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TECHNICAL REPORT ON THE ESSAKANE GOLD MINE HEAP LEACH PRE-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 Pre-feasibility Study (PFS) for its Essakane heap leach project (the Project) located in the Sahel region of Burkina Faso, West Africa. The results, which outline an economically viable project, justify the commencement of a Feasibility Study (FS) to further optimize the project development design, secure long lead equipment, and improve project economics. The purpose of this Technical Report is to disclose the results of the PFS and to support the disclosure of the June 5, 2018 Essakane Gold Mine Mineral Resource and Mineral Reserve estimate. All currency in this report is US dollars (US$) unless otherwise noted.

The Essakane Gold Mine consists of one mining permit (the Essakane Mining Permit), which contains the Essakane main zone deposit (EMZ deposit) and the Falagountou deposit, and seven exploration permits (the Essakane Exploration Permits), all located on contiguous ground. 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 Essakane Gold Mine. IAMGOLD, through its wholly-owned subsidiary Essakane S.A., owns 90% of the Essakane Gold Mine in West Africa, with the Government of Burkina Faso holding the remaining 10%.

The Essakane Gold Mine has been in operation since 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 48.0 million tonnes (Mt) in 2017 with a stripping ratio of 3.10 including 11.8 Mt of ore at an average grade of 1.17 g/t Au, for a total of 432,000 oz of gold.

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

The Mineral Resource estimate at June 5, 2018 for the Essakane Gold Mine is summarized in Table 1-1 and is reported on a 100% basis. The Mineral Resource estimates for EMZ and

Falagountou were prepared by Essakane S.A. and G Mining Services Inc. (GMSI), respectively, and are inclusive of Mineral Reserves.

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).

TABLE 1-1 MINERAL RESOURCE SUMMARY – JUNE 5, 2018

Category	Tonnage (000 t)	Grade (g/t Au)	Contained Metal (000 oz Au)
Measured	-	-	-
Indicated	159,810	0.95	4,878
Total Measured + Indicated	**159,810**	**0.95**	**4,878**
Inferred	20,744	0.88	589

Notes:

1. CIM (2014) definitions were followed for Mineral Resources.
2. Mineral Resources for the EMZ are reported at a cut-off grade of 0.33 g/t Au for saprolite, 0.43 g/t Au for transition material, and 0.30 g/t Au for fresh rock material. Cut-off grades for Falagountou are 0.36 g/t Au for saprolite, 0.46 g/t Au for transition material, and 0.52 g/t Au for fresh rock material.
3. Mineral Resources are constrained within a pit shell estimated using a long-term gold price of $1,500/oz and a US$/€ exchange rate of: 1:0.77 and a US$/CFA exchange rate of 1:0.00198.
4. Mineral Resources are inclusive of Mineral Reserves.
5. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
6. Mineral Resources are reported on a 100% basis.
7. Numbers may not add due to rounding.

The Mineral Reserve estimate at June 5, 2018 for the Essakane Gold Mine is summarized in Table 1-2 and is reported on a 100% basis. The Mineral Reserve estimates for EMZ and Falagountou were prepared by Essakane S.A.

TABLE 1-2 MINERAL RESERVE SUMMARY - JUNE 5, 2018

Category	Tonnage (000 t)	Grade (g/t Au)	Contained Metal (000 oz Au)
Proven	-	-	-
Probable	158,197	0.89	4,510
Total	**158,197**	**0.89**	**4,510**

Notes:

1. CIM (2014) definitions were followed for Mineral Reserves.
2. Mineral Reserves estimated assuming open pit mining methods.
3. Mineral Reserves are based on a gold price of $1,200/oz.

4. Mining costs ($/t mined): $2.55/t. Processing costs: $12.36/t (CIL). Processing costs $3.13/t (HL). General and Administrative costs (includes refining cost) of $3.99/t for CIL only. Heap Leach bears no G&A costs.
5. Mineral Reserves are reported on a 100% basis.
6. Mineral Reserves include material from the EMZ and Falagountou pits.
7. Numbers may not add due to rounding.

1.1.1 CONCLUSIONS

IAMGOLD has the following conclusions and observations:

1.1.1.1 GEOLOGY AND MINERAL RESOURCES

- Mineral Resources and Mineral Reserves 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.

- GMSI reviewed the information stored in the Falagountou database and found it to be in good standing.

- Drill hole spacing on the Falagountou East and West deposits is 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 Inverse Distance Cubed (ID^3) based Mineral Resource estimate for the Falagountou East and West deposits was found to be a good representation of the drill hole composites.

- Swath plots for Indicated and Inferred Mineral Resources by vertical sections for the Falagountou East and West deposits 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.

- 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.

- 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 more than sufficient to support a PFS.

1.1.1.2 MINING AND MINERAL RESERVES

- The mine design and Mineral Reserve estimate have been completed to a level appropriate for a PFS.

- 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.

1.1.1.3 METALLURGICAL TESTING AND MINERAL PROCESSING

- The metallurgical testing results indicate that the Essakane low grade material is amenable to processing by conventional heap leaching methods. Gold recovery is estimated to be 55% and reagent requirements are low.

- Detailed operating costs have been estimated based on experience and actual costs at site and are appropriate for a PFS.

- Heap leach metallurgical testing has been carried out by Kappes, Cassiday & Associates (KCA). KCA has identified the following risks that may affect the economics of the heap leach project:
    - o No metallurgical test work has been completed on the turbidite material.
    - o No metallurgical test work has been completed on the material in the stockpiles and the effects of weathering is unknown.
    - o 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.
    - o 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:
    - o Most of the column leach tests were still leaching when the tests were finished and additional recovery is likely as ore is secondarily leached through upper lifts.
    - o The high pressure grinding roll (HPGR) model selected for this study was single pass. A larger machine would allow an amount of recirculation which would result in a finer product size and potentially higher gold recovery. Test work is underway at the time of this report.
    - o The design for this study does not include cement agglomeration of the ore. Utilization of cement may increase maximum heap height or permeability requiring less liner for lower capital costs and possibly increasing gold recovery. Due to the high cost of lime, cement would only be a low added operating cost over lime. Test work is underway at the time of this report.
    - o The overall design of the crushing and stacking systems for the heap leach presented in this study is a first-pass design. The opportunity exists to optimize the general layout and individual components.

1.1.1.4 ENVIRONMENT

- No outstanding technical issues were identified for environment and permitting.

1.1.2 RECOMMENDATIONS

IAMGOLD has the following recommendations:

1.1.2.1 GEOLOGY AND MINERAL RESOURCES

- The West flank of the lithological model of the EMZ deposits should be updated for the next resource estimate in order to reflect new geological observations.

- 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.

- Estimation strategy used for EMZ could result in too much smoothing, however, reconciliation did not indicate too much smoothing in the last year. Considering a lower cut-off grade for the heap leach project, it is in the opinion of the Qualified Person (QP) that a different strategy should be investigated using the grade control results in the upcoming year. In addition to a calibration with the production, the QP suggests having an external audit to assist parameter selection.

- 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.

- GMSI suggests waiting for robust reconciliation data before making any important modifications to the Falagountou deposit block model.

- GMSI is of the opinion that the ID^3 interpolation method for the Falagountou deposit is a better global estimator compared to the Ordinary Kriging (OK) technique.

1.1.2.2 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 life of mine (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 mill operating parameters as a function of graphitic carbon concentration in the feed.

- KCA has the following recommendations:
    - Column leach tests should be conducted on the Turbidite rock type to confirm recovery.
    - Metallurgical testing should be conducted on stockpile material to check if weathering has any effect on recoveries.
    - A feasibility study is recommended to improve the reliability and accuracy of the cost estimate and form the basis for a construction decision.

1.2 ECONOMIC ANALYSIS

This section is not required as the Essakane Gold Mine is currently in production and there is no material expansion of current production.

1.3 TECHNICAL SUMMARY

1.3.1 PROPERTY DESCRIPTION AND LOCATION

The Essakane Gold Mine straddles 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 Essakane Gold Mine are subject to Burkina Faso's 2015 Mining Code No.3 036-2015/CNT, dated June 26, 2015 (the Burkina Faso Mining Law). The Essakane Gold Mine consists of one mining permit (the Essakane Mining Permit), which contains the EMZ deposit and the Falagountou deposit. The mining permit is surrounded by seven exploration permits (the Essakane Exploration Permits) belonging to Essakane Exploration SARL, the exploration subsidiary of IAMGOLD working in the region of the Essakane Gold Mine.

1.3.2 LAND TENURE

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 Essakane Gold Mine. 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 of Burkina Faso was signed on July 14, 2008.

IAMGOLD owns a 90% interest in Essakane S.A., while the Government of Burkina Faso has a 10% free-carried interest. In addition, the Government of Burkina Faso 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 1,093.19 km^2 after their second renewal. 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

The Essakane Gold Mine, 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 a heap leach facility which produced 18,000 oz of gold in 1993 but averaged between 3,000 oz and 5,000 oz 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 the mine 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 Essakane Gold Mine. On November 26, 2007, Orezone Resources became the operator and owner of a 100% interest in the Essakane Gold Mine, subject to the interest of the Government of Burkina Faso.

In April 2008, the Essakane Mining Permit was granted over an area of 100.2 km^2 containing the EMZ deposit and the Falagountou deposit. An updated FS (the UFS) was completed on June 3, 2008 and readdressed to IAMGOLD in 2009 after IAMGOLD acquired Orezone Resources and the Essakane Gold Mine 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 Essakane Gold Mine 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 zone. 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

The Essakane Gold Mine has been explored since the 1990s by geochemical sampling, mapping, trenching, Aster/Landsat image analysis and interpretation, geophysical surveys, and drilling. Exploration efforts at the Essakane Gold Mine were initially focused 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.

Reverse circulation (RC) and diamond drilling (DD) drilling has been conducted by Essakane S.A.'s Resource Development Group since January 2010. As of February 2018, a total of 2,279 RC holes 270,208 m) and 968 DD holes (267,913 m) had been drilled within the EMZ and Falagountou pits.

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 June 5, 2018.

An infill RC and DD program conducted at the Falagountou deposit, since the previous Falagountou Mineral Resource estimate in 2016, confirmed lateral continuity of mineralization oriented mostly north-south as well as an extension down-dip, which remained open. Drilling also identified a second mineralized structure, located 250 m west of the main zone. A total of 342 RC and DD holes, for a total of approximately 51,498 m, have been drilled and results incorporated into the current Mineral Resource estimate for the Essakane Gold Mine.

1.3.6 MINERAL RESOURCES

The Mineral Resource estimate 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 QP, the resource estimation reported herein is a reasonable representation of the Mineral Resources delineated at the Essakane Gold Mine as of June 5, 2018.

The Mineral Resource estimate at June 5, 2018 for the Essakane Gold Mine 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 Mineral Resource estimate as of December 31, 2017, the EMZ resource model has been updated with new drilling information. The modelling work was completed by Essakane S.A. personnel. As of June 5, 2018, total EMZ Measured and Indicated Mineral Resources are estimated at approximately 150 Mt grading 0.91 g/t Au containing 4,340 koz of gold. In addition, Inferred Mineral Resources are estimated to be approximately 18.9 Mt at 0.78 g/t Au containing 474 koz.

The Falagountou West Mineral Resource model remains unchanged since the previous estimate prepared by GMSI in October 2015. The Falagountou East Mineral Resources were estimated by GMSI in August 2016, and subsequently updated in March 2017 to include infill and extensional drilling. As of June 5, 2018, total Falagountou West and East Measured and Indicated Mineral Resources are estimated at approximately 10.7 Mt grading 1.56 g/t Au containing 539 koz of gold. In addition, Inferred Mineral Resources are estimated to be approximately 1.8 Mt at 2.00 g/t Au containing 115 koz.

The 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 June 5, 2018 for the Essakane Gold Mine is summarized in Table 1-2 and is reported on a 100% basis. The Mineral Reserve estimates for EMZ and Falagountou were prepared by Essakane S.A.

The addition of the heap leach process in 2018 has increased the EMZ Mineral Reserve inventory. As of June 5, 2018, the EMZ Mineral Reserves are estimated to total 87.9 Mt at 1.14 g/t Au of CIL Mineral Reserves and 60.5 Mt at 0.43 g/t Au of heap leach Mineral Reserves.

As of June 5, 2018, the Falagountou Mineral Reserves are estimated to be 5.1 Mt at 1.61 g/t Au in the West pit and 4.2 Mt at 1.32 g/t Au in the East pit. The Falagountou deposits have no Mineral Reserves attributed to the heap leach process due to the distance they are located from the EMZ pit and the short mine life, which ends in 2020 for the West pit and 2021 for the East pit.

In addition to the above tonnage, the Falagountou stockpile totals 0.6 Mt grading 0.81 g/t Au.

The mine design and Mineral Reserve estimate have been completed to a level appropriate for pre-feasibility studies. The Mineral Reserve estimate stated herein is consistent with 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.

The 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 48.0 Mt in 2017 with a stripping ratio of 3.10 including 11.7 Mt of ore at an average grade of 1.16 g/t Au for a total of 432,000 oz of gold.

The Essakane Gold Mine consists of several operating sites. The Essakane main pit is mined in several mining phases and accounts for over 80% of the production. The Falagountou and Essakane North satellite pits provide additional ore and operational flexibility.

Current mining production is approximately 50 Mtpa, however, a mine expansion to 70 Mtpa is planned. In 2018, SRK Consulting (Canada) Inc. (SRK) carried out a geotechnical stability assessment of the proposed Essakane main pit expansion, which indicated that the rock masses are of sufficient strength to support the proposed overall stability, and the major structures are not expected to adversely impact overall stability.

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.

All blasting activities on site are executed by an explosives supplier. Holes are loaded with bulk explosive matrix and initiated 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 four RH-120 shovels, four CAT 993K wheel loaders, two CAT 390 and six CAT 345/349 excavators. The mine's hauling fleet currently consists of 26 CAT 785C and five CAT 777F mining trucks. In view of the mine expansion, additional mining and auxiliary equipment will be required.

Mine haul roads are 20 m to 30 m wide and are constructed by the mining department to support the mine haul trucks.

Waste material is being stored in the waste dumps located east of the Essakane 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.

1.3.9 MINERAL PROCESSING

Essakane ore is processed using two stages of crushing, SAG, ball mill grinding, SABC, gravity concentration, and a CIL gold plant. The 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 mill feed has gradually increased from 2012 onwards. To maintain gold production levels, with increasing proportions of hard rock in the mill feed, an expansion was completed in 2014. The objective was to double the hard rock processing capacity from 5.4 Mtpa on a 100% hard 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 mill.

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

1.3.10 CONCENTRATOR MODIFICATIONS FOR THE TREATMENT OF GOLD LOADED HEAP LEACH CARBON

The gold loaded carbon from the heap leach process (4.5 tonnes per day, or tpd) will be transported to the Essakane concentrator to be stripped in the existing elution circuit. New unloading equipment will be added as well as a tank to store the carbon.

The elution process is designed to process 17 tonnes of carbon per batch. Once 17 tonnes of carbon from the heap leach circuit has been obtained, the carbon will be processed in the elution circuit, separately from the CIL carbon. Due to the additional carbon from the heap leach, the treatment frequency will increase from one batch per day to 1.3 batches per day. The time that the equipment is idle will be minimized and the carbon transfer time from the CIL circuit will be reduced. The acid wash and elution cycle takes up to 20 hours to complete and at the present time, a new batch is started after 24 hours. In order to reach the required treatment rate, a new cycle will be started in the acid wash column before the end of the elution cycle. Since the existing loaded carbon screen limits the transfer time of carbon from the CIL circuit, the loaded carbon screen will be replaced to increase its capacity.

In addition, two new electrowinning cells will be added. This will ensure that the cathodes are washed before and after processing the carbon from the heap leach and that the elution cycles frequency is not limited by the cathode washing stage. Using the existing carbon regeneration kiln, approximately 73% of the carbon processed will be regenerated. The target proportion of carbon regenerated will be revisited in the FS to aim at 100% regeneration. After regeneration, regenerated and fresh carbon dedicated to the heap leach will be directed to a storage tank prior to loading the carbon transport truck. One truck per day will be returned to the heap leach circuit.

1.3.11 HEAP LEACH FACILITY

The preliminary Heap Leach Facility (HLF) is designed for low environmental risk to soils, surface water, and groundwater in and around the site. The HLF will be constructed in three phases. The first phase is designed for two years of operation, the second phase is intended to come on line in Year 3 and the third phase is intended to come on line in Year 5. The staged construction of the heap is intended to minimize the up-front capital costs and help with the water balance. Phase 2 will expand the heap leach pad and the excess solution pond. The Phase 3 expansion will only increase the size of the heap leach pad.

1.3.12 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. As of May 31, 2018, the manpower status is 2,236 national workers and 95 expatriates, excluding contractors.

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 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 the Project team. An area will be designated for the main contractor use which will be fenced and a working pad will be built so the contractor can install its base camp, warehoused offices, and workshop. Potable water and power will be provided by the Owner.

Existing workshops and a construction warehouse will be reused for the 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.

1.3.13 MARKET STUDIES AND CONTRACTS

Gold is the principal commodity at the Essakane Gold Mine 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 refined the doré into bullion. The bullion is then sold directly on the open market to gold trading institutions at prevailing market prices.

1.3.14 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 Piesold 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 mill 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. An Environmental Impact Assessment (EIA) was conducted for the river diversion. 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 and the RAP report will be tabled at the end of 2018. 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 digger, farmer, etc.) were performed during the field work. However, an ongoing consultation process was put in place as part of the ESIA.

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.

A conceptual rehabilitation and closure plan (PRF) was developed in 2009 and last updated in 2013. An updated version of the closure plan will be available in December 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.

1.3.15 CAPITAL AND OPERATING COST ESTIMATES

The capital cost requirement over the LOM includes heap leach project capital expenditures; resource development costs; capitalized waste stripping; sustaining capital; mine equipment additions and replacements; equipment overhaul costs; equipment capital spares; and closure and remediation costs.

A total of $894.3M of capital is planned to be spent over the remaining LOM, which equates to $5.46/t milled (CIL + HL) or $221/oz of Au sold. The heap leach project initial capital cost is estimated at $152.7M.

The mine 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 (2018 to 2022) are shown in Table 1-3.

TABLE 1-3 LOM AND FIVE YEAR PLAN OPERATING COSTS

Area	LOM Average	Five Year Plan (2018-2022)
Mining (US$/t mined)	2.76	2.64
CIL Processing (US$/t milled)	12.00	11.86
Heap Leach Processing (US$/t milled)	3.13	3.13
G&A (US$/t milled)	3.73	3.85

The average total cash cost per ounce is US$707/oz Au while the all-in sustaining cost (AISC) averages US$946/oz Au over the LOM.

1.3.16 PROJECT EXECUTION PLAN

The Essakane heap leach 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 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 major Project milestones from the PFS are presented in Table 1-4.

TABLE 1-4 MAJOR PROJECT MILESTONES

Description	Start Date	Completion Date
Feasibility study engineering - earthworks	Q3 - 2018	Q1 - 2019
Feasibility study engineering - power	Q3 - 2018	Q1 - 2019
Feasibility study engineering - heap leach	Q3 - 2018	Q1 - 2019
Feasibility study completed	Q1 - 2019	
Detailed engineering	Q1 - 2019	Q3 - 2019
Permitting expected date	Q1- 2019	
Long lead item procurement	Q1 - 2019	Q4 - 2019
Early works	Q1- 2019	Q2-2019
Project full approval	Q2 - 2019	
Construction	Q2 - 2019	Q2 - 2020
Heap leach production	Q2 - 2020	

2 INTRODUCTION

IAMGOLD Corporation (IAMGOLD) has prepared a Pre-feasibility Study (PFS) for its Essakane heap leach project (the Project) located in the Sahel region of Burkina Faso, West Africa. The results, which outline an economically viable project, justify the commencement of a Feasibility Study (FS) to further optimize the project development design, secure long lead equipment and improve project economics. The purpose of this Technical Report is to disclose the results of the PFS and to support the disclosure of the June 5, 2018 Essakane Gold Mine Mineral Resource and Mineral Reserve estimate. 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. IAMGOLD, through its wholly-owned subsidiary Essakane S.A., owns 90% of the Essakane Gold Mine in West Africa, with the Government of Burkina Faso holding the remaining 10%. The mine has been in operation since July 2010.

2.1.1 SOURCES OF INFORMATION

This Technical Report was prepared by IAMGOLD and Kappes, Cassiday & Associates (KCA), and incorporates the work of IAMGOLD, KCA, SRK Consulting (Canada) Inc. (SRK), Soutex, and G Mining Services Inc. (GMSI) Qualified Persons (QPs). The dates of personal inspections of the Essakane Gold Mine 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
A	ampere
bbl	barrels
btu	British thermal units
°C	degree Celsius
C$	Canadian dollars
cal	calorie
cfm	cubic feet per minute
cm	centimetre
cm^2	square centimetre
d	day
dia	diameter
dmt	dry metric tonne
dwt	dead-weight ton
°F	degree Fahrenheit
ft	foot
ft^2	square foot
ft^3	cubic foot
ft/s	foot per second
g	gram
G	giga (billion)
Gal	Imperial gallon
g/L	gram per litre
Gpm	Imperial gallons per minute
g/t	gram per tonne
gr/ft^3	grain per cubic foot
gr/m^3	grain per cubic metre
ha	hectare
hp	horsepower
hr	hour
Hz	hertz
in.	inch
in^2	square inch
J	joule
k	kilo (thousand)
kcal	kilocalorie
kg	kilogram
km	kilometre
km^2	square kilometre
km/h	kilometre per hour
kPa	kilopascal
kVA	kilovolt-amperes
kW	kilowatt
kWh	kilowatt-hour
L	litre
lb	pound
L/s	litres per second
m	metre
M	mega (million); molar
m^2	square metre
m^3	cubic metre
µ	micron
MASL	metres above sea level
mbgs	metres below ground surface
µg	microgram
m^3/h	cubic metres per hour
mi	mile
min	minute
µm	micrometre
mm	millimetre
mph	miles per hour
MVA	megavolt-amperes
MW	megawatt
MWh	megawatt-hour
oz	Troy ounce (31.1035g)
oz/st, opt	ounce per short ton
ppb	part per billion
ppm	part per million
psia	pound per square inch absolute
psig	pound per square inch gauge
RL	relative elevation
s	second
st	short ton
stpa	short ton per year
stpd	short ton per day
t	metric tonne
tpa	metric tonne per year
tpd	metric tonne per day
US$	United States dollar
USg	United States gallon
USgpm	US gallon per minute
V	volt
W	watt
wmt	wet metric tonne
wt%	weight percent
yd^3	cubic yard
yr	year

3 RELIANCE ON OTHER EXPERTS

This report has been prepared by IAMGOLD. 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 Report.

4 PROPERTY DESCRIPTION AND LOCATION

The Essakane Gold Mine straddles 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. It is situated approximately 42 km east of the nearest large town and the provincial capital of Oudalan, 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.

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

4.1 MINING PERMIT

The mining and exploration permits comprising the Essakane Gold Mine 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), the completion of an Environmental and Socio-economic Impact Assessment (ESIA), and the obtaining of the Essakane Environmental Permit, the Government of Burkina Faso granted to Essakane S.A. the Essakane Mining Permit over an area of 100.2 km^2 containing the EMZ deposit and the Falagountou deposit. The Essakane Mining Permit is valid for a period of 20 years and is renewable every five years until mining 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 limits are shown in Figure 4-2.

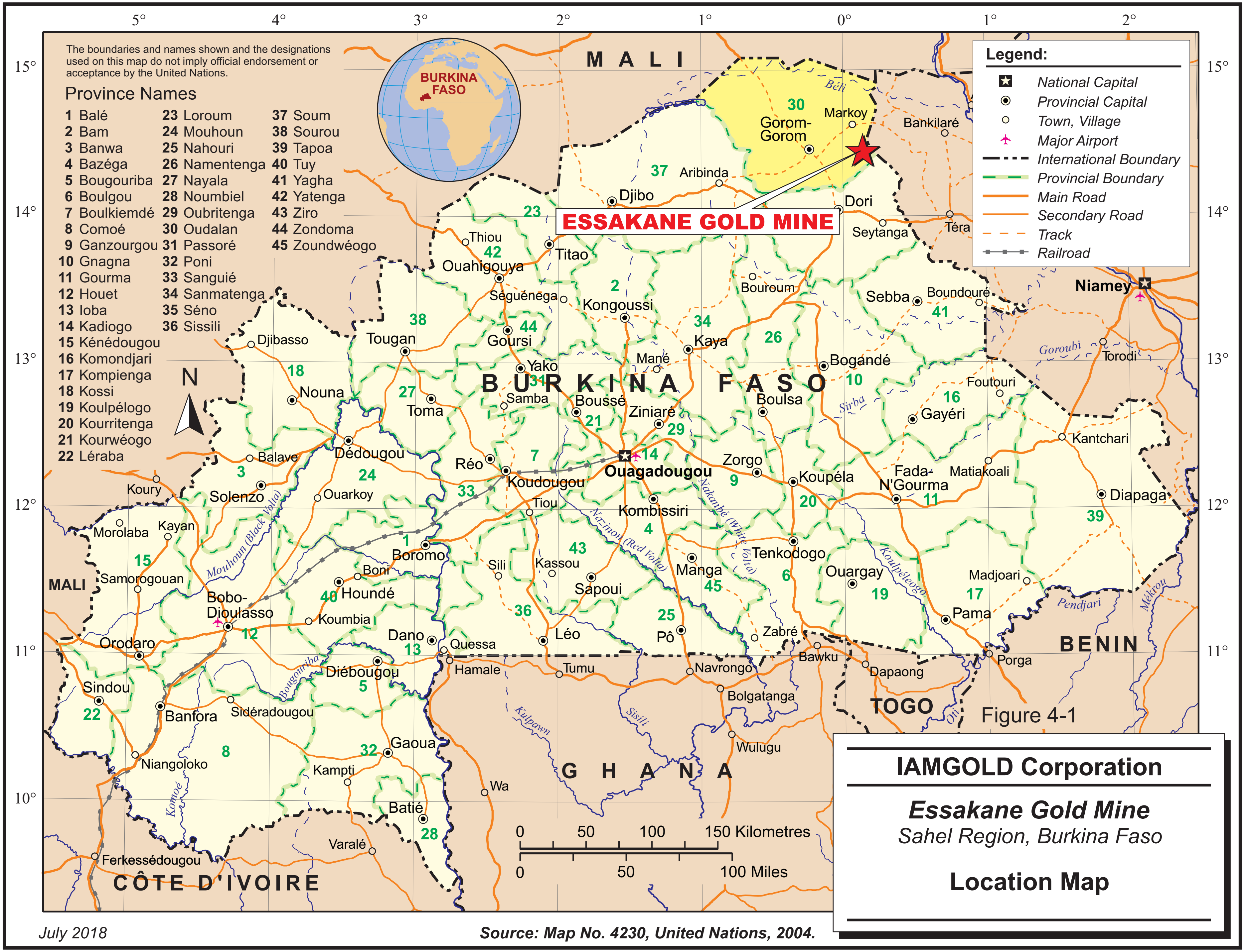
TABLE 4-1 ESSAKANE MINING PERMIT BOUNDARY COORDINATES

Points	Datum	X	Y
A	Adindan BF	177,115	1,592,488
B	Adindan BF	180,607	1,592,488
C	Adindan BF	180,607	1,594,564
D	Adindan BF	188,770	1,594,564
E	Adindan BF	188,770	1,592,379
F	Adindan BF	194,430	1,592,379
G	Adindan BF	194,367	1,587,187
H	Adindan BF	181,104	1,587,187
I	Adindan BF	181,104	1,589,186
J	Adindan BF	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 start. The mining convention describes 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 can choose to adopt them (if deemed more advantageous) or stay with the current terms of the mining convention. The mining convention between Essakane S.A. and the Government of Burkina Faso 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.

Essakane S.A is a Burkinabé company created for the purpose of developing and operating the Essakane Gold Mine. IAMGOLD owns a 90% interest in Essakane S.A., while the Government of Burkina Faso has a 10% free-carried interest. In addition, the Government of Burkina Faso 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

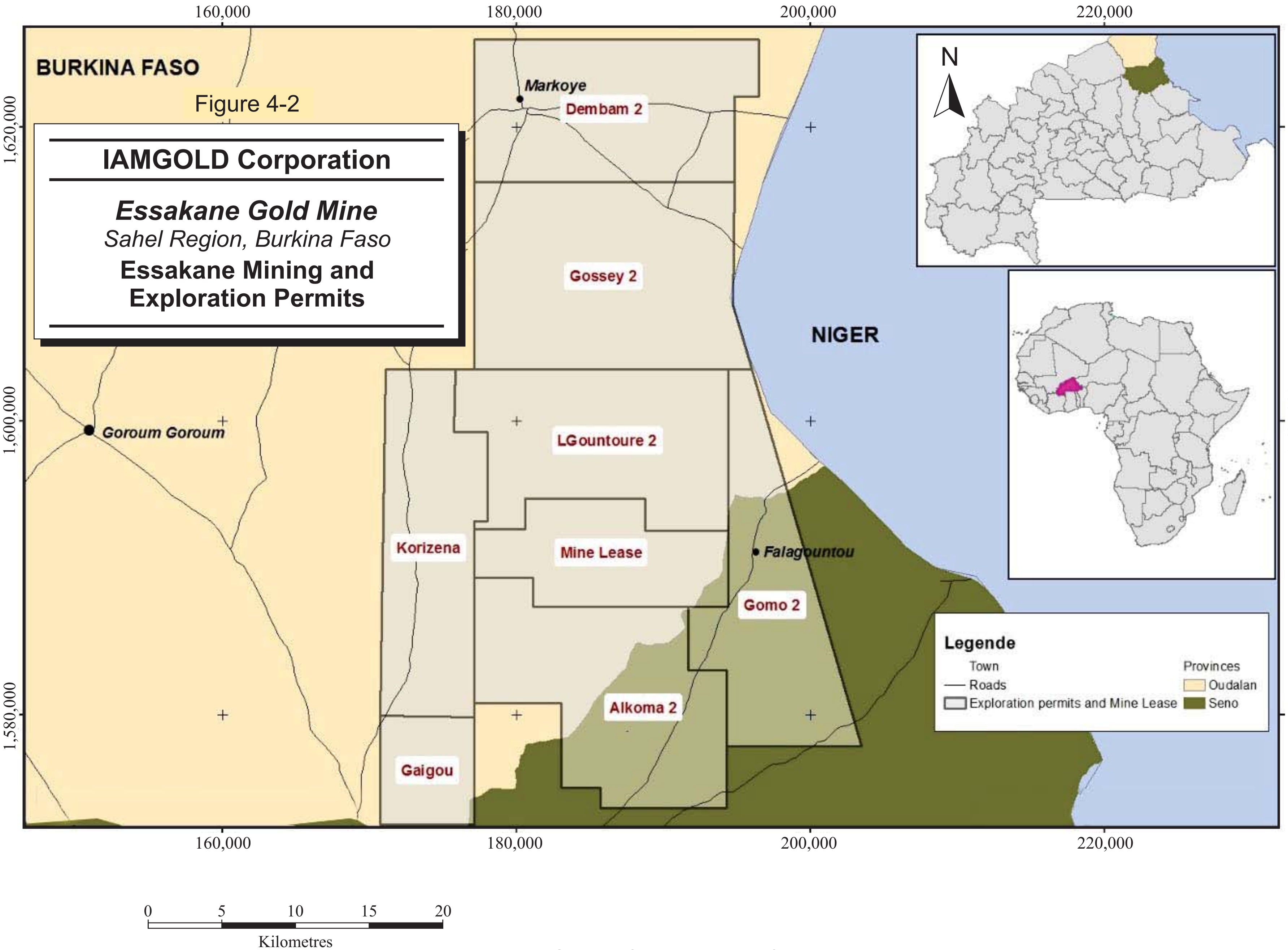
Figure 4-2

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Essakane Mining and Exploration Permits



NOTE: Refer to Table 4-1 for Essakane Mining Permit boundary coordinates.

July 2018

Source: Essakane Exploration SARL, 2018.

IAMGOLD Corporation

4.2 EXPLORATION PERMITS

The Essakane Mining Permit is surrounded by the Essakane Exploration Permits, which currently cover a total area of 1,093.19 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 francs CFA 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).

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 an arrêté 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 Minister in November 2009 for an initial three-year term ending in November 2012, and were approved for renewal by the Minister for the first three-year term on December 18, 2012. The request for a second renewal was submitted to the Minister 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 Minister to keep the original surface area.

The sixth Essakane Exploration Permit (Korizena 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 Minister on May 6, 2013. On August 18, 2015, a request for extending the actual surface area of the Korizena permit for another three year period was submitted to the Minister.

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.

At the completion of the renewal process, the total surface area of the Essakane Exploration Permits is 1,093.19 km^2.

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 arrêté numbers and expiry dates are listed in Table 4-2, and the exploration permit coordinates (projection Clark 1880; Adindan BF) are listed in Table 4-3.

TABLE 4-2 EXPLORATION PERMIT DETAILS

Permit Name	Arrêté Granted	Date Granted	Status	Surface area (km²)	Arrêté Renewed	Renewal Date	Expiry Date
ALKOMA 2	09/262/MCE/SG/DGMGC	24/11/2009	Second renewal	186.60	16/020/MEMC/SG/DGCM	24/02/2016	24/11/2018
DEMBAM 2	09/263/MCE/SG/DGMGC	24/11/2009	Second renewal	177.70	16/019/MEMC/SG/DGCM	18/12/2012	24/11/2018
GAIGOU	2013/000076/MME/SG/DGMGC	06/05/2013	First renewal	48.05	16/158/MEMC/SG/DGCMIM	01/09/2016	06/05/2019
GOMO 2	09/261/MCE/SG/DGMGC	24/11/2009	Second renewal	149.50	16/027/MEMC/SG/DGCM	25/02/2016	24/11/2018
GOSSEY 2	09/260/MCE/SG/DGMGC	24/11/2009	Second renewal	215.00	16/037/MEMC/SG/DGCM	11/03/2016	24/11/2018
KORIZENA	06/135/MCE/SG/DGMGC	21/11/2006	Second renewal (exceptional)	144.18	16/059/MEMC/SG/DGCM	01/04/2016	24/11/2018
LAO GOUNTOURÉ 2	09/264/MCE/SG/DGMGC	24/11/2009	Second renewal	172.16	16/036/MEMC/SG/DGCM	11/03/2016	24/11/2018
7 PERMITS				**1,093.19**			

TABLE 4-3 EXPLORATION PERMIT COORDINATES

Permit Name	Points	Datum	Zone	X	Y	Surface area (km²)
ALKOMA 2	A	Adindan BF	31 N	177115	1589186	186.60
	B	Adindan BF	31 N	181104	1589186	
	C	Adindan BF	31 N	181104	1587187	
	D	Adindan BF	31 N	191703	1587187	
	E	Adindan BF	31 N	191703	1582874	
	F	Adindan BF	31 N	194297	1582874	
	G	Adindan BF	31 N	194297	1573467	
	H	Adindan BF	31 N	185681	1573467	
	I	Adindan BF	31 N	185681	1574843	
	J	Adindan BF	31 N	183017	1574843	
	K	Adindan BF	31 N	183017	1580620	
	L	Adindan BF	31 N	177115	1580620	
DEMBAM 2	A	Adindan BF	31 N	177117	1625786	177.70
	B	Adindan BF	31 N	196480	1625786	
	C	Adindan BF	31 N	196480	1621932	
	D	Adindan BF	31 N	194829	1621932	
	E	Adindan BF	31 N	194829	1616114	
	F	Adindan BF	31 N	177117	1616114	
GAIGOU	A	Adindan BF	30 N	818309	1579870	48.05
	B	Adindan BF	31 N	177115	1579870	
	C	Adindan BF	31 N	177115	1572302	
	D	Adindan BF	30 N	818309	1572302	
GOMO2	A	Adindan BF	31 N	194407	1603335	149.50
	B	Adindan BF	31 N	196002	1603335	
	C	Adindan BF	31 N	203498	1577707	
	D	Adindan BF	31 N	194297	1577707	
	E	Adindan BF	31 N	194297	1582874	
	F	Adindan BF	31 N	191703	1582874	
	G	Adindan BF	31 N	191703	1587187	
	H	Adindan BF	31 N	194407	1587187	
GOSSEY 2	A	Adindan BF	31 N	177117	1616115	215
	B	Adindan BF	31 N	194829	1616114	
	C	Adindan BF	31 N	194708	1607752	
	D	Adindan BF	31 N	196002	1603335	
	E	Adindan BF	31 N	177117	1603335	
KORIZENA	A	Adindan BF	30 N	817953	1603335	144.18
	B	Adindan BF	30 N	822693	1603335	
	C	Adindan BF	30 N	822693	1599074	
	D	Adindan BF	31 N	178010	1599074	
	E	Adindan BF	31 N	178010	1593002	
	F	Adindan BF	31 N	177113	1593002	
	G	Adindan BF	31 N	177119	1579656	
	H	Adindan BF	30 N	817953	1580010	

Permit Name	Points	Datum	Zone	X	Y	Surface area (km²)
LAO GOUNTOURÉ 2	A	Adindan BF	30 N	822693	1603335	172.16
	B	Adindan BF	31 N	194407	1603335	
	C	Adindan BF	31 N	194388	1592380	
	D	Adindan BF	31 N	188769	1592379	
	E	Adindan BF	31 N	188769	1594565	
	F	Adindan BF	31 N	180605	1594565	
	G	Adindan BF	31 N	180605	1592489	
	H	Adindan BF	31 N	177113	1592489	
	I	Adindan BF	31 N	177113	1593002	
	J	Adindan BF	31 N	178010	1593002	
	K	Adindan BF	31 N	178010	1599074	
	L	Adindan BF	30 N	822693	1599074	
7 PERMITS						**1,093.19**

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 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 the Essakane Gold Mine 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 of December 26, 2007 implementing the management of a Mining Environment Preservation and Rehabilitation Fund.	Mining Environment Preservation and Rehabilitation Fund	Completed
Law No. 31/-2003/AN implementing the Mining Code in Burkina Faso	Mining permit (2008)	Completed
	Rehabilitation and closing	Ongoing
	Environmental discharge	Ongoing

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 TOPOGRAPHY, ELEVATION AND VEGETATION

The Essakane Gold Mine 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 mine site is 250 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 by a 263 km paved road to the town of Dori, and then approximately 63 km by 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 mine and daily flights are made between the mine and Ouagadougou using an aircraft owned and operated by Essakane S.A.

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

5.3 CLIMATE AND LENGTH OF OPERATING SEASON

The Essakane property is located in the northeast of Burkina Faso and the climate is typically Sahelian. Temperature ranges 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 M m^3/year. Rainfall is sporadic or absent during the rest of the year. Weather conditions have so far had minimal impact on mining operations, 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 and the Falagountou deposits, process plant, the tailings storage facility, and the waste dumps.

Electricity to the EMZ deposit is provided by on-site diesel generators. A 26 MW power plant, fueled with heavy fuel oil, 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 will provide 15 MW to the Essakane Gold Mine without any carbon-emission and will help to reduce the reliance on fossil fuels. In addition, this initiative will protect the environment.

Satellite communication is available at the mine. 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. From 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.

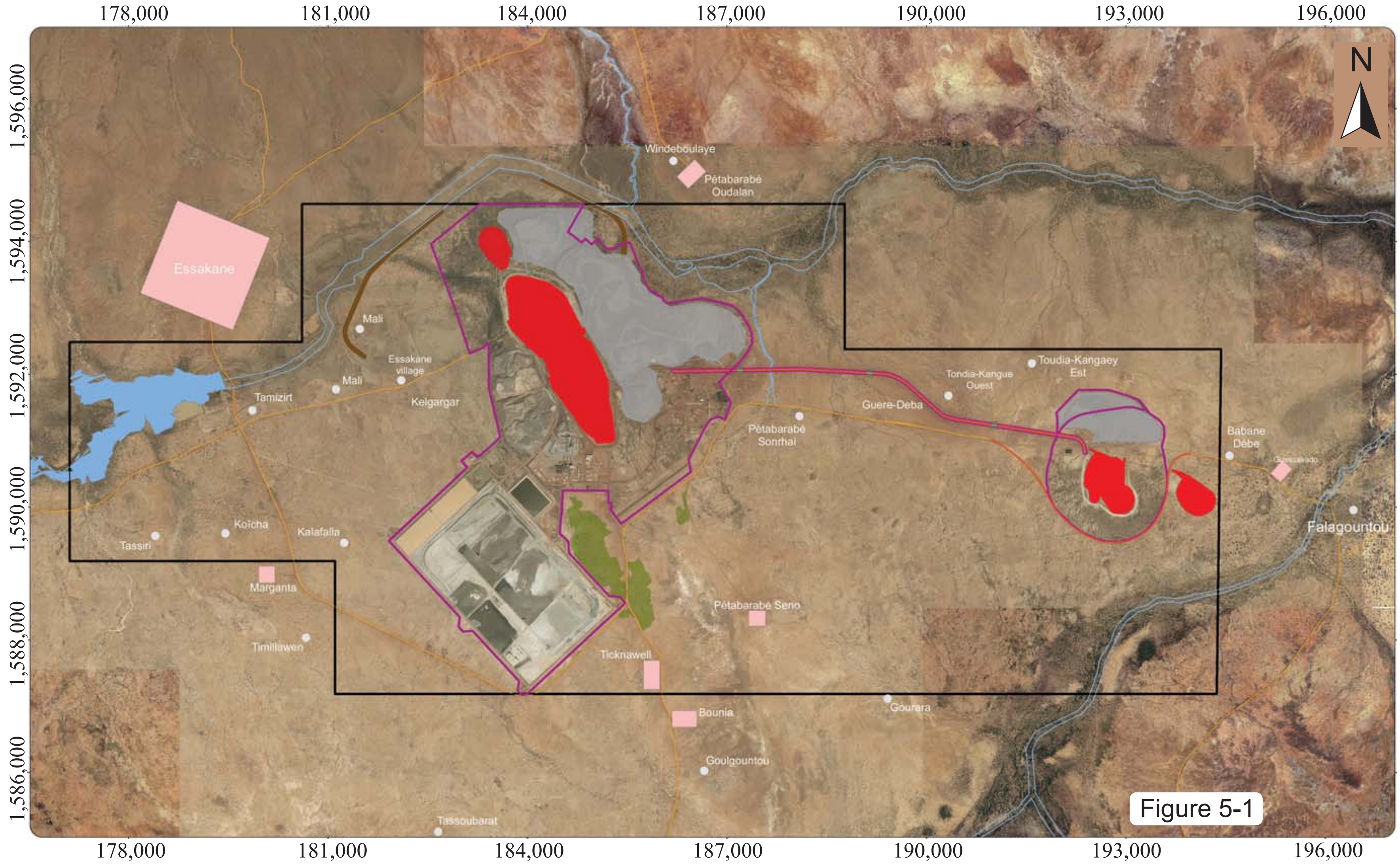


Figure 5-1

Legend:

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

0 1 2 3 4
Kilometres

Projection: UTM Zone 31
Datum: WGS 84

IAMGOLD Corporation

Essakane Gold Mine
Sahel Region, Burkina Faso
Mine Infrastructure

July 2018

Source: Essakane S.A., 2018.

IAMGOLD Corporation

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 testwork 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 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. Serious 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 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 diamond 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 Inc. (Orezone Resources). Orezone Resources became 90% owner of Essakane S.A.

Gold Fields Orogen Holding (BVI) 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 the mine until Gold Fields Essakane (BVI) Limited (GF BVI) assumed management responsibilities in January 2006.

In 2006, GF BVI carried out an exploration program on the deposit which focused 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 (BVI) 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 $8M on exploration. GF BVI increased its ownership to 60% in the Essakane Gold Mine 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 the Essakane Gold Mine. On November 26, 2007, Orezone Resources became the operator and owner of a 100% interest in the Essakane Gold Mine, subject to the interest of the Burkina Faso 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 mine to Essakane S.A., a Burkinabé anonymous company, created for the purpose of owning and operating the Essakane Gold Mine. An updated FS (the UFS) was completed on June 3, 2008.

In 2009, IAMGOLD acquired Orezone Resources and the Essakane Gold Mine was transferred to IAMGOLD Essakane S.A. The June 3, 2008 UFS was readdressed to IAMGOLD. Commercial production started on July 16, 2010.

The ownership history is summarized in Figure 6-1.

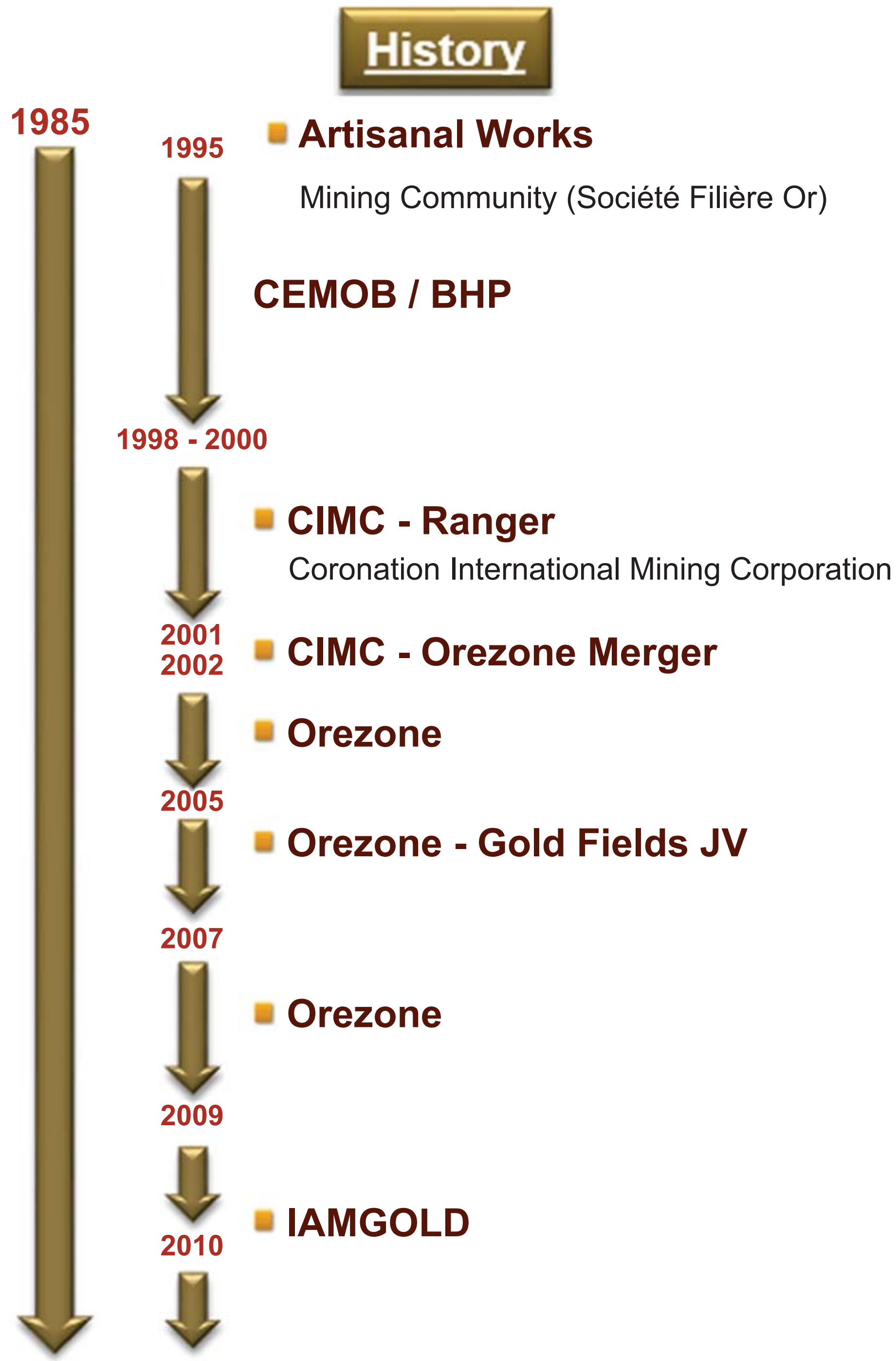


Figure 6-1

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Ownership History

July 2018

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 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)	Tonnes (000)	Grade (g/t Au)	Gold (000 oz)
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 UFS filed by IAMGOLD in March 2009, upon acquisition of the mine.

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 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 mill production from 2010 to 2015. It is estimated that a total of 2.5 million ounces of gold has been produced since 1992.

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

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

Year	Tonnes (000)	Grade (g/t Au)	Gold Ounces
1992	42	4.5	5,915
1993	116	5.1	18,388
1994	157	1.7	8,304
1995	148	1.5	6,923
1996	257	1.0	7,918
1997	165	0.8	4,321
1998	72	1.4	3,145
1999	50	2.0	3,151
Total	**1,007**	**1.9**	**58,065**

TABLE 6-3 ESSAKANE MINE AND MILL PRODUCTION 2010 TO JUNE 5, 2017

Year	Tonnes Mined (000 t)	Grade Mined (g/t Au)	Tonnes Milled (000 t)	Grade Milled (g/t Au)	Ounces Produced (000)
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	11,518	1.14	11,716	1.23	426
2016	10,921	1.21	12,005	1.22	419
2017	11,811	1.17	13,891	1.07	432
June 5, 2018	4,742	1.18	5,516	1.21	197
Total	**93,210**	**1.07**	**87,350**	**1.16**	**2,981**

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, Korizena, 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 meta-volcanoclastic, greywacke, meta-conglomerate, 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 (Hein, 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 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 (Nkuna, 2009). Gold deposits are most often related to transcurrent D2 shear zones and faults as these discontinuities have served as the main conduct of 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).

0 100 200 300 400
Kilometres

Figure 7-1

Legend:

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

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Location of Oudalan-Gorouol Greenstone Belt within West African Craton

July 2018

Source: Modified after Abouchami et al., 1990.

IAMGOLD Corporation

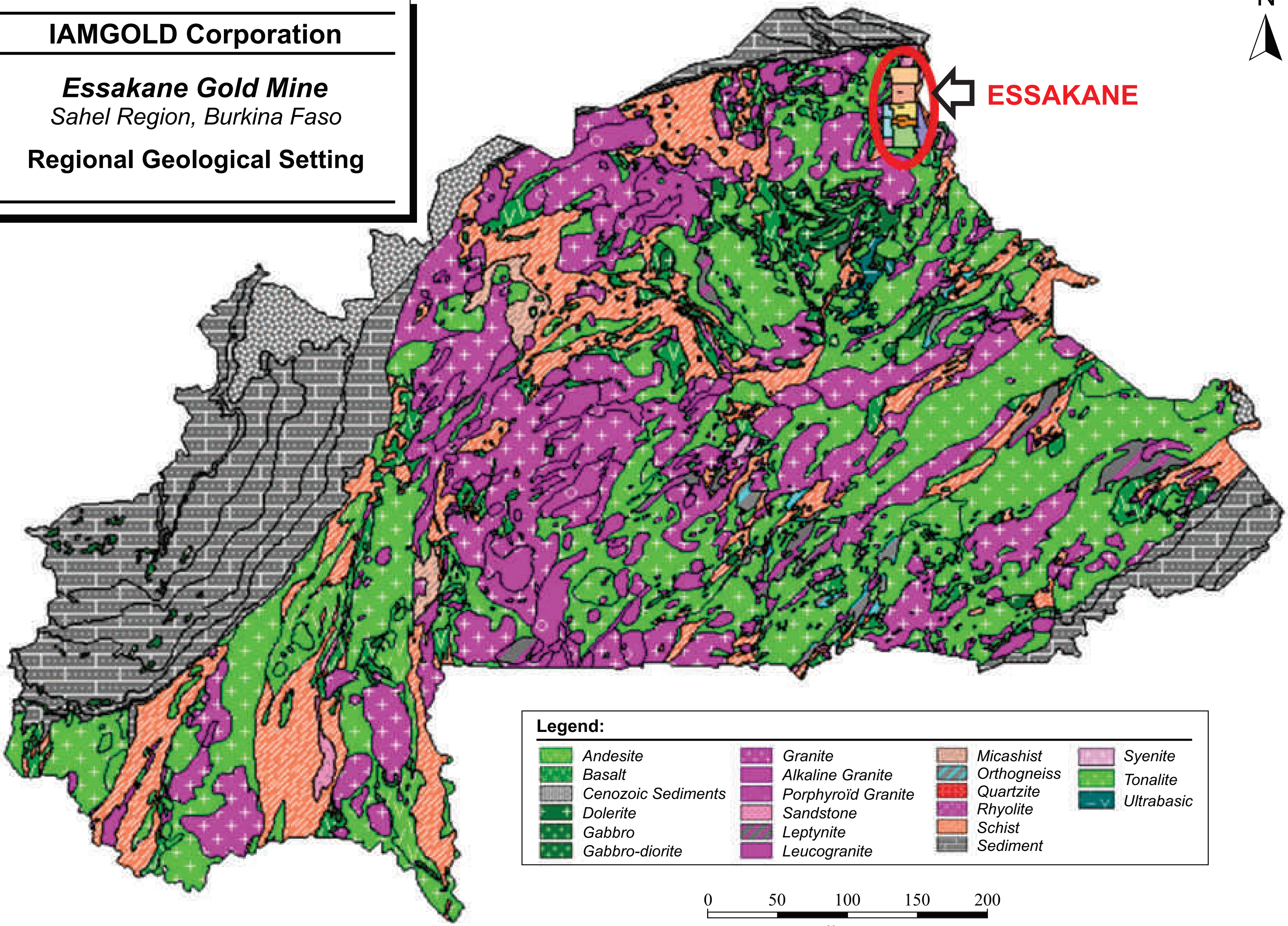
Figure 7-2

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Regional Geological Setting



July 2018

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

The Essakane Gold Mine 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 of the local geology. The bold red shape outline 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).

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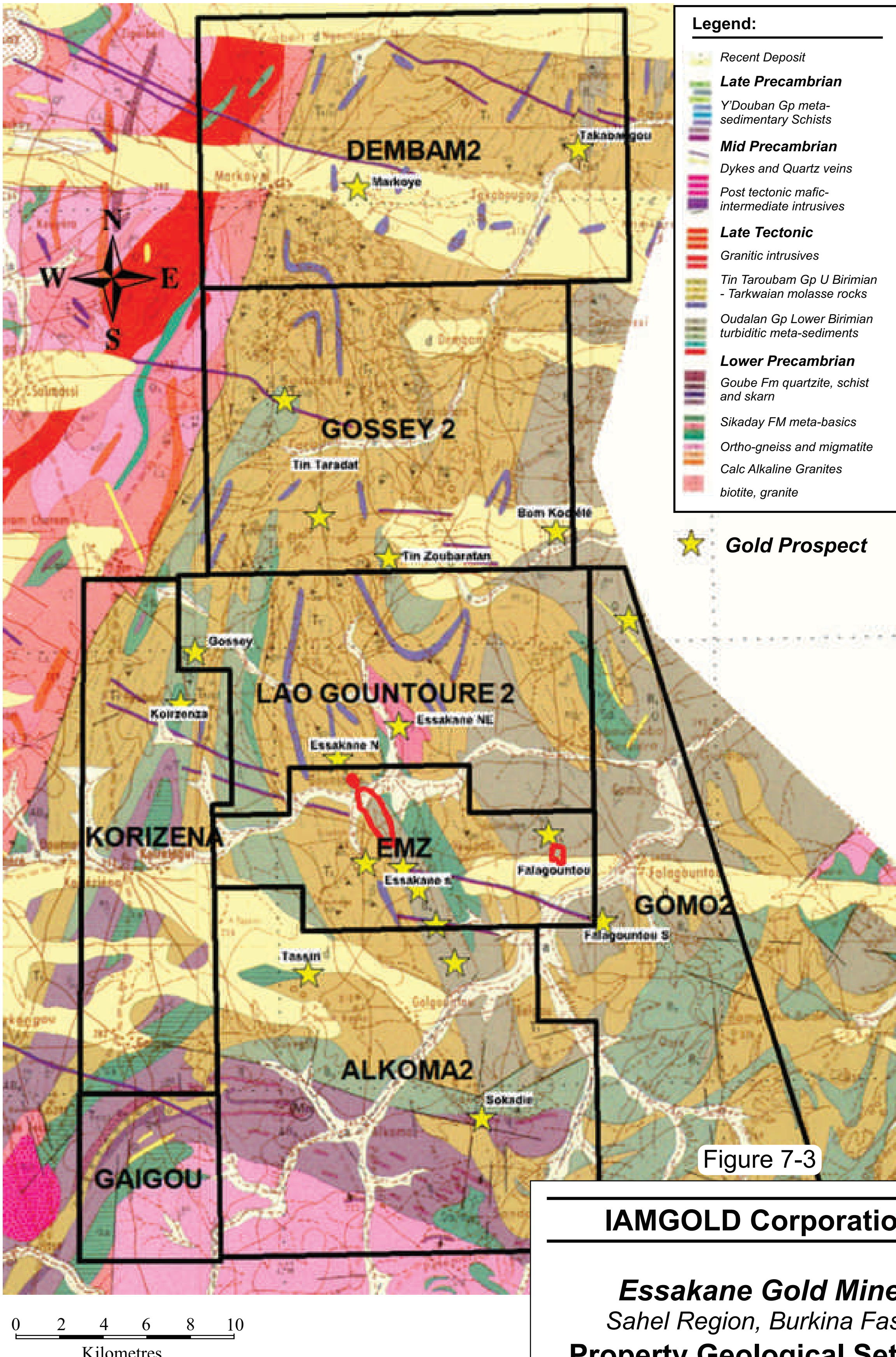

Figure 7-3

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Property Geological Setting

Source: Modified after "carte géologique de l'Oudalan au 1/2 000 000" BRGM, 1970.

July 2018

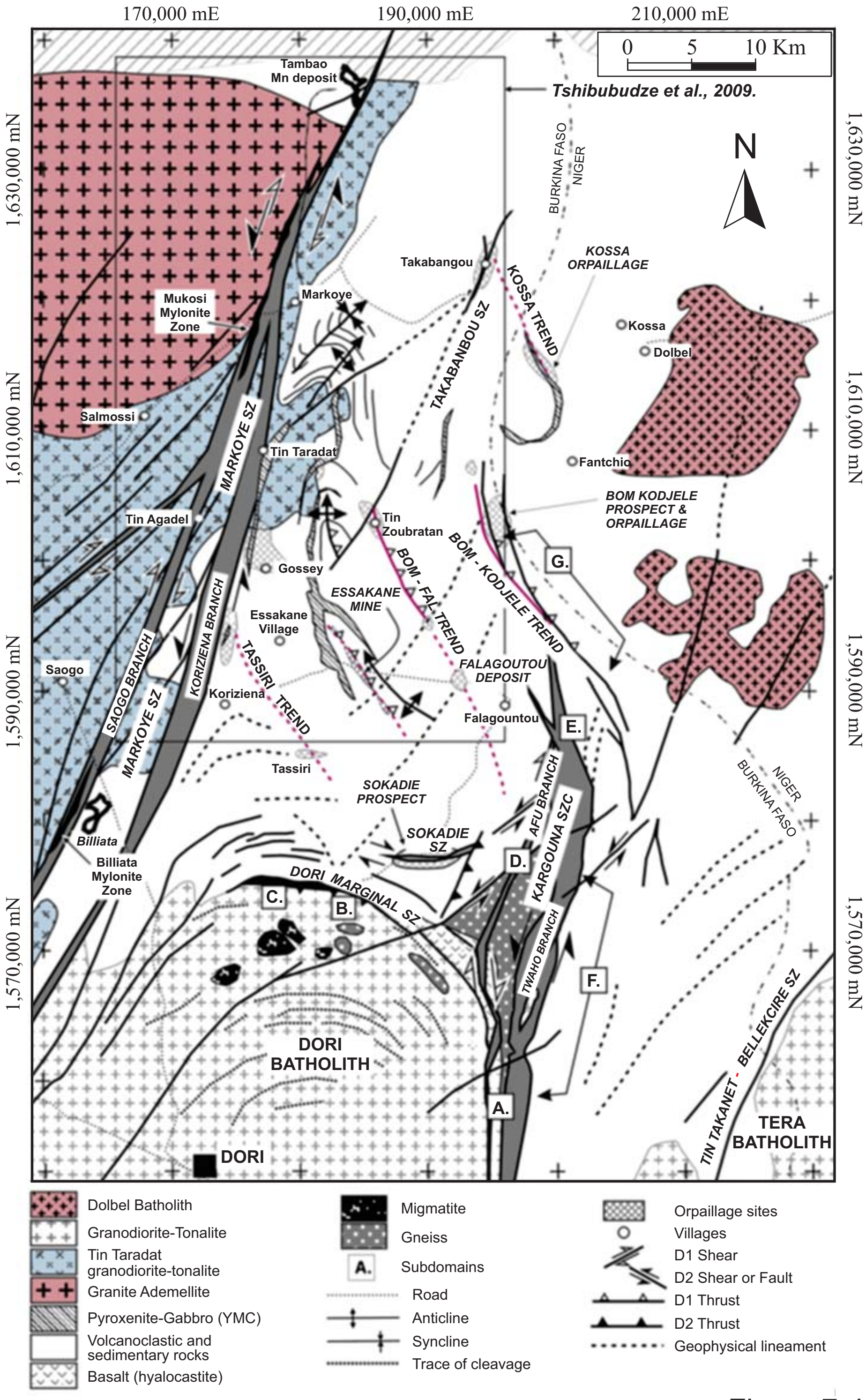

Figure 7-4

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Local Geological Map of the Oudalan-Gorouol Greenstone Belt

July 2018

Source: Tshibubudze et al., 2009.

7.3 PROPERTY GEOLOGY

The Essakane Gold Mine occurs in the Paleoproterozoic Oudalan-Gorouol greenstone belt in northeast Burkina Faso. The stratigraphy can be subdivided into a succession of lower-greenschist facies meta-sediments (argillites, arenites, and volcaniclastics), conglomerate, and subordinate felsic volcanics, and an overlying Tarkwaian-like succession comprising siliciclastic meta-sediments 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 yellow stars in Figures 7-3 and 7-5) 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 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 un-deformed andesite and metasediments. As a general rule, gold occurs 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 mine area. The Markoye Fault trends north-northeast through the western portion of the mine area and separates the Paleoproterozoic rocks from an older granite-gneiss terrane to the west.

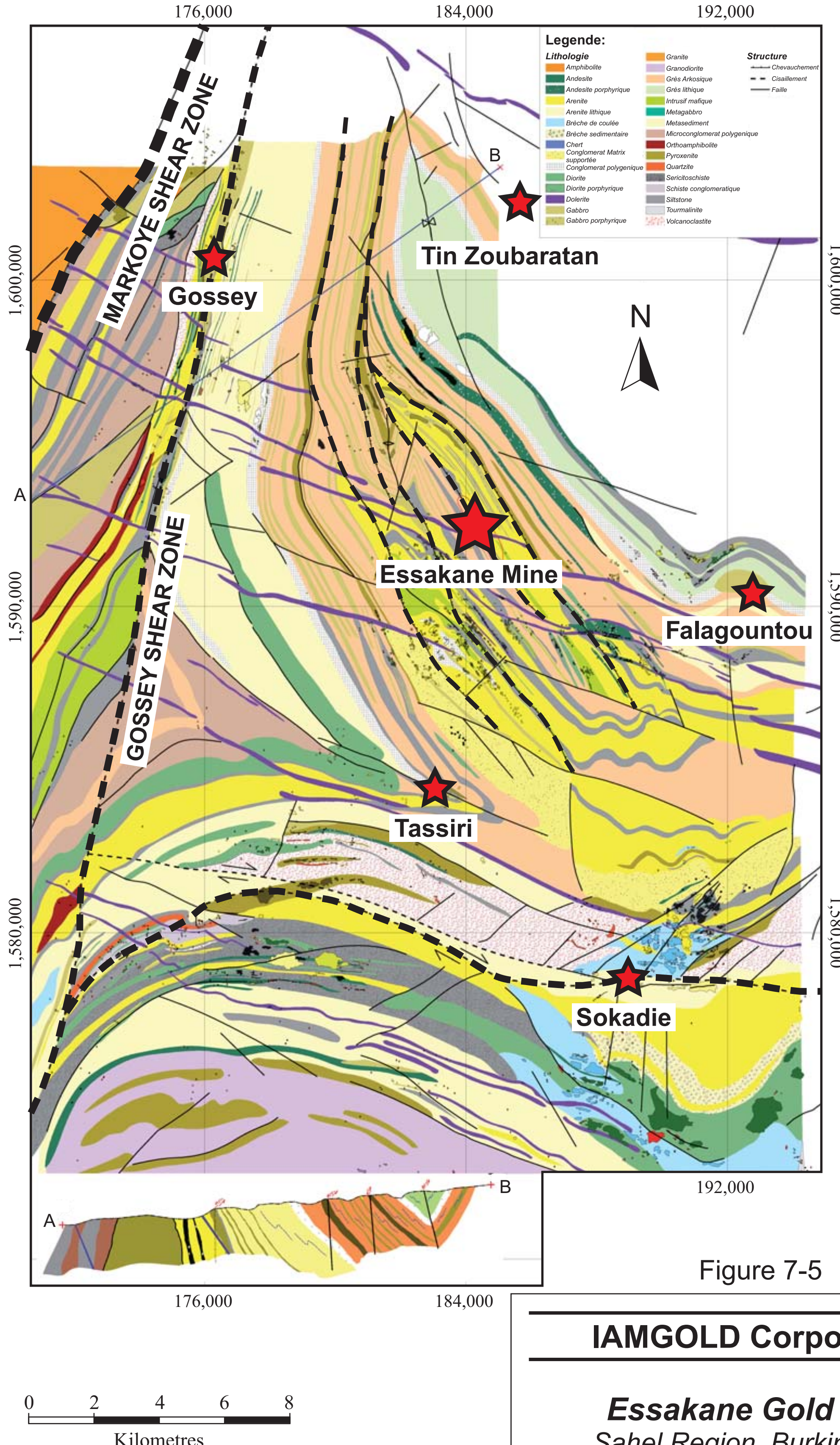

Figure 7-5

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Property Geology

July 2018

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.

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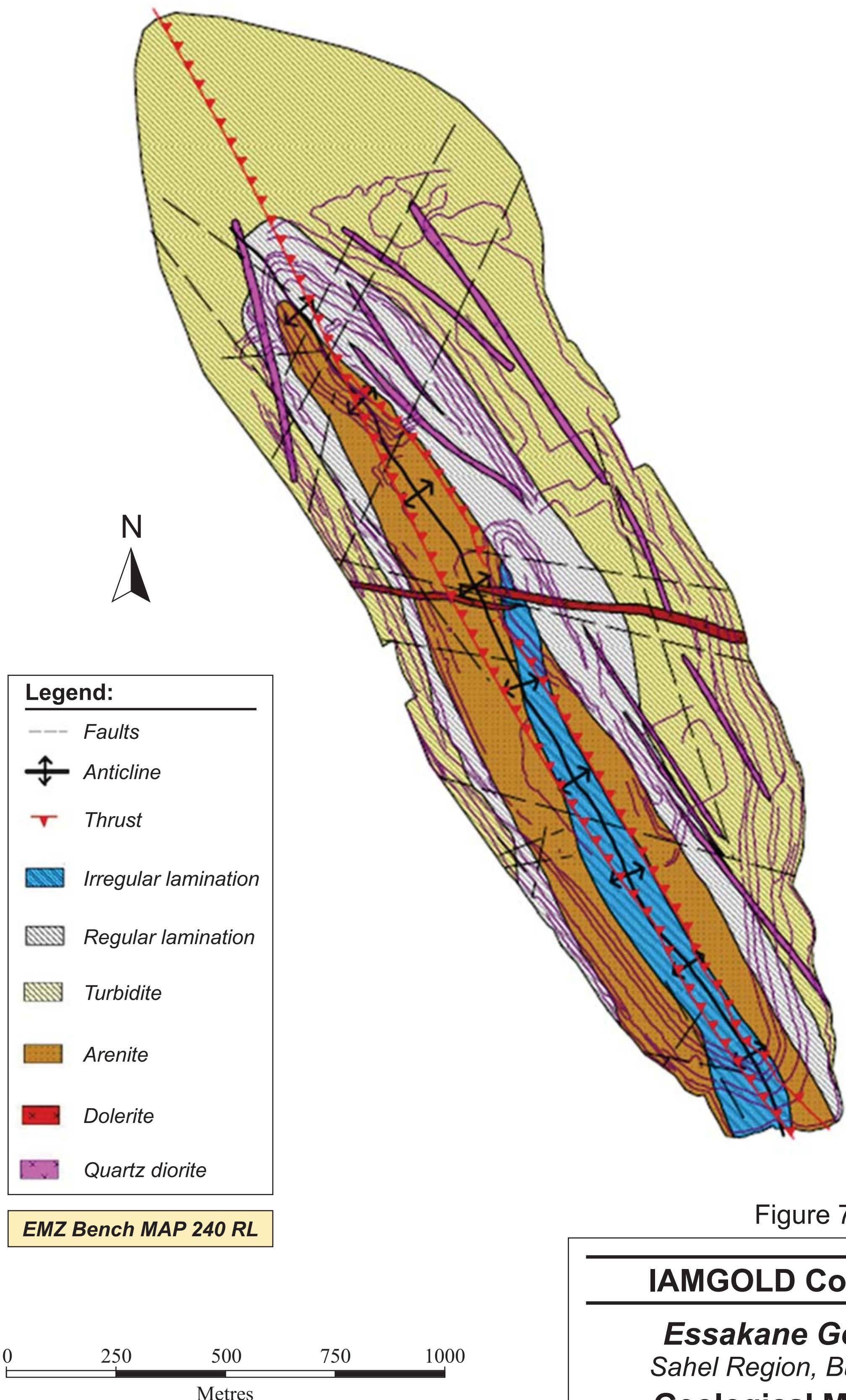


Figure 7-6

IAMGOLD Corporation

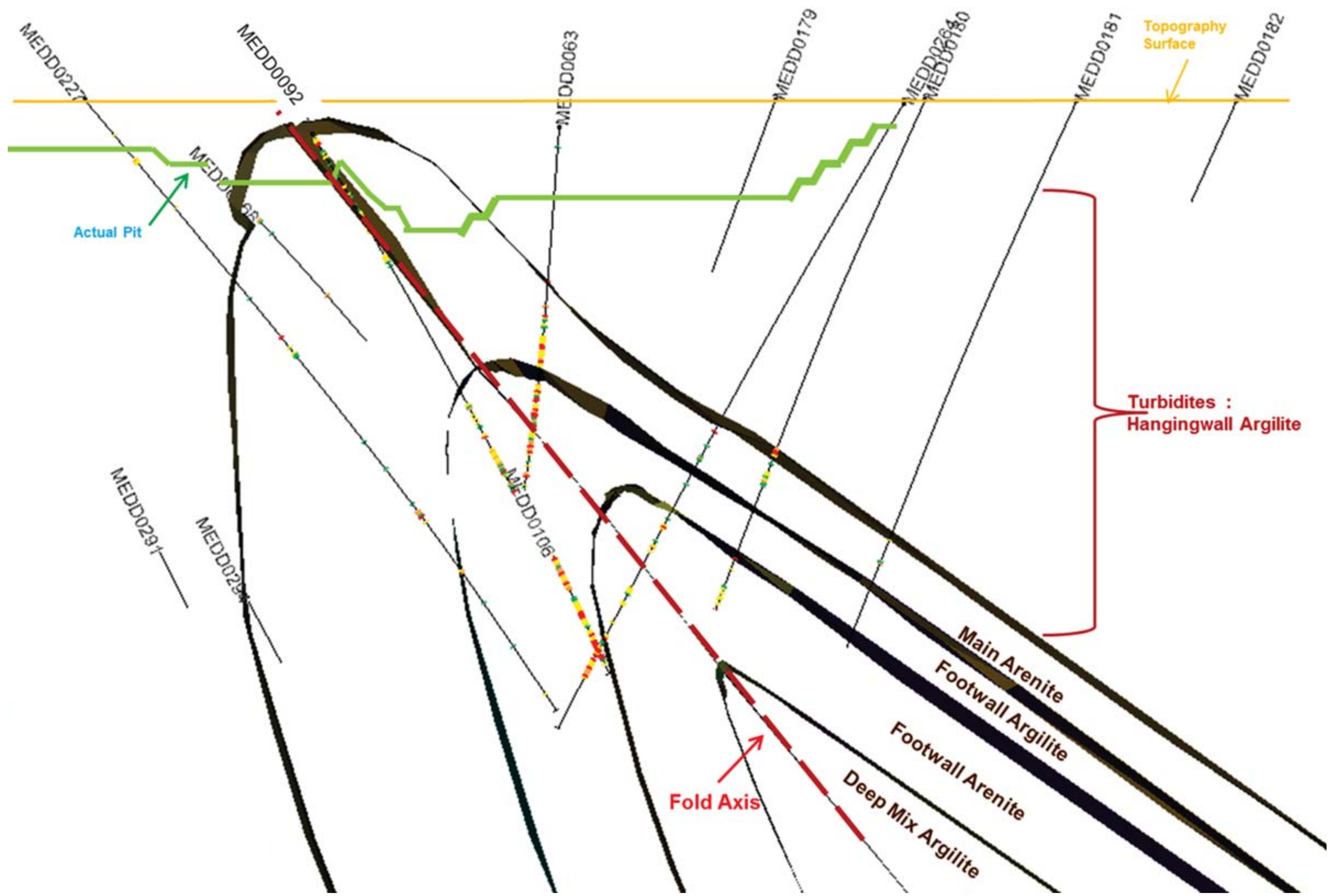
Essakane Gold Mine

Sahel Region, Burkina Faso

Geological Map of the EMZ Deposit Level 240

July 2018

Source: IAMGOLD, 2015.



Legend: Au ppm		
0.30 - 0.42		***Marginal***
0.42 - 1.50		***Low Grade***
1.50 - 3.00		***High Grade***
3.00 - 9.99		***Super High Grade***

0 50 100 150 200
Metres

Figure 7-7

IAMGOLD Corporation

Essakane Gold Mine
Sahel Region, Burkina Faso

EMZ Deposit
Cross Section 51750N

July 2018

Source: IAMGOLD, 2015.

IAMGOLD Corporation

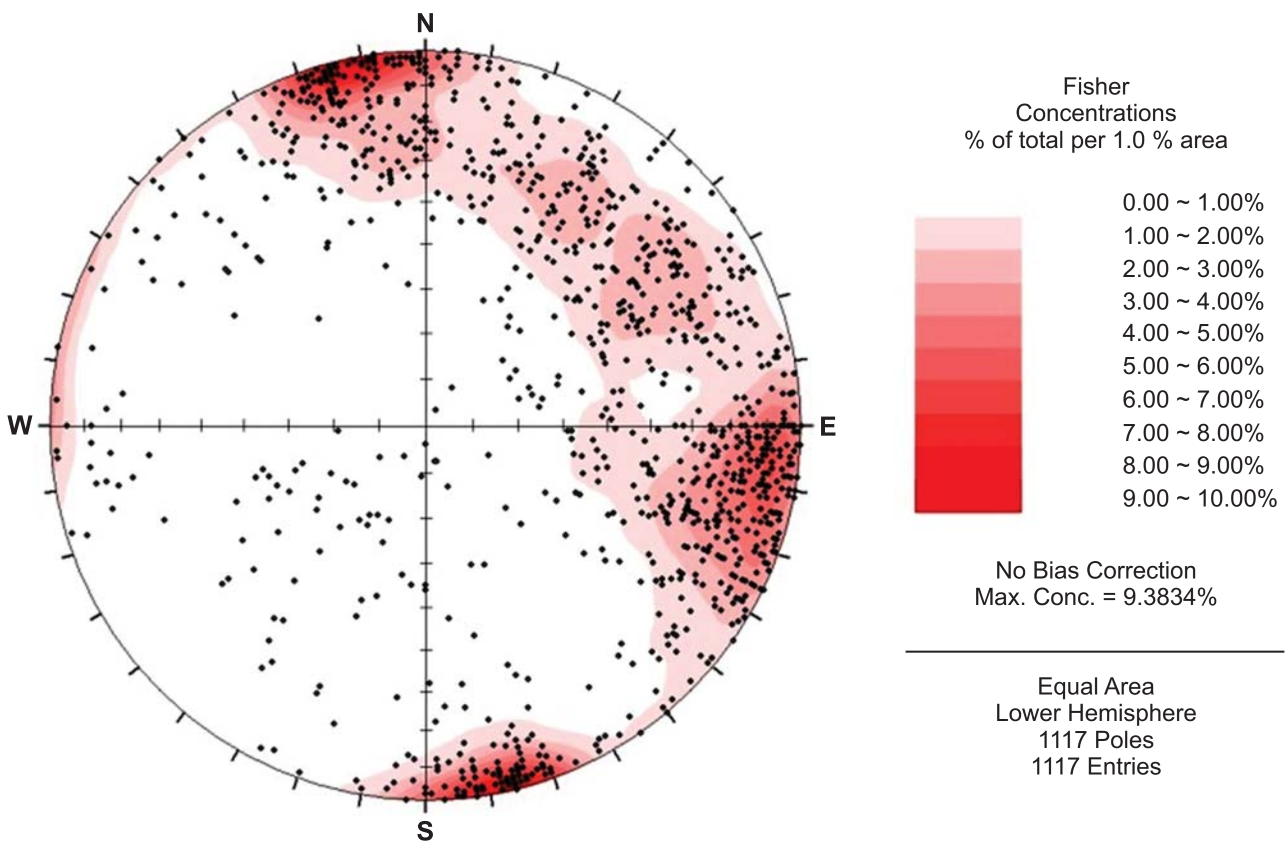


Figure 7-8

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Quartz Vein Orientations (from pit mapping)

July 2018

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 litho-structural 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 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 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 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.

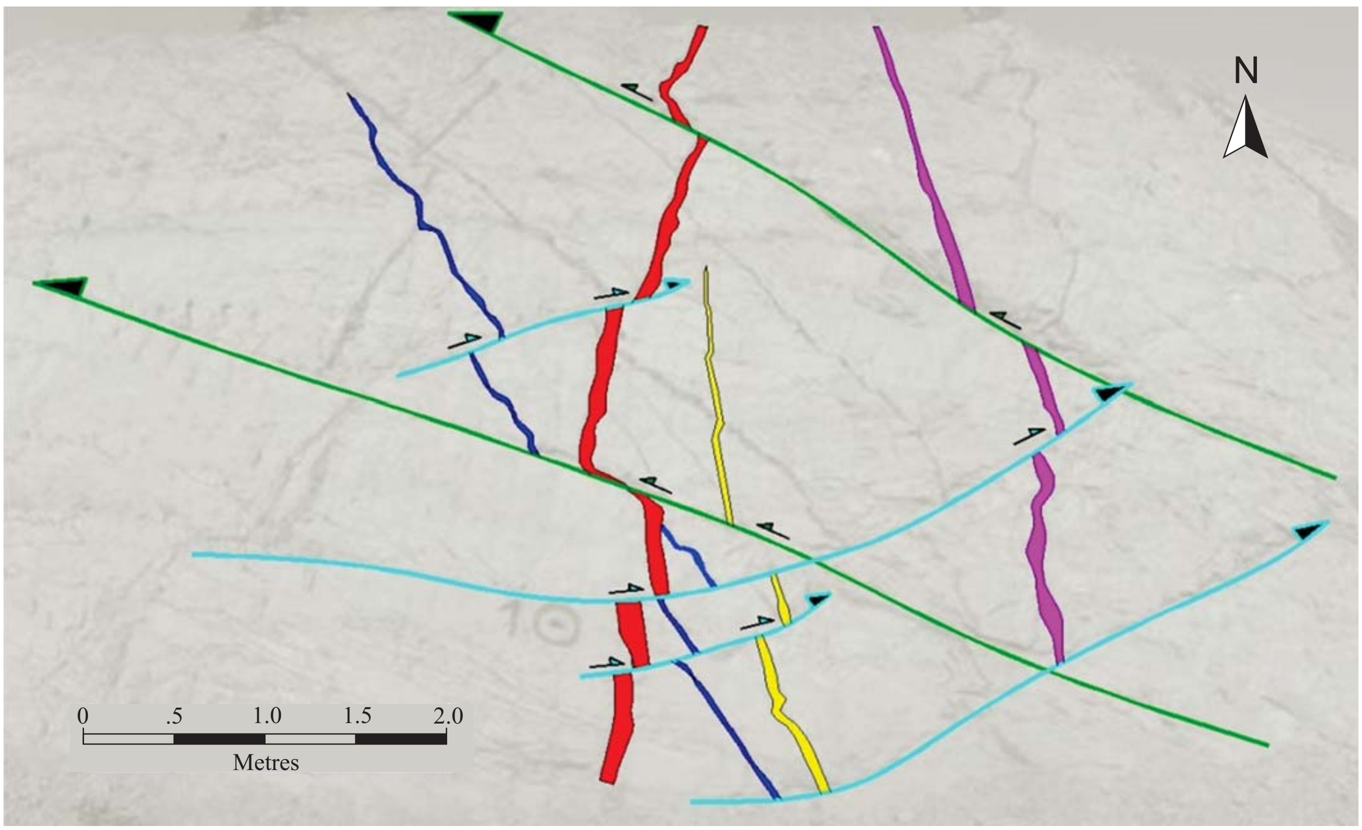


Figure 7-9

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Vein Displacements Along Minor Thrusts (West Wall EMZ Deposit)

July 2018

Source: IAMGOLD, 2015.

Arsenopyrite and pyrite occur within and adjacent to quartz veins as well as disseminated throughout areas of wallrock 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.

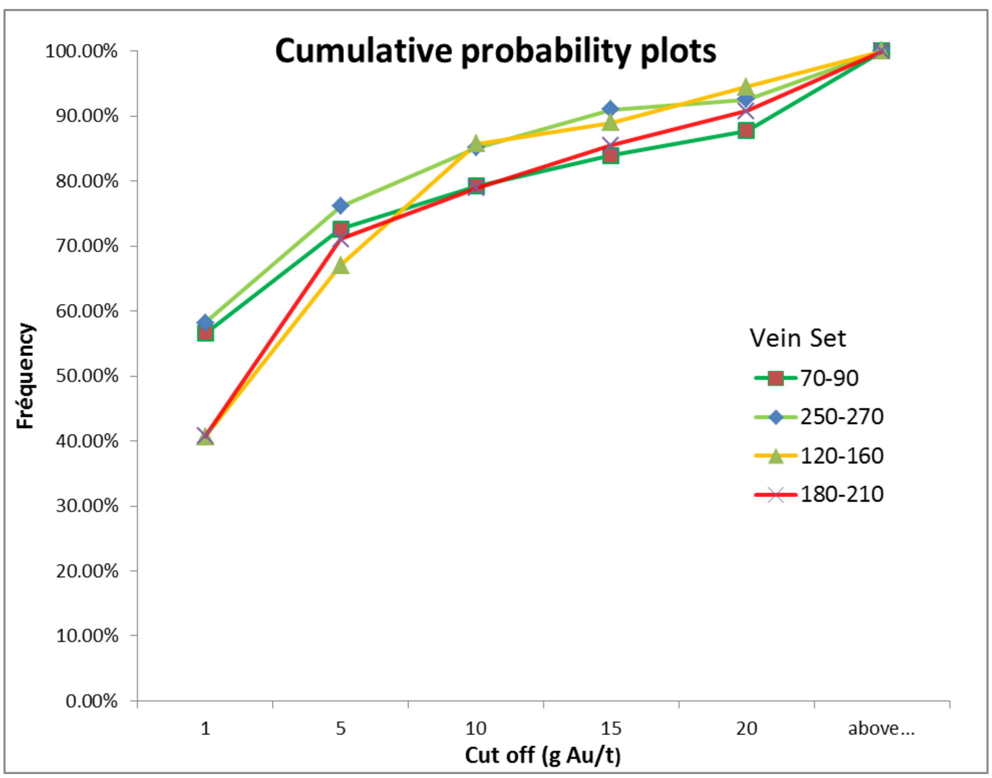
Except for vein sets located in the turbidite-hosted Birimian sills, all recorded vein sets are mineralized. The east-west and north-south vein sets both appear to show higher average gold grades than the other vein sets (Table 7-1), however, they are also more variable, with a higher proportion of the lower values also appearing in the same vein sets. Tests done 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	%	No. of samples	%	Avg Grade (g/t Au)	≤ 1 g/t	1 g/t <x ≤ 5 g/t	≤ 5 g/t	5 g/t <x ≤ 10 g/t	> 10 g/t	>20 g/t
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, as cumulative distribution functions (CDF), the proportion of samples that have returned values below a series of cut-offs for each family of veins, which shows 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° and 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 CDF OF VEIN SETS AU GRADE



7.4.2 FALAGOUNTOU DEPOSIT MINERALIZATION

Due to the intense artisanal mining ("orpaillage") activity, no detailed geological mapping has been carried out over the Falagountou 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 into the wallrock, 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 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).

7.5 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 the Essakane Gold Mine.

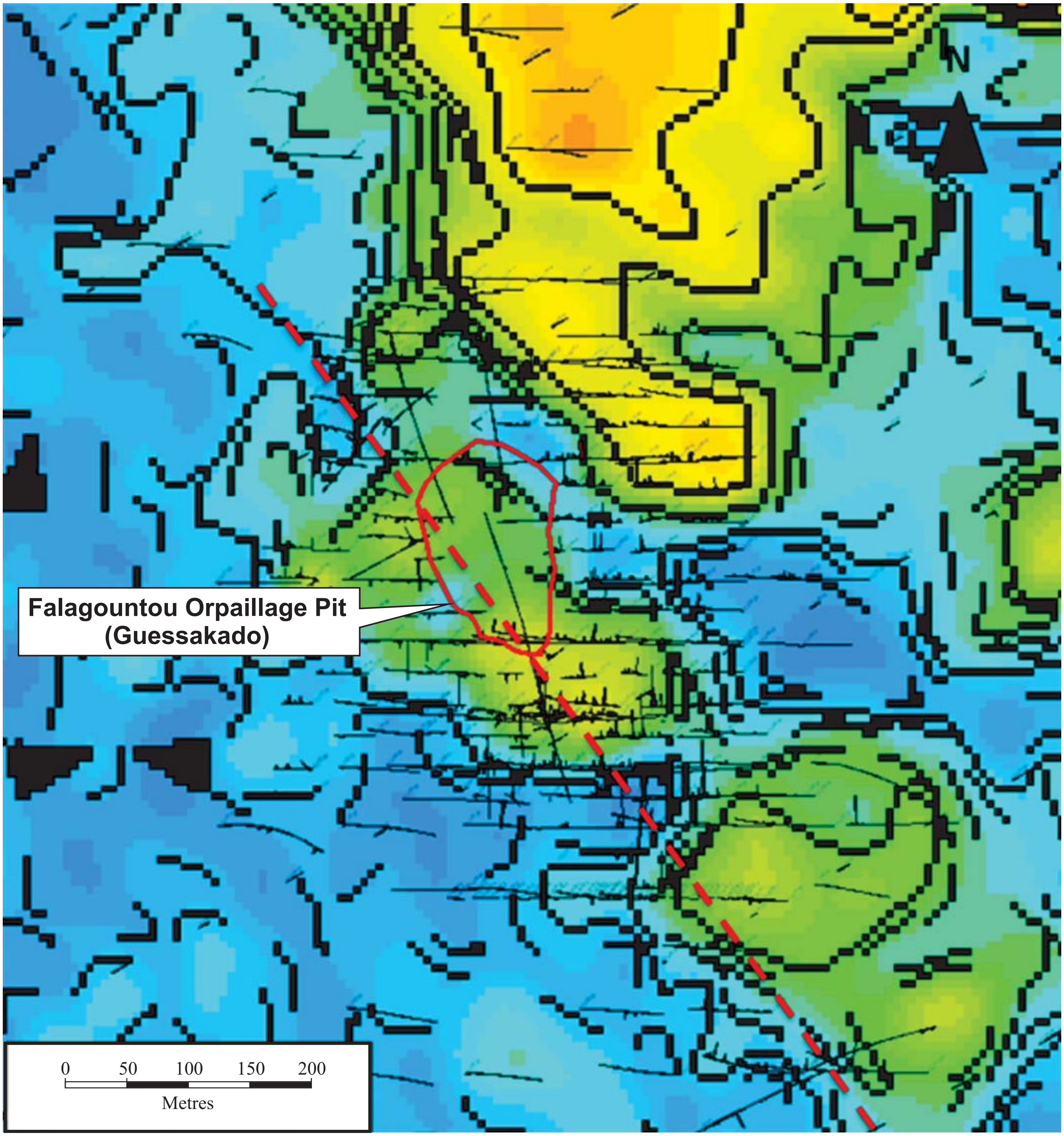


Figure 7-11

IAMGOLD Corporation

Essakane Gold Mine
Sahel Region, Burkina Faso

Total Magnetic Map of Falagountou Area

July 2018

Source: IAMGOLD, 2015.

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 dig this material without difficulty. Compared to the EMZ deposit, the saprolitic layer at the Falagountou deposit is much thinner, sometimes less than a few metres. The base of transition (or top of fresh) is gradational and the contact is placed at the Brown's value of R3 (that is, the rock can be peeled by a pocket knife 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 rocks 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 microns in diameter. Significant amounts of gold report to the +106 microns oversize despite the fine grind. Fifty per cent of the gold fraction is coarser than 106 microns 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 microns 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 (7 kg) for preparation and the use of the 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 testwork shows that the gold occurs:
    - on sulphide grain boundaries,
    - as small filamental grains concentrated along fractures within the sulphide, or as coarse flakes >100 microns 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 is the more prevalent mechanism 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 10 m 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 wallrock 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 deposit is a porphyry intrusive-hosted, orogenic style, gold deposit. 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.

Falagountou East also appears to be related to the intrusive rocks. Gold is associated with a northwest striking and northeast dipping structure affecting the sedimentary sequence that is injected locally by dioritic intrusive rocks.

9 EXPLORATION

The Essakane Gold Mine 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 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) done by GEOTECH Airborne Geophysical surveys.

The two survey areas (Tin-Taradat-Gossey-Korizena block and Gourara block) are located approximately 4 km south and 7 km west of Essakane Mine (Figure 9-1). The survey areas were flown in an East-West (N100°E azimuth) direction for the Tin-Taradat-Gossey–Korizena 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.

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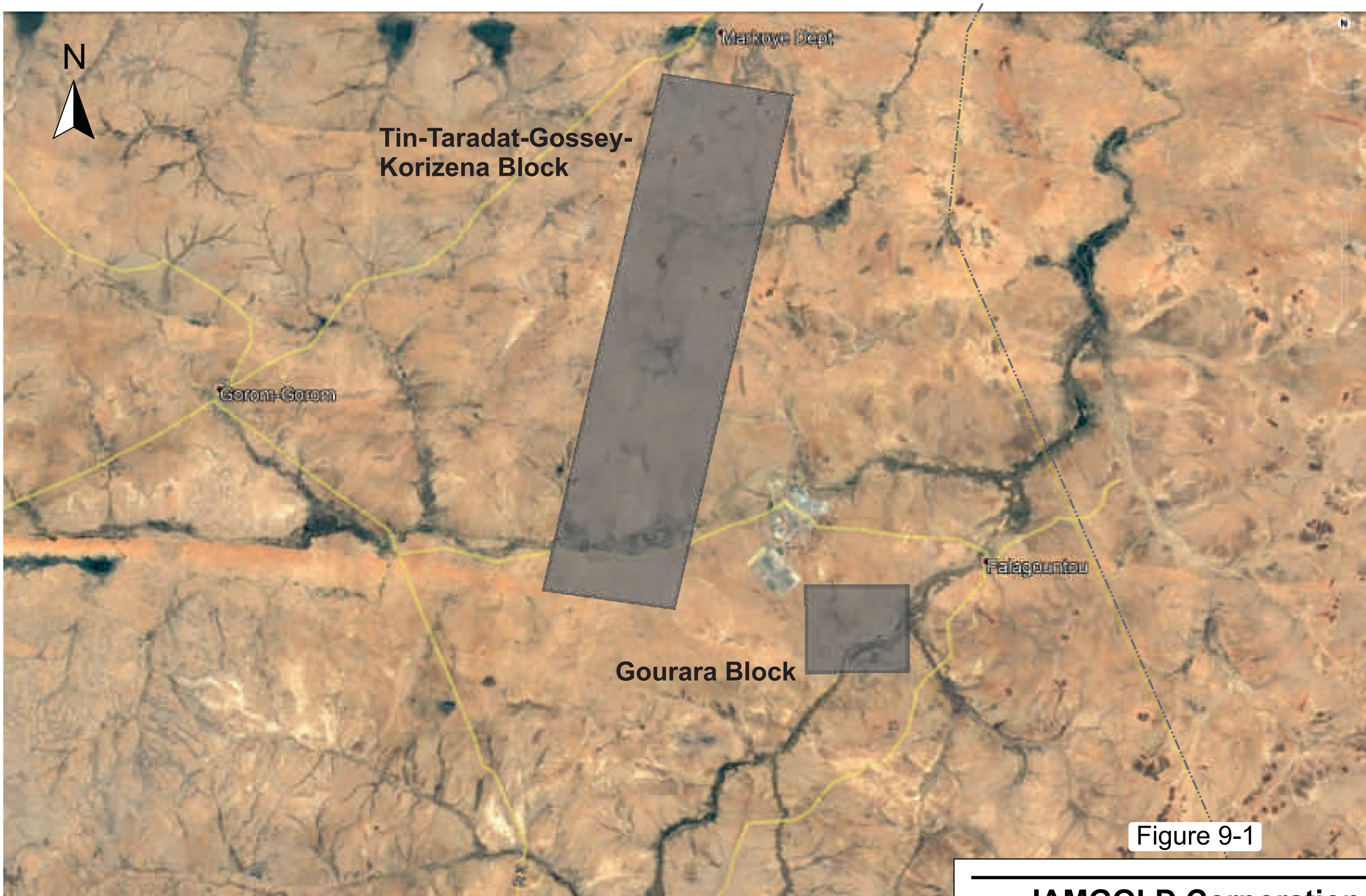


0 4 8 12 16
Kilometres

Figure 9-1

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

VTEM Survey Area
Location on Google Earth

July 2018

Source: Essakane S.A., 2018.

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-Korizena block and the Gourara block, respectively.

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 Aircore drilling was executed in 2007, after Goldfields joined Orezone Resources.

Since 2010, Essakane Exploration SARL conducts several campaigns of regional shallow and deep follow-up Air Core (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 Airmag data has been 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).

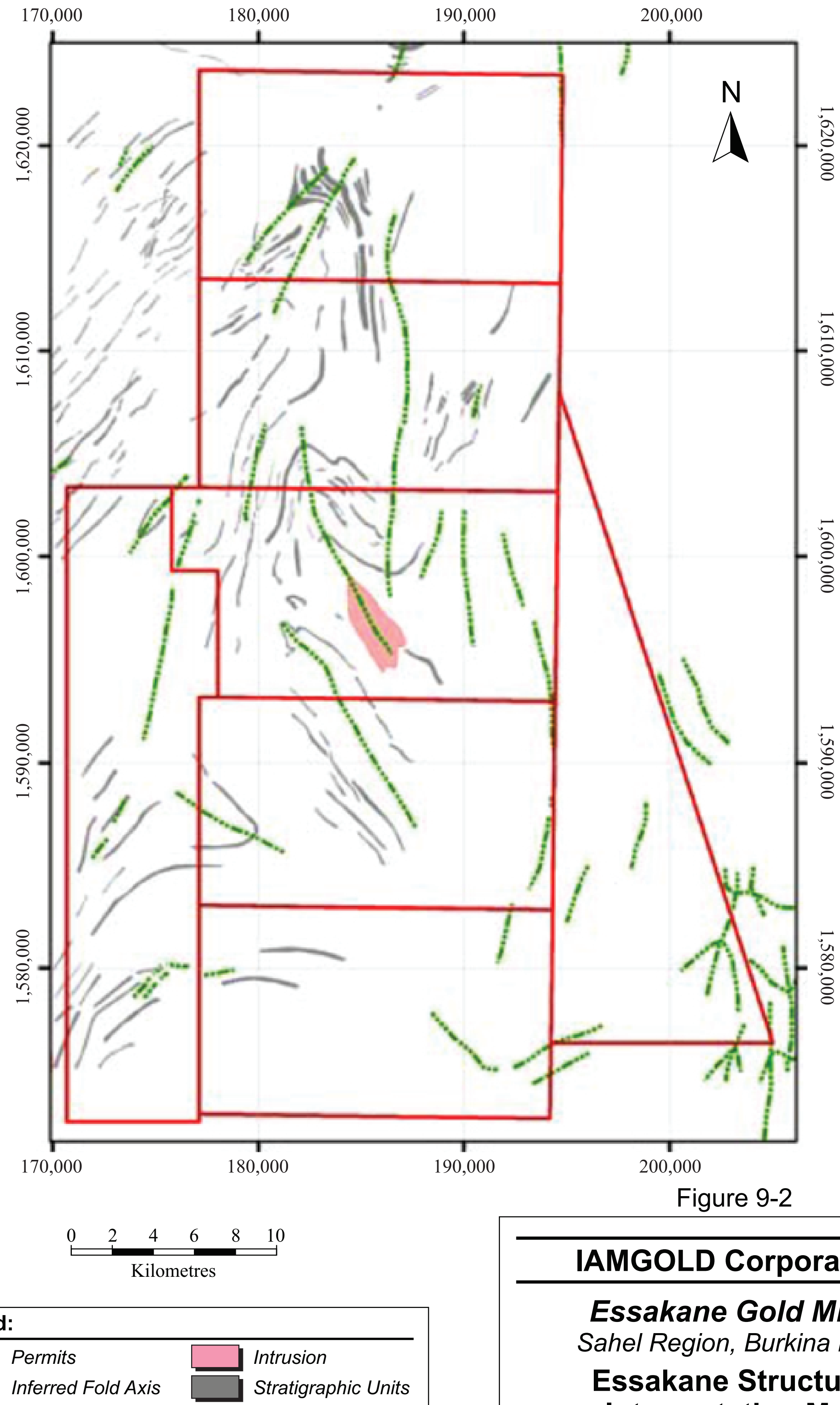


Figure 9-2

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Essakane Structural Interpretation Map

July 2018

Source: Essakane S.A., 2015.

10 DRILLING

Exploration efforts on the Essakane Gold Mine property were initially focused on identifying the potential of the entire area of the mine. In the mid-1990s, BHP undertook a widely spaced drilling program on the EMZ deposit that has been narrowed subsequently by Ranger.

Orezone Resources started 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 increasing the sample size and 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 and DD drilling has been conducted by Essakane S.A.'s Resource Development Group since January 2010. As of February 2018, a total of 2,279 RC holes 270,208 m) and 968 DD holes (267,913 m) had been drilled within the EMZ and Falagountou pits.

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 on the EMZ deposit were positive, with continuity of mineralization demonstrated at depth along the east limb of the deposit in the northern sector and 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 February 28, 2018.

An infill RC and DD program conducted at the Falagountou deposit, since the previous Falagountou Mineral Resource estimate in 2016, confirmed lateral continuity of mineralization oriented mostly north-south as well as an extension down-dip, which remained open. Drilling also identified a second mineralized structure, located 250 m west of the main zone.

The drilling 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 DD and RC drill holes as of February 28, 2018 are summarized in Table 10-1.

TABLE 10-1 ESSAKANE DRILLING PROGRAMS 1995 TO FEBRUARY 2018

		DD		RC		RCD		Total	
Year	Company	Metres	No. of Holes	Metres	No. of Holes	Metres	No. of Holes	Metres	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

		DD		RC		RCD		Total	
Year	**Company**	**Metres**	**No. of Holes**	**Metres**	**No. of Holes**	**Metres**	**No. of Holes**	**Metres**	**No. of Holes**
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	2,890	15	2,362	18	800	2	6,052	35
Total		**305,092**	**1,218**	**337,413**	**3,174**	**65,950**	**355**	**693,910**	**4,675**

Note:

1. Hole type RCD means the holes were pre-collared with RC then completed by DD.

10.1 DIAMOND DRILLING

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

HQ-size core (63.5 mm) is drilled ten metres past the saprolite horizon and then reduced to NQ core (diameter 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 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 mine-site 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 1.0 m 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 mine-site 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.

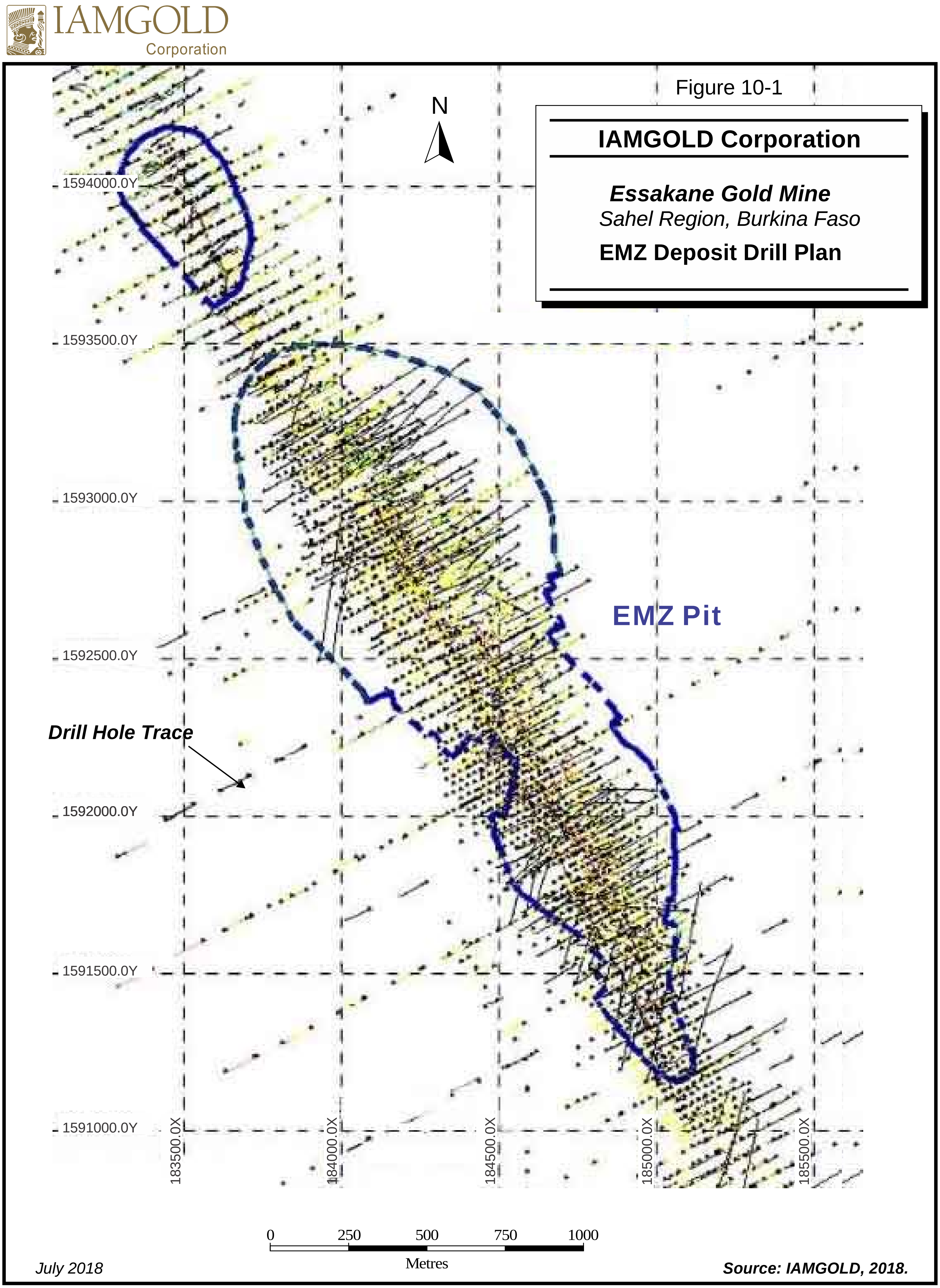
10.2 REVERSE CIRCULATION DRILLING

For RC drilling, a track-mounted Cat-Max rig is used with an attached cyclone unit that collects all the coarse (>50 microns) material over five metre runs starting at the collar. A 7 kg 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 differential GPS.

Figure 10-1 shows the drill hole plan as of February 28, 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 February 28, 2017 for the Falagountou deposit. Figure 10-4 shows a typical cross section of the drilling on the Falagountou West deposit.


IAMGOLD
Corporation
Figure 10-1
IAMGOLD Corporation
Essakane Gold Mine
Sahel Region, Burkina Faso
EMZ Deposit Drill Plan
N
EMZ Pit
Drill Hole Trace
1594000.0Y
1593500.0Y
1593000.0Y
1592500.0Y
1592000.0Y
1591500.0Y
1591000.0Y
183500.0X
184000.0X
184500.0X
185000.0X
185500.0X
0
250
500
750
1000
Metres
July 2018
Source: IAMGOLD, 2018.

IAMGOLD Corporation

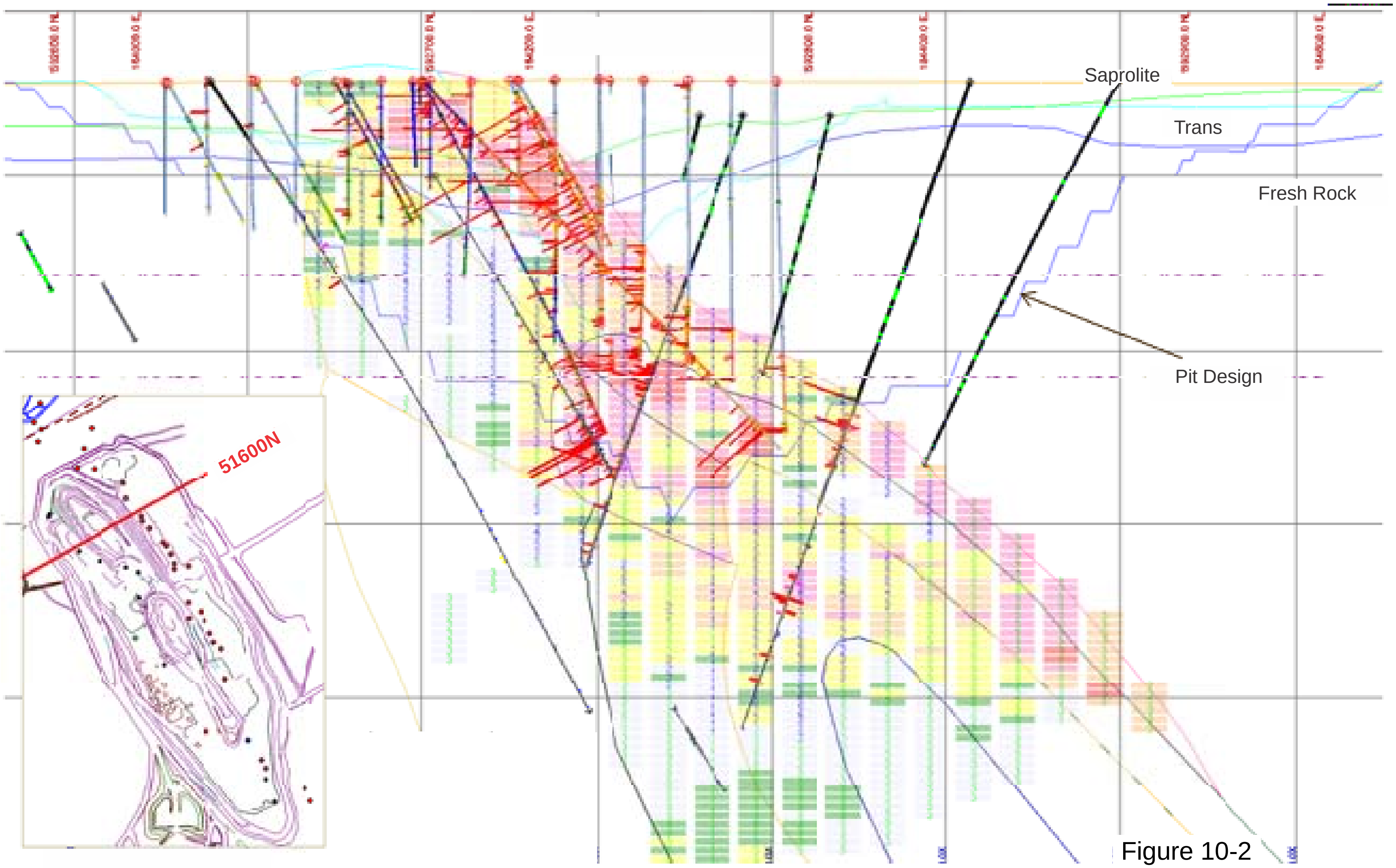


Figure 10-2

Section 51600N Looking Northwest

Au ppm

- < 0.29
- 0.29 - 0.35
- 0.35 - 0.75
- 0.75 - 1.00
- 1.00 - 1.50
- > 1.50

0 20 40 60 80 100
Metres

IAMGOLD Corporation

Essakane Gold Mine
Sahel Region, Burkina Faso

EMZ Deposit
Typical Cross Section 51600N

July 2018

Source: Essakane S.A., 2015.

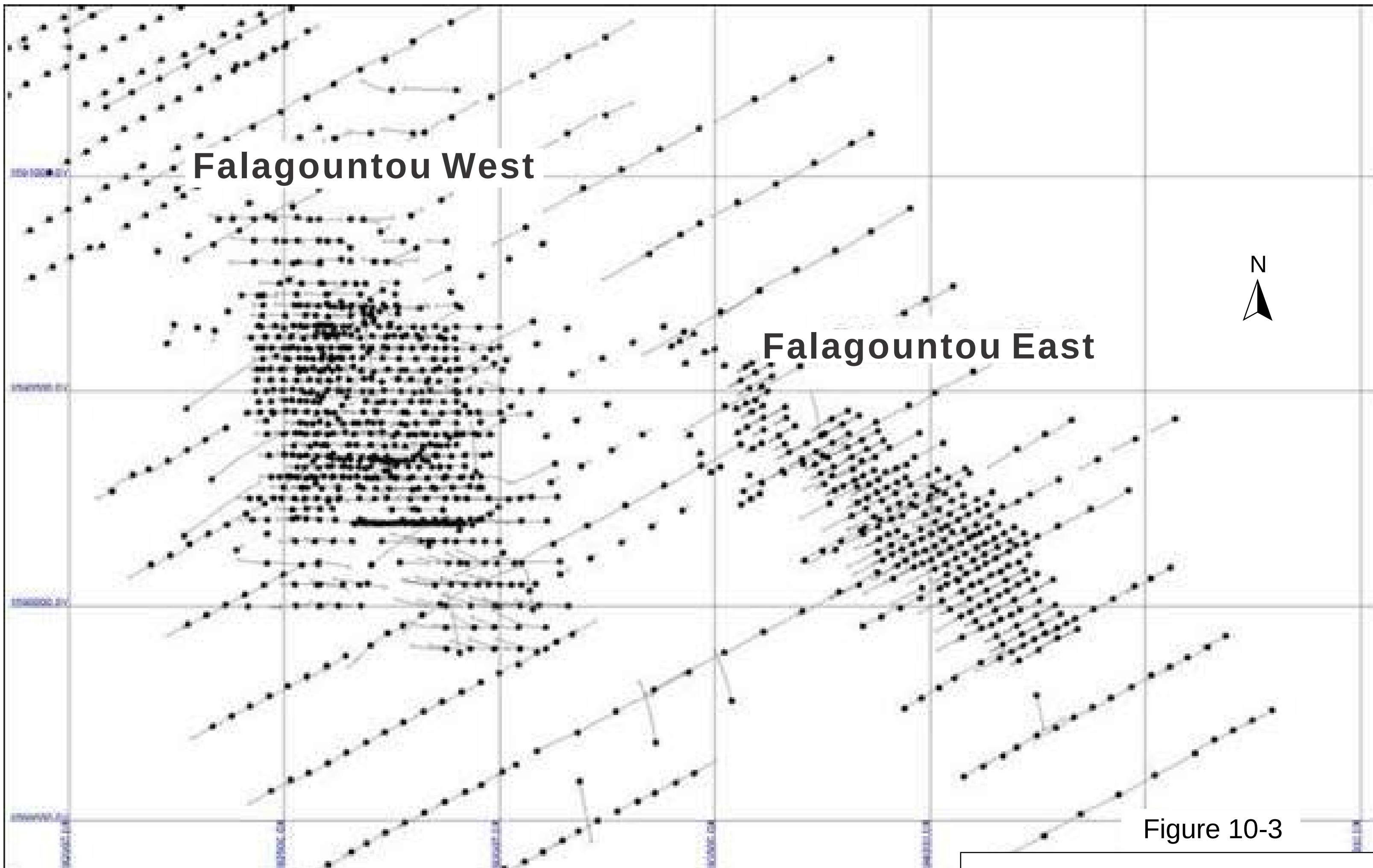


Figure 10-3

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Falagountou Deposit Drill Plan

Legend:

- *Drill Hole Collars*

0 100 200 300 400 500

Metres

July 2018

Source: Essakane S.A., 2018.

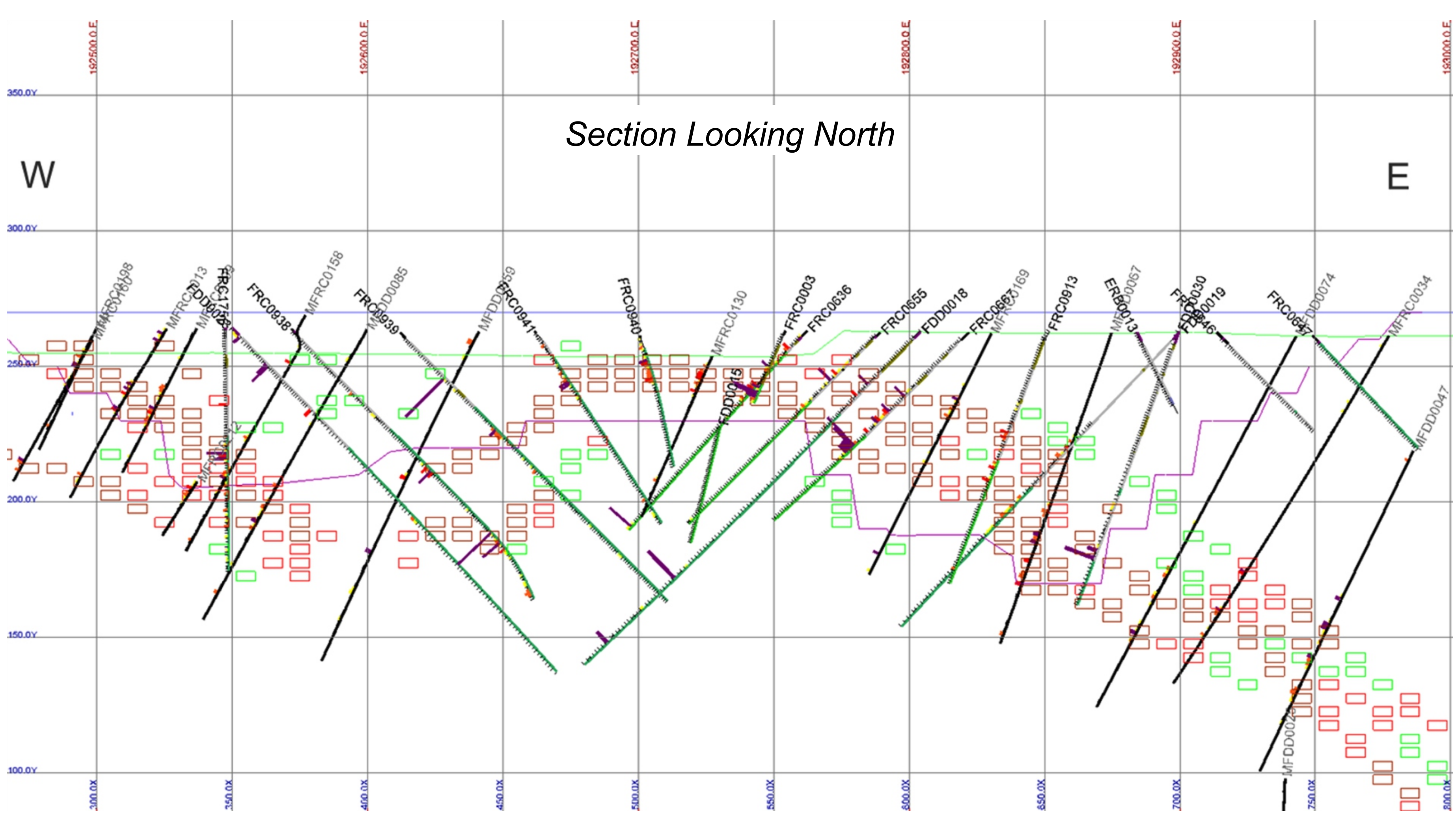
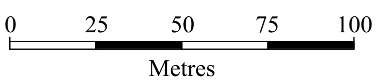


Figure 10-4



IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Falagountou West Deposit
Typical Cross Section

July 2018

Source: Essakane S.A., 2015.

IAMGOLD Corporation

10.3 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 along with 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 GEMS modelling database only after it has been duly validated in DataShed and all the assays have been received and checked. Recorded recovery averages 95%.

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

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 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 the 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 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 mine site 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.

Most of the 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 2 mm 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 95% passing 105 microns with LM-5 or with 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 had their solid residues re-assayed using fire assay. This percentage has been raised to 20% in 2014. 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 mine laboratory assay results alone resulted in a selection bias (i.e., over a long term, check samples, on average, returned lower values than the mine laboratory's results). The sampling protocols for DD samples are shown in Table 11-1.

TABLE 11-1 DD SAMPLE PREPARATION AND ASSAYING PROTOCOL

Step	Description
1. Reception	• Dry (6h @ 105ºC). • Weigh and note.
2. Crushing and pulverization	• Crush entire sample in jaw crusher down to 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 1,000g	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 all solid residues that returned > 0.3 ppm in original assay.

Since 2010, RC drilling has been carried out using 140 mm (5.5 in.) diameter holes with 5 m sample intervals down to a depth of 150 m or until the water table is intersected. The 7 kg field split is dried and pulverized to 95% passing 500 microns 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 95% passing 500 microns 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 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 send 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 microns to P_{95} of 500 microns 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 RC PREPARATION AND ASSAYING PROTOCOL

Step	Description
1. Reception	• Dry (6h @ 105ºC). • Weight and note.
2. Crushing (occasionally) and Pulverization	• Pulverize entire 7 kg sample in Keegor mills to 95% passing 500 microns.
3. Sieve analysis	• Test particle size at a frequency of 5% when prompted by the LIMS.

Step	Description
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 all solid residues that returned > 0.3 ppm in original assay.

11.2 SAMPLE SECURITY

Following IAMGOLD acquisition of Orezone Resources and Essakane Gold Mine 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 mine site 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 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 QP's opinion, the sample preparation, analysis, and security procedures at the Essakane Gold Mine are adequate for use in the estimation of Mineral Resources.

11.3 QUALITY ASSURANCE AND QUALITY CONTROL

Essakane S.A. is using a QA/QC system which involves insertion of Certified Reference Materials (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. The underlined standards are those currently in use.

TABLE 11-3 LIST OF CERTIFIED REFERENCE MATERIALS

OXIDE MATERIAL TYPE		SULPHIDE MATERIAL TYPE	
CRM	**Estimated Value (Au g/t)**	**CRM**	**Estimated Value (Au g/t)**
OXA71	0.084	SE44	0.606
<u>OXC109</u>	0.201	SE58	0.608
OXC72	0.205	SG56	1.027
OXC129	0.205	<u>SH82</u>	1.333
OXD108	0.414	SH41	1.344
OXD73	0.416	SH69	1.346
OXD87	0.417	SH65	1.348
<u>OXD144</u>	0.417	SH55	1.375
OXD107	0.452	SI54	1.78
OXE106	0.606	<u>SI64</u>	1.78
OXF105	0.8	SJ63	2.632
OXF85	0.805	SJ53	2.637
OXF65	0.805	<u>SJ80</u>	2.656
<u>OXF125</u>	0.806	<u>HISILK2</u>	3.474
OXH97	1.278	SK62	4.075
OXi96	1.802	SK78	4.134
OXi67	1.817	<u>SL77</u>	5.181
<u>OXJ95</u>	2.337	SL51	5.909
OXJ68	2.342	<u>SL61</u>	5.931
OXK79	3.532	SN60	8.595
OXK119	3.604	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 (Au g/t)	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 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

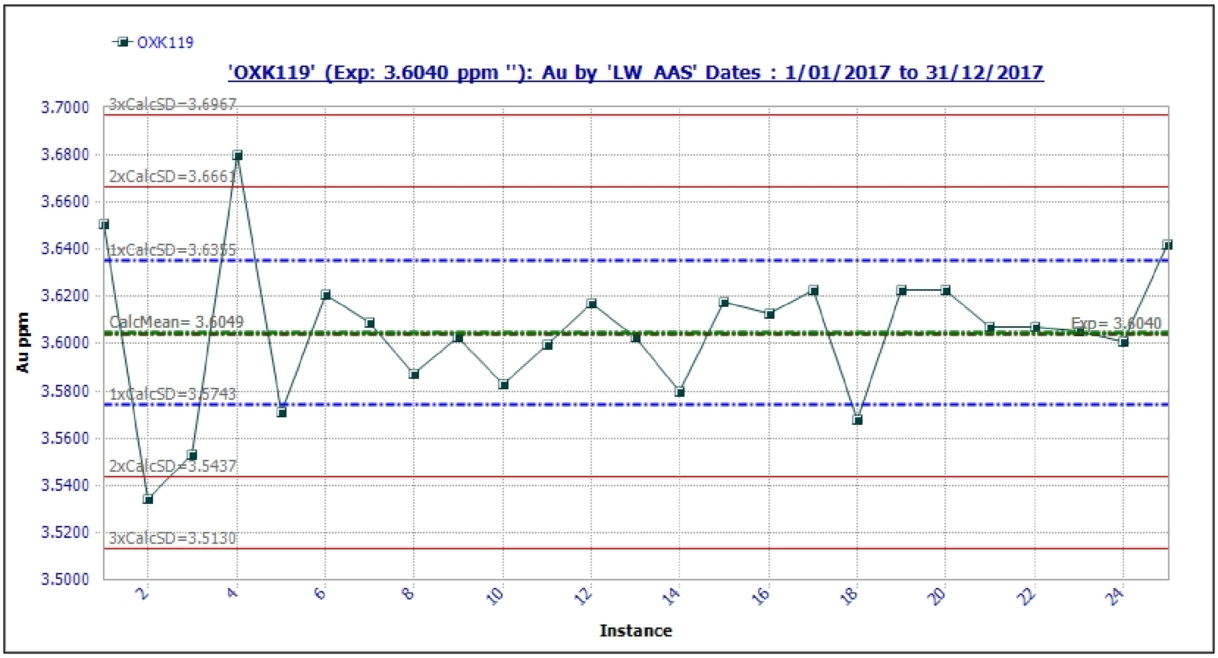


FIGURE 11-2 BLANK GRT01 PLOT

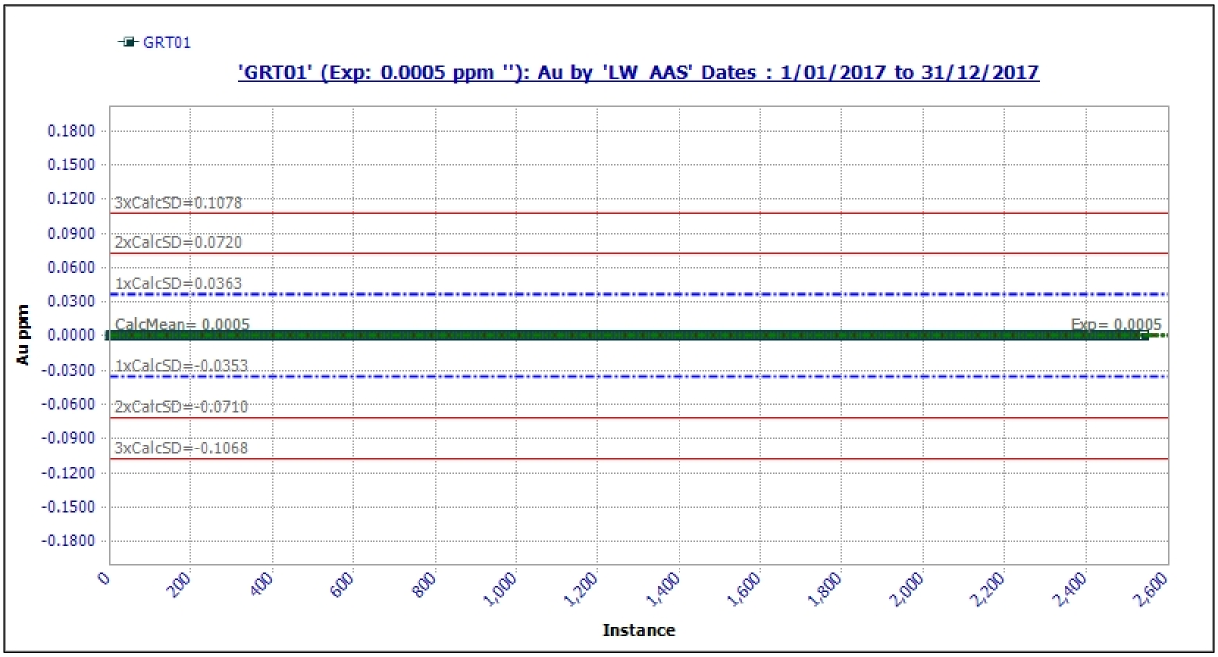


FIGURE 11-3 FIELD DUPLICATE VS. ORIGINAL SCATTERPLOT

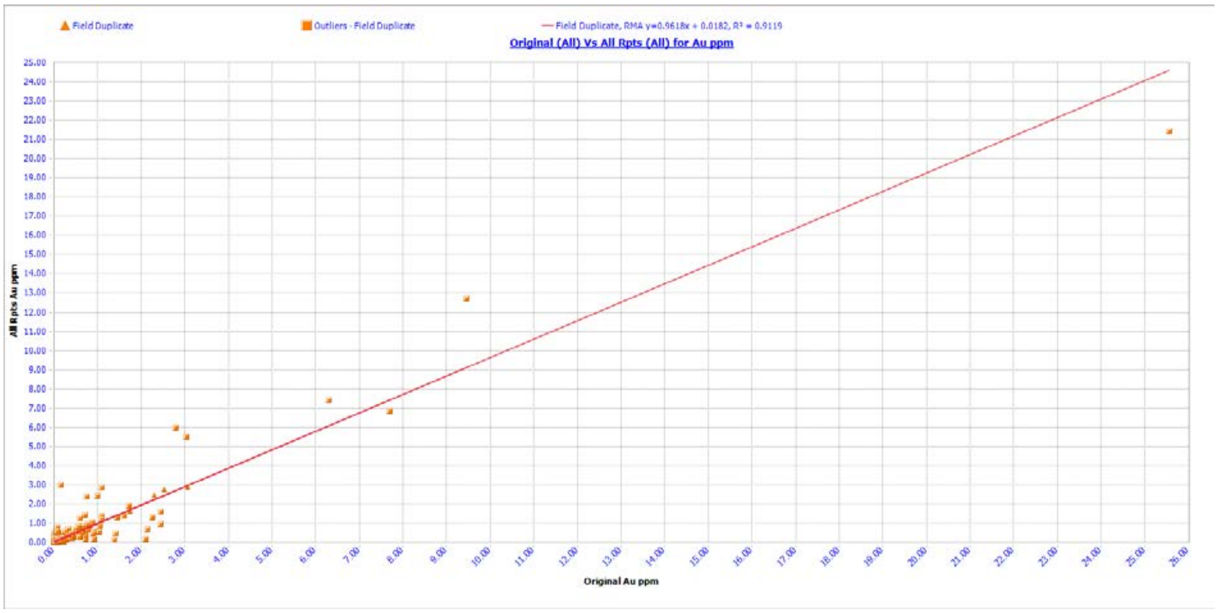


FIGURE 11-4 LOG-LOG DUPLICATE PLOT

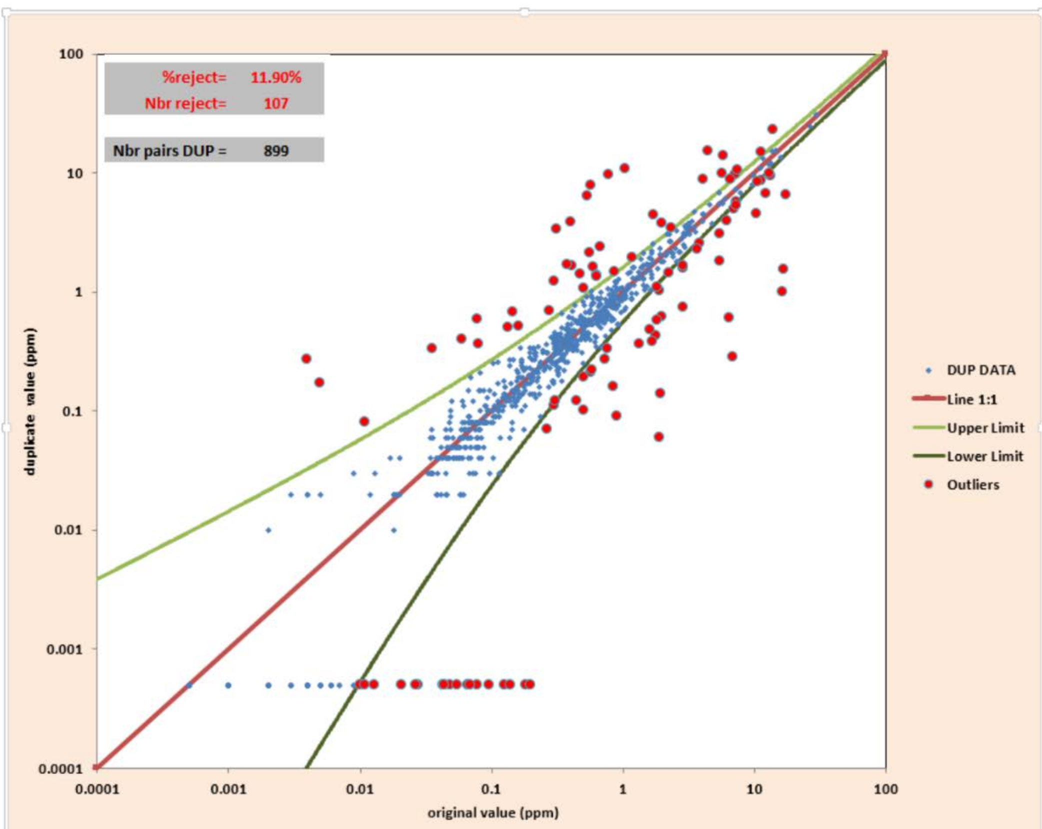
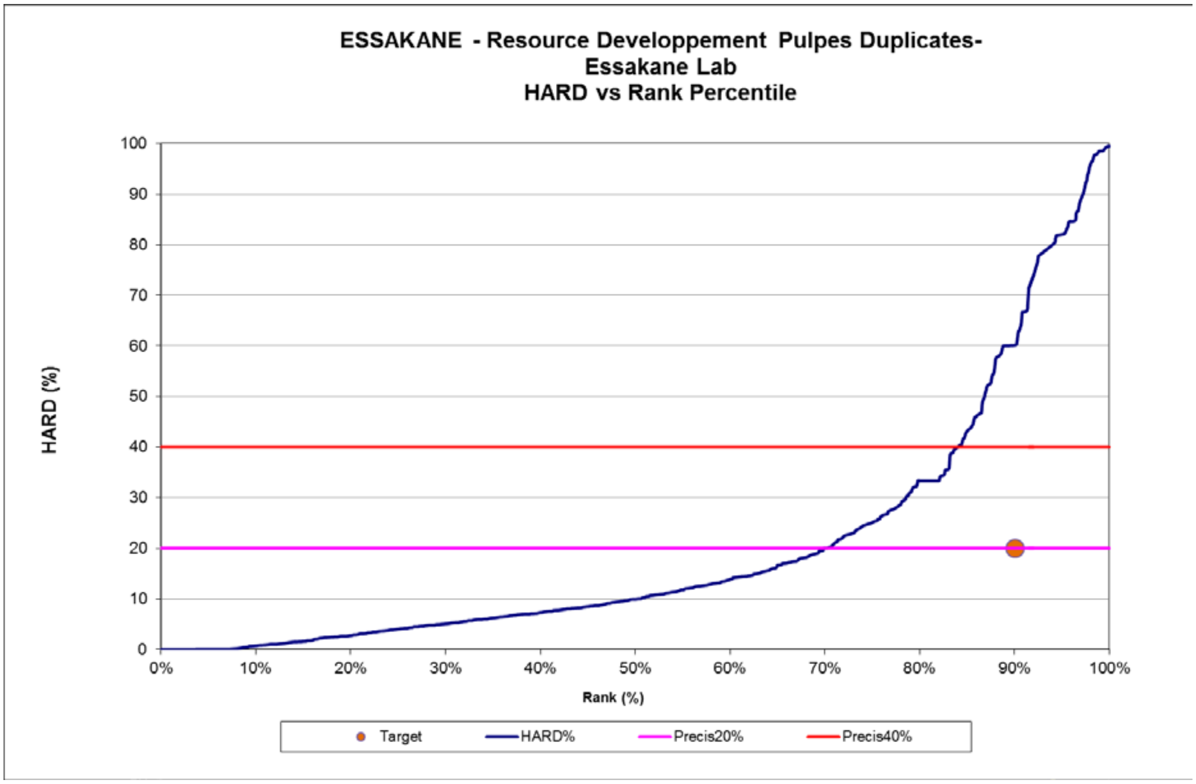


FIGURE 11-5 HARD PLOT VS. RANK PERCENTILE



12 DATA VERIFICATION

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 only 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 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.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

Testwork programs have been carried out on Essakane's major ore types by numerous international metallurgical laboratories since 1990. Testwork 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 UFS document by GMSI (2009). Since then, testwork 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.

13.1 METALLURGICAL TESTWORK FROM 1990 TO 2007

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 carbon-in-leach (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 testwork and used for design purposes are summarized in Table 13-1.

TABLE 13-1 COMMINUTION PARAMETER SUMMARY

Samples #	Ai	Cwi	Rwi	Bwi	A*b	UCS	Lab	Rock Type	Sample Designation
1	0.046	19.7	16.9	14.6		203	SGS Jo	Fresh Argellite	EMZ8 Bbwi@106um
2				14.2			SGS Jo	Fresh Argellite	EMZ8 Bbwi@150um
3	0.322	23.4	16.3	15.6		205	SGS Jo	Fresh Arenite	EMZ9 BBwi@106um
4				13.4			SGS Jo	Fresh Arenite	EMZ9 BBwi@150um
5				8.8			SGS Jo	Oxide Saprolite	EMZ10 Levin @150um
6	0.197	3.8		7.8		47	Philips	Saprock	3096 EDD 0140B (78-87m)
7			8.8	9.1	62.3		SGS	Saprock	3096 EDD 0140B (78-87m)
8				9.1			Philips	Saprolite/Arenite	3096 EDD 0154 (39-47m)
9	0.036			8.1			Philips	Saprolite/Arenite	3096 EDD 0154 (65-72m)
10	0.260	6.7		12.0		62	Philips	Fresh Arenite	3096 EDD 0154 (129-137m)
11			13.7	12.6	34.7		SGS	Fresh Arenite	3096 EDD 0154 (129-137m)
12	0.109	7.1		13.8		82	Philips	Fresh Argellite	3096 EDD 0154 (201-208m)
13			17.5	14.6	29.3		SGS	Fresh Argellite	3096 EDD 0154 (201-208m)
14	0.202	4.7		10.7		23	Philips	Fresh Arenite	3096 EDD 1671D (110-118m)
15			11.9	10.6	48.1		SGS	Fresh Arenite	3096 EDD 1671D (110-118m)
16	0.158	6.3		13.2		90	Philips	Fresh Argellite	3096 EDD 1671D (145-153m)
17			16.0	13.5	31.3		SGS	Fresh Argellite	3096 EDD 1671D (145-153m)
Ore Types	**Ai**	**Cwi**	**Rwi**	**Bwi**	**A*b**	**UCS**		**From Samples**	
1 oxide	0.036	10.0	8.8	8.7	80.0	47		5, 8-9	Oxide design (above 78m)
2 transition	0.200	15.0	10.4	9.6	55.2	47		6-7, 14-15	Transition (78-118m)
3 fresh avg	0.183	21.6	15.4	13.2	35.9	204		1-4, 10-17	Fresh Average Design (110-210m)
4 fresh hard	0.179	21.6	16.1	13.8	31.8	204		1-4, 10-13, 16-17	Fresh Hard Design (129-210m)
	estimate								
	data not used								

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 testwork 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 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 hard rock was P_{80} (80% passing) minus 125 microns. The presence of activated carbon during leaching showed improved leaching kinetics and ultimately recoveries. This observation led to the use of a CIL circuit as opposed to a Leach-CIP circuit.

13.2 RECENT METALLURGICAL TESTWORK

As part of the plant expansion, additional metallurgical testwork and ore characterization was carried out at SGS Lakefield Research Ltd (SGS) during 2011. Comminution testwork 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 testwork 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 in June 2015 to further characterize the Essakane deposit, with an emphasis on hard 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 mill 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.

- 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 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, increase recovery, and increase leaching kinetic.

- 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. The metallurgical tests included assaying, mineralogy, gravity separation, and CIL testwork. 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 hard rock samples.

- The hard 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 Mill.

A new carbon and sulphur analyzer was installed in the assay laboratory and is used to analyze mill tails samples. Onsite testing of mill samples for graphitic carbon (Cg) is regularly completed in the assay laboratory. Good correlations are observed between graphitic content and plant residues.

13.3.1 FUTURE WORK

The geometallurgy future program work includes:

- Improving the block model with available information and correlations based on the previous work;
- Integrating new parameters in mine planning. Monthly reconciliation will be used to improve the model;
- Further development work is being planned for 2018. This includes additional drilling and laboratory testing.

13.4 CIL 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 testwork, 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

In order to assess heap leach again but this time for low and marginal grade ore, laboratory testing was conducted in two separate phases by KCA in 2016 and 2017, respectively. The first phase 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.

Based on the results from the first round of metallurgical testing at KCA, a second program was developed to provide sufficient testing that would be representative of the argillite and arenite rock types expected to be sent to the heap from the EMZ. The second phase included head analysis, bottle roll leach tests, comminution testing, high pressure grinding roll (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.

13.5.2 KCA NOVEMBER 2017

This work was done on two bulk grab samples collected from the bottom of the pit, which should be representative of the orebody.

On November 30, 2016, the KCA laboratory facility received six 55 gallon drums of bulk material from the Project. The received material comprised two individual samples identified as PT6: Arenite and PT16: Argilite 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.

13.5.2.1 HEAD ANALYSIS

The samples tested at KCA were assayed for gold and silver content. The results of this analysis are presented in Tables 13-3 and 13-4.

TABLE 13-3 GOLD HEAD ANALYSIS

KCA Sample No.	Description	Assay 1 g/t Au	Assay 2 g/t Au	Average Assay g/t Au
77401 G	PT6: Arenite	0.499	0.521	0.510
77402 G	PT16: Argilite Rock/Graphite Faible	0.514	0.525	0.519

TABLE 13-4 SILVER HEAD ANALYSIS

KCA Sample No.	Description	Assay 1 g/t Ag	Assay 2 g/t Ag	Average Assay g/t Ag
77401 G	PT6: Arenite	0.62	0.62	0.62
77402 G	PT16: Argilite Rock/Graphite Faible	0.62	0.62	0.62

The head analysis of the samples tested also included carbon and sulphur content. The sulphur and carbon assays are presented in Table 13-5.

TABLE 13-5 SULPHUR AND CARBON HEAD ANALYSIS

KCA Sample No.	Description	Total Carbon, %	Organic Carbon, %	Inorganic Carbon, %	Total Sulphur, %	Sulphide Sulphur, %	Sulphate Sulphur, %
77401 G	PT6: Arenite	1.01	0.05	0.96	0.41	0.33	0.08
77402 G	PT16: Argilite Rock/Graphite Faible	1.10	0.04	1.06	0.21	0.14	0.07

13.5.2.2 COARSE BOTTLE ROLL TESTS

Bottle roll tests were conducted by KCA on the material from Essakane at a variety of crush sizes. The results indicate that the ore is amenable to cyanide leaching and that recovery is dependent on crush size. Tables 13-6 and 13-7 show the bottle roll test results and Figure 13-1 shows recovery versus crush size for the Essakane samples.

Based on the coarse bottle roll test results, it was decided that the column leach tests should be run at two different crush sizes to confirm the effects on recovery.

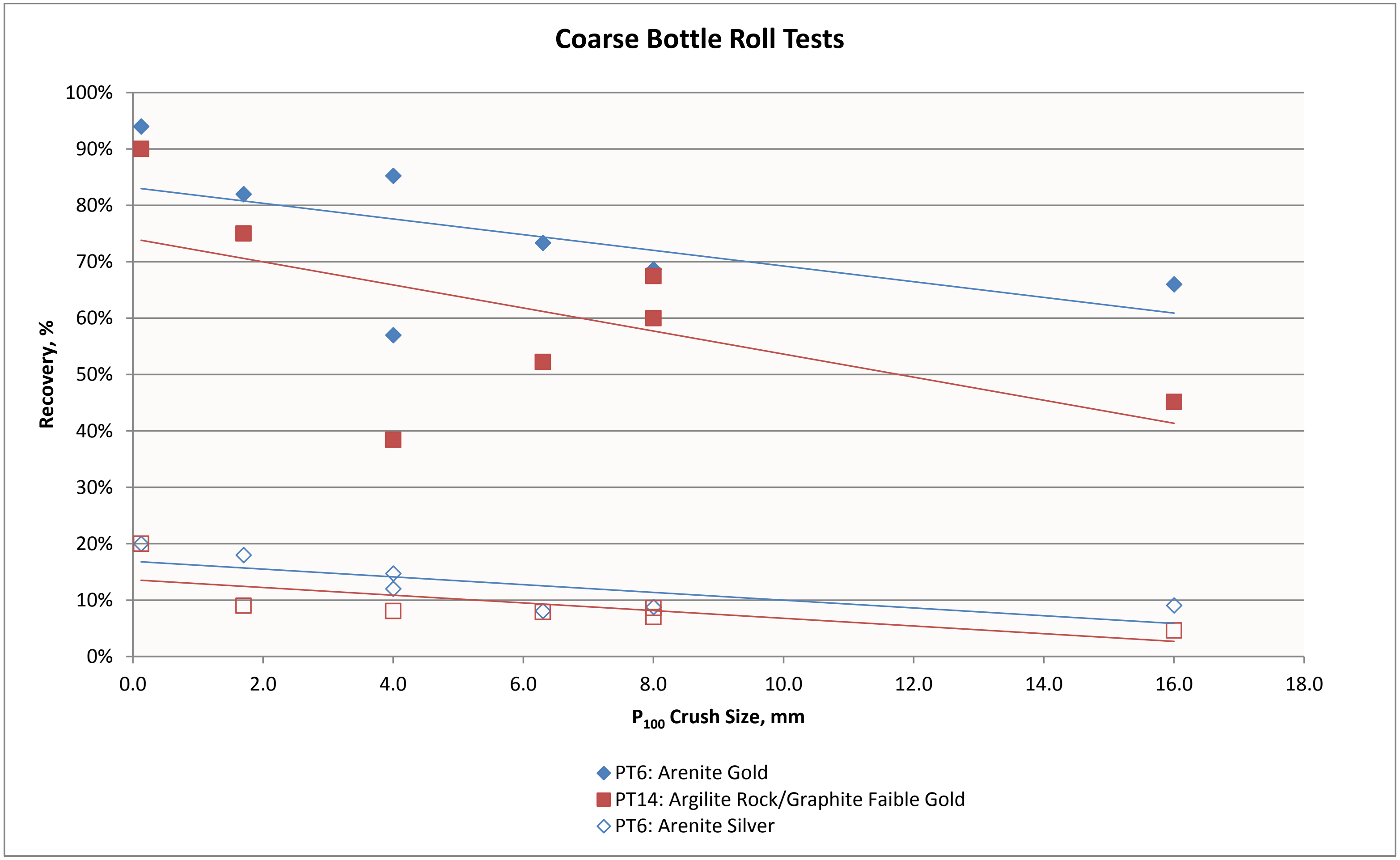
TABLE 13-6 COARSE BOTTLE ROLL TEST RESULTS – GOLD

KCA Sample No.	KCA Test No.	Description	Target p_{100} Size, mm	Calc. p_{80} Size, mm	Head Average, g/t Au	Calculated Head, g/t Au	Extracted, g/t Au	Avg. Tails, g/t Au	Au Extracted, %	Leach Time, hours	Consumption NaCN, kg/t	Addition $Ca(OH)_2$, kg/t
77401 C	77405 A	PT6: Arenite	16.0	12.63	0.510	0.511	0.337	0.174	66%	240	0.10	0.50
77401 D	77406 A	PT6: Arenite	8.0	6.49	0.510	0.571	0.392	0.179	69%	240	0.10	0.50
77401 E	77406 B	PT6: Arenite	6.3	4.85	0.510	0.501	0.367	0.133	73%	240	0.12	0.50
77401 F	77407 A	PT6: Arenite	4.0	3.04	0.510	0.786	0.670	0.116	85%	240	0.15	0.50
77401 F	77413 A	PT6: Arenite	4.0	3.02	0.510	0.642	0.368	0.274	57%	240	0.12	0.65
77401 G	77407 B	PT6: Arenite	1.7	--	0.510	0.453	0.372	0.081	82%	240	0.27	0.50
77401 G	77411 A	PT6: Arenite	0.125	--	0.510	0.455	0.427	0.027	94%	96	0.23	0.75
						0.522						
77402 C	77405 B	PT16: Argilite Rock/Graphite Faible	16.0	12.43	0.519	0.457	0.206	0.250	45%	240	0.07	0.50
77402 D	77408 A	PT16: Argilite Rock/Graphite Faible	8.0	6.03	0.519	0.659	0.444	0.214	67%	240	0.10	0.50
77402 D	77413 B	PT16: Argilite Rock/Graphite Faible	8.0	6.22	0.519	0.348	0.207	0.141	60%	240	0.13	0.40
77402 E	77408 B	PT16: Argilite Rock/Graphite Faible	6.3	4.84	0.519	0.508	0.266	0.243	52%	240	0.10	0.50
77402 F	77409 A	PT16: Argilite Rock/Graphite Faible	4.0	3.06	0.519	0.635	0.244	0.391	38%	240	0.15	0.50
77402 G	77409 B	PT16: Argilite Rock/Graphite Faible	1.7	--	0.519	0.383	0.287	0.096	75%	240	0.21	0.50
77402 G	77411 B	PT16: Argilite Rock/Graphite Faible	0.125	--	0.519	0.625	0.561	0.063	90%	96	0.35	0.50
						0.379						
Average			16.0	12.53	0.515	0.484	0.272	0.212	56%	240	0.09	0.50
Average			8.0	6.26	0.516	0.526	0.348	0.178	65%	240	0.11	0.47
Average			6.3	4.85	0.515	0.505	0.316	0.188	63%	240	0.11	0.50
Average			4.0	3.05	0.513	0.688	0.427	0.260	60%	240	0.14	0.55
Average			1.7	--	0.515	0.418	0.330	0.089	79%	240	0.24	0.50
Average			0.125	--	0.515	0.540	0.484	0.045	92%	96	0.29	0.63

TABLE 13-7 COARSE BOTTLE ROLL TEST RESULTS – SILVER

KCA Sample No.	KCA Test No.	Description	Target p_{100} Size, mm	Calc. p_{80} Size, mm	Head Average, g/t Ag	Calculated Head, g/t Ag	Extracted, g/t Ag	Avg. Tails, g/t Ag	Ag Extracted, %	Leach Time, hours	Consumption NaCN, kg/t	Addition $Ca(OH)_2$, kg/t
77401 C	77405 A	PT6: Arenite	16.0	12.63	0.62	0.50	0.05	0.45	9%	240	0.10	0.50
77401 D	77406 A	PT6: Arenite	8.0	6.49	0.62	0.57	0.05	0.52	9%	240	0.10	0.50
77401 E	77406 B	PT6: Arenite	6.3	4.85	0.62	0.62	0.05	0.57	8%	240	0.12	0.50
77401 F	77407 A	PT6: Arenite	4.0	3.04	0.62	0.65	0.10	0.56	15%	240	0.15	0.50
77401 F	77413 A	PT6: Arenite	4.0	3.02	0.62	0.60	0.07	0.53	12%	240	0.12	0.65
77401 G	77407 B	PT6: Arenite	1.7	--	0.62	0.50	0.09	0.41	18%	240	0.27	0.50
77401 G	77411 A	PT6: Arenite	0.125	--	0.62	0.51	0.10	0.41	20%	96	0.23	0.75
						0.55						
77402 C	77405 B	PT16: Argilite Rock/Graphite Faible	16.0	12.43	0.62	0.50	0.02	0.47	5%	240	0.07	0.50
77402 D	77408 A	PT16: Argilite Rock/Graphite Faible	8.0	6.03	0.62	0.58	0.05	0.53	9%	240	0.10	0.50
77402 D	77413 B	PT16: Argilite Rock/Graphite Faible	8.0	6.22	0.62	0.67	0.05	0.62	7%	240	0.13	0.40
77402 E	77408 B	PT16: Argilite Rock/Graphite Faible	6.3	4.84	0.62	0.63	0.05	0.58	8%	240	0.10	0.50
77402 F	77409 A	PT16: Argilite Rock/Graphite Faible	4.0	3.06	0.62	0.62	0.05	0.57	8%	240	0.15	0.50
77402 G	77409 B	PT16: Argilite Rock/Graphite Faible	1.7	--	0.62	0.56	0.05	0.51	9%	240	0.21	0.50
77402 G	77411 B	PT16: Argilite Rock/Graphite Faible	0.125	--	0.62	0.51	0.10	0.41	20%	96	0.35	0.50
						0.57						
Average			16.0	12.53	0.62	0.50	0.03	0.46	7%	240	0.09	0.50
Average			8.0	6.25	0.62	0.61	0.05	0.56	8%	240	0.11	0.47
Average			6.3	4.85	0.62	0.63	0.05	0.58	8%	240	0.11	0.50
Average			4.0	3.04	0.62	0.62	0.07	0.55	12%	240	0.14	0.55
Average			1.7	--	0.62	0.53	0.07	0.46	14%	240	0.24	0.50
Average			0.1	--	0.62	0.51	0.10	0.41	20%	96	0.29	0.63

FIGURE 13-1 BOTTLE ROLL RECOVERY BY CRUSH SIZE



13.5.2.3 PERMEABILITY TESTING

Preliminary percolation tests were conducted on the two samples by KCA. The samples were tested at two different crush sizes (P_{100} (100% passing) of 19 mm and P_{100} of 8 mm) and four different cement addition rates.

For the percolation tests to pass, the flow out should be 100 times the leach solution application rate of 12 L/h/m^2. Both samples failed the solution clarity portion of the tests at both crush sizes when no cement was added, indicating there is some migration of fines. Taking into account the flow, slump and solution clarity, all of the tests gave acceptable percolation results.

The preliminary percolation tests were followed up by compacted permeability tests. Based on the percolation test results, both samples were tested under a load equivalent to 50 m of heap height without any cement addition at both crush sizes. The 50 m height was selected as the test limit based on KCA's experience with finely crushed material. The compacted permeability test results are presented in Table 13-8.

All four tests gave acceptable results of less than 10% slump and a flow greater than 10 times the leach solution application rate of 12 L/h/m^2. These results indicate that the ore types tested do not require any agglomeration to stack the heap to 50 m.

TABLE 13-8 COMPACTED PERMEABILITY TEST RESULTS

KCA Sample No.	KCA Test No.	Description	Test Phase	Crush Size p_{100}, mm	Cement Added, kg/t	Calc. p_{80}, mm	Effective Height, m	Flow Rate, L/h/m^2	Flow Rate cm/sec	Incremental Slump, %	Cumulative Slump, %	Flow Pass/Fail
			Primary	19			10	1,567	0.044	2%	2%	Pass
77401 H	77417 A	PT6: Arenite	Staged Load	19	0	16	30	2,550	0.071	2%	4%	Pass
			Staged Load	19			50	2,107	0.059	2%	6%	Pass
			Primary	8			10	2,946	0.082	1%	1%	Pass
77401 I	77417 B	PT6: Arenite	Staged Load	8	0	6.3	30	2,587	0.072	3%	4%	Pass
			Staged Load	8			50	2,772	0.077	2%	6%	Pass
			Primary	19			10	3,523	0.098	1%	1%	Pass
77402 H	77418 A	PT16: Argilite Rock/Graphite Faible	Staged Load	19	0	16	30	3,544	0.098	2%	3%	Pass
			Staged Load	19			50	3,428	0.095	3%	6%	Pass
			Primary	8			10	2,827	0.079	2%	2%	Pass
77402 I	77418 B	PT16: Argilite Rock/Graphite Faible	Staged Load	8	0	6.3	30	2,603	0.072	2%	4%	Pass
			Staged Load	8			50	2,315	0.064	3%	7%	Pass

13.5.2.4 COLUMN LEACH TESTS

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 tests data are presented in Table 13-9.

TABLE 13-9 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	Extracted, % Au	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: Argilite Rock/Graphite Faible	19	0.822	0.493	0.329	62%	15.8	69	0.82	0.50
77402 I	77428	PT16: Argilite 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	Extracted, % Ag	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: Argilite Rock/Graphite Faible	19	0.52	0.09	0.43	18%	15.8	69	0.82	0.50
77402 I	77428	PT16: Argilite 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, a decision was made 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.

On March 24, 2017, the KCA laboratory facility received two additional 55 gallon drums of bulk material from the Essakane Project. The received material comprised additional material identified as PT6: Arenite and PT16: Argilite Rock/Graphite Faible.

Sample preparation was conducted to provide material for ATWAL abrasion testing.

Twenty-eight 55 gallon drums of ½ and ¼ split HQ core material from 27 drill holes representative of the main zone 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.

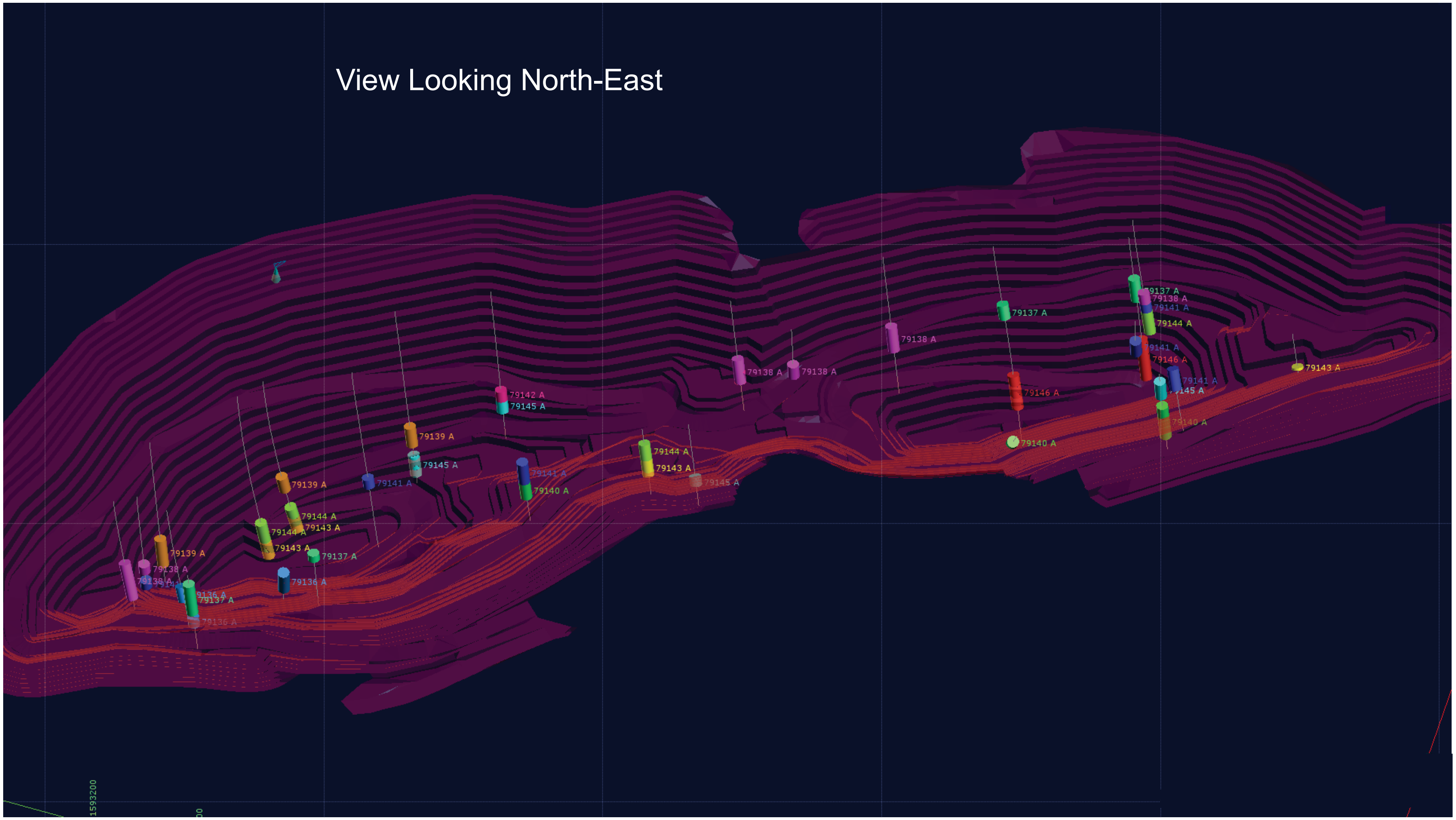
The composite samples are summarized in Table 13-10.

A diagram showing the special orientation of the metallurgical holes in the EMZ pit and composite selection is presented in Figure 13-2.

TABLE 13-10 COMPOSITE SAMPLES

KCA Sample No.	Composite Sample Name	Composite Sample Short Name	Composite Weight, kg
79136 A	Arenite West 223	AREW223	362.09
79137 A	Upper Arenite East 243	UARE243	541.05
79138 A	Middle Arenite East 243	MARE243	822.93
79139 A	Lower Arenite East 243	LARE243	541.77
79140 A	Argilite West 313	ARGW313	492.3
79141 A	Argilite East 343	ARGE343	597.58
79142 A	Argilite East 343 Refractory	ARGE343REF	88.47
79143 A	Arenite West 413	AREW413	342.08
79144 A	Arenite East 443	AREE443	564.35
79145 A	Arenite 443 & 413 Refractory	ARE400REF	459.37
79146 A	Argilite 513, 543, 623, 643	ARG550	443.19

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.



0 100 200 300 400 500
Metres

Figure 13-2

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Metallurgical Drill Holes in Essakane Main Zone Pit

July 2018

Essakane S.A., 2018.

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.

The variability samples are summarized in Table 13-11.

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 Canada for graphitic carbon, WETLABS for acid base accounting, and ThyssenKrupp Industrial Solutions in Germany for abrasion (ATWAL) testing.

TABLE 13-11 VARIABILITY SAMPLES

KCA Sample No.	Weight kg	Rock Type	KCA Sample No.	Weight kg	Rock Type
79148 A	7.16	Arenite West 223	79153 A	6.86	Argilite East 343
79148 B	7.57	Arenite West 223	79153 B	7.18	Argilite East 343
79148 C	7.45	Arenite West 223	79153 C	8.5	Argilite East 343
79148 D	7.44	Arenite West 223	79153 D	6.68	Argilite East 343
79149 A	6.75	Upper Arenite East 243	79153 E	6.47	Argilite East 343
79149 B	7.3	Upper Arenite East 243	79153 F	6.71	Argilite East 343
79149 C	7.64	Upper Arenite East 243	79153 G	7.54	Argilite East 343
79149 D	8.1	Upper Arenite East 243	79154 A	7.8	Arenite West 413
79149 E	5.51	Upper Arenite East 243	79154 B	6.98	Arenite West 413
79150 A	6.4	Middle Arenite East 243	79154 C	6.79	Arenite West 413
79150 B	6.77	Middle Arenite East 243	79155 A	7.9	Arenite East 443
79150 C	6.45	Middle Arenite East 243	79155 B	7.68	Arenite East 443
79150 D	6.73	Middle Arenite East 243	79155 C	8.41	Arenite East 443
79150 E	6.68	Middle Arenite East 243	79155 D	7.53	Arenite East 443
79150 F	7.62	Middle Arenite East 243	79155 E	6.48	Arenite East 443
79150 G	7.07	Middle Arenite East 243	79156 A	8.71	Arenite 443 & 413 Refractory
79150 H	8.21	Middle Arenite East 243	79156 B	7.23	Arenite 443 & 413 Refractory
79150 I	6.3	Middle Arenite East 243	79156 C	8.54	Arenite 443 & 413 Refractory
79151 A	6.86	Lower Arenite East 243	79156 D	7.78	Arenite 443 & 413 Refractory
79151 B	8.13	Lower Arenite East 243	79157 A	6.22	Argilite 513, 543, 623, 643
79151 C	8.25	Lower Arenite East 243	79157 B	7.62	Argilite 513, 543, 623, 643
79151 D	8.01	Lower Arenite East 243	79157 C	9.08	Argilite 513, 543, 623, 643
79151 E	7.48	Lower Arenite East 243	79157 D	6.68	Argilite 513, 543, 623, 643
79152 A	6.28	Argilite West 313	79158 A	7.39	Middle Arenite East 243
79152 B	7.11	Argilite West 313	79158 B	9.59	Argilite West 313
79152 C	6.4	Argilite West 313	79158 C	8.01	Arenite West 413
79152 D	7.23	Argilite West 313	79158 D	7.22	Argilite East 343

13.5.3.1 HEAD ANALYSIS

Head analyses for PT6: Arenite and PT16: Argilite Rock/Graphite Faible were previously presented in this report. A portion of the head material for the core composite material was crushed to 100% passing 1.70 mm. From the blended minus 1.70 mm material, duplicate portions were split out and individually ring and puck pulverized to a target grind size of 80% passing 0.075 mm.

A portion of this pulverized head material from each composite was analyzed for gold and silver by standard fire assay and wet chemistry methods. Head material was also assayed semi-quantitatively for an additional series of elements and for whole rock constituents. In addition to these semi-quantitative analyses, the head material was assayed by quantitative methods for carbon, sulphur, and mercury. A cyanide shake test was also conducted on a portion of the pulverized head material.

The gold and silver assay results for the composite samples are presented in Tables 13-12 and 13-13.

TABLE 13-12 COMPOSITE SAMPLE GOLD HEAD ANALYSIS

KCA Sample No.	Description	ALS Assay g/t Au	KCA Assay 1 g/t Au	KCA Assay 2 g/t Au	KCA Average Assay g/t Au
79136 B	Arenite West 223	0.048	0.089	0.087	0.088
79137 C	Upper Arenite East 243	0.228	0.195	0.199	0.197
79138 C	Middle Arenite East 243	0.119	0.178	0.171	0.175
79139 C	Lower Arenite East 243	0.383	0.449	0.429	0.439
79140 C	Argilite West 313	0.630	0.309	0.298	0.303
79141 C	Argilite East 343	0.376	0.552	0.573	0.562
79142 A	Argilite East 343 Refractory	0.011	0.034	0.034	0.034
79143 B	Arenite West 413	0.220	0.552	0.538	0.545
79144 C	Arenite East 443	0.647	0.456	0.480	0.468
79145 B	Arenite 443 & 413 Refractory	0.367	0.346	0.333	0.339
79146 B	Argilite 513, 543, 623, 643	0.385	0.312	0.302	0.307

TABLE 13-13 COMPOSITE SAMPLE SILVER HEAD ANALYSIS

KCA Sample No.	Description	ALS Assay g/t Ag	ALS Assay g/t Ag	KCA Assay 1 g/t Ag	KCA Assay 2 g/t Ag	KCA Average Assay g/t Ag
79136 B	Arenite West 223	<0.5	0.8	0.41	0.41	0.41
79137 C	Upper Arenite East 243	<0.5	0.2	0.41	0.41	0.41
79138 C	Middle Arenite East 243	<0.5	<0.2	0.21	0.41	0.31
79139 C	Lower Arenite East 243	<0.5	0.2	0.41	0.41	0.41
79140 C	Argilite West 313	<0.5	<0.2	0.41	0.41	0.41
79141 C	Argilite East 343	<0.5	<0.2	0.21	0.41	0.31
79142 A	Argilite East 343 Refractory	<0.5	<0.2	0.21	0.21	0.21
79143 B	Arenite West 413	<0.5	<0.2	0.41	0.41	0.41
79144 C	Arenite East 443	<0.5	<0.2	0.41	0.41	0.41
79145 B	Arenite 443 & 413 Refractory	<0.5	0.2	0.41	0.41	0.41
79146 B	Argilite 513, 543, 623, 643	<0.5	0.2	0.41	0.41	0.41

The composite samples are generally in the correct range for the expected grades for the heap leach, although a few composites (Arenite West 223, Upper Arenite East 243, Middle Arenite East 243 and Argilite East 343 Refractory) are below the cutoff grade for the heap. The silver assays are all near the detection limit for silver.

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 ore should be similar to what has been treated in the mill and the requirement for environmental controls should be the same as the existing 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. The carbon and sulphur analyses for the composite samples are presented in Table 13-14.

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.

TABLE 13-14 COMPOSITE SAMPLE CARBON AND SULPHUR ANALYSIS

KCA Sample No.	Description	Total Carbon, %	SGS Assay, Graphitic Carbon, %	Total Sulphur, %	Sulphide Sulphur, %	Sulphate Sulphur, %
79136 B	Arenite West 223	1.15	<0.05	0.19	0.10	0.09
79137 C	Upper Arenite East 243	1.19	<0.05	0.30	0.17	0.13
79138 C	Middle Arenite East 243	1.20	<0.05	0.26	0.16	0.10
79139 C	Lower Arenite East 243	1.19	0.07	0.46	0.38	0.08
79140 C	Argilite West 313	1.14	0.26	0.18	0.11	0.08
79141 C	Argilite East 343	1.36	0.18	0.26	0.18	0.08
79142 A	Argilite East 343 Refractory	1.11	0.13	0.28	0.19	0.09
79143 B	Arenite West 413	1.29	0.19	0.36	0.26	0.10
79144 C	Arenite East 443	1.35	0.16	0.43	0.35	0.08
79145 B	Arenite 443 & 413 Refractory	1.13	0.14	0.39	0.30	0.09
79146 B	Argilite 513, 543, 623, 643	1.86	0.12	0.38	0.26	0.12

13.5.3.2 HPGR CRUSHER TEST WORK

Material was crushed by KCA through a PILOTWAL High Pressure Grinding Roll (HPGR) to determine the parameters required for the sizing of an industrial HPGR and to produce HPGR product for laboratory test work. A portion of the material was sent to ThyssenKrupp Industrial Solutions for abrasion (ATWAL) testing.

The laboratory HPGR test work conducted on each sample included a single pass tests on a PILOTWAL unit at constant pressure setting and ore moisture content.

The results of the abrasion testing are presented in Table 13-15.

TABLE 13-15 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: Argilite Rock/Graphite Faible	1	< 3.15	4.0	14.87
77435 A	PT16: Argilite 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: Argilite Rock/Graphite Faible.

The HPGR crushing parameters were tested with the PILOTWAL unit at KCA. Dry screening tests were conducted to get an indication of the screen performance achievable with the partly compacted discharge of a HPGR. The PILOTWAL test results are presented in Tables 13-16 and 13-17.

TABLE 13-16 SUMMARY OF HPGR THROUGHPUT DATA

Test No.	Description	Moisture, Added %	Specific Energy, Net (dry feed) kWh/t	Specific Throughput (dry feed) t*s/(m³*h)	Net Power at Shaft kW	Specific Force N/mm²
79107	PT6: Arenite	3.0	1.5	304.6	13.4	3.6
79108	PT16: Argilite Rock/Graphite Faible	3.0	1.7	305.8	14.8	3.6
79160	Arenite West 223	3.0	1.8	272.8	14.6	3.5
79161	Upper Arenite East 243	3.0	1.8	269.0	14.4	3.4
79162	Middle Arenite East 243	3.0	1.8	279.3	14.5	3.5
79163	Lower Arenite East 243	3.0	1.7	281.6	13.7	3.4
79164	Argilite West 313	3.0	1.8	300.3	15.9	3.5
79165	Argilite East 343	3.0	1.6	304.1	14.5	3.4
79166	Arenite West 413	3.0	1.7	283.0	14.3	3.4
79167	Arenite East 443	3.0	1.7	279.7	14.2	3.4
79168	Arenite 443 & 413 Refractory	3.0	1.8	272.7	14.0	3.4
79169	Argilite 513, 543, 623, 643	3.0	1.7	274.7	13.4	3.4

TABLE 13-17 SUMMARY OF HPGR FEED AND DISCHARGE DATA

Test No.	Description	Specific Force N/mm²	Total Discharge % < 6.3 mm %	Total Discharge % < 1 mm %	Total Discharge % < 212 µm %	Center % < 6.3 mm %	Center % < 1 mm %	Center % < 212 µm %
79107	Fresh Feed	0	38.6	15.4	10.8	38.6	15.4	10.8
79107	PT6: Arenite	3.6	71.8	32.7	21.9	77.1	35.8	24.2
79108	Fresh Feed	0	31.8	11.8	7.5	31.8	11.8	7.5
79108	PT16: Argilite Rock/Graphite Faible	3.6	72.3	33.6	20.6	78.8	37.6	23.2
79160	Fresh Feed	0	30.2	7.5	2.5	30.2	7.5	2.5
79160	Arenite West 223	3.5	82.2	38.0	13.5	82.7	37.6	12.6
79161	Fresh Feed	0	5.2	0.5	0.3	5.2	0.5	0.3
79161	Upper Arenite East 243	3.4	68.9	29	9.0	76.5	33.2	9.0
79162	Fresh Feed	0	21.8	4.4	1.3	21.8	4.4	1.3
79162	Middle Arenite East 243	3.5	67.7	29.1	11.8	73.4	31.9	12.9
79163	Fresh Feed	0	32.9	9.7	3.1	32.9	9.7	3.1
79163	Lower Arenite East 243	3.4	74.8	33.5	13.8	84.1	38.9	16.3
79164	Fresh Feed	0	8.5	0.9	0.5	8.5	0.9	0.5
79164	Argilite West 313	3.5	72.7	31.3	12.1	77.5	34.5	13.4

Test No.	Description	Specific Force N/mm²	Total Discharge % < 6.3 mm %	% < 1 mm %	% < 212 µm %	Center % < 6.3 mm %	% < 1 mm %	% < 212 µm %
79165	Fresh Feed	0	26.6	7.2	2.6	26.6	7.2	2.6
79165	Argilite East 343	3.4	71.1	32.1	12.4	81.2	37.6	14.0
79166	Fresh Feed	0	15.8	3.4	1.0	15.8	3.4	1.0
79166	Arenite West 413	3.4	63.7	24.3	10.9	72.2	28.5	12.7
79167	Fresh Feed	0	15.3	2.6	0.7	15.3	2.6	0.7
79167	Arenite East 443	3.4	62.0	24.4	11.2	68.5	28.2	13.0
79168	Fresh Feed	0	22.5	4.4	1.9	22.5	4.4	1.9
79168	Arenite 443 & 413 Refractory	3.4	62	24.6	12.2	70.6	28.3	13.6
79169	Fresh Feed	0	22.7	3.2	1.2	22.7	3.2	1.2
79169	Argilite 513, 543, 623, 643	3.4	75.1	32.7	15.7	81.2	36.5	17.3

Based on preliminary data, a specific throughput of 290 (t*s)/(m^3*h) and a specific energy of 1.7 kWh/t were selected for this study. The final data presented here shows an average specific throughput of 286 (t*s)/(m^3*h) and average specific energy of 1.7 kWh/t.

13.5.3.3 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, Argilite West 313, Argilite East 343, Argilite East 343 Refractory, Arenite West 413, Arenite East 443, Arenite 443 & 413 Refractory and Argilite 513, 543, 623, 643 were submitted to Hazen Research, Inc in Golden, Colorado for comminution testing. Test work was completed to provide Bond Crusher Impact (CWi) and a Bond Abrasion Work Index (A_i) for the composite samples.

The results are summarized in Table 13-18.

TABLE 13-18 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	Argilite West 313	0.0786	20.1
79141 A	Argilite 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 & 413 Refractory	0.4139	18.4
79146 A	Argilite 513, 543, 623, 643	0.4414	13.6

The average Bond Abrasion index is 0.4187 and the material would be considered abrasive.

The average Bond Crusher Impact index is 17.31 for the material tested, which would be classified as hard to very hard.

13.5.3.4 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 number indicates that more gold is extracted from the sample with the addition of the spike than without addition of the spike.

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. Values between 10% and 20% would indicate moderate preg-robbing, while values greater than 20% would indicate highly preg-robbing. Graphitic carbon assays results for each sample are presented for comparison purposes.

The results of individual preg-robbing tests are presented in Table 13-19.

TABLE 13-19 COMPOSITE PREG ROBBING TEST RESULTS

KCA Sample No.	Description	Split	SGS Assay, Graphitic Carbon, %	Head Assay, g/t Au	Leach Results: Final pH	Leach Results: Au, mg/L	Leach Results: Extraction, g/t Au	Leach Results: Est. Ext., Au, %	Preg Robbing Results: Spike Au, mg/L	Preg Robbing Results: Direct Au, mg/L	Preg Robbing Results: Spiked Leach Au, mg/L	Preg Robbing Results: 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	Argilite West 313	A	0.26	0.309	10.4	0.06	0.120	39%	1.00	0.06	0.89	17%
79140 C	Argilite West 313	B	0.26	0.298	10.4	0.05	0.100	34%	1.00	0.05	0.88	17%
79141 C	Argilite East 343	A	0.18	0.552	10.4	0.06	0.120	22%	1.00	0.06	1.00	6%
79141 C	Argilite East 343	B	0.18	0.573	10.4	0.07	0.140	24%	1.00	0.07	1.00	7%
79142 A	Argilite East 343 Refractory	A	0.13	0.034	10.5	0.01	0.020	58%	1.00	0.01	0.87	14%
79142 A	Argilite 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	Argilite 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	Argilite 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, but there is not a direct correlation. Caution will need to be taken to minimize the amount of preg robbing material stacked on the heap.

13.5.3.5 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 a NNP value less than -20 is indicative of an acid generating material.

KCA sent head composite material from Arenite West 223 Core Material, Upper Arenite East 243, Middle Arenite East 243, Lower Arenite East 243, Argilite West 313, Argilite East 343, Argilite East 343 Refractory, Arenite West 413, Arenite East 443, Arenite 443 & 413 Refractory and Argilite 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) to WETLABS for testing. The test work showed a NNP value greater than 20 for all samples, therefore, the material has low acid producing potential.

13.5.3.6 BOTTLE ROLL TEST WORK

Standard bottle roll leach testing was conducted on a portion of the material. A 1,000 g portion of head material was pulverized to a target size of 100% passing 0.150 mm. The pulverized slurry was then utilized for leach testing.

LeachWELL bottle roll leach testing was conducted on a portion of the material. A 1,000 g portion of head material was pulverized in a laboratory rod mill to a target size of 100% passing 0.150 mm. The pulverized slurry was then utilized for leach testing.

Variability bottle roll leach testing was conducted on selected composites. A 500 g portion of the composite material was pulverized to a target size of 100% passing 0.150 mm. The pulverized slurry was then utilized for leach testing.

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 to 1.188 g/t. The sodium cyanide consumptions 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 results of standard composite bottle roll leach test work are summarized in Table 13-20.

The results of the LeachWELL composite bottle roll tests are summarized in Table 13-21.

The results of the variability bottle roll tests are summarized in Table 13-22.

TABLE 13-20 SUMMARY OF COMPOSITE STANDARD BOTTLE ROLL LEACH TESTS

KCA Sample No.	KCA Test No.	Description	Target p_{80} Size, mm	Head Average, g/t Au	Calculated Head, g/t Au	Extracted, g/t Au	Avg. Tails, g/t Au	Au Extracted, %	Leach Time, hours	Consumption NaCN, kg/t	Addition $Ca(OH)_2$, kg/t
79136 B	79171 A	Arenite West 223	0.150	0.088	0.107	0.097	0.010	90%	96	0.79	1.25
79137 C	79171 B	Upper Arenite East 243	0.150	0.197	0.185	0.172	0.014	93%	96	0.91	1.25
79138 C	79171 C	Middle Arenite East 243	0.150	0.175	0.307	0.268	0.039	87%	96	0.97	1.25
79139 C	79171 D	Lower Arenite East 243	0.150	0.439	0.577	0.519	0.058	90%	96	0.88	1.50
79140 C	79172 A	Argilite West 313	0.150	0.303	0.208	0.094	0.115	45%	96	0.91	1.25
79141 C	79172 B	Argilite East 343	0.150	0.562	1.188	1.054	0.134	89%	96	0.94	1.75
79142 A	79172 C	Argilite East 343 Refractory	0.150	0.034	0.079	0.012	0.067	15%	96	0.94	1.50
79143 B	79172 D	Arenite West 413	0.150	0.545	0.305	0.243	0.062	80%	96	1.02	1.25
79144 C	79173 A	Arenite East 443	0.150	0.468	0.409	0.381	0.027	93%	96	0.97	1.25
79145 B	79173 B	Arenite 443 & 413 Refractory	0.150	0.339	0.897	0.810	0.087	90%	96	0.99	1.50
79146 B	79173 C	Argilite 513, 543, 623, 643	0.150	0.307	0.376	0.335	0.041	89%	96	1.10	1.50

KCA Sample No.	KCA Test No.	Description	Target p_{80} Size, mm	Head Average, g/t Ag	Calculated Head, g/t Ag	Extracted, g/t Ag	Avg. Tails, g/t Ag	Ag Extracted, %	Leach Time, hours	Consumption NaCN, kg/t	Addition $Ca(OH)_2$, kg/t
79136 B	79171 A	Arenite West 223	0.150	0.41	0.38	0.07	0.31	19%	96	0.79	1.25
79137 C	79171 B	Upper Arenite East 243	0.150	0.40	0.38	0.07	0.31	19%	96	0.91	1.25
79138 C	79171 C	Middle Arenite East 243	0.150	0.30	0.28	0.07	0.21	26%	96	0.97	1.25
79139 C	79171 D	Lower Arenite East 243	0.150	0.40	0.38	0.07	0.21	19%	96	0.88	1.50
79140 C	79172 A	Argilite West 313	0.150	0.40	0.38	0.07	0.31	18%	96	0.91	1.25
79141 C	79172 B	Argilite East 343	0.150	0.30	0.32	0.12	0.21	36%	96	0.94	1.75
79142 A	79172 C	Argilite East 343 Refractory	0.150	0.20	0.23	0.02	0.31	11%	96	0.94	1.50
79143 B	79172 D	Arenite West 413	0.150	0.40	0.38	0.07	0.31	19%	96	1.02	1.25
79144 C	79173 A	Arenite East 443	0.150	0.40	0.40	0.09	0.31	23%	96	0.97	1.25
79145 B	79173 B	Arenite 443 & 413 Refractory	0.150	0.40	0.41	0.10	0.31	24%	96	0.99	1.50
79146 B	79173 C	Argilite 513, 543, 623, 643	0.150	0.40	0.40	0.09	0.31	23%	96	1.10	1.50

TABLE 13-21 SUMMARY OF COMPOSITE LEACHWELL BOTTLE ROLL LEACH TESTS

KCA Sample No.	KCA Test No.	Description	Finish	Target p_{100} Size, mm	Head Average, g/t Au	Calculated Head, g/t Au	Extracted, g/t Au	Avg. Tails, g/t Au	Au Extracted, %	Leach Time, hours	Consumption NaCN, kg/t	Addition $Ca(OH)_2$, kg/t	Soln, mg Au/L
79136 B	80260 A	Arenite West 223	AAS	0.15	0.088	0.087	0.080	0.007	92%	12	0.12	5.00	0.04
			ICP			0.098	0.091	0.007	93%				0.046
			DIBK-AAS			0.109	0.102	0.007	94%				0.051
79137 C	80260 B	Upper Arenite East 243	AAS	0.15	0.197	0.234	0.160	0.074	68%	12	<0.10	5.00	0.08
			ICP			0.223	0.149	0.074	67%				0.075
			DIBK-AAS			0.268	0.194	0.074	72%				0.097
79138 C	80260 C	Middle Arenite East 243	AAS	0.15	0.175	0.258	0.220	0.038	85%	12	0.14	5.00	0.11
			ICP			0.248	0.210	0.038	85%				0.105
			DIBK-AAS			0.316	0.278	0.038	88%				0.139
79139 C	80260 D	Lower Arenite East 243	AAS	0.15	0.439	0.643	0.580	0.063	90%	12	<0.10	5.00	0.29
			ICP			0.610	0.547	0.063	90%				0.274
			DIBK-AAS			0.767	0.704	0.063	92%				0.352
79140 C	80260 E	Argilite West 313	AAS	0.15	0.303	0.645	0.580	0.065	90%	12	0.14	5.00	0.29
			ICP			0.580	0.515	0.065	89%				0.258
			DIBK-AAS			0.737	0.672	0.065	91%				0.336
79141 C	80260 F	Argilite East 343	AAS	0.15	0.562	0.505	0.440	0.065	87%	12	0.14	5.00	0.22
			ICP			0.477	0.412	0.065	86%				0.206
			DIBK-AAS			0.585	0.520	0.065	89%				0.260
79142 A	80260 G	Argilite East 343 Refractory	AAS	0.15	0.034	0.034	0.020	0.014	59%	12	<0.10	5.00	0.01
			ICP			0.030	0.016	0.014	54%				0.008
			DIBK-AAS			0.032	0.018	0.014	57%				0.009
79143 B	80260 H	Arenite West 413	AAS	0.15	0.545	0.299	0.260	0.039	87%	12	0.12	5.00	0.13
			ICP			0.260	0.221	0.039	85%				0.111
			DIBK-AAS			0.349	0.310	0.039	89%				0.155

KCA Sample No.	KCA Test No.	Description	Finish	Target p_{100} Size, mm	Head Average, g/t Au	Calculated Head, g/t Au	Extracted, g/t Au	Avg. Tails, g/t Au	Au Extracted, %	Leach Time, hours	Consumption NaCN, kg/t	Addition $Ca(OH)_2$, kg/t	Soln, mg Au/L
79144 C	80260 I	Arenite East 443	AAS	0.15	0.468	1.822	1.760	0.062	97%	12	<0.10	5.00	0.88
			ICP			1.590	1.528	0.062	96%				0.764
			DIBK-AAS			1.836	1.774	0.062	97%				0.887
79145 B	80261 A	Arenite 443 & 413 Refractory	AAS	0.15	0.339	5.717	5.180	0.537	91%	12	<0.10	5.00	2.59
			ICP			5.364	4.827	0.537	90%				2.414
			DIBK-AAS			6.017	5.480	0.537	91%				2.740
79146 B	80261 B	Argilite 513, 543, 623, 643	AAS	0.15	0.307	0.573	0.520	0.053	91%	12	<0.10	5.00	0.26
			ICP			0.518	0.465	0.053	90%				0.233
			DIBK-AAS			0.661	0.608	0.053	92%				0.304

TABLE 13-22 SUMMARY OF VARIABILITY BOTTLE ROLL LEACH TESTS

KCA Sample No.	KCA Test No.	Composite Domain	Test Type	Finish	Target p_{100} Size, mm	Head Average, g/t Au	Calculated Head, g/t Au	Extracted, g/t Au	Avg. Tails, g/t Au	Au Extracted, %	Leach Time, hours	Consumption NaCN, kg/t	Addition $Ca(OH)_2$, kg/t
	80263 A		Standard	AAS		0.436	0.440	0.427	0.014	97%	96	0.39	1.50
	81010 A		Au Spike	AAS		0.436	0.459	0.420	0.039	91%	24	0.09	1.50
79148 A		Arenite West 223		AAS	0.15	0.436	0.426	0.405	0.021	95%			
	81001 A		LeachWELL	ICP		0.436	0.443	0.422	0.021	95%	12	0.38	5.00
				DIBK-AAS		0.436	0.461	0.440	0.021	95%			
	80263 B		Standard	AAS		0.021	0.026	0.016	0.010	61%	96	0.10	1.00
	81010 B		Au Spike	AAS		0.021	0.067	0.060	0.007	90%	24	0.18	1.00
79148 B		Arenite West 223		AAS	0.15	0.021	0.052	0.030	0.022	57%			
	81001 B		LeachWELL	ICP		0.021	0.039	0.017	0.022	43%	12	0.30	5.00
				DIBK-AAS		0.021	0.042	0.020	0.022	47%			
	80263 C		Standard	AAS		0.122	0.059	0.051	0.009	86%	96	0.34	1.00
	81010 C		Au Spike	AAS		0.122	0.097	0.090	0.007	93%	24	0.20	1.00
79148 C		Arenite West 223		AAS	0.15	0.122	0.124	0.105	0.019	85%			
	81001 C		LeachWELL	ICP		0.122	0.113	0.095	0.019	83%	12	0.39	5.00
				DIBK-AAS		0.122	0.130	0.111	0.019	85%			
	80263 D		Standard	AAS		0.087	0.523	0.265	0.257	51%	96	0.49	1.00
	81010 D		Au Spike	AAS		0.087	0.319	0.300	0.019	94%	24	0.27	1.00
79148 D		Arenite West 223		AAS	0.15	0.087	0.105	0.090	0.015	85%			
	81001 D		LeachWELL	ICP		0.087	0.128	0.113	0.015	88%	12	0.45	5.00
				DIBK-AAS		0.087	0.128	0.113	0.015	88%			
	80264 A		Standard	AAS		0.043	0.103	0.050	0.053	48%	96	0.08	1.00
	81010 E		Au Spike	AAS		0.043	0.105	0.090	0.015	86%	24	0.17	1.00
79149 A		Upper Arenite East 243		AAS	0.15	0.043	0.089	0.075	0.014	85%			
	81001 E		LeachWELL	ICP		0.043	0.084	0.071	0.014	84%	12	0.36	5.00
				DIBK-AAS		0.043	0.090	0.077	0.014	85%			
	80264 B		Standard	AAS		0.213	0.163	0.119	0.045	73%	96	0.16	1.00
	81010 F		Au Spike	AAS		0.213	0.212	0.195	0.017	92%	24	0.20	1.00
79149 B		Upper Arenite East 243		AAS	0.15	0.213	0.433	0.420	0.013	97%			
	81001 F		LeachWELL	ICP		0.213	0.461	0.449	0.013	97%	12	0.42	5.00
				DIBK-AAS		0.213	0.469	0.456	0.013	97%			

KCA Sample No.	KCA Test No.	Composite Domain	Test Type	Finish	Target p_{100} Size, mm	Head Average, g/t Au	Calculated Head, g/t Au	Extracted, g/t Au	Avg. Tails, g/t Au	Au Extracted, %	Leach Time, hours	Consumption NaCN, kg/t	Addition $Ca(OH)_2$, kg/t
	80264 C		Standard	AAS		0.051	0.062	0.050	0.012	81%	96	0.49	1.00
	81010 G	Upper Arenite East 243	Au Spike	AAS		0.051	0.120	0.105	0.015	87%	24	0.11	1.00
79149 C				AAS	0.15	0.051	0.066	0.045	0.021	69%			
	81001 G		LeachWELL	ICP		0.051	0.087	0.066	0.021	76%	12	0.42	5.00
				DIBK-AAS		0.051	0.079	0.059	0.021	74%			
	80264 D		Standard	AAS		0.151	0.294	0.121	0.173	41%	96	0.56	1.50
	81010 H	Upper Arenite East 243	Au Spike	AAS		0.151	0.233	0.180	0.053	77%	24	0.24	1.50
79149 D				AAS	0.15	0.151	0.132	0.105	0.027	79%			
	81001 H		LeachWELL	ICP		0.151	0.158	0.131	0.027	83%	12	0.32	5.00
				DIBK-AAS		0.151	0.164	0.137	0.027	83%			
	80265 A		Standard	AAS		0.158	0.137	0.118	0.019	86%	96	0.19	1.00
	81010 I	Upper Arenite East 243	Au Spike	AAS		0.158	2.844	1.215	1.629	43%	24	0.24	1.50
79149 E				AAS	0.15	0.158	0.134	0.120	0.014	90%			
	81001 I		LeachWELL	ICP		0.158	0.135	0.122	0.014	90%	12	0.54	5.00
				DIBK-AAS		0.158	0.143	0.129	0.014	90%			
	80265 B		Standard	AAS		0.441	0.432	0.340	0.093	79%	96	0.39	1.00
	81010 J	Middle Arenite East 243	Au Spike	AAS		0.441	0.363	0.330	0.033	91%	24	0.15	1.00
79150 A				AAS	0.15	0.441	0.341	0.315	0.026	92%			
	81001 J		LeachWELL	ICP		0.441	0.363	0.338	0.026	93%	12	0.42	5.00
				DIBK-AAS		0.441	0.380	0.354	0.026	93%			
	80265 C		Standard	AAS		0.283	0.217	0.189	0.027	87%	96	0.40	1.00
	81011 A	Middle Arenite East 243	Au Spike	AAS		0.283	0.234	0.210	0.024	90%	24	0.23	1.00
79150 B				AAS	0.15	0.283	0.244	0.225	0.019	92%			
	81002 A		LeachWELL	ICP		0.283	0.260	0.242	0.019	93%	12	0.30	5.00
				DIBK-AAS		0.283	0.278	0.260	0.019	93%			
	80265 D		Standard	AAS		0.264	0.222	0.189	0.033	85%	96	0.50	1.50
	81011 B	Middle Arenite East 243	Au Spike	AAS		0.264	0.224	0.195	0.029	87%	24	0.26	1.50
79150 C				AAS	0.15	0.264	0.183	0.165	0.018	90%			
	81002 B		LeachWELL	ICP		0.264	0.193	0.176	0.018	91%	12	0.09	5.00
				DIBK-AAS		0.264	0.199	0.182	0.018	91%			

KCA Sample No.	KCA Test No.	Composite Domain	Test Type	Finish	Target p_{100} Size, mm	Head Average, g/t Au	Calculated Head, g/t Au	Extracted, g/t Au	Avg. Tails, g/t Au	Au Extracted, %	Leach Time, hours	Consumption NaCN, kg/t	Addition $Ca(OH)_2$, kg/t
	80266 A		Standard	AAS		0.117	0.089	0.069	0.021	77%	96	0.55	1.50
	81011 C		Au Spike	AAS		0.117	0.139	0.120	0.019	86%	24	0.17	1.50
79150 D		Middle Arenite East 243		AAS	0.15	0.117	0.089	0.075	0.014	85%			
	81002 C		LeachWELL	ICP		0.117	0.099	0.086	0.014	86%	12	0.51	5.00
				DIBK-AAS		0.117	0.101	0.087	0.014	86%			
	80266 B		Standard	AAS		0.266	0.206	0.166	0.039	81%	96	0.33	1.00
	81011 D		Au Spike	AAS		0.266	0.165	0.150	0.015	91%	24	0.42	1.00
79150 E		Middle Arenite East 243		AAS	0.15	0.266	0.179	0.165	0.014	92%			
	81002 D		LeachWELL	ICP		0.266	0.201	0.188	0.014	93%	12	0.58	5.00
				DIBK-AAS		0.266	0.215	0.201	0.014	94%			
	80266 C		Standard	AAS		0.160	0.166	0.135	0.031	81%	96	0.65	1.50
	81011 E		Au Spike	AAS		0.160	0.107	0.090	0.017	84%	24	0.18	1.50
79150 F		Middle Arenite East 243		AAS	0.15	0.160	0.089	0.075	0.014	85%			
	81002 E		LeachWELL	ICP		0.160	0.097	0.083	0.014	86%	12	0.15	5.00
				DIBK-AAS		0.160	0.092	0.078	0.014	85%			
	80266 D		Standard	AAS		0.287	0.177	0.138	0.039	78%	96	0.43	1.00
	81011 F		Au Spike	AAS		0.287	0.227	0.210	0.017	92%	24	0.11	1.00
79150 G		Middle Arenite East 243		AAS	0.15	0.287	0.179	0.165	0.014	92%			
	81002 F		LeachWELL	ICP		0.287	0.177	0.164	0.014	92%	12	0.14	5.00
				DIBK-AAS		0.287	0.194	0.180	0.014	93%			
	80267 A		Standard	AAS		0.145	0.143	0.119	0.024	83%	96	0.56	1.50
	81011 G		Au Spike	AAS		0.145	0.159	0.135	0.024	85%	24	0.17	0.50
79150 H		Middle Arenite East 243		AAS	0.15	0.145	0.137	0.120	0.017	88%			
	81002 G		LeachWELL	ICP		0.145	0.137	0.120	0.017	88%	12	0.12	5.00
				DIBK-AAS		0.145	0.149	0.132	0.017	89%			
	80267 B		Standard	AAS		0.511	0.463	0.436	0.027	94%	96	0.43	1.00
	81011 H		Au Spike	AAS		0.511	1.074	1.035	0.039	96%	24	0.23	1.00
79150 I		Middle Arenite East 243		AAS	0.15	0.511	1.083	1.065	0.018	98%			
	81002 H		LeachWELL	ICP		0.511	1.164	1.146	0.018	98%	12	0.11	5.00
				DIBK-AAS		0.511	1.099	1.082	0.018	98%			

KCA Sample No.	KCA Test No.	Composite Domain	Test Type	Finish	Target p_{100} Size, mm	Head Average, g/t Au	Calculated Head, g/t Au	Extracted, g/t Au	Avg. Tails, g/t Au	Au Extracted, %	Leach Time, hours	Consumption NaCN, kg/t	Addition $Ca(OH)_2$, kg/t
	80275 C		Standard	AAS		0.682	0.633	0.607	0.026	96%	96	0.73	1.00
	81014 H	Middle	Au Spike	AAS		0.682	0.540	0.525	0.015	97%	24	0.32	1.00
79158 A		Arenite		AAS	0.15	0.682	0.884	0.860	0.024	97%			
	81005 H	East 243	LeachWELL	ICP		0.682	0.914	0.890	0.024	97%	12	0.62	5.00
				DIBK-AAS		0.682	1.066	1.042	0.024	98%			
	80267 C		Standard	AAS		0.387	0.263	0.235	0.027	90%	96	0.39	1.00
	81011 I	Lower	Au Spike	AAS		0.387	0.248	0.195	0.053	79%	24	0.24	1.00
79151 A		Arenite		AAS	0.15	0.387	0.311	0.285	0.026	92%			
	81002 I	East 243	LeachWELL	ICP		0.387	0.321	0.296	0.026	92%	12	0.11	5.00
				DIBK-AAS		0.387	0.329	0.303	0.026	92%			
	80267 D		Standard	AAS		0.251	0.296	0.170	0.127	57%	96	0.52	1.00
	81011 J	Lower	Au Spike	AAS		0.251	0.180	0.165	0.015	91%	24	0.36	1.00
79151 B		Arenite		AAS	0.15	0.251	0.169	0.150	0.019	89%			
	81002 J	East 243	LeachWELL	ICP		0.251	0.196	0.177	0.019	90%	12	0.12	5.00
				DIBK-AAS		0.251	0.172	0.153	0.019	89%			
	80268 A		Standard	AAS		0.528	0.393	0.338	0.056	86%	96	0.29	1.00
	81011 K	Lower	Au Spike	AAS		0.528	0.422	0.345	0.077	82%	24	0.14	1.00
79151 C		Arenite		AAS	0.15	0.528	0.447	0.420	0.027	94%			
	81002 K	East 243	LeachWELL	ICP		0.528	0.503	0.476	0.027	95%	12	0.15	5.00
				DIBK-AAS		0.528	0.471	0.444	0.027	94%			
	80268 B		Standard	AAS		0.228	0.278	0.235	0.043	85%	96	0.13	1.00
	81012 A	Lower	Au Spike	AAS		0.228	0.241	0.195	0.046	81%	24	0.17	1.00
79151 D		Arenite		AAS	0.15	0.228	0.227	0.210	0.017	92%			
	81003 A	East 243	LeachWELL	ICP		0.228	0.233	0.216	0.017	93%	12	0.09	5.00
				DIBK-AAS		0.228	0.203	0.186	0.017	92%			
	80268 C		Standard	AAS		0.144	0.341	0.300	0.041	88%	96	0.64	1.26
	81012 B	Lower	Au Spike	AAS		0.144	0.267	0.255	0.012	96%	24	0.26	1.26
79151 E		Arenite		AAS	0.15	0.144	0.239	0.225	0.014	94%			
	81003 B	East 243	LeachWELL	ICP		0.144	0.242	0.228	0.014	94%	12	0.24	5.00
				DIBK-AAS		0.144	0.218	0.204	0.014	94%			

KCA Sample No.	KCA Test No.	Composite Domain	Test Type	Finish	Target p_{100} Size, mm	Head Average, g/t Au	Calculated Head, g/t Au	Extracted, g/t Au	Avg. Tails, g/t Au	Au Extracted, %	Leach Time, hours	Consumption NaCN, kg/t	Addition $Ca(OH)_2$, kg/t
	80268 D		Standard	AAS		0.145	0.162	0.121	0.041	75%	96	0.81	1.26
	81012 C		Au Spike	AAS		0.145	0.202	0.150	0.052	74%	24	0.21	1.26
79152 A		Argilite West 313		AAS	0.15	0.145	0.139	0.120	0.019	86%			
	81003 C		LeachWELL	ICP		0.145	0.142	0.123	0.019	87%	12	0.12	5.00
				DIBK-AAS		0.145	0.122	0.104	0.019	85%			
	80269 A		Standard	AAS		0.015	0.043	0.009	0.034	20%	96	0.39	1.00
	81012 D		Au Spike	AAS		0.015	0.098	0.000	0.098	0%	24	0.23	1.00
79152 B		Argilite West 313		AAS	0.15	0.015	0.016	0.008	0.009	47%			
	81003 D		LeachWELL	ICP		0.015	0.009	0.001	0.009	8%	12	0.24	5.00
				DIBK-AAS		0.015	0.018	0.009	0.009	51%			
	80269 B		Standard	AAS		3.154	4.580	4.427	0.153	97%	96	0.28	1.00
	81012 E		Au Spike	AAS		3.154	5.709	5.580	0.129	98%	24	0.26	1.00
79152 C		Argilite West 313		AAS	0.15	3.154	5.925	5.850	0.075	99%			
	81003 E		LeachWELL	ICP		3.154	6.091	6.016	0.075	99%	12	0.18	5.00
				DIBK-AAS		3.154	5.631	5.556	0.075	99%			
	80269 C		Standard	AAS		0.683	0.576	0.305	0.271	53%	96	0.47	1.00
	81012 F		Au Spike	AAS		0.683	0.397	0.195	0.202	49%	24	0.38	1.00
79152 D		Argilite West 313		AAS	0.15	0.683	0.563	0.525	0.038	93%			
	81003 F		LeachWELL	ICP		0.683	0.489	0.452	0.038	92%	12	0.26	5.00
				DIBK-AAS		0.683	0.560	0.522	0.038	93%			
	80275 D		Standard	AAS		0.783	1.097	0.807	0.290	74%	96	0.90	1.00
	81014 I		Au Spike	AAS		0.783	0.598	0.375	0.223	63%	24	0.54	1.00
79158 B		Argilite West 313		AAS	0.15	0.783	0.643	0.600	0.043	93%			
	81005 I		LeachWELL	ICP		0.783	0.631	0.588	0.043	93%	12	0.56	5.00
				DIBK-AAS		0.783	0.619	0.576	0.043	93%			
	80269 D		Standard	AAS		0.126	0.098	0.066	0.033	67%	96	0.60	1.00
	81012 F		Au Spike	AAS		0.126	0.086	0.060	0.026	70%	24	0.38	1.00
79153 A		Argilite East 343		AAS	0.15	0.126	0.070	0.060	0.010	85%			
	81003 G		LeachWELL	ICP		0.126	0.082	0.072	0.010	88%	12	0.53	5.00
				DIBK-AAS		0.126	0.072	0.062	0.010	86%			

KCA Sample No.	KCA Test No.	Composite Domain	Test Type	Finish	Target p_{100} Size, mm	Head Average, g/t Au	Calculated Head, g/t Au	Extracted, g/t Au	Avg. Tails, g/t Au	Au Extracted, %	Leach Time, hours	Consumption NaCN, kg/t	Addition $Ca(OH)_2$, kg/t
	80270 A		Standard	AAS		0.041	0.024	0.009	0.015	36%	96	0.68	1.26
	81012 H		Au Spike	AAS		0.041	0.088	0.060	0.028	68%	24	0.26	1.26
79153 B		Argilite East 343		AAS	0.15	0.041	0.035	0.008	0.026	23%			
	81003 H		LeachWELL	ICP		0.041	0.035	0.009	0.026	26%	12	0.18	5.00
				DIBK-AAS		0.041	0.032	0.006	0.026	19%			
	80270 B		Standard	AAS		0.431	0.203	0.176	0.027	87%	96	0.59	1.00
	81012 I		Au Spike	AAS		0.431	0.368	0.315	0.053	86%	24	0.20	1.00
79153 C		Argilite East 343		AAS	0.15	0.431	0.257	0.240	0.017	93%			
	81003 I		LeachWELL	ICP		0.431	0.265	0.248	0.017	94%	12	0.16	5.00
				DIBK-AAS		0.431	0.237	0.220	0.017	93%			
	80270 C		Standard	AAS		0.300	0.310	0.127	0.183	41%	96	0.60	1.00
	81012 J		Au Spike	AAS		0.300	0.246	0.075	0.171	30%	24	0.35	1.00
79153 D		Argilite East 343		AAS	0.15	0.300	0.253	0.220	0.033	87%			
	81003 J		LeachWELL	ICP		0.300	0.295	0.262	0.033	89%	12	0.12	5.00
				DIBK-AAS		0.300	0.275	0.242	0.033	88%			
	80270 D		Standard	AAS		0.797	0.658	0.508	0.150	77%	96	0.82	1.00
	81012 K		Au Spike	AAS		0.797	0.830	0.645	0.185	78%	24	0.44	1.00
79153 E		Argilite East 343		AAS	0.15	0.797	1.148	1.100	0.048	96%			
	81003 K		LeachWELL	ICP		0.797	1.152	1.104	0.048	96%	12	0.08	5.00
				DIBK-AAS		0.797	1.192	1.144	0.048	96%			
	80271 A		Standard	AAS		0.144	0.163	0.035	0.129	21%	96	0.40	1.00
	81013 A		Au Spike	AAS		0.144	0.218	0.000	0.218	0%	24	0.17	1.00
79153 F		Argilite East 343		AAS	0.15	0.144	0.102	0.080	0.022	78%			
	81004 A		LeachWELL	ICP		0.144	0.090	0.068	0.022	75%	12	0.08	5.00
				DIBK-AAS		0.144	0.094	0.072	0.022	76%			
	80271 B		Standard	AAS		0.233	0.304	0.256	0.048	84%	96	0.85	1.26
	81013 B		Au Spike	AAS		0.233	0.246	0.195	0.051	79%	24	0.44	1.26
79153 G		Argilite East 343		AAS	0.15	0.233	0.275	0.260	0.015	94%			
	81004 B		LeachWELL	ICP		0.233	0.241	0.226	0.015	94%	12	0.12	5.00
				DIBK-AAS		0.233	0.263	0.248	0.015	94%			

KCA Sample No.	KCA Test No.	Composite Domain	Test Type	Finish	Target p_{100} Size, mm	Head Average, g/t Au	Calculated Head, g/t Au	Extracted, g/t Au	Avg. Tails, g/t Au	Au Extracted, %	Leach Time, hours	Consumption NaCN, kg/t	Addition $Ca(OH)_2$, kg/t
	80276 B		Standard	AAS		0.290	0.225	0.013	0.213	6%	96	0.83	1.00
	81014 K	Argilite East 343	Au Spike	AAS		0.290	0.405	0.000	0.405	0%	24	0.48	1.00
79158 D				AAS	0.15	0.290	0.321	0.240	0.081	75%			
	81005 K		LeachWELL	ICP		0.290	0.301	0.220	0.081	73%	12	0.52	5.00
				DIBK-AAS		0.290	0.297	0.216	0.081	73%			
	80271 C		Standard	AAS		0.986	2.387	2.260	0.127	95%	96	0.59	1.00
	81013 C	Arenite West 413	Au Spike	AAS		0.986	1.188	1.080	0.108	91%	24	0.29	1.00
79154 A				AAS	0.15	0.986	1.186	1.140	0.046	96%			
	81004 C		LeachWELL	ICP		0.986	1.201	1.155	0.046	96%	12	0.00	5.00
				DIBK-AAS		0.986	1.250	1.204	0.046	96%			
	80271 D		Standard	AAS		0.333	0.285	0.017	0.267	6%	96	0.26	1.00
	81013 D	Arenite West 413	Au Spike	AAS		0.333	0.456	0.000	0.456	0%	24	0.18	1.00
79154 B				AAS	0.15	0.333	0.300	0.240	0.060	80%			
	81004 D		LeachWELL	ICP		0.333	0.262	0.202	0.060	77%	12	0.14	5.00
				DIBK-AAS		0.333	0.278	0.218	0.060	78%			
	80272 A		Standard	AAS		0.070	0.027	0.010	0.017	36%	96	0.16	1.00
	81013 E	Arenite West 413	Au Spike	AAS		0.070	0.135	0.120	0.015	89%	24	0.14	1.00
79154 C				AAS	0.15	0.070	0.049	0.040	0.009	82%			
	81004 E		LeachWELL	ICP		0.070	0.077	0.068	0.009	89%	12	0.04	5.00
				DIBK-AAS		0.070	0.047	0.038	0.009	82%			
	80276 A		Standard	AAS		1.768	1.967	1.498	0.470	76%	96	0.88	1.00
	81014 J	Arenite West 413	Au Spike	AAS		1.768	1.668	1.200	0.468	72%	24	0.32	1.00
79158 C				AAS	0.15	1.768	1.512	1.380	0.132	91%			
	81005 J		LeachWELL	ICP		1.768	1.498	1.366	0.132	91%	12	0.58	5.00
				DIBK-AAS		1.768	1.550	1.418	0.132	91%			
	80272 B		Standard	AAS		0.315	0.255	0.228	0.027	89%	96	0.23	1.00
	81013 F	Arenite East 443	Au Spike	AAS		0.315	0.302	0.285	0.017	94%	24	0.27	1.00
79155 A				AAS	0.15	0.315	0.195	0.180	0.015	92%			
	81004 F		LeachWELL	ICP		0.315	0.255	0.240	0.015	94%	12	0.18	5.00
				DIBK-AAS		0.315	0.231	0.216	0.015	93%			

KCA Sample No.	KCA Test No.	Composite Domain	Test Type	Finish	Target p_{100} Size, mm	Head Average, g/t Au	Calculated Head, g/t Au	Extracted, g/t Au	Avg. Tails, g/t Au	Au Extracted, %	Leach Time, hours	Consumption NaCN, kg/t	Addition $Ca(OH)_2$, kg/t
	80272 C		Standard	AAS		0.091	0.086	0.069	0.017	80%	96	0.15	1.00
	81013 F	Arenite East 443	Au Spike	AAS		0.091	0.114	0.105	0.009	92%	24	0.30	1.00
79155 B				AAS	0.15	0.091	0.052	0.040	0.012	77%			
	81004 G		LeachWELL	ICP		0.091	0.100	0.088	0.012	88%	12	0.10	5.00
				DIBK-AAS		0.091	0.056	0.044	0.012	79%			
	80272 D		Standard	AAS		0.188	0.156	0.138	0.019	88%	96	0.43	1.00
	81013 H	Arenite East 443	Au Spike	AAS		0.188	0.279	0.270	0.009	97%	24	0.51	1.00
79155 C				AAS	0.15	0.188	0.130	0.120	0.010	92%			
	81004 H		LeachWELL	ICP		0.188	0.172	0.162	0.010	94%	12	0.22	5.00
				DIBK-AAS		0.188	0.136	0.126	0.010	92%			
	80273 A		Standard	AAS		0.354	0.172	0.138	0.034	80%	96	0.58	1.00
	81013 I	Arenite East 443	Au Spike	AAS		0.354	0.265	0.225	0.040	85%	24	0.50	1.00
79155 D				AAS	0.15	0.354	0.302	0.280	0.022	93%			
	81004 I		LeachWELL	ICP		0.354	0.340	0.318	0.022	93%	12	0.10	5.00
				DIBK-AAS		0.354	0.308	0.286	0.022	93%			
	80273 B		Standard	AAS		0.243	0.215	0.172	0.043	80%	96	0.59	1.00
	81013 J	Arenite East 443	Au Spike	AAS		0.243	0.290	0.225	0.065	78%	24	0.51	1.00
79155 E				AAS	0.15	0.243	1.191	1.080	0.111	91%			
	81004 J		LeachWELL	ICP		0.243	1.287	1.176	0.111	91%	12	0.14	5.00
				DIBK-AAS		0.243	1.213	1.102	0.111	91%			
	80273 C		Standard	AAS		0.628	2.687	2.486	0.201	93%	96	0.48	1.00
	81013 K	Arenite 443 & 413 Refractory	Au Spike	AAS		0.628	0.485	0.300	0.185	62%	24	0.41	1.00
79156 A				AAS	0.15	0.628	0.606	0.440	0.166	73%			
	81004 K		LeachWELL	ICP		0.628	0.618	0.452	0.166	73%	12	0.12	5.00
				DIBK-AAS		0.628	0.594	0.428	0.166	72%			
	80273 D		Standard	AAS		0.936	1.734	1.626	0.108	94%	96	0.98	1.00
	81014 A	Arenite 443 & 413 Refractory	Au Spike	AAS		0.936	1.072	0.990	0.082	92%	24	0.68	1.00
79156 B				AAS	0.15	0.936	1.225	1.160	0.065	95%			
	81005 A		LeachWELL	ICP		0.936	1.309	1.244	0.065	95%	12	0.14	5.00
				DIBK-AAS		0.936	1.275	1.210	0.065	95%			

KCA Sample No.	KCA Test No.	Composite Domain	Test Type	Finish	Target p_{100} Size, mm	Head Average, g/t Au	Calculated Head, g/t Au	Extracted, g/t Au	Avg. Tails, g/t Au	Au Extracted, %	Leach Time, hours	Consumption NaCN, kg/t	Addition $Ca(OH)_2$, kg/t
	80274 A		Standard	AAS		0.219	0.211	0.070	0.141	33%	96	0.68	1.00
	81014 B	Arenite 443	Au Spike	AAS		0.219	0.218	0.060	0.158	28%	24	0.32	1.26
79156 C		& 413		AAS	0.15	0.219	0.221	0.180	0.041	81%			
	81005 B	Refractory	LeachWELL	ICP		0.219	0.227	0.186	0.041	82%	12	0.52	5.00
				DIBK-AAS		0.219	0.207	0.166	0.041	80%			
	80274 B		Standard	AAS		0.144	0.102	0.053	0.049	52%	96	0.97	1.26
	81014 C	Arenite 443	Au Spike	AAS		0.144	0.131	0.045	0.086	34%	24	0.41	1.26
79156 D		& 413		AAS	0.15	0.144	0.159	0.120	0.039	75%			
	81005 C	Refractory	LeachWELL	ICP		0.144	0.166	0.127	0.039	76%	12	0.38	5.00
				DIBK-AAS		0.144	0.157	0.118	0.039	75%			
	80274 C		Standard	AAS		0.523	0.500	0.464	0.036	93%	96	0.75	1.00
	81014 D	Argilite	Au Spike	AAS		0.523	0.596	0.525	0.071	88%	24	0.48	1.00
79157 A		513, 543,		AAS	0.15	0.523	0.771	0.740	0.031	96%			
	81005 D	623, 643	LeachWELL	ICP		0.523	0.759	0.728	0.031	96%	12	0.56	5.00
				DIBK-AAS		0.523	0.791	0.760	0.031	96%			
	80274 D		Standard	AAS		0.648	0.827	0.762	0.065	92%	96	0.73	1.00
	81014 E	Argilite	Au Spike	AAS		0.648	1.053	0.990	0.063	94%	24	0.53	1.00
79157 B		513, 543,		AAS	0.15	0.648	0.872	0.840	0.032	96%			
	81005 E	623, 643	LeachWELL	ICP		0.648	0.854	0.822	0.032	96%	12	0.56	5.00
				DIBK-AAS		0.648	0.892	0.860	0.032	96%			
	80275 A		Standard	AAS		0.068	0.022	0.009	0.014	39%	96	0.48	1.50
	81014 F	Argilite	Au Spike	AAS		0.068	0.060	0.045	0.015	74%	24	0.57	1.50
79157 C		513, 543,		AAS	0.15	0.068	0.019	0.010	0.009	54%			
	81005 F	623, 643	LeachWELL	ICP		0.068	0.043	0.034	0.009	80%	12	0.54	5.00
				DIBK-AAS		0.068	0.021	0.012	0.009	58%			
	80275 B		Standard	AAS		0.034	0.084	0.067	0.017	80%	96	0.53	1.00
	81014 G	Argilite	Au Spike	AAS		0.034	0.074	0.060	0.014	81%	24	0.08	1.00
79157 D		513, 543,		AAS	0.15	0.034	0.037	0.020	0.017	54%			
	81005 G	623, 643	LeachWELL	ICP		0.034	0.057	0.040	0.017	70%	12	0.48	5.00
				DIBK-AAS		0.034	0.075	0.058	0.017	77%			

The standard variability bottle roll tests were compared to the standard composite tests in Table 13-23. The variability test had generally lower recoveries than the composite samples. The variability in the results may be due to the low grade and presence of coarse gold.

TABLE 13-23 VARIABILITY AND COMPOSITE BOTTLE ROLL COMPARISON

	Variability BRT				Composite BRT	
Composite Domain	Avg. Graphitic Carbon, %	Avg Preg Robbing, %	Avg. Calc. Head, g/t Au	Avg. Au Extraction, %	Avg. Calc. Head, g/t Au	Avg. 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%
Argilite West 313	0.252	17.800	1.291	64%	0.208	45%
Argilite 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%
Argilite 513, 543, 623, 643	<0.05	6.250	0.358	76%	0.376	89%

BRT – bottle roll leach test

13.5.3.7 AGGLOMERATION AND COMPACTED PERMEABILITY TEST WORK

Preliminary agglomeration test work as well as compacted permeability test work was conducted on portions of the conventional and 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 conventionally crushed (100% passing 8 millimeters) and HPGR crushed material. Separate test samples were loaded into a column and subjected to loads equivalent to 20 m, 40 m, and 60 m of overall heap height (assuming a heap density equivalent to 1.8 t/m^3).

The compacted permeability tests are summarized in Table 13-24.

TABLE 13-24 SUMMARY OF COMPACTED PERMEABILITY TESTS

KCA Sample No.	Sample Description	Crush Type	Calc. p_{80} Size, mm	Test Phase	Cement Added, kg/t	Effective Height, m	Flow Rate, L/h/m²	Flow Result Pass/Fail	Cum. Slump, % Slump	Slump Result Pass/ Fail	Overall Pass/ Fail
79107 C	PT6: Arenite	HPGR	9.27	Primary	4	20	3,579	Pass	3%	Pass	Pass
				Stage Load		40	1,833	Pass	9%	Pass	Pass
				Stage Load		60	840	Pass	12%	Fail	Fail
79118 A	PT6: Arenite	Conv.	3.39	Primary	4	20	3,870	Pass	1%	Pass	Pass
				Stage Load		40	2,887	Pass	6%	Pass	Pass
				Stage Load		60	1,546	Pass	9%	Pass	Pass
79108 C	PT16: Argilite Rock	HPGR	9.23	Primary	4	20	3,533	Pass	4%	Pass	Pass
				Stage Load		40	1,335	Pass	10%	Pass	Pass
				Stage Load		60	478	Pass	12%	Fail	Fail
79119 A	PT16: Argilite Rock	Conv.	3.47	Primary	4	20	3,898	Pass	1%	Pass	Pass
				Stage Load		40	2,826	Pass	5%	Pass	Pass
				Stage Load		60	1,604	Pass	8%	Pass	Pass
79136 B	Arenite West 223	Conv.	6.45	Primary	0	20	3,779	Pass	0%	Pass	Pass
				Stage Load		40	3,454	Pass	2%	Pass	Pass
				Stage Load		60	3,128	Pass	4%	Pass	Pass
79160 C	Arenite West 223	HPGR	9.08	Primary	0	20	2,004	Pass	4%	Pass	Pass
				Stage Load		40	1,650	Pass	6%	Pass	Pass
				Stage Load		60	1,418	Pass	7%	Pass	Pass
79137 C	Upper Arenite East 243	Conv.	6.87	Primary	0	20	3,970	Pass	1%	Pass	Pass
				Stage Load		40	3,885	Pass	3%	Pass	Pass
				Stage Load		60	3,768	Pass	4%	Pass	Pass
79161 C	Upper Arenite East 243	HPGR	9.87	Primary	0	20	2,306	Pass	4%	Pass	Pass
				Stage Load		40	1,892	Pass	7%	Pass	Pass
				Stage Load		60	1,525	Pass	9%	Pass	Pass
79138 C	Middle Arenite East 243	Conv.	6.12	Primary	0	20	3,862	Pass	0%	Pass	Pass
				Stage Load		40	3,724	Pass	2%	Pass	Pass
				Stage Load		60	3,592	Pass	3%	Pass	Pass
79162 C	Middle Arenite East 243	HPGR	9.58	Primary	0	20	2,566	Pass	5%	Pass	Pass
				Stage Load		40	2,000	Pass	6%	Pass	Pass
				Stage Load		60	1,621	Pass	7%	Pass	Pass
79139 C	Lower Arenite East 243	Conv.	6.21	Primary	0	20	3,928	Pass	0%	Pass	Pass
				Stage Load		40	3,807	Pass	2%	Pass	Pass
				Stage Load		60	3,706	Pass	4%	Pass	Pass
79163 C	Lower Arenite East 243	HPGR	8.79	Primary	0	20	982	Pass	4%	Pass	Pass
				Stage Load		40	905	Pass	6%	Pass	Pass
				Stage Load		60	741	Pass	8%	Pass	Pass

KCA Sample No.	Sample Description	Crush Type	Calc. p80 Size, mm	Test Phase	Cement Added, kg/t	Effective Height, m	Flow Rate, L/h/m²	Flow Result Pass/Fail	Cum. Slump, % Slump	Slump Result Pass/ Fail	Overall Pass/ Fail
79140 C	Argilite West 313	Conv.	6.22	Primary	0	20	3,750	Pass	1%	Pass	Pass
				Stage Load		40	3,500	Pass	3%	Pass	Pass
				Stage Load		60	3,192	Pass	5%	Pass	Pass
79164 C	Argilite West 313	HPGR	8.93	Primary	0	20	2,117	Pass	5%	Pass	Pass
				Stage Load		40	1,461	Pass	7%	Pass	Pass
				Stage Load		60	989	Pass	10%	Pass	Pass
79141 C	Argilite East 343	Conv.	5.73	Primary	0	20	2,344	Pass	0%	Pass	Pass
				Stage Load		40	1,792	Pass	2%	Pass	Pass
				Stage Load		60	1,562	Pass	4%	Pass	Pass
79165 C	Argilite East 343	HPGR	9.31	Primary	4	20	3,583	Pass	3%	Pass	Pass
				Stage Load		40	1,501	Pass	9%	Pass	Pass
				Stage Load		60	735	Pass	11%	Fail	Fail
79143 B	Arenite West 413	Conv.	6.26	Primary	0	20	3,566	Pass	0%	Pass	Pass
				Stage Load		40	3,332	Pass	3%	Pass	Pass
				Stage Load		60	3,119	Pass	4%	Pass	Pass
79166 C	Arenite West 413	HPGR	9.58	Primary	0	20	2,423	Pass	4%	Pass	Pass
				Stage Load		40	1,923	Pass	6%	Pass	Pass
				Stage Load		60	1,574	Pass	8%	Pass	Pass
79144 C	Arenite East 443	Conv.	6.05	Primary	0	20	3,268	Pass	1%	Pass	Pass
				Stage Load		40	2,954	Pass	2%	Pass	Pass
				Stage Load		60	2,606	Pass	4%	Pass	Pass
79167 C	Arenite East 443	HPGR	10.05	Primary	0	20	1,303	Pass	4%	Pass	Pass
				Stage Load		40	1,065	Pass	7%	Pass	Pass
				Stage Load		60	889	Pass	9%	Pass	Pass
79145 B	Arenite 443 & 413 Refractory	Conv.	6.24	Primary	0	20	2,923	Pass	1%	Pass	Pass
				Stage Load		40	2,395	Pass	3%	Pass	Pass
				Stage Load		60	2,122	Pass	4%	Pass	Pass
79168 C	Arenite 443 & 413 Refractory	HPGR	9.47	Primary	0	20	1,339	Pass	4%	Pass	Pass
				Stage Load		40	1,244	Pass	5%	Pass	Pass
				Stage Load		60	993	Pass	7%	Pass	Pass
79146 B	Argilite 513, 543, 623, 643	Conv.	6.14	Primary	0	20	3,363	Pass	0%	Pass	Pass
				Stage Load		40	3,122	Pass	2%	Pass	Pass
				Stage Load		60	2,859	Pass	4%	Pass	Pass
79169 C	Argilite 513, 543, 623, 643	HPGR	8.48	Primary	0	20	1,439	Pass	2%	Pass	Pass
				Stage Load		40	1,064	Pass	6%	Pass	Pass
				Stage Load		60	906	Pass	7%	Pass	Pass

All of the samples passed at 20 m and 40 m but three failed at 60 m (PT6: Arenite, PT16: Argilite Rock, and Argilite East 343). The PT6 and PT16 material passed compacted permeability tests equivalent to 50 m when conventionally crushed to 100% passing 8 mm without cement. A heap height of 50 m with no cement was selected for this study.

13.5.3.8 COLUMN LEACH TEST WORK

The PT6: Arenite and PT16: Argilite Rock/Graphite Faible samples were at KCA from the first round of column testing and new column leach 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 100% passing 4.75 mm and agglomerated with cement; and
- HPGR crushed to 100% passing 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 100% passing 8 mm and blended with hydrated lime,
- HPGR crushed to 100% passing 19 mm and blended with hydrated lime.

The column leach test results are presented in Tables 13-25 and 13-26.

TABLE 13-25 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: Argilite Rock/Graphite Faible	Conventional	0.731	78%	--	86	0.89	--	4.02
79108 C	79126	PT16: Argilite 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	Argilite West 313	Conventional	0.649	50%	6.39	61	0.42	1.25	0.00
79164 C	80242	Argilite West 313	HPGR	0.465	51%	9.59	61	0.41	1.23	0.00
79141 C	80210	Argilite East 343	Conventional	0.687	61%	5.99	61	0.32	1.77	0.00
79165 C	80245	Argilite 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	Argilite 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	Argilite 513, 543, 623, 643	Conventional	0.495	62%	6.08	61	0.26	1.50	0.00
79169 C	80257	Argilite 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 - Argilite	Conventional	0.467	53%	6.06	61	0.34	1.51	0.00
--	Average	Overall - Argilite	HPGR	0.756	60%	8.78	69	0.33	0.91	1.33

TABLE 13-26 SUMMARY OF COLUMN LEACH TEST RESULTS – SILVER

KCA Sample No.	KCA Test No.	Description	Crushing Type	Calculated Head, g/t Ag	Extracted, % Ag	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	0.46	14%	--	86	0.82	--	4.02
79107 C	79123	PT6: Arenite	HPGR	0.47	24%	--	86	0.94	--	4.05
79119 A	79132	PT16: Argilite Rock/Graphite Faible	Conventional	0.47	23%	--	86	0.89	--	4.02
79108 C	79126	PT16: Argilite Rock/Graphite Faible	HPGR	0.53	22%	--	86	0.89	--	4.07
--	Average	Essakane	Conventional	0.47	19%	--	86	0.855	--	4.02
--	Average	Essakane	HPGR	0.50	23%	--	86	0.915	--	4.06
79136 B	79187	Arenite West 223	Conventional	0.34	14%	6.30	61	0.37	1.19	0.00
79160 C	80230	Arenite West 223	HPGR	0.40	15%	10.30	61	0.36	1.26	0.00
79137 B	79190	Upper Arenite East 243	Conventional	0.35	23%	6.46	61	0.30	1.26	0.00
79161 C	80233	Upper Arenite East 243	HPGR	0.41	16%	10.38	61	0.37	1.25	0.00
79138 C	80201	Middle Arenite East 243	Conventional	0.37	26%	5.96	61	0.39	1.22	0.00
79162 C	80236	Middle Arenite East 243	HPGR	0.48	23%	9.92	61	0.37	1.25	0.00
79139 C	80204	Lower Arenite East 243	Conventional	0.31	21%	6.22	61	0.28	1.26	0.00
79163 C	80239	Lower Arenite East 243	HPGR	0.42	27%	9.63	61	0.41	1.26	0.00
79140 C	80207	Argilite West 313	Conventional	0.36	21%	6.36	61	0.42	1.25	0.00
79164 C	80242	Argilite West 313	HPGR	0.41	19%	10.49	61	0.41	1.23	0.00
79141 C	80210	Argilite East 343	Conventional	0.31	20%	5.97	61	0.32	1.77	0.00
79165 C	80245	Argilite East 343	HPGR	0.50	33%	9.54	85	0.33	0.00	4.00
79142 A	80213	Argilite East 343 Refractory	Conventional	0.24	22%	6.02	61	0.34	1.51	0.00
79143 B	80216	Arenite West 413	Conventional	0.38	17%	6.08	61	0.35	1.25	0.00

KCA Sample No.	KCA Test No.	Description	Crushing Type	Calculated Head, g/t Ag	Extracted, % Ag	Calculated Tail p_{80} Size, mm	Days of Leach	Consumption NaCN, kg/t	Addition Hydrated Lime, kg/t	Addition Cement, kg/t
79166 C	80248	Arenite West 413	HPGR	0.39	20%	10.63	61	0.38	1.25	0.00
79144 C	80219	Arenite East 443	Conventional	0.39	18%	6.10	61	0.32	1.25	0.00
79167 C	80251	Arenite East 443	HPGR	0.45	31%	10.89	85	0.64	1.26	0.00
79145 B	80222	Arenite 443 & 413 Refractory	Conventional	0.44	29%	6.08	61	0.36	1.50	0.00
79168 C	80254	Arenite 443 & 413 Refractory	HPGR	0.51	34%	9.29	85	0.66	1.51	0.00
79146 B	80225	Argilite 513, 543, 623, 643	Conventional	0.44	25%	6.17	61	0.26	1.50	0.00
79169 C	80257	Argilite 513, 543, 623, 643	HPGR	0.51	34%	9.22	62	0.26	1.50	0.00
--	--	Overall - Composite	Conventional	0.36	21%	6.16	61	0.34	1.36	0.00
--	--	Overall - Composite	HPGR	0.45	25%	10.03	68	0.42	1.18	0.40

At the conclusion of leaching, drain down tests were conducted on each column. The 24 h drain down results for the conventional columns varied from 28.3 L/t to 51.0 L/t with an average of 39.1 L/t. The 24 h drain down results for the HPGR columns varied from 15.4 L/t to 33.6 L/t with an average of 30.7 L/t. The drain down results are presented in Table 13-27.

The retained moisture for the conventional columns ranged from 45.9 L/t to 328.0 L/t with an average of 90.1 L/t. The retained moisture for the HPGR columns ranged from 67.5 L/t to 149.2 L/t with an average of 82.3 L/t. Cement addition had no noticeable effect on retained moisture. The retained moisture is presented in Table 13-28.

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 0.7% and 3.7% for the conventional and HPGR columns, respectively. The apparent bulk density averaged 1.55 t/m^3 and 1.64 t/m^3 for the conventional and HPGR columns, respectively. A bulk density for stacked ore of 1.60 t/m^3 was used for this study. The slump results are presented in Table 13-29.

Tailings screen analyses were completed on all of the columns. The results of screen analyses are shown in Figure 13-3.

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 13-30.

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. The results of the detoxification testing are presented in Table 13-31.

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. 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 plotted against the tonnes of solution applied per tonne of ore in the laboratory to determine where the curve transitions from a steep (rapid solution controlled leaching) curve to a flatter (slower time controlled leaching) curve. A heap has more tonnes per square metre of solution application than a column test and requires more time to reach the same tonnes of solution per tonne of ore for the solution controlled leaching. One laboratory day will equal one operational day for the remaining leach time. Based on the calculations described, the HPGR crushed material was assumed to have a 90 day leach time for this study. These calculations are presented in Table 13-32.

TABLE 13-27 SUMMARY OF DRAIN DOWN TEST RESULTS

KCA Sample No.	KCA Test No.	Description	Type	Sample Weight, kg	L water/$t_{dry\ ore}$			
					24 hour	48 hour	72 hour	96 hour
79118 A	79129	PT6: Arenite	Conventional	24.90	30.5	31.3	36.5	38.2
79107 C	79123	PT6: Arenite	HPGR	24.68	29.2	31.2	33.2	34.4
79119 A	79132	PT16: Argilite Rock/Graphite Faible	Conventional	24.90	33.3	36.1	38.2	40.2
79108 C	79126	PT16: Argilite Rock/Graphite Faible	HPGR	24.58	27.7	30.9	33.0	34.2
79136 B	79187	Arenite West 223	Conventional	55.76	28.3	34.1	37.1	
79160 C	80230	Arenite West 223	HPGR	48.56	31.9	37.3	41.4	44.3
79137 B	79190	Upper Arenite East 243	Conventional	59.39	38.9	43.9	46.3	48.2
79161 C	80233	Upper Arenite East 243	HPGR	48.45	32.6	37.8	41.9	43.3
79138 C	80201	Middle Arenite East 243	Conventional	51.70	37.9	43.5	46.4	48.5
79162 C	80236	Middle Arenite East 243	HPGR	48.84	31.5	37.1	40.7	42.2
79139 C	80204	Lower Arenite East 243	Conventional	57.70	40.2	45.6	48.9	
79163 C	80239	Lower Arenite East 243	HPGR	48.62	25.7	32.5	36.6	38.7
79140 C	80207	Argilite West 313	Conventional	49.88	40.3	45.5	47.9	49.7
79164 C	80242	Argilite West 313	HPGR	48.74	33.6	37.1	39.8	41.2
79141 C	80210	Argilite East 343	Conventional	50.48	50.7	57.1	60.0	
79165 C	80245	Argilite East 343	HPGR	48.80	15.4	17.6	19.3	20.9
79142 A	80213	Argilite East 343 Refractory	Conventional	54.60	35.9	41.4	43.8	46.7
79143 B	80216	Arenite West 413	Conventional	52.87	40.1	45.4	47.5	49.2
79166 C	80248	Arenite West 413	HPGR	48.62	30.6	37.6	41.1	43.0
79144 C	80219	Arenite East 443	Conventional	60.00	37.5	43.3	47.7	50.0
79167 C	80251	Arenite East 443	HPGR	48.16	21.4	27.0	29.5	
79145 B	80222	Arenite 443 & 413 Refractory	Conventional	44.69	51.0	58.4	61.8	
79168 C	80254	Arenite 443 & 413 Refractory	HPGR	48.32	38.9	45.1	48.0	51.9
79146 B	80225	Argilite 513, 543, 623, 643	Conventional	53.98	44.1	50.0	52.6	55.0
79169 C	80257	Argilite 513, 543, 623, 643	HPGR	48.62	32.9	40.5	44.4	47.9

TABLE 13-28 SUMMARY OF RETAINED MOISTURE

KCA Sample No.	KCA Test No.	Description	Type	Days Leached	Calculated Head p_{80} Size, mm	Retained Solution, $L/t_{dry\ ore}$
79118 A	79129	PT6: Arenite	Conventional	86	3.39	103.6
79107 C	79123	PT6: Arenite	HPGR	86	9.27	101.3
79119 A	79132	PT16: Argilite Rock/Graphite Faible	Conventional	86	3.47	105.2
79108 C	79126	PT16: Argilite Rock/Graphite Faible	HPGR	86	9.23	71.6
79136 B	79187	Arenite West 223	Conventional	61	6.45	59.0
79160 C	80230	Arenite West 223	HPGR	61	9.08	81.1
79137 B	79190	Upper Arenite East 243	Conventional	61	6.87	58.4
79161 C	80233	Upper Arenite East 243	HPGR	61	9.87	71.2
79138 C	80201	Middle Arenite East 243	Conventional	61	6.12	85.5
79162 C	80236	Middle Arenite East 243	HPGR	61	9.58	72.9
79139 C	80204	Lower Arenite East 243	Conventional	61	6.21	45.9
79163 C	80239	Lower Arenite East 243	HPGR	61	8.79	67.5
79140 C	80207	Argilite West 313	Conventional	61	6.22	57.3
79164 C	80242	Argilite West 313	HPGR	61	8.93	75.1
79141 C	80210	Argilite East 343	Conventional	61	5.73	67.8
79165 C	80245	Argilite East 343	HPGR	85	9.31	76.2
79142 A	80213	Argilite East 343 Refractory	Conventional	61	5.94	64.8
79143 B	80216	Arenite West 413	Conventional	61	6.26	67.1
79166 C	80248	Arenite West 413	HPGR	61	10.51	67.5
79144 C	80219	Arenite East 443	Conventional	61	6.05	63.7
79167 C	80251	Arenite East 443	HPGR	85	10.05	72.1
79145 B	80222	Arenite 443 & 413 Refractory	Conventional	61	6.24	328.0
79168 C	80254	Arenite 443 & 413 Refractory	HPGR	85	9.47	149.2
79146 B	80225	Argilite 513, 543, 623, 643	Conventional	61	6.14	64.8
79169 C	80257	Argilite 513, 543, 623, 643	HPGR	62	8.48	81.9

TABLE 13-29 PERCENT SLUMP AND FINAL APPARENT BULK DENSITY

KCA Sample No.	KCA Test No.	Description	Crush Type	Calculated Head p_{80} Size, mm	Initial Ht., m	Final Ht., m	Slump, %	Final Apparent Bulk Density, t_{dry}/m^3
79118 A	79129	PT6: Arenite	Conventional	3.39	2.467	2.394	3.0%	1.283
79107 C	79123	PT6: Arenite	HPGR	9.27	2.219	2.159	2.7%	1.410
79119 A	79132	PT16: Argilite Rock/Graphite Faible	Conventional	3.47	2.540	2.489	2.0%	1.234
79108 C	79126	PT16: Argilite Rock/Graphite Faible	HPGR	9.23	2.315	2.273	1.8%	1.334
79136 B	79187	Arenite West 223	Conventional	6.45	4.147	4.147	0.0%	1.659
79160 C	80230	Arenite West 223	HPGR	9.08	3.699	3.537	4.4%	1.693
79137 B	79190	Upper Arenite East 243	Conventional	6.87	4.661	4.661	0.0%	1.572
79161 C	80233	Upper Arenite East 243	HPGR	9.87	3.616	3.493	3.4%	1.711
79138 C	80201	Middle Arenite East 243	Conventional	6.12	3.762	3.762	0.0%	1.695
79162 C	80236	Middle Arenite East 243	HPGR	9.58	3.699	3.670	0.8%	1.641
79139 C	80204	Lower Arenite East 243	Conventional	6.21	4.455	4.455	0.0%	1.598
79163 C	80239	Lower Arenite East 243	HPGR	8.79	3.747	3.458	7.7%	1.734
79140 C	80207	Argilite West 313	Conventional	6.22	3.886	3.864	0.6%	1.592
79164 C	80242	Argilite West 313	HPGR	8.93	3.677	3.502	4.7%	1.717
79141 C	80210	Argilite East 343	Conventional	5.73	3.693	3.673	0.5%	1.695
79165 C	80245	Argilite East 343	HPGR	9.31	3.731	3.718	0.3%	1.619
79142 A	80213	Argilite East 343 Refractory	Conventional	5.94	4.134	4.102	0.8%	1.642
79143 B	80216	Arenite West 413	Conventional	6.26	4.121	4.099	0.5%	1.591
79166 C	80248	Arenite West 413	HPGR	10.51	3.642	3.508	3.7%	1.709
79144 C	80219	Arenite East 443	Conventional	6.05	4.512	4.499	0.3%	1.645
79167 C	80251	Arenite East 443	HPGR	10.05	3.727	3.543	4.9%	1.676
79145 B	80222	Arenite 443 & 413 Refractory	Conventional	6.24	4.251	4.201	1.2%	1.312
79168 C	80254	Arenite 443 & 413 Refractory	HPGR	9.47	3.693	3.489	5.5%	1.708
79146 B	80225	Argilite 513, 543, 623, 643	Conventional	6.14	4.112	4.112	0.0%	1.619
79169 C	80257	Argilite 513, 543, 623, 643	HPGR	8.48	3.616	3.454	4.5%	1.736

FIGURE 13-3 TAIL SCREEN ANALYSIS

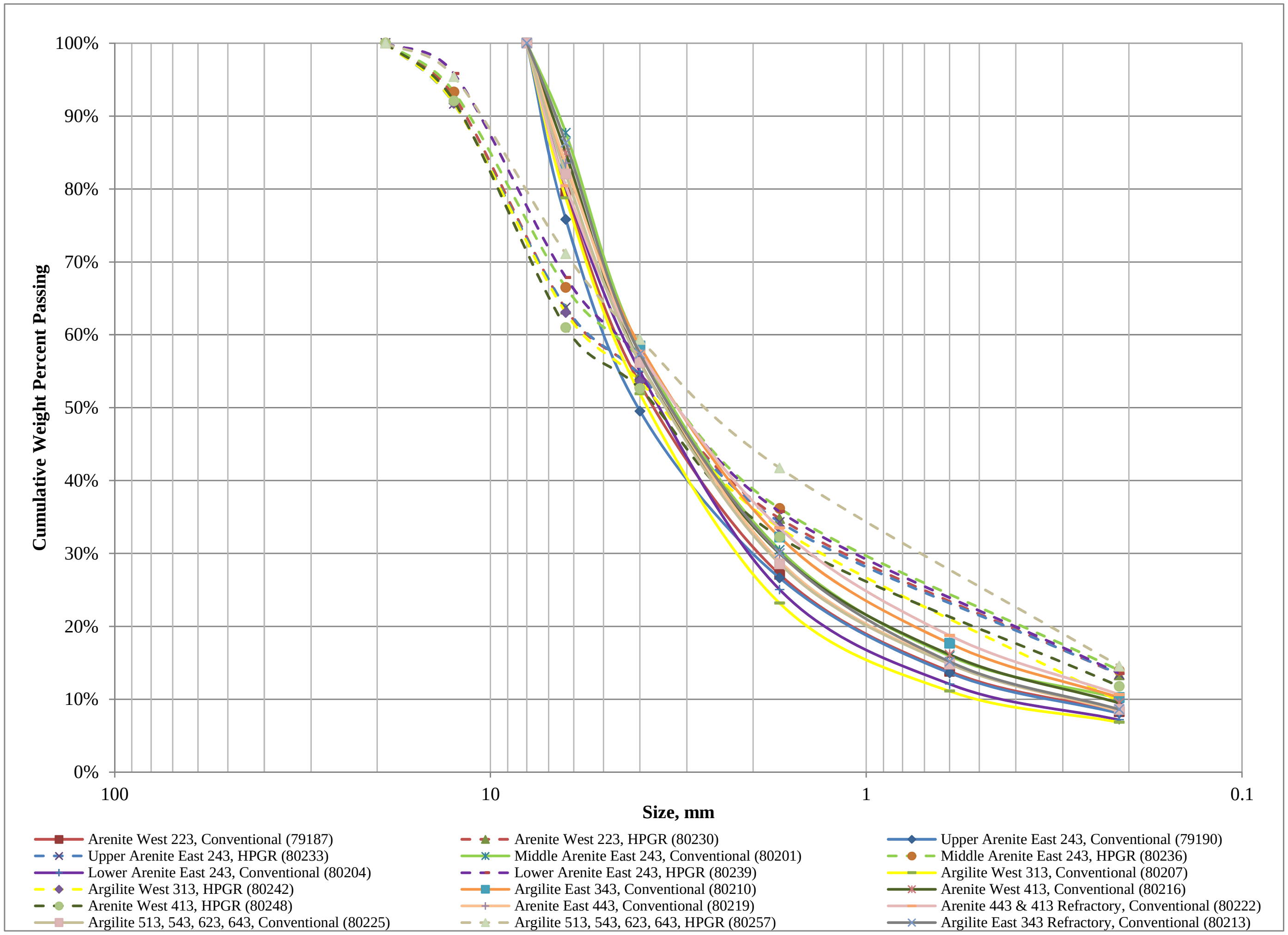


TABLE 13-30 HEAD VS. TAILS RECOVERY BY SIZE FRACTION

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

1: Note any negative recoveries were replaced with 0%

TABLE 13-31 MWMT PROFILE I ANALYSIS – WETLABS

Profile I Wet Chemistry	Units	Reporting Limit	Arenite West 223 KCA Sample No. 79136 B KCA Test No. 79187 Final Barren	Final Wash	MWMT	Lower Arenite East 243 KCA Sample No. 79139 C KCA Test No. 80204 Final Barren	Final Wash	MWMT	Argilite East 343 KCA Sample No. 79141 C KCA Sample No. 80210 Final Barren	Final Wash	MWMT	Argilite 513, 543, 623, 643 KCA Sample No. 79145 B KCA Test No. 80222 Final Barren	Final Wash	MWMT	Drinking Water Regulations US EPA
Alkalinity, Total	mg/L as $CaCO_3$	1.0	1600	120	42	1600	100	42	1200	160	45	1400	100	42	--
Bicarbonate (HCO_3)	mg/L as $CaCO_3$	1.0	110	110	42	44	96	42	ND	120	45	ND	92	42	--
Carbonate (CO_3)	mg/L as $CaCO_3$	1.0	1500	13	ND	1500	5	ND	920	40	ND	1300	12	ND	--
Hydroxide (OH)	mg/L as $CaCO_3$	1.0	ND	ND	ND	ND	ND	ND	240	ND	ND	120	ND	ND	--
Cyanide (Total)	mg/L	0.010	130	0.3	--	180	0.14	--	250	0.34	--	280	0.17	--	--
Cyanide (WAD)	mg/L	0.010	77	0.27	ND	130	0.03	ND	190	0.016	ND	190	0.018	ND	--
Total Nitrogen	Calc.	--	150	1.4	ND	220	1.7	ND	210	1.5	ND	220	1.3	ND	10
Nitrate + Nitrite Nitrogen	mg/L	0.10	1.40	ND	ND	1.60	ND	ND	1.4	ND	ND	1.1	ND	ND	--
Total Kjeldahl Nitrogen	mg/L	--	150	1.4	ND	210	1.6	ND	210	1.5	ND	220	1.3	ND	--
Extraction Fluid (Feed) pH	pH Units	--	--	--	6.32	--	--	6.32	--	--	6.32	--	--	6.32	--
MWMT Effluent pH	pH Units	--	--	--	8.59	--	--	8.61	--	--	8.75	--	--	8.67	6.5-8.5
pH	pH Units	--	10.22	8.83	8.28	10.46	8.46	8.18	10.66	9.35	8.14	10.60	8.87	8.12	--
pH - Temperature	ºC	--	22	24	21	22	24	21	23	24	22	23	24	22	--
Total Dissolved Solids	mg/L	10	2400	180	36	2600	200	38	2100	310	33	2300	190	33	500
Aluminum	mg/L	0.045	1.0	1.1	0.29	4.2	0.71	0.3	40.0	1.4	0.2	21.0	0.94	0.3	0.2 - 0.05
Antimony	mg/L	0.0040	0.33	0.026	ND	0.30	0.024	ND	0.13	0.074	0.0027	0.22	0.042	ND	0.006
Arsenic	mg/L	0.0050	12	1.0	0.15	140	5.6	0.53	31	4.7	0.39	51	5.0	0.43	0.010
Barium	mg/L	0.010	0.011	0.022	ND	ND	0.023	0.021	ND	0.032	ND	ND	0.020	ND	2
Beryllium	mg/L	0.0010	ND	ND	ND	ND	ND	ND	ND	ND	ND	ND	ND	ND	0.004
Cadmium	mg/L	0.0010	0.0027	ND	ND	0.0018	ND	ND	ND	ND	ND	0.0022	ND	ND	0.005
Calcium	mg/L	0.50	1.30	2.70	9.8	1.10	9.9	13.0	1.2	3.2	8.9	1.0	2.8	11.0	--
Chloride	mg/L	1.0	42	16	1.1	38	10	1.8	37	11	1.1	35.0	10.0	1.5	250
Chromium	mg/L	0.0050	0.016	ND	ND	0.031	ND	ND	0.030	ND	ND	0.028	ND	ND	0.1
Copper	mg/L	0.040	6.4	ND	ND	17	ND	ND	22	ND	ND	18	ND	ND	1.3
Fluoride	mg/L	0.10	1.3	ND	ND	1.1	ND	ND	1.3	ND	ND	ND	ND	ND	4.0
Iron	mg/L	0.20	19	0.20	ND	24.00	0.073	ND	13	0.95	ND	24	0.21	ND	0.3

Profile I Wet Chemistry	Units	Reporting Limit	Arenite West 223 KCA Sample No. 79136 B KCA Test No. 79187 Final Barren	Final Wash	MWMT	Lower Arenite East 243 KCA Sample No. 79139 C KCA Test No. 80204 Final Barren	Final Wash	MWMT	Argilite East 343 KCA Sample No. 79141 C KCA Sample No. 80210 Final Barren	Final Wash	MWMT	Argilite 513, 543, 623, 643 KCA Sample No. 79145 B KCA Test No. 80222 Final Barren	Final Wash	MWMT	Drinking Water Regulations US EPA
Lead	mg/L	0.0025	0.0058	0.0073	ND	ND	0.021	ND	ND	0.011	ND	ND	0.0063	ND	--
Magnesium	mg/L	0.50	ND	ND	2.2	ND	ND	2.7	ND	ND	2.4	ND	ND	2.8	--
Manganese	mg/L	0.0050	0.018	ND	0.026	ND	ND	0.018	ND	0.014	0.0074	ND	ND	0.018	--
Mercury	mg/L	0.0001	0.00014	ND	ND	0.00013	ND	ND	ND	ND	ND	ND	ND	ND	0.05
Nickel	mg/L	0.030	0.095	ND	ND	0.13	ND	ND	0.22	ND	ND	0.21	ND	ND	--
Potassium	mg/L	1.0	98.0	18	5.3	130.0	28.0	4.3	69	19	11	110	25	6.9	--
Selenium	mg/L	0.0050	ND	ND	ND	ND	ND	ND	ND	ND	ND	ND	ND	ND	0.05
Silver	mg/L	0.0050	ND	ND	ND	ND	ND	ND	ND	ND	ND	ND	ND	ND	0.10
Sodium	mg/L	0.50	920	53	4.1	1000	35	3.9	750	82	5.4	910	53	3.8	--
Sulphate Sulphur	mg/L	0.010	120	17	7	300.000	19	12	210	11	10	210	8.3	11	--
Thallium	mg/L	0.0010	ND	ND	ND	ND	ND	ND	ND	ND	ND	ND	ND	ND	0.002
Zinc	mg/L	0.020	1.0	ND	ND	1.5	ND	ND	0.94	ND	ND	0.94	ND	ND	5

Note. US EPA – United States Environmental Protection Agency

TABLE 13-32 LEACH TIME CALCULATIONS

KCA Sample No.	KCA Test No.	Description	Crush Type	Crush Size, mm	Transition, ts/to	Transition Recovery Au, %	Transition Lab Days	Transition Field Days	Total Lab Leach Days	Total Field Leach Days	Ultimate Lab Au Recovery, %	Field Deduction, %	Estd. Field Au Recovery, %
79136 B	79187	Arenite West 223	Conv.	8	0.52	69%	20	35	61	76	79%	2%	77%
79160 C	80230	Arenite West 223	HPGR	19	0.32	48%	11	21	61	71	54%	2%	52%
79137 C	79190	Upper Arenite East 243	Conv	8	0.31	45%	12	21	61	70	59%	2%	57%
79161 C	80233	Upper Arenite East 243	HPGR	19	0.32	45%	8	21	61	74	56%	2%	54%
79138 C	80201	Middle Arenite East 243	Conv	8	0.52	51%	19	35	61	77	64%	2%	62%
79162 C	80236	Middle Arenite East 243	HPGR	19	0.53	62%	15	35	61	81	71%	2%	69%

KCA Sample No.	KCA Test No.	Description	Crush Type	Crush Size, mm	Transition, ts/to	Transition Recovery Au, %	Transition Lab Days	Transition Field Days	Total Lab Leach Days	Total Field Leach Days	Ultimate Lab Au Recovery, %	Field Deduction, %	Estd. Field Au Recovery, %
79139 C	80204	Lower Arenite East 243	Conv	8	0.56	52%	12	37	61	86	65%	2%	63%
79163 C	80239	Lower Arenite East 243	HPGR	19	0.48	48%	14	32	61	79	60%	2%	58%
79140 C	80207	Argilite West 313	Conv	8	0.65	35%	20	43	61	84	49%	2%	47%
79164 C	80242	Argilite West 313	HPGR	19	0.59	39%	19	39	61	81	50%	2%	48%
79141 C	80210	Argilite East 343	Conv	8	0.56	46%	19	37	61	79	59%	2%	57%
79165 C	80245	Argilite East 343	HPGR	19	0.8	27%	25	53	85	113	46%	2%	44%
79142 A	80213	Argilite East 343 Refractory	Conv	8	0.46	29%	18	31	61	74	32%	2%	30%
79143 B	80216	Arenite West 413	Conv	8	0.39	29%	13	26	61	74	39%	2%	37%
79166 C	80248	Arenite West 413	HPGR	19	0.47	47%	13	31	61	79	54%	2%	52%
79144 C	80219	Arenite East 443	Conv	8	0.62	51%	25	41	61	77	66%	2%	64%
79167 C	80251	Arenite East 443	HPGR	19	0.45	40%	13	30	85	102	59%	2%	57%
79145 B	80222	Arenite 443 & 413 Refractory	Conv	8	0.9	57%	33	60	61	88	71%	2%	69%
79168 C	80254	Arenite 443 & 413 Refractory	HPGR	19	0.88	44%	27	59	85	117	59%	2%	57%
79146 B	80225	Argilite 513, 543, 623, 643	Conv	8	0.57	46%	20	38	61	79	60%	2%	58%
79169 C	80257	Argilite 513, 543, 623, 643	HPGR	19	0.76	72%	25	51	62	88	83%	2%	81%
		Conv Arenite Avg.			0.55	51%	19	36	61	78	63%	2%	61%
		Conv Argilite Avg.			0.56	39%	19	37	61	79	50%	2%	48%
		Conv Avg.			0.55	46%	19	37	61	79	58%	2%	56%
		HPGR Arenite Avg.			0.49	48%	14	33	68	86	59%	2%	57%
		HPGR Argilite Avg.			0.72	46%	23	48	69	94	60%	2%	58%
		HPGR Avg.			0.56	47%	17	37	68	89	59%	2%	57%

13.6 HEAP LEACH GOLD RECOVERY

The comparison between conventional crushing and HPGR crushing of Essakane ore does not show a clear benefit of one processing method over the other. The HPGR tertiary crusher has operational advantages over cone crushers.

The leach cycle, heap recovery, and reagent consumptions used in this study are based on the HPGR column leach tests completed in the May 2018 KCA program. The PT6 and PT16 samples were not included in these calculations because they are bulk samples from the pit and the core samples should be more representative. The HPGR column leach tests for the PT6 and PT16 samples were also conducted at different crush sizes. KCA typically discounts the gold recovery in column tests by 2% to 3% when estimating field recoveries and a discount of 2% for gold was used for Essakane due to the high number of column tests completed. The silver was discounted by 3% to estimate field recoveries. A summary of the column tests utilized for recovery calculations and the adjusted field recoveries are presented in Table 13-33.

TABLE 13-33 COLUMN TEST FIELD RECOVERY DISCOUNTS

KCA Test No	Rock Type	Lithology	Column Au Rec.	Field Au Rec.	Column Ag Rec.	Field Ag Rec.
80230	Arenite West	223	56%	54%	15%	12%
80233	Upper Arenite East	243	56%	54%	16%	13%
80236	Middle Arenite East	243	72%	70%	23%	20%
80239	Lower Arenite East	243	62%	60%	27%	24%
80242	Argilite West	313	51%	49%	19%	16%
80245	Argilite East	343	47%	45%	33%	30%
80248	Arenite West	413	54%	52%	20%	17%
80251	Arenite East	443	61%	59%	31%	28%
80254	Arenite Refractory	443 and 413	60%	58%	34%	31%
80257	Argilite	513, 543, 623, 643	83%	81%	21%	18%
	Arenite Average		60%	58%	24%	21%
	Argilite Average		60%	58%	24%	21%
	Arenite 443 and 413 Avg.		58%	56%	28%	25%
	Arenite 243 Average		63%	61%	22%	19%
	Overall Average		60%	58%	24%	21%

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 Argilite 123 and Turbidite. All of the column leach tests were completed on material from the EMZ pit, so the stockpiles are not represented and the effects of stockpiling are unknown. There are 7.45 Mt of stockpile material and 3.54 Mt of Turbidite from the pit, which comprises approximately 18% of the Mineral Reserve, that are not directly represented in the test work.

The estimated field recovery for gold is 55% and for silver it is 22%. The gold recovery calculations are presented in Table 13-34.

With mostly clean non-reactive ores, cyanide consumption in production heaps is typically 25% to 33% of the consumption from two metre tall laboratory column tests. The majority of the column tests on Essakane ore were conducted in four metre tall columns and a cyanide consumption factor of 80% was used. The HPGR column tests on the samples representing the orebody had a low cyanide consumption at 0.33 kg/t.

Lime requirements are the same in the field as in column tests and the average lime addition rate was 1.30 kg/t.

TABLE 13-34 GOLD RECOVERY CALCULATION

Rock Type	Lithology	Pit Tonnes	Estimated Stockpile Tonnes	Estimated Total Tonnes	Estimated Au Grade, g/t	Pit Lithology	Pit Au, kg	Pit Au Rec.	Stockpile Lithology	Stockpile Au, kg	Stockpile Au Rec.	Total Lithology	Total Au, kg	Total Au Rec.
Argilite	10203	43,098	0	43,098	0.33	Argilite Avg.	18	58%	Argilite Avg.	14	58%	Argilite Avg.	32	58%
Argilite	1023	11,909	41,477	53,386	0.32									
Argilite	1003	0	629	629	0.43									
Argilite	3103	5,995,585	146,990	6,142,575	0.43	313	2,622	49%	313	63	49%	313	2,685	49%
Argilite	313	111,837	0	111,837	0.41									
Argilite	3403	6,094,407	3,011,353	9,105,760	0.43	343	4,011	45%	343	1,435	45%	343	5,446	45%
Argilite	343	3,231,765	306,399	3,538,165	0.42									
Argilite	5103	626,352	0	626,352	0.46	513, 543, 623, 643	829	81%	513, 543, 623, 643	0	81%	513, 543, 623, 643	829	81%
Argilite	5403	1,095,191	0	1,095,191	0.45									
Argilite	543	38,448	0	38,448	0.46									
Argilite	6203	55,338	0	55,338	0.41									
Argilite	6403	42,466	0	42,466	0.35									
Arenite	2202	0	655	655	0.43	Arenite Avg.	0	58%	Arenite Avg.	0.3	58%	Arenite Avg.	0.3	58%
Arenite	4103	3,395,885	576	3,396,461	0.43	413 and 443 Avg.	4,900	56%	413 and 443 Avg.	80	56%	413 and 443 Avg.	4,980	56%
Arenite	4403	7,666,842	182,376	7,849,217	0.43									
Arenite	443	347,659	2,834	350,493	0.45									
Arenite	2203	6,343,656	893,789	7,237,445	0.42	223	2,766	54%	223	372	54%	223	3,138	54%
Arenite	223	303,904	1,208	305,112	0.43									
Arenite	2403	4,816,342	1,347,141	6,163,483	0.44	243 Avg.	6,542	61%	243 Avg.	930	61%	243 Avg.	7,472	61%
Arenite	243	10,278,793	790,990	11,069,783	0.43									
Turbidite	10303	794,912	264,301	1,059,213	0.40	Argilite Weighted Avg.	1,506	49%	Argilite Weighted Avg.	301	49%	Argilite Weighted Avg.	1,807	49%
Turbidite	1033	272,256	8,031	280,287	0.39									
Turbidite	10403	1,686,628	101,343	1,787,971	0.44									
Turbidite	1043	785,612	348,603	1,134,215	0.42									
Argilite Total		**17,346,395**	**3,506,848**	**20,853,243**	**0.431**		**7,480**	**50%**		**1,512**	**45%**		**8,992**	**49%**
Arenite Total		**33,153,081**	**3,219,569**	**36,372,649**	**0.429**		**14,208**	**58%**		**1,382**	**59%**		**15,590**	**58%**
Turbidite Total		**3,539,407**	**722,279**	**4,261,686**	**0.417**		**1,506**	**49%**		**301**	**49%**		**1,807**	**49%**
Total		**54,038,883**	**7,448,696**	**61,487,579**	**0.429**		**23,194**	**55%**		**3,195**	**52%**		**26,389**	**55%**

13.6.1 CONCLUSION FOR OVERALL ORES

The results of the metallurgical test programs indicate that the ore types tested are amenable to standard heap leaching methods. There is variability in the results and not all rock types have been tested, therefore additional test work is planned. However, the available test results are more than sufficient to support a pre-feasibility study.

A summary of the design parameters based on these test programs is as follows:

- HPGR product size: P_{100} of 19 mm (P_{80} of 8 mm);
- No agglomeration required for stacking up to 50 m;
- Leach Time: 90 days;
- Gold Recovery: 55%;
- Silver Recovery: 22%;
- Sodium Cyanide Consumption: 0.33 kg/tonne; and
- Lime Consumption: 1.30 kg/tonne.

14 MINERAL RESOURCE ESTIMATE

14.1 SUMMARY

The resource estimation methodologies, results, and validations are presented in this section.

The Mineral Resource estimate 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 QP, the resource evaluation reported herein is a reasonable representation of the Mineral Resources delineated at the Essakane Gold Mine as of June 5, 2018.

The Mineral Resource estimate at June 5, 2018 for the Essakane Gold Mine is summarized in Table 14-1 and is reported on a 100% basis. The Mineral Resource estimate is inclusive of Mineral Reserves.

The 0.30 g/t Au heap leach cut-off grade for fresh rock at EMZ is lower than the fresh rock cut-off grade at Falagountou because the Falagountou material is not considered for treatment at the heap leach facility.

Total Indicated Mineral Resources at the Essakane Gold Mine are currently estimated to be 160 Mt grading 0.95 g/t Au, totalling 4,878 koz of gold, while Inferred Mineral Resources are estimated to be 21.0 Mt grading 0.88 g/t Au, totalling 589 koz of gold. IAMGOLD's attributable Mineral Resources are 144 Mt totalling 4,390 koz of gold in Indicated Mineral Resources and 18.7 Mt totalling 530 koz of gold in Inferred Mineral Resources.

The 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.

TABLE 14-1 MINERAL RESOURCE SUMMARY – JUNE 5, 2018

Category	Tonnage (000 t)	Grade (g/t Au)	Contained Metal (000 oz Au)
Measured	-	-	-
Indicated	159,810	0.95	4,878
Total Measured + Indicated	**159,810**	**0.95**	**4,878**
Inferred	20,744	0.88	589

Notes:
1. CIM (2014) definitions were followed for Mineral Resources.
2. Mineral Resources for the EMZ reported at a cut-off grade of 0.33 g/t Au for saprolite, 0.43 g/t Au for transition material, and 0.30 g/t Au for fresh rock material. Cut-off grades for Falagountou are 0.36 g/t Au for saprolite, 0.46 g/t Au for transition material, and 0.52 g/t Au for fresh rock material.
3. Mineral Resources are constrained within a pit shell estimated using a long-term gold price of $1,500/oz and a US$/€ exchange rate of: 1:0.77 and a US$/CFA exchange rate of 1:0.00198.
4. Mineral Resources are inclusive of Mineral Reserves.
5. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
6. Mineral Resources are reported on a 100% basis.
7. Numbers may not add due to rounding.

14.2 EMZ DEPOSIT

14.2.1 DATA

The resource estimation used the results of three types of holes: 1) DD, 2) RC, and 3) holes starting in RC and ending in DD (RCD), for a total of 3,224 holes and 538,121 m drilled. Because AC and RAB sampling is more subject to segregation bias, their assay results have not been used in the estimate.

Table 14-2 details the holes included in the resource database by type, year of drilling, and series.

TABLE 14-2 RESOURCE DATABASE SUMMARY

Hole Type	Year	Series	No. of Holes	Metres Drilled
DDH	2018	MEDD0610-MEDD0627	17	3,254
		MEDD0483-MEDD0609	127	26,308
	2017	MGMT0006 - MGMT0007	2	300
	2016	MGMT001-MGMT006	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-MERC0577	18	2,362
	2017	MERC0489-MERC0557	68	4,910
	2016	MERC0384-MERC0488	105	13,654.00
	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	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	**3,247**	**538,121**

The current resource estimate includes a series of new holes. A total of 233 holes (DD, RC and RCD) for 37,762 m drilled since the last Mineral Resource estimate were added to the

EMZ resource database. Figure 14-1 shows the location of all of the drill holes available in the database as of the effective date of the Mineral Resource estimate (on the left side), as well as the location of the new drilling (right).

14.2.2 ASSAYS

The February 2018 assay database, used in the current resource update, consists of 379,326 records including 332,602 assay results above gold limit detection with an average sample length of 1.16 m, representing 430,103 assayed metres. Approximately 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.

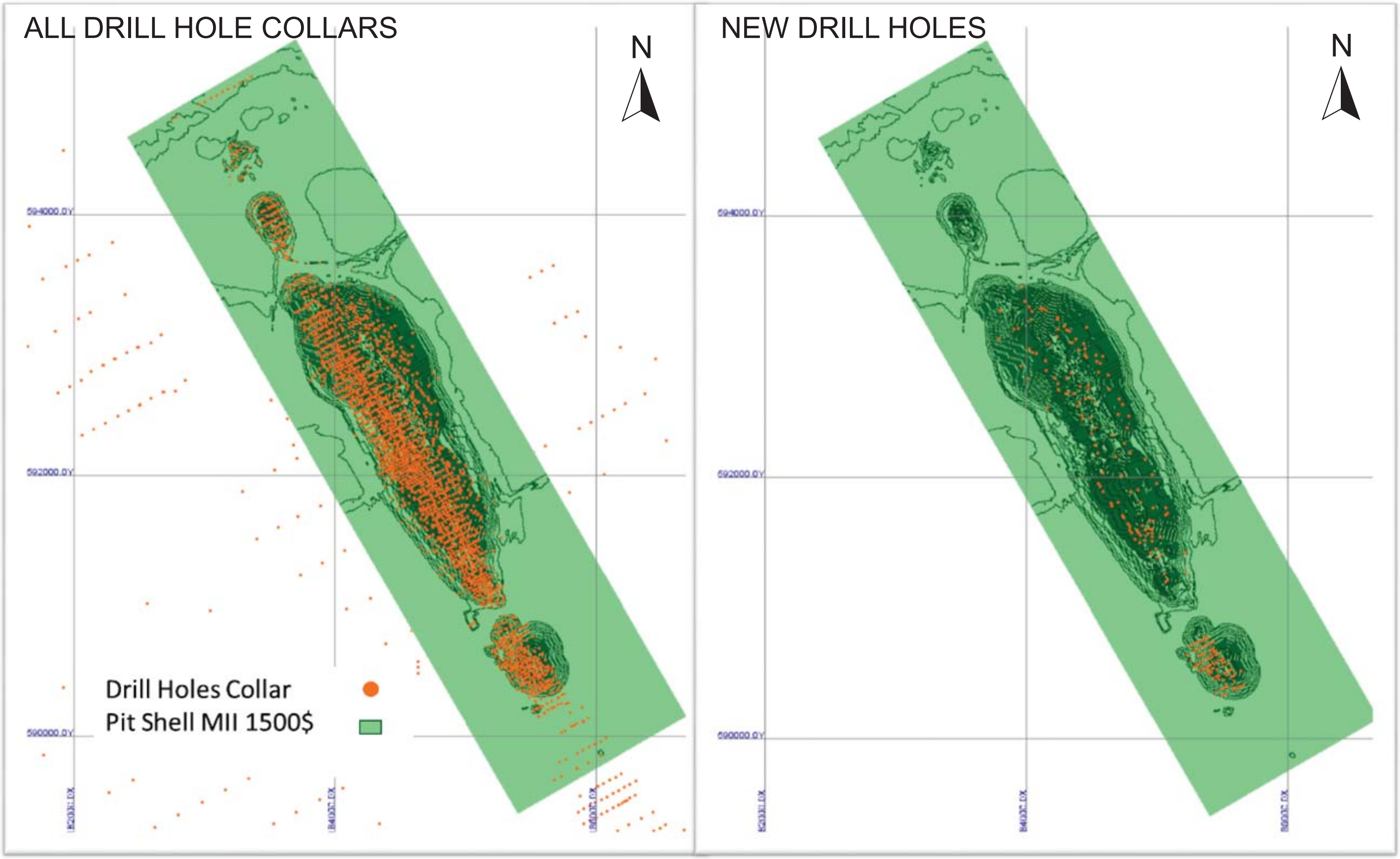
Gold grades vary from 0.0005 g/t Au to 430.0 g/t Au with an average of 0.45 g/t Au.

Note that in the EMZ deposit, a total of 187 holes have not been assayed, including abandoned holes, holes invalidated by the qualified person 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.

14.2.3 DRILL HOLE SPACING

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

1. North Satellite
2. EMZ
3. EMZ South



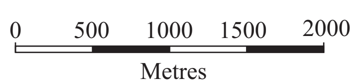


Figure 14-1

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Plan Views Showing Drill Hole Collars and New Drill Holes

July 2018

Essakane S.A., 2018.

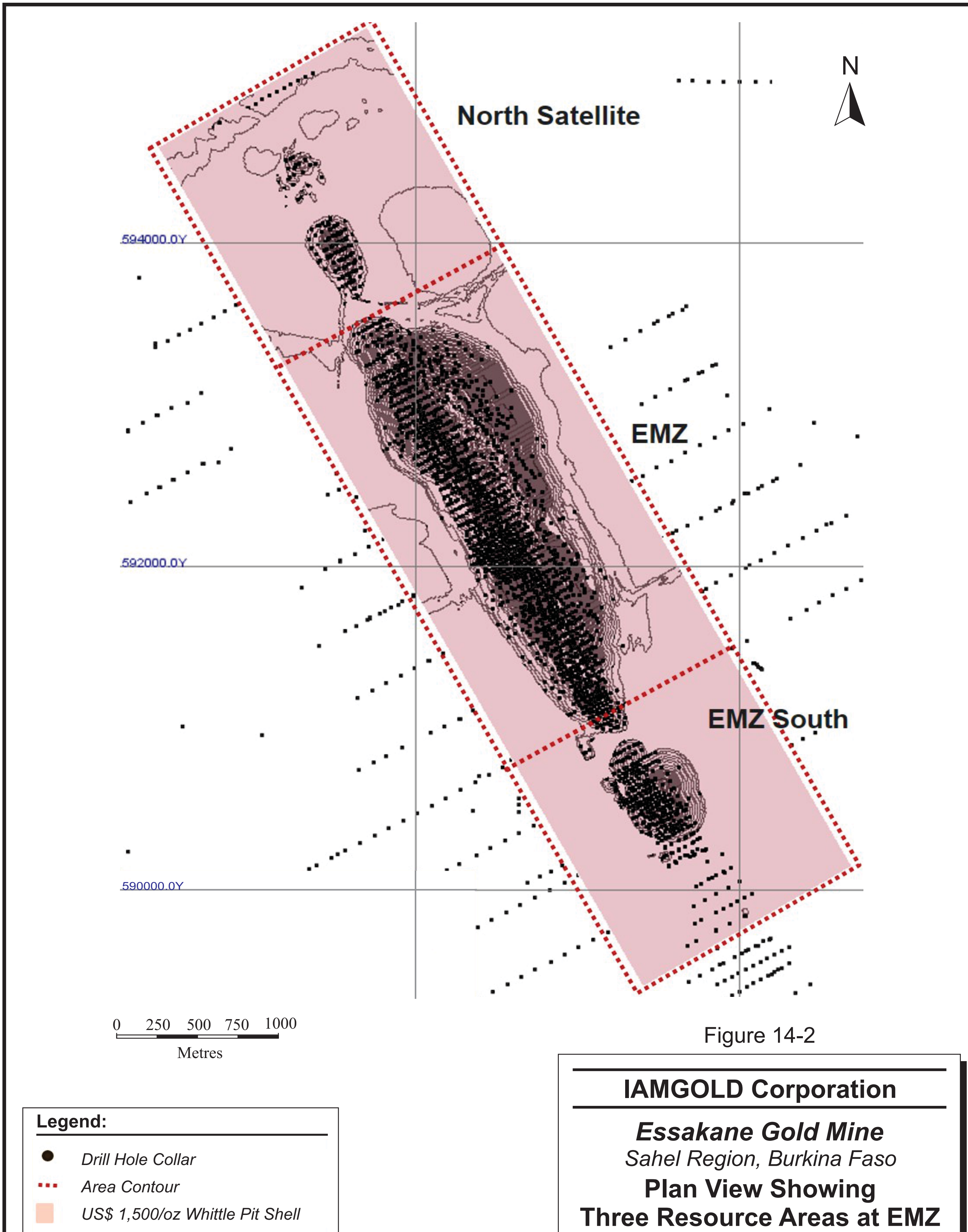


Figure 14-2

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Plan View Showing Three Resource Areas at EMZ

July 2018

Essakane S.A., 2018.

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.

14.2.4 MODELLING

The modelling work was performed by Essakane S.A. personnel. The last update on wireframes was performed at the end of January 2018. New drilling information showed a good correspondence with the actual model. The modelling was carried out using GEOVIA GEMS 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, Litho-Structural, and Topography elements are discussed in detail in the following sub-sections.

TABLE 14-3 SURFACES AND SOLIDS USED FOR THE 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/Fev18/DVR_18
6	240	Main Arenite, E Flank	3	DVR_Solide/243/Fev18/DVR_18
7	310	Footwall Argillite W Flank, lower thrust	9	DVR_Solide/313/Fev18/DVR_18
8	340	Footwall Argillite E Flank, upper thrust (N)	4	DVR_Solide/342/Fev18/DVR_18
9	410	Lower Arenite	10	DVR_Solide/413/Fev18/DVR_18
10	440	Lower Arenite	5	DVR_Solide/443/Fev18/DVR_18
11	510	Deep Argillite	11	DVR_Solide/513/Fev18/DVR_18
12	540	Deep Argillite	6	DVR_Solide/543/Fev18/DVR_18
13	620	Upper Argillite W Flank	12	DVR_Solide/620/Fev18/DVR_18
14	640	Upper Argillite E Flank	7	DVR_Solide/643/Fev18/DVR_18
15	1010	Arg Sup - W Flank	13	DVR_Solide/1013/Fev18/DVR_18
16	1020	Argil- Flanc E Flank	14	DVR_Solide/1023/Fev18/DVR_18
17	1030	Intrusif Upper- W Flank	15	DVR_Solide/1033/Fev18/DVR_18
18	1040	Intrusif Upper -E Flank	16	DVR_Solide/1043/Fev18/DVR_18
19	10	Nose		DVR_Solide/Nose/Sept2017/DVR_2017

14.2.4.1 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.

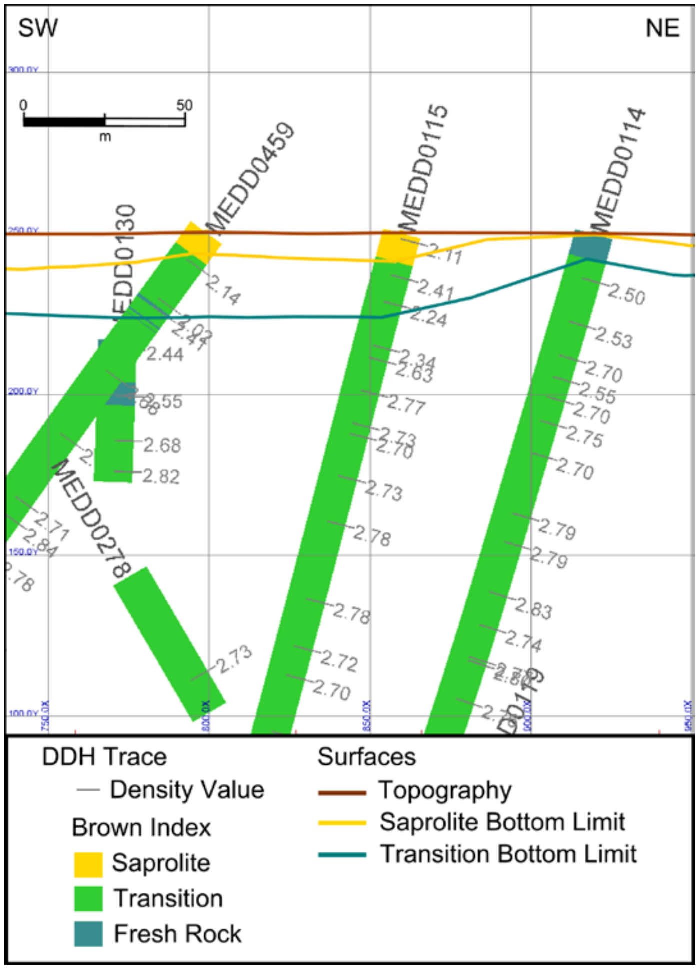
The surfaces previously used for the December 31, 2017 resource estimate have been updated with new hole information based firstly on the density measurements, where available, and by placing the limits midway between density values showing a change in the weathering zone. Where no density measurements 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 pocket knife.	5.0 – 25
	R3	Medium strong rock	Can be peeled by a pocket knife 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 pocket knife, 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



14.2.4.2 LITHO-STRUCTURAL 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 further divided into parts relating to the anticlinal fold axis, West or East flank units, and according to their positions in the folds, i.e., the nose or the limb (geometric association). These units, as illustrated on Figures 14-4 and 14-5, determined the main litho-structural domains.

FIGURE 14-4 ISOMETRIC VIEW – EMZ LITHOLOGICAL MODEL

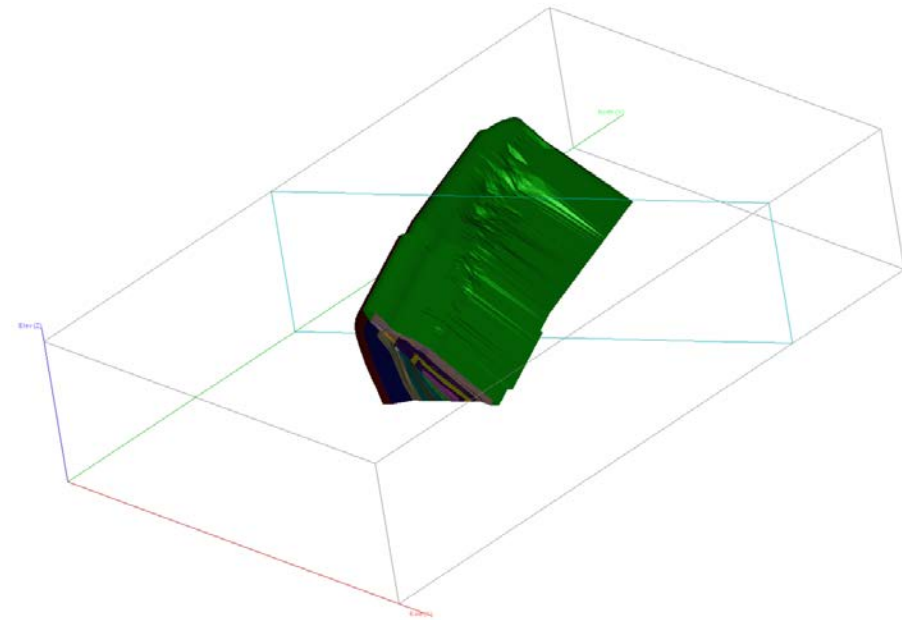
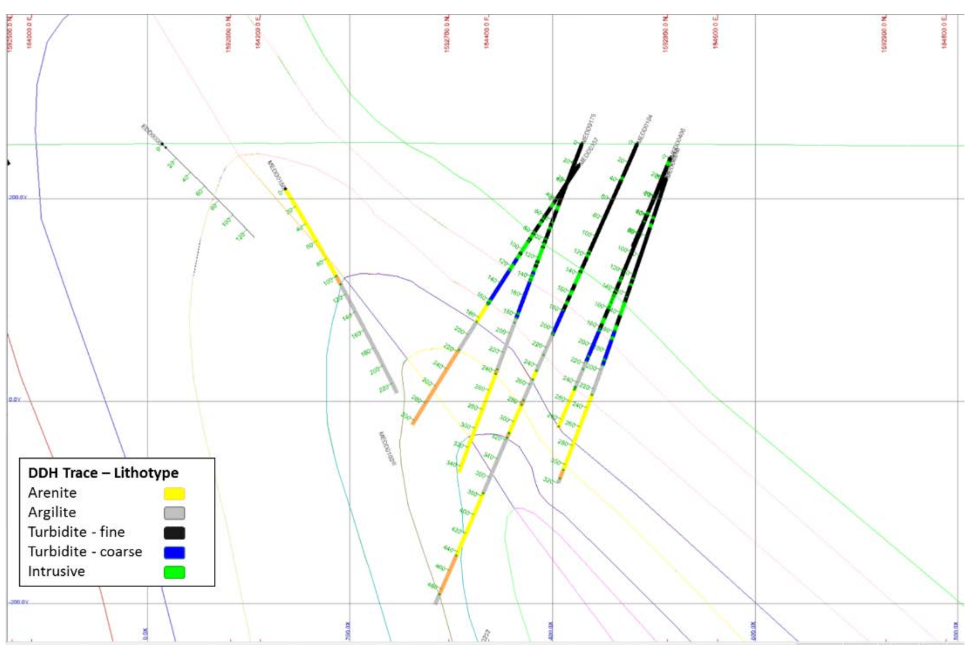


FIGURE 14-5 SECTION 51825N – EMZ LITHOLOGICAL MODEL



Following the EMZ deposit litho-structural 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 June 5, 2018 Mineral Resource, all of Essakane was modelled using the same sequence of domains.

14.2.4.3 SURFACE TOPOGRAPHY

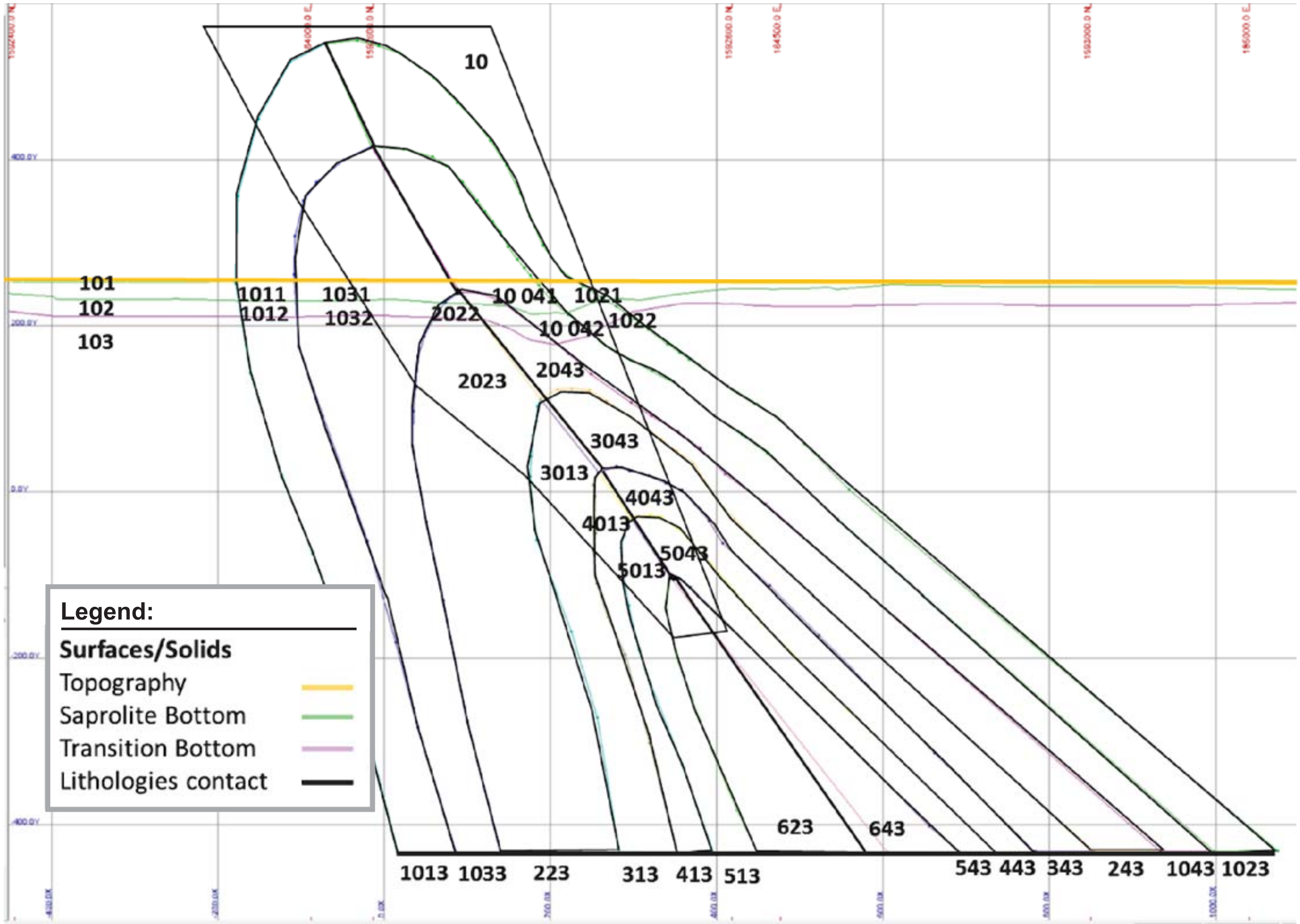
The surface (pre-mining) named “Topo_2009” was used to code all blocks above it as “Air” in the block model.

14.2.5 STATISTICAL ANALYSIS

14.2.5.1 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

Figure 14-6

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

EMZ Section 51550N – Example of Domain Coding

July 2018

Source: Essakane S.A., 2018.

Table 14-5 presents the statistics of the uncapped gold assays of each individual domain and the chosen capping value.

TABLE 14-5 STATISTICS OF THE ASSAYS GROUPED BY DOMAIN

Domain Code	Number	Statistics of Uncapped Assays Mean (g/t Au)	Max (g/t Au)	STD(1) (g/t Au)	COV(2)	Capping Level (g/t Au)	Number Capped
221	107	0.29	6.22	0.87	3.01	-	-
222	140	0.10	2.73	0.25	2.63	-	-
223	8409	0.07	14.04	0.41	5.58	6	8
241	5585	1.15	341.27	6.12	5.34	30	23
242	5310	1.19	336.16	6.92	5.82	30	19
243	27891	1.07	430.00	6.73	6.29	45	82
311	0	-	-	-	-	-	-
312	0	-	-	-	-	-	-
313	4113	0.14	44.21	1.41	9.81	13	6
341	15	0.08	0.34	0.12	1.47	-	-
342	185	0.11	2.23	0.30	2.66	-	-
343	9218	0.45	233.60	3.62	7.98	40	6
413	1291	0.18	66.68	2.22	12.26	5	3
443	1029	0.54	100.00	3.71	6.87	25	2
513	308	0.06	2.11	0.22	3.59	-	-
543	479	0.25	23.14	1.25	5.06	10	1
623	378	0.07	2.77	0.28	3.72	1	0
643	151	0.15	3.84	0.46	3.03	-	-
1011	281	0.03	0.48	0.06	2.12	-	-
1012	337	0.02	1.19	0.07	3.57	-	-
1013	1543	0.06	36.00	0.95	15.70	6	1
1021	4527	0.07	25.51	0.48	6.93	3	7
1022	3675	0.06	48.26	0.98	15.66	3	3
1023	14157	0.03	19.28	0.25	7.57	6	3
1031	1194	0.11	8.01	0.43	3.93	-	-
1032	1441	0.08	4.98	0.29	3.67	-	-
1033	7084	0.11	45.61	0.98	8.68	10	9
1041	7882	0.22	107.00	1.54	7.10	15	5
1042	3872	0.19	28.60	1.00	5.24	11	4
1043	15895	0.13	99.17	1.51	12.03	25	10
2201	3706	0.40	42.30	1.51	3.73	12	9
2202	4039	0.35	47.33	1.44	4.09	15	6
2203	22458	0.32	119.80	2.30	7.15	25	24
2401	19479	0.92	333.00	4.69	5.09	30	45

Domain Code	Number	Statistics of Uncapped Assays Mean (g/t Au)	Max (g/t Au)	STD(1) (g/t Au)	COV(2)	Capping Level (g/t Au)	Number Capped
2402	10201	0.98	160.00	4.33	4.43	30	28
2403	16307	0.88	387.00	5.16	5.85	45	25
3101	592	0.09	4.50	0.27	3.19	0.6	8
3102	942	0.13	9.74	0.51	4.03	4	3
3103	18100	0.46	109.00	3.11	6.72	30	41
3401	4872	0.25	51.90	1.52	6.12	3	55
3402	6011	0.28	120.10	2.14	7.51	3	81
3403	34377	0.57	191.00	3.36	5.95	45	42
4103	6970	0.44	73.42	2.35	5.39	30	8
4403	8907	0.65	169.92	4.25	6.53	30	24
5103	3300	0.79	100.00	4.61	5.85	20	30
5403	2544	0.95	100.00	5.81	6.14	30	14
6203	1945	0.39	100.00	3.18	8.21	30	3
6403	1189	0.35	40.71	1.64	4.64	10	3
10101	127	0.02	0.24	0.04	1.95	-	-
10102	440	0.09	3.54	0.31	3.45	-	-
10103	2572	0.07	6.02	0.31	4.16	1.5	2
10201	833	0.33	145.70	5.13	15.44	3	6
10202	2309	0.27	47.40	1.65	6.24	7	16
10203	4259	0.16	85.38	1.56	9.52	5	13
10301	2558	0.30	25.33	1.12	3.74	10	9
10302	2586	0.29	46.43	1.82	6.31	15	7
10303	10134	0.31	96.98	2.27	7.25	10	49
10401	4099	0.44	51.51	2.22	5.11	15	17
10402	4226	0.25	50.33	1.35	5.43	7	16
10403	13690	0.25	125.00	1.65	6.71	25	8

Notes:

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

Essakane S.A.'s statistical analysis showed that, 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 the high grade assays prior to compositing to restrict the influence of outliers in the composites used for grade interpolation. Capping levels for each domain were initially selected to separate outliers and were later adjusted to better match the production results.

14.2.5.2 COMPOSITING

The drill hole database coded within each interpreted domain was composited to achieve a uniform sample support. Taking into account the current bench heights of the mining operation (5 m to 10 m), 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.

14.2.5.3 STATISTICS OF THE 5 M COMPOSITES

Descriptive statistics of the 5 m composites were generated and grouped by domain. The gold grade statistics for the various estimation domains are characterized by a generally high coefficient of variation (mostly between 1.5 and 2.5), 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 5 M COMPOSITES BY DOMAIN

Domain Code	Number	Min (g/t Au)	Max (g/t Au)	Mean (g/t Au)	STD(1) (g/t Au)	COV(2)
221	23	0.01	2.46	0.27	0.54	1.98
222	32	0.00	0.61	0.09	0.11	1.31
223	2,372	0.00	3.57	0.06	0.22	3.45
241	1,159	0.00	85.50	1.10	3.23	2.93
242	1,090	0.00	70.57	1.17	3.25	2.78
243	7,028	0.00	117.08	1.04	3.74	3.58
313	1,307	0.00	16.65	0.15	0.92	6.01
341	3	0.00	0.24	0.08	0.14	1.63
342	37	0.00	0.65	0.11	0.15	1.36
343	2,508	0.00	63.98	0.40	1.85	4.68
413	407	0.00	30.43	0.20	1.69	8.37
443	321	0.00	30.13	0.52	2.20	4.22
513	98	0.00	1.18	0.07	0.16	2.37
543	152	0.00	4.80	0.22	0.54	2.49
623	115	0.00	0.99	0.08	0.18	2.33
643	49	0.00	1.87	0.15	0.33	2.13
1011	58	0.00	0.21	0.03	0.04	1.50
1012	69	0.00	0.26	0.02	0.04	1.88
1013	406	0.00	10.71	0.06	0.55	8.72
1021	1,032	0.00	5.16	0.07	0.23	3.32
1022	846	0.00	9.67	0.06	0.42	6.79
1023	3,783	0.00	4.02	0.03	0.14	4.43

Domain Code	Number	Min (g/t Au)	Max (g/t Au)	Mean (g/t Au)	STD[1] (g/t Au)	COV[2]
1031	251	0.00	3.27	0.11	0.30	2.79
1032	299	0.00	1.27	0.08	0.17	2.19
1033	1,829	0.00	9.68	0.11	0.53	4.58
1041	1,663	0.00	13.19	0.21	0.56	2.66
1042	819	0.00	11.85	0.20	0.60	3.08
1043	4,130	0.00	24.39	0.12	0.75	6.39
2201	779	0.00	17.00	0.40	0.97	2.43
2202	846	0.00	10.54	0.35	0.80	2.27
2203	5,571	0.00	28.07	0.33	1.24	3.80
2401	4,175	0.00	68.83	0.90	2.35	2.62
2402	2,156	0.00	50.24	0.96	2.21	2.30
2403	3,958	0.00	79.63	0.85	2.57	3.02
3101	147	0.00	9.62	0.15	0.80	5.34
3102	215	0.00	3.53	0.12	0.31	2.56
3103	5,050	0.00	43.13	0.46	1.87	4.03
3401	1,010	0.00	11.43	0.24	0.69	2.90
3402	1,263	0.00	33.04	0.27	1.12	4.11
3403	8,003	0.00	48.51	0.57	1.89	3.29
4103	2,282	0.00	32.18	0.41	1.43	3.49
4403	2,647	0.00	51.57	0.64	2.39	3.71
5103	967	0.00	50.08	0.74	3.02	4.08
5403	784	0.00	32.59	0.82	2.80	3.43
6203	597	0.00	30.11	0.36	1.55	4.30
6403	378	0.00	35.17	0.44	2.00	4.56
10101	28	0.00	0.10	0.02	0.02	1.25
10102	94	0.00	0.82	0.08	0.18	2.21
10103	577	0.00	1.61	0.07	0.19	2.55
10201	181	0.00	33.96	0.31	2.53	8.17
10202	494	0.00	14.53	0.26	0.91	3.56
10203	960	0.00	17.19	0.16	0.69	4.44
10301	558	0.00	9.55	0.28	0.64	2.25
10302	542	0.00	11.40	0.30	0.97	3.19
10303	2,390	0.00	44.70	0.29	1.41	4.81
10401	893	0.00	19.21	0.45	1.26	2.84
10402	881	0.00	11.01	0.27	0.82	3.06
10403	3,207	0.00	34.02	0.23	0.90	3.85

Notes:

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

14.2.5.4 DENSITY DATA

The density database contained 27,686 measurements taken from DD and RCD holes. A small number of outliers were removed from the GEMS density database. The excluded values, listed in Table 14-7, were generally too low or too high compared to neighbouring values or the weathering profile.

TABLE 14-7 EXCLUDED 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

HOLE-ID	Depth (m)	Density (g/cm^3)
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 18,513 measurements, including values ranging from 1.0 g/cm^3 to 3.58 g/cm^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. These domains were attributed a density corresponding to the median of their weathering group. For example, domain 242, which contained four density measurements, was assigned a bulk density corresponding to the median of the saprolite group at 2.32 g/cm^3.

TABLE 14-8 STATISTICS OF THE 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	241	113	1.12	2.89	1.86	1.85	1.9
	1011	31	1.42	2.64	2.11	2.04	2.0
	1021	75	1.07	2.34	1.83	1.86	1.9
	1031	15	1.00	2.26	1.91	1.99	2.0
	1041	71	1.43	2.28	1.90	1.87	1.9
	2201	46	1.66	2.59	2.12	2.12	2.1
	2401	583	1.20	2.81	1.88	1.82	1.8
	3101	1	2.12	2.12	2.12	2.12	2.0
	3401	47	1.62	2.55	1.92	1.86	1.9
	10101	2	1.81	2.00	1.91	1.91	1.9
	10201	12	1.25	2.11	1.85	1.96	1.9
	10301	57	1.65	2.36	2.02	2.03	2.0
	10401	68	1.02	2.26	1.82	1.88	1.9
	All	**1,121**	**1.41**	**2.89**	**1.94**	**1.95**	**1.9**

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)
Transition	222	2	2.57	2.64	2.61	2.61	2.3
	242	262	1.45	2.96	2.29	2.32	2.3
	342	4	2.41	2.50	2.46	2.47	2.3
	1012	34	2.11	2.72	2.49	2.53	2.5
	1022	178	1.48	2.67	2.14	2.12	2.1
	1032	11	1.90	2.77	2.34	2.27	2.3
	1042	74	1.68	2.77	2.17	2.16	2.2
	2202	65	1.77	2.77	2.44	2.49	2.5
	2402	463	1.37	3.07	2.31	2.37	2.4
	3102	16	2.04	2.68	2.37	2.31	2.3
	3402	143	1.66	2.80	2.32	2.35	2.4
	10102	18	1.76	2.53	2.03	1.89	2.3
	10202	40	1.63	2.40	1.91	1.91	1.9
	10302	81	1.80	2.62	2.17	2.11	2.1
	10402	212	1.39	2.63	2.08	2.06	2.1
	All	**1,603**	**1.80**	**3.07**	**2.27**	**2.26**	**2.3**
Fresh Rock	223	248	2.29	2.88	2.74	2.74	2.7
	243	3,025	1.76	3.48	2.73	2.74	2.7
	313	168	2.72	2.92	2.80	2.81	2.8
	343	897	2.33	3.28	2.79	2.80	2.8
	413	48	2.71	2.89	2.77	2.78	2.8
	443	50	2.57	2.93	2.76	2.77	2.8
	513	2	2.78	2.83	2.81	2.81	2.8
	543	17	2.68	2.89	2.83	2.85	2.8
	623	4	2.74	2.81	2.77	2.77	2.8
	643	1	2.83	2.83	2.83	2.83	2.8
	1013	89	2.27	2.90	2.73	2.75	2.8
	1023	965	1.73	3.00	2.62	2.73	2.7
	1033	176	2.24	2.97	2.77	2.79	2.8
	1043	928	1.57	3.41	2.70	2.76	2.8
	2203	1,135	2.02	3.33	2.73	2.74	2.7
	2403	1,736	1.62	3.16	2.71	2.73	2.7
	3103	1,036	2.04	3.36	2.79	2.80	2.8
	3403	2,647	1.75	3.47	2.76	2.79	2.8
	4103	398	2.56	3.29	2.78	2.77	2.8
	4403	731	2.12	3.20	2.78	2.77	2.8
	5103	143	2.56	3.07	2.81	2.82	2.8
	5403	150	2.48	2.94	2.79	2.80	2.8
	6203	99	2.51	2.96	2.79	2.79	2.8
	6403	81	2.61	2.90	2.76	2.75	2.8

Weathering	Domain Code	Number	Min (g/cm³)	Max (g/cm³)	Mean (g/cm³)	Median (g/cm³)	Density Used in Block Model (g/cm³)
	10103	1	2.84	2.84	2.84	2.84	2.8
	10203	70	1.90	2.99	2.59	2.69	2.7
	10303	232	1.74	3.00	2.63	2.77	2.8
	10403	712	1.77	3.58	2.54	2.60	2.6
	All	**15,789**	**2.28**	**3.58**	**2.75**	**2.77**	**2.8**

14.2.6 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.

An isotropic search was used for grade interpolation since the EMZ deposit holds three main sets of veins within its litho-structural domains. Relative pairwise variograms were computed in GEMS and completed on the five metre composites. Downhole variograms were used to confirm the nugget effect values. Variogram maps were produced to establish the main continuity direction. Models were fit in GEMS mostly with two spherical structures.

The components of the modelled variograms are summarized in Table 14-9. Generally, the nugget effect is between 50% and 65% of the total variance. The high nugget effect implies large variability within short distances.

TABLE 14-9 SEMI-VARIOGRAM PROFILES USED FOR ESSAKANE'S DOMAINS

Profile Name	Domain Code	Model Type	Nugget	1st Structure		2nd Structure		Anisotropy Rotation			
				Sill	Range 1 /2 /3 (m)	Sill	Range 1 / 2 /3 (m)	Rotation	Angle 1	Angle 2	Angle 3
D221-223	221, 222, 223		0.017	0.011	23 23/ 23	0.009	23 / 23/ 23		145	0	235
D2201-03	2201, 2202, 2203		0.210	0.070	30 / 30 / 30	0.060	100 / 100 / 100		330	6	58
D241-2	241, 242		1.770	0.280	39 / 39 / 20	0.170	211 / 211 / 110		148	-7	44
D243	243		3.100	1.210	20 / 16 / 10	1.250	67 / 59 / 36		65	-45	155
D2401-03	2401, 2402, 2403		1.240	0.340	31 / 31 / 21	0.300	202 / 202 / 133		145	-3	234
D311-313	311, 312, 313		0.124	0.110	33 / 33 / 33	0.040	141 / 141 / 141		155	0	245
D3101-02	3101, 3102	Spherical	0.016	0.009	23 / 23 / 23	-	-		0	0	0
D3103	3103		1.395	0.963	22 / 22 / 21	0.574	168 / 168 / 159		333	6	50
D341-343	341, 342, 343		0.603	0.429	37 / 27 / 22	0.316	166 / 122 / 101		151	-3	238
D3401-02	3401, 3402		0.156	0.064	23 / 23 / 23	0.030	163 / 163 / 163		155	0	245
D3403	3403		1.097	0.733	28 / 17 / 15	0.400	179 / 106 / 98		155	0	65
D413	413		0.038	0.023	24 / 24 / 24	0.026	103 / 103 / 103		65	-75	310
D4103	4103	Exponential	0.612	0.397	75 / 49 / 47			Azimuth Dip Azimuth	260	80	290
D443	443		0.910	0.606	21 / 19 / 19	0.425	162 / 149 / 146		150	0	240
D4403	4403		1.763	0.838	23 / 16 / 23	0.427	140 / 94 / 140		60	-30	150
D1011-13	1011, 1012, 1013		0.007	0.003	15 / 15 / 15	0.007	85 / 85 / 85		110	0	200
D10101-3	10101, 10102, 10103		0.006	0.006	50 / 50 / 50				0	0	90
D1021-3	1021, 1022, 1023		0.006	0.004	24 / 24 / 24	0.002	136 / 136 / 136		155	0	65
D10201-3	10201, 10202, 10203		0.770	0.580	30 / 25 / 23	0.210	172 / 144 / 134		160	25	260
D1031-3	1031, 1032, 1033	Spherical	0.087	0.051	41106	0.072	70 / 33 / 52		198	62	148
D10301-3	10301, 10302, 10303		0.270	0.280	67/ 44 / 43				345	15	85
D1041-3	1041, 1042, 1043		0.150	0.160	21 / 21 / 21	0.050	215 / 215 / 215		145	0	235
D10401-3	10401, 10402, 10403		0.290	0.170	34 / 26 / 29	0.170	218 / 165 / 184		145	0	235
DeepL	513, 543, 623, 643		0.075	0.043	34 / 34 / 34				160	25	230
DeepN	5103, 5403, 6203, 6403		1.921	1.283	25 / 25 / 25	0.422	180 / 180/ 180		145	0	235

14.2.7 BLOCK MODELLING

14.2.7.1 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 waste dumps. 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 (5 m to 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)
	East	185,400	150	10	
RES18Offic	North	1,589,400	600	10	30
	Elevation	270	48	10	

Notes:
1. In GEMS, the origin point is at the southwest corner at highest level of the block model
2. For a positive value, the direction of rotation is counter clockwise 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 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

14.2.7.2 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 at least 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 litho-structural 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 GEMS, was set to all domains. Domains were modelled as juxtaposed (no overlaps) wireframes. The rock codes attributed from the litho-structural domains were adjusted with the corresponding weathering code afterwards. 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 attributes, Weathering and Rock Type, the blocks located 99.9% above the pre-mining topography surface were defined as “Air” and coded 0.

TABLE 14-12 ROCK CODES FOUND IN THE 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

14.2.7.3 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 BLOCK MODEL

Saprolite		Transition		Fresh Rock	
Rock Code	Density (g/cm^3)	Rock Code	Density (g/cm^3)	Rock Code	Density (g/cm^3)
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
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 x100 m x 50 m (X, Y, Z). The details of the density interpolation are listed in Tables 14-14 and 14-15. The results, where estimated, overwrote the background density values previously entered.

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TABLE 14-14 INTERPOLATION DETAILS FOR THE DENSITY ESTIMATION

Block Model Parameters	Description
Data Source	Density Measurements > 1 g/cm³ and < 3.6 g/cm³ from DD & RCD Hole Types
Interpolation Method	Ordinary Kriging
Minimum/Maximum Sample	4/30
Maximum Sample per Hole	No maximum defined
Boundary Type	Soft & 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 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

Target Rock Code	Limit Target Rock Codes			
1033	1033	1043	10303	10403
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

14.2.7.4 GRADE ESTIMATION METHODOLOGY

Grade estimation for Essakane 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.

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

Interpolation Profile Name	Target Rock Code	Variography Profile Name	Limit Target Rock Codes						
17DUARG	623	D_DEEPL	623	6203	6403				
	543		543	5403	5103				
	513		513	5103	5403				
	643		643	6403	6203				
	413	D413	413	4103	4403				
	443	D443	443	4403	4103				
	6203	D_DEEPN	6203	6403	623	643			
	5403		5403	5103	513	543			
	5103		5103	5403	513	543			
	4103	D4103	4103	4403	413	443			
	4403	D4403	4403	4103	443	413			
	6403	D_DEEPN	6403	6203	643	623			
17LARG	311	D_313	311	3101	312	3102			
	312		312	311	3101	3102	313	3103	
	313	D_314	313	3103	312	3102			
	341	D341-343	341	342	3401	3402			
	342		342	343	341	3401	3402		
	343		343	342	3402	3403			
	3101	D3101-02	3101	3401	311	3102	312	3402	
	3102		3102	311	312	313	3101	3103	3401
	3103	D3103	3103	312	313	3102	3402	3403	
	3401	D3401-02	3401	341	342	3402	3101	3102	
	3402		3402	341	342	343	3101	3102	3103
	3403	D3403	3403	342	343	3102	3103	3402	
17MAREN	221	D221-223	221	222	2201	2203			
	222		222	221	223	2201	2202	2203	
	223		223	222	2202	2203			
	241	D241-2	241	242	2401	2402			
	242		242	241	243	2401	2402	2403	
	243	D243	243	242	2402	2403			
	2201	D2201-03	2201	221	222	2202	2401	2402	
	2202		2202	221	222	223	2201	2203	2401
	2203		2203	222	223	2202	2402	2403	
	2401	D2401-03	2401	241	242	2201	2202	2402	

Interpolation Profile Name	Target Rock Code	Variography Profile Name	Limit Target Rock Codes						
	2402		2402	241	242	243	2201	2202	2203
	2403		2403	242	243	2202	2203	2402	
17NORD	1011		1011	10101	1012	10102			
	1012	D1011-13	1012	1011	1013	10101	10102	10103	
	1013		1013	1012	10103	10102			
	1021		1021	1022	10201	10202			
	1022	D1021-3	1022	1021	1023	10201	10202	10203	
	1023		1023	1022	10202	10203			
	10101		10101	1011	1012	10102	10201	10202	
	10102	D10101-3	10102	1011	1012	1013	10101	10103	
	10103		10103	1012	1013	10102	10202	10203	
	10201		10201	1021	1022	10101	10102	10202	
	10202	D10201-3	10202	1021	1022	1023	10101	10102	
	10203		10203	1022	1023	10102	10103	10202	
	1031		1031	1032	10301	10302			
	1032	D1031-3	1032	1031	1033	10301	10302	10303	
	1033		1033	1032	10302	10303			
	1041		1041	1042	10401	10402			
	1042	D1041-3	1042	1041	1043	10401	10402	10403	
	1043		1043	1042	10402	10403			
	10301		10301	1031	1032	10302	10401	10402	
	10302	D10301-3	10302	1031	1032	1033	10301	10303	
	10303		10303	1033	1032	10302	10402	10403	
	10401		10401	1041	1042	10301	10302	10402	
	10402	D10401-3	10402	1041	1042	1043	10301	10302	
	10403		10403	1042	1043	10302	10303	10402	

14.2.8 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. 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 report. Consequently, no Measured Mineral Resources were defined.

Indicated Mineral Resources encompassed all blocks in the EMZ and North Satellite areas estimated in the first estimation pass using composites from a minimum of three different drill holes within domains of soft and hard boundaries inside a search sphere of 40 m of radius.

Inferred Mineral Resources corresponded to 1) the blocks of the EMZ and North Satellite areas estimated in the second pass for which composites from a minimum of one drill hole were interpolated within a search sphere of 100 m radius inside domains of soft and hard boundaries, and to 2) the blocks of the EMZ South area estimated in a single pass using composites from at least two drill holes and within an anisotropic search ellipse of ranges 75 m x 50 m x 10 m (X, Y, Z).

14.2.9 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 Vancouver office for last year's update.

14.2.9.1 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 litho-structural 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 boundaries restrictions. The grade distribution was locally influenced by this aspect of the modelling. More specifically, the mineralization between the West and East flanks 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 this geological feature is well represented for the scale of the pit but not 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 at the EMZ deposit.

14.2.9.2 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 composites that fall within a block and the interpolated grade of that block. A successful grade interpolation protocol results in block grade estimates that demonstrate a minimum amount of bias. A comparison was carried out using the weighted average of the nose and the limb 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 resource.

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

RES OFF (OK)							Nearest Neighbour			Percent Changes
Flank			Nose			Flank & Nose	Flank	Nose	Flank & Nose	Flank & Nose
Domain	No	Mean	Domain	No	Mean	Mean Pond	Mean	Mean	Mean Pond	
221	48	0.395	2201	3074	0.341	0.342	0.52	0.339	0.342	0.01%
222	128	0.202	2202	2827	0.297	0.293	0.179	0.33	0.323	-9.45%
223	7131	0.107	2203	26939	0.283	0.246	0.096	0.271	0.234	5.03%
241	3434	0.679	2401	11743	0.689	0.687	0.684	0.689	0.688	-0.16%
242	2873	0.74	2402	5372	0.796	0.776	0.727	0.779	0.761	2.05%
243	22442	0.887	2403	12782	0.785	0.850	0.864	0.783	0.835	1.84%
311			3101	516	0.091	0.091	0.145	0.07	0.137	-33.38%
312			3102	766	0.115	0.115		0.102	0.102	12.75%
313	4085	0.191	3103	25779	0.415	0.384		0.415	0.415	-7.38%
341	14	0.13	3401	3257	0.179	0.179	0.122	0.19	0.190	-5.76%
342	105	0.094	3402	3830	0.182	0.180	0.077	0.169	0.167	7.87%
343	10253	0.408	3403	31674	0.517	0.490	0.396	0.524	0.493	-0.48%
413	1127	0.098	4103	10196	0.392	0.363	0.058	0.373	0.342	6.17%
443	935	0.427	4403	12323	0.561	0.552	0.499	0.55	0.546	0.94%
513	19	0.167	5103	3081	0.795	0.791	0.118	0.824	0.820	-3.48%
543	135	0.344	5403	3177	0.678	0.664	0.286	0.625	0.611	8.71%
623	4	0.169	6203	1297	0.474	0.473	0.011	0.463	0.462	2.48%
643			6403	775	0.396	0.396		0.338	0.338	17.16%
1001	223	0.3				0.300	0.362		0.362	-17.13%
1002	163	0.297				0.297	0.325		0.325	-8.62%
1003	70	0.057				0.057	0.057		0.057	0.00%
1011	6	0.054	10101	157	0.066	0.066	0.107	0.052	0.054	21.35%
1012	41	0.048	10102	244	0.115	0.105	0.079	0.112	0.107	-1.76%

RES OFF (OK)							Nearest Neighbour			Percent Changes
Flank			Nose			Flank & Nose	Flank	Nose	Flank & Nose	Flank & Nose
Domain	No	Mean	Domain	No	Mean	Mean Pond	Mean	Mean	Mean Pond	
1013	402	0.072	10103	1460	0.07	0.070	0.079	0.076	0.077	-8.11%
1021	5366	0.059	10201	1405	0.103	0.068	0.06	0.1	0.068	-0.25%
1022	3888	0.047	10202	2157	0.192	0.099	0.041	0.196	0.096	2.52%
1023	16451	0.032	10203	3880	0.159	0.056	0.03	0.143	0.052	9.06%
1031	642	0.17	10301	2420	0.236	0.222	0.151	0.239	0.221	0.73%
1032	871	0.107	10302	2212	0.229	0.195	0.113	0.24	0.204	-4.70%
1033	6069	0.134	10303	11691	0.228	0.196	0.126	0.231	0.195	0.39%
1041	5428	0.192	10401	3552	0.263	0.220	0.188	0.259	0.216	1.85%
1042	2949	0.18	10402	2976	0.19	0.185	0.185	0.17	0.177	4.26%
1043	17273	0.095	10403	12507	0.203	0.140	0.095	0.206	0.142	-0.89%

14.2.9.3 VALIDATION USING DIFFERENT INTERPOLATION METHODS

The validation of the block model was also done using Inverse Distance Squared and Cubed (ID^2 and ID^3) interpolation to compare with the OK estimate. The same set of composites, search ellipses, and settings were used and only the interpolation technique differed. The results were compared visually. Tonnages, grades, and gold contents are similar.

14.2.9.4 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, easting vertical sections, and elevation. This validation method works as a visual means to identify possible bias in the interpolation. 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 ‘official’ block model compared to declustered five metre composites (using Nearest Neighbour interpolation). In order to facilitate the comparison, only blocks above 0.01 g/t Au were reported on the graphs.

Figures 14-7 and 14-8 illustrate swath plots for interpolated blocks by easting and elevation. The figures show a reasonable to a good correlation between blocks and composite.

FIGURE 14-7 SWATH PLOT FOR EASTING

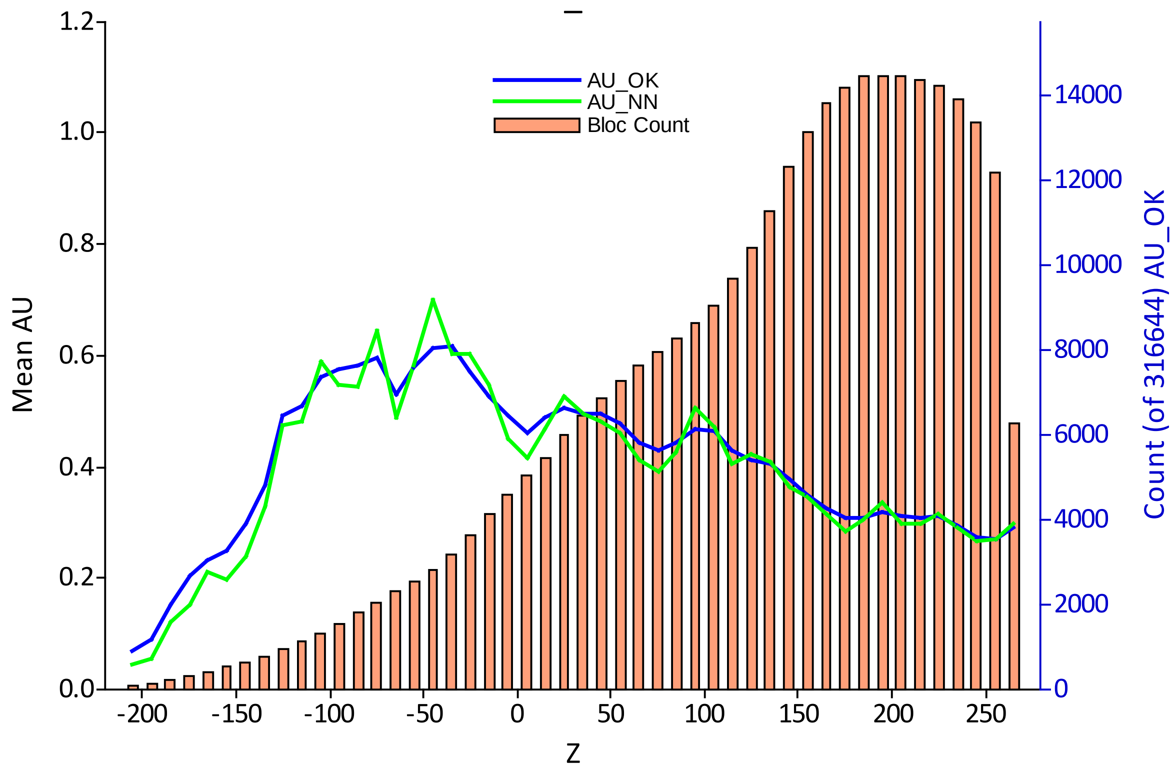
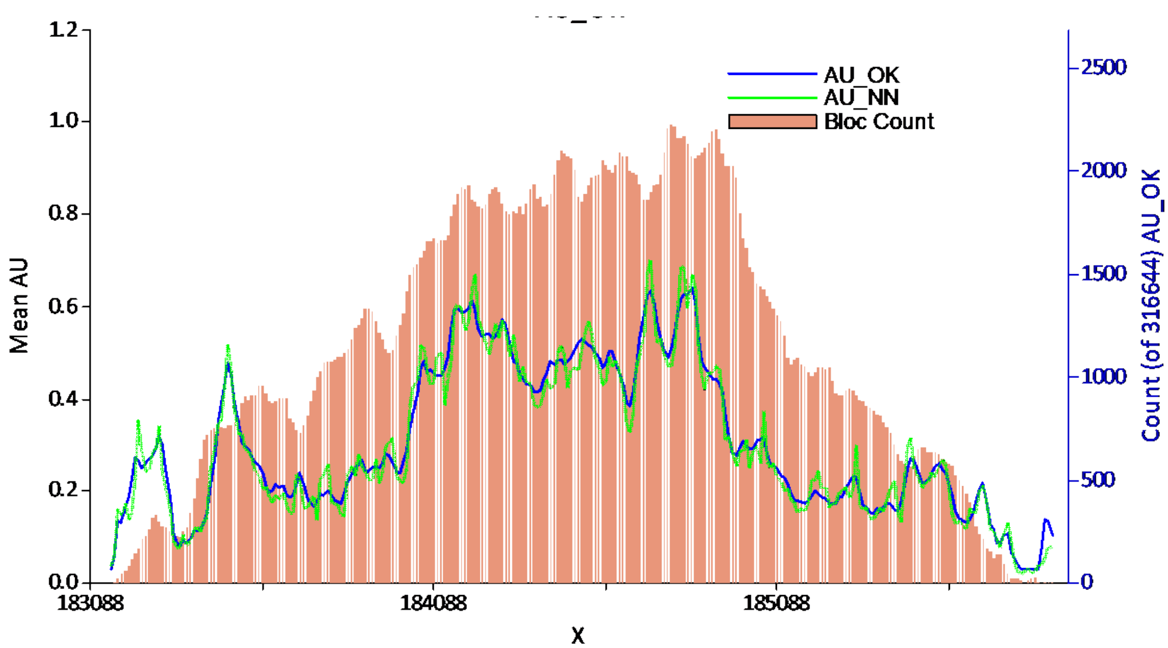


FIGURE 14-8 SWATH PLOT ELEVATION



14.3 FALAGOUNTOU DEPOSITS

The Falagountou West Mineral Resource estimate remains unchanged since the previous Technical Report (Chénard et al., 2016) as no more recent drilling has been carried out at this deposit. A Mineral Resource update of Falagountou East was undertaken by GMSI in August 2016 and subsequently updated in March 2017 to include infill and extensional drilling which is described in the following sections.

14.3.1 DATA

The GEMS database for Falagountou West was acquired by GMSI while at the Essakane Gold Mine site in February 2015 and an update to this database was received by GMSI via secured file transfer on September 26, 2015. The current resource estimates on the Falagountou West is based on the GMSI April 2015 models updated with drilling on the Falagountou West deposit completed subsequently in late 2015. For Falagountou East, GMSI received a drill hole database update via FTP transfer on February 6, 2017. The database included geotechnical and lithology logging information as well as gold assay and density sample results of the holes drilled on the Falagountou project, which includes the West and East areas. GMSI reviewed the information stored in the database and found it to be in good standing.

The GEMS project consists of 1,682 holes of different types covering both Falagountou deposits and exploration areas around these deposits (Figure 14-9). The mineralized zone modelling and resource estimation used three types of drill holes (DD, RC, and RCD). Table 14-18 lists holes used in the resource database by type, year of drilling, and series. Because AC sampling and RAB sampling are more subject to segregation bias, their results are not used in the estimate process.

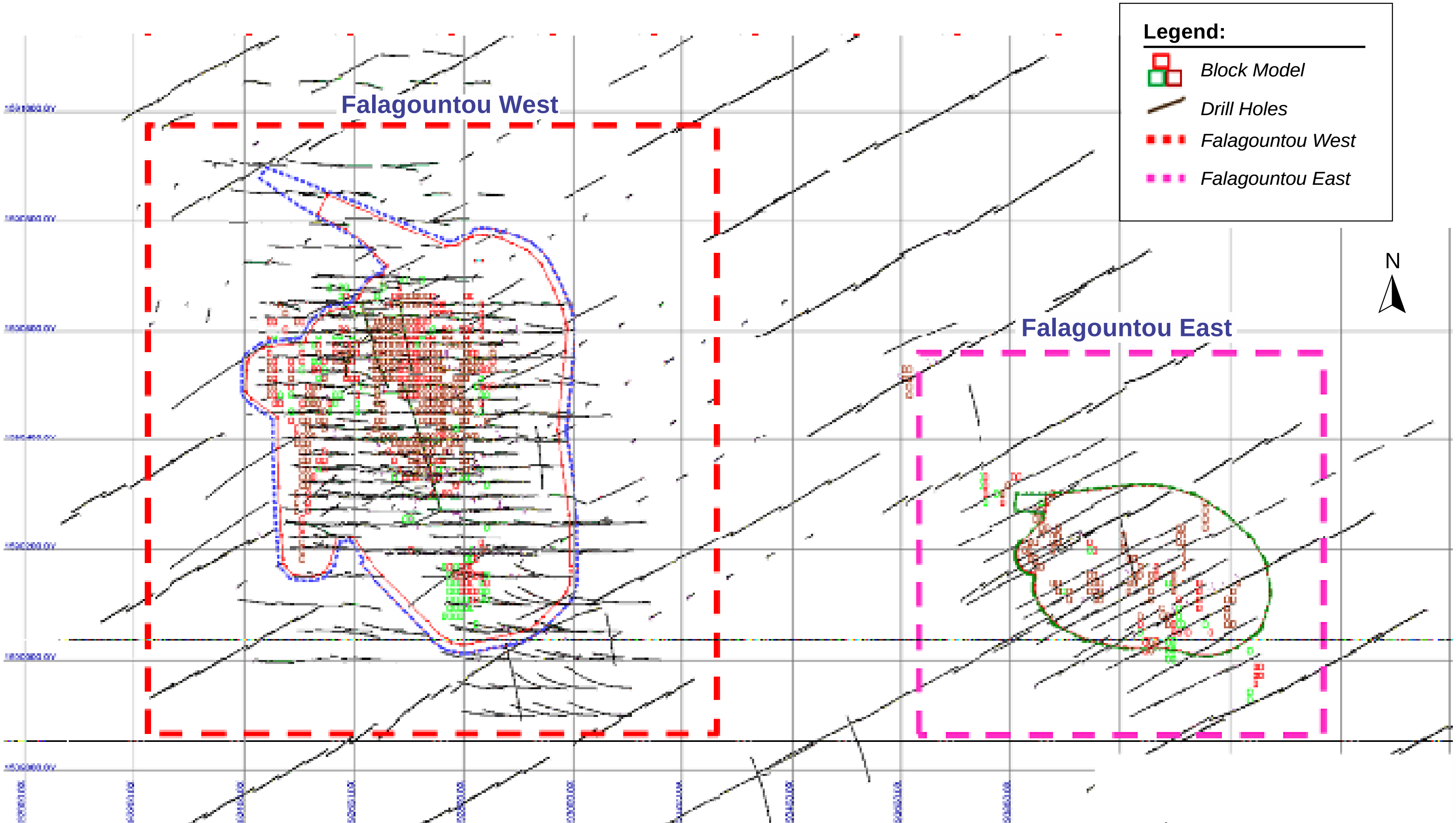


Figure 14-9

0 100 200 300 400
Metres

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Falagountou Deposit Drill Plan

July 2018

Source: Essakane S.A., 2018.

TABLE 14-18 TYPES OF HOLES USED FOR THE RESOURCE ESTIMATE (AS OF MARCH 2017)

Hole Type	Year	Series	Number of Holes	Metres Drilled
DDH	2017	MFDD0114 - MFDD0115	2	364
	2016	MFDD0113	1	151
	2015	MFDD0103 - MFDD0112	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	2,368
	2006	FDD0016 - FDD0024	9	1,384
	2004	FDD0012 - FDD0015	4	819
RC	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 FRC1908 - FRC2051	185	24,165
	2011	FRC1817 - FRC1907	91	12,497
	2010	FRC1772 - FRC1816	45	6,742
	2008	FRC1734 - FRC1771	38	2,822
	2006	FRC1704 - FRC1730 FRC1732 - FRC1733	29	3,730
	2004	FRC0568 - FRC0571 FRC0635 - FRC0670 FRC0780 - FRC0800 FRC0901 - FRC0946	107	8,882
	2003	FRC0428 - FRC0429 FRC0467 - FRC0469 FRC0478 - FRC0490	18	1,248
	1995	FRC0001 - FRC0003	3	139
RCD	2006	FRC1731D	1	225
Total			**1,202**	**142,597**

14.3.2 DRILL HOLE SPACING

Drilling at the Falagountou West deposit was carried out at a drill spacing of 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.

Drilling on the Falagountou East deposit was carried out at a similar grid spacing of approximately 25 m (between sections) by 25 m (on-section). The southern extremity of Falagountou East is drilled at 25 m (between sections) by 50 m (on-section). The drill sections are oriented at 060° and are perpendicular to mineralization, which extends along a 150° to 330° strike axis.

Drill hole spacings on the Falagountou East and West 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.

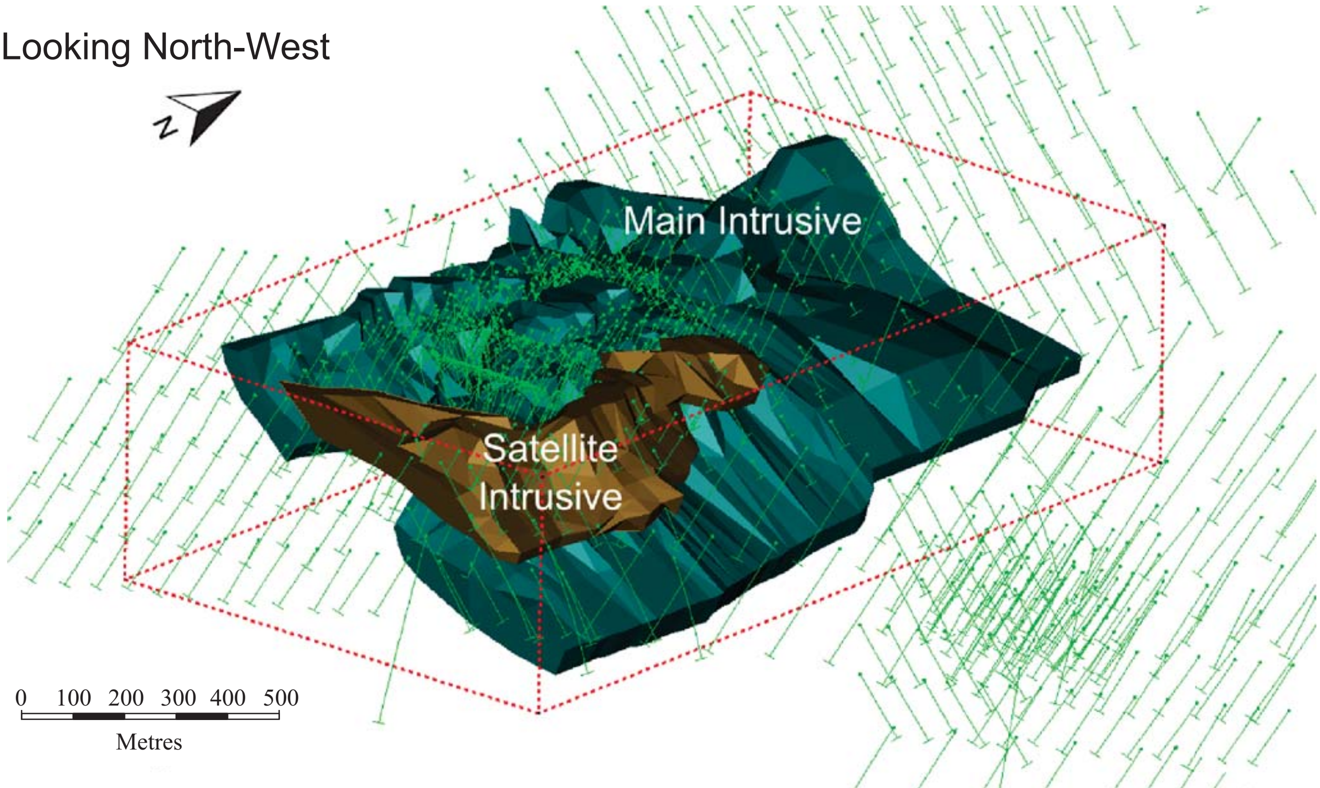
14.3.3 MODELLING

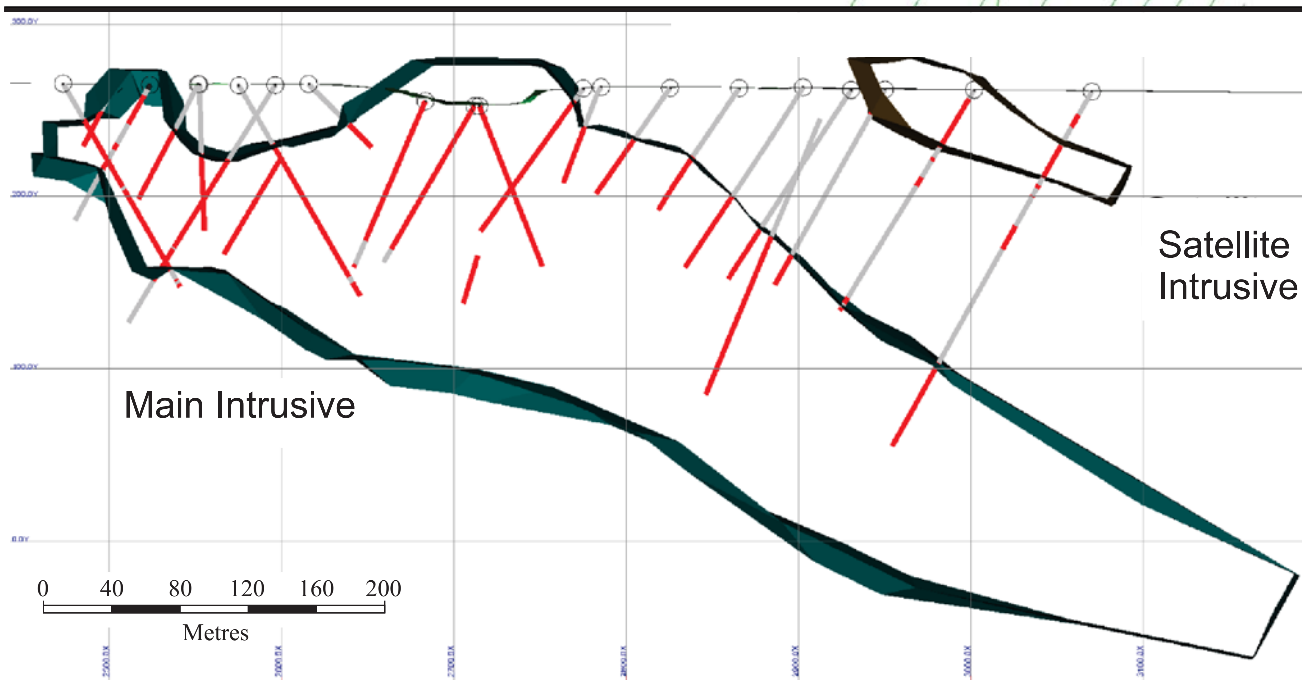
Numerous 2D and 3D modelling elements such as lithology, weathering, and mineralization solids were generated for the purpose of the current resource estimate using GEOVIA GEMS version 6.7.2.

14.3.3.1 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 14-10). The folded contact between intrusive and sedimentary rocks guided the shape of the mineralization zones. The 2015 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.

For the Falagountou East deposit, no lithology solids were used during the estimation.





Section F0450N - Looking North

Figure 14-10

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

**Intrusive Solid -
Falagountou West Deposit**

July 2018

Source: G Mining Services Inc., 2015.

TABLE 14-19 ROCK CODE DESCRIPTION

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

14.3.3.2 WEATHERING WIREFRAMES

For each DD, RC, and RCD hole in the database encompassed inside the Falagountou West and East 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 earlier in this section in Table 14-4. Based on the relationships between density measurements and hardness information, GMSI has reclassified the S5-hardness from transition to saprolite.

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.

For Falagountou West, the intervals of weathering were divided into sublayers 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 sublayers: Transition 1, Transition 2, and Transition 3. The 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 fresh rock) sublayer.

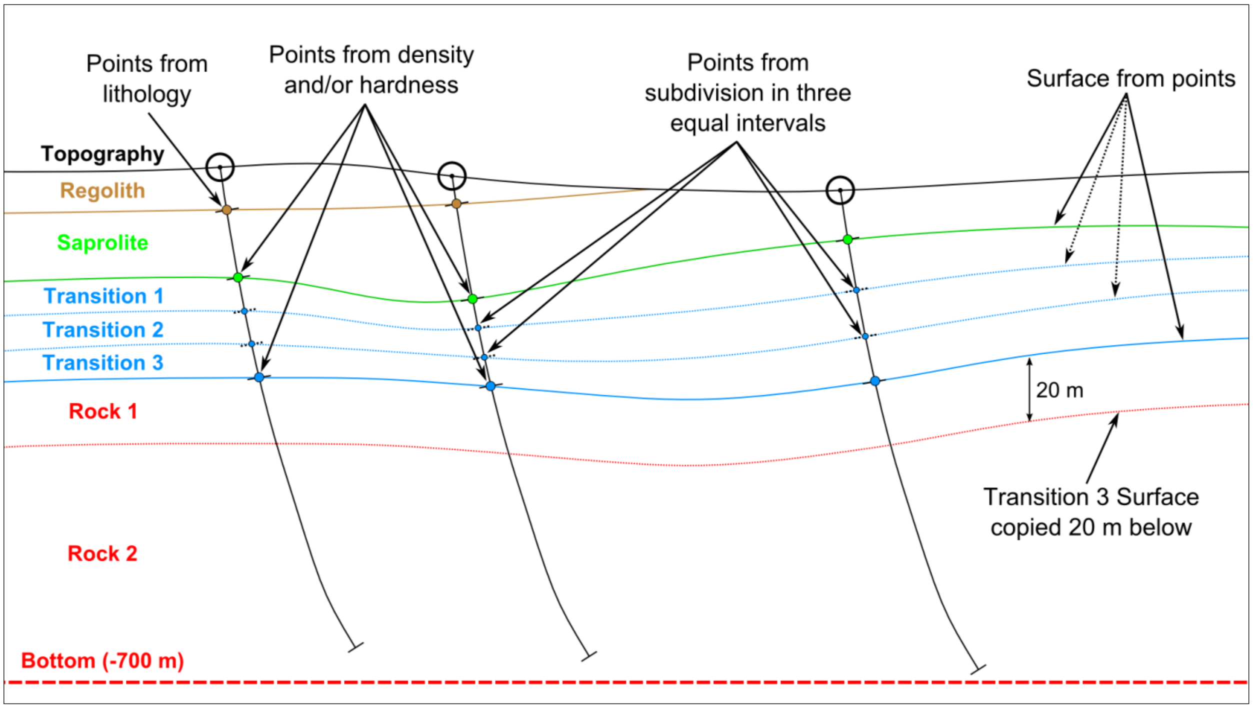
The weathering surfaces were created from their specific set of points except for the Rock 1 surface, which was generated by copying the Transition 3 surface down 20 m, and the Rock 2 bottom surface which is a flat surface located below the deepest hole.

As a last step, solids were constructed from the surfaces. The procedure of the construction of the weathering solids is summarized in Table 14-20 and illustrated in Figure 14-11.

TABLE 14-20 SUMMARY OF WEATHERING SOLID CONSTRUCTION PROCEDURE

Weathering Layer	Density Measurements Limits (t/m^3)	Code	Hardness Category	Creation Method
Regolith	-	6000	-	Points from Lithology → Surface → Solid
Saprolite 1	0 to 2.00	6000	S1, S2, S3, S4, S5	Points from Density and/or Hardness → Surface → Solid
Transition 1	2.00 to 2.55	6100	S6, R0, R1	Points resulting from Transition interval division → Surface → Solid
Transition 2		6200		Points resulting from Transition interval division → Surface → Solid
Transition 3		6300		Points from Density and/or Hardness → Surface → Solid
Rock 1	> 2.55	6400	R2, R3, R4, R5, R6	Transition 3 Surface copied down 20 m → Solid
Rock 2		6500		Flat Surface at Elevation -700 m → Solid

FIGURE 14-11 ILLUSTRATION OF WEATHERING SOLID CREATION TECHNIQUES



For Falagountou East, a simplified weathering model was built using logging codes for Regolith, Saprolite, Transition, and Fresh Rock (derived from the Brown's Index), from which solids were constructed in Leapfrog Geo. No subdivisions within weathering domains were modelled.

14.3.3.3 MINERALIZATION ZONES

The mineralization zones were designed based on the geological structure and gold assay results. Gold assay grades above 0.5 g/t Au were included in the zones and gold grades below 0.5 g/t Au were preferably left outside of the 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. All previous mineralization zones have been revised and updated in order to be used in the current resource estimate.

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. A total of eleven mineralized envelopes were modelled. On average, the zones are 3 m to 9 m thick (Table 14-21). Figure 14-12 illustrates all zones in the Falagountou West deposit.

In the Falagountou East deposit, mineralization solids were designed to delineate areas of mineralization within specific grade ranges due to a lack of grade continuity. To achieve this, all drill hole grade intervals were imported in Leapfrog Geo to generate solids with the “grade interpolant” function. In general, the trend applied during the solid creation process followed the apparent continuity of gold grades and the structure of the sedimentary beds. The following gold grade intervals were selected: (1) lower than 0.40 g/t Au; (2) 0.40 g/t Au to 0.80 g/t Au; (3) 0.80 g/t Au to 1.20 g/t Au; and (4) greater than 1.20 g/t Au (Table 14-22 and Figure 14-13).

In both Falagountou 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)
220	3.64
230	3.87
240	4.55
245	5.79
250	5.95
255	8.56
260	5.14
270	6.20
275	3.37
280	3.39
290	3.63

FIGURE 14-12 MINERALIZATION ZONES - FALAGOUNTOU WEST DEPOSIT

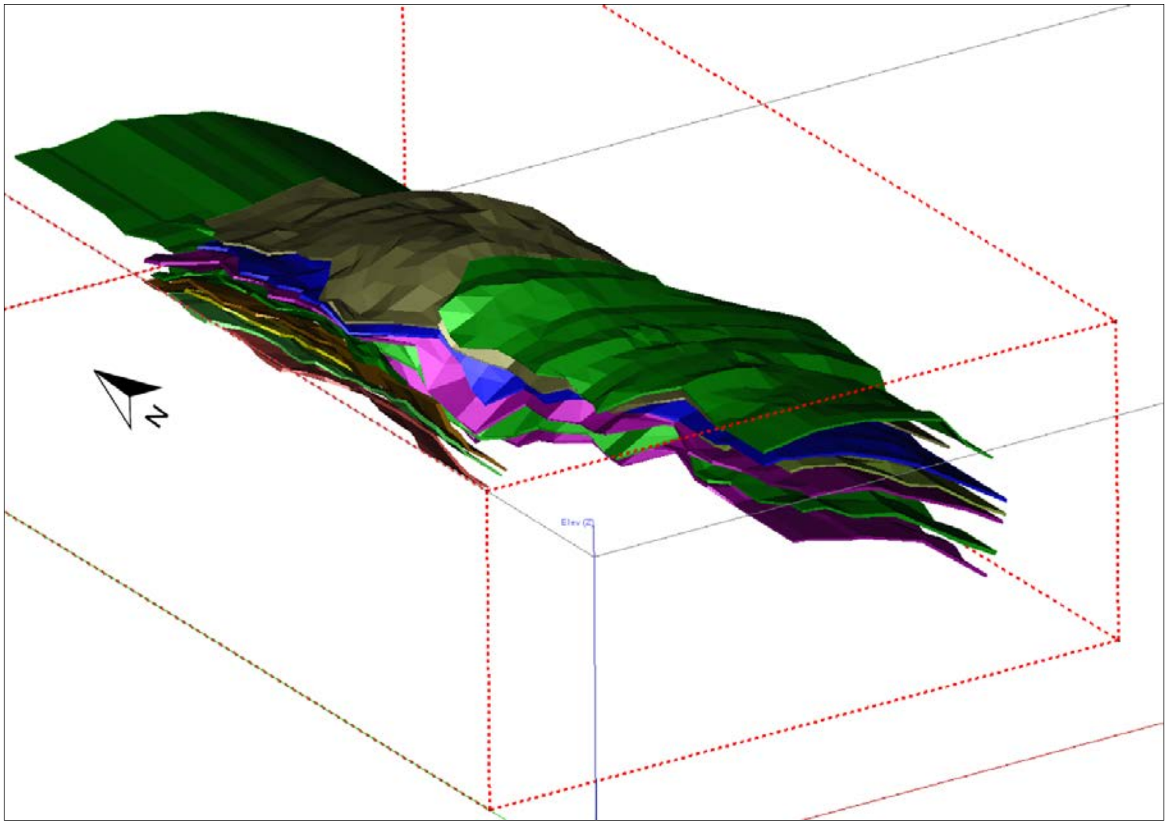
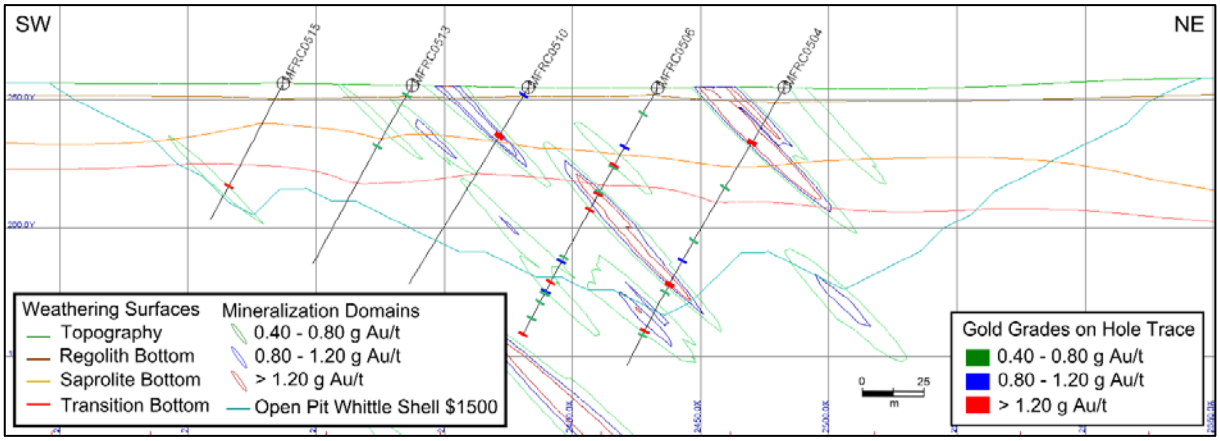


TABLE 14-22 ROCK CODES - FALAGOUNTOU EAST DEPOSIT

Zone	Description
10	< 0.4 g/t
20	0.4 g/t – 0.8 g/t
30	0.8 g/t – 1.2 g/t
40	< 1.2 g/t

FIGURE 14-13 MINERALIZATION ZONES - FALAGOUNTOU EAST DEPOSIT



14.3.3.4 TOPOGRAPHY SURFACE

A topographic surface was provided to GMSI for the resource estimation of the Falagountou West deposit. The surface named Topo Clip 27Aout12 covers most of the block model area apart for a small portion in the northeast, where no drilling is present. Blocks from the model were updated using this surface. For Falagountou East, a topography was created using the drill hole collars. The Status Mined Surface as of June 5, 2018 for the Falagountou West and East deposits was used to evaluate the resource remaining as at June 5, 2018.

14.3.4 STATISTICAL ANALYSIS

14.3.4.1 STATISTICS OF THE RAW-ASSAYS

Statistics of the raw gold assays were computed using GSLIB, a geostatistical program. Statistics were studied for assays grouped by mineralization domains for the Falagountou West deposit (Table 14-23) and as a single collection of zones for the Falagountou East deposit (at the bottom of Table 14-23).

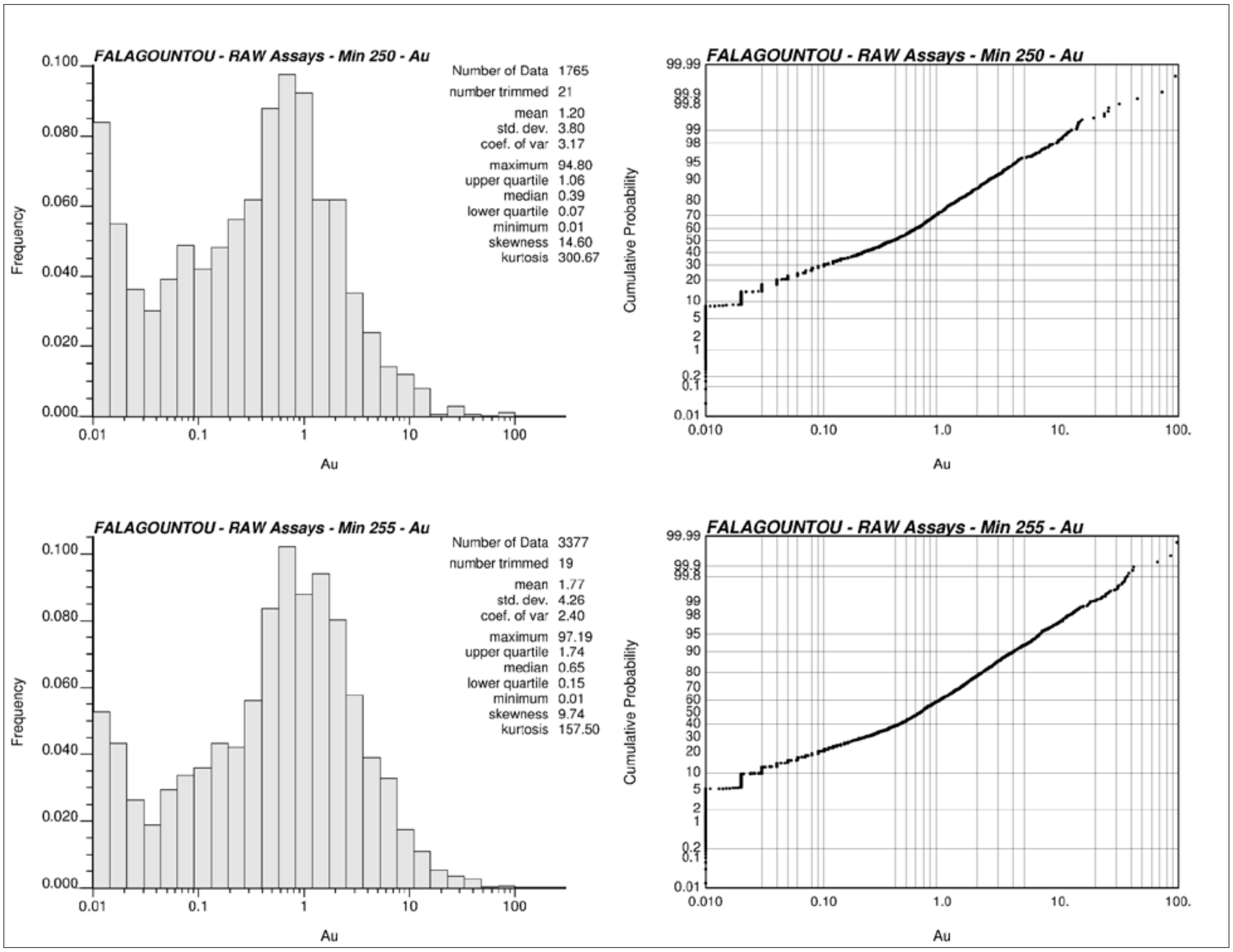
Based on raw assays, three of the eleven Falagountou West zones can be labelled as “high-grade”, with mean gold grades ranging from 1.20 g/t Au to 1.77 g/t Au (Zones 250, 255 and 270). Only two zones, 230 and 260, have mean gold grades close to the modelling cut-off grade of 0.50 g/t Au. The frequency histograms and the cumulative probability plot curves (see Figure 14-14 for graphs of zones 250 and 255) show that most of the zones display a near log-normal distribution. Upon further examination of statistical populations, two populations can be interpreted with a break around 0.50 g/t Au for most of the zones. The

Falagountou East deposit is modelled using various grade-shells, with an average gold grade of 1.34 g/t Au inside grade shells modelled at > 0.4 g/t Au.

TABLE 14-23 STATISTICS OF AU ASSAYS BY MINERALIZED ZONE - FALAGOUNTOU WEST AND EAST DEPOSITS

Zone	# of Assays	Gold Raw Assays (g/t Au)					CoV	Average Thickness (m)
		Min	Max	Average	Median	Standard Deviation		
220	272	0.01	21.4	0.61	0.05	1.79	2.94	3.64
230	596	0.01	21.9	0.47	0.06	1.46	3.08	3.87
240	1,053	0.01	28.8	0.76	0.17	2.06	2.71	4.55
245	160	0.01	21.0	0.84	0.18	2.14	2.55	5.79
250	1,765	0.01	94.8	1.20	0.39	3.80	3.17	5.95
255	3,377	0.01	97.2	1.77	0.65	4.26	2.40	8.56
260	1,076	0.01	27.7	0.57	0.08	1.49	2.61	5.14
270	607	0.01	37.2	1.76	0.63	3.59	2.04	6.20
275	150	0.01	27.2	0.95	0.24	2.68	2.82	3.37
280	244	0.01	25.1	0.78	0.11	2.42	3.08	3.39
290	201	0.01	16.9	1.00	0.44	1.86	1.86	3.63
Falagountou East 20,30,40	3,804	0.01	100.0	1.34	0.17	4.83	3.59	N/A

FIGURE 14-14 HISTOGRAMS AND CUMULATIVE PROBABILITY PLOTS FOR ZONES 250 AND 255



Based on the study of the effect of high grade values on the mean and standard deviation, and from probability and histogram plots, GMSI applied various capping limits depending on the mineralized zone for the Falagountou West deposit and as a single population for the Falagountou East deposit. Table 14-24 lists the capping levels used on the raw assays. Capping values for the Falagountou West deposit range from 6.0 g/t Au to 40.0 g/t Au and all domains at the Falagountou East deposit were capped at 30.0 g/t Au.

TABLE 14-24 GOLD CAPPING VALUES

Zone	Capping (g/t Au)	No. of Assays Capped	Metal Capped
220	6.0	3	16%
230	6.0	8	11%
240	10.0	9	9%
245	10.0	2	9%
250	20.0	8	9%
255	40.0	5	3%
260	10.0	4	5%
270	15.0	10	7%
275	10.0	2	14%
280	10.0	3	16%
290	10.0	2	3%
Falagountou East All Zones	30.0	15	8%

14.3.4.2 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).

14.3.4.3 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 Falagountou East deposit were grouped in a single population, comprising domains 20, 30, and 40, as summarized at the bottom of Table 14-25.

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

Zone	# of Composites	Gold Composites (g/t Au)					
		Min	Max	Average	Median	Standard Deviation	CoV
220	159	0.01	6.00	0.45	0.14	0.79	1.77
230	360	0.01	5.00	0.38	0.09	0.74	1.95
240	606	0.01	10.0	0.63	0.28	1.07	1.68
245	87	0.01	4.66	0.63	0.29	0.93	1.48
250	923	0.01	20.0	1.06	0.50	1.83	1.73
255	1,653	0.01	24.7	1.63	0.83	2.48	1.52
260	555	0.01	6.24	0.51	0.16	0.86	1.67
270	247	0.05	12.1	1.79	0.99	2.12	1.18
275	61	0.05	6.33	1.05	0.58	1.38	1.31
280	128	0.01	5.05	0.62	0.19	1.02	1.66
290	81	0.06	9.30	1.20	0.75	1.45	1.21
Falagountou East 20,30,40	1,616	0.01	19.94	1.16	0.30	2.25	1.94

14.3.4.4 DENSITY DATA

The density database for the 2015 Falagountou West includes density measurements for 5,210 samples. For the 2017 Falagountou East Mineral Resource update, density measurements for 5,213 samples are available within the extents of the block model. Table 14-26 summarizes the basic statistics used to establish the density model. To avoid the influence of outliers, the median value was judged as a good representation of background values for these weathering horizons.

TABLE 14-26 DENSITY DATA STATISTICS

Deposit	Weathering Profile	Code	Number of Measurements	Median (t/m^3)	Average (t/m^3)
Falagountou West 2015 Data	Saprolite	6000	115	1.87	1.87
	Trans-1	6100	109	2.08	2.12
	Trans-2	6200	112	2.29	2.27
	Trans-3	6300	144	2.38	2.35
	Rock-1	6400	638	2.69	2.67
	Rock-2	6500	4,092	2.74	2.74

Deposit	Weathering Profile	Code	Number of Measurements	Median (t/m^3)	Average (t/m^3)
Falagountou East Only 2017 Update	Overburden	9	19	1.94	1.97
	Saprolite	1	159	1.97	2.00
	Transition	2	478	2.43	2.41
	Rock	3	4,557	2.74	2.73

14.3.5 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 downhole composite for all data and for all mineralization zones in the Falagountou West deposit. Sage 2001 geostatistical software was used to perform the analysis.

For the Falagountou West deposit, considering the geometry of the mineralized zones, a series of correlograms were generated from the capped gold grades every 30° azimuth and at a single flat dip and for each of the zones. The optimal anisotropy directions were determined through regression by Sage 2001. The minimum number of composite pairs required for variography was 10. The variography model included a nugget effect and two spherical structures. All zones were grouped together to generate a global model. The global variogram model results are summarized in Table 14-27.

No variography was generated for Falagountou East.

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

			Ranges of Influence (m)										
			1st Structure				2nd Structure				Rotation		
Deposit	Semi-Variogram Profile Name	Nugget	X	Y	Z	Sill	X	Y	Z	Sill	Z	Y	Z
Falagountou West	OK2015	0.65	20	20	10	0.25	125	125	25	0.10	0	0	0

14.3.6 BLOCK MODELLING

A block model was constructed for each of the Falagountou deposits. The block models cover a sufficient area to manage pit optimizations and associated pit slopes. The block models were built using GEOVIA GEMS version 6.7.2.

14.3.6.1 BLOCK MODEL PARAMETERS – FALAGOUNTOU WEST

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

TABLE 14-28 BLOCK MODELS SETTINGS – FALAGOUNTOU WEST

Axis	Origin and Rotation (m)	Block Size (m)	Number of Blocks	
X	192,200	10	270	Columns
Y	1,589,500	15	135	Rows
Z	350	5	70	Level
Rotation	0			

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 FALA_Dec2015 and EAST_Dec2015 in the standard folder.

TABLE 14-29 LIST OF ATTRIBUTES - FALAGOUNTOU WEST

Folder Name	Model Name	Description
Standard	Alt Type	Weathering profile coding (saprolite, transition, rock)
	Min Type	Domain coding (mineralized zones)
	Rock Type	Geological coding (intrusive, sedimentary, overburden)
	Density	Specific gravity
	AuCapFINAL	Inverse Distance Cubed gold grades (g/t)
	AuCap ID2	Inverse Distance Squared gold grades (g/t)
	AuCap OK	Ordinary Kriging gold grades (g/t)
	AuCap NN	Inverse Distance power 20 gold grades (g/t)
	Pass	Interpolation pass (AuCapFINAL)
	CATEG	Resource categorization
	KRIGVAR	Ordinary Kriging variance (AuCap OK)

14.3.6.2 BLOCK MODEL PARAMETERS – FALAGOUNTOU EAST

Due to the 60° azimuth orientation of drill lines at Falagountou East, the block model was rotated to be parallel with the drill sections. The block model parameters are summarized in Table 14-30.

TABLE 14-30 BLOCK MODELS SETTINGS – FALAGOUNTOU EAST

Axis	Origin and Rotation (m)	Block Size (m)	Number of Blocks	
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 14-31 presents the list of attributes found in the block model projects EAST_MAR17_2. A percent-style block model was adopted due to the narrow nature of the grade shells. A weighted-average value for Au was produced using the percentages and grades from the four estimation domains (10, 20, 30, and 40) for reporting.

TABLE 14-31 LIST OF ATTRIBUTES - FALAGOUNTOU EAST

Folder Name	Model Name	Description
Standard	Weat_Type	Weathering profile coding (Ovb, Sap, Trans, Fresh Rock)
	Density	Specific gravity
	AU_LF_WA_New	Inverse distance cube gold grades (g/t) - Weighted average
	CATEG	Resource classification
Dom LF ##	Au_LF##	Inverse distance cube gold grades (g/t) - 10, 20, 30, 40
	Pass_##	Interpolation pass (AU_LF##) - 10, 20, 30, 40
	CMP#	Composites used in the interpolation – 10, 20, 30, 40
	LF_##	Domain coding (mineralized zones) - 10, 20, 30, 40
	Perc_##	Percentage of material in each domain (%)

14.3.6.3 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 East deposits, which were designed based on the geological structure and gold assay results and used as hard boundaries.

14.3.6.4 DENSITY MODEL

For Falagountou West, 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 shown in Table 14-32. 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. Parameters used in this single interpolation run are summarized in Table 14-33. The final density value was obtained by the combination of these two density calculation techniques, with a higher precedence to those values estimated by interpolation.

TABLE 14-32 BACKGROUND DENSITY VALUES USED IN THE MODEL – FALAGOUNTOU WEST

Weathering Profile	Code	Density (t/m^3)
Saprolite	6000	1.87
Trans-1	6100	2.08
Trans-2	6200	2.29
Trans-3	6300	2.38
Rock-1	6400	2.69
Rock-2	6500	2.74

TABLE 14-33 DENSITY INTERPOLATION PARAMETERS – FALAGOUNTOU WEST

Parameter	Interpolation Method
Calculation Method	Inverse Distance Squared
Point-Area Workspace	Dens/DENS_Oct2015
Search Ellipse	25 m x 25 m x 5 m
Target Rock Codes	6000 to 6500 inclusively
Minimum Number of Samples	1
Maximum Number of Samples	12
Outliers excluded	Below 1.0 and above 5.0 t/m^3

A summary of basic statistics of block model density values for all weathering profiles for Falagountou West is displayed in Table 14-34. All Transition-1 material was transferred in the saprolite weathering profile based on field observations. All blocks with 99% of their volume above the topography surface were coded as “Air”, with a density of 0.0 t/m^3.

TABLE 14-34 BASIC STATISTICS OF BLOCK MODEL DENSITY BY WEATHERING PROFILE – FALAGOUNTOU WEST

Weathering Profile	Code	Density Minimum (t/m^3)	Density Maximum (t/m^3)	Density Average (t/m^3)
Regolith/Saprolite	6000-6100	1.43	2.89	1.96
Transition	6200-6300	1.43	2.89	2.34
Rock	6400-6500	1.73	3.00	2.74

For Falagountou East, density values were assigned by weathering code as described in Table 14-26. No interpolation of density values was undertaken.

14.3.6.5 GRADE ESTIMATION METHODOLOGY

The final interpolation method selected for the Falagountou deposits is ID^3. OK, ID^2, and Nearest Neighbour (NN or ID^{20}) methods were also tested to compare with the ID^3 method. The ID^3 method was judged to be the most suitable to replicate composite grades throughout the Falagountou West and Falagountou East 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 6.7.2 software was used for the estimate.

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

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

- **Second Pass:** A minimum of three and a maximum of 30 composites within the search ellipse ranges. A maximum of two composites per hole were 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 two composites per hole were 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 sample search approach used to estimate the blocks for the Falagountou East deposit is summarized below:

- **First Pass:** A minimum of seven and a maximum of 20 composites within the search ellipse ranges. A maximum of three composites per hole were used for any block estimate.

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

For the first, second, and third passes, restrictions on the search ellipse ranges were applied on very high grade composites to limit their influence. This measure is judged to be prudent since the continuity of the higher grade values, within the domains, is still to be confirmed. This limit, or high grade threshold, ensures that the higher grade composites are only selected within the ranges of the half search ellipse before being used for the interpolation estimation. The high grade thresholds were chosen based on the statistical analysis of the 2.5 m composites presented earlier.

The various profiles of interpolation and search ellipses for gold composites utilized in the estimation of the resources are tabulated in Tables 14-35 to 14-38 for the Falagountou West and East deposits. The high-grade thresholds affecting the ranges of the search ellipsoid are presented by domain in Table 14-39 (Falagountou West deposit only). A single high-grade threshold value of 20 g/t Au was used for all domains of the Falagountou East deposit.

TABLE 14-35 INTERPOLATION PROFILE SETTINGS

Deposit	Profile Name	Pass	Sample Min	Sample Max	Sample Max per Hole	Target Rock Code	Ellipses Name
WEST	AUCAP_P1	1	7	30	3	See List of Rock Codes	See Table Naming of Search Ellipse Profiles (Table 14-36)
	AUCAP_P2	2	3	30	2		
	AUCAP_P2	3	1	30	2		
EAST	LF#_1	1	7	20	3	10, 20, 30, 40	EAST_1
	LF#_2	2	4	20	3		EAST_2
	LF#_3	3	1	20	3		EAST_3

TABLE 14-36 SEARCH ELLIPSE NAMES - FALAGOUNTOU WEST DEPOSIT

Rock Codes	Ellipse Profile		
	Pass 1	Pass 2	Pass 3
220	220_1	220_2	220_3
230	230_1	230_2	230_3
240	240_1	240_2	240_3
245	245_1	245_2	245_3
250	250_1	250_2	240_3
255	255_1	255_2	255_3
260	260_1	260_2	260_3
270	270_1	270_2	270_3
275	275_1	275_2	275_3
280	280_1	280_2	280_3
290	OMNI_1	OMNI_2	OMNI_3

TABLE 14-37 SEARCH ELLIPSOID SETTINGS - FALAGOUNTOU WEST DEPOSIT

Ellipse Profile Name(2)	Pass	Rotation			Anisotropy Range (m)			High Grade Range (m)(1)		
		Z	X	Z	X	Y	Z	X	Y	Z
###_1	1				25	25	25	12.5	12.5	12.5
###_2	2	0	0	0	50	50	50	25	25	25
###_3	3				100	100	100	50	50	50

Notes:
1. Refer to Table 14-39 for High Grade Transition Limit
2. Refer to Table 14-36 for Search Ellipse Profile Names

TABLE 14-38 SEARCH ELLIPSOID SETTINGS - FALAGOUNTOU EAST DEPOSIT

Ellipse Profile Name	Pass	Rotation			Anisotropy Range (m)			High Grade Transition (g/t Au)	High Grade Range (m)		
		Z	X	Z	X	Y	Z		X	Y	Z
EAST_1	1				25	25	10		12.5	12.5	5
EAST_2	2	0	40	0	50	50	15	20.0	25	25	15
EAST_3	3				100	100	15		25	25	15

TABLE 14-39 HIGH GRADE TRANSITION VALUES USED IN THE SEARCH ELLIPSE PROFILES - FALAGOUNTOU WEST DEPOSIT

Rock Codes	High Grade Transition (g/t Au)
220	2.00
230	4.00
240	5.50
245	4.00
250	8.00
255	15.00
260	6.00
270	7.50
275	6.00
280	4.50
OMNI	-

14.3.7 CLASSIFICATION AND RESOURCE REPORTING

The Mineral Resource estimate was classified in accordance with the CIM (2014) definitions (see “Classification and Resource Reporting” under “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 Falagountou East 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 Falagountou East, blocks that were estimated with more than eight composites within the third pass were included in the Indicated category.

Figure 14-15 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 14-15 RESOURCE CATEGORIES - FALAGOUNTOU WEST DEPOSIT

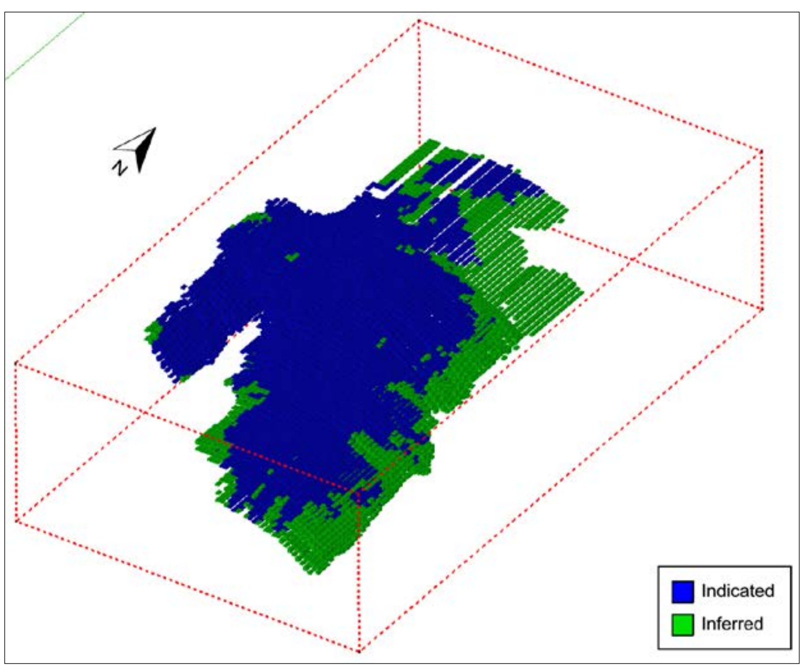
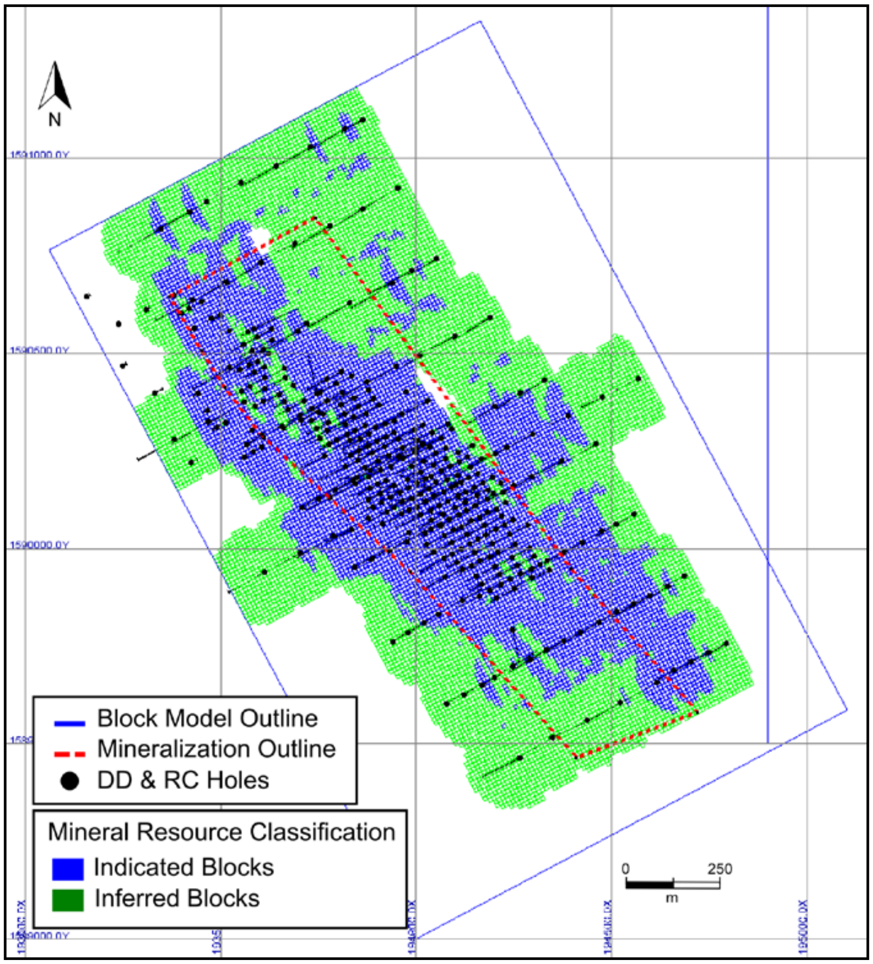


Figure 14-16 shows how the Mineral Resource categories are distributed in the Falagountou East deposit (plan view). Indicated Mineral Resources are essentially concentrated in the centre of the deposit and form half of the unconfined mineralization above 0.2 g/t Au, where the drill hole density is the highest. Inferred Mineral Resources are peripheral to Indicated, but mainly located on the northern and southern edges of the deposit where drill spacing is wider.

FIGURE 14-16 RESOURCE CATEGORIES - FALAGOUNTOU EAST DEPOSIT – LEVEL 245



14.3.8 BLOCK MODEL VALIDATION

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

14.3.8.1 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 ID^3 based Mineral Resource estimate was found to be a good representation of the drill hole composites.

14.3.8.2 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 Falagountou East deposits. Tables 14-40 and 14-41 summarize statistics between composite grades used in the interpolation process and grades of blocks interpolated for Falagountou West and Falagountou East deposits, in that order. Lower grades in the blocks of the models, compared to composites, are probably caused, in part, by the high grade transition applied to very high grade composites. Also, composites of lower grade in the periphery of mineralized zones may also influence a proportionally higher number of blocks, resulting in a lower global mean grade. This is partially demonstrated by a lower grade difference when comparing composite grades with grades from passes 1 and 2 only. Attempts were made to correct this potential bias. GMSI suggests waiting for robust reconciliation data before making any important modifications to the block model.

TABLE 14-40 AVERAGE COMPOSITE VERSUS BLOCK GRADES - FALAGOUNTOU WEST DEPOSIT

Zone	Number of Blocks	Mean Composite Grade (g/t Au)	Mean Block Grade (g/t Au)
220	1,117	0.28	0.24
230	1,485	0.27	0.20
240	2,527	0.50	0.42
245	640	0.48	0.39
250	3,172	0.96	0.89
255	5,618	1.55	1.14
260	2,724	0.45	0.39
270	715	1.48	1.28
275	196	0.78	0.70
280	393	0.55	0.49
290	348	0.89	0.81
Total	18,935	0.93	0.72

TABLE 14-41 AVERAGE COMPOSITE VERSUS BLOCK GRADES PER ESTIMATION DOMAIN - FALAGOUNTOU EAST DEPOSIT

Zone	Number of Blocks	Mean Composite (g/t Au)	Mean Block Grade (g/t Au)
20	28859	0.506	0.484
30	11441	1.019	0.992
40	8282	2.916	2.808

14.3.8.3 VALIDATION USING DIFFERENT INTERPOLATION METHODS

For Falagountou West, the validation of the block model was also carried out using different interpolation methods: OK, ID^2, and NN (or ID^{20}). The same set of composites, search ellipses, and settings were used for the different interpolations and only the estimation method differed. Tables 14-42 presents the results of confined resources with the OK interpolator as a comparison for the Falagountou West deposit. While the ID^3 method interpolates higher grades, the OK method yields more tonnage for a combined Measured, Indicated, and Inferred Mineral Resource difference of 6% of gold ounces in favour of the inverse distance technique. GMSI is of the opinion that the ID^3 interpolation method is a better global estimator compared to the OK technique, given the globally higher grades.

TABLE 14-42 COMPARISON OF ID^3 VERSUS OK INTERPOLATIONS - FALAGOUNTOU WEST DEPOSIT

			Inverse Distance Cubed			Ordinary Kriging		
Weathering Zone	**Resource Category**	**Au Cut-off Grade (g/t)**	**Tonnage (kt)**	**Au Grade (g/t)**	**Gold (koz)**	**Tonnage (kt)**	**Au Grade (g/t)**	**Gold (koz)**
Saprolite	Indicated	0.39	107	1.46	5	111	1.29	5
	Inferred		0	0.00	0	0	0.00	0
Transition	Indicated	0.43	885	1.24	35	969	1.11	34
	Inferred		14	0.64	0	14	0.82	0
Rock	Indicated	0.48	9,995	1.61	517	10,505	1.44	485
	Inferred		100	1.06	3	74	0.82	2

No other interpolation techniques were considered for Falagountou East.

14.3.8.4 SWATH PLOTS

For Falagountou West, swath plots were generated to assess the correlation between composites used in the interpolation of each block versus the total gold content estimated in the blocks. Swath plots were produced in vertical sections. This validation method works as a visual means to identify possible bias in the interpolation (e.g., a section with a significantly high gold content based on a low population of composites). In general, gold contained in each vertical section should correlate well with the amount of composite used in the interpolation.

For Falagountou East, swath plots were produced which directly compare gold grades (Au g/t) between the composites and the blocks within Indicated category blocks. This provides an indication of local grade reproducibility.

Figures 14-17 and 14-18 illustrate swath plots for Indicated Mineral Resources by vertical sections for the Falagountou West and Falagountou East deposits, respectively (Inferred Mineral Resources being marginal). Swath plots were created by grouping saprolite, transition, and rock weathering profiles. Peaks and lows in gold content generally match peaks and lows in composite frequency; no bias was found in the resource estimate in this regard.

FIGURE 14-17 SWATH PLOT OF INDICATED RESOURCES - FALAGOUNTOU WEST

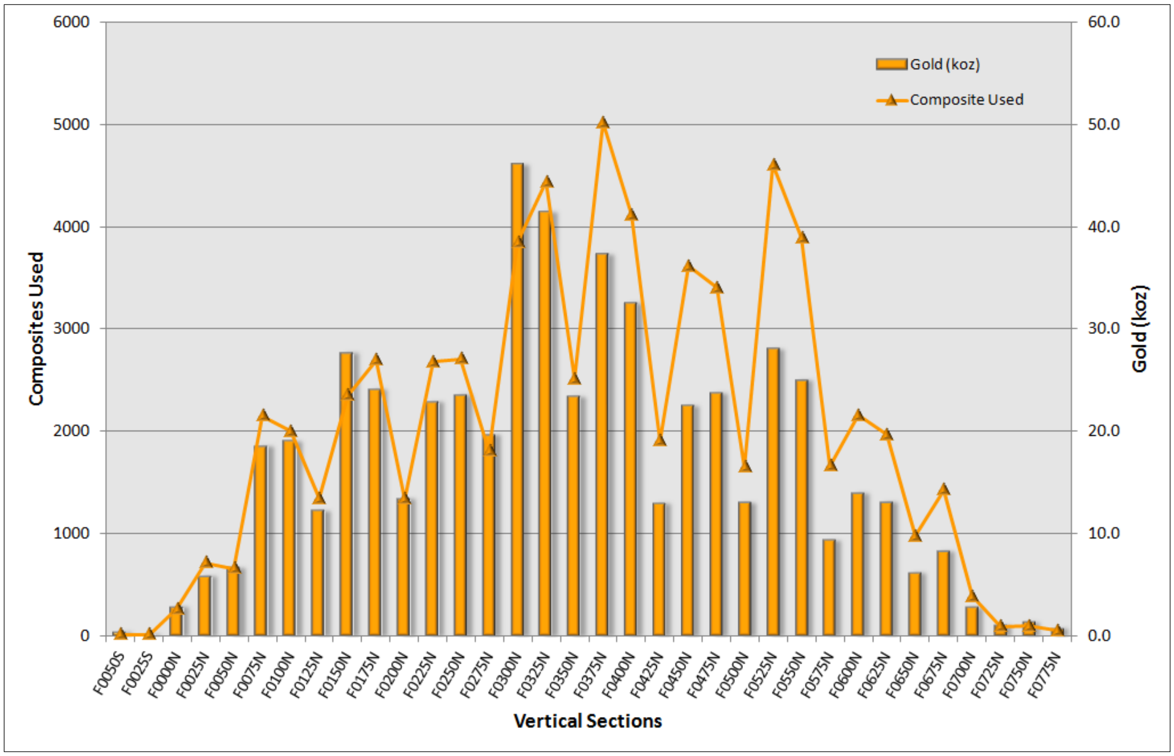
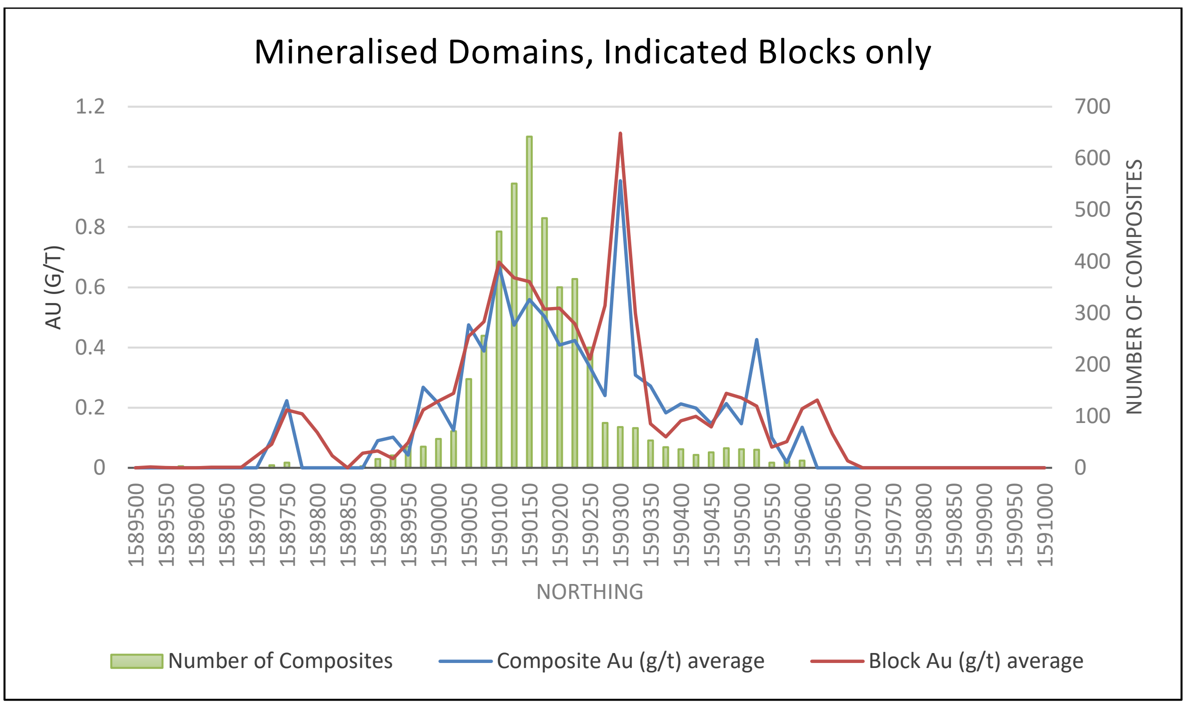


FIGURE 14-18 SWATH PLOT OF INDICATED RESOURCES - FALAGOUNTOU EAST



14.4 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 rock) was determined in this process.

14.4.1 OPTIMIZATION PARAMETERS

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

14.4.2 OPEN PIT CONSTRAINED MINERAL RESOURCES

The compilation of the Mineral Resource was carried out with the projected Mined Surfaces for June 5, 2018 for the EMZ and Falagountou West deposits.

The EMZ deposit open pit Indicated Mineral Resources are estimated to be approximately 150 Mt grading 0.91 g/t Au containing 4,340 koz of gold, including 320 koz of gold stored in stockpiles. The EMZ deposit open pit Inferred Mineral Resources are estimated to be approximately 18.9 Mt at 0.78 g/t Au containing 474 koz.

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

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

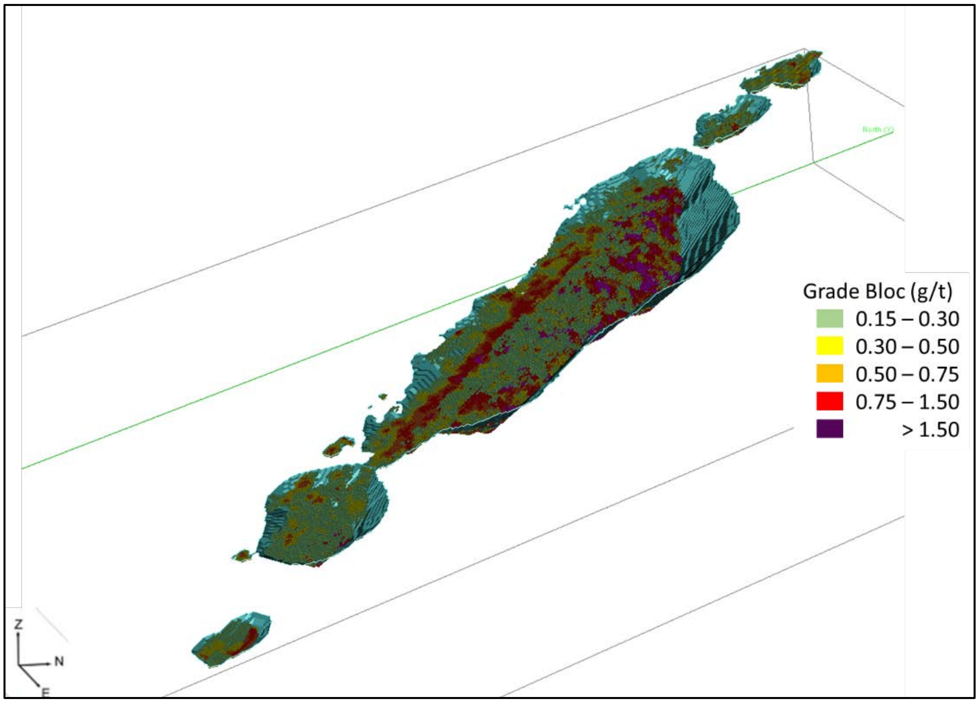
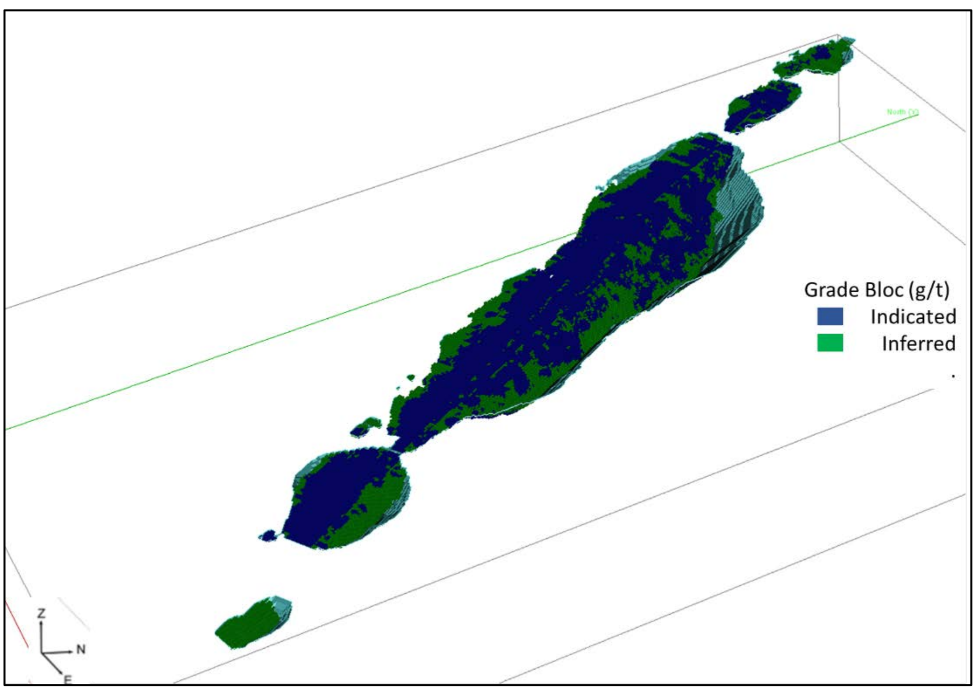


FIGURE 14-20 ISOMETRIC VIEW OF EMZ DEPOSIT RESOURCE CLASSIFICATION INSIDE US$1,500/OZ AU WHITTLE PIT SHELL



The Falagountou deposit (East and West combined) open pit Indicated Mineral Resource is estimated to be 10.7 Mt at an average grade of 1.56 g/t Au, totalling 539 koz of gold. The Falagountou deposit (East and West combined) open pit Inferred Mineral Resource is estimated to be 1.8 Mt at an average grade of 2.00 g/t Au, totalling 115 koz 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 East deposits, respectively.

Total Indicated Mineral Resources at the Essakane Gold Mine are currently estimated to be 159 Mt grading 0.95 g/t Au, totalling 4,878 koz of gold, while Inferred Mineral Resources are estimated to be 21.0 Mt grading 0.88 g/t Au, totalling 589 koz of gold. IAMGOLD's attributable Mineral Resources are 144 Mt totalling 4,390 koz of gold in Indicated Mineral Resources and 18.7 Mt totalling 530 koz of gold in Inferred Mineral Resources.

Table 14-43 shows the projected stockpile status for June 5, 2018.

TABLE 14-43 STOCKPILE STATUS AS OF JUNE 5, 2018

Material Type	Stockpiles	Tonnage (kt)	Grade (g/t Au)	Ounces (koz)
Saprolite	-	94	0.24	0.73
Transition	Stock LG	8,792	0.54	152
Fresh Rock	Stock LG	4,297	0.67	92
	Stock Marginal	3,902	0.24	59
	Stock Heap Leach	137	0.34	1.49
	Stock HG (Falagountou)	33	1.22	1.30
	Stock LG (Falagountou)	528	0.78	13.30
	ROM Pad	10	1.61	0.54
	Primary Crusher	260	1.08	9.02
	Total (Fresh Rock)	9,168	0.71	177
Total Stockpiles		**18,054**	**0.61**	330

Note. LG – low grade; HG – high grade

Details of the resource estimate are given in Table 14-44. 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 CONSTRAINED MINERAL RESOURCES: (A) GOLD GRADES AND (B) RESOURCE CATEGORIES - FALAGOUNTOU WEST DEPOSIT

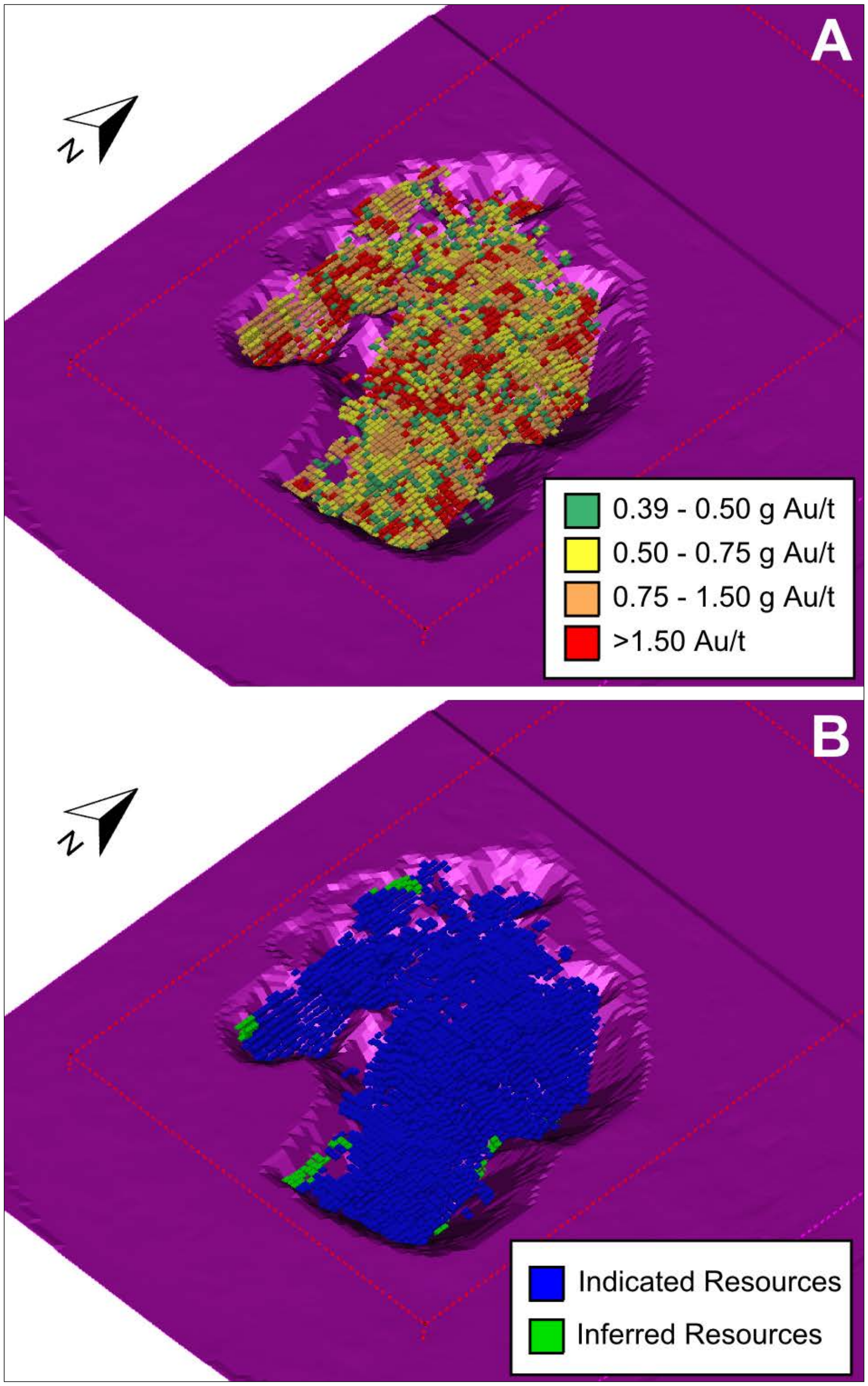


FIGURE 14-22 CONSTRAINED MINERAL RESOURCES: (A) GOLD GRADES AND (B) RESOURCE CATEGORIES - FALAGOUNTOU EAST DEPOSIT

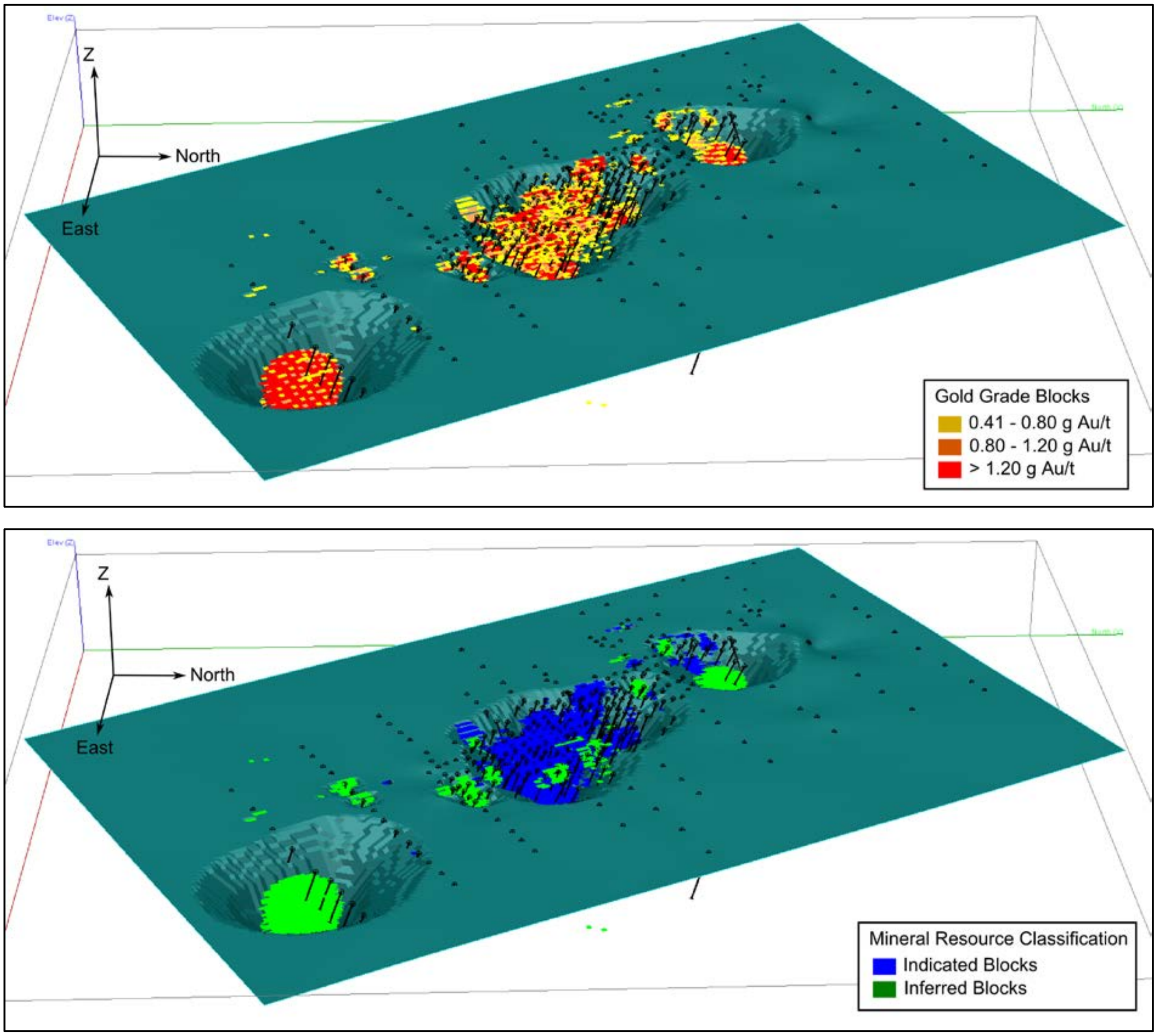


TABLE 14-44 ESSAKANE GOLD MINE JUNE 5, 2018 CONSOLIDATED MINERAL RESOURCES

Material Type	Saprolite			Transition			Fresh Rock			All Material		
Resource Category	**Tonnes (000 t)**	**Grade (g/t Au)**	**Ounces (000 oz)**	**Tonnes (000 t)**	**Grade (g/t Au)**	**Ounces (000 oz)**	**Tonnes (000 t)**	**Grade (g/t Au)**	**Ounces (000 oz)**	**Tonnes (000 t)**	**Grade (g/t Au)**	**Ounces (000 oz)**
Essakane Main Zone (EMZ)												
Cut-off Grade	**0.33 g/t Au**			**0.43 g/t Au**			**0.30 g/t Au**					
Measured	-	-	-	-	-	-	-	-	-	-	-	-
Indicated	1,345	0.46	20	917	0.62	18	128,770	0.96	3,970	131,031	0.95	4,009
Total Measured & Indicated	**1,345**	**0.46**	**20**	**917**	**0.62**	**18**	**128,770**	**0.96**	**3,970**	**131,031**	**0.95**	**4,009**
Stockpiles	94	0	1	8,792	0.54	152	9,168	0.60	177	18,054	0.57	330
Total EMZ M&I Resources	**1,439**	**0.4**	**21**	**9,708**	**0.55**	**171**	**137,938**	**1**	**4,147**	**149,085**	**1**	**4,339**
EMZ Inferred	460	0.54	8	195.26	1	3.97	18,296	1	462	18,952	1	474
Falagountou												
Cut-off Grade	**0.36 g/t Au**			**0.46 g/t Au**			**0.52 g/t Au**					
Measured	-	-	-	-	-	-	-	-	-	-	-	-
Indicated	1,209	1.05	41	665	1.18	25	8,851	1.66	473	10,725	1.56	539
Total Measured & Indicated	**1,209**	**1.05**	**41**	**665**	**1.18**	**25**	**8,851**	**1.66**	**473**	**10,725**	**1.56**	**539**
Total Falagountou M&I Resources	**1,209**	**1.05**	**41**	**665**	**1.18**	**25**	**8,851**	**1.66**	**473**	**10,725**	**1.56**	**539**
Falagountou Inferred	308	1.08	11	83	1.51	4	1,401	2.23	100	1,792	2.00	115
Consolidated Essakane Resources (EMZ & Falagountou)												
M&I Resources	2,648	0.72	61	10,373	0.59	196	146,789	0.98	4,621	159,810	0.95	4,878
Inferred Resources	769	0.76	19	279	0.90	8	19,697	0.89	562	20,744	0.88	589
Attributable M&I Resources (90%)	2,383	0.72	55	9,335	0.59	176	132,110	0.98	4159	143,829	0.95	4,390
Attributable Inferred Resources (90%)	692	0.76	17	251	0.90	7	17,727	0.89	506	18,670	0.88	530

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.

14.4.3 CONSTRAINED MINERAL RESOURCE SENSITIVITY TO CUT-OFF GRADE

14.4.3.1 EMZ DEPOSIT

The sensitivity analysis presents the 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 0.20 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: 0.33 g/t Au for saprolite, 0.43 g/t Au for transition, and 0.3 g/t Au for fresh rock. The Mineral Resources, as detailed in Table 14-45, are constrained below the mining surface as of December 31, 2017 and inside the US$1,500/oz Au Whittle pit shell optimized on Indicated and Inferred Mineral Resources.

TABLE 14-45 CONSTRAINED MINERAL RESOURCE(1) SENSITIVITY TO SELECTED CUT-OFF GRADES - EMZ

Cut-Off Grades (g/t Au)	Indicated			Inferred		
	Tonnage (kt)	Grade (g/t Au)	Ounces (koz)	Tonnage (kt)	Grade (g/t Au)	Ounces (koz)
>2.00	29,165	1.45	1,355	708	2.71	62
>1.50	39,180	1.52	1,909	1,738	2.12	118
>1.25	47,068	1.49	2,256	2,755	1.84	163
>1.00	59,111	1.41	2,689	4,141	1.60	212
>0.80	73,510	1.31	3,102	6,005	1.38	266
>0.60	94,612	1.17	3,573	9,323	1.13	339
Variable(2)	149,085	0.91	4,339	18,952	0.78	474
>0.20	175,906	0.81	4,558	24,483	0.66	520

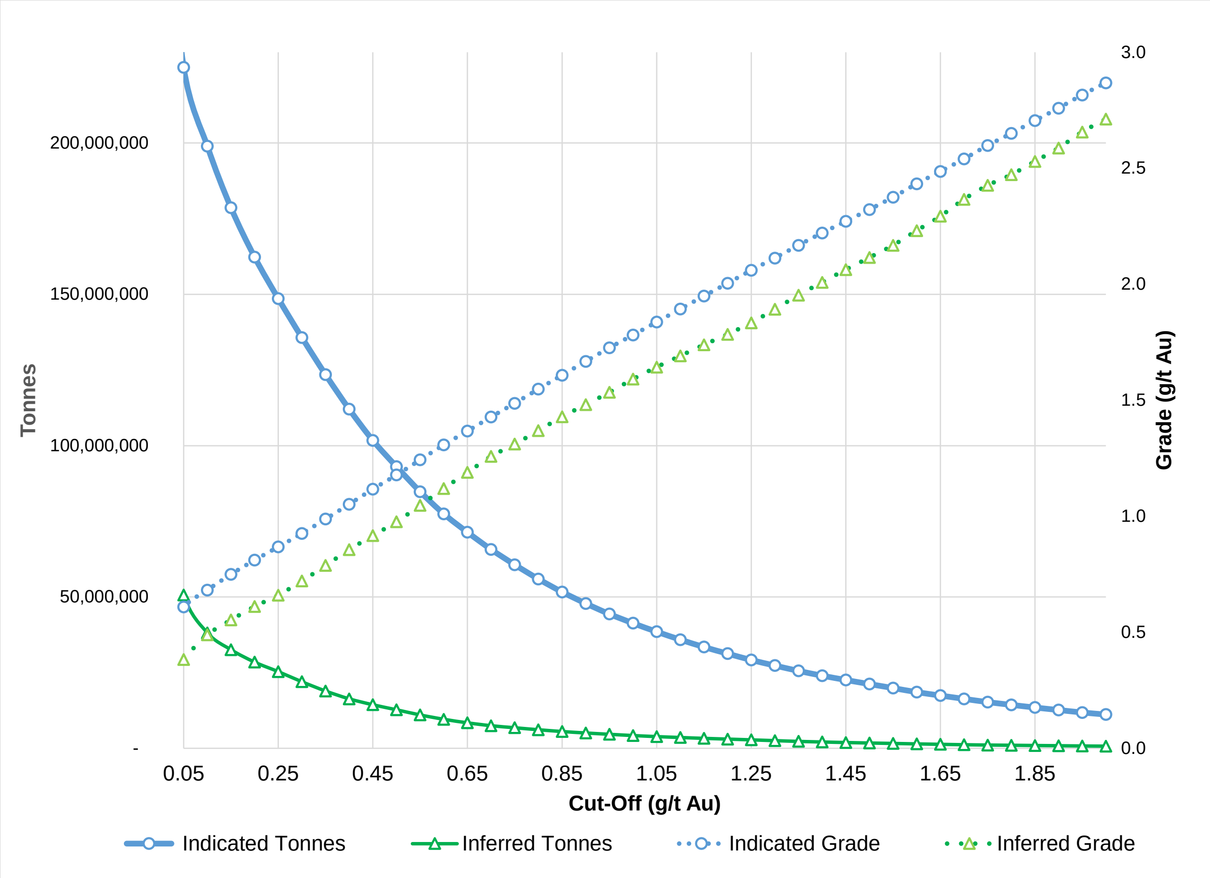
Notes:

1. Mineral Resources are constrained below the mining surface as of June 5, 2018 and inside the US$1,500/oz Au Whittle pit shell optimized on Indicated and Inferred Mineral Resources.
2. June 5, 2018 Mineral Resource Cut-off Grades : Saprolite 0.33 g/t Au; Transition 0.43 g/t Au; and Rock 0.30 g/t Au.

Figure 14-23 shows grade-tonnage curves for Indicated and Inferred Mineral Resources versus cut-off grade. 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



14.4.3.2 FALAGOUNTOU DEPOSITS

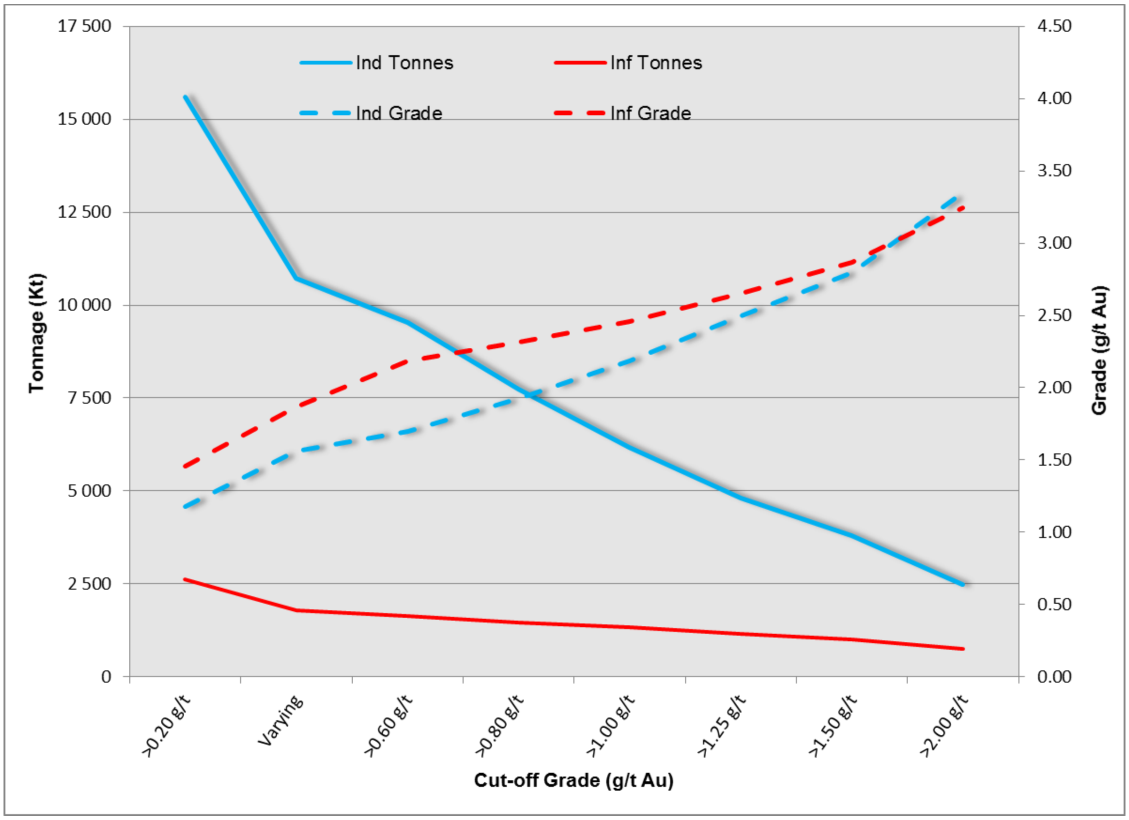
Table 14-46 summarizes the sensitivity of the constrained open pit Mineral Resources of the Falagountou deposits (West and East combined) for a series of selected gold cut-off grades. The sensitivity analysis uses gold cut-off grades between 0.20 g/t Au and 2.0 g/t Au. Figure 14-24 illustrates the grade-tonnage curves for the Indicated and Inferred Mineral Resources for the Falagountou deposits (West and East combined). The sensitivity table and graph are both a compilation of the saprolite, transition, and fresh rock weathering profiles. The Falagountou deposits do not show significant sensitivity to gold cut-off grades, expressed by relatively linear curves. The apparent kink in the Indicated tonnage curve between 0.20 g/t Au and 0.60 g/t Au is caused by the different cut-off grades used for this interval; most of the tonnage in this case comes from the "fresh rock" profile with a gold cut-off grade of 0.48 g/t Au. Gold grade sensitivity is rather linear from 0.20 g/t Au to 1.5 g/t Au.

TABLE 14-46 INDICATED MINERAL RESOURCE SENSITIVITY - FALAGOUNTOU WEST AND EAST DEPOSITS COMBINED

Cut-off Grade (g/t Au)	Indicated			Inferred		
	Tonnage (kt)	Grade (g/t Au)	Gold (koz)	Tonnage (kt)	Grade (g/t Au)	Gold (koz)
>2.0 g/t	2,492	3.35	268	743	3.24	78
>1.5 g/t	3,785	2.79	340	992	2.87	91
>1.25 g/t	4,796	2.49	385	1,156	2.65	99
>1.00 g/t	6,184	2.19	434	1,324	2.46	105
>0.80 g/t	7,741	1.93	479	1,462	2.31	109
>0.60 g/t	9,525	1.70	519	1,638	2.18	115
Varying	10,725	1.56	539	1,792	1.86	107
>0.2 g/t	15,604	1.18	590	2,630	1.46	123

Notes:
1. Mineral Resources are constrained below the mining surface as of June 5, 2018 and inside the US$1,500/oz Au Whittle pit shell optimized on Indicated and Inferred Mineral Resources (Falagountou West and East deposits combined).
2. December 31, 2015 Mineral Resource Cut-off Grades: Saprolite: 0.36 g/t Au, Transition: 0.46 g/t Au, and Fresh Rock: 0.52 g/t Au.

FIGURE 14-24 GRADE - TONNAGE CURVES OF CONSTRAINED INDICATED MINERAL RESOURCE – FALAGOUNTOU WEST AND EAST COMBINED



14.5 SENSITIVITY TO GOLD PRICE

14.5.1 ESSAKANE GOLD MINE (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 14-47, per deposit.

FIGURE 14-25 EMZ AND FALAGOUNTOU DEPOSITS INDICATED AND INFERRED MINERAL RESOURCE SENSITIVITY TO GOLD PRICES (SAP+TRANS+ROCK)

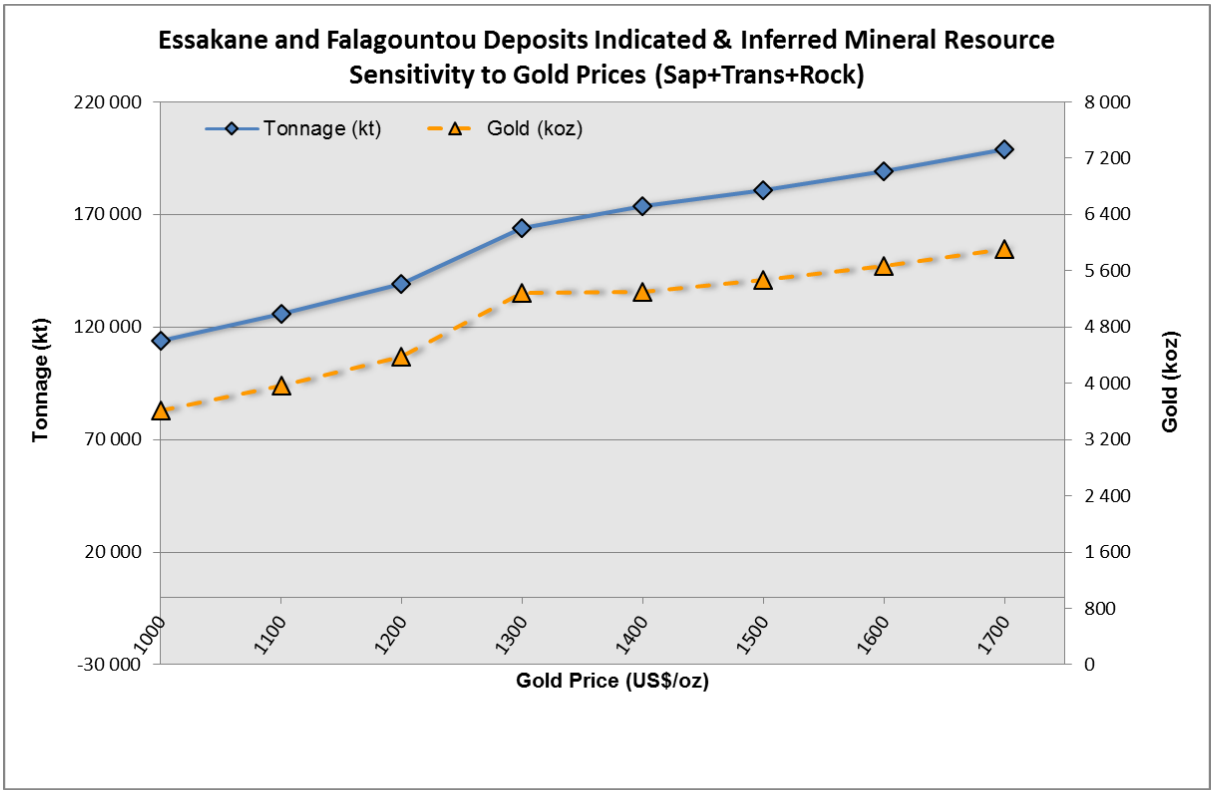


TABLE 14-47 EMZ AND FALAGOUNTOU DEPOSITS CUT-OFF GRADES FOR VARYING GOLD PRICES

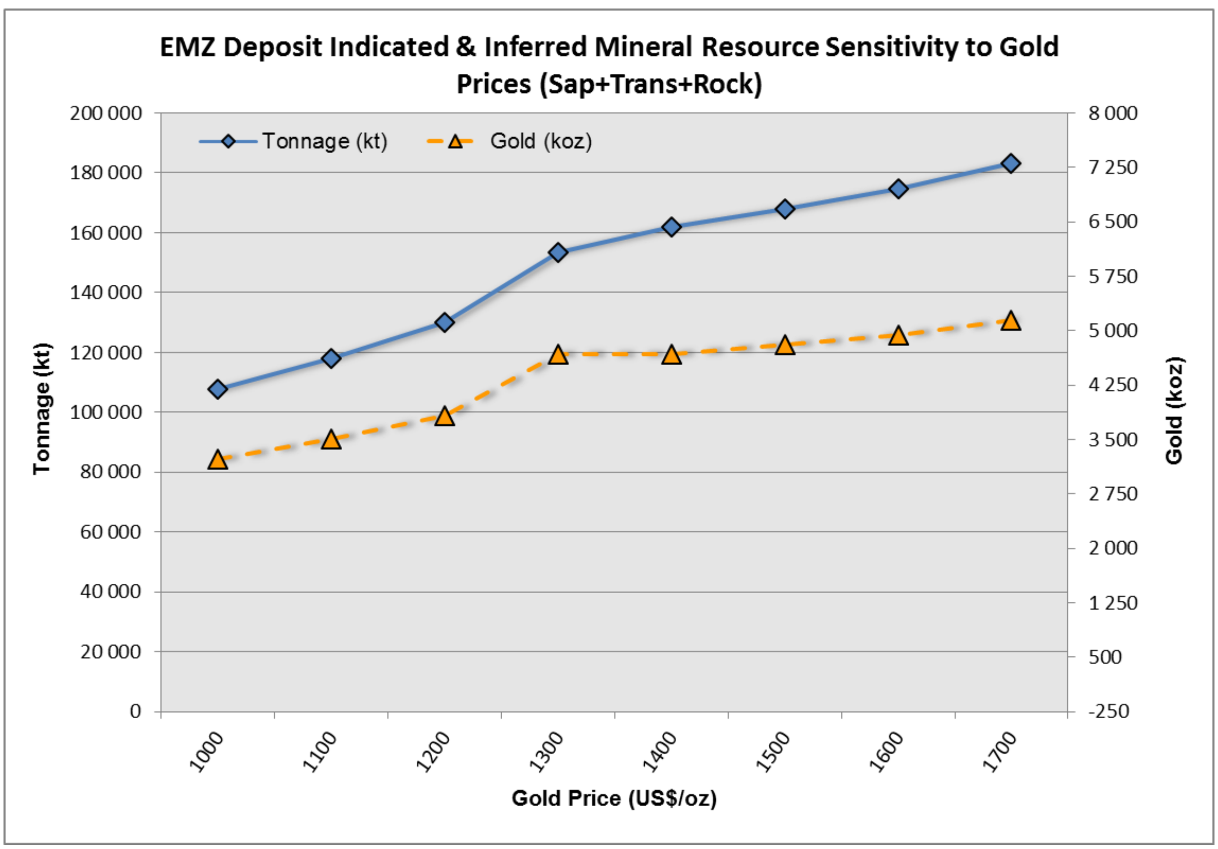
Gold Price (US$/oz)	EMZ Deposit Cut-off Grades (g/t Au)			Falagountou Deposit Cut-off Grades (g/t Au)		
	Sap	Trans	F. Rock	Sap	Trans	F. Rock
1,000	0.49	0.64	0.30	0.53	0.68	0.77
1,100	0.45	0.58	0.30	0.48	0.62	0.70
1,200	0.41	0.53	0.30	0.44	0.56	0.64
1,300	0.38	0.49	0.30	0.41	0.53	0.60
1,400	0.35	0.46	0.30	0.38	0.49	0.55
1,500	0.33	0.43	0.30	0.36	0.46	0.52
1,600	0.31	0.40	0.30	0.34	0.43	0.48
1,700	0.29	0.38	0.30	0.32	0.40	0.46

14.5.2 EMZ DEPOSIT

The sensitivity of the 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 December 31, 2017 and within the corresponding gold price Whittle pit shell and cut-off grades. The stockpiles are also included in the sensitivity analysis.

A rapid increase in tonnage and gold content at lower gold prices is followed by a steady increase of resources between gold prices of US$1,100/oz and US$1,600/oz. 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 +33% and a lower increase of +27% 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.

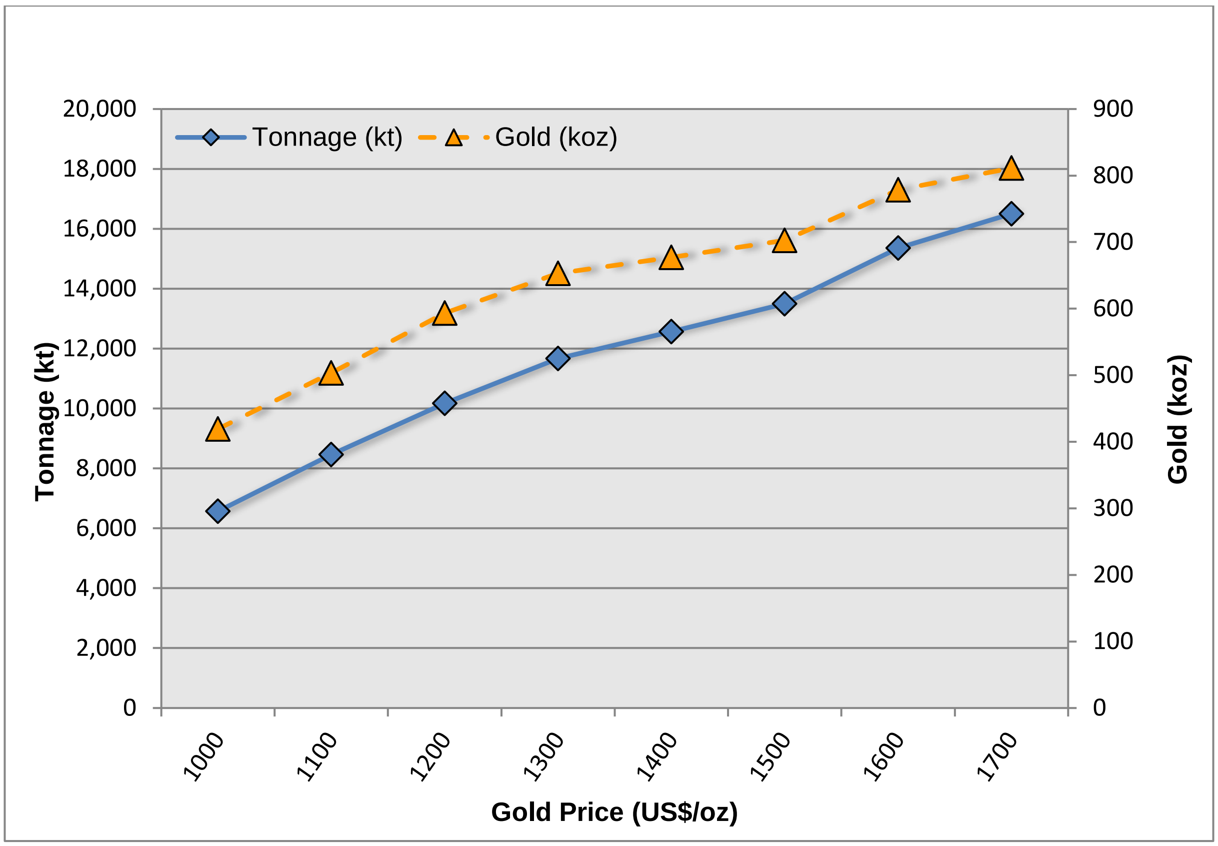
FIGURE 14-26 EMZ DEPOSIT INDICATED AND INFERRED MINERAL RESOURCES SENSITIVITY TO GOLD PRICE



14.5.3 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 14-47. The Falagountou West and East 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 32% in tonnage and 17% in gold content and a decrease of 11% in grade are anticipated.

FIGURE 14-27 MINERAL RESOURCE SENSITIVITY TO GOLD PRICE - FALAGOUNTOU DEPOSITS



14.6 COMPARISON TO PREVIOUS MODELS

14.6.1 EMZ DEPOSIT

Table 14-48 compares the Indicated and Inferred Mineral Resource estimates as of June 5, 2018 to those of December 31, 2017. As of December 31, 2017, the EMZ deposit Indicated Mineral Resources were estimated to total 3,681 koz of gold and increased by 657 koz of gold to a total estimate of 4,339 koz of gold as of June 5, 2018. Inferred Mineral Resources gained a total of 203 koz of gold from 271 koz of gold in 2017 to 474 koz of gold in 2018.

The variations expressed as differences in tonnage, grade, and ounces in Table 14-48 are influenced by multiple factors, such as various changes in the block model, update of optimization parameters and costs, mining depletion, and stockpiles variations.

The block model modifications resulted from the addition of new drill hole results. The current model links together the north, the main, and the south of Essakane by using the same litho-structural domains as the previous model. The previous model consisted of three unlinked models, and to modify it, 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 of the nose and in the limbs of the fold in order to complete variography in accordance with the location of the data within the fold.

Overall, the estimation strategy did not change from the previous model but variography was redone within the updated wireframe. The boundaries strategies between layers has been adjusted with the new layers. In order to reduce smoothing of the estimation, the maximum number of composites used for the estimation was reduced 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 density model was updated, however, the adjustments did not significantly affect the tonnages.

Also, when comparing to the December 31, 2017 block model, some changes in the resource classification were applied. Considering the unified model created in the last year, classification strategy from the EMZ was applied to Essakane South and North as well.

The differences in the EMZ deposit Mineral Resources also account for losses due to the 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 grade used for the heap leach project.

TABLE 14-48 COMPARISON OF MINERAL RESOURCES AS OF JUNE 5, 2018 TO MINERAL RESOURCES AS OF DECEMBER 31, 2017 EMZ DEPOSITS

		Saprolite			Transition			Fresh Rock			All Material		
	Material Type	**Tonnes (000 t)**	**Grade (g/t Au)**	**Ounces (000 oz)**	**Tonnes (000 t)**	**Grade (g/t Au)**	**Ounces (000 oz)**	**Tonnes (000 t)**	**Grade (g/t Au)**	**Ounces (000 oz)**	**Tonnes (000 t)**	**Grade (g/t Au)**	**Ounces (000 oz)**
Resources[1] December 31, 2017	**Cut-off grade[3]**		**0.33 g/t Au**			**0.43 g/t Au**			**0.49 g/t Au**				
	M & I	1,189	0.47	18	699	0.65	15	86,248	1.22	3,378	88,136	1.20	3,410
	Stockpiles	-	-	-	8,792	0.54	152	5,192	0.71	119	13,984	0.60	271
	Total	**1,189**	**0.47**	**18**	**9,491**	**0.55**	**167**	**91,440**	**1.19**	**3.496**	**102,120**	**1.12**	**3,681**
	EMZ Inferred	240	0.51	4	6	0.47	0	7,671	1.08	267	7,917	1.06	271
Resources[1] June 5, 2018	**Cut-off grade[3]**		**0.33 g/t Au**			**0.43 g/t Au**			**0.30 g/t Au**				
	M & I	1,345	0.46	20	917	0.62	18	128,770	0.96	3970	131,031	1	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**	**0.4**	**21**	**9,708**	**0.5**	**171**	**137,938**	**0.9**	**4147**	**149,085**	**0.9**	**4,339**
	EMZ Inferred	460	0.54	8	195	0.63	4	18,296	0.78	462	18,952	1	474
Difference	M & I	156	0.31	2	217	0.55	4	42,522	0.43	593	42,895	0.43	598
	Stockpiles	94	0.24	1	0	0	0	3,977	0.46	58	4,070	0.45	59
	Total	**250**	**0.29**	**2**	**217**	**0.55**	**4**	**46,498**	**0.44**	**651**	**46,965**	**0.44**	**657**
	EMZ Inferred	220	0.58	4	189	0.64	4	10,625	0.57	195	11,035	0.57	203

14.6.2 FALAGOUNTOU

The Falagountou West deposit was completely remodelled by GMSI in March 2015. A second update was completed by GMSI in October 2015 and mainly focussed on the Falagountou West deposit extensions and weathering profile. The updated Mineral Resource estimate was disclosed in IAMGOLD's Technical Report in 2016 (Chénard et al., 2016). This Mineral Resource estimate remains unchanged as of the effective date of June 5, 2018. The Falagountou East Mineral Resources were estimated by GMSI in August 2016, and subsequently updated in March 2017 to include infill and extensional drilling.

The current June 5, 2018 Mineral Resource estimates for the Falagountou deposits are compared with the previous 2015 Mineral Resource estimate in Table 14-49. The most significant differences between the 2015 end of year resource statement and the June 5, 2018 resource statement are a reduction of 3.8 Mt of Indicated Mineral Resource, and a 0.9 Mt gain of Inferred Mineral Resource in the fresh rock material. In addition, there has been a 1.0 Mt tonnage gain in the saprolite with a drop in grade from 1.86 g/t Au to 1.07 g/t Au, mainly attributable to the inclusion of the Falagountou East resource upgrade. A series of parameters affected the Mineral Resource estimate, including:

- Additional drilling:
  - Completely remodelled mineralization zones at Falagountou East,
  - Infill drilling refined the mineralization interpretation resulting in a reduction in volume for Indicated Mineral Resources, mainly due to a lack of grade continuity. New data included in the March 2017 resource upgrade for Falagountou East were globally 0.2 g/t Au lower than for the previous estimate (August 2016)

Modest increases in Inferred Mineral Resources were due to extensional drilling and a change in interpolation strategy:

- Weathering profile adjustments (gains and/or losses),
- Density adjustments, with new measurements and new weathering profiles

Losses in the transition material category are largely caused by the transfer of the topmost transition interval to the saprolite material class, and the simplification of the weathering model for Falagountou East.

Grades are also globally lower in the Indicated category, largely caused by the changes in the interpolation methods and the inclusion of the Falagountou East resource upgrade.

TABLE 14-49 COMPARISON OF MINERAL RESOURCES AS OF JUNE 5, 2018 TO MINERAL RESOURCES AS OF DECEMBER 31, 2015 - FALAGOUNTOU DEPOSITS (WEST AND EAST COMBINED)

		Laterite & Saprolite			Transition			Fresh Rock			Total (All Material)		
		Tonnage (kt)	**Grade (g/t Au)**	**Gold (koz)**	**Tonnage (kt)**	**Grade (g/t Au)**	**Gold (koz)**	**Tonnage (kt)**	**Grade (g/t Au)**	**Gold (koz)**	**Tonnage (kt)**	**Grade (g/t Au)**	**Gold (koz)**
December 31, 2015 (West+East)	**Cut-off grade**	**0.39 g/t Au**			**0.43 g/t Au**			**0.48 g/t Au**					
	M&I	392	1.86	23	1,484	1.44	69	13,415	1.63	702	15,290	1.61	794
	Inferred	63	2.51	5	121	3.15	12	473	2.46	38	657	2.6	55
June 5, 2018 (West+East)	**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
Difference	M&I	817	0.68	18	-819	1.66	-44	-4,564	1.56	-229	-4,565	1.74	-255
	Inferred	245	0.73	6	-38	6.56	-8	928	2.09	62	1,135	1.65	60

Notes:

1. Mineral Resources as disclosed in official December 31, 2015 year-end Mineral Resource statement.
2. Mineral Resources below mining surface as of December 31, 2015 and inside US$1,500/oz Au Whittle pit shell optimized on Measured, Indicated, and Inferred block models Fala_Dec15 and East_Dec15.
3. Cut-offs for December 31, 2015 Mineral Resources.

14.6.3 ESSAKANE GOLD MINE RESOURCE VARIATION THROUGH JUNE 2018

Factors leading to yearly changes of the Mineral Resources are summarized as follows:

- **Changes to costs**, and consequently to optimization parameters, cut-off grades, and Whittle pit shells. New parameters can be viewed in Section 15. Major changes from the December 31, 2017 estimate include:
  - More than 20% decrease in diesel and heavy fuel oil (HFO) prices,
  - 25% decrease in power costs,
  - Processing costs decrease: 15% in fresh rock and 10% in saprolite,
  - More than 50% decrease in sustaining capital costs,
  - Note that gold price assumptions for Mineral Resources remained the same at US$1,500/oz,

- **Changes to model**: while the modelling update at the EMZ deposit yielded a minor gain of resources, the remodelling and new drilling at the Essakane deposit are responsible for the addition of 81k ounces

- **Depletion**: Depletion is estimated from surveyed surface from June 5, 2018.

- **Stockpiles variation**: The stockpiles estimated for June 5, 2018 include the following piles: low grade stockpiles of transition and rock material, ROM pads (East, West and pad 2), and Primary Crushers A and B.

15 MINERAL RESERVE ESTIMATE

15.1 SUMMARY

The Mineral Reserve estimate at June 5, 2018 for the Essakane Gold Mine is summarized in Table 15-1 and is reported on a 100% basis. The Mineral Reserve was estimated using a multiple-stage process. Economic shells were designed using the Lerchs-Grossmann algorithm. Subsequently, the NPV optimization software COMET was used to accurately target the optimal shell based on several mining and economic parameters. The selected pit shell design meets the geotechnical criteria established by SRK. Finally, the validation of the Project in a financial model ensures that the Mineral Reserves in Table 15-1 are economically viable.

TABLE 15-1 MINERAL RESERVE SUMMARY – JUNE 5, 2018

Process	Category	Tonnage (000 t)	Grade (g/t Au)	Contained Metal (000 oz Au)
CIL	Proven	-	-	-
	Probable	83,716	1.27	3,415
	Stockpile	14,014	0.60	269
	Total CIL	**97,730**	**1.17**	**3,684**
Heap Leach	Proven	-	-	-
	Probable	56,427	0.42	766
	Stockpile	4,040	0.47	61
	Total Heap Leach	**60,466**	**0.43**	**827**
Total		**158,197**	**0.89**	**4,510**
Waste within Designed Pit		327,518		
Ore within Designed Pit		140,143		
Total Tonnage within Designed Pit		467,661		

Notes:

1. CIM (2014) definitions were followed for Mineral Reserves.
2. Mineral Reserves estimated assuming open pit mining methods.
3. Mineral Reserves are based on a gold price of $1,200/oz.
4. Average weighted CIL process recovery of 92.1% and Heap Leach process recovery of 55.0%.
5. Mining costs ($/t mined): $2.55/t. Processing costs: $12.36/t (CIL). Processing costs $3.13/t (HL). General and Administrative costs (includes refining cost) of $3.99/t for CIL only. Heap Leach bears no G&A costs.

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.

The mine design and Mineral Reserve estimate have been completed to a level appropriate for pre-feasibility studies. 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.

The 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

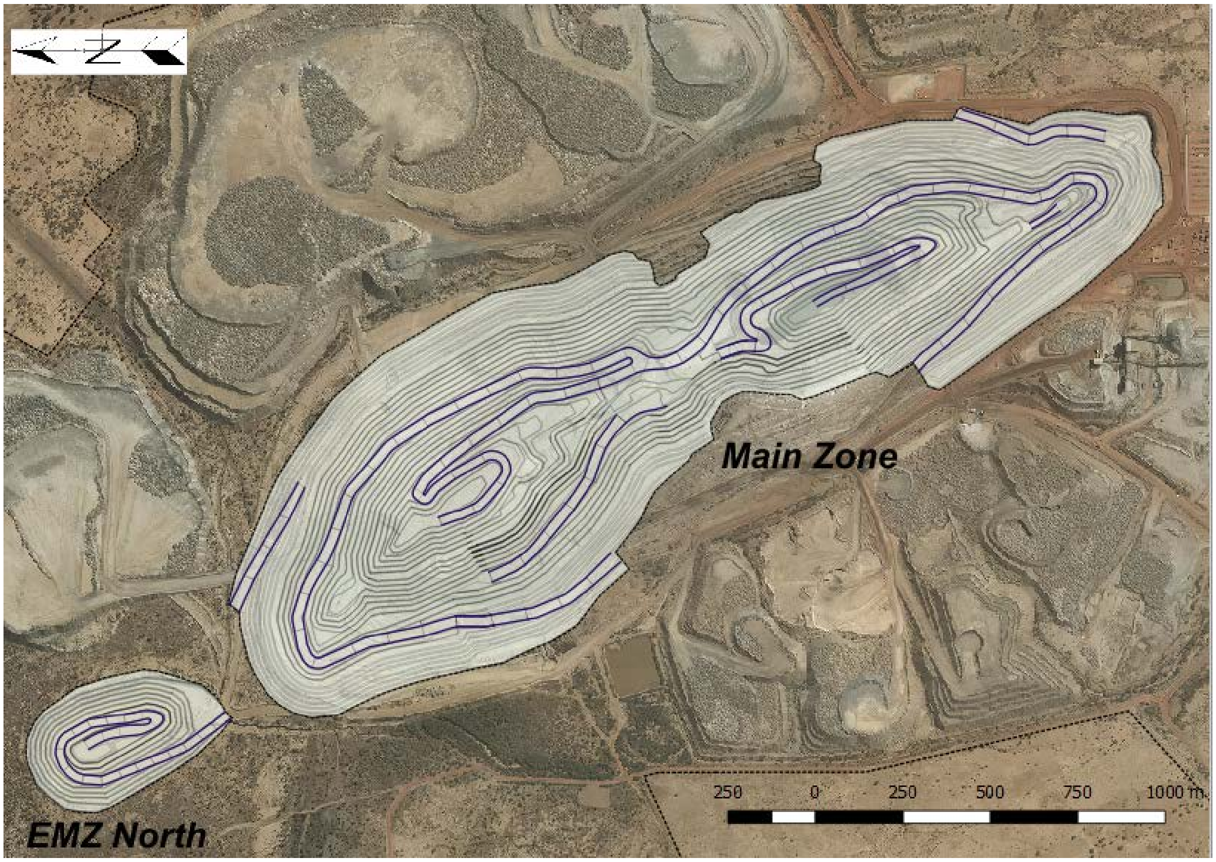
The June 5, 2018 Mineral Reserve estimate is based on separate updated resource models at year-end 2017 for the EMZ, Falagountou East, and Falagountou West deposits. The resource models were updated by Essakane S.A.'s Resource Development group. In 2017, a 16,000 m drilling campaign was performed in the EMZ deposit to increase the knowledge of the mineralization and to target low grade areas to feed a future heap leach project. Another 13,000 m campaign, carried out in early Q1 2018, increased the confidence level.

The Mineral Reserve estimate is based on a resource model published on February 28, 2018. The starting surface used for the reserve estimation was as of June 5, 2018. The surface was provided by the Surveying team from Essakane S.A.'s Technical Services and supported by an airborne LiDAR survey carried out at the end of 2017.

The EMZ pit is one of the largest in West Africa. Its final design has a length of 3,000 m and a width of 900 m, with a final depth located on level -120. Considering the regional topographical level at 260 m, the pit walls will be 380 m in certain areas.

As of June 5, 2018, the EMZ Mineral Reserves are estimated to total 87.9 Mt at 1.14 g/t Au of CIL Mineral Reserves and 60.5 Mt at 0.43 g/t Au of heap leach Mineral Reserves. These estimates include in-pit and stockpile material.

FIGURE 15-1 ESSAKANE MAIN ZONE AND EMZ NORTH



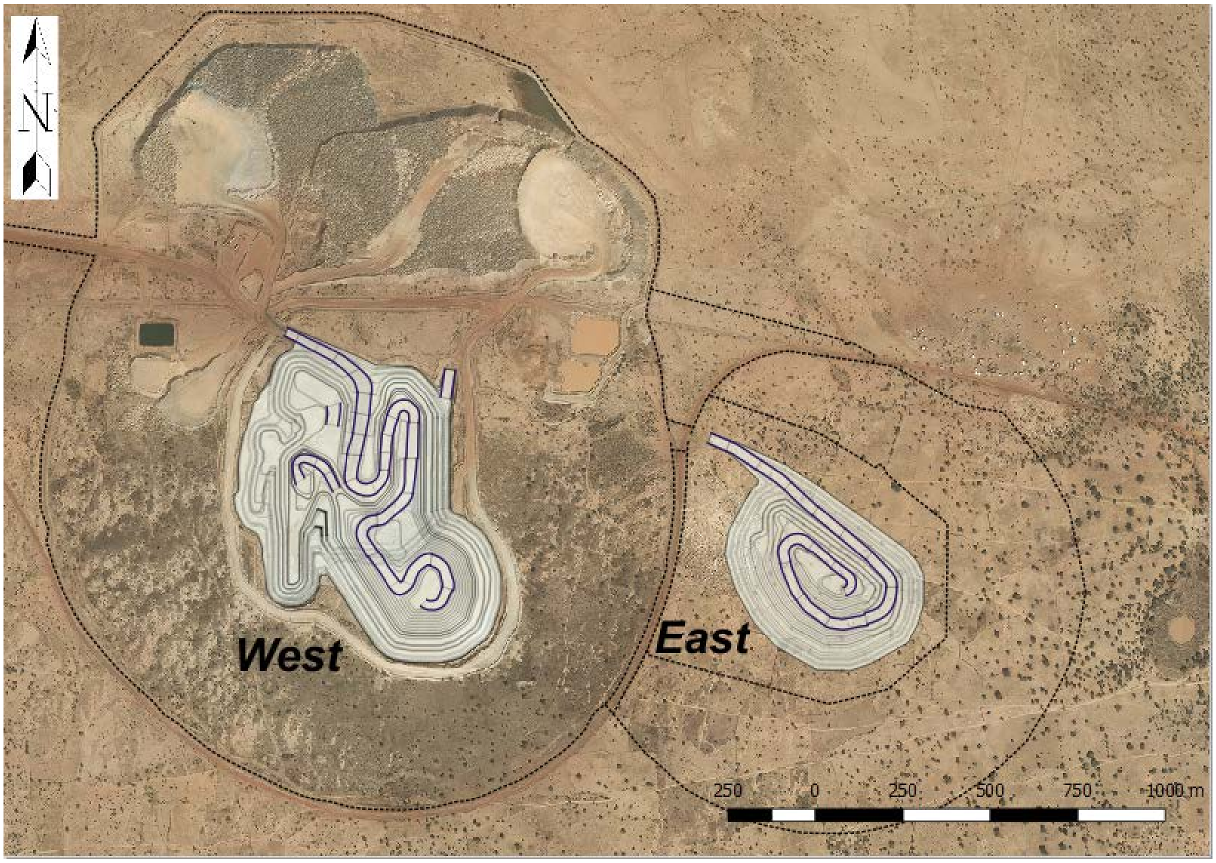
The Falagountou zone is located approximately 11,000 m east of the Essakane pit. This sector has been part of the Mineral Reserves since the start of operations. Subsequent drilling campaigns have increased the Mineral Reserves.

The Falagountou West pit began operations in 2015 and as of June 5, 2018, contains 5.1 Mt of ore at 1.61 g/t Au. Based on the current combined inventory (waste and ore) of 26.5 Mt, operations in the sector are projected to end in 2020.

The Falagountou East pit commenced production in Q1 2018. As of June 5, 2018, the pit contains Mineral Reserves of 4.2 Mt at 1.32 g/t Au. Based on the current combined inventory (waste and ore) of 29.8 Mt, operations in the sector are scheduled to end in 2021.

In addition to the above tonnage, the Falagountou stockpile totals 0.6 Mt grading 0.81 g/t Au.

FIGURE 15-2 FALAGOUNTOU WEST AND EAST PITS



15.3 DILUTION AND MINING LOSSES

The EMZ Mineral Reserve estimate and cut-off grades include a mining dilution provision of 8% for saprolite, transition, and fresh rock material. As a result of mineralization distribution and reconciliation studies, the Falagountou Mineral Reserve estimate and cut-off grades include a mining dilution provision of 8% for saprolite and 10% for transition and fresh rock material. The dilution tonnage is set at zero grade.

The use of previous reconciliation data, definition geological models (ore control), and data from the current mill has allowed control of dilution and mining recovery.

In 2018, the operational geology department, in collaboration with the mining engineering department, will continue studies on refining the parameters for mining recovery and external dilution, as well as interactions between these processes.

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, geological modelling, and external audits.

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 resources, metal prices used are slightly higher than those for 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 June 5, 2018 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	Unit	Value
Gold price	US$/oz	1,200
Long term oil price	US$/bbl	60
CFA exchange rate	CFA/USD	570
Transport & refining cost	US$/oz	3.04
Site diesel price	US$/l	1.10
Site HFO price	US$/l	0.68
Power cost	US$/kWh	0.15
Royalty (3-5%)	US$/oz	48.00
Cost of selling	US$/oz	63.04
Discount rate	%	6.00

The cut-off grade for heap leach milling was established at 0.30 g/t Au using the software COMET. Several variables and assumptions impacted this grade:

- NPV optimization,
- Using the CIL plant as a priority,
- Quantity of low-grade ore available,
- Closure of Essakane at the end of its LOM,

- Heap leach recovery grade.

Distribution of G&A costs between CIL and heap leach processes also impacts cut-off grades. If these costs are assumed to be entirely related to CIL, the cut-off grade for the heap leach process would be 0.21 g/t Au.

The next studies on the Project's economics will be focused on the optimal cut-off grade for the heap leach process and on improvements that could be made regarding the quantity of reserve in the Essakane pit.

The pit optimization parameters were consolidated by the mine engineering department with inputs provided by other departments to calculate the marginal cut-off grades (COG). The COG calculations are estimated on the basis of a long-term sustainable mill throughput of 12.1 Mtpa of fresh rock.

The metallurgical recovery assumptions for the EMZ deposit are 95.0% for saprolite, 93.0% for transition, and 92.0% for fresh rock. The metallurgical recovery assumptions for Falagountou are 0.5% higher for fresh rock at 92.5%.

The mine operating cost inputs used for pit optimization are derived from current mining costs and productivities. The costs were estimated on the basis of a diesel fuel price of $1.10/L. The reference mining costs, at surface (200 m elevation), are estimated at $2.03/t for saprolite waste, $2.46/t for transition waste, and $2.55/t for fresh rock waste. These costs are approximately $0.01/t to $0.20/t higher for mineralized material due to the added RC drilling cost. The mine operating cost includes mine sustaining capital and capital maintenance items, which total $0.50/t for the fresh rock material.

The Falagountou pit optimization parameters are the same as those for the EMZ pit, with the exception of additional ore haulage costs as the process plant is located a distance of 11 km from the pit. For fresh rock, this represents an additional cost of $0.96/t and is slightly higher for transition and saprolite due to lower densities.

The 2018 COGs, by pit, are summarized in Table 15-3 with the details for EMZ and Falagountou presented in Tables 15-4 and 15-5, respectively.

TABLE 15-3 SUMMARY OF 2018 COGS AT US$1,200/OZ AU

COG by Pit		Saprolite	Transition	Fresh Rock
EMZ CIL	(g/t Au)	0.38	0.48	0.55
EMZ Heap Leach	(g/t Au)	-	-	0.30
Falagountou	(g/t Au)	0.41	0.52	0.58

TABLE 15-4 SUMMARY OF EMZ PIT OPTIMIZATION PARAMETERS AND COG

Ore Based Cost and COG by Deposit		**EMZ Pit**		
Rock type		**Saprolite**	**Transition**	**Fresh Rock**
Metallurgical recovery	%	95.0%	93.0%	92.0%
Processing rate	Mtpa	14.98	12.48	12.07
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$/yr	55.59	58.50	58.50
Processing fixed cost	US$/t treated	3.71	4.69	4.85
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$/t treated	2.09	2.95	4.20
Other variable cost	US$/t treated	0.07	0.07	0.07
Total processing cost	**US$/t treated**	**8.25**	**10.70**	**12.36**
Mining dilution	%	8.0%	8.0%	8.0%
Ore premium mining cost	US$/t treated	0.00	0.00	0.00
Ore Feed	US$/t treated	0.04	0.04	0.04
Total fixed G&A costs	MUS$/yr	48.10	48.10	48.10
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 capital	US$/t treated	0.40	0.48	0.50
Total Ore Based Cost	**US$/t treated**	**12.07**	**15.25**	**17.06**
Full Grade Ore in-situ COG	**g/t Au**			

Reference Mining Cost by Deposit		**EMZ Pit**					
Rock type		**Saprolite**		**Transition**		**Fresh Rock**	
Material		**Ore**	**Waste**	**Ore**	**Waste**	**Ore**	**Waste**
Total Reference Mining Cost	US$/t mined	1.99	2.03	2.45	2.46	2.75	2.55
Reference elevation	RL	200 RL					
Incremental bench cost	US$/t per vert. m	0.0031					

TABLE 15-5 SUMMARY OF FALAGOUNTOU PIT OPTIMIZATION PARAMETERS AND COG

Ore Based Cost and COG by Deposit		Falagountou Pit		
Rock type		**Saprolite**	**Transition**	**Fresh Rock**
Metallurgical recovery	%	95.0%	93.0%	92.5%
Processing rate	Mtpa	14.98	12.48	12.07
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$/yr	55.59	58.50	58.50
Processing fixed cost	US$/t treated	3.71	4.69	4.85
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$/t treated	2.09	2.95	4.20
Other variable cost	US$/t treated	0.07	0.07	0.07
Total processing cost	**US$/t treated**	**8.25**	**10.70**	**12.36**
Mining dilution	%	8.0%	10.0%	10.0%
Ore premium mining cost	US$/t treated	0.98	0.97	0.96
Ore Feed	US$/t treated	0.04	0.04	0.04
Total fixed G&A costs	MUS$/yr	48.10	48.10	48.10
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 capital	US$/t treated	0.40	0.48	0.50
Total Ore Based Cost	**US$/t treated**	**13.06**	**16.23**	**18.02**
Full Grade Ore in-situ COG	**g/t Au**	**0.41**	**0.52**	**0.58**

Reference Mining Cost by Deposit		Falagountou Pits					
Rock type		**Saprolite**		**Transition**		**Fresh Rock**	
Material		**Ore**	**Waste**	**Ore**	**Waste**	**Ore**	**Waste**
Total Reference Mining Cost	US$/t mined	2.03	1.83	2.47	2.24	2.74	2.32
Reference elevation	RL	260 RL					
Incremental bench cost	US$/t per vert. m	0.0031					

15.6 MINERAL RESERVE ESTIMATES

The June 5, 2018 consolidated EMZ and Falagountou Mineral Reserves are presented in Table 15-6. No Proven Mineral Reserves have been estimated.

The addition of the heap leach process has increased Essakane's Mineral Reserve estimate.

TABLE 15-6 ESSAKANE GOLD MINE JUNE 5, 2018 CONSOLIDATED MINERAL RESERVES

	Laterite & Saprolite			Transition			Fresh Rock			All Material		
	000 t	g/t Au	000 oz Au	000 t	g/t Au	000 oz Au	000 t	g/t Au	000 oz Au	000 t	g/t Au	000 oz Au
					Essakane Main Zone (EMZ)							
Cut-off Grade CIL[1]		**0.38 g/t Au**			**0.48 g/t Au**			**0.55 g/t Au**				
Cut-off Grade Heap Leach[1]		**-**			**-**			**0.30 g/t Au**				
Proven[2]	-	-	-	-	-	-	-	-	-	-	-	-
Probable CIL[2]	34	0.47	1	510	0.66	11	73,907	1.25	2,962	74,452	1.24	2,973
Probable Heap Leach[2]							56,427	0.42	766	56,427	0.42	766
EMZ Proven & Probable	34	0.47	1	510	0.66	11	130,334	0.89	3,728	130,878	0.89	3,739
Stockpiles CIL	94	0.24	1	8,792	0.54	152	4,568	0.69	102	13,453	0.59	255
Stockpiles Heap Leach							4,040	0.47	61	4,040	0.47	61
Total EMZ Reserves	**128**	**0.30**	**1**	**9,302**	**0.55**	**163**	**138,941**	**0.86**	**3,890**	**148,371**	**0.84**	**4,054**
					Falagountou (FAL)							
Cut-off Grade[1]		**0.41 g/t Au**			**0.52 g/t Au**			**0.58 g/t Au**				
Proven[2]	-	-	-	-	-	-	-	-	-	-	-	-
Probable[2]	1,106	1.07	38	663	1.14	24	7,496	1.57	379	9,264	1.48	442
Stockpiles CIL	-	-	-	-	-	-	561	0.81	15	561	0.81	15
Total FAL Reserves	**1,106**	**1.07**	**38**	**663**	**1.14**	**24**	**8,057**	**1.52**	**394**	**9,826**	**1.44**	**456**
					Consolidated Essakane Reserves (EMZ & FAL)							
Total Reserves (100% Basis)	**1,234**	**0.99**	**39**	**9,965**	**0.58**	**187**	**146,998**	**0.91**	**4,284**	**158,197**	**0.89**	**4,510**
Attributable Reserves (90% Basis)	**1,111**	**0.99**	**35**	**8,968**	**0.58**	**169**	**132,298**	**0.91**	**3,856**	**142,377**	**0.89**	**4,059**
% of Total Reserves	0.8%			6.3%			92.9%			100%		
Waste (incl. non-reserve material)										327,518		
Strip Ratio										2.34		

Notes:
1. Cut-off grade based on Leached Au.
2. EMZ & FAL: US$1,200/oz Au cut-offs inside US$1,200/oz Au Pit Design.

Mineral Reserves were separated based on processing types, to account for the distinct processing mill recoveries. The Falagountou deposits have no Mineral Reserves attributed to the heap leach process due to the distance and short mine life, which ends in 2020 for the West pit and 2021 for the East pit.

As of June 5, 2018, there were 87.9 Mt of CIL Probable Mineral Reserve defined in the EMZ pit design and within stockpiles, at an average grade of 1.14 g/t Au totalling 3,228 koz of gold. The heap leach Probable Mineral Reserves are estimated to be 60.5 Mt in the EMZ pit design and within stockpiles, at an average grade of 0.43 g/t Au totalling 827 koz of gold.

Additionally, the Falagountou pit CIL Probable Mineral Reserves total 9.8 Mt at an average grade of 1.44 g/t Au for 456 koz of gold.

The EMZ pit contains 282.4 Mt of waste and the Falagountou pit contains 45.2 Mt of waste resulting in a total project stripping ratio of 2.34:1.0. As of June 5, 2018, the consolidated Probable Mineral Reserves are estimated to be 158.2 Mt at a grade of 0.89 g/t Au for a total of 4,510 koz of gold (in-situ).

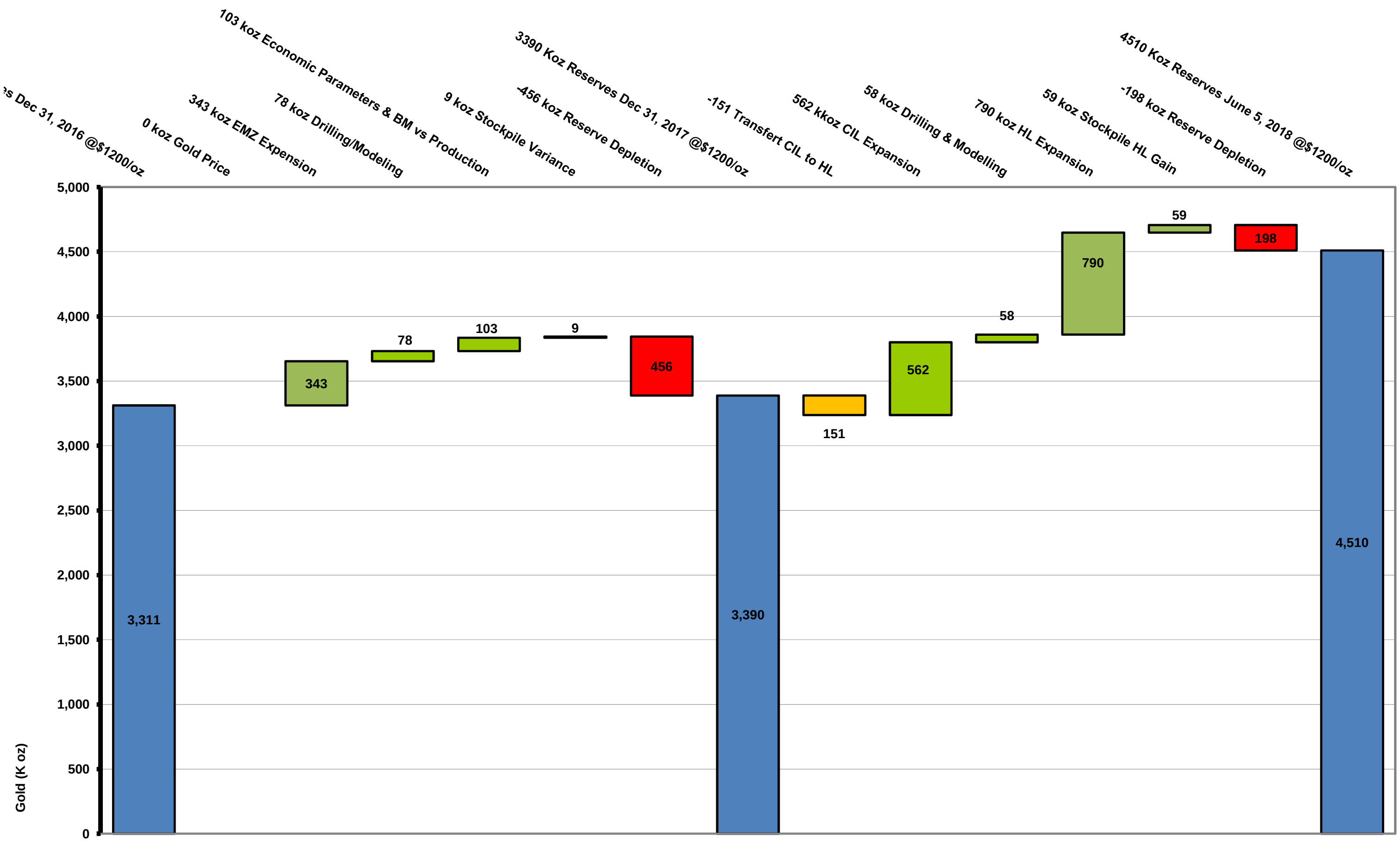
IAMGOLD's 90% attributable Probable Mineral Reserves total 142.4 Mt of ore and 4,059 koz of gold. Approximately 7% of the Mineral Reserve consists of transition and saprolite ore and the remainder (93%) is fresh rock.

The waterfall graph in Figure 15-3 shows the change in the total ounces of gold contained within the Mineral Reserves from December 31, 2016 through to June 5, 2018.

The graph shows two phases. The first phase shows December 31, 2017 Mineral Reserves at the site without the heap leach process. An increase of 78 koz gold, resulting from a 2017 drilling campaign, was made during this period, despite a depletion of 456 koz of gold, which was mainly due to a significant decrease in milling costs and update of the geological model.

The second part shows Essakane's optimization by the addition of the heap leach process. It is important to mention that the economic basis is the same for both parts. Figure 15-3 shows an increase of 1,120 koz of gold from the December 31, 2017 estimate for this second part, despite a depletion of 198 koz of gold.

FIGURE 15-3 ESSAKANE GOLD MINE MINERAL RESERVES WATERFALL GRAPH – DECEMBER 31, 2016 TO JUNE 5, 2018



A total of 562 koz of gold for the CIL process and 639 koz of gold for the heap leach process are attributable to the expansion of the Essakane main pit. An increase of 59 koz of gold from ore stockpiles is the result of a transfer of the material from the marginal non-reserve stockpile to reserves to be treated by heap leaching.

Table 15-7 shows Mineral Reserve evolution since May 2007.

TABLE 15-7 MINERAL RESERVE EVOLUTION

Year	Gold Price (US$/oz)	Tonnes (000)	Grade (g/t Au)	Ounces Au (000)
June 5/18 Heap Leach	1,200	158,197	0.89	4,510
Dec. 31/17	1,200	93,126	1.13	3,390
Dec. 31/16	1,200	89,676	1.15	3,311
Dec. 31/15	1,200	96,463	1.10	3,414
Dec. 31/14	1,300	108,821	1.11	3,886
Dec. 31/13	1,400	126,806	1.12	4,573
Dec. 31/12	1,500	114,377	1.00	3,659
Dec. 31/11	1,200	109,245	1.10	3,858
Dec. 31/10	1,000 & 850*	107,465	1.29	4,461
Dec. 31/09	850	92,911	1.44	4,301
Mar/09	600	58,122	1.67	3,121
May/07	650	46,413	1.78	2,649

Note:* Falagountou at $850/oz Au (Jan 2008 pit design)

15.6.1 STOCKPILES

The Essakane site has several stockpiles, which are separated by rock type and economic category. The arithmetic mean is applied for the assigned average gold grade. Table 15-8 lists the stockpiles and Figure 15-4 is an aerial view of the stockpiles.

The addition of the heap leach process enabled the conversion of marginal rock into Mineral Reserves, however, the marginal transition stockpile remains non-economic.

TABLE 15-8 STOCKPILE INVENTORY

Stockpile	Rock Type	Tonnage (kt)	Grade (g/t)	Destination
LG Rock	Fresh Rock	4.3	0.67	CIL & Heap Leach
LG Transition	Transition	8.8	0.54	CIL
Marginal Rock	Fresh Rock	4.0	0.47	Heap Leach
Marginal Saprolite	Saprolite	0.1	0.24	CIL
Marginal Transition	Transition	5.6	0.33	Non-Economic

FIGURE 15-4 STOCKPILE INVENTORY

16 MINING METHODS

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 48.0 Mt in 2017 with a stripping ratio of 3.10. Approximately 11.8 Mt of ore at an average grade of 1.17 g/t Au containing a total of 432 koz of gold were produced in 2017.

Table 16-1 details past production, through May 2018, at the Essakane Gold Mine.

TABLE 16-1 ESSAKANE GOLD MINE 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	4,742
Waste Mined (000 t)	11,876	15,268	1,689	30,006	32,677	35,690	36,939	36,296	16,099
Marginal Mined (000 t)	957	1,788	25,103	3,257	1,440	1,679	855	926	
Total Mined (000 t)	22,930	27,166	36,353	45,133	46,698	48,887	48,314	47,993	20,842
Strip Ratio	1.27	1.69	2.80	2.80	2.71	3.24	3.25	3.10	3.39
Mined Ore Grade (g/t Au)	1.05	1.08	1.04	0.84	0.98	1.14	1.21	1.17	1.18
Ore Milled (000 t)	2,973	7,977	10,762	10,613	11,897	11,716	12,005	13,891	5,516
Mill Grade (g/t Au)	1.49	1.53	1.10	0.89	1.06	1.23	1.22	1.07	1.21
Recovery (%)	95.7%	95.4%	91.9%	91.7%	90.7%	91.7%	89.0%	90.3%	91.5%
Gold Produced (000 oz)	136	375	350	277	369	426	419	432	197
RC Drilling (000 m)	100	209	167	227	195	148	104	78	72
Production Drilling (000 m)	12	257	541	806	724	944	869	886	338
Pre-split Drilling (000 m)	-	-	4	32	21	41	61	124	53
Tonnes Blasted (000 t)	690	12,937	24,818	43,989	37,292	42,218	43,082	49,316	19,662
Explosives (000 kg)	53	2,405	5,813	12,606	11,958	13,740	14,565	16,799	7,054
Powder Factor (kg/t)	0.12	0.19	0.23	0.29	0.32	0.33	0.34	0.34	0.36

Note *. Through May 2018

The Essakane Gold Mine consists of several operating sites. The Essakane main pit is mined in several mining phases and accounts for over 80% of the production. The Falagountou and Essakane North satellite pits provide additional ore and operational flexibility.

The Essakane main pit comprises a total of seven mining phases. Mining by phases provides a sufficient quantity of ore by postponing and scheduling the mining of waste evenly over time. The average width of push-backs is 120 m and is limited to 30 m in certain areas. Mining by phases also provides effective operational flexibility through the simultaneous opening of several fronts, and the optimization of truck cycle times through good ramp system management. Tables 16-2 and 16-3 show the mining phases for the Essakane and Falagountou pits, respectively. Figure 16-1 shows a cross section of the mining phases in the Essakane pit.

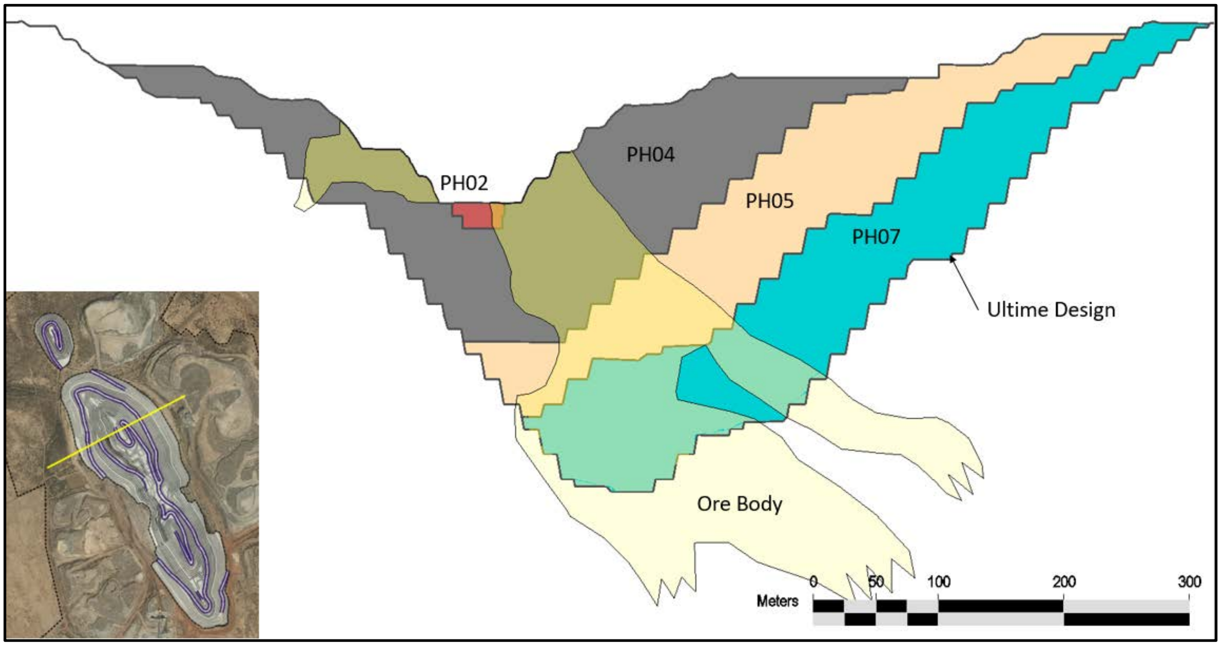
TABLE 16-2 ESSAKANE PHASES

	Ore		Waste	Total	
Zone	**Tonnage (Mt)**	**Grade (Au g/t)**	**Tonnage (Mt)**	**Tonnage (Mt)**	**Strip Ratio**
EMZ PH02	2.9	1.18	1.3	4.1	0.45
EMZ PH03	27.0	0.92	29.8	56.9	1.10
EMZ PH04	31.8	0.89	58.1	89.9	1.83
EMZ PH05	15.3	0.97	43.9	59.2	2.87
EMZ PH06	20.8	0.70	48.5	69.4	2.33
EMZ PH07	26.8	1.00	95.4	122.2	3.55
EMZ Total	**124.7**	**0.90**	**277.0**	**401.7**	**2.22**
SAT North PH01	3.3	0.63	5.4	8.6	1.63
Total EMZ	**128.0**	**0.90**	**282.4**	**410.3**	**2.21**

TABLE 16-3 FALAGOUNTOU PHASES

	Ore		Waste	Total	
Zone	**Tonnage (Mt)**	**Grade (Au g/t)**	**Tonnage (Mt)**	**Tonnage (Mt)**	**Strip Ratio**
Falagountou West PH02	0.9	1.77	1.7	2.6	2.05
Falagountou West PH03	3.9	1.57	18.4	22.4	4.68
Falagountou West Total	**4.8**	**1.61**	**20.2**	**25.0**	**4.22**
Falagountou East Total	4.3	1.30	25.0	29.3	5.75
Total Falagountou	**9.1**	**1.46**	**45.2**	**54.3**	**4.95**

FIGURE 16-1 ESSAKANE 51850N SECTION



16.1.1 MINING EQUIPMENT

Current mining production is approximately 50 Mtpa, however, the mine expansion will result in an increase to 70 Mtpa, which will require additional mining and auxiliary equipment.

The main loading equipment includes RH120 trucks and CAT-993 wheel loaders. RH120 trucks are generally utilized in the main development sectors, while front loaders are generally used to feed the primary crusher and operate in the Falagountou satellite pits.

The production truck fleet is primarily composed of CAT-785 trucks with a 135 t payload. The fleet also includes five CAT-777 trucks, which are mainly used to clear walls and maintain haul roads.

In 2016, Atlas Copco PV-235 production drill rigs were added to the fleet. These drill rigs are capable of drilling 229 mm (9-inch) holes using GPS and Wi-Fi technology. Presplitting and secondary drilling is carried out using the Atlas Copco ROC-L8 drill.

All blasting activities on site are executed by an explosives supplier. Holes are loaded with bulk explosive matrix and initiated with electric detonators.

A list of Essakane S.A.'s primary mine production equipment fleet is shown in Table 16-4.

TABLE 16-4 CURRENT PRIMARY MINE EQUIPMENT FLEET

Type	Model	Actual Number	Planned Number	Difference
Shovel	RH120 & Caterpillar 6060	4	5	+1
Excavator	Caterpillar 390	2	6	+4
	Caterpillar 345/349	6	4	-2
Loader	Caterpillar 993K	4	7	+3
Truck	Caterpillar 785C	26	41	+15
	Caterpillar 777F	5	5	-
Drilling	Atlas Copco ROCL8	2	2	-
	Atlas Copco PV-235	4	6	+2
	Sandvik DK45	5	5	-
Dozer	Bulldozer D9R	7	7	-1
	Bulldozer D10R	-	4	+4
	Wheeldozer 824	2	4	+2
Grader	Grader Cat 16M	5	5	-
Auxiliary	Caterpillar 966H	1	1	-
Tow Haul	Caterpillar 777F	1	1	-

Hauling truck calculations are completed using the mining software MineSight. The MSHaulage module optimized truck cycle time, which has improved mine waste disposal planning over time.

16.1.2 WASTE DEPOSITS

Mineral Reserves have a 2.34 stripping ratio. This tonnage represents 327 Mt of waste to be disposed of around the pits. Disposal areas must be managed in compliance with resource protection, surface water management, and the permits granted.

Disposal on dumps is carried out in 7 m layers. This technique optimizes dump density and reduces time spent using bulldozers.

Ramp systems are planned at a 6% grade and have a width of 35 m. This configuration increases the speed of waste cycles.

Figures 16-2 and 16-3 show the waste stockpile capacity for the EMZ and Falagountou pits, respectively. Table 16-5 summarizes the waste dump capacity for the EMZ and Falagountou pits.

FIGURE 16-2 EMZ WASTE STOCKPILE CAPACITY

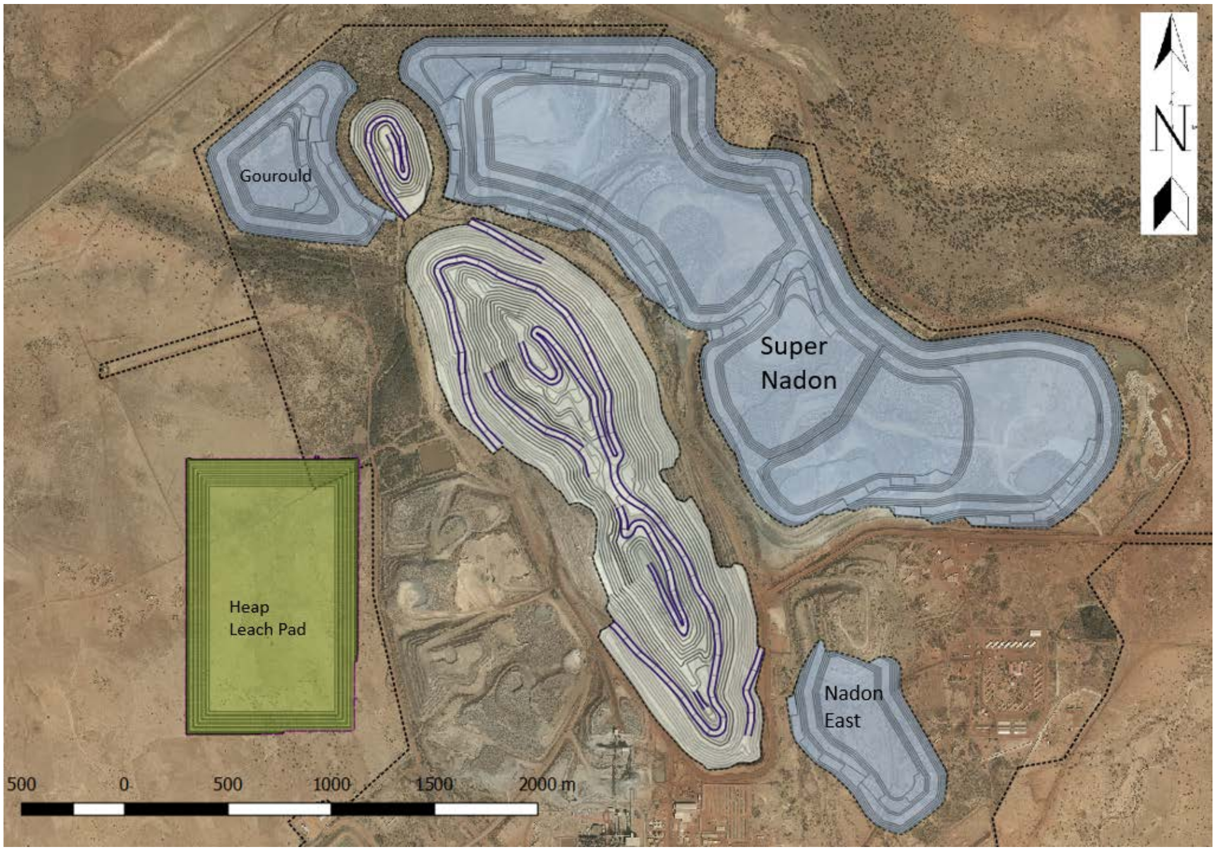


FIGURE 16-3 FALAGOUNTOU WASTE STOCKPILE CAPACITY

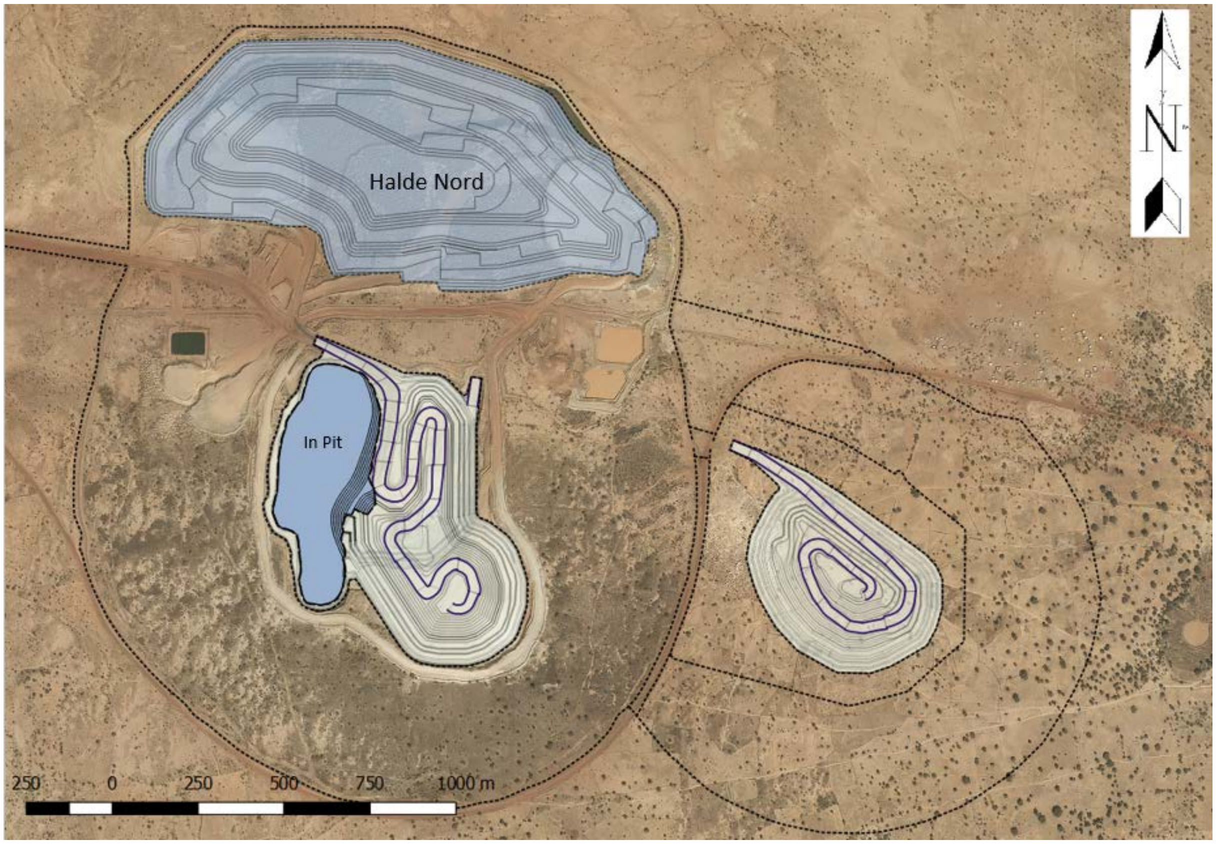


TABLE 16-5 WASTE DUMP CAPACITY

Waste Dump	Volume (Mm³)	Tonnage in situ (Mt)
Boussim	17.4	36.1
Nadon East	14.3	29.8
Super Nadon	132.3	274.9
Total EMZ	**164.1**	**340.8**
Halde Nord	28.2	58.6
In-Pit Falagountou	3.8	7.9
Total Falagountou	**32**	**66.5**

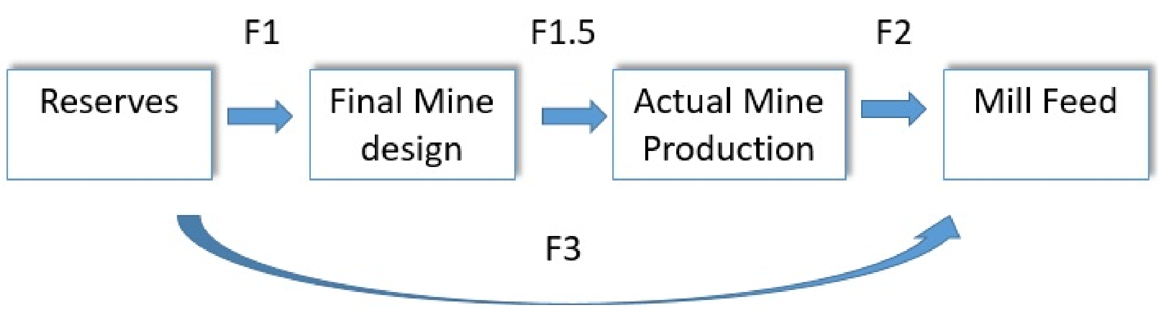
16.1.3 GRADE CONTROL

Essakane's Technical Services have a production geology team assigned to grade control and to mining reconciliation. 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.

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

The process of reconciliation is overseen by a co-corporative guidance, intended to standardize and strengthen the process. A monthly report makes it possible to measure deviations, take action in the field, and increase the accuracy of the official resource model. Figure 16-4 shows the reconciliation process.

FIGURE 16-4 RECONCILIATION PROCESS



Reconciliation results show deviations below 10% on a monthly basis, and below 5% on an annual basis.

16.2 GEOTECHNICAL DOMAINS

16.2.1 ESSAKANE PIT SLOPE DESIGN

SRK conducted a geotechnical stability assessment of the proposed ultimate open pit expansion (SRK, 2018). SRK had previously carried out a geotechnical stability assessment and developed slope design recommendation for the fresh rock units within the pit.

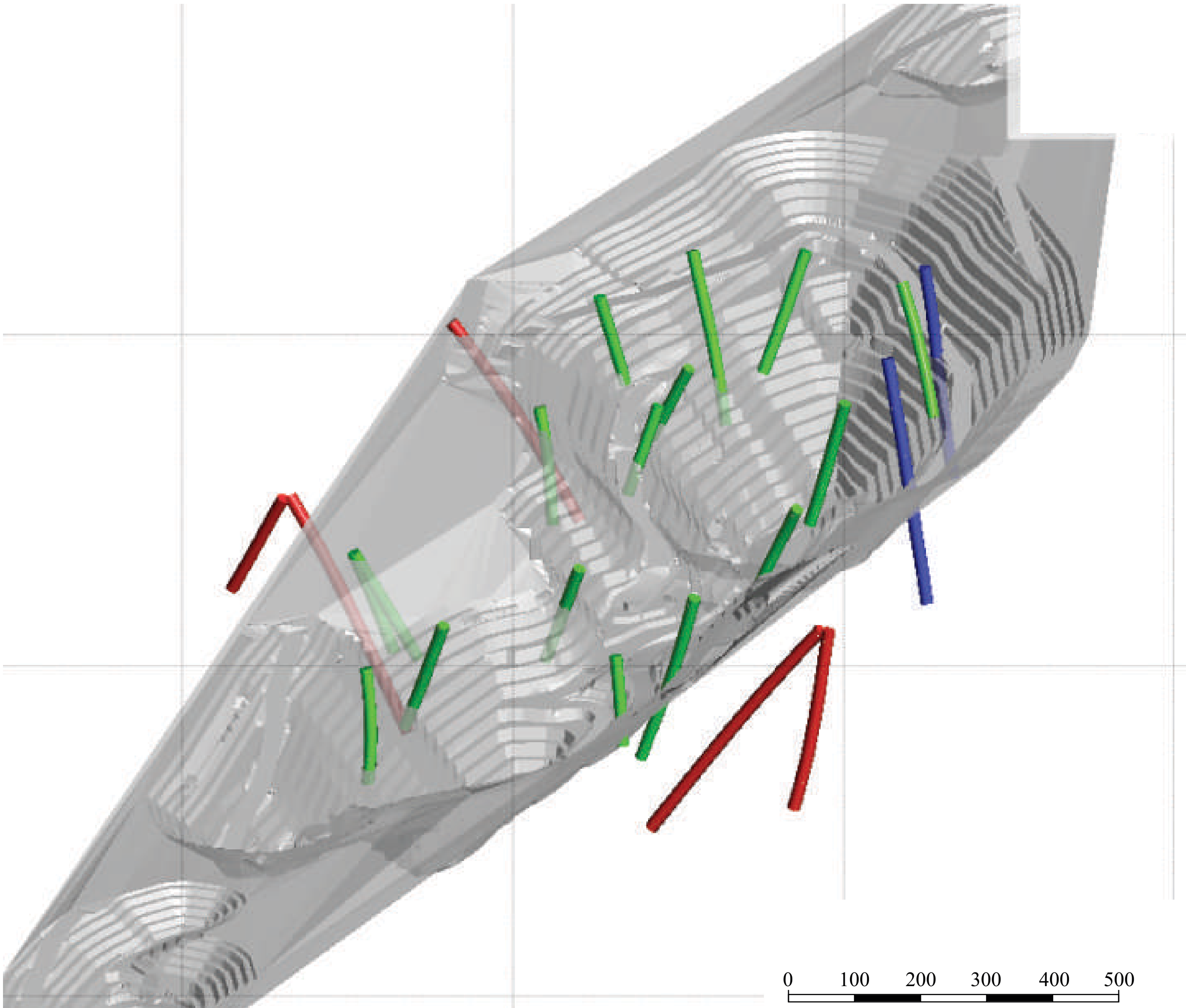
16.2.1.1 GEOTECHNICAL INVESTIGATION AND HYDROGEOLOGY MONITORING

Field investigations have been carried out to assess the open pit geotechnical and hydrogeological conditions. The information from the following investigations has been utilized for this current study:

- 2013 Geotechnical Drilling and Data Collection Program (Piteau, 2013a).
- 2015 Vibrating Wire Piezometer Installations (SRK, 2015); and
- 2017 Geotechnical Drilling Investigations (SRK, 2018).

In addition, the geotechnical data collected by SRK during the periodic site inspections, and by the Essakane Mine geotechnical and geological teams has been utilized. The locations of the geotechnical drill holes in comparison to the proposed ultimate pit shell are shown in Figure 16-5.

View Looking North-West



Legend:

Field Program

Piteau 2013

SRK HL 2017

SRK VWP 2016

Figure 16-5

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Geotechnical Drill Holes Shown on the Proposed Ultimate Essakane Pit

July 2018

SRK Consulting (Canada) Inc., 2018.

A total of eight VWPs have been installed behind the pit walls to date, as shown in Figure 16-6. During 2017, VWP07 and VWP08 were mined out as the upper bench slopes of the North wall were excavated. A replacement hole is planned near VWP07 and VWP08 as part of the 2018 VWP installation program. A new VWP hole is also proposed behind the Northeast Wall. The purpose of the installation is to improve the monitoring coverage and validate the design groundwater conditions in the SRK 2018 assessment.

FIGURE 16-6 LOCATION OF THE ESSAKANE PIT VWP

Note. . VWP07 and VWP08 Mined Out

16.2.2 SLOPE DESIGN RECOMMENDATIONS

Essakane S.A. has adopted a pit design based on slope design recommendations outlined by Piteau Associates (Piteau, 2013b) and SRK (SRK, 2015), as summarized in Table 16-6. Piteau developed design recommendations for the saprolite and transition units, and SRK for the underlying fresh rock units. SRK subsequently revised the transition and fresh rock – argillite unit within the W2 Design Sector in December 2016 (SRK, 2016b) due to significant toppling instabilities within the exposed bench slopes. The design sectors are shown in Figure 16-7.

TABLE 16-6 SUMMARY OF EXISTING PIT SLOPE DESIGN CRITERIA

Design Sector	Geotechnical Unit	Bench Face Angle (°)	Bench Height (m)	Bench Width (m)	Inter-Ramp Angle (°)	Maximum Stack Height (m)	Geotechnical Berm Width (m)
W1	Saprolite	50	10	7.5	32	40	20
	Transition	70	10	9.6	35	40	
	Fresh Rock	80	20	11.5	53	120	
W2	Saprolite	50	10	7.5	32	40	20
	Transition	70	10	9.6	35	40	
	Fresh Rock	80	20	11.5	53	120	
	Fresh Rock – Argillite*	80	20	14.5	48	120	
E1	Saprolite	50	10	7.5	32	40	20
	Transition	70	10	9.2	38	40	
	Fresh Rock	80	20	12.7	51	120	
E2	Saprolite	50	10	7.5	32	40	20
	Transition	70	10	9.2	38	40	
	Fresh Rock	80	20	12.7	51	120	

Note: * - New geological model indicates that the "Fresh Rock – Argillite" along the West Wall is the Turbidite unit.

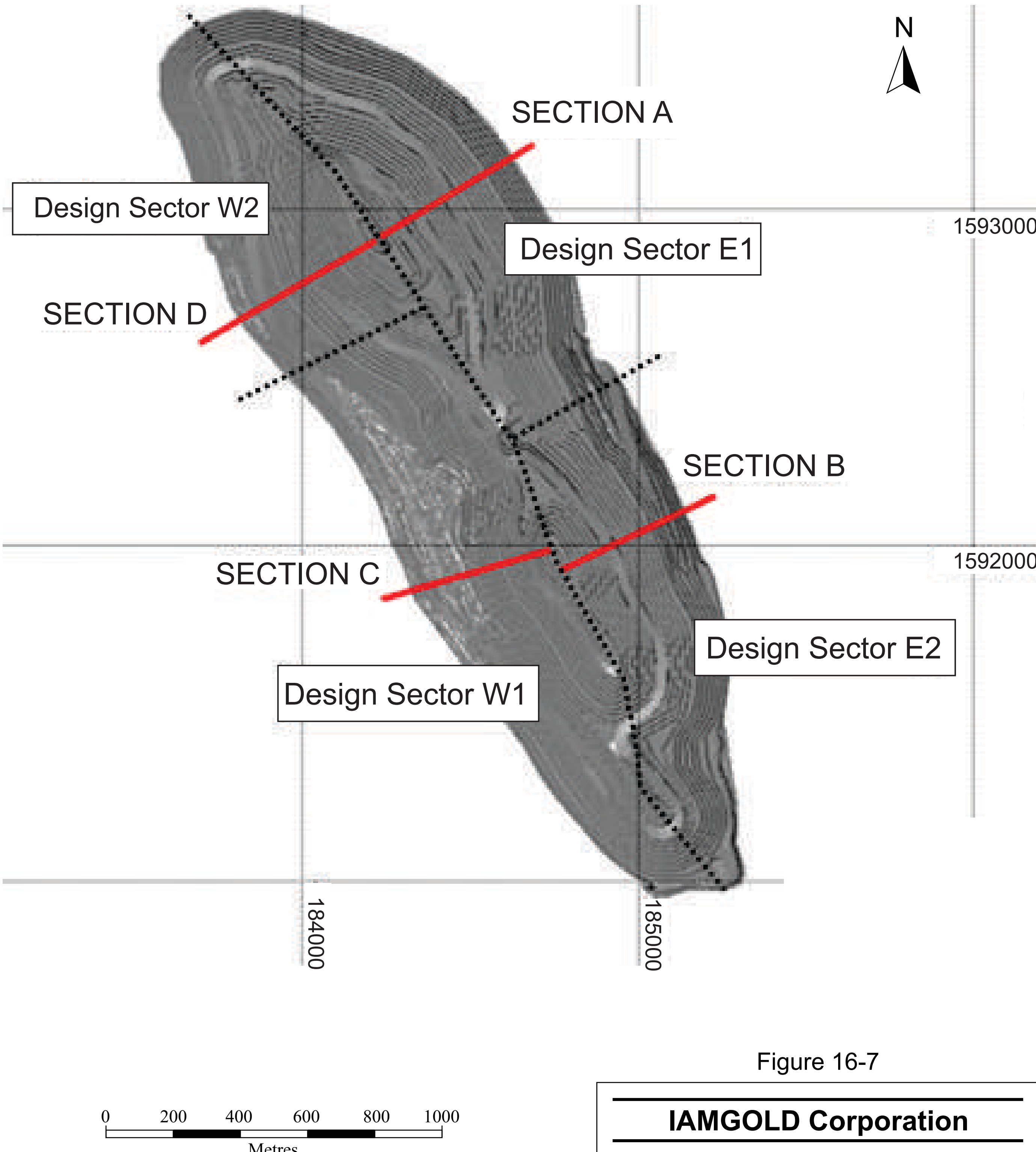


Figure 16-7

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Location of the Design Sections

July 2018

SRK Consulting (Canada) Inc., 2018.

16.2.2.1 GEOTECHNICAL AND STABILITY ASSESSMENT

SRK carried out an evaluation of the geotechnical model for the Essakane Pit (SRK, 2018), including the geology, structural geology, rock mass and hydrogeology components that control the overall stability of Essakane Pit slopes. The geotechnical model is used as input in the major fault and overall stability analyses.

Two-dimensional limit equilibrium and finite element slope stability analyses were carried out using the Rocscience software to assess the expected stability conditions for each design section in Figure 16-7. The stability analyses considered the potential for overall non-circular failure through the anisotropic rock mass.

Rock mass strength parameters were assigned to each geotechnical unit. The parameters are based on geotechnical logging data, bench face mapping, and the previous laboratory testing. Rock mass anisotropy has been incorporated into the stability modelling using the orientation of bedding and the most adverse joint set with respect to the analyzed pit wall. Two groundwater scenarios were modelled, which included a phreatic surface and pore pressure grid (PPM) approach. The PPM approach is calibrated using current VWP monitoring data at the time of the study.

The results of the slope stability analyses are summarized in Table 16-7. The results indicate that the proposed ultimate pit slopes are expected to exhibit acceptable design Factor of Safety (FOS)/SRF for the PPM and phreatic surface scenarios.

TABLE 16-7 SUMMARY OF OVERALL ESSAKANE PIT STABILITY ANALYSES

Stability Section	Design Sector	Critical Factor of Safety [Slide]		Critical Probability of Failure [RS2]	
		Phreatic Surface	PPM	Phreatic Surface	PPM
Section A	E1	2.7	2.8	1.7	1.8
Section B	E2	2.5	2.6	1.6	1.6
Section C	W1	1.4	1.4	1.4	1.9
Section D	W2	1.3	1.4	1.5	1.7

The results of the stability assessment indicate that the rock masses are of sufficient strength to support the proposed overall stability, and the major structures are not expected to adversely impact overall stability. The established slope design criteria are considered suitable for use in this current ultimate pit design. There are no current recommendations to modify the current design to improve the stability of the proposed ultimate Essakane Pit. There could be future opportunities to re-evaluate the design recommendations should there be a sufficiently long prism slope monitoring record that can be supplemented with a slope radar.

There is potential for multiple-bench instabilities due to the intersection of faults within the interim and final pit walls, and the orientation of the faults should be mapped on a continuous basis to validate the 3D structure model and evaluate the slope risks on an ongoing basis. Slope monitoring using prisms in a well-established network are essential to help manage the potential risk related to these structures. It is critical that the pit slopes are monitored to understand the overall behaviour of the walls with excavation and provide a mechanism to predict a larger scale instability.

16.2.3 FALAGOUNTOU PIT SLOPE DESIGN

SRK conducted a geotechnical stability and design study for the Falagountou Pit in 2015 (SRK, 2015). The slope design recommendations from this study are presented in Table 16-8.

TABLE 16-8 FALAGOUNTOU PIT DESIGN PARAMETERS

Parameter	Unit	Sap	Trans	Rock Sector		
				West	Northeast	South
Bench height	m	10	10	10	10	10
Berm width	m	11.5	7.5-11.5	12	7.5	8.5
Bench face angle	°	65	65	90	90	90
Geotech berm width	m	15	15	15	15	15
Inter ramp angle	°	32	32-39	40	53	50
Ramp width	m	30	30	30	30	30
Ramp gradient	°	8	10	10	10	10

16.3 MINE DESIGN

SRK was initially responsible for Essakane S.A.'s geotechnical design studies and wall stability analyses. Since June 2015, SRK was mandated to complete a third party review of the pit slope design, and has since provided overall pit slope design and recommendations for the EMZ and Falagountou pits.

SRK carries out annual site visits and remains in constant contact with the mine site's geotechnical engineers.

A Leica Geomos deformation monitoring instrument is available to Essakane for wall stability monitoring. The EMZ and Falagountou east and west pit design parameters, to ensure slope stability, are detailed in Tables 16-9, 16-10, and 16-11, respectively.

TABLE 16-9 EMZ PIT DESIGN PARAMETERS

Parameter	Unit	Sap	Trans	Fresh Rock				
Sector (rock only)				East 1	East 2	West1	West2	
							Turbidite*	other
Bench height	m	10	10	20	20	20	20	20
Berm width	m	7.5	9.6	12.7	12.7	11.5	14.5	11.5
Bench face angle	°	50	65	80	80	80	80	80
Geotech berm width	m	20	20	20	20	20	20	20
Maximum stack height	m	40	40	120	120	120	120	120
Inter ramp angle (IRA)	°	32	35	51	51	53	48	53

Stack heights in East Sectors 1 and 2 differ based on the location of the Argillic Zone in East wall.

TABLE 16-10 FALAGOUNTOU EAST PIT DESIGN PARAMETERS

Parameter	Unit	Sap	Trans			Fresh Rock			
Sector (rock only)			North & East	South	West	North	South	East	West
Bench height	m	10	10	10	10	10	20	10	10
Berm width	m	11.5	7.5	7.5	9	7.5	7.0	7.5	8.5
Bench face angle	°	65	65	50	65	90	90	90	90
Geotech berm width	m	20	20	20	20	20	20	20	20
Maximum stack height	m	40	40	40	40	80	80	80	80
Inter ramp angle (IRA)	°	32	39	32	36	53	40	53	50

TABLE 16-11 FALAGOUNTOU WEST PIT DESIGN PARAMETERS

Parameter	Unit	Sap	Trans				Fresh Rock		
Sector (rock only)			**North & East**	**South-West**	**South-East**	**West**	**South**	**North-East**	**West**
Bench height	m	10	10	10	10	10	10	10	10
Berm width	m	11.5	7.5	7.5	10	11.5	7.5	7.5	8.5
Bench face angle	°	65	65	65	65	65	90	90	90
Geotech berm width	m	15	15	15	15	15	15	15	15
Maximum stack height	m	40	40	40	40	40	100	100	80
Inter ramp angle (IRA)	°	32	39	39	34	32	50	53	40

It should be noted that for pits, pit slope parameters are continuously being optimized as mining progresses and new geotechnical information becomes available. This will include the Falagountou east pit which will be optimized following the recent commencement of slope excavation.

16.4 LIFE OF MINE PLAN

Although the Life of Mine (LOM) extends until 2026 in terms of mining, during which 142.0 Mt of ore grading 0.94 g/t Au will be extracted, the LOM plan provides a stable annual gold production above 480 koz during the operation of the two processes. The annual average production of 17.5 Mt of ore mined from pits is insufficient to feed the two 21.2 Mt processes, and as a result it is critical that the remaining ore be fed from the stockpiles.

The key constraints that guided the mining plan include:

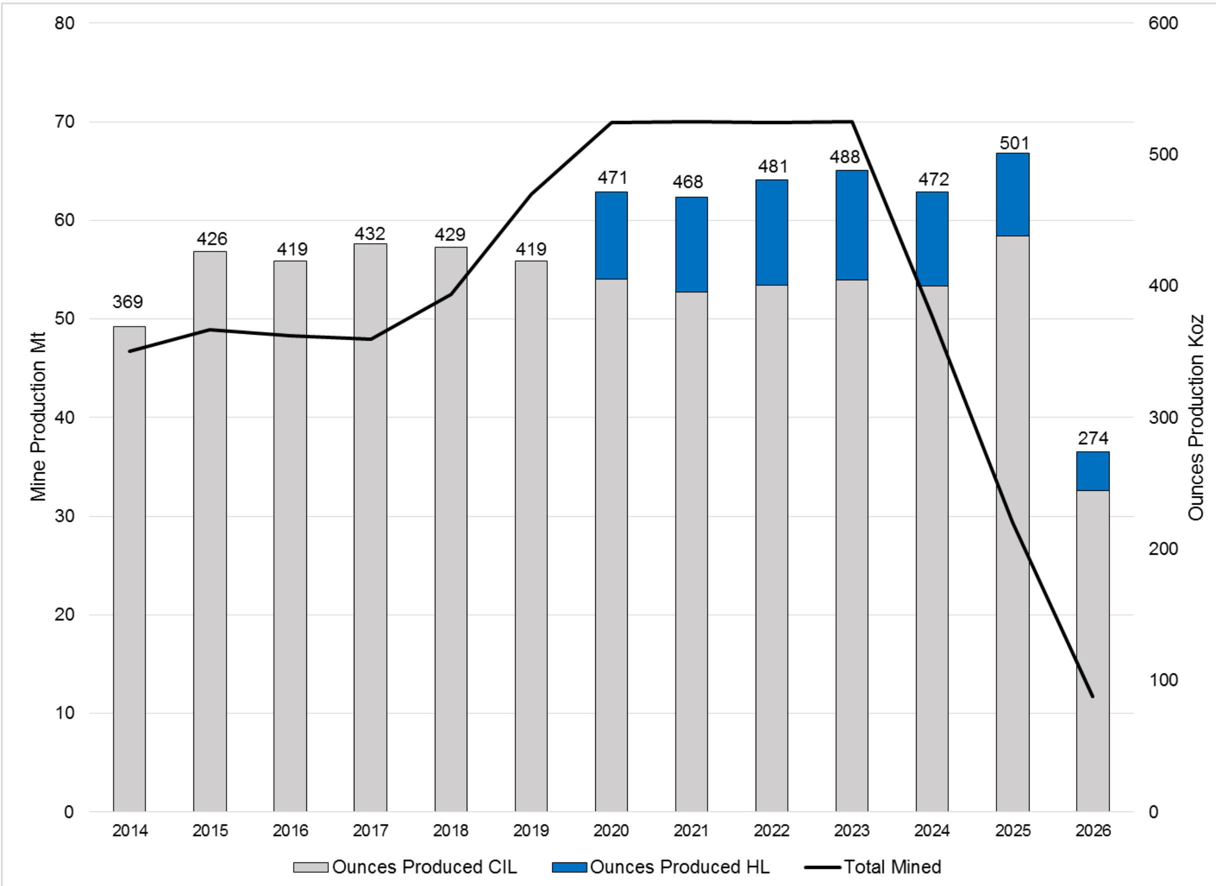
- Maximum of 70 Mt per year
- Maximum of 22 Mt per mining phase
- Limit of 7,000 tonnes of ore per day from the Falagountou sectors
- Maximum vertical advancement of 60 m to 70 m per year
- Maintaining a minimum of 400,000 oz per year at CIL plant

Table 16-12 and Figure 16-8 show the mining production over the LOM.

TABLE 16-12 ESSAKANE GOLD MINE LOM PLAN

	Unit	2018	2019	2020	2021	2022	2023	2024	2025	2026	Total
Ore Mined	Mt	12.3	17.4	18.3	15.1	19.4	18.3	18.4	14.5	8.3	**142.0**
Waste Mined	Mt	40.2	45.4	51.6	54.9	50.5	51.7	31.8	14.9	3.4	**344.5**
Total Mined	Mt	52.5	62.7	69.9	70.0	69.9	70.0	50.2	29.4	11.7	**486.5**
Strip Ratio		32.7	2.6	2.8	3.6	2.6	2.8	1.7	1.0	0.4	**2.4**
Mined Ore Grade	g/t Au	1.13	0.85	0.95	0.96	0.86	0.85	0.86	1.08	1.15	**0.94**
Ore Milled CIL	Mt	13.1	13.9	12.2	12.2	12.3	11.7	10.6	10.5	5.4	**101.9**
Mill Grade CIL	g/t Au	1.11	1.02	1.12	1.09	1.10	1.16	1.27	1.41	1.53	**1.17**
Recovery CIL	%	92.1	92.5	92.2	92.2	92.1	92.1	92.0	92.0	92.0	**92.2**
Gold Produced CIL	koz Au	429.3	419.2	405.3	395.1	400.4	404.3	400.0	437.8	244.3	**3,535.7**
Ore Mill HL	Mt	-	-	7.5	9.9	10.0	10.0	10.0	10.0	4.4	**61.8**
Mill Grade HL	g/t Au	-	-	0.50	0.41	0.46	0.47	0.40	0.36	0.38	**0.43**
Recovery HL	%	-	-	55.0	55.0	55.0	55.0	55.0	55.0	55.0	**55.0**
Gold Produced HL	koz Au	-	-	66.1	72.6	80.5	84.0	71.5	63.5	29.7	**467.9**
Total Ore Milled	Mt	13.1	13.9	19.7	22.2	22.3	21.7	20.6	20.5	9.8	**163.8**
Mill Grade	g/t Au	1.11	1.02	0.88	0.79	0.81	0.85	0.85	0.90	1.01	**0.89**
Recovery	%	92.1	92.5	84.2	83.4	82.7	82.5	83.5	84.8	85.7	**85.1**
Gold Produced	koz Au	429.3	419.2	471.4	467.8	480.9	488.3	471.5	501.3	274.0	**4,003.7**
RC Drilling	Km	145.8	154.5	154.1	116.0	117.3	114.6	97.5	68.7	35.1	**1,003.6**
Production Drilling	Km	781.4	935.4	1,011.9	1,029.5	1,056.5	1,047.2	710.0	407.7	153.4	**7,133.0**
Pre-Split Drilling	Km	120.7	162.8	178.3	211.9	202.1	141.9	211.6	156.7	37.6	**1,423.6**
Tonnes Blasted	Mt	50.7	60.7	68.4	68.7	69.9	69.7	50.2	29.4	11.7	**479.4**

FIGURE 16-8 MINING PRODUCTION MILL



17 RECOVERY METHODS

17.1 CIL RECOVERY METHODS

Ore is processed using two stages of crushing, semi-autogenous grinding, ball mill grinding, pebble crusher grinding (SABC), gravity concentration, and a CIL gold plant. The 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 mill feed has gradually increased from 2012 onwards. To maintain gold production levels, with increasing proportions of hard rock in the mill feed, an expansion was completed in 2014. The objective was to double the hard rock processing capacity from 5.4 Mtpa on a 100% hard 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 mill.

- Secondary crusher of 1 MW;
- SAG mill of 7 MW;
- Ball mill of 7 MW;
- A pebble crusher on line A and line B;
- Two gravity concentrators; and
- Eight CIL tanks.

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

The expanded mineral processing flow sheet is shown in Figure 17-1.

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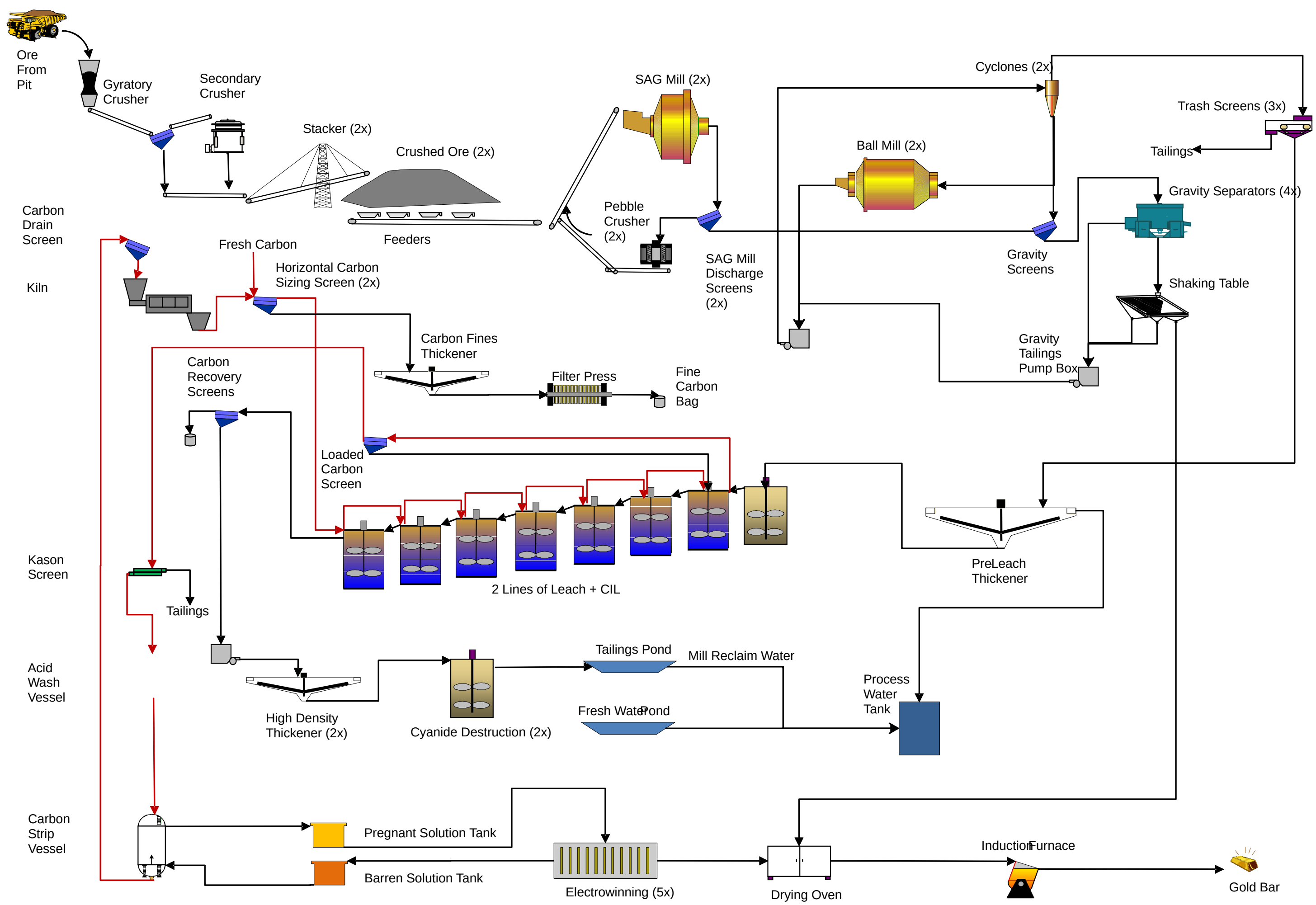

Figure 17-1

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Mineral Processing
Flow Sheet for the CIL Plant

July 2018

Source: Essakane S.A., 2018.

The ore coming from the mine 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 cathode obtained from the intensive leach reactor is then dried and smelted together with the cathodes from the elution circuit.

Plant tails are thickened and tails are stored in a tailings pond and water is recovered to the plant.

Table 17-1 summarizes mill throughput, head grade, recovery, and gold production since commissioning in July 2010. Figure 17-2 summarizes the yearly average recovery by gravity, recovery for the mill, and the head grade since 2010.

TABLE 17-1 MILL PRODUCTION SINCE COMMISSIONING IN JULY 2010

		2010	2011	2012	2013	2014	2015	2016	2017	2018 Jan-May
Throughput	000 t	2,973	7,977	10,762	10,613	11,897	11,716	12,006	13,891	5.516
Head grade	g/t Au	1.49	1.53	1.10	0.89	1.06	1.23	1.22	1.07	1.21
Recovery	%	95.7%	95.4%	91.9%	91.7%	90.7%	91.7%	89.0%	90.3%	91.5%
Gold Production	000 oz	136	375	350	277	369	426	422	432	197

FIGURE 17-2 HISTORICAL RECOVERIES AND HEAD GRADES

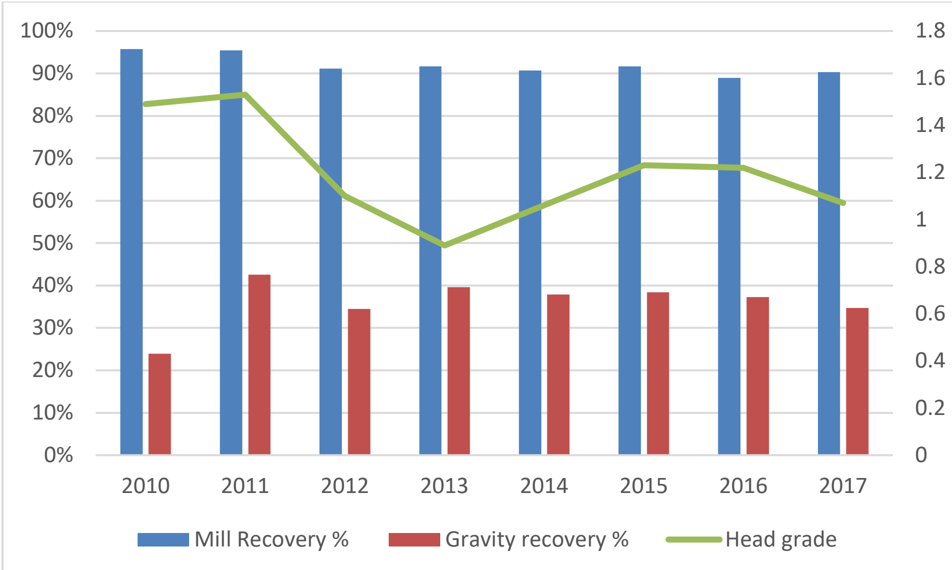

Table 17-2 summarizes the 2017 mill production.

TABLE 17-2 2017 ACTUAL MILL PRODUCTION

Month		Tonnes (000 t)	Head Grade (g/t Au)	Recovery (%)	Gold Production (000 oz)
Jan	Actual	1,188	1.05	87.4%	35
Feb	Actual	1,066	1.08	84.8%	31
Mar	Actual	1,227	1.07	87.8%	37
Apr	Actual	1,156	1.15	90.4%	39
May	Actual	1,194	1.05	91.3%	37
Jun	Actual	1,121	1.13	89.9%	37
Jul	Actual	1,126	1.08	92.8%	36
Aug	Actual	1,129	1.02	90.0%	33
Sep	Actual	1,112	1.03	92.8%	34
Oct	Actual	1,179	0.98	92.1%	34
Nov	Actual	1,241	1.02	90.0%	37
Dec	Actual	1,152	1.20	94.3%	42
Total	**Actual**	**13,891**	**1.07**	**90.3%**	**432**

Table 17-3 summarizes the actual ore tonnes milled and recovery achieved for 2016 and 2017 compared with the mine plan tonnage and recovery. In 2016, the total ore tonnes milled was slightly higher compared to the mine plan; the actual grade was higher than the mine plan as well. The actual ratio of saprolite, transition, and hard rock ore differed in 2016 from the mine plan and consequently, the recovery was lower than planned. In 2017, total tonnage treated was higher than the mine plan due to the higher than target percentage of saprolite in the mill feed and optimized SAG mill liners which permitted an increase in mill throughput. The recovery and head grade for 2017 were in line with the mine plan.

TABLE 17-3 2016 AND 2017 ACTUAL MILLING SUMMARY COMPARED TO MINE PLAN

	2016		2017	
	Mine Plan	**Actual**	**Mine Plan**	**Actual**
Ore Milled (000 t)	12,000	12,006	12,953	13,891
Saprolite (%)	0	8.5	14	11.6
Transition (%)	13	18,6	1	2.7
Hard Rock (%)	87	72.9	85	85.7
Mill Grade (g/t)	1.16	1.22	1.10	1.07
Recovery (%)	92.0	89.0	90.8	90.3

17.2 CONCENTRATOR MODIFICATIONS FOR THE TREATMENT OF GOLD LOADED HEAP LEACH CARBON

With the start of the heap leach (HL) operation, it is planned that a daily batch of 4.5 tonnes of gold loaded carbon will be transported by truck from the HL facilities to the existing Essakane concentrator.

Modifications to the concentrator will be required in the gold desorption and recovery circuits in order to process the additional carbon from the HL carbon-in-column (CIC) circuit. New carbon handling equipment, for both the loaded and the fresh/regenerated carbon, will also be required at the concentrator.

The general design criteria for the treatment of HL carbon are presented in Table 17-4.

TABLE 17-4 DESIGN CRITERIA FOR THE DESORPTION OF HL CARBON

Description	Unit	Value
Mass of CIL carbon treated	tpd	17
Gold grade on the CIL carbon	g Au/t C	1,300
Mass of HL carbon treated	tpd	4.5
Gold grade on the HL carbon	g Au/t C	2,500
Total carbon to be treated	tpd	21.5
Number of batches per day	batch/d	1.3
Quantity of carbon treated per batch	t	17
Eluted CIL carbon gold grade	g/t	100
Eluted HL carbon gold grade	g/t	100
Fraction of carbon acid washed	%	100
Fraction of carbon eluted	%	100
Fraction of carbon regenerated	%	73

A simplified flowsheet showing the HL carbon treatment is presented in Figure 17-3.

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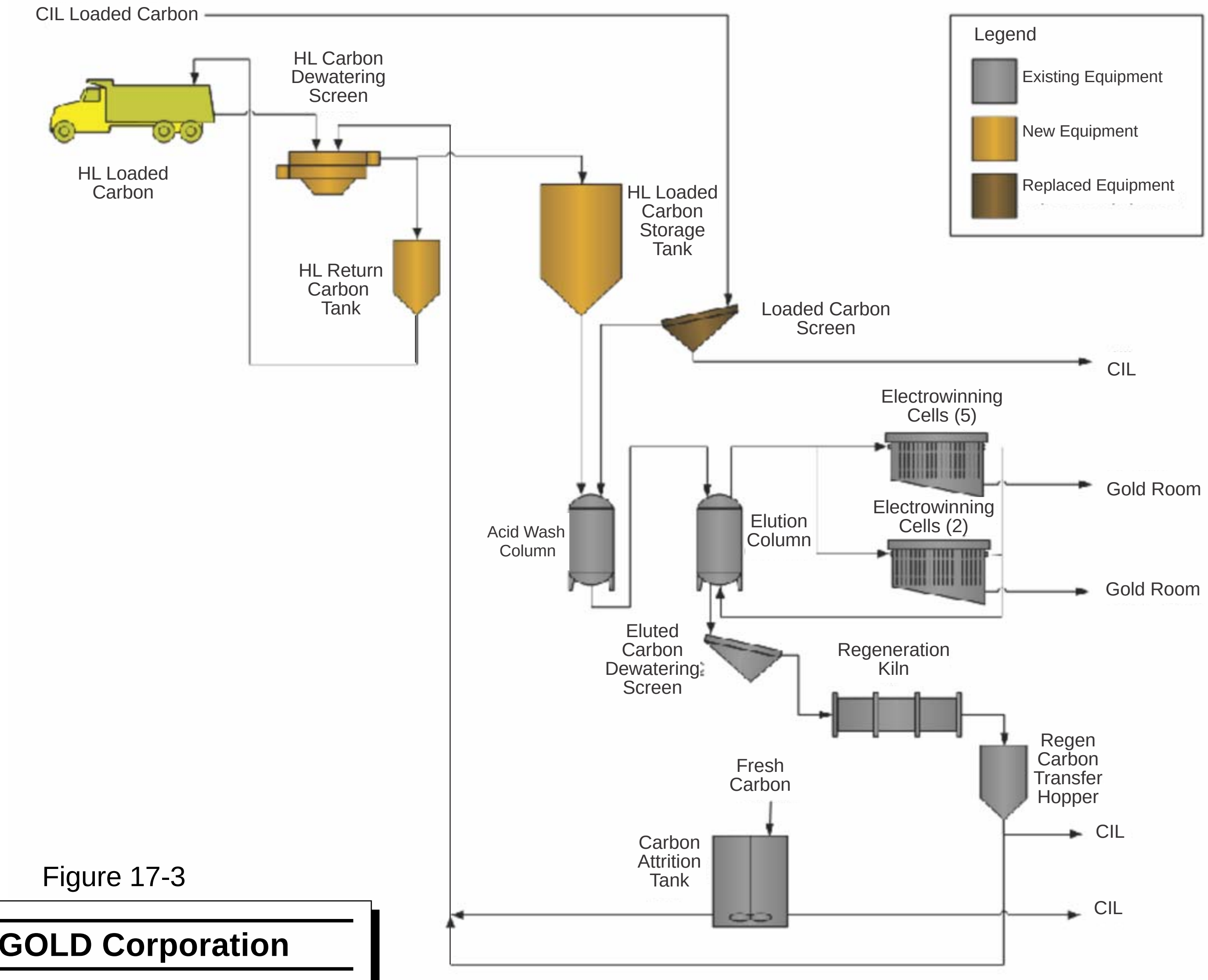

Figure 17-3

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

HL Treatment Circuit Modifications Simplified Flowsheet

Source: Kappes, Cassiday & Associates, 2018.

July 2018

17.2.1 HL LOADED CARBON RECEPTION

A 4.5 tonne batch of gold loaded carbon from the HL facilities will be trucked to the concentrator every day. At the concentrator, water will be added to the truck bin to transfer the carbon slurry to the dewatering screen. The dewatered carbon will be stored in a tank until a batch of at least 17 tonnes is obtained (after approximately four days).

The loaded carbon coming from the HL will be sampled in order to ensure a metallurgical mass balance can be completed independently for the heap leach carbon.

17.2.2 ACID WASH AND ELUTION

The existing stripping circuit is designed to process 17 tonnes of carbon batches. Striping frequencies are currently at one per day with an elution time of 11 hours. To accommodate the additional carbon from the HL, the strips frequency will be increased from one per day to 1.3 per day, thus reducing the available downtime between strips.

To allow the 1.3 elutions per day to be achieved, the existing loaded carbon screen will be replaced by a larger unit. By increasing its capacity, the transfer time between the CIL head tanks and the acid wash column will be reduced.

In between regular CIL carbon strips, an HL carbon elution cycle will be performed. The HL gold loaded carbon will be re-slurried and pumped to the existing acid wash column. It will undergo the identical acid wash and elution treatment as the CIL carbon and the cycle times will be the same.

The eluted carbon, the pregnant solution, and the barren solution will be sampled every cycle in order to account for gold.

17.2.3 ELECTROWINNING

Considering the frequency of the elutions of 1.3 per day, two additional electrowinning cells of the same model as the existing ones will be added to increase the flexibility of the operation. This will bring the number of electrowinning cells to seven units in parallel. Five electrowinning cells will be in operation during an elution cycle. The increased number of cells will allow the elution cycle to be started while the cleaning of the cathode is finalized.

The cathodes of all electrowinning cells will be cleaned before and after a cycle with HL carbon. This will ensure that the gold from each source can be accurately accounted for.

17.2.4 CARBON REGENERATION

No changes will be required to the carbon regeneration circuit. A portion (about one quarter) of the carbon will by-pass the regeneration kiln and be reused as is in the adsorption circuits.

17.2.5 FRESH AND REGENERATED CARBON LOAD-OUT

Fresh and regenerated carbon from the existing carbon regeneration circuit will be transferred to a 4.5 tonne capacity load-out tank. Prior to the tank, the carbon slurry will be dewatered using the same screen as the one used to dewater the HL loaded carbon pumped from the truck.

The carbon slurry will be pumped from the load-out tank to the truck. The truck will be drained using a built-in mesh in order to remove water prior to its transport back to the HL facilities.

17.2.6 REAGENTS AND UTILITIES

No changes to the capacity of the reagents, water and air services will be required to process the additional 4.5 tpd of loaded carbon coming from the HL.

17.2.7 POWER REQUIREMENT

The modifications made to the concentrator to treat the additional carbon from the HL circuit require a few additional electrical equipment such as pump motors, screen motors, and rectifiers for the electrowinning cells. Cable trays, grounding, lighting, and services were also added in the new equipment area. The new components require an additional installed power of 107 kW. For the electrical connections of the new loads, it was assumed that space is available in Motor Control Centres (MCC) located in nearby electrical rooms. No site survey or research in process plant single-line diagrams was done; it shall be executed at the next engineering stage.

17.2.8 PLANT LAYOUT

The new equipment that includes the HL carbon dewatering screen, the HL return carbon tank and the HL loaded carbon storage tank will be installed on a structure located in an area close to the existing elution equipment. The structure will be installed north of Leach Tank No. 2.

The dewatering screen will be installed on the top level, the HL return carbon tank will be located below and the HL loaded carbon storage tank will be located on the lower level.

The new loaded carbon screen will be located at the same location as the existing screen. The footprint of the screen is larger and an extension of the platform will have to be made. The structure holding the screen will be reinforced.

The two new electrowinning cells will be located in the refinery. One cell will be fitted with the existing cells and an extension of the building will be made to accommodate for the other cell. The opportunity of replacing the existing cells with higher capacity cells to increase the overall capacity will be evaluated in the feasibility study.

17.3 HEAP LEACH RECOVERY METHODS

17.3.1 HEAP LEACH DESIGN BASIS

In an effort to extend the mine life at the Essakane Project, the option of adding an HL circuit to the existing CIL plant has been reviewed in a scoping study completed by KCA in May 2017. The HL has the potential to turn material that would have been waste for the CIL plant into ore, giving the ability to access additional resources and extend the life of the Project.

The scoping study presented potentially economic costs utilizing a conventional crushing system and a permanent single use HL pad with a processing rate of 10 Mtpa. After the completion of the 2017 scoping study, a series of trade-off studies reviewing different processing rates, the possibility of utilizing a HPGR crusher for tertiary crushing and the possibility of adding a dump leach to supplement the heap were conducted.

Based on the trade-off studies and the first round of column testing, the Essakane HL project has been designed as an open-pit mine with HL operation utilizing a multiple-lift, single-use pad. Engineering and design of a 10.0 Mtpa processing plant was undertaken for complete crushing, leaching, and carbon adsorption systems. Material will be crushed using a three stage HPGR crushing plant. Crushed material is heap leached conventionally as a multiple lift heap in 10 m lifts. Gold and silver are leached using a dilute cyanide solution and recovered from the solution using a carbon adsorption process. The loaded carbon will be transported to the existing strip plant for doré production. A summary of the processing design criteria used for the design of the processing circuit are summarized in Table 17-5.

Testwork developed by KCA has indicated that the low grade Essakane material crushed with an HPGR is amenable to cyanide heap leaching with an estimated field gold recovery of 55% and silver recovery of 21% at a crush size of P_{80} = 8.0 mm. Based on an established processing rate of 27,397 tpd, the HL project has an estimated 6.2 year mine life.

The simplified HL process flowsheet is presented in Figure 17-4. The site general layout is presented in Figure 17-5.

Additional layout drawings are presented in Figures 17-6 through 17-7.

TABLE 17-5 PROCESSING DESIGN CRITERIA SUMMARY

Item	Design Criteria
Annual Tonnage Processed	10,000,000 tpa
Average Gold Feed Grade	0.43 g/t
Production Rate	27,397 tpd
Recovery of Gold	55%
Recovery of Silver	21%
Crushing Operation	12 hours/shift, 2 shifts/day, 7 days/week, 365 days per year
Crushed Product Size	80% passing 8.0 mm
Primary and Secondary Crusher Availability	70%
HPGR Crusher Availability	90%
Stacking Availability	70%
Heap Construction	3 Phase
Heap Leaching Cycle	90 days
Heap Pumping Availability	98%
Nominal Solution Application Rate	10 L/h/m^2
Nominal Adsorption Treatment Flow	1,541 m^3/h
Adsorption Circuit Availability	98%

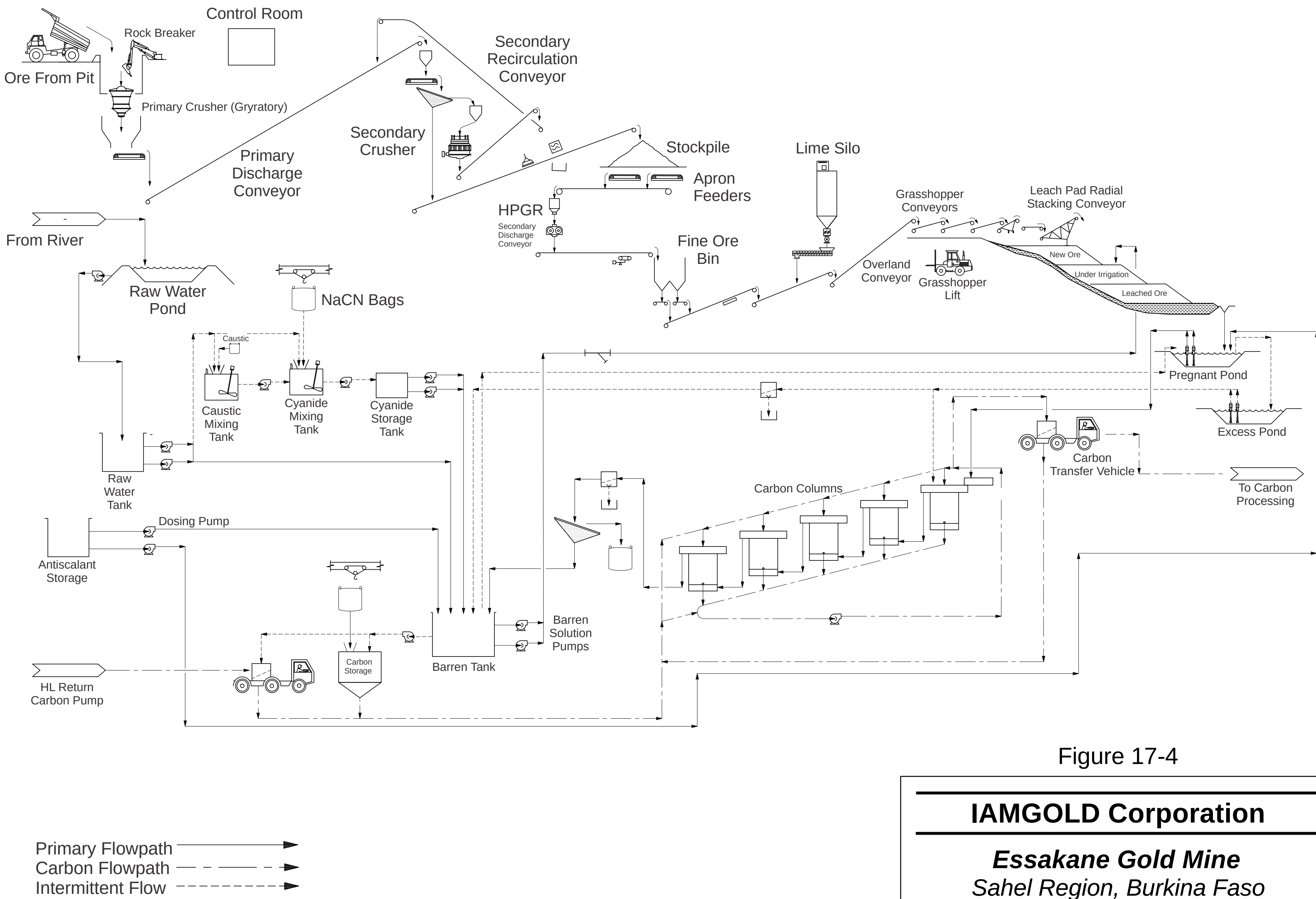

Figure 17-4

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Essakane Gold Mine

Sahel Region, Burkina Faso

Simplified Heap Leach Process Flow Sheet

July 2018

Source: Kappes, Cassiday & Associates, 2018.

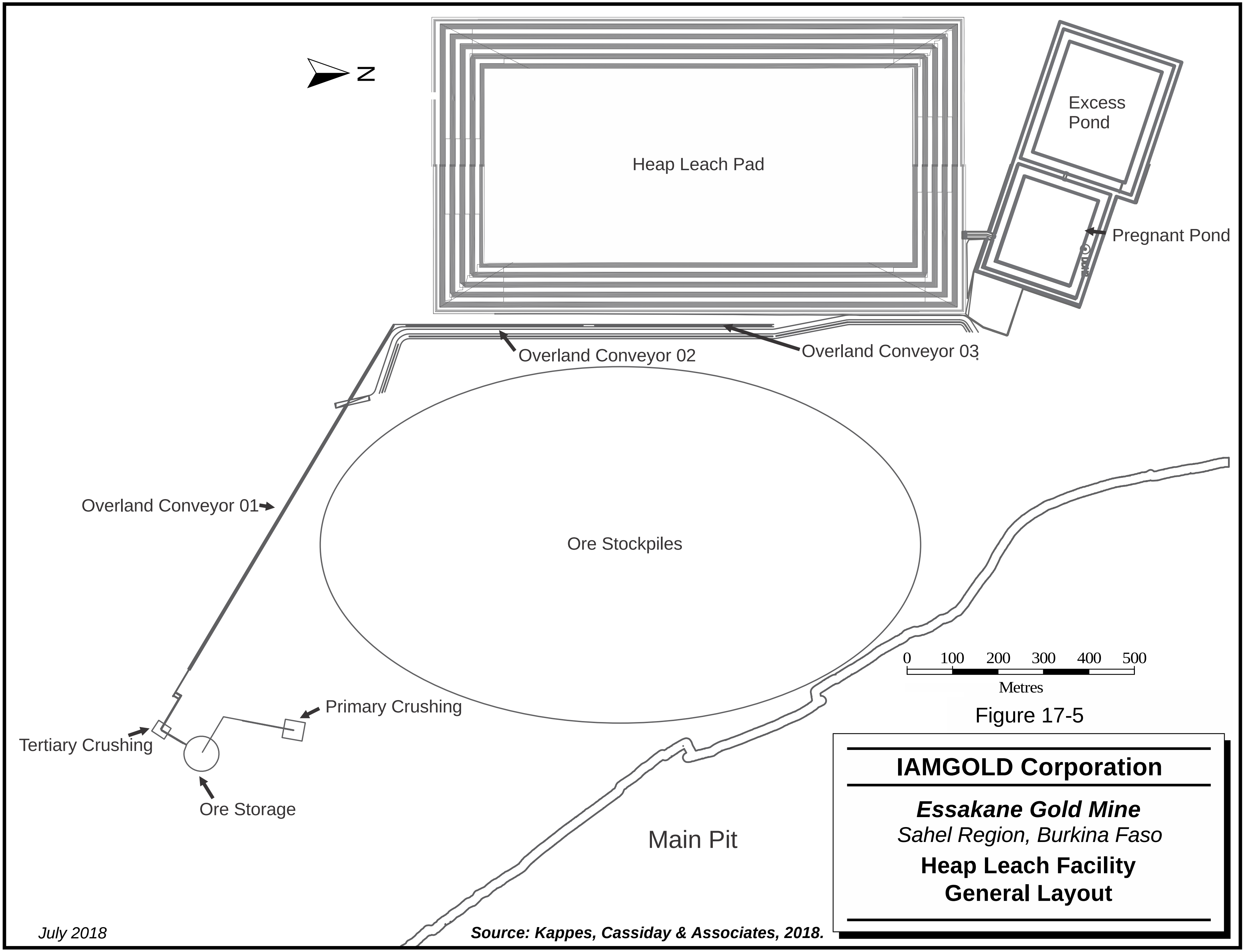


Figure 17-5

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Heap Leach Facility General Layout

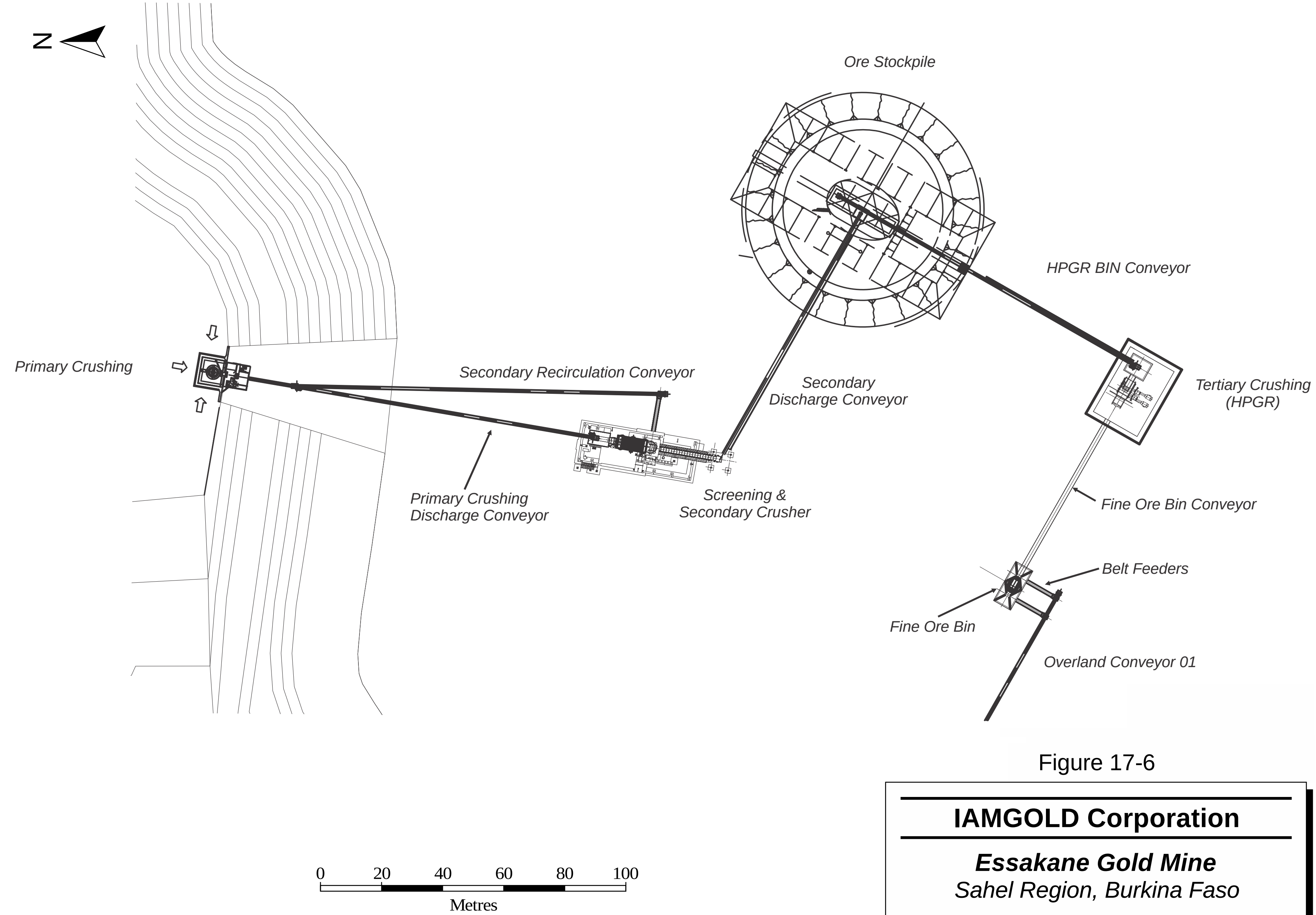


Figure 17-6

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Heap Leach Crushing Plant Layout

July 2018

Source: Kappes, Cassiday & Associates, 2018.

IAMGOLD Corporation

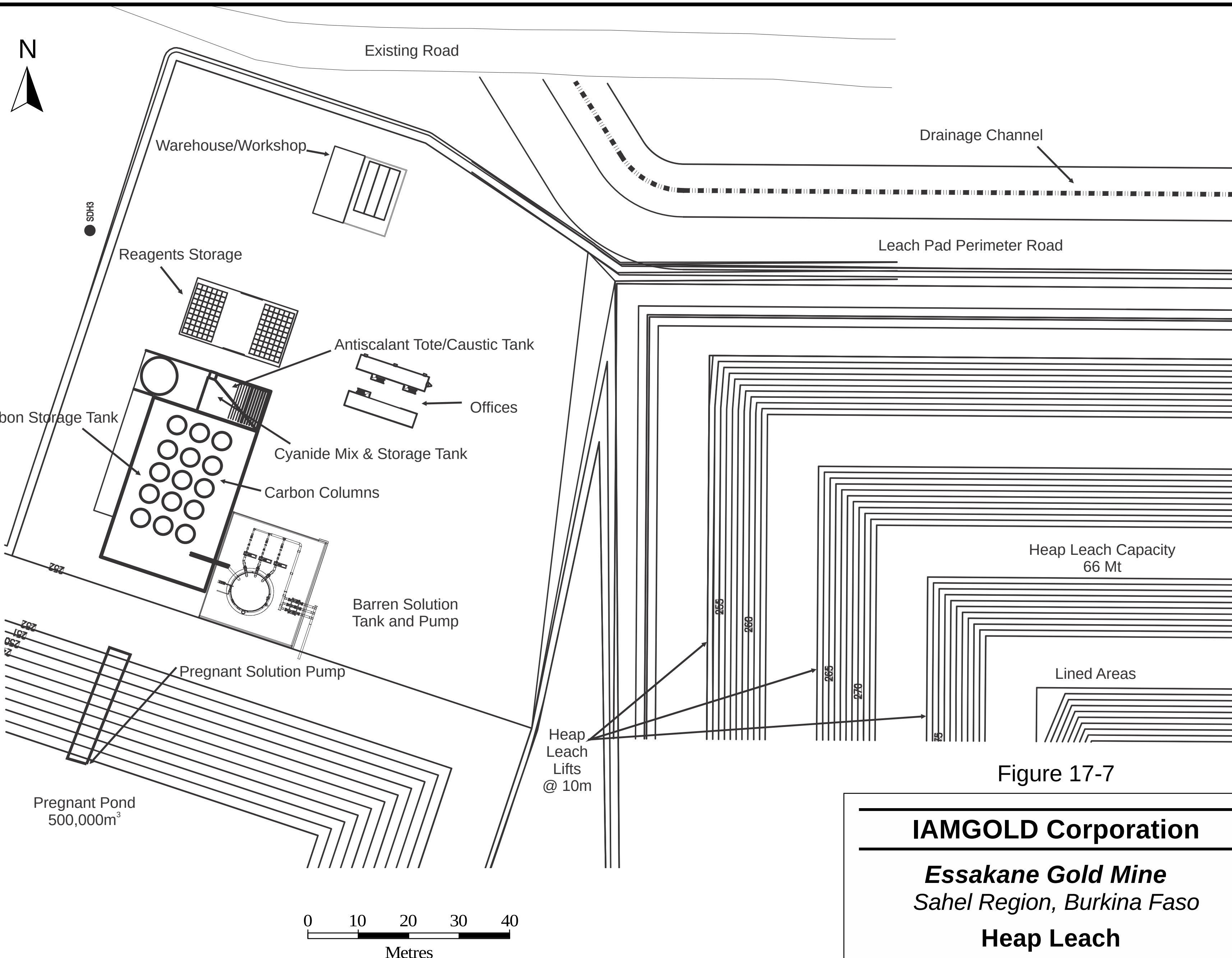


Figure 17-7

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Heap Leach
Adsorption Plant Layout

July 2018

Source: Kappes, Cassiday & Associates, 2018.

17.3.2 HEAP LEACH CRUSHING

Crushing for the Project is accomplished by a three-stage crushing system with an open primary crushing circuit, a closed secondary crushing circuit, and an open tertiary crushing circuit operating seven days/week, 24 hours/day at a rate 27,397 tpd. ROM material will be delivered and direct dumped, as much as possible, by haul trucks from the mine to gyratory crusher dump pocket. A front-end loader will also deliver material from a ROM stockpile into the gyratory crusher dump pocket. A rock breaker will be installed at the dump pocket and used to break any oversized material. ROM material from the dump pocket will be crushed using a primary gyratory crusher at an average rate of 1,631 dry tph and the primary gyratory product will discharge into a crushed ore bin directly below. The crushed ore bin is fed onto a 1,524 mm primary crusher discharge conveyor by an apron feeder. The 50-65 primary gyratory crusher is designed to crush ROM ore to 100% passing 314 mm.

Primary crushed ore is combined with the secondary cone product and is fed to the secondary screen feed bin. Material is reclaimed from the secondary feed bin with a 1,829 mm apron feeder onto the secondary screen. The secondary screen is a 4.2 m by 8.5 m double deck banana screen. The secondary screen oversize feeds directly to the MP 1250 secondary cone crusher. The secondary cone crusher discharge recycles back to the secondary screen. The secondary circuit is designed to crush ore to 80% passing 38 mm (100% passing 50 mm).

Secondary screen undersize material is placed on the secondary stockpile by the 1,219 mm secondary discharge conveyor, reclaimed using 1,829 mm apron feeders and is transferred to the tertiary crushing circuit.

The tertiary crushing circuit consists of small bin to choke feed a 170/140 HPGR crusher operated in open circuit with a final crushed product of 100% passing 19 mm. The product from the HPGR is conveyed to a fine ore bin.

The crushing system is also set up to provide the drainage gravel for the HL pad over-liner. The screen middlings can be diverted straight onto a mobile conveyor for stockpiling while the oversized material is recirculated to the cone crusher and the fines are passed through a temporary stockpile.

17.3.3 HEAP LEACH SYSTEMS

17.3.3.1 CONVEYING AND STACKING

Crushed ore from the fine ore bin is reclaimed using belt feeders and is transferred to the 1,219 mm heap feed conveyor by the fine ore discharge conveyor. Quicklime is added to the ore on the heap feed conveyor at a rate of 1.3 kg/t ore from a 100 t silo equipped with bin activators, a screw feeder, a variable speed feed conveyor, and a dust collector. The quicklime addition rate is controlled by the output of a weightometer mounted on the conveyor belts.

The heap feed conveyor discharges onto a 1,219 mm overland conveyor which conveys the crushed ore to the HL stacking system.

The HL stacking system will be constructed in 80 m wide, 10 m high lifts using a 1,219 mm mobile conveyor stacking system. The Phase 1 leach pad conveying and stacking system will consist of an overland conveyor, four mobile ramp conveyors, 22 mobile grasshopper conveyors, an index feed conveyor, a horizontal index conveyor, and a radial stacker. The overland conveyor transfers material to the mobile conveyors which feed the conveyor stacking system. As the radial stacker progresses, the system is periodically stopped to add or remove grasshopper conveyors, as needed. Phases 2 and 3 will increase the leach area without any additional equipment required.

Once a lift of cells has finished leaching, and is sufficiently drained and dry, a new lift can be stacked over the top of the old lift. The old lift will be cross-ripped with a dozer prior to stacking the new lift to break up any compacted ore sections and to redistribute material that may have been broken down by the irrigation solution or rainfall. Stacked lifts will progress in a stair-step manner.

17.3.3.2 SOLUTION APPLICATION AND LEACHING

Following stacking, the material is irrigated with a dilute sodium cyanide barren leach solution and the resulting gold and silver bearing solutions are collected into the pregnant solution pond. The Essakane project has been designed as a single pass system with no recycle of intermediate solutions to the HL. The heap will be irrigated using a drip-tube irrigation system for solution application. High Density Polyethylene (HDPE) pipes are used to distribute the solution to the drip-tubes on top of the heap. Antiscalant agent is added at the barren and pregnant solution pumps' suction inlets to reduce the potential for scaling problems within the system.

The total leach cycle of 90 days has been designed for the HL system, which is based upon metallurgical test work to date. Leach solutions will be applied to the ore at a nominal application rate of 10 L/h/m^2 with an approximate cyanide concentration of 150 ppm to the heap.

Three vertical turbine pumps operating in parallel at the barren tank will be used for the barren solution application to the heap.

The barren pumps will be mounted inside the barren tank along with a process solution pump along the side of the tank. High-strength sodium cyanide solution and an anti-scalant agent will be added to the barren tank by metering pumps. The combined nominal flow to the heap is 1,541 m^3/h.

Gold and silver bearing solutions draining from the leach pad are collected by a network of perforated drainage pipes and are directed to the pregnant solution pond. Pregnant solution is pumped from the pregnant solution pond by submersible pumps to the head tank of the carbon adsorption columns. The pregnant solution will flow by gravity through the carbon in the columns before being returned to the barren solution tank.

17.3.3.3 SOLUTION STORAGE

The solution containment and storage system includes the following facilities:

- Barren Solution Tank
- Pregnant Solution Pond
- Storm Water Excess Solution Pond

The barren solution tank is a 9.9 m diameter by 10.2 m high carbon steel tank and has been sized to provide 30 minutes of operating storage capacity at the design flow rate of 1,849 m^3/h. The operating range is between the tank overflow and the minimum depth of 4 m to provide the suction head required by the turbine pumps.

The pregnant and excess solution ponds have been sized to ensure that all the leach solutions can be managed in a controlled manner to prevent any discharges of solution to the environment. The pregnant solution pond has a capacity of 500,000 m^3 and is sized to hold a 12 hour working volume, 24 hours of drain-down and a two-year 24 hour storm.

The excess solution pond will be constructed in two phases and will have an initial capacity of 300,000 m^3 and is sized to hold the wet season accumulations plus 24 hours of heap drain-down and the 24 hour/100-year storm event (less the two-year storm event accounted for in the pregnant solution pond) over the entire lined area. During the Phase 2 heap expansion in Year 2 of operations, the event pond will be expanded to 750,000 m^3 during the dry season. The 750,000 m^3 pond will be sufficient to handle the expanded lined area for Phase 2 and Phase 3.

The pregnant pond utilizes a double 2 mm HDPE liner system on top of 300 mm of compacted soil liner. Leak detection is provided by geonet sandwiched between the two HDPE liners on top of a low permeability soil liner and a collection system to detect any solution between the liners in the event there is leakage through the primary liner. There is a second similar leak detection and collection system installed under the bottom HDPE liner and the compacted soil liner or GCL. This type of double-redundancy liner and leak detection system significantly reduces the possibility of solution entering the environment below the pond.

The excess solution pond utilizes a single 2 mm HDPE liner system on top of 300mm of compacted soil liner. There is a leak detection and collection system installed under the HDPE liner and above the compacted soil liner. The leak detection systems are checked and logged for solution each shift during operations.

17.3.4 HEAP LEACH FACILITY

The preliminary Heap Leach Facility (HLF) is designed for low environmental risk to soils, surface water, and groundwater in and around the site. The HLF will be constructed in three phases. The first phase is designed for two years of operation, the second phase is intended to come on line in Year 3 and the third phase is intended to come on line in Year 5. The staged construction of the heap is intended to minimize the up-front capital costs and help with the water balance. Phase 2 will expand the HL pad and the excess solution pond. The Phase 3 expansion will only increase the size of the HL pad.

The HLF is intended to operate as a zero discharge system; therefore, the design includes provisions to accommodate upset conditions such as severe storms and temporary loss of electric power or pumps.

The HL pad will have the following lining system from top to bottom:

- 700 mm of 30 cm – 50 cm sized drainage gravel pad cover above the lined surface.
- 2 mm Linear Low Density Polyethylene (LLDPE) single side textured geomembrane.
- 300 mm of compacted soil.

The drainage gravel pad cover over-liner is placed on the top of the geomembrane to protect the liner and act as a basal drainage layer. The HL crushing system is designed to allow for the production of the gravel pad cover. Perforated collection pipes are embedded in the gravel layer to enhance solution drainage and provide a rapid return of pregnant solution after it has passed through the ore. The piping and collection layer also minimizes the depth of solution (head) over the liner system.

An under-drain system consisting of perforated pipes is installed below the low permeability soil liner to collect and convey any near surface underground water below the pad. In addition, the under-drains act as an early leak detection system.

17.3.5 HEAP LEACH SOLUTION HANDLING AND MANAGEMENT

The Essakane HL and processing facilities are designed as zero discharge facilities for both surface water and groundwater. Pregnant solution from the heap is collected in a pregnant solution pond. The pregnant solution is pumped from the pregnant solution pond to a carbon adsorption circuit and the resulting barren solution is transferred to the barren solution tank where it is pumped back to the heap. An excess solution pond is present to accommodate seasonal variations from the process as well as storm water surges from the lined area.

Solution management for the Essakane Project is generally simple. Solution in the pregnant solution pond should be maintained in the mid-to-lower range of its working capacity. The excess solution pond should normally be maintained at empty or low levels whenever possible. When solution is diverted to the excess solution pond, it should be pumped back to the leach system at the barren tank as make-up solution as soon as practical. Every effort should be made to avoid storing excess solution over a long period of time.

17.3.6 HEAP LEACH METAL RECOVERY

Precious metals in the HL pregnant solution will be adsorbed on to activated carbon in the carbon adsorption circuit. Loaded carbon from the carbon adsorption circuit is then transferred by truck to the existing adsorption-desorption-recovery (ADR) plant at the Essakane CIL facility to recover the gold and silver.

17.3.6.1 ADSORPTION

The adsorption plant will consist of three carbon column trains each consisting of five cascade type open-top up-flow carbon adsorption columns. The number of carbon trains is selected based on the flow through the columns of 60 $(m^3/h)/m^2$ and the required minimum settled carbon depth in each column. Each of the carbon columns will have a capacity of 4.5 tonnes of activated carbon.

Pregnant solution is pumped to the carbon adsorption columns by submersible pumps in the pregnant solution pond. Anti-scalant agent is added at the pump suctions to prevent scaling of the carbon that can affect carbon loading. Barren solution exiting the last carbon columns flows through a screen to separate and capture any floating carbon from the solution.

Adsorption of gold and silver from the pregnant solution is a continuous process. Periodically the carbon contained in the lead column(s) in the series becomes loaded with gold and silver and is transferred to portable tanks on a trailer to be transferred to the existing ADR plant at the CIL facility for stripping.

Recessed impeller pumps will transfer the carbon to the portable tank, already partially filled with water to prevent carbon attrition. A screen in the bottom of the tank will allow for solution removal while keeping the carbon in the tank. After the tank load of carbon is transferred to the strip plant at the CIL plant, similar recessed impeller pumps, with the aid of flush water, will be utilized to transfer the carbon for stripping.

Carbon in the remaining columns is then advanced using recessed impeller pumps, one at a time, and a batch stripped/regenerated carbon from the CIL strip plant is transferred into the final empty column from a portable carbon tank.

Generally, the stripping of carbon from the heap will occur one or two times each week.

17.3.6.2 REAGENT MIXING AND HANDLING

The reagent mixing and handling system includes equipment used to mix and store sodium cyanide batches and to add sodium cyanide to the barren solution tank, and to the barren leach solution system. Reagent mixing and storing are at ambient temperature and pressure.

Solid, sodium cyanide briquettes are delivered to the site in 1,000 kg bulk bags. Cyanide mixing will be performed in 4,000 kg batches. During mixing, raw water or barren solution is used to partially fill the cyanide mix tank and a small amount of sodium hydroxide (pumped from the caustic storage tank) is added to the tank prior to the addition of sodium cyanide briquettes. The caustic addition will ensure that proper alkaline pH is maintained, thereby minimizing waste of cyanide by dissociation and possible generation of toxic HCN gas.

An electric hoist is used to lift the sacks to the top of the cyanide mix tank. A bag breaker system is mounted above the mix tank to discharge cyanide briquettes into the mix tank. The tank is designed to contain and dissolve solid sodium cyanide briquettes and yield a solution containing 20% (by weight) sodium cyanide. After dissolution, the cyanide solution is transferred to a storage tank from which it is distributed.

All cyanide distribution lines will be double-containment, either by "pipe-within-pipe" or "pipe-over-liner" containment systems.

17.3.7 HEAP LEACH REAGENTS AND CONSUMABLES

Average estimated annual reagent and consumable consumption quantities for the processing area are shown in Table 17-6.

TABLE 17-6 HEAP LEACH REAGENT CONSUMPTION

Reagent	Form	Annual Consumption, t
NaCN	Briquettes - 1 tonne supersacks	3,300
Quicklime	Bulk Delivery Truck	13,000
Anti-scalant	Liquid Tote, 1m3	125

17.3.7.1 QUICKLIME (CAO)

Pebble quicklime is received by pneumatic self-offloading trucks and blown into a 100 t silo. The lime is delivered to the heap feed conveyor via a variable speed screw feeder at a rate proportional to the ore feed rate for pH control in the heap.

17.3.7.2 CYANIDE

Sodium cyanide is delivered as briquettes in 1,000 kg bulk bags and is stored in a covered storage area with approximately 30 days of storage. Sodium cyanide is used to leach gold and silver from ore on the HL and is consumed at 0.33 kg/t ore processed or 9.0 tonnes per day.

17.3.7.3 SODIUM HYDROXIDE

Sodium hydroxide (caustic) is delivered as a liquid (approximately 47% by weight in water) and transferred to the caustic storage tank. Caustic is used as a pH buffer for the mixing of sodium cyanide.

17.3.7.4 ANTI-SCALANT

Anti-scalant will be received in drums or plastic tote containers. Anti-scalant will be added by metering pumps at the pregnant solution pump suction inlets and the barren tank. Anti-scaling agents will be used to prevent carbonate scaling in pumps, piping and on the carbon.

17.3.8 HEAP LEACH WATER BALANCE

17.3.8.1 PRECIPITATION AND CLIMATE DATA

Based upon the Environmental Impact Study from May 2007 and the SRK water management report from November 2017, the solution management system for the Essakane HL was designed as a zero discharge facility.

Estimates of monthly precipitation at Essakane were developed based on a review of the Environmental Impact Study. The study reported dry, wet, and average year rainfall to be 87.3

mm, 1,127.5 mm, and 397.6 mm, respectively. Monthly precipitation was estimated based on the monthly rainfall presented for three nearby stations. The pan evaporation data used is the average data from the Dori station as presented in the Environmental Impact Study. Monthly rainfall and evaporation data are summarized in Table 17-7.

TABLE 17-7 RAINFALL DATA SUMMARY IN MM

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Total
Dry Year	0.0	0.0	0.4	0.7	3.6	9.8	24.4	31.8	13.9	2.7	0.0	0.0	87.3
Wet Year	0.0	0.0	5.8	8.7	46.3	127.2	315.1	410.5	179.2	34.7	0.0	0.0	1,127.5
Average Year	0.0	0.0	2.0	3.1	16.3	44.9	111.1	144.8	63.2	12.2	0.0	0.0	397.6
Evaporation	227.0	247.9	319.6	328.8	331.4	281.0	228.6	187.4	191.6	239.8	233.7	218.6	3,035.4

Major storm events were also taken into account including a two-year, 24 hour storm from the Environmental Impact Study of 66 mm and a 100-year, 24 hour storm event from the SRK Water Management Report of 171 mm.

17.3.8.2 WATER BALANCES

KCA prepared preliminary water balances for the proposed HL facility at Essakane. The water balance model approximates the circulation of solutions within the HL and process facility, as well as the introduction of precipitation and evaporation as a function of time. The results of the water balance model predict make-up water flow rates and minimum storage capacities necessary in order to achieve a zero-discharge system.

The model uses time steps of months, which provides monthly average flow rates and volumes, as opposed to peak daily or peak instantaneous rates. This approach may attenuate the peak rate, as it averages the volumes over a monthly period.

Based on the water balance, for an average precipitation year the Essakane HL is expected to operate in a water deficit. A nominal 2,300 m^3 of make-up solution will be required each day for the process. Any solution accumulated in the excess pond (primarily during the wet months) is returned to the plant as soon as possible as make-up solution. Peak water demand for the HL process during a dry year is 125 m^3/h.

17.3.8.3 HEAP LEACH POWER REQUIREMENT

Power usage for process plant and infrastructure was derived from estimated connected loads assigned to powered equipment from the mechanical equipment list. Equipment power demands under normal operation were assigned and coupled with estimated on-stream times to determine the average energy usage.

The power estimation includes the HL, crushing circuit, conveyors, adsorption plant, and heap infrastructure.

18 PROJECT INFRASTRUCTURE

18.1 GENERAL

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 scope and extent of the general services department required to support production is very substantial. As of May 31, 2018 the manpower status is 2,236 national workers and 95 expatriates, excluding contractors.

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 mine 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. These facilities will be upgraded with the heap leach expansion. The following summarizes the modifications to the main infrastructure built for the mill expansion and the ones required for the heap leach project.

18.2 MINE TRUCK SHOP AND WAREHOUSE

The existing mine truck shop and warehouse was extensively expanded to accommodate the increased maintenance requirements for the additional mobile mining equipment required for the mill expansion. An extension of the truckshop will be carried out by the end of 2018 to implement a component renovation center. No additional expansion will be needed for the heap leach.

18.3 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 3 km fenced road was built around the Falagountou East pit.

The heap leach area will be also surrounded by a 4 km long patrol road and new fence. It is planned to re-organize the haul road leading to the crushing area in order to accommodate the new heap leach crushing arrangement. 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.

18.4 COMMUNICATION SYSTEM AND IT

All the network communication and IT related hardware are linked to the existing systems at the mine. A 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 village. The heap leach area will be linked to the existing plant control room for remote monitoring of the facility.

18.5 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.

18.6 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.

18.7 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 mill area.
- Two existing potable/fire water tanks are supplying the mining camp with a 220 gpm pump and five hydrants and fire cabinets.
- The current camp capacity is 711 rooms (1,302 beds).

The addition of lodging capacity is planned for the heap leach project. It is expected that the heap leach will require approximately 115 new employees and that the majority of the new employees will be from the area and living in the neighbouring villages. Two additional lodging unit (64 rooms) will be installed to host the heap leach construction and operation workers from outside the area.

18.8 RIVER DEVIATION

In order to expand the EMZ pit to the north, a 5 km deviation of the Gorouol River was undertaken and effectively protected the EMZ pit during seasonal rains. No modifications are required for the Project.

18.9 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 degree Celsius) 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, 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 mill expansion. Four new electrical rooms were installed to supply energy to the new grinding circuit, the pebble crusher, the new screening, and crushing circuits.

In 2018, a PV solar plant has been constructed on the mine'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 AC. It consists in nearly 130,000 panels separated in three different fields, each field being connected to two inverters and injecting power to the grid through a transformer. The addition of this plant should reduce by an estimated 18,000 tons/year the carbon dioxide emissions caused by the mine's activities.

In order to meet the new 7 MW average and 8.8 MW peak power demand from the heap leach operation, a new Wärtsilä HFO generator type 12V32 with a 5.826 MW capacity is required be installed in the existing Wärtsilä 1 powerhouse, for a total of 12 installed HFO generators on site. This will allow the continuous use of 10 HFO generators while one can be in maintenance and one can be kept in spare. To meet the demand during peaks and in high temperature situations, the contribution of the HFO generators will be increased and the six LFO generators

SDMO already located on the mine's site would be required. Minimal investment is required to bring the SDMO online and connect them on to the Essakane grid. In those situations, in order to keep at least one LFO generator in spare at all times, not more than five of them will be running simultaneously, giving a backup capacity of 4.5 MW. This capacity combined with the 5.526 MW from the HFO unit in spare provides a total power capacity of 10.0 MW.

It has been estimated that these additions will increase the mine's HFO and LFO global consumption respectively by 13,450,000 litres and 800 litres annually and that its actual storage existing installations have the capacity to absorb this increase.

For the distribution of electrical power to the new heap leach equipment, four electrical substations will be constructed and a new double circuit 6.6 kV overhead line will be installed between the mine's HFO powerhouse and the four heap leach substation. The new electrical substations will be located respectively at around 1.1 km, 1.2 km, 1.9 km and 3.4 km away from the HFO powerhouse.

18.10 ASSAY AND METALLURGICAL LABORATORIES AND MILL OFFICE

The metallurgy and mill office buildings were rebuilt per the original design before demolishing the existing structure. The metallurgy building consists of two main sections: the metallurgical lab 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.

18.11 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 division: reception, boardroom, kitchen, offices, and map room.

18.12 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 the tank which will be specially lined to ensure that the water is then acceptable for human consumption.

In order to handle waste from the additional construction camps and influx of workers, two new 100 m^3/day sewage facilities were installed beside the existing facility, one at the mine village and the other at the mill area during the expansion. No further expansion is planned for the heap leach project.

An upgrade of the potable and fire water distribution system was tied-in to the existing distribution lines to supply water to the extension of the camp. Another upgrade will be necessary with the addition of the heap leach infrastructures.

18.13 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, Off Channel Reservoir (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 need approximately 1 Mm^3 of makeup water. The BWS system is currently being upgraded as part of the mine operations and will be able to provide the heap leach makeup water.

18.14 HEAP LEACH CONSTRUCTION INFRASTRUCTURE

Some dedicated infrastructure will be required to support the heap leach operations. During construction, an area will be designated for the main contractor use. The area will be fenced and a working pad will be built so the contractor can install their base camp, warehoused offices, and workshop. Potable water and power will be provided by the Owner.

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

19 MARKET STUDIES AND CONTRACTS

Gold is the principal commodity at the Essakane Gold Mine 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 refined 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 GOLD MINE INITIAL PERMITTING

Prior to the beginning of construction work, an ESIA was conducted by Knight Piesold Consulting and submitted to the Government of Burkina Faso 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 Essakane Gold Mine 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 digger, farmer, etc.) were performed during the field work.

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.

20.4 ESSAKANE GOLD MINE 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 the Essakane Gold Mine 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 focused 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 mill infrastructure, a new satellite pit east of the mine, 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 of Burkina Faso 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 and the Resettlement Action Plan (RAP) report will be tabled by the end of 2018. This will be the first step towards obtaining the environmental and social feasibility notice.

20.5 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 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.

For the heap leach project, a third resettlement plan will be necessary. As in the previous RAP, a consultation process will be carried out through the implementation of an Advisory Committee that will include representatives from the affected villages and hamlets.

20.6 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 mentioned earlier, the heap leach project will necessitate a third phase of resettlement plan. This RAP3 will be the continuation of the first two RAP. As a result, the former processes put in place for the communities will remain the same or will be improved.

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.

20.7 WASTE ROCK AND TAILINGS DISPOSAL

20.7.1 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 waste dumps - 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 waste rock dump 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 and acidogenic studies have demonstrated that the waste is non-acid generating and may leach some arsenic. Based on the precautionary principle, a runoff water quality monitoring program is in place. Ditches were installed around the main waste dump to collect runoff water and direct it to the ponds.

Supplemental geophysical and geochemical studies are being conducted to potentially revise the final waste deposition and closure plans.

Progressive rehabilitation of the waste dumps commenced in 2011.

20.7.2 TAILINGS DISPOSAL

The mine tailings site was designed by Golder Associates Ltd. (Golder). Tailings are thickened to recover process water and deposited in 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 80 Mt of tailings, and is expected to store up to 106.0 Mt across the 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 but may 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.

20.8 ESSAKANE PIT WASTE ROCK STABILITY ASSESSMENT

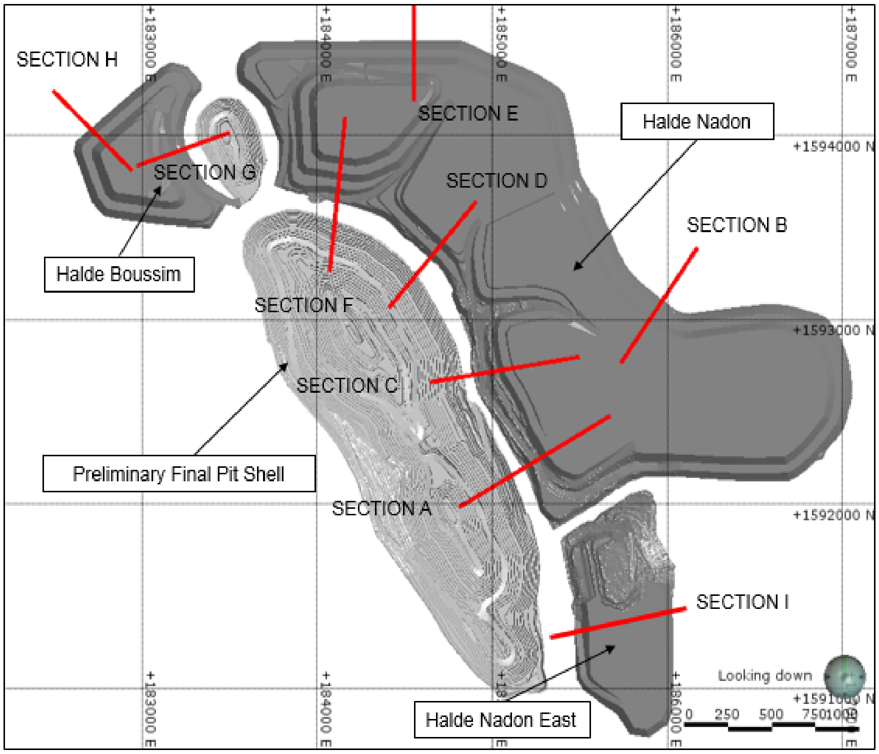
SRK carried out geotechnical and stability assessment for the proposed ultimate waste rock dump (WRD) designs (SRK, 2018). The proposed WRDs include the Halde Nadon, Halde Nadon East, and the Halde Boussim. The configurations through Section A to I are summarized in Table 20-1 and shown in Figure 20-1. 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 30°.

TABLE 20-1 SUMMARY OF PRELIMINARY FINAL WASTE ROCK DUMP DESIGN

WRD	Section	Existing Maximum Dump Height (m)	Final Maximum Dump Height (m)	Distance from Pit Crest (m)	Final Dump OSA (°)
Halde Nadon	A	45	60	100	19
	B	40	65	n/a	24
	C	40	60	135	18
	D	40	55	130	19
	E	10	85	n/a	29
	F	25	85	115	30
Halde Boussim	G	0	65	85	24
	H	0	70	N/A	30
Halde Nadon East	I	50	50	200	15

In general, the majority of the waste rock excavated from the pit will be placed into the Halde Nadon WRD that is located east and northeast of the pit. The Halde Nadon WRD will be constructed to a maximum height of 85 m and is broadly advanced toward the north, northeast, and east. The existing Halde Nadon East will extend to the south, with a final dump height less that the current WRD (50 m high). The Halde Boussim WRD is a new dump to the northwest of the pit and will be constructed to a maximum height of 70 m.

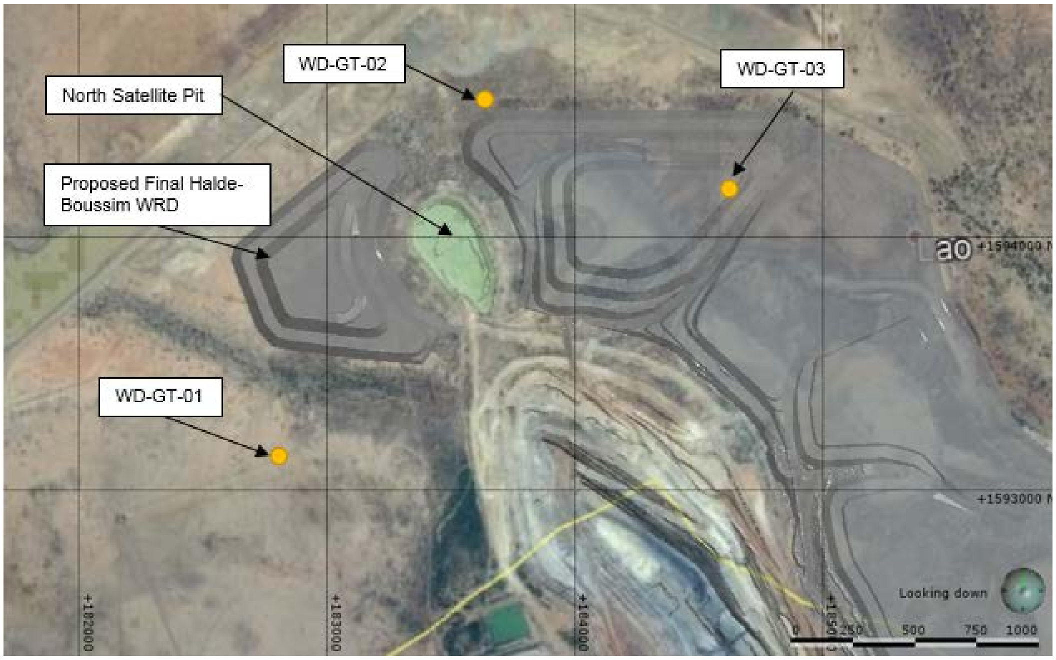
FIGURE 20-1 PROPOSED ULTIMATE DUMP DESIGNS



20.8.1 PREVIOUS SITE INVESTIGATION

Previous geotechnical site investigations have been carried out to assess subsurface conditions at the mine. This information was utilized along with new targeted site investigations in 2018 (SRK, 2018), as shown in Figure 20-2.

FIGURE 20-2 2018 WRD DRILL HOLE INVESTIGATIONS



20.8.2 FOUNDATION CONDITIONS

A summary of the soil classification testing results indicated the following:

- The particle size distribution (PSD) for the saprolite soils indicates a range of 40% to 90% passing the No. 200 sieve (0.075 mm). This indicates that the primary fraction of the saprolite is a fine-grained soil (i.e., silt and/or clay).

- The PSD testing also indicates that approximately 30% of the sample mass is retained in the medium grained sand size fraction.

- The soil plasticity testing indicates that the fine-grained portion of the tested samples is generally classified as a medium plasticity (CI) soil, with the lower bound results in the low plasticity (CL) range.

- The Pit and Borrow Area Investigation (Golder, 2008) indicates that the saprolite adjacent to the Halde Boussim is medium plasticity with results in the 40 to 50 liquid limit range.

Furthermore, the materials along the upper bench slopes along the East and West Walls, and the North Satellite Pit were inspected during the December 2017 site visit. Along the East Wall, the saprolite generally becomes thinner toward the north. The saprolite is approximately 40 m in the southern end of the pit, and less than 10 m in the northern end of the pit shows an interim bench slope within recently exposed saprolite along the Northeast Wall. The saprolite was thin (less than 10 m), fine grained sand and silty in composition, very stiff to hard consistency, and dry. Rock fabric was evident at the base of the bench slope, including the shallow, east dipping bedding relic structures.

20.8.3 WASTE ROCK DUMP PERFORMANCE

20.8.3.1 HISTORICAL WRD PERFORMANCE

Aerial photography was reviewed to understand the construction history and materials within the existing WRDs (SRK, 2018). The following general comments are made with respect to the review aerial photograph date:

- **October 2011:** the initial saprolite waste materials are placed to the east of the pit within the current Halde Nadon WRD. The materials are placed over the in-situ saprolite soils.

- **October 2014:** the Halde Nadon WRD has been advanced to the east and northeast. The previously placed saprolite waste has been covered to fresh rock, including the inside toe of the Halde Nadon along the west side of the WRD. New saprolite and/or transition rock from the northeast push-back is inferred to be placed in the northeast corner of the Halde Nadon WRD (based on the lighter material colouration). The Halde Nadon East WRD is completed to its current form, and some weaker materials appear to have been placed on the upper-most platforms.

- **December 2016:** The Halde Nadon WRD has advanced significantly to the north, northeast, and east. Newer saprolite and/or transition rock appears to have been placed toward the centre of the Halde Nadon and within the east margins of the WRD away from the outer slopes.

20.8.3.2 CURRENT WRD PERFORMANCE

During the December 2017 geotechnical pit slope site visit, the existing waste rock dumps were inspected. The WRDs were in the 30 m to 50 m height range, exhibited wide dump headings and majority of the outer slopes were constructed with hard, coarse waste rock. No significant instability issues were observed other than shallow slope face slumping along the northeast side of the Nadon Boussim WRD.

20.8.3.3 STABILITY ANALYSES

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 are summarized in Table 20-2. The results indicate that the proposed waste rock dump 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.

TABLE 20-2 SUMMARY OF WRD STABILITY ANALYSES RESULTS

WRD	Stability Section	Existing Maximum Height (m)	Final Maximum Height (m)	Distance from Pit Crest (m)	Final Dump OSA (°)	Failure Scenario	Static Conditions	
							Minimum Design FOS	Critical FOS
Halde Nadon	Section A	45	60	100	19	Shallow-seated, short term during construction	1.1	1.5
						Deep-seated foundation, long term stability	1.3-1.5	2.1
	Section B	40	65	n/a	24	Shallow-seated, short term during construction	1.1	1.5
						Deep-seated foundation, long term stability	1.3-1.5	1.9
	Section C	40	60	135	18	Shallow-seated, short term during construction	1.1	1.9
						Deep-seated foundation, long term stability	1.3-1.5	2.4
	Section D	40	55	130	19	Shallow-seated, short term during construction	1.1	1.5
						Deep-seated foundation, long term stability	1.3-1.5	2.1
	Section E	10	85	n/a	26	Shallow-seated, short term during construction	1.1	1.7
						Deep-seated foundation, long term stability	1.3-1.5	2.0
	Section F	25	85	115	26	Shallow-seated, short term during construction	1.1	1.5
						Deep-seated foundation, long term stability	1.3-1.5	2.3
Halde Boussim	Section G	New WRD	65	85	24	Shallow-seated, short term during construction	1.1	1.7
						Deep-seated foundation, long term stability	1.3-1.5	2.0
	Section H	New WRD	70	n/a	28	Shallow-seated, short term during construction	1.1	1.5
						Deep-seated foundation, long term stability	1.3-1.5	1.8

20.9 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.).

20.10 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 waste rock dumps and open pits, which may contain high suspended solids concentrations and arsenic;
- Process water: water mixed in the process plant and recovered from the tailings storage facility (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; and
- Divert clean water away from the mine site, where possible, and capture contact water.

20.10.1 WATER MANAGEMENT FACILITIES

Figure 20-3 presents proposed final water management facilities for the Essakane Mine, including the proposed HLF, Essakane Pit, the tailings management area, and the waste rock dumps. A combination of channels, and berms strategically capture and divert contact water to control ponds via pumping systems. The pumping strategy was designed based on high rainfall events recorded on site to effectively manage inflows during the wet season.

FIGURE 20-3 ESSAKANE WATER MANAGEMENT PLAN

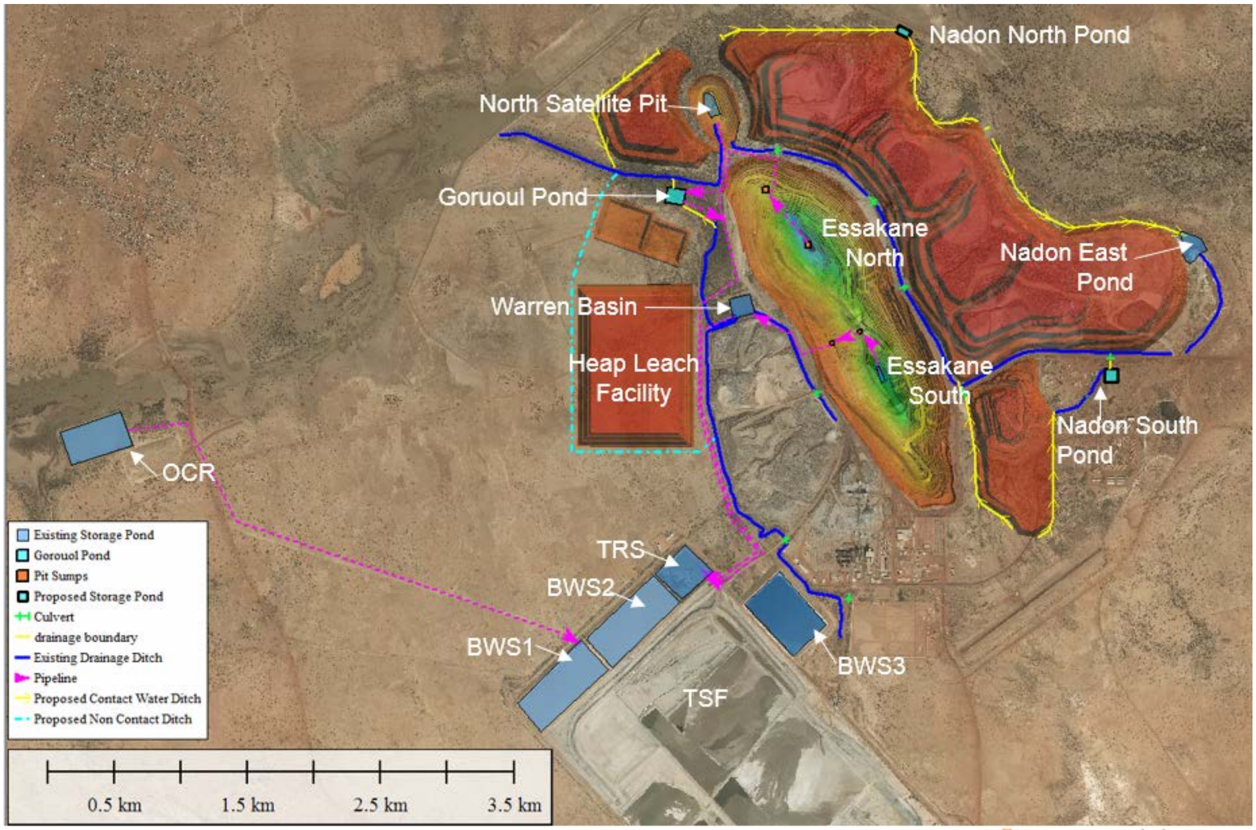


Table 20-3 presents a summary of water management infrastructure, including the overall management strategy and structure description. Grey shading is used to indicate infrastructure that has been proposed as part of the latest site water management review. All other infrastructure is already in place.

TABLE 20-3 SUMMARY OF SURFACE WATER MANAGEMENT FACILITIES

Facility	Description	Water Management Strategy
Warren Basin	An existing, unlined water storage facility, west of Essakane Pit	Contains pumped flows from pit sumps and runoff from western waste rock dumps. Dewatered to BWS Ponds. Overflow into channel currently discharging into North Satellite Pit. Channel to be diverted into Gorouol Pond once it is constructed.
North Satellite Pit	Previously mined pit, north of Essakane Pit. Situated adjacent to historic Gorouol River drainage.	Existing water storage facility for runoff around northern Nadon waste rock dump. Dewatered via direct pipeline to BWS Ponds. To be mined to final footprint in dry season.
Essakane Pit (North)	Northern extent of Essakane Pit	Dewatered via transfer stations to the Gorouol Pond.
Essakane Pit (South)	Southern extent of Essakane Pit	Dewatered via transfer stations to the Warren Basin.
Nadon North Pond	Storage pond near historic Gorouol drainage, northeast of Nadon North dump	Collects waste rock runoff. Infiltration and evaporation are primary outflows. Can be used for dust suppression.
Nadon East Pond	Storage pond east of Nadon North dump	Collects waste rock runoff. Infiltration and evaporation are primary outflows. Can be used for dust suppression.
Nadon South Pond	Storage pond south of Nadon Dump and east of Nadon East Dump	Collects waste rock runoff. Infiltration and evaporation are primary outflows. Can be used for dust suppression.
BWS Ponds	Lined water storage ponds (3) north and east of TSF	Receive pumped inflows from Warren Basin, North Satellite Pit, and freshwater from OCR Pond Water supply source for process plant.
Tailings Reclaim Sump (TRS) Pond	Unlined water storage pond north of TSF	Retains supernatant water from the TSF pond.
OCR Pond	Freshwater storage pond adjacent to Gorouol River	Captures and stores water from the Gorouol River during the wet season. Pumped to BWS ponds through dry months.
Gorouol Pond	Storage pond adjacent to historic Gorouol River drainage, south of Boussim Dump and north of heap leach	Collects runoff from waste rock runoff and freshwater runoff from perimeter of heap leach area. Contains overflow from Warren Basin. Dewatered to BWS ponds.
TSF	Lined storage cells for thickened tailings	Thickeners and dewatering pumps reclaim water to process plant.

20.10.2 WATER SUPPLY

To supply the mining camp with potable water, three wells were drilled outside the site, immediately 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. The site is currently studying a new potable water treatment system that will enable Gorouol River water to be used for potable water; this will reduce the dependence on groundwater sources which have shown signs of declining yield.

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 river bed by 1.5 m creating an off channel reservoir (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.

20.10.3 WATER MONITORING

A water quality monitoring program (surface water, groundwater, industrial water, potable water, and domestic waste water) 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).

20.11 MINE CLOSURE REQUIREMENTS AND COSTS

A conceptual rehabilitation and closure plan (PRF) was developed in 2009 and updated in 2013. Closure costs are updated annually, or 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.

An updated version of the closure plan is under development and will be available in December 2018.

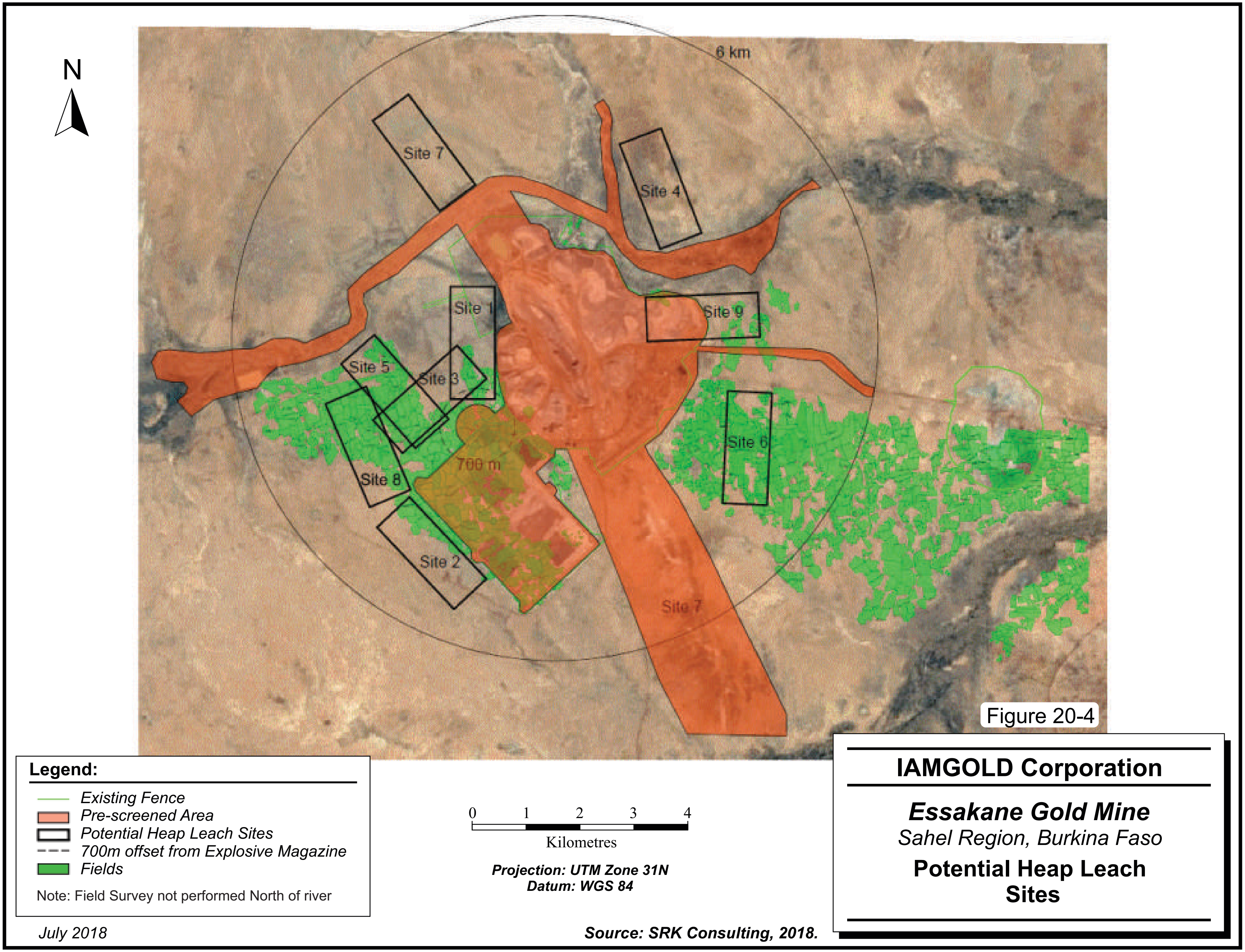
A closure plan PFS (Closure PFS) will be conducted three years prior to mine closure. This step will involve a complete review of the plan to validate the base information and to verify the status of progressive rehabilitation. At that time, consultations with stakeholders will be organized in order 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 two years prior to the closure of the mine 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.

20.12 HEAP LEACH FACILITY

20.12.1 SITING STUDY

An HLF siting analysis was performed by SRK using a multiple criteria analysis based on the Canadian Guidelines for the Assessment of Alternatives for Mine Waste Disposal. Nine potential sites of 100 m by 800 m heap leach were identified, as shown in Figure 20-4.


N
6 km
Site 7
Site 4
Site 1
Site 9
Site 5
Site 3
Site 6
Site 8
700 m
Site 2
Site 7
Figure 20-4
Legend:
Existing Fence
Pre-screened Area
Potential Heap Leach Sites
700m offset from Explosive Magazine
Fields
Note: Field Survey not performed North of river
0 1 2 3 4
Kilometres
Projection: UTM Zone 31N
Datum: WGS 84
IAMGOLD Corporation
Essakane Gold Mine
Sahel Region, Burkina Faso
Potential Heap Leach Sites
July 2018
Source: SRK Consulting, 2018.

These sites were identified based on the following criteria:

- Sites must be located within 6 km of the centre of the Essakane pit.
- Sites must not require the diversion of large water courses.
- Sites must not sterilize the potential resource.
- Sites must be located more than 400 m from the explosives magazine.
- Sites must not interfere with existing site infrastructure such as the tailings storage facility, waste rock storage areas, or the main site access road. Site 9 is an exception to this criterion, as it was sited on a portion of a waste rock dump to reduce footprint and earthworks quantities.

The nine potential heap leach sites were evaluated based on three accounts, and multiple sub-accounts as described in Table 20-4. The sites were evaluated assuming a 1,761 m by 821 m heap leach pad, sloping towards the process ponds at a slope of 1%. While both the heap leach liner slope and size of the HLF have changed since the alternatives analysis was performed, these changes have no material effect on the results of the analysis.

TABLE 20-4 POTENTIAL HEAP LEACH SITE EVALUATION

Account	Sub-Account	Rationale
Technical	Operations Infrastructure	Heap leach sites in close proximity to the pit and low grade stockpiles require less linear infrastructure (conveyors, pipelines) and the linear infrastructure requires less power during operations.
	Topography: Earthworks Required	The heap leach pad has a target grade of 1-3% towards the ponds. The local landscape generally has low constant relief. Sites that utilize the existing topographical slope and minimize earthworks are preferred.
	Foundation Conditions	Poor foundation conditions present engineering and construction challenges.
	Expandability	Facilities with the ability to be expanded in the future are preferred.
	Interference with existing infrastructure	Heap leach sites which would have no restrictions due to the existing airport and explosives magazines are preferred.
	Surface Water Management	Sites with small upstream catchment areas are preferred.
	Fencing	Sites with shorter lengths of new fence are preferred.
Social	Intercepted agricultural lands	Locations that intercept the least amount of agricultural land are most desirable.
	Village Relocation	Sites that do not require the relocation of villages are most desirable.
	Artisanal gold miners	Sites that do not interfere with artisanal gold miners are most desirable.
	Impacted regional roads	Sites that do not require relocation of regional roads are most desirable.

Account	Sub-Account	Rationale
Environmental	Mine Footprint Increase	All sites will increase the footprint associated with the mine. Sites that have the smallest increase in the mine area are most desirable.
	River Crossing	Heap leach pads located across the river from mine infrastructure pose an increased environmental risk as ore and leach solution transported over the river pose a risk of spills.
	Proximity to the river	Sites in close proximity to the river pose an increased environmental risk.

The multiple account analysis determined that Site 1 was the preferred alternative and that Site 2 is the second highest ranked alternative. Details of this analysis are provided in the SRK 2018 study. Site 1 was carried forward in the PFS.

20.12.2 HEAP LEACH FOUNDATION

20.12.2.1 PREVIOUS SITE INVESTIGATIONS

Data from geotechnical investigations previously conducted to assess subsurface conditions at the mine were used in the SRK 2018 study, including

- Geotechnical Investigation of Process Plant Area (Golder, 2008);
- Geotechnical Investigation of the Essakane Pit and Borrow Area (Golder, 2010a);
- Geotechnical Investigation of the Tailings Storage and Water Retention (Golder, 2010b); and
- Geotechnical Testing of Essakane Tailings and Saprolite for TSF Basin Lining (Golder 2012).

Following the review of the previous investigation data, five drill holes and seven test pits were completed in April 2018 to determine depth to rock and subsurface conditions within the proposed heap leach pad footprint. The locations of the drill holes, test pits, and depth to bedrock are shown in Figure 20-5. A geotechnical engineer from the SRK Ghana office supervised the drilling and geotechnical logging.

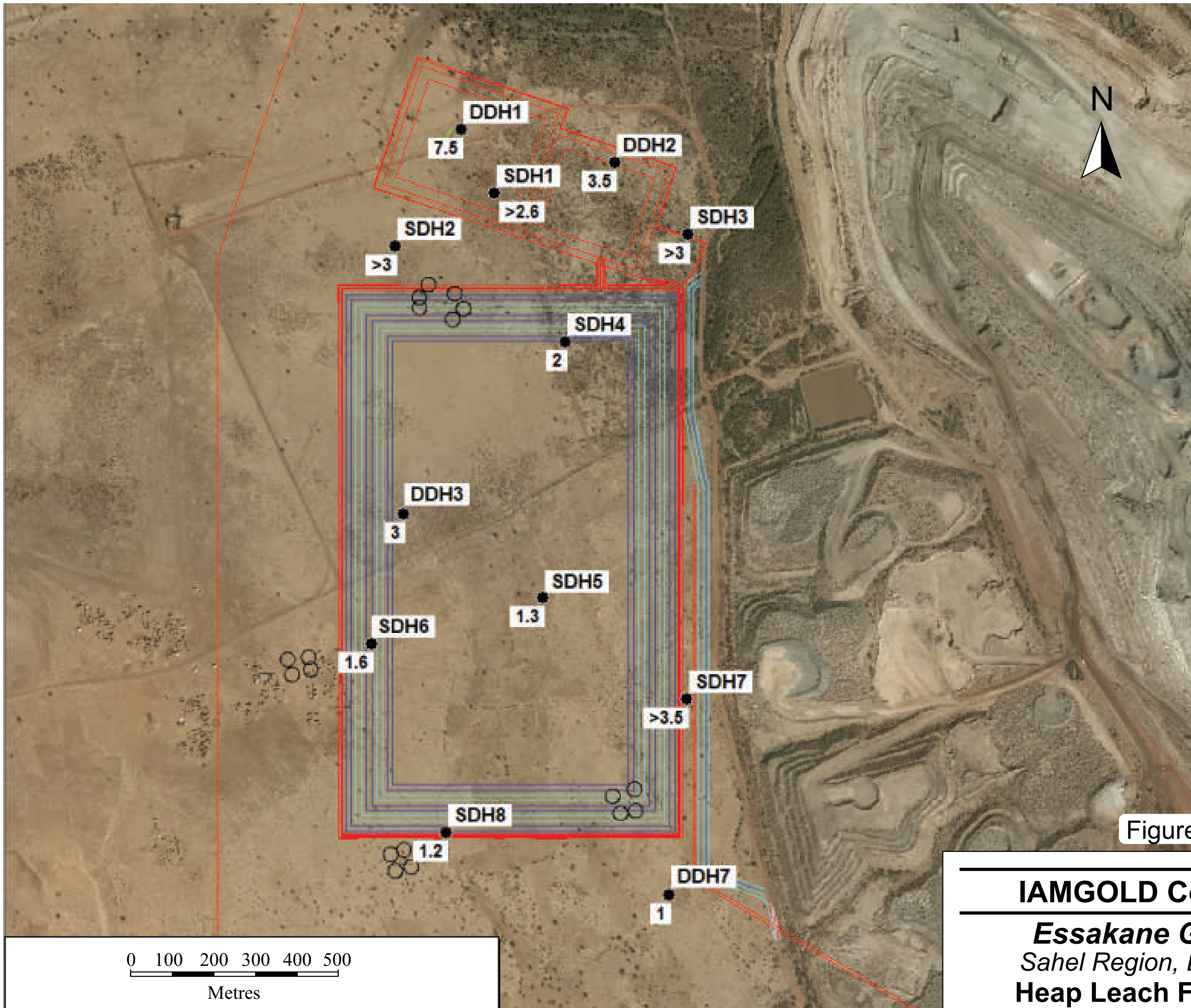


Figure 20-5

IAMGOLD Corporation

Essakane Gold Mine

Sahel Region, Burkina Faso

Heap Leach Facility 2018 Site Investigation Depth to Bedrock Results (mbgs)

July 2018

Source: SRK Consulting, 2018.

IAMGOLD Corporation

Geotechnical investigations within the footprint of the proposed HLF indicate that:

- Depth to bedrock under the proposed heap leach pad is relatively shallow, ranging from 1.2 mbgs in the southern area of the pad to 3 mbgs in the northwestern area of the pad.
- Bedrock depth under the proposed solution ponds were deeper with a maximum measured bedrock depth of 7.5 mbgs.
- Foundation soil typically consists of saprolite, which, based on particle size distribution testing, is predominately sand or silt.
- Thin layers of aeolian sands exist above the saprolite in some areas.

The thin layers of aeolian sand are assumed to be prone to collapse settlement, therefore SRK recommended that these sands be removed prior to HLF construction.

20.12.2.2 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 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 20-5. The results 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.

TABLE 20-5 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

Section	Case	Failure Mode	Factor of Safety[1] (Static)
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.

21 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COSTS

The capital cost requirement over the LOM includes the following:

- Heap leach project capital expenditures.
- Resource development costs.
- Capitalized waste stripping.
- Sustaining capital expenditures (for mill and site in general).
- Mine equipment additions and replacements.
- Equipment overhaul costs.
- Equipment capital spares (CSPARES).
- Tailing dam capital expenditures.

21.1.1 GENERAL

A total of $894.3M of capital is planned to be spent over the remaining LOM, which equates to $5.46/t milled (CIL + HL) or $221/oz of gold sold. Figure 21-1 shows the total LOM capital by year.

Capitalized waste stripping (cash portion) is the largest capital cost estimated at $368.3M, representing 41% of the LOM remaining capital expenditures. Figure 21-2 shows the distribution of the sustaining capital over the LOM.

FIGURE 21-1 LOM CAPITAL EXPENDITURES

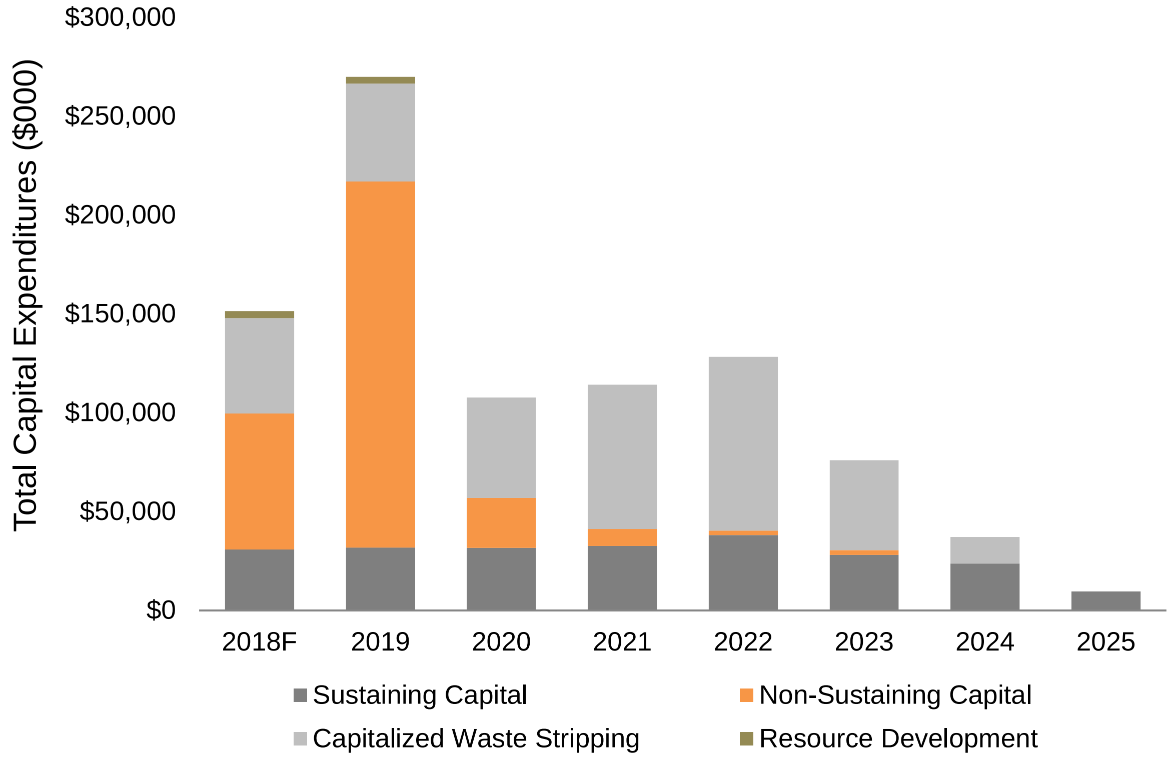
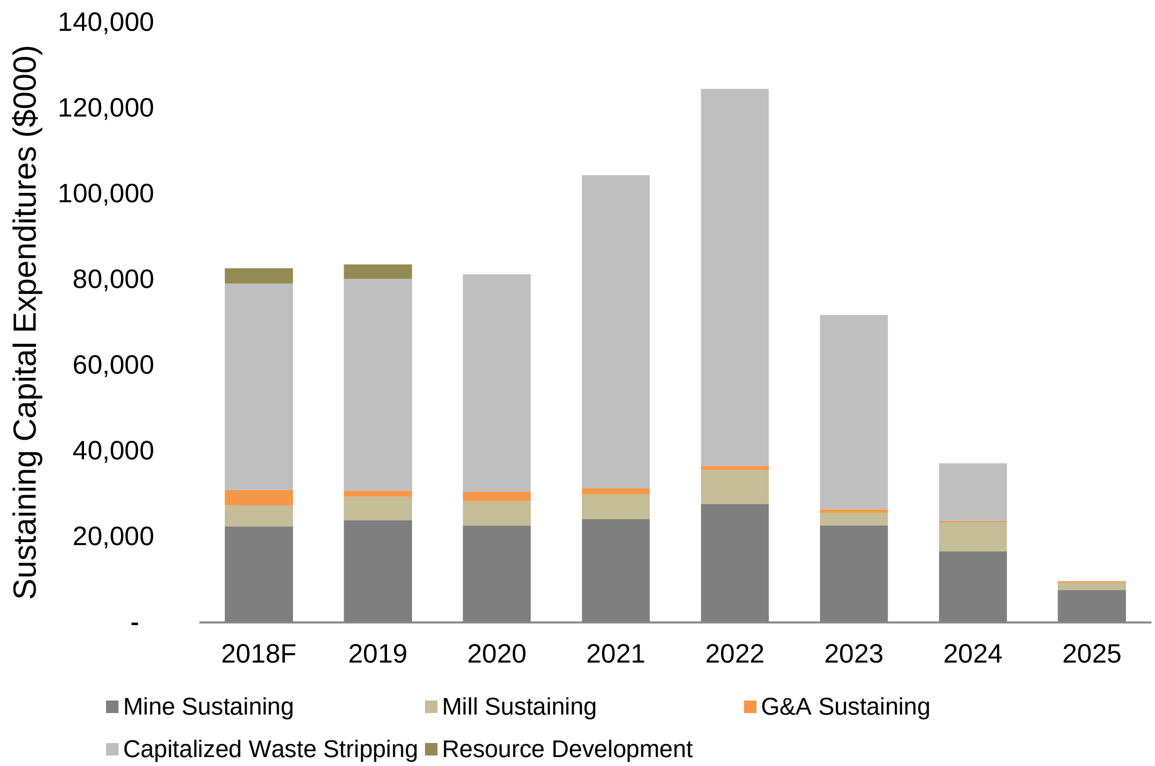


FIGURE 21-2 LOM SUSTAINING CAPITAL EXPENDITURES



21.1.2 HEAP LEACH CAPITAL COST ESTIMATE

21.1.2.1 INTRODUCTION

The Project capital cost estimate includes all direct and indirect costs. The capital cost is deemed to cover the future FS and the Implementation phase. The implementation phase includes period starting from the project approval date and finishing at commissioning activities. Start-up and ramping-up to full production are considered to be part of operating costs.

This 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, and a heap leach pad and ponds.

The Essakane heap leach project is developed in one construction major phase for most part of the scope and two additional sustaining capital expenditure phases:

- Phase I – Capital Cost (Total Project Scope including first phase of the heap leach pad)

- Phase II – Sustaining Capital (Heap leach pad extension only)
- Phase III – Sustaining Capital (Heap leach pad extension only)

21.1.2.2 ASSUMPTIONS

The following items present the assumptions that have been taken during the study estimate:

- Estimate is based on 6 days at 10 hours workweek for construction contractor;
- Estimate is based on 6 weeks in/2 weeks out rotation schedule for construction contractor;
- Estimate assumes that labour skills will range from medium to high, i.e., no unskilled nor low skill labour;
- It is assumed that origin of skilled workers will be from west African countries;
- 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 21-1 shows currency exchange rates and portion considered in this capital cost estimate are as follow:

TABLE 21-1 CURRENCY EXCHANGE RATE

Currency Code	Currency Name	May 19, 2018 (IAMGOLD April Publication)
USD	US Dollar	1.00000
EUR	Euro	1.20000
XOF (CFA)	CFA Franc BCEAO	0.00183
CAD	Canadian Dollar	0.80000

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 but 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

Tables 21-2 and 21-3 provide a summary of the project capital costs by major area and by work breakdown structure (WBS), respectively.

TABLE 21-2 CAPITAL COST BY MAJOR AREA

Major Area	Capital Expenditures	TOTAL ($000)
100	Infrastructure	9,357
300	Mill General (Material Processing including Heap Leach)	86,795
400	Plants & Equipment	1,815
700	Mining (Covered under Operating Cost)	Operating Cost
900	Project Indirect Costs (Excluding Contingency)	34,824
998	Contingency	19,918
Grand Total		**152,708**

TABLE 21-3 CAPITAL COSTS BY WBS

WBS	WBS Description	Capital Cost Initial Year ($000)	Sustaining Capital Year 3 ($000)	Sustaining Capital Year 5 ($000)
	Direct Costs	**97,966**	**5,152**	**3,070**
100	Infrastructure – General	464	-	-
103	Mine Village (Camp)	858	-	-
104	Site Roads	368	-	-
107	Village Relocation	Incl. in 919	-	-
109	LFO Storage	1,241	-	-
110	HFO Storage	4,774	-	-
114	Medical & Training Facilities	Other	-	-
116	Site Power Reticulation	1,151	-	-
123	Bulk Water Storage Reservoir (BWS)	502	-	-
301	Ore Receipt – Crushing / Stockpiling	37,426	-	-
305	CIL	347	-	-
307	Acid Wash and Elution	927	-	-
308	Carbon Regeneration & Recovery	194	-	-
310	Electrowinning and Gold Room	576	-	-
317	Water Services – Process Water	512	-	-
318	Water Services – Potable Water	Other	-	-
360	Heap Leach Process Area – General	2,536	-	-
361	Heap Leach – Conveying & Stacking	19,372	-	-
362	Heap Leach – Reagents	234	-	-
364	Heap Leach – Pad & Ponds	19,749	5,152	3,070
365	Heap Leach – CIC Plant	4,923	-	-
406	Support Equipment	1,815	-	-
	Indirect Costs	**54,742**	**1,288**	**767**
910	Construction Equipment & Tools	1,615	-	-
919	Other Construction Costs (Owner's Costs)	5,810	1,288	767
920	Construction Engineering	5,200	-	-
925	Construction Management	6,978	-	-
930	Construction Freight and tax	5,000	-	-
935	Construction Room & Board	684	-	-
940	Construction Room & Board	1,848	-	-
945	Construction Transportation	1,674	-	-
950	Initial Fills	200	-	-
955	Capital Spares	750	-	-
956	1 – 2 Years Operational Spares	Incl. in operating cost	-	-
957	Commissioning Spares	195	-	-
964	Construction Services	3,027	-	-
985	Corporate Administration	1,842	-	-
998	CONTINGENCY	19,918		
	Grand Total	**152,708**	**6,440**	**3,837**

21.1.2.6 MOBILE EQUIPMENT

Most of the major mining and auxiliary equipment are already in operation and only a minor amount of additional equipment is required for the heap leach project. Table 21-4 provides a list of small additions to that equipment fleet required for the heap leach project. The value of this additional equipment is included in WBS 406.

TABLE 21-4 INITIAL CAPITAL FOR MINING AND SERVICE EQUIPMENT

Equipment Description	QTY	Total ($000)
Major Equipment		
Loader CAT 950	1	335
Dozer CAT D9	1	1,320
Truck (Tractor) 5T (HL Carbon Transport)	1	60
Farm Tractor (For Drip Line Installation)	1	100
Grand-Total	**4**	**1,815**

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 15%, which is in line with IAMGOLD's guideline for a PFS study.

21.1.2.8 HEAP LEACH SUSTAINING CAPITAL COSTS

Sustaining capital expenditures are estimated to be $10.27 M. The sustaining costs are required to enlarge the heap leach pad and ponds. Any additional piping distribution is considered part of the operating cost. A general allowance of 25% has been used for sustaining scope indirect costs.

21.1.2.9 HEAP LEACH RECLAMATION AND CLOSURE COSTS

The reclamation and closure costs of the heap leach project has been integrated in the Essakane closure cost and included in the LOM. A more detailed cost will be provided in the FS.

21.1.3 CLOSURE AND RECLAMATION COSTS

A provision of $76.5M is estimated for the closure and reclamation cost of Essakane. This amount is planned to be re-evaluated in 2018 to reflect the updated projected disturbance until the end of the mine.

21.2 OPERATING COSTS

The mine 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 (2018 to 2022) are shown in Table 21-5.

TABLE 21-5 LOM AND FIVE YEAR PLAN OPERATING COSTS

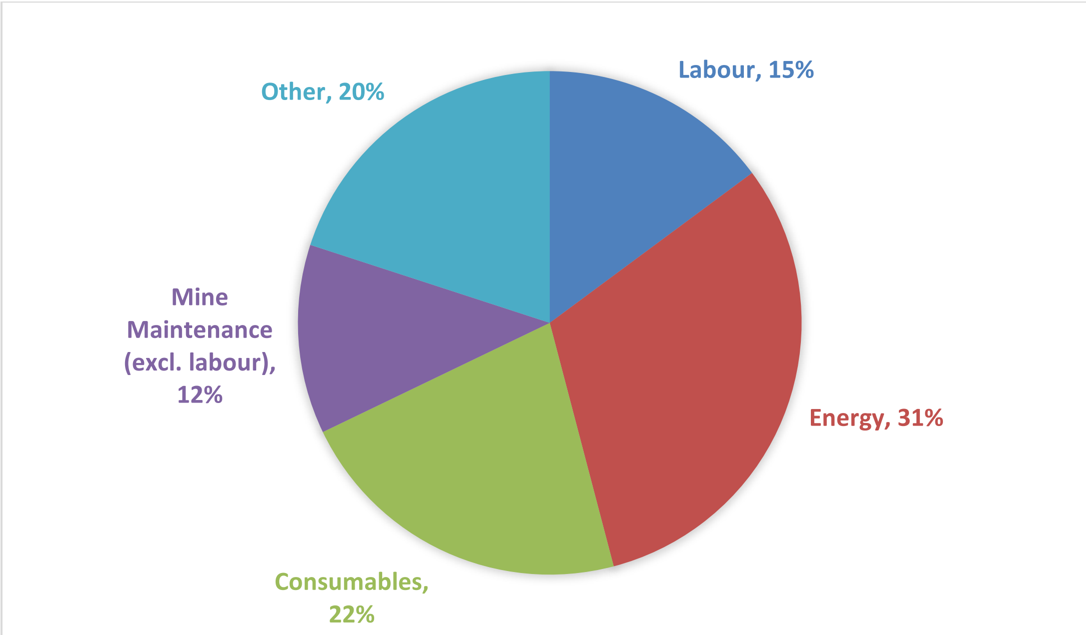
Area	LOM Average	Five Year Plan (2018-2022)
Mining (US$/t mined)	2.76	2.64
CIL Processing (US$/t milled)	12.00	11.86
Heap Leach Processing (US$/t milled)	3.13	3.13
G&A (US$/t milled)	3.73	3.85

The average total cash cost per ounce is US$707/oz Au while the all-in sustaining cost (AISC) averages US$946/oz Au over the LOM.

21.2.1 MINE OPERATING COSTS

Average mine operating costs are estimated at $2.76/t mined over the LOM (2018 included), averaging $2.64/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 as all mining activities occur at greater depth. Fuel represents $0.86/t mined and 0.78 L/t mined over the LOM, which represents 31% of the mine operating cost. The top four mining cost categories represent 80% of the mine operating costs (Figure 21-3).

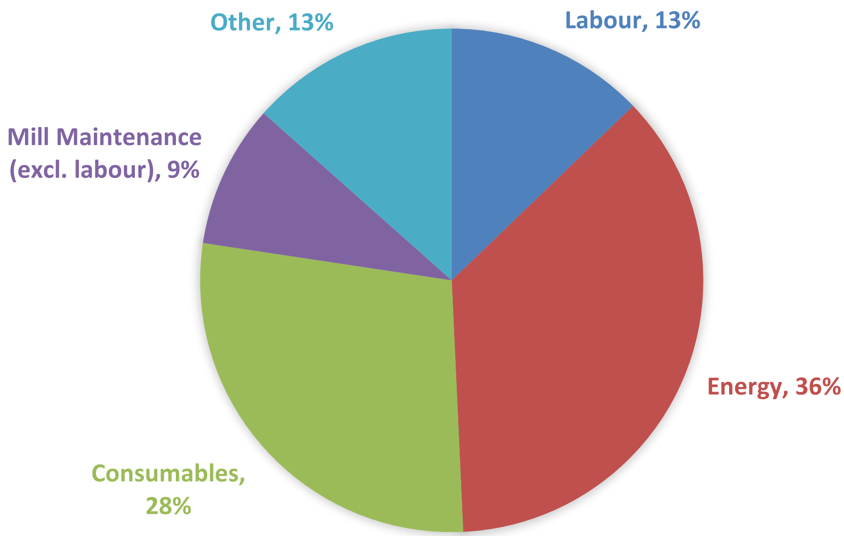
FIGURE 21-3 MINING COST CATEGORIES



21.2.2 CIL OPERATING COSTS

The average LOM milling cost is estimated at \$12.00/t milled and an average of \$11.86/t milled over the next five years. Fuel related to power generation is the primary cost and represents 30% of the CIL operating cost. The energy cost (power-HFO and solar) is \$4.37/t milled at the CIL. The total energy cost is 36% of the CIL overall costs. It is followed by cyanide (9.0%, \$1.07/t milled) and grinding media (8.9%, \$1.07/t milled). The milling cost categories are presented in Figure 21-4.

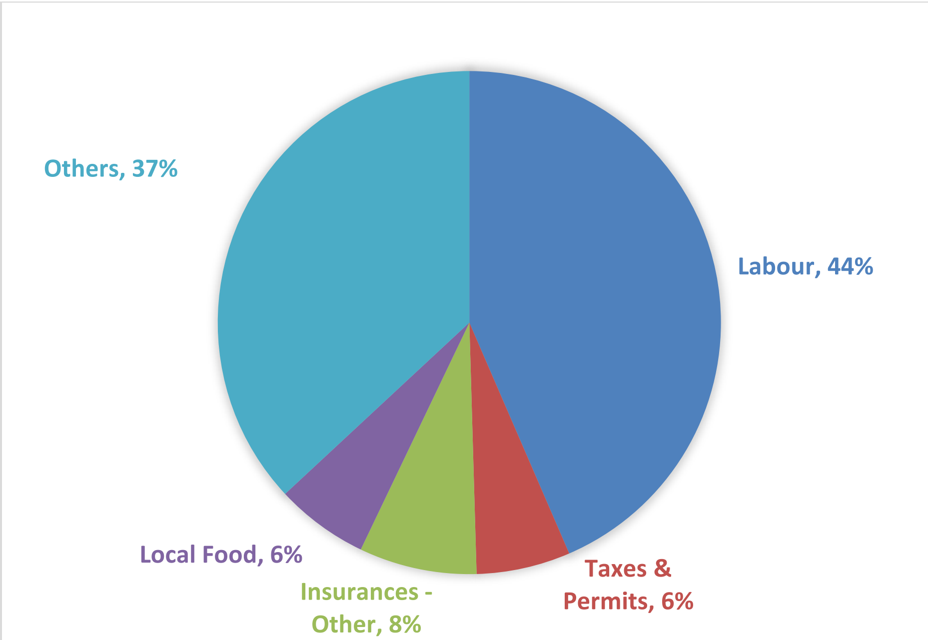
FIGURE 21-4 MILLING COST CATEGORIES



21.2.3 G&A COSTS

The average LOM G&A cost is $3.73/t milled and assumes processing at a rate above 12 Mtpa at the CIL plant until 2022. A high proportion of the G&A costs are fixed costs (i.e.: taxes) and are difficult to compress should the milling rate decrease. The top four G&A costs over the LOM represent 63% of our G&A costs (Figure 21-5) and are detailed as following: labour (44%), insurance (8%), taxes and permits (6%), and local food (6%)

FIGURE 21-5 G&A COST CATEGORIES



21.2.4 HEAP LEACH OPERATING COST

Operating costs for the heap leach expansion Project have been estimated by KCA with input from IAMGOLD.

Operating costs presented are based upon ownership of all process production equipment and site facilities, as well as the Owner employing and directing all operating, maintenance, and support personnel.

The estimated heap leach annual operating costs are based upon information presented in earlier sections of this report. LOM operating costs for heap leach processing are estimated to be US$3.13/t material processed and are presented in the following sections.

Heap leach operating costs by area are summarized in Table 21-6.

TABLE 21-6 HEAP LEACH AVERAGE ANNUAL OPERATING COSTS

Area	Costs, US$	US$/t
Area 360 - Heap Leach Process Area General	1,195,000	0.120
Area 301 - Primary Crushing	1,861,000	0.186
Area 301 - Secondary Crushing	3,208,000	0.321
Area 301 - Secondary Stockpile and Reclaim	276,000	0.028
Area 301 - Tertiary Crushing	4,321,000	0.432
Area 361 - Conveying and Heap Stacking System	3,221,000	0.322
Area 364 - Heap Leach Pad and Ponds	1,247,000	0.125
Area 365 - Heap Leach CIC Plant	1,352,000	0.135
Area 362 - Heap Leach Reagents	14,067,000	1.407
Area 317 - Water Supply, Storage & Distribution	282,000	0.028
Mobile Equipment	287,000	0.029
TOTAL COST	**31,317,000**	**3.132**

Operating costs for the heap leach have been estimated from actual costs at the existing operation, the Project's flowsheet, and first principles. Labour costs are estimated using project specific staffing, salary, wage, and benefit requirements. Unit consumption of materials, supplies, power, water, and delivered supply costs are also estimated based on vendor quotes and quoted equipment requirements.

All operating costs are presented in Q2 2018 US Dollars. Where prices were supplied in CFA Franc, an average conversion of 546.63 CFA Franc per US Dollar was used for all process operating costs. These costs do not include any Value Added Tax (VAT).

21.2.4.1 PROCESS LABOUR AND WAGES

Staffing requirements for the process and support personnel have been estimated by KCA with input from IAMGOLD. Wage, salary, and burden information for personnel was provided by IAMGOLD and has been included in the wage and salary data.

Staffing will be primarily by Burkina Faso nationals, with supply from the local labour force as a priority. The work force for the process and support will consist of approximately 115 persons including 103 persons in the plant areas and 12 persons in the laboratory.

21.2.4.2 CONSUMABLE ITEMS

Process reagent and consumables costs have been estimated based upon unit costs and consumptions. Reagent consumptions were developed from test work performed on samples of Essakane ore, as detailed in Section 13, and from process calculations. Reagent unit costs were based on actual costs at the operation or recent supplier quotes. Freight costs are included in the unit prices. Table 21-7 shows the consumption of the major consumables.

TABLE 21-7 HEAP LEACH REAGENT CONSUMPTION

Reagent	Form	Annual Consumption, t
NaCN	Briquettes - 1 tonne supersacks	3,300
Quicklime	Bulk Delivery Truck	13,000
Anti-scalant	Liquid Tote, 1m^3	125

Operating costs for these items have been distributed based on tonnage and gold production, as appropriate.

21.2.4.3 GENERAL AND ADMINISTRATIVE

The G&A cost is not included in the heap leach processing costs.

21.2.5 OVERALL OPERATING COST METRICS

Over the total LOM, 2018 to 2026, the average total cash cost per ounce of gold is $707/oz while the all-in sustaining cost (AISC) averages $946/oz of gold over the LOM.

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.

22 ECONOMIC ANALYSIS

This section is not required as the Essakane Gold Mine is currently in production and there is no material expansion of current production.

23 ADJACENT PROPERTIES

Eurasian Natural Resources Corporation PLC (ENRC, formerly Central African Mining and Exploration Company (CAMEC), Burkina Sarl, and Etablissements Sawadogo Mahamadi et Freres (ESMAF) own the Yedebere, Yevelde, and Falagountou III exploration licences located west and east of the Essakane Exploration Permits. There is no relevant information from these adjacent properties available for disclosure in this report.

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 Essakane heap leach project execution proposed to 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 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 major Project milestones from the PFS are presented in Table 24-1.

TABLE 24-1 MAJOR PROJECT MILESTONES

Description	Start Date	Completion Date
Feasibility study engineering - earthworks	Q3 - 2018	Q1 - 2019
Feasibility study engineering - power	Q3 - 2018	Q1 - 2019
Feasibility study engineering - heap leach	Q3 - 2018	Q1 - 2019
Feasibility study completed	Q1 - 2019	
Detailed engineering	Q1 - 2019	Q3 - 2019
Permitting expected date	Q1- 2019	
Long lead item procurement	Q1 - 2019	Q4 - 2019
Early works	Q1- 2019	Q2-2019
Project full approval	Q2 - 2019	
Construction	Q2 - 2019	Q2 - 2020
Heap leach production	Q2 - 2020	

24.2 RISK MANAGEMENT

A risk identification and mitigation session was organized during the PFS. This section presents the main risks and opportunities.

The main risks associated with the Project are shown in Table 24-2.

TABLE 24-2 MAIN PROJECT RISKS

Risk	Risk Response/Mitigation
Difficulty to deliver CIL and Heap Leach ore tonnage	1) Evaluate mobile equipment fleet considering requirements 2) Initiate heap leach material stockpiling 3) Management focus on CIL and HL targets
Delay in permitting due to communities agreement refusal	1) Create negotiation committee 2) Use recent Essakane and communities negotiations successes (win/win recitations)
Gold recovery at HL lower than expected	1) Include agglomeration and edge recirculation 2) Scale up factor applied from the lab to the selected financial evaluation - conservative approach
Delay in receiving approval/permitting from the government	1) Follow up 2) Design with best practices 3) Continuous communication with government throughout the process
Environmental impact	1) Follow best practices in design 2) Site investigation and lab testing during design phase 3) Maintain best operating practices

The opportunities which may improve the Project are shown in Table 24-3.

TABLE 24-3 MAIN PROJECT OPPORTUNITIES

Opportunity	Opportunity Response
Low capital cost option: maximize the ounces production while reducing the investment.	1) Gravimetric circuit survey 2) Maximize plant throughput 3) Detox plant PFS 4) Study use of surfactants
Improve HL recovery	1) Evaluate agglomeration 2) Use higher Cn concentration at the end of heap life to defeat preg robbing.
Optimize the heap leach project : Value engineering	1) Confirm crushing plant location to minimize conveyor length. 2) Optimize basin design 3) Minimize project footprint 4) Optimize the CIC plant design

25 INTERPRETATION AND CONCLUSIONS

IAMGOLD has the following conclusions and observations:

25.1 GEOLOGY AND MINERAL RESOURCES

- Mineral Resources and Mineral Reserves 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.

- GMSI reviewed the information stored in the Falagountou database and found it to be in good standing.

- Drill hole spacing on the Falagountou East and West deposits is 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 ID^3 based Mineral Resource estimate for the Falagountou East and West deposits was found to be a good representation of the drill hole composites.

- Swath plots for Indicated and Inferred Mineral Resources by vertical sections for the Falagountou East and West deposits 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.

- 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.

- 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 more than sufficient to support a PFS.

25.2 MINING AND MINERAL RESERVES

- The mine design and Mineral Reserve estimate have been completed to a level appropriate for a PFS.

- 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.

25.3 METALLURGICAL TESTING AND MINERAL PROCESSING

KCA has the following conclusions and observations:

- The metallurgical testing results indicate that the Essakane low grade material is amenable to processing by conventional heap leaching methods. Gold recovery is estimated to be 55% and reagent requirements are low.

- Detailed operating costs have been estimated based on experience and actual costs at site and are appropriate for a PFS.

- 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 Turbidite material.
    - 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:
    - Most of the column leach tests were still leaching when the tests were finished and additional recovery is likely as ore is secondarily leached through upper lifts.
    - The HPGR model selected for this study was single pass. A larger machine would allow an amount of recirculation which would result in a finer product size and potentially higher gold recovery. Test work is underway at the time of this report.
    - The design for this study does not include cement agglomeration of the ore. Utilization of cement may increase maximum heap height or permeability requiring less liner for lower capital costs and possibly increasing gold recovery. Due to the high cost of lime cement would only be a low added operating cost over lime. Test work is underway at the time of this report.
    - The overall design of the crushing and stacking systems for the heap leach presented in this study is a first-pass design. The opportunity exists to optimize the general layout and individual components.

25.4 ENVIRONMENT

- No outstanding technical issues were identified for environment and permitting.

26 RECOMMENDATIONS

IAMGOLD has the following recommendations:

26.1 GEOLOGY AND MINERAL RESOURCES

- The West flank of the lithological model of the EMZ deposits should be updated for the next resource estimate in order to reflect new geological observations.

- 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.

- Estimation strategy used for EMZ could result in too much smoothing, however reconciliation did not indicate too much smoothing in the last year. Considering a lower cut-off grade for the heap leach project, it is in the opinion of the QP that a different strategy should be investigated using the grade control results in the upcoming year. In addition to a calibration with the production, the QP suggests having an external audit to assist parameter selection.

- 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.

- GMSI suggests waiting for robust reconciliation data before making any important modifications to the Falagountou deposit block model.

- GMSI is of the opinion that the ID^3 interpolation method for the Falagountou deposit is a better global estimator compared to the OK technique.

26.2 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 mill operating parameters as a function of graphitic carbon concentration in the feed.

- KCA has the following recommendations:
    - Column leach tests should be conducted on the Turbidite rock type to confirm recovery.

- Metallurgical testing should be conducted on stockpile material to check if weathering has any effect on recoveries.

- A feasibility study is recommended to improve the reliability and accuracy of the cost estimate and form the basis for a construction decision.

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28 DATE AND SIGNATURE PAGE

This report titled "Technical Report on the Essakane Gold Mine Heap Leach Pre-Feasibility Study, Sahel Region, Burkina Faso" effective June 5, 2018 and dated July 19, 2018 was prepared and signed by the following authors:

	(Signed & Sealed) "*Vincent Blanchet*"
Dated at Longueuil, QC July 19, 2018	Vincent Blanchet, ing. Geological Engineer IAMGOLD Corporation
	(Signed & Sealed) "*Philippe Chabot*"
Dated at Longueuil, QC July 19, 2018	Philippe Chabot, ing. Mine Optimization Expert IAMGOLD Corporation
	(Signed & Sealed) "*Stéphane Rivard*"
Dated at Longueuil, QC July 19, 2018	Stéphane Rivard, ing. Director Metallurgy IAMGOLD Corporation
	(Signed & Sealed) "*Denis Isabel*"
Dated at Ouagadougou, Burkina Faso July 19, 2018	Denis Isabel, ing. Director Health Safety and Sustainability IAMGOLD Corporation
	(Signed & Sealed) "*Luc-Bernard Denoncourt*"
Dated at Longueuil, QC July 19, 2018	Luc-Bernard Denoncourt, ing. Projects Manager IAMGOLD Corporation
	(Signed & Sealed) "*Travis J. Manning*"
Dated at Reno, NV July 19, 2018	Travis J. Manning, P.E. Senior Engineer Kappes, Cassiday & Associates
	(Signed & Sealed) "*Edward Saunders*"
Dated at Vancouver, BC July 19, 2018	Edward Saunders, P.Eng. Senior Consultant (Rock Mechanics) SRK Consulting

(Signed & Sealed) "*Cam Scott*"

Dated at Vancouver, BC
July 19, 2018

Cam Scott, P.Eng.
Principal Consultant (Geotechnical Engineering)
SRK Consulting

(Signed & Sealed) "*Edith Bouchard-Marchand*"

Dated at Québec, QC
July 19, 2018

Edith Bouchard-Marchand, ing.
Process Engineer
Soutex

(Signed & Sealed) "*Réjean Sirois*"

Dated at Brossard, Quebec
July 19, 2018

Réjean Sirois, P.Eng.
Vice President Geology and Resources
G Mining Services Inc.

29 CERTIFICATE OF QUALIFIED PERSON

29.1 VINCENT BLANCHET, ING.

I, Vincent Blanchet, ing, as an author of this report entitled “Technical Report on the Essakane Gold Mine Heap Leach re-feasibility Study, Sahel Region, Burkina Faso”, effective June 5, 2018 and dated July 19, 2018, do hereby certify that:

1. I am a Geological Engineer with IAMGOLD Corporation of 1111, St-Charles Ouest - Tour Est, Suite750 Longueuil, Québec

2. I am a graduate of Université Laval 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 ten 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 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 1.3.1 to 1.3.6; 4 to 12; 14.1, 14.2, 14.4, 14.5.1, 14.5.2, 14.6.1, 14.6.3; 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 1.3.1 to 1.3.6; 4 to 12; 14.1, 14.2, 14.4, 14.5.1, 14.5.2, 14.6.1, 14.6.3; and 23; and parts of sections 3, 25, 26, and 27 of 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 19th day of July, 2018

(Signed & Sealed) "*Vincent Blanchet*"

Vincent Blanchet, ing.

29.2 PHILIPPE CHABOT, ING.

I, Philippe Chabot, ing., as an author of this report entitled “Technical Report on the Essakane Gold Mine Heap Leach Pre-feasibility Study, Sahel Region, Burkina Faso”, effective June 5, 2018 and dated July 19, 2018, do hereby certify that:

1. I am a Mine optimization expert with IAMGOLD Corporation of 1111, rue St-Charles Ouest, Tour Est, bureau 750, Longueuil (Québec), J4K 5G4.

2. I am a graduate of Université Laval, Québec in 2004 with a Bachelor 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; 16.1, 16.3, and 16.4 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 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; 16.1, 16.3, and 16.4; 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 19th day of July, 2018

(Signed & Sealed) “*Philippe Chabot*”

Philippe Chabot, ing.

29.3 STÉPHANE RIVARD, ING.

I, Stéphane Rivard, ing., as an author of this report entitled “Technical Report on the Essakane Gold Mine Heap Leach Pre-feasibility Study, Sahel Region, Burkina Faso”, effective June 5, 2018 and dated July 19, 2018, do hereby certify that:

1. I am Director Metallurgy with IAMGOLD Corporation of 111, rue Saint-Charles Ouest, Tour Est, bureau 760, Longueuil (Québec), J4K 5G4.

2. I am a graduate of LAVAL University - 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 providing also 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 June 4-11, 2018.

6. I am responsible for sections 1.3.9, 1.3.10, 1.3.11; 13.1 to 13.4; and 17.1 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 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, 1.3.11; 13.1 to 13.4; and 17.1; 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 19th day of July, 2018

(Signed & Sealed) "*Stéphane Rivard*"

Stéphane Rivard, ing.

29.4 DENIS ISABEL, ING.

I, Denis Isabel, ing., as an author of this report entitled “Technical Report on the Essakane Gold Mine Heap Leach Pre-feasibility Study, Sahel Region, Burkina Faso”, effective June 5, 2018 and dated July 19, 2018, do hereby certify that:

1. I am Director Health Safety and Sustainability with IAMGOLD Essakane SA of 146, rue 13.49, quartier Zogona, 09 BP 11 Ouagadougou 09, Burkina Faso.

2. I am a graduate Student 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 Student 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 1.3.14; 20.1, 20.2, 20.3, 20.4, 20.5, 20.6, 20.7, 20.9, 20.10, and 20.11 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 1.3.14; 20.1, 20.2, 20.3, 20.4, 20.5, 20.6, 20.7, 20.9, 20.10, and 20.11; and parts of sections 1.1, 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 19th day of July, 2018

(Signed & Sealed) "*Denis Isabel*"

Denis Isabel, ing.

29.5 LUC-BERNARD DENONCOURT, ING.

I, Luc-Bernard Denoncourt, ing., as an author of this report entitled "Technical Report on the Essakane Gold Mine Heap Leach Pre-feasibility Study, Sahel Region, Burkina Faso", effective June 5, 2018 and dated July 19, 2018, do hereby certify that:

1. I am Projects Manager with IAMGOLD Corporation,1111, St. Charles Street West, Longueuil, QC, Canada, J4K 5G4

2. I am a graduate of Laval University, Quebec City; in 2002 in Mining Engineering.

3. I am registered as a Mining Engineer in the Province of Quebec (OIQ #129874). I have worked as a mining engineer and project manager for a total of sixteen 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 and 2018.

6. I am responsible for sections 1.1, 1.2, 1.3.12, 1.3.13, 1.3.15, 1.3.16; 2; 18; 19; 21; 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 and 2018.

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, 1.3.12, 1.3.13, 1.3.15, 1.3.16; 2; 18; 19; 21; 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 19th day of July, 2018

(Signed & Sealed) "*Luc-Bernard Denoncourt*"

Luc-Bernard Denoncourt, ing.

29.6 TRAVIS J. MANNING, P.E.

I, Travis J. Manning, P.E., as an author of this report entitled "Technical Report on the Essakane Gold Mine Heap Leach Pre-feasibility Study, Sahel Region, Burkina Faso", effective June 5, 2018 and dated July 19, 2018, do hereby certify that:

1. I am Senior Engineer for Kappes, Cassiday & Associates located at 7950 Security Circle, Reno, Nevada 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 visited the Essakane Mine Site on 20 June 2017.

5. I am responsible for sections 13.5, 13.6, and 17.3 of the Technical Report. I share responsibility with my co-authors for sections 3, 25, 26, and 27.

6. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

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, 13.6, and 17.3, and parts of sections 3, 25, 26, and 27 of this Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make this Technical Report not misleading;

Dated 19th day of July, 2018

(Signed & Sealed) "*Travis J. Manning*"

Travis J. Manning, P.E.

29.7 EDWARD SAUNDERS, P.ENG.

I, Edward Saunders, P.Eng., as an author of this report entitled "Technical Report on the Essakane Gold Mine Heap Leach Pre-feasibility Study, Sahel Region, Burkina Faso", effective June 5, 2018 and dated July 19, 2018, do hereby certify that:

1. I am Edward Saunders with SRK Consulting (Canada) Inc. of 22nd Floor, 1066 West Hastings Street, Vancouver, BC, V6E 3X2, Canada

2. I am a graduate of University of Canterbury, New Zealand in 2008 with a B.Sc. degree in Geological Sciences, and University of Canterbury, New Zealand in 2009 with a Post-Graduate diploma in Engineering Geology, and University of New South Wales, Australia in 2013 with a Masters of Engineering Science degree in Geotechnical Engineering.

3. I am registered as a Professional Engineer in the Province of British Colombia (Reg.# 46438). I have worked as a rock mechanics/geological engineer for a total of eight years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Geotechnical investigation, data processing and analytical calculations for numerous greenfield studies and for operational open pit expansion projects.
   - Geotechnical pit slope stability assessment for operational mines located in sedimentary and folded geological formations. Including other Gold mines.
   - Inspections and audits for the evaluation of pit and waste rock dump slope stability performance at existing mine operations, both for regulatory and internal governance.
   - Pit slope and waste rock dump design for numerous greenfields studies and/or operations located in North America, South America, Asia, Australia, Europe and 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 Essakane Mine Site on December 7 to 15, 2017.

6. I am responsible for sections 16.2 and 20.8 of the Technical Report. I share responsibility with my co-authors for sections 3, 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 completed geotechnical site inspections and conducted stability assessment and pit slope design work for the open pits on the property that is subject of this 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 16.2 and 20.8; 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 19th day of July, 2018

(Signed & Sealed) "*Edward Saunders*"

Edward Saunders, P. Eng.

29.8 CAM SCOTT, P.ENG.

I, Cam Scott, P. Eng., as an author of this report entitled "Technical Report on the Essakane Gold Mine Heap Leach Pre-feasibility Study, Sahel Region, Burkina Faso", effective June 5, 2018 dated July 19, 2018, do hereby certify that:

1. I am a Principal Consultant (Geotechnical Engineering) with the firm of SRK Consulting (Canada) Inc. (SRK) with an office at Suite 2200, 1066 West Hastings Street, Vancouver, British Columbia, Canada.

2. In 1974, I obtained a B.A.Sc. degree in Geotechnical Engineering from the University of British Columbia. In 1984, I obtained an M. Eng. Degree in Civil Engineering from the University of Alberta.

3. I have been a Professional Engineer registered with the Association of Professional Engineers and Geoscientists of British Columbia (#11523) since 1978. My relevant experience for the purpose of the Technical Report is:
   - Continuous practice as a Geotechnical Engineer for 44 years; and
   - Engagement on the geotechnical and hydrogeological aspects of mining projects, specifically the site selection, design, permitting, operation, and closure of mine waste and heap leach facilities, over this period.

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 Site.

6. I am responsible for section 20.12 and share responsibility with my co-authors for sections 3, 25, 26, and 27 of the Technical Report.

7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.

8. I have had no 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, section 20.12 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 19th day of July, 2018

(Signed & Sealed) "*Cam Scott*"

Cam Scott, P. Eng.

29.9 EDITH BOUCHARD MARCHAND, ING.

I, Edith Bouchard Marchand, ing., as an author of this report entitled "Technical Report on the Essakane Gold Mine Heap Leach Pre-feasibility Study, Sahel Region, Burkina Faso", effective June 5, 2018 and dated July 19, 2018, do hereby certify that:

1. I am process engineer with Soutex of 357, Jackson Street, Québec, QC, Canada.

2. I am a graduate of École Polytechnique de Montréal in 2000 with a Bachelor Degree and a Master Degree in Chemical Engineering.

3. I am registered as a Professional Engineer in the Province of Quebec (Reg.# 128635). I have worked as a mining engineer for a total of 10 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Metallurgical support for various projects to increase gold recovery;
   - Metallurgical engineer for many engineering projects from scoping studies to detailed phase.

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 in November 2017.

6. I am responsible for section 17.2 and share responsibility with my co-authors for sections 3, 25, 26, and 27 of the Technical Report.

7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.

8. I have had no 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, section 17.2 and parts of sections 3, 25, 26, and 27 of the Technical Report or which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated 19th day of July, 2018

(Signed & Sealed) "*Edith Bouchard-Marchand*"

Edith Bouchard Marchand, ing.

29.10 RÉJEAN SIROIS, ING.

I, Réjean Sirois, ing., as an author of this report entitled “Technical Report on the Essakane Gold Mine Heap Leach Pre-feasibility Study, Sahel Region, Burkina Faso”, effective June 5, 2018 and dated July 19, 2018, 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 Student 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 practiced my profession continuously since 1985 and have extensive experience in estimating mineral resources 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 Essakane Mine Site on numerous occasions. My latest visit was on March 27-30, 2018.

6. I am responsible for sections 14.3, 14.5.3, and 14.6.2 of the Technical Report. I share responsibility with my co-authors for sections 3, 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 had no 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 14.3, 14.5.3, and 14.6.2, 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 19th day of July, 2018

(Signed & Sealed) “*Réjean Sirois*”

Réjean Sirois, ing.