NI 43-101 Technical Report

Feasibility Study

Zgounder Expansion Project

Kingdom of Morocco

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The quality of information, conclusions and estimates contained herein is consistent with the level of effort involved in DRA services, based on:

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1 EXECUTIVE SUMMARY

1.1 Property Description and Location

The Zgounder Project (the “Project” or “Mine”) is a silver mining project located in the Kingdom of Morocco, approximately 265 km east of the City of Agadir (pop. 575,320), in the Taroudant Province.

Aya was authorised by the *Office National des Hydrocarbures et des Mines* (“ONHYM”) to prospect and exploit base and precious metals at the Zgounder Mine. The mining title number 09/2096 and exploitation permit No. 2306 (now exploitation licence No. 393459) provide surface rights and access to the Project and allows any type of mining. The exploitation licence number 393459 covers 16 km^2. Ownership is an 85% and 15% joint venture between Aya subsidiary Zgounder Millenium Silver Mining SA (“ZMSM”) and ONHYM.

1.2 Accessibility, Climate, Local Resources, Infrastructure, and Physiography

The Zgounder Mine is accessible from the City of Agadir via well-maintained paved highways N10 and P1706 that run east for 205 km to Taliouine in the Taroudant Province. Most of the remaining 61 km to the Mine are via a paved road to the village of Askaoun. The final five (5) km drive to the Mine is via a dirt road that could be upgraded. The Zgounder Mine is also accessible via a 278 km drive on paved highway from the City of Marrakesh.

The Zgounder Mine is located between 1,925 and 2,200 metres above sea level (“MASL”), on the western flank of the Siroua Massif in the Anti-Atlas Mountains. This region is separated from the influence of the Mediterranean climate by the High Atlas Mountains to the north, and therefore is subject to the Sahara Desert climate.

The main villages are located near rivers for water sources and select vegetation (cereals, vegetables and some trees). The local population is exclusively Amazigh (aboriginal) with a semi-sedentary lifestyle. The economy is principally supported by livestock, agriculture and food trade (saffron, potatoes, dates), and manufacture of traditional carpets. Basic supplies, such as food and limited accommodation, are available at Askaoun. The larger City of Talioune offers more amenities and services. Special items must be purchased from Agadir.

The mining manpower for Zgounder resides in nearby villages, located from 5 km to 10 km from the Mine site. Skilled labour is available in nearby villages and some inhabitants were employees of previous operators of the Zgounder Mine. Mine site facilities include crew houses, offices, drill core shack, a mine portal and trails linking mine entrances. The Project is powered from two (2) substations connected to national power grid at Askaoun, the closest village to the Zgounder Mine.

In addition to the national powerline, standby power is available from three (3) 1,000 kVA (850 kW) generators.

The topography at Zgounder consists of moderately steep hills with high altitudes, in the range of 2,100 MASL, and low valleys with seasonal water flow in rivers. Vegetation is limited to minor alpine flowers, mosses, lichens and small evergreen trees. Wheat is cultivated on man-made terraces near the villages. The terraces are irrigated by springs and dams.

1.3 History

The Zgounder Silver Deposit has a long history of intermittent exploration and mining activities from ancient times to present day. The Zgounder Silver Deposit was first exploited between the 10th and 12th Century mainly in exposed oxidized zones with native silver stringers in veins. Since then, exploration campaigns and mining activities have been completed by SNAM-BRGM (1950-1979), SOMIL (1982-1990); BRPM (1990-1999); CMT (2000-2004) and Maya Gold & Silver (2012-2020).

In 2014, Maya Gold & Silver (now Aya) commissioned GoldMinds Geoservices Inc. to prepare the first NI 43-101 compliant Mineral Resource estimation and a Preliminary Economic Assessment (PEA) of the Zgounder Mine, in order to resume mining and exploitation. Aya publicly disclosed a Pre-Feasibility Study (PFS) on May 2014, which was jointly prepared by GMG and SGS. Processing operations commenced in July 2014 and Aya announced the first silver pour in August 2014 and production of the first 20 silver ingots.

The surface diamond drilling programs of 2015 and 2017 allowed Maya to increase the Mineral Resource of Zgounder and intersect silver-rich mineralisation in the East Zone, close to surface. Maya also intersected very rich silver mineralisation that probably corresponded to the extension of known underground mineralisation at elevation 1,655 MASL.

On April 12, 2017, Maya commissioned GMG to prepare an updated Mineral Resource Estimate and PEA of the Zgounder Mine. Maya publicly disclosed the updated Mineral Resources Estimate on January 8, 2018 and the PEA on February 5, 2018. The PEA was based on the January 8, 2018 updated Mineral Resources Estimate.

In 2021, Aya mandated P&E Mining Consultants Inc. (P&E) to prepare a new Mineral Resource Estimate based on new drilling information. An initial Mineral resource estimate was prepared in October 2021 and a subsequent update in December 2021

1.4 Geological Setting and Mineralisation

The Zgounder Deposit occurs within the Proterozoic Siroua Massif that occupies a transitional position between the northern mobile Panafrican Belt and the southern Eburnean Domain in the West-African Craton. The Siroua Massif is composed of geological assemblages belonging to the

Precambrian I, II and III; each separated by major discontinuities. The oldest rocks of the Siroua Massif (P1) consist of gneisses and amphibolites unconformably overlain by ophiolitic complexes, volcano-sedimentary units, alternating schist-sandstones and limestones, quartzites, and turbidites (PII). The Zgounder Deposit occurs in the PIII assemblage (Late Neoproterozoic), which is characterized by felsic calc-alkaline/alkaline volcanic units corresponding to the initiation of rifting at the start of the Infracambrian-Cambrian Transgression.

The Zgounder volcano-sedimentary assemblage forms a large EW-oriented monoclinal structure with a general southerly tilt. To the north, the assemblage sits on an andesite basement, to the west it is intruded by the Askaoun Granodioritic Massif (PIII), whereas to the east, it is overlain by volcano-sedimentary rocks of the Ouerzazate series (PII) and Neogene phonolites.

The Zgounder Series is divided into three units, which in stratigraphic (oldest to youngest) order are:

1. Blue Formation (300 m to 400 m) thick composed of sandstone, greywacke and shale with layers of tuff and quartz keratophyre followed by an orange rhyolite unit;

2. Brown Formation (350 m to 450 m thick) composed of micaceous schistose sandstone overlain by a 45 m thick dolerite sill/dyke; and

3. Black Formation (900 m thick) containing at its base a felsic volcanic complex (ignimbrite, rhyolitic breccia, devitrified rhyolite) and forming the hanging wall rock of the silver mineralisation in the Brown Formation. To the south, the Black Formation transitions into sandstone, greywacke and conglomerate.

The Zgounder shale-sandstone strikes east and dips steeply to the south, forming the south flank of an anticline generated by north-to-south compression. There are four faulting and fracturing system sets at Zgounder:

1. East to West oriented set corresponding to the opening and filling of fractures with argillaceous material and to subvertical fractures;

2. North to South oriented set;

3. NNE to NNW oriented set dipping 60º to 75º E; and

4. NNE-SSW sub-horizontal set.

The silver mineralisation occurs in three, commonly superimposed styles:

1. Mm-thick beds of well crystallized, finely disseminated pyrite associated with quartz and other sulphides found in chloritized and tuffaceous pelitic layers of the Brown Formation with silver grades of 5 g/t to 25 g/t Ag;
2. Native silver veinlets associated with proustite (Ag_3AsS_3), argentite (Ag_2S) and filling micro-fractures discordant with the stratification and suggesting stockwork-type mineralisation; and
3. Native silver dissemination with or without sulphide veinlets (sphalerite, galena, argentite and cinnabar) in brecciated sandstone-shale layers and spotted by nodules and flakes of chlorite and (or) carbonate.

The paragenetic sequence indicates two (2) stages of mineralisation: an early Fe-As stage (silver-bearing pyrite and arsenopyrite) and a later Ag-bearing polymetallic (Zn, Pb, Cu, Hg; sphalerite and chalcopyrite) stage. Native silver is the most common silver mineral and forms an amalgam with Hg. Tension gashes originally trapped the silver mineralisation within a NNE-oriented shear zone affecting the Brown Formation shale-sandstone beds containing anomalous Ag values. These mineralised structures were subsequently transposed by EW-oriented structures to form isolated Ag-mineralised lenses and bodies.

Zgounder is a low-sulphidation epithermal silver deposit hosted in Neoproterozoic age, sedimentary rocks.

1.5 Exploration Work and Drilling

Since 2013, exploration programs included geological mapping, trenching, sampling and prospecting activities at Zgounder. These activities focused mainly on mineralised fracture sets. In addition, 3-D laser scan surveys were completed in all of the underground workings.

The purpose of these laser surveys was to generate accurate 3-D maps of the underground development and stopes.

Several exploration surface and underground drilling, channelling and trenching campaigns have been completed, starting in the 1980s by previous operators followed by Maya in 2013, 2015 and 2017 through 2019 and Aya in 2020-2021. The drilling database comprised a total of 3,148 holes for 136,816 m.

Percussion drilling using air compressed hammer (T28 and YAK-T28) are routinely used for production purposes and for exploration purposes. Data gathered from T28/YAK-T28 holes is used in Mineral Resource estimation and to identify new mineralised areas for short-term mine planning.

A total of 1,653 T28/YAK-T28 holes for 31,719 m were drilled historically and until the end of January 2021 as exploration and also production holes.

In addition to drilling, underground wall and roof channel sampling were performed on all adits, galleries and stopes. A total of 658 channels for 9,071 m are used in the drilling database supporting the Mineral Resource Estimate presented in Section 14 of this Technical Report.

Reverse circulation ("RC") and percussion drill programs ("T28") were completed in 2015, 2016 and 2018 to 2019. The 2015 RC percussion hole data have not been included in the Zgounder database, due to issues with that program.

In 2016, a total of 1,598.4 m was drilled using a T28 percussion hammer rig on the 2000 and 2100 Levels of the Zgounder Mine. The percussion holes drilled in the North Zone intersected some mineralised intervals and confirmed the extension to the east of Panel 9. The data highlighted a new zone to the northeast of Corps D, above the 2100 Level. The holes have been drilled from an exploration raise in a fan and alongside the drift to delineate the shape of the mineralised body. Furthermore, the area has been drilled to delineate the geometry of the Y6 with an extension to the east of that body on level 2100. New findings occurred on the 2030 and 2000 Levels.

The 2019 reverse circulation drilling program consisted of 32 drill holes totalling 3,611 m that were drilled from the surface on the East Zone of the Zgounder Property. The RC drilling campaign aimed to provide a better understanding of the distribution, orientation and thickness of the mineralised structures, and to explore the vertical extensions of the exposed mineralised structures. The results confirmed the continuity of the known mineralised domains and new occurrences hosted in the same major East to West oriented structures. The RC drilling results of this campaign led to an underground diamond drilling program to explore the vertical extensions of the mineralisation in the western and the central part of the Zgounder Mine. Drill holes ZG-RC-1, -2, -3, -4, -5, -6, -7, and -9 intersected silver mineralisation <50 g/t, in attempt to expand mineralised zones to the south.

In 2020 and 2021, Aya also conducted localised definition drilling using T28. Hole ZG-20-30, which intersected 1,946 g/t Ag over 9 m extended the strike to the east. Furthermore, hole ZG-20-31, located east of ZG-20-30, intersected 688 g/t Ag over 17.5 m.

Drill holes ZG-SF-20-03 and ZG-SF-20-07 from the underground drill program confirm extensions of mineralised lenses toward the west at underground level 1975. Additionally, drill hole ZG-SF-20-15 intersected several significant high-grade intervals including 1,505 g/t Ag over 6 m and 855 g/t Ag over 5.5 m, confirming high-grade silver mineralisation at depth to the east and below the 1,975 m level. Drill holes ZG-SF-20-25 and ZG-SF-20-23 intersected 2,728 g/t Ag and 4,517 g/t Ag over 6 m and 3.5 m, respectively, below the previous (March 2021) Mineral Resources and to the east. Drill hole ZG-SF-20-18 intersected 854 g/t Ag over 12 m below the previous (March 2021) Mineral Resources. In the post-February 2021 period, drill hole T28-21-2125-203 intersected 2,417 g/t Ag over 12 m, drill hole T28-21-1975-253 returned 3,272 g/t Ag over 7.2 m, and drill hole T28-21-1975-253bis intersected 3,101 g/t Ag over 7.2 m. These three drill holes indicated continuity of high-grade mineralisation in a previously untested area below current operations.

Diamond drilling programs were completed at Zgounder in 2015, 2017 and 2019 through 2021. Atotal of 84,362 m has been drilled in 449 surface and underground diamond drill holes at Zgounder. In 2015, Maya completed a diamond drill program of 17 drill holes totalling 5,896 m. Native silver was observed in all eight of the holes drilled. The silver mineralisation is associated with sphalerite and galena. A 10 m sample from drill hole HL-Ext-012 was selected as a high priority sample to evaluate a sub-parallel mineralised trend north of the main Zgounder Deposit. This sample was collected from 31.3 m to 41.3 m in drill hole HL-Ext-012. GoldMind's independent assay prepared and analyzed by Fire Assay at Bourlamaque Assay Laboratories Ltd. in Val d'Or (Quebec) returned an average grade of 1,098 g/t Ag. This drill hole was collared in the valley going northward and likely intersected an extension to the east of the northern body. Three diamond holes were drilled in West Zone. Sulphide mineralisation consisting of sphalerite, galena and pyrite was encountered in altered sandstone. Multi-element analysis, particularly for zinc, lead and copper, led to identification of at least two important polymetallic corridors with horizontal widths of approximately 25 m and 40 m, extending over 1,000 m in length, with shoots of higher-grade silver mineralisation.

In 2017, Maya conducted a diamond drilling program planned and supervised by Goldminds.

The program consisted of 57 drill holes totalling 14,823 m of diamond drilling. A new zone was intersected to the East, where the mineralisation was identified at the surface. At the North Zone, hole ZG-17-03 extended mineralisation at depth from known occurrences (panels 8 & 9) at higher elevation. Similar mineralisation was observed in drill hole ZG-17-10. Drill hole ZG-17-16 is the deepest hole drilled to date at Zgounder, with a depth of 684 m (at an elevation of 1,613 MASL). The drill hole intersected disseminated native silver over 3 m at 630 m and an altered granite contact was intersected at 653 m along the drill hole. Zinc in the form of sphalerite is associated with high-grade silver reaching up to 2.38% over 1.5 m. It was the first time that the Aya intersected this type of mineralisation at depth at Zgounder.

In 2019, Maya completed an eight-hole drill program totalling 2,033.9 m. The drill program focused to the east, which corresponded to a new zone. This new zone covers an area of 200 m x 150 m that includes several mineralised envelopes, which are oriented mostly E-W with a vertical extent of 185 m below the surface intersected in drill hole ZG-19-01. The presence of the mineralisation near the surface was confirmed by the trenches results (Trench 02 11 m at 130.9 g/t Ag) and requires further work to fully define the extent of mineralisation.

In 2020, the initial diamond drilling program at Zgounder was expanded twice to follow-up on promising results, completing the year with 13,904 m of surface and underground diamond drilling. Aya extended the mineralisation approximately 90 m along the eastern strike extension and at depth. Drilling at the Zgounder Mine commenced in September 2020 and continued to mid-January 2021. Four diamond drills were operating on surface and two electric drills operated underground within the mine. High-grade mineralised extensions were encountered at 408 m and 467 m (ZG-20-04 and ZG-20-09, respectively) from surface, which indicates potential for new underground mineralised zones. In addition, drill holes ZG-20-19 and ZG-20-22 both extend the mineralisation eastward. The results confirm high-grade Ag mineralisation below the current mining operations (ZG-20-06) with an intercept of 4.0 m at 9,346 g/t Ag. In addition, drill hole ZG-20-01 confirms new high-grade mineralisation at depth at the granite contact. Results from drill hole ZG-20-13 confirm high-grade Ag mineralisation at depth with an intercept of 5.5 m at 1,273 g/t Ag. In addition, drill hole ZG-20-36 intercepted 1,587 g/t Ag over 3 m, confirming a new high-grade extension to the east.

In the post-February to December 2021 period, 166 surface and underground diamond holes were drilled totalling 33,986.90 m at Zgounder. The drilling campaign had two (2) objectives: 1) increase the confidence level of the Exploration Target area identified in March 2021; and 2) further extend mineralisation in the eastern part of the Deposit. Up to eight (8) diamond drill rigs were in operation on the mine permit. Hole ZG-SG-21-67 (underground) encountered 1,383 g/t Ag over 13.5 m to extend the mineralisation trend below the 1975 level; drill hole ZG-21-51 (surface) cut 1,615 g/t Ag over 8.5 m to confirm eastern vertical continuity 200 m below surface; hole ZG-21-50 (surface): returned 2,446 g/t Ag over 4.0 m continued definition of high-grade mineralisation below the 2030 m Level); drill hole ZG-21-43 (surface) returned 2,311 g/t Ag over 2.0 m to continue definition of eastern high-grade mineralisation. An outcome of this diamond drill program was extension of the mineralised strike length eastward by an additional 375 m.

1.6 Sample Preparation, Analysis, Security and Verification

Verification of the assay database was performed by the QPs against laboratory certificates that were obtained independently from the African Laboratory for Mining and Environment (AfriLab) in Marrakesh, Morocco. Only a few insignificant errors were found in the assays. The QPs also validated Aya's Mineral Resource database by checking for inconsistencies in analytical units, duplicate entries, interval, length or distance values less than or equal to zero, blank or zero-value assay results, out-of-sequence intervals, intervals or distances greater than the reported drill hole length, inappropriate collar locations, survey and missing interval and coordinate fields. Some minor errors were identified and corrected in the database.

Assays taken from the independent site visit samples match closely to Aya's data for all metals of interest. The authors of this Technical Report consider that the supplied assay database is robust and suitable for Mineral Resource estimation.

1.7 Metallurgy and Processing

A main composite and five (5) variability samples were selected to represent the ore body spatially, by lithology and to cover a range of grades. Comminution testing showed all the samples can be classified as very hard but only mildly abrasive. Mineralogical examination revealed the potential for nugget effects as a few unexpectedly large silver particles were observed. Gravity concentration testing yielded an average gravity recovery of 15% for conventional tests and 34% for the EGRG test. This is sufficient to warrant inclusion of a gravity concentration step. Whole ore rougher flotation tests showed an insensitivity to grind size hence a P_{80} of 100 microns was adopted for the remainder of the test program. Comparative cleaner flotation tests showed a significant shift of the grade vs recovery curve when regrinding the rougher concentrate to 80% passing 20 microns. Locked cycle tests (LCT) revealed that inclusion of a gravity step would results in a small improvement in overall silver recovery.

Cyanidation tests were performed on whole-ore, gravity tails as well as both flotation products. Whole-ore cyanidation yielded an 89% silver recovery. Leaching of flotation tails yielded silver extractions of around 65% at a NaCN consumption of 1.4 kg/t. Pre-oxygenation tests indicated that the cyanide consumption can be effectively curbed through oxidizing cyanide consumers prior to and during the initial part of the leach. The NaCN consumption decreased to 0.96 kg/t at a constantly maintained concentration of 2g/l. Further reductions are possible at lower cyanide concentrations, but the silver extraction also decreases.

Leaching of flotation concentrate showed a sensitivity to NaCN with the extraction of silver increasing by 5% when increasing the NaCN concentration from 4 to 12 g/l. Unfortunately, the cyanide consumption also doubled over this range. Tests performed using samples of the current operating flotation plant concentrate showed that it behaves similarly to the samples tested during this testwork program.

Merrill-Crowe cementation tests indicated that almost all (>99.8%) of the silver will precipitate from the combined pregnant solutions and that excess zinc is not required.

Cyanide destruction tests were conducted on barren solution from a cementation test at four (4) different hydrogen peroxide additions. It showed an optimum minimum free cyanide is achieved when adding 250% of the stoichiometric H_2O_2 requirement.

Carbon adsorption kinetic and equilibrium tests were conducted to derive modelling parameters for subsequent simulations of various CIP scenarios. These showed that a 12 t/d carbon advancement through an 8 stage carousel with 12 tonnes carbon in each tank would yield loaded carbon silver grades of around 5.5 kg/t and dissolved silver losses of 0.34 mg/l. It may be more economical to target even higher loadings in order to decrease elution costs at the expense of additional dissolved losses. When moving only 10 t/d the predicted dissolved loss can be expected to increase to 0.5 mg/l which is still acceptable and justifiable given the saving in elution costs.

Flotation concentrate samples were subjected to dynamic settling test. These showed that a 66% w/w solids underflow density is achievable at a unit rate of 0.2 m^2/(t/d). This would decrease to 59.9% w/w solids at 0.12 m^2/(t/d). A critical solids density of 68% w/w solids was established with a yield stress of 36 kPa (unsheared). CCD thickener testing and modelling showed a 5 stage CCD train would provide 99.5% washing efficiency using 1.69 m^3/t washing water.

Sedimentation testing of the flotation tailings showed that a 54% underflow density is readily achievable at a unit area of around 0.22 m^2/(t/d) but it demands 110 g/t flocculation.

Variability testing yielded overall silver recoveries ranging from 84.4% to 94% when subjecting these samples to tests mimicking the selected flowsheet.

1.8 Mineral Resource Estimate

The updated Mineral Resource Estimate incorporates drilling carried out on Zgounder between February 2018 and December 2021. The updated Mineral Resource Estimate is summarised in Table 1.1. At a cut-off grade of 65 g/t Ag, pit-constrained updated Measured and Indicated Mineral Resource totals 514 kt grading 357 g/t Ag for 5.9 Moz Ag. At a cut-off grade of 75 g/t Ag, out-of-pit updated Measured and Indicated Mineral Resource totals 9.0 Mt grading 309 g/t Ag for89.3 Moz Ag and updated Inferred Mineral Resource totals 542 kt grading 367 g/t Ag for 6.4 Moz Ag. At a cut-off grade of 50 g/t Ag, tailings updated Indicated Mineral Resource totals 272 kt grading 94 g/t Ag for 817 koz Ag.

The updated Measured and Indicated Mineral Resources for Zgounder totalling 9.8 Mt averaging 306 g/t Ag for 96.1 Moz Ag represent an increase of 116% compared to the previous (March 2021) Measured and Indicated Mineral Resources of 44.4 Moz Ag. The updated Inferred Mineral Resources for Zgounder totalling 542 kt averaging 367 g/t Ag for 6.4 Moz Ag represents an increase of 1,519% compared to the previous (March 2021) Inferred Mineral Resources of 395 koz Ag.

Table 1.1 – Updated Mineral Resource Estimate (1-13)

Area	Class	Cut-Off (Ag g/t)	Tonnes (k)	Ag (g/t)	Ag (koz)
Pit Constrained	Measured	65	108	477	1,656
	Indicated	65	406	325	4,242
	Measured +Indicated	65	514	357	5,898
	Inferred	-	-	-	-
Out-of-Pit	Measured	75	3,403	343	37,527
	Indicated	75	5,576	289	51,810
	Measured +Indicated	75	8,979	309	89,337
	Inferred	75	542	367	6,395
Tailings	Measured	-	-	-	-
	Indicated	50	272	94	817
	Measured +Indicated	50	272	94	817
	Inferred	-	-	-	-
Total	Measured	-	3,511	347	39,183
	Indicated	-	6,254	283	56,869
	Measured +Indicated	**-**	**9,765**	**306**	**96,052**
	Inferred	-	542	367	6,395

1. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues. There is no certainty that Mineral Resources will be converted to Mineral Reserves. No additional Inferred Mineral Resources were reported for this update.
2. Mineral Resources are reported inclusive of Mineral Reserves
3. The Inferred Mineral Resource in this estimate has a lower level of confidence that that applied to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of the Inferred Mineral Resource could be upgraded to an Indicated Mineral Resource with continued exploration.
4. The Mineral Resources in this news release were estimated in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions (2014) and Best Practices Guidelines (2019) prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council.
5. A silver price of US$22.5/oz with a process recovery of 90%, US$20/t rock process cost, US$16.5/t tailings process cost and US$7/t G&A cost were used.
6. The constraining pit optimization parameters were US$15/t of mineralised material (including waste mining) and 50º pit slopes with a 65 g/t Ag cut-off.
7. The out-of-pit parameters used a US$22/t mining cost. The out-of-pit Mineral Resource grade blocks were quantified above the 75 g/t Ag cut-off, below the constraining pit shell and within the constraining mineralised wireframes. Out–of-pit Mineral Resources exhibit continuity and reasonable potential for extraction by the cut and fill underground mining method.
8. The tailings parameters were at a US$9/t mining cost, and Mineral Resource grade blocks were quantified above the 50 g/t Ag cut-off.
9. Individual calculations in tables and totals may not sum correctly due to rounding of original numbers.
10. Grade capping of 6,000 g/t Ag was applied to composites before grade estimation.
11. Bulk density was determined from measurements taken from drill core samples.
12. 1.2 m composites were used during grade estimation.
13. Previously mined areas of the deposit were depleted from the Mineral Resource Estimate.

The Updated Mineral Resource Estimate incorporates drilling carried out on Zgounder from February 2018 to September 2021. The Mineral Resource database update consists of 516 drill holes (surface and underground combined) for 41,932 m completed at Zgounder. The drilling campaign had two (2) objectives: 1) increase the confidence level of the Exploration Target area identified in March 2021; and 2) and further extend mineralisation in the eastern part of the Deposit. The two (2) objectives were achieved, in that the drilling extended the east-west strike length of the mineralisation from 775 m to 1,100 m and at depth.

Three-dimensional block models were created for the Zgounder Deposit and for the historical tailings located a few hundred metres northwest of the mine site. A geological rock code system was introduced and assigned to the various lithological units and mineralised domains. Continuity directions were assessed based on the orientation of the domains and the spatial distribution of silver. Separate variograms were generated for 1.2 m down-hole silver composites within each domain. Mineralisation modelling, grade estimation and Mineral Resource reporting were conducted using GemcomTM, LeapfrogTM, Snowden SupervisorTM and NPV SchedulerTM software. Ordinary kriging was used for grade estimation into 2.0 m x 2.0 m x 2.0 m model blocks.

This Mineral Resource Estimate forms the basis of Aya's initial Mineral Reserve Estimate in conjunction with the Feasibility Study of Zgounder.

1.9 Mineral Reserves Estimate

Mineral Reserves for the Zgounder mine were estimated using HxGN MinePlan's MSOPit module to determine the economic pit shell for the open pit portion, and Deswik.SO to determine the underground reserves. The historical tailings were converted from Mineral Resources to Mineral Reserves using economic parameters and calculations. Only Measured and Indicated Mineral Resource categories were considered for the Mineral Reserves.

For the open pit mining and historical tailings reclamation, a standard open pit truck and shovel operation was assumed, with no drill & blast requirements for the historical tailings. For the underground mining, a combination of drift-and-fill and long-hole stoping was used. A combined ore production of 2.7 ktpd, combining the new mill at 2.0 ktpd and the existing mill at 0.7 ktpd, was used.

Mine designs were created for the open pit and the underground portions of the mine. The operational pit was designed using the economic pit shell as a guide, adding 12 m wide ramps to accommodate the chosen 8x6 trucks, and ensure an appropriate mining width is respected. Developments to access the stopes were designed for the underground mine to ensure access to the ore.

The Mineral Reserves are estimated at 3.1 Mt proven reserves grading 289 g/t Ag and 5.8 Mt probable reserves grading 231 g/t Ag, for a total of 8.9 Mt ore grading 251 g/t Ag. The access the

Mineral Reserves, a total of 27.6 Mt of waste will need to be extracted. Table 1.2 presents a summary of the Mineral Reserves Estimate for the Zgounder mine.

Table 1.2 – Mineral Reserves Estimate – Effective December 13, 2021

Description	Classification	Tonnage	Ag Grade	In-Situ Ag
		(Mt)	(g/t)	(Moz)
Open Pit	Proven Reserves	0.6	312	5.7
	Probable Reserves	1.6	233	12.1
	Total Open Pit Reserves	*2.2*	*253*	*17.8*
Historical Tailings	Probable Reserves	0.3	77	0.8
Underground	Proven Reserves	2.5	283	23.0
	Probable reserves	3.6	256	29.3
	Total Underground Reserves	*6.1*	*267*	*52.3*
Total	Proven Reserves	3.1	288	28.7
	Probable Reserves	5.5	239	42.1
	Total Reserves	***8.6***	***257***	***70.9***

NOTES:

1. The Mineral Reserve is estimated in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions (2014) and Best Practices Guidelines (2019) prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council.
2. The Mineral Reserve is estimated with a variable COG which was calculated by mining method.
3. Ag content (oz) are estimated as in-situ
4. An ONHYM royalty of 3% is included in the Mineral Reserve Estimate.
5. The Mineral Reserve is estimated with a mining recovery of 95%.
6. The Mineral Reserve includes both internal and external dilution. The external dilution included a mining dilution of 0.3 m width on the hanging wall and footwall for the long-hole mining method and a 0.1 m width on the hanging wall and footwall for the cut-and-fill mining methods.
7. A minimum mining width of 4m was used for the long-hole and cut-and-fill mining methods.
8. The economic viability of the Mineral Reserve has been demonstrated.
9. For the historical tailings Reserves Estimate, a silver price of US$20/oz with a process recovery of 92%, a process cost of $20.93/t (including G&A), and a mining cost of $4.31/t (including haulage) were used.
10. For the Open-pit Reserves Estimate, a silver price of US$20/oz with a process recovery of 92%, a process cost of US$22.91/t (including G&A), and a mining cost of $4.00/t (including haulage) were used.
11. For the Underground Reserves Estimate, a silver price of $20/oz with a process recovery of 92%, a process cost of US$22.91/t (including G&A), and a mining cost of $24.13/t (including haulage and backfill) were used for the combined cut-and-fill and long-hole methods.
12. The reserves estimate has an effective date of December 13, 2021.
13. Totals may not add due to rounding.

1.9.1 Mining Methods

The Zgounder Project will be mined in a combination of open pit mining, reclamation of historical tailings, and underground mining. The mine will operate year-round, seven (7) days a week, twenty-four (24) hours per day (two (2) 12-hour shifts). Two weeks of adverse weather conditions per year are considered in the mine plan.

1.9.1.1 Open Pit Mining

Conventional open pit mining with 8x6 trucks and matching shovels, undertaken by a local mining contractor, was chosen for open pit portion of the Zgounder mine. The material extracted from the pit will be loaded into 8x6 trucks and hauled to its destination. The pit will be mined over a 7-year period, with two years of pre-production mining and an additional year for stockpile rehandling at the end of the mine life.

Ore material will be sent to either the ore pass or an ore stockpile; the stockpiles will separate the low-grade and high-grade material. Rehandled stockpile material will be loaded onto the trucks and brought to the ore pass. Waste material will be sent to the waste stockpile located near the pit. Some waste material will be sent underground for use in backfill.

The mine will be operated by a local mining contractor, who will supply all equipment and staff necessary for the operation.

1.9.1.2 Historical Tailings Reclamation

The nearby historical tailings will be reclaimed at the end of the open pit mine life using a loader and truck operation. No drilling and blasting will be required. Additionally, the tailings material will not require any crushing.

The material will be loaded on to the 8x6 trucks and hauled to the mill. No waste material will be excavated. The reclamation will take place over the course of two (2) years.

The historical tailings reclamation will be undertaken by the open pit mining contractor.

1.9.1.3 Underground Mining

The underground mine will be mined using a combination of drift-and-fill and long-hole stoping. The underground mine will be accessed from surface and from the historic underground drift excavations that will require rehabilitation. The main ramp will begin from the underground on 2000L main level and be excavated up and down to the 2092L and 1648L Levels, respectively.

The development of the underground mine will be undertaken by mining contractors while the ore production will be undertaken by Aya staff. The underground operation will be undertaken over 11 years.

1.9.2 COMBINED MINE PLAN

Table 1.3 presents the combined mine plan for the open pit, historical tailings and underground portions of the Project.

Table 1.3 – Combined Mine Plan

		Unit	PP2	PP1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Total
Open Pit Mine														
Ore	Pit to Mill	kt	-	-	200.9	192.6	185.8	251.0	248.6	234.2	202.8	-	-	1,516
	Pit to Low-Grade Stockpile	kt	-	8.1	20.3	113.2	98.9	36.7	36.7	29.2	29.3	-	-	372.4
	Pit to High-Grade Stockpile	kt	-	62.6	48.2	80.8	45.7	28.9	11.3	4.1	8.4	-	-	290.2
	Low-Grade Stockpile to Mill	kt	-	-	14.5	0.1	7.6	30.1	9.5	45.3	76.8	-	188.6	372.4
	High-Grade Stockpile to Mill	kt	-	-	46.6	95.3	94.7	6.9	29.9	8.4	8.4	-	-	290.2
	Total Mill Feed	kt	-	-	262.0	288.0	288.0	288.0	288.0	288.0	288.0	-	188.6	2,178.6
	Silver Grade at Mill	g/t	-	-	350.0	303.3	202.2	279.5	311.7	231.8	230.1	-	61.8	253.5
Waste		kt	-	3,171.2	4,320.0	3,940.0	3,880.7	3,295.8	2,509.6	1,393.3	937.4	-	-	23,447.9
Total Mined		kt	-	3,241.9	4,589.5	4,326.5	4,211.1	3,612.4	2,806.2	1,660.9	1,177.9	-	-	25,626.5
Stripping Ratio		w/o	-	44.8	16.0	10.2	11.7	10.4	8.5	5.2	3.9	-	-	10.8
Tailings Reclamation														
Ore	Tonnage to Mill	kt	-	-	-	-	-	-	-	-	-	288.0	30.8	318.8
	Silver Grade	g/t	-	-	-	-	-	-	-	-	-	80.5	39.5	7.6
Underground Mine														
Ore	Tonnage to Mill	kt	226.3	251.4	594.9	671.6	673.1	673.1	673.2	673.2	673.2	745.9	237.5	6,093.4
	Silver Grade	g/t	264.4	286.4	284.5	249.8	283.3	291.9	322.8	243.8	257.6	225.2	208.5	267.5
Waste		kt	566.3	655.0	671.7	663.0	704.2	307.3	196.7	136.2	156.9	104.1	33.0	4,194.5
Total Mined		kt	792.64	906.44	1266.59	1334.61	1377.33	980.44	869.86	809.42	830.14	849.97	270.50	10,287.9
Combined Mine Plan														
Ore to ROM Pad		kt	226.3	251.4	856.9	959.6	961.1	961.1	961.2	961.2	961.2	1,033.9	456.9	8,590.8
Silver Grade		g/t	264.4	286.4	304.5	265.9	259.0	288.2	319.5	240.2	249.4	184.9	136.6	256.7
Waste		kt	566.3	3,826.2	4,991.7	4,603.0	4,584.9	3,603.1	2,706.3	1,529.5	1,094.3	104.1	33.0	27,642.5
Total Mined		kt	792.6	4,077.6	5,848.6	5,562.6	5,546.0	4,564.2	3,667.5	2,490.7	2,055.5	1,138.0	489.9	36,233.3

1.10 Recovery Methods

The processing complex at Zgounder will comprise of the following three (3) facilities:

1. Existing Pant #1 (cyanidation plant);
2. Existing Plant #2 (flotation plant); and
3. New Plant #3.

Products from the existing plants will be processed by the new facility. The existing Plant # 1 (Cyanidation Plant) will continue to treat 180 t/d of mineralised material feed and its silver sludge will be transferred to the New Plant #3 for refining. The existing Plant # 2 (Flotation Plant) will continue to treat 500 t/d of mineralised material and it will produce a concentrate slurry which will be pumped to the New Plant #3 for leaching and refinery. Additionally, its flotation tails will also be transferred to the New Plant #3, where it will combine with the new plant's flotation tailings in the Leaching-CIP circuit.

The new mineral processing facility has been designed to treat mineralised material at a nominal throughput rate of 2,000 t/d and at a head grade of 210 g/t of silver. The plant will comprise of the following unit operations:

- Crushing: two stage crushing circuit closed out by a vibrating screen;
- Grinding: single stage ball milling circuit closed out by hydrocyclones to grind the mineralised material to 80% passing 100 microns;
- Gravity Separation: gravity concentrator integrated within the grinding circuit. Fed by diverting some cyclone underflow to the scalping screen. Gravity tailings will return to the ball mill feed chute;
- Flotation: One rougher stage followed by regrinding of the rougher concentrate to 80% passing 45 microns prior to three cleaner stages;
- Intensive cyanidation: of the gravity concentrate with the pregnant solution combining with two (2) other high grade solutions (eluate and CCD overflow) ahead of the Merrill-Crowe section;
- Concentrate Leaching and Counter-Current Decantation (CCD): Cyanidation of the 3rd flotation cleaner concentrate at elevated cyanide concentrations followed by liquid-solid separation in a train of CCD thickeners with the CCD overflow reporting to the Merrill-Crowe and the barren CCD underflow reporting to the tailings leach;
- Tailings Cyanidation and CIP: leaching of flotation tailings and other tailings streams followed by adsorption onto activated carbon in a carousel CIP circuit;

- ADR process: acid-washing and elution circuit to strip silver from the carbon and regenerate it for re-use;
- Merrill-Crowe: to recover silver from the combined pregnant solutions through zinc cementation; and
- Refinery: drying and smelting of sludge to produce doré silver bars.

Key design parameters (most from testwork results) are as follows:

- Crushing circuit utilisation of 68.5% and the remainder of the plant at 91.3%;
- Silver recovery into the gravity concentrate of 16% with 84% of the remaining silver recovered into the flotation concentrate;
- A flotation mass pull of 3.5%
- Cyanidation extraction efficiencies of 96% for both the gravity concentrate and the flotation concentrate and 60% extraction from the flotation tailings;
- An overall extraction of 91 to 95%;
- A hardness Axb of 23.2 units which classifies the ore as very hard;
- A ball mill work index of 23.1 kWh/t which confirms the hardness classification;
- Concentrate and tailings leach residence times of 48 hrs and 36 hrs respectively;
- Carbon elution rate of 162 tpd (Split AARL elution circuit)
- CIP slurry residence time of 9.6 hrs (8 tanks of 1.2 hrs each)

1.11 Project Infrastructure

1.1.1 EXISTING INFRASTRUCTURE

The Zgounder mine has been in production since 2019 (for the flotation plant) and has all the necessary infrastructure required to support the current mining operation. This includes, but is not limited to laboratory, fuel storage, offices, warehouse and storage, 700 t/d processing capacity (flotation and cyanidation plants combined), camp, underground mine and related infrastructure, waste stockpiles and TSF capacity for 5 years production:

1.1.2 NEW INFRASTRUCTURE FOR ZGOUNDER EXPANSION

The new process plant #3 facility has nine (9) areas: crushing, grinding, flotation, intensive leach reactor, counter-current decantation, cyanidation, ADR process, Merrill-Crowe.

The other major facilities and services outside the new process plant, which are included as part of the expansion Project, include: new electrical line and substation, new emergency power facilities, additional fuel storage tanks, new open pit, including waste storage stockpiles, and new automation and telecommunication system.

1.11.1 SURFACE WATER MANAGEMENT

Englobe was retained by Aya Gold & Silver (Aya) to complete the global water balance for the Zgounder mine expansion project as part of the Zgounder feasibility study. The objectives were to validate the hydraulic conditions to supply the new processing plant with water and to size the new water management infrastructure for the mine expansion. During the development of this study, one of the requirements was to develop a strategy that did not include any additional freshwater wells. No site visit was conducted due to COVID-19 restrictions.

Therefore, Englobe developed a water management strategy and a water balance model to accommodate those challenges and requirements. The water management strategy utilizes the large catchment areas of the project site and existing and proposed infrastructure to collect and convey fresh water, contact water and non-contact water to the TSF Site C storage basin. Englobe proposes a flexible system that can adapt to variable precipitation conditions by selectively collecting and discharging water on site, using a system of one large water reservoir, two (2) new collection ponds, new and existing diversion ditches, and pumping stations.

Based on obtained results, Englobe recommends to maintain a minimal available water volume of 150,000 m^3 in the TSF Site C for dry year conditions, and a maximal additional free water storage capacity in the new TSF Site C of 470,000 m^3 for wet year water storage. As the maximal free water storage capacity is significant, the volume could be reduced over time following more precise water input modelling. With the proposed storage capacity, the site water management system would then be able to provide an adequate supply of process water to the plant. The water treatment plant does not seem to be required under the analyzed conditions.

1.1.3 NEW TSF

For the purposes of this FS, using the current information concerning the characteristics of the construction materials and the foundation, the TSF design was carried out with the aim of defining a budget estimate for the needs of the FS and was designed in three (3) phases for a total capacity of 10 years of expanded capacity beyond the existing TSF capacity of 5 years

- Phase 1: Volume stored: 2.48 Mm^3, 2.85 years;
- Phase 2: Volume stored (Phase 1 + Phase 2): 4.84 Mm^3, 7 years;
- Phase 3 Volume stored: 2.66 Mm^3, 3 years.

1.12 Market Studies and Contracts

The end product that is planned to be marketed from the Zgounder Expansion plant is in the form of silver doré bars (silver ingots). The silver ingots produced at Zgounder will be of high purity, typically in excess of 98% of Ag content by weight. The silver doré bars are delivered to refineries where they will be refined to commercially marketable 99.9% pure silver bars.

Silver is considered a global liquid commodity, and is predominantly traded on the London Bullion Market Association (LBMA) and COMEX in New York.

A recommendation as to acceptable consensus pricing is put forward to the company executives, and a decision is made to set the metal price guidance for Mineral Resource and Mineral Reserve estimates. This guidance is updated at least annually, or on an as-required basis.

Metal prices used for the December 2021 Mineral Resource Estimate (P&E Report) and for the Mineral Reserves Estimate as part of this Technical Report are listed in the Table 1.4.

For the economic analysis of the Project, a silver price of $22.00/oz was used.

Table 1.4 – Metal Prices Used for the Mineral Resources and Mineral Reserves Estimates

Metal	Unit	Mineral Resource Estimate	Mineral Reserves Estimate
Silver	$US/oz	22.50	20.00

1.13 Environmental Studies, Permitting and Social or Community Impact

The first Environmental Impact Assessment (EIA) study of the Zgounder mine was prepared in 2013 by Hydraumet, Morocco. Subsequently, operating permit No. 2306, which included exploration permit, surface rights, access to property and any type of mining operations, was issued to Maya Gold and Silver Inc. by ONHYM. On August 15, 2014, the operation of the Zgounder mine by ZMSM obtained its environmental acceptability from the prefecture of Agadir Ida-Outanane. An environmental monitoring program was developed by ENGITECH/TEVARI in 2014 and implemented in 2015.

In December 2021, NOVEC submitted a new Environmental and Social Impact Assessment (ESIA) as part of the Zgounder Silver Mine Expansion Project. This expansion project includes an open pit mine, a waste dump, a new 2,000 t/d concentrator and a new tailings impoundment. The International Finance Corporation's Performance Standards were applied when defining the scope and terms of reference of this new ESIA.

Under the current regulatory framework, no new permits are required for the Zgounder Expansion Project.

1.14 Capital and Operating Cost Estimates

1.14.1 Capital Cost Estimate (Capex)

The initial Capex estimate for the Zgounder Expansion Project includes all the Projects' direct and indirect costs to be expended during the implementation of the Project, inclusive of an upcoming basic engineering as well as the execution phase, complete with detailed engineering. The Capex is deemed to cover the period starting at the approval by Aya of this Report and finishing after commissioning is achieved.

All capital costs are expressed in United States Dollars (USD). Currency exchange rates are dated Q4 2021. Inflation and risk are not included in the estimate.

The initial Capex for the Zgounder Expansion Project is estimated at $ 139.4M USD. Details are presented in Table 1.5.

Table 1.5 – Initial Capex Summary by Major Area (USD)

WBS	Major Area	Total Cost ($ USD)
1000	Mining – UG equipment & infrastructures	9,713,352
2000	Mining – Open pit pre-stripping	2,943,170
4000	New Processing Plant	60,770,216
5000	Power Generation and Distribution	8,643,184
6000	TSF	5,536,670
	Sub-Total Direct Costs	*87,606,593*
9000	Indirect Costs	30,808,881
10000	Owner's Costs	5,386,250
20000	Contingency	15,621,809
	Grand Total	**139,423,533**

1.14.2 Operating Cost Estimate (Opex)

The Opex is presented in United States Dollars (USD) and uses prices obtained in Q4 2021. DRA developed these operating costs in conjunction with Aya.

The following are examples of cost items specifically excluded from the Opex:

- Value Added Tax (VAT);
- Project financing and interest charges.

Table 1.6 presents the operating costs summary by major Project area over the LOM.

The average operating cost, including transport is $55.42/t of ore.

Table 1.6 – Operating Costs Summary

Description	LOM Cost	Cost ($/t)[1]	Cost ($/oz)	Total Cost (%)
Mining – Underground	226,634,161	26.38	3.50	47.6
Mining – Open Pit Ore and Historical Tailings to ROM Pad	24,777,360	2.88	0.38	5.2
Process (average)	163,450,368	19.03	2.53	34.4
General and Administration	50,264,337	5.85	0.78	10.6
ESG	10,972,885	1.28	0.17	2.3
Total	**476,099,911**	**55.42**	**7.36**	**100.0**

1. Figures may not add due to rounding.

1.15 Economic Analysis

A financial model has been developed to include the relevant study results in order to estimate and evaluate project cash flows and economic viability. The evaluation method takes into account mill feed tonnages and grades (including dilution) for the ore and the associated recoveries, silver price, operating costs, transport and refining charges, government royalties and capital expenditures (both initial and sustaining). The project has been evaluated on a 100% ownership basis, with no debt financing.

The economic analysis demonstrates that the project has positive economics under the assumptions used. On a before tax basis, the project has a 5% NPV of $471 M and an IRR of 57%. On an after-tax basis, the project has a 5% NPV of $373 M and an IRR of 48%. Total undiscounted cash flow over the life of mine equals $522 M and payback period is estimated at 1.7 years post expansion.

The Project also demonstrates a favourable cost structure with an all-in sustaining cost of $9.58 per ounce of silver produced.

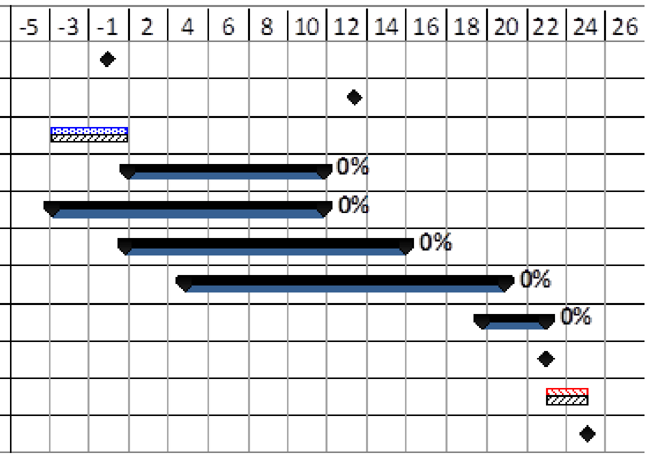
1.16 Other Relevant Information

1.16.1 PROJECT EXECUTION SCHEDULE

A milestone schedule has been developed for the Zgounder Expansion Project covering the main activities of studies, permitting, engineering, procurement, construction, commissioning and ramp-up. The Level-one schedule in presented in Figure 1.1.

Figure 1.1 – Project Schedule

Zgounder Expansion Project	Start	Finish
Financing In Place	M00	M00
Mills Delivery On Site	M+12	M+12
FEED Stage	M-03	M+01
Detail Design & Drafting	M+01	M+10
Procurement	M-03	M+10
Fabrication and Deliveries	M+01	M+14
Construction	M+04	M+19
Commissioning	M+19	M+21
Start Hot Commissioning	M+21	M+21
Production ramp up	M+22	M+23
Start Commercial Production	M+24	M+24



1.16.1.1 Schedule Assumptions

The Project milestone schedule has been developed based on the following assumptions:

- Project assumes EPCM / EPC construction strategy
- Geotechnical studies and survey reports (in their final version) are received by the EPCM / EPC contractor (s) before the start of basic engineering;
- Hydrogeological surveys and reports (in their final version) are received by the EPCM / EPC contractor (s) before the start of basic engineering and are favourable to the Project;
- All permits required will be awarded before the beginning of construction;
- Design criteria, process flowsheet and scope of work will be frozen and agreed upon by all stakeholders before the start of basic engineering;
- Qualified resources will be available for the EPCM / EPC contractor (s);
- Qualified construction workers will be available at the time of construction.

1.16.2 Risk Review

A risk review meeting was held in March 2022 between DRA and Aya personnel as part of the Feasibility Study. The risks covered geology, mining, mineral processing, tailings, environmental, social and permitting project Capex, Opex, construction, and general risks.

A total of 58 risks were identified by the group. Of these, five (5) were resolved during the meeting or judged as obsolete, leaving 53 active risks. From this list, four (4) were classified as High risk, 24 were classified as medium risk, and 25 were classified as low risk in the pre-mitigation rating. Post mitigation, 53 out of 54 risks were downgraded to low risk, and one remained as a medium risk. No risks remained at a high rating after mitigation.

In order to continue to mitigate project risks, it is recommended that sufficient risk management effort be included in the next phase of the Project (EPCM). Specifically, it is recommended that (a) a second risk review be held at the onset of the next phase to continue to identify and detail any special scope required early-on, and (b) particular emphasis be placed on conducting a full HAZOP review as per standard engineering practices.

1.17 Interpretation and Conclusions

1.17.1 Geology and Mineral Resources

The mineral exploration results for the Zgounder Silver Property have been very positive with a significant upgrade in the Mineral Resources since March 2021. The Property shows further upside potential and additional exploration is warranted.

At a cut-off grade of 65 g/t Ag, pit-constrained updated Measured and Indicated Mineral Resource totals 514 kt grading 357 g/t Ag for 5.9 Moz Ag. At a cut-off grade of 75 g/t Ag, out-of-pit updated Measured and Indicated Mineral Resource totals 9.0 Mt grading 309 g/t Ag for 89.3 Moz Ag, and updated Inferred Mineral Resource totals 542 kt grading 367 g/t Ag for 6.4 Moz Ag. At a cut-off grade of 50 g/t Ag, tailings updated Indicated Mineral Resource totals 272 kt grading 94 g/t Ag for 817 koz Ag. The effective date of this updated Mineral Resource Estimate is December 13, 2021.

The updated Measured and Indicated Mineral Resources for Zgounder totalling 9.8 Mt averaging 306 g/t Ag for 96.1 Moz Ag represent an increase of 116% compared to the previous (March 2021) Measured and Indicated Mineral Resources of 44.4 Moz Ag. The updated Inferred Mineral Resources for Zgounder totalling 542 kt averaging 367 g/t Ag for 6.4 Moz Ag represents an increase of 1,519% compared to the previous (March 2021) Inferred Mineral Resources of 395 koz Ag.

The Updated Mineral Resource Estimate incorporates drilling carried out on Zgounder from February 2018 to September 2021. The Mineral Resource database update consists of 516 drill holes (surface and underground combined) for 41,932 m completed at Zgounder. The drilling successfully extended the east-west strike length of the Zgounder silver mineralization from 775 m to 1,100 m and at depth in successfully intersecting silver mineralization in the Exploration Target established by P&E (2021).

Three-dimensional block models were created for the Zgounder Deposit and for the historical tailings located a few hundred metres northwest of the mine site. A geological rock code system was introduced and assigned to the various lithological units and mineralized domains. Continuity directions were assessed based on the orientation of the domains and the spatial distribution of silver. Separate variograms were generated for 1.2 m down-hole silver composites within each domain. Mineralization modelling, grade estimation and Mineral Resource reporting were conducted using GemcomTM, LeapfrogTM, Snowden SupervisorTM and NPV SchedulerTM software. Ordinary kriging was used for grade estimation into 2.0 m x 2.0 m x 2.0 m model blocks.

Mineral Resources have been estimated using Ordinary Kriging of capped composites. Potentially economic mineralization has been identified by categorizing blocks based on a Nearest Neighbor ("NN") assignment. Blocks assigned a NN grade of 40 g/t Ag or higher are categorized as potentially economic mineralisation, whereas blocks assigned a NN grade less than 40 g/t Ag are categorised as waste. The Mineral Resource Estimates have classified into Measured, Indicated and Inferred based on a series of expanding search ellipsoids.

1.17.2 Process

Metallurgical testing confirmed that the ore is amenable to the flowsheet consisting of crushing, grinding, gravity concentration, flotation, cyanidation of both concentrates and the flotation tailings and carbon adsorption followed by silver recovery from pregnant solutions through zinc cementation.

1.17.3 Mineral Reserves and Mining Methods

The Mineral Reserves are estimated at 8.9 Mt of ore grading 251 g/t Ag, combining the open pit, historical tailings reclamation and underground portions of the mine. Further Mineral Reserves could be defined by reclamation the pillar located between the bottom of the open pit and the top of the underground at the end of the mine life.

The Report for the Zgounder Project is based on an 11-year mine life combining the open pit mining, historical tailings reclamation, and underground mining. The mine will operate year-round, seven (7) days per week, twenty-four (24) hours per day (two (2) 12-hour shifts). Two (2) weeks of adverse weather conditions per year are considered in the mine plan.

Approximately 71% of the ore will be coming from the underground portion of the operation while 29% will come from the open pit and historical tailings reclamation. Over the LOM, 8.6 Mt of ore will be mined or reclaimed, of which 92% will be sent directly to the crusher and mill and 8% will be sent to an ore stockpile to be rehandled later. A total of 27.6 Mt of waste material will be mined to access the ore.

1.17.4 Recovery Methods

The flowsheet is complex and provides only a small improvement in recovery above that of a much simpler whole-ore cyanidation flowsheet. A major benefit of the flotation circuit is that the footprint of the CCD thickener train is much smaller than that of a comparative whole-ore cyanidation plant. Another benefit is the opportunity to oxidize cyanide consumers in the flotation concentrate in a more intensive pre-oxidation step. These benefits need to be traded-off against the complexity of the flotation circuit and the downside of being forced to destroy cyanide in the reclaim solution prior to its introduction into the flotation circuit.

1.17.5 Project Infrastructure

1.17.5.1 Tailings Storage Facility (TSF)

Given the production volumes contemplated at this stage of the Project, and the estimated duration of the mine, the topographic and hydrographic conditions of Sites C and A have shown that the tailings dam can be built in three (3) phases, allowing for a distribution of the capital investments over the entire period of the operation thus reducing the initial Capex.

The first two (2) phases will be constructed at Site C, and will support approximately 7 years of the LOM. The last phase will be constructed at Site A, for an initial period of 3 years of operation. It is important to note that Site A TSF could be extended in the future if new mineralised zones are discovered and the mine life is extended.

1.17.5.2 Surface Water Management

The water management strategy and a water balance model were developed to account for dry, average and wet years. The water management strategy utilizes the large catchment areas of the project site and existing and proposed infrastructure to collect and convey fresh water, contact water and non-contact water to the TSF Site C storage basin. Engobe proposes a flexible system that can adapt to variable precipitation conditions by selectively collecting and discharging water on-site, using a system of one (1) large water reservoir, two (2) new collection ponds, new and existing diversion ditches, and pumping stations.

Based on obtained results, Englobe concludes that the site can manage the water supply to the processing plant during variable conditions by operating a water reservoir with a maximum initial capacity of 620,000 m³ and maintain the minimum water volume at around 150,000 m³. Those volumes can be refined to suit the staged TSF pond construction sequence as more climatic and operational information become available. The water treatment plant does not seem to be required under the analyzed conditions.

For the water management strategy to be effective, the site must be appropriately instrumented and controlled. All data must be stored and analysed to make valuable conclusions about the interaction between the mine site operation, the local climate and environment. The proposed system, comprised of physical infrastructure and instrumentation, can be adapted to the variable annual and seasonal precipitation, changing climate and operational requirements, and ensure that all available water resources are accounted for and effectively managed.

1.17.6 Environmental Studies, Permitting and Social or Community Impact

The Zgounder Mine site has a very long mining history and can be described as a "brownfield site". The surface mineralized showings were exploited as far back as the 10th century. More recent mining and processing activities took place between the 1950s and 1970s, and later from the 1980s to 1990s with a cyanide-leaching process plant. There is evidence of heavy metal leaching and cyanide contamination in surface water and potentially groundwater water and soils on site. Since 2014, Aya has undertaken civil work to reduce the impacts and risks, targeting especially the tailings impoundments' containment, stability, and revegetation.

The ESIA identifies the relevant risks and proposes mitigation measures. The environmental management plan and monitoring program will ensure that the mining activities comply with their permits and the applicable laws and regulations for mining operations in Morocco.

1.18 Recommendations

1.18.1 Geology and Mineral Resources

It is recommended that issues noted in the database be corrected, and that the methodology implemented for the Mineral Resource Estimate as described be continuously reconciled and validated against actual production results.

An exploration budget of US$6.6 M is recommended for Zgounder in 2022, for a total of 46,000 m of step-out and infill drilling on the Zgounder Mine Property. The drill program is to focus on:

1. Expanding the Mineral Resources along strike, particularly to the east, and at depth to the granite contact; and
2. Advance Inferred Mineral Resources to Indicated Mineral Resources to support Mineral Reserve Estimates.

1.18.2 Mineral Reserves and Mining Methods

In the next phase of the Project, DRA recommends looking at the possibility of mining the crown pillar at the end of the mine life. DRA also recommends looking at the possibility of in-pit waste stockpiling in the mined-out portions of the open pit to minimise the ex-pit waste stockpile size and its related environmental footprint.

The COG was estimated according to the available information at the time of this study and should be reviewed and optimised if the Project has more updated circumstances or cost rates to improve the Project's profitability and Mineral Reserves.

Improvement opportunities still remain and can be included in the next phase. Special open pit and underground sequencing and redesigning relying on new geological targets currently under development can be undertaken:

Mine design should be reviewed and redesigned taking into account new geological targets and Mineral Resources that are being currently developed. Some of the newly identified areas are close to the main decline and could bring in ore production sooner than the Main deposit.

Due to the complexity of the geometry of the deposit, definition drilling should be done planned during detail engineering, and the mining method selection should be revisited.

Drill and blast parameters in the study were designed according to typical stope geometry in each area. During detail engineering and in the operations phase, determination of optimal burden and spacing should be reviewed stope by stope to optimise drilling and blasting costs.

Battery operated mining fleets are currently being developed by the major mine equipment manufacturers and implemented in more and more operations to reduce mine ventilation needs. It is recommended that this technology be considered when replacing the mine fleet in a few years and that this new technology has proven itself at other operation.

For the CRF, DRA recommends that UCS testing be performed for CRF to gain understanding of strength development with regards to the target strength of 235 kPa.

RockEng geotechnical review of the current mine design has identified the following recommendations to be explored further during later phases of the Project.

- Level stacking with small level spacing introduces pillar stability risk between level, particularly in intersections and wide span excavations. It is recommended that level layouts be adjusted to avoid direct stacking of lateral development level-to-level.
- Cross-cut stacking in long-hole stopes will introduce pillar stability risk for drill-horizons during overhand advance. This may be de-risked by implementing double-lift long-hole

stopes (i.e., effectively skipping every other level for long-hole mining blocks). It is recommended that opportunities to achieve double-lift long-hole stopes be investigated further.

- Backfill strength requirements are based on 25 m level spacing. For shorter vertical exposures there may be opportunity to reduce backfill strength. It is recommended that backfill strengths be reviewed as stope designs and sequencing plans advance.

1.18.3 Process

While the selected flowsheet does improve overall recoveries marginally (~2%) it is complex and somewhat costly and it requires detoxification of reclaim water in order to minimise the risk of cyanide affecting the performance of the flotation section. The value of a simpler flowsheet needs to be revisited.

The concept, assumed during the preparation of the FS, was that peroxide would be injected into the reclaim water line at the tailings dam and that the residence time between this point and the discharge into the water tanks at the plant would be sufficient to destroy most of the free and WAD cyanide. The tests conducted were performed for 60 minutes which likely exceed the residence time in the reclaim pipeline. The requirement for a small reactor tank to add residence time for this reaction would need to be evaluated in future phases of the Project.

1.18.4 Recovery Methods

It is recommended to perform additional confirmatory testwork on the whole-ore cyanidation and CCD flowsheet using several cyanide concentrations for leaching and on variable head grade samples. This would allow comparison of this simplified flowsheet against the more complex flowsheet that formed the basis of this FS.

1.18.5 Project Infrastructure

1.18.5.1 Tailings Storage Facility

Currently, testwork is ongoing at the geotechnical laboratory and therefore, the conclusions of this testwork are not included in this FS. It is recommended to complete the stability analysis of the Site C TSF design including the results of the ongoing geotechnical testwork and depending on the results, a modification of the TSF Site C could be considered in order to increase its storage capacity.

Also, additional design work is recommended to consider the borrow materials for the TSF dam construction to be excavated from inside the footprint of the TSF Site C, and hence increase its storage capacity.

1.18.5.2 *Surface Water Balance and Infrastructure*

The proposed water management strategy and resulting water balance consider several assumptions that need to be refined before the next phase of the Project. The following list presents the main assumptions and limitations that should be refined at the next design phase:

- A monthly water balance should be developed;
- The current and future TSFs should have a tailings and water management manual (OMS manual) where the water management principles are presented and integrated with the tailings management principles;
- The proposed water management strategy requires actions and procedures based on a time-sensitive understanding of the current site water storage conditions and forecasting of short-term environmental conditions. A flexible water management tool should be developed and implemented;
- The current and future operations should monitor the TSFs and document input and output parameters to better refine the water balance model;
- The water balance model in this study relies on weather data that is not site-specific. The site weather station should be used at the next phase of the Project to better define the site-specific weather data; and,
- Mine dewatering is an important input to the global water balance. A more precise dewatering plan would increase the precision of the proposed water balance.
- Incorporate the new waste dump water management and infrastructure into the overall site water balance.

Finally, it is recommended to execute the geotechnical, hydraulic, and hydrological studies required to move to the detailed engineering phase.

1.18.6 Environmental Studies, Permitting and Social or Community Impact

- Develop a water management program, including hydrological and hydrogeological characterisations, to ensure the project's design compliance with commitments, permits and legislation requirements.
- With the increase of the project footprint try to reduce the number of effluents with the collection ditches. This will help defining the required water treatment if required.

- Environmental monitoring program should clearly identify sampling locations (water, soil, air) and coordinates and maintain the sampling labels for traceability.
- More sampling locations of soil samples should be added, near petroleum storage tanks.
- Sediments or alluvion along oueds should also be characterised.
- Continue geochemical characterisation on waste rock and tailings to verify the potential of acid mine drainage and metal leaching, as recommended in the report from Lamont inc, 2021.
- Continue to investigate water and soil contamination upstream, at the mine site and downstream, as recommended in the ESIA.
- Revise and increase the frequency of the mine effluents monitoring for a more efficient control within the operations. In Canada for example, monitoring is performed on a weekly basis and reported on a monthly basis.
- Introduce treatment at the source with the addition of a cyanide destruction process (example SO_2 air);
- Continue to treat tailings to increase pH and help the precipitate heavy metals and arsenic within the tailings.
- Ensure that the emergency responses plan is known and tested and the intervention material is available.
- Perform root cause analysis for recurrent accidental spills in order to identify appropriate solutions.
- Consider in the closure plan to fill the open pit with waste rock from the waste dump to reduce project footprint and remediate old tailings pond after the processing of tailings.
- Ensure a sufficient number of environmental staff in order to meet regulatory performance.

2 INTRODUCTION

2.1 Terms of Reference and Scope of Study

The purpose of the Feasibility Study (FS) is to review and define the optimum configuration for the mine and processing arrangement for the Zgounder Expansion Project to 2,700 t/d based on the latest available test work and Mineral Resource Estimates (MRE). Following completion of the engineering deliverables, a capital and operating cost estimates was prepared as well as a subsequent economic evaluation to determine the Project's viability. The definitions are followed by estimation and confirmation of project economics.

The following Technical Report (Report) summarises the results of the FS for the complete Project. DRA is the overall lead consultant in collaborating with other consultants (for scope that outside of DRA's mandate) and compiled a final report that is inclusive of the work and deliverables performed by such consultants.

DRA has focussed on the following key issues that require addressing in the Technical Report:

- Develop open pit and underground mining methods, designs and mine plans;
- Estimate of Mineral Reserves;
- Process test work review and validation of test reports;
- Design process plant based on following work and test work;
- Develop of new process plane # 3 on-site infrastructure;
- Develop Capex and Opex Estimates and Financial modelling;
- Compile work carried out by other consultants including geology/resources, geotechnical studies, tailings storage facility, hydrogeology, and water management.

2.2 Source of Information

DRA is the overall lead consultant in collaborating with other consultants (for scope that fall outside of DRA's responsibility) and compile a final report that is inclusive of the work and deliverables performed by such consultants.

DRA Global Limited (DRA) has provided engineering and integration services for all aspects of the NI 43-101 Technical Report Feasibility Study on the Zgounder Expansion Project (Project) with the participation of other consulting companies. The Technical Report includes the Resource Estimation (by P&E Mining Consultants Inc. ("P&E")), Geotechnical design of the underground mine (RockEng Inc. (RockEng)), Tailing Storage Facility Consultant (GCIM), Water Management, (Englobe).

This Report is based in part on technical reports, maps, published government reports, and public information, as listed in Section 27 "References" of this Report. Sections from reports authored by other consultants listed above may have been directly quoted/extracted, translated and/or summarised in this Report, and are so indicated, where appropriate.

The information, conclusions and opinions contained herein are based on:

- Review of the available literature and reports;
- Zgounder area visits;
- Meetings with Aya personnel.

2.3 Qualified Persons

The responsibilities for the preparation of the different sections of this Report are shown in Table 2.1.

Table 2.1 – Qualified Persons and their Respective Sections of Responsibilities

Section	Title of Section	Qualified Person	Company
1	Summary	Daniel M. Gagnon and related QPs	ALL
2	Introduction	Daniel M. Gagnon	DRA
3	Reliance on Other Experts	Daniel M. Gagnon	DRA
4	Property Description and Location	William Stone	P & E
5	Accessibility, Climate, Local Resources, Infrastructure and Physiography	William Stone	P & E
6	History	William Stone	P & E
7	Geological Setting and Mineralisation	William Stone	P & E
8	Deposit Type	William Stone	P & E
9	Exploration	Antoine Yassa	P & E
10	Drilling	Antoine Yassa	P & E
11	Sample Preparation, Analyses, and Security	Jarita Barry	P & E
12	Data Verification	Antoine Yassa, Jarita Barry	P & E
13	Mineral Processing and Metallurgical Testing	Daniel Morrison	DRA
14	Mineral Resource Estimates	Fred Brown, Eugene Puritch	P & E
15	Mineral Reserve Estimates		
15.3 to 15.4 and 15.6	Open Pit	Daniel M. Gagnon	DRA

Section	Title of Section	Qualified Person	Company
15.1, to 15.2 and 15.5 to 15.6	Underground	André-François Gravel	DRA
16	Mining Methods		
16.1 to 16.3 and 16.7	Open Pit except for 16.2.2	Daniel M. Gagnon	DRA
16.2.2	Geotechnical – OP	Claude Bisaillon	DRA
16.4	Underground except for 16.4.1	André-François Gravel	DRA
16.4.1	Geotechnical – UG	Kathy Kalenchuk	RockEng
16.5	Cement Rock Fill	André-François Gravel	DRA
16.6	Mine Ventilation	Hugo Dello Sbarba	Howden
17	Recovery Methods	Daniel Morrison	DRA
18	Project Infrastructure		DRA
18.1 to 18.4	Project Infrastructure	Daniel M. Gagnon	DRA
18.5	Surface Water	Philippe Rio Roberge	Englobe
18.6	Tailings Management Facility	Richard Barbeau	GCIM
18.7	Water Management		
18.7.1	Context	Richard Barbeau	GCIM
18.7.2 to 18.7.4	Surface water management infrastructures	Philippe Rio Roberge	Englobe
19	Market Studies and Contracts	Daniel M. Gagnon	DRA
20 except for 20.7	Environment Studies, Permitting and Social or Community Impact	Julie Gravel	DRA
20.7	Mine Closure	Daniel M. Gagnon	DRA
21	Capital and Operating Costs	Daniel M. Gagnon	DRA
22	Economic Analysis	Stephen Coates	Normetal Consulting
23	Adjacent Properties	William Stone	P & E
24	Other Relevant Data and Information	Daniel M. Gagnon	DRA
25	Interpretation and Conclusions	Daniel M. Gagnon and related QPs	ALL
26	Recommendations	Daniel M. Gagnon and related QPs	ALL
27	References	Daniel M. Gagnon and related QPs	ALL
28	Abbreviations	Daniel M. Gagnon and related QPs	ALL

2.4 Site Visit

This Section provides details of the personal inspection on the Property by some of the Qualified Persons.

Name	Company	Site Visit (Yes or No)	Date
Antoine Yassa, P. Geo.	P&E	Yes	January 11 to 13, 2021 November 19 to 21, 2021
Daniel M. Gagnon, P. Eng.	DRA	No	
Daniel Morrison, P. Eng.	DRA	No	
Claude Bisaillon, P. Eng.	DRA	Yes	January 18 and 19, 2011
André-François Gravel, P. Eng.	DRA	Yes	May 25 and 26, 2021
Hugo Dello Sbarba, P. Eng.	Holden	No	
Kathy Kalenchuk	RockEng	No	
Julie Gravel, P. Eng.	DRA	No	
Stephen Coates, P. Eng.	Normetal Consulting	No	

2.5 Units and Currency

In this Report, all prices and costs are expressed in United States Dollars ($ US or USD) unless otherwise stated. If other currencies are utilised, their symbols are specified (i.e. Canadian Dollars (CAD), Moroccan Dirhams (MAD), etc.). Quantities are given in the “International System of Units (SI) metric units, the standard Canadian and international practice, including metric tonne (tonne or t) for weight, and metre (m) or kilometre (km) for distance.

Abbreviations used in this Report are listed in Section 28.

3 RELIANCE ON OTHER EXPERTS

The QPs prepared this Report using reports and documents as noted in Section 27. The Authors wish to make clear that they are Qualified Persons only in respect to the areas in this Report identified in their “Certificates of Qualified Persons”, submitted with this Report.

A draft copy of the Report has been reviewed for factual errors by Aya. Any changes made as a result of these reviews did not involve any alteration to the conclusions made. Hence, the statement and opinions expressed in this Document are given in good faith and in the belief that such statements and opinions are neither false nor misleading at the date of this Report.

The Qualified Persons (QP) who prepared this Report relied on information provided by experts who are not QPs. The QPs who authored the sections in this Report believe that it is reasonable to rely on these experts, based on the assumption that the experts have the necessary education, professional designations, and relevant experience on matters relevant to the Technical Report.

The QPs used their experience to determine if the information from previous reports was suitable for inclusion in this Technical Report and adjusted information that required amending. This Report includes technical information, which required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the QPs do not consider them to be material.

DRA has relied upon market studies provided by Aya. Section 19 summarises the key information regarding the silver market overview and outlook. DRA has reviewed the content of the market study presentation and believes that it provides a reasonable overview of the current market as well as projections according to various recognised sources.

DRA has relied on reports and opinions provided by Aya for information in Section 20 pertaining to Environment Studies, Permitting and Social or Community Impact. DRA has reviewed the content of this Section and believes that it provides current and reliable information on environmental, permitting and social or community factors related to the Project.

DRA has relied on Aya for guidance on applicable taxes, royalties, and other government levies or interests, applicable to revenue or income from the Project. An independent or audited verification of applicable taxes was not completed by DRA or another firm.

DRA is relying on the previous NI 43-101 reports and its referenced documents in relation to all pertinent aspects of the Property. The Reader is referred to these data sources, which are outlined in Section 27 of this Report, for further details.

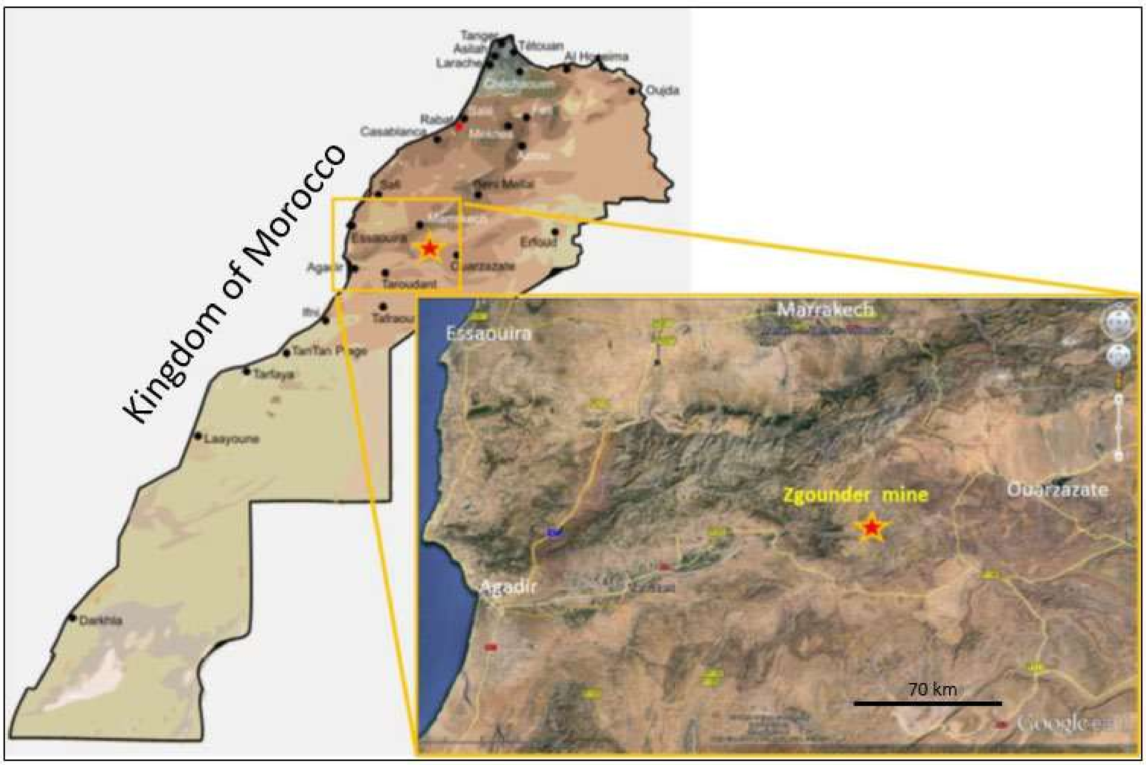
4 PROPERTY DESCRIPTION AND LOCATION

This Section has been summarised and/or largely extracted from the Report available on SEDAR entitled: "Technical Report and Updated Mineral Resource Estimate of the Zgounder Silver Project, Kingdom of Morocco, prepared for Aya Gold & Silver Inc., dated January 28, 2022, prepared by P&E Mining Consultants Inc., Report # 415."

4.1 Location

The Zgounder Property is located approximately 265 km east of the City of Agadir and 220 km west of Ouarzazate (Figure 4.1) in the central part of the Anti-Atlas Mountains, Kingdom of Morocco.

Figure 4.1 – Location of the Zgounder Mine Property Between Agadir and Ouarzazate

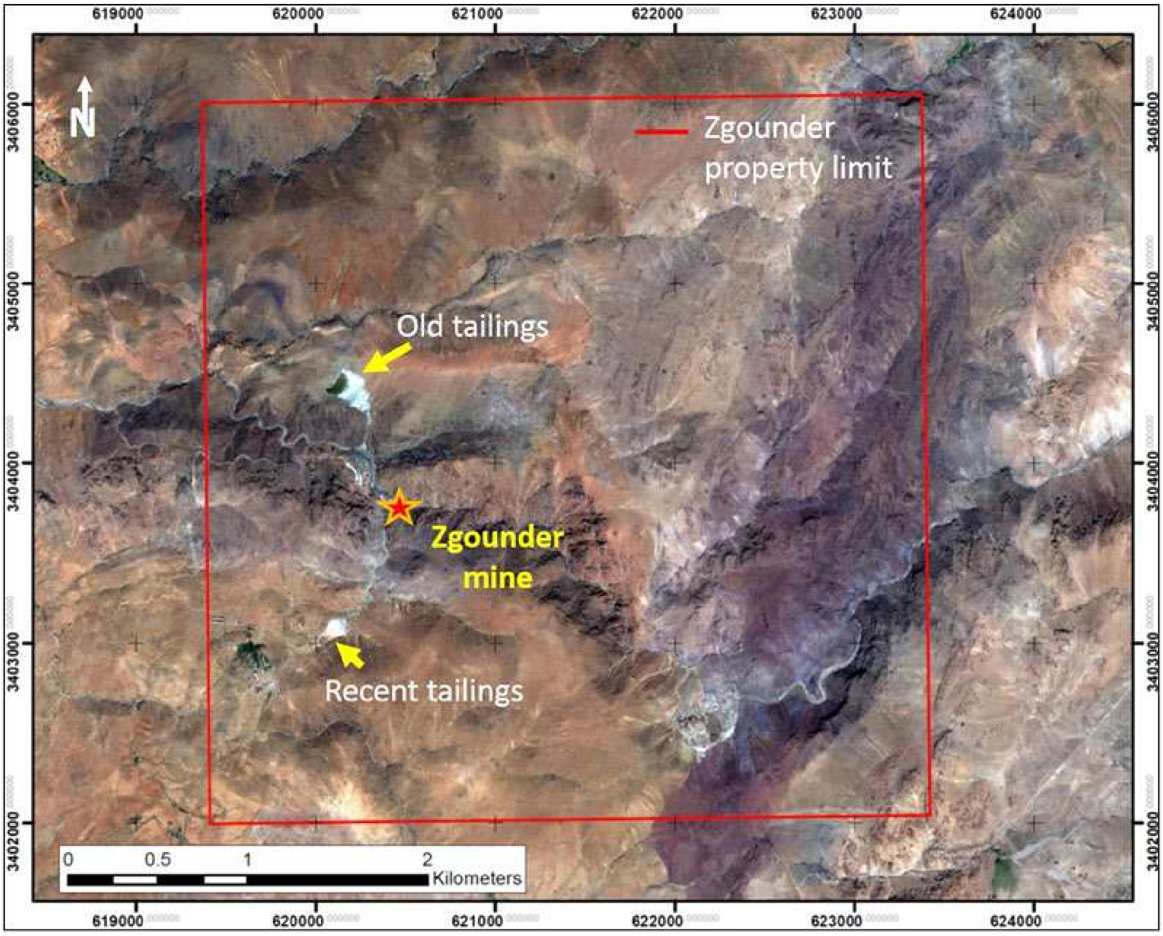


Source: GoldMinds (2018)

4.2 Property Description

The Zgounder Property covers an area of 4 km x 4 km for a total of 16 km^2 (Figure 4.2).

Figure 4.2 – Property Mining Permit Limits of Zgounder Mine



Source: Audet (2021)

The approximate coordinates of the Zgounder Property in the Lambert conformal conic projection are as follows: X 276,000; Y 420,355. Details regarding the coordinate system are:

- Maroc_LCC_zone2
- Authority: Custom
- Projection: Lambert_Conformal_Conic
- False_Easting: 500000.0
- False_Northing: 300000.0
- Central_Meridian: -5.4
- Standard_Parallel_1: 28.102913
- Standard_Parallel_2: 31.288494
- Scale_Factor: 1.0

- Latitude_Of_Origin: 29.7
- Linear Unit: metre (1.0)
- Geographic Coordinate System: GCS_Merchich_Degree
- Angular Unit: Degree (0.0174532925199433)
- Prime Meridian: Greenwich (0.0)
- Datum: D_Merchich
- Spheroid: Clarke_1880_IGN
- Semimajor Axis: 6378249.2
- Semiminor Axis: 6356515.0
- Inverse Flattening: 293.4660212936265

The centre of the Zgounder Property is located at approximate Longitude 7° 44’ 50” W and Latitude 30° 43’ 42” N and, in UTM NAD83 Zone 29N, 6,199,930 m E and 3,400,170 m N.

The elevation ranges from 2,000 metres to 2,180 metres above sea level (MASL).

4.3 Land Tenure

Aya Gold & Silver Inc. was authorized by the Office National des Hydrocarbures et des Mines (“ONHYM”) to prospect and exploit base and precious metals at the Zgounder Mine.

The mining title number 09/2096 and exploitation permit No. 2306 (now exploitation licence 393459) provide surface rights and access to the Property and allow any type of mining.

The exploitation licence No. 393459 covers 16 km^2 and is valid until October 17, 2027 (as of November 15, 2021 Date of Opinion for Zgounder by Dentons Sayarh Menjra Avocats, for Aya Gold & Silver Inc.).

4.4 Land Agreements and Royalties

In 2012, Aya and ONHYM signed an agreement for the development and operation of the Zgounder Mine. As part of the agreement, ONHYM received a 15% free-carried interest in the local company, Zgounder Millennium Silver Mining SA (“ZMSM”) with Aya retaining 85%.

The ONHYM participation becomes a participating interest once the historical resource containing 6.76 Moz is mined. Also, as part of the agreement, ONHYM receives a net-smelter return royalty of 3% on all silver sold on the permit without expiry.

On May 17, 2013, the company entered into a net profit interest agreement with Global Works, Assistance and Trading S.A.R.L whereby the company would pay a royalty equal to 5% of revenues from silver sales less mining and processing costs.

Any dispute related to the validity of the interpretation or execution of the Agreement between ONHYM and Aya shall be resolved amicably by conciliation between the two parties. It is appropriate to note that in case of the failure of this approach, the dispute shall be submitted to the arbitration process of the International Chamber of Commerce in Paris, under the rules of this chamber that apply to Moroccan laws.

4.5 Environmental and Permitting

The necessary authorisation for the use of public water, including the use of spring water or groundwater necessary for the process plant, was obtained from the Water Basin Agency of Souss Massa Draa. Following usage, wastewater is to be discharged into the tailings pond.

4.6 Other Properties of Interest

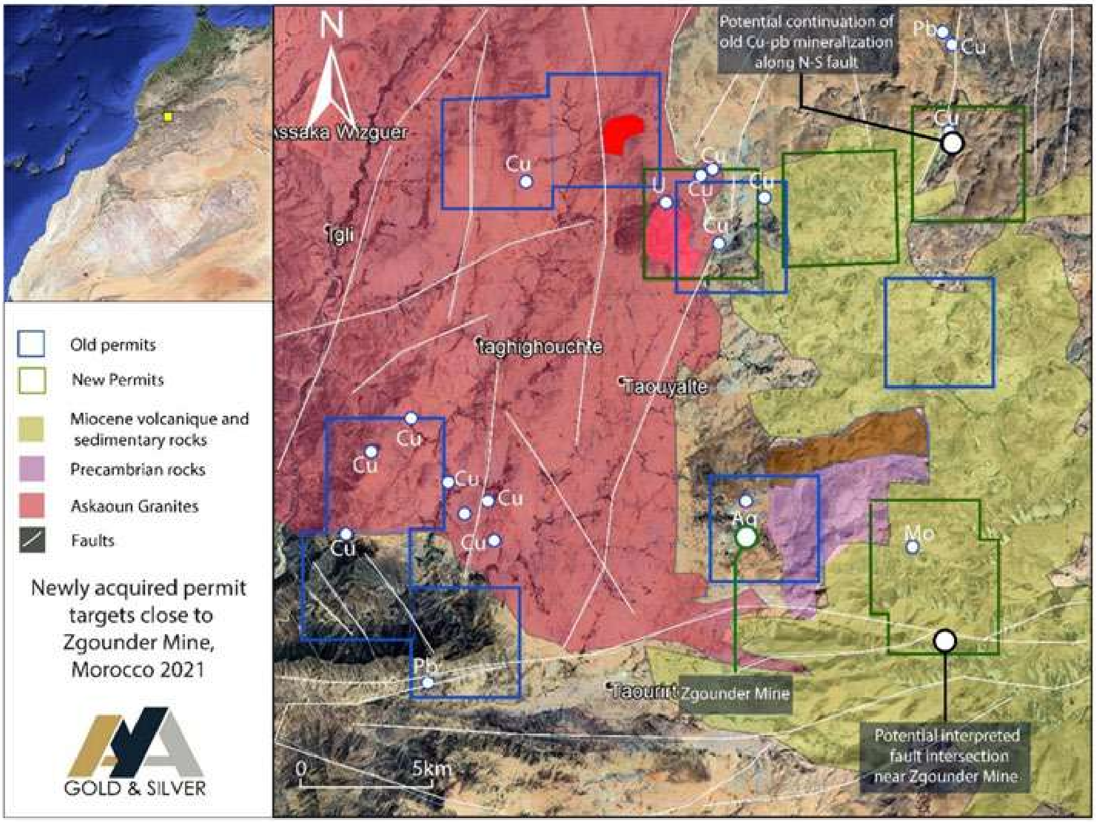
Aya have seven (7) research permits in the Zgounder Mine area. These permits are not contiguous with the Zgounder exploitation licence (Figure 4.3) and collectively cover 96 km^2.

In a press release dated June 28, 2021, Aya announced that five (5) additional permits had been granted in the Zgounder Regional area, increasing its total exploration land holdings from 112 km^2 to 168 km^2. These additional permits are also not contiguous with the Zgounder exploitation licence.

4.7 Other Significant Factors and Risks

To the extent known to the author of this Technical Report section, there are no other significant factors and risks that may affect access, title, or the right or ability to perform work on the Zgounder Property that have not been discussed in this Technical Report.

Figure 4.3 – Other Properties of Interest in the Zgounder Mine Property Area



Source: Aya (2022)

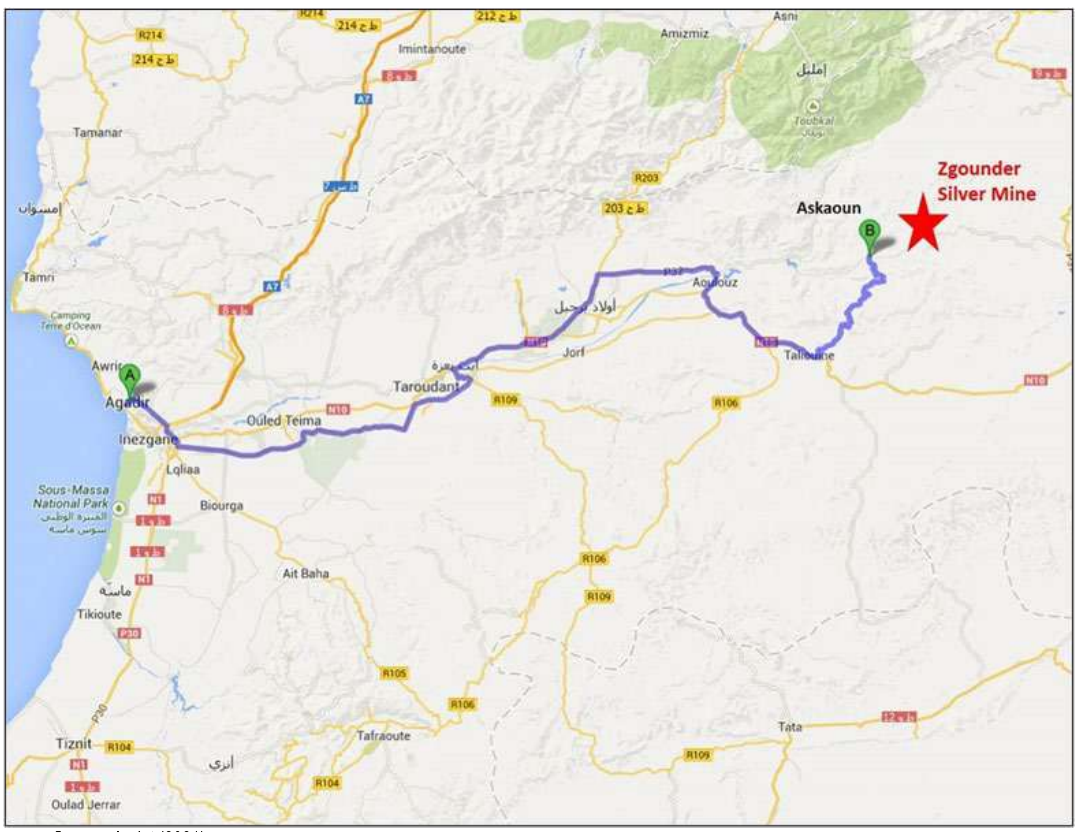
5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

This Section has been summarised and/or largely extracted from the Report available on SEDAR entitled: "Technical Report and Updated Mineral Resource Estimate of the Zgounder Silver Project, Kingdom of Morocco, prepared for Aya Gold & Silver Inc., dated January 28, 2022, prepared by P&E Mining Consultants Inc., Report # 415."

5.1 Accessibility

The Zgounder Mine is located in the province of Taroudant, approximately 265 km east of the port City of Agadir. The mine site is accessible from Agadir by a well-maintained paved road (N10) traversing 216 km east to Taliouine. From Taliouine, a hillside paved road heads north 50 km to the Village of Askaoun. The mine site is accessible from Askaoun by a well maintained 5 km gravel road (Figure 5.1). A 260 km paved road from Marrakesh to Askaoun, via Agouim, is presently under construction.

Figure 5.1 – Access to the Zgounder Mine from Agadir



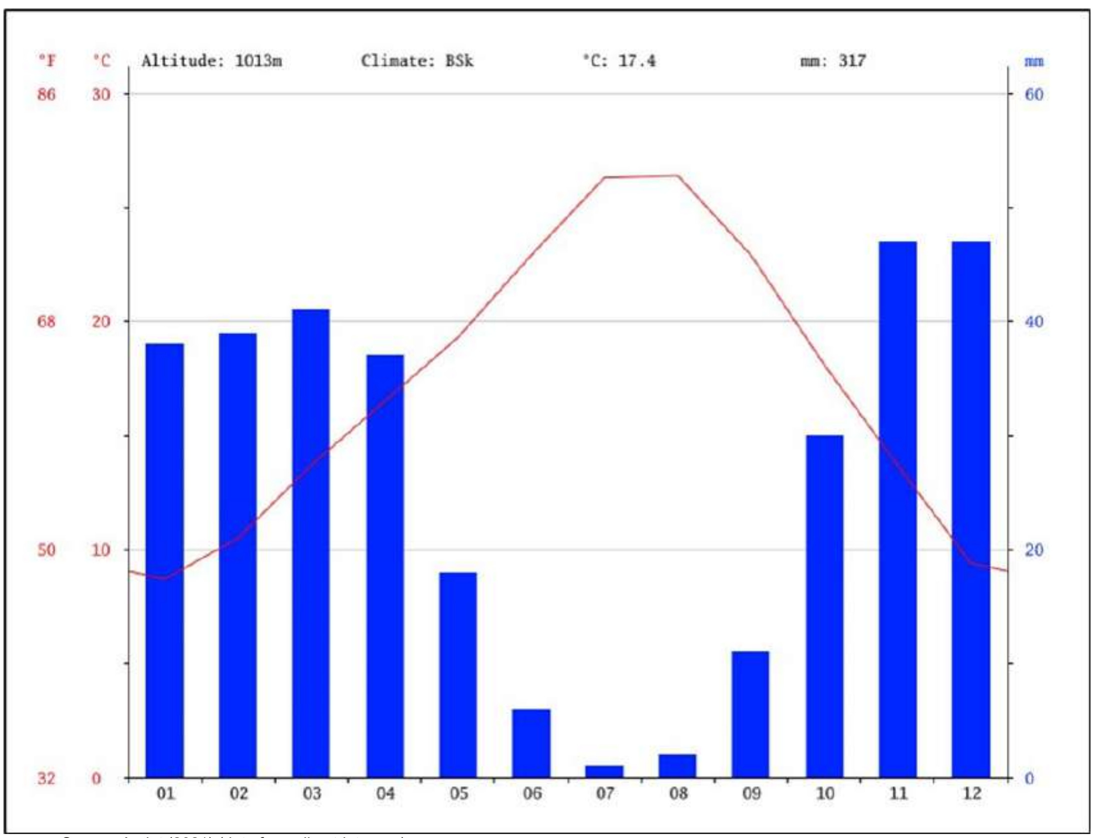
Source: Audet (2021)

5.2 Climate

The Zgounder Mine is located between 1,925 and 2,200 MASL, on the western flank of the Siroua Massif of the Anti-Atlas Mountains. This region is separated from the influence of the Mediterranean climate by the High Atlas Mountains to the north, and therefore, shares the Sahara climate. The area is semi-arid, since it is <50 km from the Sahara Desert. Winters are cool to cold; snowfalls of up to 0.5 m occur during the first quarter of the year in areas above 1,600 MASL. The average annual rainfall is approximately 317 mm; the driest month is July with 1 mm of rain and December has the highest rate of precipitation with an average of 48 mm (Figure 5.2). The average annual temperature is approximately 12.2°C; summers are warm to hot and essentially dry. Seasonal and monthly temperature variations are significant (Figure 5.3).

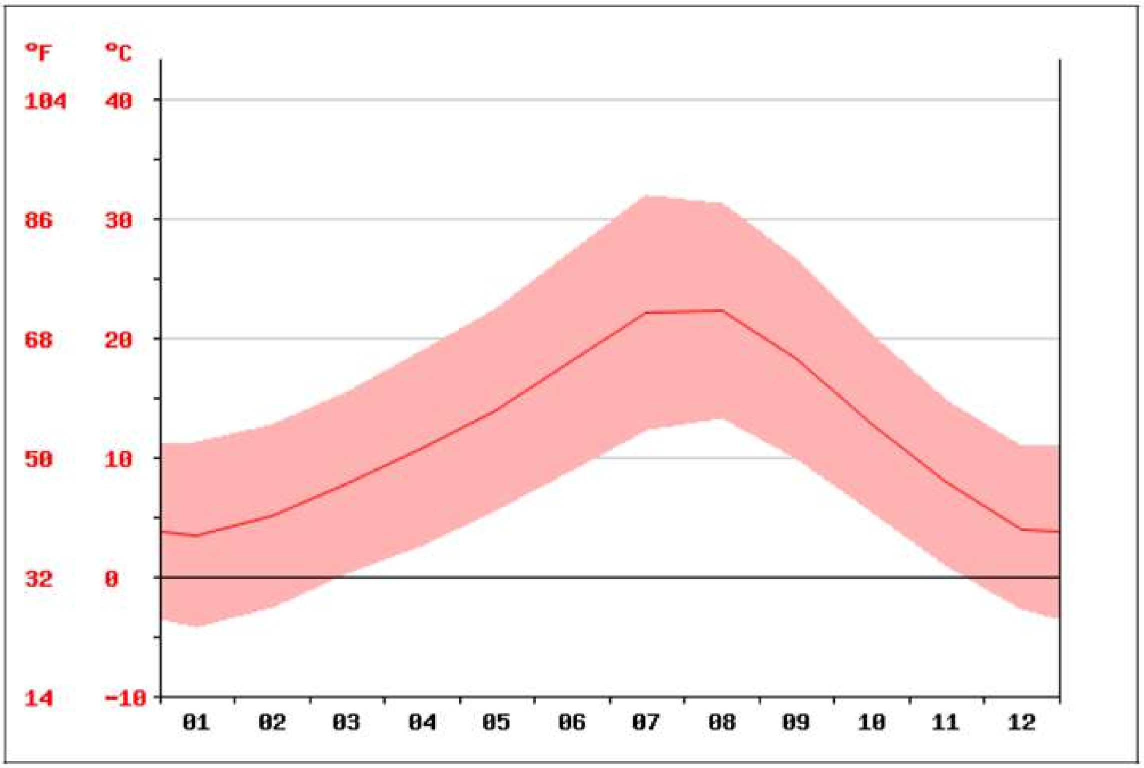
The hottest month of the year is August, with an average temperature of 22.2°C. The coldest month is January, with an average temperature of 3.5°C (Figure 5.4).

Figure 5.2 – Monthly Precipitation and Temperature Averages at Askaoun



Source: Audet (2021) (data from climatdata.org.)

Figure 5.3 – Monthly Temperature Averages at Askaoun Village



Source: GoldMinds (2018) (data from climatdata.org.)

Figure 5.4 – Monthly Precipitation and Temperature Averages at Askaoun

month	1	2	3	4	5	6	7	8	9	10	11	12
mm	60	55	67	60	32	14	3	4	20	45	69	71
°C	3.5	5.1	7.8	10.7	14	18.1	22.1	22.2	18.2	12.8	7.9	4
°C (min)	-4.3	-2.5	0.2	2.6	5.6	8.9	12.3	13.2	9.9	5.5	1	-2.8
°C (max)	11.3	12.8	15.5	18.9	22.5	27.3	31.9	31.2	26.6	20.2	14.8	10.9
°F	38.3	41.2	46	51.3	57.2	64.6	71.8	72	64.8	55	46.2	39.2
°F (min)	24.3	27.5	32.4	36.7	42.1	48	54.1	55.8	49.8	41.9	33.8	27
°F (max)	52.3	55	59.9	66	72.5	81.1	89.4	88.2	79.9	68.4	58.6	51.6

Source: Goldminds (2018) (data from climatdata.org.)

5.3 Local Resources

The main villages are located near rivers (oueds) for water sources and select vegetation (certain cereals, vegetables, and some trees). The local population is exclusively Amazigh (aboriginal population) with a semi-sedentary lifestyle. The economy is principally supported by livestock, agriculture and food trade (saffron, potatoes, dates), and manufacture of traditional carpets. The Siroua Region is a popular tourist destination from spring to late fall.

Basic supplies, such as food and limited accommodation, are available at Askaoun. The larger City of Talioune offers more amenities and services. Special items must be purchased from Agadir.

Mining in Morocco has existed for centuries, and skilled labour is readily available. The mining manpower for Zgounder resides in nearby villages, located from 5 km to 10 km away from the mine site. Skilled labour is available in nearby villages and some inhabitants are previous employees of SOMIL and CMT, former operators of the Zgounder Mine.

5.4 Infrastructure

The Zgounder Mine is easily accessible via a well-maintained gravel road from the Village of Askaoun (Figures 5.5 and 5.6). Mine site facilities include crew houses, offices, drill core shack, a mine portal and trails linking mine entrances (Figure 5.7). Figure 5.8 depicts the existing local infrastructure at Zgounder Mine.

Figure 5.5 – A) Paved Road and B) Gravel Road

A) From Taliouine to Askaoun

B) On-Site from Askaoun

Source: GoldMinds (2018)

Figure 5.6 –Gravel Trail from 2,000 to 2,175 MASL at Zgounder Mine

Source: Audet (2021)

Figure 5.7 – Mine Site Facilities

A) Main Office and Mining Crew Houses

Source: Audet (2021)

B) Mine Portal at 2,000 MASL

Figure 5.8 – Local Infrastructure at Zgounder Mine

Source: GoldMinds (2018)

Figure 5.8 description: view looking west at the Zgounder diesel stand by generators (black arrow) and plant installations showing the conveyor (yellow arrows), storage bins with crushers (white arrows), and two (2) cyanidation lines with counter-current decantation (blue arrows).

During the first years of production, Maya used the trapped water from the 2000 Level in the Mine. Presently, water for the Mine is sourced from the Zgounder River, which flows through the Property (Figure 5.9). The Mine now also provides potable water by gravity from an artesian well five km from the mine office. A small dam helps to maintain an adequate water supply during the dry summer months.

Figure 5.9 – Zgounder River (Yellow Arrow) Flowing through the Property

Source: GoldMinds (2018)

Process plant installations at Zgounder include a crusher, a grinding bay, a cyanide leaching facility with counter-current decantation circuit and a flotation plant (Figures 5.1 to 5.12).

The site also has an analytical geochemical laboratory with equipment for processing and analysis of mine samples.

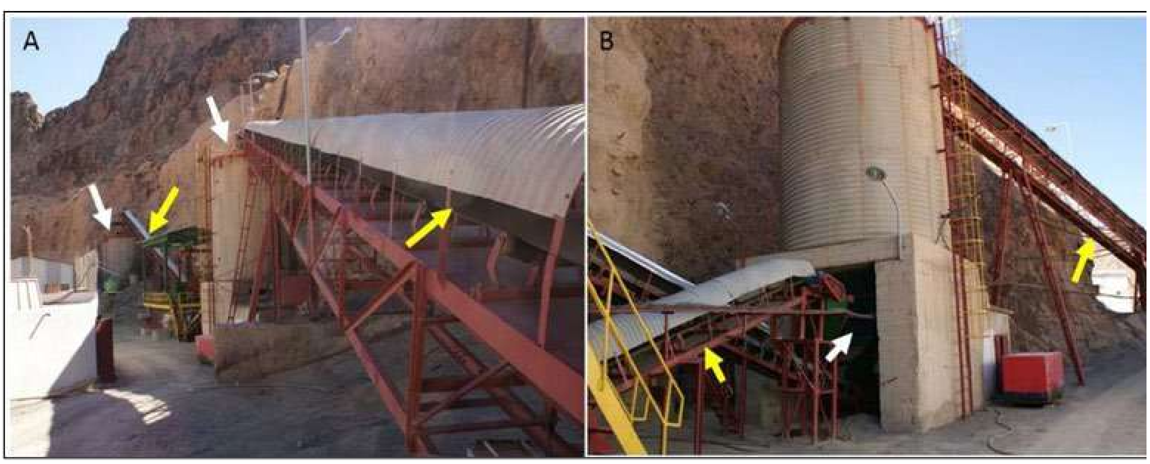
Figure 5.10 – View of the Cyanide Leaching Tanks and Counter-Current Decantation Thickeners

Source: Audet (2021)

Figure 5.11 – Conveyors and Crushers at Zgounder Mine

A) Conveyors (yellow arrows) and coarse storage bins with cone crushers at base (white arrows)

B) Close-up view of the secondary crusher (white arrow)



Source: Audet (2021)

Figure 5.12 – Zgounder Flotation Plant

Source: Audet (2021)

Description: View of the flotation plant above the clouds.

5.5 Physiography

The Jbel Siroua Region (elevation 3,304 MASL, Adrar n'Siroua) is the highest point of the Anti-Atlas, located 14 km southeast of the Zgounder Mine. The Siroua Massif is crossed by numerous rivers, mainly Assif n'Tifnout, Assif n'Oumarigh, and Assif n'Iriri. The Zgounder Deposit is situated on the northern flank of a steep ridge between the Talat N'ouna River to the north and the Zgounder River to the south and west. The ridge rises steeply from an elevation of 2,000 MASL at the Zgounder River to 2,230 MASL at the far eastern end of the Deposit.

The topography at Zgounder is characterised by moderately steep hills with high altitudes, in the range of 2,100 MASL, and low valleys with seasonal flow in rivers (Figure 5.13). Vegetation is limited to minor alpine flowers, mosses, lichens and small evergreen trees. Wheat is cultivated on man-made terraces near the villages. The terraces are irrigated by springs and dams.

Figure 5.13 – Photograph of the Zgounder Mine Area

Source: Audet (2021)

Description: A) a cultivated valley between Taliouine and Askaoun, flanked by moderately steep hills; and B) hills of moderate elevation and sparse vegetation.

6 HISTORY

This Section has been summarised and/or largely extracted from the Report available on SEDAR entitled: "Technical Report and Updated Mineral Resource Estimate of the Zgounder Silver Project, Kingdom of Morocco, prepared for Aya Gold & Silver Inc., dated January 28, 2022, prepared by P&E Mining Consultants Inc., Report # 415."

6.1 Exploration and Mining History

The Zgounder Silver Deposit has a long history of intermittent exploration and mining activities from ancient times to present day.

6.1.1 Ancient Works

The Zgounder Silver Deposit was first exploited between the 10th and 12th Century, mainly in exposed shallow oxidized zones with native silver stringers hosted within E to W, N to S,

NW and NE-trending veins. Excavation scars from those old mining operations can exceed 60 m in depth (Figure 6.1). Evidence of those ancient operations is found locally (Figure 6.2) and many of the excavation sites have been mapped.

Figure 6.1 – Excavation Scars from Ancient Workings at Zgounder

Source: GoldMinds, 2018

Figure 6.2 – Evidence of Ancient Operations

Source: GoldMinds, 2018

Description: A) a granite wheel used in the medieval period to reduce the size of extracted mineralised rocks; and B) slag (the remainders of molten metal) found at the surface, close to the entrance of 2100 Level at the Zgounder Mine.

6.1.2 SNAM-BRGM PERIOD (1950 TO 1979)

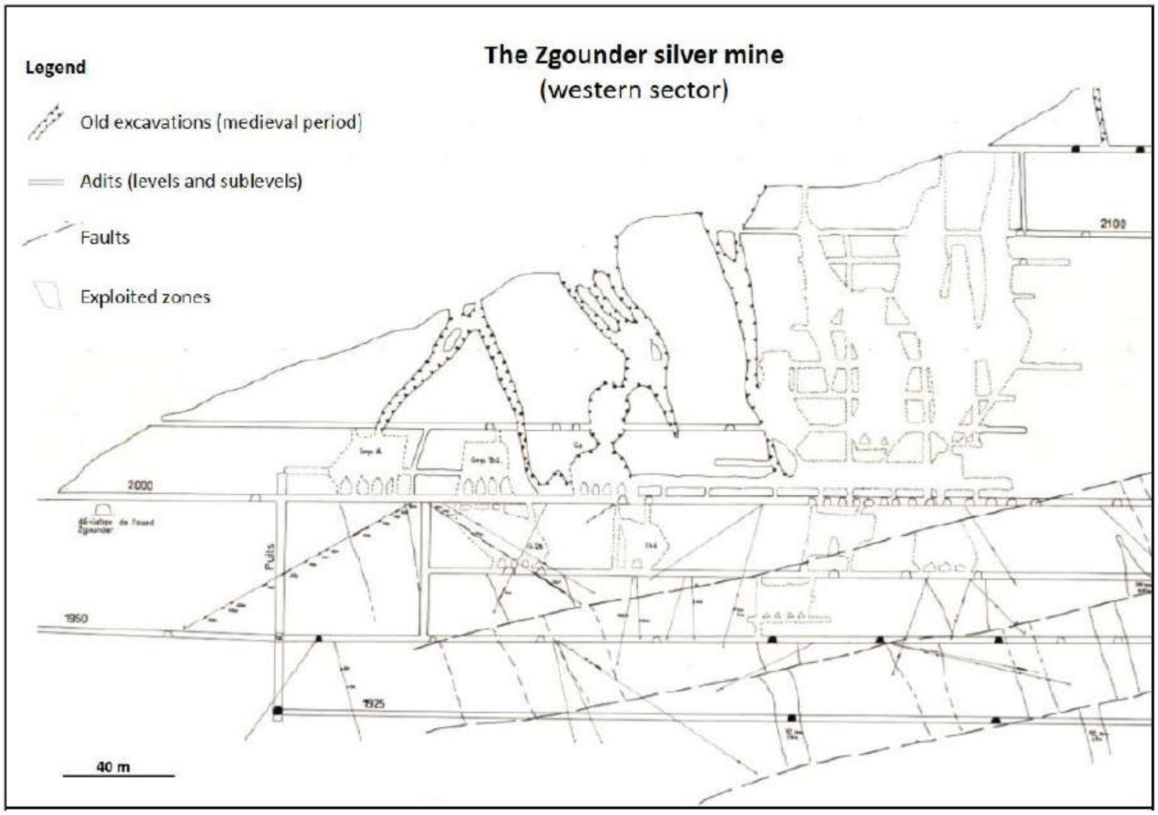
Earlier exploration campaigns and mining activities were completed by SNAM (1950-1955), BRPM (1956-1965; 1969-1972), SACEM-BRPM jointly (1971-1972), and BRPM-PNUD (1978).

6.1.3 SOMIL PERIOD (1982 TO 1990)

The Société Minière de Sidi Lahcen (SOMIL) operated the Zgounder Mine from 1982 to 1990. Mine development included several underground drifts and adits (9,220 m in total) connected by raises (1,200 m in total). The highest adit level was excavated at 2,175 MASL at the eastern end of the Mine. The lowest adit level was at 1,925 MASL in the Western Zone (Figure 6.3).

Drilling was conducted on several levels and sublevels totalling 15,383 m (percussion and diamond drilling). These drill holes were named according to their collar elevation. The 2000 Level (the main mine entrance) corresponds to the principal level for draw points and two (2) Alimak raises were developed. The run-of-mine product was transported by wagon to the entrance/exit of the 2000 Level, and then onwards to the primary crusher (Figure 6.4).

Figure 6.3 –Longitudinal Projection of the Western Zone by CMT Displaying the Ancient Excavation Areas at the Zgounder Silver Mine



Source: GoldMinds and SGS (2014)

6.1.4 BRPM PERIOD (1990 TO 1999)

In 1990, the BRPM pursued underground exploration with ten (10) drill holes totalling 2,282 m to delimit the extension of silver mineralisation in the Western Zone.

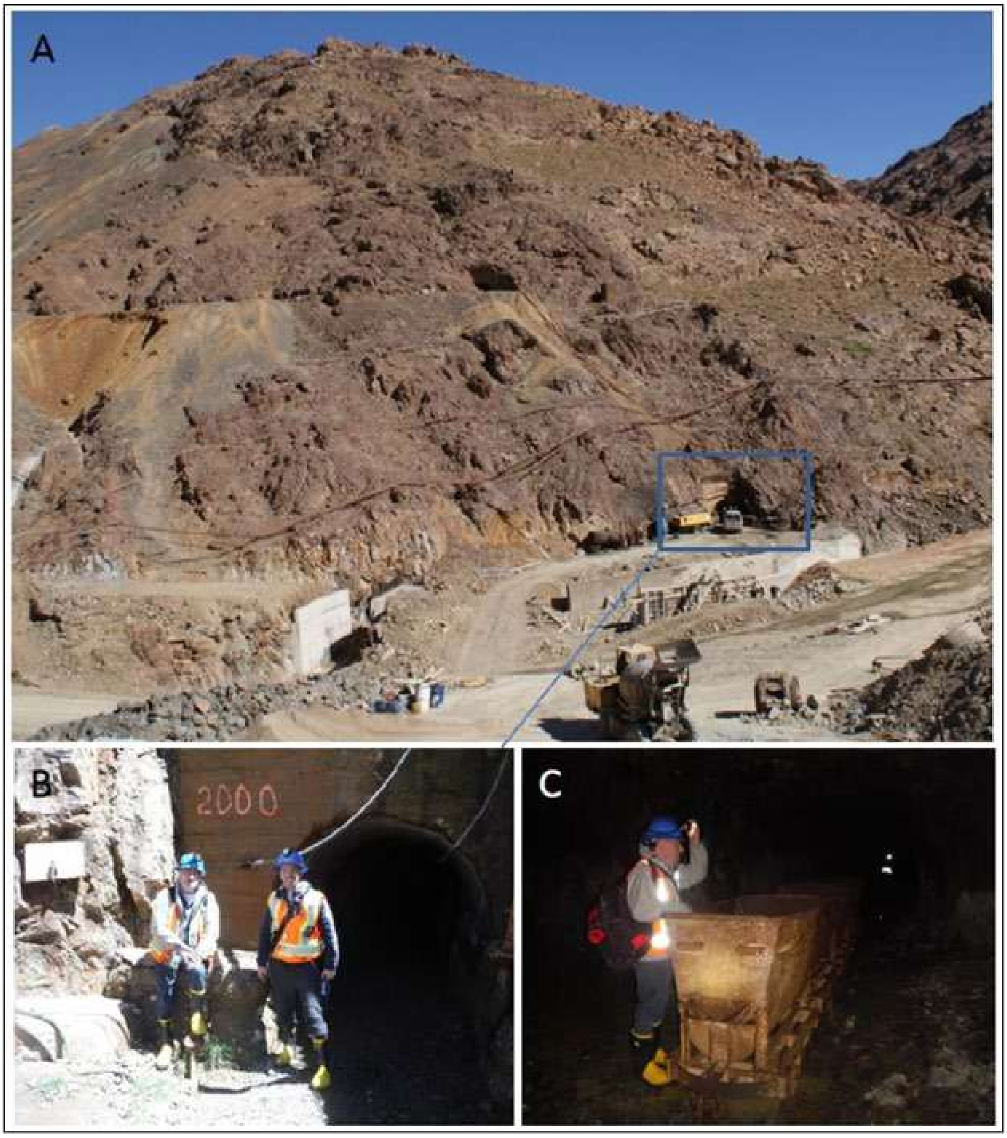
The BRPM started an exploration campaign in 1997 consisting of mapping and sampling the mineralised structures followed by a drill program. Seven (7) surface holes were drilled along strike of mineralised zones totalling 1,761 m of drill core. The BRPM interpreted these zones as new mineralised zones parallel to, and stratigraphically beneath, the dolerite contact zone.

However, ACA Howe (ACA Howe International, 1999) considers that these intersections can be explained by mineralised cross fractures. ACA Howe verified the available drill core left after the 1997 campaign and collected rock samples from key surface areas of the Mine.

The mine waste dump consists of surface reject material covering the southern side of the River Tlat Nouna (Figures 6.5 and 6.6). This waste dump was referred to as the ‘ancient tailings’ by

Icelandic Gold Corporation, forming a superficial thin layer of coarse (5 cm to 10 cm) and fine (1 cm to 3 cm) debris that range from 0.5 m to 2.0 m thick (Figure 6.6).

Figure 6.4 – Infrastructure from the SOMIL Period at Zgounder



Description: A) view looking east showing the entrance of the 2000 and 2025 Level adits, a network of water pipes and several bridges were in construction at the time of this photo; B) entrance to the 2000 Level prior to recent modifications; and C) an old small mine rail car on the 2,000 Level.

Source: GoldMinds and SGS (2014)

Figure 6.5 – Southward View of Oxidized Waste Dump Resulting from the Ancient Mine Workings at the Zgounder Silver Mine

Source: GoldMinds and SGS (2014)

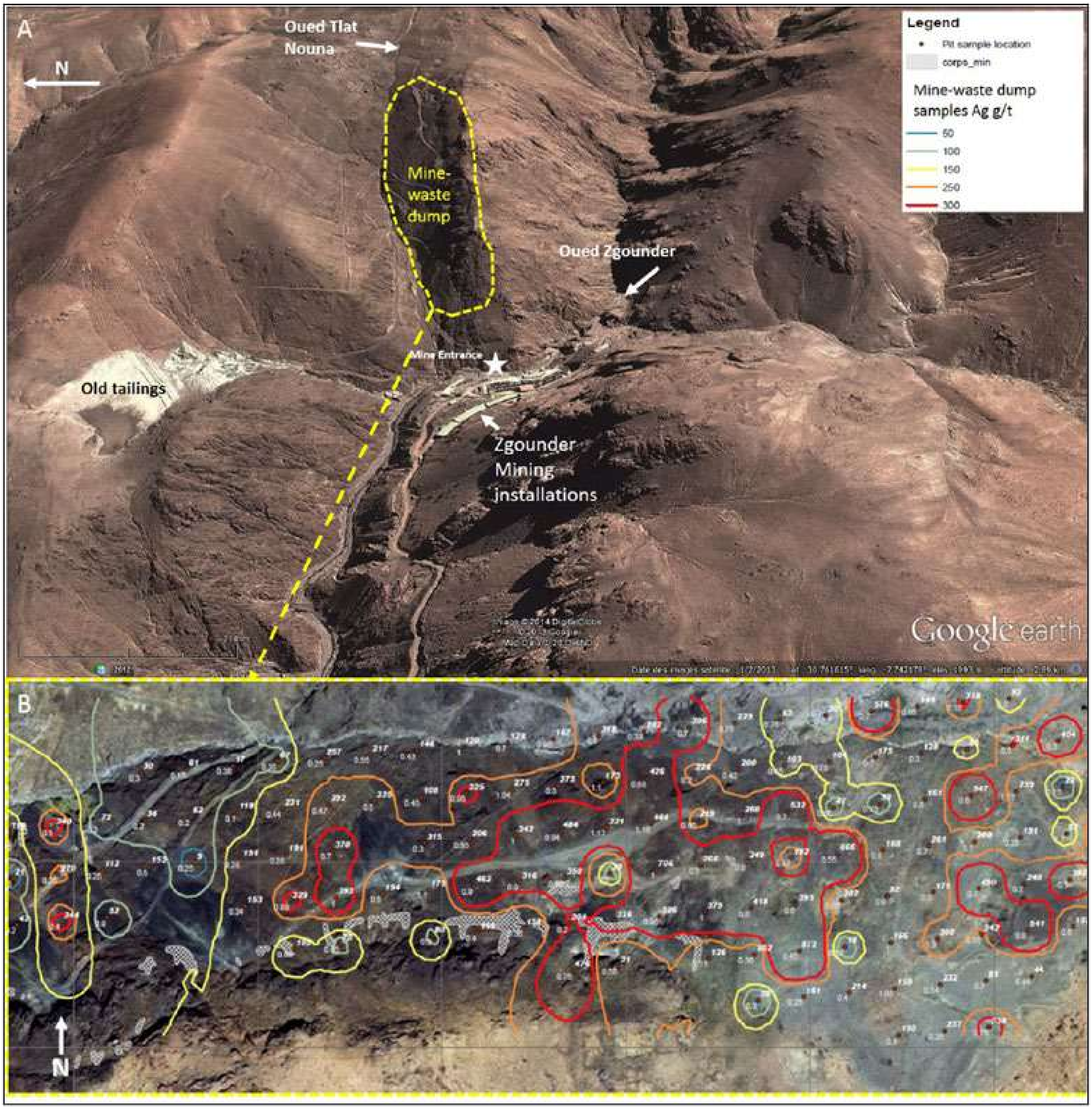
Icelandic Gold Corporation collected 20 surface samples from the mine-waste dump averaging 377 g/t Ag (or 438 g/t Ag from 17 samples). Another eight (8) surface samples from an EW traverse along the mine waste dump were collected having an average of 573.6 g/t Ag (ACA Howe International report, 1999). The old tailings located at the north of the Zgounder Mine installation correspond to the residues processed between 1980 and 1990, which were deposited in a V-shaped valley (Figure 6.6A). A channel sample 9 m deep taken by ACA Howe returned an average of 224 g/t Ag (Icelandic Gold Corporation Report, 1999).

6.1.5 CMT PERIOD (2002 TO 2004)

From 2002 to 2004, the Compagnie Minière de Touissit (CMT) conducted surface and underground exploration programs to delimit the mineralised zones in the northern zone of the Zgounder Mine and to verify the historical mineral resource estimate as previously defined by BRPM. CMT also continued to search for new mineralised zones in the central and eastern sectors of the Mine. The CMT exploration programs for 2002 to 2004 are summarised in

Table 6.1.

Figure 6.6 – BRPM Waste Dumps at Zgounder



Description: A) Location of the mine-waste dump at the Zgounder Mine (from Google Earth); and B) Location of samples collected by BRPM from the mine waste dump.

Source: GoldMinds and SGS (2014)

Table 6.1 – CMT Exploration Programs from 2002 to 2004[1]

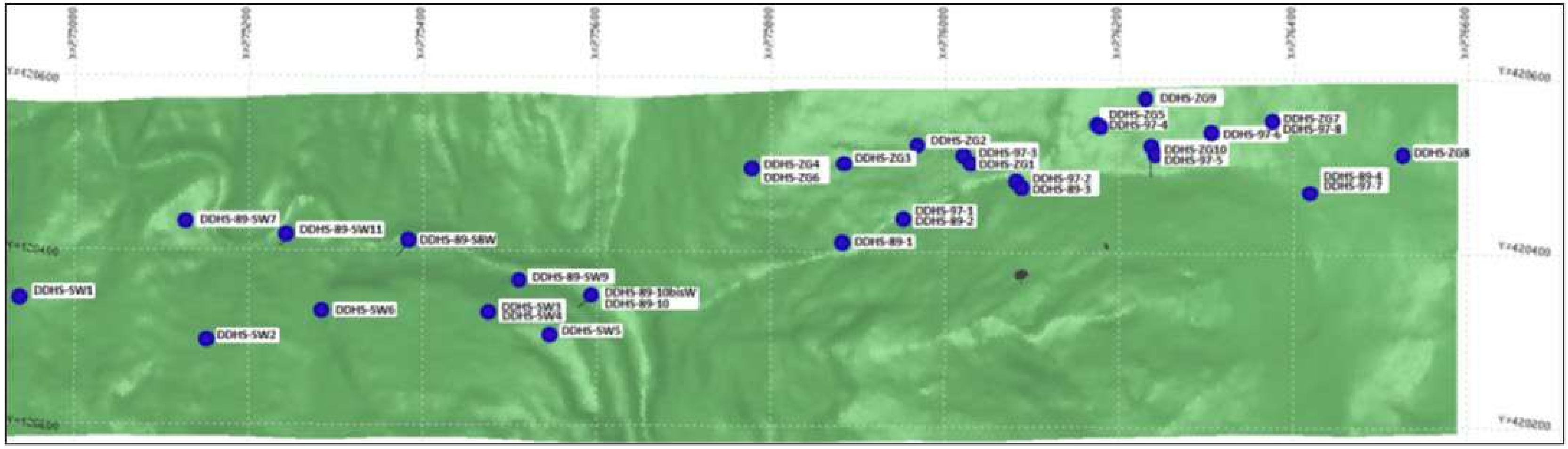
Item	2002	2003	2004	Total
Drilling (m)				
Surface diamond drilling	2,728	0	0	2,728
Underground diamond drilling	619	2,647	835	4,100
Percussion drill T28	56	7,733	3,998	11,787
Percussion drill Yak	0	2,529	320	2,849
Total Drilling	**3,403**	**12,909**	**5,153**	**21,464**
Mining Development (m)				
Adits	65	470	443	978
Raises	35	19	98	151

Source: GoldMinds and SGS (2018)

The CMT exploration program consisted of 10 surface drill holes and 26 underground drill holes totalling 6,827.9 m (Figures 6.7 and 6.8), and mining development (977.9 m) of new cross-cuts at the 2150, 2125, 2100, and 2000 Levels) to define new silver-rich zones at the Zgounder Mine (Table 6.2).

[1] CMT FINAL REPORT, 2004

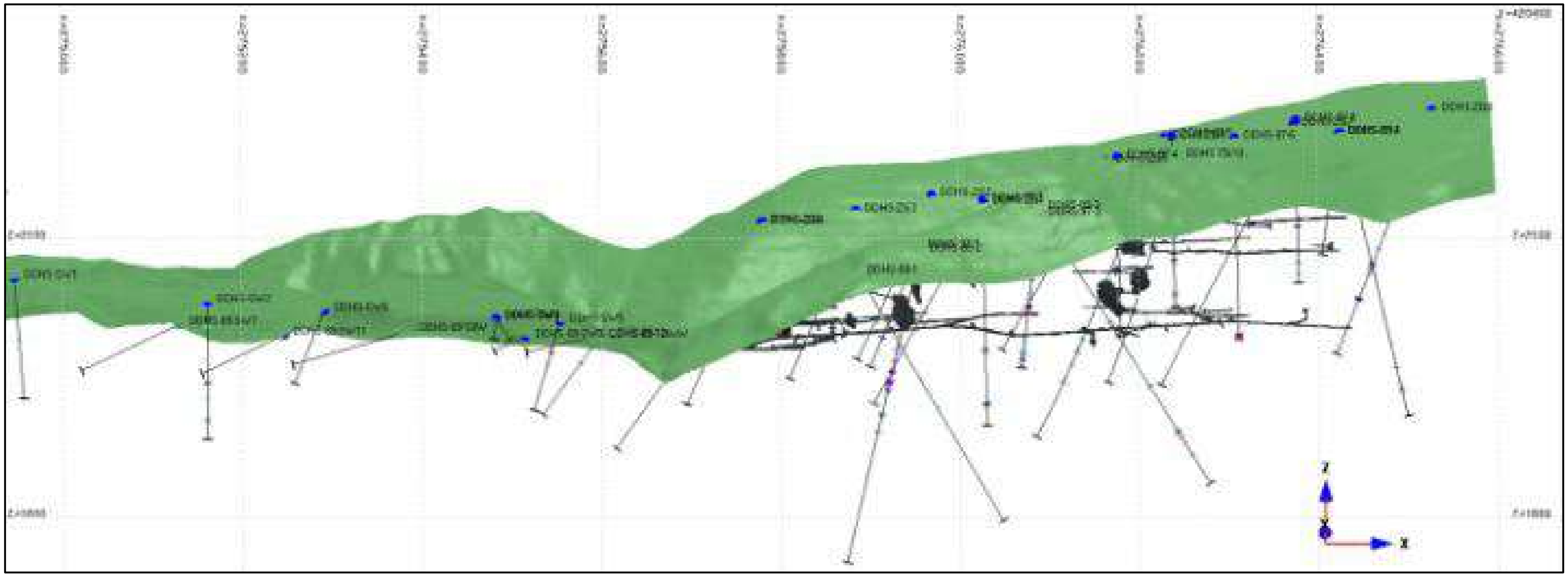
Figure 6.7 – Plan View of Surface Exploration CMT Diamond Drill Holes on Zgounder Property



Note: topographic surface is green

Source: GoldMinds and SGS (2014)

Figure 6.8 – 3-D Longitudinal Projection of Surface Exploration Diamond Drill Holes on Zgounder Property



Note: topographic surface is green

Source: GoldMinds and SGS (2014)

Table 6.2 – Summary of CMT Underground Workings from 2002 to 2004[2]

Level / Zone	Adit Length (m)	Purpose / Results
2150	2.0	Silver-rich sedimentary schist layers in altered dolerite
2125	153.4	Reaching the southern end of the level
2100	20.0	Fractured sandstone/shale associated with rhyolitic rocks
2000	802.6	East, northwest and west of the Central Zone
Including		
Northwest	313.0	Strongly fractured shale/sandstone. Clay-filed faults containing quartz, lead, and zinc sulphides, disseminated pyrite and chlorite. Silver mineralisation was encountered in and out of the fault zone.
Source: GoldMinds and SGS (2014)		

The Icelandic Gold Corporation referred to the old and recent tailings as 'modern tailings' (Figures 6.9 and 6.10).

Figure 6.9 – Views of the Old Tailings Storage Facility

Source: GoldMinds and SGS (2014)

[2] CMT FINAL REPORT, 2004

Figure 6.10 – Northeast View of the Recent Tailings Storage Facility

Source: GoldMinds and SGS (2014)

6.1.6 MAYA PERIOD (2012 TO 2020)

In 2012, Maya Gold & Silver ("Maya", now "Aya" as of July - 2020) and the Office National des Hydrocarbures et des Mines (ONHYM) agreed to negotiate an agreement for Maya to acquire 85% of the Zgounder Silver Deposit. Maya was granted the license (No. 2306) and the surface rights and access to the Property for any type of exploration and exploitation. The agreement on the Zgounder Silver Deposit was signed January 06, 2012.

Maya conducted a chip sampling on the 2000 m Level adit wall faces; specifically at six (6) Ag mineralised sites where historical sampling and assay results from the BRPM (Boily, 2012).

Forty-one (41) face wall samples were collected from nine (9) stations (1 to 9) eight (8) of them located at the western part of the 2000 m Level at the already exploited zones by SOMIL. Each sample represents an interval of one metre and their location is determined with ±2 m precision.

No previous evidence of face sampling was observed within the 2000 m Level. The weights of the new samples vary from 0.97 kg to 3.33 kg and are deemed representative of the in-situ silver mineralisation. The face samples represent well stratified arenaceous schists with mineralised fractures containing <0.5% pyrite and traces of sphalerite and galena (Boily, 2012).

These samples were fire assayed and the results are generally higher by a factor of x2 to x15 compared to the historical data. These differences could partially be explained by different sampling techniques and (or) assay methods (Boily, 2012).

6.2 Historical Mineral Resource and Mineral Reserve Estimates

Readers are cautioned that the authors of this Technical Report have not performed sufficient work to classify the historical estimates as current Mineral Resources or Mineral Reserves; and the Issuer is not treating the historical estimates as current Mineral Resources nor as Mineral Reserves.

6.2.1 SNAM-BRGM

During the SNAM-BRGM Period (1950-1979), historical mineral resources of approximately 435,000 t at 385 g/t Ag were defined (Reminex, 2009). In 1978, the BRPM and PNUD excavated pits on a 20 m grid over the ancient tailings waste dump and outlined a historical mineral reserve of 66,000 t grading 325 g/t Ag, assuming a bulk density of 1.5 g/cm^3 (ACA Howe International, 1999).

6.2.2 SOMIL

The North Zone of the 2,000 Level was the subject of underground exploration drilling between 1984 and 1987 with a total historical mineral resource of 110,280 t at 277 g/t Ag, including10,700 t at 180 g/t Ag.

Between 1982 and 1990, SOMIL extracted a total of 500,000 t at 330 g/t Ag totalling 5.3 Moz of silver using backfill, shrinkage, and open long-hole mining methods and applying a cut-off grade of 125 g/t Ag (Reminex, 2009).

6.2.3 BRPM-Icelandic

'Reserves' were calculated for the Zgounder Mine on June 30, 1990, shortly before closure due to low silver prices. In summary, the 'historical mineral reserves' stated by ACA Howe International (1999) are shown in

Table 6.3.

Table 6.3 – Distribution of Various Historical Mineral Reserve Blocks[3]

Zone	Unit	Proven	Probable	Possible	Total
Central	Tonnes	15,377	27,260	17,077	59,714
	Ag g/t	388	336	314	343
Central at Depth	Tonnes		34,760	27,614	62,374
East	Tonnes		55,570	69,585	125,155
	Ag g/t		374	394	385.12
North	Tonnes		25,413	63,602	89,015
	Ag g/t		197	318	283
Total	Tonnes	15,377	143,003	177,878	336,258
	Ag g/t	388	348	367	360
Ancient Tailings	Tonnes		66,117	(not included in the in-situ total)	
	Ag g/t		325		

Source: GoldMinds and SGS (2014)

Readers are cautioned that the authors of this Technical Report have not performed sufficient work to classify the historical estimates as current Mineral Resources or Mineral Reserves; and the Issuer is not treating the historical estimates as current Mineral Resources nor as Mineral Reserves.

In 1999, BRPM calculated the Zgounder Mine historical mineral reserves to be 4.58 Moz of silver at a cut-off grade of 150 g/t. Icelandic calculated an additional potential historical mineral resource of 1.4 Moz to 2.5 Moz of Ag hosted in the 400,000 t tailings contained within the old dam (Icelandic Gold Corporation Report, 1999) (See Figure 6.9).

6.2.4 CMT

Research conducted by BRPM and CMT resulted in a historical mineral reserve calculation totalling 869,650 t grading 405.4 g/t Ag, divided between the South Zone (357,400 t at 468.3 g/t Ag) and North Zone (449,625 t at 375.9 g/t Ag) of the Zgounder Mine (CMT, 2004). The total historical mineral resource estimate included mineral resources remaining within existing stopes and the "mineral reserves" calculated from the old tailings and is listed in Table 6.4.

[3] ACA HOWE INTERNATIONAL REPORT, 1999

Table 6.4 – Comparison of CMT Historical Mineral Reserves Between 2002 and 2004

	Reserves 31 December 2002			Reserves 31 December 2003			Observations
	ttv	Ag (g/t)	metal Ag (kg)	ttv	Ag (g/t)	metal Ag (kg)	metal Ag (kg)
Reserves Certaines							
Corps sud, secteur Central	38,858	334.2	12,897	38,844	344.2	9,929	-3,059
Corps sud, secteur Central Aval	32,055	494.1	15,838	32,055	494.1	15,838	0
Corps sud, secteur Nord	10,700	180.0	1,926	10,700	180.1	1,926	0
Corps sud, secteur Est	17,740	321.0	5,694	50,897	650.2	33,096	27,401
Total reserves certaines, mine	99,353	366.8	36,445	122,496	496.2	60,788	24,343
Haldes	52,637	242.8	12,781	52,637	239.9	12,628	-153
Stock extrait mine, carreau	0		0	4,598	453.9	2,087	2,087
Stock haldes, carreau	4,481	270.7	1,213	4,481	270.7	1,213	0
Total reserves certaines	**156,471**	**322.4**	**50,439**	**184,212**	**416.5**	**76,715**	**26,276**
Reserves Probables							
Corps sud, secteur Central	15,136	402.2	6,087	15,139	402.2	6,087	0
Corps sud, secteur Central Aval	18,385	497.5	9,146	18,385	497.5	9,146	0
Corps sud, secteur Nord	14,181	399.0	5,658	31,866	349.0	11,120	5,462
Corps sud, secteur Est	117,082	411.5	48,185	71,797	472.1	33,897	-14,288
Total reserves probables, mine	**164,784**	**419.2**	**69,076**	**137,184**	**439.2**	**60,250**	**-8,828**
Reserves Possibles							
Corps sud, secteur Central	14,765	353.6	5,220	14,764	353.6	5,220	0
Corps sud, secteur Central Aval	37,093	400.4	14,851	37,093	400.4	14,851	0
Corps sud, secteur Nord	39,894	438.6	17,499	39,894	438.6	17,499	0
Corps sud, secteur Est	67,374	407.9	27,484	61,960	410.5	25,437	-2,047
Total reserves possibles, mine	**159,125**	**408.8**	**65,054**	**153,711**	**409.9**	**63,007**	**-2,047**
Total Reserves	**480,380**	**384.2**	**184,569**	**475,107**	**420.9**	**199,972**	**15,403**

Note: ttv = *tonne tout venant* equiv. to tonne ore.
Source: GoldMinds and SGS (2014)

Readers are cautioned that the authors of this Technical Report have not performed sufficient work to classify the historical estimates as current Mineral Resources or Mineral Reserves; and the Issuer is not treating the historical estimates as current Mineral Resources nor as Mineral Reserves.

The central part of the South Zone is heavily mined above the 2000 m Level and this Zone extends to the east. The East Zone of the 2000 m Level was therefore the subject of the drilling campaign supervised by GMG, followed by the Central and North Zones.

6.2.5 MAYA

In 2014, Maya released a Preliminary Economic Assessment ("PEA"; GoldMinds, 2014a) and a Pre-Feasibility Study ("PFS"; GoldMinds, 2014b). The PEA was supported by a Mineral Resource Estimate based on historical information and new analytical data sampled from an underground percussion drilling program completed in 2013. A total of 23 mineralised envelopes and 67 panels (small, mineralised bodies) were utilised in the Mineral Resource estimation. Using a cut-off grade (COG) of 125 g/t Ag, base case Measured Resources totalled 1.4 Moz of silver (142,000 t averaging 304 g/t Ag), Indicated Mineral Resources totalled 4.6 Moz of silver (400,000 t averaging 357 g/t Ag), and Inferred Mineral Resources total 5.3 Moz of silver (353,000 t averaging 463 g/t Ag) (Table 6.5). Contained silver in Measured and Indicated Mineral Resources totalled 6 Moz of silver (540,000 t averaging 343 g/t Ag).

The 2014 PFS estimated Mineral Reserves (GoldMinds, 2014b), in accordance with the definitions and guidelines adopted from the Canadian Institute of Mining, Metallurgy and Petroleum (CIM Standards on Mineral Resources and Reserves). The estimated Mineral Reserves were based entirely on Measured and Indicated Mineral Resources, which were converted to Proven and Probable Mineral Reserves, respectively (Table 6.6). Aya commenced process plant operations at its Zgounder Mine in July 2014 and announced the first silver pour in August 2014 with the production of the 20 silver ingots. As of December 2017, Aya produced a total of 1.432 Moz of silver ingots from the Zgounder Mine.

A subsequent updated Mineral Resource Estimate was published in an Aya press release dated January 8, 2018 (Table 6.7), and subsequently supported the 2018 PEA (GoldMinds, 2018).

Readers are cautioned that the authors of this Technical Report have not performed sufficient work to classify the historical estimates as current Mineral Resources or Mineral Reserves; and the Issuer is not treating the historical estimates as current Mineral Resources nor as Mineral Reserves.

Table 6.5 – Zgounder Base Case (>125 g/t Ag) Historical Mineral Resource Estimate[4]

Classification	Tonnes[1]	Ag (g/t)	Ag (oz)
Measured	142,000	304	1,391,000
Indicated	397,000	357	4,560,000
Inferred	353,000	463	5,254,000
Measured and Indicated	**539,000**	**343**	**5,948,000**

1. Totals may not sum correctly due to rounding

Source: GoldMinds (2014b)

Table 6.6 – Estimated Mineral Reserves – 2014 Pre-Feasibility Study

Classification	Tonnes[1]	Ag (g/t)	Ag (oz)
Proven	151,659	281.2	1,371,182
Probable	421,175	330.4	4,473,519
Proven and Probable	**572,834**	**317.3**	**5,844,701**

1. Totals may not sum correctly due to rounding

Source: GoldMinds (2014b)

[4] GoldMinds, 2014b - ZGOUNDER PRE-FEASIBILITY STUDY

Table 6.7 – Mineral Resource Estimate at Zgounder Silver Mine - 2018 Zgounder PEA

Area	Class	Tonnes (t)	Ag (g/t)	Ag (oz)
In-pit Resource Estimate	Measured	208,000	315	2,108,000
	Indicated	616,000	293	5,794,000
	Measured +Indicated	824,000	298	7,902,000
	Inferred	1,886,000	248	15,012,000
High-Grade Underground Resource Estimate	Measured	34,000	482	527,000
	Indicated	132,000	377	1,601,000
	Measured +Indicated	166,000	398	2,128,000
	Inferred	1,051,000	332	11,209,000
Old Tailings Inferred Resources	Measured	-	-	-
	Indicated	-	-	-
	Measured +Indicated	-	-	-
	Inferred	500,000	132	2,122,000
Total	Measured	242,000	338	2,633,000
	Indicated	748,000	308	7,395,000
	Measured +Indicated	990,000	315	10,028,000
	Inferred	3,437,000	256	28,343,000

Source: GoldMinds and SGS (2018) and Aya (at the time Maya Gold & Silver) company press release dated January 8, 2018.

6.3 Previous Mineral Resource Estimate

The previous Zgounder updated Mineral Resource Estimate by P&E was published in an Aya press release dated March 16, 2021 and is summarized below in Table 6.8. At a cut-off grade of 70 g/t Ag, pit-constrained updated Measured and Indicated Mineral Resource totalled 684 kt grading 277 g/t Ag for 6.1 Moz Ag. At a cut-off grade of 125 g/t Ag, out-of-pit updated Measured and Indicated Mineral Resource totalled 3.9 Mt grading 296 g/t Ag for 37.5 Moz Ag, and updated Inferred Mineral Resource totalled 59 kt grading 209 g/t Ag for 395 koz Ag. At a cut-off grade of 50 g/t Ag, tailings updated Indicated Mineral Resource totalled 272 kt grading 94 g/t Ag for 817 koz Ag.

Table 6.8 – P&E Updated Mineral Resource Estimate (Effective March 1, 2021) (1-12)

Area	Class	Cut-Off	Tonnes	Ag	Ag
		(Ag g/t)	(k)	(g/t)	(koz)
Pit Constrained	Measured	70	534	301	5,158
	Indicated	70	150	190	916
	Measured +Indicated	70	684	277	6,074
	Inferred	-	-	-	-
Out-of-Pit	Measured	125	3,052	303	29,704
	Indicated	125	885	275	7,815
	Measured +Indicated	125	3,937	296	37,519
	Inferred	125	59	209	395
Tailings	Measured	-	-	-	-
	Indicated	50	272	94	817
	Measured +Indicated	50	272	94	817
	Inferred	-	-	-	-
Total	Measured	70 & 125	3,586	302	34,862
	Indicated	50, 70 & 125	1,307	227	9,548
	Measured +Indicated	50, 70 & 125	4,893	282	44,410
	Inferred	125	59	209	395

Notes:
1) Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues. There is no certainty that Mineral Resources will be converted to Mineral Reserves.
2) The Inferred Mineral Resource in this estimate has a lower level of confidence than that applied to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of the Inferred Mineral Resource could be upgraded to an Indicated Mineral Resource with continued exploration.
3) The Mineral Resources in this news release were estimated in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (the "CIM") Standards on Mineral Resources and Mineral Reserves Definitions (2014) and Best Practices Guidelines (2019) prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council, as may be amended from time to time.
4) A silver price of US$20/oz with a process recovery of 85%, US$30/t rock process cost, US$20/t tailings process cost and US$7/t G&A cost were used.
5) The constraining pit optimization parameters were US$2/t mineralised and waste material mining cost and 50º pit slopes with a 70 g/t Ag cut-off.
6) The out-of-pit parameters used a US$30/t mining cost. The out-of-pit Mineral Resource grade blocks were quantified above the 125 g/t Ag cut-off, below the constraining pit shell and within the constraining mineralised wireframes. Out-of-pit Mineral Resources exhibit continuity and reasonable potential for extraction by the cut and fill underground mining method.
7) The tailings parameters were at a US$2/t mining cost, and Mineral Resource grade blocks were quantified above the 50 g/t Ag cut-off.
8) Individual calculations in tables and totals may not sum correctly due to rounding of original numbers.
9) Grade capping of Ag outliers using thresholds based on statistical distribution for each geological domain were as follows: Domain SE100 (7.4 kg/t), Domain NW200 (6.5 kg/t), Domain NE300 (2.3 kg/t) and Domain SW400 (7.4 kg/t).
10) Block bulk density was determined from measurements taken from core samples and averaged 2.75 t/m^3 for the main mineralised host rock type.
11) 1.2 m composites were used during grade estimation.
12) Previously mined areas of the Deposit were depleted from the Mineral Resource Estimate.

This Updated Measured and Indicated Mineral Resources for Zgounder totalling 4.9 Mt averaging 282 g/t Ag for 44.4 Moz Ag represent an increase of 340% compared to the previous (February 2018) Measured and Indicated Mineral Resources of 10 Moz Ag. The updated Measured Mineral Resource increased from 2.6 Moz to 34.9 Moz Ag, a 1,242% increase compared to February 2018. The Updated Indicated Mineral Resource increased from 7.4 Moz to 9.5 Moz, an increase of 28% compared to February 2018. On the other hand, the Updated Inferred Mineral Resource decreased from 28.3 Moz to 0.40 Moz Ag compared to February 2018. The Updated Mineral Resource Estimate incorporates drilling carried out on Zgounder between February 2018 and the end of December 2020.

This Updated Mineral Resource Estimate is superseded by that presented in Section 14 of this Technical Report.

From December 2017 through September 2021, Aya produced a total of 2.604 Moz of silver from Zgounder (Aya press releases, 2019 to 2021).

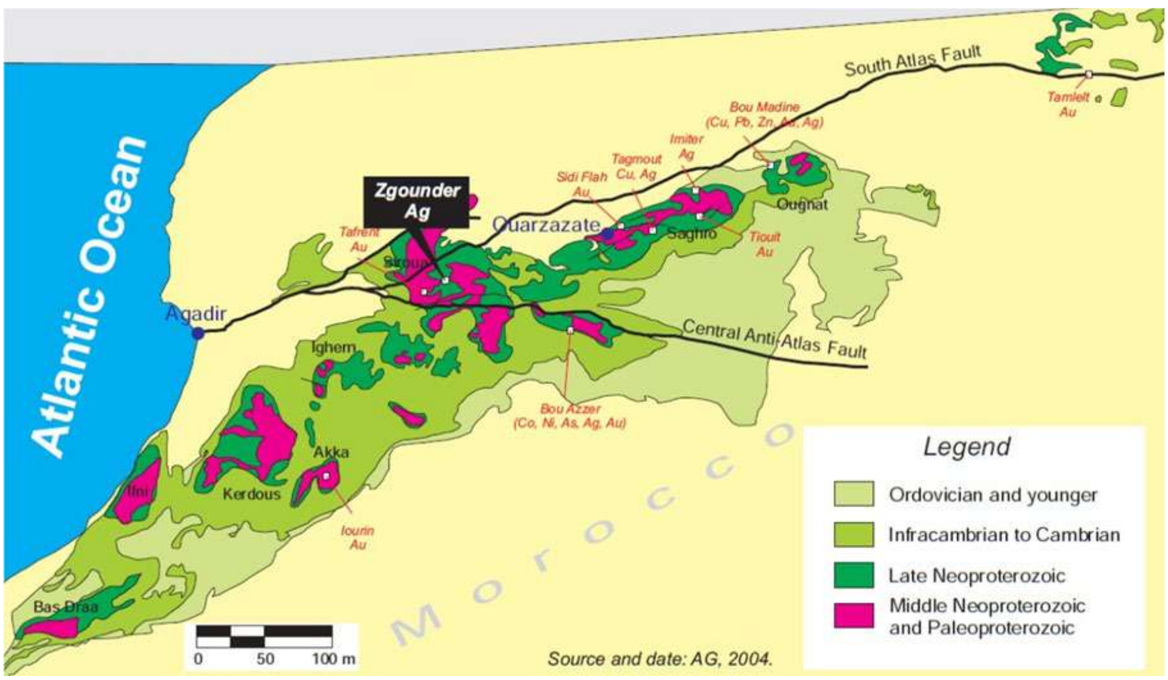
7 GEOLOGICAL SETTING AND MINERALISATION

This Section has been summarised and/or largely extracted from the Report available on SEDAR entitled: "Technical Report and Updated Mineral Resource Estimate of the Zgounder Silver Project, Kingdom of Morocco, prepared for Aya Gold & Silver Inc., dated January 28, 2022, prepared by P&E Mining Consultants Inc., Report # 415."

7.1 Regional Geology

The Zgounder Silver Deposit is located in the central Anti-Atlas Mountain Chain on the northwest flank of the Siroua Massif hosted in the Pan-African Orogenic Belt (680 Ma to 580 Ma). The Pan-African Orogeny started during the Middle Precambrian (Clauer, 1974) with the formation of a back-arc basin filled by a series of synorogenic volcano-sedimentary rocks. The back-arc basin was covered at the end of the Precambrian by the Adoudounian marine sediments, as a result of a marine transgression affecting the whole Anti-Atlas Mountain Chain (Figure 7.1).

Figure 7.1 – Regional Geology of the Anti-Atlas Mountain Chain



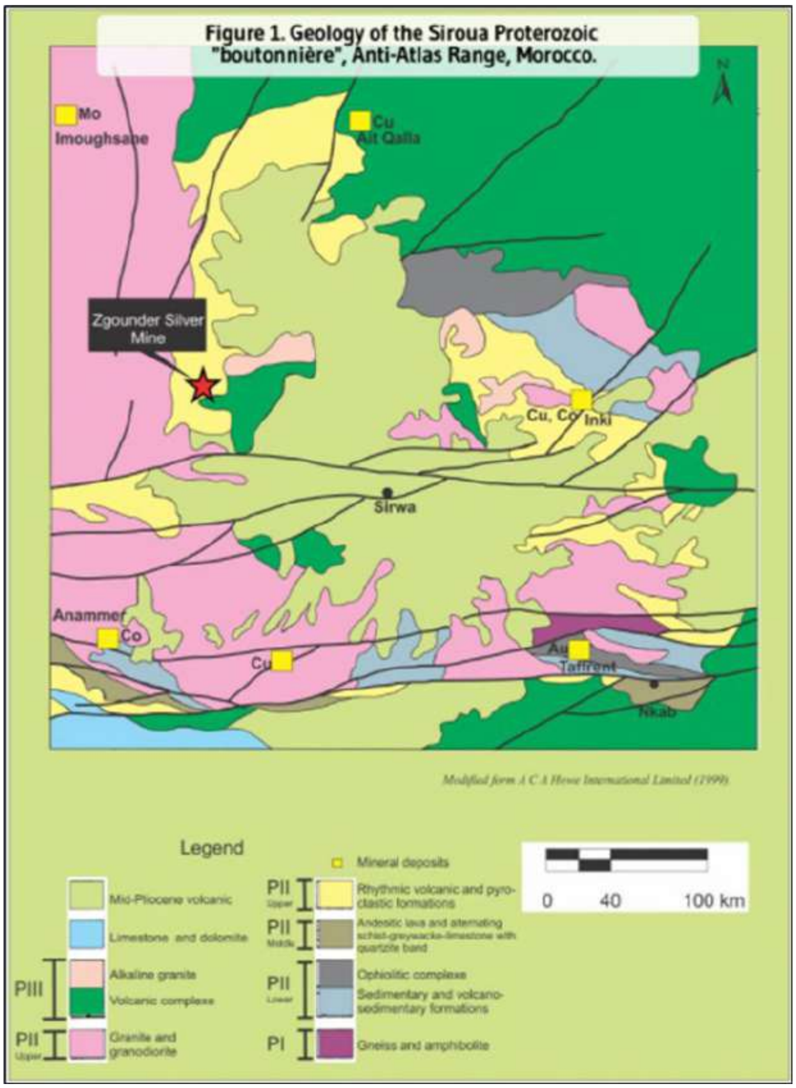
Source: GoldMinds and SGS (2018)

The Siroua Massif is located between two major structural zones, namely a northern unit attached to the Pan-African Domain and a southern unit generated by the Eburnian Orogeny and accreted onto the West African Craton. The Siroua Massif consists of a Pan-African bedrock (gneiss and amphibolite) that is unconformably overlaid by ophiolitic complexes and volcano-sedimentary units of alternating schist-sandstone, limestone, quartzite and turbidite. The Zgounder mineralisation occurred during felsic calc-alkaline/alkaline volcanic activity marking the commencement of rifting and the Infracambrian–Cambrian Transgression in the Late Neoproterozoic (Buggisch and Flügel, 1988).

7.2 Property Geology

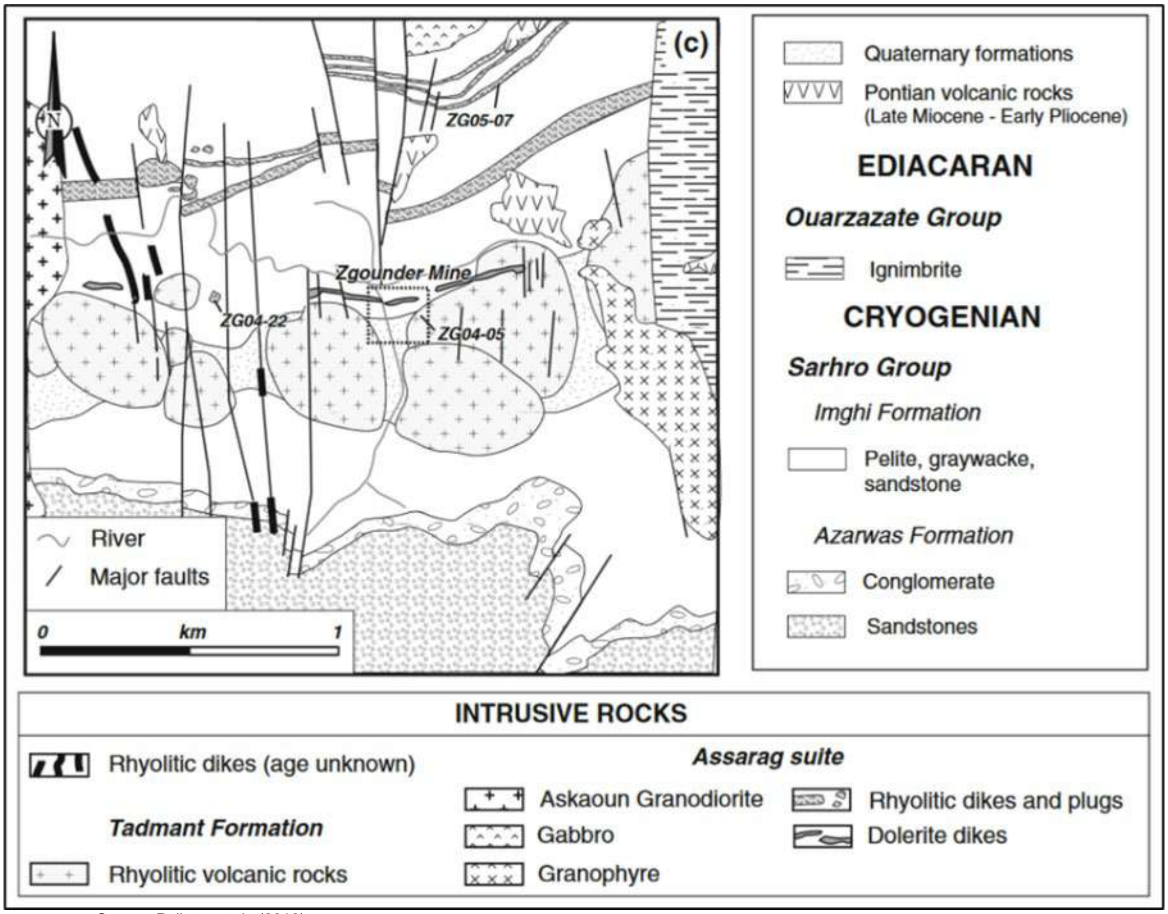
The geological series at Zgounder consist mainly of volcano-sedimentary formations attributed to the Precambrian II (PII), which are intruded to the west by the Askaoun Granodioritic Massif (later Precambrian II-III), (Demange, 1977) (Figures 7.2 and 7.3). The series are overlaid in the east by the volcano-sedimentary rocks of the Ouarzazate series (PII) and Neogene phonolites.

Figure 7.2 – Geology of the Siroua Proterozoic Massif, Anti-Atlas Mountain Chain, Morocco



Source: Aya website (2021)

Figure 7.3 – Geology of the Zgounder Mine Property Area

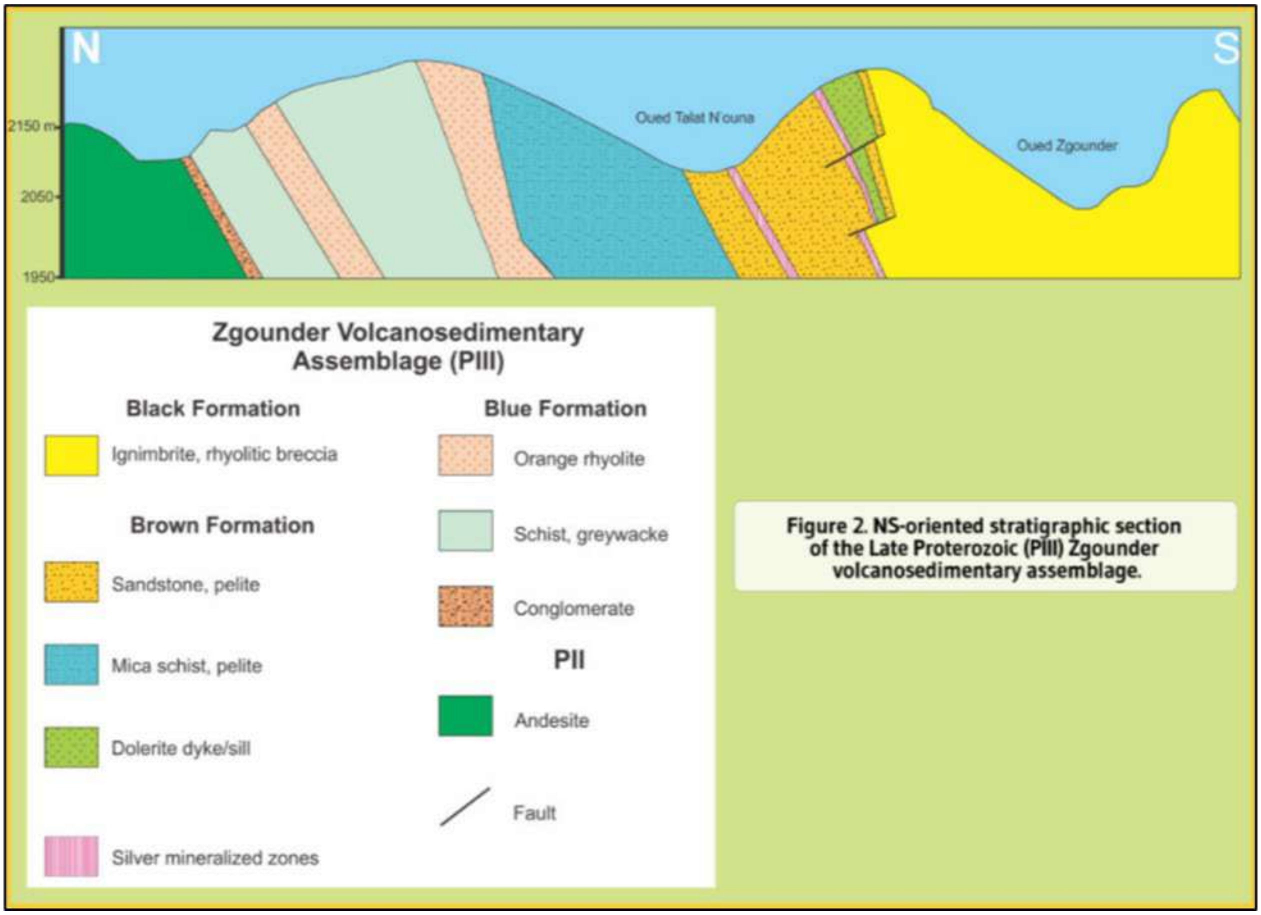


Source: Pelleter et al., (2016)

The Zgounder volcano-sedimentary series consist of a mixed sequence of metavolcanic rocks, metasedimentary rocks, dolerite and granodiorite intrusions. Outcrops in the form of a window of PII rocks occur on the south limb of a large east-west trending monocline, dipping steeply to the south. The series is surrounded by PIII volcanics and volcaniclastics to the east, basal PII formations to the north, and the Askaoun Granite to the west and southwest.

The geological series at Zgounder were divided into three (3) formations (Demange, 1997); two (2) with a major clastic component intercalated with volcanics (the 'Blue' and 'Brown' Formations) overlaid by an acid ignimbrite volcanic complex (the 'Black Formation') (Figures 7.4 and 7.5).

Figure 7.4 – N-S Oriented Stratigraphic Cross-Section of the Late Proterozoic (PIII) Zgounder Volcano-Sedimentary Assemblage

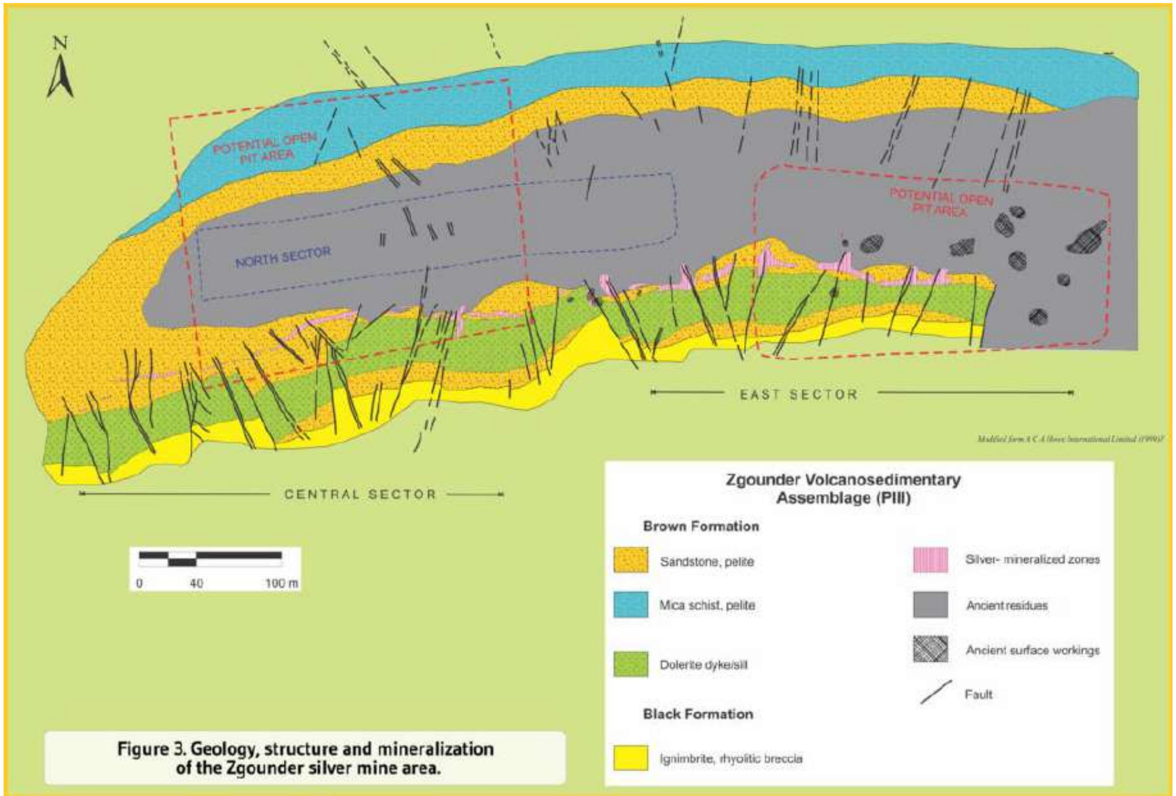


Source: Aya (website, 2021)

7.2.1 Blue Formation

As depicted in Figure 7.4, the Blue Formation is 300 to 400 m in thickness and composed of sandstone, greywacke and pelites with interbedded tuffs and quartz-keratophyre. The formation terminates in an orange rhyolitic unit, which forms the topographic ridge to the north of the mineralised zone.

Figure 7.5 – Geology, Structure, and Silver Mineralisation of the Zgounder Mine Area

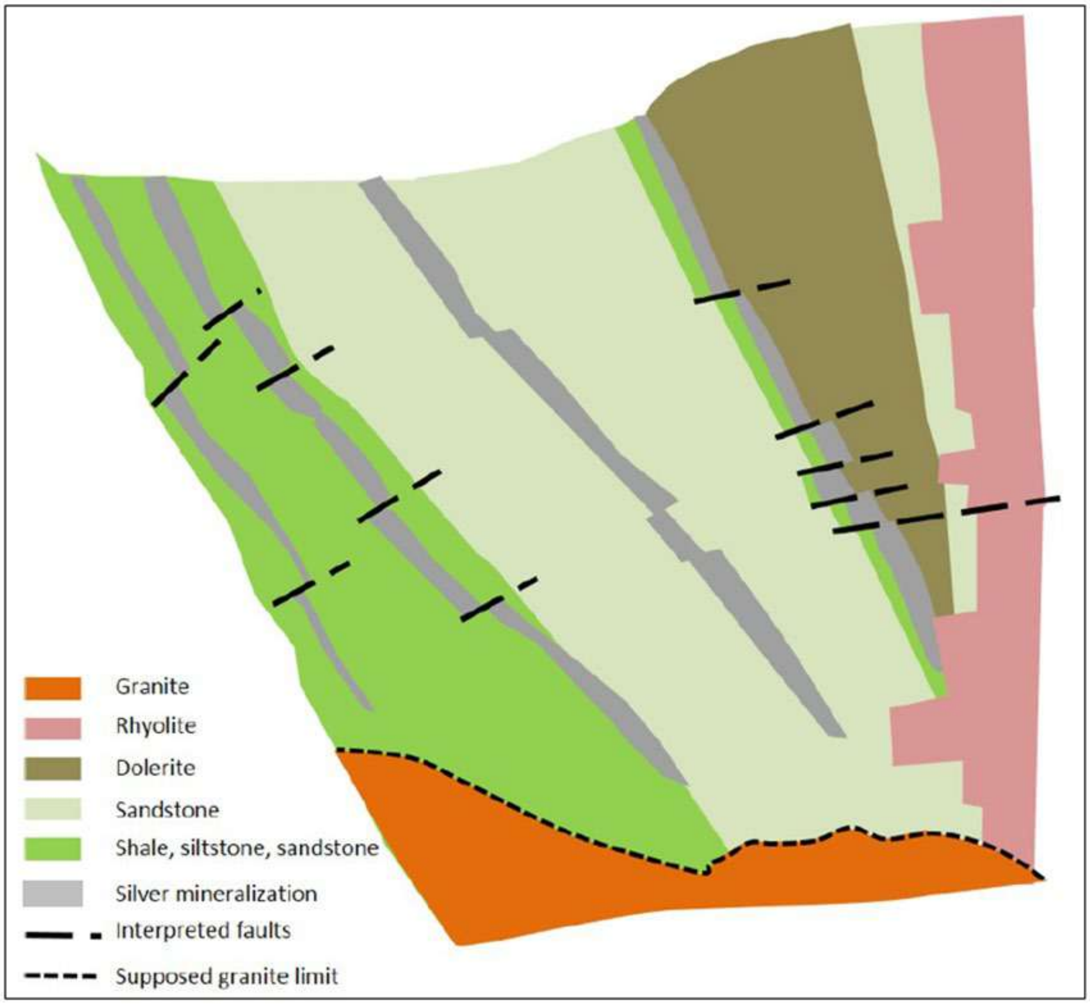


Source: Aya (website December, 2021)

7.2.2 BROWN FORMATION

As depicted in Figure 7.5, the Brown Formation is 350 to 400 m in thickness and consists of mica schist, arenaceous schist, breccia intercalations, and pelite (containing green volcanic clasts) overlain by a 45 m thick dolerite sill/dyke. The Brown Formation is affected by epizonal metamorphism, as indicated by weak schistosity, which is difficult to distinguish from the stratification. This formation is composed of two units: Unit 1 is 120 m in thickness and composed of heavily oxidized, coarse mica schist located north of Talat N'ouna; and Unit 2 is 280 m in thickness and largely covered by the ancient tailings on the southern flank of the Talat N'ouna River. Unit 2 consists of a coarse-grained pelite with millimetric clasts in sericitic or chloritic tuffaceous bands. The bands have a volcano-sedimentary origin and host polymetallic mineralisation (pyrite, sphalerite, galena, arsenopyrite, silver sulphide and native silver) (Figure 7.6).

Figure 7.6 – Conceptual Geologic Model of Zgounder (GMG 2017)



Source: Modified by P&E (December, 2021), from GoldMinds and SGS (2018)

7.2.3 BLACK FORMATION

As depicted in Figure 7.5, the Black Formation is 900 m in thickness and consists of a basal felsic volcanic complex (ignimbrite, rhyolitic breccias, devitrified rhyolite, pyroclastic rocks) that forms the hanging wall of the Ag-mineralisation in the upper part of the Brown Formation. Farther south, the upper part of the Black Formation is composed by sandstone, greywacke and some thin intercalations of polymictic conglomerate (Figure 7.6).

7.3 Deposit Geology and Structure

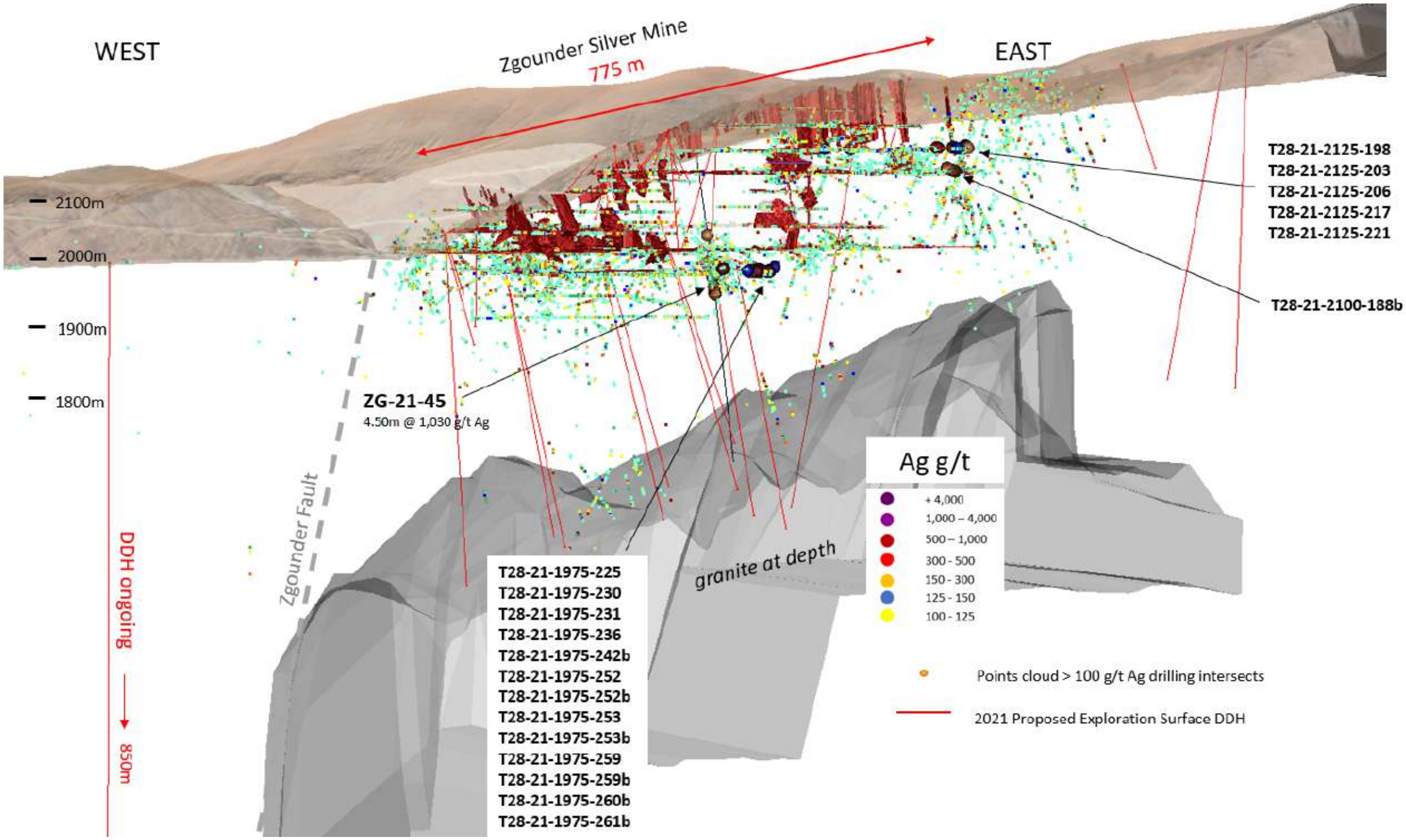
The Zgounder Silver Deposit is cross-cut by fractures of variable orientations. There are at least four (4) fracture systems:

1. Late sub-vertical E-W fractures and shear zones;
2. N-S fractures/faults dipping steeply to the east;
3. NNE-NNW-oriented system dipping 60° at a strike of 75°E; and
4. A sub-horizontal system of fractures oriented NNE and NNW, which displaced the Brown Formation to the north with depth (Bounajma, 2002).

Based on the information collected from the recent diamond drilling programs, Aya modelled the surface of the granite at Zgounder Mine (Figure 7.7). The deepest silver mineralisation was intersected during the 2017 drilling program (Hole 16), which corresponds to the deep extension of the mineralisation, just above the granite surface. The diamond holes drilled at the East Zone intersect the granite at depths shallower than the West Zone, with a minimum depth of elevation approximately 1,945 m.

A granitoid intrusion occurs to the northeast of the Zgounder Mine and granitic contacts and finger-shaped bodies have been found at depth, which suggests that the high-grade silver mineralisation is associated with this intrusion. The southern contact with the rhyolite is irregular with steeply dipping mineralisation blocks offset southward/northward and southward along flat-lying faults.

Figure 7.7 – Granite Surface Model Contact at Depth Plunging Westward



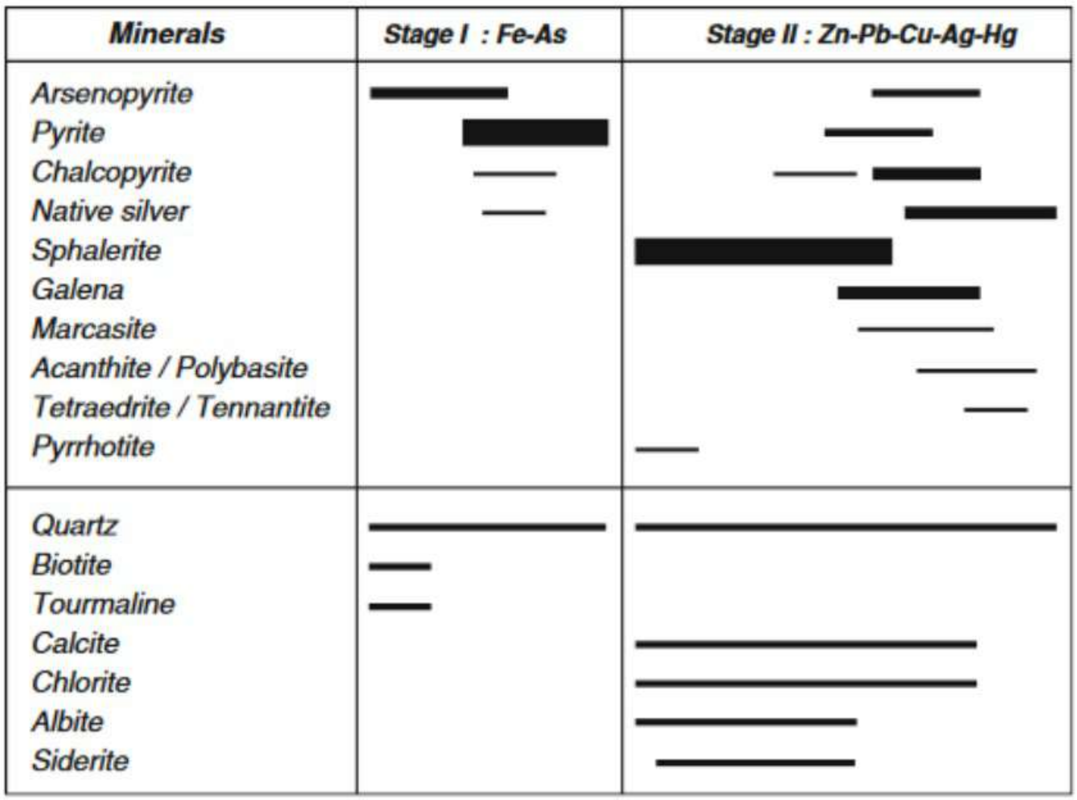
Source: Aya (Press Release dated October 5, 2021 and Corporate Presentation, November 2021)

7.4 Mineralisation

The Zgounder silver mineralisation occurs at the top of the Brown Formation (sandstone), predominantly along the contact and within the dolerite sill. The economic silver concentrations at Zgounder are present mainly as vertical bodies, complex clusters, shear zones, and veinlets, and at the intersection of the E-W and N-S fractures, though preferentially at the contact zones between schist and dolerite (Petruk, 1975; Popov et al., 1989). Native silver occurs in complex sets of microfractures, mainly at intersections with sulphide veinlets, and locally accompanied by chlorite-rich alteration. Small Ag grains (average size = 50 µm) occur in corrosion zones of early sulphides or disseminated within the schist and dolerite.

According to Marcoux et al. (2015), the paragenetic sequence shows two successive stages of mineralisation: 1) an early Fe–As stage and 2) an Ag-bearing polymetallic (Zn–Pb–Cu–Hg) stage (Figure 7.8). The early-stage mineralisation is composed of pyrite (70.3%) and arsenopyrite (6.1%). Pyrite shows rare silver inclusions (20 µm). The late polymetallic stage of mineralisation consists of dominantly sphalerite (17.9% of the mineralised material). Electronic microprobe data suggest the presence of two generations of sphalerite: an iron-free generation (<1% Fe) and an iron-rich generation (up to 8% Fe). Silver has not been detected in the sphalerite. Chalcopyrite is rare (1.8% of the mineralised material), carrying very rare Ag-poor grey copper patches (<40 µm), and Ag-free galena (2.3% of the mineralised material).

Figure 7.8 – Paragenetic Mineral Sequence in the Zgounder Deposit (Marcoux and Wadjinny, 2005)



Source: Pelleter et al., (2016)

Native silver is by far the most common silver mineral (Figure 7.9), representing 1.07% of mineralised material concentrate, and 65% to 90% of Zgounder silver. The native silver is Ag–Hg amalgam, rather than pure silver, forming inclusions 25 µm to 480 µm in size (average 150 µm to 250 µm). Electronic microprobe analyses (Marcoux and Wadjinny, 2005) revealed presence of two (2) generations of large Ag-rich patches containing 85% to 95% Ag (average structural formula $Ag_{17}Hg$), which likely corresponds to remobilisation slightly post-dating the major silver mineralisation event. The latter is characterised by smaller silver blebs containing 72% to 80% Ag (Ag_5Hg, similar to that of eugenite), which represent the majority of the native silver mineralisation at Zgounder. Native silver patches show irregular variation of Hg grade and grain size (up to 1,920 µm), and carry myriads of native silver inclusions (<5 µm). Small patches of polybasite ($Ag_{16}Sb_2S_{11}$) and pearceite ($Ag_{16}As_2S_{11}$) are rare. Tennantite and tetrahedrite are very rare phases. Silver contents are variable, but low (average: 4% Ag). Acanthite (Ag_2S) is the main silver sulphide, but is much less abundant than native silver and contains inclusions of native silver (Marcoux et al., 2015).

Figure 7.9 – Native Silver in 2020 Drill Core

Source: Audet (2021)

Tension gashes originally trapped the silver mineralisation within a NNE-oriented shear zone affecting the shale-sandstone beds (Brown Formation) that contain anomalous Ag values (Figure 7.10). The silver was likely remobilised by E-W oriented structures forming isolated Ag-mineralised lenses and fissures (Figure 7.11). The silver mineralisation extends laterally over 1,000 m and dips sub-vertically to the south. Vertical mineralised zones are offset by sub-horizontal faults with a northward movement of 10 m to 30 m, creating steps or blocks of mineralisation (Bounajma, 2002).

Based on lead isotope ratios ($^{206}Pb/^{204}Pb$ = 17.89 and $^{207}Pb/^{204}Pb$ = 15.57) measured on galena grains from the polymetallic stage of silver mineralisation, the calculated age for the Zgounder Silver Deposit is approximately 510 Ma (Marcoux and Wadjinny, 2005), using the Stacey and Kramers (1975) model. The Zgounder lead isotopic ratios are similar to those measured at Imiter ($^{206}Pb/^{204}Pb$: ≈18.10; $^{207}Pb/^{204}Pb$ ≈15.5), with a mineralisation age of approximately 550 Ma (Late-Proterozoic; Pasava, 1994; Cheilletz et al., 2002). The similar ages of Zgounder and Imiter (eastern Anti-Atlas, Morocco), imply that the former is another example of a Neoproterozoic epithermal deposit in the Anti-Atlas of Morocco (Baroudi et al., 1999; Essarraj et al., 1998).

Figure 7.10 – Tension Gashes Filled by Sphalerite and Galena

Source: Goldminds (2018)

Figure 7.11 – E-W Vertical Fault

Filled with Galena in the Ceiling (top) and Tension Gashes Filled with Fine Pyrite (bottom

Source: Audet (2021)

8 DEPOSIT TYPES

This Section has been summarised and/or largely extracted from the Report available on SEDAR entitled: "Technical Report and Updated Mineral Resource Estimate of the Zgounder Silver Project, Kingdom of Morocco, prepared for Aya Gold & Silver Inc., dated January 28, 2022, prepared by P&E Mining Consultants Inc., Report # 415."

Zgounder is a Neoproterozoic age, sedimentary rock-hosted, low-sulphidation epithermal silver deposit.

Epithermal deposits are a type of mineral deposit that consists of veins and disseminated mineralisation formed at or near (≤1.5 km) the Earth's surface (Hedenquist, 2000; Taylor, 2007; Figure 8.1). The deposits occur in association with hot springs and young volcanic centres. The host rocks are generally volcanic and volcaniclastic sedimentary, sedimentary and metamorphic rocks. The mineralisation is dominated by silver and gold. However, copper, lead and zinc may also be present in variable amounts.

Epithermal gold deposits are classified based on the sulphidation state of the sulphide mineralogy, as being one of three (3) sub-types (Hedenquist, 2000; Taylor, 2007):

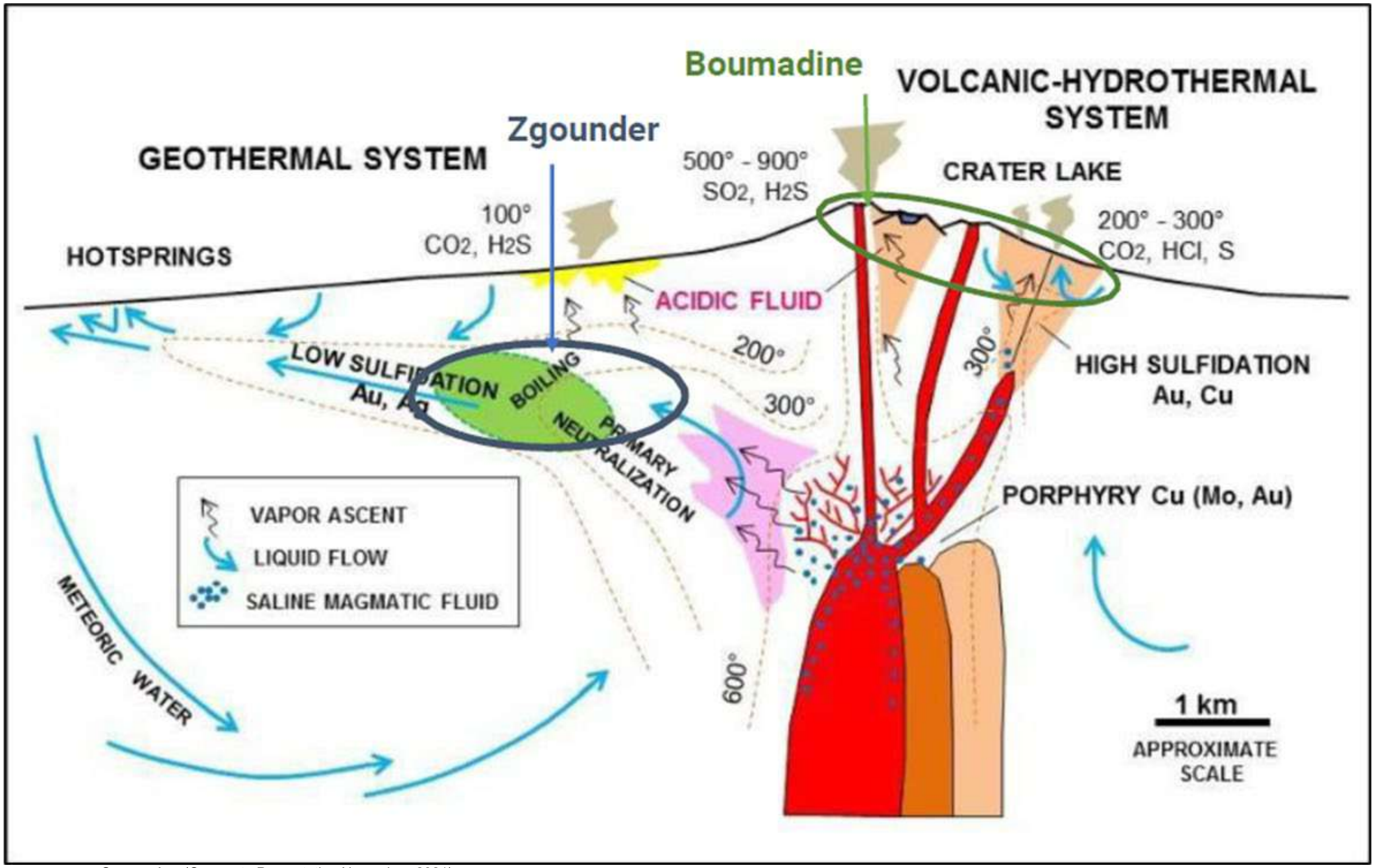
1. Low-Sulphidation: previously called adularia-sericite, these low-sulphidation sub-type deposits are considered to be formed by near-neutral pH fluids, as a result of being dominated by meteoric waters, but containing some magmatic C and S;

2. High-Sulphidation: previously called quartz-(kaolinite)-alunite, alunite-kaolinite, enargite-Au, or high-sulphur deposits, these highly acidic deposits generally occur proximal to magmatic sources of heat and volatiles and form from acidic hydrothermal fluids containing magmatic S, C and Cl; and

3. Intermediate-Sulphidation: some deposits with mostly low-sulphidation characteristics have sulphide mineralised assemblages that represent a sulphidation state between that of high-sulphidation and low-sulphidation deposits.

In low-sulphidation epithermal systems, silver and gold occur in vein deposits that contain only up to a few percent sulphides. The principal gangue minerals are calcite, chlorite, adularia, barite, rhodochrosite, fluorite and sericite. Calcite is characteristic, because of the low acidity of the hydrothermal mineralising fluids. Low-sulphidation deposits form in fault zones, sedimentary rocks and vein complexes relatively more distal from magmatic intrusions (Taylor, 2007).

Figure 8.1 – Low-Sulphidation Epithermal Deposit Model



Source: Aya (Corporate Presentation November, 2021)

The Zgounder Deposit formed during two (2) distinct stages of hydrothermal fluid alteration and mineralisation (Essaraj et al., 1998, 2016, 2017):

- Deposition of quartz with minor biotite and As-Co minerals from a variety of H_2O-CO_2-CH_4 rich fluids in equilibrium with the metasedimentary host rocks. The fluids were at high temperatures (400ºC to 450°C) over a wide range of pressures, during the early brittle deformation of the Brown Formation, following emplacement of the Askaoun Granite; and
- The main phase of (Cu-Zn)-Ag (Hg) mineral deposition. Silver deposition occurred following crystallization of quartz-sphalerite-chalcopyrite veins, but the Cu-Zn and Ag(Hg) mineralising fluids were NaCl-$CaCl_2$ brines at minimum temperatures of about 160°C to 200°C, during a period of hydrothermal albitization.

The mineralising fluids evolved from an early high-temperature stage of likely magmatic origin, followed by a rapid decrease in temperature and dilution and cooling of metal-laden Na-Ca brines, due to mixing with highly saline groundwater in an evolving hydrothermal fluid system. Based on alteration of magmatic zircon from felsic intrusions associated with the mineralisation, Pelleter et al., (2016) proposed an age of 564 ± 15 Ma for the timing of the hydrothermal albitization and mineralisation in the Zgounder Silver Deposit.

According to Marcoux et al. (2005), Zgounder is an epithermal silver–mercury deposit very rich in silver. The high mercury content of the native silver demonstrates that this mineral is primary hypogene mineralisation and not a secondary mineral produced by cementation. Zgounder shows strong geological, structural, mineralogical and geochemical similarities to the Imiter Ag-Hg deposit. Zgounder and Imiter could therefore be two manifestations of a regional epithermal Neoproterozoic mineralisation event within the Anti-Atlas Domain.

9 EXPLORATION

This Section has been summarised and/or largely extracted from the Report available on SEDAR entitled: "Technical Report and Updated Mineral Resource Estimate of the Zgounder Silver Project, Kingdom of Morocco, prepared for Aya Gold & Silver Inc., dated January 28, 2022, prepared by P&E Mining Consultants Inc., Report # 415."

9.1 Geological Activities

In 2018 and 2019, ZMSM mapped and sampled the Northern area of the Property and excavated seven small trenches at the eastern sector of the Zgounder Property. No other significant geological work was performed until the start of the drilling program in September 2020.

9.1.1 2018-19 Surface Sampling

ZMSM undertook surface sampling at the North Zone of the Zgounder Mine. Samples of the Brown Formation were composed mainly of metamorphosed argillite (schist) and sandstone. The North Zone is affected by NE-SW, NW-SE and E-W fractures. The tension gashes within a NNE-oriented shear zone contain generally elevated Ag values.

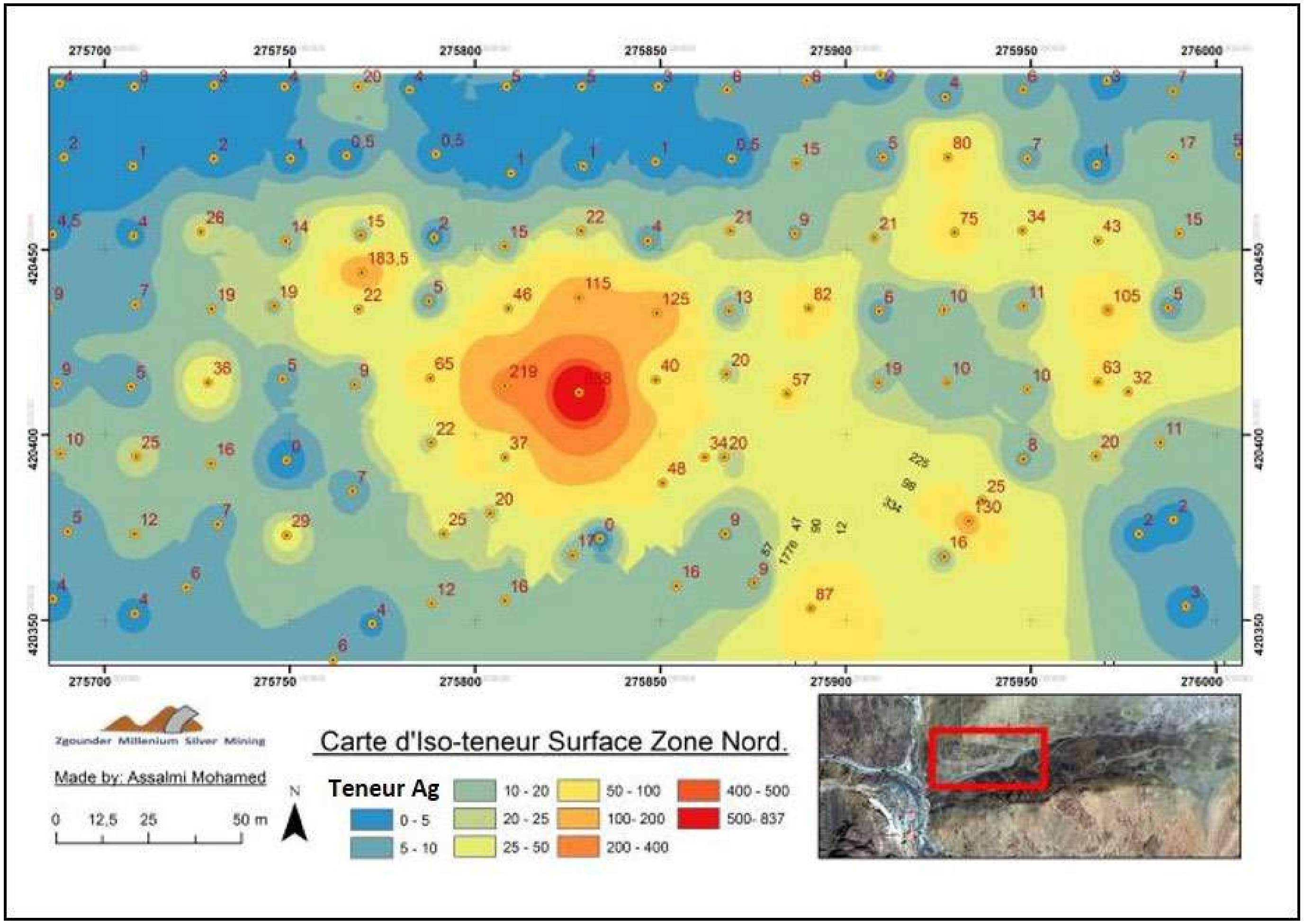
Systematic sampling was done on a regular grid spacing of 20 m by 20 m. The direction of the sampling profiles was North-South, perpendicular to the direction of stratification (Figures 9.1 and 9.2).

The West Zone, the zone west of the Zgounder River, was also subject of surface sampling (Figure 9.3). The surface samples were taken from fractures-oriented WNW-ESE, NW-SE and NNW-SSE in the sandstone.

9.1.2 Surface Mapping

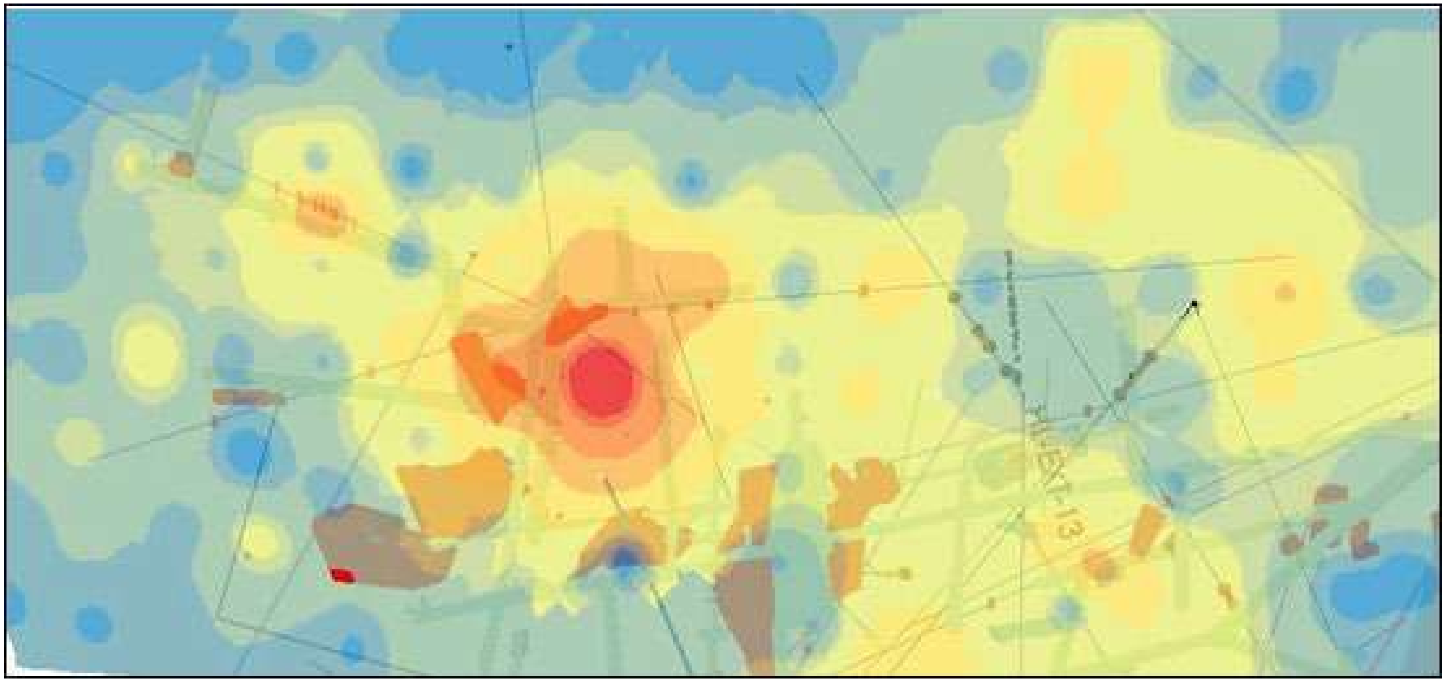
Surface mapping was completed on the West Zone and approximately 35 measurements of fault and fracture directions recorded. The main fault/fracture groups are oriented WNW-ESE, NW-SE, and NNW-SSE (Figures 9.4 and 9.5).

Figure 9.1 – Isocontours Map – Ag, Surface Sampling – North Zone



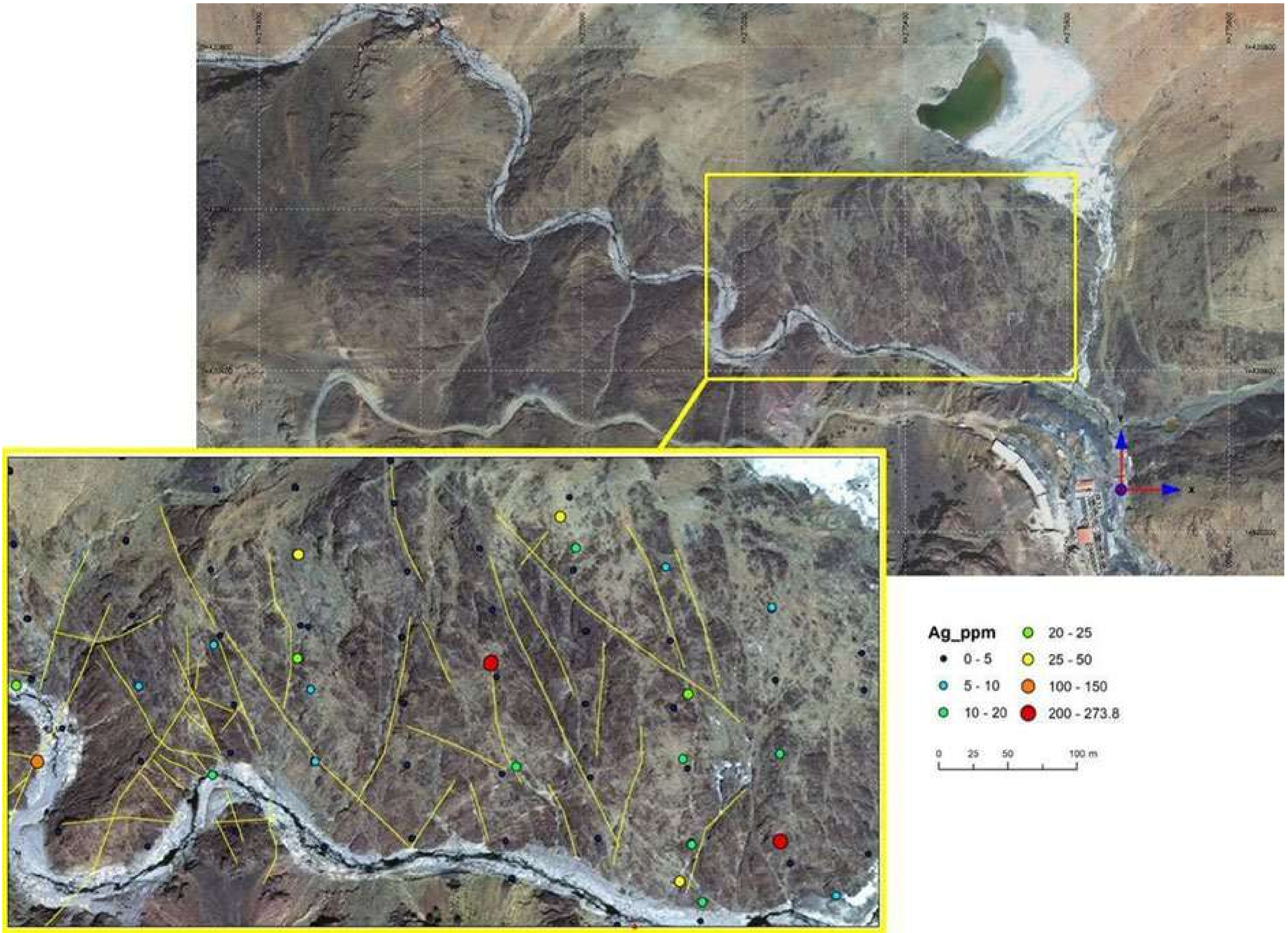
Source: GoldMinds and SGS (2018)

Figure 9.2 – Ag Surface Sampling Results Superimposed on Mineralised Envelope Positions



Source: GoldMinds and SGS (2018)

Figure 9.3 – Ag Surface Sampling Results Superimposed on Topography

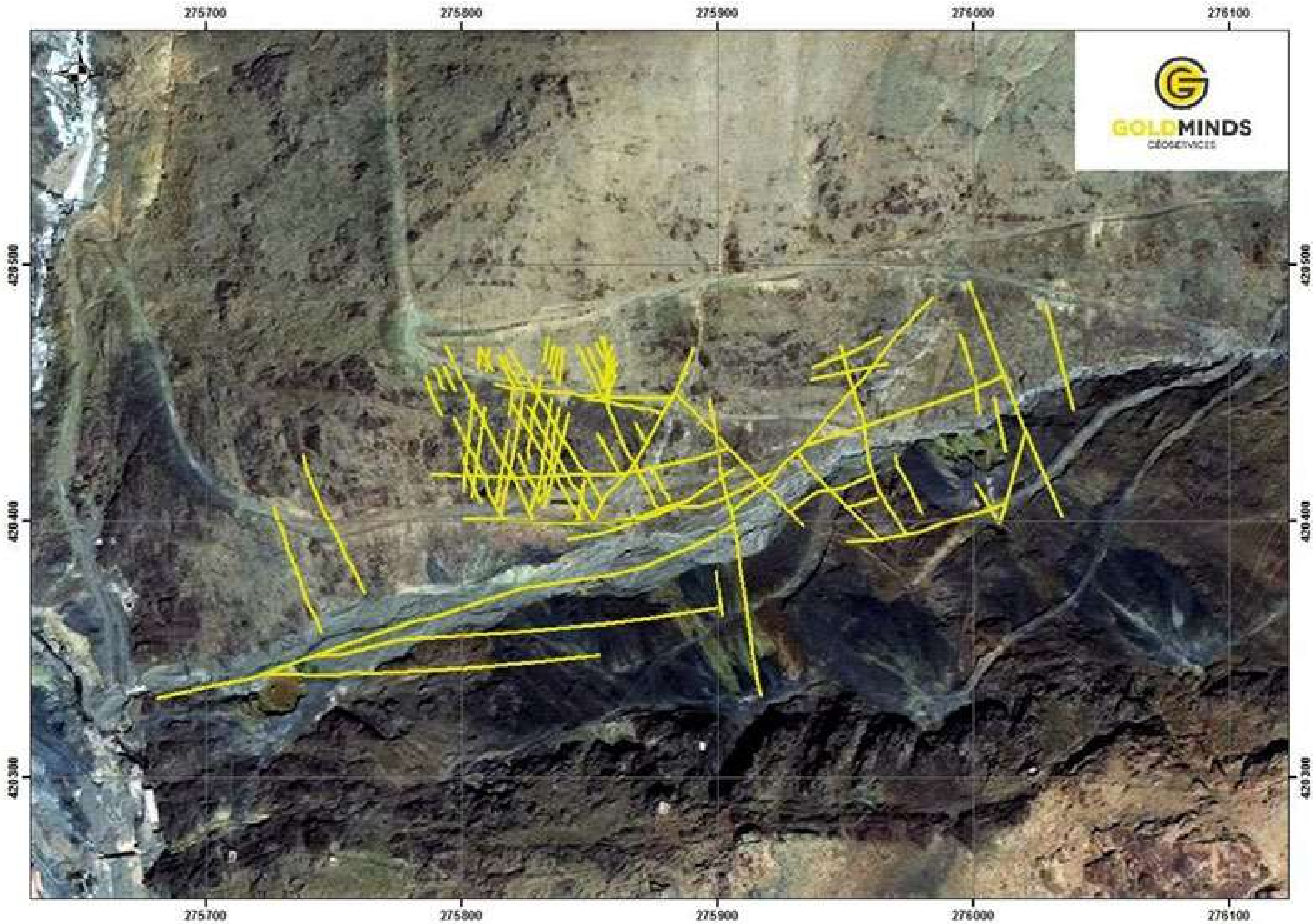


Source: GoldMinds and SGS (2018)

Figure 9.4 – Faults and Fractures on the West Zone

Source: GoldMinds and SGS (2018)

Figure 9.5 – Faults and Fractures in the North Zone



Source: GoldMinds and SGS (2018)

9.1.3 2019 Trenching Program

A trenching program was initiated in June 2019 on the East Zone of the Zgounder Mine (Figure 9.6). The trenches were excavated with a saw and oriented to cross-cut mineralised structures trending E-W, NNW-SSE and filled by quartz-carbonate with traces of malachite and pyrite. Trenches were up to 80 m long, sample lengths were one metre, and their locations determined using a handheld GPS. Trench samples returned up to 576 g/t Ag (Table 9.1).

Table 9.1 – 2019 Trench Sampling Highlights

Trench	Samples	From	To	Length	Ag
		(m)	(m)	(m)	(g/t)
TR02	E34 to E41	33	41	8	150
TR03	E3	2	3	1	576
TR03	E10 to E13	9	13	4	500
TR03	E80	79	80	1	526
TR03	E31	30	31	1	184

Source: Aya press release dated October 7, 2019

9.2 3-D Laser Scanning Survey

A 3-D laser scan survey was conducted for the first time at Zgounder Mine in April 2013, using a laser scanner Faro Photon 120 and Faro Focus. Underground drifts, adits (levels and sublevels), and openings were subject to the 3-D laser scanning.

The five (5) main levels (2000, 2100, 2125, 2150 and 2175) and five (5) sublevels (Level 2025, 2030N, 2035E, 2050S, 2175, and 2087), and stopes at levels 2035E, 2087 and 2100 (accessible at that time) were scanned (Figures 9.7 to 9.13). Some of these sublevels were only partially surveyed, due to lack of access.

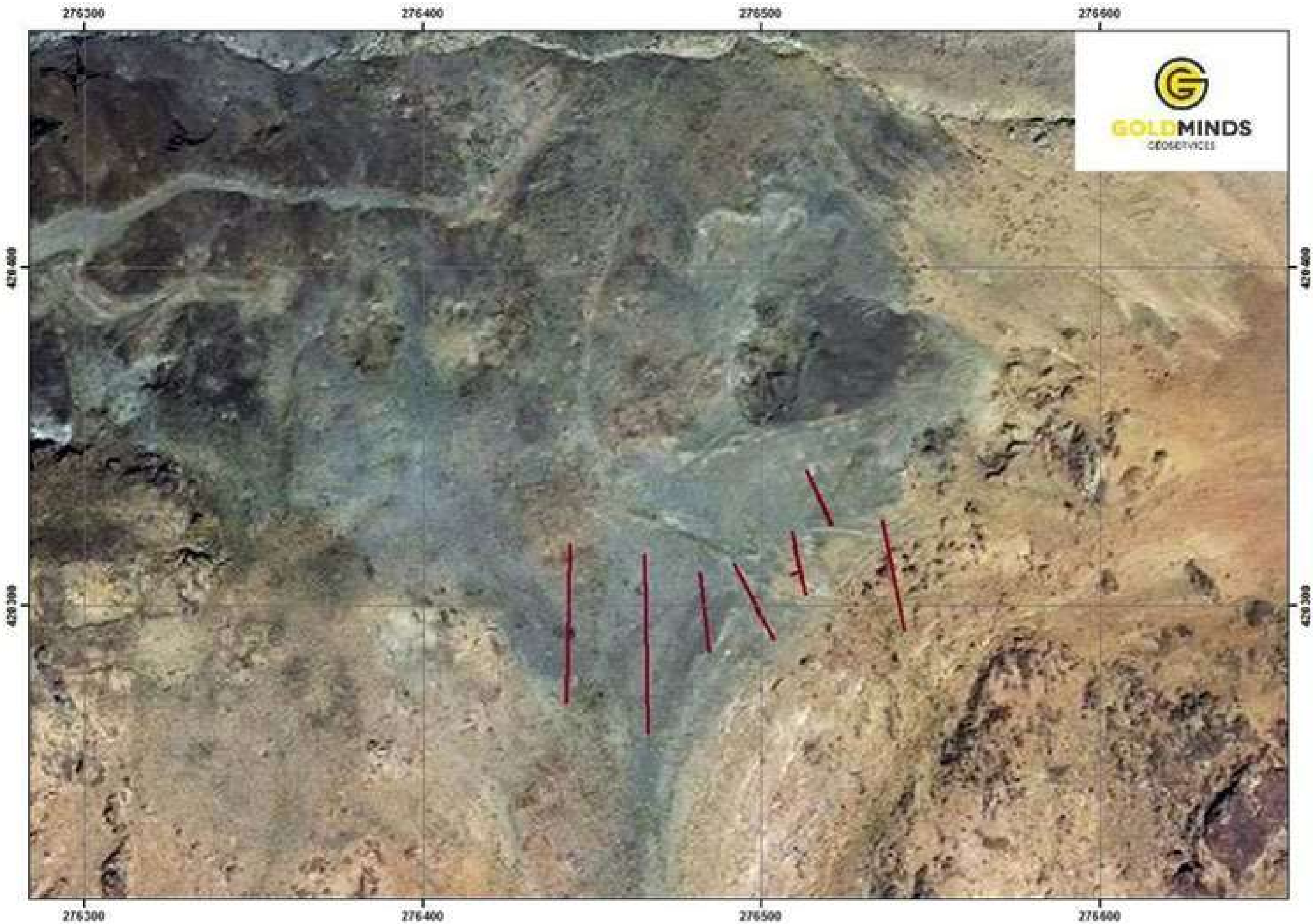
In November 2021, Aya updated the 3-D laser scan of the entire underground mine and operation.

The results of the 3-D monitoring survey have been integrated with the 3-D Mineral Resource model using GEMSTM (modelling and estimation software) and in LeapfrogTM Geo (geological modelling software). This 3-D modelling allowed visualisation of the mined material within the mineralised envelopes and to account for this missing volume during Mineral Resource estimation (see Section 14 of this Technical Report).

9.3 Airborne Geophysics

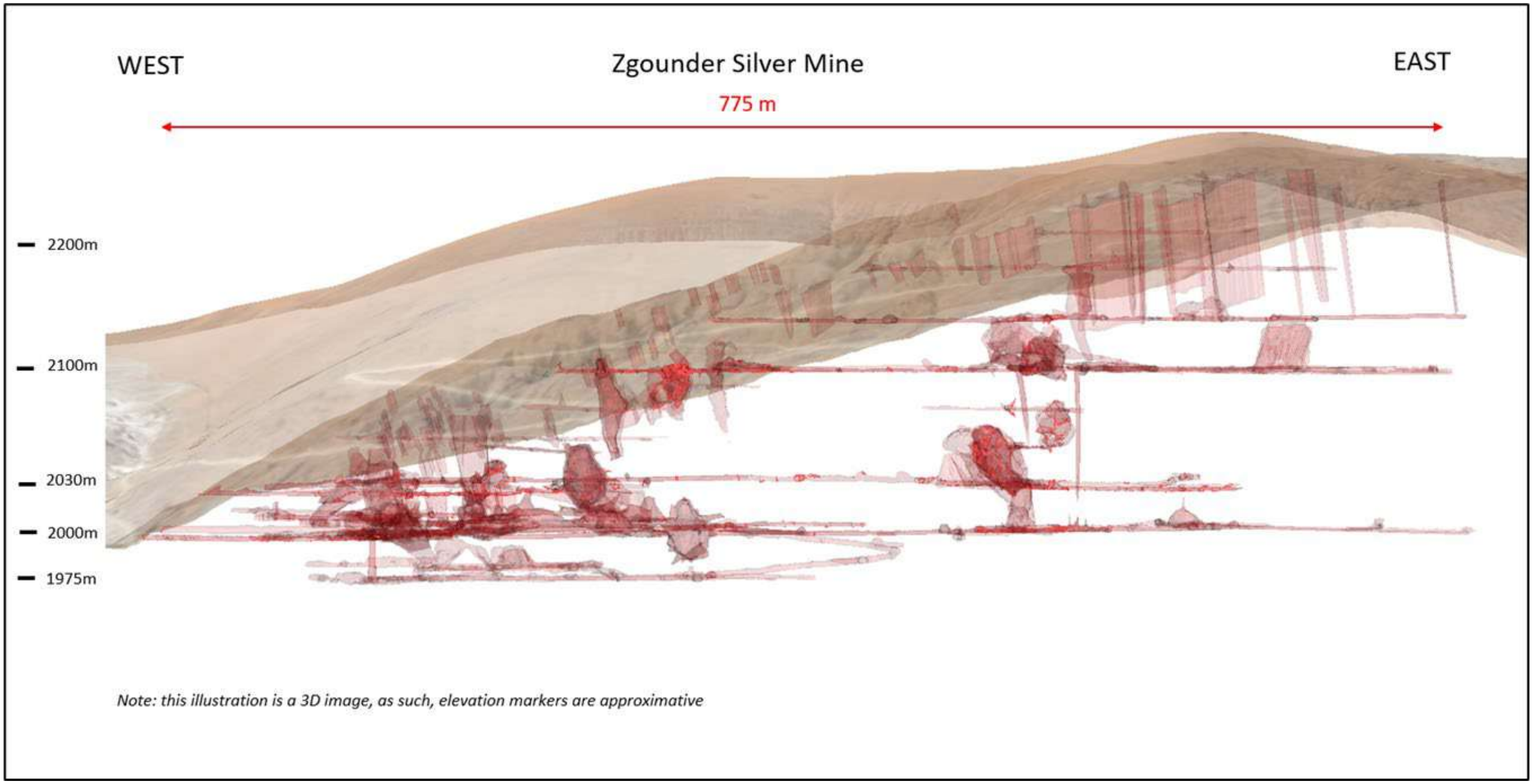
In a press release dated November 15, 2021, Aya announced that it had mandated Geotech to fly a VTEM™ heliborne terrain, magnetic and radiometric geophysical survey over the entire Zgounder Regional Project before year end 2022.

Figure 9.6 – Trench Locations on the East Zone



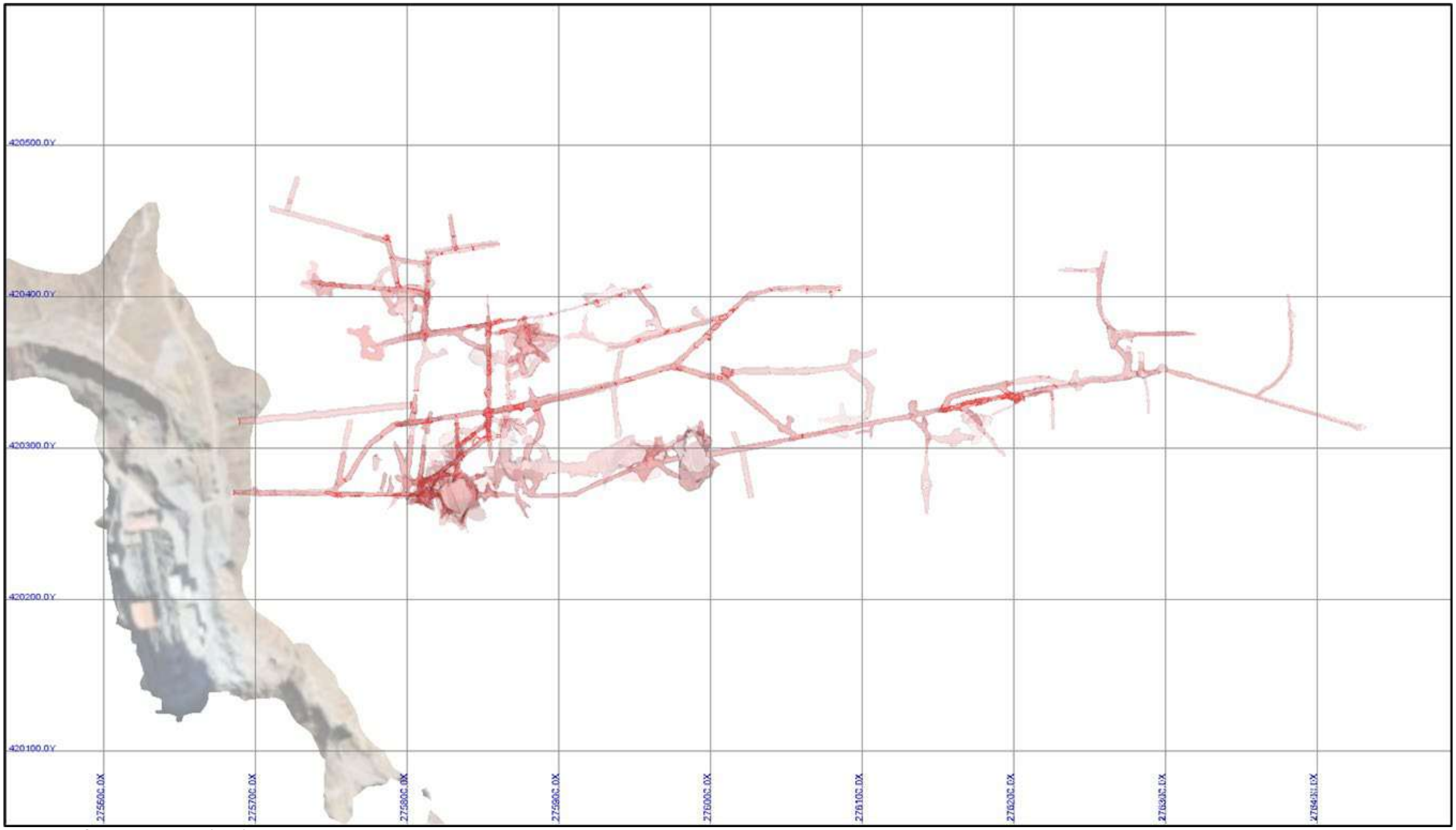
Source: GoldMinds and SGS (2018)

Figure 9.7 – 3-D Laser Scan of Various Levels and Sublevels at Zgounder



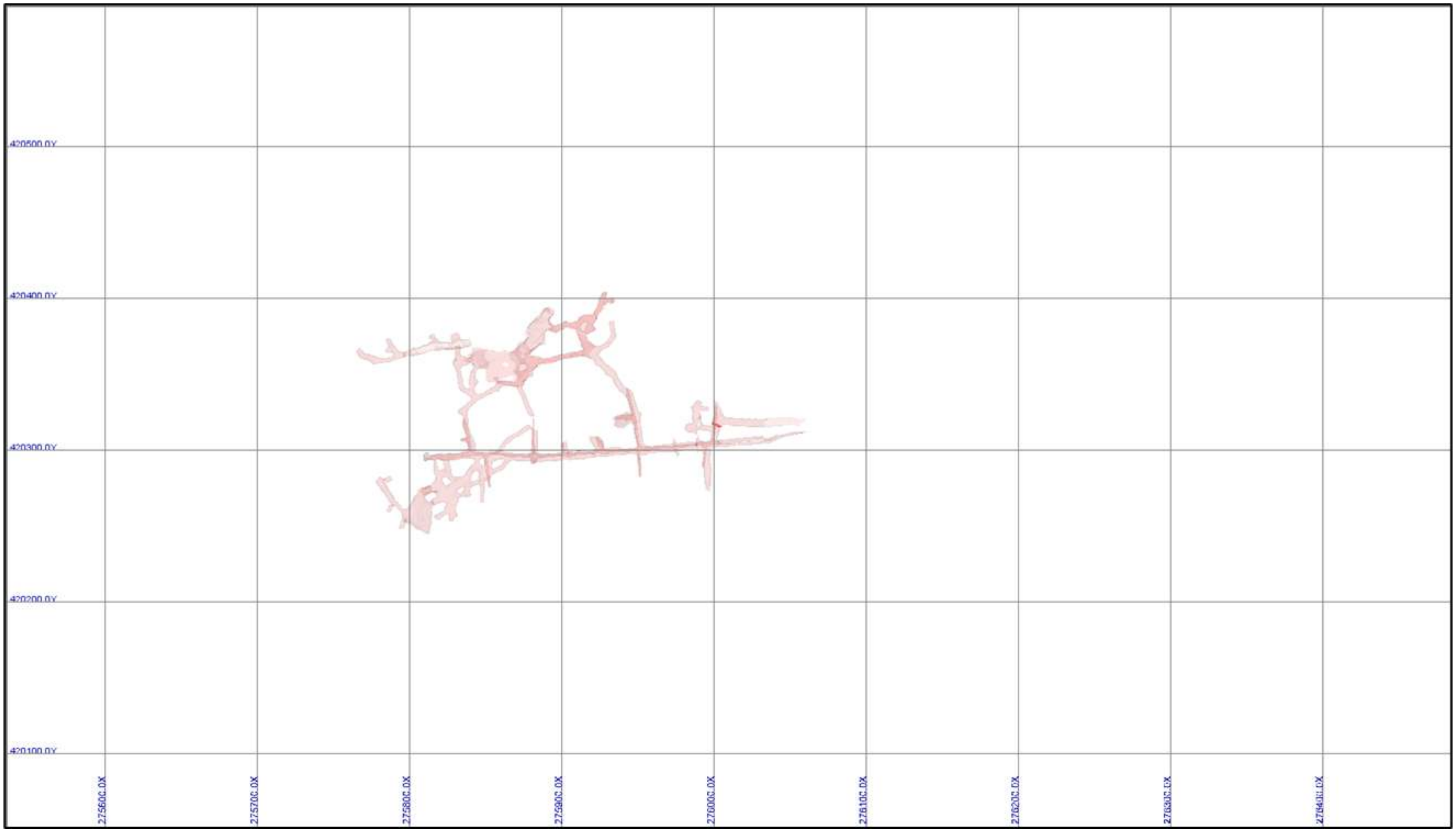
Source: Aya website (2021)

Figure 9.8 – 3-D Scan of Level 2000, Zgounder Mine



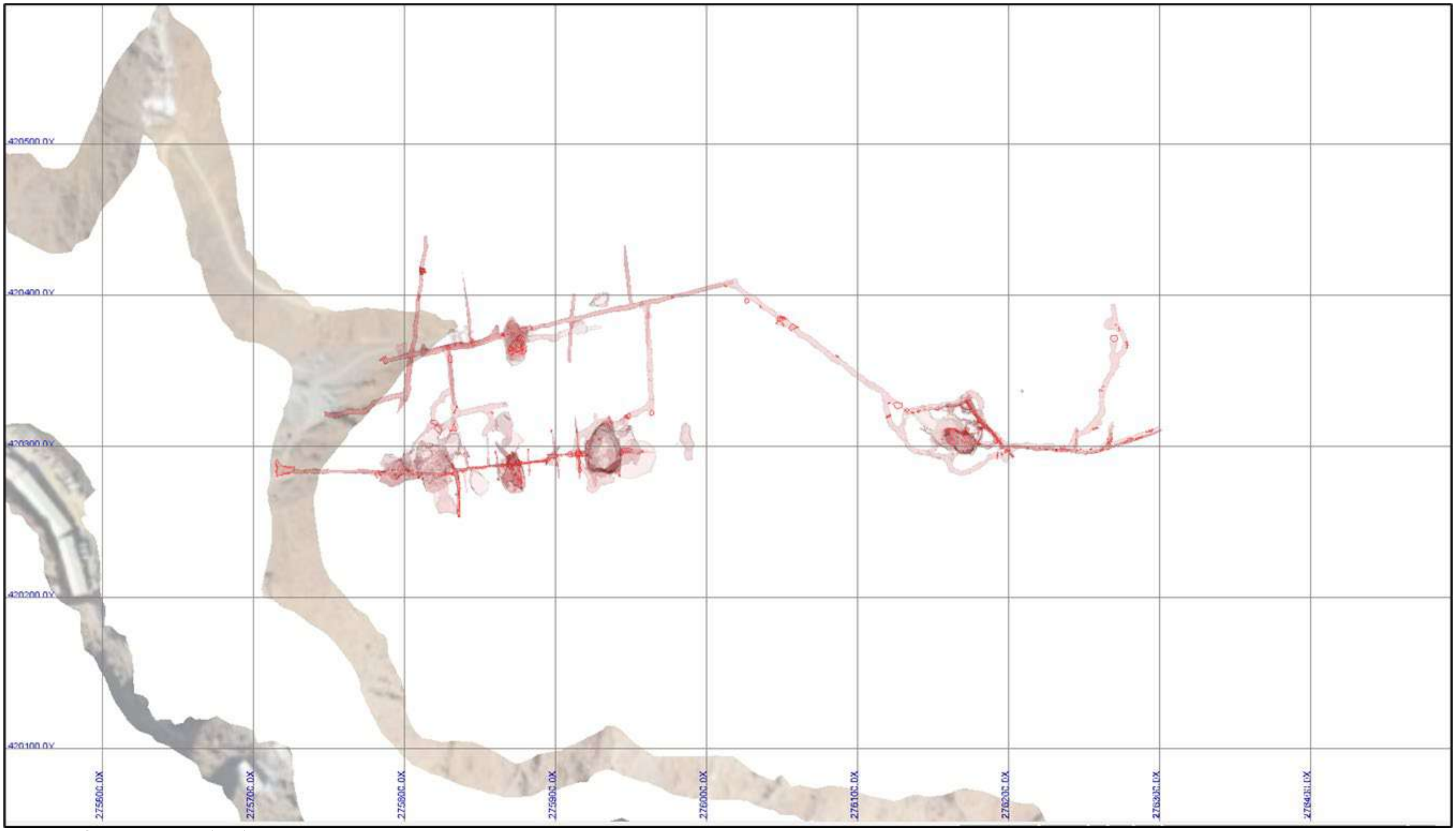
Source: Aya website (2021)

Figure 9.9 – 3-D scan of Level 1975, Zgounder Mine



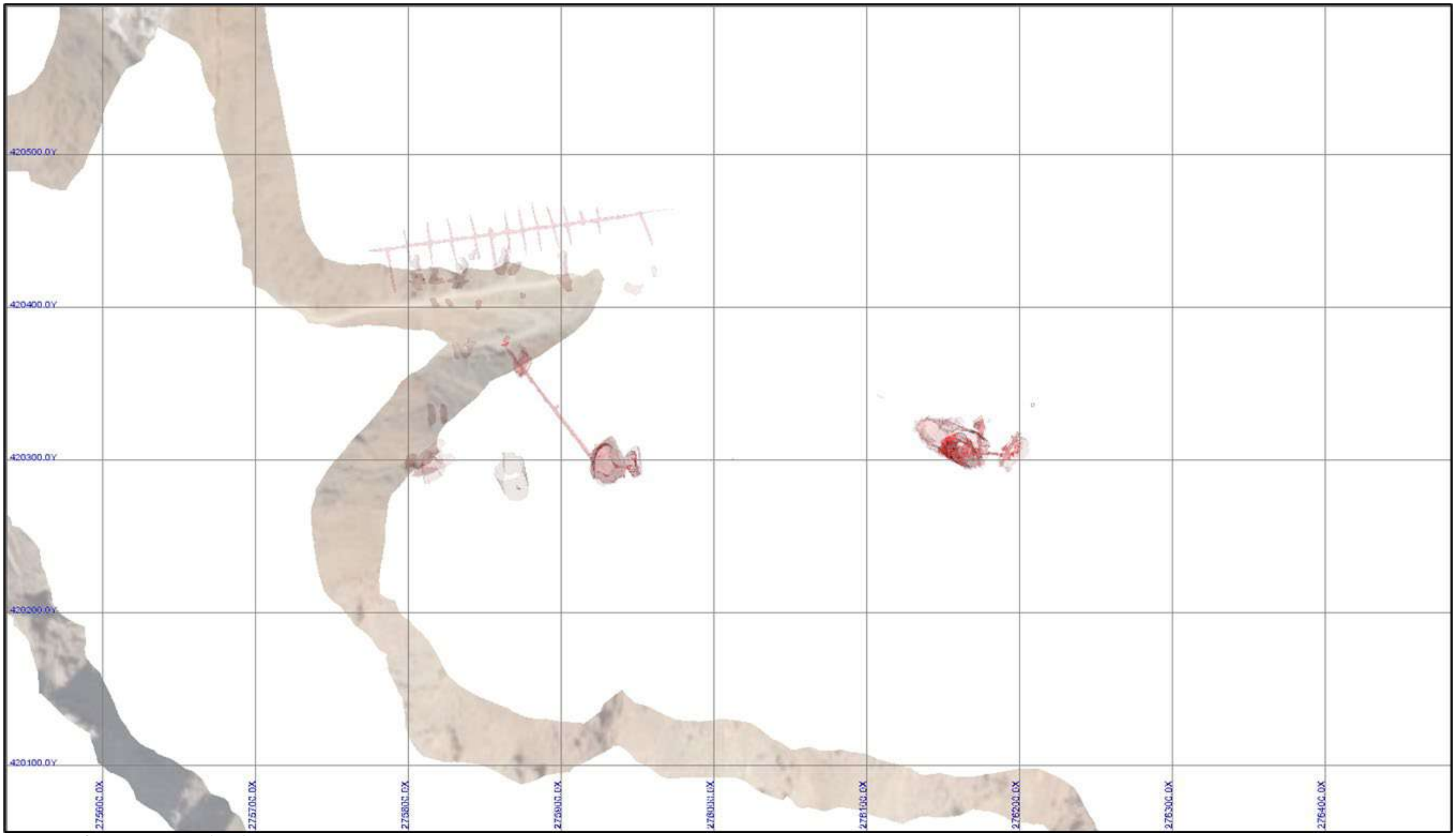
Source: Aya website (2021)

Figure 9.10 – 3-D Scan of Level 2035, Zgounder Mine



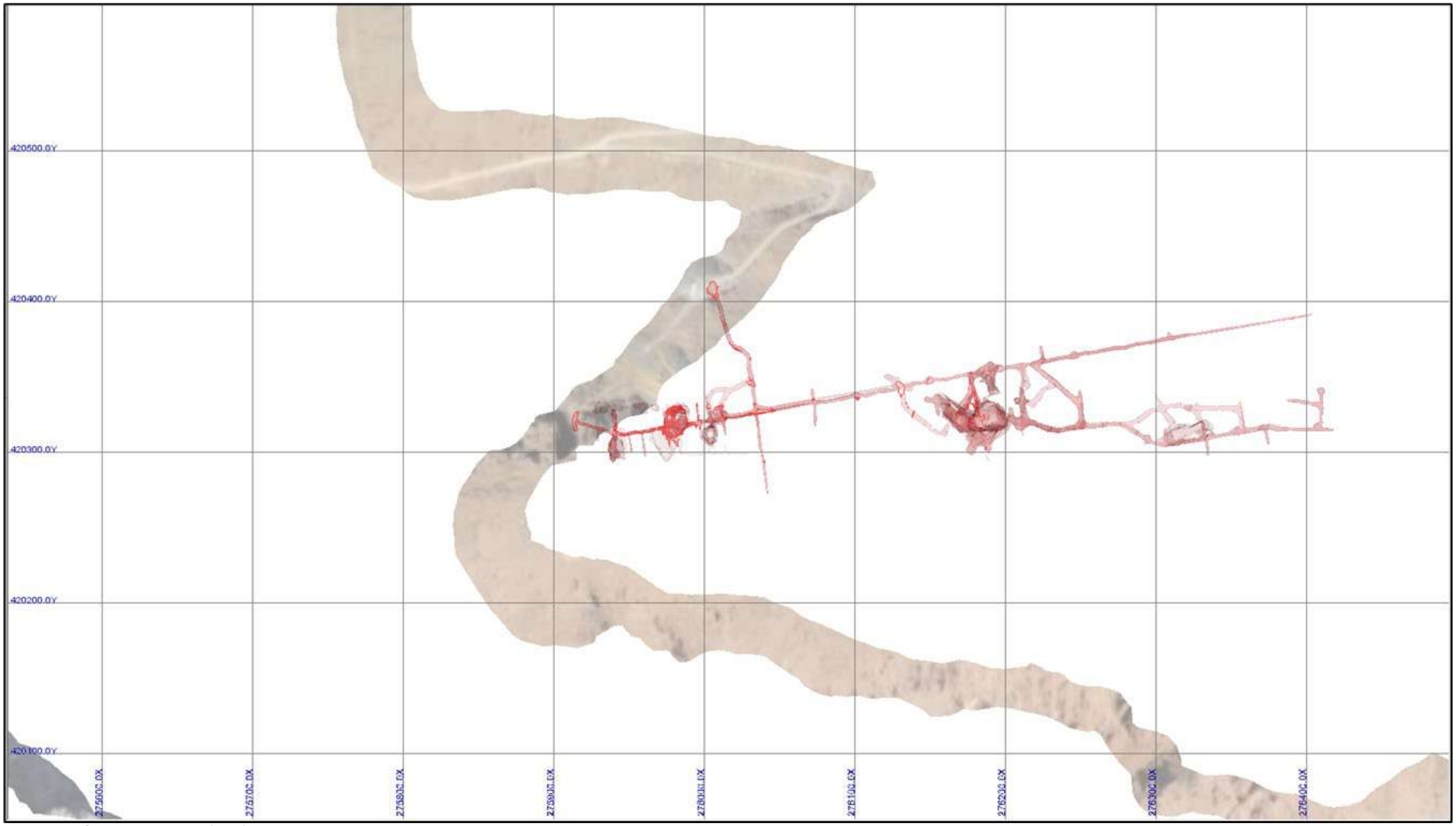
Source: Aya website (2021)

Figure 9.11 – 3-D Scan of Level 2050, Zgounder Mine



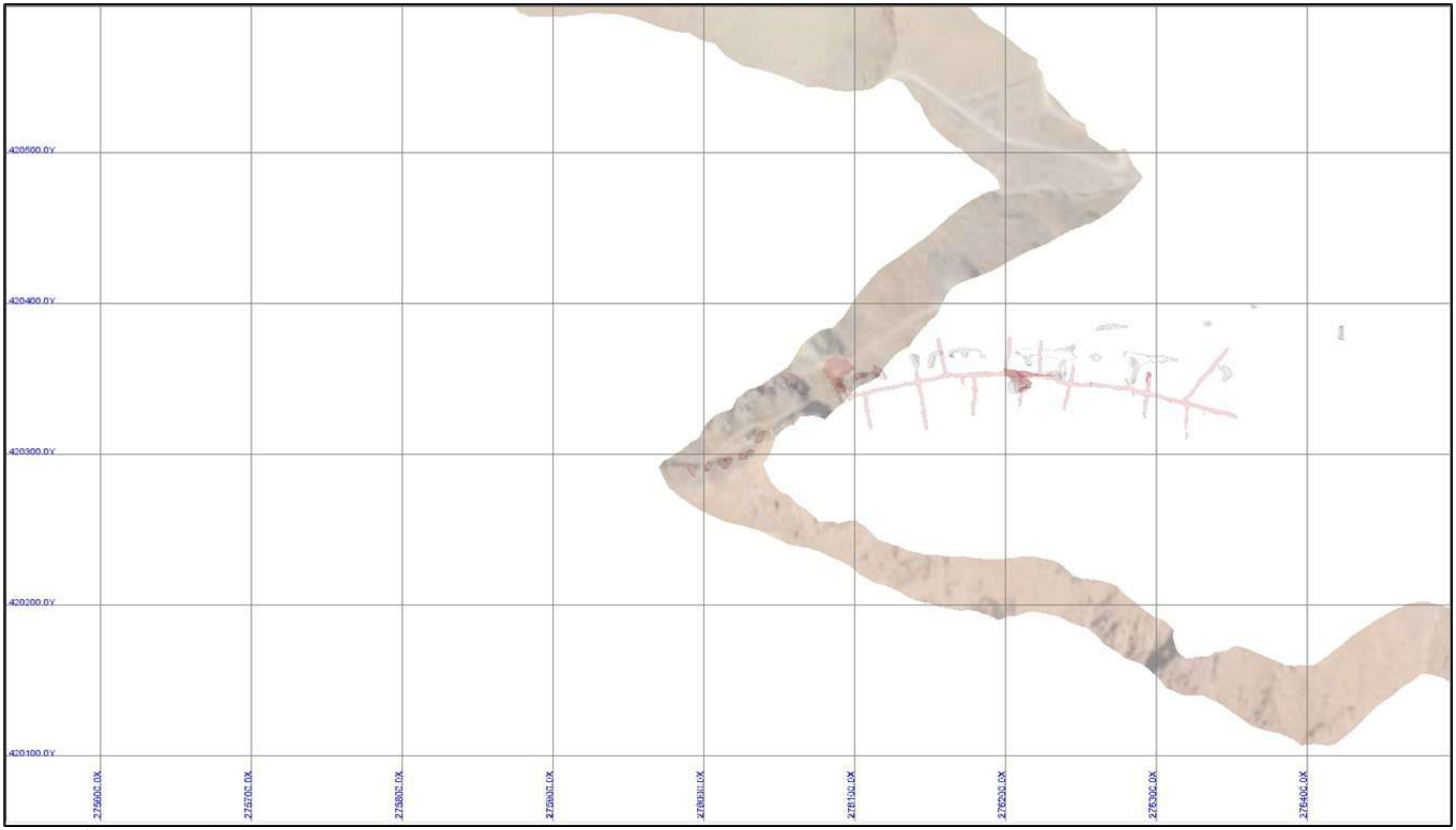
Source: Aya website (2021)

Figure 9.12 – 3-D Scan of Level 2100, Zgounder Mine



Source: Aya website (2021)

Figure 9.13 – 3-D Scan of Level 2150, Zgounder Mine



Source: Aya website (2021)

10 DRILLING

This Section has been summarised and/or largely extracted from the Report available on SEDAR entitled: "Technical Report and Updated Mineral Resource Estimate of the Zgounder Silver Project, Kingdom of Morocco, prepared for Aya Gold & Silver Inc., dated January 28, 2022, prepared by P&E Mining Consultants Inc., Report # 415."

10.1 Summary

Several exploration drilling campaigns have been completed at Zgounder, starting in the 1980s, and subsequently in 2013, 2015, and 2017 through 2021. A 19,000 m drilling program on surface and underground began in September 2020 and ended mid-February 2021. In the balance of 2021, an additional 41,914 m were drilled on surface and underground. The drilling database consisted of a total of 3,148 holes for 136,816 m. A breakdown of the hole numbers, different hole types (including channels and trenches), and total meterage is presented in Table 10.1.

Table 10.1 - Zgounder Drilling Database per Hole Type

Hole Type	No. of Drill Holes	Metre (m)
DDH Surface	339	69,530
DDH Underground	110	14,832
Reverse Circulation (RC)	30	3,462
T28	1,920	37,779
YAK-T28	84	1,867
Channels	658	9,071
Trenches	7	275
Total	**3,148**	**136,816**

The collar locations for the holes are presented in Appendix G and the mineralised intersections using a cut-off grade of 75 g/t Ag are presented in Appendix H of P&E (Report # 415, January 2022).

Percussion drilling using air compressed hammer (T28 and YAK-T28) are routinely used for production purposes and also for exploration. Data gathered from T28 is not only used for the ongoing mineral resource estimations, it also delineates new mineralised areas for short-term mine planning. A total of 2,004 T28/YAK-T28 exploration and production holes for 39,646 m have been drilled historically and until the end of September 2021.

In addition to drilling, underground wall and roof channel sampling were performed on all adits, galleries and stopes. A total of 658 channels for 9,071 m are used in the drilling database that supports the Mineral Resource Estimate presented in Section 14 of this Technical Report.

10.2 Percussion Drilling

Surface reverse circulation ("RC") and underground percussion drill programs ("T28") were completed in 2015, 2016 and 2018 to 2019. The 2015 RC percussion hole data have not been included in the Zgounder database, due to issues with that program.

10.2.1 2016 TO 2017 – T28 PROGRAMS

In 2016, a total of 1,598.4 m was drilled using a T28 percussion hammer rig on the 2000 and 2100 Levels of the Zgounder Mine.

The percussion holes drilled in the North Zone intersected some mineralised intervals and confirmed the extension to the east of Panel 9. The data highlighted a new zone to the northeast of mining operation above the 2100 Level. The holes have been drilled from an exploration raise in a fan and alongside the drift to define the shape of the body. Furthermore, the area has been drilled to define the geometry of the Y6 with an extension to the east of that body on Level 2100. New findings occurred on the 2030 and 2000 Levels.

Independent analysis was conducted at Bourlamaque Laboratory in Val d'Or (Quebec) on ten samples from hole 2100-T28-17-64. These independent samples returned a mineralised interval grading 2.63 kg/t Ag over 12 m. The hole was drilled on elevation 2,106, on azimuth 200º N at 17º dip. The mineralised zone is irregular and sub-vertical in orientation.

Lead and zinc sulphide minerals were observed at the surface to the west near the Zgounder River. However, the percussion holes drilled close to the mine entrance did not yield significant results, which suggests a plunge component to the silver mineralisation. On the other hand, holes drilled to the north of the Talat N'ouna River intersected some mineralised occurrences near the surface.

10.2.2 2018 TO 2019 – RC PROGRAMS

The reverse circulation drilling program consisted of 32 drill holes totalling 3,611 m that were drilled from the surface on the East Zone of the Zgounder Property. The RC drilling campaign aimed to provide a better understanding of the distribution, orientation and thickness of the mineralised structures, and to explore the vertical extensions of the exposed mineralised structures. The results confirmed the continuity of the known mineralised envelopes and new occurrences hosted in the same major East to West oriented structures.

The RC drilling results of this campaign led to an underground diamond drilling program to explore the vertical extensions of the mineralisation in the western and the central part of the Zgounder Mine. Holes ZG-RC-1, -2, -3, -4, -5, -6, -7, and -9 intersected silver mineralisation below 50 g/t, in attempt to expand mineralised zones to the south. Drill hole ZG-RC-12B lacked core sample return due to faulting. Drill hole 8 was not drilled.

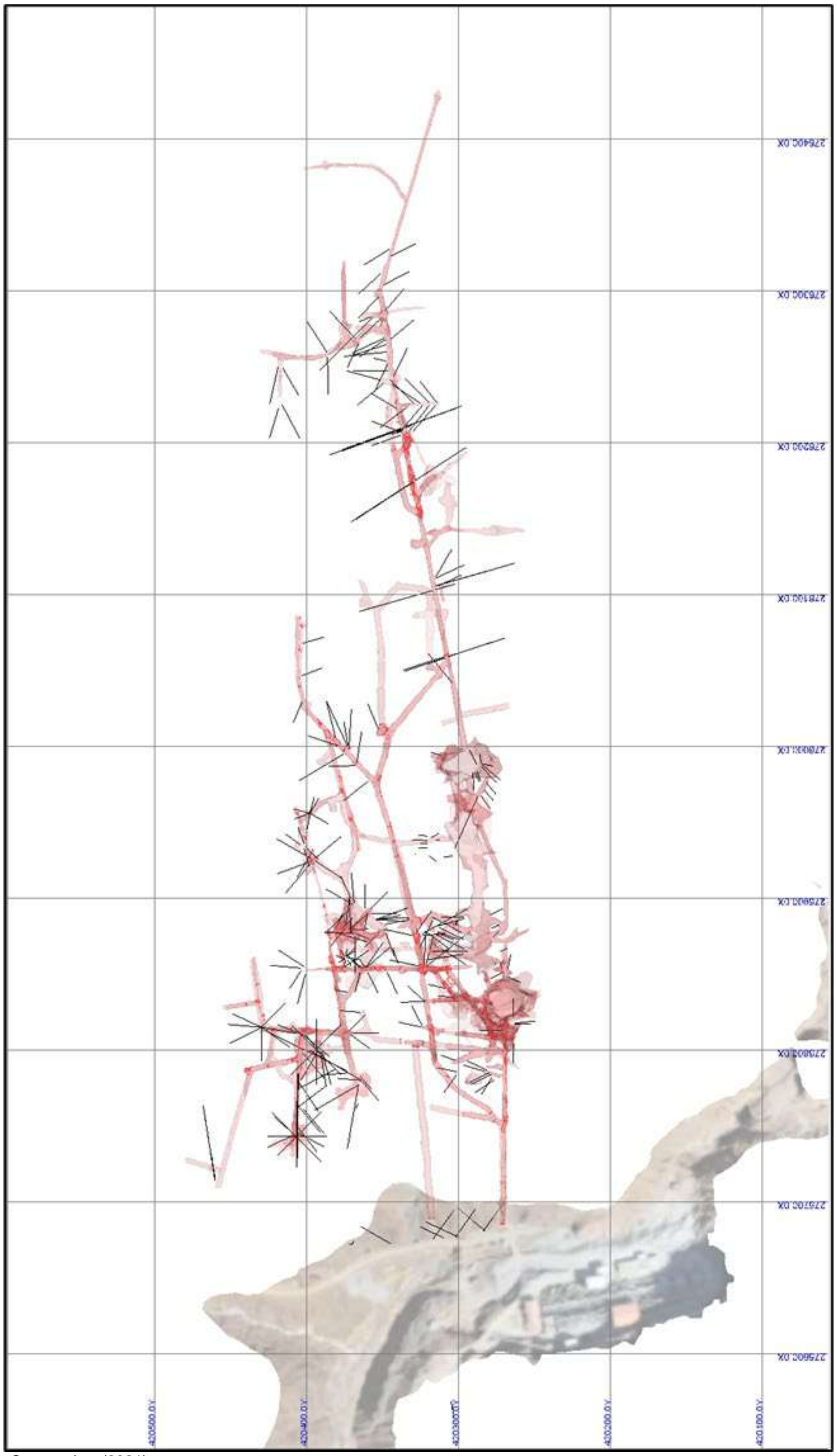
10.3 2020 to February 2021 – Percussion Drilling (T28) and Diamond Drilling

Aya also conducted localised definition drilling using T28. From the diamond drilling program, drill hole ZG-20-30, which intersected 1,946 g/t Ag over 9 m extended the strike to the east. Furthermore, drill hole ZG-20-31, located east of ZG-20-30, intersected 688 g/t Ag over 17.5 m.

Drill holes ZG-SF-20-03 and -07 from the underground drill program confirm extensions of mineralised lenses toward the west at underground Level 1975. Additionally, drill hole ZG-SF-20-15 intersected several significant high-grade intervals including 1,505 g/t Ag over 6 m and 855 g/t Ag over 5.5 m, confirming high-grade silver mineralisation at depth to the east and below the 1,975 m level. Drill holes ZG-SF-20-25 and ZG-SF-20-23 intersected 2,728 g/t Ag and 4,517 g/t Ag over 6 m and 3.5 m, respectively, below the current Mineral Resource and to the east. Drill hole ZG-SF-20-18 intersected 854 g/t Ag over 12 m below the current Mineral Resources.

Figure 10.1 shows the location the T28 holes drilled on the Level 2000, Figure 10.2 depicts percussion drilling and the collecting of chips samples, each sample is 1.2 m in length and Figure 10.3 shows the final bagged sample. T28 drill hole traces and point cloud drill data >100 g/t Ag is presented in Figure 10.4. Selected underground drill locations are presented in Figure 10.5.

Figure 10.1 – Plan View of the Percussion Drill Holes (T28) on Level 2000



Source: Aya (2021)

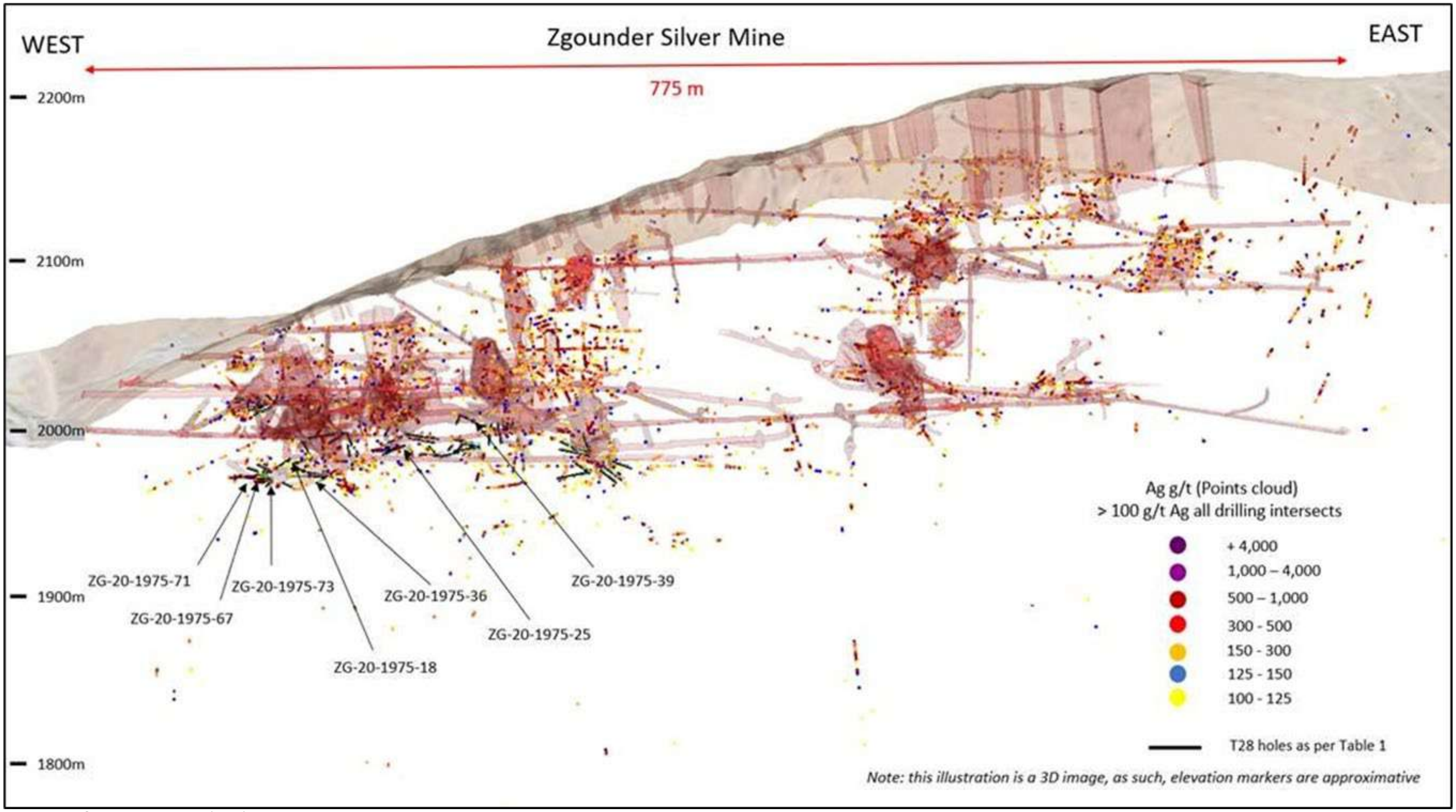
Figure 10.2 – Typical Surface Percussion Drilling at Zgounder Silver Mine

Source: Aya (2021)

Figure 10.3 – Underground T28 Percussion Drill Hole Samples

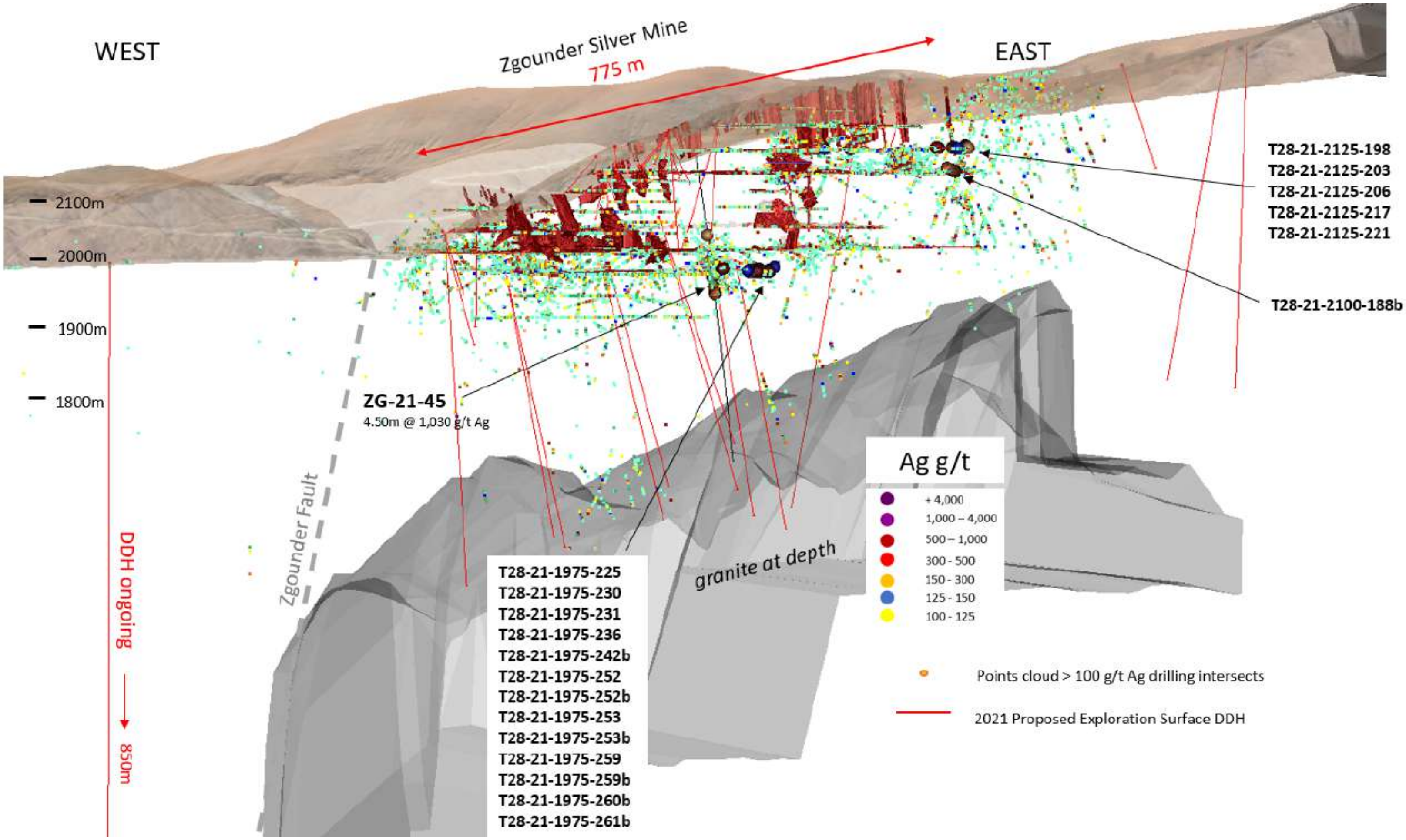
Source: Aya (2021)

Figure 10.4 – T28 Drill Hole Traces and Point Cloud Drill Data >100 g/t Ag



Source: Aya website (2021)

Figure 10.5 – Zgounder Post-February 2021 T28 Drilling Results



Source: Aya (Press Release dated October 5, 2021 and Corporate Presentation, November 2021)

10.4 Post-February 2021 Percussion Drilling (T28)

Under the 2021 exploration program, a total of 42,000 m was planned to be drilled at Zgounder. Two (2) T28 percussion drill rigs were in operation, in addition to more numerous diamond drill rigs. In total, 351 T28 drill holes were completed totalling 7,926.60 m in the post-February 2021 period.

T28 underground electric percussion drill holes at Zgounder intersected high-grade silver mineralisation along strike to the east and at depth below the previous Mineral Resources. These holes were drilled on the 1975 m, 2100 m and 2125 m Levels to the east in the mine. Significant intercepts from the T28 underground drilling at Zgounder (in core lengths) are as follows:

- T28-21-1975-120: intersected 1,599 g/t Ag over 8.4 m, including 7,143 g/t Ag over 1.2 m and 2.903 g/t Ag over 1.2 m (extended mineralisation from the 1975 m Level)
- T28-21-2100-142: intersected 1,931 g/t Ag over 4.8 m
- T28-21-2100-144: intersected 2,553 g/t Ag over 1.2 m (extended mineralisation on-strike of the 2100 m Level)
- T28-21-1975-21E: intersected 3,065 g/t Ag over 7.2 m, including 12,640 g/t Ag over 1.2 m and 4,000 g/t Ag over 1.2 m
- T28-21-1975-21Ebis: intersected 4,801 g/t Ag over 3.6 m, including 12,638 g/t Ag over 1.2 m (extended mineralisation on-strike to the east of the 1975 m Level)
- T28-21-1975-26E: intersected 1,300 g/t Ag over 4.8 m, including 4,873 g/t Ag over 1.2 m (continued high-grade mineralisation below current mine workings)
- T28-21-2100-188bis: 480 g/t Ag over 13.2 m, including 2,151 g/t Ag over 1.2 m and 2,113 g/t Ag over 1.2 m (validate vertical continuity of mineralisation below the 2100 m Level)
- T28-21-2125-203: intersected 2,417 g/t Ag over 12 m, including 13,727 g/t Ag over 1.2 m, 3,029 g/t Ag over 1.2 m, 4,808 g/t over 1.2 m and 1,370 g/t Ag over 1.2 (Figure 10.5)
- T28-21-1975-253: returned 3,272 g/t Ag over 7.2 m
- T28-21-1975-253bis: intersected 3,101 g/t Ag over 7.2 m, including 13,671 g/t Ag over 1.2 m and 3,680 g/t Ag over 1.2 m (extension of high-grade underground mineralisation in a previously untested area below current operations)

In particular, percussion drill holes T28-21-2125-203, T28-21-1975-253, and T28-21-1975-253bis are among the best intercepts at Zgounder. Intersections of these high-grade mineralised pods below the current mining levels validate continuity at depth and to the east at Zgounder.

10.5 Diamond Drilling

Diamond drilling programs were completed at Zgounder in 2015, 2017, 2019 through February 2021, and post-February 2021. A total of 84,362 metres in 449 surface and underground drill holes have been completed on the Zgounder Property. The drill programs on the Zgounder Property summarised by year are listed in Table 10.2.

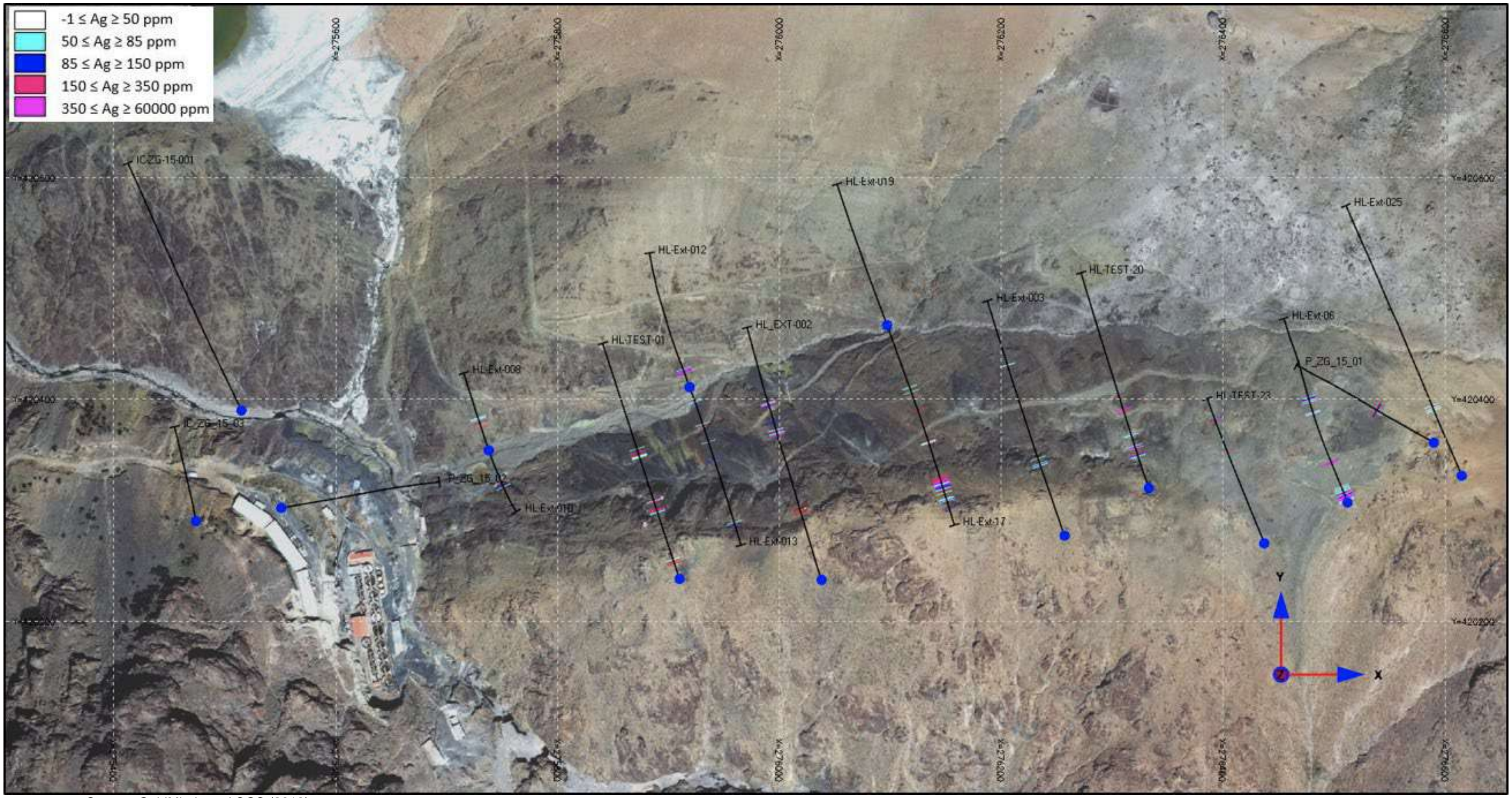
Table 10.2 – Summary of Diamond Drilling Programs

Hole Type	Year	No. of Drill Holes	Metre (m)
Surface	1980s	102	5,396
Surface	1989	10	2,282
Surface	1997	8	1,761
Surface	2002	27	4,005
Surface	2015	2	788
Surface	2017	57	14,823
Surface	2018	11	4,515
Surface	2019	8	2,034
Surface	2020	38	11,922
Surface	2021	76	22,004
Surface	*Sub-Total*	*339*	*69,530*
Underground	2019	5	867
Underground	2020	15	1,982
Underground	2021	90	11,983
Underground	*Sub-Total*	*110*	*14,832*
Surface and Underground	**Total**	**449**	**84,362**

10.5.1 2015 Drilling Program

In 2015, Maya completed a 17-hole diamond drill program totalling 5,896 m (Figure 10.6). The main objectives of the program were to verify presence of widespread silver mineralisation, explore lateral extensions of the known deposit to the north and east, and explore possible extensions of the known deposit at depth.

Figure 10.6 – Plan View – 2015 Diamond Drill Holes



Source: GoldMinds and SGS (2018)

Native silver was observed in eight of the holes drilled. The silver mineralisation is typically associated with zinc (sphalerite) and lead minerals (galena). A 10 m sample from drill hole HL-Ext-012 was selected as a high priority sample to evaluate a sub-parallel mineralised trend north of the main Zgounder Deposit. This sample was collected from 31.3 m to 41.3 m in drill hole HL-Ext-012. GoldMind's independent assay prepared and analyzed by Fire Assay at Bourlamaque Assay Laboratories Ltd in Val d'Or (Quebec) returned an average grade of 1,098 g/t Ag. This drill hole was collared in the valley going northward and likely intersected an extension to the east of the northern body.

Three (3) diamond holes were drilled in West Zone. Sulphide mineralisation consisting of sphalerite, galena and pyrite was encountered in an altered sandstone unit. A total of 3,055 samples were prepared and assayed at the ALS Laboratory in Spain and another 1,167 samples were assayed at the ALS Laboratory in Ireland.

Other metals have also been the subject of exploration at Zgounder Mine. Multi-element analysis, particularly for zinc, lead and copper, led to identification of at least two (2) important polymetallic corridors with horizontal widths of approximately 25 m and 40 m, extending over 1,000 m in length, with shoots of higher-grade silver mineralisation.

10.5.2 2017 Drilling Program

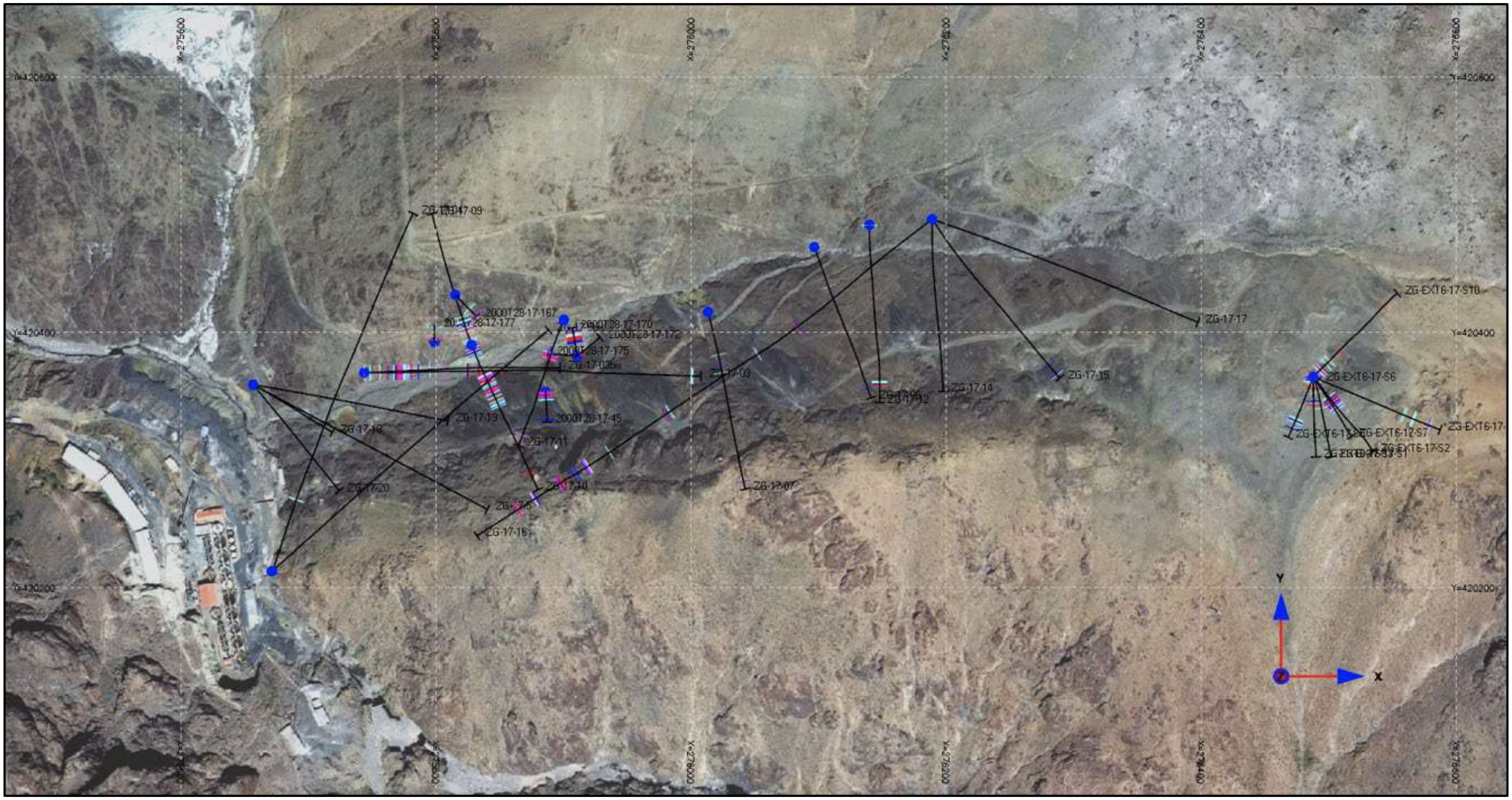
Maya conducted a diamond drilling program planned and supervised by GoldMinds. The drilling program consisted of 57 diamond drill holes totalling 14,823 m.

The objectives of the drilling program were to examine widespread mineralisation across the known Deposit, explore lateral extensions of the known Deposit to the North and to the East, explore possible extensions of the known Deposit at depth, refine the quality of the Mineral Resources to enable possible conversion to Mineral Reserves and increase the Mineral Resource to evaluate a larger mine operating plan.

A new zone was intersected to the East, where the mineralisation was identified at the surface. At the North Zone, drill hole ZG-17-03 extended mineralisation at depth from known occurrences (Panels 8 and 9) at higher elevation. Similar mineralisation was observed in drill hole ZG-17-10.

Drill hole locations are shown in Figure 10.7. Drill hole ZG-17-16 is the deepest hole drilled to date at Zgounder, with a depth of 684 m (at an elevation of 1,613 MASL). The drill hole intersected disseminated native silver over 3 m at 630 m, an altered granite contact was intersected at 653 m along the hole. Zinc in the form of sphalerite is associated with high-grade silver reaching up to 2.38% over 1.5 m. It was the first time that Maya intersected this type of mineralisation at depth at Zgounder.

Figure 10.7 – Plan View – 2017 Diamond Drill Holes



Source: GoldMinds and SGS (2018)

10.5.3 2019 Drilling Program

In 2019, Maya completed an eight-hole drill program totalling 2,033.9 m. The drill program focused to the east, which corresponded to a new zone. This new zone covers an area of 200 m x 150 m that includes several mineralised envelopes, which are oriented mostly E-W with a vertical extent of 185 m below the surface intersected in drill hole ZG-19-01.

The purpose of diamond drilling was to learn more about the continuity of the mineralised envelopes. The presence of the mineralisation near the surface was confirmed by the trenches results (Trench 02 11 m at 130.9 g/t Ag) and requires further work to fully define the extent of mineralisation.

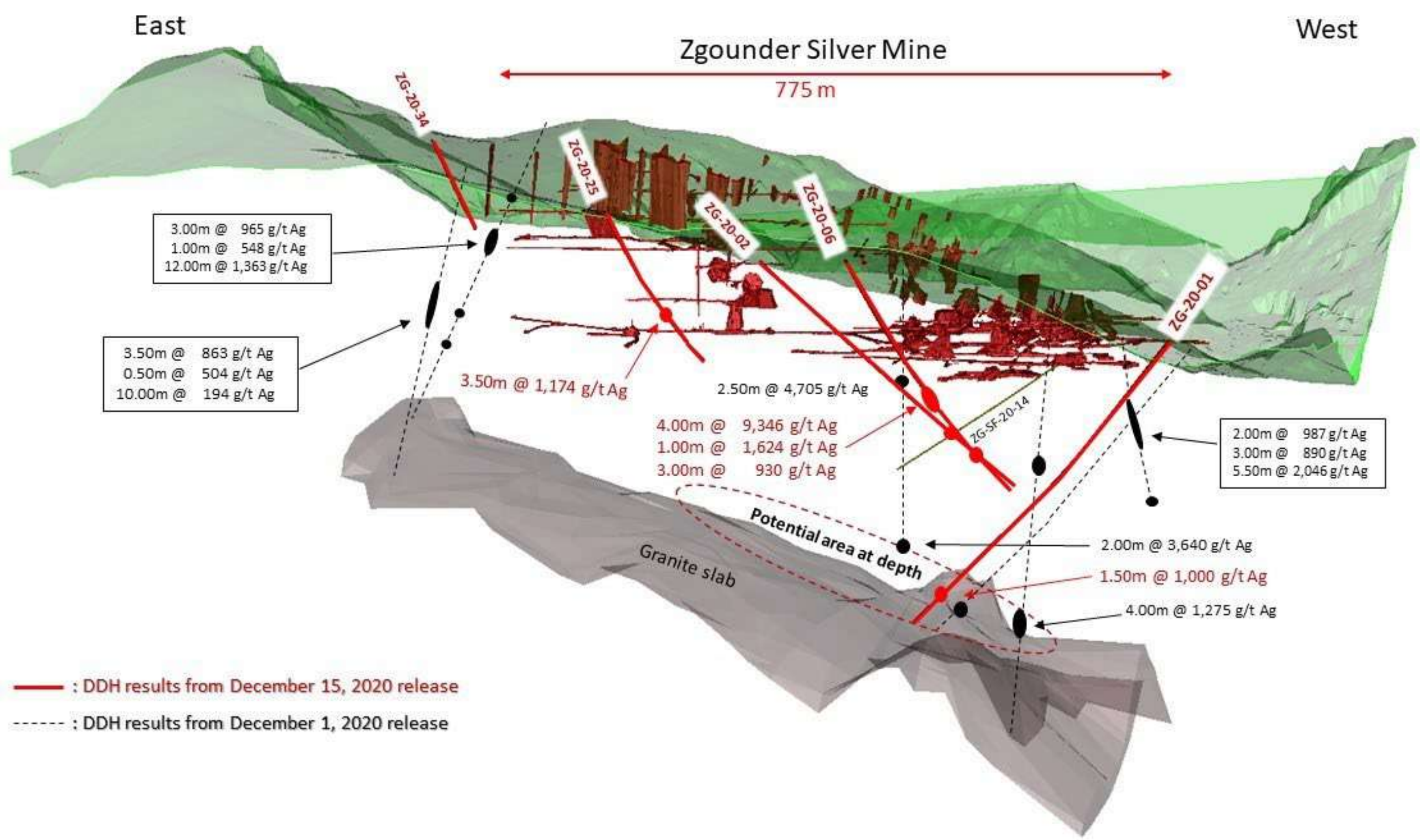
10.5.4 2020 Drilling Program

In 2020, the initial exploration program at Zgounder was expanded twice to follow-up on promising results, completing the year with 13,904 m of surface and underground diamond drilling.

The exploration program had two (2) objectives: 1) to increase the confidence level of the Mineral Resources by converting the 28.3 Moz of silver in Inferred Mineral Resources to Measured and Indicated Mineral Resources; and 2) to identify new prospective mineralisation at depth and better characterize the Mineral Resource potential in the eastern part of the Deposit. Aya extended the mineralisation approximately 90 m along the eastern strike extension and at depth.

Drilling at the Zgounder Mine started in September of 2020, four (4) diamond drills were operating on the surface, and two (2) electric drills worked within the mine. High-grade mineralised extensions were encountered at 408 m and 467 m (ZG-20-04 and ZG-20-09, respectively) from surface, which indicates potential for new underground mineralised zones. In addition, drill holes ZG-20-19 and ZG-20-22 both extend the mineralisation eastward. The results confirm high-grade Ag mineralisation below the current mining operations (ZG-20-06) with an intercept of 4.00 m at 9,346 g/t Ag. In addition, drill hole ZG-20-01 confirms new high-grade mineralisation at depth at the granite contact (Figure 10.8). Results from drill hole ZG-20-13 confirm high-grade Ag mineralisation at depth with an intercept of 5.5 m at 1,273 g/t Ag. In addition, drill hole ZG-20-36 intercepted 1,587 g/t Ag over 3 m, confirming a new high-grade extension to the east (Figure 10.8).

Figure 10.8 – Location and Results of 2020 Diamond Drill Holes

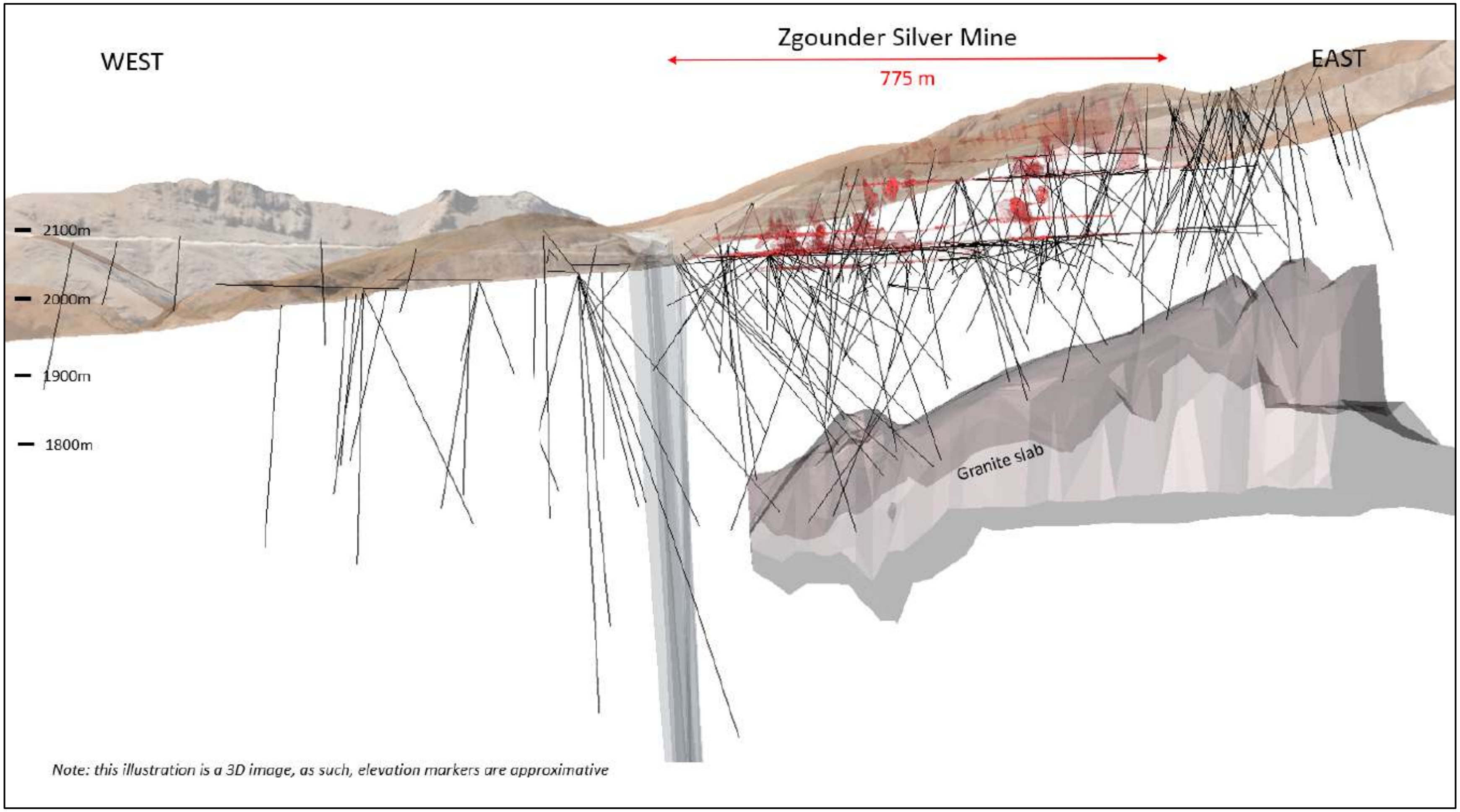


Source: Aya (Press Release dated December 15, 2020 and website (2021))

10.5.5 2021 Drilling Program: Through February 2021

From September 2020 to mid-February 2021, an intensive drilling program totalling 284 diamond drill holes (surface and underground combined) for 19,000 m was carried out at the Property. Up to six (6) DDH rigs were used simultaneously on surface and two DDH rigs were drilling underground. The drilling program had two (2) objectives: 1) to increase the confidence level of the Mineral Resources by converting the 28.3 Moz Ag in Inferred Mineral Resources to Measured and Indicated Mineral Resources; and 2) to identify new prospective mineralisation at depth and better characterize the Mineral Resource potential in the eastern part of the Zgounder Deposit. Some of the drill hole traces are represented in Figure 10.9.

Figure 10.9 – Longitudinal 3-D Projection of the Zgounder Mine (in red) with DDH Traces Shown



Source: Aya website (2021)

10.5.6 2021 Drilling Program: Post-February 2021

In the post-February 2021 period, 166 surface and underground diamond holes were completed totalling 33,986.90 m at Zgounder. Up to eight (8) diamond drill rigs were in operation on the mine permit. Significant intercepts from the diamond drill program at Zgounder (in core lengths) are summarised as follows (from numerous Aya press releases):

- ZG-SF-20-25 (underground): intersected 2,728 g/t Ag over 6.0 m
- ZG-SF-20-23 (underground): cut 4,517 g/t Ag over 3.5 m and 1,635 g/t Ag over 7.5 m
- ZG-SF-20-18 (underground): intersected 854 g/t Ag over 12.0 m
- ZG-20-38 (surface): intersected 1,623 g/t Ag over 4.0 m
- ZG-20-13bis (surface): intersected 2,100 g/t Ag over 2.0 m (intersection of high-grade silver mineralisation at depth below the previous Mineral Resources and along strike to the east)
- ZG-21-06 (surface): intersected 412 g/t Ag over 1.0 m
- ZG-21-05 (surface): intersected 100 g/t Ag over 0.5 m (extending mineralisation potential to the west)
- ZG-21-15 (surface): returned 6,437 g/t Ag over 6.5 m, including 24,613 g/t Ag over 0.5 m and 11,483 g.t Ag over 0.5 m and 12,775 g/t Ag over 0.5 m
- ZG-SF-21-04 (underground): returned 1,599 g/t Ag over 8.4 m, including 7,143 g/t Ag over 1.2 m and 2,903 g/t Ag over 1.2 m (extended high-grade silver mineralisation to the east)
- ZG-21-21 (surface): intersected 2,887 g/t Ag over 5.5 m, including 5,920 g/t Ag over 0.5 m, 3,120 g/t Ag over 0.5 m, 8,800 g/t Ag over 0.5 m and 9,160 g/t Ag over 0.5 m (extension of the mineralisation at depth to the granite contact within the Exploration Target established by P&E, 2021)
- ZG-21-22 (surface): intersected 1,010 g/t Ag over 5 m, including 4,320 g/t Ag over 0.5 m, 2,560 g/t Ag over 0.5 m and 1,608 g/t Ag over 0.5 m (continuation of the mineralised system definition of the new eastern zone)
- ZG-SF-21-12 (underground): intersected 1,879 g/t Ag over 6.0 m, including 6,422 g/t Ag, 3,929 g/t Ag over 0.5 m, 3,249 g/t Ag over 0.5 m and 2,871 g/t Ag over 0.5 m (continuation of high-grade mineralisation below the current mine workings)
- ZG-SF-21-39 (underground): intersected 1,299 g/t Ag over 15 m, including 13,138 g/t Ag over 0.5 m, 5,621 g/t Ag over 0.5 m and 5,168 g/t Ag over 0.5 m (extension of on-strike underground mineralisation below current operations)
- ZG-21-40 (surface): returned 845 g/t Ag over 6.0 m, including 3,825 g/t Ag over 0.5 m and 1,358 g/t Ag over 0.5 m

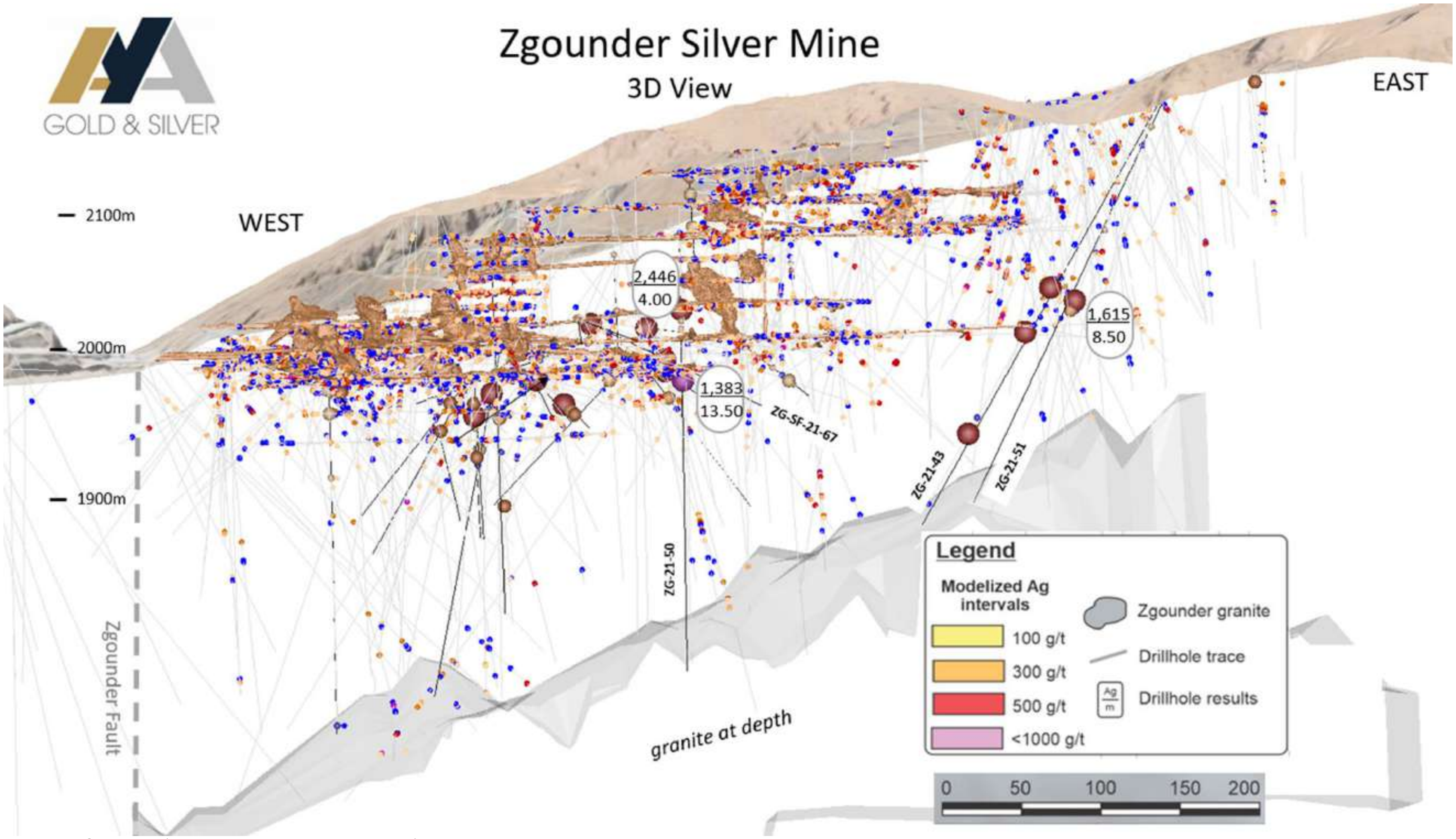
- ZG-21-41 (surface): returned 448 g/t Ag over 10.5 m, including 16,396 g/t Ag over 0.5 m and 3,527 g/t Ag over 3.5 m (discovery of new mineralised structure below 2,000 m in the Zgounder East Zone)
- ZG-21-36 (surface): intersected 1,298 g/t Ag over 9.5 m, including 10,753 g/t Ag over 0.5 m, 2,963 g/t Ag over 0.5 m, and 2,794 g/t Ag over 0.5 m (extension of strike and ongoing definition of new Zgounder East Zone)
- ZG-21-30bis (surface): returned 1,863 g/t Ag over 1 m and 162 g/t Ag over 15.5 m (expand eastern mineralised strike length)
- ZG-SF-21-25 (underground): intersected 1,230 g/t Ag over 2.5 m (extend mineralisation at depth to the granite contact within the Exploration Target established by P&E, 2021).

As a result of the positive drilling results received to the end of Q3, the 2021 drill program at Zgounder was expanded from 35,000 m to 42,000 m into year-end. Result from this drilling are summarized below:

- ZG-SG-21-67 (underground): encountered 1,383 g/t Ag over 13.5 m, including 10,960 g/t Ag over 0.5 m, 10,435 g/t Ag over 0.5 m, 5,000 g/t Ag over 0.5 m, 3,002 g/t over 0.5 m and 1,598 g/t Ag over 0.5 m (Figure 10.10) (extended mineralisation trend below the 1975 level)
- ZG-21-51 (surface): cut 1,615 g/t Ag over 8.5 m, including 11,360 g/t Ag over 0.5 m, 8,840 g/t over 0.5 m, 1,840 g/t Ag over 0.5 m and 1,752 g/t Ag over 0.5 m (confirmed eastern vertical continuity 200 m below surface)
- ZG-21-50 (surface): returned 2,446 g/t Ag over 4.0 m, including 10,635 g/t Ag over 0.5 m and 8,080 g/t Ag over 0.5 m (continued definition of high-grade mineralisation below the 2030 m Level)
- ZG-21-43 (surface): returned 2,311 g/t Ag over 2.0 m, including 2,080 g/t Ag over 2.0 m and 1,821 g/t Ag over 3.0 m (continued definition of eastern high-grade mineralisation)

Some of these drill result highlights are shown in Figure 10.10. The drill program succeeded in extending the strike length of the Zgounder Deposit mineralisation eastward by an addition 375 m. The Zgounder Deposit remains open to expansion by further drilling along strike and at depth.

Figure 10.10 – Zgounder Post-February 2021 Surface and Underground Drilling Results



Source: Aya (press release dated November 17, 2021)

11 SAMPLE PREPARATION, ANALYSIS AND SECURITY

This Section has been summarised and/or largely extracted from the Report available on SEDAR entitled: "Technical Report and Updated Mineral Resource Estimate of the Zgounder Silver Project, Kingdom of Morocco, prepared for Aya Gold & Silver Inc., dated January 28, 2022, prepared by P&E Mining Consultants Inc., Report # 415."

11.1 Sample Preparation

11.1.1 Logging and Sampling

Logging drill core or RC chipped materials and sampling are performed at the Zgounder Mine drill core shed facility. Internationally accepted procedures and standards are applied by Aya's technical team.

Drill core logging is carried out in hand-written format and all the information transferred into Excel spreadsheets. This method provides duplicate records of all the drill core logging and sampling information, and facilitates data verification and validation. Aya considers that these advantages outweigh the additional time required to copy the data.

Digital photographs are taken of the drill core and drill core recovery, RQD, basic geotechnical information, geological and structural elements are recorded in the drill logs (Figure 11.1). Samples for bulk density determination are also selected.

Nominal drill core sample intervals are 1.0 m and 1.5 m, but are adjusted to respect lithological contacts or abrupt changes in mineralisation, generally between 0.3 m to 2.0 m. Hard drill core material is cut using a diamond-blade saw, with up to four diamond-blade saws operating simultaneously during the peak of the 2020 to 2021 drilling campaigns (Figure 11.2). The rock saw operator cuts along contacts between samples along a line drawn by the logging geologists.

One-half of the drill core is placed into a polyethylene bag with a sample tag and sent to the assay laboratory for analysis, and the remaining half-drill core is carefully returned to its original position in the drill core boxes (Figure 11.2). An arrow to mark downhole direction is drawn along each drill core sample by the geologist, for future reference. Paper sample tags are stapled to the drill core boxes at the end of the sample intervals. Sample books were utilized with pre-recorded, unique sequential number tags reserved for QC samples at pre-determined locations.

Figure 11.1 – Drill Core at the Logging Facility

Description: Top) staff recording RQD on core at the logging facility (top); and Bottom) geologists and technicians inspecting drill core during the 2020 drilling program (bottom).

Figure 11.2 – Drill Core Storage and Processing Facilities at Zgounder Mine

Description: Top) up to four diamond saws were operating during the 2020 - 2021 drilling program; and Bottom) drill core logging and core storage facilities.

11.1.2 BULK DENSITY DETERMINATION

Bulk density determination is performed onsite by Aya geologists, with the water immersion method selected as an appropriate method to determine the bulk density of rocks at Zgounder (Figure 11.3). Bulk density determinations are completed in a dedicated area, where the equipment is protected from disturbances, such as drafts that might influence balance readings.

Aya's protocol calls for the determination of wet (moisture percent) and dry densities of mineralised and barren samples. Full drill core pieces of approximately 10 cm to 15 cm are used for the determinations. When this process is complete, the drill core is cut and one-half is returned to the original location in the drill core box, with a piece of flagging tape stapled to the box to aid with future sample identification.

Figure 11.3 – Geologist Taking Bulk Density Measurements

Bulk density for different lithologies were measured using standard procedures. The results are presented in Table 11.1.

Table 11.1 – Bulk Density Factor for Each Rock Type and Mineralised Material Observed at the Zgounder Project

Facies	Rock Code	No. Samples	Dry Bulk Density
Brown Formation			
Schist + Ag	300	62	2.76
Schists not mineralised	200, 310, 320, 330, 350	256	2.75
Schist + Pyrite	340	20	2.77
Volcanics			
Andesite	400	7	2.54
Diorite	500	19	2.70
Rhyolite	475	30	2.67
Intrusives			
Pink Granite	600	5	2.61
Granodiorite	650	26	2.71
Fault (BX, Cis, FZ)	70	0	2.70
	Total	**425**	

11.1.3 Preparation and Analysis

Sample preparation and analysis for samples collected from T28 production drilling (Figure 11.4) are carried out in Aya's laboratory facilities at the Zgounder Mine. Chip samples collected from the T28 drilling operation are collected on a 1.2 m length basis. The samples are dried and analyzed for silver (Ag) at the Zgounder Mine laboratory using aqua regia (1/3 HNO_3 and 2/3 HCL) with atomic absorption (AA) finish. In October 2020, Aya's geologists began inserting certified reference materials (CRMs"), blanks and duplicates, in accordance with industry accepted quality assurance/quality control ("QA/QC" or "QC") procedures. Select pulps were also sent to ALS in Seville, Spain laboratory for check assaying of Ag only, using aqua regia and atomic absorption spectroscopy ("AAS") finish.

Sample preparation and analysis for samples generated from diamond drill holes (Figure 11.4) are performed at African Laboratory for Mining and Environment ("AfriLab") in Marrakesh, Morocco. All individual samples represent approximately one-metre length ½-cut drill cores, with half of the drill core length stored on-site for reference (Figure 11.2). Each ½-drill core sample is sent for preparation and analysis to AfriLab.

All samples are analyzed for silver, copper, iron, lead, and zinc, using aqua regia with an AAS finish. Samples grading above 200 g/t Ag are re-analyzed by fire assay method.

Figure 11.4 – Drill Rigs at Zgounder Mine in 2020

Description: Top) diamond drill rig at surface; and
Bottom) an electric diamond drill rig on 1975 Level.

Table 11.2 lists various independent and reputable Spanish, Moroccan, and Canadian laboratories used since the 1980s and includes the laboratory certification/accreditation details.

Table 11.2 – Summary of the Independent and Reputable Assay Laboratories Used Since 1980s

Drill Program	Sample Preparation	Analytical Laboratory	Analytical Methods	Accreditation / Independent of Aya	
1980s to present	Aya	Aya	Aqua Regia ICP-AES	None	No
2013	ALS Val d'Or, Canada	ALS Val d'Or	AG-GRA21	ISO/IEC 17025:2017	Yes
2015 to present	AfriLab, Morocco	AfriLab	Aqua Regia ICP-AES	SGS MA20/819942595	Yes
2015	ALS, Seville, Spain	ALS Spain	AG-GRA21	INAB NO.173T	Yes
2019 to 2021	AfriLab, Morocco	ALS Spain-Ireland	AG-GRA21	INAB NO.173T	Yes

11.1.3.1 *Zgounder Laboratory Sample Preparation and Analysis*

Prior to the 2020 drilling program, all percussion drilling (T28- YAK-T28) samples were prepared and analyzed at the Zgounder Mine laboratory (Figures 11.5 and 11.6). Samples are completely crushed to 80% passing 2 mm and riffle-split to obtain a 100 g subsample, which is then pulverized to a pulp 80% passing 75 µm.

Each sample is subject to chemical digestion using the bi-acid (1/3 nitric acid and 2/3 hydrochloric acid), in order to dissolve trace elements within the sample into solution. The solutions are analyzed by atomic absorption spectrometer (AA ICE 3500). Fire assay is used for high-grade silver samples. The assay results are then sent in a file format, supported in Microsoft Excel, to the geological department for integration.

Figure 11.5 – Sample Preparation of Drill Core Samples

(A) Crusher (B) Splitter (C) Pulverizer (D) Sample Number Confirmation

Figure 11.6 – Chief Chemist Discussing Laboratory Procedures

Note: Figure 11.6 top and bottom.

11.1.3.2 *AfriLab Laboratory Sample Preparation and Analysis*

The total sample is crushed to <2 mm with a passing rate of 85% using a ROCKLABS jaw crusher. A sieving operation is used to ensure the sample is 85% <2 mm. To control the risk of contamination, the jaw crusher is cleaned thoroughly between each sample using compressed air and local waste rock.

The crushed sample is then divided using a Riffle splitter, in order to have a sub-sample of between 250 g to 300 g. The splitter is cleaned thoroughly between each sample using compressed air.

The sub-sample (of 250 g to 300 g) is pulverized using a ROCKS LABS pulverizer. Pulverizing performance is targeted to a size of 85% of the sample at <75 µm. One sample in twenty is selected at random to verify this performance, by wet sieve test (standard 75 µm sieve).

Silver is analyzed by atomic absorption after aqua regia mixture (HCl and HNO_3) preparation. Silver grades of >200 g/t Ag are further analyzed by a fire assay method.

11.1.3.3 *ALS Laboratory Sample Preparation and Analysis*

After ALS logs the sample into its tracking system, the sample is crushed, riffle split, and then pulverized to >85% passing 75 µm screen. Routine QC pulp testing is randomly carried out on at least one in 50 samples. Silver is analyzed by fire assay fusion with gravimetric finish.

11.1.4 Security – Chain of Custody

Drill core is under ZMSM's control from the drill site, where ZMSM geologists supervise operations, to the drill core shed at the mine site, where drill core boxes are transported at the end of each shift for logging, cutting and sampling. Prepared samples are stored at the Aya facility until a sufficient number of samples have accumulated, at which time samples are packed into 50 litre plastic drums and transported to the AfriLab facility in Marrakesh, or the ALS laboratory in Seville, using a commercial transport group. Samples analyzed by ALS in Ireland are shipped directly to the ALS facility in Ireland from the ALS laboratory in Seville, Spain and tracked through ALS's Global Enterprise Management System (GEMS).

All samples remain under constant surveillance until delivery to the laboratory facility, thereby preserving a continuous chain of custody.

When logging and sampling are completed, the drill core boxes are safely stored at the warehouse (Figure 11.2) with the coarse reject and pulp samples returned from the laboratory.

11.1.5 Conclusion

It is the opinion of the author of this section that sample preparation, security, and analytical procedures for the Zgounder Project were adequate for the purposes of the Mineral Resource Estimate in this Technical Report.

11.2 Aya Quality Assurance/Quality Control Review

Aya implemented and monitored a thorough QA/QC program for the drilling undertaken at the Zgounder Project over the 2020 to 2021 period, in addition to the internal QC protocol implemented at the laboratories. QC protocol at Zgounder included the insertion of CRMs into every batch of drill core and T28 samples sent for analysis, including CRMs, blanks, and field duplicates. Samples prepared at the logging facility are numbered sequentially, such that drill samples and QC samples are not differentiated to the laboratory.

CRMs, blanks and duplicates are inserted sequentially every 1 in 50 samples to the diamond drill hole sample flow and 1 in 25 samples to the T28 sample flow.

Example for a drill hole sample flow:

- CRMs are inserted at a rate of 1 in 50 samples;
- Blank samples are inserted at a rate of 1 in 50 samples to monitor for instrumentation carry-over and contamination at the laboratory;
- Duplicate samples are inserted 1 in every 50 samples; and

- Check analysis at a secondary laboratory is carried out on one in every 50 samples, representing between 5% to 10% of the global primary laboratory sample flow.

Geologists in-charge must prepare the notebooks dedicated to sampling by identifying the numbers corresponding to all control samples. Geologists and technicians are sampling according to preset sample books (Table 11.3).

Table 11.3 – Example of Preset Sampling Protocol at Zgounder

Sample number ending in	Sample type	Sample number ending in	Sample type
1	Blank	41	Normal Sample
2	Normal Sample	42	Normal Sample
3	Normal Sample	43	Normal Sample
4	Normal Sample	44	Normal Sample
5	Normal Sample	45	Normal Sample
6	Normal Sample	46	Normal Sample
7	Normal Sample	47	Normal Sample
8	Normal Sample	48	Normal Sample
9	Normal Sample	49	Normal Sample
10	Normal Sample	50	Blank
11	Normal Sample	51	Normal Sample
12	Normal Sample	52	Normal Sample
13	Normal Sample	53	Normal Sample
14	Duplicate du 13	54	Normal Sample
15	Normal Sample	55	Normal Sample
16	Normal Sample	56	Normal Sample
17	Normal Sample	57	Normal Sample
18	Normal Sample	58	Normal Sample
19	Normal Sample	59	Normal Sample
20	Normal Sample	60	Normal Sample
21	Normal Sample	61	Normal Sample
22	Normal Sample	62	Normal Sample
23	Normal Sample	63	Normal Sample
24	Normal Sample	64	Duplicate du 63
25	Standard (A or B or C)	65	Normal Sample
26	Normal Sample	66	Normal Sample
27	Normal Sample	67	Normal Sample
28	Normal Sample	68	Normal Sample
29	Normal Sample	69	Normal Sample
30	Normal Sample	70	Normal Sample
31	Normal Sample	71	Normal Sample
32	Normal Sample	72	Normal Sample
33	Normal Sample	73	Normal Sample
34	Normal Sample	74	Normal Sample
35	Normal Sample	75	Standard (A or B or C)
36	Normal Sample	76	Normal Sample
37	Normal Sample	77	Normal Sample
38	Normal Sample	78	Normal Sample
39	Normal Sample	79	Normal Sample
40	Normal Sample	80	Normal Sample

Two (2) different CRMs were used during the 2020 drilling program; the OREAS 621 and OREAS 604B and a further two (2) CRMs (the OREAS 605B and OREAS 139) were introduced at a later stage in 2021. All four (4) CRMs were purchased from ORE Research & Exploration Pty Ltd ("ORE") in Australia and are certified for Ag, Au, Cu and Zn (except for OREAS 139, which is not certified for Au). The certified mean values for each CRM are outlined in Table 11.4. A record of CRM used during sample preparation is made in the QC information list, as shown in Table 11.3.

Table 11.4 – Summary of Certified Reference Materials Used at Zgounder in 2020 - 2021

Certified Reference Material	Certified Mean Value			
	Ag (g/t)	Au (g/t)	Cu (%)	Zn (%)
OREAS 621	68.5	1.25	0.368	5.22
OREAS 604B	507	1.69	2.12	0.117
OREAS 605B	1015	1.72	5.03	0.240
OREAS 139	76.7	--	0.0271	13.63

The first sample inserted into each batch is a blank material (Table 11.3), that has been sourced and prepared by Aya from a granite unit outside the camp area. The blank material is safely stored away from any source of contamination in the Zgounder camp.

A summary of QC samples inserted into the DDH and T28 samples stream and analyzed at AfriLab and the Zgounder Mine laboratory is outlined in Table 11.5.

Table 11.5 – Summary of QC Samples used by ZMSM in DDH Sample Flow Sent to AfriLab and the T28 Sample Flow Analysed at the ZMSM Laboratory in 2020-2021

Drilling Period	Certified Reference Material (n=)	Blanks (n=)	Duplicates (n=)	QC Samples (%)
DDH 2020-21	1,044	1,157	1,026	7.3
T28 2020-21	181	177	63	13.1

11.2.1 2020 TO FEBRUARY 2021 DIAMOND DRILL HOLE PROGRAM – ANALYSES PERFORMED AT AFRILAB

11.2.1.1 Performance of Certified Reference Materials

CRMs were inserted into the analysis stream approximately every 50 samples. Two (2) CRMs were used during the 2020 to February 2021 diamond drilling program to monitor for silver performance; OREAS 621 and OREAS 604B (Tables 11.4 and 11.5). Criteria for assessing CRM performance are based as follows: data falling within ±2 standard deviations from the accepted mean value, pass;

data falling outside ±3 standard deviations from the accepted mean value, or two (2) consecutive data points falling between ±2 and ±3 standard deviations on the same side of the mean, fail. The CRM results are presented in Figures 11.7 and 11.8. No failures were recorded for either CRM. However, a slight negative bias was noted for the OREAS 621 CRM.

Figure 11.7 – Performance of CRM OREAS 621 for Ag

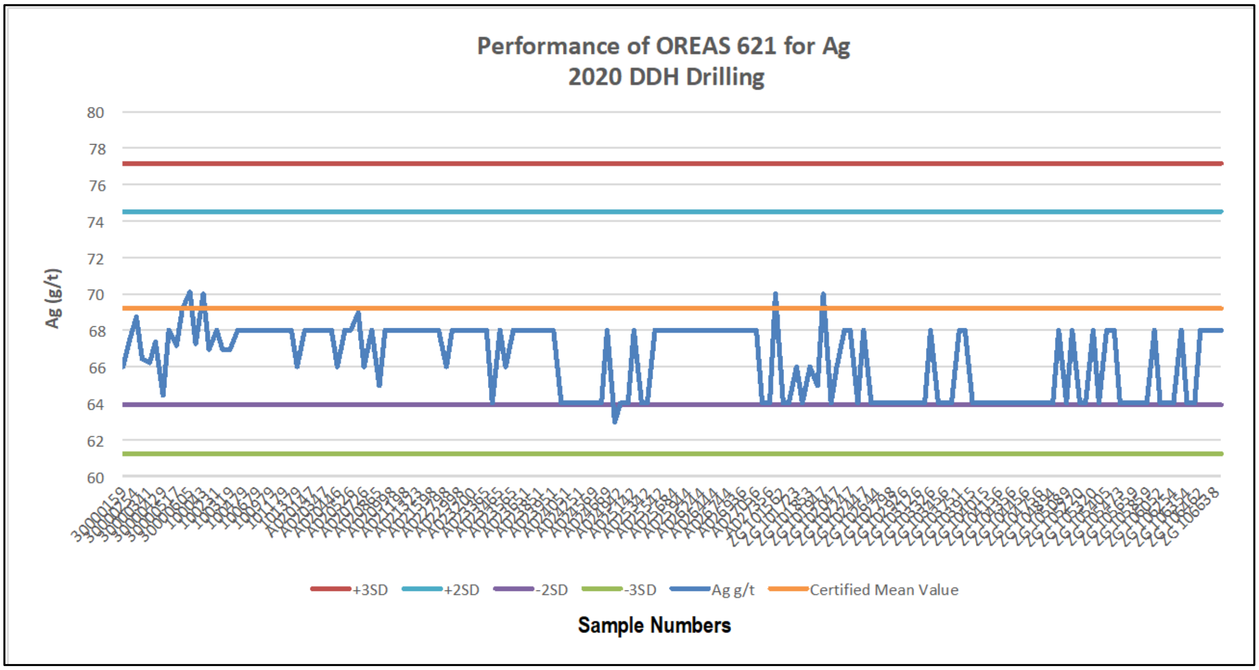
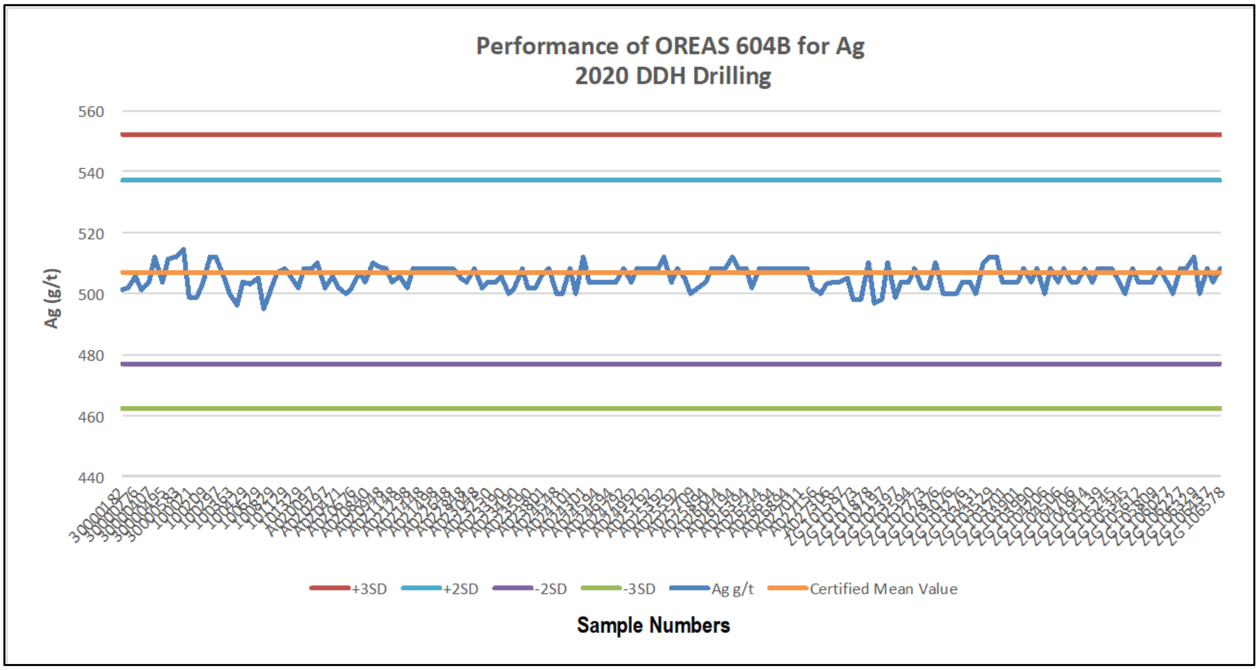


Figure 11.8 – Performance of CRM OREAS 604b for Ag



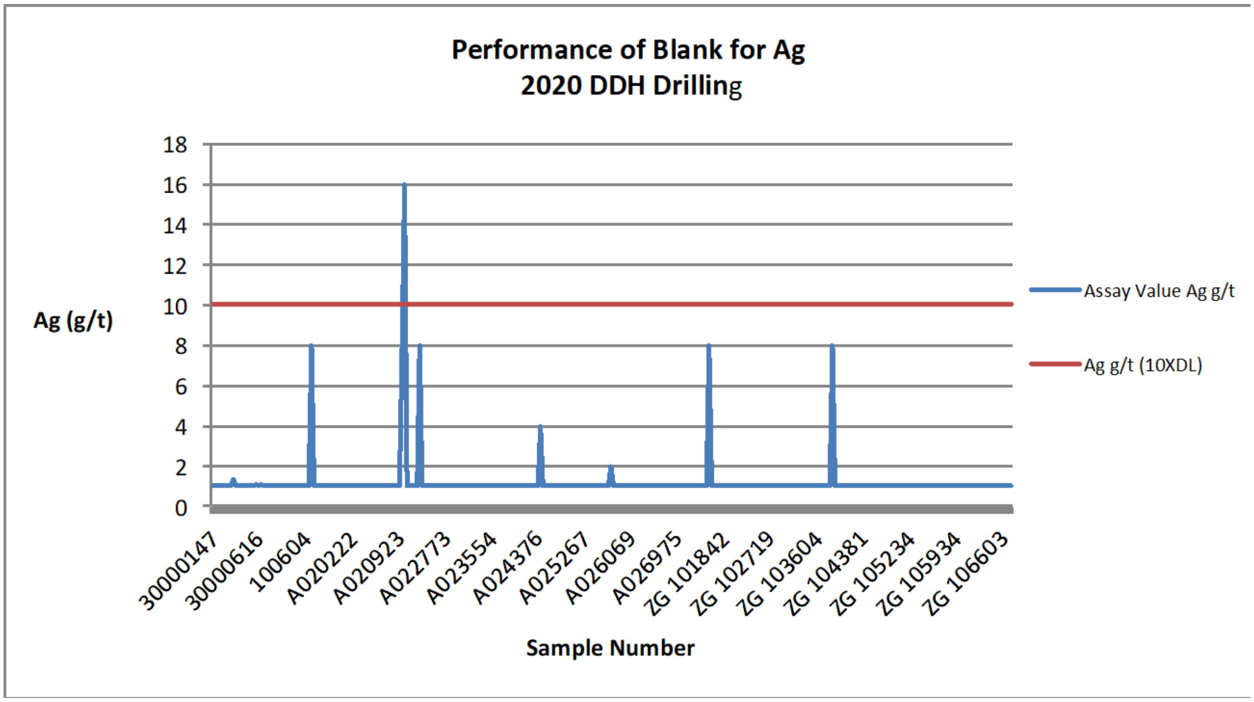
The author of this Technical Report section considers that the CRMs demonstrate acceptable accuracy in the Zgounder 2020-February 2021 diamond drilling data.

11.2.1.2 *Performance of Blanks*

Blanks were inserted into the analysis stream approximately every 50 samples. All blank data for Ag are graphed (Figure 11.9). If the assayed value in the certificate was indicated as being less than detection limit, the value was assigned the value of the detection limit for data treatment purposes (e.g. <1 = 1). An upper tolerance limit of ten times the detection limit was set. There were 345 data points to examine.

The vast majority of data plotted below the set tolerance limit, with one data point plotting above the set tolerance limit. The author of this Technical Report section does not consider this to significantly impact the data.

Figure 11.9 – Performance of Blanks



The author of this Technical Report section does not consider contamination to be an issue for the Zgounder 2020-February 2021 diamond drilling data.

11.2.1.3 Performance of Duplicates

Duplicate data for silver were examined for the 2020 to February 2021 diamond drilling program at the Zgounder Project. There were 302 duplicate pairs in the dataset. The data were graphed (Figures 11.1 and 11.11) and found to have acceptable precision.

Figure 11.10 – Performance of Duplicates (All Data Points Included)

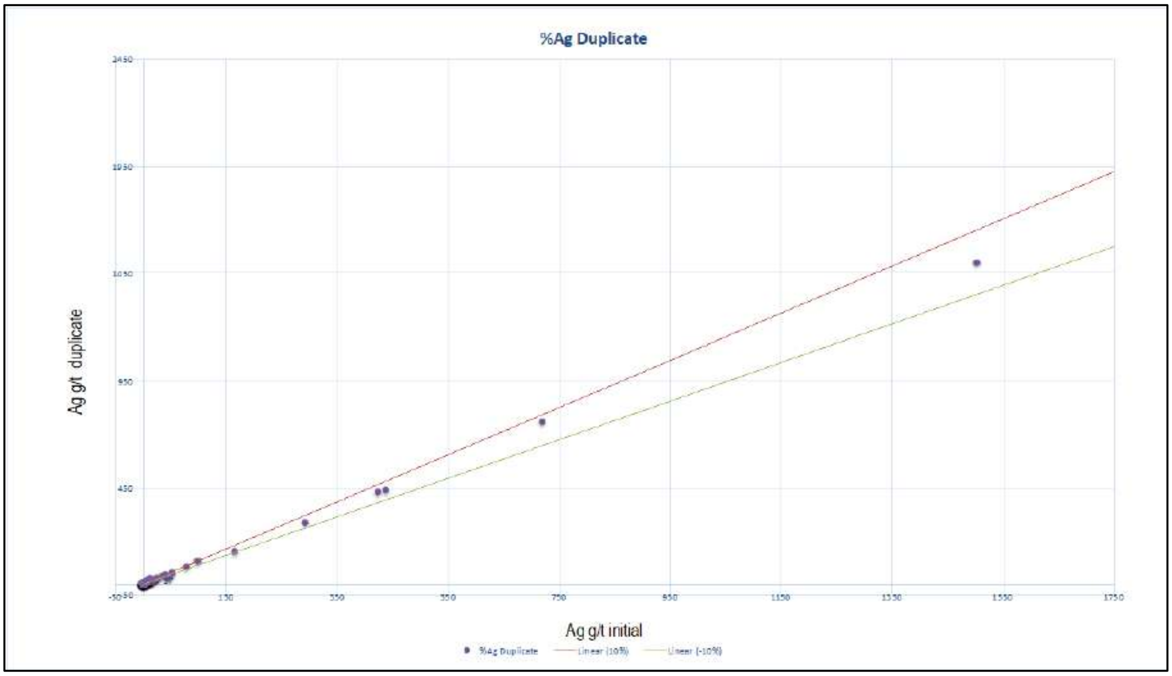
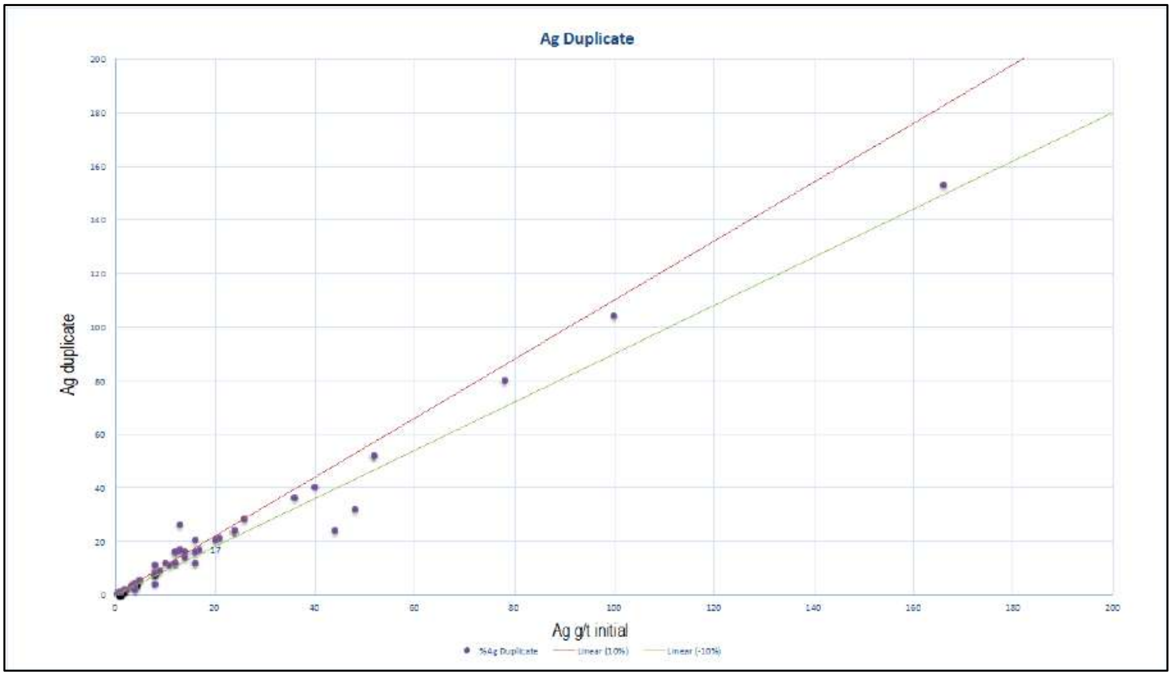


Figure 11.11 – Performance of Duplicates Data points <200 g/t Ag only



11.2.2 2020-February 2021 T28 Production Drilling – Analyses Performed at Zgounder Laboratory

The same OREAS CRMs, OREAS 621, and OREAS 604B, were used throughout the 2020 to February 2021 T28 production drilling. CRMs were inserted at a rate of approximately 1 in 25 samples. The CRM results are presented in Figures 11.12 and 11.13. No failures were recorded for either CRM.

Figure 11.12 – Performance of CRM OREAS 621 for Ag

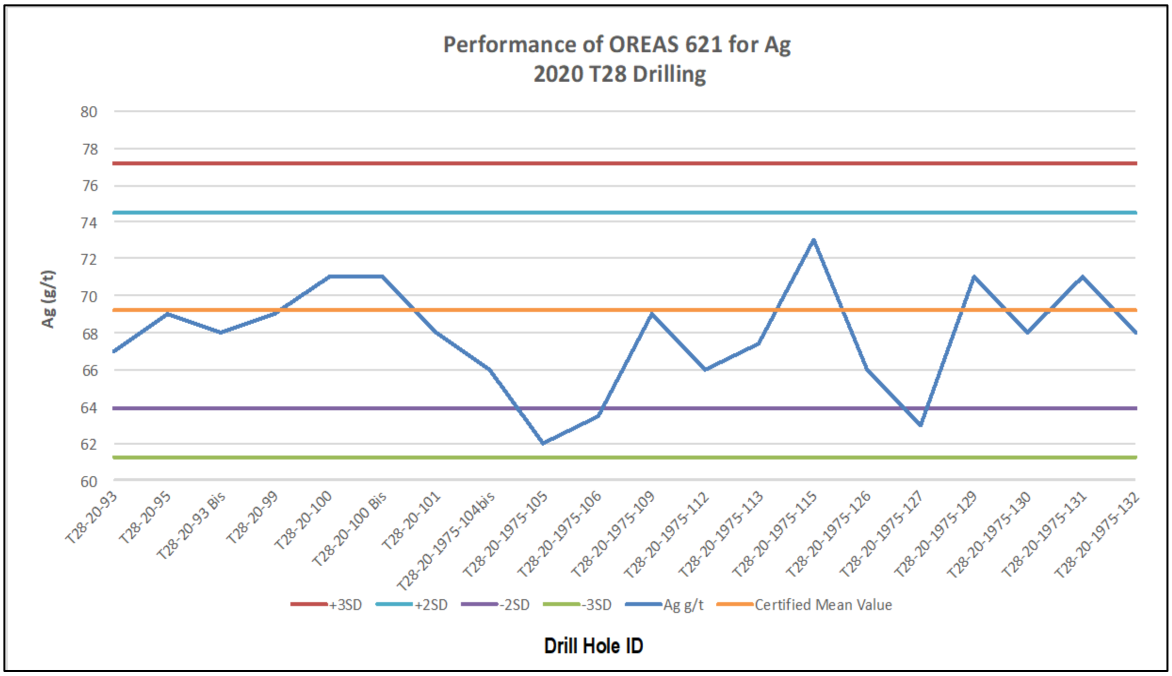
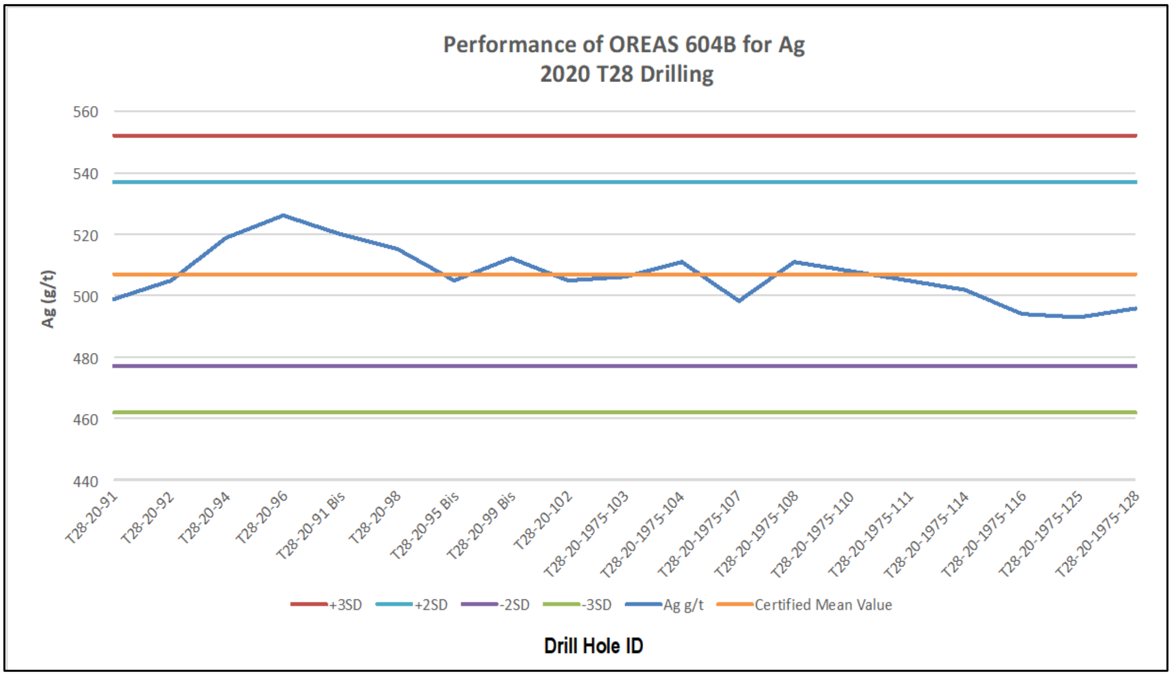


Figure 11.13 – Performance of CRM OREAS 604B for Ag

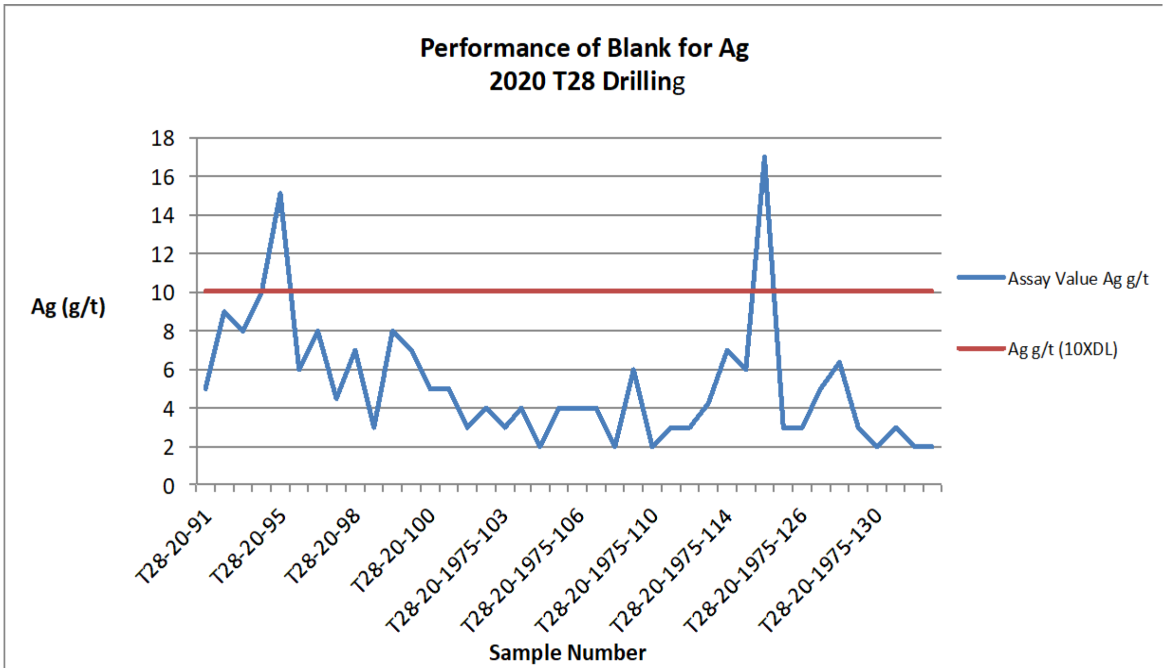


The author of this Technical Report section considers that the CRMs demonstrate acceptable accuracy in the Zgounder 2020 to February 2021 T28 production drilling.

11.2.2.1 *Performance of Blanks*

Blanks were inserted into the analysis stream approximately every 25 samples. All blank data for Ag are graphed (Figure 11.14). The vast majority of data plotted below the set tolerance limit, with only two (2) data points plotting above the set tolerance limit. The Qualified Person does not consider this to be of significant impact to the data.

Figure 11.14 – Performance of Blanks



The author of this Technical Report section does not consider contamination to be an issue for the Zgounder 2020 to February 2021 T28 production drilling data.

11.2.2.2 Performance of Duplicates

Duplicate data for silver were examined for the 2020 to February 2021 T28 production drilling at the Zgounder Mine. There were 40 duplicate pairs in the dataset. The data were graphed (Figures 11.15 and 11.16) and found to have acceptable precision.

Figure 11.15 – Performance of Duplicates All data points included

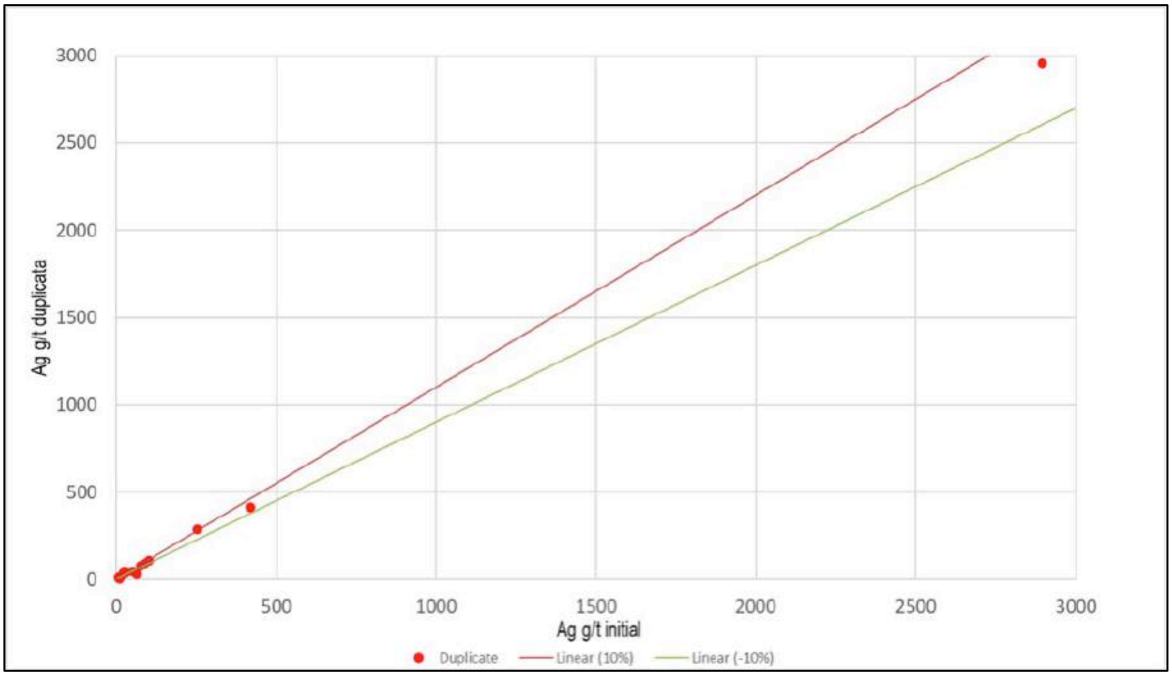
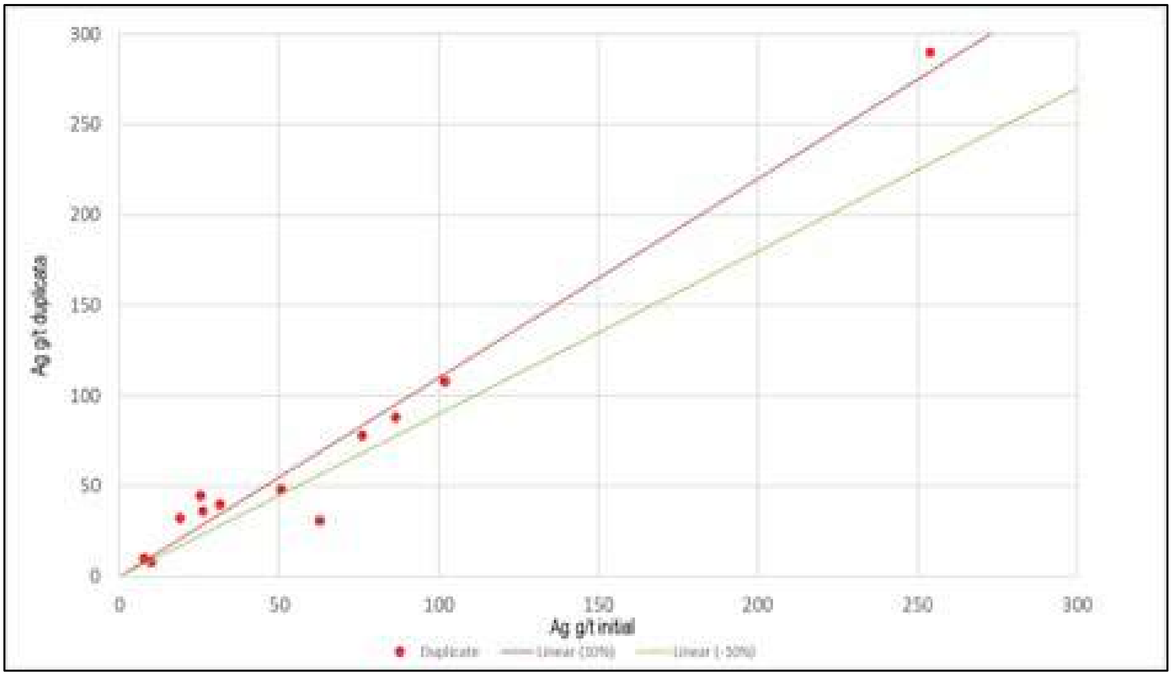


Figure 11.16 – Performance of Duplicates Data points <300 g/t Ag only



11.2.3 2020 Tailings Drilling Program – Analyses Performed at AfriLab, Marrakesh Laboratory

A total of 320 tailings samples were sent to AfriLab, Marrakesh for analysis, along with 80 QC samples, including CRMs, blanks and duplicates, making a total of 400 samples sent for analysis. Three (3) intervals, out of the 320 tailings samples are without results, due to poor recovery or intersecting a cavity.

The same two (2) OREAS CRMs, OREAS 621 and OREAS 604B, were used for the 2020 tailings drilling program. CRMs were inserted at a rate of approximately 1 in 15 samples. The CRM results are presented in Figures 11.17 and 11.18. One (1) low failure only was recorded for the OREAS 621 CRM (Figure 11.17), which the author of this Technical Report section does not consider to be of significant impact to the current Mineral Resource Estimate data.

Figure 11.17 – Performance of CRM OREAS 621 for Ag

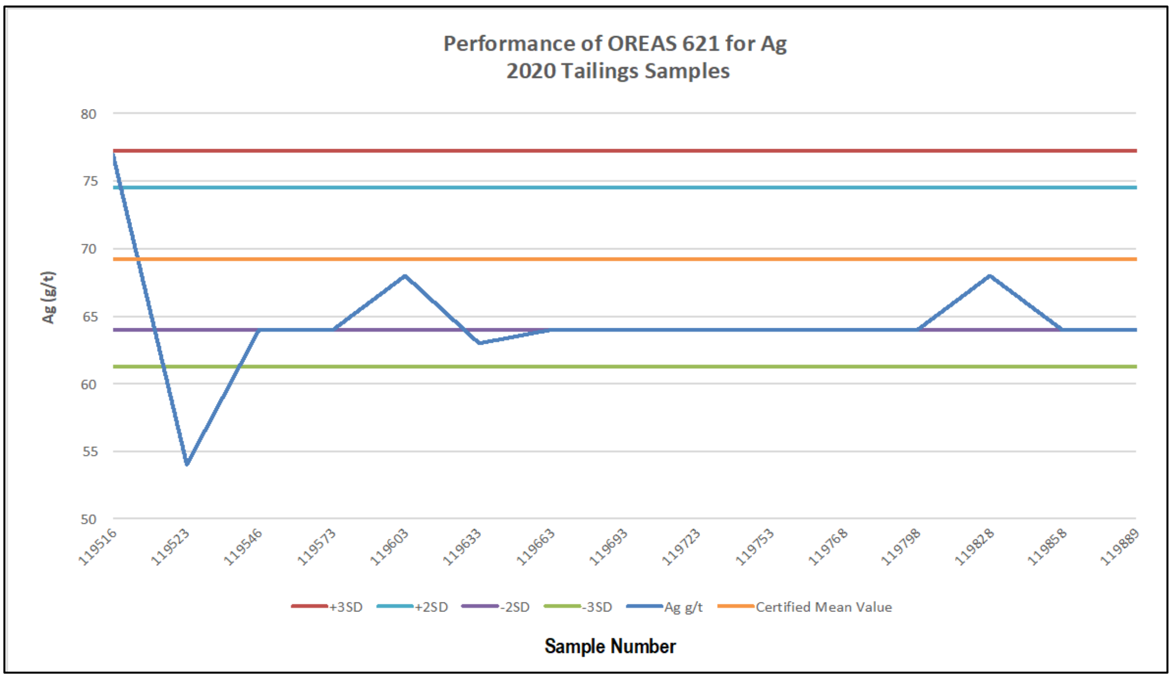
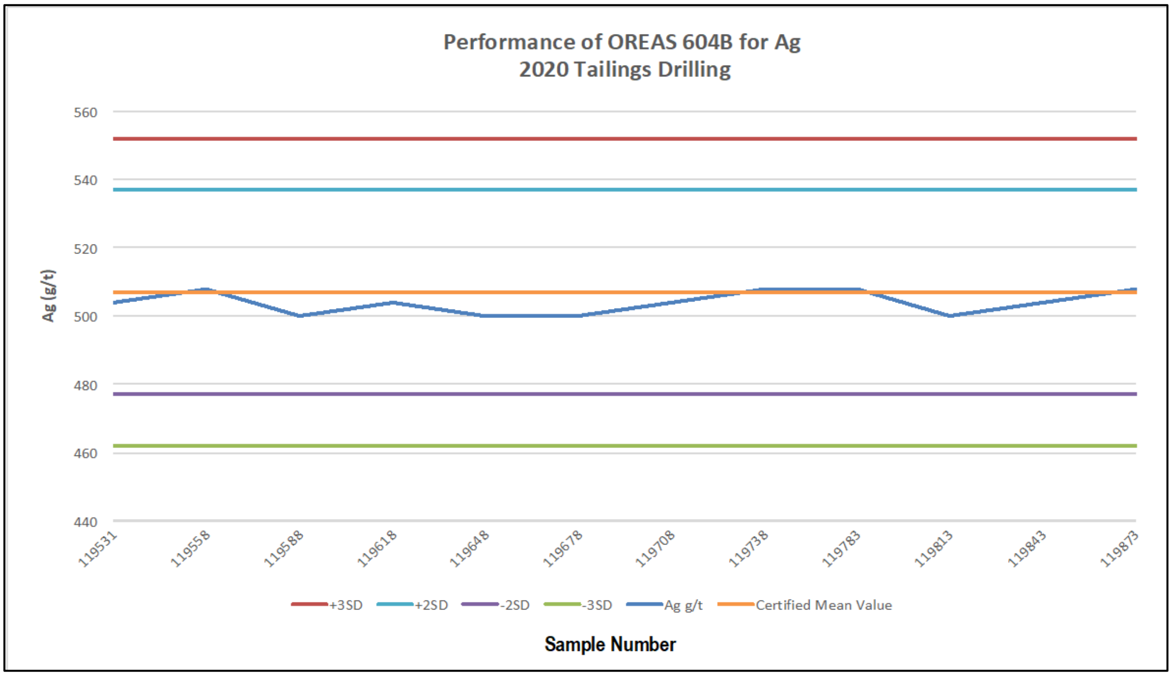


Figure 11.18 – Performance of CRM OREAS 604b for Ag

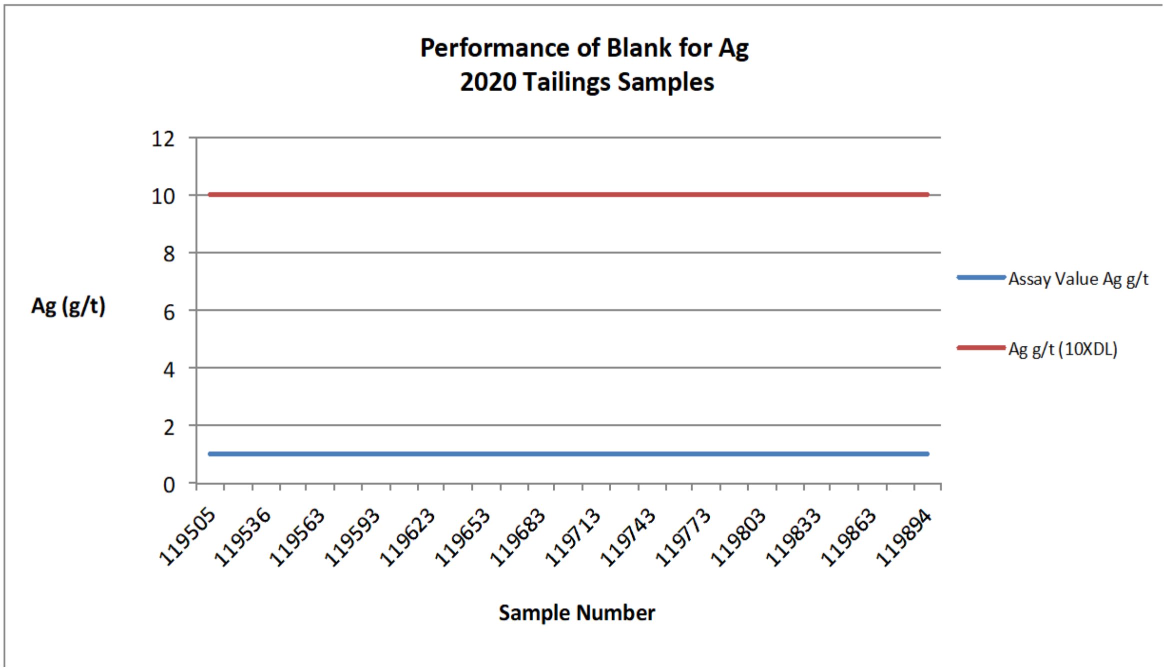


The author of this Technical Report section considers that the CRMs demonstrate acceptable accuracy in the Zgounder 2020 tailings drilling data.

11.2.3.1 Performance of Blanks

Blanks were inserted into the analysis stream approximately every 15 samples. All blank data for Ag are graphed (Figure 11.19). All data plot below the set tolerance limit.

Figure 11.19 – Performance of Blanks

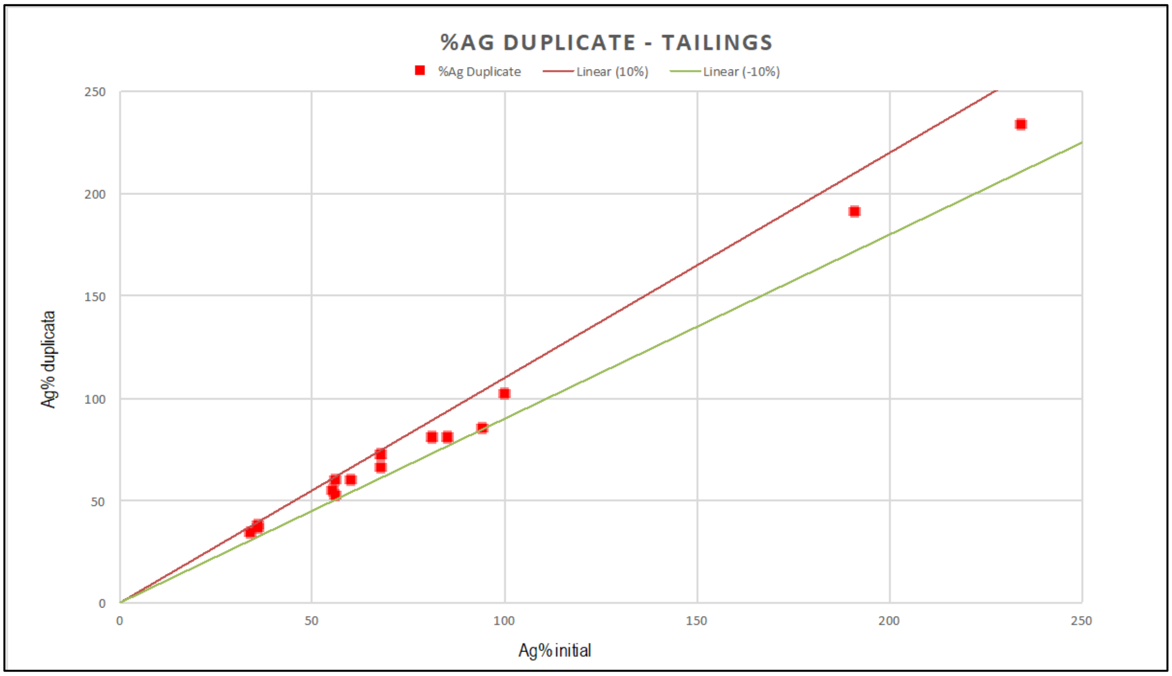


The QP of this Technical Report section does not consider contamination to be an issue for the Zgounder 2020 Tailings drilling data.

11.2.3.2 Performance of Duplicates

Duplicate data for silver were examined for the 2020 Tailings drilling program at the Zgounder Project. There were 25 duplicate pairs in the dataset. The data were graphed (Figure 11.20) and found to have acceptable precision.

Figure 11.20 – Performance of Duplicates



11.3 2020 AYA Umpire Sampling Program

Aya carried out a comprehensive umpire-sampling program to confirm the integrity of the analytical results from several of the Company's drilling campaigns. Select pulverized pulp samples were submitted for check assaying at a secondary laboratory (umpire lab), to check original analyses performed at a primary laboratory. The majority of check assays were conducted at ALS in Seville, Spain, with the remainder analyzed at AfriLab, in Marrakesh (Table 11.6).

Table 11.6 – Summary of Check Samples sent to a Secondary Lab in 2020

Drilling Period	Primary Lab	Secondary Lab	No. Check Samples	QC Samples (%)
RC 2013	ALS Val d'Or	ALS, Seville, Spain	106	6.9
T28 prior 2019	ZMSM	AfriLab	209	3.6
T28 2020 to Feb 2021	ZMSM	ALS, Seville, Spain	292	8.1
DDH 2020	AfriLab	ALS, Seville, Spain	1,071	11.0

A total of 1,071 check samples were submitted to ALS, Spain during the 2020 diamond drilling campaign, representing 11.0% of the total batch of samples sent to the primary laboratory, AfriLab, Marrakesh. Both the original samples and check assays were analyzed by fire assay with gravimetric finish.

Additional check analyses were also completed at a secondary laboratory on samples from the following drilling programs:

- 2013 RC Drilling:

  106 samples were sent for check analyses at ALS, Spain for comparison with original analyses performed at ALS Val d'Or in Canada. Both the original samples and check assays were analyzed by fire assay with gravimetric finish;

- Prior to 2019 T28 Production Drilling:

  209 samples were sent for check analyses at AfriLab, Marrakesh for comparison with original analyses performed at the Zgounder Mine laboratory. The original samples were analyzed by aqua regia method with ICP- AES finish and check assays were analyzed by fire assay with gravimetric finish; and

- 2020 T28 Production Drilling:

  292 samples were sent for check analyses at ALS, Spain for comparison with original analyses performed at the Zgounder Mine laboratory. The original samples were analyzed by aqua regia method with ICP-AES finish and check assays were analyzed by fire assay with gravimetric finish.

A range of QC samples were also included with the pulp samples selected for check assaying for the 2020 DDH and 2020 – February 21 T28 drilling, including CRMs, blanks and duplicates. Unfortunately, none of the CRMs could be assayed, due to insufficient material. No external QC blanks or duplicates were included in the 2013 RC or prior to 2019 T28 drilling check assaying data.

QC results of the blank and duplicate data for the 2020 DDH and 2020 to February 2021 T28 drilling data were reviewed by the author and all blank data fell below four times the lower detection limit for silver. Duplicate data were viewed on scatter plots and the 2020 DDH and majority of the 2020 to February 2021 T28 drilling data showed reasonable reproducibility.

The author of this Technical Report section reviewed the umpire assay results for all four (4) programs, and comparisons were made between the primary lab results and the umpire lab results with the aid of scatter plots (Figures 11.21 to 11.28). As expected, lower grades are less reproduceable closer to lower detection limits for all four comparisons and data for both T28 drilling comparisons were also less reproducible, due to the difference in analytical methods between the primary and secondary laboratories.

Comparison of the 106 samples from the 2013 RC drilling program, between primary lab ALS Val d'Or, Canada and ALS, Seville, Spain, reveals a low bias in the reported primary lab results and an R^2 value of 0.936 (Figure 11.21). Removal of 12 probable outliers yields an improved R^2 value of 0.997 and decreases the bias shown in the original data (Figure 11.22). Both the original samples and check assays were analyzed by fire assay with gravimetric finish.

The 209 samples from the from T28 production drilling carried out between 2015 through 2019, between primary lab at Zgounder and AfriLab, reveal a low bias in the reported primary lab results and an R^2 value of 0.823 (Figure 11.23). Removal of 19 probable outliers gives an improved R^2 value of 0.961 and decreases the bias shown in the original data (Figure 11.24). As noted previously, the original samples were analyzed by Aqua Regia method with ICP-AES finish and the check assays were analyzed by fire assay with gravimetric finish.

The 292 samples from the T28 production drilling carried out during 2020 to February 2021, between primary lab ZMSM and AfriLab, reveals a low bias in the reported primary lab results and an R^2 value of 0.852 (Figure 11.25). Removal of 46 probable outliers gives an improved R^2 value of 0.977 and decreases the bias shown in the original data (Figure 11.26). As for the 2019 and earlier T28 drilling data, the original samples were analyzed by Aqua Regia method with ICP-AES finish and the check assays were analyzed by fire assay with gravimetric finish.

The 1,071 samples from the from 2020 DDH check assaying, between primary lab AfriLab and ALS (Spain), reveals a slight low bias in the reported primary lab results and an R^2 value of 0.980 (Figure 11.27). Removal of 57 probable outliers gives an improved R^2 value of 0.990 and virtually eliminates the slight bias shown in the original data (Figure 11.28). Both the original samples and check assays were analyzed by fire assay with gravimetric finish.

Probable outliers were removed from the datasets to reflect the global reproducibility of the Mineral Resource data and overall performance of Aya's primary laboratories. Excluded outliers likely reflect sample mix-ups in the field or lab, or potential issues with lab processes. The latter possibility, however, is difficult to investigate without sufficient QC samples inserted into the Umpire Lab assaying sample stream.

The author of this Technical Report section does not consider any biases exhibited in the data to be of material impact to the current Mineral Resource Estimate, as the primary laboratories are generally under-reporting silver results. There may, however, be material impacts on the local scale. Overall, the author of this Technical Report section considers the data to be acceptable for use in the current Mineral Resource Estimate.

Figure 11.21 – 2013 RC Drilling Umpire Sampling Results for Ag (All Data)

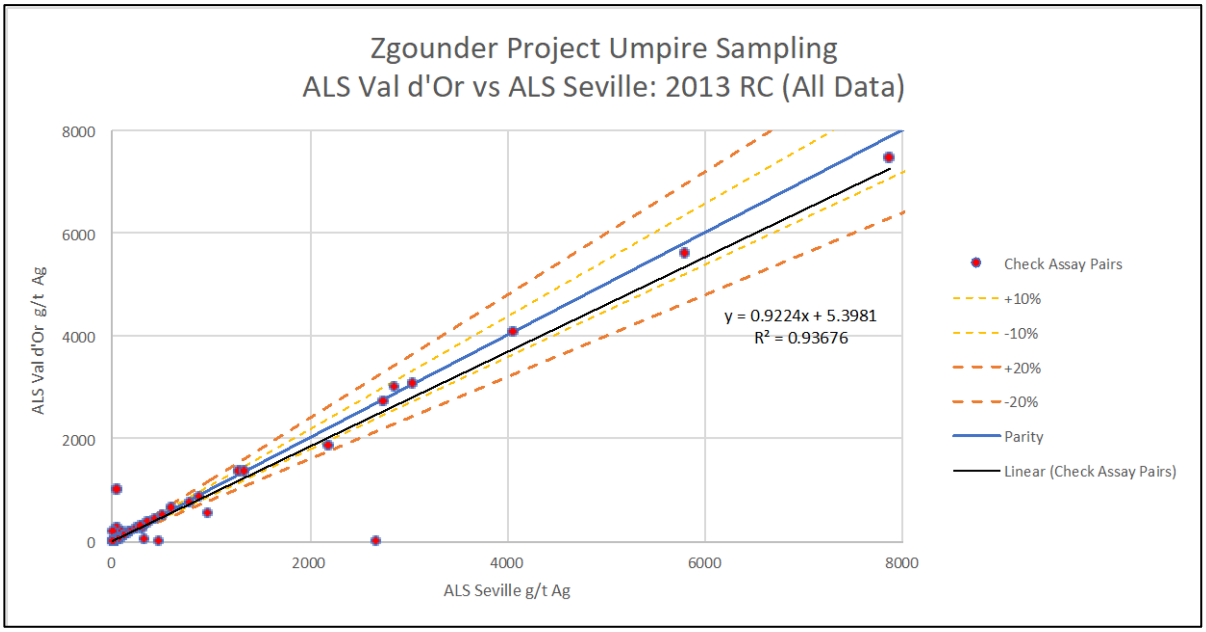


Figure 11.22 – 2013 RC Drilling Umpire Sampling Results for Ag (Probable Outliers Excluded)

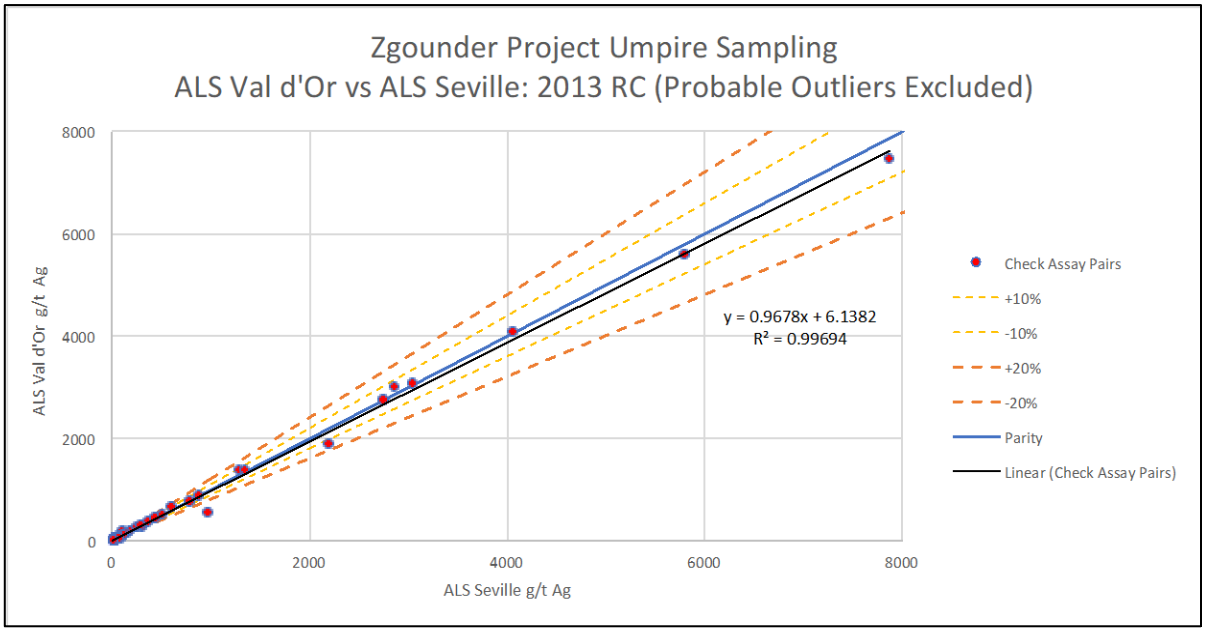


Figure 11.23 – Prior to 2019 T28 Production Drilling Umpire Sampling Results for Ag (All Data)

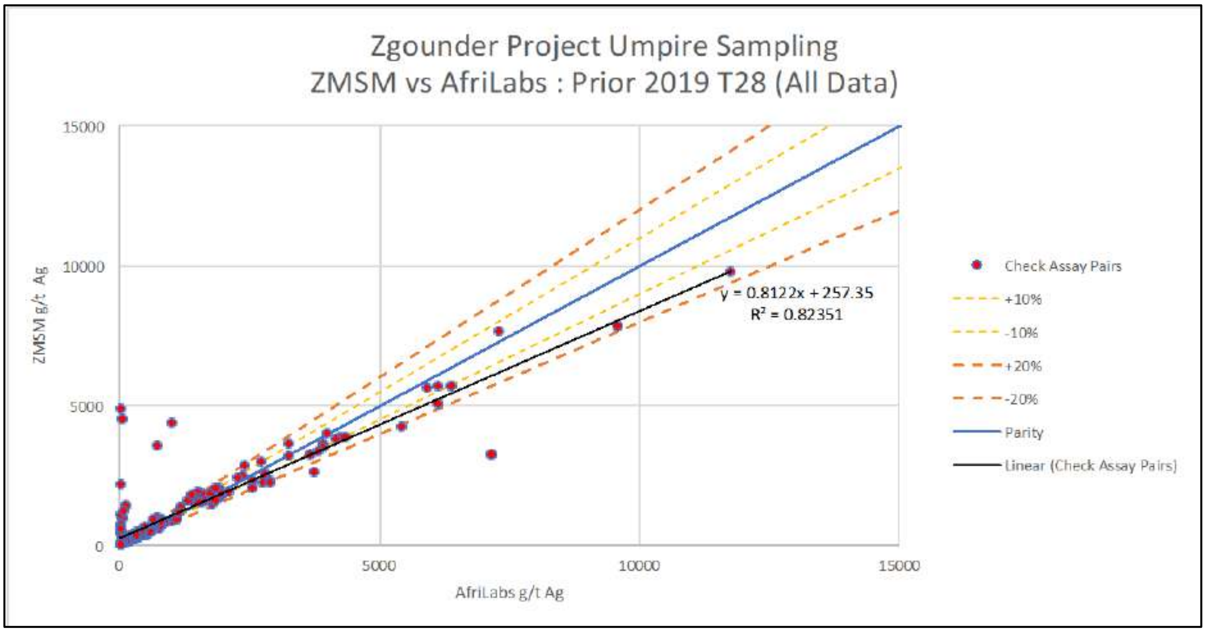


Figure 11.24 – Prior to 2019 T28 Production Drilling Umpire Sampling Results for Ag (Probable Outliers Excluded)

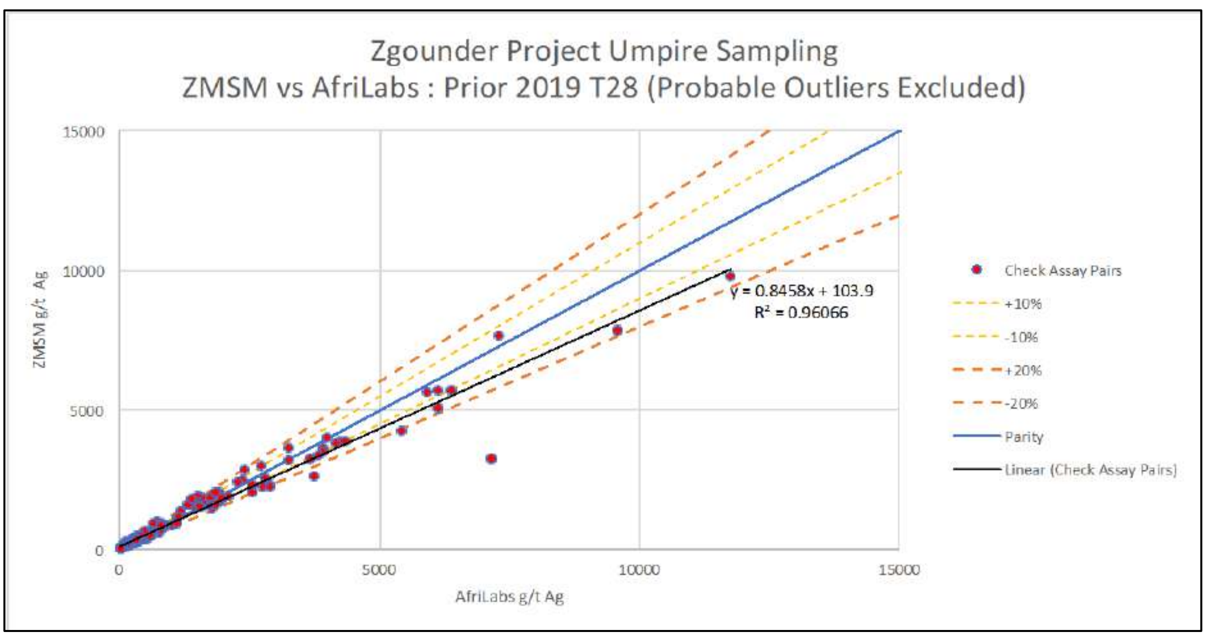


Figure 11.25 – 2020-2021 T28 Production Drilling Umpire Sampling Results for Ag (All Data)

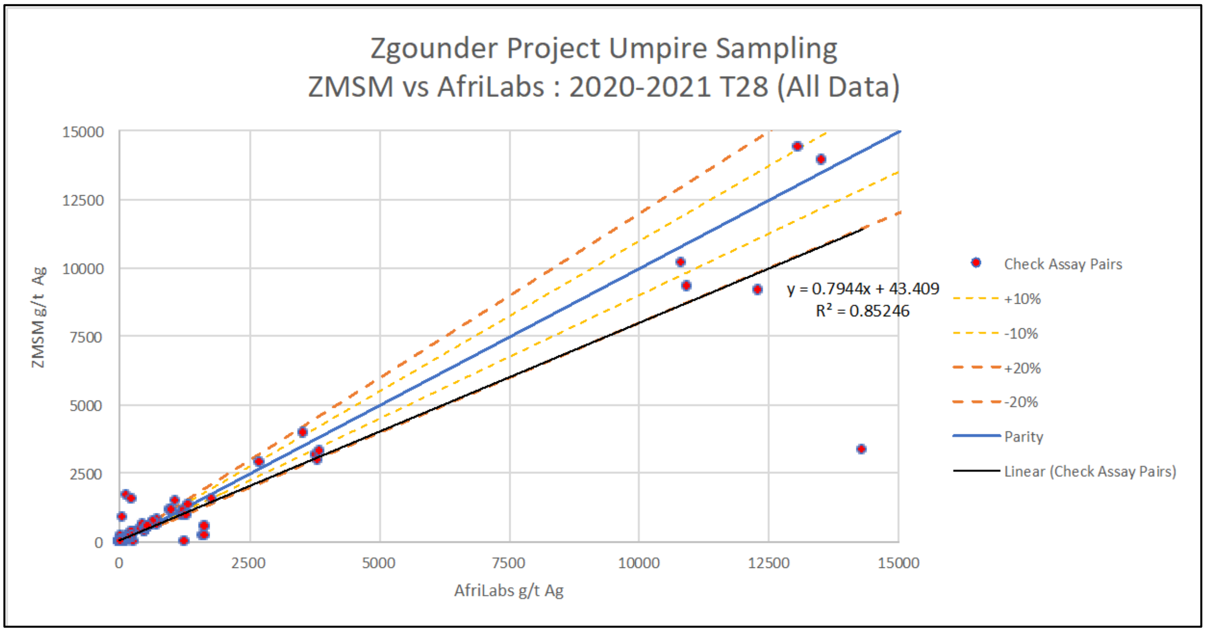


Figure 11.26 – 2020 to February 2021 T28 Production Drilling Umpire Sampling Results for Ag (Probable Outliers Excluded)

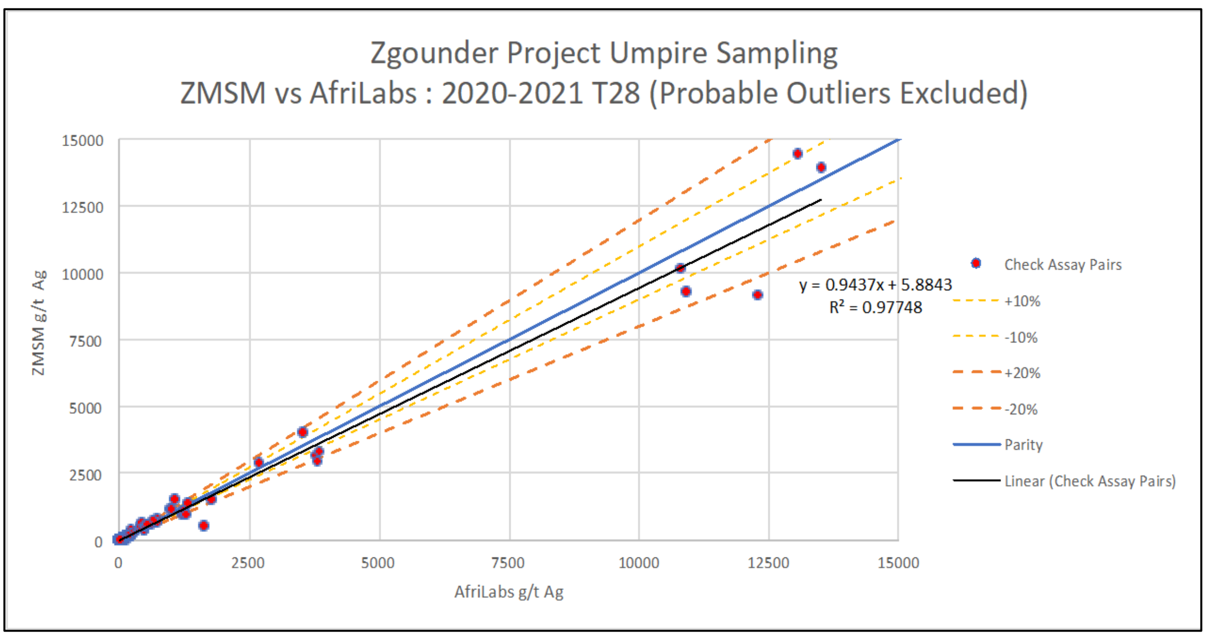


Figure 11.27 – 2020 Diamond Drilling Umpire Sampling Results for Ag (All Data)

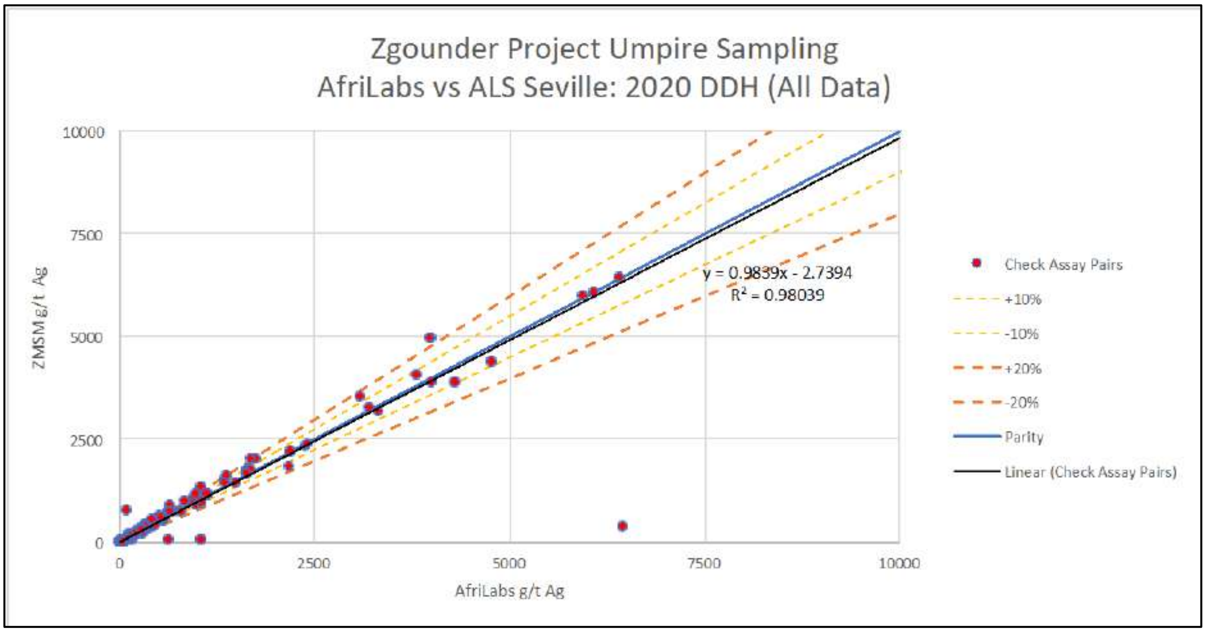
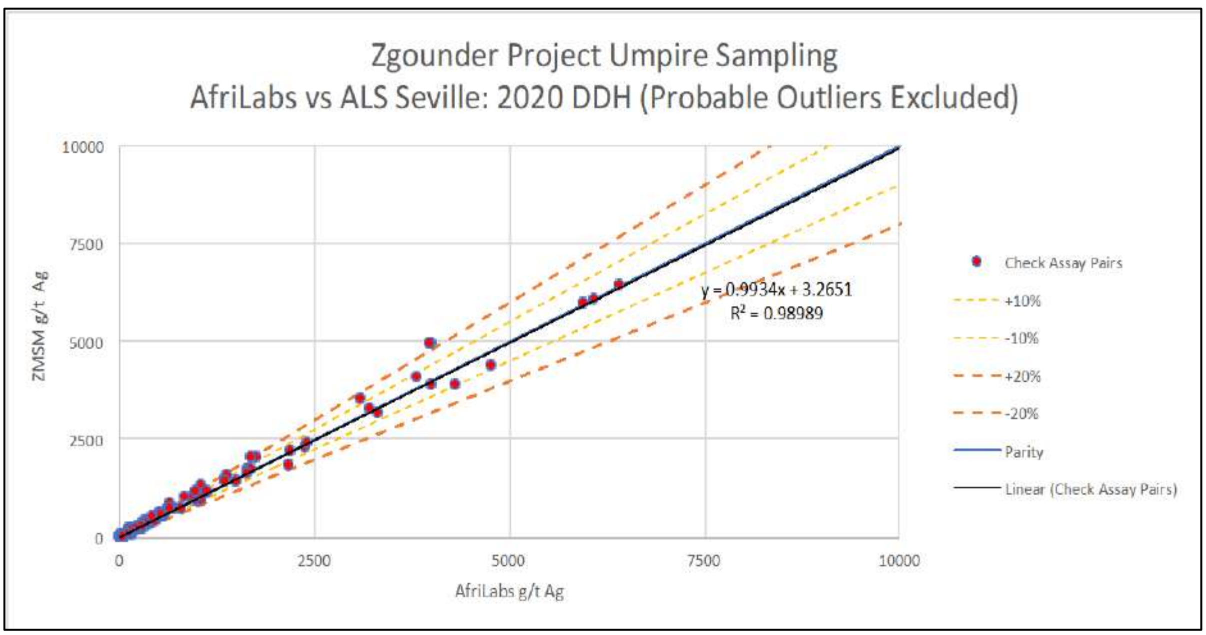


Figure 11.28 – 2020 Diamond Drilling Umpire Sampling Results for Ag (Probable Outliers Excluded)



11.3.1 Post-February 2021 DDH Program – Analyses Performed at AfriLab

11.3.1.1 Performance of Certified Reference Materials

CRMs continued to be inserted into the analysis stream approximately every 50 samples. Four (4) CRMs were used during the post-February 2021 diamond drilling program to monitor silver performance; OREAS 621, OREAS 604B, OREAS 605B and OREAS 139 (Tables 11.4 and 11.5). Criteria for assessing CRM performance is described in Section 11.2.1.1. The CRM results are presented in Figures 11.29 through 11.32. There were 113 data points for OREAS 621, 165 for OREAS 604B, 189 for OREAS 605B and 253 for OREAS 139. No failures were recorded for any of the CRMs. However, a slight negative bias was noted for the OREAS 621 and OREAS 605B CRMs.

The author of this Technical Report section considers that the CRMs demonstrate acceptable accuracy in the Zgounder post-February 2021 diamond drilling data.

11.3.1.2 Performance of Blanks

Blanks were inserted into the analysis stream approximately every 50 samples. All blank data for Ag are graphed (Figure 11.33) and data treated as described in Section 11.2.1.2. There were 813 data points to examine. All blank data plotted below the set tolerance limit.

The author of this Technical Report section does not consider contamination to be an issue for the Zgounder post-February 2021 diamond drilling data.

11.3.1.3 Performance of Duplicates

Duplicate data for silver were examined for the post-February 2021 diamond drilling program at the Zgounder Project. There were 724 duplicate pairs in the dataset. The data were graphed (Figures 11.34 and 11.35). Data falling closer to the lower detection level was found to be less precise (Figure 11.35), as expected, however the data overall was found to have acceptable precision.

Figure 11.29 – Performance of C RM OREAS 621 For Ag

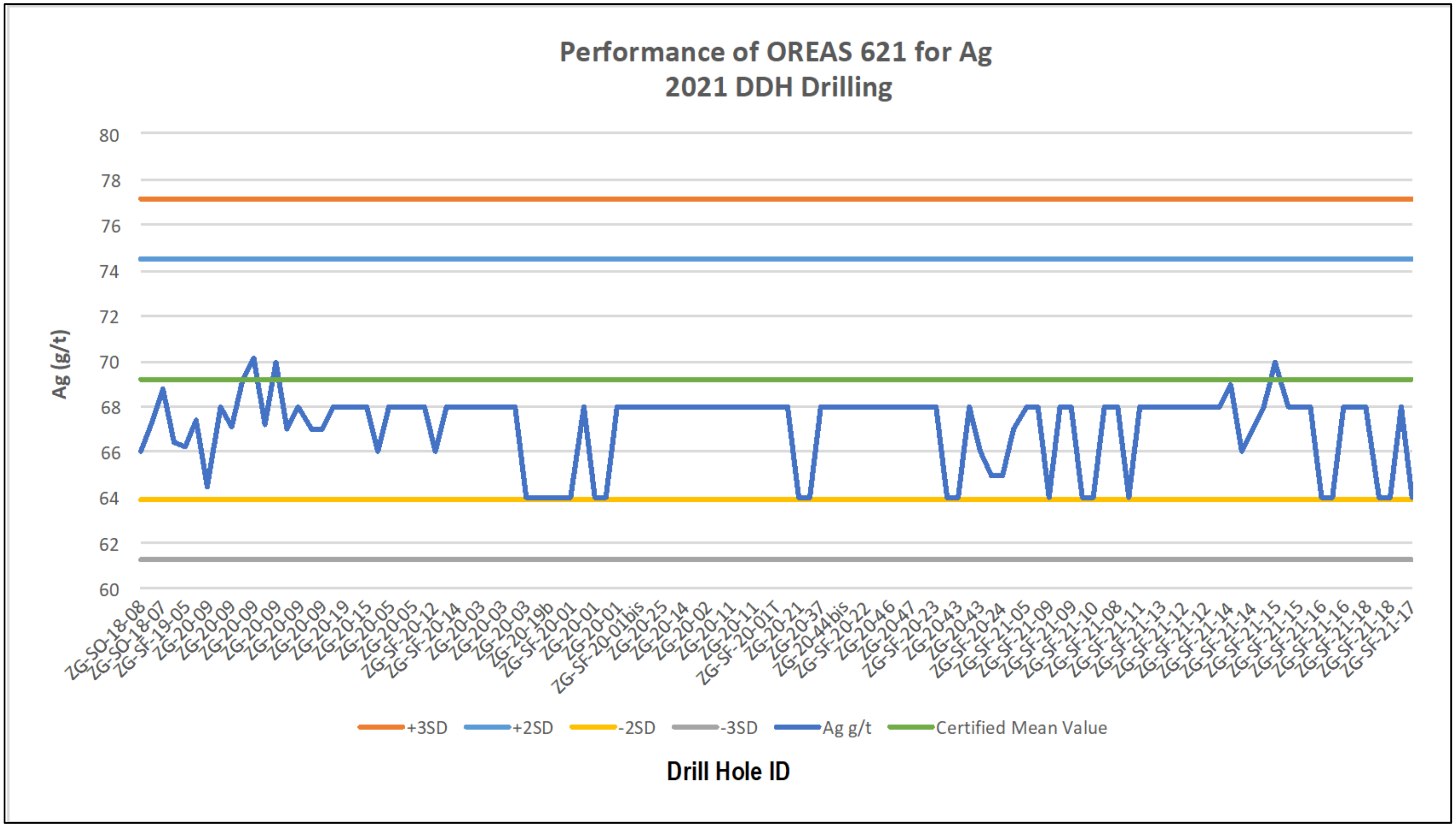


Figure 11.30 – Performance of CRM OREAS 604b For Ag

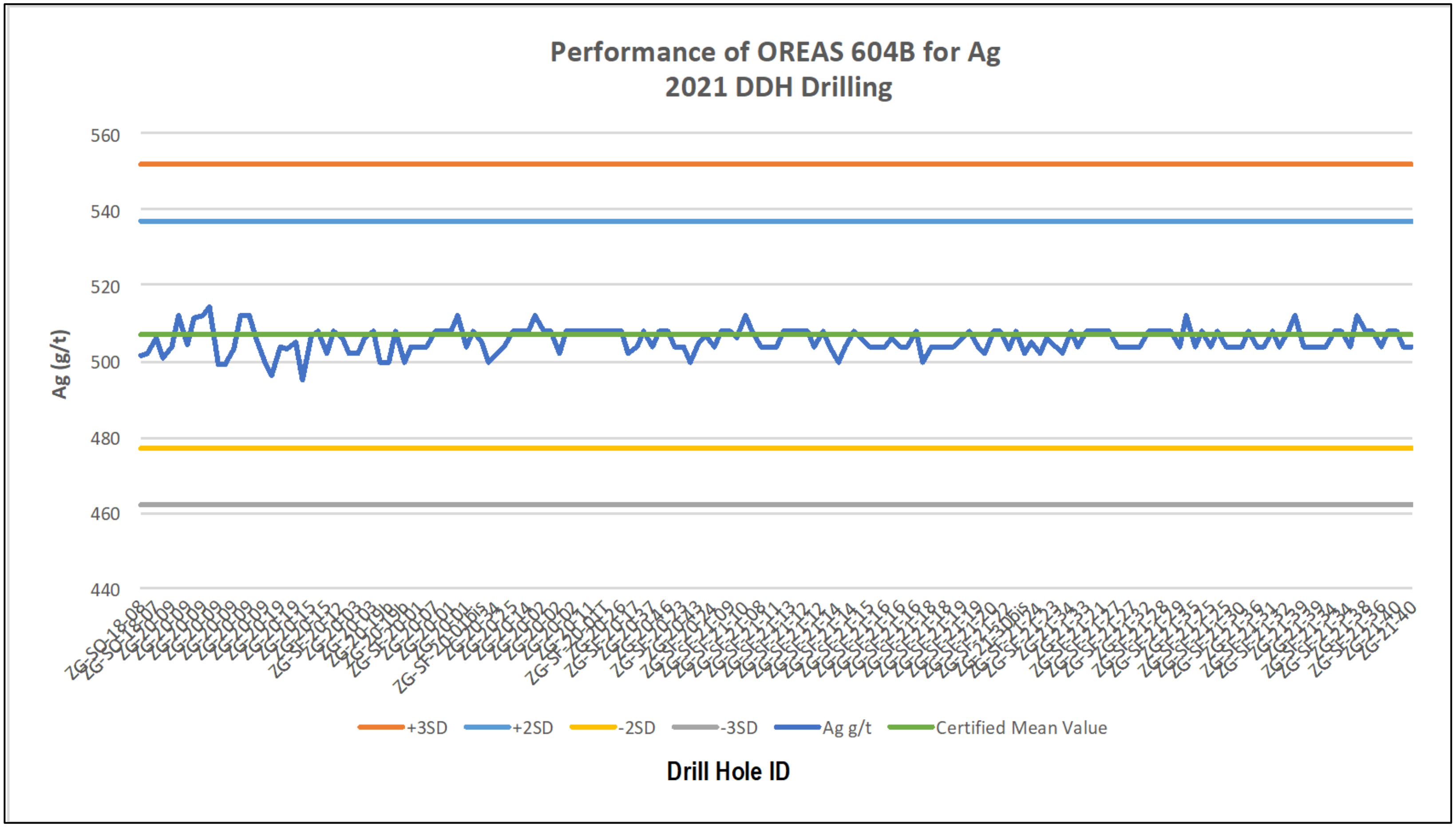


Figure 11.31 – Performance of CRM OREAS 605b For Ag

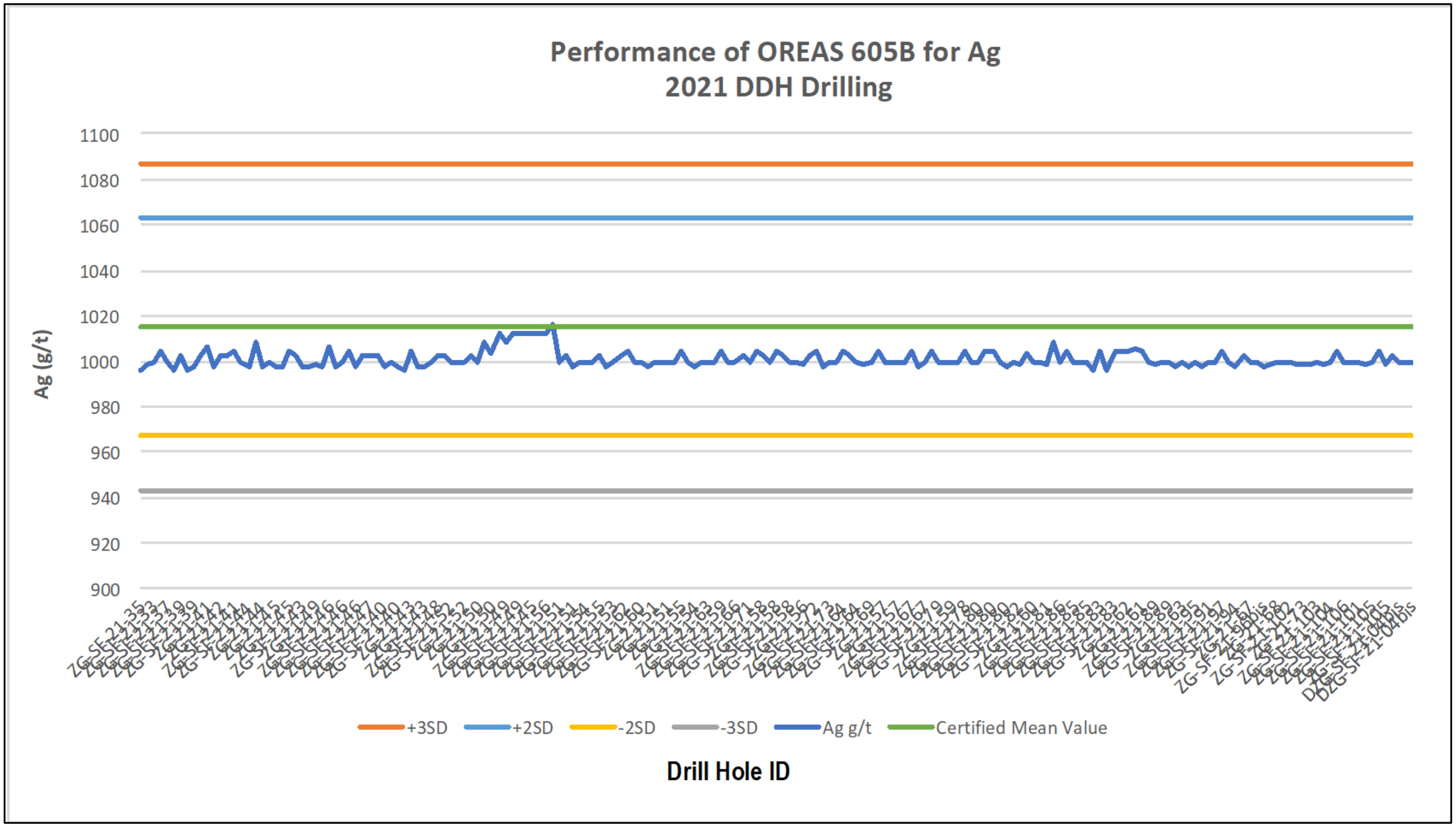


Figure 11.32 – Performance of CRM OREAS 139 For Ag

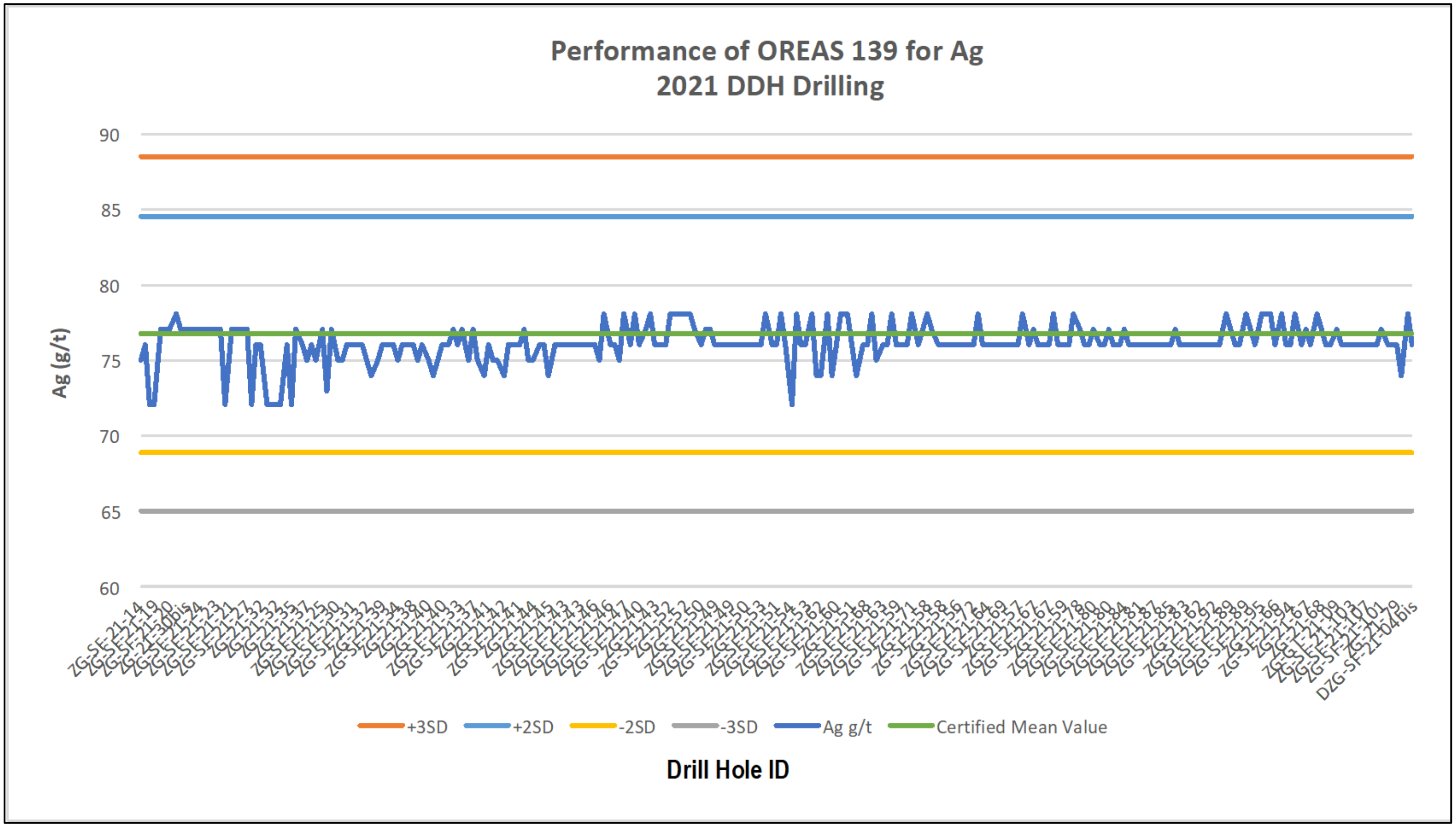


Figure 11.33 – Performance of Blanks

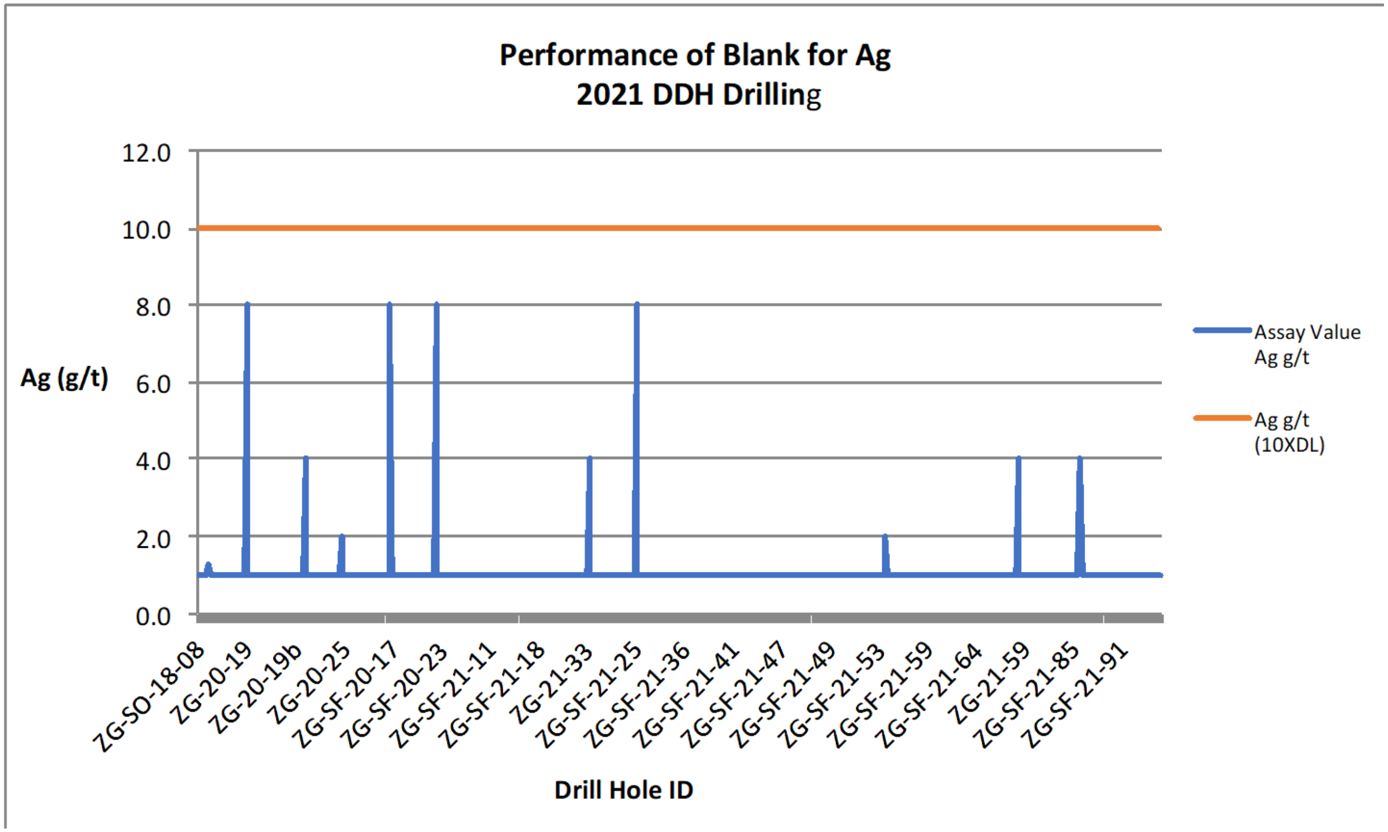


Figure 11.34 – Performance of Duplicates (All Data Points Included)

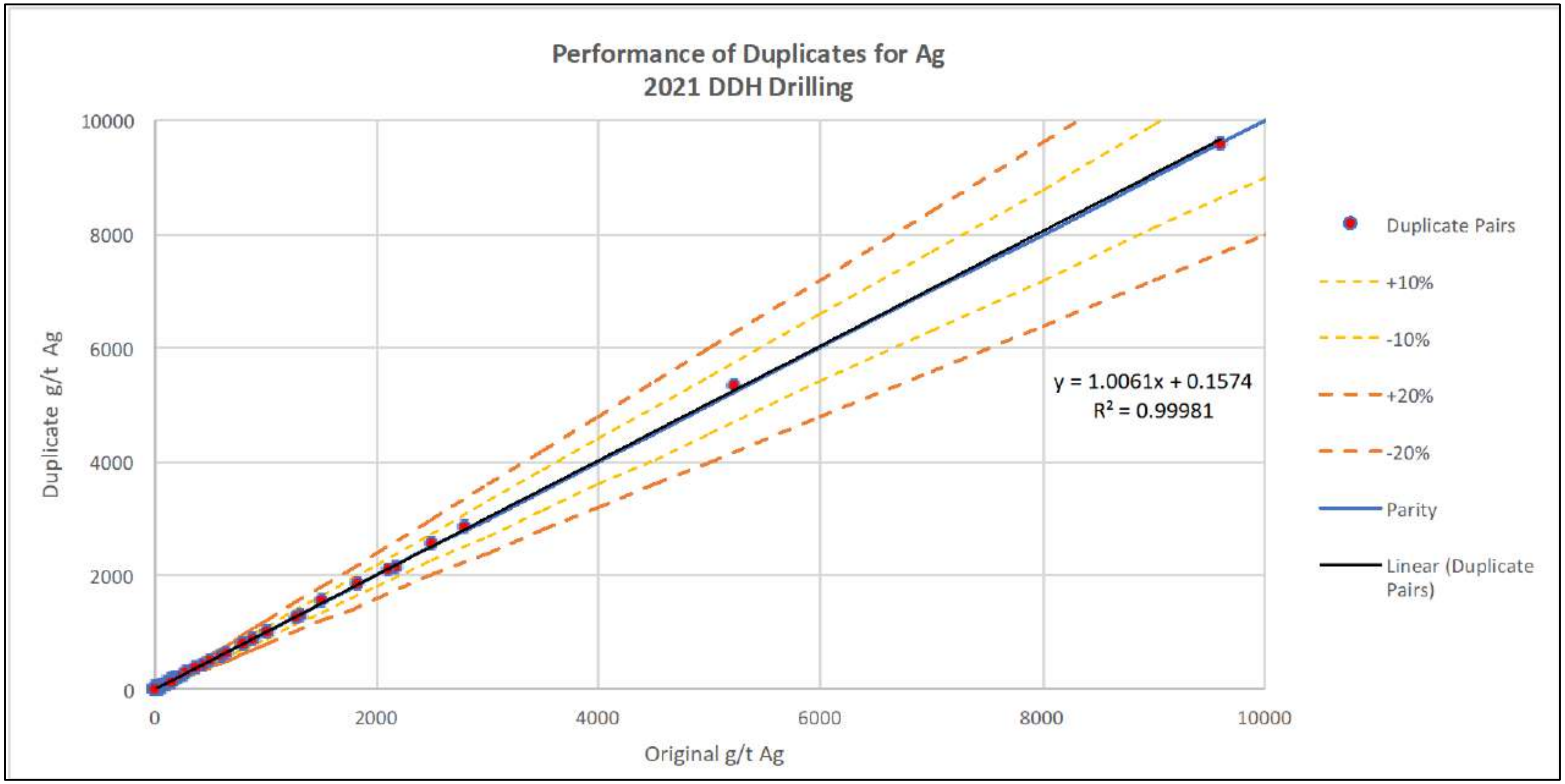
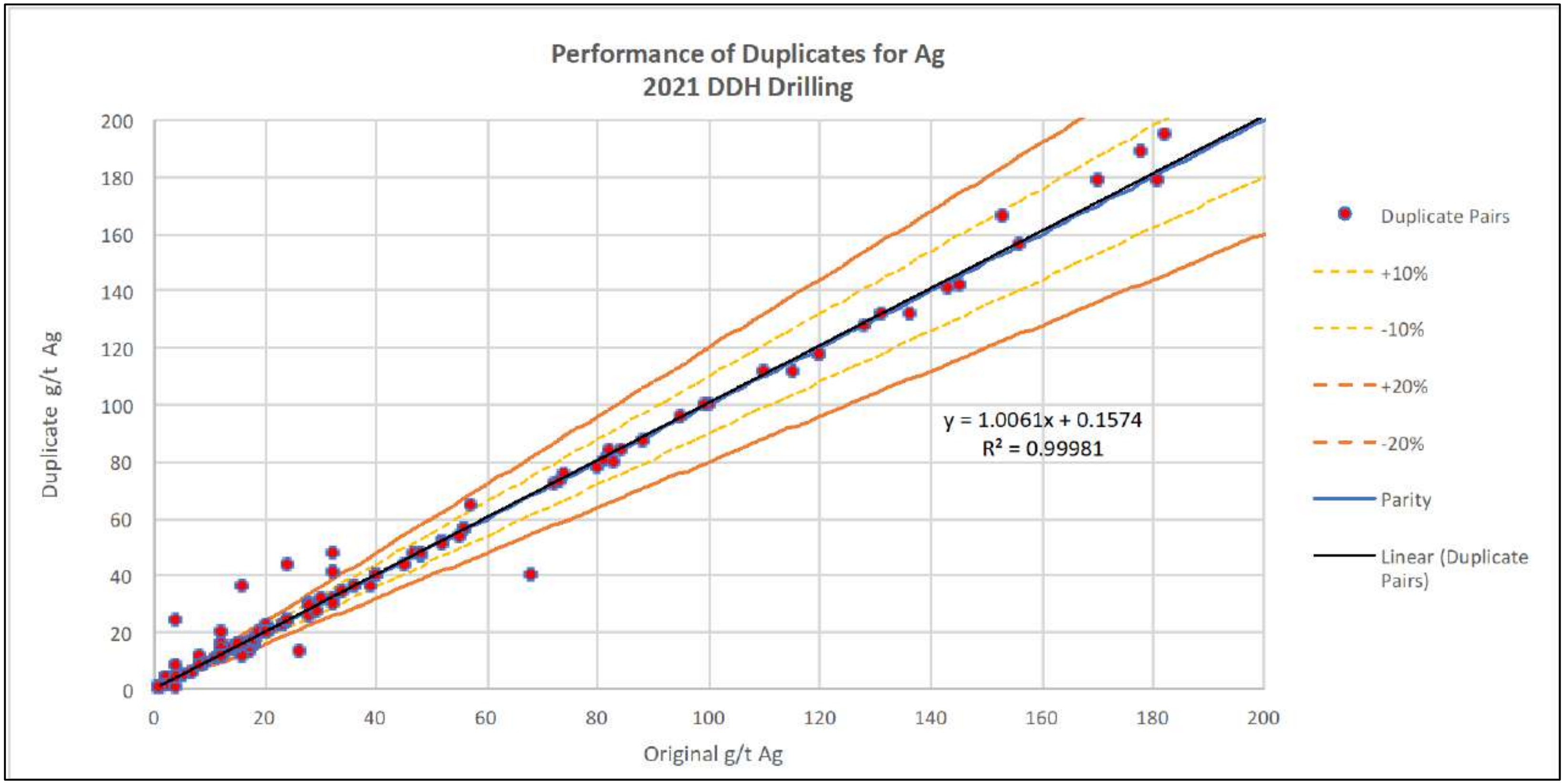


Figure 11.35 – Performance of Duplicates (Data Points <200 g/t Ag Only)



11.3.2 POST-FEBRUARY 2021 T28 PRODUCTION DRILLING – ANALYSES PERFORMED AT ZMSM LABORATORY

The same four (4) OREAS CRMs were used in the post-February 2021 diamond drilling for the T28 production drilling. CRMs were inserted at a rate of approximately 1 in 25 samples. Figures 11.36 to 11.39 depict the CRM results. There were 23 data points for OREAS 621, 24 for OREAS 604B, 45 for OREAS 605B, and 48 for OREAS 139. There was only one (1) failure recorded for the OREAS 621 CRM (Figure 11.36). No failures were recorded for the remaining three (3) CRMs, however low biases were noted for the OREAS 621 and OREAS 605B CRMs.

The author of this Technical Report section considers that the CRMs demonstrate acceptable accuracy in the Zgounder Post-February 2021 T28 production drilling.

11.3.2.1 Performance of Blanks

Blanks were inserted into the analysis stream approximately every 25 samples. All blank data for Ag are graphed (Figure 11.40) and data treated as described in Section 11.2.1.2. There were 137 data points to examine. All blank data plotted below the set tolerance limit.

The author of this Technical Report section does not consider contamination to be an issue for the Zgounder Post-February 2021 T28 production drilling data.

11.3.2.2 Performance of Duplicates

Duplicate data for silver were examined for the Post-February 2021 T28 production drilling at the Zgounder Mine. There were 23 duplicate pairs in the dataset. The data were graphed (Figure 11.41) and found to have acceptable precision.

Figure 11.36 – Performance of C RM OREAS 621 For Ag

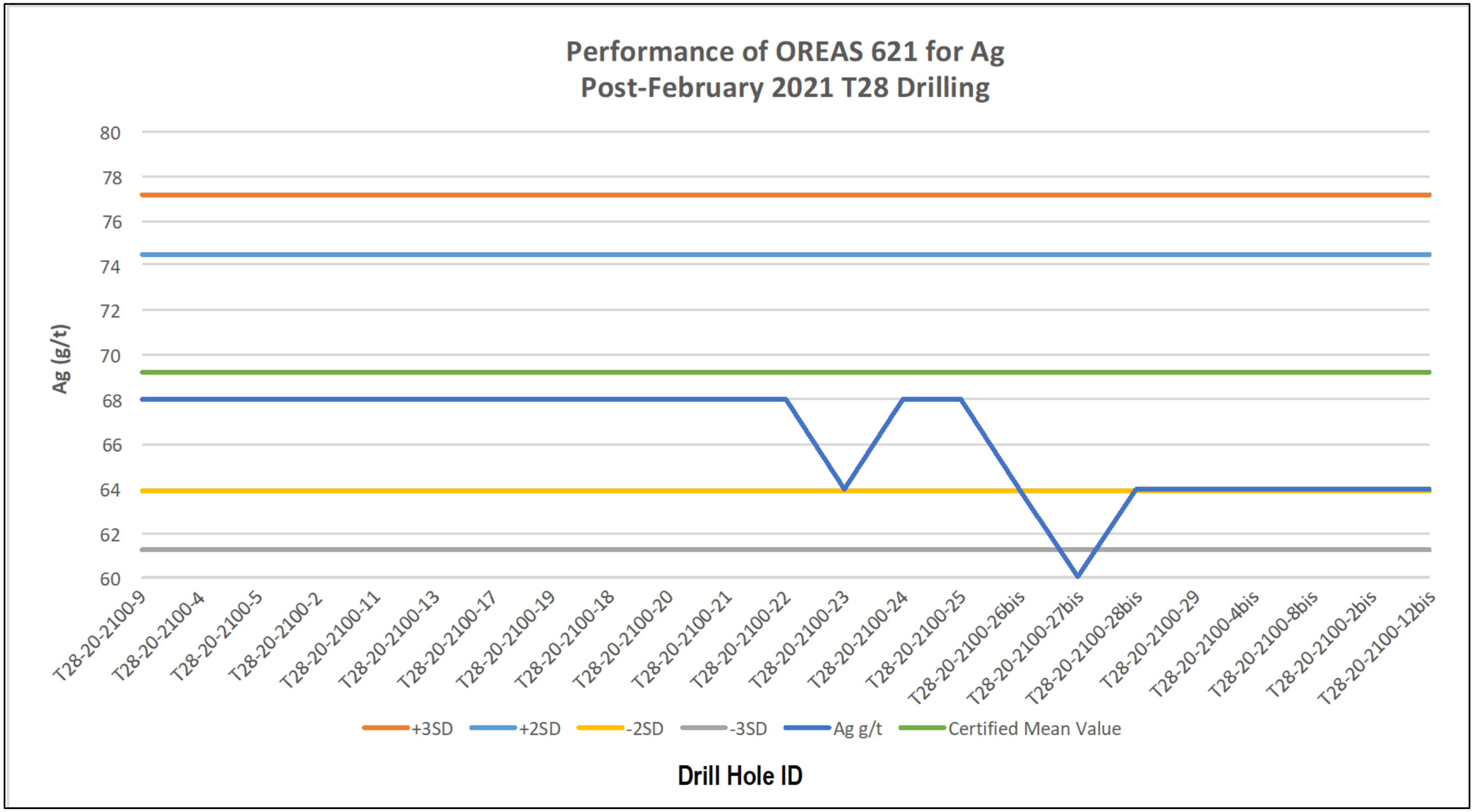


Figure 11.37 – Performance of CRM OREAS 604B For Ag

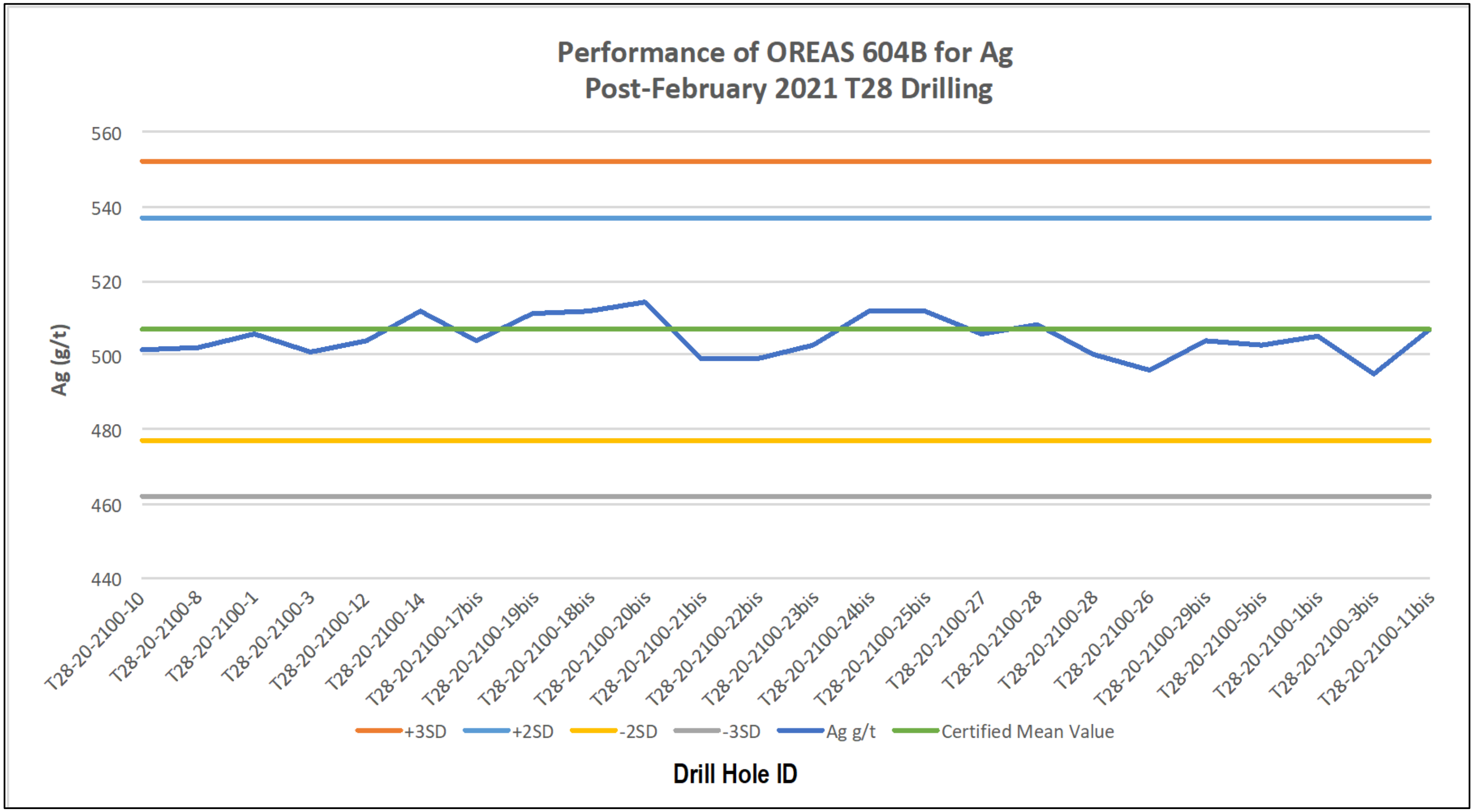


Figure 11.38 – Performance of CRM OREAS 605B For Ag

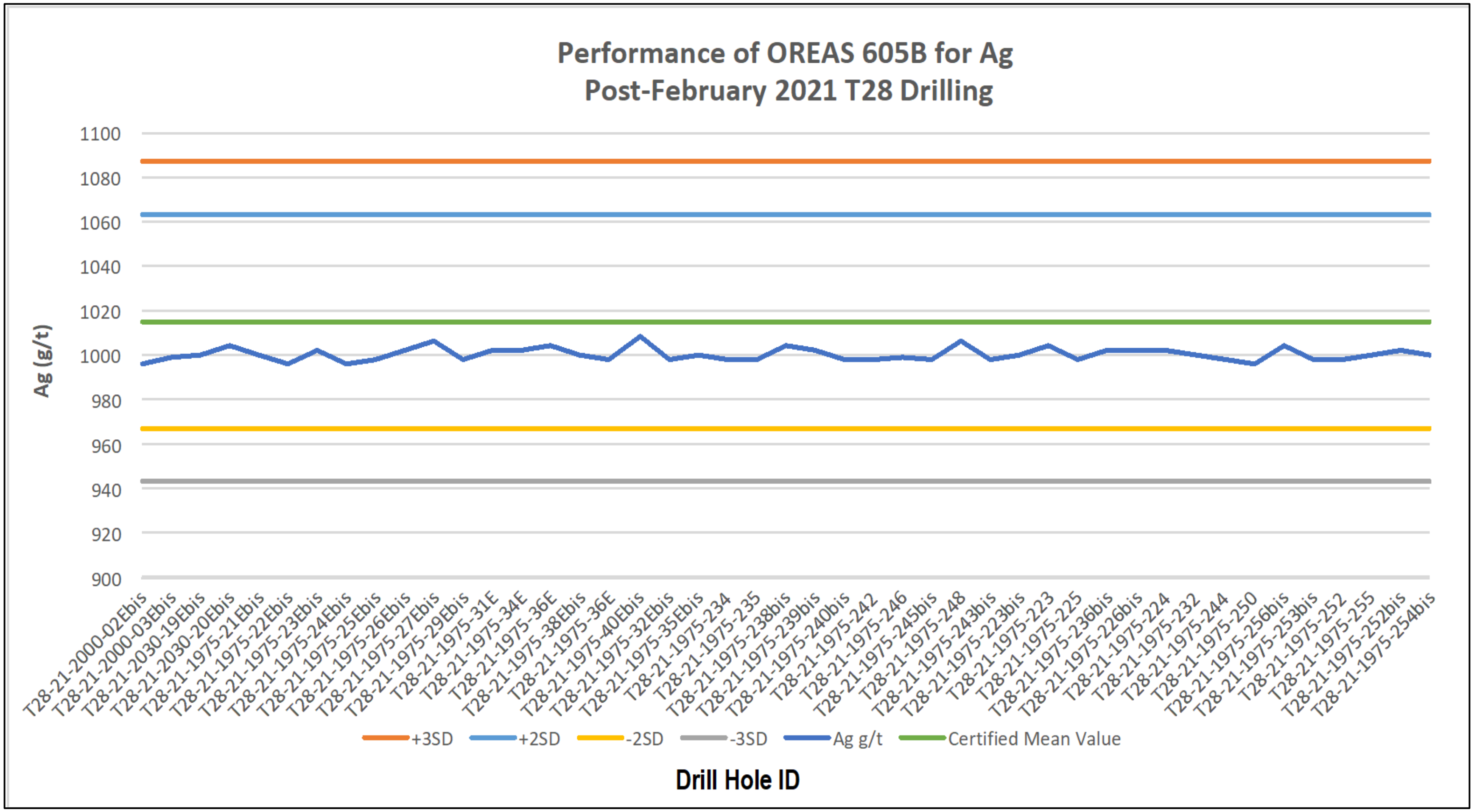


Figure 11.39 – Performance of CRM OREAS 139 For Ag

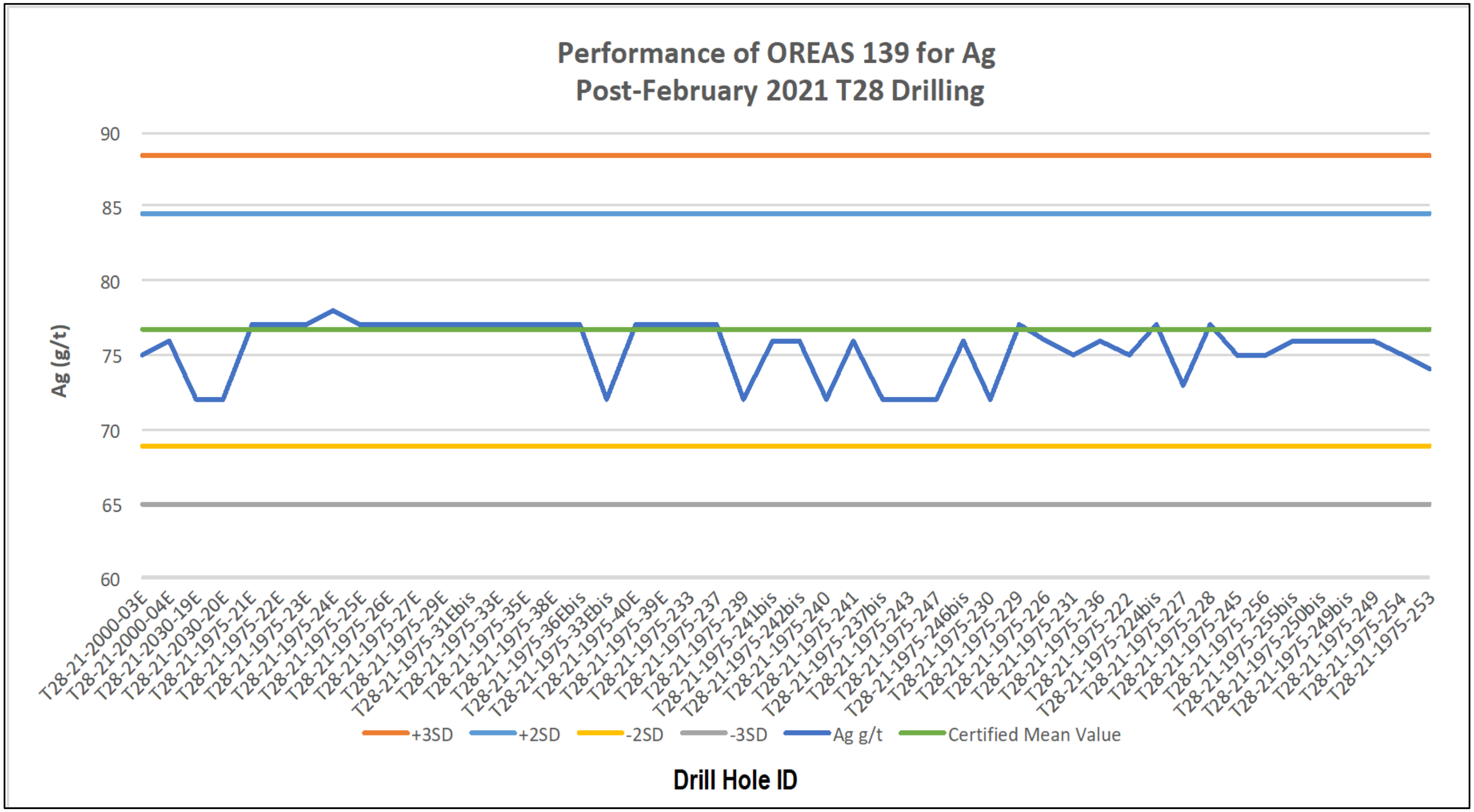


Figure 11.40 – Performance of Blanks

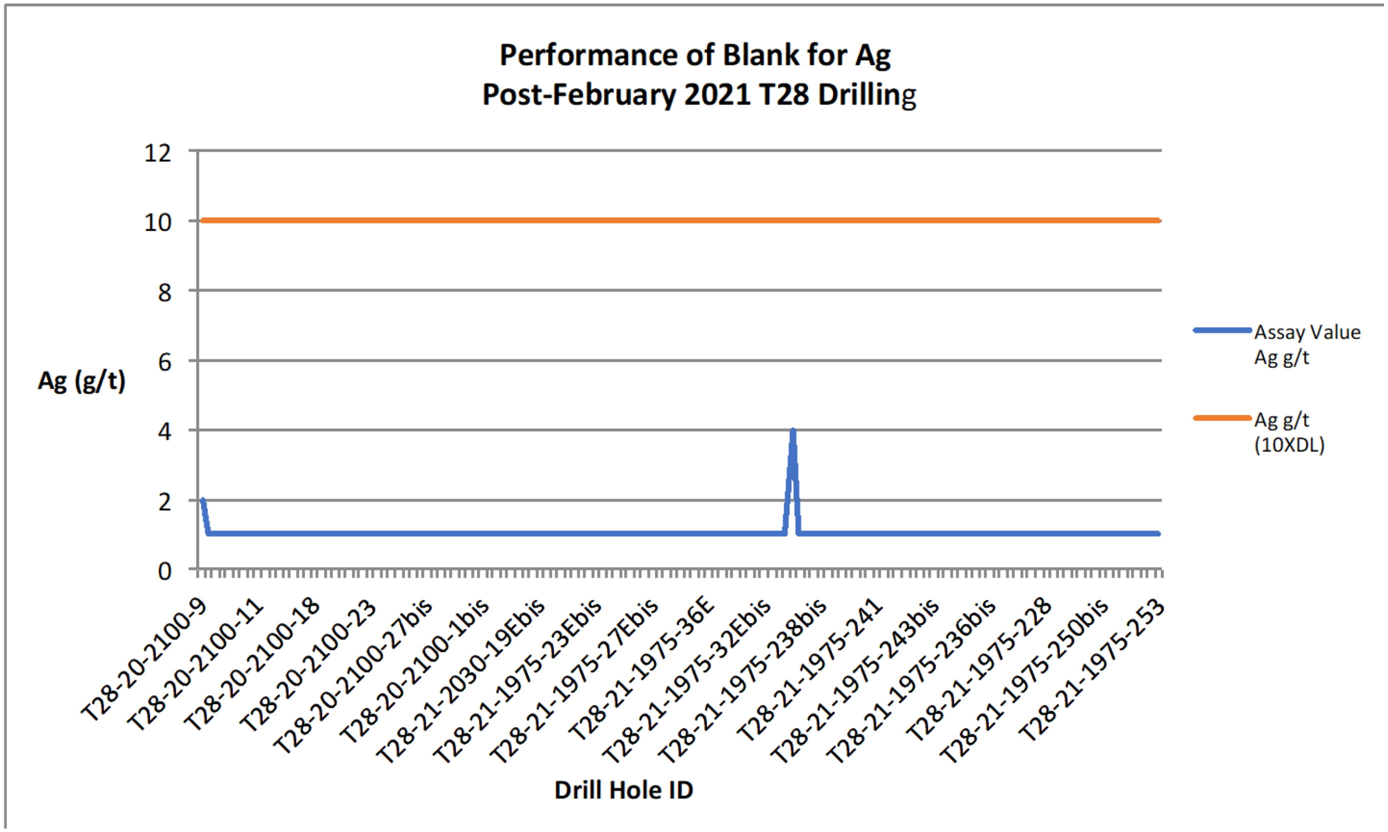
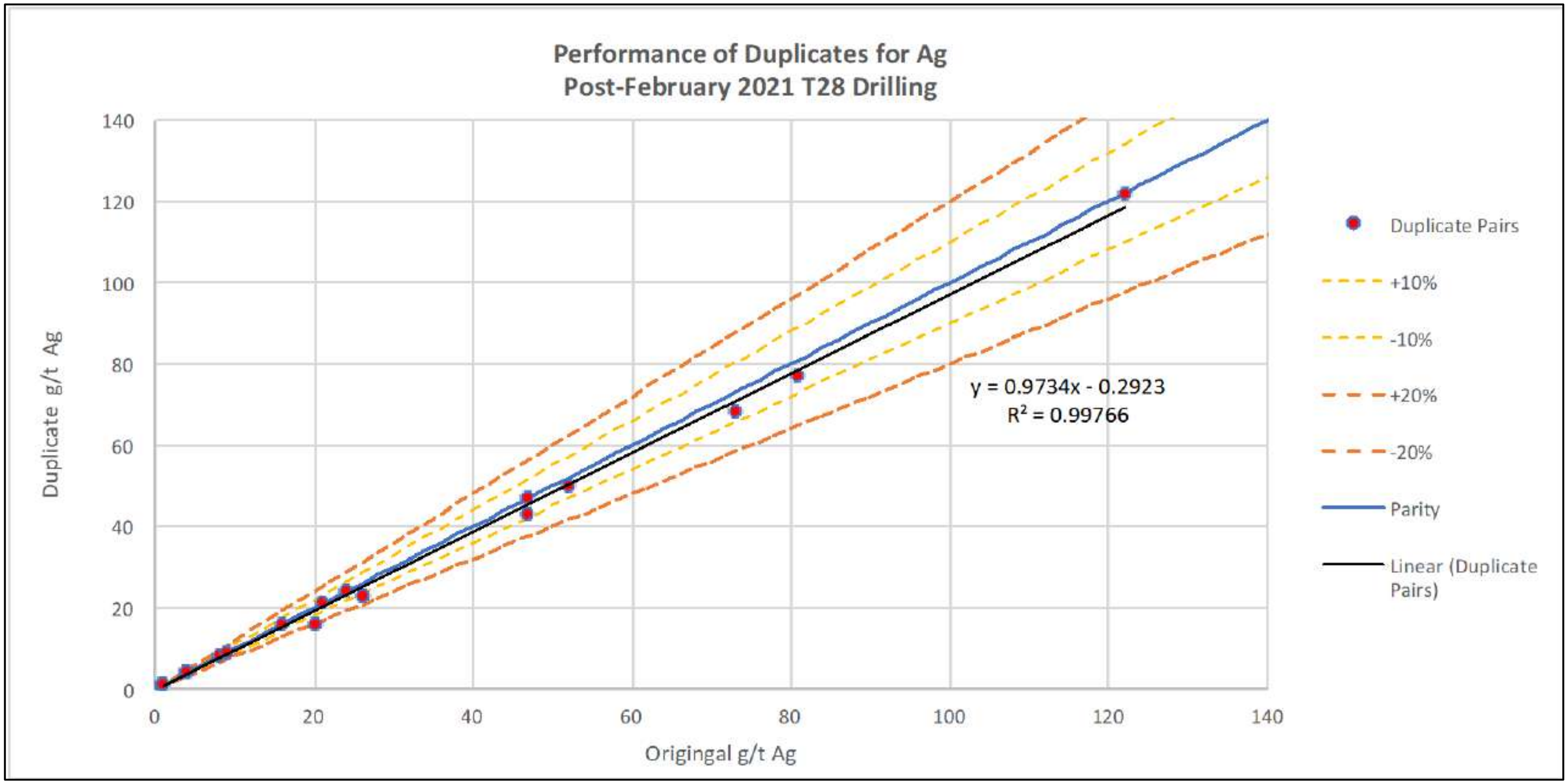


Figure 11.41 – Performance of Duplicates (All Data Points Included)



11.4 Conclusion

Aya implemented and monitored a thorough QA/QC program for the drilling completed at the Zgounder Project over the 2020-2021 period and also completed comprehensive umpire assaying to confirm sampling and analyses undertaken during all drilling campaigns. Examination of QA/QC results for all recent sampling indicates no material issues with accuracy, contamination or precision in the data, and all check assaying programs generally confirm the original data. The author of this Technical Report section recommends that Aya continue with the current QC protocol and continue to monitor QC data in an ongoing basis.

In the opinion of the author of this Technical Report section that sample preparation, security and analytical procedures for the Zgounder Project are adequate and that the data is of good quality and satisfactory for use in the current Mineral Resource Estimate.

12 DATA VERIFICATION

This Section has been summarised and/or largely extracted from the Report available on SEDAR entitled: "Technical Report and Updated Mineral Resource Estimate of the Zgounder Silver Project, Kingdom of Morocco, prepared for Aya Gold & Silver Inc., dated January 28, 2022, prepared by P&E Mining Consultants Inc., Report # 415."

12.1 February 2021 – Drill Hole Database

The authors of this Technical Report section conducted verification of the Zgounder Mine drill hole assay database for silver, by comparison of the database entries with assay certificates, supplied directly to P&E by AfriLab, Marrakesh, Morocco, and Aya's laboratory facilities at the Zgounder Mine. Assay certificates were supplied in comma-separated values (csv) format and Portable Document Format (.pdf) file format.

Assay data ranging from 2014 through February 2021 were verified for the Zgounder Project by the authors. Approximately 16% (10,955 out of 68,774 samples) of the entire database were verified for silver and approximately 10% (4,631 out of 45,839 samples) of the constrained database for silver were verified.

Very few minor discrepancies were encountered in the data, which were not material to the current Mineral Resource Estimate.

12.2 December 2021 – Drill Hole Database

The authors of this Technical Report section conducted further verification of the Zgounder Mine drill hole assay database in December 2021. Data added to the database after February 2021, were verified for silver, by comparison of the database entries with assay certificates, supplied directly to P&E by AfriLab, Marrakesh, Morocco. Assay certificates were supplied in comma-separated values (csv) format and Portable Document Format (.pdf) file format.

Approximately 58% (2,362 out of 4,056 samples) of the constrained data were verified for silver. Very few minor discrepancies were encountered in the data, which were not material to the current Mineral Resource Estimate.

12.3 January 2021 – P&E Site Visit and Independent Sampling

The Zgounder Project was visited by Mr. Antoine Yassa, P.Geo., of P&E, from January 11 to 13, 2021, for the purpose of completing a site visit that included:

- visiting surface and underground drilling and outcrop sites;
- inspection of onsite laboratory;
- GPS location verifications;

- review of database, surveyor, logging, sampling and QC procedures;
- discussions and;
- verification sampling.

Mr. Yassa collected 38 verification samples from 11 diamond and T28 drill holes. All samples were selected from holes drilled in 2020. A range of high, medium and low-grade samples were selected from the stored drill core. Samples were collected by taking a quarter drill core, with the other quarter drill core remaining in the drill core box. Individual samples were placed in plastic bags with a uniquely numbered tag, after which all samples were collectively placed in a larger bag. Mr. Yassa delivered the samples to AfriLab, a certified laboratory in Marrakesh, Morocco for shipment directly to AGAT Laboratories in Mississauga, Ontario for analysis. Samples at AGAT were analyzed for silver by multi-acid digest with ICP-OES finish and also by fire assay with gravimetric finish. Density determinations were measured by pycnometer on all of the samples.

AGAT is an independent lab that developed and implemented at each of its locations a Quality Management System (QMS) designed to ensure the production of consistently reliable data. The system covers all laboratory activities and takes into consideration the requirements of ISO standards. AGAT maintains ISO registrations and accreditations. ISO registration and accreditation provide independent verification that a QMS is in operation at the location in question. AGAT Laboratories is certified to ISO 9001:2015 standards and is accredited, for specific tests, to ISO/IEC 17025:2017 standards.

Results of the Zgounder site visit verification samples for silver are presented in Figures 12.1 and 12.2. Both methods of analyses carried out by AGAT are represented and compared to original Zgounder assays.

Figure 12.1 – Results of January 2021 Ag Verification Sampling by P&E

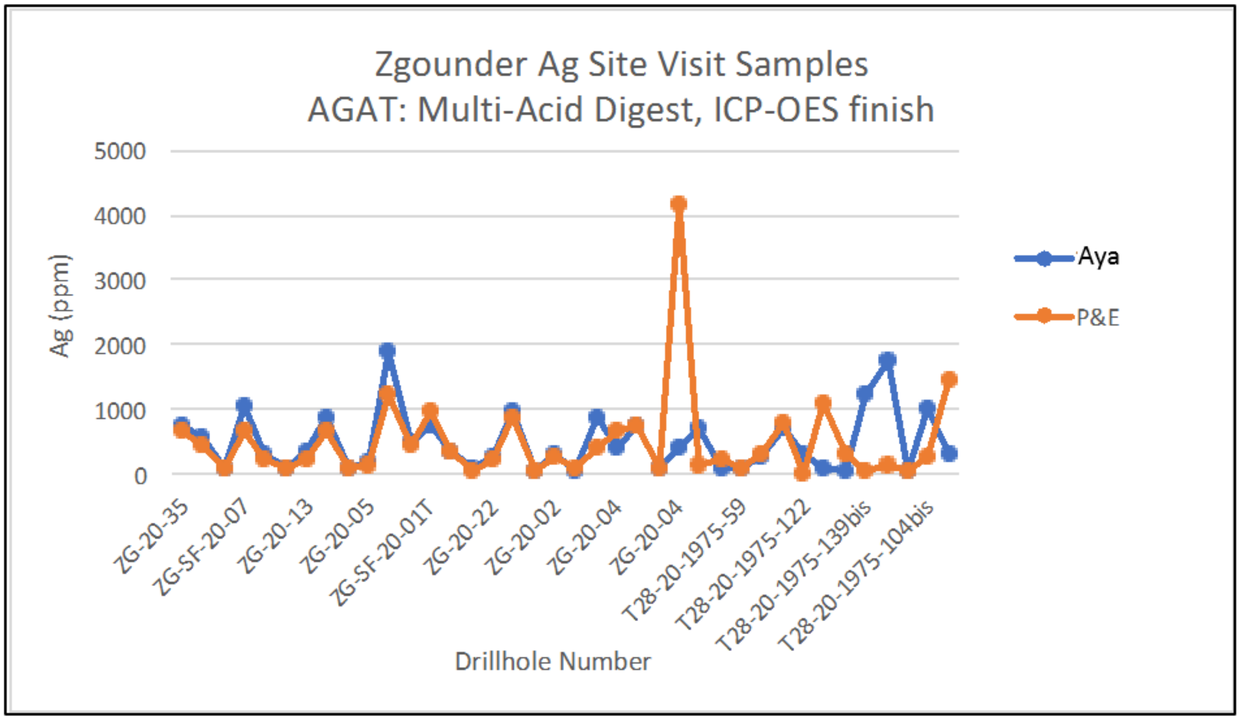
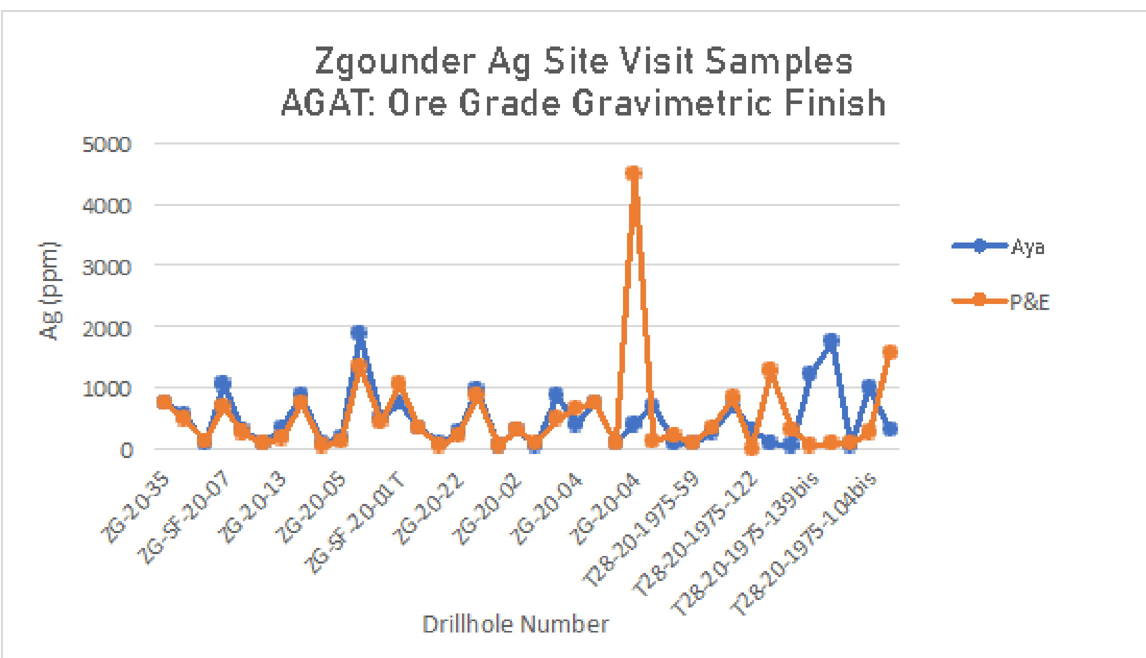


Figure 12.2 – Results of January 2021 Ag Verification Sampling by P&E



12.4 November 2021 – P&E Site Visit and Independent Sampling

The Zgounder Project was visited again by Mr. Yassa, from November 19 to 21, 2021, to carry out a site visit that included:

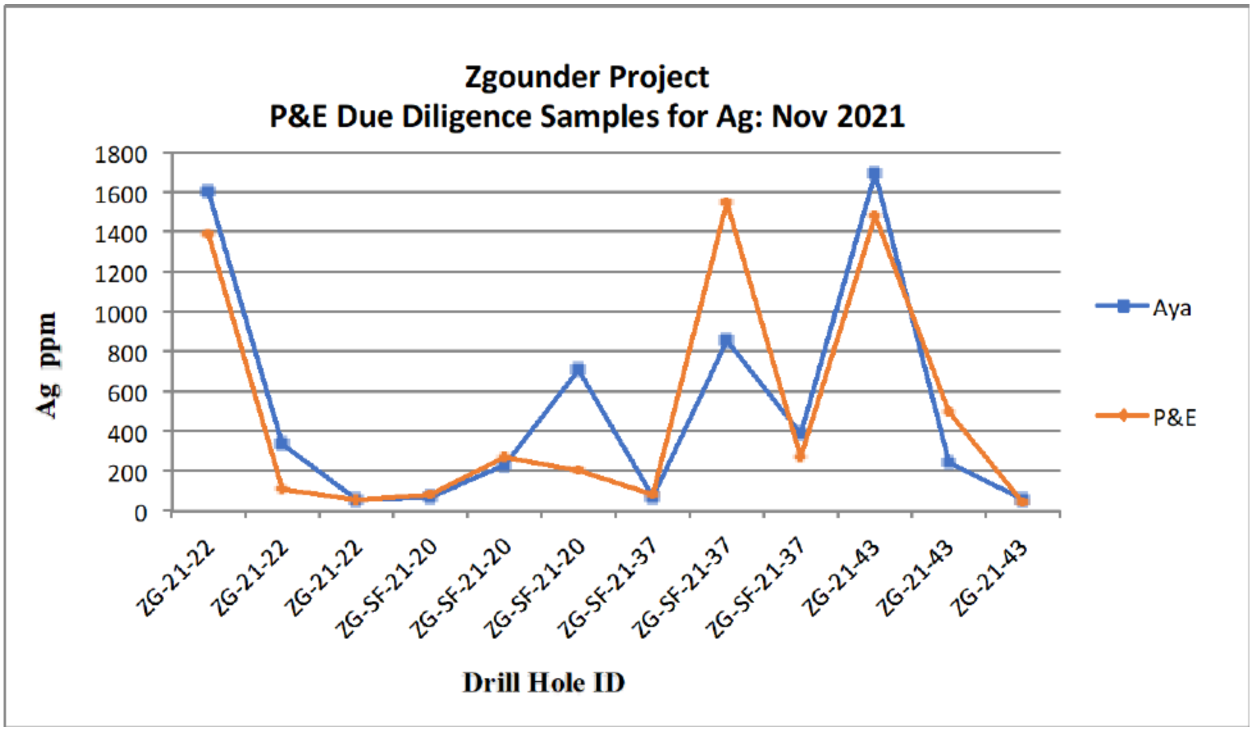
- visiting surface and underground drilling and outcrop sites;
- GPS location verifications;
- review of database, surveyor, logging, sampling and QC procedures;
- discussions on mineralisation and reconciliation and;
- verification sampling.

Mr. Yassa also conducted a site visit at Afrilab's laboratory facility in Marrakesh, Morocco on November 22, 2021 and carried out an audit on the Zgounder database on the same day.

Mr. Yassa collected 12 verification samples from four diamond drill holes. All samples were selected from holes drilled in 2021. Sampling was undertaken using the same protocol applied during the January 2021 site visit. Mr. Yassa delivered the samples to AfriLab for preparation and shipment of the pulps directly to Actlabs, of Ancaster, Ontario, for analysis. Samples at Actlabs were analyzed for silver by 4-acid digest with ICP-OES/ICP-MS finish. Samples returning results >100 ppm silver were also analyzed by fire assay with gravimetric finish. Density determinations were measured by pycnometer on all of the samples.

Actlabs is independent of Aya and maintains a Quality System that is accredited to international quality standards through ISO/IEC 17025:2017 and ISO 9001:2015. The accreditation program includes ongoing audits, which verify the QA system and all applicable registered test methods. Actlabs is also accredited by Health Canada. Results of the Zgounder site visit verification samples for silver are presented in Figure 12.3.

Figure 12.3 – Results of November 2021 Ag Verification Sampling by P&E



12.5 Conclusion

The authors of this Technical Report section consider that a good correlation exists between the Ag assay values in Aya's database and the independent verification samples collected by P&E and analyzed at AGAT and Actlabs, and that the data are of good quality and appropriate for use in the current Mineral Resource Estimate.

13 MINERAL PROCESSING AND METALLURGY TESTING

13.1 Introduction

This Section includes summaries of past and recent test work that were used to develop the process flowsheet and plant design for treating the ore based on the mineral resource estimation process from reports and technical notes listed in Section 27 of the Report. These test work programs will be referenced throughout this Section of the Report.

The interpretation and analysis of the test work results were carried out by DRA. This analysis was then used to determine the process design basis and flowsheet of the new Plant # 3.

The overall Project consists of three (3) processing facilities:

a. An existing Cyanidation Plant – Plant #1:

   The existing Plant # 1 operates at 180 t/d of ore feed and is operated independently of the flotation plant. The silver sludge recovered from this cyanidation plant will be transferred to the New Plant #3 for refining.

b. An existing Flotation Plant – Plant #2:

   The existing Plant # 2 treats 500 t/d of mineralised material, its silver concentrate slurry product will be pumped to the New Plant #3 for leaching and refinery. The flotation tailings from Plant #2 will be transferred to the New Plant #3 to be combined with its new flotation tailings in the Leaching-CIP circuit.

c. A new Processing Plant – Plant #3:

   The new mineral processing facility has been designed to produce silver ingots, and to process on average 2000 t/d of mineralised material.

13.2 Previous Testwork

13.2.1 Testwork prior to 2021

Aya has several options to process whole-rock material, a sulphide concentrate, and sulphide flotation tailings separately, or a combination of the three (3) as a part of the processing operations for the Zgounder Silver Project.

13.2.1.1 1979 - Bureau de Recherche et Participation Minière ("BRPM")

The PEA reported that a pilot plant test achieved cyanidation leach silver recovery of 93.8% on flotation tailings.

13.2.1.2 2003 - Bureau de Recherche Géologique et Minière ("BRGM")

Bench-scale flotation and subsequent cyanidation testwork resulted in 82.6% silver flotation recovery and 54.1% cyanidation recovery of the flotation concentrate.

The mineralized material was reported as relatively hard with a Bond ball mill work index of 14 to 17 kWh/t.

13.2.1.3 2015 - Zgounder's Laboratory

A test employing potassium amyl xanthate ("PAX") as the collector at a rate of 32 g/t, and methyl isobutyl carbinol ("MIBC") as the frothing agent at a rate of 25 g/t, yielded a silver recovery of 78.7%. Approximately 18.3% of the original feed mass was floated.

Another series of tests was conducted on fresh process plant feed ground to a F80 of 50 µm. The best result was achieved at 30.2% w/w solids with the use of 27 g/t of PAX, 28 g/t of MIBC and 30 g/t of sulphidrate at natural pH. A silver recovery of 82.2% was achieved while the mass recovery was 23.6%.

13.2.1.4 2016 - Yantai Xinhai Mining Research and Design Co., Ltd

A sample of 30 kg from the Zgounder Deposit was used to determinate the optimal process technologies for the recovery of silver with a view to providing design information for the 500 t/d process plant. The raw material bulk density was 2.73 t/m³.

A two-stage gravity concentration test was performed on material that was ground to 86.5% -200 mesh. A gravity silver concentrate was produced grading 17,957 g/t Ag with a silver recovery of 19.3% and a weight recovery of 0.34%.

Xinhai tested different flotation reagents (CuSO4, SIX, Butylamine Aeroflot, Na2S, Na2CO3, etc.) as well as grind size. A rougher flotation silver concentrate was produced assaying 983 g/t with a silver recovery of 88.5% and a mass recovery of 27.9%.

Open circuit flotation tests were conducted. The test procedure included two stages of scavenging and two stages of cleaning. A final flotation concentrate was produced with an Ag grade of 2,689 g/t at an Ag recovery of 80.9% and a mass recovery of 9.3%. The feed silver grade was 309 g/t Ag.

Open circuit flotation tests including two stages of scavenging and two stages of cleaning after gravity concentration. A gravity silver concentrate was produced grading 17,585 g/t Ag with a silver recovery of 18.7% and a weight recovery of 0.33%. A final flotation concentrate was produced of 2,424 g/t Ag with a silver recovery of 64.3% and a weight recovery of 8.2%. The feed silver grade was 310 g/t Ag.

Locked cycle circuit flotation tests were performed. These tests comprised of two stages of scavenging and two stages of cleaning after gravity concentration. A gravity silver concentrate was produced with a grade of 17,577 g/t Ag, a silver recovery of 19.1% and a mass recovery of 0.34%. The final flotation concentrate assayed at 3,962 g/t Ag with a silver recovery of 64.9% and a mass recovery of 5.1%. The feed silver grade was 313 g/t Ag.

The tailings settling test yielded an underflow density of 56.9% w/w solids.

13.2.1.5 2017 - Yantai Xinhai Mining Research and Design Co., Ltd

Xinhai performed cyanide leaching and flotation tests on the silver concentrates obtained during the 2016 metallurgical test campaign and from the old process plant tailings (1982 to 1990) with the key observations as follows:

- Cyanide leach tests of old process plant tailings grading 102.5 g/t Ag yielded a 24-hour leach recovery of 76.1%, using 1 kg/t NaCN at a slurry density of 33% w/w solids;
- Cyanide leach tests were conducted on high-grade mineralized ore grading 1,067 g/t Ag after grinding to 85% -200 mesh. These tests yielded a 48-hour leach recovery of 97.1%, using NaCN of 3.5 kg/t at 40% solids;
- Locked-cycle flotation tests were performed after gravity concentration and is summarised in Table 13.1.

Table 13.1– New Gravity-Flotation Locked-Cycle Test Results

Stream	Mass Recovery (%)	Ag (g/t)	Silver Recovery (%)
Gravity Concentrate	0.95	6,730	20.5
Flotation Concentrate	8.46	2,397	64.9
Combined Concentrate	9.41	2,834	85.4
Final Tailings	90.59	50.5	14.6
Feed	100.00	312	100.0

- Cyanide leach tests on low-grade flotation concentrate grading 1,063 g/t Ag at 95% -200 mesh yielded a 40-hour leach recovery of 86.3%, using NaCN of 2 kg/t at 33% w/w solids.

13.2.1.6 2019 - SGS Minerals

SGS performed additional testwork on flotation concentrate samples in 2019. Head analysis of the flotation concentrate sample showed that it contained 2,773 g/t Ag.

A cyanidation test was conducted on the as-received concentrate and a second test after grinding the concentrate to a P80 of 33 µm. The as-received concentrate had a P80 of 80 µm. The test revealed that finer grinding had no effect on silver recovery, which was excellent for both tests (≈94%). A test was conducted with activated carbon in the leach ("CIL"), but this also did not affect silver recovery appreciably (≈95%). This was the baseline test against which the Albion and POX procedures could be evaluated.

Silver recovery by CIL decreased from 95% in the baseline test to 74% after Albion processing and to 52% after POX, hot curing and lime boiling. Cyanide and lime consumptions were also higher after Albion and POX processing than in the baseline test.

13.2.1.7 2019 - Multi-disciplinary Digital Publishing Institute ("MDPI")

MDPI performed numerous flotation tests on low-grade tailings to increase the Ag grade prior to leaching. A composite tailings sample was formed which had a grade of 148 g/t Ag.

Flotation tests were done with the objective of optimizing different parameters; specifically, particle size distribution (regrinding), collector type and dosage, pH, D80, activator, and sulphidizing agents. The optimum result achieved was a concentrate grade of 1,745 g/t Ag, with a silver recovery grade of 84% and a mass recovery of approximately 40%.

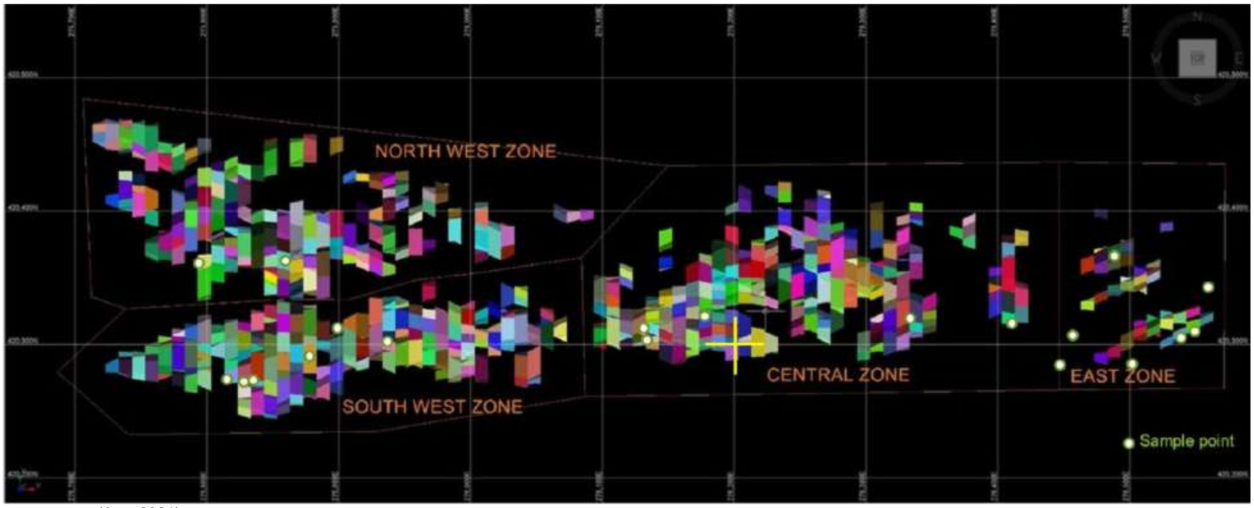
13.3 2021 Metallurgical Testwork

The information presented in this section is, for the most part, largely drawn and/or summarised from the Report entitled "An Investigation into the Metallurgical Response of Samples from the Zgounder Deposit", SGS, Project # 16525.05-DRAFT – Final Report by SGS Canada Inc., issued December 10, 2021.

13.3.1 Sample Selection

DRA selected samples from the Central, Southwest, Northwest, and East Zones for testwork. Mining in the East Zone will be delayed for more than three (3) years; therefore, this zone was excluded from the main composite sample. However, the East Zone was included in the variability samples. Figure 13.1 shows a plan view of the location of the sample points (white circles with a green rim) based on the provided coordinates.

Figure 13.1 – Preliminary Mining Zone Map



(Aya, 2021)

The main composite will be based on mining zone ratios, while respecting first the silver feed grade and second the mineralogy.

Aya collected the samples for the metallurgical testwork. The samples were split into nine (9) lots, namely Lots #1 to #5 and Lots A to D. The numbered lots are from the Northwest, Southwest and Central Zones, whereas the lettered lots are from the East Zone. To produce the main composite, the samples from each lot were split proportionately to represent the first three-years of mining activities. The variability samples were selected to represent distinct zones and lithologies.

Samples were prepared for metallurgical testwork in support of this Feasibility Study ("FS") which evaluates the addition of a third treatment plant to the existing facilities. The 2021 metallurgical testing was performed at an independent laboratory, namely SGS Canada Inc. at Lakefield, Ontario, Canada. The following metallurgical testing was recommended and accepted by Aya:

- Chemical characterisation;
- Physical characterisation;
- Gravity concentration tests;
- Flotation tests;
- Bottle roll cyanidation;
- Merrill-Crowe cementation tests;
- Hydrogen peroxide cyanide destruction;
- Carbon adsorption modelling
- Dewatering tests;

- Overall Balances;
- Variability Testing; and
- Mineralogy (Main Comp and VAR-5).

13.3.2 CHEMICAL CHARACTERISATION

13.3.2.1 Sample Preparation

From January to April 2021, SGS received three (3) pallets of drums for the test program. The samples from January 2021 arrived in small drums on five (5) skids and were prepared into a Main Composite (Main Comp) as well as five (5) variability samples (VAR-1 to VAR-5). In April 2021, SGS also received Zgounder Historical Tailings and Plant Concentrate samples for the test program.

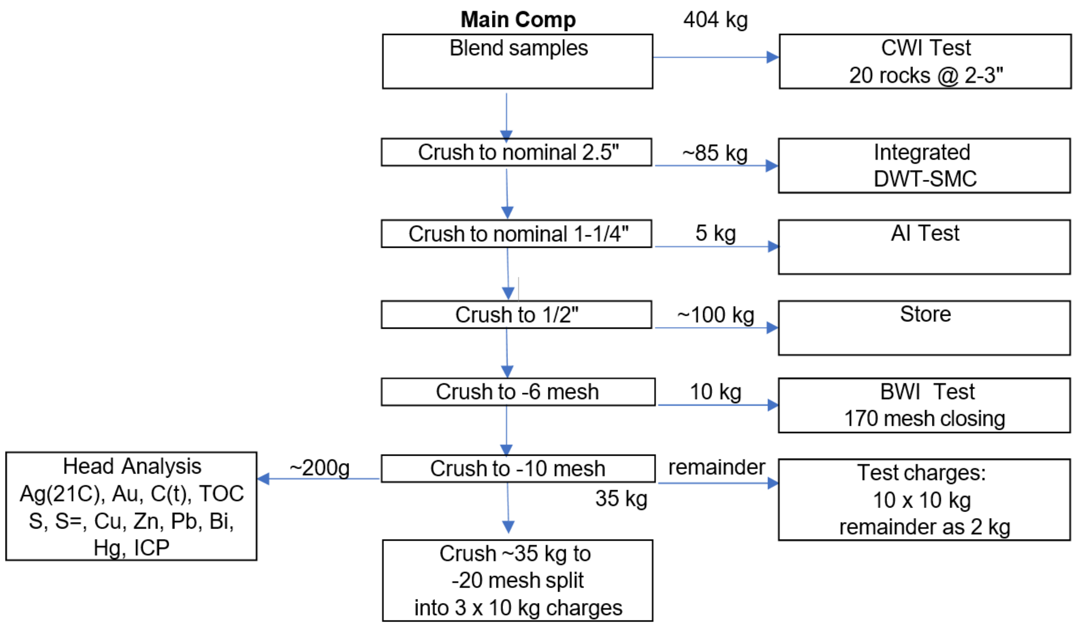
The Main Comp was prepared by blending selected samples. Table 13.2 summarises the composition of this sample by main lithology, zone, and the percentage contribution of each zone by mass%.

Table 13.2 – Main Composite Lithology

Lithology	% Lithology in Main Comp	Zone
Schist	72	Central
Dolerite	9	South West
Rhyolite	4	North West
Schist + Dolerite	15	East

Rocks were proportionally recovered from each component for the crushing work index test (CWI). The remainder of the sample was then blended and crushed to the required sizes for the various comminution tests, removing the required sample mass at each crush size. The remainder of the sample was then crushed to pass 10 mesh. A small (~200 g) representative sample was removed for chemical analysis, which included silver (Ag), gold (Au), total carbon (C(t), total organic carbon (TOC), total sulphur (S), sulphide sulphur (S2-), copper (Cu), zinc (Zn), lead (Pb), bismuth (Bi), mercury (Hg), and a multi-element ICP scan. Approximately 35 kg of the sample was split out and further crushed to pass 20 mesh. The remaining -10 mesh sample as well as the -20 mesh sample was then split into 10 kg test charges which was stored in a freezer prior to testing. The flowsheet outlining the Main Comp sample preparation is shown in Figure 13.2.

Figure 13.2 – Main Composite Sample Preparation Flowsheet



The deposit zones for each variability sample are provided in Table 13.3. Figure 13.3 (VAR-1) and Figure 13.4 (VAR-2 to -5) illustrate the sample preparation flowsheet.

Table 13.3 – Variability Sample Deposit Zone

Sample	Zone
VAR-1	Central
VAR-2	South West
VAR-3	North West
VAR-4	East 1
VAR-5	East 2

The samples were blended, crushed to the required sizes for the comminution and metallurgical tests, split into test charges and a small sample was removed for chemical analysis. The VAR-1 sample was low weight and the rejects and unused feed from the SMC and BWI test were used to prepare the metallurgical samples as depicted in Figure 13.3.

Figure 13.3 – VAR-1 Sample Preparation Flowsheet

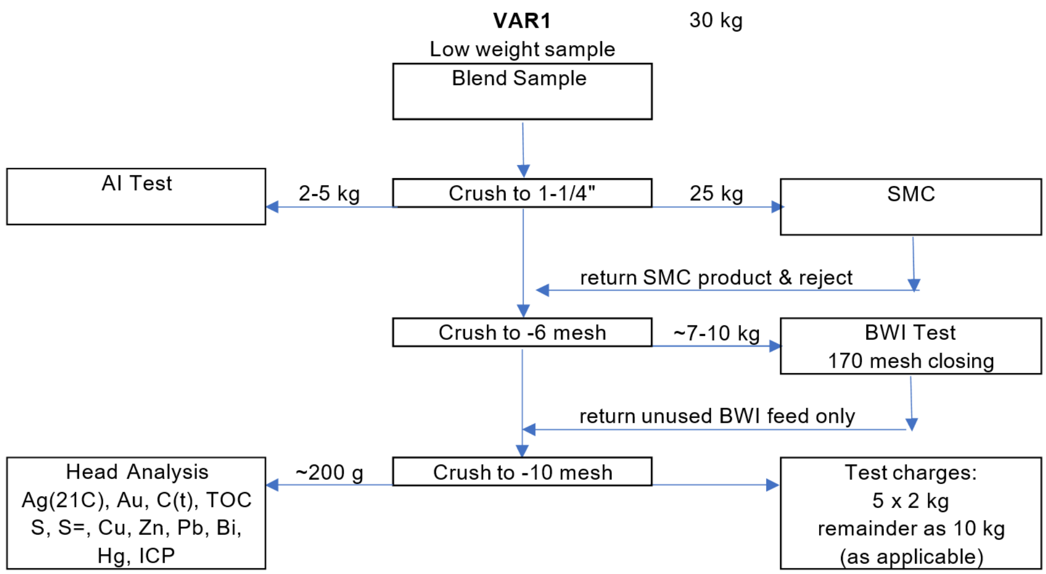
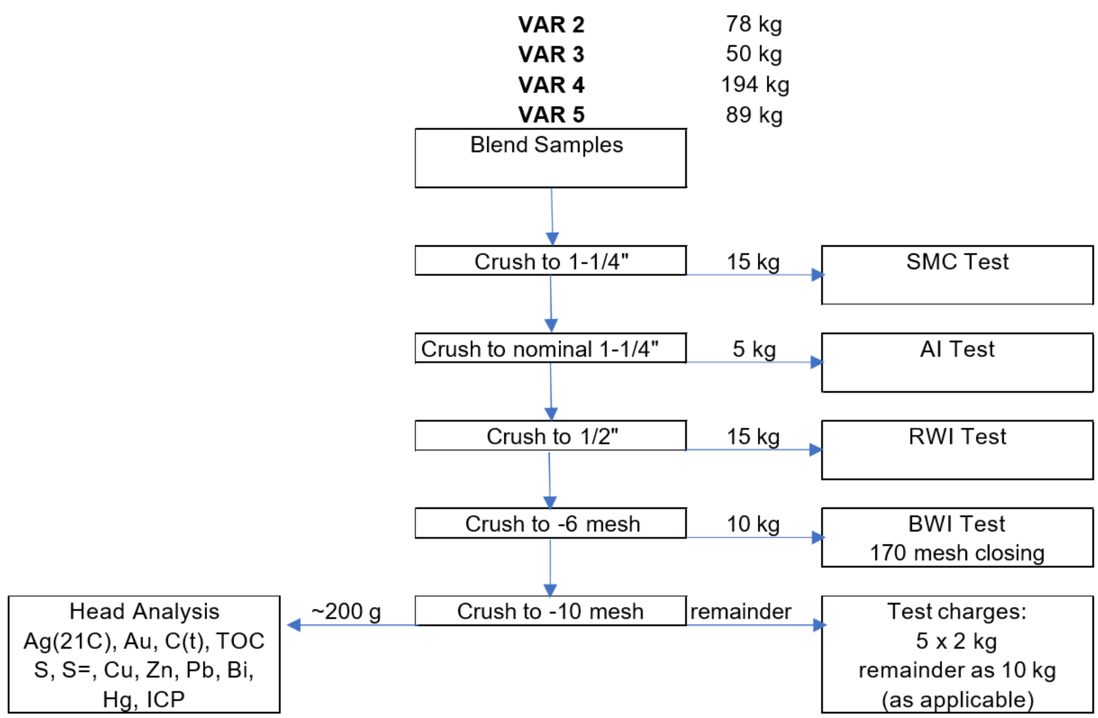


Figure 13.4 – VAR-2-5 Sample Preparation Flowsheet



Twenty-five (25) samples of the Zgounder historical tailings, with a total mass of 29.7 kg, were blended and split into test charges. A representative sample was analyzed for particle size distribution and then submitted for chemical analysis for the same elements as the Main Comp.

The Zgounder plant flotation concentrate was delivered as a wet filter cake, approximately 25 kg (dry equivalent), and was split into the required test charges for the program. A sample was submitted for silver analysis.

The chemical analyses of all the samples are provided in Table 13.4.

Table 13.4 – Head Assays

Element	Unit	Sample ID						Zgounder	
		Main Comp	VAR-1	VAR-2	VAR-3	VAR-4	VAR-5	Historical Tailings	Plant Conc.
Au	g/t	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	-
Ag	g/t	200	22	252	302	99	107	106	4,347
Bi	%	<0.002	<0.002	<0.002	0.014	<0.002	<0.002	<0.002	-
C(t)	%	0.05	0.04	0.05	0.05	0.03	0.06	0.13	-
S	%	0.83	0.45	0.77	0.86	0.58	0.83	0.59	-
S2-	%	0.80	0.46	0.74	0.84	0.56	0.79	0.51	-
Cu	%	0.028	<0.01	0.020	0.044	0.029	0.46	0.022	-
Zn	%	0.51	0.18	0.51	0.52	0.49	0.44	0.31	-
Pb	%	0.21	0.056	0.20	0.30	0.23	0.28	0.076	-
Hg	g/t	12.2	3.4	9.3	14.3	11.3	5.2	19.6	-
Al	g/t	97,900	88,900	94,800	118,000	104,000	104,000	96,600	-
As	g/t	93	58	79	< 30	541	259	304	-
Ba	g/t	620	857	589	777	617	665	652	-
Be	g/t	2.41	2.42	2.38	3.04	2.86	2.8	2.73	-
Ca	g/t	8,670	14,500	7,650	4,000	1,570	1,700	1,6200	-
Cd	g/t	< 30	< 30	< 30	21	< 30	< 30	14	-
Co	g/t	75	43	59	39	36	52	62	-
Cr	g/t	92	72	55	107	64	70	106	-
Fe	g/t	50,900	54,800	46,900	41,800	41,300	40,200	66,600	-
K	g/t	33,700	34,900	32,600	46,200	40,000	43,000	26,900	-
Li	g/t	61	50	65	49	72	63	65	-
Mg	g/t	14,300	16,600	12,700	10,900	11,200	12,100	18,400	-
Mn	g/t	953	1510	795	409	519	505	1950	-
Mo	g/t	< 10	< 10	< 10	< 20	16	47	14	-
Na	g/t	13,900	14,600	14,200	15,700	9,360	10,700	17,100	-
Ni	g/t	38	22	48	38	33	33	50	-

Element	Unit	Sample ID						Zgounder	
		Main Comp	VAR-1	VAR-2	VAR-3	VAR-4	VAR-5	Historical Tailings	Plant Conc.
P	g/t	442	562	386	484	319	367	814	-
Sb	g/t	< 30	< 30	< 30	< 10	< 30	< 30	27	-
Se	g/t	< 30	< 30	< 30	< 30	< 30	< 30	< 30	-
Sn	g/t	< 20	< 20	< 20	< 20	< 20	< 20	< 20	-
Sr	g/t	94.8	144	84.1	95	54.1	45.4	120	-
Ti	g/t	6,900	8,530	6,250	4,790	4,650	4,790	9,570	-
Tl	g/t	< 30	< 30	< 30	< 30	< 30	< 30	< 30	-
U	g/t	< 20	< 20	< 20	< 20	< 20	< 20	< 40	-
V	g/t	105	119	92	103	85	85	138	-
Y	g/t	25.8	26	25.6	24	21	18.9	27.9	-

13.3.2.2 Mineralogy

Based on the SGS Mineralogy Report (September 23, 2021) on the Main Comp sample, the dominant silver mineral host, identified from the limited number of surfaces inspected, was the Ag-Hg phase accounting for 49.5% of the overall silver, while native Ag is the second most abundant host at 37.5%. Acanthite and proustite are also present and represent 5.7% and 2.9% of the overall silver, respectively. The remainder is compositionally variable Ag-Cu sulfosalts or Ag-As-Cu sulfosalt where each can be ± Fe, Sb, Pb. Collectively, these other silver bearing minerals account for <5% of the overall silver distribution. Two grains accounted for 78% of the distribution - one grain of the Ag-Hg phase accounted for 49.4% and one native Ag grain represented 28.8% of the silver distribution. Therefore, the silver distribution is strongly based on a “nugget” effect of a few large Ag mineral grains. Additional polished section evaluations are recommended to decrease any statistical bias.

Overall, liberated silver-minerals (which combines all the silver minerals and considers them as one group) account for 65.2% of the total. For the non-liberated silver minerals, the most common occurrence was exposed grains (attached to) occurring as ternary particles with sulfides and silicates (29.4%) and to a lesser extent, binaries exposed with sulfides (2.0%), or as binaries exposed in silicates (2.3%) or binaries exposed in oxides (1.6%). Locked associations account for 1.0% with silicate, 0.7% with oxides, 0.2% with sulfide, 0.1% with sulfide and oxide ternaries and 0.3% as sulfide and silicate ternaries. By frequency, roughly 89% of the silver-minerals are very fine-grained at less than 6 μm in size.

Mineralogically, as well as the test work program completed for the Main Comp sample, the anticipated silver recoveries were expected to be in the high 80% to low 90% range.

To confirm the observations from mineralogical examinations, an aqua regia test was performed on the residue from Test CN18 to evaluate the amount of locked silver in acid digestible sulphur minerals and in indigestible silicates. Before the addition of aqua regia, the residue was re-pulped to 50% solids and then heated to 80°C before commencing the four-hour leach. The resultant filtrate and filter cake were separately assayed for silver contents. Table 13.5 indicates that 62% of Ag in the leach tailings were locked in sulphides while the remaining 38% was locked in silicates.

Table 13.5 – Aqua Regia Acid Leach Test Results

CN18 Residue	% Distribution	% Overall Distribution
Hot Aqua Regia Acid Leach Extraction of gold possibly associated with pyrite, arsenopyrite and/or other sulphides	62	4.6
Remaining Material Gold locked in silicates, or associated with fine sulphides locked in silicates	38	2.8

The Main Comp and one (1) variability sample (VAR-4) was submitted for mineralogy, which included QEMSCAN and scanning electron microscopy. The VAR-4 sample illustrated very low gravity recovery, poor flotation response, and problems with viscosity in the grinding mill. The higher abundance of sericite/muscovite (54.8%) in the VAR-4 sample might be one of the causes of the poor viscosity and flotation response, and the poor grinding when compared to the previously studied Main Comp sample.

Table 13.6 compares the modal abundances between the two (2) samples and highlights the differences between the two (2). Most notable is the higher abundance of feldspars in the Main Comp (13.6% difference) while the sericite/muscovite is higher in VAR-4 (29.6% higher).

Table 13.6 – Comparison of the Model Mineral Abundance of the Main Composite Sample and the VAR-4 Sample

Sample ID	Main Comp	VAR-4	Difference
Minerals	Mass (%)		
Pyrite	1.9	1.0	0.9
Sphalerite	1.0	0.6	0.4
Chalcopyrite	0.1	0.0	0.1
Galena	0.1	0.2	0.1
Arsenopyrite	0.1	0.1	0.0
Ag-Minerals	0.1	0.0	0.1
Other Sulphides	0.0	0.0	0.0

Sample ID	Main Comp	VAR-4	Difference
Minerals	Mass (%)		
Quartz	23.3	24.6	1.3
Feldspars	23.9	10.3	13.6
Sericite/Muscovite	25.2	54.8	29.6
Chlorite	11.0	5.6	5.4
Amphibole	5.7	0.1	5.6
Clays	2.6	1.1	1.5
Other Silicates	0.8	0.6	0.2
Fe-Oxides	0.5	0.2	0.3
Rutile	2.4	0.6	1.8
Other Oxides	0.8	0.0	0.8
Carbonates	0.2	0.0	0.2
Apatite	0.3	0.1	0.2
Other	0.1	0.1	0.0

In the VAR-4 study, acanthite hosts 74.9% of the silver, followed by galena ~9%, and various Ag-Cu-As sulfosalts, which account for ~13.7% of the overall silver. Only two (2) native silver particles were observed and accounted for 0.5% of the distribution.

From the Main Comp report, it was noted that two (2) silver bearing particles accounted for ~79% of the total silver-mineral distribution, indicating a micro nuggeting effect. Despite the more uniform grain size of the silver minerals; it was noted that one acanthite particle accounted for 28% of the silver mineral distribution.

The VAR-4 samples had very low gravity recovery and this could be attributed to the fine grain size, but also poor liberation.

13.3.3 Physical Characterisation

The Main Comp and five (5) variability samples were submitted for comminution testing. This included DWT and SMC tests, the Bond low-energy impact (CWI) test, the Bond rod mill (RWI) test, the Bond ball mill (BWI) grindability tests and the Bond abrasion (AI) test.

The comminution parameters summary is presented in Table 13.7.

Table 13.7 – Comminution Tests Summary

Sample Name	Relative Density	JK Parameters				Work Indices (kWh/t)			AI (g)
		A x b1	A x b2	ta	SCSE	CWI	RWI	BWI	
Main Comp	2.81	22.4	21.0	0.22	13.6	12.6	23.8	23.9	0.171
VAR-1	2.81	-	24.4	0.22	13.0	-	-	24.9	0.217
VAR-2	2.78	-	23.4	0.22	13.2	-	23.8	25.0	0.219
VAR-3	2.79	-	24.3	0.23	12.9	-	24.0	24.4	0.093
VAR-4	2.64	-	24.3	0.24	12.4	-	19.9	19.3	0.009
VAR-5	2.80	-	23.2	0.21	13.3	-	23.4	22.5	0.043
Average	2.77	22.4	23.4	0.22	13.1	12.6	23.0	23.3	0.125
Std. Dev.	0.07	-	1.3	0.01	0.4	-	1.7	2.2	0.090
Rel. Std. Dev.	2	-	6	5	3	-	7	9	72
Min.	2.64	-	24.4	0.24	12.4	-	19.9	19.3	0.009
Max.	2.81	-	21.0	0.21	13.6	-	24.0	25.0	0.219

1 A x b from DWT 2 A x b from SMC

The DWT, SMC, RWI, and BWI tests classified all samples tested as very hard, whereas the CWI classified the Main Comp as moderately hard. The samples were also generally mildly abrasive. VAR-4 was generally the softest of all the samples.

13.3.4 Gravity Concentration Tests

13.3.4.1 Knelson/Mozley Testing

Silver recovery through gravity concentration was evaluated by standard Knelson/Mozley testing. The testwork procedure started with rod mill grinding of the samples (2 to 12 kg) to a P80 ranging from 116 µm to 140 µm, with most tests conducted at 125 µm. The ground sample was then passed through a Knelson MD-3 concentrator. The resultant Knelson concentrate was further upgraded on a Mozley table. Roughly 0.05 to 0.1% of the original mass was collected as Mozley concentrate which was assayed to extinction along with two sub-samples of the combined tailings. For test G1 the secondary Mozley table step was omitted and the primary Knelson concentrate was submitted for intensive leaching testing. The assay and mass balance results are summarised in Table 13.8.

Silver recovery to the Knelson concentrate in Test G1 was 27.7%. Silver recoveries to the Mozley concentrates ranged from 7.4 to 21.6%. The weighted average gravity silver recovery in the Knelson/Mozley tests (Tests G2-G9) was 15.3%.

Table 13.8 – Gravity Separation Test Results Summary

Gravity Test	P_{80}	Wt	Gravity Concentrate			Ag Head Grade
#	µm	kg	Mass %	Assay Ag, g/t	% Rec'y Ag	g/t
G1*	140	10	1.40	4,023	27.7	203
G2	125	12	0.11	36,624	18.2	229
G3	125	2	0.055	64,676	15.9	222
G4	125	2	0.074	40,869	13.2	229
G5	125	12	0.074	46,985	13.6	254
G6	125	1.8	0.049	30,103	7.4	200
G7	125	2	0.079	32,727	10.5	246
G8	125	8	0.048	79,045	17.0	223
G9	116	6	0.123	33,155	21.6	189

*Knelson only

A single Knelson only test was completed to produce a gravity concentrate for intensive cyanide leaching, which extracted 97.2% of the silver from the concentrate. The gravity concentrate was leached under the following conditions:

Pulp Density: 10% (w/w);

Dissolved Oxygen: > 20 ppm with hydrogen peroxide;

Kinetic solution samples: 2, 4, 7, 24 h;

Retention time: 48 h;

[NaCN]: 20 g/L maintained with NaCN.

13.3.4.2 E-GRG Testing

The main objective of the GRG test is to confirm and determine the suitability of a gravity recovery circuit. The GRG tests are typically conducted under optimum conditions, which result in production of the maximum amount of GRG. Actual plant recoveries (and laboratory gravity separation results) would be lower.

The E-GRG protocol requires that the entire concentrate and a representative subsample of the tailings from each stage be subjected for size fraction analysis for silver to evaluate recovery as a function of particle size. Concentrates are always assayed to extinction. The test procedure consisted of treating a 20 kg sample in a laboratory Knelson MD-3 concentrator in three stages. In the first stage, the sample was crushed to pass 20 mesh and then passed through the Knelson concentrator. The Knelson concentrate was submitted for size fraction analysis followed by silver assaying to extinction per size fraction. A representative sample of the tailings was submitted for size fraction analysis. The first pass Knelson tailings were decanted and ground to ~216 µm. The mill discharge was passed through the Knelson concentrator a second time collecting a concentrate and tailings, which were sampled and assayed as in the first pass. The second pass tailings were decanted and ground to ~96 µm and passed through the Knelson a third time. The results are shown in Table 13.9.

Table 13.9 – E-GRG Test Results Summary

Stage	Knelson Tailing Particle Size (µm)	Concentrate, Cumulative		
		% Mass Rec'y	Assay Ag, g/t	% Ag Rec'y
1	658	0.4	4654	8.4
2	216	0.8	5897	21.9
3	96	1.2	6221	33.8

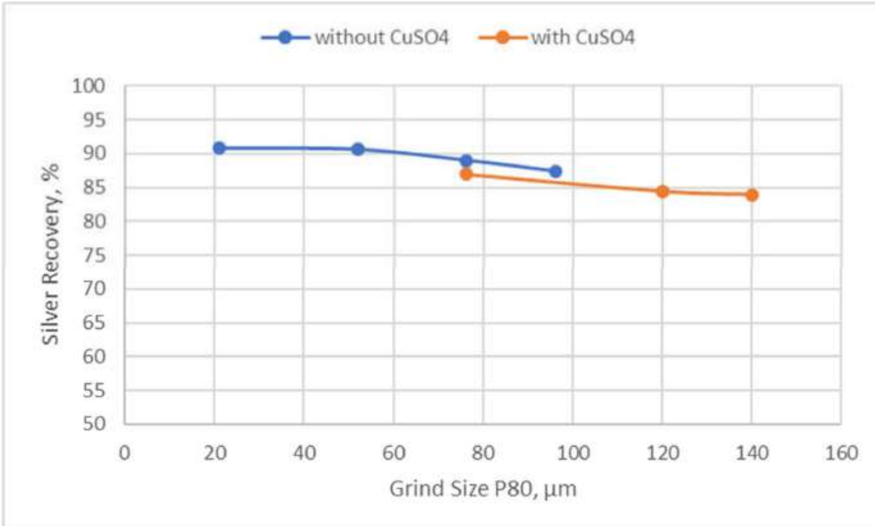
The E-GRG number for the Main Comp was 33.8%. This number is sufficiently high to justify the inclusion of a gravity step in the flowsheet.

13.3.5 FLOTATION TESTS

13.3.5.1 Whole Ore Rougher Kinetic

Two (2) series of whole ore rougher kinetic flotation tests were conducted on the Main Comp sample to investigate the effects of grind size and copper sulphate addition on silver recovery. Figure 13.5 presents the resultant silver recoveries. Both sets of tests show a moderate increase in silver recovery with a decrease in the fineness of the grind. However, grinding finer than 52 microns did not improve recovery any further. There was also only a 5% improvement in recovery when decreasing from as coarse as 120 microns to the 52-micron size. Due to this marginal benefit, a relatively coarse grind size of 80% passing 100 microns was adopted into the design criteria.

Figure 13.5 – Whole Ore Rougher Flotation – Effect of Grind Size with and without CuSO4



The addition of copper sulphate did not appear to appreciably affect silver recovery. Consequently, all further testing was completed without the addition of copper sulphate. Based on these results, the grind sizes, chosen for subsequent testing, were 52 and 120 µm as these represent the minimum and maximum grinds before the curves flattens out. The reagent suite was also limited to PAX, AERO 241 and MIBC.

13.3.5.2 *Whole Ore Cleaner Flotation*

Whole ore cleaner tests were performed on the Main Comp samples to examine the effects of grind size, regrinding of the concentrate before cleaning, and reagent addition on flotation performance. The tests comprised of a 20-minute rougher stage followed by upgrading of the concentrate in three (3) cleaning stages accompanied by a first cleaner scavenger step. Reducing the grind size from 76 µm to 52 µm resulted in an increase in final concentrate silver grade and a ~2% increase in Ag recovery. Including a regrind to 80% passing 20 µm before cleaning improved the grade vs recovery curves appreciably as shown in Figure 13.6.

Reducing the PAX and Aero241 additions did not have a detrimental effect on overall silver recovery. Test conditions are summarized in Table 13.10 and the results are summarized in Table 13.11. Note that the recoveries shown in Figure 13.6 are the cleaner stage recoveries and not overall recoveries i.e. it shows how much silver was recovered from the rougher concentrate as opposed to from the original head sample.

Figure 13.6 – Whole Ore Cleaner Flotation – Grade-Recovery Curve

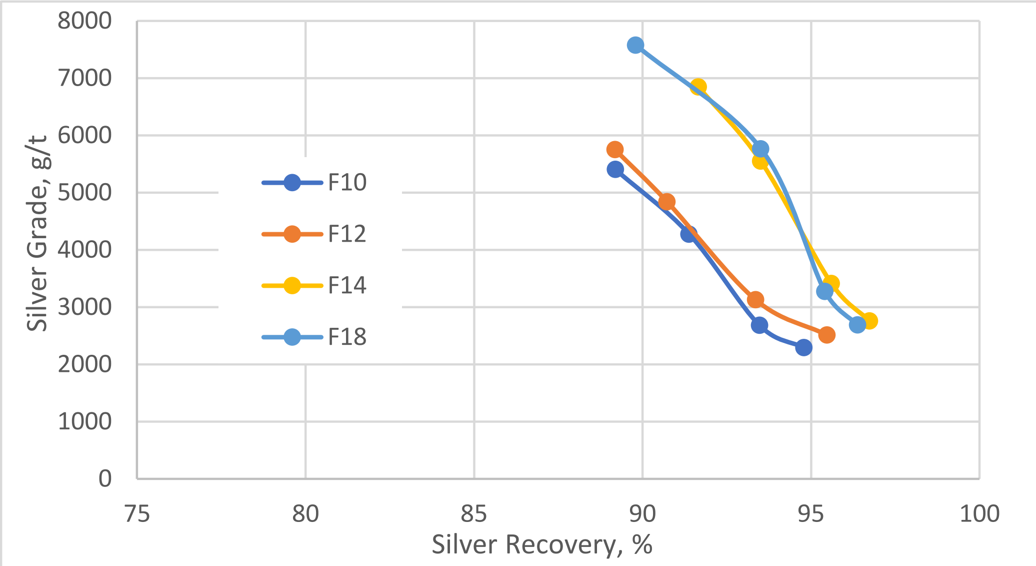


Table 13.10 – Whole Ore Cleaner Flotation Test Conditions

Test	P_{80}, µm Primary	P_{80}, µm Regrind	Reagents, g/t PAX	Aero241	MIBC
F10	76	---	250	165	30
F12	52	---	250	165	30
F14	52	21	250	165	30
F18	52	23	170	125	30

Table 13.11 – Whole Ore Cleaner Flotation Test Results Summary

Tests	Product	Wt %	Assay, %, g/t		Distribution, %	
			Ag	S^{2-}	Ag	S^{2-}
F10	3rd Cl Conc	3.1	5408	22.0	78.1	85.0
	Ro Conc	25.8	726	2.90	87.5	93.5
	Rougher Tail	74.2	36.0	0.07	12.5	6.5
	Head (calc.)	100.0	214	0.80	100.0	100.0
F12	3rd Cl Conc	3.1	5756	20.9	79.7	83.8
	Ro Conc	25.6	779	2.81	89.3	93.2
	Rougher Tail	74.4	32.0	0.07	10.7	6.8
	Head (calc.)	100.0	223	0.77	100.0	100.0

Tests	Product	Wt %	Assay, %, g/t		Distribution, %	
			Ag	S^{2-}	Ag	S^{2-}
F14	3rd Cl Conc	2.6	6853	25.0	81.6	84.4
	Ro Conc	24.2	812	3.00	89.0	93.2
	Rougher Tail	75.8	32	0.07	11.0	6.8
	Head (calc.)	100.0	221	0.78	100.0	100.0
F18	3rd Cl Conc	2.3	7582	28.4	80.1	80.5
	Ro Conc	23.4	834	3.19	89.2	91.5
	Rougher Tail	76.6	31	0.09	10.8	8.5
	Head (calc.)	100.0	219	0.82	100.0	100.0

13.3.5.3 Locked Cycle Flotation Testing

Three (3) locked cycle tests (LCT) were completed using the master composite as is (whole ore) and the gravity tailings. The first cleaner tailings and rougher tailings combined forms the final tailings of this procedure. The first cleaner scavenger concentrate was circulated back to the regrind stage and the second and third cleaner tailings, for Tests LCT 1 (whole ore) and LCT 2 (gravity tailings), were recycled to the previous cleaner in closed circuit (see Figure 13.7). The third test (LCT 3, gravity tailings) included only one cleaning stage (see Figure 13.8), and the cleaner tailings were taken as final tailings and the first cleaner scavenger concentrate was circulated back to the regrind.

The locked cycle test on LCT1 achieved recoveries of 84% Ag and 87% S2- to the third cleaner concentrate which assayed 6495 g/t Ag and 24.3% S2- at a 2.8% mass pull. The comparable locked cycle test on the gravity tailings resulted in an overall combined third cleaner concentrate and gravity concentrate assaying at 13,001 g/t Ag with 85% silver recovered in 2.9% of the mass. Test LCT 3 yielded 88% Ag recovery in the combined first cleaner and gravity concentrate, with a grade of 6,889 g/t Ag, in 6.3% mass pull. The results are summarized in Table 13.12.

The result shows that inclusion of the gravity step allows for a small improvement in both the grade of the final product as well as a small improvement in overall silver recovery. The implication is that a flowsheet which includes a gravity step would yield more revenue. It would also allow for a smaller capital expenditure with respect to the size of the CCD thickeners required to treat the final concentrate emanating from the flotation section. These benefits are attractive when weighed against the relatively small Capex and Opex associated with the gravity concentration step. Including a gravity step in the flowsheet also minimises the risk of losing coarse metal particles that may not float nor leach well. Given these benefits, inclusion of a gravity step is recommended.

Figure 13.7 – Locked Cycle Tests (LCTs) 1 and 2 Flowsheet

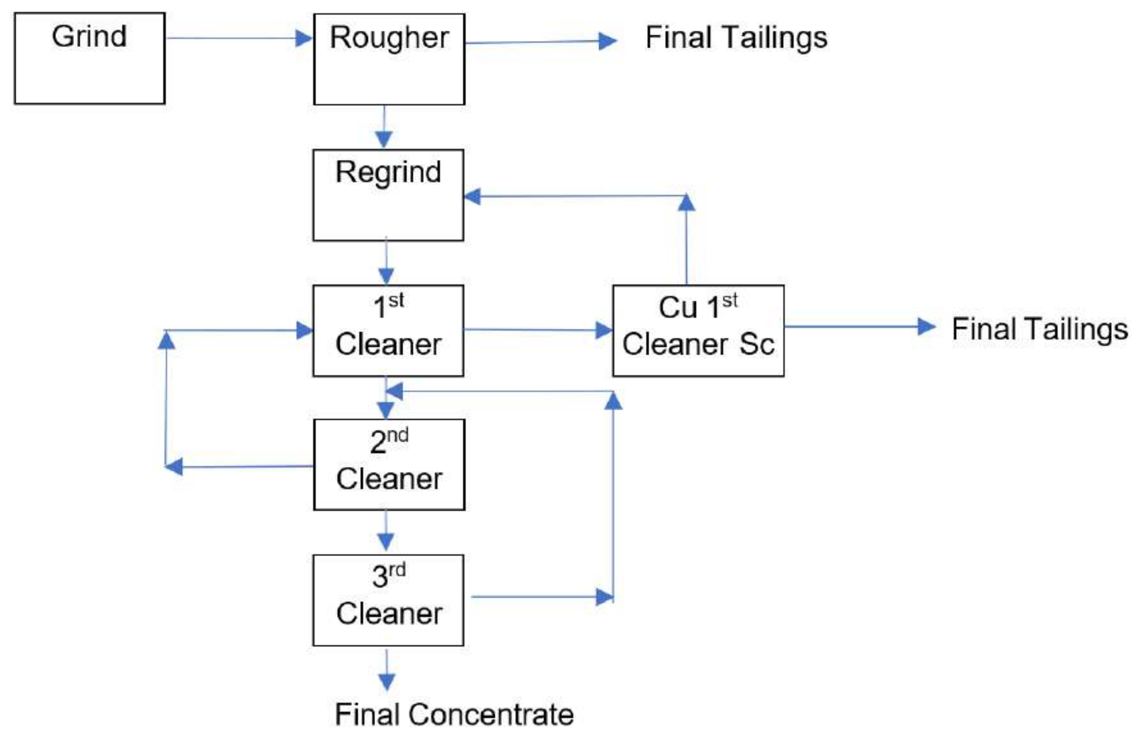


Figure 13.8 – Locked Cycle Test (LCT) 3 Flowsheet

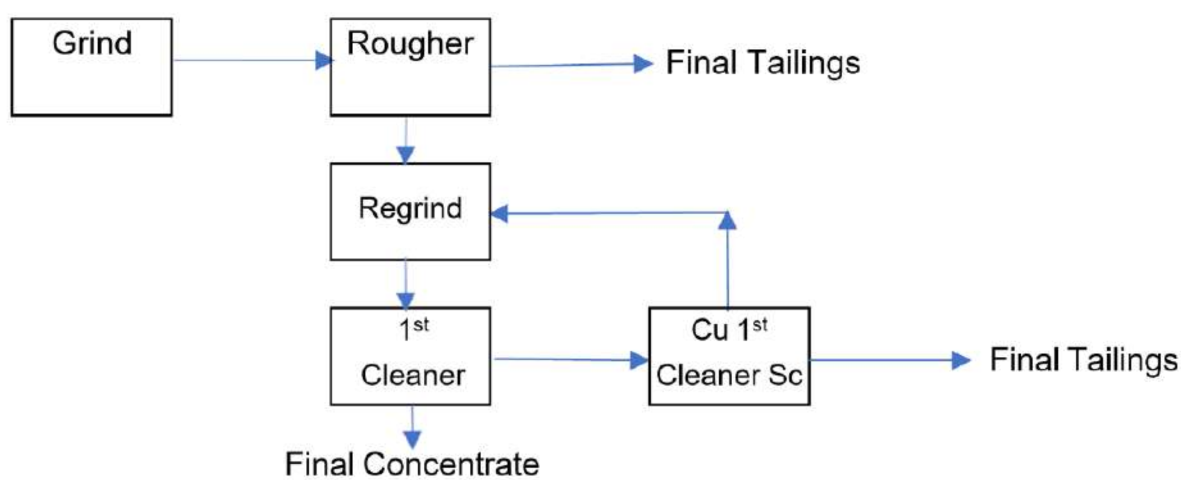


Table 13.12 – Locked Cycle Test Results Summary

Sample	Test No.	Product	Weight %	Assays, g/t, % Ag	Assays, g/t, % S^{2-}	Distribution, % Ag	Distribution, % S^{2-}
Main Comp Whole Ore	LCT 1	3rd Cleaner Conc	2.8	6495	24.3	84.2	86.9
		Combined Tailings	97.2	34.9	0.11	15.8	13.1
		Head	100.0	215	0.78	100.0	100.0
Main Comp Gravity Tailings (with three cleaners)	LCT 2	Combined Concentrate*	2.9	13001		85.0	
		Combined Tailings	97.1	34.1	0.083	15.0	11.6
		Head	100.0	222	0.70	100.0	100.0
Main Comp Gravity Tailings (with one cleaner)	LCT 3	Combined Concentrate*	6.3	6889		87.7	
		Combined Tailings	93.7	32.7	0.065	12.3	7.3
		Head	100.0	249	0.84	100.0	100.0

*Combined concentrate = Mozley + final cleaner concentrate

13.3.6 Bottle Roll Cyanidation

Standard bottle roll cyanidation tests were completed using the master composite on the whole ore, gravity tailings, and flotation products. Conditions were as follows:

Whole Ore, Gravity Tailings and Flotation Tailings

Pulp density: 50% w/w solids

pH: 10.5 – 11; maintained with lime

Dissolved oxygen: 8-9 ppm with air sparging

Kinetic solution samples: 8, 24, 48, and 72 h

Retention time: 96 h

[NaCN] whole ore tests: 1, 2, 4, 6, and 8 g/L maintained with NaCN

[NaCN] gravity tailings: 2, 4, 6, and 8 g/L maintained with NaCN

[NaCN] flotation tailings: 1, 2, and 4 g/L maintained with NaCN

Flotation Concentrate

Pulp density: 10% and 50% w/w solids

pH: 10.5-11; maintained with lime

Dissolved oxygen: 20 ppm with air/oxygen mix

Kinetic solution samples: 1, 2, 4, 6, 8, 12, 16, 24 h

Retention time: 48 h

[NaCN]: 1, 2, 4, and 12 g/L maintained with NaCN

13.3.6.1 *Whole Ore Cyanidation*

While not included in the flowsheet, whole ore cyanidations were performed to serve as a baseline for comparison against the eventual flowsheet. As such, the detailed results are not included in this report. The silver extraction of 88.5% appeared to be optimal at a grind of 80% passing 71 µm in the indicated three (3) tests conducted at different grinds. Silver extraction and reagent consumptions increased with NaCN concentration and resulted in >90% extraction at the highest cyanide concentration. Adding lead nitrate did not improve the results. Increasing the dissolved oxygen level to exceed 30 ppm improved both the leaching kinetics as well as final extraction.

The leach kinetic curves are shown in Figure 13.9 (various free NaCN concentrations at 71 µm) and Figure 13.10 (various grind sizes at 2 g/l NaCN). It is evident that the initial kinetics are fast up to about the 20-hour mark. This is followed by a period of slow but steady leaching up to and possibly beyond the 96-hr termination point. This is possibly due to the presence of coarse silver that simply takes a long time to leach, and it validates the inclusion of a gravity step in the flowsheet as the coarse silver will report to the gravity concentrate where it can be subjected to an intensive cyanidation, which is designed to specifically target the slow leaching components.

Figure 13.9 – Main Comp Whole Ore Cyanidation Leach Kinetics at 71 µm

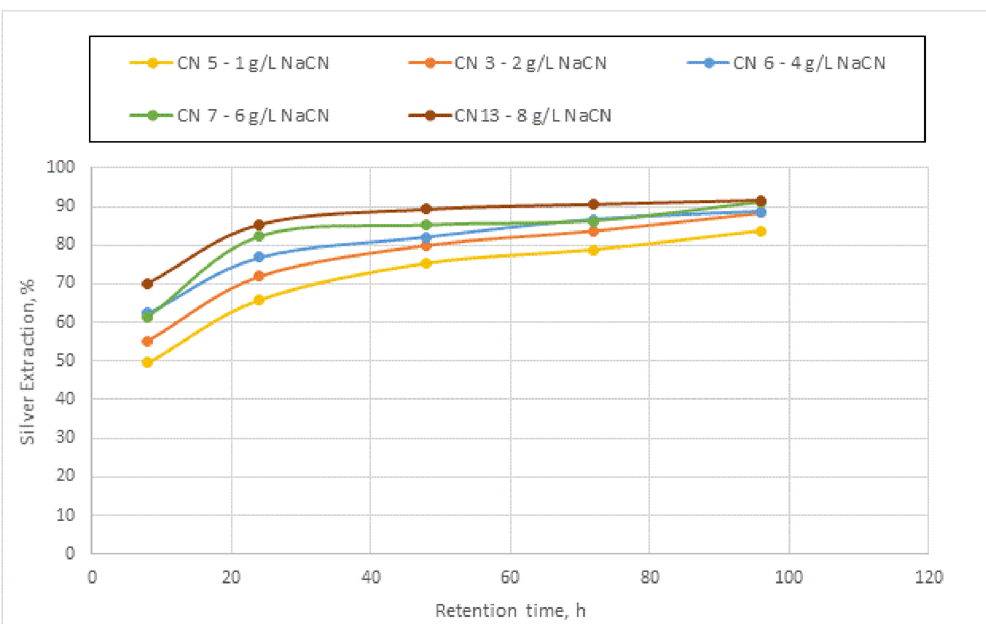
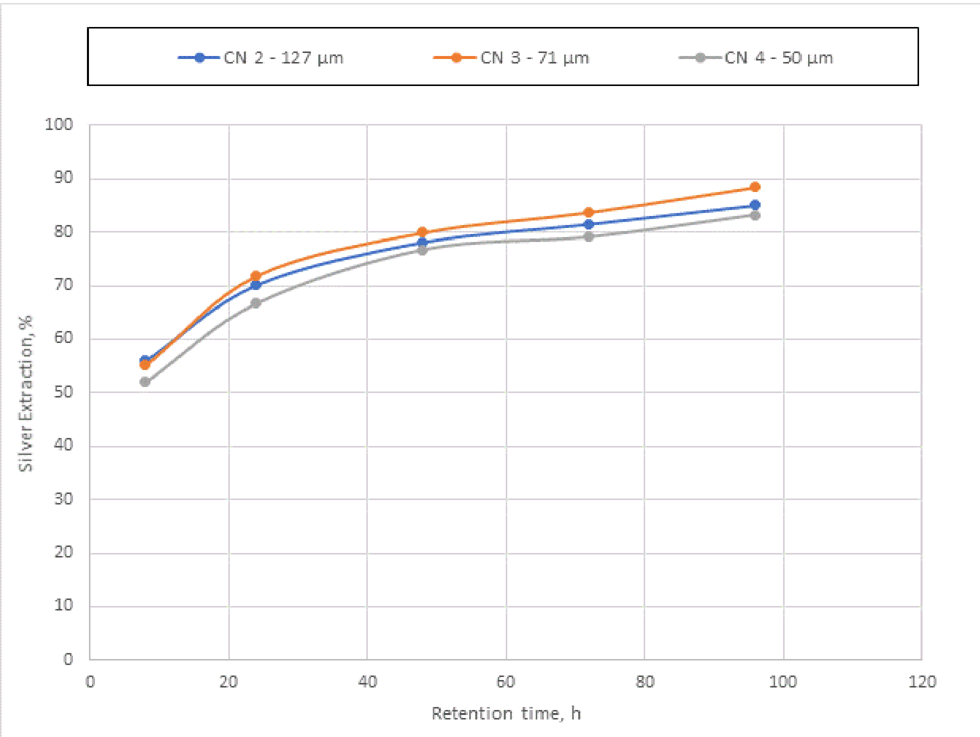


Figure 13.10 – Main Comp Whole Ore Cyanidation Leach Kinetics at 2 g/l NaCN



13.3.6.2 Gravity Tailings

Four (4) bottle roll cyanidation tests were performed to evaluate a flowsheet where gravity concentration is added ahead of the cyanidation. In this flowsheet, the gravity concentrate would be subjected to an intensive cyanidation step while the gravity tailings would be leached conventionally. This set of tests evaluated how well the gravity tailings would respond to cyanidation. The effect of different NaCN concentrations was tested on the Main Comp Test G1 gravity tailings. Test conditions and reagent addition and consumption values are shown in Table 13.13 while the silver distribution results are shown in Table 13.14. All four (4) tests were conducted at a grind P80 of 68 microns, a pulp density of 50% w/w solids and a total duration of 96 hrs. The gravity silver recovery was the same for all four tests at 27.7%.

The final silver extractions were similar and are therefore, independent of cyanide strength. However, leach kinetics improved as the NaCN concentrate increased, as did the consumption of cyanide.

Table 13.13 – Main Comp Gravity Tailings Cyanidation Conditions and Reagents Summary

Sample	CN	[NaCN],	Dissolved Oxygen	Reagent Add'n, kg/t of CN Feed		Reagent Cons, kg/t of CN Feed	
	Test No.	g/l	ppm	NaCN	CaO	NaCN	CaO
Main Comp –G1 Gravity Tailings	CN15	2	9.0	3.54	1.47	1.97	1.44
	CN16	4	9.2	6.56	1.27	3.16	1.17
	CN17	6	9.1	9.37	1.02	4.13	0.84
	CN18	8	9.1	12.1	0.94	5.03	0.67

Table 13.14 – Main Comp Gravity Tailings Cyanidation Test Results Summary

CN Test No.	Ag Stage Extraction, %					Overall Ag Recovery	Ag Residue	Final PLS Assays, mg/l	
	8 h	24 h	48 h	72 h	96 h	%	g/t	Cu	Zn
CN15	61.1	74.1	83.9	87.3	88.3	91.5	19	83.8	111
CN16	37.3	42.5	88.8	90.4	90.6	93.2	15	62.2	144
CN17	76.5	85.7	87.1	88.6	89.8	92.6	17	81.0	159
CN18	79.1	87.7	88.1	87.0	89.7	92.6	17	87.0	185

13.3.6.3 Flotation Tailings

Bottle roll tests were conducted using flotation tailings samples to evaluate the effects of grind size (52, 66, 95, and 112 µm), NaCN concentration, temperature and the presence of carbon on leach performance.

Silver extractions were similar at all the NaCN concentrations tested. The leaching kinetics and NaCN consumptions increased with increasing cyanide concentration. Higher temperature (50°C) was not beneficial for silver extraction, but the NaCN consumption doubled. Three (3) grind sizes were tested, and final extractions were similar for all three tests. The addition of carbon did not affect the leach outcomes appreciably. This indicates that preg-robbing is not a concern for this application. The timed samples indicate that leaching of silver from the tailings was substantially completed within 36 hours and this was adopted as the residence time required for this section of the plant.

The addition of 3 kg/t NaCN was adopted as the silver extraction did not improve considerably at higher addition rates. At this addition rate, the cyanide consumption is expected to be 1.44 kg/t. A lime addition rate of 1.4 kg/t was also adopted in the PDC based on these test results.

Subsequent to this, a set of tests was conducted to evaluate whether pre-oxygenation and a high dissolved oxygen (DO) concentration for the initial 12 h of the 36 h leach would reduce cyanide consumption. The tests were performed with 2 h of pre-oxygenation at 20 ppm DO and on the rougher tailings from test F47. Two (2) pairs of duplicate tests were performed at controlled cyanide concentrations of 2 g/l (Tests CN71 and CN72) and 1 g/l (Tests CN73 and CN74), respectively.

The results show a significant decrease in NaCN consumption to 0.96 kg/t and 0.56 kg/t for the tests performed at 2 g/l and 1 g/l, respectively. Additionally, the silver extraction at 2 g/l was similar to what had been recorded before at 65.2%, but it decreased to 60.7% at the lower cyanide level of 1 g/l.

Given the decrease in silver extraction, the 2 g/l scenario is recommended, i.e., a NaCN addition of around 2.5 kg/t maintained at around 2 g/l for the duration of the leach. It is also recommended that pre-oxygenation should be implemented to curb cyanide consumption.

13.3.6.4 Flotation Concentrate

Bottle roll cyanidation tests were conducted on various rougher and cleaner concentrate samples to determine its response to leaching. These samples were all from tests conducted using the main composite sample as well as the variability samples, Zgounder Historical Tailings and Plant Concentrate. The test parameters, varied in this set of tests, were pulp density, retention time, grind size and NaCN concentration. Oxygen was sparged throughout and alkaline conditions were maintained by manually adding lime as required. The conditions and results are summarised in Tables 13.15 and 13.16.

Flotation rougher concentrate cyanidations showed that leaching at a higher pulp density (50%) and high NaCN (12 g/l) yielded comparable results to what was achieved at the lower pulp density (10%) and lower NaCN levels (4 g/l) extraction. Tests comparing leaching at 4 and 12 g/l NaCN resulted in a modest 5% increase in Ag extraction at the higher NaCN addition, but with twice the NaCN consumption. Residual cyanide would also need to be destroyed before any reclaimed water can be re-used in a flotation circuit which adds to the incentive to minimize cyanide usage.

Bottle roll leach tests were also completed on the high-grade first cleaner concentrates from an open circuit batch test (F29) and the combined first cleaner concentrates from LCT3. The effect of [NaCN] was tested (1, 2, and 4 g/l NaCN) and there was a steady increase in Ag extraction from around 81% to 92% at the higher NaCN addition compared to the lowest addition. The result shows that the concentrates samples will leach better at elevated cyanide concentrations.

From a silver recovery perspective, the best results were achieved with the first two (2) test (i.e. where the rougher concentrate was leached and only the rougher tailings were discarded). The tests where cleaner concentrates were leached yielded worse silver recoveries implying that a more complicated flotation flowsheet (i.e. including cleaner steps) would only make sense if the tailings are also leached. The flowsheet for this FS included these steps as it allows for smaller CCD thickeners but also includes leaching of the tailings in order to still maintain high silver recoveries.

Table 13.15 – Flotation Concentrate Cyanidation Test Conditions Summary

Sample	Grav Test No.	CN Test No.	Grind Size P_{80}, µm	Pulp Density % (w/w)	Retention Time, h	[NaCN], g/L	Dissolved Oxygen, ppm	Reagent Add'n, kg/t of CN Feed		Reagent Cons, kg/t of CN Feed	
								NaCN	CaO	NaCN	CaO
F25 Rougher Concentrate	G3	CN19	29	10	48	4	21.3	50.9	1.71	7.35	-
		CN20		50	96	12	11.1	27.5	0.10	14.7	-
F26 First Cleaner Concentrate	G4	CN21	36	10	48	4	18.2	49.8	1.33	15.0	0.00
F29 First Cleaner Concentrate	G8	CN30	27	10	48	1	21.2	20.3	1.76	11.3	1.64
		CN31				2	25.5	28.2	0.99	13.2	0.78
		CN32				4	31.0	50.5	1.10	18.3	0.55
F30 Rougher Concentrate	G9	CN34	53	10	48	4	25.9	48.1	1.84	12.1	0.51
F31 Rougher Concentrate		CN35	52			4	24.4	48.6	1.38	16.5	0.29
F31 Rougher Concentrate		CN36	52			12	22.5	160	0.74	36.2	0.74
F32 Rougher Concentrate		CN37	35			4	24.4	49.9	1.98	12.6	0.93
LCT 3 First Cleaner Concentrate	G5	CN22	36	10	48	1	26.1	14.5	2.36	6.46	2.36
		CN23				2	25.1	24.6	2.28	7.80	2.28
		CN24				4	23.7	48.5	2.27	15.8	2.27

Table 13.16 – Main Comp Flotation Concentrate Cyanidation Test Results Summary

Sample	Ag Extraction, %				Residue	Head, Ag, g/t (calc.)	Ag Recovery		
	1 h	8 h	24 h	48 h	g/t		Gravity %	Flotation %	Grav +Flot + CN, %
F25 Rougher Concentrate	58	75	90	94.5	37	658	15.9	87.6	85.5
	-	85	91	92	25	591			86.6
F26 First Cleaner Concentrate	41	71	92	90.6	246	2,615	13.2	81.7	77.4
F29 First Cleaner Concentrate	16	54	70	81.4	911	4,897	17.0	80.9	65.9
	25	70	81	84.1	671	4,229			68.0
	32	73	89	92.1	344	4,362			74.5
F30 Rougher Concentrate	51	71	82	89.5	134	1,270	15.1	84.2	79.1
F31 Rougher Concentrate	49	68	82	91.8	129	1,575	15.1	83.5	80.7
F31 Rougher Concentrate	68	88	94	96.1	65	1,653	15.1	83.5	83.2
F32 Rougher Concentrate	48	72	84	92.2	108	1,385	15.1	87.4	83.5
LCT 3 First Cleaner Concentrate	32	59	74	76.3	602	2,540	13.6	74.0	62.4
	35	62	77	79.7	485	2,385			64.6
	49	65	81	88.1	290	2,428			69.9

13.3.6.5 Zgounder Plant Concentrate

The new plant will receive concentrate from the existing flotation plant. To test its compatibility with the proposed overall flowsheet, a sample of concentrate was obtained from the existing operation and subjected to cyanidation testing using similar conditions as determined before for the main composite sample. It also confirmed that the cyanide consumption will be similar. The conditions and results are summarised in Tables 13.17 and 13.18.

Silver extractions exceeded 90% for all the conditions tested as shown in Table 13.18. This marginally exceeds the recoveries achieved from leaching the master composite concentrates. Leaching the current operating plant concentrate with 4 g/l NaCN at 10% solids yielded ~96% Ag extraction. A similar recovery was achieved with 12 g/l NaCN at 50% solids, with faster leaching kinetics, but Cu and Zn values in the leach pregnant solutions were very high. A single test was completed with 500 g/t lead nitrate, which indicates that the addition of lead nitrate did not improve leach kinetics or overall Ag extraction.

Generally, the sample from the current operation behaved similarly to the master composite and the results indicate that the conditions chosen for the master composite would also work for the concentrate from the existing plant.

Table 13.17 – Zgounder Plant Flotation Concentrate Cyanidation Test Conditions Summary

Sample	CN Test No.	Grind Size P_{80}, µm	Pulp Density % (w/w)	Retention Time, h	[NaCN] g/l	Lead Nitrate, g/t	Dissolved Oxygen, ppm	Reagent Add'n, kg/t of CN Feed		Reagent Cons, kg/t of CN Feed	
								NaCN	CaO	NaCN	CaO
Plant Flotation Conc.	CN9	79	10	48	1	-	30.9	21.9	1.61	14.3	0.28
	CN10				2		31.9	32.8	1.77	17.0	0.03
	CN11				4		31.3	55.1	1.76	21.9	0.04
	CN12				1	500	30.0	21.6	1.81	14.0	0.42
Plant Flotation Conc. - 2 kg*	CN25	79	10	48	4	-	26.5	51.8	1.61	19.2	0.00
Plant Flotation Conc.	CN46	84	50	48	12	-	30.1	34.2	0.79	25.5	0.00

*PLS submitted for Merrill-Crowe and Electrowinning

Table 13.18 – Zgounder Plant Flotation Concentrate Cyanidation Test Results Summary

CN Test	Silver Extraction, %					Ag Residue, g/t			Head, g/t Ag	Final PLS Assays, mg/l		
No.	4 h	8 h	12 h	24 h	48 h	A	B	Avg.	Calc.	Cu	Pb	Zn
9	55	82	85	91	93.5	278	270	274	4,226	185	0.6	73.8
10	64	86	88	93	95.2	207	209	208	4,332	210	1.0	119
11	74	89	87	90	95.9	173	176	175	4,294	220	1.7	154
12	57	83	85	94	94.1	254	251	253	4,251	119	1.0	80.6
25	70	79	80	83	91.6	366	373	370	4,423	245	1.9	164
46	83	88	88	97	96.3	174	180	177	4,770	1098	-	903

13.3.7 MERRILL-CROWE (MC) CEMENTATION TESTS

Cementation testwork was completed using the pregnant leach solution from Test CN 25, which was performed with the Zgounder Plant Concentrate.

The cementation test conditions are outlined in Table 13.19. The cyanide concentration and pH of the feed solution were high enough that additional cyanide and lime were not needed. The standard test procedure is to start with a minimum of 0.25 g/L NaCN and pH 11. The stoichiometric requirement of zinc was added based on the mass of silver in the pregnant leach solution (PLS). Accordingly, 0.271 g of zinc was added. A quarter of this mass of Lead nitrate (Pb(NO3)2) was also added. The sample was deaerated before the addition of the zinc and lead nitrate. The overall reaction is expressed by Clennell's equation as follows:

$$2\,NaAg(CN)_2 + 4\,NaCN + 2\,Zn + 2\,H_2O \rightarrow 2\,Na_2Zn(CN)_4 + 2\,Ag + H_2 + 2\,NaOH$$

Table 13.19 – Merrill-Crowe Test Conditions

Test No.	Solution		Reagent Addition					Nitrogen Addition			
								De-Aeration		Cementation	
	Vol (l)	pH	NaCN (g)	Ca(OH) (g)	Stoich. Factor	Zn (g)	Pb(NO3)2 (g)	Time (min)	Flow (l/min)	Time (min)	Flow (l/min)
MC1-A	1,000	11.7	0.00	0.000	1	0.271	0.07	30	3.0	30	3.0
MC1-B	1,000	11.7	0.00	0.000	1.5	0.407	0.10	30	3.0	30	3.0
MC1-C	1,000	11.7	0.00	0.000	2	0.543	0.14	30	3.0	30	3.0

Tests were completed at 1, 1.5, and 2 times the stoichiometric zinc requirement. The results are provided in Table 13.20 while the feed and barren solution analyses are given in Table 13.21.

Table 13.20 – Merrill-Crowe Test Results

Test No.	Precipitate Weight	Barren Solution		% PPT
	g	Au, mg/L	Ag, mg/L	Ag
MC1-A	0.692	<0.01	0.13	100.0
MC1-B	0.690	<0.01	0.37	99.9
MC1-C	0.917	<0.01	0.14	100.0

Based on Table 13.21, the silver in the Merrill-Crowe barren solution was low for all tests, with precipitation efficiencies of ~100%. The tests confirmed that low barren silver concentrations could be achieved when adding the stoichiometric zinc requirement and that an excess zinc is not required.

Table 13.21 – Merrill-Crowe Barren Solution Analysis

Element	Unit	Sample ID			
		CN 25 48h PLS	MC-1A Final Solution	MC-1B Final Solution	MC-1C Final Solution
Ag	mg/l	448	0.13	0.37	0.14
Al	mg/l	6.8	6.5	6.4	6.4
As	mg/l	17	16	16	15
Ba	mg/l	0.012	0.01	0.01	0.01
Be	mg/l	< 0.002	< 0.002	< 0.002	< 0.002
Bi	mg/l	< 1	< 1	< 1	< 1
Ca	mg/l	13.8	16.7	16.7	16.4
Cd	mg/l	< 1	< 0.09	< 0.09	< 0.09
Co	mg/l	0.3	0.4	0.5	0.5
Cr	mg/l	< 0.1	< 0.1	< 0.1	< 0.1
Cu	mg/l	245	218	219	220
Fe	mg/l	80.5	85.2	85.9	88.8
K	mg/l	11	11	11	11
Li	mg/l	< 2	< 2	< 2	< 2
Mg	mg/l	< 0.07	< 0.07	< 0.07	< 0.07
Mn	mg/l	0.07	0.06	0.06	0.05
Mo	mg/l	2.1	2.1	2.0	2.0
Na	mg/l	3000	2580	2530	2540
Ni	mg/l	1.7	1.7	1.5	1.5
P	mg/l	< 5	< 5	< 5	< 5
Pb	mg/l	1.9	< 2	4	< 2
Sb	mg/l	< 1	< 1	< 1	< 1
Se	mg/l	< 3	< 3	< 3	< 3
Sn	mg/l	< 2	< 2	< 2	< 2
Sr	mg/l	0.043	0.045	0.044	0.043
Ti	mg/l	< 0.02	< 0.02	< 0.02	< 0.02
Tl	mg/l	< 3	< 3	< 3	< 3
U	mg/l	< 1	< 1	< 1	< 1
V	mg/l	< 0.2	< 0.2	< 0.2	< 0.2

Element	Unit	Sample ID			
		CN 25 48h PLS	MC-1A Final Solution	MC-1B Final Solution	MC-1C Final Solution
W	mg/l	< 2	< 2	< 2	< 2
Y	mg/l	< 0.02	< 0.02	< 0.02	< 0.02
Zn	mg/l	164	328	336	351

13.3.8 HYDROGEN PEROXIDE CYANIDE DESTRUCTION

It is important to eliminate cyanide from recycling solutions before it enters the flotation circuit as cyanide acts as a depressant during flotation. Hydrogen peroxide (H_2O2) can be used to oxidize free and weakly-complexed cyanide into cyanate (OCN-) while metals, such as copper, are precipitated as hydroxides (OH-). Stronger metal-cyanide complexes, such as those of iron, can be removed by adding copper salts to precipitate the iron as copper (II) ferrocyanide according to:

$$2\,Cu^{2+} + [Fe(CN)_6]^{4-} \rightarrow Cu_2Fe(CN)_6$$

Four different tests were conducted on the barren solution from test CN 81 to determine the optimum dosage of hydrogen peroxide addition as detailed in Table 13.22. The test duration was similar for all four tests at 60 minutes. The results show that the minimum free cyanide concentration is achieved when adding 250% the stoichiometric addition of H_2O2, i.e. the optimum peroxide addition to the barren solution is at the dosage of 3.27 g of 100% H_2O2 per g CNWAD. Since this result became available only after completion of the financial analysis, it was not incorporated in the FS but will need to be adopted for future phases of the Project. The value used in this FS was 20% lower at 2.6 g/g CNWAD and was based on typical industry reported ratios.

Table 13.22 – Hydrogen Peroxide Cyanide Destruction Test Conditions and Results

Test ID	Final pH	EMF AgCl, mV	H_2O2				Lime	Solution Analysis				
			Conc. used, %	Addition, g	Stoich. (100 % H_2O2)	Dosage, g of 100% H_2O2 per g CN WAD	Addition, g	CNT mg/L	CNWAD mg/L	CNfree mg/L	Cu mg/L	Fe, mg/L
Feed	10.6	-500	…	…	…		…	1120	849	828	20.9	97.0
H1	10.3	-413	30	2.78	150	1.96	0.00	349	27.4	12.6	14.8	91.3

Test ID	Final pH	EMF AgCl, mV	H_2O2				Lime	Solution Analysis				
			Conc. used, %	Addition, g	Stoich. (100 % H_2O2)	Dosage, g of 100% H_2O2 per g CN WAD	Addition, g	CNT mg/L	CNWAD mg/L	CNfree mg/L	Cu mg/L	Fe, mg/L
H2	10.4	-7.3	30	4.63	250	3.27	0.00	198	8.2	3.5	4.72	113
H3	10.4	7.8	30	9.25	500	6.54	0.00	132	5.2	4.1	1.09	50.8
H4	8.8	200	30	16.7	900	11.8	0.36	115	5.3	4.7	0.59	38.7

13.3.9 Carbon Adsorption Modelling

Carbon adsorption tests were conducted to establish model parameters for this particular slurry and activated carbon. The model parameters were then used to perform various simulations to explore the effect of carbon movement and carbon inventory on adsorption efficiency in a carousel Carbon-in-pulp (CIP) Plant. Seven (7) different scenarios were simulated as shown in Table 13.23. The following parameters were identical for all seven (7) scenarios:

- Feed material silver head grade = 30 g/t;
- Leach residence time prior to adsorption = 36 hrs;
- Leach extraction from combined feed products prior to CIP = 62.5%;
- Number of CIP stages = 8;
- Slurry residence time per adsorption stage = 80 mins.

Table 13.23 – Carbon Adsorption Modelling Results

Description	Unit	Scenario #						
		1	2	3	4	5	6	7
Daily carbon transfer / batch elution capacity	kg/day	12,000	12,000	18,000	18,000	18,000	18,000	9,900
Carbon inventory per stage	kg	12,000	18,000	12,000	18,000	18,000	18,000	12,000
Carbon frequency advance	% in 24 hrs	100	67	150	100	100	100	83
Carbon residence time per stage	h	24	36	16	24	24	24	29
Silver in final barren solution	mg/l	0.340	0.256	0.222	0.189	0.104	0.360	0.490

Description	Unit	Scenario #						
		1	2	3	4	5	6	7
Silver in loaded carbon	g/t	5,409	5,429	3,658	3,663	3,626	3,736	6,494
Silver on stripped carbon	g/t	100	100	100	100	50	200	100
Silver Lock-Up on Carbon	kg	153.3	212.8	91.2	125.6	118.8	139.1	202.5
Overall Silver Adsorption Efficiency	%	98.5	98.9	99.1	99.2	99.6	98.5	97.9
Overall Silver Recovery	%	61.6	61.8	61.9	62.0	62.2	61.5	61.2

Scenario 1 represents the base case which reflects the design criteria adopted by DRA for the FS. For this scenario, the carbon inventory per stage was 12 tonnes and it was moved once every 24 hrs. The eluted carbon loading was assumed to be 100 g/t. The simulation predicts a loaded carbon grade of 5,409 g/t, a dissolved silver loss of 0.34 mg/l, and a total silver lock-up of 153.3 kg in the CIP circuit.

Scenarios 2, 3, and 4 were designed to lower the soluble silver losses from the last stage of the carousel CIP while Scenarios 5 and 6 illustrated the significance of good and poor elution, respectively. Scenario 2 looked at increasing the carbon inventory but slowing down the cycle time to maintain the same elution rate of 12 t/d. The silver losses decreased marginally, but at the expense of an increase in silver lock-up.

Scenario 3 used the same inventory as the base case, but moved the carbon batch every 16 hours which increases the elution rate to 18 t/d. Both the dissolved loss as well as lock up improved relative to the base case, but the increased cost of eluting more carbon is expected to render this scenario less attractive from an economic perspective.

Scenario 4 is a combination of 2 and 3 and saw further improvements in dissolved losses, but again at a net increase in costs. From Scenarios 2 to 4, it was determined that the most economical way of operating this plant will be to slow the carbon advance rate and push the loading in stage 1 as close to the equilibrium isotherm as possible, which was later examined in Scenario 7.

By pushing the carbon loading up to the isotherm limit (~6500 g/t Ag), the barren solution silver losses in Scenario 7 increased to 0.49 mg/l Ag, which represents a decrease in silver revenue compared to Scenario 1. However, the decrease in elution costs is substantial as the elution rate was decreased from 12 to 9.9 t/d only. This scenario is viewed as the most economical of all scenarios tested and should be adopted in future phases of the Project.

As these simulations demonstrated that the DRA design criteria was sufficiently close to the optimum, no changes were made to the PDC as a result of this work. Reducing the carbon circuit from a 12 to 10 t/d circuit is an opportunity for future phases to reduce capital costs marginally.

13.3.10 DEWATERING TESTS

13.3.10.1 Flotation Concentrate

a. Dynamic Thickening

Dynamic thickening test results indicated that the Flotation Concentrate sample responded well to BASF Magnafloc 10 flocculant at a dosage of 25 g/t. Thickener unit areas examined ranged from 0.20 to 0.12 m^2/tpd. The underflow density was 66.1% w/w solids at 0.20 m^2/tpd and decreased to 59.9% w/w solids at 0.12 m^2/tpd. Overflow Total Suspended Solids (TSS) remained at ~30 mg/L for all the tests. A 30-minute period of extended thickening, without feed or raking, increased the underflow density from 66.1% to 68.3% solids when operating at 0.20 m^2/tpd unit area. The corresponding yield stress increased from 29 Pa at 66.1% solids to 42 Pa at 68.3% solids. The period of extended thickening was included to investigate the thickening response of the thickened bed during a simulated period of thickener standby operation. Thickening results are summarised in Table 13.24.

Table 13.24 – Flotation Concentrate – Summary of Thickening Results by Unit Area

Dosage flocc't, g/t	Unit Area, m^2/(t/d)	Solids Loading, t/m^2/h	Net Rise Rate m^3/m^2/d	Underflow, %w/w solids	Overflow TSS, mg/L	Residence Time, h	U/F Yield Stress, Pa
25	0.20	0.21	38.0	66.1	30	0.76	29
25	0.18	0.23	42.2	65.6	24	0.66	21
25	0.16	0.26	47.4	63.2	28	0.58	11
25	0.14	0.30	54.2	61.3	30	0.51	7
25	0.12	0.35	63.3	59.9	33	0.44	6
Underflow extended for 30 minutes:				68.3			42

Bed height was maintained around 150 mm

b. Underflow Rheology

The Critical Solids Density (CSD) of the flotation concentrate sample was ~68% solids. This sample exhibited a yield stress of 36 Pa (unsheared) and 23 Pa under sheared conditions. Thixotropic behaviour was exhibited by the sample when test densities exceeded 68% solids. Thixotropic response is a “flow-friendly” behaviour whereby the resistance to flow decreases during constant shearing. Plug flow responses were exhibited by the sample at 71.5% solids, during the unsheared sample measurement.

c. CCD thickening tests and modelling

Flotation concentrate leached pulp from test CN62 was subjected to a program of thickening tests, combined with CCD modelling, to determine the optimal design parameters for this unit operation.

Static tests were initially performed to select the most appropriate flocculant Magnaflocc333, a very high molecular weight anionic polyacrylamide was selected.

Based on initial modelling, a 5 stage CCD circuit with no filtration of the final barren pulp was selected for validation testing and modelling. This circuit yielded an overall washing efficiency of 99.5% with a water consumption of 1.39 m^3/tonnes of dry solids washed. This scenario was selected as it achieves a reasonably high washing efficiency for intermediate capital expenditure and mid-range water consumption.

Three sets of dynamic settling tests were conducted with slurry that simulated the solution feed to CCD thickeners 1, 3 and 5 respectively. The purpose of these tests was to determine the optimum flocculant addition in each thickener as well as the optimum unit area requirements. Based on the tests and modelling, the design criteria established are dilution of the feed to 20%w/w solids, stages 1 to 3 dosed at 40 g/t flocculant and stages 4 and 5 at 30 g/t flocculant. The underflow density will be 64% w/w solids and the overall wash efficiency 99.5%. However, the water consumption will be marginally higher than originally predicted at 1.69 m^3/t.

d. Vacuum and Pressure Filtrations

Vacuum and pressure filtration tests were completed on the flotation concentrate underflow at 66% solids. Vacuum filtration was conducted at a 15 inHg (0.51 bar) vacuum level. Micronics 8963 polypropylene cloth was selected for the vacuum filtration tests. The filtration test cake thicknesses ranged from 14 to 40 mm. The resulting specific filtration rate ranged from 290 to 839 kg/m^2h. The final cake residual moisture content ranged from 19.6% to 21.3% moisture.

Pressure filtration was conducted at 5.5 bar and 6.9 bar pressure levels. Testori P4408 TC polypropylene cloth was selected for the tests. The test cake thickness ranged from 25 to 44 mm. Filter throughput (estimated full cycle time) ranged from 240 to 330 kg/m^2h. The discharge cake moisture content ranged from 12.4 to 14.6%.

13.3.10.2 Flotation Tailings

a. Dynamic Thickening

Dynamic thickening test were performed using the LCT 3 Combined Flotation Tailing sample. The results showed that the sample responded well to BASF Magnafloc 504 flocculant at a dosage of 110 g/t.

During each test, the feed rate was adjusted to test different specific unit areas. These ranged from 0.28 to 0.16 m^2/(t/d). The underflow density reached 57.7% w/w solids at 0.28 m^2/(t/d) and decreased to 49.8% w/w solids at 0.16 m^2/(t/d). The total suspended solids (TSS) content in the overflow stream increased from 96 mg/l to 142 mg/l as the unit area was decreased. A 30-minute period of extended thickening, without feed or raking, allowed the underflow density to increase marginally from 57.7% solids to 58.6% solids when operating at 0.28 m^2/tpd unit area. The corresponding yield stress increased from 63 Pa at 57.7% solids to 79 Pa at 58.6% solids. Thickening results are summarised in Table 13.25.

Table 13.25 – Flotation Tailings – Summary of Thickening Results by Unit Area

Dosage flocc't, g/t	Unit Area, m^2/(t/d)	Solids Loading, t/m^2/h	Net Rise Rate m^3/m^2/d	Underflow, %w/w solids	Overflow TSS, mg/L	Residence Time, h	U/F Yield Stress, Pa
110	0.28	0.15	63.9	57.7	96	0.74	63
110	0.25	0.17	71.6	55.9	109	0.66	46
110	0.22	0.19	81.4	53.7	120	0.58	34
110	0.19	0.22	94.2	51.6	136	0.50	28
110	0.16	0.26	111.9	49.8	142	0.42	15
Underflow extended for 30 minutes:				58.6			79

Bed height was maintained around 150 mm

b. Underflow Rheology

The Critical Solids Density (CSD) of the flotation tailings sample was ~57% w/w solids, which exhibited a yield stress of 28 Pa under unsheared flow conditions and 18 Pa under sheared conditions. Thixotropic behaviour was exhibited by the sample at test densities exceeding 56.6% solids. Plug flow responses were exhibited by the sample at 61.6% solids, during the unsheared sample measurement.

c. Vacuum and Pressure Filtrations

Vacuum and pressure filtration tests were completed using a sample of the flotation concentrate underflow with the slurry at 56% w/w solids. Vacuum filtration was conducted at 15 a inHg (0.51 bar) vacuum level. Micronics 8963 polypropylene cloth was selected for the vacuum filtration tests. The filtration test cake thicknesses ranged from 10 to 21 mm. The resulting specific solids filtration rates ranged from 64 to 237 kg/m^2h. The discharge cake residual moisture content ranged from 23.1% to 25.3% moisture. Pressure filtration was conducted at 5.5 bar and 6.9 bar pressure levels. Testori P4408 TC polypropylene cloth was selected for the tests. The test cake thickness ranged from 15 to 30 mm. Filter throughput per unit area (estimated full cycle time) ranged from 122 to 199 kg/m^2h. The product cake moisture content ranged from 16.1 to 17.9%.

d. Variability Static Settling Test

To examine the settling response at an elevated pH, a LCT 3 Combined 1st Cleaner Scavenger and Rougher Tailings Cycles A-F sample at pH 10.5 was subjected to an additional static settling test. In this case, a coarser blend of F33 Flotation Tailings (refer to Table 13.26 for the sample characterisations) underwent two (2) stages of static settling tests (i.e. feed solids density and flocculant dosage optimisation stages) in 2 L graduated cylinders fitted with a rotating "picket-style" rake. Table 13.27 indicates that both results achieved similar response compared to the original benchmark test at pH 7.8 (highlighted in green) with only half of flocculant dosage. Even though the settling response was also improved for F33 Flotation Tailings, it is not as effective as the pH adjustment. It should be noted that all values across these tables were calculated without a safety factor.

Table 13.26 – Sample Characterisation of F33 Flotation Tailings

Sample I.D.	Particle Sizing			SG of Dried Solids	Testing pH	Liquor Density, kg/l
	[1]d_{80}, µm	[1]<20 µm % vol	[1]<1 µm % vol			
F33 Flotation Tailings	98	45.0	4.4	2.81	7.3	1.00

1 Determined using laser diffraction (Malvern).

Table 13.27 – Variability Static Settling Test Results Summary

Sample ID	Testing pH	Reagent Magnafloc	Flocc't Dosage g/t	1Feed % w/w	2U/F % w/w	Unit Area $m^2/(t/d)$	3ISR $m^3/m^2/d$	4Supernatant Clarity	5TSS mg/l
LCT3 Comb 1st Cleaner Scavenger and Ro Tailings Cycles A-F	7.8	504	71	5	53	0.28	457	Hazy	30
	10.5	504	36	5	50	0.26	411	Hazy	36
F33 Flotation Tailings	7.3	504	63	5	52	0.24	410	Hazy	22

1 Diluted Thickener Feed
2 Final Thickened "Underflow" Density
3 Initial Settling Rate
4 Supernatant Visual Clarity at 10 minutes of elapsed settling time
5 Supernatant Total Suspended Solids (TSS) at 10 minutes of elapsed settling time

13.3.11 Overall Balances

13.3.11.1 Main Composite Overall Recoveries

SGS created Table 13.28 to show the silver distribution, overall recoveries (gravity + flotation + cyanidation) and reagent consumptions for four scenarios based on the testwork outputs. Three (3) grind sizes (80% passing 116, 95 and 66 microns) are shown, and for the 95 micron grind, the effect of a high cyanide addition (12 g/l as opposed to 4 g/l) is also shown. The overall reagent consumptions, which combines the cyanide used in all three (3) leach sections, were high, ranging from 2.96 kg/t at the coarsest grind to 6.37 kg/t for the high cyanide addition scenario. Overall, silver recoveries ranged from 82.9% to 90.6%, with the best outcome for the test with the elevated NaCN addition (12 g/l) in the flotation concentrate leach. Note that the recoveries were calculated from the residue and head solids assays and not calculated from less accurate solution assays.

Table 13.28 – Main Composite Overall Recoveries

K_{80}	Gravity %	NaCN, g/l		% Distr'n Ag		% Rec'y Ag		Overall Reagent Cons, kg/t		% O'all Rec'y	Comb Tail
µm	Rec'y Ag	Flot Conc	Flot Tail	Flot Conc	Flot Tail	Conc CN	Tail CN	NaCN	CaO	Grav+Flot+ CN, Ag	g/t Ag
116	21.2	4	4	71.5	13.4	64.0	9.3	2.96	0.75	82.9	34.3
95	18.8	4	4	72.5	14.3	66.6	9.8	3.60	0.83	86.1	27.9
95	18.8	12	4	72.5	14.3	69.7	9.8	6.37	0.89	90.6	18.9
66	18.1	4	4	76.9	11.1	70.9	8.1	3.33	0.92	86.5	27.0

13.3.11.2 Zgounder Historical Tailings

The Zgounder Historical Tailings sample was processed through the gravity-flotation-cyanidation flowsheet and the results are summarised in Table 13.29. The sample was ground to 80% passing 92 microns which is similar to the design value used in the FS. Gravity recovery was low at 0.7%. Rougher flotation recovered 44% of the silver into a rougher concentrate. Leaching of this rougher concentrate extracted 92% of the contained silver, while leaching of the rougher tailings extracted 68% of its contained silver. The overall gravity-flotation-cyanidation silver recovery was 79.5%, which is more than acceptable for a tailings sample.

Table 13.29 – Zgounder Historical Tailings – Overall Balance

Sample	K_{80}	Gravity %	% Distr'n Ag		% Rec'y Ag		Overall Reagent Cons, kg/t ore		% O'all Rec'y	Comb Tail Assa	Head Ag g/t	
	µm	Rec'y Ag	Flot Conc	Flot Tail	Conc CN	Tail CN	NaCN	CaO	Grav+Flot+ CN, Ag	g/t Ag	Calc	Direct
F34	92	0.7	44.4	55.6	40.8	38.0	2.42	0.94	79.5	22.4	111	106

13.3.12 VARIABILITY TESTING

The five (5) variability samples were processed through gravity, gravity tailings flotation, and cyanidation of the flotation products. Knelson/Mozley gravity Ag recoveries ranged from 3.9 to 16.4%, with an average of 11.4%.

Flotation tests on the gravity tailings were completed employing the standard conditions of 20 minutes of roughing at natural pH with PAX, AERO 241 and MIBC addition. The flotation tests recovered 59 to 69% of the total silver in the original feed samples to a rougher concentrate at mass pulls of 5 to 12%.

Additional tests were completed with longer froth times to try to improve recoveries. Ag recoveries increased by ~5% with longer froth recovery times, but with higher mass pulls (12.6% to 23.9%).

The rougher concentrates and tailings were leached with cyanide under the same conditions as applied to the Main Comp flotation concentrates and tailings. The NaCN addition for the concentrate leaches was 12 g/l, and for the tailings, 4 g/l. The stage extraction of silver when leaching from the flotation concentrate ranged from 82.4% to 98.7%, but with very high NaCN consumptions of 20.7 kg/t to 113 kg/t. Expressing this in terms of the original head grade, these recoveries ranged from 58.1% to 66.8% as shown in Table 13.30.

The flotation tailings tests reported Ag extractions of 75% to 86% from the float tailings with NaCN consumptions of 1.0 to 2.5 kg/t of cyanide feed. Expressed in terms of the head grade, the recovery into solution from the tailings samples ranged from 13.8% to 30.8% as shown in the table below.

The overall gravity-flotation-cyanidation results are shown in Table 13.30. The best overall recovery was achieved with VAR-2 at 94.4%.

Table 13.30 – Variability Samples Overall Gravity-Flotation-Cyanidation Balances

Sample	K_{80}	Gravity %	% Distr'n Ag		% Rec'y Ag		Overall Reag		% O'all Rec'y	Comb Tail Assa	Head Grade Assa
	µm	Rec'y Ag	Flot Conc	Flot Tail	Conc CN	Tail CN	NaCN	CaO	Grav+Flot+ CN, Ag	g/t Ag	g/t Ag
VAR-1	99	10.3	67.7	22.0	59.2	16.5	2.40	0.73	86.0	2.5	22
VAR-2	102	16.4	59.0	24.6	58.2	19.8	3.12	0.56	94.4	14.1	252
VAR-3	107	13.1	68.9	18.0	66.8	13.8	4.00	0.50	93.7	16.0	302
VAR-4	67	3.9	60.3	35.9	58.1	30.8	3.23	1.10	92.8	5.9	99
VAR-5	88	10.2	69.1	20.8	58.2	16.1	15.9	0.45	84.4	7.7	107

13.4 Deleterious Elements

Head sample assays yielded mercury grades of 12.2 g/t for the Main Comp sample and less than 15 g/t for the variability samples. Two (2) mercury retorts have been included in the flowsheet to remove mercury ahead of the smelting step.

Arsenic assays yielded 541 and 259 g/t for variability samples VAR-4 and VAR-5 respectively. All the other samples assayed at below 100 g/t for arsenic. The pregnant leach solution, as well as the four Merrill Crowe effluent solutions, contained only 16 mg/L of arsenic. These solutions also contained more than 80 mg/l iron, which exceeds the rule-of-thumb of a 3:1 iron to arsenic ratio which ensures that arsenic is stabilized as a ferric-arsenate precipitate. Consequently, there is no need to provide for active arsenic removal in the design.

13.5 Interpretation of Results

From the given test results, the following findings should be implemented to the existing flowsheets and design criteria:

- Based on the mineralogical examination, the anticipated silver recoveries were expected to be in the high 80% to low 90% range. This was confirmed throughout the test program.
- The E-GRG number for the Main Comp was 33.8%, indicating that it would likely be beneficial to include a gravity circuit in the final flowsheet. Silver recovery into the Mozley table concentrate, which more realistically mimics full-scale gravity concentration operations, was more modest at around 15%. However, considering the high head grade, this is still a significant amount of silver which can be removed early in the flowsheet.
- Flotation tests confirmed that the gravity tailings are amenable to flotation and that inclusion of this step would allow the production of a low volume concentrate which, ultimately, reduces the size of the CCD thickeners appreciably.

- Cyanidation testwork showed that the best recoveries are achieved at extended durations and high cyanide levels. Unfortunately, these conditions also elevate cyanide consumption and necessitates the need to include cyanide detoxification to allow re-use of the reclaim water in a flotation circuit. Additional tests with pre-oxygenation and elevated oxygen levels showed that reduced cyanide consumption is achievable when oxidizing cyanide consumers prior to and during the initial phase of leaching.
- The optimum peroxide addition to the barren solution is at the dosage of 3.27 g of 100% H_2O2 /g CNWAD so that the free cyanide is minimized.
- The carbon modelling work showed that the optimum configuration would be at a marginally smaller carbon movement rate of around ~10 t/d. However, for the purposes of this FS, the initial batch size of a 12-tonne carbon circuit was retained.

The following tests have been conducted by SGS Canada Inc. in 2021, yet excluded from this section, since they do not directly form part of the final flowsheet:

- Electrowinning testing; and
- Davis Tube testing.

14 MINERAL RESOURCES

This Section has been summarised and/or largely extracted from the Report available on SEDAR entitled: "Technical Report and Updated Mineral Resource Estimate of the Zgounder Silver Project, Kingdom of Morocco, prepared for Aya Gold & Silver Inc., dated January 28, 2022, prepared by P&E Mining Consultants Inc., Report # 415."

The Mineral Resource Estimate presented herein has been prepared following the guidelines of the Canadian Securities Administrators' National Instrument 43-101 and Form 43-101F1, and in conformity with generally accepted "CIM Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines (2019). Mineral Resources have been classified in accordance with the "CIM Standards on Mineral Resources and Reserves: Definition and Guidelines" as adopted by CIM Council (2014).

> *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. A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.*

> *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 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. An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.*

> *An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are 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. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.*

Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no guarantee that all or any part of the Mineral Resource will be converted into a Mineral Reserve. Confidence in the estimate of Inferred Mineral Resources is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure.

The author of this Technical Report section is not aware of any known permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource Estimate. All of the Mineral Resource estimation work reported herein was reviewed by Eugene Puritch, P.Eng., FEC, CET., an independent Qualified Person as defined by National Instrument 43-101 by reason of education, affiliation with a professional association, and past relevant work experience.

The effective date of the Mineral Resource Estimate is December 13, 2021.

14.1 Introduction

The 2021 Mineral Resource Estimate was completed by Marc-Antoine Audet, Ph.D. P.Geo., Geological Consultant and Aya Gold & Silver's "Qualified Person" and was reviewed by the author of this Technical Report section.

A summary of the 2021 Mineral Resources effective December 13, 2021 is provided in Table 14.1.

Table 14.1 - Updated Mineral Resource Estimate

Area	Class	Cut-Off (Ag g/t)	Tonnes (k)	Ag (g/t)	Ag (koz)
Pit Constrained	Measured	65	108	477	1,656
	Indicated	65	406	325	4,242
	Measured +Indicated	65	514	357	5,898
	Inferred	-	-	-	-
Out-of-Pit	Measured	75	3,403	343	37,527
	Indicated	75	5,576	289	51,810
	Measured +Indicated	75	8,979	309	89,337
	Inferred	75	542	367	6,395
Tailings	Measured	-	-	-	-
	Indicated	50	272	94	817
	Measured +Indicated	50	272	94	817
	Inferred	-	-	-	-
Total	Measured	-	3,511	347	39,183
	Indicated	-	6,254	283	56,869
	Measured +Indicated	**-**	**9,765**	**306**	**96,052**
	Inferred	-	542	367	6,395

14.2 Mineral Resource Database

A GEMS project developed by M. Audet was provided for review. The project folder contains drilling and sampling data, block models and estimation profiles. Geology zones, underground workings and a topographic surface are also included. The coordinate reference system used is Merchich/Sud Maroc (EPSG 26192).

The Mineral Resource Estimate data are stored as a Microsoft Access database. Drilling and sampling data include collar, downhole survey, assay, lithology and bulk density tables.

The supplied collar table contains 3,139 records with a total length of 138,468.92 m (Table 14.2). The drilling extends approximately two km along strike. (*Appendix A*)

Table 14.2 – Drill Hole Summary

Type	Count	Total (m)
DDH	337	70,167.52
DDHGT	2	616.10
DDHU	193	17,264.48
RC	30	3,462.00
Trench	7	264.00
UG Channel	658	9,071.42
UG Drilling	1,912	37,623.41
Total	**3,139**	**138,468.93**

Industry standard validation checks were carried out on the supplied databases, and minor corrections made where necessary. The author of this Technical Report section typically validates a Mineral Resource database by checking for inconsistencies in naming conventions or analytical units, duplicate entries, interval, length or distance values less than or equal to zero, blank or zero-value assay results, out-of-sequence intervals, intervals or distances greater than the reported drill hole length, inappropriate collar locations, and missing interval and coordinate fields.

No significant issues were noted with the database. However, the author of this Technical Report section did note that the assay database contains 504 “-1” Ag grade entries and 2,600 “0” Ag grade entries. M. Audet reports that a zero-grade entry should be considered a valid result corresponding to “below-detection-limit”. The author of this Technical Report section also noted 59 zero length assays, which were subsequently corrected.

The author of this Technical Report section considers that the supplied drill hole database is suitable for Mineral Resource estimation.

14.3 Economic Considerations

Based on knowledge of similar projects, review of available data, and consideration of potential mining scenarios, the economic parameters listed in Table 14.3 were deemed appropriate for the Mineral Resource Estimate. The Ag price corresponds to the approximate 24-month trailing average as of November 31, 2021.

Table 14.3 – Economic Parameters

Item	Unit	Value
Ag Price	$US/oz	22.50
Process Recovery	%	90
Processing Cost	$US/t	20.00
Tailings Processing Cost	$US/t	16.50
G&A	$US/t	7.00
Open Pit Mining Cost	$US/t	15.00
Underground Mining Cost	$US/t	22.00
Tailings Mining Cost	$US/t	9.00
Open Pit Cut-off	g/t	65
Underground Cut-off	g/t	75
Tailings Cut-off	g/t	50

14.4 Block Model

An orthogonal block model was established with the block model limits selected in order to cover the overall extent of the mineralization (Table 14.4). The block model consists of separate variables for estimated grades, volume percent domain inclusion, rock codes, bulk density and classification attributes. It is noted that the orientation of the block model does not correspond to the principal orientation of the existing underground workings.

Table 14.4 – Block Model Setup

Direction	Origin	Number of Blocks	Block Size (m)
Minimum X	275,532	680	2
Minimum Y	420,086	392	2
Maximum Z	2,250	326	2
Rotation	No Rotation 0°		

A model block size of 2 m x 2 m x 2 m was selected by M. Audet in order to constrain mineralization boundaries and facilitate mine planning. It is noted that the block size selected does not reflect the dimensions of a reasonable Selective Mining Unit, and that estimation with small blocks may over-smooth the kriged estimates and distort the global Mineral Resource Estimate summary.[5]

[5] Armstrong M and Champigny N (1989). *A study on kriging small blocks*. CIM Bulletin. Vol. 82, No. 923, pp.128-133.

14.5 Mineralisation Domain

Geological zones were developed by Marc-Antoine Audet based on the local geology, structure, surface and underground drilling, assay grades and logged drill hole lithologies. A total of five individual geological zones were defined. (*Appendix B*)

Table 14.5 – Geological Zones

Zone	Rock Code	Strike Length (m)
NW	210	430
SW	220	430
NE	230	630
SE	240	500
SS	250	430

Each block within a defined geological zone was subsequently categorized by assigning the grade of the nearest 1.20 m composite to the block using an oriented search ellipsoid. The orientation of the search ellipsoid is vertical, rotated 75º clockwise, with a long axis of 40 m, a vertical intermediate axis of 40 m, and a minor axis of 20 m. Blocks with a resulting grade of 40 g/t Ag or higher were categorized as potentially mineralized material and assigned a Rock Code of 210, 220, 230, 240 or 250. Blocks with a nearest composite grade of less than 40 g/t Ag were categorised as waste and assigned a Rock Code of 40. Blocks intersecting mined-out areas were assigned a block code of 10.

The resulting block categorization was then used to back-tag the assay, bulk density and composite tables with unique rock codes. The back-tags were derived directly from the categorized block model (Table 14.3).

14.6 Exploratory Data Analysis

The average length of the surface drill holes is 208 m, the average length of the underground drilling is 20 m, and the average length of the underground channel sampling is 14 m. Summary statistics for the assay data by geological zone are listed in Table 14.6.

Table 14.6 – Ag g/t Assay Summary Statistics

Zone	0	210	220	230	240	250	Total
Count	14,939	22,167	22,440	20,267	21,833	1,196	**102,842**
Minimum	-1	-1	-1	-1	-1	-1	**-1**
Maximum	6,525	60,000	31,050	24,613	17,280	9,480	**60,000**

Zone	0	210	220	230	240	250	Total
Mean	11	77	103	43	102	54	**72**
Std Dev	115	700	543	401	526	379	**515**
CoV	10.15	9.12	5.28	9.23	5.15	7.00	**7.20**

The supplied database contains a total of 1,235 bulk density measurements taken from drill hole core, of which one sample reported a zero-sample length. The average bulk density of the remaining 1,234 measurements is 2.75 tonnes per cubic metre (t/m^3) and the median bulk density is 2.77 t/m^3 (Table 14.7).

Table 14.7 – Summary of Bulk Density Statistics (t/ m^3)

Zone	0	210	220	230	240	250	Total
Count	270	286	191	308	158	21	**1,234**
Minimum	2.16	1.38	1.95	2.06	1.49	2.6	**1.38**
Maximum	4.42	4.78	3.2	3.01	4.56	2.84	**4.78**
Mean	2.73	2.78	2.75	2.75	2.76	2.73	**2.75**
Std Dev	0.17	0.17	0.13	0.09	0.24	0.08	**0.16**
CoV	0.06	0.06	0.05	0.03	0.09	0.03	**0.06**
Median	2.64	2.79	2.77	2.77	2.76	2.76	**2.77**

14.7 Compositing

Assay sample lengths range from 0.10 m to 18.8 m, with an average sample length of 1.22 m, a median sample length of 1.20 m and a mode of 1.00 m. A total of 29% of the assay sample lengths are equal to 1.00 m and 23% are equal to 1.20 m.

The assay samples were composited to a length of 1.20 m down-the-hole in order to ensure equal sample support. Unsampled intervals in the data were assigned a zero grade prior to compositing. Residual composites less than 0.60 m were discarded. A rock code was subsequently assigned to each composite based on the block categorization described in Section 14.5.

It is noted that “-1” and “0” Ag assay grades in the database were incorporated into the compositing process. Surface trench assays were also included in the compositing process. It is also noted that the compositing process generated a small number of spatial duplicates from two (2) drill holes (Table 14.8). The inclusion of spatial duplicates can generate unexpected results during grade estimation.

Table 14.8 – Composite Spatial Duplicates

HOLE ID	X	Y	Z	Ag g/t
T28-21-1975-39E	275,793.7	420,284.1	1,984.05	4
T28-21-1975-40E	275,793.7	420,284.1	1,984.05	4
T28-21-1975-39E	275,793.9	420,285.3	1,984.16	12
T28-21-1975-40E	275,793.9	420,285.3	1,984.16	4
T28-21-1975-39E	275,794.1	420,286.4	1,984.26	16
T28-21-1975-40E	275,794.1	420,286.4	1,984.26	4
T28-21-1975-39E	275,794.3	420,287.6	1,984.37	12
T28-21-1975-40E	275,794.3	420,287.6	1,984.37	4
T28-21-1975-39E	275,794.5	420,288.8	1,984.47	20
T28-21-1975-40E	275,794.5	420,288.8	1,984.47	4
T28-21-1975-39E	275,794.8	420,290.0	1,984.58	8
T28-21-1975-40E	275,794.8	420,290.0	1,984.58	4
T28-21-1975-39E	275,794.9	420,291.1	1,984.68	28
T28-21-1975-40E	275,794.9	420,291.1	1,984.68	12
T28-21-1975-39E	275,795.2	420,292.3	1,984.78	24
T28-21-1975-40E	275,795.2	420,292.3	1,984.78	12
T28-21-1975-39E	275,795.4	420,293.5	1,984.89	20
T28-21-1975-40E	275,795.4	420,293.5	1,984.89	20
T28-21-1975-39E	275,795.6	420,294.7	1,984.99	48
T28-21-1975-40E	275,795.6	420,294.7	1,984.99	28

Summary composite statistics are listed in Table 14.9.

Table 14.9 – Ag g/t Composite Summary Statistics

Zone	0	210	220	230	240	250	Total
Count	82,896	4,733	6,714	2,035	4,960	159	**101,497**
Minimum	0	0	0	0	0	4	**0**
Maximum	8,221	59,500	29,096	15,196	16,272	9,480	**59,500**
Mean	14	267	307	238	319	312	**65**
StdDev	86	1,169	886	757	813	883	**422**
CoV	6.03	4.38	2.89	3.18	2.55	2.83	**6.48**

14.8 Treatment of Extreme Values

A capping threshold of 6,000 g/t Ag was applied on composite values prior to grade estimation. M. Audet states that the 6,000 g/t Ag threshold for composites was derived as follows:

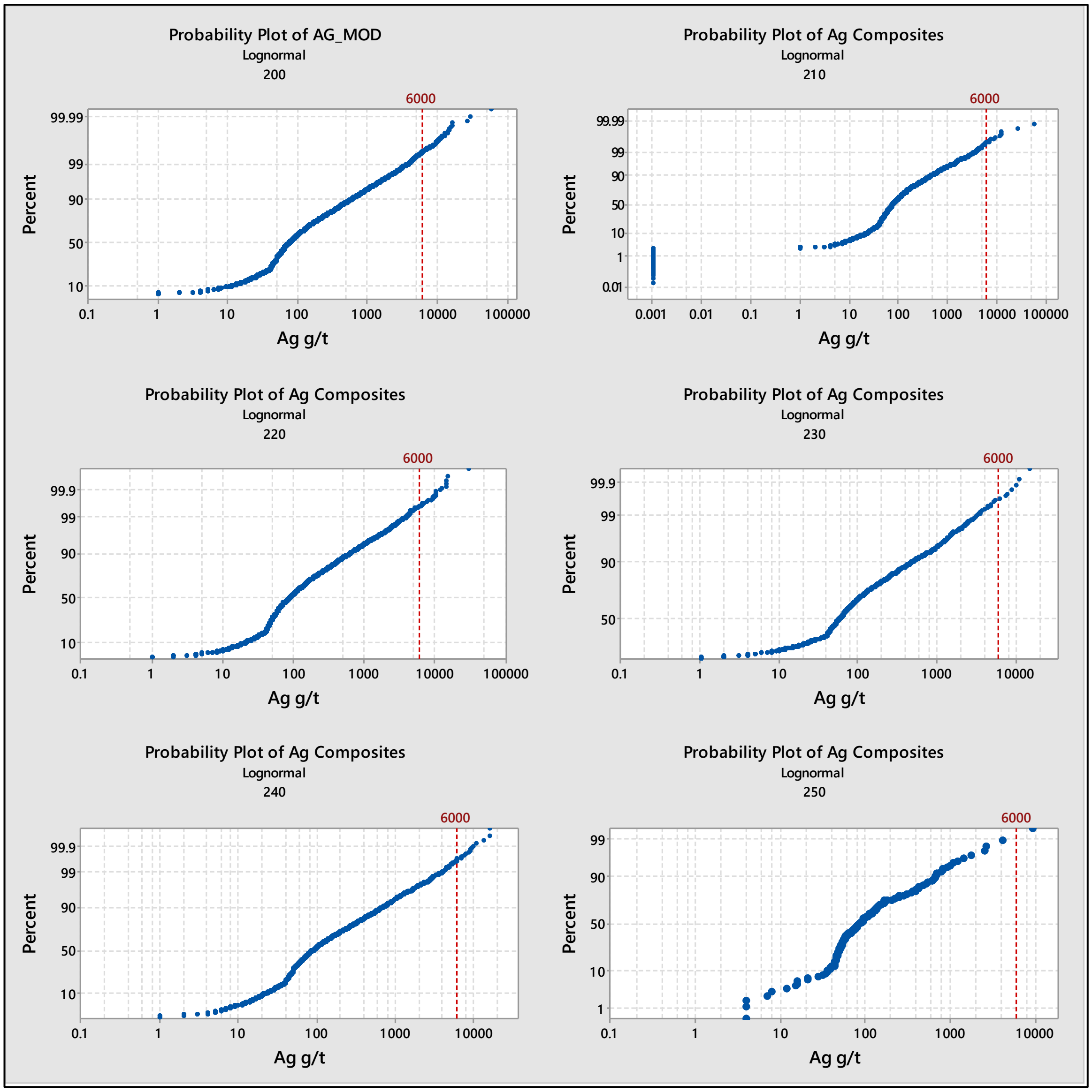
- 102,341 (assay) samples ranging from 0 g/t to 60,000 g/t Ag
- 18,205 (assay) samples greater than 40 g/t Ag
    - Average for the 18,205 (assay) samples: 363 g/t
    - Average of (assay) samples from 40 g/t to 4,500 g/t: 279 g/t
    - Average of (assay) samples from 40 g/t to 6,000 g/t: 299 g/t
- Relatively small variation of the global average between 4,500 g/t and 6,000 g/t
- Large variation when samples above 6,000 are included.
- Capping at 6,000 g/t seems reasonable.

As a check on the “reasonableness” of the selected composite capping threshold, the author of this Technical Report section generated log probability graphs of uncapped composites. The author notes that the log probability graphs suggest that the capping threshold selected is slightly conservative (Figure 14.1).

14.9 Continuity Analysis

Mr. Audet supplied a Supervisor project folder with the variography used for the Mineral Resource Estimate. The Ag variography was modelled on domain-coded uncapped composite data using a normal-scores transformation within each domain. Standardized spherical models were used to model the experimental semi-variograms in normal-score transformed space with four structures. (*Appendix C*) Experimental semi-variograms were developed by Mr. Audet for domains 210, 220, 230 and 240, and then back-transformed. The experimental semi-variograms as entered in the GEMS project file are tabulated in Table 14.10.

Figure 14.1 – Log-Probability Graphs for Composites



Source: P&E, 2022

Table 14.10 – Experimental Semi-Variogram of Ag Composites

Ag 210	Direction 1	Direction 2	Direction 3
Vector	0 > 080	0 > 350	90 > 000
C0	0.68	0.68	0.68
C1	0.18	0.18	0.18
C2	0.10	0.10	0.10
C3	0.04	0.04	0.04
R1	10.0	7.0	6.0
R^2	38.0	53.0	13.0
R3	175.0	60.0	33.5
Ag 220	**Direction 1**	**Direction 2**	**Direction 3**
Vector	0 > 100	0 > 010	90 > 000
C0	0.45	0.45	0.45
C1	0.38	0.38	0.38
C2	0.13	0.13	0.13
C3	0.04	0.04	0.04
R1	8.0	6.0	6.5
R^2	20.5	22.5	14.0
R3	150.0	70.0	19.0
Ag 230	**Direction 1**	**Direction 2**	**Direction 3**
Vector	0 > 090	0 > 000	90 > 000
C0	0.34	0.34	0.34
C1	0.53	0.53	0.53
C2	0.11	0.11	0.11
C3	0.02	0.02	0.02
R1	10.0	3.0	2.5
R^2	50.0	17.0	14.0
R3	75.0	20.0	30.0
Ag 240	**Direction 1**	**Direction 2**	**Direction 3**
Vector	0 > 090	0 > 000	90 > 000
C0	0.43	0.44	0.44
C1	0.33	0.33	0.33
C2	0.17	0.17	0.17

C3	0.07	0.06	0.06
R1	8.5	9.5	5.0
R^2	35.0	23.5	9.0
R3	160.0	25.0	35.0

14.10 Grade Estimation and Mineral Resource Classification

Block grades were estimated by Ordinary Kriging ("OK") using between two and thirteen capped composites, with a maximum of two composites from the same drill hole. Grade estimation and classification were carried out in three-passes using a series of expanding search ellipsoids (Table 14.11):

- First Pass: blocks estimated on the first pass are classified as Measured.
- Second Pass: un-estimated blocks estimated on the second pass are classified as Indicated.
- Third Pass: un-estimated blocks estimated on the third pass are classified as Inferred. If a block is informed by three or more drill holes, it is upgraded to Indicated.

The search ranges selected by M. Audet for estimation and classification correspond roughly to the modelled variography, with the First Pass search range similar to R1, the Second Pass search range similar to R^2, and the Third Pass range set to 50 m x 40 m x 50 m.

The author of this Technical Report section notes that the 3-D orientations of the experimental semi-variogram, the anisotrophy implemented for estimation, and the orientation of the search ellipsoid do not correspond.

Discretization of the 2 m x 2 m x 2 m blocks was set at 3 x 3 x 3. An Inverse Distance Squared ("ID2") model was also generated using the same grade estimation strategy. Bulk density was estimated using the same strategy as that defined for the Ag block grades. Block model cross-sections and plans for Ag and classification are presented in *Appendix D* and *Appendix E*.

Table 14.11 – Search Ellipsoids Ranges (m)

Domain	**Measured** (m)	**Indicated** (m)	**Inferred** (m)
NW 210	15 x 10 x 15	22 x 15 x 22	50 x 40 x 50
SW 220	9 x 6 x 6.5	20 x 18 x 19	50 x 40 x 50
NE 230	10 x 3 x 2.5	25 x 17 x 14	50 x 40 x 50
SE 240	10 x 9.5 x 5	25 x 23 x 9	50 x 40 x 50
SS 250	8 x 6 x 5	25 x 25 x 10	50 x 40 x 50

14.11 Mineral Resource Estimate

Pit constrained Mineral Resources have been reported using a cut-off of 65 g/t Ag. See *Appendix F*. Out of Pit Mineral Resources have been reported using a cut-off of 75 g/t Ag, and tailings material have been reported using a cut-of of 50 g/t Ag. The Mineral Resource Estimate has an effective date of December 13, 2021 (Table 14.12).

The author of this Technical Report section independently calculated and confirmed the volumetrics totals from the supplied Mineral Resource block model.

Highlights of the Mineral Resource Estimate include:

- Measured and Indicated Mineral Resources of 9.8 million tonnes containing 96 million Ag ounces with an average grade of 306 g/t; and
- Inferred Mineral Resources of 542 thousand tonnes containing 6.4 million Ag ounces with an average grade of 367 g/t.

Table 14.12 - Updated Mineral Resource Estimate (1-13)

Area	Class	Cut-Off	Tonnes	Ag	Ag
		(Ag g/t)	(k)	(g/t)	(koz)
Pit Constrained	Measured	65	108	477	1,656
	Indicated	65	406	325	4,242
	Measured +Indicated	65	514	357	5,898
	Inferred	-	-	-	-
Out-of-Pit	Measured	75	3,403	343	37,527
	Indicated	75	5,576	289	51,810
	Measured +Indicated	75	8,979	309	89,337
	Inferred	75	542	367	6,395
Tailings	Measured	-	-	-	-
	Indicated	50	272	94	817
	Measured +Indicated	50	272	94	817
	Inferred	-	-	-	-
Total	Measured	-	3,511	347	39,183
	Indicated	-	6,254	283	56,869
	Measured +Indicated	**-**	**9,765**	**306**	**96,052**
	Inferred	-	542	367	6,395

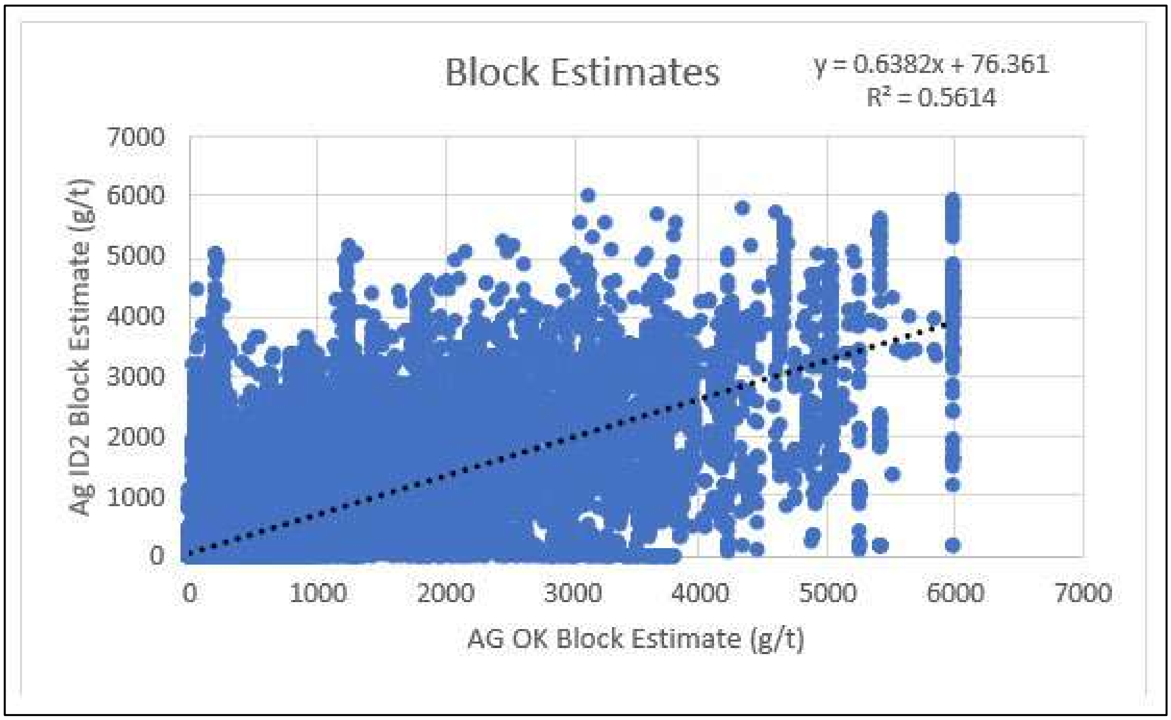
1. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues. There is no certainty that Mineral Resources will be converted to Mineral Reserves. No additional Inferred Mineral Resources were reported for this update.
2. Mineral Resources are reported inclusive of Mineral Reserves
3. The Inferred Mineral Resource in this estimate has a lower level of confidence that that applied to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of the Inferred Mineral Resource could be upgraded to an Indicated Mineral Resource with continued exploration.
4. The Mineral Resources in this news release were estimated in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions (2014) and Best Practices Guidelines (2019) prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council.
5. A silver price of US$22.5/oz with a process recovery of 90%, US$20/t rock process cost, US$16.5/t tailings process cost and US$7/t G&A cost were used.
6. The constraining pit optimization parameters were US$15/t of mineralised material (including waste mining) and 50º pit slopes with a 65 g/t Ag cut-off.
7. The out-of-pit parameters used a US$22/t mining cost. The out-of-pit Mineral Resource grade blocks were quantified above the 75 g/t Ag cut-off, below the constraining pit shell and within the constraining mineralised wireframes. Out–of-pit Mineral Resources exhibit continuity and reasonable potential for extraction by the cut and fill underground mining method.
8. The tailings parameters were at a US$9/t mining cost, and Mineral Resource grade blocks were quantified above the 50 g/t Ag cut-off.
9. Individual calculations in tables and totals may not sum correctly due to rounding of original numbers.
10. Grade capping of 6,000 g/t Ag was applied to composites before grade estimation.
11. Bulk density was determined from measurements taken from drill core samples.
12. 1.2 m composites were used during grade estimation.
13. Previously mined areas of the deposit were depleted from the Mineral Resource Estimate.

14.12 Validation

The block model was validated visually by the inspection of successive section lines in order to confirm that the block models correctly reflect the distribution of high-grade and low-grade values.

The author of this Technical Report section compared the Inverse Distance Squared ("ID2") model as supplied by M. Audet to the OK model, and notes that the OK model appears to overestimate the ID2 model (Figure 14.2).

Figure 14.2 – ID2 versus OK Block Models



Source: P&E, 2022

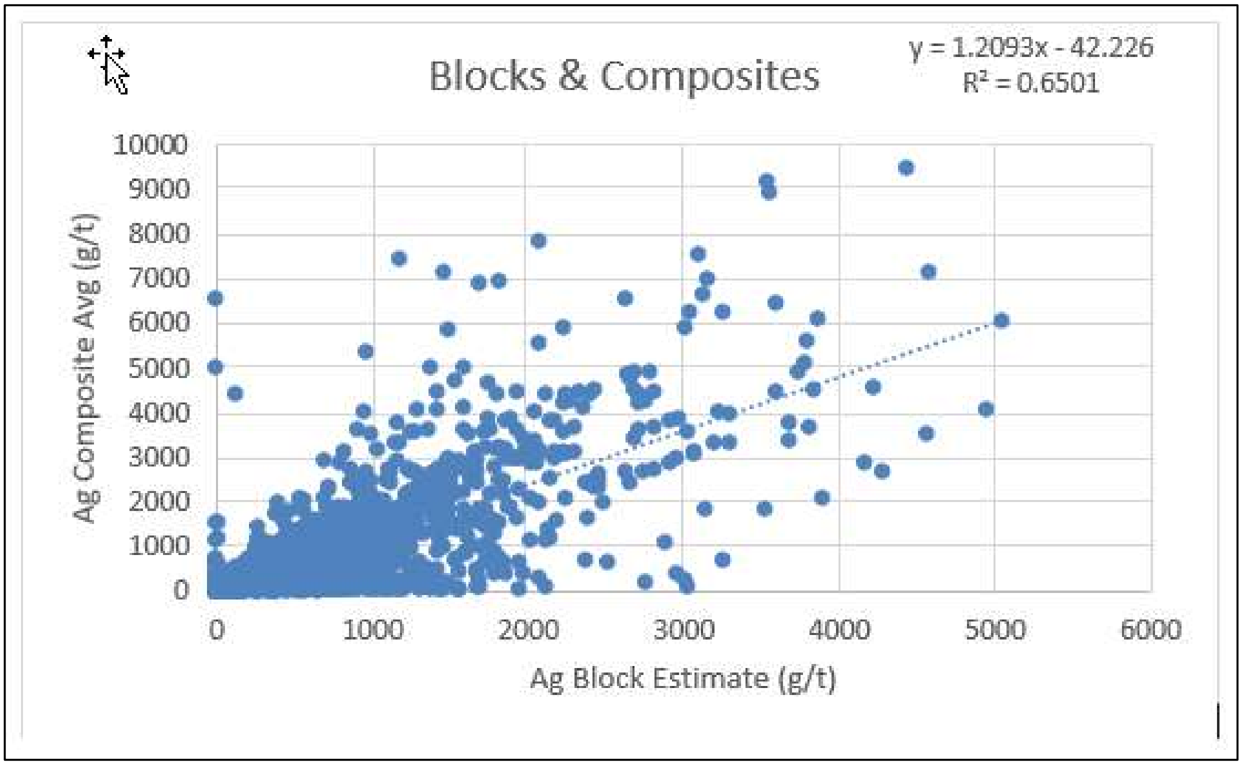
An additional validation check was completed by comparing the average grade of the composites falling within a block to the corresponding block grade estimate (Figure 14.3). The results fall within acceptable limits for linear grade estimation.

The author of this Technical Report section further calculated summary statistics for the block model estimated bulk density:

- Minimum: 2.55
- Maximum: 2.78
- Average: 2.77
- Median: 2.77

The results are comparable to the bulk density measurements derived from drill core.

Figure 14.3 – Block and Composite Grades



Source: P&E, 2022

15 MINERAL RESERVES

15.1 Introduction

The Mineral Reserves for the Zgounder mine were prepared by DRA. The Mineral Reserves have been developed using best practices in accordance with CIM Best Practices Guidelines and follow National Instrument 43-101 reporting requirements. The effective date of the Mineral Reserve estimate is December 13, 2021. The Zgounder mine is planned as a combined open pit and underground operation with an ore production rate of 2.7 ktpd.

The Mineral Reserves were derived from the Mineral Resource Block Model that was presented in Section 14. The Mineral Reserves are the Measured and Indicated Mineral Resources that have been identified as being economically extractable and which incorporate mining loses and the addition of waste dilution. The Mineral Reserves form the basis for the mine plan presented in Section 16.

15.2 Geological Information

15.2.1 GEOLOGICAL MODEL

The FS was based on Mineral Resources with an effective date of December 13, 2021, which is detailed in Section 14 of this Report. Table 15.1 presents the block model limits and block sizes for the block model used for open pit and underground mining and historical tailings.

Table 15.1 – Block Model Limits

Direction	Minimum (m)	Maximum (m)	Block Size (m)	Block Count
Open Pit and Underground Block Model				
X	275,632	276,992	2	680
Y	420,086	420,770	2	342
Z	1,598	2,300	2	351
Historical Tailings Block Model				
X	275,400	275,650	25	10
Y	420,625	420,825	25	8
Z	2,055	2,100	5	9

The following items, presented in Table 15.2, from the 3D block model were imported in HxGn Mine plan for the open pit and historical tailings mineral reserves and in Deswik Cad for the underground mineral reserves.

Table 15.2 – Block Model Items

Item	Description
ROCK	Rock Type: 10 – Void (previous underground excavations) 40 – Waste 100 – SW additional mineralised zone 210 – NW mineralised zone 220 – SW mineralised zone 230 – NE mineralised zone 240 – SE mineralised zone 250 – Mineralised zone near graphite
DENS	Density of rock; t/m^3
AG	Silver grade; g/t
AGDIL	Diluted silver grade; g/t
CU	Copper grade; %
CUDIL	Diluted copper grade; %
ZN	Zinc grade
ZNDIL	Diluted zinc grade; %
RES	Resource category: 1 – Measured 2 – Indicated 3 – Inferred
TOPO%	Percentage of block below the topography; %

15.2.2 Topographical Data

The mine design for the Feasibility Study was carried out using a lidar topographic surface dated June 2021 that was supplied to DRA by Aya.

15.2.3 Existing Underground Development

The underground mine design for the Feasibility Study was carried out using a lidar and CMS survey of underground openings dated November 19, 2021, that was supplied to DRA by Aya.

15.2.4 Material Properties

The material properties for the different rock types are outlined below. These properties are important in estimating the Mineral Reserves, the equipment fleet requirements as well as the rock pile and stockpile design capacities.

15.2.4.1 Density

Density values were included in the resource block model for open pit and underground and vary per rock type as shown in Table 15.3. A default density of 2.76 t/m^3 was used for blocks without a previously assigned density value, based on the average block density. Void blocks (Rock Type 10) were not assigned a density as they are empty volumes.

For the historical tailings, a density of 1.56 t/m^3 was used.

Table 15.3 – Block Densities for Open Pit and Underground

Rock Type	Number of Blocks	Minimum Density (t/m^3)	Maximum Density (t/m^3)	Average Density (t/m^3)
10	N/A	N/A	N/A	N/A
40	6,022,380	2.75	2.78	2.76
100	216,843	2.67	2.78	2.76
210	140,171	2.70	2.78	2.77
220	198,106	2.65	2.78	2.77
230	157,663	2.55	2.78	2.77
240	122,705	2.67	2.78	2.77
250	21,148	2.70	2.78	2.77
Overall	**6,878,814**	**2.55**	**2.78**	**2.76**

15.2.4.2 Moisture Content

Due to the dry conditions of rock in the region, a moisture content of 2% was used.

15.2.4.3 Swell Factor

A swell factor of 1.35 was used for all material, and a compaction factor of 0.92 was used for the stockpiled waste material, for an overall factor of 1.25.

15.2.4.4 Angle of Repose

The overall angles of repose used for the ore and waste stockpiles are 25° and are described in Sections 16.2.1.2 and 16.2.1.3, respectively.

15.3 Open Pit Mineral Reserves

15.3.1 Pit Optimisation

The open pit optimisation was conducted on the deposit to determine the economic pit limits. The optimisation was carried out using initial cost, sales price, and pit and plant operating parameters. The optimisation was completed in HxGN MinePlan's MSOPit module, using the Pseudoflow algorithm. The optimiser operates on a net value calculation for all the blocks in the model (i.e., revenue from silver sales minus operating costs).

Only Measured and Indicated Resources from the 3D block model have been considered in the optimisation and mine plan. Table 15.4 presents the optimisation parameters. These were developed assuming a standard open pit truck and shovel operation and a production rate of rate of 288,000 tonnes of ore per year. Process recoveries and all production costs were estimated by DRA based on metallurgical testing and detailed cost calculations.

Table 15.4 – Mineral Reserve Pit Optimisation Parameters

Description	Unit	Value
Economics		
Silver price	$/oz	20.00
Refinery Cost	$/oz	0.20
Royalties, Transport and Tax	%	3.00
Processing		
Silver recovery	%	92.00
Processing cost	$/t milled	19.36
G&A cost	$/t milled	3.55
Mining		
Ore mining cost	$/t ore	4.00
Waste mining cost	$/t waste	2.00
Mining recovery	%	95.00
Mining dilution	%	15.00
Pit slope angles	°	Average slope: 52.50°

Since the mine will run concurrently with an underground operation, a limit at Level 2000 was placed to ensure an adequate pillar remains between the bottom of the open pit operation and the top of the underground operation.

15.3.1.1 *Silver Price*

The selected silver price of $20.00 US/oz is based on Aya Gold & Silver's current price for Mineral Reserve estimation for its operations and other mining projects. This value is considered conservative as silver is currently trading at $23.60 US/oz (as of February 16, 2022) and the average three-year price (2019-2022) is $21.03 US/oz.

15.3.1.2 *Overall Slope Angles*

The slope angles used in the pit optimisation followed the recommended pit slopes as developed by DRA as shown in Table 15.5 and detailed in Section 16.2 of the Report.

Table 15.5 – Summary of Pit Slope Parameters

Domain	BFA	Max. Bench Height	Planned Berm Width	Design IRA	Max. Stack Height	Geotechnical Berm Width	Max. Overall Slope Angle
	(°)	(m)	(m)	(°)	(m)	(m)	(°)
North	75	5	5	52.5	10	12	50
South	75	5	5	52.5	10	12	50
East	80	5	5	55.9	10	12	50

15.3.1.3 *Mine Dilution and Mining Recovery*

Given the Zgounder deposit geometry, a dilution of 15% was applied to account for the waste that will be mined with the ore. In addition, a 95% recovery was applied.

15.3.1.4 *Mining Costs*

Mining costs to build economic pit limits were derived from preliminary discussions with local Moroccan contractors and verified against first principles calculation and similar operations. The final costs received from the local contractors based on the mine plan and haulage distances were lower than the initial costs therefore DRA considers the unit prices used in economic pit limits and reserve estimation conservative and appropriate. Details on the final estimated operating costs are provided in Section 21 of the Report.

15.3.1.5 *Silver Process Recovery*

The silver process recovery was based on metallurgical test work and recovery tests and described in Section 13 of the Report.

15.3.1.6 *Process Cost*

Process unit costs were estimated by DRA process engineers based on the process flowsheet for the new mill at a production rate of 2,000 t/d and 700 t/d for the existing installations. The process parameters are presented in Section 17 and operating costs in Section 21.

15.3.1.7 *G&A Costs*

G&A costs were provided by Aya on a yearly basis and converted to a cost per tonne based on a 2,700 t/d production at the new and existing mills. Details on G&A costs are provided in Section 21 of the Report.

15.3.1.8 *Payable Metal*

The payable silver bars, after refinery charges, royalties and transportation and taxes is estimated at 96% of silver value. Details are given in Section 21 of the Report.

15.3.1.9 *Cut-Off Grade*

The Open Pit cut-off grade (COG) has been calculated according to Equation 15.1. The COG is used to determine whether the material being mined will generate a profit after paying for the mining, processing and administrative costs and is the basis for defining the economic pit. The COG grade of 47.38 g/t Ag was used in the optimisation.

Equation 15.1 – Cut-Off Grade Calculation

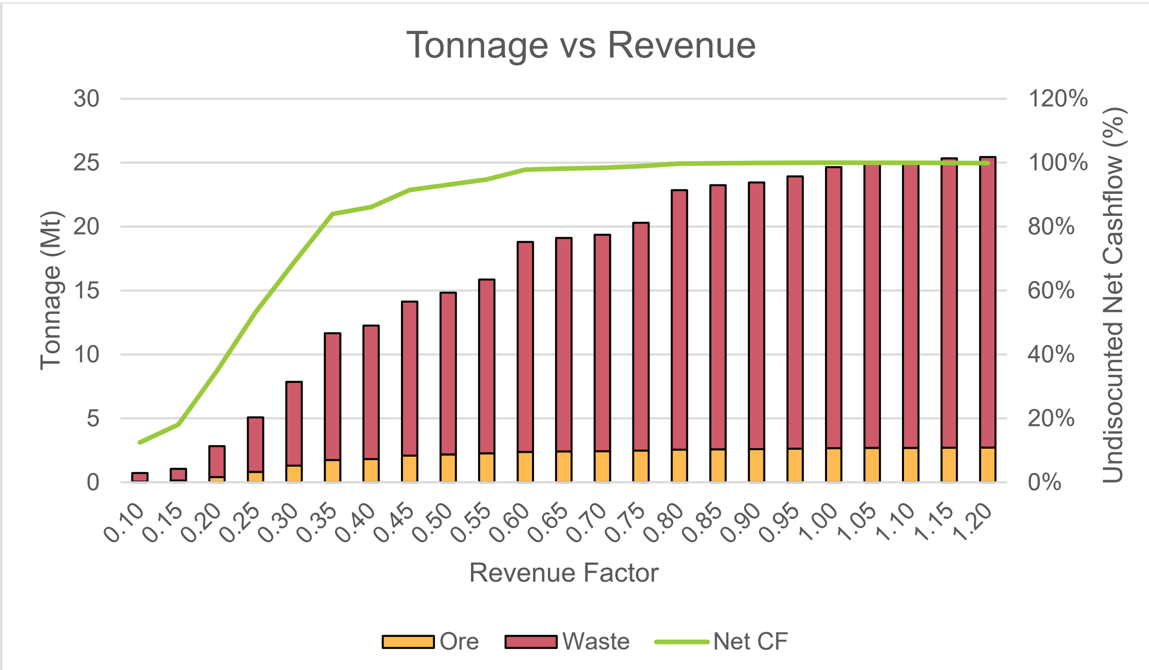
$$COG = \frac{Processing\ Cost\left(\$/_{t}\right) + G\&A\ Cost\left(\$/_{t}\right) + Mining\ Cost\left(\$/_{t}\right)}{Process\ Recovery\ (\%) \times \left(Silver\ Price\left(\$/_{oz}\right) - Refinery\ Cost\left(\$/_{oz}\right) - Royalties, Transport, Tax\left(\$/_{oz}\right)\right)}$$

15.3.1.10 *Pit Optimisation Results*

An optimal open pit mining limit was established using HxGN MinePlan's MSOPit module, with the Pseudoflow algorithm and the parameters listed in Table 15.4. When varying silver prices from $2.00 US/oz to $24.00 US/oz, as shown in Figure 15.1. the undiscounted net cashflow increases until reaching a maximum at the revenue factor 1.00 pit ($20.00 US/oz). From this point, the cashflow decreases slightly as the costs exceed the revenue.

The revenue factor 0.80 pit ($16.00 US/oz) was selected as the optimised pit since it contains 96% of the total ore and 98% of the available revenue. This pit shell contains 2.6 Mt of ore and 20.3 Mt of waste, for a strip ratio of 7.9 to 1 and a mine life of 32 months.

Figure 15.1 – Pit Optimisation Results



Source: DRA, 2022

15.3.2 Pit Design

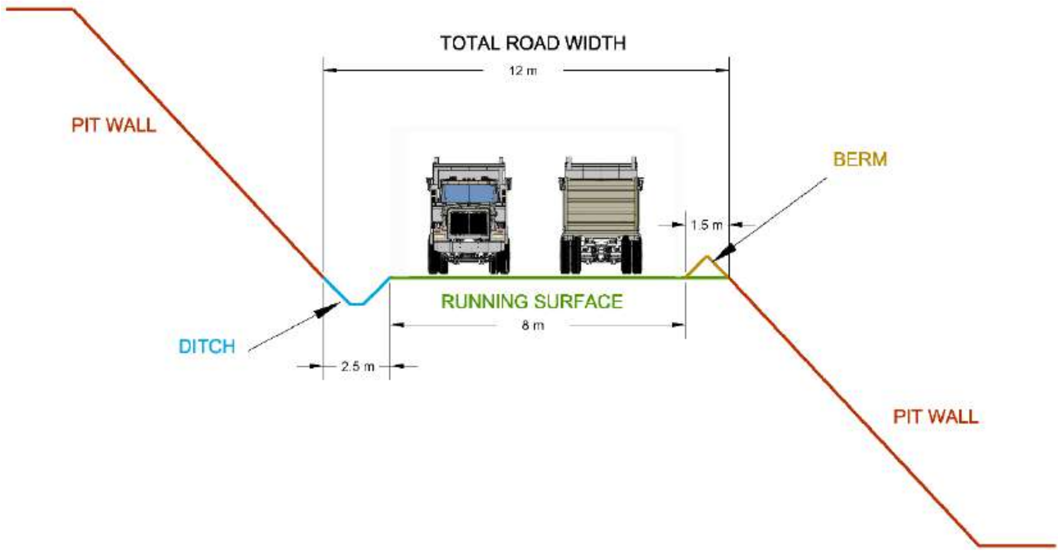
The next step in the Mineral Reserves estimation process is to design an operational pit that will form the basis of the production plan. This pit design uses the selected economic pit shell as a guideline and includes smoothing the pit walls, adding ramps to access the pit bottom and ensuring that the pit can be mined using the selected equipment. The pit designs were designed in HxGN MinePlan based on the 3D Mineral Resource block model and the revenue factor 0.80 Pit (Section 15.3.1.10). The following section provides the parameters that were used for the open pit design and presents the results. The pit was designed assuming 5m high benches in ore and waste and 40 t 8x6 type trucks and matching excavators.

15.3.2.1 Haul Road Design

The ramps and haul roads were designed with an overall width of 12 m. For double lane traffic, industry practice dictates that the running surface width be a minimum of 3 times the width of the largest truck. The overall width of a 8x6 type truck is 2.5 m which results in a running surface of approximately 7.5 m. The allowance for berms and ditches increases the overall haul road width to 12 m. In certain areas of the pit where low traffic is expected, the width of the roads was reduced to 8 m for single-lane traffic.

A maximum ramp grade of 10% was considered. Figure 15.2 presents a typical section of the in-pit haul road or ramp design.

Figure 15.2 – Haul Road Design



Source: DRA, 2022

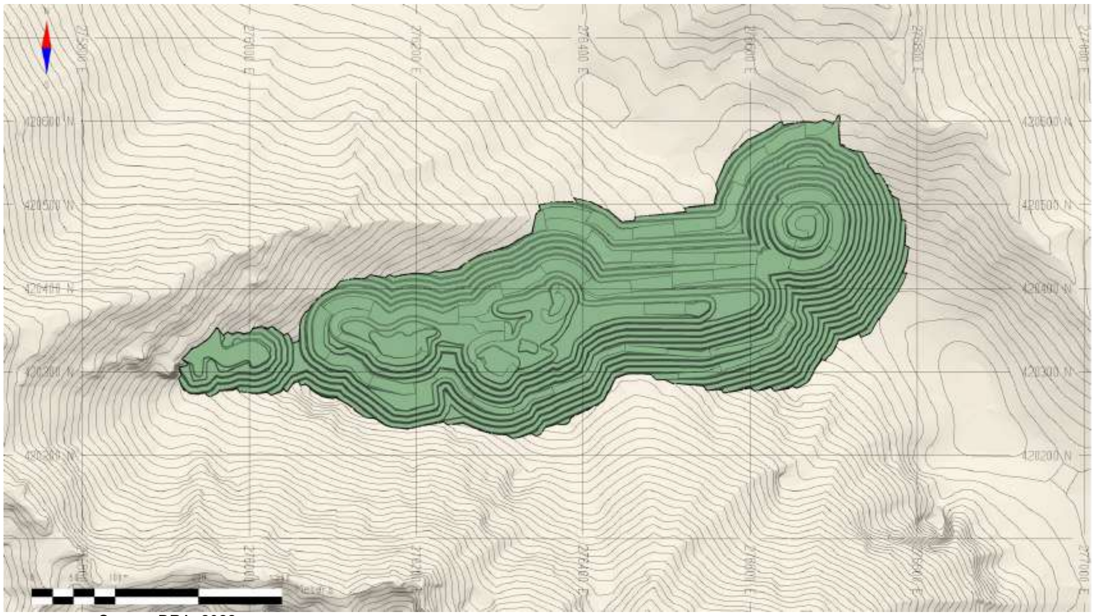
15.3.2.2 Pit Slopes

The pit designs followed the recommended geotechnical pit slopes as described in Section 16.2.2 of the Report.

15.3.2.3 Open Pit Design Results

Figure 15.3 shows the open pit mine design for the Zgounder pit. This pit was used for the estimation of the Mineral Reserves. The final pit design delineates 25.6 Mt of combined ore and waste, compared to 22.9 Mt in the pit shell. The difference is a 15% decrease in ore tonnage and a 26% increase in waste tonnage.

Figure 15.3 – Zgounder Pit Design



Source: DRA, 2022

15.3.3 Open Pit Mineral Reserves

Table 15.6 – Open Pit Mineral Reserves

Description	Classification	Tonnage (Mt)	Ag Grade (g/t)	In-Situ Ag (Moz)
Open Pit	Proven	0.6	312	5.7
	Probable	1.6	233	12.1
	Total Open Pit Reserves	2.2	253	17.8
	Waste	23.4		
	Strip Ratio	10.6		

NOTES:
1. The Mineral Reserve is estimated in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions (2014) and Best Practices Guidelines (2019) prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council.
2. The Open Pit Mineral Reserve is estimated with a COG of 47.38 g/t
3. Ag content (oz) are estimated as in-situ
4. An ONHYM royalty of 3% is included in the Mineral Reserve Estimate.
5. The Mineral Reserve is estimated with a mining recovery of 95%.
6. The Mineral Reserve includes both internal and external dilution estimated at 15%.
8. The economic viability of the Mineral Reserve has been demonstrated.
9. For the Open-pit Reserves Estimate, a silver price of US$20/oz with a process recovery of 92%, a process cost of US$22.91/t (including G&A), and a mining cost of $4.00/t (including haulage) were used.
10. The reserves estimate has an effective date of December 13, 2021.
11. Totals may not add due to rounding.

15.4 Historical Tailings Mineral Reserves

15.4.1 INTRODUCTION

There is an existing tailings facility from a previous mining operation in the vicinity of the Zgounder mine that contains residual silver that was drilled, modelled, tested and converted to Mineral Resources as described in Sections 13 and 14 of the Report. These historical tailings were converted to Mineral Reserves using the parameters listed in Table 15.7. The parameters are based on those used for the open pit optimisation (Table 15.4) but adjusted to account for the lack of drilling and blasting, different haulage times and a reduction in crushing and griding requirements.

Table 15.7 – Mineral Reserves Historical Tailings Parameters

Description	Unit	Value
Economics		
Silver price	$ US/oz	20.00
Refinery Cost	$ US/oz	0.20
Royalties, Transport and Tax	%	3.00
Processing		
Silver recovery	%	92.00
Processing cost	$ US/t milled	17.38
G&A cost	$ US/t milled	3.55
Mining		
Ore mining cost	$ US/t ore	4.31
Mining recovery	%	90.00

15.4.1.1 Silver Price

The selected silver price of $20.00 US/oz is based on Aya's current price for Mineral Reserve estimation for its operations and other mining projects. This value is considered conservative as silver is currently trading at $23.60 US/oz (as of February 16, 2022) and the average three-year price (2019-2022) is $21.03 US/oz.

15.4.1.2 Overall Slope Angles

No overall slope angle was calculated for the historical tailings as they are relatively shallow and will be excavated in a bench-by-bench manner.

15.4.1.3 Mine Dilution and Mining Recovery

The historical tailings account a dilution of approximately 17% and a mining recovery of 90%.

15.4.1.4 Mining Costs

Mining costs were adjusted from local Moroccan contractors for the open pit portion of the mine and verified first principles calculations based on similar operations. The contractor costs were adjusted to account for the lack of drilling and blasting, and the difference in haulage distances. Details on the operating costs are provided in Section 21 of the Report.

15.4.1.5 Silver Process Recovery

The silver process recovery was based on metallurgical test work and recovery tests and described in Section 13 of the Report.

15.4.1.6 Process Cost

Process unit costs were estimated by DRA process engineers based on the process flowsheet for the new mill at a production rate of 2,000 t/d and the existing mill at 700 t/d. The process parameters are presented in Section 17 and operating costs in Section 21.

15.4.1.7 G&A Costs

G&A costs were provided by Aya on a yearly basis and converted to a cost per tonne based on a 2,700 t/d production at the new and existing mills. Details on G&A costs are provided in Section 21 of the Report.

15.4.1.8 Payable Metal

The payable silver bars, after refinery charges, royalties and transportation and taxes is estimated at 96% of silver value. Details are given in Section 21 of the Report.

15.4.2 HISTORICAL TAILINGS CUT-OFF GRADE

The cut-off grade (COG) has been calculated according to Equation 15.2. The COG is used to determine whether the material being mined will generate a profit after paying for the mining, processing and administrative costs and is the basis for converting the tailings Mineral Resources to Mineral Reserves. A COG grade of 44.43 g/t was used for the conversion to Mineral Reserves.

Equation 15.2 – Cut-Off Grade Equation

$$COG = \frac{Processing\ Cost\ \left(\$/_t\right) + G\&A\ Cost\ \left(\$/_t\right) + Mining\ Cost\ \left(\$/_t\right)}{Process\ Recovery\ (\%) \times \left(Silver\ Price\ \left(\$/_{oz}\right) - Refinery\ Cost\ \left(\$/_{oz}\right) - Royalties, Transport, Tax\ \left(\$/_{oz}\right)\right)}$$

15.4.3 Historical Tailings Mineral Reserves

Table 15.8 – Historical Tailings Mineral Reserves

Description	Classification	Tonnage	Ag Grade	In-Situ Ag
		(Mt)	(g/t)	(Moz)
Tailings	Probable	0.3	77	0.8

NOTES:
1. The Mineral Reserve is estimated in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions (2014) and Best Practices Guidelines (2019) prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council.
2. The Historical Tailings Mineral Reserve is estimated with a COG of 44.43 g/t
3. Ag content (oz) are estimated as in-situ
4. An ONHYM royalty of 3% is included in the Mineral Reserve Estimate.
5. The Mineral Reserve is estimated with a mining recovery of 90%.
6. The Mineral Reserve includes both internal and external dilution estimated at 17%.
8. The economic viability of the Mineral Reserve has been demonstrated.
9. For the Historical Tailings Reserves Estimate, a silver price of US$20/oz with a process recovery of 92%, a process cost of $20.93/t (including G&A), and a mining cost of $4.31/t (including haulage) were used.
10. The reserves estimate has an effective date of December 13, 2021.
11. Totals may not add due to rounding

15.5 Underground Mineral Reserves

15.5.1 Underground Parameters

DRA has designed an underground mine to produce 1,870 t/d of silver ore from the Zgounder deposit. The mine will be accessed by from a new portal at the 2000 Level from where a ramp will be driven to reach the upper and lower levels. The deposit will be mined using conventional mechanized transverse long-hole, longitudinal long-hole and cut-and-fill mining methods. Ore will be trucked to the concentrator via a 3 km surface road using the underground trucks. The parameters used for the estimation of the Underground mineral reserves is given in Table 15.9 below.

15.5.1.1 Stope Optimizer Parameters

The stope has been designed using the Deswik Stope Optimizer. The stope sizes and the mining methods has been mainly determined by the geometry of the deposit and to produce high grade ore with minimal dilutions. Therefore, the maximum opening sizes the stopes are small with the maximum opening is well under the geotechnical recommendations. Detail parameters of the stope design are in Section 16.4.3 of the Report.

Table 15.9 – Zgounder Underground Mineral Reserves Parameters

Description	Unit	Value
Economics		
Silver price	$/oz	20.00
Refinery Cost	$/oz	0.20
Royalties, Transport and Tax	%	3.00
Processing		
Silver recovery	%	92.00
Processing cost	$/t milled	19.36
G&A cost	$/t milled	3.55
Mining		
Drift-and-Fill cost	$/t mined	24.00
Long-Hole cost	$/t mined	18.03
Mining recovery (average)	%	95.00
Mining dilution	%	variable

15.5.1.2 *Mine Dilution*

Dilution is the material (ore, waste, or backfill) that breaks off from stope and drift walls, backs and walls and which is inherent to underground mining. The dilution was calculated as per the formula and parameters below.

$$\%\ Dilution = \frac{Stope\ volume + dilution\ shell}{Stope\ volume}$$

Longitudinal Long-hole:

- Near and Far walls dilution (North and South, included in SO parameters): 0.1 m.
- Unsupported stope back dilution: 0.1 m (top access drive)
- Bottom level floor dilution: 0.3 m.
- Ore/Waste contact limit dilution (East and West): 0.3 m.
- Internal dilution from backfilled stopes: 0.3 m (each side).

Transverses Long-hole primary stopes:

- Near and Far walls dilution (North and South, included in SO parameters): 0.3 m
- Unsupported stope back dilution: 0.3 m.
- Bottom level floor dilution: 0.3 m.
- Ore/Waste contact limit dilution (East and West): 0.3 m.
- Internal dilution from secondary stopes: 0 m.

Transverses Long-hole secondary stopes:

- Near and Far walls dilution (North and South, included in SO parameters): 0.3 m.
- Hanging wall and footwall dilution: 0.3 m.
- Unsupported stope back dilution: 0.3 m.
- Bottom level floor dilution: 0.3 m.
- Ore/Waste contact limit dilution (East and West): 0.3 m.
- Internal dilution from primary backfilled stopes: 0.3 m (each side).

Drift-and-Fill:

- Near and far walls and dilution (North and South, included in SO parameters): 0.1 m.
- Unsupported stope back dilution: 0.1 m.
- Bottom level floor dilution: 0.1 m.
- Ore/Waste contact limit dilution (East and West): 0.1 m.
- Internal dilution from primary backfilled stopes: 0.1 m (each side).

15.5.1.3 Ore Recovery

DRA assessed the stope recovery factor based on the geometry of the deposit and the stopes and the mining methods. The overall ore recovery for the cut-and-fill and the 12 m high long-hole-stope is estimated at 95% assuming a geological loss of 1.5% and a mining loss of 3.5%.

15.5.1.4 Mining Costs

The mining cost for each mining methods have been calculated on the principal that Aya will self-perform the production of the mine and includes direct labour, equipment operation, permanent material, consumables and indirect. The mining cost to define the drift-and-fill and the long-hole stopes were estimated at respectively $ 24.00 per tonne and $ 18.03 per tonne. Details of the final underground operating costs are given in Section 21 of the Report.

15.5.1.5 Silver Process Recovery

The silver process recovery was based on metallurgical test work and recovery tests and described in Section 13 of the Report.

15.5.1.6 Process Cost

Process unit costs were estimated by DRA process engineers based on the process flowsheet for the new mill at a production rate of 2,000 t/d and 700 t/d for the existing mill. The process parameters are presented in Section 17 and operating costs in Section 21.

15.5.1.7 G&A Costs

G&A costs were provided by Aya on a yearly basis and converted to a cost per tonne based on a 2,700 t/d production at the new and existing mills. Details on G&A costs are provided in Section 21 of the Report.

15.5.1.8 Payable Metal

The payable silver bars, after refinery charges, royalties and transportation and taxes is estimated at 96% of silver value. Details are given in Section 21 of the Report.

15.5.2 CUT-OFF GRADE

The COG value was estimated for each mining methods with the following results:

- Drift-and-fill: 85 g/t;
- Long-hole Longitudinal: 74 g/t;
- Long-hole Transverses: 70 g/t.

The overall mineral reserves economic COG for Zgounder underground reserves was selected at 85 g/t.

15.5.3 UNDERGROUND MINERAL RESERVES

The mineral underground reserves for Zgounder are estimated at 6,093,425 tonnes of recoverable and diluted ore grading 267 g/t Ag using a cut-off grade of 85 g/t Ag. The mineral underground reserves are comprised of 42% in proven category (2,532,973 tonnes grading 283 g/t Ag) and 58% in probable category (3,560,452 grading 256 g/t Ag). Reserves are inclusive of dilution and ore loss.

The mineral reserves are based on the Production Schedule presented in Section 16.7. The mineral reserves are the stope and development tonnage (including material above the cut-off and internal low-grade dilution) to which dilution and recovery factors are added.

Details of the mineral reserves are given in Table 15.10.

Table 15.10 – Zgounder Underground Mineral Reserves Estimate

Category	Tonnage (Mt)	Ag Grade (g/t)	In-Situ Ag (Moz)
Proven	2.5	283	23.0
Probable	3.6	256	29.3
Proven and Probable	**6.1**	**267**	**52.3**

Notes:

1. The Mineral Reserve is estimated in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions (2014) and Best Practices Guidelines (2019) prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council.
2. The Underground Mineral Reserve is estimated with a variable COG which was calculated by mining method.
3. Ag content (oz) are estimated as in-situ
4. An ONHYM royalty of 3% is included in the Mineral Reserve Estimate.
5. The Mineral Reserve is estimated with a mining recovery of 95%.
6. The Mineral Reserve includes both internal and external dilution. The external dilution included a mining dilution of 0.3 m width on the hanging wall and footwall for the long-hole mining method and a 0.1 m width on the hanging wall and footwall for the cut-and-fill mining methods.
7. A minimum mining width of 4m was used for the long-hole and cut-and-fill mining methods.
8. The economic viability of the Mineral Reserve has been demonstrated.
9. For the Underground Reserves Estimate, a silver price of $20/oz with a process recovery of 92%, a process cost of US$22.91/t (including G&A), and a mining cost of $24.13/t (including haulage and backfill) were used for the combined cut-and-fill and long-hole methods.
10. The reserves estimate has an effective date of December 13, 2021.
11. Totals may not add due to rounding.

15.6 Combined Mineral Reserves

The Mineral Reserves for the Project are estimated at 2.2 Mt of open pit ore at a grade of 253.5 g/t Ag, 0.3 Mt of tailings ore at a grade of 76.6 g/t Ag, and 6.1 Mt of underground ore at a grade of 267.25 g/t Ag, for a total of 8.6 Mt of ore at a grade of 256.68 g/t Ag. Table 15.11 presents combined open pit, historical tailings, and underground Mineral Reserves of the Project.

Table 15.11 – Mineral Reserves Estimate – Effective December 13, 2021

Description	Classification	Tonnage	Ag Grade	In-Situ Ag
		(Mt)	(g/t)	(Moz)
Open Pit	Proven Reserves	0.6	312	5.7
	Probable Reserves	1.6	233	12.1
	Total Open Pit Reserves	*2.2*	*253*	*17.8*
Historical Tailings	Probable Reserves	0.3	77	0.8
Underground	Proven Reserves	2.5	283	23.0
	Probable reserves	3.6	256	29.3
	Total Underground Reserves	*6.1*	*267*	*52.3*
Total	Proven Reserves	3.1	288	28.7
	Probable Reserves	5.5	239	42.1
	Total Reserves	8.6	257	70.9

NOTES:
1. The Mineral Reserve is estimated in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions (2014) and Best Practices Guidelines (2019) prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council.
2. The Mineral Reserve is estimated with a variable COG which was calculated by mining method.
3. Ag content (oz) are estimated as in-situ
4. An ONHYM royalty of 3% is included in the Mineral Reserve Estimate.
5. The Mineral Reserve is estimated with a mining recovery of 95%.
6. The Mineral Reserve includes both internal and external dilution. The external dilution included a mining dilution of 0.3 m width on the hanging wall and footwall for the long-hole mining method and a 0.1 m width on the hanging wall and footwall for the cut-and-fill mining methods.
7. A minimum mining width of 4m was used for the long-hole and cut-and-fill mining methods.
8. The economic viability of the Mineral Reserve has been demonstrated.
9. For the historical tailings Reserves Estimate, a silver price of US$20/oz with a process recovery of 92%, a process cost of $20.93/t (including G&A), and a mining cost of $4.31/t (including haulage) were used.
10. For the Open-pit Reserves Estimate, a silver price of US$20/oz with a process recovery of 92%, a process cost of US$22.91/t (including G&A), and a mining cost of $4.00/t (including haulage) were used.
11. For the Underground Reserves Estimate, a silver price of $20/oz with a process recovery of 92%, a process cost of US$22.91/t (including G&A), and a mining cost of $24.13/t (including haulage and backfill) were used for the combined cut-and-fill and long-hole methods.
12. The reserves estimate has an effective date of December 13, 2021.
13. Totals may not add due to rounding.

16 MINING METHODS

16.1 Introduction

The Zgounder Project will be mined in a combination of open pit mining, reclamation of historical tailings, and underground mining.

- The mining method selected for the open pit portion of the Zgounder Project is a conventional open pit, drill & blast, truck and shovel operation. The open pit consists of a single pit.
- The tailings reclamation will be mined using a loader and truck operation; no drilling and blasting will be required.
- The underground mine will be mined using a combination of drift-and-fill mining and long-hole stoping.

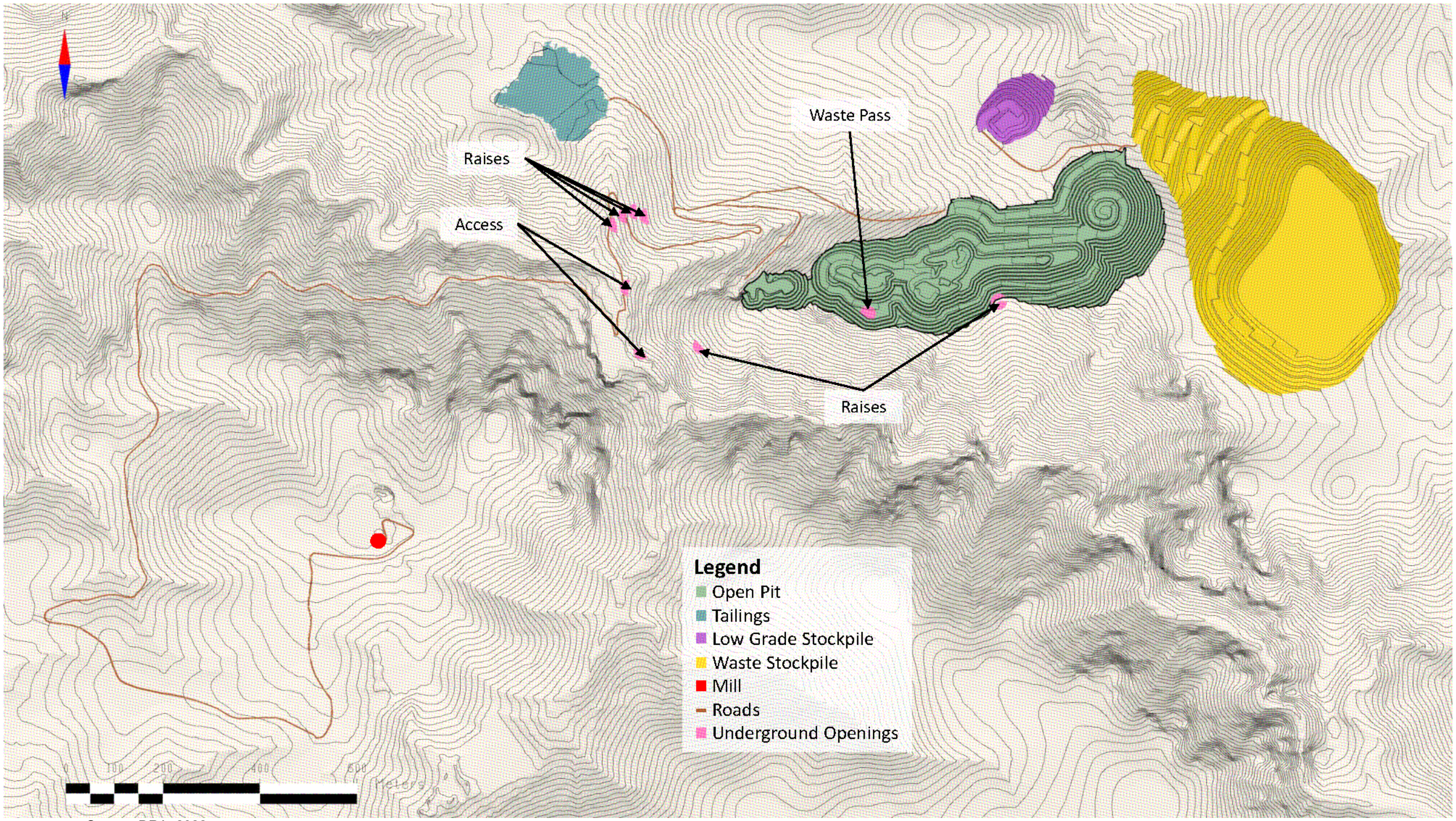
For both the open pit and the tailings reclamation, the ore will be transported by truck to the ore pass, where the material will be brought underground to be processed with the underground ore. Waste material will be deposited in a waste stockpile located near the pit. Ore will also be stockpiled in low-grade and high-grade stockpiles near the pit for future rehandling and processing.

All mining open pit and historical tailings reclamation will be undertaken by mining contractors, who will be responsible for all drilling, blasting, loading, transportation for both the ore and waste material, as well as any pre-split drilling requirements. The development of the underground mine will be undertaken by mining contractors while ore production mining will be undertaken by Aya staff.

The mine will operate year-round, seven (7) days per week, twenty-four (24) hours per day (two (2) 12-hour shifts). Two weeks of adverse weather conditions per year are considered in the mine plan.

Figure 16.1 depicts the Mine Project Layout.

Figure 16.1 – Zgounder Project Layout



Source: DRA, 2022

16.2 Open Pit

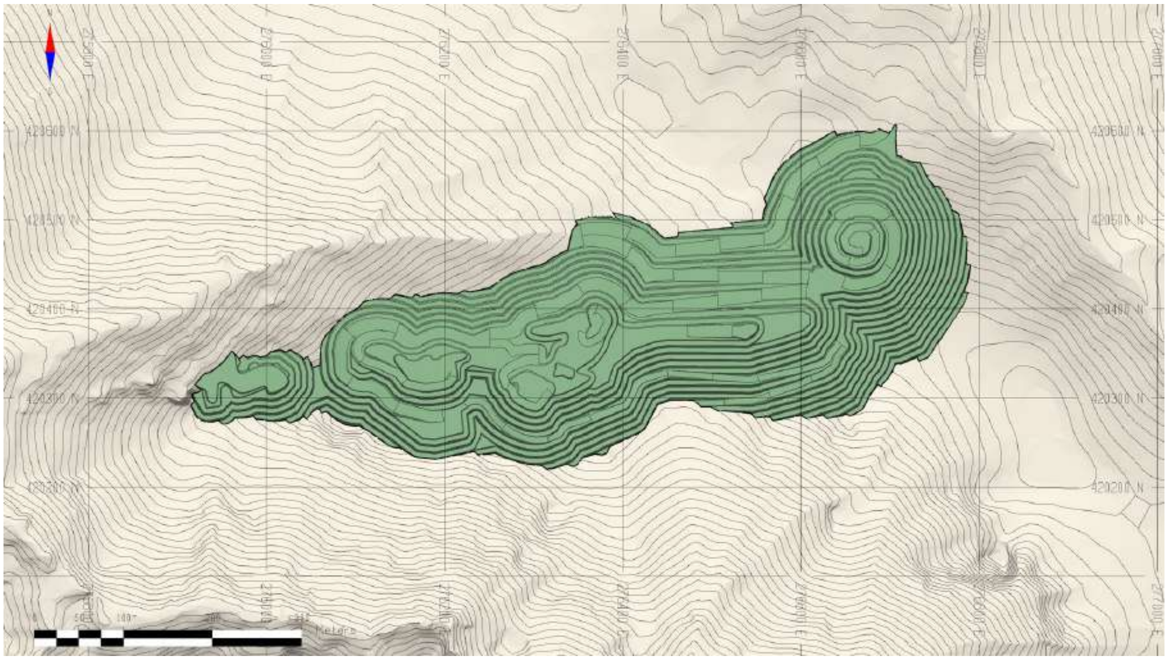
16.2.1 Mine Design

Mineral Reserves were estimated for the Zgounder open pit based on the economic and pit design parameters detailed in Section 15. The total tonnage to be mined from the pit is estimated at 25.6 Mt, ore and waste combined. This material will be mined over an 8-year period (including pre-production stripping) and in conjunction with the tailings reclamation and underground mine.

16.2.1.1 Pit Design

The final pit designed for the Zgounder Project follows the recommended geotechnical parameters defined in Section 16.2.2. The final pit design is presented in Figure 16.2.

Figure 16.2 – Zgounder Final Pit Design



Source: DRA, 2022

16.2.1.2 Ore Stockpile Design

Low grade material will be stockpiled near the pit to be rehandled and processed later. The stockpile was designed with the following parameters:

- Lift height: 10 m;
- Face angle: 30°;

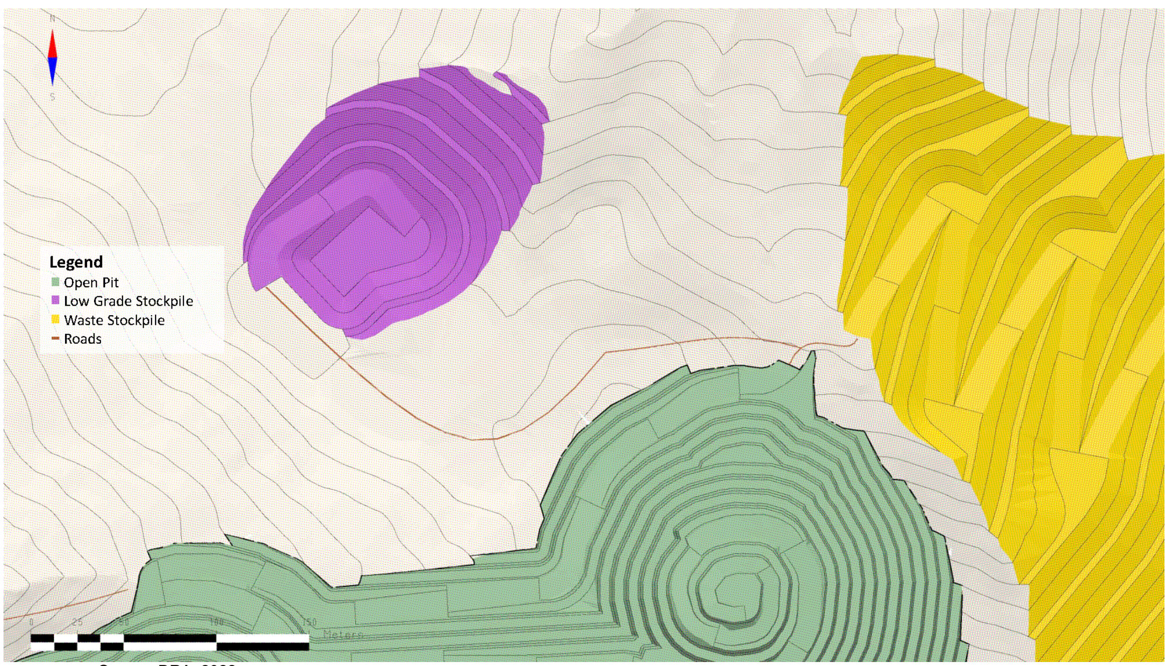
- Berm width: 3 m;
- Overall slope: 25°;
- Swell factor: 1.30;
- Compaction factor: 0.93.

The low grade stockpile's capacity is presented in Table 16.1 and its location relative to the pit is presented in Figure 16.3. High grade material can be stockpiled near the mill.

Table 16.1 – Ore Stockpile Capacity

Description	Tonnage (kt)	Volume (km^3)
Low Grade	259.4	136.6

Figure 16.3 – Zgounder Ore Stockpile Location



Source: DRA, 2022

16.2.1.3 *Waste Stockpile Design*

Waste material mined will be stored in a waste stockpile located Northeast of the pit. The stockpile was designed with the following parameters:

- Lift height: 10 m;
- Face angle: 30°;

- Berm width: 3 m;
- Overall slope: 25°;
- Swell factor: 1.30;
- Compaction factor: 0.93.

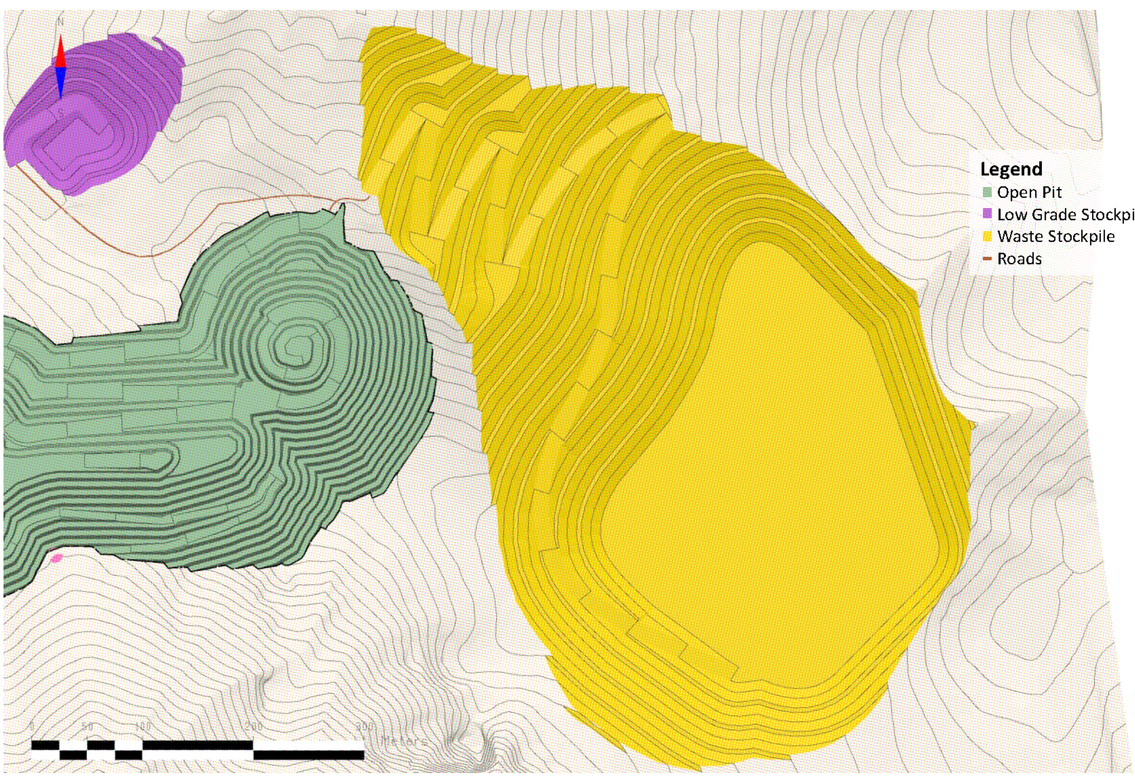
The waste stockpile's capacity is presented in Table 16.2 and its location, relative to the pit, is presented in Figure 16.4. A portion of the open pit waste (2 Mm^3) will be sent underground for backfilling purposes via waste passes close to the open pit edge.

At the time of this FS, the waste rock pile is outside of the surface water balance boundary and will have to be integrated at a later stage of the Project.

Table 16.2 – Waste Stockpile Capacity

Description	Unit	Value
Tonnage	Mt	21.0
Volume	Mm^3	8.8

Figure 16.4 – Zgounder Waste Stockpile Location



Source: DRA, 2022

16.2.2 Geotechnical

This section is based on information provided by Aya for open pit as well as a geotechnical report entitled "Geotechnical Analyses for Zgounder Feasibility Study" prepared by RockEng, Draft Version and issued March 2, 2022 for the underground.

16.2.3 Pit Slope Angles

16.2.3.1 Overview

DRA provided open pit slope design parameters for the proposed open pit portion of the Project. The design parameters are based on geotechnical site investigations, available local and regional geological data, and well-established geotechnical design methods used to estimate the Project design pit slope angles. Due to the 2020-2021 Health and Safety emergency preventing international travel, all geotechnical work was performed by Aya personnel supervised online by RockEng and DRA personnel.

A total of eleven (11) HQ-sized triple-tube oriented geotechnical drill holes, for a total of 1,975 m, were drilled, logged, and sampled to collect geotechnical information for the planned open pit and underground operation.

DRA identified geotechnical rock mass units associated with the primary rock and alteration types, based on the results of the on-site investigation by Aya personnel. Major geological structures (faults and foliation) have been included in the geotechnical slope stability analyses for the pit. Slope stability analyses were conducted using industry standard limit-equilibrium methodology.

16.2.3.2 Pit Design

A multi-component site investigation program was completed to provide data for the open pit and underground design work. A total of approximately 1,975 m of oriented geotechnical drilling was completed, distributed over eleven (11) core holes. Aya logged the geotechnical holes.

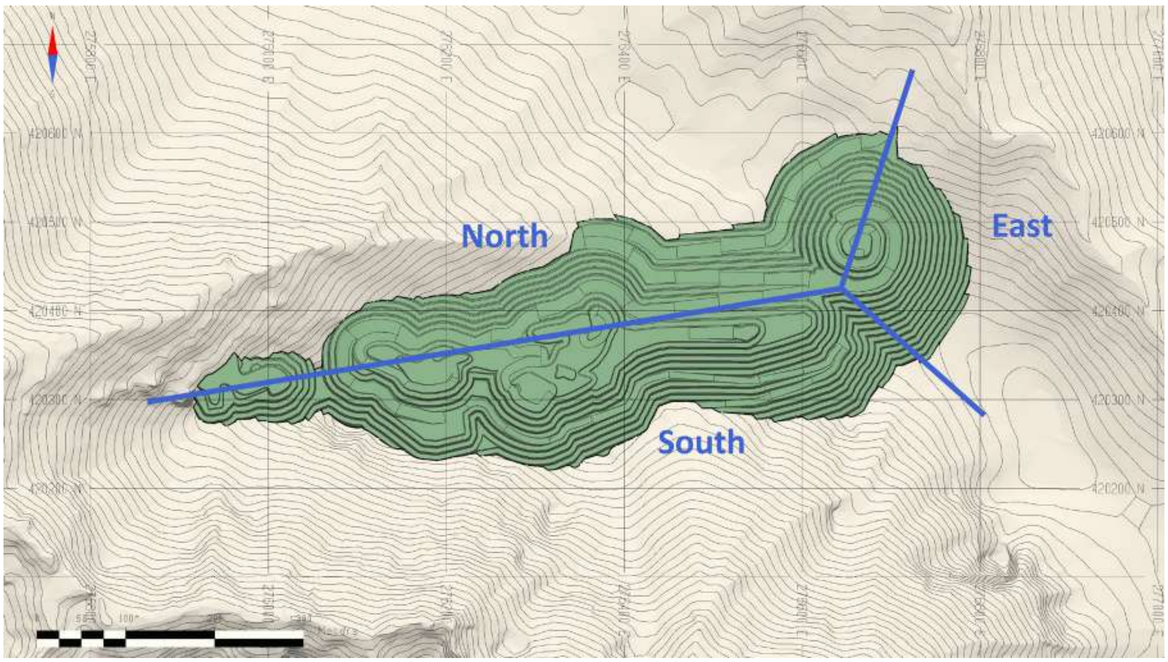
A laboratory testing program was completed, consisting of the following tests for the three (3) main lithologies:

- Uniaxial compressive strength (24 tests);
- Triaxial (15 tests);
- Brazilian tensile strength (11 tests).

A sufficient quantity of quality data was collected to characterise the geological units of the study area and support FS-level slope designs.

The proposed open pit was divided into three (3) geotechnical domains based on wall orientation and the different structural geology fabrics in the area. Discontinuity sets and geotechnical units for each domain were identified for use in the slope designs. Design sectors were based on the anticipated main orientations of the proposed pit walls. The domains are presented in Figure 16.5.

Figure 16.5 – Plan View of the Ultimate Pit Shell Showing the Section Lines and Design Domains



Source: DRA, 2022

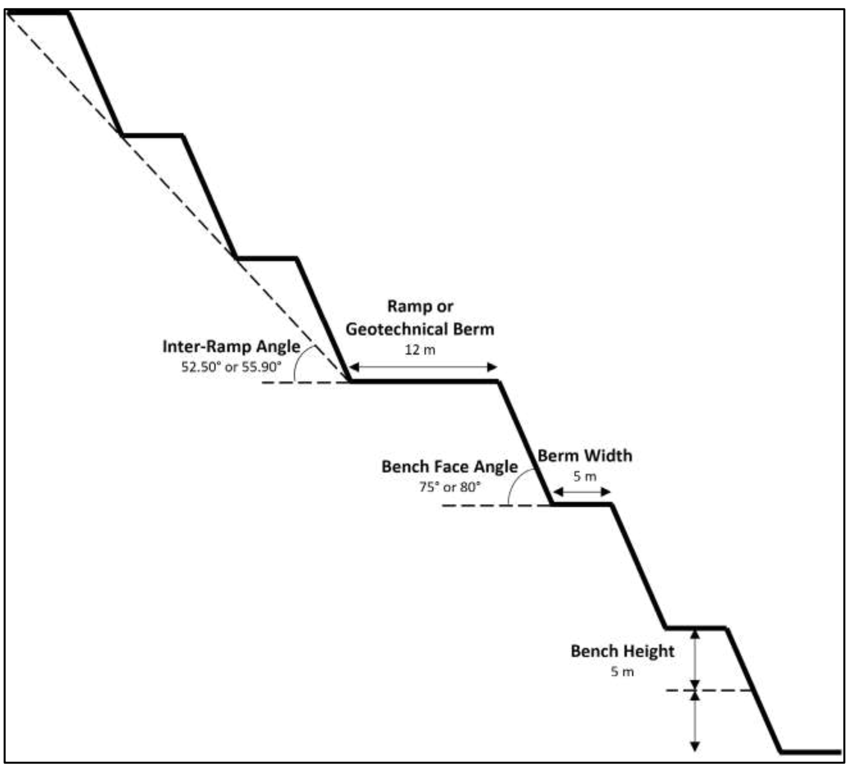
The recommended inter-ramp slope angles vary from 52.5° to 55.9° based on wall orientation and wall height. Inter-ramp slope heights are limited to 130 m, after which a geotechnical berm (or ramp) with a minimum width of 12 m is required. The inter-ramp height limits and geotechnical berms provide flexibility in the mine plan to mitigate potential slope instability, access for slope monitoring installations, and working space for potential future in-pit wells, drains and other water management infrastructure. All final pit slopes are assumed to be excavated using controlled blasting.

The open pit slope designs are outlined in Table 16.3 and Figure 16.6.

Table 16.3 – Zgounder Pit Slope Design Parameters

Domain	BFA	Max. Bench Height	Planned Berm Width	Design IRA	Max. Stack Height	Geotechnical Berm Width	Max. Overall Slope Angle
	(°)	(m)	(m)	(°)	(m)	(m)	(°)
North	75	5	5	52.5	10	12	50
South	75	5	5	52.5	10	12	50
East	80	5	5	55.9	10	12	50

Figure 16.6 – Typical Open Pit Wall Design



Source: DRA, 2022

16.2.4 Pit Dewatering

Pit dewatering is not expected to be required at the Zgounder Project. In the case of rare rainfall, any water gathered at the bottom of the pit will pumped out using portable electric pumps.

16.2.5 Mine Plan

A mine plan (or schedule) was prepared to estimate a probable production scenario for the Project and assess the capital and operating costs for the Project's financial model. The mine plan was based on an ore production rate of 961 kt/a to the mill, combining the open pit, the tailings, and the underground mining.

Mine planning was performed using HxGN MinePlan's MPSO module based on the final pit design. Due to the small size of the pit (2.3 years of ore production at the full production rate), it was not separated into phases.

16.2.5.1 Mine Production Schedule

The mine production schedule aimed to maximise the grade at the mill while balancing the ore tonnage production with the tailings and underground ore productions for a steady mill feed. Low-grade material was stockpiled to be rehandled and processed later in the mine life in order to maximise the grade earlier.

The resulting open pit mine schedule is presented in Table 16.4, and the end of period maps are presented in Figures 16.7 to 16.12. The material extracted in that period is represented in green.

Table 16.4 – Open Pit Mine Production Schedule

Description		Unit	PP	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Total
Ore	Pit to Mill	kt	-	200.9	192.6	185.8	251.0	248.6	234.2	202.8	-	-	1,516
	Pit to Low-Grade Stockpile	kt	8.1	20.3	113.2	98.9	36.7	36.7	29.2	29.3	-	-	372.4
	Pit to High-Grade Stockpile	kt	62.6	48.2	80.8	45.7	28.9	11.3	4.1	8.4	-	-	290.2
	Low-Grade Stockpile to Mill	kt	-	14.5	0.1	7.6	30.1	9.5	45.3	76.8	-	188.6	372.4
	High-Grade Stockpile to Mill	kt	-	46.6	95.3	94.7	6.9	29.9	8.4	8.4	-	-	290.2
	Total Mill Feed	kt	-	262.0	288.0	288.0	288.0	288.0	288.0	288.0	-	188.6	2,178.6
	Silver Grade at Mill	g/t	-	350.0	303.3	202.2	279.5	311.7	231.8	230.1	-	61.8	253.5
Waste		kt	3,171	4,320	3,940	3,881	3,296	2,510	1,393	937	-	-	23,448
Total Mined		kt	3,242	4,589	4,326	4,211	3,612	2,806	1,661	1,178	-	-	25,626
Stripping Ratio		w/o	44.8	16.0	10.2	11.7	10.4	8.5	5.2	3.9	-	-	10.8

Figure 16.7 – Pre-Production – Month 12 – Open Pit End of Period Map

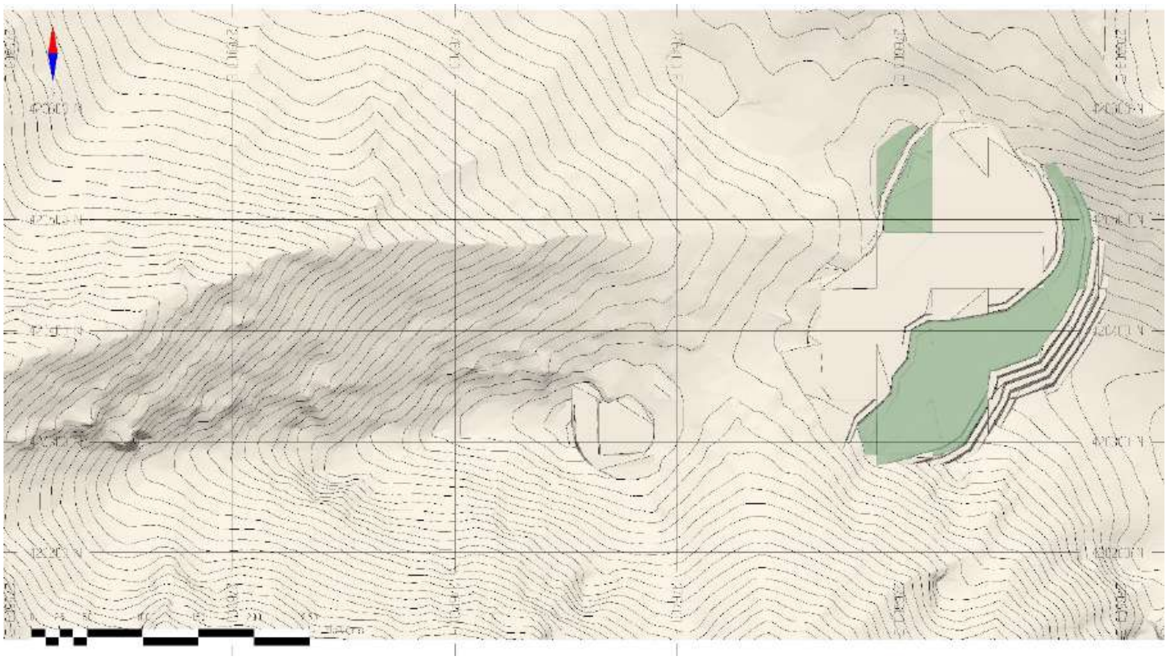
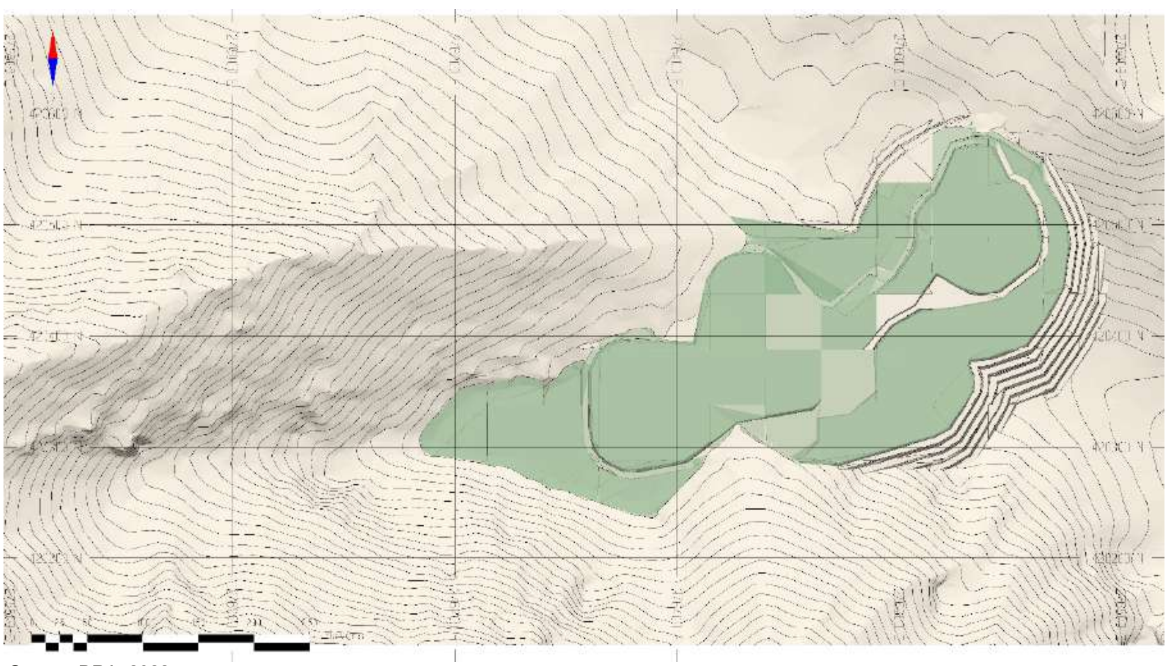
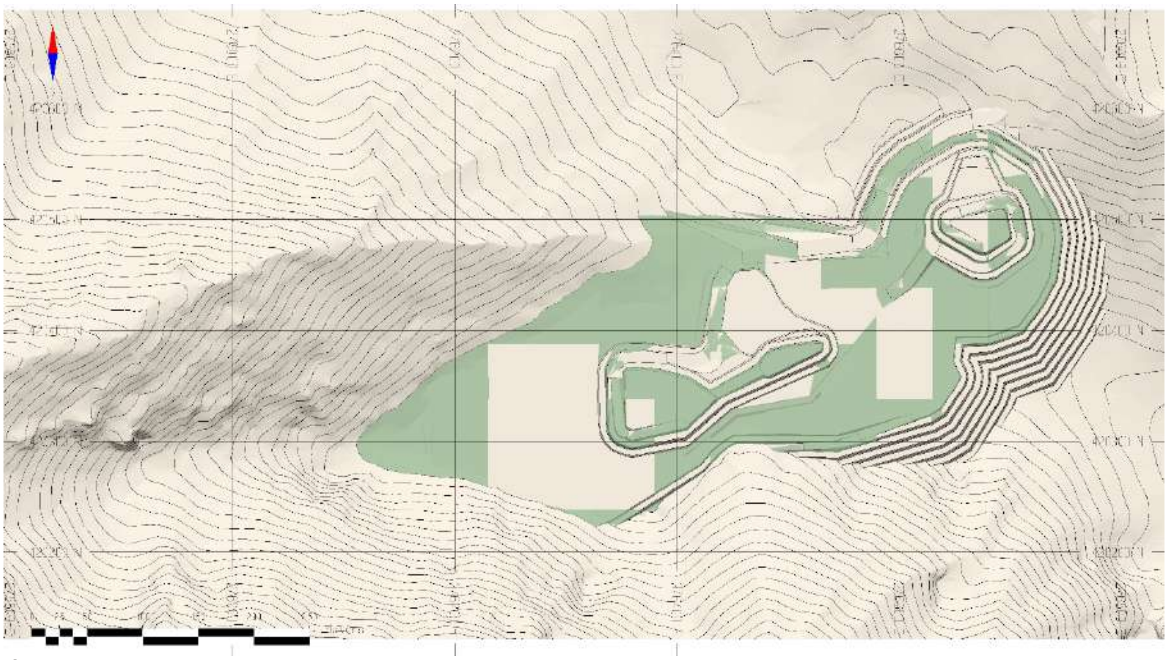


Figure 16.8 – Year 1 – Open Pit End of Period Map



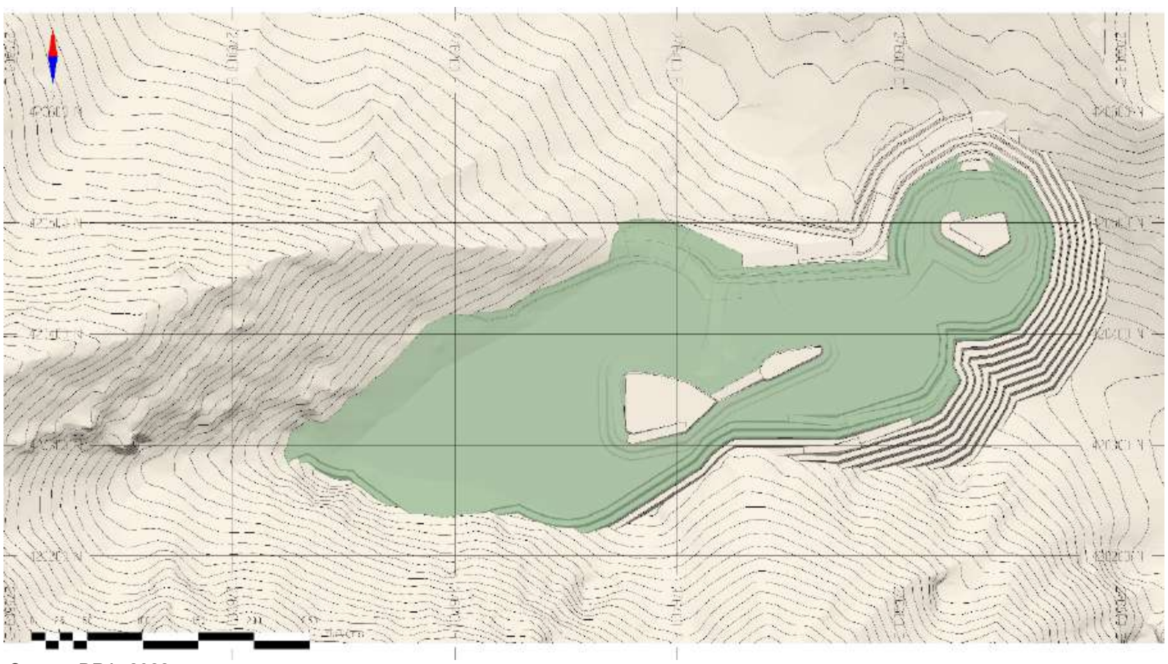
Source: DRA, 2022

Figure 16.9 – Year 2 – Open Pit End of Period Map



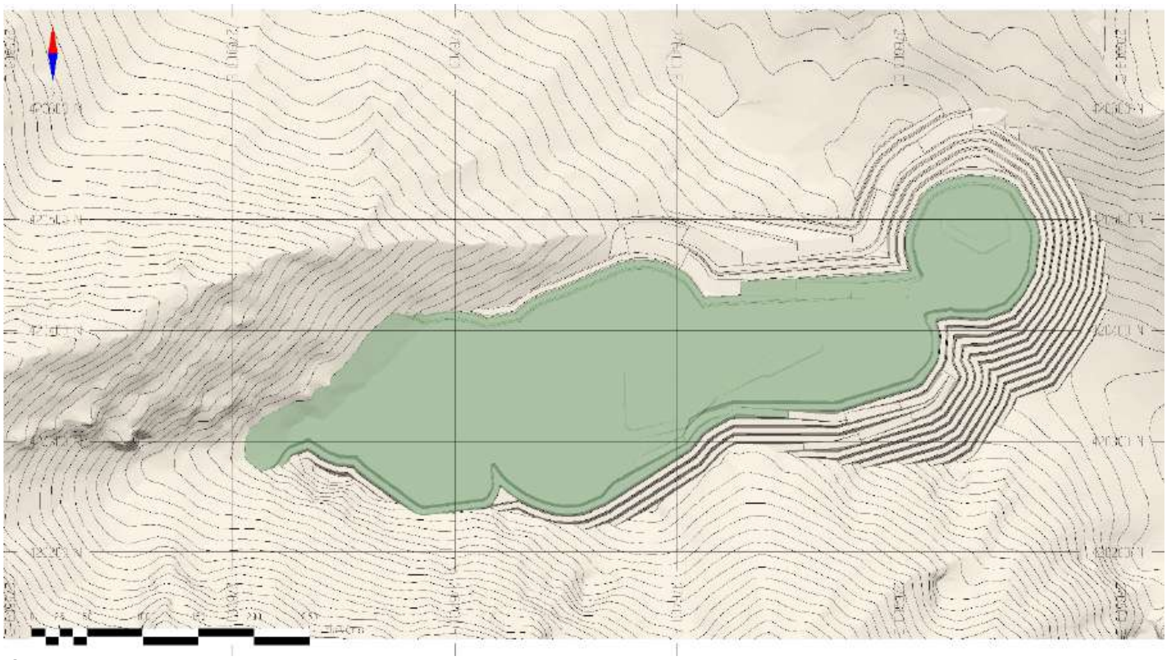
Source: DRA, 2022

Figure 16.10 – Year 3 – Open Pit End of Period Map



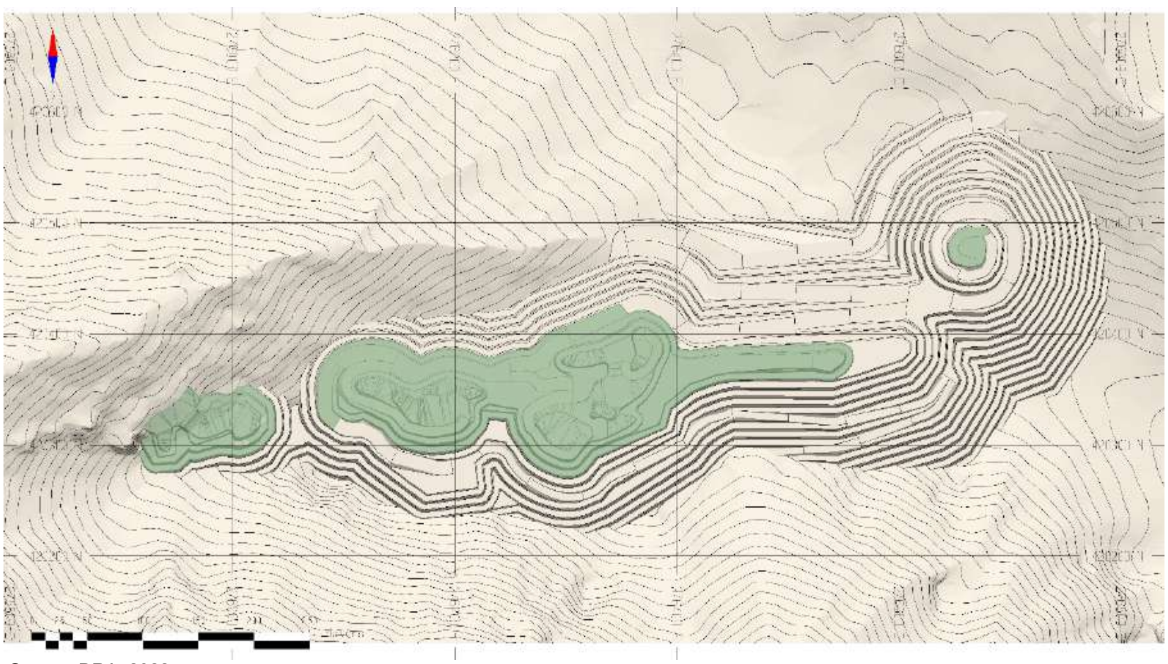
Source: DRA, 2022

Figure 16.11 – Year 4 – Open Pit End of Period Map



Source: DRA, 2022

Figure 16.12 – Year 7 – Open Pit End of Period Map



Source: DRA, 2022

16.2.6 Mine Equipment and Manpower Requirements

The open pit mine will be operated by an open pit contractor, who will be providing all mining equipment and manpower. The equipment used will be appropriate for the designed bench, ramp and road size. Any owner equipment and staff are included in the underground portion of the operation.

Local contractors typically use 8 x 6 type haul trucks and 5 m^3 shovels and loaders. It is anticipated that a peak of 8 trucks will be required.

16.3 Historical Tailings Reclamation

16.3.1 Design

Mineral reserves were estimated for the Zgounder open pit based on the economic parameters detailed in Section 15. The total tonnage to be reclaimed from the tailings is estimated at 0.3 Mt, ore; and no waste material will be reclaimed. This material will be reclaimed over a 2-year period, in conjunction with the open pit and underground mines. No design was required for the tailings, which will be reclaimed in a similar manner to ore stockpiles.

16.3.2 Reclamation Plan

A reclamation plan (or schedule) was prepared to estimate a probable reclamation scenario for the Project's tailings and assess the capital and operating costs for the financial model. The reclamation plan was based on an ore production rate of 961 ktpa to the mill, combining the open pit, the tailings and the underground mining.

The reclamation planning was performed using HxGN MinePlan's MPSO module based on the existing tailings facility.

16.3.2.1 Reclamation Schedule

The reclamation schedule aimed to maximise the mill feed at the end of the open pit and underground mine life. The resulting tailings reclamation schedule is presented in Table 16.5

Table 16.5 – Tailings Reclamation Schedule

Description		Unit	Y8	Y9	Total
Ore	Tonnage to Mill	kt	288.0	30.8	318.8
	Silver Grade	g/t	80.5	39.5	76.6

16.3.3 Equipment and Manpower Requirements

The tailings reclamation will be undertaken by an open pit contractor, who will be providing all mining equipment and manpower. Any owner equipment and staff are included in the underground portion of the operation.

16.4 Underground Mining

16.4.1 Geotechnical Assessment

The information presented in this section is, for the most part, largely extracted, summarised from the Report entitled "Geotechnical Analyses for Zgounder Feasibility Study" Report # 20038-120, Draft Version, prepared by RockEng., issued March 2, 2022.

RockEng Inc. (RockEng) was mandated to provide a feasibility level geomechanical analysis of the Zgounder underground project (RockEng, 2022). The objectives of the geomechanical study were to support the feasibility study by characterising the engineering properties of the major lithological units and to identify major geomechanical risks or opportunities that could affect the economic viability of the project. The complete scope of work consists of:

- Geomechanical site characterisation and domain parameterization;
- Numerical stress modelling and interpretation of staged stope extraction sequence;
- Ground support recommendations for permanent and temporary development;
- In stope ground support;
- Stope designs and dimensioning constraints;
- Dilution predictions;
- Crown pillar assessment;
- Backfill strengths and recommendations;
- Mine sequence recommendations; and
- Infrastructure siting recommendations.

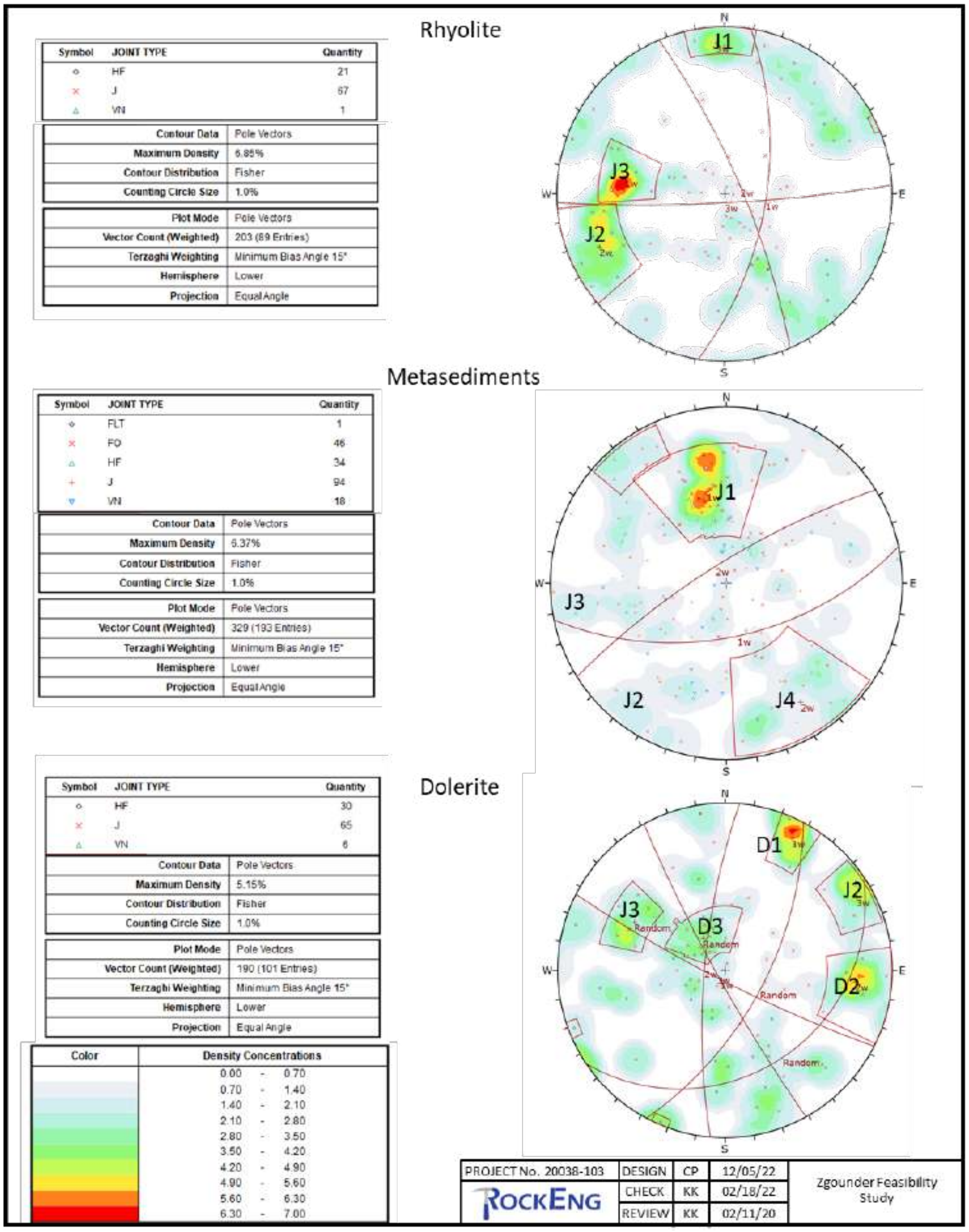
The mine design, mine sequence and mining parameters provided at the time to RockEng were of the previous design. Some differences with the latest design were identified by RockEng following the design exercise. These differences will be addressed in future design exercises and/or next phase of the Project.

16.4.1.1 *Geotechnical Characterisation*

The Zgounder geotechnical site characterisation has incorporated geological and geotechnical drill core logs and laboratory strength testing data, as well as existing geology logs, and site lithological and fault models. Detailed review and analyses of all available data at this stage has led to the following conclusions:

- Structural Features: Structural domains vary by lithology. These are presented below and illustrated in Figure 16.13.
  - The rhyolite features three dominant joint sets:
    - One (J1) dips steeply to the south, similar to the foliation/bedding in the metasediments (however rhyolite does not appear to be foliated); and
    - Two (2) are believed to be associated with late-stage brittle deformation: (J2) steeply inclined dipping to the northeast and (J3) inclined dipping east).
  - The metasediments have two well defined joint sets and two minor joint sets:
    - The bedding/foliation parallel jointing (J1) is the dominant trend inclined dipping to the south; and
    - One major joint set (J4: inclined dipping to the northwest) and two weaker trends (J2: inclined dipping to the north east, and J3: inclined dipping east) are believed to be associated with late-stage brittle deformation.
  - The dolerite has 6 joint sets:
    - Three (3) are believed to be associated with intrusive cooling (D1, D2, D3) and are roughly orthogonal to one another; and
    - Three (3) are believed to be associated with late-stage brittle deformation.
- Intact Rock Strength: Most rock units at Zgounder are medium strong to strong. The average unconfined compressive strength (UCS) based on laboratory testing are: rhyolite = 47.5 MPa, dolerite= 56.5 MPa and metasediments = 38.5 MPa.
- Rock Mass Classification: Geotechnical domaining has demonstrated that, for feasibility study design purposes, the rock mass can be spatially categorized by lithology. The rock mass quality is, on average, poor to fair. Average Q' values by lithology are rhyolite = 4.6, dolerite= 2.4 and metasediments = 3.8.
- In-situ Stress: The far field in situ stress state is unknown. The in-situ stress tensor applied for this study assumes a local horizontal stress ratio of 1.3 ($\sigma h=\sigma H$), with principal horizontal stresses orientated north-south and east-west.

Figure 16.13 – Stereonets for Each Lithology Encountered in the Geotechnical Drilling Campaign



Source: (RockEng, 2022)

16.4.1.2 *Geotechnical Mine Design*

Based on the current understanding of the deposit and the data available for the Project, RockEng provided geotechnical guidelines and recommendations for the mine design. The design assumptions and approach can be consulted in their feasibility-level geomechanical study (RockEng, 2022).

a. Ground Support Requirements

All underground excavations will require adequate ground support installed. No workers shall work under unsupported grounds or in any areas deemed unsafe by a competent internal or external geotechnical professional. The ground support requirements provided by RockEng can be consulted below:

- Standard Drift Development – For excavation spans up to 5 m.
  - Category 1 (Q >= 1): Mesh and 2.4 m Swellex on a 1.2 m x 1.2 m pattern with mesh and bolts across the back and down walls to mid-drift height.
  - Category 2 (Q < 1): Mesh and 2.4 m Swellex on a 1.1 m x 1.1 m pattern with mesh and bolts across the back and down walls to within 0.5 m of the floor.
  - Category 3 (Q < 0.1): Add 75mm shotcrete to Category 2.
- Wide spans and intersections up to 8 m – Secondary support (installed in addition to primary support) is required to account for larger potential wedges. In addition to the standard primary support, the following options can be used for secondary support:
  - 3 m Super Swellex (25 tonne capacity) on a 1.5 m x 1.5 m pattern; or
  - 3.5 m Super Swellex (25 tonne capacity) on a 2 m x 2 m pattern; or
  - Alternatively, cable bolts (of the same length, and with the same pattern) may be used instead of Super Swellex bolts.
- Drawpoint Support – The drawpoint support should consist of standard ground support with an addition of three rings of 3 m long Super Swellex with 2 bolts per ring across the drift back.

b. Stope Stability Recommendations

The current longhole stope design (see Sections 16.4.3.1 and 16.4.3.2) includes long support in the longhole stope's back for longhole stopes exceeding a back span of 6 m x 12 m (hydraulic radius of 2 m). These represent approximately half (47%) of the longhole stopes. No long support will be installed in the hanging wall, footwall nor end walls of the longhole stopes.

In addition to the long support installed, standard ground support will be installed in the longhole stope overcut and undercut as per Section 16.4.1.2a. Standard ground support will also be installed in drift-and-fill stopes as per Section 16.4.1.2a.

c. Crown Pillar Stability Recommendations

Empirical crown pillar analyses conclude that, for short term crown pillar stability with probability of failure 5 to 10%, mining should be restricted as follows:

- Drift-and-fill (restricted to 5 m spans and cut lengths up to 40 m) will need a 35 m thick crown for stopes in metasediments and a 45 m thick crown for stopes in dolerite.
- Longhole stoping (restricted to 10 m HW to FW span with 20 m strike length span) will need to be more than 50 m depth from surface (to back of stope) for metasediments and 65 m for dolerite.

Crown pillar ore will be recovered from the underground in the late stages of the Project when open pit operations will be completed in its vicinity. A detailed engineering plan (backfill plan, detailed stope design and sequencing intentions, ventilation, water management strategies, geotechnical monitoring strategies, etc.) will need to be put in place to safely recover this ore.

The long-term risk of crown pillar failure to surface for the Zgounder Project will be in part or completely negated by the placement of backfill to within 5 m of the back on the upper-most stoping level. This will be accomplished by a LHD vehicle equipped with a push-plate on site.

d. Backfill Strength Requirements

The current mining methods intend to use consolidated cemented rockfill for both the drift-and-fill and the longhole stoping mining methods. For hanging wall (HW) to footwall (FW) spans up to 15 m, a backfill unconfined compressive strength (UCS) of approximately 235 kPa will be required for overhand stopes.

Assuming that adequate consolidation of CRF is achieved in drift-and-fill stopes, undercut backfill strengths will have to be in the order of 2 to 3 MPa (span dependent). These backfill strengths will need to be achieved to recover the sill pillars throughout the underground mine.

e. Historic Underground Excavations

For historic mine excavations, a void risk assessment will need to be completed prior to advanced construction and operations to better define hazards (safety, production and environmental) and consequences associated with historical mine voids, confidence in void locations and geometry and necessary actions for risk management and mitigation. Detailed backfill and ground support design in man-entry undercuts will be required as mine planning advances prior to initializing underhand drift-and-fill or longhole stoping activities.

16.4.2 MINING METHODOLOGY

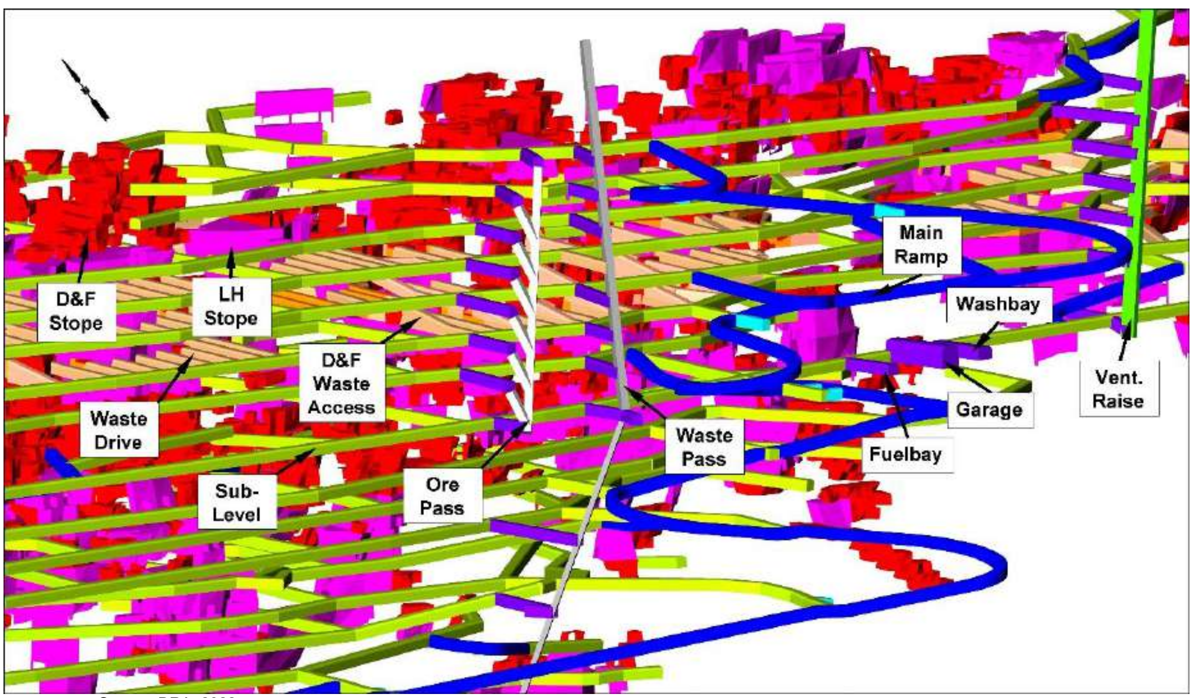
Underground mining will occur concurrently with the open pit mine. The underground mine has been designed to produce 1,844 t/d of ore from an underground operation using conventional highly mechanized mining equipment routinely used in similar operations around the world.

Due to the complexity of the deposit and historic mined excavations, the mining methods selected for the Zgounder mine will be transverse long-hole stoping, longitudinal long-hole stoping and drift-and-fill mining. The mine layout is therefore complex and adjustments in the design are expected as more information is gathered and mining progresses.

The underground mine will be accessed from surface and from the historic underground drift excavations that will need to be rehabilitated. These access points can be viewed in Figure 16.1. The main ramp will start from the underground on the 2000L main level and be excavated up and down to the 2092L and 1648L respectively. Historic underground stope excavations were assumed to be cemented backfilled prior to developing new drifts nearby.

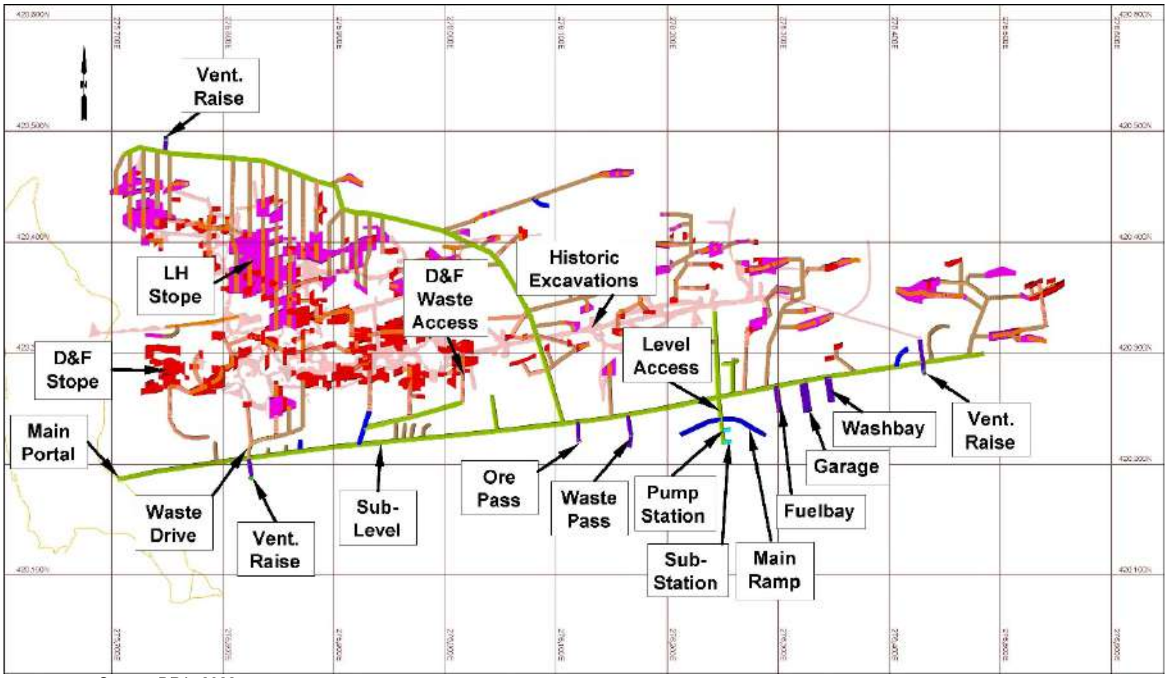
A general overview of the mine design and typical layouts of the different levels are presented in Figures 16.14 to 16.17.

Figure 16.14 – General Mine Layout (Isometric View)



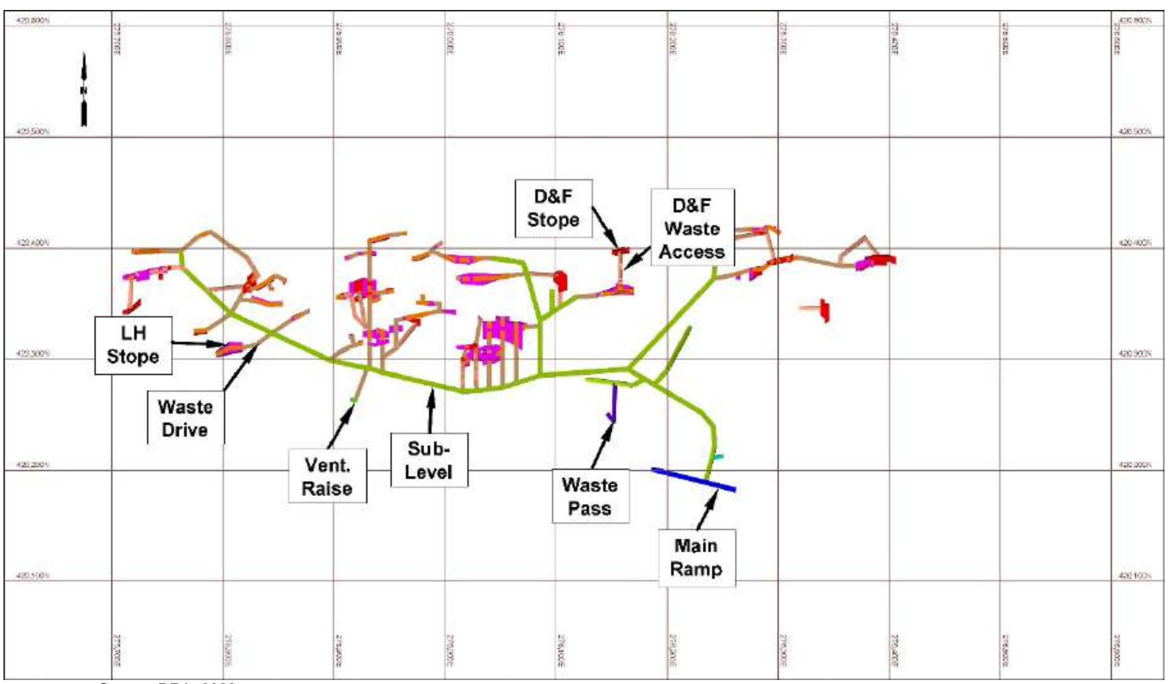
Source: DRA, 2022

Figure 16.15 – Main Level (2000L)



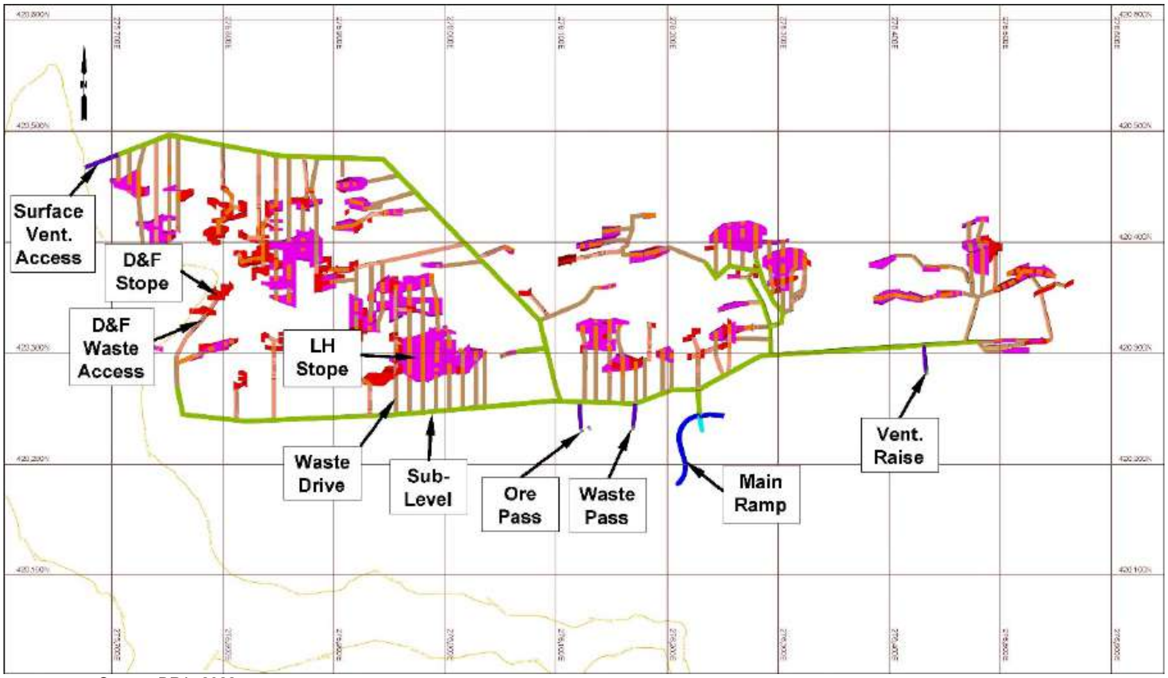
Source: DRA, 2022

Figure 16.16 – Typical Level Layout Below 2000L



Source: DRA, 2022

Figure 16.17 – Typical Level Layout Above 2000L



Source: DRA, 2022

16.4.3 Stope Design

Stopes were designed using a two-pass method. The first pass design was done using the Stope Optimizer (SO) module in Deswik software (Deswik.SO) to design long-hole stopes and drift-and-fill stopes based on defined parameters (minimum and maximum stope dimensions, minimum cut-off, dilution and recovery, etc.). The SO parameters used for Zgounder are presented in Tables 16.6 and 16.7. The second pass was to combine the two (2) mining methods by selecting the most optimal method based on the productivity, the deposit geometry and the grade.

Table 16.6 – SO Parameters for Long-hole Stoping

Description	Value
Width (Longitudinal-Length)	12 m
Height	12 m
Length (Longitudinal-Width)	2 to 10 m
Dip	0° (Flat)
Wall Minimum and Maximum Dip	55° to 115°
Internal Maximum Dilution	40 % (in marginal waste)
External Dilution	0.3 m
Ag Cut-off Grade	85 g/t

Table 16.7 – SO Parameters for Drift-and-Fill Stoping

Description	Value
Width	3 to 7 m
Height	4 m
Length	3 to 100 m
Dip	0° (Flat)
Wall Minimum and Maximum Dip	60° to 120°
Internal Maximum Dilution	40 % (in marginal waste)
External Dilution	0.1 m
Ag Cut-off Grade	85 g/t

16.4.3.1 Transverse Long-hole Stoping

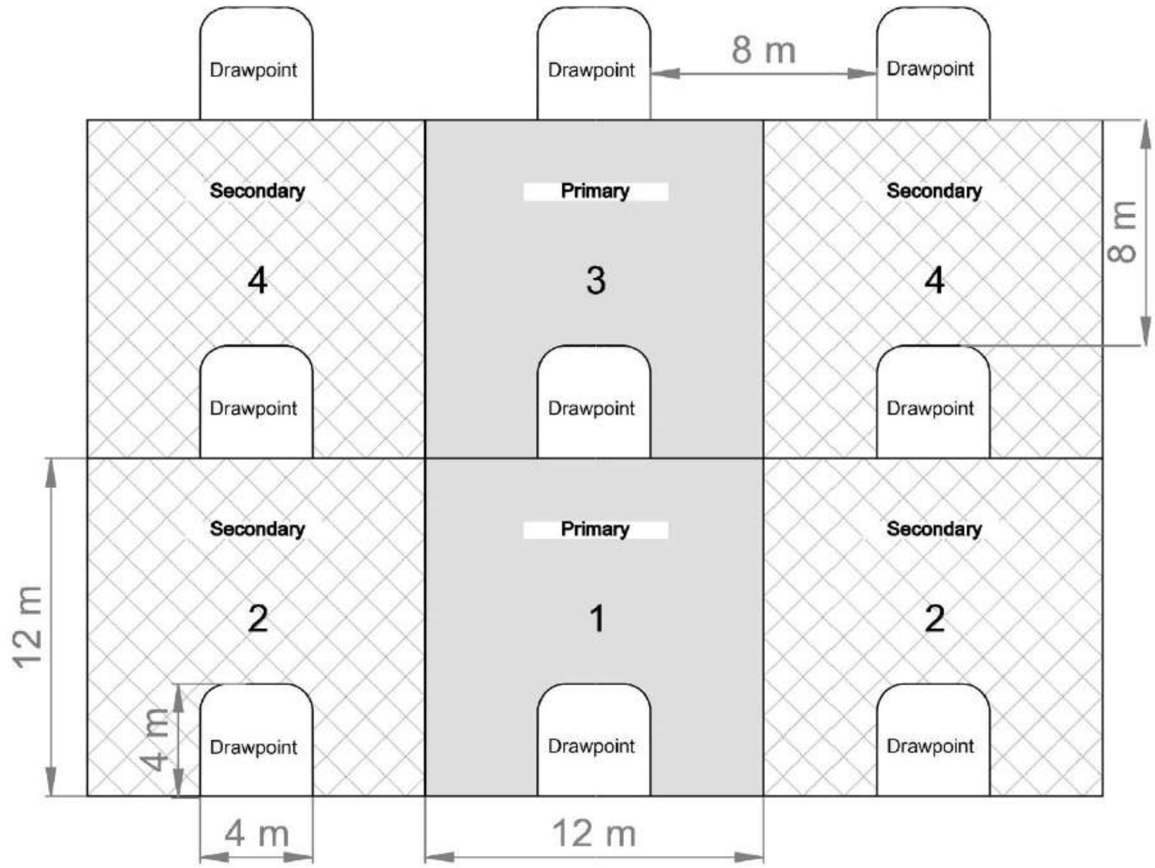
Several areas of the mine will be developed for transverse long-hole (LH) stoping. The transverse stopes will be mined in a primary-secondary sequence, with primary and secondary stopes measuring 12 m wide x 12 m high x 15 m deep. Generally, mining for transverse stopes will start at the far side (North) and retreat to the near side (South). In some instances, the far side will be mined from South to the near side (North).

The transverse sequences will start at the lower level of a given production zone. The primary stopes will be excavated first and then backfilled as the sequence advances. Once the curing is completed, the production of the adjacent secondary stope can begin. Once the primary and secondary stopes are completed on a given level in a production zone, the production in that zone can then move to the upper level.

Ore handling and stope backfilling are described in Sections 16.4.3.4 and 16.4.3.5.

The general mining sequence for transverse long-hole stoping is illustrated in Figure 16.18.

Figure 16.18 – Transverse Long-hole Stoping Mining Method



Source: DRA, 2022

Production drilling will be done by ITH long-hole drills. The selected drilling diameter is 89 mm. A 3 m x 3 m cut will be drilled using the same drills. Table 16.8 presents the drilling parameters for the transverse long-hole stope blasting parameters.

Table 16.8 – Transverse Long-hole Stope Blasting Parameters

Item	15 m LHOS
Top Access Drive Width	4 m
Top Access Drive Height	4 m
Bottom Access Drive Width	4 m
Bottom Access Drive Height	4 m
Level Spacing	12 m
Stope Width	12 m
Stope Height	12 m

Item	15 m LHOS
Stope Length	15 m
Ring Burden	1.50 m
Volume per Stope	2,160 m^3
Tonnes per Stope	5,940 m^3
Hole Spacing	1.88 m
Hole Stemming (Average)	3.5 m
Drillhole Diameter	89 mm
Hole Sub-Drilling	0.18 m
Explosive Specific Gravity	0.95 g/cm^3
Powder Factor	0.69 kg/t

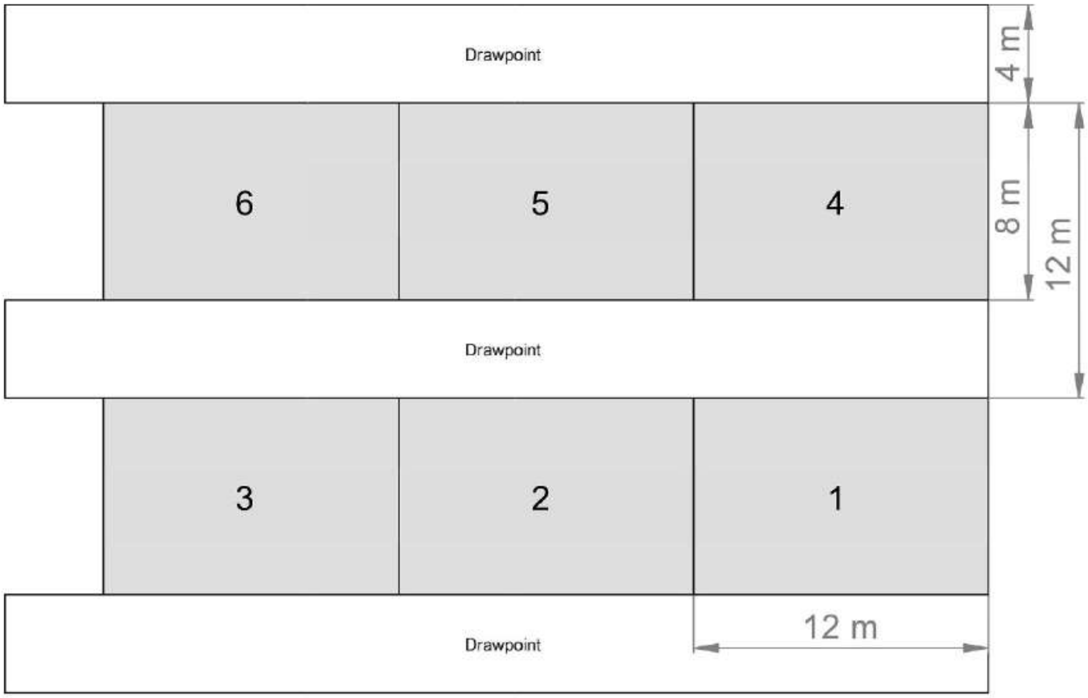
16.4.3.2 Longitudinal Long-hole Stoping

Several areas of the mine will be excavated using the longitudinal long-hole (LH) stoping method. The longitudinal stopes will be mined retreating from East to West and also from West to East. The longitudinal stopes width will vary from 3 m to 10 m, with an average length of 12 m using the same sublevel spacing of 12 m. The retreating of the stope will start at the bottom level of a given longitudinal mining zone. The first stopes are excavated then backfilled. The next stope in the longitudinal sequence will be ready for production when the curing of the previous stope will be completed. Once a production level is completed in a longitudinal mining zone, the production on the level above, in the same given zone, can begin.

Ore handling and stope backfilling are described in Sections 16.4.3.4 and 16.4.3.5.

The general mining sequence for longitudinal long-hole stoping is illustrated in Figure 16.19.

Figure 16.19 – Longitudinal Longhole Stoping Mining Method



Source: DRA, 2022

Production drilling will be done by ITH long-hole drills. The selected drilling diameters is 89 mm. A 3 m x 3 m cut will be drilled using the same drills. Table 16.9 presents the drilling parameters for the transverses long-hole stopes.

Table 16.9 - Longitudinal Long-hole Stope Blasting Parameters

Item	15 m LHOS
Top Access Drive Width	4 m
Top Access Drive Height	4 m
Bottom Access Drive Width	4 m
Bottom Access Drive Height	4 m
Level Spacing	12 m
Stope Width	3 to 10 m
Stope Height	12 m
Stope Length	18 m
Ring Burden	1.50 m
Volume per Stope	791 m^3
Tonnes per Stope	2,175 m^3

Item	15 m LHOS
Hole Spacing	1.80 m
Hole Stemming (Average)	1.78 m
Drillhole Diameter	89 mm
Hole Sub-Drilling	0.18 m
Explosive Specific Gravity	0.95 g/cm^3
Powder Factor	0.79 kg/t

16.4.3.3 Drift-and-Fill

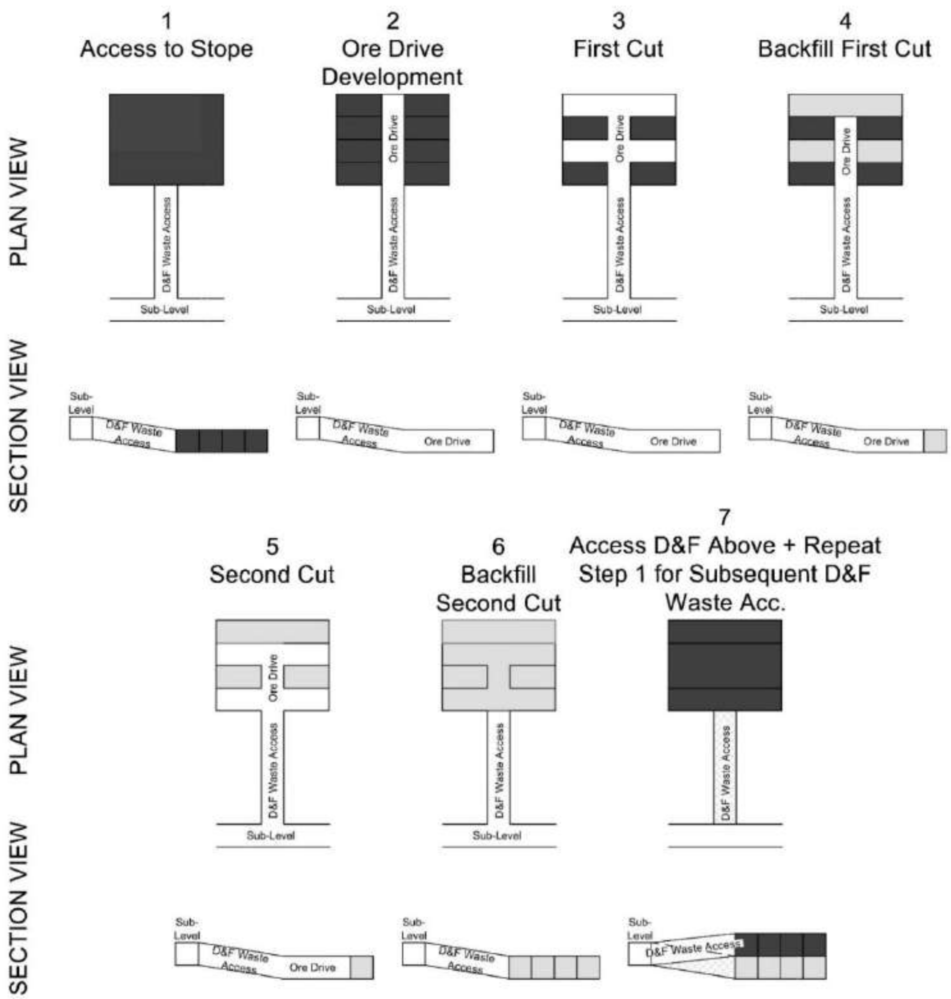
Drift-and-fill (D&F) mining is one of the preferred mining methods due to its low operating cost in the region and its selective approach (high recovery and low dilution). D&F mining will be accomplished using lateral development equipment the same used to develop accesses to the long-hole stopes. The lateral round will be drilled using a 1-boom hydraulic jumbo with an advance rate of 3.6 m. The selected height of the D&F stopes is 4 m for an optimum combined recovery and productivity. The width of the stope will vary following the mineralization. The minimum width of the stope is 3 m, to fit the production equipment. The maximum width, or open span, is 8 m, to minimize the ground support requirement. Ore handling and stope backfilling are described in the Sections 16.4.3.4 and 16.4.3.5.

The drift-and-fill mining sequence is mainly bottom-up drift-and-fill waste drive to access the drill-and-fill stopes on the next sub-level. Figure 16.20 illustrates the drift-and-fill mining method.

16.4.3.4 Ore Handling

Mucking of the ore will be done using 10-tonne Load-Haul-Dump (LHD) machines. LHDs will be equipped with a remote-control system for mucking in the long-hole stopes. For the levels below 2000 level, broken ore will be brought to the intersection of the level access where loading of the haul trucks will occur. The back of these intersections will be slashed to 5 m to allow loading. The 20-tonne underground mining trucks will haul the ore material to the surface through the ramp(s) and the main level. For levels above 2000 Level, the ore will be brought to the ore pass. A truck loading bay chute will be installed at the bottom to load haulage trucks.

Figure 16.20 – Drill-and-Fill Mining Method



Source: DRA, 2022

16.4.3.5 *Stope Backfilling*

All stopes will be backfilled with cemented rockfill (CRF). A waste pass will be driven from underground to the surface open pit to ensure enough waste is available for backfilling the stope. At each level, an LHD will move the waste from the waste pass access to the stopes. The cement milk will be added to the LHD bucket in a selected cut-out on the sub-level. The cement milk will come from the surface plant through a pipe installed in the main level, the main ramp and through drill holes between levels. The mixing of the rock with the cement milk will be done naturally while unloading the cemented rockfill in the stope. For the long-hole stopes, the cemented rockfill will simply be dumped from the top access. For the drift-and-fill stopes, the cemented rockfill will be rammed inside the stope tight to the back.

16.4.4 Mine Design

The Zgounder mine will be developed with underground headings to handle at least 1,844 t/d of ore plus related waste development. The main ramp and headings are sized to handle up to 20-tonne haulage trucks.

16.4.4.1 Portals / Main Level

A new portal will be excavated in the diorite south of the deposit to access the new underground mine while operating the existing mine without interfering with the operations. This portal will have access directly to the main level on the 2000 Level. The main level will be driven to the east end of the deposit and will remain the main haulage way of the underground operation throughout the life of mine. It will connect to the main ramp through the level access in the eastern part of the mine. It will also access some of the access to the productions stope. Therefore, the production can be assumed from the new main access at an early stage of the project.

The main level is sized 4.0 m high by 4.5 m wide to allow for 20-tonne underground trucks, the larger equipment in the mine, during development and during production. The main level will carry all the mine services from the portal to the ramp, and the workshop location in that same level. The main service available will include a:

- 4” compressed air line;
- 4” wastewater line;
- 2” freshwater line;
- 2” diesel steel pipes;
- High-voltage cables;
- Leaky feeder with radio; and
- Ventilation duct of 0.9 m diameter (during construction only until the permanent ventilation system is operational).

The ground support installed will be mainly 2.4 m Swellex bolts and mesh screen (see Section 16.4.1.2a for more details). Mucking of the headings will be completed by 10-tonne LHD to the nearest remuck bay to be thereafter loaded in 20-tonne trucks and hauled to the main portal. The excavation of the main level will be performed by a qualified mining contractor.

The portal locations are shown in Figure 16.1.

16.4.4.2 Main Ramp

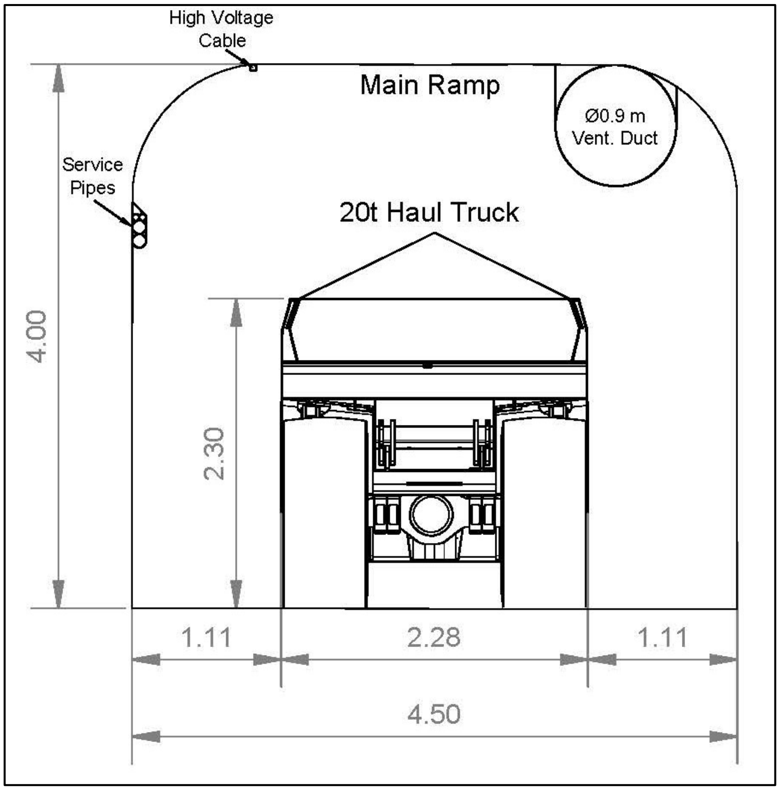
The main ramp will connect to the main level and to all other levels of the underground mine. The main ramp was placed in the center of the deposit to optimize the ore and backfill handling.

The ramp is sized 4.0 m high by 4.5 m wide to allow for 20-tonne underground trucks, the larger equipment in the mine, during development and during production. The ramp will be equipped with the same services as the main level with the exception that of the high-voltage cables which will be installed only in some segments of the ramp to carry power to the different levels. Some drill holes between levels will also be used to connect cables between level accesses. During construction until the permanent ventilation system is operational, the ramp will be equipped with a ventilation duct of 0.9 m diameter. At the production phase, the ventilation duct will only remain in the lower part of the underground mine to force fresh air to the lower levels and the ramp will be the main exhaust of the mine from the 1648 level to the 2000 level.

The ramp will be driven at a maximum slope of 15%, which is typical for underground mines. At the intersection of the level access, the ramp will be sloped 3% toward the level access intersection so the running water can be diverted to the level and the sump. Also, the fairly flat intersection will help the circulation of the mobile equipment.

The same development method as the main level will be used for the main ramp. The general arrangement of the ramp is shown in Figure 16.21.

Figure 16.21 – Main Level and Main Ramp General Arrangement with 20-t Truck



Source: DRA, 2022

16.4.4.3 Level Access

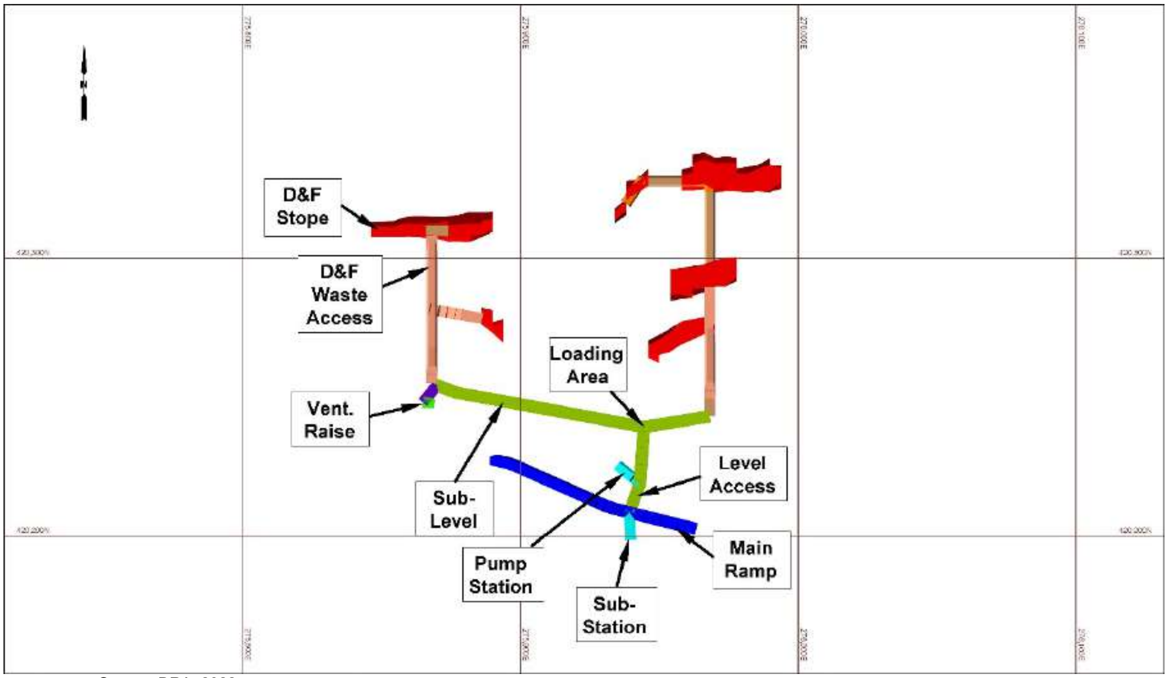
The level access has been designed at 4.0 m high by 4.5 m wide to fit a 20-tonne truck and a 10-tonne LHD. They will connect the main ramp to the sub-levels. From the level access, a sump will be excavated to catch the water running down from the level but also from the main ramp. A substation is also excavated in the level access. This allows level services to be centralised in the level access.

During the life of mine, the majority of the haulage trucks will be loaded at the intersection of the level access and the sub-level.

The level access has been designed and located considering the geological setting. The excavation will be completed by a contractor. The primary ground support remains the same as the ramp. The waste will be loaded into trucks and carried to the surface or to be used as backfill for the stopes as described in Section 16.4.3.5.

A typical level access layout is shown in Figure 16.22. The general arrangement of the main haulage drifts is shown in Figure 16.23.

Figure 16.22 – Typical Level Access Layout, 1744L (Plan View)



Source: DRA, 2022

16.4.4.4 Sub-Level

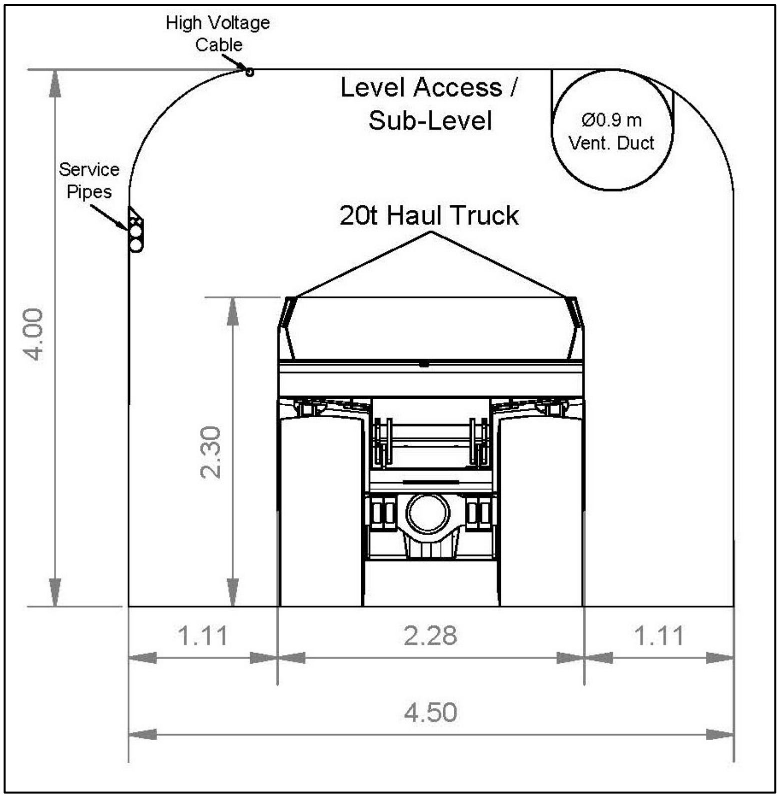
The mining areas will be accessed to the east and to the west by the sub-levels. Those excavations are designed at 4.0 m high by 4.5 m wide to fit a 20-tonne truck and a 10-tonne LHD. They will connect the waste drive and the drift-and-fill waste access to the level access located near the centre of the level. During production, the main activity in the sub-level will be by LHDs carrying the ore from the stope to the loading area. Furthermore, this access will be equipped with services to carry compressed air, water, power, communication and blasting and ground cables, and ventilation conduit to the working areas.

All sub-levels will be driven at a 3% slope to drain the ground water and drilling water towards the main ramp, where the water management system will be located.

The sub-level has been designed and located considering the geological setting. The excavation will be completed by a contractor. The primary ground support remains the same as the ramp. The waste will be loaded into trucks and carried to the surface or to be used as backfill for the stopes as described in Section 16.4.3.5.

The general arrangement of the main haulage drifts is shown in Figure 16.23.

Figure 16.23 – Level Access and Sub-Level General Arrangement with 20-t Truck



Source: DRA, 2022

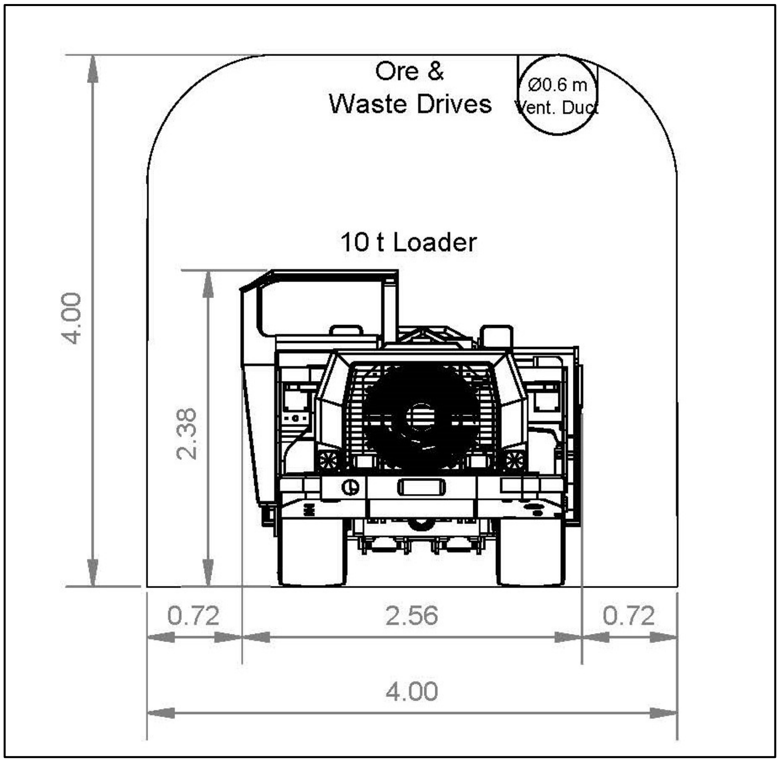
16.4.4.5 Ore Drive

Ore Drives are designed at 4.0 m high by 4.0 m wide to fit 10-tonne LHDs used for the ore mucking. Temporary flexible ventilation ducts will be installed to ventilate the excavation and the mucking operations. Temporary services will be installed for longer access. Adequate ground support will be installed based on RockEng recommendations (see Section 16.4.1).

156 kt of ore is planned to be excavated for the ore drives throughout the life of mine. The ore handling is described in the previous Section 16.4.3.4. For the long-hole stopes, the ore drive will be used to drill from the top and to muck the stopes from the bottom. For the drift-and-fill stopes, ore drive(s) will be excavated within ore to access the stopes.

The excavation of the ore drive will be completed by the production crew. The general arrangement of the ore drive is shown in Figure 16.24.

Figure 16.24 – Ore and Waste Drive General Arrangement



Source: DRA, 2022

16.4.4.6 Waste Drive

The waste drive will connect the ore drive to the sub-level, in the long-hole operation. Like the ore drive, the waste drive is designed at 4.0 m high by 4.0 m wide to fit 10-tonne LHDs used for the ore mucking. Temporary flexible ventilation ducts will be installed to ventilate the excavation while in development and production. Temporary services will be installed for longer access. Also, temporary ground support will by installed following geotechnical recommendations. The proximity of the waste drive to the stope will maximize the mucking waste to the backfill operation.

The excavation of the ore drive will be completed by the production crew.

The general arrangement of the waste drive is shown in Figure 16.24.

16.4.4.7 Drift-and-Fill Waste Access

The drift-and-fill waste access will connect the ore drive to the sub-level, in the drift-and-fill operation. They are also at 4.0 m high by 4.0 m wide to fit 10-tonne LHDs used for the ore mucking. The described access is a slash of the back of the bottom access to give access to the upper level. The drift-and-fill waste access is illustrated in Figure 16.20 of Section 16.4.3.3.

Temporary flexible ventilation ducts will be installed to ventilate the excavation operations. Temporary services will be installed for longer access. Also, temporary ground support will by installed following geotechnical recommendations. The proximity of the waste drive to the stope will maximize the mucking of the waste to the backfill operation.

The excavation of the ore drive will be completed by the production crew.

16.4.4.8 Ore Pass

A 3 m by 3 m ore pass will by driven from the main level access, at the 2000 Level, to the 1976 level. Finger raises will also be driven to connect the ore pass to the ore pass accesses and to the sub-levels.

The ore pass and the finger raises will be driven with an Alimak by a local contractor. The Alimak method will also be used to drive the other raises.

16.4.4.9 Waste Pass

Part of the backfill circuit, a 3 m by 3 m waste pass will be driven from the main level access, at the 2000 Level, to the surface pit. As the underground mine gets deeper, the waste pass will be driven further down in segments to lower levels to extend with the existing ones. The mine is designed to get the waste from the pit all the way down to the 1776 level. For the lower levels, the waste will be trucked or directly hauled from nearby waste material with the LHD.

The waste passes segments will be driven with an Alimak by a local contractor. The Alimak method will also be used to drive the other raises.

16.4.4.10 Fresh Air Raises and Escapeways

Three (3) raises will be driven South of the deposit from the 1708 Level to the main level (2000L). In the North, one (1) raise will connect the 1984 Level to the 2020 Level. As the mine gets deeper, below the 2000 level, fresh air raises will be driven in segments from the levels below. The fresh air network on the South will connect to the main portal (2000L) which is the main intake in the mine. The fresh air raises will be driven on a 3 m by 3 m section. Escapeways will be installed in the fresh air raises so that the underground personnel can use as a second egress in case of emergency.

Like the waste pass, the fresh air raise will be excavated using an Alimak by a local contractor.

16.4.4.11 Exhaust Raises

A 3 m by 5 m exhaust raise will be excavated from the main level (2000L) to the surface. The surface breakthrough is located at the top of the mountain on the edge of the current design of the open pit mine. This raise will connect to the levels above the main level (2000L). Its purpose is to exhaust air from these levels to the surface. The main ventilation fans will be installed at the breakthrough of the raise and will suck the air out.

The exhaust raise will be excavated in two phases. First, a 3 m by 3 m Alimak pilot will be driven up. Secondly, the pilot will be slashed to 5 m wide from the surface using an Alimak while retreating the rail. The raise will be excavated by a local contractor.

16.4.4.12 Underground Infrastructure

As part of the mine development, other miscellaneous infrastructure will be excavated in the mine. The main approach is to excavate crosscuts from the level to the size required.

On the east end of the 2000 Level, an underground garage; a fuel bay and a wash bay will be excavated.

The underground workshop will be installed in the early stages. The service bays will be equipped with all the services required underground to maintain and service the full fleet of underground mobile and fixed equipment, including, but not limited to:

- Garage;
  - Two (2) light service bays;
  - One (1) heavy service bay;
  - One (1) tire bay;
  - Toolboxes;
  - Maintenance office.
- Washbay;
- Fuel bay and fuel storage;

Most of the major repair services for the underground mobile equipment will be done underground. The underground mobile equipment will be on a maintenance schedule to minimize equipment breakdowns during operations. Only emergency repairs will be done underground on mobile equipment. A small underground shop bay in the garage will carry a minimal parts inventory to quickly respond to equipment breakdowns. The fuel and lube trucks will be able to fill the hydraulic systems underground when needed.

The underground shop will carry spare pumps, pump motors, auxiliary ventilators and motors, in case of a breakdown. The pump and fans will be replaced underground and serviced in the surface shop. The pumps will also be on a maintenance schedule and brought to the surface shop on a regular basis. The same principle will apply for the ventilation fans. Spare fans will be kept in inventory for all motor fans.

A diesel line will bring the fuel from the surface to the underground fuelbay.

16.4.5 Explosives Storage and Explosives Handling

The explosive storage facility will be a separate underground excavation from outside the underground mine. This excavation will have its own portal to surface and it will always be gated and guarded. The explosive supplier will deliver directly into this magazine.

The explosive storage magazines will be accessible from the access road. One (1) magazine will store the detonators and the other will store the ammonia nitrate/fuel (ANFO), detonating cord, sticks of packaged emulsion and perimeter explosives. A dedicated vehicle will take the explosives directly from the magazines. The blasting crew will be responsible for marking the explosives and detonators used in the inventory logbooks located in each magazine. The site superintendent will be responsible for arranging regular powder magazine inventory checks as required by regulations.

16.4.6 Services

16.4.6.1 Potable Water

No potable water line will be installed in the underground mine. Water bottles will be available in the refuge chamber and other designated areas.

16.4.6.2 Main Return Water

Industrial water will be supplied mainly for the drilling work. The industrial water will come from the surface and will be recycled. The water line will be installed in the main ramp as well as in all the lateral development of the mine.

16.4.6.3 Compressed Air

The compressed air facilities will be on the surface and will supply the underground mine through a 4” Victaulic reticulation. The compressed air will mainly assist the drilling and explosive loading operations.

16.4.6.4 Diesel

A diesel line will be installed on the main level (2000L) to supply diesel to the fuel tank and station in the 2000L fuel bay. Portable fuel stations will be installed in underground cut-outs in three (3) more levels. The diesel line will be run through the main ramp to supply diesel to these stations.

16.4.6.5 Power

The substation will be excavated from the level accesses. The main power distribution is done to these substations from the main ramp or though drilled holes. For the auxiliary power distribution, a cable will be installed in the lateral development of the mine and will provide power to the drilling, auxiliary pumping and auxiliary ventilation equipment.

16.4.6.6 Toilets

Portable toilets will be available underground in the refuge stations. Some portable toilets will also be installed temporarily near the advance faces to follow the lateral development.

16.4.7 Mine Dewatering

It is not expected to get a significant water inflow in the underground mine. The current mine dewatering strategy is to develop a sump in every level access below the main level (2000L). A dewatering pipe will be installed between the main ramp and the main portal (2000L). The water will be pumped from one sump to the other all the way to the surface with a pumping capacity of 10 l/s.

16.4.8 Refuge Stations

Portable self-contained refuge stations will be purchased and installed to follow the lateral development. In most instances, the portable refuge will be installed near a fresh air intake, and equipped with oxygen bottles.

Eight (8) refuge stations, one every four (4) sub-levels on average, will be commissioned. The refuge stations have been planned to be excavated close to the fresh air raise intake and escapeway of the level. They will be accessible from the sub-level, and they will be connected to the compressed air distribution of the mine.

16.4.9 Mine Plan – Production Schedule

A mine plan was produced from the mine design and by applying production rates and several constraints using the scheduling module in Deswik (Deswik.Sched). The priority was to mine high-grade ore in the first five years of the LOM, while accessing the different mining levels of the mine.

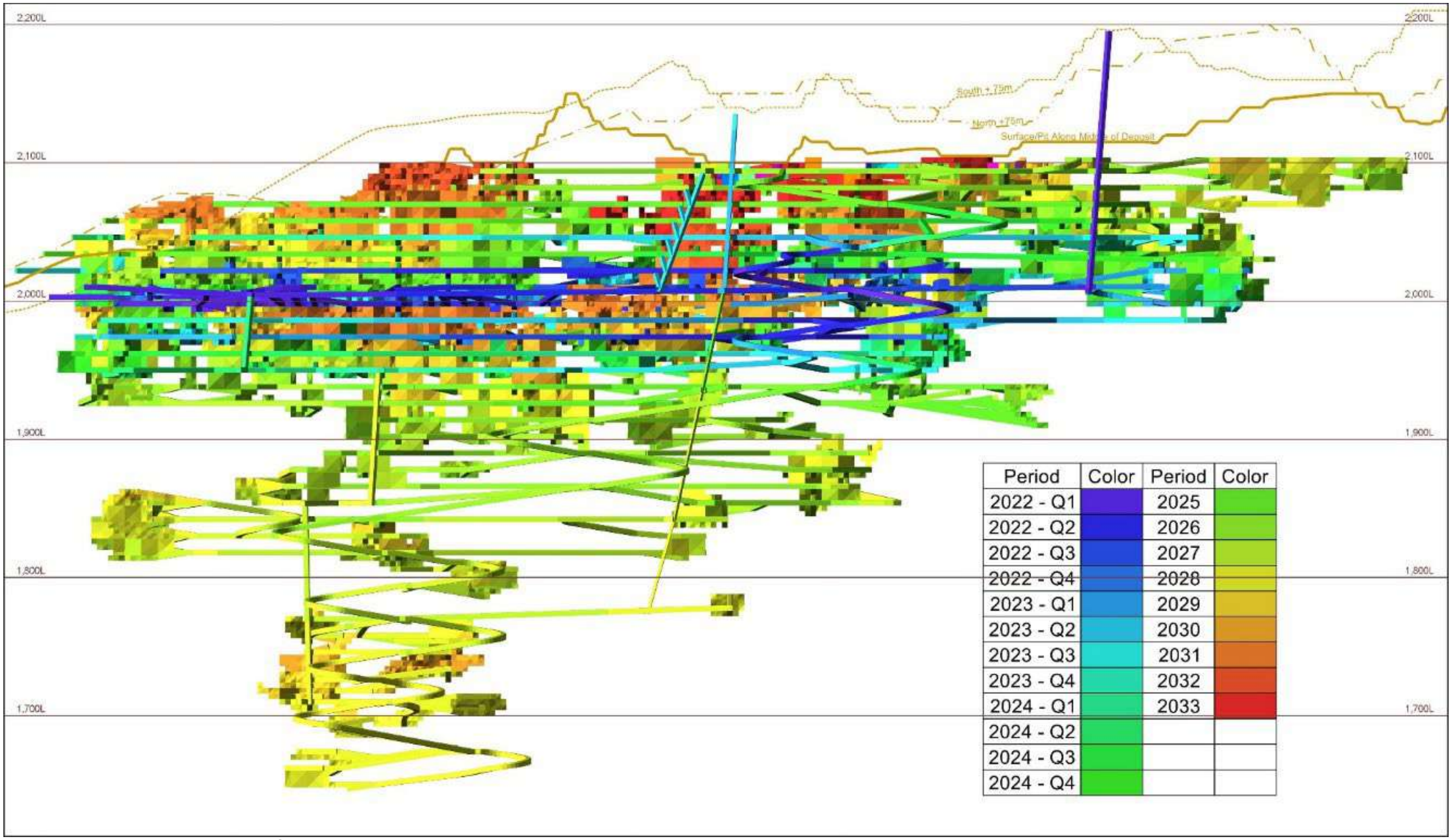
Sequencing of the stopes has been described in Section 16.4.3, but generally remains a bottom-up operation.

The resulting mine production schedule is shown in Table 16.10. Figure 16.25 illustrates the mine plan per quarter for the first three years of the LOM and subsequently each year for the remaining of the LOM.

Table 16.10 – Mine Plan

Description	Unit	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	Total
Silver Production	koz	1,924	2,315	5,441	5,393	6,128	6,317	6,985	5,272	5,576	5,400	1,592	52,343
Ore Production	kt	226.3	251.4	594.9	671.6	673.1	673.1	673.2	673.2	673.2	745.9	237.5	6,093.4
Ore Development	kt	101.3	125.3	221.3	181.1	179.8	59.5	48.7	70.5	75.1	51.6	27.0	1,141.1
Long-Hole Longitudinal	kt	1.8	6.1	83.3	71.3	121.3	153.6	156.3	63.4	34.9	51.6	78.5	822.0
Long-Hole Transverse	kt	20.4	27.9	106.8	220.5	205.6	305.0	299.7	292.8	292.0	430.7	35.9	2,237.4
Drift-and-Fill	kt	102.8	92.1	183.4	198.7	166.5	155.0	168.5	246.5	271.1	212.1	96.1	1,892.9
Silver Grade	g/t	264	286	284	250	283	292	323	244	258	225	209	267
Ore Development	m	2,351	2,906	5,141	4,206	4,174	1,382	1,131	1,634	1,744	1,198	628	26,495
Waste Lateral Development	m	4,500	5,000	5,000	5,000	5,967	2,933	1,613	-	2	-	-	30,013
Waste Vertical Development	m	249	115	79	-	70	258	120	-	-	-	-	891

Figure 16.25 – Mine Plan / Period Plot (Longitudinal View)



Source: DRA, 2022

16.4.10 Mining Equipment

16.4.10.1 Mining Development Equipment

All lateral development, ramps and production drilling for the drift-and-fill stopes will be drilled using single-boom hydraulic jumbos. The single-boom jumbo was selected based on the majority of the production headings sizes being 4 m high by 4 m wide but with the capacity of drilling a heading of 5 m high by 5 m wide. For the 4 m by 4 m headings, the jumbo will drill an average of 45 drill holes of 45 mm diameter and four (4) 100 mm diameter reamed holes in the cut. This will result in a 4.2 m drilled round and break at 3.6 m. At full production, the mine will require four (4) single boom jumbos.

The development face will be loaded with explosives using a manual ANFO loader. This loader will also be used to load the production holes for the stopes. This equipment was selected for its low cost and productivity. At the peak of the mining development and production, the mine will require four (4) ANFO loaders on site.

16.4.10.2 Ground Support Equipment

The ground support will be done using stopers and jacklegs on a scissor lift platform. The Swellex bolts will be inflated using a Swellex portable pump. Two workers on the scissor lift platform will be required to complete the scaling, mesh installation and bolting of a heading. At full production, the mine will need five (5) scissor lift platforms, five (5) jacklegs, five (5) stopers and two (2) Swellex pumps.

16.4.10.3 Mucking Equipment

Mucking of the development round will be done using 10-tonne LHDs. In the early development phase, the waste from the development round will be mucked with an LHD to the closest waste transfer bay (i.e. remuck bay). From there, it will be loaded into 20-tonne haulage trucks and dumped at the surface on the transfer pad where the surface haulage truck will be loaded. The waste will also be mucked to the backfill circuit when stope production will begin.

The ore development and production rounds will be mucked with 10-tonne LHDs. For the levels below the 2000 Level, the ore will be loaded in 20-tonne haulage trucks. For the levels above 2000 Level, most of the ore will be dumped with the LHDs directly into the ore pass and rehandled on the 2000 Level. For haulage distances above 200 m, the LHDs will be loaded in 20-tonne haulage trucks.

At full production, the mine will require six (6) LHDs and three (3) 20-tonne haulage trucks.

16.4.10.4 *Production Drilling*

The production long-hole drilling will be accomplished by a down-the-hole hammer drill. The production drills will be mobile. The blast hole diameter will be 89 mm diameter and a 760 mm diameter boring head will be used to drill the first cut. The current plan is to ramp up production to one (1) drill working full time.

16.4.10.5 *Service and Auxiliary Vehicles*

The underground mining operation will be serviced by a full complement of service and auxiliary equipment such as boom trucks, graders, pick-up trucks.

16.4.11 MINING OPERATION

The mining operation strategy for the Zgounder project is to contract all waste capital development activities and to self perform all the production activities. Therefore, Aya Gold will be responsible to self perform production mining, ore handling, mine ventilation, mine backfill, and certain mine services to support the contractor and the self performed production. The contractor will be responsible to perform all the lateral and vertical development to access all the levels of the mine.

16.5 Cemented Rock Fill

16.5.1 BACKGROUND

Rock Fill (RF) is currently used as the backfill method. The expansion project underway, has indicated that a better underground backfill method needs to be implemented to facilitate ore exploitation.

DRA carried out a preliminary trade-off study on the type of backfill to be used at the Zgounder Project. The trade-off study considered Cemented Paste Fill (CPF), Cemented Aggregate Fill (CAF) and Cemented Rock Fill (CRF). CRF was chosen as the most suitable solution for the Project based on technical and financial considerations. This section provides details on the CRF backfill system.

16.5.2 CEMENTED ROCK FILL DESIGN CRITERIA

The backfill voids schedule developed from the latest mine design data, was used to determine the requirements of the cemented rock fill plant over the life of mine. The assumptions made using DRA's intensive backfill design database information, is presented in Table 16.11.

The backfill strength was determined by RockEng based on the stope sizes provided on the previous design (RockEng, 2022). Refer to Section 16.4.1.2d for details on the backfill strength requirements.

Table 16.11 – Backfill Design Input Data and Assumptions

Description	Units	Value
In-Situ Rock Density (SG)	SG	2.75
Backfill Void Fill Ratio	%	65
CRF Plant Utilisation	%	62.5
CRF Plant Excess Factor	%	10
Days per year	days	337
Days per month	days	28
Hours per month	hr	421
Utilisation Hours per day	hr	15
CRF Plant Operation	hpa	5,055
Uniaxial Compressive Strength (UCS)	kPa	235
CRF Density (w/w)	%	94
CRF Placed Density	SG	2.55
Annual Underground Ore Production	t	673,184
CRF Plant Annual Design Rate	m^3	182,984
CRF Plant Annual Design Rate	t	453,800
CRF Design Rate	m^3/h	36
CRF Design Rate	t/h	90

16.5.3 Backfill Requirement

Backfill requirement is approximately 182,984 m^3 per year, representing 65% void fill factor. The remaining 35% voids will be filled with waste rock or left empty. The CRF system has been designed to operate at volumetric flow rate of 36 m^3/h and throughput rate of 90 t/h.

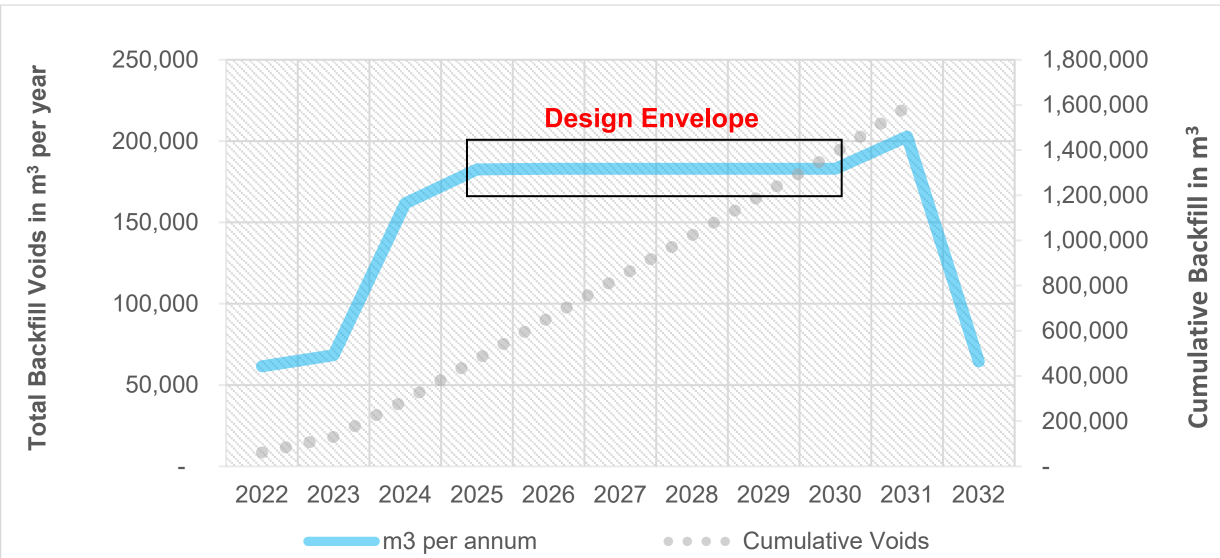
Backfill requirement increases from Year 2022 to Year 2025. From Year 2025 to 2030, backfill requirement reaches a 6 years plateau. Maximum backfill requirement occurs in the Year 2031 – this “end of life peak” is not used to size the CRF plant. Backfill requirement drops rapidly in the Year 2032.

Table 16.12 present the backfill requirement. Figure 16.26 illustrates backfill volumetric requirement over life of mine and Figure 16.27 presents backfill life of mine requirement in dry tonnes for CRF.

Table 16.12 – Backfill Requirement

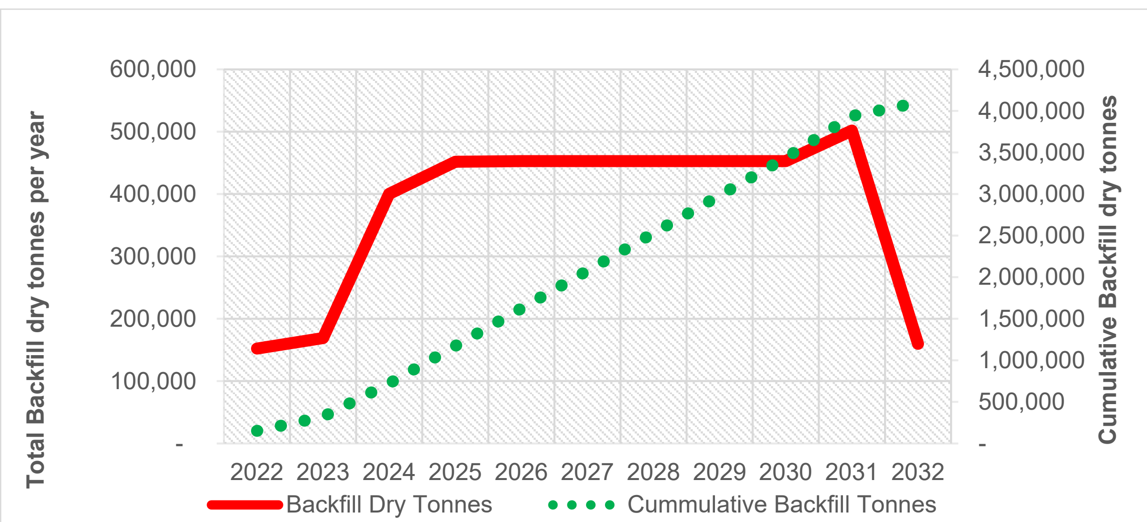
Description	Unit	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032
Backfill Flowrate	m³/h	12	14	32	36	36	36	36	36	36	40	13
CRF Throughput	t/h	30	33	79	89	90	90	90	90	90	99	32

Figure 16.26 – Backfill Voids Per Year and Cumulative Backfill Over LOM



Source: DRA, 2022

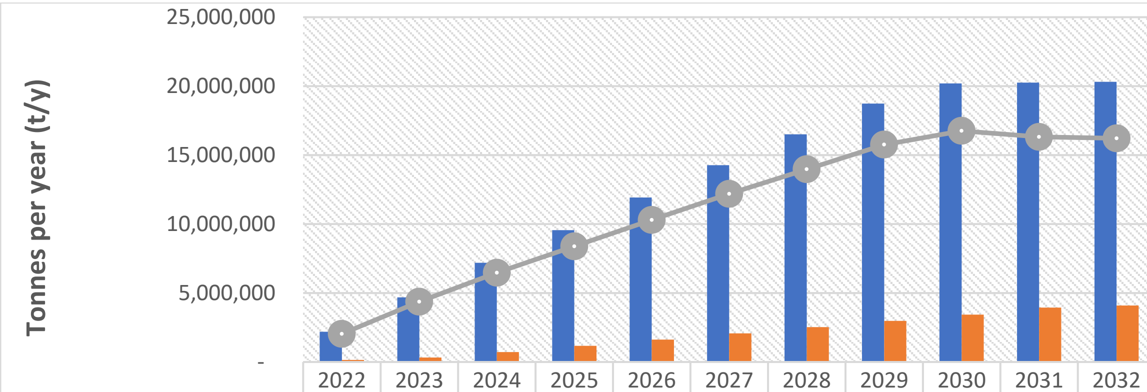
Figure 16.27 – CRF Backfill Dry Tonnes Per Year and Cumulative Backfill Over LOM



Source: DRA, 2022

There is an abundance of source material to produce CRF, most of which comes from the surface open pit and some from the underground waste development. Figure 16.28 shows that supply of backfill material far exceeds backfill demand over life of mine. Waste rock generation capacity from underground and the open pit is 400 t/h (on average) which is 310 t/h more than the backfill requirement of 90 t/h.

Figure 16.28 – Backfill Requirement v/s Supply over LOM



	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032
Waste t Cumulative	2,191,	4,691,	7,190,	9,557,	11,925	14,270	16,506	18,741	20,194	20,262	20,318
Backfill t Cumulative	152,25	321,38	721,63	1,173,	1,626,	2,079,	2,532,	2,985,	3,438,	3,939,	4,099,
Waste t Remaining	2,039,	4,369,	6,468,	8,384,	10,298	12,191	13,973	15,756	16,756	16,322	16,218

Source: DRA, 2022

16.5.4 CEMENTED ROCK FILL DESCRIPTION

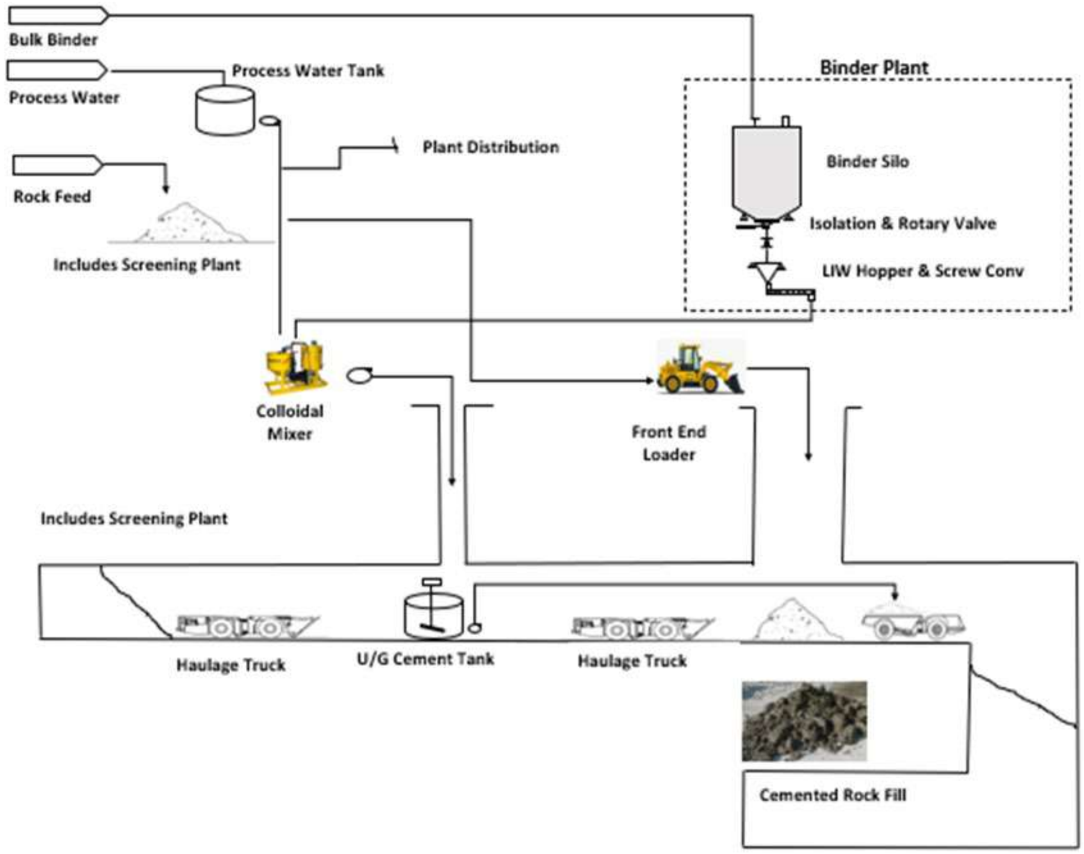
The CRF system includes but not limited to:

- Surface Infrastructure (Process Equipment and Mobile Equipment);
- Surface and UDS (Surface Water Lines, Boreholes, Rock Passes);
- Mobile Equipment as applicable;
- Other Infrastructure.

The CRF backfill system consist of waste rock from underground waste development and waste rock from the surface open pit. The feed is screened and sized using a rock breaker to achieve a top size of 150mm.

The surface paste backfill plant includes a 450t binder storage-transfer plant and a colloidal mixer for preparing cement slurry. Surface and underground mobile equipment used for rock handling purposes. The feed is delivered underground by a waste pass (refer to Section 16.4.4.9). Cement slurry is delivered underground by a slurry borehole. Figure 16.29 depicts the CRF backfill system.

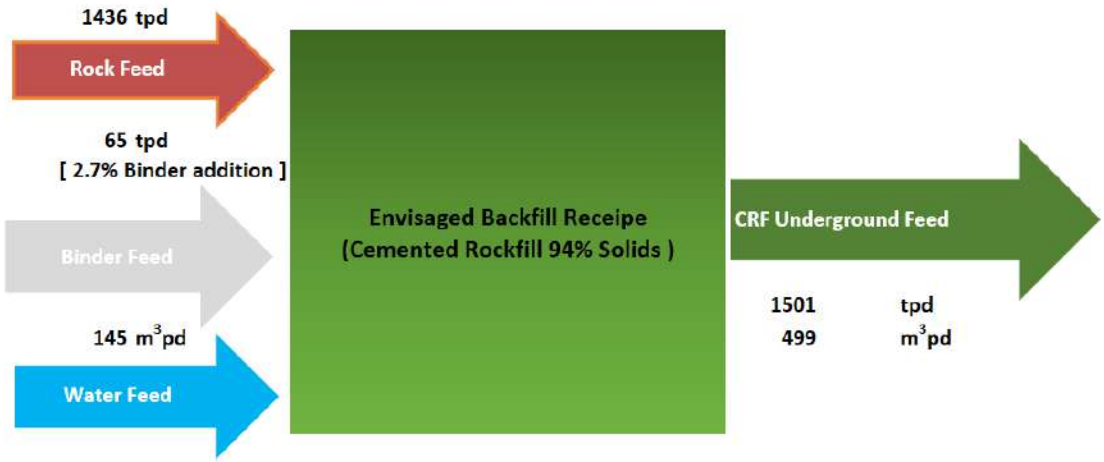
Figure 16.29 – CRF Backfill System



Cement slurry will be brought into an underground cement tank complete with mechanical agitation. This tank will be equipped with appropriately designed duty-standby positive displacement pumps. Waste rock from underground development and the open pit will be transferred into the haulage trucks. Cement slurry will be pumped into the haulage trucks via spray bars to produce cemented rockfill.

Cement slurry positive displacement pumps will be capable to distribute cement slurry to the furthest stopes. Figure 16.30 depicts the preliminary mix design.

Figure 16.30 – CRF Backfill System Preliminary Mix Design



16.6 Mine Ventilation

Howden were engaged to assist Aya with the ventilation design work. Howden was responsible for developing the strategy for an underground ventilation system.

The fresh air requirement has been established for the mine. Table 16.13 depicts the factors used for contaminants dilution.

Table 16.13 – Fresh Air Requirement

Regulatory Fresh Air Demand (m³/s/kW)	0.06
NO2/Nox (%)	0.2
TLV CO (ppm)	25
TLV DPM (mg/m3)	0.1
TLV NO2 (ppm)	0.5

In estimating the aggregate rate of fresh airflow for the entire mine, a utilisation rate has been applied to account when machines may be mechanically unavailable, or simply not in use.

Table 16.14 – List of Mobile Equipment

Equipment Type	Power (kW)	Power (hp)	Quantity
Jumbo	55	73	2
Production Drill	110	147	1

Equipment Type	Power	Power	Quantity
	(kW)	(hp)	
Loader	235	313	6
Truck	224	299	6
Scissor Lift	120	160	3
Boom Truck	120	160	1
Fuel-Lube	120	160	1
Forklift	120	160	1
Grader	142	189	1
Mine Pickup	95	127	3
Mine Tractor	44	59	3

Multiple iterations to the mine plan and equipment list were considered. The utilisation rates are, 80% for production equipment, 50% for most service equipment, and 25% for equipment that are mainly operating on electric power such as Jumbos and production drills. Based on this utilisation rate, DRA established a maximum fresh air demand of 203 m^3/s (430 kcfm) to meet the demand required for all zones. Air fresh demand was calculated by adding 15% contingency to the total air requirement assuming that all the equipment was available and in use in the mine.

To determine the operating fan pressure, a worst-case scenario was established by directing most of the fresh airflow towards the deepest production levels assuming a 60-40% air fresh ratio between lower levels (2000L to 1648L) and upper levels (2020L to 2092L). To reach this ratio, all the upper level regulators were set to inject a minimum amount of fresh air (10 kcfm).

Table 16.15 presents the fan operating point for the mine.

Table 16.15 – Fresh Air and Pressure Requirement

Zone	Max Fresh Air Req.	Max Collar Pressure Req.
	m^3/s (kcfm)	Pa (in. w.g.)
East Exhaust	203 (430)	2,500 (10)

16.6.1 Zgounder Ventilation Network and Infrastructure Description

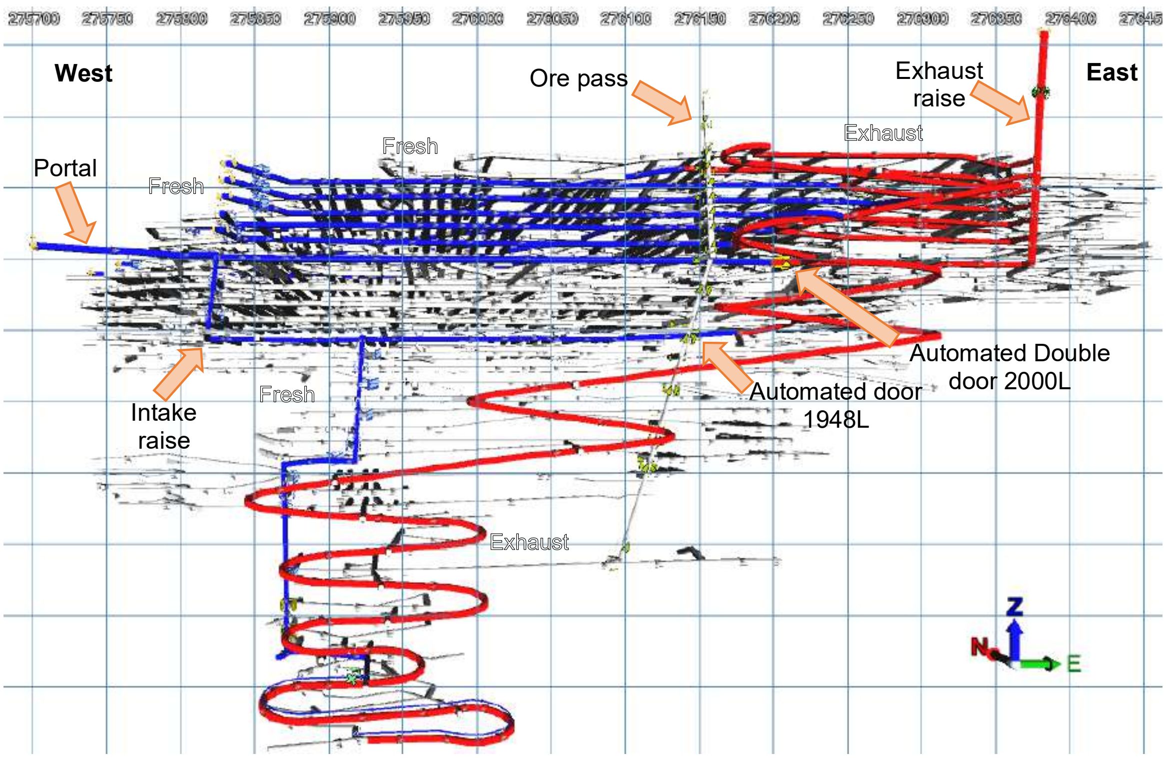
The Zgounder mine ventilation system consists of twelve independent air intakes located on the west area of the mine and an exhaust raise located on the East area. A parallel fan arrangement of 450 kW (600 hp) will be supplied.

The ramp will serve as return air pathway towards the exhaust raise located to the east.

Two (2) doors will control fresh airflow on Level 2000 and Level 1948 preventing fresh air from being forced out by the exhaust raise before mixing with contaminants.

Figure 16.31 shows a longitudinal view of the Zgounder mine ventilation network.

Figure 16.31 – General View of the Zgounder Ventilation Network



Source: Howden, 2022

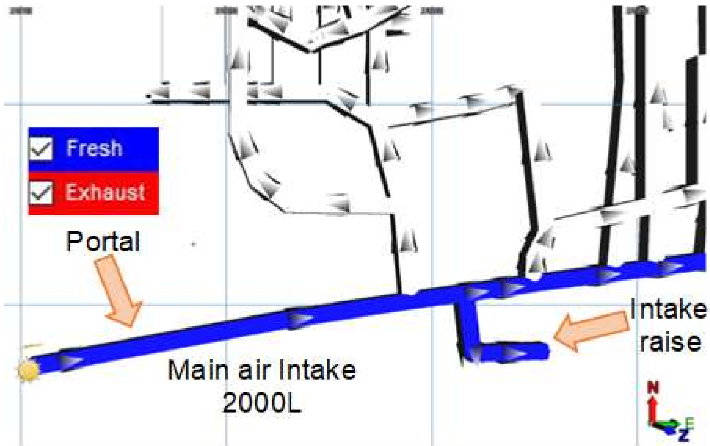
The ventilation raises have all been sized at 3 x 5 m (Alimak).

An ore pass connects the surface to Level 1776. This raise was not considered in the ventilation network since it will remain blocked continuously.

Eleven (11) drifts connected to surface provide fresh air from Level 2020 to Level 2080. The main fresh air intake for the lower levels is a drift that connects the portal to an Intake raise located on Level 2000 west. This airway injects air from Level 2000 to Level 1708 through a raise breaking through each one of the production levels.

Figure 16.32 shows a view of the portal and the Intake raise.

Figure 16.32 – Portal and Intake Raise View

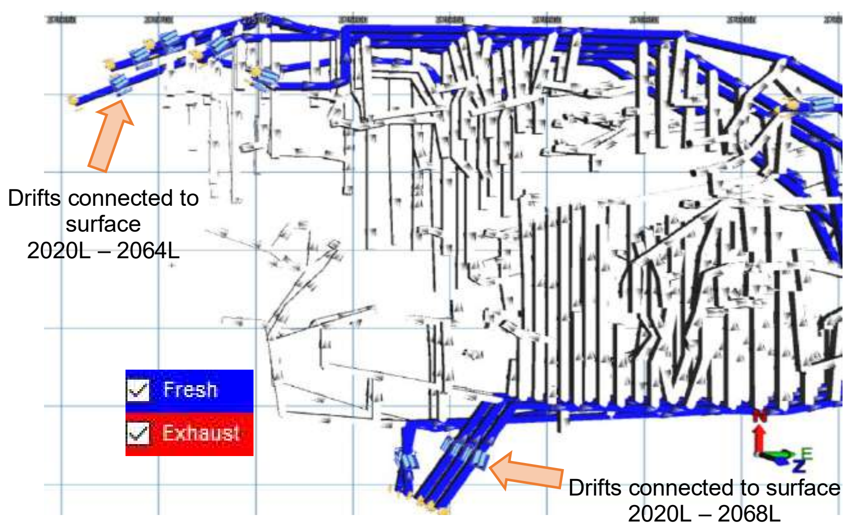


Source: Howden, 2022

Bulkheads with regulators will control the airflow on the intake upper levels. The exhaust of contaminated air on this zone will be conducted through the ramp (from 2000L to 2092L) and the drifts connected to the Exhaust raise. Auxiliary ventilation will be required on the headings to ensure a proper fresh airflow.

Figure 16.33 shows a view of regulators on the intake drifts.

Figure 16.33 – Intake Drifts View



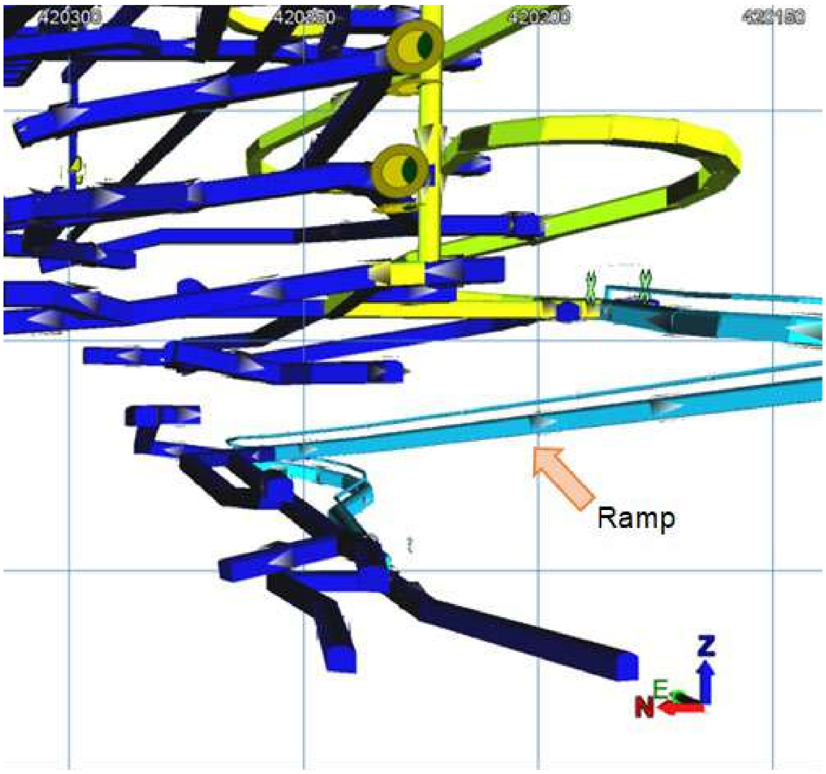
Source: Howden, 2022

Regulators will control the lower levels air intake directing the airflow towards the return air pathway of the ramp through each level access. Auxiliary ventilation will be required on the headings to ensure a proper fresh airflow.

In addition, bulkheads with drop board regulators will control the air intakes on the lower levels. On these levels, the exhaust of contaminated air will be conducted through the exhaust raise and through each level access. Auxiliary ventilation will be required on the headings to ensure a proper fresh airflow.

Figure 16.34 shows a view of drop board regulators on the Intake raise of levels 1724 and 1744.

Figure 16.34 – Drop Board Regulators View



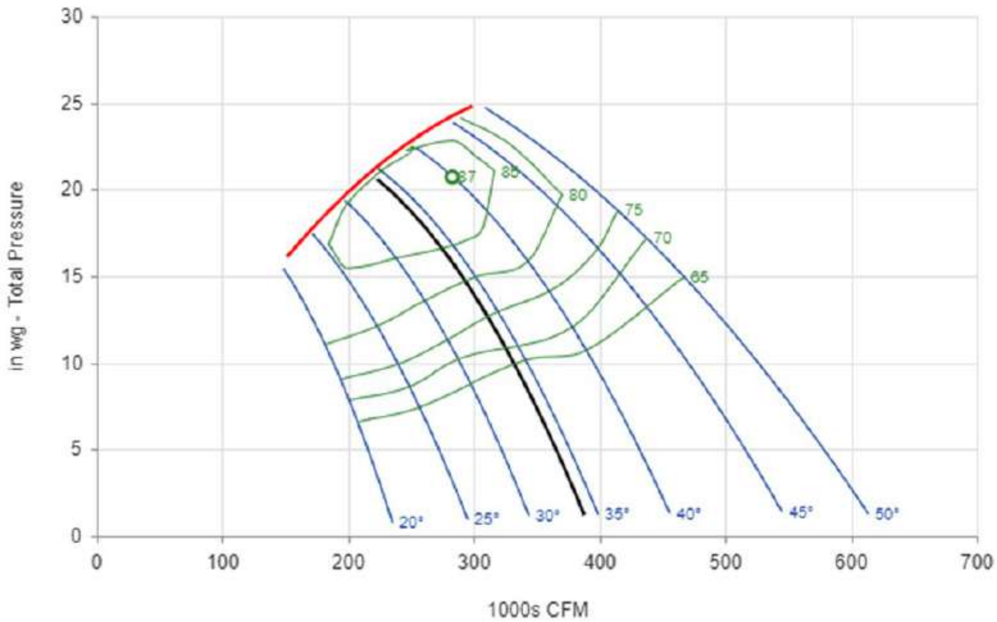
Source: Howden, 2022

16.6.2 Zgounder East Exhaust Fan Data

The fan model proposed for East Exhaust is 8400-AMF-5500. A parallel fan arrangement is set up to match the determined flow of 430 kcfm and 10 in w.g. pressure required. Fans will therefore be equipped with variable speed drives to be able to accommodate advanced ventilation controls to optimize its speed and consequently reduce energy costs.

Figure 16.35 shows the operating point for the Exhaust fans.

Figure 16.35 – Operating Point for East Exhaust, Parallel Configuration



Source: Howden, 2022

16.6.3 Production and Auxiliary Ventilation

The production auxiliary ventilation system has been designed based on the following assumptions:

Vehicle assumptions:

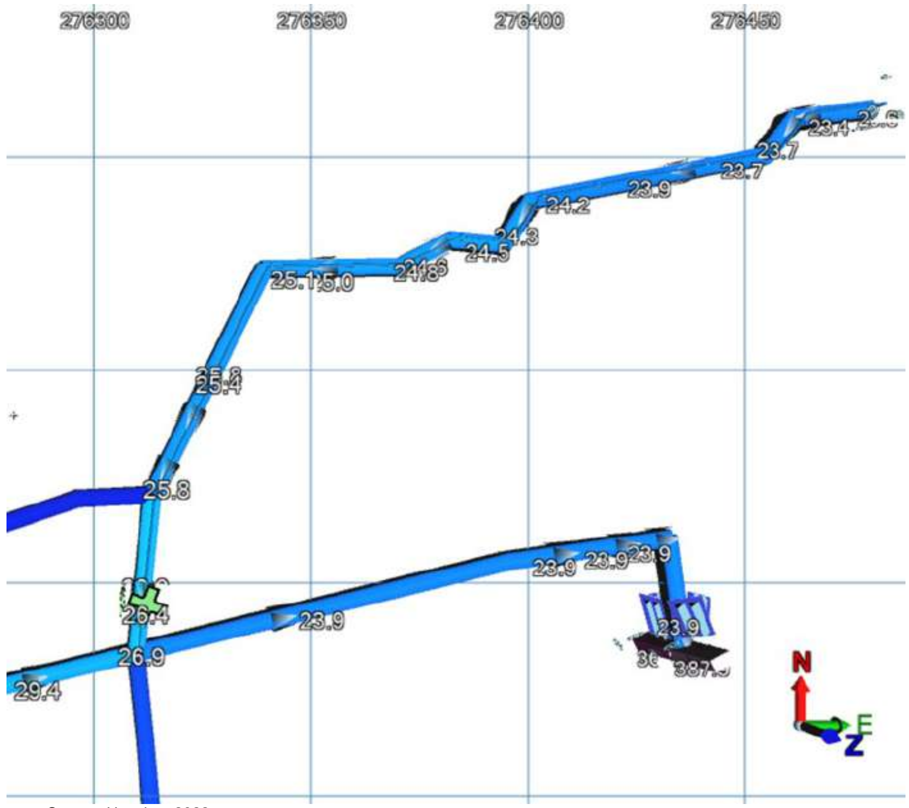
- Trucks do not have access to the mineralized zones and will not pass beyond the access of each level.
- LHD will have access to mineralized zones and the ore pass.
- The fresh air requirement for a worst-case LHD on the heading is 11 m^3/s (23.3 kcfm).
- The fresh air requirement for a worst-case LHD and a truck on the ramp and main access is 22 m^3/s (46.6 kcfm).

Duct assumptions:

- The ramp and main access auxiliary system will use 0.96 m (38 in) flexible ducting (friction factor = 0.0029 kg/m^3).
- The heading auxiliary system will use 0.66 m (26 in) flexible ducting (friction factor = 0.0029 kg/m^3).

Figure 16.36 shows a view of a typical ducting layout.

Figure 16.36 – Isometric View of Typical Ducting Layout on a Level to Heading



Source: Howden, 2022

16.7 Combined Mine Plan

Table 16.16 presents the combined mine plan for the open pit mine, the tailings reclamation, and the underground mine.

Table 16.16 – Combined Mine Plan

		Unit	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	Total
Open Pit Mine														
Ore	Pit to Mill	kt	-	-	200.9	192.6	185.8	251.0	248.6	234.2	202.8	-	-	1,516
	Pit to Low-Grade Stockpile	kt	-	8.1	20.3	113.2	98.9	36.7	36.7	29.2	29.3	-	-	372.4
	Pit to High-Grade Stockpile	kt	-	62.6	48.2	80.8	45.7	28.9	11.3	4.1	8.4	-	-	290.2
	Low-Grade Stockpile to Mill	kt	-	-	14.5	0.1	7.6	30.1	9.5	45.3	76.8	-	188.6	372.4
	High-Grade Stockpile to Mill	kt	-	-	46.6	95.3	94.7	6.9	29.9	8.4	8.4	-	-	290.2
	Total Mill Feed	kt	-	-	262.0	288.0	288.0	288.0	288.0	288.0	288.0	-	188.6	2,178.6
	Silver Grade at Mill	g/t	-	-	350.0	303.3	202.2	279.5	311.7	231.8	230.1	-	61.8	253.5
Waste		kt	-	3,171.2	4,320.0	3,940.0	3,880.7	3,295.8	2,509.6	1,393.3	937.4	-	-	23,447.9
Total Mined		kt	-	3,241.9	4,589.5	4,326.5	4,211.1	3,612.4	2,806.2	1,660.9	1,177.9	-	-	25,626.5
Stripping Ratio		w/o	-	44.8	16.0	10.2	11.7	10.4	8.5	5.2	3.9	-	-	10.8
Tailings Reclamation														
Ore	Tonnage to Mill	kt	-	-	-	-	-	-	-	-	-	288.0	30.8	318.8
	Silver Grade	g/t	-	-	-	-	-	-	-	-	-	80.5	39.5	7.6
Underground Mine														
Ore	Tonnage to Mill	kt	226.3	251.4	594.9	671.6	673.1	673.1	673.2	673.2	673.2	745.9	237.5	6,093.4
	Silver Grade	g/t	264.4	286.4	284.5	249.8	283.3	291.9	322.8	243.8	257.6	225.2	208.5	267.5
Waste		kt	566.3	655.0	671.7	663.0	704.2	307.3	196.7	136.2	156.9	104.1	33.0	4,194.5
Total Mined		kt	792.6	906.44	1266.59	1334.61	1377.33	980.44	869.86	809.42	830.14	849.97	270.5	10,287.9
Combined Mine Plan														
Ore to ROM Pad		kt	226.3	251.4	856.9	959.6	961.1	961.1	961.2	961.2	961.2	1,033.9	456.9	8,590.8
Silver Grade		g/t	264.4	286.4	304.5	265.9	259.0	288.2	319.5	240.2	249.4	184.9	136.6	256.7
Waste		kt	566.3	3,826.2	4,991.7	4,603.0	4,584.9	3,603.1	2,706.3	1,529.5	1,094.3	104.1	33.0	27,642.5
Total Mined		kt	792.6	4,077.6	5,848.6	5,562.6	5,546.0	4,564.2	3,667.5	2,490.7	2,055.5	1,138.0	489.9	36,233.3

17 RECOVERY METHOD

This Section of the Report covers two (2) areas of the overall process design. Section 17.1 describes the existing Plant Nos 1 and 2 in place and DRA's current mandate in Section 17.2 provides the design for new Processing Plant # 3.

The processing complex at Zgounder will comprise of the following three (3) facilities:

- Existing Pant #1 (cyanidation plant);
- Existing Plant #2 (flotation plant); and
- New Processing Plant #3.

17.1 Mineral Processing Facility Design

The existing Plant # 1 (Cyanidation Plant) operates at 180 t/d of mineralised material feed and is operated independently of the flotation plant. The silver sludge recovered from this cyanidation plant will be transferred to the New Plant #3 for refining.

The existing Plant # 2 (Flotation Plant) treats 500 t/d of mineralised material, its silver concentrate product will be transferred to the New Plant #3 for leaching and refinery. The flotation tails from Plant #2 will be transferred to the New Plant #3 to be combined with its new flotation tailings in the Leaching-CIP circuit.

Table 17.1 summarises the production data of Plants #1 and 2.

Table 17.1 – Production Data of Plants #1 and 2

Description	Unit	2021				
		Q1	Q2	Q3	Q4	Total
Treated - Ore	t	13,273	14,375	14,544	16,074	58,266
Average head grade - Ore	ppm	512	484	418	405	452
Treated - conc	t	-	-	-	-	-
Head grade - conc	ppm	-	-	-	-	-
Recovery	%	83.73	78.79	79.17	82.11	81.01
Ag Produced	oz	190,621	174,786	154,331	172,654	692,392
Treated	t	35,200	41,943	39,325	49,781	166,249
Average head grade	ppm	214	233	177	197	205
Recovery	%	81.88	84.24	82.44	82.66	82.90
Ag Produced	oz	198,511	264,364	184,293	261,087	908,254

Source: Aya (2021)

17.1.1 PLANT # 1 - CYANIDATION PLANT

The existing Plant # 1 is composed of two (2) identical production lines, referred to as the North Line and South Line, producing silver ingots.

Silver bearing mineralized material from the mine is crushed using a 55 kW jaw crusher and a 75 kW cone crusher.

Crushed mineralised material is stored in two silos, one each dedicated to the two (2) identical production lines (North and South). The crushed mineralised material is further reduced in size by a grinding circuit in closed-circuit with hydro-cyclones. The pump box receives the 132 kW ball mill discharge and a downstream thickener overflow. The slurry is pumped to the cyclones where the underflow is fed to the ball mill feed chute along with the conveyed crushed ore from the silo. The cyclones overflow is transferred to a thickener prior to cyanidation.

The cyclones overflow of each line is densified in its own thickener, whereas the underflow is pumped to the first cyanidation tank of each dedicated leaching circuit. The thickener overflow, as previously noted, is recirculated back upstream to the pump box of the grinding circuit.

Each cyanide leaching circuit (North Line and South Line) consists of four identical agitated tanks operating in series. NaCN and H_2O_2 are added to the leaching circuit.

From the last leaching tank, the material is transferred to five counter-current decantation ("CCD") thickeners. Each thickener is 12 m in diameter. The overflow solution from each thickener is enriched by pumping it upstream to the preceding thickener. The slurry underflow is depleted from its silver content by being pumped downstream to the next thickener. The barren underflow from the final CCD thickener is pumped to a tailings pump box, from where it is pumped to the existing tailings pond. The overflow solution from the first CCD thickener is transferred for silver recovery.

The silver recovery section is in a covered building. The North Line and South Line overflow solutions are combined in this section. Silver recovery is done through the Merrill-Crowe process which features zinc dust cementation.

The filter-press sludge (silver mud) from the Merrill-Crowe is dried, calcined in a static furnace, and then smelted. Silver ingots are the final product from the furnace.

Table 17.2 summarises the existing comminution equipment at the cyanidation plant and Table 17.3 lists the reagents and grinding media that are currently being utilised.

Table 17.2 – Existing Comminution Equipment

Equipment	Motor (kW)
Jaw crusher	55
Cone crusher	75
Ball mill (North Line)	132
Ball mill (South Line)	132

Source: Aya (2021)

Table 17.3 –Reagents and Grinding Media Used

List of Consumables
Sodium cyanide
Lime powder
Zinc powder
Lead nitrate
Balls 70 mm
Borax
Hydrogen peroxide
Iron sulphate

Source: Aya (2021)

17.1.2 Plant # 2 - Existing Flotation Plant

The existing plant # 2 beneficiates ore from the mine into a silver concentrate that is bagged on-site. Its main processing unit operations are crushing, grinding, flotation, and dewatering.

Run-of-mine ore is fed into a bin either by haul trucks or by a front-end loader. The mineralized material is withdrawn from the bin by a feeder to feed a 75 kW primary jaw crusher.

The jaw crusher crushes the mineralized material, which is then conveyed to a dry crushing product screen with two decks. The oversize of the top deck falls by gravity into a 37 kW jaw crusher. The crushed product from this jaw crusher is combined with the screen bottom deck's oversize. Both streams combined are conveyed to a 160 kW secondary cone crusher.

The secondary cone crusher product combines with the primary jaw crusher product and is circulated back to the double-deck screen in closed circuit. This screen undersize is the crushed product from the crushing area and is conveyed to a crushed mineralized material silo.

From the crushed mineralized material silo, the product is reclaimed by vibrating feeders and conveyed to the primary grinding circuit. This circuit comprises of a 320 kW primary ball mill, which operates in closed circuit with a spiral classifier. The overflow product from this primary grinding circuit contains the fines and is transferred to a secondary grinding circuit.

The secondary grinding circuit consists of a 280 kW ball mill in closed circuit with a cluster of hydro-cyclones. The primary grinding circuit product combines with the secondary ball mill discharge into a pump box. The slurry is pumped to the cyclones. The purpose of this last grinding step in the existing process plant is to reach the final particle size for the flotation process.

The cyclones underflow is reground in the secondary ball mill, whereas the cyclones overflow gravitates to a conditioning tank. Reagents are added to the agitated tank in order to condition the surfaces of the material particles prior to the flotation process.

The conditioned slurry flows to the rougher flotation bank which consists of three (3) mechanically agitated cells to start the beneficiation process. Rougher concentrate overflows from the flotation cells into a launder and is fed to the cleaner flotation stage for further upgrading. The rougher tailings are fed to the scavenger flotation circuit which consists of a total of seven (7) cells.

The scavenger concentrate is circulated back to the rougher feed. The scavenger tailings constitutes the final tailings and are pumped to the existing Tailings Storage Facility (TSF).

There are two (2) stages of cleaner flotation. The first cleaners are a bank of two (2) cells, whereas the second cleaner stage is composed of one (1) flotation cell. The cleaner tailings are circulated back to the rougher flotation feed. The cleaner concentrate is the final flotation concentrate and is pumped to the existing concentrate thickener. The thickener overflow is returned into the system and re-used as process water.

The thickened silver concentrate underflow is pumped to a horizontal plate-and-frame pressure filter which targets 12% moisture in the filter cake. The cake is discharged by opening the plates unto a belt conveyor. The silver concentrate is then bagged for shipment.

Table 17.4 summarises the existing and operating comminution equipment at the flotation plant, while Table 17.5 provides a list of reagents and grinding media used.

Table 17.4 – Existing and Operating Comminution Equipment

Equipment	Motor (kW)
Primary jaw crusher	75
Secondary cone crusher	160
Tertiary jaw crusher	37
Primary ball mill	320
Secondary ball mill	280

Source: Aya (2021)

Table 17.5 – List of Reagents and Grinding Media Used

List of Consumables
Balls 100 mm
Balls 60 mm
Ammonium Dibutyl Dithiophosphate
Sodium Butyl Xanthate
Copper sulphate
Pine oil

Source: Aya (2021)

17.1.3 New Processing Plant #3

The new mineral processing facility has been designed to treat mineralised material at a nominal throughput rate of 2,000 tpd and at a head grade of 210 g/t of silver annually.

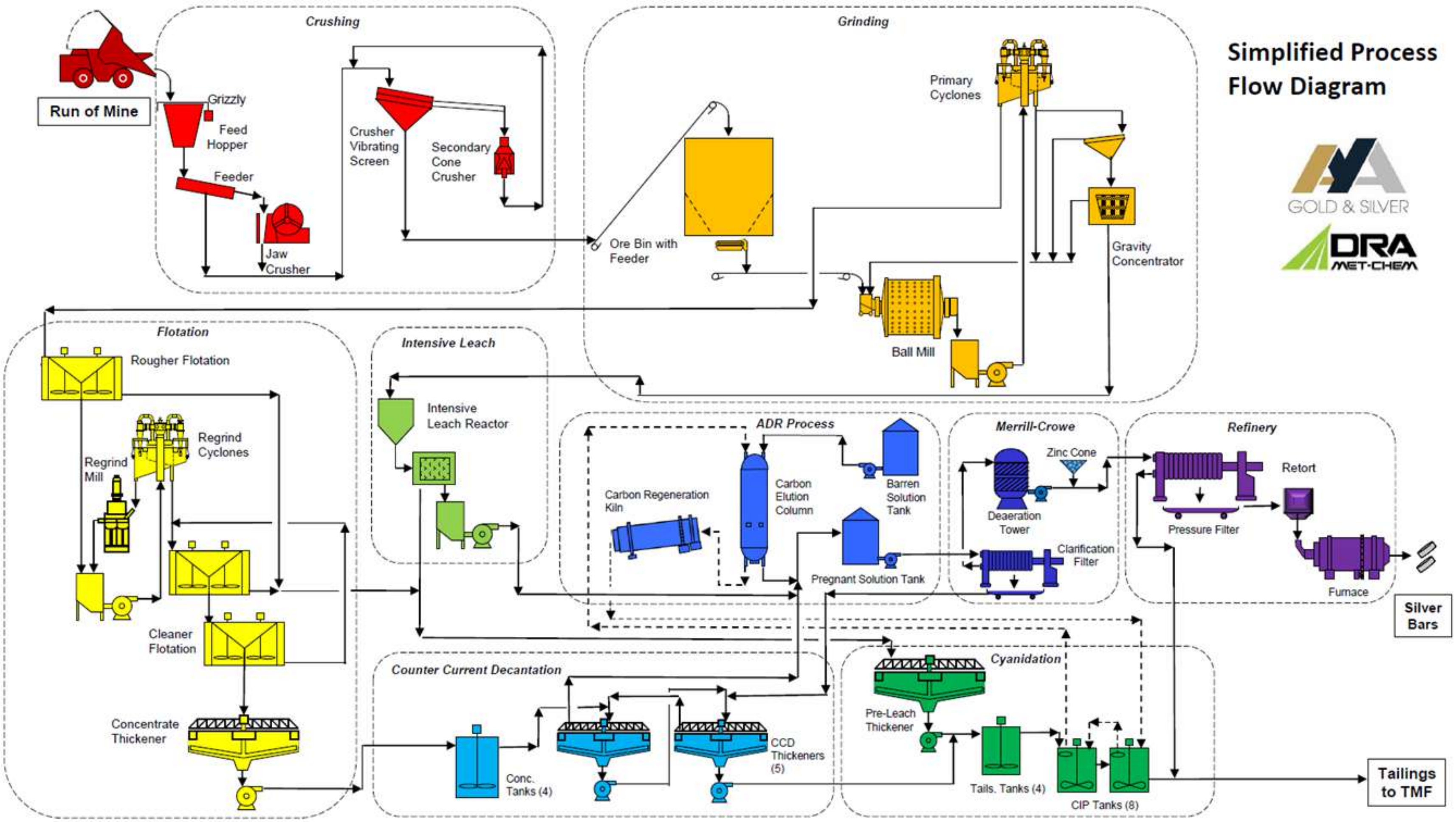
The new plant will treat 2,000 tpd and comprises the following circuits:

- Crushing: two stage crushing circuit closed out by a vibrating screen;
- Grinding: single stage ball milling circuit closed out by hydrocyclones;
- Gravity Separation: integrated within the grinding circuit with feed from diverting some cyclone underflow and gravity tailings returning to the ball mill feed chute;
- Flotation: One rougher stage followed by regrinding of the rougher concentrate and three subsequent cleaner stages;
- Intensive leach of the gravity concentrate with the pregnant solution combining with two other high grade solutions (eluate and CCD overflow) ahead of the Merrill-Crowe section;

- Concentrate Leaching and Counter-Current Decantation (CCD): Cyanidation at elevated cyanide concentrations followed by liquid-solid separation in CCD thickeners with the CCD overflow reporting to the Merrill-Crowe and the CCD underflow reporting to the tailings leach;
- Tailings Cyanidation and CIP: leaching of flotation tailings and other tailings streams followed by adsorption onto activated carbon in a carousel CIP circuit;
- ADR Process: an elution circuit to strip silver from the carbon and regenerate it for re-use;
- Merrill-Crowe: to recover silver from solution through cementation; and
- Refinery: drying and smelting of sludge to produce doré silver bars.

A simplified process flow sheet is depicted in Figure 17.1 and summarises the flow routing within the major circuits of the new process plant facility. More detailed process information is presented in Section 17.3 of the Report.

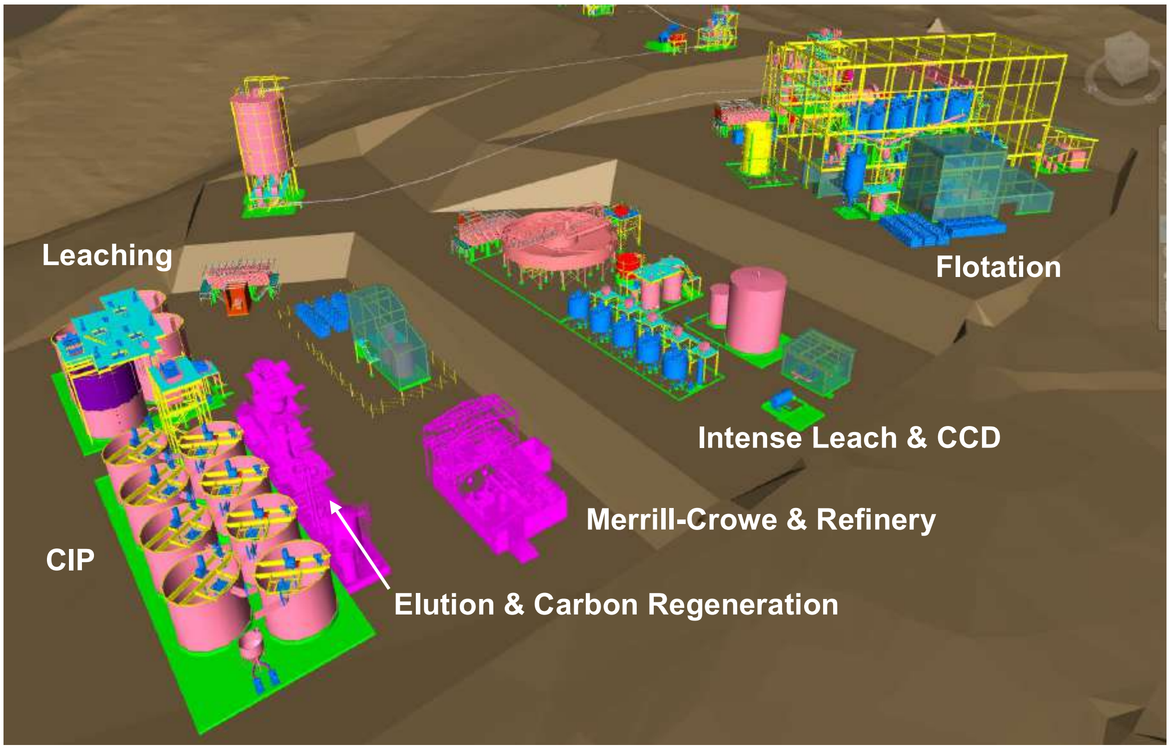
Figure 17.1 – Simplified Flow Sheet (New Plant #3)



Source: DRA, 2022

Figure 17.2 depicts a 3D model cut of the new processing plant # 3.

Figure 17.2 – 3D Model Cut of the New Processing Plant #3



Source: DRA, 2022

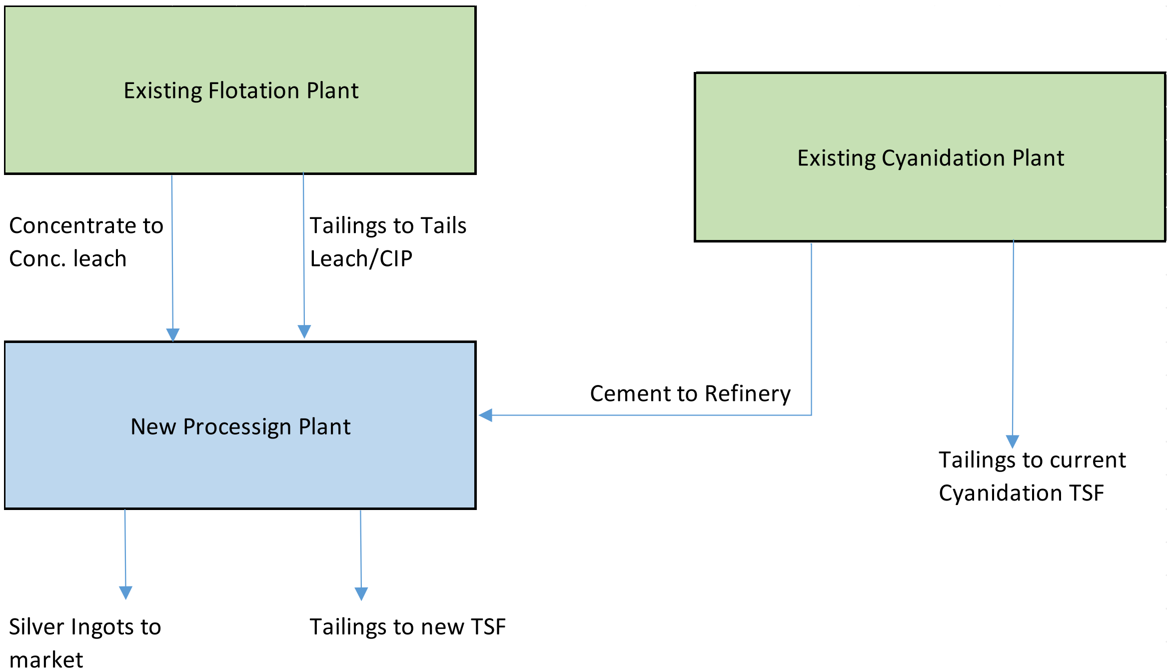
17.1.4 INTEGRATION OF THE THREE PROCESSING PLANTS

Cementation sludge, produced by the existing cyanidation plant, will be transported to the new plant's refinery where it will be smelted. The tailings produced by this existing cyanidation plant will continue to be deposited in the existing Tailings Storage Facility (TSF).

Flotation concentrate, produced by the existing flotation plant, will be pumped to the new plant's concentrate leach section where it will combine with the flotation concentrate produced by the new plant. Tailings from the existing flotation plant will be pumped to the tailings leach section of the new plant, where it will combine with the tailings of the new plant and leached to recover residual silver.

All of the silver produced by the three (3) facilities will be poured as doré bars (ingots) in the new Processing Plant #3 refinery. Combining the final products allows for additional silver extraction for the ore treated through the existing plants and hence increased overall silver recovery. Figure 17.3 illustrates how the three (3) processing facilities ties-in together. Refer to Section 18 for site layout drawing.

Figure 17.3 – Simplified Block Flow Diagram of Integrated Plants



Source: DRA, 2022

17.2 Key Design Criteria – New Plant #3

The crusher and concentrator will operate 24-hour per day, 7-day per week, 52-weeks per year. The crusher plant will operate at 68.5% and the concentrator targeted operating availability is 91.3%.

The concentrator capacity has been established at an average rate of 2,000 dry t/d or a nominal throughput rate of 91.3 dry t/h. Table 17.6 summarises the design basis for the plant.

Table 17.6 – Main Design Criteria: New Plant # 3

Parameter	Unit	Value
Total ore processing rate	dry t/y	730,000
Average concentrator ore processing rate	dry t/d	2,000
Silver ore grade (average)	g/t	210
Crusher operating time	%	68.5
Concentrator operating time	%	91.3
Nominal ore processing rate	dry t/h	91.3
Flotation concentrate grade from existing plant #2	Ag g/t	4,400

Parameter	Unit	Value
Silver Recovery		
Gravity concentrator	%	16
Gravity concentrate leach	%	96
Flotation	%	84
Flotation concentrate leach	%	96
Flotation tailings leach	%	65
Silver recovery (overall)	%	93.4

Table 17.7 summarises the design basis for individual unit operations. Note that the values in this table are not for nominal operations, but for the design case.

Table 17.7 – Design Criteria Summary – Unit Operations

Parameter	Unit	Value
Comminution		
A x b (15th percentile)	-	23.2
CWi (85th percentile)	kWh/t	12.6
BWi (85th percentile)	kWh/t	23.1
RWi (85th percentile)	kWh/t	22.7
Crushing		
ROM feed top size	mm	500
Sec. crusher feed top size	mm	200
Sec. screen circulating load	%	147
Fine ore bin capacity	h	17.3
Grinding		
Gravity circuit mass pull	%	0.05
Cyclone underflow split to gravity circuit	%	39
Primary grind P_{80}	µm	100
Primary mill specific energy	kWh/t	20.6
Circulating load	%	321
Flotation		
Flotation mass pull (from new feed)	%	3.5
Rougher concentrate mass pull	%	28.8
Rougher stages	#	1

Parameter	Unit	Value
Rougher residence time	mins	50
Cleaner stages	#	3
First cleaner residence time	mins	17.5
Second cleaner residence time	mins	9
Third cleaner residence time	mins	7.5
Regrind mill P80	µm	45
Regrind circuit circulating load	%	230
Regrind cyclones overflow density	% solids	27
Flotation Concentrate Cyanidation		
Flotation concentrate leach residence time	h	48
Concentrate leach tanks	#	4
CCD wash ratio	-	2
Concentrate thickener underflow density	% solids	50
Flotation Tailings Cyanidation		
Flotation tailings leach residence time	h	36
Concentrate thickener underflow density	% solids	50
Flotation tailings leach number of tanks	#	4
Adsorption		
Configuration		Carousel CIP
Number of stages	#	8
CIP loaded carbon grade	g/t	4,000+-500
Carbon advancement (nominal to design)	t/d	12 to 19
CIP residence time per stage	h	1.2
CIP carbon inventory per stage	t	12
Carbon ADR Circuit and Refinery		
Configuration		Split AARL
Carbon batch size	t	12
Eluate flowrate	BV/h	2
Carbon kiln specific energy requirement	kWh/kg	2
Total volume of solution to Merrill-Crowe circuit	m^3/d	425
Cyanide Detoxification		
Hydrogen peroxide injection into reclaim water line		
H_2O_2 addition rate	kg/kg CNWAD	2.6

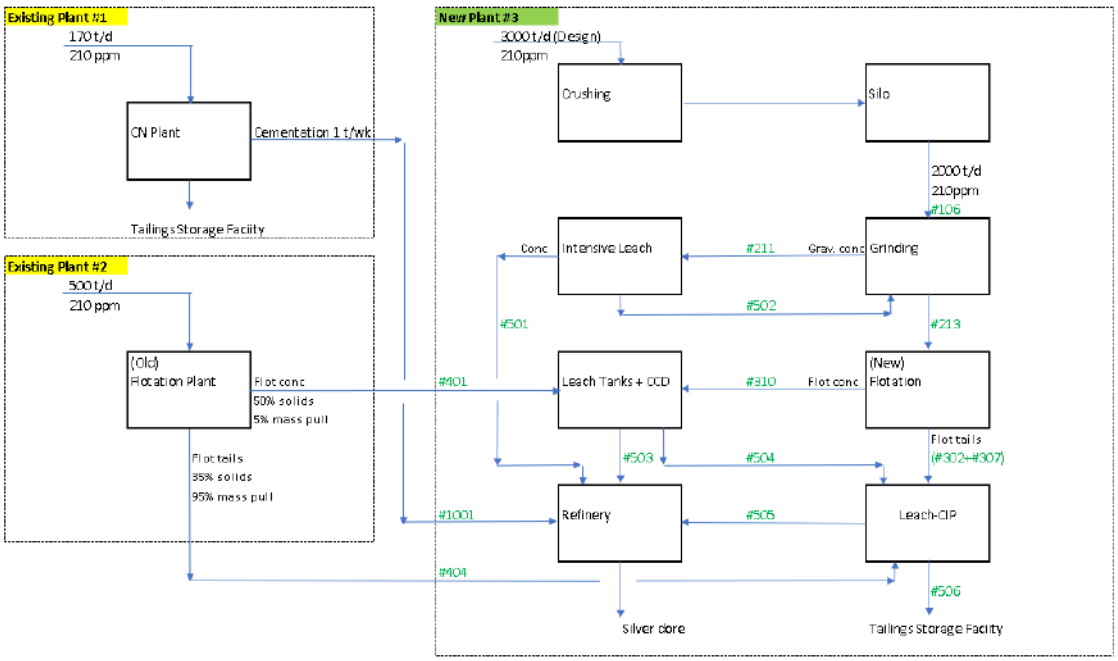
Parameter	Unit	Value
Major Reagents and Consumables		
Ball mill media wear rate	g/kWh	32
Regrind mill media wear rate	g/kWh	81
Cyanide	kg/t	5.8
Lime	kg/t	0.79
Flocculant (conc. thickener)	g/t	144
Flocculant (tails thickener)	g/t	72
PAX (collector)	g/t	167
Aero (activator)	g/t	81
MIBC (frother)	g/t	35

17.2.1 MASS BALANCE AND WATER BALANCE

A process plant mass balance was calculated based on the key design criteria above and the process flow sheet as depicted in Figure 17.4. Figure 17.4 expands on the simplified block flow diagram (BFD) presented in Figure 17.1 by adding more internal details showing the links between unit operations.

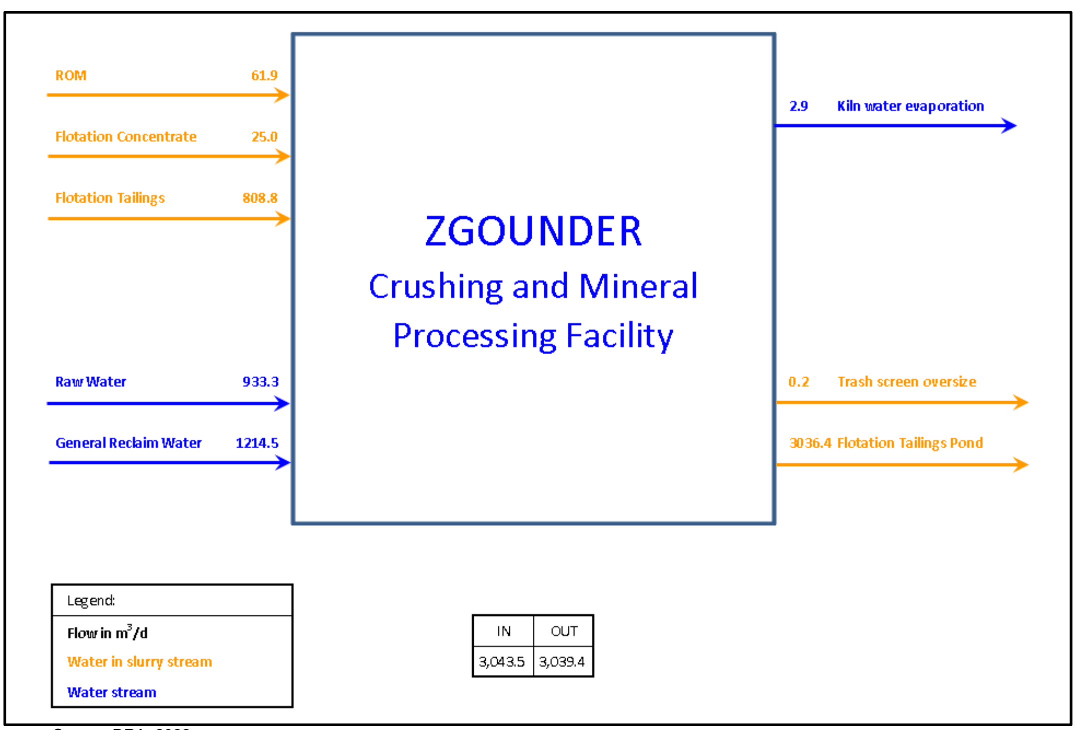
Figure 17.5 depicts a simplified water balance. There are three (3) sources of water in slurries, these being the surface water entering the plant with the ROM ore, water entering the plant with the flotation concentrate and tailings slurries from the existing flotation plant. These account for approximately 29% of the water used by the new processing plant. Raw water, at 933 m^3/day, will be abstracted from the raw water source and is used to mix reagents, for water spraying duties and to make-up the process water shortfall. Raw water accounts for approximately 31% of the total water consumption. Reclaim water, at 1,215 m^3/day, is pumped from the supernatant pond on the TSF to the process water tank and it constitutes the remaining 40% of the total water demand.

Figure 17.4 – Block Flow Diagram



Source: DRA, 2022

Figure 17.5 – Water Balance Summary – New Process Plant #3



Source: DRA, 2022

17.3 Process Description

The simplified process flow sheet is depicted in Figure 17.1 in Section 17.1. The mineral processing facility has nine (9) areas: crushing, grinding, flotation, intensive leach reactor, counter-current decantation, cyanidation, ADR process, Merrill-Crowe, and refinery.

17.3.1 Crushing

The run-of-mine (ROM) ore is transported by truck to the ROM hopper, where the rock breaker assists in crushing any oversize ore unable to pass through the static grizzly. The ore then passes over the primary feeder, with the oversize reporting to the primary jaw crusher, while the underflow drops directly onto the sacrificial conveyor. The primary crusher product joins the feeder undersize on the sacrificial conveyor and is then screened by the secondary screen. The screen oversize is transported through the coarse ore bin and subsequently the secondary feeder to be re-crushed by the secondary cone crusher. The secondary crusher discharge recycles back to the screen. Secondary screen undersize is directed to the fine ore bin through its feed conveyor. The fine ore bin has a live storage capacity of approximately 1,800t.

17.3.2 Grinding

The screen undersize is then conveyed to a fine ore bin before being transferred by two (2) feeders and a mill feed conveyor to the primary ball mill. The ball mill circuit is designed to produce a product with a nominal P_{80} of 0.10 mm.

The ball mill was sized to accommodate the 85th percentile of hardness as derived by testwork conducted at SGS on representative samples of the Zgounder deposit. The work indices determined for this ore places it in the very hard category of ores tested by SGS. The ball mill operates in closed circuit with a set of hydro-cyclones. The ball mill discharge is pumped to the ball mill hydro-cyclones. While the cyclone overflow passes through the trash screen to the rougher flotation circuit, the cyclone underflow returns, by gravity, to the ball mill. Some of the underflow is diverted to the gravity concentrator. The gravity feed is first screened, with oversize returning to the ball mill, and undersize feeding the gravity concentrator. The concentrate from the gravity concentrator is subsequently fed to the intensive leach reactor circuit. Residue after intensive cyanidation returns to the primary mill discharge pumpbox.

17.3.3 Intensive Leach Reactor

Leaching of gravity concentrates is performed in a batch type reactor tank, which is operated at elevated concentrations of cyanide and dissolved oxygen. The pregnant solution is transferred to a pregnant solution tank ahead of the Merrill-Crowe circuit and barren solution is returned to the grinding circuit.

17.3.4 FLOTATION

The hydrocyclones overflow report to rougher flotation circuit. The rougher flotation circuit consists of six (6) mechanical cells.

The silver rougher flotation concentrate is pumped to the regrind mill to liberate the base metal sulphide minerals present to allow for cleaning and separation. The regrind circuit consists of a vertical mill operating in closed-circuit with hydro-cyclones. The cyclone overflow at a P80 of 45 µm is further treated in the first cleaner flotation circuit. The cyclone underflow is reground through the vertical mill. The mill discharge slurry is then combined with the rougher concentrate to be re-classified in the cyclone.

Concentrate from the first cleaner flotation circuit is further concentrated in the second and subsequently third cleaner flotation circuits before being fed to the concentrate thickener. Tailings from the rougher and first cleaner cells are pumped to the pre-leach thickener. Tailings from the second cleaner flotation circuit join both vertical mill discharge and rougher concentrate to be re-classified in the regrind cyclone cluster.

17.3.5 CONCENTRATE THICKENING, LEACHING AND COUNTER-CURRENT DECANTATION

Flotation concentrate is sent to the pre-leach thickener to increase the slurry density prior to leaching and to recover some of the process water. The flotation concentrate from the existing flotation plant also reports to this thickener. The underflow of the pre-leach thickener is leached in four (4) concentrate leach tanks.

The discharge of the concentrate leaching circuit is pumped through the counter-current decantation (CCD) feed pumpbox to the first CCD thickener. The overflow of this thickener reports to the pregnant solution tank. The CCD circuit consists of 5 identical thickeners, each with its own rake mechanism, feed tank, overflow standpipe, underflow and overflow pumps. Wash solution is introduced into the final CCD stage to wash silver from the slurry. The overflow solution from each thickener (except from the first one) is pumped up to the preceding one such that the wash solution moves counter-current to the direction of slurry flow through the train. This progressively displaces silver from the slurry solution. The underflow slurry (except from the last stage) is depleted from its silver content by being pumped down to the subsequent thickener feed box where it mixes with the solution from the downstream stage. The barren underflow from the last CCD thickener then joins the underflow from the tailings pre-leach thickener to feed the first tailings leach tank.

17.3.6 FLOTATION TAILINGS THICKENING, LEACHING AND CARBON-IN-PULP

Tailings from both rougher and first cleaner flotation circuits are pumped through the respective pumpboxes to the tailings pre-leach thickener so that its overflow can be re-used as process water.

The tailings leaching circuit comprises of four (4) tanks, where cyanide is added to the first tank so as to effect the leaching of silver. Discharge from the last leach tank is then pumped to the carousel CIP feed distribution launder. In the carousel CIP circuit, the slurry flows from one tank to another through interstage carbon retention screens. The final effluent from the CIP circuit reports to a carbon safety screen. The undersize of this carbon safety screen is then pumped to the Tailings Storage Facility (TSF) while the oversize (i.e. activated carbon that may have leaked from the CIP circuit) goes to the carbon recovery bin.. The counter-current movement of carbon and slurry is affected by changing the slurry feed and discharge tanks in the carousel instead of pumping carbon as is done in a conventional CIP circuit.

17.3.7 Elution

The silver desorption circuit includes acid washing and rinsing, elution, and carbon regeneration. Loaded carbon from the CIP will be transferred to the acid-wash column every 18 hours. Both the acid-wash and elution columns will be sized for a 12-tonne capacity. The carbon will be washed in a dilute hydrochloric acid solution prior to being rinsed with water. The acid wash and rinse will help prevent scale or inorganic compound build-up on the carbon and allow for increased silver adsorption and desorption kinetics. Spent acid and rinse solutions will be discarded. The washed loaded carbon will be transferred to the elution column for silver desorption.

Elution will be performed using the split-AARL process. Lean solution from the previous elution cycle will be used as eluate and will be recirculated through the column and heaters as the column heats up. Once the column reaches the targeted temperature, the full volume of hot lean solution will be pumped through the column and transferred to the pregnant eluate tanks. Once this is complete, the final 2 to 4 BV of the strip will commence with fresh water used as eluate to strip the remaining silver. The dilute pregnant solution from this step will not report to the pregnant solution tank, instead it will be diverted to report to the lean solution tank and will be used as the eluate for the next elution cycle.

Once elution is complete, the column will be filled with water, and the carbon will be transferred to the carbon dewatering screen to be dewatered and transferred into the carbon regeneration kiln feed hopper. The carbon will be regenerated in a propane-fired carbon regeneration kiln operating at 700°C. The regenerated carbon will discharge from the kiln onto the carbon fine screen to remove any carbon fines generated during the elution and regeneration processes before being quenched in raw water. Any fresh makeup carbon will be introduced into the carbon attrition tank where it will be agitated in water and discharged onto the carbon fine screen at this stage and screened prior to use. All carbon fines will be washed into a tank and continuously pumped through the carbon filter. The regenerated and fresh carbon will be pumped back to the CIP circuit via a carbon transfer pump.

The carbon stripped in the elution column is dewatered and then reactivated through the carbon regeneration kiln so that it can be reused for adsorption in the CIP circuit.

17.3.8 MERRILL-CROWE AND REFINERY

The Merrill-Crowe process is used to recover silver from a cyanide leach solution by using zinc powder cementation. The plant processes include Solution Clarification, Deaeration, Zinc Addition and Precipitation, Filtration, and Refining. The pregnant eluate is cooled down and subsequently combined in the pregnant solution tank with the pregnant solution from the intensive leach reactor and the overflow from the first CCD thickener before being filtered through the clarification filters to produce a clear solution. The clarified solution is deoxygenated by passing the solution through the Crowe tower. Zinc powder is then added to the solution to precipitate the silver from sodium cyanide solution. This metallic silver is then filtered and dried in an oven. A mercury retort extracts any mercury in the sludge. The dried sludge is then mixed with appropriate fluxes and smelted in the electric furnaces to produce silver bullion. The barren solution from this Merrill-Crowe process is pumped back to the CIP tailings pumpbox.

17.4 Equipment Sizing and Selection

The equipment was sized based on the design criteria developed, flow sheet drawings, the mass balance, and to satisfy layout considerations.

Design factors used were: 20% for crushing equipment, 15% for grinding and flotation equipment, 20 % for leaching, CCD and ADR equipment, and 5 % for slurry pumps.

17.4.1 CRUSHING

The main crushing plant equipment is an Astec Standard 3042 Jaw Crusher and SBS 52 C/C Cone Crusher. The crushed ore is transported via conveyor to be stored in fine ore bin. The fine ore bin has a storage capacity 17.3 hours. The crushing plant discharges rocks with a particle size distribution of 80 % less than (P_{80}) 13 mm.

17.4.2 GRINDING, GRAVITY CONCENTRATION AND FLOTATION

Ore is withdrawn from the bottom of the fine ore bin using two feeders. The ball mill is 4.9 m in diameter by 8.1 m long with 3,100 kW installed power. The ball mill operates in closed circuit with a hydrocyclone pack. The hydrocyclone pack comprises of six (6) Model gMAX15-20-3229 cyclones, with four (4) operating and two (2) standbys. The cyclone overflow flows into the rougher flotation circuit, this circuit is composed of six (6) mechanical cells with a volume of 40 m^3 each. The rougher flotation concentrate flows to the regrind cyclone, while the overflow flows to first cleaner flotation and the underflow flows to regrind mill.

The hydrocyclone underflow flows, by gravity, to both the ball mill and gravity concentrator. Part of the cyclone underflow is diverted to the gravity circuit. It is first screened, with oversize returning to the ball mill for further grinding, and the undersize then feeding the gravity concentrator. The concentrator is sized to accommodate a feed rate of 88 tph dry solids. The concentrate from the gravity concentrator is subsequently fed to the intensive leach reactor circuit. The intensive leach circuit is sized to accommodate the expected mass pull of 0.13 to 0.15% of fresh feed. Tailings from the intensive leach reactor is returned to the primary mill discharge pumpbox.

First, second and third cleaner flotation circuit comprise of six (6) first cleaner mechanical cells with volume of 8 m^3 each; and four (4) second and two (2) third cleaner mechanical cells of with volume of 3 m^3 each. These three cleaner flotation circuits recover silver concentrates with the tailings sent to pre-leach thickener.

The ball mill, cyclone and rougher flotation and cleaners flotation circuit design are based on test work and DRA experience.

17.4.3 INTENSIVE LEACH

A SLR1000 leach reactor consists of two tanks (leaching and agitated) linked by a peristaltic pump. The concentrate holding tank is equipped with load cells to track batch sizes. The leach tank and impeller are rubber lined to minimize wear. The SLR1000 uses either peroxide or oxygen gas to achieve the elevated levels of dissolved oxygen required to accelerate the leaching process.

17.4.4 THICKENING, CONCENTRATE LEACHING AND COUNTER-CURRENT DECANTATION

A 6 m diameter thickener is selected for densifying silver flotation concentrate (from existing plant #2 and new plant #3) prior to leaching and CCD. A 25 m diameter thickener is selected as pre-leach thickener to densify flotation tailings prior to leaching-CIP. The four (4) concentrate leach tanks are sized at 4 m diameter with 4.5 m height each.

Five (5) 6 m diameter thickeners are selected as counter current decantation thickeners to recover dissolved silver into pregnant solution for Merrill-Crowe process.

17.4.5 TAILINGS LEACHING AND CARBON-IN-PULP

The pre-leach thickener receives the following feed streams:

- Rougher flotation tailings;
- Cleaner flotation tailings;
- Flotation tailings from existing flotation plant (#2);

These streams are combined in the thickener feed well where flocculant is added to facilitate solids settling. The thickener overflow is recycled to the process water tank. Thickener underflow is pumped to the tailings leach tanks for further treatment.

The four (4) tails leach tanks are sized 12.9 m diameter with 13.4 m height each.

The eight (8) carbon-in-pulp tanks are sized 10 m diameter with 11 m height each.

17.4.6 CARBON STRIPPING

Loaded carbon from the adsorption tanks will be transferred to the acid-wash column approximately every 18 hours. The acid-wash column will be sized for 12 tonnes of carbon.

Elution will be performed using the split-AARL process. The elution column will also be sized for a 12-t capacity.

17.4.7 MERRILL-CROWE AND REFINERY

Pregnant solution is sent to a 426 m^3 storage tank and then filtered and clarified through two pressure leaf clarifiers. Clean pregnant solution from the filters is sent to deaerator in removing dissolved oxygen.

After deaeration, clarified pregnant solution is pumped to two (2) 800 mm filter presses. Before being pumped, zinc dust is added to the solution to initiate a precipitation reaction. Daily production of PLS is about 426 m^3 with a grade of 1,324 g/m^3 Ag.

The precipitate is dried and then smelted in two (2) induction furnaces, producing 23-kg doré bars with a purity ranging from 80%-90% Ag.

17.4.8 REAGENTS

17.4.8.1 PAX collector

Potassium amyl xanthate (PAX) is used in sulphide flotation, it is mixed with raw water before being distributed to both rougher and cleaner flotation circuits as well as to the primary mill cyclone cluster. The expected consumption is 96.7 t/y.

17.4.8.2 AERO 241

AERO 241 is used as a collector and distributed to both rougher and cleaner flotation circuits as well as to the primary mill cyclone cluster.

17.4.8.3 MIBC

Methyl isobutyl carbinol (MIBC) is used as a frother distributed to both rougher and cleaner flotation circuits. Its expected consumption is 20 t/y.

17.4.8.4 Flocculant

Flocculant is used in all thickeners to aid the settling of concentrate and tailings. It is mixed with potable and subsequently process water before being distributed to the pre-leach, concentrate, and all CCD thickeners. Its total expected consumption is 43.2 t/y.

17.4.8.5 Sodium Cyanide

Cyanide is mixed with raw water before being distributed to the intensive leach reactor, the barren solution tank, the first tailings leach tank, and also to the first three (3) concentrate leach tanks. Its total expected consumption is 957 t/y.

17.4.8.6 Lime

Lime is mixed with process water before being distributed to the gravity-intense leach reactor, the concentrate and pre-leach thickeners, the first tailings leach tank, and also to the first two concentrate leach tanks. Its total expected consumption is 447 t/y.

17.4.8.7 Hydrochloric Acid

A tanker truck will transport 33% w/w hydrochloric acid solution to the processing plant. Upon arrival, the acid will be transferred to the hydrochloric acid tank. The acid will be pumped to the acid-wash column via the hydrochloric acid distribution pump. Prior to use, it will be diluted with water at the discharge of the distribution pump.

17.4.8.8 Sodium Hydroxide

Sodium hydroxide will be used to raise the pH within the elution process. A tanker truck will transport 50% w/w sodium hydroxide solution to the processing plant. Upon arrival, sodium hydroxide solution will be transferred to the sodium hydroxide tank. The sodium hydroxide will be pumped to the elution circuit via the sodium hydroxide distribution pump. Prior to use, it will be diluted with water at the discharge of the distribution pump.

17.4.8.9 Activated Carbon

Silver is recovered from the leach solution by activated carbon in the CIP (carbon-in-pulp) process. The expected consumption is 40 t/y.

17.4.8.10 Pre-Coat

Pre-coat is transferred to both clarification filters and filter presses.

17.4.8.11 Lead Nitrate

Lead nitrate is mixed with raw water and subsequently with zinc powder before being added to the Crowe tower discharge. Its expected consumption is 300 kg/y.

17.4.8.12 Zinc Powder

Zinc dust is added to precipitate silver from cyanide solution. Its expected consumption is 1.7 t/y.

17.4.8.13 Hydrogen Peroxide

Hydrogen peroxide is used in the reclaim water from TSF to treat residual cyanide; optimum dosage is 3.3 g H_2O2/g weak acid dissociable (WAD).

17.5 Utilities

17.5.1 Compressed Air

Compressors will supply the plant with compressed air. The flotation circuit will utilise a dedicated compressor to provide compressed air with distribution to all rougher and cleaner flotation cells.

17.5.2 Water Services

17.5.2.1 Raw Water

Raw water is pumped to the carbon stripping (i.e. acid preparation, fine carbon, and barren solution tanks), PAX collector, flocculant, cyanide, lime, hydrochloric acid, sodium hydroxide, lead nitrate, and process water areas.

17.5.2.2 Process Water

The process water is reclaimed from the TSF and also from the overflow of both concentrate and pre-leach thickeners.

17.5.2.3 Fire Water

Fire water is supplied from Raw Water and Fire Water Tank. Under normal circumstances, the flow rate is 0. However, the fire water capacity is 391m^3, if required.

17.5.2.4 Safety Showers

The safety showers are located in the carbon stripping, flotation, CIP, refinery, PAX collector, MIBC, cyanide, lime, hydrochloric acid, sodium hydroxide, and lead nitrate areas.

18 PROJECT INFRASTRUCTURE

18.1 Existing Infrastructure

The Zgounder mine has been in production since 2019 and has all the necessary infrastructure required to support the current mining operation. This includes, but is not limited to:

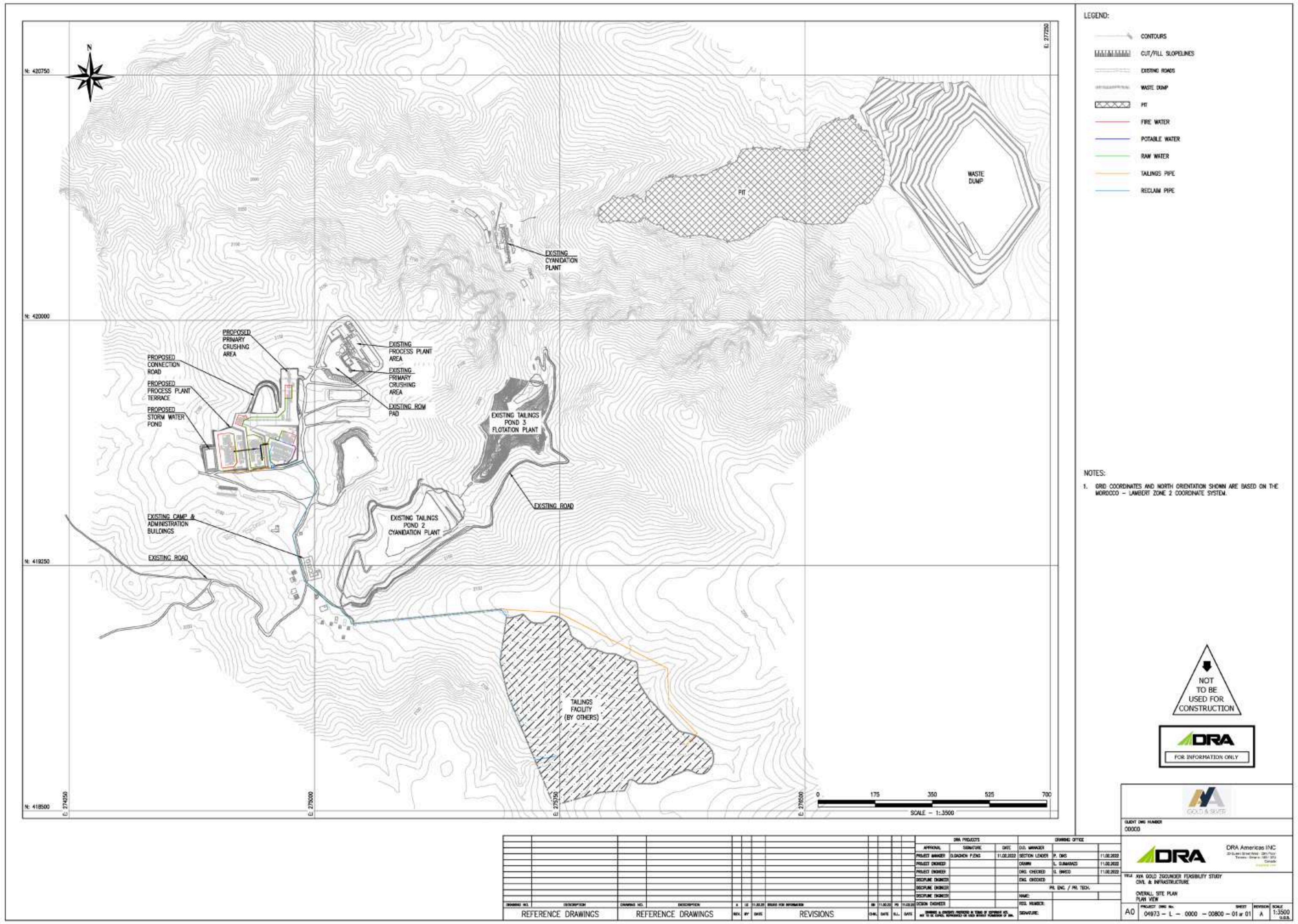
- Laboratory for preparation and analysis of process samples,
- Fuel storage of 20,000 L capacity;
- Chemical storage for storing process reagents;
- Power supply of 2,160 KVa capacity;
- Water supply for 20 m^3 per hour;
- Crushing facilities of 700t/d capacity
- Flotation and cyanidation facilities for 700 t/d operation
- Tailings ponds for 5 years of current production capacity for the flotation tailings, and 10 years of capacity for the cyanidation plant tailings;
- Camp for 80 people;
- Waste facility for disposal of industrial waste;
- Core shack for logging and preparing drill core as well as core storage;
- Offices for management and technical personnel;
- Warehouses;
- Workshops to sustain the current operation;
- Underground mine and related infrastructure including ventilation fans, compressors, offices, storage, maintenance shops and mine dry;
- Explosives storage;
- Ore stockpile of 10,000 t capacity.

18.2 Zgounder Expansion Infrastructure

For the Expansion Project, this section describes the main Project elements related to process, followed by support infrastructure. Figure 18.1 shows all existing and planned infrastructure and locations of the plant and mines and Figure 18.2 depicts the process plant area.

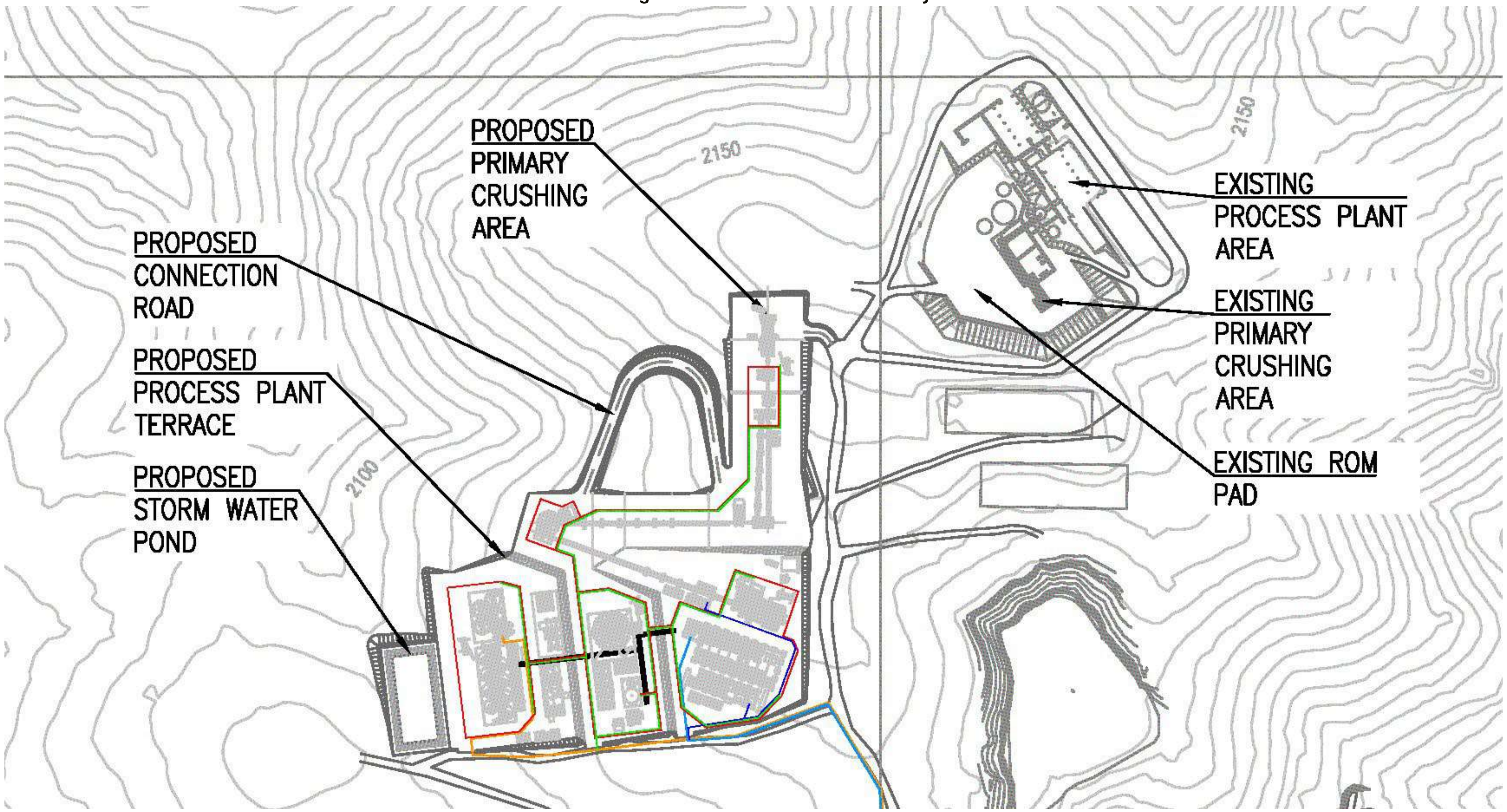
All plant infrastructure design takes into consideration the data gathered during site geotechnical investigations and a geotechnical report performed by GCIM in December 2021

Figure 18.1 - General Overall Site Plan



Source: DRA 2022

Figure 18.2 – Process Plant Site Layout



Source: DRA 2022

18.3 New Infrastructure, Services and Facilities

The new process plant #3 facility has nine (9) areas: crushing, grinding, flotation, intensive leach reactor, counter-current decantation, cyanidation, ADR process, Merrill-Crowe, and refinery which are described in Section 17 of the Report.

Major facilities and services outside the new process plant, which are included as part of the expansion Project, are described in the following sub-sections:

18.3.1 Fuel Farm

Currently, diesel and gasoline are purchased in bulk and stored on-site. Each tank has a capacity of 20,000 L and is constructed with full containment systems in the event of tank rupture. Mining and on-site diesel-powered mobile equipment are fuelled at the storage tanks.

As a result of the Expansion Project, additional fueling capacity of 20,000 L is required to supplement the existing tank farm.

18.3.2 Camp

There is no requirement to modify or expand the camp for the Expansion Project.

18.3.3 Waste Disposal facility

All on-site waste is separated at source into domestic and similar waste, recyclable non-hazardous waste (wood, scrap metals, papers, and plastics), and hazardous waste (oils, lubricants, adhesives, paints, reagents, solvents, and batteries). Aya has engaged a contractor to collect the wastes on a scheduled basis.

18.3.4 Water Supply and Distribution

For the Expansion Project, the new process plant # 3, the three (3) water circuits (Raw Water, Fire Water, and Process Water) have been developed to support the requirements of the plant, and in the case of the potable water circuit, the surrounding infrastructure.

18.3.5 Power Supply and Distribution

18.3.5.1 Power Supply to Concentrator Site and to Mine Site

The actual plant is supplied by a power line 22 kV, 54 km length, provided from the distribution grid of "*Office National de l'Électricité et de l'Eau potable*" (ONEE), post Taliouine.

The actual maximum available power for the Zgounder mine is 2,500 kVA.

The new plant and the mine, including the existing facilities, will be supplied by a new overhead 60 kV power line, single circuit, 90 km length, supplied from the post Igli. The new site substation 60 kV – 22 kV (Aya's scope) will include a 12 MVA transformer and a power factor compensation unit with a capacity of: 3 x 1.5 MVAR = 4.5 MVAR.

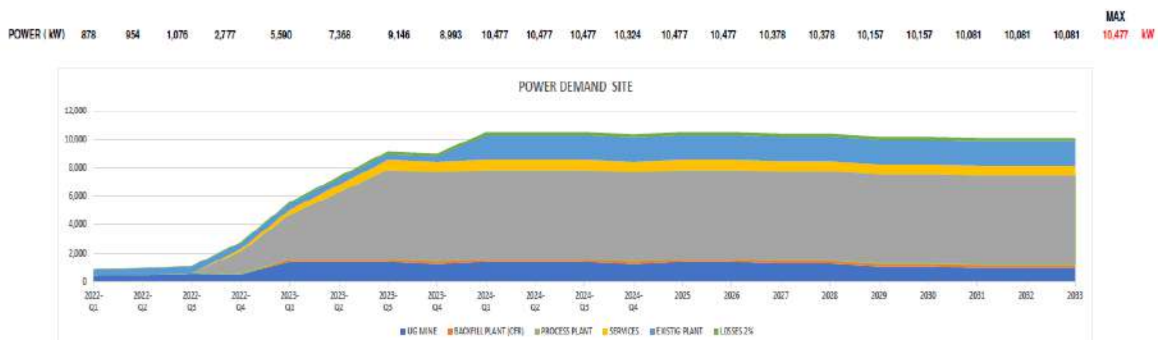
18.3.5.2 *Power Demand*

The total maximum power demand is estimated at 10.5 MW, and this power is estimated to be required in 2025, 2026 being related to the development of the underground mine.

As important note, this power includes the power requirement of the existing plants:1,740 kW.

Figure 18.3 depicts the total power demand from 2022 until 2033.

Figure 18.3 – Total Power Demand from 2022 Until 2033



Source: DRA, 2022

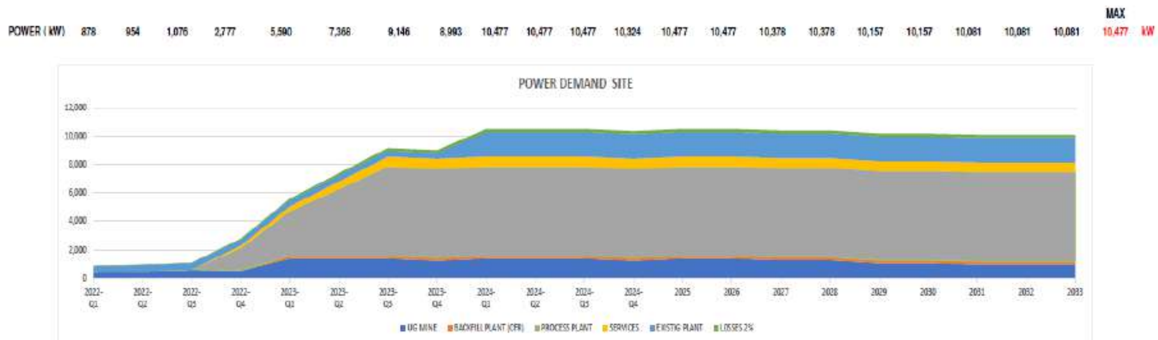
The process power demand is estimated at 6.3 MW. The power demand was estimated based on data from the Mechanical Equipment List.

The Backfill Plant (CFR) power demand is estimated at 194 kW.

In the same document is presented the mine power demand, with the evolution on the period 2022 -2034; the maximum power demand is 1,365 kW.

Figure 18.4 presents the mine power demand from 2022 until 2033.

Figure 18.4 – Mine Power Demand from 2022 Until 2033



Source: DRA, 2022

The Services power demand estimated at 692 kW is necessary for: Administration, Offices, Laboratory, Electric Rooms, Plant Warehouse, Guard House, Change Rooms, Mine and Geology Offices, Site Clinic, Prayer Room.

At the end of the calculation, are added the losses in transformers and feeders.

The process power demand was estimated using Data from the Mechanical Equipment List. A breakdown by areas is presented below.

Table 18.1 – Power Demand Requirement

Description	Power Demand Requirements (kW)
41 -Sub-Total Crushing	529
42 -Sub-Total Primary Grinding	2,070
43-Sub-Total Rougher Flotation and Regrind, Cleaner Flotation	805
44-Sub-Total Dewatering	163
45-Sub-Total Intensive Leach Reactor, Flotation Concentrate Leaching and Counter Current Decantation, Carbon-In-Pulp	1,740
46-Sub-Total Carbon Stripping, Merrill - Crowe and Refinery	420
48-Sub-PAX Collector, MIBC, AERO 241, flocculant, NaCN, LIME	22
49-Sub-Total Utilities - Raw Water and Fire Water, Process water, Air Services	533
Total Process Power Demand	6,281
Total Backfill Power Demand	194
Sub-Total Mining Materials Handling	0
Sub-Total Mine UG Pump Station	60

Description	Power Demand Requirements (kW)
Sub-Total Development crew /crew	119
Sub-Total Drift-and-Fill /crew	75
Sub-Total Room&Pillar / crew	0
Sub-Total Long-hole / crew	97
Sub-Total Mine UG (include 2 Main Fans)	746
Total UG Mine Power Demand	1,097
Total General Services Power Demand	692
Total General - Process and Services	8,262
Loses in transformers and feeders	165
Total General Services Power Demand (excluding existing plants)	8,428

18.3.5.3 *Emergency Power*

An emergency power system will be provided as a standby source of power to feed essential services (emergency and exit lighting, fire pumps, etc.) as well as critical process loads in the event of power loss from the generating plant. The standby power source shall consist of one Diesel Generator (1 MVA, PF = 0.8) located in the neighbourhood of the Main-Substation.

18.3.6 CONTROL SYSTEM

18.3.6.1 *Automation Process Network*

The Process Control System ("PCS") will be based on an Ethernet backbone network in a ring type topology. The network will link all the main automation equipment, such as Supervisory Control and Data Acquisition ("SCADA") system, Historian, Human Machine Interface ("HMI") and Process Control System processors.

The proposed network includes fibre optic linking of the following main areas of the plant:

- ROM Control Room;
- Grinding Area Control Room (Main Control Room);
- Electrical Room ER-5210 (Main Substation 60 kV – 22 kV);
- Electrical Room ER-4001 (Crushing Area);

- Electrical Room ER-4002 (Concentrator Area – Grinding and Regrinding);
- Electrical Room ER-4003 (Concentrator Area – Leach, CCD, and Refinery);
- Mine Substation 22 kV – 5.5 kV (Skid Mounted);
- UG Mine RMUs (Mine Equipment and Surface Ventilators);
- Administration;
- Offices;
- Plant Warehouse;
- Workshop;
- Site Clinic;
- Existing Process Plant.

Network automation communication services are:

- SCADA stations located in the control rooms and in the field;
- Process Control System processors inter-communication;
- PCS/Remote Input/Output ("I/O") communication;
- PCS direct interface to the Motor Control Centers (MCCs);
- IEC61850 interface to the power distribution equipment;
- Field device communication including communication with 3rd parties Programmable Logic Controller ("PLC") supplied with mechanical equipment;
- Camera system installed in the plant for process control viewing purposes;
- Camera system installed in the plant for security purposes;
- IP phone system for plant site;

18.3.6.2 Process Control System

The process control system will be of PLC type. A PCS system will be supplied to control strategic areas of the plant with remote I/O racks located generally in the Electrical Rooms.

The main processors will control the following sectors: crushing, concentrator (all areas), and remote loads connected to the 22-kV pole line.

Major equipment like primary ball mill could come with their own PLC and with a Local Control Panel.

The 400 V MCC's should be equipped with an ″intelligent″ protection relay able to communicate.

The protection relays shall be equipped with Ethernet ports.

A local control station shall be installed near each motor.

The central SCADA system will have the capacity to control and supervise all remote PCS equipment. If a communication outage occurs, critical equipment will be controlled locally.

18.3.6.3 *Camera System*

A process and security camera system, with recorder and a viewer, will be installed in the main control room. Viewer for security purposes will be installed in the gate house. Other cameras will be installed in the plant for process control purposes. One (1) process viewing station will be installed in each control room.

18.3.7 COMMUNICATION SYSTEM (LOCAL & EXTERNAL)

18.3.7.1 *Telecommunications Local System*

The telecommunications system will be based on Ethernet links throughout the plant buildings and administrative buildings following generally the electrical reticulation network (buried and/or installed on the pole lines).

Single-mode fibre optic backbone will be deployed through the plant to accommodate both automation and corporate services on the same fibre cable on different fibre.

18.3.7.2 *Telecommunications and Mobile Radio Systems*

The telecom service includes the tower located in a high elevation zone of the plant. It will be supplied by a third-party provider, and it will communicate with the plant communication interface.

The telecom system will include:

- IP phone system;
- Mobile radio system;
- Fire detection system;
- Access control system (gate, door).

The mobile radio system will be provided for the construction phase and the operation of the mine and plant site.

18.3.7.3 *Telecommunication Services*

The site will be connected to a local Internet Service Provider (ISP).

The back system will use a cellular modem or satellite technology.

The IP phone system will be connected to an internal private branch exchange (PBX).

18.3.7.4 *Telecommunications Distribution*

During the construction phase, all communication services, such as Internet and phone, will be distributed via Wi-Fi, Wimax and Microwave point-to-point radios to reach all areas of the plant site.

All mine trucks and pick-ups will be equipped with a Wimax/Wi-Fi antenna that shall also act as a Wi-Fi local access point.

The telecommunication distribution will be through the plant's fiber optic network covering the crushing and concentrator areas, Administration Office, Camp and Cafeteria, and Mine Office.

If necessary, wireless communication will be provided for other auxiliary buildings outside of the plant area.

18.3.7.5 *Corporate Network*

The automation Ethernet backbone network, in a ring type topology described in the previous section will be used for automation, camera and security videos, IP phone system and corporate network applications.

All the major network equipment will be located in dedicated server rooms located in the administrative office, telecom shelter, control room, and electrical rooms.

Corporate services are:

- Wired/Wireless Phones and System Server;
- Process and Security Camera System;
- Access Control System (gate, door);
- Fire Detection.

18.4 Hydrogeology

A Phase 1[6] groundwater exploration project was undertaken in order to identify sectors with potentially favourable characteristics for the development of a water supply that can provide sufficient quantities of water for mine processes. The Reader is referred to Section 20 of the Report for details.

18.5 Surface Water Balance

The information presented in this section is, for the most part, largely extracted, summarised from the Report entitled "*Aya Gold & Silver inc, Zgounder Feasibility Study - Zgounder Mine Expansion Project, Site Water Balance*" Project # 057-P-0023332-0-01-330-GS-R-0003-00, Final Version, prepared by Englobe Corp., issued November 30, 2021.

18.5.1 Introduction

In 2020, Englobe was retained by Aya Gold & Silver (Aya) to undertake the global water balance for the Zgounder mine expansion project as part of the Zgounder feasibility study. The current mine is operating at around 700 t/d and with the combined processing plants, the mine is expected to operate around 2,700 t/d.

Englobe's mandate was to validate the hydraulic conditions to supply the new processing plant with water and to size the new water management infrastructure for the mine expansion.

18.5.2 Climate Data - Precipitation

In the absence of a weather station on site, Englobe's team used the data from the closest weather station, located at Iguidi, approximately 18 km from the site. It should be noted that the elevation of the weather station is ~1,000 m below the mine.

Statistical analysis reveals significant variability in annual precipitation, ranging from 68.9 to 753.5 mm/year. The highest precipitation was detected in 2009-2010 and 2010-2011 when the global El Niño event took place. The maximum daily precipitation in any given year fluctuates from 11.5 mm/d to 120.9 mm/d. Comparison between the annual and the maximum daily precipitation reveals strong correlation between the two. Also, one day of maximum precipitation accounts for a large share (12%-35%) of annual total.

For instance, in 2017-2018, the annual total was 142.8 mm, of which 49.5 mm or 35% of annual precipitation fell on a single day. This illustrates unevenness of the precipitation throughout the year and poses a challenge in collection and management of large, infrequent precipitation events.

[6] Englobe 2021. *Étude hydrogéologique Phase 1, Recherche en eau souterraine Mine d'argent de Zgounder*, Final Report. Prepared for Aya Gold & Silver, by H. Tezkratt. Issued March 2021, 79 p.

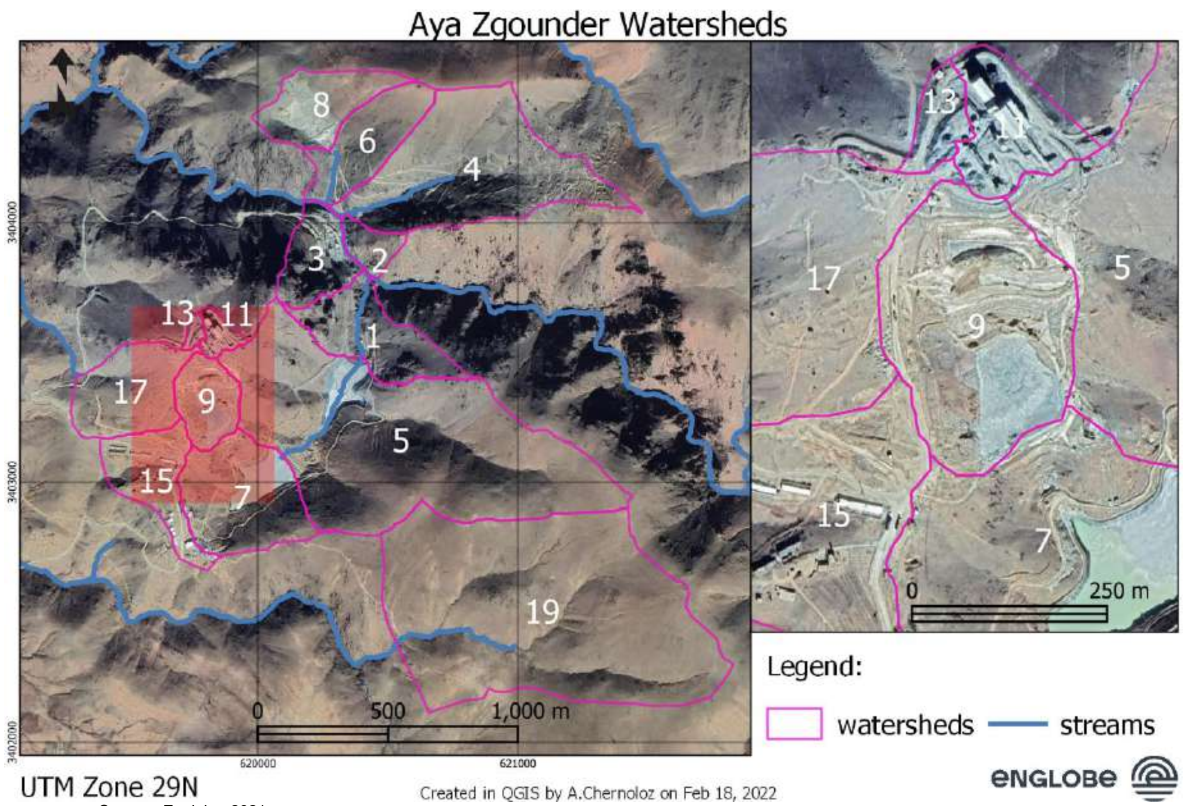
Temperatures have very large seasonal variations: hot to very hot summer and cold in winter. Daily amplitudes can exceed 20°C. In winter, however, low daytime temperatures are pleasant, while nighttime temperatures drop to around 0°C (night frosts occur in December and January).

18.5.3 HYDROLOGY AND WATERSHEDS

The mine site is located primarily in the Oued Zgounder (or Achkouchi) watershed, with the exception of the administrative area with worker camps to the southwest of the site and the site of the new tailings storage facility (Site C). The mine site elevations range from 2,000 to 2,200 m.

Sub-watersheds have been determined for the following infrastructure or areas which is shown in Figure 18.5.

Figure 18.5 – Main Watersheds On-Site



Source: Englobe, 2021

The labels in Figure 18.5 correspond to the following infrastructure:

- Abandoned tailings facility (6 and 8);
- Decommissioned tailings facility (9);
- Cyanide tailings facility (7);

- Flotation tailings facility (5);
- Camp/Administration Area (15);
- Old Dam on Oued Zgounder (0);
- Flotation plant area (13 and 11);
- Cyanidation plant area, garage, storage, and laboratory (3);
- Future pit area and waste dump (4);
- Future plant area (17);
- Future tailings facility area (19);
- Other buildings and infrastructure areas (1 and 2).

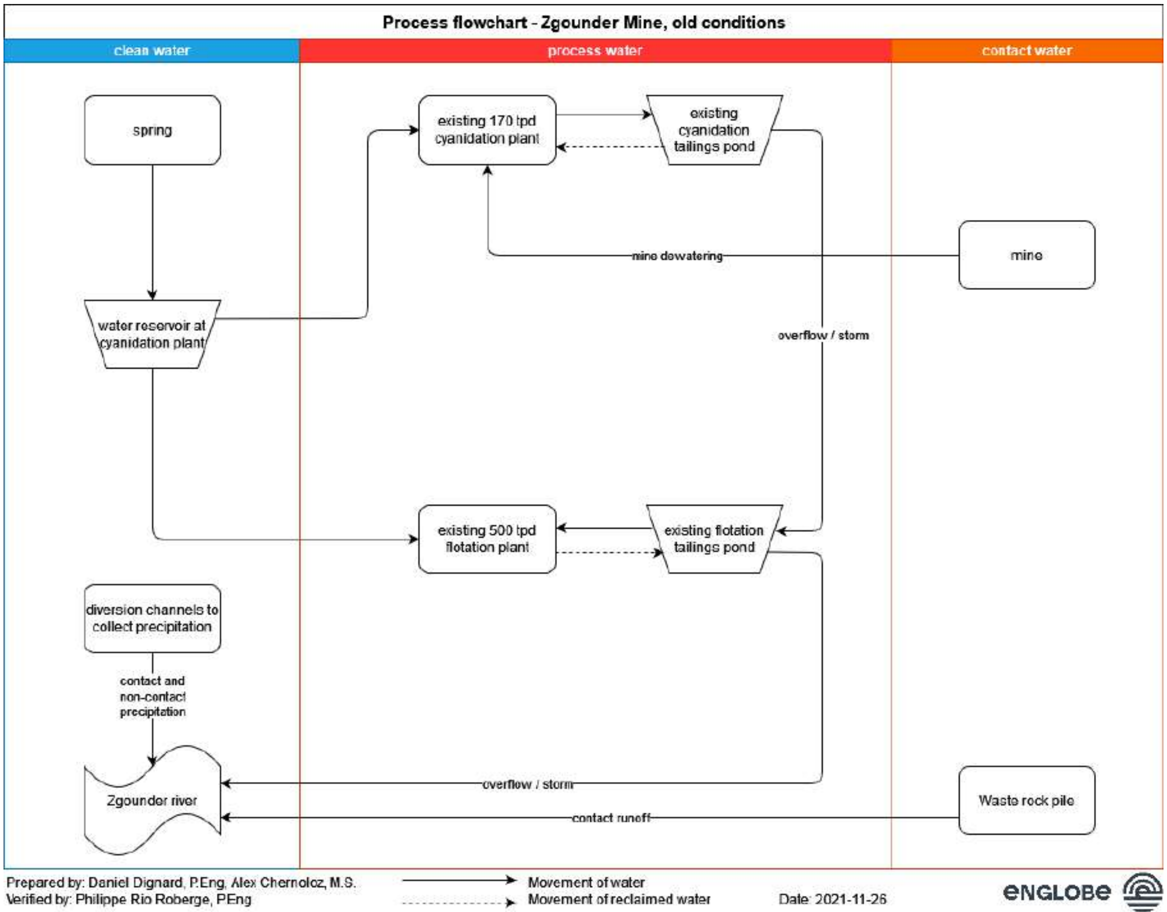
18.5.4 Water Management System

Understanding the existing water management operation is fundamental to developing the water management strategy for the expansion project. For the current project, no site visit was conducted by Englobe's design team due to COVID-19 restrictions. Therefore, all information on the existing operation was provided by Aya and could not be validated by Englobe. The following sections present information on the existing and expanded operations.

18.5.4.1 Existing Water Management System

There are three (3) types of water in the water management process: clean water, process water and mine water. Clean water comes from the surface water spring. Process water is recycled from the process and mine water is from the dewatering of the mine. The existing overall water management system is shown in Figure 18.6.

Figure 18.6 – Existing Water Management System Process Flow Diagram



Source: Englobe, 2021

Clean water for the mining operations comes from a spring about 7 km from the mine site. This spring supplies the cyanidation plant and the flotation plant as well as the mining camp. Clean water for the mine site is delivered through a pipeline with a tapping and manual shut-off valve.

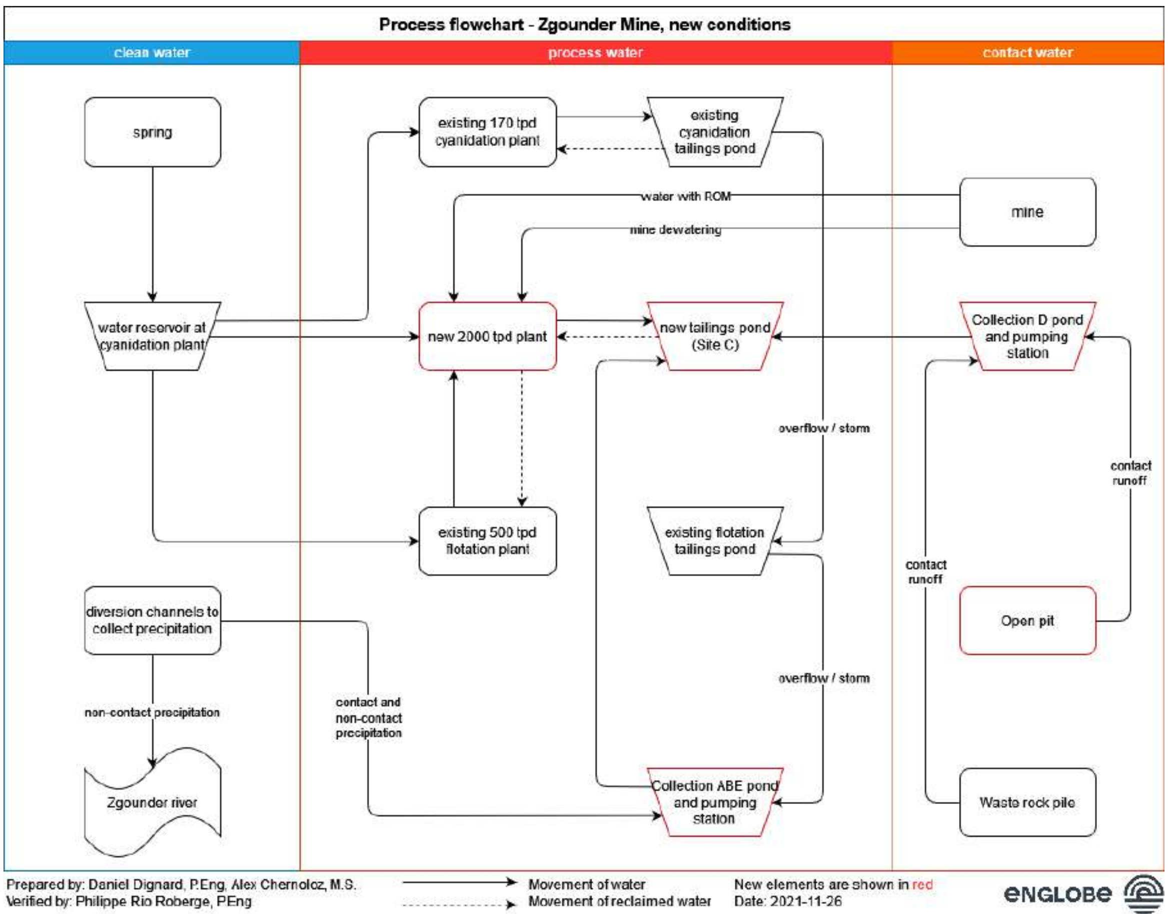
According to Aya, the average flow rate measured from the spring is 23.3 m³/h. The clean water is mostly used as a water source for the existing plants. Two (2) pumping systems with basins downstream of the cyanidation TSF and the flotation TSF are used to recirculate process water from the TSF to the plants.

No water balance or water management strategy for the current two (2) TSFs were available to be incorporated in the water balance. Therefore, it was assumed that under average precipitation conditions, the existing plants water management system was in equilibrium (water inputs=water outputs from the system). This assumption was derived from the fact that the current plants operate normally; however, certain conditions result in a deficit of process water and signs of contamination downstream of the current TSF are apparent. It is assumed that during larger rainfall events the overflow water from the cyanidation and flotation ponds will overflow and cascade into the Zgounder Oued.

18.5.4.2 *Expanded Water Management System*

The mine is undergoing a process of expansion in which a new 2,000 t/d plant will be constructed. A new TSF (Site C) will be constructed and will include a water reservoir, to be used as storage for the Project. The expanded water management system is shown in Figure 18.7. Note that the components in black are existing and the components in red are proposed infrastructure.

Figure 18.7 – New Water Management System



Source: Englobe, 2021

18.5.5 Water Balance Model

18.5.5.1 Methodology

The Englobe represented Zgounder site as an open model, with inputs equal to outputs on an annual basis, drawing the system boundary around the Project.

Englobe collected the information from various sources, including hydrological reports from GCIM and GEOBEL, the preliminary metallurgical plant water balance by DRA (J4973-PROC_MB_001 Process Mass Balance Rev A), the relevant weather data, information from the field studies and data provided by AYA.

Englobe computed the net balance as the difference between the inputs and outputs from the system. When inputs exceeded outputs, the resulting difference was recorded as surplus. This surplus can be mitigated by transferring the excess water into storage, reducing the inputs, releasing non-contact water to the environment, etc. Conversely, when outputs exceeded the inputs, the resulting difference was recorded as water deficit, which can be mitigated by taking the shortage from the storage, increasing the inputs, etc. The discussion of the water balance scenarios is presented in Section 18.5.6.

18.5.5.2 Assumptions

Englobe assumes the following:

- The Zgounder mine acts as one (1) system of connected elements, including natural (watersheds, streams, outlets) and manmade (water wells, reservoirs, tailings storage facilities, pipelines, channels, etc.).
- Inputs to the system include water from the existing spring, precipitation into the lined ponds, precipitation into the watersheds (contact water), precipitation into the unimpacted watersheds (non-contact water), moisture in the run of the mill and the mine dewatering.
- Outputs from the system include evaporation, infiltration, entrainment of water in tailings voids (EIE) in the TSFs, infiltration in the watersheds, discharge into the environment, the mining camp consumption, processing losses (carbon reactivation kiln evaporation, etc.) and water for cemented rock fill (CRF).
- Internal flows include processing water pumped from processing plants as slurry into the tailings ponds, and reclaimed water pumped from tailings ponds to the processing plants. These flows were approximated by Aya for the existing operation and calculated by DRA for the expansion Project.
- Surface external flow (outside the mine impacted watersheds) inputs and outputs are assumed to be 0, since the water from the Zgounder river is not intercepted, and there is no data about

the discharge of water. There is no official information about the tailings dams overflowing intentionally into the environment.

- GCIM assumes the recovery of water from the new TSF pond to be 60%, based on observations and experience, with the remaining 40% lost to EIE.
- Englobe undertook its own estimate of existing EIE based on the average year scenario, as discussed in Section 18.5.5.4.

18.5.5.3 Input Data – Processing and Hydrology

Key inputs for the water balance model used by Englobe and the sources of those inputs are listed in Table 18.2.

Table 18.2 – Summary of Input Data

Parameter	Value	Unit	Source
Precipitation, dry year	70.4	mm/y	Weather data, station Iguidi, 5th percentile
Precipitation, avg year	227.2	mm/y	Weather data, station Iguidi, median
Precipitation, wet year	679.0	mm/y	Weather data, station Iguidi, 95th percentile
Runoff coefficient, small watershed	0.6	(-)	GCIM
Total area of lined ponds	22.9	ha	Estimated by Englobe
Total area of contact watersheds	88.1	ha	Estimated by Englobe
Total area of non-contact watersheds	114.5	ha	Estimated by Englobe
Water from mine dewatering	10	m^3/h	Aya
Water input from the spring	23.3	m^3/h	Aya
Water for cemented rock fill	124	m^3/d	Aya
Water with slurry, existing cyanidation and flotation plants	1221	m^3/d	Aya
Water with the run of mine	105.3	m^3/d	Projected by 3rd party consultant (DRA)
Water with slurry, new plant including existing flotation plant	1984	m^3/d	Projected by 3rd party consultant (DRA)
Water recovery ratio, existing process and tailings storage	48	%	Computed by Englobe from Aya data
Water recovery ratio, new process and tailings storage	60	%	Estimated by 3rd party consultant (GCIM)

18.5.5.4 *Computation of Existing Evaporation, Infiltration, Entrainment in Voids (EIE)*

The losses from the existing system occur in the form of evaporation (E), infiltration (I) and entrainment in voi(E). Because of limited input info and instrumentation, it is not currently possible to separate E, I & E. Therefore, EIE is reviewed as a single quantity, not separated into its individual components. Because of lack of instrumentation and reliable data, Englobe assumed constant EIE based on average year (median precipitation = 227.2 mm/y). Realistic modelling of EIE will likely see its increase in dry years and decrease in wet years, driven by increased evaporation and water heads.

Englobe computed the EIE based on the inputs from Aya, on the assumption that the system is balanced (inputs = outputs) in an average year, and the sum of inputs (the spring, the precipitation into two (2) existing ponds and their watersheds) plus the recovered water from the TSFs are equal to the outputs (the slurry to the TSFs and the mining camp needs) with no deficit or surplus of water. According to Engobe's estimate based on inputs and outputs, the EIE is 52%, and conversely, the recovery of reclaim water from the existing TSFs is 48%. This value agrees with the range of recoveries provided by GCIM (40-70%).

18.5.5.5 *Mine Dewatering*

The average projected mine dewatering flow was estimated at 10 m³/h; this value was provided by Aya following a 24-hour site pumping test from the mine well during the dry season. This number may vary depending on the season, the mining plan and encountered geology.

18.5.5.6 *"Dry", "Average" and "Wet" Scenarios*

The Englobe team decided to take 5%, 50% and 95% percentiles of precipitation as the representative of dry, normal, and wet years, as summarised in Table 18.3.

Table 18.3 – Weather Statistics for Dry, Normal, Wet Years

Scenario	Percentiles (Inclusive)	Annual Precipitation (mm)	Precipitation (days)
Wet	95%	679.0	76.5
Normal	50%	227.2	43.0
Dry	5%	70.4	18.7

18.5.5.7 *Hydrological Modelling*

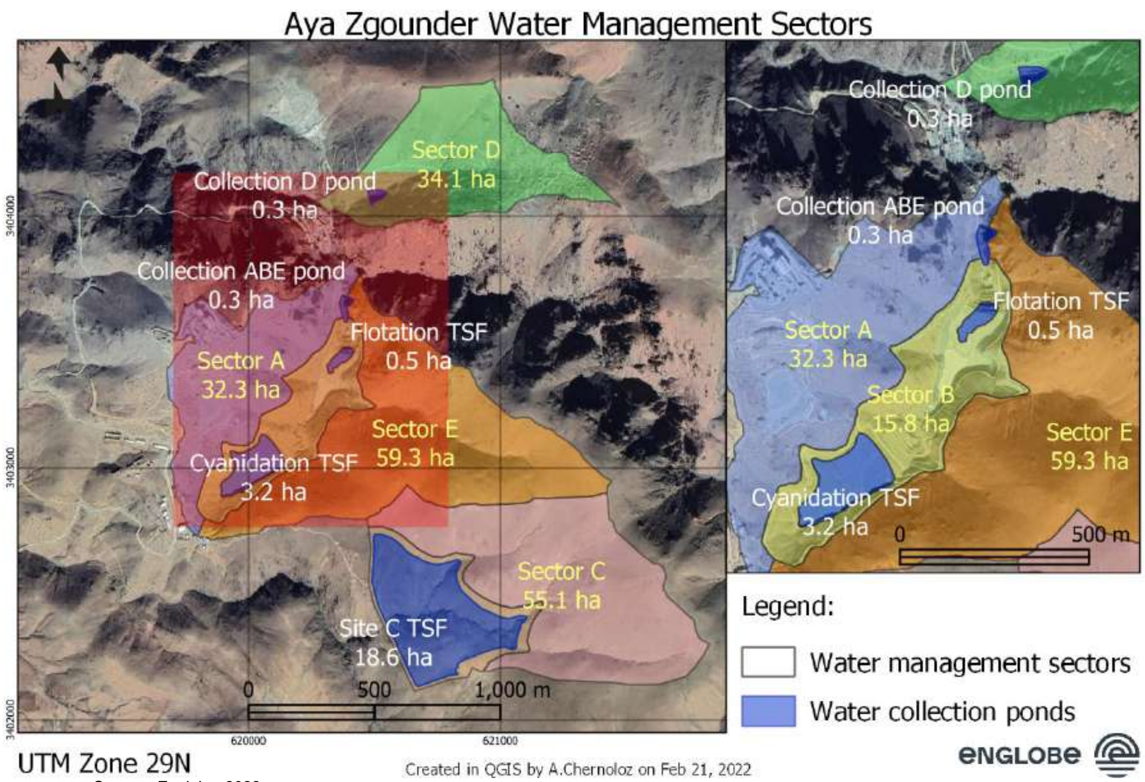
Since the Iguidi station is the closest to the Zgounder mine, data from this station was used to develop intensity-duration-frequency (IDF) analysis.

Using the Generalized Extreme Value Distribution, 1:100-year, 24-hour precipitation is estimated at 134.1 mm.

18.5.6 Results of Global Water Balance

The global water balance was computed as the difference of all inputs and outputs. The area of the Project was divided into five (5) water management Sectors A to E, defined by watershed boundaries and manmade water diversion structures. Sectors E and C collect non-contact water, with the remaining sectors collecting contact water. The water management areas were overlain by the existing and proposed ponds as shown in Figure 18.8.

Figure 18.8 – Water Management Sectors



Source: Englobe, 2022

18.5.6.1 Summary of the Global Water Balance

Englobe computed the water balance for the Project for dry, average, and wet years. The summary of this computation is presented in Table 18.4.

Table 18.4 – Water Balance by Precipitation Scenario

		item	Dry year (m³/day)	Dry year (m³/year)	Average year (m³/day)	Average year (m³/year)	Wet year (m³/day)	Wet year (m³/year)
inputs (I)	fresh	existing spring	559.2	204,108	559.2	204,108	559.2	204,108
	mine	run of mine	105.3	38,435	105.3	38,435	105.3	38,435
		mine dewatering	240.0	87,600	240.0	87,600	240.0	87,600
	precipitation	cyanidation TSF	6.1	2,229	19.7	7,193	58.9	21,498
		flotation TSF	1.0	353	3.1	1,140	9.3	3,407
		Site C TSF	35.9	13,094	115.8	42,259	346.0	126,294
		collection ABE pond	0.6	211	1.9	682	5.6	2,037
		collection D pond	0.6	211	1.9	682	5.6	2,037
		contact watersheds (A, B, D) w/o ponds	102.1	37,256	329.4	120,234	984.5	358,919
		non-contact watersheds (C, E) w/o bermed area and ponds	132.4	48,323	427.3	155,950	1,276.9	466,473
	recovery	recycled from cyanidation TSF - 170 tpd	144.0	52,544	144.0	52,544	144.0	52,578
		recycled from Site C TSF - 2000+500 tpd	1,190.2	434,408	1,190.2	434,408	1,190.2	434,408
outputs (O)	slurry	sent to cyanidation TSF - 170 tpd	301.0	109,865	301.0	109,865	301.0	109,865
		sent to Site C TSF - 2000+500 tpd	1,983.6	724,014	1,983.6	724,014	1,983.6	724,014
	other	cemented rock fill (CRF)	124.0	45,260	124.0	45,260	124.0	45,260
		carbon reactivation kiln evaporation	0.9	329	0.9	329	0.9	329
		trash screen oversize to tailings	0.2	73	0.2	73	0.2	73
		mining camp needs	4.0	1,460	4.0	1,460	4.0	1,460
		Total inputs	2,517.2	918,772	3,137.6	1,145,235	4,925.4	1,797,760
		Total outputs	2,413.7	881,001	2,413.7	881,001	2,413.7	881,001
		Deficit (-) or surplus (+)	103.5	37,772	723.9	264,235	2,511.7	916,760

Source: Englobe, 2021

All three (3) base scenarios show the surplus of water, ranging from 38,000 to 917,000 m³/y, before any mitigation measures are applied.

18.5.6.2 *Balance Mitigation Measures*

Considering the wet year scenario, if pumping from the existing spring is completely stopped and the entire volume of non-contact water is diverted, the excess of water is reduced to 247,000 m³/y. This excess contact water must be stored or treated prior to release into the environment.

Under the evaluated mitigated wet year scenario, by removing all spring water supply and non-contact precipitation from the model during the wet year and during the following years, it would take roughly 3 years of average precipitation intensity to consume the surplus water.

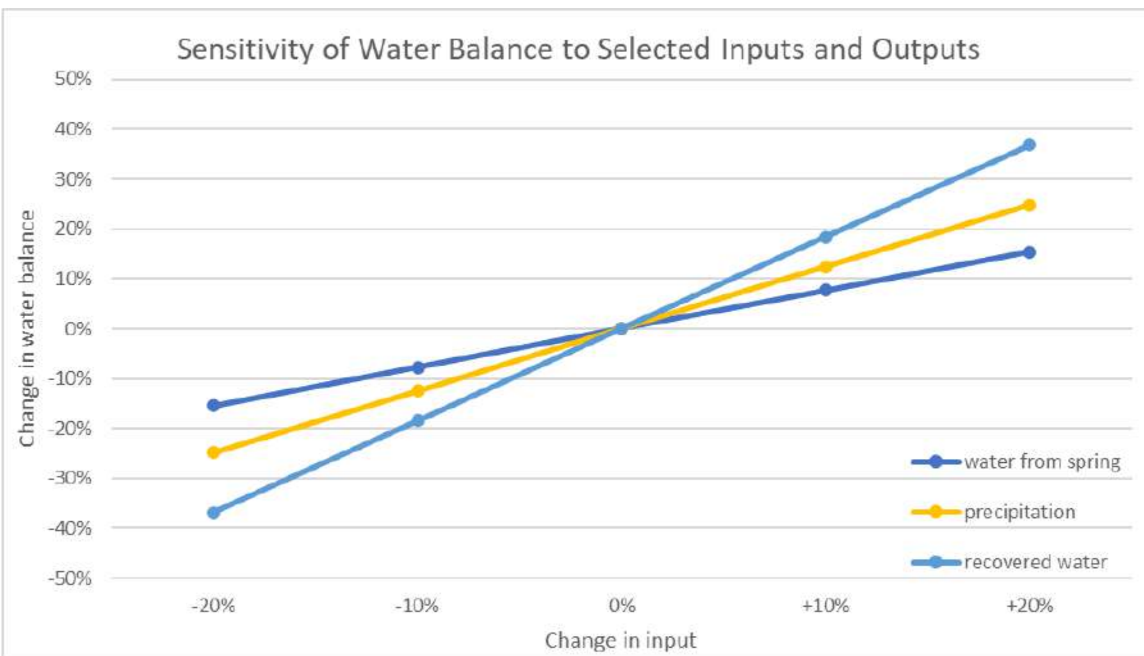
18.5.6.3 *Sensitivity Analysis*

Given the uncertainties in certain input information, Englobe completed a sensitivity analysis on the average year to discover which inputs and outputs have the largest impact on the water balance. The summary of this analysis is presented in Figure 18.9:

The dominant factors influencing the water balance are listed below without preserving their order of importance.

- The amount of recovered water had the largest impact on water balance. The effects of recovery ratio (and conversely, the loss to EIE) on the water balance illustrates the importance of this factor.
- The amount of collected precipitation is a function of the area, the precipitation amount, and the runoff coefficient. The area of the watersheds is measured with high precision, but the amount of local precipitation remains uncertain, and the runoff coefficient is an estimate based on general literature review. The sensitivity analysis highlights the need to measure the runoff coefficient in the field and seek better sources of local climate data.
- The freshwater flow from the existing spring is the least important factor of the three (3) factors reviewed.

Figure 18.9 – Sensitivity of Water Balance to Selected Inputs and Outputs



Source: Englobe, 2021

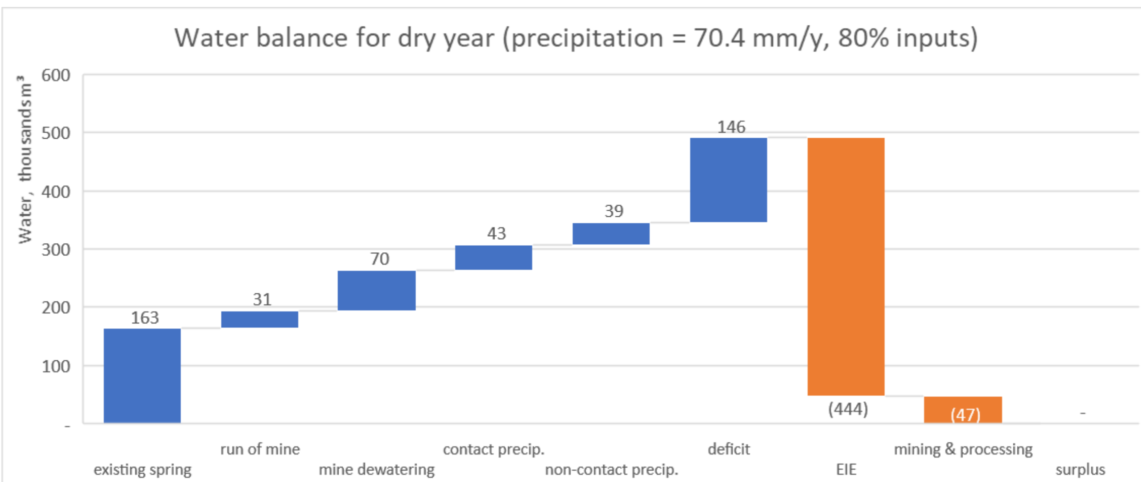
Based on the sensitivity analysis results, extreme scenarios can be derived to determine a reasonable minimum and maximum water storage capacity in TSF Site C.

18.5.6.4 *Limit Scenarios*

a. Extreme Dry Scenario

To account for the uncertainties in the instrumentation, weather data and assumptions made, Englobe subjected the system to the hypothetical scenario in which all water inputs were reduced by 20% during the dry year. In this scenario, the deficit of water totals 146,000 m³/year, as shown in Figure 18.10 below. This scenario result should be reflected in the water management strategy to avoid interruptions of the mining and processing operations.

Figure 18.10 – Extreme Dry Scenario

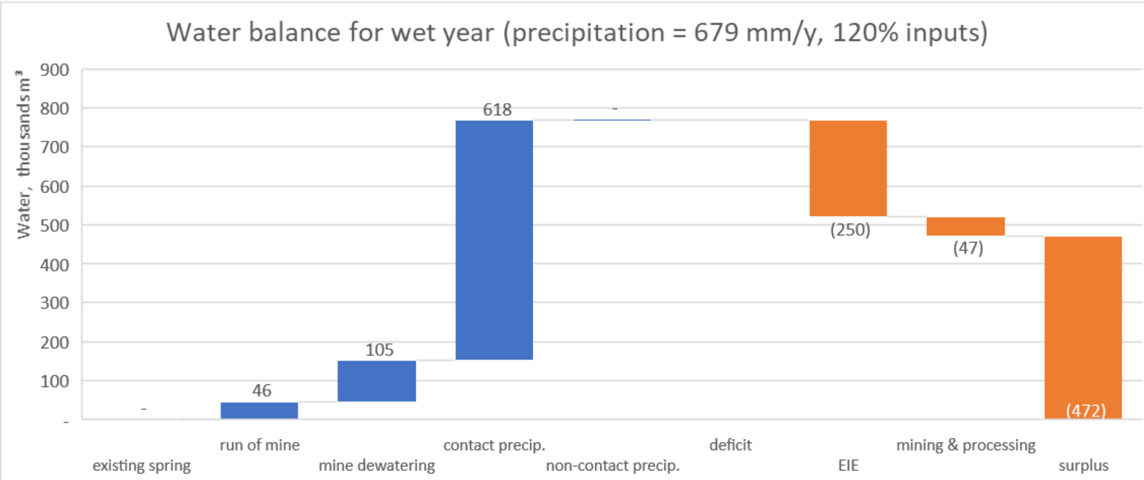


Source: Englobe, 2021

b. Extreme Wet Scenario

Similar to the extreme dry scenario, discussed above, Englobe subjected the system to the extreme wet scenario, in which all inputs were increased by 20%. In this case, after the mitigative measures, the surplus of water reaches approximately 472 000 m³/y, as illustrated in Figure 18.11.

Figure 18.11 – Extreme Wet Scenario



Source: Englobe, 2021

18.5.6.5 Recommendations for the Conceptual Infrastructure Design

The water management strategy requires the infrastructure to be designed in order meet the global water balance conclusions. Englobe approached the conceptual design aiming to achieve the following engineering objectives:

- Ensure continuous operation, unaffected by water shortage or excess;
- Create a flexible system that can adapt to changing conditions;
- Simplify the water management infrastructure to increase its robustness and effectiveness;
- Minimize the need for new fresh water sources and water treatment on site.

Based on the water balance conclusions, Englobe proposes the following water management infrastructure:

- Create a water reservoir with a maximum initial capacity of 620,000 m³ and maintain the minimum water volume at around 150,000 m³. Those volumes can be refined to suit the staged TSF pond construction sequence as more climatic and operational information become available;
- Build water diversion ditches and water diverting structures in the diversion ditches to collect precipitation, separate it into contact and non-contact streams, and convey it in or out of the system; and,
- Build two (2) water collection basins and pumping stations to convey water to the TSF Site C water basin.

18.6 Tailings Management Facility

ZMSM has retained the services of Epoch Resources (Pty) Ltd ("Epoch") for the role of Engineer of Record (EoR). As such, Epoch is the firm responsible for confirming that the current and future tailings facility are designed, constructed, and decommissioned with appropriate concern for integrity of the facility, and that it aligns with and meets applicable regulations, statutes, guidelines, codes, and standards. Epoch is a South African based company with extensive design, project execution and ongoing monitoring/inspection of tailings storage facilities ("TSFs") within Africa and particularly in West Africa.

The information presented in this section is, for the most part, largely extracted, translated and/or summarised from the Report entitled "*Étude de la digue a stériles du projet Zgounder - Etude de faisabilité de la digue phasage & BOQ des travaux*" No Projet - Zgd1-2020, prepared by GCIM, issued as revision A on January 21, 2022.

18.6.1 Design Criteria

The following elements were selected for the Tailings Storage Facility (TSF) design criteria based on discussions with Aya:

- The tailings stored in the TSF are coming entirely from the cyanidation circuit;
- The TSF construction will be by embankment with the possibility of downstream elevation;
- The elevations will be carried out under the same conditions as the primary embankment;
- Daily production: 2,700 t/d;
- Plant operation: 360 days per year;
- Project life: 10 years;
- An overcapacity of 500,000 m^3 for possible water storage;
- Pulp density: 1.4 t/m^3.

Accounting for the information above, the total storage volume would be 6,942,900 m^3 + 500,000 m^3 for water storage.

18.6.2 Hydrological Conditions of the Different Potential Site Locations

GCIM applied three (3) different methods in order to determine the water flows generated by the catchment areas of the sites surveyed in the Zgounder mine area as well as different return periods:

- Empirical formulas approach to calculate floods of rare frequencies and in particular of return frequency 1/10 necessary for the Gradex Method.

- Direct transposition of peak flows by the Francou-Rodier method from the sites of hydrological stations.
- Use of the rain Gradex from the estimated ten-year peak flow.

The estimates are based on rainfall data from October 1988 to August 2020 from the Iguidi substation located about 18 km from the study area.

For the assessment of water input and flash flood events, the data from the Iguidi hydrological station located in the Tifnout wadi basin was also used. It has been in operation since 1989, and the processing of the hydrological information was updated until August 2013, at which time, the station was damaged by a flash flood.

Examination of the topography of the watersheds of the potential storage sites made it possible to delimit the watershed lines, identify the longest rivers, the extreme elevations as well as their morphological characteristics and is summarised in Table 18.5.

Table 18.5 – Catchment Areas Characteristics

Site Name	Area	Length	Grade	Perimeter	Altitude Difference		
	km²	km	%	[km]	Elev Max	Elev Min	Diff.
Site A	2.571	2.412	10.36	6,199	2,370	2,120	250
Site B	0.935	2.260	12.17	4,582	2,390	2,115	275
Site C	0.854	1.295	10.04	3,679	2,240	2,110	130
Site D	0.981	1.842	11.40	4,090	2,400	2,190	210
Site E	0.262	0.640	18.75	2,292	2,170	2,050	120

The flows of the floods retained are the result of an analysis and a comparison of the flows calculated by the different calculation methods.

- For watersheds with an area greater than 1 km², the flows retained correspond to the average of all the empirical formulas, the Gradex formula and the transposition.
- For small watersheds with a surface area of less than 1 km², flood flows are deducted from the average of the Gradex formula and the transposition.

Table 18.6 – Floods at Different Return Periods

Catchment Area		Site A	Site B	Site C	Site D	Site E
Area [km²]		2.57	0.94	0.85	0.98	0.26
Estimated Flows	Qp 10 [m^3/s]	14.24	9.73	9.21	10.11	3.33
	V [m^3]	41,359	20,800	16,411	22,622	2,629

Catchment Area		Site A	Site B	Site C	Site D	Site E
	Qp 20 [m^3/s]	13.73	8.14	7.69	8.44	3.07
	V [m^3]	39,872	17,396	13,690	18,867	2,425
	Qp 50 [m^3/s]	15.93	9.37	8.85	9.71	3.62
	V [m^3]	46,249	20,029	15,755	21,709	2,852
	Qp 100 [m^3/s]	17.59	10.34	9.76	10.71	4.05
	V [m^3]	51,071	22,108	17,389	23,953	3,196

18.6.3 Site Selection

Based on the topographic backgrounds at 1:50,000, the topographic survey submitted by AYA, satellite photos and digital terrain models, a preliminary choice of potentially interesting sites, has been carried out.

The analysis of the available documents made it possible to identify five (5) possible storage sites, designated as Sites A to E distributed throughout the Project area:

- Site A is located about 2 km northeast of the plant and closes a fairly wide thalweg with an east-west direction. The catchment area of the embankment is 2.57 km².
- Site B is located on a thalweg adjoining the south side of the thalweg of site A, approximately 1.7 km northeast of the plant. The catchment area of the embankment is 0.94 km². The direction of the flow is from southeast to northwest.
- Site C is located approximately 1,000 m to the southeast of the plant on a thalweg with a general east-west direction. The catchment area of the embankment is 0.85 km².
- Site D is located about 2.5 km east of the plant and closes a thalweg with a northeast-southwest direction. The catchment area of the embankment is 0.98 km².
- Site E: raise of the current cyanidation Plant TSF. However, based on the available topographic backgrounds, the maximum storage capacity of the elevation of the cyanide dike is estimated at 1.9 to 2.0 Mm^3, which is insufficient, given the total forecast storage needs.

A comparison of the performance of each site was made, based on the following characteristics:

- Maximum storage capacity conditioning the duration of operation of the structure;
- Maximum height of embankment;
- Hydrology of the respective watersheds to assess the magnitude of the floods of the watersheds controlled by the planned works;
- Geological and geotechnical conditions;

- Preliminary estimate of the quantities of work required to construct the dikes.

During the field visits, it was possible to examine the conditions of the five (5) potential sites for carrying out the embankment and identifying their main advantages and constraints. Based on the analysis of their characteristics, Sites B, D, and E were eliminated.

After an assessment of Sites A and C, Site C was retained due to its proximity to the plant and the reduced surface area of the catchment area upstream of the embankment.

However, Site A has been kept as a fallback solution in the event of saturation of Site C, which may happen if the production of the mine increases following new extensions of the deposit and/or a possible extension of its operating life.

18.6.4 TSF Design by Phase

For the purposes of this FS, using the current information concerning the characteristics of the construction materials and the foundation, the TSF design was carried out with the aim of defining a budget estimate for the needs of the FS.

It should be noted that during subsequent Project phases, after the completion of all the reconnaissance work, this estimate will be detailed and could be adjusted to suit any changes adopted during further studies.

Also, it is important to note that to precisely define the measurements of the planned works, especially the volumes of the embankments to be put in place, stability calculations have been made on the Site C designed embankment. The geotechnical investigation campaign on Site C and its borrow areas was still in progress at the time of the write up of the FS, and stability calculations were carried out on the basis of geological and geotechnical knowledge of the Project area, while using the geomechanical parameters measured as per previous investigations carried out on the various mine existing infrastructures.

The conclusion of the stability assessment that the minimum embankment slopes to be adopted (taking into account the quality of the construction materials, the foundation and the different load cases examined) are 2.25h/1v for the slopes downstream and upstream of the embankment wall.

18.6.4.1 Phase 1

This phase includes the construction of the starter dam in colluvial materials on Site C. The embankment will have the following characteristics:

- Final elevation of the dam - Phase 1: 2,204.5 MASL;
- Crest width: 4 m;

- Dam upstream slope: 2.25H/1V;
- Dam downstream slope: 2.25H/1V;
- Volume stored: 2.48 Mm^3, 2.85 years;
- The embankment upstream of the dam will be protected by a geomembrane (2 mm) – anti-puncture geotextile (500 g/m^2), anchored in a trench filled with clay;
- The downstream dam will be protected against gullying with rockfills.

A drainage network made up of draining collectors placed in trenches filled with draining materials will be installed under the geomembrane. This network will collect any potential leaks under the geomembrane and will protect the TSF against any rising water table. A collection pond will be installed downstream of the dam to collect any percolation water.

Excess water in the TSF will be collected and sent back to the plant by means of a floating barge equipped with pumps and a HDPE return water line.

Figures 18.12 and 18.13 show the details of the Phase 1 design.

18.6.4.2 Phase 2

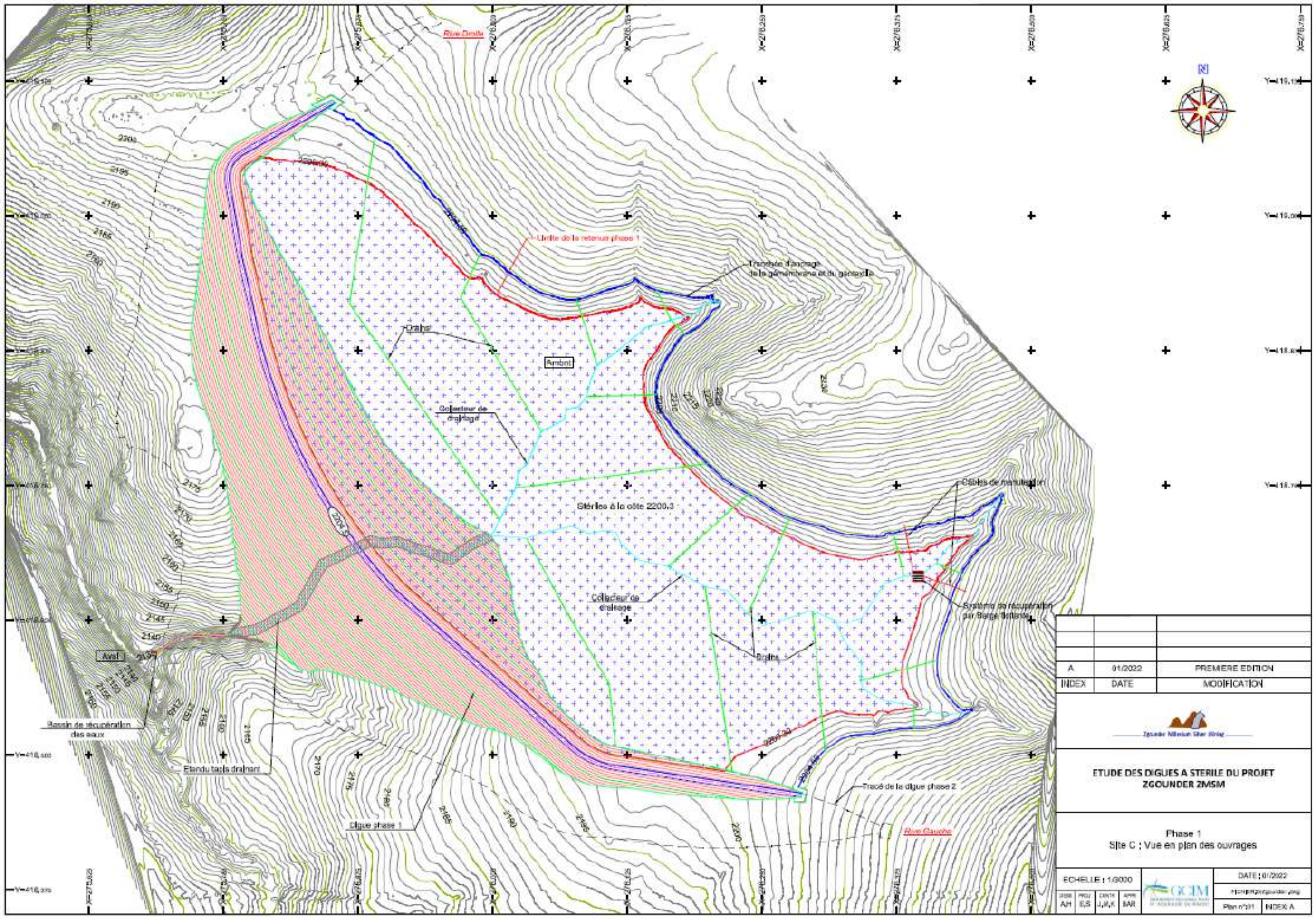
This phase includes raising the dam downstream in colluvial materials on Site C, with the following characteristics:

- Final elevation of the dam - Phase 2: 2,214 MASL;
- Crest width: 4 m;
- Dam upstream slope: 2.25H/1V;
- Dam downstream slope: 2.25H/1V;
- Volume stored (Phase 1 + Phase 2): 4.84 Mm^3, 7 years;
- Extension of the geomembrane and the geotextile on the upstream embankment of the dam.

The drainage network will be extended with the extension of the geomembrane and the protective geotextile.

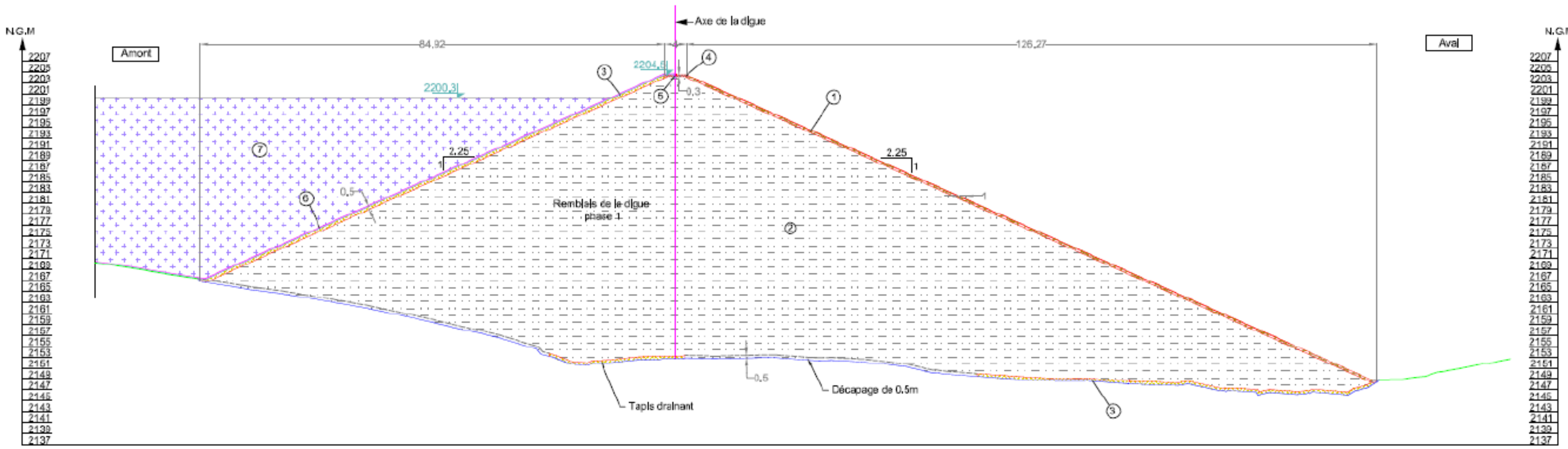
Figures 18.14 and 18.15 show the details of Phase 2 design.

Figure 18.12 – Plan View of TSF Phase 1



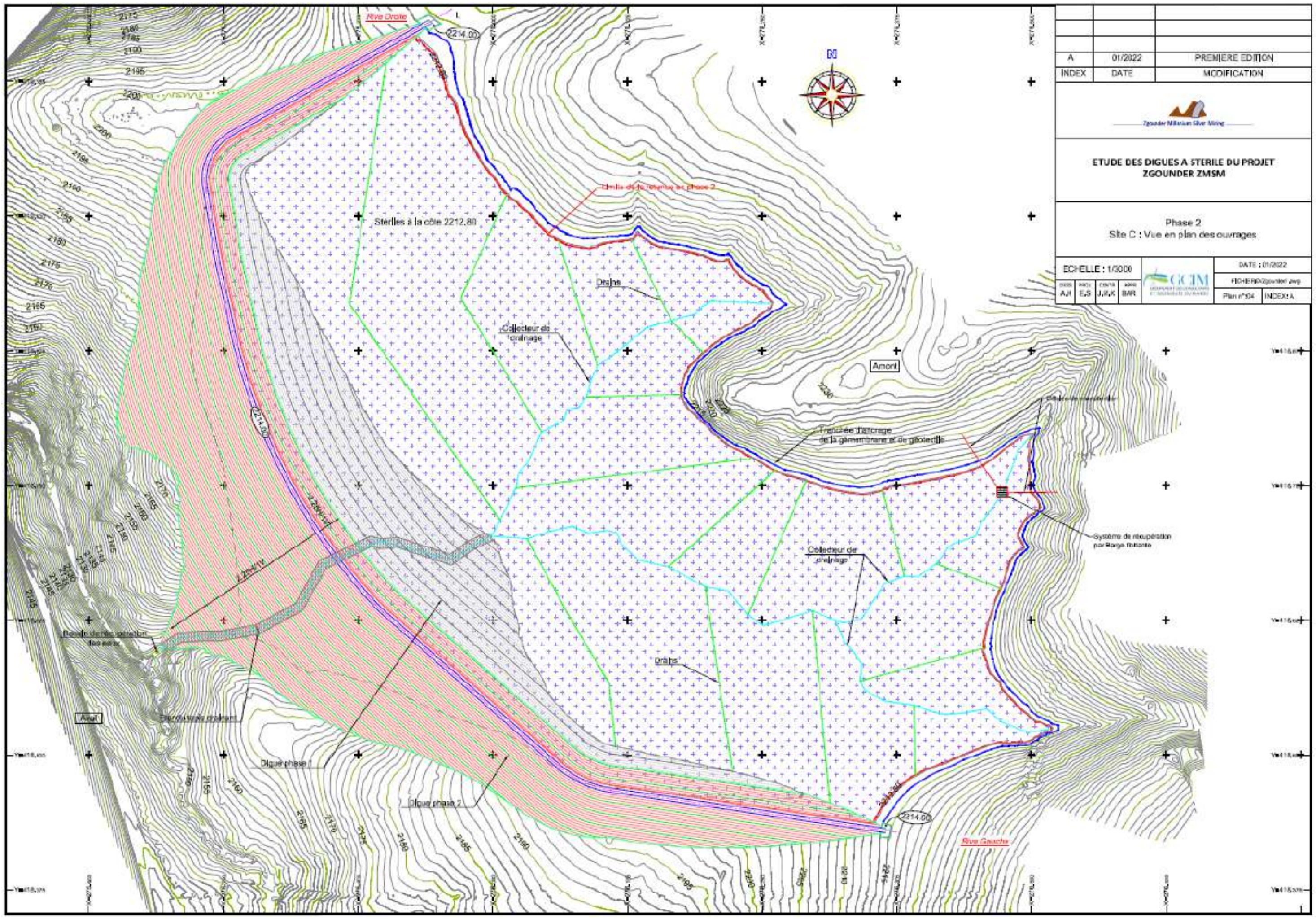
Source: GCIM, 2022

Figure 18.13 – Typical Cross Section for Phase 1 TSF Dam



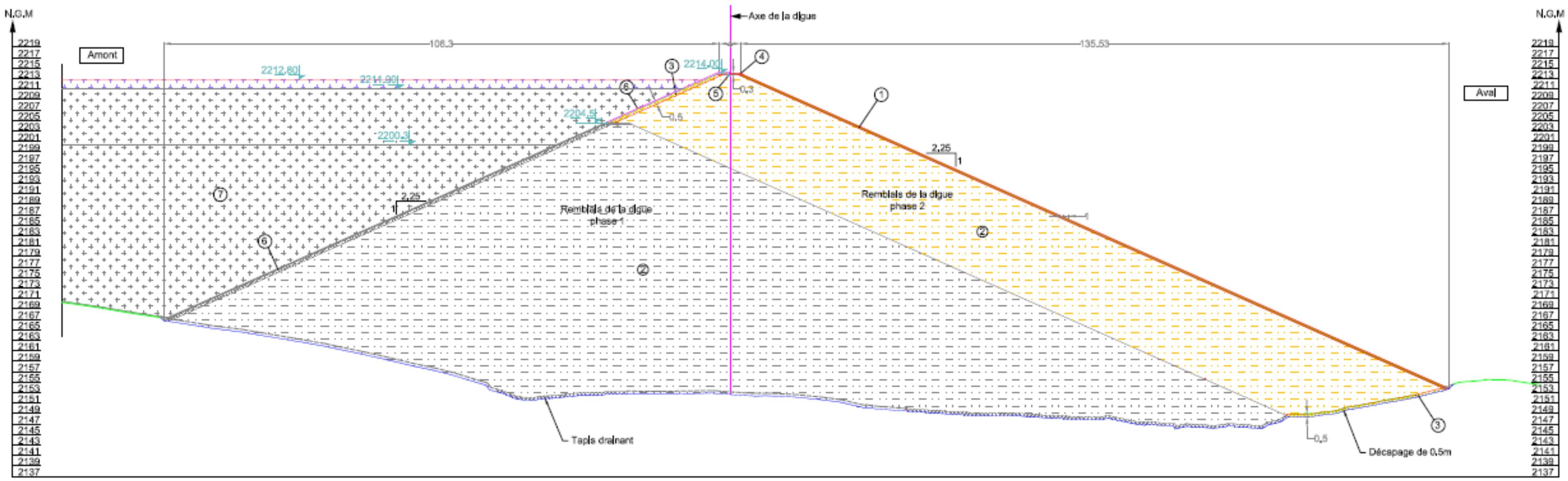
Source: GCIM, 2022

Figure 18.14 – Plan View of TSF Phase 2



Source: GCIM, 2022

Figure 18.15 – Typical Cross Section for Phase 2 TSF Dam



Source: GCIM, 2022

18.6.4.3 Phase 3

Phase 3 consists of the construction of an additional TSF located at Site A with the following characteristics:

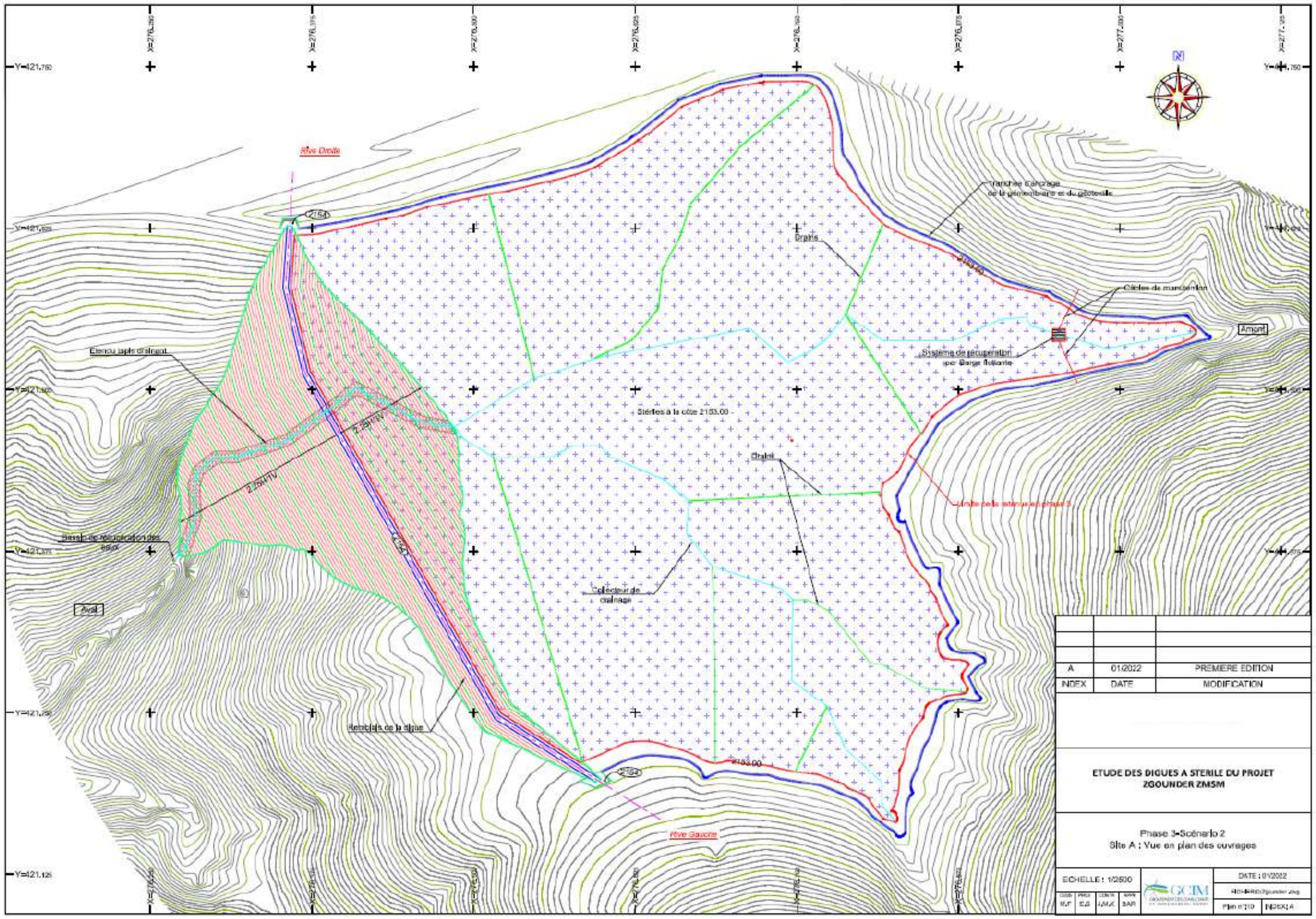
- Final elevation of the dam - Phase 1: 2,154 MASL;
- Crest width: 4 m;
- Dam upstream slope: 2.25H/1V;
- Dam downstream slope: 2.25H/1V;
- Volume stored: 2.66 Mm^3, 3 years;
- The embankment upstream of the dam will be protected by a geomembrane (2 mm) – anti-puncture geotextile (500g/m^2), anchored in a trench filled with clay;
- The downstream dam will be protected against gullying with rockfills.

Similarly to Site C, a drainage network made up of draining collectors placed in trenches filled with draining materials will be installed under the geomembrane, and will emerge in a concrete collection pond which will be constructed downstream from the dam.

Excess water will be collected and returned to the plant by means of a floating barge on which pumps and a HDPE return water pipe will be installed. A similar barge planned for Site C could be installed at Site A.

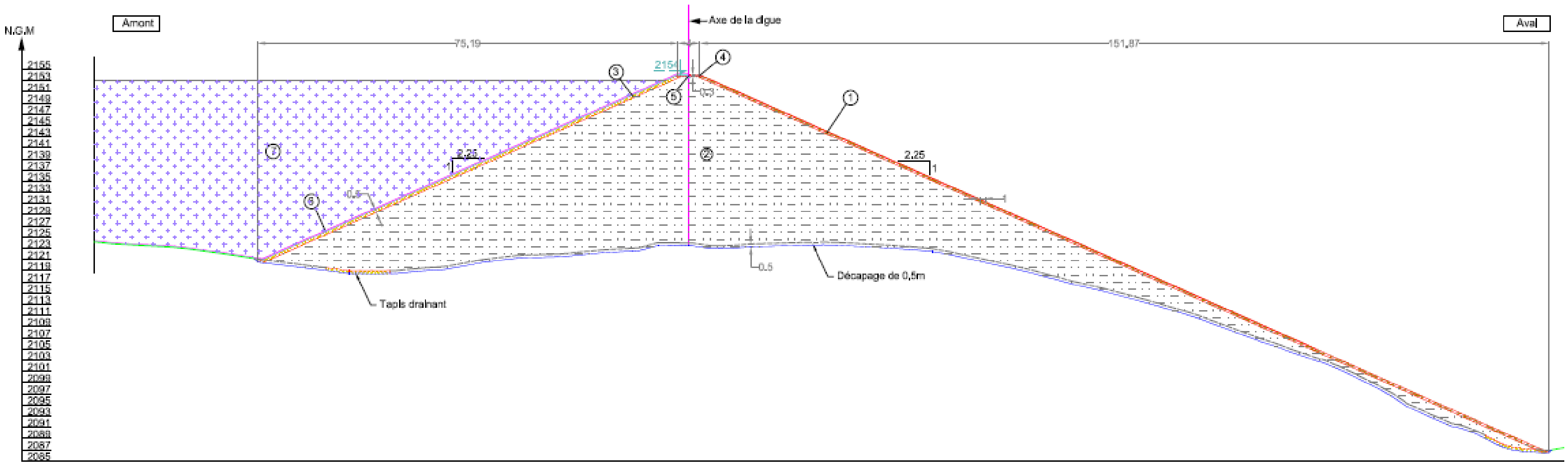
Figures 18.16 and 18.17 show the details of the Phase 3 design for Site A.

Figure 18.16 – Plan View of TSF Phase 3



Source: GCIM, 2022

Figure 18.17 – Typical Cross Section for Phase 3 TSF Dam



Source: GCIM, 2022

18.7 Surface Water Management Infrastructure

The information presented in this section is, for the most part, largely extracted, summarised from the Report entitled "*Aya Gold & Silver inc, Zgounder Feasibility Study, Water Management Infrastructure*" Project # 057-P-0023332-0-01-350-GS-R-0001-00, Final Version, prepared by Englobe Corp., issued December 10, 2021.

18.7.1 Context

Following the development of the Zgounder mine feasibility level global water balance, Englobe designed, at a conceptual level, the infrastructure required to achieve the water management strategy. The following sections present the proposed design.

Based on the water management strategy and design principles developed by Englobe in Reports entitled "*Aya Gold & Silver inc, Zgounder Feasibility Study - Zgounder Mine Expansion Project, Site Water Balance*" and "*Aya Gold & Silver inc, Zgounder Feasibility Study - Zgounder Mine Expansion Project, Water Management Infrastructure*" which are summarised in Sections 18.5 and 18.7 of this Report, GCIM has advanced the design of the water management infrastructure, previously prepared by Englobe, based on their assumptions and knowledge and has therefore developed the surface water management infrastructure Capex and is presented in Section 21.1 of this Report.

18.7.2 Objectives

Englobe approaches the water management infrastructure design aiming to achieve the following engineering objectives:

- Ensure continuous operation of the mineral processing plant, unaffected by water shortage or excess due to weather variability;
- Create a flexible system that can adapt to changing conditions;
- Minimise the need for new water wells as well as water treatment on-site;
- Reduce the mines environmental impact by collecting and managing all contact water within the site.

18.7.3 Proposed Water Management Strategy

Based on the completed water balance modelling, Englobe proposes the following water management strategy:

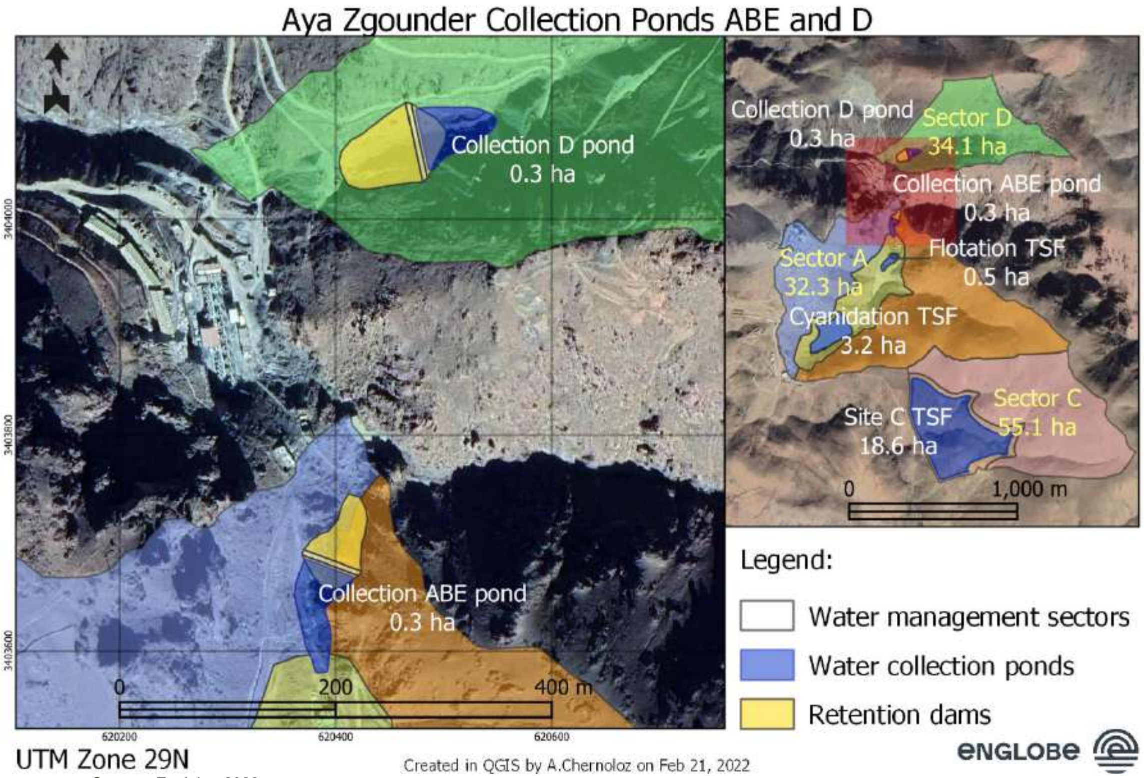
- Manage the mineral processing plant water supply between dry and wet years by maintaining water levels in the TSF between the minimal and maximal recommended water storage capacities.

- Build a flow-smoothing pumping basin downstream of the existing flotation tailings storage facility to collect contact and non-contact water of that valley and convey it to TSF Site C.
- Use and upgrade the existing Eastern diversion ditch around the existing cyanidation and flotation TSFs to collect and convey non-contact water from that valley into or out of the system.
- Build a flow-attenuating pumping basin downstream of the mine and waste rock storage facility to collect all contact water of that valley and convey it to TSF Site C.
- Build a water diversion ditch to collect and convey non-contact waters into or out of the new TSF Site C.
- Install flow diversion devices into non-contact diversion ditches in order to collect in the system or discharge to the environment water depending on the operational water balance requirement.
- Store a portion of intense precipitations reporting to the existing cyanidation and flotation TSFs into their respective water ponds.
- Maintain the desired storage water level on target by manipulating water inputs:
    - The flow from the existing spring;
    - Non-contact water runoff diversion.

18.7.4 New Water Management Infrastructure

Figures 18.18 and 18.19 depict the proposed Ponds ABE and D as well as the water management infrastructure along with their reporting watersheds.

Figure 18.18 – Proposed Ponds ABE and D



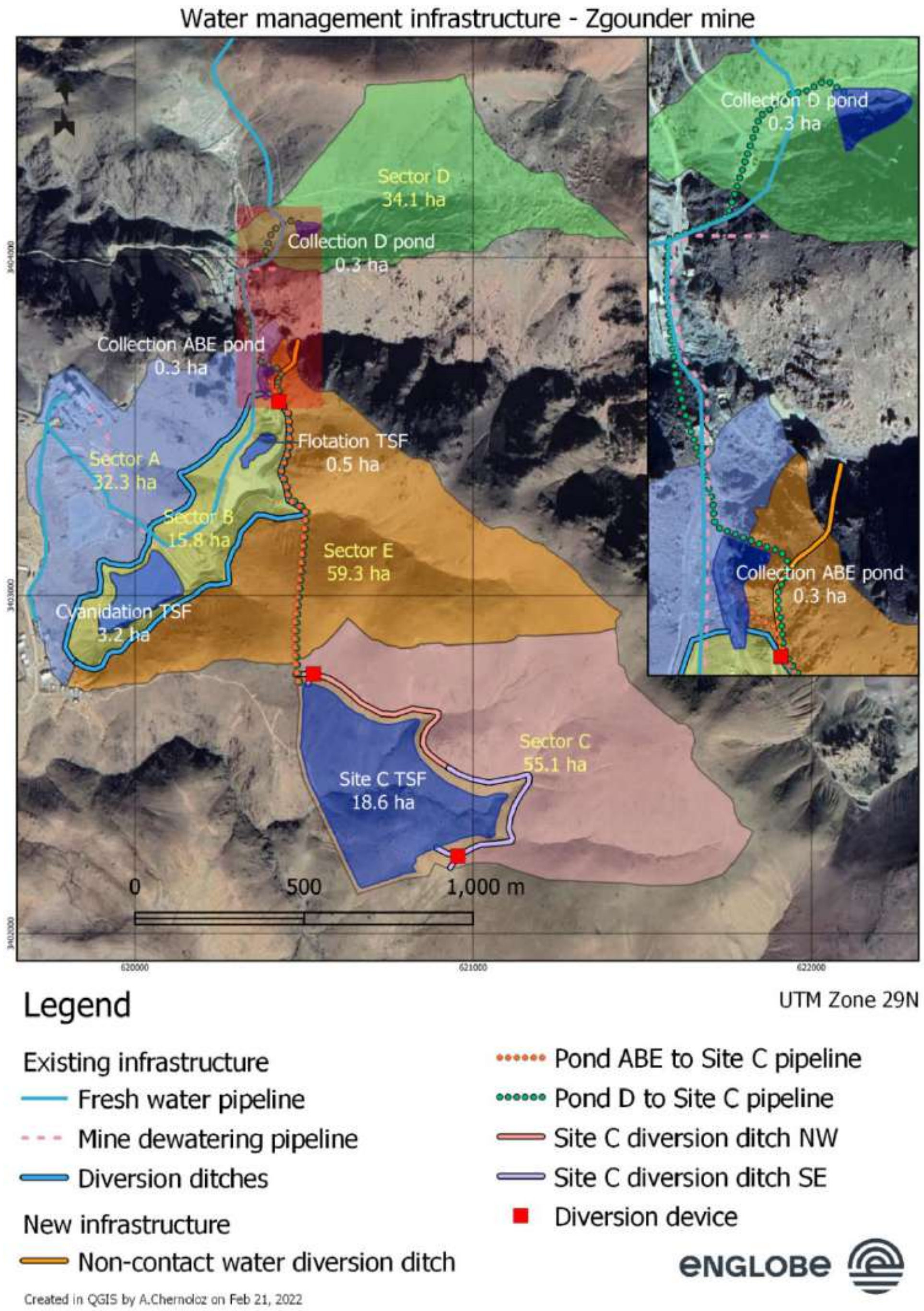
Source: Englobe, 2022

In order to achieve the water management strategy objectives, new infrastructure, as depicted in Figure 18.19, will be required, namely:

- A collection pond for runoff water from Sectors A, B and E (Pond ABE);
- A pumping station in Pond ABE;
- An emergency spillway on the retention dam of Pond ABE;
- The upgrade of the existing diversion ditch of the non-contact water from Sector E;
- A flow diversion device in Sector E diversion ditch;
- Pipelines for water conveyance between pumping station ABE and TSF Site C;
- An extension of Sector E non-contact water diversion ditch around Pond ABE;
- A smoothing and collection pond for runoff water from Sector D (Pond D);
- A pumping station in Pond D;
- An emergency spillway on retention dam of Pond D;
- Pipelines for water conveyance between pumping station D and TSF Site C;
- Diversion ditches for non-contact water from Sector C;

- Flow diversion devices in Sector C diversion ditch;
- Monitoring and control systems.

Figure 18.19 – Water Management Infrastructure and Water Management Sectors



Source: Englobe, 2022

Table 18.7 presents a summary of the new pond and dam parameters.

Table 18.7 – Summary of New Pond and Dam Parameters

Parameter	Unit	Pond ABE	Pond D
Dam crest elevation	MASL	2,000.0	2,016
Downstream Dam toe elevation	MASL	1,975.0	1,987
Crest width	m	5.0	5.0
Slopes	-	1V:2.5H	1V:2.5H
Dam volume	m³	19,000	41,000
Dam footprint	m²	4,500	7,000
Pond surface elevation	MASL	1,999	2,015
Pond bottom	MASL	1,992	2,006
Pond volume	m³	8,500	9,300
Pond plan area	m²	3,000	2,700

18.7.4.1 Water Management Infrastructure

Pond ABE will be located downstream of the existing flotation TSF. The pond is designed to smooth the precipitation inflow from the design flood (up to 1:100-year) and enable its transfer to Site C water reservoir / TSF. The pond collects non-contact and contact water from the valley. It is not intended to be used as a retention pond; therefore, it is not lined with impervious material. The location of the water dam and the resulting storage capacity drove the minimum pumping capacity of 1.0 m³/s to adequately collect and convey a 1:100-year storm event. Discharge pipelines transfer water from Pond ABE to Site C water reservoir/TSF.

The non-contact water from Sector E can be either collected to Pond ABE, or diverted to Zgounder River, using existing Sector E diversion ditch. The diversion of non-contact water into the Zgounder River will be required when TSF pond C is near capacity and in cases of high intensity precipitations. This ditch will be fitted with an automated flow diversion structure upstream of pond ABE. This flow diversion device will be used to divert non-contact water to Zgounder river under two (2) conditions:

- High intensity rainfall events, when pond ABE nears its capacity, to reduce the amount of water reported to pond ABE;
- Wet seasons where the system nears maximum storage capacity in TSF Site C.

An extension to the current eastern diversion ditch needs to be constructed between the flow diversion device and the valley below Pond ABE, leading to the river.

Pond D collects runoff water from Sector D, where the mining activity takes place. This sector is situated on the right bank of Zgounder river. Like Pond ABE, Pond D is not intended to be used as a storage pond; it is designed to smooth the precipitation inflow from a design flood event (up to 1:100-year) and enable its transfer to the TSF Site C water reservoir. The location of the water dam and the resulting storage capacity drove the minimum pumping capacity of 1.0 m³/s to adequately collect and convey a 1:100-year storm event. Collected water from the Pond D will be transferred to TSF Site C water reservoir via pipelines.

Both pumping stations would be located near the water collecting ponds, with an inlet structure that will supply the pumps. With a significant difference between the peak design flow and average flow, each pumping station could have multiple pumps equipped with variable frequency drives (VFDs). This would provide a wide range of pumping capabilities. At this stage, to keep the design simple and to limit the pumps horsepower, each pumping station is designed to have four (4) pumps with four (4) forcemain pipes, where each pump would discharge in an individual forcemain pipe. Each pump would be sized for 25% of the peak design flow.

Due to high pressure in the system for both pumping stations, an HDPE pipe vs. steel pipe could be installed to minimize the pressure surges. In addition, each forcemain would be equipped with a check valve isolation valve and pressure relief with surge anticipator valve. A drain line could be installed on each forcemain pipe to be able to drain these lines back to the pond.

Due to the high-power requirements from the pumps, each pumping station would have a high voltage service. With this high voltage service, it is expected that multiple transformers would be required to power the pumps and equipment of the station.

The pumping stations could be controlled by a programmable logic controller (PLC) panel to activate the pumps depending on the basin water depth and to ensure uniform wear of the pumping equipment.

The TSF at Site C is the main water storage reservoir on site, situated in the southernmost part of the mine site. The proposed water management strategy calls for the construction of water diversion ditches fitted with manual flow diversion structures, such as sluice gates or valves; this will enable the operator with the flexibility to either collect the water into the pond or divert into the environment the non-contact water from upstream of the reservoir. This diversion system is required in certain average and wet years to reduce the amount of water reporting in the system.

19 MARKET STUDIES AND CONTRACTS

19.1 Market Consideration

The end product that is planned to be marketed from the Zgounder Expansion plant is in the form of silver doré bars (silver ingots). The silver ingots produced at Zgounder will be of high purity, typically in excess of 98% of Ag content by weight. The silver doré bars are delivered to refineries where they will be refined to commercially marketable 99.9% pure silver bars.

Silver is considered a global liquid commodity, and is predominantly traded on the London Bullion Market Association (LBMA) and COMEX in New York.

Silver production will be sold on the spot market, with no plan currently to hedge any sales.

19.2 Commodity Price Guidance

Aya has established a standard procedure to determine the medium and long-term silver metal price to be used for Mineral Resource and Mineral Reserves estimates. This procedure considers the consensus of future metal price forecasts, projections from financial analysts specializing in the mining and metals industry, and metal price forecasts used by other peer mining companies in public disclosures.

Based on the above information, a recommendation as to acceptable consensus pricing is put forward to the company executives, and a decision is made to set the metal price guidance for Mineral Resource and Mineral Reserve estimates. This guidance is updated at least annually, or on an as-required basis.

Metal prices used for the December 2021 Mineral Resource Estimate (P&E Report) and for the Mineral Reserves Estimate as part of this Technical Report are listed in Table 19.1.

For the economic analysis of the Project, a silver price of $22.00/oz was used.

Table 19.1 – Metal Prices Used for the Mineral Resources and Mineral Reserves Estimates

Metal	Unit	Mineral Resource Estimate	Mineral Reserves Estimate
Silver	$US/oz	22.50	20.00

19.3 Product and Sales Contracts

Aya has been producing silver ingots for a number of years and has already a contract in place with a refinery for the delivery of silver ingots. Aya receives normally 99.7% of the value of its sale of silver ingots on delivery to the refinery, and there is also a transport charge of $0.15 per ounce and a $0.20 per ounce refinery charge.

19.4 Supply and Services Agreement

Contracts and agreements are currently in place for the supply of goods and services necessary for the current mining and processing operations.

These include, but are not limited to, contracts for diamond drilling services, mine development, ore haulage, specialised maintenance service for plant equipment, supply of diesel, supply of electrical power with the national electrical power provider, supply of explosives, supply of consumables and process reagents including grinding media, sodium cyanide, collectors, and transportation and logistics services including catering and personnel transportation.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

The information presented in this Section of the Technical Report *has been either translated, extracted and/or summarised* based on the Environmental and Social Impact Assessment (ESIA) report prepared by NOVEC entitled " *Etude d'impact Environnemental et Social de l'expansion du Projet Minier Zgounder*", Final Version, dated December 28, 2021 and February 2022 as well as Aya Environmental Surveillance and Monitoring Program prepared by Aya Gold and Silver - Zgounder Millenium Silver Mining Report entitled "*Etude d'Impact Environnemental et Social de l'expansion du projet minier Zgounder* - *Programme de Surveillance et de Suivi Environnemental*" dated December 2021. Earlier Technical Reports 43-101, closure plan as well as the hydrogeological study prepared by Englobe in March 2021 along with various other information provided by Aya have been consulted and are listed in *Section 27 of the Report.*

20.1 Regulatory Overview

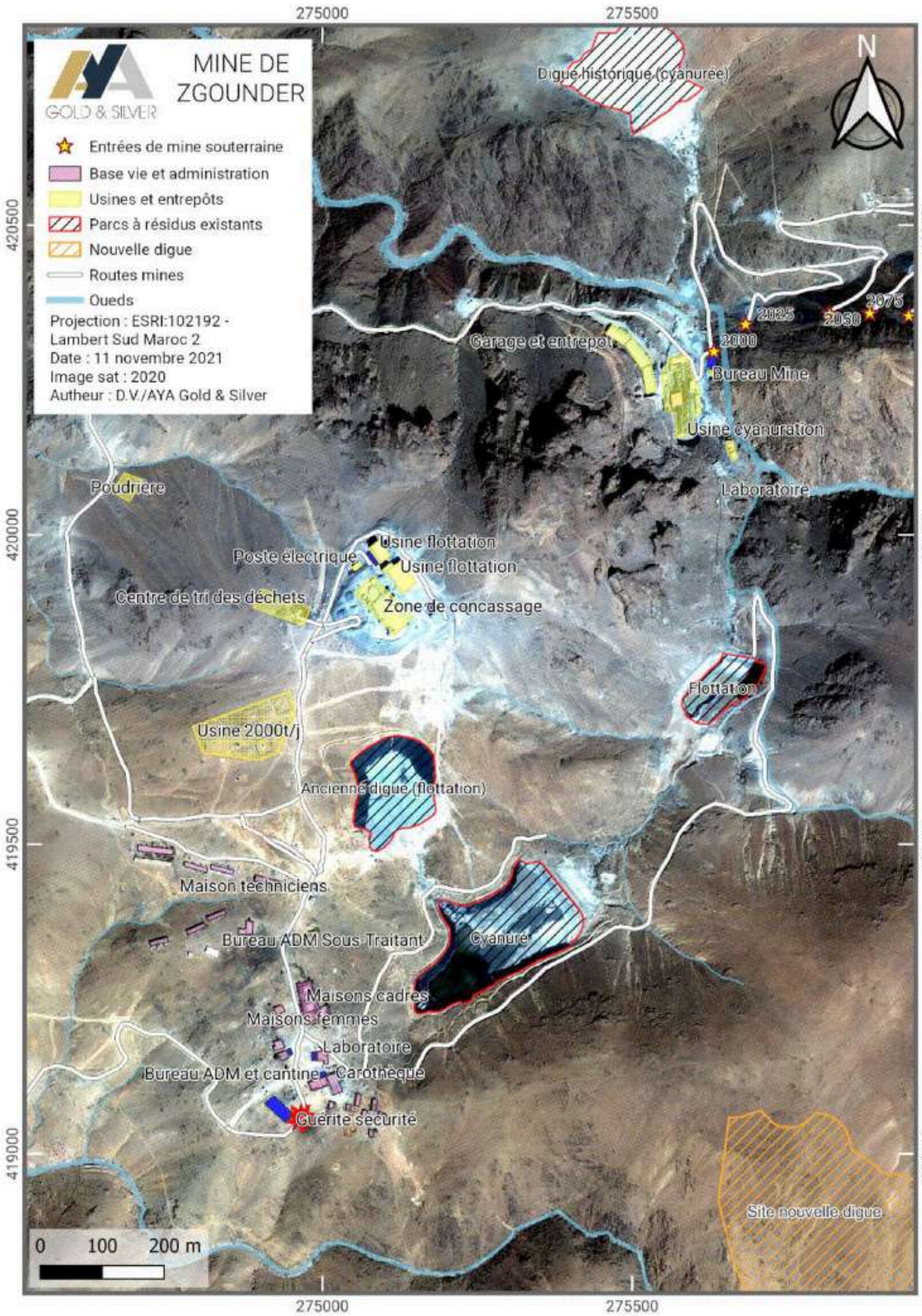
20.1.1 Introduction

The first Environmental Impact Assessment (EIA) study of the Zgounder mine was prepared in 2013 by Hydraumet Maroc. The area covered by this impact study is registered under Title No. 09/2096 issued by the Rabat Mines Directorate. Following the EIA, operating permit No. 2306, which included exploration permit, surface rights, access to property and any type of mining operations was issued to Maya Gold and Silver Inc. by ONHYM. On August 15, 2014, the operation of the Zgounder mine by ZMSM received environmental acceptability from the prefecture of Agadir Ida-Outanane.

In December 2021, NOVEC submitted a new Environmental and Social Impact Assessment (ESIA) as part of the Zgounder Silver Mine Expansion Project. The International Finance Corporation's Performance Standards were applied when defining the scope and terms of reference of this new ESIA.

This expansion project includes an open pit mine, a waste dump area, a new 2,000 t/d concentrator and a new tailing impoundment. Figure 20.1 shows the mine facilities with the future infrastructure indicated in colour.

Figure 20.1 – Zgounder Mine Existing Facilities



Reference: NOVEC ESIA, 2021

According to NOVEC's ESIA, no new permits are required for the Zgounder Silver Mine Extension Project under the current regulatory framework.

Impact studies are governed by Law 12-03, which aims to harmonise the procedures for preparing and reviewing impact studies at the national level. It delimits the scope of the law enforceable against public and private projects which, by reason of their size or nature, are likely to have an impact on the environment. It defines the objectives and content of an impact study and makes the granting of any authorisation for the realization of said projects conditional on obtaining an "Environmental Acceptability" decision. The law provides for a mandatory public inquiry, the conditions of application of which are laid down in Decree No. 2-04-564.

According to NOVEC's ESIA, the mine expansion is not subject to an environmental impact assessment under the said Act.

20.1.2 Legislative and Regulatory Framework

Morocco has developed an environmental policy aimed at preserving ecosystems and promoting sustainable development. The Moroccan legislative framework includes several legislative texts aimed at protecting public health and the environment.

Mining activities are governed by several legislation, a few of these are listed below:

- Law 33-13 (2016), Mining Code;
- Decree n°2-15-807 (April 20, 2016), on the application of Law 33-13;
- Dahir no 1-03-59 - Law No. 11-03 on the protection and enhancement of the environment;
- Dahir no 1-03-60 - Law No. 12-03 on environmental impact assessments studies and their implementing Decrees;
    - Decree No. 2-04-563 of November 4, 2008, on the functions and operation of the national committee and regional committees studying environmental impacts;
    - Decree No. 2-04-564 04 November 2008 listing rules for the organization and conduct of the public inquiry into the Project subject to the studies of environmental impacts;
- Dahir no 1-03-61 - Law No. 13-03 on the fight against air pollution;
- Law No. 49-17 on Environmental Assessments (August 13, 2020);
- Law 99-12 relative to a national charter on the environment and sustainable development;
- Law 42-6 concerning Paris Agreement on climate change;
- Law No. 28-00 on waste management and disposal;
- Law 36-15 concerning water;
- Dahir No. 1-69-170 of July 1969 on the protection and restoration of soil;
- Dahir 20 Hijja 1335 (1917) on forest conservation and exploitation.

20.1.3 LEGAL CONVENTIONS

Eight conventions ratified by Morocco are applicable to the Project:

- Paris Climate Accords;
- Ramsar Convention on Wetlands of International Importance Especially as Waterfowl Habitat;
- UNESCO Convention Concerning the Protection of the World Cultural and Natural Heritage;
- Convention on International Trade in Endangered Species of Wild Fauna and Flora
- Bonn Convention on Migratory Species;
- Vienna Convention for the Protection of the Ozone Layer;
- United Nations Framework Convention on Climate Change;
- Convention on Biological Diversity.

20.2 Biophysical Baseline Environment

20.2.1 SOILS AND LAND CAPABILITY

The topography of the Zgounder mine site is characterized by steep hills with elevations of around 2,100 m with low valleys and seasonally flowing rivers (oueds).

In the study area, the risk of landslide is medium. Indeed, the rainfall regime of the area, the slope of the ground, the geology, the characteristics of the soil, the soil cover, and the earthquakes probability lead to an average risk of localized landslides.

ENGITECH/TEVARI is responsible for the environmental monitoring of the Zgounder Mine developed since 2014. The results from the July 2014 sampling campaign are considered as reference samples, even though they are not representative of the initial state of the area before mining activities. The metal concentrations were above agricultural norms within the mine site and tailings impoundments.

As part of the ongoing monitoring program, topsoil serving as a substrate for the crops located downstream of the mine are sampled. The parameters analyzed are Cu, Zn, Pb, As and Hg. The concentrations are below the threshold applied to agricultural land (Tevari, September 2020). Due to lack of information, the sampling location is not specified.

A soil characterisation was performed by Labomag in May 2021. The soil samples were taken approximately 2.5 km downstream of the mining activities along the Zgounder oued. The soils were defined as having a remarkably coarse texture. The parameters analyzed were Cu, Zn, Fe, Mn, Cd, Pb, As, Ni, Cr, Co, Ba, Hg, and CNtot. The petroleum hydrocarbons were not characterised. There were significant concentrations of lead and cyanide found at two (2) locations.

Since the soil is coarse, then the risk of infiltration and migration of contamination is high. The identification of cyanide in soil at 2.5 km downstream is a major concern. As recommended by NOVEC in the ESIA, further investigation is required.

20.2.2 Flora

In the study area, natural vegetation such as argan trees, olive trees and almond trees, is found mainly on the edge of the Souss oued and its tributaries, and in wildlife parks.

Near the rivers, the vegetation is diverse with, oleanders, reeds, colchiques, and exotic species, such as Berber cacti. Land use is dominated by the practice of annual crops and arboriculture. Within this perimeter, citrus, almond and wild olive trees occupy a large fraction of the area.

Elsewhere vegetation has been modified by man. The vegetation introduced by man are eucalyptus, poplar, and fruit trees (almond, olive, fig). The Argan tree, an endemic plant of the region is not present. In the Souss valley, jujube bushes are found. Low crops, namely cereals (wheat, barley, corn) and vegetables (tomato, potato, pepper, alfalfa, etc.) are cultivated.

Near the project site, the vegetation is limited to mosses, lichens, and a few trees.

20.2.3 Fauna

In the watershed of the Oued Souss, there is a multitude of species of mammals, including: red fox, wild boar, African golden wolf, cape hare, squirrel of getulie, Algerian hedgehog, gerbil, gray mouse, bat, etc. The species most represented and best adapted to the environment are rodents and chiroptera. The mining site, being practically bare does not attract avifauna.

20.2.4 Water Resource

20.2.4.1 Hydrology

The Zgounder mine site is located within two (2) watersheds, the Zgounder Basin and the Aoulouz Basin. These two (2) basins are located in the upstream part of the large watershed of Souss-Massa. They are drained respectively by the oued Zgounder (or Achkoukchi) site and the oeud Aoulouz. These two streams flow into the main river of Souss. The runoff on the site is generally from east to west, from the high mountains of Siroua to the plain of Souss-Moussa.

The oued Zgounder which crosses the mine site is an intermittent stream with seasonal flow. This river sometimes experiences strong seasonal variations in flow regime, with high water usually recorded from November to May and low water levels from June to September.

20.2.4.2 Surface Water Quality

In the ESIA, a characterisation of surface water was performed by Labomag laboratory. A total of ten (10) water samples were taken and analyzed in May 2021. As shown in Figure 20.2, five (5) samples were taken in the vicinity of the mine site, while the others were taken downstream of the Project along the Zgounder oued. The results were compared to Decree no 1276-01 of 2002 criteria for surface water quality. All the sampling stations showed total Kjeldahl nitrogen (NTK) values above criteria. In the mine area, the concentrations were above criteria for COD, total phosphor, zinc, iron, cupper, manganese, NTK, cadmium, nickel, arsenic, detergents, cyanide, phenols, and total hydrocarbons. The concentrations at the sampling stations located 2.5 km (Eau 7) and 4.7 km (Eau 8) downstream met the norms, except for anionic detergents and phenols.

Figure 20.2 – Surface Water Monitoring Stations in 2021

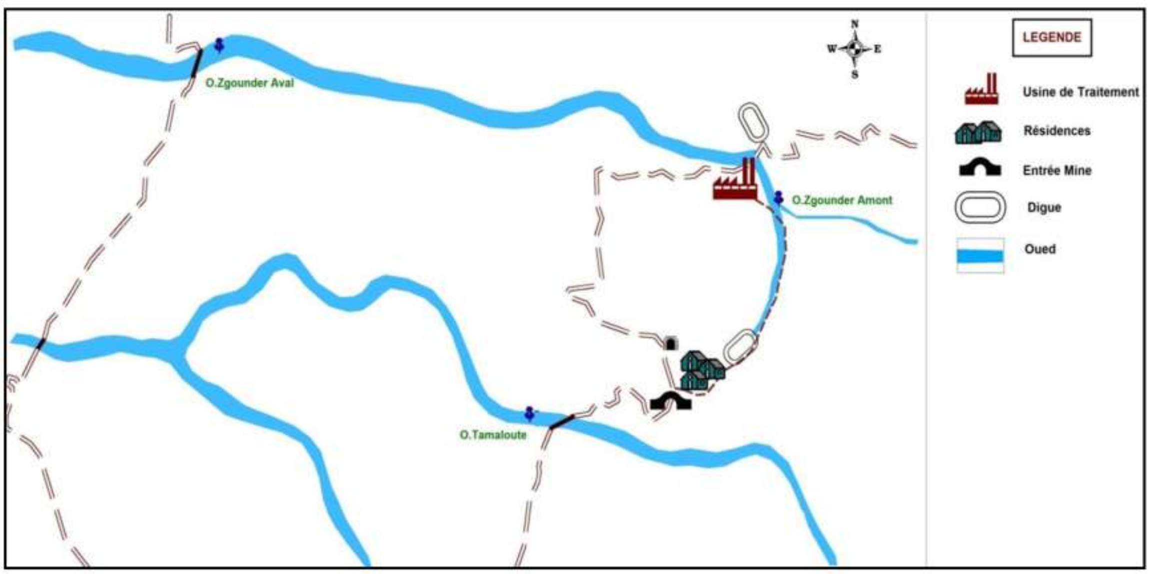
Source: NOVEC ESIA, 2021

ENGITECH/TEVARI is responsible for the environmental monitoring at the Zgounder Mine site. As mentioned in the Section 20.2.1, results are compared to a 2014 campaign which is not representative of the water quality prior of the mining operation. Sample "Oued Zgounder Amont" is also used as a reference for the water quality upstream of the mining site, even though this sampling point is located downstream of the abandoned tailings impoundment, which could be a source of contamination.

Figure 20.3 shows the location of the sampling stations. The parameters analyzed are pH, conductivity, MES, DBO, DCO, NTK, Pt and microbiology) and metals (Cu, Zn, Pb, As, Hg, and Cn) at a frequency of every 3 months. Frequently, the streams have little flow and the water has a neutral pH.

The sampling showed that the water quality has been periodically impacted in the immediate surrounding of the Zgounder Mine site. TEVARI noted anomalic concentrations in Cu, As, CN near the treatment plant where accidental spills tend of occur. In September 2020, Tevari mentions concerns about increasing concentrations in arsenic. However, the sample station located 3 km downstream is consistently meeting water quality norms. In 2021, a new sampling station was added nearly 300 metres downstream of the mine site along the Zgounder river, concentrations in Cu , Zn and As were above irrigation criterias.

Figure 20.3 – Environmental Program Surface Waster Monitoring Stations



Source: TEVARI, September 2020

In January 2020, Géobel mandated by Englobe sampled and analyzed surface water in the Zgounder mine and obtained similar results in terms of contamination in metals and cyanide.

As a recommendation, more sampling stations should be added upstream of the mine site and near mine discharge points. The frequency should also be increased to help improve intervention and control of the operations.

20.2.4.3 Hydrogeology

A hydrogeological study prepared by Englobe Corp. and was issued in March 2021[7]. The objective of this study was to collect existing geological and hydrogeological information in the area of the mine, in order to evaluate the aquifer potential for future need in process water.

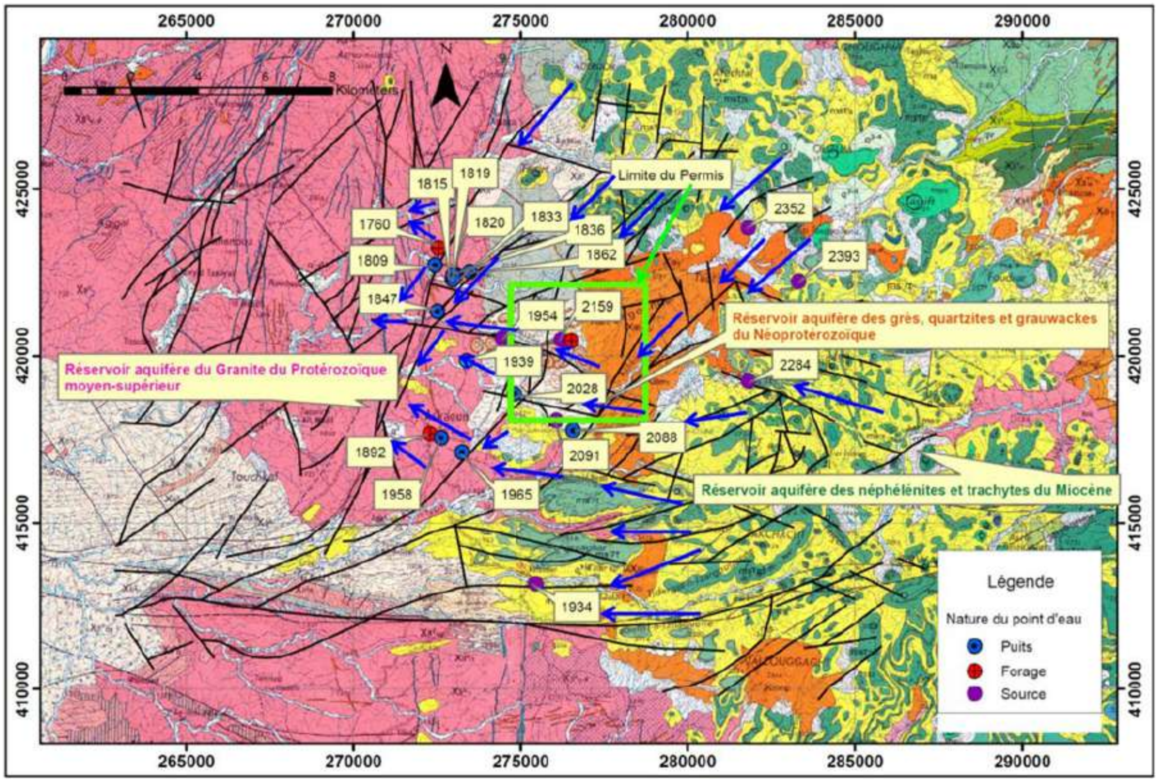
The Project area is classified as a discontinuous aquifer area directly related to the existing fracturing network, in sedimentary, metamorphic, and volcanic rocks, which could be productive, since recharged by precipitation and snowmelt. In this region, water sources are observed along faults and at the contact level of permeable and impermeable layers, which confirms the presence of an aquifer. Several types of wells are observed in the valleys, from traditional wells and others equipped with pumps to provide potable water and in some case irrigation water and are less then 15 m depth. There are no wells at the mine site.

The majority of the wells are sinked into granite faults and the water level varies between 3 and 22 m.

In general, based on topography, fracturing zones and the hydrographic network, regional groundwater flow in the area is oriented northeast to southeast. Figure 20.4 depicts the presumed groundwater flow axes of the Zgounder region.

[7] Englobe 2021. *Étude hydrogéologique Phase 1, Recherche en eau souterraine Mine d'argent de Zgounder*, Final Report. Prepared for Aya Gold & Silver, by H. Tezkratt. Issued March 2021, 79 p.

Figure 20.4 – General Direction of Groundwater Flow



Source: Englobe, Hydrogeological Study, March 2021

Since 2014, the environmental monitoring and surveillance program has been monitoring groundwater from a single well identified as downstream of Zgounder located 5 km west of the mine site. The parameters analyzed are the same as surface water. The water quality is, in compliance with regulatory norms, but concentrations in nitrates (NO_3) are high (June 2020: 22.8 mg/L and September 2020: 85.10 mg/L) compared to surface water sampling stations at the mine (< 5 mg/L). Nitrates are commonly used in explosives.

Information concerning the local hydrogeology at the mine site is insufficient and considering the high potential of infiltration previously mentioned and the presence of contaminants near the mine infrastructure, it is recommended that more monitoring wells be installed and that a hydrogeological investigation be performed. Petroleum hydrocarbons should also be added to the list of parameters.

It must be noted that many improvements have been made to the operation site. The new tailings impoundment is lined with an impermeable geomembrane, which protects infiltration from reaching the water table. This is a very good and effective mitigation measure.

20.2.5 Air Quality

20.2.5.1 Climate

The Zgounder mine is located between 1925 and 2200 MASL on the western side of the Siroua massif of the Anti-Atlas. This region is separated from the influence of the Mediterranean climate by the High Altas mountains to the north and therefore shares the climate of the Sahara. The region is semi-arid as the Sahara Desert is less than 50 km away. Winters are cold and up to 50 cm of snowfall can occur above 1600 m altitude during the first quarter of the year. Average annual precipitation is 500 mm, with the driest month being July with 3 mm of rain, and December the rainiest with an average of 70 mm. The mean annual temperature is approximately 12.2°C, summers are warm to hot and mostly dry.

The Siroua massif is usually covered with snow during winter. The rivers in the area are fed directly by snowmelt.

20.2.5.2 Air Quality

NOVEC has mandated SGS to perform ambient air quality measurements in 2021. The sampling took place at four (4) locations: three (3) at the mine site mine and one (1) in the municipality of Askaouen. Samples and measurements were carried out in accordance with Decree No. 2-09-286 of 20 hija 1430 (8 December 2009) establishing air quality standards and air monitoring procedures. The parameters measured were SO_2, NO_2, O_3 CO, HCN, PM10 and heavy metals: As, Mn, Cr, Ni, Pb, Hg.

In comparison to the Moroccan reference limit values mentioned in ministry decrees, all results are lower than the threshold values, except for dust PM10 at three (3) locations. The source of emissions is assumed to be hauling and dumping of trucks and, wind erosion which can be controlled with water trucks, paved roads and/or installation of wind barriers.

Monitoring of ambient air was not part of the environmental surveillance and monitoring program initiated in 2014 and it has been included and updated in 2021.

20.2.6 Environmental Noise

NOVEC conducted in March 2021 a three-day noise measurement campaign at three (3) locations in the Zgounder mine: the cyanidation plant, the camp, and the access to the underground mine. The measurements at the mining site respect the legislation criteria.

Noise monitoring was not part of the 2014 surveillance program. It is, however, included in the proposed 2021 surveillance and monitoring program (pre-construction and construction).

20.3 Socio-Economic Baseline Environment

Socio-economic development activities in the Kingdom induce in general an improvement in living conditions, which in turn induce an increase in the population. Table 20.1 describes the demographic situation of the municipalities in the project area.

Table 20.1 – Demography in the Project Area

Municipality	Population	Households	Household Size
Askaouen	7,458	1,306	5.7
Taouyalate	7,465	1,370	5.4
Ouzioua	7,670	1,425	5.4

The household size in the study area is higher by approximately one person when compared to the national average (4.6).

It is also known that in the area:

- The population is generally young, since the percentage of people between the ages of 15 and 59 exceeds 50%;
- The illiteracy rate of the population is relatively high, exceeding 50% of the population for the commune of Taouyalate;
- The ratio of the active population compared to the total population, varies from 50.7% in Askaouen, to 44.1% in Taouyalate. It is the lowest in Ouzioua with 34.4%.

In terms of sectors of activity, agricultural activities are the most practised at the level of the study area. The percentage of working women is 45% in the municipality of Askaouen. On the other hand, the female labour force in Ouzioua is only 7%.

The study area is mountainous, the nearest villages are Aoulouz located 2.5 km west of the mine site and Myal and Ait Himmi located 2.7 km and 3.4 km north-west of the mine.

The villages are located near rivers and streams (oueds) allowing agriculture (cereals, vegetables and some trees). The local population is exclusively Amazigh (indigenous population) with a semi-sedentary way of life. The local economy is mainly supported by farming, agriculture and trade (saffron, potatoes, dates, and henna) including the weaving of renown carpets (the Tazenakht carpets). The Siroua region has significant tourist potential and can be visited from spring to early autumn.

20.4 Stakeholder Consultation and Disclosure

In the 2021 ESIA, no public consultation is mentioned. However, the previous ESIA, prepared in 2013 by Hydraumet, refers to a socio-economic survey conducted in Askaoun and in the villages of the Province of Taroudant. The following impacts related to the Project were discussed:

- Air quality: mainly dust control from the roads and tailings impoundments;
- Sound characterisation and protection;
- Storm water runoff damages;
- Groundwater controls: chemical and petroleum product storage tanks;
- Wastewater discharges: identification and control;
- Soil protection: revegetation to limit erosion;
- Fauna and flora: mitigation measures to bring diversity;
- Management of heavy metal and cyanide.

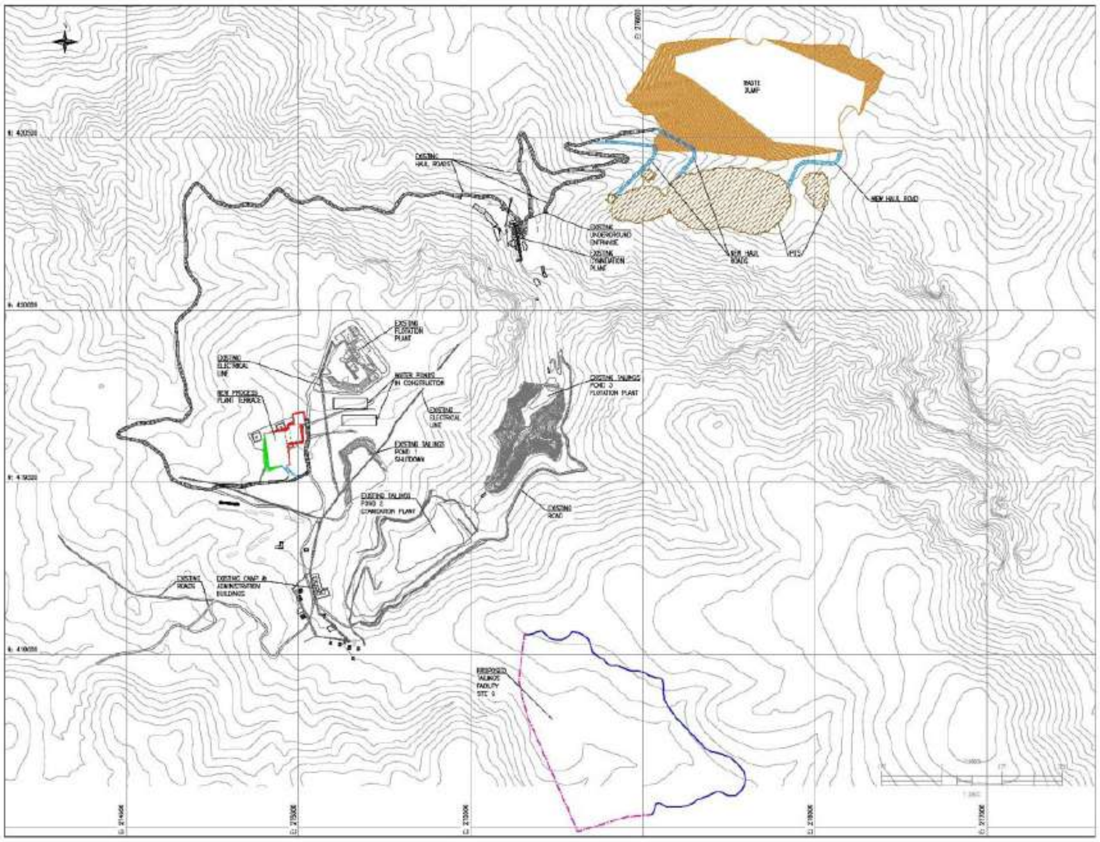
The 2021 Environmental surveillance and monitoring program addresses these topics.

20.5 Environmental and Social Risks and Impacts

20.5.1 General Information concerning existing and new infrastructures description

The main features of the Zgounder Mine are shown in Figure 20.5. The new infrastructure such as the open pit, waste dump and tailings are also indicated in colour.

Figure 20.5 – Zgounder Mine Existing and Future Infrastructure



Source: NOVEC ESIA, December 2021

The ore is extracted from the underground mine and haul to the 200 t/d cyanide plant and the recently constructed 500 t/d flotation plant. There are four (4) tailings impoundments on-site. The historical dam located in the northern portion of the site has been closed since 1990 and contains approximately 500,000 tonnes of material. As part of the Project expansion, it is planned to treat the tailings again to extract the remaining silver content. The tailings impoundment located south of the flotation plant closed in 2020 and is being revegetated. The other two (2) tailings are in operation and are referred to as the cyanide and the flotation dams.

Besides the process of the new treatment plant and location of the new infrastructure, little information is given concerning the open pit mine, the waste dump design and slope stability, the water management and effluent discharge location. Information concerning the equipment fleet required for the new open pit mine is not specified, nor the need to expand the maintenance facilities and petroleum production storage, if required.

The location of the new Tailings Storage Facility was selected considering the advantageous topography for the installation of the geomembrane. The final height of the dam will be of 48 m. No information is given concerning the design criteria and stability factor.

The workers camp capacity may need to be increased and the capacity of the associated sewage system may also require to be upgraded.

20.5.2 Environmental and Social Risk

The 2021 NOVEC ESIA examined and measured the magnitude of the impacts, both positive and negative, on the physical, biological, and social environment during the different stages of the Zgounder Mine.

The components of the environment, which are likely to suffer impacts, are grouped according to the medium concerned and classified according to their sensitivity. The analysis of this sensitivity makes it possible to define the level of resistance that the element presents in relation to the Project.

This sensitivity is the crossing of the apprehended impact and the value of the element as presented in Table 20.2.

Table 20.2 – Environmental Sensitivity Evaluation

Description	Elements	Anticipated impact	Value	Sensitivity
Physical Environment	Ground	Weak	Average	Weak
	Ambient air	Medium	Average	Average
	Water quality	Medium	Strong	Strong
	Natural landscape	Medium	Weak	Weak
Biological Environment	Flora	Weak	Average	Weak
	Fauna	Weak	Average	Weak
	Wetlands & Protected areas	Weak	Weak	Weak
Social Environment	Population and Habitation	Medium	Average	Average
	Health and safety	Medium	Average	Average
	Noise	Weak	Average	Weak
	Socio-economy activities / Employment +++	Medium	Average	Average
	Infrastructure and equipment	Medium	Average	Average
	Archeology and heritage	Weak	Weak	Weak

+++ : Positive Impact
Source: NOVEC ESIA, 2021

Detailed and specific mitigation measures are presented for each significant impact in the NOVEC ESIA report. These measures are also found in the 2021 Environmental and surveillance program report and are part of Aya's commitment to preventing and controlling contamination.

20.6 Management and Monitoring Plans

An Environmental Monitoring Plan (EMP) was validated by the Moroccan authorities in 2014, and the firm TEVARI has conducted quarterly site visits since 2015. Water and soil quality and tailings stability were the main aspects covered. The reports from 2015 to 2021 are presented in the ESIA appendices, and the January to September 2021 were made available for consultation

AYA is committed to maintaining environmental monitoring throughout the various phases of the mine life, to the implementation of mitigation measures and to reporting. This plan must be followed by qualified personnel and/or delegated entity for external monitoring and control.

An updated version of the environmental monitoring and surveillance plan was prepared in December 2021 with the addition of new elements and indicators. Table 20.3 is a summary of the proposed monitoring and surveillance plan.

Table 20.3 – Proposed Environmental Surveillance and Monitoring Program

Element	Indicator	Location	Frequency
Noise	Noise Level	Surroundings of the project area	1 x / Month
Soil	Reporting accidental spills Contaminant, quantity and quantities of soil excavated	Incident location	1 x / Quarter
Air quality	Monitoring of air quality indicators (PM10, PM^2.5, CO, CO_2, NOx, NO_2, etc.) and mercury	Surroundings of the project area	1 x / Quarter
Landscape integration	Visual appearance of all equipment and at the discharge point	Study area	1 x / Year
Fauna and Flora	Status of presence of species: Number of dead birds per year and lost habitat, ornithological indicators: Kilometric Abundance Index (IKA) and Point Abundance Index (IPA) State of flora in temporary storage areas	Study area	1 x / Year
Affected Community	Incidence of pollution-related illnesses and deaths Water quality and availability	Study area	2 x / Semester
	Socio-economic development indicators	Study area	1 x / Year

Element	Indicator	Location	Frequency
Surface and groundwater	Level and quality of groundwater and surface water (MES content) at the catch area	Determined stations	Monthly
	Accidental spill quarter quality	Study area	At each incident
Potable water	Water quality	STEP	1 x / 3 months

In terms of best practices, here is a list of recommendations:

- Add noise monitoring stations near sensitive receptors such as the camp and the closest village;
- Maintain a logbook of the dust condition along the mine haul road, waste dump and tailings and of the mitigation measures applied;
- Maintain a register of public complaints;
- Increase the amount of groundwater monitoring wells and add petroleum hydrocarbons to the list of parameters;
- Increase the surface water sampling frequency and add petroleum hydrocarbons to the list of parameters.

20.7 Mine Closure

AYA has developed a Conceptual Closure Plan (CCP) to ensure a safe and sustainable environmental and physical closure of the Zgounder mine at the end of operations. In the absence of a Moroccan legislative framework, Aya has followed the objectives of international good practices, such as:

- Protect public health and safety;
- Reduce or prevent environmental degradation;
- Allow appropriate time for revegetation;
- Return the mine-affected area to a state that is consistent with stakeholders' land use expectations.

The CCP takes into consideration, certain assumptions are as follow:

- Runoff and seepage quality within the industrial site (waste rock storage, tailings) will be acceptable in accordance with Moroccan legislation and will not require additional treatment;
- Groundwater quality will be acceptable;
- Progressive rehabilitation will take place during the mining operation;

- The costs of decontamination, dismantling and relocation of the equipment of the treatment plants will be offset by the selling price;
- The buildings (offices, workshops, accommodation camp, etc.) will be in an acceptable condition to be transferred to a third party;
- Roads necessary for inspection and monitoring will be maintained;
- No re-sloping or rehandling of material of dikes or waste pile slopes are required;
- An allocation of USD 1 million is planned for the general cleaning of the site, the removal of materials and equipment, the dismantling of permanent electrical and piping connections to modular buildings, etc.;
- An allocation of $50,000 is provided for the treatment of hydrocarbon contamination on the surface.

In general, the mine pit will be secured to avoid accidents and the industrial site, including the tailings dykes and surface, will be covered with 20 cm of topsoil and seeded with grass and bushes.

Public consultation will take place before finalising the mine closure plan. A closure surveillance and monitoring will be defined in collaboration with the authorities. The program will remain in place during the decommissioning period and at least for a 5-year period after closure.

The water runoff and seepage will be monitored as well as the stability of dykes of waste rock and tailings. Areas above underground works will also be inspected in order to identify signs of settlements. The success of revegetation activities will be monitored to ensure viable and self-sustaining vegetation growth in rehabilitated areas and to determine whether other vegetation support activities are warranted.

The estimated closure cost for the Zgounder mine is USD 6.6 million. This includes an estimated 10% of the total for management fees and 20% for contingencies.

21 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimate

The Capital Cost Estimate ("Capex") is based on the scope of work as presented in earlier sections of this Report. The Capex consists of direct and indirect capital costs as well as contingency. Provisions for sustaining capital are included in Aya's financial model. The Capex is reported in United States Dollars ($).

21.1.1 Basis of the Estimate

The initial Capex estimate includes all Projects' direct and indirect costs to be expended during the implementation of the Zgounder Project, inclusive of an upcoming basic engineering as well as the execution phase, complete with detailed engineering. The Capex is deemed to cover the period starting at the approval by Aya of this Report and finishing after commissioning is achieved. It should hence be understood that this Capex excludes transfer to Aya operations, performance test, start-up, ramp up and operations.

The sustaining Capex estimate includes all Projects' direct and indirect costs to be expended throughout the life of mine.

21.1.2 Mandate

For this Report, DRA is responsible for estimating and compiling the initial and sustaining Capex for the entire Project, including Owner's costs.

DRA prepared the Capex for all facilities except for the new TSF Site C.

GCIM prepared the design and quantities for the initial and sustaining capital cost estimates for the TSF. The capital expenditure associated with the TSF has been determined to a Class 3 AACE accuracy of +25%/-15%, based on quantities measured by GCIM and rates sourced from Moroccan earthworks tender enquiries undertaken and provided by DRA. The Capex has been apportioned across the LOM, with the first phase of TSF 1 as the initial Capex, and the sustaining capital comprising the remaining phases.

21.1.3 Capex Estimate Accuracy

The accuracy of the initial Capex estimate is assumed at ±20.

21.1.4 Deliverables

The Capex is developed based on the following list of deliverables:

- Project description;
- Mine plans, complete with initial mining equipment and pre-production costs;
- Mechanical equipment list;
- Material Take-Offs (MTO) for all the civil works, earthworks, concrete, structural and miscellaneous steel, buildings
- MTO for major electrical equipment, including the power plant;
- MTO for tailings storage, including tailings' roads, as well as tailings and reclaim water pipelines;
- Overall general arrangement plan.

21.1.5 Exchange Rates

The base currency for this Report is the US Dollar (USD). For information purposes, the following is a preliminary list of currencies that could potentially be used, along with the conversion exchange rates based on an average of the previous 90 days prior to finalisation of the Capex. The currency exchange rates shown in Table 21.1 will form the basis for any currency exchanges used in the Capex. The rates are based on averages in November 2021.

Table 21.1 – Currency Conversion Rates

Currency	Code Name	Exchange Rate	US Dollar
USD	US Dollar	1.000	1.000
CAD	Canadian Dollar	1.318	0.759
AUD	Australian Dollar	1.408	0.710
EUR	Euro	0.8990	1.112
CNY	Chinese Yuan	6.724	0.149
ZAR	South African Rand	14.558	0.069
MAD	Moroccan Dirhams	9.55	0.105

21.1.6 Estimating Software

The Capex estimate was developed using MS Excel.

21.1.7 Methodology

21.1.7.1 Plant Equipment and Bulk Quantities and Material Costs

A mechanical equipment list was developed by the engineering team from DRA. Semi-detailed estimates, supplemented by general arrangements drawings, were used for civil works, including earthworks, concrete, and structural steel. To ensure the entire scope coverage, some allowances were added, based on DRA's experience. Piping as well as instrumentation and controls, were factored from mechanical costs.

21.1.7.2 Reference Documents

The estimate is developed in accordance with the project criteria. Specifically, the estimate was based on the following:

- Project EPCM Schedule;
- Process Design Criteria;
- Engineering Discipline Design Criteria;
- Process Flowsheets;
- Single Line Diagrams;
- Mechanical and Electrical Equipment Lists;
- General Arrangement and Stockpile Layout Drawings;
- Discipline MTOs;
- Vendor Quotations;
- Historical Data;
- Specific unit rates computed from adjudicated contractor's bids;
- Drawings.

21.1.7.3 Estimate Execution

The capital cost estimate is organised in accordance with the Project Work Breakdown Structure (WBS) defined for both direct and indirect areas.

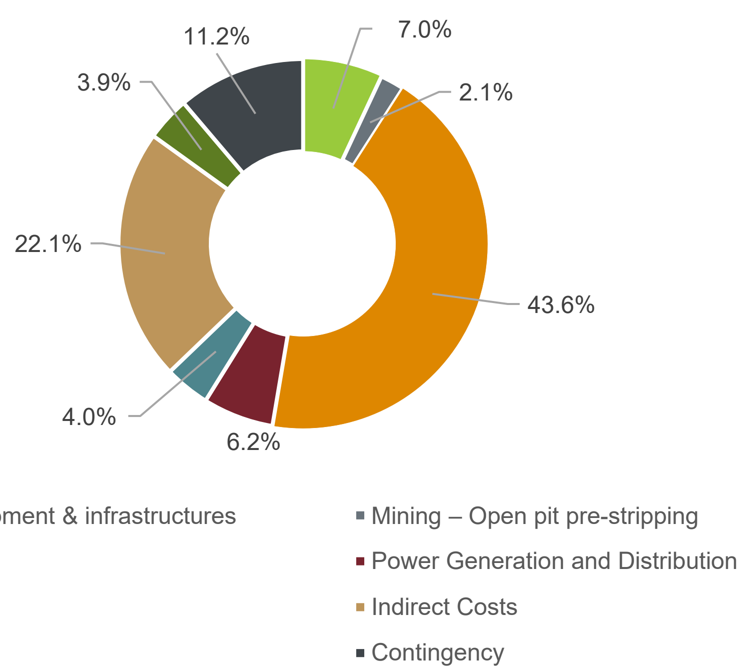
21.1.8 Capital Cost Estimate Summary

The initial Capex for the Zgounder expansion is presented in Table 21.2. The sustaining Capex covering foe the existing facilities as well as the expansion is distributed over the LOM, but not included in the initial Capex. All costs relate to the area within DRA's scope unless otherwise noted. Owner's costs, its contingencies and risk amounts are included in this estimate.

Table 21.2 – Initial Capex Summary by Major Area (USD)

WBS	Major Area	Total Cost ($ USD)
1000	Mining – UG equipment & infrastructures	9,713,352
2000	Mining – Open pit pre-stripping	2,943,170
4000	New Processing Plant	60,770,216
5000	Power Generation and Distribution	8,643,184
6000	TSF	5,536,670
	Sub-Total Direct Costs	***87,606,593***
9000	Indirect Costs	30,808,881
10000	Owner's Costs	5,386,250
20000	Contingency	15,621,809
	Grand Total	**139,423,533**

Figure 21.1 – Initial Capex Summary by Major Area (USD)



21.1.9 Basis of Estimate – Direct Costs

21.1.9.1 Quantities

Engineering MTOs are based on "neat" quantities derived from project drawings and sketches. No allowances are included in the MTO's, for any discipline. Quantities are estimated where drawing information is not available.

Metric units are assumed throughout the estimate except for piping. Pipe sizing is defined in inches of nominal diameter.

21.1.9.2 Unit Quantity Preparation

Quantity preparation is the responsibility of the discipline engineering leads. The MTOs, by discipline, are reviewed on an on-going basis as MTO assumptions are defined.

MTOs are produced primarily from drawings generated either during the Project or as received from Aya.

21.1.9.3 Civil

All earthworks quantities are taken off neat in place, with no allowance for swell or compaction of materials.

Mass earthworks estimates are based on soil studies information, AutoDesk, land desktop models, drawings and sketches. Waste allowance is included as per discipline by the estimating team. Production rates for mass earthworks are not used, rather all-in rate prices for the local area as obtained from adjudicated all-in direct cost rate obtained from bids received from three (3) vendors were utilised. It is assumed that the all-in rate included for material cost if needed, equipment, small tools, labour and benefit cost, contractor's overhead and profit.

Detailed excavation and backfill quantities for buildings and structures were developed for each area, based on estimated foundation sizes. An allowance for hauling materials one (1) kilometre is included. Costs include loading, crushing, screening and delivery to the area of operation.

21.1.9.4 Concrete

Concrete MTOs are taken off neat, without allowances for over pour or wastage. No allowance for these items within the concrete supply price is included. Production rates for concrete are not used, rather all-in rate prices for the local area as obtained from adjudicated all-in direct cost rate obtained from bids received from three (3) vendors were utilised.

It was assumed that the all-in rate includes for material cost, equipment, small tools, labour and benefit cost, contractor's overhead and profit. Quantities for rebar, formwork and miscellaneous steel have their all-in price build-up calculation separate from the concrete pricing.

MTOs are developed based on concrete shape and forms. Each concrete shape includes concrete, rebar, formwork and additives. Inserts, embedded material, coatings, liners and painting are itemized separately in the estimate.

21.1.9.5 Structural Steel

Structural steel MTOs are developed from sketches taken from general arrangement and layout drawings. A Discipline allowance is applied for cut and waste, connections, clips and hardware.

A supply only (delivery cost excluded) rate for the local area as obtained from adjudicated all-in direct supply rate obtained from bids received from three (3) vendors were utilised and installation production rate was then multiplied by the productive factor and the unit labour rate for structural steel. The supply cost and installation cost were then added together to arrive at the total cost for all the various steel profile including light, medium, heavy and extra heavy. Structural steel delivery cost is added to the project freight cost.

21.1.9.6 Architectural

Architectural MTOs are developed from general arrangement and layout drawings for the appropriate area as required.

All building components (i.e. interior partitions, doors, windows, toilets and accessories, etc.) have been quantified and prices.

All buildings are assumed stick-built with separate allowances for plumbing, electrical, furnishings and grounding.

A supply only rate for the local area as obtained from adjudicated all-in direct supply rate obtained from bids received from four (4) vendors were utilized and installation production rate was then multiplied by the productive factor and the unit labour rate for architectural buildings. The supply cost and installation cost were then added together to arrive at the total cost for all the various items including walls, roofing, block-walls, bricks, partitions, hardware, etc. Required foundations and detailed earthworks were also provided in the MTO's and priced accordingly.

21.1.9.7 Mechanical

The mechanical equipment descriptions, quantities, sizes, capacities, equipment specifications and powers are provided in the project Mechanical Equipment List.

Budgetary quotations were obtained for almost all process plant equipment and plateworks, mining equipment and all the power plant and major electrical equipment, this is equivalent to 98% of total direct equipment costs; the balance of the plant equipment costs was generally developed based on an internal database, representing 2% of total direct equipment costs. Some equipment costs were estimated where no relevant data was available.

Detailed budgetary pricing from at least three (3) vendors for each process equipment vendor were solicited. Then a detailed technical and commercial adjudication was carried out to select the cost carried in the estimate for each process equipment. Similar methodology was adopted for the mining and electrical equipment to arrive at the preferred cost adopted in the estimate.

No allowance is included in the estimate for HVAC equipment as most of the buildings are open, where air conditioning is needed, it has been included in the architectural account.

21.1.9.8 Electrical

Electrical costs within battery limits by WBS area are based on single line diagrams, engineering lengths, size, cable types and cable tray run from equipment to the various MCCs and electrical panels as applicable.

Supplier budgetary quotations for major electrical equipment including switchgear, transformers and MCC and the power plant from at least two (2) vendors suppliers and installation of bulk materials companies were obtained and analyzed both for technical and commercial compliance. The recommended supply costs and rates were utilized in the estimate.

Fuel tank farm was provided for in the Project, but the cost was excluded from the estimate as it is envisaged it will be operated on a turnkey project. This cost is therefore be carried as an operating cost.

21.1.9.9 Instrumentation

Instrumentation within battery limits is based on equipment factor allowances.

The plant distributed control system (DCS) is based on equipment factor allowances and includes vendor support for programming and system configuration.

21.1.9.10 Piping

Process and service piping within the battery limits of the process plant is estimated on an equipment factor allowance basis. Base documents are plot plans, flowsheets and general arrangement drawings.

Tailings and reclaim water piping runs are estimated from engineering drawings and subsequently priced in detail from the pipe size and specifications provided.

Standard industry man-hours are used for estimating all piping installation. Unit costs for hydro testing and pipe degreasing are included in the unit rates and factored allowances.

21.1.9.11 Labour Costs

Labour manhours were developed from adjudicated contractors' bids as well as obtain pricing for each site scope of work. The productivity factors (PF) vary as a function of the expected qualifications, as well as of the building height and the congestion; they vary from 1.00 to 1.25. It should be noted that a PF of 1.0 refers to projects being executed with better-than-average skill, base 40-hour workweek, within reasonable commuting distance, limited in-plant movement, favorable weather, etc.

Labour rates were developed based on salary information reflecting local Moroccan labour. They are inclusive of salaries, allowances for vacation, overtime and premium work, and exclusive of tools and consumables, construction equipment, overhead and profit.

It is assumed that the local community of Zgounder can accommodate the direct and indirect workforce estimated for the Project, including occasional site visits and vendor representatives. The workforce is estimated to reach 240, with an average of 170. Local accommodation and rotational transportation costs are included as part of construction field indirect costs.

21.1.9.12 Preliminary and General Cost (P&G's)

P & G's are contractor's indirect cost but are accounted for as part of the project or owner's direct cost. These costs were provided by each contractor as part of their bid by scope of work. These costs typically include:

- Contractor's contractual requirements such as plans, procedures, security and all other expenses incurred in executing this contract.
- Contractor's mobilisation cost for all temporary works, labour, equipment and mobile equipment, facilities, supplies, materials, tools, personnel, medical, testing including cleaning services. Fuel storage if needed will be included in this account as well as vehicles, generators, portable washrooms and washing facilities.
- Contractor's demobilization of all temporary equipment and facilities
- Management supervision, field engineering, etc.

21.1.10 Direct Cost Summary – Underground Mining (1000)

Table 21.3 presents the initial capital costs for the Underground Mining area.

Table 21.3 – Capex Summary- Underground Mining

WBS	Major Area	Total Cost ($ USD)
	Underground Mobile Equipment	7,785,031
	Underground Infrastructure and Ventilation	1,928,322
	Total Underground Mining	**9,713,353**

21.1.11 Direct Cost Summary – Open Pit Mine Development (2000)

Table 21.4 depicts the initial capital costs for the Open Pit Mine Development area. The pricing is based on contractor quotations.

Table 21.4 – Capex Summary- Open Pit Mine Development

WBS	Major Area	Total Cost ($ USD)
	Contract Mining – Waste	2,677,454
	Contract Mining – Ore	194,792
	Contract Mining – Pre-Split	27,914
	Contract Mining - Mobilisation	43,011
	Total Open Pit Mine Development	**2,943,170**

21.1.12 Direct Cost Summary – Process Plant (4000)

Table 21.5 depicts the initial capital costs for the Process Plant Facilities.

Table 21.5 – Capex Summary- Process Plant Facilities

WBS	Major Area	Total Cost ($ USD)
	Process Plant Feed Circuit (plant terracing, feed circuit, crushed ore stockpile & conveyors)	7,161,536
	Grinding	11,027,183
	Flotation	8,507,694
	Leaching	14,147,884

WBS	Major Area	Total Cost ($ USD)
	Silver Recovery	10,122,485
	Reagent & Grinding Media	2,815,354
	Tailings & Ponds	710,676
	Dewatering	2,409,168
	Utilities	3,868,235
	Total Process Plant	**60,770,216**

The process plant area covers the ROM stockpile, crushing and crushed ore stockpile, the plant feed circuit, the process plant including grinding, flotation, dewatering, reagents and utilities.

The plant terracing refers to the initial cut and fill and site preparation activities prior to detailed excavation for the facilities.

21.1.13 Direct Cost Summary – Process Site – Surface Infrastructure (5000)

Table 21.6 details the summary for the Surface Infrastructure in the Process Area.

Table 21.6 – Capex Summary- Process Surface Infrastructure

WBS	Major Area	Total Cost ($ USD)
	Buildings	541,400
	Site Power Generation and Distribution	8,101,784
	Total Process Site – Surface Infrastructure	**8,643,184**

21.1.14 Direct Cost Summary – Tailings and Water Management (6000)

Table 21.7 details the summary for the Tailings and Water Management.

Table 21.7 – Capex Summary- Tailings and Water Management

WBS	Major Area	Total Cost ($ USD)
	Tailings and Water Management	5,536,670
	Total Tailings and Water Management	**5,536,670**

21.1.15 Indirect Cost

The indirect costs are those costs that specifically pertain to the design and construction of the Project, but not considered direct costs such as equipment and materials used in the construction of the facilities. The indirect costs include design, procurement and construction management activities, vendor representatives, spare parts and first fills, and contractor's indirect costs. Table 21.8 presents the costs for the indirect expenses.

Some elements of the indirect costs are further expanded in this sub-section.

Table 21.8 – Capex Summary- Indirect Costs

WBS	Major Area	Total Cost ($ USD)
	Contractor Indirect Costs	8,763,986
	Spares, Fills and Inventory	4,095,656
	Freight and Logistics	2,234,197
	EPCM Costs (External Engineering)	14,678,242
	Vendor Representatives, Start-Up, and Commissioning	1,036,800
	Total Indirect Costs	**30,808,881**

21.1.15.1 Construction Field Indirect

This component includes all temporary buildings and services required during construction and commissioning. These costs are normally estimated in the construction execution plan, but are factored as a percentage of total direct cost and include if needed, for the following:

- Offices;
- Temporary warehouses;
- Temporary construction services;
- Construction water supply;
- Sewage facilities;
- Construction communications;
- Laydown areas;
- Roads and maintenance;
- Heavy lifting cranes rentals (including mobilisation and demobilisation above 30 t);
- Materials handling management (off-loading and loading);
- Vehicles.

21.1.15.2 *Construction Support*

Temporary construction services include office janitorial and garbage services for the EPCM and Owner's project teams; QA surveying; site access control; security services; personnel physicals, safety induction and badges, safety, first aid, medical supplies and services. The costs for these services are based on the EPCM and Owner's organizational charts, project construction schedule, and experience with projects of similar size and duration.

An allowance is made for soils, concrete and piping non-destructive testing (NDT), freight and duty rates for construction management.

Allowances for personnel protective equipment (PPE) for the EPCM and Owner's project teams including hardhats, safety glasses, and safety shoes are included in the Owner's cost.

This item is calculated as a percentage of the total direct cost.

21.1.15.3 *Dust Suppression and Loss Productivity*

Dust control is a normal site condition in this region of world and in a hot climate, provision has been made in the estimate for dust suppression particularly on the roads around the process plant duration construction. In-plant dust suppression within scope areas is assumed to be included in the scope of work within that area.

21.1.15.4 *Construction Camp, Catering and Services*

As site is within short proximity to local towns, no allowance is included for construction camp and catering. It is assumed that the construction crews will have or find local accommodations. Provision to accommodate for the site EPCM and Owner's team in nearby rent villas (3) for the duration of the construction period has been provided at the current rate of $1,000 USD per month per villa paid by the Owner. Catering and accommodations services has also been computed using the current rate of $10 USD per man-day paid by the Owner.

21.1.15.5 *Health, Safety and Environmental Protection*

An allowance is provided in the estimate to provide for an HSE program.

21.1.15.6 *Construction Fuel*

Diesel fuel needed to operate the diesel generators at the EPCM staff accommodation (3# 60kW gen set) and 1 # 150 kW gen set at the construction site have been computed based on each generator's consumption rate and duration of day operation multiplied by the anticipated construction duration to arrive at the total volume of fuel required. Added to this is an allowance to provide fuel for the construction equipment. The total volume is then multiplied by the current negotiated delivered to site price for diesel. Additional cost is carried as an allowance for the storage tanks.

21.1.15.7 Spare Parts

Major spares as defined as critical spares needed for the smooth operation of the plant. This account is included in the operational expense of the plant (Opex).

21.1.15.8 First Fills

The estimated cost to supply plant first fills includes such items as ball charge, lubricants, fuels, and reagents. First fills do not include general warehouse inventory and staff. First fills are calculated based on specification and data provided by the process and mechanical engineering team.

21.1.15.9 Freight

Freight costs as provided with vendor quotations are included. The remaining freight costs are factored as a percentage of the equipment cost based on 5.0% ocean freight, and 1.5% overland to site as the project strategy is to procure the silver transportation fleet ahead of time and use these to conveying the process equipment and other materials from the port of Monrovia to the Zgounder site in Morocco. Allowances have also been included for port charges at 0.5%, forwarding fees at 0.5% and custom brokerage fees at 0.5% of the equipment cost based on experience on other projects.

It is assumed that all bulk material prices include transportation, i.e. delivered to site basis with the exception for structural steel which is priced ex-works. Freight cost for structural steel has therefore been included in the estimate based on the above assumptions. Duties are not included in this estimate.

21.1.15.10 Vendor Representatives

Costs for vendor representatives at site are developed based on information provided by vendors. Travel time to or from site of one day is included with the time required on site. Airfares, lodging, and other out-of-pocket expenses are allowed for in the rate per round trip. All-in daily rates provided by vendors with their equipment pricing are adjusted to reflect the planned construction work week circle.

Vendor representative costs are included where vendors state they must be on-site for installation to maintain equipment warranty. Vendor representative assistance have also been included during commissioning for some major equipment.

21.1.16 Owner's Costs

Owner's costs have been included in the estimate as provided by Aya and include, but are not limited to:

- Preproduction labour costs including process, maintenance and administration labour. Mining & geology labour is estimated in the mine Capex;
- Transport from town to site;
- Potable water supply during construction;
- Communication cost during construction;
- Training for personnel;
- Construction insurance;
- Legal costs for contracting;
- Land acquisition;
- Community engagement budget;
- Preliminary mining operations.

21.1.16.1 Engineering and Procurement Services

The engineering and procurement (EP) costs are costs to design all elements and processes for the scope of work defined within the project and to procure all equipment and services necessary to construct, commission and operate the new facilities. This also includes office engineering support during the construction phase.

21.1.16.2 Construction Management

The construction management (CM) estimate is based on previous DRA project experience. The execution basis of the CM estimate is that multiple contractors will work on unit price, or lump sum contracts.

The CM estimate covers the field or site-based services required for constructing and commissioning the process expansion facilities and associated infrastructure.

The CM estimate includes the following site-based services:

- Project management;
- Field engineering;
- Site document control;
- Construction management;

- Industrial relations;
- Construction supervision to general superintendent;
- Health, safety, environment and community;
- Site administration;
- Field human resources;
- Site quality assurance and control;
- Site project controls (cost control and schedule);
- Field accounting;
- Site computers and information technology services;
- Site procurement;
- Field receiving and warehousing;
- Field contract administration.

The CM costs are calculated as a percentage of the total direct cost and indirect cost excluding engineering and procurement, contingency and Owner's Cost.

Support expenses for CM staff are included in the construction indirect field costs. These expenses include offices, vehicles, communications and transportation.

21.1.16.3 Commissioning

The commissioning estimate includes trade crews to support commissioning for a period of three (3) months. The cost for commissioning assistance by the EPCM contractor, based on providing technical staff, is determined based on the execution schedule and manpower plan. This also includes an allowance for commissioning spares, which are based on vendor recommendations and (where not given) a factored allowance on each equipment. The account is based allowed for as a factor of the total mechanical equipment cost.

21.1.16.4 Third Party Engineering

This allowance is included in the Estimate to allow for if needed third party engineering for items such as additional soil test for roads, equipment foundations, etc.

21.1.17 CONTINGENCY

Contingency is a monetary provision in the estimated cost of a project to cover uncertainties or unforeseeable elements of time and cost within the project scope as estimated. Contingency is also meant to cover normal inadequacies that are inherent in design definition, execution definition, and estimating deficiencies, but it does not cover scope changes. Inadequacies are inherent in any project, due to the dynamic nature of project engineering and construction.

Contingency is developed for the estimate based on the degree of definition of scope and budgetary bids from vendors. The final project contingency in this case is 12.6%.

21.1.18 QUALIFICATIONS

All estimates are developed within a frame of reference defined by assumptions and exclusions, grouped underestimate qualifications. Assumptions and exclusions are listed in the following paragraphs.

21.1.18.1 Assumptions

The following items are assumptions concerning the Capex:

- The Capex reflects an Engineering, Procurement and Construction Management (EPCM) type execution wherein an EPCM contractor will provide the design, procurement and construction activities for all aspects of the Project. All sub-contracts would be managed by the EPCM contractor;
- The Capex is based on no delays in execution from time of contract award to the selected EPCM contractor as a result of either of the following:
  - Aya's financing charges;
  - Aya's permitting delays.
- Estimate is based on rotations schedule of 4 and 2, i.e. 4 weeks in and 2 weeks R&R, with traveling during the 2 weeks R&R;
- Estimate is based on 6 days at 8 hours per day workweek;
- Estimate assumes that labour skills will be medium;
- Estimate assumes all equipment and materials will be new;
- Estimate assumes aggregates used for fill, adequate both in terms of quality and quantity, will be available within a 5 km radius from site;
- Estimate assumes overburden disposal will be within a 5 km radius from the construction site;
- Estimate assumes fresh water, adequate both in terms of quality and quantity, is available locally at no costs and does not need any treatment to be used for concrete mix, leak/hydro testing, flushing, cleaning, etc.;
- Estimate assumes drinking water will be bottled;

- Estimate assumes EPCM and Owner's teams will be in sufficient quantity so as not to delay contractors;
- Estimate assumed smooth coordination between contractors' battery limits;
- Estimate assumes 40% of manual labour will be sourced within the Zgounder area, while 60% will be a combination of remote Moroccan workers and expats from neighbouring countries;
- Estimate assumes no labour decree is in effect in Morocco;
- Estimate assumes no camp or catering;
- Estimate assumes no limitation to site access;
- Estimate assumes construction contract types will be either lump sum, cost plus, or unit rates;
- Estimate assumes no underground obstructions of any nature;
- Estimate assumes no hazardous materials in excavated materials;
- Estimate assumes no delay in Client's decision-making;
- Estimate assumes no delay in obtaining permits and licenses of any kind;
- Estimate assumes no interruption in job continuity;
- Estimate assumes normal BFSk workforce;
- Estimate assumes engineering progress prior to the execution will be sufficient so as to avoid rework.

21.1.18.2 Exclusions

The following items are not included in the Capex:

- Currency fluctuations;
- Any and all scope change;
- Cost related to any force majeure;
- Operating cost;
- Working capital;
- Inflation beyond the Capex estimate base date;
- Expected Monetary Value (EMV) of identified risks;
- Financing and interest charges during construction;
- Changes to design criteria;
- Scope changes or accelerated schedule;
- Schedule delays exceeding two (2) weeks and associated costs;
- Delays resulting from community relation, permitting, project financing, etc.;

- Any and all taxes, customs charges, excises, etc.;
- Unidentified ground conditions;
- Extraordinary climatic events;
- Force majeure;
- Labour disputes;
- Insurance, bonding, permits and legal costs;
- Receipt of information beyond the control of EPCM contractors;
- Changes in Moroccan or Canadian law.

21.1.19 SUSTAINING CAPITAL

The sustaining costs as shown in Table 21.9 are those capital costs that must be incurred to achieve a successful mining operation but are not necessary to be incurred during the preproduction phase of the project.

For the mining operation, sustaining capital includes the addition or replacement of mining equipment, development of ramps and declines, new underground conveying systems to feed the raise, ventilation fans, piping and pumps for the dewatering systems, expansion of the underground telecommunications network and backfill reticulation. Underground mining equipment replacement is included in the mining contractor rates.

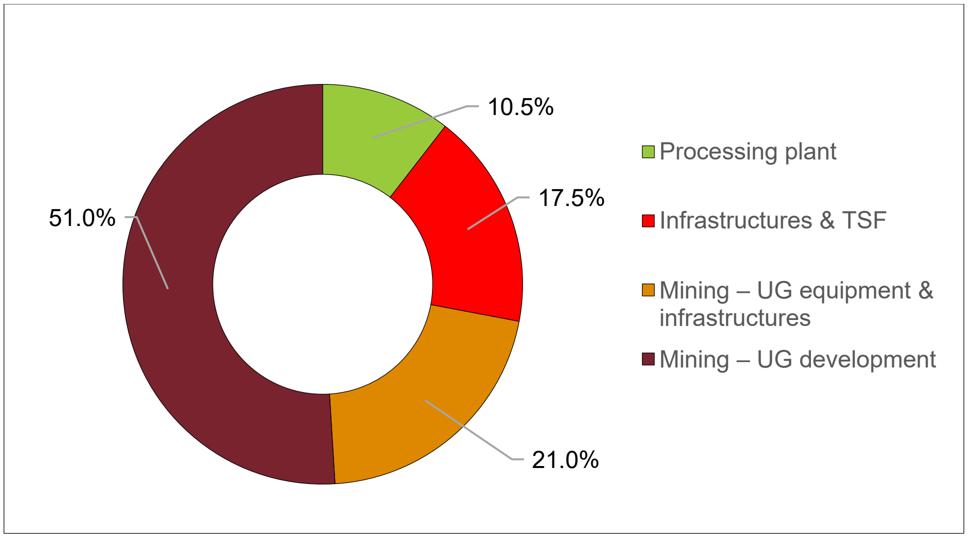
The sustaining capital requirements for the process plant includes for the purchase of spare parts for equipment, and for replacement of equipment, when required.

The tailings area covers the expansion of the TSF as the tailings storage increases in area.

Table 21.9 – Sustaining Capital Cost Estimate

Major Area	Total Cost ($ USD)
Processing plant	7,315,580
Infrastructure & TSF	12,177,070
Mining – UG equipment & infrastructure	14,663,444
Mining – UG development	35,523,599
Total Sustaining Capital	**69,679,693**

Figure 21.2 – Sustaining Capital Cost by Major Area



Source: DRA, 2022

21.1.20 CLOSURE COST

The provisional cost estimate for the rehabilitation and closure of the mine, based on the rehabilitation and closure plan framework is provided in Table 21.10. A detailed closure plan is to be developed during detail design. The closure liability will be funded through an appropriate instrument and the actual closure liability updated annually by comparing planned rehabilitation against actual rehabilitation.

Table 21.10 – Summary Estimate of Financial Provision Required for Closure

Description	Total ($ USD)
General Cleaning	1,000,000
Soil Decontamination	100,000
Underground Infrastructures	156,695
Processing Facilities Area	1,611,878
Warehouse Area	71,631
Waste Storage Facility (Waste Rock + TSF)	2,064,000
Post-Closure Monitoring	75,000
Sub-Total:	*5,079,204*
Contingency @a 20%	1,015,841
Administration Fees @ 10%	507,920
Total (excluding taxes):	**6,602,965**

21.2 Operating Cost Estimate

21.2.1 Basis of the Estimate

This Section describes the basis of estimate and approach taken in calculating the operating costs for the Project. The operating costs cover the costs related to underground mining and open pit mining, ore processing, and General and Administration ("G&A") costs

The Opex is presented in United States Dollars (USD) and uses prices obtained in Q4 2021. DRA developed these operating costs in conjunction with Aya, with specific inputs provided by external consultants for tailings disposal costs and concentrate transportation.

The following are examples of cost items specifically excluded from the Opex:

- Value Added Tax (VAT);
- Project financing and interest charges.

Table 21.11 presents the operating costs summary by major project area over the LOM.

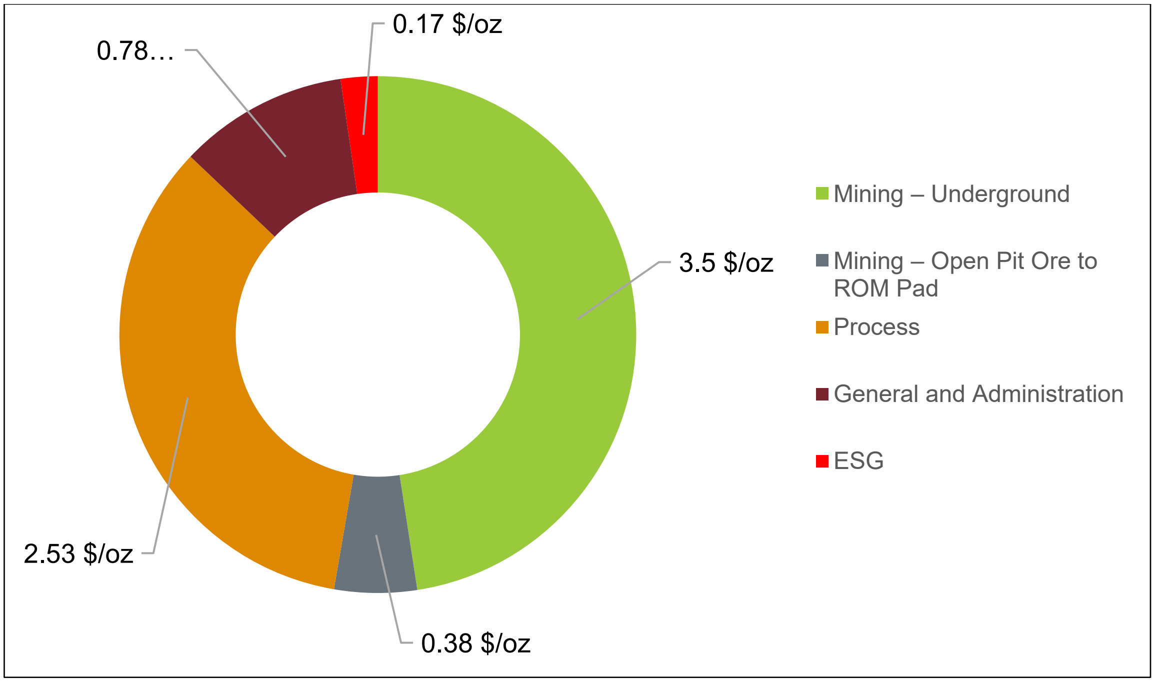
The average operating cost, including transport is $55.42/t of ore.

Table 21.11 – Operating Costs Summary

Description	LOM Cost	Cost ($/t)[1]	Cost ($/oz)	Total Cost (%)
Mining – Underground	226,634,161	26.38	3.50	47.6
Mining – Open Pit Ore and Historical Tailings to ROM Pad	24,777,360	2.88	0.38	5.2
Process (average)	163,450,368	19.03	2.53	34.4
General and Administration	50,264,337	5.85	0.78	10.6
ESG	10,972,885	1.28	0.17	2.3
Total	**476,099,911**	**55.42**	**7.36**	**100.0**

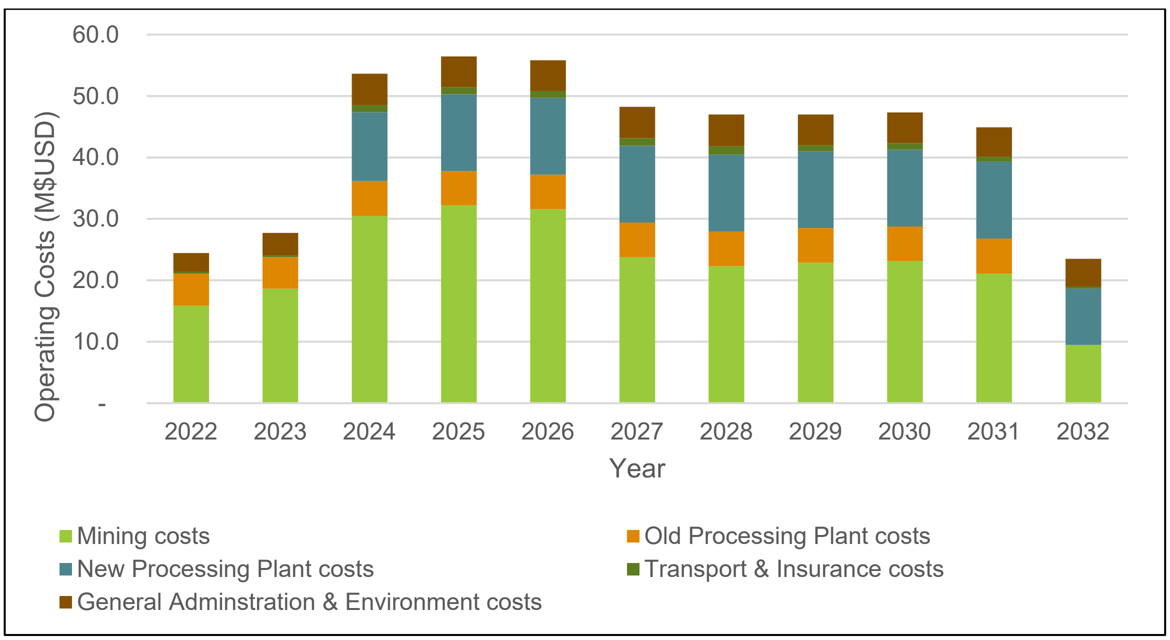
1. Figures may not add due to rounding.

Figure 21.3 – Operating Cost Summary by Area



Source: DRA, 2022

Figure 21.4 –Zgounder Project LOM Operating Costs



Source: DRA, 2022

21.2.2 OPEN PIT MINING OPERATING COSTS

The open pit mining operating costs breakdown is summarised in Table 21.12.

The open pit mine operating cost was estimated average $ 1.09/t mined for the life of the open pit, an average $ 9.92/t of ore and is based on local contractor quotes. The cost is based on equipment operation costs, mine-related manpower, explosives costs as well as the costs associated with dewatering, road maintenance and other activities. Open Pit Costs also include the mining of the historical tailings.

Table 21.12 – Open Pit Opex

Description	Total Cost	$/t Mined	$/t Open Pit Ore
Contract Mining Waste	17,119,635	0.75	6.86
Contract Mining Ore	6,860,970	0.30	2.75
Contract Pre-Split	440,513	0.02	0.18
Ore Rehandling to Crusher	356,242	0.02	0.14
Total	**24,777,360**	**1.09**	**9.92**

1. Figures may not add due to rounding.

21.2.3 UNDERGROUND OPERATING COSTS

The underground mine operating cost was estimated average $ 37.19/t mined for the life for combined Long-hole and Drift-and-Fill mining methods, ore and waste level development, backfill, surface hauling, ore rehandling and Owner's indirects, The underground mining operating costs breakdown is summarised in Table 21.13 and is based on owner's operated mining fleet and personnel.

Table 21.13 – Underground Mining Opex Summary

Description	LOM Cost[1]	Ave Annual Cost[2]	$/t Underground Ore
Waste Development	65,600,133	5,963,648	10.77
Ore Development	44,156,194	4,014,199	7.25
Long-hole Longitudinal Stoping	5,206,718	473,338	0.85
Long-hole Transverse Stoping	12,048,643	1,095,331	1.98
Drift-and-Fill Stoping	24,056,820	2,186,983	3.95
Backfill	31,005,191	2,818,654	5.09

Description	LOM Cost[1]	Ave Annual Cost[2]	$/t Underground Ore
Surface Haulage	12,186,851	1,107,896	2.00
Ore Rehandling	53,763	4,888	0.01
Owner's Indirect Costs	32,319,848	2,938,168	5.30
Total Mining Opex	**226,634,161**	**20,603,106**	**37.19**

1. Figures may not add due to rounding
2. Total underground mine production tonnes LOM: 6,093,425 t

21.2.4 New Processing Plant Operating Costs

The new processing plant is based on the plant design nameplate feed throughput capacity of 730,000 t/y, the estimated process operating costs for the new plant are divided into five (5) main components: Manpower, electrical power, grinding media and reagent consumption, consumables and wear items, and spare parts and miscellaneous.

The cost breakdown for the new plant is summarised in Table 21.14.

Table 21.14 – Summary of Estimated Annual New Process Plant Opex

Operating Cost	LOM	Cost ($/a)	Cost ($/t)[1]	Total Costs (%)
Manpower	33,714,327	3,618,990	4.96	30.0
Electrical Power	15,335,983	1,645,285	2.25	13.6
Grinding Media and Reagent Consumption	51,194,557	5,499,394	7.53	45.6
Consumables and Wear Items	7,450,680	800,000	1.10	6.6
Spare parts and miscellaneous[3]	4,656,675	500,000	0.68	4.1
Total Operating Costs	**112,352,222**	**12,063,668**	**16.53**	**100.0**

1. Based on mill feed throughput of 730,000 t/y and total of LOM of 6,208,900 tonnes through the new plant
2. Figures may not add due to rounding.
3. Spare parts, estimated as 2.5% of total equipment capital cost and 15% for transportation cost from client (7% sea transport and 8% land transport).

21.2.4.1 Manpower Costs – New Plant

It is estimated that there will be 112 employees. This includes:

- Five (5) plant administrative staff;
- Fifty (50) personnel dedicated to mill operations;
- Thirty-Six (36) personnel dedicated to mill maintenance;

- Twenty-One (21) metallurgy personnel.

Table 21.15 presents the manpower for the process facility.

Table 21.15 – Concentrator Plant Manpower Opex

Description	Number	Cost ($/year)
Administration	5	497,700
Operations	50	1,194,480
Maintenance	36	985,920
Metallurgy	21	940,890
Total	**112**	**3,618,990**

The total annual cost for manpower is estimated at $ 3.6 M per year.

21.2.4.2 Electrical Power Costs – New Plant

Electrical power is required to operate equipment in the processing plant such as conveyors, crushers, mills, screens, pumps, agitators, bagging system, services (compressed air and water), etc. The unit cost of on-site generated electricity was established at $ 0.0508/kWh. The total annual cost for process plant electrical power is estimated between $1.6 M per year.

The electrical power consumption was derived from the mechanical equipment list and from equipment suppliers power requirements. The estimated electrical operating costs is based on the plant operating 24 hours per day, 7 days per week, with a run time of 91.3% operating percentage, for a feed throughput of 730,000 t/y.

21.2.4.3 Grinding Media and Reagent Consumption Costs – New Plant

Grinding mills will need addition of steel balls to replace the worn balls to maintain the steel load in the mills and to perform proper size reduction on the material. Also, the drum scrubber and the polishing mills will require addition of ceramic media to replace the worn media. Consumption of the steel balls and ceramic media is based on abrasion index, power consumption and experience.

Diesel as silver collector and MIBC as frother are the reagents required throughout the various stages of flotation. Flocculant is required for both thickeners' operation. The quantities were determined based on the test work.

The total annual cost for grinding media and reagent consumption is estimated at $ 5.5 M per year. Table 21.16 shows the costs related to grinding media and to plant reagents and, when processing fresh rocks in the feed blend.

Table 21.16 – Concentrator Grinding Media and Reagents Costs

Description	Cost ($/year)[1]
Grinding Media	496,724
Plant Reagents	5,002,670
Total	**5,499,394**

1. Based on feed throughput of 730,000 t/y.

21.2.4.4 *Consumables and Wear Items Costs – New Plant*

The consumption and costs for the mineral sizers wear parts, grinding mill liners, polishing mill liners, flotation cell wear parts, screen deck panels, pump wear parts, filter cloths, dryer wear parts, etc. for different equipment was obtained from the equipment suppliers and from experience with similar operations. The total annual cost for consumables and wear items is at $ 0.8 M per year.

21.2.4.5 *Spare Parts and Miscellaneous Costs – New Plant*

The strategic spare parts and miscellaneous costs were estimated as 2.5% of the total equipment capital costs and 15 for transportation cost from client (7% sea transport and 8% land transport). The total annual cost is estimated at $ 0.5 M per year.

21.2.5 OLD PROCESSING PLANT COSTS

The operating cost breakdown for the old plant is summarised in Table 21.17.

Table 21.17 – Summary of Estimated Annual Old Process Plant Opex

Operating Cost	LOM	Cost ($/y)	Cost ($/t)[1]	Total Costs (%)
Cyanidation Plant	16,056,803	1,459,709	32.00	28.9
Flotation Plant	39,482,884	3,589,353	21.00	71.1
Total Operating Costs	**55,539,687**	**5,049,062**	**23.32**	**100.0**

1. Based on mill feed throughput of 218,000 tonnes per year and LOM tonnage of 502,000 tonnes to the old cyanidation plant and 1,880,000 tonnes through the old flotation plant.
2. Figures may not add due to rounding.

21.2.6 GENERAL AND ADMINISTRATION COSTS

The General and Administration (G&A) costs include the following categories:

- Manpower;
- Subcontracting;

- Leasing;
- Supplies and Consumables
- Parts and Equipment
- Administrative costs;
- Transportation & Insurances.

The overall G&A annual costs are estimated at $ 4.6M/year or $5.85/t of feed to the plant. Given the nature of G&A costs, plant operations and throughput have little to no impact on these costs. As a result, G&A was assumed to be constant over the LOM except for the first two years before the commissioning of the new plant. Table 21.18 summarises the G&A breakdown.

Table 21.18 – G&A Costs

Description	LOM Costs ($)	2022 Costs ($) [1]	2023 Costs ($) [1]	2024+ Costs ($) [1]	Unit Cost ($/t)	Unit Cost ($/oz)
Manpower	19,427,701	1,668,902	1,835,792	1,769,223	2.26	0.30
Subcontracting	12,176,916	760,277	1,140,415	1,141,802	1.42	0.19
Leasing	3,171,546	176,181	193,799	311,285	0.37	0.05
Supplies and Consumables	1,513,257	84,062	92,468	148,525	0.18	0.02
Parts and Equipment	680,330	37,594	44,920	66,424	0.08	0.01
Administrative costs	3,585,260	199,162	219,079	351,891	0.42	0.06
Transportation & Insurance	9,709,328	244,141	258,701	1,022,943	1.13	0.15
Total	**50,264,338**	**3,170,319**	**3,785,174**	**4,812,093**	**5.85**	**0.78**

1. Excludes last year
2. Figures may not add due to rounding

22 ECONOMIC ANALYSIS

The economic analysis performed for this report was based on an annual cash flow model over the life of the project. The cash flow projections were estimated using the project development and production schedule and incorporate sales revenue, CAPEX, OPEX, and other costs. CAPEX estimates include initial capital expenditures, sustaining capital expenditures and closure costs. OPEX estimates include mining costs, processing costs and general and administration costs. Other expenses include royalties, transport and insurance, and taxes and depreciation.

The economic analysis results present net present value ("NPV") and internal rate of return ("IRR") on a pre-tax and post-tax basis. Payback period and total undiscounted cash flow are also presented. A sensitivity analysis was performed on key parameters.

22.1 Assumptions

Key assumptions used to build the project cash flow model include:

- Metal price of silver on a US$/oz basis;
- Light fuel oil price on a US$/l basis;
- Exchange rates for both the Canadian dollar and Moroccan dirham relative to the US dollar.

22.1.1 Metal Price

The silver price was set at $22.00 per ounce based on current market conditions and was kept constant over the life of mine.

22.1.2 Fuel Price

The cost of light fuel oil was set at $0.85 per liter based on current market conditions and was kept constant over the life of mine.

22.1.3 Exchange Rates

Exchange rates were used to convert all different costs to US dollars and were set based on current market conditions as follows:

- 0.759 United States dollars to the Canadian dollar;
- 0.108 United States dollars to the Moroccan dirham.

22.2 Production Schedule

The mine plan is based on the mineral reserves for the project with a total of 8.6Mt of ore processed at an average grade 257 g/t of silver. The average mill recovery over the life of mine is 91.3% for a total production of 64.7M ounces of silver. Figure 22.1 and Figure 22.2 highlight the processing and production schedule for the project. A detailed yearly production schedule is presented in Table 22.1 highlighting the key physicals used to build the cash flow model.

Figure 22.1 – Processing Schedule

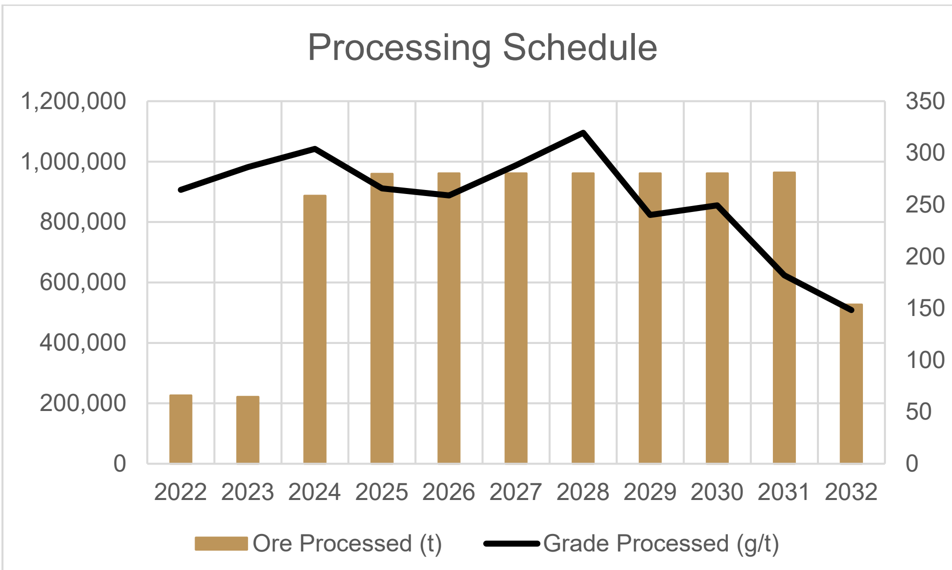
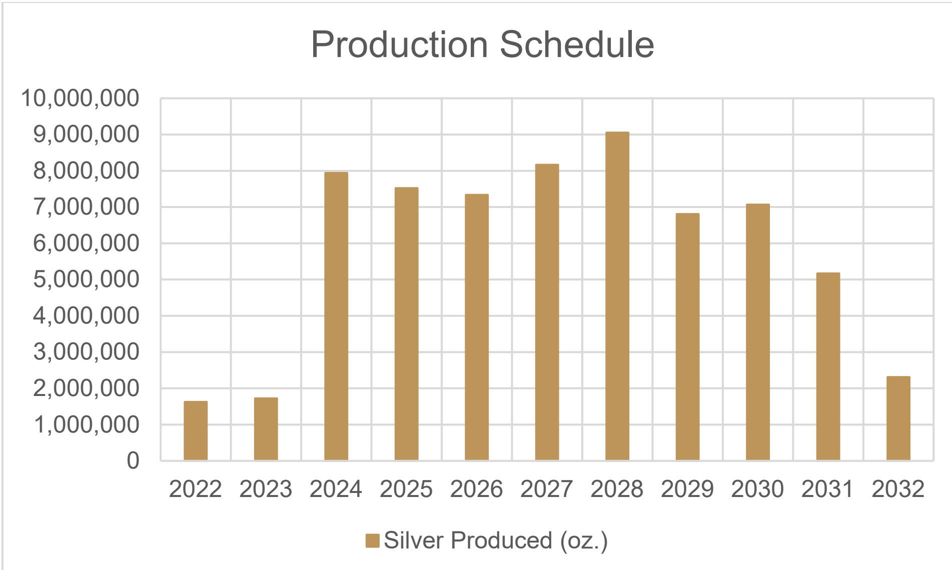


Figure 22.2 – Production Schedule



22.3 Revenue

Revenue was calculated using silver pricing and exchange rate assumptions with 99.7% of produced silver as payable. A refining charge equal to $0.20 per ounce produced diminished revenue.

Table 22.1 – Production Schedule

Production Schedule	Units	Total	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032
Mining													
Open Pit Waste Mined	(kt)	23,448	0	3,171	4,320	3,940	3,881	3,296	2,510	1,393	937	0	0
Open Pit Ore Mined	(kt)	2,497	0	71	270	387	330	317	297	268	241	288	31
Underground Ore Mined	(kt)	6,093	226	251	595	672	673	673	673	673	673	746	238
Average Grade Mined	(g/t)	256	264	268	304	242	245	283	316	244	259	185	189
Ore Rehandled	(kt)	763	0	0	91	95	102	37	39	54	85	0	259
Processing													
Ore Processed	(kt)	8,591	226	221	887	960	961	961	961	961	961	964	527
Grade Processed	(g/t)	257	264	286	304	266	259	288	319	240	249	182	148
Recovery	%	91.3%	84.6%	84.6%	91.7%	91.7%	91.7%	91.7%	91.7%	91.7%	91.7%	91.7%	92.0%
Production													
Silver Produced	(koz.)	64,729	1,628	1,725	7,943	7,522	7,338	8,166	9,053	6,806	7,067	5,170	2,312

22.4 Capital Expenditures

Capital expenditures for the project include initial capital expenditures incurred before the completion of the expansion and sustaining capital expenditures incurred thereafter. Working capital requirements for the expansion project are estimated as being minimal considering that the mine is currently in operation.

22.4.1 Initial Capital Expenditures

Initial capital expenditures are incurred over a 24-month construction period, including commissioning. Total initial costs to expand the Zgounder plant is estimated at $139.4M, which includes equipment and infrastructures, open pit pre-stripping, indirect costs, owner costs and contingency, as summarised in Table 22.2. The total initial capital expenditures include a contingency of $16.6M which is 13.5% of the total CAPEX before contingency.

22.4.2 Sustaining Capital Expenditures and Closure Costs

Total sustaining costs are estimated at $69.7M and includes costs infrastructure, equipment and underground mine development. Underground mine development costs are scheduled to match the expected spend profile in the underground mine plan. Closure costs are estimated at $6.6M including the offset of equipment salvage value. Sustaining capital expenditures and closure costs breakdown is presented in Table 22.2.

Table 22.2 – Capital Expenditure

Capital Expenditure	Initial ($M)	Sustaining ($M)
Processing Plant	60.8	7.3
Infrastructures & TSF	6.6	12.2
New Power Line	7.6	-
Mining – Open Pit Pre-Stripping	2.9	-
Mining – UG Equipment & Infrastructures	8.8	14.7
Mining – UG Development	-	35.5
Indirect Costs	30.8	-
Indirect Contractors	8.8	-
Initial Spares & First Fills	4.1	-
Transport & Freight	2.2	-
EPCM & Commissioning	15.7	-
Direct & Indirect Cost Subtotal	*117.5*	-

Capital Expenditure	Initial ($M)	Sustaining ($M)
Owner Costs	5.4	-
Contingency	16.6	-
Total	**139.4**	**69.7**
Closure Costs	**-**	**6.6**

22.5 Operating Costs

Operating costs are comprised of mining costs, processing costs and general and administration costs, which average $54.29 per tonne milled over the life of mine. A summary of unit operating costs estimate per tonne milled is presented in Table 22.3.

Table 22.3 – Operating Costs

Operating Costs	Unit Cost ($/t milled)
Mining	29.27
Processing	19.03
G&A & ESG	6.00
Total	**54.29**

22.6 Other Expenses

Other expenses include royalties, transport and insurance, taxes and depreciation.

22.6.1 Royalties

A 3% net smelter return royalty is applicable on the metal sales from Zgounder and payable to the ONYHM. This royalty represents a cost of $42.2M over the life of mine. In addition, a 2.75% technical assistance fee payable to Aya Gold and Silver is applied to gross revenue and has the impact of reducing income taxes.

22.6.2 Transport and Insurance

Silver transport and insurance costs are equal to $0.15 per ounce of silver produced.

22.6.3 Taxes and Depreciation

A $3.00 per tonne Moroccan mining tax is applied on processed tonnage. Initial capital expenditures are depreciated over a 10-year period and sustaining capital expenditures are depreciated over a 5-year period. The resulting taxable income is taxed at a rate of 20% by the Kingdom of Morocco. Total mining tax and total income taxes over the life of mine equals $25.8M and $125.0M respectively.

22.7 Annual Cash Flow Forecast

Table 22.4 and Figure 22.3 present the cash flow on an annual basis for the project.

Figure 22.3 – Cash Flow Summary

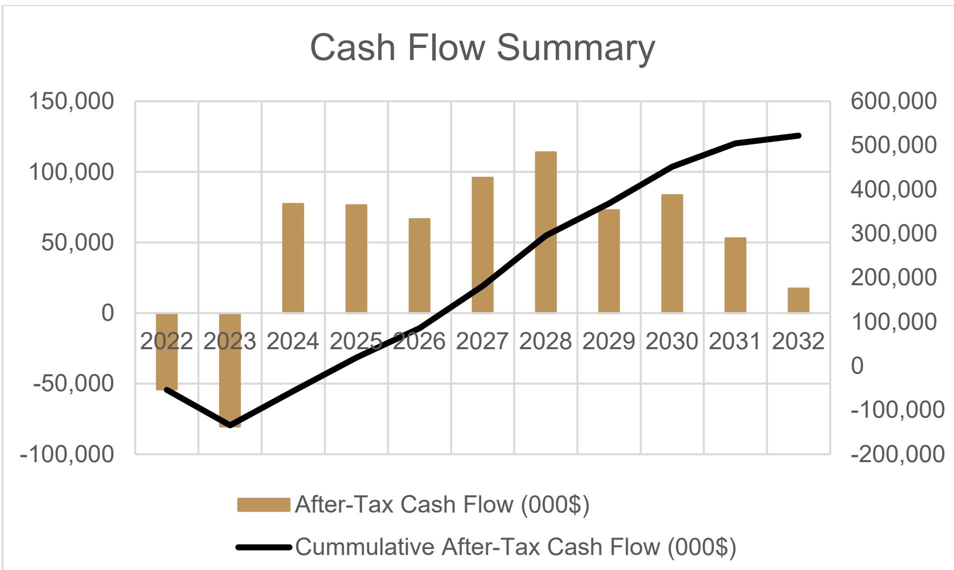


Table 22.4 – Cash Flow Summary

Cash Flow Summary (000$)	Total	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032
Revenue	**1,406,856**	**35,375**	**37,485**	**172,628**	**163,483**	**159,488**	**177,487**	**196,760**	**147,935**	**153,594**	**112,368**	**50,252**
Mining Cost	251,412	15,862	18,634	30,472	32,201	31,574	23,785	22,307	22,867	23,126	21,098	9,486
Processing Cost	163,450	5,250	5,136	16,882	18,091	18,140	18,140	18,141	18,141	18,142	18,229	9,158
G&A & ESG Costs	51,528	3,089	3,699	5,083	5,041	5,023	5,106	5,194	4,970	4,996	4,806	4,520
Silver transport & Insurance	9,709	244	259	1,191	1,128	1,101	1,225	1,358	1,021	1,060	776	347
Total OPEX	**476,099**	**24,445**	**27,728**	**53,628**	**56,462**	**55,837**	**48,255**	**47,000**	**47,000**	**47,323**	**44,909**	**23,511**
Initial Capital Expenditures	139,424	55,316	79,105	5,003	0	0	0	0	0	0	0	0
Sustaining Capital Expenditures	76,283	7,673	8,908	11,262	7,382	15,137	5,930	3,933	7,176	910	911	7,061
Total CAPEX	**215,706**	**62,989**	**88,014**	**16,265**	**7,382**	**15,137**	**5,930**	**3,933**	**7,176**	**910**	**911**	**7,061**
ONHYM Royalties (on sales)	42,206	1,061	1,125	5,179	4,904	4,785	5,325	5,903	4,438	4,608	3,371	1,508
Mining Taxes	25,772	679	966	2,593	3,175	3,011	2,969	2,909	2,822	2,741	3,102	805
Income Taxes	125,005	229	0	17,698	15,183	14,258	19,178	23,155	13,613	14,570	7,121	0
Total Other Expenses	**192,983**	**1,969**	**2,091**	**25,470**	**23,262**	**22,053**	**27,472**	**31,967**	**20,873**	**21,919**	**13,594**	**2,313**
After-Tax Cash Flow	**522,068**	**-54,027**	**-80,348**	**77,265**	**76,377**	**66,461**	**95,830**	**113,859**	**72,886**	**83,442**	**52,954**	**17,368**

22.8 Results

The economic analysis demonstrates that the project has positive economics under the assumptions used. On a before tax basis, the project has a 5% NPV of $471M and an IRR of 57%. On an after-tax basis, the project has a 5% NPV of $373M and an IRR of 48%. Total undiscounted cash flow over the life of mine equals $522.0M and payback period is estimated at 1.7 years post expansion.

The project also demonstrates a favorable cost structure with an all-in sustaining cost of $9.58 per ounce of silver produced as shown in Table 22.5.

Table 22.5 – AISC

AISC	Unit Cost ($/oz.)
General & Administration	0.78
CSR & Environment	0.17
UG Mine Opex	3.50
OP Mine OPEX	0.38
Process OPEX	2.53
Sustaining Capital	1.18
Royalties & Mining Taxes	1.05
Total	**9.58**

22.9 Sensitivity Analysis

A sensitivity analysis was performed by variating certain parameters to validate the impact on after-tax NPV and IRR as well as payback period and total undiscounted cash flow. Capital costs and operating costs were raised and lowered by 25% in 5% increments. The silver price was variated between $16.00 and $36.00 per ounce representing a variation of -27% to +64% compared to base case assumptions. The sensitivity results are presented in Table 22.6 and spider diagrams representing the impacts of the variated parameters on after-tax NPV and IRR are illustrated in Figures 22.4 and 22.5.

Table 22.6 – Sensitivity Results

Sensitivity Results											
Silver Price ($)	**16.00**	**19.00**	**20.00**	**22.00**	**23.50**	**26.00**	**28.00**	**30.00**	**32.00**	**34.00**	**36.00**
Silver Price Variation	**-27%**	**-14%**	**-9%**	**Base Case**	**+7%**	**+18%**	**+27%**	**+36%**	**+45%**	**+55%**	**+64%**
After-Tax 5% NPV ($M)	132.0	252.8	293.0	373.3	433.2	532.8	612.2	691.1	769.9	848.6	927.4
After-Tax IRR (%)	21.1%	34.4%	38.8%	47.5%	54.1%	65.5%	74.9%	84.7%	94.9%	105.8%	117.4%
Undiscounted Cash Flow ($M)	213.3	367.8	419.3	522.1	598.9	726.4	827.9	928.9	1,029.8	1,130.7	1,231.5
Payback Period (years)	3.7	2.5	2.2	1.7	1.5	1.2	1.0	0.9	0.8	0.7	0.6
Capital Costs Variation	**-25%**	**-20%**	**-15%**	**-10%**	**-5%**	**Base Case**	**+5%**	**+10%**	**+15%**	**+20%**	**+25%**
After-Tax 5% NPV ($M)	416.3	407.8	399.2	390.6	381.9	373.3	364.7	356.1	347.4	338.8	330.1
After-Tax IRR (%)	65.0%	60.7%	56.9%	53.5%	50.4%	47.5%	45.0%	42.6%	40.4%	38.4%	36.5%
Undiscounted Cash Flow ($M)	567.3	558.4	549.4	540.3	531.2	522.1	513.0	503.9	494.8	485.6	476.5
Payback Period (years)	1.2	1.3	1.4	1.5	1.6	1.7	1.9	2.0	2.1	2.2	2.3
Operating Costs Variation	**-25%**	**-20%**	**-15%**	**-10%**	**-5%**	**Base Case**	**+5%**	**+10%**	**+15%**	**+20%**	**+25%**
After-Tax 5% NPV ($M)	450.5	435.1	419.6	404.2	388.8	373.3	357.8	342.2	326.5	310.8	295.1
After-Tax IRR (%)	58.0%	55.8%	53.7%	51.6%	49.5%	47.5%	45.6%	43.6%	41.6%	39.7%	37.8%
Undiscounted Cash Flow ($M)	619.8	600.3	580.7	561.2	541.6	522.1	502.5	482.7	462.9	443.1	423.3
Payback Period (years)	1.4	1.5	1.5	1.6	1.7	1.7	1.8	1.9	2.0	2.1	2.2

Figure 22.4 – Sensitivity - After-Tax 5% NPV

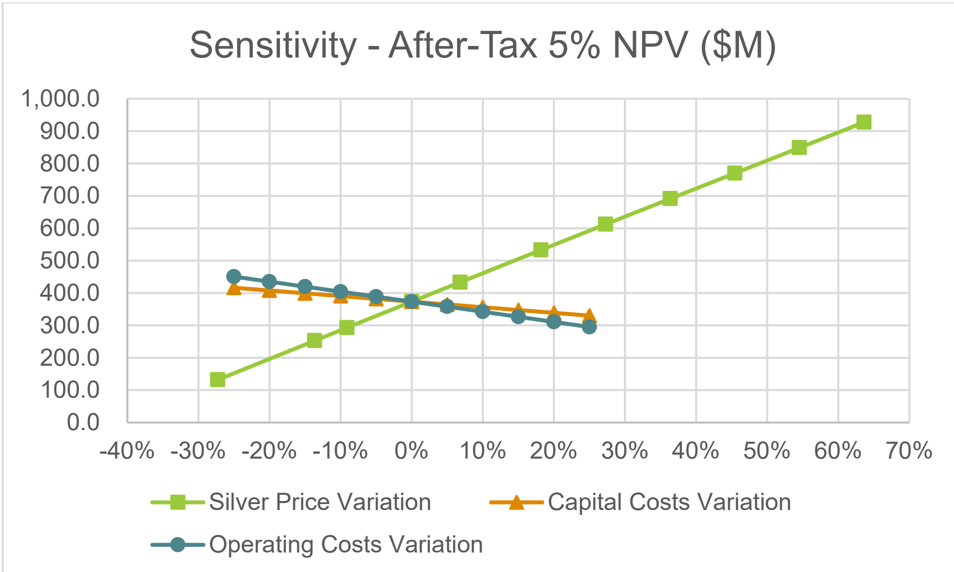
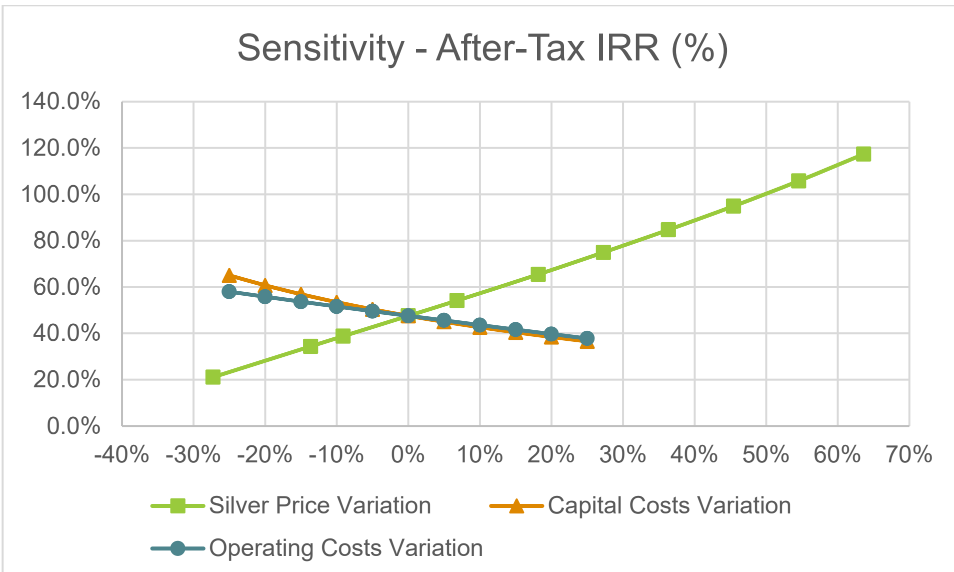


Figure 22.5 – Sensitivity - After-Tax IRR



23 ADJACENT PROPERTIES

There are no adjacent properties contiguous with the Zgounder Property and the author of this Technical Report section is not aware of any significant exploration activities by other companies. The closest mines are Hajjar (Zn, Pb, Cu and Ag) to the north and Imini (Mn) to the east, which are each approximately 50 km from the Zgounder Silver Mine.

Altus Strategies Plc. recently acquired the Izouga copper-silver project, approximately 20 km southeast of Zgounder Property. Izouga covers an area of 24 km^2.

24 OTHER RELEVANT INFORMATION

This section of the Report covers other relevant information regarding the Zgounder Expansion Project.

24.1 Project Status

This should section be filled in by Aya to discuss current status of the project expansion project, namely the Feed study and any purchased long lead items or awarded contracts relevant to the project

This Report refers to the Feasibility Study ("FS") of the Zgounder expansion project. This FS was started in the first quarter of 2021, and will be completed by the first quarter of 2022.

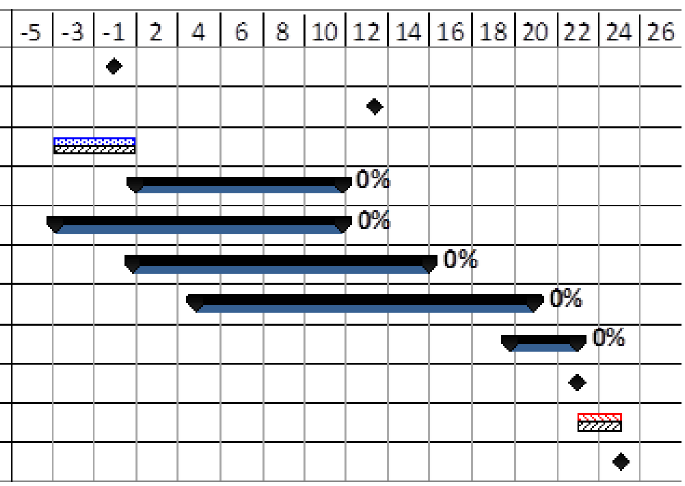
In order to fast-track the development of the project, Aya decided to launch the Front End Engineering Design ("FEED") phase of the project in December 2021. The goal was to be in a position to be able to order some of the longest lead mechanical equipment of new process plant shortly after the completion of the FS.

24.2 Project Execution Schedule

A milestone schedule has been developed for the Zgounder Expansion Project covering the main activities of studies, permitting, engineering, procurement, construction, commissioning and ramp-up. The Level-one schedule in presented in Figure 24.1.

Figure 24.1 – Project Schedule

Zgounder Expansion Project	Start	Finish
Financing In Place	M00	M00
Mills Delivery On Site	M+12	M+12
FEED Stage	M-03	M+01
Detail Design & Drafting	M+01	M+10
Procurement	M-03	M+10
Fabrication and Deliveries	M+01	M+14
Construction	M+04	M+19
Commissioning	M+19	M+21
Start Hot Commissioning	M+21	M+21
Production ramp up	M+22	M+23
Start Commercial Production	M+24	M+24



24.2.1 Schedule Assumptions

The Project milestone schedule has been developed based on the following assumptions:

- Project assumes EPCM / EPC construction strategy
- Geotechnical studies and survey reports (in their final version) are received by the EPCM / EPC contractor (s) before the start of basic engineering;
- Hydrogeological surveys and reports (in their final version) are received by the EPCM / EPC contractor (s) before the start of basic engineering and are favourable to the Project;
- All permits required will be awarded before the beginning of construction;
- Design criteria, process flowsheet and scope of work will be frozen and agreed upon by all stakeholders before the start of basic engineering;
- Qualified resources will be available for the EPCM / EPC contractor (s);
- Qualified construction workers will be available at the time of construction;

24.2.2 Key Project Milestones

Table 24.1 – Key Project Milestones

Milestone	Months After Project Approval
Financing in Place	0
Project Execution Kick-Off	1
Long Lead Equipment Packages Awarded	2
Start Process Plant Construction	4
Start Earthworks and Main Access Road Construction	4
Start Concrete Construction	7
60KVA Construction Power Available	18
Detailed Engineering Complete	11
Mill Delivery on Site	13
Construction Completion	20
Commence Hot Commissioning	22
Start Commercial Production	24

24.2.3 CRITICAL PATH

The critical path for the process plant is the procurement, manufacture and delivery of the Regrind Mill, followed by installation and commissioning of the comminution circuit.

24.2.4 SUB-CRITICAL PATH

Sub-critical path is running through the Civil and Concrete construction of the new Process Plant. Due to the construction sequence, the Structural, Mechanical, Piping, Electrical and Controls in the Flotation is the last area to be constructed.

24.3 Risk Review

A risk review meeting was held in March 2022 between DRA and Aya personnel as part of the Feasibility Study. The risks covered geology, mining, mineral processing, tailings, environmental, social and permitting project Capex, Opex, construction, and general risks.

A total of 58 risks were identified by the group. Of these, five (5) were resolved during the meeting or judged as obsolete, leaving 53 active risks. From this list, four (4) were classified as High risk, 24 were classified as medium risk, and 25 were classified as low risk in the pre-mitigation rating. Post mitigation, 53 out of 54 risks were downgraded to low risk, and one remained as a medium risk. No risks remained at a high rating after mitigation.

The High risks (pre-mitigation) are associated with silver price fluctuation affecting revenue, inflation affecting operation costs, unplanned mine dilution affecting grade, and construction delays with the new 60 KVA power line. All these risks reduce to low rating by implementing appropriate mitigation measures, It is recommended that Aya continues to follow these risks during the construction and operations phases.

In order to continue to mitigate project risks, it is recommended that sufficient risk management effort be included in the next phase of the Project (EPCM). Specifically, it is recommended that (a) a second risk review be held at the onset of the next phase to continue to identify and detail any special scope required early-on, and (b) particular emphasis be placed on conducting a full HAZOP review as per standard engineering practices.

The 53 risks identified during the risk review meeting are presented in Table 24.2. showing pre-mitigation and post mitigation levels.

Table 24.2 – Identified Risks

Ref	Risk Description	Pre mitigation / Current Rating						Risk Mitigation Actions	Post Mitigation Rating					
		Probability	Impact	Prob score	Imp score	risk score	Risk Rating		Probability	Impact	Prob score	Imp score	risk score	Risk Rating
General														
G1	**Silver (Ag) Price Fluctuation** Price of Silver reduces by 25%	M	H	2	3	6	H	Mine higher grade stopes while waiting for price to recover	L	M	1	2	2	L
G2	**Foreign Exchange and Inflation** High inflation in Morocco and/or rapid change in Forex (USD:MAD or USD:else)	M	M	2	2	4	M	Edge part of the local currency for daily operation and construction	M	L	2	1	2	L
G3	**Country Risk**	L	L	1	1	1	L	Morocco is a Mining friendly country and Aya already in operation for anumber of years	L	L	1	1	1	L
G4	**Legislative Risk**	L	M	1	2	2	L	Zgounder has the required permit. No evidence that non-compliance issues have been specified by authorities.	L	M	1	2	2	L
G5	**Supply Chain** delay in delivery of critical piece of equipment	M	M	2	2	4	M	Proceed with purchase of long lead item w/ increased built-in delay of 2 months	L	M	1	2	2	L
G6	**Health & Safety risk during construction and operation phases** New covid like virus apparance during the construction phase	L	M	1	2	2	L	Covid related requirements for contractor in contracts	L	L	1	1	1	L
G7	**Training / Operation risk (Process & Mining)**	L	L	1	1	1	L	Process : Already 2 process plants running at the site with similar technologies Mining: Proper training & continuous education Build in-house training program Bring in outside labour at start of mine	L	L	1	1	1	L
G8	**Water Supply risk** not enough water stored on site to start the commissionning of the new plant	L	H	1	3	3	M	consider construction of Water supply dam start discussion with local private land owners where Water Spring have been identified Increase mine dewatering capacities	L	M	1	2	2	L
G9	**Budget risk Closure**	L	L	1	1	1	L	Good planning and adjust to changing regulations	L	L	1	1	1	L
G10	**Budget risk OPEX** Cost increase from OP contractors in the future	M	M	2	2	4	M	Many contractors in Morocco with similar equipment	L	L	1	1	1	L
G11	**Budget risk OPEX** Inflation	H	M	3	2	6	H	Review mine plan to temporarily mine higher grade ore	L	L	1	1	1	L
G12	**Budget risk OPEX**	M	M	2	2	4	M	Cyanide consumption is a major risk on OPEX. Implementation of an oxygen plant to maintain high level of Dissolve Oxygen will reduce the cyanide consumption and risk on OPEX	L	M	1	2	2	L

Resources, Reserves & Mining Risks															
M1	**Resources and Reserves Calculations** Resources are lower (tonnage and/or Grade)	M	M	2	2	4	M	Continue infill drilling program, and increase measured resource portion, Deposit is open at depth and laterally	L	L	1	1	1	L
M2	**Mining Dilution and ore revovery** Mining dlution increases compared to FS	M	M	2	2	4	M	Improve grade-control, do definition drilling to better define stope limits, increase ground support, improve mining method	L	M	1	2	2	L
M3	**Mining Dilution and ore revovery** Lower ore recovery due to poor mining practice or "sterilization" of an ore zone because of poorly backfilled mined out stope nearby	H	M	3	2	6	H	implement good backfill practices, identification of potential problematic areas and implementation of specific action plans, including increased ground support, multiple access, etc..	M	L	2	1	2	L
M4	**Hydrology, and Groundwater Conditions** unexpected water inflow into the mine or higher than anticipated water inflow into the mine	M	M	2	2	4	M	Install dewatering station at the lower level of the mine, seal old workings	L	L	1	1	1	L
M5	**Risk of Pit Wall Failure** Wall/ bench failure causing production delays or fatality	L	H	1	3	3	M	Monitor wall movement (GCMP) Regular geotechnical drilling/mapping campaigns to confirm/supplement data Regular independent consultant site visits	L	M	1	2	2	L
M6	**Health & Safety risk during construction and operation phases** Stability due to simultaneous mining of OP & UG	L	H	1	3	3	M	Monitoring movement in slopes and tunnels/UG openings Coordinate drill & blast short term planning Ensure crown pillar is maintained throughout Proper surveying	L	M	1	2	2	L
M7	**Health & Safety risk during construction and operation phases** Fatalities and serious injuries due to the collapse of the existing UG Opening	L	H	1	3	3	M	Backfill existing stopes and other larges UG excavations with CRF or another cemented type backfill. Monitoring movement in slopes and tunnels/UG openings Ground control program (GCMP) Monitor wall/roof stability Ground control studies Regular inpendent consultant site visits Plan for alternate access/egress	L	H	1	3	3	M
M8	**Rock fall/Fall of Ground** Mine/area shutdown	L	H	1	3	3	M	Ground control program (GCMP) + review existing studies based on new geological interpretations/models Monitor wall/roof stability Ground control studies Regular inpendent consultant site visits Plan for alternate access/egress	L	M	1	2	2	L
M9	**Delay in Construction / Development in Early Years** development of UG infrastructures delayed	M	M	2	2	4	M	multiple UG contractor working on site in order to avoid lost time and increase reactivity to potential issues	L	M	1	2	2	L
M10	**Delay in Construction / Development in Early Years** Not enough feed to new mill	M	M	2	2	4	M	Proper stockpiling Review scenarios with Higher OP feed production	L	M	1	2	2	L
M11	**Underground Temperature** Excessively high temperatures	L	L	1	1	1	L	Proper ventilation and temperautre monitoring	L	L	1	1	1	L
M12	Dust Emission (Health and safety hazard)	H	L	3	1	3	M	Sufficient water truck to maintain haul roads + sprinklers	L	L	1	1	1	L
M13	Petroleum Accidental Spill	M	L	2	1	2	L	Storage facilities equiped with containment, emergency response equipment, emergency plan and training	L	L	1	1	1	L
M14	Runoff quality	M	L	2	1	2	L	Settling ponds, monitoring	L	L	1	1	1	L
M15	**Power supply** Electrical line (60KVa) construction delay	M	H	2	3	6	H	Get project calendard updated start ASAP Ensure 22KVa line can be used as backup, Engage discussion with genset rental	L	M	1	2	2	L
M16	**Emergency Power** Power outage underground	M	M	2	2	4	M	Back up generator	L	L	1	1	1	L
M17	**Fire/Explosion Underground** Burning	L	H	1	3	3	M	Fire extinguishers at regular intervals in tunnels Emergency response plan in place and mine rescue team Monitor flammable air particulars	L	M	1	2	2	L
M18	**Mining Infrastructure** Improper implementation of design	L	L	1	1	1	L	Supervise construction work by qualified engineer Perform shallow geotechnical analysis	L	L	1	1	1	L
M19	**Mining Infrastructure** Collision risk	M	M	2	2	4	M	Add traffic signs Make roads/ramps/drifts wide enough for multiple vehicles Training program about minesight driving safety for employees and contractors	L	M	1	2	2	L
M20	**Mine Personnel Safety** Contractor H&S misaligned with company H&S	L	L	1	1	1	L	Communication about H&S during contractor vetting Regular check ins with employees/contractors on site about implementation Proper shift boss training	L	L	1	1	1	L
M21	**Mine Personnel Safety** Miscommunication between employees due to language barriers	L	L	1	1	1	L	Provide materials in main languages spoken on site	L	l	1	1	1	L

Processing and TSF Risks														
	Processing													
P1	**Risk of not achieving the productivity target** Lower throughput than expected	L	M	1	2	2	L	Design consideration for an increase of grinding capacity 'Effective maintenance practices and process control. Identify and resolve bottlenecks. Effective operations procedures and staff training.	L	L	1	1	1	L
P2	**Lower recovery** Lower recovery than expected	M	M	2	2	4	M	Effective grind size control. Effective cyanide and lime addition automation.	L	M	1	2	2	L
P3	**OPEX exceeds target**	L	M	1	2	2	L	Confirmatory variability testwork in future project phases. Effective process control and automation following effective commissioning.	L	L	1	1	1	L
P4	**Insufficient or unreliable water or power supply**	L	M	1	2	2	L	Ensure sufficient surge capacity. Minimize consumption as far as possible.	L	L	1	1	1	L
P5	**Ineffective cyanide detoxification affecting flotation performance**	M	L	2	1	2	L	More detailed testwork in future project phases. Ensure sufficient margins on peroxide addition system. L	L	L	1	1	1	L
P6	**Risk of water contamination (cyanide water into non-cyanide water)** risk of ending with cyanide water into the non cyanide process water circuit	L	H	1	3	3	M	Utilisation of Hydrogen Peroxide to remove cyanide for non-cyanide process water. Inplementation cyanide of on-line analyser on process water	L	M	1	2	2	L
P7	**Risk of high Cu/Zn content in the Concentrate Leaching circuit** due to high Cu/Zn ore feed concenctrate leaching gets very high Cu/Zn values and pregnant solution gets conaminated with Cu/Zn	M	L	2	1	2	L	Blending strategy to smooth the grade of Cu/Zn of ore fed into the plant	L	L	1	1	1	L
P8	**Risk of Carbon contamination in the CIP circuit** Carbon in CIP circuit contaminated with Cu-Zn/Hg/flotation collector or something else	M	L	2	1	2	L	CIP design to increase the carbon rotation cycle and avoid carbon poisoning	L	L	1	1	1	L
	Tailings Storage Facility (TSF)													
T1	**TSF collapse**	L	H	1	3	3	M	Develop & implement OMS Increase dam buttrest design and acces	L	L	1	1	1	L
T2	**TSF failure** TSF Dam failure causing unexpected tailings deversal into the environment	L	M	1	2	2	L	Design of the TSF as per Global Industry Standard on Tailings Management TSF raises only downstream Develop & implement OMS	L	M	1	2	2	L
T3	**Contamination of Ground Water** Leak in TSF geomembrane	L	M	1	2	2	L	2mm geomembrane plus geotextile underneath drainage system under the TSF provision for high quality geomembrane installation company	L	L	1	1	1	L
T4	**Tailings Pipeline Failure**	L	M	1	2	2	L	Orient slope toward current impoudment Inspect pipe condition using ultrasound	L	L	1	1	1	L
T5	**Runoff contamination**	M	M	2	2	4	M	Surveillance and recycling of water	M	L	2	1	2	L
T6	**Process water recovery** Reduction in water recovery from TSF from evaporation	M	M	2	2	4	M	Develop additional water recovery source from the mining site (Spring, mine dewatering, small dams) and pumping infrastructure (Ex: Mobile diesel pump w/ succion)	L	L	1	1	1	L
T7	**TSF overflow in the environnment** Overabondance of water in the TSF from process and precipitations	L	M	1	2	2	L	Maintain higher freeboard in OMS Inquire about evaporator	L	L	1	1	1	L
Environmental and Social Risks														
S1	**Community Relation (construction and operation phases)** Risk of not reaching an agreement with local communities for land occupation	M	L	2	1	2	L	Establish community committee and address environmental issues	L	L	1	1	1	L
S2	**Surface water contamination towards villages**	L	H	1	3	3	M	Ongoing measurments, Plan to capture water run-off and send to TSF	L	L	1	1	1	L
S3	**Closure Plan expectations**	L	L	1	1	1	L	Public consultations to be held	L	L	1	1	1	L
E1	**Environmental risk (construction and operation phases)** Risk of some materials being acid-generating due to a lack of conclusive information	M	L	2	1	2	L	Complete field leaching test Put env compliance inside contract	L	L	1	1	1	L

25 INTERPRETATION AND CONCLUSIONS

25.1 Conclusion

25.1.1 GEOLOGY AND MINERAL RESOURCES

The mineral exploration results for the Zgounder Silver Property have been very positive with a significant upgrade in the Mineral Resources since March 2021. The Property shows further upside potential and additional exploration is warranted.

At a cut-off grade of 65 g/t Ag, pit-constrained updated Measured and Indicated Mineral Resource totals 514 kt grading 357 g/t Ag for 5.9 Moz Ag. At a cut-off grade of 75 g/t Ag, out-of-pit updated Measured and Indicated Mineral Resource totals 9.0 Mt grading 309 g/t Ag for 89.3 Moz Ag, and updated Inferred Mineral Resource totals 542 kt grading 367 g/t Ag for 6.4 Moz Ag. At a cut-off grade of 50 g/t Ag, tailings updated Indicated Mineral Resource totals 272 kt grading 94 g/t Ag for 817 koz Ag. The effective date of this updated Mineral Resource Estimate is December 13, 2021.

The updated Measured and Indicated Mineral Resources for Zgounder totalling 9.8 Mt averaging 306 g/t Ag for 96.1 Moz Ag represent an increase of 116% compared to the previous (March 2021) Measured and Indicated Mineral Resources of 44.4 Moz Ag. The updated Inferred Mineral Resources for Zgounder totalling 542 kt averaging 367 g/t Ag for 6.4 Moz Ag represents an increase of 1,519% compared to the previous (March 2021) Inferred Mineral Resources of 395 koz Ag.

The Updated Mineral Resource Estimate incorporates drilling carried out on Zgounder from February 2018 to September 2021. The Mineral Resource database update consists of 516 drill holes (surface and underground combined) for 41,932 m completed at Zgounder. The drilling successfully extended the east-west strike length of the Zgounder silver mineralization from 775 m to 1,100 m and at depth in successfully intersecting silver mineralization in the Exploration Target established by P&E (2021).

Three-dimensional block models were created for the Zgounder Deposit and for the historical tailings located a few hundred metres northwest of the mine site. A geological rock code system was introduced and assigned to the various lithological units and mineralized domains. Continuity directions were assessed based on the orientation of the domains and the spatial distribution of silver. Separate variograms were generated for 1.2 m down-hole silver composites within each domain. Mineralization modelling, grade estimation and Mineral Resource reporting were conducted using GemcomTM, LeapfrogTM, Snowden SupervisorTM and NPV SchedulerTM software. Ordinary kriging was used for grade estimation into 2.0 m x 2.0 m x 2.0 m model blocks.

Mineral Resources have been estimated using Ordinary Kriging of capped composites. Potentially economic mineralization has been identified by categorizing blocks based on a Nearest Neighbor ("NN") assignment. Blocks assigned a NN grade of 40 g/t Ag or higher are categorized as potentially economic mineralisation, whereas blocks assigned a NN grade less than 40 g/t Ag are categorised as waste. The Mineral Resource Estimates have classified into Measured, Indicated and Inferred based on a series of expanding search ellipsoids.

25.1.2 Mineral Processing and Metallurgical Testing

Metallurgical testwork indicated that silver recoveries of 92% is achievable for this ore when processing it through a flowsheet consisting of gravity concentration followed by intensive cyanidation of the gravity concentrate and flotation of the gravity tailings. The flotation concentrate is leached and the slurry then reports to a series of CCD thickeners to recover a pregnant solution to the Merril-Crowe refinery. The flotation tailings combines with the CCD barren slurry and is then leached. Silver is recovered from this slurry in a carousel CIP. The strip solution from the carbon elution circuit also reports to the Merril-Crowe step, where cementation is used to produce a sludge for smelting.

The ore hardness was determined to be very hard but, fortunately, met testing indicated that silver recoveries are not overly sensitive to grind hence a primary grind of 80% passing 100 microns was adopted.

25.1.3 Mineral Reserves and Mining Methods

The Mineral Reserves are estimated at 8.9 Mt of ore grading 251 g/t Ag, combining the open pit, historical tailings reclamation and underground portions of the mine. Further Mineral Reserves could be defined by reclamation the pillar located between the bottom of the open pit and the top of the underground at the end of the mine life.

The Report for the Zgounder Project is based on an 11-year mine life combining the open pit mining, historical tailings reclamation, and underground mining. The mine will operate year-round, seven (7) days per week, twenty-four (24) hours per day (two (2) 12-hour shifts). Two (2) weeks of adverse weather conditions per year are considered in the mine plan.

Approximately 71% of the ore will be coming from the underground portion of the operation while 29% will come from the open pit and historical tailings reclamation. Over the LOM, 8.6 Mt of ore will be mined or reclaimed, of which 92% will be sent directly to the crusher and mill and 8% will be sent to an ore stockpile to be rehandled later. A total of 27.6 Mt of waste material will be mined to access the ore.

25.1.4 RECOVERY METHODS

The Zgounder expansion project consists on the addition of a new mineral processing facility that has nine (9) areas: crushing, grinding, flotation, intensive leach reactor, counter-current decantation, cyanidation, ADR process, Merrill-Crowe, and refinery.

This new processing facility will integrate the production coming from the two existing processing facilities, and all of the silver produced by the three facilities will be poured as doré bars (ingots) in the new processing facility refinery.

25.1.5 PROJECT INFRASTRUCTURE

25.1.5.1 Tailings Storage Facility (TSF)

Given the production volumes contemplated at this stage of the Project, and the estimated duration of the mine, the topographic and hydrographic conditions of Sites C and A have shown that the tailings dam can be built in three (3) phases, allowing for a distribution of the capital investments over the entire period of the operation thus reducing the initial Capex.

The first two (2) phases will be constructed at Site C, and will support approximately 7 years of the LOM. The last phase will be constructed at Site A, for an initial period of 3 years of operation. It is important to note that Site A TSF could be extended in the future if new mineralised zones are discovered and the mine life is extended.

25.1.5.2 Surface Water Management

The water management strategy and a water balance model were developed to account for dry, average and wet years. The water management strategy utilises the large catchment areas of the project site and existing and proposed infrastructure to collect and convey fresh water, contact water and non-contact water to the TSF Site C storage basin. Engobe proposes a flexible system that can adapt to variable precipitation conditions by selectively collecting and discharging water on-site, using a system of one (1) large water reservoir, two (2) new collection ponds, new and existing diversion ditches, and pumping stations.

Based on obtained results, Englobe concludes that the site can manage the water supply to the processing plant during variable conditions by operating a water reservoir with a maximum initial capacity of 620,000 m^3 and maintain the minimum water volume at around 150,000 m^3. Those volumes can be refined to suit the staged TSF pond construction sequence as more climatic and operational information become available. The water treatment plant does not seem to be required under the analyzed conditions.

For the water management strategy to be effective, the site must be appropriately instrumented and controlled. All data must be stored and analysed to make valuable conclusions about the interaction between the mine site operation, the local climate and environment. The proposed system, comprised of physical infrastructure and instrumentation, can be adapted to the variable annual and seasonal precipitation, changing climate and operational requirements, and ensure that all available water resources are accounted for and effectively managed.

25.1.6 Environmental Studies, Permitting and Social or Community Impact

The Zgounder Mine site has a very long mining history and can be described as a “brownfield site”. The surface mineralised showings were first exploited as far back as the 10th to 12th Century. More recent mining and processing activities existed between the 1950s and 1970s, and later 1980s to 1990s with a cyanide-leaching process plant. There are evidence of heavy metal leaching and cyanide contamination found in surface water and potentially groundwater water and soils on site. Since 2014, Aya has undertaken civil works to reduce the impacts and risks, especially for the tailings impoundments containment, stability and revegetation.

The ESIA has identified the relevant risks and mitigation measures are presented. The Environmental Management Plan and monitoring program ensure that the mining activities will comply with its permits and the applicable laws and regulations for mining operations in Morocco.

26 RECOMMENDATIONS

Considering the positive outcome of this Report, it is recommended to pursue the next phase of the Project through various aspects need to be monitored or done are listed below.

26.1 Geology and Mining

26.1.1 GEOLOGY AND MINERAL RESOURCES

It is recommended that issues noted in the database be corrected, and that the methodology implemented for the Mineral Resource Estimate as described be continuously reconciled and validated against actual production results.

An exploration budget of US$6.6M is recommended for Zgounder in 2022, for a total of 46,000 m of step-out and infill drilling on the Zgounder Mine Property. The drill program is to focus on:

1. Expanding the Mineral Resources along strike, particularly to the east, and at depth to the granite contact; and

2. Advance Inferred Mineral Resources to Indicated Mineral Resources to support Mineral Reserve Estimates.

26.1.2 MINING METHODS

In the next phase of the Project, DRA recommends looking at the possibility of extending the open pit to the 2000 Level. DRA also recommends looking at the possibility of in-pit waste stockpiling in the mined-out portions of the open pit to minimise the ex-pit waste stockpile size and its related environmental footprint.

The COG was estimated according to the available information at the time of this study and should be reviewed and optimised if the Project has more updated circumstances or cost rates to improve the Project's profitability and Mineral Reserves.

Improvement opportunities still remain and can be included in the next phase. Special open pit and underground sequencing and redesigning relying on new geological targets currently under development can be undertaken:

Mine design should be reviewed and redesigned taking into account new geological targets and Mineral Resources that are being currently developed. Some of the newly identified areas are close to the main decline and could bring in ore production sooner than the Main deposit.

Due to the complexity of the geometry of the deposit, definition drilling should be done planned during detail engineering, and the mining method selection should be revisited.

Drill and blast parameters in the study were designed according to typical stope geometry in each area. During detail engineering and in the operations phase, determination of optimal burden and spacing should be reviewed stope by stope to optimise drilling and blasting costs.

Battery operated mining fleets are currently being developed by the major mine equipment manufacturers and implemented in more and more operations to reduce mine ventilation needs. It is recommended that this technology be considered when replacing the mine fleet in a few years and that this new technology has proven itself at other operation.

For the CRF, DRA recommends that UCS testing be performed for CRF to gain understanding of strength development with regards to the target strength of 235 kPa.

26.1.3 Geotechnical Considerations

RockEng geotechnical evaluations (RockEng, 2022) utilised a previous version of the Zgounder mine design. The most significant change to the underground mine plan is a reduction in level spacing from 25 m to 12 m. A geotechnical review of the current mine design by RockEng has identified the following recommendations to be explored further during later phases of the Project.

- Level stacking with small level spacing introduces pillar stability risk between level, particularly in intersections and wide span excavations. It is recommended that level layouts be adjusted to avoid direct stacking of lateral development level-to-level.
- Cross-cut stacking in long-hole stopes will introduce pillar stability risk for drill-horizons during overhand advance. This may be de-risked by implementing double-lift long-hole stopes (i.e., effectively skipping every other level for long-hole mining blocks). It is recommended that opportunities to achieve double-lift long-hole stopes be investigated further.
- Backfill strength requirements are based on 25 m level spacing. For shorter vertical exposures there may be opportunity to reduce backfill strength. It is recommended that backfill strengths be reviewed as stope designs and sequencing plans advance.

26.2 Process and Recovery Methods

26.2.1 Process

While the selected flowsheet does improve overall recoveries marginally (~2%) it is complex and somewhat costly and it requires detoxification of reclaim water in order to minimise the risk of cyanide affecting the performance of the flotation section. The value of a simpler flowsheet needs to be revisited.

The concept, assumed during the preparation of the FS, was that peroxide would be injected into the reclaim water line at the tailings dam and that the residence time between this point and the discharge into the water tanks at the plant would be sufficient to destroy most of the free and WAD cyanide. The tests conducted were performed for 60 minutes which likely exceed the residence time in the reclaim pipeline. The requirement for a small reactor tank to add residence time for this reaction would need to be evaluated in future phases of the Project.

26.2.2 RECOVERY METHODS

Considering the positive outcome of this Report, it is recommended to pursue the next phase of the Project through various aspects need to be monitored or done are listed below.

Note that the mineral resource estimate has been updated at the end of the feasibility study and its supporting testwork to show an increased head grade of around 270 g/t. There is a risk that the plant may experience throughput limitations and/or a decrease in recovery as a result of the elevated head grade. However, it is our belief that this risk is marginal as it is likely that the additional silver will report to the gravity concentrate or flotation concentrate and that it will simply result in a higher than expected silver concentration in the pregnant solution reporting to the refinery. This assertion is supported by anecdotal evidence from the current operations, where an increase in grade is normally expected to lead to improved recoveries as opposed to a decrease. However, this remains a speculative expectation as it is not based on testwork results. It is recommended that additional testing should be conducted at the higher grade to confirm that the design will be able to handle it.

During the feasibility study, DRA completed a trade off study concerning the flow sheet to be developed. The outcome showed minor financial differences between the whole-ore leaching flowsheet ("full CCD") and the retained flowsheet ("flotation - CCD - CIP"). It is recommended to perform additional confirmatory testwork on the "full CCD" flowsheet using several cyanide concentrations for leaching and on variable head grade samples. By doing these new series of test, a potential flowsheet optimisation could be contemplated, while reaching potentially comparable recovery levels. The process water circuits could potentially be simplified (no need to separate the "cyanide" process water from the "non-cyanide" process water), and Process Plant capex could be optimized. It is recommended to look at this opportunity into more details during the next phase of engineering.

26.3 Project Infrastructure

26.3.1 Tailings Storage Facility

Currently, testwork is ongoing at the geotechnical laboratory and therefore, the conclusions of this testwork are not included in this FS. It is recommended to complete the stability analysis of the Site C TSF design including the results of the ongoing geotechnical testwork and depending on the results, a modification of the TSF Site C could be considered in order to increase its storage capacity.

Also, additional design work is recommended to consider the borrow materials for the TSF dam construction to be excavated from inside the footprint of the TSF Site C, and hence increase its storage capacity.

26.3.2 Surface Water Balance and Infrastructure

The proposed water management strategy and resulting water balance consider several assumptions that need to be refined before the next phase of the Project. The following list presents the main assumptions and limitations that should be refined at the next design phase:

- A monthly water balance should be developed;
- The current and future TSFs should have a tailings and water management manual (OMS manual) where the water management principles are presented and integrated with the tailings management principles;
- The proposed water management strategy requires actions and procedures based on a time-sensitive understanding of the current site water storage conditions and forecasting of short-term environmental conditions. A flexible water management tool should be developed and implemented;
- The current and future operations should monitor the TSFs and document input and output parameters to better refine the water balance model;
- The water balance model in this study relies on weather data that is not site-specific. The site weather station should be used at the next phase of the Project to better define the site-specific weather data; and,
- Mine dewatering is an important input to the global water balance. A more precise dewatering plan would increase the precision of the proposed water balance.
- Incorporate the new waste dump water management and infrastructure into the overall site water balance.

Finally, it is recommended to execute the geotechnical, hydraulic, and hydrological studies required to move to the detailed engineering phase.

26.4 Environmental Studies, Permitting and Social or Community Impact

- Develop a water management program, including hydrological and hydrogeological characterisations, to ensure the project's design compliance with commitments, permits and legislation requirements.
- With the increase of the project footprint try to reduce the number of effluents with the collection ditches. This will help defining the required water treatment if required.
- Environmental monitoring program should clearly identify sampling locations (water, soil, air) and coordinates and maintain the sampling labels for traceability.
- More sampling locations of soil samples should be added, near petroleum storage tanks.
- Sediments or alluvion along oueds should also be characterised.
- Continue geochemical characterisation on waste rock and tailings to verify the potential of acid mine drainage and metal leaching, as recommended in the report from Lamont inc, 2021.
- Continue to investigate water and soil contamination upstream, at the mine site and downstream, as recommended in the ESIA.
- Revise and increase the frequency of the mine effluents monitoring for a more efficient control within the operations. In Canada for example, monitoring is performed on a weekly basis and reported on a monthly basis.
- Introduce treatment at the source with the addition of a cyanide destruction process (example SO_2 air);
- Continue to treat tailings to increase pH and help the precipitate heavy metals and arsenic within the tailings.
- Ensure that the emergency responses plan is known and tested and the intervention material is available.
- Perform root cause analysis for recurrent accidental spills in order to identify appropriate solutions.
- Consider in the closure plan to fill the open pit with waste rock from the waste dump to reduce project footprint and remediate old tailings pond after the processing of tailings.
- Ensure a sufficient number of environmental staff in order to meet regulatory performance.

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28 ABBREVIATIONS

The following abbreviations may be used in this Report.

Abbreviation	Meaning or Units
'	Feet
"	Inch
$	Dollar Sign
$/m²	Dollar per Square Metre
$/m³	Dollar per Cubic Metre
$/t	Dollar per Metric Tonne
$M	dollars, millions
%	Percent Sign
% w/w	Percent Solid by Weight
¢/kWh	Cent per Kilowatt hour
°	Degree
°C	Degree Celsius
°C	degree Celsius
2D	Two Dimensions
3D	Three Dimensions
ACA Howe	ACA Howe International
AfriLab	African Laboratory for Mining and Environment
Ag	Silver
ALS	ALS Global / ALS Laboratory
ANDZOA	*Agence Nationale de Développement des Zones Oasiennes et l'Arganier*
AOI	Area of Influence
AP	Acid potential
AQG	Air Quality Guidelines
AQSRs	Air Quality Sensitive Receptors
ARD	Acid Rock Drainage
As	arsenic
ASL	Above Sea Level
ASPT	Average Score per Taxon.
ASTM	American Standard Test Method
Au	gold
AWG	American Wire Gauge
Aya	Aya Gold & Silver Inc.

Abbreviation	Meaning or Units
Az	Azimuth
Bank	Bank Cubic Metre
BAU	Business as Usual
BDF	Bulk Density Factors
BFA	Bench Face Angle
BFS	Bankable Feasibility Study
BIF	Banded Iron Formation
BIGS	BIGS Global Burkina
BOD	Biological Oxygen Demand
BOF	Basic Oxygen Furnace
BQ	Drill Core Size (3.65 cm diameter)
BRGM	Bureau de Recherche Géologique et Minière
BRPM	*Bureau de Recherches et de Participations Minières (Morocco)*
BRPM	Bureau de Recherche et Participation Minière
BS	Base Saturation
BSG	Bulk Specify Gravity
BTU	British Thermal Unit
BWI	Bond Ball Mill Work Index
Ca	Calcium
$CaCO_3$	Calcium Carbonate
CAD	Canadian Dollar
Capex	Capital Expenditures/Capital Cost Estimate
CBD	Convention on Biological Diversity
Cc	Calcite
CCD	counter-current decantation
CD	*Calcaires et dolomites inférieures*
CDE	Canadian Development Expenses
CDP	Closure and Decommissioning Plan
Ce	Cesium
CEC	Cation Exchange Capacity
cfm	Cubic Feet per Minute

Abbreviation	Meaning or Units
CFR	Cost and Freight
CH	Critical Habitat
CICS	Indicative Criteria for Soil Contamination
CIF	Cost Insurance and Freight
CIL	Carbon-in-Leach
CIM	Canadian Institute of Mining, Metallurgy, and Petroleum
CIP	Carbon-in-Pulp
Cl	Clay
CL	Concentrate Leach
cm	centimetre(s)
CMT	*Compagnie Minière de Touissit*
CNDH	National Council for Human Rights
Co	Cobalt
CO	Carbon Monoxide
CO_2	Carbon Dioxide
COG	Cut-Off Grade
COV	Coefficient of Variation
CPM	Commission of Phytosanitary Measures
CR	Critically Endangered
CREI	Regional Committee for Environmental Impact Studies
CRM or standards	Certified Reference Materials
CTAE	*Comité Technique d'Analyse Environnementale*
CTMP	*Centre de Technologie Minérale et de Plasturgie Inc.*
Cu	Copper
Cu	copper
Cu_2S or Csi	Chalcocite
Cu_5FeS_4 or Bo	Bornite
CuEq	Equivalent Copper
$CuFeS_2$ or Cpy	Chalcopyrite
CWI	Crusher Work Index
D	Day
d/w	Days per Week
d/y	Days per Year

Abbreviation	Meaning or Units
D2	Second Generation of Deformation
D3	Third Generation of Deformation
D4	Fourth Generation of Deformation
dB	Decibel
dBA	Decibel with an A Filter
DDH	Diamond Drill Hole
DDH	diamond drill hole
Deg	Angular Degree
DEM	Digital Elevation Model
DFS	Definitive Feasibility Study
DGPS	Differential Global Positioning System
Dia.	Diametre
DMS	Dense Media Separator
DOL	Direct-On-Line
Dpm	Diesel Particulate Matter
DT	Davis Tube
DTM	Digital Terrain Model
Dvr	Dissected Valley Rolling
DWI	Drop Weight Index
DWT	Drop Weight Test
DXF	Drawing Interchange Format
E	East
E	east
EA	Environmental Assessment
EAA	Ecological Area of Analysis.
EAB	Environmental Assessment Board
EAF	Electric Arc Furnace
EBS	Environmental Baseline Study
EC	European Commission
EC	Electrical conductivity
EDS	Energy-dispersive X-ray spectroscopy
EHS	Environmental, Health and Safety
EI	Ecological Importance
EIA	Environmental Impact Assessment
EIR	Environmental Impact Report
EIS	Environmental Impact Statement
ELVs	Emission Limit Values

Abbreviation	Meaning or Units
EM	Electromagnetic
EMP	Environmental Management Plan
EMPr	Environmental Management Program
EN	Endangered
ENP	European Neighbourhood Policy
EOH	End of Hole
EP	Equator Principles
EP	Environmental Permit
EPA	Environmental Protection Agency
EPCM	Engineering, Procurement and Construction Management
EPFIs	Equator Principles Financial Institutions
EPPO	European and Mediterranean Organization for the Protection of Plants
EQA	Environmental Quality Act
ER	Electrical Room
ESBS	Environmental and Social Baseline Study
ESIA	Environmental and Social Impact Assessment
ESP	Exchangeable Sodium Percentage
ET	Evapotranspiration
FA	Fire Assay
FAO	Food and Agriculture Organisation
Fe	Iron
FEED	Front End Engineering Design
FEL	Front End Loader
FeS_2 or Py	Pyrite
FOB	Free on Board
FS	Feasibility Study
ft	Feet
Fu	Flat to undulating
FW	Foot wall
g	Gram(s)
G&A	General and Administration
g/l	Grams per Litre
g/t	Grams per Tonne
gal	Gallons

Abbreviation	Meaning or Units
GCP	Ground Control Points
GCW	Gross Combined Weight
GDP	Gross Domestic Product
GEMS	Global Earth-System Monitoring Using Space
GHG	Greenhouse Gas
GIIP	Good International Industry Practice
GLCs	Ground Level Concentrations
GMG/GoldMinds	GoldMinds Geoservices Inc.
GOH	Gross Operating Hours
GPS	Global Positioning System
Gr	Granular
H	Horizontal
h	Hour
h/d	Hours per Day
h/y	Hour per Year
H_2	Hydrogen
H_2O	Water
H_2SO_4	Sulphuric Acid
ha	Hectare(s)
HCEFLCD	The High Commission for Water and Forests and the Fight against Desertification
HCl	Hydrochloric Acid
HDPE	High Density PolyEthylene
HF	Hydrofluoric Acid
HFO	Heavy Fuel Oil
HG	High Grade
Hg	mercury
HL	Heavy Liquid
HNO_3	nitric acid
Hp	Horse Power
HQ	Drill Core Size (6.4 cm Diameter)
HTH	High Test Hypochlorite
HVAC	Heating Ventilation and Air Conditioning
HW	Hanging wall
Hz	Frequency in Hertz
I/O	Input / Output

Abbreviation	Meaning or Units
IBA	Important Bird Area
IBMWP	Iberian Biomonitoring Working Party
ICMM	The International Council on Mining and Metals
ICP-AES	Inductively Coupled Plasma – Atomic Emission Spectroscopy
ICP-MS	Inductively Coupled Plasma – Mass Spectroscopy
ICP-OES	Inductively Coupled Plasma – Optical Emission Spectroscopy
ID	Identification
ID^2	inverse distance squared
IDW	Inverse Distance Method
IDW^2	Inverse Distance Squared Method
IEC	International Electro-Technical Commission
IFC	International Finance Corporation
In	Inches
INDH	National Initiative for Human Development
IPAs	Important Plant Areas
IPPC	International Convention for the Protection of Plants
IR	Infrared Radiation
IRA	Inter-Ramp Angle
IRR	Internal Rate of Return
ISG	Industry Sector Guidelines
ISO	International Standards Organisation
ISRIC	International Soil Reference Centre
IT	Interim target
IT	Information Technology
IUCN	International Union for the Conservation of Nature
IWUL	Integrated Water Use Licence
IWULA	Integrated Water Use Licence Application
IWWMP	Integrated Water and Waste Management Plan
JORC	Joint Ore Reserves Committee
JV	Joint Venture

Abbreviation	Meaning or Units
K	Kelvin
k	thousand(s)
KCl	Potassium Chloride
KE	Kriging Efficiency
Kg	Kilogram(s)
kg/l	Kilogram per Litre
Kg/t	Kilogram per Metric Tonne
Kl	Kilolitre
km	Kilometre(s)
km/h	Kilometre per Hour
kPa	Kilopascal
KSR	Kriging Slope Regression
kt	Kilotonne
kV	Kilovolt
kVA	Kilo-volt-ampere = 1,000 volt-ampere (amps)
kW	Kilowatt
kWh	Kilowatt-hour
kWh/t	Kilowatt-hour per Metric Tonne
L	Line
l	Litre(s)
l/h	Litre per hour
lbs	Pounds
LC	Least Concern.
Level	mine working level referring to the nominal elevation (m RL), e.g. 4285 level (mine workings at 4285 m RL)
LFO	Light Fuel Oil
LG	Low Grade
LG-3D	Lerchs-Grossman – 3D Algorithm
LHOS	Long-hole Open Stoping
Li	Lithium
LIMS	Laboratory Information Management Systems
LIMS	Low Intensity Magnetic Separator
LMo	Monin Obukhov Length
LOI	Loss on ignition
LOM	Life of Mine

Abbreviation	Meaning or Units
LP	Sound pressure level (in dB)
LPA	*Lumière polarisée analysée*
LPNA	*Lumière polarisée non-analysée*
Ltd	Limited
LV	Low Voltage
LW	Sound Power Level (in dB)
m	Metre
M	million(s)
m	metre(s)
m/h	Metre per Hour
m/s	Metre per Second
m^2	square metre(s)
m^2	Square Metre
m^3	Cubic metre(s)
m^3/d	Cubic Metre per Day
m^3/h	Cubic Metre per Hour
m^3/y	Cubic Metre per Year
Ma	Malachite
mA	Milliampere
Ma	millions of years
MAAQS1	Morocco Ambient Air Quality Standards
MAPM	*Ministère de l'Agriculture et des Pêches Maritimes*
MASL	Meters above sea level
MATHUE	*Ministère de l'Aménagement du territoire, de l'Habitat, de l'Urbanisme et de l'Environnement*
MCC	Motor Control Centre
MDPI	Multi-disciplinary Digital Publishing Institute
MEB	*Microscopie électronique à balayage*
MEMEE	*Ministère de l'Energie, des Mines, de l'Eau et de l'Environnement*
Meq	Mili equivalent
MES	Minimum Emission Standards (SA)
mg	Milligram(s)
Mg	Magnesium
mg/kg	Miligram per kilogram
mg/l	Milligram per Litre

Abbreviation	Meaning or Units
$mg/m^2/day$	Milligram per metre squared per day
MH	Modified Habitat
MIBC	methyl isobutyl carbinol
MIBK	Methyl Isobutyl Ketone
min	Minute
min/h	Minute per Hour
min/shift	Minute per Shift
ml	Millilitre
ML	Metal Leaching
MLA	Mineral Liberation Analyzer
mm	millimetre
mm/d	Millimetre per Day
Mm^3	Million Cubic Metres
MM5	Fifth Generation NCAR / Penn State Mesoscale Model
MMER	Metal Mining Effluent Regulation
MMU	Mobile Manufacturing Units
Mn	Manganese
MOLP	Multiple Objective Linear Programming
MOU	Memorandum of Understanding
Moz	million ounces
Ms	Mountains and Steep Slopes
Mt	Million Metric Tonnes
Mtpa or Mt/a	million tonne per annum
Mtpy or Mt/y	Million of Metric Tonne per year
MV	Medium Voltage
MVA	Mega Volt-Ampere
MW	Megawatts
MWh/d	Megawatt Hour per Day
My	Million Years
N	Nitrogen
N	North
NAAQS	National Air Quality Standards
NaCN	Sodium cyanide
NAD	North American Datum
NAG	Non-Acid Generating
NaHS	Sodium hydrosulfide

Abbreviation	Meaning or Units
Nb or Nº	Number
NCCC	National Climate Change Committee
ND	No Data
NDCs	Nationally Determined Contributions
NE	Northeast
NEMA	National Environmental Management Act
NEMAQA	National Environment Management Air Quality Act
NFPA	National Fire Protection Association
NGO	Non-Governmental Organisation
NGR	Neutral Grounding Resistor
NH	Natural Habitat
Ni	Nickel
NI 43-101	National Instrument 43-101
NLG	Noise level guidelines
Nm³/h	Normal Cubic Metre per Hour
NN	nearest neighbour
NNE	north-northeast
NNP	Net neutralisation potential
NNW	north-northwest
NOx	Oxides of nitrogen
NP	Neutralisation Potential
NPI	National Pollutant Inventory (Australia)
NPR	Neutralisation potential ratio
NPV	Net Present Value
NQ	Drill Core Size (4.8 cm diameter)
NRs	Noise receptors
NSR	Net Smelter Return
NT	Near Threatened
NTP	Normal Temperature and Pressure
NTS	National Topographic System
NW	Northwest
O/F	Overflow
O_3	Ozone
OB	Overburden
OC	Organic Carbon
OF	Ferric oxides

Abbreviation	Meaning or Units
OK	Ordinary Kriging
OK	ordinary kriging
ONEE	*Office national de l'électricité et de l'eau potable*
ONHYM	*Office National des Hydrocarbures et des Mines* "The National Office of Mines and Hydrocarbons"
OP	Open Pit
Opex	Operating Expenditures
ORE	ORE Research & Exploration Pty Ltd.
ORMVA	*Office Régional de Mise en Valeur Agricole*
oz	Ounce (troy)
oz/t	Ounce per Short Ton
p	Pressure in Pa
P&E	P&E Mining Consultants Inc.
P&ID	Piping and Instrumentation Diagram
P.Eng.	Professional Engineer
P.Geo.	Professional Geoscientist
P_{80}	80% percent passing
Pa	Pressure in Pascal
PAG	Potential Acid Generating
PAGER	Program for the Grouped Supply of Drinking Water to Rural Populations
PANE	*Plan d'Action National pour l'Environnement*
PAOI	Project Area of Influence
PAPs	Project Affected Persons
PAX	potassium amyl xanthate
Pb	Lead
Pd	Palladium
PEA	Preliminary Economic Assessment
PERG	Global Rural Electrification Program
PET	Potential Evapotranspiration
PF	Power Factor
PFS	Pre-Feasibility Study
PFS	Pre-Feasibility Study
PGE	Platinum-Group Element

Abbreviation	Meaning or Units
PGGS	Permit for Geological and Geophysical Survey
Ph	Phase (electrical)
pH	Potential Hydrogen
PIII	Basement
PIR	Primary Impurity Removal
PLC	Programmable Logic Controllers
PM	Particulate Matter
PNUD	United Nations Development Program
PP	Preproduction
ppb	part per billion
ppm	parts per million
PPP	Public Participation Process
pref	Reference pressure, 20 µPa
Project	Zgounder Project or Zgounder Mine
Property	the Zgounder Property that is the subject of this Technical Report
PS	Performance Standards
Psi	Pounds per Square Inch
Pt	Platinum
P-T	Pressure-Temperature
Pty	Proprietary
PVC	Polyvinyl Chloride
PVT	Pollution Tolerant Valves (diatoms).
Q1, Q2, Q3, Q4	first quarter, second quarter, third quarter, fourth quarter of the year
QA/QC	Quality Assurance/Quality Control
QDGC	Quarter Degree Grid Cell
QKNA	Quantitative Kriging Neighbourhood Analysis
QMS	Quality Management System
QP	Qualified Person
RAP	Resettlement Action Plans
RBA	*Reserve de Biosphere Arganeraie*
RC	Reverse circulation
RCIA	Rapid Cumulative Impact Assessment
RCMS	Remote Control and Monitoring System
RER	Rare Earth Magnetic Separator

Abbreviation	Meaning or Units
RMR	Rock Mass Rating
ROM	Run of Mine
Rpm	Revolutions per Minute
RQD	Rock Quality Designation
RWI	Bond Rod Mill Work Index
S	South
S	Sulphur
S/R or SR	Stripping Ratio
SA	South Africa
SA NAAQS	South African National Ambient Air Quality Standards
SA NDCR	South African National Dust Control Regulations
SACEM	Société Anonyme Chérifienne d'Études Minières du Maroc
SAG	Semi-Autogenous Grinding
SANAS	South African National Accreditation System
SANS	South African National Standard
Sc	Scandium
SCC	Species of Conservation Concern
Scfm	Standard Cubic Feet per Minute
SCIM	Squirrel Cage Induction Motors
SE	Southeast
sec	Second
SEDAR	System for Electronic Document Analysis and Retrieval
SEM	Scanning Electronic Microscope
SEP	Stakeholder Engagement Plan
Set/y/unit	Set per Year per Unit
SG	Soil Groups
SG	Specific Gravity
SGS-Lakefield	SGS Lakefield Research Limited of Canada
SIA	Social Impact Assessment
SIR	Secondary Impurity Removal
SLM	Sound Level Meter
SMC	SAG Mill Comminution
SMU	Soil Mapping Unit

Abbreviation	Meaning or Units
SNAM	*Société Nationale des Autoroutes du Maroc*
SNRC	*Système National de Référence Cartographique*
SO_2	Sulphur dioxide
SOMIL	Société Minière de Sidi Lahcen
SoW	Scope of Work
SPI	Specific Pollution Index (diatoms).
SPI	SAG Power Index
SPLP	Synthetic Precipitation Leaching Procedure
SPT	Standard Penetration Tests
SSE	South southeast
SSW	South southwest
STRM	Shuttle Radar Topography Mission
SW	South West
SW	Switchgear
SW	southwest
t	metric tonne(s)
t/d or tpd	Metric Tonne per Day
t/h or tph	Metric Tonne per Hour
t/h/m	Metric Tonne per Hour per Metre
$t/h/m^2$	Metric Tonne per Hour per Square Metre
t/m or tpm	Metric Tonne per Month
t/m^2	Metric Tonne per Square Metre
t/m^3	tonnes per cubic metre
t/m^3	Metric Tonne per Cubic Metre
T+C	Tailings and cement
T28	T28 percussion hammer drill rig (Jackleg drill)
Ta	Tantalum
TAL	Total Alkalinity
TCLP	Toxicity Characteristic Leaching Procedure
TD	Tamjout Dolomite
Technical Report	this NI 43-101 Technical Report
the Company	the Aya Gold & Silver Inc. company that the report is written for

Abbreviation	Meaning or Units
TIN	Triangulated Irregular Network
ton	Short Ton
tonne or t	Metric Tonne
ToR	Terms of Reference
TOS	Trade-Off Study
t/a or tpa	Metric Tonne per Annum
t/d or tpd	tonnes per day
TSF	Tailings Storage Facility
TSP	Total Suspended Particulates
TSS	Total Suspended Solids
U	Uranium
U/F	Under Flow
UG	Underground
Uh	Undulating to hilly
UK	United Kingdom
ULC	Underwriters Laboratories of Canada
UNESCO	United Nations Educational, Scientific, and Cultural Organization
UNFCCC	United Nations Framework Convention on Climate Change
US	United States
US EPA	United States Environmental Protection Agency
USA	United States of America
USD or $ US	United States Dollar
USGPM	US Gallons per Minute
USGS	United States Geological Survey
UTM	Universal Transverse Mercator grid system
V	Vertical
V	Volt
VAC	Ventilation and Air Conditioning
VFD	Variable Frequency Drive
VLF	Very Low Frequency
VOC	Volatile Organic Compounds
VU	Vulnerable
W	Watt
W	West

Abbreviation	Meaning or Units
w/v	Weight/volume
w:o	Waste to ore ratio
WAC	West African Archean Craton
WB	World Bank
WHIMS	Wet High Intensity Magnetic Separation
WHO	World Health Organization
WRA	Whole Rock Analysis Method
WRB	World Reference Base
WRD	Waste Rock Dump
WSD	World Steel Dynamics
Wt	Wet Metric Tonne
wt	weight
wt%	weight percent
WWTW	Wastewater Treatment Works
X	X Coordinate (E-W)
XRD	X-Ray Diffraction
XRF	X-Ray Fluorescence
y	Year
Y	Y coordinate (N-S)
Z	Z coordinate (depth or elevation)
Zgounder Deposit	Zgounder Silver Project
Zgounder Property	Zgounder Silver Project
ZMSM	Zgounder Millenium Silver Mining SA
Zn	Zinc
µg	Microgram(s)
µg/m³	Micrograms per cubic meter
µm	Microns, Micrometre
µPa	Pressure in micro-pascal

29 CERTIFICATE OF QUALIFIED PERSON

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CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*NI 43-101 Technical Report – Feasibility Study for the Zgounder Expansion Project, Kingdom of Morocco"* filed with an original date of March 31, 2022 and an amended date of June 16, 2022, with an effective date of December 13, 2021 (the "**Technical Report**"), prepared for Aya Gold and Silver Inc. ("**Aya**" or the "**Corporation**").

I, William Stone*, Ph.D., P.Geo.* do hereby certify:

1. I am an independent senior geological consultant working for P&E Mining Consultants Inc. located at 201 Country Court Blvd, Suite 304, Brampton, Ontario, Canada L6W 4L2.
2. I am a graduate from of Dalhousie University with a Bachelor of Science (Honours) degree in Geology (1983). In addition, I have a Master of Science in Geology (1985) and a Ph.D. in Geology (1988) from the University of Western Ontario. I have worked as a geologist for a total of 36 years since obtaining my M.Sc. degree.
3. I am a registered member of the Professional Geoscientists of Ontario (License No 1569) and the Professional Engineers & Gescientists Newfoundland & Labrador (License No. 10221).
4. My relevant work experience includes:

• Contract Senior Geologist, LAC Minerals Exploration Ltd.	1985-1988
• Post-Doctoral Fellow, McMaster University	1988-1992
• Contract Senior Geologist, Outokumpu Mines and Metals Ltd.	1993-1996
• Senior Research Geologist, WMC Resources Ltd.	1996-2001
• Senior Lecturer, University of Western Australia	2001-2003
• Principal Geologist, Geoinformatics Exploration Ltd.	2003-2004
• Vice President Exploration, Nevada Star Resources Inc.	2005-2006
• Vice President Exploration, Goldbrook Ventures Inc.	2006-2008
• Vice President Exploration, North American Palladium Ltd.	2008-2009
• Vice President Exploration, Magma Metals Ltd.	2010-2011
• President & COO, Pacific North West Capital Corp.	2011-2014
• Consulting Geologist	2013-2017
• Senior Project Geologist, Anglo American	2017-2019
• Consulting Geoscientist	2020-Present
• Participation and author of many NI 43-101 Technical Reports.	

5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.

6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am responsible for the preparation of Sections 4 to 8 and 23 of the Technical Report. I am also responsible for the relevant portions of Sections 1, 25 to 27 of the Technical Report.
8. I personally visited did not the property that is the subject to the Technical Report.
9. I have had prior involvement with the property that is the subject of the Technical Report.
    - *"Technical Report and Updated Mineral Resource Estimate of the Zgounder Silver Project, Kingdom of Morocco",* Prepared for Aya Gold & Silver Inc., dated April 30, 2021 (with an effective date of March 1, 2021), prepared by P&E Mining Consultants Inc. Report # 395.
    - *"Technical Report and Updated Mineral Resource Estimate of the Zgounder Silver Project, Kingdom of Morocco",* Prepared for Aya Gold & Silver Inc., dated January 28, 2022 (with an effective date of December 13, 2021), prepared by P&E Mining Consultants Inc. Report # 415.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 16 day of June, 2022

""Original Signed and Sealed on file""
William Stone, Ph.D., P.Geo.
Senior Associate Geologist
P&E Mining Consultants Inc.

P&E MINING CONSULTANTS INC.
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I, Fred H. Brown, *P. Geo.,* do hereby certify:

1. I am an independent geological consultant working for P&E Mining Consultants Inc. located at 201 Country Court Blvd, Suite 304, Brampton, Ontario, Canada L6W 4L2.
2. I graduated with a Bachelor of Science degree in Geology from New Mexico State University in 1987. I obtained a Graduate Diploma in Engineering (Mining) in 1997 from the University of the Witwatersrand and a Master of Science in Engineering (Civil) from the University of the Witwatersrand in 2005.
3. I am a registered member of the Association of Professional Engineers and Geoscientists of British Columbia as a Professional Geoscientist (171602) and the Society for Mining, Metallurgy and Exploration as a Registered Member (#4152172).
4. My relevant work experience includes:
    - Underground Mine Geologist, Freegold Mine, AAC 1987-1995
    - Mineral Resource Manager, Vaal Reefs Mine, Anglogold 1995-1997
    - Resident Geologist, Venetia Mine, De Beers 1997-2000
    - Chief Geologist, De Beers Consolidated Mines 2000-2004
    - Consulting Geologist 2004-2008
    - P&E Mining Consultants Inc. – Associate Geologist 2008-Present
    - Participation and author of several NI 43-101 Technical Reports.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am responsible for the co-authoring Section 14. I am also responsible for the relevant portions of Sections 1 and 25 to 27 of the Technical Report.

8. I personally did not visit the property that is the subject to the Technical Report.
9. I have had prior involvement with the property that is the subject of the Technical Report.
    - *"Technical Report and Updated Mineral Resource Estimate of the Zgounder Silver Project, Kingdom of Morocco",* Prepared for Aya Gold & Silver Inc., dated April 30, 2021 (with an effective date of March 1, 2021), prepared by P&E Mining Consultants Inc. Report # 395*.*
    - *"Technical Report and Updated Mineral Resource Estimate of the Zgounder Silver Project, Kingdom of Morocco",* Prepared for Aya Gold & Silver Inc., dated January 28, 2022 (with an effective date of December 13, 2021), prepared by P&E Mining Consultants Inc. Report # 415.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 16 day of June, 2022

""Original Signed and Sealed on file""
Fred H. Brown, P.Geo.
Associate Geologist
P&E Mining Consultants Inc.

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I, Jarita Barry*, P. Geo.,* do hereby certify:

1. I am an independent geological consultant working for P&E Mining Consultants Inc. located at 201 Country Court Blvd, Suite 304, Brampton, Ontario, Canada L6W 4L2.
2. I am a graduate of of RMIT University of Melbourne, Victoria, Australia, with a B.Sc. in Applied Geology. I have worked as a geologist for over 15 years since obtaining my B.Sc. degree.
3. I am a registered member of I am a geological consultant currently licensed by Engineers and Geoscientists British Columbia (License No. 40875), Professional Engineers and Geoscientists Newfoundland & Labrador (License No. 08399) and Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (License No. L3874). I am also a member of the Australasian Institute of Mining and Metallurgy of Australia (Member No. 305397);
4. My relevant work experience includes: Brief summary of relevant experience.

• Geologist, Foran Mining Corp.	2004
• Geologist, Aurelian Resources Inc.	2004
• Geologist, Linear Gold Corp.	2005-2006
• Geologist, Búscore Consulting	2006-2007
• Consulting Geologist (AusIMM)	2008-2014
• Consulting Geologist, P.Geo. (APEGBC/AusIMM)	2014-Present
• Participation and author of several NI 43-101 Technical Reports.	

5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am responsible for the preparation of Section 11 and co-authoring Section 12 of the Technical Report. I am also responsible for the relevant portions of Sections 1 and 25 to 27 of the Technical Report.

8. I personally did not visit the property that is the subject to the Technical Report

9. I have had prior involvement with the property that is the subject of the Technical Report.

    - *"Technical Report and Updated Mineral Resource Estimate of the Zgounder Silver Project, Kingdom of Morocco",* Prepared for Aya Gold & Silver Inc., dated April 30, 2021 (with an effective date of March 1, 2021), prepared by P&E Mining Consultants Inc. Report # 395.
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10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.

11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 16 day of June, 2022

""Original Signed and Sealed on file""
Jarita Barry, P.Geo.
Associate Geologist
P&E Mining Consultants Inc.

P&E MINING CONSULTANTS INC.
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I, Eugene Purtich*, P. Eng., FEC, CET*, do hereby certify:

1. I am an independent mining consultant and President of P&E Mining Consultants Inc. located at 201 Country Court Blvd, Suite 304, Brampton, Ontario, Canada L6W 4L2.
2. I am a graduate of The Haileybury School of Mines, with a Technologist Diploma in Mining, as well as obtaining an additional year of undergraduate education in Mine Engineering at Queen's University. In addition, I have also met the Professional Engineers of Ontario Academic Requirement Committee's Examination requirement for a Bachelor's degree in Engineering Equivalency.
3. I am a registered member of I am a mining consultant currently licensed by the: Professional Engineers and Geoscientists New Brunswick (License No. 4778); Professional Engineers, Geoscientists Newfoundland and Labrador (License No. 5998); Association of Professional Engineers and Geoscientists Saskatchewan (License No. 16216); Ontario Association of Certified Engineering Technicians and Technologists (License No. 45252); Professional Engineers of Ontario (License No. 100014010); Association of Professional Engineers and Geoscientists of British Columbia (License No. 42912); and Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (No. L3877). I am also a member of the National Canadian Institute of Mining and Metallurgy.
4. My relevant work experience includes:
   - Mining Technologist - H.B.M.& S. and Inco Ltd., 1978-1980
   - Open Pit Mine Engineer – Cassiar Asbestos/Brinco Ltd., 1981-1983
   - Pit Engineer/Drill & Blast Supervisor – Detour Lake Mine, 1984-1986
   - Self-Employed Mining Consultant – Timmins Area, 1987-1988
   - Mine Designer/Resource Estimator – Dynatec/CMD/Bharti, 1989-1995
   - Self-Employed Mining Consultant/Resource-Reserve Estimator, 1995-2004
   - President – P&E Mining Consultants Inc, 2004-Present
   - Participation and author of several NI 43-101 Technical Reports.

5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.

6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.

7. I am responsible for the co-authoring Section 14 of the Technical Report. I am also responsible for the relevant portions of Sections 1 and 25 to 27 of the Technical Report.

8. I personally or did not visit) the property that is the subject to the Technical Report.

9. I have had prior involvement with the property that is the subject of the Technical Report.
   - *"Technical Report and Updated Mineral Resource Estimate of the Zgounder Silver Project, Kingdom of Morocco",* Prepared for Aya Gold & Silver Inc., dated April 30, 2021 (with an effective date of March 1, 2021), prepared by P&E Mining Consultants Inc. Report # 395*.*
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10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.

11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 16 day of June, 2022

""Original Signed and Sealed on file""

Eugene Puritch, P.Eng., FEC, CET
President
P&E Mining Consultants Inc.

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I, Antoine R. Yassa*, P. Geo*, do hereby certify:

1. I am an independent senior geological consultant with P&E Mining Consultants Inc., located at 201 Country Court Blvd, Suite 304, Brampton, Ontario, Canada L6W 4L2.
2. I am a graduate of Ottawa University at Ottawa, Ontario with a B.Sc. (HONS) in Geological Sciences (1977) with continuous experience as a geologist since 1979.
3. I am a registered member of the Order of Geologists of Québec (License No 224) and by the Association of Professional Geoscientists of Ontario (License No 1890).
4. My relevant work experience includes:
   - Minex Geologist (Val-d'Or), 3-D Modeling (Timmins), Placer Dome 1993-1995
   - Database Manager, Senior Geologist, West Africa, PDX, 1996-1998
   - Senior Geologist, Database Manager, McWatters Mine 1998-2000
   - Database Manager, Gemcom Modeling & Resources Evaluation (Kiena Mine) 2001-2003
   - Database Manager & Resources Evaluation at Julietta Mine, Bema Gold Corp. 2003-2006
   - Consulting Geologist 2006-present
   - Participation and author of several NI 43-101 Technical Reports.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am responsible for the preparation of Sections 9 and 10 and co-authoring of Sections 12 and 14 of the Technical Report. I am also responsible for the relevant portions of Sections 1 and 25 to 27 of the Technical Report.

8. I personally visited property that is the subject to the Technical Report on November 19 to 21, 2021 and January 11 to 13, 2021.

9. I have had prior involvement with the property that is the subject of the Technical Report.

   - *"Technical Report and Updated Mineral Resource Estimate of the Zgounder Silver Project, Kingdom of Morocco",* Prepared for Aya Gold & Silver Inc., dated April 30, 2021 (with an effective date of March 1, 2021), prepared by P&E Mining Consultants Inc. Report # 395*.*
   - *"Technical Report and Updated Mineral Resource Estimate of the Zgounder Silver Project, Kingdom of Morocco",* Prepared for Aya Gold & Silver Inc., dated January 28, 2022 (with an effective date of December 13, 2021), prepared by P&E Mining Consultants Inc. Report # 415.

10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.

11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 16 day of June, 2022

""Original Signed and Sealed on file""
Antoine Yassa, P.Geo.
Senior Associate Geologist
P&E Mining Consultants Inc.

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CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*NI 43-101 Technical Report – Feasibility Study for the Zgounder Expansion Project, Kingdom of Morocco"* with an effective date of December 13, 2021 (the "**Technical Report**"), originally issued on March 31, 2022 and amended on June 16, 2022, prepared for Aya Gold and Silver Inc. ("**Aya**" or the "**Corporation**").

I, *Daniel Morrison, P. Eng.*, do hereby certify:

1. I am a Principal Process Engineer, with DRA Global Limited with an office at 20 Queen Street West, 29th Floor, Toronto, Ontario, Canada.
2. I am a graduate from the University of Stellenbosch, South Africa with a Bachelor of Chemical Engineering in 1990.
3. I am a registered member of the Professional Engineers of Ontario (#100134360).
4. I have worked continuously as a Metallurgical and Process Engineer for more than 28 years since my graduation.
5. My relevant work experience includes:
    - Review and report on mineral processing and metallurgical operations and projects around the world for due diligence and regulatory requirements;
    - Engineering study from PEA to EPCM projects work on many minerals processing and metallurgical and hydrometallurgical projects around the world;
    - Design and supervision and implementation of process flow programs;
    - Hands-on experience in cyanidation of gold and silver ores around the world;
    - Operational experience in operations management and operational support positions in metallurgical and hydrometallurgical operations in Africa and in the Americas.
    - Participation and author of several NI 43-101 Technical Reports.
6. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, 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 the Company applying all the tests in Section 1.5 of NI 43-101.

8. I am responsible for the preparation of Sections 13 and 17. I am also responsible for the relevant portions of Sections 1, 21.2, and 25 to 27 of the Technical Report.

9. I did not visit the property that is the subject to the Technical Report.

10. I have had prior involvement with the property that is the subject of the Technical Report.

    - "*Technical Report and Updated Mineral Resource Estimate of the Zgounder Silver Project, Kingdom of Morocco*", Prepared for Aya Gold & Silver Inc., dated January 28, 2022 (with an effective date of December 13, 2021), prepared by P&E Mining Consultants Inc. Report # 415.

11. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.

12. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 16 day of June 2022

"Original Signed and Sealed on file"

Daniel Morrison, P.Eng.
Principal Process Engineer
DRA Global Limited

555 René-Lévesque Blvd West / 6th floor / Montréal / Quebec / Canada / H2Z 1B1
T +1 514 288-5211 / **E** info@draglobal.com / https://www.draglobal.com/

CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*NI 43-101 Technical Report – Feasibility Study for the Zgounder Expansion Project, Kingdom of Morocco"* with an effective date of December 13, 2021 (the "**Technical Report**"), originally issued on March 31, 2022 and amended on June 16, 2022, prepared for Aya Gold and Silver Inc. ("**Aya**" or the "**Corporation**").

I, *André-François Gravel, P. Eng., PMP.,* do hereby certify:

1. I am a Senior Mining Engineer with DRA Global. Limited, with an office at 555 René-Lévesque Blvd West, Montreal, Quebec, Canada.
2. I am a graduate of École Polytechnique de Montréal, Montreal, Quebec, Canada in 2000 with a bachelor degree in Mining Engineering.
3. I am registered as a Professional Engineer in the Province of Quebec (Reg. #123135).
4. I have worked as an Engineer in the Mining & Metals industry continuously since my graduation from university.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("N 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
6. My relevant work experience includes:
   - Design, scheduling, cost estimation and Mineral Reserve estimation for several underground and open pit studies similar to Zgounder in Canada, the USA, South America, Asia and West Africa.
   - Technical assistance in mine design and scheduling for mine operations in Canada, the US and Morocco.
   - Participation and author of several NI 43-101 Technical reports.
7. I am independent of the Company applying all the tests in Section 1.5 of NI 43-101.
8. I have participated in the preparation of this Technical Report and am responsible for Sections 15, 16.4, and 16.5 except for 15.3, 15.4, and 16.4.1. I am also responsible for the relevant portions of Sections 1, 21.2, and 25 to 27 of the Technical Report.

9. I personally visited the property that is the subject to the Technical Report on May 25 and 26, 2021.

10. I have had no prior involvement with the property that is the subject of the Technical Report.

11. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.

12. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 16 day of June 2022

"Original Signed and Sealed on file"

André-François Gravel, P.Eng., PMP
Senior Mining Engineer
DRA Global Limited

555 René-Lévesque Blvd West / 6th floor / Montréal / Quebec / Canada / H2Z 1B1
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CERTIFICATE OF QUALIFIED PERSON

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I, *Claude Bisaillon, P. Eng.*, do hereby certify:

1. I am Senior Geotechnical Engineer with DRA Global Limited located at 555 René Lévesque West, 6th Floor, Montreal, Quebec Canada H2Z 1B1.
2. I am a graduate from *Concordia University in Montreal in 1991 with a B.Sc. in geology and from the Université Laval in Quebec City in 1996 with a B.Ing. in geological engineering.*
3. I am a registered member of "Ordre des Ingénieurs du Québec" (#116407). I am a Member of the Canadian Institute of Mining, Metallurgy and Petroleum.
4. I have worked as an engineer continuously since graduation from University in 1996. My relevant work experience includes:
    - Over 24 years of consulting in the field of Mineral Resource estimation, orebody modelling, mineral resource auditing and geotechnical engineering in Canada, the USA, Asia, and South America.
    - Participation and author of several NI 43-101 Technical Reports.
    - QP Review, audits, due diligence, interpretation of geoscientific data for several projects.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am responsible for the preparation of Section 16.2.2. I am also responsible for the relevant portions of Sections 1, 25 to 27 of the Technical Report.

8. I personally visited the property that is the subject to the Technical Report on January 18 and 19, 2011 as a consultant for SGS Canada.

9. I have had prior involvement with the property that is the subject of the Technical Report.

   - *"Due Diligence of Azegour and Zgounder Properties, Kingdom of Morocco",* Prepared for Maya Gold & Silver Inc., dated February 22, 2011 (Revised on April 06, 2011), prepared by SGS Canada. An internal report prepared to support the potential acquisition of the Zgounder mine.

10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.

11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 16 day of June 2022

"Original Signed and Sealed on file"
Claude Bisaillon, P. Eng.
Senior Geotechnical Engineer
DRA Global Limited

920 Princess St., Suite 310, Kingston, Ontario, Canada K7L 1H1
T: 613-507-7575 F: 613-507-7576

Certificate of Qualified Person

To accompany the Report entitled "NI 43-101 Technical Report – Feasibility Study for the Zgounder Expansion Project, Kingdom of Morocco" filed with an original date of March 31, 2022 and an amended date of June 16, 2022, with an effective date of December 13, 2021 (the "Technical Report"), prepared for Aya Gold and Silver Inc. ("Aya" or the "Corporation").

I, Kathy Kalenchuk, P. Eng., PE, do hereby certify:

1. I am President and Principal Consultant, with RockEng Inc. located at 920 Princess St., Suite 310, Kingston, Ontario, Canada.
2. I am a graduate of the University of Alberta, Edmonton, Alberta, Canada in 2004 with a Bachelors degree in Mining Engineering. I am a graduate of the Queen's University, Kingston, Ontario, Canada in 2007 with a Master's degree in Geomechanical Engineering and 2010 with a Ph.D. in Geomechanical Engineering.
3. I am registered as a Professional Engineer in the Province of Ontario (Reg. #100164661), the Province of British Columbia (Reg. # 189685), and the State of Montana (Reg. #PEL-PE-LIC-59819). I have worked as a Geotechnical Engineer for a total of 11 years continuously since my graduation.
4. My relevant work experience includes:
   - Due Diligence Reviews: Review of mine plans and designs for investors as well as third-party review for technical quality and validity of geomechanical studies.
   - Underground Mine Design: All geomechanical components of underground mine design such as, mining method, mine sequencing, stope sizing, excavation stability, static and dynamic ground support, hazard identification and risk management, infrastructure siting, construction method evaluations.
   - Operational Geomechanics and Ground Control: Routine site support for operating mines, including audits and inspections, incident investigations, regular design reviews, support evaluations, instrumentation data interpretation, development and review of Ground Control Management Plans, and provision of ground control training to site personnel.
   - Site Characterization: Core logging, geotechnical mapping, damage mapping, data analysis, domain delineation, material parameterization, in situ stress.
   - Participation and author of several NI 43-101 Technical Reports.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.

7. I am responsible for the preparation of Section 16.4.1. I am also responsible for the relevant portions of Sections 1 and 26 of the Technical Report.
8. I have not personally visited the property that is the subject to the Technical Report.
9. I have had prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 16th day of June 2022.

"Original Signed and Sealed on file"

Kathy Kalenchuk, Ph.D., P.Eng., PE
President & Principal Consultant

16th of June 2022

Howden Ventsim
1381 Rue Hocquart
St-Bruno-de-Montarville,, Québec.
Canada
J3V 6B5

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ventsim@howden.com
www.howden.com

CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*NI 43-101 Technical Report – Feasibility Study for the Zgounder Expansion Project, Kingdom of Morocco"* filed with an original date of March 31, 2022 and an amended date of June 16, 2022, with an effective date of December 13, 2021 (the "**Technical Report**"), prepared for Aya Gold and Silver Inc. ("**Aya**" or the "**Corporation**").

I, *Hugo Dello Sbarba, P.Eng, M.Sc.*, do hereby certify:

I am a Mine Ventilation Engineer and the Director of Ventsim with Howden Canada Inc. located at 1381 Rue Hocquart, St-Bruno-de-Montarville, Quebec J3V 6B5, CANADA.

1. I am a graduate from *Concordia University, Mtl, Qc, Canada in Mechanical Engineering with a specialty in HVAC graduated in 2008 and a Master's of Science Degree from Laval University, Quebec, Qc, Canada in Mining Engineering graduated in 2012.*
2. I am a registered member of the Professional Engineers of Ontario (100197915).
3. My relevant work experience includes:
    - A total of 12 years of experience in Mine Ventilation since my graduation
    - 3 years of experience working at an operating underground mine.
    - Participation of several NI 43-101 Technical Reports

4. I have read the definition of “qualified person” set out in the NI 43-101 – Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
5. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
6. I am responsible for the preparation of Section 16.6. I am also responsible for the relevant portions of Sections 1, 25 to 27 of the Technical Report.
7. I personally did not visit the property that is the subject to the Technical Report.
8. I have had no prior involvement (if any previous involvement, provide explanation) with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
10. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 16 day of June, 2022

““Original Signed and Sealed on file””
Hugo Dello Sbarba, P. Eng. M. Sc.
Director of Ventsim
Howden

CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*NI 43-101 Technical Report – Feasibility Study for the Zgounder Expansion Project, Kingdom of Morocco"* filed with an original date of March 31, 2022 and an amended date of June 16, 2022, with an effective date of December 13, 2021 (the "**Technical Report**"), prepared for Aya Gold and Silver Inc. ("**Aya**" or the "**Corporation**").

I, *Philippe Rio Roberge, P. Eng., PMP.*, do hereby certify:

1. I am the director of mining services with Englobe located at 100, rue Jean-Coutu, bureau 101, Varennes, Quebec J3X 0E1.
2. I am a graduate from of the University of Sherbrooke, QC., Canada, Bachelor of Civil Engineering 2006.
3. I am a registered member of "Ordre des Ingénieurs du Québec", license 142781.
4. My relevant work experience includes more than 15 years of civil, geotechnical, mining and environmental engineering. I have participated in mine waste and water management, construction and operation projects as well as numerous feasibility level economic studies.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am responsible for the preparation of Section 18.5 and 18.7.2 to 18.7.4. I am also responsible for the relevant portions of Sections 1, 25 to 27 of the Technical Report.
8. I did not visit the property that is the subject to the Technical Report.
9. I have had no prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 16th day of June, 2022

““Original Signed and Sealed on file””
Philippe Rio Roberge, P. Eng., PMP.
Director of mining services
Englobe

7, rue Zemrane, Avenue mohamed VI - Rabat
Tel : +212 537 658 330 Fax : +212 537 635 369
E-Mail : gcim@gcim.ma

CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*NI 43-101 Technical Report – Feasibility Study for the Zgounder Expansion Project, Kingdom of Morocco"* filed with an original date of March 31, 2022 and an amended date of June 16, 2022, with an effective date of December 13, 2021 (the "**Technical Report**"), prepared for Aya Gold and Silver Inc. ("**Aya**" or the "**Corporation**").

I, Richard Barbeau, *Ing, P. Eng. M.Sc.,* do hereby certify:

1. I am a Mining, Industrial and Mechanical Engineer and a Qualified Person with GCIM, located 7, rue Zemrane, Avenue Mohamed VI in Rabat, Morocco.
2. Graduated of Mining Engineer Degree in 1981 – Laval University, Mining Engineer Department, education from 1977 to 1981, Quebec, Canada;
3. Graduated of Industrial Engineer DESS Degree in 2004 – Laval University, Mechanical Engineer Department, education from 2003 to 2004, Quebec, Canada;
4. Graduated of Mechanical Engineer Master Degree in 2009 – Laval University, Mechanical Engineer Department, education from 2005 to 2009, Quebec, Canada
5. Member of "Ordre des Ingénieurs du Québec (OIQ-36572)" and Professional Engineers of Ontario (PEO-100157701)".
6. My relevant work experience includes: Tailing Dam Master Degree in Tar Sands at Syncrude Canada, Tailings Management - State of Art - Technical Paper 2008, Tailing Storage Management – Technical Paper 2007; QP Balabag Pre-Feasibility Study (Philippines), QP True Gold BF, Tailing Dam construction and supervising of construction works at LaRonde Canada, Design and Construction of Water Storage Process Plant Reservoir at Karma BF, Leach Pad Design and Construction at Taparko BF, Tailing Dam Raising Construction Design and supervising of construction works at Taparko BF, Dam 4 Water Dam Raising and supervising of construction works at Taparko BF, West Pit 3/5 Rehabilitation and supervising of rehabilitation works at Taparko BF, Mine access roads design in mountains at CBG Guinea, Geotechnical engineer at QCM, Century Mining and Agrium Mine.
7. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of

7, rue Zemrane, Avenue mohamed VI - Rabat
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E-Mail : gcim@gcim.ma

Disclosure for Mineral Projects (“NI 43-101”) and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.

8. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
9. I am responsible for Sections 18.6 & 18.7.1 related to hydrology and Tailing Management Facility. I am also responsible for the relevant portions of Sections 1, 25 to 27 of the Technical Report.
10. I never visited the property that is the subject to the Technical Report.
11. I have had no prior involvement with the property that is the subject of the Technical Report.
12. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
13. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 16th day of June, 2022

“Original Signed and Sealed on file”
Richard Barbeau, Ing, P. Eng. M.Sc.
GCIM

EcoMine

CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*NI 43-101 Technical Report – Feasibility Study for the Zgounder Expansion Project, Kingdom of Morocco"* with an effective date of December 13, 2021 (the "**Technical Report**"), originally issued on March 31, 2022 and amended on June 16, 2022, prepared for Aya Gold and Silver Inc. ("**Aya**" or the "**Corporation**").

I, *Julie Gravel, P. Eng.*, do hereby certify:

1. I am an Environmental Engineer and President with Ecomine located at 996 Chemin Francisco, Rivière-Rouge Quebec J0T 1T0.
2. I am a graduate from a bachelor degree in Geological Engineering, *Ecole Polytechnique de Montréal*, Québec, Canada in 1989.
3. I am a registered member of *Ordre des ingénieurs du Quebec* (OIQ # 103517).
4. My relevant work experience includes:
    - Environmental advisor for various mining projects;
    - Mining Environmental Impact Assessment and surveillance and closure plan;
    - Federal and provincial permitting;
    - Participation and author of several NI 43-101 Technical Reports.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am responsible for the preparation of Section 20 except for Section 20.7. I am also responsible for the relevant portions of Sections 1, 25 to 27 of the Technical Report.
8. I personally did not visit the Property that is the subject to the Technical Report.
9. I have had no prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 16 day of June 2022

"Original Signed on file"
Julie Gravel, P. Eng.
President
Ecomine

CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*NI 43-101 Technical Report – Feasibility Study for the Zgounder Expansion Project, Kingdom of Morocco*" with an effective date of December 13, 2021 (the "**Technical Report**"), originally issued on March 31, 2022 and amended on June 16, 2022, prepared for Aya Gold and Silver Inc. ("**Aya**" or the "**Corporation**").

I, Stephen Coates, P. Eng., do hereby certify:

1. I am Vice-President of Normetal Consulting Inc. located at 1068 Chemin des Franciscains, Mont-Tremblant, Quebec, J8E 3M6.
2. I am a graduate from McGill University located in Montreal, Quebec, Canada with a bachelor's degree in Mining Engineering.
3. I am a registered member of l'Ordre des Ingénieurs du Québec (#5047905).
4. My relevant work experience includes 10 years in mining operations, technical study delivery, mine financing and strategic consulting.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43-101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am responsible for the preparation of Section 1.15 and 22 of the Technical Report.
8. I personally did not visit the property that is the subject to the Technical Report.
9. I have had no prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 16 day of June 2022

"Original Signed and Sealed on file"

Stephen Coates, P. Eng.
Vice-President
Normetal Consulting Inc.

555 René-Lévesque Blvd West / 6th floor / Montréal / Quebec / Canada / H2Z 1B1
T +1 514 288-5211 / **E** info@draglobal.com / https://www.draglobal.com/

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I, *Daniel M. Gagnon, P. Eng.,* do hereby certify:

1. I am Vice President Mining, Geology and General Manager Montréal with DRA Global Limited located at 555 René Lévesque West, 6th Floor, Montreal, Quebec Canada H2Z 1B1.
2. I am a graduate of École Polytechnique de Montréal, Montreal, Quebec, Canada in 1995 with a bachelor degree in Mining Engineering.
3. I am registered as a Professional Engineer in the Province of Quebec (Reg. #118521).
4. I have worked as a Mining Engineer for a total of 26 years continuously since my graduation. My relevant work experience includes:
    - Design, scheduling, cost estimation and Mineral Reserve estimation for several open pit studies similar to Zgounder Expansion Project in Canada, the US, South America, West Africa and Morocco.
    - Technical assistance in mine design and scheduling for mine operations in Canada, the US, and Morocco.
    - Participation and author of several NI 43-101 Technical Reports.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
6. I am independent of the issuer applying all the tests in Section 1.5 of NI 43-101.
7. I am responsible for the preparation of Sections 2, 3, 15.3 ,15.4, 15.6, 16.1 to 16.3, 16.7, 18.1 to 18.4, 19, 20.7, 21.1, and 24 with the exception of Sections 16.2.2. I am also responsible for the relevant portions of Sections 1, 21.2, and 25 to 27 of the Technical Report.

8. I personally did not visit the property that is the subject to the Technical Report.

9. I have had no prior involvement with the property that is the subject of the Technical Report.

10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.

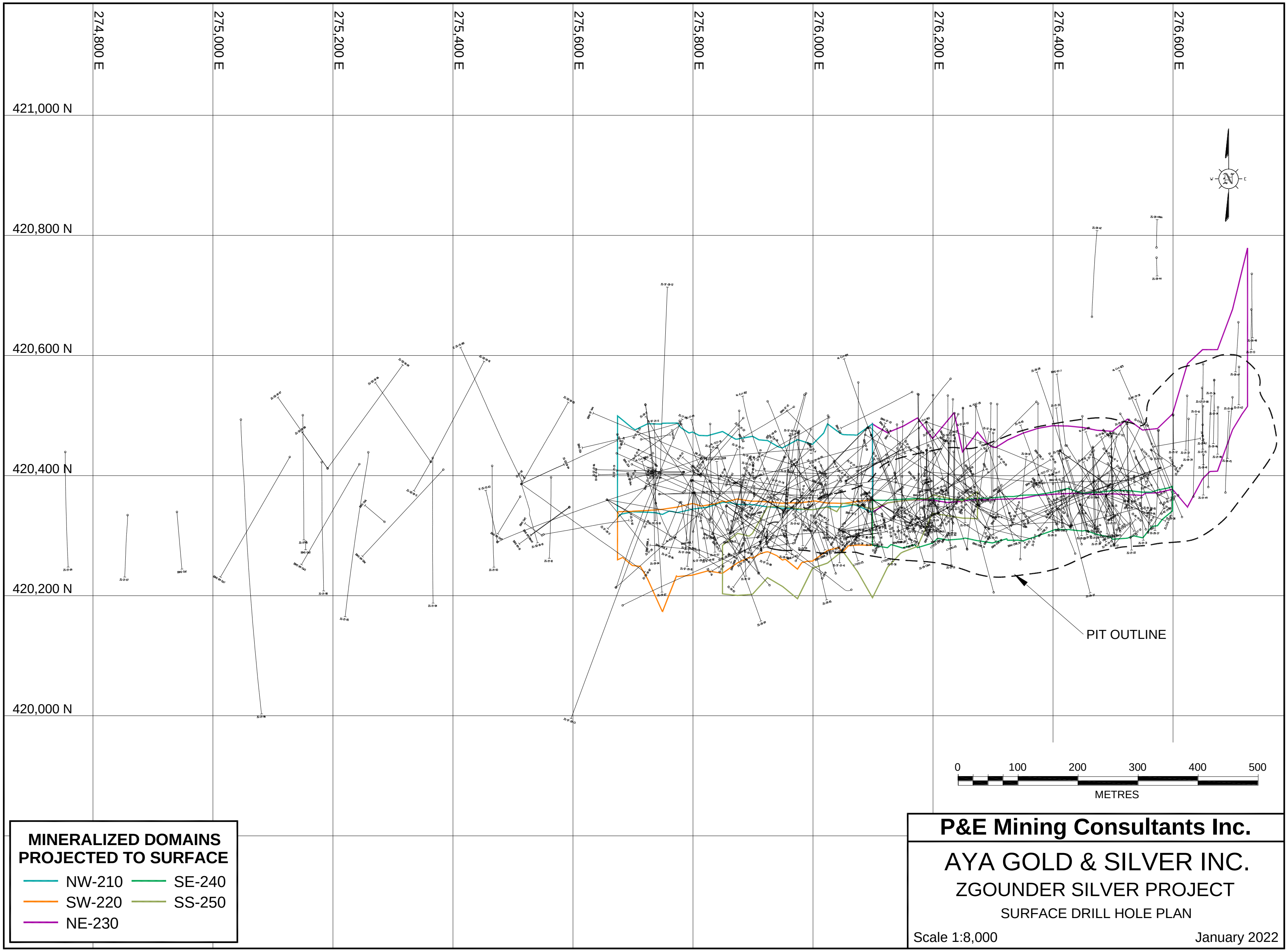
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 16 day of June 2022

"Original Signed and Sealed on file"

Daniel M. Gagnon, P. Eng.
VP Mining, Geology and GM Montréal
DRA Global Limited

Appendix A – Drill Hole Plan



P&E Mining Consultants Inc.

AYA GOLD & SILVER INC.

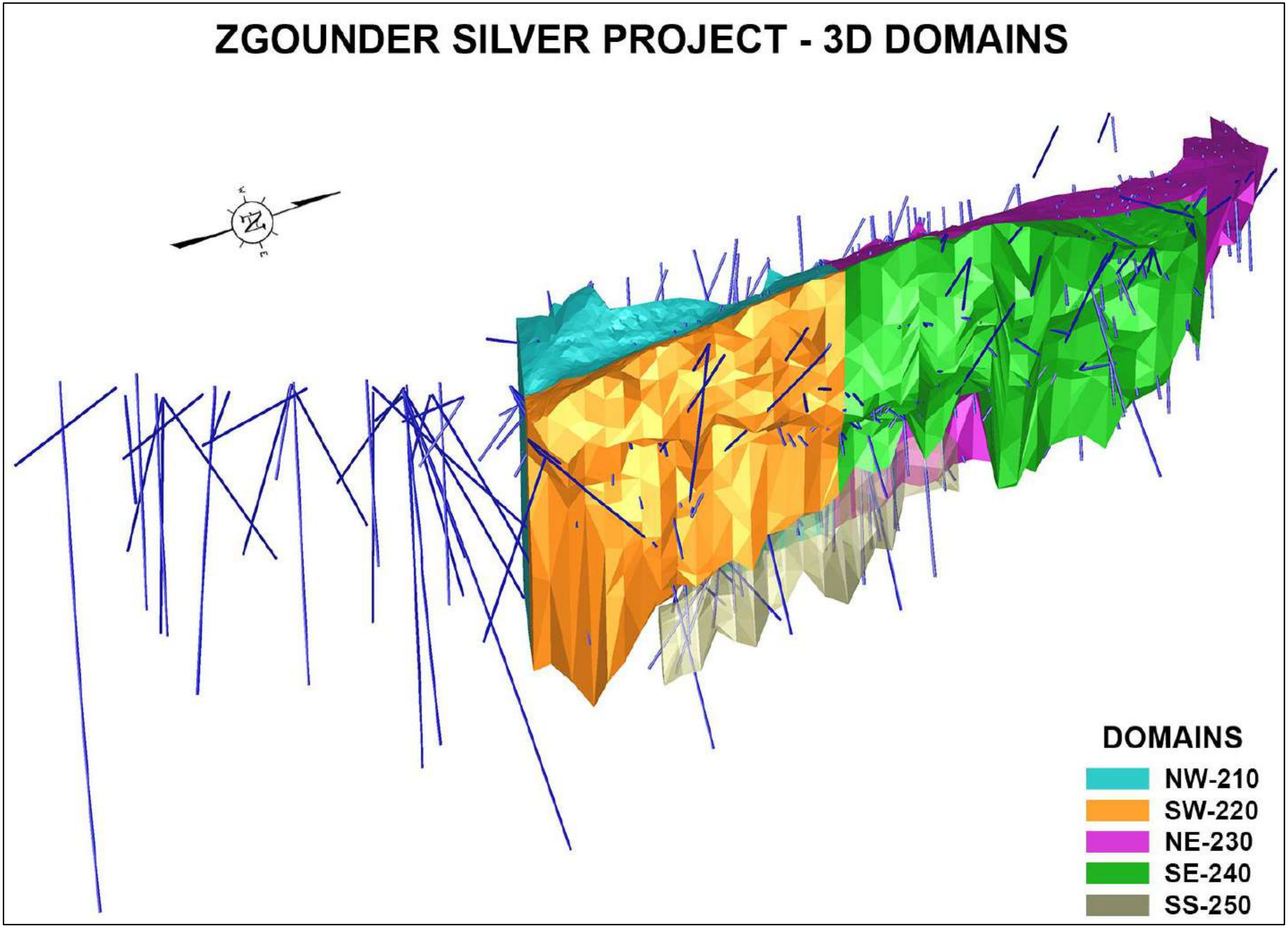
ZGOUNDER SILVER PROJECT

SURFACE DRILL HOLE PLAN

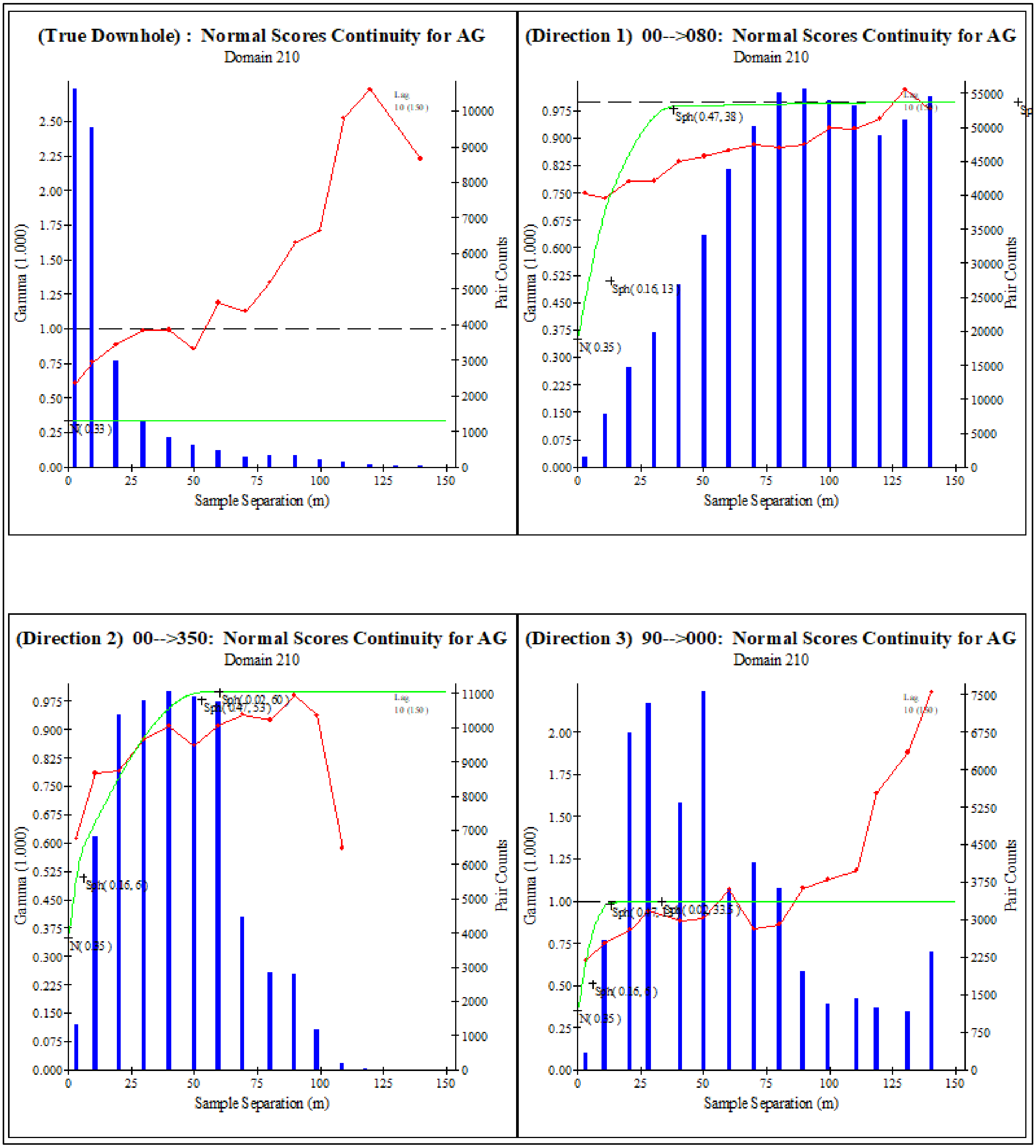
Scale 1:8,000

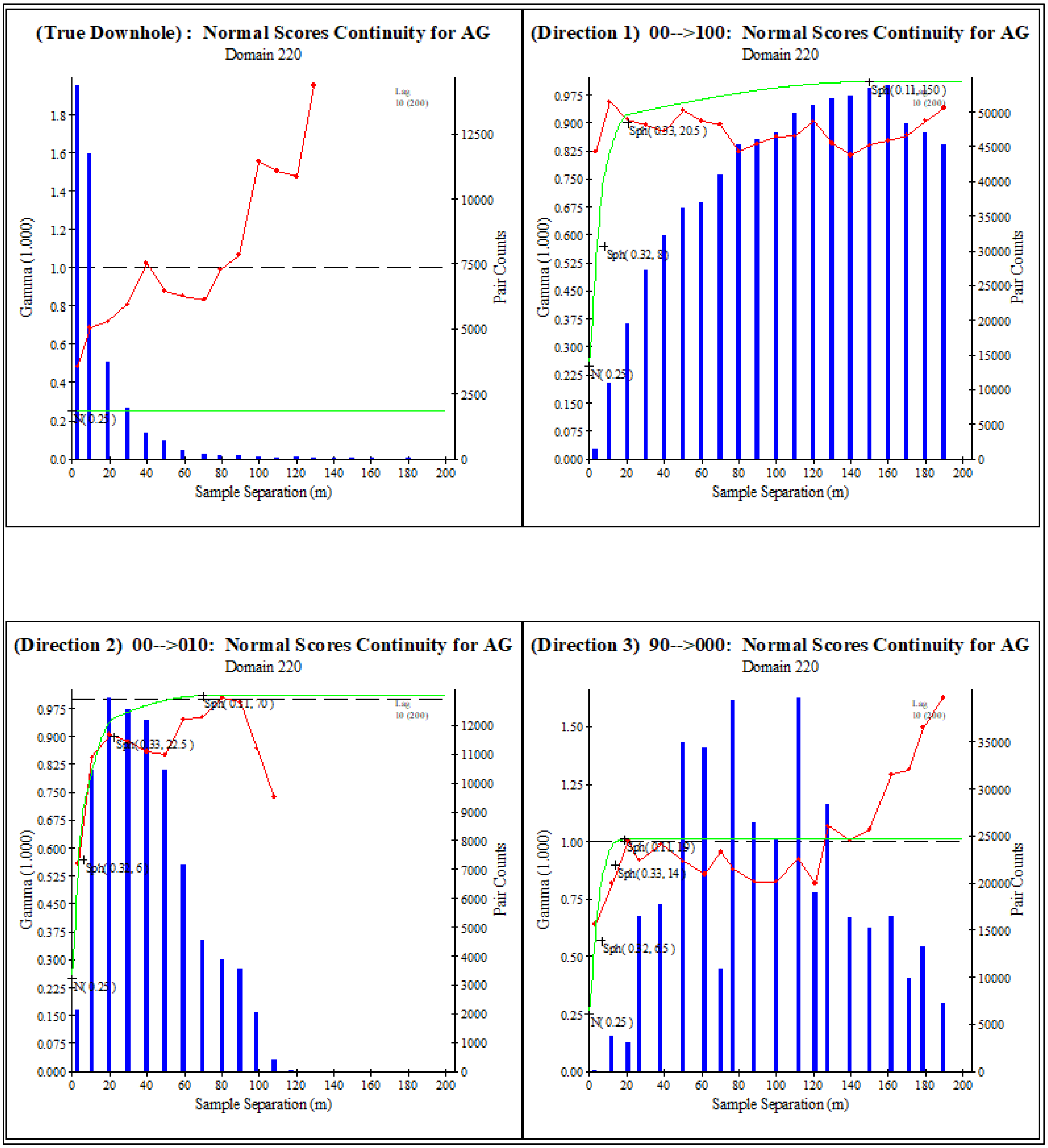
January 2022

Appendix B – 3-D Domains

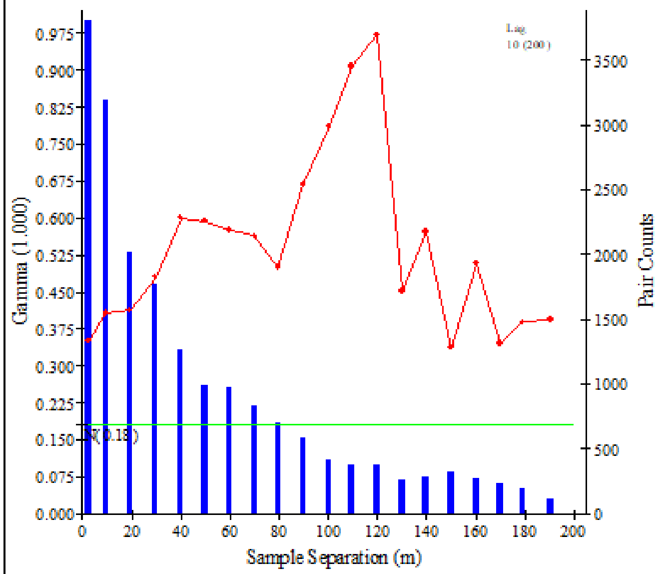

ZGOUNDER SILVER PROJECT - 3D DOMAINS
DOMAINS
NW-210
SW-220
NE-230
SE-240
SS-250

Appendix C – Variograms

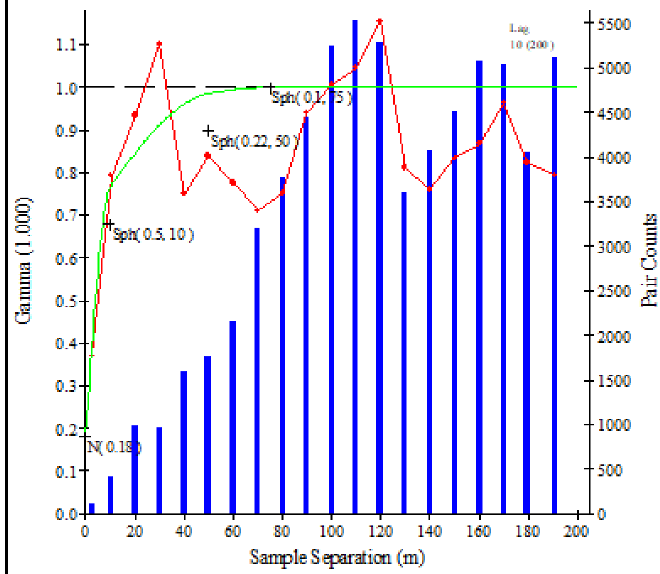

(True Downhole) : Normal Scores Continuity for AG
Domain 210
Gamma (1.000)
Pair Counts
Sample Separation (m)
N(0.33)
(Direction 1) 00-->080: Normal Scores Continuity for AG
Domain 210
Gamma (1.000)
Pair Counts
Sample Separation (m)
N(0.35)
Sph(0.16, 13)
Sph(0.47, 38)
(Direction 2) 00-->350: Normal Scores Continuity for AG
Domain 210
Gamma (1.000)
Pair Counts
Sample Separation (m)
N(0.35)
Sph(0.16, 6)
Sph(0.47, 53)
Sph(0.02, 60)
(Direction 3) 90-->000: Normal Scores Continuity for AG
Domain 210
Gamma (1.000)
Pair Counts
Sample Separation (m)
N(0.35)
Sph(0.16, 6)
Sph(0.02, 33.5)


(True Downhole) : Normal Scores Continuity for AG
Domain 220
Gamma (1.000)
Pair Counts
Sample Separation (m)
N(0.25)
(Direction 1) 00-->100: Normal Scores Continuity for AG
Domain 220
Gamma (1.000)
Pair Counts
Sample Separation (m)
N(0.25)
Sph(0.32, 8)
Sph(0.33, 20.5)
Sph(0.11, 150)
(Direction 2) 00-->010: Normal Scores Continuity for AG
Domain 220
Gamma (1.000)
Pair Counts
Sample Separation (m)
N(0.25)
Sph(0.31, 6)
Sph(0.33, 22.5)
Sph(0.11, 70)
(Direction 3) 90-->000: Normal Scores Continuity for AG
Domain 220
Gamma (1.000)
Pair Counts
Sample Separation (m)
N(0.25)
Sph(0.32, 6.5)
Sph(0.33, 14)
Sph(0.11, 19)

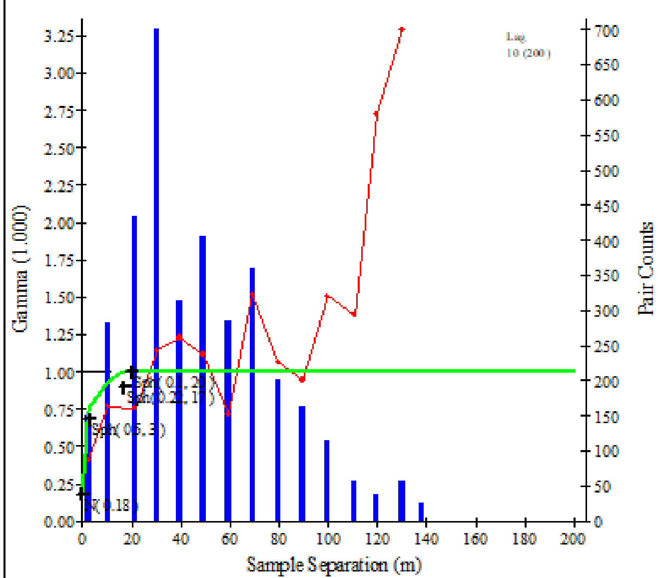
(True Downhole) : Normal Scores Continuity for AG
Domain 230



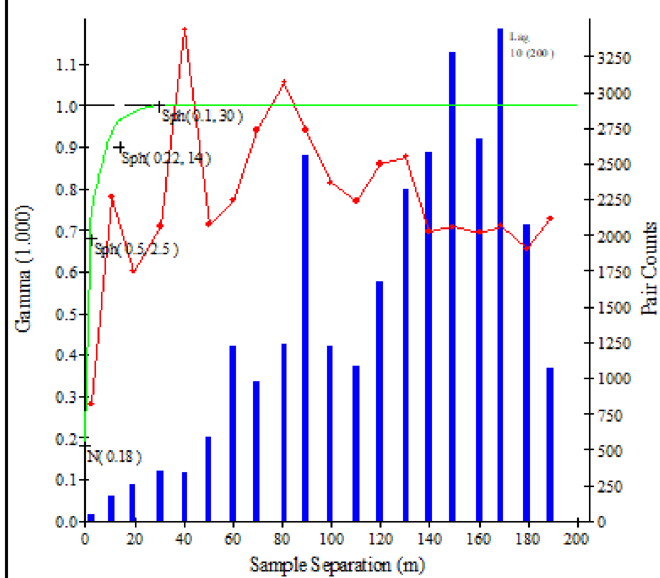
(Direction 1) 00-->090: Normal Scores Continuity for AG
Domain 230

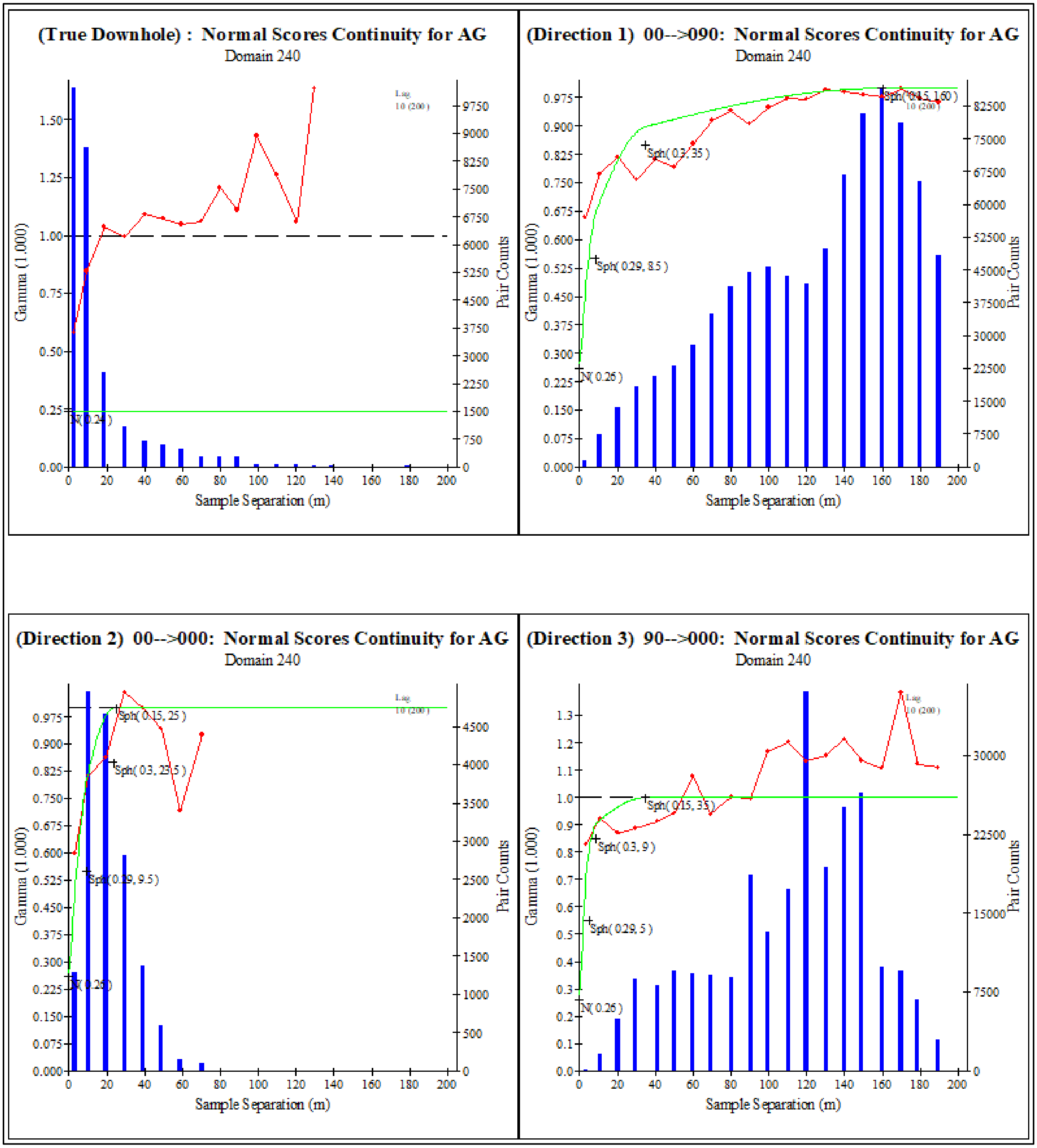


(Direction 2) 00-->000: Normal Scores Continuity for AG
Domain 230

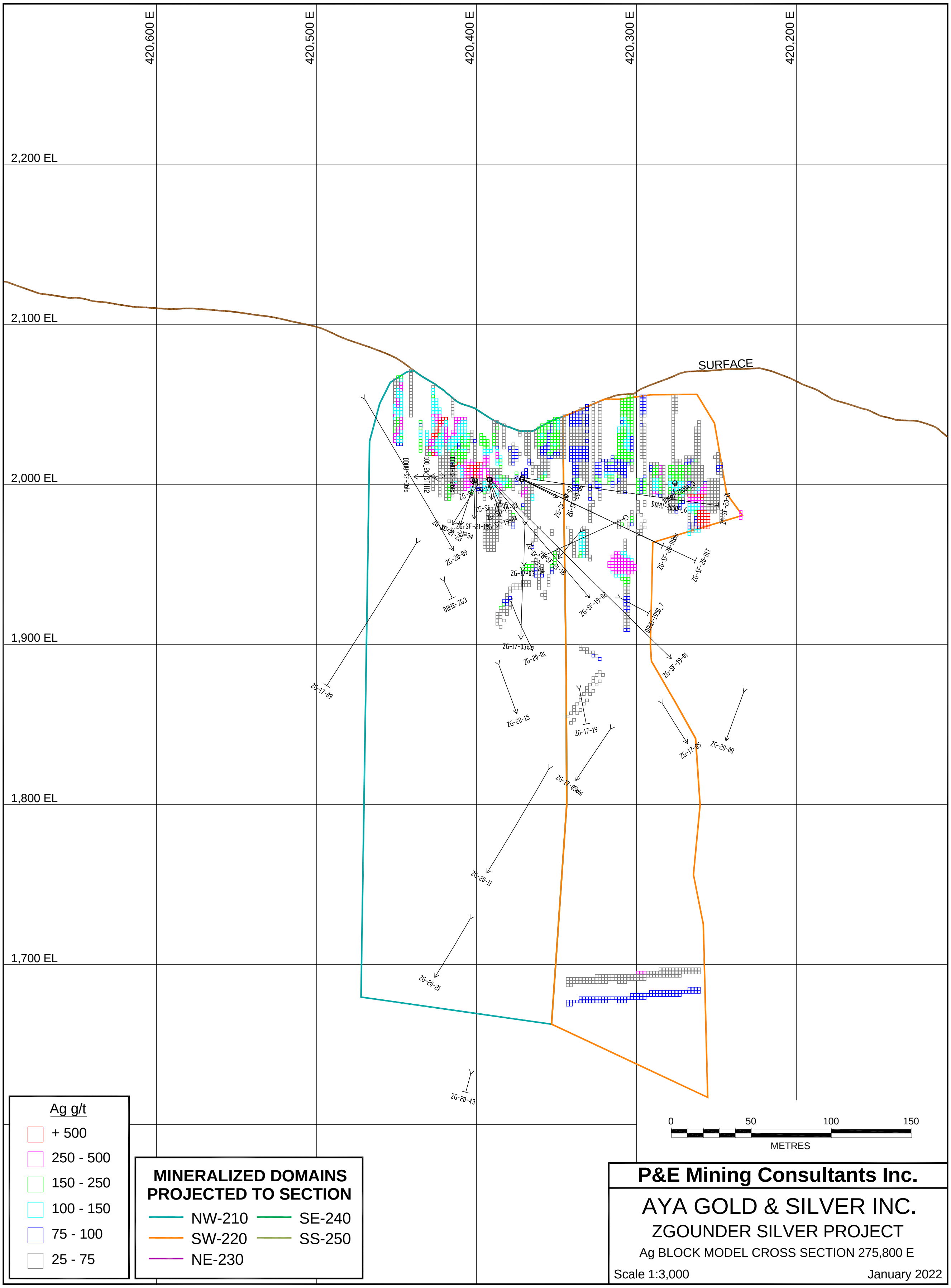


(Direction 3) 90-->000: Normal Scores Continuity for AG
Domain 230

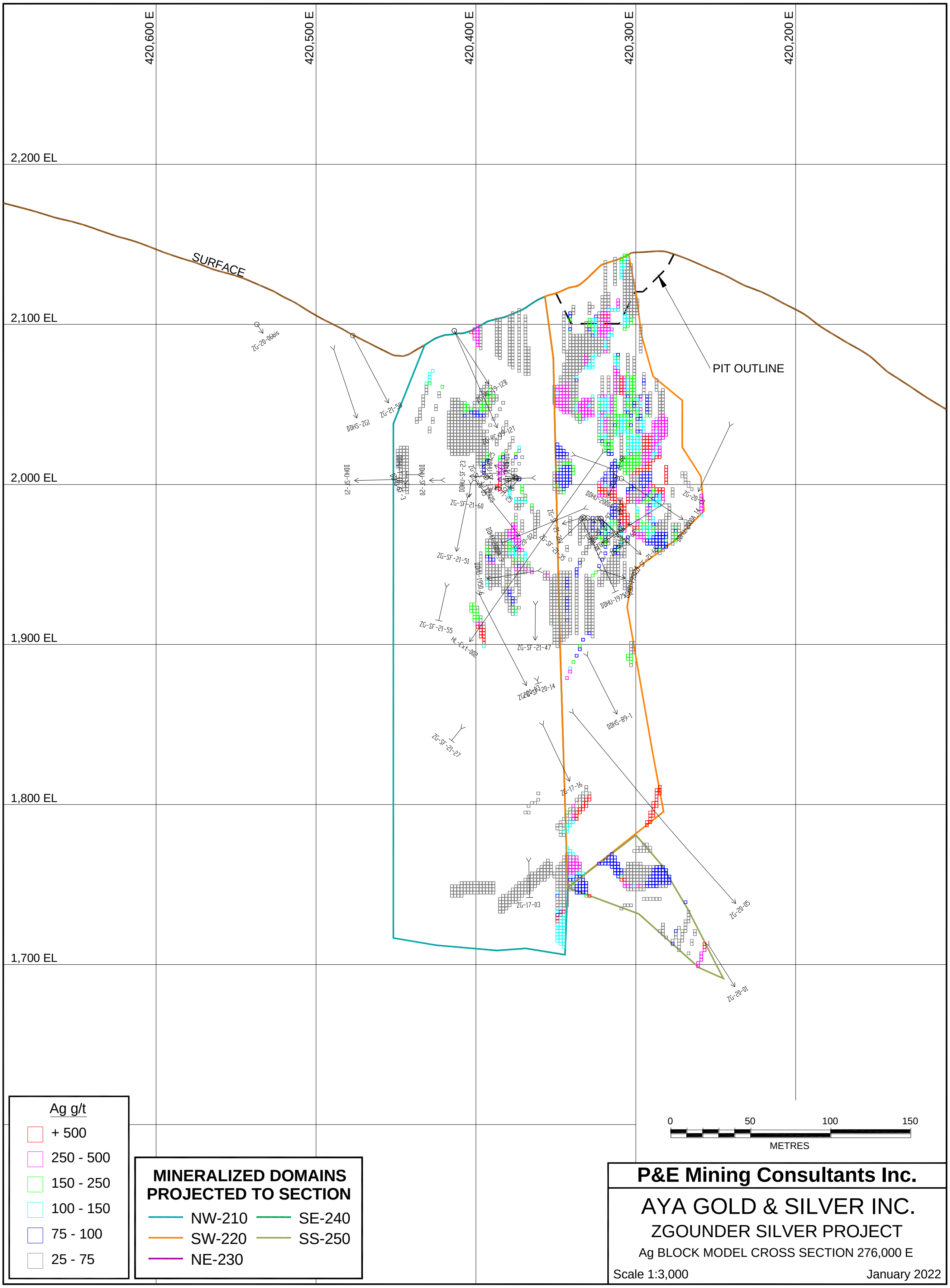


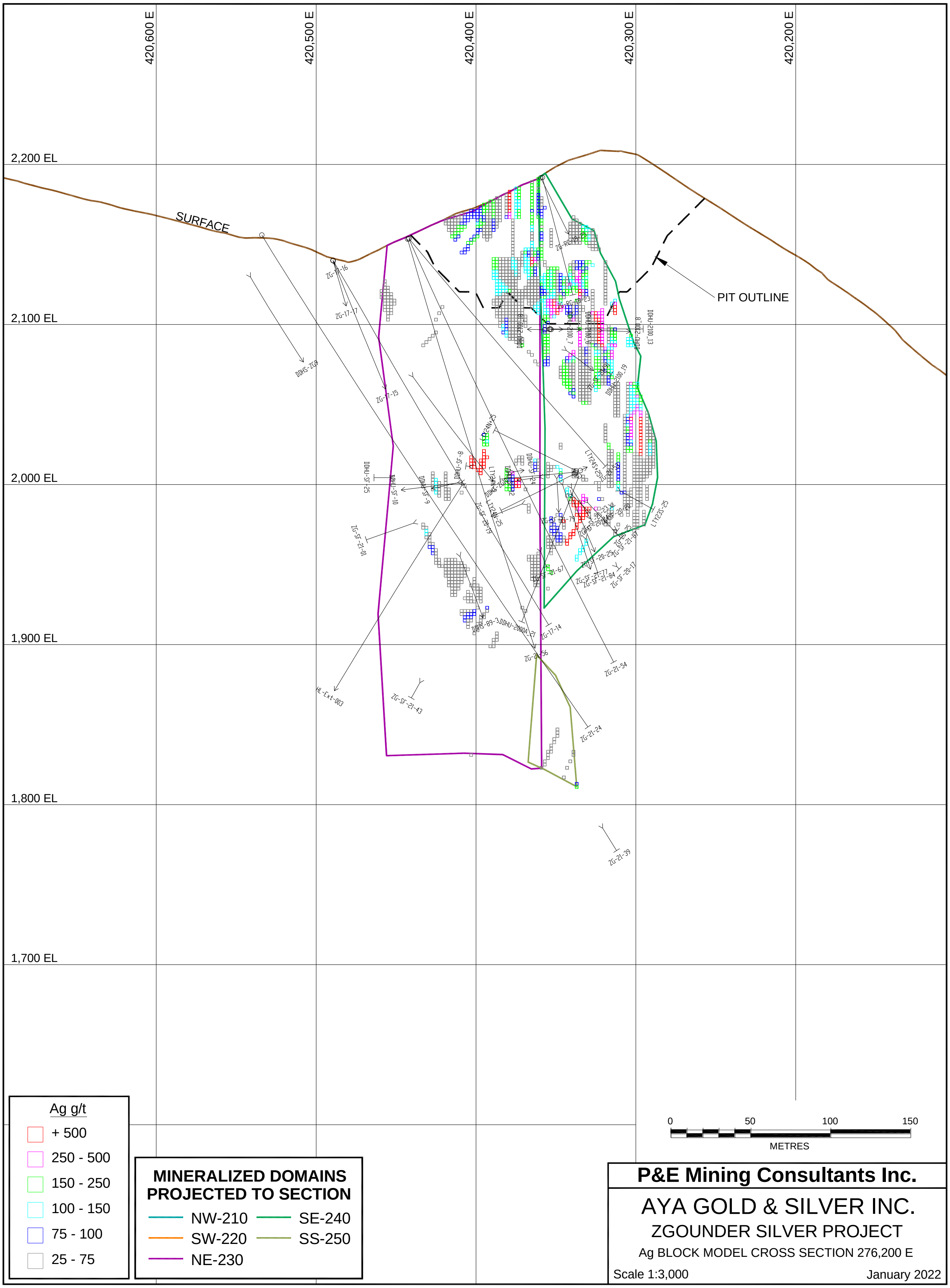

(True Downhole) : Normal Scores Continuity for AG
Domain 240
Gamma (1.000)
Pair Counts
Sample Separation (m)
N(0.24)
Lag
10 (200)
(Direction 1) 00-->090: Normal Scores Continuity for AG
Domain 240
Gamma (1.000)
Pair Counts
Sample Separation (m)
N(0.26)
Sph(0.29, 8.5)
Sph(0.3, 35)
Sph(0.15, 160)
(Direction 2) 00-->000: Normal Scores Continuity for AG
Domain 240
Gamma (1.000)
Pair Counts
Sample Separation (m)
N(0.26)
Sph(0.29, 9.5)
Sph(0.3, 23.5)
Sph(0.15, 25)
(Direction 3) 90-->000: Normal Scores Continuity for AG
Domain 240
Gamma (1.000)
Pair Counts
Sample Separation (m)
N(0.26)
Sph(0.29, 5)
Sph(0.3, 9)
Sph(0.15, 35)

Appendix D –
Ag Block Model Cross Sections and Plans

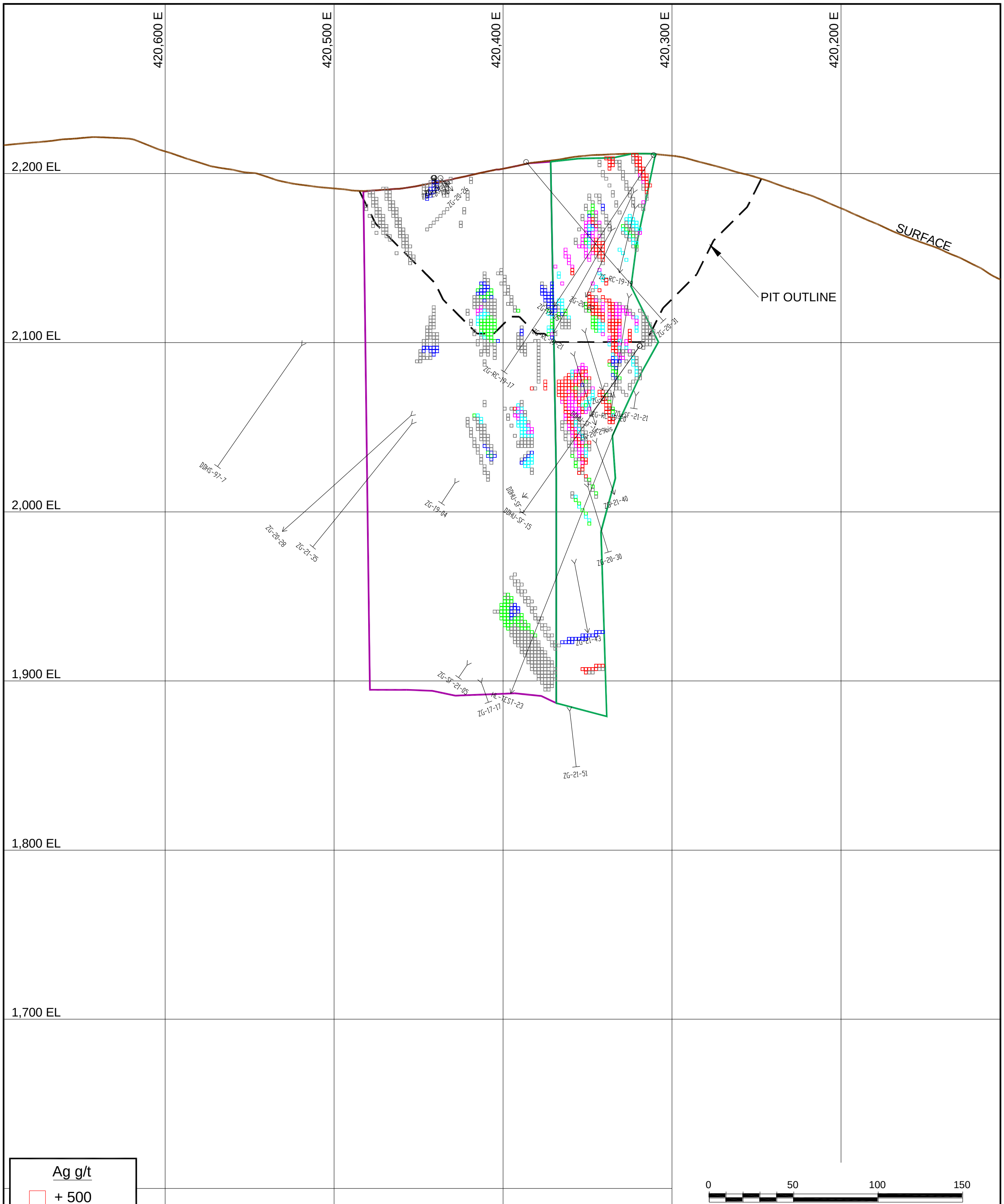


420,600 E
420,500 E
420,400 E
420,300 E
420,200 E
2,200 EL
2,100 EL
2,000 EL
1,900 EL
1,800 EL
1,700 EL
SURFACE
Ag g/t
+ 500
250 - 500
150 - 250
100 - 150
75 - 100
25 - 75
MINERALIZED DOMAINS PROJECTED TO SECTION
NW-210
SE-240
SW-220
SS-250
NE-230
0
50
100
150
METRES
P&E Mining Consultants Inc.
AYA GOLD & SILVER INC.
ZGOUNDER SILVER PROJECT
Ag BLOCK MODEL CROSS SECTION 275,800 E
Scale 1:3,000
January 2022


420,600 E
420,500 E
420,400 E
420,300 E
420,200 E
2,200 EL
2,100 EL
2,000 EL
1,900 EL
1,800 EL
1,700 EL
SURFACE
PIT OUTLINE
Ag g/t
+ 500
250 - 500
150 - 250
100 - 150
75 - 100
25 - 75
MINERALIZED DOMAINS PROJECTED TO SECTION
NW-210
SW-220
NE-230
SE-240
SS-250
0
50
100
150
METRES
P&E Mining Consultants Inc.
AYA GOLD & SILVER INC.
ZGOUNDER SILVER PROJECT
Ag BLOCK MODEL CROSS SECTION 276,000 E
Scale 1:3,000
January 2022



420,600 E
420,500 E
420,400 E
420,300 E
420,200 E
2,200 EL
2,100 EL
2,000 EL
1,900 EL
1,800 EL
1,700 EL
SURFACE
PIT OUTLINE
Ag g/t
+ 500
250 - 500
150 - 250
100 - 150
75 - 100
25 - 75
MINERALIZED DOMAINS PROJECTED TO SECTION
NW-210
SE-240
SW-220
SS-250
NE-230
0
50
100
150
METRES
P&E Mining Consultants Inc.
AYA GOLD & SILVER INC.
ZGOUNDER SILVER PROJECT
Ag BLOCK MODEL CROSS SECTION 276,200 E
Scale 1:3,000
January 2022



Ag g/t

- + 500
- 250 - 500
- 150 - 250
- 100 - 150
- 75 - 100
- 25 - 75

MINERALIZED DOMAINS PROJECTED TO SECTION

- NW-210
- SE-240
- SW-220
- SS-250
- NE-230

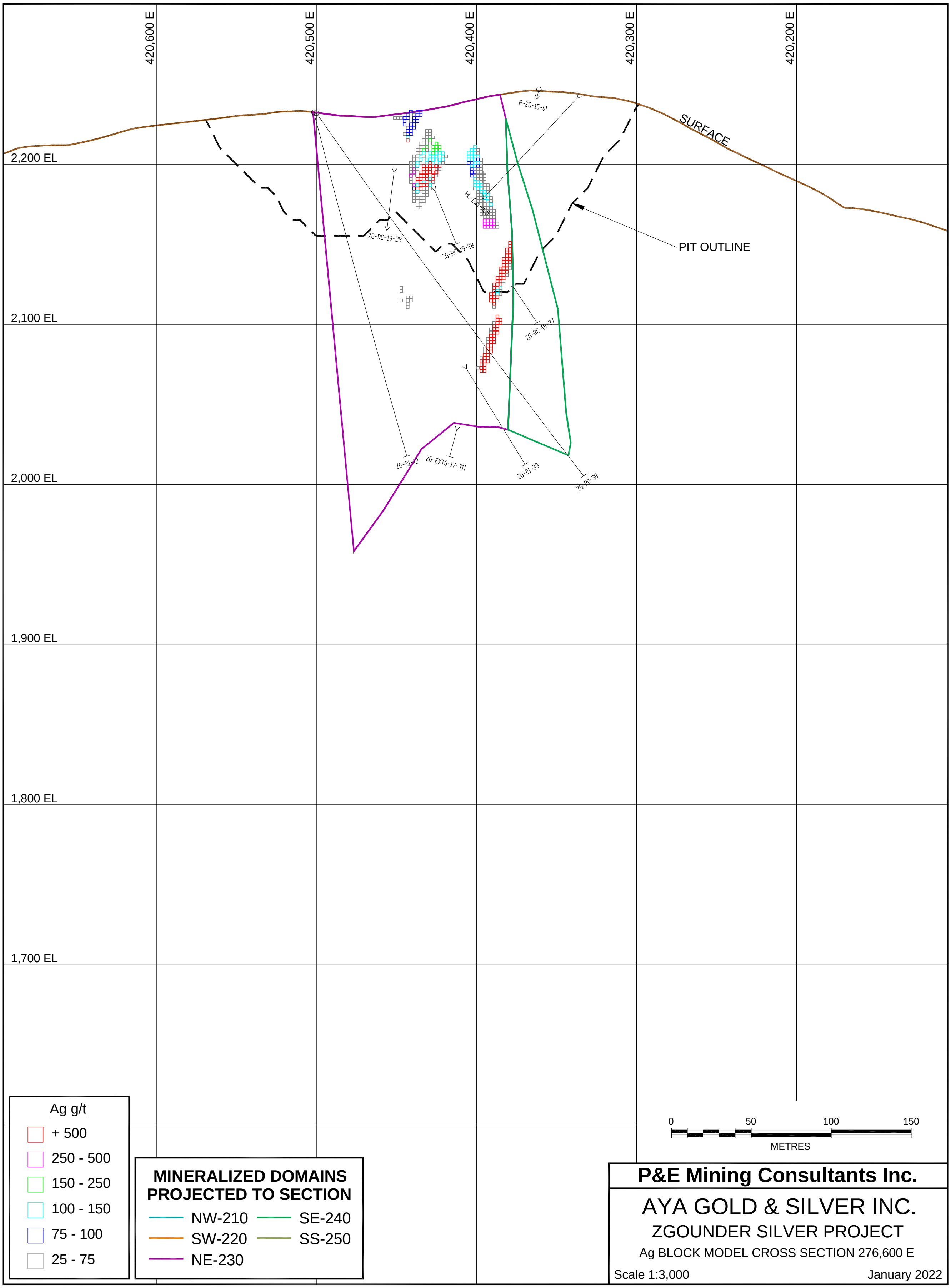
P&E Mining Consultants Inc.

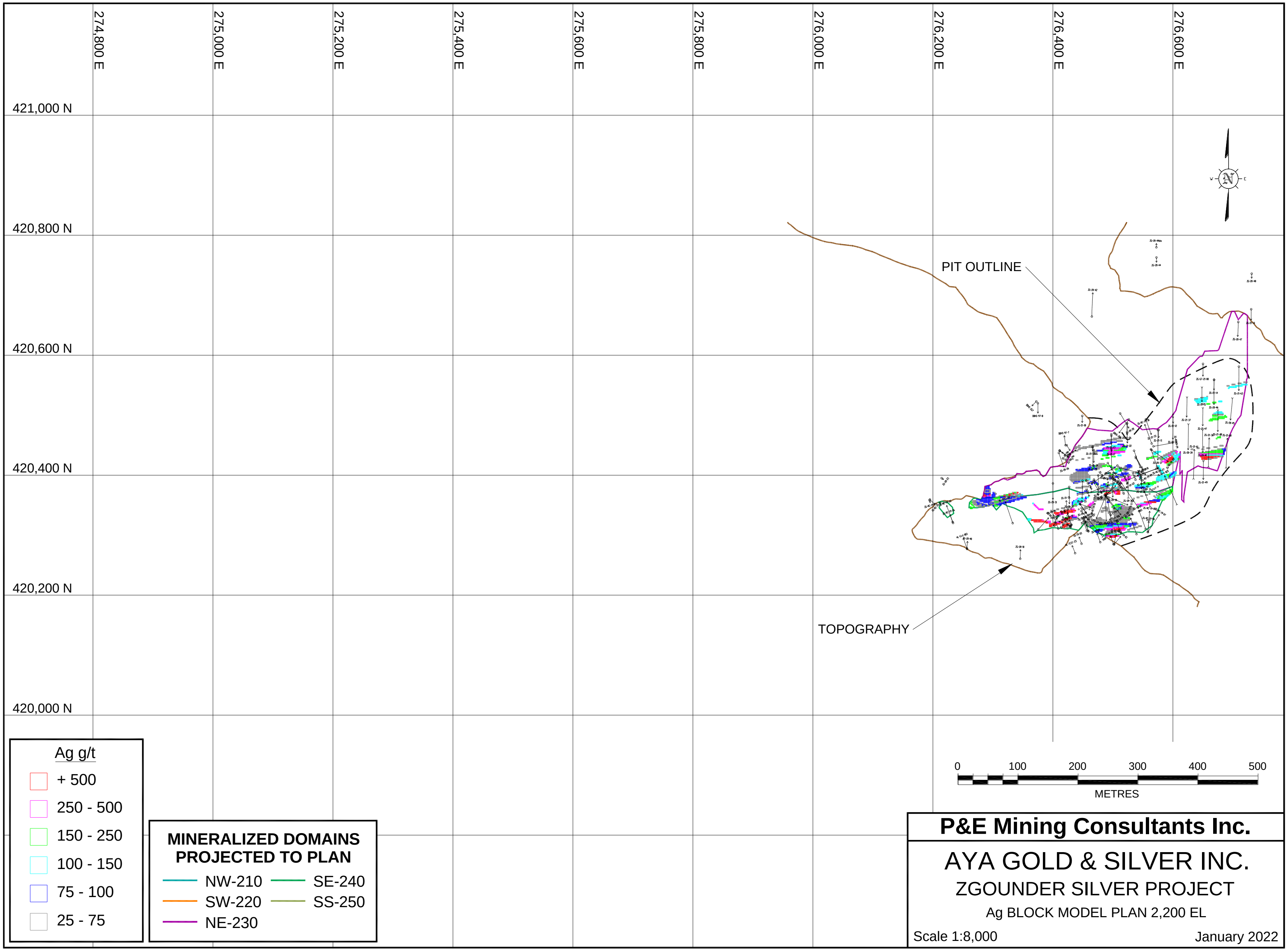
AYA GOLD & SILVER INC.

ZGOUNDER SILVER PROJECT

Ag BLOCK MODEL CROSS SECTION 276,400 E

Scale 1:3,000 January 2022


420,600 E
420,500 E
420,400 E
420,300 E
420,200 E
2,200 EL
2,100 EL
2,000 EL
1,900 EL
1,800 EL
1,700 EL
SURFACE
PIT OUTLINE
Ag g/t
+ 500
250 - 500
150 - 250
100 - 150
75 - 100
25 - 75
MINERALIZED DOMAINS
PROJECTED TO SECTION
NW-210
SE-240
SW-220
SS-250
NE-230
0
50
100
150
METRES
P&E Mining Consultants Inc.
AYA GOLD & SILVER INC.
ZGOUNDER SILVER PROJECT
Ag BLOCK MODEL CROSS SECTION 276,600 E
Scale 1:3,000
January 2022



P&E Mining Consultants Inc.

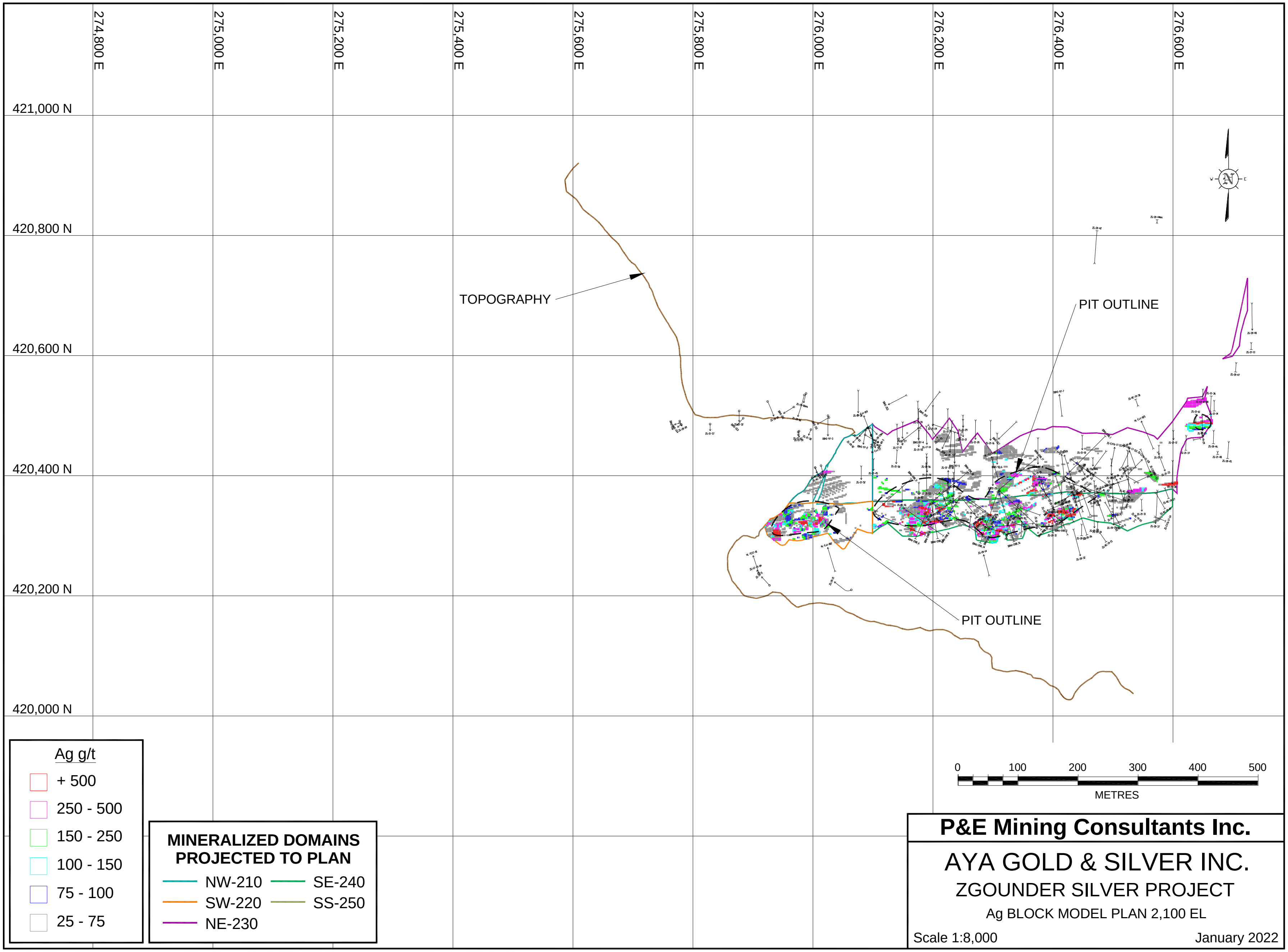
AYA GOLD & SILVER INC.

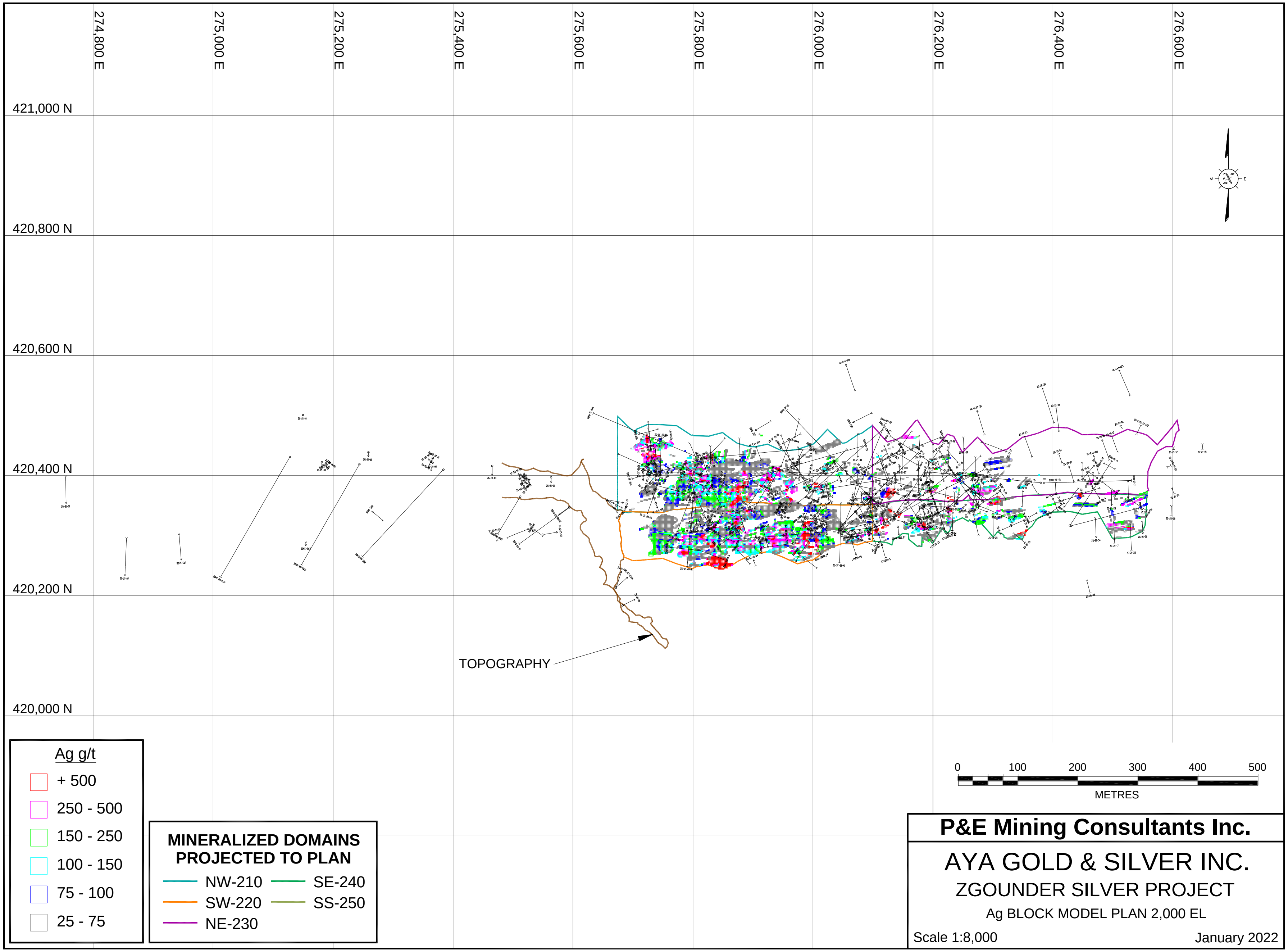
ZGOUNDER SILVER PROJECT

Ag BLOCK MODEL PLAN 2,200 EL

Scale 1:8,000

January 2022


274,800 E
275,000 E
275,200 E
275,400 E
275,600 E
275,800 E
276,000 E
276,200 E
276,400 E
276,600 E
421,000 N
420,800 N
420,600 N
420,400 N
420,200 N
420,000 N
TOPOGRAPHY
PIT OUTLINE
PIT OUTLINE
N
Ag g/t
+ 500
250 - 500
150 - 250
100 - 150
75 - 100
25 - 75
MINERALIZED DOMAINS PROJECTED TO PLAN
NW-210
SE-240
SW-220
SS-250
NE-230
0
100
200
300
400
500
METRES
P&E Mining Consultants Inc.
AYA GOLD & SILVER INC.
ZGOUNDER SILVER PROJECT
Ag BLOCK MODEL PLAN 2,100 EL
Scale 1:8,000
January 2022



P&E Mining Consultants Inc.

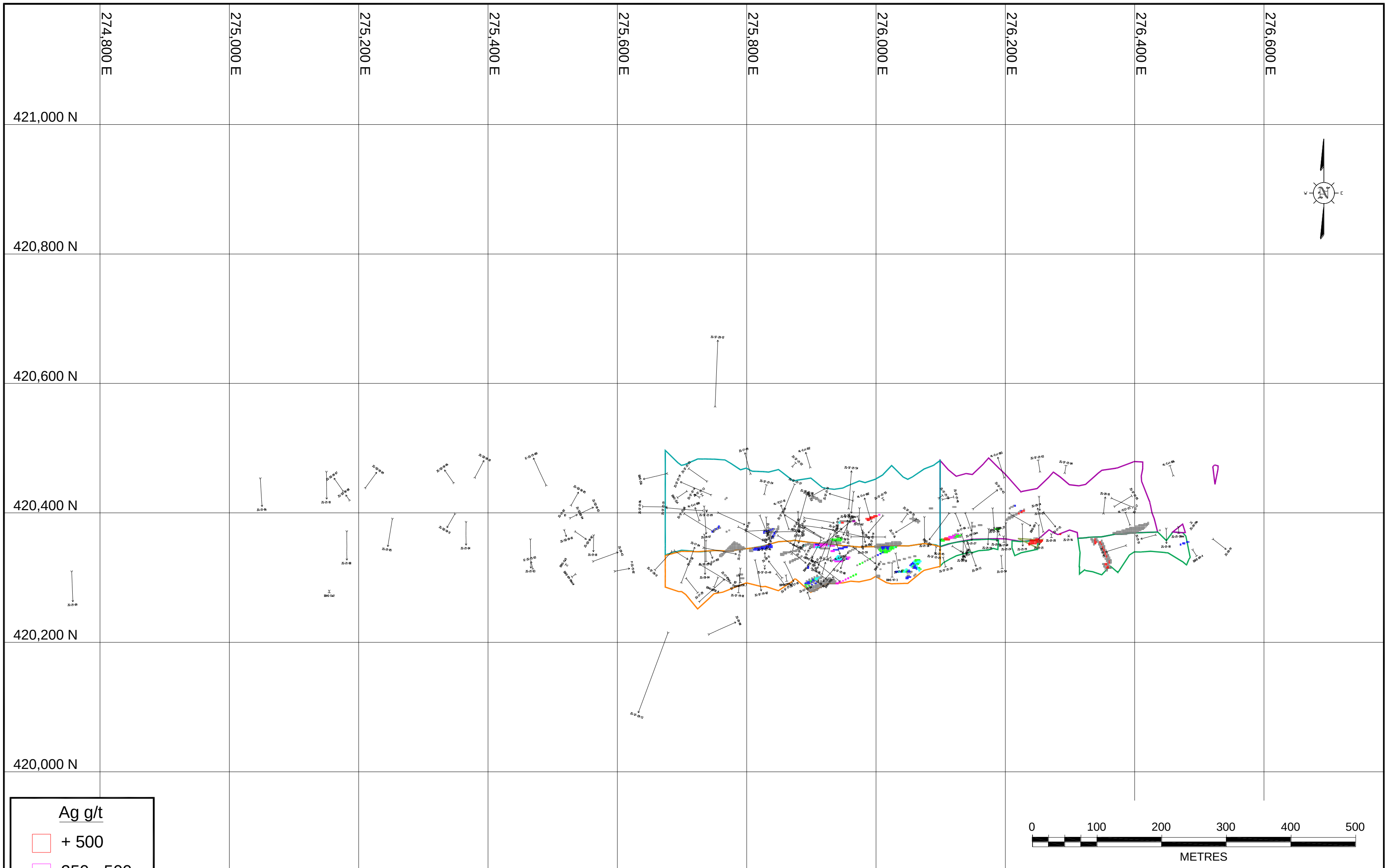
AYA GOLD & SILVER INC.

ZGOUNDER SILVER PROJECT

Ag BLOCK MODEL PLAN 2,000 EL

Scale 1:8,000

January 2022



P&E Mining Consultants Inc.

AYA GOLD & SILVER INC.

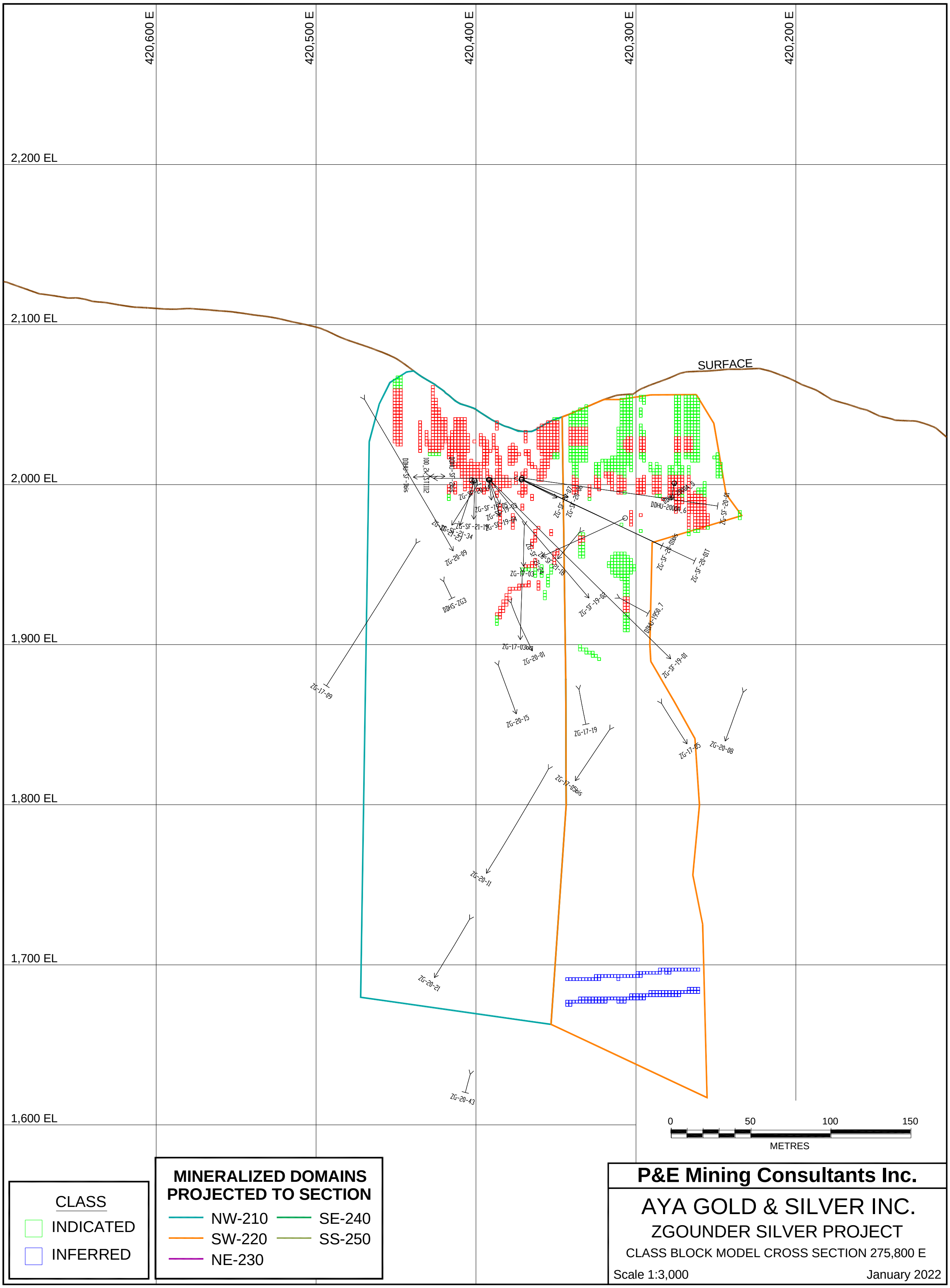
ZGOUNDER SILVER PROJECT

Ag BLOCK MODEL PLAN 1,900 EL

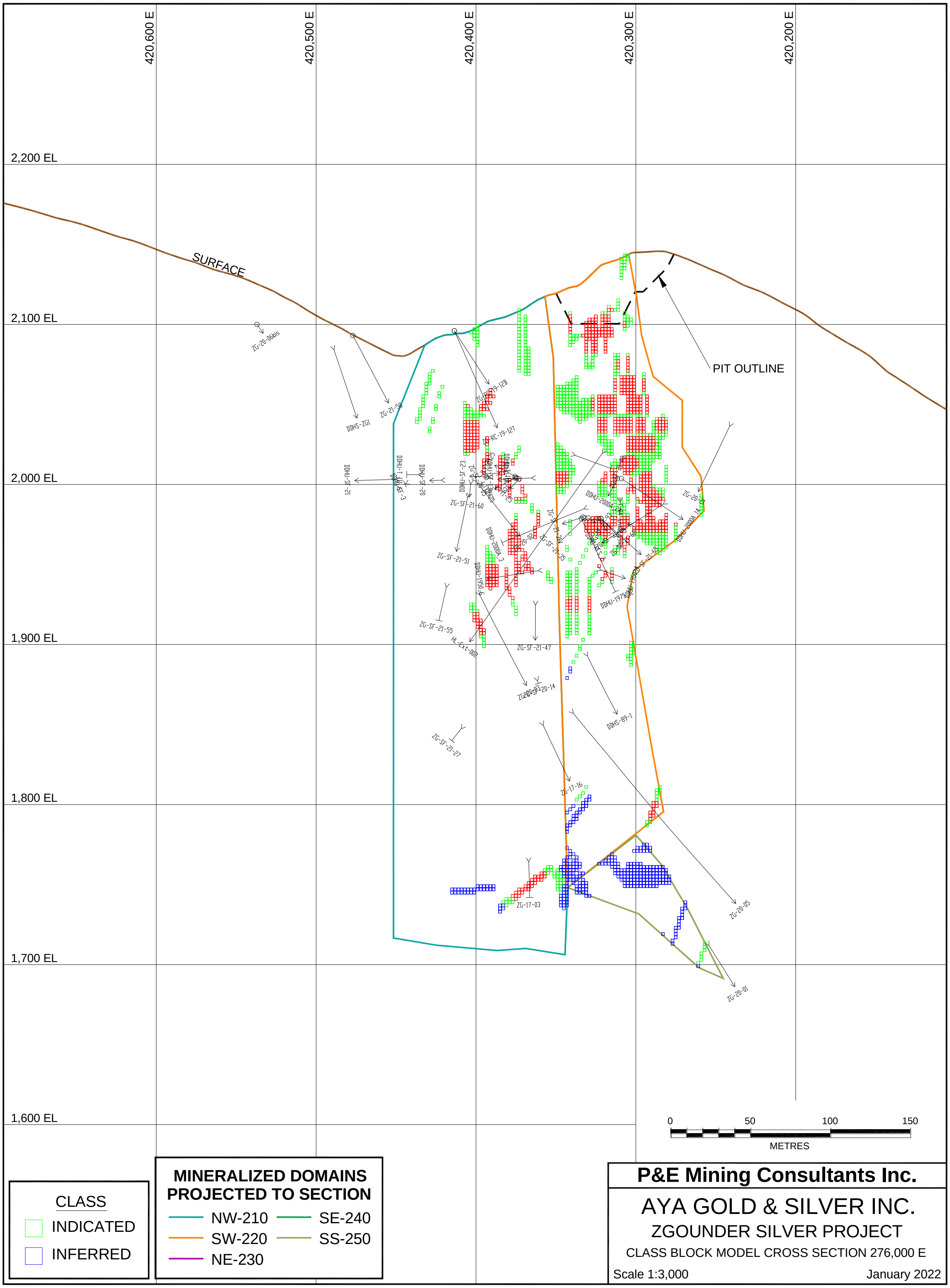
Scale 1:8,000

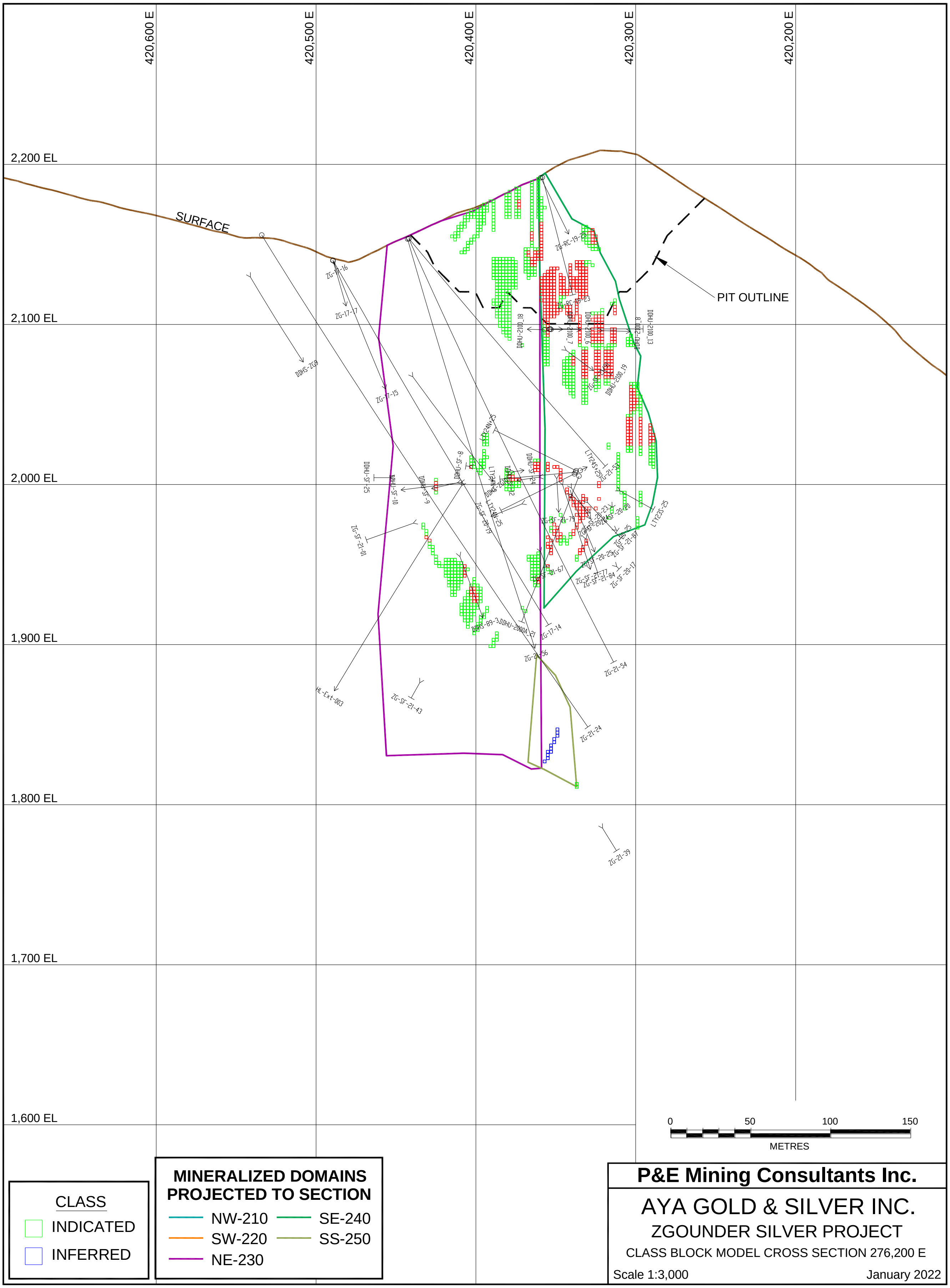
January 2022

Appendix E – Classification Block Model Cross Sections and Plans



420,600 E
420,500 E
420,400 E
420,300 E
420,200 E
2,200 EL
2,100 EL
2,000 EL
1,900 EL
1,800 EL
1,700 EL
1,600 EL
SURFACE
ZG-17-09
ZG-20-15
ZG-17-19
ZG-17-05bis
ZG-20-11
ZG-20-21
ZG-20-43
ZG-17-15
ZG-20-08
ZG-SF-19-01
ZG-SF-19-02
ZG-20-01
ZG-20-09
DDHS-ZG3
0
50
100
150
METRES
CLASS
INDICATED
INFERRED
MINERALIZED DOMAINS
PROJECTED TO SECTION
NW-210
SE-240
SW-220
SS-250
NE-230
P&E Mining Consultants Inc.
AYA GOLD & SILVER INC.
ZGOUNDER SILVER PROJECT
CLASS BLOCK MODEL CROSS SECTION 275,800 E
Scale 1:3,000
January 2022


420,600 E
420,500 E
420,400 E
420,300 E
420,200 E
2,200 EL
2,100 EL
2,000 EL
1,900 EL
1,800 EL
1,700 EL
1,600 EL
SURFACE
PIT OUTLINE
0
50
100
150
METRES
CLASS
INDICATED
INFERRED
MINERALIZED DOMAINS
PROJECTED TO SECTION
NW-210
SE-240
SW-220
SS-250
NE-230
P&E Mining Consultants Inc.
AYA GOLD & SILVER INC.
ZGOUNDER SILVER PROJECT
CLASS BLOCK MODEL CROSS SECTION 276,000 E
Scale 1:3,000
January 2022



P&E Mining Consultants Inc.

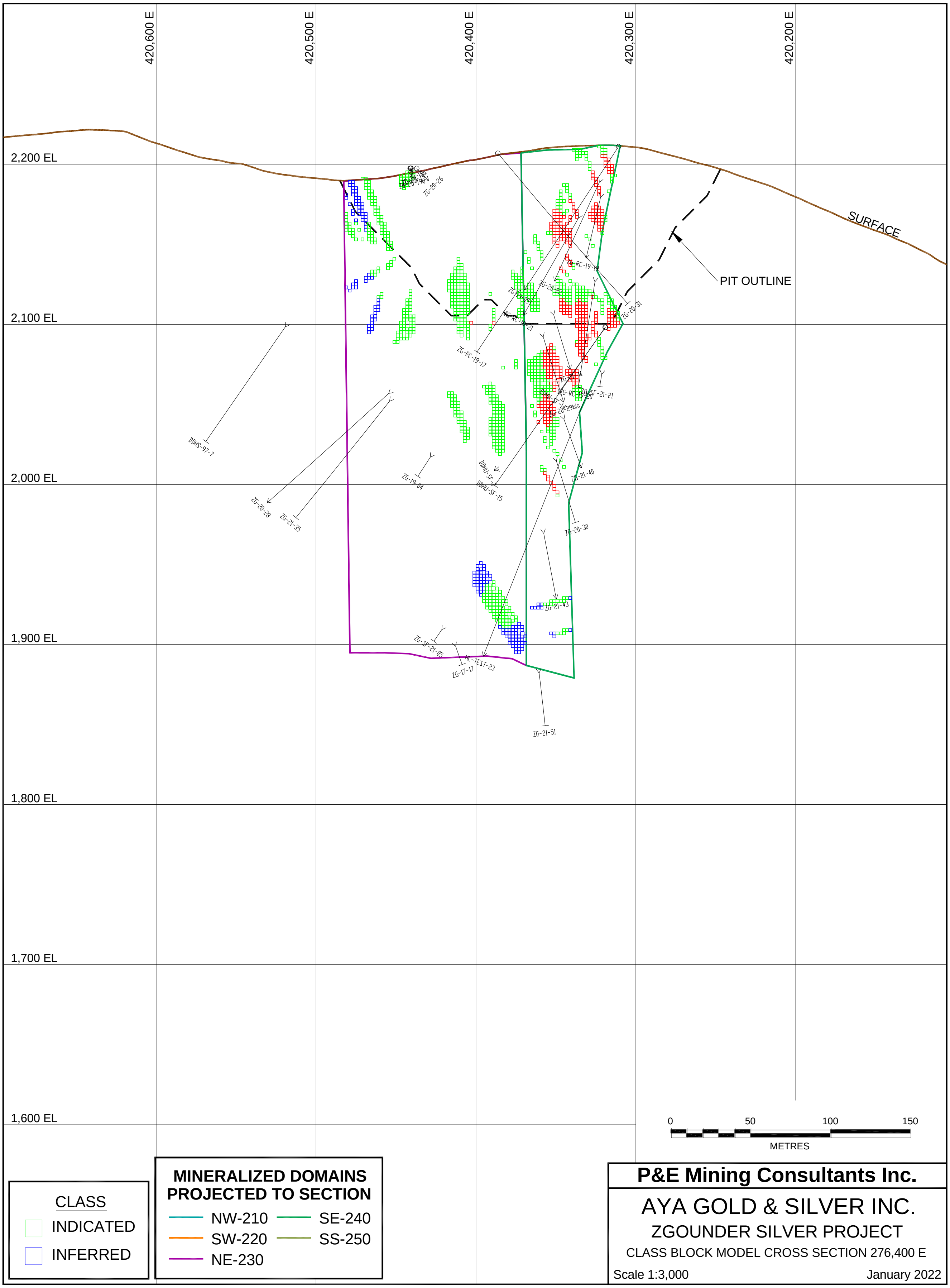
AYA GOLD & SILVER INC.

ZGOUNDER SILVER PROJECT

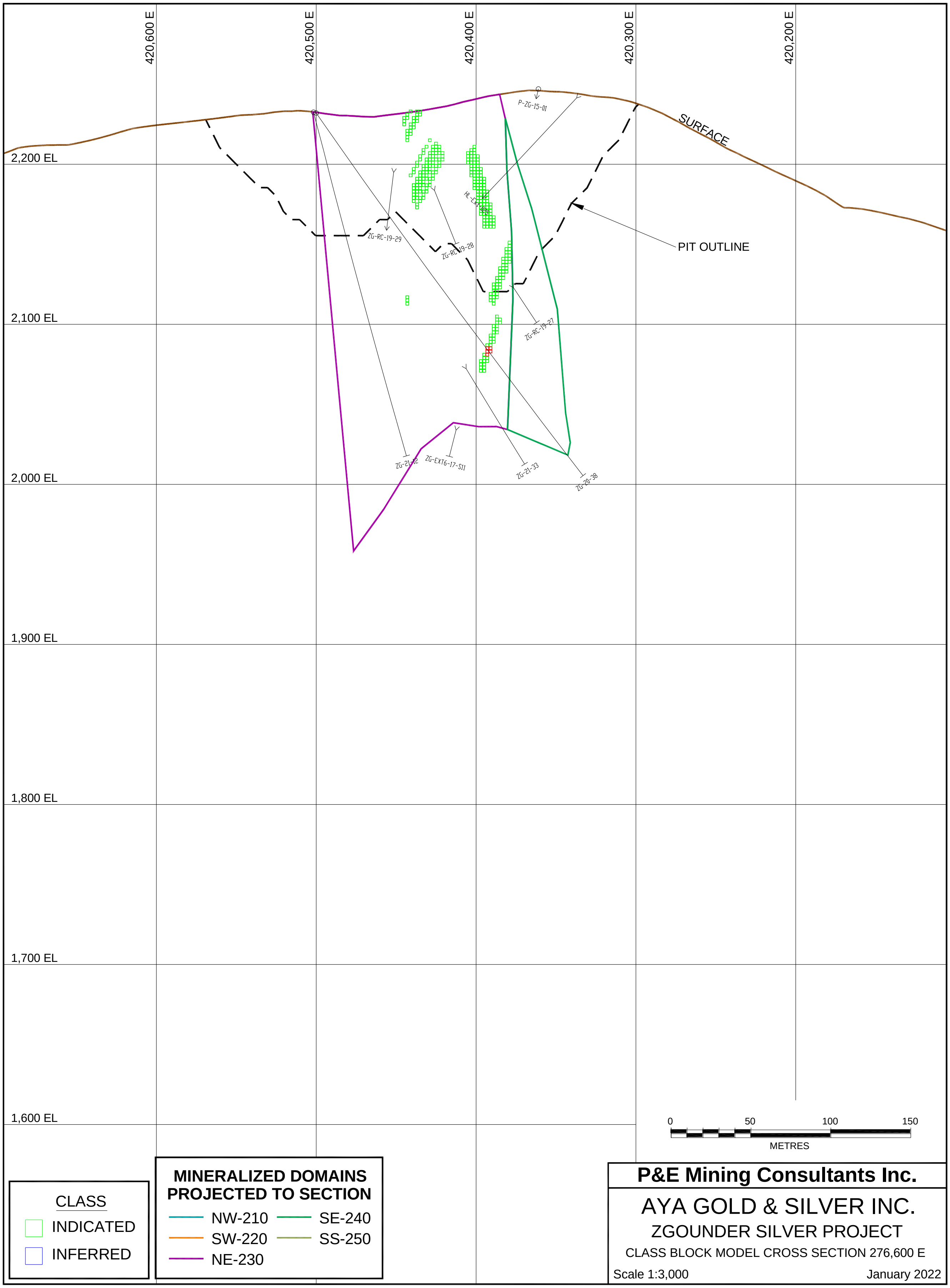
CLASS BLOCK MODEL CROSS SECTION 276,200 E

Scale 1:3,000

January 2022



420,600 E
420,500 E
420,400 E
420,300 E
420,200 E
2,200 EL
2,100 EL
2,000 EL
1,900 EL
1,800 EL
1,700 EL
1,600 EL
SURFACE
PIT OUTLINE
0
50
100
150
METRES
CLASS
INDICATED
INFERRED
MINERALIZED DOMAINS
PROJECTED TO SECTION
NW-210
SE-240
SW-220
SS-250
NE-230
P&E Mining Consultants Inc.
AYA GOLD & SILVER INC.
ZGOUNDER SILVER PROJECT
CLASS BLOCK MODEL CROSS SECTION 276,400 E
Scale 1:3,000
January 2022



P&E Mining Consultants Inc.

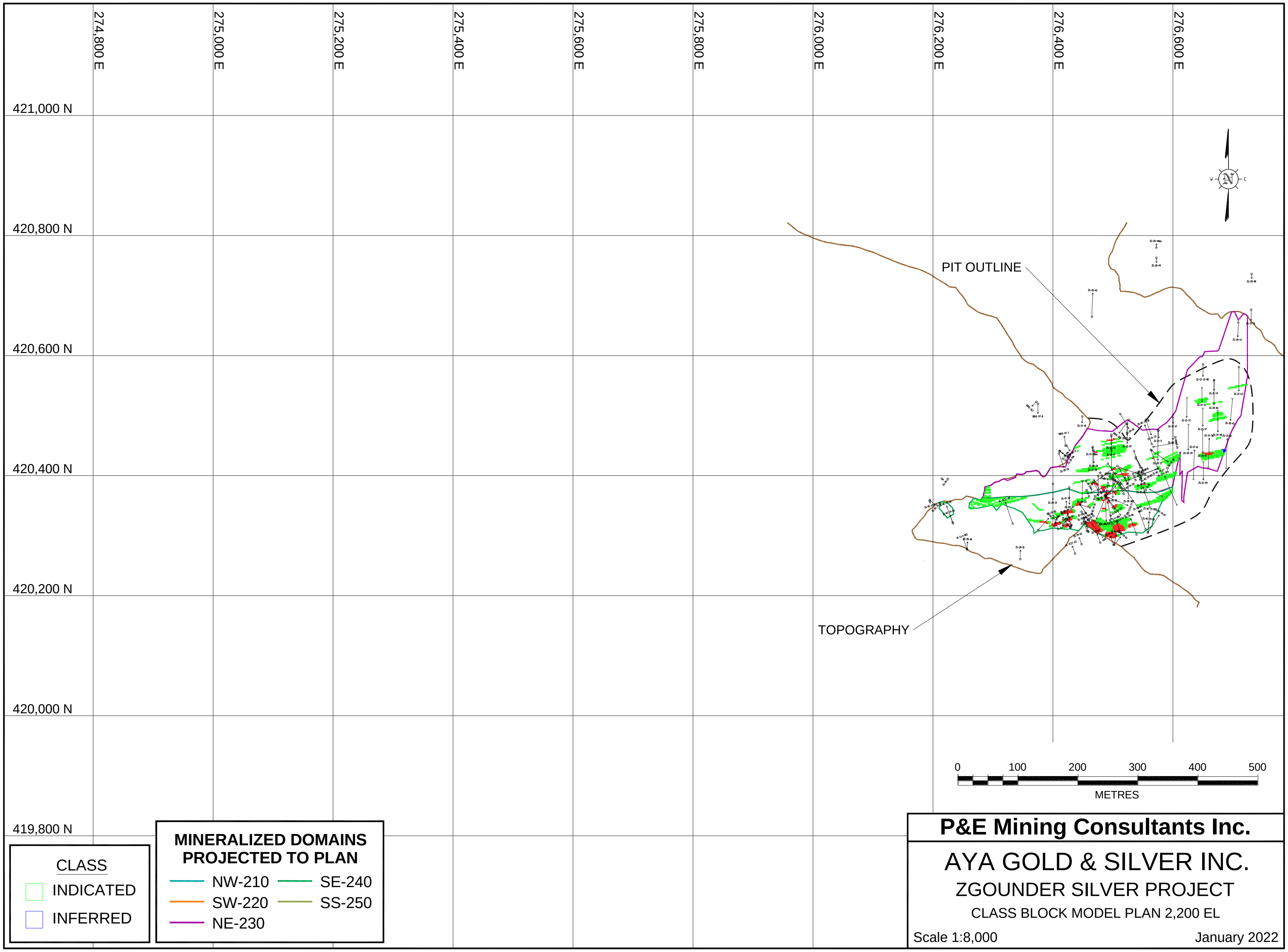
AYA GOLD & SILVER INC.

ZGOUNDER SILVER PROJECT

CLASS BLOCK MODEL CROSS SECTION 276,600 E

Scale 1:3,000

January 2022



P&E Mining Consultants Inc.

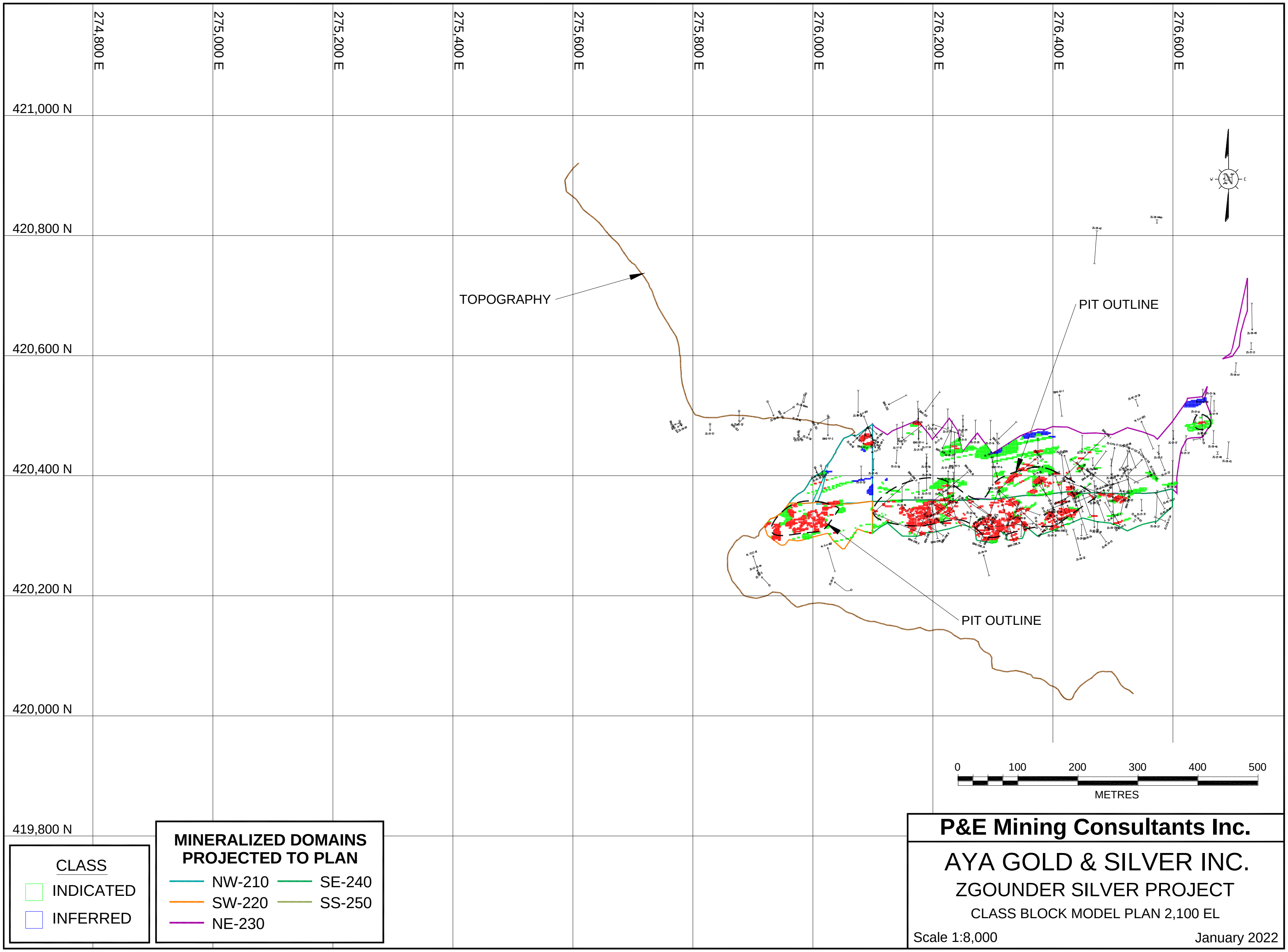
AYA GOLD & SILVER INC.

ZGOUNDER SILVER PROJECT

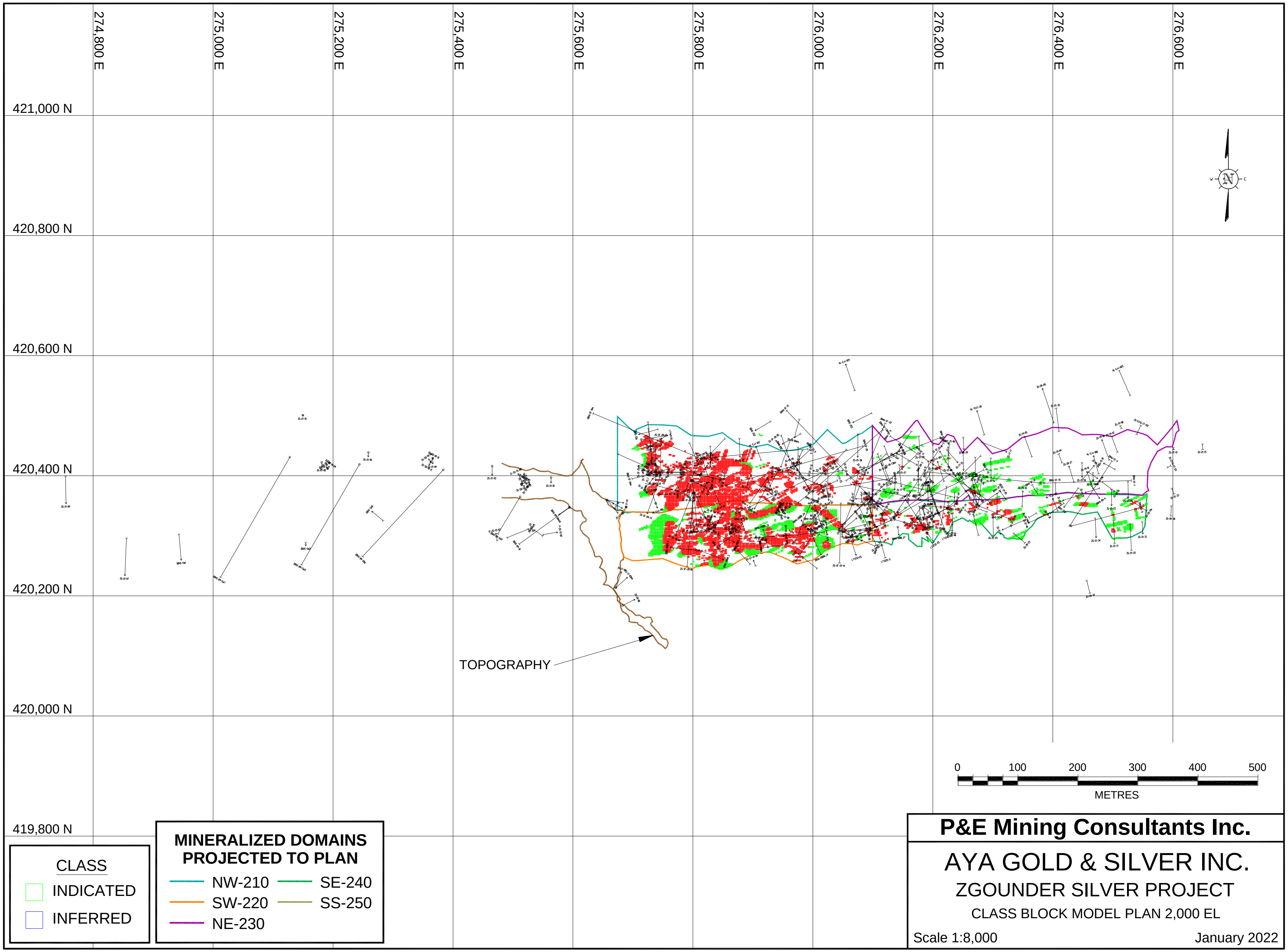
CLASS BLOCK MODEL PLAN 2,200 EL

Scale 1:8,000

January 2022



274,800 E
275,000 E
275,200 E
275,400 E
275,600 E
275,800 E
276,000 E
276,200 E
276,400 E
276,600 E
421,000 N
420,800 N
420,600 N
420,400 N
420,200 N
420,000 N
419,800 N
TOPOGRAPHY
PIT OUTLINE
PIT OUTLINE
0
100
200
300
400
500
METRES
CLASS
INDICATED
INFERRED
MINERALIZED DOMAINS PROJECTED TO PLAN
NW-210
SE-240
SW-220
SS-250
NE-230
P&E Mining Consultants Inc.
AYA GOLD & SILVER INC.
ZGOUNDER SILVER PROJECT
CLASS BLOCK MODEL PLAN 2,100 EL
Scale 1:8,000
January 2022



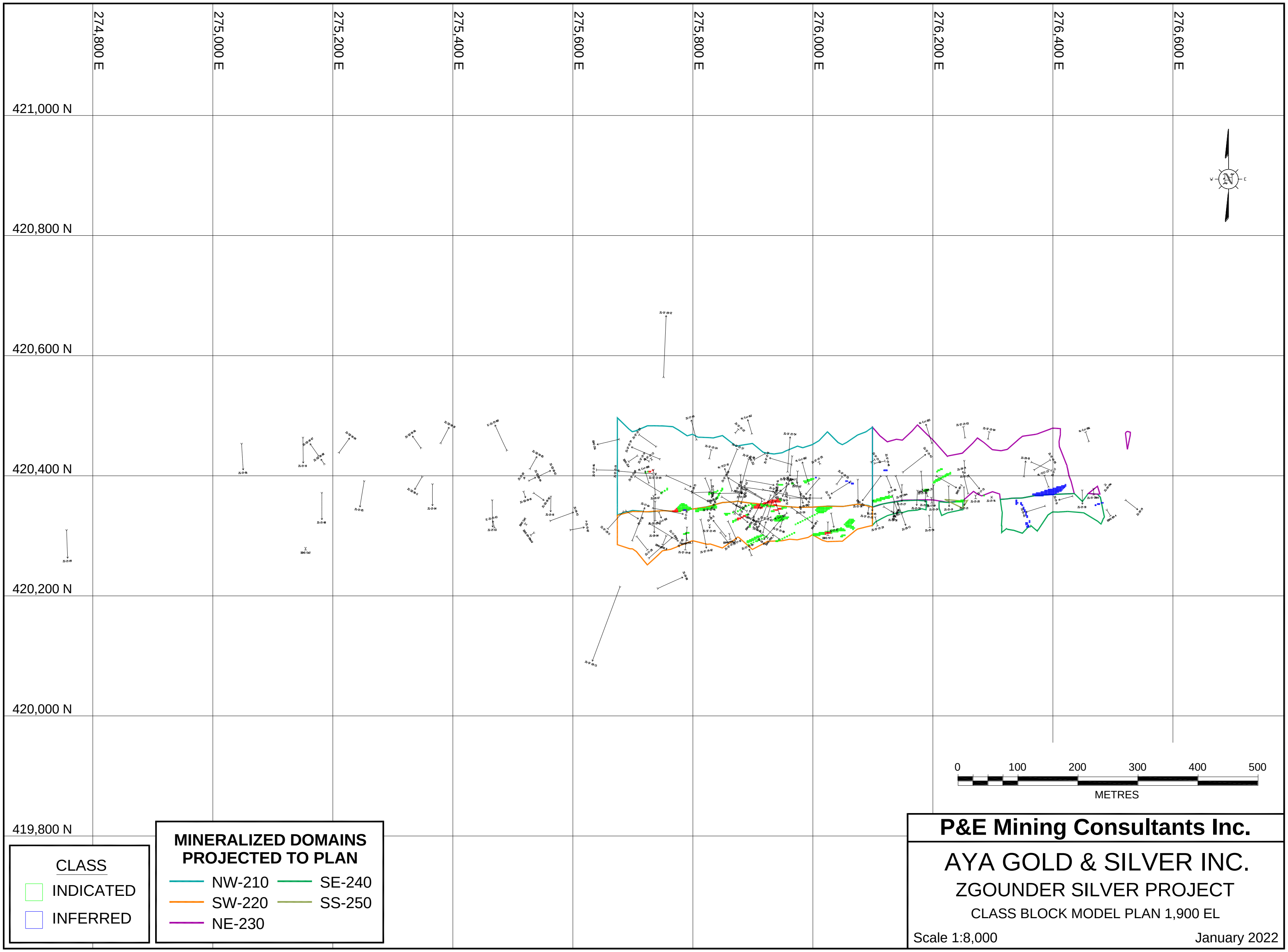
CLASS
INDICATED
INFERRED

MINERALIZED DOMAINS PROJECTED TO PLAN
NW-210
SE-240
SW-220
SS-250
NE-230

P&E Mining Consultants Inc.

AYA GOLD & SILVER INC.
ZGOUNDER SILVER PROJECT
CLASS BLOCK MODEL PLAN 2,000 EL

Scale 1:8,000
January 2022


274,800 E
275,000 E
275,200 E
275,400 E
275,600 E
275,800 E
276,000 E
276,200 E
276,400 E
276,600 E
421,000 N
420,800 N
420,600 N
420,400 N
420,200 N
420,000 N
419,800 N
N
0
100
200
300
400
500
METRES
CLASS
INDICATED
INFERRED
MINERALIZED DOMAINS
PROJECTED TO PLAN
NW-210
SE-240
SW-220
SS-250
NE-230
P&E Mining Consultants Inc.
AYA GOLD & SILVER INC.
ZGOUNDER SILVER PROJECT
CLASS BLOCK MODEL PLAN 1,900 EL
Scale 1:8,000
January 2022

Appendix F – Optimised Pit Shell

ZGOUNDER SILVER PROJECT
OPTIMIZED PIT SHELL