

AYA GOLD & SILVER INC.

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TECHNICAL REPORT

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TECHNICAL REPORT

Preliminary Economic Assessment for the Boumadine Polymetallic Project, Kingdom of Morocco

LONGITUDE 4°55'18" WEST AND LATITUDE 31°24'40" NORTH
UTM WGS 84 ZONE 30R 317,310 m EAST AND 3,476,770 m NORTH



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1.0 EXECUTIVE SUMMARY

1.1 INTRODUCTION

This Technical Report (the "Report") presents a Preliminary Economic Assessment ("PEA") for the Boumadine Polymetallic Project, Kingdom of Morocco. This PEA is based on the Mineral Resource Estimate ("MRE") published by Aya on February 24, 2025. The current Estimate is an update of the MRE prepared by P&E Mining Consultants Inc. ("P&E"), published on May 8, 2024. The Report has an effective date of November 4, 2025.

Aya, a reporting issuer, trades on the TSX under the symbol "AYA" and has its head office located at Suite 132, 1320 Graham, Ville Mont-Royal, Québec, H3P 3C8.

1.2 PROPERTY DESCRIPTION AND LOCATION

The Boumadine Property (also known as Boumaadine, Boumâadine, and Bou Madine) is located in the Province of Errachidia, Kingdom of Morocco, ~220 km east of the City of Ouarzazate and 70 km southwest of the City of Errachidia.

Aya's property in the Boumadine area ("Boumadine Property") consists of 12 mining permits and 19 exploration permits totaling 341 km² in size. The "Boumadine Mining License", which contain Boumadine Deposit and is the focus of this Boumadine Report, consists of mining permit LE-383661 and covers the historical Boumadine Mine, the Boumadine Camp, and the current Mineral Resource Estimate described in this report, which total 32 km² in area. The other thirty permits are distributed within a 25 km radius of the Boumadine Deposit and collectively cover an additional 309 km² in area. Furthermore, an Authorization of Exploration of 600 km² was granted to Aya in January 2025.

On October 9, 2012, Aya and ONHYM signed a joint venture agreement for the acquisition, development and exploitation of the Boumadine Deposit. Under the terms of the said agreement, Aya acquired 85% of mining license LE-383661 for total cash payments of MAD 28 million, being approximately USD 2.8 million at such time. A new Moroccan company - BGM, was created with Aya and ONHYM as 85%-15% shareholders. The mining title of the Boumadine Mining License was transferred to BGM by ONHYM. The participation of ONHYM is subject to dilution if they fail to invest 15% in the budget after Aya matches all the previous investment from ONHYM. ONHYM will receive a 3% royalty and Aya will receive a 2.75% management fee on BGM sales revenue from the first year of operation.

In addition to its ownership of the Boumadine Mining License, Aya, through its subsidiaries, has; 100% ownership of a total of 11 mining licenses and exploration permits and; an option to earn 100% interest in 19 other mining licenses and exploration permits.

1.3 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The Boumadine Property is located in the Errachidia Province of the Meknès-Tafilalet Region, in the Anti-Atlas Mountains. It is accessible via the National Highway 10 (N10), ~220 km east-northeast from Ouarzazate City or ~70 km southwest from Errachidia City. Tinejdad, the nearest town, is ~16 km north from the historical Boumadine Mine. The nearest village, Bouyouid, is 4 km away. The Property is accessible from Tinejdad by all-terrain vehicle on a paved and gravel road.

The Property has a Sahara climate, which falls under the category of a hot desert climate, also known as a "hot arid climate" (Köppen climate classification BWh). Summers are hot, with daytime temperatures >40°C. Winters are generally mild, with daytime temperatures ranging from mild to warm; nighttime temperatures can be cool, but freezing conditions are uncommon. The region receives very little rainfall throughout the year, most of which occurs in winter. Field work at Boumadine can be carried out year-round.

The Property is accessible by two roads: 1) a 16-km dirt road southwards from Tinejdad; and 2) a 2.8-km dirt road from the east through the village of Bouyouid. The National Highway 10 (N10) goes through the City of Tinejdad and connects to Ouarzazate City to the west and Errachidia City to the east of Tinejdad. The Boumadine project is connected to the national power line.

There are numerous dirt roads and paths that lead to former shafts and other remnants of the historical mining infrastructure. Water is currently sourced from historical underground workings and wells. Electricity on site is provided via the national electricity grid. The facilities on-site are adapted for exploration operations. They include an office, drill core shack, the AfriLab sample preparation laboratory, drill contractors workshops, and drill contractors camps.

The physiography of Boumadine Property is characterized by its desert setting, with influences from the nearby Atlas Mountains. The topography of the area is marked by ridges and hills mostly, with altitude ranging between 980 and 1,300 m asl. The site and its surroundings exhibit characteristics typical of a desert landscape, with vast expanses of arid and rocky terrain. Vegetation in the Boumadine area consists mainly of desert plants, such as Acacia Raddiana and Tamarix Amplexicaul. Drought-resistant grasses may be found and provide some ground cover. Oases and palm trees are notable features in this region.

The Boumadine Mining License is in close proximity to the Ziz river, which flows through the centre of the Town of Tinejdad. The river, along with its valley, contributes to the oasis environment with palm groves, particularly date palms. Group of nomads pass by the Boumadine Property with their livestock of camels or goats. Historical artisanal mining activities have been recognized at several places on the Boumadine Property. Historical extraction work focused on barite and lead veins.

Mine workers and other personnel are available from surrounding villages and Tinejdad. Errachidia City (formerly Ksar souk) is the closest major urban centre. Errachidia has an international airport with access to Casablanca and it is also accessible by road from Marrakech, ~420 km away. Basic supplies, such as food and limited accommodation, are available at Tinejdad; Errachidia offers greater diversity in supplies. Special items must be purchased from Casablanca or Marrakech.

1.4 HISTORY

The historical Boumadine Mine is one of the oldest known mines in the Kingdom of Morocco. It was probably exploited by the Portuguese in the 15th and 16th centuries. They extracted the oxidized part of the polymetallic veins to a depth of as much as 20 m. Such workings are found along a north-south strike length of 4.2 km on the Boumadine Property.

Between 1956 and 1998, exploration and mining activities in the Boumadine area were completed by the Bureau de Recherches et de Participations Minières ("BRPM"), with and without partners. These activities included mineral prospecting, geophysical surveys, drilling, mineralogical studies, mineral resource estimations, metallurgical testwork, engineering and economic studies, shaft excavation, and underground development and mining. Underground mining from 1986 to 1992 produced 261,485 t of mineralized material from four mining levels for mineral processing on-site.

In 2013, Maya Gold and Silver Inc. ("Maya"; precursor entity to Aya Gold & Silver) acquired the Project through a Joint Venture with the ONHYM (new entity of the BRPM) on a 85%/15% basis, respectively. Between 2013 and 2016, Maya completed geological mapping and grab sampling of the historical mineralized structures. In 2017, Maya completed a drilling program to confirm the historical mineral resources. Fourteen drill holes totalling 3,158 m were completed over the Central, South and Tizi Zones. Between 2018 and 2020, Maya completed a sampling program on two historical tailings deposits and 9,503 m of diamond drilling on the South, Central, North, Imariren and Tizi Zones. In addition, Maya completed a drone survey over the Boumadine Property.

Maya announced its name change to Aya in a Company press release dated July 31, 2020.

1.5 GEOLOGICAL SETTING, MINERALIZATION, DEPOSIT TYPE

The Boumadine Property is located at the eastern end of the Anti-Atlas Mountain Range, which extends east-northeast to west-southwest, over approximately 600 km from the Atlantic Ocean in the west to the interior of the African Plate in the east. The Anti-Atlas basement rocks are mainly Neoproterozoic in age and consist of ophiolites, island arc-related gneiss and intrusive rocks, particularly near to the northern edge of the West African Craton.

The Boumadine polymetallic deposit (Ag, Au, Cu, Pb, Zn) is located on the northwest side of the Ougnat Massif (or Boutonnière). The geology of the Massif consists of a Neoproterozoic metasedimentary basement overlain unconformably by a Late Neoproterozoic volcano-sedimentary rock sequence and by Paleozoic lacustrine sedimentary and minor volcanic rocks. The basement consists of sandstone, pelites and greywackes that are intruded locally by granite, granodiorite, and diorite bodies. The volcano-sedimentary sequence consists of felsic and mafic volcanic rock units separated by volcano-sedimentary units.

The volcanic and volcano-sedimentary rock unit have been grouped into three formations, which from the oldest to youngest are:

- Tamerzaga-Timrachine Formation ("TTF"): Consists of ignimbrites, rhyodacites and andesites;
- Isilf-Quinou-Oufroukh Formation: Consists of volcano-sedimentary rocks, specifically tuffs and breccia, andesite flows; and fine- to coarse-grained sedimentary rocks; and
- Aoujane-Aissa-Akchouf Formation: Formed of ignimbrites, dacite domes and flows, and andesite flows.

These three formations are intruded by dolerite, microdiorite and andesite dykes. At the Boumadine Mining License, only the andesite dykes are present and trend north-south.

The Ougnat Massif area was subjected to a Neoproterozoic shearing, which generated regional-scale faults trending N30°E and associated secondary fractures. The area has also been affected by a late-stage series of north-south extensional fractures that were subsequently reactivated by a compressive Hercynian tectonic event.

The TTF volcanoclastic sequence of felsic tuffs and mafic tuffs host the Boumadine Deposit. The felsic tuffs consist of angular to rounded cm-size felsic fragments, quartz eyes, plagioclase grains, and locally mafic fragments. This felsic sequence is homogeneous and massive, and sits unconformably on mafic tuffs. Mafic tuffs consist of amphibole and fragments/clasts of sedimentary rocks. Mafic tuffs are interpreted as underwater-deposited volcanoclastic eruptives.

Many intrusions are observed on the Boumadine Property. The intrusions are divided into a pre- to syn-mineralization group and a post-mineralization group. The pre- to syn-mineralization intrusions are mainly felsic to intermediate in composition, show aphanitic to porphyritic textures, and form dykes and sills. Locally porphyritic mafic dykes, similar in composition to mafic tuffs, crosscut the felsic tuff sequence and syn-mineralization dykes, suggesting bimodal magmatism.

The post-mineralization intrusions consist of rhyolite subvolcanic domes associated with normal faults. These domes are interpreted as being synchronous with a post-mineralization deformation episode that disrupted the Boumadine mineralized zones. Subsequently, a swarm of regionally extensive mafic dykes intruded every lithological unit on the Boumadine Property.

Two events of hydrothermal alteration are observed on the Boumadine Mining License. The first alteration event affects the felsic tuff sequence as phyllic alteration (quartz-sericite-pyrite). Proximal to massive sulphide veins (1 to 5 m thick), there is an advanced clay alteration composed of kaolinite and pyrophyllite. The second sequence of alteration affects mainly the underlying mafic tuffs and consists of propylitic alteration (epidote and chlorite). Near the veins, the alteration minerals are black chlorite, pyrophyllite and pyrite. The transition between these two alteration events is relatively sharp and consistent with the change in tuff composition.

Due to the extensive weathering to clay minerals, the Boumadine Deposit has a very light colour that contrasts with the surrounding landscape. The mantos, “chapeau de fer” or “iron cap” alteration extends from 5 to 10 m depth. The mantos consists principally of goethite and jarosite with sparse hematite and no lepidochrosite. This mineralogical assemblage indicates that the oxidation fluids were strongly acidic. In this case, Mn, Zn, Cd, Ni, and Co, are highly mobile in the acid and sulphur-rich fluids and are commonly leached at surface. However, Ag, Au, Ba, Sr and Pb are immobile and form stable sulphosalts. The hydroxide-rich “mantos” has been partially mined out by artisanal workers for ochre and precious metals.

The Boumadine Deposit has been traced on surface and in drilling for approximately 6.0 km along strike. Strike direction varies from mainly northwest to northerly and dips vary from steeply northeast to steeply southwest. The Boumadine Deposit consists of 45 mineralized domains that have been grouped into five separate zones. The South and Central Zones consist of 13 stacked mineralized vein domains. From the south end of the South Zone to the north end of the Central Zone, these domains extend for 4,800 m along strike, up to 300 to 400 m across strike and up to 1,000 m down-dip. The South Zone appears to be offset dextrally along a northeast-trending fault from the Central Zone. The north end of the Central Zone appears to be offset sinistrally along a northeast-trending fault from the North Zone. The North Zone consists of eight closely-spaced mineralized vein domains. This Zone is 650 m long, 5 to 10 m in thickness and 500 m down-dip. It strikes northwest and dips steeply southwest. The Imariren Zone and the Tizi Zone are two sub-parallel, single mineralized vein domains that are 200 m apart in the south and 500 m apart in the north, strike northerly, and dip vertically. The Tizi Zone has been extended to 2.0 km in length, while Imariren has been traced for 1.2 km. Both zones extend 600 m down-dip.

The Boumadine Deposit mineralized zones consist of 1 to 4 m-wide massive sulphide lenses/veins oriented N20°W and dipping 70° east. The massive sulphide veins (approximately 70% sulphide) consist of pyrite, sphalerite, galena, arsenopyrite and chalcopyrite, with subordinate amounts of cassiterite, silver-rich sulphosalts, stannite, enargite, bismuthinite, native copper and bismuth. The main mineralization zone is surrounded by a 1 to 10 m thick halo of 10 to 30% disseminated pyrite and two types of veinlets: 1) quartz-carbonate-galena-sphalerite veinlets; and 2) massive pyrite veinlets. Geochemically, there is a strong positive correlation of gold with silver and copper and a weaker correlation of zinc with lead and molybdenum. The Boumadine Deposit has been described in literature as being an epithermal polymetallic deposit in a shallow submarine setting, but field and drilling evidence may suggest a deeper environment of formation.

1.6 EXPLORATION AND DRILLING

Exploration activities completed by Aya on the Boumadine Property since 2020, other than drilling, include surface trenching, satellite-based hyperspectral surveys, aerial electromagnetic, magnetic and radiometric survey, mineral prospecting, geological mapping, grab sampling and assaying.

From May 2022 to September 2025, Aya completed 660 diamond drill holes, 94 multipurpose holes (MP) and 49 Reverse circulation holes (RC), totaling 192,957 m. In 2025, Aya completed 273 drill holes, amounting to 109,240 m, on the Boumadine Property. Of these, 214 drill holes totaling 44,514 m were completed along the Boumadine Deposit and utilized for the 2025 Mineral Resource Estimate (MRE). The drilling programs aimed to extend the mineralization of the North, Central, South, Tizi and Imariren Zones and test targets located farther from the main mineralized trend.

1.7 SAMPLE ANALYSES AND DATA VERIFICATION

Aya implemented and monitored a thorough QA/QC program for the drilling completed at the Boumadine Deposit over the 2018 to 2025 period. Examination of QA/QC results for all recent sampling indicates no material issues with accuracy, contamination, or precision in the data. The current Authors have reviewed the QA/QC procedures and results, and the previous verification work conducted by the site visit QP, and confirm that the sample preparation, security, and analytical procedures are adequate. The data are considered to be of satisfactory quality and suitable for use in the current Mineral Resource Estimate.

The current author performed a verification of the Boumadine Deposit database for the 2024 to 2025 period, which included verifying drill hole assay data. No errors were found, and the data is deemed suitable for inclusion in the current Mineral Resource Estimate.

Verification of the Boumadine Deposit data, used for the previous Mineral Resource Estimate, had been reviewed independently by the site visit QP, including a site visit in March 2024, due diligence sampling, verification of drill hole assay data from electronic assay files, and assessment of the available QA/QC data. The site visit QP stated at the time that sufficient verification of the Project data had been undertaken and that the supplied data are of satisfactory quality and suitable for use in the previous Mineral Resource Estimate.

1.8 MINERAL PROCESSING AND METALLURGICAL TESTING

Based on the 2022 and 2024 metallurgical testwork completed by SGS Canada Inc., the process plant design for the Project considers a conventional flotation flowsheet. Lead, zinc and pyrite concentrates will be recovered and sold to market.

OMC used its testwork database to infer missing data and to recommend further testwork for subsequent studies. While both BWi values from testwork fell outside the database range for other similar resources in the area, the average BWi fell at the 81st percentile of the range, aligning well with OMC's typical strategy of designing to the 85th percentile hardness. As such, the average of both test results was used in the comminution circuit design (13.1 kWh/t). The remainder of the comminution circuit design criteria were estimated from OMC's database.

The conditions tested during the 2022 and 2024 SGS locked cycle testing campaigns were used as the basis for the concentrate recovery circuit design, since the testing incorporated recycles at steady state and produced concentrates of acceptable grade and recovery. Using locked cycle data provides a higher design certainty since it more closely represents plant operation by the incorporation of recycles and since it validates the process conditions at steady state.

The design criteria derived from the 2022 and 2024 locked cycle testing includes:

- General circuit design (i.e. orientation, number of stages, implementation of regrinding).
- Reagent addition.
- Concentrate mass pulls, grades and recoveries.
- Residence times.

Next steps include continued refinement of the metallurgical testwork, particularly comminution, variability and dewatering.

1.9 MINERAL RESOURCE ESTIMATE

The Mineral Resource Estimate ("MRE") of the Boumadine Deposit is amenable to conventional open-pit and to underground mining methods.

The MRE contains an Indicated Mineral Resource of 5,2 Mt grading 91 g/t Ag, 2.78 g/t Au, 2.8% Zn and 0.85% Pb containing an estimated 15.1 Moz of Ag, 462 koz of Au, 145 kt of Zn and 44 kt of Pb, and an Inferred Mineral Resource of 29.2 Mt grading 82 g/t Ag, 2.63 g/t Au, 2.11% Zn, and 0.82% Pb containing an estimated 76.8 Moz of Ag, 2.5 Moz of Au, 615 kt of Zn and 237 kt of Pb, as shown in Table 1.1. The MRE has an effective date of February 24, 2025. Approximately 49% of the Inferred Mineral Resource is pit-constrained and reported above a cut-off NSR value of \$95/t, and 51% is deemed for underground development and reported above a cut-off NSR value of US\$125/t. The sensitivity of the out-of-pit Mineral Resource to changes in potentially economic NSR cut-off value was also calculated and the results are listed in Table 1.2.

A total of 428 drill holes for 142,268 m were available for Mineral Resource modelling. Mineralization models were developed by Aya, reviewed and accepted by the Authors. Forty-five individual mineralized domains were identified through drilling and surface sampling. The modelled mineralized domains are constrained by individual wireframes, based on sulphide content and a nominal 100 g/t AgEq cut-off value. Mineralized wireframes were used as hard constraining boundaries for the purposes of block coding, statistical analysis, compositing limits, and estimation of the Mineral Resources.

A rotated three-dimensional block model, with 2.5 m x 5.0 m x 5.0 m blocks, was used for the MRE. The block model consists of estimated Au, Ag, Cu, Pb and Zn grades, bulk density, block volume inclusion percent, and classification criteria. Net smelter return ("NSR"), AuEq and AgEq block values were subsequently calculated from the estimated Ag, Au, Cu, Pb and Zn grades, incorporating metal prices, metallurgical recoveries, concentrate freight and smelter charges.

Sampled assays were composited to a 1.00 m standard length. Grades were estimated using Inverse Distance Squared (ID2) estimation, with two estimation passes. Composites were capped prior to estimation. Composite samples were selected within an oriented search ellipse, based on domain orientation and grade trends. Bulk density values specific to each mineralized domain were assigned based on bulk density measurements obtained from drill core.

Classification criteria were determined from observed grade, geological continuity and variography. Grade blocks estimated in the first pass that used a minimum of two drill holes and with an average distance between composites of <50 m were classified as Indicated, and all remaining estimated grade blocks were classified as Inferred.

Pit-constrained Mineral Resources have been estimated within an optimized pit shell for the purpose of reporting Mineral Resources and includes Indicated and Inferred Mineral Resources. The pit-constrained Mineral Resources are reported using a NSR cut-off value of US\$95/t. Out-of-pit Mineral Resources are reported beneath the pit shell which exhibit historical continuity and reasonable potential for extraction by longhole mining methods. Out-of-pit Mineral Resources are reported using an NSR cut-off of US\$125/t.

Table 1-1 Boumadine MRE as of February 24, 2025

Boumadine MRE as of February 24, 2025 ⁽¹⁻¹²⁾																	
	Cut-off NSR US\$/t	Tonnes (kt)	Average Grade							Contained Metal							
			Ag (g/t)	Au (g/t)	Cu (%)	Pb (%)	Zn (%)	AgEq (g/t)	AuEq (g/t)	Ag (koz)	Au (koz)	Cu (kt)	Pb (kt)	Zn (kt)	AgEq (koz)	AuEq (koz)	
Pit-Constrained																	
Indicated	95	3,920	94.3	2.99	0.13	0.84	2.95	476.5	5.3	11,881	377	5.1	33	116	60,051	667	
Inferred	95	14,258	89.7	2.89	0.1	0.81	2.38	450	5	41,135	1,325	14.3	115	339	206,293	2,293	
Out-of-Pit																	
Indicated	125	1,249	80.1	2.11	0.08	0.87	2.32	358.2	3.98	3,216	85	1	11	29	14,382	160	
Inferred	125	14,938	74.3	2.39	0.07	0.82	1.85	356.9	3.97	35,669	1,148	10.5	122	276	171,393	1,905	
Total																	
Indicated	95/125	5,169	90.8	2.78	0.12	0.85	2.8	447.9	4.981	15,097	462	6.1	44	145	74,433	827	
Inferred	95/125	29,196	81.8	2.63	0.08	0.82	2.11	402.4	4.473	76,804	2,473	24.8	237	615	377,686	4,198	

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 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.
4. A silver price of US\$24/oz with a process recovery of 89%, a gold price of US\$2,200/oz with a process recovery of 85%, a zinc price of US\$1.20/lb with a process recovery of 72%, a lead price of US\$1.00/lb with a process recovery of 85%, and a copper price of US\$4.00/lb with a process recovery of 75% were used.
5. $AgEq = Ag(g/t) + (Au(g/t) * Au\ price/gram * Au\ recovery) / (Ag\ price/gram * Ag\ recovery) + Zn(\%) * Zn\ price/lb * Zn\ recovery / (Ag\ price/gram * Ag\ recovery) * 685.7147973 + Pb(\%) * Pb\ price/lb * Pb\ recovery / (Ag\ price/gram * Ag\ recovery) * 685.7147973 + Cu(\%) * Cu\ price/lb * Cu\ recovery / (Ag\ price/gram * Ag\ recovery) * 685.7147973$
6. $AuEq = Au(g/t) + (Ag(g/t) * Ag\ price/gram * Ag\ recovery) / (Au\ price/gram * Au\ recovery) + Zn(\%) * Zn\ price/lb * Zn\ recovery / (Au\ price/gram * Au\ recovery) * 685.7147973 + Pb(\%) * Pb\ price/lb * Pb\ recovery / (Au\ price/gram * Au\ recovery) * 685.7147973 + Cu(\%) * Cu\ price/lb * Cu\ recovery / (Au\ price/gram * Au\ recovery) * 685.7147973$
7. The constraining pit optimization parameters were US\$3.5/t for mineralized material mining. US\$2/t for waste mining US\$89/t for processing and US\$6/t for G&A totalling US\$95/t for a cut-off and 50-degree pit slopes
8. The out-of-pit parameters used a US\$30/t mining cost, US\$89/t processing cost and US\$6/t G&A totalling US\$125/t for a cut-off The out-of-pit Mineral Resource grade blocks were quantified above the US\$125 NSR cut-off, below the constraining pit shell and within the constraining mineralized wireframes. Out-of-pit Mineral Resources exhibit continuity and reasonable potential for extraction by the long hole underground mining method.
9. Individual calculations in tables and totals may not sum due to rounding of original numbers.
10. Grade capping of 800 g/t Ag, 30 g/t Au, 28% Zn, 10% Pb and 1.4% Cu was applied to composites before grade estimation.
11. Bulk density was evaluated separately for each individual vein with values ranging from 3.20 to 4.00 t/m³ determined from drill core samples and used for the MRE. For oxidized and transitional material, a bulk density of 2.65 t/m³ was used.
12. 1.0 m composites were used during grade estimation.

Table 1-2 Cut-Off Sensitivity MRE ⁽¹⁻¹²⁾

Indicated & Inferred Out-of-Pit Resources												
Cutoff	Tonnes	Ag	Ag	Au	Au	Cu	Pb	Zn	AgEq	AgEq	AuEq	AuEq
NSR US\$/t	(kt)	(g/t)	(koz)	(g/t)	(koz)	(%)	(%)	(%)	(g/t)	(koz)	(g/t)	(koz)
145	12,476	83	33,164	2.45	985	0.07	0.91	2.03	394	157,918	4.38	1,755
140	13,331	80	34,430	2.40	1,028	0.07	0.88	1.98	384	164,578	4.27	1,829
135	14,159	79	35,820	2.34	1,064	0.07	0.87	1.96	376	170,951	4.17	1,900
130	15,098	77	37,306	2.27	1,103	0.07	0.85	1.94	367	177,957	4.08	1,978
125	16,186	75	38,885	2.21	1,148	0.07	0.82	1.91	357	185,775	3.97	2,065
120	17,311	73	40,374	2.15	1,194	0.07	0.80	1.86	347	193,392	3.86	2,150
115	18,376	71	41,803	2.09	1,235	0.06	0.79	1.83	339	200,397	3.77	2,227
110	19,289	69	42,987	2.04	1,267	0.06	0.78	1.80	332	206,197	3.70	2,292
80	20,539	68	44,787	1.98	1,306	0.06	0.77	1.77	324	213,754	3.60	2,376
Indicated & Inferred In-Pit Resources												
Cutoff	Tonnes	Ag	Ag	Au	Au	Cu	Pb	Zn	AgEq	AgEq	AuEq	AuEq
NSR US\$/t	(kt)	(g/t)	(koz)	(g/t)	(koz)	(%)	(%)	(%)	(g/t)	(koz)	(g/t)	(koz)
120	16,018	98	50,290	3.21	1,653	0.11	0.84	2.43	490	252,356	5.45	2,805
115	16,423	96	50,830	3.16	1,666	0.11	0.84	2.43	483	255,215	5.37	2,837
110	16,850	95	51,366	3.10	1,679	0.11	0.84	2.42	476	258,136	5.30	2,869
105	17,373	93	52,000	3.03	1,695	0.11	0.83	2.41	468	261,474	5.20	2,906
100	17,792	92	52,504	2.98	1,707	0.10	0.82	2.40	462	264,055	5.13	2,935
95	18,178	91	53,016	2.94	1,716	0.10	0.82	2.39	456	266,344	5.07	2,961
90	18,504	90	53,404	2.90	1,723	0.10	0.81	2.38	451	268,214	5.01	2,981
85	18,885	89	53,811	2.85	1,730	0.10	0.81	2.38	445	270,310	4.95	3,005
80	19,550	87	54,616	2.77	1,739	0.10	0.81	2.38	436	273,966	4.84	3,045
75	19,977	86	55,082	2.72	1,745	0.10	0.80	2.38	430	276,129	4.78	3,069
70	20,491	84	55,570	2.66	1,753	0.10	0.80	2.37	423	278,585	4.70	3,097

Notes 1-12 listed below Table 1.1.

1.10 MINING OPERATIONS

The Boumadine deposit is divided into three main zones, the Center, North, and South. Mining will commence with Open Pit (OP) operations, followed by the introduction of Underground (UG) mining starting the third year of the project. Open pit production is expected to contribute 62.6% of the total mined tonnage, while UG mining will account for the remaining 37.4%. The current Life-of-Mine (LOM) spans 12 years, with OP mining scheduled from Year 1 through Year 10, and UG mining beginning in Year 3 and continuing through Year 12.

1.10.1 OP Mining

OP mining is scheduled to begin pre-stripping in Year 0, with mineralized material production ramping up to production by Year 1. Initial OP production will be phased to target near surface mineralized material and initial waste removal.

The mining method selected is standard truck and shovel pairing with a contractor fleet selected for all OP operations. The mining operation will be handled by a mining contractor. The pit roads will be 20 metres (m) wide for a double lane road at a maximum grade of 10%. The waste material will be stored in proximity to the pits in waste rock dumps with a maximum height of 100 m.

1.10.2 UG Mining

UG mining is scheduled to begin two years after the start of OP operations. This phased approach is intended to defer the capital expenditure associated with UG development and enhance the project's economic viability, provided that OP production remains sufficient to meet the plant's feed requirements. Initial UG production will focus on the highest-grade zones, particularly in the Center and North areas. Access to these zones will be established via surface ramps.

Based on the geometry and geotechnical conditions of the deposit, the selected mining method is modified Avoca, a variation of the longhole stoping method that uses rock fill as backfill. Stopes are designed with dimensions of 2 m minimum width, 20 m in height, and 20 m in length. Mined material will be hauled by 30-ton trucks, via a ramp system, to three designated surface stockpiles.

The various zones of the deposit will be accessed via ramps driven either from surface or from the existing OPs. Additional lateral development will include level access drifts, production (sill) drifts, and other necessary excavations to accommodate mine infrastructure. Vertical development will consist of ventilation raises and rock passes, which are planned to be excavated, using either raise boring, or conventional raise mining methods.

Capital and operating lateral and vertical development will be performed by a contractor, while all subsequent mine production activities will be performed by owner crews.

1.11 PROCESSING AND RECOVERY OPTIONS

The process plant design for the Project is based on a conventional flotation flowsheet. Lead, zinc and pyrite concentrates will be recovered and sold to market.

The plant design criteria were selected to minimize operating costs and maximize the use of proven technology. The key criteria for equipment selection are suitability for duty, safety, reliability, and ease of maintenance.

The process plant design comprises the following unit operations:

- Single stage primary crushing with a jaw crusher to produce an 80% Passing (P_{80}) crushed product size of 136 mm.
- Crushed rock stockpile with a live capacity of approximately 8,115 tonnes to provide 24 hours of live storage. During extended periods of primary crusher equipment maintenance, additional crushed material inventory can be generated in the weeks leading up to the planned shutdown by dozing

crushed material from this stockpile to the area adjacent to the stockpile. This material can then be reclaimed during the shutdown by front-end loader to feed the grinding circuit.

- SAB type grinding circuit consisting of a SAG mill and a ball mill to produce a ground product with a P_{80} of 58 microns (μm), with hydrocyclones for particle size classification.
- Lead rougher flotation, classification, regrinding and cleaner flotation to produce a lead concentrate of 29.6% grade. The lead concentrate will be thickened, filtered, and bagged prior to shipment.
- Zinc rougher flotation, classification, regrinding and cleaner flotation to produce a zinc concentrate of 57.4% grade. The zinc concentrate will be thickened and filtered for stockpiling prior to shipment.
- Pyrite rougher flotation to produce a pyrite concentrate with 4.2 Au g/t and 81.0 Ag g/t. The pyrite concentrate will be thickened and filtered for stockpiling prior to shipment.
- Tailings thickening and storage in a tailings management facility.

Run-of-mine (ROM) material will be delivered to a feed bin by mine truck to supply the crushing plant. The crushed product will be stockpiled before entering the grinding circuit, which will include a SAG mill, ball mill, and hydrocyclones for particle size classification. The ground product will feed the lead rougher flotation circuit. The lead rougher concentrate will proceed to a classification and regrind circuit, followed by three cleaner flotation stages and a cleaner-scavenger stage. The final lead concentrate will be thickened, filtered and bagged for shipment.

The lead rougher tails and cleaner-scavenger tails will feed the zinc rougher flotation circuit. The zinc rougher concentrate will follow a similar regrind and cleaning process, including three cleaning stages and a cleaner-scavenger stage. The final zinc concentrate will be thickened, filtered, and stockpiled for shipment.

The zinc rougher tails and cleaner-scavenger tails will feed the pyrite rougher flotation circuit. The pyrite rougher tails will be thickened and pumped to the flotation tailings management facility. The pyrite concentrate will be thickened, filtered and stockpiled for shipment. Filtrate and thickener overflows from all circuits will return to the process water pond.

1.12 INFRASTRUCTURE

Infrastructure on the Project includes an integrated mining and processing operation including open pits, an underground mine, and a central processing plant supported by crushing, flotation, stockpiling, and concentrate handling facilities. Site access is via the National 10 highway with upgraded roads connecting all major areas.

Key infrastructure includes the ROM pad, workshops, laboratories, security and administration buildings, and extensive haul and service roads. Waste rock storage, a lined TSF with staged raises, and on-site water harvesting and treatment support operational needs. Power is provided through a new 225 kV ONEE supply to a 40 MVA substation. Additional services include fuel storage, sewage treatment, waste management, security, limited accommodation, and concentrate transport to Nador-West Port.

1.13 CAPITAL AND OPERATING COSTS

1.13.1 Capital Costs

The capital cost estimates for the Boumadine Project PEA was compiled by Lycopodium. All costs are expressed in United States Dollars (US\$) unless otherwise stated and are based on Q4 2025 pricing and deemed to have an overall accuracy of +50%/-30%. The capital cost estimate conforms to AACE International (Association for the Advancement of Cost Engineering) Class 5 estimate standards as prescribed in recommended practice 47R11.

The total initial capital cost for the Boumadine Project is \$446 M and the LOM sustaining cost is \$340 M. The initial capital and sustaining capital, for the Project is summarized in Table 1-3 by major area.

Table 1-3 Capital Expenditures including Sustaining and Closing Costs

Capital Expenditures (\$M)	Initial	Sustaining	Total
Direct Costs	288	340	628
Open Pit Mining	54	58	112
Underground Mining	-	250	250
Processing Plant	167	-	167
Shipping Infrastructure	11	-	11
Electrical Line	17	-	17
Raw Water Supply	30	-	30
Tailings Storage Facility	9	22	31
TSF Closure Costs	-	9	9
Indirects Costs	63	-	63
Subtotal	351	340	691
Contingency	96	-	96
Total	446	340	786

1.13.2 Operating Costs

The operating cost estimate conforms to a preliminary economic assessment level estimate with a +50/-30% accuracy. The operating cost estimate was developed in Q4 2025 using data from projects, studies, and previous operations from Lycopodium, WSP, and Aya.

The LOM average unit operating cost is \$66.16/t and the total LOM cost including cash cost and sustaining cost is \$119.72/t. Table 1-4 provides a summary of the operating costs for the Project.

Table 1-4 Operating Costs for First 5 Years of Operation and for LOM

Item	Units	Year 1-5	LOM
Mining	\$/t milled	48.93	42.83
Processing	\$/t milled	17.28	17.28
G&A	\$/t milled	5.43	5.58
Tailings, Environmental and Water Management	\$/t milled	0.46	0.48
Total On-Site Operating Cost	\$/t milled	72.10	66.16
Transportation	\$/t milled	38.70	35.56
Royalties	\$/t milled	8.52	6.75
Mining Tax	\$/t milled	0.36	0.32
Total Cash Cost	\$/t milled	119.68	108.78
OP Sustaining Capital	\$/t milled	3.06	1.87
UG Sustaining Capital	\$/t milled	7.95	8.06
TSF Sustaining Capital	\$/t milled	0.95	1.01
Total Costs Including Sustaining	\$/t milled	131.65	119.72

1.14 ECONOMIC ANALYSIS

The economic analysis was performed assuming a 5% discount rate typical for silver-gold projects. Cash flows have been discounted to the start of construction, assuming that the project execution decision will be made, and major project financing will be carried out at this time.

The pre-tax NPV discounted at 5% is \$2.2B, with a pre-tax IRR of 69% and payback period of 1.3 years. On an after-tax basis, the NPV discounted at 5% is \$1.5B, with an after-tax IRR of 47%, and payback period of 2.1 years. Cumulative after-tax unlevered free cash flow totals \$2.8B. Tax calculations are based on Aya's understanding of current Moroccan tax regulations as of the effective date of this report.

Readers are cautioned that the PEA is preliminary in nature. It includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves and there is no certainty that the PEA will be realized. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

A summary of the project economics is listed in Table 1-5 and Table 1-6, and after-tax free cash flow is shown graphically in Figure 1-1.

Table 1-5 Key Economic Results

Project Economics	Units	Base Case	
		Pre-tax	Post-tax
NPV _{5%}	\$B	2.2	1.5
IRR	%	69%	47%
Payback	Years	1.3	2.1
NPV: Capex ¹	-	5.0	3.3
Revenue LOM	\$B	7.0	0.0
Avg. Annual Revenue	\$M/y	629	-
EBITDA LOM	\$B	3.4	-
Avg. Annual EBITDA	\$M/y	308	-
Cumulative FCF LOM	\$B	2.8	2.0
Avg. Annual FCF	\$M/y	254	176

1. NPV:Capex ratio is the ratio of Net Present Values, discounted at 5%, to the initial capital expenditure.

Table 1-6 Additional Economic Model Outputs

Item	Units	Year 1-5	LOM
Processed Grade			
Au	g/t	3.15	2.43
Ag	g/t	85.8	72.5
Zn	%	2.05	1.91
Pb	%	0.66	0.70
Au-eq	g/t	4.76	3.85
Ag-eq	g/t	443	358
Recoveries			
Au rec (%)	%	96.1%	96.1%
Ag Rec (%)	%	96.4%	96.4%
Zn rec (%)	%	74.7%	74.7%
Pb Rec (%)	%	82.0%	82.0%
Metal Production			
Gold Production	koz	1,351	2,337
Silver Production	koz	36,894	69,874

Item	Units	Year 1-5	LOM
Zinc Production	MIbs	468	975
Lead Production	MIbs	166	392
Gold-Equivalent Production (Au-eq)	koz	2,006	3,643
Silver-Equivalent Production (Ag-eq)	koz	187,261	340,038
Avg. Annual AuEq Production	koz/y	401	328
Avg. Annual AgEq Production	koz/y	37,452	30,611
Operating Cost per Ounce			
Total Cash Costs ¹	\$/oz AuEq	827	928
Total AISC ²	\$/oz AuEq	910	1021
Total Cash Costs ¹	\$/oz AgEq	8.9	9.9
Total AISC ²	\$/oz AgEq	9.8	10.9

1. Cash costs include mine-site operating costs such as mining, processing, and direct site G&A, as well as product shipping, royalties and mining taxes.
2. AISC is calculated as the sum of treatment and refining charges, onsite operating costs, sustaining capital costs, and closure costs, divided by the quantity of ounces equivalent produced.

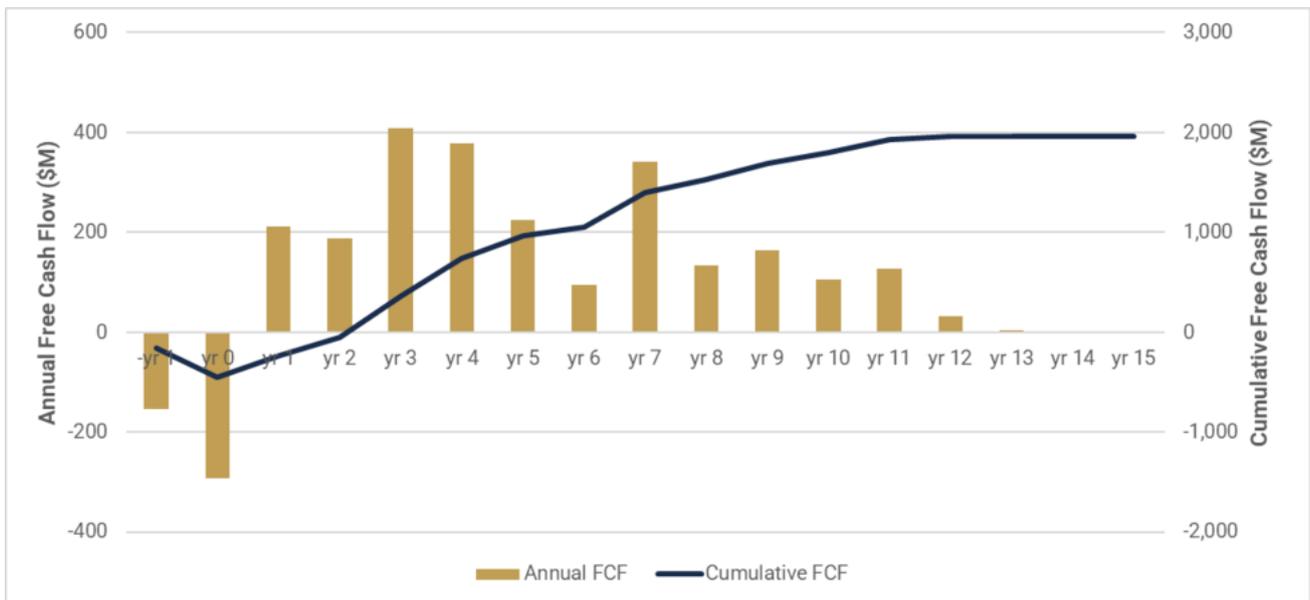


Figure 1-1 Boumadine Project - Annual Free Cash Flow Profile

1.15 ENVIRONMENTAL STUDIES, PERMITS AND SOCIAL OR COMMUNITY IMPACT

The Project is located in the Drâa-Tafilalet region of western Morocco, approximately 19 km south of the nearest town of Tinejdad. The biophysical and social characteristics of the Project area are defined by its semi-arid climate and relatively rural and remote nature. Urban settlements are concentrated along main transport routes with villages located along valleys with access to water resources. Climatic conditions influence local ecosystems and land uses including agriculture, concentrated in areas of water availability.

Limited environmental studies have been completed in the past. As part of the environmental and social work currently being undertaken, an environmental baseline sampling program was initiated by BGM in September 2024 and a high-level screening for critical biodiversity habitat was undertaken by SLR in September 2024.

The baseline sampling, covering soils, air quality and water, was undertaken by AfriLabs, a Moroccan-based laboratory specializing in mineral and environmental analysis.

Additional baseline sampling is planned together with subsequent E&S studies as part of the international ESIA process outlined below. These studies will cover the topics of biodiversity, hydrology, hydrogeology, soils and sediments, air quality, noise, GHG, climate change, stakeholder engagement, labour and working conditions, community health and safety, and other social elements.

The Project will require national permits to support the development of the mine and the requirement to carry out an Environmental Impact Assessment under Moroccan Law. In addition to meeting national permitting requirements, the Project is being undertaken to meet international ESIA lender standards. At the time of writing, the Project is yet to initiate the national EIA permitting process. The plan is to initiate this process in 2026, at the same time as undertaking an ESIA to international lender standards.

A conceptual mine closure plan governed by Aya's HSEC Policy will be prepared as part of the ESIA.

1.16 CONCLUSIONS AND RECOMMENDATIONS

The Mineral Resources currently estimated for the Boumadine Polymetallic Project consist of:

- An Indicated Mineral Resource of 5.2 Mt grading 91 g/t Ag, 2.78 g/t Au, 2.8% Zn and 0.85% Pb containing an estimated 15.1 Moz of Ag, 462 koz of Au, 145 kt of Zn and 44 kt of Pb;
- An Inferred Mineral Resource of 29.2 Mt grading 82 g/t Ag, 2.63 g/t Au, 2.11% Zn and 0.82% Pb containing an estimated 76.8 Moz of Ag, 2.5 Moz of Au, 615 kt of Zn and 237 kt of Pb;

The PEA provides a base case assessment for developing the Boumadine mineral resource by conventional open pit and underground mining methods, and a crushing-milling-flotation processing plant with a tailings dam.

The PEA economic analysis shows the Boumadine Polymetallic Project has an after-tax NPV5% of \$1.5B, IRR of 47%, and a payback period of 2.1 years. The PEA economics support a decision to continue to advance the Project and carry out additional detailed studies, including pre-feasibility level work and completion of a feasibility study.

Additional expenditures are recommended by the Boumadine Report Authors for the following activities:

- Drilling to advance Inferred to Indicated Mineral Resources.
- Drilling down-dip to develop additional Mineral Resources at depth.
- Follow-up geological mapping, mineral prospecting, and assays.
- Development of a comprehensive bulk density model.
- Investigate grade capping thresholds by individual mineralized domain.
- Review grade anisotropy by individual mineralized domain.
- Advance metallurgical testwork, including spatial composites and variability samples.
- Undertake detailed studies for infrastructure, including TSF design, raw water sourcing, electrical line connection and concentrate logistics studies.
- Update and upgrade mining and processing designs through pre-feasibility and feasibility level designs.
- Conduct hydrogeology and geotechnical studies to improve design parameters for mining.
- Complete the ESIA, which will inform a comprehensive Environmental and Social Management and Monitoring Plan to be developed for the Project.

The Boumadine Report Authors also recommend that Aya continues with the current QC protocol and monitor QC data and continue refining the metallurgical testwork for improved process recoveries.

This estimated cost of the recommended work program is US\$109.0M, which includes 10% contingency (without applicable taxes). This is shown in Table 1-7.

Table 1-7 Recommended Programs and Budgets for 2026-2027

Year	Item	Activity	Unit (m)	Cost Estimate (\$M, USD)
Exploration Phase 1 - 2026				
2026		Drilling (all-in costs)	200,000	42.0
		Administration and Management		4.2
		Sub-Total		46.2
		Contingency (10%)		4.6
		Total - 2026		50.8
Exploration Phase 2 - 2027				
2027		Drilling (all-in costs)	200,000	42.0
		Administration and Management		4.2
		Sub-Total		46.2
		Contingency (10%)		4.6
		Total - 2026		50.8
Feasibility Study - 2026-2027				
2026		Feasibility Pre-Work and Feasibility Study Commencement		5.9
2027		Feasibility Study Completion		0.7
		Sub-Total		6.7
		Contingency (10%)		0.7
		Total - FS		7.3

2.0 INTRODUCTION AND TERMS OF REFERENCE

2.1 TERMS OF REFERENCE

This Technical Report on the Boumadine (also known as Boumaadine, Boumâadine, and Bou Madine) Deposit was prepared by Aya Gold and Silver Inc. (“Aya” or the “Company”) at the request of Mr. Benoit La Salle, President & CEO of Aya Gold & Silver Inc. Aya is a public, TSX listed company trading under the symbol “AYA”, with its head office located at: Suite 132, 1320 Graham, Ville Mont-Royal, Québec, H3P 3C8. This Technical Report has an effective date of November 4, 2025.

This Technical Report (“the Report”) has been prepared to provide a fully compliant NI 43-101 Technical Report and Preliminary Economic Assessment (“PEA”) of the existing mineralization at the Boumadine Deposit (or the “Boumadine Property” or the “Boumadine Mining License”), located in the Kingdom of Morocco. The Boumadine Mining License is a joint venture owned by Aya (85%) and l’Office National des Hydrocarbures et des Mines (“ONHYM”) (15%) of the Kingdom of Morocco. Aya (known as Maya Gold & Silver at the time of the Acquisition Agreement) and Office National des Hydrocarbures et des Mines (“ONHYM”) signed an agreement on October 9, 2012, for the acquisition of the Boumadine Mining License. A new Moroccan company, Boumadine Global Mining (“BGM”), was created with Aya (85%) and ONHYM (15%) as shareholders. The mining title of the Boumadine Mining License was transferred to BGM by ONHYM.

The name change from Maya Gold & Silver Inc. to Aya Gold & Silver Inc. was announced in a press release dated July 30, 2020. The Updated Mineral Resource Estimate reported herein is based on up-to-date drilling results and appropriate metal pricing, and is fully conformable to the “CIM Standards on Mineral Resources and Reserves – Definitions (2014) and Best Practices Guidelines (2019)”, as referred to in National Instrument (“NI”) 43-101 and Form 43-101F, Standards of Disclosure for Mineral Projects.

2.2 SOURCES OF INFORMATION

2.2.1 Site Visit

Mr. Antoine Yassa, P. Geo., of P&E and an independent Qualified Person under the terms of NI 43-101, completed a site visit to the Boumadine Property from March 12 to 14, 2024. A data verification sampling program was completed on-site (see Section 12 of this Report). Mr. Yassa is a professional geologist with ~45 years of experience in mineral exploration and mining operations, including several years working in precious metal and polymetallic deposits.

Two additional independent Qualified Persons have visited the site. Cortney Palleske, M.A.Sc., P.Eng., visited the Boumadine Property from June 5 to 7, 2024. Alex Pheiffer, PrSciNat, visited the site from September 1 to 3, 2024.

2.2.2 Additional Information Sources

This Report is based, in part, on internal Company technical reports, and maps, published government reports, company letters and memoranda, and public information as listed in the references (Section 27) of this Report. Several sections from reports authored by other consultants have been directly quoted or summarized in this Technical Report, and are cited where appropriate.

The Authors of this Report have used selected portions or excerpts from material contained in the following NI 43-101 compliant Technical Reports, which are publicly available on SEDAR+ under Aya’s profile:

- Boily, M. 2014. The Boumadine Polymetallic (Au, Ag, Zn, Pb, Cu) Deposit, Errachidia Province, Kingdom of Morocco. Prepared for Maya Gold & Silver Inc. by Boily, Michel. Effective Date April 2, 2014. Issue Date April 2, 2014. 217 pages.
- GoldMinds Geoservices Inc. 2019. NI 43-101 Technical Report Preliminary Economic Assessment [of the] Boumadine Polymetallic Deposit, Kingdom of Morocco. Prepared for Maya Gold & Silver Inc. by

Duplessis, C., Rachidi, M., Dufort, D. and Rousseau, G. Effective Date: April 24, 2019. Issue Date: May 24, 2019. 304 p.; and

- P&E Mining Consultants Inc. 2024. NI 43-101 Technical Report and updated Mineral Resource Estimate of the Boumadine Polymetallic Project, Kingdom of Morocco. Prepared for Aya Gold & Silver Inc. by Stone, W., Brown, F., Barry, J., Yassa, A., Feasby, G. and Puritch, E. Effective Date: May 08, 2024. Issue Date: May 31, 2024. 214 p.;
- Aya Gold & Silver 2025. NI 43-101 Technical Report and Updated Mineral Resource Estimate of the Boumadine Polymetallic Project, Kingdom of Morocco. Prepared for Aya Gold & Silver Inc. by Lalonde, D, and Pérez, P. Effective Date: February 24, 2025. Issue Date : March 31, 2025. 193 p.

Since 2024, Aya has continued extensive mineral exploration and drilling programs. The results of these programs were disclosed publicly in numerous Aya press releases and are summarized in Sections 9 and 10 of this Report.

The Authors and Co-authors of each section of this Report are listed in Table 2-1, who in acting as Qualified Persons as defined by NI 43-101, take responsibility for those sections of this Report as outlined in the “Certificate of Author” included in Section 28 of this Report.

Table 2-1 Qualified Persons Responsible for this Report

Qualified Person	Company	Sections of Technical Report
Preetham Nayak, P.Eng.	Lycopodium Minerals Canada Ltd	2, 3.2, 19, 21.1.1, 21.1.3, 21.1.6-21.1.10, 21.2.4, 22, 24.1 and Co-Author 1, 25, 26, 27
Ruan Venter, P.Eng.	Lycopodium Minerals Canada Ltd	13, 17, 21.2.2 - 21.2.3, 24.2 and Co-Author 1, 25, 26, 27
Zuned Shaikh P.Eng.	Lycopodium Minerals Canada Ltd	18.1 to 18.3, 18.6 to 18.13, and Co-Author 1, 25, 27
Benjamin Berson, P.Eng.	WSP Canada Inc.	16.1, 16.3-16.12, 18.4, 21.1.2, 21.2.1 and Co-Author 1, 25, 26, 27
Alex Pheiffer, PrSciNat	SLR Consulting France SAS	20 and Co-Author 1, 25, 26, 27
George Papageorgiou PrEng, PhD, MSc, BSc, Eng (Civil), Wits	Epoch Resources (Pty) Ltd	18.5, 21.1.4-21.1.5, 21.2.5, and Co-Author 1, 25, 26, 27
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Cortney Palleske, M.A.Sc., P.Eng.	RockEng Inc.	16.2 and Co-Author 25, 27

2.3 UNITS AND CURRENCY

In this Technical Report, all currency amounts are stated in US dollars (“\$”) unless otherwise stated. Commodity prices are typically expressed in US dollars (“US\$”) and will be duly noted where appropriate. Quantities are generally stated in Système International d’Unités (“SI”) metric units including metric tons (“tonnes”, “t”) and kilograms (“kg”) for weight, kilometres (“km”) or metres (“m”) for distance, hectares (“ha”) for area, grams (“g”) and grams per tonne (“g/t”) for metal grades. Platinum group metal (“PGM”), gold and silver grades may also be reported in parts per million (“ppm”) or parts per billion (“ppb”). Copper metal values are reported in percentage (“%”) and parts per billion (“ppb”). Quantities of PGM, gold and silver may also be reported in troy ounces (“oz”), and quantities of copper in pounds (“lb”). Abbreviations and terminology are summarized in Table 2-2 and units in Table 2-3.

Grid coordinates for maps are given in the UTM WGS 84 Zone 30 R or as longitude and latitude.

Table 2-2 Terminology and Abbreviations

Abbreviation	Meaning
\$	dollar(s)
°	degree(s)
°C	degrees Celsius
<	less than
>	greater than
%	percent
3-D	three-dimensional
AAS	atomic absorption spectrometry
Actlabs	Activation Laboratories Ltd.
Ag	silver
AgEq	silver equivalency
ALS	ALS Laboratory (Australian Laboratory Services), ALS Limited
ANCOLD	Australian National Committee on Large Dams
As	arsenic
Au	gold
AuEq	gold equivalency
avg	average
Aya	Aya Gold & Silver Inc.
Ba	barium
BGM	Boumadine Global Mining
BRGM	Bureau des Recherches Géologiques et Minières
BRPM	Bureau de Recherches et de Participations Minières
BWI	ball mill work index
CaO	calcium oxide
Cd	cadmium
CDA	Canadian Dam Association
CGG	Compagnie Générale de Géophysique
CIL	carbon-in-leach
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
cm	centimetre(s)
CN	cyanide
Co	cobalt
Company, the	Aya Gold & Silver Inc.
CRI	Comité Régional d'Investissement (centre of investment)
CRM(s)	certified reference material(s)
CSA	Canadian Securities Administrators
CSAMT	controlled-source audio magnetotelluric survey
Cu	copper
d	day
D ₈₀	80% passing size
DAR	Direction des Affaires Rurales
Deposit, the	Boumadine Deposit
E	east
EBRD	European Bank for Reconstruction and Development
ESMS	environmental and social management system
ESIA	Environmental and Social Impact Assessment

Abbreviation	Meaning
FA	fire assay
Fe	iron
g	gram
G&A	General and administration
g/L	grams per litre
g/t	grams of metal per tonne
GISTM	Global Industry Standard on Tailings Management
GoldMinds	GoldMinds Geoservices Inc.
ha	hectare(s)
HC	Hot Curing
HSEC	Health and Safety, Environment and Community
ICOLD	International Commission on Large Dams
ICP	inductively coupled plasma
ICP-OES	inductively coupled plasma-optical emission spectroscopy
ID	identification
ID²	inverse distance squared
IFC	International Finance Corporation
ISO	International Organization for Standardization
ISO/IEC	International Organization for Standardization/International Electrotechnical Commission
k	thousand(s)
kg	kilogram(s)
kg/t	kilogram(s) per tonne
km	kilometre(s)
km²	square kilometre(s)
koz	thousands of ounces
kt	kilotonne(s), thousands of tonnes
kWh/t	kilowatt hour per tonne
L	litre(s)
lb	pound(s) (weight)
LB	Lime Boiling
LCT(s)	locked cycle testwork(s)
level	mine working level referring to the nominal elevation (m RL), e.g. 4285 level (mine workings at 4285 m RL)
LoM	life of mine
M	million(s)
m	metre(s)
m³	metres cubed
m asl	metres above sea level
Ma	millions of years
MAD	Moroccan Dirham
mag	magnetic(s)
max.	maximum
Maya	Maya Gold and Silver Inc.
mbs or MBS	metres below surface
Mg	magnesium
min.	minimum
mL	millilitre(s)
mm	millimetre(s)

Abbreviation	Meaning
Mn	manganese
Moz	million ounce(s)
MRE	Mineral Resource Estimate
Mt	million(s) tonnes per year
MTEDD	le ministère de transition énergétique et durable or minister of durable energetic transition
N	total number of observations/population size
N	north
N10	National Highway 10
NaCN	sodium cyanide
Ni	nickel
NI or NI 43-101	National Instrument or National Instrument 43-101
NN	Nearest Neighbour
NPV	net present value
NSR	net smelter return
OK	Ordinary Kriging
ONHYM	Office National des Hydrocarbures et des Mines
ORE	ORE Research & Exploration Pty Ltd.
oz	ounce(s)
P₈₀	80% passing size
P&E	P&E Mining Consultants Inc.
PAX	potassium amyl xanthate
PEA	Preliminary Economic Assessment
Pb	lead
P.Eng.	Professional Engineer
P.Geo.	Professional Geoscientist
POX	pressure oxidation
ppm	parts per million
Property, the	Boumadine Property
Q1, Q2, Q3, Q4	first quarter, second quarter, third quarter, fourth quarter of the year
QA	quality assurance
QA/QC	quality assurance / quality control
QC	quality control
R²	the coefficient of determination
Report, the	this Technical Report
RQD	rock quality designation
S	south
S	sulphur
SEDAR	System for Electronic Document Analysis and Retrieval
SGS	SGS Canada Inc. / SGS Lakefield Research
Si	silicon
SIPX	sodium isopropyl xanthate
SODECAT	Société de Développement du Cuivre de l'Anti Atlas
SODIM	Société de Développement de l'industrie maricole
Sr	strontium
t	metric tonne(s)
t/m³	tonnes per cubic metre
Technical Report	(this) NI 43-101 Technical Report

Abbreviation	Meaning
TEM	Transmission Electron Microscopy
TIMA-X	Tescan Integrated Mineral Analyses
TSF	tailings storage facility
TSX-V	Toronto Venture Stock Exchange
TTF	Tamerzaga-Timrachine Formation
URSTM	Unité de recherche et de service en technologie minérale
US\$	United States dollars
UTM	Universal Transverse Mercator
VTEM	Versatile Time Domain Electromagnetic (survey)
W	west
w/w	weight by weight
wireframe	polygons joined together to represent a specific 3-D domain or unit
wt%	weight percent
XRD	x-ray diffraction
Zn	zinc

Table 2-3 Unit Measurement Abbreviations

Abbreviation	Meaning	Abbreviation	Meaning
µm	microns, micrometre	m ³ /h	cubic metre per hour
\$	dollar	m ³ /s	cubic metre per second
\$/t	dollar per metric tonne	m ³ /y	cubic metre per year
%	percent sign	mØ	metre diameter
% w/w	percent solid by weight	m/h	metre per hour
¢/kWh	cent per kilowatt hour	m/s	metre per second
°	degree	Mt	million tonnes
°C	degree Celsius	Mtpy	million tonnes per year
cm	centimetre	min	minute
d	day	min/h	minute per hour
ft	feet	mL	millilitre
GWh	Gigawatt hours	mm	millimetre
g/t	grams per tonne	Mt	million tonnes or megatonnes
h	hour	MV	medium voltage
ha	hectare	MVA	mega volt-ampere
hp	horsepower	MW	megawatts
Hz	hertz	oz	ounce (troy)
k	kilo, thousands	Pa	Pascal
kg	kilogram	pH	Measure of acidity
kg/t	kilogram per metric tonne	ppb	part per billion
kHz	kilohertz	ppm	part per million
km	kilometre	s	second
kPa	kilopascal	t or tonne	metric tonne
kt	thousands or tonnes or kilotonnes	tpd	metric tonne per day
kV	kilovolt	t/h	metric tonne per hour
kW	kilowatt	t/h/m	metric tonne per hour per metre
kWh	kilowatt-hour	t/h/m ²	metric tonne per hour per square metre
kWh/t	kilowatt-hour per metric tonne	t/m	metric tonne per month
L	litre	t/m ²	metric tonne per square metre
L/s	litres per second	t/m ³	metric tonne per cubic metre
lb	pound(s)	T	short ton
M	million	tpy	metric tonnes per year
m	metre	V	volt
m ²	square metre	W	Watt
m ³	cubic metre	wt%	weight percent
m ³ /d	cubic metre per day	yr	year

3.0 RELIANCE ON OTHER EXPERTS

3.1 EXPERT REPORTS

3.1.1 Introduction

The Authors of this Technical Report have relied on the following other expert reports, which provided information regarding mineral rights, surface rights, property agreements and royalties, sections of this Report.

3.1.2 Mineral Tenure and Surface Rights

The Authors have not independently reviewed ownership of the Property area and any underlying mineral tenure, and surface rights. The Authors have fully relied on, and disclaim responsibility for, information derived from Aya, and legal experts retained by Aya for this information through the following document:

- Dentons Morocco June 18, 2025 Legal Opinion / Boumadine Mining Title.

This information is used in Section 4 of this Report.

3.2 RELIANCE ON EXTERNAL REPORTS AND EXPERTS

3.2.1 Logistics

The QP for Section 18.13 and Section 22 has relied upon the report titled « Étude logistique du transport en vrac de minerai », prepared by SCX Technology in October 2025. The QP has not independently verified the information contained in this study and considers such reliance reasonable for the purposes of this Technical Report.

3.2.2 Marketing

Lycopodium has relied on external information for Request for Interest documents provided by Aya and reviewed by Lycopodium.

3.2.3 Taxes

The Qualified Person (QP) responsible for Section 22, has relied upon information, interpretations, and assumptions related to taxation provided by Aya based on current Morocco tax regulations. Lycopodium has not independently verified the taxation inputs and considers such reliance reasonable for the purposes of the Technical Report.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION

The Boumadine Property is located in the Province of Errachidia, Kingdom of Morocco, approximately 220 km east of the City of Ouarzazate and 70 km southwest of the City of Errachidia (Figure 4-1). The historical Boumadine Mine is located at approximately Longitude 4°55'18" West, Latitude 31°24'40" North (and altitude 1,145 m asl), or in UTM WGS 84 Zone 30 R 317,310 m East and 3,476,770 m North.

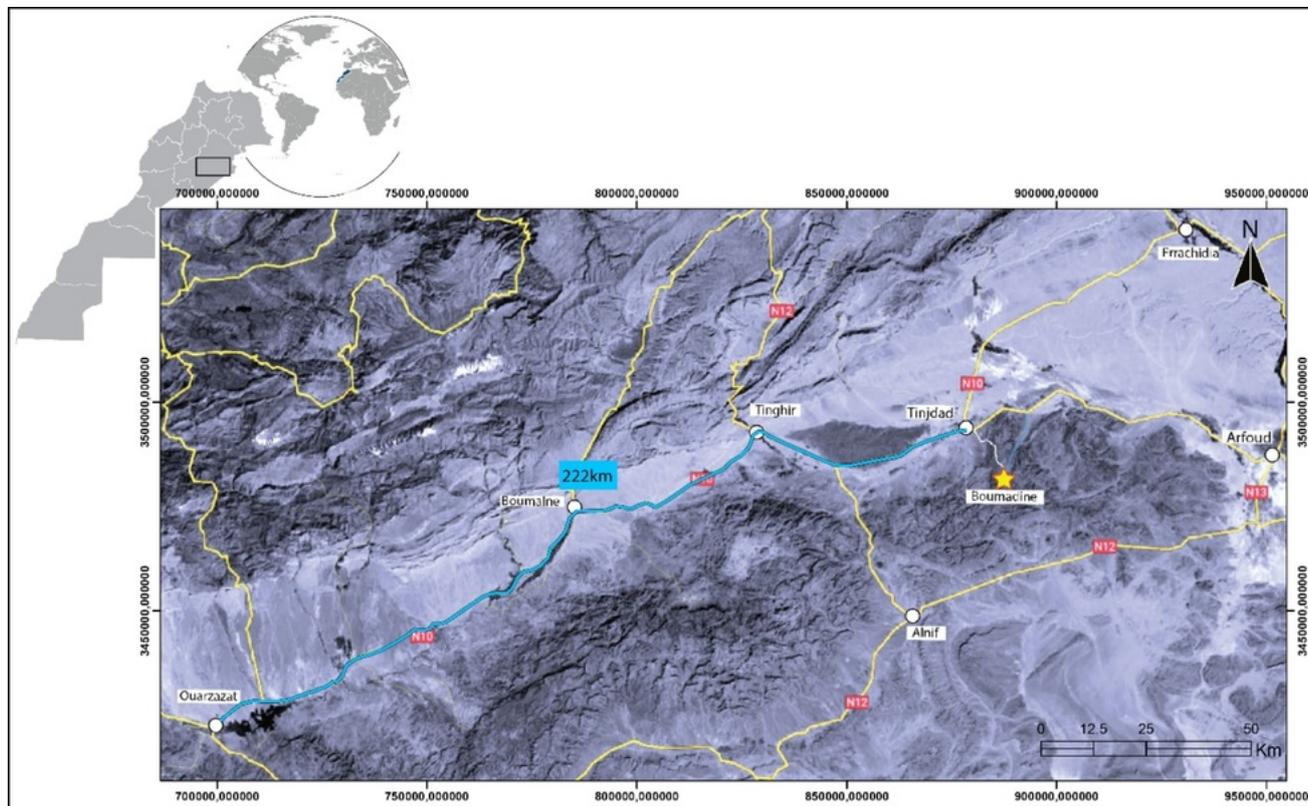


Figure 4-1 Location of the Boumadine Property between Ouarzazate and Errachidia, Kingdom of Morocco

Source: Aya (May 2024)

4.2 PROPERTY DESCRIPTION AND TENURE

Aya's property in the Boumadine area ("Boumadine Property") consists of 12 mining permits and 19 exploration permits totaling 341 km² in size. The "Boumadine Mining License", which contain Boumadine Deposit and is the focus of this Boumadine Report, consists of mining permit LE-383661 and covers the historical Boumadine Mine, the Boumadine Camp, and the current MRE described in this Report. Mining Permit LE-383661 covers a total of 32 km² in area. The additional thirty permits are distributed within a 25 km radius of the Boumadine Deposit and collectively cover an additional 309 km² in area (see Figure 4-2 and Table 4-1). In addition, an Authorization of Exploration of 600 km² was granted to Aya in January 2025.

In addition to its ownership of the Boumadine Mining License, Aya, through its subsidiaries, has: 100% ownership of a total of 11 mining licenses and exploration permits and; an option to earn 100% interest in 19 other mining licenses and exploration permits, as summarized in Table 4-1 below.

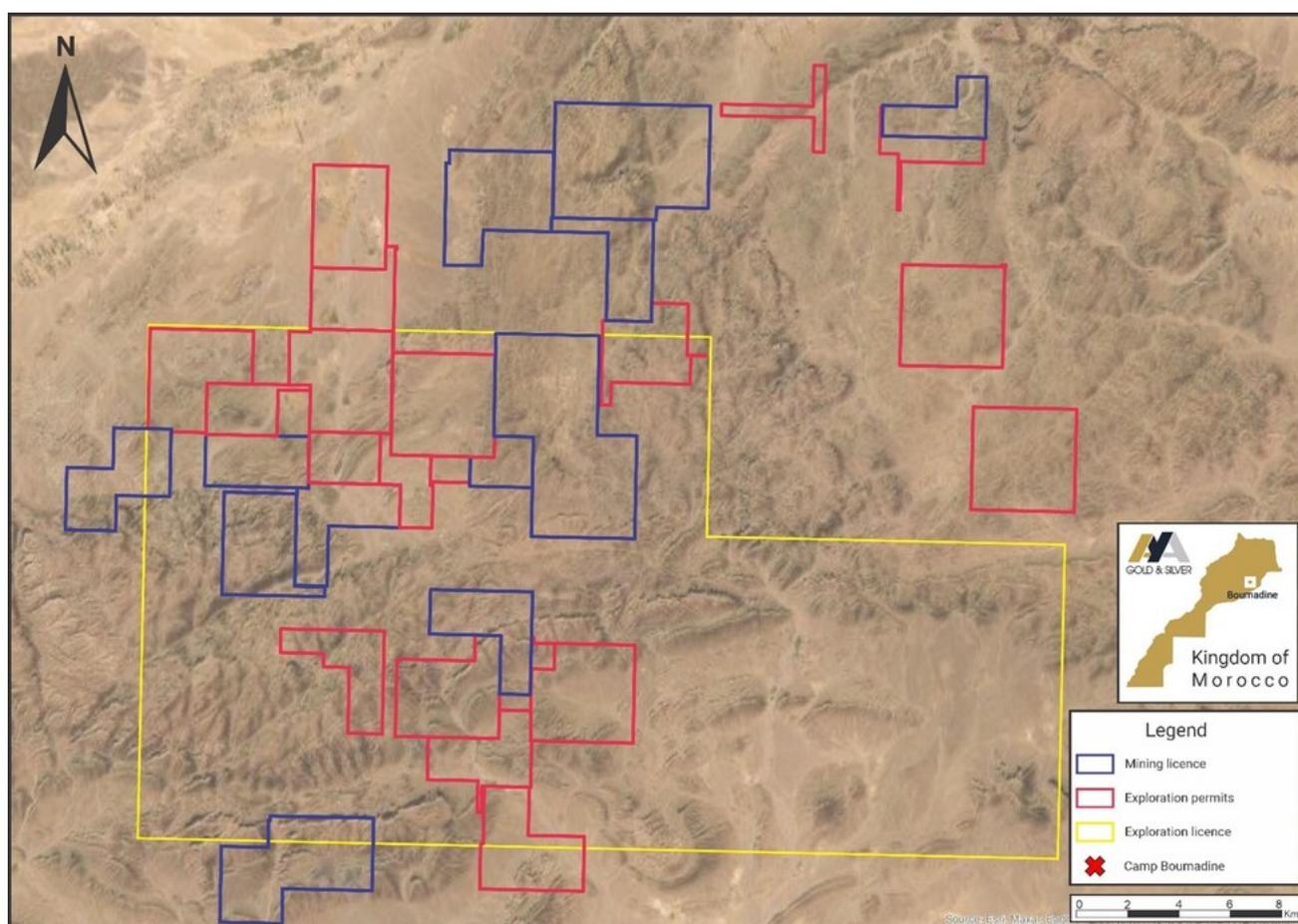


Figure 4-2 Land Tenure in the Boumadine Property Area

Source: This Report

Note: Mineral Tenure information effective June 18, 2025.

Table 4-1 Aya Mining and Exploration Permits in the Boumadine Property Area

Permit ID	Permit Type	Interest of Aya*	Area (km ²)	Granted	Expires
LE-383661*	Licence d'exploitation	85% ownership	31.67	May 17, 2016	May 16, 2026
LE-383631**	Licence d'exploitation	Option to own 100%	10.34	July 13, 2017	July 12, 2027
LE-383657**	Licence d'exploitation	Option to own 100%	8.00	October 18, 2015	October 17, 2025
LE-383685**	Licence d'exploitation	Option to own 100%	14.67	June 18, 2017	June 17, 2027
LE-383692**	Licence d'exploitation	Option to own 100%	4.02	May 14, 2016	May 13, 2026
LE-383722**	Licence d'exploitation	Option to own 100%	9.53	September 27, 2014	September 26, 2024 ⁴
LE-383724**	Licence d'exploitation	Option to own 100%	6.16	November 25, 2017	November 24, 2027
LE-383852**	Licence d'exploitation	Option to own 100%	9.49	October 16, 2018	October 15, 2028
LE-383853**	Licence d'exploitation	Option to own 100%	17.59	December 21, 2018	December 21, 2028
LE-383856**	Licence d'exploitation	Option to own 100%	25.87	July 20, 2016	July 19, 2026
LE-383874**	Licence d'exploitation	Option to own 100%	8.10	December 2, 2015	December 1, 2025
PE-3052**	Permis d'exploitation	Option to own 100%	11.68	July 17, 2014	July 16, 2026
PR-3842950**	Permis de recherche	Option to own 100%	15.57	June 27, 2023	June 26, 2026
PR-3843051**	Permis de recherche	Option to own 100%	16.00	June 10, 2023	June 9, 2026
PR-3843056**	Permis de recherche	Option to own 100%	15.89	June 10, 2023	June 9, 2026
PR-3843057**	Permis de recherche	100% Ownership	9.43	June 10, 2023	June 9, 2026
PR-3843146**	Permis de recherche	Option to own 100%	8.54	June 14, 2023	June 13, 2026
PR-3843156**	Permis de recherche	100% Ownership	11.72	June 14, 2023	June 13, 2026
PR-3843332**	Permis de recherche	100% Ownership	4.70	October 12, 2023	October 11, 2026
PR-3843342**	Permis de recherche	100% Ownership	11.40	June 14, 2023	June 13, 2026
PR-3843370**	Permis de recherche	100% Ownership	3.99	March 8, 2024	March 7, 2027
PR-3843371**	Permis de recherche	100% Ownership	3.03	March 8, 2024	March 7, 2027
PR-3843372**	Permis de recherche	100% Ownership	7.99	March 8, 2024	March 7, 2027
PR-3843387**	Permis de recherche	100% Ownership	8.28	June 29, 2024	June 28, 2027
PR-3843388**	Permis de recherche	100% Ownership	5.73	June 29, 2024	June 28, 2027
PR-3843389**	Permis de recherche	100% Ownership	1.56	June 29, 2024	June 28, 2027
PR-3843390**	Permis de recherche	100% Ownership	14.34	June 29, 2024	June 28, 2027
PR-3843391**	Permis de recherche	100% Ownership	13.00	June 29, 2024	June 28, 2027
PR-3843448**	Permis de recherche	100% Ownership	12.55	May 20, 2025	May 19, 2028

Permit ID	Permit Type	Interest of Aya*	Area (km ²)	Granted	Expires
PR-3843449**	Permis de recherche	100% Ownership	14.37	May 20, 2025	May 19, 2028
PR-3843450**	Permis de recherche	100% Ownership	5.59	May 20, 2025	May 19, 2028

Notes :

1. Mineral Tenure information effective September 30, 2025.
2. « Permis de recherche » means « exploration permit » and « licence d'exploitation » means « mining license ».
3. Permits marked with * are within the Boumadine Mining License and permits marked with ** are located outside the Boumadine Mining License.
4. Mining License LE-383722 is currently being renewed for an additional 10 years
5. For this table, Aya means Aya or one of its subsidiaries, AGSM, ZMSM or BGM.

4.3 MINERAL TENURE IN MOROCCO

In Morocco, Mineral Tenure are attributed according to the mining code 33-13 (2013) by le Ministère de Transition Énergétique et Durable (Minister of Durable Energetic Transition or "MTEDD"). There are three types of permits:

- a) Autorisation d'exploration (Authorization of Exploration). These authorizations grant the holder the right to carry preliminary exploration work within a defined perimeter. It is granted for a period of 2 years, and is renewable once for an additional year. The authorization provides a priority right to obtain one or more exploration permits within the same area, provided the request is submitted during the validity period of the authorization;
- b) Permis de recherche (Exploration Permit). These permits grant the holder the right to conduct geological research (i.e., exploration) to identify mineral deposits within a specified area. Initially, the permit is issued for a period of 3 years and can be renewed once for an additional 4 years under specific conditions. After that period, the permit must be converted into a Mining License; and
- c) Licence d'exploitation (Exploitation or Mining License). These licenses are granted once a viable mineral deposit has been discovered. They allow the holder to extract and commercialize the minerals. Mining Licenses are issued for longer durations (generally 10 years) and require compliance with environmental and safety regulations. They are renewable.

"Permis d'exploitation" were granted prior to the new mining code and are progressively being replaced by "Licence d'Exploitation".

4.4 ACQUISITION AGREEMENT

On October 9, 2012, Maya and ONHYM signed a joint venture agreement for the acquisition, development and exploitation of the Boumadine Deposit. Under the terms of said agreement, Maya acquired 85% of mining license LE-383661 for total cash payments of MAD 28 million, being approximately USD 2.8 million at such time. A new Moroccan company - BGM, was created with Maya and ONHYM as 85%-15% shareholders, respectively. The mining title of the Boumadine Mining License was transferred by ONHYM to BGM. ONHYM's shares are fully participating and are not a free-carry. ONHYM will receive a 3% royalty and Aya will receive a 2.75% management fee on BGM sales revenue from the first year of operation.

In addition to its ownership of the Boumadine Mining License, Aya, through its subsidiaries, other than BGM, has: 100% ownership of a total of 11 mining licenses and exploration permits and; an option to earn 100% interest in 19 other mining licenses and exploration permits.

4.5 ROYALTIES AND ENCUMBRANCES

Aya and ONHYM signed an agreement for the development and exploitation of the Boumadine polymetallic deposit; Aya holds 85% and ONHYM holds a fully participating non free-carry 15% stake in its share capital of the Boumadine Mining Licence through BGM. ONHYM will receive a 3% royalty on BGM sales revenue from the first year of operation. Aya will receive a management fee equal to 2.75% of the revenue from BGM.

4.6 ENVIRONMENTAL AND PERMITTING

In order to convert an exploration permit to a mining permit, an environmental impact assessment needs to be submitted to the regional centre of investment ("CRI").

Aya is committed to carrying out exploration work, Mineral Resource estimation, mine and infrastructure design work, and metallurgical testing in the next 60 months. Aya agreed to periodically inform the Regional Department of Energy and Mines and ONHYM of its work progress.

4.7 OTHER SIGNIFICANT FACTORS AND RISKS

To the best of the knowledge of the Authors, there are no environmental considerations or other significant factors or risks that may affect access, title, or the right or ability to perform work on the Boumadine Property.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESS

The Boumadine Property is located in the Errachidia Province of the Meknès-Tafilalet Region, in the Anti-Atlas Mountains. It is accessible via the National Highway 10 (N10), ~220 km east-northeast from Ouarzazate City or ~70 km southwest from Errachidia City (Figure 5-1). The nearest town is Tinejdad, ~16 km north from the historical Boumadine Mine. The nearest village, Bouyou, is 4 km away from the site. The Property is accessible from Tinejdad by all-terrain vehicle on a paved and gravel road.

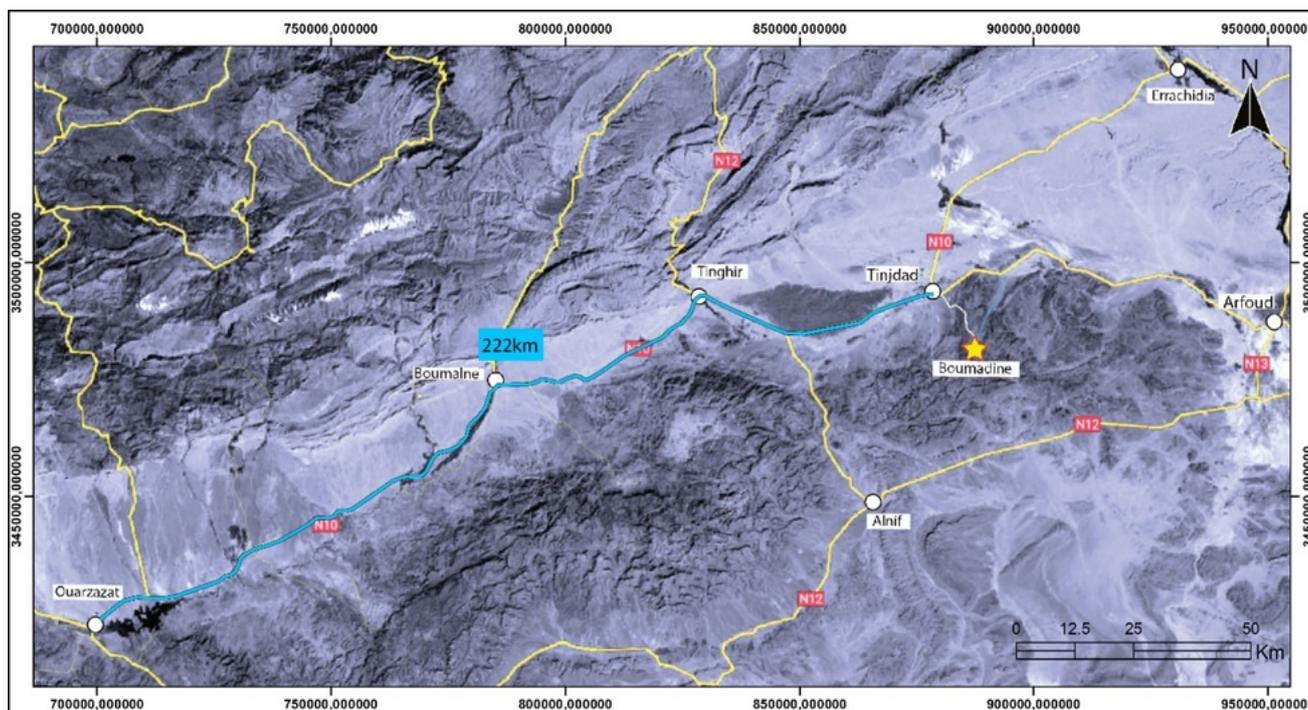


Figure 5-1 Location of the Boumadine Deposit from Ouarzazate

Source: Aya (May 2024)

5.2 CLIMATE

The Project is located on the Ougnat 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 climate falls under the category of a hot desert climate, also known as a “hot arid climate” (Köppen climate classification BWh).

Summers are hot, with daytime temperatures exceeding 40°C (Table 5-1). Winters are generally mild, with daytime temperatures ranging from mild to warm; nighttime temperatures can drop, but freezing temperatures are uncommon. The region receives very little rainfall throughout the year; most of the rainfall occurs during the winter months. Field work at Boumadine can be performed year-round.

Table 5-1 Monthly Temperature and Precipitation Average at Tinejdad from 1991 to 2021

Monthly Temperature and Precipitation Average at Tinejdad from 1991 to 2021	Jan	Feb	Mar	Apr	May	June	July	Aug	Sep	Oct	Nov	Dec
Avg. Temp. (°C)	8.6	10.6	14.9	19.2	23.1	28	31.5	30.3	25.5	20.1	13.4	9.5
Min. Temp. (°C)	1.7	3.2	7	11.2	15	19.2	22.3	22	17.9	13.3	6.9	3.4
Max. Temp. (°C)	15	17.1	21.5	25.7	29.4	34.4	38.2	36.7	31.7	25.9	19.2	15.4
Precipitation / Rainfall (mm)	8	13	9	8	9	4	2	4	9	17	12	7
Humidity (%)	40	36	29	23	21	17	14	17	25	34	41	47
Rainy Days (d)	1	1	2	1	1	1	0	1	1	2	2	1
Avg. Sun hours (hours)	9	9.6	10.6	11.6	12.3	12.7	12.6	11.9	11	10.1	9.2	8.7

Source: *climatedata.org*

5.3 INFRASTRUCTURE

The Boumadine Property can be accessed by two roads: 1) a 16-km dirt road southwards from Tinejdad; and 2) a 2.8-km dirt road from east through the Village of Bouyouud. The National Highway 10 (N10) goes through the City of Tinejdad and connects to Ouarzazate City to the west and Errachidia City to the east of Tinejdad. The Boumadine camp is connected to the national grid.

There are many dirt roads and paths that lead to former shafts and other remnants of the historical mining infrastructure. Water is currently sourced from historical underground workings and wells. The Boumadine camp and infrastructures are connected to the national electrical grid, to provide electricity.

The facilities on-site are adapted for exploration operations. They include an office, drill core shack, the Afrilabs sample preparation laboratory, drill contractor workshop, and drill contractor camp (Figure 5-2).

5.4 PHYSIOGRAPHY AND VEGETATION

The physiography of Boumadine Property is characterized by its desert setting, with influences from the nearby Atlas Mountains. The topography of the area is marked by ridges and hills mostly, with altitude ranging between 980 and 1,300 m asl (Figure 5-3a). The site and its surroundings exhibit characteristics typical of a desert landscape, with vast expanses of arid and rocky terrain.

The vegetation of the Boumadine region consists mainly of desert plants such as Acacia Raddiana and Tamarix Amplexicaul. Certain types of drought-resistant grasses may be found in the region, providing some ground cover. Oases and palm trees are a notable feature in this region (Figure 5-3b). Date palms are cultivated in this type of area.



Figure 5-2 Boumadine Site Infrastructure

Source: Aya (April 2024)



Figure 5-3 Physiography and Vegetation

Source: Aya (April 2024)

Figure 5-3 Description: a) Physiography at Boumadine; and b) Cultivated oasis near Boumadine

5.5 LOCAL RESOURCES

The Boumadine Mining Licence is in close proximity to the Ziz river, which flows through the center of Tinejdad. The river, along with its valley, contributes to the oasis environment with palm groves, particularly date palms. Other types of agricultural activities can be found in these areas.

Group of nomads pass by the Boumadine Property with their livestock of camels or goats (Figure 5-4a). The nomad community is composed of Amazigh (Berber) people.

Historical artisanal mining activities have been recognized at several places on the Boumadine Property (Figure 5-4b). Historical extraction work focused on barite and lead veins.

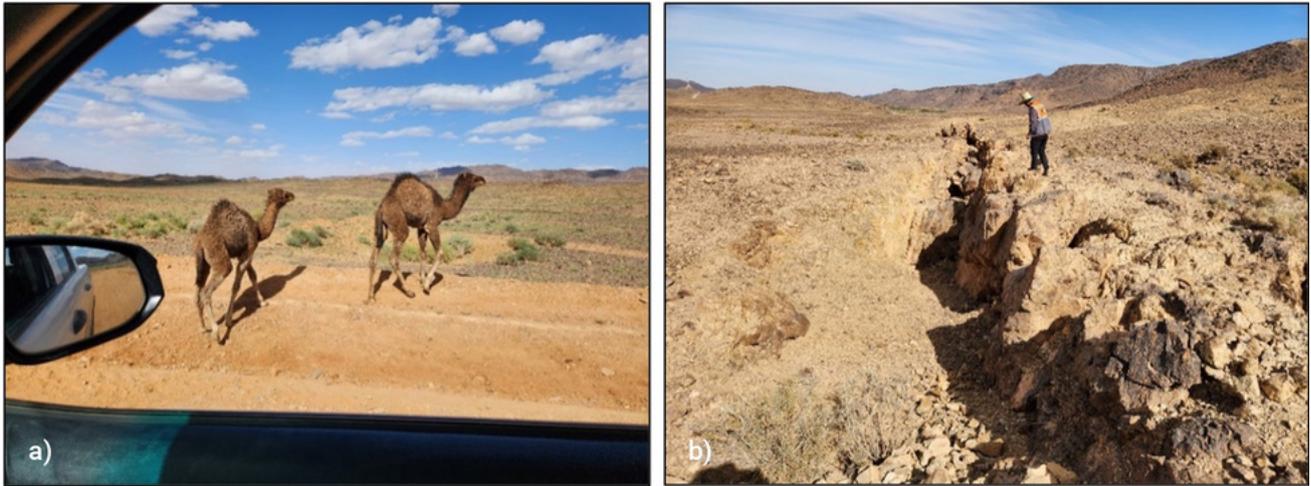


Figure 5-4 Nomadic Livestock and Artisanal Activity

Source: Aya (April 2024)

Figure 5-4 Description: a) Camels near the Boumadine Historical Mine Site; and b) an artisanal excavation on the Boumadine Property.

Mine workers and other personnel are available from nearby villages or Tinejdad. The City of Errachidia (formerly called Ksar souk) is the closest major urban centre. Errachidia has an international airport with access to Casablanca and it is also accessible by road from Marrakech (~420 km away).

Basic supplies, such as food and limited accommodation, are available at Tinejdad; the larger City of Errachidia offers greater diversity in supplies. Special items must be purchased from Casablanca or Marrakech.

6.0 HISTORY

6.1 MINING AND EXPLORATION HISTORY

6.1.1 Antiquity and 15th to 16th Centuries

The historical Boumadine Mine is one of the oldest known mines in the Kingdom of Morocco. It was probably exploited by the Portuguese during the 15th and 16th centuries. They extracted the oxidized part of the polymetallic veins to a depth of as much as 20 m. Such workings are found along a north-south strike length of 4.2 km on the Boumadine Property.

6.1.2 BRPM: 1956 to 1964

In 1956, the Bureau de Recherches et de Participations Minières ("BRPM"), the Morocco filial of the Bureau des Recherches Géologiques et Minières ("BRGM") commenced modern exploration at Boumadine. Initially, BRPM defined three targets:

1. The north-south historical workings, ±4 km extension starting from the south area and finishing to the north at 'Imariren';
2. The N70°E veins, enriched in galena and chalcopryite. At least five of these veins were defined over the property; and
3. The southwest-northeast 'Bou Guedoud' regional faults, which is an alignment of different intrusions.

Exploration was prioritized at the north-trending historical workings. In 1962, exploration activities increased and by the end of 1964, BRPM had completed 6,248 m of drilling and 27 m of shaft excavation on the Mining License.

6.1.3 BRPM: 1964 to 1966

In 1964, a geophysical survey testing different methods (magnetism, electromagnetism) was completed on the Mining License. The results were inconclusive.

Metallurgical testing was completed in July 1965, leading to the first economic study and the next phase of exploration, which involved underground excavation to certify probable and possible mineral reserves. Exploration workings were extended in the Central Zone, followed by mineralized material development and underground drilling. During that period, BRPM completed 1,984 m of drilling, 122 m of vertical underground development, and 859 m of horizontal underground development.

6.1.4 BRPM: 1966 to 1975

Between 1966 and 1970, the BRPM continued mineralogical, metallurgical, and economical studies. BRPM contracted the USSR agency Giprotsvetmet for an independent technical economic review. The USSR agency considered the opportunity to treat the mineralized material through a flotation, roaster, and cyanidation circuit. Their first conclusion came in 1968 and was negative, based on the remote location and low mineral resource quantity at the time.

In 1969, BRPM conducted an electromagnetism survey (Thuram) to better define extension and geometry of Boumadine polymetallic veins.

Exploration continued from 1973 to 1975. The BRPM completed an additional 7,132 m of drilling to target the Central Veins extension below underground working and to define new polymetallic veins in the northern area. A new shaft was opened in the northern area, whereas one central shaft was deepened to -150 m below surface ("MBS"). The BRPM completed 678 m of mineralized material development at the -150 mbs central level and 288 m of mineralized material development in the northern area.

6.1.5 BRPM: 1975 to 1985

From 1975 to 1981, BRPM did not conduct any exploration activities; Exploration was resumed in 1981–1982. An infill campaign targeted a high-grade panel with 340 m of underground drifts and two shafts of 37.5 m each. Many underground channel samples were taken in the central area at that time.

Exploration continued in 1984 to help define the mineral resource of the Central Area. Additional underground workings, diamond and sludge drill holes were completed to define parallel polymetallic veins. Two new areas, Tizi and South Zones, were explored by two new shafts and additional underground workings.

6.1.6 SODIM: 1986 to 1989

In 1986, the society SODIM in partnership with the BRPM started industrial testwork for mining production. Between July 1986 and October 1987, SODIM treated 23,700 tonnes grading 1.38% Pb and 6% Zn; precious metals recovery was low at <30%. All materials were sourced from the -50 m level of the Central Zone.

In June 1988, the BRPM retook ownership of the Deposit to pursue industrial tests. From June 1988 to May 1989, the BRPM treated 42,785 tonnes grading 1.57% Pb and 4.97% Zn through a flotation circuit. Precious metals recovery was still low; all materials were sourced from the -50 m level of the Central Zone.

In July 1989, the BRPM loaned the Project to the Société de Développement du Cuivre de l'Anti Atlas ("SODECAT") to continue production testwork.

6.1.7 SODECAT: 1989 to 1992

SODECAT re-optioned the Property in 1989 and invested in the mining infrastructure. The development of underground workings were concentrated at the -100 m level in the Central Zone and -120 m level in the South Zone. From July 1989 to August 1992, 1,570 m of surface drilling, 111 m of shaft, 1,058 m of stopes and drifts, and 187 m of raises were completed. A committee was organized in 1991 between BRPM and SODECAT to decide the future of the Project. Due to marginal profit, the decision was made to mine out high-grade stopes and cease production at the end of 1991. During their production period, the SODECAT extracted 191,000 t of mineralized material grading 0.94 wt.% Pb, 328 g/t Ag and 3.69 g/t Au from the South and Central Zones.

6.1.8 BRPM: 1993 to 1998

An electromagnetism geophysical survey was completed in 1994. Results confirmed extension of polymetallic veins below -300 m below surface. The recommendation from the geophysical survey report for completing additional deep drill holes was not followed-up.

The BRPM issued its final report in 1998, which summarize all the work programs completed on the Boumadine Mining License (also known as Boumaadine), including a summary of all its drilling (Table 6-1; Figures 6-1 and 6-2).

Table 6-1 Summary of BRPM Exploration Work

Period	Diamond Holes		Shafts/Raises	Drifts
	Number	(m)	(m)	(m)
1956 to 1957	2	144	27	191
1962 to 1964	40	6,248	-	-
1964 to 1966	36	1,984	77	981
1973 to 1975	25	7,132	152	874
1981 to 1982	-	-	82	340
1984 to 1985	13	1,029	140	1,885
1989 to 1992	unknown	1,570	298	1,376

Source: Aya (April 2024)

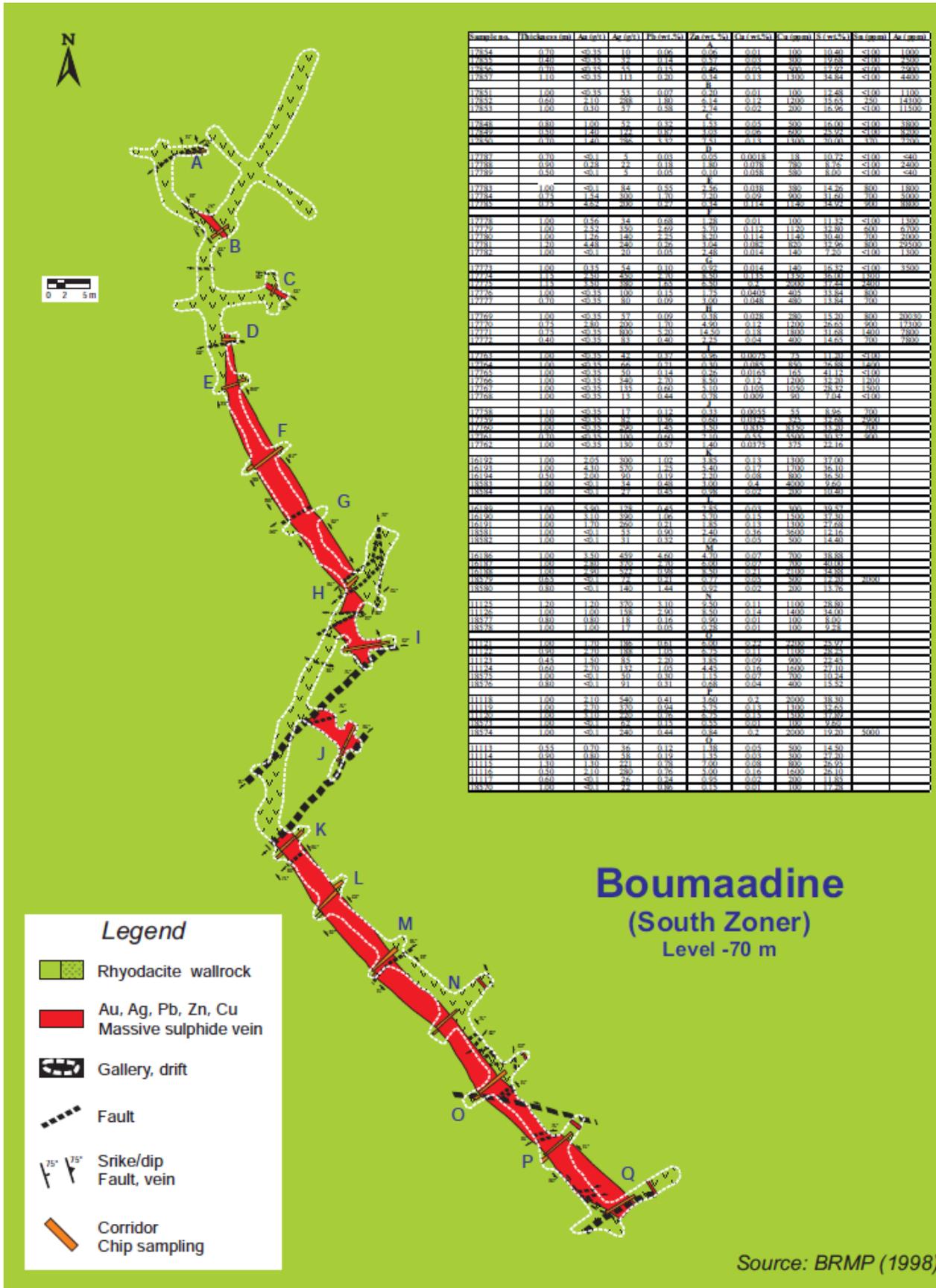


Figure 6-1 -70 m Level Plan of South Zone, with Chip Sample Assays

Source: BRPM (1998)

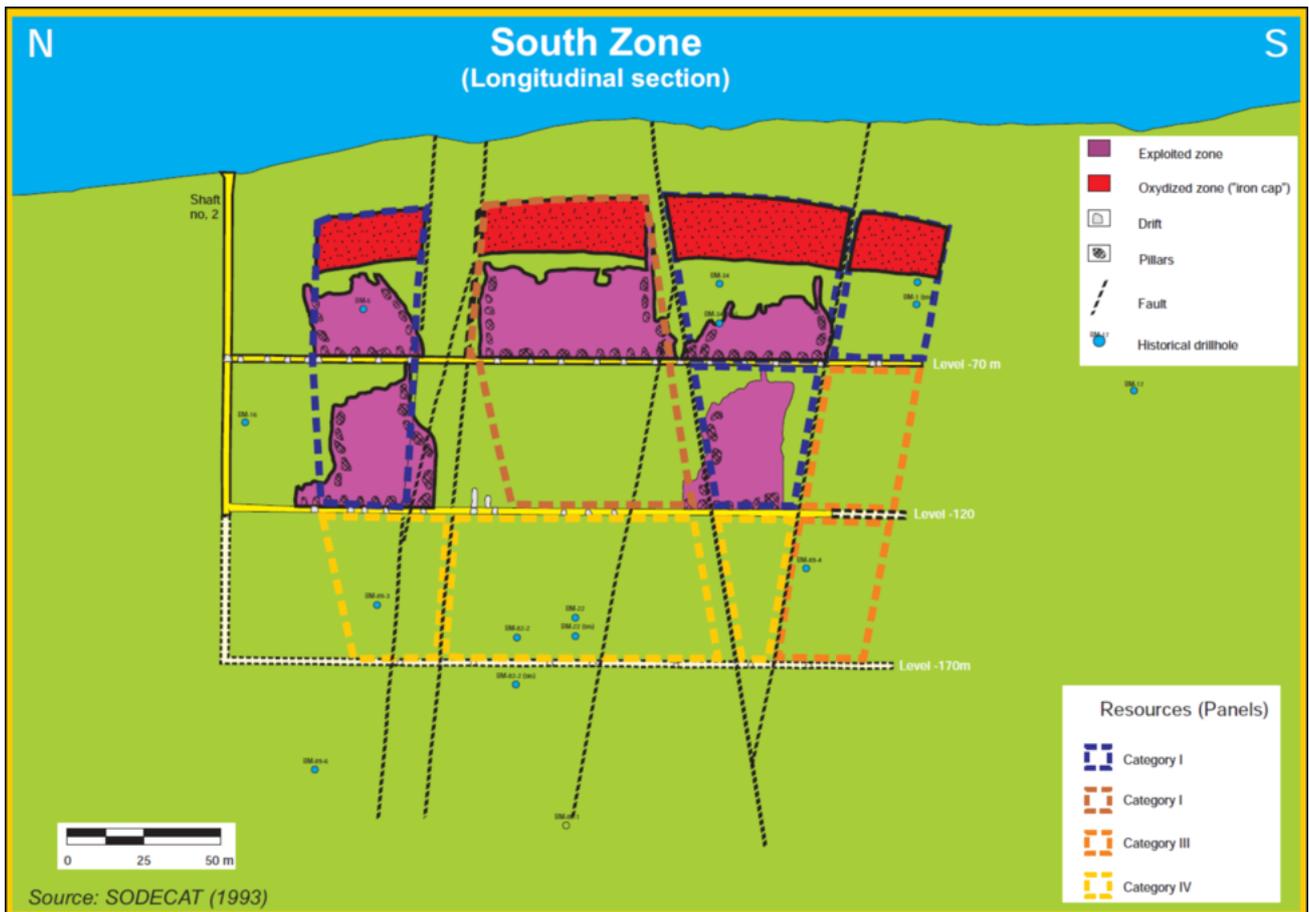


Figure 6-2 Longitudinal Projection of the Principal Polymetallic Vein Mined in the South Zone

Source: BRPM, (1998)

Note: Panels or blocks defined according to the historical mineral resource category. Category I to IV corresponds to internal BRPM mineral resource classification.

6.1.9 Maya: 2013 to 2019

In 2013, Maya Gold and Silver Inc (Maya: former entity of Aya Gold & Silver) acquired the Boumadine Mining License through a Joint Venture with the ONHYM (new entity of the BRPM) owned 85% / and 15%, respectively.

Between 2013 and 2016, Maya only completed surface exploration work. This work included the collection of several grab samples from historical mineralized structures to investigate the historical mineral resources. Surface mapping of the mineralized structures was completed to identify targets for the surface sampling program.

In 2017, Maya conducted drilling with the objective to certify the historical mineral resource. Fourteen drill holes were completed over the Central, South and Tizi Zones for a total of 3,158 m.

The next drilling campaign was designed to increase the mineral resource at the Boumadine Deposit. Between 2018 and 2020, Maya completed 9,503 m of diamond drilling over South, Central, North, Imariren and Tizi Zones. In addition, Maya conducted a drone survey over part of the Boumadine Mining License. A summary of the drilling completed by Maya is presented in (Table 6-2).

Table 6-2 Summary of Diamond Drilling Completed by Maya from 2017 to 2020

Summary of Diamond Drilling Completed by Maya from 2017 to 2020	Zone	Metres Drilled	Objective
2017	Centre	1,490.4	Historical resource and old workings verification
	Sud	1,137.6	
	Tizi	530.0	
2018	Centre	1,597.0	Check southern continuity of the vein
	Sud	608.0	
	Tizi	466.1	
	Imariren Est	804.9	Follow-up on grab samples positive results
	Imariren Ouest	1,168.4	
2019	Centre	1,446.7	Increase Imariren resource
	Imariren Est	670.8	
	Imariren Ouest	1,297.8	Test northern historical resource
	Nord	543.9	
2020	Nord	899.0	Increase Zone Nord resource
Total		12,661.0	

Source: Aya (April 2024)

In 2018, a sampling campaign was completed over two historical tailing deposits. A soil core sample were taken every meter in 49 drill holes for a total of 186.63 m (Figure 6-3).

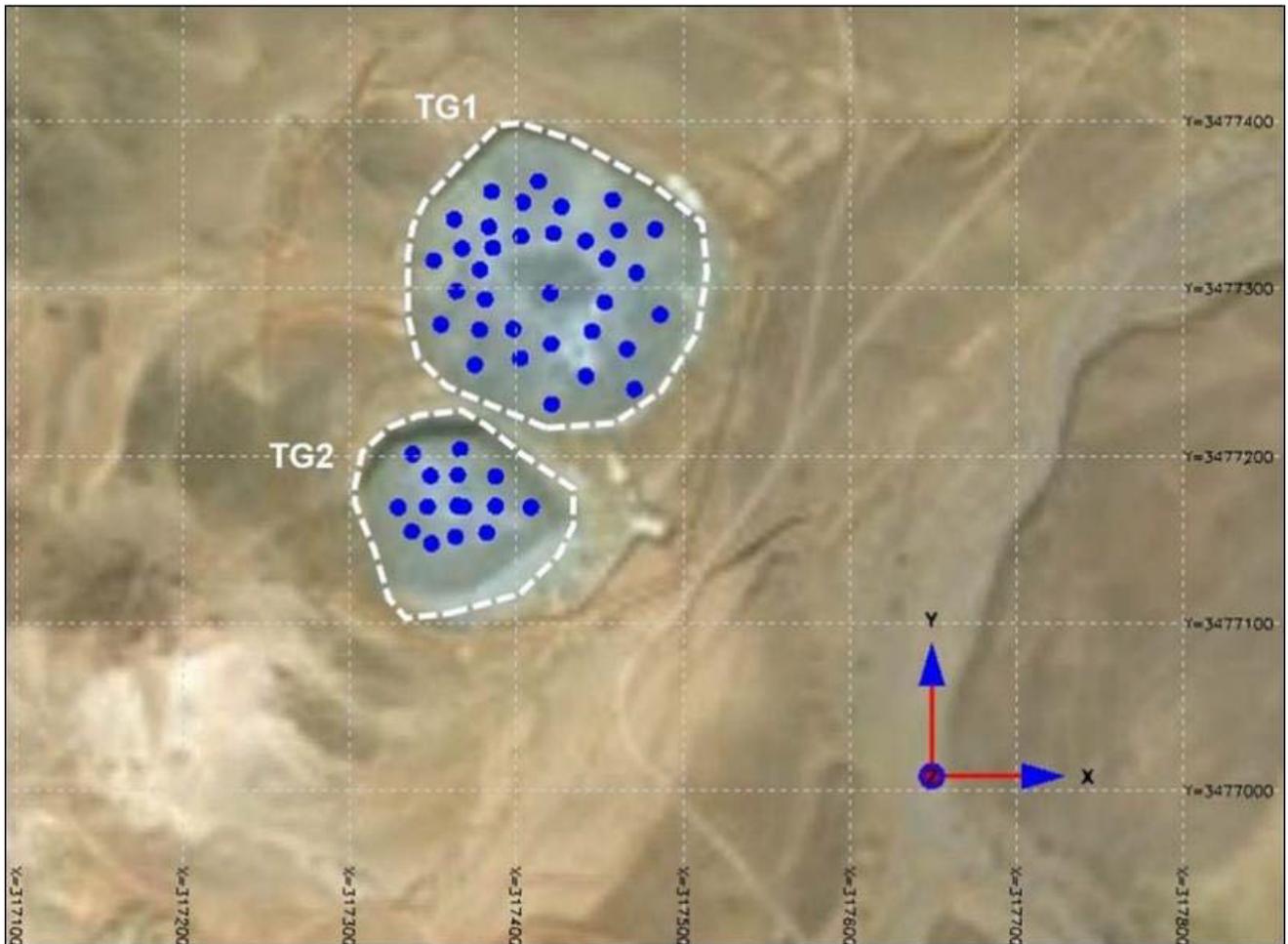


Figure 6-3 Location of Soil Core Samples Completed by Maya in 2018

Source: GoldMinds (2019)

6.2 GEOPHYSICAL SURVEYS

BRPM completed several geophysical surveys over the Boumadine Property (Table 6-3). Electromagnetic surveys delivered results that helped to better define polymetallic veins extension on surface (Figure 6-4).

Table 6-3 Geophysical Surveys Completed by BRPM

Years	Type
1964	Self Potential
	Seismic
	Resistivity
1974	Electromagnetic - TURAM
1985	Electromagnetic - TURAM
1990	Electromagnetic - TURAM
	Electric - Self Potential
	Induced Polarization
1994	Electromagnetic CSAMT & TEM

Source: Aya (April 2024)

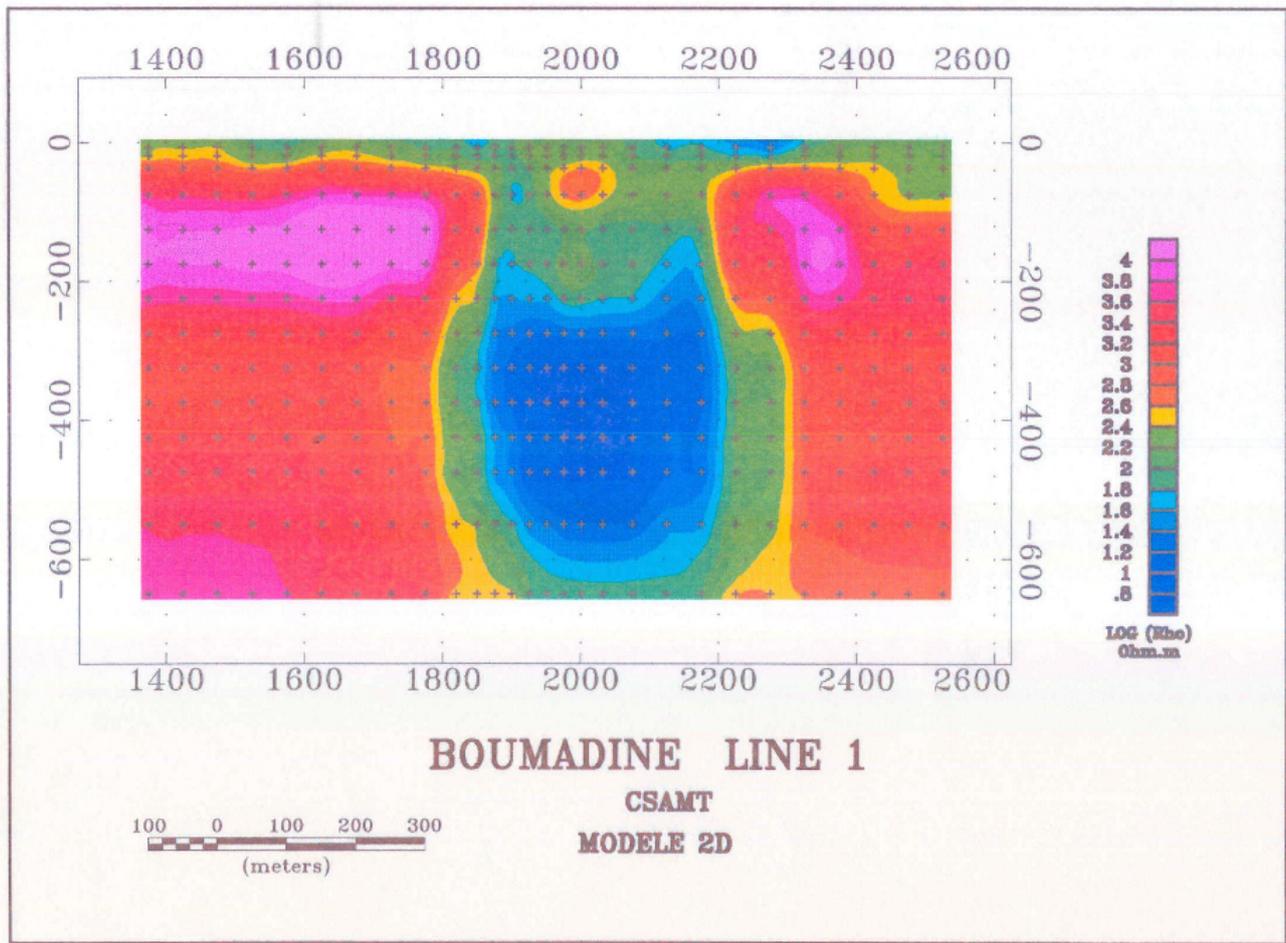


Figure 6-4 2-D Section of a CSAMT Survey*

Source: BRPM (1994)

* CSAMT = controlled-source audio magnetotelluric survey

6.3 PAST PRODUCTION

BRPM published a summary of production figures in 1998 (Table 6-4), including flotation results (Table 6-5) and metallurgical recoveries (Table 6-6).

Table 6-4 Tonnes Extracted

Period	Location	Tonnes
1986-1987	Central level -50	22,355
1988-1989	Central level -50	48,130
1989-1992	Central level -50 & -100	132,000
1989-1992	South - level 70 & -120	59,000
Total		261,485

Source: BRPM (1998)

Table 6-5 Flotation Results

Pb Concentrate		Zn Concentrate		Grade - Rejects	
Pb (%)	41.6	Zn (%)	44.5	Pb (%)	0.21
Ag (g/t)	4,711	Ag (g/t)	565	Zn (%)	0.96
Au (g/t)	24.6	Au (g/t)	3.7	Ag (g/t)	125
				Au (g/t)	2.38

Source: BRPM (1998)

Table 6-6 Metallurgical Recovery by Concentrate Type

Metallurgical Recovery by Concentrate Type	Pb	Zn	Ag	Au
Pb Concentrate	69.08%	-	32.01%	13.52%
Zn Concentrate	-	77.04%	18.03%	9.53%

Source: BRPM (1998)

6.4 HISTORICAL MINERAL RESOURCE ESTIMATES

The Authors are unable to verify the historical mineral resource estimates, because the supporting data are incomplete, they were not prepared by an independent party, and all predate the requirements set forth in NI 43-101. The historical mineral resource estimates are relevant in that they provide historical context and a framework on which to plan work programs to define NI 43-101 compliant Mineral Resources or Mineral Reserves. Aya is not treating the historical mineral resource estimates as current Mineral Resources, and therefore they cannot be relied on and may not be indicative of future mining at Boumadine.

6.4.1 BRPM

BRPM produced several mineral estimates during its tenure; however, none of these mineral estimates were reported in compliance with NI 43-101 or other internationally recognized reporting standards. The final BRPM mineral resource was reported in 1998 and included mining depletion (Table 6-7).

Table 6-7 Boumadine Deposit 1998 Mineral Resource Report

Boumadine Deposit 1998 Mineral Resource Report	Tonnes	Au (g/t)	Ag (g/t)	Zn (%)	Pb (%)
Certain	280,775	3.17	171	3.60	0.70
Very Probable	267,670	2.27	155	3.27	1.02
Probable	231,230	1.06	175	3.03	0.86
Possible	450,000	1.52	95	3.51	1.05

Source: (BRPM (1998))

Note: *BRPM internal classification categories, not NI-43-101 compliant

The work was completed prior to the implementation of current NI 43-101 standards, does not conform to present-day standards, and should not be relied upon. The Authors have not completed sufficient work to classify the historical mineral resource estimates as current Mineral Resource Estimates. Aya is not treating the historical mineral resources as current Mineral Resource Estimates, and they should not be relied upon.

6.4.2 Maya 2019 PEA

Maya commissioned GoldMinds Geoservices Inc. (GoldMinds) to complete a Preliminary Economic Assessment (“PEA”) for the Boumadine Deposit. The PEA was reported in May 2019 with an effective date of April 24, 2019.

6.4.3 P&E 2024 MRE

On behalf of Aya, P&E prepared a Mineral Resource Estimate for the Boumadine Deposit. The MRE was published on May 8, 2024 with an effective date of February 23, 2024 (Table 6-8).

Table 6-8 Boumadine MRE as of April 15, 2024 ⁽¹⁻¹²⁾

Class	Cut-off NSR US\$/t	Tonnes (kt)	Average Grade							Contained Metal						
			Ag (g/t)	Au (g/t)	Cu (%)	Pb (%)	Zn (%)	AgEq (g/t)	AuEq (g/t)	Ag (koz)	Au (koz)	Cu (kt)	Pb (kt)	Zn (kt)	AgEq (koz)	AuEq (koz)
Pit-Constrained																
Indicated	95	995	145	2.71	0.2	1.6	5.9	634	7.4	4,647	87	2	16	59	20,299	237
Inferred	95	8,474	103	2.97	0.1	0.8	2.5	475	5.54	28,087	808	8	67	215	129,478	1,510
Out-of-Pit																
Indicated	125	1,046	82	2.33	0.1	0.6	2.8	402	4.69	2,751	78	1	6	29	13,533	158
Inferred	125	15,096	76	2.42	0.1	0.9	2.2	389	4.53	36,653	1,175	11	131	330	188,663	2,198
Total																
Indicated	95/125	2,042	113	2.51	0.1	1.1	4.3	515	6.01	7,398	165	3	22	88	33,832	395
Inferred	95/125	23,569	85	2.62	0.1	0.8	2.3	420	4.89	64,740	1,983	19	198	546	318,141	3,708

- 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 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.
 4. A silver price of US\$21/oz with a process recovery of 89%, a gold price of US\$1,900/oz with a process recovery of 85%, a zinc price of US\$1.20/lb with a process recovery of 72%, a lead price of US\$1.00/lb with a process recovery of 85%, and a copper price of US\$4.00/lb with a process recovery of 75% were used.
 5. $AgEq = Ag(g/t) + (Au(g/t) * Au\ price/gram * Au\ recovery) / (Ag\ price/gram * Ag\ recovery) + Zn(\%) * Zn\ price/lb * Zn\ recovery / (Ag\ price/gram * Ag\ recovery) * 685.7147973 + Pb(\%) * Pb\ price/lb * Pb\ recovery / (Ag\ price/gram * Ag\ recovery) * 685.7147973 + Cu(\%) * Cu\ price/lb * Cu\ recovery / (Ag\ price/gram * Ag\ recovery) * 685.7147973$
 6. $AuEq = Au(g/t) + (Ag(g/t) * Ag\ price/gram * Ag\ recovery) / (Au\ price/gram * Au\ recovery) + Zn(\%) * Zn\ price/lb * Zn\ recovery / (Au\ price/gram * Au\ recovery) * 685.7147973 + Pb(\%) * Pb\ price/lb * Pb\ recovery / (Au\ price/gram * Au\ recovery) * 685.7147973 + Cu(\%) * Cu\ price/lb * Cu\ recovery / (Au\ price/gram * Au\ recovery) * 685.7147973$
 7. The constraining pit optimization parameters were US\$3/t for mineralized material mining. US\$2/t for waste mining US\$89/t for processing and US\$6/t for G&A totalling US\$95/t for a cut-off and 50-degree pit slopes.
 8. The out-of-pit parameters used a US\$30/t mining cost, US\$89/t processing cost and US\$6/t G&A totalling US\$125/t for a cut-off. The out-of-pit Mineral Resource grade blocks were quantified above the \$125 NSR cut-off, below the constraining pit shell and within the constraining mineralized wireframes. Out-of-pit Mineral Resources exhibit continuity and reasonable potential for extraction by the longhole underground mining method.
 9. Individual calculations in tables and totals may not sum due to rounding of original numbers.
 10. Grade capping of 800 g/t Ag, 30 g/t Au, 28% Zn, 10% Pb and 1.4% Cu was applied to composites before grade estimation.
 11. Bulk density was evaluated separately for each individual vein with values ranging from 3.20 to 4.00 t/m³ determined from drill core samples and used for the MRE. For oxidized and transitional material, a bulk density of 2.65 t/m³ was used.
 12. 1.0 m composites were used during grade estimation.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

The Boumadine Property is located at the eastern end of the Anti-Atlas Mountain Range, which extends east-northeast to west-southwest, over approximately 600 km from the Atlantic Ocean in the west to the interior of the African Plate in the east (Figure 7-1). The Anti-Atlas basement rocks are mainly Neoproterozoic in age and consist of ophiolites, island arc-related gneiss and intrusive rocks, particularly near to the northern edge of the West African Craton.

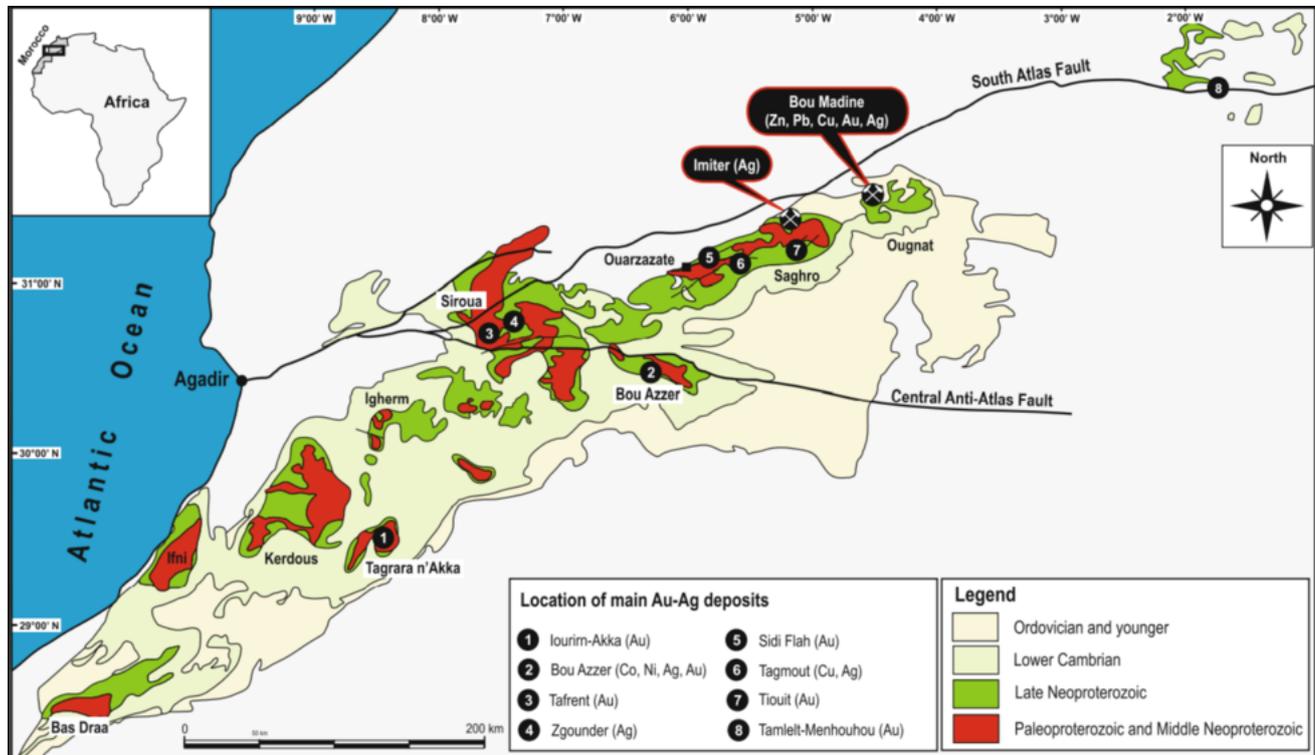


Figure 7-1 The Regional Geology of the Anti-Atlas Displaying Proterozoic Windows

Source: Boily (2014)

Note: Boumadine is also known as Bou Madine

The Boumadine polymetallic deposit (Ag, Au, Cu, Pb, Zn) is located on the northwest side of the Ougnat Massif (or Boutonnière). The geology of the Massif consists of a Neoproterozoic metasedimentary basement overlain unconformably by a Late Neoproterozoic volcano-sedimentary rock sequence and by Paleozoic lacustrine sedimentary and minor volcanic rocks. The basement consists of sandstone, pelites and greywackes that are intruded locally by granite, granodiorite, and diorite bodies. The volcano-sedimentary sequence consists of felsic and mafic volcanic rock units separated by volcano-sedimentary units (Figures 7-2 and 7-3).

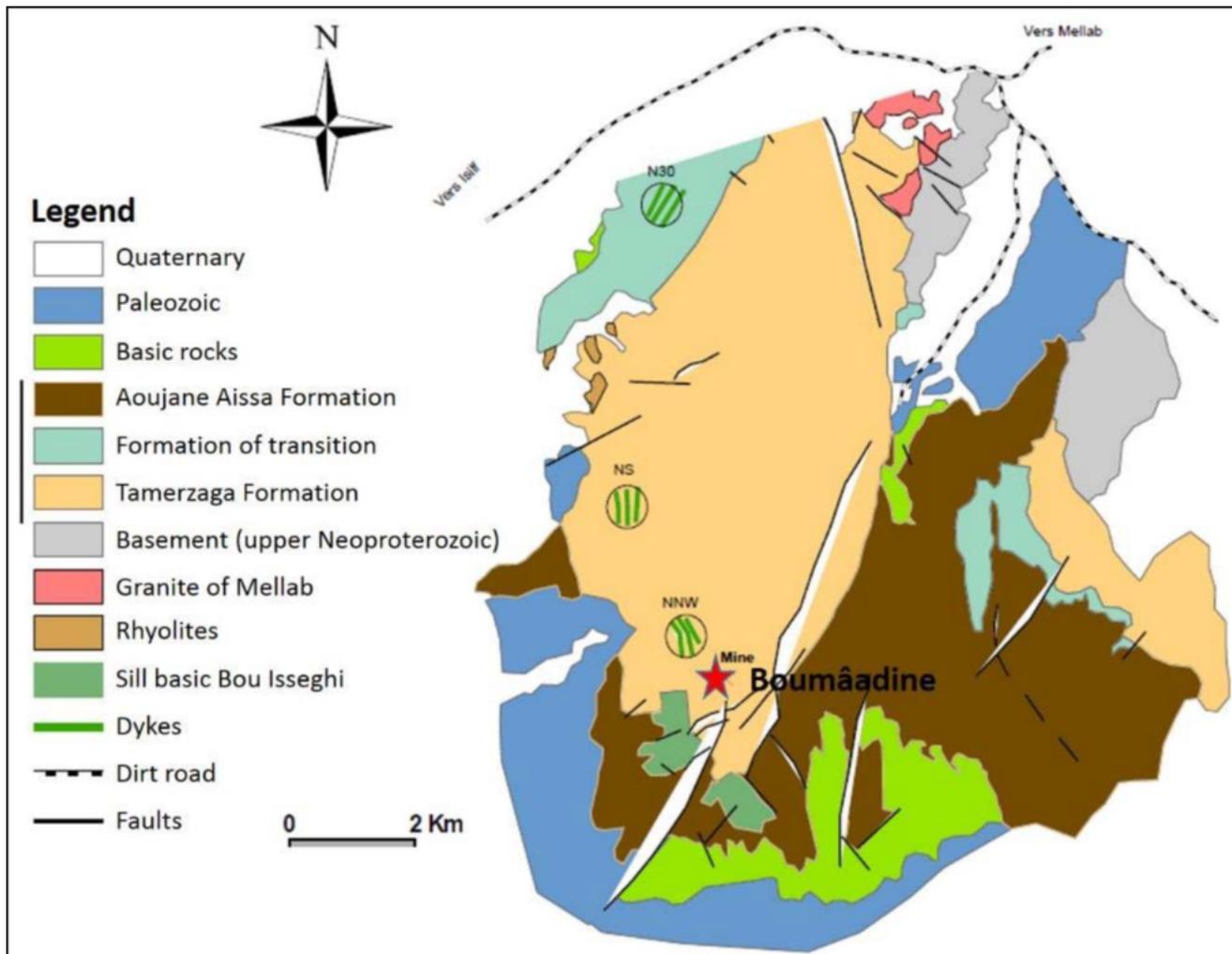


Figure 7-2 Geology of the Ougnat Massif

Source: Boily (2014)

Note: Boumadine is also known as Boumâadine

The volcanic and volcano-sedimentary rock unit have been grouped into three formations (Figure 7-3), which from the oldest to youngest are:

1. Tamerzaga-Timrachine Formation ("TTF"): Consists of ignimbrites, rhyodacites and andesites;
2. Isilf-Ouinou-Oufroukh Formation ("IOF"): Consists of volcano-sedimentary rocks, specifically tuffs and breccia, andesitic flows; and fine- to coarse-grained sedimentary rocks; and
3. Aoujane-Aissa-Akchouf Formation ("AAA"): Formed of ignimbrites, dacite domes and flows, and andesitic flows.

These three Formations are intruded by dolerite, microdiorite and andesite dykes. At the historical Boumadine Mine, only the andesite dykes are present and trend north-south.

The Ougnat Massif area was subjected to a Neoproterozoic shearing, which generated regional-scale faults trending N30°E and associated secondary fractures. The area has also been affected by a late-stage series of north-south extensional fractures that were subsequently reactivated by a compressive Hercynian tectonic event.

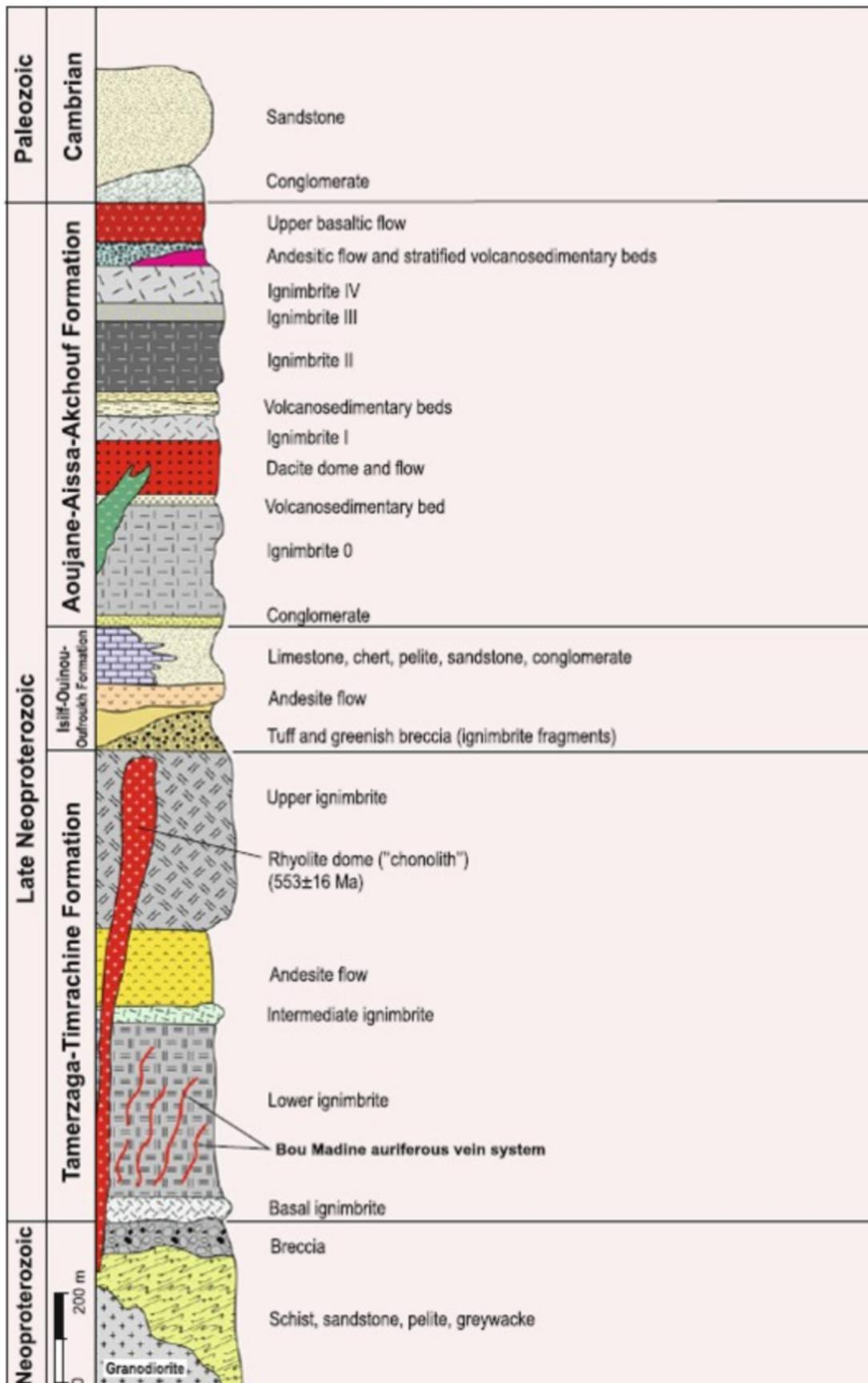


Figure 7-3 Schematic Stratigraphic Column of the Central Ougnat Massif

Source: Modified by Bouabdellah and Levesse (2016) from Paile (1983)

7.2 PROPERTY GEOLOGY

7.2.1 Volcanic and Intrusive Rocks

The TTF volcanoclastic sequence of felsic tuffs and mafic tuffs host the Boumadine Deposit (Figures 7-3 and 7-4). The felsic tuffs consist of angular to rounded cm-size felsic fragments, quartz eyes, plagioclase grains, and locally mafic fragments. This felsic sequence is homogeneous and massive, and sits unconformably on mafic tuffs. Mafic tuffs consist of amphibole and fragments/clasts of sedimentary rocks. Mafic tuffs are interpreted as underwater-deposited volcanoclastic eruptives.

Many intrusions are observed on the Boumadine Property. The intrusions are divided into a pre- to syn-mineralization group and a post-mineralization group. The pre- to syn-mineralization intrusions are mainly felsic to intermediate in composition, show aphanitic to porphyritic textures, and form dykes and sills. Locally porphyritic mafic dykes, similar in composition to mafic tuffs, cross cut the felsic tuff sequence and syn-mineralization dykes, suggesting bimodal magmatism.

The post-mineralization intrusions consist of rhyolite subvolcanic domes associated with normal faults. These domes are interpreted as being synchronous with a post-mineralization deformation episode that disrupted the Boumadine mineralized zones. Subsequently, a swarm of regionally extensive mafic dykes intruded every lithological unit on the Property.

7.2.2 Hydrothermal Alteration

Two events of hydrothermal alteration are observed on the Mining License. The first alteration event affects the felsic tuff sequence as phyllic alteration (quartz-sericite-pyrite). Proximal to massive sulphide veins (1 to 5 m thick), there is an advanced clay alteration composed of kaolinite and pyrophyllite.

The second sequence of alteration affects mainly the underlying mafic tuffs and consists of propylitic alteration (epidote and chlorite). Near the veins, the alteration minerals are black chlorite, pyrophyllite and pyrite. The transition between these two alteration events is relatively sharp and consistent with the change in tuff composition.

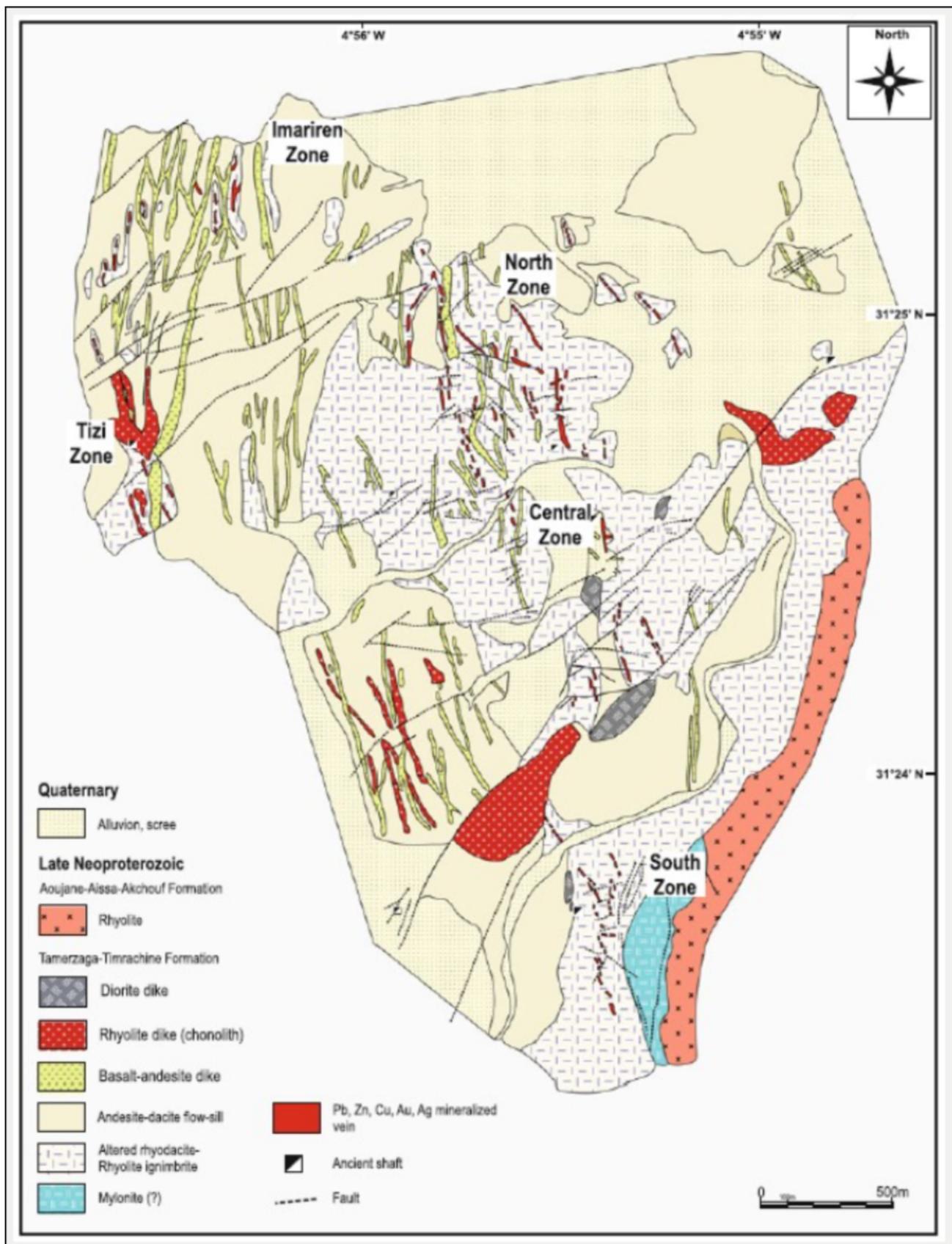


Figure 7-4 Boumadine Mining License Geology Map

Source: Aya website (April 2024)

7.2.3 Structure

This section is summarized largely from Boily (2014).

The most important structures in the Boumadine Mining License are represented by km-scale N30°E-oriented faults, limiting several tectonic blocks in the Ougnat Massif and potentially corresponding to tectonic movement of basement rocks. These faults, associated with tectonic breccias, first manifested normal-sinistral movement (striae with a N30°E/ 0–15°S plunge), and subsequently re-activated as normal faults. The tectonic fractures correspond either to normal faults associated with a north-northeast to south-southwest-oriented tension gash or N160°E tension gash controlling the emplacement of the andesite dykes, “chonolith” rhyolites, and polymineralized Boumadine veins (Figure 7-5). These features correspond to the shortening direction associated with the early sinistral activity on the N30°E faults (Abia et al., 2003), as represented by the patterns of dykes distributed in “en-echelon arrays” in potential conjugate shear zones developed along N10°–30°E and N130°–140°E directions. The movements along the N30°E sinistral faults induced shearing associated with the opening of a collapsing basin and extension during emplacement of the TTF.

Some felsic and andesite dyke swarms strike N0°–N30°E, at variance with the N160°E strike of earlier andesite dykes. The emplacement of these dyke swarms is likely related to the reactivation of the early N30°E faults as normal brittle faults (associated with brecciation) that represent tension cracks. At Boumadine, ductile shear-zones striking N150°E developed C/S structures with a schistosity trending N135°E, consistent with dextral movement under N30°–N40°E shortening direction (Abia et al., 2003). These shear zones developed in corridors already affected by a strong pyrophyllitization and syn- and late-kinematic pyrophyllite veinlets occur within the shear-zones coeval with the shearing (Figure 7-6). The shearing overprinted late rhyolite domes and andesite dykes (Ait Sadi, 1992), and the shear-zones are sealed by the upper basalt flows (Freton, 1988). Thus, a return to compressional regime tectonics occurred at the end of the Late Proterozoic.

During the Phanerozoic era (Variscan period), reactivation of some N30°E-oriented faults as reverse faults occurred in relation with doming of the Proterozoic basement and the overlying Paleozoic cover. Cu–Pb–Zn mineralized quartz-carbonate veins were emplaced in N75°E fractures in Paleozoic and Proterozoic terrains. These veins correspond to the Imariren-type mineralization of Abia et al. (1999) and fill tension joints associated with N45°E and N105°E conjugate faults. N40°–50°E-oriented brittle faults dextrally displace the “pyrophyllite shear-zones” (up to tens of metres) and the polymetallic mineralized veins.

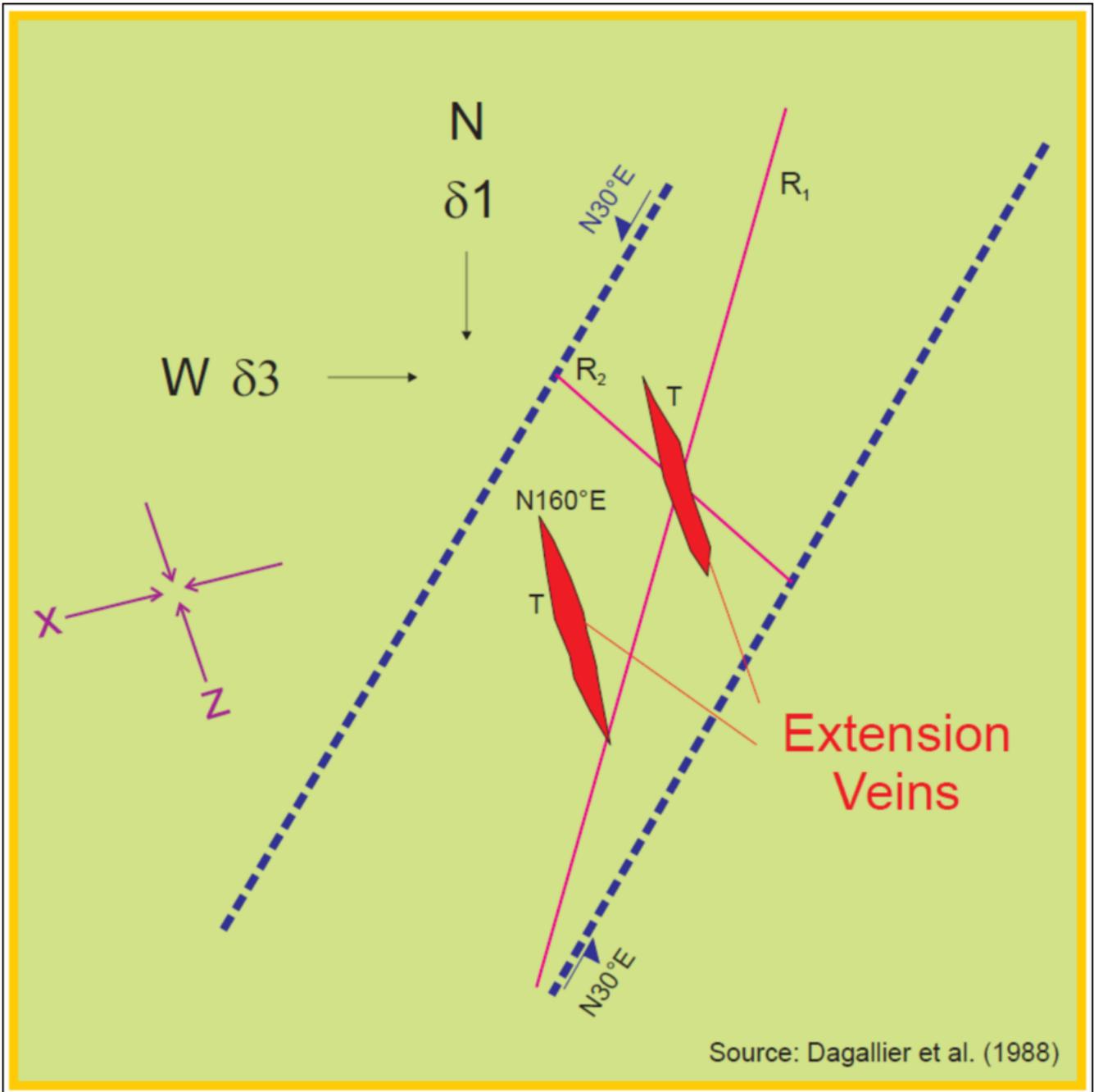


Figure 7-5 Stress Distribution in a Sinistral Shear Zone

Source: Boily (2014)

Figure 7-5 Description: Z = maximum shortening; X = maximum elongation; $R1-R2$ = conjugated Reidel system; T = extension gash; $\delta 1$ = maximum stress during deformation; $\delta 3$ = minimum stress during deformation.



Figure 7-6 Interpreted Traces of N150°E-Trending Shear Zones

Source: Boily (2014)

Figure 7-6 Description: Dextral shear zones developed in corridors previously affected by strong pyrophyllitization.

7.2.4 Supergene Weathering

Due to the extensive weathering to clay minerals, the Boumadine Deposit has a very light color that contrasts with the surrounding landscape. The mantos, “chapeau de fer” or “iron cap” alteration extends from 5 to 10 m depth. The mantos consists principally of goethite and jarosite with sparse hematite and no lepidochrosite. This mineralogical assemblage indicates that the oxidation fluids were strongly acidic. In this case, Mn, Zn, Cd, Ni and Co are highly mobile in the acid and sulphur-rich fluids and are commonly leached at surface. However, Ag, Au, Ba, Sr and Pb are immobile and form stable sulphosalts. The hydroxide-rich “mantos” has been partially mined out by artisanal workers for ochre and precious metals (Figure 7-7).



Figure 7-7 Oxidized Mineralized Vein at the Central Zone

Source: Boily (2014)

7.3 DEPOSIT GEOLOGY

The Boumadine Deposit has been traced on surface and in drilling as a curvilinear vein system for ~6,000 m along strike (Figure 7-8). Strike direction varies from mainly northwest to northerly and dip from steeply northeast to steeply southwest (Figure 7-9).

The Deposit consists of 45 mineralized domains that have been grouped into five separate zones. The South and Central Zones consist of up to 13 stacked mineralized vein domains. From the south end of the South Zone to the north end of the Central Zone, these domains extend for 4,800 m along strike, as much as 300 to 400 m across strike and a maximum of 1,000 m down-dip. The south end of the South Zone appears to be open to expansion by drilling along strike to the south. The South Zone appears to be offset dextrally along a northeast-trending fault from the Central Zone. The north end of the Central Zone appears to be offset sinistrally along a northeast-trending fault from the North Zone.

The North Zone consists of eight closely-spaced mineralized vein domains. This Zone is 650 m long, 5 to 10 m thick and 500 m down-dip. It strikes northwest and dips steeply southwest.

The Imariren Zone and the Tizi Zone are two sub-parallel, single mineralized vein domains that are 200 m apart in the south and 500 m apart in the north, strike northerly, and dip vertically. The Tizi Zone has been extended to 2.0 km in length, whereas Imariren has been traced over 1.2 km. Both zones extend 600 m down-dip.

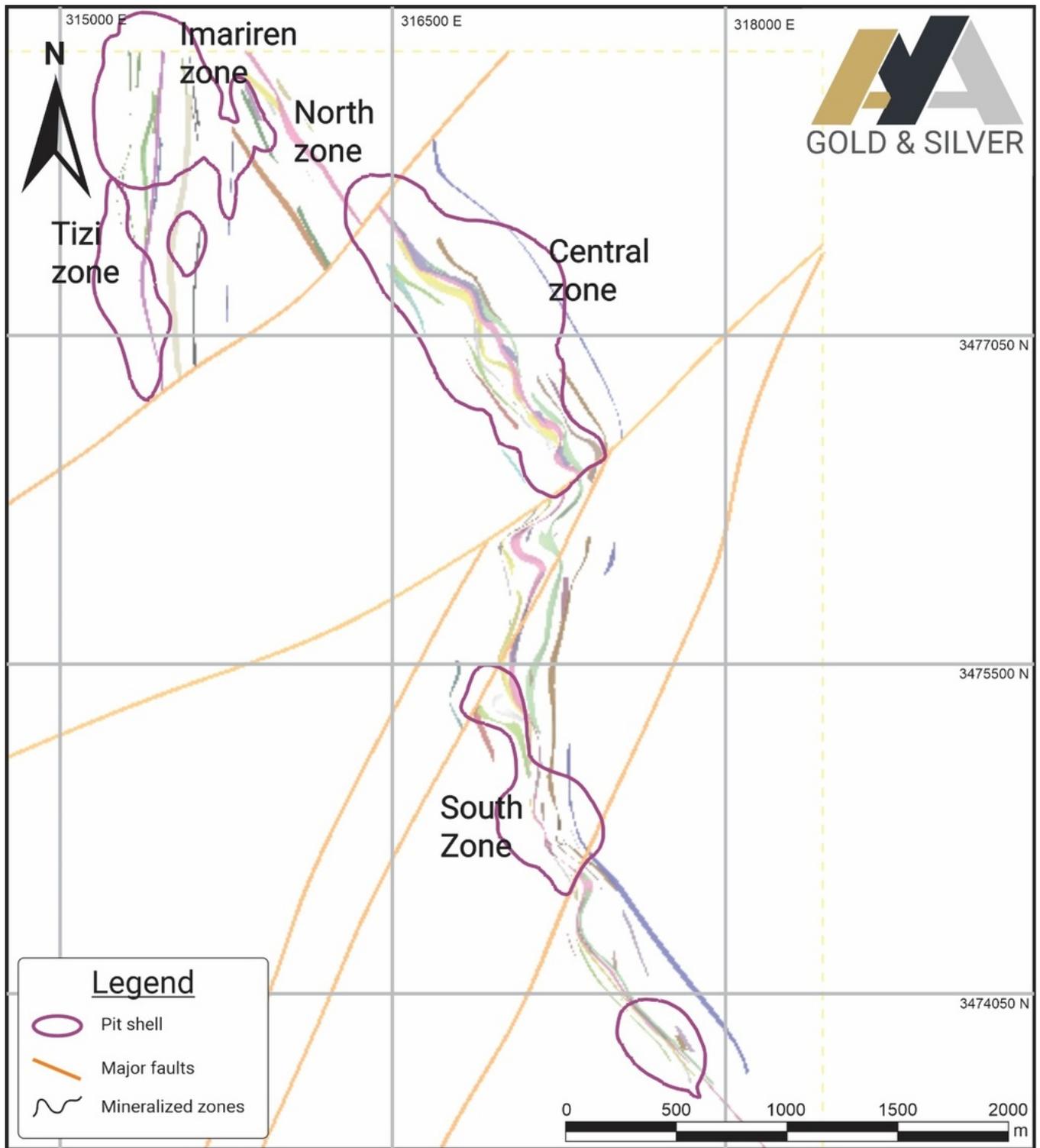


Figure 7-8 Surface Plan View of the Boumadine Deposit Mineralized Zones

Source: Aya press release dated February 24, 2025

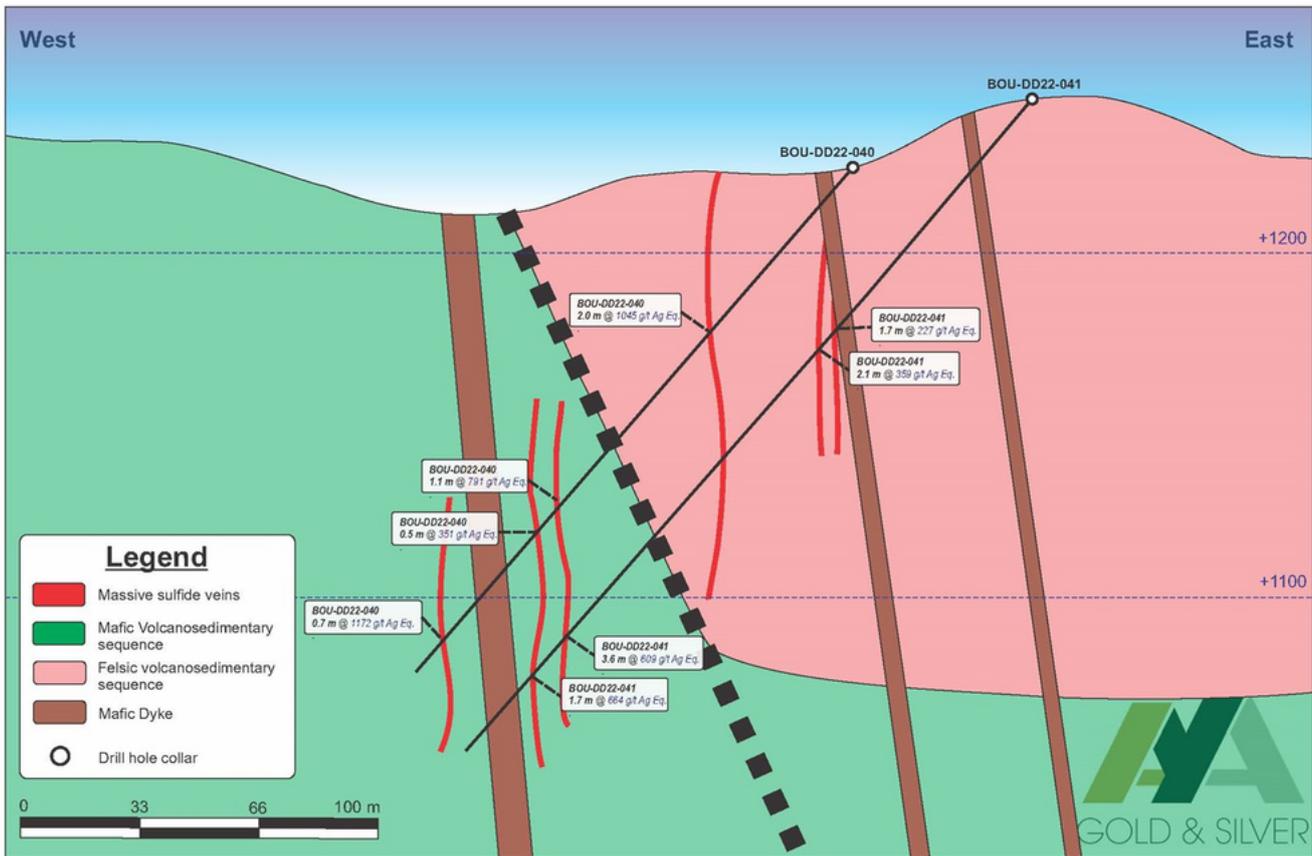


Figure 7-9 Cross-Sectional Projection 3,477,070 N of the Central Zone

Source: Aya press release dated February 14, 2023.

Note: The heavy dashed line marks a fault.

7.4 MINERALIZATION

The Boumadine mineralized zones generally consist of 1 to 4 m-wide massive sulphide lenses/veins oriented N20°W and dipping 70° east. The massive sulphide veins (>70% sulphide) are composed mainly of pyrite, sphalerite, galena, arsenopyrite and chalcopyrite, with subordinate amounts of cassiterite, silver-rich sulphosalts, stannite, enargite, bismuthinite, native copper and bismuth (Figures 7-10 to 7-13). The main mineralization zone is generally surrounded by a 1 to 10 m (locally up to 20 m) thick halo of 10 to 30% disseminated pyrite and two types of veinlets: 1) quartz-carbonate-galena-sphalerite veinlets; and 2) massive pyrite veinlets (Freton, 1988).

Within massive sulphide veins, zones of breccias with silicified angular fragments and round fragments have been completely replaced by pyrite. Those breccia zones underline the presence of syn-volcanic faults, which probably served as fluid pathways for the mineralization. In weathered felsic tuffs, breccia fragments can be replaced by pyrite, which locally forms large, mineralized sub-zones as much as 10 m thick. Those thick sub-zones are interpreted to be the upper part of the hydrothermal system. Geochemically, there is a strong positive correlation of gold with silver and copper and a weaker correlation of zinc with lead and molybdenum.



Figure 7-10 Brecciated and Slightly Oxidized Pyrite-Rich Mineralization

Source: Boily (2014)

Figure 7-10 Description: Typical brecciated and slightly oxidized pyrite-rich mineralization with quartz veinlets mined from polymetallic veins and collected from the Central Zone muck pile.



Figure 7-11 Galena-Rich Mineralization

Source: Boily (2014)

Figure 7-11 Description: Galena-rich mineralization with pyrite, pyrrhotite, chalcopyrite, sphalerite and second stage quartz veins that contain some of the Au and Ag mineralization. Collected from the Central Zone muck pile.



Figure 7-12 Pyrite-Sphalerite Mineralized Material from the Central Zone

Source: Boily (2014)

Figure 7-12 Description: Mineralized material sample BOU2012-02 collected from the muck pile near Shaft A of the Central Zone, historical Boumadine Mine.

U/Pb single zircon dating from a “chonolithic” rhyolite intrusion cutting the mineralized veins yielded an age of 553±16 Ma (Levresse, 2001). This result is consistent with a late Neoproterozoic maximum age for the mineralization.

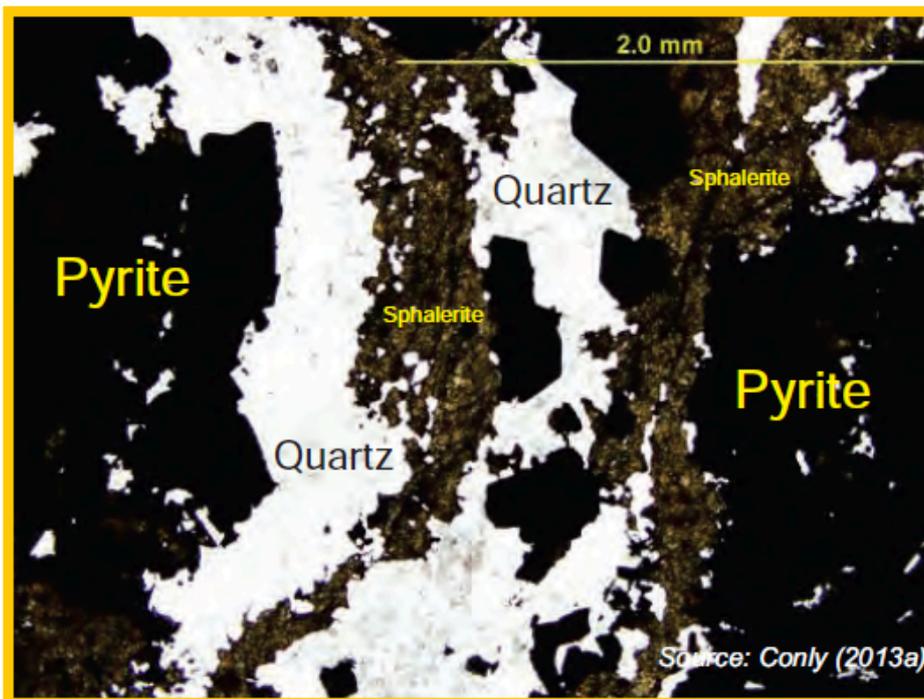
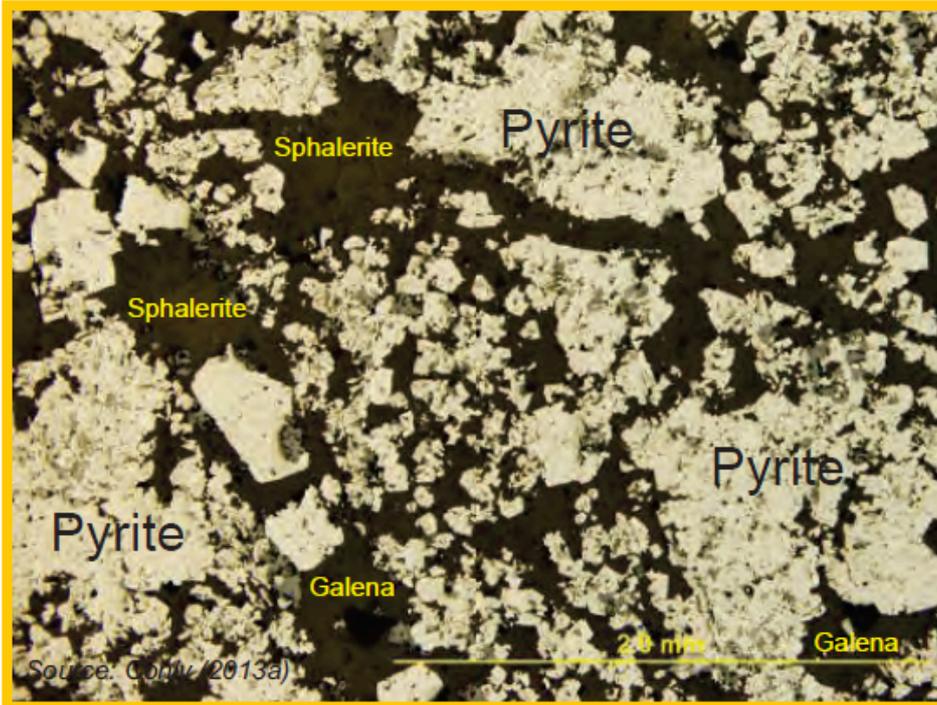


Figure 7-13 Massive Sulphide Minerals and Textures at Boumadine

Source: Boily (2014)

Figure 7-13 Description: A) Reflected light microphotograph showing the paragenetic relationship between pyrite, galena and sphalerite. **Sample BOU2012-02.** B) Transmitted light photomicrograph showing mineralogical and textural variations in sphalerite + quartz veins in sample BOU2012-02. Low Fe-bearing sphalerite (yellow-brown) occurs in the core of a diverging quartz vein that is cutting recrystallized coarse-grained massive pyrite.

8.0 DEPOSIT TYPES

Boumadine mineralization traditionally was considered to be hosted only in felsic tuffs (Abia et al., 2003; Bouabdellah and Levresse, 2016; Duplessis et al., 2019). However, drilling completed from 2017 to 2024 intercepted mineralized veins in mafic tuffs. Furthermore, the mineralized veins in the mafic tuffs appear to be more continuous, richer and thicker than in the felsic tuffs.

Hydrothermal alteration at Boumadine resembles that found in high sulphidation (acid) epithermal systems. Aluminous alteration is more proximal to sulphide-rich mineralized zones and the propylitic alteration more distal. Propylitization is generally induced by the convection of surface fluids, whereas aluminous alteration results from the contribution of acidic magma-derived fluids during degassing of andesite/diorite subvolcanic intrusions.

On the other hand, high-sulphidation deposits are composed of sulphides rich in S and Cu, such as tennantite and enargite. At Boumadine, the sulphides are mainly pyrite and minor sphalerite and galena, and trace chalcopyrite. This sulphide mineral affiliation is more like that of a volcanogenic massive sulphide type.

The current Company-preferred interpretation is that the mineralizing system at Boumadine developed under shallow submarine conditions in a graben setting (Bouabdellah and Levresse, 2016). In this model, magma-derived high-temperature acidic fluids/vapours containing Au and Ag ascend from the subvolcanic andesite/diorite intrusions and mix with circulating low-temperature, seawater-derived chlorinated fluids containing Fe, Zn and Pb. Mixing, cooling and wall rock reactions drive metal precipitation and deposition in volcanoclastic rocks below the seafloor.

Other shallow submarine, seawater-entrained polymetallic epithermal systems have been reported elsewhere (Hannington et al., 2005), with perhaps the best known example being at Milos Island (Greece) in the Aegean Sea (Alfieris et al., 2013).

9.0 EXPLORATION

Exploration activities completed by Aya on the Boumadine Mining License include trenching, hyperspectral surveys, airborne geophysics, mineral prospecting, geological mapping, grab sampling and drilling. Each of these activities is summarized below.

9.1 TRENCHING

One trench (37m) was excavated during the early 2022 drilling campaign on section 8125N between drill holes BOU-DD22-001 and BOU-DD22-006. The goal was to find mineralization at surface. Assaying failed to return any significant values. The trench was too shallow to expose the target and it was judged unnecessary to extend it, because mineralization was intersected in the subsurface by drill hole BOU-DD22-006.

9.2 HYPERSPECTRAL

At the end of 2021, Compagnie Générale de Géophysique – Veritas (“CGG”) carried out a satellite-based geological and mineral mapping study for Aya over the Boumadine Mining License. The aim of the study was to process satellite imagery and identify areas of possible mineral alteration and place these in a geological context. The study enabled the generation of geologically valid target areas that were subsequently visited in the field during geological mapping in 2022 and 2023. The program was completed in two stages: 1) a regional project outlining the main structural controls and areas of alteration carried out at 1:25,000 scale; and 2) a detailed 1:5,000 scale project that focused on the historical permit (LE-383661).

Several satellite datasets were supplied by CGG, and processed and interpreted over the Boumadine area. For the regional project, Sentinel-2 imagery was used for the geological interpretation with Copernicus elevation data being merged to enhance this process. ASTER imagery was utilized for its spectral range, which enables clay/iron/carbonate alteration related to hydrothermal alteration associated with intrusions to be mapped. For the detail mapping, WorldView-3 imagery was used for both the structural and spectral mineral mapping.

The mineral outputs exhibited a strong correlation with the main structures mapped (at 1:25,000 scale). There is significant argillitic alteration along the northwest-trending fault in the northeastern area of the Boumadine Mining License (Figure 9-1). There is also significant argillitic alteration along a northeast-trending fault slightly to west of the Boumadine Mining License. Mineral outputs also display a strong correlation with the underlying geology in the Northern Zone on the license area, with the altered rhyodacite-rhyolite ignimbrites clearly highlighted.

In early 2024, Aya re-engaged CGG to conduct a new satellite-based study over larger part of the Boumadine Property. The project has been carried out using high resolution satellite imagery to provide detailed 1:5,000 scale structural interpretation and 1:10,000 scale lithological classification with key areas of alteration mapped to 1:5,000 scale. This time the study integrated Geological maps and other spatial geological datasets provided by Aya.

The use of WorldView-3 satellite imagery has significantly improved lithological classification (up to 1:10,000 scale) and structural mapping (1:5,000 scale) compared to the previous 1:200,000 regional map. The higher resolution provided clearer boundaries and more detailed structural features. Additionally, CGG’s ASTER Bare Earth+ multi-spectral imagery was utilized to map large alteration footprints while filtering out non-geological artifacts. This integration identified new areas of interest beyond the Boumadine Deposit. The enhanced spatial and spectral resolution of WorldView-3 imagery allowed for the mapping of narrow intrusive dykes and faults (Figure 9-2), and the identification of intrusive not visible in optical imagery. At 1:5,000 scale, the structural mapping has helped establish relationships between faults, dyke intrusions, and alteration zones.

In June 2025, Aya asked CGG to complete a new satellite-based study to increase the land coverage to 1500 km². The study should be completed by mid-November with the delivery of the same outputs as the 2024 study.

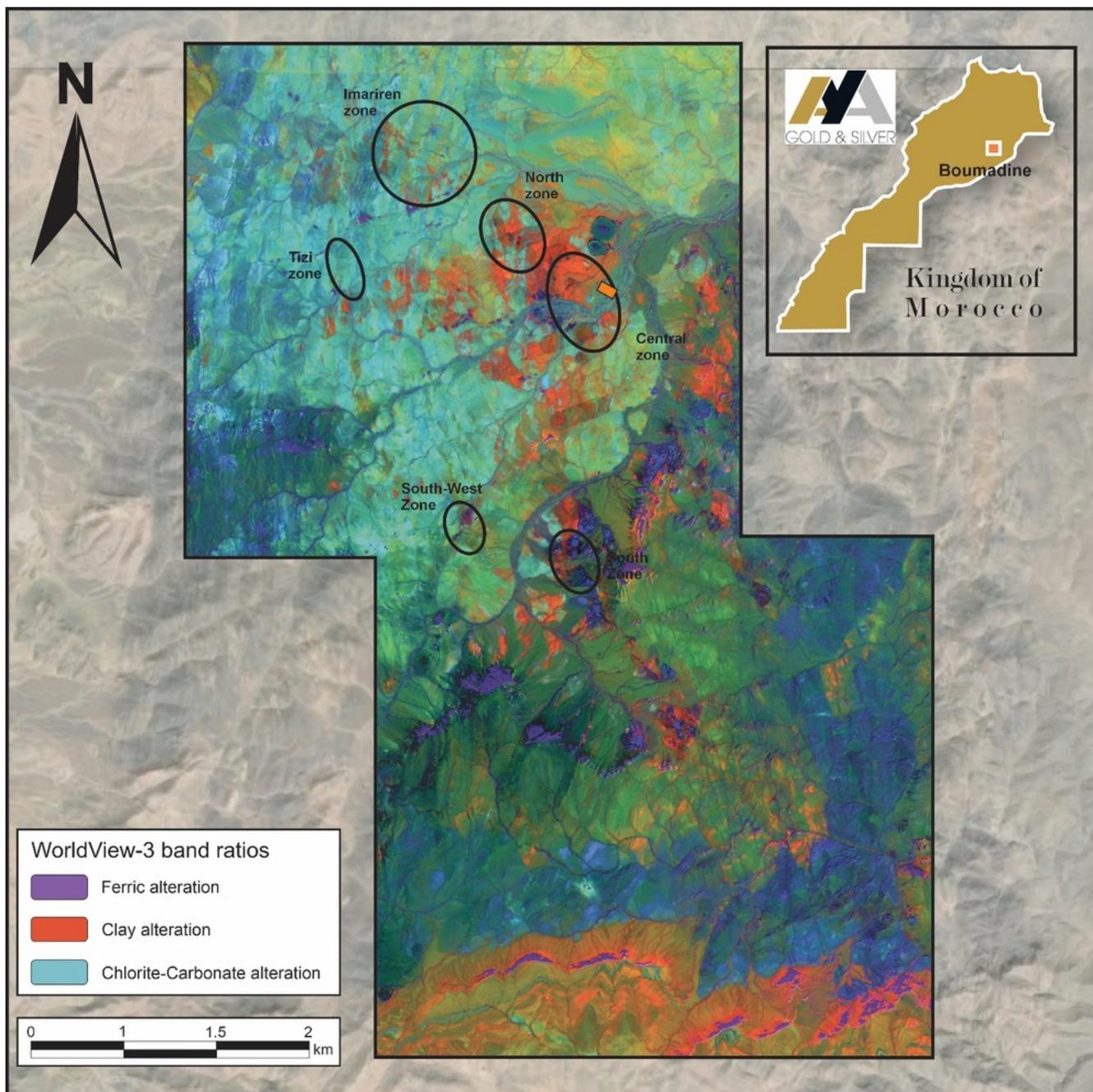


Figure 9-1 Map of the 2022 Hyperspectral Survey Completed on Boumadine

Source: Aya (April 2024)

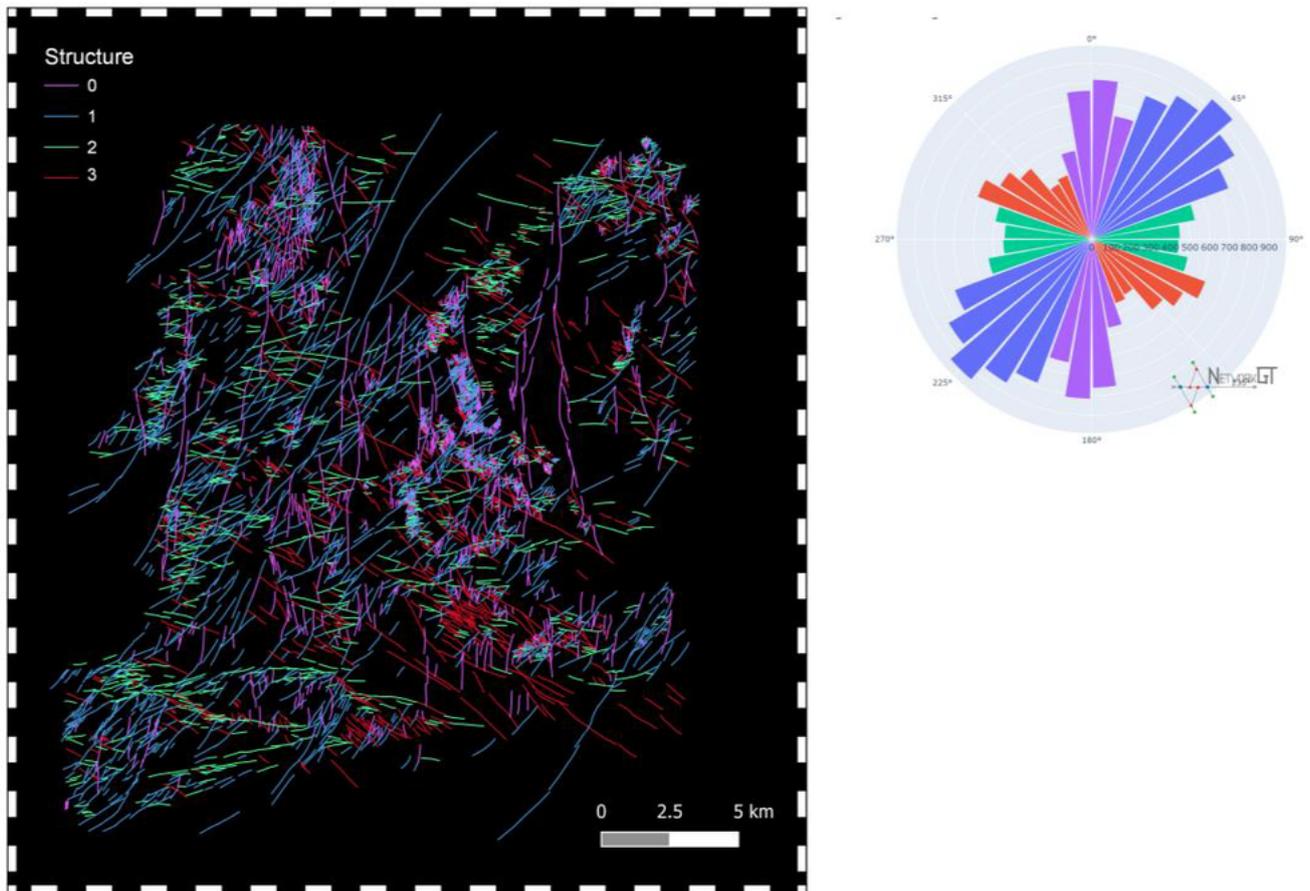


Figure 9-2 Map of the Interpreted Structures in the Boumadine Area, Colored by Orientation Set

Source: This Report

9.3 AIRBORNE GEOPHYSICAL SURVEYS

In March 2022, Geotech Ltd. completed an airborne geophysical survey over the Boumadine Mining License. A total of 366 linear-km (33 km²) was flown in a west - east (N90°E) direction with traverse line spacing of 100 m. Tie lines were flown perpendicular to the traverse lines at a spacing of 1,000 m.

Principal geophysical sensors included a versatile time domain electromagnetic (“VTEM”) system, airborne magnetics using a cesium magnetometer, and RSI ARGs RSX-5 spectrometer system.

The purpose of the survey was to provide magnetic, resistivity and radiometry coverage over the mining license with sufficient resolution to map the footprint of the known mineralization and any potential extensions.

The airborne results of both methods were high quality and meaningful. Electromagnetically, Boumadine features a prominent, large (>6.0 x 1.5 km), variably conductive, and north-south elongate resistivity low feature in the centre of the block that roughly coincides with all the historical mining work (Figure 9-3).

The resistivity low also coincides well with the northern half of the magnetic low (Figure 9-4). Those anomalies accurately mapped the known mineralization and supports extension of the favorable prospective corridor to the north and the south.

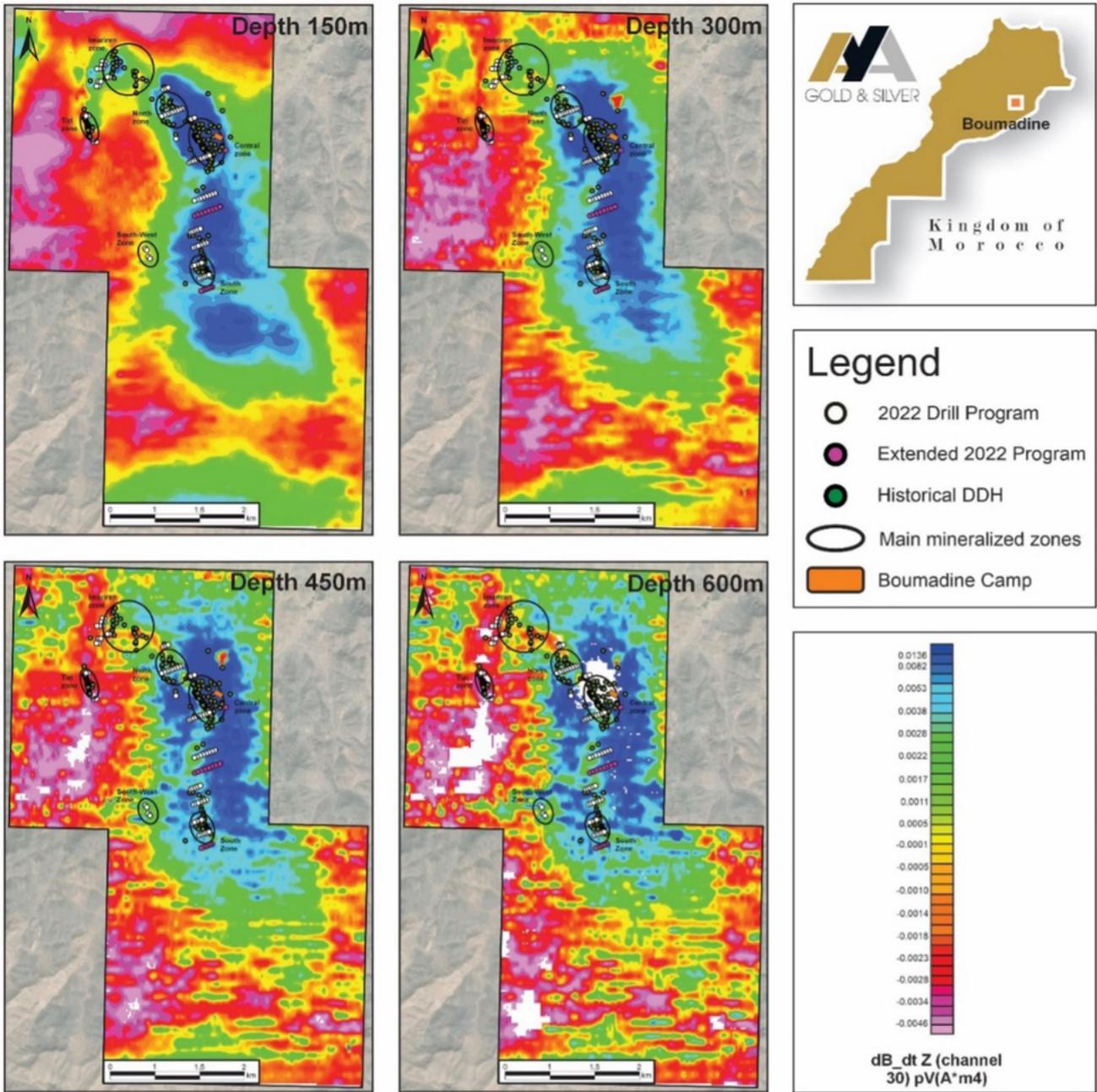


Figure 9-3 Boumadine 2022 VTEM Survey Depth Slices

Source: Aya (April 2024)

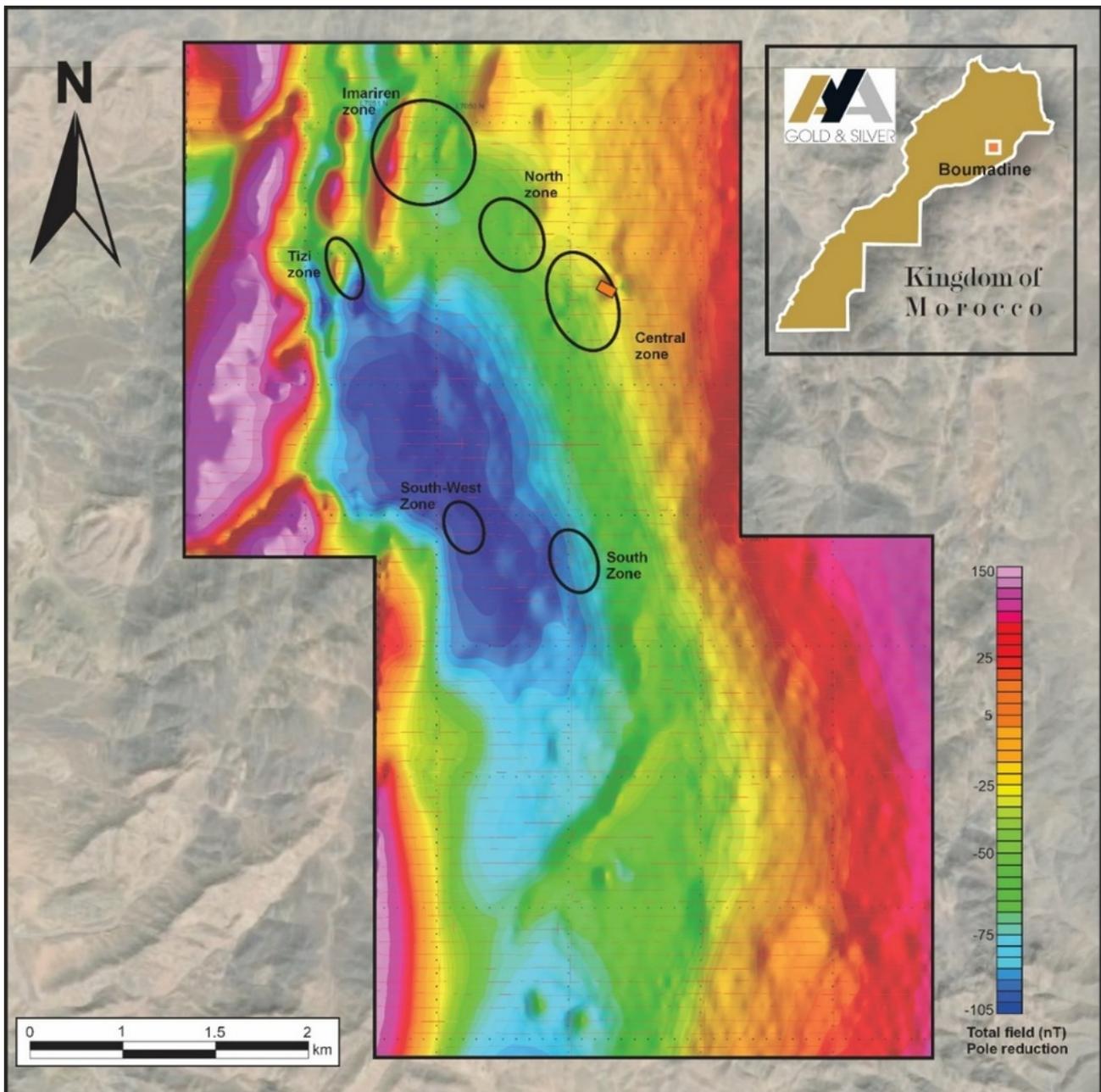


Figure 9-4 2022 Magnetic Survey Completed at Boumadine

Source: Aya (April 2024)

Following the success of the 2022 geophysical survey, a new airborne geophysical survey commenced on February 2, 2024, and concluded on July 18, 2024. Completed by Expert Geophysics Limited (EGL), this helicopter-borne MobileMT electromagnetic and magnetic survey covered three blocks: Boumadine, Boumadine West, and Boumadine East, encompassing all of Aya's permits in the Boumadine Property at the time of the survey (Figure 9-5).

The survey involved 105 production flights, covering a total of 14,353 line-km. Specifically, the survey was separated in three blocks: a) Boumadine covered 6,771 line-km over 609 square km, b) Boumadine West covered 4,535 line-km over 414 square km, and c) Boumadine East covered 3,047 line-m over 278 square km.

Electromagnetic readings were obtained using an EGL AFMAG & VLF MobileMT system, which includes an airborne three-component magnetic sensor and a base station with two horizontal electric components.

Additionally, a cesium vapor magnetometer in a separate towed-bird was used to measure the intensity of the Earth's magnetic field.

The geophysical acquisition was successful, in identifying multiple potentially parallel, on-trend conductive anomalies similar to those previously identified at Boumadine Deposit. Notably, a very large potential conductive anomaly was detected approximately 5 km west of Boumadine, that exhibit a similar orientation but stronger intensity than the Boumadine Deposit conductor. This extensive system also includes strong potential conductors oriented east-west. Additionally, the survey revealed the continuation of the conductivity anomaly; south of Boumadine Deposit and a series of new N340 and north-south oriented potential conductive anomalies.

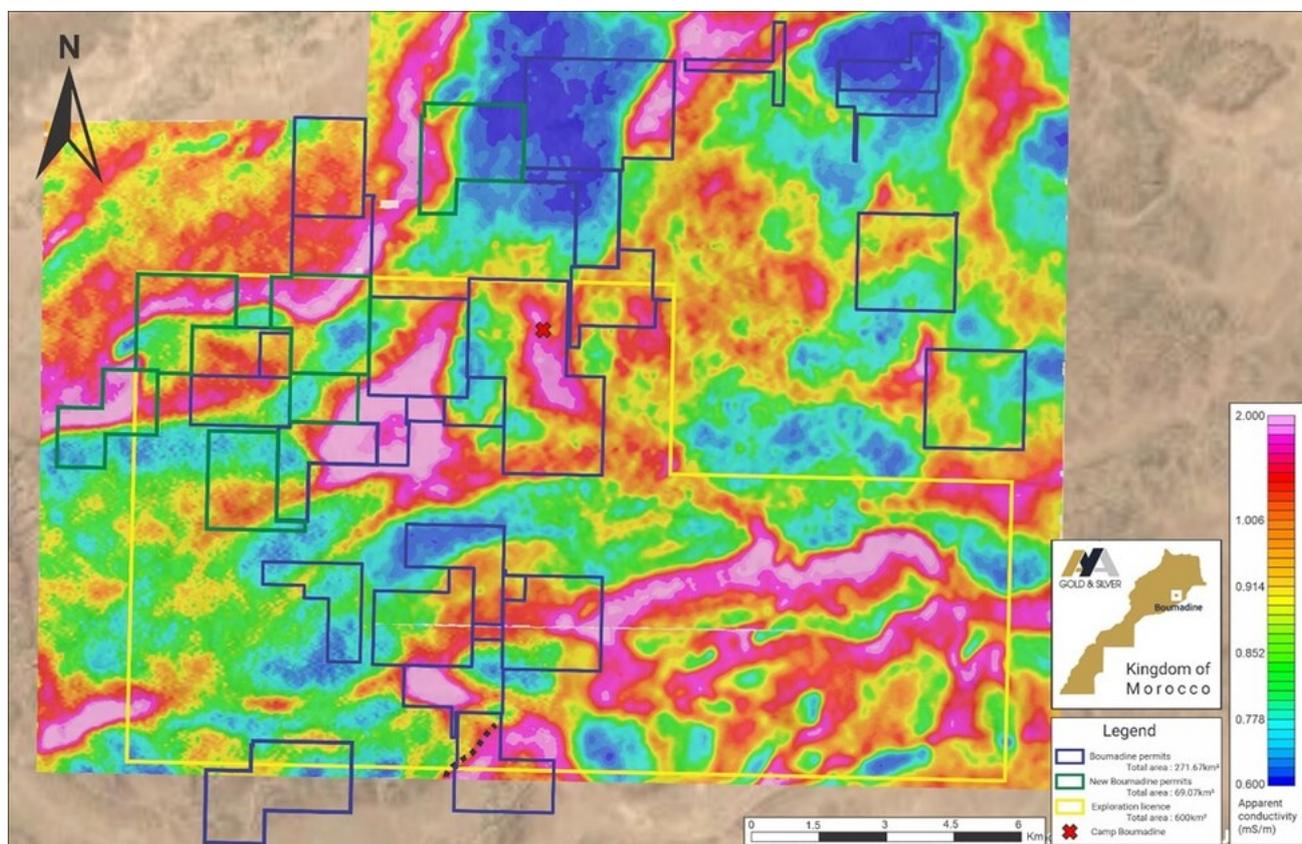


Figure 9-5 Location of New Boumadine Permits with 2024 Airborne Geophysics

Source: Aya press release dated February 24, updated October 2025

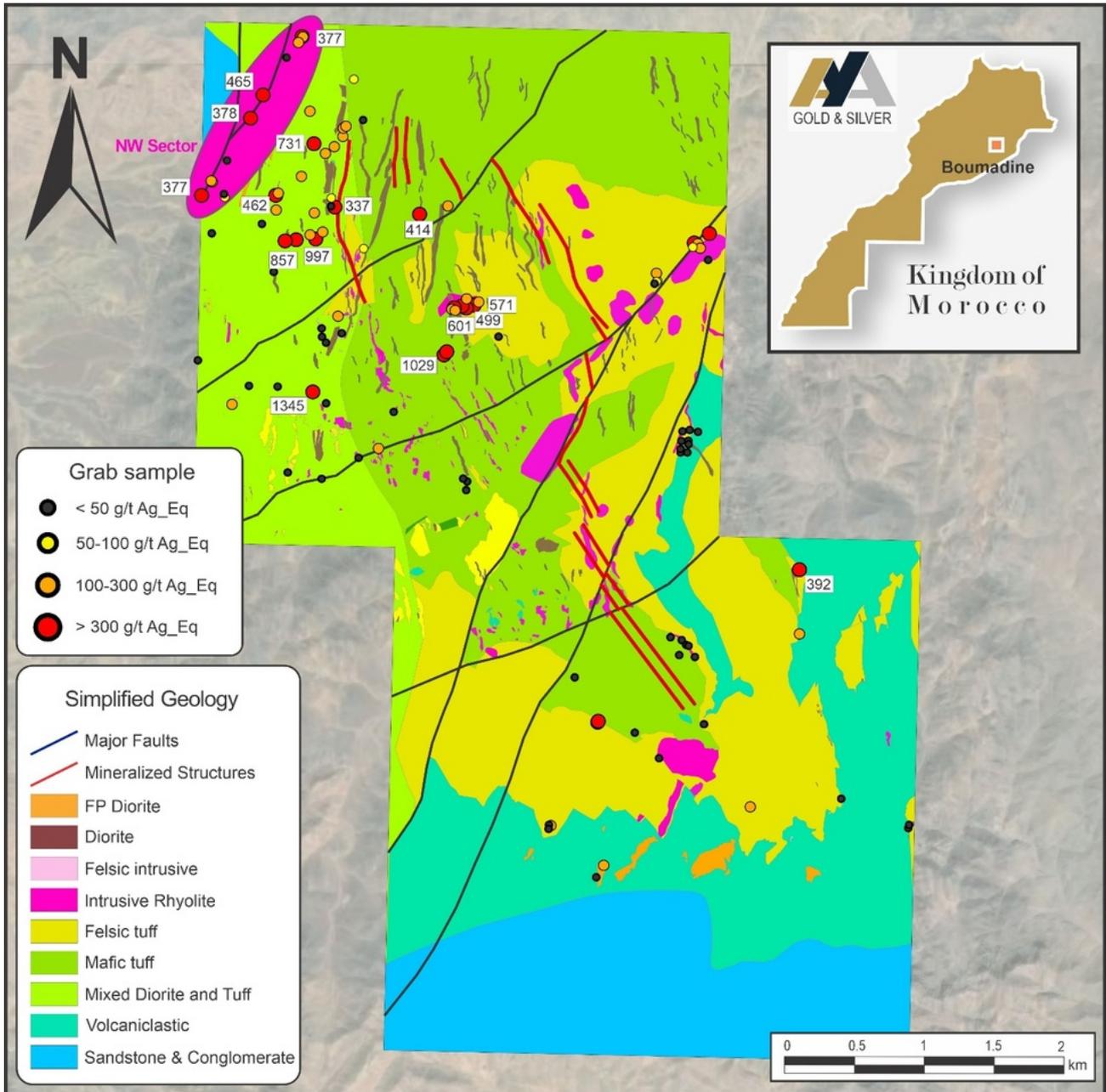
Note: Showing apparent conductivity at 175Hz

9.4 GEOLOGICAL MAPPING

Detailed mapping was conducted on Boumadine Mining License, with the objective to improve geological understanding of the mineralization and geological events.

From the mapping work, two major fault sets were recognized: 1) a fault event N030 that intersects the main Boumadine corridor (N340) and could be responsible for an Au enrichment and the Zn mineralization event; and 2) an N70°E fault event cutting both the N20°W and N70°E structures that appears to be responsible for a Ag-Pb +/- Cu mineralization event (Figure 9-6).

Following the proximal mapping around the deposit, a larger scale mapping of our regional permit was launched and remains ongoing. Six areas of interest with the highest potential have been subjected to more detailed mapping and will be drilled before the end of 2025. The different areas and the location of the samples are presented in Figure 9-7.



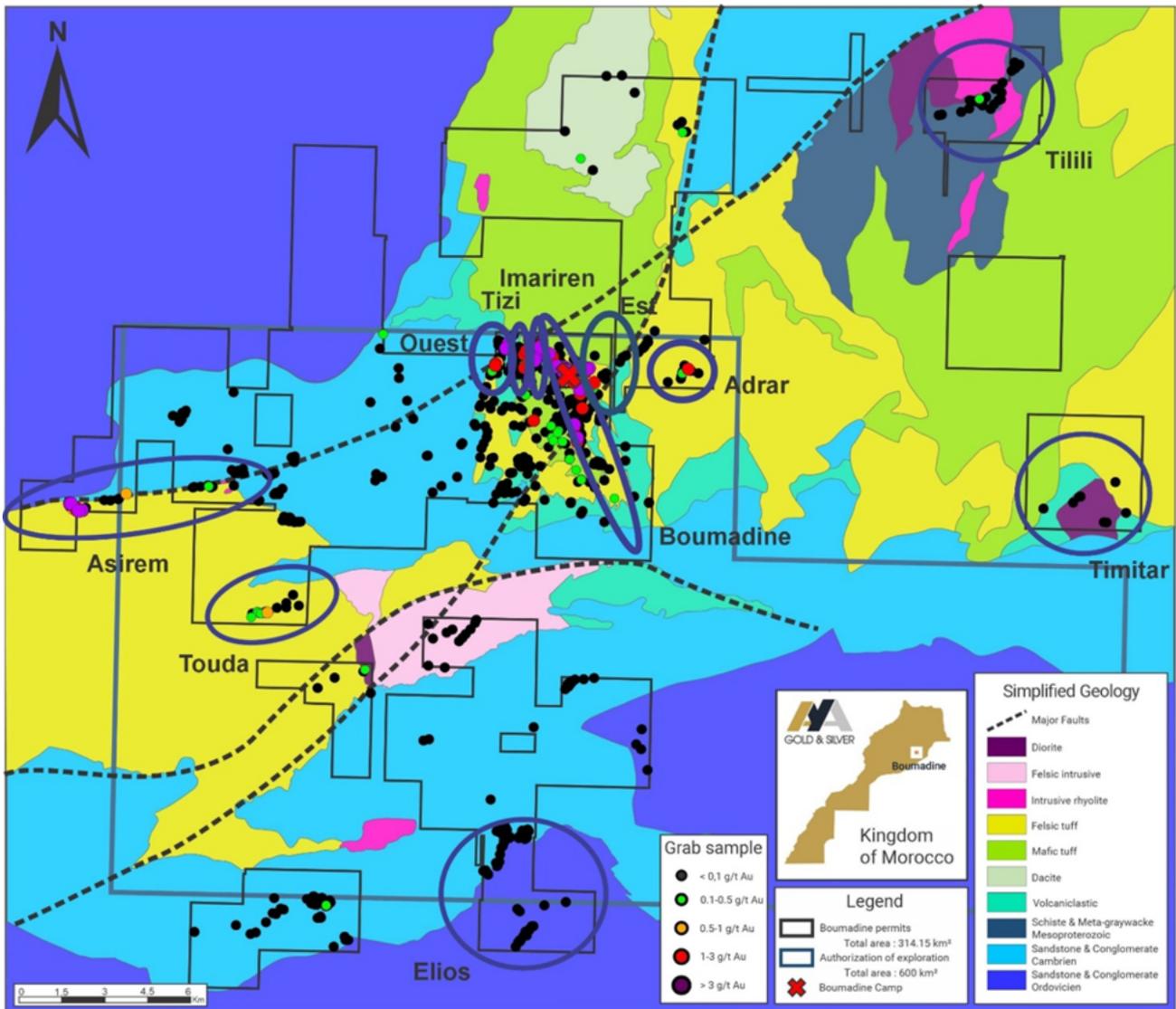


Figure 9-7 Simplified regional Geological Map with 2025 Surface Sample Locations

Source: Aya's Press Release dated July 7, 2025

9.5 GRAB SAMPLING

In 2023, 127 surface grab samples were taken, leading to the identification of a new mineralized structure to the northwest of the Property (see Figure 9-5 above). Samples from the structure, which can be followed for >1.5 km, graded up to 3.45 g/t Au (sample 2260129), 186 g/t Ag (sample 2274547), 9.40% Cu (sample 2274534), 27.40% Pb (sample 2274545), and 1.80% Zn (sample 2274547) (Table 9-1). The mineralization exhibits stockwork quartz-pyrite-chalcopyrite veinlets associated with silicified felsic dykes injected into a corridor of faults located at the contact of volcanic rocks and sedimentary rocks. This discovery shows the mineralization potential outside of the main Boumadine corridor.

Mapping and grab sampling activities continued throughout 2024 and 2025, extending to properties beyond the Boumadine Mining License. A total of 993 grab samples were collected on the Boumadine permits since 2022.

Table 9-1 Grab Sampling Assay Highlights

Sample ID	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)
2393526	12.20	226	4.10	0.78	0.18
2393529	7.40	211	4.20	1.64	0.33
2393527	4.78	81	1.00	0.32	0.04
2394147	3.93	163	0.05	0.43	0.84
2393509	3.50	90	3.25	0.62	0.02
2260129	3.41	40	0.75	0.16	0.06
2393498	3.38	58	36.64	0.02	0.01
2260130	3.37	23	0.28	0.04	0.03
2393412	3.34	32	4.04	0.68	0.04
2260131	2.78	44	0.54	0.06	0.02
2394143	2.66	132	0.13	0.28	1.00
2393516	2.28	9	0.49	0.06	0.03
2393518	2.13	16	1.12	0.84	0.08
2393470	2.13	26	0.27	0.05	0.01
2394146	2.02	82	0.03	0.43	0.47
2393251	0.03	747	59.58	0.01	0.01
2393421	0.16	482	0.12	8.49	0.01
2393505	0.11	475	0.40	18.93	0.54
2394526	0.03	410	22.47	1.38	0.10
2393295	0.03	359	1.33	39.98	0.33
2393287	0.03	354	1.32	43.43	10.52
2393318	0.15	326	7.90	9.13	0.56
2393549	0.19	323	11.18	19.29	1.81
2390608	0.03	300	29.56	0.01	0.01
2393467	0.03	17	48.15	0.04	0.02
2389945	0.03	74	34.46	0.05	0.05
2393329	0.06	5	23.97	0.01	0.01
2393482	0.03	24	20.91	0.02	0.02
2393537	0.03	32	20.18	0.28	0.02
2393473	0.03	17	19.76	0.02	0.01
2394519	0.03	56	19.27	5.07	13.46
2393368	0.06	116	18.24	0.55	0.60

Source: Aya's Press Release dated July 7, 2025

10.0 DRILLING

10.1 SUMMARY

From May 2022 to September 2025, Aya completed 660 diamond drill holes, 94 multipurpose holes (MP) and 49 Reverse circulation holes (RC), totaling 192,957 m. In 2025, Aya completed 273 drill holes, amounting to 109,240 m, on the Boumadine Property. Of these, 214 drill holes totaling 44,514 m were completed along the Boumadine Deposit and utilized for the 2025 Updated Mineral Resource Estimate (MRE) (Table 10-1). The drilling programs aimed to extend the mineralization of the North, Central, South, Tizi and Imariren Zones and test targets located farther from the main mineralized trend.

In addition, all historical drill holes from 2018 to 2021 were re-logged and resampled by Aya in 2023 for a total of 77 drill holes and 9,510 m of drill core. The historical BRPM drill holes and Maya's 2017 drill holes have not been retrieved in the Aya drill core yard. Therefore, it was not possible to proceed with re-sampling and the decision was made to include those drill holes for geological interpretation and exclude them from the MRE database. Information from the drill holes completed in 1992 by the SODECAT-BRPM has not been found, and those drill holes are not included in Aya's MRE database.

Table 10-1 Aya Diamond Drilling at Boumadine

Period	Company	Surface		Underground		Total Drill Holes	Total Metres	Comments
		Drill Holes	Metres	Drill Holes	Metres			
1964 to 1985	BRPM	70	13,467	48	2,124	138	15,591	Excluded from 2024 uMRE
2017	MAYA	14	3,158	---	---	14	3,158	
2018 to 2020	MAYA	77	9,507	---	---	77	9,507	Included in 2025 PEA
2022	AYA	86	19,702	---	---	86	19,702	
2023	AYA	171	65,572	---	---	171	65,572	
2024	AYA	219	107,683	---	---	219	107,683	
2025 (end September)	AYA	273	109,240	---	---	260	109,240	214 drill holes are included in PEA

Source: Aya press release dated September 30, 2025

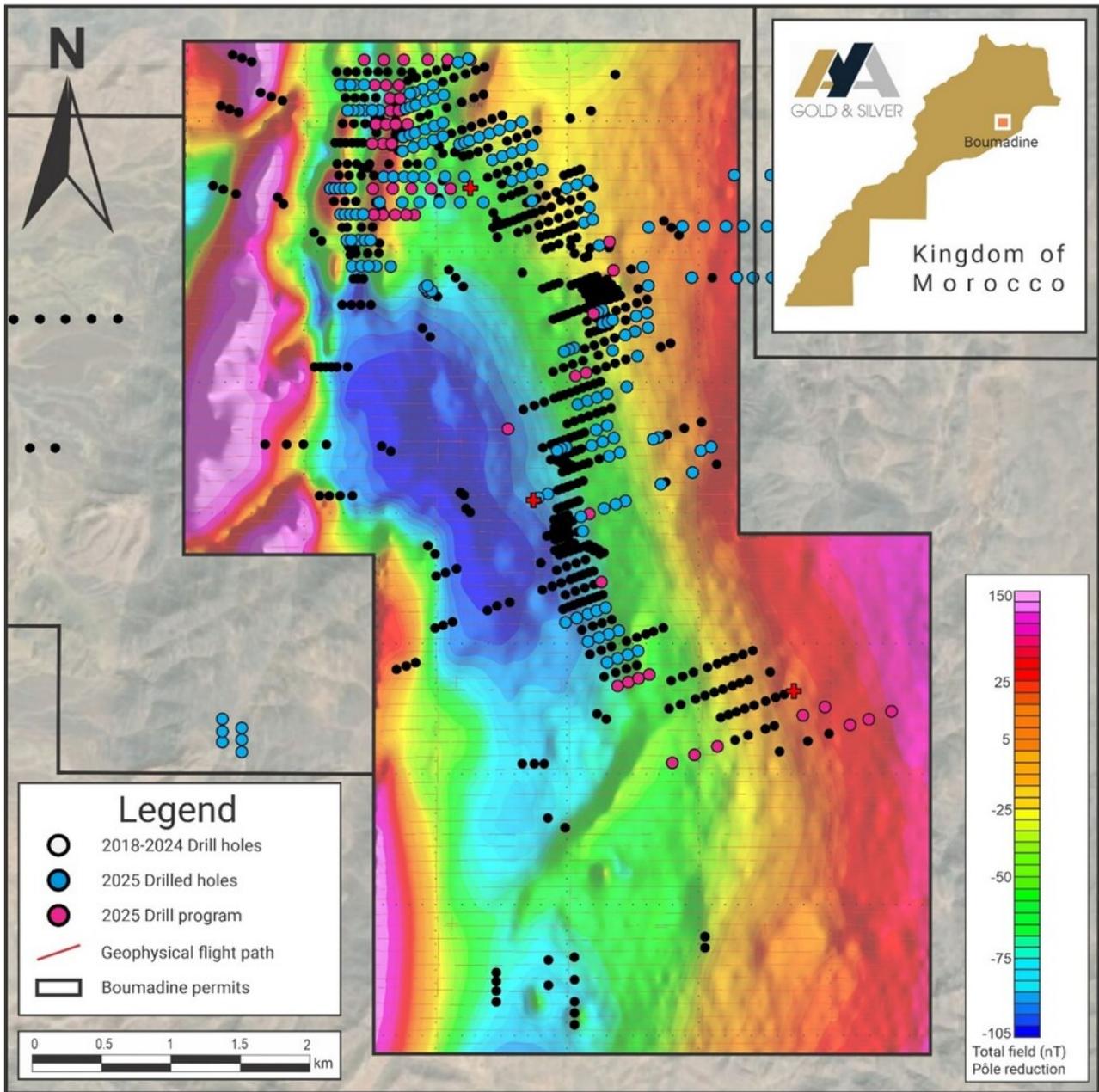


Figure 10-1 Aya Current MRE Drill Hole Collar Location Map

Source: Aya press release dated September 30, 2025

Note: Showing Magnetic Data – residual total field

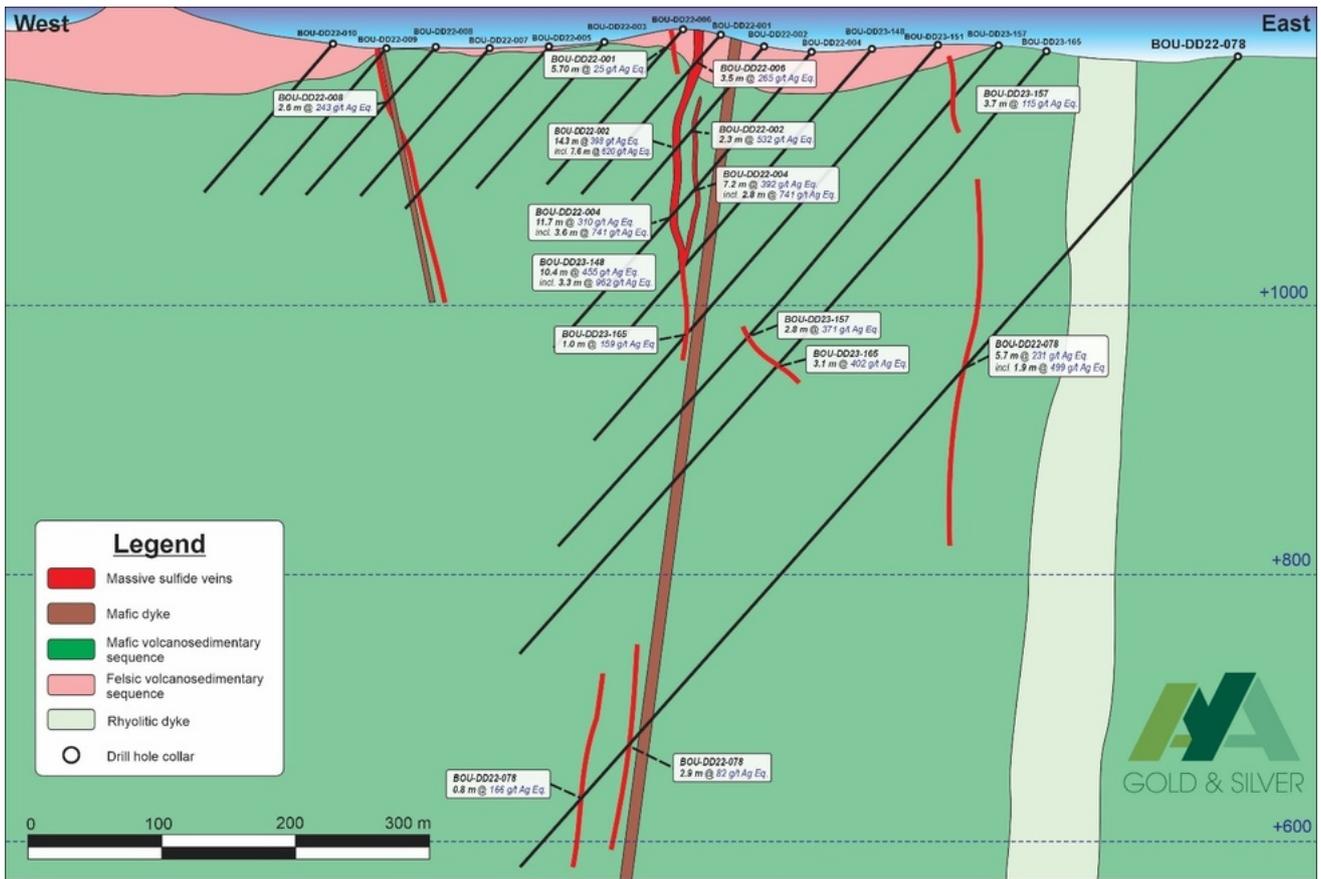


Figure 10-2 Interpreted Drill Hole Cross-Section Projection 8125N

Source: Aya (April 2024)

Figure 10-2 Description: Ag equivalent is based on 100% recovery with the following ratios: 1 g/t Au: 93.4 g/t Ag; 1% Cu: 130.4 g/t Ag; 1% Pb: 31.8 g/t Ag; 1% Zn: 54.1 g/t Ag. All assay values are uncut. All intersections are in core lengths, as true thickness remains undetermined at this stage.

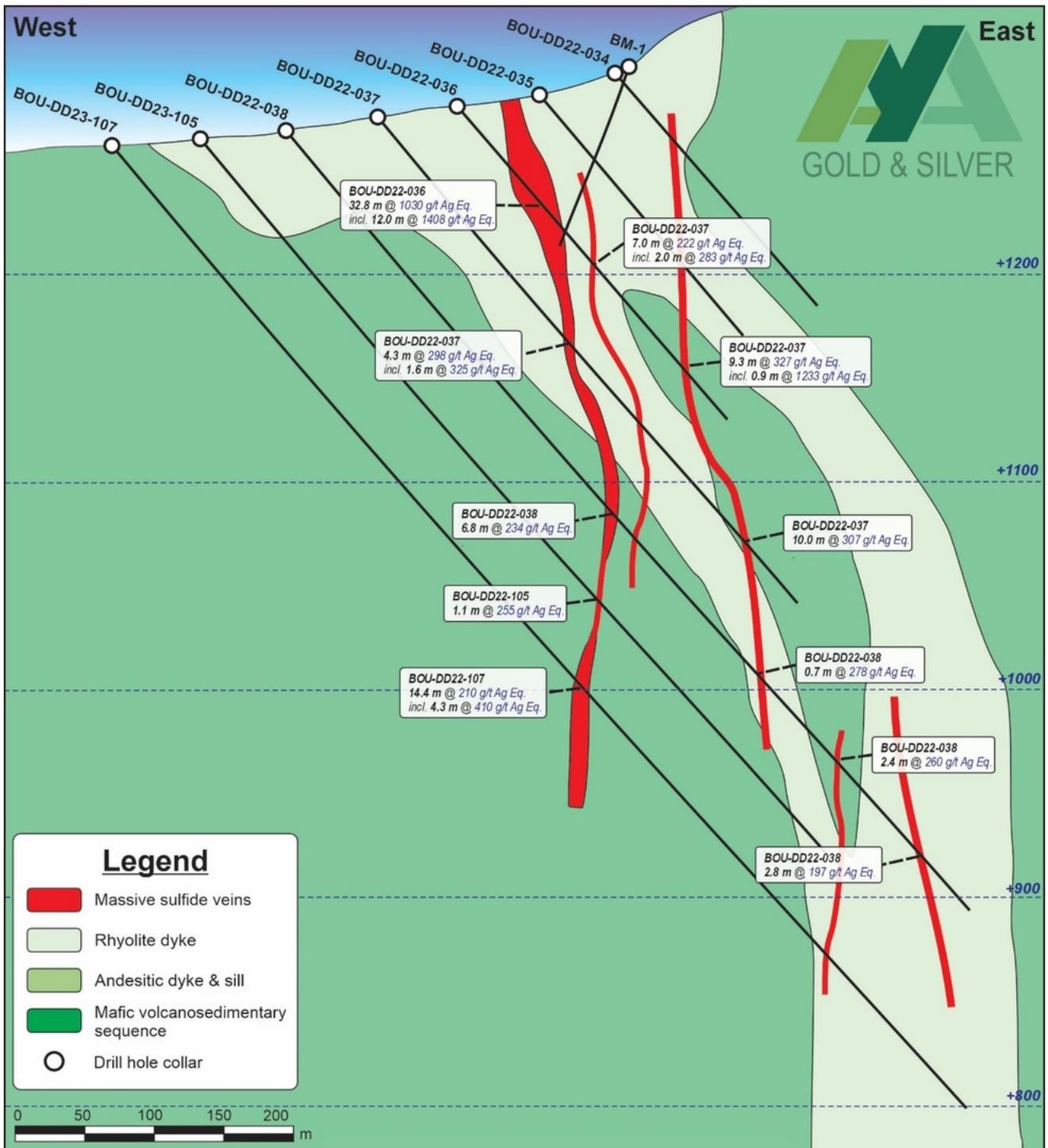


Figure 10-3 Interpretation of Drill Cross-Section Projection 6400N

Source: Aya (April 2024)

Figure 10-3 Description: Ag equivalent is based on a 100% recovery with the following ratios: 1 g/t Au: 93.4 g/t Ag; 1% Cu: 130.4 g/t Ag; 1% Pb: 31.8 g/t Ag; 1% Zn: 54.1 g/t Ag. All assay values are uncut. All intersections are in core lengths, as true thickness remains undetermined at this stage.

10.2 DRILLING PROCEDURES

Aya typically employs a drill pattern consisting of a series of drill lines-oriented perpendicular to the trend of mineralization. Most drill holes are oriented either N70°E or S70°W, depending on the terrain. For Tizi and Imariren, the drill holes are oriented E-W. The standard drill hole spacing for most of the Boumadine Deposit is 100 m. Locally, this spacing has been reduced to 50 m, whereas in other areas, such as the edges of the Boumadine Deposit, the spacing is 200 m.

GEOSOND Maroc SARL completed out the drilling program between 2022 and 2023. In 2024, FTE Drilling joined GEOSOND to continue executing the drill program. All core drilling to date has been completed using either HQ or NQ core size.

Prior to the start of drilling, the collars are set out in the field with a standard hand-held GPS with a precision of ± 3 m in easting and northing. Following completion of the drill holes, the collar locations are surveyed by a professional surveyor with a DGPS Trimble R8s. Drill rigs were aligned using a standard compass with back and front site pickets.

During drilling, the drill core is placed in standard plastic drill core boxes and every 3m run is marked by a labelled plastic drill core block. Each drill core box is labelled with the drill hole ID and a sequential box number. The drill core boxes are delivered by the drilling contractor every morning at the end of the night shift to the Boumadine drill core shack.

GEOSOND completed downhole orientation measurements using a Reflex EZ-Shot (by Reflex Instruments Inc.) until drill hole BOU-DD23-192. For all subsequently completed drill holes, a Devico Devi-flex was utilized. FTE systematically employed a Reflex EZ-Shot. Downhole surveys were performed by the drillers, with measurements initially taken at 12 metres and subsequently at 25-metres intervals. The data were communicated to the geologists each morning along with the daily drill report.

Following removal of the drill rig, a 1 m PVC tube is inserted into the drill hole and it is cemented. The drill hole number is written on the concrete base (Figure 10-4).



Figure 10-4 Drill Hole Collar Locations

Source: This Report

At the drill core shack, “from-to” of every drill core box is measured by geologists, and the core logging information is recorded in the logging software (GeoticLog). After logging and sampling, drill core boxes are stacked outside with one drill hole per cross-pile within a fenced and guarded enclosure around the Boumadine drill core shack (Figure 10-5).



Figure 10-5 Stacked Drill Core at Boumadine

Source: P&E (May 2024)

10.3 DRILLING RESULTS

In general, exploration and definition drilling has identified and further defined the distribution of mineralization in five areas: North Zone, Central Zone, South Zone, Tizi and Imariren. Drilling results on the main structure show strong continuity of the mineralization.

High-grade mineralization was intersected at Tizi, with two drill holes, BOU-DD24-306 and BOU-DD24-310, showing high gold concentrations (20.05g/t Au over 1.5m; and 23.34 g/t Au over 1.6m, respectively). The mineralization remains hosted in massive sulphide veins, characterized by a higher amounts of arsenopyrite.

A new style of mineralization has also been identified from in a drill hole along an east-west structure (BOU-DD24-329) which has returned high-grade Ag results (1,937 g/t Ag over 1.9m). This structure shows stockworks of brecciated carbonate-quartz-pyrite-chalcopyrite ± tetrahedrite veinlets with low temperature texture.

Table 10-2 Significant intercepts from the 2024 program

Drill Hole	Section	Zone	From	To	Length*	Au	Ag	Cu	Pb	Zn	Mo	Ag Eq**
			(m)	(m)	(m)	(g/t)	(g/t)	(%)	(%)	(%)	(g/t)	(g/t)
BOU-DD23-214	8850N	Main	214.0	223.0	9.0	4.77	61	0.12	0.07	0.19	5	535
Including			216.2	221.5	5.3	6.61	90	0.18	0.10	0.27	6	749
BOU-DD23-218	8850N	Para	244.3	247.5	3.2	14.72	19	0.02	0.19	0.15	4	1,411
BOU-DD23-218	8850N	Para	252.6	256.8	4.2	13.59	115	0.10	0.13	0.12	3	1,409
BOU-DD23-218	8850N	Main	280.3	286.1	5.8	9.21	80	0.21	0.06	0.14	8	978
Including			280.3	284.3	4.0	13.05	108	0.29	0.07	0.19	8	1,377
BOU-DD23-220	6575N	Main	105.0	115.9	10.9	1.77	91	0.09	1.72	4.53	133	575
Including			112.3	114.7	2.4	6.26	261	0.26	1.24	6.56	16	1,275
BOU-DD23-220	6575N	Para	133.4	136.8	3.4	0.89	76	0.26	2.47	6.97	7	649
BOU-DD23-223	6525N	Main	131.6	169.9	38.3	1.53	311	0.04	1.80	4.40	101	763
Including			144.9	155.9	11.0	2.34	494	0.06	1.89	3.93	36	996
BOU-DD23-225	9325N	Para	47.4	53.7	6.3	1.44	54	0.02	0.85	5.34	12	508
Including			50.8	53.7	2.9	2.88	86	0.04	0.96	9.90	21	927
BOU-DD23-227	9325N	Main	259.7	268.3	8.6	3.34	18	0.07	0.13	0.45	6	369
Including			263.5	268.3	4.8	5.42	21	0.07	0.08	0.11	7	545
BOU-DD23-228	6300N	Main	267.2	276.1	8.9	1.99	81	0.02	1.03	3.36	59	488
Including			267.7	273.0	5.3	3.09	119	0.03	1.45	4.66	87	715
BOU-DD23-229	6525N	Para	111.9	116.0	4.1	1.6	175	0.12	1.86	7.32	259	810
BOU-DD23-230	6575N	Main	166.6	184.2	17.6	2.64	247	0.27	1.24	7.74	86	991
Including			177.6	181.3	3.7	2.91	651	0.08	3.18	11.49	106	1,662
BOU-DD23-230	6575N	Para	188.2	202.3	14.1	2.78	97	0.24	0.41	6.52	31	755
BOU-DD23-245	6450N	Main	198.4	206.5	8.1	0.73	122	0.02	2.32	4.68	68	524
Including			198.4	200.2	1.8	1.83	299	0.06	4.83	13.08	269	1,355
BOU-DD23-248	6450N	Main	329.5	335.4	5.9	5.94	59	0.13	0.95	8.75	23	1,136
BOU-DD23-251	6450N	Main	345.9	355.3	9.4	2.66	32	0.04	0.21	4.39	14	531
Including			346.4	348.8	2.4	7.99	91	0.13	0.30	15.77	19	1,719
Including			314.0	318.8	4.8	6.76	36	0.06	0.11	0.16	0	569

Drill Hole	Section	Zone	From	To	Length*	Au	Ag	Cu	Pb	Zn	Mo	Ag Eq**
			(m)	(m)	(m)	(g/t)	(g/t)	(%)	(%)	(%)	(g/t)	(g/t)
BOU-DD23-265	8850N	Main	338.0	341.1	3.1	16.25	86	0.13	0.14	0.12	0	1,355
BOU-DD23-265	8850N	Para	366.0	374.0	8.0	4.51	58	0.23	0.20	0.32	0	442
Including			369.7	372.3	2.6	12.17	160	0.69	0.31	0.51	0	1,186
BOU-DD24-284	9950N	Imariren	439.7	441.6	1.9	15.7	91	0.16	0.06	0.05	4	1,317
BOU-DD24-306	3478100	Tizi	314.1	317.1	3.0	11.48	89	0.24	0.15	0.78	3	1,021
Including			314.1	315.6	1.5	20.05	133	0.37	0.20	1.37	2	1,755
BOU-DD24-310	34777500	Tizi	58.0	71.7	13.7	4.90	42	0.06	0.37	0.35	9	445
Including			58.0	59.6	1.6	23.34	148	0.20	0.41	0.50	17	1,988
BOU-DD24-310	34777500	Tizi	281.4	282.4	1.0	0.08	7,820	0.17	5.50	1.46	7	8,036
BOU-DD24-329	East-West	East-West	142.1	144.0	1.9	0.03	1,937	1.66	0.16	0.17	7	2,110
BOU-DD24-353	East-West	East-West	354.0	356.3	2.3	0.06	107	0.26	4.78	36.67	5	1,300

Source: Aya press releases dated January 21,2025

Notes:

* All assay values are uncut. All intersections are core lengths, as true width remains undetermined at this stage.

**Ag equivalent is based on 100% recovery with the following ratios: 1 g/t Au: 93.4 g/t Ag; 1% Cu: 130.4 g/t Ag; 1% Pb: 31.8 g/t Ag; 1% Zn: 54.1 g/t Ag.

Table 10-3 Significant intercepts from the 2025 program

DDH No.	Section	Zone	From	To	Au	Ag	Length*	Cu	Pb	Zn	Mo	Ag Eq**
			(m)	(m)	(g/t)	(g/t)	(m)	(%)	(%)	(%)	(g/t)	(g/t)
BOU-DD24-387	5000N	Boumadine	378.3	381.0	0.37	297	2.7	0.0	3.7	3.2	74	497
BOU-DD24-442	6425N	Boumadine	173.7	178.3	0.20	73	4.6	0.1	1.3	4.9	178	248
BOU-DD24-453	7025N	Boumadine	164.4	169.6	2.68	38	5.2	0.2	0.2	0.7	12	284
Including			164.4	168.1	3.44	45	3.7	0.2	0.1	0.2	14	341
BOU-DD24-465	9050N	Boumadine	128.8	139.4	0.48	81	10.6	0.0	1.3	2.0	9	199
BOU-DD24-472	3477600	Tizi	125.8	128.7	3.78	29	2.9	0.1	0.7	1.8	17	394
BOU-DD24-473	10250N	Boumadine	282.7	288.1	1.48	54	5.4	0.1	1.1	1.7	6	243
BOU-DD24-474	3477600	Tizi	48.0	52.0	2.38	37	4.0	0.0	0.6	2.5	56	302
BOU-DD24-475	10250N	Boumadine	359.2	369.3	0.94	50	10.1	0.1	1.0	1.4	6	189
BOU-DD24-475	10250N	Boumadine	405.4	407.1	5.62	257	1.7	0.4	0.3	3.5	6	820

DDH No.	Section	Zone	From	To	Au	Ag	Length*	Cu	Pb	Zn	Mo	Ag Eq**
			(m)	(m)	(g/t)	(g/t)	(m)	(%)	(%)	(%)	(g/t)	(g/t)
BOU-DD24-478	3477600	Tizi	459.4	462.7	4.61	75	3.3	0.1	0.1	0.5	11	460
BOU-DD24-487	9050N	Boumadine	483.1	486.5	3.36	225	3.4	0.1	0.1	0.0	2	502
BOU-DD25-493	9050N	Boumadine	423.0	431.0	1.19	16	8.0	0.0	0.2	1.1	3	142
BOU-DD25-497	9850N	Imariren	184.6	190.8	1.93	18	6.2	0.1	0.1	0.9	3	201
BOU-DD25-500	9050N	Boumadine	537.0	543.6	2.41	50	6.6	0.1	0.2	0.1	2	255
Including			540.1	543.6	3.11	60	3.5	0.1	0.2	0.1	4	321
BOU-DD25-501	9850N	Imariren	93.0	99.0	1.73	48	6.0	0.1	0.2	0.5	16	210
Including			93.6	96.0	3.78	93	2.4	0.2	0.1	0.2	15	413
BOU-DD25-501	9850N	Imariren	192.5	200.1	1.68	7	7.6	0.1	0.0	0.0	6	146
BOU-DD25-502	9250N	Boumadine	161.5	169.3	0.99	26	7.8	0.1	0.4	3.0	17	191
Including			161.5	162.7	3.06	95	1.2	0.3	1.5	15.1	10	763
BOU-DD25-504	10050N	Imariren	141.0	147.6	3.69	46	6.6	0.0	0.2	0.3	3	349
Including			141.0	142.2	10.34	37	1.2	0.0	0.1	0.1	3	851
BOU-DD25-506	9850N	Boumadine	297.3	302.3	3.19	48	5.0	0.1	0.1	0.1	2	312
Including			297.3	299.5	4.45	50	2.2	0.1	0.0	0.1	2	413
BOU-DD25-509	10050N	Imariren	282.1	291.4	2.18	62	9.3	0.1	0.7	1.5	2	296
Including			289.8	291.4	7.16	215	1.6	0.5	0.2	3.2	3	897
BOU-DD25-511	10050N	Imariren	311.1	316.0	1.95	270	4.9	0.1	0.1	0.6	8	449
BOU-DD25-512	10050N	Imariren	338.0	339.0	15.86	1	1.0	0.0	0.1	0.3	4	1247
BOU-DD25-513	9850N	Boumadine	365.3	367.1	5.19	118	1.8	0.5	0.2	1.0	2	591
BOU-DD25-513	9850N	Boumadine	374.8	377.7	5.52	109	2.9	0.3	0.2	5.2	4	698
BOU-DD25-516	9050N	Boumadine	498.0	507.0	1.24	6	9.0	0.1	0.1	0.2	2	114
BOU-DD25-516	9050N	Boumadine	648.5	649.0	66.66	111	0.5	0.1	1.0	1.6	6	5373
BOU-DD25-544	3477200	Tizi	31.9	44.2	0.31	60	12.3	0.0	1.1	1.9	4	161
BOU-DD25-547	3477200	Tizi	117.0	126.0	0.42	80	9.0	0.0	1.8	3.6	5	248
BOU-DD25-550	9050N	Boumadine	54.9	59.5	1.33	101	4.6	0.0	1.1	1.6	8	272
BOU-DD25-572	8850N	Boumadine	351.8	364.0	2.60	23	12.2	0.1	0.1	0.0	5	232
Including	8850N	Boumadine	352.6	354.8	5.28	49	2.2	0.1	0.1	0.1	4	473
Including	8850N	Boumadine	359.3	361.7	6.29	48	2.4	0.1	0.1	0.0	7	551
BOU-DD25-572	8850N	Boumadine	381.6	388.1	2.66	21	6.5	0.0	0.2	0.4	9	246

DDH No.	Section	Zone	From	To	Au	Ag	Length*	Cu	Pb	Zn	Mo	Ag Eq**
			(m)	(m)	(g/t)	(g/t)	(m)	(%)	(%)	(%)	(g/t)	(g/t)
Including	8850N	Boumadine	383.0	385.0	6.49	25	2.0	0.0	0.2	0.2	8	544
BOU-DD25-572	8850N	Boumadine	573.0	574.8	9.77	65	1.8	0.2	0.7	2.8	19	929
BOU-DD25-584	8750N	Boumadine	325.7	334.7	4.04	41	9.0	0.1	0.1	0.1	7	369
Including	8750N	Boumadine	327.0	331.2	5.98	51	4.2	0.2	0.1	0.1	8	535
BOU-DD25-584	8750N	Boumadine	594.0	595.9	4.11	17	1.9	0.0	0.1	0.3	3	345
BOU-DD25-589	8750N	Boumadine	341.0	351.2	2.90	29	10.2	0.1	0.1	0.1	11	271
Including	8750N	Boumadine	345.4	348.0	7.46	71	2.6	0.4	0.1	0.1	15	692
BOU-MP24-015	3478300	Tizi	475.0	476.6	0.05	774	1.6	0.1	0.1	0.1	4	788
BOU-MP25-026	9250N	Boumadine	238.5	243.2	0.58	36	4.7	0.1	1.3	7.2	12	296
BOU-MP25-028	9250N	Boumadine	392.1	396.4	3.27	19	4.3	0.1	0.1	0.3	8	286
Including			392.1	394.2	5.46	26	2.1	0.1	0.1	0.0	10	460
BOU-MP25-029	9450N	Boumadine	93.0	95.0	1.16	100	2.0	0.1	7.4	4.7	53	498
BOU-MP25-069	8075N	Boumadine	79.1	91.0	1.45	35	11.9	0.1	0.3	1.4	18	195
BOU-RC25-024	7175N	Boumadine	107.0	119.0	1.11	19	12.0	0.1	0.6	1.3	15	157

Source: Aya press releases, 2025

Notes:

* All assay values are uncut. All intersections are core lengths, as true width remains undetermined at this stage.

**Ag equivalent is based on 100% recovery with the following ratios: 1 g/t Au: 93.4 g/t Ag; 1% Cu: 130.4 g/t Ag; 1% Pb: 31.8 g/t Ag; 1% Zn: 54.1 g/t Ag.

11.0 SAMPLE PREPARATION, ANALYSIS AND SECURITY

The following section discusses the sample preparation, analyses and security procedures carried out by Aya for the Boumadine Property between 2018 and 30 September 2025.

11.1 SAMPLE PREPARATION

11.1.1 Logging and Sampling

Logging and sampling of drill core are performed at Aya's onsite logging facility; a large warehouse with ample space for logging tables and direct vehicular access for drill core box delivery.

Geotechnical personnel align drill core pieces and check for gaps. Devicore BTT is utilized during drilling to indicate the drill core orientation of the bottom surface of the drill core, and during drill core logging this mark is continued along the entirety of the drill core, where possible in a straight line. Logging procedure includes using core orientation to determine the azimuth and dip of each structure encountered (e.g., veins, contacts, faults). Digital photographs are taken of the drill core (wet and dry) and drill core recovery, RQD, basic geotechnical information, geological and structural elements are recorded in the drill core logs. Sample intervals are marked and samples for bulk density determination are also selected. Drill core recovery is generally good; however, when poor, the samples are shorter and there are small gaps in the sampled drill core to show where it was lost.

All data are entered using Geotic software and logging is regularly supervised with sign-off on all steps by a supervisor. When logging is complete, the data are audited in a spreadsheet available to all personnel involved before being imported into a master file with limited access to select authorized personnel only. Nominal drill core sample intervals are 1.0m, which are adjusted to respect lithological contacts or abrupt changes in mineralization, with smaller intervals of 0.5m where needed.

Drill core samples are cut in half lengthwise using a diamond-blade saw. 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 the remaining half-drill core is carefully returned to its original position in the drill core boxes. Field duplicates are made by halving the already halved drill core again and both ¼-drill cores are sent as duplicates to the lab, leaving the remaining ½-drill core archived in the drill core box. 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.

11.1.2 Bulk Density Determinations

Bulk density determination is performed onsite by Aya geologists, with the water immersion method selected to determine the bulk density of rocks at Boumadine, as there is none to very limited porosity in Boumadine drill core, this method is judged adequate by the Authors. Bulk density determinations are completed in a dedicated area, where the equipment is protected from disturbances, such as vibration or drafts, which might influence balance readings.

Aya's protocol requires the determination of wet (moisture percent) and dry bulk densities of mineralized and barren samples. Full drill core pieces of ~10 to 15cm are used for the determinations. When this process is complete, the drill core is cut and one-half 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.

The equipment is calibrated on a daily basis with 0.5 and 1.0 kg reference materials used for wet and dry tests, and the balance is calibrated weekly with dry certified weights. The set-up is rudimentary, although acceptable and the equipment has been upgraded in 2024 .

As discussed in Section 14.4 of this Report, the average bulk density for the constrained sulphide material is 3.72 t/m³. For the current Mineral Resource Estimate a bulk density of 2.61 t/m³ was assigned to oxide and transitional blocks. For sulphide blocks, the median sulphide bulk density was assigned for each modelled domain.

11.1.3 Sample Preparation and Analysis

Samples were prepared by AfriLab at its Boumadine prep-laboratory facility or at its Zgounder prep-lab. A total of 250 g of pulverized sample material was subsequently submitted for analysis to Afrilab in Marrakech. When received at the analytical lab, drill core samples are 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 barren waste rock.

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

The sub-samples are pulverized using a ROCKLABS pulverizer. Pulverizing performance is targeted to a size of 85% of the sample at <75 µm. One sample in twenty is selected randomly to verify this performance, by wet sieve test (standard 75 µm sieve).

Ag, Zn, Pb, Cu, Fe, Sn, As and Mo are analyzed by Inductively Coupled Plasma (“ICP”) spectrometry after 4-acid digestion. Gold is analyzed by fire assay method with AAS finish. Silver grades of >200 g/t Ag are further analyzed by fire assay method with gravimetric finish.

11.1.4 Security – Chain of Custody

Drill core remains under Aya’s control from the drill site, where Company geologists supervise operations, to the on-site drill core logging facility, where drill core boxes are transported at the end of each shift for logging, cutting and sampling. When logging and sampling are completed, the plastic drill core trays are stored outside, on-site and cross-piled within a gated compound that is guarded by a security guard around the clock. Sample chain of custody is simplified by the presence of the on-site AfriLab preparation laboratory. Prepared samples are then shipped to the AfriLab facility in Marrakech and tracked through AfriLab’s internal management system.

11.2 QUALITY ASSURANCE/QUALITY CONTROL REVIEW

Aya implemented and monitored a thorough quality assurance / quality control (“QA/QC” or “QC”) program for the drilling undertaken at the Boumadine Deposit during the 2018 to 2025 period. In addition to the internal QC protocol implemented at the laboratories, QC protocol at Boumadine included the sequential insertion of certified reference materials (“CRMs”), blanks and field duplicates into every batch of drill core samples sent for analysis (each batch contains 25 samples). Samples prepared at the drill core logging facility are numbered sequentially, such that drill core samples and QC samples are not able to be differentiated by the laboratory.

QC sample insertion rates are as follows:

- A range of CRMs over varying grades are inserted at a rate of 1 in 25 samples;
- Blank samples are inserted at a rate of 1 in 25 samples to monitor for instrumentation carry-over and contamination at the laboratory;
- Field duplicate samples were also inserted into the drill core sample stream, but not as systematically as the CRMs and blanks, at a rate of ~1 in every 50 samples from 2022 to 2025. Prior to this, four field duplicates only were inserted into three drill holes during 2018 and none were inserted during the 2019 drill program;
- At the end of each month, a selection of 5% of the coarse reject samples is submitted to Afrilab; and
- Check analyses at an umpire laboratory (ALS in Seville, Spain) are carried out on one in every 50 samples, representing ~2% of the global primary laboratory sample flow.

The QA/QC procedures from 2018 to 2024 were previously evaluated by P&E and documented in the May 2024 Mineral Resource Estimate (MRE) report. The author determined that the sample preparation, security, and analytical procedures for the Boumadine Deposit were satisfactory, and that the data were of high quality, suitable for inclusion in the current MRE.

The current author has reviewed P&E's findings and agrees with their conclusions. For further reference, performance graphs from the previous MRE report are provided in Appendix H.

New data from the 2024-2025 period, included in this MRE update, have been reviewed by the current author. The performance evaluations are detailed in the subsequent sections.

11.2.1 2024 to 2025 Diamond Drill Hole Programs

11.2.1.1 Performance of Certified Reference Materials

A total of 21 different OREAS certified reference materials ("CRMs") were used during the 2024 to 2025 drilling at the Boumadine Deposit, to monitor accuracy at the lab for gold, silver, lead, zinc and copper. A summary of CRMs inserted into the drill sample stream and analyzed at AfriLab is outlined in Figure 11-1. All 13 CRMs were purchased from ORE Research & Exploration Pty Ltd ("ORE") in Australia and the corresponding certified mean value for each individual CRM is indicated in Table 11-1.

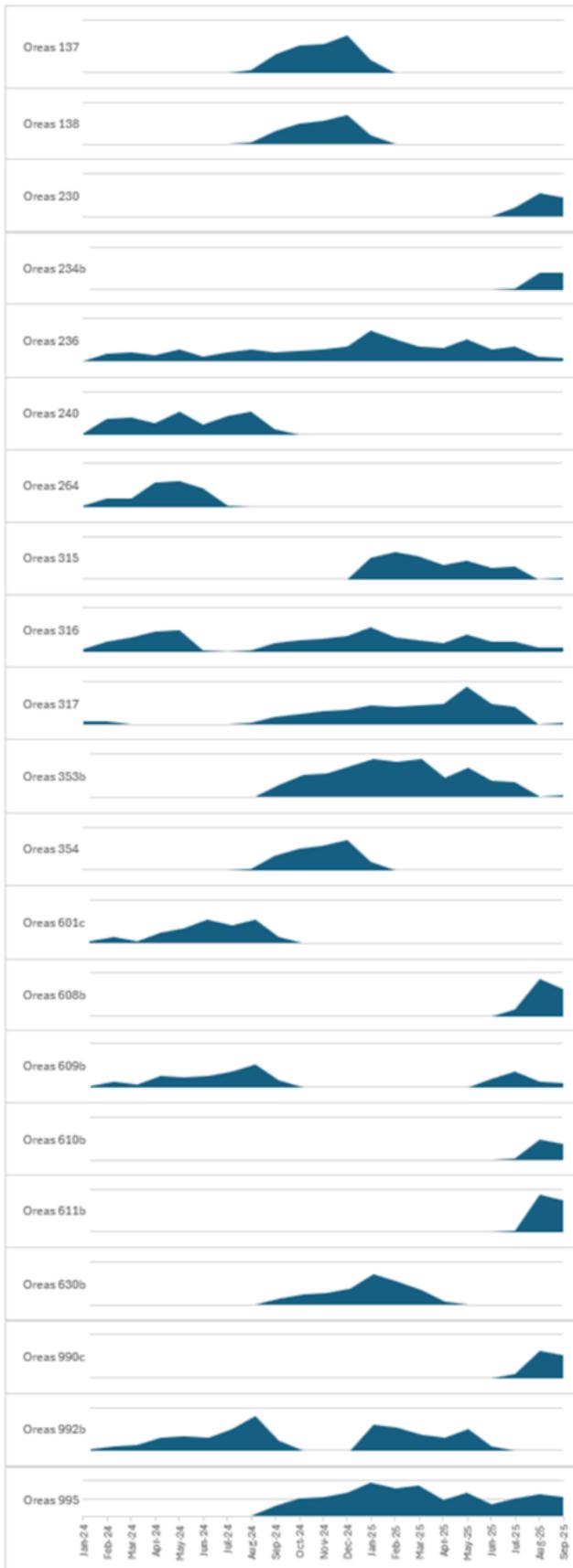


Figure 11-1 CRM usage at Boumadine 2024 TO 2025

Source: This Report

Table 11-1 Summary of Reference Materials used at Boumadine in 2024 to 30 September 2025

Reference Material	Certified Mean Value				
	Au	Ag	Pb	Zn	Cu
	(g/t)	(g/t)	(%)	(%)	(%)
OREAS137	--	25.9	0.673	4.92	0.0246
OREAS138	--	45.2	1.23	8.19	0.0266
OREAS230	0.34	0.13	0.00085	0.0098	0.0172
OREAS234b	1.23	0.35	0.00262	0.0143	0.0162
OREAS236	1.85	0.48	0.003	0.014	0.017
OREAS240	5.51	1.3	0.003	0.014	0.017
OREAS264	0.31	1.29	0.001	0.022	0.009
OREAS315	--	72.5	3.79	5.45	0.0785
OREAS316	--	103	5.02	11.16	0.161
OREAS317	--	232	12.13	17.45	0.413
OREAS353b	--	2,184.00	59.18	3.83	0.431
OREAS354	--	98	1.58	49.77	0.1387
OREAS601c	1	50.3	0.033	0.043	0.116
OREAS608b	1.29	15.2	0.0377	0.0651	--
OREAS609b	4.97	24.6	0.045	0.131	0.498
OREAS610b	8.54	46.9	0.0758	0.2187	0.92
OREAS611b	14.38	76.1	0.097	0.2803	--
OREAS630b	0.36	19	0.411	1.11	0.052
OREAS992b	15	340	0.374	0.862	44.73
OREAS995	4.52	37.3	0.2697	1.29	22.6

Source: This Report

CRMs were inserted into the analysis stream approximately every 25 samples. Criteria for assessing CRM performance are based as follows: data falling outside ± 3 standard deviations from the accepted mean value, or two consecutive data points falling between ± 2 and ± 3 standard deviations on the same side of the mean, fail. The CRM results are presented in Figure 11-2 to Figure 11-11. The Author of this Technical Report section considers that the CRMs demonstrate acceptable accuracy in the Boumadine 2024 to 2025 diamond drilling data and the relatively few failures indicate no material issues with accuracy.

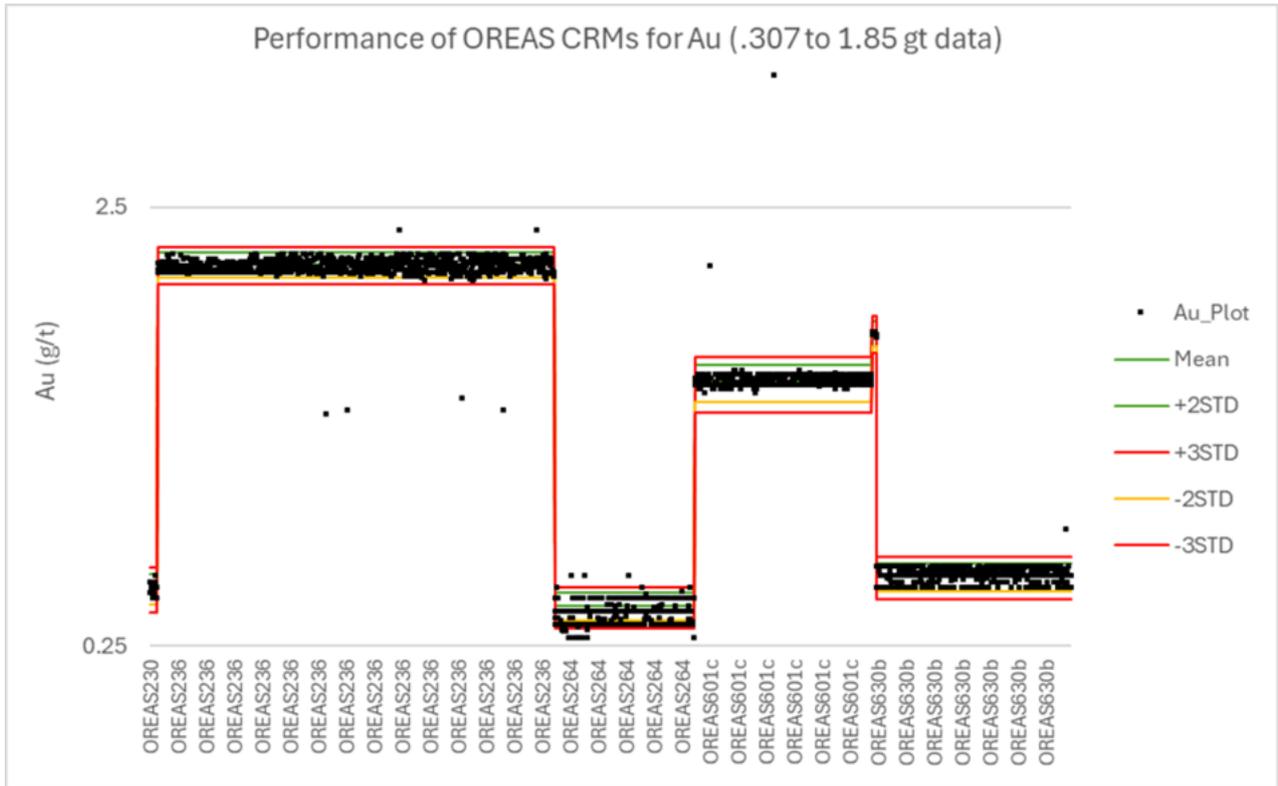


Figure 11-2 Performance of OREAS CRMs for Au (0.307 to 1.85 g/t data)

Source: This Report

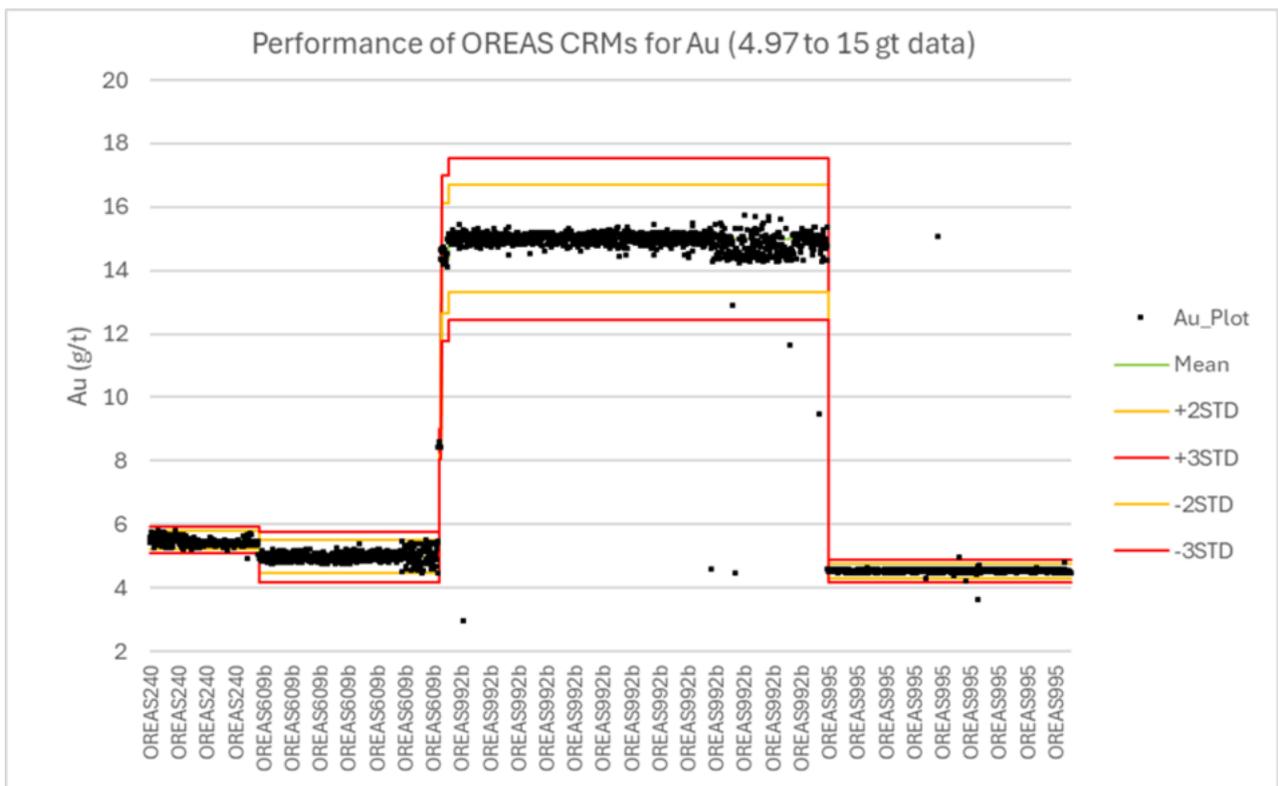


Figure 11-3 Performance of OREAS CRMs for Au (4.97 to 15 g/t data)

Source: This Report

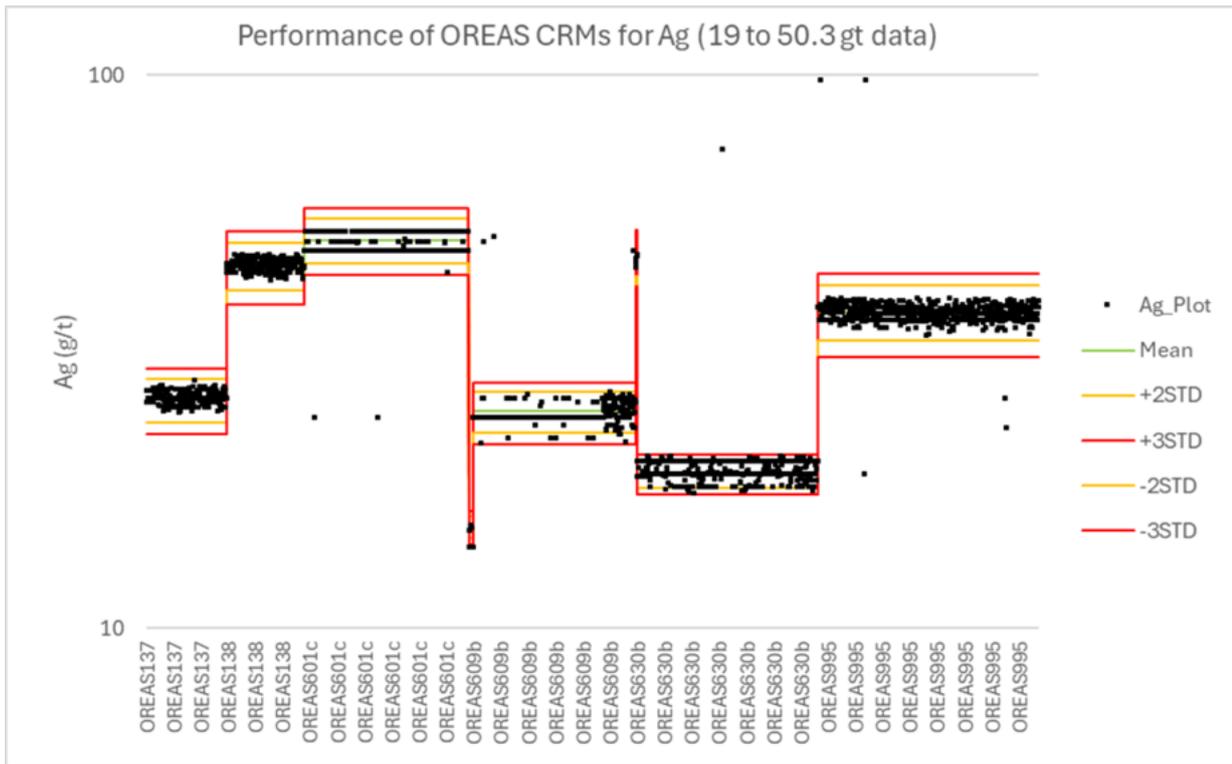


Figure 11-4 Performance of OREAS CRMs for Ag (19 to 25.9 g/t data)

Source: This Report

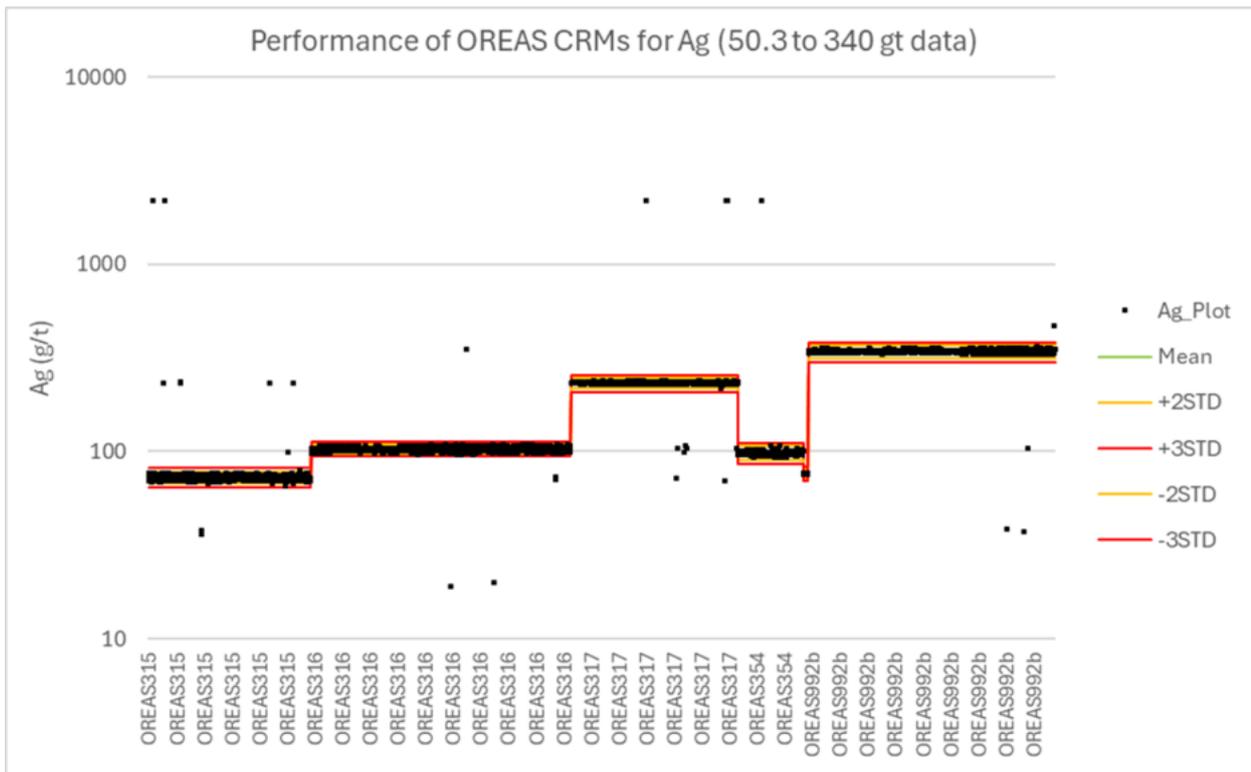


Figure 11-5 Performance of OREAS CRMs for Ag (50.3 to 340 g/t data)

Source: This Report

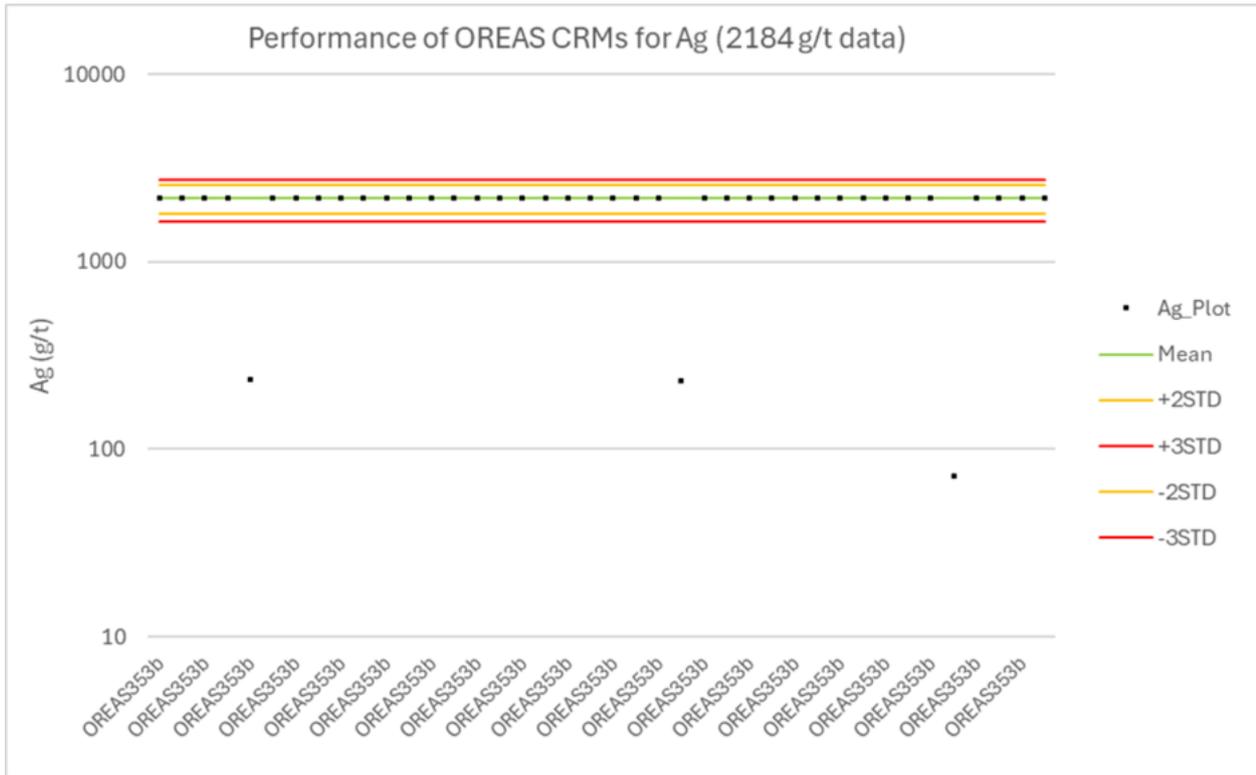


Figure 11-6 Performance of OREAS CRMs for Ag (2184 g/t data)

Source: This Report

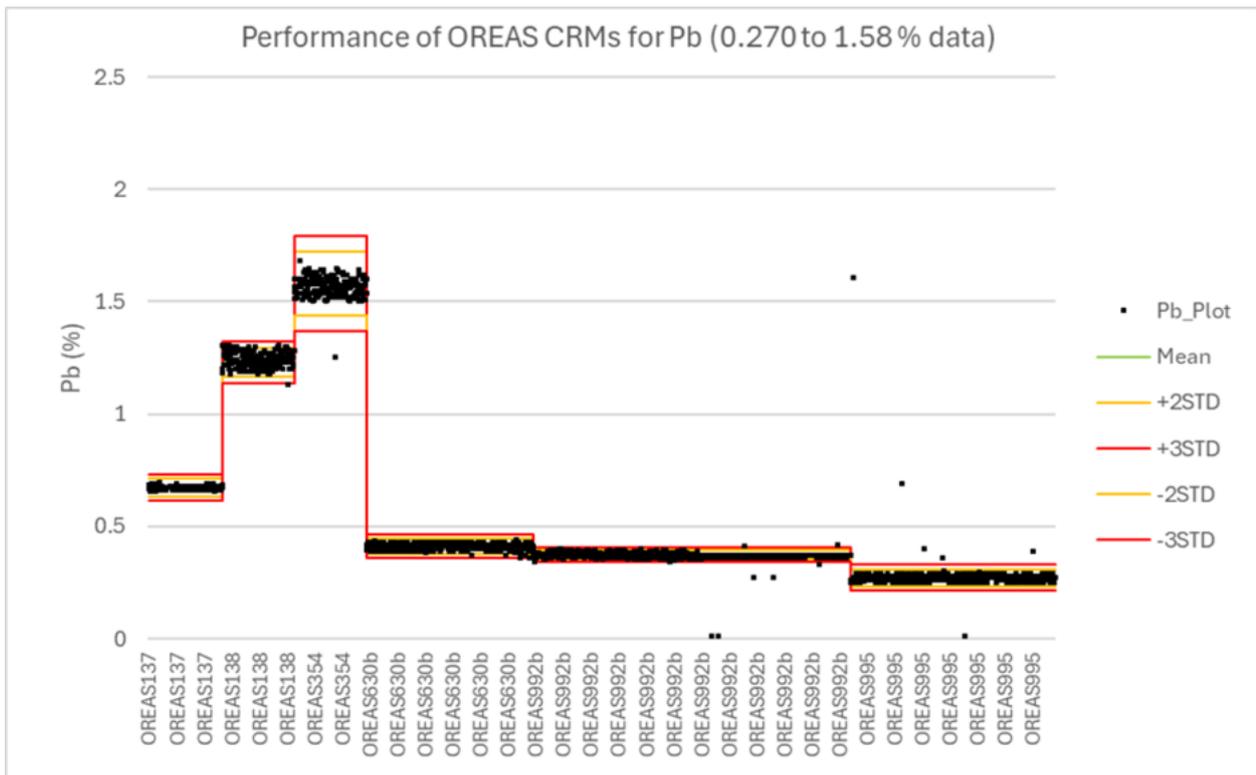


Figure 11-7 Performance of OREAS CRMs for Pb (0.270 to 1.58 % data)

Source: This Report

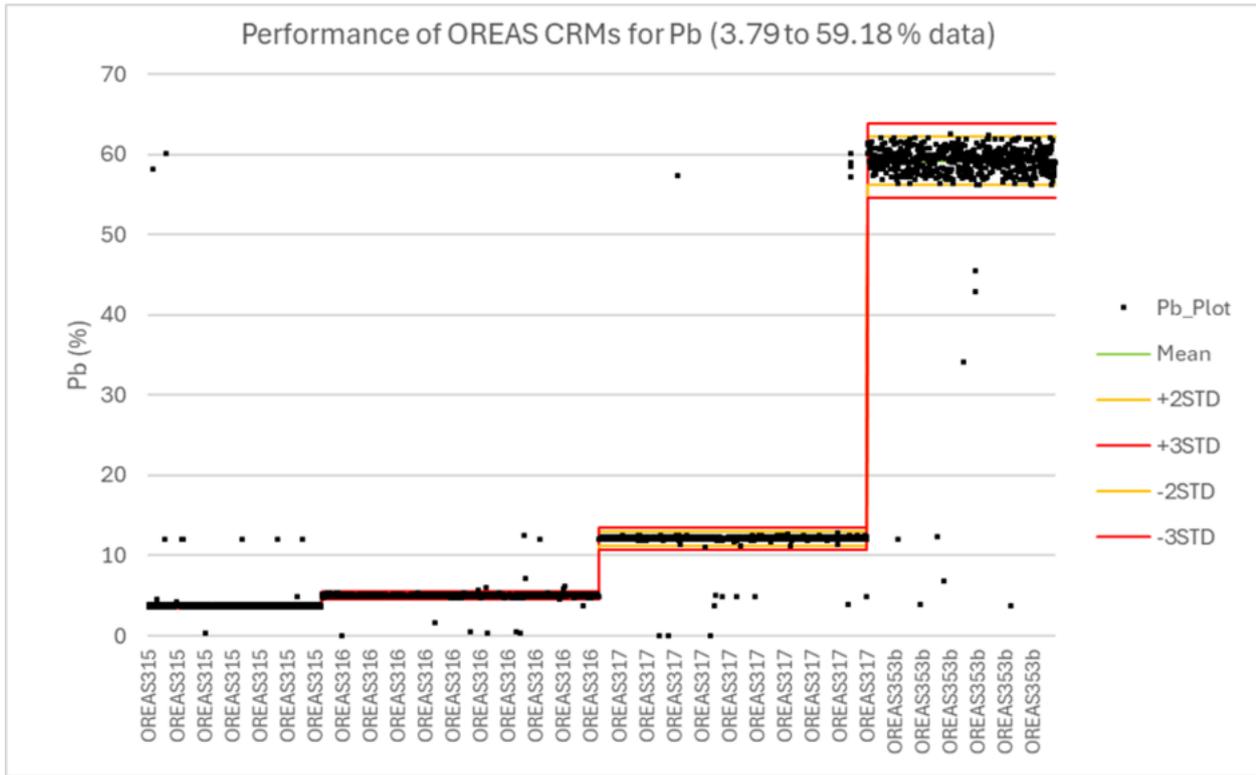


Figure 11-8 Performance of OREAS CRMs for Pb (3.79 to 59.18% data)

Source: This Report

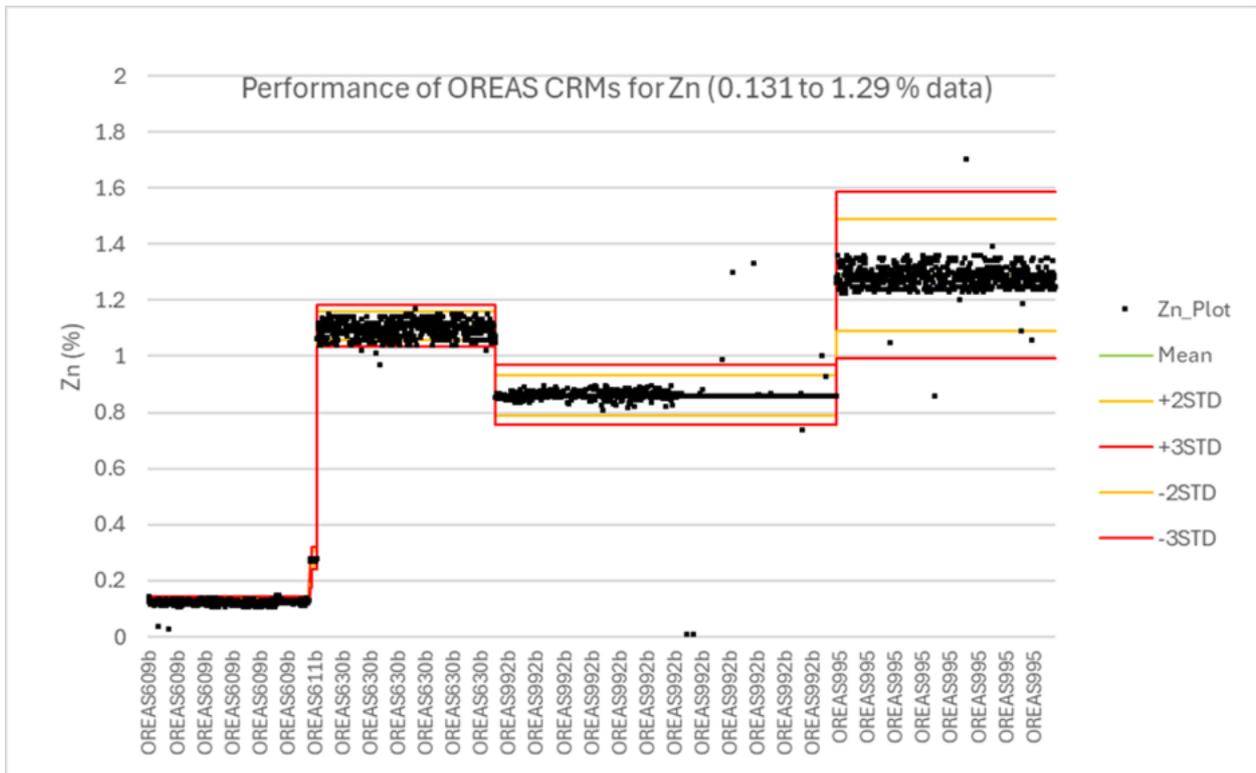


Figure 11-9 Performance of OREAS CRMs for Zn (0.131 to 1.29% data)

Source: This Report

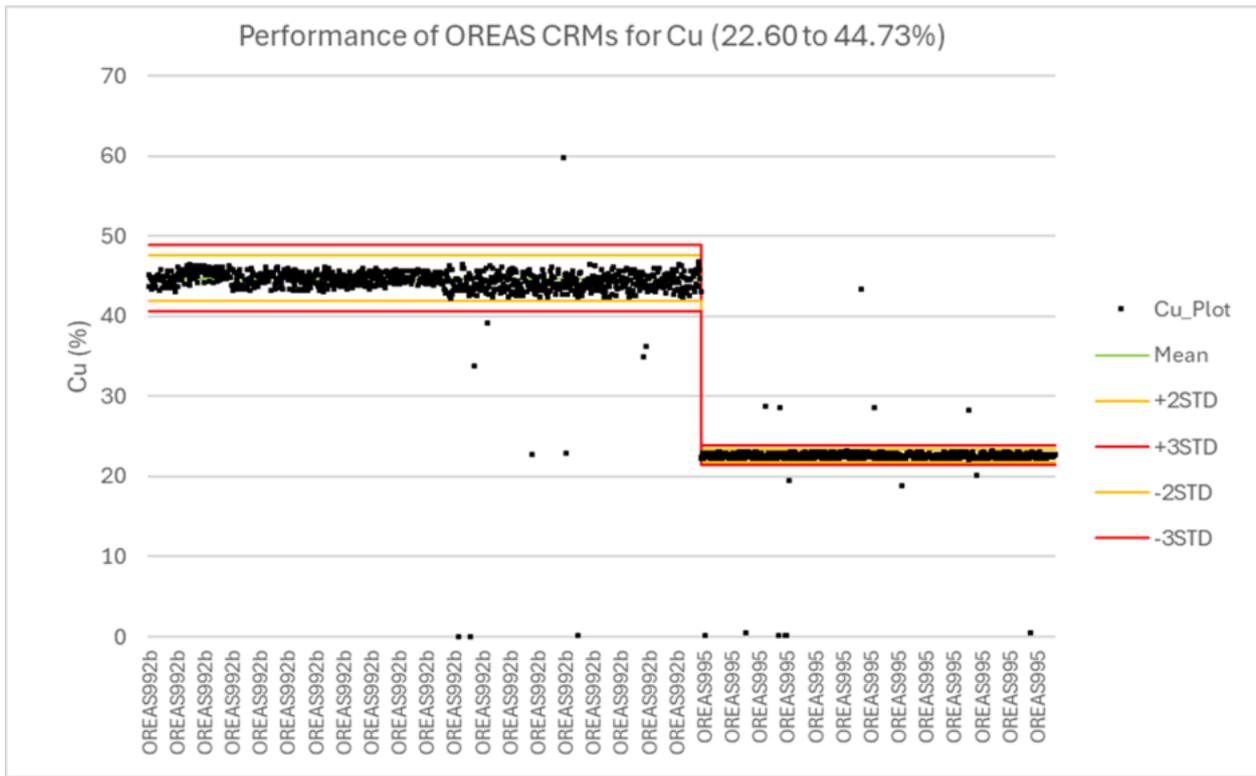


Figure 11-12 Performance of OREAS CRMs for Cu (22.6 to 44.73%)

Source: This Report

11.2.1.2 Performance of Blanks

The blank material used by Aya is sourced and prepared from local cobbles of arenite and a variety of barren sandstone. The blank material is safely stored away from any source of contamination in plastic drums on-site. Blanks are inserted into the analysis stream approximately every 25 samples. All blank data for gold, silver, lead, zinc, and copper are graphed (Figure 11-13 to Figure 11-17). If the assayed value in the certificate was indicated as being less than detection limit, the value was assigned half the value of the detection limit for data treatment purposes (e.g., <1 = 0.5). An upper tolerance limit of +3 standard deviations from the calculated mean was set. There were 4,843 data points to examine.

The vast majority of data plotted below the set tolerance limit, with very few data points plotting above the set tolerance limit. The Author of this Report section does not consider contamination to be an issue with the 2024 to 2025 Boumadine Deposit data.

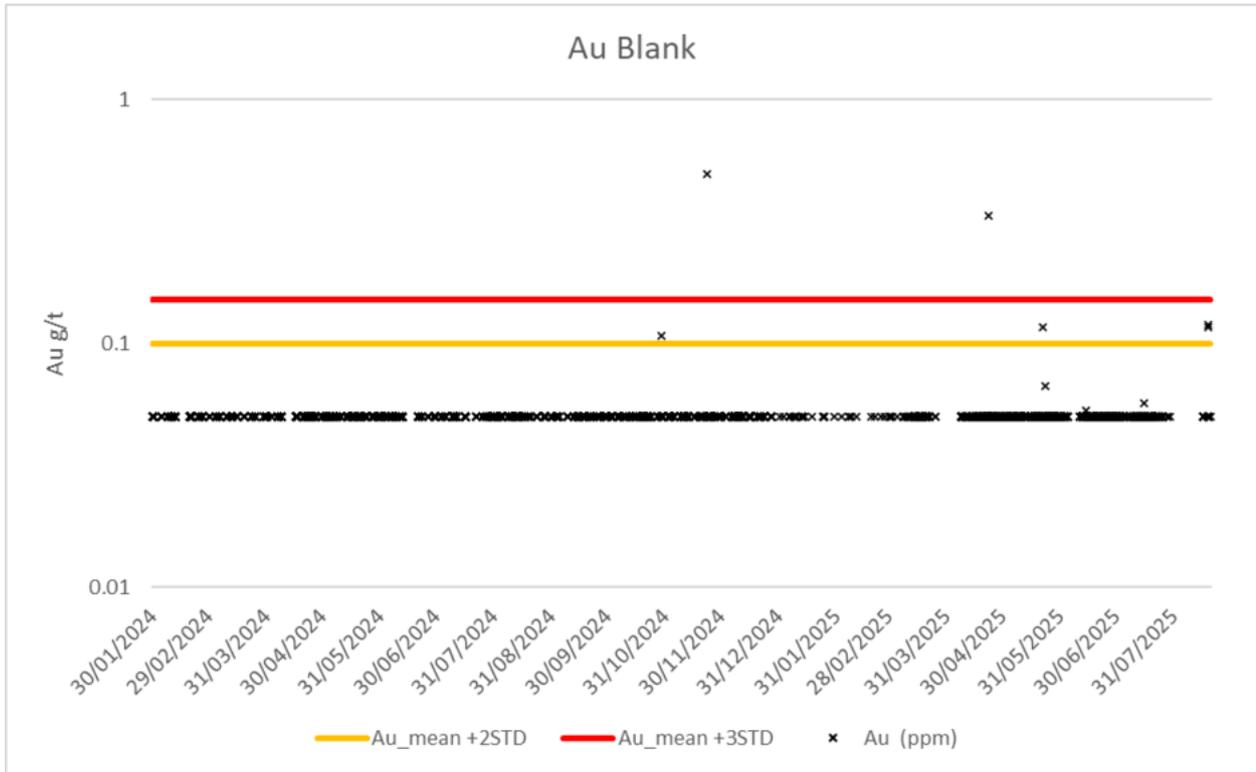


Figure 11-13 Performance of Blanks for Au

Source: This Report

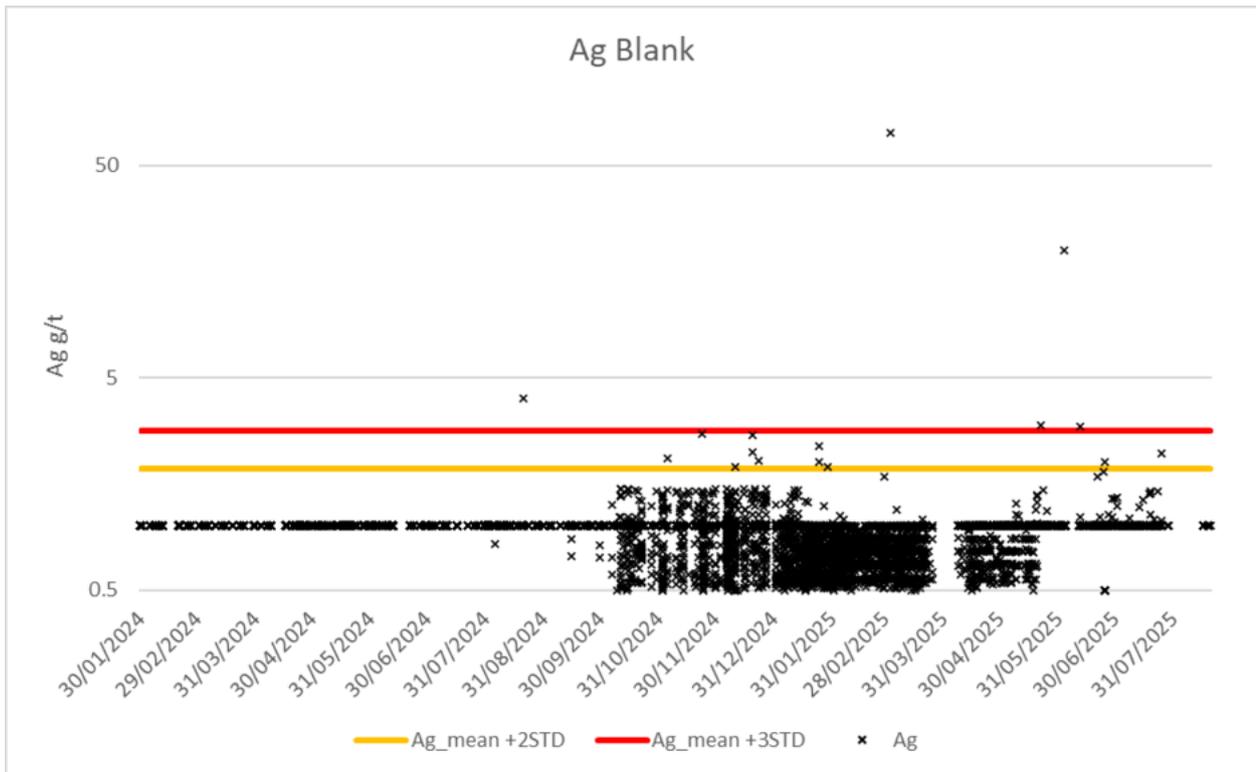


Figure 11-14 Performance of Blanks for Ag

Source: This Report

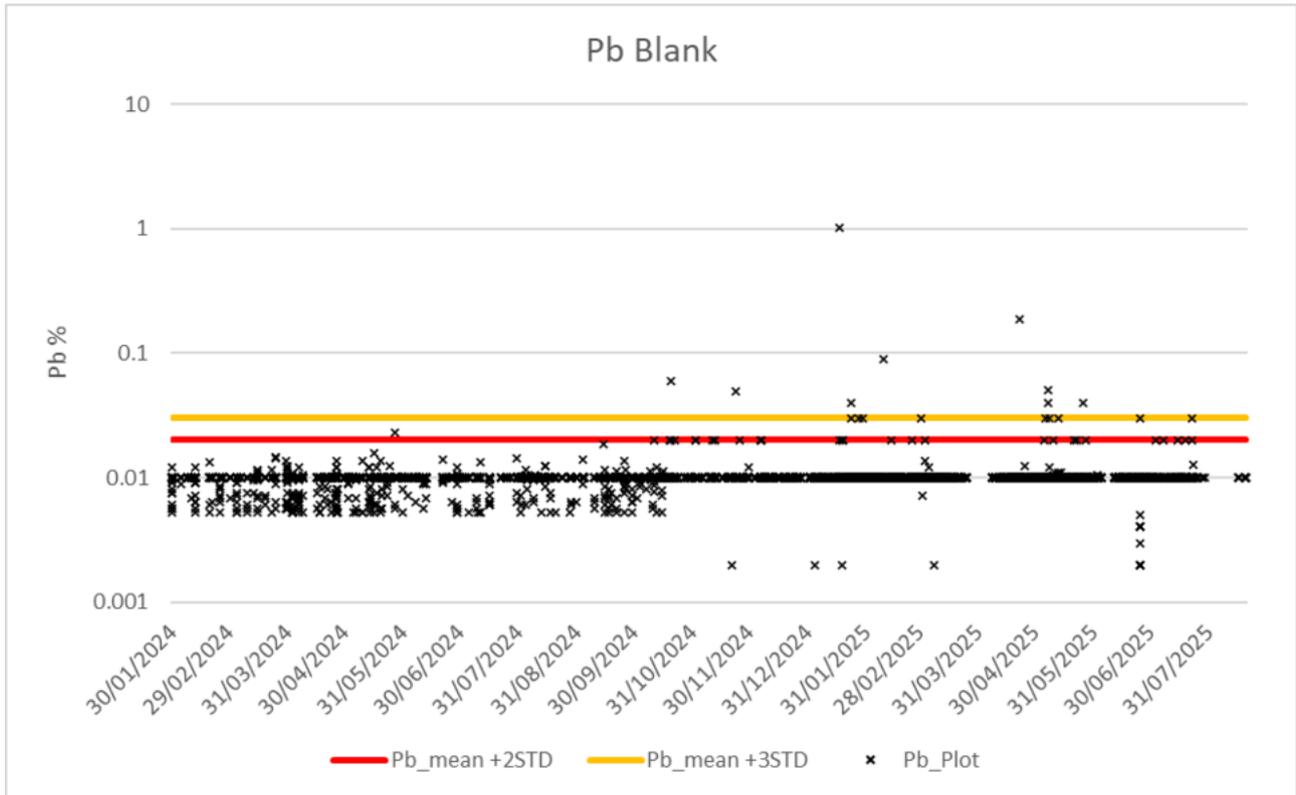


Figure 11-15 Performance of Blanks for Pb

Source: This Report

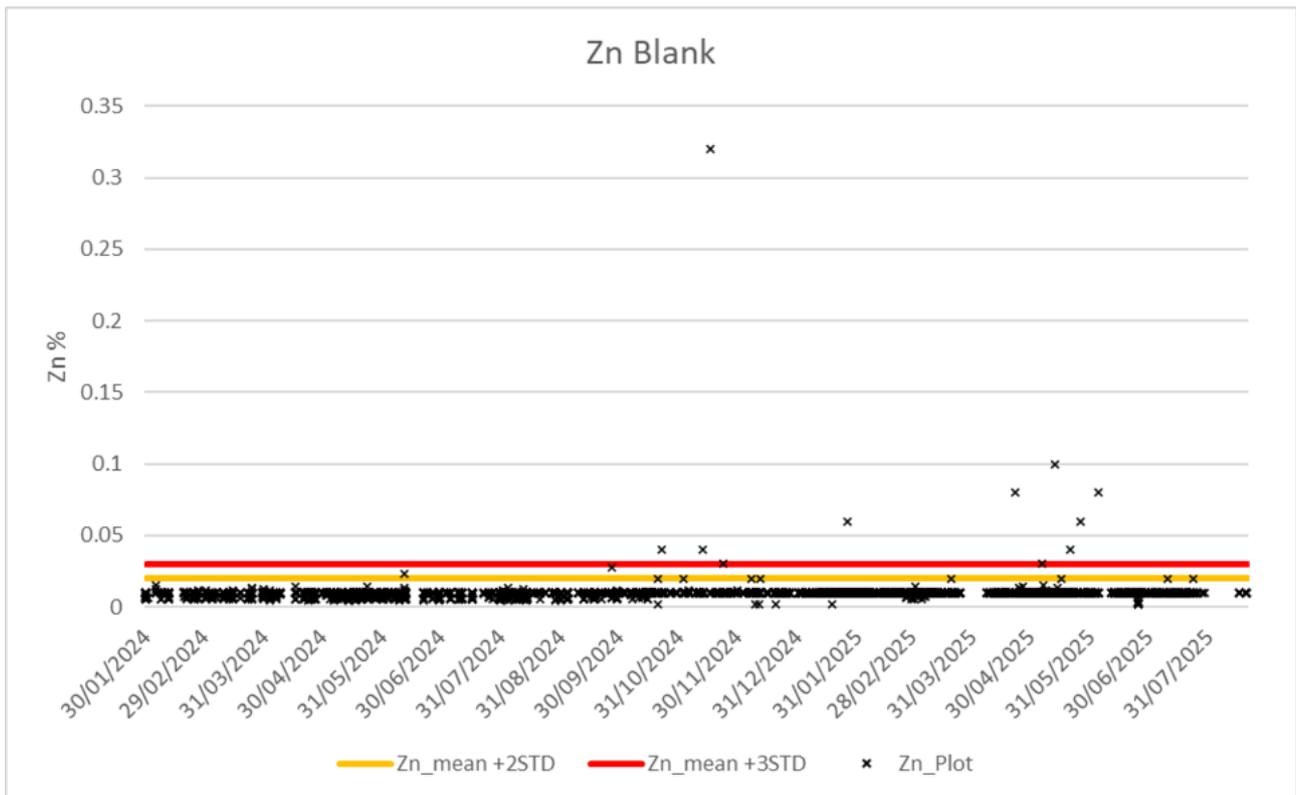


Figure 11-16 Performance of Blanks for Zn

Source: This Report

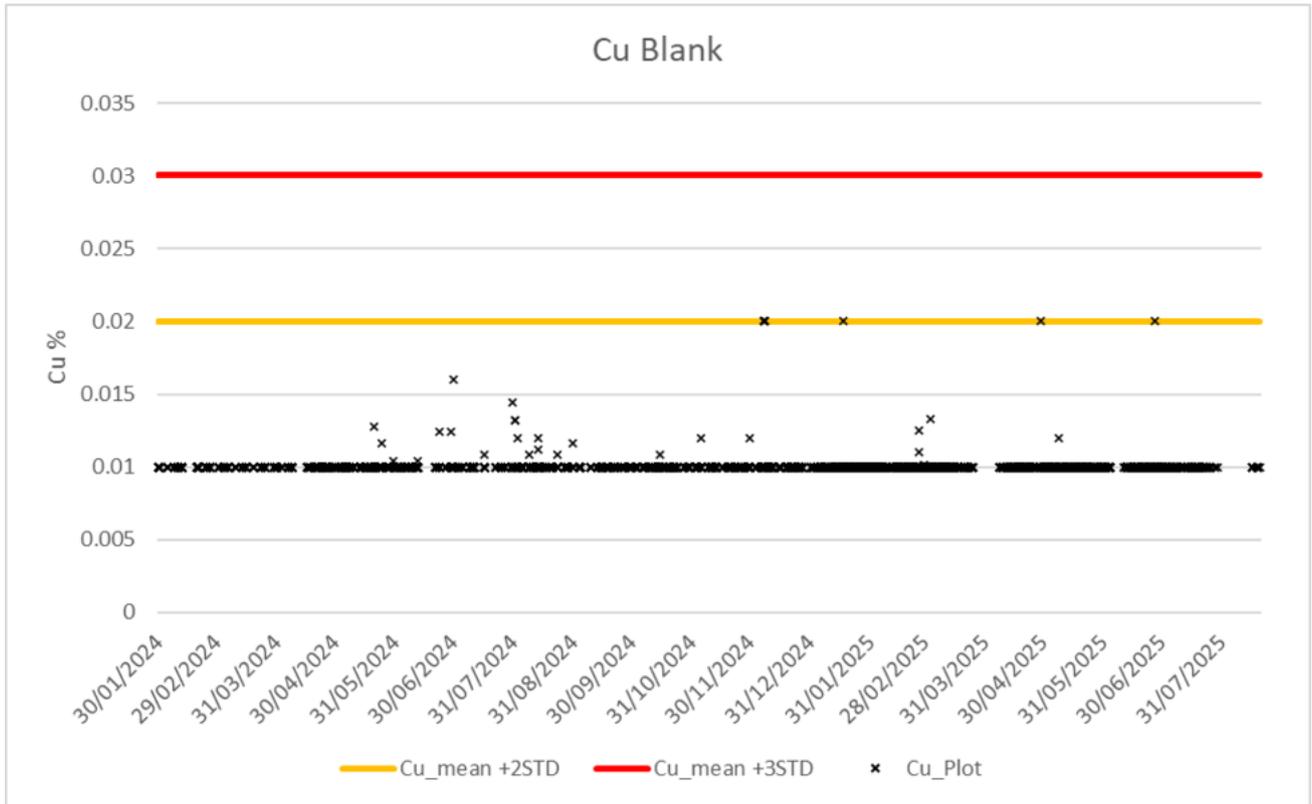


Figure 11-17 Performance of Blanks for Cu

Source: This Report

11.2.1.3 Performance of Duplicates

Field duplicate samples were inserted into the drill core sample stream, at a rate of ~1 in every 25 samples from 2022 to 2025. Prior to this, four field duplicates only were inserted into three drill holes during 2018 and none were inserted during the 2019 drill program. At the end of each month, a selection of 5% of the coarse reject samples is also submitted to Afrilab for duplicate analysis and the primary lab also assays pulp samples for duplicate analysis.

Core duplicate data for gold, silver, copper, lead and zinc were examined by the current Author for the 2024 to 2025 diamond drilling at the Boumadine Deposit. The data were graphed (Figure 11-18 and Figure 11-22) and found to have acceptable precision.

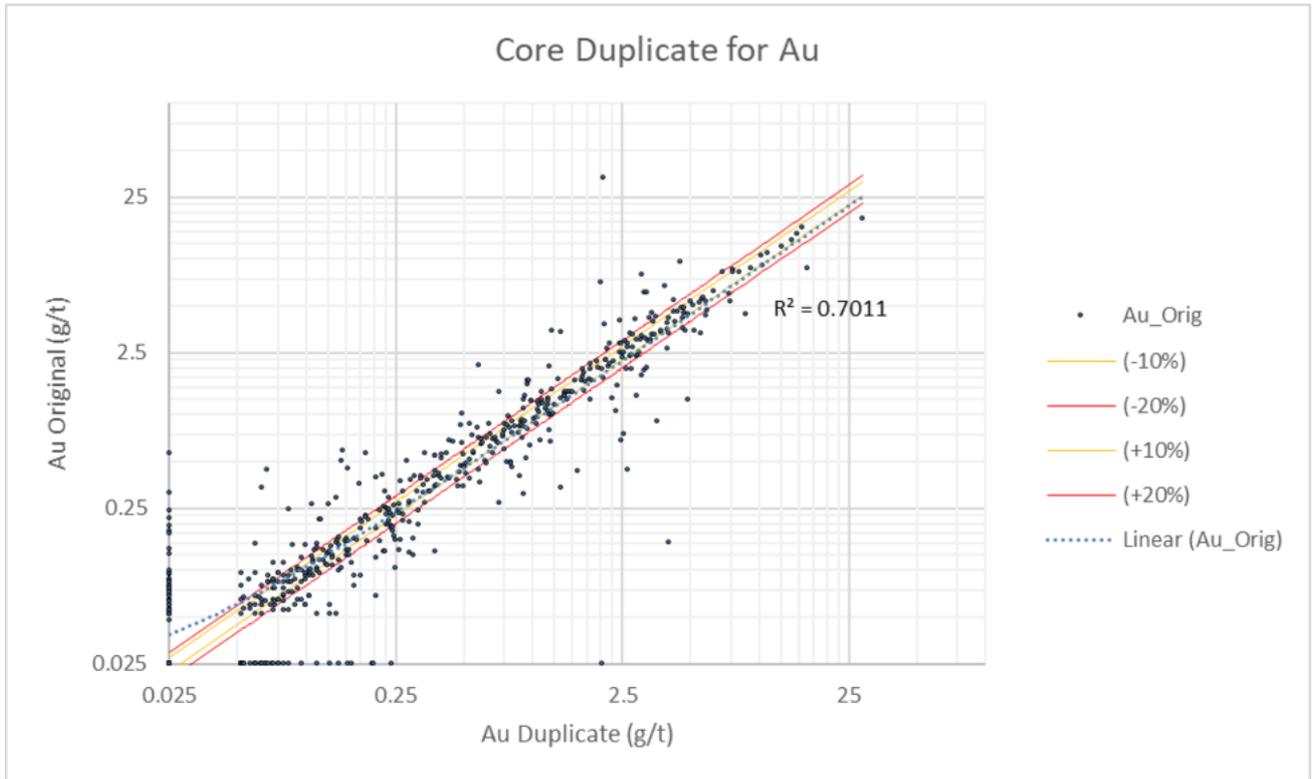


Figure 11-18 Performance of Field Duplicates for Au

Source: This Report

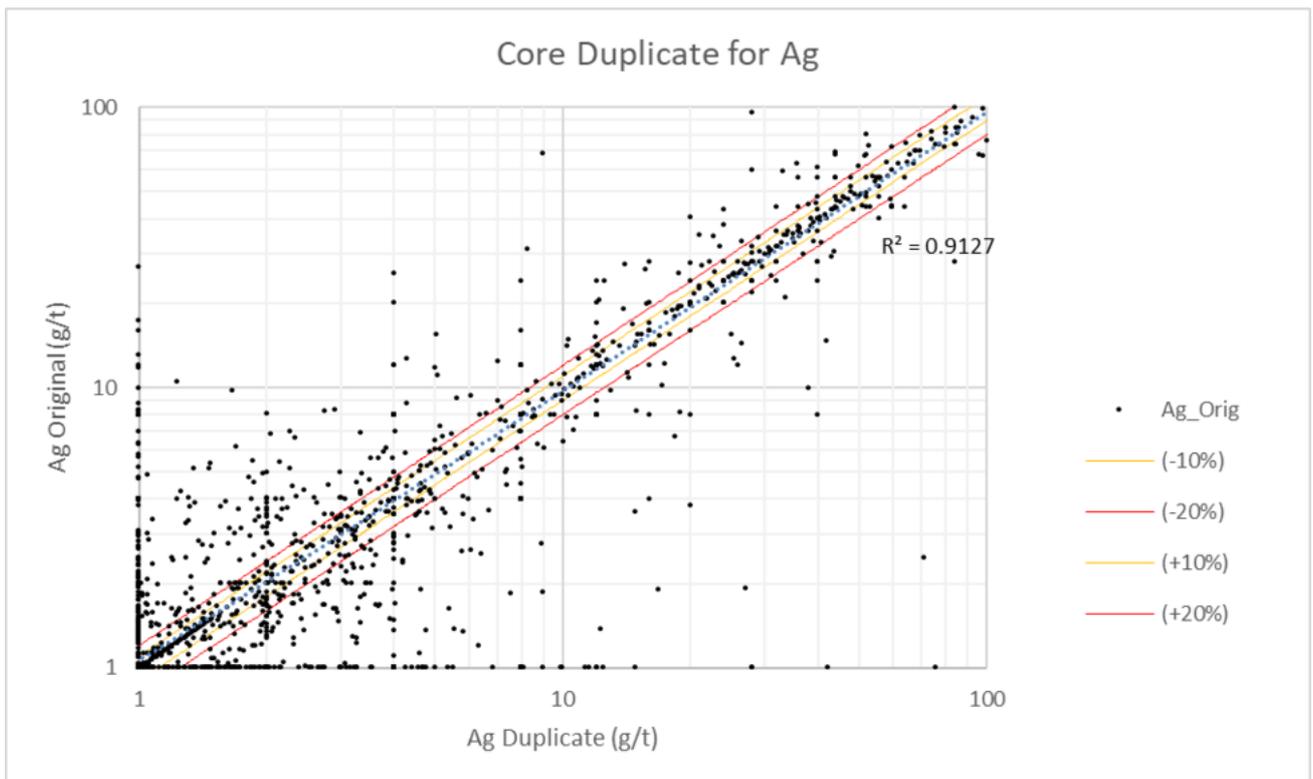


Figure 11-19 Performance of Field Duplicates for Ag

Source: This Report

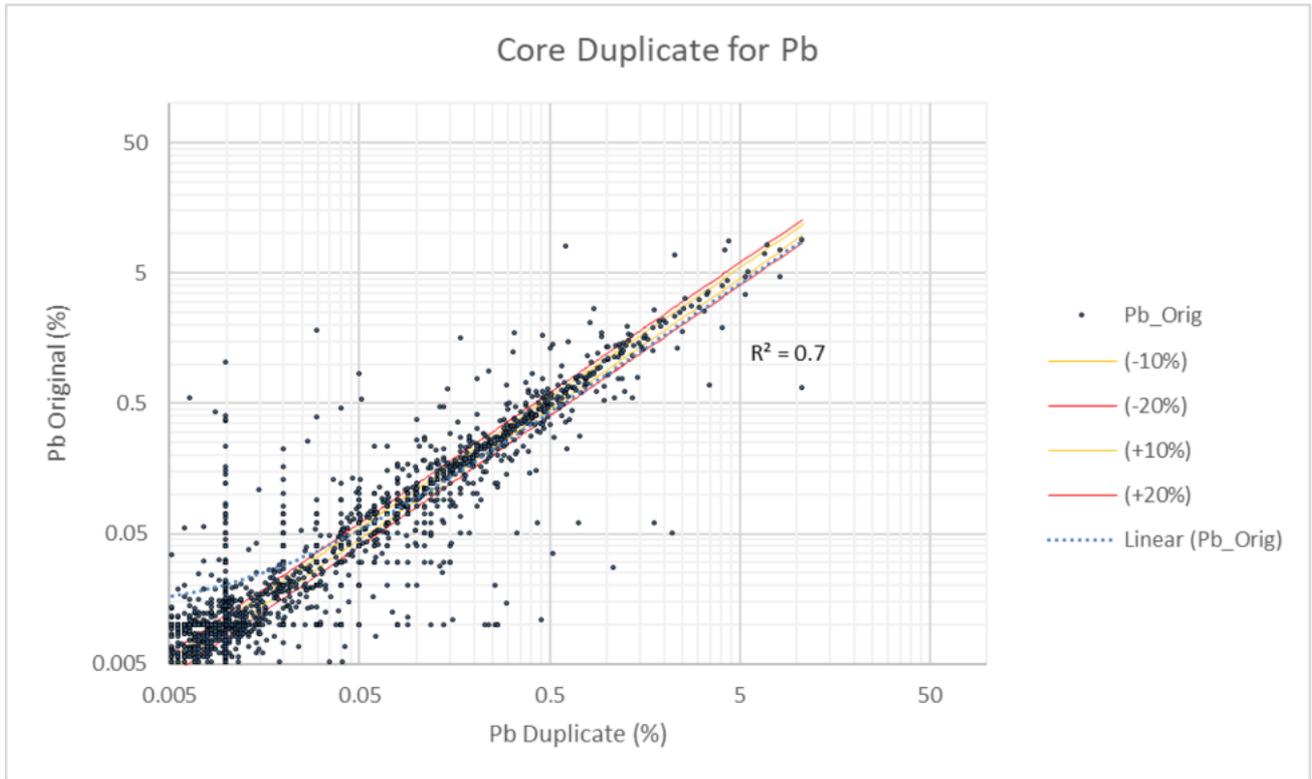


Figure 11-20 Performance of Field Duplicates for Pb

Source: This Report

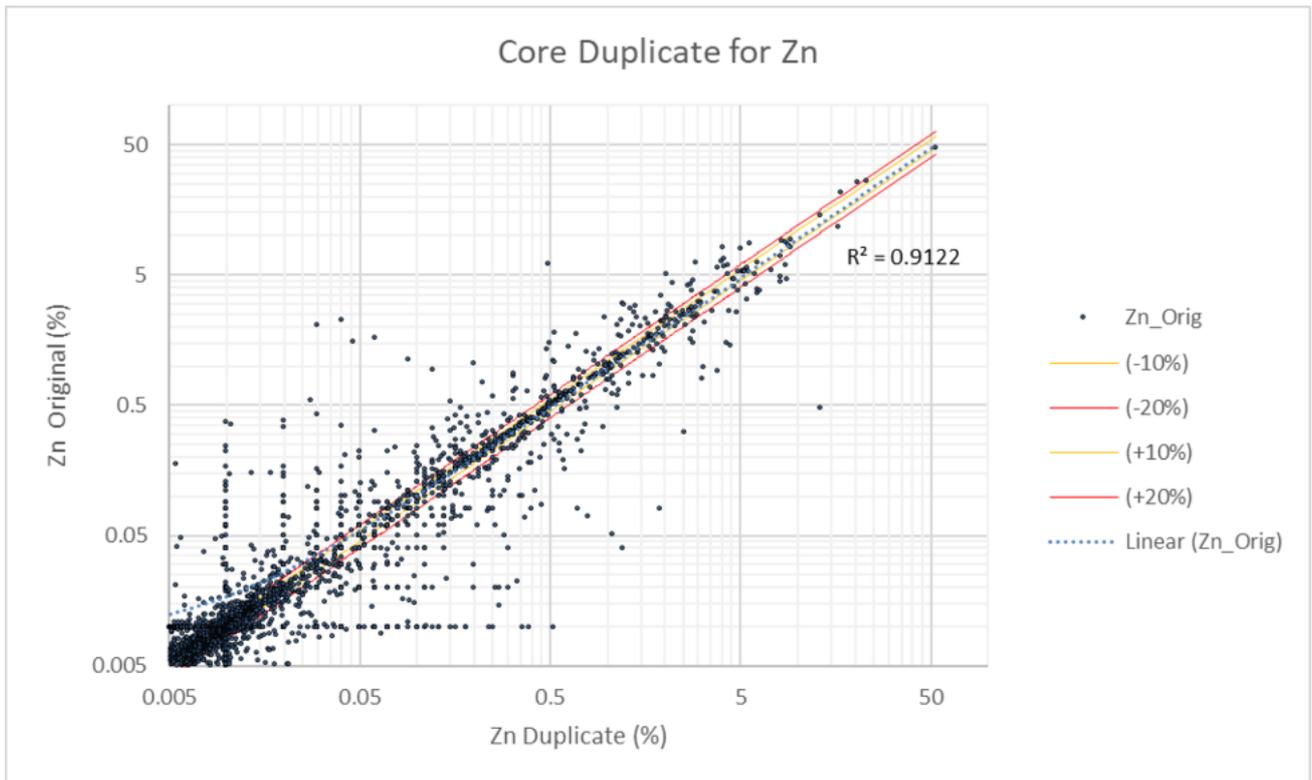


Figure 11-21 Performance of Field Duplicates for Zn

Source: This Report

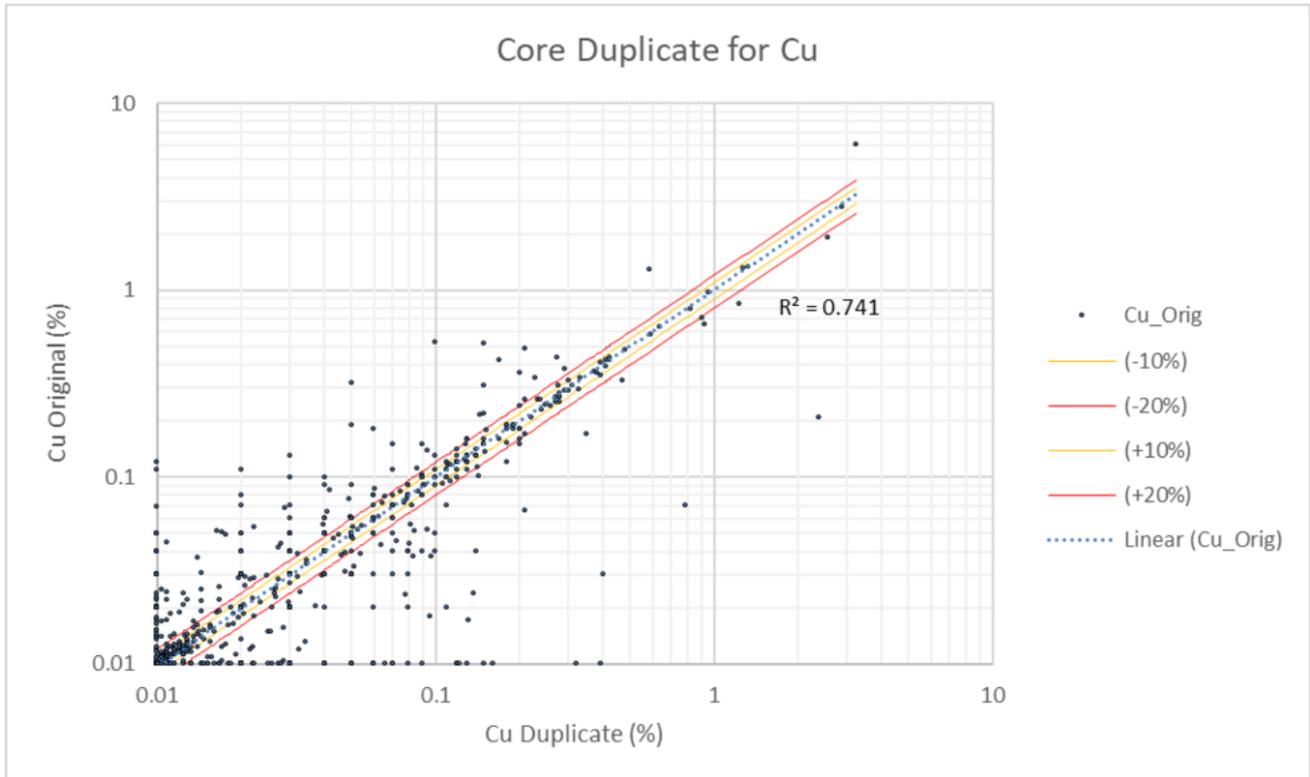


Figure 11-22 Performance of Field Duplicates for Cu

Source: This Report

11.2.1.4 Umpire Sampling

Aya conducted umpire sampling to verify the integrity of the analytical results produced by AfriLab, the primary laboratory, for geochemical testing at the Boumadine Deposit. Select pulverized pulp samples were submitted for check assaying at a secondary laboratory (umpire lab) to validate the original analyses performed by AfriLab. These check analyses were conducted on one in every 50 samples, representing approximately 2% of the samples sent for analysis, and were completed at ALS in Seville, Spain.

For the 2024 to 2025 period, a new ALS laboratory opened in Morocco at Aya's Zgounder site. Umpire samples for Boumadine were sent to this new lab. However, as the facility was only operational by the end of 2024, none of these samples were available for the current study.

P&E, the Author of the 2024 MRE and Technical Report, reviewed the umpire assays for gold and silver for the Q3 and Q4 2022 drilling, comparing 422 samples. P&E noted some dispersion in the gold results below 2 ppm and around 8 ppm Au (Figure 11-22), along with a high bias in the reported umpire lab results and an R² value of 0.891. Increased dispersion below 80 ppm Ag was observed, as expected near the lower detection limit, while good correlation was seen above these lower grades (Figure 11-23). No significant bias was detected, with an R² value of 0.964.

The current author verified P&E's observations and work. The current author does not consider the biases exhibited in the gold data to have a material impact on the current Mineral Resource Estimate, as the primary laboratory may be under-reporting these results. Overall, the current author considers the data acceptable for use in the current Mineral Resource Estimate.

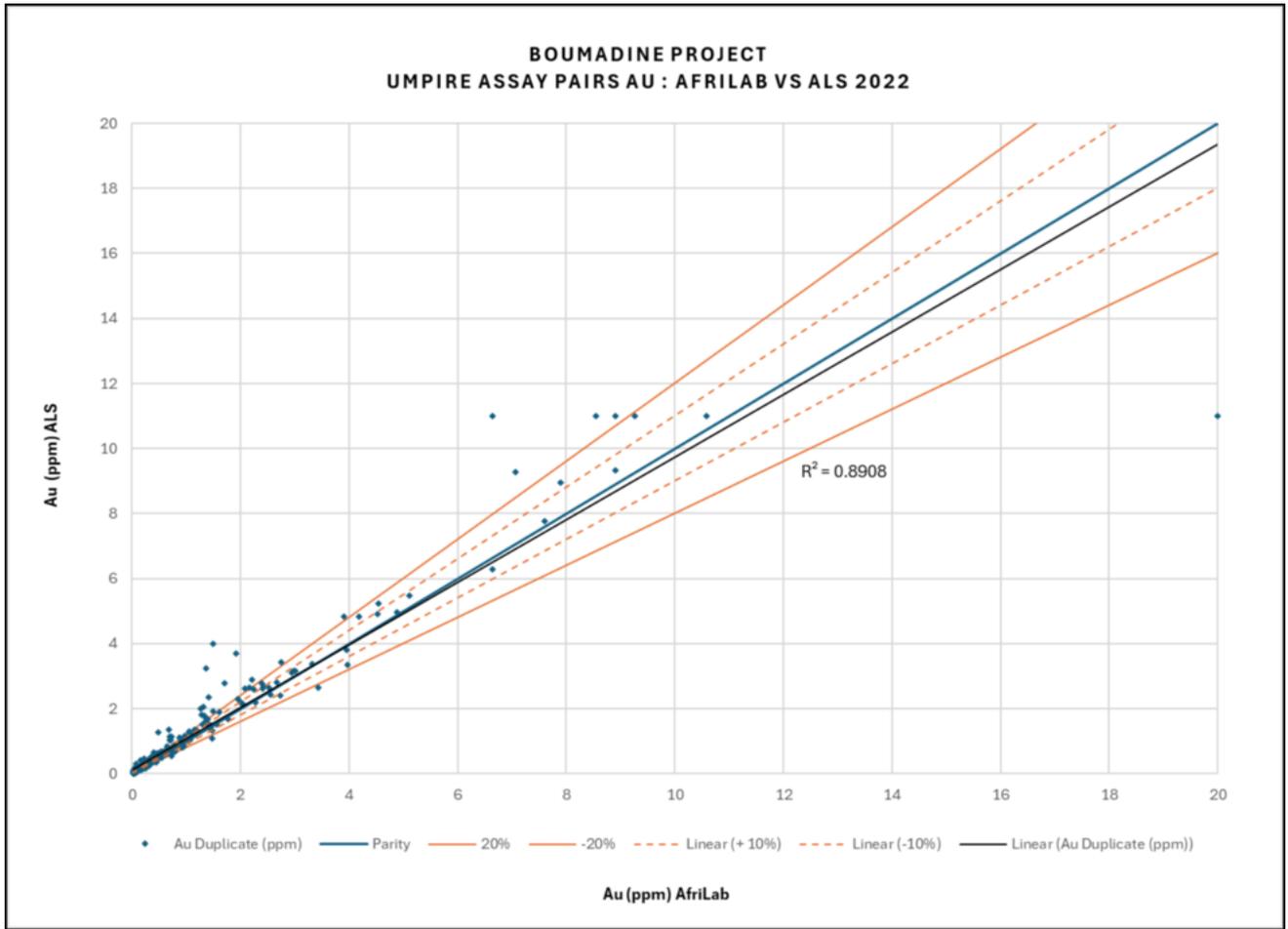


Figure 11-22 Umpire Assay Comparison for Au: Afrilab Versus ALS 2022

Source: P&E (May 2024)

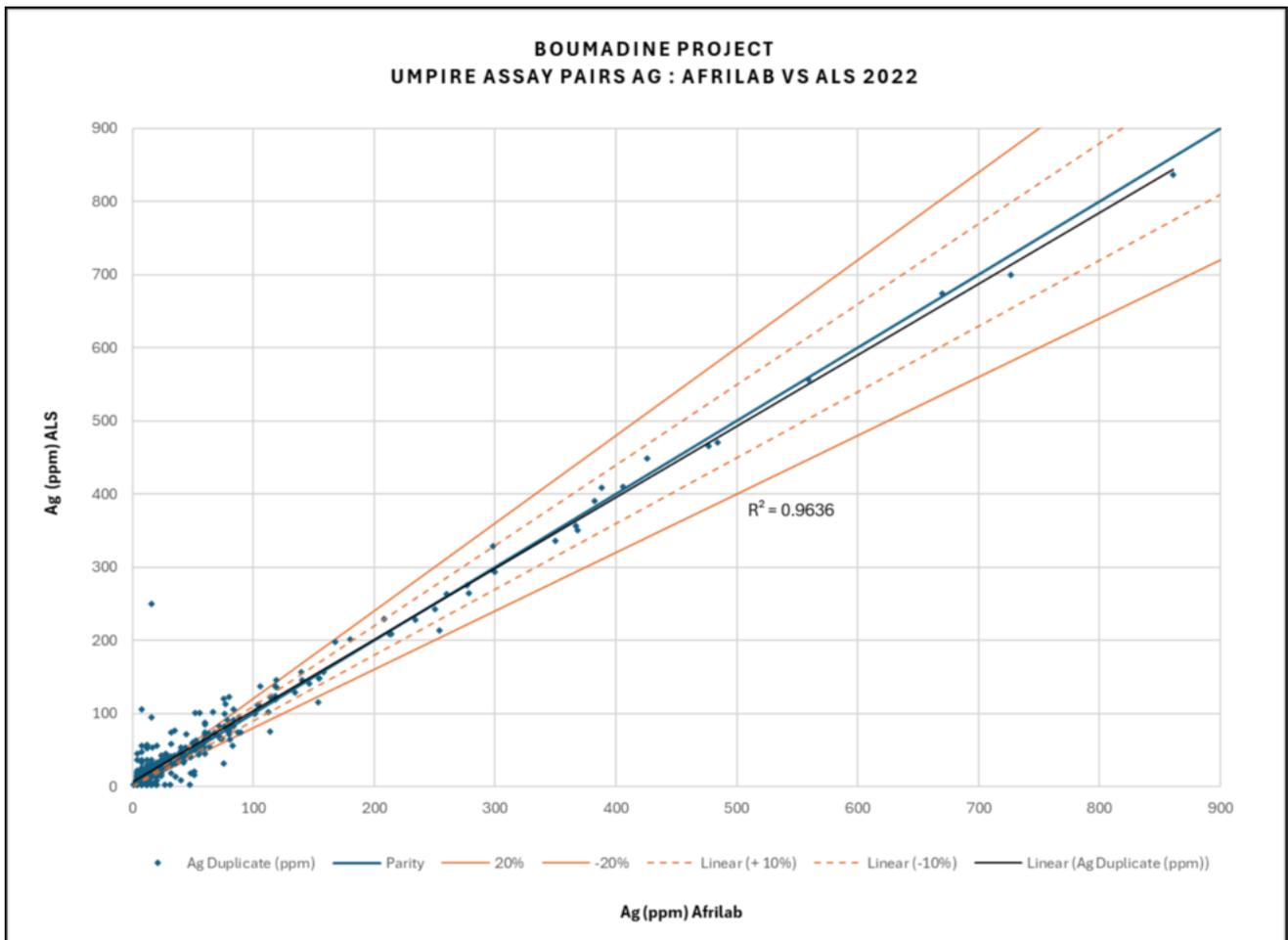


Figure 11-23 Umpire Assay Comparison for Ag: Afrilab Versus ALS 2022

Source: P&E (May 2024)

11.3 CONCLUSION

Aya implemented and monitored a thorough QA/QC program for the drilling completed at the Boumadine Deposit during the 2018 to 2024 period and also completed umpire assaying to confirm sampling and analyses undertaken during the drilling. Examination of QA/QC results for all recent sampling indicates no material issues with accuracy, contamination or precision in the data, and the late-2022 check assaying reviewed by the Author and P&E generally confirms the original data.

It is the opinion of the current Author that sample preparation, security and analytical procedures for the Boumadine Deposit are adequate and that the data are of good quality and satisfactory for use in the current Mineral Resource Estimate.

12.0 DATA VERIFICATION

12.1 DRILL HOLE DATABASE VERIFICATION

12.1.1 MRE 2025 Assay Verification

The current authors conducted a verification of the Boumadine Deposit drill hole assay data for silver, gold, copper, lead, and zinc for the 2024 to 2025 period. Certificates provided by Afrilabs were re-imported into an Excel template, and assay results were checked against the Geotic database and core photos. Approximately 5% of the certificates were verified by the author, and no errors were encountered during the verification process.

Assay data from 2018 to 2024 for the Boumadine Deposit were verified independently by P&E in the previous MRE report (P&E, May 2024). Approximately 15% of the overall data were verified for silver, gold, copper, lead, zinc, molybdenum and iron. Approximately 89% of the constrained data were verified for gold and molybdenum and ~16% for silver, copper, iron, lead and zinc. No errors were encountered in the data during the verification process.

The database verification undertaken by P&E is summarized in Table 12-1.

Table 12-1 Boumadine Database Verification Summary: May 2024

Element	All Recent Data (N=57,364)		Constrained Recent Data (N=1,591)	
	No. Verified	% Verified	No. Verified	% Verified
Au	8,758	15.3	1,415	88.9
Mo	8,760	15.3	1,409	88.6
Ag	8,566	14.9	259	16.3
Cu	8,566	14.9	259	16.3
Fe	8,566	14.9	259	16.3
Pb	8,566	14.9	259	16.3
Zn	8,566	14.9	259	16.3

Source: P&E May 2024

12.1.2 Drill Hole Data Validation

The Authors also validated the Mineral Resource database in Leapfrog Geo Sequent 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 drill hole collar locations, survey and missing interval and coordinate fields. A few errors were identified and corrected in the database.

12.2 QP SITE VISIT AND INDEPENDENT SAMPLING

Throughout 2024 and 2025, the current Authors made several visits to the Boumadine Property.

Mr. Antoine Yassa, P.Geo., of P&E and an independent Qualified Person under the terms of NI 43-101, completed a site visit to the Boumadine Property from March 12 to 14, 2024 as part of the independent QP visit for the 2024 MRE. The site visit included the following activities:

- Visiting various surface drilling sites;
- Inspection of onsite drill core logging and drill core storage facilities;
- GPS location verifications along the main mineralized trend from North to South
- Inspection of Afrilab in Marrakesh;
- Review of database, drill hole collar surveying, logging, sampling and QC procedures;
- Technical discussions and;
- Drill core verification sampling.

Mr. Yassa also collected 21 verification drill core samples from seven diamond drill holes. Samples were selected from holes drilled in 2019, 2022 and 2023. 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 sample preparation and pulp shipment directly to Actlabs Laboratories in Ancaster, Ontario for analysis. The pulp samples at Actlabs were analyzed for silver and gold by fire assay with gravimetric finish, and by 4-acid digest with ICP-OES finish for silver, copper, molybdenum, lead and zinc.

Actlabs is independent of Aya and P&E and runs 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.

Results of the Boumadine site visit verification samples are presented in Figures 12-1 to 12-7.

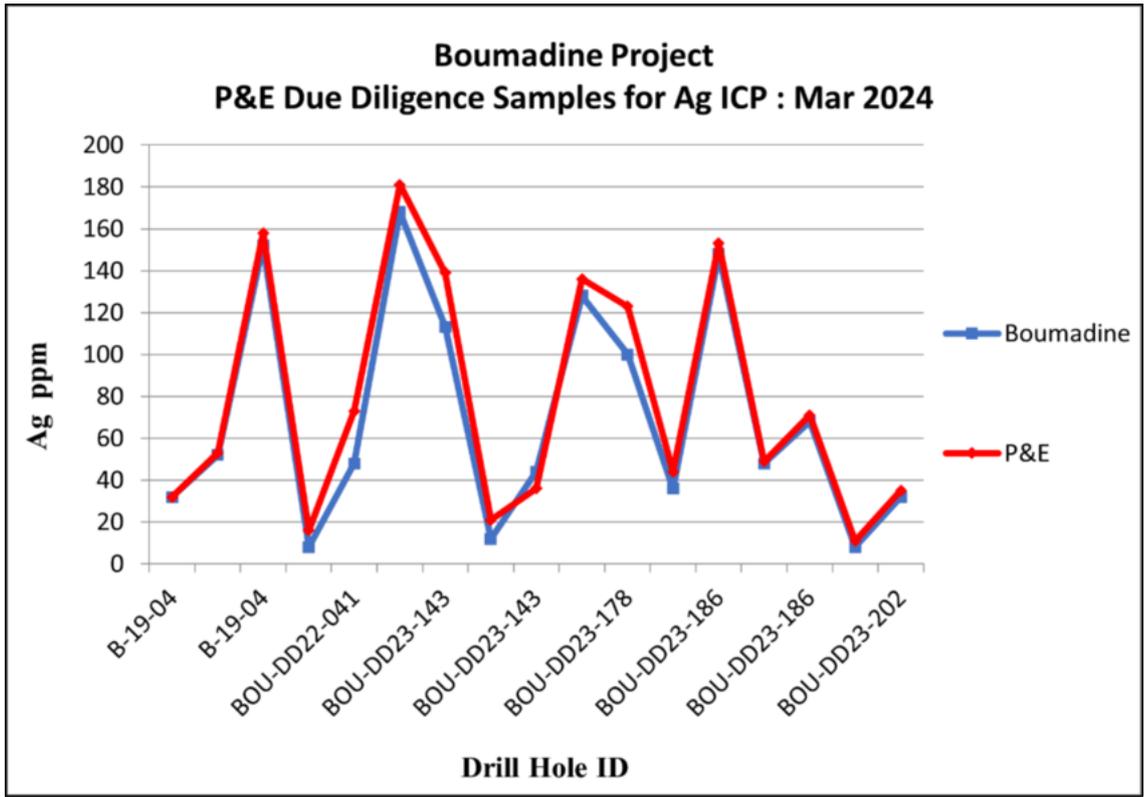


Figure 12-1 Results of March 2024 Ag (ICP) Verification Sampling by P&E
 Source: P&E (May 2024)

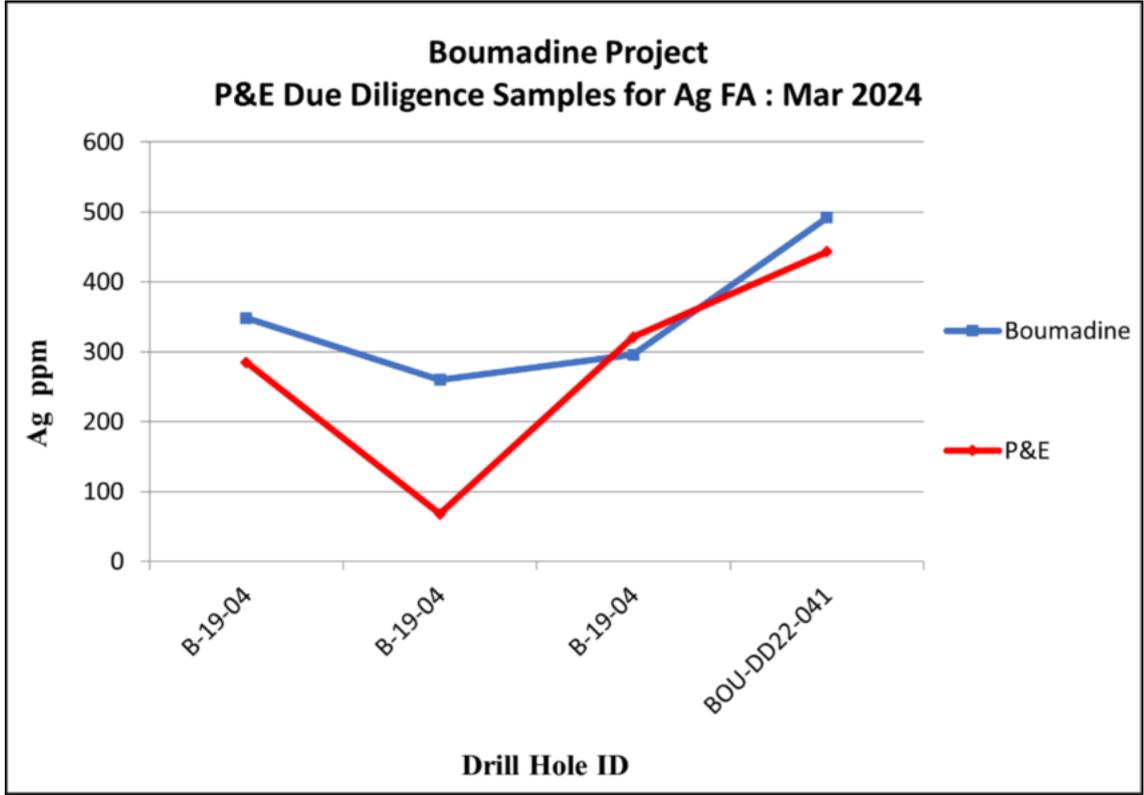


Figure 12-2 Results of March 2024 Ag (FA) Verification Sampling by P&E
 Source: P&E (May 2024)

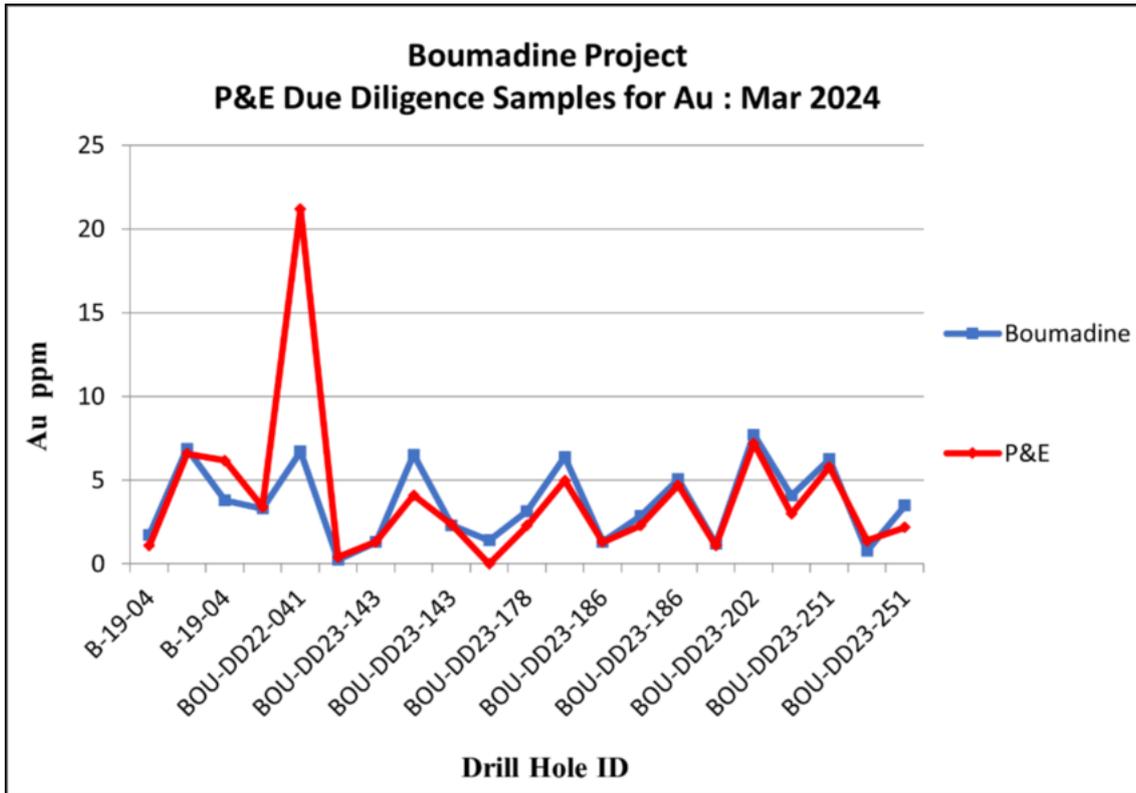


Figure 12-3 Results of March 2024 Au Verification Sampling by P&E

Source: P&E (May 2024)

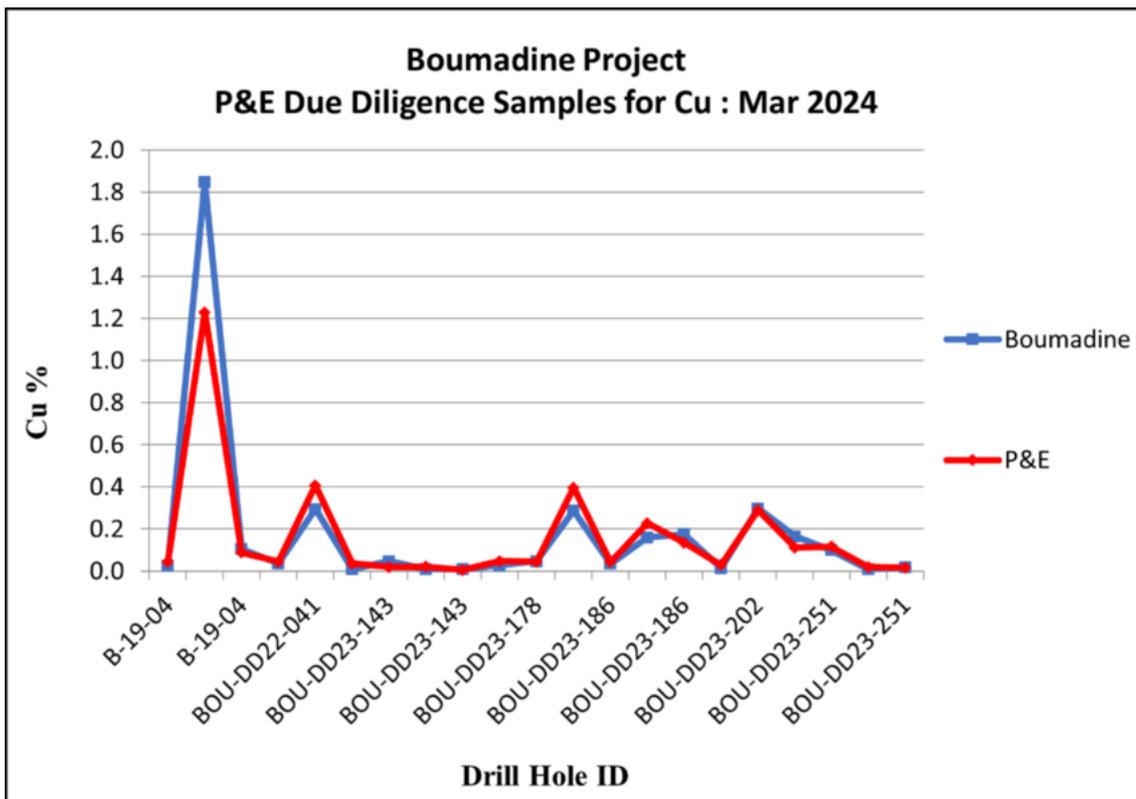


Figure 12-4 Results of March 2024 Cu Verification Sampling by P&E

Source: P&E (May 2024)

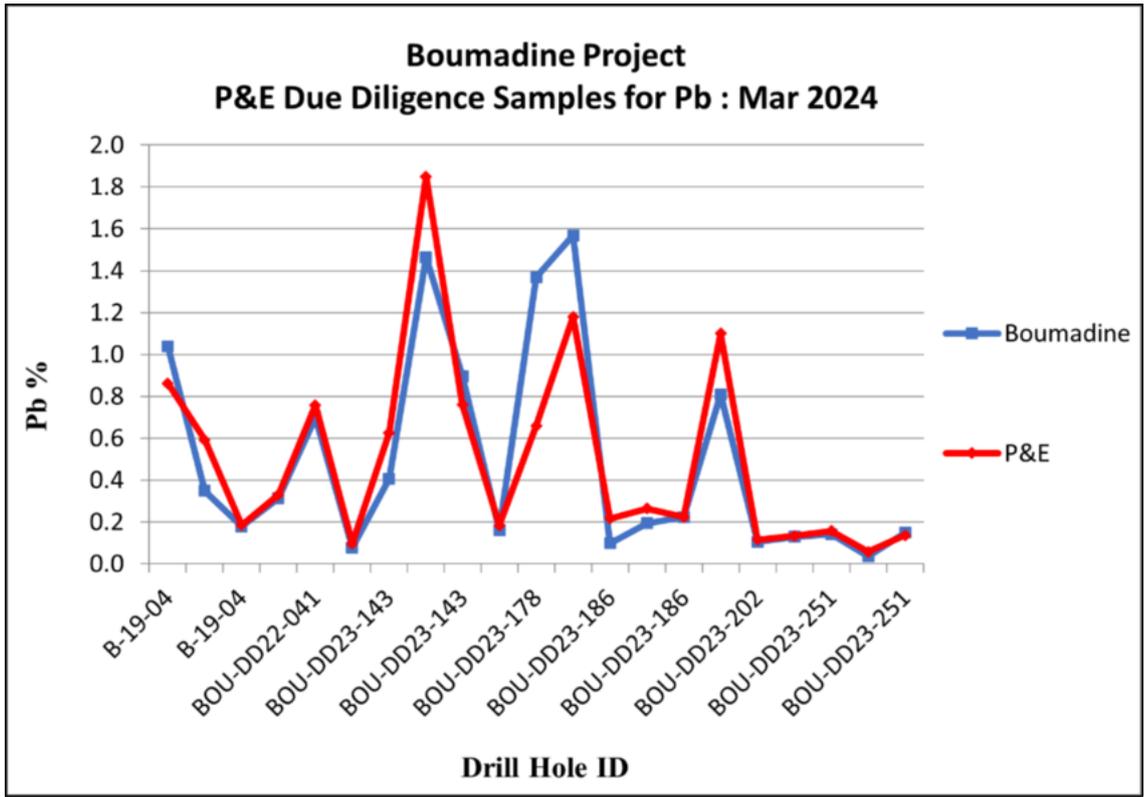


Figure 12-5 Results of March 2024 Pb Verification Sampling by P&E
 Source: P&E (May 2024)

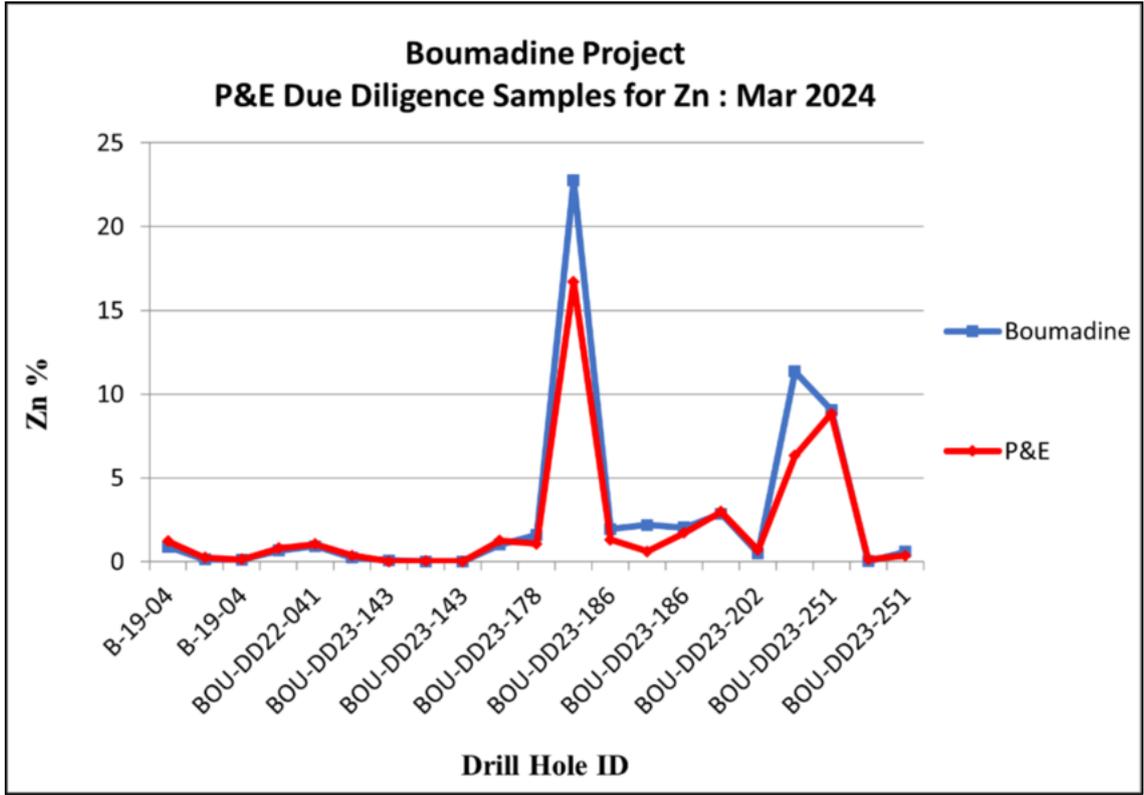


Figure 12-6 Results of March 2024 Zn Verification Sampling by P&E
 Source: P&E (May 2024)

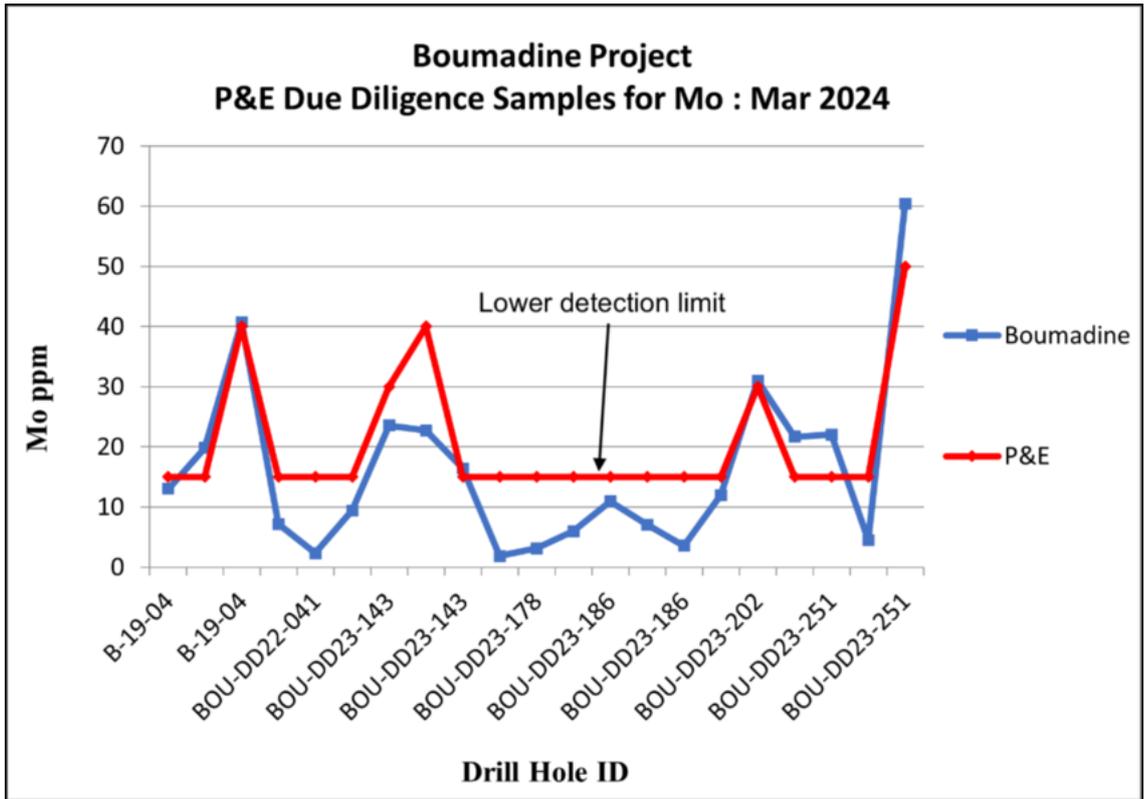


Figure 12-7 Results of March 2024 Mo Verification Sampling by P&E

Source: P&E (May 2024)

12.3 CONCLUSION

The current Author has verified the Boumadine Deposit data used for the current Mineral Resource Estimate. This verification included site visits, validation of drill hole assay data from electronic files, and an assessment of the available QA/QC data.

In March 2024, Mr. Antoine Yassa, P.Geo., of P&E, an independent Qualified Person, performed an independent due diligence review, site visit, and sampling. The findings indicated a strong correlation between the assay values in Aya’s database and the independent verification samples collected and analyzed by P&E at Actlabs. It was concluded that the supplied data were suitable for use in a Mineral Resource Estimate.

The current Author concludes that sufficient verification of the project data has been conducted and that the supplied data are of good quality and appropriate for use in the current Mineral Resource Estimate.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 INTRODUCTION

Several testwork programs have been carried out at different laboratories for the Boumadine Project, with the first pilot test program carried out between 1986-1991 and the latest lab test program completed in 2024. Tested materials have included historical tailings, fresh rock, and produced pyrite concentrate samples. A summary of both the historical and recent test programs is provided in Table 13-1.

Table 13-1 Summary of Testwork Programs

Testwork Program	Mineralogy	Comminution	Gravity	Flotation	Direct Cyanidation/CIL	Chlorination	Roast – CIL/Cyanidation	POX – CIL/Cyanidation	BIOX – CIL/Cyanidation	Albion – CIL/Cyanidation	MACH Reactor - Cyanidation
SODIM / BRPM SODECAT 1986-1991 (Pilot-scale)				✓							
URSTM 2011 (tailings sample)					✓						
Nicromet 2013 (tailings sample)						✓					
CINF 2017 (tailings sample)					✓		✓	✓			
SGS Lakefield 2018 (tailings and fresh rock samples)		✓		✓	✓		✓	✓			
SGS Lakefield 2019 (fresh rock sample)	✓			✓	✓			✓		✓	
SGS Lakefield 2022 (composited fresh rock sample)	✓	✓	✓	✓	✓		✓		✓	✓	
AFRILAB 2023 Sample Campaign (fresh rock)	✓										
SGS Lakefield 2023-2024 (composited fresh rock and pyrite concentrate samples)	✓		✓	✓			✓	✓			
Mintek 2024 (pyrite concentrate sample)					✓						✓
BGRIMM 2025 (bulk pyrite concentrate sample)	✓						✓				

Details of the historical testwork (1986-2017) can be found in the published Technical Report and Preliminary Economic Assessment on the Boumadine Project issued on 24 May 2019. Since the results have not been used in the development of the process design criteria, they will not be further discussed in this report. Details of the testwork done by Mintek (2024) have also not been provided since the results were not used toward the development of the process plant design and contained confidential information.

The metallurgical sampling locations for the material tested in the recent programs (2021-present) are shown by disks in the cross-sectional model renderings shown in Figure 13-1.

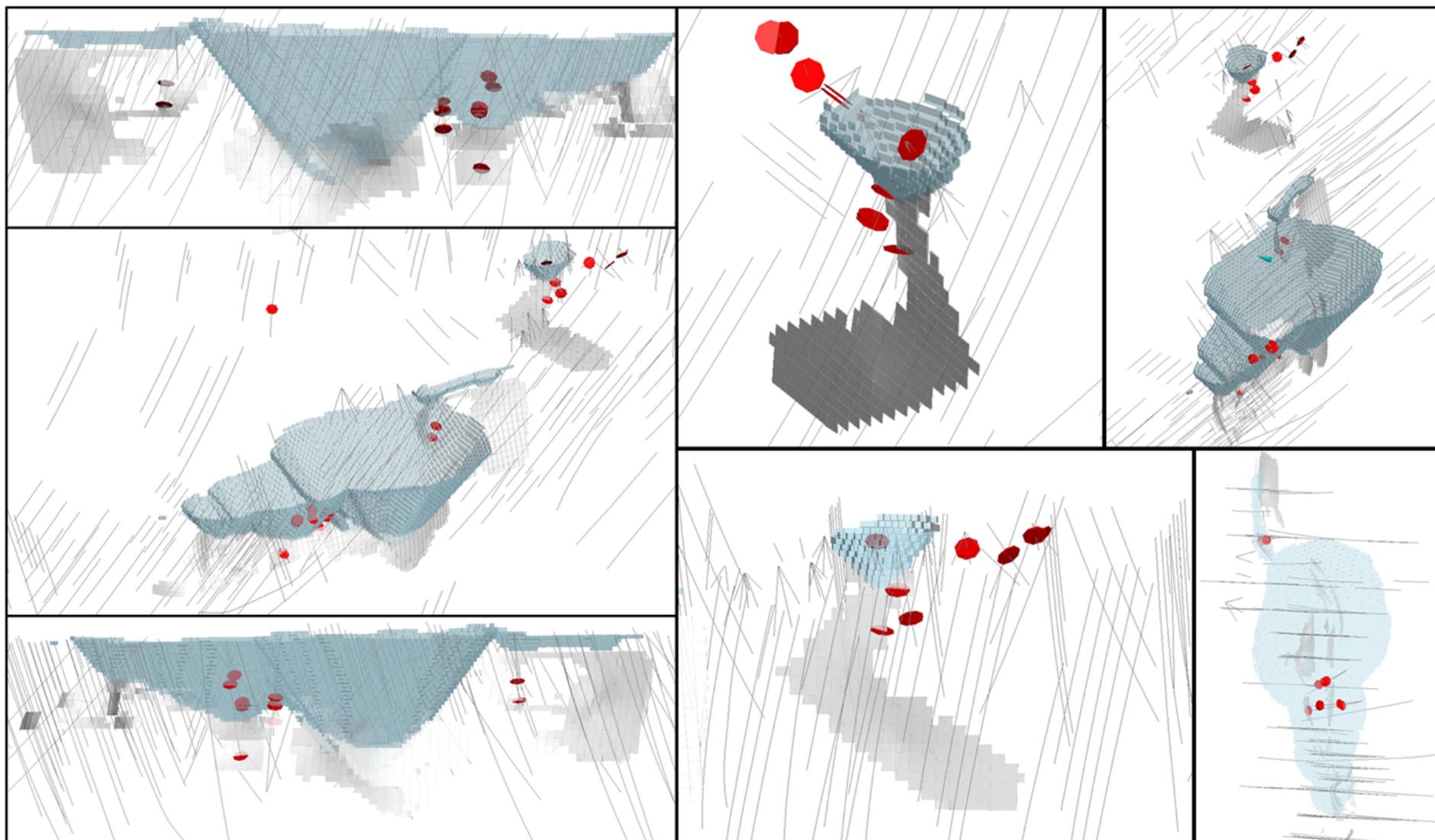


Figure 13-1 Metallurgical Sampling Locations

13.1.1 SGS Lakefield Testwork Program – 2018

In 2018, SGS Lakefield conducted a metallurgical test program on fresh rock and tailings samples from the Boumadine Project. The fresh rock sample (Sample #1) was screened at 10 mesh and any oversize was crushed to pass 10 mesh. It was then homogenized and rotary split into the required test charges. A total of nine tailings samples (Sample #2, 10350, 10351, 10352, 10353, 10358, 10359, 10360 and 10361) were received, which were also homogenized (individually) and split the required test charges.

The testwork program included:

- Head analysis.
- Grindability testing (for fresh rock sample only).
- Flotation testing.
- Cyanidation testing.
- Roast-CIL testing.
- POX-CIL testing.

13.1.1.1 Head Analysis

Representative aliquots of each sample were submitted for chemical analysis including whole rock analysis, ICP scanning, fire assay for Au and Ag, LECO for total sulphur and sulphides, XRF for Pb, Zn, and Cu by, and comprehensive analysis for other elements by ICP-MS and fire assay / ICP. The results for elements of interest are shown in Table 13-2.

Table 13-2 Head Analysis of Fresh Rock and Tailings Samples

Element	Units	Sample ID									
		Sample #1	Sample #2	10350	10351	10352	10353	10358	10359	10360	10361
S.G.	-	4.04	3.4	2.31	2.41	2.31	3.1	-	-	-	-
Fire Assay											
Au	g/t	4.29	3.20	1.79	1.34	1.58	2.16	1.75	2.10	3.26	2.52
Ag	g/t	-	-	-	-	-	-	120	165	201	136
LECO											
S	%	37.0	27.8	12.4	14.9	13.0	23.3	14.4	24.1	22.8	28.7
S=	%	36.1	24.5	0.40	5.21	0.53	20.0	-	-	-	-
XRF											
Pb	%	0.41	0.30	0.31	0.23	0.33	0.64	0.35	0.55	0.31	0.30
Zn	%	1.34	0.098	0.77	1.00	0.68	1.03	0.60	1.19	0.21	1.12
Cu	%	0.18	0.012	0.028	0.045	0.043	0.17	0.044	0.12	0.016	0.051

13.1.1.2 Grindability Testwork

The fresh rock sample was submitted for the Bond ball mill grindability test performed at 100 mesh of grind. The Bond ball mill work index (BWi) was determined to be 10.7 kWh/t, falling in the moderately soft range of SGS's database.

13.1.1.3 Flotation Testwork

Rougher Kinetic Flotation Testing

Open circuit batch rougher kinetic flotation tests were conducted on the Sample #1 (fresh rock) and Sample #2 (tailings).

The fresh material sample responded reasonably well to conventional lead / zinc / pyrite flotation process. Open circuit batch rougher flotation tests on the fresh rock sample resulted in 69.7% recovery of lead to a lead rougher concentrate at a grade of 7.8% Pb, and 16.7% of the gold also reported to the lead concentrate. The zinc rougher concentrate graded 36.8% Zn with 81.9% recovery at low mass pull (3.3%). The pyrite rougher circuit recovered 92% of the sulfur with high mass pull of 68.6%. The pyrite rougher concentrate sulphur grade was 50.7% and recovered 79.5% of the gold.

For the tailings sample, it was discovered that the pH of the pulped sample was very low following grinding. The quantity of lime required to raise the pH to 8.5 for flotation was very high and weight loss was observed during flotation. For the second test on Sample #2 material, the sample was washed, ground, filtered and washed again prior to flotation. The filtrate and wash solutions were analyzed and showed a high concentration of iron, indicating the presence of soluble salts which were contributing to the low pH and high lime consumption. A bulk sulphide flotation was performed on the washed tailings, producing a pyrite rougher concentrate recovering 99.6% of the sulphides and 91.3% of the gold at grades of 49.3% S and 6.1 g/t Au.

Cleaner Flotation Testing

Open circuit batch cleaner flotation tests were performed on Sample #1. The first test attempted to produce lead, zinc, and pyrite concentrates. A lead 4th cleaner concentrate grading 33.1% Pb was produced with 56.7% Pb recovery. The zinc 3rd cleaner concentrate assayed 51% Zn with 78% Zn recovery. The pyrite rougher concentrate contained 49.8% S and 3.4 g/t Au with recovery rates of 92.4% S and 73% Au. The pyrite rougher tails assayed 0.2 g/t Au. Most of the gold losses were in the lead circuit.

A second test was performed without lead flotation. A zinc concentrate was produced grading 46.4% Zn, but with only 47.6% recovery. The pyrite rougher concentrate assayed 49.8% S and 5.28 g/t Au, with 97.5% S recovery and 87.5% Au recovery. The gold loss to the zinc circuit was 11%, and the pyrite rougher tails assayed 0.22 g/t Au. 85.2% of the contained lead reported to the pyrite concentrate.

A third test was performed to produce pyrite concentrate for downstream testing (roast / CIL). A lead rougher stage was included to reduce the amount of lead in the pyrite concentrate. A zinc concentrate grading 54.2% Zn was produced with 79.5% Zn recovery. The pyrite concentrate graded 51% S and 5.18 g/t Au with sulphur recovery at 93.9% and gold recovery at 80.9%. The major gold loss was to the lead rougher concentrate at 15.5%. The pyrite concentrate was submitted for a roast / CIL test.

13.1.1.4 Cyanidation Testwork

Five cyanide bottle roll tests were completed on tailings samples (Sample #2, 10350, 10351, 10352 and 10353) and two tests on pyrite concentrate samples after fine grinding to <10 microns (μm).

The tailings samples were washed before leaching. Lime and NaCN consumptions were high and further weight losses occurred during the leaching period. The normalized recoveries (based on residue and direct head grades) ranged from 25-69%.

The lime and NaCN consumptions for the pyrite concentrate samples were also high, and the overall normalized recoveries were 38-41%.

Silver recoveries were higher with the overall normalized recoveries ranging between 47-98% for the tailings samples and 59-73% for the pyrite concentrate samples.

13.1.1.5 Roast-CIL Testwork

Four roasting tests were completed – two on the obtained flotation concentrate and two on the tailings composite samples. The tests were conducted as two-stage roasts, in a static muffle furnace for a total of four hours (two hours per stage). The sample was stirred every 15 minutes during the second stage. All sulphide sulphur was oxidized during roasting.

Each calcine was subsequently subjected to a standard cyanidation test as CIL, with 2 g/L NaCN, 10 g/L carbon for 24 hours. The RST-2 calcine (CN-9 feed) was ground in an attrition mill to <10 μm prior to cyanidation. The CIL testing results are listed in Table 13-3.

Gold extraction from the calcines ranged from 45% to 85%, whereas silver recovery ranged from 5% to 72%. The poor recovery from the fully oxidized calcine could have been due to the presence of 0.15% to 0.25% lead in the pyrite concentrates, which was known to cause problems in the roasting process.

13.1.1.6 POX-CIL Testwork

Seven pressure oxidation (POX) tests were completed on the obtained flotation concentrate and tailings composite samples. The POX tests were carried out in standard Parr 2 L titanium autoclaves. The pulp density in the POX tests was dictated by the sulphide sulphur grade in the POX feed and the need to create autothermal heating conditions. The POX feed was pre-acidulated to pH 1.5 for 30 minutes by addition of sulphuric acid. After POX, the pulp was filtered, and the solids were washed. POX residues were sub-sampled for analysis and the remaining residue forwarded for CIL. POX-4 through POX-7 tests evaluated the effect of Hot Curing (HC) and Lime Boiling (LB) on subsequent lime consumption and silver recovery during CIL. The POX-CIL results are summarized in Table 13-4.

Gold recovery from the flotation concentration after POX 1 pressure oxidation (POX CIL1) was high, at 97.4%. However, lime consumption was extremely high at 321 kg/t (POX feed basis). Silver recovery was low at 49%. The results are related to the chemistry in the pressure oxidation process, with high sulphide feeds tending to produce significant amounts of basic iron sulphate and silver jarosite. The sulphate ion in basic iron sulphate does not react with limestone and can only be neutralized with lime. Therefore, copious amounts of lime were consumed during neutralization of the POX residue prior to and during cyanidation. In addition, it is well known that silver jarosite is insoluble in cyanide solution. These are well known phenomena in processing of refractory gold sulphide concentrates by pressure oxidation.

The consumption of lime by the POX 4 residue was reduced significantly to 16.8 kg/t in the POX CIL 4 test, by employing a Hot Cure stage after pressure oxidation. Silver recovery in this test, however, dropped from 49% in POX CIL 1 to just 1.2%, indicating that more silver jarosite formed during hot curing.

In the final test on the flotation concentrate, POX CIL 6, the autoclave slurry was first subjected to Hot Curing, followed by solid-liquid separation and washing of the residue. The washed solids were then subjected to a Lime Boil stage, which is designed to breakdown the silver jarosite complex. Excess lime is added to the slurry (generally 100 to 200% stoichiometric excess of the sulphate in the solids), which is then held near boiling temperature for several hours. The overall lime requirement was 60 kg/t, and recovery of both gold and silver were high after lime boiling, at approximately 98%.

Table 13-3 Roast-CIL Results

Test ID	Feed	Temperature °C		Sulphide S %		Reagent Addition kg/t		Reagent Consumption kg/t		CIL Recovery %	
		Stage 1	Stage 2	Stage 1	Stage 2	NaCN	CaO	NaCN	CaO	Au	Ag
RST-1	F7 Pyrite Con	400	700	49.6	<0.05	2.82	1.16	1.02	1.16	62.5	9.1
RST-2	240g F7 Pyrite Con +10g Pb Con	400	700			4.17	4.29	3.91	4.27	84.8	72.5
RST-3	Oxidized Tailings Comp	650	800	8.62	<0.05	2.62	1.19	1.02	1.18	68.7	7.0
RST-4	Non-Oxidized Black Tailings Comp	650	800	24.3	<0.05	3.39	1.54	1.54	1.52	44.8	4.7

Table 13-4 POX-CIL Results

Test ID	Feed	Product to CIL	Lime Boil CaO addition		Reagents (kg/t of POX feed)				CIL Recovery %	
			kg/t LB feed	kg/t POX feed	Added		Consumed		Au	Ag
					NaCN	CaO	NaCN	CaO		
POX CIL 1	F7 Pb Ro Con & F7 Pyrite Con	POX 1 residue	-	-	7.24	320.8	3.55	320.8	97.4	49.2
POX CIL 2	Oxidized Tailings Comp	POX 2 residue	-	-	1.96	93.2	1.00	93.2	87.7	12.1
POX CIL 3	Non-Oxidized Tailings Comp	POX 3 residue	-	-	2.21	73.7	0.61	73.6	95.4	3.5
POX CIL 4	F7 Pb Ro Conc & F7 Pyrite Conc	POX 4 HC residue	-	-	6.56	16.8	4.45	16.8	97.6	1.2
POX CIL 5	Non-Oxidized Tailings Comp	POX 5 HC residue	-	-	1.49	2.4	0.37	2.2	43.7	65.7
POX CIL 6	F7 Pb Ro Conc & F7 Pyrite Conc	POX 6 HC-LB residue	263	59.2	2.11	0.6	0.90	0.6	98.2	97.6
POX CIL 7	Non-Oxidized Tailings Comp	POX 7 HC-LB residue	40.1	23.8	1.39	0.4	0.25	0.3	71.6	81.4

13.1.2 SGS Lakefield Testwork Program - 2019

In 2019, further metallurgical testwork was carried out on the fresh rock sample that was used in previous 2018 program. The sample was stored in the freezer and therefore was considered with the same characteristics to samples tested in the previous program.

Sequential flotation was carried out on the ground sample to produce a pyrite concentrate for gold recovery testwork, including baseline cyanidation, POX, and Albion Process™. The Albion Process™, developed by Xstrata Pty Ltd. (Glencore Technology Pty Limited) is a combination of fine grinding and exothermic oxidative leaching without the need for pressure treatment or external heating.

Open circuit batch rougher flotation tests resulted in a lead rougher concentrate with a grade of 10.4% Pb at a recovery of 67.0%, along with 12.9% of the gold and 43.7% of the silver, which also reported to the lead concentrate of grades of 19.8 g/t and 6,906 g/t, respectively. The zinc rougher concentrate graded 51.1% Zn at 68.8% recovery with a low mass pull (1.9%). The pyrite rougher circuit recovered 93.9% of the sulphur to the concentrate, with a high mass pull of 70%.

A full chemical head analysis and a semi-quantitative XRD analysis were conducted on the pyrite concentrate. The concentrate had a sulphur grade of 51.0% with 4.96 g/t Au and 310 g/t Ag, and the concentrate sample was found to be composed of 94% pyrite with just over 5% quartz and less than 1% each of arsenopyrite and chalcopyrite.

A bottle roll cyanidation test was conducted on the pyrite concentrate to serve as a baseline for comparing the gold recoveries after processing the pyrite concentrate by either the Albion or POX processes. The concentrate was first attrition-ground (250 g at 50% solids for 30 minutes), resulting in a product D_{80} of 4.47 μm (determined via Malvern analysis). The CIL conditions were 5.0 g/L NaCN with oxygen sparging at 40% solids with 15 g/L carbon for 24 hours.

The Albion Process™ testing at SGS Lakefield is accredited by Glencore Technology. The attrition-ground solids were pulped to 10%, heated to 95 °C and the slurry was first acidified using sulphuric acid to a pH of 4.5 until the pH started to fall naturally. Oxygen was sparged into the pulp at ~1 L/min and alkali addition in the form of 20% solids limestone slurry (CaCO_3) was added via a pH controller to control the pH at 5.5. The test ran for a period of ~52 hours with constant monitoring of pH, ORP and limestone additions were recorded. Hot water was added to the pulp throughout the leach to make up for any evaporative losses. The residue from Albion leaching was then subjected to CIL, under the same conditions as for the baseline test.

With the relatively low level of sulphide oxidation obtained in the first Albion leach test (Neutral Albion-1R), a repeat test was carried out. In the repeat Albion test (Neutral Albion-1R), the pyrite concentrates leach residence time was increased from 52 to 72 hours and the oxygen sparging rate was increased from 1 L/min to 1.7 L/min. Initial acid addition to kick-start oxidation was 2 kg/t lower than in the first test at 21 kg/t H_2SO_4 . Alkali addition of 1,209 kg/t CaCO_3 was almost three times the amount used in the first test (440 kg/t).

Three identical POX tests were conducted, and the products were combined for downstream testing (hot cure, lime boil and CIL). The pulp was pre-acidulated for 30 minutes at pH 1.5 with sulphuric acid in 2 L titanium autoclave vessels. After 30 minutes, the autoclave was sealed, pressure tested and heated to target temperature. The tests were run at 225 °C for 60 minutes, with 690 kPa (~100 psi) of oxygen over pressure applied. The three POX pulps were combined and heated to 95°C for four hours in a 4 L Pyrex vessel with a lid and overhead mechanical agitation (Hot Curing process, to re-dissolve basic iron sulphate to ferric sulphate in solution). The washed hot cure residue was pulped to 10% solids and boiled at 95°C for six hours at a reagent addition of 35.8 kg/t POX feed (Lime Boiling process, to break down the silver jarosite complex). After six hours, the pulp was forwarded for CIL testing.

The CIL testing results of the three gold recovery processes are summarized in Table 13-5. The baseline test had high addition and consumption rates for NaCN and demonstrated good silver recovery at 84%, but poor recovery of 17%. CIL on the Albion leached product (NA-1, obtained from 52 hours Albion leaching) resulted in slightly higher gold recovery to 45% with similar silver extraction (85%) as the baseline. The repeated Albion test (NA-2, 72 hours) resulted in CIL leaching recovery of 81% for gold and 92.3% for silver. It was anticipated that with further optimization of the Albion conditions, further sulphide oxidation can likely be achieved and gold extraction from the residue increased even further. Pressure Oxidation of the pyrite concentrate,

followed by hot curing and lime boiling, provided the best extraction of the three processes – with extractions of 98% and 96%, respectively.

Table 13-5 CIL Results Summary

Test	Reagents (kg/t of CIL feed)				Recovery %		CIL Residue g/t	
	Added		Consumed		Au	Ag	Au	Ag
	NaCN	CaO	NaCN	CaO				
CIL-1 Baseline	24.9	1.83	22.3	1.60	17.3	83.9	4.07	48
CIL-2 Neutral Albion-1	3.85	11.7	3.52	11.6	45.4	84.8	1.65	29.4
CIL-4 Neutral Albion-1R	5.46	6.79	5.11	6.79	81.0	92.3	0.37	8.6
CIL-3 POX-HC-LB*	2.23	3.98	1.52	3.98	98.2	96.6	0.41	47.6

*35.8 kg/t CaO added during LB prior to CIL-3.

13.1.3 SGS Lakefield Testwork Program - 2022

The 2022 test program by SGS Lakefield was carried out on a composite of 75 samples received from the Boumadine deposit (called the Main Composite). The samples were crushed, blended and split to portions for chemical and mineralogy analysis, comminution testing (BWi), and metallurgical testing including:

- Gravity separation.
- Flotation (open circuit and locked cycle).
- Direct CIL cyanidation of pyrite flotation concentrate.
- BIOX-CIL.
- Albion Leach™-CIL.
- Roast-CIL.

13.1.3.1 Head Analysis

The chemical analysis included gold and silver by fire assay, total organic carbon, total carbon, sulphide sulphur, mercury, lead, zinc, and a multi-element ICP scan. The results for the elements of significant interest are shown in Table 13-6.

Table 13-6 Composite Sample Head Analysis

Element	Au g/t	Ag g/t	As g/t	Cu %	S= %	Pb %	Zn %
Assay	3.33	113	834	1.94	26.5	1.00	2.25

13.1.3.2 Mineralogy

X-Ray Diffraction

The XRD results provided the overall mineral composition of the composite sample as summarized in Table 13-7. Slightly more than 40% of the sample weight were sulphides with the most common being pyrite.

Table 13-7 Composite Sample Mineralogy

Mineral	% Weight
Quartz	34.6
Pyrite	32.6
Muscovite	17.1
Chlorite	3.3
Sphalerite	3.2
Gypsum	2.9
Arsenopyrite	2.8
Biotite	1.7
Galena	0.9
Chalcopyrite	0.9
Total	100

TIMA-X

A portion of the sample was stage-crushed, targeting a P₈₀ of 75 µm but obtaining a true P₈₀ of 220 µm. The sample was screened into +53 µm and -53 µm fractions. The modal mineralogy, liberation and association characteristics of sulphides, and gold and silver deportment of the samples were characterized by Tescan Integrated Mineral Analyzer (TIMA-X) technology using the High-Resolution Mapping method. The results for both size fractions were combined to represent the composite sample.

Modal Mineralogy

The sulphides in the composite sample were found to consist of mainly of pyrite (55.5%), sphalerite (4.1%), arsenopyrite (3.2%), galena (1.4%), and trace amounts of chalcopyrite (0.4%), pyrrhotite (0.1%), and tetrahedrite (0.06%). The remainder of the sample was found to be composed of quartz (21.7%), micas/chlorite/clays (10.6%), and trace amounts (<1% each) of other minerals.

A discrepancy was observed between the quartz and pyrite grades obtained by the XRD and TIMA analyses, which was attributed to potential gravity settling during sample preparation.

Sulphide Liberation and Association

At the grind size P₈₀ of 220 µm, the liberation and mineral association of the key minerals of interest were as follows:

- A high amount of the contained pyrite was found to be free and liberated (approximately 94%). The increase of pyrite liberation from the +53 µm to -53 µm size fraction was minimal at 2%.
- Approximately 77% of the contained arsenopyrite was free and liberated. The increase of arsenopyrite liberation between the +53 µm to -53 µm size fractions was significant at 24%.
- Approximately 78% of the contained sphalerite was free and liberated. The increase of sphalerite liberation between the +53 µm to -53 µm size fractions was significant at 47%.
- Approximately 65% of the contained galena was free and liberated. The increase of galena liberation between the +53 µm to -53 µm size fractions was significant at 60%.
- The remainder of the arsenopyrite, sphalerite and galena occurred as complex particles and middlings with pyrite and quartz / feldspars.

Based on individual particle liberation and grade, mineralogically limited grade-recovery analyses were carried out to indicate theoretical maximum achievable grade by recovery of the sulphide minerals. The analysis projected:

- Galena grades between 100% and 58% for recoveries of between 54% and 87%.

- Sphalerite grades between >99% and 63% for recoveries of between 69% and 96%.
- Arsenopyrite grades between >99% and 72% for recoveries of between 56% and 96%.

Gold and Silver Mineralogy

69% of the contained gold in the composite sample was determined to be native gold and 30% was electrum. The gold grains were generally very fine (<15 µm) with 78% of the gold grains below <3 µm, accounting for 23% of the gold by mass. The gold minerals were poorly liberated, with pure, free and liberated grains accounting for only 12.4% of the total. Non-liberated grains occurred as middlings with pyrite (45.8%), and sphalerite (1.8%), and complex particles (40.0%). These phenomena suggested that achieving high gold concentrations by physical means would be unrealistic, and that leaching extraction would need to follow a high degree of pyrite oxidation.

Silver was found to be widely distributed as native silver (20%) with the remainder occurring as acanthine, electrum, tetrahedrite, polybasite, freibergite, jalpaite, native gold, and others. Only 1.6% of the silver minerals were free, pure and liberated, with the remaining non-liberated particles occurring as middlings with pyrite (25.8%) and quartz / feldspars (4.1%), and predominantly as complex particles (67.4%).

13.1.3.3 Comminution Testing

The Main Composite sample was submitted for the Bond ball mill grindability test performed at a 150 mesh of grind. The measured Bond ball mill work index (BWi) was 15.4 kWh/t, falling in the moderately hard range when compared to SGS's database.

13.1.3.4 Gravity Separation Testing

1.8 kg of the Main Composite sample was used for the test feed. The feed was ground in a laboratory rod mill to a P₈₀ of 78 µm. The mill discharge was passed through a Knelson MD-3 Concentrator, collecting a concentrate and producing tailings. The Knelson concentrate was further upgraded on a Mozley mineral separator. The final concentrate represented 0.1% of the feed mass and graded 312 g/t Au and 747 g/t Ag. The combined Knelson and Mozley tailings assayed 1.55 g/t Au and 117 g/t Ag. The recovery to the final Mozley concentrate was 19% Au and 0.7% Ag. The results indicated that gravity separation would add little value to the flowsheet.

13.1.3.5 Flotation Testwork

Open circuit rougher and cleaner tests were completed on the Main Composite sample, attempting to produce a saleable lead / gold / silver concentrate and a pyrite concentrate with the maximum amount of gold and silver, along with a zinc concentrate that would be floated prior to pyrite flotation. Locked cycle testing was also completed to determine the effect of recirculating middling products, to test the robustness of the conditions and to produce a pyrite concentrate for downstream testing.

Rougher Flotation Tests

The baseline lead-zinc-pyrite kinetic test was completed at a primary grind P₈₀ of 58 µm. The reagents and grind size were varied in the subsequent tests. These test conditions produced a lead-gold-silver concentrate grading 9.39% Pb, 3.47% Zn, 1.60 % As, 30.8% S, 12.4 g/t Au, and 678 g/t Ag in the baseline test. The concentrate recovered 87% of lead, 32.8% of gold, and 53.9% of Ag. However, 14.8% of the Zn and 13% of the As also reported to the lead rougher concentrate.

Further tests were conducted with the objective of increasing the grade of the lead-gold-silver concentrate and a pyrite concentrate containing the remainder of the gold and silver. The tests investigated different reagent schemes and grind sizes. The main objective of these tests was to reduce the amount of zinc reporting to the lead concentrate. Various zinc / sulphide depressants, in addition to ZnSO₄ and NaCN, were tested, such as sodium sulphide, sodium sulphite, and sodium metabisulphite. None of these alternatives significantly reduced the level of zinc in the rougher concentrate. The zinc recovered to the lead concentrates ranged from 14% to 25%. The combined lead + pyrite concentrates all achieved recoveries of greater than 90% gold and 85-87% silver.

Two flotation tests were performed at finer grind sizes of P_{80} of 37 μm . The finer grind did not appear to have any effect on overall grades or recoveries, although reagent additions also differed from the coarser ground tests.

Bulk Sulphide Flotation Test

A single bulk sulphide flotation test was conducted at a neutral pH, producing a concentrate with 1.53% Pb, 3.52% Zn, 1.79% As, 44.1% S, 5.16 g/t Au and 185 g/t Ag. The recovery of all elements was very high (>96%), but the concentrate mass pull was also very high at 58.7%.

Cleaner Flotation Tests

Four open circuit batch cleaner flotation tests were performed on the Main Composite sample, trialling various circuit arrangements, grind sizes and flotation reagents conditions. The results showed that adding sodium cyanide to the lead flotation circuit helped to depress sphalerite and pyrite flotation, decreasing the amount of zinc in the lead concentrate by 50%. Implementing a zinc regrind helped improve the zinc concentrate grade slightly but the recovery increased more significantly from 59.7% to 74.8%. Implementing a finer lead grind produced a higher-grade concentrate with lower zinc content but reduced overall recovery. These trials were able to achieve sufficient lead recovery and zinc grade but had high grades of gold and silver in the lead concentrate, lead grades below the saleable target, and low zinc recovery. The results suggested that to achieve concentrate grades of 50% or more for both lead and zinc, low recoveries would be expected.

Locked Cycle Flotation

A locked cycle test was completed with the goals of assessing overall metallurgical performance with the incorporation of internal recycles and producing concentrate for downstream testing. The test was completed over six cycles with 2 kg of feed per cycle. Products were recirculated in a typical counter-current manner. The lead and zinc first cleaner tailings were advanced to the next circuit roughing stage. The tested flowsheet, including the recirculation of middling streams, is shown in Figure 13-2.

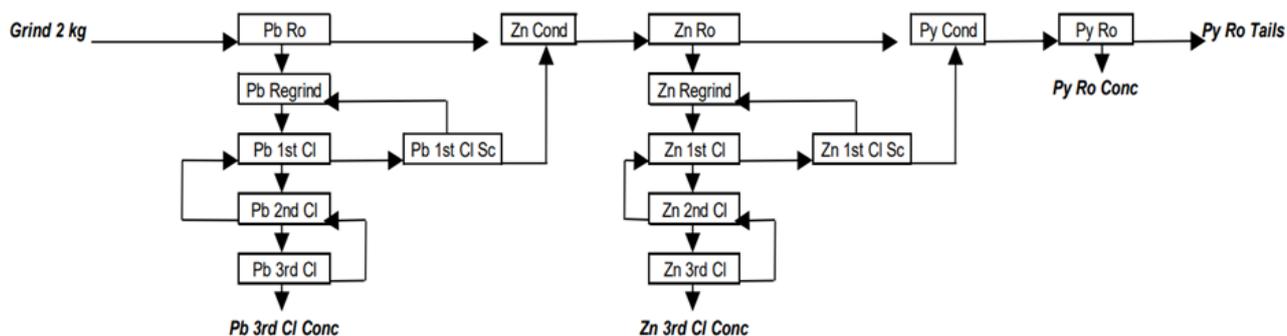


Figure 13-2 Locked Cycle Test Flowsheet

The conditions for the locked cycle test were based on the results from the batch tests, with minor adjustments in the reagent dosages. The primary grind P_{80} target was 58 μm , lead regrind D_{80} was 18 μm , and zinc regrind D_{80} was 17 μm . The test conditions are summarized in Table 13-8.

The test generally exhibited satisfactory stability except for zinc, where the stability was lower than the other elements, suggesting that steady state may not have been achieved in the zinc circuit. The results of the locked cycle tests based on the out-products from the final 3 cycles are presented in Table 13-9.

Table 13-8 Locked Cycle Test Conditions

Pb Circuit	Reagents Dosed (g/t)					Time (min)		Cond.
	Lime	Zn(SO ₄).7H ₂ O	NaCN	Aero3418A	MIBC	Grind	Cond.	
Grind	-	300	100	5	-	70		
Pb Rougher 1	-	-	-	-	10			1
Pb Rougher 2	-	15	5	5	5			1
Pb Rougher 3	-	15	5	5	5			1
Pb Re grind	-	30	10	-	-	15		
Pb 1 st Cleaner	-	-	-	5	5			1
Pb 1 st Cl Scav	-	-	-	2.5	5			1
Pb 2 nd Cleaner	-	-	-	-	5			1
Pb 3 rd Cleaner	-	-	-	-	5			1
Total	0	360	120	22.5	40			7
Zn Circuit (Pb Ro + 1 st Cl Sc Tails)	Reagents Dosed (g/t)				Time (min)			Froth
	Lime	Cu(SO ₄).5H ₂ O (10% w/w)	SIPX (1% w/w)	MIBC	Grind	Cond.	Froth	
Zn Conditioner 1	1,400	-	-	-		3		
Zn Conditioner 2	-	225	-	-		3		
	265	-	-	-		2		
Zn Rougher 1	-	-	10	10		1		1
Zn Rougher 2	165	-	10	7.5		1		2
Zn Re grind	250	-	5	-	5			
Zn 1 st Cleaner	60	-	-	7.5		1		2
Zn 1 st Cl Scav	-	-	5	7.5		1		1
Zn 2 nd Cleaner	50	-	-	7.5		1		1.5
Zn 3 rd Cleaner	140	-	2.5	5		1		1.5
Total	930	225	32.5	45		11		9
Sulphide Circuit (Zn Ro + Zn 1 st Cl Tails)	Reagents Dosed (g/t)			Time (min)		pH	Eh	
	PAX	MIBC	H ₂ SO ₄	Cond.	Froth			
Sulphide Conditioner	-	-	1,000	1		8.0	100	
Sulphide Rougher 1	100	10	-	1	2	8.2	75	
Sulphide Rougher 2	100	7.5	-	1	4	8.4	50	
Sulphide Rougher 3	100	10	-	1	8	8.5	50	

Table 13-9 Locked Cycle Test Results Summary

Product	Weight %	Assays							% Distribution						
		Cu %	Pb %	Zn %	As %	S %	Au g/t	Ag g/t	Cu	Pb	Zn	As	S	Au	Ag
Pb Conc	3.0	4.81	26.7	4.35	1.67	35.0	26.6	1,923	75.3	84.5	7.1	4.7	4.1	26.5	53.1
Zn Conc	2.3	0.37	0.48	58.1	0.27	32.9	1.22	166	4.3	1.2	72.0	0.6	2.9	0.9	3.5
Pyrite Conc	54.9	0.061	0.18	0.65	1.69	42.4	3.80	78.5	17.4	10.6	19.5	86.8	89.9	69	39.6
Pyrite Tails	39.9	0.015	0.089	0.063	0.21	2.02	0.27	10.5	3.0	3.7	1.4	7.9	3.1	3.6	3.8
Head (calc)	100	0.19	0.95	1.83	1.07	25.9	3.02	109	100	100	100	100	100	100	100
Head (direct)	-	0.19	1.00	2.25	0.083	-	3.33	113	-	-	-	-	-	-	-

The following observations were made from the locked cycle test:

- The final lead concentrate had a good lead recovery of 84.5%, but low grade of 26.7% Pb. The main diluents were copper and zinc.
- The final zinc concentrate grade was good at 58.1% Zn in 2.3% of the total mass, but the recovery was modest at 72%.
- The pyrite concentrate recovered 69% of the gold, 39.6% of the silver, 89.9% of the S, and 86.8% of the arsenic, in a high mass pull of 54.9%.
- Gold and silver were mainly recovered to the lead concentrate, with 26.5% of the gold and 53.1% of the silver reporting to the lead concentrate.

13.1.3.6 Pyrite Concentrate Testing

The pyrite concentrates from cycles the six were combined and blended, then split into small charges for direct CIL, Albion, BIOX, and roasting testwork. The P₈₀ size of the concentrate composite was determined to be 42 µm, and the composition of elements of interest is provided in Table 13-10.

Table 13-10 Pyrite Concentrate Analysis

	Au g/t	Ag g/t	Pb %	Zn %	Fe %	As %	S %	S= %
Assay	3.83	76.2	0.27	0.45	35.8	1.72	42.8	41.4

Direct CIL Testing

Two cyanidation tests were completed on the pyrite concentrate at two grind sizes: one at the “as received” grind size of P₈₀ 42 µm and the other reground to a D₈₀ of 6.3 µm. The other test conditions were held constant across both trials. The results of the tests are summarized in Table 13-11.

Table 13-11 Direct CIL Extraction Results

Test No.	P ₈₀ µm	Reagent Added kg/t CN Feed		Reagent Cons. kg/t of CN Feed		Au Extraction %	Ag Extraction %
		NaCN	CaO	NaCN	CaO		
CIL 1	42	20.3	4.25	4.39	3.91	9.5	36.4
CIL 2	6.3	28.5	15.4	14.5	15.3	22.4	80.2

Gold extraction was improved with fine grinding but was still very low for both trials. Better extractions were seen for silver, reaching 80% for the finely ground sample. The reagent consumptions were high and increased significantly with fine grinding.

Albion Testing

Glencore recommended Albion leach parameters that were in line with their standard neutral leach protocol. The Albion oxidation test conditions and results are summarized in Table 13-12.

Table 13-12 Pyrite Concentrate Albion Testing Results

Test No.	Retention Time h	P ₈₀ µm	Limestone Addition kg/t	Sulphide Oxidation %
NAL-1	96	6.3	1,647	96.3
NAL-2	78	10	1,261	74.7

Sulphide oxidation was high (96.3%) for the test at the finer grind size P₈₀ of 6.3 µm and longer retention time but dropped significantly to 74.7% when the grind was coarsened to 10 µm and the retention time decreased to 78 hours.

The Albion residues were washed before they were subjected to cyanide leaching testing. The CIL test results are provided in Table 13-13.

Table 13-13 CIL Leaching Results for Albion Residues

Sample	Reagent Added kg/t CN Feed		Reagent Cons kg/t of CN Feed		Au Extraction %	Ag Extraction %
	NaCN	CaO	NaCN	CaO		
NAL-1 Residue	7.9	2.5	3.2	2.4	94.4	94.8
NAL-2 Residue	8.2	3.3	2.3	3.3	84.9	91.3

Gold and silver extractions were greatly improved after the Albion pre-treatment. The test at 6.3 µm grind size achieved gold and silver extractions of 94.4% and 94.8%, respectively. The gold extraction decreased significantly with the coarser grind and shorter retention time. Reagent consumptions for both trials were reduced in comparison with the direct CIL tests.

Bacterial Oxidation (BIOX) Testing

The inoculum build up was performed in preparation of the bioleach tests. For each of the tests, the pyrite concentrate sample was slurried with the bacteria, with conditions maintained for optimum bacterial activity (heated to 40°C in an acidic solution, targeting ~10.5% pulp density). After 18, 26, or 35 days, the pulps were filtered, and the filtrate was collected and measured. A filtrate sample was submitted for analysis and the rest was retained. The filter cake was repulped with water, agitated and re-filtered. The filter cake was then washed three times, targeting one displacement volume of water each time a representative subsample of the washed filter cake was submitted for chemical analysis. The remainder of the filter cake was forwarded to cyanidation. The bio-oxidation liquor would have to be neutralized prior to disposal, but no neutralization testwork was conducted at this stage. The results of bio-oxidation are provided in Table 13-14.

Table 13-14 Bio-Oxidation Testing Results

Test No.	Days	Sulphide Oxidation %
BAT-2	18	60.6
BAT-3	26	88.0
BAT-1	35	92.8

Sulphide oxidation improved with increased retention time, ranging from 60.6% for the 18-day test to 92.8% for the 35-day test. It should be noted that a 35-day residence time in a batch test translates to an ~5-day retention in a continuous process.

The results of CIL on the bio-oxidation residue are provided in Table 13-15.

Table 13-15 Bio-Oxidation Residue CIL Results

Sample	Test No.	Reagent Added kg/t CN Feed		Reagent Cons. kg/t of CN Feed)		Au Extraction %	Ag Extraction %
		NaCN	CaO	NaCN	CaO		
BAT-2 Residue	CIL 4	17.7	9.0	7.9	8.8	75.2	75.5
BAT-3 Residue	CIL6	15.3	11.4	8.8	11.2	82.2	40.1
BAT-1 Residue	CIL 7	23.6	16.2	23.4	16.2	85.1	37.0

The CIL gold extractions improved from 75 to 85% with the longer oxidation period (i.e. higher sulphide oxidation), however silver extractions decreased drastically. This result is most likely due to jarosite formation and may be overcome with the inclusion of a lime boil stage.

Reagent consumption in the CIL tests was very high. High cyanide consumption in commercial bacterial leaching plants is generally associated with the formation of elemental sulphur as an intermediate sulphide oxidation product in the BIOX process, and the reaction between cyanide and sulphur to produce thiocyanate ions in solution.

Roast-CIL Testing

Five roasting tests were completed on the pyrite concentrate. The tests were conducted in a static muffle furnace under various roasting conditions (changing the number of stages, temperature and/or aeration). The samples were rabbled every 15 minutes. The results are summarized in Table 13-16.

Table 13-16 Roast Testing Results

Test	Roast Conditions	Sulphide Oxidation %
RST-1	1 st stage 400°C and 2 nd stage 700°C	99.9
RST-2	1 st stage 400°C and 2 nd stage 600°C	97.6
RST-3	1 st stage 400°C, 2 nd stage 600°C with 1L/min oxygen, 3 rd stage 600°C with 2L/min oxygen	99.8
RST-6	1-stage 500°C	89.3
RST-7	1-stage 600°C	95.1

The results showed that oxidation improved with more than one roasting stage with higher temperatures. While oxidation was high with aeration, only one trial was conducted so it is difficult to attribute the results to aeration versus the longer retention time and temperature.

The roasted product from each trial was then split in half, with one half left as-is, and the other half reground in a lab attrition mill to a D₈₀ of <10 µm. The calcines were then subjected to standard CIL cyanidation tests. Results of the CIL tests are provided in Table 13-17.

Table 13-17 Roast Residue CIL Results

Sample	P ₈₀ µm	Reagent Added kg/t CN Feed		Reagent Cons. kg/t of CN Feed		Au Extraction %	Ag Extraction %
		NaCN	CaO	NaCN	CaO		
RST 1 Calcine	46	21.9	12.0	6.5	12.0	61.1	19.9
	7.4	24.0	6.9	4.8	6.6	71.1	44.9
RST 2 Calcine	41.5	24.1	-	13.3	-	55.6	65.9
	6.7	31.4	12.2	12.7	12.2	65.9	60.3
RST 3 Calcine	44.3	27.3	8.6	10.3	8.6	59.3	32.5
	7.5	27.3	5.9	5.1	5.5	66.7	51.6
RST 6 Calcine	51.3	29.1	34.7	13.6	34.7	52.5	51.8
	8.1	37.2	14.6	22.5	14.6	63.9	61.8
RST 7 Calcine	36.5	30.5	13.5	13.9	13.5	56.4	29.6
	5.9	26.8	9.3	11.8	9.1	66.7	50.4

Gold extraction from the calcines was poor, ranging from 55.6% to 71.1%, whereas silver extraction ranged from 19.9% to 60.3%. SGS stated that they were uncertain why gold and silver extractions were so poor, even after near complete oxidation of the sulphides. The formation of an impervious molten phase during the roasting process was suggested as a possibility.

13.1.4 SGS Testwork Program – 2023-2024

In 2024, more testing was conducted by SGS on the Main Composite and the pyrite concentrate composite produced during lock-cycle testing obtained from the previous 2022 program. The main objective of the 2024 program was to investigate methods of separating the arsenopyrite from the pyrite in the sulphide

concentrate prior to oxidative treatment. The idea was that improving the gold grade and Au:S ratio would enhance the economics of sulphide oxidation by separating the higher gold grade / lower sulphide grade arsenopyrite from the pyrite in the pyrite concentrate. By treating the two streams separately, favorable conditions for each could be discerned to produce a more optimized and economic process. Several methods were tested, including gravity separation and Jameson Cell flotation.

After none of the tested methods proved suitable, the 2024 program was then re-focused on the pre-oxidation of the concentrate prior to cyanide leaching. A locked cycle test was completed on the Main Composite to produce a lead concentrate for mineralogy and a pyrite concentrate for pressure oxidation and roasting. The oxidative tests were performed to verify the recoveries achieved in previous programs between 2018-2022.

13.1.4.1 Mineralogy Testing

The pyrite concentrate from the 2022 locked cycle tests had been submitted for chemical analysis at that time but was submitted for bulk mineralogy analysis as part of the 2024 testwork. The main sulphide in the sample was pyrite with trace pyrrhotite, followed by minor arsenopyrite, and sphalerite. Quartz and muscovite were the main silicates. The liberation of Fe-sulphides (pyrite and pyrrhotite) was very good at 98%, and that of arsenopyrite was 87%. Approximately 7% of the arsenopyrite was associated with Fe-sulphides. The P_{80} of Fe-sulphides was 41 μm ; that of arsenopyrite was 38 μm .

Eighty-three gold grains were identified, ranging in size from <3 to 10 μm . The gold minerals were associated mainly with pyrite and arsenopyrite and they occurred as poorly exposed attachments (47%) and locked inclusions (53%) in the sulphides. This indicated that very fine regrinding would be required to liberate or expose the gold minerals associated with the sulphides. The submicroscopic gold would not be amenable to cyanidation, while the poorly exposed gold attachments would be theoretically amenable to cyanidation.

D-SIMS analysis of pyrite yielded an average gold concentration of 1.62 ppm, and of arsenopyrite 13.5 ppm. The total amount of gold (both microscopic and submicroscopic) associated with Fe-sulphides accounted for 63%, arsenopyrite 14%, and both Fe-sulphides and arsenopyrite 18%. More than 95% of the total microscopic and submicroscopic gold was associated with Fe-sulphides and arsenopyrite. Therefore, recovery of the sulphides will also yield the maximum recovery of gold.

A total of 474 silver grains were identified, including acanthite (argentite), native silver, Ag-Cu-Sb sulphides / sulfosalts, and hessite. The silver was not liberated and mainly associated with iron sulphides (83.6%), arsenopyrite (3.5%), complex particles (10.9%), and with iron sulphides and arsenopyrite (1.8%). Approximately 17% of the silver minerals occurred as inclusions while the remainder (83%) were variably exposed. Silver minerals were fine and ranged in size from <3 to 15 μm .

13.1.4.2 Gravity Separation Testing

Two 91 g aliquots of the pyrite concentrate were used to determine the potential for upgrading the gold and reducing the arsenic grade using a Mozley Mineral Separator table. One of the aliquots was ground in a lab attrition mill to a D_{80} grind size of 7 μm while the other was tested without a regrind. A Mozley concentrate and three Mozley middling samples, along with the Mozley tailings, were collected, filtered, dried, and assayed for Au, Ag, and As. The gold and arsenic recovery were calculated, and their relationship was plotted. It was observed that gold followed the arsenopyrite, indicating no separation of arsenic was achieved.

13.1.4.3 Flotation Testing

Open Circuit Batch Cleaner Flotation

A 2 kg test charge of the Main Composite was ground in a laboratory rod mill at 60% solids for 70 minutes obtaining a P_{80} particle size of 58 μm . The mill discharge was subjected to lead rougher flotation followed by zinc rougher flotation before producing a pyrite rougher concentrate from the zinc rougher tailings, using conditions established in the 2022 testwork campaign. The pyrite rougher concentrate was reground to 37 μm in a laboratory rod mill with ceramic media. The reground concentrate was cleaned in three stages and with a first cleaner scavenger. The flowsheet of the open circuit batch cleaner flotation test is presented in Figure 13-3.

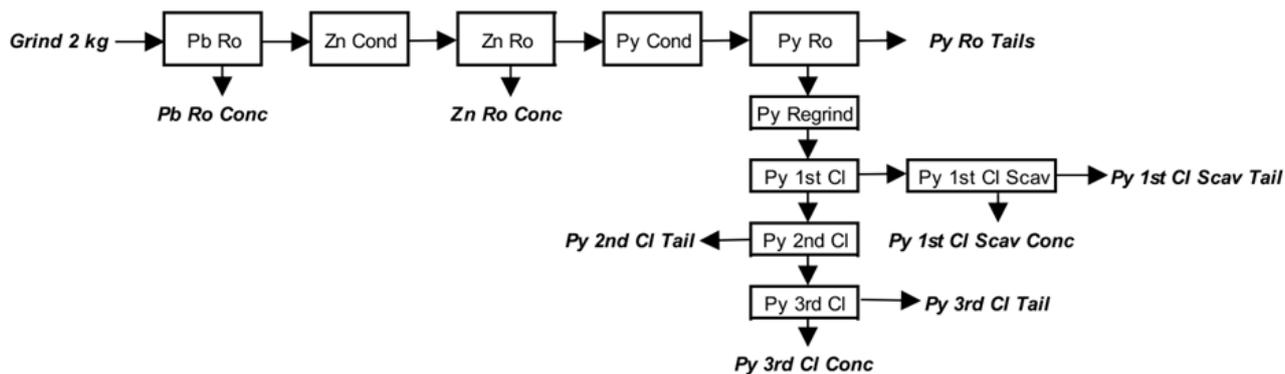


Figure 13-3 Cleaner Flotation Flowsheet for Separating Arsenopyrite / Pyrite

Similar to the gravity separation tests, the Au:S ratio was not improved. While the pyrite 3rd cleaner concentrate gold grade increased from 1.41 g/t Au to 4.34 g/t Au, the sulphur grade increased even more, from 9.8% to 52.9%. Significant gold losses were observed in the cleaning stages.

Jameson Cell Dilution Test

A bench flotation test was performed to evaluate the potential of a Jameson flotation cell to upgrade gold in a cleaner application. This test followed the bench dilution test procedures supplied by Glencore Technologies in which dilute flotation conditions are used to simulate the effects of wash water on the system. A sequential test was performed to remove the lead and zinc first, then the tailings were used to evaluate the pyrite cleaner circuit.

The results indicated no benefit to dilution cleaning of the pyrite concentrate, but there was a slight improvement comparing to cleaner flotation.

Arsenopyrite-Pyrite Separation Testing

A 4 kg test charge of the Main Composite was ground to a P₈₀ of 58 µm in a laboratory rod mill. The mill discharge was subjected to sequential lead, zinc and pyrite rougher flotation to produce enough pyrite concentrate for several tests that would investigate separating arsenopyrite from the pyrite. The obtained pyrite concentrate was split into four equal aliquots for the testwork.

Various reagent combinations were examined during the tests. The most promising results were observed using a simple combination of lime, copper sulphate and xanthate at a high pH, with 80% of the arsenic reporting to the concentrate. However, there was no significant improvement in the gold grades in the concentrate in these tests, and there was a direct correlation between gold and sulphur recoveries with R2 values very close to 1.

Locked Cycle Test

A single 6-cycle locked cycle test was completed to produce pyrite concentrate for POX and roasting tests, using the same flowsheet as the 2022 locked cycle test. The same test conditions from the 2022 locked-cycle test were used, including residence times and reagent additions (all conditions are shown in Table 13-7). The only variance was lime addition, which was added to the zinc circuit as required to maintain the target pH. The results are summarized in Table 13-18.

Table 13-18 Locked Cycle Test Results Summary

Product	Weight %	Assays						% Distribution					
		Cu %	Pb %	Zn %	S %	Au g/t	Ag g/t	Cu	Pb	Zn	S	Au	Ag
Pb Conc	2.1	6.02	32.5	4.36	30.3	32.0	2,244	71.4	79.4	4.9	2.4	21.0	47.7
Zn Conc	2.6	0.41	0.80	56.6	32.6	1.35	220	6.0	2.4	77.4	3.2	1.1	5.7
Pyrite Conc	52.1	0.07	0.22	0.59	46.2	4.61	83.5	20.2	13.3	16.2	90.0	73.7	43.2
Pyrite Tails	43.2	0.01	0.1	0.07	2.73	0.31	7.78	2.4	4.9	1.5	4.4	4.1	3.3
Head (calc)	100.0	0.18	0.88	1.91	26.8	3.25	101	100.0	100.0	100.0	100.0	100.0	100.0
Head (direct)	-	0.19	1.00	2.25	26.5*	3.33	113	-	-	-	-	-	-

The results were similar to results from the previous 2022 program by SGS (shown in Table 13-8), further validating the test conditions. The gold, silver and sulphur grades and recoveries were slightly higher, and the gold and sulphur ratios in the pyrite concentrates were essentially the same (0.09 and 0.1, respectively).

With a significant silver grade, the lead concentrate from the third cleaner was submitted for silver mineralogy. The analysis found that the silver was fine-grained with most particles in tetrahedrite form, ranging widely from 0.6 to 31% w/w Ag.

13.1.4.4 Pyrite Oxidative Treatment Testing

To verify the previous test results, one roast test and one POX test were conducted on the locked-cycle pyrite concentrate samples.

Roast-CIL Testing

A total of 500 g of concentrate was roasted in two 250 g batches as a two-stage roasting process, in a static muffle furnace. Each stage was two hours, and the sample was rabbled every 15 minutes. The first stage was conducted at 400°C and the second was at 700°C. Neither stage was aerated. Both batches were able to achieve sulphide oxidation of 99.9%.

After the roast, the two calcines were blended and split in half. One half was left as-is and the other half was reground in a lab attrition mill to a D_{80} particle size of 4 μm . The calcines were then subjected to a standard cyanidation / CIL test. The results are summarized in Table 13-19.

Table 13-19 Roast Residue CIL Results

Sample	P_{80} μ	Test No.	Reagent Added kg/t CN Feed		Reagent Cons. kg/t of CN Feed		Au Extraction %	Ag Extraction %
			NaCN	CaO	NaCN	CaO		
Roast Calcine	49	CIL 2	5.84	5.97	0.62	5.97	73.0	11.3
	4	CIL 3	7.6	5.39	2.81	5.39	80.4	30.2

Gold recovery was 80% from the finely ground calcine by CIL and 73% from the as-received calcine. Silver recoveries were lower at 11.3% and 30.2% for the coarse and fine sizes, respectively. The comparative test from the previous program achieved a gold recovery of 71.1% and silver recovery of 44.9% after grinding to 7.4 μm (shown in Table 13-16).

It was also observed that the addition and consumption of NaCN and CaO for the program were significantly lower than the previous program (results shown in Table 13.2.16). There was no explanation on this discrepancy in the SGS report and requires further investigation. The lesser quantity of reagent added and consumed could have led to the significant drop of silver recoveries observed in the 2024 test program.

POX-HC-LB-CIL Testing

Due to the high sulphide content in the feed, the POX test was conducted at 8% w/w pulp density in DI water to simulate autothermal heating conditions in the autoclave. The pulp was pre-acidulated for 30 minutes at a pH of 1.5, and then it was sealed and heated to the target temperature of 225°C. The test was run for 690 kPa of oxygen over-pressure applied.

After the pressure oxidation test, the pulp was subjected to hot curing to re-dissolve the basic iron sulphate to reduce the lime consumption in the subsequent CIL step. The hot curing process involved heating the pulp to 95°C for four hours in the autoclave. An 80% weight loss was observed when the iron sulphates were re-dissolved, upgrading the gold and silver grades to 19.5 g/t and 430 kg/t, respectively. 96.5% of the sulphur was oxidized during the POX step.

After the hot cure, lime boiling was conducted to break down the silver jarosite formed during POX, since it is refractory to cyanidation. The washed hot cure residue was pulped to 10% w/w solids and boiled at 95°C for 6 hours with 294 kg/t feed of CaO added, relative to the lime boil feed (46 kg/t based on the POX feed). The

lime addition rate was based on the estimated sulphate concentration of the hot cure residue. Following the lime boil, the pulp was sent to CIL testing.

CIL testing of the POX-HC-LB pulp resulted in extractions of 96.9% Au and 93.4% Ag. These results compared well with the previous test program results, through which 98.2% of the contained gold and 96.6% of the contained silver were extracted (results shown in Table 13-4).

13.1.5 AFRILAB 2023 Sample Campaign

The African Laboratory for Mining and Environment (AFRILAB) assayed 117 samples to determine the total sulfur content in each. Silver, zinc, lead, and iron content were determined by XRF analysis on drill cores logged by the exploration team. The results of the 2023 Sample Campaign were used to produce a sulphur content model based on elemental compositions and mineralogical associations. The multi-linear regression analysis indicated a strong correlation ($R^2 = 0.98$) between the calculated values and the actual laboratory test results, demonstrating the development of a valid relationship between the metal elements' content and the total sulfur content. The equation and graph representing this correlation are shown in Equation 13.2.1 and Figure 13-4, respectively.

$$\%S = 0.0154 \cdot Ag + 0.344 \cdot Zn + 0.593 \cdot Pb + 1.00 \cdot Fe - 0.25 \tag{13.2.1}$$

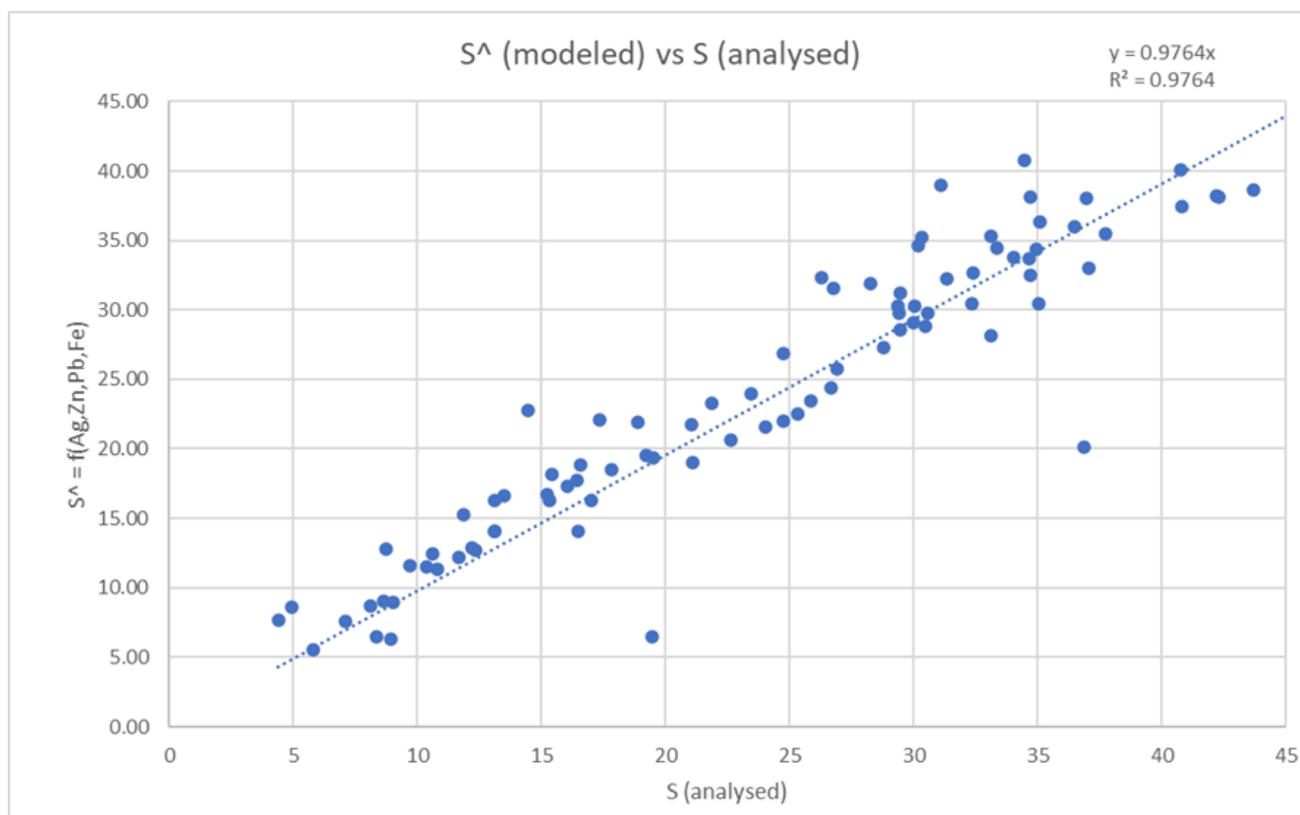


Figure 13-4 Correlation Between Analyzed and Calculated Sulphur Content

13.1.6 BGRIMM 2025 Testwork Program

In 2025 BGRIMM was contracted to perform further testwork on SGS produced samples with the primary tasks included conducting fundamental test analyses of the sample and performing experimental studies on processes such as roasting and cyanidation for gold leaching of the pyrite concentrate. The objective was to

determine the optimal process conditions and parameters for the main flowsheet, thereby achieving the highest leaching recovery rates for gold and silver under the best process conditions.

SGS prepared the BGRIMM sample utilising the test conditions identified in the 2022 and 2023 locked cycle tests. It should be noted the bulk flotation conditions were open circuit resulting in slightly lower gold and silver in the pyrite concentrate than would be expected.

Table 13-20 SGS Bulk Flotation Results

Product	Assays, %, g/t						
	Pb	Zn	Fe	As	S=	Au	Ag
Pb Ro Conc	9.57	5.33	27.4	2.68	33.6	15.5	711
Zn Ro Conc	0.56	29.1	10.8	0.95	30	2.17	128
Py Ro Conc	0.28	0.23	40.9	1.8	44.4	4.1	46.6
Pyrite Rougher Tails	0.06	0.07	2.54	0.091	1.16	0.19	7.9
Head (calc.)	0.88	1.86	19.2	0.97	21.1	2.9	81
Head (direct)	0.88	2.45	19.7	1.03	22.5	4.11	96

13.1.6.1 Feed Sample Properties

BGRIMM received 60 kg of the bulk pyrite concentrate, measuring a d(0.9) particle size of 77.53 µm, true density of 4491.2 kg/m³, and elemental analysis provided in Table 13-21.

Table 13-21 Elemental Analysis Results of Pyrite Concentrate (%)

Element	Au(g/t)	Ag(g/t)	As	Fe	S	C	Si	Al	Mg	Sb
Content	3.96	80.10	1.86	37.95	45.64	0.044	3.24	0.65	0.067	0.021
Element	Pb	Ca	Cd	Bi	Ti	Bi	Zn	Mn	Cu	Sn
Content	0.22	0.082	0.021	0.007	0.017	0.007	0.24	0.0086	0.063	0.011

X-ray diffraction analysis indicated the received sampled primarily contains pyrite as the dominant mineral phase with a minor presence of a mica mineral phase, while chemical phase analysis of gold indicated that the majority of gold is encapsulated with ~15% of the gold exposed.

13.1.6.2 Roasting Condition Testing

Single Stage Roasting Condition Test

To investigate the roasting behavior of this pyrite concentrate and determine the optimal single-stage roasting temperature, single-stage roasting temperature experiments were conducted at temperatures ranging from 500 - 950°C under a strongly oxidizing atmosphere to ensure complete reaction at each temperature.

The primary objective of roasting under a strongly oxidizing atmosphere is to remove sulfur (S). Experimental results indicate that as the roasting temperature increases, the calcine yield gradually declines, the arsenic (As) removal rate initially decreases and then rises, while the sulfur (S) removal rate generally exhibits an upward trend.

As the temperature increases, the Pb volatilization rate rises. When the roasting temperature exceeds 650°C, the Pb volatilization rate from the material accelerates significantly. Pb vapor tends to cause clogging and corrosion of equipment pipelines. Moreover, Pb volatilization may entrain fine gold particles into the flue gas, resulting in gold losses. Therefore, the roasting temperature should be maintained below 650°C.

Each roasting product was grinded and further subjected to 48 h leach tests at 33% w/w pulp density and 6 kg/t NaCN with the results indicated in Figure 13-5.

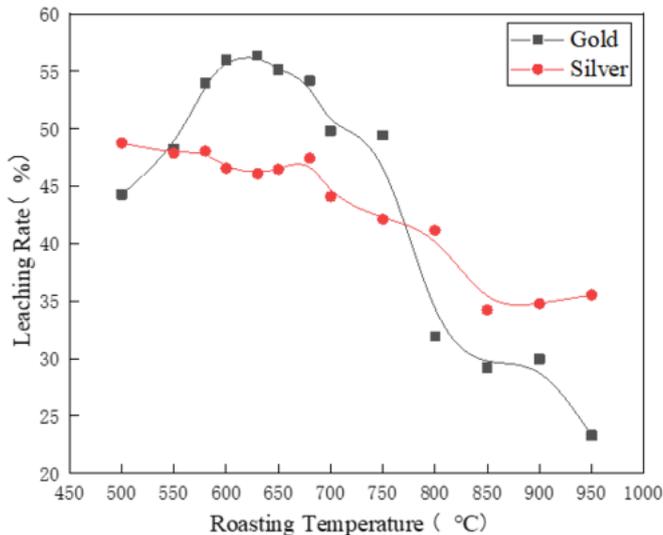


Figure 13-5 Au and Ag Leach Rates at Different Roasting Temperatures

As the temperature gradually increases, the Au leaching rate first rises and then declines, reaching a maximum of ~56% at approximately 630°C. In contrast, the Ag leaching rate decreases progressively with increasing temperature, peaking at ~48% at 500°C. Based on this evaluation, the optimal single stage roasting temperature for this pyrite concentrate is determined to be 630°C.

Two Stage Roasting Condition Test

Building on the single stage roasting test, a two-stage roasting temperature condition tests were conducted under varying first stage calcination temperatures (450-630 °C), with the first stage using a weakly oxidizing atmosphere and the second stage a strongly oxidizing atmosphere. Each roasting step was performed for 2 hours.

The experimental results indicate that when the second-stage calcination temperature is fixed at 630°C, variations in the first-stage calcination temperature have minimal impact on calcine yield, desulfurization rate, and deleaching rate. The calcine yield remains stable between 70%~71%, while the desulfurization rate consistently stays around 99.5%. However, the first-stage calcination temperature significantly affects arsenic (As) removal efficiency. The arsenic removal rate initially increases and then decreases with rising temperature, reaching a maximum of 86.34% at a calcination temperature of 580°C.

Leaching was performed on the two stage roasted product at the same conditions used for the single stage roast products. The experimental results demonstrate that when the first stage calcination temperature is set at 530°C, the leaching rate of gold (Au) reaches its highest value of ~63%, while the silver (Ag) leaching rate under these conditions is ~46%. After comprehensive evaluation, the calcination method for this concentrate was determined to be a two-stage process at 530-630°C.

Roasting Time Condition Test

Under the optimal two stage roasting temperature conditions, the roasting time for each stage was set in 30 min increments from 30 min to 3 hours. According to the test results, as the roasting duration increases, the calcine yield gradually decreases, while the removal rates of As and S generally show an upward trend. When the roasting time exceeds 2hours, the sulfur content in the calcine essentially ceases to decrease. When the roasting time exceeds 2.5hours, the arsenic removal rate no longer shows significant improvement.

Utilizing the same leaching conditions the test results indicated, as the roasting time increases, the leaching rates of gold and silver gradually rise. When the total time of the two stage roasting exceeds 2.5 hours, the leaching rates of Au and Ag from the calcine basically stop increasing. Therefore, the optimal roasting time for each stage is determined to be 2.5 hours.

Sodium Roasting Exploratory Test

"Sodium roasting" is an important technology in mineral processing and metallurgical pretreatment. Its core idea is: at high temperatures (usually in the range of 500-1000°C), sodium salts are added to mineral raw materials, causing them to chemically react with the target components or impurities in the raw materials. This changes the chemical form or physical properties of the target components, making them easier for subsequent leaching, separation, or purification. "Sodium roasting" also plays a certain role in the oxidative pretreatment of some refractory gold ores (encapsulated in sulfides, arsenides, or silicates).

Three samplers were roasting, a no additive control sample, one with sodium carbonate, and one with sodium hydroxide at the two stage roasting conditions already determined. The results indicate that the addition of sodium salts leads to an increase in calcine yield but a decrease in the removal rates of harmful elements.

Leaching of the products indicate that soda roasting does not improve the gold leaching rate but significantly enhances the silver leaching. However, the high content of harmful elements in the iron concentrate after soda roasting compromises its quality. Consequently, the addition of sodium salts for roasting is not recommended.

Hydrated Lime Roasting Exploratory Test

Addition of 6 g of Ca(OH)₂ with 100 g pyrite concentrate sample and utilizing the optimum roasting, acid-, and cyanide leaching conditions resulted in reduction of sulfur and arsenic removal rate as well as a reduction of ~12% gold extraction rate and ~14% silver extraction rate. Roasting with hydrated lime is not recommended at this stage.

13.1.6.3 Cyanidation Pre-Treatment Testing

A large quantity of calcine was produced under optimal roasting conditions (530 °C for 2.5 hours, and 630 °C for 2.5 hours) for subsequent experiments achieving an average calcine yield rate of 69.7%.

Leach vs CIL Testing

Test conditions were grinding to P₉₀ 30µm, 33%w/w solids, pH 11, 6kg/t NaCN at 48 hour leaching, with and without 20 g/L activated carbon.

The results show either leaching method is suitable for this material, with similar extraction rates for with and without activated carbon addition.

Grind Sensitivity Testing

The same leaching conditions were used but the grind size was varied from P₉₀ 0.3-4µm. The results indicated that the material had to be ground to below 1µm to see some improvement in gold recovery with no difference in silver recovery. The energy requirements and material handling would be cost prohibitive to explore further at industrial scale.

Cyanidation Pre-Treatment Testing

Utilizing the established leaching conditions in the leach vs CIL testing, four pre-treatments methods were tested to establish the effect on the final leach rates, namely:

1. Roasting – Cyanidation
2. Roasting – Water Quenching – Cyanidation
3. Roasting – Acid Leaching – Cyanidation
4. Roasting – Water Quenching – Acid Leaching – Cyanidation

Roasting conditions were the established optimum, and leaching conditions were pH of 1, liquid-to-solid ration of 4, at 90 °C for 4 hours. Results are shown in Table 13-22.

Table 13-22 Cyanidation Pre-Treatment Evaluation

No	Calcine Yield (%)	Acid Leaching Residue Rate (%)	Gold Grade (g/t)		Gold Leaching Rate (%)	Silver Grade (g/t)		Silver Leaching Rate (%)
			Raw Material	Cyanidation Residue		Raw Material	Cyanidation Residue	
1	69.73	-	3.96	1.98	65.14	80.10	58.10	49.42
2	69.73	-	3.96	1.88	66.90	80.10	59.60	48.12
3	69.73	95.34	3.96	1.82	69.45	80.10	43.00	64.31
4	69.73	94.08	3.96	1.72	71.51	80.10	43.90	64.05

Water quenching improves gold recovery by ~1% and acid leaching improves gold and silver recovery by ~5% and ~15% respectively. It is recommended to adopt the roasting-water quenching-acid leaching-cyanidation process for the pretreatment of pyrite concentrate.

13.1.6.4 Acid Leaching Condition Testing

Liquid-to-Solid Ratio Testing

The liquid-to-solid ratio during acid leaching is one of the critical factors governing the effectiveness of both acid leaching and subsequent cyanidation leaching processes. If the liquid-to-solid ratio is too low, the fluidity of the acid solution becomes poor, potentially leading to insufficient contact between the acid and the slurry, incomplete reactions, and low impurity removal efficiency. Conversely, if the liquid-to-solid ratio is too high, excessive acid solution may be used, increasing the costs associated with subsequent filtration and waste acid treatment. Therefore, it is necessary to determine the optimal acid leaching liquid-to-solid ratio experimentally.

Test conditions for leaching were pH 1 with sulfuric acid, 90 °C, 4 hour acid leach time with varying L/S ratios (4:1, 5:1, 6:1, and 7:1), followed by the established cyanide leaching conditions. Based on the test results a L/S ratio of 6:1 is recommended for acid leaching.

Temperature Condition Testing

Maintaining the previous acid leaching (with L/S ratio of 6:1) and cyanidation leaching test conditions and varying the acid leach temperature from 60 – 90 °C in 10 °C increments, indicated that the gold and silver leach extraction gradually increase with increasing acid leaching temperature. By increase the temperature from 60 to 90 °C the gold and silver leach rate increased by ~2% respectively.

Leach Time Testing

Maintaining the previous acid leaching (with 90 °C Temperature) and cyanidation leaching test conditions and varying acid leach time from 1 to 5 hours in 1 hours increments, indicated that the gold and silver leach extraction increase by ~2% respectively from 1 to 3 hours with no further improvement beyond that.

13.1.6.5 Cyanidation Condition Testing

A Bulk sample at the optimum roasting and acid leaching conditions were prepared.

Grind Optimization Testing

Cyanidation leach conditions were set at pH 11 with lime, 6 kg/t NaCN, 33%w/w pulp density, 48 hour leach time and varying the grind from unground (P₉₀ 80µm) to P₉₀ 38, 22, 17, and 15 µm. The gold extraction increased by ~3% between unground to P₉₀ 38µm with no further improvement in the finer grinds and the grind had no effect on the silver extraction rate.

Cyanide Dosing Optimization Testing

Cyanidation leach conditions were set at previous conditions (with P₉₀ at 38 µm) and varying the cyanide dosage rate between 1 to 5 kg/t in 1kg/t increments. Gold extraction was found to improve by ~2% from 1 to

3 kg/t NaCN with no further improvement at higher dosages and silver had a marginal increase in extraction rate from 1 to 2 kg/t NaCN with no increase beyond that.

Leach Time Optimization Testing

Cyanidation leach conditions were set at previous conditions (with 3 kg/t NaCN dosage rate) and varying the leach time from 12 to 48 hours in 12 hour increments. The optimum leach time was determined as 36 hours with ~72% gold dissolution and ~64% silver dissolution.

13.1.6.6 Verification Testing

Four tests were performed at the selected optimum conditions, and the repeatability of the results were confirmed.

From the results of the fluidized bed roasting test, in the small-scale fluidized bed roaster, the average calcine yield is 69.27%, the average removal rate of Pb is 11.83%, the average removal rate of S is 99.60%, and the average removal rate of As is 84.08%. The calcine yield and the removal rates of harmful elements are close to those of the muffle furnace roasting results.

From the cyanidation results, after the calcine roasted in the fluidized bed roaster undergoes water quenching, acid leaching, grinding and cyanidation, the average cyanide leaching rate of gold is 72.11% and that of silver is 63.48%, which are consistent with the results of muffle furnace roasting.

13.1.7 Hazen 2025 Testwork Program

In 2025 Hazen Research Inc. was contracted to perform roast leach extraction testwork to recover gold and silver from pyrite flotation concentrate with a summary below extracted from the ongoing testwork campaign.

Hazen has trialed varying temperatures, roasting gas compositions, additives, and additional grinding prior to leaching. The testwork campaign is still ongoing with gold recoveries ranging from 53 to 80% and silver recoveries ranging from 25 to 86%, preliminary observations as follow:

- Reducing roast with nitrogen feed gas did not improve metal recoveries.
- 6% Lime addition improved silver recovery by 23.5%.
- Increasing roasting temperature (600 to 800 °C) reduced gold and silver recoveries by ~50% and ~40% respectively. Sintering of the material is expected to be the cause of the sharp decrease in recoveries with 600 to 650 °C was chosen for follow on tests.
- Sodium hydroxide shows no improvement over lime with plugging observed and agglomerates formed in the calcine product.
- Crushing of the calcine (12 minutes in a ring and puck mill) prior to leaching provided inconsistent results with questionable PSD measurements of the material.
- Gold and silver balances were overall poor, 77 to 125% and 69 to 111% respectively, likely due to the "nugget" effect.

13.2 RESULTS INTERPRETATION AND CONCLUSIONS

This section describes how the results presented in Section 13.2 have been interpreted and used in the development of the process design criteria. At the time that the process design criteria were being developed, only preliminary results from the SGS 2022 and SGS 2024 locked-cycle testing campaigns were available, so these results were used with the intention that future testwork results would be confirmatory. The criteria developed from these results are outlined in Table 17-1 in Chapter 17. Orway Mineral Consultants, a subsidiary of Lycopodium, provided the design for the comminution circuit.

13.2.1 Mineralogy / Metallurgy

The head grades used in the process plant design were the calculated head grades from the 2022 locked cycle tests flotation tests conducted by SGS (detailed in Section 13.2.4). The selection of these grades allows for complete metallurgical accounting and direct alignment with reagents consumed during testwork and the recoveries observed.

13.2.2 Comminution Design

OMC used its testwork database to infer missing data and to recommend further testwork for subsequent studies. While both BWi values from testwork fell outside the database range for other similar resources in the area, the average BWi fell at the 81st percentile of the range, aligning well with OMC's typical strategy of designing to the 85th percentile hardness. As such, the average of both test results was used in the comminution circuit design (13.1 kWh/t). The remainder of the comminution circuit design criteria were estimated from OMC's database.

13.2.3 Flotation Circuit Design

The conditions tested during the 2022 and 2024 SGS locked cycle testing campaigns were used as the basis for the concentrate recovery circuit design, since the testing incorporated recycles at steady state and produced concentrates of acceptable grade and recovery (detailed in Section 13.1.3.5 and Section 13.1.4.3 respectively). Using locked cycle data provides a higher design certainty since it more closely represents plant operation by the incorporation of recycles and since it validates the process conditions at steady state.

The design criteria derived from the 2022 and 2024 locked cycle testing includes:

- General circuit design (i.e. orientation, number of stages, implementation of regrinding).
- Reagent addition.
- Concentrate mass pulls, grades and recoveries.
- Residence times.

The flotation recoveries for the various metals were applied to the head grades from the mine plan to estimate annual concentrate productions. The lead, zinc and sulphur grades from the locked-cycle tests were used to determine the production rates of lead, zinc and pyrite concentrates, respectively. The gold and silver grades of each concentrate was then calculated using the recovery of the metals to each concentrate, based on the locked-cycle tests.

13.2.4 Pyrite Concentrate Production

The correlation developed between the total sulphur and metals' content in the drill core samples (Equation 13.2.1, described in Section 13.1.5) was used to develop a model to estimate the life of mine pyrite concentrate produced, assuming constant concentrate grades and recoveries based on SGS locked cycle testing (2022 and 2024). This model was applied to the project financial model to estimate pyrite concentrate production based on the mine plan. The details of the financial model are further discussed in Chapter 22.

13.2.5 Pyrite Concentrate Treatment

Throughout recent testwork campaigns, several methods of oxidizing and leaching the pyrite concentrate were trialled to achieve the highest recoveries of gold and silver (while minimizing costs associated with reagent consumption, energy required, and residence time). Leaching tests to date have shown that the material is highly refractory and resistant to direct cyanidation for gold and silver recovery. A comparison of tested methods is provided in Table 13-23 (including only testwork conducted on fresh material, excluding the historic tailings).

Table 13-23 Comparison of Pyrite Treatment Methods

Treatment Process	Testwork Campaign	Process Conditions	Sulphide Oxidation %	Au Extraction ¹ %	Ag Extraction ¹ %	NaCN Added kg/t feed	Lime Added kg/t feed	NaCN Consumed kg/t feed	Lime Consumed kg/t feed
Direct Cyanidation (CIL)	SGS 2018	30 h pre-aeration, 40% pulp density, 48-h leach	N/A	19	58	12.3	31.3	9.96	31.3
		30 h pre-aeration, 40% pulp density, 48-h leach	N/A	32	70	12.1	29.8	7.99	29.7
	SGS 2019	D ₈₀ 4.47 µm, 40% pulp density	N/A	17.3	83.9	24.9	1.83	22.3	1.60
	SGS 2022	D ₈₀ 42 µm, 4-h pre-aeration (pH 10.51, 20 ppm DO, 250 g/t PbNO ₃), 72-h leach at pH 10.5-11	N/A	9.5	36.4	20.3	4.25	4.39	3.91
		D ₈₀ 6.3 µm, 4-h pre-aeration (pH 10.51, 20 ppm DO, 250 g/t PbNO ₃), 72-h leach at pH 10.5-11	N/A	22.4	80.2	28.5	15.4	14.5	15.3
	Mintek 2024	24-h leach, 25% solids, 20 g/L CN, DO>10.5, then 8-h CIL	N/A	29	40	-	-	2.11	39.1
POX-CIL	SGS 2018	5.6% pulp density, pre-acidulation (pH 1.25), 225°C, 690 kPa, 120 min	99	97.4	49.2	7.24	320.8	3.55	320.8
		5.6% pulp density, pre-acidulation (pH 1.39), 225°C, 690 kPa, 60 min, hot cure	99	97.6	1.2	6.56	16.8	4.45	16.8
		5.6% pulp density, pre-acidulation (pH 1.30), 225°C, 690 kPa, 60 min, hot cure and lime boil	-	98.2	97.6	2.11	0.6 + 59.2 kg/t for lime boil	0.90	0.6 + 59.2 kg/t for lime boil
	SGS 2019	5.5% pulp density, 30-min pre-acidulation (pH 1.50), 225°C, 690 kPa, 60 min, hot cure and lime boil	99.8	98.2	96.6	2.23	3.98	1.52	3.98 + 35.8 kg/t for lime boil
	SGS 2024	8% pulp density, 30-min pre-acidulation (pH 1.50), 225°C, 690 kPa, 60 min, hot cure and lime boil	96.5	96.9	93.4	2.52	0 + 46.1 for lime boil	0.58	0 + 46.1 for lime boil
Albion-CIL	SGS 2019	52-h, 95°C, 10% pulp density, pre-acidulation (pH 4.5), O ₂ sparge 1 L/min, limestone addition to maintain pH 5.5, 23 kg/t H ₂ SO ₄ and 440 kg/t CaCO ₃	28%	45.4	84.8	3.85	11.7	3.52	11.6
		72-h, 95°C, 10% pulp density, pre-acidulation (pH 4.5), O ₂ sparge 1.7 L/min, limestone addition to maintain pH 5.5, 21 kg/t H ₂ SO ₄ and 1,209 kg/t CaCO ₃	68%	81.0	92.3	5.46	6.79	5.11	6.79
	SGS 2022	96-h, 95°C, D ₈₀ 6.3 µm, 10% pulp density, limestone addition to maintain pH 5.5	96.3	94.4	94.4	7.92	2.48	3.22	2.40
		72-h, 95°C, D ₈₀ 10 µm, 10% pulp density, limestone addition to maintain pH 5.5	74.7	84.9	91.3	8.25	3.32	2.29	3.27

Treatment Process	Testwork Campaign	Process Conditions	Sulphide Oxidation %	Au Extraction ¹ %	Ag Extraction ¹ %	NaCN Added kg/t feed	Lime Added kg/t feed	NaCN Consumed kg/t feed	Lime Consumed kg/t feed
Roast-CIL	SGS 2018	Two-stage roast, 2 hours per roast, at 400°C and 700°C	49.6	62.5	9.1	2.82	1.16	1.02	1.16
		Two-stage roast, 2 hours per roast, at 400°C and 700°C, ground to <10 µm before CIL	-	84.8	72.5	4.17	4.29	3.91	4.27
	SGS 2022	Two-stage roast, 2 hours per roast, at 400°C and 700°C	99.9	61.1	19.9	21.9	12.0	6.54	12.0
		Two-stage roast, 2 hours per roast, at 400°C and 700°C. Pre-CIL re-grind to 7.4 µm	99.9	71.1	44.9	24.0	6.88	4.83	6.63
		Two-stage roast, 2 hours per roast, at 400°C and 600°C	97.6	55.6	51.3	24.1	N/A	13.3	N/A
		Two-stage roast, 2 hours per roast, at 400°C and 600°C. Pre-CIL re-grind to 6.7 µm	97.6	65.9	60.3	31.4	12.2	12.7	12.2
		Three-stage roast, 2 hours per roast, at 400°C and 600°C (stage 2 and 3). Aeration during 2 nd and 3 rd stage (1 L/min and 2 L/min)	99.8	59.3	32.5	27.3	8.56	10.3	8.56
		Three-stage roast, 2 hours per roast, at 400°C and 600°C (stage 2 and 3). Aeration during 2 nd and 3 rd stage (1 L/min and 2 L/min). Pre-CIL re-grind to 7.5 µm	99.8	66.7	51.6	27.3	5.93	5.10	5.54
		One-stage roast, 2 hours, at 500°C	89.3	52.5	51.8	29.1	34.7	13.6	34.7
		One-stage roast, 2 hours, at 500°C. Pre-CIL re-grind to 8.1 µm	89.3	63.9	61.8	37.2	14.6	22.5	14.6
		One-stage roast, 2 hours, at 600°C	95.1	56.4	29.6	30.5	13.5	13.9	13.5
		One-stage roast, 2 hours, at 600°C. Pre-CIL re-grind to 5.9 µm	95.1	66.7	50.4	26.8	9.3	11.8	9.1
	SGS 2024	Two-stage roast, 2 hours per roast, at 400°C and 700°C	99.9	73.0	11.3	5.84	5.97	0.62	5.97
		Two-stage roast, 2 hours per roast, at 400°C and 700°C. Pre-CIL re-grind to 4 µm	99.9	80.4	30.2	7.6	5.39	2.81	5.39
	BGRIMM 2025	Roasting: Mild oxidative roasting (530 °C, 2.5-h), Intensive oxidative roasting (630 °C, 2.5-h); Water Quenching; Acid Leaching: pH 1, 90 °C, L/S 6:1, 3-h; Cyanide Leaching: P ₉₀ 38 µm, 3 kg/t NaCN, 36-h, 33% pulp density	99.6	72.0	62.9	3	N/A	N/A	8.75 g/L (Acid Leach) 3.28 kg/t (NaCN Leach)
Hazen 2025	Best 3 Tests: Roast at 600 °C with 6% Lime and Air with 10%SO ₂ gas, crushed, leached, recrushed and re-leached.	N/A	77.5	84.0	N/A	N/A	25.7	9.1	
	Average Tests: Roast at 600 °C with 6% Lime and Air with 10%SO ₂ gas, crushed, leached, (excluding: recrushed and re-leached)	N/A	54.4	71.8	N/A	N/A	8.4	14.1	
BIOX-CIL	SGS 2022	24-h stabilization, 18-day residence, 10.5% pulp density, 40°C, air sparge, pH 1.6	60.6	75.2	75.5	17.7	9.01	7.91	8.77
		24-h stabilization, 26-day residence, 10.5% pulp density, 40°C, air sparge, pH 1.6	88.0	82.2	40.1	15.3	11.4	8.75	11.2
		24-h stabilization, 26-day residence, 10.5% pulp density, 40°C, air sparge, pH 1.6	92.8	85.1	37.0	23.6	16.2	23.4	16.2
MACH Reactor-CIL	Mintek 2024	6-h pre-oxidation, 4-h boosted leach, 72-h bottle roll cyanidation	16-18	40	70	-	-	36	273
		22-h pre-oxidation, 4-h boosted leach, 72-h bottle roll cyanidation	54	60	75	-	-	32	639

13.3 FUTURE TESTWORK RECOMMENDATIONS

The following testwork is recommended to progress the design to the next phase:

Sample Requirement:

- Spatial composites to represent different physical locations throughout the deposit
- Samples from different zones and depths for variability testing.
- Sufficient sample sizes to conduct downstream testwork.
- Number of representative samples to be determined by AGS.
- Sufficient sample size to generate representative concentrate samples for off-taker evaluation.

Comminution Testwork:

- Additional tests to determine:
 - Crushing work index (CWi)
 - SMC parameters: abrasivity (Ai) and impact breakage (A x b)
 - BWi (confirmatory)
- Variability of these parameters throughout the deposit and determine correlations to lithology.
- Flotation regrind tests to determine specific energy; engage with vendors for criteria relating to equipment sizing and specifications.

Ore Characterization Testwork:

- Assess material variability by assaying samples from different zones without blending according to mine plan and to create design ranges.
- Particle size distribution of ROM material and of final concentrates in support of dewatering testwork.
- Rheology testwork to determine apparent viscosities at varying shear rates and varying slurry % solids in support of slurry pipe and pump design.
- Physical characteristics impacting material handling, such as feed moisture content, specific gravity, bulk density, and stacking and clumping / compaction properties.

Flotation Testwork:

- Variability testing – open circuit and locked cycle to determine the impacts of feed variance and process upsets on grades and recoveries. Develop relations between open circuit and locked cycle results
- Produce enough of each concentrate to support dewatering testwork and further pyrite concentrate treatment testwork.

Dewatering Testwork:

- Particle settling testwork to determine the flux rate for thickener sizing.
- Flocculant screening tests.
- Filtration testwork to determine:
 - Filter type
 - Optimum feed pulp density
 - Achievable cake moisture levels
 - Transport moisture limit targets
 - Filter cycle times
 - If any filter aids are required.

Pyrite Concentrate Treatment Testwork:

- Continue development of pyrite oxidation option.
- Assess variability of pyrite concentrate via spatial composites and variability testing.
- Generate sufficient pyrite concentrate for off-taker evaluation.

Water Analysis:

- Surface water quality analysis.
- Ground water quality analysis.

14.0 MINERAL RESOURCE ESTIMATES

14.1 INTRODUCTION

This section presents the updated MRE for the Boumadine Deposit, incorporating new drilling data and geological modeling refinements. It updates the previous MRE dated April 16, 2024. The Mineral Resource Estimate has been prepared in accordance with the Canadian Securities Administrators' National Instrument 43-101 and Form 43-101F1, following industry best practices outlined in the "CIM Estimation of Mineral Resources and Mineral Reserves Best Practices" guidelines. Mineral Resources have been classified in compliance with the CIM Standards on Mineral Resources and Reserves: Definition (2014) and Best Practices (2019), as adopted by the CIM Council.

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. Although the Authors are not experts with respect to environmental permitting, legal, title, taxation, socio-economic, marketing, or political matters, they are not aware of any unusual factors relating to these matters that may materially affect the estimated Mineral Resources as of the effective date of this Report.

The Authors consider the block model used for the Mineral Resource Estimates and Mineral Resource classification, which was developed by Aya, to represent a reasonable estimation of the Mineral Resources for the Boumadine Deposit with regard to compliance with generally accepted industry standards and guidelines, the methodology used for grade estimation, the classification criteria used and the actual implementation of the methodology in terms of Mineral Resource estimation and reporting. The Mineral Resources have been estimated in conformity with the requirements of the CIM "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines as required by the Canadian Securities Administrators' National Instrument 43-101. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

Wireframe modelling was developed by Aya using Seequent Leapfrog Geo™ software. Mineral Resource estimation and variography were completed by Aya using Isatis.neo™ software. Open-pit optimization was developed by Aya using the GEOVIA Whittle software.

14.2 DATA USED

The resource estimate is based on a dataset of 428 drill holes, totaling 142,268 m. This updated MRE incorporates an additional 93 drill holes, contributing 44,514 m of new drilling data. The drilling extends approximately 6.0 km along strike (Figure 14-1 and Appendix A). The database was closed on January 31, 2025.

Drill hole data was exported from Geotic software in .csv format and included tables for collar, survey, assay, lithology, and bulk density. Assay data comprise gold (Au g/t), silver (Ag g/t), copper (Cu%), lead (Pb%), and zinc (Zn%) grades.

The coordinate system used is WGS 84 UTM Zone 30N (EPSG 32630). A Digital Terrain Model surveyed by drone was used as a reference surface for the project. Veins were extracted as .msh files from Leapfrog Seequent software.

Industry standard validation checks were carried out on the database, and minor corrections were made where necessary. The Authors typically validate 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 errors were noted within the database. The Authors consider the drill hole database to be suitable for Mineral Resource estimation.

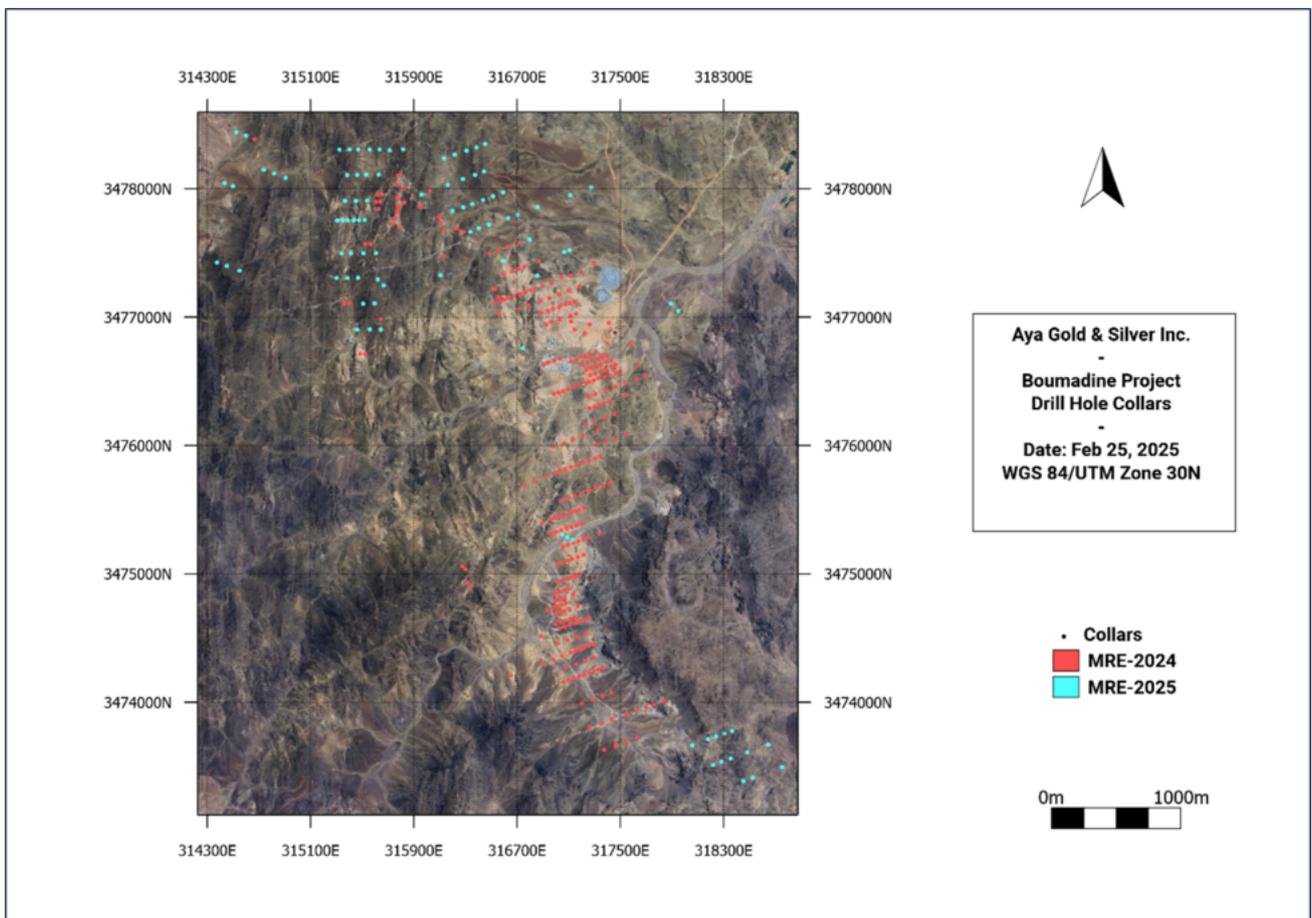


Figure 14-1 Drill Hole Plan View

Source: This Report

14.3 ECONOMIC CONSIDERATIONS

For the Mineral Resource model, Aya selected the economic parameters listed in Table 14-1.

Table 14-1 Economic Parameters

NSR ASSESSMENT - BOUMADINE DEPOSIT Feb 25					
Zn Concentrate	Metal Price	Concentrate	Smelter	Refining Chg.	Average Grade
Element	\$US/lb or oz	Recovery	Payable	\$US/lb or oz	% or g/t
Pb	\$1.00	2.17%	0%	\$0.00	1.00%
Zn	\$1.20	72.00%	85%	\$0.00	1.00%
Au	\$2,200	1.73%	0%	\$4.00	1.00
Ag	\$24.00	5.54%	0%	\$4.00	1.00
Concentration Ratio		31.04	3.22%		
Trucking/Storage/Ship Loading US \$/t per WMT		\$50			
Ocean Freight /Unloading \$US/t per WMT		\$0			
Smelter Treatment Charge US \$/t per DMT incl \$6 Penalty		\$156			
Humidity Factor		8.0%			
	NSR				
Element	US \$ Val/tonne				
Pb	\$0.00				
Zn	\$16.19				
Au	\$0.00				
Ag	\$0.00				
Subtotal	\$16.19				
Less Smelter Charges US \$/t	\$5.03				
Less Shipping & Handling Charges US \$/t	\$1.74				
Subtotal NSR US \$/tonne of milled from Zn Concentrate	\$9.43				
Pb Concentrate	Metal Price	Concentrate	Smelter	Refining Chg.	Average Grade
Element	\$US/lb or oz	Recovery	Payable	\$US/lb or oz	% or g/t
Pb	\$1.00	84.50%	95%	\$0.00	1.00%
Cu	\$4.00	75.30%	95%	\$0.00	1.00%
Zn	\$1.20	3.93%	0%	\$0.00	1.00%
Au	\$2,200	26.50%	95%	\$30.00	1.00
Ag	\$24.00	53.10%	95%	\$1.50	1.00
Concentration Ratio		32.91	3.04%		
Trucking/Storage/Ship Loading US \$/t per WMT		\$50			
Ocean Freight /Unloading \$US/t per WMT		\$0			

Table 14-2 Calculation Formulas for NSR, Silver Equivalent, and Gold Equivalent

	Au (oz)	Ag (oz)	Zn (lb)	Pb (lb)	Cu (lb)
Prices in \$USD	\$2,200	\$24	\$1.20	\$1.00	\$4.00
Recovery in %	85.20%	89.10%	72.00%	84.50%	75.30%
NSR (\$/t)	$(Pb\% \times \$10.74) + (Zn\% \times \$13.59) + (Au \text{ g/t} \times \$50.89) + (Ag \text{ g/t} \times \$0.56) + (Cu\% \times 63.08)$				
Ag Equivalent (g/t)	$Ag(g/t) + (Au(g/t) \times Au \text{ price/oz} \times Au \text{ recovery}) / (Ag \text{ price/oz} \times Ag \text{ recovery}) + Zn(\%) \times Zn \text{ price/lb} \times Zn \text{ recovery} / (Ag \text{ price/oz} \times Ag \text{ recovery}) \times 685.7147973 + Pb(\%) \times Pb \text{ price/lb} \times Pb \text{ recovery} / (Ag \text{ price/oz} \times Ag \text{ recovery}) \times 685.7147973 + Cu(\%) \times Cu \text{ price/lb} \times Cu \text{ recovery} / (Ag \text{ price/oz} \times Ag \text{ recovery}) \times 685.7147973$				

Table 14-3 Calculation Formulas for NSR, Ag Equivalent, and Au Equivalent

	Au (oz)	Ag (oz)	Zn (lb)	Pb (lb)	Cu (lb)
Prices in \$USD	\$2,200	\$24	\$1.20	\$1.00	\$4.00
Recovery in %	85.20%	89.10%	72.00%	84.50%	75.30%
NSR (\$/t)	$(Pb\% \times \$10.74) + (Zn\% \times \$13.59) + (Au \text{ g/t} \times \$50.89) + (Ag \text{ g/t} \times \$0.56) + (Cu\% \times 63.08)$				
Ag Equivalent (g/t)	$Ag(g/t) + (Au(g/t) \times Au \text{ price/oz} \times Au \text{ recovery}) / (Ag \text{ price/oz} \times Ag \text{ recovery}) + Zn(\%) \times Zn \text{ price/lb} \times Zn \text{ recovery} / (Ag \text{ price/oz} \times Ag \text{ recovery}) \times 685.7147973 + Pb(\%) \times Pb \text{ price/lb} \times Pb \text{ recovery} / (Ag \text{ price/oz} \times Ag \text{ recovery}) \times 685.7147973 + Cu(\%) \times Cu \text{ price/lb} \times Cu \text{ recovery} / (Ag \text{ price/oz} \times Ag \text{ recovery}) \times 685.7147973$				
Au Equivalent (g/t)	$Au(g/t) + (Ag(g/t) \times Ag \text{ price/oz} \times Ag \text{ recovery}) / (Au \text{ price/oz} \times Au \text{ recovery}) + Zn(\%) \times Zn \text{ price/lb} \times Zn \text{ recovery} / (Au \text{ price/oz} \times Au \text{ recovery}) \times 685.7147973 + Pb(\%) \times Pb \text{ price/lb} \times Pb \text{ recovery} / (Au \text{ price/oz} \times Au \text{ recovery}) \times 685.7147973 + Cu(\%) \times Cu \text{ price/lb} \times Cu \text{ recovery} / (Au \text{ price/oz} \times Au \text{ recovery}) \times 685.7147973$				

14.4 MINERALIZED DOMAINS

Interpreted mineralized wireframes (domains) were based on logged drill holes, lithology and assay grades. Aya identified continuous zones of mineralization with a massive sulphide percentage of 70% or greater, and zones where the assay grades were ≥ 100 g/t AgEq and the massive sulphide percentage was $\geq 30\%$. The selected intervals include lower-grade mineralization where necessary to maintain continuity between drill holes. Three-dimensional wireframes linking drill hole cross-sections were constructed using the Leapfrog™ Radial Basis Function, with hanging wall and footwall surfaces snapped directly to the selected drill hole intercepts.

A total of 45 individual mineralized veins were developed (Figure 14.2 and Appendix B). The mineralized vein wireframes were used to back-tag the assay, bulk density and composite tables with unique category codes (Table 14-4).

Table 14-4 Mineralized Veins

Vein Code - North Area	Vein Code - Central Area	Vein Code - South Area
North - Tizi Main	Central - Main	South - Main
North - Tizi West	Central - East 1	South - East 1
North - Tizi East	Central - West 1	South - West 1
North - Tizi East 0	Central - West 2	South - West 2
North - Tizi West 0	Central - East 0	South - East 0
North - Tizi West 001	Central - East 3	South - East 3
North - Tizi West 01	Central - East 2	South - East 2
North - Tizi Imariren	Central - FarFarEast	South - East 4
North - Tizi Imariren 2	Central - West 3	South - FarFarEast
North - Tizi Imariren 3	Central - West 4	South - West 3
North - Main	Central - FarWest	South - West 4
North - East 1	Central - East 33	South - FarWest
North - West 1	Central - Nord West	South -East 33
North - West 2		
North - East 2		
North - Imariren		
North - Nord West		
North - North West 0		
North North West 01		

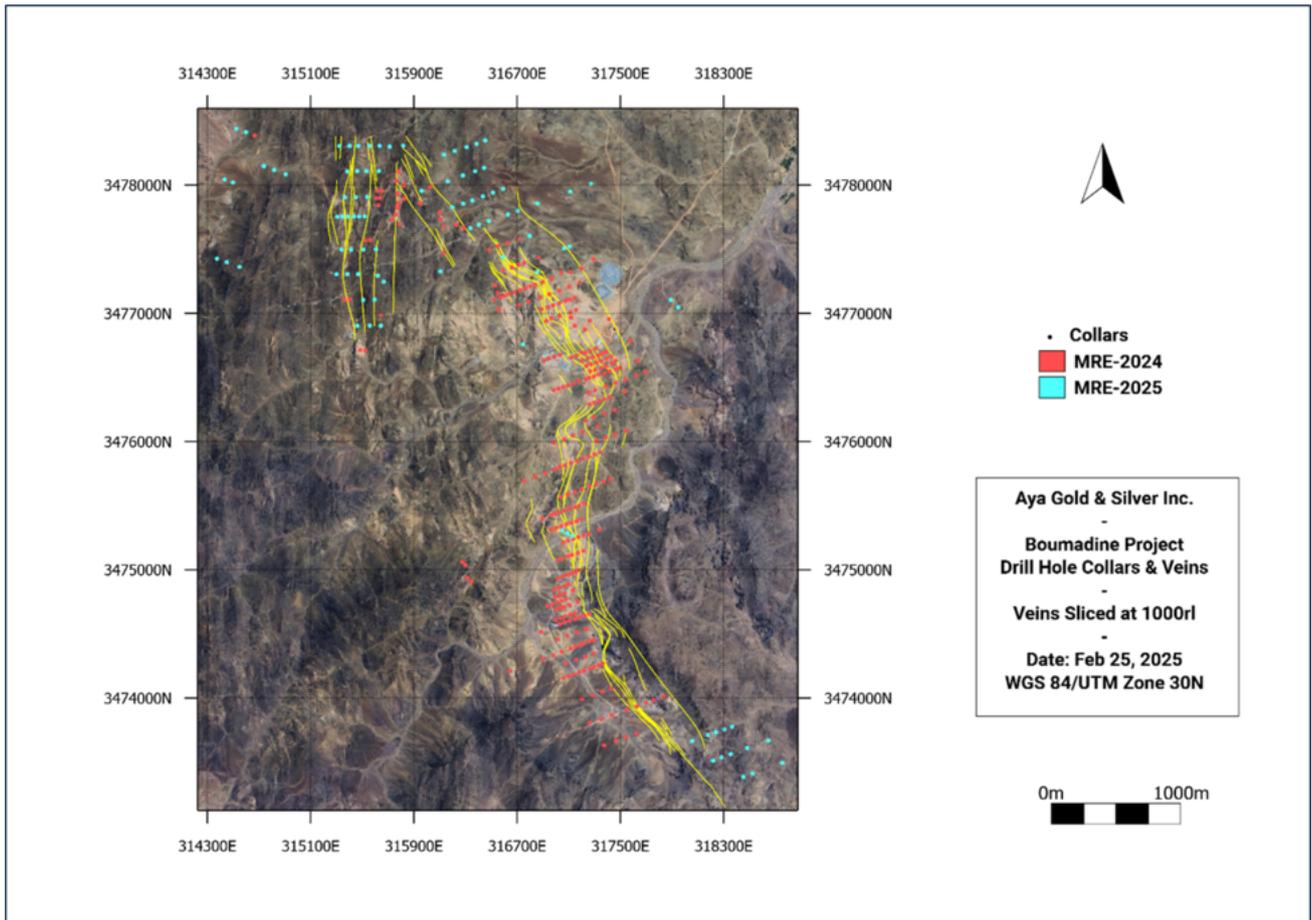


Figure 14-2 Mineralized Veins

Source: This Report

14.5 EXPLORATORY DATA ANALYSIS

The average nearest neighbor drill hole collar distance for the Boumadine drilling is 48 m. The average length of the drill holes is 328 m. Summary statistics for the drill hole assay data constrained by the modelled domains are listed in Table 14-5.

Table 14-5 Constrained assays summary statistics

Assay	Count	Average	St Dev	CoV	Minimum	Maximum
Ag (g/t)	3,828	56.11	171.51	3.057	0.81	7,720.00
Au (g/t)	3,821	1.50	3.12	2.082	0.05	74.48
Cu (%)	3,816	0.07	0.18	2.670	0.01	3.22
Pb (%)	3,816	0.67	1.40	2.099	0.01	24.40
Zn (%)	3,814	1.89	3.50	1.847	0.01	39.71

Note: St Dev = standard deviation, CoV = coefficient of variation.

Bulk density measurements were determined by Aya using the water immersion method on drill core samples. The average bulk density for the constrained sulphide material is 3.72 t/m³.

For the Mineral Resource Estimate a bulk density of 2.61 t/m³ was assigned to oxide and transitional blocks. For sulphide blocks the median sulphide bulk density was assigned for each modelled domain. Minor veins with insufficient density measurements were combined into spatial sub-domains (Table 14-6).

Table 14-6 Summary Statistics for bulk density measurements

Domain	Count	Average Bulk Density (t/m ³)	Median Bulk Density (t/m ³)
Major Vein			
1_Main	314	3.87	3.97
1_West1	124	3.64	3.67
1_West2	91	3.59	3.49
1_West3	45	3.64	3.72
1_East0	45	3.66	3.71
1_East1	147	3.64	3.68
1_East2	48	3.74	3.90
1_East3	88	3.49	3.47
3_Tizi Main	33	3.90	3.93
Eastern Veins			
FarFar East			
East4	44	3.60	3.74
East33			
Western Veins			
West4			
FarWest	27	3.62	3.64
Vein between Tizi-Imariren			
Between Tizi Imariren			
Between Tizi Imariren 2	17	3.65	3.76
Between Tizi Imariren 3			
Imariren -Vein System			
Imariren			
North West			
North West 01	51	3.93	4.11
North West 0			
Tizi Other			
Tizi_East			
Tizi_East 0			
Tizi West 001			
Tizi West	40	3.90	3.91
Tizi West 01			
Tizi West 0			
Total			
Total	1114	3.72	3.78

14.6 COMPOSITING

Constrained assay sample lengths within the defined mineralized domains range from 0.40 to 1.30 m, with an average sample length of 0.85 m and a mode of 1.00 m. A total of 31% of the constrained assay sample lengths equal 1.00 m.

Based on the distribution of assay sample lengths, Aya chose to composite all constrained assay samples to 1.00 m in order to ensure equal length sample support. Length-weighted composites were calculated within the defined mineralized veins. A small number of unsampled intervals in the data were treated as null intervals.

The compositing process started at the first point of intersection between the drill hole and the mineralized vein intersected and was halted on exit from the mineralization. Downhole residual composites that were less than half the compositing length were merged with the preceding interval. The wireframes that represent the mineralized veins were used to back-tag a rock code attribute into the composite workspace.

The composite data were visually validated against the mineralized wireframes and extracted for analysis and grade estimation. Summary composite statistics are listed in Table 14-7.

Table 14-7 Composite Summary Statistics

Assay	Count	Average	St Dev	CoV	Minimum	Maximum
Ag (g/t)	3,730	52.75	165.33	3.134	0.81	7,720.00
Au (g/t)	3,730	1.38	2.72	1.978	0.05	60.28
Cu (%)	3,730	0.06	0.17	2.639	0.01	3.22
Pb (%)	3,730	0.64	1.27	1.986	0.01	19.86
Zn (%)	3,730	1.80	3.13	1.741	0.01	39.71

Note: St Dev = standard deviation, CoV = coefficient of variation.

14.7 TREATMENT OF EXTREME VALUES

Aya selected capping thresholds based on the low Coefficient of Variation of the constrained composite grades and analysis of individual composite log-probability distributions for the composite sample populations (Figure 14-3). Composites were capped to the defined threshold prior to grade estimation (Table 14-8).

Table 14-8 Composite Capping Thresholds

Veins	Capping Threshold (g/t)	Average Uncapped Value	Number Capped	Capping Percentile	Average Capped Value
Ag (g/t)	800	52.75	11	99.7	49.73
Au (g/t)	30	1.38	2	99.9	1.36
Cu (%)	1.4	0.06	11	99.7	0.06
Pb (%)	10	0.64	9	99.7	0.63
Zn (%)	28	1.80	3	99.9	1.79

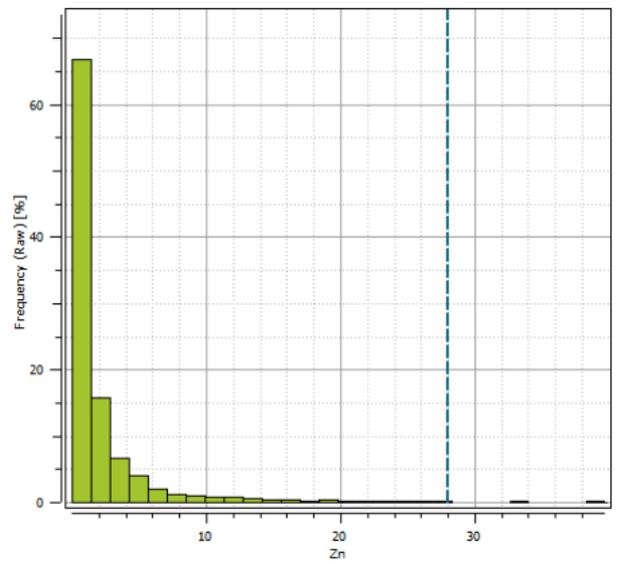
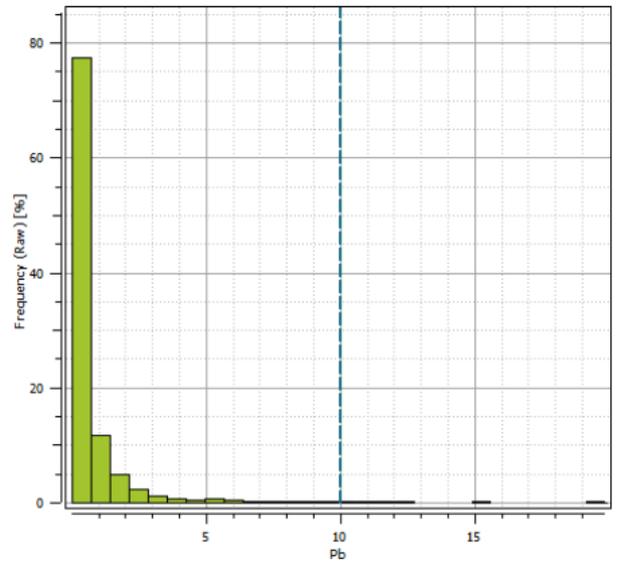
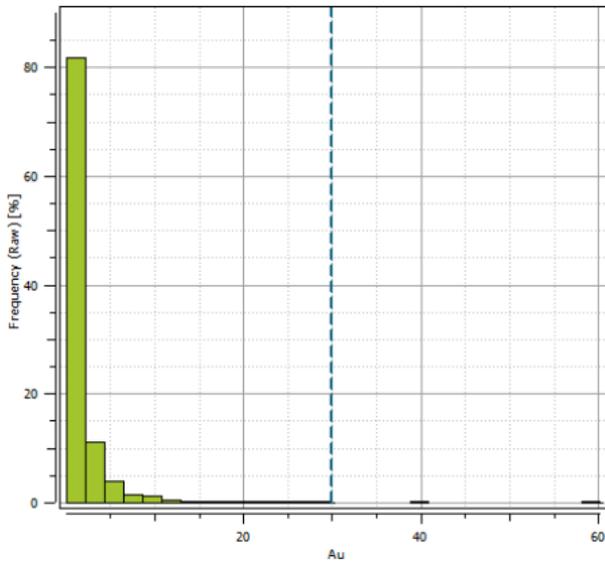
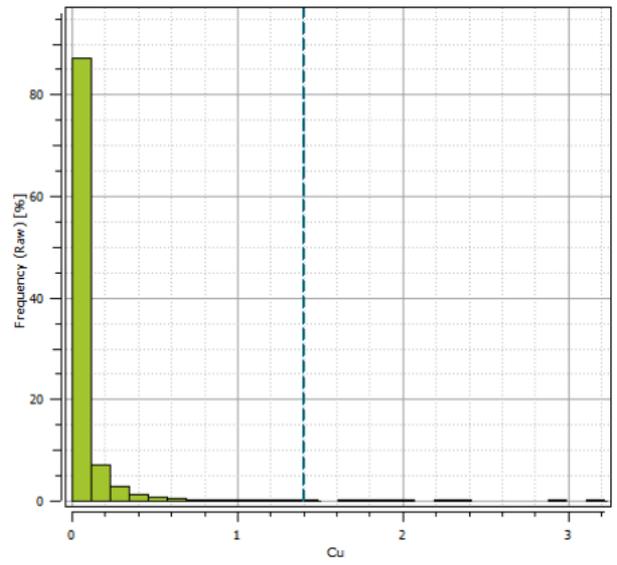
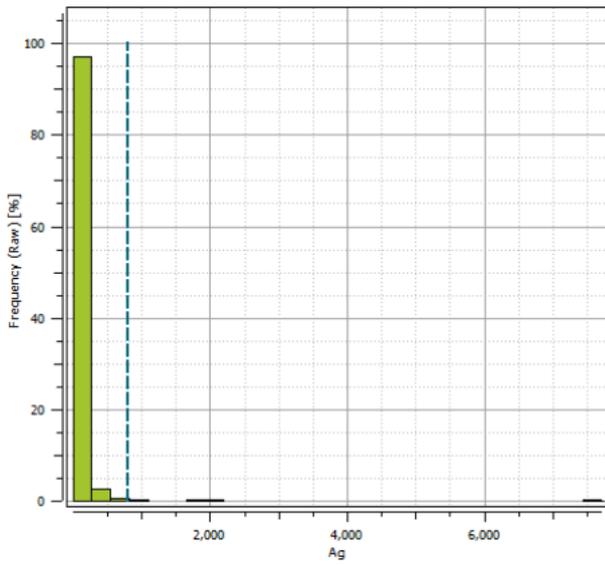


Figure 14-3 Histogram for Composites

Source: This Report

14.8 CONTINUITY ANALYSIS

Three-dimensional continuity analyses (variography) were conducted by Aya on domain-coded uncapped composite data. In general, an acceptable semi-variogram could only be developed for the Main domain. These semi-variograms were used to guide the search ellipse parameters for inverse distance estimation. Standardized spherical models were used to model the experimental semi-variograms (Figure 14-4).

The experimental semi-variogram for the northern Main is listed in Table 14-9.

Table 14-9 North Main Zone Semi-Variogram Results

North Main	Dip (°)	Dip Azimuth	Pitch	Ranges UVW (m)			Sill
Nugget							0.1
Spherical	75°	60°	30°	120	40	5	0.3032
Spherical	75°	60°	30°	215	75	9	0.5941

The experimental semi-variogram for the southern Main is listed in Table 14-10.

Table 14-10 South Main Zone Semi-Variogram Results

South Main	Dip (°)	Dip Azimuth	Pitch	Ranges UVW (m)			Sill
Nugget							0.1
Spherical	75°	60°	30°	55	40	3	0.3282
Spherical	75°	60°	30°	230	160	12	0.5702

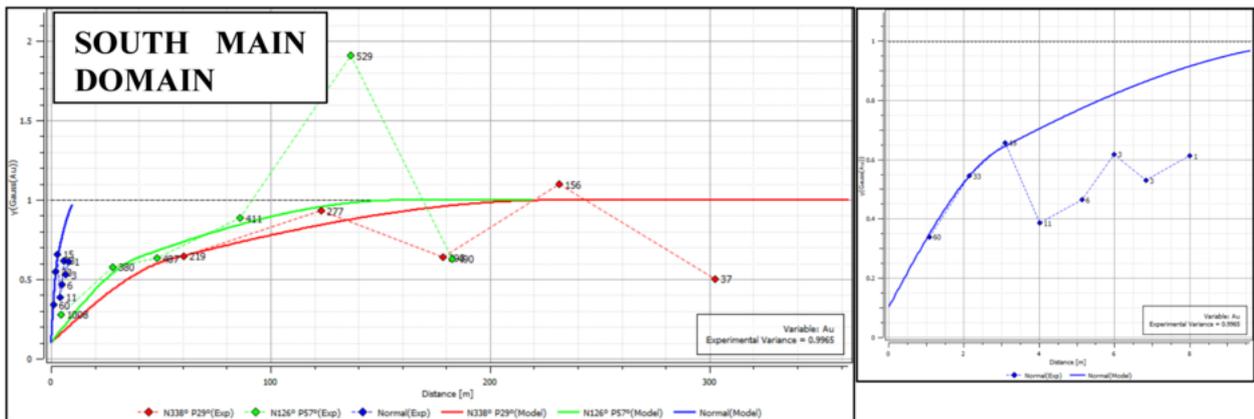
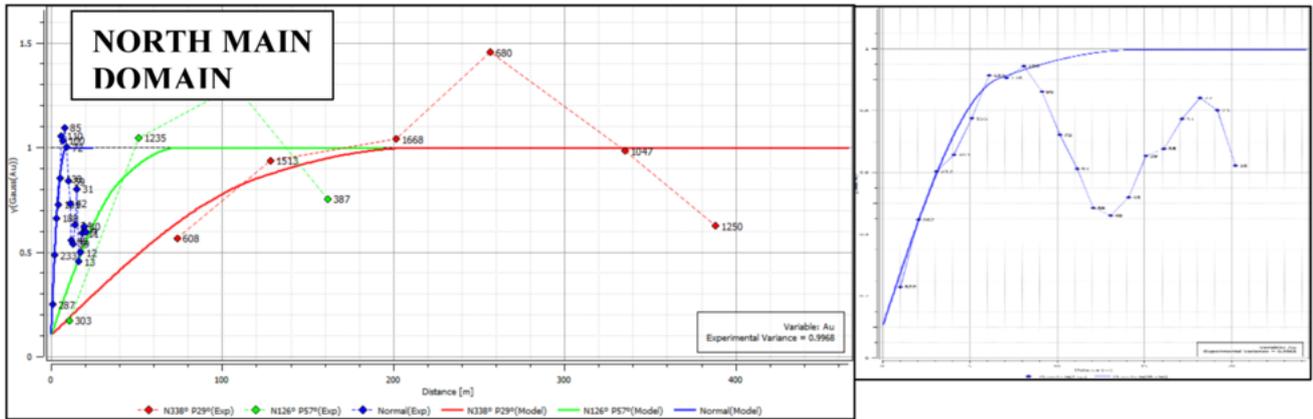


Figure 14-4 Main Zone Semi-Variograms

Source: This Report

14.9 BLOCK MODEL

Aya developed a rotated block model with the block model limits selected, in order to cover the extent of the mineralized structures, the potential open pit dimensions, and to reflect the general nature of the mineralized zones (Table 14-11). The block model consists of estimated Ag, Au, Cu, Pb and Zn grades, bulk density, block volume inclusion percentage and classification criteria. NSR, Au and Ag equivalent block values were subsequently calculated from the estimated grades of Au, Ag, Cu, Pb and Zn.

Table 14-11 Block Model Setup

Direction	Origin (Corner)	Number of Blocks	Block Size (m)
Minimum X	316,550	880	2.5
Minimum Y	3,472,800	1,150	5.0
Minimum Z	460	174	5.0
Rotation	20° counter-clockwise		

14.10 GRADE ESTIMATION AND MINERAL RESOURCE CLASSIFICATION

Mineral Resource grade estimation and variography were undertaken by using Isatis.neo™ software.

Aya assigned bulk density values for each domain based on the median bulk density values by domain. A bulk density of 2.61 t/m³ was assigned to oxide and transition blocks.

Block grades were estimated from capped composites with two passes. The orientation of the search ellipsoid was defined by the orientation of the modelled domains and observed grade trends. Composite samples were selected within an oriented 200 m x 130 m x 60 m ellipsoid for the first pass and a 400 m x 260 m x 120 m ellipsoid for the second pass. Search and grade estimation were constrained by the individual mineralized veins, which define hard boundaries for grade estimation.

Inverse Distance Squared (ID²) estimation was used for all domains. For the estimate, a minimum of four and a maximum of twelve composites were used in the first pass, while the second pass applied a minimum of four and a maximum of eight composites. Overall, approximately 40% of the modeled domains were estimated for grade.

Block grades were estimated separately for each domain and subsequently combined in one model space using the domain block inclusion percentages (see Appendices C, D, E and F).

The parameters used to define the classification limits included spatial analysis, drill hole spacing, and the observed continuity of the mineralization. Mineral Resources were classified algorithmically based on the local drill hole spacing within each individual mineralized domain. Grade blocks estimated in the first pass that used a minimum of two drill holes and an average distance between composites of <50 m were classified as Indicated, and all additional estimated blocks were classified as Inferred. Classification block model cross-sections and plans are presented in Appendix G.

14.11 MINERAL RESOURCE ESTIMATE

Open-pit Mineral Resources reported herein have been constrained within an optimized pit shell. The results from the optimized pit shell are used solely for the purpose of reporting Mineral Resources and include Indicated and Inferred Mineral Resources. Historical mining is minimal, poorly defined, and has not been depleted from the Mineral Resource Estimate.

The constraining pit optimization parameters were 50° pit slopes, US\$3.5/t for mineralized material mining, US\$2/t for waste mining, US\$89/t for processing, and US\$6/t for G&A, totalling US\$95/t for the NSR cut-off value (Figure 14.5). The optimized pit shell is shown in Appendix H.

The out-of-pit parameters used a US\$30/t mining cost, US\$89/t processing cost, and US\$6/t G&A totalling US\$125/t. The out-of-pit Mineral Resource grade blocks were quantified above the US\$125/t NSR cut-off value, below the constraining pit shell, and within the constraining mineralized wireframes. Out-of-pit Mineral Resources have been reported if they exhibit continuity and reasonable potential for extraction by the longhole mining method.

Highlights of the Boumadine Mineral Resource Estimate include:

- The Mineral Resource is amenable to conventional open-pit and underground mining methods;
- An Indicated Mineral Resource of 5.2 Mt grading 91 g/t Ag, 2.78 g/t Au, 2.8% Zn and 0.85% Pb containing an estimated 15.1 Moz of Ag, 462 koz of Au, 145 kt of Zn and 44 kt of Pb;
- An Inferred Mineral Resource of 29.2 Mt grading 82 g/t Ag, 2.63 g/t Au, 2.11% Zn and 0.82% Pb containing an estimated 76.8 Moz of Ag, 2.5 Moz of Au, 615 kt of Zn and 237 kt of Pb; and
- 49% of the Inferred Mineral Resource is pit-constrained above a US\$95/t NSR cut-off value, with the remaining 51% suitable for underground development and reported above an NSR cut-off value of US\$125/t.

The Mineral Resource has an effective date of February 24, 2025 (Table 14-12).

The sensitivity of the Mineral Resource to changes in cut-off grade was also calculated across a range of potentially economic NSR cut-offs (Table 14-13).

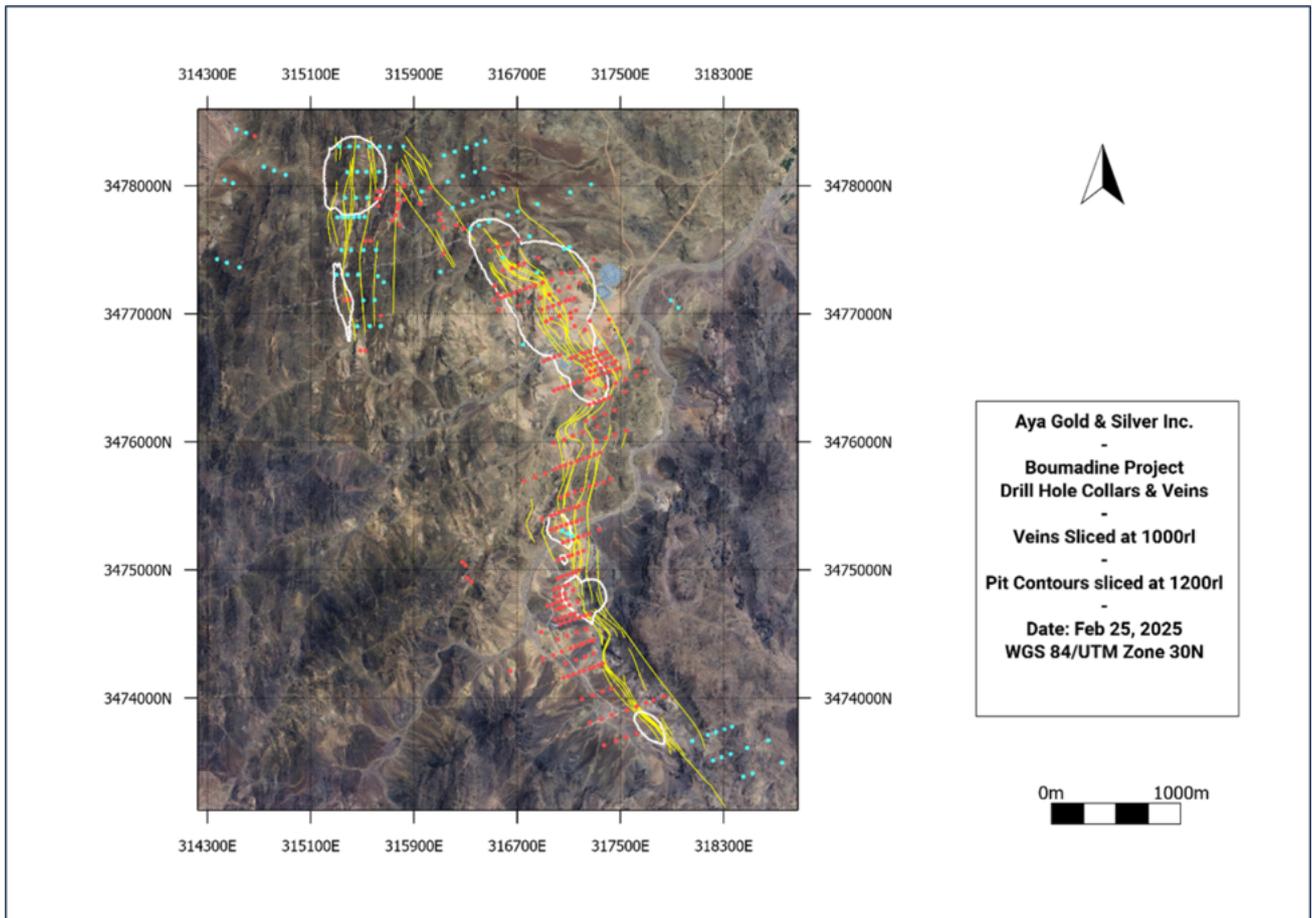


Figure 14-5 Boumadine Pit Shell Plan View

Source: This Report

Table 14-12 Boumadine MRE as of February 24, 2025

Class	Cutoff	Tonnes	Average Grade							Contained Metal						
			Ag	Au	Cu	Pb	Zn	AgEq	AuEq	Ag	Au	Cu	Pb	Zn	AgEq	AuEq
	NSR US\$/t	(kt)	(g/t)	(g/t)	(%)	(%)	(%)	(g/t)	(g/t)	(koz)	(koz)	(kt)	(kt)	(kt)	(koz)	(koz)
Pit-Constrained																
Indicated	95	3,920	94.3	2.99	0.13	0.84	2.95	476.5	5.3	11,881	377	5.1	33	116	60,051	667
Inferred	95	14,258	89.7	2.89	0.10	0.81	2.38	450.0	5.0	41,135	1,325	14.3	115	339	206,293	2,293
Out-of-Pit																
Indicated	125	1,249	80.1	2.11	0.08	0.87	2.32	358.2	3.98	3,216	85	1.0	11	29	14,382	160
Inferred	125	14,938	74.3	2.39	0.07	0.82	1.85	356.9	3.97	35,669	1,148	10.5	122	276	171,393	1,905
Total																
Indicated	95/125	5,169	90.8	2.78	0.12	0.85	2.80	447.9	4.981	15,097	462	6.1	44	145	74,433	827
Inferred	95/125	29,196	81.8	2.63	0.08	0.82	2.11	402.4	4.473	76,804	2,473	24.8	237	615	377,686	4,198

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. Mineral Resources 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
4. Individual calculations in tables and totals may not sum due to rounding of original numbers.

Table 14-13 Mineral Resource Estimate Sensitivity

Indicated Out-of-Pit Resources												
Cutoff	Tonnes	Ag	Ag	Au	Au	Cu	Pb	Zn	AgEq	AgEq	AuEq	AuEq
NSR US\$/t	(kt)	(g/t)	(koz)	(g/t)	(koz)	(%)	(%)	(%)	(g/t)	(koz)	(g/t)	(koz)
145	1,014	82	2,679	2.48	81	0.07	0.90	1.99	387	12,621	4.38	143
140	1,064	80	2,732	2.42	83	0.07	0.88	1.94	380	13,012	4.27	146
135	1,132	78	2,847	2.36	86	0.07	0.86	1.92	372	13,524	4.18	152
130	1,177	76	2,890	2.29	87	0.07	0.84	1.90	366	13,858	4.08	154
125	1,249	74	2,982	2.22	89	0.07	0.82	1.87	358	14,382	3.97	159
120	1,311	72	3,036	2.16	91	0.06	0.80	1.83	352	14,819	3.86	163
115	1,415	70	3,199	2.11	96	0.06	0.79	1.79	341	15,500	3.77	171
110	1,479	69	3,278	2.06	98	0.06	0.78	1.77	334	15,906	3.69	176
105	1,559	67	3,380	1.99	100	0.06	0.76	1.73	327	16,391	3.59	180
90	1,771	62	3,550	1.81	103	0.06	0.72	1.67	309	17,577	3.33	189

Indicated In-Pit Resources												
Cutoff	Tonnes	Ag	Ag	Au	Au	Cu	Pb	Zn	AgEq	AgEq	AuEq	AuEq
NSR US\$/t	(kt)	(g/t)	(koz)	(g/t)	(koz)	(%)	(%)	(%)	(g/t)	(koz)	(g/t)	(koz)
120	3,458	101	11,244	3.22	358	0.14	0.85	2.97	513	57,011	5.70	634
115	3,561	99	11,378	3.15	361	0.14	0.85	2.95	504	57,739	5.61	642
110	3,659	98	11,512	3.09	364	0.13	0.85	2.92	496	58,409	5.52	649
105	3,756	97	11,657	3.04	366	0.13	0.85	2.90	489	59,040	5.43	656
95	3,920	94	11,881	2.94	370	0.13	0.84	2.87	476	60,051	5.30	667
90	3,987	93	11,972	2.90	372	0.13	0.84	2.86	471	60,431	5.24	672
85	4,066	92	12,067	2.86	373	0.12	0.83	2.84	466	60,864	5.17	677
80	4,168	91	12,199	2.80	375	0.12	0.82	2.83	458	61,414	5.09	683
75	4,261	90	12,303	2.75	376	0.12	0.82	2.82	452	61,877	5.02	688
60	4,513	86	12,511	2.62	380	0.11	0.80	2.77	434	62,994	4.83	700

Inferred Out-of-Pit Resources												
Cutoff	Tonnes	Ag	Ag	Au	Au	Cu	Pb	Zn	AgEq	AgEq	AuEq	AuEq
NSR US\$/t	(kt)	(g/t)	(koz)	(g/t)	(koz)	(%)	(%)	(%)	(g/t)	(koz)	(g/t)	(koz)
145	11,463	82	30,296	2.48	915	0.07	0.90	1.99	394	145,297	4.38	1,615
140	12,267	80	31,485	2.42	956	0.07	0.88	1.94	384	151,566	4.27	1,685
135	13,028	78	32,773	2.36	989	0.07	0.86	1.92	376	157,428	4.18	1,750
130	13,921	76	34,194	2.29	1,026	0.07	0.84	1.90	367	164,099	4.08	1,824
125	14,938	74	35,669	2.22	1,068	0.07	0.82	1.87	357	171,393	3.97	1,905
120	16,000	72	37,062	2.16	1,112	0.06	0.80	1.83	347	178,572	3.86	1,985
115	16,962	70	38,355	2.11	1,149	0.06	0.79	1.79	339	184,898	3.77	2,055
110	17,810	69	39,466	2.06	1,179	0.06	0.78	1.77	332	190,290	3.69	2,115
105	18,980	67	41,154	1.99	1,215	0.06	0.76	1.73	323	197,363	3.59	2,194
90	22,662	62	45,428	1.81	1,322	0.06	0.72	1.67	299	218,136	3.33	2,425

Inferred In-Pit Resources												
Cutoff	Tonnes	Ag	Ag	Au	Au	Cu	Pb	Zn	AgEq	AgEq	AuEq	AuEq
NSR US\$/t	(kt)	(g/t)	(koz)	(g/t)	(koz)	(%)	(%)	(%)	(g/t)	(koz)	(g/t)	(koz)
120	12,560	97	39,047	3.21	1,295	0.10	0.84	2.29	484	195,345	5.38	2,171
115	12,861	95	39,452	3.16	1,305	0.10	0.84	2.28	478	197,476	5.31	2,195
110	13,191	94	39,854	3.10	1,315	0.10	0.83	2.28	471	199,726	5.23	2,220
105	13,617	92	40,343	3.03	1,329	0.10	0.82	2.27	462	202,434	5.14	2,250
95	14,258	90	41,135	2.93	1,345	0.10	0.81	2.25	450	206,293	5.00	2,293
90	14,517	89	41,432	2.89	1,351	0.10	0.81	2.25	445	207,784	4.95	2,310
85	14,819	88	41,744	2.85	1,357	0.09	0.81	2.25	440	209,447	4.89	2,328
80	15,381	86	42,418	2.76	1,364	0.09	0.80	2.26	430	212,552	4.78	2,363
75	15,716	85	42,778	2.71	1,369	0.09	0.80	2.26	424	214,252	4.71	2,381
60	16,798	81	43,684	2.56	1,384	0.09	0.78	2.24	406	219,083	4.51	2,435

14.12 VALIDATION

The block model was validated visually by the inspection of successive cross-section lines, in order to confirm that the block models correctly reflect the distribution of high-grade and low-grade values (see Appendices).

An additional validation check was completed by comparing the correlation of the average grade of the composites within a block to the corresponding estimated block grade.

As a further check on the model the average estimated block grades were compared to the Nearest Neighbour block model grades generated by Aya, and to the average capped composite grades and declustered composite grades (Table 14-14 and Figure 14-6).

The current Authors consider the validation results to be acceptable for linear estimation.

Table 14-14 Grade Block Model Check

Grade Element	Composite Average	Declustered Composite Average	Block Average	NN Block Average
Ag (g/t)	49.73	48.19	43.63	47.04
Au (g/t)	0.06	0.06	0.05	0.06
Cu (%)	0.06	0.06	0.06	0.07
Pb (%)	0.63	0.66	0.59	0.70
Zn (%)	1.79	1.76	1.44	1.61

Note: NN = Nearest Neighbour

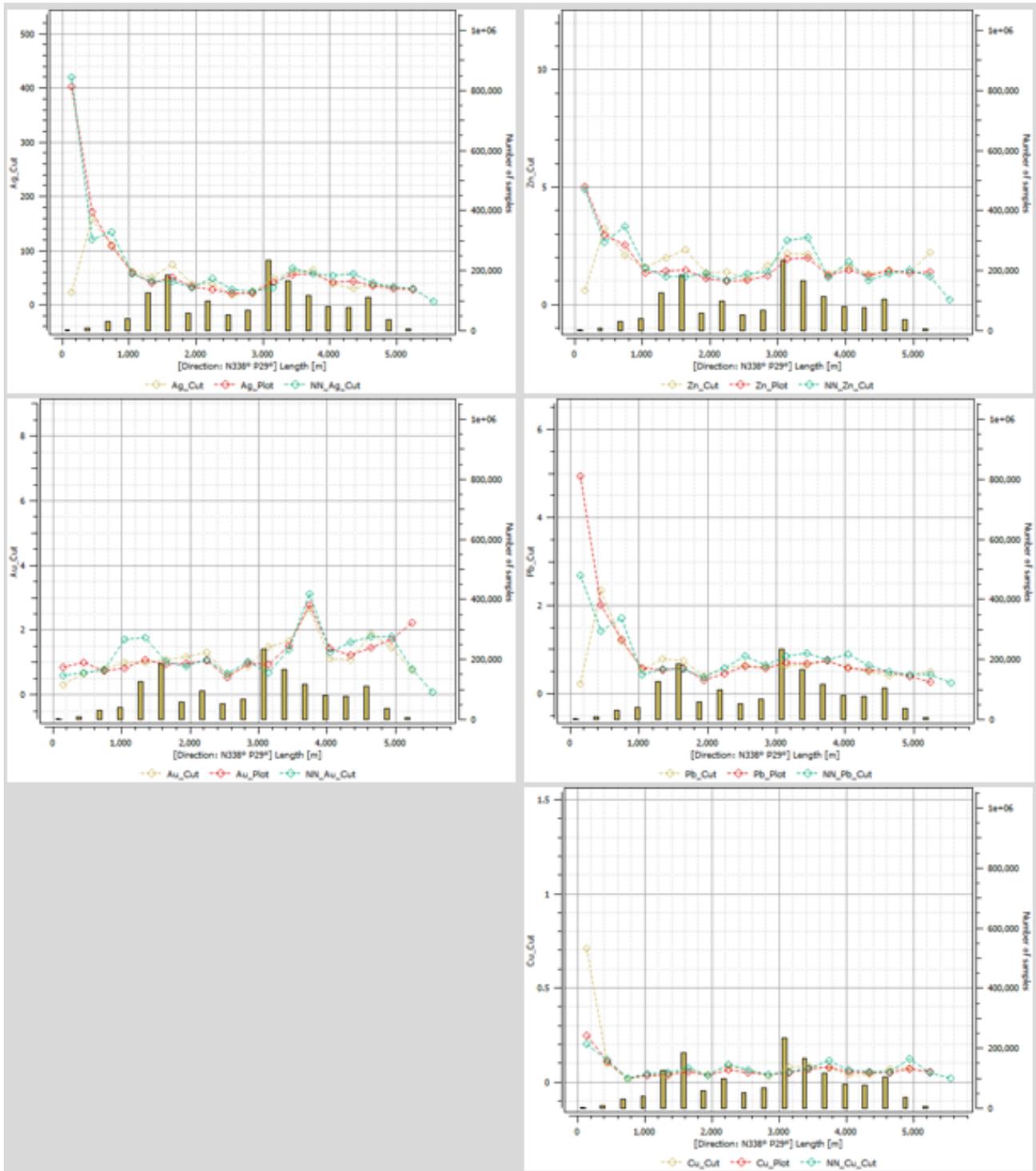


Figure 14-6 Composite Grade Versus Swath Plots

Note: Ag_Cut = Composite Grade; Ag_Plot=Model Grade; NN_Ag_Cut=Nearest Neighbourhood block estimate

Source: This Report

15.0 MINERAL RESERVE ESTIMATES

This section is not applicable to this Report.

16.0 MINING METHODS

16.1 GENERAL DESCRIPTION OF THE MINING PROJECT

The Boumadine deposit is a large mineralized polymetallic system, containing Au, Ag, Zn and Pb, delineated over more than 5 km with historical mining infrastructures in the center and south zones. The mineralization at Boumadine starts from surface in some occurrences and is generally in the form of vertical, or sub-vertical, structures. A mix of Open Pit mining (“OP”) and Underground mining (“UG”) operation is planned for Boumadine.

16.2 GEOTECHNICAL

16.2.1 Open Pit

16.2.1.1 Open Pit Slope Design

Geotechnical evaluations of the Boumadine pits have been completed by empirical evaluation to produce slope design constraints dominated by on slope dip direction (see RockEng report #24017-102 for complete documentation of geotechnical site characterization and analyses). The pit slope design guidance may be applied to varying pit shell geometries with reliance on slope orientation (rather than geographical location).

Overall slope angle (OSA) constraints are summarized in Table 16-1). It is noted that these are maximum allowable OSA values; slopes may be limited to a flatter angle once benches, ramps, slope breaks etc. are accounted for in geometry.

Table 16-1 Summary of OSA recommendations.

Slope Dip Direction	Overall Slope Angle (OSA)		
	Slope Height up to 380m	Slope Height up to 220m	Slope Height up to 100m
5	48	52	60
65	38	42	48
125	48	52	60
185	48	52	60
245	48	52	60
305	48	52	59

Further to the OSA summarized above, pit slope designs are constrained by the following:

- Adverse jointing is expected at Boumadine in east-northeast dipping slopes due to the presence of a prominent ENE dipping joint set. Here, the inter ramp slope angle (IRA) should not exceed the dip of the adverse joint (48°).
- Minimum catch bench width are defined as follows (Ryan & Pryor, 2000 in Read & Stacey, 2009):

$$\text{Catch Bench Width (m)} = 0.2 * \text{Bench Height} + 4.5\text{m}$$
- Where overburden is present, it will be stripped and laid back at a 2H:1V slope. A bench between the toe of the overburden and the crest of rock should be left around the pit perimeter to prevent overburden from running into the pit (bench width should be equal to overburden depth).
- The first 2 benches will be single bench height to account for the likely occurrence of poorer ground conditions near surface (weathering horizon).
- Below the initial two benches in rock, 20 m high double benches are geotechnically feasible.
- Geotechnical slope breaks will be established every 100 vertical metres; preliminary design consider geotechnical slope breaks to be not less than double the width of a typical catch bench.

16.2.2 Underground

Geotechnical evaluations of the Boumadine underground mine have been completed by empirical evaluations (see RockEng report #24017-102 for complete documentation of geotechnical site characterization and analyses).

16.2.2.1 Mining Method Selection

Longhole open stoping is a geotechnically sound mining method for the Boumadine underground. Based on deposit body geometry, longitudinal open stoping and/or Avoca style stoping is deemed amenable from a geotechnical perspective. Longitudinal stoping provides operational flexibility to adjust stope panel lengths, thus there is an opportunity for optimizing design for local variances (good or bad) in ground conditions. This is particularly important where the deposit body is narrow as production performance in narrow zones is likely to be more sensitive to HW and FW dilution.

At relatively shallow depths, low stress conditions allow for operational flexibility with mine sequencing. At this phase of study, one key aspect of sequencing that has been identified pertains to inter-lens pillar stability and adjacent stope performance. The lead/lag between adjacent lenses is minimized as an opportunity to mitigate loss of confinement.

16.2.2.2 Open Stope Design and Overbreak Potential

Stope stability has been evaluated by empirical methods considering vertical stope sidewalls, HW and FW faces dipping ENE at $\sim 85^\circ$ (based on modelled stope geometries), and horizontal stope backs. It is assumed that both sulphides and Tuff may be exposed in stope faces. The recommended hydraulic radii limitations are provided in Table 2. It has been concluded that 20 m level spacing (25 m stope height), with 20 m strike lengths are well suited for Boumadine underground. Larger stopes may be achieved if cable bolting is implemented. At these dimensions, with typical HW to FW spans, stope backs and endwalls are expected to perform well, with stope back cable bolting required rarely (where HR exceeds 4.2). The predicted overbreak (for 20 m strike x 20 m level spacing) is roughly 0.5 m for stope HWs and blast damage only for stope FWs.

Table 16-2 Recommended stope sizing constraints by stope face and rock type.

Stope Face	Unsupported HR	Supported HR
HW/FW	5.6	7.5
Back	4.2	6.2
Endwalls	7.5	-

16.2.2.3 Pillar Stability

Inter-lens Stope Pillars

Inter-lens pillars exist between stacked (HW to FW) mineralized zones. The Boumadine underground mine plans intend to utilize cemented rock fill, and so the largest mining risk associated with inter-lens stope pillar stability is dilution and oversize entering open stopes (as opposed to global mine stability). The following pillar dimensioning constraints have been established by empirical methods.

- At W:H = 0.3 (6 m for a 20 m stope strike length), total pillar loss predicted for HW to FW sequencing is 2.3 m, and for FW to HW sequencing is 2.2 m.
- At W:H = 0.4 (8 m pillar for a 20 m stope strike length), total pillar loss for HW to FW sequencing is 1.8 m, and for FW to HW sequencing is 1.5 m.
- At W:H = 0.5 (10 m pillar for a 20 m stope strike length), total pillar loss for HW to FW sequencing is 1.1 m, and for FW to HW sequencing is 1.0 m.

Rib & Sill Pillars

Rib pillar dimensioning rely on 1:0.7 width (rib pillar horizontal thickness) to height (HW to FW span) aspect ratio with a minimum dimension of 3 m. Permanent sill pillar dimensioning relies on 1:1 width (vertical dimension) to height (HW to FW span) aspect ratio with a minimum dimension of 5 m.

Crown Pillar

Based on empirical crown pillar evaluations, a 20 m pillar (in rock, negating overburden thickness) is deemed appropriate for stable crown pillar conditions during active mining operations. This dimension relies on the assumption that backfill will be utilized to manage long term stability for mine closure. Further, it is not unreasonable to expect that, with appropriate engineering and planning, full crown pillar recovery may be achieved (assuming surface disturbance is permitted, and environmental risks are tolerably low).

16.2.2.4 Infrastructure Siting

Given the relatively shallow mining depth, the geotechnical risks associated with stress induced damage to permanent infrastructure are considered low; thus, allowing for reasonable design flexibility, general guidance are as follows: Cross-cut lengths ensure that haulage drive stand-offs are at least 15 m from the deposit body contact. Given the near-vertical nature of the deposit body, consideration may be given to either HW or FW infrastructure placement. Lateral development lay outs ensure that wide spans and intersections are not stacked level-to-level.

16.2.2.5 Ground Support

Standard lateral development (4 to 6 m spans) will be supported by 2.4 m rebar at a 1.25 x 1.25 m square pattern applied to the back and walls, with welded wire mesh in the back and shoulders. Cable bolts will be utilized in intersections and wide spans with bolt lengths equal to ½ the span, at a 2 m square pattern. It is assumed that that 2% of development will require shotcrete.

16.3 HYDROGEOLOGY

Deemed unnecessary for this stage of the project as it is a Preliminary Economic Assessment (PEA).

16.4 MINING METHOD

16.4.1 Mining Method Selection

This section details the assumptions and procedures used to develop a mine plan for the project. The Boumadine deposit extends from surface to depth well beyond what could be mined using an OP method exclusively. It has been decided early on that the mining method used would be a mixture of OP and UG.

As a first step, pit shells were created using the Whittle software, and UG stopes created using Deswik Stope Optimizer (DSO). With this initial design, it was possible to make a preliminary evaluation of the interaction between OP and UG, and to work towards an OP design that would recover as much high-grade material at a reasonable cost before the UG method could become more economical.

Historical exploration activities, including UG development, have been conducted on the deposit in the 1980's and 1990's; and therefore, it will be necessary in the future to locate and survey these workings, so they can be considered in the detailed mine design and planning.

16.4.2 NSR Cut-offs determination

For the Boumadine Project, Aya used an NSR cut-off to evaluate the economic viability of mining a tonne of diluted mineralized material. The viability is determined by comparing the material's NSR value with the

applicable NSR cut-off. An NSR cut-off represents the onsite cost to mine and process one tonne of mineralized material, including general and administrative (G&A) expenses.

The NSR cut-offs incorporate mining, processing, and G&A costs based on a preliminary cost model, which has then been confirmed by the operating costs estimates developed in Section 21.

Table 16-3 presents the calculations used to derive the NSR cut-offs applied in this PEA study for the open pit, underground stopes, and underground development.

Table 16-3 NSR Cut-off Calculations

GENERAL ASSUMPTIONS			
Area	Value	Unit	Source
Royalties	3	%	
Discount Rate	5	%	
OPEN-PIT MINING			
Area	Value	Unit	Source
Mining	\$ 4.60	USD/t	
Processing Cost	\$ 57.85	USD/t	
G&A	\$ 4.30	USD/t	
Total	\$ 66.75	USD/t	
Waste Mining	\$ 2.00	USD/t	
Sustaining Capital	\$ 2.27	USD/t	Zgounder MRE
UNDERGROUND MINING			
Area	Value	Unit	Source
Mining	\$ 62.86	USD/t	
Processing Cost	\$ 57.85	USD/t	
G&A	\$ 4.30	USD/t	
Total	\$ 125.00	USD/t	
UNDERGROUND DEVELOPMENT			
Area	Value	Unit	Source
Mining	\$ 18.30	USD/t	
Processing Cost	\$ 57.85	USD/t	
G&A	\$ -	USD/t	
Total	\$ 76.15	USD/t	

NB - assumes development is required by block - decision to sent to mill vs waste

16.4.3 Pit optimization

GEOVIA Whittle (2022) was used to perform pit optimization process for Boumadine. Using the Lerch-Grossman (LG) algorithm inside of Whittle, nested shells were generated, and the optimal pit shell was selected for each deposit. These pit shells were then leveraged for mine design and ultimately scheduling process.

Table 16-4 describes the economic and mining inputs used for pit optimization:

- Estimated costs and cut-off grade values were provided by AYA. These costs were compared to cost benchmarked against similar project before being implemented in the Whittle.

Table 16-4 Whittle Parameters

Mining Parameters	Value	Unit
Mining Cost (Ore)	4.60	USD/t
Mining Cost (Waste)	2.00	USD/t
Processing Cost	57.85	USD/t
G&A	4.30	USD/t
Mining Recovery	100	%
Mining Dilution	0	%
Processing Plant Throughput	8,000	t/d
Cut off Grade (NSR)	66.75	USD%/g/t

Based on recommendations from the Geotechnical team, Table 16-5 presents the pit slopes parameters used for pit optimization depending on the ultimate depths of the pits.

Table 16-5 Geotechnical Parameters

Slope Dip Direction	Overall Slope Angle (OSA)		
	Slope Height up to 380m	Slope Height up to 220m	Slope Height up to 100m
5	48	52	60
65	38	42	48
125	48	52	60
185	48	52	60
245	48	52	60
305	48	52	59

Taken from the geotechnical report

The first iteration of the pit shells was generated in four distinct pits, two located in the south, one central pit, and one in the north. The initial block model was then modified to create four separate block models, each constrained to the outlines of the pits generated during the first calculation. Each pit was subsequently optimized individually, allowing for a more precise analysis tailored to the geological and economic characteristics of each area.

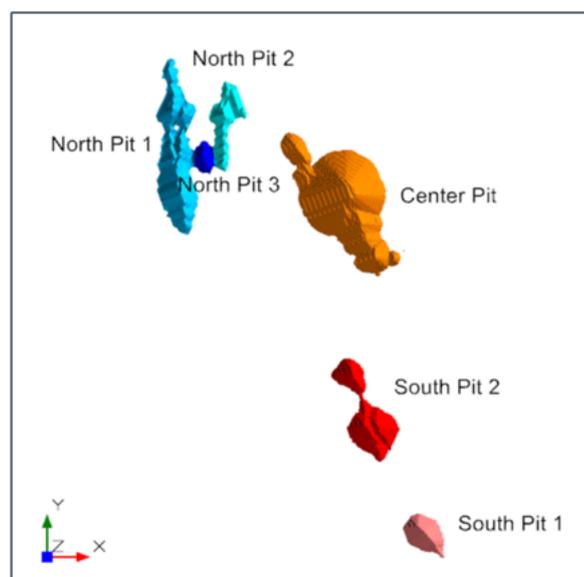


Figure 16-1 Initial Pit Shells Generated using the Large Block Model

Source: WSP (2025)

The methodology and optimization parameters used for the Central zone is presented in this section. A similar method and parameters was used for the other pits.

The first step consisted of re-blocking the block models to dimensions of 10 x 10 x 10 m (X, Y, Z), to ensure adequate resolution for optimization analyses.

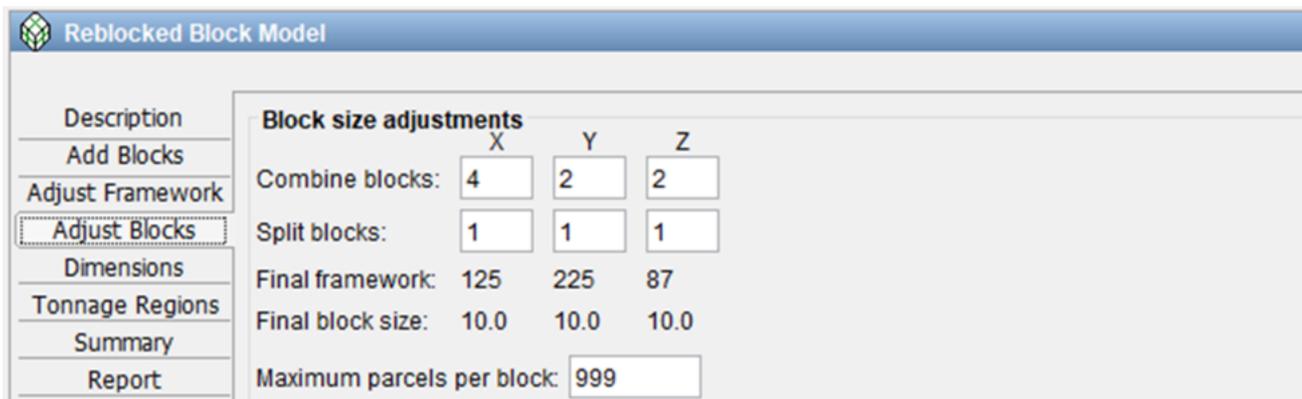


Figure 16-2 Whittle Reblocked Block Model Parameters

Source: WSP (2025)

Next, the pit slope angles were defined according to the geotechnical recommendations, with variations, depending on pit depth, to reflect stability constraints.

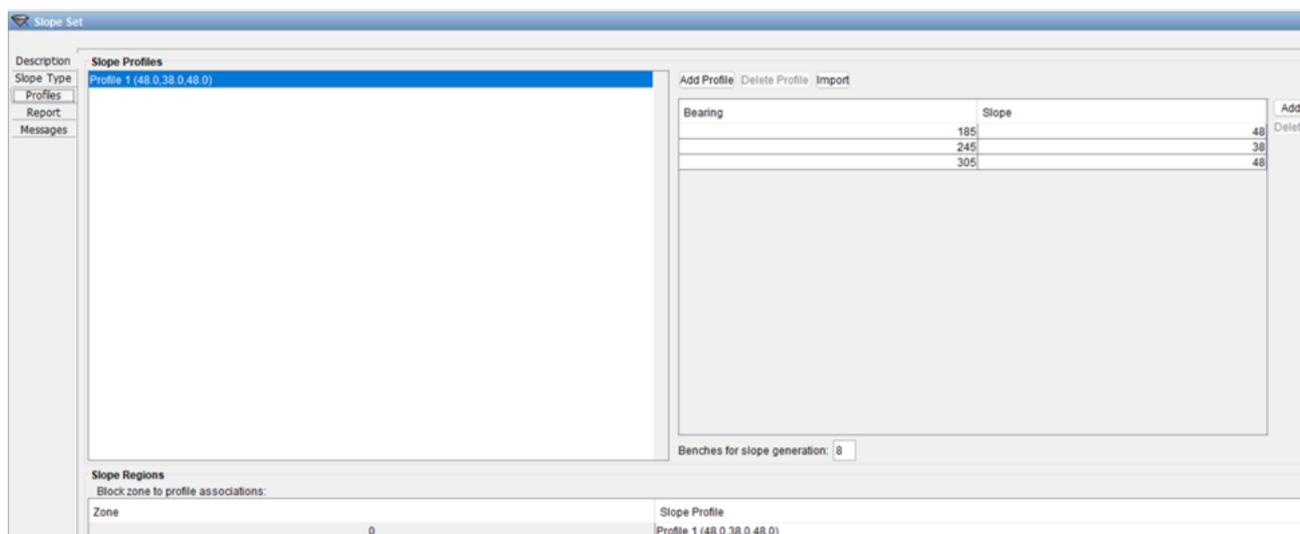


Figure 16-3 Whittle Slope Setting

Source: WSP (2025)

For each pit, optimized pit shells were generated using a Revenue Factor (RF), ranging from 0.4 to 1.1, with increments of 0.02. Based on these results, pit-by-pit graphs were developed, incorporating a 5% discount rate and a selling price of 0.97cent/\$ to account for a 3% royalty, thereby allowing the development of optimistic and pessimistic scenarios.

Pit Shells										
Description										
Mining										
Processing										
Selling										
Optimization										
Constraints										
Expressions										
User Element										
Output										
Report										
Messages										
Number of pit shells generated : 36										
Summary data for each pit:										
Pit	Rev Factor	Rock	Ore	Strip	Max Bench	Min Bench	nsr_Units	nsr_Grade		
1	0.4	27 728 247	1 848 277	14	80	58	559 472 358	302.7		
2	0.42	31 304 817	2 070 377	14.12	80	57	613 499 250	296.32		
3	0.44	50 930 865	2 788 016	17.27	81	57	821 091 948	294.51		
4	0.46	114 811 706	4 513 899	24.44	82	50	1 367 458 327	302.94		
5	0.48	121 148 483	4 812 927	24.17	82	50	1 437 787 455	298.74		
6	0.5	143 050 026	5 384 274	25.57	82	46	1 608 367 326	298.72		
7	0.52	154 564 569	5 806 810	25.62	82	45	1 710 050 694	294.49		
8	0.54	162 779 344	6 302 835	24.83	82	44	1 804 265 573	286.26		
9	0.56	169 972 812	6 882 851	23.7	82	44	1 901 183 396	276.22		
10	0.58	187 232 287	7 599 983	23.64	82	44	2 046 846 677	269.32		
11	0.6	190 372 499	8 206 255	22.2	82	44	2 126 442 184	259.12		
12	0.62	199 841 108	8 694 208	21.99	82	43	2 212 321 848	254.46		
13	0.64	201 559 549	8 936 167	21.56	82	43	2 243 658 589	251.08		
14	0.66	206 273 928	9 228 532	21.35	82	43	2 288 740 483	248.01		
15	0.68	226 240 856	9 727 404	22.26	82	41	2 399 028 540	246.63		
16	0.7	233 968 871	10 133 219	22	82	41	2 458 610 282	242.63		
17	0.72	233 948 798	10 323 909	21.66	82	41	2 479 054 663	240.13		
18	0.74	239 081 154	10 697 013	21.56	82	40	2 518 484 331	237.66		
19	0.76	245 379 858	11 109 232	21.09	82	40	2 581 638 033	232.39		
20	0.78	246 339 809	11 248 160	20.9	82	40	2 596 284 061	230.82		
21	0.8	252 169 806	11 483 320	20.96	82	40	2 631 242 188	229.14		
22	0.82	253 404 050	11 683 740	20.69	82	40	2 650 980 735	226.9		
23	0.84	263 575 232	12 042 889	20.89	82	40	2 705 103 975	224.62		
24	0.86	277 382 684	12 472 488	21.24	82	40	2 772 044 036	222.25		
25	0.88	287 776 491	12 722 953	21.62	82	39	2 815 605 998	221.3		
26	0.9	288 303 388	12 812 816	21.5	82	39	2 823 564 625	220.37		
27	0.92	299 461 938	13 043 045	21.96	82	38	2 865 633 797	219.71		
28	0.94	300 035 166	13 153 638	21.81	82	38	2 874 857 811	218.56		
29	0.96	329 659 556	13 585 479	23.27	82	36	2 969 502 510	218.58		
30	0.98	331 560 804	13 743 328	23.13	82	36	2 984 400 669	217.15		
31	1	344 470 439	13 999 038	23.61	82	35	3 028 325 672	216.32		
32	1.02	398 870 005	14 919 749	25.73	82	34	3 200 223 334	214.5		
33	1.04	410 782 983	15 243 657	25.95	82	34	3 245 073 126	212.88		
34	1.06	414 631 994	15 406 981	25.91	82	34	3 263 006 861	211.79		
35	1.08	431 987 003	15 714 470	26.49	82	33	3 316 046 654	211.02		
36	1.1	434 933 427	15 872 665	26.4	82	33	3 331 239 305	209.87		

Figure 16-4 Whittle Pit Shells Iteration – Pit Center

Source: WSP (2025)

Once the pits were generated, and after running several scheduling scenarios for each pit individually, the mining rate that would enable optimal extraction of the majority of the pits was evaluated.

The total mining rate, including mineralized material and waste, was set at 137,000 tonnes per day (t/d), or 50Mt per year.

The pit-by-pit graphs were then recalculated to adjust production schedules and select the optimal pit shells based on this rate.

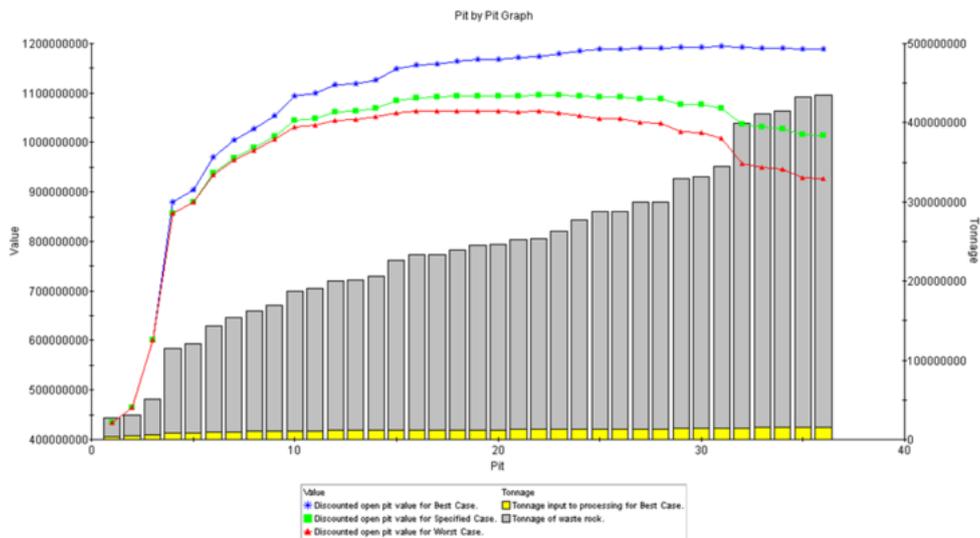


Figure 16-5 Whittle Pit Shells Iteration – Pit Center

Source: WSP (2025)

For the Center Pit iteration 18 was selected has the optimal shell, with a revenue factor of 0.74.

16.4.4 UG Mining

The currently identified UG section of the Boumadine deposit extends to approximately 640 m below surface and spans up to ~5.6 km along strike. It is divided into three main zones: North, Center, and South. The deposit's dip is relatively steep, varying from around 70° to around 80°, and its average width ranges between about 1 and 4 m. Higher grade mineralized areas are concentrated in the Center zone beneath the pit also in the far North zone, as shown in Figure 16-6.

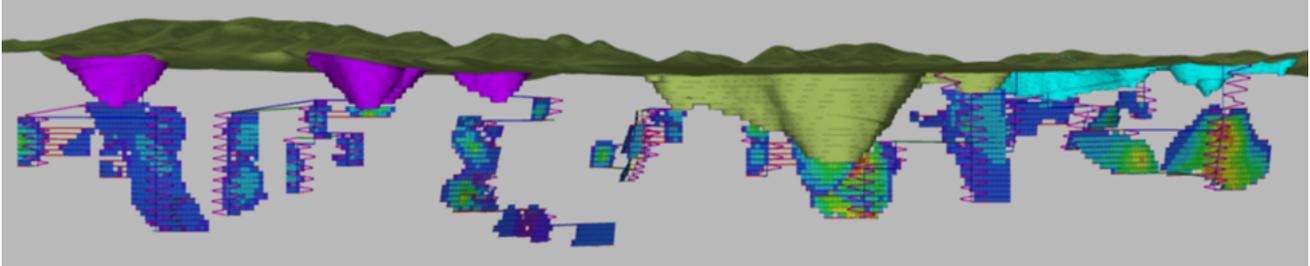


Figure 16-6 Boumadine Mining Design

Source: WSP (2025)

Given the deposit's geometry (dip and width) and its geotechnical parameters (Rock Mass Characterization) the long-hole stoping method was judged appropriate. And because of its productivity, flexibility, and other advantages, the modified Avoca method was selected

The deposit is planned to be mined in a bottom-up sequence using the modified Avoca method, a variant of conventional Avoca adapted for situations lacking full dual-end access or ideal geometry. It is essentially a longitudinal retreat long-hole stoping technique.

This method divides the deposit into sublevels, and the mineralized material between them is mined as stopes. Most of the mineralized material on each sublevel will be mined with parallel longitudinal stopes extending along the strike of the deposit. Each series of longitudinal stopes requires a production drive on the upper sublevel for drilling and loading explosives and a production drive on the lower sublevel for mucking blasted mineralized material. These production drives will be driven from a central access drift connected to the ramp.

The longitudinal stopes will be mined in a retreating fashion, advancing in series, one after the other, towards the access drift. Blast holes are drilled between sublevels. To initiate each bench, a slot is drilled and blasted to form or access a void; subsequent blast hole rings are fired into that void until the desired hydraulic radius is achieved. As mining proceeds, stopes are backfilled progressively with waste rock from the far end, maintaining a controlled void ahead of the fill front. The cycle of drilling, blasting, mucking, and backfilling repeats until the entire level is extracted. The method is illustrated in Figure 16-7.

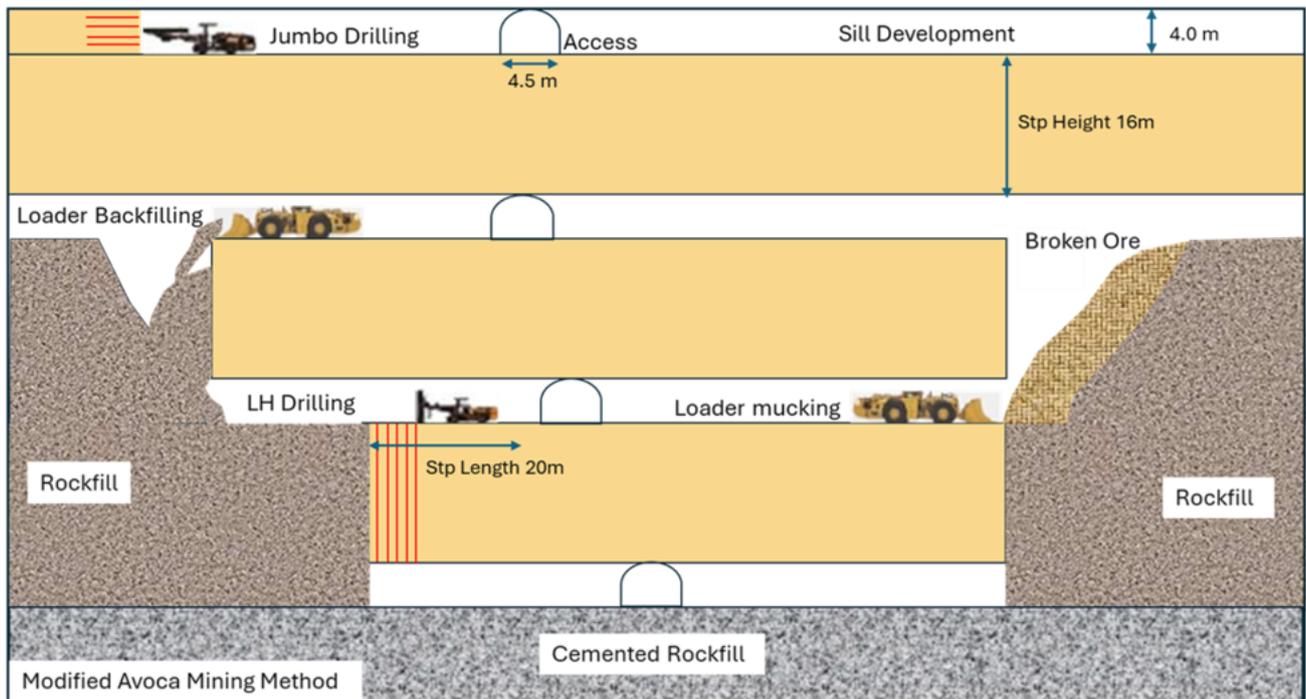


Figure 16-7 Modified Avocat Mining Method Illustration

Source: WSP (2025)

16.5 MINE DESIGN

16.5.1 Open Pit Mine Design

Pit designs were developed for the three deposit areas and are based on the optimized shells and the geotechnical design parameters from the report developed by RockEng, and are as follows:

- Bench height: 10 to 20 m
- Minimum catch bench width (m) = $0.2 * \text{Bench Height} + 4.5$
- Inter Ramp Slope Angle < 48 degrees for east dipping slopes
- Include a slope break every 100m: Slope break width is double a standard bench width
- No pre-split blasting is deemed necessary at this stage.

The pit locations are displayed in Figure 16-8.

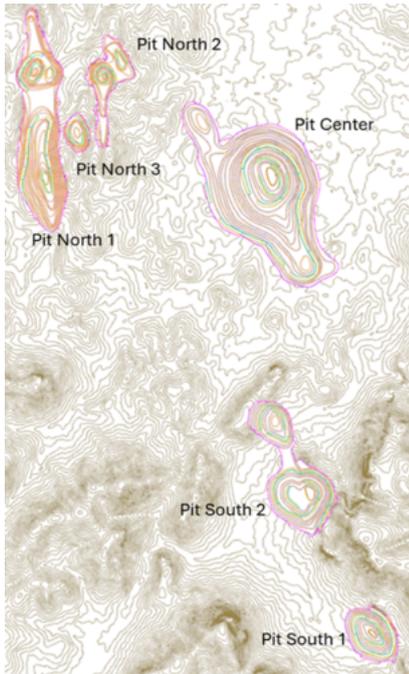
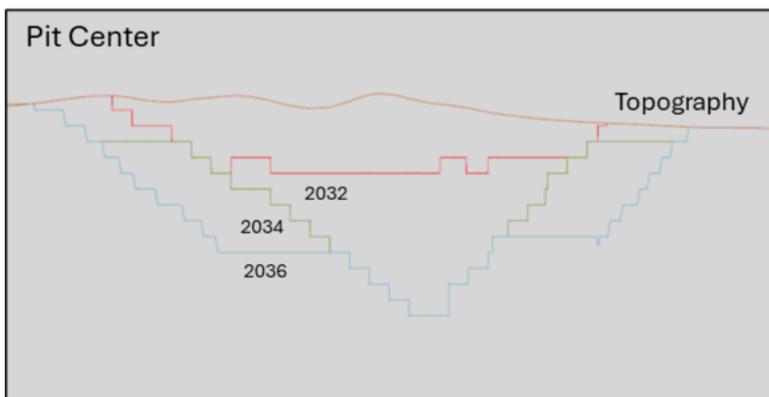
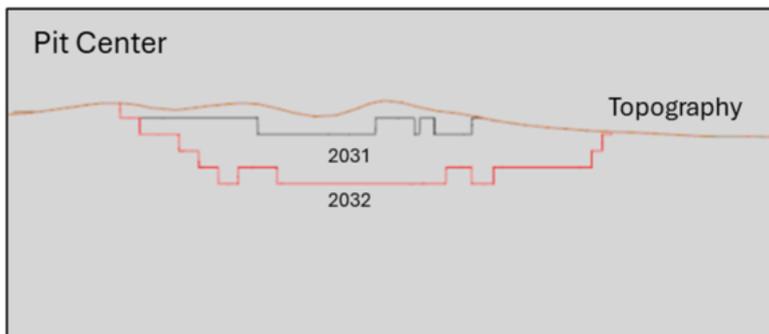


Figure 16-8 Boumadine pit locations

The Center pit design consists of three phases of successive pushbacks around the entire pit perimeter as illustrated in Figure 16-9 a-b-c



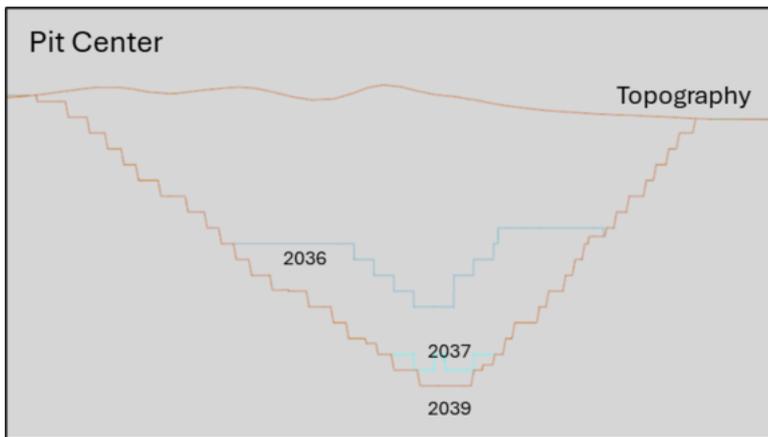


Figure 16-9 Pit Cross section

Figure 16-9 Description: Center Pit cross section for years : (a) 2031-2032 (b) 2032-2036 (c) 2036-2039

The other small pit designs consist of one or two phases which can be scheduled independently following the same approach typically used for satellite pits

The mining parameters are based on a production of 137 000 t/d. The production will be performed by a contractor that will have to select their equipment. For design purposes, it is assumed that a mix of 90 tonne CAT 777 Truck and 140-tonne CAT 785 Truck or similar truck size, will be used and paired with a 200-t Class CAT 6020 and/or 300-t Class CAT 6030 Shovel or similar shovel size. It has been assumed that all the Open Pit mining activities will be accomplished by a mining contractor.

With this assumed equipment. To following parameters were deduced:

- Minimum Mining Width: 30 m
- Ramp Width (Double Lane): 20 m
- Ramp Width (Single Lane): 12 m
- Turning Radius: 20m
- Maximum ramp grade: 10%

16.5.2 UG Stope design

Stope dimensions were based on recommendations from the PEA Geomechanics Report. Each stope was designed to be 20 m in height and 20 m in length, and mining will proceed in a retreating sequence.

Stope shapes were generated using DSO software, considering the deposit geometry, as well as geotechnical and economic criteria. Multiple DSO scenarios were evaluated to determine the optimal results in terms of tonnage and grade.

The DSO Shapes are visually checked and sequenced to suit the mining method (long hole longitudinal retreat) and scheduled in Deswik scheduler to produce the production profile and LOM. The stope design parameters are summarized in Table 16-6.

Table 16-6 UG Stope Design Parameters

UG Stope Design parameters	
Parameter	Value
Stope parameters	
Mining Method	Modified Avoca
UG NSR COG	125 \$/t
Minimum Width	2 m
Maximum Width	8 m
Stope Height (Floor to Floor)	20 m
Stope Length	20 m
Pillar between veins	8 m
Minimum dip angle	55 deg
Mining Dilution ELOS HW	0.5 m
Mining Dilution ELOS FW	0.0 m
Mining recovery	95%
Backfill Dilution	5%
Longhole Drilling	
Drilling Layout	Downhole Dice Five
Hole Diameter	64 mm
Burden	2.1 m
Spacing	1.6 m
Drilling Factor	4.57 t/m-drilled
Longhole Blasting	
Explosive	Bulk Emulsion
Powder factor	1.13 kg/m ³
Backfill	
Backfill	Rock fill and CRF

A summary of the stopping shapes properties for the different zones is presented in Table 16-7.

Table 16-7 Zones Stope Geometry Proprieties

	DSO Shape Number	DSO Shape Width (m)			DSO Shape Dip (deg)		
	Unit	Min	Max	Average	Min	Max	Average
North	755	2	7.9	2.6	55	80	77
Centre	765	2	4.9	2.9	55	87	79
South	1295	2	8.3	2.8	55	80	76

Preliminary economic check was conducted on satellite mining areas to ensure they generate enough value to pay for the required development and infrastructure costs. After this exercise a few non-economic areas were eliminated from the production plan.

16.5.3 UG development design

Lateral development will include the main access ramps (declines), level access drifts, the production drifts, and all the necessary excavations for infrastructures.

Vertical development will consist of ventilation raises and rock passes, which are planned to be excavated using either raise boring, or conventional raise mining methods.

All lateral and vertical development activities are assumed to be carried out by a contractor. Aya has provided the unit rates for these activities, based on quotes received from selected contractors.

The decline is driven at a maximum gradient of -15% and will provide access to mining levels for all zones. The mining area will be accessed by driving level access and sill drifts above and below the stoping area.

For the purposes of this Preliminary Economic Assessment, a preliminary ground support design has been developed based on recommendations from the geotechnical report. The standard support in development headings consists of Swellex bolts installed in a 1.25 m x 1.25 m square pattern, with mesh applied across the back and down the walls to mid-drift height.

In intersections and areas with wide spans, additional support is assumed in the form of cable bolts, with lengths equal to half the span, installed in a 2.0 m x 2.0 m square pattern.

The main UG development heading profiles are presented in Table 16-8.

Table 16-8 Development Heading Profiles

	Tunnel Profile (without overbreak)			
	Width (m)	Height (m)	Length (m)	Max grade (%)
Lateral Development				
Spiral Ramp	4.5	4	-	±15
Level Access	4.5	4	-	±3
Ventilation Drift	4.5	4	-	±3
Connection Drift	4.5	4	-	±15
SILL Rock drift	4	4	-	±3
Rock Pass ACCESS	4.5	4	-	±3
Electrical Main Substation	4.5	4	10	±3
Main Pumping Station	4.5	4	20	±3
Ore Remuck Bay	4.5	4	15	±3
Level Sump (Gravity Drain/Borehole)	4.5	4	15	±15
Temporary Electrical Station	4.5	4	10	±3
Portable Refuge Bay	4.5	4	7	±3
Safety Bay	1.5	2	1	±0
Vertical Development				
Ventilation Raises			3 and 4 m Diameter	NA
Ore Pass			4 m Diameter	NA

Each level includes electrical stations, sumps, ventilation accesses, truck turnaround areas, and remuck bays, as illustrated in the schematic plan of a typical level shown in Figure 16-10.

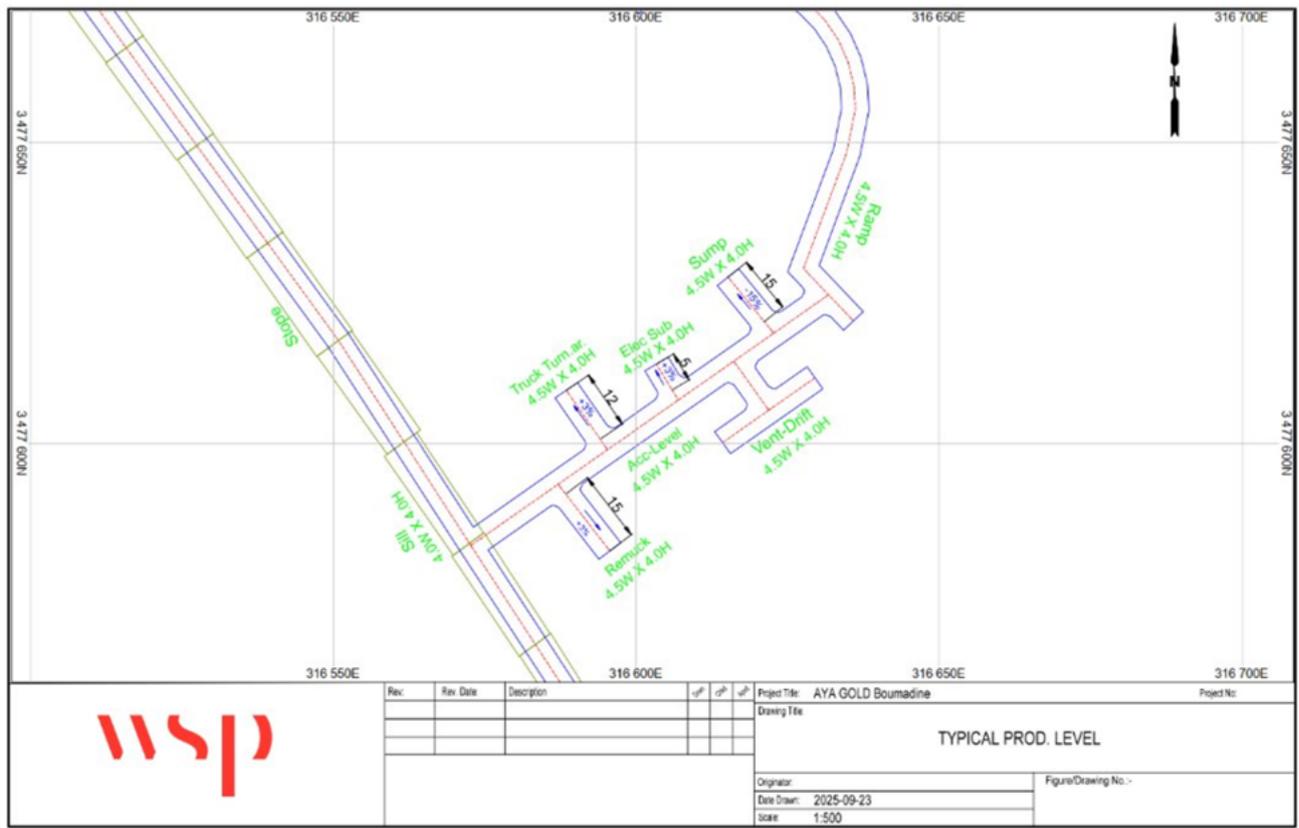


Figure 16-10 Typical Level Layout

16.6 MINE SAFETY INFRASTRUCTURE

The Boumadine UG mining zones will have the following infrastructure related to mine safety:

- The ventilation raises connecting the sublevels will be equipped with ladderways, providing an emergency escape alternative to the spiral ramp.
- Mine-rescue equipment will be provided for use by the mine-rescue team.
- The mine site will have a dedicated ambulance for emergency medical response.
- Portable refuge station will be strategically positioned throughout the mine, each designed and equipped to accommodate up to eight people for 36 hours.
- The mine will have a stench gas system to provide warning in an emergency.

16.7 MINE EQUIPMENT

The equipment fleet has been detailed for production activities only, as all development work will be performed by a contractor, who will be responsible for supplying the necessary equipment and materials.

The Mobile equipment units needed per year for each equipment type is based on their estimated productivities and the planned production targets outlined in the LOM plan.

The equipment fleet includes the main production vehicles as well as auxiliary vehicles, to support operations.

Vertical development will be completed by contractors providing the equipment as required. The raising and development equipment is excluded from the equipment list. The LOM mobile equipment fleet is summarized in Table 16-9.

Table 16-9 Production Mobile Equipment

	Model	Max Units Required
Drilling & Blasting		
Prod Drill, ITH	Sandvik DL311	3
Emulsion Charger	PAUS 40 ST-LG	2
Mucking & Haulage		
30 tonnes Mine Truck	Epiroc MT436	6
10 tonnes LHD 10 t	Epiroc ST1030	7
Ground Support		
Cable Bolter	Sandvik DS 421	2
Transmixer	Partindus HURON 45	1
Service		
Boom trucks	PAUS TSL 853 T7	2
Grader	XCMG GR135C	1
Fuel Lube Truck	PAUS UNI 40SV	2
Water Truck	PAUS UNI 40SV	1
Pickup Truck	Toyota Hilux	15
Mine rescue Truck	ACCES	1

16.8 MINE PERSONNEL

Table 16-10 presents the annual owner’s staffing requirements for a full production year, covering UG production as well as the technical, supervisory, and support teams. It excludes contractor personnel involved in OP production and UG development. This roster includes provisions for absences, vacations, and temporary vacancies.

The mine will operate year-round, seven (7) days per week, twenty-four (24) hours per day (two (2) 12-hour shifts). Most of the workforce will be recruited locally, supplemented by a few expatriate technical staff. In addition, Aya plans to mobilize experienced personnel from its Zgounder Mine in Morocco to provide on-Site training and to help ensure that operations are conducted safely and efficiently.

Table 16-10 Mine Personnel List

Group / Role	Max Required
UG Mining	
Longhole Drillers	12
Longhole Blasters	8
Mine Truck Operators	24
LHD Operators	28
Cable Bolter Operators	8
Transmixer Operators	4
Scissor Lift Operators	16
Boom Truck Operators	24
Grader Operators	4
Fuel Lube Truck Operators	8

Group / Role	Max Required
Water Truck Operators	4
Total	140
UG Maintenance	
Mobile Mechanics	11
Fixed Mechanics	4
Fixed Mechanics - Batch Plant	4
Mobile Electricians	4
Fixed Electricians	8
Fixed Electricians - Batch Plant	4
Total	35
Operations and technical services (UG and OP)	
Mine Supervisors - Direct	18
Maintenance Supervisors - Direct	5
Mine Manager	1
OP Captains	1
UG Captains	3
OP Mine Admin	1
UG Mine Admin	2
Maintenance Planning	2
OP Technical Services	20
UG Technical Services	36
Total	89

16.9 PRODUCTION PLAN

Both the UG and OP mining sequences were developed using Deswik Scheduler. The Boumadine project LOM production plan extends over twelve (12) years, including pre-production.

To improve the overall project economics, the production schedule prioritizes the mining of higher-grade material during the early years. Additionally, UG production is intentionally delayed by two (2) years following the start of OP operations. This phased approach allows for earlier cash flow generation from the OP while deferring the capital-intensive UG development to a later stage in the mine life.

The Net Smelter Return (NSR) is used to evaluate the economics and feasibility of Boumadine project. The NSR value relates to the expected net value received for metal in saleable concentrate, net of costs for transportation and associated smelting and refining charges.

The NSR of for each block in the resource model was calculated based on the block head grade and the parameters presented in Table 16-11 .

Table 16-11 Parameters of Calculating NSR values

Element	Metal Price \$US/lb or oz	Concentrate Recovery	Smelter Payable
Lead Concentrate			
Pb	\$1.00	84.5%	95%
Cu	\$4.00	75.3%	95%
Zn	\$1.20	3.9%	0%
Au	\$2,200	26.5%	95%
Ag	\$24.00	53.1%	95%
Zinc Concentrate			
Pb	\$1.00	2.2%	0%
Zn	\$1.20	72.0%	85%
Au	\$2,200	1.7%	0%
Ag	\$24.00	5.5%	0%
Pyrite Concentrate			
Au	\$2,200	69.0%	60%
Ag	\$24.00	39.6%	60%

The NSR formulas used for the development of the Mine Plan differs slightly from the one used for the mineral Resource estimate, and is based on the metallurgical recoveries for pyrite flotation.

NSR - Mine values were calculated as follows:

$$\text{NSR } \$/t = (\text{Pb}\% \times \$10.74) + (\text{Zn}\% \times \$9.43 \times 1.44) + (\text{Au g/t} \times \$46.85) + (\text{Ag g/t} \times \$0.55) + (\text{Cu}\% \times 63.08)$$

Mineralized material in the production schedule has been classified into three distinct stockpiles based on both the overall mineralized Net Smelter Return (NSR-Mine) and the NSR-Pyrite. The NSR-Pyrite was calculated on a per tonne of pyrite basis, with the intention of showing the potential return per tonne of pyrite produced from each tonne of feed to the mill. Therefore, it is possible for the NSR-Pyrite to be higher than the NSR-Mine.

The NSR-Pyrite was calculated based on the parameters presented in Table 16-12. The gold and silver values are used to calculate the value of the precious metals in the pyrite concentrate, while the iron value is used to calculate the mass pull to the pyrite concentrate.

Table 16-12 Parameters of Calculating NSR Pyrite values

Element	Metal Price \$US/oz	Concentrate Recovery	Smelter Payable
Pyrite Concentrate			
Au	\$2,200	69.0%	60%
Ag	\$24.00	39.6%	60%
Fe	\$0	90.0%	0%

NSR-Pyrite values were calculated as follows:

$$\text{NSR-Pyrite } \$/t = [(\text{Au g/t} \times \$29.28) + (\text{Ag g/t} \times \$0.18)] / (\text{Fe}\% \times 0.0246)$$

The criteria and thresholds used for the categorization of the three stockpiles are illustrated in Figure 16-11. and can be summarized as follows:

- If the NSR-Mine is less than \$100/t, the material is placed in the low-grade stockpile.
- If the NSR-Mine is greater than \$100/t and the NSR-Pyrite is less than \$160/t, the material is placed in the low-grade pyrite stockpile.

- If the NSR-Mine is greater than \$100/t and the NSR-Pyrite greater than \$160/t, the material is placed in the high-grade pyrite stockpile.

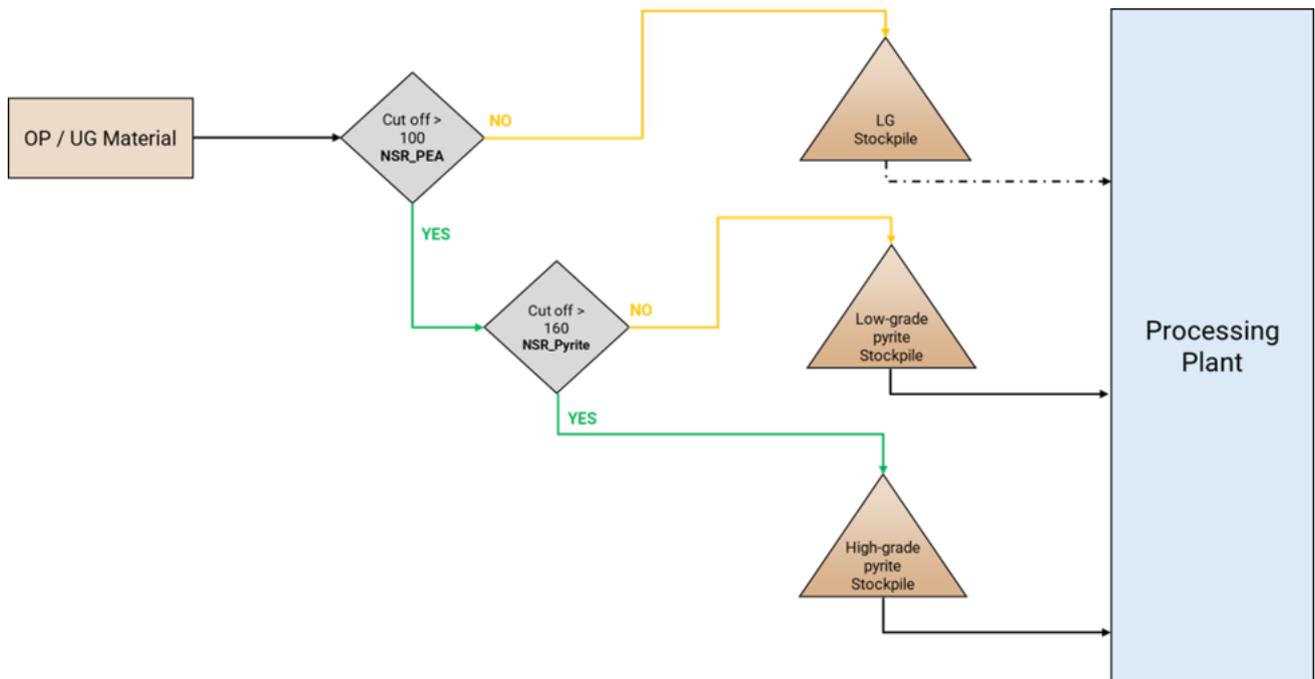


Figure 16-11 Stockpiles Classification Parameters

16.9.1 Pre-Production Plan

Production is scheduled to begin early 2029, during the construction phase of the project (“year 0” of the financial model). As such, pre-production activities will need to commence sufficiently ahead of time to allow for critical early works. The principal activities during the pre-production period will include installation of roads, terrasses and surface stockpile. Pre-stripping for the OPs will be executed by a contractor. The waste rock dump facilities will need to be prepared before the stripping can start.

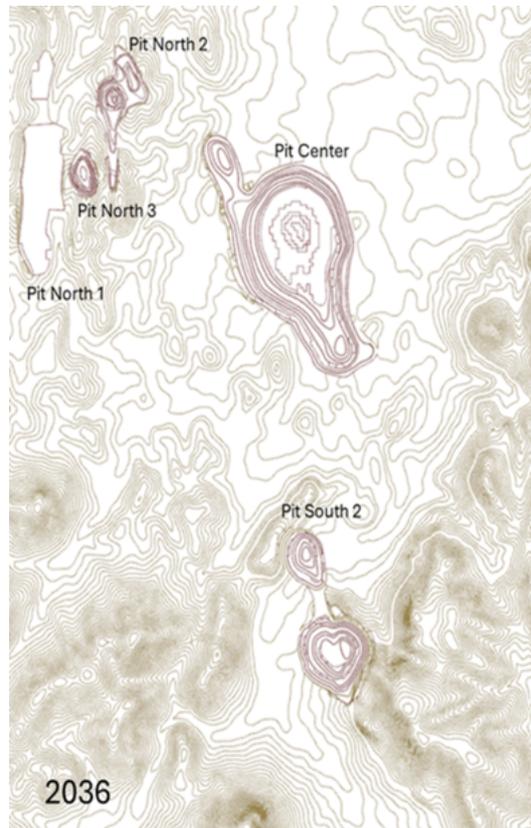
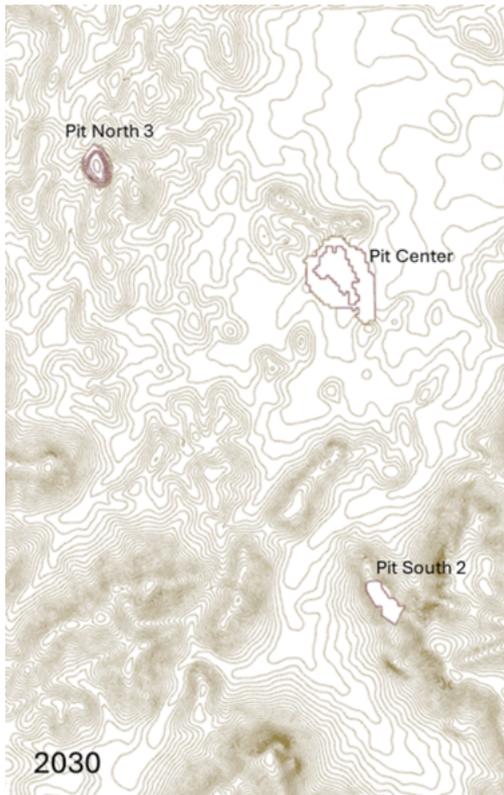
16.9.2 OP Production Plan

OP mining will begin in three primary areas Pit Center, Pit South 2, and Pit North 3. The Pit Center, which contains higher-grade material, will be the main pit and will be mined first, starting in 2029. Mining of this pit will continue for approximately eight (8) years. In parallel, Pit South 2 and Pit North 3 will be developed more gradually to supplement production from the Pit Center.

Pit South 2 is expected to be mined over a period of six (6) years, while Pit North 3 will be mined only during the first year of the OP operation. Subsequent phases of OP mining will include Pit North 1 and Pit North 2, which are scheduled to be mined between 2034 and 2037.

OP production will conclude with Pit South 1, which is expected to be completed in 2038. The yearly production, all material included, set for the pits in the schedule is 50Mt per year.

The open pits sequence is illustrated in figures 16-12 a-b-c-d-e-f.



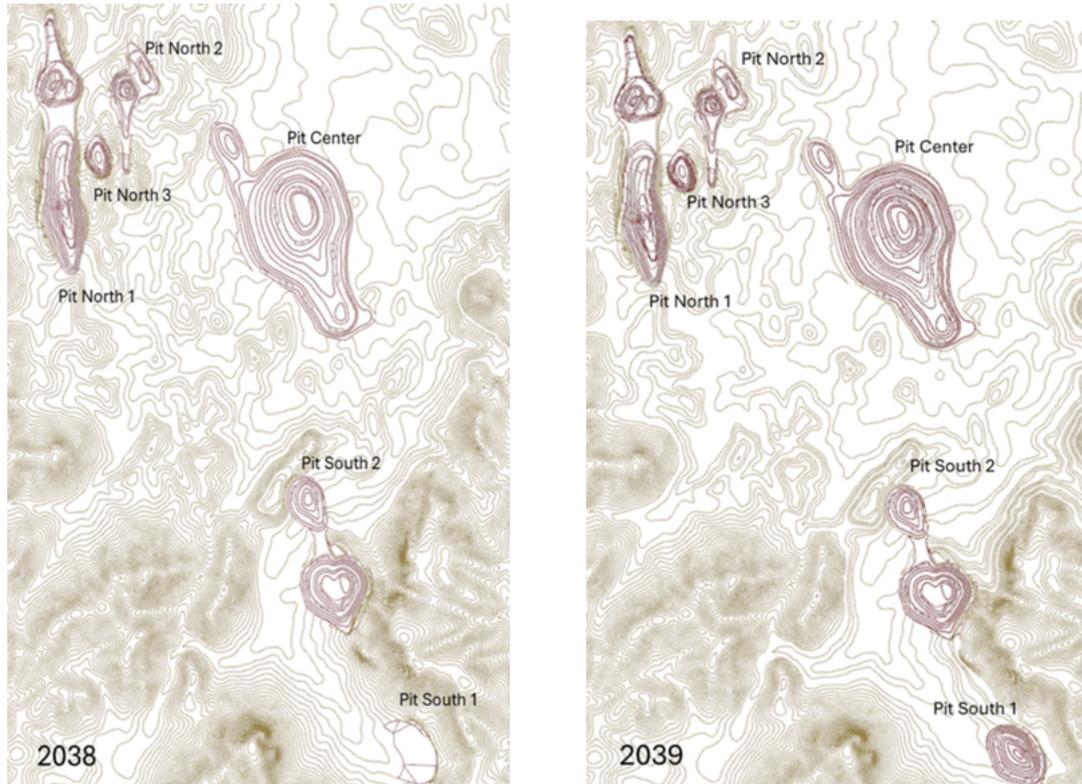


Figure 16-12 Open Pit Sequence from 2030 to 2039

Figure 16-12 Description: Surface view of open pit circuit for the following years : (a) 2030 (b) 2032 (c) 2034 (d) 2036 (e) 2038 (f) 2039

Table 16-13 provides a breakdown of the OP LOM production by stockpile and Table 16-14 provides a breakdown of the OP LOM production by Pit zone.

Figure 16-13 represents the yearly distribution of the Open pit mineralized tonnes per Zone and Figure 16-14 represents the yearly distribution of the Open pit Waste tonnes per Zone.

Table 16-13 OP LOM Production by Stockpile

Description	Units	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	Row total
Total Pits												
Ore Tonnes	tonnes	1,997,787	3,588,632	2,025,290	3,788,908	1,340,316	1,037,757	1,394,911	2,261,544	1,460,085	546,525	19,441,753
NSR Average	\$/tonnes	157.9	195.9	146.7	231.7	186.1	160.3	157.9	265.1	158.4	180.9	193.4
Low Grade Stockpile												
Ore Tonnes	tonnes	670,435	629,485	763,365	555,725	248,140	119,720	218,587	221,982	194,468	11,893	3,633,801
NSR Average	\$/tonnes	77.8	81.2	74.8	74.4	70.8	74.1	75.3	77.2	72.0	74.7	76.1
Low Grade Pyrite Stockpile												
Ore Tonnes	tonnes	484,911	1,282,317	650,908	1,095,540	555,242	504,769	702,257	533,046	1,058,604	484,916	7,352,510
NSR Average	\$/tonnes	123.9	124.6	135.5	137.8	142.8	142.2	140.0	126.0	154.4	181.7	139.7
High Grade Pyrite Stockpile												
Ore Tonnes	tonnes	842,442	1,676,829	611,017	2,137,643	536,933	413,269	474,066	1,506,516	207,012	49,716	8,455,443
NSR Average	\$/tonnes	241.3	293.4	248.5	320.7	284.3	207.4	222.4	342.0	259.6	198.4	290.4
Waste Tonnes	tonnes	18,000,646	36,410,341	47,974,382	46,208,802	48,659,423	48,975,925	48,613,949	47,744,026	45,938,677	18,317,528	406,843,699
Strip Ratio		9.0	10.1	23.7	12.2	36.3	47.2	34.9	21.1	31.5	33.5	20.9
Total Tonnes	tonnes	19,998,434	39,998,972	49,999,673	49,997,710	49,999,738	50,013,682	50,008,859	50,005,569	47,398,762	18,864,053	426,285,452

Table 16-14 OP LOM Production by Zone

Description	Units	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	Row total
Total Pits												
Ore Tonnes	tonnes	1,997,787	3,588,632	2,025,290	3,788,908	1,340,316	1,037,757	1,394,911	2,261,544	1,460,085	546,525	19,441,753
NSR Average	\$/tonnes	157.9	195.9	146.7	231.7	186.1	160.3	157.9	265.1	158.4	180.9	193.4
Waste Tonnes	tonnes	18,000,646	36,410,341	47,974,382	46,208,802	48,659,423	48,975,925	48,613,949	47,744,026	45,938,677	18,317,528	406,843,699
PIT CENTER												
Ore Tonnes	tonnes	1,774,628	3,156,451	1,432,820	2,866,656	831,929	555,290	1,143,980	2,025,805	53,846	-	13,841,404
NSR Average	\$/tonnes	150.1	188.3	115.6	238.8	144.4	143.1	155.5	280.9	519.4	-	194.0
Waste Tonnes	tonnes	13,225,060	21,842,338	33,565,946	32,133,091	39,165,856	39,444,178	38,852,557	37,971,859	956,613	-	257,157,498
PIT NORTH 3												
Ore Tonnes	tonnes	222,933	-	-	-	-	-	-	-	-	-	222,933
NSR Average	\$/tonnes	220	-	-	-	-	-	-	-	-	-	220.2
Waste Tonnes	tonnes	3,749,869	-	-	-	-	-	-	-	-	-	3,749,869
PIT SOUTH 2												
Ore Tonnes	tonnes	227	432,181	592,470	922,252	508,387	370,448	-	-	-	-	2,825,965
NSR Average	\$/tonnes	72	251	222	210	254	194	-	-	-	-	224.5
Waste Tonnes	tonnes	1,019,294	14,568,002	14,408,436	14,075,712	9,493,567	3,895,102	-	-	-	-	57,460,113
PIT NORTH 2												
Ore Tonnes	tonnes	-	-	-	-	-	112,019	148,512	-	-	-	260,531
NSR Average	\$/tonnes	-	-	-	-	-	136	191	-	-	-	167.5
Waste Tonnes	tonnes	6,424	-	-	-	-	5,636,645	2,548,245	-	-	-	8,191,314
PIT NORTH 1												
Ore Tonnes	tonnes	-	-	-	-	-	-	102,419	235,738	1,395,392	-	1,733,549
NSR Average	\$/tonnes	-	-	-	-	-	-	136	129	144	-	141.8
Waste Tonnes	tonnes	-	-	-	-	-	-	7,213,148	9,772,166	42,832,064	-	59,817,378
PIT SOUTH 1												
Ore Tonnes	tonnes	-	-	-	-	-	-	-	-	10,847	546,525	557,372
NSR Average	\$/tonnes	-	-	-	-	-	-	-	-	175.3	180.9	181
Waste Tonnes	tonnes	-	-	-	-	-	-	-	-	2,150,001	18,317,528	20,467,529

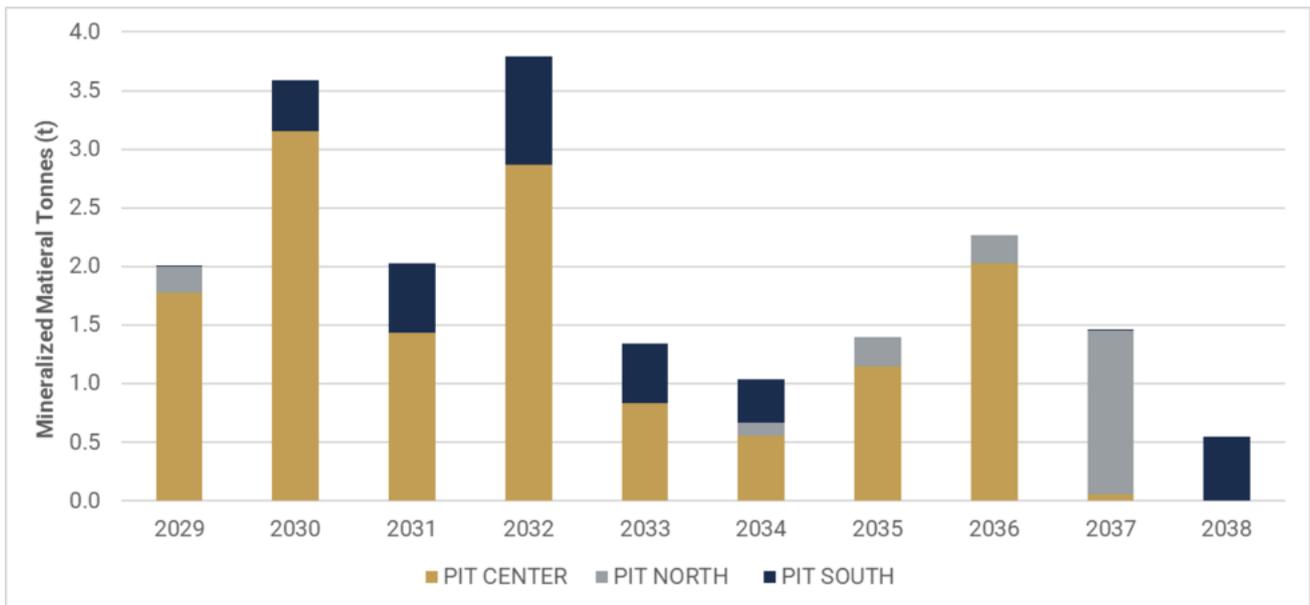


Figure 16-13 Open pit Mineralized tonnes per year and per Zone

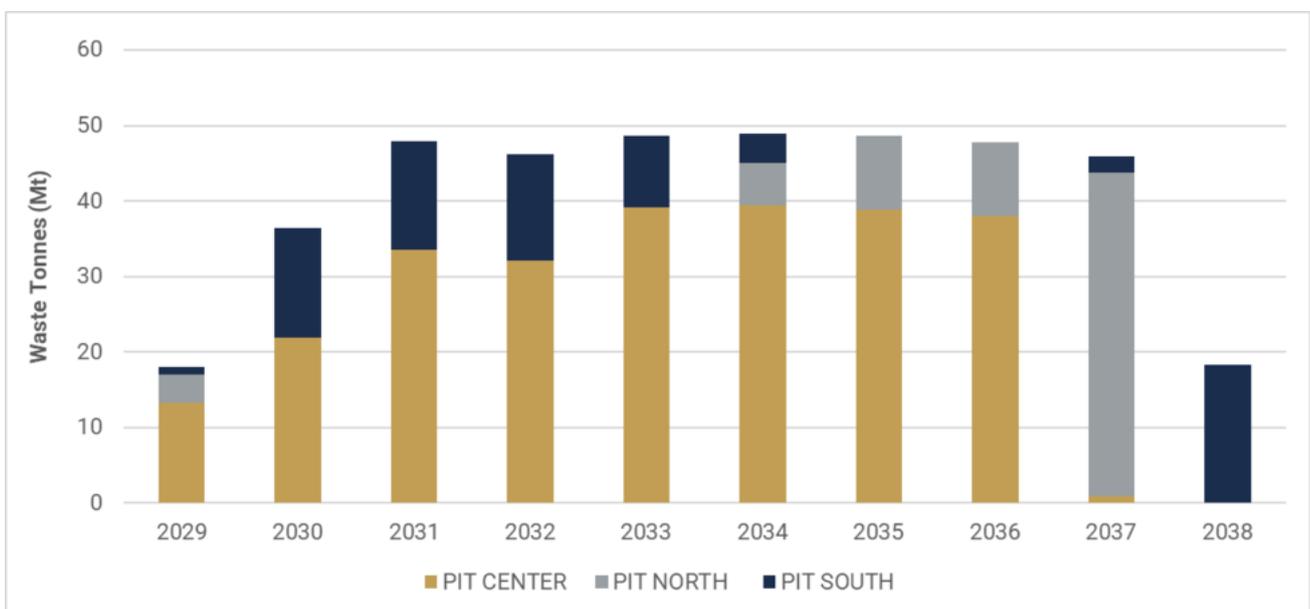


Figure 16-14 Open pit Waste tonnes per year and per Zone

16.9.3 UG Development Plan

Development of the UG facility is scheduled to begin in 2031 and will require approximately one year to reach full production capacity. All development activities will be executed by a contractor. The pre-production phase for the UG operations will include equipment procurement, excavation of terraces for the portal and ventilation shafts, and the recruitment of qualified mining personnel to support the commencement of UG mining activities.

All UG mining zones in the Boumadine deposit will be accessed by developing spiral ramp either from the surface or from existing, developed pits. Ramps will be utilized for:

- Production haulage
- Material and personnel movement

- Fresh air intakes for the mine
- Main dewatering path for the mine

Ramps will provide access to all levels of the mine and, subsequently, to the production sublevels, which will be spaced at 20-m intervals. Return air ventilation raises, excavated in parallel with the ramp, will connect the sublevels and be equipped with ladderways, offering an emergency escape route as an alternative to the main ramp.

Additional supporting infrastructure are located throughout the mine such as dewatering sumps, main pumping station, electrical substations, explosive magazine, emulsion bay, fuel bay and backfill stations.

Vertical development will consist of a surface ventilation raise, which will be excavated using raise boring. In addition, ventilation return air raises connecting the sublevels will be constructed and equipped with ladderways to serve as secondary egress routes. Rock passes will also be developed in selected areas and will be excavated using conventional raising methods. Principal development headings are presented in Figure 16-15.

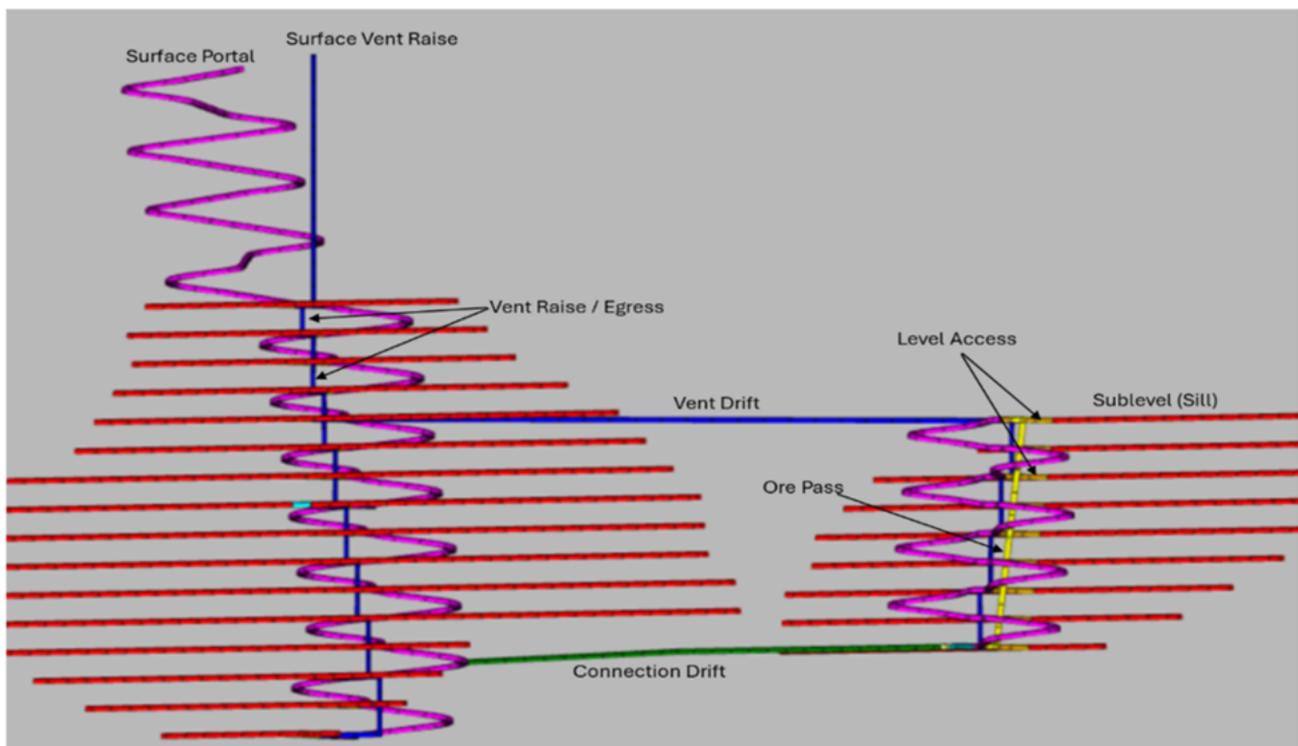


Figure 16-15 Principal Development Headings (North Zone Area)

The lateral development rate used for the project is 8 m per day and was based on performance data from the Zgounder Mine, reflecting the multiple face advance rate. The vertical advance rate was set at 3 m per day.

For development material, a dilution factor of 0% and a recovery rate of 100% were assumed. The NSR cut-off grade applied was \$77 per tonne.

Table 16-15 presents the LOM development plan summarized by development type and separated into Capital and Operating.

Table 16-15 Life-of-Mine Development Plan

Description	Units	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	Row total
Lateral Development															
Ramp	m	-	-	5,444	2,722	1,218	4,751	3,485	5,477	2,785	2,539	7,434	2,708	-	38,561
Level Access	m	-	-	782	945	1,159	1,416	1,981	1,538	1,427	1,273	1,748	1,995	-	14,264
Connection Drift	m	-	-	358	428	-	417	66	319	160	730	324	-	-	2,802
Sublevel (Sill)	m	-	-	599	6,818	8,799	6,994	6,215	6,205	8,852	8,410	5,707	8,435	323	67,358
Ventilation Drift	m	-	-	763	348	453	938	229	1,256	275	846	924	985	-	7,018
Dev Infra	m	-	-	111	78	51	69	99	138	88	126	192	131	-	1,082
Total Lateral Meters	m	-	-	8,057	11,340	11,680	14,585	12,075	14,934	13,586	13,924	16,330	14,255	323	131,086
Verical Development															
Ventilation Raises	m	-	-	548	421	180	467	630	476	405	325	295	1,031	39	4,816
Ore Pass	m	-	-	-	-	162	-	-	-	-	-	-	80	-	242
Total Vertical Meters	m	-	-	548	421	342	467	630	476	405	325	295	1,111	39	5,058
Development Cost Type															
CAPEX Dev	m	-	-	8,005	4,942	3,222	8,058	6,490	9,204	5,139	5,838	10,918	6,931	39	68,787
OPEX Dev	m	-	-	599	6,818	8,799	6,994	6,215	6,205	8,852	8,410	5,707	8,435	323	67,358
Total Devl Meters	m	-	-	8,604	11,760	12,022	15,052	12,705	15,409	13,991	14,248	16,625	15,366	362	136,144

16.9.1 UG Production Plan

The UG production sequence will follow the same strategy as the OP, by prioritizing high-grade zones. For this reason, UG mining will begin in 2031 simultaneously in the Center and North zones.

One more key reason for initiating UG production in the Center zone is to access and extract the high-grade stopes located beneath the Center zone. These stopes represent some of the richest portions of the deposit and starting them early ensures that this material is not sterilized or lost due to OP operations above.

As the UG operations will be conducted in parallel with OP mining for these zones, the decision has been made to start the access ramps to these zones from the surface independently of pit operations. This approach ensures timely access to UG stopes without being constrained by the OP mining sequence.

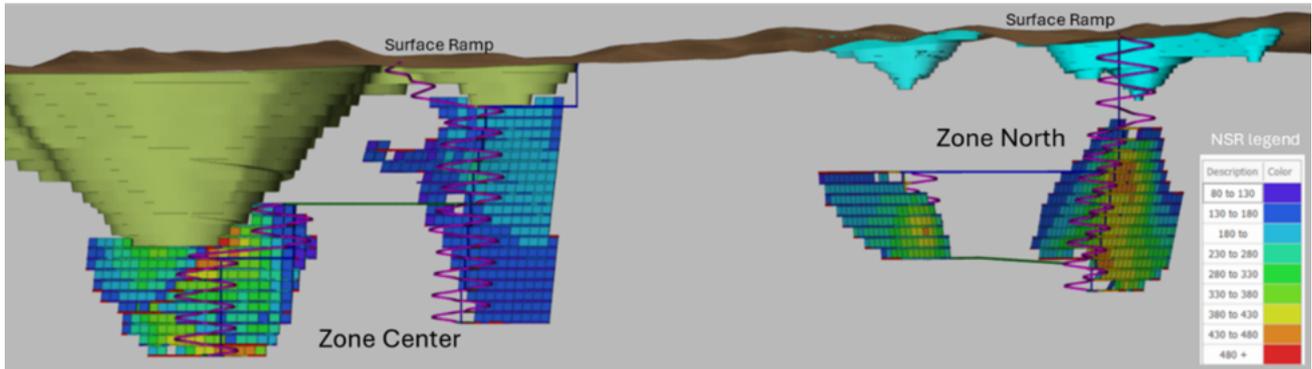


Figure 16-16 Primary UG Mining Area in the Center and North Zones

The start of UG production in the remaining areas of the Center and North zones, as well as in the South zone, is linked to the production schedule of the corresponding OPs, as access to these UG zones will be gained through the pits once they are sufficiently developed. The portals of these declines will be excavated in the pit walls.

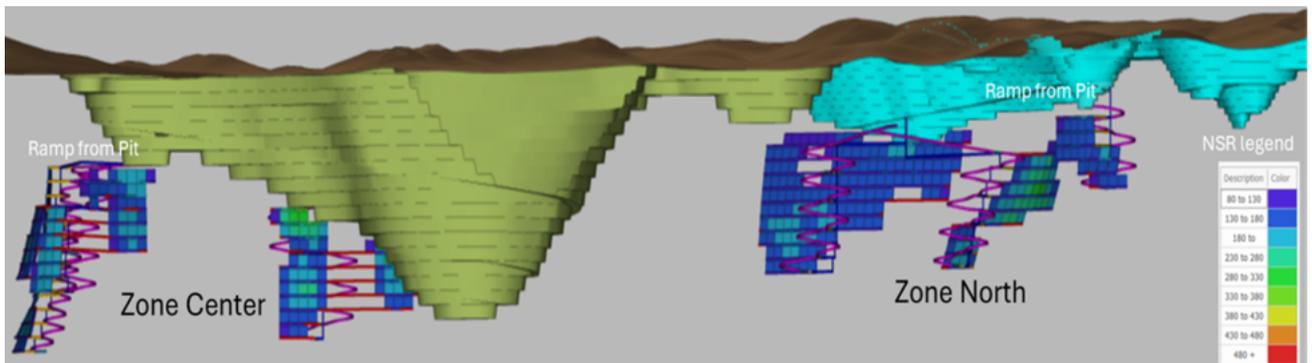


Figure 16-17 Secondary UG Mining Area in the Center and North Zones

Production will start in 2034 in the South zones and in 2039 in the remaining North zones.

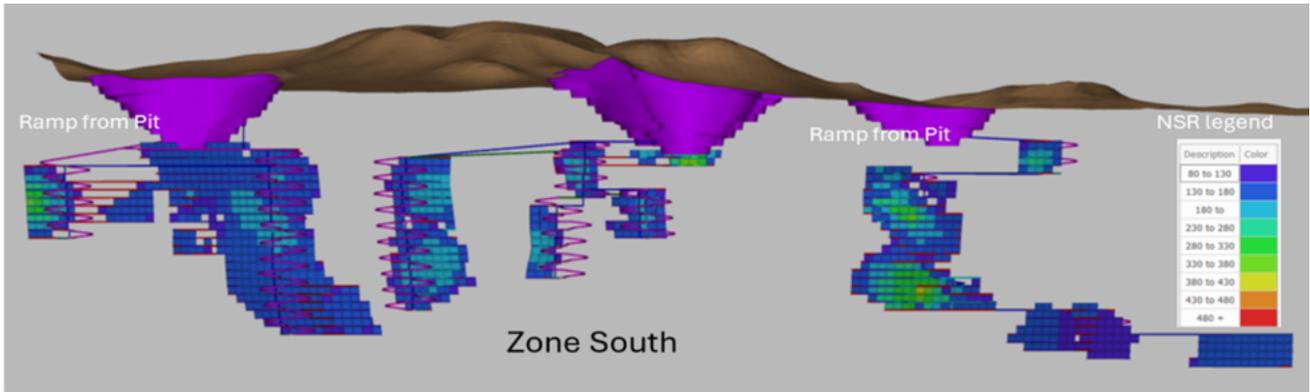


Figure 16-18 Remaining UG Mining Area in the South Zone

In zones where depth and mine design allow, the deposit has been divided into two or more mining horizons, separated by a sill pillar level. The first mineable level in the horizon will be backfilled with Cemented Rock Fill (CRF) upon completion, enabling safe and stable mining operations to proceed below it.

The stope production cycle time was estimated based on a typical stope with an average width of 2.5 m, a height of 16 m (floor to back), and a length of 20 m. using an average mineralized material density of 3.7 t/m³, a dilution of 5% and a recovery of 95%, the average stope tonnage is 2510 tonnes, to account for broken rock volume increase a swell factor of 36.5% was used in the calculation. Results are presented in Table 16-16.

Table 16-16 Stope Production Cycle Time

Activity	Duration	Rate	Unit
Stope Preparation	2		d/stp
Production Drilling		345	m/d
Blasting	0.56		d/stp
Mucking		1 530	t/d
Backfill Prep	1		d/stp
Rock filling		1 637	t/d
Stope cycle time	9.1		day
Tonnes per stope per day		276	t/d/stope

The overall UG production plan is presented in Table 16-17 and Table 16-18.

Table 16-17 shows the Boumadine UG production plan sorted by stockpile category.

Table 16-17 Boumadine UG Production Plan Sorted by Stockpile Category

Description	Units	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	Row total
Total Underground															
Total Mineralized Tonnes	tonnes	-	-	30,784	845,409	1,580,951	1,653,772	1,181,957	991,179	1,093,573	1,599,170	1,053,730	1,292,922	316,411	11,639,858
Total NSR	\$/t	-	-	242.0	262.6	276.8	235.4	155.0	159.6	150.3	156.5	138.6	156.2	163.5	190.0
Dev Mineralized Tonnes	tonnes	-	-	30,784	312,566	400,612	294,061	312,774	256,135	335,300	338,979	253,871	336,986	18,394	2,890,462
Dev NSR	\$/t	-	-	242.0	254.4	238.6	163.5	135.4	136.1	119.8	133.5	120.1	126.1	142.0	162.2
Stope Mineralized Tonnes	tonnes	-	-	-	532,844	1,180,339	1,359,711	869,183	735,043	758,273	1,260,191	799,859	955,936	298,018	8,749,396
Stope NSR	\$/t	-	-	-	267.4	289.8	250.9	162.0	167.8	163.8	162.7	144.5	166.8	164.8	199.2
Low Grade Stockpile															
Mineralized Tonnes	tonnes	-	-	2,093	14,550	24,386	39,114	46,102	56,743	115,669	53,285	87,818	73,480	2,810	516,051
NSR Average	\$/t	-	-	95.0	86.4	91.0	88.8	91.8	88.5	88.3	90.1	90.9	90.7	87.1	89.7
Low Grade Pyrite Stockpile															
Mineralized Tonnes	tonnes	-	-	8,474	168,737	317,272	607,303	991,800	348,161	732,970	1,119,553	729,820	708,518	201,517	5,934,123
NSR Average	\$/t	-	-	139.4	152.0	155.6	170.6	156.3	154.7	142.6	144.0	137.4	145.9	156.3	149.9
High Grade Pyrite Stockpile															
Mineralized Tonnes	tonnes	-	-	20,218	662,122	1,239,293	1,007,355	144,055	586,274	244,934	426,333	236,092	510,924	112,084	5,189,684
NSR Average	\$/t	-	-	300.2	294.7	311.5	280.2	166.4	169.4	202.8	197.6	159.9	179.8	178.4	245.9
UG WASTE TONNES															
UG WASTE TONNES	tonnes	-	-	360,484	244,149	192,495	425,431	304,635	479,542	335,856	370,790	544,792	372,403	11,007	3,641,586
ROCKFILL TONNES															
ROCKFILL TONNES	tonnes	-	-	-	356,514	1,198,051	1,415,029	866,075	757,898	701,032	1,265,605	796,760	991,106	337,485	8,685,555
CRF TONNES															
CRF TONNES	tonnes	-	-	-	203,983	101,401	60,680	13,828	24,310	114,950	46,385	-	-	-	565,536

Table 16-18 Boumadine UG Production Plan Sorted by Mining Zone

Description	Units	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	Row total
Total Underground															
Total Mineralized Tonnes	tonnes	-	-	30,784	845,409	1,580,951	1,653,772	1,181,957	991,179	1,093,573	1,599,170	1,053,730	1,292,922	316,411	11,639,858
Total NSR	\$/t	-	-	242.0	262.6	276.8	235.4	155.0	159.6	150.3	156.5	138.6	156.2	163.5	190.05
ZONE CENTRE															
Mineralized Tonnes	tonnes	-	-	-	523,342	743,433	923,251	641,413	11,375	-	-	22,236	254,039	143,898	3,262,987
NSR Average	\$/t	-	-	-	263.9	264.1	223.1	158.0	179.9	-	-	184.7	177.0	158.8	219.3
ZONE NORTH															
Mineralized Tonnes	tonnes	-	-	30,784	322,067	837,518	698,699	-	-	-	-	141,472	560,636	130,762	2,721,939
NSR Average	\$/t	-	-	242.0	260.5	288.2	256.6	-	-	-	-	132.7	144.3	167.7	232.8
ZONE SOUTH															
Mineralized Tonnes	tonnes	-	-	-	-	-	31,822	540,543	979,803	1,093,573	1,599,170	890,022	478,246	41,752	5,654,932
NSR Average	\$/t	-	-	-	-	-	126.5	151.5	159.4	150.3	156.5	138.4	159.0	166.5	152.6

16.9.2 Combined Production Plan

The total mineralized material tonnage for the mine is estimated at 31.1 million tonnes, with an average NSR grade of 192.1 \$/t. OP mining will contribute with 62.6% of the total tonnage, while UG mining will account for the remaining 37.4%. In the final three years of the mine life, production will be exclusively from UG operations.

As the total tonnage from the OPs and UG has not been fully balanced, the strategy is to use the different stockpiles to regulate and optimize the feed to the processing plant, ensuring a consistent tonnage and grade profile over the LOM. Boumadine production plan is summarized in Table 16-19, Figure 16-19 and Figure 16-20.

Table 16-19 Boumadine LOM Production Plan

Description	Units	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041	Row total
Total Pits															
Ore Tonnes	tonnes	1,997,787	3,588,632	2,025,290	3,788,908	1,340,316	1,037,757	1,394,911	2,261,544	1,460,085	546,525	-	-	-	19,441,753
NSR Average	\$/tonnes	157.9	195.9	146.7	231.7	186.1	160.3	157.9	265.1	158.4	180.9	0.0	0.0	0.0	193.4
Total Underground															
Ore Tonnes	tonnes	-	-	30,784	845,409	1,580,951	1,653,772	1,181,957	991,179	1,093,573	1,599,170	1,053,730	1,292,922	316,411	11,639,858
NSR Average	\$/t	-	-	242.0	262.6	276.8	235.4	155.0	159.6	150.3	156.5	138.6	156.2	163.5	190.1
Mine Total															
Ore Tonnes	tonnes	1,997,787	3,588,632	2,056,075	4,634,317	2,921,266	2,691,529	2,576,867	3,252,722	2,553,657	2,145,695	1,053,730	1,292,922	316,411	31,081,611
NSR Average	\$/t	157.9	195.9	148.2	237.3	235.2	206.5	156.6	233.0	154.9	162.7	138.6	156.2	163.5	192.1

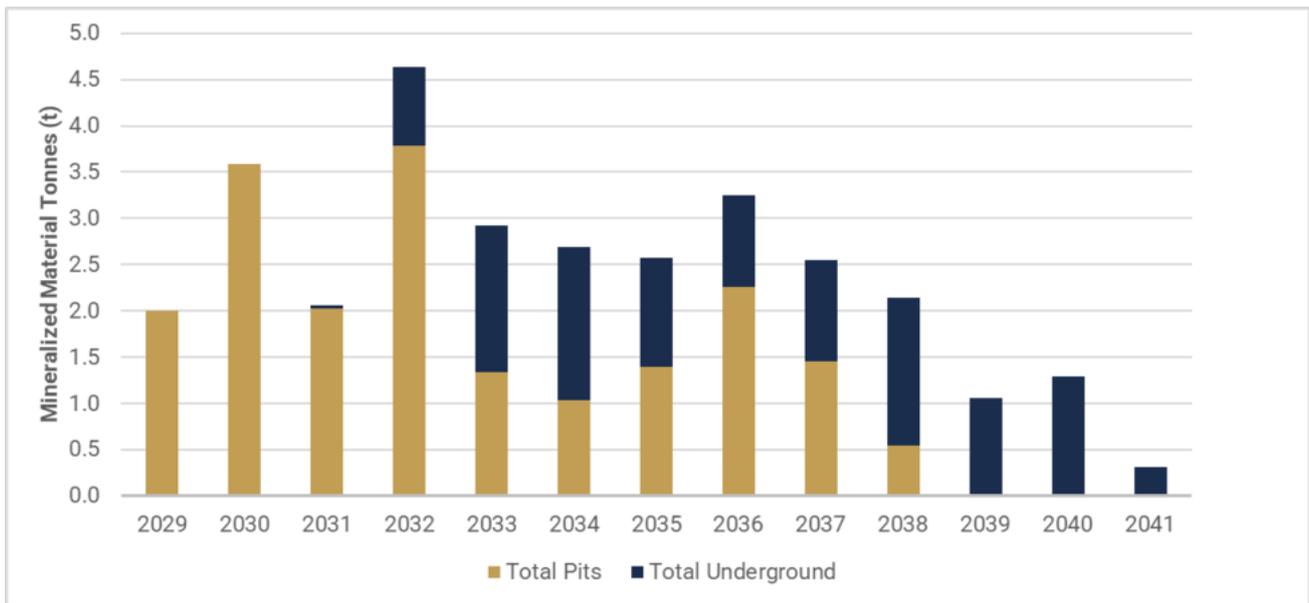


Figure 16-19 Boumadine LOM Production Plan breakdown by OP and UG Mining Methods

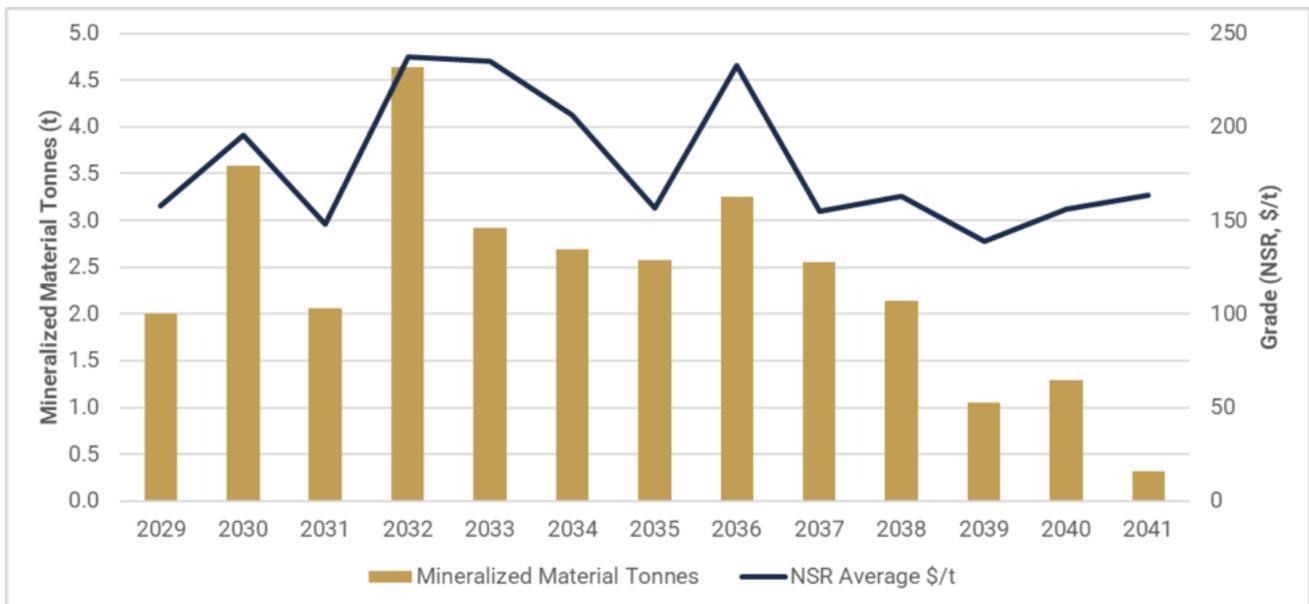


Figure 16-20 Boumadine LOM Production Plan Tonnes and NSR

16.10 ELECTRICAL UG DISTRIBUTION

The electrical distribution for the underground mine’s medium voltage is planned from the portals and service boreholes to provide a redundancy loop. Main substations equipped with 1.5-MW transformers and low voltage distribution switchgear are planned for every three levels. These substations are expected to be skid-mounted, featuring standard equipment and distribution breakers. As underground mine development progresses, some main substations will be relocated from completed areas to newly developed zones.

Low voltage will be supplied via boreholes from the main substations to both upper and lower levels. Each level will therefore contain an electrical bay with distribution panels and motor starters to supply equipment and services, including jumbo drills, fans, pumps, lighting, and power outlets. All equipment must be certified by the appropriate regulatory bodies and allow for padlocking to ensure safe zero-energy maintenance.

Electrical connections for the underground mine and surface ventilator feed will come from the surface overhead line.

For communications, a robust fiber optics cable system will be installed during mine development, ensuring high-speed and reliable data transmission throughout all underground operations. This advanced network will be integrated with patch panels and junction boxes located in each electrical bay, providing effective connectivity for underground mine communications. The fiber optics infrastructure will allow real-time monitoring, remote equipment operation, and emergency response capabilities, enhancing both operational efficiency and safety across the mine. Additionally, the system is designed for scalability, allowing future upgrades to accommodate evolving technology and increased data demands as mine development progresses.

16.11 VENTILATION

This section of the report outlines the ventilation design and strategy for the Boumadine Project, based on the proposed mine layout, production schedule, and diesel equipment fleet. The design aims to ensure safe and efficient airflow distribution across all zones of the mine, while optimizing energy consumption and fan performance. All calculations and design assumptions comply with Canmet (Canada, 2021) ventilation standards. MSHA (MSHA, n.d.) will be used in case Canmet standards are not available. In the absence of specific standards, a default airflow rate of 0.06 m³/s/kW will be used as per industry standard. Additional assumptions are defined for diesel engine ventilation:

- New vehicles: Tier 4 emissions standard
- Contractor fleet: Tier 2

16.11.1 Design Criteria

Ventilation demand is based on the equipment fleet used throughout the LOM. Airflow requirements per vehicle type were calculated assuming the following assumptions:

- LHD 10T (ST1030): Tier 2, Canmet requirements
- Mine truck 30T (MT46): Tier 3, Canmet requirements
- Remainder of fleet: 0.06 m³/s/kW

Using the equipment fleet, the number of equipment and utilization factor to capture the worst-case scenario, the airflow demand for the project is shown in Table 16-20 for a total of 413 m³/s. Utilization factor for LHDs and trucks have been assumed at 100%, 50% for most of utility vehicles and 25% for equipment that primarily operates on electric such as Jumbos and production drills.

Table 16-20 Airflow Requirements Calculations

Mining Equipment airflow requirement					
Equipment	Engine kW	No. of units	Utilization rate	Requirement per unit m ³ /s	Total requirement m ³ /s
Drilling & Blasting					
Prod Drill, ITH w/ Raise Head	129	12	25%	7.7	6
Emulsion Charger	120	8	35%	7.2	5
Mucking & Haulage					
30 tonnes Mine Truck Tier 3	350	9	100%	10.8	65
10 tonnes LHD 10 t	256	9	100%	11.2	78
Ground Support					
Cable Bolter	119	12	25%	7.1	4
Transmixer	185	9	25%	11.1	3
Service					

Mining Equipment airflow requirement					
Equipment	Engine kW	No. of units	Utilization rate	Requirement per unit m ³ /s	Total requirement m ³ /s
Telehandler	75	4	25%	4.5	5
Boom trucks	120	7	25%	7.2	4
Grader	123	15	100%	7.4	7
Fuel Lube Truck	126	8	50%	7.6	8
Water Truck	126	8	50%	7.6	4
Mobile rock breaker	200	11	25%	12	3
Pickup Truck	150	5	50%	9	68
Mine rescue Truck	150	16	0%	9	0
Development					
Jumbo		3	25%	7	5
Scissor Lifts		3	50%	7	11
Boom Truck		3	50%	7	11
Shotcrete Sprayer		1	25%	7	2
Pickup Truck		6	50%	9	27
LHD		3	100%	8	24
Truck		6	100%	12.7	76
				Total	413

Other design criteria for the ventilation design are defined as follows:

- Maximum air velocity in ramps: 7 m/s
- Main fan pressure limit: 5000 Pa to avoid high pressures in return air raise system for safety and bulkhead design purposes to access the egress.
- Fan efficiency of main fans: 75%
- Fan efficiency of auxiliary fans: 80%
- Mains fans power utilization rate: 80%.
- Motor efficiency: 95%
- Heating or cooling of mine air is not required.
- Any open stopes are sealed with a ventilation curtain
- Optimize blast clearing times
- Leakage factor: 50 mm²/m² for auxiliary duct network
- Hydraulic resistance of closed regulator and personnel doors bulkheads: 15 N s²/m⁸
- When airflow is divided between two raises on a given zone, a 70/30% split is assumed for worst-case pressure balance.
- Electricity cost: \$US 90/MWh
- Main fans operate 365 days/year, 24 hours/day
- Auxiliary fans operate 365 days/year, 24 hours/day with On/Off starters only

16.11.2 Ventilation Demand and Distribution

The mine is divided in the ventilation zones shown in Figure 16-16. Each ventilation zone share the same main fans and are connected UG.

- Zone South 1

- Zone South 2
- Zone South 3
- Zone South 4
- Zone South 5
- Zone Center
- Zone North 1
- Zone North 2
- Zone North 3

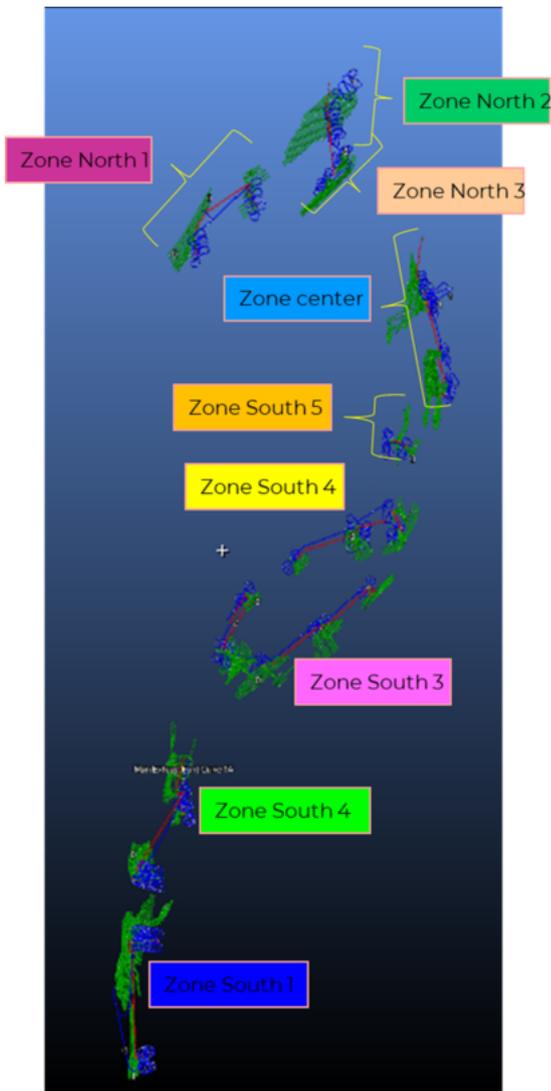


Figure 16-21 Ventilation Zones

Based on the mine plan, the annual mining activity proportion per ventilation zones in terms of development and production was estimated as shown in Table 16-21. The airflow distributed for each zone is assumed to be proportional to the mining activity split by multiplying it to the total airflow requirement of 413 m³/s. A contingency of 20% contingency is then added to ensure total airflow meets worst-case operational scenarios. The maximum airflow requirement per zone is shown in Table 16-22. The main fans sizing has been performed based on this airflow requirement.

Table 16-21 Production Profile Based On Ventilation Zones

Year/zone	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041
Nord 1											30%	30%
Nord 2	50%	50%	50%	50%	50%							
Nord 3											10%	10%
Centre	50%	50%	50%	50%	50%							
Sud 5											30%	30%
Sud 4											30%	30%
Sud 3						5%	5%	50%	50%	50%		
Sud 2						90%	90%					
Sud 1						5%	5%	50%	50%	50%		

Main fans are sized to meet the maximum airflow requirements defined by the production profile, ensuring adequate ventilation capacity across all operational zones. The fan pressure has been calculated using the Ventsim™ model assuming that the air is sent to the lower levels while allowing leakage at each of the bulkheads with a hydraulic resistance of 15 N s²/m⁸ and a 70/30% split if the Return Air Raise (RAR) system is divided. A pressure of 500 Pa is added to each of the operating point to account for the fan installation pressure losses. The power is then estimated with the assumed fan efficiency.

Table 16-22 Main Fans Sizing

Zone	Max % of prod.	Max airflow req. (m ³ /s)	Fan pressure	Fan type	Nominal motor power (kW)
North 1	50%	206	1882	Surface horizontal parallel arrangement	2 x 300
North 2	70%	288	2671	Surface horizontal parallel arrangement	2 x 600
North 3	30%	124	1379	Surface horizontal parallel arrangement	2 x 150
Center	70%	288	4367	Surface horizontal parallel arrangement	2 x 1000
South 5	50%	206	2417	Surface horizontal parallel arrangement	2 x 400
South 4	50%	206	2009	Surface horizontal parallel arrangement	2 x 350
South 3	70%	288	4270	Parallel Horizontal fans in portal in bulkhead	2 x 1000
South 2	1	412	2930	Parallel Horizontal fans in portal in bulkhead	2 x 1000
South 1	0.7	288	3561	Surface horizontal parallel arrangement	2 x 850

16.11.3 Main Ventilation system Design

The main ventilation system design for a typical zone is shown in Figure 16-17. Each zone has a single intake portal and is connected to return air network converging toward the surface main exhaust fans. This design was chosen to optimize blast clearing times. The airflow is distributed to the different ramps and production levels with the use of bulkheads and drop board regulators. The drop board regulators are adjusted manually to supply enough air on each of the levels. When a level becomes inactive, the regulator is completely sealed. Zones Nord 1, Nord 2, Nord 3, Centre, Sud 1, and Sud 5 have one exhaust raise with surface horizontal parallel arrangement fans (total of two fans per zone) as illustrated in Figure 16-17 and Zones Sud 2, Sud 3, and Sud 4 have one dedicated exhaust drift connected to surface with parallel UG fans in wall arrangement (total of two fans per zone) as illustrated in Figure 16-18.

To comply with maximum airspeed and fan pressure limits, in addition to the intake portals, Zones Sud 1, Sud 2, Sud 3, Sud 5, Centre, and Nord 2 have each a new Fresh Air Raise (FAR), as shown in Figure 16-19. No fans are required for the FARs.

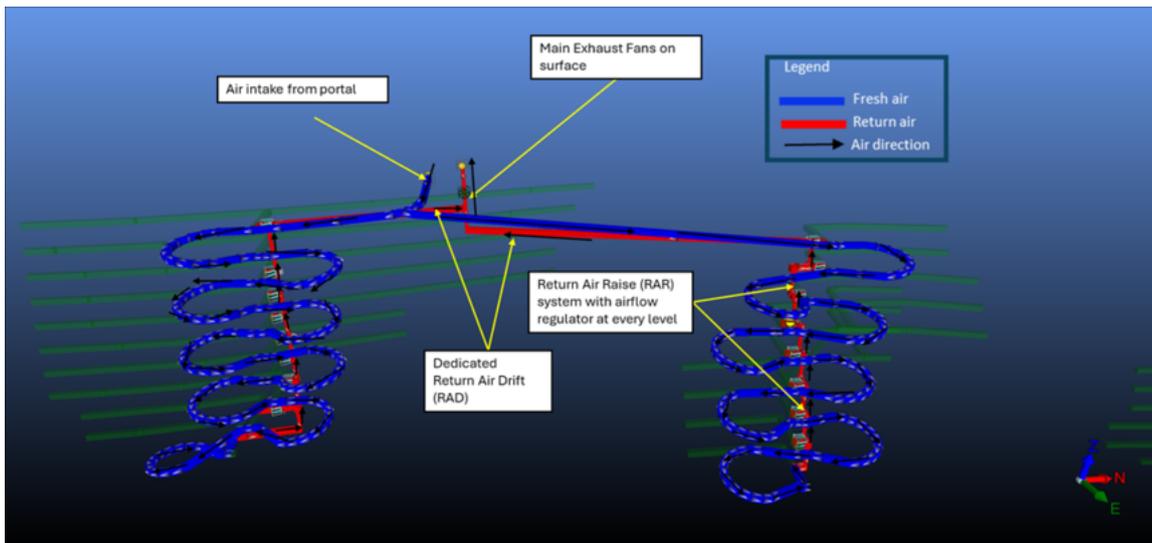


Figure 16-22 Main Ventilation Design Zone Nord 1

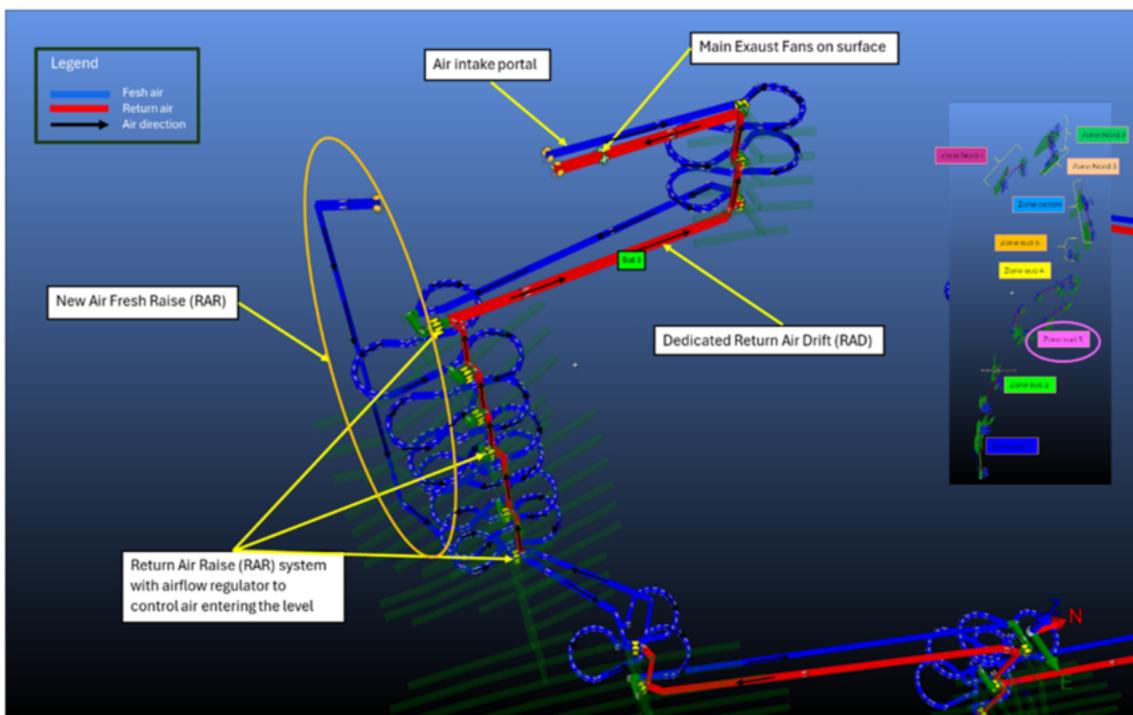


Figure 16-23 Main Ventilation Design Zone Sud 3

Raise sizing was performed based on the available sizes and to avoid having excessive fan power and pressure. All ventilation raises to surface will be raisebore type with maximum of 5.0 m diameter for the RARs. The FARs are raisebore type and have been sized at 3.0 and 4.0 m diameter to comply with the design requirements. All UG ventilation raise have been assumed as internal drop raises at a maximum size of 4.5 × 4.5 m and minimum of 3.0 m x 3.0 m. The RAR system is also used as an egress using laddertube type. The laddertube is maintained open to minimize the impact on the hydraulic resistance of the RAR system. Double personnel doors are therefore installed in each of the bulkheads for egress purposes.

The main fans are equipped with Variable Frequency Drives (VFDs), enabling speed modulation and energy optimization throughout the ventilation system. Main fans are assumed to operate at an average power utilization of 80% which is equivalent to an average speed reduction of 8%. The speed of the main fans will be modulated manually based on the expected fleet scheduled for the zone. The average annual power cost for the main fans is estimated at **\$US1.8 M**, assuming \$90/MWh electricity rate, as detailed in Table 16-23.

Table 16-23 Energy Consumption of Main Fans

Zone	Max % of prod.	Max airflow req. (m ³ /s)	Fan pressure	2030	2031	2032	2033	2034	2035	2036	2037	2038	2039	2040	2041
Nord 1	50%	206	1882											435	435
Nord 2	70%	288	2671	865	865	865	865	-							
Nord 3	30%	123	1379											191	191
Centre	70%	288	4367	1,414	1,414	1,414	1,414								
Sud 5	50%	206	2417											559	559
Sud 4	50%	206	2009											465	465
Sud 3	70%	288	4270					519	519	1,383	1,383	1,383	1,383		
Sud 2	100%	412	2930					1,355	1,355						
Sud 1	70%	288	3561					432	432	1,153	1,153	1,153	1,153		
Total (kW)				2279	2279	2279	2279	2306	2306	2536	2536	2536	2536	1650	1650
Annual (MWh)				19,964	19,964	19,964	19,964	20,204	20,204	22,214	22,214	22,214	22,214	14,458	14,458
Annual energy cost (\$M US)				\$1.797	\$1.797	\$1.797	\$1.797	\$1.818	\$1.818	\$1.999	\$1.999	\$1.999	\$1.999	\$1.301	\$1.301

16.11.4 Auxiliary Ventilation system Design

For both production and ramp development systems, a leakage factor of $50 \text{ mm}^2/\text{m}^2$ has been considered to account for ventilation losses. The systems operate with a fan efficiency of 65% and a motor efficiency of 95% to account for the difference in duct lengths based on the advanced rate and production levels geometry.

Ramp Development

For the ramp development, the system consists of two flexible ducts with a diameter of 1.0 m and a maximum length of 800 m. The ramp system will use the fresh air from the main ventilation network assuming that the RAR system will be connected at every five levels. 200 kW auxiliary fans have been sized for each duct system to supply $37 \text{ m}^3/\text{s}$ to support the operation of a truck, LHD, bolter, and pickup. Once the RAR is connected, the fans and duct would be moved to the new connection with the lower raise. It is assumed that a total of two ramp development systems are operating at any given time. The schematic of a typical arrangement for a ramp development ventilation system is shown in Figure 16-24.

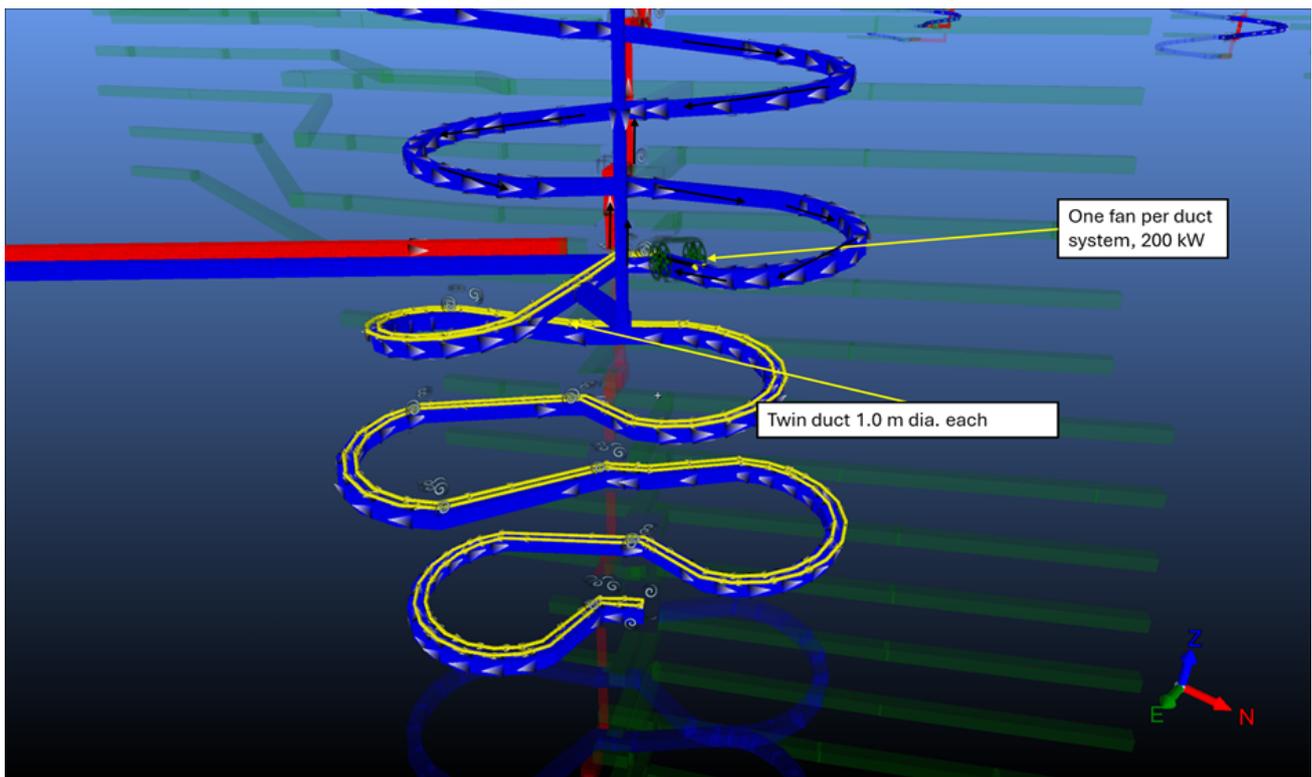


Figure 16-24 Auxiliary Ventilation on Ramp Development Zone Centre

Production Levels

Each active level is equipped with a 150-kW main auxiliary fan and a 15-kW auxiliary fan for secondary headings. The regulator opening is adjusted to supply enough fresh air for the auxiliary ventilation fans to avoid re-circulation while ensuring that the other active levels have sufficient air. Ventilation requirements include one primary heading (servicing LHD and truck) requiring $21 \text{ m}^3/\text{s}$ of airflow, and one secondary heading (LHD only) requiring $10 \text{ m}^3/\text{s}$. Flexible ducts have a diameter of 1.0 m, with average lengths of 300 m for main headings and 200 m for secondary headings. Across the mine, a total of eleven levels operate with fans at any given time; four active, four on standby, and three in development. For both production levels and ramp development, a leakage factor of $50 \text{ mm}^2/\text{m}^2$ is applied to estimate the pressure and airflow supplied to the working face with fan. The layout for the auxiliary ventilation system of a typical production level is shown in Figure 16-25.

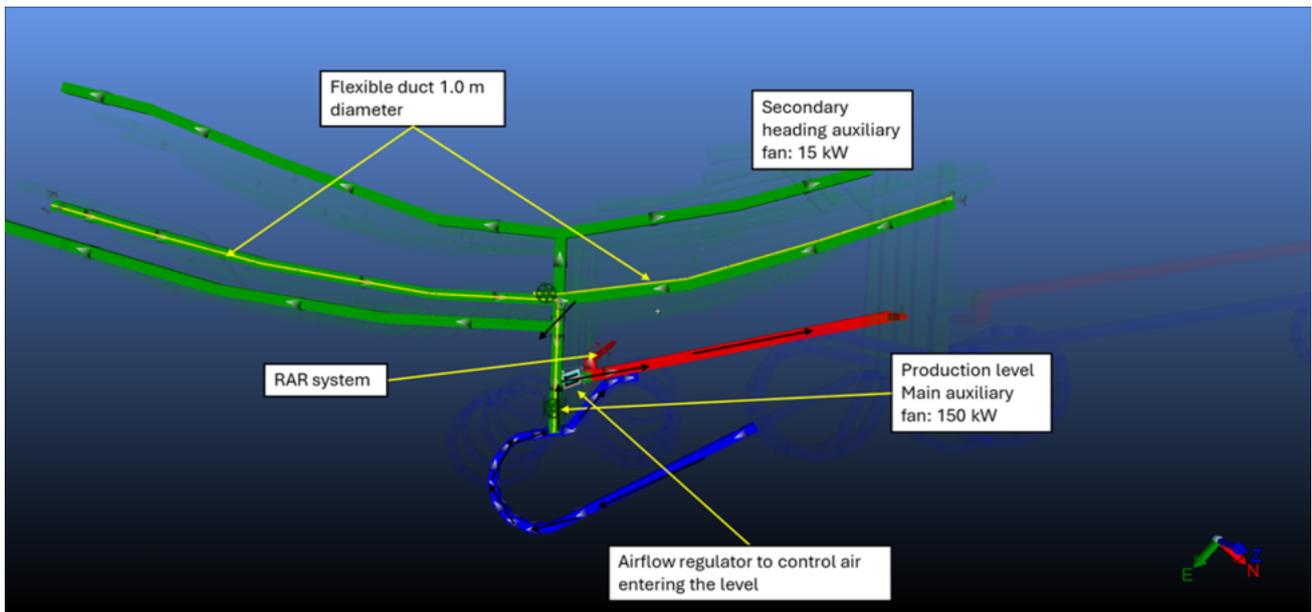


Figure 16-25 Auxiliary Ventilation on Production Levels Zone Centre

The annual energy cost for the auxiliary and ramp development fans is estimated at \$US1.9 M/yr, assuming \$90 /MWh electricity rate, as detailed in Table 16-24.

Table 16-24 Energy Consumption of Auxiliary and Ramp Development Fans

	Airflow (m ³ /s)	Pressure (Pa)	Consumed power per fan (kW)	Average number of fans operating	Annual energy consumption (MWh)	Annual energy cost (\$US)
Production level main auxiliary fan	21	4117	140	11	13,491	\$ 1,214,234
Production level secondary fan	10	656	11	11	1,024	\$ 92,131
Ramp development fan	18.5	6364	191	4	6,681	\$ 601,273
				Total	21,196	\$ 1,907,638

The combined annual energy cost for auxiliary and main ventilation systems is estimated at **\$US 3.7 M/yr**.

16.12 MINE BACKFILL

Given that the mining method adopted for the UG mine is modified Avoca, the main type of backfill will be rock fill using waste rock generated either from the UG mine development or from the OP operations. CRF will be used in the stopes located at the lowest extraction levels within each mining horizon. The use of CRF at these levels will provide sufficient ground support to enable the safe extraction of the overlying sill pillar stopes.

Rock fill will be placed in the stopes by direct dumping from access levels, depending on stope geometry and accessibility. The material will consist of waste rock with a controlled size distribution to ensure adequate compaction and minimize voids. Placement will generally proceed in lifts, with each lift allowed to settle before subsequent filling to maintain stability and prevent bridging. The rock fill mass will act primarily as passive support rather than structural strength to provide confinement and maintain stability during adjacent stope extraction.

The rock fill system should be designed to ensure efficient delivery, placement, and performance. The system will comprise a network of UG and surface fill raises and loading bays integrated into the mine's development layout. Where necessary, drawpoints or remote-controlled dumping systems will be incorporated to manage

fill distribution. The overall system must provide operational flexibility and efficiency enabling continuous backfilling concurrent with mineralized material extraction.

The secondary mine backfill would be CRF. CRF slurry will be mixed in a surface batch plant and then stored in an agitated tank for delivery into a transmixer.

The transmixer will transport the CRF slurry UG and dump it onto a measured amount of waste rock stored in an available remuck bay near the stope to be backfilled. It is important that the correct proportion of CRF slurry be added to the correct amount of waste rock to ensure the cured target strength of the CRF is achieved. An LHD will thoroughly mix the CRF slurry into the waste rock, ensuring a homogeneous mixture is delivered to the stope.

Following the placement of the CRF in the stope, time must be allowed to cure to the target backfill strength required prior to adjacent mining activities.

WSP recommends conducting a Particle Size Distribution (PSD) test of waste material to confirm its suitability for use as rock fill or as aggregate in cemented rock fill. In addition, it is recommended to determine the required strength parameters for the CRF and develop and optimize the mix design accordingly to ensure that the specified mechanical performance and stability criteria are achieved.

16.13 MINE DEWATERING

Mine dewatering is planned with typical pumping station with submersible pumps are located in the low point of each level, lifting infiltration water to the upper level, then up to the surface for collection to the final effluent treatment plant. When needed, additional floor sumps are added, collecting water through ditches by gravity in each low point. The liquid level inside the pumping station sump is monitored and controlled with level transmitters, indicators and controllers, so the submersible pumps can start and stop accordingly. Typical motor ranges are 2-5.5 kW for sump pumps, and 15-25 kW for the submersible pumps.

Table 16-25 Mine Dewatering

UG Mine	# of pumping stations
UG Center	9
UG South	23
UG North	9

17.0 RECOVERY METHODS

17.1 PROCESS DESIGN

The process plant design for the Project is based on a conventional flotation flowsheet. Lead, zinc and pyrite concentrates will be recovered and sold to market.

The plant design criteria were selected to minimize operating costs and maximize the use of proven technology. The key criteria for equipment selection are suitability for duty, safety, reliability, and ease of maintenance.

17.1.1 Selected Process

The process plant design comprises the following unit operations:

- Single stage primary crushing with a jaw crusher to produce an 80% Passing (P_{80}) crushed product size of 136 mm.
- Crushed rock stockpile with a live capacity of approximately 8,115 tonnes to provide 24 hours of live storage. During extended periods of primary crusher equipment maintenance, additional crushed material inventory can be generated in the weeks leading up to the planned shutdown by dozing crushed material from this stockpile to the area adjacent to the stockpile. This material can then be reclaimed during the shutdown by front-end loader to feed the grinding circuit.
- SAB type grinding circuit consisting of a SAG mill and a ball mill to produce a ground product with a P_{80} of 58 microns (μm), with hydrocyclones for particle size classification.
- Lead rougher flotation, classification, regrinding and cleaner flotation to produce a lead concentrate of 29.6% grade. The lead concentrate will be thickened, filtered, and bagged prior to shipment.
- Zinc rougher flotation, classification, regrinding and cleaner flotation to produce a zinc concentrate of 57.4% grade. The zinc concentrate will be thickened and filtered for stockpiling prior to shipment.
- Pyrite rougher flotation to produce a pyrite concentrate with 4.2 Au g/t and 81.0 Ag g/t. The pyrite concentrate will be thickened and filtered for stockpiling prior to shipment.
- Tailings thickening and storage in a tailings management facility.

The unit operations incorporated in the selected process flowsheet are shown in Figures 17-1 and 17-2.

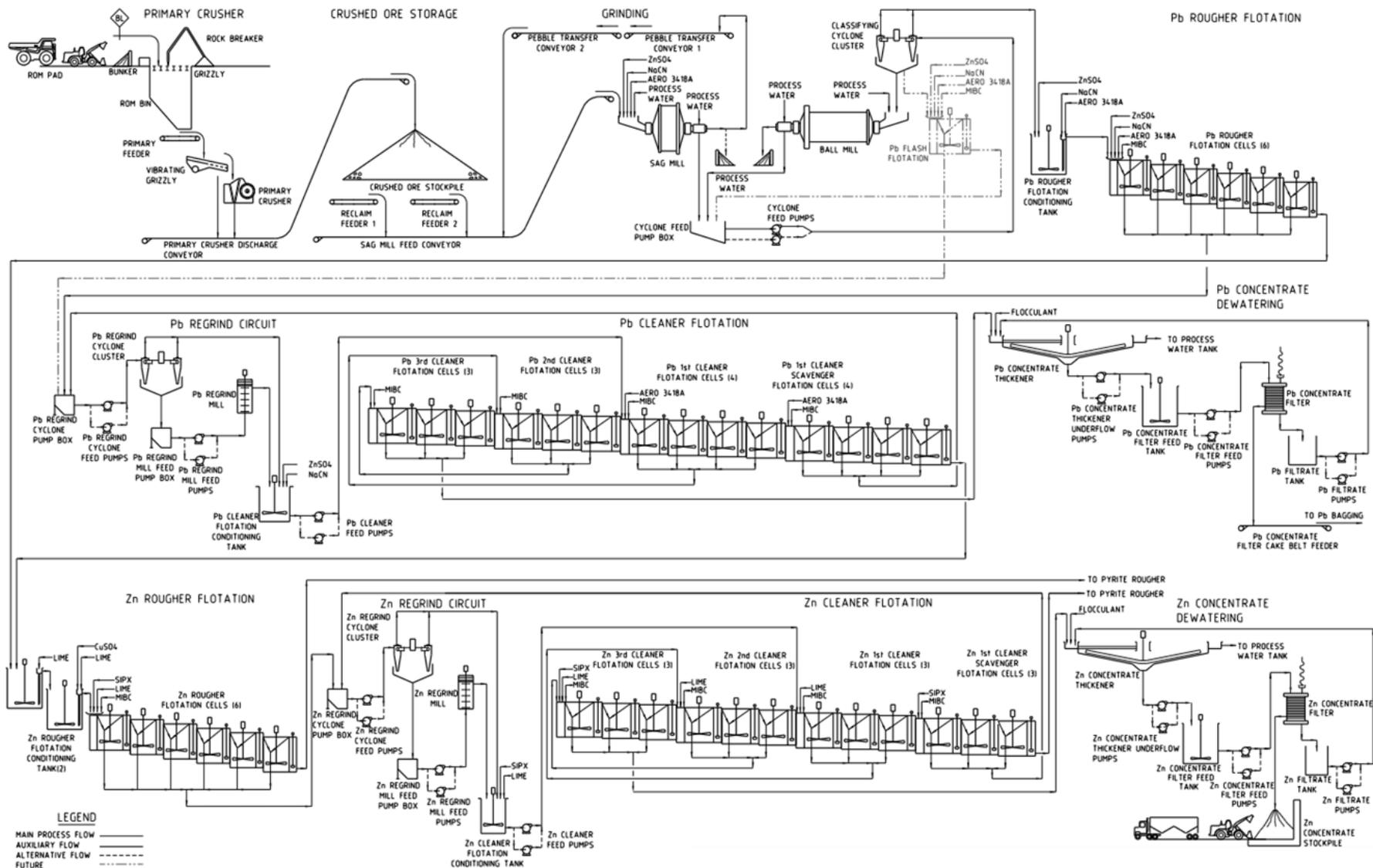


Figure 17-1 Simplified Process Flowsheet

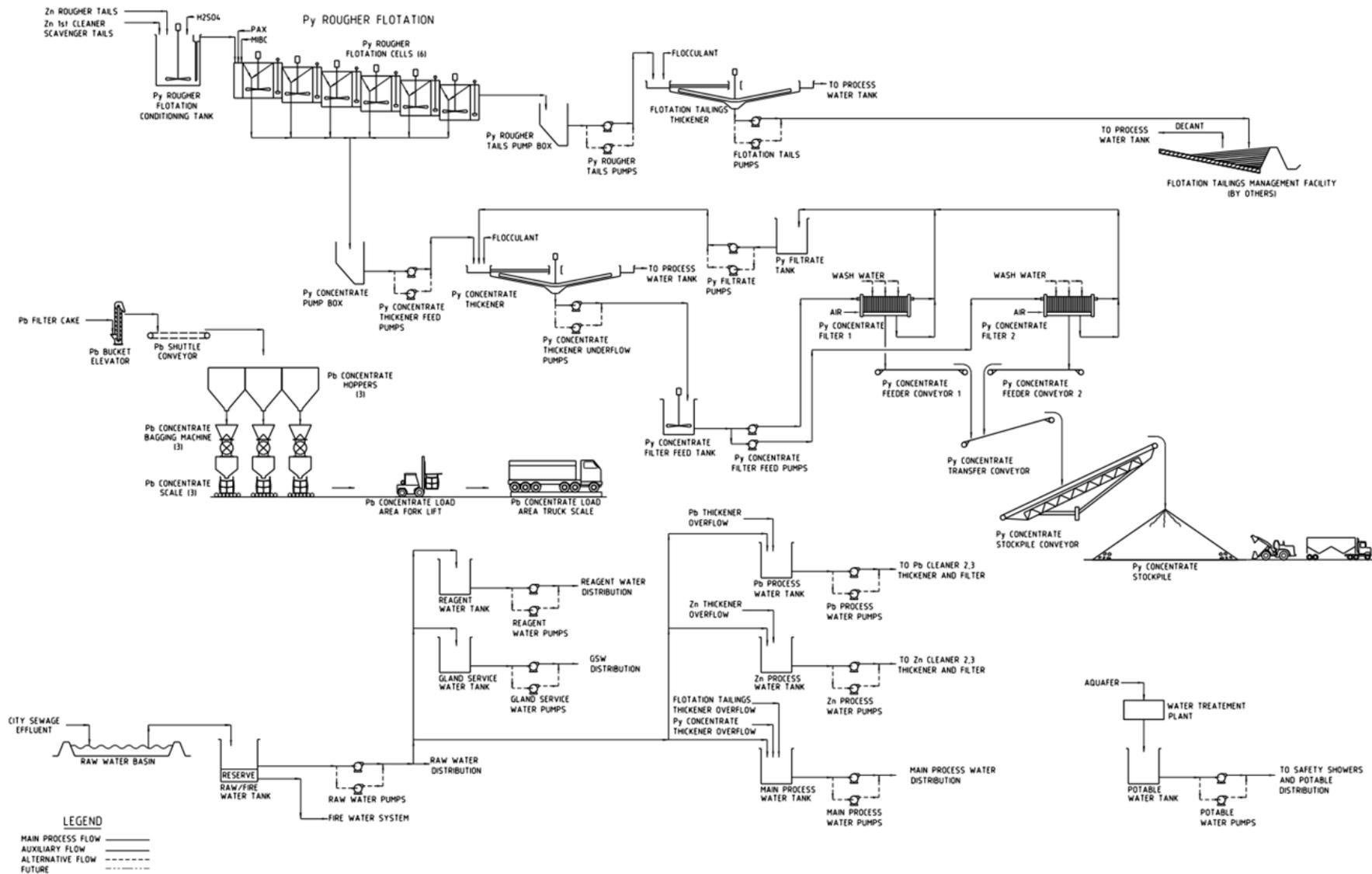


Figure 17-2 Simplified Process Flowsheet (Continued)

17.1.2 Key Process Design Criteria

The key design criteria for the process plant are outlined in Table 17-1.

Table 17-1 Key Process Design Criteria

Criteria Description	Units	Nominal	Source
Plant Throughput	Mtpa	2.92	AGS
Life of Mine	y	11	AGS
Head Grade - Design	g/t Au	3.15	Testwork
	g/t Ag	105	Testwork
	% Pb	0.92	Testwork
	% Zn	1.87	Testwork
Flotation Recovery:			
• Pb to Pb Concentrate	%	82.0	Testwork
• Zn to Zn Concentrate	%	74.7	Testwork
• Au to Py Concentrate	%	71.4	Testwork
• Ag to Py Concentrate	%	41.4	Testwork
Concentrate Grades:			
• Pb grade in Pb Concentrate	%	29.6	Testwork
• Zn grade in Zn Concentrate	%	57.4	Testwork
• Au grade in Py Concentrate	g/t	4.2	Testwork
• Ag grade in Py Concentrate	g/t	81.0	Testwork
Reagent Addition¹:			
• Zinc Sulphate (ZnSO ₄ ·7H ₂ O)	g/t	360	Testwork
• Sodium-Diisobutyl Dithiophosphinate (Aerophine 3418A)	g/t	22.5	Testwork
• Methyl Isobutyl Carbinol (MIBC)	g/t	106.3	Testwork
• Copper Sulphate (CuSO ₄ ·5H ₂ O)	g/t	225	Testwork
• Sodium Isopropyl Xanthate (SIPX)	g/t	32.5	Testwork
• Potassium Amyl Xanthate (PAX)	g/t	300	Testwork
• Sulphuric Acid (H ₂ SO ₄)	g/t	1,000	Testwork
• Sodium Cyanide (NaCN)	g/t	120	Testwork
• Lime (@91% CaO)	g/t	2,350	Testwork
Crushing Plant Availability	%	75	Agreed
Milling and Flotation Circuit Availability	%	91.3	Agreed
Filtration Circuit Availability	%	80	Agreed
Crushing Work Index (CWi)	kWh/t	10.2	Assumption
Bond Ball Mill Work Index (BWi)	kWh/t	13.1	Testwork
SMC Axb	0	46.2	Assumption
Bond Abrasion Index (Ai)	0	0.402	Assumption
Grind Size, P₈₀	µm	58	Testwork
Pb Flotation Conc. Mass Pull – Rougher / Cleaner	%	9.7 / 2.6	Testwork
Pb Regrind Size, P₈₀	µm	16	Testwork
Pb Concentrate Thickener Solids Loading	t/h/m ²	0.2	Assumption
Zn Flotation Conc. Mass Pull – Rougher / Cleaner	%	4.4 / 2.4	Testwork
Zn Regrind Size, P₈₀	µm	15	Testwork
Zn Concentrate Thickener Solids Loading	t/h/m ²	0.2	Assumption
Py Rougher Flotation Conc. Mass Pull	%	53.5	Testwork
Py Concentrate Thickener Solids Loading	t/h/m ²	0.5	Assumption
Flotation Tailings Thickener Solids Loading	t/h/m ²	0.5	Assumption
Tailings Disposal	-	Pumped to TSF (by others)	AGS

17.2 PROCESS PLANT DESCRIPTION

ROM material will be delivered to a local feed bin by a mine truck to feed the crushing plant. The crushed product will be stockpiled prior to feeding the grinding circuit, consisting of a SAG mill and a ball mill with hydrocyclones for particle size classification. The ground product will feed the lead rougher flotation conditioning tank, which will feed the lead rougher flotation cells. The lead rougher flotation concentrate will proceed to a classification and regrind circuit, while the tails will feed the zinc rougher flotation conditioning tanks. Following regrinding, the ground lead rougher concentrate will report to a lead cleaner flotation circuit, consisting of three cleaning stages and one cleaner-scavenger stage. The lead cleaner tailings will report to zinc rougher flotation, while the final lead cleaner concentrate will be thickened, filtered and bagged prior to shipment. The filtrate from the lead filter will circulate back to the lead concentrate thickener and the lead thickener overflow will report to the lead water circuit and any excess will report back to the process water pond.

The lead rougher tails and the lead first cleaner-scavenger tails will feed the zinc rougher flotation conditioning tank, which will feed the zinc rougher flotation cells. The zinc rougher flotation concentrate will proceed to a classification and regrind circuit, while the tails will feed the pyrite rougher flotation conditioning tanks. Following regrinding, the zinc rougher concentrate will report to a zinc cleaner flotation circuit, consisting of three cleaning stages and one cleaner-scavenger stage. The zinc cleaner-scavenger tailings will report to pyrite rougher flotation, while the third zinc cleaner concentrate will be thickened and filtered prior to stockpiling for truck loading and shipment. The filtrate from the zinc filter will circulate back to the zinc concentrate thickener and the zinc thickener overflow will report to the zinc water circuit and any excess will report back to the process water pond.

The zinc rougher tails and the zinc first cleaner-scavenger tails will feed the pyrite rougher flotation conditioning tank, which will feed the pyrite rougher flotation cells. The pyrite rougher flotation tailings will be thickened in a tailings thickener before getting pumped to the flotation tailings management facility. The flotation tailings thickener overflow will report to the process water pond. The pyrite concentrate will be thickened, filtered and stockpiled by a radial stacker from where it is loaded on trucks and shipped. The filtrate from the pyrite concentrate filter will report to the pyrite concentrate thickener, and the pyrite concentrate thickener overflow will report to the process water pond.

17.2.1 Crushing Circuit

The crushing circuit will be fed through the ROM bin by direct tipping of the mine truck or reclaimed by a front-end loader. The ROM bin will be fitted with a static grizzly with an aperture size of 800x800 mm to prevent oversized rocks from entering the crusher. Oversized rocks will be crushed by a fixed rock breaker. The ROM bin will discharge onto an apron feeder that will feed a vibrating grizzly with an aperture size of 90 mm. The grizzly undersize will bypass the primary crusher to reduce wear on the crusher, while the grizzly oversize will directly feed the primary crusher. The primary crusher will be a 160 kW jaw crusher, and will crush the grizzly oversize at a rate of 444 dry t/h from an F_{80} of 501 mm to a P_{80} of 156 mm. The crushed rock will discharge onto the primary crusher discharge conveyor together with the grizzly undersize to feed the 24h live capacity crushed rock stockpile.

17.2.2 Grinding Circuit

Crushed material will be reclaimed from the crushed rock stockpile by two apron feeders, which may be operated together or with one on standby. The apron feeders will discharge on to the SAG mill feed conveyor to feed the primary mill. Flotation reagents zinc sulphate, sodium cyanide and Aero 3418A will be added to the primary mill with process water to pre-condition the material and to achieve a target moisture level of 75 %w/w in the mill.

The primary mill will be a SAG mill with a diameter of 7.92 m and effective grinding length of 4.39 m. It will have a high-speed variable speed drive and installed motor power of 6.1 MW. It will be charged with forged steel balls with a maximum size of 125 mm, equipped with a grate discharge and a trommel screen. The trommel screen oversize will discharge into a scats bunker or onto one of two pebble transfer conveyors for recirculation back to the SAG mill via the SAG mill feed conveyor. The trommel screen undersize will discharge

to the cyclone feed pump box. The trommel screen will be equipped with a spray water mechanism to improve screening efficiency.

The secondary mill will be a ball mill with a diameter of 6.10 m and an effective grinding length of 9.32 m. Process water will be fed to the mill to achieve a target moisture content of 75%. The ball mill will have a fixed speed drive with a wound round induction motor and liquid resistance starter. The installed motor power will be 6.1 MW. It will be charged with high-chromium steel balls with a maximum size of 50 mm, discharging by overflow and equipped with a trommel screen. The trommel screen oversize will discharge to a scats bunker, and the trommel screen undersize will discharge to the cyclone feed pump box. The trommel screen will be equipped with a spray water mechanism to improve screening efficiency.

The combined SAG and Ball Mill discharge will be pumped to the cyclones for classification from the cyclone feed pump box, targeting an overflow P_{80} of 58 μm . There will be a total of twelve installed cyclones with six duty and six standby. The cyclones will be operated at a pressure of 85 kPa(g) and are expected to have a nominal recirculation rate of 382%. The cyclone overflow will report to the lead rougher flotation circuit, and the underflow will feed the secondary mill.

17.2.3 Lead Rougher Flotation Circuit

The cyclones' overflow will gravitate to the lead rougher conditioning tank. Zinc sulphate, sodium cyanide and Aero 3418A will be fed to the agitated conditioning tank where the rougher feed will be conditioned for five minutes. The conditioned slurry will then feed the lead rougher flotation cells. The circuit will consist of a single train of six forced-air mechanical cells. Zinc sulphate, sodium cyanide, Aero 3418A, and MIBC will be added to the lead rougher cells to maximize lead recovery. The lead rougher flotation concentrate will report to the lead regrind and cleaner flotation circuits, while the tails will feed the zinc rougher flotation circuit.

17.2.4 Lead Regrind Circuit

The lead rougher concentrate and the lead first cleaner-scavenger concentrate will report to the lead regrind cyclone pump box and will be pumped to the lead regrind cyclone cluster for classification and to increase the percent solids of the slurry feeding the regrind mill. The lead cyclone cluster will consist of sixteen cyclones in total, with 11 duty and 5 standby. The cyclones will be operated at a pressure of 64 kPa(g) and will target an overflow particle d_{50} of 21 μm . The lead regrind cyclones' overflow will report directly to the lead cleaner flotation surge tank, while the underflow will gravitate to the lead regrind mill feed pump box before getting pumped to the lead regrind stirred media mill (SMM). The SMM will reduce the particle size from an F_{80} of 58 μm to a P_{80} of 16 μm . It will be charged with zirconia toughened alumina ceramic beads and will have an installed motor power of 1.6 MW. The SMM discharge will combine with the cyclone overflow in the lead cleaner flotation surge tank.

17.2.5 Lead Cleaner Flotation Circuit

Zinc sulphate and sodium cyanide will be added to the lead cleaner flotation surge tank for conditioning ahead of flotation. The flotation feed will be conditioned for five minutes prior to feeding the first lead cleaner flotation circuit. The lead cleaner flotation circuit will consist of three cleaning stages and one cleaner-scavenger stage. The first cleaner stage will contain one train of four forced-air mechanical cells and all other stages will contain one train of three forced-air mechanical cells. MIBC will be added to each stage and Aero 3418A will be added to the second and third cleaner flotation circuits to promote lead recovery.

The concentrate and tails will travel through the circuit counter-currently. The lead first cleaner flotation concentrate will feed the lead second cleaner flotation cells, the lead second cleaner flotation concentrate will feed the lead third cleaner flotation cells, and the lead third cleaner flotation concentrate will feed the lead concentrate thickener. The lead third cleaner flotation tails will report to the lead second cleaner flotation cells, the lead second cleaner flotation tails will feed the lead first cleaner flotation cells, and the lead first cleaner flotation cell tails will feed the lead first cleaner-scavenger flotation cells. The lead first cleaner-scavenger flotation concentrate will be recirculated back to the lead regrind cyclone pump box, while the tails will report to the zinc rougher flotation conditioning tank with the lead rougher flotation tails.

17.2.6 Lead Concentrate Thickening, Filtration, and Bagging Circuit

The lead third cleaner flotation concentrate will be pumped to the lead concentrate thickener, where flocculant will be added to the feed well to promote particle settling. The thickener will be high rate with a diameter of 11 m. The slurry will be thickened to 55% (w/w) solids in the underflow. The thickener overflow will report to the lead process water tank, while the thickener underflow will be pumped to the lead concentrate filter feed tank. The filter feed tank will be sized to hold 24 hours of feed slurry. The slurry will be pumped to a pressure filter to dewater the concentrate cake to 10% (w/w) moisture. The thickener and filter design criteria will be further defined upon the completion of dewatering testwork. The filtrate from the lead concentrate filter will be discharged to a filtrate tank and pumped back to the lead concentrate thickener, while the lead concentrate filter cake will be discharged via a belt feeder, bucket elevator and shuttle conveyor into lead concentrate hoppers. The hoppers will feed automatic bagging systems, with the bags transported and stacked by forklift prior to loading on highway trucks for shipment offsite.

17.2.7 Zinc Rougher Flotation Circuit

The first zinc rougher conditioning tank will receive the lead rougher flotation tails and the lead first cleaner scavenger tails. Lime will be added to the first conditioning tank to achieve a target pH of 11.5, and the second conditioning tank will be dosed with additional lime and copper sulphate. The zinc rougher flotation feed will be conditioned in the agitated conditioning tanks for a total of 10 minutes – five minutes in each tank. The conditioned slurry from the second conditioning tank will feed the zinc rougher flotation cells. The circuit will consist of a single train of six forced-air mechanical cells. SIPX, lime and MIBC will be added to the zinc rougher cells to maximize zinc recovery. The zinc rougher flotation concentrate will report to the zinc regrind and cleaner flotation circuits, while the tails will feed the pyrite rougher flotation circuit.

17.2.8 Zinc Regrind Circuit

The zinc rougher concentrate and the zinc first cleaner-scavenger concentrate will report to the zinc regrind cyclone pump box and will be pumped to the zinc regrind cyclone cluster for classification and to increase the percent solids of the feed entering the regrind mill. The zinc cyclone cluster will consist of eight cyclones in total, with 5 duty and 3 standby. The cyclones will be operated at a pressure of 81 kPa(g) and will target an overflow particle d_{50} of 20 μm . The zinc regrind cyclones overflow will report directly to the zinc cleaner flotation surge tank, while the underflow will gravitate to the zinc regrind mill feed pump box before being pumped to the zinc regrind stirred media mill (SMM). The SMM will reduce the particle size from an F_{80} of 58 microns to a P_{80} of 15 μm . It will be charged with zirconia toughened alumina ceramic beads and will have an installed motor power of 700 kW. The SMM discharge will combine with the cyclone overflow in the zinc cleaner flotation surge tank.

17.2.9 Zinc Cleaner Flotation Circuit

SIPX and lime will be added to the zinc cleaner flotation surge tank for conditioning ahead of flotation. The flotation feed will be conditioned for five minutes prior to feeding the first zinc cleaner flotation circuit. The zinc cleaner flotation circuit will consist of three cleaning stages and one cleaner-scavenger stage. Each stage will contain one train of three forced-air mechanical cells. MIBC will be added to each stage, SIPX will be added to the first cleaner-scavenger and the third cleaner, and lime will be added to the first through third cleaners to maximize zinc recovery.

The concentrate and tails will travel through the circuit counter-currently. The zinc first cleaner flotation concentrate will feed the zinc second cleaner flotation cells, the zinc second cleaner flotation concentrate will feed the zinc third cleaner flotation cells, and the zinc third cleaner flotation concentrate will feed the zinc concentrate thickener. The zinc third cleaner flotation tails will report to the zinc second cleaner flotation cells, the zinc second cleaner flotation tails will be recycled to the zinc first cleaner flotation cells, and the zinc first cleaner flotation cell tails will report to the zinc first cleaner-scavenger flotation cells. The zinc first cleaner-scavenger flotation concentrate will be recirculated back to the zinc regrind cyclone pump box, while the tails will report to the pyrite rougher flotation conditioning tank with the zinc rougher flotation tails.

17.2.10 Zinc Concentrate Thickening and Filtration Circuit

The zinc third cleaner flotation concentrate will be pumped to the zinc concentrate thickener, where flocculant will be added to the feed well to promote particle settling. The slurry will be thickened in a 11 m diameter, high-rate thickener to 55% (w/w) solids in the underflow. The thickener overflow will report to the zinc process water tank, while the thickener underflow will be pumped to the zinc concentrate filter feed tank. The filter feed tank will be sized to hold 24 hours of feed slurry. The slurry will be pumped to a pressure filter to dewater the concentrate cake to 10% (w/w) moisture. The thickener and filter design criteria will be further defined upon the completion of dewatering testwork. The zinc concentrate filter filtrate will be discharged to filtrate tank and pumped back to the zinc concentrate thickener, while the zinc concentrate filter cake will be discharged to a stockpile in a covered shed prior to loadout. The zinc concentrate will be loaded into highway haulage trucks by a front-end loader for shipment offsite.

17.2.11 Pyrite Rougher Flotation Circuit

The pyrite rougher flotation conditioning tank will receive the tails from the zinc rougher flotation and the zinc first cleaner-scavenger flotation. Sulphuric acid will be fed to the agitated conditioning tank to lower the pH between 7-8.5. The pyrite rougher feed will be conditioned for five minutes. The conditioned slurry will feed the pyrite rougher flotation cells. The circuit will consist of a single train of six forced-air mechanical cells. PAX and MIBC will be added to the pyrite rougher cells to maximize pyrite recovery. The pyrite rougher flotation concentrate will report to pyrite concentrate thickening and filtration, while the tails will report to the flotation tailings thickener.

17.2.12 Pyrite Concentrate Thickening and Filtration Circuit

The pyrite rougher flotation concentrate will be pumped to the pyrite concentrate thickener, where flocculant will be added to the feed well to promote particle settling. The slurry will be thickened to 65% (w/w) solids in the underflow in a 32 m diameter high-rate thickener. The thickener design criteria will be further defined upon the completion of dewatering testwork. The thickener overflow will report to the process water pond, while the thickener underflow will be pumped to the pyrite concentrate filter feed tank. The filter feed tank will be sized to hold 24 hours of feed slurry. From the filter feed tank, the pyrite concentrate slurry is pumped to one of two parallel pressure filters to dewater the concentrate cake to 15% (w/w) moisture for stockpiling.

The filtrate from the pyrite concentrate filter will recirculate back to the pyrite concentrate thickener. The filter design criteria will be further defined upon the completion of dewatering testwork. The stockpile will be a radially stacked dead stockpile with a live capacity of 7,250 m³, holding approximately 24 hours of filtered pyrite concentrate. The pyrite concentrate will be loaded into highway haulage trucks by a front-end loader for shipment offsite.

17.2.13 Flotation Tailings Disposal

The pyrite rougher flotation tails will be pumped to the flotation tails thickener, where flocculant will be added to the feed well to promote particle settling. The thickener will be a high-rate type with a diameter of 22 m. The slurry will be thickened to 40% (w/w) solids in the underflow. The thickener overflow will report to the process water pond, while the thickener underflow will be pumped to the flotation tailings management facility. It is estimated that 129 m³/h of water will be decanted from the flotation tailings management facility back to the process water pond. The thickener design criteria will be further defined upon the completion of dewatering testwork.

17.2.14 Reagents

Plant operations will require a reagent inventory of flocculant, zinc sulphate, Aero 3418A, MIBC, copper sulphate, SIPX, PAX, sulphuric acid, sodium cyanide, lime, and anti-scalant. The site will maintain six weeks' worth of supply of each reagent onsite in the reagent shed. All reagents will be dosed to their respective injection sites by metered dosing pumps (one per dosing site), except for lime and flocculant. Lime will be dosed by a centrifugal pump and flocculant will be dosed by helical screw pumps, with one per dosing application.

17.2.14.1 Flocculant

Flocculant will be used in each of the thickeners to promote particle settling. It will be delivered to site as a solid powder in 750 kg bags. The bags will be split by a bag breaker and mixed for five hours with filtered process water. The mixing concentration will be 0.25% (w/v). The flocculant will be diluted in-line with process water to a concentration of 0.025% (w/v). The type and quantity of flocculant used will be further defined upon the completion of thickening testwork.

17.2.14.2 Zinc Sulphate

Zinc sulphate is an activator in lead flotation. It will be fed to the SAG mill, lead rougher flotation and lead cleaner flotation feed. It will be delivered to site as a heptahydrate crystal in 1,000 kg bags at 98% purity. It will be diluted to 25% (w/v) concentration for dosage. Three bags will be required for the mixing of one batch, which is enough to supply 22 hours of operation.

17.2.14.3 Aero 3418A

Aero 3418A will be used as a collector and promoter in lead flotation. It will be fed to the SAG mill, lead rougher flotation, the lead first cleaner and the lead first cleaner scavenger. It will be delivered to site and stored in 1 m³ IBC totes.

17.2.14.4 MIBC

MIBC will be delivered as a liquid in 1 m³ IBC totes for usage as a frothing agent in all flotation circuits and stages.

17.2.14.5 Copper Sulphate

Copper sulphate will be used as an activator in zinc rougher flotation. It will be fed to the second zinc conditioning tank. The copper sulphate will be delivered as solid pentahydrate crystals at 98% purity in 1,200 kg bulk bags. The copper sulphate will be diluted to 20% (w/v) for dosage. Two tonnes of crystals will be required per mix, which is enough to supply operations for 24 hours.

17.2.14.6 Sodium Isopropyl Xanthate (SIPX)

SIPX will be used as a collector in zinc rougher and cleaner flotation. It will be fed to the zinc rougher, zinc regrind feed, zinc first and second cleaners, and the zinc first cleaner scavenger. SIPX will be delivered to site in 1,000 kg bulk bags as pellets with a purity of at least 90%. One tonne of material will be used per mix, through which the SIPX will be diluted to 10% (w/v). One mix will last for approximately three days of operation.

17.2.14.7 Potassium Amyl Xanthate (PAX)

PAX will be used as a collector in the pyrite rougher flotation, fed directly to the rougher cells. PAX will be delivered to site in pellet format packed in 1,000 kg bulk bags, with a purity of at least 90%. Three tonnes of PAX will be mixed per makeup batch, which will dilute the reagent to 10% (w/v) and will last for 25 hours of operation.

17.2.14.8 Sulphuric Acid

Sulphuric acid will be used for pH adjustment prior to pyrite flotation, fed to the pyrite conditioning tank. It will be delivered to site as a liquid in 1 m³ IBC totes with a purity of 98%. Three totes will be kept in the dosing area, which will be enough supply for 15 hours of operation.

17.2.14.9 Sodium Cyanide

Sodium cyanide will be used in lead flotation and will be fed to the SAG mill, lead conditioning tank, lead rougher flotation cells and to the lead cleaner flotation surge tank. It will be delivered to site as solid

briquettes packed in one-tonne bulk bags in boxes. The briquettes will be mixed at a rate of one tonne per batch, diluted to 20% (w/v). This will be sufficient supply for 22 hours of operation.

17.2.14.10 Lime (CaO)

Lime will be used in zinc flotation and will be dosed to the zinc conditioning tanks, zinc rougher flotation, zinc cleaner flotation surge tank, and the first, second and third zinc cleaners. It will be delivered to site as a quicklime powder. It will be stored in a 150-tonne silo and slaked for dosage. The lime slurry will be prepared at a 20% (w/w) solids concentration. The storage tank will have a 24-hour residence time, and the dosage system will have a ring main for constant recirculation to prevent scaling. Lime will be delivered to site in full truckloads pneumatically offloading into the silo, requiring approximately one truckload per day.

17.2.14.11 Anti-Scalant

Anti-scalant will be used to prevent process water scaling and will be dosed to the process water pond pump suction line. It will be delivered to site as a liquid in 1 m³ IBC totes. One tote will service approximately 3.6 days of operation.

17.2.15 Plant Services

17.2.15.1 Air

Plant and instrument air will be supplied from air compressors. The air will be dried before it is distributed to various air receivers and users.

17.2.15.2 Water

Raw water will be sourced from the city sewage treatment effluent into a Raw Water Basin and transferred into a combined raw and fire water tank located in the process plant. Raw water will be used for gland service water, reagent mixing, dust suppression in the crushing circuit, process water makeup and the fire water reserve.

The main process water will be stored in a pond and will be supplied by thickener overflows, tailings management facility decant return, and raw water makeup. Separate lead and zinc process water tanks are fed from the lead and zinc concentrate thickeners respectively and used for the last cleaning and filter stages in each respective circuits to avoid contamination of the final products.

Potable water will be sourced from an aquifer and will be treated through an onsite treatment plant prior to use at the camp and in the operations facilities.

17.2.15.3 Fuel

Plant mobile equipment will be powered by diesel, which will be stored onsite in a tanker and distributed to fueling stations.

17.3 PLANT CONSUMPTION

17.3.1 Water

Nominal water consumption figures are provided in Table 17.3.1.

Table 17-2 Daily Water Consumption (Nominal)

Type	Daily Consumption (m ³ /d)
Raw Water	2,997
Potable Water	128

17.3.2 Energy

A breakdown of the estimated annual energy consumption for the process plant is provided in Table 17-3.

Table 17-3 Annual Energy Consumption

Area	Annual Energy Consumption (MWh/y)
Feed Preparation – Crushing & Stockpiling	2,012
Milling – Grinding & Classification	93,487
Flotation & Concentrate Handling	36,512
Tails Thickening & Handling	2,701
Reagents	1,841
Water Services	2,692
Air Services	4,576
Fuel Storage & Distribution	102
Water Supply, Sewage, TSF, Plant Infrastructure (Buildings & Offices)	2,494
Total	172,396

17.3.3 Reagents and Consumables

Annual consumption figures for each reagent and major consumables are shown in Table 17-4.

Table 17-4 Annual Consumption of Reagents and Consumables

Reagent	Annual Consumption ¹ (t/y)
Flocculant	60
Zinc Sulphate	1,051
Aero 3418A	66
MIBC	310
Copper Sulphate	657
SIPX	95
PAX	876
Sulphuric Acid	2,920
Sodium Cyanide	350
Lime	6,862
Anti-Scalant	91
Grinding Media – SAG Mill	1,679
Grinding Media – Ball Mill	2,307
Grinding Media – Re grind Mills	59

¹Reagent consumptions are provided at supply concentration

18.0 PROJECT INFRASTRUCTURE

18.1 OVERALL SITE DEVELOPMENT

The overall site plan is shown in Figure 18-1 and includes major facilities of the Project, including the proposed open pit mines, underground mine, crushing and feed preparation facilities, flotation process plant, electrical substations, administrative buildings, and tailings facilities.

18.2 SITE ACCESS ROAD

18.2.1 Access to Site

The project site is located approximately 450 kilometers from the capital of Marrakesh, with the main access being via the National 10 highway. The N10 passes through the town of Tinejdad, which is connected to the mine site through an upgraded dirt road.

The Port of Agadir, approximately 600 km away, will enable the transportation of goods and products for shipping, with an expected truck round trip from the mine to the port and back taking about 24 hours. The Nador Beni Ansar Port, also located approximately 600 km away, can also be utilized. All ports are accessible via national highways.

18.2.2 Project Site Roads

Internal roads will provide access between the open pits, underground portals, crushing facilities, process plant area, mine service areas, and tailings storage facilities. These roads will be six metres wide and constructed flush with bulk earthworks pads to facilitate stormwater management.

18.3 PROCESS PLANT FACILITIES

The infrastructure will support mining, processing, and construction activities. Located north-east of the central zone open pit, the main site will host a ROM pad, crushing and feed preparation facilities, stockpile with live reclaim, processing plant operations, and a security and site office buildings. Open pit mines will operate independently but connect to the central site via haul roads and mine access roads. A portion of the mining activity will also be carried out through an underground portal.

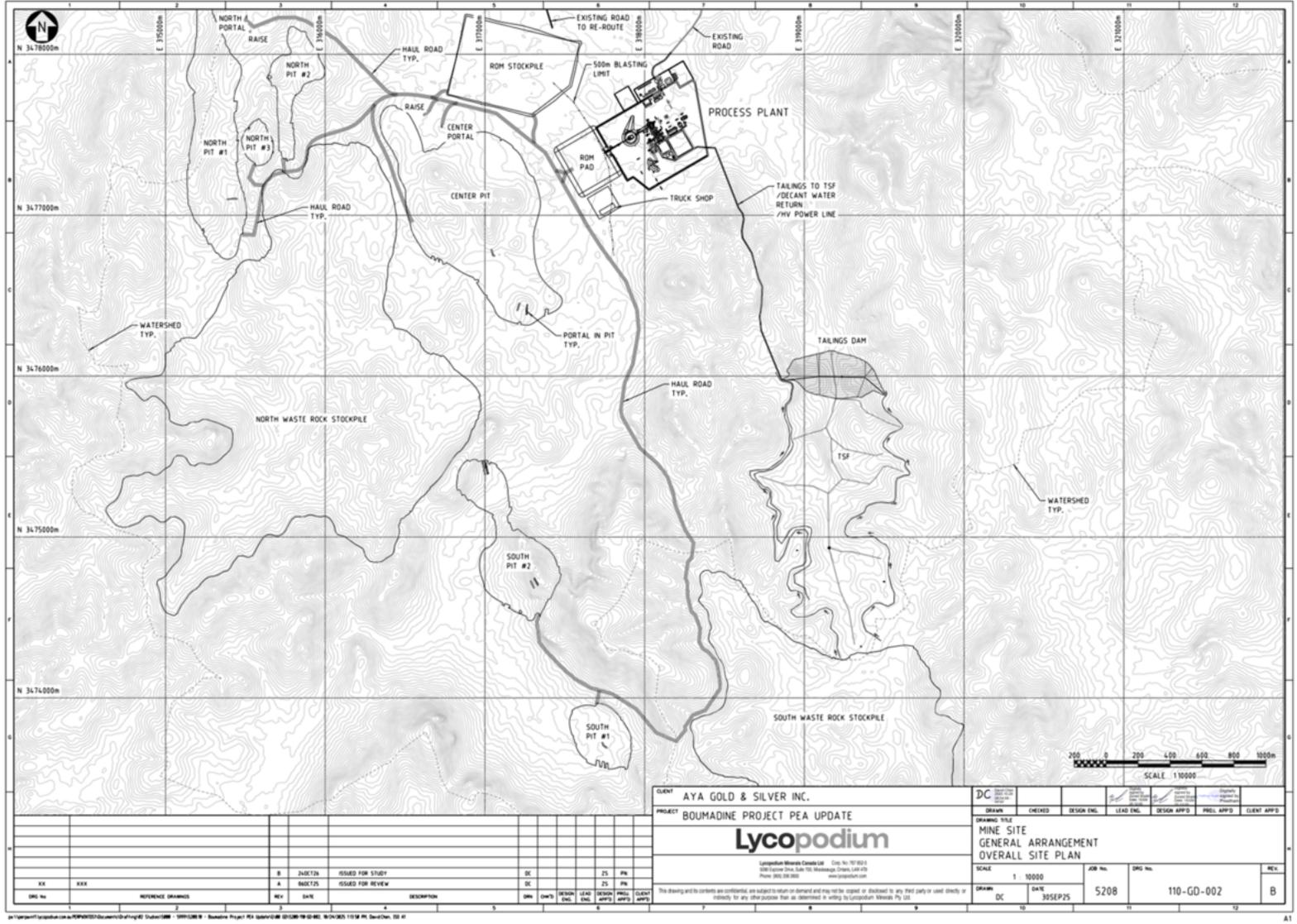


Figure 18-1 Overall Site Plan

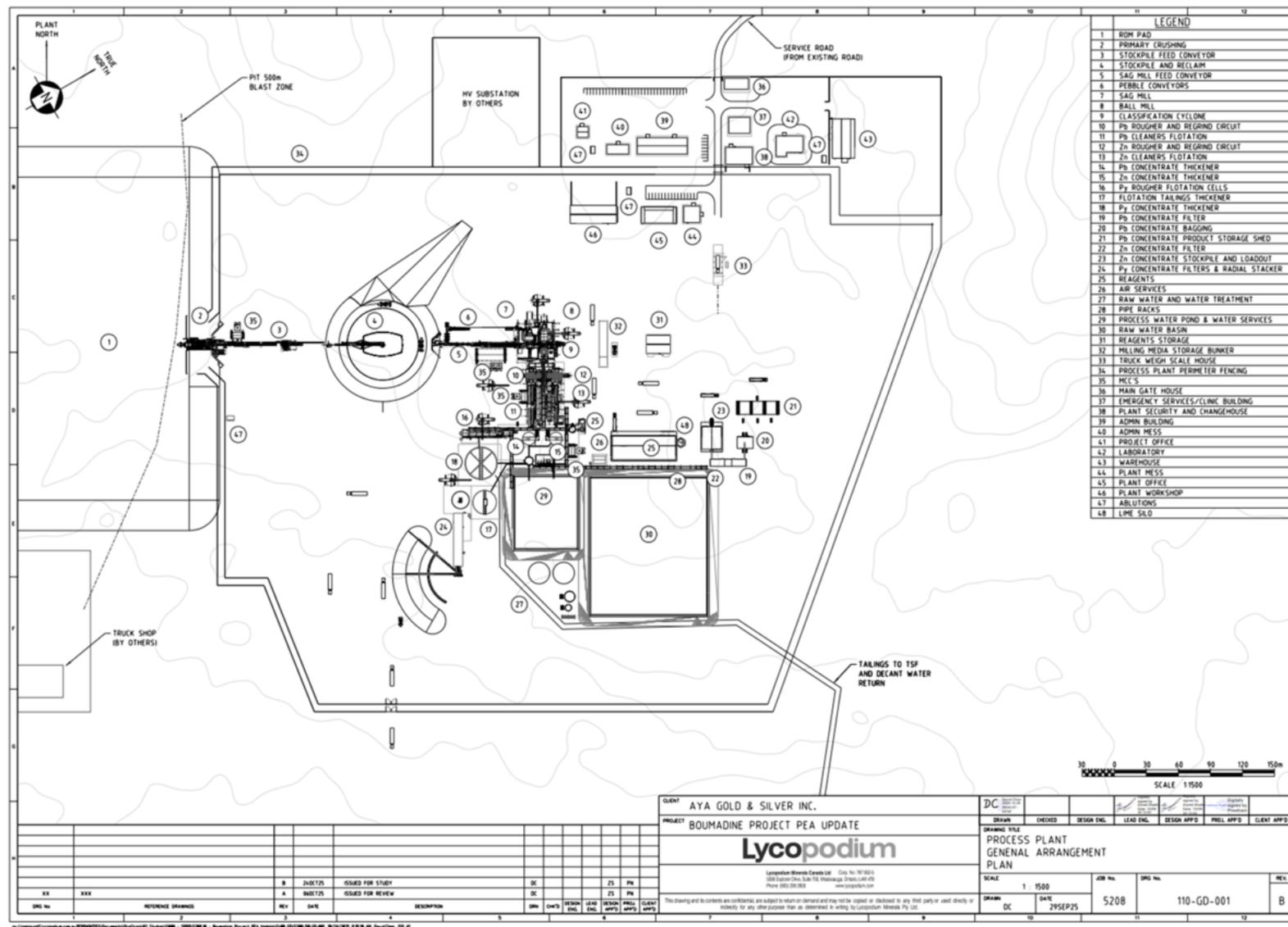


Figure 18-2 Process Plant Layout

Key infrastructure includes:

- Run of mine (ROM) pad.
- Crushing and feed preparation.
- Stockpile with reclaim chamber.
- Lead and zinc flotation plant.
- Lead and Zinc Concentration Handling
- Reagents Storage.
- Plant security and site office building.
- Plant administration and first aid building.
- Mine services area.
- Waste rock facility.
- Laboratory.
- Explosive magazine.
- Fuel and lubricant storage.
- Maintenance workshops and warehouse.
- Electrical Switchrooms.

Services provided cover power generation and distribution, water supply and distribution, surface water management, sewage and waste management, mine pit access, and haul roads. Portable / prefabricated structures will house security office and assay lab, while maintenance shops and warehousing will be brick and mortar buildings on concrete slabs.

Electrical Switchrooms will be located near areas of use.

Due to scarce water sources, an on-site water harvesting reservoir will be constructed near the process plant. A site water balance assessment will determine storage needs. A containerized water treatment system will ensure potable water availability for safety showers and personnel consumption.

Wastewater storage and treatment facilities will be established on-site. Solid waste, excluding domestic waste, will be managed off-site by licensed contractors, with on-site incineration for domestic waste. Dedicated areas for waste collection and sorting will facilitate disposal off-site, with no on-site long-term storage planned.

18.4 MINING FACILITIES

18.4.1 Truck Shop

A 1,000 m² truck shop will be adjacent to the ROM Pad for heavy equipment and light vehicles maintenance. There will be three bays dedicated to maintenance and one enclosed wash bay. All heavy equipment garage doors will be roll-type to maximize inside working space.

A 10-ton overhead crane equipped with two hooks will be installed in the heavy equipment maintenance area, while a single 5-ton overhead crane will cover the light vehicle area. Other maintenance equipment will be available for the workers, considering both the mining and the site maintenance fleets.

Various sections will be installed in the truck shop, including a tool crib, an oil and grease deposit with a carousel distribution system, a mechanical / electrical room, storage rooms, a foreman office, bathrooms, locker rooms, a dining room, and a meeting room.

All water on the truck shop floor will be collected to an oil separation system with an integrated coalescent filter to ensure efficiency even if emulsification occurs. A sand pre-separator is included in the wash bay.

18.4.2 Site Roads

The haulage roads will be 20 m wide based on the mine trucks selected and will connect the open pits, the underground mine portals, the mineralized material stockpile, the ROM pad and the waste rock stockpiles. The service roads to access ventilation raises and other mine services will be 10 m wide.

18.4.3 Low-grade Material Stockpile

The low-grade mineralized material stockpile will be located near the ROM pad to allow for the deposition of low-grade material for eventual blending or mineralized material storage, should the crushing or process plant be shut down. The stockpiled material will be rehandled by a loader and will have a capacity of 3.5 Mt (1.7 Mm³), with a planned height of 5 meters. A more detailed design will be developed during the next stage of the project, to possibly increase the height of the stockpile.

In the absence of geochemical characterization results, it is assumed that material may be potentially acid-generating and/or leachable. Based on this, it is assumed that the mineralized material stockpile will require a foundation consisting of a 2 mm thick HDPE geomembrane with a 300 mm sand protection layer on each side. This assumption will be revisited at the next stage of the project.

Additionally, the stockpile will be located at least 60 meters from the surrounding watercourses and 50 meters from the open pit wall.

18.4.4 Rom Pad

Mineralized material that is over the cut-off for low grade material, as described in 16.9, will be trucked and stored on the mill ROM pad. The material will be separated based on the NSR-Pyrite value into two stockpiles – low- and high-grade pyrite. The ROM will be sufficiently sized for the estimated LOM stockpile volume requirements.

The feed to the concentrator will be blended from the two stockpiles, and material will be fed to the crushing plant with a front-end loader.

18.4.5 Waste Rock Stockpiles

Two waste rock stockpiles with a combined capacity of 215 million cubic meters (Mm³), equivalent to 430 million tonnes (Mt), are planned. The North waste rock stockpile will have a capacity of 150 Mm³ and will be required at the start of mining operations; however, its southern section is scheduled for construction later during ongoing operations. The South waste rock stockpile will have a capacity of 65 Mm³ and will receive waste rock from both the South pits and a portion of the Central pit.

Because a large portion of the waste rock will be used for backfilling in the underground mine, only 5 million tonnes will be added to the waste rock stockpile throughout the underground mining operations.

The design of the waste rock stockpiles will adhere to all relevant mining industry standards, utilizing conservative parameters at this stage due to the absence of geotechnical studies and geochemical characterization results. It is currently assumed that the stored waste rock could be potentially acid-generating and/or leachable. In this optic, it is assumed that each pile will require a foundation consisting of a 2 mm thick HDPE geomembrane with a 300 mm sand protection layer on both sides, or a similar design that would prevent potential contamination. This assumption will be revisited at the next stage of the project.

Additionally, each stockpile will be situated at least 60 meters from surrounding watercourses and 50 meters from the open pit wall.

The following design criteria were used for the waste rock stockpiles:

- Bench slope 2.5H: 1V

- Bench maximum height: 10 m
- Bench maximum width: 10 m
- Maximum height: 100 m
- Waste in-situ density: 2.75 t/m³
- Swelling factor : 1.36

18.5 TAILINGS STORAGE FACILITY

Boumadine Global Mining SA has appointed Epoch Resources (Pty) Ltd (Epoch) to review and provide the Qualified Person's signature for the update to the Pre-Economic Assessment (PEA) for a Flotation Tailings Storage Facility (TSF). A preliminary site selection study was completed in 2024 by an independent third-party Moroccan based consultancy, Groupement des Consultants et Ingénieurs de Maroc (GCIM) for two potential TSFs. The study design criteria have since been updated from two TSFs to only one Flotation TSF, with an updated storage capacity and tailings production rate based on the Life of Mine (LOM) production.

A summary of the key design criteria for the TSF is summarized in Table 18-1 below.

Table 18-1 Design Criteria

Item	Design Criteria	Flotation TSF	Source / Comment
1	Tailings Annual Deposition Rate (Mtpa)	1.65	Aya
2	Life of Mine Storage (Years)	11.3	Aya
3	Total tonnage stored (Million)	18.6	Aya
4	Lining Requirement	Lined	Best Practice
5	Construction Methodology	Full Containment Walls	GISTM compliance

The conceptual design for the preferred TSF site comprises the components that affect the major costs at a high level. The tailings will be deposited from a slurry pipeline into a lined basin, behind earth-fill downstream wall raises. Drainage measures are included along the upstream wall slope and within the basin. Outside of the TSF basin, storm diversion channels along the perimeter of the TSF and an emergency basin located downstream of the TSF manage storm events, and a return water basin receives water pumped from a floating barge on the TSF pool.

The key design parameters entail calculating quantities according to the dimensions of the TSF as per the PEA mine plan. The construction phasing will include five lifts, with each lift's height increased to accommodate the additional tonnage required by the design criteria.

18.6 WATER SUPPLY

18.6.1 Raw Water Supply System

Due to water scarcity, off-site water sourcing has been evaluated. This could be done by using treated wastewater from nearby communities. Three raw water lines will provide up to 4230 m³/d to the site, from the towns of Tinejdad, Goulmima and Tinghir. An on-site water treatment plant will be used to ensure that the water quality is acceptable for the process. The raw water will be stored in the on-site water harvesting reservoir prior to being transferred to the open top raw water tank for distribution.

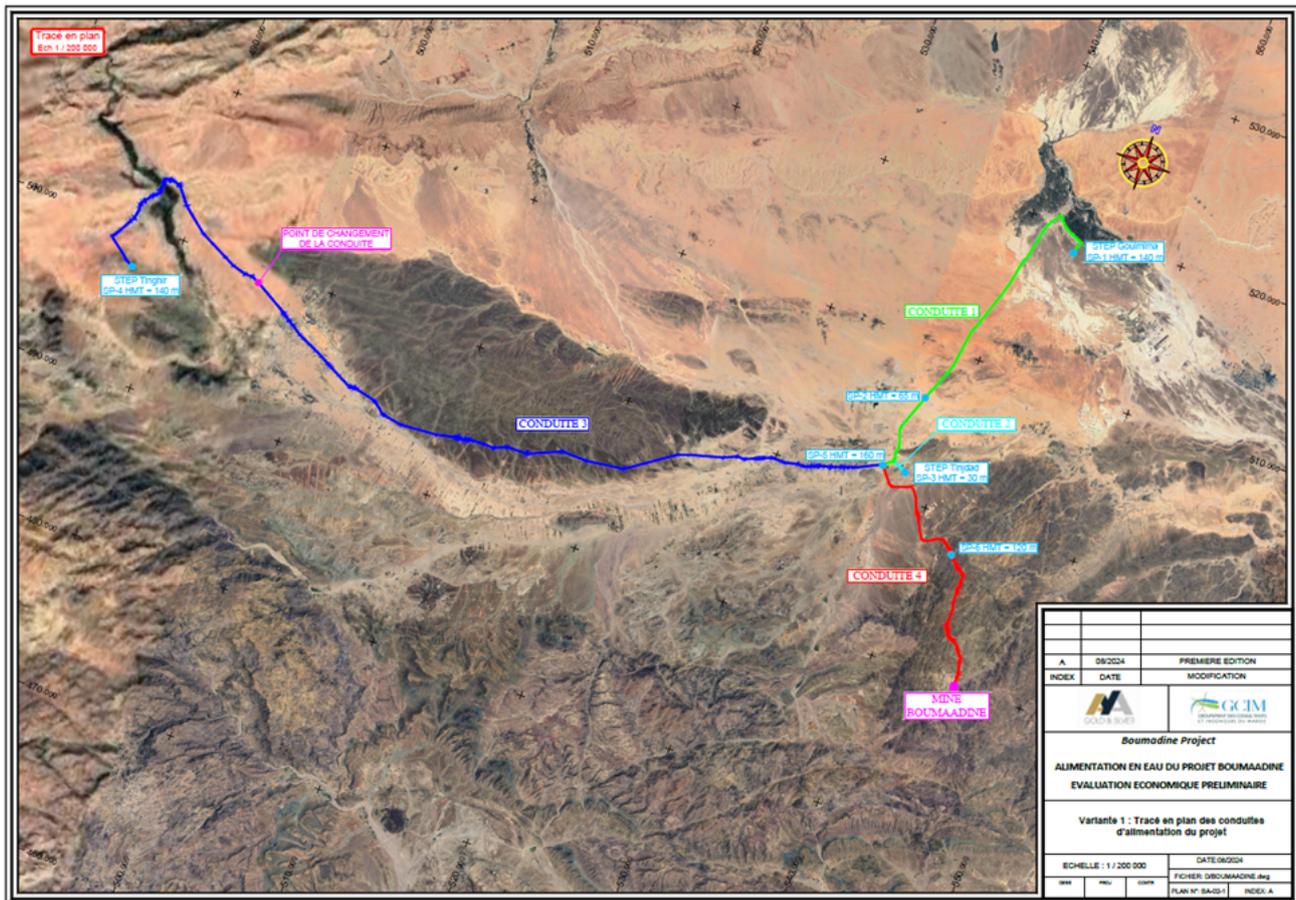


Figure 18-3 Raw Water Sourcing

It is assumed that raw water will also be obtained from nearby boreholes and non-contact surface run-off water. A rainwater catchment pond will also harvest water for raw water usage and supplement in the flotation plant.

Raw water will be provided from the raw water tank for use as fire water. Feed to the potable water treatment plant and other users at the process plant will be sourced directly from boreholes and other non-contact water sources.

18.6.2 Fire Water Supply System

Fire water will be piped to all main facilities via buried underground fire water ring mains around each of the facilities. The raw water tank live volume will hold a four-hour reserve of fire water equalling 288 m³. In addition, all buildings will be equipped with hose cabinets and supplemented with handheld fire extinguishers of two types - general purpose extinguishers for inside plant areas, and dry type extinguishers for inside electrical and control rooms.

18.6.3 Potable Water Supply

The potable water treatment plant will be designed to local drinking water guidelines. The water treatment plant is expected to include multimedia filtration for reduction of turbidity, followed by ultraviolet disinfection for primary disinfection, and the addition of sodium hypochlorite for secondary disinfection. Treated potable water from the potable water treatment plant will be stored in the plant potable water tank and distributed via the plant potable water pump in a piping ring main to serve all potable water users in the process plant, heap leach, feed preparation and crushing facilities.

18.7 WATER MANAGEMENT

Studies and assessments required to complete a site wide water management will be done as part of the next phase of project.

18.8 POWER SUPPLY

The Plant is estimated to have a Maximum Demand (running) of an estimated 26.3 MVA and a connected load of approximately 30.9 MVA. The main substation is sized at 40 MVA @ 65.75% load capacity.

Primary power to the Boumadine site will be provided by Office National de l'Electricité et de l'Eau Potable (ONEE) via a 220 kV high-voltage overhead transmission line. This newly constructed high-voltage overhead line will tee-off from the existing high-voltage overhead line that supplies power to the town of Tinejdad. The overhead line will connect to the main substation located north-east of the process plant. Power will be supplied to all areas of the site via the main 40 MVA substation.

18.9 SEWAGE AND SOLID WASTE MANAGEMENT

The project will utilize a modular sewage treatment plant based on a bio-oxidation process. Sewage and wastewater from various areas of site will be collected and transported to the plant via pump stations, strategically located to ensure efficient gravity flow. The plant will treat the water to the required environmental standards before being released into the environment. Solid waste will be periodically removed from the facility for off-site disposal.

18.10 SECURITY

The Process Plant site area will be surrounded by fencing and access shall be through a main gate located at plant north direction. A 'Main Gate House' at the entrance shall provide station to any security personnel with a more security resources stationed at the 'Plant Security and Changehouse' Building.

18.11 ACCOMMODATIONS

Effort of local hiring will be made given the proximity of the nearby town of Tinejdad and reduce the need for onsite accommodation.

Offsite accommodation shall include 200 prefabricated rooms. Senior staff and expatriate employees will be accommodated in rental units in the town of Tinejdad.

A construction/ permanent camp of 200-300 people capacity will be setup to accommodate any on-site accommodation needs.

18.12 ADMINISTRATION AND PLANT BUILDINGS

Administration and Plant Buildings area will be situated at the plant north entrance. Administration area will include the following:

- Main gate House, Emergency Services / Clinic Building, Plant Security and Changehouse, Admin Building, Admin Mess, Project Office, Laboratory, Warehouse.

Plant area buildings, which are accessible after passing by the Plant Security and Changehouse, will include the following:

- Plant Office, Plant Mess, Plant Workshop.

Three Ablutions facilities are located between the Administration and Plant Buildings. One Ablutions facility will also be located inside the Process Plant area close to the ROM Pad.

18.13 CONCENTRATE TRANSPORTATION AND LOADOUT

The three concentrate products will be shipped by road to the port of Nador-West, located 641 km from Boumadine. The zinc and pyrite concentrates will be shipped in bulk, while the lead concentrate will be shipped in big bags. The road haulage of concentrate will be contractor operated, with transportation along national highways.

A concentrate handling facility will be located near or at the port of Nador-West. This facility will be covered and allow for approximately 250,000 t of storage capacity over a surface area of 13,500 m².

19.0 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

The commodities produced at Boumadine are gold, silver, zinc and lead, in the form of concentrates. The concentrates are sold under the terms of preliminary offers received and detailed in Section 19.2. As most of the value is contained in the gold and silver, AYA considered there were no need to carry out a proper market studies at this stage of the project.

19.2 Commodity price projections

This study used the following prices for the different commodities:

- Gold: \$2800/oz
- Silver: \$30/oz
- Lead: \$1.00/lbs
- Zinc: \$1.20/lbs

Metals prices were kept constant for the entire life of mine plan of Boumadine.

19.3 Contracts

19.3.1 Concentrates sales

Aya Gold & Silver contacted several potential off-taker for the different concentrates that will be produced at Boumadine. Three different commodity traders have well received the request of interest sent by Aya Gold & Silver and have provided preliminary terms for off-take agreement for the purchase of the Lead-concentrate, Zinc-concentrate and Pyrite-concentrate of the Boumadine project. These terms are summarized as follows:

- Lead concentrate:
 - Lead: payable at 95%, subject to a minimum deduction of 3 units
 - Silver: payable at 95%, subject to a minimum deduction of 50 grams
 - Gold: payable at 95%, subject to a minimum deduction of 1 gram
 - Treatment charges: based on the annual Lead Asian Benchmark, with a \$10/dmt in favor of the seller.
 - Refining charges: for silver, based on annual benchmark, with \$0.15 premium per Ag payable ounces in favor of the seller; for gold, \$20 per gold payable ounces.
 - Penalty: for arsenic penalty, \$2.5 for each 0.10% above 0.6% arsenic content in the lead concentrate
- Zinc concentrate:
 - Zinc: payable at 85%, subject to a minimum deduction of 8 units
 - Silver: deduction of 3 ounces, then payable at 70%
 - Gold: deduction of 1 gram, then payable at 70%
 - Treatment charges: based on annual Zinc Asian Benchmark, with a \$15/dmt in favor of the seller.
 - Refining charges: for silver, based on annual benchmark, with \$0.15 premium per Ag payable ounces in favor of the seller; for gold, \$20 per gold payable ounces.
 - Penalty: for Arsenic penalty, \$2.5 for each 0.10% above 0.6% arsenic content in the zinc concentrate; for Cadmium penalty, \$2.5 for each 0.1% above 0.3% cadmium content in the zinc concentrate

- Pyrite concentrate:
 - Gold: if the gold content is between 2g/t and 4g/t in the pyrite concentrate, payable of 57%; if gold content is between 4g/t and 6g/t, payable of 64%,
 - Silver: payable of 60% if silver content is above 40 g/t in the each tonne of pyrite concentrate,
 - Penalty: Arsenic penalty of \$3.5 for each 0.10% above 1% arsenic content between 1% and 1.5%, and penalty of \$4 for each 0.10% above 1.5%
 - No refining charges for gold or silver in the Pyrite concentrate

19.3.2 Other contracts

There are no other material contracts or agreements in place as of the effective date of this Report. AYA Gold & Silver has not hedged nor committed any of its production to an off-take agreement with any of the commodity traders that have expressed their interest to the Boumadine project.

20.0 ENVIRONMENTAL STUDIES, PERMITS, AND SOCIAL OR COMMUNITY IMPACTS

20.1 Project Setting

The Project is located in the Drâa-Tafilalet region of western Morocco, approximately 19 km south of the nearest town of Tinejdad. The biophysical and social characteristics of the Project area are defined by its semi-arid climate and relatively rural and remote nature. Urban settlements are concentrated along main transport routes with villages located along valleys with access to water resources. Climatic conditions influence local ecosystems and land uses including agriculture, concentrated in areas of water availability.

20.2 Environmental and Social Studies

Limited environmental studies have been completed in the past. As part of the environmental and social work currently being undertaken, an environmental baseline sampling program was initiated by BGM in September 2024 and a high-level screening for critical biodiversity habitat was undertaken by SLR in September 2024.

The baseline sampling, covering soils, air quality and water, was undertaken by AfriLabs, a Moroccan-based laboratory specializing in mineral and environmental analysis. Preliminary findings from the surveys and screening assessment reflect the following:

- **Air Quality:** Preliminary results indicate elevated particulate matter less than 10µm in diameter PM₁₀ and dust deposition levels, likely due to a combination of natural arid conditions and historical mining activities on site. Gaseous pollutants (NO₂, SO₂) were well below guideline thresholds.
- **Water Resources:** water resources are influenced by the semi-arid nature of the Project site. The Oued Akrouz, which crosses the area, is ephemeral but plays a key role in local irrigation feeding the Barrage d'Akrouz and acting as a pathway for aquifer recharge. Several wells are located along the valley of the Oued supplying local villages and farmers with domestic and irrigation water. Preliminary results indicate background water quality in the wells around the douars, , with some elevated concentrations of metals and metalloids reflecting the local geology and geochemistry. In the area of historical mining activities, water in the old shafts reflect the local mineralized geology with high acidity and elevated concentrations of some metals, metalloids, and major ion compared to the background groundwater quality from the irrigation well samples..
- **Soils and Sediments:** Background soil samples, i.e. soil samples which are distant from the area of historical mining activities on site, show some differences in chemical composition compared to the soil and sediment samples on site. Preliminary results of all samples indicate elevated levels of heavy metals and trace elements related to the local geology and geochemistry, and in some instances due to historical mining activities. While the survey included a preliminary review of the suitability of the soil for agriculture, environmental factors such as rainfall and evaporation / evapotranspiration, are probably the key limiting factor on soil use for agriculture in the local area.
- **Biodiversity:** most of the survey area includes a varying terrain of hills, small mountains and rocky outcrops all sparsely vegetated with a low density of plant cover. Preliminary flora and fauna surveys have identified the potential for some species of importance to occur on site. This will be confirmed through detailed seasonal surveys. The closest protected area to the Project is the Aferdou (El Kheng) Biological Reserve located 35km northeast of the site. While the Project site falls within an internationally recognised area, the Oasis du Sud Marocain Biosphere Reserve, the core zone of this reserve is located approximately 200km west of the Project site. Further engagement is required during the ESIA to better understand the biosphere reserve and any requirements for the ESIA and Project.

Additional baseline sampling is planned together with subsequent E&S studies as part of the international ESIA process outlined below. These studies will cover the topics of biodiversity, hydrology, hydrogeology, soils and sediments, air quality, noise, GHG, climate change, stakeholder engagement, labour and working conditions, community health and safety, and other social elements.

20.3 Permitting and Applicable Standards

The Project will require national permits to support the development of the mine and the requirement to carry out an Environmental Impact Assessment under Moroccan Law. In addition to meeting national permitting requirements, the Project is being undertaken to meet international ESIA lender standards. At the time of writing, the Project is yet to initiate the national EIA permitting process. The plan is to initiate this process in 2026, at the same time as undertaking an ESIA to international lender standards.

In this regard, applicable standards that will be applied to the ESIA preparation include:

- The existing legal framework in Morocco with respect to environmental protection, EIA processes, sustainable development, climate change, water management, soil protection, waste management, landfill management, oil management, energy efficiency, air quality, pollution prevention and control, biodiversity protection and conservation, cultural heritage protection, property expropriation, mining, labour and working conditions, and occupational health and safety, as well as any regulation that may influence the content of the ESMP measures either for construction or operation;
- The Lenders policies, including the 2024 EBRD Environmental and Social Policy and supporting E&S Requirements, the 2012 IFC Performance Standards, and the World Bank Group Environmental, Safety, and Health Guidelines (general and sector-specific guidelines), and supporting good practice E&S reference documents; and
- The relevant AYA corporate policies, sustainability framework and requirements.

20.4 Key Environmental and Social Issues

20.4.1 Potentially Affected Communities

Available preliminary information on potential project affected persons reflects a Project site that is unoccupied and a study area that, in general, is sparsely populated. Villages are located mainly, along the valleys of the wadis where there is access to water resources and related livelihoods such as date and olive farming, and livestock. The closest settlements to the Project site are the downstream villages and farms located along the Oued Akrouz; the nearest is the village and farms of Bouydoud, about 1.5km to the northeast. These are reported by BGM to be home to large families who have their origins in the area. Farming is subsistence and/or for local markets. To the southwest and west, the villages and farms situated in the adjacent valley, along the Oued Taghoucht, lie approximately 7.5 km from the Project site. These villages are reported by BGM to be younger in age. Similarly, associated farms are for subsistence and/or local markets. With the allocation of land plots (visible from aerial imagery) and the establishment of the new barrage (Barrage de Taghoucht) it can be expected that occupation and farming in this area will increase over time. It was reported by BGM that people in the villages to the south do not usually travel north to the Project area. No villages or farms are located along the planned access road.

Occasionally, groups of nomads pass through the Project site accompanied by their livestock, primarily camels and goats. The BGM exploration team, familiar with the region, observed a decline in nomadic lifestyles attributed to the influence of modernity. Local nomadic populations utilise the land seasonally and predominantly for grazing purposes. Access to water is via two water wells, one of which is located within proximity of the Project site.

The Project site is understood to be located on communal land belonging to local tribes. The land acquisition for the Project facilities is expected to be done through a long-term lease of the required area, with initial discussions underway with local authorities.

Social baseline surveys and a social impact assessment will be undertaken as part of the ESIA.

20.4.2 Environmental and Social Management

As part of the ESIA, potential impacts associated with the Project will be identified and assessed in a systematic approach. The impact assessment will inform a comprehensive Environmental and Social Management and Monitoring Plan to be developed for the Project. It will include:

- Mitigation measures for construction and operations
- Framework management plans covering biodiversity, erosion and sediments, waste, air and noise, water, community health and safety, community development and investment, supply chain, cultural heritage, rehabilitation
- Stakeholder engagement plan including grievance management
- Monitoring plans for Project performance and the receiving environment (water, air, noise and biodiversity)

Water management planning will include the following elements:

- Characterization of the groundwater systems on site
- Environmental testwork to understand the geochemical composition and acid generating potential of the host rock and mineral residues
- Development of a site-wide climatic water balance to understand the water circuit and confirm water use, storage and discharge volume requirements.
- Establishment of an engineered stormwater runoff system catering for the diversion of clean water runoff around Project sites, separation of clean and potentially dirty water runoff on site, storage of potentially dirty water runoff in appropriately designed and engineered facilities, and collection and re-use of potentially dirty water runoff.
- Flood protection analysis and measures, if needed.
- Prioritizing water reuse and recycling.
- Water treatment for potable water supply, raw water supply, and discharge to the environment (if discharge is required).

20.4.3 Mine Closure

A conceptual mine closure plan governed by Aya's HSEC Policy will be prepared as part of the ESIA.

21.0 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COST ESTIMATE

21.1.1 Summary

The capital cost estimates for the Boumadine Project PEA was compiled by Lycopodium and is presented here in summary format. The capital cost estimate reflects the Project scope as described in this report.

All costs are expressed in United States Dollars (US\$) unless otherwise stated and are based on Q4 2025 pricing and deemed to have an overall accuracy of +50%/-30%. The capital cost estimate conforms to AACE International (Association for the Advancement of Cost Engineering) Class 5 estimate standards.

The capital estimate for the Project is summarized in Table 21-1 by the major cost elements.

Table 21-1 Capital Estimate Summary

Item	Directs	Indirects	Contingency	Contingency (%)	Total
OP Mining - Infrastructure	7,358,216	0	1,839,554	25.00%	9,197,770
Open Pit Pre-Stripping Costs	46,523,915	0	11,630,979	25.00%	58,154,894
Process Plant	166,643,296	61,895,704	62,631,157	27.41%	291,170,157
Shipping Infrastructure	10,870,368	0	5,435,184	50.00%	16,305,552
Electrical Line	17,204,211	0	4,301,053	25.00%	21,505,264
Raw Water Supply	30,426,944	536,842	7,606,736	24.57%	38,570,522
Tailings Storage Facility (TSF)	8,817,164	263,158	2,204,291	24.28%	11,284,613
Total	287,844,114	62,695,704	95,648,954	27.29%	446,188,772

The following exchange rates have been used in the compilation of the estimate:

Table 21-2 Currency Exchange Rate

Currency	USD
AUD	0.65
USD	1
GBP	1.35
Euro	1.15
Rand	0.06
CAD	0.72

The capital cost estimate is broken down into mining capex, process plant and infrastructure capex, and tailings facility.

21.1.2 Mining Capex

All costs are reported in 2025 USD unless otherwise specified and conversion rates used as per Table 21-2. Escalation of 3% has been applied to estimates from prior years where required. Contingency is excluded from this estimate and will be applied separately.

Capital costs for mining activities include:

- General surface infrastructure – Roads, electrical distribution and other surface facilities.
- Open Pit facilities – Mineralized material and waste stockpiles
- Underground development – Lateral and vertical

- Underground mobile equipment – Acquisition and sustaining costs
- Underground mine infrastructure – Ventilation, electrical facilities, dewatering and facilities

Equipment costs for open-pit mobile equipment are included in the contractor mining cost. No equipment is forecast to be purchased by the owner. Other than the first year of mining, all waste mining costs for the open-pit operations are classified as an operating cost. A portion of these costs could eventually be reclassified as capital stripping at a later phase. First year mining costs have been capitalized.

A summary of the capital costs is shown in Figure 21-1 and Table 21-3.

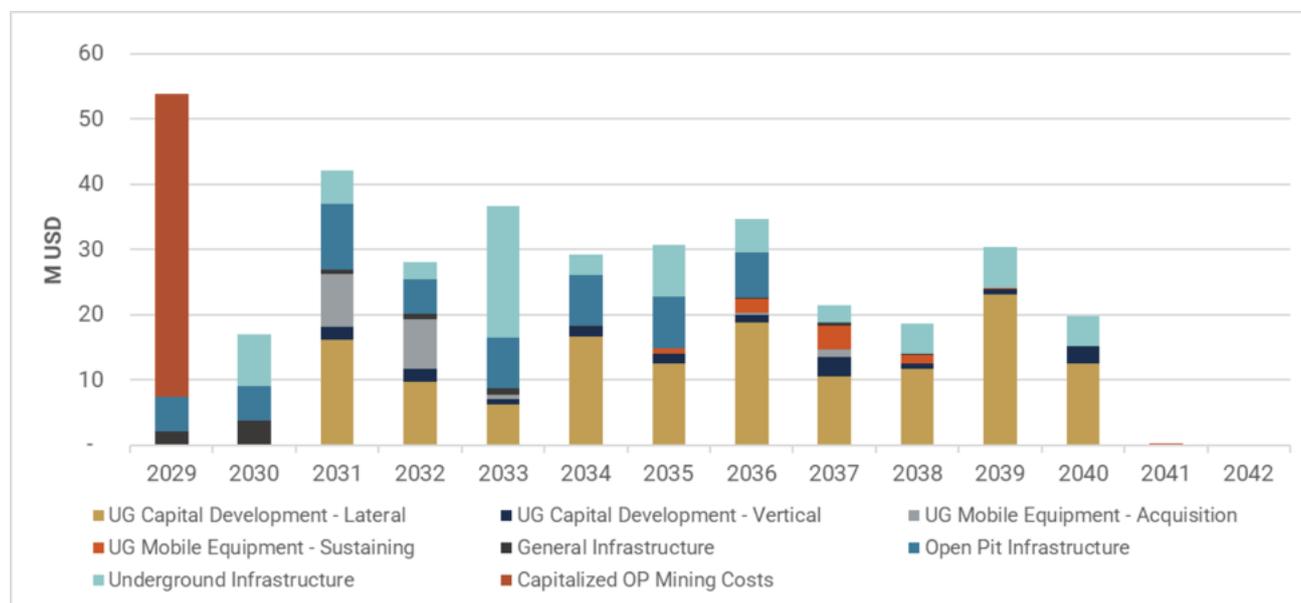


Figure 21-1 Capital Cost Summary – Mining

Table 21-3 Capital Cost Estimate - Mining

Area	\$M
UG Capital Development - Lateral	137.9
UG Capital Development - Vertical	16.7
UG Mobile Equipment - Acquisition	17.8
UG Mobile Equipment - Sustaining	8.2
General Infrastructure	9.0
Open Pit Infrastructure	56.5
Underground Infrastructure	69.8
Capitalized OP Mining Costs (Y1)	46.5
Total	362.5

21.1.2.1 General Surface Infrastructure

Surface facilities include roads, electrical distribution, and a truck shop. Costs are summarized in Table 2.

The plan includes 10 km of site roads and 2 km of service roads, with respective unit costs of \$252,000/km and \$84,000/km. Site roads are 20 m wide, while the service road linking the ventilation raise to the plant is 10 m wide. Electrical distribution will cover the total road length of 12 km at a unit cost of \$95,000/km.

The truck shop has an estimated cost of \$5.1 M. This includes three service bays as well as auxiliary bays, office space and a warehouse.

Table 21-4 General Facilities Cost

Area	\$M
Site Road	2.6
Service Road	0.2
Electrical Distribution	1.1
Surface Facilities	5.1
Total	9.0

21.1.2.2 Open Pit Facilities

Open pit mining will be performed by a contractor, who will provide and maintain all equipment and materials required.

Construction of mineralized material and waste stockpile facilities is included in the OP estimate. Stockpiles assume that geomembrane is required throughout; this requirement may be reduced at a later date as a result of further environmental testing. No clearing is required due to lack of vegetation.

Unit costs include surface preparation, foundations, and construction costs. Construction of waste stockpiles will be distributed over the mine life according to the production plan. Costs are summarized in Table 21-5.

Table 21-5 Open Pit Facilities Cost

Area	m ²	\$/m ²	\$M
Ore Stockpiles	370,000	13.00	4.8
Waste Stockpiles	4,920,509	10.51	51.7
Total			56.5

21.1.2.3 Underground Development

All underground development is forecast to be completed by a contractor. An average unit cost of \$2,150 was provided by Aya for this estimate, based on current contracts. This applies to ramp and lateral development and includes all labour, materials and equipment.

A vertical cost of \$2,442/m was provided for 3-meter diameter raisebores. This quote was scaled to obtain an estimate of \$2,900/m for 4-meter diameter raises. Raisebore costs include allowances for mobilisation/demobilisation.

Table 21-6 summarises the underground development costs.

Table 21-6 Underground Development Cost

Area	\$M
Ramp	82.9
Lateral Development	55.0
Raisebores - 3 m	13.5
Raisebores - 4 m	3.1
Total	154.6

21.1.2.4 Underground Mobile Equipment

Underground mobile equipment is included for all production activities. Mine development is expected to be completed by the contractor who will provide all required equipment.

Recent quotes were provided by Aya for most of the mobile equipment costs. Quotes for similar recent projects were used to estimate costs for the remaining equipment.

Pickup trucks are assumed to be replaced after the useful life of 5 years. An overhaul of 50% of the initial acquisition cost was included for all other equipment at mid-life. Equipment exceeding useful life was replaced at initial cost.

Table 21-7 summarizes the parameters used to develop the mobile equipment estimate.

Table 21-7 Mobile Equipment

Activity	Unit	Quantity	Unit price	Life Years	Sustaining
Drilling & Blasting					
Production Drill	Sandvik DL 311 ITH	3	\$931,000	12	\$465,500
Mobile Compressors for Drill	Atlas Copco	3	\$113,000	12	\$56,500
Emulsion Charger	PAUS	2	\$373,000	8	\$186,500
Mucking & Haulage					
30 tonnes Mine Truck	Epiroc MT436	6	\$720,000	9	\$360,000
10 tonnes LHD 10 t	Epiroc ST1030	7	\$634,000	9	\$317,000
Ground Support					
Cable Bolter	BoltecS	2	\$817,000	12	\$408,500
Transmixer	PARTINDUS	1	\$343,000	9	\$171,500
Service					
Scissor Lift	Paus	2	\$188,000	8	\$94,000
Boom Truck	Paus	3	\$363,000	7	\$181,500
Grader	XCMG GR135	1	\$96,000	15	\$48,000
Fuel Lube Truck	Paus	2	\$287,000	8	\$143,500
Water Truck	Paus	1	\$287,000	8	\$143,500
Pickup Truck	Toyota Hilux	15	\$44,000	5	REPLACE
Mine rescue Truck	Access	1	\$93,000	16	\$46,500
Total - Mine Life			\$17,788,000		\$8,204,000

21.1.2.5 Underground Mine Facilities

Underground facilities costs include egress, ventilation, electrical, dewatering, and rock passes. The estimate is summarised in Table 21-8.

Table 21-8 Underground Facilities Cost

Area	\$M
Egress Raises	3.1
Ventilation – Main Fans	30.3
Ventilation – Auxiliary Fans	2.0
Ventilation – Mercaptan	0.3
Electrical Stations	12.2
Pumping Stations	12.1
General Services - GFP, Lighting and Vent	8.7
Ore Passes	1.2
Total	69.9

Egress assumes safescapes will be installed in all egress raises at a unit cost of \$700/m. A fixed cost of \$75,000 is also included.

Main fans costs have been assumed based on existing quotes for similar projects in Europe and Canada, the cost includes the Electrical substation, the fan with dampers and accessories, mechanical and electrical installation, 15% of material cost to account for freight and civil works. In total there are nine different fan system, one for each zone, the total cost for each fan is shown in the Table 21-9 below.

Table 21-9 Main fans cost

Ventilation Zone	\$M
North 1	2.1
North 2	4.2
North 3	1.6
Centre	4.3
South 5	2.2
South 4	2.2
South 3	4.2
South 2	5.2
South 1	4.3
Total	30.3

Auxiliary fans cost has been estimated based assuming a total of 27 fans are required initially at the start of the project, 22 for the production levels (two (2) per level) and five (5) for the ramp development. It then includes an allowance of 20% for spares per year.

Underground bulkheads with drop board regulators and double personnel doors for egress have not been included in the cost and need to be included in the next phase of the project.

A mercaptan system is included for all surface raises at a unit cost of \$26,000, based on a quote for a similar project.

Electrical stations are estimated at a unit cost of \$329,000. This includes electrical panels and components, cable and installation. Estimate assumes that 30% of electrical panels and components will be reused as the underground mine moves from area to area. Cables and installation are included for each station. A total of 37 electrical stations will be required over the life of the underground mines.

Pumping stations are estimated at a unit cost of \$294,000. This includes pumps, starters, structural allowances, instrumentation, cabling and installation. A total of 41 pumping stations will be required over the life of the underground mines.

An allowance of \$344/m is included for all capital development to account for all general services, such as local electrical distribution, lighting and ventilation ducting.

Ore passes are estimated to cost \$1.2 M for the South and North mining areas. 13 dump points are forecast in this area. The Center mining area includes only trucking to surface. Rock pass costs include truck chutes, grizzlies, lining, discharging gates as well as construction/installation costs.

21.1.3 Process Plant and Infrastructure Capex

The Table 21-10 below shows the Capital cost breakdown of the Process Plant and Infrastructure.

Table 21-10 Process Plant and Infrastructure Capital Cost Breakdown

Process Plant Main Areas	Supply Cost	Installation Hours	Labour Installation Cost	Freight Cost	Subtotal Cost	Project Contingency	Project Total
000 Construction Distributables	3,886,500	5,000	198,620	-	4,085,120	1,021,280	5,106,400
100 Treatment Plant Cost	89,252,329	1,243,741	31,605,949	12,470,446	133,328,724	36,760,703	170,089,427
200 Reagents & Plant Services	11,506,610	138,552	3,680,704	1,595,996	16,783,311	4,791,217	21,574,527
300 Infrastructure	10,063,976	81,544	1,132,876	1,249,289	12,446,141	4,584,031	17,030,173
500 Management Costs	19,965,140	49,913	-	-	19,965,140	4,991,285	24,956,425
600 Owners Project Costs	41,327,536	397,295	-	603,028	41,930,564	10,482,641	52,413,205
Infrastructure							
Shipping Infrastructure	-	-	-	-	10,870,368	5,435,184	16,305,552
Electrical Line	-	-	-	-	17,204,211	4,301,053	21,505,264
Raw Water Supply	-	-	-	-	30,426,944	7,606,736	38,570,522
Grand Total	176,002,091	1,916,045	36,618,149	15,918,760	287,040,523	79,974,130	367,551,495

21.1.3.1 Estimating Methodology

Engineering lists, general arrangement drawings and a layout 2D model have been produced with sufficient detail to permit the assessment of the engineering quantities for concrete, steelwork, mechanical and electrical for the process plant and associated infrastructure. Quantities have also been benchmarked against similar flowsheet and throughput executed Lycopodium projects.

The unit rates and labour rates are based on historical in-house information or budgetary quoted and reflect current project market conditions. Budget pricing for major mechanical equipment was obtained from suitable, reputable suppliers and contractors.

21.1.3.2 Quantity Development

The Project works were quantified to represent the defined scope of work and to enable the application of rates to determine costs.

Quantity information was derived from a combination of sources and categorized to reflect the maturity of design information as follows:

- Material Take Offs: detailed quantities derived from drawings and Navis work model of Boumadine.
- Estimated: includes quantities derived from sketches or redline mark-ups of previous project drawings/data by estimating or similar projects.
- Factored: quantities derived from percentages applied as a factor based on experience or similar projects.
- Allowance: lump sum added to the cost estimate to account for costs that are certain to occur, but they cannot be identified with any accuracy.

The derivation of quantities is provided in Table 21-4 below, weighted by value of the direct permanent works (i.e. excluding temporary works, construction services, commissioning assistance, engineering costs, escalation and contingency).

Table 21-11 Derivation of Quantities

Classification	MTO Prepared	Factored
Earthwork		1
Concrete		1
Structural Steel		1
Platework		1
Mechanical Equipment	1	0
Process Piping		1
Overland Piping		1
Electrical Bulks		1
Electrical Equipment		1
Instrumentation and Control		1
Buildings		1

The table below identifies the sourcing of costs included in the estimate.

Table 21-12 Supply Cost Source

Classification	Total Supply Cost	Estimated / Historical Pricing	Budgetary Quote
F Mechanical	59,877,365	0.48	0.52

21.1.3.3 Installation Basis

This component represents the cost to install the plant equipment and bulk materials on site or to perform site activities. Installation costs are further divided between direct labour, equipment and construction indirect costs.

The labour component reflects the cost of the direct workforce required to construct the Project scope. The labour cost is the product of the estimated work hours spent on site multiplied by the cost of labour, inclusive of overtime premiums, statutory overheads, and payroll burden.

The equipment component reflects the cost of the construction equipment and running costs required to construct the Project. The equipment cost also includes cranes, vehicles, small tools, consumables, and Personal Protective Equipment (PPE).

The installation of the mechanical equipment has been built up using historical data of similar projects. The other disciplines installation costs were estimated as a percentage of mechanical costs and validated against historical data from similar projects.

21.1.3.4 Field Indirects

Construction indirect costs encompass the remaining cost of installation and include items such as offsite management, onsite staff and supervision above trade level, crane drivers, equipment and labour mobilization and demobilization.

Construction indirect costs for all direct labour is included for all works in the capital estimate. This is inclusive of PPE, travel and clothing. Earthworks rates are inclusive of fuel, maintenance and running costs of machinery. Construction equipment and project cranes are included in the capital cost estimate.

Field In-directs costs have been calculated based on a percentage of direct installation cost. The percentage applied is derived from similar projects.

21.1.3.5 Engineering Services

The estimate for engineering design and construction management services was factored based on previous Lycopodium experience for similar sized projects and based on an allowance typical for projects of similar size and nature.

21.1.3.6 Vendor Representatives

Some equipment will require vendor representation during construction and or commissioning. A provision has been included in the estimate to cover the vendor representatives' services, including mob/demob and site expenses. It is based on an allowance typical for projects of similar size and nature.

21.1.3.7 Freight

The freight estimate was derived from the following:

- By calculation as a percentage of supply costs for items where freight tonnes and bulk volumes are yet to be determined.
- By benchmarking against similar projects.

21.1.3.8 Owners Costs

A summary of the Owners costs is given below based on estimates. A total cost, including contingency is shown to be \$52.4M. This includes some general costs, Plant and Admin Pre-Production, Spare Parts and Plant Mobile Equipment. Owner's costs were developed by Aya, and include costs for engineering studies, project-specific home office costs, field staffing, pre-production in-country expenses, and a construction camp.

Table 21-13 Owners Project Cost Total

Plant Area	Supply Cost	Installation Hours	Labour Installation Cost	Freight Cost	Subtotal Cost	Project Contingency	Project Total
610 Owners Costs - General	21,572,167	107,861	-	-	21,572,167	5,393,042	26,965,209
620 Plant & Admin Pre-Production	11,192,110	289,434	-	-	11,192,110	2,798,027	13,990,137
640 Spare Parts	5,750,000	-	-	-	5,750,000	1,437,500	7,187,500
670 Plant Mobile Equipment	2,813,259	-	-	603,028	3,416,288	854,072	4,270,359
Total	41,327,536	397,295	-	603,028	41,930,564	10,482,641	52,413,205

21.1.4 Tailings Storage Facility Capex

The high-level cost estimate was determined for the Capital Expenditure (CAPEX), Engineering Fees, Operating Expenditure (OPEX) and Closure CAPEX costs, as described below and detailed in *Update to the Pre-Economic Assessment of the Boumadine Tailings Storage Facilities (Epoch, 2025)*. The total LoM capital cost estimate is \$42.7 M (406M MAD (Moroccan Dirhams)), and comprises the following:

- The initial LoM CAPEX is \$11.3M (107 MAD),
- The sustaining capital, including contingency, is \$22.0M (209M MAD),
- Engineering consultant and construction supervision fees for the TSF over the LoM amounts to \$0.7M (6,5M MAD), and
- The rehabilitation and closure of the TSF and post-closure monitoring costs amounts to \$9.0M (85,8M MAD).

Table 21-14 LoM Capital Costs (Million MAD)

LoM Costs	TSF (\$M)
TSF Capex	8.8
Contingency	2.2
Engineering Consultant Fees: Monitoring and EOR	0.3
Total Initial Capex	11.3
Sustaining CAPEX (Including Contingency)	22.0
Engineering Consultant Fees: Monitoring and EOR	0.4
Closure CAPEX	9.0
Total	42.7

21.1.5 TSF Closing Costs

The rehabilitation and closure and post-closure monitoring costs were estimated to be \$9.0 M (85.8M MAD) for the study, and includes the following:

- The closure design work amounts to \$0.1M (1.1M MAD).
- Closure CAPEX is \$8.7M (82.6M MAD) phased over three years. The assumptions of the CAPEX estimates are based on the following items and associated works.
 - Spillway channel, and
 - TSF basin capping layer.
- The construction supervision of closure works amounts to \$0.2M (2,1M MAD).

21.1.6 Sustaining and Closing Costs

The sustaining and closing costs are detailed in each respective section.

The overall capital costs for the project, including sustaining and closing costs, are shown in Table 21-15.

Table 21-15 Capital Expenditures including Sustaining and Closing Costs

Capital Expenditures (\$M)	Initial	Sustaining	Total
Direct Costs	288	340	628
Open Pit Mining	54	58	112
Underground Mining	-	250	250
Processing Plant	167	-	167
Shipping Infrastructure	11	-	11
Electrical Line	17	-	17
Raw Water Supply	30	-	30
Tailings Storage Facility	9	22	31
TSF Closure Costs	-	9	9
Indirects Costs	63	-	63
Subtotal	351	340	691
Contingency	96	-	96
Total	446	340	786

21.1.7 Contingency

An amount of contingency has been provided in the estimate to cover anticipated variances between the specific items allowed in the estimate and the final total installed project cost. The contingency does not cover scope changes, etc., or the listed qualifications and exclusions.

Contingency has been applied to the estimate as a percentage allowance and is varied for different cost elements but typically around 25% for all areas except electrical costs where the contingency applied was 50%. The average project contingency is 27%. It should be noted that contingency is not a function of the specified estimate accuracy which should be measured against the project total that includes contingency.

21.1.8 Qualifications and Assumptions

The capital estimate is qualified by the following assumptions:

- The base date for the bulk of pricing for the estimate is Q32025
- Prices of materials and equipment with an imported content have been converted to US\$ at the rates of exchange stated previously in this document. All pricing has been entered into the estimate utilizing native currencies wherever possible.
- Suitable construction/fill materials will be available from borrow pits within 2 km of the work fronts. It must be reviewed following the results of the geotechnical investigation.
- There is no allowance for unforeseen blasting in the bulk earthworks cost estimates.
- It has been assumed mobile equipment purchased and used by the owner's construction team will be handed over to the owner's operations team upon completion of construction. No allowance for additional mobile equipment has been made for operations.

21.1.9 Exclusions

The following items are specifically excluded from the capital cost estimate:

- Permits and licenses
- Project sunk costs
- Federal and state taxes and duties
- Exchange rate variations
- Sustaining Capital Costs

Fuel Distribution

21.1.10 Process Plant Pre-Production and Working Capital Costs

The costs incurred by operations during the latter stages of construction and commissioning are included in the capital cost estimate but are derived in this estimate. Pre-production costs associated with mining are excluded.

21.1.10.1 Pre-Production Labour

The pre-production site administration expenses cost covers the establishment of operations during the eight months preceding start-up and includes provision for power consumption, mobile equipment, and other expenses during this period.

21.1.10.2 First Fill Reagents and Opening Stocks

Costs have been allowed in the project financial model cash flow estimates to purchase the mill balls, regrind ceramic media and reagents needed to commission the plant. Other consumables and reagents required for

the process plant first fill and opening stocks are expected to be paid for in the commissioning and ramp up period, from the operating cost budget.

Sufficient first fill reagents and consumables have been estimated for the initial ball mill steel ball load to fill the reagent tanks, fill the regrind mills with ceramic media, and for other plant consumable requirements. Opening stocks refer to the purchase of the reagents and consumables required to sustain the operations for 6 weeks, which is the minimum on-site start-up storage quantity nominated by Aya Gold and Silver.

Quantities allowed have been based on either consumption over the minimum period or minimum shipping quantities, considering package size.

21.1.10.3 Vendor Representatives

This cost allows for specialist vendor representatives to oversee commissioning of their processing equipment and include allowances for labour, airfares, and expenses.

21.2 OPERATING COST ESTIMATE

21.2.1 Mining Opex

Operating costs for mining activities include:

- Open-pit variable costs – mineralized material and waste mining, grade control and haulage
- Open-pit fixed costs – owners’s labour
- Underground variable costs – definition drilling, developing, stoping, haulage and backfill
- Underground fixed costs – labour, maintenance, electricity, mine services

A summary of the operating costs is shown in Figure 2 and Table 7. Year 1 open pit mining costs have been capitalized.

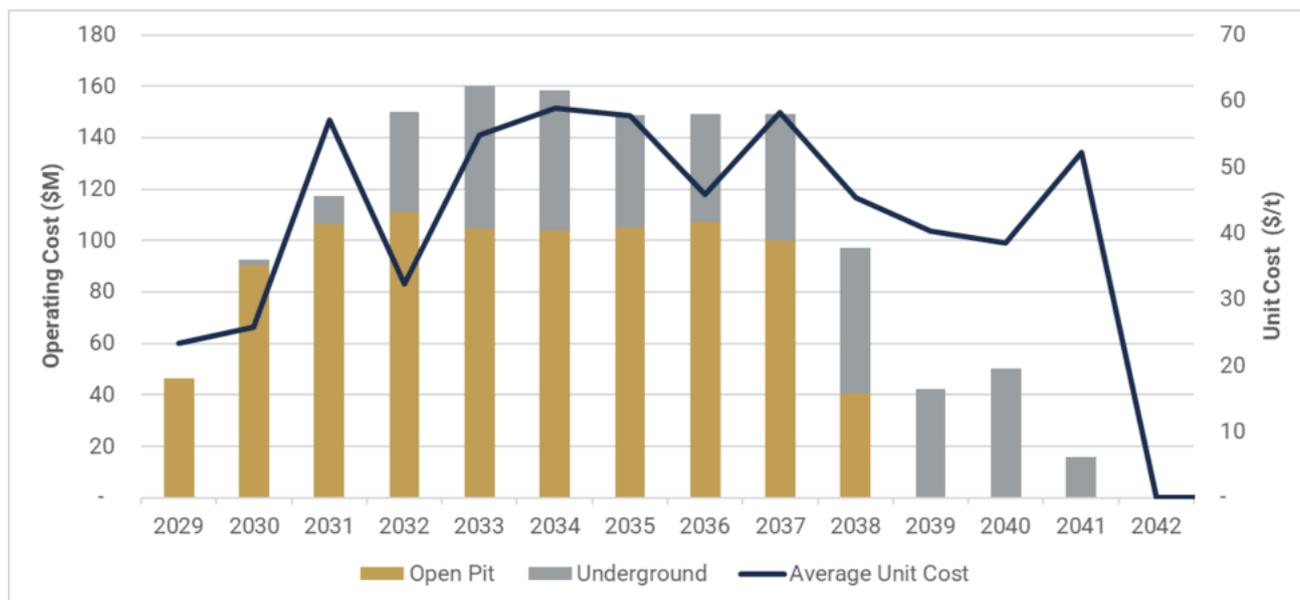


Figure 21-2 Operating Cost Summary – Mining

Table 21-15 Operating Cost Estimate - Mining

Area	\$M
Open-Pit Variable Costs	856.6
Open-Pit Fixed Costs	13.3
Underground Variable Costs	311.8
Underground Fixed Costs	152.2
Total	1,331,1
Unit Cost	\$43/t

21.2.1.1 Open-Pit Costs

Open pit mining will be performed by a contractor. Unit costs were provided by Aya based on current contracts. The following costs were used.

Table 21-16 Open-Pit Unit Costs

Area	\$/t
Ore Mining	1.50
Waste Mining	2.00
Grade Control	1.10
Surface Haulage	2.00

Owner's labour was estimated on an annual basis based on the Aya salary tables. Senior managers are budgeted as expatriate labour while technical services are a blend of local and expatriate labour.

Table 21-17 Open-Pit Owner's Team

Area	QUANTITY
Captains	1
Mine Admin	1
Technical Services	20
Annual Labour Cost	\$1.3 M

21.2.1.2 Underground Variable Costs

Variable costs were calculated for all stoping activities, based on production cycles and provided consumables cost. Costs for definition drilling and development were provided as per current contracts. Cemented rockfill costs were benchmarked from a similar project. Overall cost of backfill in the unit cost considers 94% rockfill and 6% CRF as per the production plan

Unit costs include all consumables (including a 10% loss allowance) and operating costs for direct equipment. Drilling costs include electricity. As no provision is made for a compressed air network, mobile compressors are included for all production drills. Costs are shown in Table 21-18.

Table 21-18 Underground Variable Costs

Activity	\$/T
Definition Drilling	3.00
Development	\$2,150/m
Drilling	3.20
Blasting	2.69
Mucking	1.96
UG Haulage	3.04
Backfill – Rockill	1.18
Backfill – CRF	7.94

21.2.1.3 Underground Fixed Costs

Fixed costs include labour, maintenance of service equipment, electricity and mine services.

LOM costs are summarized in Table 21-19.

Table 21-19 Underground Fixed Costs

Area	\$M
Mining Labour	27.6
Maintenance - Parts and Diesel	9.9
Maintenance Labour	10
Staff and Technical Services	49.5
Electricity	48.9
Infrastructure Maintenance and Mine Services	6.3
Total	152.2

All labour is budgeted on an annual basis according to provided salary tables. All labour is local except for certain managers that are expatriate. 1 supervisor is allocated for every 8 employees. The labour estimate is shown in Figure 21-3.

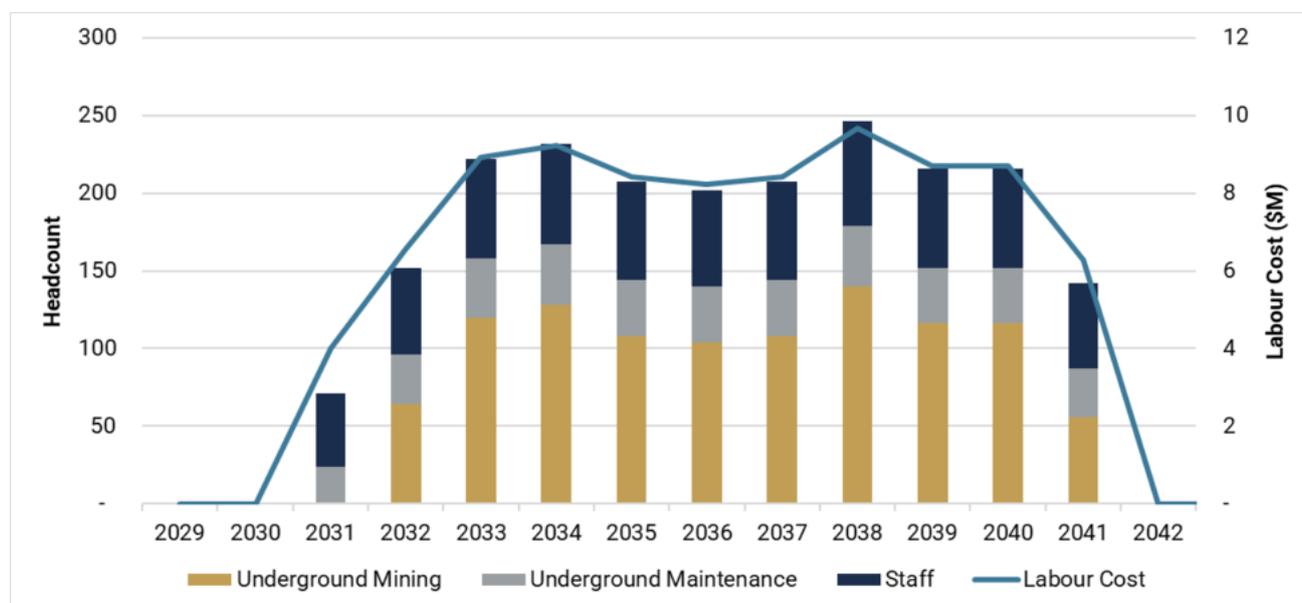


Figure 21-3 Underground Labour Profile

Maintenance costs were developed based on annual operating hours. Annual maintenance costs are based on a 20-30% factor of initial acquisition cost and then converted to hourly rates. Diesel is also factored based on operating hours. Mobile maintenance labour is estimated using a factor of 4 units per mechanic and 12 units per electrician. Twelve (12) mechanics and 12 electricians are included to maintain the mine infrastructure. Diesel is budgeted at \$1.30/L. Mobile maintenance costs are summarized in Table 21-20.

Table 21-20 Mobile Maintenance Costs

Activity	Unit	Annual Hours	Annual Mtc Factor	Parts \$/hr	Diesel \$/hr
Drilling & Blasting					
Production Drill	Sandvik DL 311 ITH	1,500	20%	\$124.13	\$5.57
Mobile Compressors for Drill	Atlas Copco	1,500	20%	\$15.07	\$13.36
Emulsion Charger	PAUS	1,500	20%	\$49.73	\$1.78
Mucking & Haulage					
30 tonnes Mine Truck	Epiroc MT436	5,000	30%	\$43.20	\$29.68
10 tonnes LHD 10 t	Epiroc ST1030	5,000	30%	\$38.04	\$18.55
Ground Support					
Cable Bolter	BoltecS	3,000	20%	\$54.47	\$2.41
Transmixer	PARTINDUS	1,500	20%	\$45.73	\$7.11
Service					
Scissor Lift	Paus	2,500	20%	\$15.04	\$3.29
Boom Truck	Paus	2,000	20%	\$36.30	\$4.57
Grader	XCMG GR135	2,000	20%	\$9.60	\$5.85
Fuel Lube Truck	Paus	2,000	20%	\$28.70	\$4.99
Water Truck	Paus	1,500	20%	\$38.27	\$3.92
Pickup Truck	Toyota Hilux	2,000	20%	\$4.40	\$1.45
Mine rescue Truck	Access	1,200	20%	\$15.50	\$0.59

Electricity for production drills is included in the drilling unit cost. Electricity consumption for the mine is based on the following factors:

- 4.85 kWh/T for dewatering
- 1.37 kW/T for lighting and services
- ~20,000 MWh/year for main fans
- ~20,000 MWh/year for auxiliary fans

Electricity is budgeted at \$0.09/kWh for a total of \$48.9M.

Other fixed costs included in estimate include an annual allowance of \$150,000 for infrastructure maintenance, \$100,000 for mine services, and \$326,000 for ventilation maintenance (1% of initial capital).

21.2.2 Process Operating Cost Estimate

21.2.2.1 Introduction

The operating costs have been compiled by Lycopodium based on costs developed by Aya Gold & Silver and Lycopodium for all processing related costs.

Operating costs for Mining and TSF are detailed separately in Section 21.2.1 and Section 21.2.3 Respectively.

The estimate is considered to have an accuracy of +50 / -30%, is presented in US\$ and is based on information obtained or adjusted for the third quarter of 2025 (Q3 2025).

21.2.2.2 Process Plant Operating Costs

Processing operating costs have been developed by Lycopodium for a life of mine (LOM) blend. It is expected that the plant will operate on a range of mineralized material blends. The LOM processing costs are a weighted average of the various mineralized material type processing costs based on the LOM blend.

Processing operating costs have been developed for a plant with an annual throughput equivalent to 2.92 Mtpa of fresh mineralized material plant feed at a P₈₀ grind size of 58 µm for the SAB circuit, 16 µm for the Pb regrind circuit, and 15 µm for the Zn regrind circuit, based on a 24 hour per day operation, 365 days per year. The plant will operate at the following utilization:

- 6,570 h/y (75%) - Primary jaw crusher with a crushed rock stockpile.
- 8,000 h/y (91.3%) - SAB grinding circuit and all flotation, regrind, thickening, reagents, and services processing plant sections.
- 7,008 h/y (80%) – Concentrate filtration and handling.

The operating costs have been compiled from a variety of sources, including the following:

- Labour pay rates and manning as advised by Aya Gold & Silver.
- Laboratory rates as advised by Aya Gold & Silver.
- Grid power costs as advised by Aya Gold & Silver at a rate of 0.090 US\$/kWh.
- Consumable prices from supplier budget quotations obtained in Q3 2025. Aya Gold & Silver advice, and the Lycopodium database.
- Crushing and grinding consumable prices from supplier budget quotations, OMC modelled crushing and grinding energy and consumables, based on physical mineralized material characteristics determined from comminution testwork and estimated material properties from OMC database.
- Reagent consumptions, mass pulls and recoveries based on the results from a metallurgical testwork program.
- First principal estimates based on typical operating data / standard industry practice.

The processing operating cost estimate is summarized in Table 21-21. The relative proportions of each operating cost center for the processing and process plant (processing plus G&A) operating costs are shown in Figure 21-4.

The operating costs include all direct costs to allow production of concentrates. The battery limits for the processing operating costs are as follows:

- Mineralized material delivered to the ROM bin.
- Tailings discharged onto the tailings storage facility (TSF).
- Pb concentrate in bags, with Zinc and Pyrite concentrates on stockpiles.

Qualifications

The operating cost estimate presented in this section excludes the following:

- All sunk costs.
- ROM stockpile rehandling costs.
- General and Administration:
- All head office costs and corporate overheads.
 - Withholding taxes and other taxes.
 - Any impact from fluctuations in foreign exchange rate.
 - Any escalation from the date of the estimate.

- Project finance costs.
- Interest charges.
- Corporate Overheads.
- Political risk insurance.
- GST and/or VAT.
- All costs associated with areas beyond the battery limits of the Study.
- Plant rehabilitation costs.
- Land or crop compensation costs (included in CAPEX).
- Subsidies to local communities.
- License fees.
- Royalties.
- No allowance for contingency.
- Any business interruption costs.
- Concentrate:
 - Pyrite concentrate processing or further refining costs.
 - In transit insurance costs.
 - Transport or transit costs.
- Process / Tailings:
 - Tailings storage costs, including future lifts and rehabilitation.
 - Tailings dust suppression costs.
 - External Government required monitoring and compliance costs.
 - Raw water sourcing costs.
- Environmental:
 - Any rehabilitation or closure costs.
- Labour:
 - Union fees.
 - Contract labour other than maintenance (generally self-performed by Aya Gold and Silver).
 - Overtime allowances.
- First fill/opening stocks are captured in the CAPEX

The operating cost estimate includes the following:

- Import duties on consumable unit costs (in the consumables cost).
- Costs for the preparation and assaying of mine grade control samples and routine laboratory tests on the site water samples (in the laboratory cost).

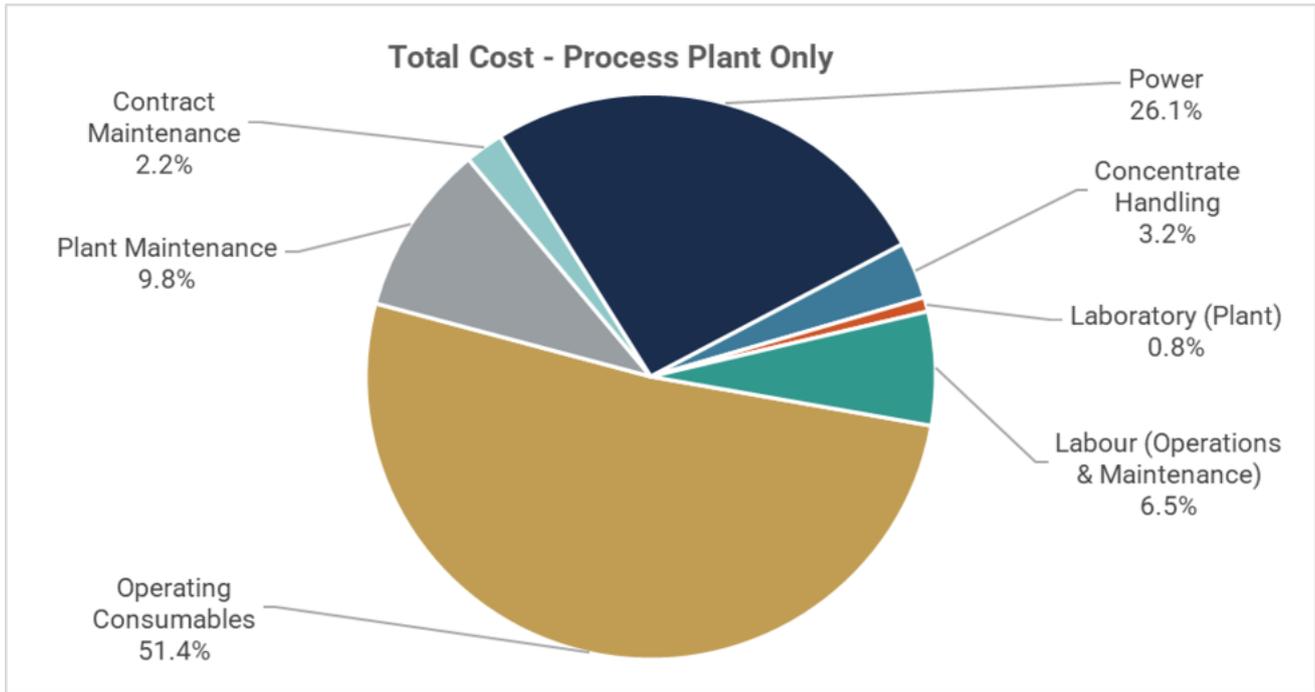


Figure 21-4 Processing Plant Operating Cost Breakdown

Table 21-21 Process Plant Operating Cost Summary (US\$, Q3 2025, +50 / -30%)

Process Stream Costs	LOM Blend	
	US\$/y	US\$/t
Operating Consumables	25,928,100	8.88
Maintenance	4,923,734	1.69
Contract Maintenance	1,102,125	0.38
Power (Plant)	13,177,570	4.51
Concentrate Handling	1,637,364	0.56
Laboratory	396,000	0.14
Process Plant Labour	3,282,230	1.12
Total Process Plant	50,447,122	17.28
General & Administration		
Labour (G&A)	3,460,906	1.19
Laboratory (Exploration, Mining, Environmental)	2,628,793	0.90
Miscellaneous Costs	8,982,750	3.08
Total G&A	15,072,450	5.16
Total Overall OPEX	65,519,572	22.44

21.2.2.3 Power

The power cost estimate has been based on grid power at a unit cost of US\$0.09/kWh. The average continuous power draw and power cost for the LOM blend by plant area is summarized in Table 21-22.

Table 21-22 LOM Blend - Power Cost Summary (US\$, Q3 2025, +50 / -30%)

Plant Area	Connected Power kW	LOM Average Continuous Power Draw kW	LOM Power Cost US\$/y
120 – Feed Preparation – Crushing and Stockpiling	437	230	181,043
130 – Milling – Grinding and Classification	14,774	10,672	8,413,850
150 – Flotation and Concentrate Handling	5,941	4,168	3,286,037
180 – Tails Thickening and Handling	446	308	243,132
210 – Reagents	350	210	165,695
220 – Water Services	423	307	242,274
240 – Air Services	776	522	411,866
250 – Fuel Storage and Distribution	16	12	9,173
330/350/360/370 – Water Supply, Sewage, TSF, and Plant Infrastructure	605	285	224,499
410 – Mining General ¹	7,414	5,171	N/A
Total	31,182	21,885	13,177,570

Note 1: Mining Power Costs are covered in Section 21.2.1

The power consumption for the crushers and ball mills has been calculated by OMC based on the physical mineralized material properties determined from comminution testing and database estimation. The power consumption for the remainder of the individual plant mechanical equipment items has been calculated from the load list derived from the vendor supplied equipment and estimated based on the installed motor size of individual items of equipment, excluding standby equipment, adjusted by efficiency, load, and utilization factors to arrive at the annual average power draw. This is then multiplied by total hours per annum and the electricity price to obtain the power cost.

21.2.2.4 Operating Consumables

Costs for processing operating consumables, including reagents, liners, fuels, and process supplies have been estimated and are summarized for the LOM blend by plant area in Table 21-23.

Table 21-23 Consumables Cost Summary (US\$, Q3 2025, +50 / -30%)

Plant Area	LOM Blend	
	US\$/y	US\$/t
Crushing	733,074	0.25
Milling	9,464,632	3.24
Pb Re grinding	310,115	0.11
Pb Concentrate Handling	36,595	0.01
Zn Re grinding	267,019	0.09
Zn Concentrate Handling	32,126	0.01
Py Concentrate Handling	506,517	0.17
Reagents	12,632,419	4.33
Fuel (Plant)	1,945,602	0.67
Total	25,928,100	8.88

The consumables cost for mining is included in the mining operating cost.

The consumption of reagents and other consumables has been calculated from laboratory testwork and comminution circuit modelling at estimated material properties, calculated from first principles, or has been assumed based on experience with other operations. No additional allowance for process upset conditions and wastage of reagents has been made.

Reagent costs have been sourced from a combination of budget quotations, Client supplied costs, and in-house data relating to similar projects in the region. Transport and freight to site and import duties and taxes have been added based on vendor information. In absence of vendor information, a cost of 17.5% is added for transport, customs, and duties.

A diesel price, delivered to site, of US\$1.30 per litre has been used, as provided by the Client. Diesel will be used in plant mobile equipment and the diesel consumption is based on industry standard vehicle consumption rates and estimated equipment utilization.

Allowances have been made for water treatment reagents.

21.2.2.5 Maintenance

The plant maintenance cost allowance has been factored from the capital supply cost using factors from the Lycopodium database and is summarized for the LOM blend.

The allowance covers mechanical spares and wear parts, but excludes crushing and grinding wear components, grinding media, liners, and general consumables which are allowed for in the consumables cost.

The maintenance cost excludes payroll maintenance labour which is included in the labour cost. No additional contract labour has been allowed for.

Allowances for plant mobile equipment and general maintenance expenses have been made.

The mobile equipment allowance is based on unit costs for maintenance of the light vehicles, portable generators, and other mobile equipment for the process plant.

General maintenance expenses include specialist maintenance software, maintenance manuals, training costs, vendor visits, and control system maintenance and license fees.

21.2.2.6 Labour

The labour rates, manning levels and rosters used to determine the labour operating cost estimate have been provided by Aya Gold & Silver.

The plant labour cost includes all labour costs associated with site-based administration, plant operations, laboratory, and maintenance personnel. The plant labour cost excludes all mining personnel (included in the Mining cost category) and all head office (included in the General & Administrative Overheads costs).

The estimate of the labour contingent has been based on a four shift operation (two shifts working 12 hours per day, tow rotation shifts), to provide continuous coverage for the plant operation with allowance for leave and absenteeism coverage. Provision has been made in the manning numbers to accommodate annual and sick leave requirements.

21.2.2.7 Laboratory Costs

The laboratory cost has been provided by Aya Gold and Silver for a similar operation in the region. The laboratory cost includes, 21,000 mining production samples, 6,000 exploration samples, 4,451 processing plant samples, 177 met lab samples, and 80 environmental samples per month.

21.2.2.8 Services and Utilities

Mobile Equipment

Plant mobile equipment requirements have been agreed with Aya Gold and Silver. Mobile equipment costs provide for the rental fees, fuel, and maintenance of the mobile equipment fleet (excluding the mining fleet

and mining light vehicles). The purchase cost of this equipment has been included in the capital cost estimate.

The rental fees, fuel, and maintenance costs for the mobile equipment are included in the maintenance cost center.

Water Supply

As informed by Aya Gold & Silver, no direct water supply costs are incurred and allowance have been made related to the mechanical equipment maintenance and associated power consumption for pumping water, which are captured under the respective headings.

21.2.3 Site General and Administration

The general and administration costs is based on Lycopodium's internal database, and from information and costs provided by Aya Gold and Silver. Costs were benchmarked with other similar operations by Lycopodium.

The general and administration expenses include the following ongoing operating expenses:

- Site office expenses including communications and communication maintenance, office equipment and supplies, computer supplies and software licenses.
- Insurance expenses covering industrial special risks, third party liability, motor vehicle and security. Labour associated insurances (medical, death and disability and workers liability insurances) are included in the labour costs.
- Financial expenses including banking charges, legal fees, auditing costs and accounting consultants. Concentrate refining and royalties are excluded as they are deducted directly from revenue in a separate cost category.
- Government charges including permits and environmental inspection fees.
- Personnel expenses such as first aid and medical costs, safety supplies, travel and accommodation, expatriate travel, international expat recruiting / relocation costs, training, recreational and local facilities costs, professional memberships and subscriptions, and entertainment allowances. The allowances for expatriate travel (international and regional) and international expat recruiting / relocation include all personnel (mining as well as administration and process plant).
- Contract costs for personnel transport (in country charter flights and on site bussing), camp, catering and cleaning, environmental compliance testing, OH&S and other consultants. The camp catering and cleaning contract cost includes all personnel (mining as well as administration and process plant).
- Community relations expenses including general expenses, community projects and scholarships.
- Infrastructure operating costs
- Co-disposal facility operating costs

21.2.4 Concentrate Shipping and Handling Costs

Ground concentrate shipping costs were provided by Aya, and were developed in cooperation with local Moroccan logistics company. These costs were developed based on the concentrate production from the mine plan, and developed from first principles.

The shipping costs include road transportation, handling at the port, and associated port fees. The total of these costs were estimated to be 479 MAD/t, or \$50.44/t.

Maritime shipping costs were estimated at \$25/t based on costs for similar operations and publicly available data for bulk transportation of concentrates along similar shipping routes.

21.2.5 TSF Opex

The OPEX of \$0.3M to \$0.4M (3.1 to 3.8M MAD) per annum for the TSF comprises:

- \$30k to \$60k (0,27 to 0.57M MAD) per annum for pumping of recovering water,
- \$0.1M to \$0.2m (1.1 to 2.0M MAD) per annum for operational management, pipeline management and equipment maintenance, and
- \$0.2M (1.5M MAD) per annum for quarterly inspections, monitoring and quarterly reports by the design engineer.

22.0 ECONOMIC ANALYSIS

22.1 INTRODUCTION

The preliminary economic assessment is preliminary in nature, it includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the preliminary economic assessment would be realized. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

A preliminary economic analysis has been carried out for the Project using a cash flow model. The model is constructed using annual cash flows by taking into account annual processed tonnages and grades. The associated process recoveries, metal prices, operating costs and refining charges, royalties and capital expenditures (both initial and sustaining) were also taken into account. The financial model was completed on a 100% project basis and includes a 3% gross royalty to the ONHYM (Office National des Hydrocarbures et des Mines). As a general rule, the financial assessment of projects of this nature is carried out on a '100% equity' basis, i.e. the debt and equity sources of capital funds are ignored. No provision is made for the effects of inflation. Results are given before and after taxation. All amounts in this chapter are presented in United States Dollars (US\$) and unless otherwise stated, are referred to '\$'.

The model reflects the base case and technical assumptions shown in the preceding chapters of this report. Discounting has been applied from the first year of mine construction (Year 0 onward).

According to Canadian securities law, the outcomes of the economic assessments mentioned in this section constitute forward-looking information. Results depend on inputs that could differ considerably from those predicted here due to known and unknowable risks, uncertainties, and other factors

22.2 FISCAL AND ECONOMIC PARAMETERS

22.2.1 Royalties and Duties

The economic analysis incorporates royalties due to the ONHYM. The Government of Morocco also collects various taxes and duties on the importation of fuels, supplies, equipment, and outside services as specified in the Mining Code.

The ONHYM is entitled to collect a 3% royalty on the total revenues from metal production.

22.2.2 Taxes

Aya's understanding of current Morocco tax regulations are applied to assess the tax liabilities. Taxation calculations are based on a 35% corporate tax rate as prescribed by Moroccan tax regulations. It also includes a mining tax rate of \$0.32/t extracted.

Depreciation allowance uses a linear method based on life of mine (11 years) for initial investment as prescribed by Moroccan tax regulations. Investments made during the course of the operations are amortized on the remaining years of life.

22.2.3 Economic Parameters

All costs have been estimated in US Dollars. All capital and operating costs are provided in Q4 2025 money terms. Inflation rates have not been applied in the financial model as the evaluation has been carried out on a real terms constant money basis.

No price escalation has been applied to the model, and cost input parameters were considered fixed for the life-of-mine for the purposes of this financial valuation.

No salvage value has been considered within the economic model for the Project equipment or infrastructure such as buildings, earth works, or remediation works.

The financial valuation has been completed using base-case prices shown in Table 22-1 below for the various metals produced, with a sensitivity analysis provided for Gold and Silver.

Table 22-1 Base Case Prices and Sensitivity Ranges

Metal Prices	Units	Base Case Price	Sensitivity Range
Gold Price	\$/oz	\$2,800	\$1567 - \$4000
Silver Price	\$/oz	\$30	\$17 - \$48
Zinc Price	\$/lb	\$1.20	-
Lead Price	\$/lb	\$1.00	-

22.3 PROJECT TIMING

Key assumptions with respect to project timing used in developing the financial model are:

- For the purposes of the financial valuation, project capital expenditure is spent over 2 years prior to production (Year 0). In the actual planned ramp up, it is estimated that 40% of the capital cost is likely to be incurred in Year -1 (2 years before production) of the implementation period, and the remaining 60% of the capital cost incurred in Year 0.
- For the financial valuation it has been assumed that the capital expenditure is incurred in full during the year prior to project commissioning and ramp-up.
- Commissioning, ramp-up and initial mining occurs during Year 1 of the schedule.
- Process plant commissioned and first material in the plant in Year 1 of the schedule.
- It is assumed that the project will reach 100% of name plate capacity within six (6) months of commissioning, in line with experience at other similar projects that have been constructed by Lycopodium elsewhere in Africa during the last few years.

22.4 FINANCIAL MODELLING

The financial assessment of the Project was carried out on a 100% equity basis, not accounting for potential sources of funding which may include debt. The technical parameters and key assumptions described elsewhere in this report are reflected in the financial model, with the LOM Project cash flow shown in Table 22-2.

Table 22-2 Key PEA Assumptions and Economic Parameters

Parameter	Units	Value	Source
General			
Tonnes per day processed	tpd	8000	Aya
Process Plant Capacity Year 1	%	75%	Aya / Lycopodium
Process Plant Capacity Subsequent Years	%	100%	Aya / Lycopodium
Shipping Capacity Year 1	%	75%	Aya / Lycopodium
Shipping Capacity Subsequent Years	%	100%	Aya / Lycopodium
Process Plant CAPEX year 1 Split	%	40%	Aya
Metal Prices			
Gold Price	\$/oz	\$2,800	Aya/Lycopodium
Silver Price	\$/oz	\$30	Aya/Lycopodium
Zinc Price	\$/lb	\$1.20	Aya/Lycopodium
Lead Price	\$/lb	\$1.00	Aya/Lycopodium

Parameter	Units	Value	Source
Other Expenses			
ONHYM Royalties	%	3.00%	Aya
Aya Technical Assistance	%	2.75%	Aya
Mining Tax (per mineralized tonne)	\$/t	0.32	Moroccan Tax Law
Income Taxes	%	35.0%	Moroccan Tax Law
Payables			
Lead Concentrate			
Pb Payable	%	95	Prel. off-taker terms
Pb Minimum Deductible	%	3	Prel. off-taker terms
Au Payable	%	95	Prel. off-taker terms
Au Minimum Deductible	g/t	1	Prel. off-taker terms
Ag Payable	%	95	Prel. off-taker terms
Ag Minimum Deductible	g/t	50	Prel. off-taker terms
Zinc Concentrate			
Zn Payable	%	85	Prel. off-taker terms
Zn Minimum Deductible	%	8	Prel. off-taker terms
Au Deduction	g/t	1	Prel. off-taker terms
Au Payable (After Deduction)	%	70	Prel. off-taker terms
Ag Deduction	g/t	93.3	Prel. off-taker terms
Ag Payable (After Deduction)	%	70	Prel. off-taker terms
Pyrite Concentrate			
<i>Au Payable Level 1</i>			
Au Minimum Grade	g/t	2	Prel. off-taker terms
Au Payable	%	57	Prel. off-taker terms
<i>Au Payable Level 2</i>			
Au Minimum Grade	g/t	4	Prel. off-taker terms
Au Payable	%	64	Prel. off-taker terms
<i>Ag Payable</i>			
Ag Minimum Grade	g/t	40	Prel. off-taker terms
Au Payable	%	60	Prel. off-taker terms
Deductibles			
Lead Concentrate			
Treatment Charge	\$/t	100	Estimated based on historical TC Values
Au Refining Charge	\$/oz	20	Prel. off-taker terms
Ag Refining Charge	\$/oz	1.5	Prel. off-taker terms
Zinc Concentrate			
Treatment Charge	\$/t	200	Estimated based on historical TC Values
Au Refining Charge	\$/oz	20	Prel. off-taker terms
Ag Refining Charge	\$/oz	1.5	Prel. off-taker terms
Penalties			
Lead Concentrate			
<i>Arsenic Penalty Level 1</i>			
As grade threshold	%	0.6	Prel. off-taker terms
As penalty interval	%	0.1	Prel. off-taker terms
As penalty per interval	\$	2.5	Prel. off-taker terms
Zinc Concentrate			
<i>Arsenic Penalty Level 1</i>			

Parameter	Units	Value	Source
As grade threshold	%	0.6	Prel. off-taker terms
As penalty interval	%	0.1	Prel. off-taker terms
As penalty per interval	\$	2.5	Prel. off-taker terms
<i>Cadmium Penalty Level 1</i>			
Cd grade threshold	%	0.3	Prel. off-taker terms
Cd penalty interval	%	0.1	Prel. off-taker terms
Cd penalty per interval	\$	2.5	Prel. off-taker terms
Pyrite Concentrate			
<i>Arsenic Penalty Level 1</i>			
As grade threshold	%	1	Prel. off-taker terms
As penalty interval	%	0.1	Prel. off-taker terms
As penalty per interval	\$	3.5	Prel. off-taker terms
<i>Arsenic Penalty Level 2</i>			
As grade threshold	%	1.5	Prel. off-taker terms
As penalty interval	%	0.1	Prel. off-taker terms
As penalty per interval	\$	4	Prel. off-taker terms

Table 22-3 Cash Flow Model

Parameter	Unit	yr -1	yr 0	yr 1	yr 2	yr 3	yr 4	yr 5	yr 6	yr 7	yr 8	yr 9	yr 10	yr 11	yr 12	yr 13	yr 14	yr 15	Total
Mining																			
Open Pit Mining																			
Mineralized Material	kt	-	1,998	3,589	2,025	3,789	1,340	1,038	1,395	2,262	1,460	547	-	-	-	-	-	-	19,442
Waste	kt	-	18,001	36,410	47,974	46,209	48,659	48,976	48,614	47,744	45,939	18,318	-	-	-	-	-	-	406,844
Strip Ratio		-	9.0	10.1	23.7	12.2	36.3	47.2	34.8	21.1	31.5	33.5	-	-	-	-	-	-	20.9
Underground Mining																			
Mineralized Material	kt	-	-	-	31	845	1,581	1,654	1,182	991	1,094	1,599	1,054	1,293	316	-	-	-	11,640
Waste to Surface	kt	-	-	-	360	244	192	425	305	480	336	371	545	372	11	-	-	-	3,642
Processing																			
Feed to Processing Plant	kt	-	-	2,190	2,920	2,920	2,920	2,920	2,920	2,920	2,920	2,920	2,920	2,295	316	0	0	0	31,082
Pb	%	-	-	0.63	0.86	0.69	0.68	0.45	0.52	0.72	1.05	0.82	0.61	0.66	0.51	0.00	0.00	0.00	0.70
Zn	%	-	-	2.41	2.95	1.80	1.69	1.48	1.89	1.50	2.09	1.87	1.88	1.48	1.59	0.00	0.00	0.00	1.91
Au	g/t	-	-	2.85	1.96	4.04	3.85	2.99	2.22	3.85	1.49	1.26	0.88	1.28	1.97	0.00	0.00	0.00	2.43
Ag	g/t	-	-	107	91	92	90	55	46	67	74	82	43	59	72	0	0	0	73
Fe	%	-	-	17.3	16.4	18.6	17.5	16.1	18.5	18.4	15.0	12.8	10.4	11.2	14.8	0.0	0.0	0.0	15.7
As	%	-	-	0.6	0.5	0.6	0.6	0.7	1.1	0.8	0.5	0.4	0.4	0.4	1.3	0.0	0.0	0.0	0.6
Cu	%	-	-	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.0	0.0	0.0	0.0	0.1	0.0	0.0	0.0	0.1
S	%	-	-	19.9	19.1	20.9	19.7	17.5	19.9	20.2	17.3	15.0	11.8	12.8	16.5	0.0	0.0	0.0	17.7
Au-Eq ¹	g/t	-	-	4.77	3.85	5.67	5.42	4.08	3.31	5.12	3.02	2.76	1.92	2.42	3.27	0.00	0.00	0.00	3.85
Ag-Eq ¹	g/t	-	-	444	359	528	505	380	308	477	281	256	179	225	304	0	0	0	358
Lead Concentrate	kt			38	70	56	55	36	42	58	85	66	49	42	4	0	0	0	601
Pb Grade	%	-	-	29.6	29.6	29.6	29.6	29.6	29.6	29.6	29.6	29.6	29.6	29.6	29.6	0.0	0.0	0.0	29.6
Au Grade	g/t	-	-	39.1	19.5	50.1	48.9	57.5	36.9	45.9	12.2	13.3	12.3	16.7	33.2	0.0	0.0	0.0	29.9
Ag Grade	g/t	-	-	3,116	1,912	2,430	2,414	2,237	1,608	1,690	1,279	1,822	1,272	1,620	2,584	0	0	0	1,892
As Grade	%	-	-	1.6	1.0	1.4	1.5	2.8	3.8	1.9	0.7	0.8	1.2	0.9	4.3	0.0	0.0	0.0	1.5
Zinc Concentrate	kt			69	112	68	64	56	72	57	79	71	71	44	7	0	0	0	772
Zn Grade	%	-	-	57.4	57.4	57.4	57.4	57.4	57.4	57.4	57.4	57.4	57.4	57.4	57.4	0.0	0.0	0.0	57.4
Au Grade	g/t	-	-	0.9	0.5	1.7	1.7	1.6	0.9	2.0	0.5	0.5	0.4	0.7	1.0	0.0	0.0	0.0	1.0
Ag Grade	g/t	-	-	157	108	181	187	131	85	157	125	154	81	140	161	0	0	0	134
As Grade	%	-	-	0.1	0.1	0.2	0.2	0.2	0.3	0.2	0.1	0.1	0.1	0.1	0.4	0.0	0.0	0.0	0.1
Pyrite Concentrate	kt			886	1,132	1,237	1,166	1,037	1,181	1,198	1,027	887	701	597	106	0	0	0	11,156
Au Grade	g/t	-	-	5.0	3.6	6.8	6.9	6.0	3.9	6.7	3.0	3.0	2.6	3.5	4.2	0.0	0.0	0.0	4.8
Ag Grade	g/t	-	-	110	97	90	93	64	47	67	87	111	74	93	89	0	0	0	83.7
As grade	%	-	-	1.3	1.1	1.2	1.3	1.8	2.4	1.7	1.1	1.1	1.6	1.2	3.4	0.0	0.0	0.0	1.5
S Grade	%	-	-	44.3	44.3	44.3	44.3	44.3	44.3	44.3	44.3	44.3	44.3	44.3	44.3	44.3	44.3	44.3	44.3
Metal Profile																			
Metal Produced																			
Pb	Mlbs	-	-	25	45	36	36	24	27	38	55	43	32	27	3	0	0	0	392
Pb - Pb Concentrate	Mlbs	-	-	25	45	36	36	24	27	38	55	43	32	27	3	0	0	0	392
Zn	Mlbs	-	-	87	142	86	81	71	91	72	100	90	90	56	8	0	0	0	975
Zn - Zn Concentrate	Mlbs	-	-	87	142	86	81	71	91	72	100	90	90	56	8	0	0	0	975
Au	koz	-	-	193	176	365	347	270	201	347	135	114	79	91	19	0	0	0	2,337
Au - Pb Concentrate	koz	-	-	48	44	90	86	67	50	86	33	28	20	22	5	0	0	0	577
Au - Zn Concentrate	koz	-	-	2	2	4	4	3	2	4	1	1	1	1	0	0	0	0	24
Au - Py Concentrate	koz	-	-	143	131	271	258	200	149	258	100	85	59	67	14	0	0	0	1,735
Ag	koz	-	-	7,278	8,192	8,356	8,108	4,960	4,129	6,032	6,687	7,391	3,869	4,162	710	0	0	0	69,874

Parameter	Unit	yr -1	yr 0	yr 1	yr 2	yr 3	yr 4	yr 5	yr 6	yr 7	yr 8	yr 9	yr 10	yr 11	yr 12	yr 13	yr 14	yr 15	Total
Ag - Pb Concentrate	koz	-	-	3,805	4,283	4,369	4,239	2,593	2,159	3,154	3,496	3,864	2,023	2,176	371	0	0	0	36,532
Ag - Zn Concentrate	koz	-	-	347	391	399	387	237	197	288	319	353	185	199	34	0	0	0	3,334
Ag - Py Concentrate	koz	-	-	3,125	3,518	3,589	3,482	2,130	1,773	2,591	2,872	3,174	1,662	1,787	305	0	0	0	30,008
Au-eq¹	koz	-	-	317	341	504	482	362	294	456	269	247	171	169	31	0	0	0	3,643
Ag-eq¹	koz	-	-	29,612	31,843	47,060	44,968	33,778	27,399	42,561	25,128	23,055	15,937	15,761	2,937	0	0	0	340,038
Metal Payable																			
Pb	Mlbs	-	-	22	41	33	32	21	24	34	50	39	29	25	3	0	0	0	352
Pb - Pb Concentrate	Mlbs	-	-	22	41	33	32	21	24	34	50	39	29	25	3	0	0	0	352
Zn	Mlbs	-	-	74	120	73	69	61	77	61	85	77	77	47	7	0	0	0	829
Zn - Zn Concentrate	Mlbs	-	-	74	120	73	69	61	77	61	85	77	77	47	7	0	0	0	829
Au	koz	-	-	137	116	260	248	192	132	248	88	74	51	59	14	0	0	0	1,619
Au - Pb Concentrate	koz	-	-	45	41	86	82	63	47	81	31	26	18	21	5	0	0	0	546
Au - Zn Concentrate	koz	-	-	0	0	1	1	1	0	1	0	0	0	0	0	0	0	0	4
Au - Py Concentrate	koz	-	-	92	75	173	165	128	85	165	57	48	33	38	9	0	0	0	1,069
Ag	koz	-	-	5,588	6,218	6,439	6,252	3,789	3,115	4,632	5,101	5,672	2,919	3,186	545	0	0	0	53,457
Ag - Pb Concentrate	koz	-	-	3,615	4,069	4,150	4,027	2,464	2,051	2,996	3,321	3,671	1,922	2,067	352	0	0	0	34,705
Ag - Zn Concentrate	koz	-	-	98	38	136	136	47	0	82	57	97	0	46	10	0	0	0	747
Ag - Py Concentrate	koz	-	-	1,875	2,111	2,153	2,089	1,278	1,064	1,554	1,723	1,904	997	1,072	183	0	0	0	18,005
Au-eq¹	koz	-	-	237	249	372	356	266	207	336	197	182	126	123	23	0	0	0	2,673
Ag-eq¹	koz	-	-	22,088	23,228	34,735	33,202	24,860	19,342	31,325	18,351	16,950	11,753	11,444	2,191	0	0	0	249,470
Revenue																			
Lead Concentrate	\$M			247	264	382	367	262	206	338	221	209	128	137	24	0	0	0	2,786
Pb	\$M	-	-	22	41	33	32	21	25	34	50	39	29	25	3	0	0	0	352
Au	\$M	-	-	127	116	240	228	177	132	228	86	73	50	59	13	0	0	0	1,529
Ag	\$M	-	-	108	122	125	121	74	62	90	100	110	58	62	11	0	0	0	1,041
Penalties	\$M	-	-	-1	-1	-1	-1	-2	-3	-2	0	0	-1	0	0	0	0	0	-13
Deductions	\$M	-	-	-6	-7	-8	-8	-5	-4	-6	-6	-6	-3	-4	-1	0	0	0	-63
TC	\$M	-	-	-4	-7	-6	-6	-4	-4	-6	-9	-7	-5	-4	0	0	0	0	-60
Zinc Concentrate	\$M			78	123	81	77	64	78	68	88	80	77	49	7	0	0	0	869
Zn	\$M	-	-	89	145	88	83	73	93	74	103	92	92	57	9	0	0	0	995
Au	\$M	-	-	0	0	3	3	2	0	4	0	0	0	0	0	0	0	0	12
Ag	\$M	-	-	3	1	4	4	1	0	2	2	3	0	1	0	0	0	0	22
Penalties	\$M	-	-	0	-1	0	0	0	0	0	0	0	0	0	0	0	0	0	-4
Deductions	\$M	-	-	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	-1
TC	\$M	-	-	-14	-22	-14	-13	-11	-14	-11	-16	-14	-14	-9	-1	0	0	0	-154
Pyrite Concentrate	\$M			305	268	541	514	368	204	478	206	188	110	135	21	0	0	0	3,337
Au	\$M	-	-	257	209	485	462	359	238	462	160	135	94	107	26	0	0	0	2,993
Ag	\$M	-	-	56	63	65	63	38	32	47	52	57	30	32	6	0	0	0	540
Penalties	\$M	-	-	-8	-5	-9	-11	-30	-66	-31	-5	-4	-14	-4	-10	0	0	0	-197
Deductions	\$M	-	-	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
TC	\$M	-	-	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Total Revenues	\$M	-	-	629	654	1,004	957	694	489	884	515	476	315	322	53	0	0	0	6,991
Operating Expenditures																			
Open Pit Mining Cost	\$M	-	-	91	107	111	105	104	105	107	100	41	0	0	0	0	0	0	870
UG Mining Cost	\$M	-	-	2	11	39	56	55	44	42	49	57	42	50	16	0	0	0	461
Processing Charges	\$M	-	-	38	51	51	51	51	51	51	51	51	51	40	6	0	0	0	537
G&A	\$M	-	-	15	15	15	15	15	15	15	15	15	15	15	8	0	0	0	173

Parameter	Unit	yr -1	yr 0	yr 1	yr 2	yr 3	yr 4	yr 5	yr 6	yr 7	yr 8	yr 9	yr 10	yr 11	yr 12	yr 13	yr 14	yr 15	Total
TSF	\$M	-	-	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	4
Raw water cost	\$M	-	-	1	1	1	1	1	1	1	1	1	1	1	1	0	0	0	11
Concentrate Shipping	\$M	-	-	88	116	120	113	100	114	116	105	90	72	60	10	0	0	0	1,105
Total Costs	\$M	-	-	234	300	337	341	325	330	332	321	254	182	166	40	0	0	0	3,162
ONHYM Royalties (on sales	\$M	-	0	19	20	30	29	21	15	27	15	14	9	10	2	0	0	0	210
Mining Taxes	\$M	-	1	1	1	2	1	1	1	1	1	1	0	0	0	0	0	0	10
Aya technical assistance	\$M	-	0	17	18	28	26	19	13	24	14	13	9	9	2	0	0	0	192
Total of Expenses	\$M	-	1	272	338	396	397	366	359	384	351	282	200	185	43	0	0	0	3,573
Taxes (Excluding Interest)																			
EBITDA	\$M	0	-1	358	316	608	561	328	130	500	163	194	115	136	10	0	0	0	3,418
Amortization	\$M	0	0	57	62	65	70	75	80	88	93	101	116	136	0	3	3	3	-
Use of LCF	\$M	0	0	-1	0	0	0	0	0	0	0	0	0	0	-1	0	0	0	-
Taxable income	\$M	0	-1	300	254	543	491	253	50	412	70	93	-2	0	8	-3	-3	-3	-
VAT	\$M	0	0	42	12	7	1	-3	1	0	-2	-12	-13	-3	-23	-7	0	0	-
Taxes payable	\$M	0	0	147	101	197	172	86	18	145	23	21	-13	-3	-20	-7	0	0	866
Capital Expenditures																			
Initial Capital Expenditure	\$M	155	292	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	446
Sustaining Capital Expenditure	\$M	-	-	17	45	31	37	36	31	39	21	23	30	20	0	3	3	3	340
Total Capital Expenditure	\$M	155	292	17	45	31	37	36	31	39	21	23	30	20	0	3	3	3	786
Cashflow Statement																			
Revenue	\$M	0	0	629	654	1,004	957	694	489	884	515	476	315	322	53	0	0	0	6,991
Production Costs	\$M	0	0	234	300	337	341	325	330	332	321	254	182	166	40	0	0	0	3,162
Royalties	\$M	0	0	19	20	30	29	21	15	27	15	14	9	10	2	0	0	0	210
Mining Taxes	\$M	0	1	1	1	2	1	1	1	1	1	1	0	0	0	0	0	0	10
Aya technical assistance	\$M	0	0	17	18	28	26	19	13	24	14	13	9	9	2	0	0	0	192
Income Taxes	\$M	0	0	147	101	197	172	86	18	145	23	21	-13	-3	-20	-7	0	0	866
Capex	\$M	155	292	17	45	31	37	36	31	39	21	23	30	20	0	3	3	3	786
Pre-Tax Free Cash Flow	\$M	-155	-292	358	289	604	550	311	113	485	156	184	93	125	11	-3	-3	-3	2,824
Pre-Tax Free Cash Flow - 5% Discount	\$M	-155	-292	350	268	535	464	250	86	353	108	121	59	75	6	-2	-2	-2	2,224
Post-Tax Free Cash Flow	\$M	-155	-292	211	188	408	378	226	94	341	134	163	106	128	31	4	-3	-3	1,958
Post-Tax Free Cash Flow - 5%	\$M	-155	-292	206	175	361	319	181	72	248	93	108	67	77	18	2	-2	-2	1,475

¹Au-Eq and Ag-Eq are calculated at metal prices of Au = \$2800/oz, Ag = \$30/oz, Pb = \$1.00/lb, Zn = \$1.20/lb

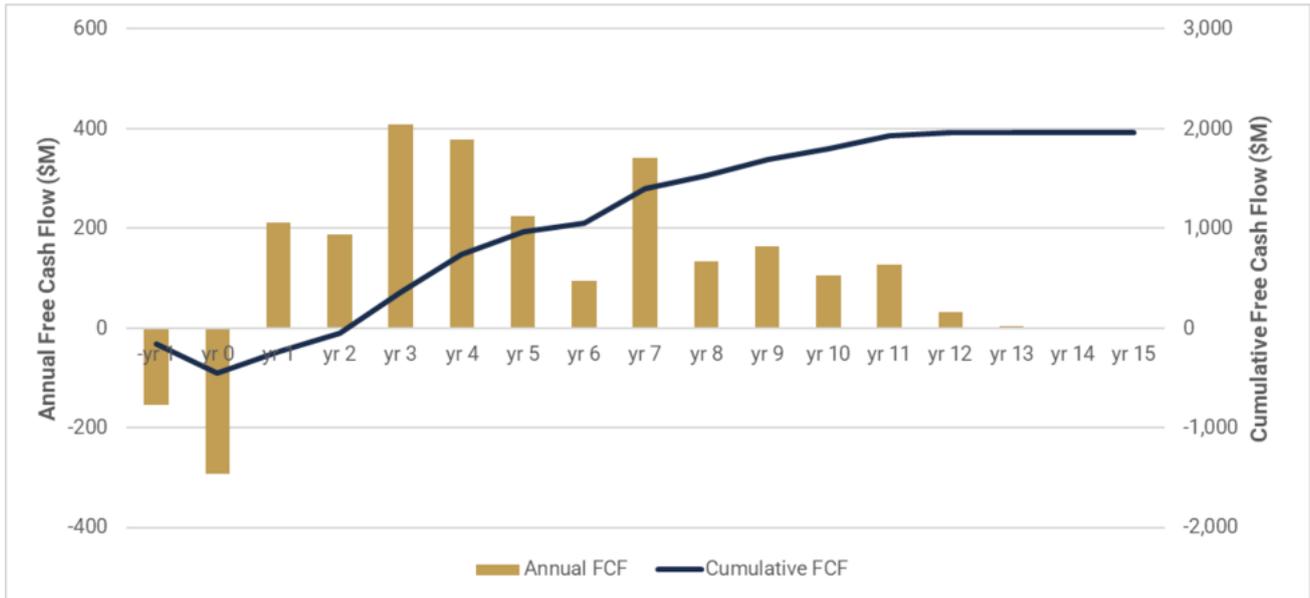


Figure 22-1 Boumadine Project - Annual Free Cash Flow Profile

22.5 RESULTS

22.5.1 Key Economic Results

On a pre-tax basis, the Project has a Net Present Value (NPV) of \$2,224M at a discount rate of 5%, an Internal Rate of Return (IRR) of 69%, and a payback period of 1.3 years. On a post-tax basis, the NPV is \$1,475M at a discount rate of 5%, the IRR is 47%, and the payback period is 2.1 years. The payback period corresponds to the time at which the cumulative cash flow becomes positive.

The key economic results of the PEA for the Boumadine Project are presented in the tables below.

Table 22-4 Key Economic Results

Project Economics	Units	Base Case	
		Pre-tax	Post-tax
NPV _{5%}	\$B	2.2	1.5
IRR	%	69%	47%
Payback	Years	1.3	2.1
NPV: Capex ¹	-	5.0	3.3
Revenue LOM	\$B	7.0	-
Avg. Annual Revenue	\$M/y	629	-
EBITDA LOM	\$B	3.4	-
Avg. Annual EBITDA	\$M/y	308	-
Cumulative FCF LOM	\$B	2.8	2.0
Avg. Annual FCF	\$M/y	254	176

1. NPV:Capex ratio is the ratio of Net Present Value, discounted at 5%, to the initial capital expenditure.

Table 22-4 Additional Economic Model Outputs

Item	Units	Year 1-5	LOM
General			
Total Mine Life	years	-	11.1
Open Pit Strip Ratio ¹	-	19.4	20.9
Throughput Capacity	tpd	8,000	8,000
Total Tonnes Processed	Mt	13.9	31.1
Open-pit	Mt	10.4	19.4
Underground	Mt	3.5	11.6
Processed Grade			
Au	g/t	3.15	2.43
Ag	g/t	85.8	72.5
Zn	%	2.05	1.91
Pb	%	0.66	0.70
Au-eq	g/t	4.76	3.85
Ag-eq	g/t	443	358
Recoveries			
Au rec (%)	%	96.1%	96.1%
Ag Rec (%)	%	96.4%	96.4%
Zn rec (%)	%	74.7%	74.7%
Pb Rec (%)	%	82.0%	82.0%
Metal Production			
Gold Production	koz	1,351	2,337
Silver Production	koz	36,894	69,874
Zinc Production	Mlbs	468	975
Lead Production	Mlbs	166	392
Gold-Equivalent Production (Au-eq)	koz	2,006	3,643
Silver-Equivalent Production (Ag-eq)	koz	187,261	340,038
Avg. Annual AuEq Production	koz/y	401	328
Avg. Annual AgEq Production	koz/y	37,452	30,611
Cost per tonne milled			
Mining	\$/t milled	48.93	42.83
Processing	\$/t milled	17.28	17.28
G&A	\$/t milled	5.43	5.58
Tailings, Environmental and Water Management	\$/t milled	0.46	0.48
Total On-Site Operating Cost	\$/t milled	72.10	66.16
Transportation	\$/t milled	38.70	35.56
Royalties	\$/t milled	8.52	6.75
Mining Tax	\$/t milled	0.36	0.32
Total Cash Cost	\$/t milled	119.68	108.78
OP Sustaining Capital	\$/t milled	3.06	1.87
UG Sustaining Capital	\$/t milled	7.95	8.06
TSF Sustaining Capital	\$/t milled	0.95	1.01
Total Costs Including Sustaining	\$/t milled	131.65	119.72
Operating Cost per Ounce			
Total Cash Costs ¹	\$/oz AuEq	827	928
Total AISC ²	\$/oz AuEq	910	1021

Item	Units	Year 1-5	LOM
Total Cash Costs ¹	\$/oz AgEq	8.9	9.9
Total AISC ²	\$/oz AgEq	9.8	10.9

1. Cash costs include mine-site operating costs such as mining, processing, and direct site G&A, as well as product shipping, royalties and mining taxes.
2. AISC is calculated as the sum of treatment and refining charges, onsite operating costs, sustaining capital costs, and closure costs, divided by the quantity of ounces equivalent produced.

22.5.2 Life of Mine Cash Flow Analysis

Figure 22-2 shows the cumulative and annual cash flow for the project at a base case metal prices.

The cash flows starting for years 13 through 15 relate to closure and environmental monitoring.

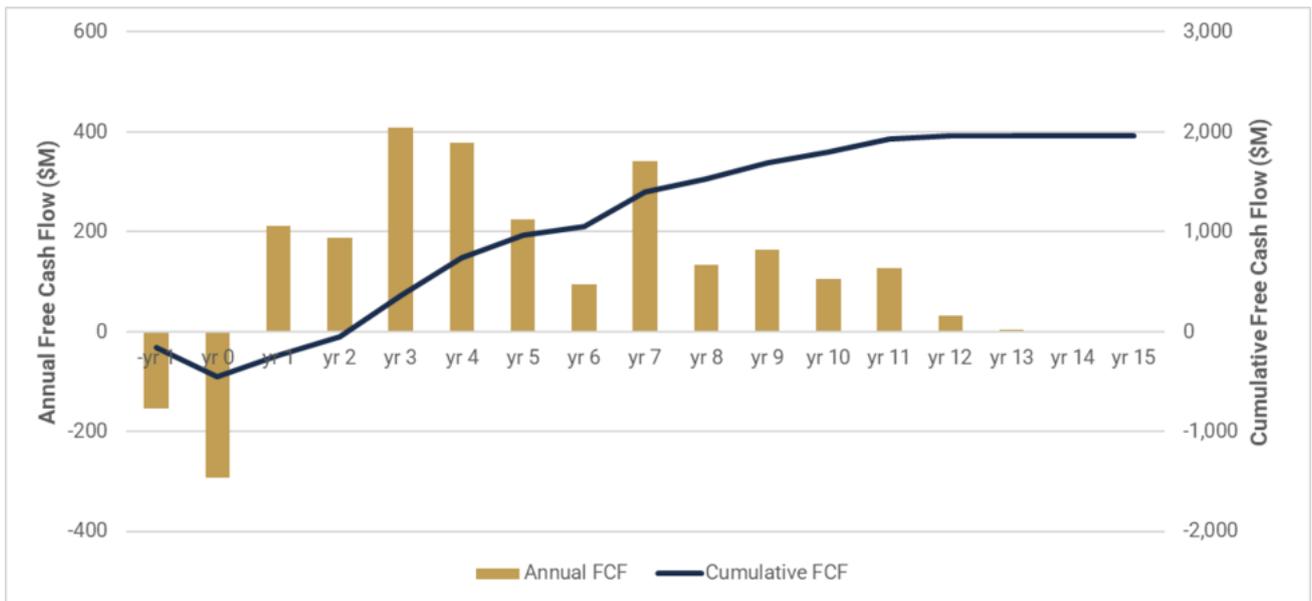


Figure 22-2 Annual Free Cash Flow

22.5.3 Project Parameter Sensitivities

Sensitivity analysis was carried out to determine the effect of changes to input parameters on the base-case financial model. Each sensitivity analysis was performed independent of others.

An overall Projects Economic Sensitivity based on Gold and Silver Price is shown in Table 22-5 below. The objective was to show a range of possible prices for gold and silver to best show the range of economic viability.

Table 22-5 Project Economics Sensitivity - Gold and Silver Price

Parameter	Units	Zero-NPV ¹	- 25%	Base Case	+ 25%	Spot Price ²
Gold Price	\$/oz	1,567	2,100	2,800	3,500	4,000
Silver Price	\$/oz	17	22.5	30	37.5	48
NPV5% Pre-Tax	\$M	150	1,032	2,224	3,416	4,479
NPV5% Post-Tax	\$M	0	657	1,475	2,262	2,963
IRR Pre-Tax	%	13%	42%	69%	90%	107%
IRR Post-Tax	%	4%	27%	47%	63%	77%
LOM Revenue	\$M	4,322	5,457	6,991	8,526	9,896
LOM EBITDA	\$M	902	1,972	3,418	4,864	6,156
FCF-Unlevered (Pre-Tax)	\$M	235	1,336	2,824	4,312	5,642
FCF-Unlevered (Post-Tax)	\$M	90	927	1,958	2,940	3,818
Payback Period (Pre-Tax)	Years	3.2	2.2	1.3	0.9	0.7
Payback Period (Post-Tax)	Years	6.8	2.8	2.1	1.5	1.2
NPV5% : CAPEX Ratio	-	-	1.5	3.3	5.1	6.6

1. Gold and silver prices at which NPV5% - post-tax is equal to \$0M.

2. Assumed Spot Prices as of 2025-10-31

22.5.3.1 Gold and Silver Price Sensitivity Based Parameters

A series of sensitivity analyses were performed at different combinations of gold and silver prices. The ranges are shown in Table 22-6.

Table 22-6 Base Case Prices and Sensitivity Ranges

Metal Prices	Units	Base Case Price	Sensitivity Range
Gold Price	\$/oz	\$2,800	\$2200 - \$3400
Silver Price	\$/oz	\$30	\$24 - \$36
Zinc Price	\$/lb	\$1.20	-
Lead Price	\$/lb	\$1.00	-

The results are shown in Tables 22-7 to 22-10.

Table 22-7 Gold and Silver Price Based IRR - Pre-Tax

		Au Price						
		2200	2400	2600	2800	3000	3200	3400
Ag Price	24	47%	53%	58%	64%	68%	73%	78%
	26	49%	55%	60%	65%	70%	75%	79%
	28	51%	57%	62%	67%	72%	76%	81%
	30	53%	59%	64%	69%	73%	78%	82%
	32	55%	60%	66%	70%	75%	80%	84%
	34	57%	62%	67%	72%	77%	81%	85%
	36	59%	64%	69%	74%	78%	83%	87%

Table 22-8 Gold and Silver Price Based IRR - Post-Tax

		Au Price						
		2200	2400	2600	2800	3000	3200	3400
Ag Price	24	31%	35%	39%	43%	47%	51%	54%
	26	32%	37%	41%	45%	48%	52%	55%
	28	34%	38%	42%	46%	50%	53%	56%
	30	35%	39%	43%	47%	51%	54%	58%
	32	37%	41%	45%	48%	52%	55%	59%
	34	38%	42%	46%	50%	53%	57%	60%
	36	39%	43%	47%	51%	54%	58%	61%

Table 22-9 Gold and Silver Price Based NPV_{5%} Pre-Tax in \$M

		Au Price						
		2200	2400	2600	2800	3000	3200	3400
Ag Price	24	1,220	1,473	1,725	1,978	2,230	2,483	2,735
	26	1,302	1,555	1,807	2,060	2,312	2,565	2,817
	28	1,384	1,637	1,889	2,142	2,394	2,647	2,899
	30	1,466	1,719	1,972	2,224	2,477	2,729	2,982
	32	1,549	1,801	2,054	2,306	2,559	2,811	3,064
	34	1,631	1,883	2,136	2,388	2,641	2,893	3,146
	36	1,713	1,965	2,218	2,470	2,723	2,976	3,228

Table 22-10 Gold and Silver Price Based NPV_{5%} Post-Tax in \$M

		Au Price						
		2200	2400	2600	2800	3000	3200	3400
Ag Price	24	791	963	1,135	1,306	1,478	1,646	1,813
	26	848	1,020	1,192	1,363	1,533	1,700	1,867
	28	905	1,077	1,248	1,420	1,588	1,754	1,921
	30	962	1,134	1,305	1,475	1,642	1,809	1,975
	32	1,019	1,191	1,362	1,529	1,696	1,863	2,029
	34	1,076	1,247	1,417	1,584	1,750	1,917	2,084
	36	1,133	1,304	1,471	1,638	1,805	1,971	2,138

22.5.3.2 CAPEX and OPEX Sensitivity Based Parameters

Sensitivity analyses were also evaluated based on capital expenditures and operating expenses.

The results are shown in Tables 22-11 to 22-14.

Table 22-11 CAPEX and OPEX Based IRR - Pre-Tax

		CAPEX						
		-30%	-20%	-10%	0%	10%	20%	30%
OPEX	-30%	108%	98%	89%	81%	75%	69%	64%
	-20%	104%	93%	84%	77%	71%	65%	61%
	-10%	99%	88%	80%	73%	67%	62%	57%
	0%	94%	84%	76%	69%	63%	58%	53%
	10%	88%	79%	71%	64%	59%	54%	49%
	20%	83%	74%	66%	60%	54%	50%	45%
	30%	77%	68%	61%	55%	50%	45%	41%

Table 22-12 CAPEX and OPEX Based IRR - Post-Tax

		CAPEX						
		-30%	-20%	-10%	0%	10%	20%	30%
OPEX	-30%	79%	70%	63%	57%	52%	48%	44%
	-20%	75%	66%	60%	54%	49%	45%	41%
	-10%	70%	63%	56%	51%	46%	42%	38%
	0%	66%	59%	52%	47%	43%	39%	35%
	10%	62%	54%	49%	44%	39%	35%	32%
	20%	57%	50%	45%	40%	36%	32%	29%
	30%	52%	46%	40%	36%	32%	28%	25%

Table 22-13 CAPEX and OPEX Based NPV_{5%} Pre-Tax in \$M

		CAPEX						
		-30%	-20%	-10%	0%	10%	20%	30%
OPEX	-30%	3,180	3,109	3,038	2,967	2,895	2,824	2,753
	-20%	2,932	2,861	2,790	2,719	2,648	2,577	2,506
	-10%	2,685	2,614	2,543	2,472	2,401	2,329	2,258
	0%	2,437	2,366	2,295	2,224	2,153	2,082	2,011
	10%	2,190	2,119	2,048	1,977	1,906	1,835	1,764
	20%	1,942	1,871	1,800	1,729	1,658	1,587	1,516
	30%	1,695	1,624	1,553	1,482	1,411	1,340	1,269

Table 22-14 CAPEX and OPEX Based NPV_{5%} Post-Tax in \$M

		CAPEX						
		-30%	-20%	-10%	0%	10%	20%	30%
OPEX	-30%	2,177	2,106	2,035	1,964	1,893	1,822	1,751
	-20%	2,014	1,943	1,872	1,801	1,730	1,659	1,588
	-10%	1,851	1,780	1,709	1,638	1,567	1,496	1,425
	0%	1,688	1,617	1,546	1,475	1,404	1,333	1,262
	10%	1,519	1,448	1,377	1,306	1,235	1,164	1,093
	20%	1,347	1,276	1,205	1,134	1,063	992	921
	30%	1,173	1,102	1,031	960	889	818	747

22.6 CONCLUSION

An economic analysis of the mine schedule at the metal base prices shown earlier in Table 22-6 shows the project is financially viable. The project is highly sensitive to the metal price and recovery. However, the project remains economically viable in all sensitivity cases shown in this section.

On a pre-tax basis, the Project has a Net Present Value (NPV) of \$2,224M at a discount rate of 5%, an Internal Rate of Return (IRR) of 69%, and a payback period of 1.3 years.

On an post-tax basis, the NPV is \$1,475M at a discount rate of 5%, the IRR is 47%, and the payback period is 2.1 years.

23.0 ADJACENT PROPERTIES

Several vein occurrences of Ba, Pb, Cu, and Zn that vary in size and thickness are recognized regionally. Some of these veins are subject of artisanal exploitation by individuals or small Moroccan companies.

Regionally, Boumadine is located near several mineral deposits that have been recently exploited, are currently active, or are under development. The largest nearby deposit is the Imiter Silver Mine. Imiter is a major silver deposit that is owned by the Moroccan mining company Managem. The Imiter Mine is located on the north side of the Saghro Massif, ~80 km west of the Boumadine Property (Figure 23-1).

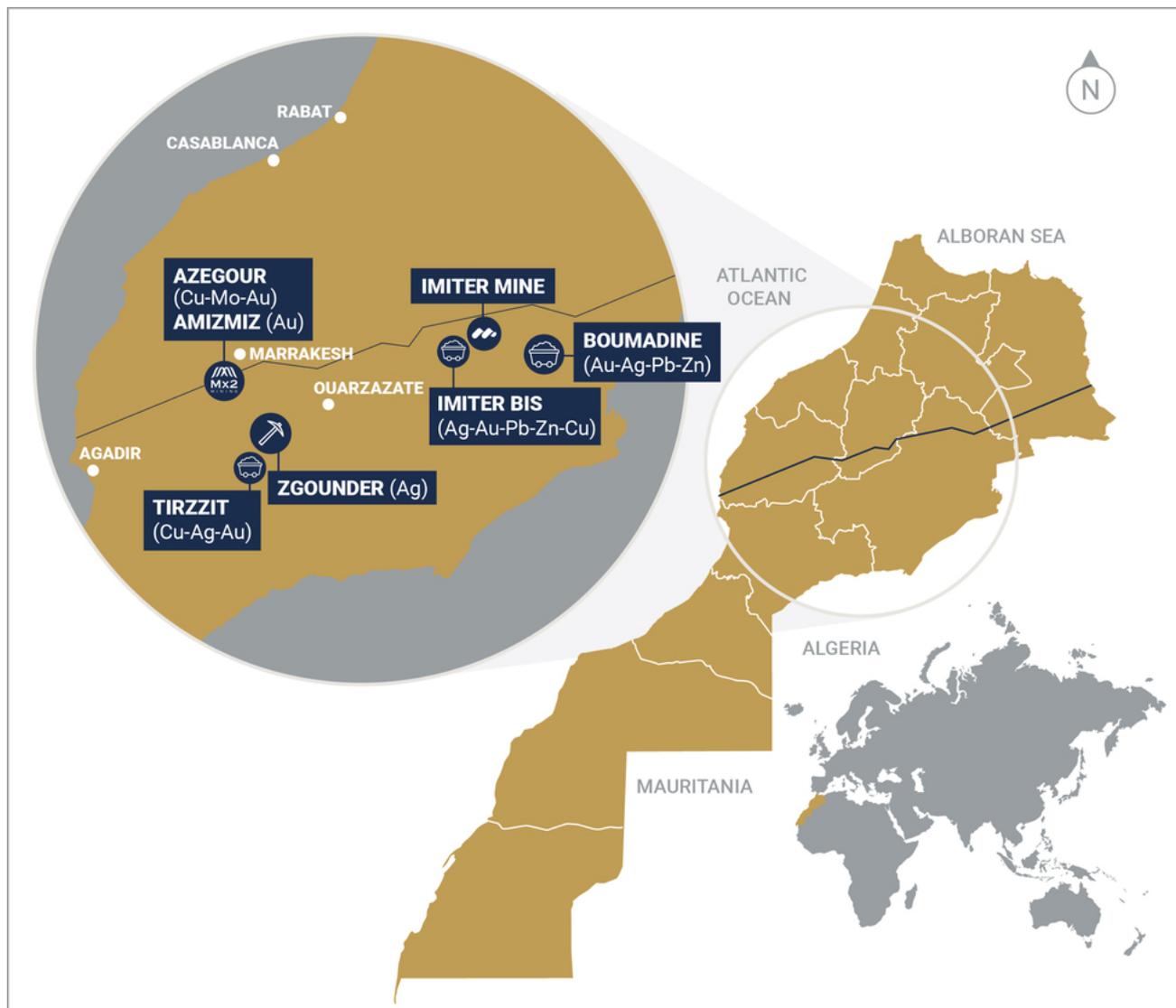


Figure 23-1 Location of the Imiter Mine

Source: Aya corporate presentation dated April 2024

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Execution Schedule

The Project Execution Schedule was developed to provide a high level overview of all activities required to complete the project. The project execution plan is summarized and presented in Figure 24-1.

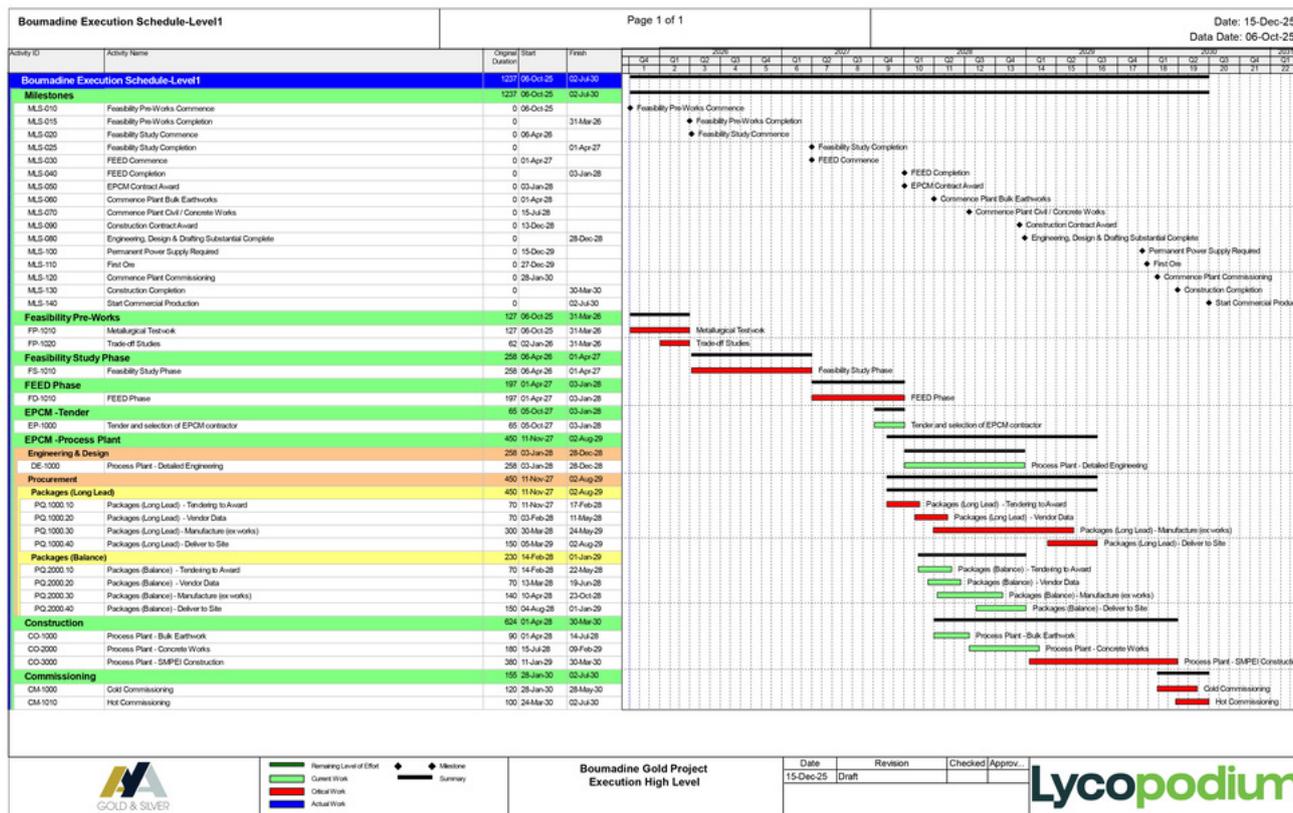


Figure 24-1 Project Execution Schedule

Source: Lycopodium 2025

24.2 Pyrite Treatment

While not included within this report, Aya has continued to work on the metallurgical development for a pyrite treatment circuit. A portion of the gold and silver included in Boumadine is refractory to direct cyanide leaching. The refractoriness to cyanidation is due to the high sulfur content in the pyrite concentrate. Boumadine’s gold is in the form of fine grains (below 15 µm) and mainly associated with pyrite. A pre-treatment is necessary to oxidize the sulfur of pyrite prior to cyanide leaching.

Internal and external trade-off studies have indicated that roasting is the best option for pyrite pre-treatment prior to conventional cyanidation. Roasting would allow for reduced tailings generation and lower operating costs while also producing sulfuric acid, for which there is a high demand within the Kingdom of Morocco.

Testwork is on-going to determine optimal conditions for operation of a pyrite roasting plant. A flowsheet has been developed, and engineering will continue to be progressed in parallel to the mine and concentrator designs. This will provide flexibility for Aya to decide whether it will sell all of the pyrite or if a portion can be strategically treated within the country to produce doré ingots and a sulfuric acid byproduct.

25.0 INTERPRETATION AND CONCLUSIONS

25.1 Geology

The Boumadine Property is located in the Province of Errachidia, in the eastern part of the Kingdom of Morocco. Aya's Property in the Boumadine area consists of 12 mining licenses and 19 exploration permits totalling 341 km² in size. The Boumadine Mining License, which contains the Boumadine Deposit and is the focus of this Boumadine Report, consists of mining license LE-383661 and covers the historical Boumadine Mine, the Boumadine Camp, and the current MRE described in this Report, which total 32 km² in area. The additional thirty exploration permits and mining licenses are distributed within a 25 km radius of the Boumadine Deposit and collectively cover an additional 309 km² in area. In addition, an Exploration authorization of 600 km² was granted to Aya in January 2025.

The historical Boumadine Mine is one of the oldest known mines in the Kingdom of Morocco. It was probably exploited by the Portuguese in the 15th and 16th centuries. Between 1956 and 1998, exploration and mining activities in the Boumadine area were completed by the BRPM.

The Boumadine Property is located at the eastern end of the Anti-Atlas Mountain Range, which extends east-northeast to west-southwest, >600 km, from the Atlantic Ocean in the west to the interior of the African plate in the east. The Boumadine polymetallic deposit (Ag, Au, Cu, Pb, Zn) is located on the northwest side of the Ougnat Massif (or Boutonnière). The geology of the Massif consists of a Neoproterozoic metasedimentary basement overlain unconformably by a Late Neoproterozoic volcano-sedimentary rock sequence and Paleozoic lacustrine sedimentary and minor volcanic rocks. The basement consists of sandstone, pelites and greywackes that are intruded locally by granite, granodiorite, and diorite bodies. The volcano-sedimentary sequence consists of felsic and mafic volcanic rock units separated by volcano-sedimentary units.

The Boumadine Deposit, including Tizi and Imariren Zones has been traced on surface and in drilling for over 6.0km along strike. The Boumadine Deposit mineralized zones consist of 1 to 4 m-wide massive sulphide lenses/veins oriented N20°W and dipping 70° east. The massive sulphide veins (approximately 70% sulphide) consist of pyrite, sphalerite, galena, arsenopyrite and chalcopyrite, with subordinate amounts of cassiterite, silver-rich sulphosalts, stannite, enargite, bismuthinite, native copper and bismuth. The main mineralization zone is surrounded by a 1 to 10 m thick halo of 10 to 30% disseminated pyrite and two types of veinlets: 1) quartz-carbonate-galena-sphalerite veinlets; and 2) massive pyrite veinlets. Geochemically, there is a strong positive correlation of gold with silver and copper and a weaker correlation of zinc with lead and molybdenum. The Boumadine Deposit has been described in literature as being an high-sulphidation epithermal polymetallic deposit in a shallow submarine setting. However, field and drilling evidences suggests a deeper environment of formation.

Exploration activities completed by Aya on the Boumadine Property since 2020, other than drilling, include surface trenching, satellite-based hyperspectral surveys, aerial electromagnetic and magnetic survey, mineral prospecting, geological mapping, grab sampling and assaying. Between May 2022 and September 2025, Aya completed 660 diamond drill holes, 94 multipurpose holes (MP) and 49 Reverse circulation holes (RC), totaling 192,957 m. The drilling programs aimed to extend the mineralization of the North, Central, and South Zones while also testing targets located farther from the main mineralized trend. In addition, all historical drill holes from 2018 to 2021 were re-logged and resampled in 2023 for a total of 77 drill holes and 9,510 m of drill core.

Aya implemented and monitored a thorough QA/QC program for the drilling completed at the Boumadine from 2018 to 2025. Examination of QA/QC results for all recent sampling indicates no material issues with accuracy, contamination or precision in the data. It is the Author's opinion that sample preparation, security and analytical procedures are adequate and that the data are of good quality and satisfactory for use in the current Mineral Resource Estimate. Verification of the Boumadine Deposit data, had also been undertaken by independent Qualified Person in the previous MRE, including a site visit in March 2024, due diligence sampling, verification of drill hole assay data from electronic assay files, and assessment of the available QA/QC data. Independent Authors stated that sufficient verification of the Project data has been undertaken and that the supplied data are of satisfactory quality and suitable for use in the Mineral Resource Estimate.

The updated MRE of the Boumadine Deposit is amenable to conventional open-pit and to underground mining methods. The updated MRE for Boumadine contains an Indicated Mineral Resource of 5.2 Mt grading 91 g/t Ag, 2.78 g/t Au, 2.8% Zn and 0.85% Pb containing an estimated 15.1 Moz of Ag, 462 koz of Au, 145 kt of Zn

and 44 kt of Pb, and an Inferred Mineral Resource of 29.2 Mt grading 82 g/t Ag, 2.63 g/t Au, 2.11% Zn, and 0.82% Pb containing an estimated 76.8 Moz of Ag, 2.5 Moz of Au, 615 kt of Zn and 237 kt of Pb. Pit-constrained Mineral Resources have been estimated within an optimized pit shell for the purpose of reporting Mineral Resources and includes Indicated and Inferred Mineral Resources. The pit-constrained Mineral Resources are reported using a NSR cut-off value of US\$95/t. Out-of-pit Mineral Resources are reported beneath the pit shell that exhibit continuity and reasonable potential for extraction by longhole mining methods. Out-of-pit Mineral Resources are reported using an NSR cut-off of US\$125/t. The Mineral Resource has an effective date of February 24, 2025. The sensitivity of the Mineral Resource to changes in cut-off grade was also calculated across a range of potentially economic NSR cut-offs.

The Mineral Resource Estimates have been classified with respect to CIM Standards as Indicated Mineral Resources and Inferred Mineral Resources, according to the geological confidence and sample spacing that currently define the Deposits, with Indicated Mineral Resources requiring <50 m spaced drill hole centres. All additional estimated grade blocks were classified as Inferred. The Authors are of the opinion that the current Mineral Resource Estimate meets the reasonable prospect of eventual economic extraction. The Authors have experience with other similar projects and are of the opinion that the cut-off grade and cost assumptions are reasonable.

The Authors are not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that may materially affect the Mineral Resource Estimate. A material decrease in metal prices below those utilized for the current Mineral Resource Estimates or a significant increase in operating costs could materially affect the cut-off and average grades, and potentially result in a revised lower Mineral Resource Estimate tonnage.

25.2 Mining

The OP mining area is present at shallow depths and accounts for 63 % of the total production, with an average NSR value of \$193 per tonne. LOM for the OP operation is projected at 9 years. Mining will be carried out by conventional truck and shovel pairing; the bench height varies between 10 and 20 m depending on the pit. Material will be excavated and hauled by a contractor fleet to three separate surface stockpiles, which will supply the processing plant based on operational needs and overall project strategy.

The UG mining area is an extension of the mineralized zones from the OPs and accounts for 37 % of the total production, with an average NSR value of \$190 per tonne. LOM for the UG operation is projected at 10 years. Mining will be carried out using the modified Avoca method, with stope dimensions of 2.0-m minimum width, 20 m in height, and 20 m in length. Material will be hauled by 30-ton trucks, via a ramp system, to three separate surface stockpiles.

Capital and operating lateral and vertical development will be carried out by a contractor, while all subsequent mine production activities will be performed by owner crews.

WSP has identified several risks associated with the UG operations, which are summarized as follows:

- Interaction between OP and UG operations: Since both activities will be conducted simultaneously, delays in OP operations could directly impact the UG schedule.
- Critical pre-production period: The pre-production phase is essential to the overall success of the project. As such, a detailed and well-coordinated schedule for this period is crucial.
- Dependence on rock fill availability for the Avoca method: The effectiveness of the modified Avoca mining method relies on a consistent supply of rock fill. This necessitates a robust system for managing waste material, whether generated UG or transported from the surface.

25.2.1 Mining Geotechnical

Project risks and opportunities identified during mining geotechnical evaluations are as follows:

Risks:

- Some major faults have been delineated in the Boumadine geological model, however these do not transect intended production areas. The spatial distribution of adverse geological features is a large unknown that will have to be resolved to achieve FS level of study.
- The mining depths evaluated in this study are shallower than those in the updated mine plan. Stress magnitudes are likely to be more adverse than those estimated in this study.

Opportunities:

- Reasonable conservatism has been applied to the overall pit slope angle recommendations. There may be opportunity to increase slope angles once additional geotechnical data is available.
- Due to data limitations, this study has applied reasonably conservative global rock strength values to the design evaluations. Laboratory testing efforts will substantially improve confidence in material strength parameterization and help to resolve geological controls and spatial variability. Opportunity to increase stope sizing (or reduce dilution expectations) may be resolved from more detailed material strength data.
- In situ stress assumptions applied to stope design are somewhat conservative. Later phases of study should rely on numerical stress modelling to improve confidence in likely induced stress conditions. Opportunity to increase stope sizing (or reduce dilution expectations) may be resolved from more detailed estimates of in situ stresses.
- Longitudinal stoping provides operational flexibility to adjust stope panel lengths, thus there is an opportunity for optimizing design for local variances (good or bad) in ground conditions. This is particularly important where the deposit body is narrow as production performance in narrow zones is likely to be more sensitive to HW and FW dilution.
- Level spacing in the updated mine plan is 15 m. There is an opportunity to reduce development costs by increasing this spacing (e.g. this study assumed a 20 m level spacing as development plans were not available in July, 2024).

25.3 Processing

Conclusions from the metallurgical and comminution testwork conducted on the Boumadine deposit include:

- The average BWi from two tests was used in the comminution circuit design (13.1 kWh/t), with the remainder of comminution design criteria estimated from OMC's database.
- Locked cycle flotation tests with three circuits (Pb, Zn and Py) achieved acceptable recoveries with grades in the salable range required. The lead and zinc circuits performed best with regrind after rougher flotation, including three cleaning stages and one cleaner-scavenger stage.
- Sulfur content can be estimated based on the metals' content in the material, proven via multilinear regression analysis showing a strong correlation between the laboratory and calculated values ($R^2 = 0.98$). By assuming constant concentrate grades and recoveries, the LOM pyrite concentration production could be estimated based on the sulfur content equation for varying metals' grades in the deposit.
- The Boumadine mineralized material is highly refractory and resistant to direct cyanidation.

Based on the metallurgical testwork completed thus far, the following unit operations have been selected for the PEA flowsheet:

- Single stage primary crushing to reduce the ROM material size to a P_{80} of 136 mm.

- Crushed rock stockpile with a live capacity of approximately 8,115 tonnes, allowing for 24 hours of live storage and additional capacity for periods of primary crusher maintenance.
- SAB type grinding circuit consisting of a SAG mill and a ball mill to produce a ground product P₈₀ of 58 microns (µm), with hydrocyclones for particle size classification.
- Lead rougher flotation, classification, regrinding and cleaner flotation to produce a lead concentrate of 29.6% grade. The lead concentrate will be thickened, filtered, and bagged prior to shipment.
- Zinc rougher flotation, classification, regrinding and cleaner flotation to produce a zinc concentrate of 57.4% grade. The zinc concentrate will be thickened and filtered for stockpiling prior to shipment.
- Pyrite rougher flotation to produce a pyrite concentrate with 4.2 Au g/t and 81.0 Ag g/t. The pyrite concentrate will be thickened and filtered for stockpiling prior to shipment.
- Tailings thickening and storage in a tailings management facility.

The process plant operating cost estimate was developed in Q3 2025 using data from database projects, studies, and previous operations by Lycopodium, and Aya Gold & Silver. The operating cost estimate conforms to preliminary economic assessment accuracy of +50%/-30%. The estimate covers general and administrative (G&A), and processing costs. The unit operating cost per tonne of material milled is estimated at US\$22.44/t processed.

Lycopodium have identified the following risks for the project:

- Additional variability testing is required focussing on specific pit locations to verify material properties used for design parameters and recovery predictions for the deposit body correlates with the established mine plan.
- Thickening and filtration tests should be performed to confirm design assumptions and sizing. The selected filtration technology should also conform to required transport moisture limits yet to be determined.
- Filtration of ultrafine material can prove to be challenging possibly requiring the use of drying pads to further reduce product moisture content prior to shipment. The additional material handling is not currently costed and has the potential to impact final product quality if not performed well.
-

25.4 Infrastructure

The Project is supported by a fully integrated site layout including open-pit mines, underground access, processing facilities, and support services.

Site access via the N10 highway through Tinejdad; internal haulage and service roads link pits, portals, ROM pad, process plant, and stockpiles.

Process plant facilities include ROM pad, crushing and feed preparation, flotation plant, concentrate handling, reagent storage, laboratories, workshops, fuel storage, and administration buildings.

Water supply relies on pipelines from nearby towns, boreholes, runoff capture, and an on-site water harvesting reservoir; containerized treatment systems ensure process and potable water quality.

Mining facilities include a 1,000 m² truck shop, low-grade stockpile (3.5 Mt), ROM stockpiles, and two waste rock stockpiles (215 Mm³ total) designed with HDPE liners and conservative geotechnical assumptions.

Tailings management uses a single lined Flotation TSF with downstream raises, internal drainage, storm diversion channels, an emergency basin, and a return water system.

Power supply delivered through a new 220 kV line feeding a 40 MVA substation serving all site loads.

Additional infrastructure includes sewage treatment, security fencing, communications, explosives storage, and limited on-site accommodation supplemented by housing in Tinejdad.

Concentrate transport will be contractor-operated, shipping by road to Nador-West Port with a covered 250,000-t storage facility.

The Authors have identified the following risks associated with the project infrastructure:

- The supply of water will rely on off-site sources as the area receives little rainfall annually. It will be important to include redundancy and storage capacity to reduce risks associated with water supply.
- The transportation of pyrite will require well-coordinated logistics due to the large quantities of material to transport annually.

25.5 Environmental and Social

The Project is located in the Drâa-Tafilalet region of western Morocco, approximately 19 km south of the nearest town of Tinejdad. The biophysical and social characteristics of the Project area are defined by its semi-arid climate and relatively rural and remote nature. Urban settlements are concentrated along main transport routes with villages located along valleys with access to water resources. Climatic conditions influence local ecosystems and land uses including agriculture, concentrated in areas of water availability.

Limited environmental studies have been completed in the past. As part of the environmental and social work currently being undertaken, an environmental baseline sampling program was initiated by BGM in September 2024 and a high-level screening for critical biodiversity habitat was undertaken by SLR in September 2024.

The baseline sampling, covering soils, air quality and water, was undertaken by AfriLabs, a Moroccan-based laboratory specializing in mineral and environmental analysis.

Additional baseline sampling is planned together with subsequent E&S studies as part of the international ESIA process outlined below. These studies will cover the topics of biodiversity, hydrology, hydrogeology, soils and sediments, air quality, noise, GHG, climate change, stakeholder engagement, labour and working conditions, community health and safety, and other social elements.

The Project will require national permits to support the development of the mine and the requirement to carry out an Environmental Impact Assessment under Moroccan Law. In addition to meeting national permitting requirements, the Project is being undertaken to meet international ESIA lender standards. At the time of writing, the Project is yet to initiate the national EIA permitting process. The plan is to initiate this process in 2026, at the same time as undertaking an ESIA to international lender standards.

A conceptual mine closure plan governed by Aya's HSEC Policy will be prepared as part of the ESIA.

25.6 Capital Cost Estimate

The preliminary economics of the Boumadine Project can be assessed using the capital and operational cost estimates detailed in this PEA. The calculations are based on an open pit and underground mining operation concept, the development of a processing plant, infrastructure, and the Owner's expenses and provisions.

The capital cost estimate conforms to Class 5 guidelines for a preliminary economic assessment level estimate with a +50%/-30% accuracy according to the Association for the Advancement of Cost Engineering International (AACE International). The capital cost estimate was developed in Q4 2025 based on Lycopodium's in-house database of projects and studies, experience from similar operations and inputs from WSP and Epoch Resources.

The total initial capital cost for the Boumadine Project is \$446 M and the life-of-mine sustaining cost is \$340 M.

25.7 Operating Cost Estimate

The operating cost estimate was developed in Q4 2025 using data from projects, studies, and previous operations from Lycopodium’s internal database. The operating cost estimate conforms to preliminary economic assessment accuracy of +50%/-30%. The estimate covers the general and administrative (G&A), and mining and processing. The unit operating cost per tonne of material milled is \$66.16/t.

25.8 Economic Analysis

The economic analysis was performed assuming a 5% discount rate typical for silver-gold projects. Cash flows have been discounted to the start of construction, assuming that the project execution decision will be made, and major project financing will be carried out at this time.

The pre-tax NPV discounted at 5% is \$2.2B, with a pre-tax IRR of 69% and payback period of 1.3 years. On an after-tax basis, the NPV discounted at 5% is \$1.5B, with an after-tax IRR of 47%, and payback period of 2.1 years. Cumulative after-tax unlevered free cash flow totals \$2.8B. Tax calculations are based on Aya’s understanding of current Moroccan tax regulations as of the effective date of this report.

Readers are cautioned that the PEA is preliminary in nature. It includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves and there is no certainty that the PEA will be realized. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

A summary of the project economics is listed in Table 25-1 , and after-tax free cash flow is shown graphically in Figure 25-1.

Table 25-1 Key Economic Results

Project Economics	Units	Base Case	
		Pre-tax	Post-tax
NPV _{5%}	\$B	2.2	1.5
IRR	%	69%	47%
Payback	Years	1.3	2.1
NPV: Capex	-	5.0	3.3
Revenue LOM	\$B	7.0	-
Avg. Annual Revenue	\$M/y	629	-
EBITDA LOM	\$B	3.4	-
Avg. Annual EBITDA	\$M/y	308	-
Cumulative FCF LOM	\$B	2.8	2.0
Avg. Annual FCF	\$M/y	254	176

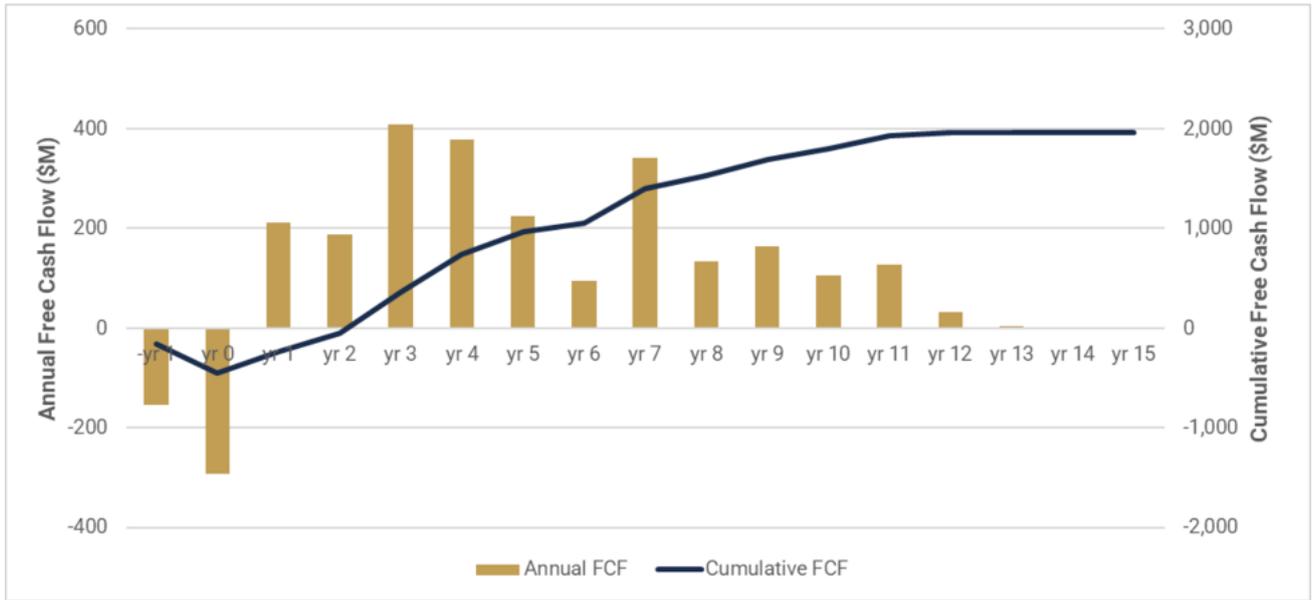


Figure 25-1 Boumadine Project - Annual Free Cash Flow Profile

25.9 Conclusion

The Mineral Resources currently estimated for the Boumadine Polymetallic Project consist of:

- An Indicated Mineral Resource of 5.2 Mt grading 91 g/t Ag, 2.78 g/t Au, 2.8% Zn and 0.85% Pb containing an estimated 15.1 Moz of Ag, 462 koz of Au, 145 kt of Zn and 44 kt of Pb;
- An Inferred Mineral Resource of 29.2 Mt grading 82 g/t Ag, 2.63 g/t Au, 2.11% Zn and 0.82% Pb containing an estimated 76.8 Moz of Ag, 2.5 Moz of Au, 615 kt of Zn and 237 kt of Pb;

The PEA provides a base case assessment for developing the Boumadine mineral resource by conventional open pit and underground mining methods, and a crushing-milling-flotation processing plant with a tailings dam.

The PEA economic analysis shows the Boumadine Polymetallic Project has an after-tax NPV5% of \$1.5B, IRR of 47%, and a payback period of 2.1 years. The PEA economics support a decision to continue to advance the Project and carry out additional detailed studies, including a pre-feasibility study.

26.0 RECOMMENDATIONS

26.1 Summary

The results presented in this technical report demonstrate that the Boumadine Project is technically and economically viable. It is recommended that the Project advance from the current PEA stage into the subsequent study phases, including Pre-Feasibility work toward a full Feasibility Study. Table 26-1 summarizes the proposed budget to advance the Project through the Feasibility stage.

Table 26-1 Budget for Advancing Project to Next Stages

Area	\$M, USD
MRE Reporting	0.3
Metallurgical Testwork	1.4
Processing Plant	1.5
Mining Consultant	0.5
Tailings Storage Facility	0.4
Geotechnical	0.9
Infrastructure	0.2
Hydrogeology	0.9
Environmental Studies, Permitting and Social Considerations	0.7
Contingency	0.7
Total	7.3

26.2 Geology

Aya owns or controls 31 mining licenses and exploration permits in the Boumadine Property area (341 km²) in the eastern part of the Kingdom of Morocco. At Boumadine, silver–gold polymetallic sulphide mineralization is hosted within a vein system striking approximately 6 km.

Additional expenditures are recommended by the Authors for the following activities:

- Drilling to advance Inferred to Indicated Mineral Resources;
- Drilling down-dip in order to develop additional Mineral Resources at depth;
- Follow-up geological mapping, mineral prospecting, and assays;
- Development of a comprehensive bulk density model;
- Review grade anisotropy by individual mineralized domain; and
- Advance the project to its next development phase

The Authors also recommend that Aya continue with the current QC protocol and monitor QC data on an ongoing basis, and continue refining the metallurgical work for improved recoveries.

The estimated cost of the recommended work program is US\$101.6M, which includes 10% contingency (without applicable taxes) (Table 26-2). Phase 1 of the recommended work program should be completed in 2026 and Phase 2 in 2027. Phase 2 is contingent on a successful Phase 1.

Table 26-2 Recommended programs and budgets for 2026-2027

Year	Item	Activity	Unit (m)	Cost Estimate (\$M, USD)
Phase 1 - 2026				
2026		Drilling (all-in costs)	200,000	42.0
		Administration and Management		4.2
		Sub-Total		46.2
		Contingency (10%)		4.6
		Total - 2026		50.8
Phase 2 - 2027				
2027		Drilling (all-in costs)	200,000	42.0
		Administration and Management		4.2
		Sub-Total		46.2
		Contingency (10%)		4.6
		Total - 2027		50.8

26.3 Mining

The PEA study has demonstrated the positive potential of the Boumadine Project. Based on these results, WSP recommends that Aya undertake the following additional studies in preparation for a Pre-Feasibility Study:

- Conduct further geotechnical investigations to validate the data used for determining wall angles, stope design parameters and ground support requirements.
- Conduct hydrogeological investigations to determine ground water inflows in the pits and the UG workings.
- Perform a trade-off study comparing the planned longitudinal Avoca mining method with alternative methods such as shrinkage stoping and Alimak mining, to identify the most suitable approach for the project.
- Refine estimates of unplanned dilution and mining recovery factors based on the selected mining method, site-specific geotechnical conditions, and backfill properties.
- Consider implementing dedicated escapeway raises instead of installing ladderways in ventilation raises. Ladderways within ventilation raises reduce airflow efficiency due to increased friction losses and can be accommodated in smaller diameter raisebores than those required for ventilation.
- Incorporate truck bypasses into the ramp design and conduct a haulage simulation to assess the ramp traffic and confirm the haulage cycle and the optimal number of trucks required.
- Evaluate the potential for automated and tele-remote operation of mining equipment. These technologies could enhance equipment utilization during shift changes and improve productivity following blasting and smoke clearance.

26.4 Processing

The following testwork is recommended to progress the design to the next phase:

Sample Requirement:

- Spatial composites to represent different physical locations throughout the deposit

- Samples from different zones and depths for variability testing.
- Sufficient sample sizes to conduct downstream testwork.
- Number of representative samples to be determined by AGS.
- Sufficient sample size to generate representative concentrate samples for off-taker evaluation.

Comminution Testwork:

- Additional tests to determine:
 - Crushing work index (CWi)
 - SMC parameters: abrasivity (Ai) and impact breakage (A x b)
 - BWi (confirmatory)
- Variability of these parameters throughout the deposit and determine correlations to lithology.
- Flotation regrind tests to determine specific energy; engage with vendors for criteria relating to equipment sizing and specifications.

Ore Characterization Testwork:

- Assess material variability by assaying samples from different zones without blending according to mine plan and to create design ranges.
- Particle size distribution of ROM material and of final concentrates in support of dewatering testwork.
- Rheology testwork to determine apparent viscosities at varying shear rates and varying slurry % solids in support of slurry pipe and pump design.
- Physical characteristics impacting material handling, such as feed moisture content, specific gravity, bulk density, and stacking and clumping / compaction properties.

Flotation Testwork:

- Variability testing – open circuit and locked cycle to determine the impacts of feed variance and process upsets on grades and recoveries. Develop relations between open circuit and locked cycle results
- Produce enough of each concentrate to support dewatering testwork and further pyrite concentrate treatment testwork.

Dewatering Testwork:

- Particle settling testwork to determine the flux rate for thickener sizing.
- Flocculant screening tests.
- Filtration testwork to determine:
 - Filter type
 - Optimum feed pulp density
 - Achievable cake moisture levels
 - Transport moisture limit targets
 - Filter cycle times
 - If any filter aids are required.

Pyrite Concentrate Treatment Testwork:

- Continue development of pyrite oxidation option.
- Assess variability of pyrite concentrate via spatial composites and variability testing.

Water Analysis:

- Surface water quality analysis.
- Ground water quality analysis.

26.5 Tailings Storage Facility

The following recommendations are applicable to the next stage of the study:

- As per Requirement 5.1, a Multiple Accounts Analysis (MAA) should be conducted to assess alternative tailings technologies in the design of the TSF. The selection of an alternative technology to that described in the PEA may alter the cost estimates for the TSF, as well as for that of the tailings processing and TSF conveyance system.
- The appropriate site location selected for the TSF should be confirmed during the MAA, incorporating the input from experts in the relevant fields, including environmental, social, process, etc, to assess the consequences of hazards posed by the TSF. The selection of an alternative site location to that described in the PEA may alter the cost estimates for the TSF, as well as that of the tailings conveyance system.
- The following design aspects of TSF, documented by GCIM, may be reviewed in the next stage of the study and may impact the TSF CAPEX estimate:
 - Investigate the suitability of a single 2mm HDPE liner, incorporating the consideration of the permeability of the material upon which it is installed, to mitigate seepage in the case of a liner leak,
 - Investigate the suitability of the inclusion of the geotextile beneath the liner. This entails considering both the use as a liner protection measure as well as the resulting seepage in the case of a liner leak,
 - Significant floods are generated by the catchments upstream of the TSF. As stated by GCIM, the design of the storm diversion channels is required, adequately sized to cater for the design floods,
 - Sizing of the return water and emergency basins based on a monthly TSF pool water balance.

26.6 Environmental and Social

As part of the ESIA, potential impacts associated with the Project will be identified and assessed in a systematic approach. The impact assessment will inform a comprehensive Environmental and Social Management and Monitoring Plan to be developed for the Project. It will include:

- Mitigation measures for construction and operations
- Framework management plans covering biodiversity, erosion and sediments, waste, air and noise, water, community health and safety, community development and investment, supply chain, cultural heritage, rehabilitation
- Stakeholder engagement plan including grievance management
- Monitoring plans for Project performance and the receiving environment (water, air, noise and biodiversity)

Water management planning will include the following elements:

- Characterization of the groundwater systems on site
- Environmental testwork to understand the geochemical composition and acid generating potential of the host rock and mineral residues

- Development of a site-wide climatic water balance to understand the water circuit and confirm water use, storage and discharge volume requirements.
- Establishment of an engineered stormwater runoff system catering for the diversion of clean water runoff around Project sites, separation of clean and potentially dirty water runoff on site, storage of potentially dirty water runoff in appropriately designed and engineered facilities, and collection and re-use of potentially dirty water runoff.
- Flood protection analysis and measures, if needed.
- Prioritizing water reuse and recycling.
- Water treatment for potable water supply, raw water supply, and discharge to the environment (if discharge is required).

26.7 Project Next Phases

To progress the project, it is recommended to conduct pre-feasibility level trade offs, design, and engineering to improve the accuracy of cost estimation. This shall include optimizing the mine plan, updating the process design using results from additional metallurgical test data, and conduct engineering of other disciplines such as mechanical, civil, structural and electrical engineering producing MTOs. The information from this phase of project will then feed into a Feasibility Study.

The advancement to a Feasibility Study will be dependent on project economics from pre-feasibility level designs and progress of drilling to advance classification of resources.

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28.0 CERTIFICATES

CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled “*Technical Report - Preliminary Economic Assessment for the Boumadine Polymetallic Project, Kingdom of Morocco*” which is effective as of November 4th, 2025 and issued on December 18th, 2025 (the “*Technical Report*”) prepared for AYA Gold and Silver (the “*Company*”).

I, *Cortney Palleske, P. Eng.*, do hereby certify:

1. I am currently employed as a Principal Geomechanics Consultant by RockEng Inc. (RockEng), located at 920 Princess Street, Suite 310, Kingston ON, Canada K7L 1H1.
2. I graduated from Queen’s University with a Bachelor of Applied Science in 2005 and a Master of Applied Science in 2014.
3. I am a registered member of EGBC (Engineers and Geoscientists of British Columbia, membership #49303), OIQ (Ordre des Ingénieurs du Québec, membership #6076971), and PEO (Professional Engineers Ontario, membership #100548665).
4. I have worked as an Engineer in Geomechanics since my graduation from university in 2005. I have 20 years of experience in geomechanical engineering consulting and have worked on open pit and underground mining projects at all stages from scoping to operations and closure planning.
5. I have read the definition of “qualified person” set out in the National Instrument 43-101 and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be an independent qualified person for the purposes of NI 43-101;
6. I am independent of the issuer in accordance with the application of section 1.5 of National Instrument 43-101.
7. I have not had any prior involvement with the property that is the subject of the *Technical Report*.
8. I am responsible for Sections 16.2 and parts of Sections 25, 27.
9. I have visited the Boumadine project site from June 5 to 7, 2024.
10. I have no personal knowledge as of the date of this certificate of any material fact or change, which is not reflected in this *Technical Report*.
11. I have read NI 43-101 and Form 43-101F1 and have prepared the parts of the *Technical Report* for which I am responsible in compliance with NI 43-101 and Form 43-101F1; and as of the date of the certificate, to the best of my knowledge, information and belief, the *Technical Report* contains all scientific and technical information that is required to be disclosed to make the *Technical Report* not misleading.

Dated this 18th day of December 2025

“*Original Signed and Sealed*”

Cortney Palleske, P.Eng.
RockEng Inc.

CERTIFICATE OF QUALIFIED PERSON

JARITA BARRY, P.GEO.

I, Jarita Barry, P.Geo., residing at 9052 Mortlake-Ararat Road, Ararat, Victoria, Australia, 3377, do hereby certify that:

1. I am an independent geological consultant contracted by P&E Mining Consultants Inc.
2. This certificate applies to the Technical Report titled "Preliminary Economic Assessment for the Boumadine Polymetallic Project, Kingdom of Morocco", (The "Technical Report") with an effective date of November 4, 2025.
3. I am a graduate of RMIT University of Melbourne, Victoria, Australia, with a B.Sc. in Applied Geology. I have worked as a geologist for over 17 years since obtaining my B.Sc. degree. I am a geological consultant currently licensed by Engineers and Geoscientists British Columbia (License No. 40875) and Professional Engineers and Geoscientists Newfoundland & Labrador (License No. 08399). I am also a member of the Australasian Institute of Mining and Metallurgy of Australia (Member No. 305397);

I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

My relevant experience for the purpose of the Technical Report is:

- 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. (EGBC/AusIMM) 2014-Present

4. I have not visited the Property that is the subject of this Technical Report.
5. I am responsible for Authoring Section 11, and Co-authoring Sections 1, 12, 25, 26 and 27 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have been previously involved with the Boumadine Polymetallic Project as a "Qualified Person" for a Technical Report titled "Technical Report and Updated Mineral Resource Estimate of the Boumadine Polymetallic Project, Kingdom of Morocco", by P&E Mining Consultants Inc. dated May 31, 2024.
8. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: November 4, 2025

Signed Date: December 18, 2025

{SIGNED AND SEALED}

[Jarita Barry]

Jarita Barry, P.Geo.

CERTIFICATE OF QUALIFIED PERSON

ANTOINE YASSA, P.GEO.

I, Antoine Yassa, P.Geo. residing at 3602 Rang des Cavaliers, Rouyn-Noranda, Quebec, J0Z 1Y2, do hereby certify that:

1. I am an independent geological consultant contracted by P&E Mining Consultants Inc.
2. This certificate applies to the Technical Report titled "Preliminary Economic Assessment for the Boumadine Polymetallic Project, Kingdom of Morocco", (The "Technical Report") with an effective date of November 4, 2025.
3. 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. I am a geological consultant currently licensed by the Order of Geologists of Québec (License No 224) and by Professional Geoscientists Ontario (License No 1890);

I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

My relevant experience for the purpose of the Technical Report is:

- 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 and Resources Evaluation (Kiena Mine) 2001-2003
 - Database Manager and Resources Evaluation at Julietta Mine, Bema Gold Corp. 2003-2006
 - Consulting Geologist 2006-present
4. I have visited the Property that is the subject of this Technical Report on March 12 to 14, 2024.
 5. I am responsible for Authoring Section 10, and Co-authoring Sections 1, 12, 25, 26 and 27 of this Technical Report.
 6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101. I am independent of the Vendor and the Property.
 7. I have been previously involved with the Boumadine Polymetallic Project as a "Qualified Person" for a Technical Report titled "Technical Report and Updated Mineral Resource Estimate of the Boumadine Polymetallic Project, Kingdom of Morocco", by P&E Mining Consultants Inc. dated May 31, 2024.
 8. I have read NI 43-101 and Form 43-101F1. This Technical Report has been prepared in compliance therewith.
 9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: November 4, 2025

Signed Date: December 18, 2025

{SIGNED AND SEALED}

[Antoine Yassa]

Antoine Yassa, P.Geo.

CERTIFICATE OF QUALIFIED PERSON

WILLIAM STONE, PH.D., P.GEO.

I, William Stone, Ph.D., P.Geo, residing at 4361 Latimer Crescent, Burlington, Ontario, do hereby certify that:

1. I am an independent geological consultant working for P&E Mining Consultants Inc.
2. This certificate applies to the Technical Report titled "Preliminary Economic Assessment for the Boumadine Polymetallic Project, Kingdom of Morocco", (The "Technical Report") with an effective date of November 4, 2025.
3. I am a graduate 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 35 years since obtaining my M.Sc. degree. I am a geological consultant currently licensed by the Professional Geoscientists Ontario (License No 1569).

I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

My relevant experience for the purpose of the Technical Report is:

• 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

4. I have not visited the Property that is the subject of this Technical Report.
5. I am responsible for authoring Sections 3.1, 4 to 9 and 23 and Co-authoring Sections 1, 25, 26 and 27 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have been previously involved with the Boumadine Polymetallic Project as a "Qualified Person" for a Technical Report titled "Technical Report and Updated Mineral Resource Estimate of the Boumadine Polymetallic Project, Kingdom of Morocco", by P&E Mining Consultants Inc. dated May 31, 2024.
8. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: November 4, 2025

Signed Date: December 18, 2025

{SIGNED AND SEALED}

[William Stone]

William Stone, Ph.D., P.Geo.

CERTIFICATE OF QUALIFIED PERSON

EUGENE PURITCH, P. ENG., FEC, CET

I, Eugene Puritch, P. Eng., FEC, CET, residing at 44 Turtlecreek Blvd., Brampton, Ontario, L6W 3X7, do hereby certify that:

1. I am an independent mining consultant and President of P&E Mining Consultants Inc.
2. This certificate applies to the Technical Report titled "Preliminary Economic Assessment for the Boumadine Polymetallic Project, Kingdom of Morocco", (The "Technical Report") with an effective date of November 4, 2025.
3. 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. 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.

I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

I have practiced my profession continuously since 1978. My summarized career experience is as follows:

- 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
4. I have not visited the Property that is the subject of this Technical Report.
 5. I am responsible for Co-authoring Sections 1, 14, 25, 26 and 27 of this Technical Report.
 6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
 7. I have been previously involved with the Boumadine Polymetallic Project as a "Qualified Person" for a Technical Report titled "Technical Report and Updated Mineral Resource Estimate of the Boumadine Polymetallic Project, Kingdom of Morocco", by P&E Mining Consultants Inc. dated May 31, 2024.
 8. I have read NI 43-101 and Form 43-101F1. This Technical Report has been prepared in compliance therewith.
 9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: November 4, 2025

Signed Date: December 18, 2025

{SIGNED AND SEALED}

[Eugene Puritch]

Eugene Puritch, P.Eng., FEC, CET

CERTIFICATE OF QUALIFIED PERSON

FRED BROWN, P.GEO.

I, Fred H. Brown, of PO Box 332, Lynden, WA, USA, do hereby certify that:

1. I am an independent geological consultant and have worked as a geologist continuously since my graduation from university in 1987, specialising in gold, silver, base metals, PGEs, diamonds, industrial minerals and other commodities.
2. This certificate applies to the Technical Report titled "Preliminary Economic Assessment for the Boumadine Polymetallic Project, Kingdom of Morocco", (The "Technical Report") with an effective date of November 4, 2025.
3. I graduated with a Bachelor of Science degree in Geology from New Mexico State University in 1987. I obtained a Diploma in Datametrics in 1993 from the University of South Africa, 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. In 2015 I obtained a Citation in Applied Geostatistics from the University of Alberta, and a Geographic Information Systems Certificate from the University of California San Diego in 2016. I am registered with 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).

I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

My relevant experience for the purpose of the Technical Report is:

- Mineral Resource Manager, AngloGold Corporation, South Africa 1988-1997
- Chief Geologist, De Beers Consolidated Mines, South Africa 1997-2004
- Consulting Geologist 2004-2015 & 2016-Present
- Senior Geostatistician, Twin Creeks & Phoenix Mines, Nevada 2015-2016
- P&E Mining Consultants Inc. – Sr. Associate Geologist 2008-Present

4. I have not visited the Property that is the subject of this Technical Report.
5. I am responsible for Co-authoring Sections 1, 14, 25, 26 and 27 of this Technical Report.
6. I am independent of the Issuer applying the test in Section 1.5 of NI 43-101.
7. I have been previously involved with the Boumadine Polymetallic Project as a "Qualified Person" for a Technical Report titled "Technical Report and Updated Mineral Resource Estimate of the Boumadine Polymetallic Project, Kingdom of Morocco", by P&E Mining Consultants Inc. dated May 31, 2024.
8. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance therewith.
9. As of the effective date of this Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: November 4, 2025

Signed Date: December 18, 2025

{SIGNED AND SEALED}

[Fred Brown]

Fred Brown, P.Geo.

CERTIFICATE OF AUTHOR

To accompany the Report entitled "Technical Report - Preliminary Economic Assessment for the Boumadine Polymetallic Project, Kingdom of Morocco" which is effective as of November 4, 2025, and issued on December 18, 2025 (the "Technical Report") prepared for AYA Gold and Silver (the "Company").

I, Alessandra (Alex) Pheiffer, PrSciNat, do hereby certify that:

- 1 I am Technical Director: ESIA with SLR Consulting France SAS, of 2 Square Roger Genin, 38000, Grenoble.
- 2 I am a graduate from Rand Afrikaans University, in South Africa, in 2004 with a M,Sc Environmental Management.
- 3 I am registered as a Professional Natural Scientist (PrSciNat) in Environmental Science (Reg. No. 400183/05) with the South African Council for Natural Scientific Professions.
- 4 I have worked as an environmental scientist for a total of 23 years. My relevant experience for the purpose of the Technical Report is:
 - a) Review and reporting as a consultant on many mining operations and projects for feasibility, due diligence and regulatory requirements.
 - b) Operational experience for a period of one year as an Assistant to the Chief Environmental Officer within Anglo Platinum's Waterval Smelter in South Africa.
 - c) Experience as an Environmental Scientist in coal, gold, silver, lead, zinc, copper, nickel, PGMs, iron, manganese, uranium, chrome, ferrochrome, bauxite.
- 5 I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be an independent qualified person for the purposes of NI 43-101.
- 6 I am independent of the Issuer applying the test set out in Section 1.5 of National Instrument 43-101.
- 7 I am responsible for Section 20 and co-authored Sections 1, 25, 26, and 27 of the Technical Report.
- 8 I visited the Boumadine project site between September 1-3, 2024.
- 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 have prepared the Technical Report in compliance with NI 43-101 and Form 43-101F1. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, Section 20 and parts of Sections 1,

25, 26, and 27 contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated the 18th day of December, 2025

"Original Signed & Sealed"

*Alessandra (Alex) Pheiffer, PrSciNat
ESIA Consultant*

CERTIFICATE OF QUALIFIED PERSON

To accompany the Report titled "Technical Report - Preliminary Economic Assessment for the Boumadine Polymetallic Project, Kingdom of Morocco" which is effective as of November 4, 2025 and issue on December 18, 2025 (the "Technical Report") prepared for AYA Gold & Silver (the "Company").

I, *Dr George Papageorgiou, PrEng*, do hereby certify:

1. I am a Tailings Engineer with Epoch Resources (Pty) Ltd, with an office at Building Block B, Viscount Rd Office Park, 8 Viscount Rd, Bedfordview, Johannesburg, South Africa.
2. I graduated with a BSc Eng (Civil) (1991), a MSc Eng (Civil) (1994) and PhD Eng (Civil) (2004) from the University of the Witwatersrand, Johannesburg, South Africa.
3. I am a Registered Professional Engineer (Pr.Eng) with the Engineering Council of South Africa (ECSA), Registration Number 202503451
4. I have worked as an engineer in the fields of mining waste management and mine closure for a total of 34 years since my graduation in 1991. This includes twenty years as a Founding Partner and Director of Epoch Resources (Pty) Ltd.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined by NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I have not visited the site of the Project.
7. I have participated in the preparation of this Technical Report and am responsible for Section 18.5, 21.1.4, 21.1.5, 21.2.5, and contributed part of Sections 1, 25, 26 and 27.
8. I am independent of the issuer as described in section 1.5 of NI 43-101.
9. I have had no prior involvement with the Project
10. I have no personal knowledge as of the date of this certificate of any material fact or change, which is not reflected in this Report.
11. Neither I, nor any affiliated entity of mine, have earned the majority of our income during the preceding three (3) years from the Company, or any associated or affiliated companies.
12. I have read NI 43-101 and have prepared the Technical Report in compliance with NI 43-101 and; and have prepared the report in conformity with generally accepted mining industry practice, and as of the date of the certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

<p>Physical Address</p> <p>Postal Address</p> <p>Telephone</p> <p>Facsimile</p> <p>Web Address</p> <p>Company Registration</p> <p>Directors</p> <p>Associate Consultant</p>	<p>Ground Floor, Block A, Viscount Road Office Park 8 Viscount Road, Bedfordview, 2008</p> <p>PO Box 751306, Gardenview, 2047</p> <p>+27 (11) 656 0380/1</p> <p>+27 (11) 502 3857</p> <p>www.epochresources.co.za</p> <p>Epoch Resources (Pty) Ltd, No 2005/007908/07</p> <p>GJ Wiid, G Papageorgiou, AC Savvas</p> <p style="text-align: right;">Prof G Heymann</p>
---	--

Dated this 18th day of December 2025

"Signed and sealed"
Dr George Papageorgiou, PrEng

Lycopodium

CERTIFICATE OF QUALIFIED PERSON

I, Preetham Nayak, P. Eng., as an author of this report entitled "Technical Report - Preliminary Economic Assessment for the Boumadine Polymetallic Project, Kingdom of Morocco" ("Technical Report"), prepared for Aya Gold & Silver Inc. with an effective date of November 4, 2025, do hereby certify that:

- 1) I am a Senior Study Manager with Lycopodium Minerals Canada Ltd., with a business address at 5090 Explorer Dr, Suite 700, Mississauga, ON, L4W 4T9.
- 2) I am a graduate of the University of British Columbia with a Master of Applied Science degree, honours Mining Engineering, 2015 and a graduate of the National Institute of Technology Karnataka, India with a Bachelors of Technology degree in Mining Engineering, 2010.
- 3) I am a member of Engineers and Geoscientists British Columbia and registered as a Professional Engineer in the province of British Columbia (License Number 47553). I have practiced my profession in the mining and metals industry continuously since graduation.
- 4) My relevant experience for the purpose of the Technical Report includes:
 - Over 9 years of experience with design and management of mining projects
 - Review and report as a consultant on numerous process facilities and mining projects around the world for due diligence and regulatory requirements
 - Study manager for several preliminary economic assessment studies in Africa, Australia and Asia.
 - Development, execution and interpretation of capital cost estimate and cashflow modelling in precious metals and base metals projects
- 5) I have read the definition of 'qualified person' set out in National Instrument 43-101 ('NI 43-101') and certify that by virtue of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a 'qualified person' for the purposes of NI 43-101.
- 6) I have not visited the Boumadine Project site.
- 7) I am responsible for Sections 1.1, 1.13.1, 1.14, 1.16, 2, 3.2, 19, 21.1.1, 21.1.3, 21.1.6 - 21.1.10, 21.2.4, 22, 24.1, 25.6, 25.8, 25.9, 26.1, 26.7 and 27 of the Technical Report.
- 8) I am independent of the issuer as described in section 1.5 of NI 43-101.

Lycopodium

- 9) I have not been involved in the Boumadine Project previously.
- 10) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
- 11) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this 18th day of December 2025

Signed and Sealed

Preetham Nayak, P. Eng.



CERTIFICATE OF QUALIFIED PERSON BENJAMIN BERSON

I, Benjamin Berson, state that:

- (a) I am a Principal Mining Engineer at:
WSP Canada Inc.
1600, boul. René-Lévesque Ouest, 16^e étage
Montréal, Québec H3H 1P9 Canada
- (b) This certificate applies to the technical report titled Technical Report - Preliminary Economic Assessment for the Boumadine Polymetallic Project, Kingdom of Morocco; with an effective date of November 4, 2025 (the "Technical Report").
- (c) I am a "qualified person" for the purposes of National Instrument 43-101 ("NI 43-101"). My qualifications as a qualified person are as follows. I am a graduate of Ecole Polytechnique of Montréal with a Bachelor of Science degree in Mining Engineering from 2007, I am a member in good standing of the Ordre des Ingénieurs du Québec (145202). My relevant experience after graduation, for the purpose of the Technical Report, includes over 18 years of experience of working as a mine engineer in the field of mine geotechnics and mine planning for underground and open pit mine, including 15 years in gold mines operating in Canada and 3 years working in consulting services for multiple mining projects located in Canada and abroad.
- (d) I did not complete a personal inspection of the property described in the Technical Report.
- (e) I am responsible for Item 16.1, 16.3-16.12, 18.4, 21.1.2, 21.2.1 and portions of Items 1, 25, 26, 27 of the Technical Report.
- (f) I am independent of the issuer as described in section 1.5 of NI 43-101.
- (g) I have not had any involvement with the property that is the subject of the Technical Report.
- (h) I have read NI 43-101 and the parts of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101; and
- (i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible, contain(s) all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Montréal, Québec, Canada this December 18, 2025.

"Signed and Sealed"

Benjamin Berson; P.Eng, PMP.

V1.2021
2021 NI 43-101 QP Certificate_WSP.docx

CERTIFICATE OF QUALIFIED PERSON

I, Ruan Pierre Venter, P.Eng., as an author of this report entitled "Technical Report - Preliminary Economic Assessment for the Boumadine Polymetallic Project, Kingdom of Morocco", prepared for Aya Gold & Silver inc., with an effective date of November 4, 2025 ("Technical Report"), do hereby certify that:

- 1) I am a Manager of Process - Americas with Lycopodium Canada Ltd. My office address is 5090 Explorer Dr, Suite 700, Mississauga, ON, L4W 4T9.
- 2) I am a graduate of the North-West University, South Africa in 2011 with a Bachelor of Engineering degree in Chemical Engineering with Specialization in Mineral Processing.
- 3) I am a member of Professional Engineers Ontario and registered as a Professional Engineer with membership #100592368. I have worked as a metallurgical and process engineer for a total of 15 years since my graduation. My relevant experience for the purpose of the Technical Report is:
 - Metallurgical and operational experience in platinum flotation, chromite foundry sands, and gold operations within South Africa and East Africa; and
 - Lead process engineer on gold and sulphide flotation projects ranging from testwork management, all phases of studies and detail design up to commissioning of numerous projects as well as commissioning manager on three in Africa and Canada.
- 4) I have read the definition of 'qualified person' set out in National Instrument 43-101 ('NI 43-101') and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a 'qualified person' for the purposes of NI 43-101.
- 5) I have not visited the Boumadine Polymetallic Project.
- 6) I am responsible for all of preparation of Item Nos. 1.8, 1.11, 1.13.2, 13, 17, 21.2.2, 21.2.3, 24.2, 25.3, 25.7, 26.4 of the Technical Report.
- 7) I am independent of the Issuer applying the test set out in Section 1.5.(4) of NI 43-101.
- 8) I have not been involved previously in the Boumadine Polymetallic Project.
- 9) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

Lycopodium

- 10) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 18th day of December, 2025

Signed and Sealed

Ruan Pierre Venter, P.Eng.

Lycopodium

CERTIFICATE OF QUALIFIED PERSON

I, Zunedbhai Shaikh, P.Eng., as an author of this report entitled "Technical Report - Preliminary Economic Assessment for the Boumadine Polymetallic Project, Kingdom of Morocco for the Boumadine Polymetallic Project, Kingdom of Morocco, prepared for Aya Gold & Silver inc., with an effective date of November 4, 2025 ("Technical Report"), do hereby certify that:

- 1) I am a Lead Mechanical Engineer with Lycopodium Canada Ltd. My office address is 5090 Explorer Dr, Suite 700, Mississauga, ON, L4W 4T9.
- 2) I am a graduate of the Toronto Metropolitan University (Formerly Ryerson University) in 2008 with a Bachelor of Engineering degree in Mechanical Engineering.
- 3) I am a Member of Professional Engineers of Ontario and registered as a Professional Engineer in the Province of Ontario (Registration No. 100137621). I have worked as a Mechanical Engineer for a total of 17 years since my graduation. My relevant experience for the purpose of the Technical Report is:
 - review and report as a consultant on numerous process facilities and mining projects around the world for due diligence and regulatory requirements
 - Lead Mechanical Engineer on a number of feasibility studies and detailed designs in the gold industry in Africa, Australia and Asia.
 - Commissioning Engineer at a number of gold mines in Africa, Australia and Asia
- 4) I have read the definition of 'qualified person' set out in National Instrument 43-101 ('NI 43-101') and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a 'qualified person' for the purposes of NI 43-101.
- 5) I have not visited the Boumadine Project site.
- 6) I am responsible for all of preparation of Item Nos. 1.12, 18.1 to 18.3, 18.6 to 18.13 and 25.4 of the Technical Report.
- 7) I am independent of the Issuer applying the test set out in Section 1.5.(4) of NI 43-101.
- 8) I have not been involved previously in the Boumadine Project.

Lycopodium

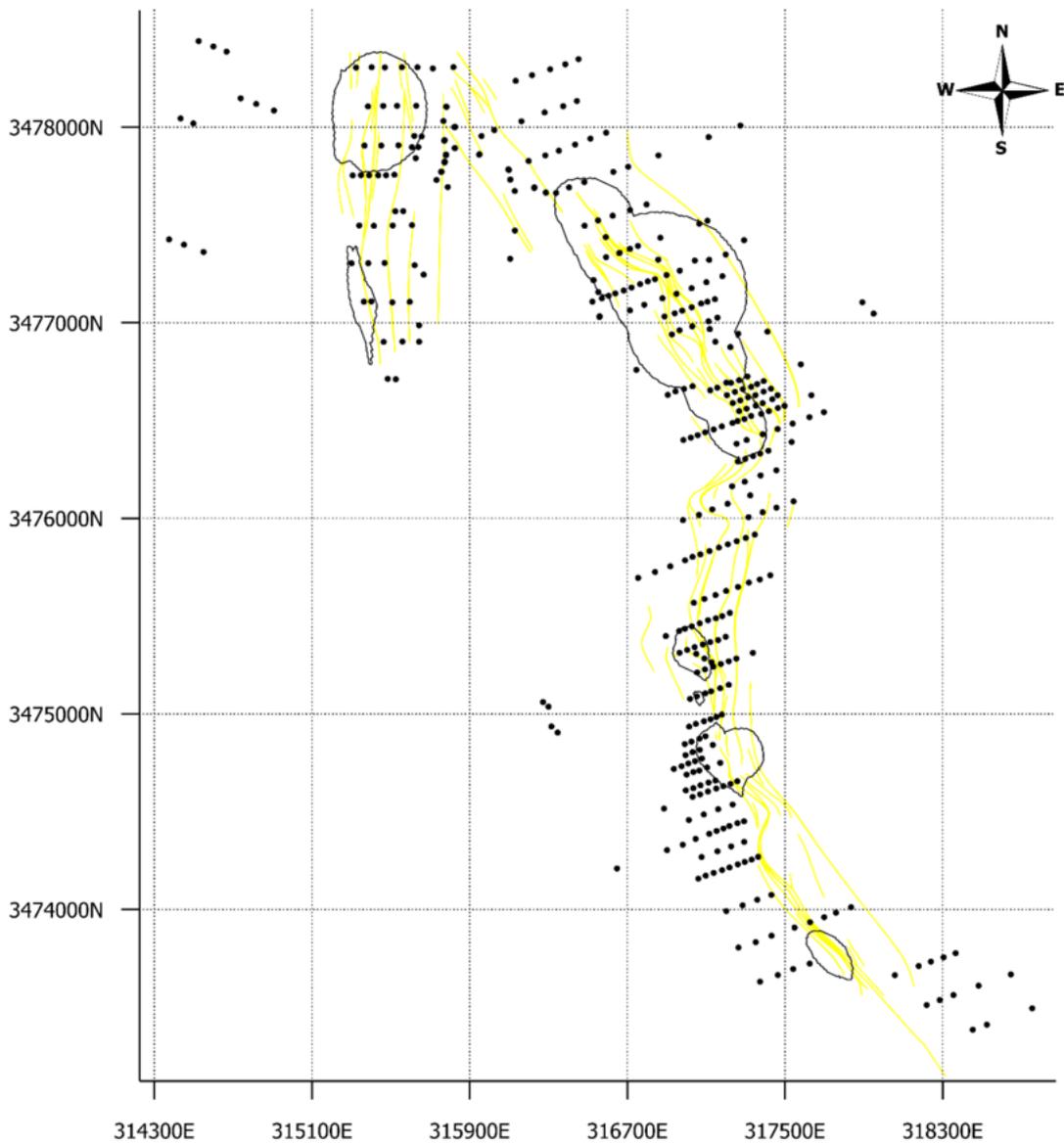
- 9) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
- 10) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

| Dated this 18th day of December, 2025

Signed and Sealed

Zunedbhai Shaikh, P.Eng.

Appendix A DRILL HOLE PLAN



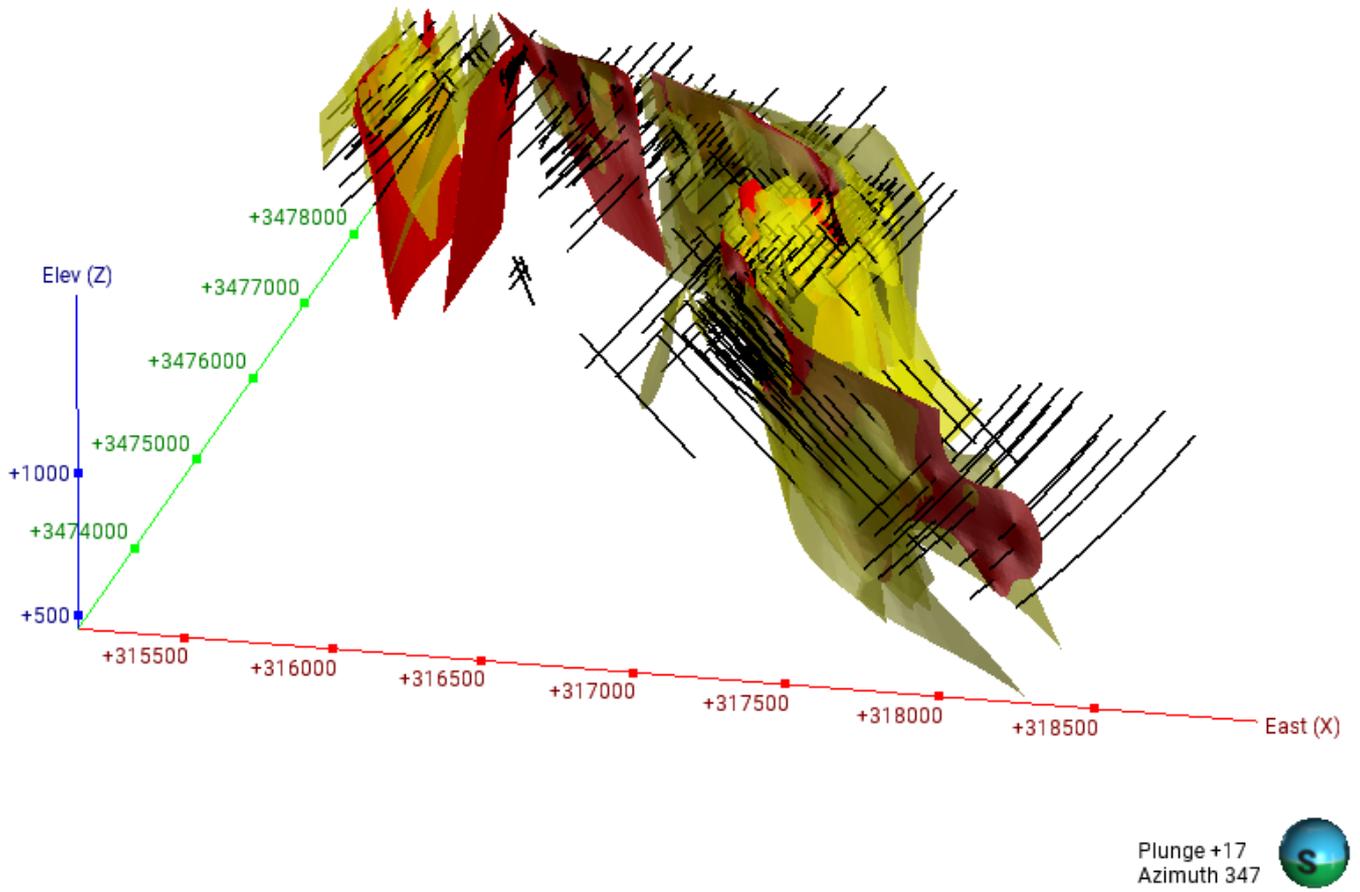
Aya Gold & Silver Inc.
-
Boumadine Deposit
Surface Drill Hole Plan
MRE - February 2025
-
Veins Sliced at 1000rl
-
Pit Contours sliced at 1200rl
-
Date: Feb 25, 2025
WGS 84/UTM Zone 30N



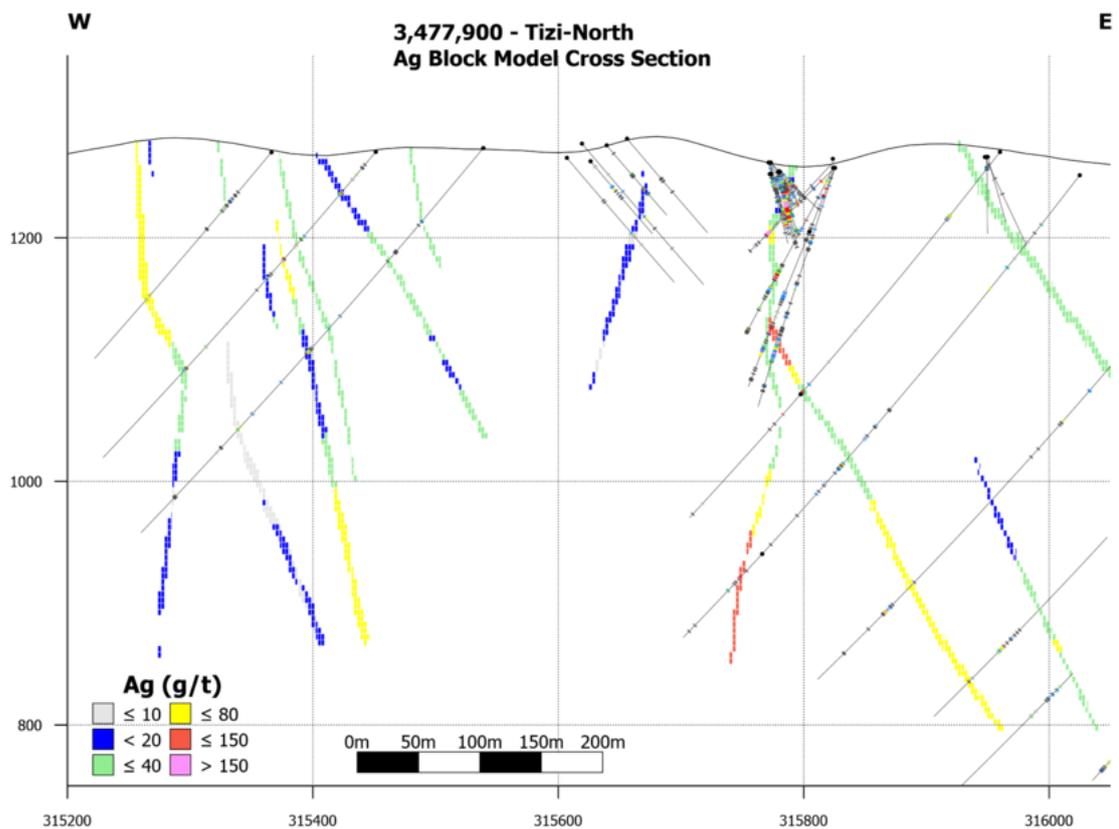
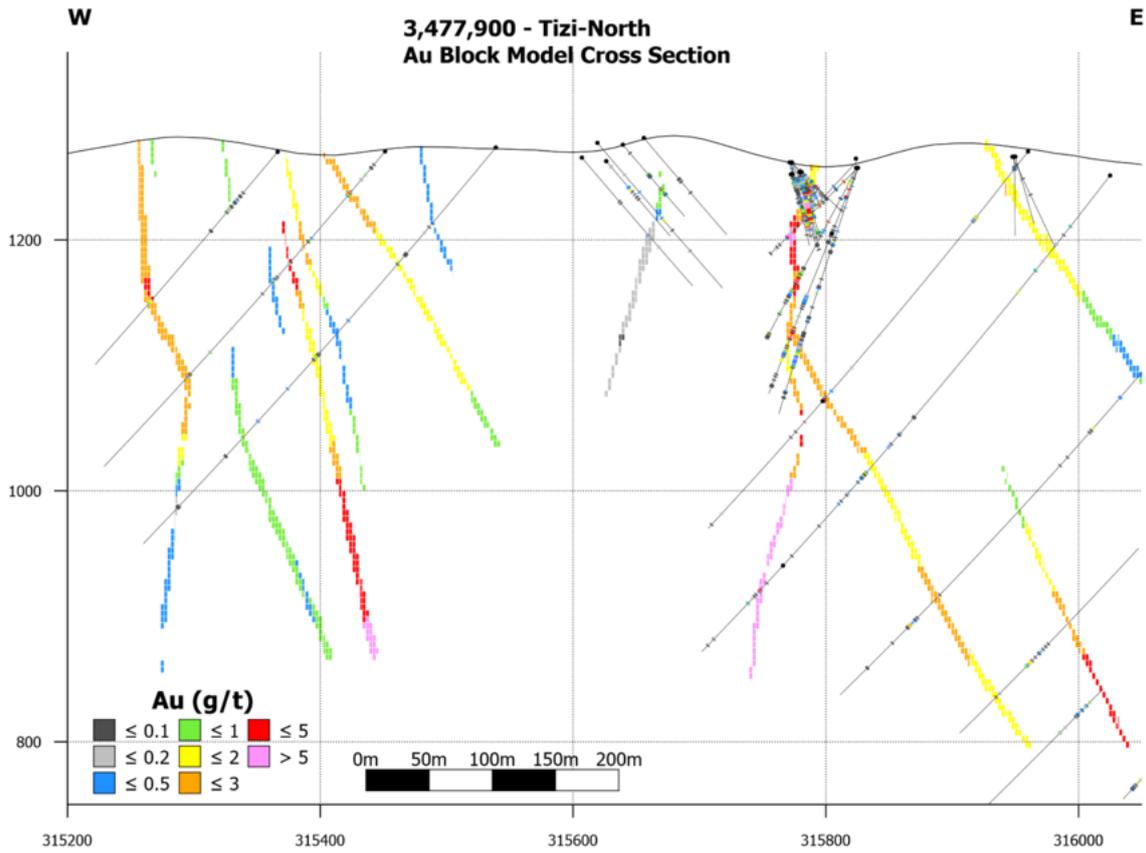
Appendix B 3-D DOMAINS

Boumadine 3D-Domains

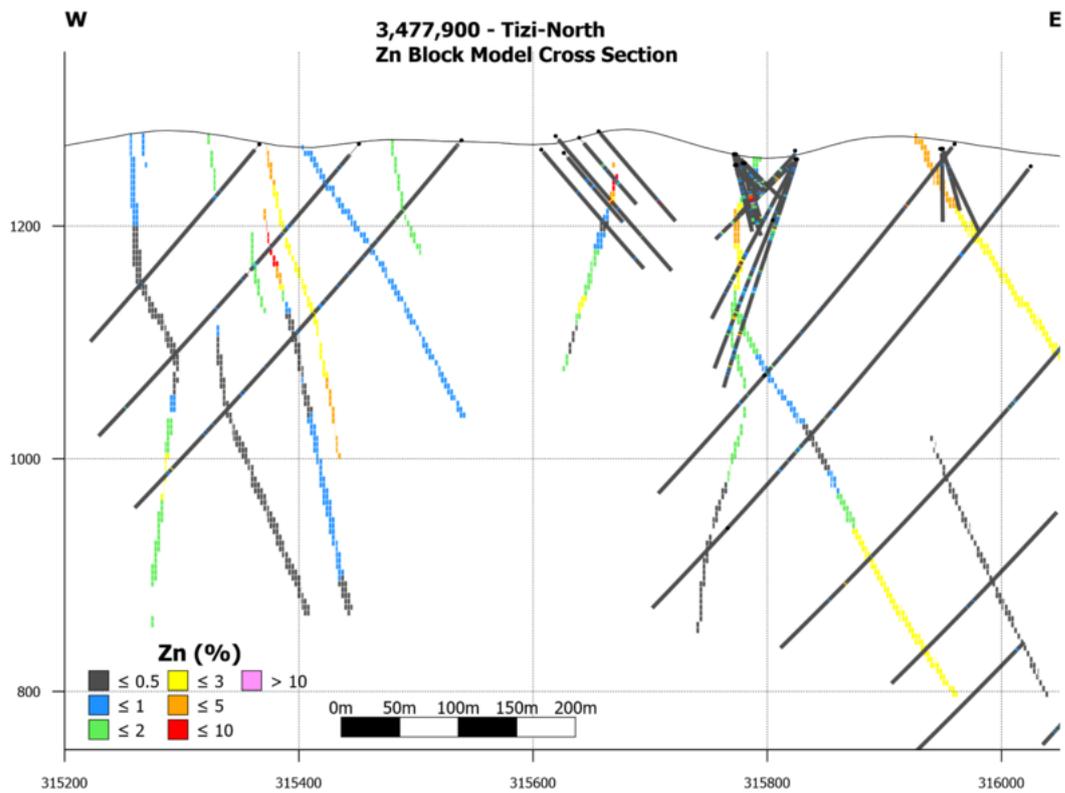
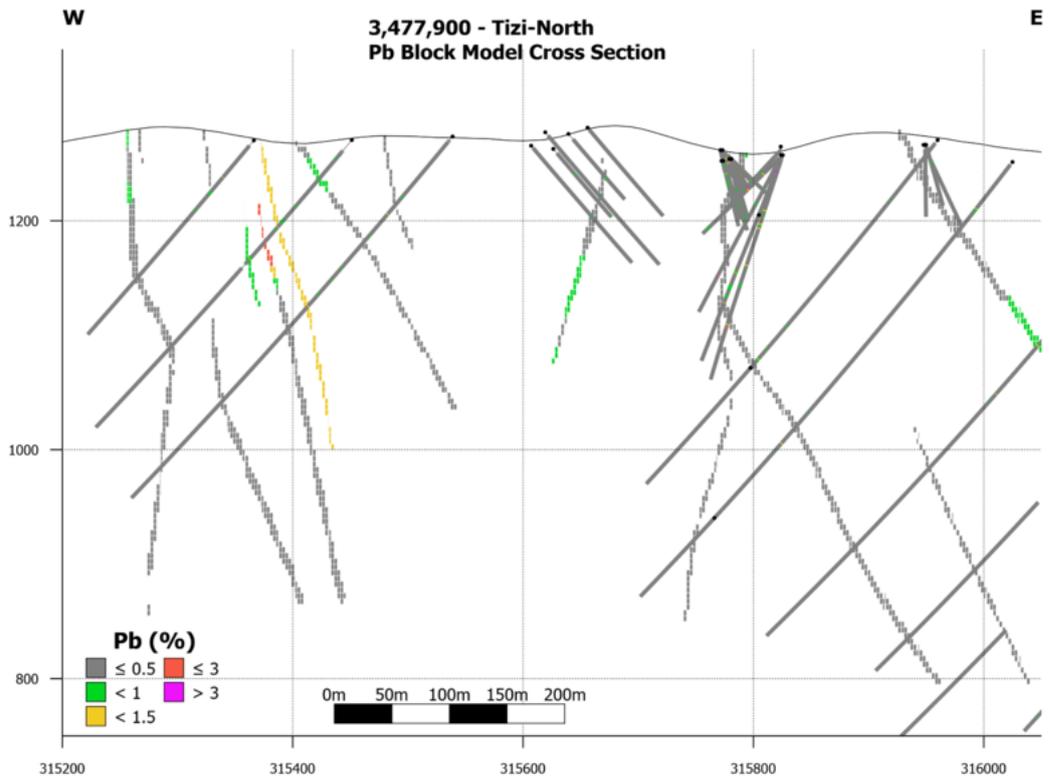
main veins in red (Central_main, South_main, Tizi_Main, Imariren)

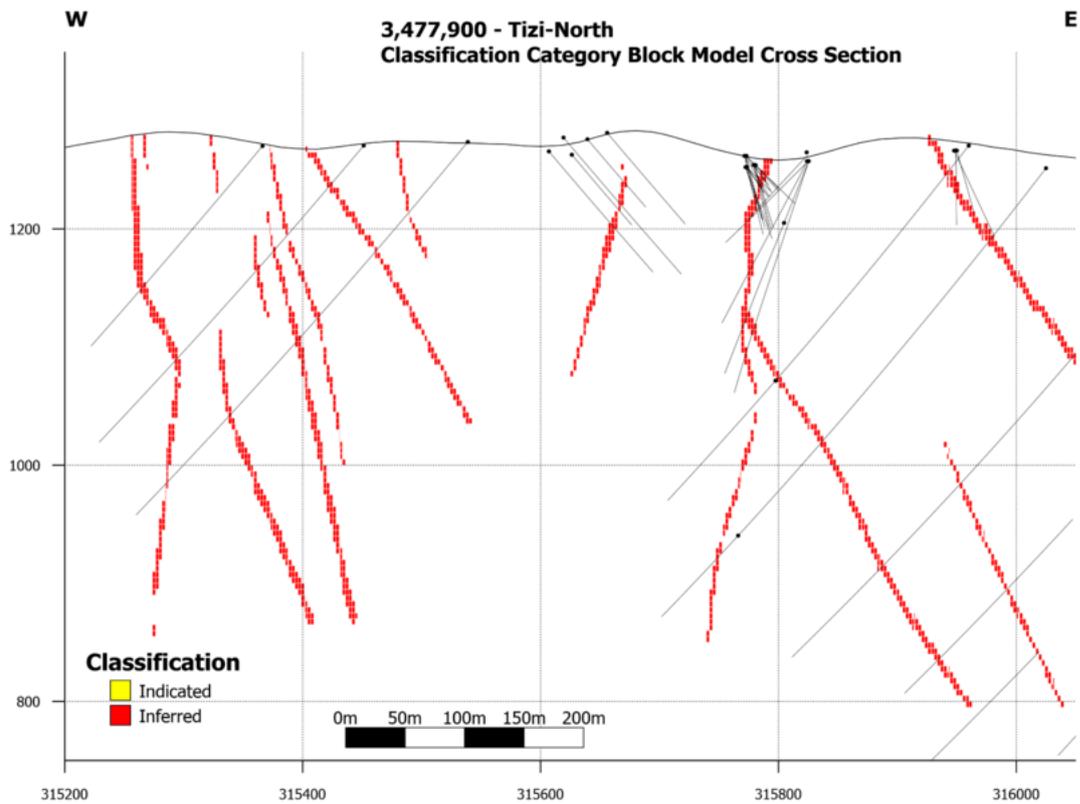
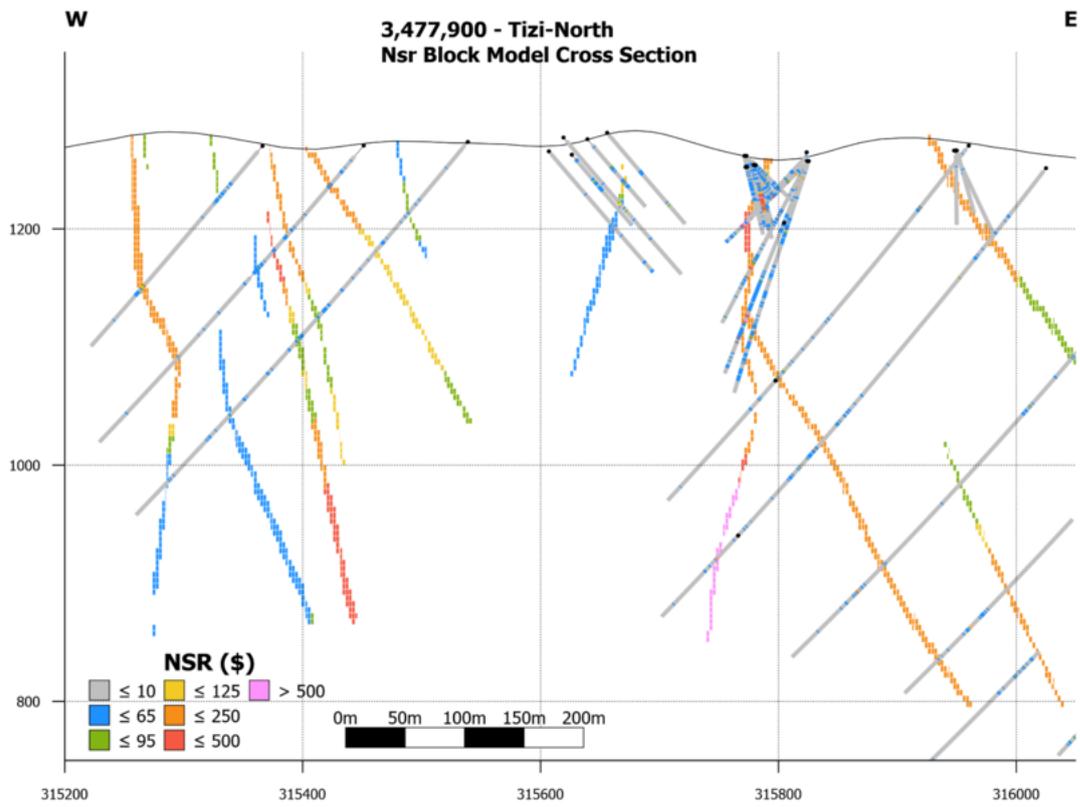


Appendix C TIZI & NORTH ZONES BLOCK MODEL CROSS SECTIONS

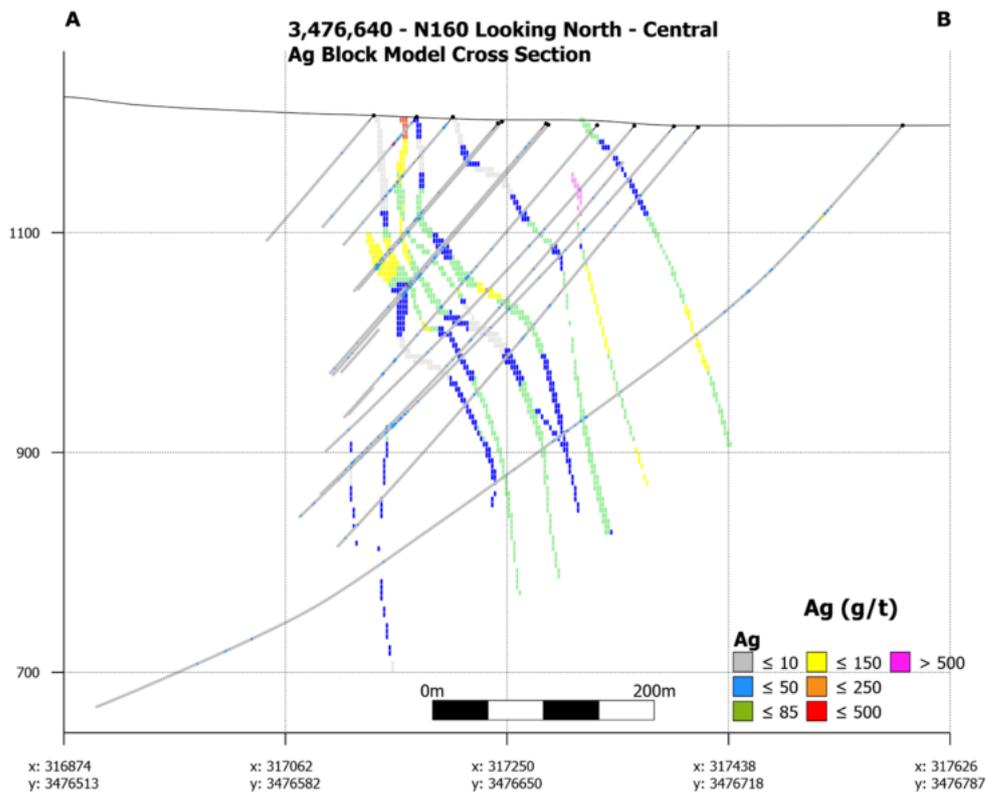
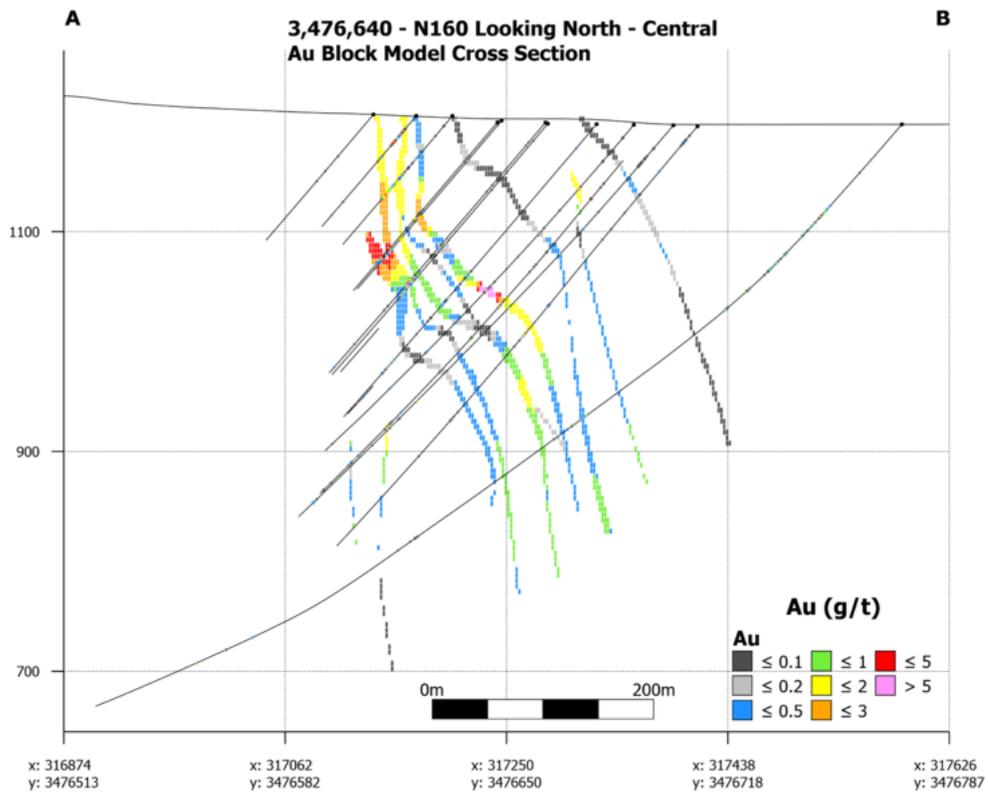


TIZI |

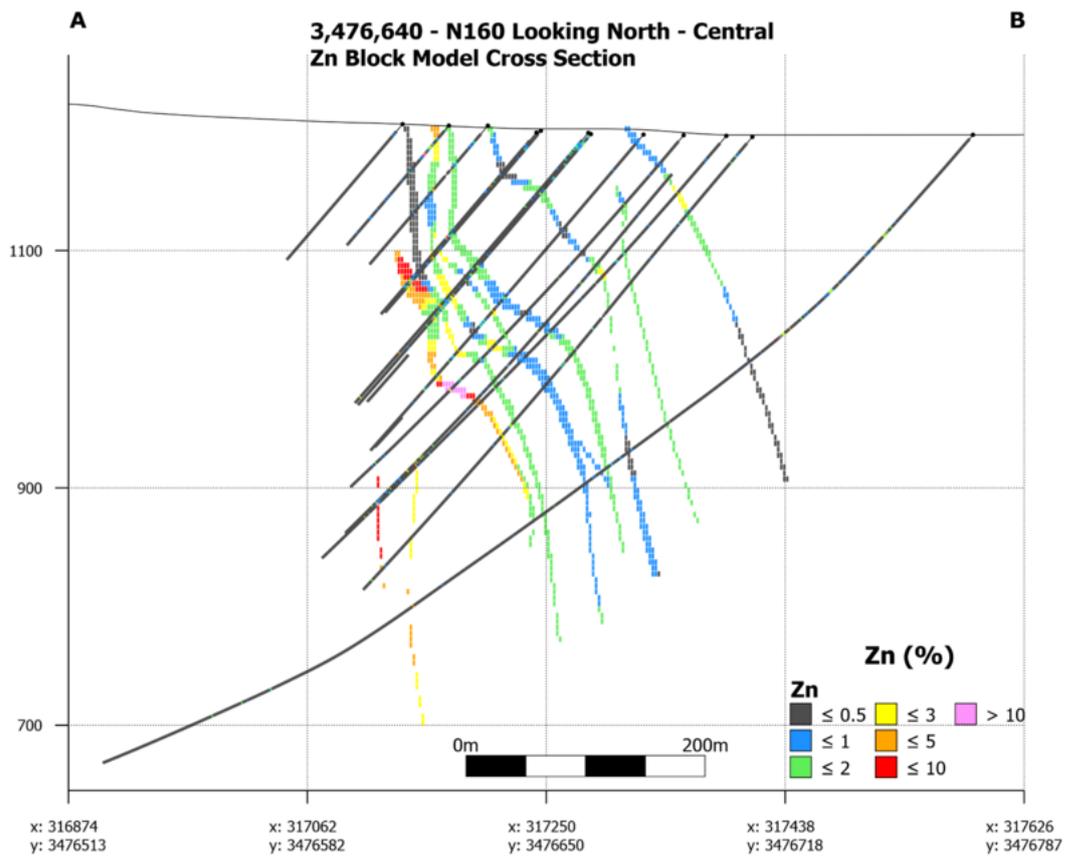
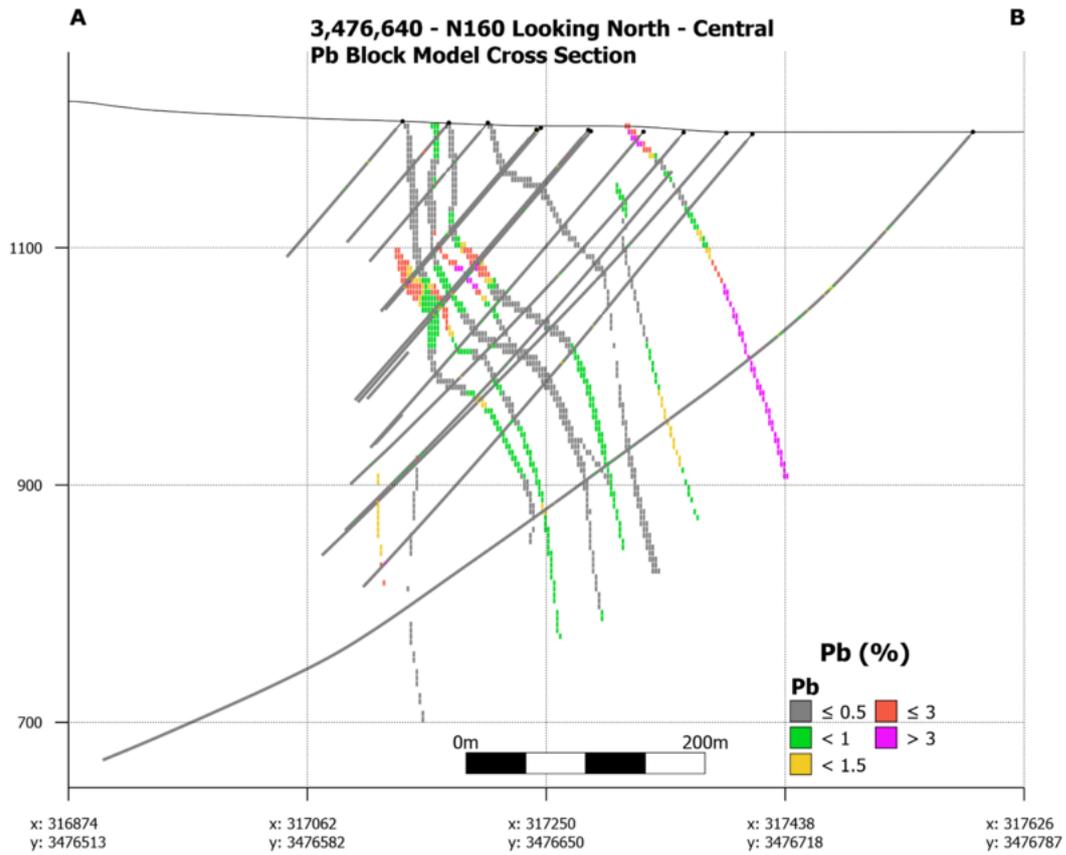


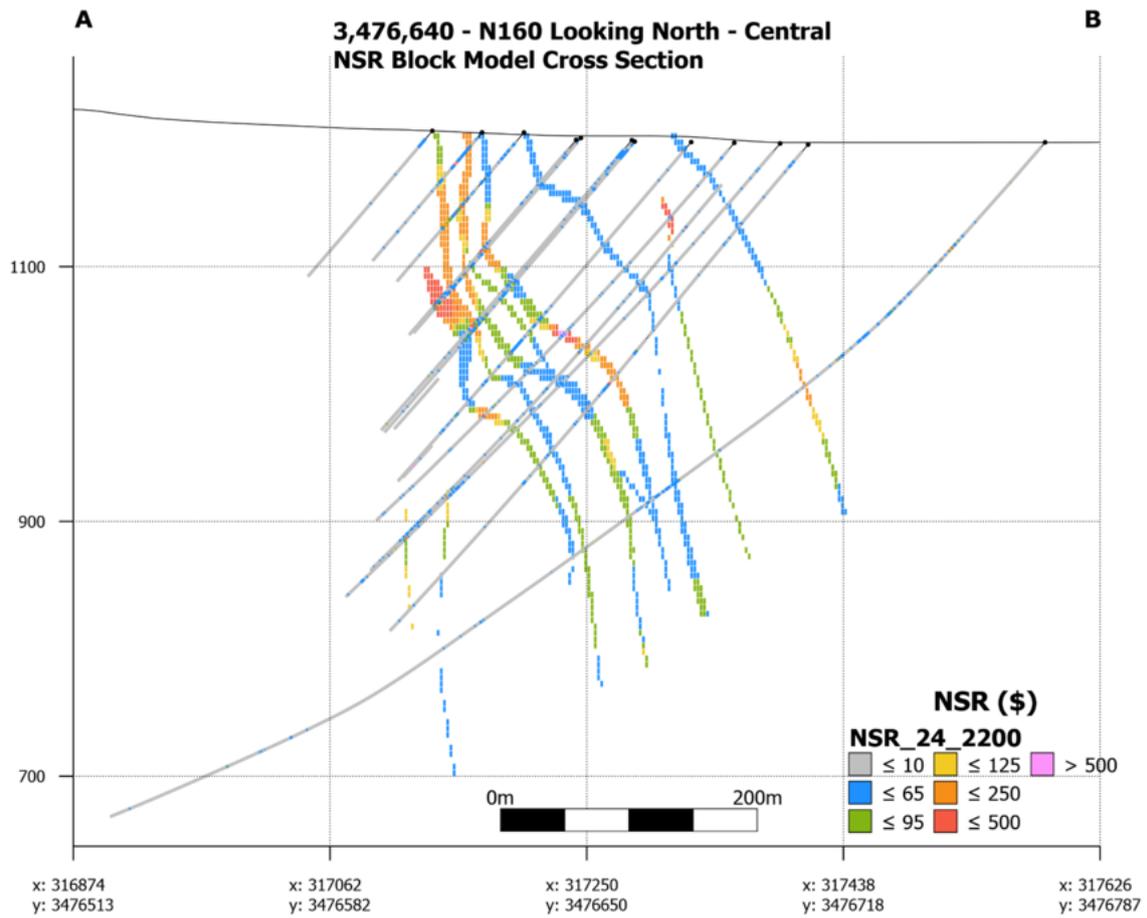


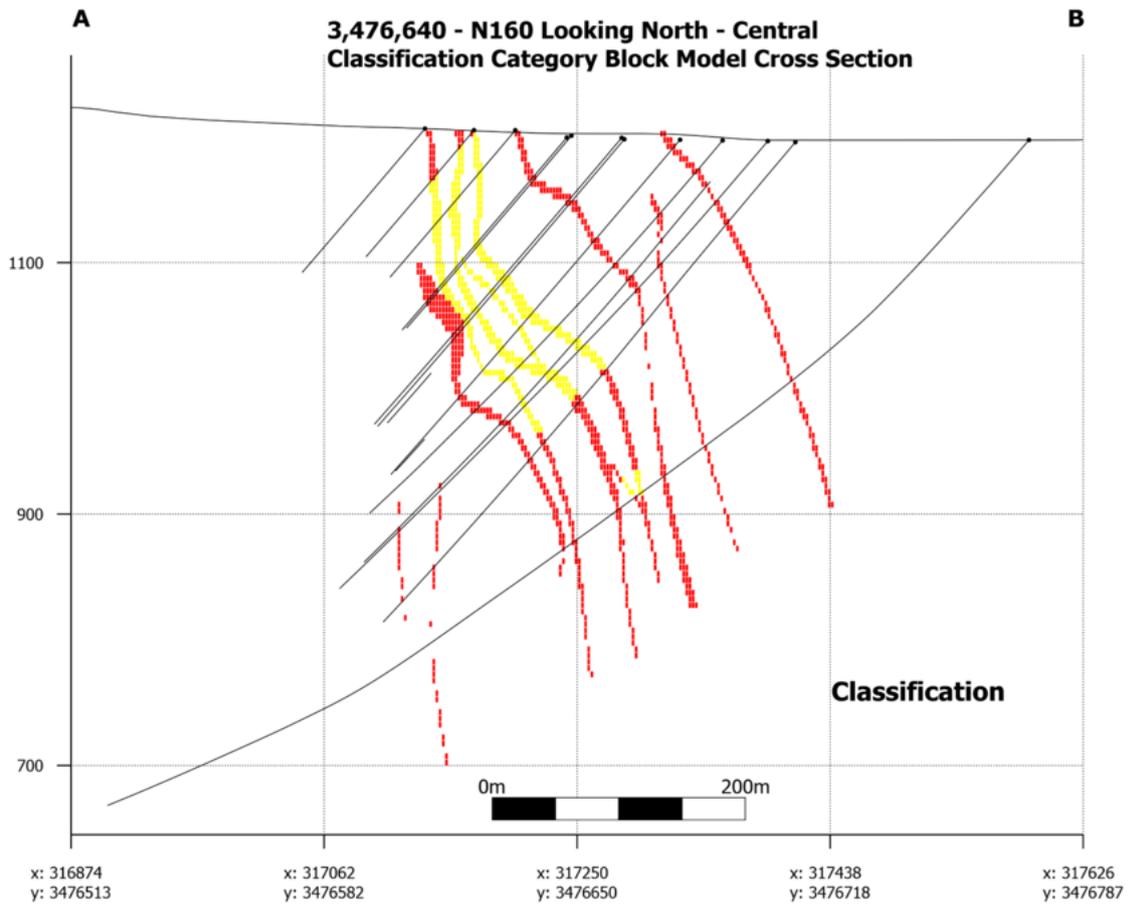
Appendix D CENTRAL ZONE BLOCK MODEL CROSS SECTIONS



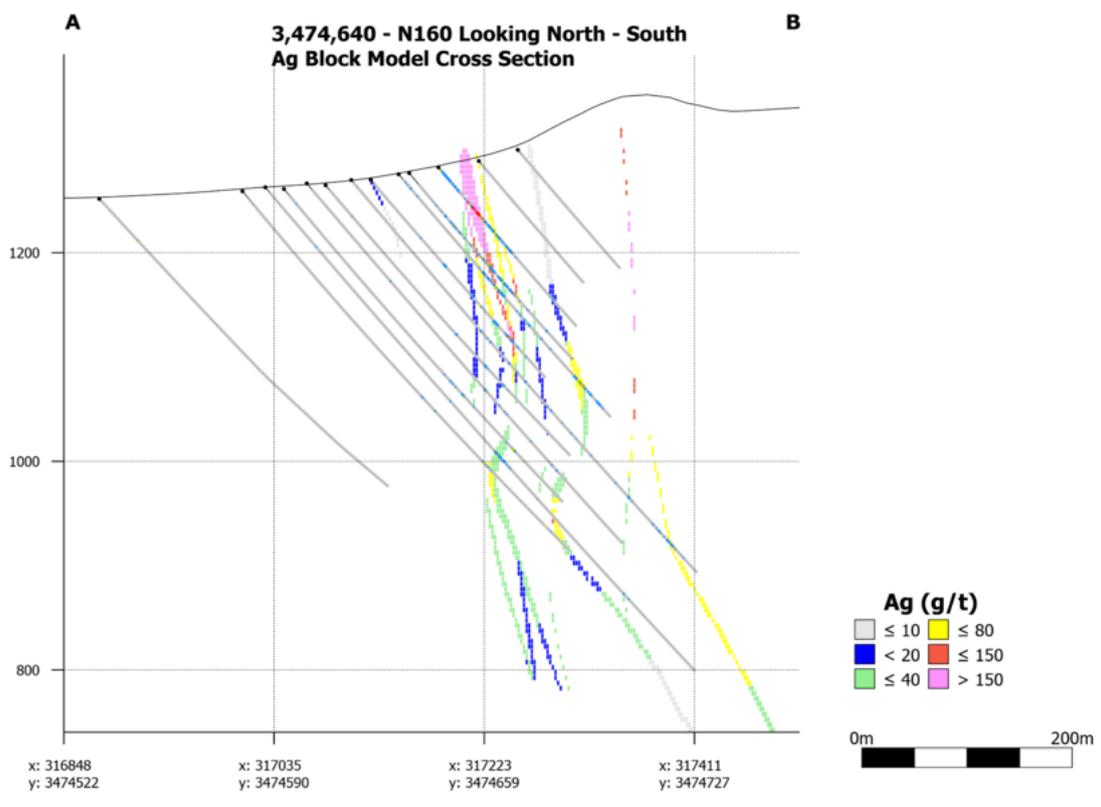
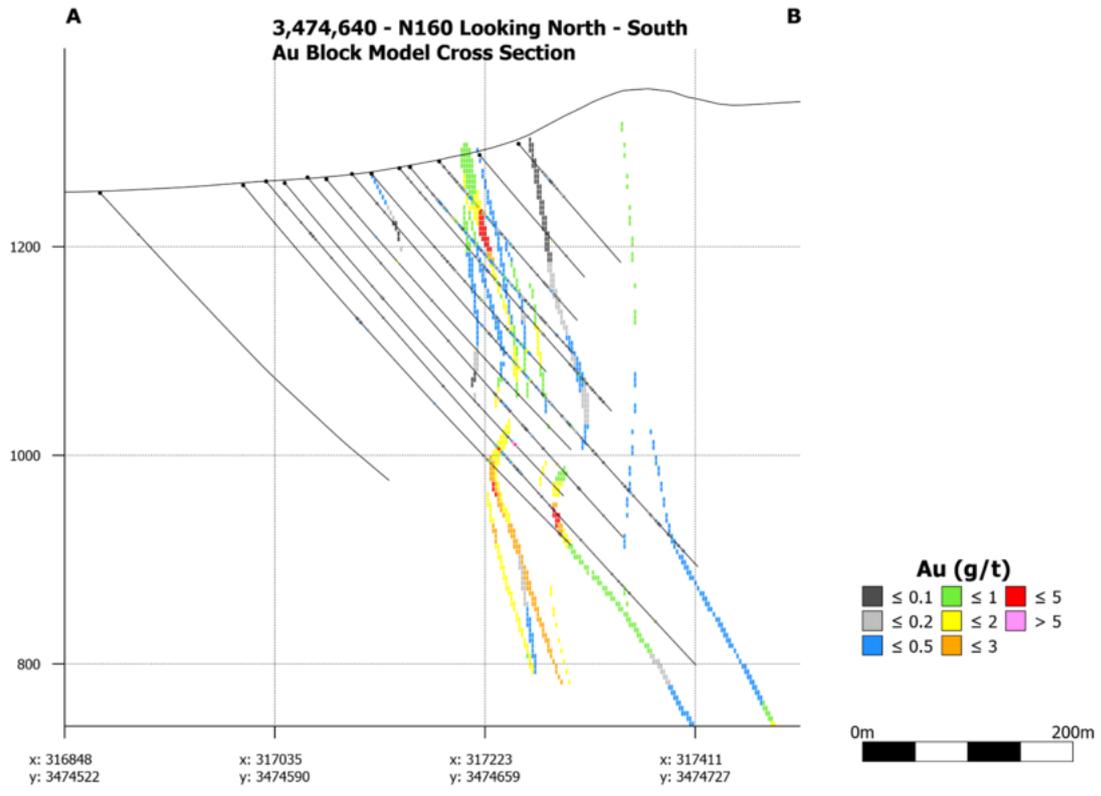
TIZI |



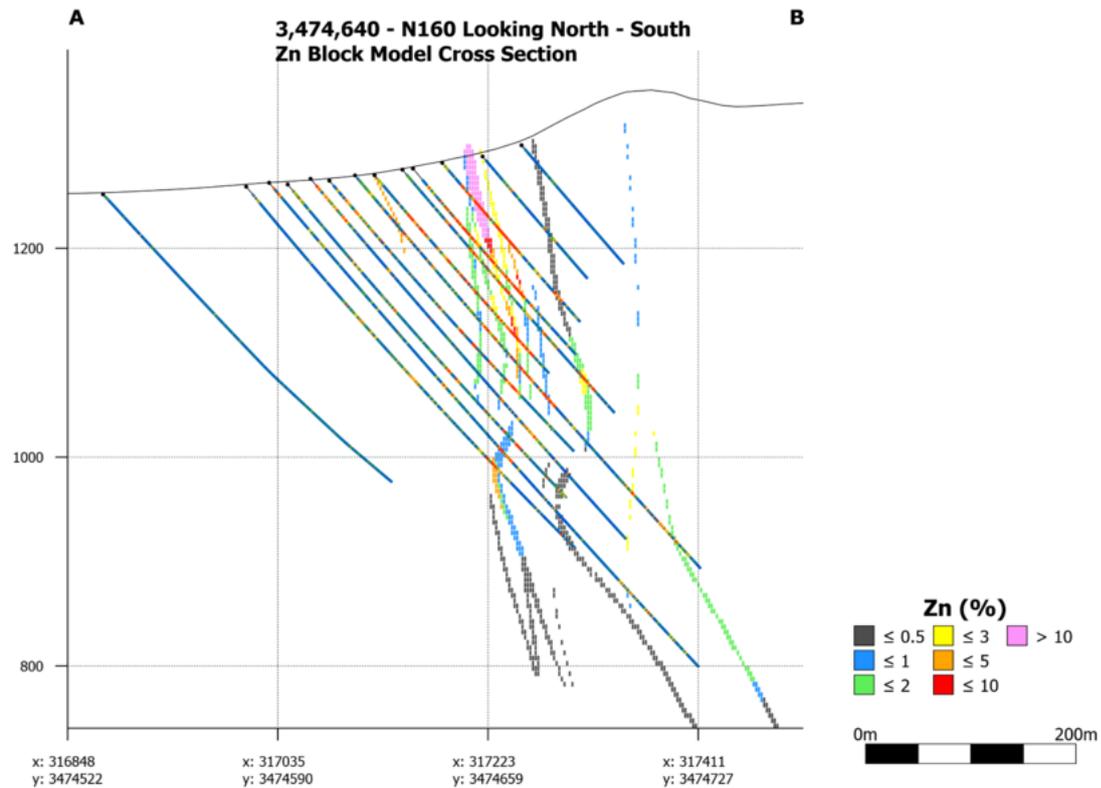
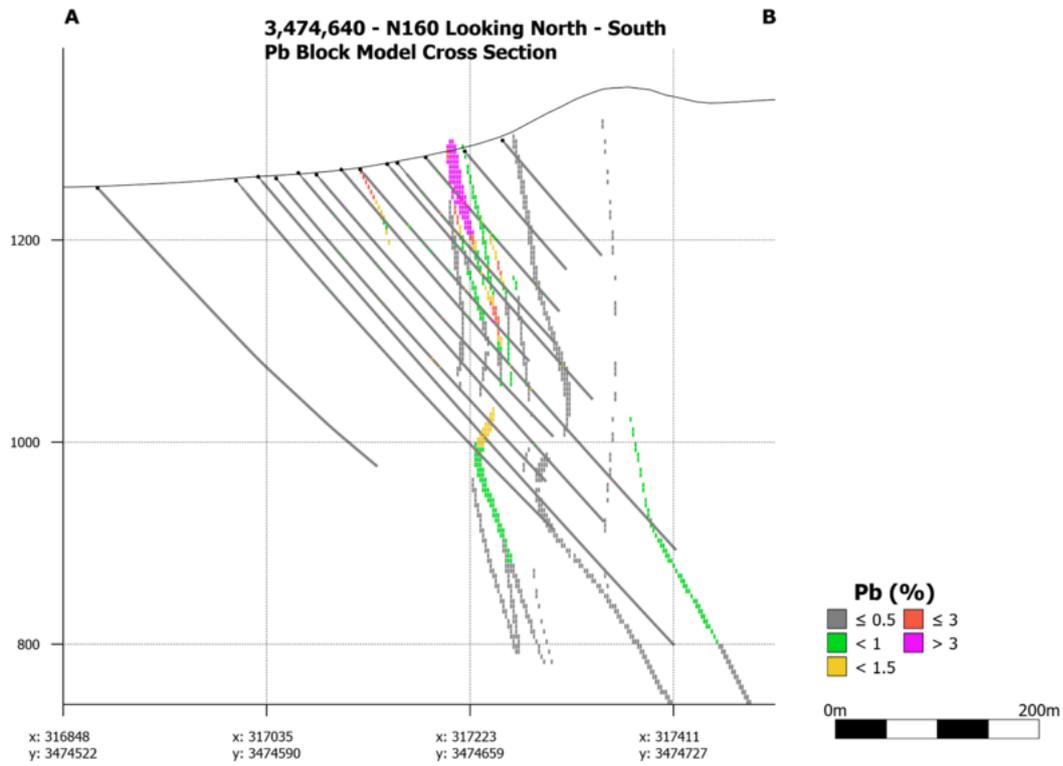


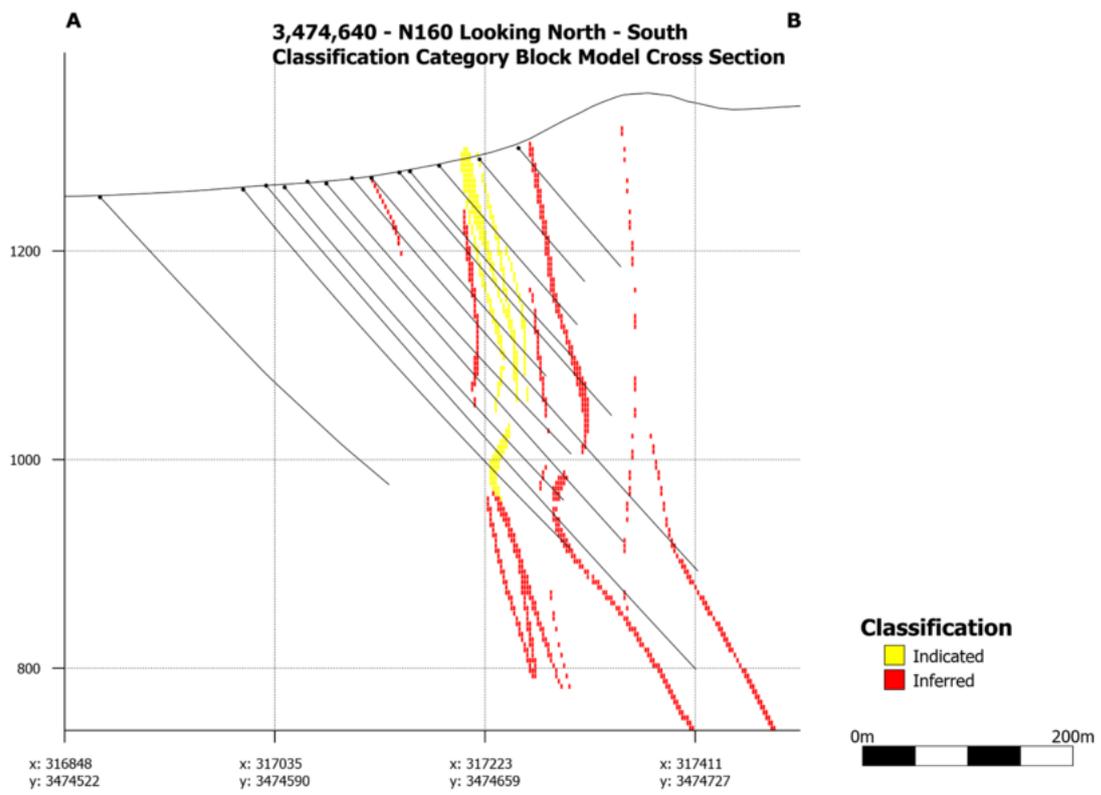
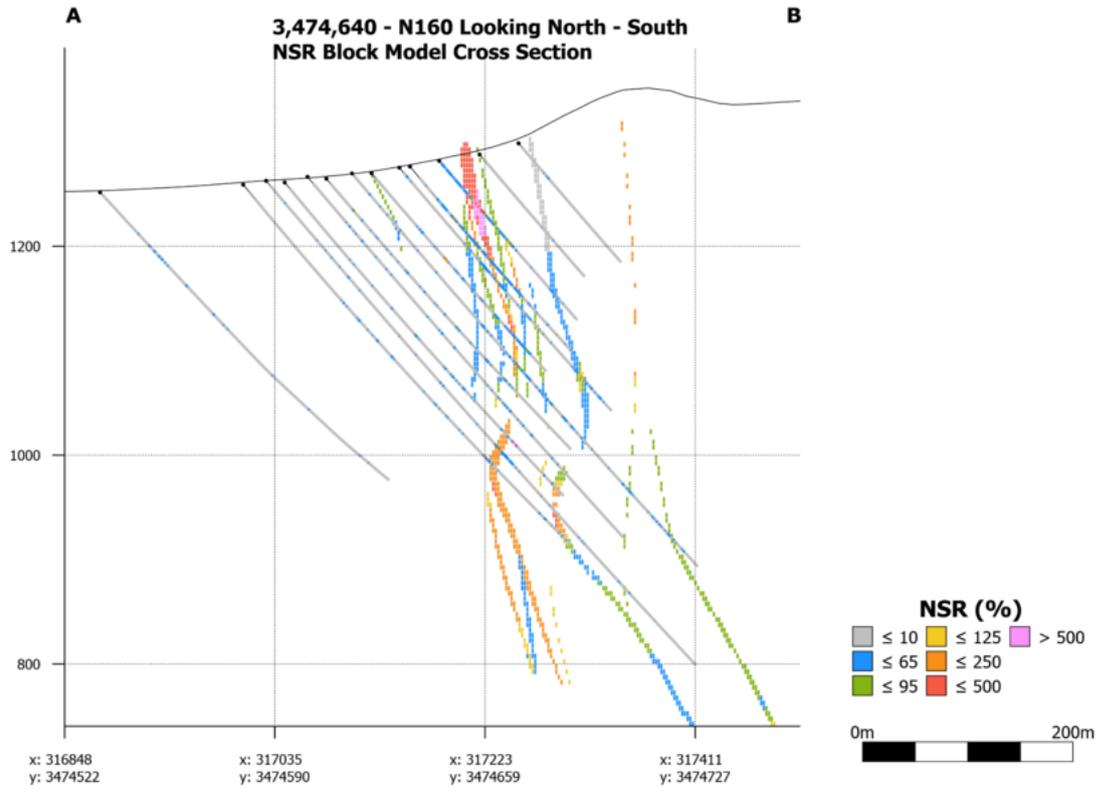


Appendix E SOUTH ZONE BLOCK MODEL CROSS SECTIONS

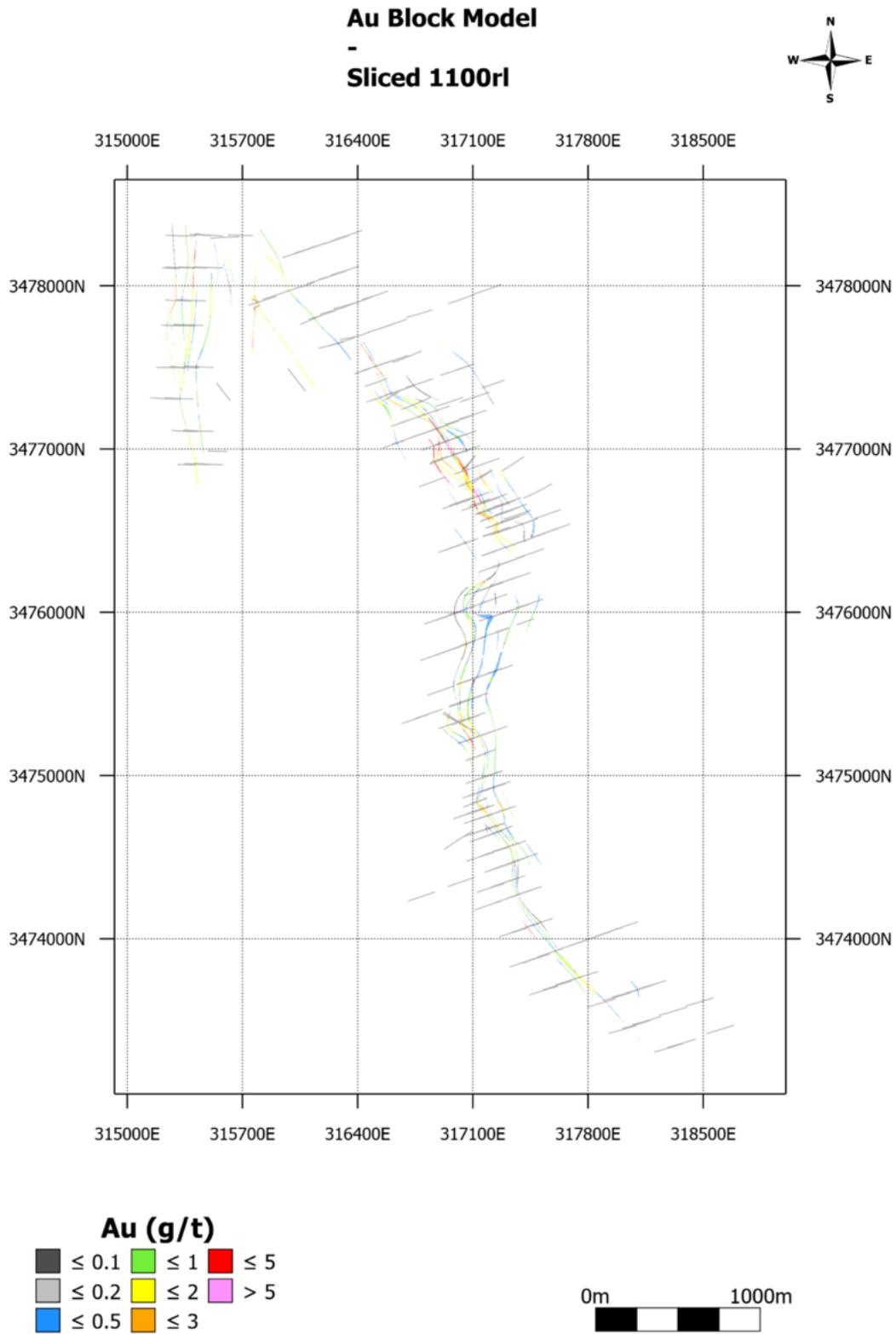


TIZI |

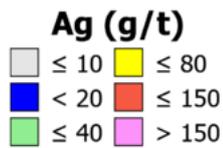
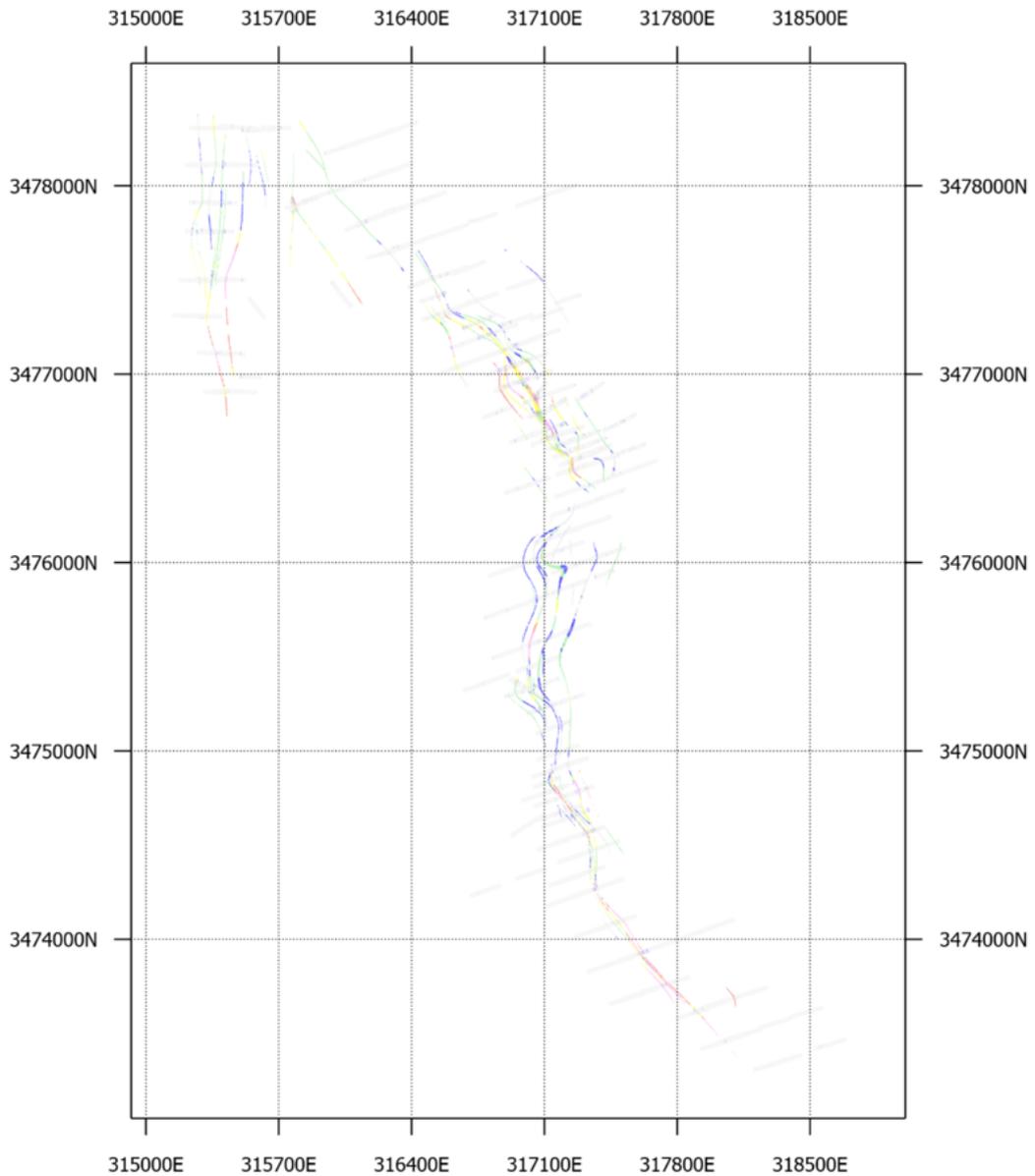




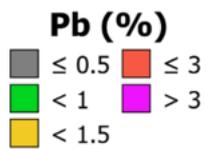
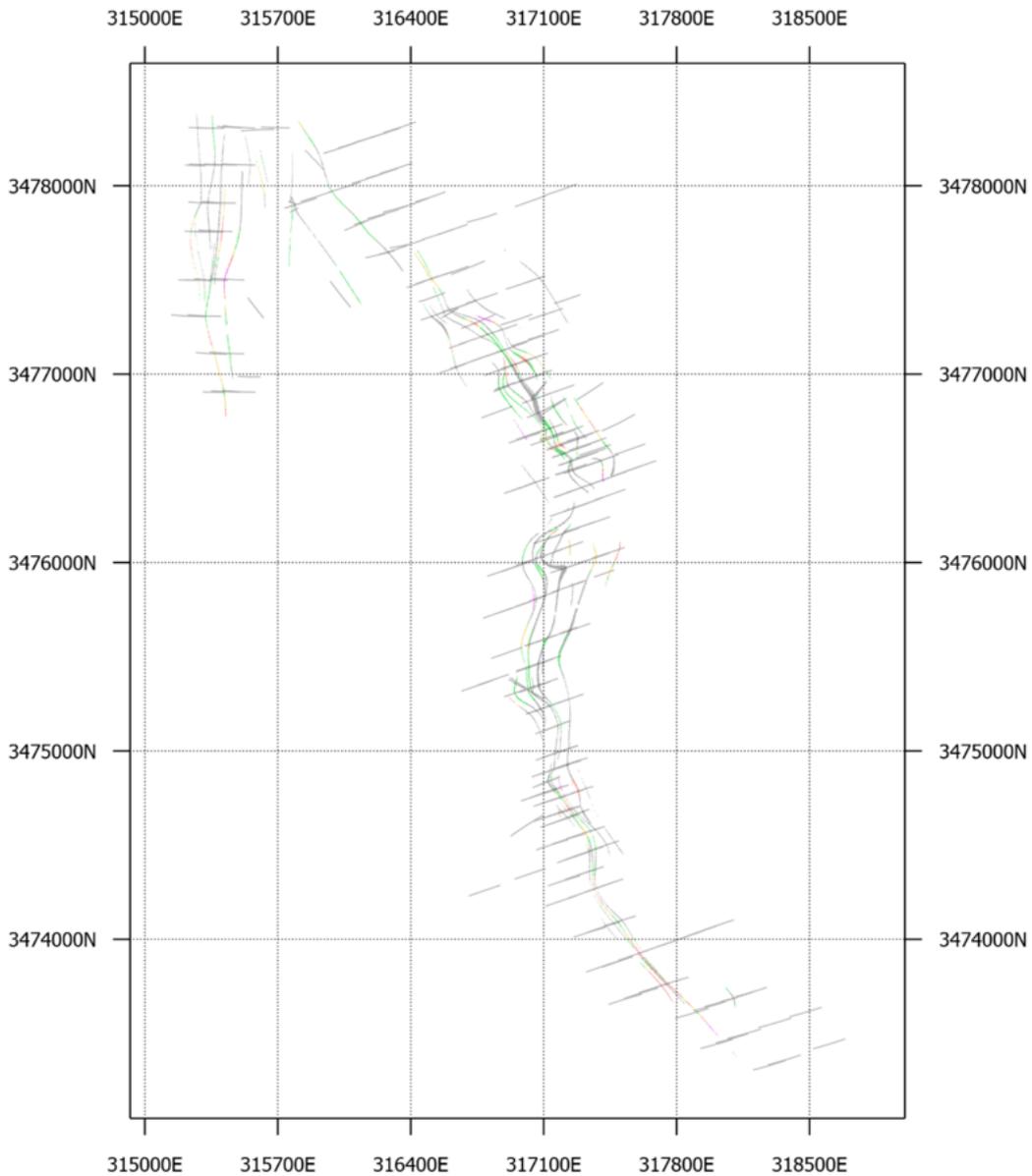
Appendix F BLOCK MODEL PLAN VIEWS



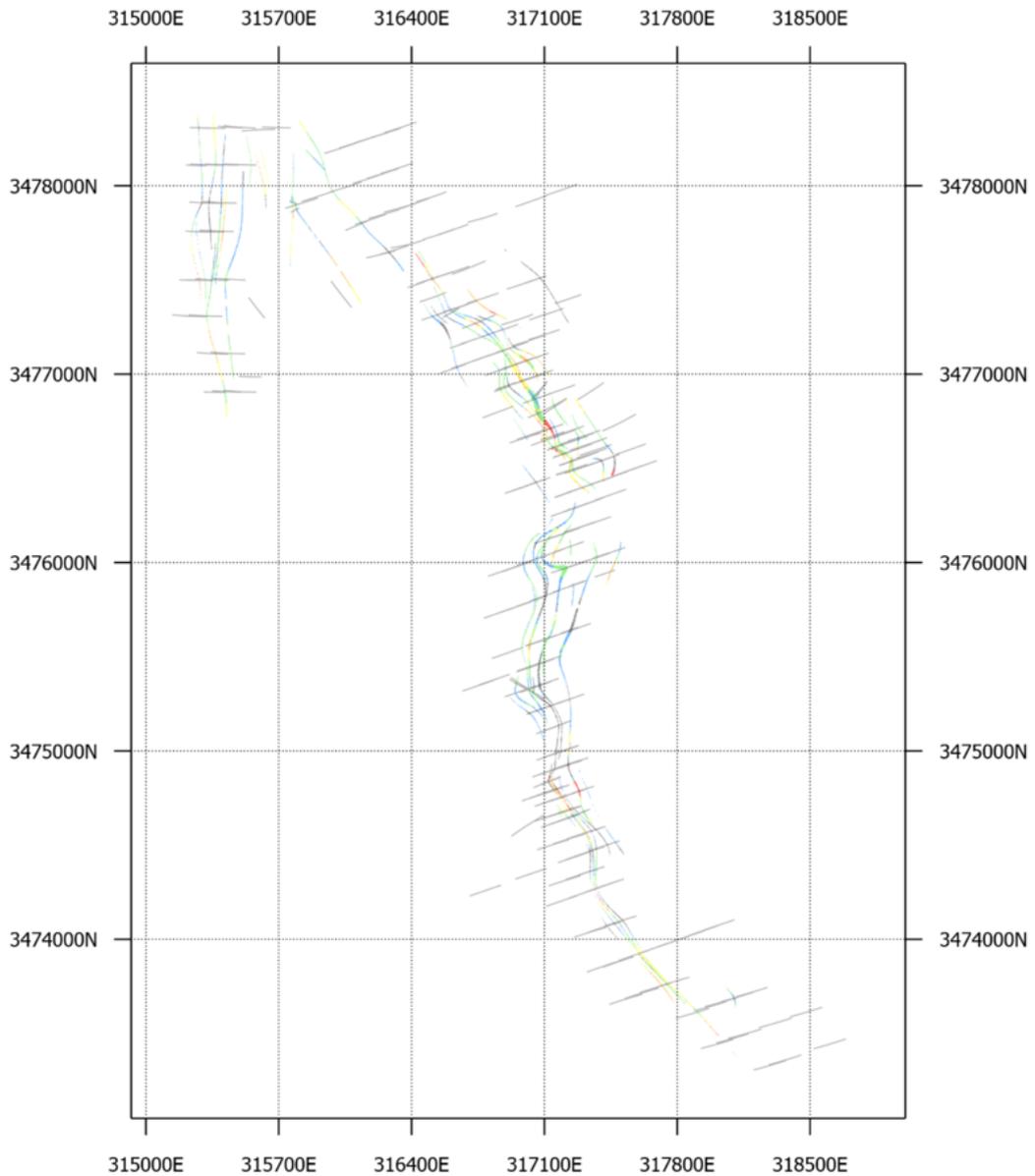
Ag Block Model - Sliced 1100rl



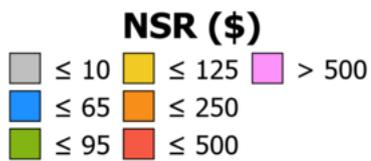
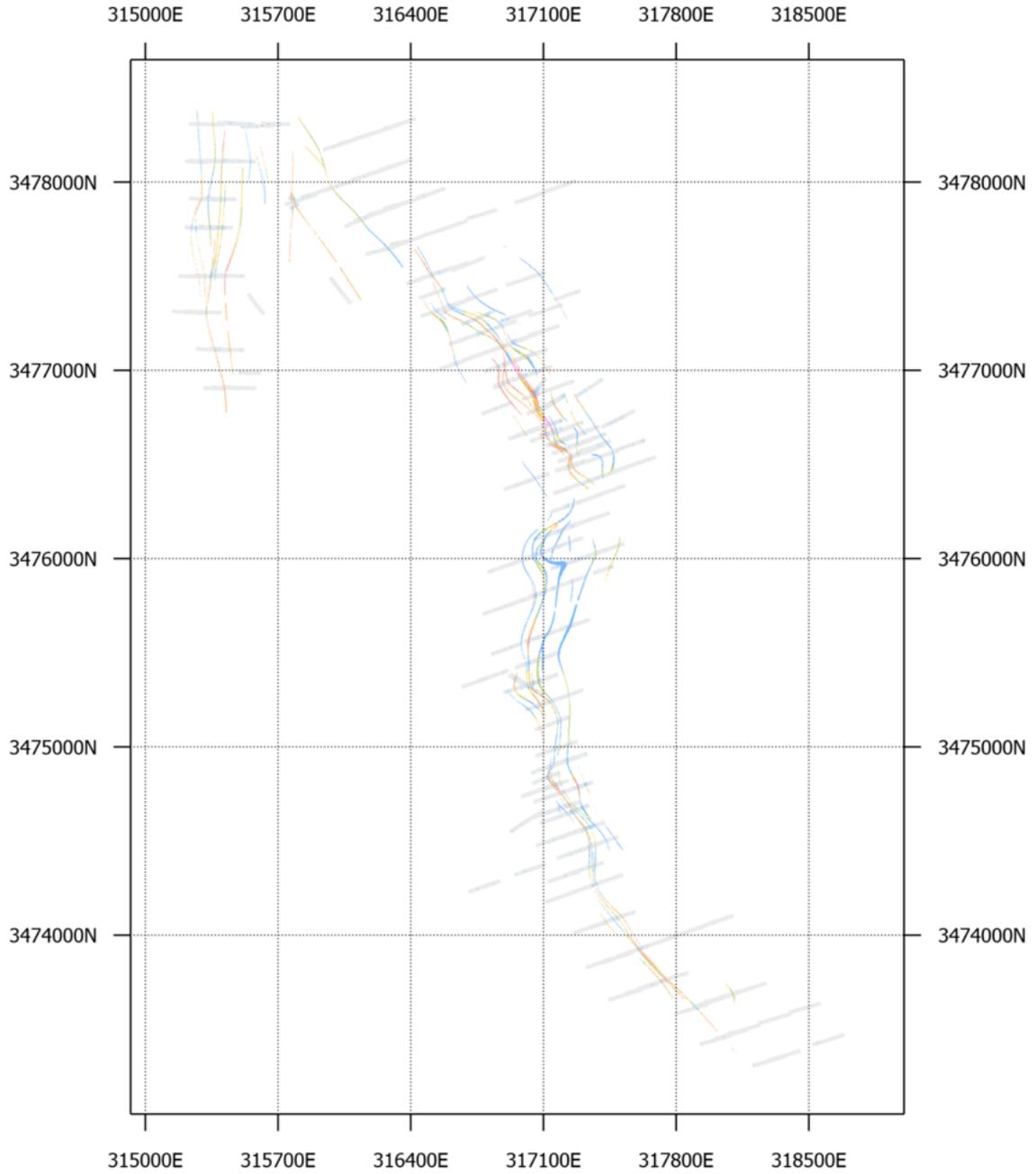
Pb Block Model
-
Sliced 1100rl



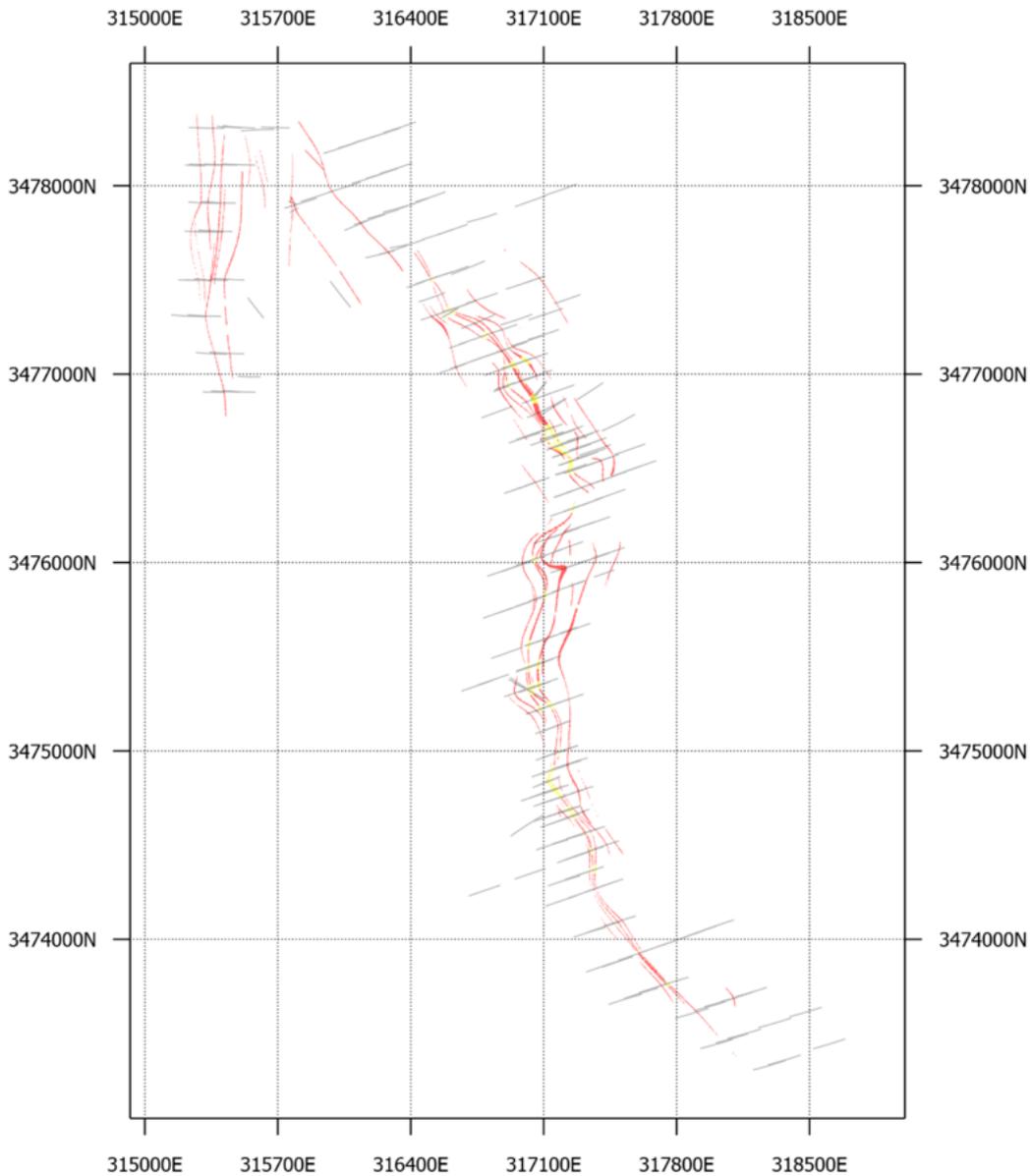
Zn Block Model - Sliced 1100rl



NSR Block Model - Sliced 1100rl



Classification Category Block Model - Sliced 1100rl



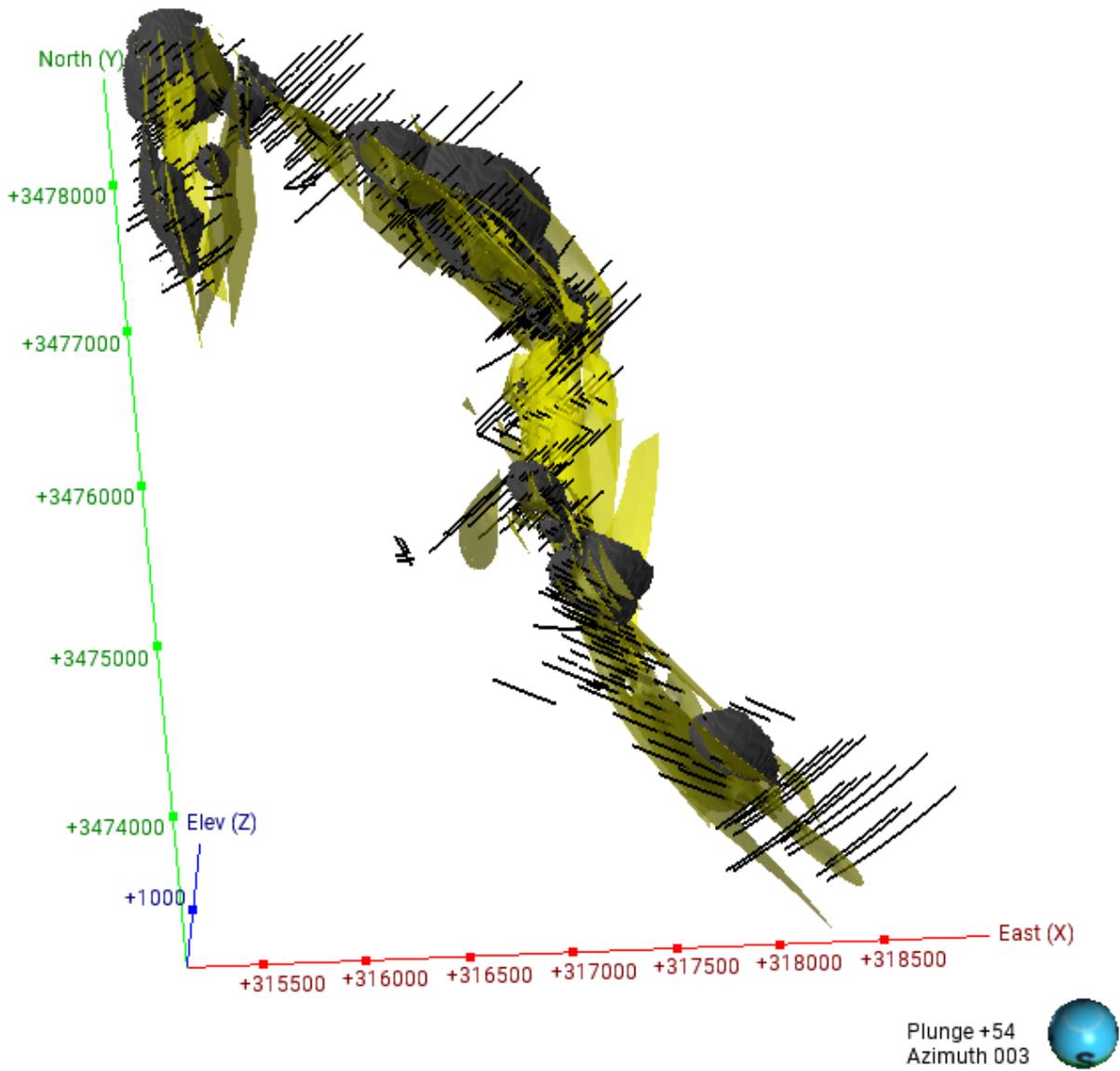
Classification

-  Indicated
-  Inferred



Appendix G OPTIMIZED PIT SHELLS

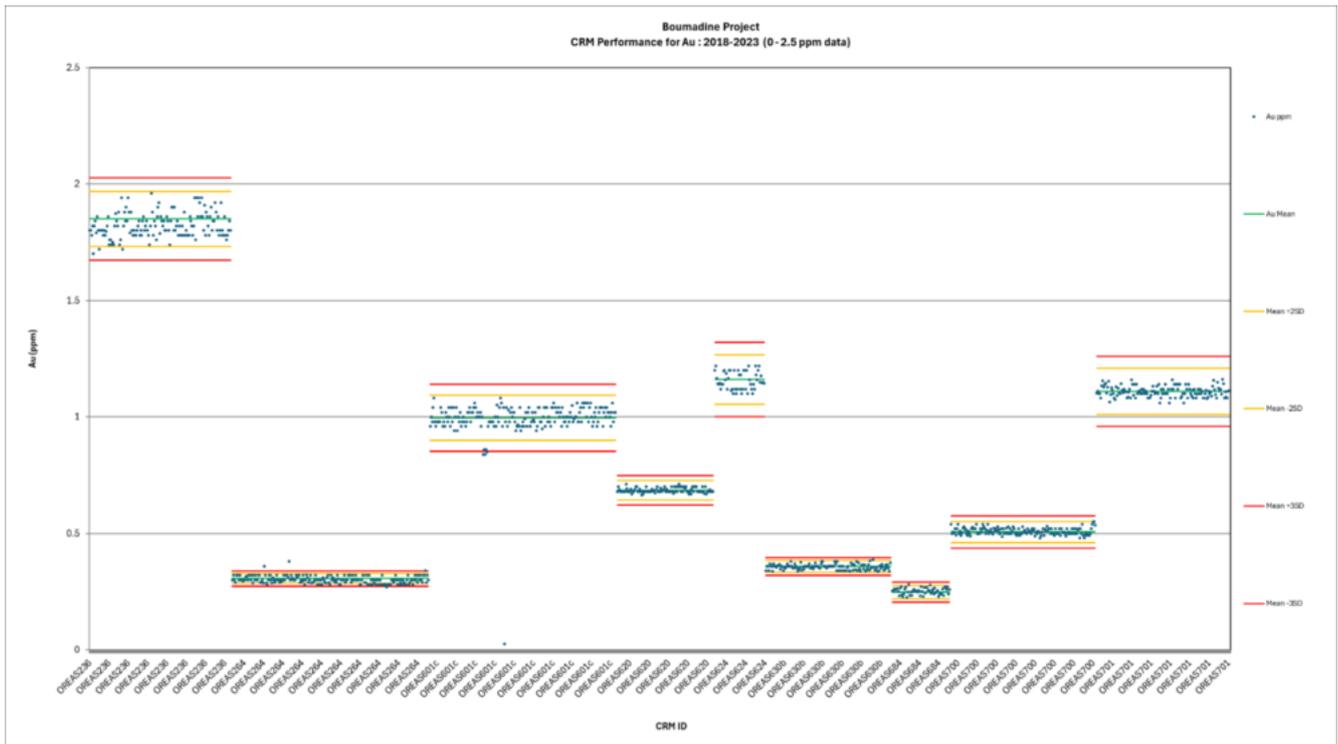
Optimized Pit Shells in 3D



Appendix H QAQC– 2018 TO 2024 DIAMOND DRILL HOLE PROGRAMS

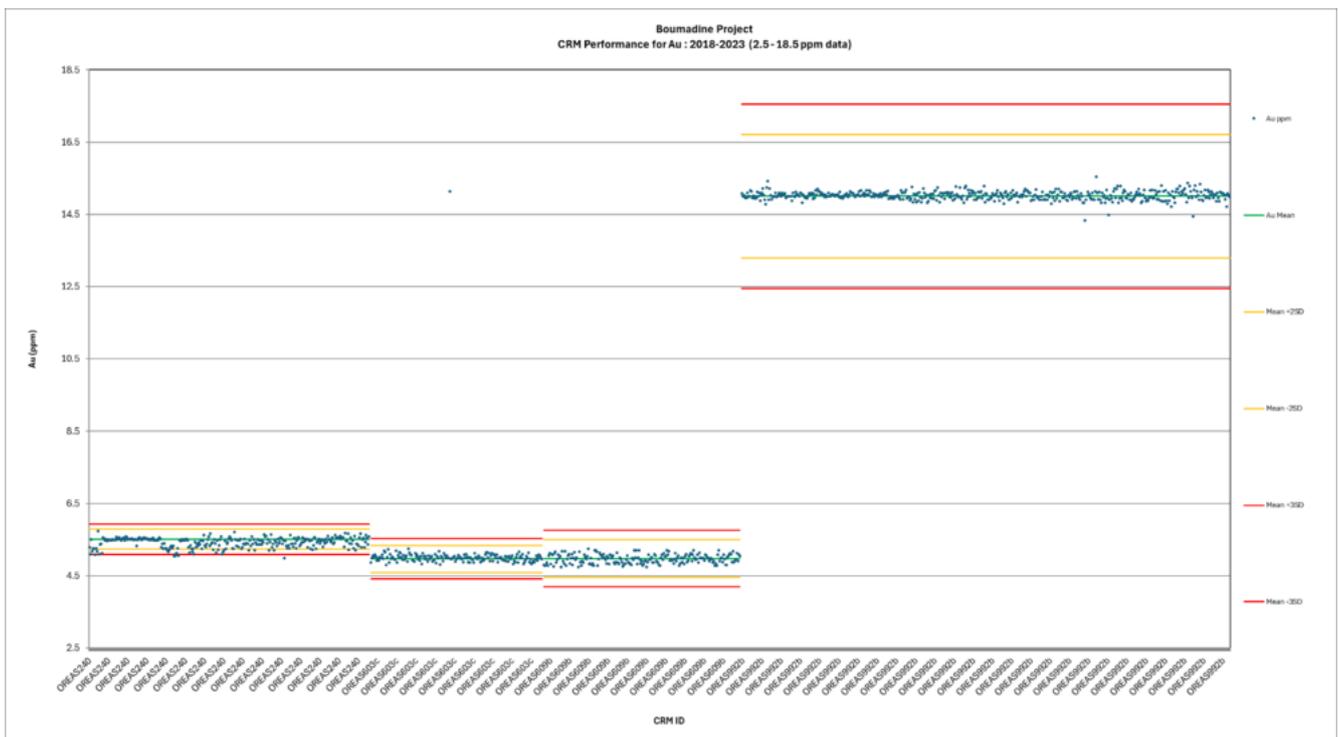
Performance of CRM Materials – 2018 to 2024

OREAS CRMs for Au (0 to 2.5ppm)



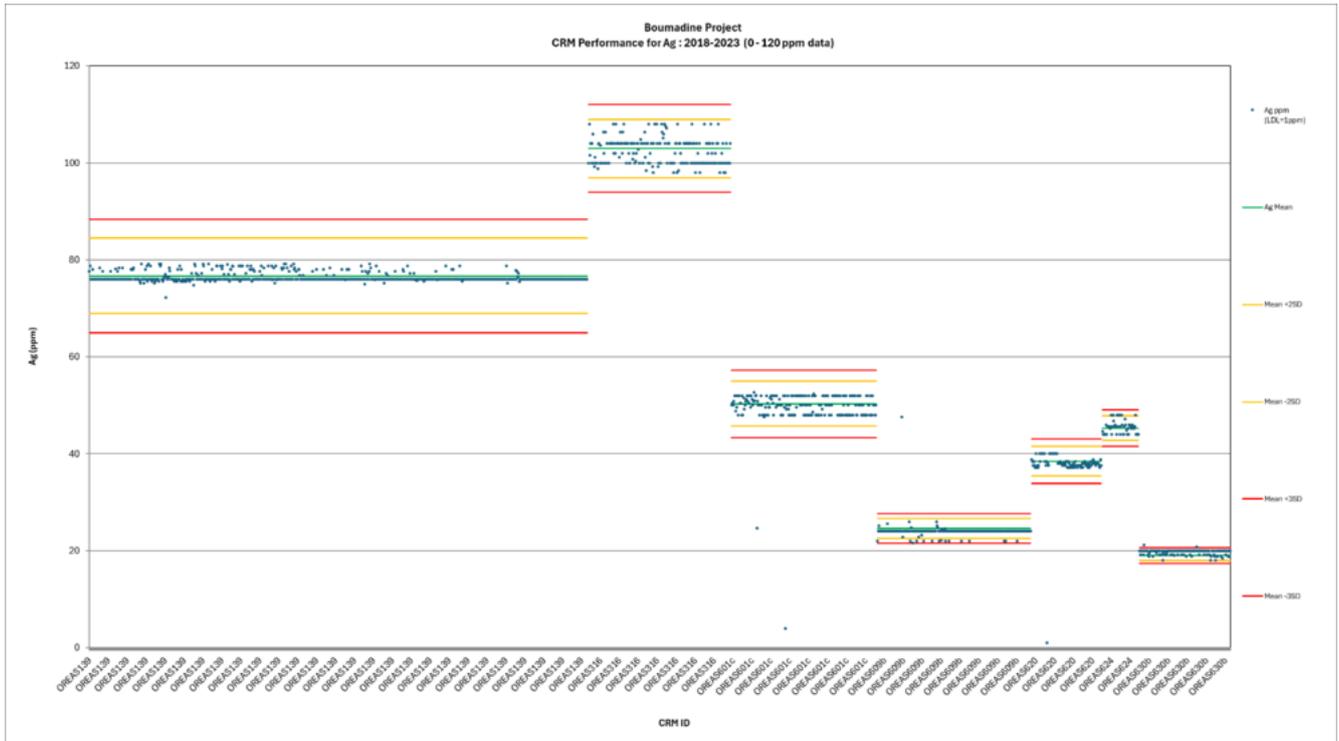
Source: P&E (2024)

OREAS CRMs for Au (2.5 to 18.5ppm)



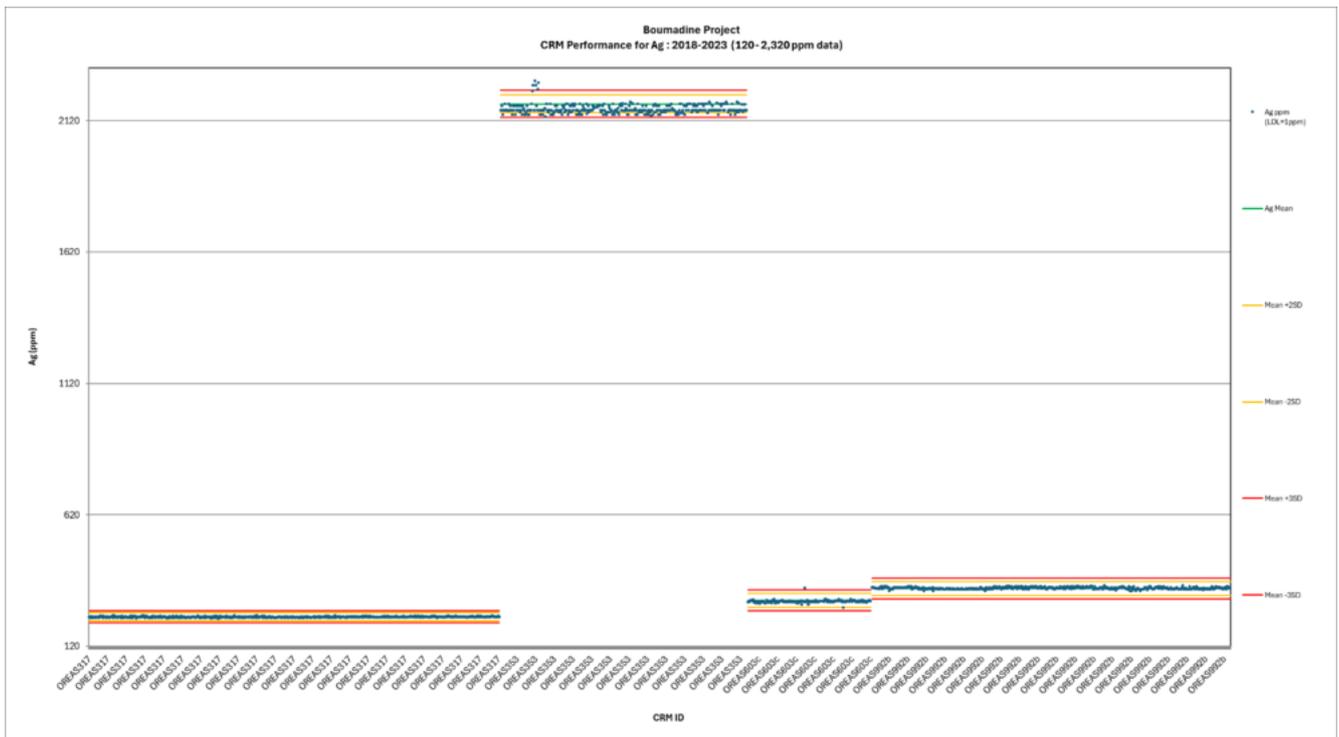
Source: P&E (2024)

OREAS CRMs for Ag (0 to 120 ppm)



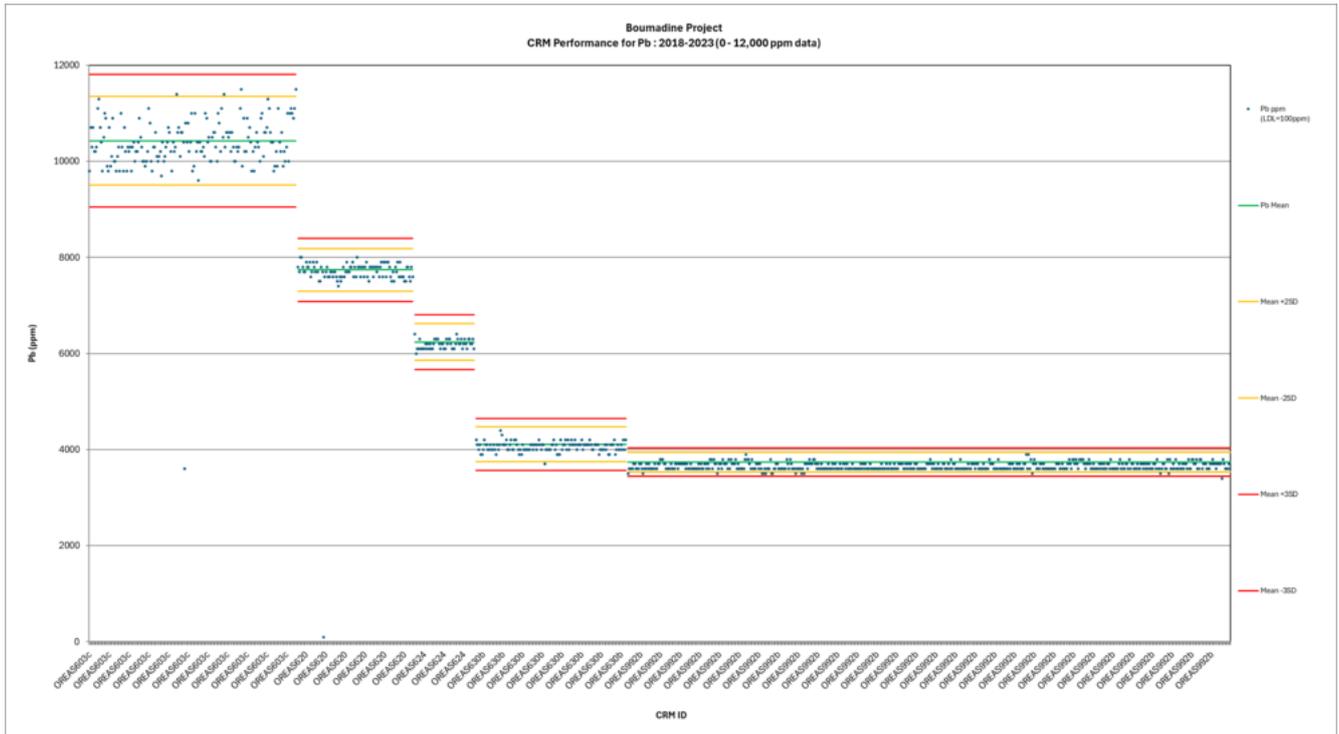
Source: P&E (2024)

OREAS CRMs for Ag (120 to 2,300 ppm)



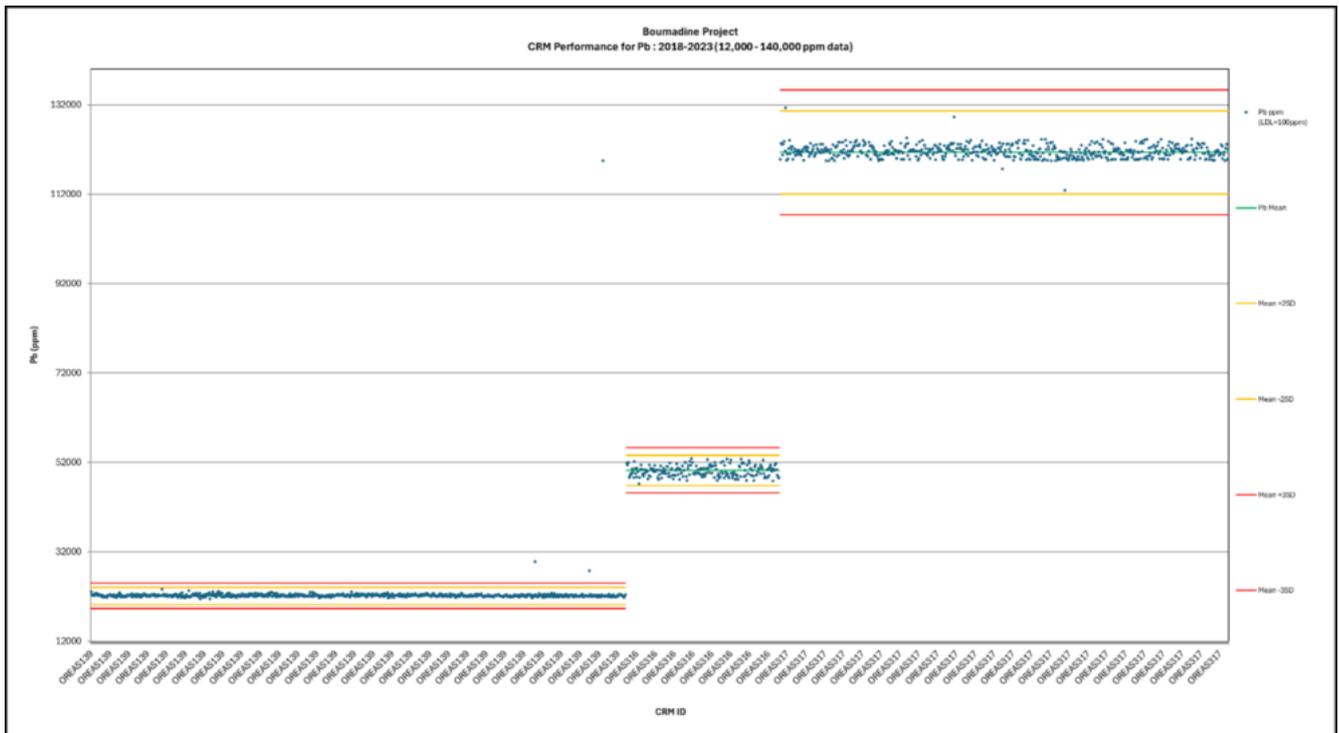
Source: P&E (2024)

OREAS CRMs for Pb (0 to 12,000 ppm)



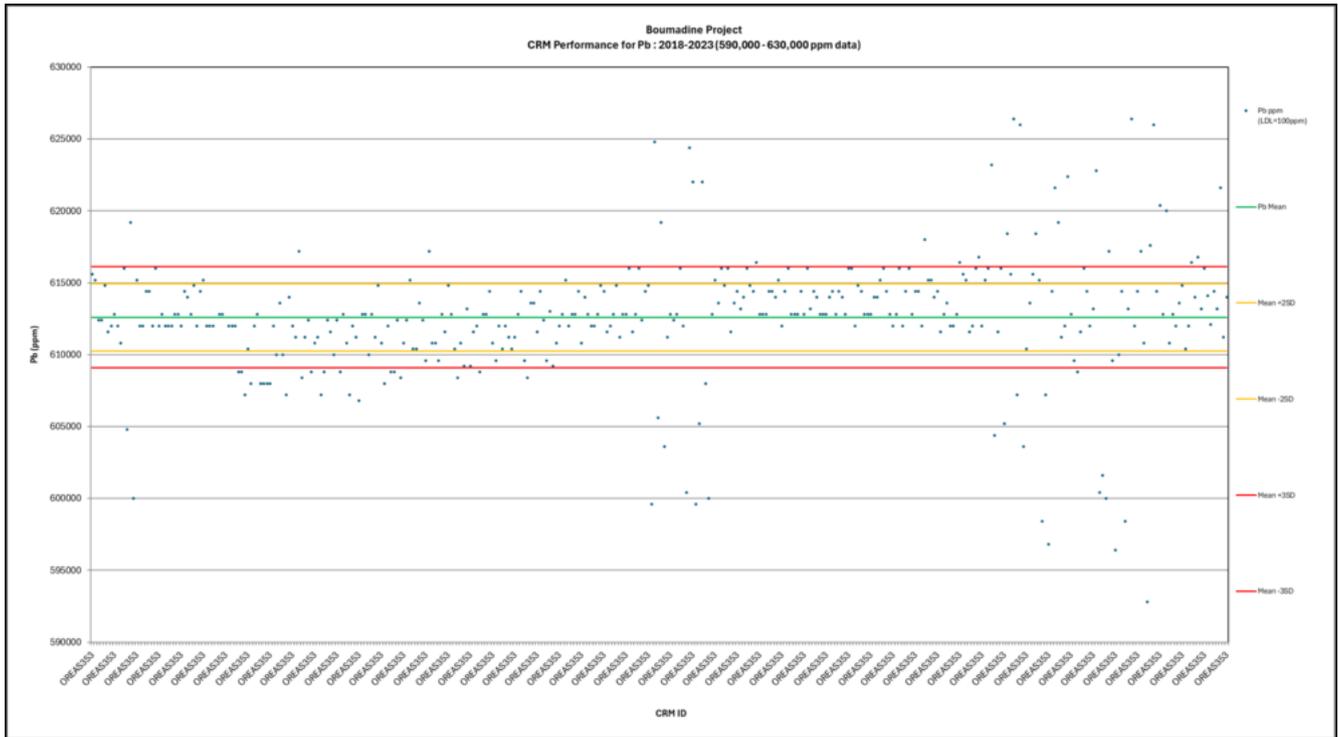
Source: P&E (2024)

OREAS CRMs for Pb (12,000 to 140,000 ppm)



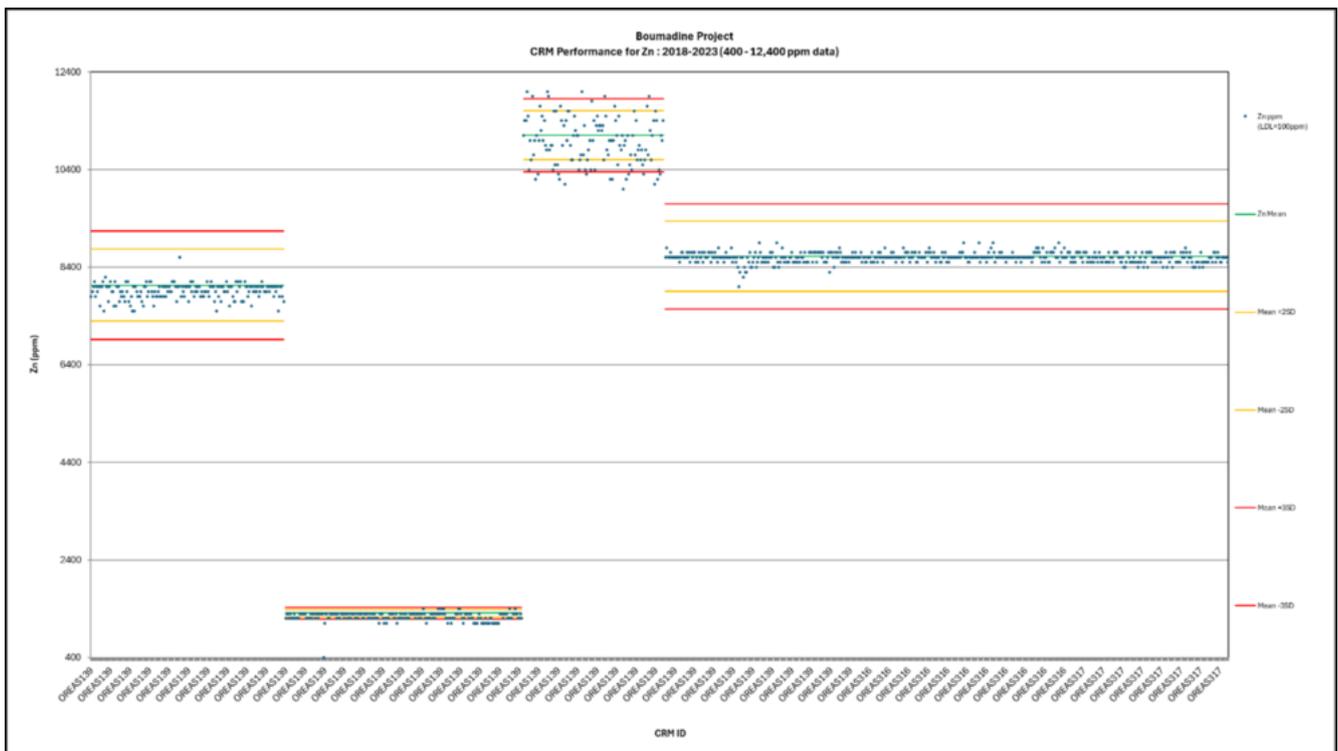
Source: P&E (2024)

OREAS CRMs for Pb (590,000 to 630,000 ppm)



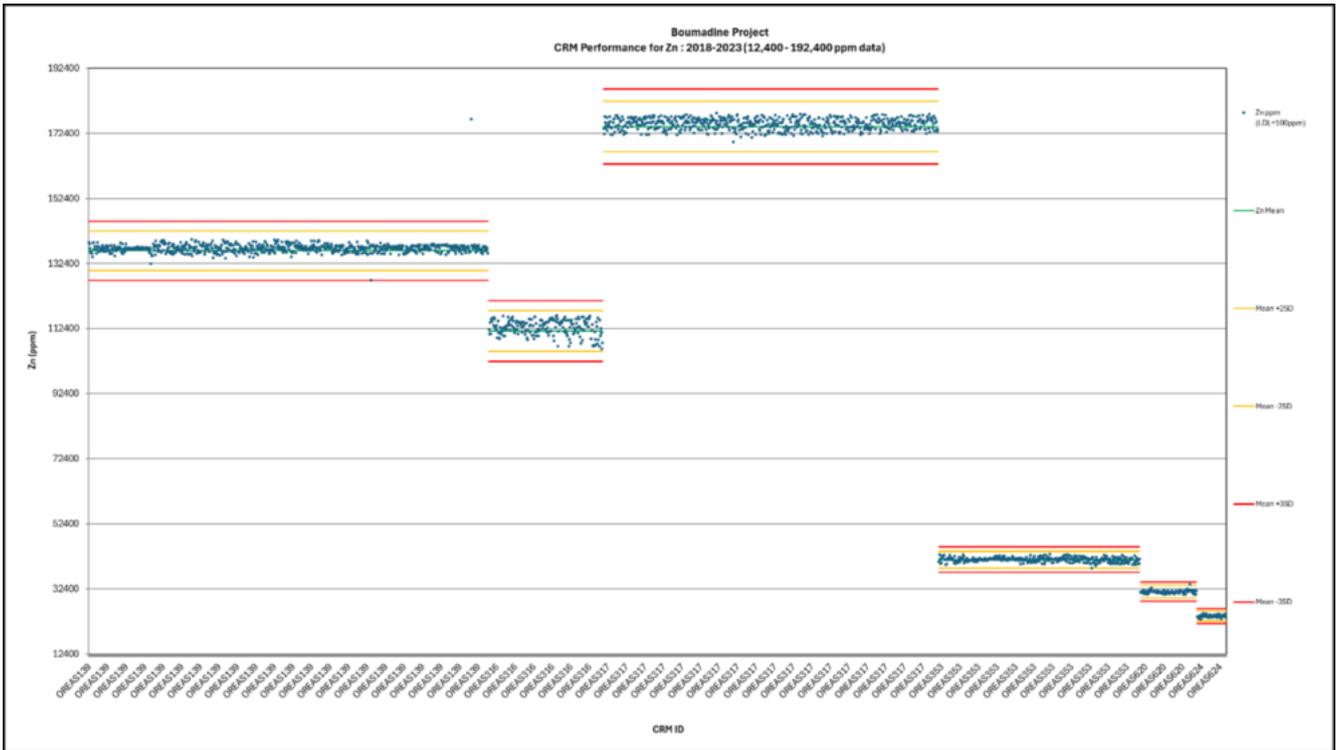
Source: P&E (2024)

OREAS CRMs for Zn (400 to 12,400 ppm)



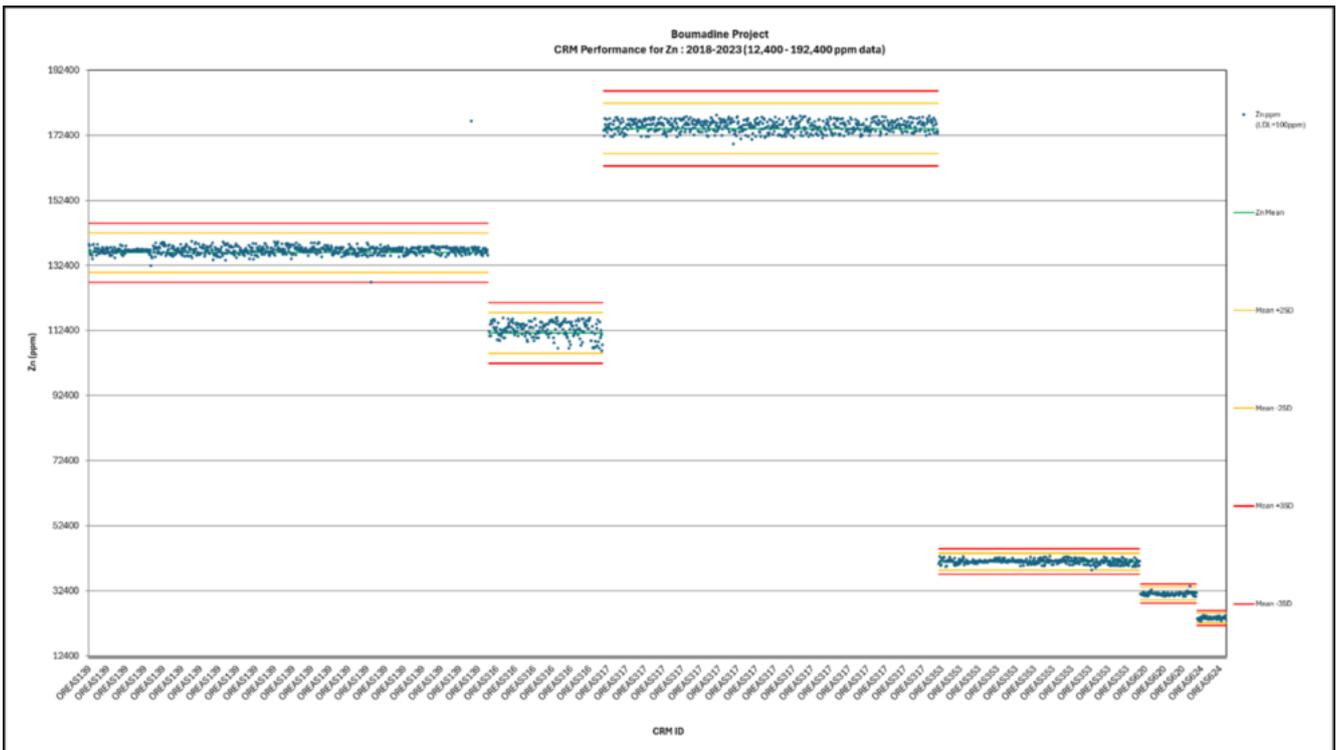
Source: P&E (2024)

OREAS CRMs for Zn (12,400 to 192,400 ppm)



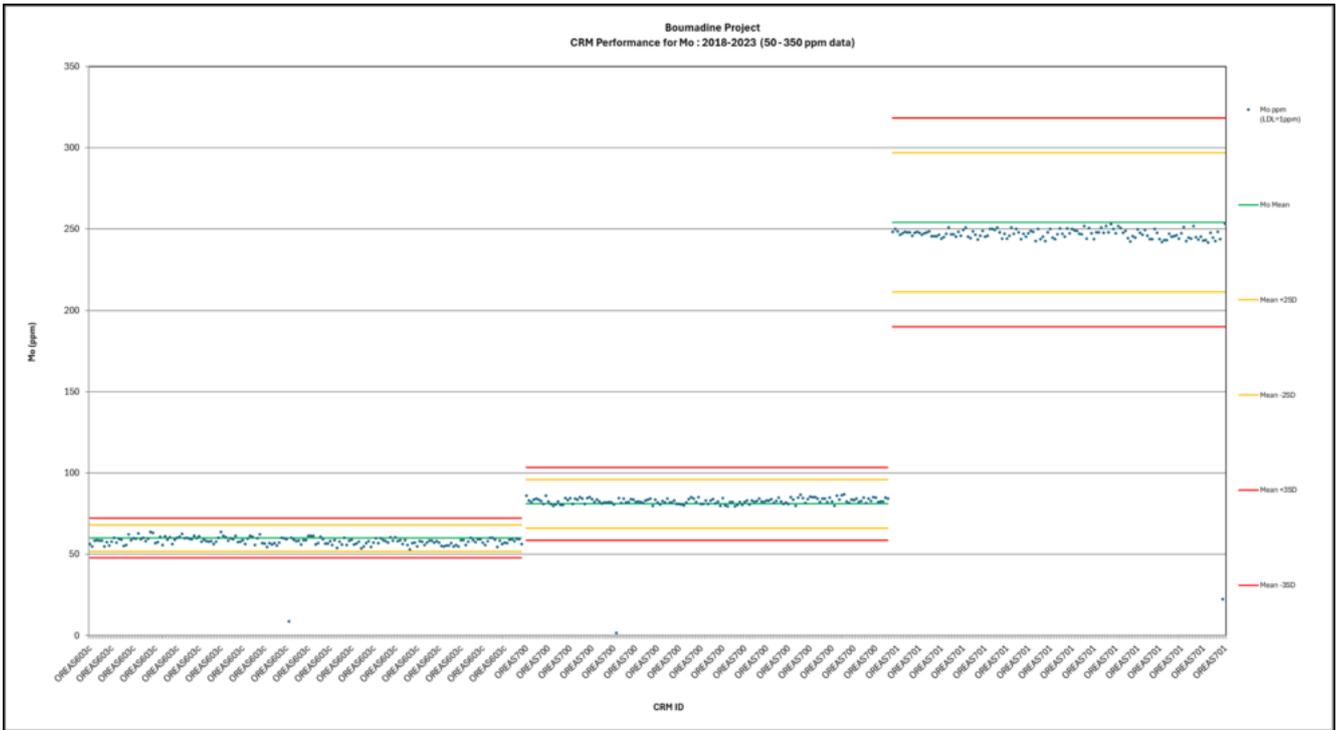
Source: P&E (2024)

OREAS CRMs for Mo (0 to 50 ppm)



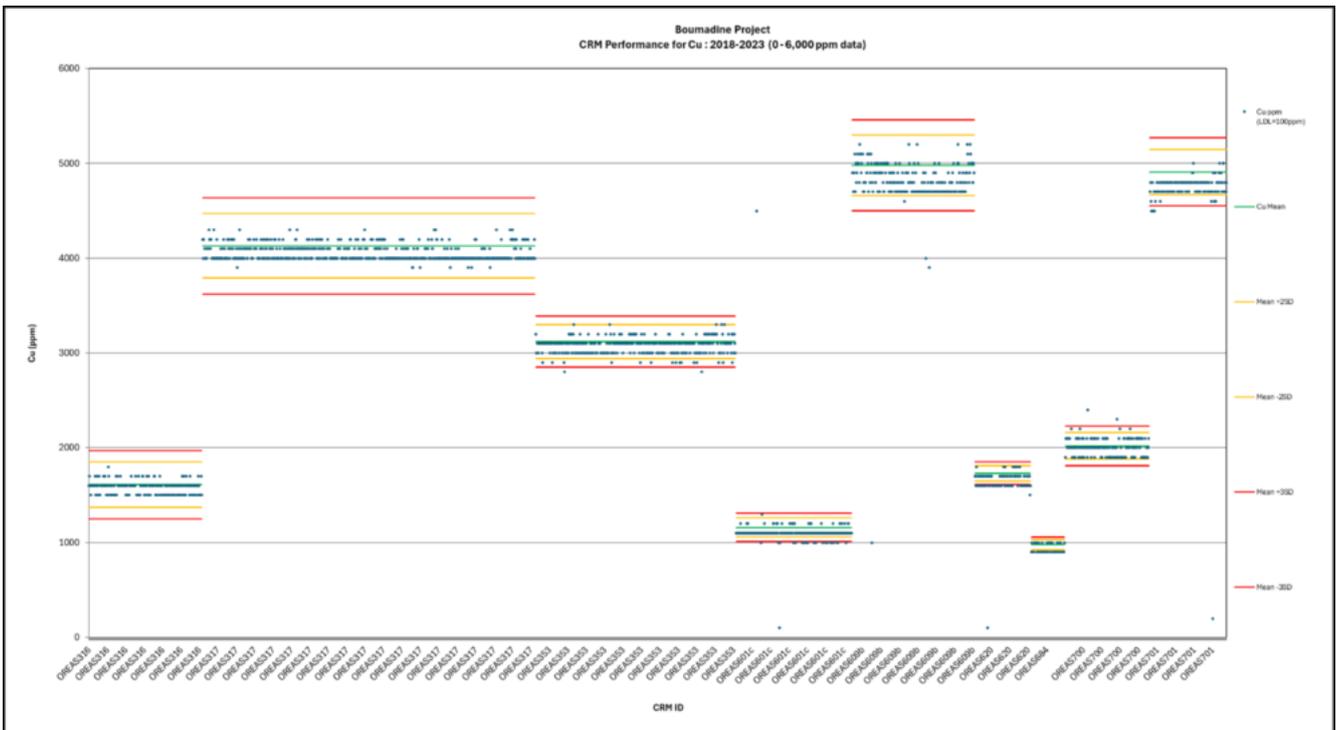
Source: P&E (2024)

OREAS CRMs for Mo (50 to 350 ppm)



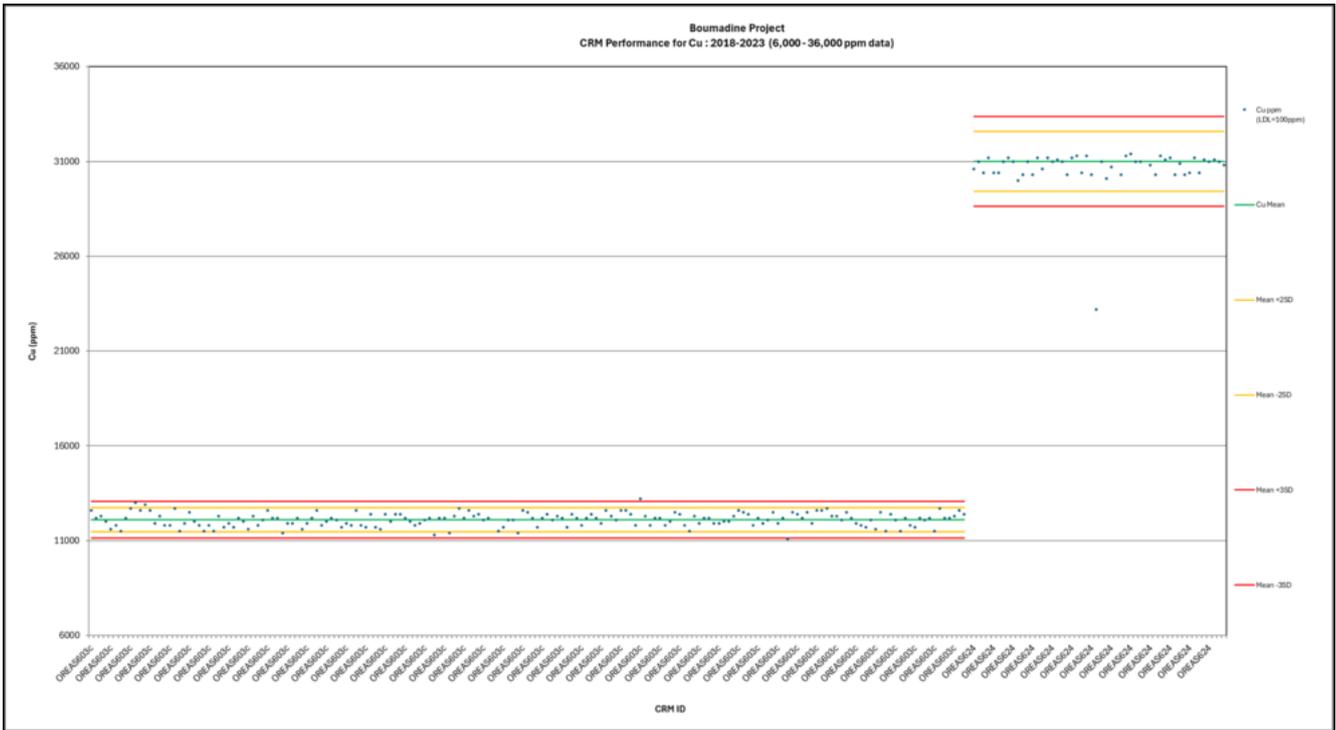
Source: P&E (2024)

OREAS CRMs for Cu (0 to 6,000 ppm)



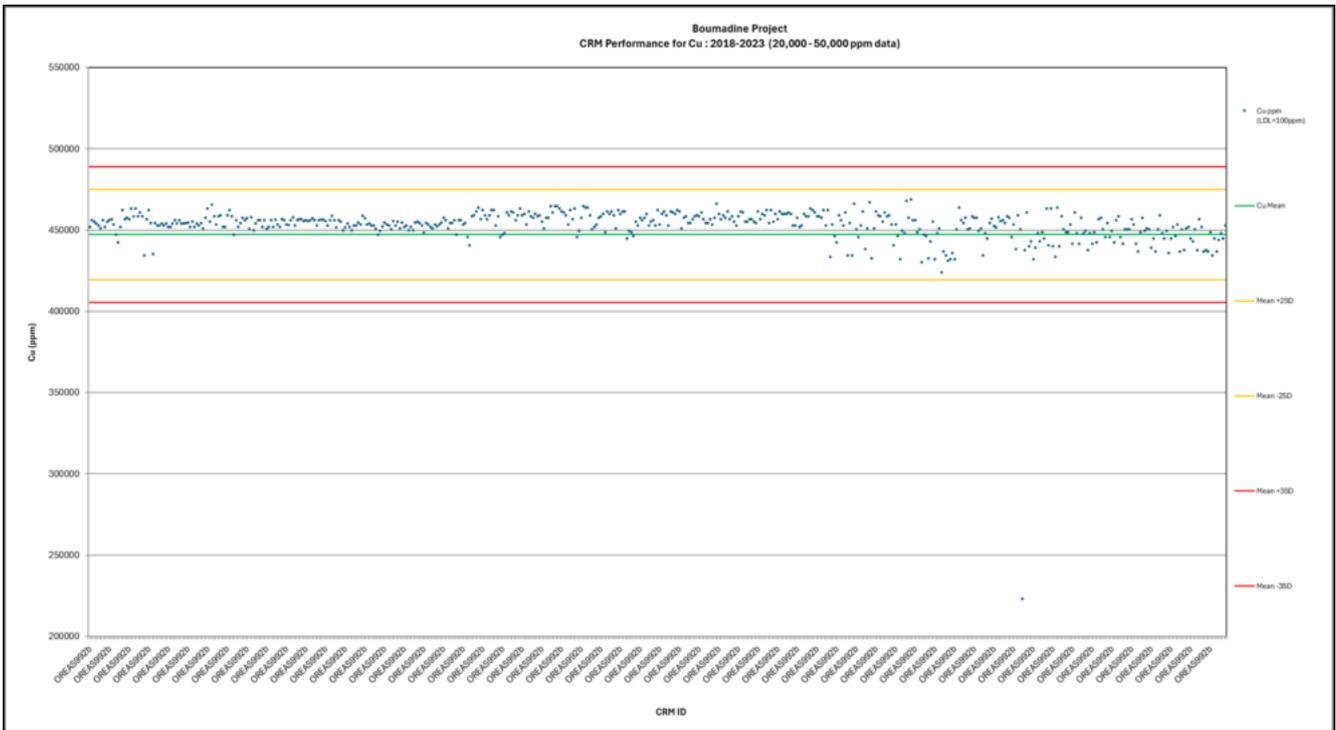
Source: P&E (2024)

OREAS CRMs for Cu (6,000 to 36,000 ppm)



Source: P&E (2024)

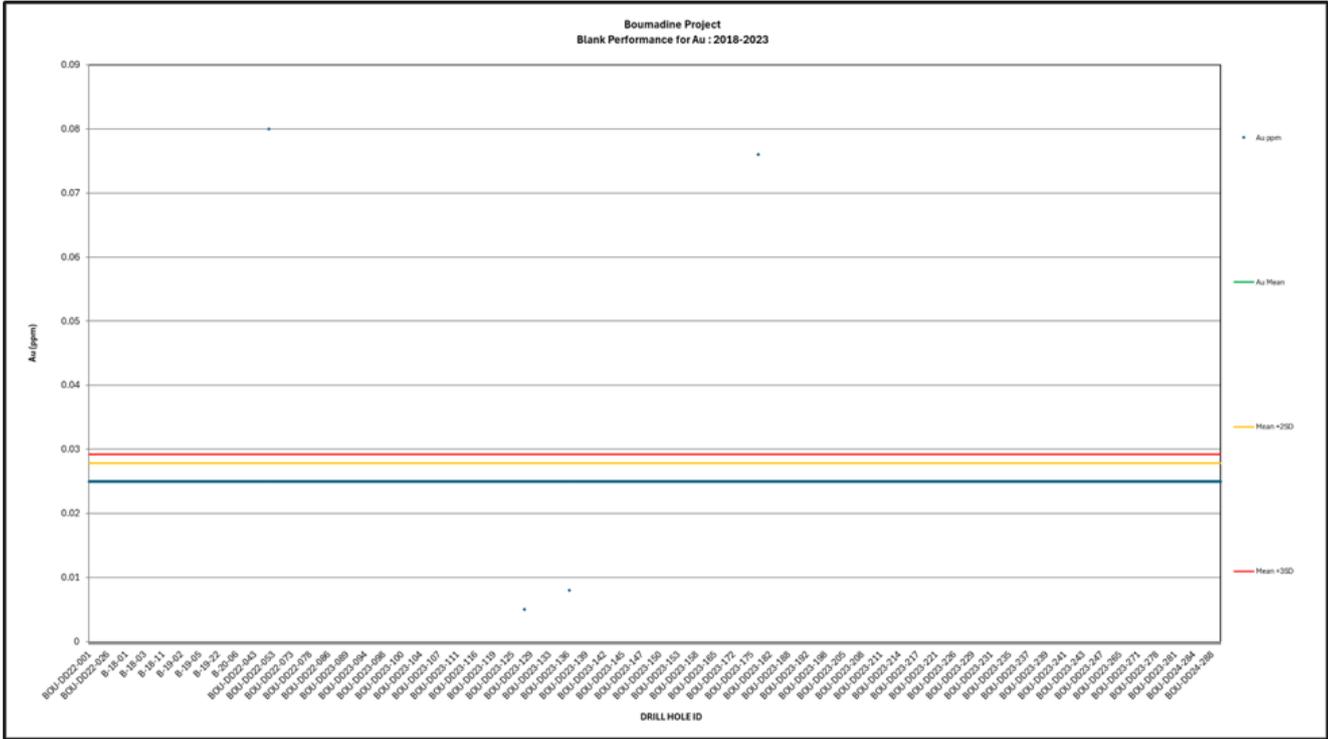
OREAS CRMs for Cu (20,000 to 50,000 ppm)



Source: P&E (2024)

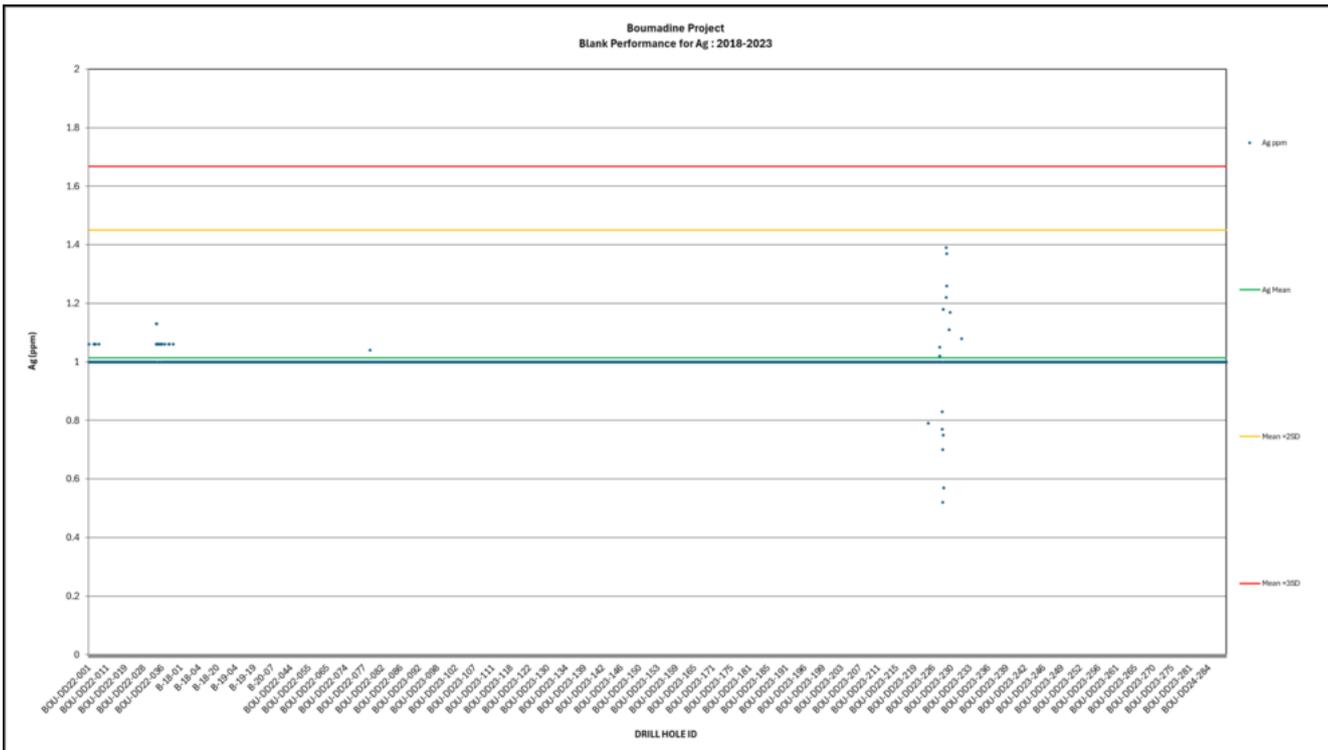
Performance of Blanks – 2018 to 2024

Performance of Blanks for Au



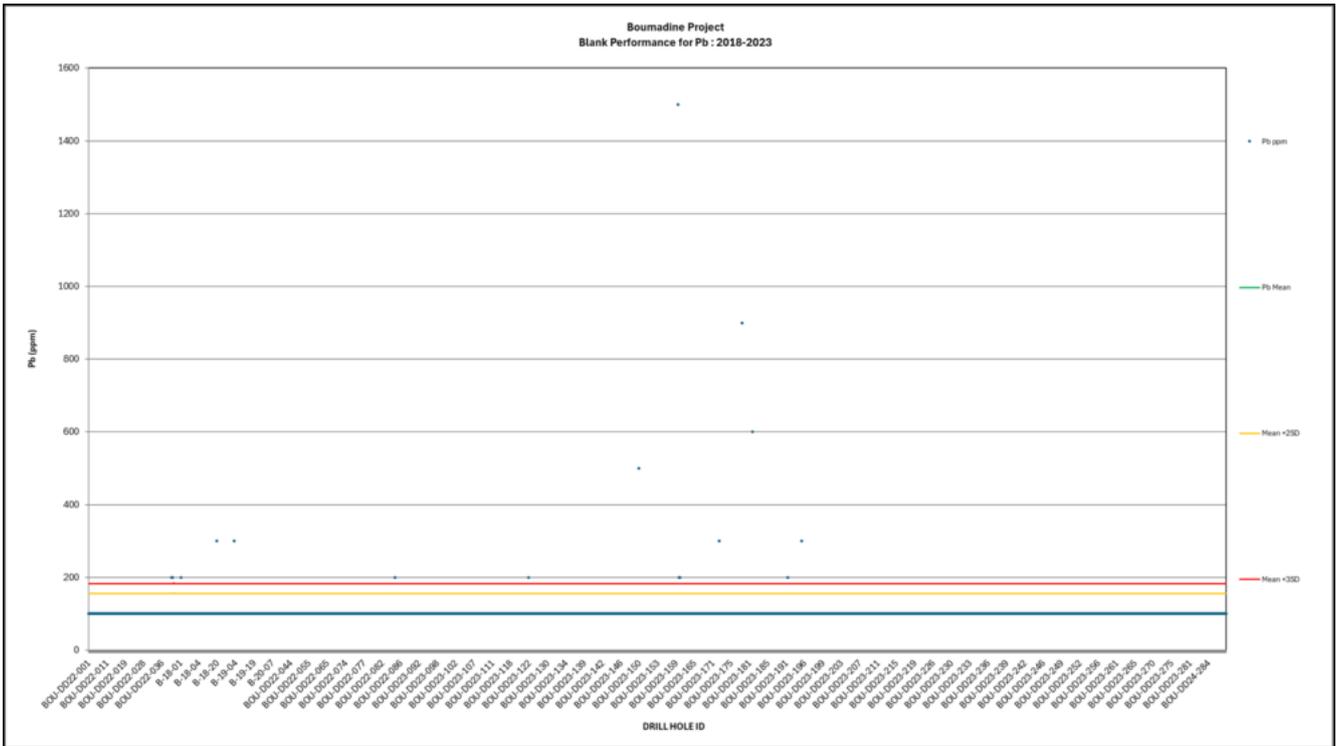
Source: P&E (2024)

Performance of Blanks for Ag



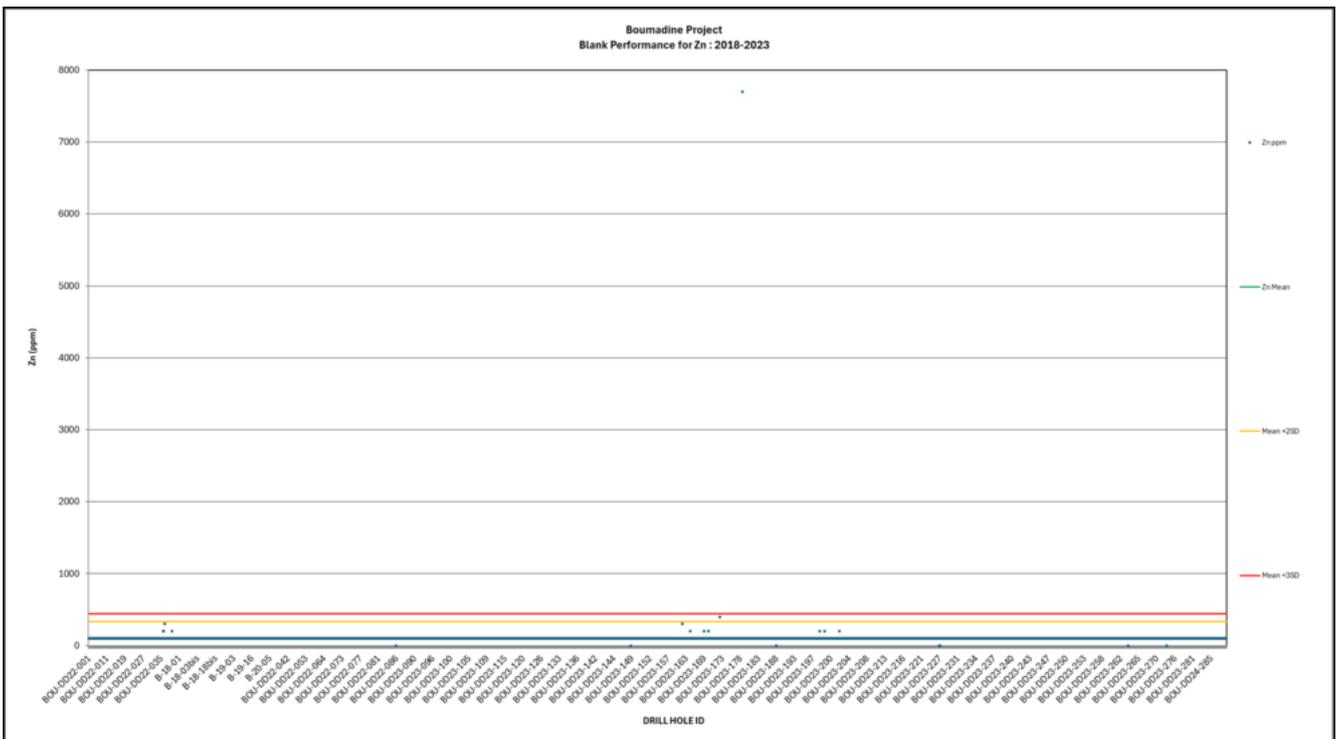
Source: P&E (2024)

Performance of Blanks for Pb



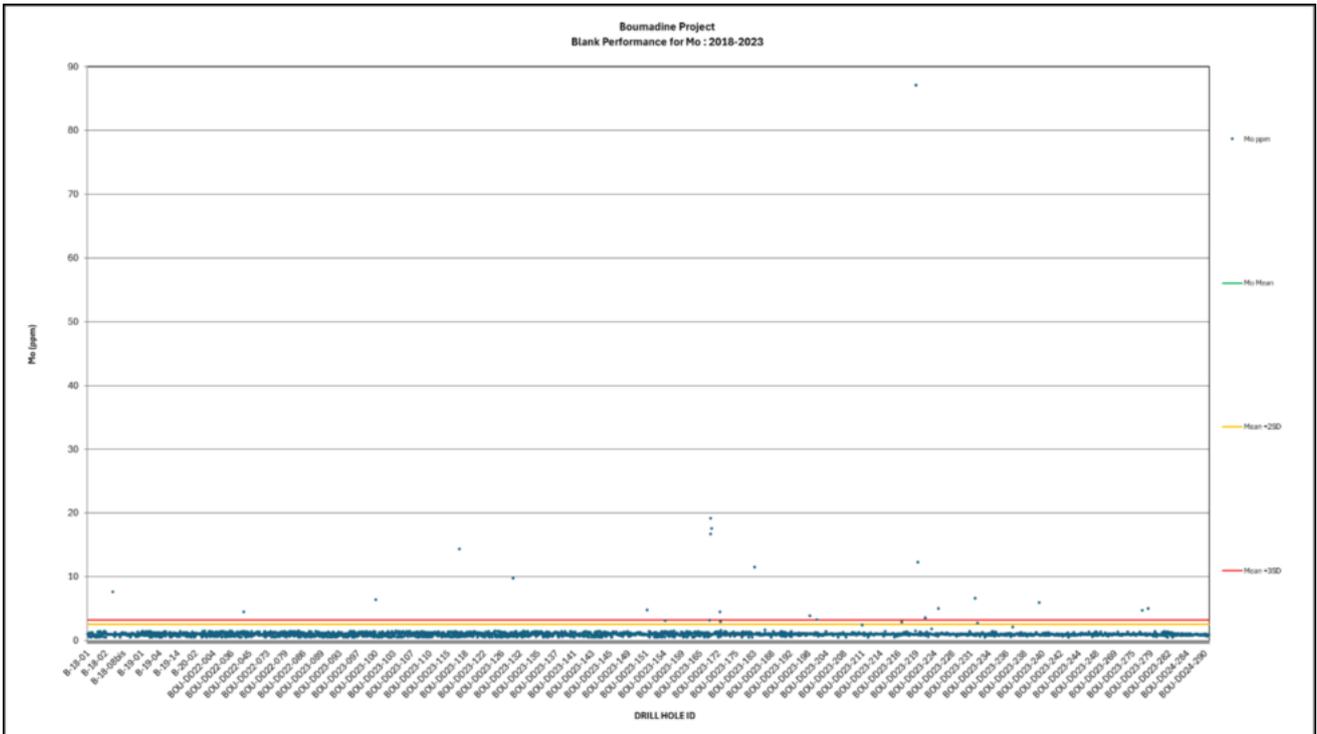
Source: P&E (2024)

Performance of Blanks for Zn



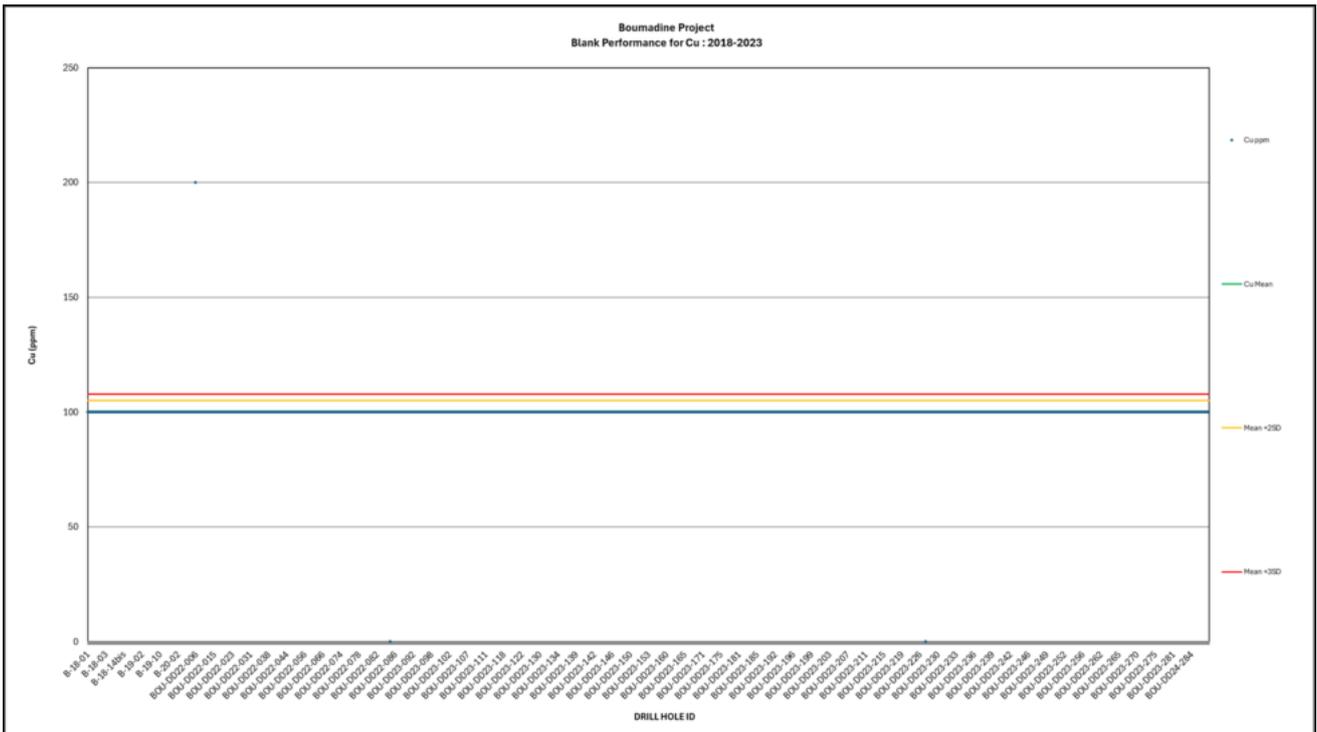
Source: P&E (2024)

Performance of Blanks for Mo



Source: P&E (2024)

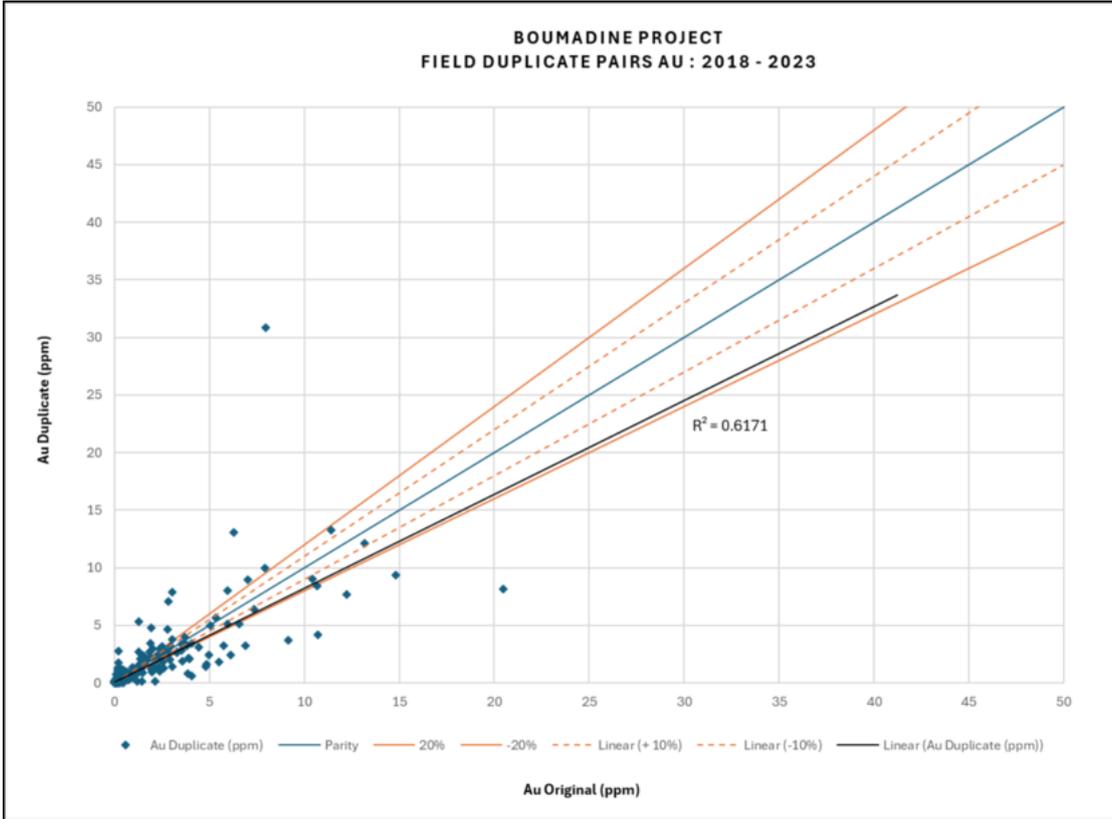
Performance of Blanks for Cu



Source: P&E (2024)

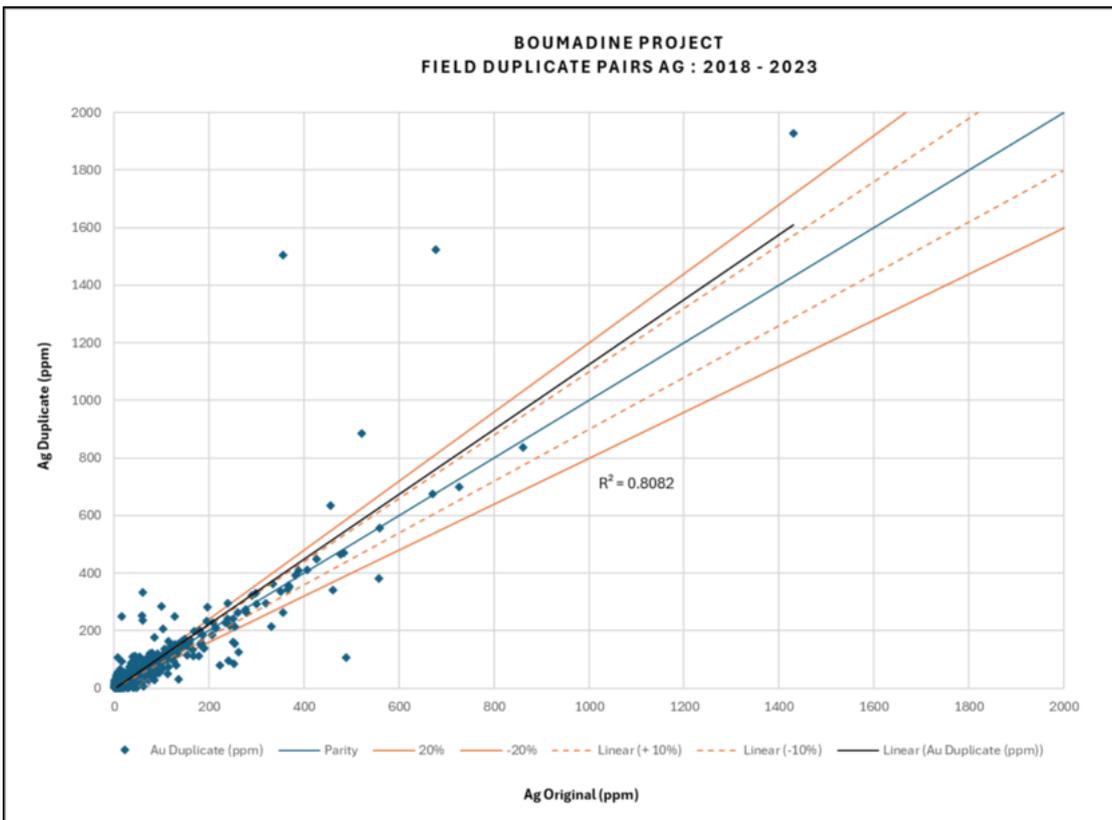
Performance of Field Duplicates – 2018 to 2024

Performance of Field Duplicates for Au



Source: P&E (2024)

Performance of Field Duplicates for Ag



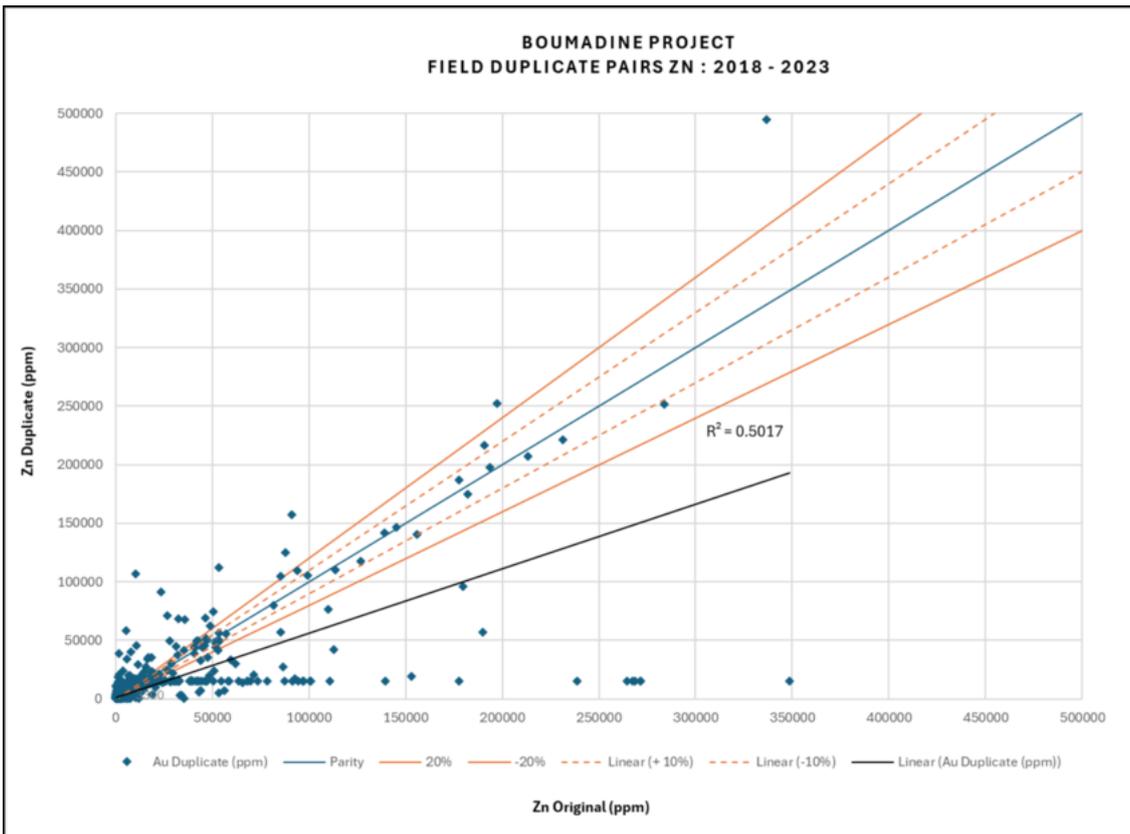
Source: P&E (2024)

Performance of Field Duplicates for Pb



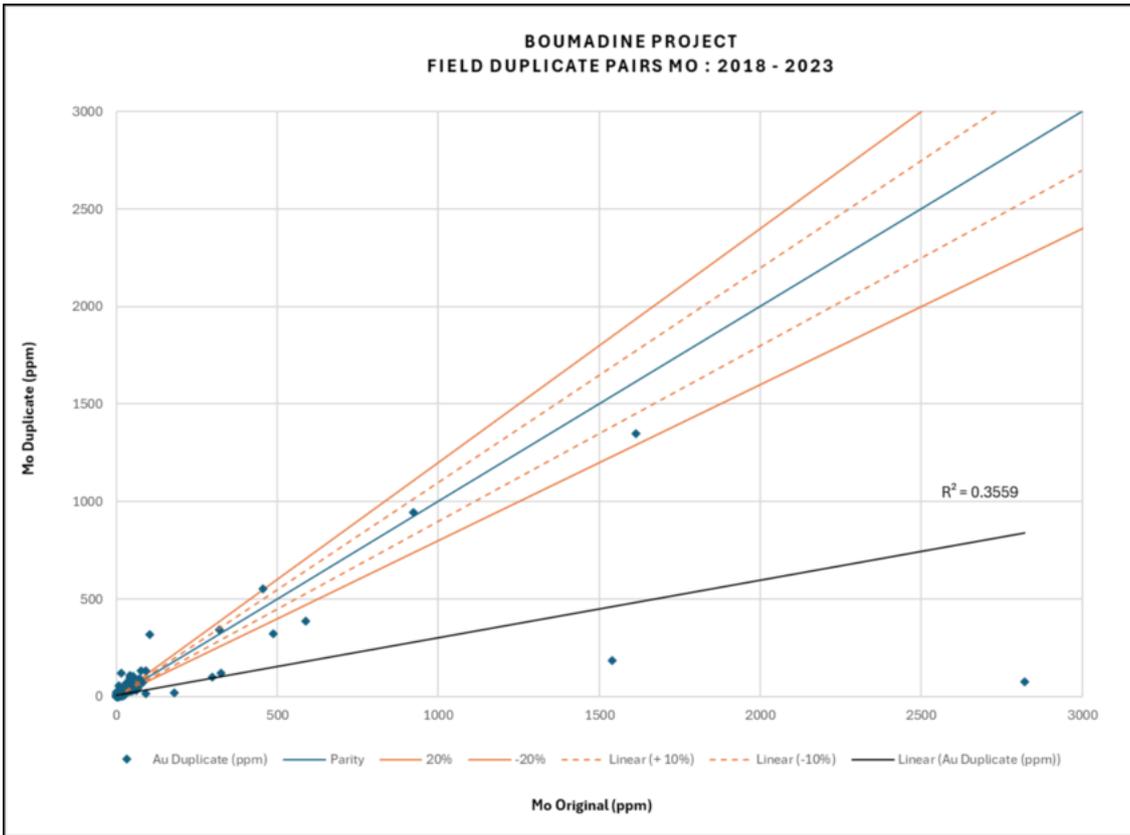
Source: P&E (2024)

Performance of Field Duplicates for Zn



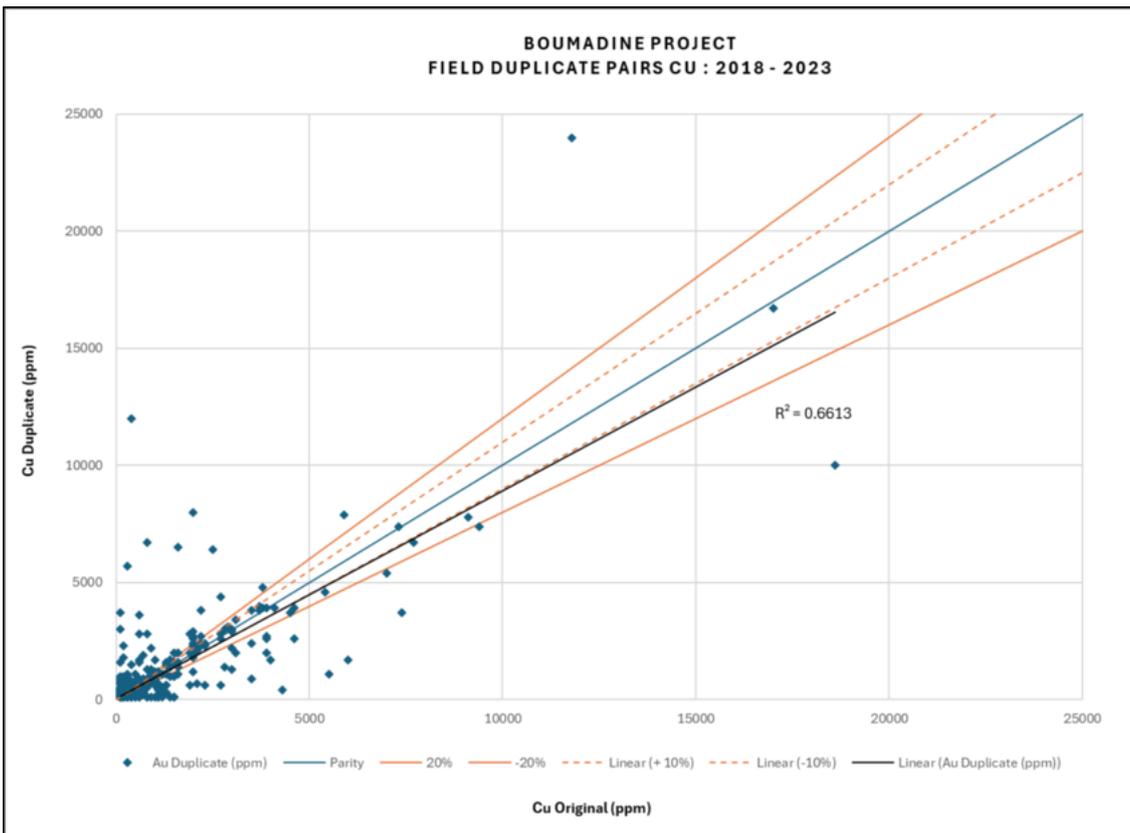
Source: P&E (2024)

Performance of Field Duplicates for Mo



Source: P&E (2024)

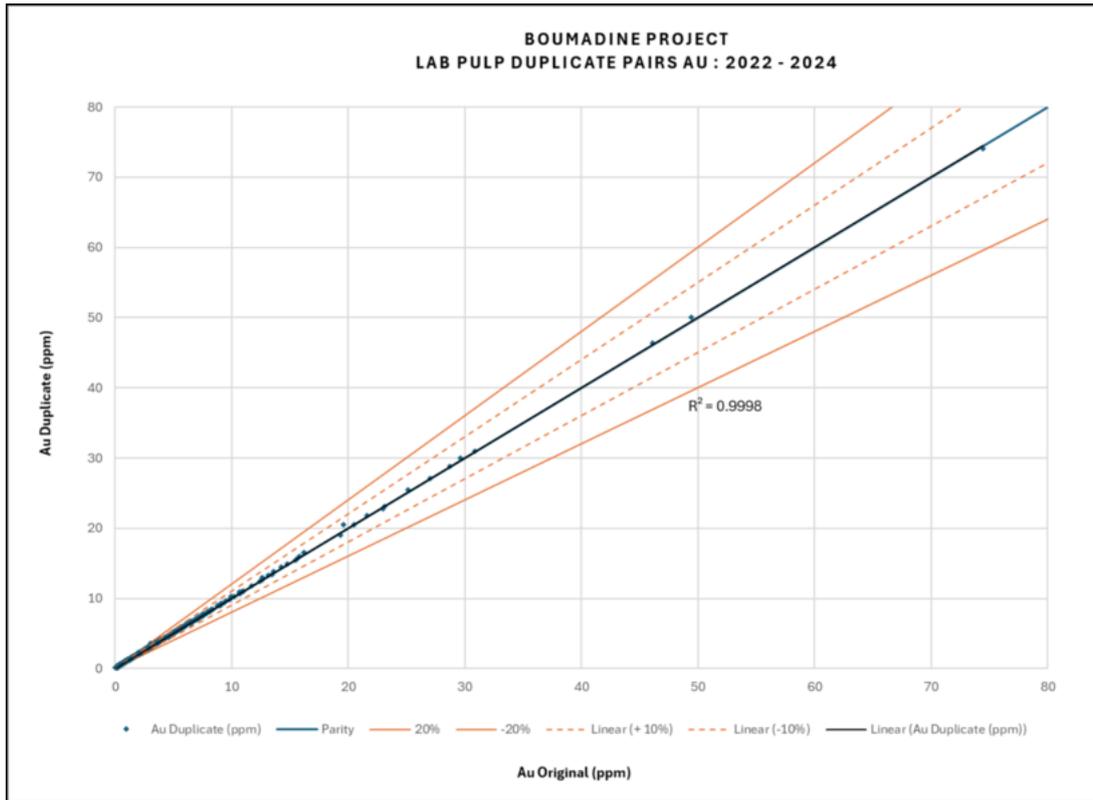
Performance of Field Duplicates for Cu



Source: P&E (2024)

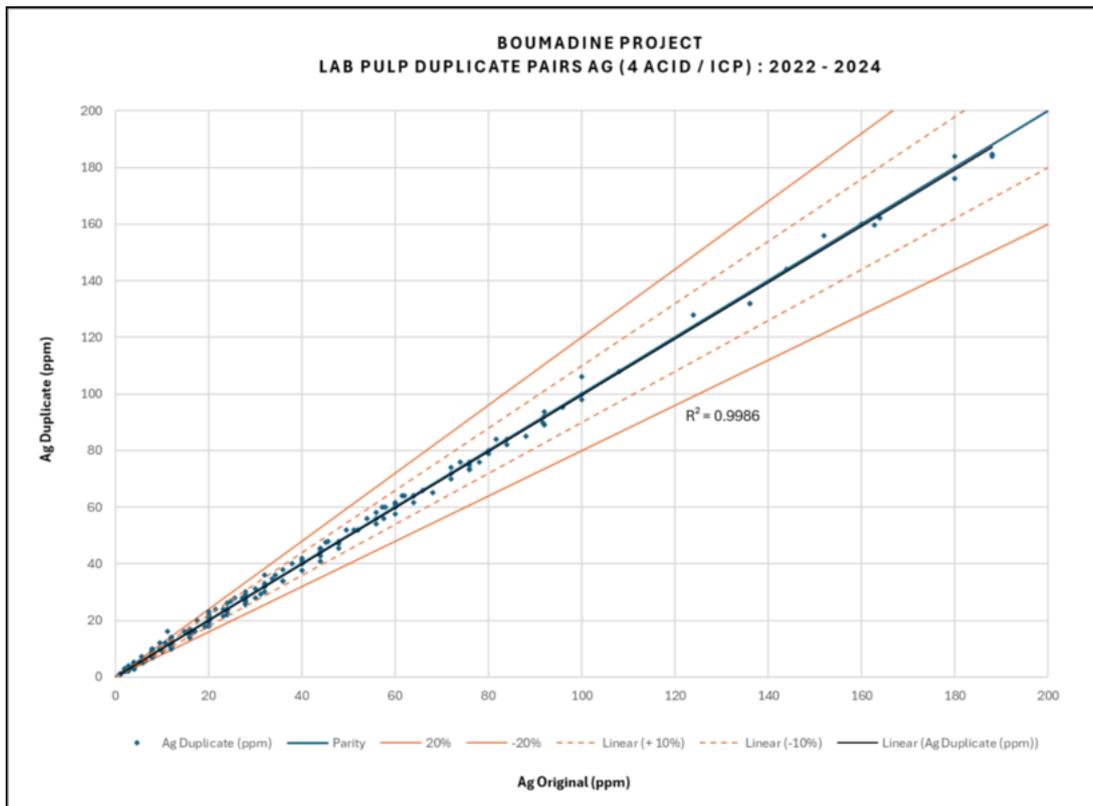
Performance of Lab Pulp Duplicates – 2018 to 2024

Performance of Lab Pulp Duplicates for Au



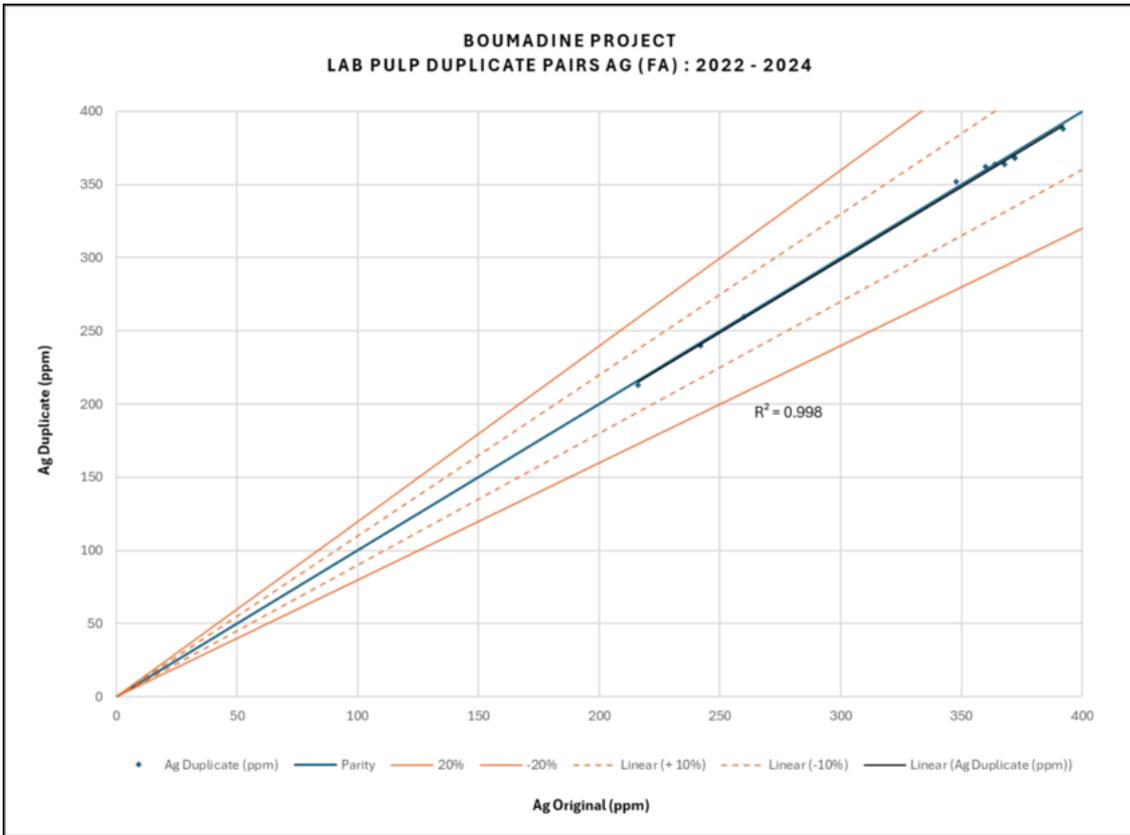
Source: P&E (2024)

Performance of Lab Pulp Duplicates for Ag (4 Acid/ICP)



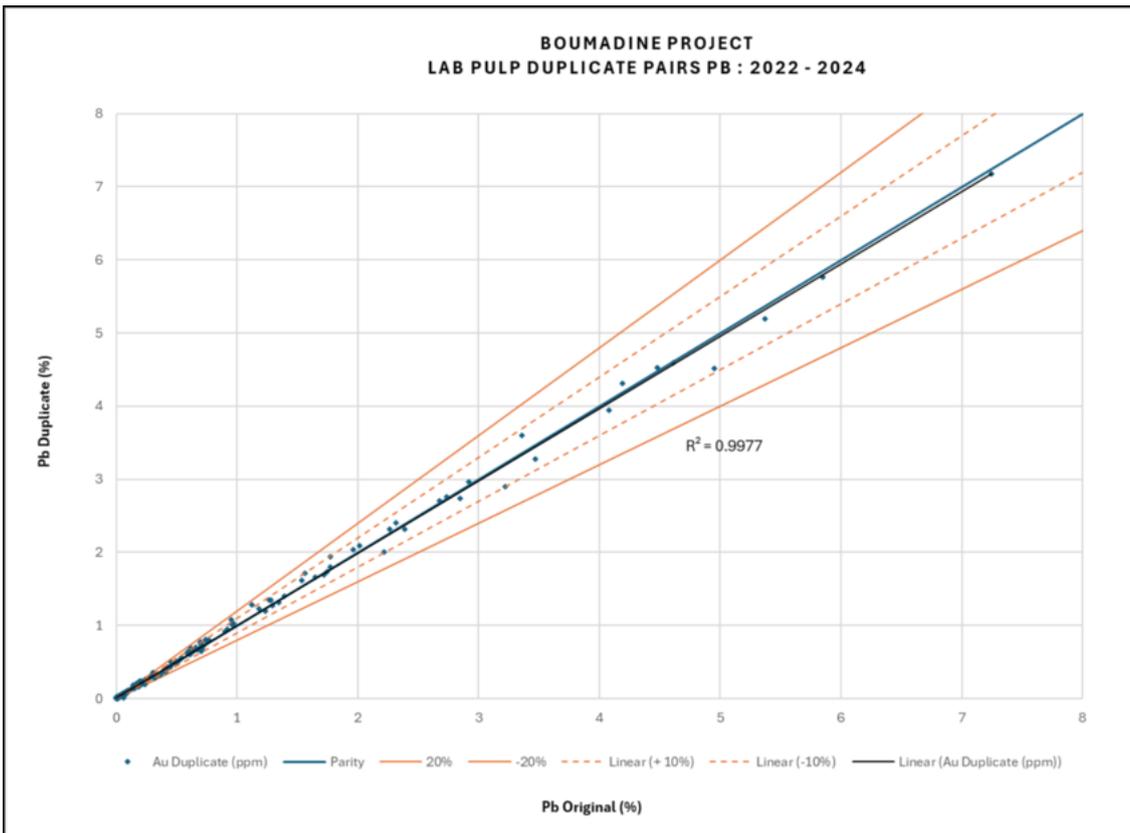
Source: P&E (2024)

Performance of Lab Pulp Duplicates for Ag (Fire Assay)



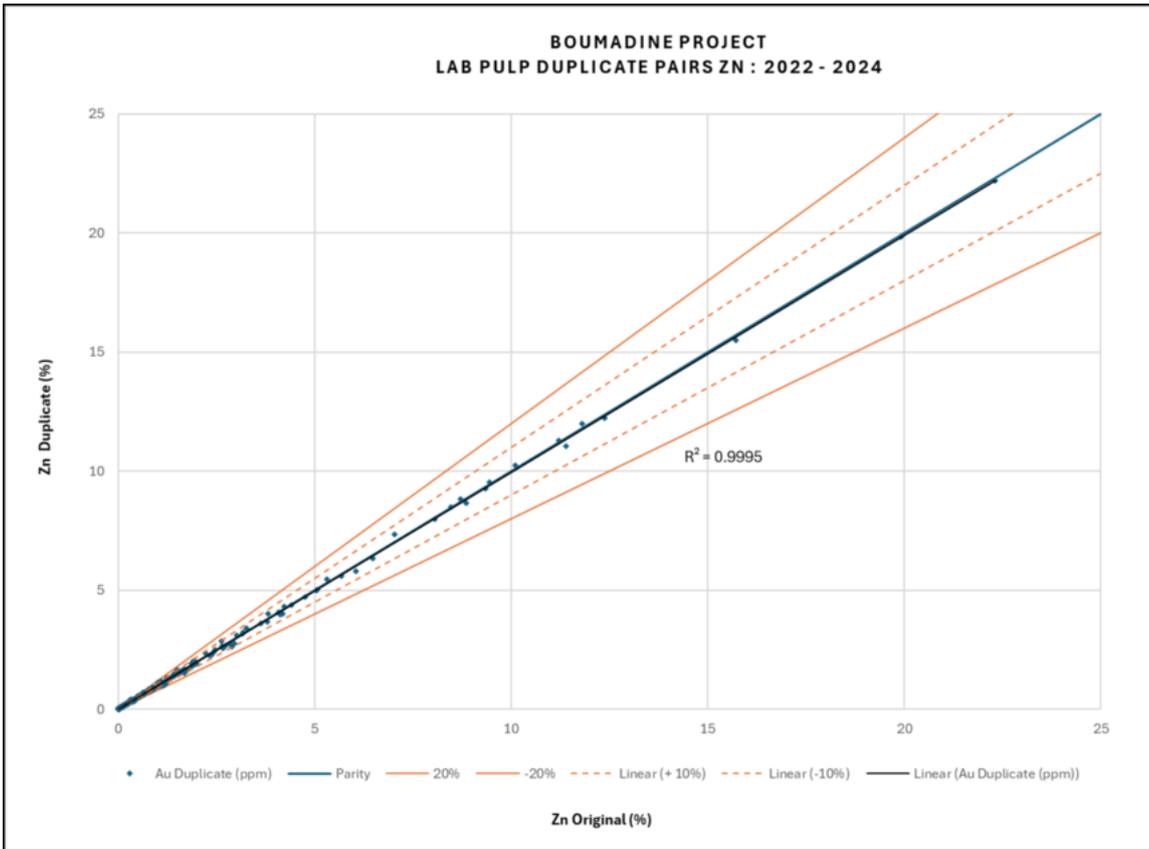
Source: P&E (2024)

Performance of Lab Pulp Duplicates for Pb



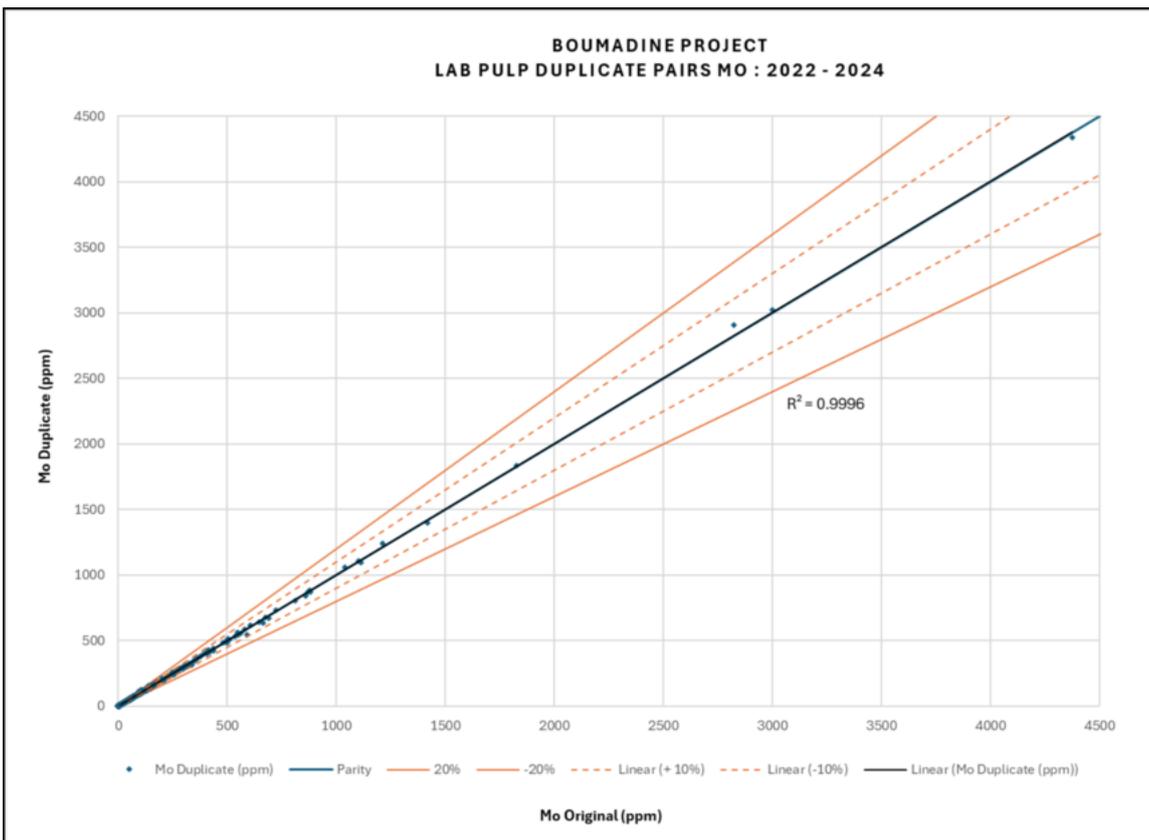
Source: P&E (2024)

Performance of Lab Pulp Duplicates for Zn



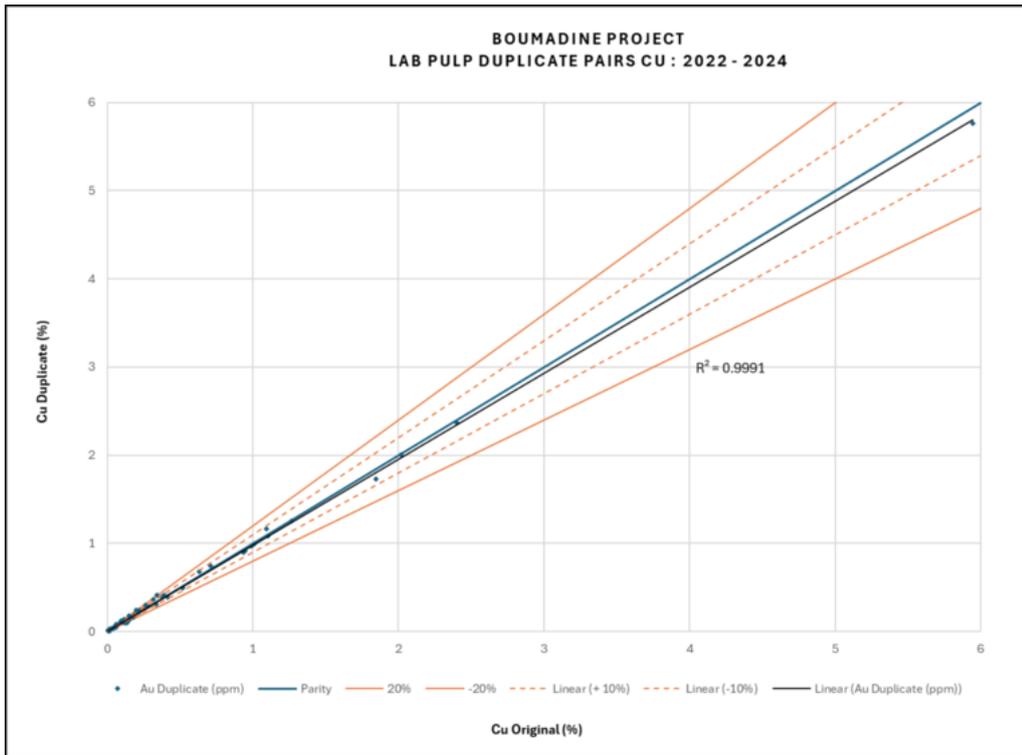
Source: P&E (2024)

Performance of Lab Pulp Duplicates for Mo



Source: P&E (2024)

Performance of Lab Pulp Duplicates for Cu



Source: P&E (2024)