Lycopodium

Newcore Gold Ltd.

Enchi Gold Project

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NEWCORE GOLD

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DISCLAIMER

This report has been prepared for Newcore Gold Ltd (Newcore) by Lycopodium (Americas) Ltd (Lycopodium) as an independent consultant and is based in part on information furnished by Newcore and in part on information not within the control of either Newcore or Lycopodium. While it is believed that the information, conclusions and recommendations will be reliable under the conditions and subject to the limitations set forward herein, Lycopodium does not guarantee their accuracy. The use of this report and the information contained herein shall be at the user's sole risk, regardless of any fault or negligence of Lycopodium.

1.0 SUMMARY

1.1 Introduction

Newcore Gold Ltd. (Newcore or the Company) engaged Lycopodium (Americas) Ltd. (Lycopodium) to prepare a Pre-Feasibility Study (PFS) for the Enchi Gold Project (Enchi, the Project, or the Property). The PFS has been completed in accordance with the Canadian disclosure requirements of National Instrument 43-101 (NI 43-101) and the associated requirements of Form 43-101F1.

To support the preparation of the PFS, Newcore retained a number of independent engineering and consulting firms to provide specialized technical input. The roles and responsibilities of the consultants contributing to this report are summarized below:

- Lycopodium managed and coordinated the development of this report, developed PFS-level designs for the process plant and general on-site infrastructure, developed the consolidated cost estimates and reviewed and signed off on the financial model that was prepared by the Company.
- Knight Piésold Consulting (KP) supervised geotechnical investigations, provided geotechnical recommendations, conducted environmental studies, and developed the PFS-level design and cost estimate of the tailings and water management facilities.
- DRA Americas Inc. (DRA) developed the mineral resource estimate.
- Fuse Advisors Inc. (Part of SLR Consulting) prepared the mineral reserve estimate, mining methods and mining capital cost and operating cost estimates.

1.2 History

The exploration activities across the entire Project area date back to the 1800s, with activities completed sporadically and by various individuals as well as companies.

Alluvial and bedrock gold were prospected and exploited by several generations of galamsey (local artisanal gold miner) workings to the present day. European companies explored, developed, and mined in several phases since 1900. The result is that erratic gold in quartz vein mineralization was ‘opened up’ in numerous pits, shafts, and drives, notably at the Sewum, Tokosea, Alatakrom, Achimfo, Nkwanta, and Kojina Hill prospects. Only the historic Sewum and Tokosea mines appear to have any significant development and production history although this is poorly recorded. Since the 1940s, artisanal mining activities have continued in the area on a very limited scale. No modern commercial-scale mining has been carried out on the Project.

Modern exploration in the form of soil sampling, surface trenching, rotary air blast (RAB) drilling, reverse circulation (RC) drilling, and diamond drilling has been completed by various operators, including EQ Resources in the late 1980s, Leo Shield from 1995 to 1998, Red Back from 2003 to 2006, Edgewater from 2011 to 2012, and Newcore (previously known as Pinecrest) from 2014 to present.

1.2.1 Previous Resources and Reserves

Resource estimations were previously completed on the Project in 2011, 2015, 2020, 2021, and 2023. The most recent historic Mineral Resource Estimate (completed for Newcore by Todd McCracken from BBA Consultants and dated effective January 25, 2023) is summarized in Table 1.2.1. No Mineral Reserves were reported for the Project at the time.

Table 1.2.1 Summary of 2023 Mineral Resources (McCracken and Meadow Smith, 2023)

Classification	Zone	Open Pit (OP)/ Underground (UG)	Tonnes	Au g/t	Au Oz
Indicated	Sewum	OP	20,925,000	0.48	323,300
	Boin	OP	13,020,000	0.62	258,200
	Nyam	OP	7,791,000	0.65	162,000
	Total	**OP**	**41,736,000**	**0.55**	**743,500**
Inferred	Sewum	OP	21,154,000	0.47	317,600
		UG	644,000	2.68	55,500
	Boin	OP	15,884,000	0.68	349,600
	Nyam	OP	1,852,000	0.68	40,600
		UG	829,000	2.41	64,000
	Kwakyekrom	OP	3,970,000	0.64	81,000
		UG	274,000	1.86	16,300
	Tokosea	OP	1,949,000	0.75	46,900
	Total	**OP/UG**	**46,556,000**	**0.65**	**972,000**

Cut-offs varied between 0.14 and 0.27 g/t Au for open-pit resources; underground resources used a cut-off grade of 1.50 g/t Au.

Newcore considers the previous estimate to be relevant as they provide an indication of the potential of the Project; however, the Qualified Person (QP) has not done sufficient work to classify these previous estimates as current mineral resources or reserves, and Newcore does not consider these estimates as current mineral resources or reserves.

1.3 Geology and Mineralization

The Project is located in southwestern Ghana, in a region well known for gold mineralization which hosts numerous historical and current operating mines along strike. Located to the southwest of Enchi is the Afema gold project in Côte d'Ivoire (Turaco Gold) and located to the northeast of Enchi are the operating Chirano and Bibiani gold mines (Asante Gold Corporation). In 2021, Ghana became the largest gold producer in Africa and as of 2025 continued to hold that position with approximately 187.3 tonnes of gold produced (www.gold.org). The Project covers a 40 km strike length of the Bibiani Shear Zone along the eastern margin of the Sefwi Belt stretching from the Côte d'Ivoire border in the southwest to neighbouring claims to the northeast. The Bibiani Shear is known to host significantly large lode-gold deposits such as Bibiani and Chirano which are operating mines located to the northeast of Enchi.

The Project is situated on the contact between the Sefwi Belt to the west and the Kumasi Basin to the east. The Sefwi Belt is dominated by mafic volcanics, metasediments, and intrusive granitoids. The Kumasi Basin contains wide basins of marine clastic sediments. All the rocks of the region have been extensively metamorphosed to greenschist facies.

Extensive faulting, on local and regional scales, occurs along the margins of the volcanic- sedimentary belts. These northeast-trending structures are fundamentally important in the development of gold deposits in the region. The major shear system within the Project area is located at the boundary of the Sefwi Belt and the Kumasi Basin and is called the Bibiani Shear Zone. Gold deposits are typically located on second or third order structures or splays off the Bibiani Shear.

The Project contains mineralized zones that are characteristic of mesothermal quartz vein-style gold deposits. This type of mineralization is the most important type of gold occurring within West Africa and is commonly referred to as the Ashanti-type.

Mineralization occurs as both sulphide (within fresh and transitional facies) and non-sulphide (within transitional and oxide facies) styles, depending on the host weathering profile. Sulphide mineralization is characterized by early stage disseminated sulphides of primarily pyrite and/or arsenopyrite, hosting significant gold content, which is overprinted by late-stage quartz veining with minor amounts of visible gold and accessory polymetallic sulphides. Non-sulphide mineralization is characterized by gold hosted within non-sulphide (e.g., oxide and hydroxide) minerals, in either the early or later stage-mineralizing event. Extensive oxidation has occurred within the weathered profile throughout the Property.

1.4 Deposit Types

The Project's mineralized zones have the characteristics of epigenetic, mesothermal quartz vein-style gold deposits with an overlying gold-bearing saprolite/laterite layer. This type of mineralization is the most common style of gold occurrence in West Africa and is commonly referred to as the Ashanti type.

1.5 Exploration Work

Exploration activities, mainly consisting of line cutting, soil sampling, trenching, auger drilling and topographic surveys, have been carried out by various operators during the history of the Project. The principal targets were anomalies generated from airborne geophysical data. The work included both wide-spaced and detailed surveys. Results included anomalous gold-in-soil, trenches and auger samples, which warranted additional follow-up work.

Trenching has been a valuable exploration tool allowing for the definition of gold mineralized structures within the broad gold-in-soil anomalies identified across the Project. Exploration work at Enchi continues to define near-surface, gold mineralized structures on the Project. Trenching completed in 2021 and 2022 focused on a number of high-priority gold targets that are defined by kilometre-scale gold-in-soil anomalies located across the Property.

Drone topographic surveys completed in 2022 and 2025 have covered the vast majority of the Property with high-resolution elevation data. All RC holes, diamond core drillholes and trenches completed at the Project have been corrected to the drone topographic survey elevation data.

1.6 Drilling

Drilling at the Project can be subdivided into three phases, including:

- Historic drilling by previous owners (pre-2006).
- Recent drilling by Newcore predecessors (2010–2018).
- Recent drilling by Newcore (2020 to present).

A total of 68,017 m in 340 diamond drill (DDH) holes, 144,100 m in 1,130 reverse circulation (RC) holes, and 14,380 m in 363 rotary air blast (RAB) holes have been completed and reported on the Project from its inception to the effective date for the current report. This drilling is summarized by time period and target area in Table 1.6.1 and Table 1.6.2 respectively.

Table 1.6.1 Summary of Drilling Completed on the Project by Time Period

Total Drilling Years	DDH		RC		RAB	
	Holes	m	Holes	m	Holes	m
Pre 2006	3	129	288	26,031	363	14,380
2011-2012 and 2017-2018	192	24,005	74	9,493	-	-
2020-2025	94	27,233	768	108,576	-	-
2025-2026*	51	16,650	-	-	-	-

Total Drilling Years	DDH		RC		RAB	
	Holes	m	Holes	m	Holes	m
Totals	**340**	**68,017**	**1,130**	**144,100**	**363**	**14,380**

*Drilling completed and reported post Mineral Resource Estimate database cutoff date of 6 October 2025 and up until the report date.

Table 1.6.2 Summary of Drilling Completed on the Project by Target Area

Target	DDH		RC		RAB	
	Holes	m	Holes	m	Holes	m
Boin	119	24,716	373	51,088	275	9,341
Sewum	109	20,308	282	33,262	45	3,102
Nyam	84	18,848	216	26,180	11	421
Kwakyekrom	3	741	105	15,319	32	1,516
Tokosea	-	-	82	10,238	-	-
Nkwanta	-	-	1	100		
Kojina Hill	1	150	71	7,913		
Eradi	24	3,254	-	-		
Totals	**340**	**68,017**	**1,130**	**144,100**	**363**	**14,380**

Drilling, logging and sampling protocols were implemented to acceptable industry standards throughout the various phases of drilling at the Project.

1.7 Sample Preparation, Analysis and Security

There is little documentation available with respect to sample preparation, analysis or Quality Assurance / Quality Control (QA/QC) measures for the historic drilling programs completed prior to Newcore. However, the procedures used by Newcore (formerly Pinecrest) during the most significant and recent drill programs in the 2010s to present indicate that industry-best practices were being followed. Overall, the QP considers the dataset to be of sufficient quality for Mineral Resource estimation.

1.8 Mineral Processing and Metallurgical Testing

Newcore has conducted a number of metallurgical testwork programs since 2012 at SGS Lab (SGS), the University of Mines and Technology in Tarkwa (UMaT), Jet-Rom Engineering Ltd. (Jet-Rom), Odeleb Ltd., Intertek Minerals Ltd., as well as Intertek Base Met Labs (Intertek). The testwork was conducted on diamond drill hole (DDH), reverse circulation (RC), composite and trench samples, and included ore representing a range of weathering profiles from the various deposits. Test results from both historical and recent testwork were used to support the PFS design.

A summary of the relevant comminution and metallurgical testwork results supporting the development of a conventional milling and CIL gold processing plant is provided below:

- Comminution testwork indicates that fresh ore is significantly harder than oxide and transition ore and therefore governs the crushing and grinding circuit design. The comminution circuit was designed using conservative design parameters based on the 85th percentile of available test results.

- CIL testwork demonstrated that gold recovery from oxide ore is relatively insensitive to grind size between P_{80} 53 µm and 75 µm. However, fresh ore recovery improved by approximately 3% to 9% when the grind size was reduced from P_{80} 75 µm to P_{80} 53 µm. The grinding circuit was therefore designed to achieve a target grind size of P_{80} 53 µm to maximize overall gold recovery.

- Thickener testwork showed that oxide material exhibits the poorest settling characteristics and therefore governs thickener sizing criteria. A pre-leach thickener solids loading rate of 0.21 t/m^2/h was selected to achieve an underflow density of approximately 45% solids.

- Estimated cyanide consumption ranges from 0.83 to 1.49 kg/t, while lime consumption ranges from 1.41 to 2.61 kg/t, depending on ore type and grind size. Cyanide consumption values used for design were reduced by 50% from laboratory testwork results based on benchmarking and the assessment that cyanide may have been overdosed during testwork.

- Estimated overall gold recoveries after accounting for process losses range from 86.9% to 95.0%, depending on ore type and deposit. Fresh ore recoveries are estimated to range from 87.9% to 90.0%, while oxide and transition ore recoveries range from 86.9% to 95.0%.

- The metallurgical testwork confirms that all ore types are amenable to processing by a conventional milling and CIL flowsheet, with high gold recoveries expected across the deposits under the appropriate leach conditions.

1.9 Mineral Resource Statement

The Mineral Resource Estimate statement for the Project prepared by DRA is summarized in Table 1.9.1. Additional details on mining and processing modifying factors are also provided in the corresponding footnotes. The Resources for each deposit have all been reported using a constraining resource pit at a gold price of $3,200/oz. Mineral Resources are reported inclusive of Mineral Reserves; Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 1.9.1 Mineral Resource Estimate – Effective Date of October 6, 2025

Zone	Classification	Tonnes ('000)	Au Grade (g/t)	Contained Au (ounces)
Boin	Indicated	23,477	0.73	550,000
	Inferred	9,237	0.60	178,000
Sewum	Indicated	41,233	0.43	573,000
	Inferred	24,246	0.39	308,000
Nyam	Indicated	13,458	0.66	287,000
	Inferred	5,471	0.68	120,000
Kwakyekrom	Indicated	5,447	0.52	92,000
	Inferred	1,156	0.52	19,000
Total Indicated		**83,615**	**0.56**	**1,502,000**
Total Inferred		**40,111**	**0.49**	**626,000**

Notes for Mineral Resource Estimate:

1. Canadian Institute of Mining Metallurgy and Petroleum (CIM) definition standards were followed for the resource estimate.
2. The effective date of the Resource is 6 October 2025.
3. All figures are rounded to reflect the relative accuracy of the estimate and numbers may not add due to rounding.
4. The resource models used a combination of ordinary kriging (OK) and inverse distance weighting (IDW) grade estimation techniques within a three-dimensional block model with mineralized zones defined by wireframed solids and constrained by pits shells for Sewum, Boin, Nyam and Kwakyekrom. Validations were completed using alternative interpolation techniques for each deposit.
5. Open pit cut-off grades varied from 0.1 to 0.2 g/t A u based on mining and processing costs as well as the recoveries in different weathered material.
6. A $3,200/ounce gold price was used to determine the cut-off grade.
7. Metallurgical recovery of 85% was applied to oxide and transition mineralization, and 91.7% for fresh mineralization.
8. The pit optimization considered the following costs: mining cost based on mineralization type of $1.97/tonne for oxide, $2.62/tonne for transition, and $3.15/tonne for fresh; waste mining costs of $1.64/tonne for oxide, $2.34/tonne for transition, and $2.87/tonne for fresh; processing and G&A costs assumed of $8.74/tonne for oxide, $8.49/tonne for transition, and $19.29/tonne for fresh.
9. Average densities of mineralized material varied between 1.53 and 2.15 g/cm^3 for oxide, 1.86 and 2.38 g/cm^3 for transition, and 2.48 and 2.74 g/cm^3 for fresh rock. Average densities of waste rock varied between 1.45 and 1.77 g/cm^3 for oxide, 1.81 and 2.15 g/cm^3 for transition, and 2.45 and 2.74 g/cm^3 for fresh rock.
10. Optimization pit slope angles varied by deposit and mineralized area, with an overall strip ratio including all pits of 3.35.
11. Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
12. The mineral resource estimate was prepared by Ryan Wilson, P.Geo, Matthew Halliday, P.Geo, Schadrac Ibrango, P.Geo of DRA Americas Inc. in accordance with NI 43-101. These individuals are independent qualified persons (QP) as defined by NI 43-101.

13. As of the Report's date, the QPs, to the best of their knowledge, are not aware of any metallurgical, environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other risk factors that might materially affect the estimate of Mineral Resources.

1.10 Mineral Reserves

The Enchi Gold Project contains Probable Mineral Reserves of 51.3 Mt grading 0.64 g/t Au, containing approximately 1.055 Moz of gold, as estimated from the Boin, Sewum, Nyam, and Kwakyekrom deposits. The Mineral Reserve estimate was prepared in accordance with the 2014 CIM Definition Standards and is based on detailed pit optimization, mine design, metallurgical test work, mining recovery assumptions, operating cost estimates, and economic parameters including a gold price of $3,200/oz. Only Indicated Mineral Resources were considered for conversion to Mineral Reserves, with appropriate modifying factors applied.

The largest reserve contributors are Boin, containing 15.9 Mt at 0.84 g/t Au for 429 koz, and Sewum, containing 26.2 Mt at 0.48 g/t Au for 406 koz. Additional reserves are contained within the Nyam deposit, which hosts 7.2 Mt at 0.76 g/t Au for 175 koz, and the Kwakyekrom deposit, which hosts 2.0 Mt at 0.70 g/t Au for 44 koz. Table 1.10.1 summarizes the Mineral Reserves estimation by deposit.

Table 1.10.1 Open Pit Mineral Reserves by Deposit

Probable Mineral Reserves					
Mining Area	**Material Type**	**Tonnes**	**Au Grade g/t**	**Contained Gold oz**	**Reserve Cut-off Grade g/t Au**
Boin	Oxide	2,517,000	0.77	62,000	0.14
	Transition	5,164,000	0.77	128,000	0.18
	Fresh	8,225,000	0.90	239,000	0.27
	Total	**15,906,000**	**0.84**	**429,000**	
Sewum	Oxide	12,059,000	0.41	159,000	0.14
	Transition	8,760,000	0.49	138,000	0.18
	Fresh	5,394,000	0.63	110,000	0.27
	Total	**26,213,000**	**0.48**	**406,000**	
Nyam	Oxide	1,026,000	0.70	23,000	0.18
	Transition	2,532,000	0.67	55,000	0.22
	Fresh	3,635,000	0.83	97,000	0.30
	Total	**7,193,000**	**0.76**	**175,000**	

Probable Mineral Reserves					
Mining Area	**Material Type**	**Tonnes**	**Au Grade g/t**	**Contained Gold oz**	**Reserve Cut-off Grade g/t Au**
Kwakyekrom	Oxide	513,000	0.48	8,000	0.18
	Transition	811,000	0.61	16,000	0.22
	Fresh	654,000	0.98	21,000	0.34
	Total	**1,978,000**	**0.70**	**44,000**	
Total	Oxide	16,115,000	0.49	252,000	
	Transition	17,267,000	0.61	336,000	
	Fresh	17,908,000	0.81	467,000	
	Total	**51,290,000**	**0.64**	**1,055,000**	

Notes:

1 CIM definition standards were followed for the reserve estimate.

2. The effective date of the Mineral Reserve Estimate is 23 June 2026.

3. All figures are rounded to reflect the relative accuracy of the estimate and numbers may not add due to rounding.

4. A $3,200 per ounce gold price was used to determine the Mineral Reserve cut-off grade.

5. Reserve cut-off grades range from 0.14 g/t Au to 0.34 g/t Au depending on the material type and deposit.

6. A mining recovery of 96.0% was assumed for all deposits.

7. Dilution has been accounted for within the block model regularization process and is considered appropriate for the deposit type, varying by deposit at approximately 6 - 7%.

8. The Mineral Reserve estimate considered the following costs: Mining cost ranged from $1.94 to $4.32/tonne mined depending on the material type and deposit; Mineral processing cost of $9.78/tonne for oxide, $13.36/tonne for transition, $20.00/tonne for fresh ore, with an additional ore transportation cost included for the satellite mining deposits of $4.09/tonne for Nyam and $3.49/tonne for Kwakyekrom; Mine site G&A cost of $1.97/tonne processed assumed for all deposits.

9. Processing recoveries are variable, dependent on material type and mining area, ranging from 80% to 95%.

10. Mineral Reserves are stated in terms of delivered tonnes; gold ounces are reported as contained and do not include allowances for processing losses.

11. The mineral reserve estimate was prepared by Grant Carlson, P. Eng of Fuse Advisors in accordance with NI 43 101. This individual is an independent qualified persons ("QP") within the meaning of NI 43-101.

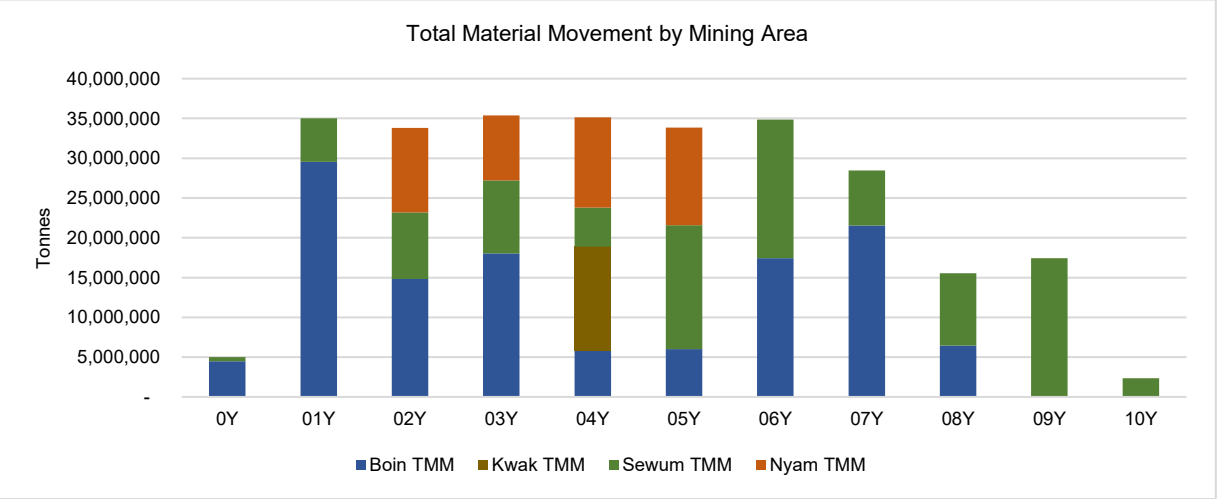
12. As of the effective date of the Mineral Reserve Estimate, the QP, to the best of their knowledge, is not aware of any metallurgical, environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other risk factors that might materially affect the potential development of the Mineral Reserves.

1.11 Mining Methods

The Enchi Gold Project is designed as a conventional open-pit, truck-and-shovel mining operation utilizing contractor mining services. Mining will occur from 11 open pits across the Boin, Sewum, Nyam, and Kwakyekrom deposits, divided into 17 mining phases over a mine life of approximately 9.3 years. Ore will be mined using drilling, blasting, loading, and hauling methods and processed at a centrally located CIL plant.

The life-of-mine (LOM) plan includes the mining of approximately 277 Mt of total material, comprising 51 Mt of ore and 226 Mt of waste, at an average strip ratio of 4.4:1, inclusive of 5 Mt of pre-strip waste mining tonnage. Annual mill throughput is planned at approximately 5.5 Mtpa, with early mine sequencing focused on higher-grade portions of the deposits to maximize project value and support strong early gold production. The production sequence by mining area is presented in Figure 1.11.1.

Figure 1.11.1 LOM Production Profile by Mining Area



Material will be mined as oxide, transition, and fresh rock. Oxide material and a significant portion of the transition material are expected to be free-dig and will not require drilling and blasting, while the lower transition zone and fresh rock will require conventional drill-and-blast methods. Free-dig material accounts for approximately 60% of total mined tonnage, while the remaining 40% is expected to require drilling and blasting. The LOM material movement by material type is summarized in Table 1.11.1.

Table 1.11.1 LOM Totals by Material Type

Material Type	Ore Tonnes Mt	Au Grade g/t	Contained Gold Koz	Waste Tonnes Mt	Total Tonnes Mt
Oxide	16	0.49	252	84.2	100
Transition - Dig	11	0.61	208	55.5	66
Transition - Blast	7	0.60	128	28.0	35
Fresh	18	0.81	467	57.8	76
Total	**51**	**0.64**	**1,055**	**226**	**277**

Mining will utilize a combination of 91-tonne rigid-frame haul trucks as the primary production fleet and 41-tonne articulated dump trucks (ADTs) for selective mining, pre-stripping activities, and mining in constrained or narrow pit-bottom areas where larger equipment is unable to operate efficiently.

Mining is planned on 10 m benches at Boin and Sewum and 12 m benches at Nyam and Kwakyekrom, with each bench mined in two flitches of 5 m and 6 m, respectively, to improve ore selectivity, grade control, and separation of ore and waste contacts.

Waste rock will be placed in dedicated waste rock storage facilities adjacent to each mining area, while ore from the satellite Nyam and Kwakyekrom deposits will be stockpiled and rehandled for transport to the process plant by a secondary ore haulage fleet.

1.12 Recovery Methods

The crushing circuit is designed to accommodate two different ore types. ROM ore will be trucked from the pits to the ROM pad, and either be directly tipped or reclaimed by FEL to the fresh or oxide ore ROM bins.

Softer oxide and diggable transition (Dig) ores are processed through a mineral sizer, which reduces ROM ore to a P_{80} of 45 mm before conveying it directly to the SAG mill feed conveyor. Harder fresh and blasted transition (Blast) ores are processed through a primary jaw crushing circuit, where ROM ore is screened, crushed to a P_{80} of 152 mm, and routed via a surge bin and transfer conveyors to the SAG mill feed conveyor. The milling circuit will be an SAB circuit (SAG and Ball mill), operating in closed circuit with classification cyclones. Ball mill trommel screen undersize will flow by gravity to the gravity feed standpipe in which a portion of this slurry will be pumped to the gravity circuit, while the standpipe will overflow to the mill discharge hopper which feeds the classification cyclones.

Cyclone underflow will be returned to the ball mill, while cyclone overflow will flow by gravity to a trash screen for removal of coarse debris ahead of the pre-leach thickener. The thickener overflow will be reused as process water, while the pre-leach thickener underflow will be pumped to the CIL circuit as leach feed. The leaching circuit will consist of a single train of seven CIL tanks. Lead nitrate solution will be added to improve the leach kinetics in this circuit.

Carbon elution will utilize a split-AARL elution method with a carbon capacity of 18 tonnes. Gold will be recovered from the pregnant eluate by electrowinning, sludge filtering and smelting to produce doré bars. Carbon will be regenerated in a diesel-fired, vendor-supplied carbon regeneration kiln.

The CIL tailings slurry will pass through a carbon safety screen to recover any residual activated carbon. The screen undersize will discharge by gravity to a tailings hopper, from which the slurry will flow by gravity to the Tailings Storage Facility (TSF).

1.13 Project Infrastructure

The Enchi Gold Project's infrastructure plan outlines facilities and systems supporting mining, processing, tailings storage facility and construction. Key Infrastructure is detailed below and includes the following scope:

Roads

Internal roads are planned to provide light vehicle access between crushing, process plant and mine services areas including the mine pits, mine dry complex, explosives magazine and fuel and lubricant storage areas. These roads will be 6 metres wide and constructed flush with bulk earthworks pads to facilitate stormwater management.

The mining haul road network for the Enchi Project has been designed to accommodate the selected mining fleet and operational requirements, ensuring safe and efficient movement of personnel, equipment, and materials throughout the site. All haul roads have been designed with a minimum running surface width of 16.8 m to accommodate the largest mining trucks in a double-lane operating configuration, supporting two-way traffic and operational flexibility.

Waste Rock Storage Facilities

The Waste Rock Storage Facilities (WRSFs) have been designed to provide sufficient capacity for the Life-of-Mine waste rock production schedule, with a total design capacity of approximately 152 million loose cubic meters (LCM) compared to a storage requirement of 147 million LCM. Waste rock is distributed across nine strategically located dump facilities, which are designed using 20 m lifts and 15 m berms at a 37° angle of repose, with additional capacity available through future lift expansions if required.

Buildings

The project infrastructure includes a range of administrative, operational, and support facilities required for mining, construction, and process plant operations. Most buildings are planned as prefabricated modular structures constructed from semi-permanent materials such as steel cladding and corrugated roofing.

The administration and operational compound, located east of the process plant near the site entrance, will include the gatehouse, emergency services / clinic, security and changehouse, administration building, and administration mess hall. Within the process plant area, facilities will include a laboratory, plant office, plant mess, warehouse and workshop, reagent storage, and storage yard. Four male and female ablution facilities will be distributed across the site.

A dedicated gold room, located south of the leach tank area, will house gold recovery and refining equipment, including electrowinning, smelting, and secure bullion storage facilities. The gold room will incorporate enhanced security measures, including controlled access, CCTV surveillance, alarm systems, and secure bullion transfer arrangements.

Power Supply and Distribution

The Plant is estimated to have a connected load of approximately 37MW with a maximum demand (running) of an estimated 33MW.

There are currently two options for power supply for mining operations: Both options remain technically feasible and capable of supplying the forecast project electrical demand with option 2 being the preferred due to the avoidance of significant upfront capital expenditure for power infrastructure for the mining operations is currently being evaluated through two potential supply arrangements:

- **Option 1 – Grid Power Supply**

Supply from the grid via the GRIDCo 161 kV Network. This would involve construction of a dedicated 161 kV transmission line and step-down substation that would be connected to the Ghana GRIDCo 161kV network.

- **Option 2 – Onsite Gas-Fired Power Generation**

Supply from an onsite gas-fired generation via a build-own-operate (BOO) arrangement with Genser Energy. This involves development of a dedicated onsite gas-fired power station and natural gas pipeline infrastructure owned and operated by Genser.

Based on preliminary information, the Genser Option 2 (onsite gas-fired generation) is the preferred power supply solution.

Tailings Storage Facility

The Project's tailings management strategy was developed in accordance with applicable regulatory requirements and the best international practices adopted for the PFS. A multi-criteria assessment of six potential TSF sites identified TSF 1 and TSF 2 as the preferred facilities to accommodate the projected 49 Mt life-of-mine tailings, with combined storage capacity of approximately 51.2 Mt, while TSF 4 was identified as a future expansion option. Both downstream-constructed facilities incorporate engineered containment measures, including compacted clay liners and a perimeter tailings distribution system, to support safe, environmentally responsible, and operationally efficient tailings storage.

1.14 Environmental, Permitting and Social Considerations

The project's environmental and social setting comprise managing various valuable environmental components and social assets. Environmental and social impacts predicted and identified are manageable through implementation of initial recommended mitigation measures. Some small communities within the Project footprint are anticipated to require relocation and resettlement. A final Resettlement Action Plan (RAP) must be developed to provide finite financial, and asset estimate for future relocation. As of this report, it is assumed that any required resettlement would occur prior to construction, and as such costs related to the relocation and resettlement of the community are not included in the Pre-Feasibility Study. An Environmental Impact Assessment (EIA) for the Project must be undertaken to obtain an Environmental Permit from the Environmental Protection Authority (EPA) as the project advances prior to construction.

1.15 Cost Estimation

The economic evaluation of the Enchi Gold Project is based on the capital and operating cost estimates developed in this Pre-Feasibility Study. All costs are presented in United States Dollars (USD), unless otherwise noted, and are based on first quarter 2026 pricing. The overall estimate is classified with an accuracy range of -20% to +30%.

1.15.1 Capital Cost Estimate

The total capital cost estimate reflects the process plant design to support a 9.3-year mine plan and production schedule, with Year-1 and Year-4 adopted as representative design cases for the two primary ore types that will be encountered across the mine life.

The total initial capital cost is estimated at $351.2 million. The following table provides a detailed breakdown per area of the total project cost.

Table 1.15.1 Total Project Capital Cost Summary

Description	Initial $M	Sustaining $M	Closure $M	LOM $M
Construction Distributables	$34.3	-	-	$34.3
Treatment Plant Costs	$113.5	-	-	$113.5
Reagents and Plant Services	$18.7	-	-	$18.7
Infrastructure - Tailings Dam	$25.8	$108.4	-	$134.3
Infrastructure - Power Supply	$4.9	-	-	$4.9
Other Site Infrastructure and Services	$16.9	-	-	$16.9
Mining (Incl. Pre-Production)	$23.3	$8.6	-	$32.0
EPCM Costs	$38.6	-	-	$38.6

Description	Initial $M	Sustaining $M	Closure $M	LOM $M
Owners Projects Costs	$37.4	-	-	$37.4
Closure Capital	-	-	$18.5	$18.5
Contingency	$37.7	$17.7	$2.8	$58.2
Total Capital Costs	**$351.2**	**$134.8**	**$21.3**	**$507.2**

1.15.2 Operating Cost Estimate

Processing operating costs have been estimated for the plant with an ore feed rate of 5.5 Mtpa, based on continuous operation of 24 hours per day and 365 days per year, for three different scenarios.

The Oxide (75 µm) processing scenario applies to both oxide ore and diggable upper transition ore during Year 1. After Year 1, the Oxide (53 µm) processing scenario applies to the remaining diggable upper transition ore. Fresh (53 µm) processing scenario also applies to lower transition ore that requires drilling and blasting. The finer grind size reflects the need to blend this material with fresh ore or drilled-and-blasted lower transition ore to achieve the required processing characteristics.

A summary of the processing operating costs is presented in Table 1.15.2 with a breakdown of the costs:

Table 1.15.2 Operating Cost Summary

Plant Throughput	Oxide (@ P_{80} - 75 µm)		Oxide (@ P_{80} - 53 µm)		Fresh (@ P_{80} - 53 µm)	
	5,500,000 tpa Ore		5,500,000 tpa Ore		5,500,000 tpa Ore	
Cost Centre	Total Operating Cost		Total Operating Cost		Total Operating Cost	
	US$/year	US$/t Ore	US$/year	US$/t Ore	US$/year	US$/t Ore
Process Plant						
Operating Consumables	27,775,107	5.05	30,443,154	5.54	44,621,233	8.11
Power	16,203,714	2.95	17,562,878	3.19	27,705,862	5.04
Plant Maintenance	4,331,696	0.79	4,364,818	0.79	4,516,975	0.82
Laboratory	658,666	0.12	658,666	0.12	658,666	0.12
Labour (O&M)	4,433,134	0.81	4,433,134	0.81	4,433,134	0.81
Total - Process Plant	**53,402,316**	**9.71**	**57,462,649**	**10.45**	**81,935,869**	**14.90**

Reagent and consumable consumption rates have been estimated based on laboratory testwork results, comminution circuit modelling using average mineralized material characteristics, first-principles calculations, and operating experience from comparable projects. No additional allowances have been included for process upsets or reagent losses. Processing consumable costs, including reagents, grinding media, liners, fuels, and other process supplies, have been estimated.

Process plant power consumption has been determined from the installed power requirements of plant equipment, as defined in the electrical load list, and includes power demand for both process facilities and site accommodation infrastructure.

Annual maintenance costs have been estimated by applying a maintenance factor equivalent to 5.0% of the equipment supply cost across all plant areas.

Labour costs have been developed using benchmark compensation data from mining operations in Ghana. Staffing levels, operating rosters, and labour rates have been established based on continuous plant operation using two 12-hour shifts per day, with additional personnel allowances included to provide coverage for leave, training, and absenteeism.

Laboratory and assay cost estimates include routine plant solids and solution analyses, as well as the assessment of a minimum of 150 grade-control samples per day in support of mining activities. Allowances have been made for both onsite and external analytical services.

The requirements for plant mobile equipment, including a mobile rock breaker for the mineral sizing circuit, were established in consultation with the Client and benchmarked against comparable projects operating in similar jurisdictions.

The General and Administration (G&A) costs are based on information and cost data provided by Newcore and Lycopodium's in-house database, based on an agreement with Newcore and referencing similar projects within comparable geographic regions.

1.15.3 Sustaining Cost Estimate

Sustaining capital costs represent the ongoing capital expenditure required to maintain the asset's productive capacity and operational integrity over the life of mine. These costs have been estimated and classified in accordance with the same criteria and methodology applied to the initial capital expenditure (CAPEX), ensuring consistency in scope definition, cost structuring, and reporting standards across the estimate. Closure costs are not included in this estimate and fall outside the scope of sustaining capital. The sustaining capital cost is estimated at $134.8 million with corresponding cost allocations as shown in Table 1.15.3:

Table 1.15.3 Sustaining Cost Summary

Description	Sustaining $M
300 Infrastructure - Tailings Dam	108.4
400 Mining	8.7
Contingency	17.7
Sustaining Capital Costs	**134.8**

1.15.4 Closure Cost

A Pre-Feasibility Study (PFS)-level closure cost estimate of approximately $18.5 million was prepared by Knight Piésold Consulting (KP) and Fuse Advisors, covering physical closure and reclamation activities. A 15% contingency was incorporated into the cash flow model, resulting in a total estimated closure cost of $21.3 million. These activities encompass the dismantling and removal of key infrastructure, demolition of concrete and masonry structures, and rehabilitation of disturbed areas through the re-establishment of native vegetation. Closure costs assume $40,000 per year over five years for care and maintenance of rehabilitated areas, as well as $150,000 per year over five years for environmental monitoring. These costs first occur in Year 10.

A summary of the mine closure cost estimate is presented in Table 1.15.4.

Table 1.15.4 Closure Costs Estimate Summary

Description	Closure Costs $M
Mining Demobilization	$2.4
Site Demobilization and Reclamation	$15.1
Post-Closure Monitoring and Maintenance	$1.0
Salvage Value	-
Total Pre-Contingency	**$18.5**
Contingency (15%)	$2.8
Total Closure Capital Costs	**$21.3**

1.16 Economic Analysis

A high-level illustrative economic analysis of the potential of the Enchi Gold Project has been carried out using a cash flow model. The financial model is based on annual cash flows, incorporating yearly processed tonnages and grades in the process plant feed. It also accounts for process recoveries, metal prices, operating costs, refining costs, royalties, and capital expenditures, including both initial and sustaining capital.

The model has been developed on a 100% project basis. A royalty on revenues is payable to the Government of Ghana based on a sliding-scale structure ranging from 5% to 12%, with a rate of 12% when the gold price is at or above $4,500/oz. At the base case gold price of $3,800/oz a government royalty rate of 10% is applicable. Additionally, a 2% net smelter return (NSR) royalty is payable to Triple Flag Precious Metals Corp (with 1% subject to a buy-back option for a lump sum payment of $3.5 million at any time held by Royal Gold Inc.). Consistent with standard practice for projects of this nature, the financial evaluation is conducted on a "100% equity" basis, with no consideration given to debt and equity financing structures. Inflation has not been incorporated into the analysis. Results are presented on both a pre-tax and after-tax basis, with current Ghanaian tax regulations applied to determine corporate and mining tax obligations. The Government of Ghana retains a 10% free carried interest in the Project.

All monetary values are expressed in United States dollars (US$) and, unless otherwise noted, are denoted as "$". The model reflects the base case and technical assumptions outlined in the preceding sections of this report. A mid-year discounting is applied from the second year of mine construction (Year -1 onwards).

On a pre-tax basis, the Project has a Net Present Value (NPV) at a discount rate of 5% of $794M, with an Internal Rate of Return (IRR) of 51% and a payback period of 1.3 years. On an after-tax basis, the $NPV_{5\%}$ is $496M, the IRR is 37%, and the payback period is 1.6 years. The payback period corresponds to the time at which the cumulative cash flow becomes positive.

The key assumptions and economic parameters used in the PFS are as follows:

Table 1.16.1 Key Economic Results

Key Assumptions	
Base Case Gold Price	$3,800/oz
Production Profile	
Total Tonnes Processed (mt)	51.3
Total Tonnes Waste (mt)	220.5
Strip Ratio (1)	4.3
Gold Grade Processed (g/t Au)	0.64
Mine Life (years)	9.3
Throughput (mtpa)	5.5
Gold Recovery	90.5%
LOM Payable Gold Production (ozs Au)	953,350
LOM Average Annual Gold Production (ozs Au)	104,162
Peak Gold Production (ozs Au)	136,709

Key Assumptions	
Unit Operating Costs	
LOM Average Operating Cost (2) (US$/oz Au)	$1,689
LOM Average Cash Cost (3) (US$/oz Au)	$2,149
LOM AISC (Cash Cost plus Sustaining Cost) (4) (US$/oz Au)	$2,290
Initial Production – Years 1 to 3	
Average Annual Gold Production (ounces)	128,924
Mill Head Feed Grade (g/t Au)	0.80
Average Operating Costs (1) (US$/oz Au)	$1,399
Average Cash Costs (2) (US$/oz Au)	$1,859
AISC (Cash Costs plus Sustaining Capital) (3) (US$/oz Au)	$1,967
Capital Costs	
Initial Capital Cost (US$M)	$351
LOM Sustaining Capital Cost (US$M)	$135
Closure Cost (US$M)	$21

Notes:

1. Strip ratio excludes pre-strip tonnes mined (included in initial capex).
2. Cash costs consist of mining costs, processing costs, and mine-site G&A.
3. Cash costs consist of operating costs plus treatment and refining charges, and royalties.
4. AISC consists of cash costs plus sustaining capital (excluding closure costs and taxes.)

An economic evaluation of the mine schedule, based on a gold price of $3,800/oz, indicates that the project is financially viable. The analysis shows that project economics are most sensitive to changes in gold price, gold recovery, and operating costs, while exhibiting lower sensitivity to variations in capital costs.

1.17 Interpretations and Conclusions

The Mineral Resources currently defined for the Enchi Gold Project was analyzed for each of the four deposits Boin, Sewum, Nyam, and Kwakyekrom. The Mineral Resources Estimate consists of an Indicated Mineral Resource of 1,502,000 ounces of gold (83.6 million tonnes at an average grade of 0.56 g/t gold) and an Inferred Mineral Resource of 626,000 ounces of gold (40.1 million tonnes at an average grade of 0.49 g/t gold). A $3,200/ounce gold price was used to determine the cut-off grade. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

The Enchi Gold Project contains Probable Mineral Reserves of 51.3 Mt grading 0.64 g/t Au, containing approximately 1.055 Moz of gold, as estimated from the Boin, Sewum, Nyam, and Kwakyekrom deposits. The Mineral Reserve estimate was prepared in accordance with the 2014 CIM Definition Standards and is based on detailed pit optimization, mine design, metallurgical test work, mining recovery assumptions, operating cost estimates, and economic parameters including a gold price of US$3,200/oz.

The process plant is designed to treat both oxide / transition and fresh ore through two separate crushing circuits feeding a common SAB grinding circuit, followed by CIL circuit, with gold recovered via carbon elution, electrowinning, and smelting to produce doré bars.

Highlights of the PFS include:

- The Enchi Gold Project LOM plan forecasts 277 Mt be mined over a period of 9.3 years.
- Processing of ore material through a CIL Plant is forecasted to be 51.3 Mt at an average ROM gold grade of 0.64 g/tonne with an expected 1,055,000 oz of contained gold.
- Average LOM operating costs estimated at $1,689/oz, cash costs estimated at $2,149/oz, and all-in sustaining costs estimated at $2,290/oz gold.
- Initial capital expenditure of $351M, sustaining capital expenditure of $135M, and closure capital costs of $21M.
- At a gold price of $3.800/oz: $794M pre-tax $NPV_{5\%}$ and a 50.9% pre-tax IRR with a 1.3-year payback; $496M after-tax $NPV_{5\%}$ and a 37.4% after-tax IRR with a 1.6-year payback.

1.18 Recommendations

1.18.1 Work Program

The results presented in this report demonstrate that the Enchi Gold Project is technically viable. It is recommended that the project be advanced to the feasibility study (FS) phase.

Table 1.18.1 summarizes the proposed budget to advance the Project through the feasibility stage along with additional drilling and development work.

Table 1.18.1 Proposed Feasibility Study Budget Summary

Table	Cost $
Drilling	3,500,000
Geology and Resource	100,000
Mining	700,000
Metallurgy	1,000,000
Process and Infrastructure	1,500,000
Mine Water and Water Management	250,000
Environment and Engagement	750,000
Tailings Storage Facility	1,000,000

Table	Cost $
Value Engineering and Trade-off Studies	250,000
Total	**9,050,000**

1.18.2 Metallurgical Testwork Program

The following testwork is recommended to progress the design in the FS phase:

- Collect representative master composite, variability, and bulk samples for all mineralization types based on the mine plan.
- Complete additional comminution testwork, including SMC testing, to improve design confidence and optimize milling circuit performance.
- Undertake further ore characterization and mineralogical studies to better understand ore variability, gold deportment, and material handling properties.
- Conduct additional gravity and leach testwork to confirm flowsheet design, optimize recovery, and refine gravity, CIL, and oxygen plant sizing criteria.
- Perform dynamic settling and rheology testing to confirm thickener sizing and slurry handling requirements.
- Complete cyanide detoxification testwork to support detox circuit (if required) and water treatment plant design.
- Carry out water quality and lime buffering tests to support process water and reagent consumption design criteria.
- Undertake comprehensive tailings geotechnical and geochemical characterization to support TSF design, environmental assessments, and closure planning.

1.18.3 Geology and Mineral Resource

Key recommendations highlighted by the QPs include:

- In addition to the chip boards and pulps currently retained for storage, a certain proportion of remaining original RC sample material should be stored in a secure location for further sampling, re-sampling or independent check assay programs. All coarse reject materials from mineralized zones should also be retained for the same purposes.

- An increased proportion of diamond drilling should be completed to help better define the weathering profile between oxide, transition and fresh rock for all deposits.
- Further review of the historical QA/QC and currently available standards should be conducted; three representative standards for each of oxide and fresh mineralization types (six (6) in total) should be selected with no overlap in performance (i.e., low, medium and high grades).
- A real-time quality detection system should be developed and implemented in order to act immediately on failures and maintain an updated action table for documentation.
- Additional bulk density should be collected for both Nyam and Kwakyekrom to improve the understanding of density in these deposits for future block models.
- Increased representative density measurements should be collected within each lithology, mineralization and weathering type.
- It is recommended to conduct regular check assay programs (on the order of 5%) for both RC and DD campaigns to better understand the relatively high variability ranges observed during DRA's QP check program.
- It is also recommended that 3D modelling is conducted in real time; dowhhole surveys should be imported and reviewed during drilling, allowing erroneous surveys to be discovered in real time. Additional multishot or gyro surveys can also be conducted at hole closure if warranted.
- Additional attention should be made to lithological, alteration and oxidation modelling. Vein modelling data should be standardized and potentially warranted to have its own data sheet. All veins should make it to the single sources of truth, even if they are primary lithology, secondary lithology, thin structure, etc.
- The Checkerboard zone at Sewum has two directions of mineralization, one orthogonal to the primary orientation. It is recommended to conduct additional exploration to resolve the orientation and location, width and grade distribution of secondary structures.

1.18.4 Mineral Reserves

- Incorporation of updated pit optimization inputs based on further details that were determined during the PFS study completion, including revised G&A cost assumptions and royalty rates.
- Reassess ore loss and mining recovery factors based on anticipated equipment fleet, bench configuration, and ore-waste contact complexity.
- Validate Selective Mining Unit (SMU) dimensions against planned mining equipment sizes, ore selectivity capabilities, and grade control practices.

- Targeted infill drilling programs within economically significant Inferred Mineral Resource areas with the potential for upgrading to an Indicated or Measured classification.

1.18.5 Mining Methods

- Review and optimize secondary ore haulage requirements throughout the mine life, including identification of the economic transition point where ore should be transferred from the primary mining fleet to a dedicated secondary haulage fleet.

- Evaluate potential haul road alignments and required earthworks to identify the optimum balance between upfront road construction costs and long-term haulage cost savings.

- Refine the life-of-mine haulage network plan incorporating phased road development, pit access requirements, and integration with key mine infrastructure and material destination points.

- Evaluate WRSF locations considering geotechnical, environmental, permitting, and operational factors, including assessment of surface preparation requirements, development sequencing, and associated mining costs.

- Optimize drill-and-blast design parameters and fragmentation targets to reduce waste mining costs, improve loading and hauling productivity, and support mine-to-mill optimization of ore material.

1.18.6 Pit Geotechnical

- Complete additional geotechnical drilling across all planned pits and deposits to reduce uncertainty related to lithological variability, weathering profiles, structural controls, and geotechnical domains, supporting refinement of inter-ramp and overall slope angle designs.

- Maximize the geotechnical data collection from ongoing resource drilling programs to help refine IRA and OSA parameters and strengthen confidence in the geotechnical model.

- Complete additional geotechnical and rock mechanics testing, particularly within fresh and transition materials and the varying lithologies across the Shear Zone, to expand the geotechnical database and improve confidence in the selected geotechnical design parameters.

1.18.7 Infrastructure

The following engineering activities are recommended to advance the site infrastructure from the Pre-Feasibility Study (PFS) to the Feasibility Study (FS) level.

- Refine the overall site layout to optimize the arrangement of the process plant, administration facilities, workshops, warehouses, stockpiles, utilities, and supporting infrastructure while maintaining operational efficiency, maintenance access and future expansion capability.

- The overall process plant layout should be reviewed and optimized, including consideration of terraced facility arrangements and revised platform elevations, to reduce cut-and-fill quantities, minimize retaining structures where practical, improve constructability, and maintain adequate site drainage.

- Undertake detailed geotechnical investigations across all major infrastructure footprints, including process plant facilities, proposed power plant location, stockpiles, and utility corridors, to confirm foundation design parameters.

- Complete detailed engineering of all site access roads, mine haul roads, and process plant roads, including pavement design, drainage structures, culverts, erosion protection measures, and optimized road alignments.

- Develop detailed material take offs (MTOs) for concrete, structural steel, platework and piping to support a Class 2 capital cost estimate.

- Complete a detailed trade-off study between grid power and onsite power generation considering capital cost, operating cost, implementation schedule, reliability and expansion capability.

1.18.8 TSF Design

Key TSF design recommendations for advancing the TSF include:

- Complete additional geotechnical investigations to confirm subsurface conditions, foundation competence, and groundwater conditions at the selected TSF sites

- Confirm the availability, quantity, and suitability of construction materials required for embankment fill materials, liner systems, filters, and drains.

- Conduct further testing to define tailings geotechnical and geochemical properties, including density, strength, deposition behaviour, and seepage characteristics

- Progress detailed analyses of embankment geometry, staged construction, storage capacity, seepage control, stability, water balance, and operational requirements.

- Develop design measures for long-term stability, capping, surface re-sloping, diversion drainage, and seepage control.

- Refine designs related to tailings delivery pipelines, decant systems, underdrainage, leak detection, diversion channels, and emergency spillways.

- Update cost estimates and implementation schedules using refined quantities, confirmed borrow requirements, and updated design assumptions.

- Continue design development for TSF 1 and TSF 2, with TSF 1 as the preferred option, TSF 2 supporting current Life-of-Mine storage requirements, and TSF 4 reserved for potential future expansion.

- Refine and optimize the overall infrastructure layout, including process plant facilities, roads, utilities, and supporting infrastructure, to improve constructability, operational efficiency, maintenance access, future expansion capability, and reduce earthworks quantities through grading and terraced layouts where appropriate.

- Advance the infrastructure design to Feasibility Study level by completing geotechnical investigations, detailed engineering of civil and electrical infrastructure, detailed material take-offs (MTOs), and optimization studies, including evaluation of the preferred project power supply option, to support a Class 2 capital cost estimate.

1.18.9 Environment, Permitting and Social Considerations

- Undertake additional baseline studies to support EIA and engineering designs. Seasonal baseline studies will provide comprehensive pre-project baseline information or knowledge base against which to assess potential impacts and evaluate the Project against national and international requirements. Future studies should consider an Ecosystem Services Benefit (ESB) study and a Biodiversity Management Plan to preserve and conserve species of conservation importance.

- Social baseline studies should include detailed mapping of catchment communities, settlements and hamlets, as well develop a full RAP for potential displacement or resettlement of people.

- Undertake continuous monitoring such as a weather and air quality stations at relevant locations.

- Undertake a regional hydrocensus and water quality testing programme to generate comprehensive dataset to improve confidence in modelling outcomes (e.g., hydrogeological, water balance, solute transport models, etc.)

- Undertake a comprehensive stakeholder engagement programme and document all stakeholder interactions to assist the Community Relations team as reference point for future engagements.

- Ensure ongoing community investment is done in a strategic and sustainable way so that any supported projects / infrastructure can be independently managed and maintained.
- Integrate sensitive and protected areas into the Global Information System (GIS) used by the exploration team, to minimize the risk for damage, for example to cultural heritage sites and known wildlife habitats.
- Ensure exploration drill holes and trenches are properly sealed, to minimize land disturbance and avoid future problems with water connectivity – establish a formal procedure for this and ensure the closure of all drill sites is properly documented, in particular near the proposed TSF location.
- Update engineering designs, models, analyses, and cost estimates as necessary, and incorporate environmental and social risks.

2.0 INTRODUCTION

This report was prepared and compiled by the Qualified Persons (QPs) employed by Lycopodium (Americas) Ltd. (Lycopodium), Knight Piésold Consulting. (KP), Fuse Advisors Inc. (Part of SLR Consulting) (Fuse), and DRA Americas Inc. (DRA) at the request of Newcore Gold Ltd. (Newcore or the Company). The purpose of this report is to present a technical report for the Enchi Gold Project (Enchi, the Project, or the Property) in accordance with the Canadian Securities Administrators National Instrument 43-101 (NI 43-101) and Form 43-101 F1.

The responsibilities of the engineering companies contracted by Newcore Gold to prepare this Report are outlined below:

- Lycopodium managed and coordinated the development of this report, developed PFS-level designs for the process plant and general on-site infrastructure, developed the consolidated cost estimates and reviewed and signed off on the financial model that was prepared by the Company.
- Knight Piésold Consulting supervised geotechnical investigations, provided geotechnical recommendations, conducted environmental studies, and developed the PFS-level design and cost estimate of the tailings and water management facilities.
- DRA developed the mineral resource estimate.
- Fuse prepared the mineral reserve estimate, mining methods and mining capital cost and operating cost estimates.

2.1 Purpose of Report

The purpose of this report is to publish a technical report on the Enchi Gold Project summarizing the geology, past exploration activities, Mineral Resource Estimate, Mineral Reserve Estimate, and Pre-Feasibility Study (PFS) completed for the Project.

2.2 Terms of Reference

Newcore engaged the services of the authors in January 2026 to write an independent NI 43-101 Technical Report on the Enchi Property in Ghana. This report was prepared in accordance with NI 43-101 and Form NI 43-101 F1.

2.3 Newcore Gold Ltd.

Newcore's corporate offices are located at Suite 610 - 815 West Hastings St., Vancouver, British Columbia, V6C 1B4, Canada. Newcore is a public company listed on the TSX Venture Exchange (TSX-V: NCAU) and also trades on the OTCQX® in the United States (OTCQX: NCAUF). Newcore and its subsidiaries engage principally in the advancement and development of the Enchi Gold Project in Ghana.

2.4 Qualification of Consultants

The consultants responsible for preparing this technical report are qualified specialists in geology, mining, mineral processing, and mine infrastructure. The consultants and their associates involved in the preparation of this report have no beneficial interest in Newcore. The consultants are not insiders, associates, or affiliates of Newcore. The conclusions presented in this technical report are not contingent on any prior agreements regarding the outcomes to be reached, and there are no undisclosed understandings related to future business dealings between Newcore and the consultants. The consultants were compensated for their services in accordance with standard professional consulting practices.

2.5 Report Responsibility and Qualified Persons

The following individuals are considered Qualified Persons (QPs), as defined under NI 43-101, by virtue of their education, professional experience, and membership in good standing with appropriate professional institutions.

- Preetham Nayak, P. Eng., Lycopodium (Americas) Ltd.
- Ryda Peung, P. Eng., Lycopodium (Americas) Ltd.
- Zuned Shaikh, P. Eng., Lycopodium (Americas) Ltd.
- Ama Nketiah, P.E., Knight Piésold Consulting
- George Lartey-Young, C. Env., Knight Piésold Consulting
- Hugues Nkulu Ngoy, Knight Piésold Consulting
- Martiens Prinsloo, Pri.Nat.Sci, Knight Piésold Consulting
- Jorge Velazquez, Knight Piésold Consulting
- Grant Carlson, P. Eng., Fuse Advisors Inc. (Part of SLR Consulting).

- Ryan Wilson, P. Geo., DRA Americas Inc.
- Matthew Halliday, P. Geo., DRA Americas Inc.
- Schadrac Ibrango, P. Geo., DRA Americas Inc.
- Claude Bisaillon, P. Eng., DRA Americas Inc.

The aforementioned QPs contributed to the preparation of this Report and have provided QP certificates, which are included at the beginning of this Report. The certificates identify the specific sections of the Report for which each QP is responsible. Each QP also contributed figures, tables, and relevant portions of Chapters 1 (Summary), 2 (Introduction), 3 (Reliance on Other Experts), 21 (Capital and Operating Costs), 25 (Interpretation and Conclusions), 26 (Recommendations), and 27 (References). Table 2.5.1 summarizes the allocation of responsibilities for the applicable sections of the Report and identifies the corresponding Qualified Person.

Table 2.5.1 Qualified Persons and Areas of Report Responsibility

Chapter	Description	Lycopodium	DRA	Knight Piésold	Fuse
1	Summary		MH		
1.1	Introduction	PN			
1.2	History		RW		
1.3	Geological Setting and Mineralization		RW		
1.4	Deposit Types		RW		
1.5	Exploration		RW		
1.6	Drilling		RW		
1.7	Sample Preparation, Analyses and Security		MH		
1.8	Mineral Processing and Metallurgical Testing	RP			
1.9	Mineral Resource Estimate		RW		
1.10	Mineral Reserve Estimate				GC
1.11	Mining Methods				GC
1.12	Recovery Methods	RP			
1.13	Project Infrastructure	ZS			
1.14	Environmental Studies, Permitting, and Social or Community Impact			GLY	
1.15	Cost Estimation				

Chapter	Description	Lycopodium	DRA	Knight Piésold	Fuse
1.15.1	Capital Cost Estimate	PN			
1.15.2	Operating Cost Estimate	RP			
1.15.3	Sustaining Cost Estimate	PN			
1.15.4	Closure Cost			GLY	
1.16	Economic Analysis	PN			
1.17	Interpretations and Conclusions	PN	RW		
1.18	Recommendations				
1.18.1	Work Program	PN			
1.18.2	Metallurgical Testwork Program	RP			
1.18.3	Geology and Mineral Resource		RW		
1.18.4	Mineral Reserves				GC
1.18.5	Mining Methods				GC
1.18.6	Pit Geotechnical		CB		
1.18.7	Infrastructure	ZS			
1.18.8	TSF Design			AN	
1.18.9	Environment, Permitting and Social Considerations			GLY	
2	Introduction	PN			
3	Reliance on other Experts	PN			
4	Project Property Description and Location		RW		
5	Accessibility, Climate, Local Resource, Infrastructure and Physiography		RW		
6	History		RW		
7	Geological Setting and Mineralization		RW		
8	Deposit Types		RW		
9	Exploration		RW		
10	Drilling		RW		
11	Sample Preparation, Analyses and Security		MH		
12	Data Verification		SI		
13	Mineral Processing and Metallurgical Testing	RP			
14	Mineral Resource Estimate				

Chapter	Description	Lycopodium	DRA	Knight Piésold	Fuse
14.1	Mineral Resource Estimate Definition and Procedure		RW		
14.2	General Description		RW		
14.3	Supporting Data		RW		
14.4	Boin Deposit		SI		
14.5	Sewum Deposit		MH		
14.6	Nyam Deposit		RW		
14.7	Kwakyekrom Deposit		RW		
14.8	Mineral Resource Statement		RW		
14.9	Open-Pit Sensitivity Analysis		RW		
15	Mineral Reserve Estimate				GC
15.5.1	Geotechnical Design Criteria		CB		
16	Mining Methods				
16.1	Introduction				GC
16.2	Pit Design				GC
16.2.1	Geotechnical Design Criteria		CB		
16.2.2	Haul Road Design Criteria				GC
16.3	Waste Rock Storage Facility (WRSF) Design				GC
16.4	Mine Layout				GC
16.5	Mine Phases				GC
16.6	Mine Schedule				GC
16.7	Mining Operations				GC
16.7.1	Load and Haul				GC
16.7.2	Drill and Blast				GC
16.7.3	Grade Control				GC
16.7.4	Ancillary Equipment				GC
16.7.5	In-pit Water Management				GC
17	Recovery Methods	RP			
18	Project Infrastructure				
18.1	Introduction	ZS			
18.2	Roads	ZS			
18.2.1	Project Site Roads	ZS			
18.2.2	Mining Haul Roads				GC
18.3	Waste Rock Storage Facilities				GC
18.4	Process Plant Infrastructure	ZS			

Chapter	Description	Lycopodium	DRA	Knight Piésold	Fuse
18.5	Power Supply and Distribution	ZS			
18.6	Tailings Storage Facility			AN	
18.7	TSF to Plant Site Access Road			AN	
18.8	Mine-Wide Water Balance			JV	
18.9	Mine-Wide Stormwater Management Plan			JV	
18.10	Mine-Wide Hydrogeology				
18.10.1	Hydrostratigraphy			HNN	
18.10.2	Groundwater Levels and Flow Patterns			HNN	
18.10.3	Hydrochemistry and Geochemistry			MP	
18.10.4	Three-Dimensional Conceptual Hydro-Stratigraphic Model			HNN	
18.10.5	Groundwater Numerical Model			HNN	
18.10.6	Predictive Simulations and Model Results			HNN	
18.10.7	Conceptual Dewatering and Water Supply			HNN	
18.11	Water Systems	ZS			
18.12	Sewage Treatment	ZS			
18.13	Fuel Storage and Distribution				GC
18.14	Communication				GC
18.15	Accommodation Camp	ZS			
19	Market Studies and Contracts	PN			
20	Environmental Studies, Permitting, and Social or Community Impact			GLY	
21	Capital and Operating Costs				
21.1	Introduction	PN			
21.2	Overall Project Cost	PN			
21.3	Capital Cost Estimate				
21.3.1	Introduction	PN			
21.3.2	Area 000 – Construction Distributables	PN			
21.3.3	Area 100 – Treatment Plant Cost	PN			

Chapter	Description	Lycopodium	DRA	Knight Piésold	Fuse
21.3.4	Area 200 – Reagent and Plant Services	PN			
21.3.5	Area 300 – Infrastructure	PN			
21.3.6	Area 400 – Mining				GC
21.3.7	Area 500 – Management Costs	PN			
21.3.8	Area 600 – Owner Project Costs	PN			
21.4	Operating Cost Estimate				
21.4.1	Introduction	RP			
21.4.2	Mining Operating Cost Estimate				GC
21.4.3	Process Plant Operating Cost Estimate	RP			
21.4.4	TSF Operating Cost Estimate			AN	
21.4.5	General and Administration	RP			
21.5	Sustaining Capital				
21.5.1	Mining				GC
21.5.2	Process Plant	PN			
21.5.3	Tailings Storage Facilities			AN	
21.6	Closure Costs			GLY	
22	Economic Analysis	PN			
23	Adjacent Properties		RW		
24	Other Relevant Data and Information	PN			
25	Interpretation and Conclusions				
25.1	Introduction	PN			
25.2	Geology and Mineral Resource Estimate		RW		
25.3	Mineral Processing and Metallurgical Testwork	RP			
25.4	Mining and Mineral Reserves				GC
25.5	Recovery Methods	RP			
25.6	Infrastructure	ZS			
25.7	Environmental, Permitting and Social Considerations			GLY	
25.8	Capital Cost Estimate	PN			
25.9	Operating Cost Estimate	RP			
25.10	Economic Analysis	PN			

Chapter	Description	Lycopodium	DRA	Knight Piésold	Fuse
25.11	Risks and Opportunities				
25.11.1	Geology and Resources Risks		RW		
25.11.2	Mineral Reserves Risks				GC
25.11.3	Mining Methods Risks		CB		GC
25.11.4	Process Design Risks	RP			
25.11.5	Infrastructure Risks	ZS			
25.11.6	Tailings Storage Facility Risks			AN	
25.11.7	Geology and Resources Opportunities		RW		
25.11.8	Mineral Reserves Opportunities				GC
25.11.9	Mining Methods Opportunities				GC
25.11.10	Process Design Opportunities	RP			
25.11.11	Infrastructure Opportunities	ZS			
25.11.12	TSF Design Opportunities			AN	
25.11.13	Environment, Permitting and Social Considerations Opportunities			GLY	
25.12	Conclusion	PN			
26	Recommendations				
26.1	Path Forward	PN			
26.2	Geology and Mineral Resource		RW		
26.3	Mining and Mineral Reserves				GC
26.3.3	Geotechnical		CB		
26.4	Metallurgy	RP			
26.5	Infrastructure	ZS			
26.6	Mine Water and Water Management				GC
26.7	Tailings Storage Facility			AN	
26.8	Environmental, Social and Community Engagement			GLY	
26.9	Feasibility Study	PN			
27	References	PN/RP	RW/MH/CB		

2.6 Site Visit

The following QP's have completed property site visits:

Qualified Person	Company	Date of Site Visit
Preetham Nayak	Lycopodium (Americas) Ltd	N/A
Ryda Peung	Lycopodium (Americas) Ltd	N/A
Zuned Shaikh	Lycopodium (Americas) Ltd	N/A
Ama Nketiah	Knight Piésold Consulting	N/A
George Lartey-Young	Knight Piésold Consulting	February 09-13, 2026
Hugues Nkulu Ngoy	Knight Piésold Consulting	N/A
Martiens Prinsloo	Knight Piésold Consulting	N/A
Jorge Valazquez	Knight Piésold Consulting	N/A
Grant Carlson	Fuse Advisors (Part of SLR)	N/A
Ryan Wilson	DRA Americas Ltd.	N/A
Matthew Halliday	DRA Americas Ltd.	N/A
Schadrac Ibrango	DRA Americas Ltd.	September 23-25, 2025
Claude Bisaillon	DRA Americas Ltd.	N/A

2.7 Effective Date and Declaration

The issue date of this report is 7 August 2026. The effective date of the technical report is 23 June 2026. As of the date of this report, the authors are not aware of any material fact or material change with respect to the subject matter of this technical report that is not presented herein, or which the omission to disclose could make this report misleading.

2.8 Currency, Units of Measure, and Calculations

Unless otherwise specified or noted, the units used in this report are metric. Every effort has been made to clearly display the appropriate units being used throughout the report.

- Coordinates within this report use WGS 84 UTM Zone 30N, unless otherwise stated.
- Currency is in United States dollars (US$ or $), unless otherwise noted.
- All ounce units are reported in troy ounces, unless otherwise stated: 1 oz (troy) = 31.1035 g.

This report includes technical information that required subsequent calculations to derive subtotals, totals, and weighted averages. Such calculations inherently involve rounding and may, therefore, introduce minor variances. Where such variances occur, the QPs consider them to be immaterial.

2.9 Acknowledgement

The Project benefited from the specific input of Greg Smith, P. Geo. (VP, Exploration), Luke Alexander (President, CEO & Director), Danny Lee (CFO), and Mal Karwowska (VP, Corporate Development & Investor Relations) of Newcore.

3.0 RELIANCE ON OTHER EXPERTS

3.1 Introduction

The QPs have relied on external expert reports, which provide information on mineral rights, surface rights, property agreements, royalties, environmental considerations, permitting, social licence, closure, taxation, and marketing for relevant sections of this Report.

3.2 Property Agreements, Mineral Tenure, Surface Rights and Royalties

The QPs have not independently verified ownership of the Project area or the associated property agreements, mineral tenure, surface rights, and royalty arrangements. The QPs have relied fully on information provided by Newcore for Sections 1.2 and 4 of this Report. Details related to property agreements, mineral tenure, surface rights, and royalties are also incorporated into the economic analysis presented in Section 22.

3.3 Environmental, Permitting and Social Considerations

The QPs have relied upon Abakus Geosocial Consult Limited, for matters pertaining to environment, permitting, and social impact (Section 20). The QPs believe that it is reasonable to rely on this expert, based on the assumption that the expert has the necessary education, professional designations, and relevant experience on matters relevant to the technical report.

3.4 Taxation

The QPs have fully relied upon, for, information supplied by Newcore related to taxation as applied to the financial model, as received by email titled 2026.06 - Newcore - Enchi Gold Project - 2026 PFS Cash Flow Model - FINAL. This information is used in the economic analysis (Section 22).

3.5 Markets

The QPs have fully relied upon, information derived from Newcore for use in Section 19 of the Report.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Project is located 290 km west of the capital of Accra and 50 km southwest of the Chirano Mine operated by Asante Gold Corporation (Figure 4.2.1). The Project is centered on 5°47′ North latitude and 2°42′ West longitude. The town of Enchi is located 10 km to the west of the Project.

The Project covers a 40 km strike length of the eastern margin of the Sefwi Belt, stretching from the Côte d'Ivoire border in the southwest to neighbouring claims to the northeast. The Project comprises nine prospecting licences, totaling 248 km² located in the Enchi and Aowin Suaman Districts, in the southwestern region of Ghana.

4.2 Mineral Disposition

The nine licences that make up the Project are summarized in Table 4.2.1 and are also displayed in Figure 4.2.2. Licence boundaries are defined by a list of latitude and longitude coordinates of the corners (pillar points) submitted to the Minerals Commission (Mincom). The boundaries are not physically marked on the ground and have not been surveyed by Newcore.

The application process for a prospecting licence, which is required for drilling and excavation work, is as follows:

- Application submitted to Mincom.
- Mincom completes paperwork and checks maps.
- Mincom generates a letter that is sent to the local authorities and is posted for three weeks; this provides an opportunity for objections to the licence application.
- Local authorities write back to Mincom if no objections are presented.
- Application proceeds to a technical committee for review.
- Upon technical committee approval, the licence is prepared and sent to the Mincom Minister for signature.

The entire process typically takes two years or more to complete. Once an application is submitted, work under the licence is allowed to proceed.

A series of letters were received from the Ministry of Lands and Natural Resources in April 2026 advancing the renewals of the Sewum, Enkye, Nyamebekyere (Nyam), Yehiwakrom, and Omanpe Licences. The Enkye and Yehiwakrom Licences have officially been extended until 8 April 2029.

For the Sewum, Nyam, and Omanpe Licences, the Ministry has sent letters to the Minerals Commission confirming that the licences will be extended upon payment of the required processing fee, which is being completed. It is anticipated that the Abotia Licence will also be extended.

During the renewal process, the licences are not subject to a reduction in land size. Holding fees required to maintain the licences on an annual basis total approximately $100,000 as of the date of this Report.

Nyame Esa, Anguzu and Nkwanta are ongoing licence applications and are required to proceed through the full application process. These licences were submitted in 2019 and letters for each of the applications from Mincom to the Minister recommending approval were received in September 2024.

Figure 4.2.1 Location Map of the Enchi Gold Project



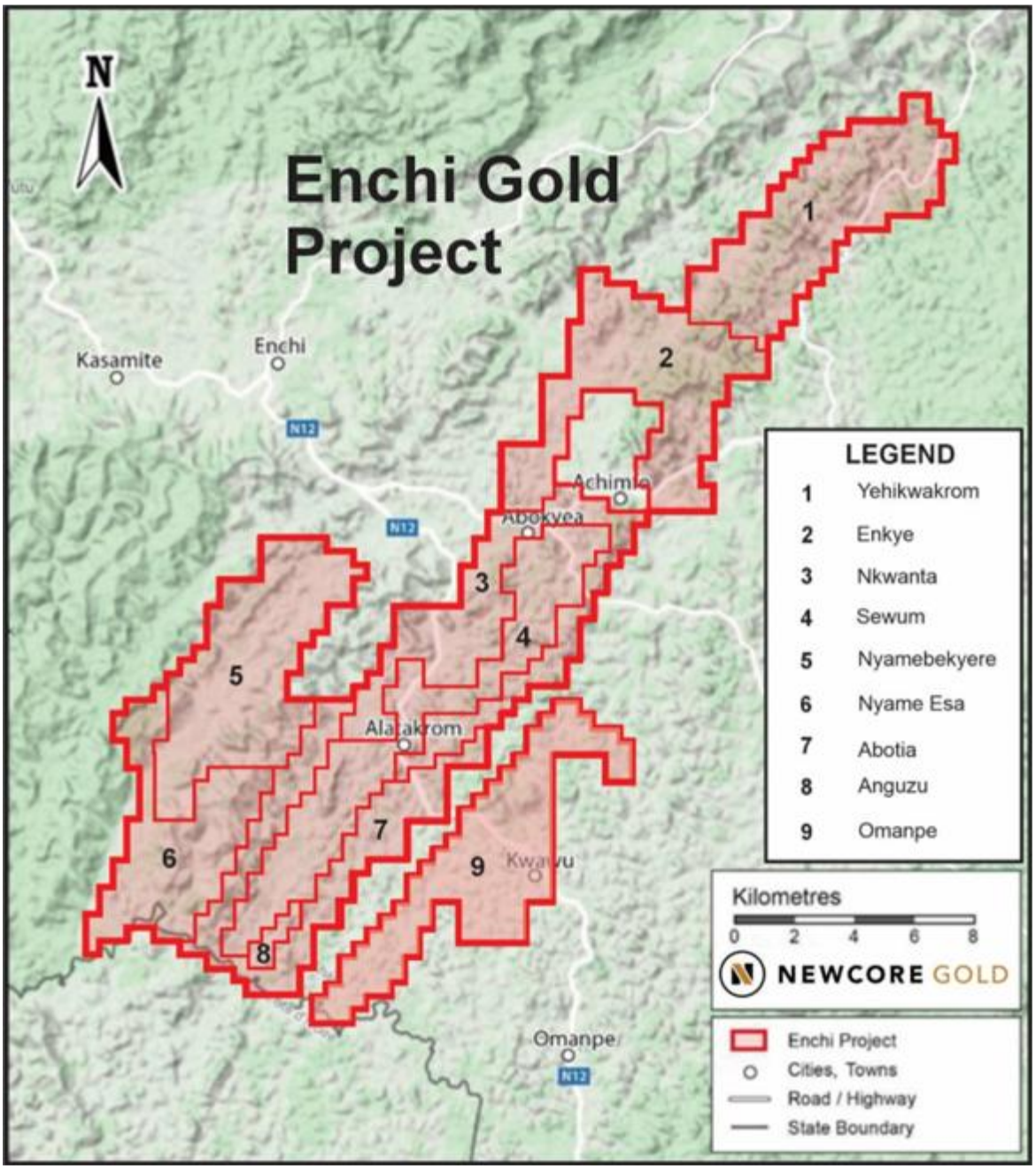
Source: Newcore, 2024

Table 4.2.1 List of Project Licences

Name	Type	Number	Area km²	Current Holding Company	Status
Sewum	PL	PL 2/424	32.55	Cape Coast Resources Ltd.	50% shed off completed. All maps and application for 3-year extension submitted. Confirmation of extension until 2029 sent from Ministry to Mincom on 8 April 2026. Process proceeding.
Enkye	PL	PL 2/404	34.65	Cape Coast Resources Ltd.	8 April 2026 Licence extended to April 8, 2029.
Nyamebekyere	PL	PL 2/406	35.91	Cape Coast Resources Ltd.	Confirmation of extension until 2029 sent from Ministry to Mincom on 8 April 2026. Process proceeding.
Yehikwawkrom	PL	PL 2/405	29.82	Cape Coast Resources Ltd.	8 April 2026 Licence extended to April 8, 2029.
Abotia	PL	PL 2/119	25.83	Cape Coast Resources Ltd.	Application for 3-year extension submitted 3 November 2019. Renewal letter confirming 3-year extension received on 11 April 2023 with extension to 2026. Application for extension to 2029 submitted.
Omanpe	PL	PL 2/436	32.13	Cape Coast Resources Ltd.	Confirmation of extension until 2029 sent from Ministry to Mincom 8 April 2026.
Nyame Esa	PL	not assigned	24.36	Boin Resources Limited	Re-application for portion of 50% shed off from original Nyamebekyere PL by BRL Mineral Commission has forwarded recommendation for approval to Minister for signature of PL
Anguzu	PL	not assigned	1.89	Boin Resources Limited	Re-application for portion of 50% shed off from original Nyamebekyere PL by BRL Mineral Commission has forwarded recommendation for approval to Minister for signature of PL
Nkwanta	PL	not assigned	30.87	Boin Resources Limited	Re-application for the 50% shed off from original Sewum PL by BRL. Mineral Commission has forwarded recommendation for approval to Minister for signature of PL.

Cape Coast Resources Ltd. and Boin Resources Limited are wholly-controlled subsidiaries of Newcore.

Figure 4.2.2 Enriched Licence Map



Source: Newcore, 2026

4.3 Tenure Rights

Edgewater Exploration Ltd. (Edgewater) executed a definitive Option Agreement dated 5 May 2010 that outlined the terms of an Option-Joint Venture agreement with Red Back, whereby Edgewater at the time could earn a 51% interest in Red Back's ownership interest in the Project.

To earn the 51% interest, Edgewater had to spend a total of CA$5.0 M on work expenditures on the Project within 26 months, including CA$2.0 M in the first 14 months. Edgewater would be the operator of the Option-Joint Venture agreement and would continue to be the operator of the Joint Venture as long as Edgewater held the larger equity interest in the Joint Venture.

On 17 September 2010, Kinross Gold Corporation (Kinross) announced that it had successfully completed the transaction to acquire all outstanding shares of Red Back for $7.1 B, and that Red Back would become a wholly owned subsidiary of Kinross.

On 22 May 2012, Edgewater announced that it had completed the earn-in requirements of the 2010 Option Agreement with Kinross. As a result, Edgewater then held a 51% interest in the Enchi Gold Project, and a joint venture company was to be formed.

On 22 May 2014, Newcore (at the time named Pinecrest Resources Ltd.) announced that it had entered into an agreement to earn 100% interest of the Project from Kinross and Edgewater. The terms of the transaction were as follows.

For Newcore to acquire Kinross' 49% interest:

- Red Back to receive 19.9% of the issued and outstanding common shares of Newcore post-closing of the transaction.
- Red Back to retain a 2% net smelter return (NSR) royalty on production from the Project with Newcore retaining the right at any time to buy back 50% (1%) of the NSR royalty for $3.5 M. Newcore's buyback option was subsequently transferred to Sandstorm Gold Ltd. (now Royal Gold Inc.) in 2014 while Kinross also sold its 2% NSR royalty to Maverix Metals (now Triple Flag Precious Metals Corp.) in 2019.
- Red Back to receive a payment of $10 per ounce of gold on any new NI 43-101 Measured and Indicated Resource estimate included in a Feasibility Study, or on any ounce of gold mined, whichever occurs first. Such amount would be payable in cash or, at Newcore's sole discretion, common shares of Newcore provided that, Newcore shall not be entitled to elect to pay in common shares if such issuance would result in Red Back holding more than 20% of the issued and outstanding shares of Newcore.
- Red Back to have first right to process material from the Project at its Chirano Mill if toll processing is considered.

- Red Back to receive 5,000,000 share purchase warrants priced at CA$0.40 per warrant exercisable for a 5-year term from closing of the transaction. These warrants subsequently expired unexercised.

For Newcore to acquire Edgewater's 51% interest:

- Upon closing of the transaction, Edgewater to receive one Newcore post-consolidated common share (the Acquisition Shares) for every five common shares of Edgewater issued and outstanding on the closing, which represented approximately 40% of the issued common shares of Newcore post-closing of the transaction.

- Edgewater was to agree to distribute the Acquisition Shares pro-rata to its shareholders as soon as reasonably practicable after the closing of the transaction.

- Newcore was to pay to Edgewater a cash payment of CA$150,000.

On 5 December 2014, Newcore announced that it had completed the acquisition of a 100% interest in the Enchi Gold Project from Edgewater Exploration Ltd. and Red Back Mining Ghana Limited, an indirectly wholly owned subsidiary of Kinross Gold Corporation at the time. The Government of Ghana is entitled to a 10% free carried interest in the Project.

On 6 August 2020, the company officially announced a company name change from Pinecrest Resources Ltd. to Newcore Gold Ltd.

4.4 Royalties and Related Information

A 2% NSR royalty on production from the Project is held by Triple Flag Precious Metals Corp. (with 1% subject to a buy-back option for a lump sum payment of $3.5 M at any time held by Royal Gold Inc.).

A royalty on revenues is due to the Government of Ghana (calculated based on the international market value of gold) on the revenues from gold production on the Property covered by the exploitation permit. The royalty rate is based on a sliding-scale structure ranging from 5% to 12%, with a rate of 12% when the gold price is at or above $4,500/oz.

4.5 Environmental Liabilities

The QP is not aware of any known environmental liabilities on the Property. Newcore is not responsible for small-scale artisanal and alluvial mining that has occurred across the Property and maintains good relations with the local communities.

4.6 Permits

All required permits for conducting exploration on the licences have been granted or have been applied for and are awaiting government approval.

4.7 Other Relevant Factors

In areas where there is no existing surface holder, Newcore is not required to pay any compensation or fees. In areas where there is an established surface holder, Newcore is required to pay compensation when properties are disturbed; in most cases, this is related to the disturbance of crops during establishment of access for exploration activities.

The risk to the Project would come in the form of the licence applications being denied by Mincom and work needing to be halted although this is not anticipated.

There are no other significant risk factors which could affect access, title, or the right or ability to perform work. Newcore has completed successive and extensive exploration programs covering the majority of the licences over the last 12 years.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Site Topography, Elevation and Vegetation

The Project area is primarily drained by the Tano River and its tributaries, which flow generally in an easterly direction. Much of the Project area comprises steep topography incised by river tributaries with scattered flat plateaus with an average height of about 300 metres above sea level (masl).

A portion of the Project area is covered by subsistence farmland. The main food crops grown locally are cocoa, plantain, maize, cocoyam, cassava, and rice (Figure 5.1.1).

Figure 5.1.1 Cocoa Plantation



Source: Newcore, 2021

The northern part of the Project lies adjacent to forest reserves, and is covered by tall, primary, semi deciduous rain forest (Figure 5.1.2). Most of this area, which does not form a part of the Project, is reserved for commercial timber production.

Figure 5.1.2 Local Landscape



Source: Newcore, 2021

5.2 Access

The Project is located in the southwestern region of Ghana and is accessed from Accra on sealed roads via the regional port city of Sekondi (Takoradi) or the mining centre of Kumasi. From either of these centres, access to the town of Enchi (population of 20,451 in 2021), the capital of Aowin-Suaman district, is available by paved road (Elubo-Enchi Road or the Asankragua-Enchi Road). Access through the remainder of the Project area is by dirt and gravel roads (Figure 5.2.1).

Accra has daily international flights to and from Europe, the U.S., and various African locations. Domestic flight services are available with scheduled flights between Accra and Kumasi, which is located 170 km northeast of the Project.

Figure 5.2.1 Project Access Map



Source: Newcore, 2024

5.3 Climate

The Aowin District, within which the Project is based, is situated in the Wet Semi Equatorial Climatic Zone. The climate is typically warm and humid with a mean-monthly temperature of 27°C. There are two rainy seasons: the major rainy season from May to July, and a shorter rainy season from September to October. The district receives an annual rainfall of between 1,500 and 1,800 mm. During the dry season, predominantly December to March, Harmattan winds (dry hot continental fronts from the Sahara) blow over the country resulting in drier warm days and cool nights.

Rainfall data is based on historic government measurements from a weather station located in the town of Enchi and has been confirmed by a regional weather station and on-site rainfall measurements from three rainfall stations installed by Newcore at each of the Sewum, Boin, and Nyam Deposits. Additionally, two weather stations have been installed at Mochekrom and Sewum which monitor rainfall, temperature, wind, wind gusts, wind direction, humidity, heat index, and solar radiation.

Distribution of the rainfall at Enchi, as detailed in the most recently completed Baseline Study, is such that the higher rainfall months are April to July and September to October, with no month that is excessively higher than average. The overall effect of the rainfall is mitigated by the evaporation rates which are particularly high such that for seven months (January to April, August, November to December) evaporation exceeds the average rainfall. Annual evaporation in the Enchi area exceeds the annual average rainfall.

Exploration and mining operations can be conducted on the Project year-round.

5.4 Infrastructure

The Project area has moderate infrastructure. A paved road crosses the central portion of the Project leading to the town of Enchi. The remainder of the Project is serviced by a series of gravel and dirt roads. The district capital of Enchi is located 10 km west of the Project.

Access to the Project site is by an existing gravel road that connects the village of Sewum and Alatakrom. The Ghana National Highway N12 passes through the village of Alatakrom which is located within the Project site. The nearest seaport, Takoradi, is 205 km southeast of the Property by way of the N1 and N12 paved highways. Accra, the capital of Ghana and the main point of entry by sea or air, is 427 km east of the Project by road.

The town of Enchi provides most services including accommodations, restaurants, a hospital, gas stations, and numerous stores for local supplies. Potable water must either be trucked into the area or supplied via water wells.

The area has a fixed telephone line and mobile phone service tower. Mobile cell service exists over much of the Project area.

The town of Enchi is located 77 km north of the substation at Elubo, serviced by a 225 kV line, and 122 km southwest of the substation at Asawinso, serviced by a 161 kV line. Ghana's eight thermal plants have a total generation capacity of 1,684 MW; three of these plants are located 200 km southeast of Enchi in the coastal city of Takoradi. These plants operate on light crude oil or natural gas sourced from Ghanaian producers.

Genser Energy operates four natural gas pipelines totalling 430 km. The pipelines feed power plants containing gas turbines which are built and operated by Genser Energy. Genser Energy currently operates power plants at five separate mines within Ghana, including the Chirano Gold Mine (owned by Asante Gold Corporation) which is located 50 km northeast of the Project.

The region has a long history of mining, and there is a large population base of skilled and unskilled labour to draw upon for exploration and development programs.

Modern seaports at Takoradi and Tema are located 207 km and 447 km southeast of the Project, respectively, and provide logistical support for the development and construction of large-scale mines and industrial projects in the country.

Generally, there is sufficient availability of surface rights for potential future mining operations, including potential tailings and waste disposal areas, potential processing plant sites, power, mining personnel, as well as potential sources of water from existing drainages, rainfall and water wells.

6.0 HISTORY

6.1 Introduction

The exploration activities across the entire Project area date back to the 1800s, with activities completed sporadically and by various individuals as well as companies.

Alluvial and bedrock gold were prospected and exploited by several generations of galamsey (local artisanal gold miner) workings to the present day. European companies explored, developed, and mined in several phases since 1900. The result is that erratic gold in quartz vein mineralization was 'opened up' in numerous pits, shafts, and drives, notably at the Sewum, Tokosea, Alatakrom, Achimfo, Nkwanta, and Kojina Hill prospects. Only the historic Sewum and Tokosea mines appear to have any significant development and production history although this is poorly recorded. Since the 1940s, artisanal mining activities have continued in the area on a very limited scale. No modern commercial-scale mining has been carried out on the Project.

Table 6.1.1 summarizes the exploration activities that have taken place within the boundaries of the Project as currently held by Newcore. Due to the scattered nature of the work and the various licence holders, the QP cautions that the history may not be complete. Most information prior to 2010 was derived from reports and digital data acquired from EQ Resources, Leo Shield Exploration Ghana NL (Leo Shield), Mutual Ghana Ltd. (Mutual), and Kinross. Trenching and drilling procedures and results are summarized in Sections 9 and 10, respectively. Metallurgical testwork is discussed in Section 13.

The extensive work completed by the previous landholders has resulted in the identification of more than 25 targets, with 15 advanced gold zones or prospects which are described in detail in Section 7.

Table 6.1.1 Project History

Year	Company	Activities
1987	EQ Resources	2,837 soil samples collected on a 100 m x 25 m spaced grid.
1993	Mt. Edon	3,260 soil samples collected on a 6 km by 3 km spaced grid, followed by a 100 m x 25 m spaced grid. 250 rock chip and float samples collected.

Year	Company	Activities
1994-1997	Mutual	Spot imagery. Helicopter magnetic and electromagnetics on 100 m spaced lines. Fixed wing magnetic and radiometric survey on 200 m spaced lines. 2,837 soil samples on 100 m x 25 m grid spacing. 2,257 soil samples on 200 m x 40 m spaced grid. 34 trenches totalling 2,396 m. 6 diamond drillholes (DDH) totalling 464 m. 18 Reverse Circulation (RC) drillholes totalling 1,202 m.
1995-1998	Leo Shield	14,470 soil samples on 400 m x 50 m spaced grid. 89 trenches totalling 10,240 m. Adit sampling at Kojina Hill and Achimfo. Stream sediment sampling (76 pits). 121 RC holes totalling 7,621 m. 49 rotary air blast (RAB) holes totalling 2,028 m.
2003	Red Back	Historical data compilation and assessment work.
2004	Red Back	237 regional stream sediment samples. 16,728 soil samples. 148 rock chip samples.
2005	Red Back	695 soil samples. 69 trenches totalling 5,750 m. 102 RAB holes totalling 5,261 m. 80 RC holes totalling 9,715 m.
2006	Red Back	Ground magnetic survey. Induced polarization (IP) survey. 2,221 soil samples. 38 trenches totalling 3,564 m. 217 RAB holes totalling 7,182 m. 73 RC holes totalling 7,403 m.

<table>
<tr><th>Year</th><th>Company</th><th>Activities</th></tr>
<tr><td>2011</td><td>Edgewater</td><td>Edgewater optioned the Project from Red Back (Red Back was subsequently acquired by Kinross).
9,441 soil samples over 461-line km.
12 trenches at Nyam totalling 396 m.
3 trenches at Sewum totalling 781 m.
8 trenches at Boin totalling 359 m.
7 trenches at Eradi totalling 1,294 m.
Versatile Time Domain Electromagnetic (VTEM) / magnetic / radiometric survey totaling 3,084 –line-km.
182 DDH and 13 RC holes totalling 23,697 m.
Resource estimation completed on Boin, Sewum and Nyam.</td></tr>
<tr><td>2012</td><td>Edgewater</td><td>25 RC holes totalling 4,058 m.
Bottle roll tests.
Soil and rock sampling, auger drilling, and trenching.</td></tr>
<tr><td>2014</td><td>Pinecrest</td><td>Acquisition of the Project from Edgewater and Kinross by Newcore (formerly known as Pinecrest).</td></tr>
<tr><td>2015</td><td>Pinecrest</td><td>Completion of an initial Preliminary Economic Assessment (PEA) Study.</td></tr>
<tr><td>2017</td><td>Pinecrest</td><td>28 RC holes totalling 3,406 m.</td></tr>
<tr><td>2020</td><td>Newcore</td><td>Company changed name from Pinecrest Resources to Newcore.

<table>
<tr><th rowspan="2">Zone</th><th colspan="2">RC</th><th colspan="2">DDH</th><th colspan="2">Trenches</th></tr>
<tr><th>Holes</th><th>m</th><th>Holes</th><th>m</th><th>Holes</th><th>m</th></tr>
<tr><td>Boin</td><td>26</td><td>4,269</td><td>1</td><td>61</td><td></td><td></td></tr>
<tr><td>Sewum</td><td>10</td><td>1,375</td><td></td><td></td><td></td><td></td></tr>
<tr><td>Sewum South</td><td>10</td><td>1,375</td><td></td><td></td><td>3</td><td>234</td></tr>
<tr><td>Nyam</td><td>8</td><td>1,030</td><td></td><td></td><td></td><td></td></tr>
<tr><td>Kwakyekrom</td><td>9</td><td>1,080</td><td></td><td></td><td></td><td></td></tr>
</table>

Resource estimation completed on Boin, Sewum and Nyam.
Bottle roll tests.</td></tr>
</table>

<table>
<tr><th>Year</th><th>Company</th><th>Activities</th></tr>
<tr><td>2021</td><td>Newcore</td><td>
<table>
<tr><th rowspan="2">Zone</th><th colspan="2">RC</th><th colspan="2">DDH</th><th colspan="2">Trenches</th></tr>
<tr><th>Holes</th><th>m</th><th>Holes</th><th>m</th><th>Holes</th><th>m</th></tr>
<tr><td>Boin</td><td>121</td><td>18,177</td><td>17</td><td>4,535</td><td></td><td></td></tr>
<tr><td>Sewum</td><td>93</td><td>13,506</td><td>15</td><td>4,791</td><td></td><td></td></tr>
<tr><td>Nyam</td><td>84</td><td>12,799</td><td>12</td><td>4,258</td><td></td><td></td></tr>
<tr><td>Kwakyekrom</td><td>59</td><td>9,714</td><td>2</td><td>640</td><td></td><td></td></tr>
<tr><td>Tokosea</td><td>23</td><td>2,524</td><td></td><td></td><td></td><td></td></tr>
<tr><td>Kojina Hill</td><td>5</td><td>670</td><td></td><td></td><td></td><td></td></tr>
<tr><td>Eradi</td><td>14</td><td>2,190</td><td></td><td></td><td></td><td></td></tr>
<tr><td>Sewum South</td><td></td><td></td><td></td><td></td><td>17</td><td>3,726</td></tr>
<tr><td>Nkwanta</td><td></td><td></td><td></td><td></td><td>6</td><td>1,411</td></tr>
<tr><td>Nyam</td><td></td><td></td><td></td><td></td><td>2</td><td>26</td></tr>
<tr><td>Eradi</td><td></td><td></td><td></td><td></td><td>1</td><td>48</td></tr>
</table>
Bottle roll and column tests.
Resource estimation on Boin, Sewum, Nyam and Kwakyekrom.
Completed an updated PEA Study.
</td></tr>
<tr><td>2022</td><td>Newcore</td><td>
<table>
<tr><th rowspan="2">Zone</th><th colspan="2">RC</th><th colspan="2">Trenches</th></tr>
<tr><th>Holes</th><th>m</th><th>Holes</th><th>m</th></tr>
<tr><td>Boin</td><td>7</td><td>832</td><td></td><td></td></tr>
<tr><td>Tokosea</td><td>59</td><td>7,714</td><td></td><td></td></tr>
<tr><td>Nkwanta</td><td></td><td></td><td>6</td><td>1,021</td></tr>
<tr><td>Kojina Hill</td><td></td><td></td><td>12</td><td>2,125</td></tr>
<tr><td>Agyeikrom</td><td></td><td></td><td>8</td><td>1,852</td></tr>
<tr><td>Tokosea</td><td></td><td></td><td>2</td><td>74</td></tr>
</table>
Density measurements.
Structural geology review.
Bottle roll and column tests.
Drone topographic survey over Boin, Sewum, and Nyam.
</td></tr>
</table>

<table>
<tr><th>Year</th><th>Company</th><th>Activities</th></tr>
<tr><td>2023</td><td>Newcore</td><td>5 DDH holes at Nyam totalling 2,155 m.
Metallurgical trenches at Nyam, Boin, and Sewum.
Density measurements.
Fresh material metallurgical testing for Nyam and Sewum.
Bottle roll and column tests.
Pilot Heap Test completed at the University of Mines and Technology (UMaT).
Completed an updated Mineral Resource Estimate.</td></tr>
<tr><td>2024</td><td>Newcore</td><td><table><tr><th rowspan="2">Zone</th><th colspan="2">RC</th></tr><tr><th>Holes</th><th>m</th></tr><tr><td>Boin</td><td>48</td><td>7,152</td></tr><tr><td>Sewum</td><td>33</td><td>2,922</td></tr><tr><td>Nyam</td><td>13</td><td>1,415</td></tr><tr><td>Kojina Hill</td><td>2</td><td>230</td></tr></table>
Metallurgical testing including column tests and bottle rolls.
In-fill soil sampling.
Baseline environmental and social studies.
Drone topographic and magnetic study (Kojina Hill only).
Geotechnical and hydrogeological work.
Completed an updated PEA Study.</td></tr>
<tr><td>2025</td><td>Newcore</td><td><table><tr><th rowspan="2">Zone</th><th colspan="2">RC</th><th colspan="2">DDH</th></tr><tr><th>Holes</th><th>m</th><th>Holes</th><th>m</th></tr><tr><td>Boin</td><td>37</td><td>5,191</td><td>15</td><td>4,845</td></tr><tr><td>Sewum</td><td>68</td><td>7,039</td><td>9</td><td>1,769</td></tr><tr><td>Nyam</td><td>13</td><td>1,415</td><td>4</td><td>1,691</td></tr><tr><td>Kwakyekrom</td><td>24</td><td>3,843</td><td></td><td></td></tr><tr><td>Kojina Hill</td><td>12</td><td>1,684</td><td></td><td></td></tr></table>
Metallurgical testing including column tests and bottle rolls.
In-fill soil sampling.
Baseline environmental and social studies.
Topographic surveys (areas between resource zones and; proposed processing area).
Geotechnical and hydrogeological work.</td></tr>
</table>

<table>
<tr><th>Year</th><th>Company</th><th>Activities</th></tr>
<tr><td>2026</td><td>Newcore</td><td>
<table>
<tr><th rowspan="2">Zone</th><th colspan="2">DDH</th></tr>
<tr><th>Holes</th><th>m</th></tr>
<tr><td>Boin</td><td>15</td><td>5,914</td></tr>
<tr><td>Sewum</td><td>11</td><td>3,848</td></tr>
<tr><td>Nyam</td><td>16</td><td>5,612</td></tr>
</table>
Baseline environmental and social studies.
Geotechnical and hydrogeological work.
Diamond drilling at Boin, Sewum, and Nyam
Updated Mineral Resource Estimate
</td></tr>
</table>

Resource estimations were previously completed on the Project in 2011, 2015, 2020, 2021, and 2023. The most recent historic Mineral Resource Estimate (completed for Newcore by Todd McCracken from BBA Consultants and dated effective 25 January 2023) is summarized in Table 6.1.2. No Mineral Reserves were reported for the Project at the time.

Table 6.1.2 Summary of 2023 Mineral Resources (McCracken and Meadows Smith, 2023)

Classification	Zone	Open Pit (OP)/ Underground (UG)	Tonnes	Au g/t	Au Oz
Indicated	Sewum	OP	20,925,000	0.48	323,300
	Boin	OP	13,020,000	0.62	258,200
	Nyam	OP	7,791,000	0.65	162,000
	Total	OP	41,736,000	0.55	743,500
Inferred	Sewum	OP	21,154,000	0.47	317,600
		UG	644,000	2.68	55,500
	Boin	OP	15,884,000	0.68	349,600
	Nyam	OP	1,852,000	0.68	40,600
		UG	829,000	2.41	64,000
	Kwak	OP	3,970,000	0.64	81,000
		UG	274,000	1.86	16,300
	Tokosea	OP	1,949,000	0.75	46,900
	Total	OP/UG	46,556,000	0.65	972,000

Cut-offs varied between 0.14 and 0.27 g/t Au for open-pit resources; underground resources used a cut-off grade of 1.50 g/t Au.

Newcore considers the previous estimates to be relevant as they provide an indication of the potential of the Project; however, the Qualified Person (QP) has not done sufficient work to classify these previous estimates as current mineral resources or reserves, and Newcore does not consider these estimates as current mineral resources or reserves.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Enchi concession is located within southwest Ghana and straddles the boundary between the Sefwi Volcanic Belt to the west and the Kumasi Sedimentary Basin to the east. The Sefwi Belt and Kumasi Basin are comprised predominantly of Birimian-age rocks (2.17 to 2.18 Ga) (Davis et al., 1994) (Figure 7.1.1).

The Sefwi Belt is dominated by mafic volcanics, metasediments, and intrusive granitoids that are sandwiched between sedimentary basins (Sunyani Basin to the west and the Kumasi Basin to the east). The Sefwi Belt is traceable for 260 km along strike and is usually 30 to 60 km wide. The metavolcanic and metasedimentary sequences are believed to be contemporaneous, with the sediment deposited in basins eroded from the adjacent volcanic terrains (Asiedu et al., 2004).

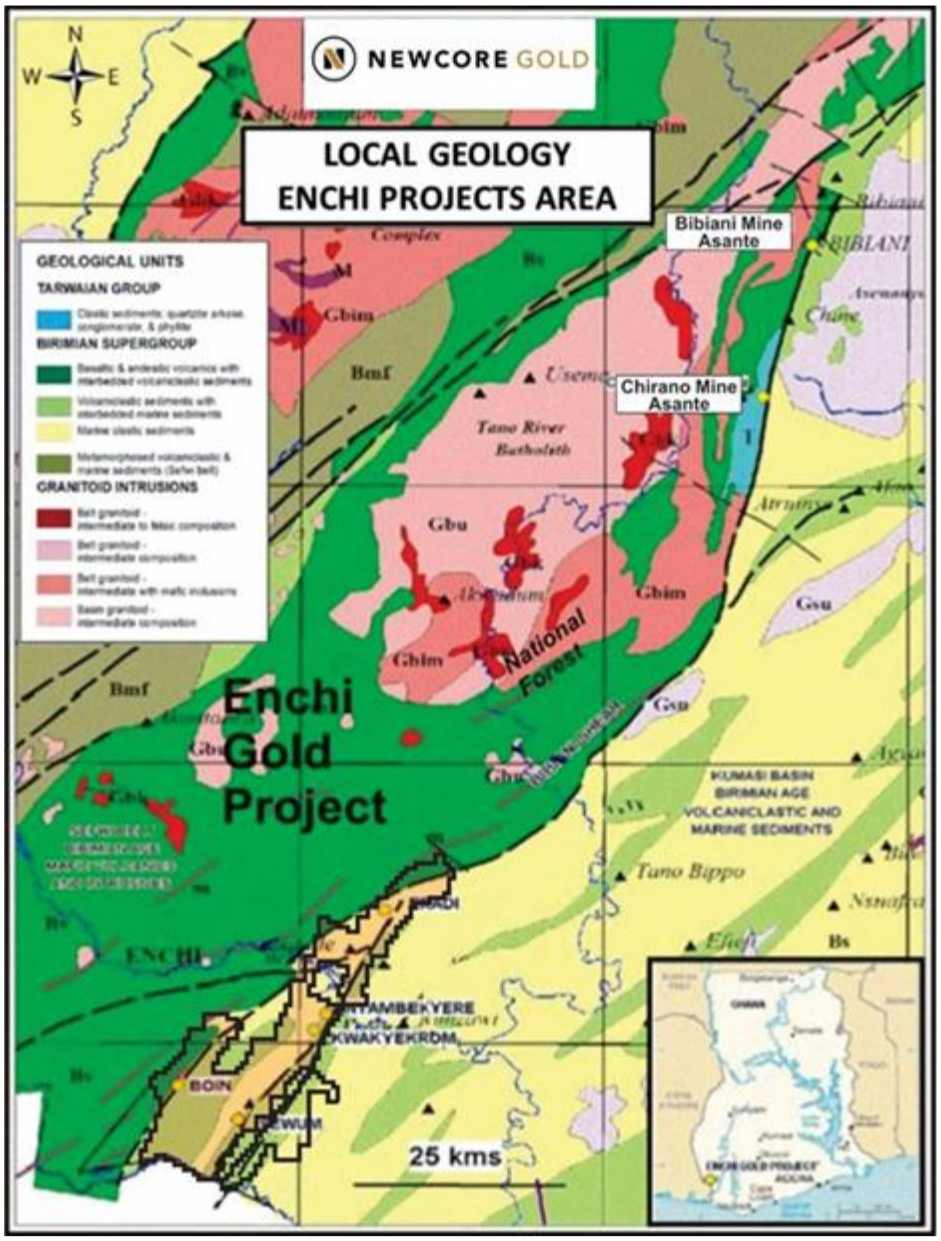
The Kumasi Basin is characterized by wide sequences of marine clastic sediments (quartzite, conglomerates, phyllites). Both the Birimian sediments and volcanics have been extensively metamorphosed to greenschist facies, locally to amphibolite facies. The boundary between the volcanic belts and basins can be gradational yet, is typically faulted with the faults most likely representing basin margin growth faults along which basin subsidence occurred (Hirdes and Leube, 1989).

Granitoid intrusions are common within the belt and basin terrains and can be divided into two types: Belt Type (Dixcove) and Basin Type (Cape Coast) granitoids. Belt type granitoids (2,180 Ma) range from tonalite to granodiorite in composition and are confined to the metavolcanic belts. Basin granitoids (approximately 2,116 to 2,088 Ma) are mainly granodiorite in character and contain more potassium and rubidium relative to the belt granitoids and are concentrated in the central portions of the Birimian metasedimentary basins (Hirdes and Leube, 1989).

Extensive faulting occurred along the margins of the volcanic-sediment belts. Observed at local and regional scales, these northeast-trending structures are fundamentally important in the development of gold deposits for the region. The major shear system within the Enchi concession at the boundary of the Sefwi Belt and Kumasi Basin is termed the Bibiani Shear Zone. Gold deposits are located in third-order structures that splay-off the second-order structures and are sub-parallel to the overall trend of the Bibiani Shear Zone. The Bibiani Shear Zone has been traced for 40 km on the Project area. Major structures within the Project are named from west to east, the Bibiani Shear (BS), the West Sewum Shear (WSS), and the Nyam Shear (NS).

Multiple tectonic events have affected virtually all Birimian rocks. The dominant event is compressional folding and thrusting from the Eburnean Orogeny (2.1 to 2.2 Ga) (Schofield, 2006; Eisenlohr, 1989).

Figure 7.1.1 Regional Geology Map



Source: Newcore, 2024 (modified from Davis et al., 1994)

7.2 Project Geology and Structure

The Project covers 40 km of the belt-basin contact on the east side of the Sefwi Volcanic Belt, north of the Côte d'Ivoire border. The contact is marked by a major fault known as the Bibiani Shear Zone, which also hosts the Chirano and Bibiani gold mines located 50 km and 70 km respectively north of the Enchi licences (Figure 7.2.1).

The Project is characterized by variably degraded laterite to residual soil profiles with minor caps of indurated ferro-duricrust across the main hilltops. Rock outcrops are rare due to the thick tropical weathering and jungle cover. Most rock exposures are found in road cuttings and by trenching.

Numerous other major faults splay off the Bibiani Shear Zone and pass through the licence area, such as the Boin Fault, Sewum Fault, and Nyam Fault. Many gold deposits in the Enchi District are localized along or adjacent to these structures.

The regional scale shears are believed to have been originally formed as thrusts during northwest-southeast compression with later movements dominated by left-lateral strike slip shearing (Griffis et al., 2002).

The principal rock types found on the Project are defined below:

- Volcanics / Intrusives (MB): massive, very fine-grained, without texture, weathered white to brown, to deep pink and red; igneous rock generally evident as un-deformed rafts; fault-bound, within foliated and sheared volcaniclastics and pelitic sediments.

- Volcaniclastics (SVC): hanging wall, fine to medium-grained, lithic to crystal volcaniclastic wacke, with a characteristic porous, spongy, honeycombed texture. Weathers to light pink and is variably graphitized and foliated to sheared proximal to the late faults.

- Greywacke (SGW): dark to light grey, coarse- to medium-grained with fine clay matrix along with poorly sorted, angular grains of quartz, feldspar, and rock fragments.

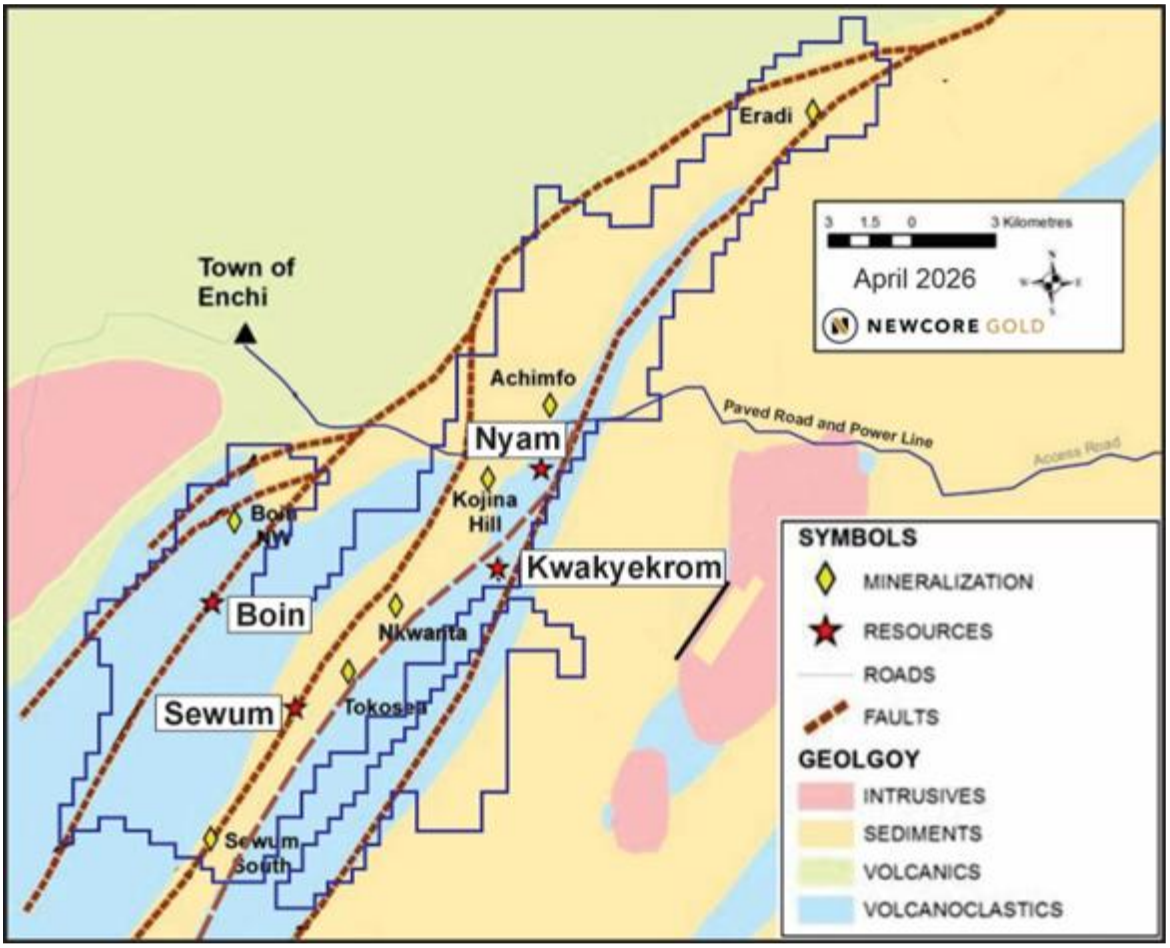
- Turbidites (SPH): footwall, metre-thick, cyclically bedded, turbidite sequence of graded, fine to medium-grained, grey to black, phyllitic pelite-psammite beds. The finer pelite horizons are preferentially strained and the coarser units are preferentially fractured.

- Graphitic Phyllites (SPG): black, very fine to fine-grained carbonaceous and graphitic altered phyllites and schists. Each of the host rock-types may be preferentially graphitized ± silicified and sheared proximal to the reactivated faults and shears, becoming increasingly assimilated to SPG. Within and proximal to the main SPG deformation zones, texture was the main discriminating feature used to distinguish and map the SVC-SPH contact.

- Quartz Veins (QV): massive 0.5 to 5 m wide, white to smoky, blue polyphase quartz veins variably faulted and graphitized and mineralized. The major quartz zones represent the main hanging wall deformation zone developed as a result of the progressive movement along the basal contact shear zone.

- Basic, Intermediate and Felsic Dykes and Sills: coarse-grained granodiorite to diorite and finer grained equivalent andesites to dolerites have been logged. The felsic and intermediate dykes tend to be layered parallel, altered, and structurally deformed within the surrounding host volcanics and sediments. The dolerites are generally much later, crosscutting. They were traditionally mapped as post-deformational, though they are often crosscut and displaced by late reactivation. There is evidence for multiple generations of dolerites through to post-Cretaceous times.

Figure 7.2.1 Project Geology

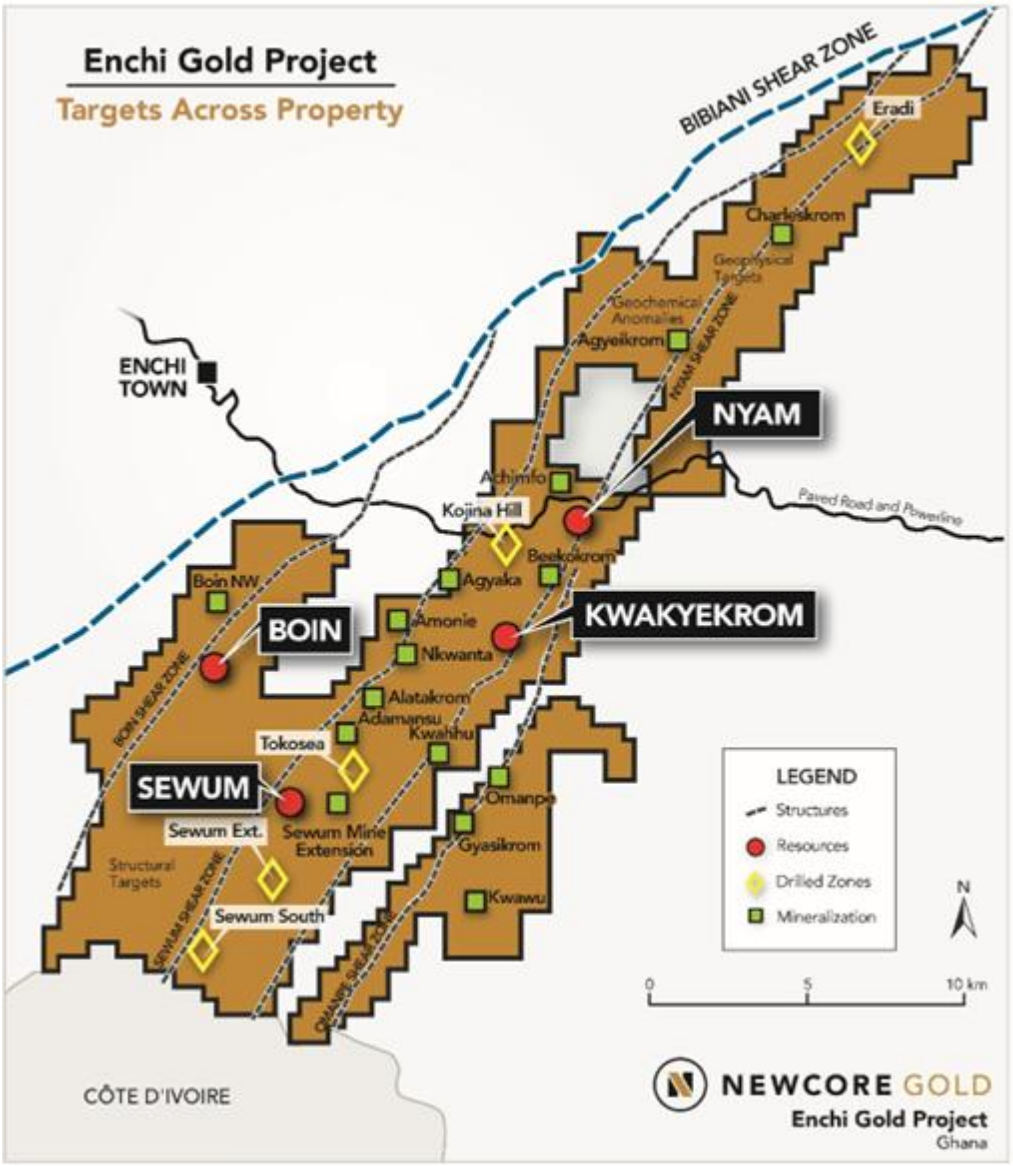


Source: Newcore, 2026

7.3 Mineralization

More than 25 targets have been identified, and fifteen advanced gold anomalous areas or prospects are described in detail in the following section. The locations of the zones are illustrated in Figure 7.3.1.

Figure 7.3.1 Summary of Mineral Zones



Source: Newcore, 2026

7.3.1 Sewum

The Sewum and Sewum Extension prospects are found along the eastern contact of a thrust-bounded volcanic sliver, outcropping 6 km to the east of the Boin Zone. The gold mineralization is associated with late D2 to D4 deformation phases. It is structurally controlled within, and adjacent to, late graphitic faults focused on the margins of poly-phase quartz veins within faults. The veins developed along the axial planes of hinges and limbs of earlier hanging-wall D3 drag folds and/or intrusives.

The Sewum Gold prospects form a continuous 40 km strike length from Sewum Extension, northeast through Kojina Hill and the Nyam Zone, up to the Eradi Prospect in the north.

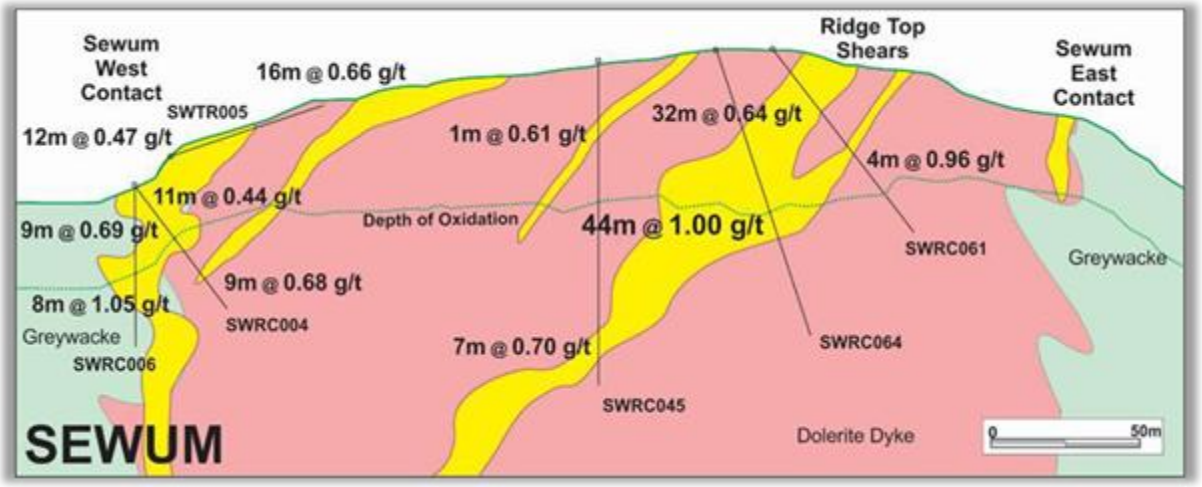
The main relief of Sewum Hill is characterized by a relict and generally thin indurated, duricrust, or ferricrete plateau along the main hilltop, degraded breakaways forming the slope crests and variably mixed and transported upper-slope soils progressing into residual mid- and lower-slope soils. The duricrust mantle is geochemically subdued and potentially ferricrete bearing. Various surrounding hilltops have similar remnant duricrust caps and should be evaluated with care to understand and develop the regolith model for the region. Sewum Hill has a deep weathering profile.

The Sewum setting differs, however, in the scale of shear zones, as compared to those expressed at Boin, and has proportionally more volcanic igneous rocks and late-stage, intrusive intermediate and felsic dykes or sills.

The Sewum prospects are situated along several major thrust zones that crop out across the regional 3 km wide north-south corridor, south of Tokosea. The structures comprise (west to east) (Figure 7.3.2):

- Road Zone (SRZ).
- Sewum Ridge Top Shear Zone (SRTSZ)
- Main Contact Zone (MCZ).
- Sewum Zone (SWZ).
- Sewum-Tokosea Mine Trend (SETO).

Figure 7.3.2 Generalized Geology Section at Sewum



Source: Newcore, 2023

The host rocks at Sewum include interbedded carbonaceous siltstone and sandstone (turbidite). The sediments have been regionally deformed to greenschist facies, are steeply dipping, and typically strike north-northeast (30º) parallel to the regional structural grain. A steeply dipping dolerite dyke 3 km long and up to 500 m wide has been intersected in the drilling and acts as an important host to gold mineralization in the Sewum area.

Three styles of mineralization have been identified at Sewum:

- Quartz – sericite – carbonate replacement of sheared dolerite and sediment localized along moderately (40°) dipping shears hosted within dolerite, e.g., Sewum Ridge Top Shears Zone (SRTSZ).

- Breccia and stockwork formed in sediment and dolerite developed at the margin of the dolerite dyke and replaced and in-filled by quartz-sericite ankerite and minor sulphides, e.g., Checkerboard Hill, East Contact Zone (ECZ), and West Contact Zone (WCZ).

- Minor disseminated arsenopyrite associated with quartz veining and silicification in sheared sediment, e.g., Sewum Shear Zone.

The Sewum Shear Zone represents a major regional structure that can be traced within Ghana for 25 km south from where the shear branches off the Bibiani Shear Zone and continues across the Ghana border into Côte d'Ivoire. The shear has a complex anastomosing geometry with numerous splays and has played a major role in localizing gold mineralization in the Sewum area (e.g., Sewum deposit and Tokosea target currently operated as small-scale mines).

Striking north-northeast, the Sewum Shear is typically vertical to steep west dipping and can be up to 100 m wide. Mylonitic fabric has been observed within the shear zone, in places. Gold mineralization within the Sewum Shear is related to a phase of quartz veining with associated arsenopyrite.

Mineralization is concentrated in discrete zones and appears to be related to an early phase of quartz veining that has been brecciated by later movement along the Sewum Shear.

The dolerite dyke at Sewum has acted as a solid 'node' with the bulk of the regional scale deformation absorbed by the surrounding host fine-grained carbonaceous sediment. Branches of the Sewum Shear have anastomosed around the dolerite dyke and in places mark the contact which is also mineralized.

One of the most significant zones of continuous gold mineralization identified in drilling at Sewum is the Ridge Top Shears Zone (RTSZ), related to a series of close-spaced, moderately dipping shears up to 20 m thick hosted within the dolerite dyke.

The relationship of these shears with the Sewum Shear Zone is unclear but they are most likely temporally related. The shears within the dolerite may be thrust faults or faults that link between the steep shears that anastomose around the dolerite dyke.

The dolerite intrusive emplacement has not been controlled by pre-existing faults; indeed, features typical of intrusive contacts, such as frictional 'intrusive breccia' or hornfelsing of adjacent sediment, have been observed along the dyke's west margin, in addition to the presence of chilled margins within the intrusive.

The age of the dolerite dyke is not certain. However, the partially sheared east contact, spatial relationship with gold mineralization and some drill core features indicative of soft sediment deformation at the intrusive contact, indicate the intrusive was probably emplaced during the Eburnean Orogeny similar to most other mafic intrusives in the region. It is also possible the dyke may have been intruded as a sill along bedding planes and later tilted vertically during regional deformation along with the host sediment.

The size and composition of the intrusives at Sewum are more akin to the 'Belt' style intrusives than the 'Basin' style intrusives which tend to be larger, coarser grained, and felsic in composition (Griffis, et al., 2002).

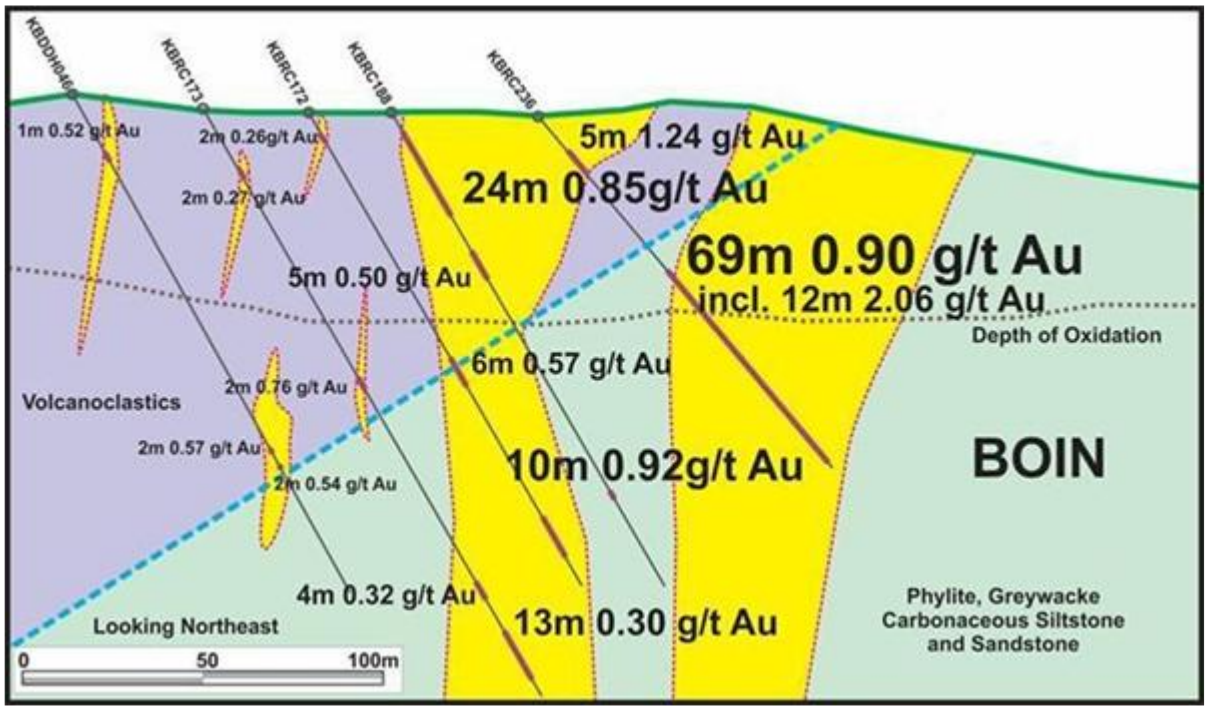
The presence of the dolerite body within the Sewum Shear Zone is significant in that the intrusive represents a more competent rock type compared to the surrounding sediment and is more likely to deform in a brittle manner during faulting and deformation, potentially making a better (more permeable) host to mineralization similar to the Chirano Gold Mine (brecciated granite host).

Mineralized breccia and stockwork are commonly found along the margin of the dolerite dyke (ECZ and WCZ). The breccia is composed of angular clasts of siltstone and dolerite in a clast-supported fabric cemented by quartz, carbonate, and minor pyrite. The breccia texture indicates very little milling and mixing of fragments have occurred and was formed by hydraulic fracturing, probably in response to fault movement near the intrusive contact.

7.3.2 Boin

The Boin Shear Zone is one of a number of major structures that splay off the Bibiani Shear and pass through the Project. The Boin Shear Zone is interpreted as a thrust fault, dipping moderately west, and is responsible for the development of the zone of mineralized quartz veins at Boin. Eleven km of the Boin Shear Zone has been drill-tested at shallow depths over regular intervals across the structure. A generalized section is shown in Figure 7.3.3.

Figure 7.3.3 Generalized Geology Section at Boin



Source: Newcore, 2023

The Boin Shear Zone is formed adjacent to this major second-order, west-dipping, thrust contact between mafic volcanic to volcaniclastic sediments which overthrust turbidites to the east. The whole contact is expressed as a 10 to 30-m wide graphitic shear zone, which trends 025° to 040° and dips west 30° to 70°. The Boin thrust is an early, regionally second-order splay or replication off the main basin-boundary contact further to the west. Multiple sets of crosscutting fabrics, veins, and faults have been recorded within the core and trench logging. The gold is mostly found in the hanging wall quartz zone and is characterized by massive 20 to 30-m wide zones of intensive quartz veining cut and fractured by late, graphitic faults.

There are multiple generations of pyrite developed within the Boin structures. The early, barren, non-auriferous pyrite tends to be intense, well formed, coarse, and cubic. The later, possibly remobilized, auriferous pyrite tends to form as fine to very fine, disseminated cubic crystals within graphitic fault margins, or amorphous ribbons, rims, or coatings within quartz veins.

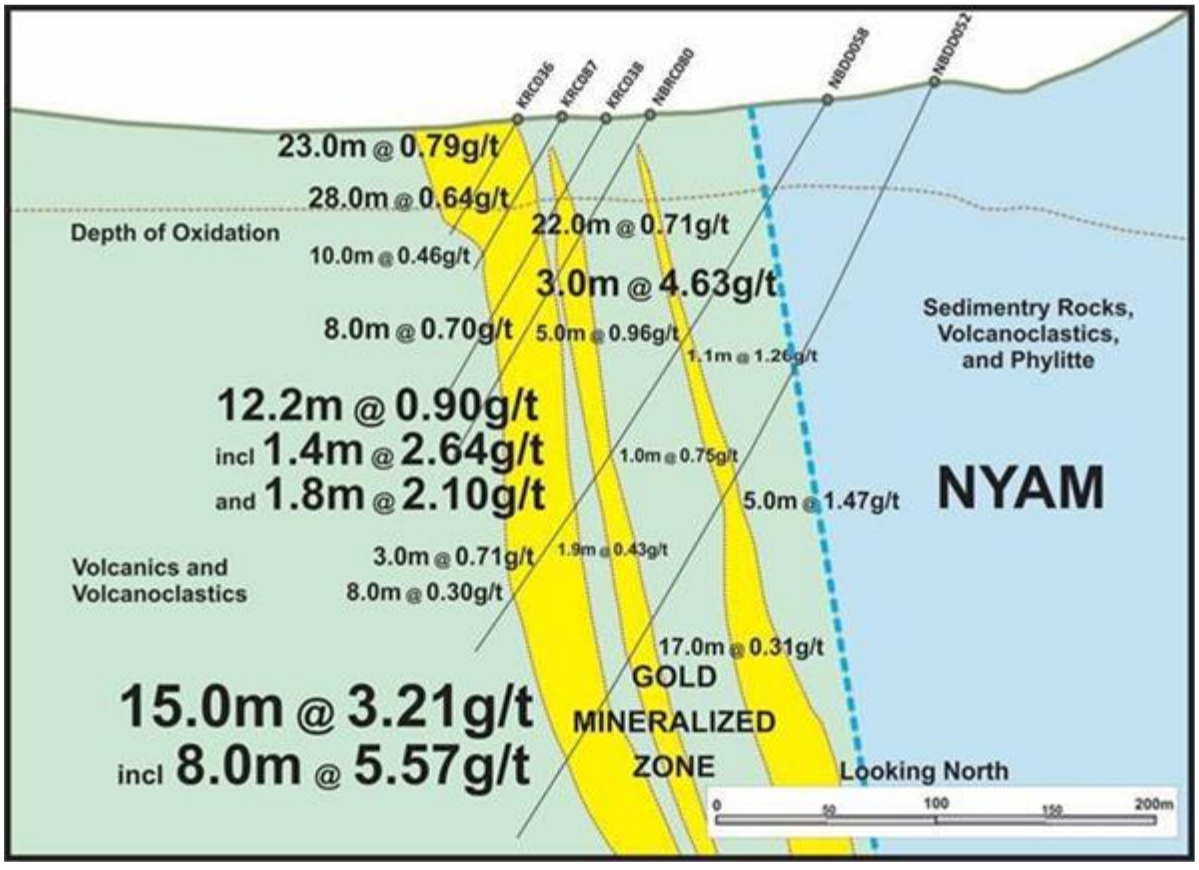
Hydrothermal alteration displays a typical greenschist assemblage (gold + quartz + sericite ± graphite ± chlorite ± epidote ± ankerite). Chlorite + epidote clots are observed within, or proximal to, the gold mineralization within the brecciated quartz veins. These probably result from remobilization associated with regional alteration.

No visible bleaching or other styles of alteration have been observed in the host sediment related to the quartz veining apart from narrow silicified vein selvedges. At the Boin Zone, the depth of weathering reaches up to 100 m in places. Weathering is deepest where the mineralization is best developed suggesting the greater intensity of veining and fracturing may have enhanced the weathering over the deposit.

7.3.3 Nyam

The Nyam Zone strikes over a distance of 2.1 km, hosted by altered phyllite, 200 to 300 m west of the interpreted position of the second order NS structure. The zone of mineralization lies in the hanging wall of a northeast-striking shear that dips 70° east and is up to 30 m thick. Nyam mineralization is part of a continuous 15 km strike length of gold prospects on the Project that spans from Nyam southwest to Sewum in the south and includes the adjacent Kojina Hill. An extensive envelope of anomalous gold mineralization (more than 0.25 g/t) dips sub-vertically and strikes 030° (Figure 7.3.4).

Figure 7.3.4 Generalized Geology Section at Nyam



Source: Newcore, 2023

Mineralization at the Nyam Zone is composed of veined and brecciated sediment, phyllite and moderate intrusive rocks cemented by quartz, carbonate (ankerite), and albite, and has been traced continuously in trenching and drilling for over 2 km along strike.

Alteration associated with the zone of veining and brecciation consists of bleaching due to replacement by sericite, quartz, ankerite, albite, rutile, and minor pyrite. Pyrite typically makes up less than 1% of the infill and alteration minerals. No visible gold, arsenopyrite or base metal sulphides have been identified in any core samples to date.

The footwall of the mineralization is marked by discontinuous carbonaceous shears and a 2 to 3 m wide zone of green coloured fuchsite-magnesium chlorite alteration. The fuchsite is believed to represent an alteration front where chromium leached from the altered volcaniclastic sandstone beds has been re-deposited in micas, replacing the basal shear adjacent to the quartz-carbonate-sericite alteration zone.

The zone of quartz-sericite-carbonate bleaching has a gradational upper contact and is not always mineralized. Rare carbonaceous shears cut through the mineralization, indicating that the shear zone has continued to move after the mineralization event. Post-mineralization deformation is also supported by petrologic studies that describe stylolites, recrystallization, strained and sutured quartz, and albite grains in the vein material (England, 2011).

Rare sphalerite and anhedral grains of chalcopyrite less than 0.1 mm in size, rimmed by tetrahedrite–tennantite have been observed in the quartz veins during petrological studies (England, 2011).

7.3.4 Kwakyekrom

The Kwakyekrom Zone is located 3 km south of the Nyam Zone and is interpreted to be related to the extension of the same structure. Drilling has tested the Kwakyekrom Zone over a strike distance of 1.3 km and to a depth of approximately 150 m. The zone is hosted by altered phyllite, 700 to 800 m west of the interpreted position of the second order NS structure. The phyllite has been intersected by metre-scale dolerite dykes similar in composition to the larger intrusive bodies encountered at Sewum.

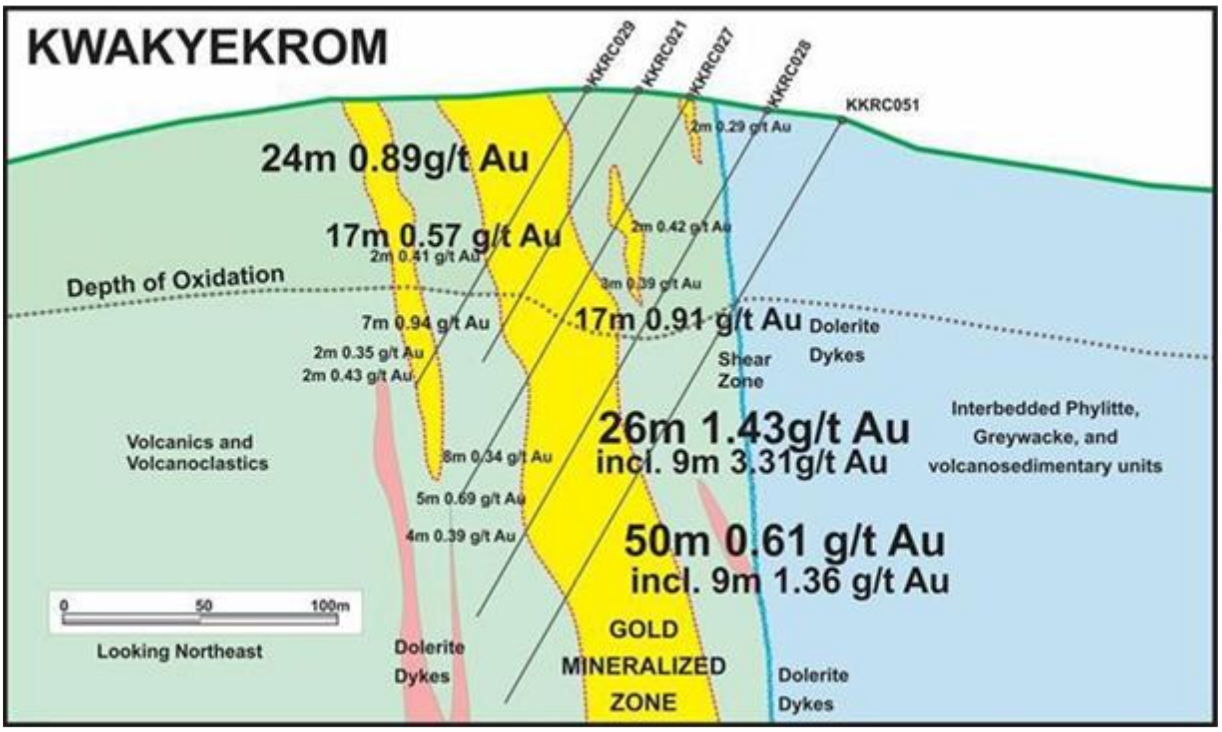
Kwakyekrom mineralization is part of a continuous 15 km strike length of gold prospects on the Project that stretches from Nyam southwest through Kwakyekrom to Sewum in the south. Gold mineralization (more than 0.20 g/t) is hosted in a series of sub-parallel zones ranging in width between 5 and 25 m (Figure 7.3.5).

Kwakyekrom mineralization is associated with sediments showing intense ductile strain, with centimetre- to metre-scale quartz veins focused within brittle-ductile deformation zones. Additionally, sediment-dolerite contacts are often the site of quartz veins and variable gold mineralization.

The NNE-SSW-striking, steeply- to moderately dipping metasedimentary package is consistent with high degrees of ductile strain and possibly the presence of tight folds in the stratigraphy. As with Nyam, the main fabric is overprinted by a moderately developed crenulation that dips to the NW.

The alteration associated with the zone of veining and brecciation consists of bleaching due to replacement by sericite, quartz, ankerite, albite, and minor pyrite but is not as well developed as at Nyam. Fine-grained pyrite is focused around discrete quartz veins ranging in width from <0.1 m to more than 1.5 m. No visible gold, arsenopyrite or base metal sulphides have been identified in any samples to date. A series of crosscutting graphitic sheared structures ranging in width between 0.2 and >1.5 m are present.

Figure 7.3.5 Generalized Geology Section at Kwakyekrom



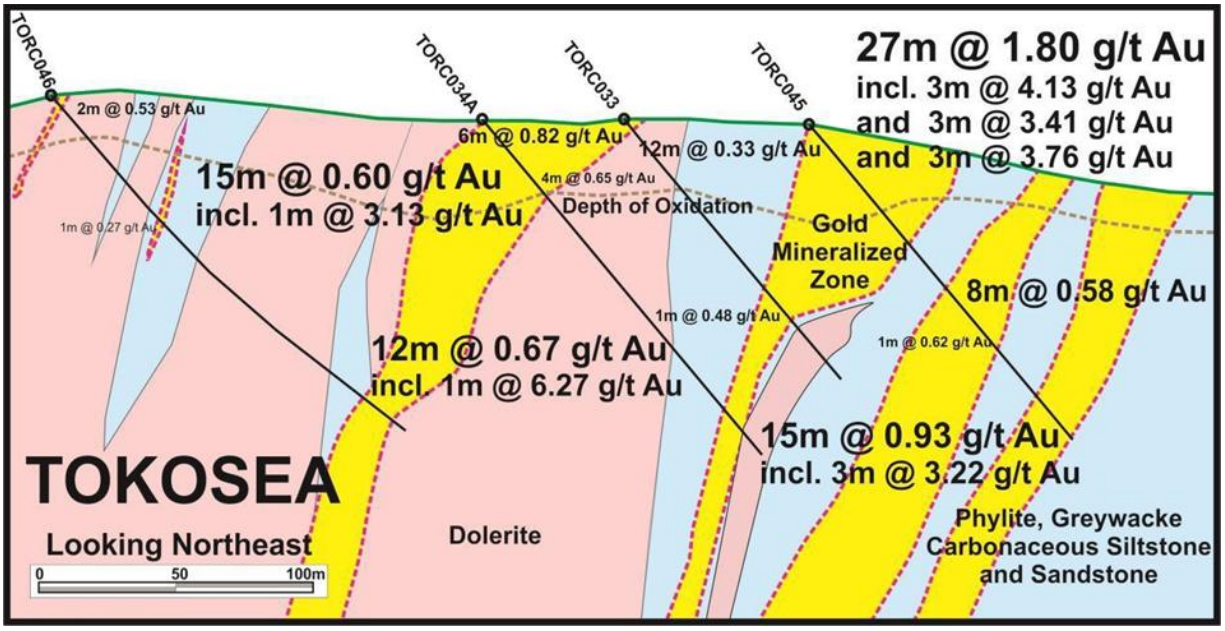
Source: Newcore, 2023

7.3.5 Tokosea

The Tokosea prospect is located on a volcanic / sediment contact similarly to that associated with the Sewum Mine, although offset by faulting south of Adamansu. The prospect includes the workings of the historic Tokosea Mine along with several parallel and en-echelon gold mineralized quartz veined zones some 30 m to the east, including the Tokosea East prospect. The historic mine at Tokosea included a small open pit and underground development on the 18 m, 27 m, and 45 m Levels.

Significant gold mineralization is hosted by sub-vertical quartz veined structures in phyllite with some gold in quartz veinlets within the sediment and volcaniclastic units. The main structure developed in the Tokosea Mine is a shear-hosted, thin (0.3 to 1 m) lenticular quartz veined zone averaging 5 to10 m in width, dipping 85° northwest, and following a contact between a dominantly argillaceous (phyllite) footwall (eastern) and a dominantly volcaniclastic hanging wall (western) unit. The immediate host rock is a black carbonaceous phyllite. The general strike is 030°.

Figure 7.3.6 Generalized Geology Section at Tokosea



Source: Newcore, 2023

7.3.6 Kojina Hill

The Kojina Hill prospect is located 1 km west of the Nyam deposit. Previous small-scale gold mining has been reported to have occurred in the area but is poorly documented. The zone consists of a closed-spaced gold mineralized structures striking NE-SW and dipping west at 80°, and which plunges steeply to moderately north. Mineralization is hosted by a zone of deeply weathered quartz-veined phyllite. Fuchsite-altered greywacke is also noted. The central portion is exposed on the side of a prominent hill and has been defined along strike by trenching both to the north and south for more than 1 km in each direction.

7.3.7 Eradi

The Eradi prospect is located in the north of the Enchi licence area where the regional structures converge and gradually change strike from north-northeast to northeast. Very little outcrop exposure is present at Eradi due to the thick weathering profile and laterite development. All geology mapped comes from trenches and drillholes.

The Eradi Zone is the northernmost occurrence along the Nyam Shear Zone (NSZ), one of a number of major structures that splay off the Bibiani Shear and pass through the Enchi licence area. Mineralization at Eradi is developed within a second-order shear that parallels and lies 300 m west of the NSZ. Gold mineralization at Eradi is entirely hosted in quartz veins. The veins are variable in shape, size, and orientation, rarely exceeding 1 m in thickness, and tend to dip moderately (20° to 60°) east. The intensity of veining varies between drill sections. Quartz in the veins is composed of white, less than 10 mm anhedral grains that are often fractured and recrystallized by later shearing. The quartz veins are generally quite pure, containing rare carbonate minerals and no sulphides.

No visible bleaching or other styles of alteration have been observed in the host sediment related to the quartz veining apart from narrow silicified vein selvedges. No intrusives have been identified in trenches or drill core at Eradi.

The host rocks at Eradi are dominated by interbedded carbonaceous siltstone and sandstone (turbidite). The sediments have been regionally deformed to greenschist facies, are steeply dipping, and typically strike northeast (040°) parallel to the regional structural grain. Gold mineralization at Eradi is hosted in irregular quartz veining localized along northeast striking shear zones with a near vertical dip.

7.3.8 Nkwanta

The Nkwanta prospect is located in the central portion of the Project where exploration has defined a gold anomalous target 2.5 km by 1.0 km. The area is associated with the same phyllite / volcaniclastic contact as that located near Tokosea.

An adit at the Nkwanta prospect tests a weakly mineralized, narrow quartz vein over a strike of 300 m. The quartz vein is hosted by phyllite, within a contact zone, with volcaniclastics to the west. The contact zone is possibly the strike extension of that in the Tokosea Mine 3 km to the south.

7.3.9 Agyeikrom

The Agyeikrom prospect is located in the north-central portion of the Project where exploration has defined a gold anomalous target 4.5 km by 2.0 km. The area is associated with the same phyllite / volcaniclastic contact as that located near Kojina Hill. Mineralization is hosted by a zone of deeply weathered quartz-veined phyllite and fuchsite-altered greywacke in a series of zones dipping moderately to the west.

7.3.10 Sewum South

The Sewum South prospect is located 3 km south of the Sewum deposit. Soil sampling has generated the largest individual anomaly on the Project measuring 6.0 km by 2.5 km. The anomaly is associated with a wide and complex conductive zone in the airborne electromagnetic survey suggesting structural and geological similarities to the Sewum deposit area. Some of the lower-lying portions of the Sewum South area have been the site of artisanal gold mining activity.

7.3.11 Achimfo

Several thin (less than 1 m wide) quartz veined structures are hosted by phyllite exposed in old workings including small shafts and galamsey workings over strike-lengths of up to 400 m and depths of up to 40 m. Erratic high-grade gold is hosted by quartz veining. The vein hosting structures are considered as steep southeast dipping thrusts that juxtaposed folded finer - and coarser-grained metasediments (carbonate altered siltstones, pyrite altered quartzite, and greywacke).

7.3.12 Adamansu

Quartz veining is hosted by phyllite, within a contact zone, with volcaniclastics to the west. The contact zone is possibly the fault-displaced strike extension of that at the Sewum mine, and the southern extension of that at the Tokosea Mine.

7.3.13 Alatakrom

The Alatakrom prospect is along strike, northeast of the Tokosea East prospect. Several conformable sub-vertical gold mineralized quartz vein zones are hosted by phyllite, within 50 m of a contact with volcaniclastics to the west.

7.3.14 Beekokrom

The prospect straddles projected strike positions of mineralized structures defined at the Kwakyekrom prospect, 2 km to the southeast.

7.3.15 Sewum Mine

The Sewum Mine developed two narrow (0.5 to 1 m wide) quartz veins, the Main Reef and West Reef, over a strike of 450 m. The veins dip southeast at 45° to 60° within a strongly deformed carbonaceous phyllite near a contact with less deformed volcaniclastics to the west. The Sewum Mine is possibly hosted by a bedding concordant splay from the second order splay.

From 1940 to 1951, the Kwahu Mining Co. deepened the main shaft to 120 m and developed the 45 m and 78 m Levels. No production was recorded (Kesse, 1985).

8.0 DEPOSIT TYPES

The Project's mineralized zones have the characteristics of epigenetic, mesothermal quartz vein-style gold deposits with an overlying gold-bearing saprolite / laterite layer. This type of mineralization is the most common style of gold occurrence in West Africa and is commonly referred to as the Ashanti type.

Mesothermal mineralization has a strong structural control and brittle-ductile deformational style that is related to large tectonic corridors (more than 50 km long and several km wide). These deformational zones display evidence of complex multiphase displacement with mineralization typically associated with second- and third-order structures (Roberts, 1988). Auriferous veins are best developed at dilatational sites where structural or compositional irregularities occur within the shear structure. Favourable sites include conjugate or branching shear zone intersections, major flexures within the shear plane, axial planes of hinges and limbs of folds, and compositional variations associated with major lithological contacts or incorporated dyke material.

The most common host rock is usually a fine-grained metasediment in close proximity to graphitic or siliceous chemical sediments. However, in some areas, intrusive rocks are known to host significant gold mineralization such as at the Chirano Gold Mine (owned by Asante Gold Corporation) located 50 km northeast of the Project.

Mesothermal alteration is generally more visible within greenschist facies settings. Alteration usually occurs as chloritization, pyritization, silicification, and tourmalinization, with minor amounts of potassic and alkali feldspar alteration as well as potassic phyllosilicate (sericite, muscovite, and biotite) alteration. Carbonate alteration is pervasive (ankerite and calcite) on regional and deposit scales.

Mineralization can occur as both refractory and non-refractory styles. Refractory mineralization is characterized by early-stage, disseminated sulphides of primarily pyrite, and/or arsenopyrite hosting significant gold content, which is overprinted by late-stage quartz veining with minor amounts of visible gold and accessory polymetallic sulphides. Examples of the refractory-style deposits include Obuasi (AngloGold) and Bogoso-Prestea (Heath Goldfields Ltd.). Non-refractory mineralization is described as gold not hosted within sulphide minerals of either the early or late-stage mineralization events. Examples of non-refractory mineralization include Chirano (Asante Gold Corporation) and Ahafo (Newmont).

The gold mineralization that occurs in the oxidized zone is released from the hypogene deposits by physical disaggregation and chemical dissolution. Dissolution and reprecipitation of gold in the saprolite appear to take place in-situ with little evidence of supergene enrichment. The mineralization can be covered by centimetres to several metres of kaolinite-mica forest soils. The saprolite zone of leached rock can generally extend down 60 to 70 m (Bowell, 1992), and at Enchi can exceed 100 m.

Newcore's exploration and development programs are based on the well-documented epigenetic, mesothermal quartz vein-style of gold mineralization hosted in the bedrock and the overlying weathered rocks. The deposits of Ghana lying along major fault zones forming the contact between the sedimentary and the volcanic-plutonic units, like the deposits at Enchi, are traditionally the reference of gold deposits of West Africa in international literature.

9.0 EXPLORATION

Exploration activities, mainly consisting of line cutting, soil sampling, trenching, auger drilling and topographic surveys, have been carried out by various operators during the history of the Project. The principal targets were anomalies generated from airborne geophysical data. The work included both wide-spaced and detailed surveys. Results included anomalous gold-in-soil, trenches and auger samples, which warranted additional follow-up work.

The procedures for each exploration method summarized below are excerpts from the "Geologist's Procedures Manual, Version 1.0, October 1, 2005" generated by Red Back Mining Inc. (Red Back, 2005), which have been carried forward in subsequent exploration campaigns.

9.1 Soil Sampling

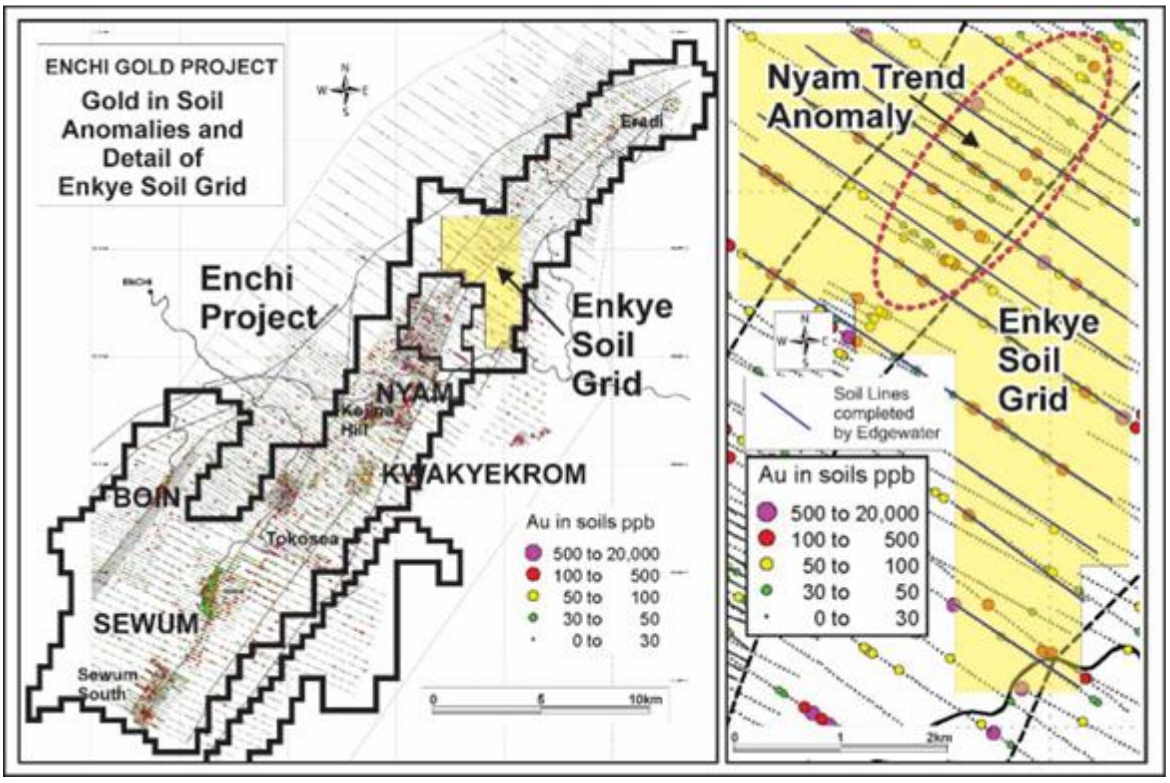
All soil sampling was conducted in the presence of a geologist and was not carried out by technicians alone. Samples were collected from ±50 cm depth and weighed from 2 to 3 kg. Duplicate samples were collected every 25 samples. To collect the duplicate, a larger hole was dug to collect 5 to 6 kg of sample and mixed thoroughly on a plastic sheet. The material was then coned and quartered into two samples. The results of the soil survey were disclosed in a previous technical report (McCracken et al., 2016). Table 9.1.1 summarizes the soil work completed.

Table 9.1.1 Soil Survey Summary

Prospect	Area Covered km²	No. of Lines	Grid Spacing	Total Line Length km	No. of Samples	Type of Sample
Enkye	35	10	400 m x 50 m	60	986	Soil

The Enkye grid soil sample results defined a moderately anomalous zone 2.5 km long and averaging 1.25 km in width, on trend from the Nyam Anomaly located 4 km south (Figure 9.1.1). The anomalous area is defined by a series of results greater than 50 ppb gold, with common results greater than 100 ppb gold, and isolated results greater than 500 ppb gold. The samples are representative of the material tested and generally no sample bias has been identified beyond the normal variability of the weathered and soil profile.

Figure 9.1.1 Summary Map of Anomalous Soil Results



Source: Newcore, 2026

9.2 Trenching

Trenching has been a valuable exploration tool allowing for the definition of gold mineralized structures within the broad gold-in-soil anomalies identified on the Project.

The trenches are dug 1.0 to 1.5 m in width with a maximum depth of 3.5 m. The name of a trench consists of a two-letter prospect prefix, followed by "TR" and then a sequential numbering.

For consistency, trenches start at the western end (collar) and intervals are measured along the surface using slope distance, not horizontal distance. This allows correct plotting of the trench as a three-dimensional entity. To allow routine plotting of the trench as a drillhole, each segment must be considered as a separate trench, with its own collar, and with its sample intervals starting at zero at its western end. The segments of a trench are identified by suffixes, for example CHTR798A, CHTR798B, from west to east.

Completed trenches are measured by marking out intervals along the surface starting from zero at the western end. Strings may be dropped down the sides of the trench to help the marking of the 1 or 2 m sampling intervals near the base of the trench.

The trenches are surveyed as a three-dimensional entity, and trench data is stored in the standard drilling tables of the database (collar, survey, assay, geology).

The collar coordinates are determined by tape and compass, GPS, DGPS, or EDM survey depending on the stage of the Project.

The surface trace of the trench is surveyed from the collar to the end using tape, compass, and clinometer to produce a 'downhole' survey file. The intervals are chosen to match inflection points in the trench trace.

The 'from and to' measurements are slope measurements along the surface and are not corrected to horizontal distances.

The survey is usually done by a geologist and an assistant. The assistant holds a pole with a mark at the geologist's eye height. The geologist stands at the collar, the assistant at the first inflection point, and the geologist sights on the mark on the pole to record the inclination and azimuth.

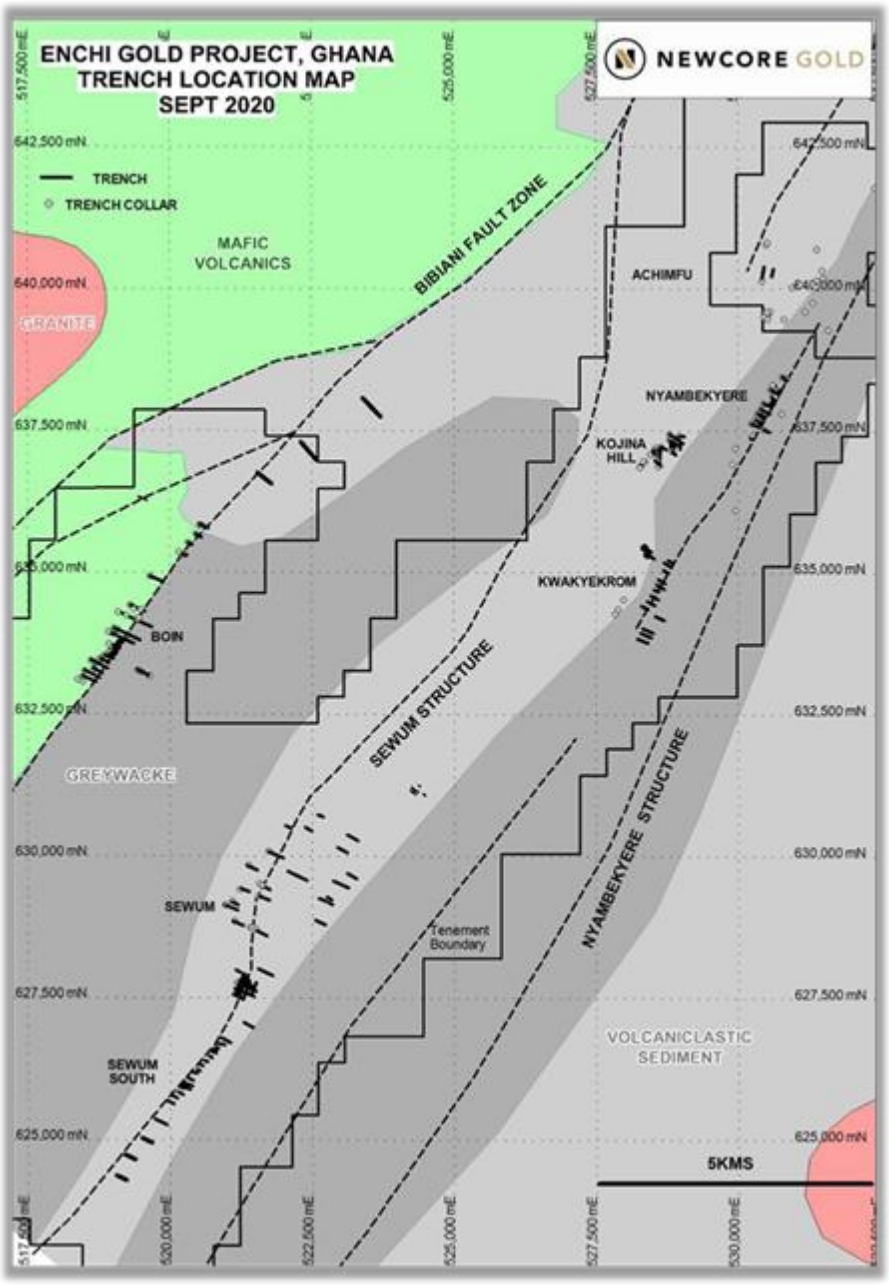
Continuous channel samples are cut from the centre line of the floor of the trench. The trench must be checked by a geologist prior to sampling to ensure saprolite has been reached. The base of the trench must be cleaned by brushing or using a spade prior to sampling. Trenches are sampled by lithology, routinely using 2 m intervals with a minimum interval of 0.5 m.

Duplicates were taken every 25 samples along a second channel cut either just above or below the original sample.

9.2.1 2020 Trenching

One hundred and eighty (180) trenches totalling 17,019 m were completed on the Project during 2020. The criteria for reporting trench results were 4 m minimum length and a minimum 0.3 g/t average grade over the interval. Figure 9.2.1 shows the location of the trenches at the Project site.

Figure 9.2.1 Summary Map of 2020 Enchi Trench Locations



Source: Newcore, 2020

9.2.2 2021 – 2022 Trenching

Exploration work at Enchi, including trenching, continued to define near-surface, gold mineralized structures on the Project. Trenching completed in 2021 and 2022 focused on a number of high- priority gold targets that are defined by kilometre-scale gold-in-soil anomalies located across the Project. Trenching intersected gold mineralization with similar grades and widths to prior trench results associated with the current resource zones.

As part of this program, Newcore completed 62 trenches totalling 11,037 m with a total of 49 trenches encountering gold-bearing structures of which 37 encountered multiple gold mineralized zones.

Intercept highlights from the 2021 and 2022 exploration are detailed in the following Table 9.2.1.

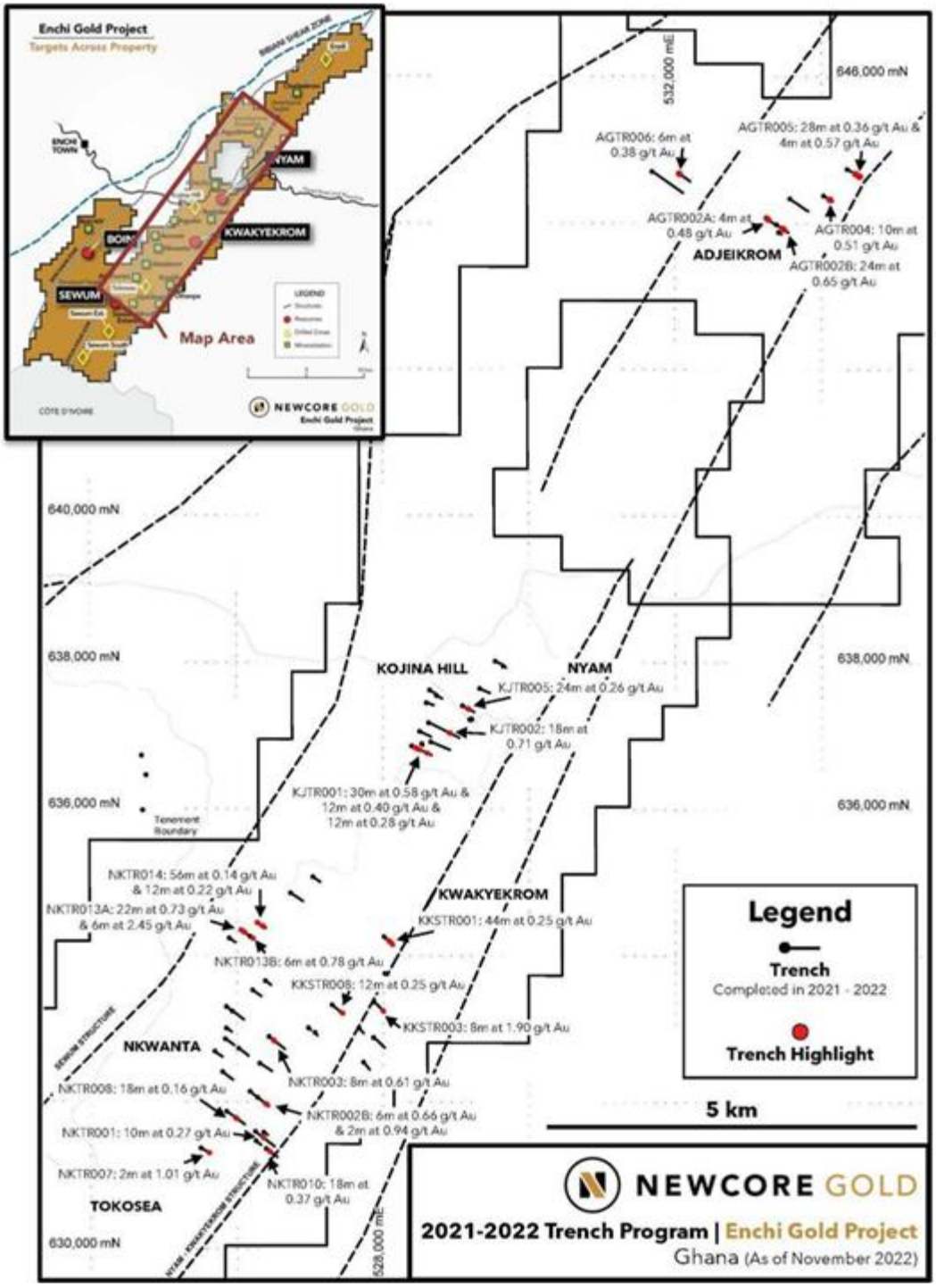
Table 9.2.1 Current Trench Intercepts Summary

Trench ID	Target	From m	To m	Int. m	Au g/t
AGTR002A	Agyeikrom	26.0	30.0	4.0	0.48
AGTR002B	Agyeikrom	98.0	122.0	24.0	0.65
AGTR003	Agyeikrom	112.0	114.0	2.0	0.86
AGTR004	Agyeikrom	98.0	108.0	10.0	0.51
AGTR005	Agyeikrom	104.0	106.0	2.0	0.75
AGTR005	Agyeikrom	116.0	122.0	6.0	0.25
AGTR005	Agyeikrom	128.0	132.0	4.0	0.57
AGTR005	Agyeikrom	174.0	202.0	28.0	0.36
AGTR006	Agyeikrom	2.0	8.0	6.0	0.38
KJCH001	Kojina Hill	2.0	24.0	22.0	0.13
KJCH004	Kojina Hill	0.0	9.0	9.0	2.01
KJTR001	Kojina Hill	48.0	78.0	30.0	0.58
KJTR001	Kojina Hill	92.0	98.0	6.0	0.27
KJTR001	Kojina Hill	110.0	122.0	12.0	0.40
KJTR001	Kojina Hill	222.0	234.0	12.0	0.28
KJTR002	Kojina Hill	106.0	110.0	4.0	1.04
KJTR002	Kojina Hill	234.0	276.0	42.0	0.24
KJTR002	Kojina Hill	308.0	326.0	18.0	0.71
KJTR005	Kojina Hill	76.0	100.0	24.0	0.26
KJTR005	Kojina Hill	124.0	126.0	2.0	0.81
KJTR008A	Kojina Hill	42.0	50.0	8.0	0.21
KJTR008B	Kojina Hill	240.0	246.0	6.0	4.59

Trench ID	Target	From m	To m	Int. m	Au g/t
NKTR001	Nkwanta	132.0	142.0	10.0	0.27
NKTR002B	Nkwanta	140.0	146.0	6.0	0.66
NKTR002B	Nkwanta	160.0	162.0	2.0	0.94
NKTR003	Nkwanta	68.0	76.0	8.0	0.61
NKTR007	Nkwanta	128.0	130.0	2.0	1.01
NKTR008	Nkwanta	148.0	166.0	18.0	0.16
NKTR010	Nkwanta	48.0	66.0	18.0	0.37
NKTR0013A	Nkwanta	2.0	8.0	6.0	0.48
NKTR0013A	Nkwanta	38.0	60.0	22.0	0.73
NKTR0013A	Nkwanta	46.0	52.0	6.0	2.45
NKTR0013B	Nkwanta	46.0	52.0	6.0	0.78
NKTR0014	Nkwanta	0.0	12.0	12.0	0.22
NKTR0014	Nkwanta	80.0	136.0	56.0	0.14
NKTR021	Nkwanta	58.0	66.0	8.0	0.25
NKTR026	Nkwanta	108.0	118.0	10.0	0.58
NKTR027	Nkwanta	50.0	64.0	14.0	0.19
KKSTR001	Kwakyekrom	82.0	126.0	44.0	0.25
KKSTR003	Kwakyekrom	174.0	182.0	8.0	1.90
KKSTR008	Kwakyekrom	170.0	182.0	12.0	0.25

The Intervals in the above table are trench lengths, with true width estimated to be 75-85%, and length-weighted averages from uncut assays. Health, safety, and the environment are prioritized throughout the trenching process, and all trenches are backfilled and reclaimed once sampling and mapping are completed.

Figure 9.2.2 Summary Map of 2021 – 2022 Enchi Trench Locations



Source: Newcore, 2022

9.3 Auger Drilling

Auger holes, which were completed with hand augers, are vertical (-90º) and therefore no azimuth is required in the collar file. In the survey file, a -90º dip will be required at 0 m and at the end of the hole in the downhole survey file. The average sample depth was 3 m.

Sampling was carried out on the basis of regolith geology. Lateritic soils, mottled clays, and saprolite were sampled separately. The A soil horizon was not sampled. Duplicates were taken every 25 samples.

The results of the auger survey are summarized in Table 9.3.1.

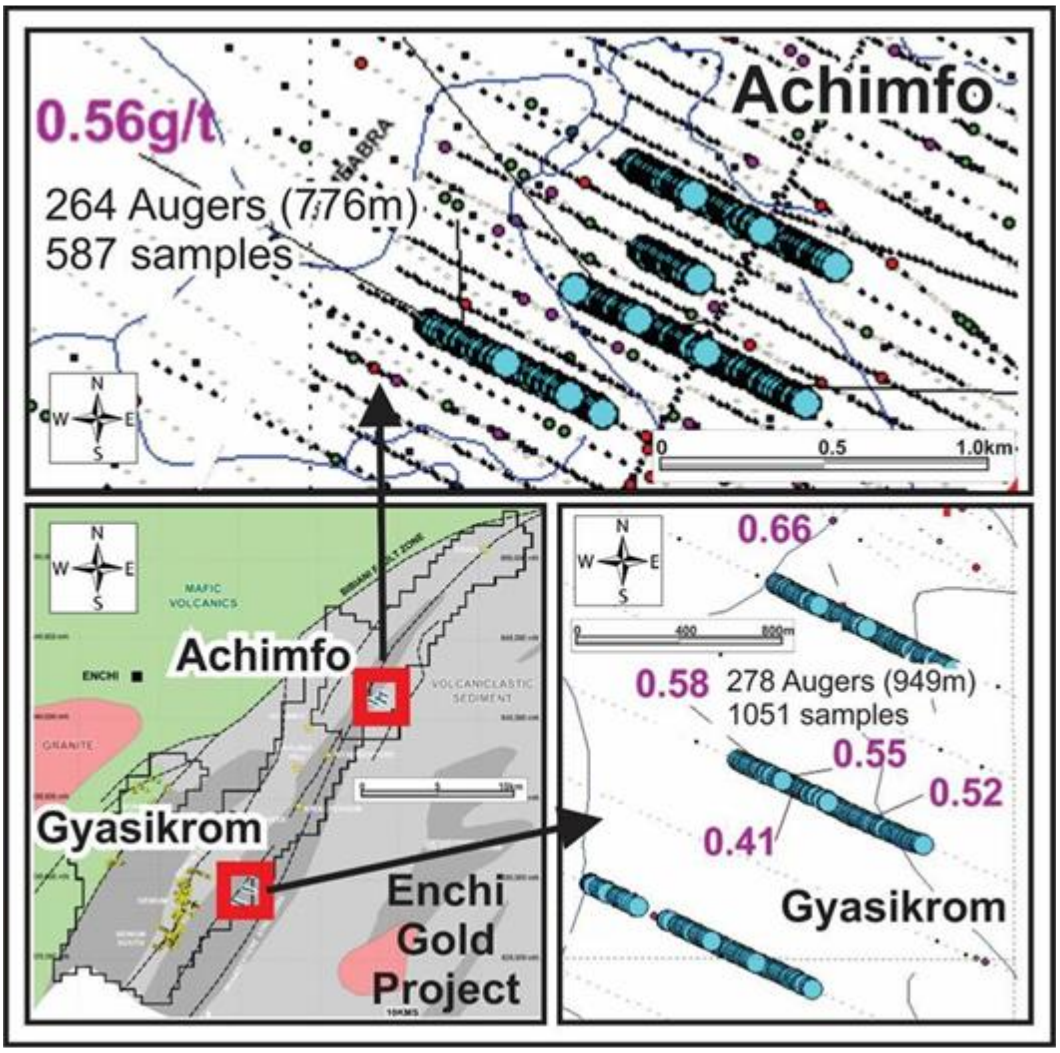
Table 9.3.1 Auger Drilling Summary

Prospect	Area Covered km^2	No. of Samples	No. of Holes	Total Depth m	Significant Results g/t	Type of Sample
Achimfo	1.00	587	264	776.0	Assays to 0.5 g/t Au	Auger
Gyasikrom	1.55	1,051	278	949.0	Assays to 0.5 g/t Au	Auger

Figure 9.3.1 is a map summarizing the significant auger results from the Achimfo and Gyasikrom Prospects.

In the Achimfo and Gyasikrom area, individual auger sample results returned irregularly spaced values considered to be anomalous with greater than 500 ppb gold. No anomalous areas of significant size were outlined by the auger drilling.

Figure 9.3.1 Summary Map of Significant Auger Results



Source: Newcore, 2020

9.4 Drone Topographic Surveys

9.4.1 2022 Survey

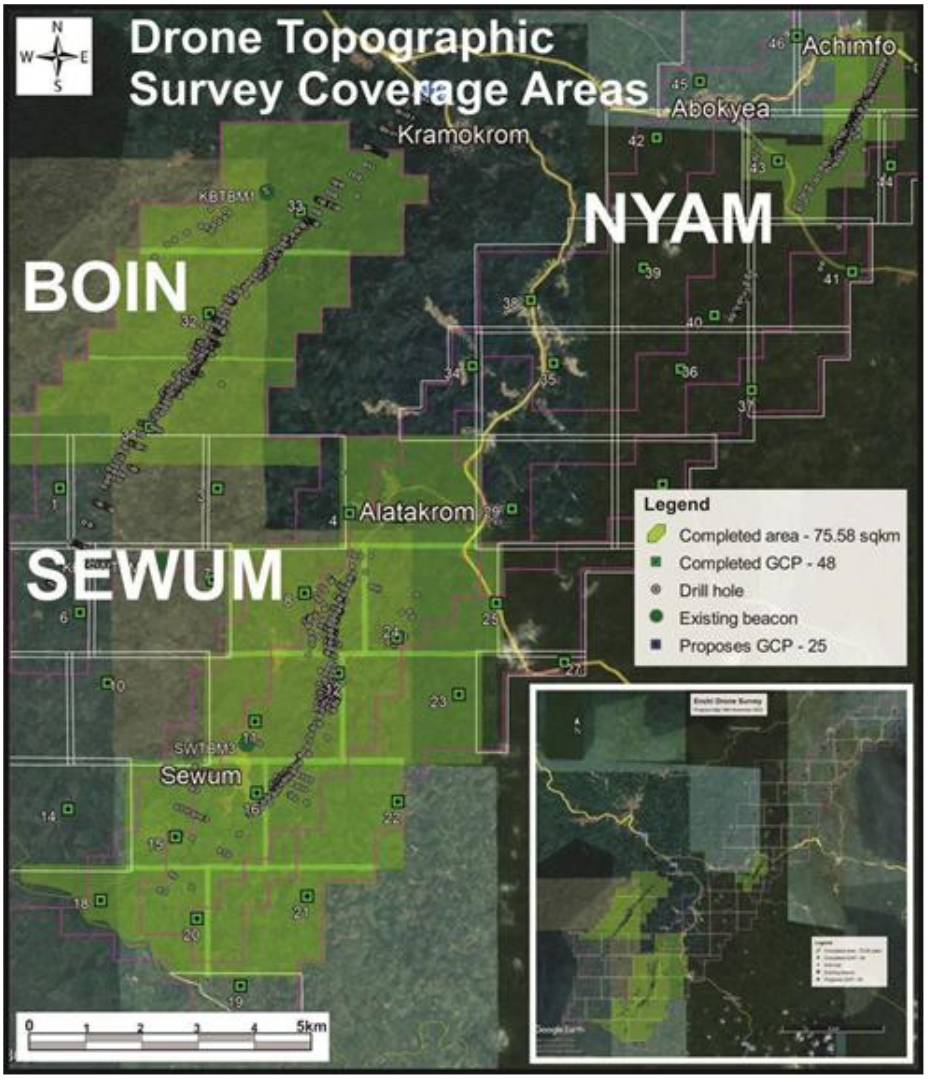
A drone topographic survey was completed in 2022 over the Boin, Sewum, and Nyam deposits with a total surveyed area of 75.58 km^2. To complete the survey a total of 48 ground control points were established at 1 km centres surveyed off existing control beacons within the Project area.

The survey covered an area at Boin of approximately 7 km by 4 km, at Sewum of 8 km by 5 km, and at Nyam of 3 km by 2 km (Figure 9.4.1). The surveys were completed using east-west oriented flight lines spaced 50-100 m apart. The survey employed a DeltaQuad Pro #MAP equipped with a Post-Processing

Kinematic (PPK) kit, which is proven to be a more reliable and accurate solution for drone surveys. Drones equipped with PPK solutions offer greater data dependability because of the GPS correction technology incorporated into the units.

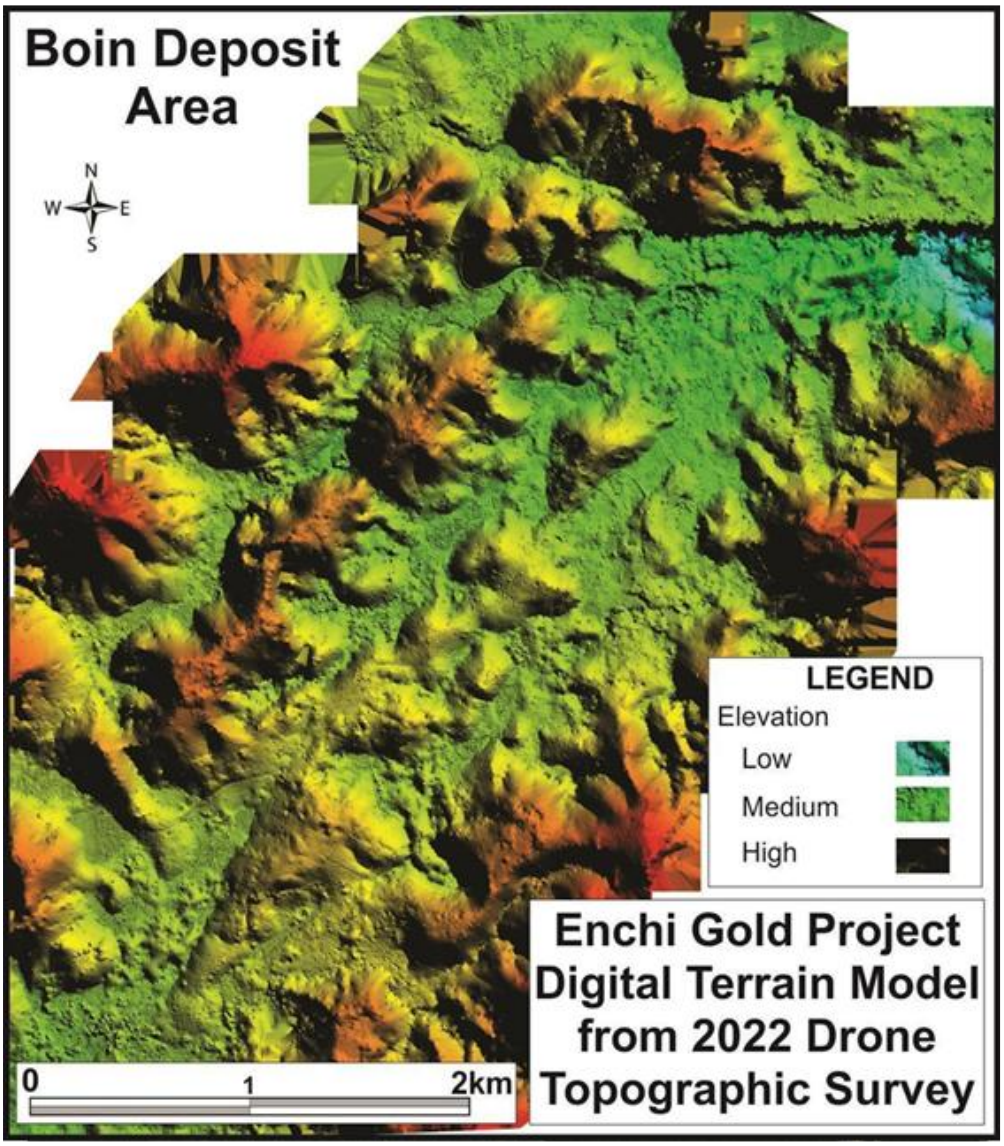
All RC holes, diamond drillholes and trenches at Boin, Sewum and Nyam were corrected to the drone topographic survey elevations completed in 2022. A digital terrain model (DTM) was created by flying a drone survey over the resource zones. Orthophotos (geometrically corrected photos) were generated from photos and data collected during the drone survey. The elevation (Z) data for drill collars was modified to fit the DTM surface (Figure 9.4.2). The Project database drill collar survey file therefore comprises handheld GPS coordinates (X & Y) and modified elevation (Z) data.

Figure 9.4.1 Topographic Drone Survey Coverage



Source: Newcore, 2022

Figure 9.4.2 Digital Terrain Model for Boin from Drone Topographic Survey



Source: Newcore, 2022

9.4.2 2024 Survey

Between September and December 2024, SEMS Exploration completed a drone magnetic geophysical survey over a 3.6 km^2 block of the Project which included a portion of the Kwakyekrom deposit and the Kojina Hill target. One GEMSYS Airbird magnetometer system slung from a drone was used. A Geometrics Standard Proton magnetometer was deployed as a base-station and used to record diurnal data for post processing the line data. GPS positioning was provided by the GEMSYS magnetometer. All positional XY information has been provided in the WGS84 projection system.

9.4.3 2025 Survey

SEMS Exploration completed a topographic LiDAR survey on the central area of the Project, where terrain is hilly with thick vegetation, mostly forest. The survey area covered 66.42 km^2. Survey data has an average 6 cm accuracy over the entire survey area. The first step was the creation of survey pillars and identification of drone launch sites. The survey started on 29 September 2024 and was completed on 2 November 2024. Data processing was completed on 26 December 2024, and the final report was issued in January 2025. An initial DEM with resolution of 50 cm was created. The WGS84 UTM projection is a standard Universal Transverse Mercator projection using the standard zone number 30 north. The DGPS equipment is composed of a DGPS rover unit and a base station unit, both Stonex brand. The LiDAR is composed of one D-RTK2 antenna, one DJI Matrice 300 RTK and one DJI Zenmuse L2 LiDAR camera. For this mission SEMS Exploration used eight drone batteries. Data consistency with good flight overlaps for visibility and LiDAR data resulted in good accuracy at centimetre-scale for LiDAR and orthophoto.

9.5 Recent Exploration Results

9.5.1 Agyeikrom Target

The exploration work in 2021 - 2022 includes first-pass trenching on the Agyeikrom target which is located in the north-central portion of the Project. The gold-in-soil anomaly at Agyeikrom extends 4.5 km by 2 km with no previous trenching or drilling completed. A total of eight trenches (1,852 m) tested 1.2 km of strike length with results including 0.65 g/t Au over 24 m (AGTR002B) and secondary intervals of 0.20 g/t Au over 4.3 m, 0.36 g/t Au over 28 m and0.57 g/t Au over 4 m (AGTR005), 0.51 g/t Au over 10 m (AGTR004), and 0.48 g/t Au over 4 m (AGTR002A).

9.5.2 Kojina Hill Target

At the Kojina Hill target trenching consisted of 15 trenches (2,168 m) with step-out trenching on a previously outlined mineralized area extending the defined gold mineralization more than 500 m north and south of the prior drilling. Trench KJTR008B, located 300 m to the south of previous work, intersected eight gold mineralized structures highlighted by 4.59 g/t Au over 6 m. Trench KJCH004, located 100 m to the east on a subparallel structure intersected 2.01 g/t Au over 9 m. Trench KJTR001, located 500 m to the south of previous work and on the southern limited of the currently tested area, intersected multiple gold mineralized structures with results including 0.58 g/t Au over 30 m, 0.40 g/t Au over 12 m, and 0.28 g/t Au over 12 m. Additional trenches intercepted 0.71 g/t Au over 18 m, 0.24 g/t Au over 42 m, 0.26 g/t Au over 24 m, and 1.04 g/t Au over 4 m.

9.5.3 Nkwanta Target

At the Nkwanta target trenching has defined a series of gold mineralized structures within one of the strongest gold-in-soil anomalies identified on the Project, which stretches for 2.5 km by 1.5 km. A total of 30 trenches (5,610 m) tested multiple structures with results including 0.73 g/t Au over 22 m with a higher-grade core of 2.45 g/t Au over 6.0 m, 0.14 g/t Au over 56 m, 0.37 g/t Au over 18 m, 0.58 g/t Au over 10 m, 0.61 g/t Au over 8 m, and 0.78 g/t Au over 6 m.

9.5.4 Kwakyekrom Extension

Trenching work on the southern extension of the Kwakyekrom deposit extended the defined gold mineralization in preparation of additional drilling to be completed. The trenching consisted of a total of nine trenches (1,407 m) with results including 1.90 g/t Au over 8 m, 0.25 g/t Au over 44 m, 0.25 g/t Au over 12 m, 0.23 g/t Au over 6 m, and 0.22 g/t Au over 6 m.

10.0 DRILLING

10.1 Overview

The Project is considered an advanced property by definition of NI 43-101. As such, this Report is not required to meet NI 43-101F1 Item 10(c). Location maps are provided in this Section to disclose the collar locations of the drillholes. Generalized cross sections of the drilling and geology for Sewum, Boin, Nyam, and Kwakyekrom are also available in Section 7 of this Report.

Further details on individual drilling campaigns can be found in previous technical reports (McCracken, 2010; McCracken, 2012; McCracken, 2014; McCracken et al., 2016; McCracken and Smith, 2020; McCracken et al., 2021; McCracken and Meadows Smith, 2023 Lycopodium, 2024, and DRA, 2026.

Drilling at the Project can be subdivided into three phases, including:

- Historic drilling by previous owners (pre-2006).
- Drilling by Newcore predecessors (2010 to 2018).
- Recent drilling by Newcore (2020 to present).

A total of 68,017 m in 340 diamond drill (DDH) holes, 144,100 m in 1,130 reverse circulation (RC) holes, and 14,380 m in 363 rotary air blast (RAB) holes have been completed and reported on the Project from its inception to the effective date for the current report. This drilling is summarized by time period and target area in Table 10.1.1 and Table 10.1.2, respectively.

Table 10.1.1 Summary of Drilling Completed on the Project by Time Period

Total Drilling Years	DDH		RC		RAB	
	Holes	m	Holes	m	Holes	m
Pre 2006	3	129	288	26,031	363	14,380
2011-2012 and 2017-2018	192	24,005	74	9,493	-	-
2020-2025	94	27,233	768	108,576	-	-
2025-2026*	51	16,650	-	-	-	-
Totals	340	68,017	1,130	144,100	363	14,380

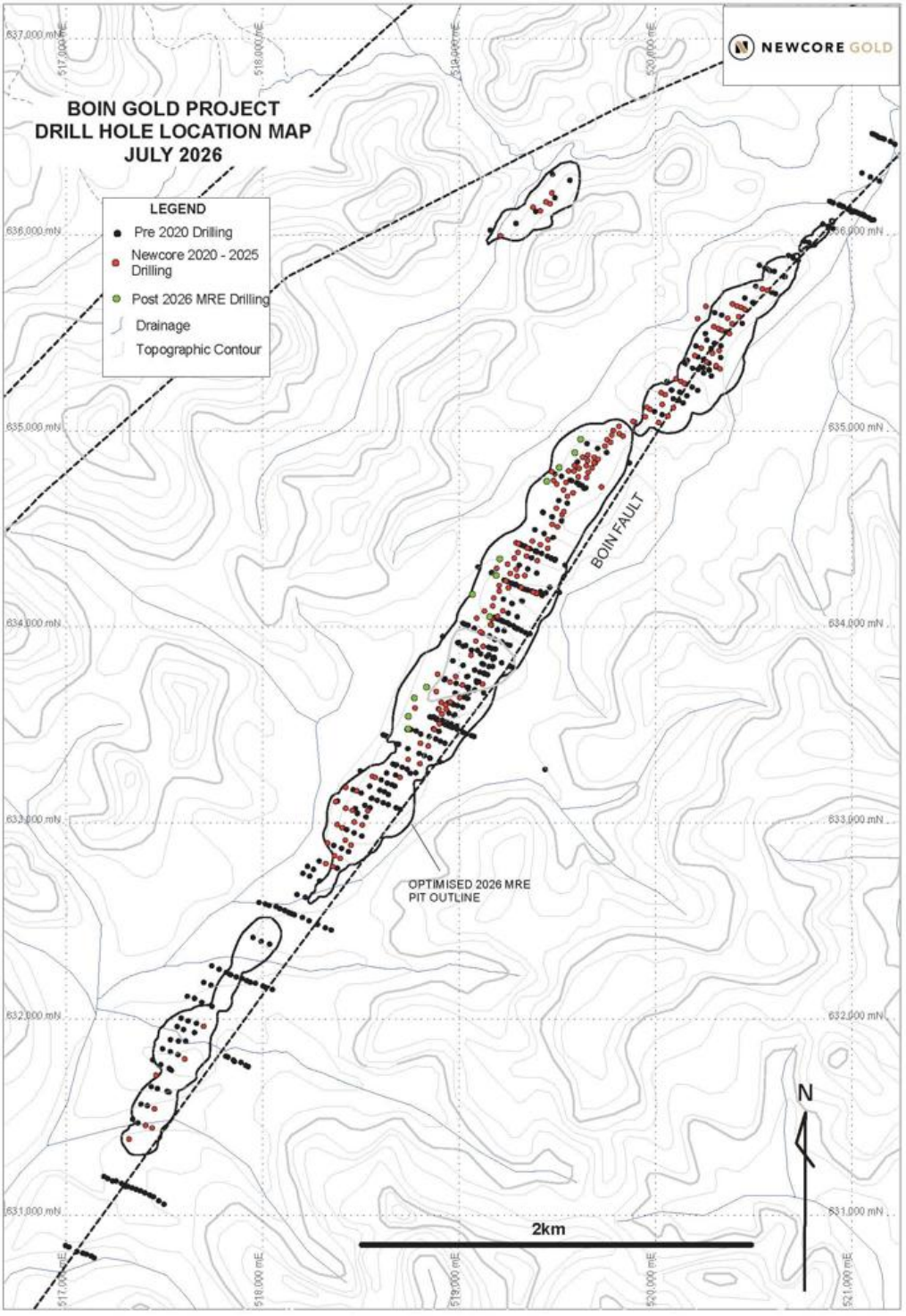
*Drilling completed and reported post Mineral Resource Estimate database cutoff date of 6 October 2025 and up until the report date.

Table 10.1.2 Summary of Drilling Completed on the Project by Target Area

Target	DDH		RC		RAB	
	Holes	m	Holes	m	Holes	m
Boin	119	24,716	373	51,088	275	9,341
Sewum	109	20,308	282	33,262	45	3,102
Nyam	84	18,848	216	26,180	11	421
Kwakyekrom	3	741	105	15,319	32	1,516
Tokosea	-	-	82	10,238	-	-
Nkwanta	-	-	1	100		
Kojina Hill	1	150	71	7,913		
Eradi	24	3,254	-	-		
Totals	**340**	**68,017**	**1,130**	**144,100**	**363**	**14,380**

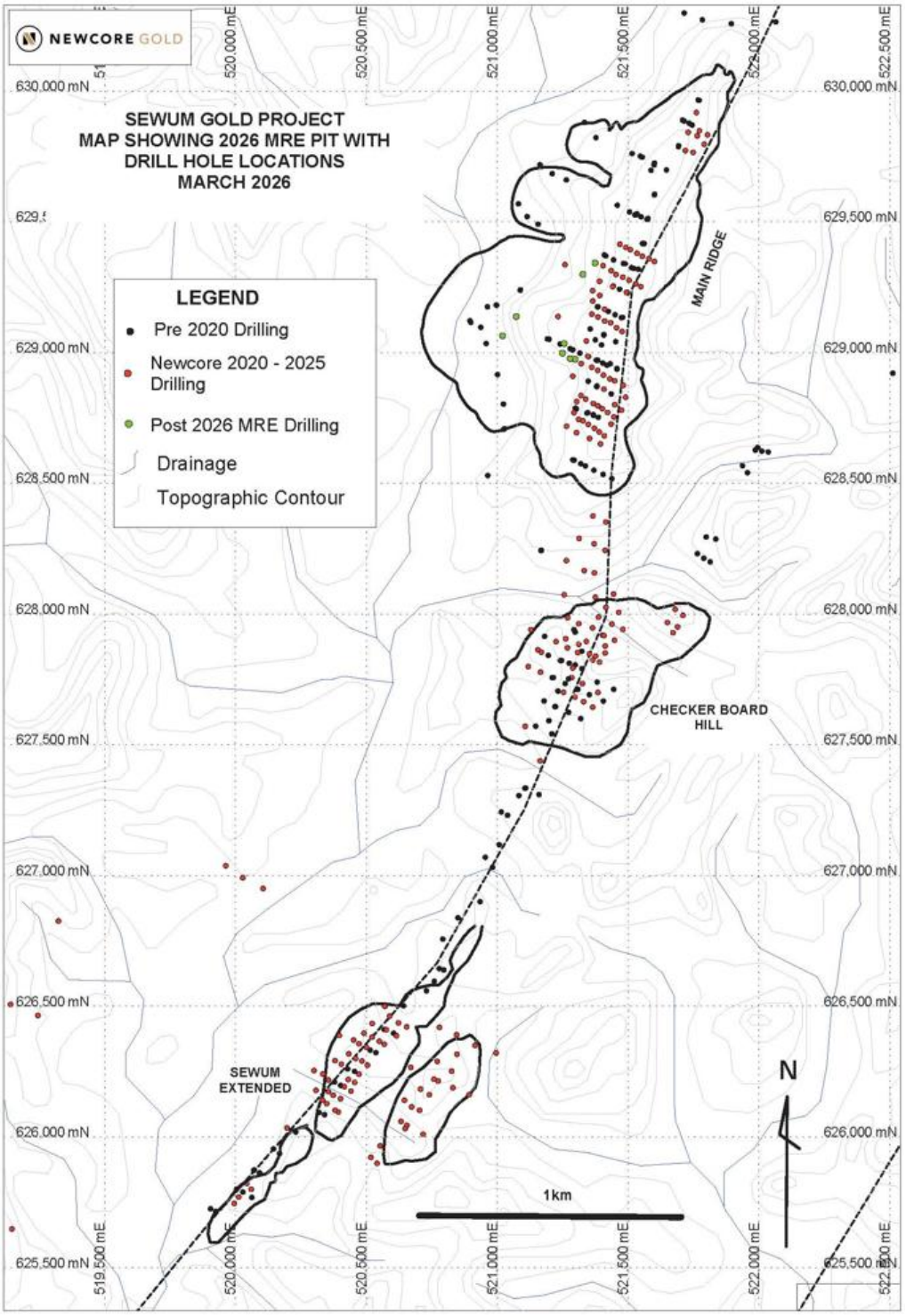
Drilling completed and reported up to the report date at the Boin, Sewum, Nyam and Kwakyekrom target areas is also summarized by time period in Figure 10.1.1 to Figure 10.1.4, respectively.

Figure 10.1.1 Summary of Drilling at the Boin Target Area by Time Period



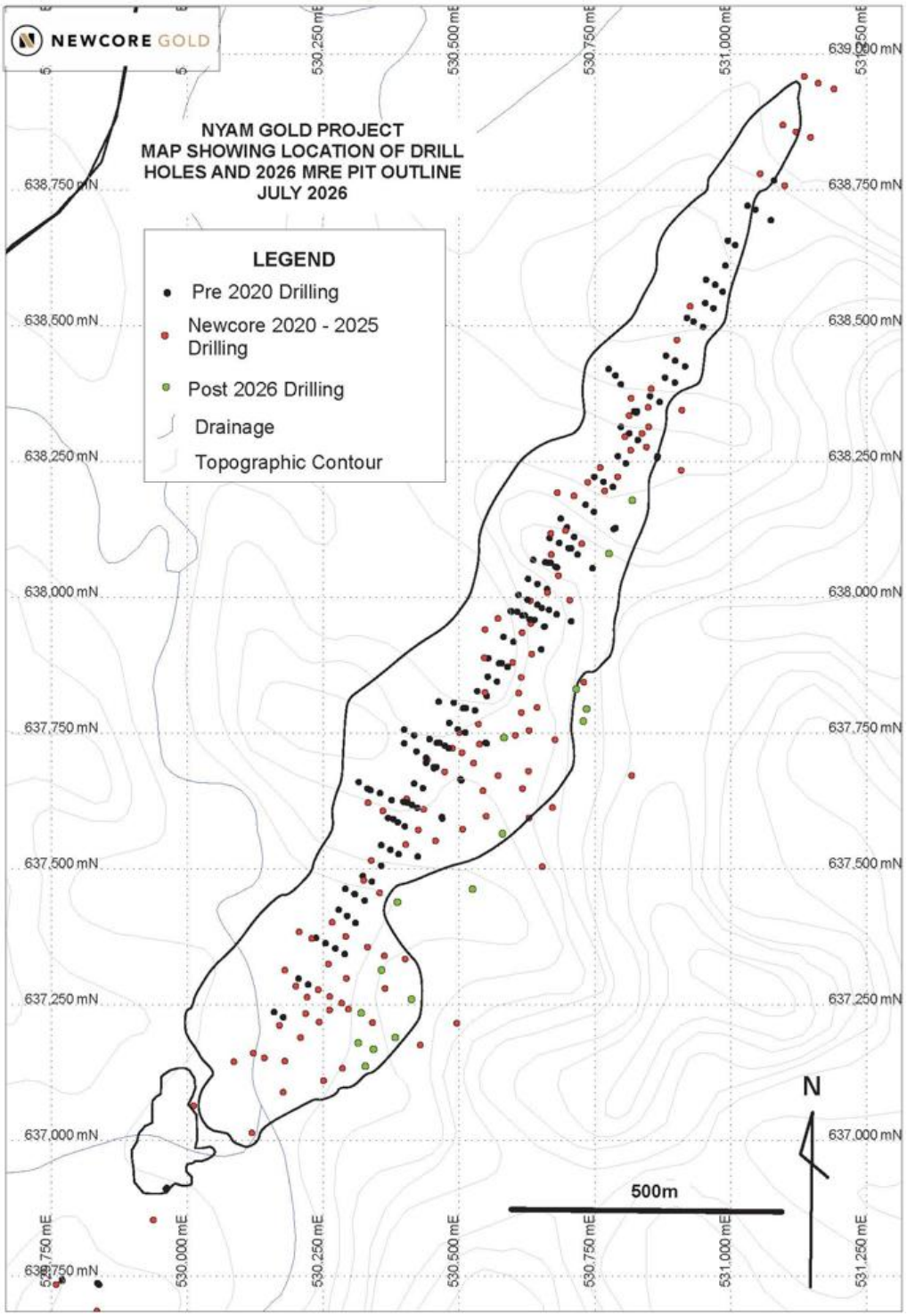
Source: Newcore, 2026

Figure 10.1.2 Summary of Drilling at the Sewum Target Area by Time Period



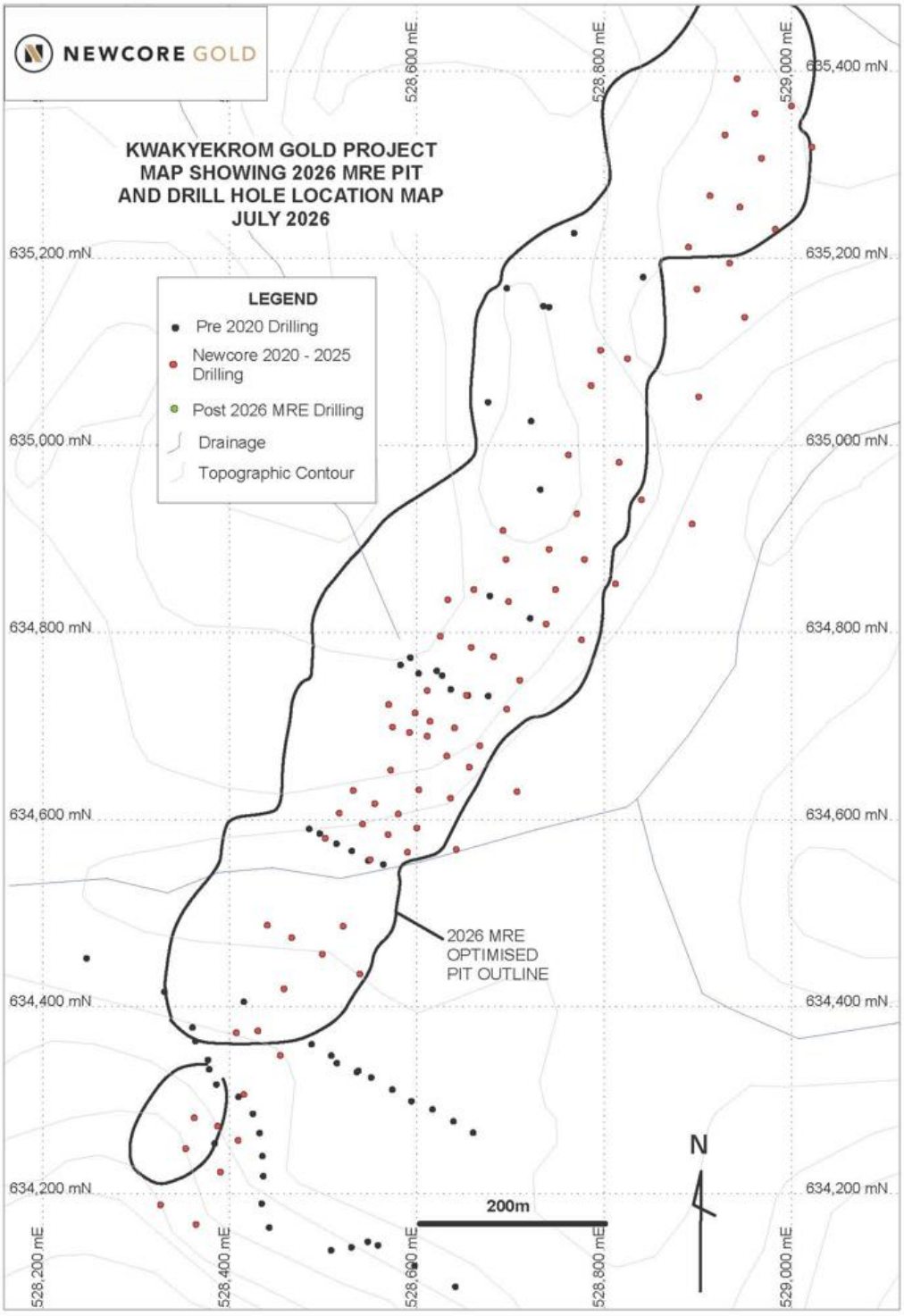
Source: Newcore, 2026

Figure 10.1.3 Summary of Drilling at the Nyam Target Area by Time Period



Source: Newcore, 2026

Figure 10.1.4 Summary of Drilling at the Kwakyekrom Target Area by Time Period



Source: Newcore, 2026

10.2 Property Drilling Summary

10.2.1 Historic Drilling, Previous Owners (Pre 2006)

Drilling on the Project was completed by previous owners comprised of Mutual Resources and Leo Shield from 1994 to 1998 and Red Back Mining (subsequently purchased by Kinross) during 2005-2006. Drilling consisted of shallow holes including 363 RAB holes totalling 14,380 m, 288 RC holes totalling 26,031 m, and 3 DDH holes totalling 129 m (Table 10.2.1). Drilling was concentrated at Boin, Sewum, and Nyam with the majority of the RC and RAB holes targeting the Boin area. Limited drilling was also completed at Kwakyekrom and Kojina Hill.

Table 10.2.1 Summary of Historic Drilling (Pre 2006) at the Project

Target	Pre 2006					
	DDH		RC		RAB	
	Holes	m	Holes	m	Holes	m
Boin	3	129	112	13,033	275	9,341
Sewum	-	-	51	4,490	45	3,102
Nyam	-	-	79	5,458	11	421
Kwakyekrom	-	-	13	682	32	1,516
Kojina Hill	-	-	33	2,368	-	-
Totals	3	129	288	26,031	363	14,380

The majority of the drilling focused on gold mineralization at the Boin and Sewum target areas with results including 1.79 g/t Au over 39 m from 111 m (KBRC052), 2.10 g/t Au over 33 m from 31 m (KBRC012), 1.92 g/t Au over 30 m from 26 m (KBRC008), 2.56 g/t Au over 21 m from 105 m (KBRC011), 4.11 g/t Au over 13 m from 19 m (KBRC100), 3.23 g/t Au over 16 m from 5 m (KBRC067), 0.63 g/t Au over 42 m from 58 m (SWRC005), and 0.87 g/t Au over 24 m from 44 m (SWRC002).

10.2.2 Drilling, Newcore Predecessors (2010–2018)

Predecessor companies (Edgewater and Pinecrest) of the current Owner completed drilling campaigns in 2011-2012 and 2017-2018. The majority of the RC and DDH drilling was completed at Boin, Sewum, and Nyam, with a minimal amount of drilling also completed at the Kojina Hill and Eradi targets. Drilling consisted of 74 RC holes totalling 9,493 m, and 192 DDH holes totalling 24,005 m (Table 10.2.2).

Table 10.2.2 Summary of Drilling by Newcore Predecessors

Target	2011-2012 / 2017-2018			
	DDH		RC	
	Holes	m	Holes	m
Boin	62	7,563	22	2,334
Sewum	71	9,994	27	3,930
Nyam	47	5,133	15	2,186
Kwakyekrom	1	101	-	-
Kojina Hill	1	150	10	1,043
Eradi	10	1,064	-	-
Totals	192	24,005	74	9,493

The drilling outlined gold mineralization over several kilometres at the Boin, Sewum and Nyam targets with results including 2.65 g/t Au over 35 m from 6 m (KBDD045), 0.35 g/t Au over 200 m from 1 m (SWDD038), 2.01 g/t Au over 29 m from 57 m (SWDD001), 1.41 g/t Au over 34 m from 45 m (KBDD011), 0.53 g/t Au over 81 m from 48 m (SWDD051), 1.12 g/t Au over 38 m from 77 m (SWRC064), 1.72 g/t Au over 24 m from 44 m (KBDD003), and 1.15 g/t Au over 35 m from 97 m (KBRC137).

10.2.3 Recent Drilling, Newcore (2020 to Present)

Newcore has completed drilling on the Project since 2020, with RC and DDH drilling primarily focused on Boin, Sewum, and Nyam. A smaller amount of drilling has also been completed at the Kwakyekrom and Tokosea areas, along with exploration drilling completed at the Kojina Hill and Eradi targets. Drilling in the resource areas as of the database cutoff for the Mineral Resource Estimate (6 October 2025) consisted of 657 RC holes totalling 93,736 m and 80 DDH holes totalling 25,044 m (Table 10.2.3).

Table 10.2.3 Summary of Recent Drilling by Newcore (2020-2025) at the Project

Target	2020-2025*			
	DDH		RC	
	Holes	m	Holes	m
Boin	33	9,741	239	35,721
Sewum	24	6,559	204	24,842
Nyam	21	8,103	122	18,536
Kwakyekrom	2	640	92	14,637
Tokosea	-	-	82	10,238
Nkwanta	-	-	1	100
Kojina Hill	-	-	28	4,502

Target	2020-2025*			
	DDH		RC	
	Holes	m	Holes	m
Eradi	14	2189.5	-	-
Totals	**94**	**27,233**	**768**	**108,576**

*Includes drilling up until the database cutoff for this Report (6 October 2025)

Drilling completed since 2020 has expanded the gold mineralization along strike and to depth at Boin, Sewum, Nyam, and Kwakyekrom, and has identified gold mineralized structures at Tokosea, Kojina Hill and Eradi.

Results from drilling completed at Boin include 204.3 g/t Au over 1 m from 59 m (KBRC301), 95.16 g/t Au over 2 m from 109 m (KBRC250), 2.25 g/t Au over 56 m from 96 m (KBRC352), 1.96 g/t Au over 62 m from 139 m (KBRC288), 4.41 g/t Au over 24 m from 126 m (KBRC357), 1.00 g/t Au over 103 m from 113 m (KBRC276), and 1.28 g/t Au over 74 m from 121 m (KBDD064).

Results from drilling completed at Sewum include 1.85 g/t Au over 62 m from 1 m with a second zone of 0.75 g/t Au over 68 m from 99 m (SWRC194), 0.82 g/t Au over 103 m from 61 m (SWRC093), 0.73 g/t Au over 68 m from 62 m (SWRC213), 1.98 g/t Au over 25 m from 3 m (SWRC164), 3.04 g/t Au over 16 m from 104 m and 0.91 g/t Au over 49 m from 39 m (SWRC076), 0.63 g/t Au over 77 m from 4 m (SWRC107), 1.97 g/t Au over 22 m from 86 m (SWRC083), 1.52 g/t Au over 28 m from surface (SWRC073), and 2.33 g/t Au over 14 m from 104 m (SWDD071B).

Results from drilling completed at Nyam include 2.04 g/t Au over 35 m from surface (NBRC075), 1.57 g/t Au over 40 m from 179 m (NBRC045), 1.38 g/t Au over 45 m from 8 m (NBRC017), 4.51 g/t Au over 13 m from 236 m (NBDD048), 1.94 g/t Au over 25 m from 222 m (NBRC047), 3.21 g/t Au over 15 m from 321 m (NBDD052), 0.78 g/t Au over 57 m from 74 m (NBRC073), 1.79 g/t Au over 24 m from 38 m (NBRC092), 0.85 g/t Au over 50 m from 1 m (NBRC076), and 1.62 g/t Au over 24.8 m from 248.7 m (NBDD056).

Results from drilling completed at Kwakyekrom include 0.79 g/t Au over 53 m from surface (KKRC045), 1.43 g/t Au over 26 m from 111 m (KKRC028), 0.61 g/t Au over 50 m from 148 m (KKRC051), 0.99 g/t Au over 29 m from 81 m (KKRC080), 4.71 g/t Au over 6 m from 34 m (KKRC023), and 0.94 g/t Au over 29 m from 160 m (KKRC056).

Post Database Cutoff Date Drilling (6 October 2025)

Newcore continues to drill on the Project and has released results for a total of 51 diamond drill holes (16,650 m) since the database cutoff date (6 October 2025) for the current MRE (Table 10.2.4). This drilling targeted mineralization below the constraining pits presented in this Report and focused on testing the depth extensions at the Boin, Nyam and Sewum targets.

DRA does not consider any of this drilling to be material to the Mineral Resource Estimate presented in this Report.

Table 10.2.4 Summary of Drilling Completed Post Database Cutoff Date (6 October 2025)

Target	2025-2026*	
	DDH	
	Holes	m
Boin	21	7,283
Nyam	16	5,612
Sewum	14	3,755
Totals	51	16,650

*Drilled and released since database cutoff date (6 October 2025).

Results from drilling completed at Boin include 173.75 g/t Au over 1.0 m from 264 m (KBDD098); 147.50 g/t Au over 1.0 m from 310 m (KBDD100) which notably included the first visible gold encountered by drilling on the Project; 3.54 g/t Au over 23.0 m from 253 m (KBDD097); 3.22 g/t Au over 17.0 m from 327 m, with a second interval of 1.70 g/t Au over 25.5 m from 285 m (KBDD102); and 1.23 g/t Au over 33.0 m from 337.0 m, including 2.16 g/t Au over 13.0 m from 357.0 m (KBDD107).

Results from drilling completed at Sewum include 1.59 g/t Au over 15.5 m from 147.5 m, including a higher-grade interval of 3.22 g/t Au over 6.4 m from 152.9 m (SWDD107); 0.69 g/t Au over 41.0 m from 21.5 m, with a second interval of 1.00 g/t Au over 10.5 m from surface and a third interval of 0.41 g/t Au over 25.2 m from 102.4 m (SWDD112); and 1.37 g/t Au over 24.0 m from 94.0 m including 1.93 g/t Au over 12.0 m from 103.0 m, and 0.55 g/t Au over 16.5 m from 30.0 m (SWDD121).

Results from drilling completed at Nyam include 3.51 g/t Au over 21.0 m from 245 m, with a higher-grade interval of 5.91 g/t Au over 9.0 m from 251 m (NBDD086); and 1.61 g/t Au over 14.0 m from 278 m with a higher-grade interval of 2.94 g/t Au over 7.0 m from 284 m (NBDD085).

Additional results from drilling completed at Sewum include 1.37 g/t Au over 24.0 m from 94 m, including a higher-grade interval of 1.93 g/t Au over 12.0 m from 103 m (SWDD121); and 0.74 g/t Au over 10.2 m from 54.3 m with a second zone of 9.24 g/t Au over 1.9 m from 251.4 m (SWDD115).

Additional results from drilling completed at Nyam include 3.67 g/t Au over 25.3 m from 307.7 m, (NBDD088); and 1.26 g/t Au over 18.0 m from 300.0 m including 2.31 g/t Au over 9.0 m from 300.0 m (NBDD091).

10.3 Drilling Procedures

10.3.1 Surveying

Collar Surveys

Prior to drilling a hole, the proposed collar position is located by tape and compass survey from the nearest point whose coordinates are accurately known, or by handheld GPS. When a surveyor is on site, the collar is located by Electronic Distance Measurement (EDM) survey (Figure 10.3.1).

The inclination is set using a clinometer attached to the rod tracks while the mast is tilted and checked and approved by the geologist prior to the start of drilling. Comparison of the first downhole surveys with the nominal collar dip and azimuth is completed by the geologist.

Prior to 2017, all collar locations were accurately surveyed using EDM, operated by qualified and experienced surveyors. It was the responsibility of the geologist to enter all collar details from each day of drilling into the relevant computer file.

Collar coordinates from drilling completed from 2020 to 2026 were determined by hand-held GPS. Under normal conditions, the Garmin 64 handheld unit can attain up to a 3 m accuracy in the X & Y coordinates. After a drill hole was completed, the geologist recorded the position of the drill collar using a handheld GPS. This XYZ data was then entered into the Project database.

Differential GPS surveying was subsequently completed in 2025 and 2026 and included the collars for >90% of the holes at Boin, Sewum, Nyam, and Kwakyekrom.

Drill collars are well-preserved with PVC pipes encased in concrete pillars with drill hole details displayed (Figure 10.3.2).

Figure 10.3.1 Collar Survey



Source: Newcore, 2026

Figure 10.3.2 Preserved Drill Collar



Source: Newcore, 2022

Downhole Surveys

A minimum of two downhole surveys are completed on each hole. For holes less than 100 m, the survey was completed at half-depth and at the end of the hole. Holes greater than 100 m are surveyed at 50 m intervals and at the end of the hole. All surveys are completed during the drilling process.

For diamond drilling completed from 2020 until 2022, drillholes were surveyed using a TruShot (BLY in-house design) downhole survey instrument. This is not a gyroscope, so azimuth readings are liable to interference from iron / magnetic substances. The TruShot tool is therefore attached to 3 m aluminium rods that protrude through the drill bit. Surveys are taken at 12 m and 30 m downhole and then at 30 m intervals. The technician supervising the drilling contractor photographed each downhole survey reading and texted it to the Project geologist for approval before drilling was allowed to proceed. These photographs are archived to act as a record of downhole survey data for each hole.

Drill core was also oriented by the drillers using a TruCore (BLY in-house design) orientation tool for collecting structural measurements. In competent rock, drill core was orientated for each 3 m drill run. If the ground was broken, the drillers used shorter drill runs, and the core was orientated for each run. The drillers placed a wax pencil mark on the bottom of the core. Newcore technicians placed the core on an 'angle iron' bench beside the drill rig. The individual core pieces were aligned, and the orientation mark traced along the base of the core as a solid black line. Downhole direction tick marks were drawn at regular intervals along the orientation line. If the technician was not certain of the core alignment, a dashed line was drawn.

All diamond drilling since 2020 has included core orientation. Downhole surveys are completed every 30 metres using downhole gyro survey instruments. The instrumentation includes a Reflex ACT 111 RD Downhole Unit and a Reflex EZTRAC Survey Instrument.

Digital Terrain Model

A digital terrain model (DTM) was created by flying a drone survey over the resource zones. Orthophotos were also generated from data and photos collected during the drone survey. The elevation (Z) data for drill collars was modified to fit the DTM surface. The Project database drill collar survey file therefore comprises handheld GPS coordinates (X & Y) and modified elevation (Z) data.

10.3.2 Drilling

Rotary Air Blast (RAB) Drilling

A downhole hammer was used to penetrate the ground with compressed air used to lubricate and cool the bit and carry drill cuttings to the surface. The drill cuttings were carried up the hole outside of the drill steel by compressed air. The drill cuttings were collected from the collar at 1 m intervals. The hole was flushed with compressed air after each 1 m interval to minimize downhole contamination.

Reverse Circulation (RC) Drilling

Only face-sampling hammers are used. A length of PVC casing is inserted into the top of the hole at a sufficient depth to create a secure seal at the top of the hole.

The hole is cleaned out at the end of each rod by blowing the hole in order to keep the hole dry and to reduce any potential contamination (Figure 10.3.3).

The cyclone is also cleaned after every hole to minimize contamination between holes.

Figure 10.3.3 Reverse Circulation Drill



Source: Newcore, 2021

Diamond Drilling (DDH)

Diamond drilling is completed using a wireline system to obtain a combination of PQ, HQ and NQ sized core (Figure 10.3.4). Holes are converted to HQ when poor ground conditions are encountered. Core is retrieved at 3 m interval runs and the core is placed in the core box by the drillers. Geotechnicians monitor the drill rig operation 24 hours per day. Forms are filled out during each shift recording the type of work completed and the time taken, such as rig shifts, pulling rods, changing the bit, drilling, breakdowns, and downhole surveys.

Figure 10.3.4 Diamond Drilling



Source: Newcore, 2022

10.3.3 Logging Procedures

Chip Logging

RC and RAB drill logs are completed manually on standard logging forms. All necessary fields are completed using a standard set of codes for documentation purposes. Records are kept of the percentage sample recovery for each 1 m interval (estimated by visual comparison) in the geological log report.

Samples are examined and logged on site, and washed chips are glued to chip boards (prior to 2017) or placed in chip trays (2017 to present) for future reference (Figure 10.3.5 to Figure 10.3.7). Chip boards and/or chip trays are stored at the Newcore field warehouse facility in Enchi.

Figure 10.3.5 RC Chip Logging



Source: Newcore, 2025

Figure 10.3.6 Chip Board Preparation



Source: Newcore, 2017

Figure 10.3.7 Example Chip Tray



Source: Newcore, 2022

Diamond Drill Core Logging

Full core boxes are collected at the end of each shift and taken to the Enchi site office. All drill core is systematically marked out, logged by geologists using geotechnical and geological logs, photographed, split in half using a diamond core saw and sampled at approximately 1.0 to 1.5 m intervals taking into account lithological contact and boundaries to visible mineralization. Very soft, clay-rich core is halved using a broad-bladed steel spatula (paint scraper). The following is a detailed description of the logging procedure used for the Project:

- Prior to logging, all drill core trays are laid out on logging shelves for geologists to confirm mark ups made by the technicians and to label the 1 m intervals on the core trays. All core is then photographed on a stand with a digital camera, a single box at a time together with a white board describing the date, borehole number, box number, and interval.

- After the entire core has been photographed, the core is laid out to be logged by geologists. Two logging forms are used: a descriptive geological form and a geotechnical form. The descriptive logs are used to record core recoveries, intensity of weathering, rock types, alteration styles and intensities, percentage and types of sulphides and other general information that cannot be recorded on the geotechnical logs. The geotechnical forms are primarily used to record detailed structural information (alpha - beta measurements) from the oriented drill core but also contain such information as rock quality designation (RQD), fracture and joint data, and core hardness.

- Structural measurements are recorded for veins, bedding and cleavages on a paper logging sheet. These readings are taken twice for each feature, first using a Konometer to record Alpha and Beta angles and secondly using a 'rocket launcher' core mount to record dip, dip direction and strike.

- Once completed, all the logging data is entered into the drillhole database.

10.3.4 Sampling Approach

Reverse Circulation Sampling

Sampling is completed at the rig. The standard form and ticket books are completed by a technician and signed off by the project geologist.

A 1 m sampling interval is used in all holes with the entire hole sampled.

Dry Samples

Each sample is collected in a large plastic bag clamped tightly onto the base of the cyclone.

In 2012, each sample was weighed, then a split was taken for analysis using a four-inch PVC tube splitter (Figure 10.3.8). Care was taken to ensure the tube was speared down the centre of the bag to the base of the plastic. Beginning in 2017, samples are passed through a riffle splitter, and an approximate 3 kg split is collected for submittal to the assay lab (Figure 10.3.9).

The sample split is placed in pre-numbered calico sample bags for dispatch to the geochemical laboratory. A record is made on the geological log and in the ticket books, at the drill site, of the sample identity numbers and corresponding intervals.

The splitter is thoroughly cleaned between samples to avoid cross-contamination.

Figure 10.3.8 Reverse Circulation Sampling Using Tube Splitter



Source: Newcore, 2012

Figure 10.3.9 Reverse Circulation Sampling Riffle Splitter



Source: Newcore, 2020

Wet Samples

Wet samples are collected in Fabrene bags and placed in the sun to allow the excess water to drain, and whenever possible, left to settle before subsequent sampling, using the same procedure as with the dry samples (Figure 10.3.10).

The samples are transported each day to Newcore's core storage facility to await shipment to the analytical laboratory. The core storage facility maintains a night watch person on the Property to ensure samples and equipment are not tampered with.

Figure 10.3.10 Drying Wet RC Samples



Source: Newcore, 2025

Diamond Drill Core Sampling

The following is the diamond drill core sampling procedure carried out on the Project:

- Once geologists have completed logging, the core is ready to be sampled and two aluminium tags are placed at each 1 m interval; one tag stating the depth down the hole, the other with the sample number for the 1 m interval.

- All diamond drill core is sampled at approximately 1 m intervals. When the core is too soft to be sawn using a diamond saw, the samplers use chisels or paint scrapers to halve the core. When cut using a saw (Figure 10.3.11), the core is cut in half (Figure 10.3.12) following the markings made by the geotechnicians at the rig site.

- The half-core not sampled is retained in the core boxes and stored for future reference, petrological work, further geochemical sampling, SG, or other engineering tests.

- All sampling is monitored by geologists. The half-core samples are placed in a numbered clear plastic bag and the numbered aluminium tag for that specific interval placed in the bag with the sample. During sampling, forms are completed recording the hole number, sample interval, sample number, and core loss.

- Every 10th sample is a QA/QC sample. These samples are prepared prior to core sampling and are placed in the sample stream. Every 20th sample is a duplicate, and between the duplicates either a standard or a blank is used. Duplicate samples are prepared at the laboratory. The entire sample is crushed to -2 mm and two splits (more than 1.5 kg) are collected from the one sample using a Jones Splitter and the splits are then processed as separate samples.

Figure 10.3.11 Core Cutting Area



Source: Newcore, 2022

Figure 10.3.12 Example Cut Drill Core



Source: Newcore, 2022

- Once sampling of an entire drillhole is complete, the 1 m samples are placed into nylon rice sacks, ten samples per sack. Each sack is tied and labelled with the company name and sample numbers the sack contains. All samples from a single drillhole are delivered to independent commercial laboratories (Intertek or SGS) as a single batch. If the samples are not sent the same day to the lab, they are stored in a room inside the Enchi site office until ready to be transported.

- Each batch of samples is delivered using the company's vehicles and drivers directly from site to the SGS or Intertek labs in Tarkwa, approximately a 4-hour drive from the Project. Each batch of samples is submitted to SGS or Intertek with a sample submission form outlining the method of preparation and analysis. Once the samples are delivered, the laboratory staff sign and date Newcore's copy of the sample submission form acknowledging receipt of the samples. Each time a delivery is made to the laboratory, any pulps available are collected and brought back to the Enchi site office for storage. The pulps from selected drillholes are regularly sent for umpire assaying at different laboratories as a check on the results that are received from SGS and Intertek.

- Assay results are received both electronically and in hard copy form.

10.4 QP's Opinion

It is the QP's opinion that the procedures employed by Newcore for all geological functions, including drilling and survey requirements, data capture and management, sampling and chain of custody, meet acceptable industry standards and the resulting information is appropriate for Mineral Resource Estimation.

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Rotary Air Blast (RAB)

11.1.1 Sample Preparation

The following is summarized from the Red Back Geologist's Procedures Manual (Red Back, 2005).

All RAB chip samples were prepared at the Intertek laboratory in Tarkwa using preparation code PT01/SP02.

Below is a brief description of the sample preparations procedure:

- Samples are sorted and dried at 105°C.
- Once dried, the entire sample is crushed to a 75% passing at 2 mm.
- The sample is then split to get a sample up to 2 kg in weight for pulverizing.
- The entire split sample is then pulverized to allow a 95% passing of 75 microns (µm).
- The pulp is split to 150 g for analysis.

Red Back drivers delivered the samples to the Intertek Tarkwa facility, which operated under the umbrella of Intertek / Genalysis Services Pty Ltd. The facility is certified with the following credentials: ISO 17025 and NATA certificate 3244.

At no time was an employee, officer, director, or associate of Newcore involved in the preparation of the samples.

11.1.2 Analytical Procedure

The following is summarized from the Red Back Geologist's Procedures Manual (Red Back, 2005).

The samples were assayed using the Fire Assay (FA) method (30-g aliquots). A 30-g portion of pulverized sample is weighed, mixed with a fluxing reagent containing litharge (PbO) and then placed into a fusion furnace and fused at approximately 1,100°C. During this stage, the reduced lead collects the precious metals and forms a button. The sample is then removed from the furnace and cooled. The lead button is separated from the silicate slag.

The second stage of FA is called cupellation. During the cupellation process, at approximately 950°C the lead in the button oxidizes and is absorbed into the cupel leaving a precious metal bead known as a prill. The resultant prill is digested with Aqua Regia, first by adding nitric acid to dissolve the silver, and then hydrochloric acid. Gold content is determined by Atomic Absorption Spectrometer (AAS). The detection threshold limits are in the range of 0.01 ppm to 100 ppm.

At no time was an employee, officer, director, or associate of Newcore involved in the analysis of the samples.

11.2 Reverse Circulation

11.2.1 Sample Preparation

All RC chip samples were prepared at the Intertek laboratory in Tarkwa using preparation code PT01/SP02.

Below is a brief description of the sample preparations procedure.

- Samples are sorted and dried at 105°C.
- Once dried, the entire sample is crushed to a 75% passing at 2 mm.
- The sample is then split to get a sample up to 2 kg in weight for pulverizing.
- The entire split sample is then pulverized to allow a 95% passing of 75 µm.
- The pulp is split to 150 g for analysis.

Each batch of samples is delivered using Newcore vehicles and drivers directly from site or picked up on site by representatives of the independent commercial lab operated by Intertek laboratory in Tarkwa, approximately 130 km from Enchi. Each batch of samples is submitted to Intertek with a sample submission form outlining the method of preparation and analysis. Once the samples are delivered, the laboratory staff sign and date Newcore's copy of the sample submission form acknowledging receipt of the samples.

The Intertek Tarkwa facility is independent of Newcore. The facility is certified with the following credentials: ISO 17025 and NATA certificate 3244.

Of the samples sent to the laboratories, 10% were either a duplicate sample, blank, or standard.

At no time was an employee, officer, director, or associate of Newcore involved in the preparation of the samples.

11.2.2 Analytical Procedure

The samples were assayed using the FA method (50-g aliquots). The same procedures as those described in Section 11.1.2 were followed.

At no time was an employee, officer, director, or associate of Newcore involved in the analysis of the samples.

11.3 Diamond Drilling

11.3.1 Sample Preparation

All drill core samples were prepared at the SGS or Intertek laboratory in Tarkwa using preparation code PRP89 or SP12 respectively:

- Samples are sorted and dried.
- Once dried, less than 3 kg of the sample is crushed to a 75% passing at 2 mm.
- Sample is split to get a 250 g sample for pulverizing.
- 250 g of the crushed sample is then pulverized to allow an 85% passing of 75 µm.

Each batch of samples is delivered using company vehicles and drivers directly from site or picked up on site by representative of the independent commercial labs operated by Intertek or SGS in Tarkwa, approximately a 4 hour drive from Enchi. Each batch of samples is submitted to SGS or Intertek with a sample submission form outlining the method of preparation and analysis. Once the samples are delivered, the laboratory staff sign and date Newcore's copy of the sample submission form acknowledging receipt of the samples. Of the samples sent to the laboratories, 10% were either a duplicate sample, blank, or standard.

Each time a delivery is made to the SGS or Intertek laboratory, any pulps available are collected and brought back to the Enchi site office for storage. The pulps from selected drillholes are sent regularly for umpire assaying and are sent to Intertek laboratories to check for gold FA and ICP multi trace element analysis.

At no time was an employee, officer, director, or associate of Newcore involved in the preparation of the samples.

11.3.2 Analytical Procedure

Samples were assayed for gold using a 50-g charge FA (code: FAA505, SGS or FA51 Intertek) using the following detection limits: Gold 0.01 ppm – 100 ppm; 50 g, FA, AAS finish.

A few selected holes were analysed for trace elements using the ICP12B method, which is based on a two-acid digest (aqua regia: a combination consisting of nitric acid and hydrochloric acid). Once the material is digested, the solution is analysed either by inductively coupled plasma-atomic emission spectroscopy (ICP-AES) or by inductively coupled plasma-mass spectrometry (ICP-MS) or by both. Two-acid digestion methods are the weakest of the digestions and silicate material is not affected, resulting in partial results for most elements.

The ICP12B method is recommended for samples with organic or high sulphide content.

SGS has geochemical accreditation and certification that conform with the requirements of CAN-P-1579 and CAN-P-4E (International Organization for Standardization / International Electrotechnical Commission), complemented by ISO/IEC 17025:2005 standard general requirements for the competence of testing and calibration laboratories. The Intertek Tarkwa facility is certified with the following credentials: ISO 17025 Standard and NATA certificate #3244.

At no time was an employee, officer, director, or associate of Newcore involved in the analysis of the samples.

11.4 Soil Sample Preparation and Analysis

Sample preparation and analyses were completed at the independent analytical facility of SGS or Intertek in Tarkwa, Ghana.

Soil samples were dried and pulverized to 90% -75 µm.

The analysis was completed with a 50-g FA with aqua regia digest and di-isobutyl ketone (DIBK) extraction with AAS finish with a lower detection limit of 1 ppb.

11.5 Trench Sample Preparation and Analysis

Trench samples were dried and pulverized to 90% -75 µm.

The analysis was completed with a 50-g FA with aqua regia digest with AAS finish at a 10 ppb detection limit.

11.6 Auger Sample Preparation and Analysis

Sample preparation and analysis was completed at the independent analytical facility of SGS or Intertek in Tarkwa, Ghana.

Auger samples were dried and pulverized to 90% passing -75 µm.

The analysis was completed with a 50-g FA with aqua regia digest and DIBK extraction with AAS finish with a detection limit of 10 ppb.

11.7 QA/QC

QA/QC programs were carried out during each drilling and trenching program. The QP generated and reviewed QA/QC charts for each program.

11.7.1 Soil

Blanks were inserted at a frequency of one every 50 samples with a minimum of one per batch. The material consisted of red-brown soils (2.5 kg) collected in Accra. Commercial standards were inserted at a frequency of one every 50 samples with a minimum of one per batch.

The QP has not reviewed the QA/QC results for the soil survey program. The soil results are not material to the Mineral Resource Estimate.

11.7.2 Auger

Blanks were inserted at a frequency of one every 50 samples with a minimum of one per batch. The material consisted of red-brown soils (2.5 kg) collected in Accra.

Commercial standards were inserted at a frequency of one every 50 samples with a minimum of one per batch.

The QP has not reviewed the QA/QC results for the auger survey program. The auger results are not material to the Mineral Resource Estimate.

11.7.3 Trench

Blanks were inserted at a frequency of one every 50 samples with a minimum of one per batch. The material consisted of oxide rock fragments supplied from Accra.

Commercial standards were inserted at a frequency of one every 50 samples with a minimum of one per batch.

The results of the trench QA/QC samples were incorporated with the drill results and charted accordingly.

11.8 Duplicates

Nine thousand five hundred and seventy-one (9,571) duplicates were available in the database; comparing the means and standard deviations of the duplicates with the original sample data results in similar distributions. Additionally, when conducting a t-test for paired means, the QP does not reject the null hypothesis.

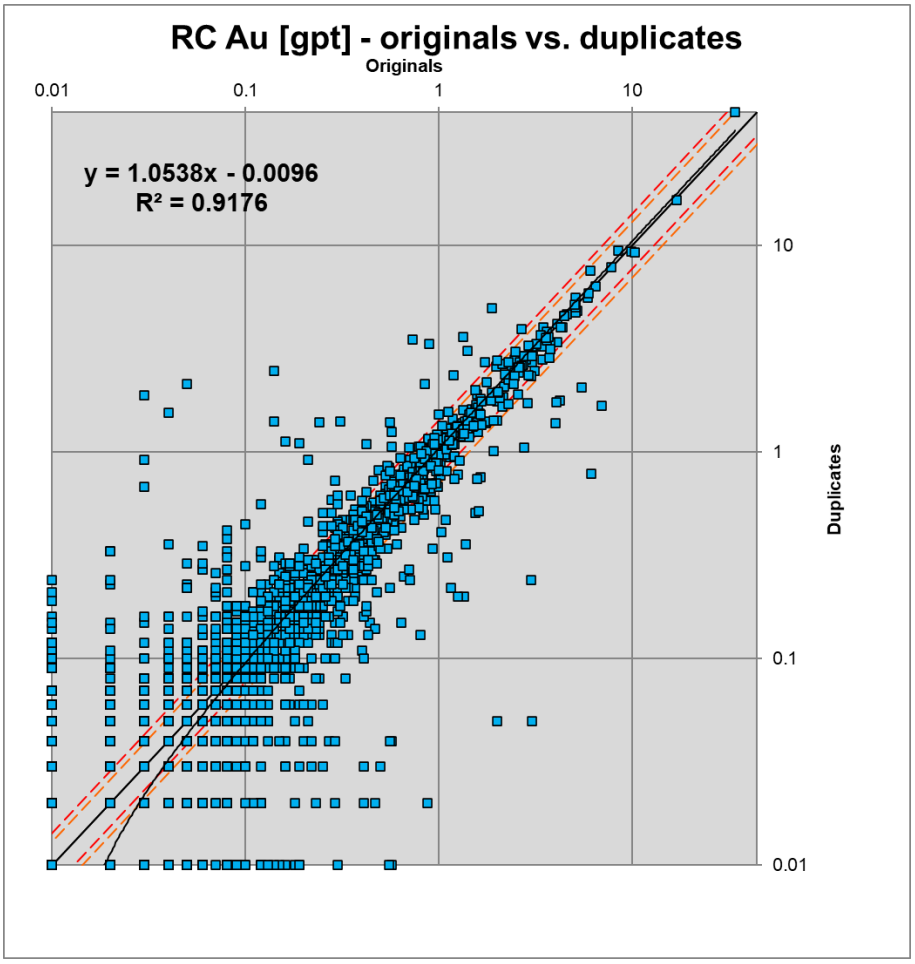
Table 11.8.1 Duplicates Comparison Statistics

Sheet	Pairs	Original Mean	Duplicate Mean	Original SD	Duplicate SD	T-Test P	Result
Duplicates	9,571	0.135	0.136	0.682	0.642	0.628	Do Not Reject
TRENCH	667	0.087	0.084	0.236	0.223	0.485	Do Not Reject
RC	6,601	0.146	0.148	0.755	0.686	0.564	Do Not Reject
HCORE	2,104	0.114	0.114	0.534	0.597	0.994	Do Not Reject
RAB	180	0.126	0.130	0.560	0.517	0.890	Do Not Reject
RD	19	0.111	0.219	0.173	0.470	0.147	Do Not Reject

11.8.1 RC Drilling

The scatter plot (Figure 11.8.1) shows that near the detection limit there is a lot of noise, this is not uncommon with duplicates, above 5 g/t there appears to be tight control. There is a considerable number of samples that fall outside the 20% interval between 0.5 g/t and 5 g/t; however, the majority of samples still plot within the limits. There is a strong correlation as can be evidenced from the high R2 of 0.92 and the small intercept. In addition to possible nugget effects, there may be issues retaining a homogeneous split of RC cuttings. The QP believes there is reasonable correlation between the originals and the duplicates.

Figure 11.8.1 RC Duplicates

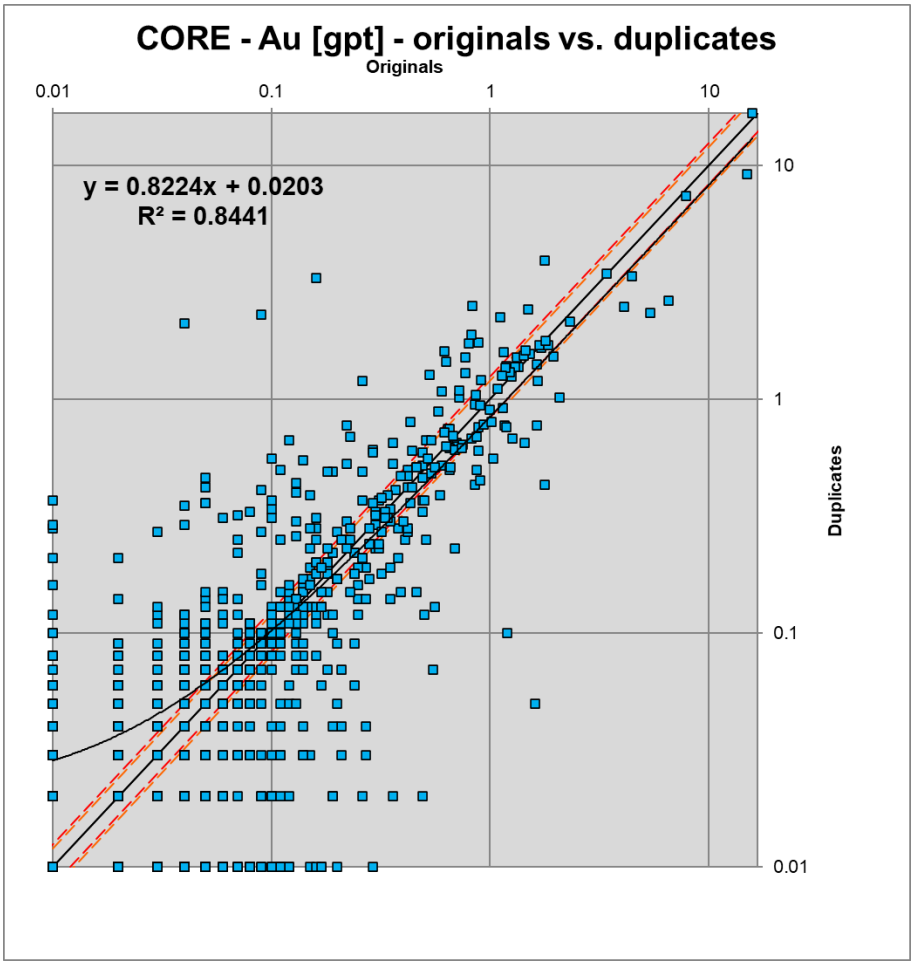


Source: DRA, 2026

11.8.2 Diamond Drilling

The scatter plot (Figure 11.8.2) shows that near the detection limit there is a lot of noise, this is not uncommon with duplicates, above 5 g/t there appears to be tight control. There are a considerable number of samples outside the 20% although the duplicates are relatively equally split across the 1:1 line. There is a moderately strong correlation as can be evidenced from the R2 of 0.82 and the small intercept. The QP believes there is reasonable correlation between the originals and the duplicates. Additional care should be taken to conduct checks on spurious duplicates.

Figure 11.8.2 Core Duplicates



Source: DRA, 2026

11.8.3 Conclusion

Duplicates have reasonable reproducibility for all sample types and are suitable for resource estimation.

11.9 Standards

Newcore has used a variety of standards over the duration of the Project (Table 11.9.1), the standards have been selected with appropriate grade ranges, from industry standard suppliers, there is a mix of matrices that are suitable for the deposit.

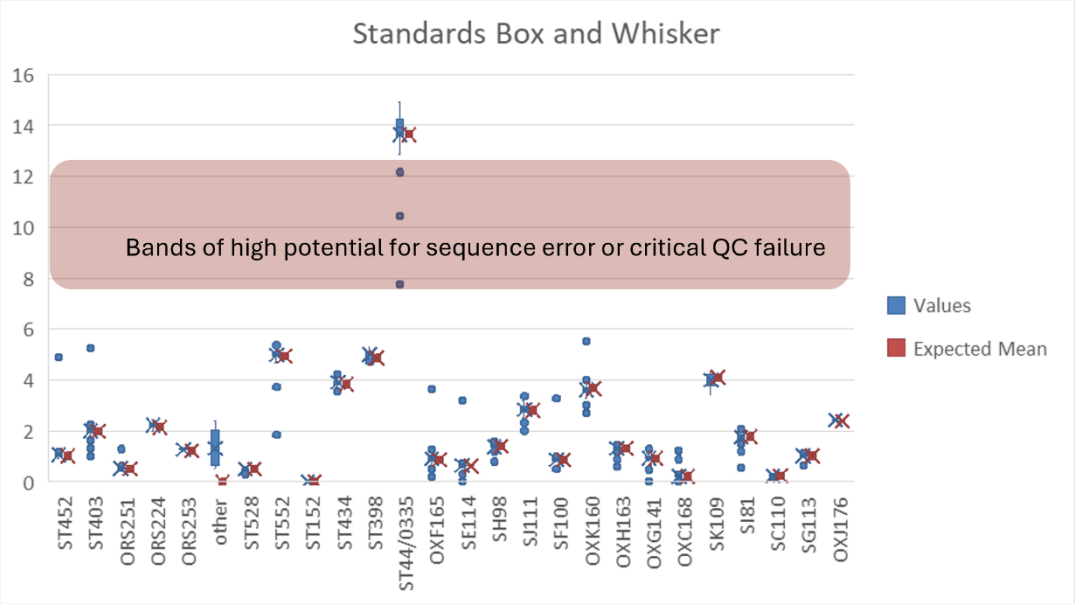
Table 11.9.1 List of Certified Reference Materials Used at Enchi

STD	Certifier	Count	Au g/t	SD
ORS224	OREAS	91	2.154	0.053
ORS251	OREAS	118	0.504	0.015
ORS253	OREAS	117	1.220	0.044
OXC168	ROCKLABS	141	0.218	0.006
OXF165	ROCKLABS	137	0.875	0.017
OXG141	ROCKLABS	95	0.930	0.016
OXH163	ROCKLABS	185	1.313	0.026
OXJ176	ROCKLABS	1	2.385	0.048
OXK160	ROCKLABS	148	3.674	0.078
SC110	ROCKLABS	130	0.235	0.009
SE114	ROCKLABS	279	0.634	0.016
SF100	ROCKLABS	147	0.860	0.016
SG113	ROCKLABS	97	1.024	0.019
SH98	ROCKLABS	243	1.400	0.028
SI81	ROCKLABS	171	1.790	0.03
SJ111	ROCKLABS	149	2.812	0.068
SK109	ROCKLABS	22	4.102	0.098
ST152	Gannet Holdings Pty	43	0.016	
ST398	Gannet Holdings Pty	97	4.870	0.17
ST403	Gannet Holdings Pty	575	1.990	0.09
ST434	Gannet Holdings Pty	15	3.840	0.15
ST44/0335	Gannet Holdings Pty	44	13.650	
ST452	Gannet Holdings Pty	199	1.030	0.05
ST528	Gannet Holdings Pty	128	0.510	0.03
ST552	Gannet Holdings Pty	232	4.930	0.18

11.9.1 Review of Outliers

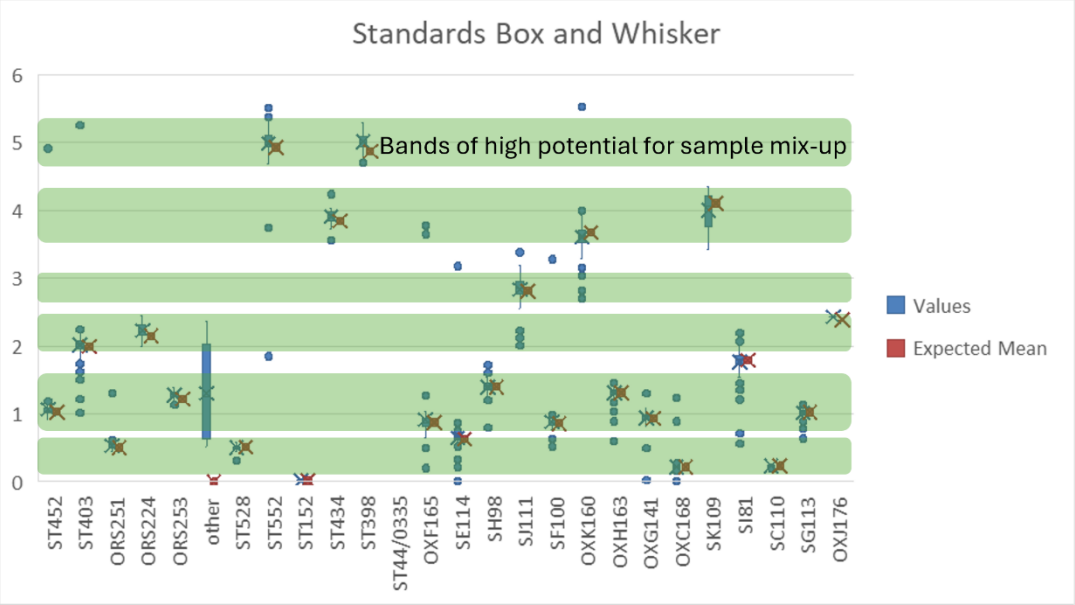
A thorough study has not been conducted at this time, but previous studies have reported on QA/QC samples by drill campaign. There are potential mix ups of standards and possible sequence order issues. For the size of the drillhole database and the number of quality samples there is no reason to believe that there would be any sizable impact on resources.

Figure 11.9.1 Box and Whisker Plots – All Standards



Source: DRA, 2026

Figure 11.9.2 Box and Whisker Plots – Excluding st44 for Scale



Source: DRA, 2026

11.9.2 Statistical Review of Standards

Table 11.9.2 presents a summary of some basic statistics reviewing the similarity of the standard values distribution to the expected; there is a certain amount of variability with some comparisons more similar than others.

Table 11.9.2 T-Test and Binomial Test and CUSUM Data for Standards – Major Outliers Removed

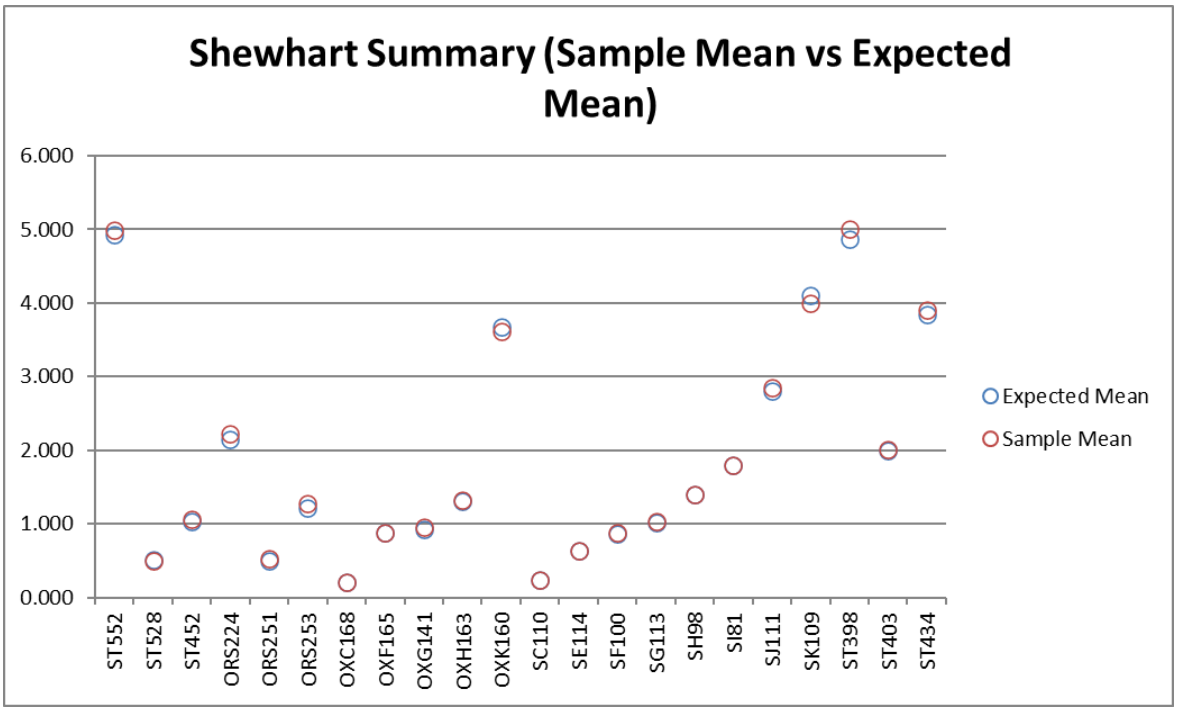
Standard	N	Sample Mean	Expected Mean	T Stat	T-Test P	T-Test Result	Binomial P	Binomial Result	CUSUM Max	CUSUM Alert
ST552	225	4.985	4.930	2.47	0.014	Reject Null	0.616	Do Not Reject	9.0	Drift Alert
ST528	127	0.501	0.510	-3.07	0.003	Reject Null	0.233	Do Not Reject	2.0	In Control
ST452	197	1.064	1.030	1.72	0.088	Do Not Reject	0.379	Do Not Reject	84.3	Drift Alert
ORS224	75	2.228	2.154	6.74	0.000	Reject Null	0.000	Reject Null	68.6	Drift Alert
ORS251	101	0.532	0.504	3.42	0.001	Reject Null	0.000	Reject Null	138.0	Drift Alert
ORS253	99	1.282	1.220	13.07	0.000	Reject Null	0.000	Reject Null	90.0	Drift Alert
OXC168	133	0.211	0.213	-1.15	0.252	Do Not Reject	0.000	Reject Null	28.0	Drift Alert
OXF165	131	0.878	0.875	0.49	0.623	Do Not Reject	0.000	Reject Null	49.3	Drift Alert
OXG141	92	0.956	0.930	4.75	0.000	Reject Null	0.000	Reject Null	108.3	Drift Alert
OXH163	181	1.319	1.313	1.01	0.315	Do Not Reject	0.000	Reject Null	26.4	Drift Alert
OXK160	144	3.615	3.674	-4.00	0.000	Reject Null	0.000	Reject Null	13.8	Drift Alert
SC110	130	0.234	0.235	-1.00	0.319	Do Not Reject	0.011	Reject Null	12.3	Drift Alert
SE114	270	0.641	0.634	2.12	0.035	Reject Null	0.000	Reject Null	61.1	Drift Alert
SF100	134	0.876	0.860	3.36	0.001	Reject Null	0.000	Reject Null	77.5	Drift Alert
SG113	86	1.027	1.024	1.13	0.262	Do Not Reject	0.007	Reject Null	8.2	Drift Alert
SH98	239	1.394	1.400	-1.23	0.219	Do Not Reject	0.000	Reject Null	15.1	Drift Alert
SI81	163	1.791	1.790	0.09	0.932	Do Not Reject	0.000	Reject Null	42.7	Drift Alert
SJ111	141	2.857	2.812	4.54	0.000	Reject Null	0.000	Reject Null	57.3	Drift Alert
SK109	22	4.001	4.102	-1.78	0.090	Do Not Reject	0.000	Reject Null	3.3	In Control
ST398	97	5.012	4.870	12.66	0.000	Reject Null	0.927	Do Not Reject	32.4	Drift Alert
ST403	566	2.015	1.990	3.26	0.001	Reject Null	0.236	Do Not Reject	35.8	Drift Alert
ST434	15	3.913	3.840	1.89	0.080	Do Not Reject	0.927	Do Not Reject	4.4	In Control

Outliers not removed for ST series

CUSUM = Cumulative Sum

There have been some sample mislabels in the historical dataset, with most of the obvious outliers having been removed. Overall, there is reasonable accuracy, but the precision is not meeting the characteristics for some of the control standards.

Figure 11.9.3 Summary of Sample Means by Standard Type



Source: DRA, 2026

The sample standards were all plotted on Shewhart charts to review performance over time, and the average sample mean and expected means were compared for all standard types. Figure 11.9.3 highlights that, among all standard types, there is close accuracy, some perform slightly above reference average and some below. There is good overall control on accuracy, with the means of the standards similar to the expected means.

The QP reviewed the precision and bias further and the results are tabulated in Table 11.9.3. It can be seen that, based on DRA's criteria, the bias is low. Table 11.9.4 to Table 11.9.6 and Equations 11.1 present the classification. However, the precision is not performing up to the referenced levels. It is recommended to continue to work on improving the precision to reduce the spread of the data. Additionally, Shewhart graphs were reviewed using the standard deviation of the independent lab instead of the reference material; there appears to be normal distribution except for a few potential drifts. It is the QP's opinion that the dataset is sufficient for resource estimation for the current block size and open-pit style mineralization.

Table 11.9.3 Precision and Bias Analysis of Analytical Standards

Standard	No	Mean	Ref Mean	Bias %	Bias	SD	RSD %	Precision	Precision Bias Matrix	Precision and Bias Rank
ST552	225	4.985	4.930	1.1	Low bias	0.332	6.7	Moderate precision	1	Precision issue
ST528	127	0.501	0.510	-1.7	Low bias	0.032	6.4	Moderate precision	1	Precision issue
ST452	197	1.064	1.030	3.3	Low bias	0.280	26.3	Low precision	1	Precision issue
ORS224	75	2.228	2.154	3.4	Low bias	0.095	4.3	High precision	0	Excellent
ORS251	101	0.532	0.504	5.5	Moderate bias	0.082	15.4	Low precision	3	Failing QA/QC
ORS253	99	1.282	1.220	5.1	Moderate bias	0.047	3.7	High precision	2	Systematic Bias
OXC168	133	0.211	0.213	-1.1	Low bias	0.023	11.1	Low precision	1	Precision issue
OXF165	131	0.878	0.875	0.3	Low bias	0.068	7.8	Moderate precision	1	Precision issue
OXG141	92	0.956	0.930	2.8	Low bias	0.053	5.5	Moderate precision	1	Precision issue
OXH163	181	1.319	1.313	0.4	Low bias	0.078	5.9	Moderate precision	1	Precision issue
OXK160	144	3.615	3.674	-1.6	Low bias	0.178	4.9	High precision	0	Excellent
SC110	130	0.234	0.235	-0.5	Low bias	0.013	5.6	Moderate precision	1	Precision issue
SE114	270	0.641	0.634	1.0	Low bias	0.051	7.9	Moderate precision	1	Precision issue
SF100	134	0.876	0.860	1.9	Low bias	0.056	6.3	Moderate precision	1	Precision issue
SG113	86	1.027	1.024	0.3	Low bias	0.028	2.8	High precision	0	Excellent
SH98	239	1.394	1.400	-0.4	Low bias	0.070	5.0	Moderate precision	1	Precision issue
SI81	163	1.791	1.790	0.0	Low bias	0.110	6.1	Moderate precision	1	Precision issue
SJ111	141	2.857	2.812	1.6	Low bias	0.118	4.1	High precision	0	Excellent
SK109	22	4.001	4.102	-2.5	Low bias	0.266	6.7	Moderate precision	1	Precision issue
ST398	97	5.012	4.870	2.9	Low bias	0.110	2.2	High precision	0	Excellent
ST403	566	2.015	1.990	1.3	Low bias	0.182	9.0	Moderate precision	1	Precision issue
ST434	15	3.913	3.840	1.9	Low bias	0.150	3.8	High precision	0	Excellent

RSD = Relative Standard Deviation

Table 11.9.4 Bias Classifier and Precision Classifier

Bias %	Qualitative Classifier	RSD %	Qualitative Classifier
0 – 5	Low bias	0 – 5	High precision
5 – 10	Moderate bias	5 - 10	Moderate precision
10+	High bias	10+	Low precision

Equations 11.1 – Relative Standard Deviation and Percent Bias

$$RSD\% = \frac{SD}{\bar{x}} \times 100$$

$$Bias(\%) = \frac{\bar{x} - \mu_{ref}}{\mu_{ref}}$$

Table 11.9.5 Precision and Bias Matrix

	High Precision	Moderate Precision	Low Precision
Low bias	0	1	1
Moderate bias	2	3	3
High bias	2	3	4

Table 11.9.6 Precision and Bias Matrix Classifier

Rank	Classifier
0	Excellent
1	Precision issue
2	Systematic Bias
3	Failing QA/QC
4	QA/QC Out of Control

11.9.3 Rocklabs OxC 168 – Example of Review Charts

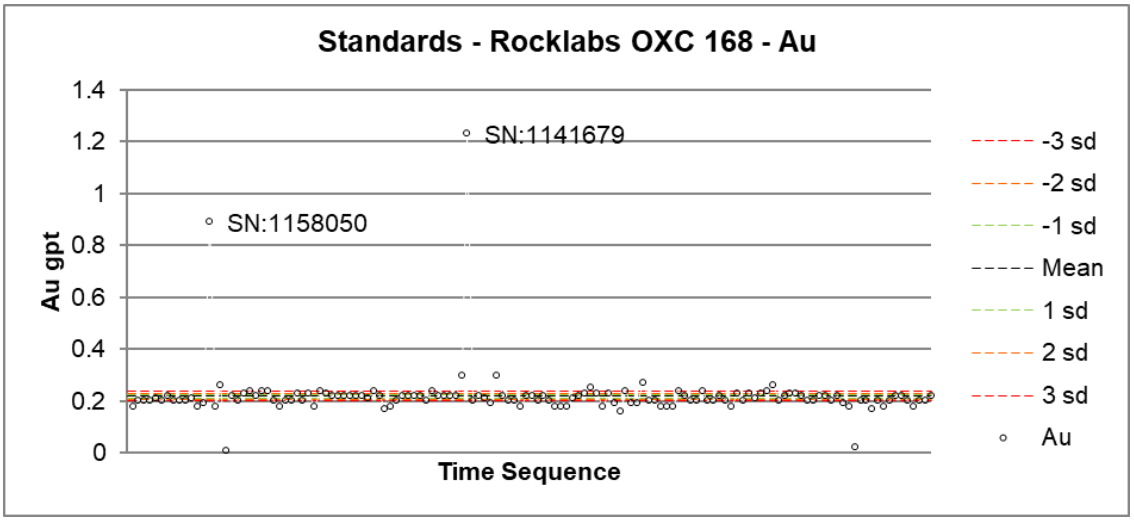
For Pb Fire Assay: Au Gold (ppm) certified value: 0.218, Standard Deviation 0.006

"*Origin of Reference Material: Basalt and feldspar minerals with minor quantities of finely divided gold-containing minerals that have been screened to ensure there is no gold nugget effect.*" (Rocklabs OxC 168 certificate)

Table 11.9.7 Rocklabs OxC 168 – Sample Descriptive Statistics

OxC168	Sample	SRM
Count	133	
Mean	0.211	0.218
Median	0.21	
Minimum	0.16	
Maximum	0.3	
Mode	0.2	
Std Dev	0.02	0.006
Variance	0.00	
Skewness	0.86	
Kurtosis	2.12	

Figure 11.9.4 Rocklabs OxC 168 Outliers and Mislabelled Samples



Source: DRA, 2026

Table 11.9.8 Rocklabs OxC 168 – Outliers Removed

Hole	Sample	Standard	Batch	Au (g/t	Comments
SWDD082	1141679	OxC168	NCD_021	1.23	N/A
SWRC132	1158050	OxC168	4577	0.89	N/A
NBRC093	1153930	OxC168	4545	0.01	N/A
ERDD022	1146921	OxC168	NCD_029	0.02	N/A

Outliers are not exhaustive, in-depth review of surrounding sample values not conducted for possible sequence errors.
N/A: Not Applicable

Figure 11.9.5 Using Certified Mean and Standard Deviation

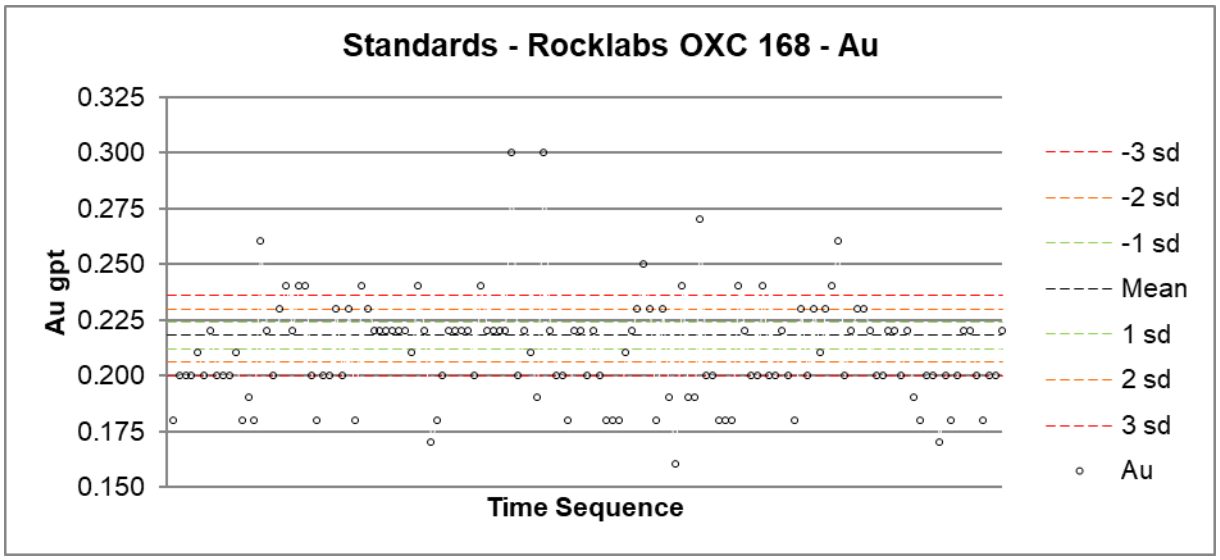
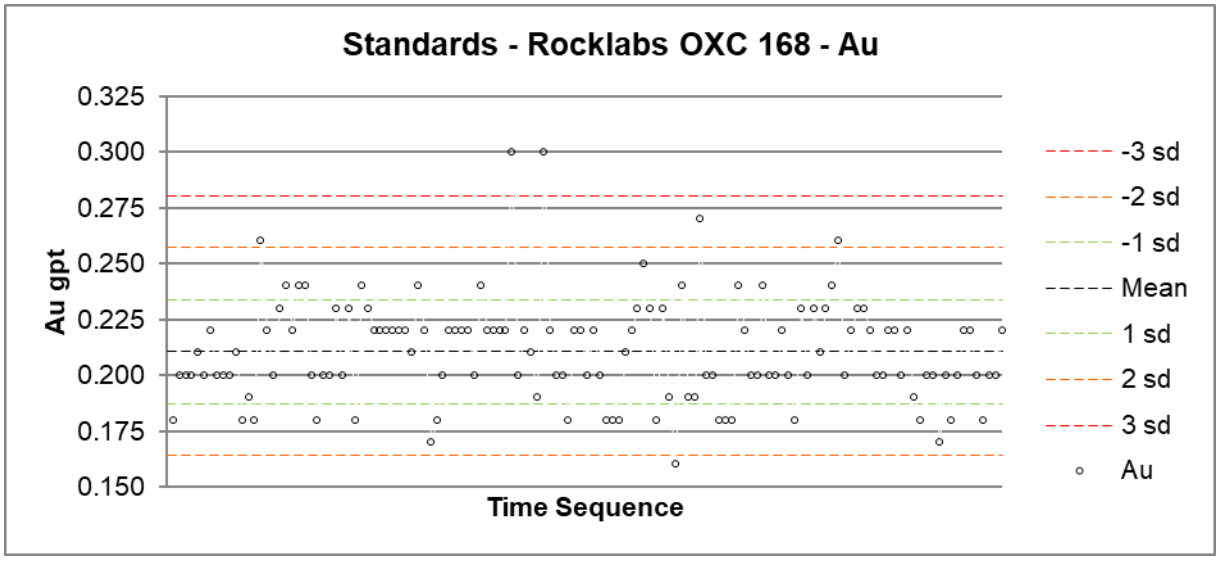


Figure 11.9.6 Using Sample Population Mean and Standard Deviation



11.9.4 Conclusion

Upon review of the bias and precision globally of all the standards, there appears to be issues with precision, and this is reflected in the control charts; however, there is reasonable global accuracy. It is the QP's opinion that the dataset is good for use for resource estimation.

11.10 Blanks

The master database records 5,557 blanks, 65 of which (1% of data) are above the 10x detection limit, affecting about 30 batches. Most of these 65 samples have been reviewed and appear to be related to sample mix-up or sequence errors, and not as a result of contamination. Only two batch sample sequences may have contamination, both are RC holes, neither appear detrimental, and based on the style of mineralization the width of zones would not be skewed particularly in relation to RC sampling precision. Most blanks are less than 10x detection and most at or below detection limit. Data is fit for purpose.

11.11 QP's Opinion

It is the QP's opinion that the sample preparation, analytical procedures, and security measures put in place for the trenches, reverse circulation, and diamond drill programs met acceptable industry standards at the time and that the information can be used for geological and mineral resource modelling.

12.0 DATA VERIFICATION

12.1 Site Inspection (Pre-DRA)

12.1.1 2010

Mr. Todd McCracken, P.Geo., was a qualified person (QP) and co-author of the Project NI 43-101 compliant Technical Report, dated 19 April 2023 entitled "NI 43-101 Technical Report – Mineral Resource Estimate for the Enchi Gold Project, Ghana" (McCracken and Meadows Smith, 2023). Mr. McCracken visited the Property and the Accra offices of Red Back between 18-22 March 2010 inclusive. Mr. McCracken inspected drill collar locations, trench locations, property geology, drill core and chip boards.

The QP is not treating the 2010 site inspection as a current inspection.

12.1.2 2011

Mr. McCracken visited the Property from 11-16 December 2011. Mr. McCracken inspected drill collar locations, property geology and chip boards.

The QP is not treating the 2011 site inspection as a current inspection.

12.1.3 2014

Mr. McCracken visited the Property from 28 April-1 May 2014. Mr. McCracken inspected drill collar locations, property geology and chip boards.

The QP is not treating the 2014 site inspection as a current inspection.

12.1.4 2017

Mr. Joe Amanor, MAusIMM (CP), was a qualified person (QP) and co-author of the 2021 report entitled "NI 43-101 Technical Report Preliminary Economic Assessment for the Enchi Gold Project, Enchi, Ghana." (McCracken et al., 2021). Mr. Amanor visited the Property from 6-7 June 2017, and from 5-6 September 2017. Drilling was confirmed through a site inspection, which included review of chip trays of representative material and original logs from RC drilling completed in 2017 and 2018, as well as field inspections of the locations for the drillholes which are marked by concrete monuments.

The QP is not treating the 2017 site inspection as current.

12.1.5 2020 – 2021

Mr. Amanor visited the Property from 5-8 November 2020 as well as from 2-6 June 2021 (inclusive). Drilling was confirmed through a site inspection, which included review of chip trays of representative material and original logs from RC drilling completed in 2020 and 2021, as well as field inspections of the locations for the drillholes, which are marked by concrete monuments. Locations were confirmed through verification with adjacent drillholes and with GPS checks.

The QP is not treating the 2020 and 2021 site inspection as current.

12.1.6 2022

Mr. Simon Meadows Smith, Fellow of the Institute of Materials, Minerals and Mining (FIMMM) with registration number 49627 of SEMS Exploration Services Ltd is a QP and co-author of the 2023 report (McCracken and Meadows Smith, 2023).

The QP site visit included a field inspection of the Boin, Sewum, Nyam, Kwakyekrom, and Tokosea areas at Enchi. The field inspection included confirmation of selected holes from the drilling completed in 2021 and 2022, verifying the locations for the drillholes which are marked by concrete monuments. Locations were confirmed through GPS checks. Coordinates in UTM WGS84 zone 30N were recorded for eight drill collars using a GARMIN GPSMAP 64x handheld GPS. Downhole information, etched into the concrete pillars, was also recorded for each of the eight holes. A comparison of drillhole survey data recorded by the QP and presented in the Enchi Gold Project database confirmed the data.

A visit was completed to the diamond drill on site (on a scheduled day off during the visit) consisting of a Boart Longyear (BLY), track-mounted LF 900 diamond core rig which was found to be relatively new and in good working condition. Discussions were held on drilling techniques, downhole surveying, and core orientation process.

The Newcore core yard was inspected, which included core handling, logging, cutting, sampling, and storage facilities. Additionally, the site inspection included a review of chip trays of representative material and original logs from RC drilling completed in 2021 and 2022, as well as an inspection of diamond drill core completed during this same time frame. The site visit also included an inspection of the geological office and designated density measurement room, both found to be in good condition.

Figure 12.1.1 2022 QP Site Visit



Source: Meadows Smith, 2022

12.1.7 2023

Mr. Amanor visited Enchi for from 16-18 November 2023. During his visit to the Project, Mr. Amanor inspected five diamond drill holes that were completed at Nyam (NBDD060 to NBDD064).

Drill collar locations were confirmed through GPS checks on each hole. Coordinates in UTM WGS84 zone 30N were recorded using a GARMIN GPSMAP 64x handheld GPS. Downhole information, etched into the concrete pillars, was also recorded for each hole. A comparison of drillhole survey data recorded by the QP and presented in the Enchi Gold Project database confirmed the data.

12.2 Drill Collar Validation (Pre-DRA)

12.2.1 Pre-2011

A validation of the Red Back RC drill collars was conducted during the 2010 site visit. Seventeen collars representing 11% of the RC drilling completed by Red Back were surveyed using a handheld Garmin GPSMAP 60CSx. GPS readings were collected in UTM WGS 84 coordinate system. Two of the Boin Zone collars had substantial errors, likely due to the collar number being incorrectly recorded in the field. The accepted error for the handheld GPS is typically 3 to 5 m of which all other collars but one passed (SWRC005).

Validation of the RAB holes could not be completed during the site visit as there were no monuments marking the location of the RAB holes.

12.2.2 2011

A validation of the Edgewater diamond and RC drill collars was conducted during the 2011 site visit. Twenty-one collars representing 2% of the drilling completed on the Project at the time were surveyed using a handheld Garmin GPSMAP 60CSx. GPS readings were collected in Universal Transverse Mercator (UTM) World Geodetic System (WGS) 84 coordinates system.

The accepted error for the handheld GPS is typically 3 to 5 m in the X and Y coordinates. There appeared to still be issues with the Z coordinates in the database relative to the GPS. However, the Z coordinates calculated by the handheld GPS were generally degraded by about three times the X and Y accuracy.

12.2.3 2012

A validation of the Newcore 2012 RC drill collars was conducted during the 2014 site visit. Twenty-one collars, representing 2% of the drilling completed on the Project at the time, were surveyed using a handheld Garmin GPSMAP 62. GPS readings were collected in Universal Transverse Mercator (UTM).

The accepted error for the handheld GPS is typically 3 to 5 m in the X and Y coordinates. Three collars were outside the customary error range. There appeared to be issues with the Z coordinates in the database relative to the GPS. Although the Z coordinates from a handheld GPS tend to have a large error, the elevation of the drill collars did not match the topographic file provided.

12.2.4 2017

A validation of the Newcore 2017 RC drill collars was conducted during the 2017 site visits. Locations were confirmed through verification with adjacent drillholes. While no GPS readings were conducted, 24 of the 28 collars were in close proximity to prior drill collars, which remained clearly marked in the field. Likewise, the 2017 drill collars were marked by concrete monuments. A total of 26 of the 28 drill collars were inspected during the site visits.

12.2.5 2020 – 2021

A validation of the drill collars for the RC drilling completed in 2018, 2020 and the first half of 2021 was conducted during the 2020 and 2021 site visits. Forty-two collars were surveyed using a handheld Garmin Etrex-10. GPS readings were collected in UTM WGS 84 coordinate system.

12.2.6 H2 2021 – 2022

A validation of RC and DDH collars for drilling completed in the second half of 2021 and in 2022 was conducted during the 2022 site visit. Locations were confirmed through GPS checks and included eight holes on the five deposit areas. Coordinates in UTM WGS84 zone 30N were recorded for eight drill collars using a GARMIN GPSMAP 64x handheld GPS. (Figure 12.2.1). Table 12.2.1 contains the results of the collar checks.

Figure 12.2.1 H2 2021 – 2022 Collar Validation



Source: Meadows Smith, 2022

Table 12.2.1 H2 2021 – 2022 Collar Validation

Prospect	Hole	QP Visit			Database		
		East m	North m	Dip °	East m	North m	Dip °
Boin	KBRC275	520,242	635,366	-55	520,245	635,365	-55
Boin	KBDD064	519,240	634,233	-57	519,239	634,235	-60
Boin	KBRC271	517,380	631,564	-50	517,381	631,566	-50
Sewum	SWRC164	520,403	626,142	-60	520,402	626,147	-60
Sewum	SWRC107	521,408	627,916	-50	521,408	627,918	-50
Tokosea	TORC045	523,739	630,276	-50	523,739	630,276	-50
Nyam	NBRC024	530,262	637,266	-60	530,262	637,265	-60
Kwakyekrom	KKRC080	528,824	635,094	-60	528,825	635,093	-60

Prospect	Hole	Difference		
		East	North	Dip
Boin	KBRC275	-3	1	0
Boin	KBDD064	1	-2	3
Boin	KBRC271	-1	-2	0
Sewum	SWRC164	1	-5	0
Sewum	SWRC107	0	-2	0
Tokosea	TORC045	0	0	0
Nyam	NBRC024	0	1	0
Kwakyekrom	KKRC080	-1	1	0

12.2.7 H2 2021 – 2023

A validation of the drill collars for diamond drilling completed from 2021 to 2023 was conducted in November 2023. Locations were confirmed through GPS checks and included all five holes drilled at Nyam. Coordinates in UTM WGS84 zone 30N were recorded using a GARMIN GPSMAP 64x handheld GPS (Figure 12.2.2).

Figure 12.2.2 2023 Collar Validation



Source: Amanor, 2023

12.3 DRA Review (2025)

12.3.1 Site Visit Summary

A DRA representative visited the Project independently from 23-25 September 2025 (Schadrac Ibrango PhD, P.Geo.). The main objectives of the visit were to hold technical discussions with Newcore personnel, understand the nature of the alteration and mineralization with respect to the host rocks (i.e., core review and outcrop / trench exposures), review current interpretations and modelling methodologies, and address all geological functions, including:

- Drilling, logging and sampling procedures (Figure 12.3.1).
- Data collection, treatment and storage (Figure 12.3.1, Figure 12.3.4, Figure 12.3.5).
- Analytical procedures (including QA/QC).
- Core/sample chain of custody and storage processes.
- Inspection of the general site layout, office and core shack facilities to ensure all protocols aligned with industry-best practices (Figure 12.3.5).
- The independent sampling campaign focused on a variety of locations and drill campaigns (Figure 12.3.1), as well as Verification of available drill collar locations (Figure 12.3.2 and Figure 12.3.3).

Figure 12.3.1 Core Review and Independent Sampling



Source: DRA, 2026

Figure 12.3.2 Collar Verification



Source: DRA, 2026

Figure 12.3.3 Collar Verification 2



Source: DRA, 2026

Figure 12.3.4 Trench Sampling Methodology Review



Source: DRA, 2026

Figure 12.3.5 Core Shack Site Inspection



Source: DRA, 2026

12.3.2 Collar Validation

A validation of the Newcore drill collars was conducted during the 2025 site visit. Over 70 collars were surveyed using a handheld Garmin64. GPS readings were collected in Universal Transverse Mercator (UTM) World Geodetic System (WGS) 84 coordinate system.

The average distance from the recorded collar was 3.79 m (see compared distances in Table 12.3.1) which corresponds well with the precision of a handheld GPS unit. One collar (SWDD006) was recorded as off by ~19 m, requiring additional follow-up.

Table 12.3.1 Collar Verification Surveys 2025 Site Visit

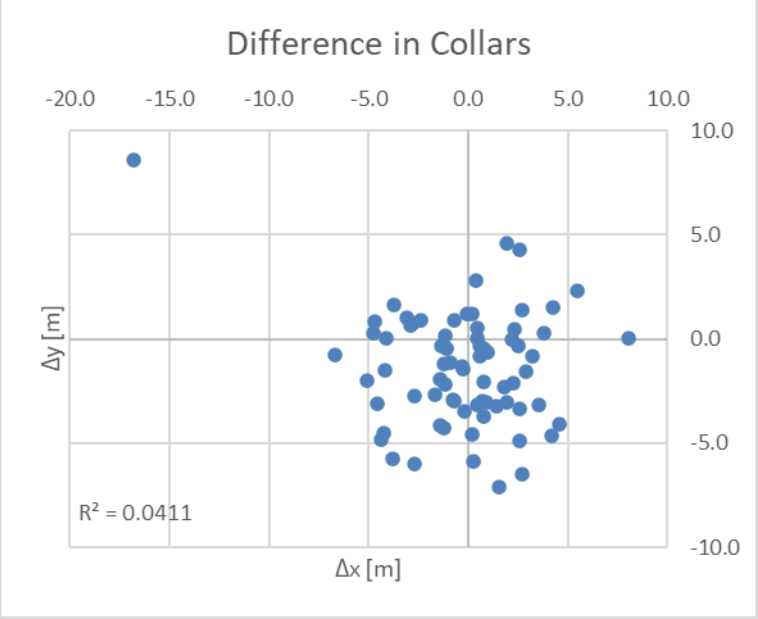
Hole ID	Site Visit		Newcore Database		Calculations		
	X	Y	DB_X	DB_Y	Δx	Δy	Distance
KBRC275	520,241.9	635,366.0	520,245.0	635,365.0	-3.1	1.0	3.2
KBRC276	520,217.8	635,374.9	520,217.0	635,377.0	0.8	-2.1	2.2
KBDD072	520,201.7	635,381.0	520,201.0	635,384.0	0.7	-3.0	3.1
KBDD071	520,260.2	635,349.1	520,261.0	635,352.0	-0.8	-2.9	3.0
KBRC250	520,275.6	635,340.1	520,273.0	635,345.0	2.6	-4.9	5.5
KBRC249	520,308.7	635,325.7	520,309.0	635,327.0	-0.3	-1.3	1.4
KBRC234	520,037.0	635,079.0	520,035.0	635,082.0	2.0	-3.0	3.6
KBRC333	520,060.2	635,069.3	520,064.0	635,075.0	-3.8	-5.8	6.9
KBRC354	519,781.6	634,975.6	519,779.0	634,979.0	2.6	-3.4	4.2
KBRC327	519,752.6	634,950.9	519,754.0	634,955.0	-1.4	-4.1	4.4
KBRC351	519,684.6	634,860.2	519,689.0	634,865.0	-4.4	-4.8	6.5
KBRC326	519,707.3	634,853.0	519,708.0	634,856.0	-0.7	-3.0	3.0
KBRC349	519,693.6	634,832.9	519,689.0	634,837.0	4.6	-4.1	6.2
KBRC350	519,657.8	634,845.7	519,656.0	634,848.0	1.8	-2.3	2.9
KBRC352	519,628.8	634,864.8	519,630.0	634,866.0	-1.2	-1.2	1.7
KBRC164	519,647.8	634,817.7	519,649.0	634,822.0	-1.2	-4.3	4.4
KBRC348	519,660.7	634,781.5	519,658.0	634,788.0	2.7	-6.5	7.1
KBDD064	519,239.9	634,232.0	519,239.0	634,235.0	0.9	-3.0	3.2
KBRC308	519,202.3	634,209.3	519,205.0	634,212.0	-2.7	-2.7	3.8
KBRC307A	519,250.6	634,198.0	519,252.0	634,200.0	-1.4	-2.0	2.4
KBRC306	519,193.5	634,291.9	519,192.0	634,299.0	1.5	-7.1	7.3
KBRC342	518,885.2	633,665.1	518,885.0	633,671.0	0.2	-5.9	5.9
SWRC238	521,373.5	628,710.7	521,371.0	628,711.0	2.5	-0.3	2.5
SWRC239	521,348.5	628,722.8	521,348.0	628,726.0	0.5	-3.2	3.2
SWRC240	521,328.8	628,736.3	521,328.0	628,740.0	0.8	-3.7	3.8
SWRC241	521,311.4	628,747.8	521,311.0	628,745.0	0.4	2.8	2.8
SWRC172	521,290.3	628,764.0	521,293.0	628,770.0	-2.7	-6.0	6.5
SWRC179	521,445.5	628,797.9	521,450.0	628,801.0	-4.5	-3.1	5.5
SWRC177	521,445.9	628,755.2	521,446.0	628,754.0	-0.1	1.2	1.2
SWRC178	521,484.2	628,875.0	521,482.0	628,875.0	2.2	0.0	2.2

Hole ID	Site Visit		Newcore Database		Calculations		
	X	Y	DB_X	DB_Y	Δx	Δy	Distance
SWRC263	521,435.4	629,313.0	521,435.0	629,313.0	0.4	0.0	0.4
SWRC262	521,459.5	629,301.3	521,454.0	629,299.0	5.5	2.3	5.9
SWRC261	521,480.6	629,287.2	521,480.0	629,288.0	0.6	-0.8	1.0
SWRC260	521,502.9	629,274.8	521,504.0	629,277.0	-1.1	-2.2	2.5
SWRC258	521,549.2	629,247.4	521,549.0	629,252.0	0.2	-4.6	4.6
SWRC105	521,417.6	628,030.3	521,415.0	628,026.0	2.6	4.3	5.0
SWDD084	521,405.9	627,918.4	521,403.0	627,920.0	2.9	-1.6	3.3
SWRC107	521,407.1	627,916.9	521,408.0	627,918.0	-0.9	-1.1	1.4
SWRC212	521,285.8	627,756.5	521,285.0	627,757.0	0.8	-0.5	0.9
SWRC195	521,297.1	627,794.0	521,289.0	627,794.0	8.1	0.0	8.1
SWRC213	521,326.4	627,730.8	521,325.0	627,734.0	1.4	-3.2	3.5
SWRC205	521,300.5	627,679.8	521,297.0	627,683.0	3.5	-3.2	4.8
SWRC204	521,333.2	627,664.1	521,330.0	627,665.0	3.2	-0.9	3.3
SWRC169	520,444.8	626,176.3	520,441.0	626,176.0	3.8	0.3	3.8
SWDD006	520,324.2	626,094.4	520,341.0	626,085.7	-16.8	8.6	18.9
SWRC103	520,352.3	626,129.5	520,348.0	626,128.0	4.3	1.5	4.5
SWRC102	520,334.3	626,142.4	520,332.0	626,142.0	2.3	0.4	2.4
SWRC163	520,380.7	626,101.6	520,381.0	626,103.0	-0.3	-1.4	1.5
KKRC085A	528,994.9	635,361.0	529,000.0	635,363.0	-5.1	-2.0	5.5
KKRC093	528,699.2	634,873.4	528,695.0	634,878.0	4.2	-4.6	6.2
KKRC037	528,656.3	634,846.8	528,661.0	634,846.0	-4.7	0.8	4.7
KKRC092	528,626.3	634,834.2	528,633.0	634,835.0	-6.7	-0.8	6.7
KKRC017A	528,693.8	634,828.5	528,698.0	634,833.0	-4.2	-4.5	6.2
KKRC100	528,747.8	634,842.6	528,748.0	634,846.0	-0.2	-3.4	3.4
KKRC097	528,775.4	634,869.3	528,777.0	634,872.0	-1.7	-2.7	3.1
KKRC021	528,587.9	634,693.0	528,592.0	634,693.0	-4.1	0.0	4.1
KKRC029	528,570.3	634,700.7	528,574.0	634,699.0	-3.7	1.7	4.1
KKRC032	528,568.9	634,722.5	528,570.0	634,723.0	-1.1	-0.5	1.2
KKRC025	528,500.7	634,579.7	528,502.0	634,580.0	-1.3	-0.3	1.4
KKRC034	528,517.6	634,606.7	528,517.0	634,607.0	0.6	-0.3	0.7
KKRC066	529,454.2	636,214.3	529,459.0	636,214.0	-4.8	0.3	4.8
NBRC024	530,260.9	637,265.1	530,262.0	637,265.0	-1.1	0.1	1.2
NBRC025	530,281.1	637,253.7	530,284.0	637,253.0	-2.9	0.7	2.9
NBRC031	530,299.7	637,243.4	530,297.0	637,242.0	2.7	1.4	3.1
NBDD048	530,343.3	637,214.9	530,341.0	637,217.0	2.3	-2.1	3.1

Hole ID	Site Visit		Newcore Database		Calculations		
	X	Y	DB_X	DB_Y	Δx	Δy	Distance
NBRC111	530,255.8	637,323.5	530,260.0	637,325.0	-4.2	-1.5	4.4
NBDD071	530,583.5	637,742.5	530,583.0	637,742.0	0.5	0.5	0.7
NBRC105	530,738.9	638,216.6	530,737.0	638,212.0	1.9	4.6	5.0
NBRC085	530,573.0	637,960.4	530,572.0	637,961.0	1.0	-0.6	1.1
NBRC086	530,594.2	637,974.8	530,597.0	637,974.0	-2.8	0.8	2.9
NBRC077	530,487.3	637,722.9	530,488.0	637,722.0	-0.7	0.9	1.1
NBRC075	530,404.2	637,630.2	530,404.0	637,629.0	0.2	1.2	1.2
NBRC022	530,432.6	637,610.9	530,435.0	637,610.0	-2.4	0.9	2.5

There doesn't appear to be any bias in the direction of measured collar locations and recorded collar locations, as evidenced by Figure 12.3.6; the offset appears random around the collar location and there is no clear correlation between them. Ignoring the outlier, the distribution is generally as expected as the sum of the shifts should gravitate to zero and be randomly distributed.

Figure 12.3.6 Horizontal (X,Y Coordinates) Offset in Collar Locations



Source: DRA, 2026

This verification represents 3% of all the diamond drillholes and 6% of all the RC holes completed as of the report date, with a higher proportion focused in the areas of interest for resource estimation.

12.3.3 Analytical Verification

DRA took 70 verification samples (Table 12.3.2) for analysis from drill core selected on site during the site visit (Figure 12.3.1), with a select number of samples from each deposit used in the resource estimation. Twenty-five samples from Boin, 13 from Sewum, 15 from Nyam, and 17 from Kwakyekrom.

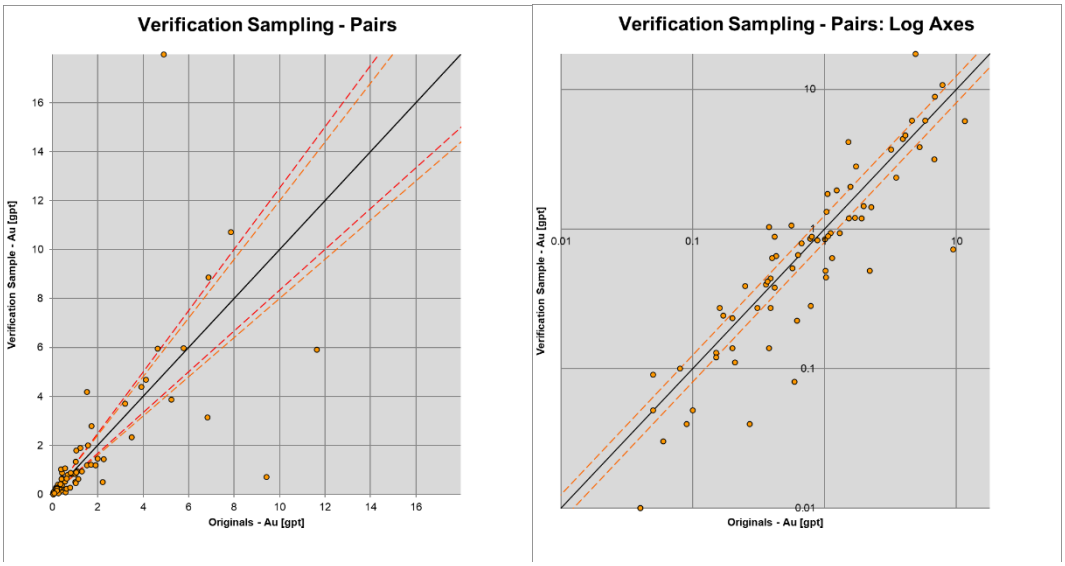
Table 12.3.2 2025 Verification Samples

Project	Hole ID	From m	To m	Check Sample ID	Check Sample Au g/t	Newcore DB Au g/t
Boin	KBDD064	134	135	03112	0.59	0.08
Boin	KBDD064	135	136	03113	0.88	0.83
Boin	KBDD064	136	136.7	03114	1.24	1.89
Boin	KBDD064	136.7	137.4	03115	4.91	17.97
Boin	KBDD064	137.4	138.5	03116	0.38	0.14
Boin	KBDD064	138.5	139.5	03117	0.78	0.85
Boin	KBDD067	184	185.2	03118	0.09	0.04
Boin	KBDD067	185.2	186	03119	3.49	2.33
Boin	KBDD067	186	186.7	03120	7.85	10.72
Boin	KBDD067	186.7	188	03121	5.25	3.86
Boin	KBDD067	188	189	03122	0.39	0.27
Boin	KBDD067	250	251	03123	2.21	0.5
Boin	KBDD067	251	252	03124	6.87	8.87
Boin	KBDD067	252	253	03125	0.39	0.44
Boin	KBDD067	253	254.1	03126	0.25	0.39
Boin	KBDD067	254.1	255	03127	0.67	0.79
Boin	KBDD067	255	256	03128	1.3	0.93
Boin	KBDD067	256	257.2	03129	0.31	0.27
Boin	KBDD067	32.1	33.3	03130	0.15	0.13
Boin	KBDD067	33.3	34.5	03131	1.53	1.19
Boin	KBDD067	34.5	35.7	03132	1.52	4.19
Boin	KBDD067	35.7	37	03133	0.42	0.88
Boin	KBDD067	37	38	03134	1.58	2
Boin	KBDD067	38	39	03135	0.36	0.4
Boin	KBDD067	39	40	03136	0.05	0.05
Nyam	NBDD052	321	322	03137	0.16	0.27
Nyam	NBDD052	322	323	03138	5.79	5.98

Project	Hole ID	From m	To m	Check Sample ID	Check Sample Au g/t	Newcore DB Au g/t
Nyam	NBDD052	323	324	03139	4.63	5.95
Nyam	NBDD052	326	327	03142	1.71	1.2
Nyam	NBDD052	327	328	03143	3.93	4.4
Nyam	NBDD052	328	329	03144	2.26	1.43
Nyam	NBDD052	329	330	03145	11.64	5.9
Nyam	NBDD052	330	331	03146	1.01	0.84
Nyam	NBDD057	108	109	02460	0.05	0.09
Nyam	NBDD057	109	110	02461	1.04	1.33
Nyam	NBDD057	110	111	02462	0.8	0.88
Nyam	NBDD057	111	112	02463	1.11	0.93
Nyam	NBDD057	112	113	02464	0.21	0.11
Nyam	NBDD057	113	114	02465	0.43	0.64
Nyam	NBDD057	114	115	02466	0.42	0.38
Kwakyekrom	KKDD001	33	34	02467	0.62	0.22
Kwakyekrom	KKDD001	34	35	02468	6.83	3.14
Kwakyekrom	KKDD001	35	36	02469	3.21	3.71
Kwakyekrom	KKDD001	36	37	02470	9.43	0.71
Kwakyekrom	KKDD001	37	38	02471	4.12	4.68
Kwakyekrom	KKDD001	38	39	02472	1.73	2.79
Kwakyekrom	KKDD001	39	40	02473	1.06	0.89
Kwakyekrom	KKDD002	181	182	02474	0.57	0.52
Kwakyekrom	KKDD002	182	183	02475	1.91	1.19
Kwakyekrom	KKDD002	183	184	02476	0.06	0.03
Kwakyekrom	KKDD002	184	185	02477	0.08	0.1
Kwakyekrom	KKDD003	187	188	02478	0.04	0.01
Kwakyekrom	KKDD003	188	189	02479	0.63	0.65
Kwakyekrom	KKDD003	189	190	02480	0.4	0.62
Kwakyekrom	KKDD003	190	191	02481	0.17	0.24
Kwakyekrom	KKDD003	191	192	02482	0.2	0.23
Kwakyekrom	KKDD003	192	193	02483	0.15	0.12
Sewum	SWDD083	198.6	199.2	02484	0.04	0.01
Sewum	SWDD083	199.2	200.6	02485	1.02	0.5
Sewum	SWDD083	200.6	201.4	02486	0.37	0.42
Sewum	SWDD083	201.4	202.2	02487	0.38	1.03

Project	Hole ID	From m	To m	Check Sample ID	Check Sample Au g/t	Newcore DB Au g/t
Sewum	SWDD083	202.2	202.8	02488	1.05	1.78
Sewum	SWDD083	202.8	204	02489	0.27	0.04
Sewum	SWDD073	158.2	159	03147	0.2	0.14
Sewum	SWDD073	159	159.9	03148	0.56	1.06
Sewum	SWDD073	159.9	160.8	03149	1.14	0.62
Sewum	SWDD073	160.8	162	03150	0.79	0.28
Sewum	SWDD073	162	163	02451	1.99	1.46
Sewum	SWDD073	163	164.1	02452	1.03	0.45
Sewum	SWDD073	164.1	165	02453	0.1	0.05

Figure 12.3.7 Verification Sampling – Duplicate Plots



*Verification Control Lines are ±20%

There was a higher spread in the data than anticipated from the drill core samples, with a larger proportion of samples outside the 20% difference line than expected. Nyam and Boin had better control than Sewum and Kwakyekrom. The cause of this could be heterogeneities in weathering and oxidations, nugget effect, and core box handling of historic core. Regardless, the null hypothesis cannot be rejected for t-tests performed both globally and on each subset; as such, it is assumed there are equal means with no clear sign-based bias. Currently, verification sampling is satisfactory, but additional sampling is recommended to better understand higher variations, continued review of lab precision, and modeling of the nugget effect. Independent sampling validates the continued use of the dataset and affirms moderate nugget value from variography modelling.

12.4 QP's Opinion

Considering the results of the QP independent check assay program and bearing in mind the variable nature of the mineralization (i.e. nugget effect), the QP is of the opinion that the supplied data is of sufficient quality to support Mineral Resource Estimation. It is recommended that additional sampling be completed to better understand the elevated variability as part of the next phase of project development and prior to upgrading the Inferred and Indicated Resources into higher categories.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

Newcore has conducted a number of metallurgical testwork programs since 2012 at SGS Lab (SGS), the University of Mines and Technology in Tarkwa (UMaT), Jet-Rom Engineering Ltd. (Jet-Rom), Odeleb Ltd., Intertek Minerals Ltd., as well as Intertek Base Met Labs (Intertek). The testwork was conducted on diamond drill hole (DDH), reverse circulation (RC), composite and trench samples, and included ore representing a range of weathering profiles from the various deposits.

The sections to follow summarize available historical and recent testwork results, with a focus on the data relevant to supporting the PFS design.

13.2 Summary of Testwork Programs

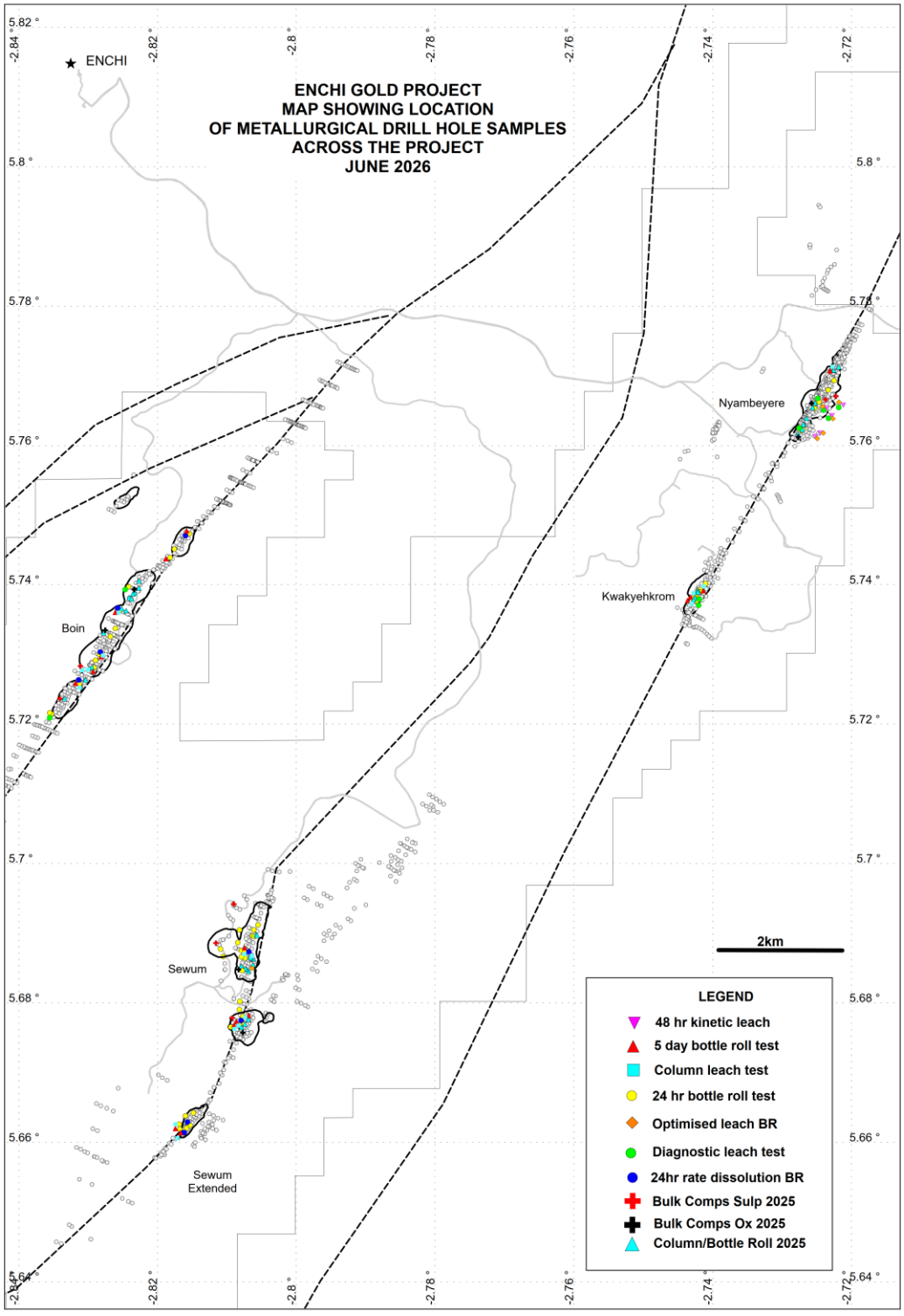
A summary of the testwork programs conducted from 2012 to 2026 is shown in Table 13.2.1. Figure 13.2.1 to Figure 13.2.5 illustrate the sampling locations for all the metallurgical testwork.

Table 13.2.1 Summary Table of Testwork Programs

Program	Size Dist. Analysis	Cyanide Assays	Mineralogy	Flotation-Leach Tests	Bottle Rolls (BR)	Gravity Gold Recovery	24-hr Dissolution BR	Diagnostic Leach BR	48-Hr Kinetic Leach	Optimized Leach BR	Preg-Rob Index (PRI)	5-10 Day BR	Column Tests	Agglomeration	Percolation	Pilot Heap	Comminution	Solid-Liquid
SGS 2012		✓			✓													
Intertek 2020-2021	✓				✓		✓	✓				✓	✓					
Intertek 2022	✓											✓	✓					
UMaT 2022-2023												✓	✓					
Intertek 2023	✓				✓	✓		✓	✓	✓			✓					
UMaT 2023																✓		
Jet-Rom 2023																	✓	
Odeleb Ltd. 2023																	✓	
Odeleb Ltd. 2024												✓					✓	
Intertek 2024					✓			✓					✓					
UMaT 2024												✓	✓					
Intertek 2025-2026*			✓	✓	✓	✓		✓		✓	✓		✓	✓	✓		✓	✓

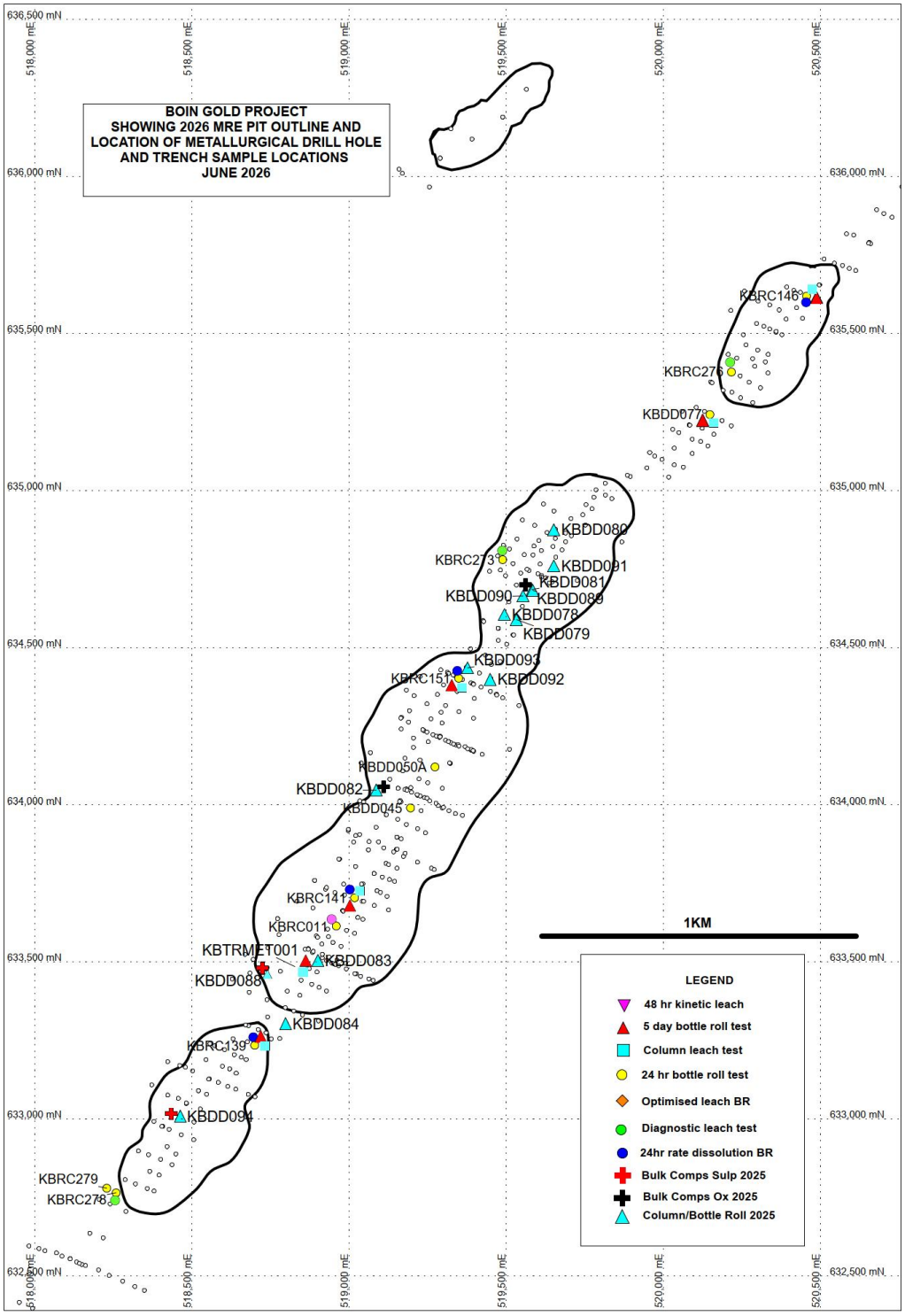
* Denotes the more recent testwork conduct with both Intertek in Ghana and Intertek Base Met Labs in Canada.

Figure 13.2.1 Map of Metallurgical Sampling Locations



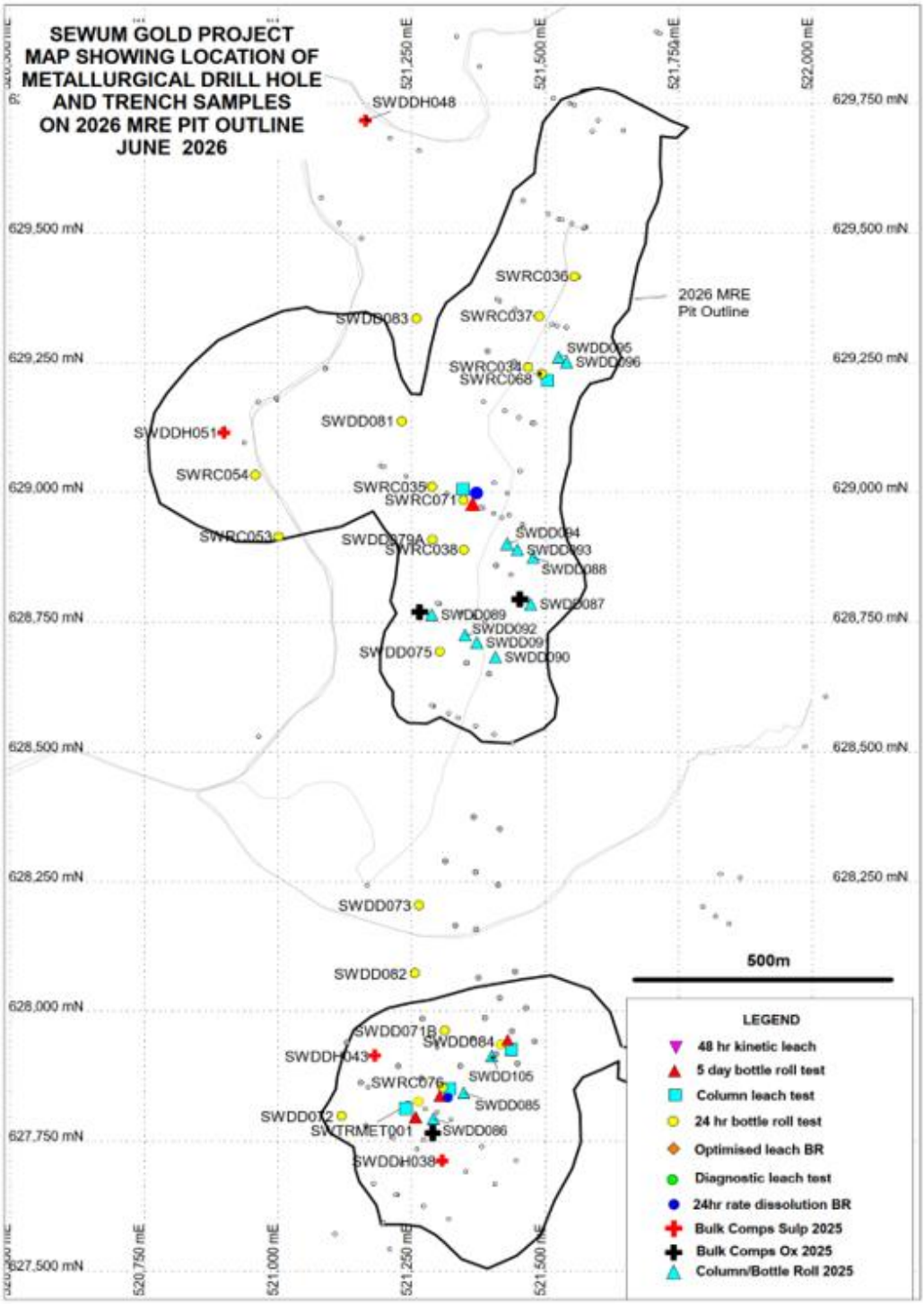
Source: Newcore, 2026

Figure 13.2.2 Boin Metallurgical Sampling Locations



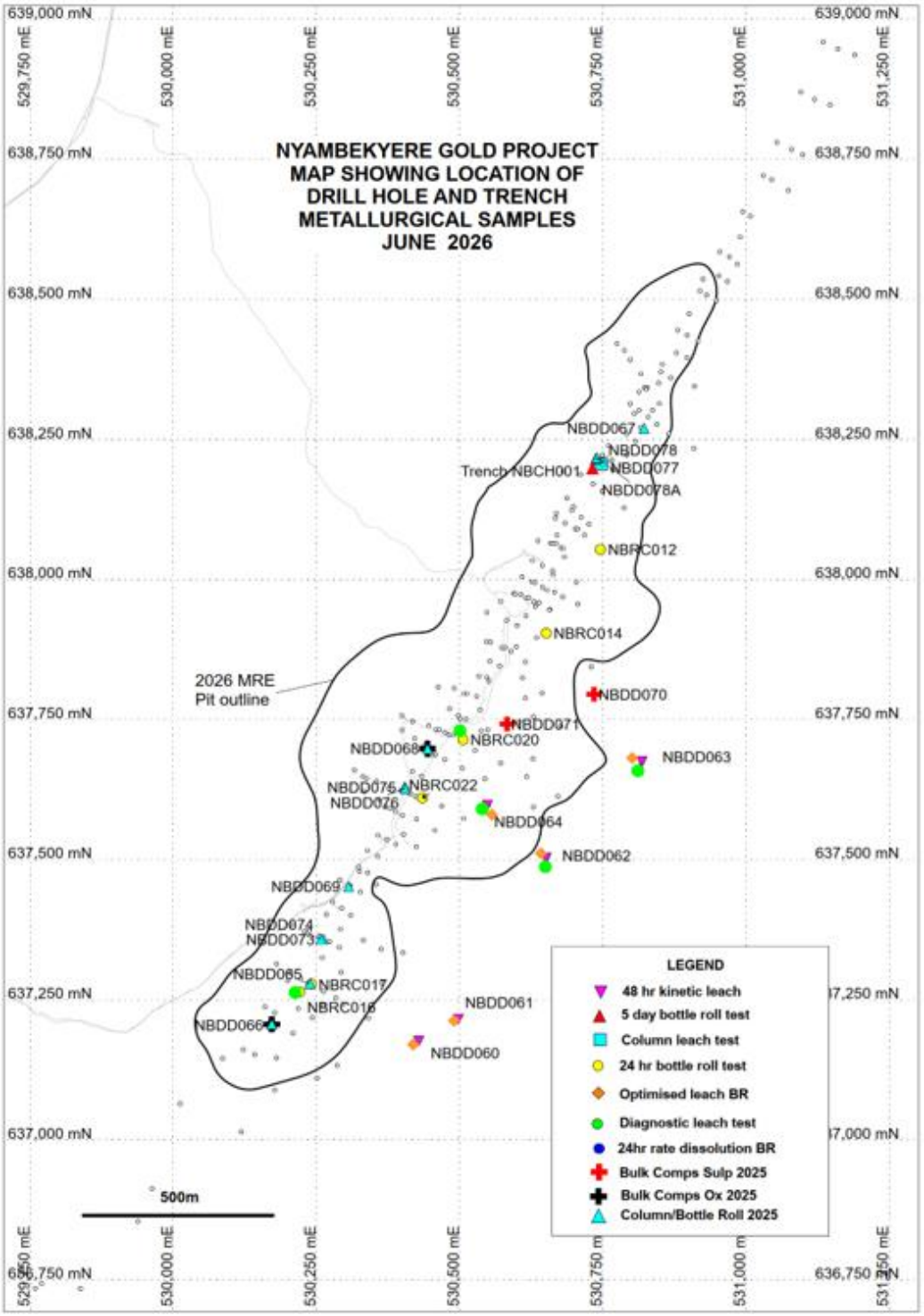
Source: Newcore, 2026

Figure 13.2.3 Sewum Metallurgical Sampling Locations



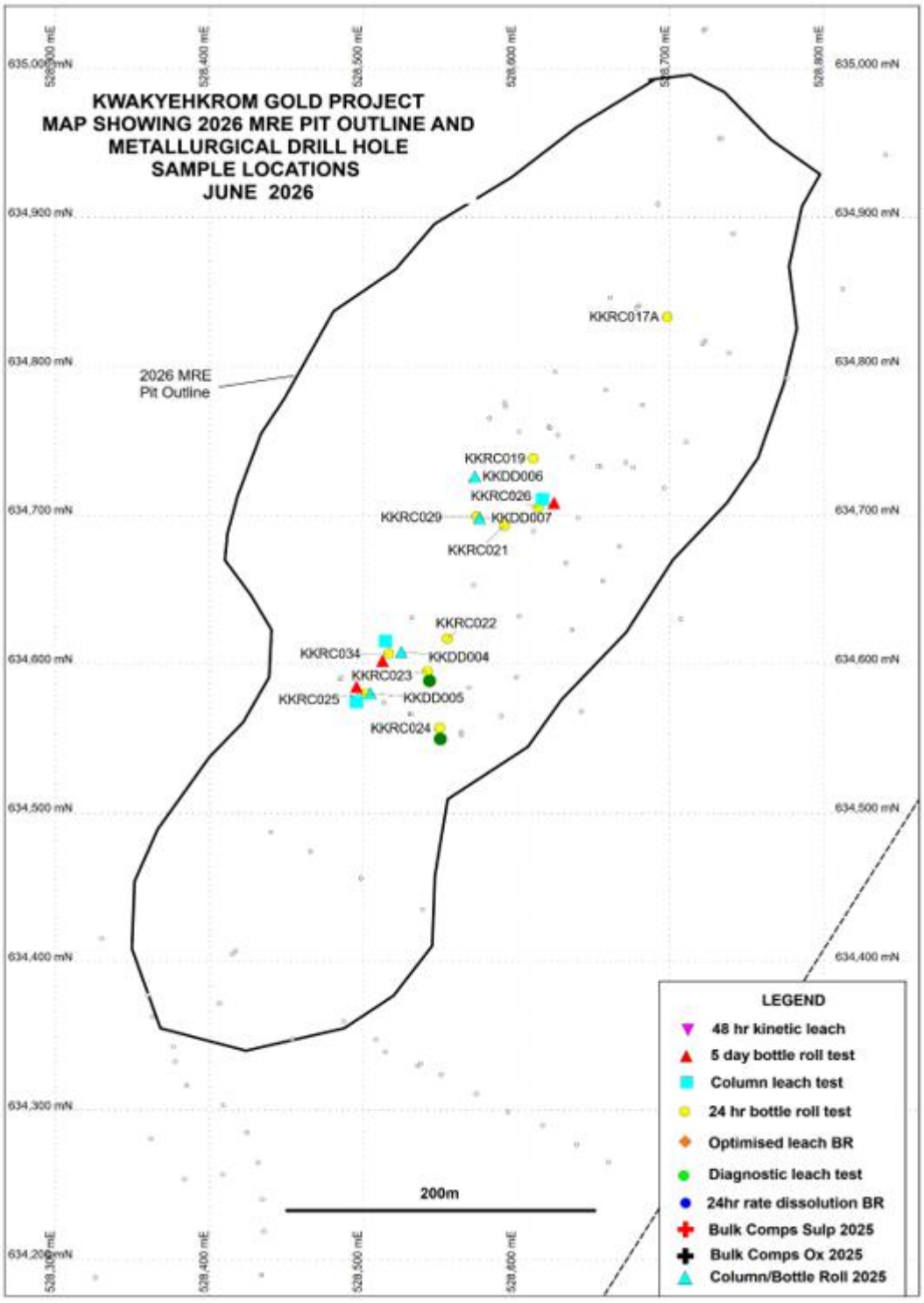
Source: Newcore, 2026

Figure 13.2.4 Nyam Metallurgical Sampling Locations



Source: Newcore, 2026

Figure 13.2.5 Kwakyekrom Metallurgical Sampling Locations



Source: Newcore, 2026

13.3 Summary of Historical Testwork Programs (2012 to 2024)

A summary of the historical metallurgical and comminution testwork per zone is shown in Table 13.3.1 and Table 13.3.2, respectively.

Table 13.3.1 Summary of Historical Metallurgical Tests from 2012 to 2024 by Zone

Metallurgical Test	Boin	Sewum	Nyam	Kwakyekrom	Tokosea	Total
Cyanide Assays	21					21
Bottle Rolls	74	50	30	45	4	203
Gravity Gold Recovery	2	6				8
Size Distribution Analyses	7	12		1		20
24-hr Dissolution Bottle Roll	5	5				10
Diagnostic Leach Bottle Roll	7	5	6	4		22
48-hr Kinetic Leach Bottle Roll			27			27
24-48h Kinetic Leach Bottle Roll		45				45
Optimized leach Bottle Roll			14			14
5-10 Day Bottle Roll	7.5	13.5	2	1		24
30-90 Day Column Tests	9	14	3	1		27
Pilot Heap	1	1				2
Total	**132.5**	**152.5**	**82**	**52**	**4**	**423**

Decimal values denote partial samples for that test

Table 13.3.2 Summary of Historical Comminution Tests from 2023 to 2024 by Zone

Comminution Test	Sewum	Boin	Nyam	Crush Rock	Total
BWi	24	4	7	0	35
UCS	28	8	4	0	40
Corr. Strength	0	9	11	0	20
Point Load	28	17	15	0	60
DWi	0	1	1	0	2
SMC	1	2	2	0	5
CWi	1	2	2	3	8
Total	**82**	**43**	**42**	**3**	**170**

The drill holes / trenches where comminution samples were sourced from are as follows:

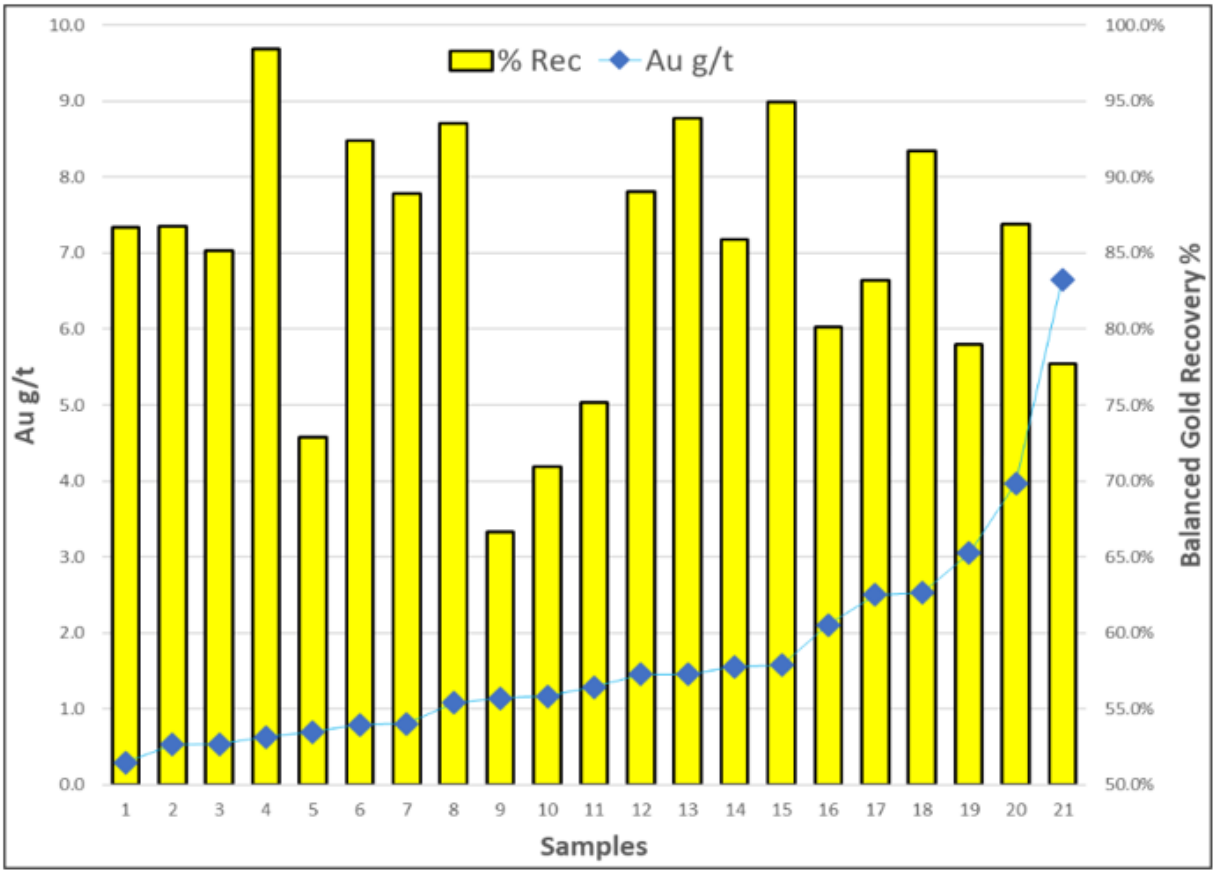
- Oxide samples from trenches SWMETTR01 (3 samples) and BNMETTR01 (2 samples).

- Nyam samples from NBDD060, NBDD061, NBDD063 and NBDD064.
- Sewum fresh samples from SWDD071B, SWDD072, SWDD073, SWDD075, SWDD076, SWDD078, SWDD079A, SWDD081, SWDD082 and SWDD083.
- Boin fresh samples from KBDD061, and KBDD073.

13.3.1 Cyanide Solution Assays

Early metallurgical testing for the Enchi Gold Project consisted of cyanide soluble analyses conducted at SGS in 2012 on 21 samples from drill hole KBRC011 drilled at the Boin deposit. The results are presented in Figure 13.3.1. No strong correlation was established from the results.

Figure 13.3.1 Boin Cyanide Assay Gold Grade vs. Gold Recovery



Source: Newcore, 2024

13.3.2 Head Analysis

In 2023, Intertek completed multi-element ICP analysis of 27 Nyam fresh composites from drill holes NBDD060 to NBDD064, and five oxide trench samples Boin drill holes BNMETTR01, BNMETTR02, and Sewum drill holes SWMETTR01, SWMETTR02 and SWMETTR03. The results indicated that the fresh samples had an average gold grade of 1.7 g/t Au (range 0.5 to 9.6 g/t Au) and negligible silver content. Base metal concentrations (Cu, Pb, and Zn) were generally low in both fresh and oxide samples, with no significant correlation to gold grade. Fresh mineralized samples contained notably higher concentrations of mobile elements (Ca, Mg, Na, and S) and moderately higher concentrations of Co, Mn, Ni, and Sr compared to the oxide samples, while most other elements showed similar concentrations across both sample types.

13.3.3 Bottle Rolls

Bottle rolls testwork were conducted at SGS in 2012, Intertek in 2020 to 2021, and 2022 to 2023. The results are summarized in Table 13.3.3. Note that the 5 day and 10 day bottle rolls are not included in this table.

Table 13.3.3 Summary of Standard Bottle Rolls (2012 – 2023)

Hole Id	Type	Zone	Mat. Type	Metallurgical Test	# Samples	Au g/t	Avg Rec	Lab	Year
KBDD045	DDH	Boin	Ox	Bottle Roll	2	1.96	92.2%	SGS	2012
KBDD050A	DDH	Boin	Ox	Bottle Roll	2	1.43	47.6%	SGS	2012
SWRC034	RC	Sewum	Trans	Bottle Roll	1	1.03	66.7%	SGS	2012
SWRC035	RC	Sewum	Fresh	Bottle Roll	1	1.01	65.2%	SGS	2012
SWRC036	RC	Sewum	Fresh	Bottle Roll	1	0.54	12.6%	SGS	2012
SWRC037	RC	Sewum	Fresh	Bottle Roll	1	1.00	28.2%	SGS	2012
SWRC038	RC	Sewum	Trans	Bottle Roll	2	0.81	9.6%	SGS	2012
SWRC053	RC	Sewum	Trans	Bottle Roll	2	1.50	29.1%	SGS	2012
SWRC054	RC	Sewum	Ox	Bottle Roll	2	1.24	20.1%	SGS	2012
NBRC012	RC	Nyam	Trans	Bottle Roll	2	1.76	66.1%	SGS	2012
NBRC014	RC	Nyam	Trans	Bottle Roll	3	1.48	81.2%	SGS	2012
SWRC073	RC	Sewum Ext	Ox	24 hr Bottle Roll	5	0.56	77.8%	Intertek	2020-21
SWRC074	RC	Sewum Ext	Ox	24 hr Bottle Roll	5	1.32	96.9%	Intertek	2020-21
SWRC076	RC	Sewum CH	Trans	24 hr Bottle Roll	10	0.85	95.6%	Intertek	2020-21
SWRC068	RC	Sewum Ridge	Ox/Trans	24 hr Bottle Roll	5	1.03	97.6%	Intertek	2020-21
SWRC071	RC	Sewum Ridge	Ox	24 hr Bottle Roll	5	0.74	86.5%	Intertek	2020-21
KBRC139	RC	Boin S	Ox	24 hr Bottle Roll	5	1.30	77.0%	Intertek	2020-21
KBRC141	RC	Boin C	Ox/Trans	24 hr Bottle Roll	5	1.56	87.6%	Intertek	2020-21
KBRC151	RC	Boin C	Trans	24 hr Bottle Roll	5	1.16	98.2%	Intertek	2020-21
KBRC146	RC	Boin N	Trans	24 hr Bottle Roll	5	1.37	85.5%	Intertek	2020-21

Hole Id	Type	Zone	Mat. Type	Metallurgical Test	# Samples	Au g/t	Avg Rec	Lab	Year
KKRC017A	RC	Kwakyekrom	Trans	24 hr Bottle Roll	3	0.33	74.9%	Intertek	2021
KKRC019	RC	Kwakyekrom	Trans	24 hr Bottle Roll	2	0.47	71.9%	Intertek	2021
KKRC034	RC	Kwakyekrom	Trans	24 hr Bottle Roll	5	1.06	83.1%	Intertek	2021
KKRC029	RC	Kwakyekrom	Ox/Trans	24 hr Bottle Roll	7	1.81	76.9%	Intertek	2021
KKRC026	RC	Kwakyekrom	Ox	24 hr Bottle Roll	3	0.80	86.7%	Intertek	2021
KKRC025	RC	Kwakyekrom	Trans	24 hr Bottle Roll	5	0.66	83.4%	Intertek	2021
KBRC271	RC	Boin	Trans/Fresh	24 hr Bottle Roll	13	1.57	49.7%	Intertek	2021
KBRC272	RC	Boin	Trans/Fresh	24 hr Bottle Roll	2	0.81	39.8%	Intertek	2021
KBRC273	RC	Boin	Trans/Fresh	24 hr Bottle Roll	10	2.06	32.4%	Intertek	2021
KBRC276	RC	Boin	Trans/Fresh	24 hr Bottle Roll	17	1.45	35.3%	Intertek	2021
KBRC278	RC	Boin	Trans/Fresh	24 hr Bottle Roll	7	1.52	37.4%	Intertek	2021
KBRC279	RC	Boin	Fresh	24 hr Bottle Roll	1	1.04	16.4%	Intertek	2021
KKRC024	RC	Kwakyekrom	Trans	24 hr Bottle Roll	2	0.98	27.7%	Intertek	2021
KKRC023	RC	Kwakyekrom	Trans/Fresh	24 hr Bottle Roll	5	2.99	19.6%	Intertek	2021
KKRC022	RC	Kwakyekrom	Trans/Fresh	24 hr Bottle Roll	8	0.81	17.4%	Intertek	2021
KKRC021	RC	Kwakyekrom	Trans/Fresh	24 hr Bottle Roll	5	0.80	7.2%	Intertek	2021
NBRC016	RC	Nyam	Trans/Fresh	24 hr Bottle Roll	5	1.23	54.5%	Intertek	2021
NBRC017	RC	Nyam	Ox/Trans/Fresh	24 hr Bottle Roll	15	1.11	69.4%	Intertek	2021
NBRC020	RC	Nyam	Fresh	24 hr Bottle Roll	4	1.69	54.4%	Intertek	2021
NBRC022	RC	Nyam	Ox	24 hr Bottle Roll	1	2.19	94.1%	Intertek	2021
SWRC107	RC	Sewum Ox	Ox	24 hr Bottle Roll	5	0.77	96.5%	Intertek	2021
SWRC107	RC	Sewum Sulp	Fresh	24 hr Bottle Roll	5	1.46	82.4%	Intertek	2021
TOCH-003	Trench	Tokosea	Ox	24 hr Bottle Roll	4	0.53	88.70%	Intertek	2023
NBDD060 to 064	DDH	Nyam	Fresh	48-hour kinetic leach	27	1.81	82.50%	Intertek	2023
SWDD071B, 072, 073, 082, 075, 079A, 081, 083, 076, 077, 078, 080	DDH	Sewum	Fresh	Optimized	45	0.79	82.80%	Intertek	2024

Fourteen of the 27 tests completed on Nyam fresh samples were performed under optimized conditions which applied finer grind size, lead nitrate addition and oxygen addition. The tests yielded an average gold recovery of 91.7% at an average gold head grade of 2.08 g/t. Those samples with lower recoveries in the original 48-hour bottle rolls increased the most, averaging over 14%, with the higher original recoveries increasing over 5%. The optimized leach result for Nyam fresh are shown in Table 13.3.4.

Table 13.3.4 Optimized Leach Tests for Nyam Fresh Samples (Intertek 2023)

Hole Id	From	To	Type	Zone	Mat. Type	Lith.	Sample ID	Metallurgical Test	Head Au g/t	% Rec
NBDD063	395.2	399.2	DDH	Nyam	W1	SPH/MD	2001237	Optimized Leach	2.09	90.4%
NBDD060	360.0	365.0	DDH	Nyam	W1	MD	2001225	Optimized Leach	2.89	82.7%
NBDD063	496.0	500.0	DDH	Nyam	W1	MD	2001236	Optimized Leach	3.64	91.0%
NBDD063	489.0	496.0	DDH	Nyam	W1	MD	2001240	Optimized Leach	1.29	91.1%
NBDD060	302.5	367.6	DDH	Nyam	W1	SPG/MD	2001226	Optimized Leach	3.63	91.1%
NBDD062	350.5	419.0	DDH	Nyam	W1	MD	2001232	Optimized Leach	0.74	94.4%
NBDD061	278.4	420.4	DDH	Nyam	W1	MD	2001230	Optimized Leach	0.73	97.2%
NBDD063	497.0	499.0	DDH	Nyam	W1	MD	2001235	Optimized Leach	9.34	95.0%
NBDD063	396.1	423.2	DDH	Nyam	W1	MD	2001244	Optimized Leach	0.62	87.1%
NBDD064	223.2	250.2	DDH	Nyam	W1	MD	2001249	Optimized Leach	0.98	93.8%
NBDD063	403.0	508.3	DDH	Nyam	W1	MD	2001241	Optimized Leach	0.91	90.2%
NBDD063	465.8	491.0	DDH	Nyam	W1	MD	2001242	Optimized Leach	0.78	92.3%
NBDD064	209.6	247.3	DDH	Nyam	W1	MD	2001251	Optimized Leach	0.73	89.0%
NBDD063	472.8	487.2	DDH	Nyam	W1	MD	2001243	Optimized Leach	0.78	98.0%

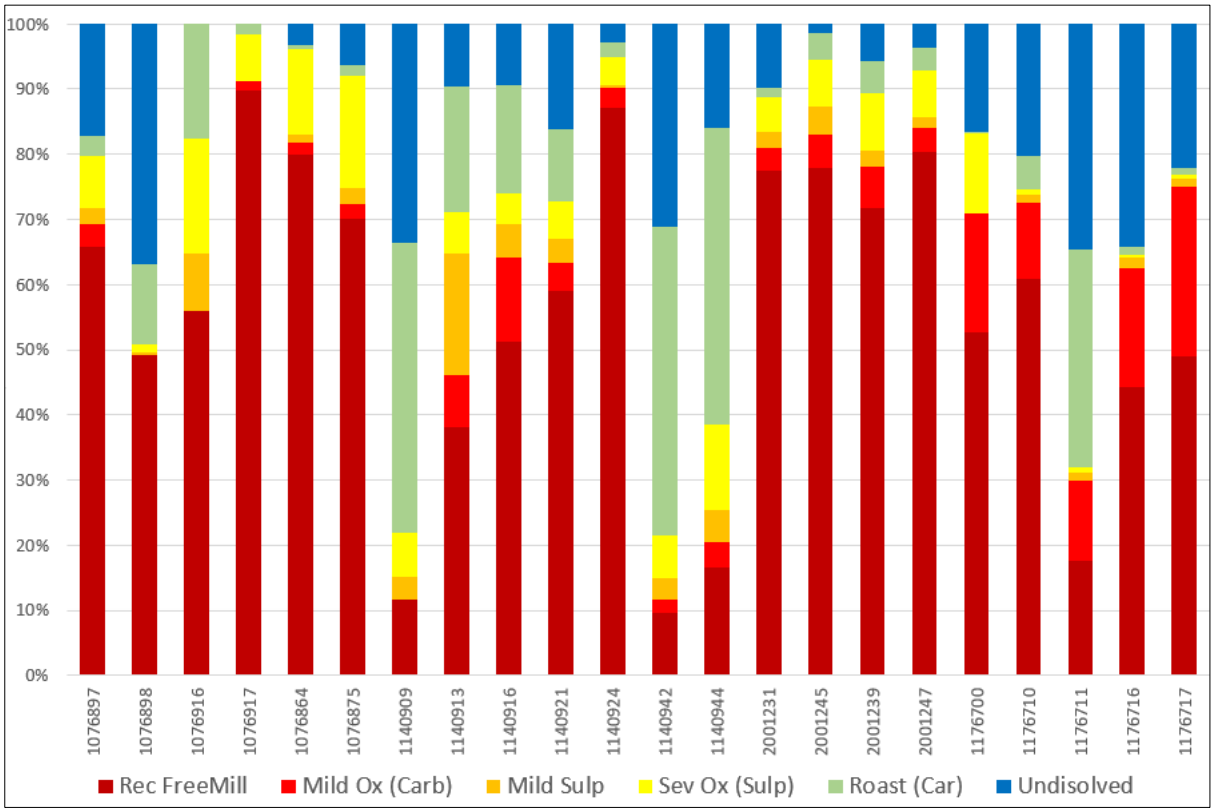
Notes:
LT = Laterite, SAP = Saprolite, MD = Dolerite, QV = Quartz Vein, SPH = Phyllite, SGW = Greywacke, CY = Clay, SZ = Shear Zone

13.3.4 Diagnostic Leach Testwork

A total of 22 diagnostic leach tests were completed on samples, including six from Nyam fresh, two from Kwakyekrom transition, two from Kwakyekrom fresh, seven from Boin fresh, and five from Sewum fresh. The transition and fresh material were sourced from eight RC holes (two from Nyam, two from Kwakyekrom, four from Boin) and 3 DDH holes for Nyam fresh and eight DDH holes for Sewum fresh. The RC samples represent a selected subset of predominantly fresh material from RC holes at Boin, Nyam, and Kwakyekrom. Diagnostic leach analyses were conducted on selected samples that returned moderate to low recoveries in 24-hour bottle roll tests, as part of a larger sample set.

Diagnostic leach results include free-milling gold, gold extracted by a mild oxidative pre-leach, acid-soluble gold, gold extracted via a severe oxidative pre-leach, gold recoverable through complete oxidation (roasting), and undissolved gold remaining in the residue. The results are summarized in Figure 13.3.2.

Figure 13.3.2 Diagnostic Leach – Recovery Gold Distribution



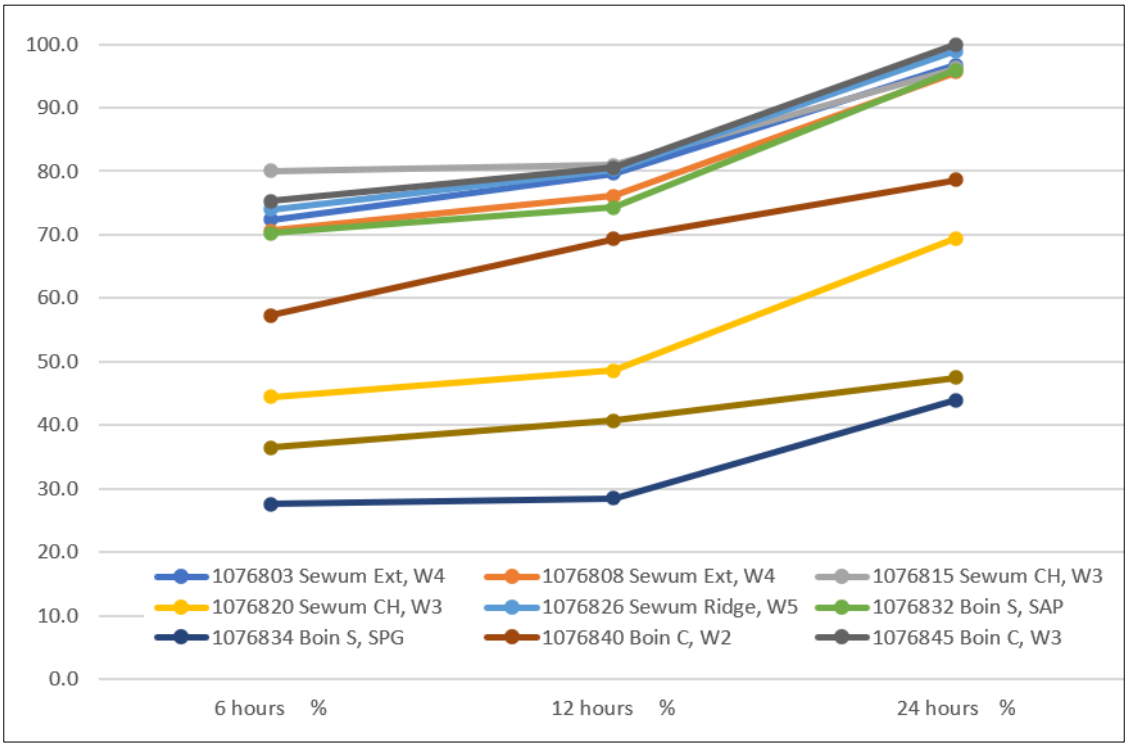
Source: Newcore, 2025

All tests were completed between 2020 and 2024 at Intertek Laboratory in Tarkwa, Ghana. The samples contained an average of 15% undissolved gold, with values of up to 37%. Four samples showed a significant improvement in gold recovery through roasting, with an additional 33 to 47% recovery, whereas the remaining samples averaged only 6% additional recovery. Acid digestion, primarily targeting sulphide-associated gold, contributed an average of 17% additional gold recovery. Across all 22 samples, the average head grade was 1.91 g/t Au, with an overall average gold recovery of 85.0% when all recovery components were combined.

13.3.5 Rate Dissolution Testwork

Rate dissolution tests were completed on 10 samples of oxide and transition material from eight RC holes, four each from Boin and Sewum. The samples were randomly selected from a larger set previously analysed by 24-hour bottle roll tests, representing a range of gold recoveries. Both sets of tests were conducted between 2020 and 2021 at Intertek Laboratory in Tarkwa, Ghana, and the results are generally comparable. Refer to Figure 13.3.3 for more details on the dissolution testwork.

Figure 13.3.3 Gold Dissolution – Recovery vs. Time



Source: Newcore, 2024

Dissolution rates were consistent across all samples, being rapid from 0 to 6 hours, slower from 6 to 12 hours, and moderate from 12 to 24 hours. The rate dissolution samples averaged 1.05 g/t Au, with an overall average recovery of 75.3%. Samples from Sewum showed higher recoveries, averaging 91.4%.

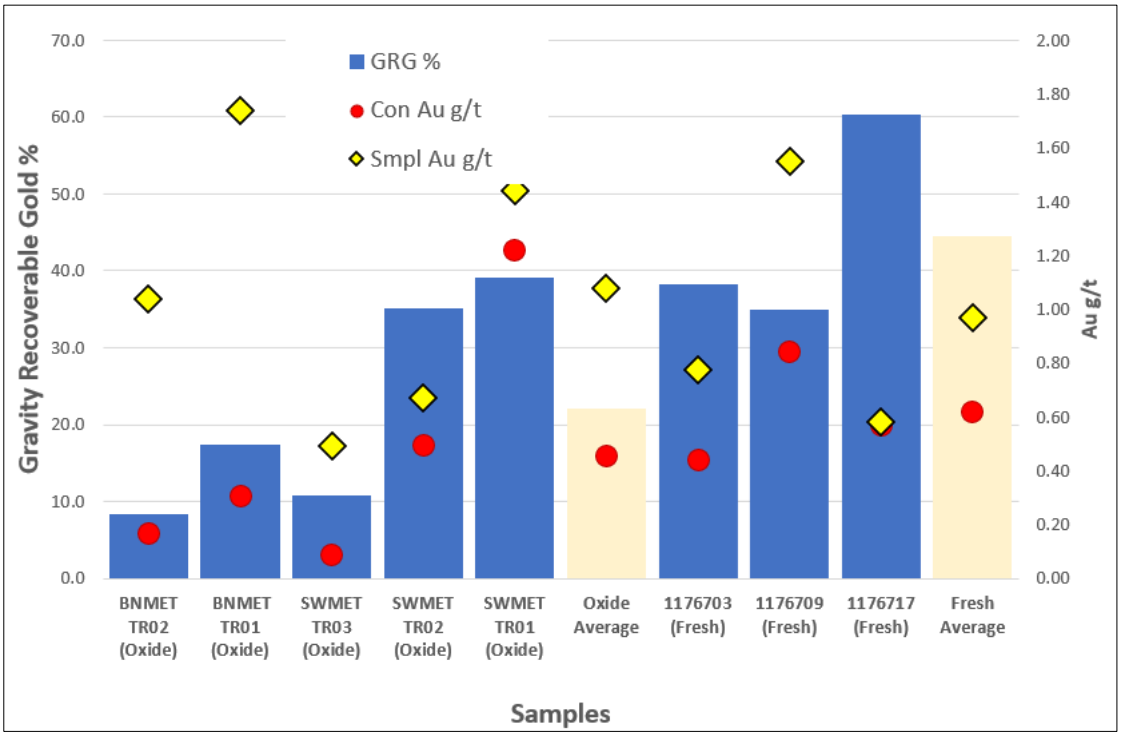
13.3.6 Gravity Recoverable Gold Tests

In 2023, five gravity recoverable gold (GRG) tests were completed on oxide samples, including two from Boin and three from Sewum. The oxide material was sourced from trenches at Boin and Sewum and was subsequently used for bottle roll and column leach tests. The trench samples are composites created from 20 m wide mineralized intervals, with an average gold grade of 1.04 g/t Au, ranging from 0.41 to 1.61 g/t Au. All tests were conducted at Intertek Laboratory in Tarkwa, Ghana. The samples averaged 1.06 g/t Au, with an average gravity recovery of 22.1% and an average concentrate grade of 15.9 g/t Au.

In 2024, three GRG tests were completed on Sewum fresh core samples at Intertek as well. The samples averaged 0.97 g/t Au, with an average gravity recovery of 44.5% and an average concentrate grade of 21.6 g/t Au.

Refer to Figure 13.3.4 for a visual representation of the gravity testwork.

Figure 13.3.4 Gravity Recoverable Gold vs. Gold Grade vs Conc. Gold Grade



Source: Lyco, 2026

13.3.7 Five To Ten – Day Bottle Rolls

Twenty-four 5 to 10-day bottle roll tests were completed on samples submitted for column testing. The material included oxide and transition samples from Boin, Sewum, and Kwakyekrom. All tests were conducted between 2020 and 2024 at UMaT and Intertek Laboratories in Tarkwa, Ghana. The 5 to 10-day bottle roll tests were performed on coarse material identical to that used in the corresponding column leach tests, with the material crushed to a relatively coarse grind.

The samples averaged 0.97 g/t Au, with values ranging from 0.33 to 4.2 g/t Au, and average recovery was 76.9%, ranging from 30.0% to 96.9%. In almost all cases, gold leaching was ongoing at the end of the five- or ten-day period, indicating that ultimate recoveries would likely be higher.

13.3.8 Column and Pilot Heap Testwork

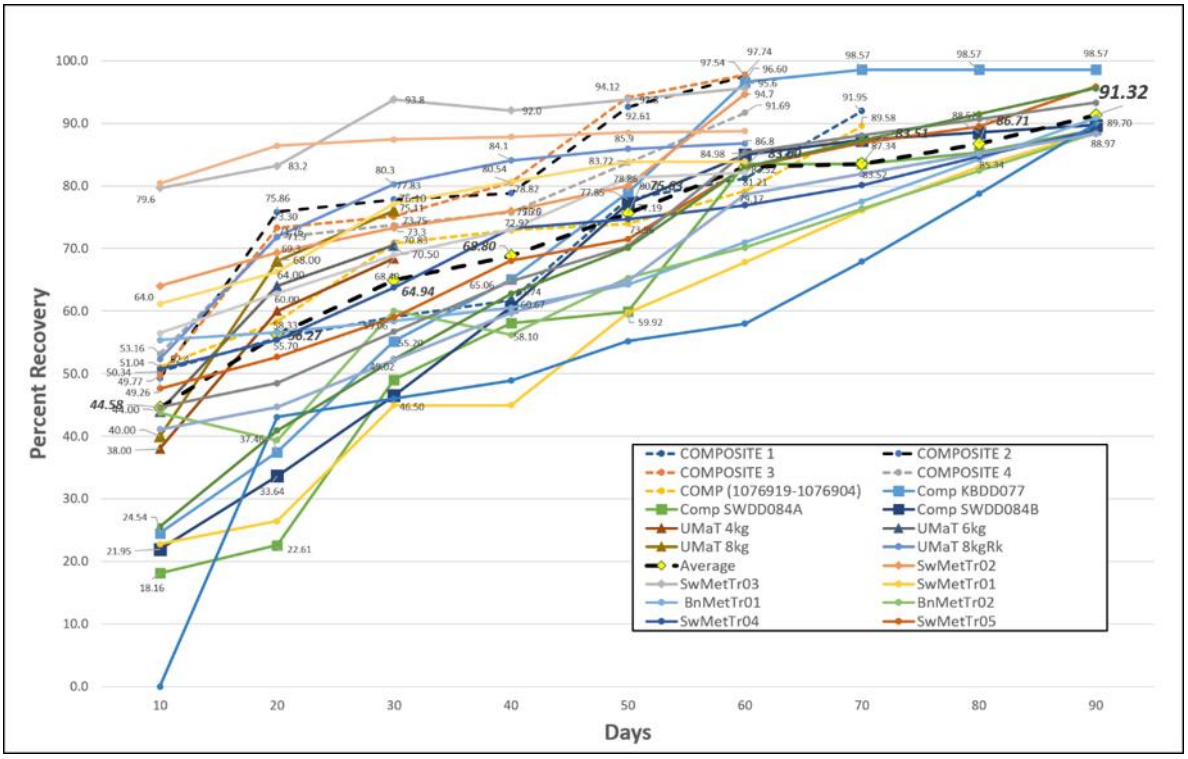
Column and pilot heap tests were conducted by Intertek and UMaT between 2020 and 2024. The sample details and results are summarized in Table 13.3.5 and in Figure 13.3.3 and Figure 13.3.4.

Table 13.3.5 Summary of Column Leach and Pilot Heap Testwork

Hole Id	Type	Zone	Mat. Type	Lith.	Sample ID	Metallurgical Test	Head Au g/t	% Rec	Lab	Year
KBRC141/151	RC	Boin Central	W3/W4	QV/SAP	COMPOSITE 1	70-day Column	1.49	92.0%	Intertek	2021
KBRC139/146	RC	Boin N & S	W2/W3/W4	SAP/SPH	COMPOSITE 2	60-day Column	1.02	97.5%	Intertek	2021
SWRC068/071	RC	Sewum Ridge	W3/W4/W5	QV/LG	COMPOSITE 3	60-day Column	1.11	97.7%	Intertek	2021
SWRC074/076	RC	Sewum CH & Ext.	W3/W4	SAP	COMPOSITE 4	60-day Column	1.51	91.7%	Intertek	2021
KKRC025/026/034	RC	Kwakyekrom	W3/W4	QV/SAP	COMP (KWAK)	70-day Column	0.96	89.6%	Intertek	2021
KBDD077	DDH	Boin	W3/W4/W5	CY/SZ/QV	Comp KBDD077	90-day Column	0.51	98.6%	Intertek	2022
SWDD084A	DDH	Sewum	W3/W4/W5	CY/QV	Comp SWDD084A	90-day Column	0.55	89.0%	Intertek	2022
SWDD084B	DDH	Sewum	W3	MD/SZ/QV	Comp SWDD084B	90-day Column	0.85	89.7%	Intertek	2022
077/084cmp	DDH	Boin/Sewum	W3/W4/W5	CY/MD/SZ/QV	UMaT 4kg	30-day Column	0.68	68.4%	UMaT	2022
077/084cmp	DDH	Boin/Sewum	W3/W4/W5	CY/MD/SZ/QV	UMaT 6kg	30-day Column	0.68	70.5%	UMaT	2022
077/084cmp	DDH	Boin/Sewum	W3/W4/W5	CY/MD/SZ/QV	UMaT 8kg	30-day Column	0.68	76.1%	UMaT	2022
077/084cmp	DDH	Boin/Sewum	W3/W4/W5	CY/MD/SZ/QV	UMaT 8kg+Rk	60-day Column	0.68	86.8%	UMaT	2022
BnMeTTr	Trench	Boin	W5	SAP	BnMeTTr01	90-day Column	1.79	90.9%	Intertek	2023
BnMeTTr	Trench	Boin	W5	SAP	BnMeTTr02	90-day Column	1.04	88.4%	Intertek	2023
SwMeTTr	Trench	Sewum	W5	SAP	SwMeTTr01	90-day Column	1.56	88.6%	Intertek	2023
SwMeTTr	Trench	Sewum	W5	SAP	SwMeTTr02	60-day Column	0.70	94.7%	Intertek	2023
SwMeTTr	Trench	Sewum	W5	SAP	SwMeTTr03	60-day Column	0.59	95.6%	Intertek	2023
SWTR_MET_002	Trench	Sewum	W5	MZ/DU	SwMetTr04	90-day Column	0.84	88.86%	Intertek	2024
SWTR_MET_002	Trench	Sewum	W5	MZ/DU	SwMetTr05	90-day Column	0.50	95.88%	Intertek	2024
SWTR_MET_002	Trench	Sewum	W5	LT	SwMetTr06	90-day Column	0.55	93.30%	Intertek	2024
SWTR_MET_004	Trench	Sewum	W5	SPH	SwMetTr07	90-day Column	0.66	90.16%	Intertek	2024
SWTR_MET_004	Trench	Sewum	W5	SGW	SwMetTr08	90-day Column	0.32	89.82%	Intertek	2024
BNTR_MET_002	Trench	Boin	W5	SAP/SPH	BnMetTr03	120-day Col'n	0.60	98.29%	Intertek	2024
BNTR_MET_003	Trench	Boin	W5	SPH	BnMetTr04	90-day Column	0.48	95.59%	Intertek	2024
NBCH-001	Trench	Nyam	W5	SAP/QV	NBCH_001HG	60-day Column	4.20	88.73%	Intertek	2024
NBCH-001	Trench	Nyam	W5	SAP	NBCH_001LG	60-day Column	0.40	82.14%	Intertek	2024
SWCH-001	Trench	Sewum	W5	SAP/QV	SWCH-001	60-day Column	0.30	83.88%	Intertek	2024
KBTR_MET_001	Trench	Boin	W5	SAP	SwMeT	60-day Pilot	0.81	93.5%	UMaT	2023
SWTR_MET_001	Trench	Sewum	W5	SAP	BnMeT	60-day Pilot	1.09	90.3%	UMaT	2023

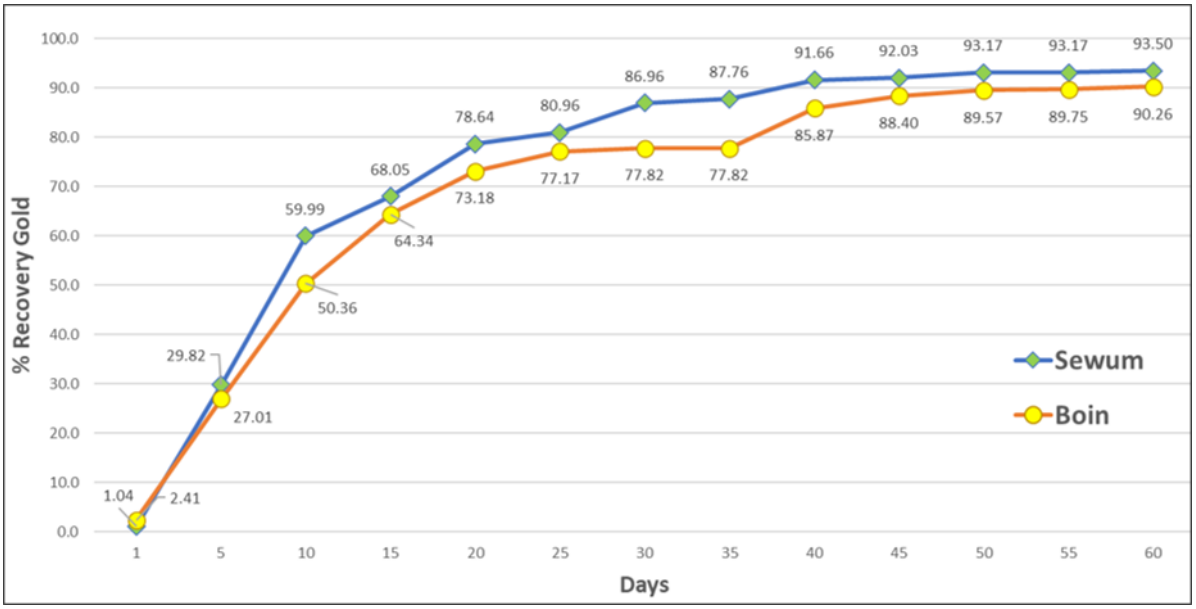
Notes:
CY = Clay, DU = Duricrust, LT = Laterite, SAP = Saprolite, MD = Dolerite, MZ = Mottled Zone, QV = Quartz Vein, SPH = Phyllite, SGW = Greywacke, SZ = Shear Zone

Figure 13.3.5 Summary of Column Leach Tests



Source: Newcore, 2025

Figure 13.3.6 Summary of Column Leach Tests



Source: Newcore, 2024

13.3.9 Size Distribution Analyses

A total of 20 size diagnostic tests were completed between 2020 and 2023 on samples previously tested by bottle roll and column leach tests at Intertek Laboratory in Tarkwa, Ghana. A further 13 tests were completed on tails material from the column tests. The samples include trench, RC, and DDH material. The trench material was sourced from metallurgical trenches at Boin and Sewum. The RC material consists of four coarse reject composites from holes at Boin and Sewum, and one from Kwakyekrom. DDH material includes one composite from Boin and two from Sewum.

Across all samples, an average of 40% of the material was naturally coarser than 2-2.5 mm and contained 51% of the gold, while 35% was finer than 212 to 250 µm and contained 29% of the gold. Refer to Table 13.3.6 to Table 13.3.7 for details.

Table 13.3.6 Size Distribution Analyses (+212/250 µm)

Source	Material	Mesh Size	% Mass Dist	% Passing	Au Dist %
BnMETTR01	Trench	+212µm	64.00	36.00	83.39
BnMETTR02	Trench	+212µm	72.52	27.48	90.75
BnMETTR03	Trench	+212µm	61.45	38.55	75.36
BnMETTR04	Trench	+212µm	43.03	56.97	81.19
SwMETTR01	Trench	+212µm	59.00	41.00	62.95
SwMETTR02	Trench	+212µm	86.07	13.93	81.99
SwMETTR03	Trench	+212µm	92.02	7.98	93.93
SwMETTR04	Trench	+212µm	91.17	8.83	96.40
SwMETTR05	Trench	+212µm	93.16	6.84	95.69
SwMETTR06	Trench	+212µm	95.10	4.90	98.30
SwMETTR07	Trench	+212µm	70.36	29.64	59.26
SwMETTR08	Trench	+212µm	83.35	16.65	87.55
Composite 1	RC	+250µm	42.40	57.60	46.23
Composite 2	RC	+250µm	42.50	57.50	52.29
Composite 3	RC	+250µm	48.99	51.01	37.85
Composite 4	RC	+250µm	36.65	63.35	36.31
Comp Kwak	RC	+250µm	26.11	73.89	50.58
KBDD077	DDH	+212µm	65.37	34.63	86.11
SWDD084 A	DDH	+212µm	59.30	40.70	42.89
SWDD084 B	DDH	+212µm	68.83	31.17	61.46

Table 13.3.7 Size Distribution Analyses (+2-2.5 mm)

Source	Material	Mesh Size	% Mass Dist	% Passing	Au Dist %
BnMETTR01	Trench	+2.5mm	35.20	64.80	63.79
BnMETTR02	Trench	+2.5mm	34.35	65.65	47.66
BnMETTR03	Trench	+2.5mm	45.98	54.02	75.88
BnMETTR04	Trench	+2.5mm	25.95	74.05	57.57
SwMETTR01	Trench	+2.5mm	33.39	66.61	28.15
SwMETTR02	Trench	+2.5mm	48.85	51.15	41.56
SwMETTR03	Trench	+2.5mm	57.33	42.67	76.92
SwMETTR04	Trench	+2.5mm	46.63	53.37	69.82
SwMETTR05	Trench	+2.5mm	53.67	46.33	67.20
SwMETTR06	Trench	+2.5mm	78.75	21.25	91.37
SwMETTR07	Trench	+2.5mm	46.16	53.84	35.93
SwMETTR08	Trench	+2.5mm	68.71	31.29	74.42
Composite 1	RC	+2mm	24.05	75.95	28.00
Composite 2	RC	+2mm	12.57	87.43	27.60
Composite 3	RC	+2mm	16.93	83.07	14.64
Composite 4	RC	+2mm	9.49	90.51	10.91
Comp Kwak	RC	+2mm	26.11	73.89	50.58
KBDD077	DDH	+2.36mm	42.44	57.56	63.23
SWDD084 A	DDH	+2.36mm	36.85	63.15	24.24
SWDD084 B	DDH	+2.36mm	50.85	49.15	31.48

13.3.10 Comminution Testwork

Table 13.3.8 and Table 13.3.9 summarize the bench-scale comminution test results for the project on oxide and fresh material in 2023. The overall dataset includes 30 Bond ball mill work index tests (BWi) and 20 unconfined compressive strength (UCS) tests. The Bond BWi tests were conducted at a closing screen of 106 µm.

Table 13.3.8 Bench-Scale Comminution Testwork at Jet-Rom Engineering Ltd.

Lab	Sample ID	Weathered State	BWi, kWh/t	F_{80}, µm	P_{80}, µm	Gpr	BWi, Closing µm
Jet-Rom	SWMETTRO1	Oxide	10.3	1,499	53.1	1.59	106
Jet-Rom	SWMETTRO2	Oxide	11.7	2,006	57.7	1.39	106
Jet-Rom	SWMETTRO3	Oxide	10.3	2,256	54	1.54	106
Jet-Rom	BNMETTRO1	Oxide	9.8	1,430	54.4	1.73	106

Lab	Sample ID	Weathered State	BWi, kWh/t	F_{80}, µm	P_{80}, µm	Gpr	BWi, Closing µm
Jet-Rom	BNMETTRO2	Oxide	10.8	1,638	43.6	1.29	106
Jet-Rom	2001228	Fresh	10.8	1,794	82.4	2.04	106
Jet-Rom	2001229	Fresh	11.5	2,356	84.3	1.85	106
Jet-Rom	2001238	Fresh	10.5	2,005	77.2	1.97	106
Jet-Rom	2001246	Fresh	10.6	2,031	72.6	1.85	106
Jet-Rom	2001248	Fresh	11.6	2,184	80.4	1.78	106

Table 13.3.9 Bench-Scale Comminution Testwork at Odeleb Ltd.

Lab	Sample ID	Weathered State	BWi, kWh/t	F_{80}, µm	P_{80}, µm	Gpr	BWi, Closing, µm	Ave, UCS MPa
Odeleb	1176698	Fresh	24.6	2,312	74.2	0.67	106	168.5
Odeleb	1176699	Fresh	16.1	2,224	69.1	1.06	106	144
Odeleb	1176700	Fresh	17.1	2,564	72.5	1.00	106	152.4
Odeleb	1176701	Fresh	16.5	2,501	68.6	1.01	106	118.9
Odeleb	1176702	Fresh	16.9	2,561	68.1	0.97	106	205.2
Odeleb	1176703	Fresh	19.1	2,516	68.5	0.85	106	145.3
Odeleb	1176704	Fresh	28	2,233	74.6	0.57	106	218.2
Odeleb	1176705	Fresh	17.3	2,204	62.6	0.91	106	181.7
Odeleb	1176706	Fresh	18.2	2,213	64.9	0.87	106	216.9
Odeleb	1176707	Fresh	18.1	2,251	68.4	0.91	106	113.7
Odeleb	1176708	Fresh	17.1	2,189	75.2	1.06	106	175.8
Odeleb	1176709	Fresh	14.1	2,150	67.8	1.24	106	51.3
Odeleb	1176710	Fresh	19.4	2,187	54.0	0.71	106	168.6
Odeleb	1176711	Fresh	20.2	2,288	68.8	0.80	106	187.1
Odeleb	1176712	Fresh	16.3	2,099	74.3	1.11	106	73.3
Odeleb	1176713	Fresh	11.6	2,045	66.6	1.56	106	25.5
Odeleb	1176714	Fresh	17.4	1,991	61.4	0.90	106	68.3
Odeleb	1176715	Fresh	16.3	1,995	55.5	0.91	106	69.2
Odeleb	1176716	Fresh	18.4	2,059	65.7	0.88	106	47.6
Odeleb	1176717	Fresh	18	1,925	64.2	0.90	106	68.1

In 2024, additional comminution testwork was completed by Odoleb Ltd. on Boin and Nyam fresh samples, and Boin, Sewum and Nyam oxide and transition samples. Refer to Table 13.3.10 to Table 13.3.12 for details.

Table 13.3.10 Unconfined Compressive Strength Testwork at Odeleb Ltd.

Lab	Sample ID	Deposit	Oxidation	UCS Mpa
Odeleb	1176776	Nyam	Sulp	31.54
Odeleb	1176777	Nyam	Sulp	35.10
Odeleb	1176778	Nyam	Sulp	31.80
Odeleb	1176779	Nyam	Sulp	15.90
Odeleb	1176780	Nyam	Sulp	59.55
Odeleb	1176781	Nyam	Sulp	34.61
Odeleb	1176782	Nyam	Sulp	40.39
Odeleb	1176783	Nyam	Sulp	81.63
Odeleb	1176784	Nyam	Sulp	23.08
Odeleb	1176785	Nyam	Fresh	32.30
Odeleb	1176786	Nyam	Fresh	40.50
Odeleb	1176787	Boin	Fresh	28.87
Odeleb	1176788	Boin	Fresh	44.52
Odeleb	1176789	Boin	Fresh	51.81
Odeleb	1176790	Boin	Fresh	88.27
Odeleb	1176791	Boin	Fresh	70.94
Odeleb	1176792	Boin	Fresh	23.69
Odeleb	1176793	Boin	Fresh	65.29
Odeleb	1176794	Boin	Fresh	41.30
Odeleb	1176795	Boin	Fresh	27.70
Odeleb	1174938	Boin	Trans	2.04
Odeleb	1174940	Boin	Trans	3.55
Odeleb	1174942	Boin	Trans	9.02
Odeleb	1174944	Boin	Trans	10.90
Odeleb	1174946	Boin	Trans	5.26
Odeleb	1174948	Boin	Trans	18.22
Odeleb	1174950	Sewum	Trans	4.44
Odeleb	1174952	Sewum	Trans	7.57
Odeleb	1174954	Sewum	Trans	12.40
Odeleb	1174956	Sewum	Trans	10.04
Odeleb	1174958	Sewum	Trans	23.44
Odeleb	1174960	Sewum	Trans	32.47
Odeleb	1174962	Nyam	Trans	9.92
Odeleb	1174964	Nyam	Trans	5.90
Odeleb	1174966	Nyam	Trans	2.33
Odeleb	1174968	Boin	Ox	0.08
Odeleb	1174970	Boin	Ox	0.20
Odeleb	1174972	Sewum	Ox	0.51

Lab	Sample ID	Deposit	Oxidation	UCS Mpa
Odeleb	1174974	Sewum	Ox	0.19
Odeleb	1174976	Nyam	Ox	0.06

Table 13.3.11 Point Load Index Testwork at Odeleb Ltd.

Lab	Sample ID	Deposit	Oxidation	Point Load Index, Is50 Mpa
Odeleb	1176756	Nyam	Fresh	3.51
Odeleb	1176757	Nyam	Fresh	5.25
Odeleb	1176758	Nyam	Fresh	0.76
Odeleb	1176759	Nyam	Fresh	5.07
Odeleb	1176760	Nyam	Fresh	5.07
Odeleb	1176761	Nyam	Fresh	7.59
Odeleb	1176762	Nyam	Fresh	4.09
Odeleb	1176763	Nyam	Fresh	8.12
Odeleb	1176764	Nyam	Fresh	3.83
Odeleb	1176765	Nyam	Fresh	6.37
Odeleb	1176766	Nyam	Fresh	6.54
Odeleb	1176767	Boin	Fresh	6.88
Odeleb	1176768	Boin	Fresh	6.69
Odeleb	1176769	Boin	Fresh	7.81
Odeleb	1176770	Boin	Fresh	6.33
Odeleb	1176771	Boin	Fresh	4.83
Odeleb	1176772	Boin	Fresh	3.35
Odeleb	1176773	Boin	Fresh	3.97
Odeleb	1176774	Boin	Fresh	4.31
Odeleb	1176775	Boin	Fresh	3.39
Odeleb	1174937	Boin	Trans	0.07
Odeleb	1174939	Boin	Trans	0.13
Odeleb	1174941	Boin	Trans	0.43
Odeleb	1174943	Boin	Trans	0.31
Odeleb	1174945	Boin	Trans	0.59
Odeleb	1174947	Boin	Trans	0.66
Odeleb	1174949	Sewum	Trans	0.30
Odeleb	1174951	Sewum	Trans	0.49
Odeleb	1174953	Sewum	Trans	0.36
Odeleb	1174955	Sewum	Trans	0.52
Odeleb	1174957	Sewum	Trans	0.44
Odeleb	1174959	Sewum	Trans	1.21

Lab	Sample ID	Deposit	Oxidation	Point Load Index, Is50 Mpa
Odeleb	1174961	Nyam	Trans	0.04
Odeleb	1174963	Nyam	Trans	0.57
Odeleb	1174965	Nyam	Trans	0.20
Odeleb	1174967	Boin	Ox	0.00
Odeleb	1174969	Boin	Ox	0.06
Odeleb	1174971	Sewum	Ox	0.00
Odeleb	1174973	Sewum	Ox	0.04
Odeleb	1174975	Nyam	Ox	0.02

Table 13.3.12 BWi, DWi, SMC and CWi Testwork at Odeleb Ltd.

Lab	Sample ID	Deposit	Oxidation	BWi kWh/t	DWi kWh/t	SMC	CWi kWh/t
Odeleb	1176801	Crush Rock	sulp				21.57
Odeleb	1176802	Crush Rock	sulp				19.17
Odeleb	1176803	Crush Rock	sulp				23.64
Odeleb	Comp A	Boin	sulp	14.78	18.8	16.84	19.03
Odeleb	Comp B	Nyam	sulp	14.62	15.8	15.50	18.90
Odeleb	Comp DecA	Boin	Trans/Ox	4.67		5.26	10.46
Odeleb	Comp DecB	Sewum	Trans/Ox	7.27		5.54	14.53
Odeleb	Comp DecC	Nyam	Trans/Ox	3.55		5.90	13.78

13.4 Summary of 2025 Heap Leach Testwork Program

In 2025, Intertek conducted column leach and column percolation testwork on composite ore samples received from Newcore, to evaluate their amenability to heap leaching. The objective was to assess the suitability of the ore for heap leach processing, determine potential metal recoveries, and estimate reagent consumption for low-grade ore. Although a heap leach design was not considered in this PFS, this work was completed on the back of prior studies that evaluated the Project's amenability to heap leach processing, and this additional work was used to determine the optimal processing methodology for the Project for the PFS. A summary of the 2025 heap leach amenability testwork is summarized in this section.

Column leach testwork was conducted on composite ore samples received from Newcore in two rounds from the Nyam, Boin, and Sewum deposits. Round 1 comprised 12 ore samples of various weathering types received in May 2025, while Round 2 comprised nine transition ore samples received in September 2025. Refer to Figure 13.2.1 to Figure 13.2.5 for the drill hole locations from which the samples were collected.

Table 13.4.1 summarizes the column leach test results for different ore samples at various crush sizes. Gold head grades varied from approximately 0.36 to 1.12 g/t, with transition samples showing a higher average head grade of about 0.83 g/t. The recorded head grade of 1.67 g/t for Nyam 4% cement, 44 mm, sample may indicate a coarse or nugget gold effect that may require further investigations.

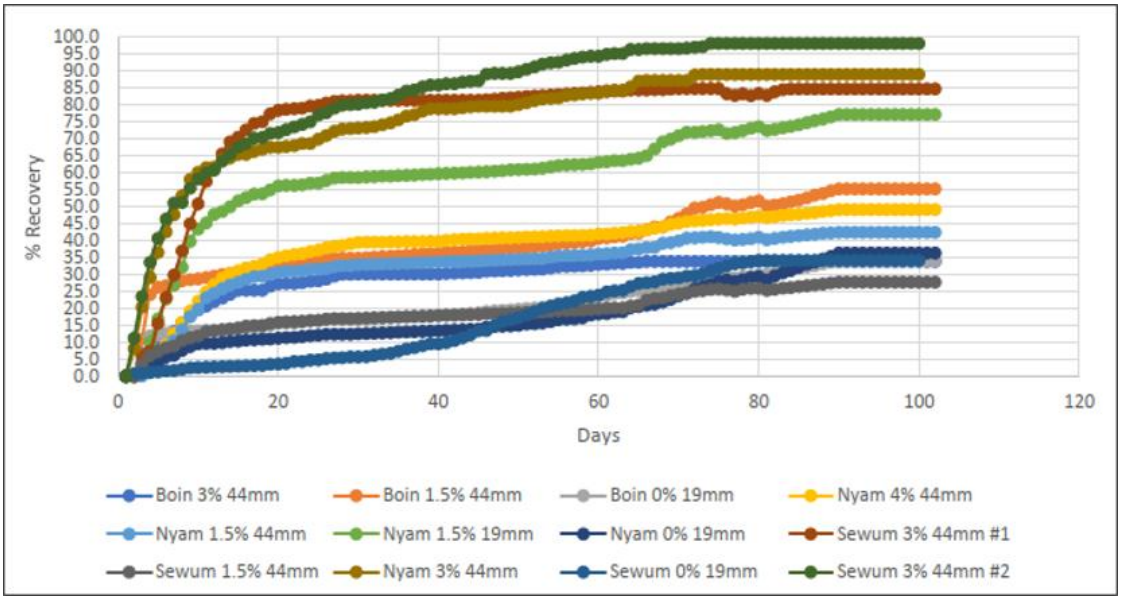
Gold recoveries were generally higher for the transition samples, except for the Round-1 Sewum deposit samples. Average gold extraction after 90 days of leaching was about 91% for the Round-2 transition samples and slightly above 55% for the Round-1 samples. Cyanide and lime consumptions were higher for the Round-2 transition samples, with average cyanide consumption of 1.2 kg/t and 1.5 kg/t, and average lime consumption of 0.16 and 0.18 kg/t for the Round-1 and Round-2 samples, respectively.

Table 13.4.1 Summary of Column Leach Recovery and Reagent Consumption

Sample ID		Slump	Head Assay C	Recovery C	Reagent Consumption g/t		
		%	Au, g/t	%	NaCN	Lime	Cement
Boin Ox 3%, 44 mm	Round-1	0.7	0.79	38.4	1.26	0.19	30
Boin Trans 1.5%, 44 mm		0.5	0.37	16.6	1.14	0.15	15
Boin Fresh 0%, 19 mm		0.9	0.36	8.6	1.22	0.12	0
Nyam Ox 4%, 44 mm		1.5	1.67	94.6	1.47	0.2	40
Nyam Trans 1.5%, 44 mm		1.3	0.69	57.9	1.19	0.14	15
Nyam Trans 1.5%, 19 mm		0.6	0.47	67.8	1.06	0.13	15
Nyam Fresh 0%, 19 mm		0.3	0.52	33.1	1.29	0.13	0
Sewum Ox 3%, 44 mm (1)		0.6	0.48	97.9	1.3	0.2	30
Sewum Trans 1.5%, 44 mm		0.3	0.66	23.8	1.29	0.14	15
Sewum Ox 3%, 44 mm		0.6	0.51	88.2	1.14	0.21	30
Sewum Fresh 0%, 19 mm		0.3	0.58	36.0	0.97	0.11	0
Sewum Ox 3%, 44 mm (2)		0.5	0.47	100.0	1.27	0.19	30
Boin Trans 4%, 44 mm	Round-2	2.1	1.12	86.6	1.13	0.16	40
Boin Trans 4%, 19 mm		2.1	0.92	91.3	1.2	0.17	40
Boin Trans 4%, 9.5 mm		1.8	0.77	95.5	1.21	0.17	40
Nyam Trans 4%, 44 mm		2.4	1.05	88.5	2.1	0.21	40
Nyam Trans 4%, 19 mm		1.2	0.82	86.0	1.56	0.17	40
Nyam Trans 4%, 9.5 mm		2.1	1.07	92.6	1.44	0.16	40
Sewum Trans 4%, 44 mm		1.8	0.54	94.5	1.71	0.19	40
Sewum Trans 4%, 19 mm		2.1	0.58	94.8	1.68	0.19	40
Sewum Trans 4%, 9.5 mm		1.2	0.58	96.5	1.47	0.19	40

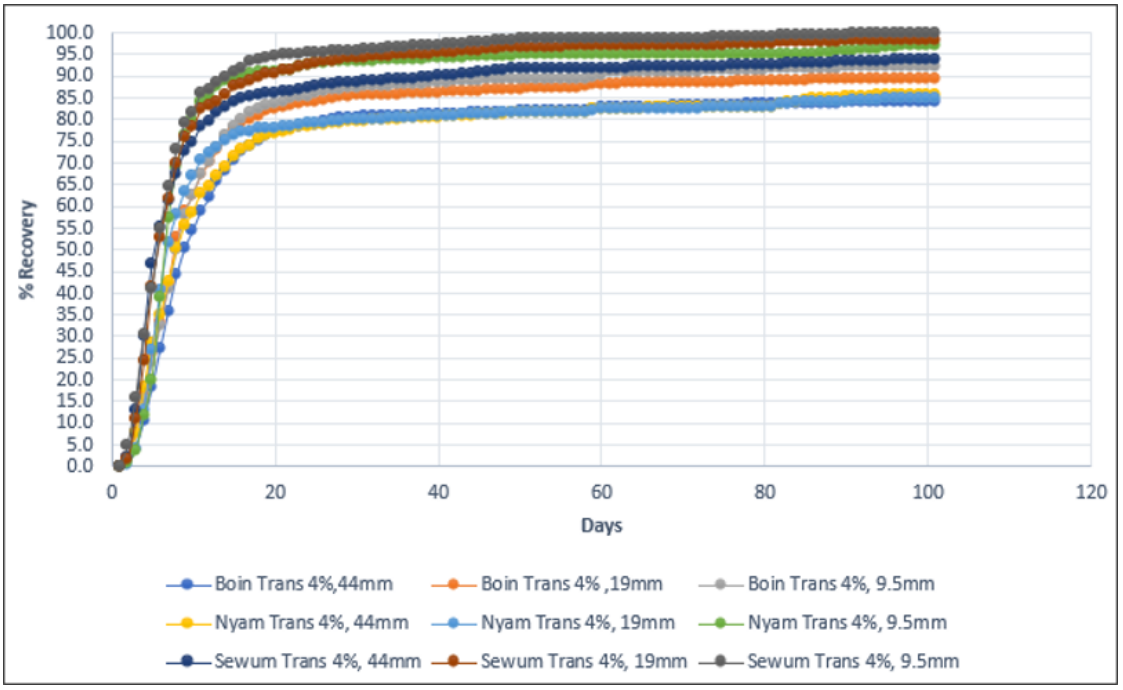
Note:
The %" in the sample ID refers to the % cement in the sample.

Figure 13.4.1 Column Leach Recovery vs Leach Days (Round-1 Samples)



Source: Intertek, 2026

Figure 13.4.2 Column Leach Recovery vs Leach Days (Round-2 Samples)



Source: Intertek, 2026

Figure 13.4.1 and Figure 13.4.2 show corresponding leach rate profiles for Round-1 and Round-2 samples. Gold recovery increased significantly during the early stages of leaching, particularly for the Round-2 diggable transition ore samples, and then gradually slowed as the leach progressed. Most of the recovery for the Round-2 samples was achieved within the first 40 days, reaching approximately 85–97%, depending on the deposit. In contrast, Round-1 samples required a longer leach period of about 75–90 days to approach the highest recovery, with final recoveries ranging in 8-100% depending on the deposit. For the Round-2 transition ores, gold recovery saturated at different times by deposit, with the Boin deposit samples reaching saturation after about 60 days, Nyam deposit samples after 70 days, and Sewum deposit samples after approximately 70 days.

Slump was calculated based on the change in ore height before and after irrigation in the columns. Slump is used to assess the stability and settlement of the leached material. The slump was observed to be in the range of 0.3-2.4%, with an average of about 0.7% for the Round-1 ore samples and about 1.9% for the Round-2 samples – which was below the 10% threshold, indicating good heap leach stability.

Many of the Round-1 samples (Nyam 4%, 44 mm; Boin 1.5%, 44 mm; Sewum 3%, 44 mm; and Boin 0%, 19 mm) showed poor accountability, with values either below 50% or above 150%, potentially indicating a nugget effect where the probability of gold not being evenly distributed across the sample fractions may be fairly high. Another indicator of nugget effect was the presence of elevated gold grades in coarse size fractions during size distribution analysis. For example, in Nyam 4% (44 mm), the +25 mm fraction recorded about 28 g/t Au. This indicated that gold occurred as coarse particles or nuggets rather than being finely disseminated, contributing to variability in assay results and leach recovery performance.

From the PRI analysis, it was observed that PRI values varied across ore sample types and size fractions. Round-1 Boin, 0% composite sample exhibited the highest PRI value, exceeding 1.0. On an average, the Round-2 transition ore samples had stronger preg-robbing potential than the Round-1 samples, as indicated in Table 13.4.2.

Table 13.4.2 Summary of Preg-Robbing Results

Sample ID		Average PRI %
Boin Ox 3%, 44 mm	Round-1	0.16
Boin Trans 1.5%, 44 mm		0.59
Boin Fresh 0%, 19 mm		1.46
Nyam Ox 4%, 44 mm		0.07
Nyam Trans 1.5%, 44 mm		0.33
Nyam Trans 1.5%, 19 mm		0.33
Nyam Fresh 0%, 19 mm		0.00
Sewum Ox 3%, 44 mm (1)		0.24
Sewum Trans 1.5%, 44 mm		0.81

Sample ID		Average PRI %
Sewum Ox 3%, 44 mm		0.24
Sewum Fresh 0%, 19 mm		0.80
Sewum Ox 3%, 44 mm (2)		0.20
Boin trans 4%, 44 mm		0.80
Boin trans 4%, 19 mm		0.70
Boin trans 4%, 9.5 mm		0.90
Nyam trans 4%, 44 mm		0.70
Nyam trans 4%, 19 mm	Round-2	0.70
Nyam trans 4%, 9.5 mm		0.50
Sewum trans 4%, 44 mm		0.90
Sewum trans 4%, 19 mm		0.90
Sewum trans 4%, 9.5 mm		1.00

13.5 Summary of Recent CIL Testwork Program (2024 to 2026)

From 2025 to 2026, Intertek and Intertek Base Met Labs conducted metallurgical testing geared towards a conventional carbon-in-leach (CIL) gold plant for Enchi composite samples to assess metallurgical performance and mineral characteristics across the main deposits, Nyam, Boin and Sewum. Drill core and coarse reject material was received in four shipments, and the composites were prepared to represent fresh, transition, and oxide mineralization from the various deposits, including an MRE-grade Sewum blend, consisting of 75 kg at a grade of 0.98 g/t Au, designed to reflect expected mill feed. The drill holes from which these samples were collected from are:

- Nyam – NBDD070, and NBDD071.
- Boin – KBDD094, and KBDD088.
- Sewum – SWDD038, SWDD051, SWDD048, and SWDD043.

The program evaluated the ore's amenability to conventional free milling gold plant processing routes, namely crushing and milling, followed by gravity recovery with cyanide leaching of the gravity tails via a CIL process. Testwork included in this program was comminution, head analysis, mineralogy, preg-robbing analysis, gravity recovery, cyanide leaching, flotation-leach, and tailings dewatering.

13.5.1 Comminution Test Results

The Bond ball work index (BWi) tests were conducted on six composites provided by Newcore with a closing screen size of 106 µm. The BWi of these samples ranged between 5.50 and 16.3 kWh/t, which would be considered moderately hard. The Bond abrasion indices (Ai) ranged between 0.066 and 0.299, with lowest value measured for Nyam W3 -19 +12, indicating low to moderate abrasiveness. Refer to Table 13.5.1 for a summary of the latest comminution test results from Intertek.

Table 13.5.1 Comminution Test Results (Intertek 2025-2026)

Sample ID	Bond Ball Mill Work Index				Abrasion Index
	F_{80} µm	P_{80} µm	Gpr	BWi kWh/t	
Sewum-W3	590	47	2.34	8.04	-
Nyam-W3	1,289	65	4.09	5.5	-
Boin-W3	1,189	49	1.42	11.0	-
Boin Bulk AB	2,414	81	1.25	15.44	0.316
Sewum Bulk AB	2,583	75	1.09	16.34	0.22
Nyam Bulk AB	2,381	77	1.34	14.19	0.205
Nyam-W3 -19 +12	-	-	-	-	0.066
Boin-W3 -19 +12	-	-	-	-	0.299

13.5.2 Head Analysis and Preg Robbing Analysis

The composite samples displayed highly variable gold grades ranging from 0.32 to 4.49 g/t Au, alongside low silver values, with the highest gold grade along with elevated arsenic (up to ~2,266 g/t As) both concentrated in the Sewum Fresh-High Grade sample. Sulphur content remains low to moderate (0.02–2.08%) and is predominantly sulphide-based, while total carbon peaks at ~4.5% in fresh zones despite generally low Total Organic Carbon (0.04–0.62%). Refer to the Table 13.5.2.

Table 13.5.2 Composite Head Analysis (Intertek 2025-2026)

Comp ID	Cu %	Fe %	Au g/t	Ag g/t	As g/t	S %	C %	$S(SO_4)$ %	S^{2-} %	TOC %	Cg %
Nyam Fresh	0.007	4.11	0.85	0.2	156	1.24	2.14	<0.01	1.24	0.11	
Boin Fresh	0.015	3.08	0.79	0.35	133	0.86	1.71	0.025	0.83	0.62	
Sewum Fresh (Low Grade)	0.009	5.51	0.47	0.2	243	1.11	4.38	0.01	1.11	0.2	
Sewum Fresh (High Grade)	0.008	6.64	4.49	0.2	2266	2.08	4.54	0.032	2.05	0.12	0.05
Sewum W3	0.003	4.88	0.46	0.15	276	0.02	0.06	<0.01	0.02	0.04	
Nyam W3	0.004	3.86	0.32	0.35	94.5	0.04	0.07	<0.01	0.04	0.07	
Boin W3	0.002	2.05	0.6	0.45	152	0.02	0.05	<0.01	0.02	0.06	
Boin Oxide (+850 /+150 µm)			1.03			1.15					
Boin Oxide (+2 mm +6.3 mm)				0.46		0.39					

Comp ID	Cu %	Fe %	Au g/t	Ag g/t	As g/t	S %	C %	$S(SO_4)$ %	S^{2-} %	TOC %	Cg %
Sewum Fresh – MRE Grade*			0.91			1.1					
Sewum Oxide Composite			0.51								
Boin Oxide Composite			1.02								
Nyam Oxide Composite			0.84								

* Consisting of 75 kg at a grade of 0.98 g/t Au.

13.5.3 Mineralogical Analysis

Mineralogical characteristics of six composites were measured using the QEMSCAN Bulk Mineral Analysis routine. Pyrite was the main sulphide for mineral across the composites accounting for 99% of total sulphur, for Nyam Fresh, Boin Fresh and Sewum Fresh-Low Grade, and 56 to 73% for Nyam-W3 Comp, Boin-W3 Comp, and Sewum-W3 Comp. The remaining mass was comprised mainly of quartz, feldspar and mica. Refer to Table 13.5.3 for details on sulphur deportment.

Table 13.5.3 Sulphur Deportment (Intertek 2025-2026)

Mineral	% S Distribution					
	Nyam Fresh	Boin Fresh	Sewum Fresh - Low Grade	Nyam - W3 Comp	Boin - W3 comp	Sewum - W3 Comp
Pyrite	98.5	98.5	98.9	71.8	55.9	73.1
Other Sulphides	1.22	1.23	1.10	25.6*	34.9*	26.0*
Other	0.28	0.25	0.01	2.63**	9.23**	0.91*

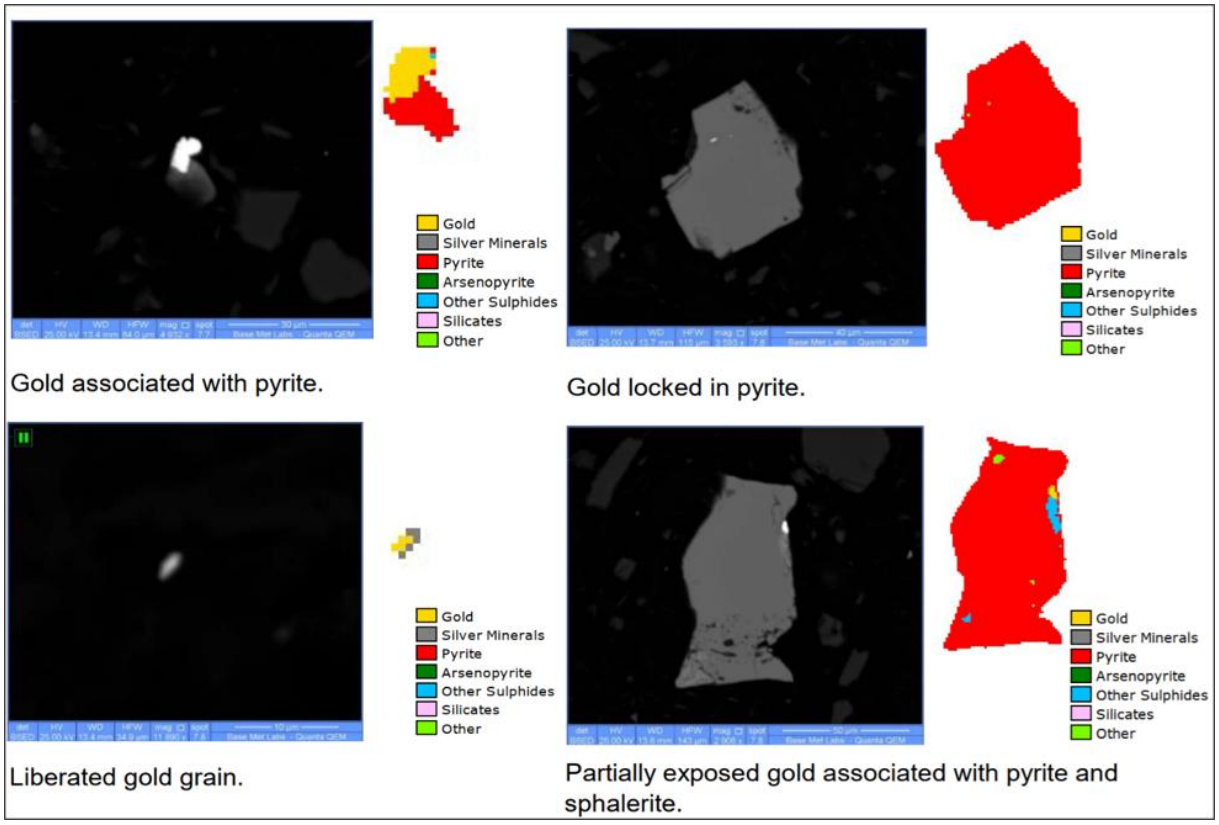
Deportment based on low S grade and low particle statistics.
*Mostly Enargite and Chalcopyrite
**Mostly K-Al-Si-Sulphates

A study of the nature of the gold was also completed on two samples through QEMSCAN Trace Mineral Search (TMS). Table 13.5.4 describes the compositions of gold grains detected. As seen in Figure 13.5.1, the gold grains had various degrees of liberation and exposure, with some grains fully liberated while other grains were locked in pyrite. Additional gold deportment study is recommended to better assess the degree of gold liberation.

Table 13.5.4 Gold Grain Mineral Content (Intertek 2025-2026)

Comp ID	Particle	Gold-Bearing Mineral	Element Mass %		
			Au	Ag	Total
Boin Oxide U/S PSD (+850+150 µm)	Grain - 1	Electrum	9.7	90.3	100
	Grain - 2	Electrum	9.9	90.1	100
	Grain - 3	Electrum	40.7	59.3	100
	Grain - 4	Electrum	19.9	80.1	100

Figure 13.5.1 Gold Grain Images



Source: Intertek, 2026

13.5.4 Preg-Robbing Analysis

Preg-robbing analysis, as seen in Table 13.5.2, indicated negligible preg-robbing potential for most composites; however, the Boin Fresh and Sewum Fresh-Low Grade composites exhibited elevated preg-robbing behaviour.

Table 13.5.5 Preg-Robbing Analysis (Intertek 2025-2026)

Sample ID	Intensive Leach	IL with Spike	Spike Standard	PRV	% Factor	PRI
	24 hr Sol'n Assay - g/tonne		ppm			
Nyam Fresh	0.25	5.15	5.00	0.10	2%	0.02
Boin Fresh	1.07	0.01	5.00	6.06	121%	1.21
Sewum Fresh	0.01	2.39	5.00	2.62	52%	0.52
Nyam W3	0.16	5.16	5.00	0.00	0%	0.00
Boin W3	0.23	5.18	5.00	0.05	1%	0.01
Sewum W3	0.25	5.16	5.00	0.09	2%	0.02

13.5.5 Direct Leaching and Gravity-Leaching Tests

Whole ore or direct leaching tests were conducted to evaluate metallurgical performance without gravity recovery. Refer to Table 13.5.2 for a summary of the results.

As seen in Table 13.5.6, the oxide samples are not sensitive to the grind size. Transition and oxide samples generally exhibited faster leach kinetics and higher ultimate recoveries relative to fresh material. Furthermore, the addition of kerosene during the grinding stage successfully passivated naturally occurring carbonaceous material, yielding significantly improved recoveries of approximately 20 to 25% higher than the baseline cyanidation.

Table 13.5.6 Direct Leaching Test

Test ID	Comp ID	Mat. Type	Grind Size P_{80} µm	Kerosene g/t	Leach Time hr	Total Extraction % Au	Calc'd Head Grade g/t Au	Tail Grade g/t Au	NaCN kg/t	Lime kg/t
CIL-04	Nyam Fresh	Fresh	53	-	48	89.7	1.47	0.15	1.8	0.79
CIL-05	Boin Fresh	Fresh	53	-	48	63.4	0.57	0.21	2.25	0.96
CIL-15	Boin Fresh	Fresh	53	250	48	83.2	1.04	0.18	1.26	1.95
CIL-06	Sewum Fresh	Fresh	53	-	48	60.2	0.55	0.22	2.87	1.35
CIL-08	Nyam W3	Trans.	53	-	48	91.4	0.29	0.03	1.62	3.6
CIL-10	Boin W3	Trans.	53	-	48	96.6	0.59	0.02	1.05	1.85
CIL-12	Sewum W3	Trans.	53	-	48	96.6	0.44	0.02	1.16	2.37
CIL-16	Sewum Blend	Trans.	53	-	48	61.9	0.94	0.36	2.72	0.84
CIL-17	Sewum Blend	Trans.	53	250	48	87.9	0.91	0.11	2.51	0.93
CIl Test February 2026	Sewum_Ox	Oxide	53	-	24	96.0	0.51	0.02	0.83	1.7
	Sewum_Ox	Oxide	75	-	24	98.1	0.51	0.01	0.81	1.7
	Boin_Ox	Oxide	53	-	24	89.0	1.09	0.12	0.76	1.7
	Boin_Ox	Oxide	75	-	24	91.9	0.99	0.08	0.76	1.5
	Nyam_Ox	Oxide	53	-	24	90.9	0.88	0.08	0.89	1.7
	Nyam_Ox	Oxide	75	-	24	88.5	0.87	0.10	0.85	1.5

Table 13.5.7 provides a summary of the gravity-leaching tests. The results indicated that reducing the primary grind size from 75 to 53 µm increased overall gold recovery significantly for the fresh samples. Additionally, the inclusion of gravity recovery generally improved overall gold recovery relative to direct cyanidation.

Table 13.5.7 Gravity-Leaching Test

Test	Comp ID	Mat. Type	Grind Size P_{80} µm	Kerosene g/t	Leach Time hr	Grav Recovery %Au	Total Extraction % (Grav + CIL	Tail Grade (g/t Au)	NaCN kg/t	Lime (kg/t)
GCIL-01	Nyam Fresh	Fresh	53	-	48	16.4	91.4	0.12	1.61	0.71
GCIL-25	Nyam Fresh	Fresh	75	-	48	22.3	82.4	0.2	0.93	0.91
GCIL-02	Boin Fresh	Fresh	53	-	48	40.6	78.6	0.14	1.7	1.3
GCIL-21	Boin Fresh	Fresh	53	250	48	38.9	89.4	0.11	1.88	0.89
GCIL-23	Boin Fresh	Fresh	75	-	48	38.2	86.7	0.13	0.96	2.01
GCIL-03	Sewum Fresh	Fresh	53	-	48	27.6	72.8	0.2	2.49	1.32
GCIL-07	Nyam-W3	Trans.	53	-	48	14.6	90.6	0.03	1.53	3.19
GCIL-09	Boin-W3	Trans.	53	-	48	7.8	96.6	0.02	0.65	2.25
GCIL-11	Sewum-W3	Trans.	53	-	48	9.2	97.7	0.01	1.14	2.45
GCIL-22	Sewum Blend	Trans.	53	250	48	39	90.2	0.1	2.3	1.08
GCIL-24	Sewum Blend	Trans.	75	-	48	34.1	87.1	0.12	1.44	1.39

Note that all the fresh and transition samples were leached at 48 hours with 250 ppm lead nitrate as a pre-treatment. The oxide samples were leached at only 24 hours to achieve gold recoveries in the high 80% to high 90%.

13.5.6 Flotation and Cyanide Leaching

Gravity-flotation-leach testing was completed on selected fresh samples. Following gravity separation via panning, gravity tailings underwent rougher flotation using Potassium Amyl Xanthate (PAX), with subsequent leaching of individual products to assess extraction performance.

Leaching of the flotation concentrate resulted in substantially lower recoveries for the Boin and Sewum composites due to the concentration of naturally occurring carbonaceous material within the flotation concentrate. The results indicate that flotation is unlikely to provide a processing advantage unless carbon rejection or carbon passivation strategies are incorporated.

13.5.7 Diagnostic Leaching

A three-stage diagnostic leach was completed on the Boin Fresh flotation tailings to characterize the occurrences of residual gold. The investigation indicated that while 35% of the residual gold remained directly amenable to cyanidation, no significant sulphide-locked gold was identified. Instead, approximately 65% of the remaining gold was encapsulated within silicate and gangue minerals, indicating that additional fine grinding alone is unlikely to substantially increase metallurgical recovery for this material, highlighting the importance of a controlled addition of a hydrocarbon reagent (kerosene) to achieve a substantial increase in gold recovery.

13.5.8 Tailings Settling Tests

Initial flocculant screening tests identified the anionic flocculants Magnafloc AN913SH and Magnafloc AN 9609 VHM as the preferred reagents for the thickening applications, with AN913SH evaluated at dosages ranging from approximately 5 to 40 g/t for the fresh composites, and AN 9609 VHM evaluated at dosages from 15 to 120 g/t for the oxide material. Refer to Table 13.5.8 for a summary of the thickening testwork results for both fresh and oxide samples.

Table 13.5.8 Static Settling Tests

Zone	Mat. Type	Floc. Type	Floc. Dosage g/t	pH	Initial Density %	Final Unraked Density %	Final Raked Density %	Free Settling Rate mm/s	Calc'd Solids Loading t/m^2h	Overflow Clarity TAU
Nyam	Fresh	-	-	10.5	13.6	61.7	63.1	0.25	0.13	100
Nyam	Fresh	AN913SH	5	10.5	13.6	58.1	60.5	1.29	0.69	46
Nyam	Fresh	AN913SH	10	10.5	13.6	58.2	60.5	1.5	0.80	15
Nyam	Fresh	AN913SH	20	10.5	13.6	56	60.5	1.75	0.93	7
Nyam	Fresh	AN913SH	40	10.5	13.6	56	58.2	2.03	1.08	8
Boin	Fresh	-	-	10.5	13.7	56.2	57.3	0.17	0.09	75
Boin	Fresh	AN913SH	5	10.5	13.6	54.1	56.1	0.48	0.25	35
Boin	Fresh	AN913SH	10	10.5	13.7	54.2	58.4	0.52	0.28	10
Boin	Fresh	AN913SH	20	10.5	13.6	52.2	54.1	0.56	0.30	5
Boin	Fresh	AN913SH	40	10.5	13.7	48.8	53.2	0.89	0.48	4
Sewum	Fresh	-	-	10.5	13.7	65.8	65.8	0.16	0.09	114
Sewum	Fresh	AN913SH	5	10.5	13.6	60.4	62.9	1.67	0.89	74
Sewum	Fresh	AN913SH	10	10.5	13.6	60.5	63	1.47	0.78	45
Sewum	Fresh	AN913SH	20	10.5	13.6	56	59.3	2.09	1.11	7
Sewum	Fresh	AN913SH	40	10.5	13.6	53.9	59.3	2.78	1.48	5
Boin	Oxide	-	-	10.5	15.2	41.1	41.1	0.25	0.15	-
Boin	Oxide	AN 9609 VHM	15	10.5	15.6	44.1	44.7	0.34	0.20	-
Boin	Oxide	AN 9609 VHM	30	10.5	15	44.1	44.7	0.42	0.24	-
Boin	Oxide	AN 9609 VHM	60	10.5	15.1	46.7	46.7	0.43	0.25	-
Boin	Oxide	AN 9609 VHM	117	10.5	15.4	51.8	51.8	0.46	0.27	-
Sewum	Oxide	-	-	10.5	15	39.8	40.3	0.12	0.07	-
Sewum	Oxide	AN 9609 VHM	15	10.5	15.1	46.7	47.3	0.31	0.18	-
Sewum	Oxide	AN 9609 VHM	29	10.5	15.3	47.3	48	0.33	0.19	-
Sewum	Oxide	AN 9609 VHM	59	10.5	15.3	50.1	50.1	0.36	0.21	-
Sewum	Oxide	AN 9609 VHM	117	10.5	15.4	50.3	51.1	0.42	0.25	-

Zone	Mat. Type	Floc. Type	Floc. Dosage g/t	pH	Initial Density %	Final Unraked Density %	Final Raked Density %	Free Settling Rate mm/s	Calc'd Solids Loading t/m²h	Overflow Clarity TAU
Nyam	Oxide	-	-	10.5	15	40.8	41.3	0.12	0.07	409
Nyam	Oxide	AN 9609 VHM	15	10.5	15	45.3	45.9	0.36	0.21	372
Nyam	Oxide	AN 9609 VHM	30	10.5	15.1	48	48	0.37	0.21	347
Nyam	Oxide	AN 9609 VHM	60	10.5	15.1	48.6	49.4	0.37	0.21	270
Nyam	Oxide	AN 9609 VHM	120	10.5	15	50	50	0.38	0.22	100

Static settling tests demonstrated that fresh (sulfide) tailings exhibit highly favorable dewatering characteristics compared to the oxide material. The fresh tailings yielded unraked underflow densities of 53.2% to 65.8% solids (settling at 0.16–0.89 mm/s) with flocculant dosing up to 40 g/t, whereas the oxide tailings achieved only 39.8% to 48.0% solids (settling at 0.12–0.37 mm/s) with flocculant rate up to 30 g/t.

13.6 Results Interpretation

This section describes how the results in Sections 13.3, 13.4 and 13.5 are interpreted for application to a CIL process, which forms the basis of the 2026 PFS plant design.

13.6.1 Comminution Design

Table 13.6.1 summarizes the comminution parameters by weathering zone and deposits and provides design parameters used by OMC for the crushing, mineral sizing and grinding circuits. Where parameters were unavailable, OMC used database or benchmark information to infer ore properties; these inferred values are denoted in italic font. Oxide and transition ore properties were used for the mineral sizer circuit design, and fresh / ore properties were used for the primary crushing circuit design, respectively. The 85th percentile of available BWi and CWi test results was selected for milling circuit power modeling. The milling circuit design is based on a representative annual ore blend corresponding to the year with the highest proportion of fresh rock in the mine plan provided by Newcore at the time of the study.

Table 13.6.1 Summary of Comminution Test Results and Design Parameters

Ore Type & Deposit		Blend Year 4 %	Ai g	CWi kWh/t	BWi (106 µm) kWh/t	BWi F_{80} µm	BWi P_{80} µm	Axb	ta	Ore SG
Fresh Rock	Boin	58.9	0.316	19	20.5	1987	69	40.1	0.57	2.88
	Sewum	-	0.22	15.6	19.9	2222	69	32.1	0.41	2.84
	Nyam	17.7	0.205	17.5	14.5	2038	76	32.7	0.34	2.78
Oxide and Transition	Boin	13.9	0.299	10.5	10.9	1419	49	150	2.0	2.38
	Sewum	-	0.220	14.5	11.1	1588	53	150	2.0	2.47
	Nyam	9.5	0.066	13.8	5.5	1289	65	150	2.0	2.26

Ore Type & Deposit	Blend Year 4 %	Ai g	CWi kWh/t	BWi (106 µm) kWh/t	BWi F_{80} µm	BWi P_{80} µm	Axb	ta	Ore SG
Design (Calculated)	100	0.27	17.1	16.7	1903	73	47.2	0.77	2.71

13.6.2 Selected 80% Passing Grind Size

Selection of the leach grind size P_{80} has been based on the recovery vs. grind size information observed in the latest Intertek CIL testwork (2025-2026).

For oxide ore, leach performance is not sensitive to a grind size (P_{80}) of 53 µm or 75 µm. In contrast, leach performance for fresh ore is sensitive to grind size, as shown in Table 13.6.2. Gold recovery for fresh ore increased by approximately 3% to 9% when the grind size was reduced from a P_{80} of 75 µm to P_{80} of 53 µm. The milling circuit was therefore designed to be able to produce ore slurry at a P_{80} of 53 µm.

Table 13.6.2 Fresh Ore Leach Test Results - P_{80} of 53 µm vs. 75 µm

Zone	Test		Grind Size P_{80} µm	48 Hrs Recovery %	% Recovery Difference
Boin Fresh	GCIL-21	Grav-Leach (Kerosene)	53	89.4	2.8
	GCIL-23	Grav-Leach (Kerosene)	75	86.7	
Nyam Fresh	GCIL-01	Grav-Leach	53	91.4	9.0
	GCIL-25	Grav-Leach	75	82.4	
Sewum Fresh	GCIL-22	Grav-Leach (Kerosene)	53	90.2	3.1
	GCIL-24	Grav-Leach (Kerosene)	75	87.1	

13.6.3 Thickener Sizing

Static settling testwork indicated that the fresh tailings samples yielded unraked underflow densities of 53.2% to 65.8% solids (settling at 0.16–0.89 mm/s) with flocculant dosing up to 40 g/t, whereas the oxide tailings achieved only 39.8% to 48.0% solids (settling at 0.12–0.37 mm/s) with flocculant dosing up to 30 g/t. There are no pre-leach thickener testwork, therefore, the pre-leach thickener design is based on tails thickener testwork. A solids loading rate calculated from on the oxide tails thickener test results will be used as a sizing parameter as oxide is the more challenging ore type to settle. For a targeted 45% solids in the pre-leach thickener underflow, a calculated solids loading rate of 0.21 t/m^2h was used.

13.6.4 Major Reagents Consumption

Reagents consumption for the Project has been estimated based on the consumption observed in the latest Intertek CIL testwork (2025-2026) for oxide, transition and fresh ore. The following cyanide and lime consumptions were estimated:

- Cyanide:
  - 0.83 kg/t for diggable oxide and transition ore at grind size P_{80} of 75 µm.
  - 1.00 kg/t for diggable oxide and transition ore at grind size P_{80} of 53 µm.
  - 1.49 kg/t for blast transition and fresh ore at grind size P_{80} of 53 µm.
  - A 50% reduction was applied to the testwork cyanide consumption values based on a benchmarking exercise conducted in agreement with Newcore and feedback indicating that the laboratory may have overdosed cyanide during the testwork.
- Lime:
  - 2.61 kg/t for diggable oxide and transition ore at grind size P_{80} of 75 µm.
  - 2.54 kg/t for diggable oxide and transition ore at grind size P_{80} of 53 µm.
  - 1.41 kg/t for blast transition and fresh ore at grind size P_{80} of 53 µm.

13.6.5 Gold Recovery Estimation

Table 13.6.3 presents the optimized gold extraction values observed in the latest Intertek CIL testwork (2025-2026).

Table 13.6.3 Gold Extraction for CIL Process

Zone	Gold Extraction %		
	Oxide	Transition	Fresh
Sewum	96.0	97.7	90.2
Boin	89.0	96.6	89.4
Nyam	90.9	91.4	91.4
Kwakyekrom*	90.9	91.4	91.4

*Kwakyekrom had no testwork data, hence, the same recoveries as the Nyam deposit were assumed due to the proximity of the deposits.

The gold losses are based on the predicted solution and carbon losses as per Table 13.6.4.

Table 13.6.4 Estimated Gold Losses

Parameter	Units	Oxide Diggable	Transition Diggable	Transition Blast	Fresh Blast	Source
		Tails Solution Gold Losses				
Design Tails Grade	g Au/m³	0.015	0.015	0.015	0.015	PDC
Predicted Actual Tails Grade	g Au/m³	0.01	0.01	0.01	0.01	Newcore/Fuse
Tails Pulp Density	% w/w	43.21	43.21	47.77	47.77	PDC
Tails Solution S.G.	-	1.0	1.0	1.0	1.0	PDC
Solution Losses (Equivalent g Au/t ROM feed)	g Au/t	0.013	0.013	0.011	0.011	Calculated
		Carbon Gold Losses				
Carbon Losses (Equivalent g Au/t ROM feed)	g Au/t	0.003	0.003	0.003	0.003	PDC/Calculated

Accounting for solution and carbon losses, an estimation of the gold recovery for each ore type and deposit is presented in Table 13.6.5.

Table 13.6.5 Estimated Gold Recovery

Zone	Gold Recovery % (after losses)			
	Oxide Diggable	Transition Diggable	Transition Blast	Fresh Blast
Sewum	92.1	94.3	95.0	88.0
Boin	86.9	94.5	94.8	87.9
Nyam	88.6	89.1	89.1	89.7
Kwakyekrom	87.6	88.3	89.4	90.0

13.7 Metallurgical Conclusions

The following conclusions can be drawn from the metallurgical and comminution testwork during the PFS phase:

Comminution Circuit:

- Comminution circuit design parameters were developed using testwork results supplemented by benchmark data where necessary. Design values were based on the 85th percentile of BWi and CWi results to provide a conservative basis for sizing and power estimation.

- The milling circuit design is based on the representative mine plan blend containing the highest proportion of fresh rock, resulting in a design ore with a BWi of 16.7 kWh/t and CWi of 17.1 kWh/t.

- Oxide and transition ores exhibit significantly softer grinding characteristics than fresh ores and were used as the basis for mineral sizer design, while fresh ores governed the primary crushing and grinding circuit design.

Leach Circuit:

- CIL testwork demonstrated that oxide ore gold recovery is relatively insensitive to grind size between P_{80} 53 µm and 75 µm, whereas fresh ore recovery improves substantially at finer grind sizes.

- Fresh ore gold recoveries increased by approximately 3% to 9% when grind size was reduced from P_{80} 75 µm to P_{80} 53 µm. Accordingly, the processing plant was designed to achieve a target grind size of P_{80} 53 µm to maximize overall gold recovery for the fresh ore.

Thickener Sizing:

- Thickener sizing was based on tailings static settling testwork, with oxide material identified as the more challenging ore type for settling performance.

- A solids loading rate of 0.21 $t/m^2/h$ was selected for pre-leach thickener design to achieve a target underflow density of approximately 45% solids.

Major Reagent Consumptions:

- Cyanide consumption is expected to range from 0.83 to 1.49 kg/t, with higher consumption associated with transition and fresh ores treated at the finer P_{80} 53 µm grind size.

- Cyanide consumption values used for design were reduced by 50% from laboratory testwork results based on benchmarking and the assessment that cyanide may have been overdosed during testwork.

- Lime consumption is expected to range from 1.41 to 2.61 kg/t, with lower consumption observed for fresh ore relative to oxide and transition ores.

Estimated Gold Recoveries:

- Estimated overall gold recoveries after accounting for solution and carbon losses range from 86.9% to 95.0%, depending on ore type and deposit.

13.8 Future Testwork Recommendations

The following testwork is recommended to progress the design in the FS phase:

Sample Requirement:

- A set quantity of master composites, variability samples and some bulk samples for each mineralization type using interval composite samples from drill cores. Number of representative samples to be determined by Newcore based on the mine plan.

Comminution Testwork:

- Additional comminution testwork is recommended for ore types and deposits where design parameters were inferred rather than determined through direct testing, to further improve confidence in process design and equipment sizing.
- Specifically, additional SMC testwork for BWi, Ai, A x b for diggable transition and blast transition ore is recommended to improve the comminution database and support further optimization of milling circuit design and power requirements.

Ore Characterization Testwork:

- Full head assays for Au, Ag, As, Fe, Sb, Hg, Te, Cu, CuCNSol, Ctotal, Corg, CO32-, Stotal, Ssulphide, and multi-element ICP Scan for master and variability samples.
- Additional mineralogy testwork to further assess gold deportment in each mineralization type.
- Bulk materials handling testwork to confirm bulk density, angle of repose, angle of surcharge, and other bulk solids handling properties.

Gravity-Leach Testwork:

- Additional extended gravity separation testwork along with gravity circuit modelling to confirm gravity concentrator selection.
- Additional leach testwork on master composites to further optimize leach conditions.
- Additional variability samples to further assess variation in gold recovery across different orebody.
- Carbon kinetic tests to generate data on carbon loading capacity and kinetics for use in the CIL modelling or elution plant sizing.

- Oxygen uptake rate tests to simulate oxygen consumption in the leaching process for proper oxygen plant sizing.
- Gravity-leach testing on bulk samples to confirm flowsheet at optimum conditions at a larger scale.

Settling Testwork:

- Dynamic settling testwork on master composites and bulk samples to confirm pre-leach thickener sizing under continuous mode.
- Rheology testwork to determine critical slurry viscosity and settling rate for pump design.

Cyanide Detoxification Testwork:

- Conduct cyanide detox tests on both tails slurry and tails solution to generate data for a detox circuit or water treatment plant design.

Water Testwork:

- Conduct water analysis for site water samples (include pH, S.G.).
- Buffer curve determination (pH vs. lime addition).

Tailings Testwork:

- Geochemical and geotechnical testwork to characterize the tailings for the TSF design.
- Geochemical testing typically include:
  - Acid-base accounting (ABA).
  - Net acid generation (NAG) testing.
  - Shake flask extraction testing.
  - Humidity cell testing for long-term leaching and drainage characterization.
- Geotechnical testing typically include:
  - Particle size distribution (sieve and hydrometer analysis).
  - Specific gravity determination.

- Atterberg limits.
- Standard and Modified Proctor compaction tests.
- Consolidation testing (oedometer and constant rate of strain).
- Soil-moisture characteristic testing.
- Shear strength testing (triaxial and direct shear).
- Permeability testing.

The cost of a full FS testwork program is estimated to range from $1.0 million to $1.5 million, depending on the number of samples collected and the scope of testing required to support a feasibility design.

14.0 MINERAL RESOURCE ESTIMATES

14.1 Mineral Resource Estimate Definition and Procedure

The current mineral resource estimate for the Project has been prepared following the CIM standards and definitions, as required under NI 43-101 regulations. The standards and definitions are as follows:

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

"A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction."

"The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling. "

"Material of economic interest refers to diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals."

"The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of Modifying Factors. The phrase 'reasonable prospects for eventual economic extraction' implies a judgment by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. The Qualified Person should consider and clearly state the basis for determining that the material has reasonable prospects for eventual economic extraction. Assumptions should include estimates of cut-off grade and geological continuity at the selected cut- off, metallurgical recovery, smelter payments, commodity price or product value, mining and processing method and mining, processing and general and administrative costs. The Qualified Person should state if the assessment is based on any direct evidence and testing."

"Interpretation of the word 'eventual' in this context may vary depending on the commodity or mineral involved. For example, for some coal, iron, potash deposits and other bulk minerals or commodities, it may be reasonable to envisage 'eventual economic extraction' as covering time periods in excess of 50 years. However, for many gold deposits, application of the concept would normally be restricted to perhaps 10 to 15 years, and frequently to much shorter periods of time."

14.1.1 Measured Mineral Resource

"A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are 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."

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

14.1.2 Indicated Mineral Resource

"An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics 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."

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

14.1.3 Inferred Mineral Resource

"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."

"An Inferred Mineral Resource is based on limited information and sampling gathered through appropriate sampling techniques from locations such as outcrops, trenches, pits, workings and drill holes. Inferred Mineral Resources must not be included in the economic analysis, production schedules, or estimated mine life in publicly disclosed Pre-Feasibility or Feasibility Studies, or in the Life of Mine plans and cash flow models of developed mines. Inferred Mineral Resources can only be used in economic studies as provided under NI 43-101."

"There may be circumstances, where appropriate sampling, testing, and other measurements are sufficient to demonstrate data integrity, geological and grade / quality continuity of a Measured or Indicated Mineral Resource, however, quality assurance and quality control, or other information may not meet all industry norms for the disclosure of an Indicated or Measured Mineral Resource. Under these circumstances, it may be reasonable for the Qualified Person to report an Inferred Mineral Resource if the Qualified Person has taken steps to verify the information meets the requirements of an Inferred Mineral Resource."

14.2 General Description

Four block models were used to compile the mineral resource statement presented in this Report, including separate models for each of the Boin, Sewum, Nyam and Kwakyekrom deposits. The resource estimates were prepared by three QPs all of DRA Americas Inc., as follows: Boin – Schadrac Ibrango, P.Geo.; Sewum – Matthew Halliday, P.Geo., and; Nyam and Kwakyekrom – Ryan Wilson, P.Geo.

14.3 Supporting Data

14.3.1 Drillhole Database

The Newcore technical team provided the diamond drillhole database used by DRA to estimate the Mineral Resources reported herein for the Project. Further information regarding the database and its verification can be found in Section 12 of this Report.

14.3.2 Topography

The topographic surface used in the Mineral Resource Estimate is a Digital Terrain Model (DTM) based on drone surveys completed at Sewum, Boin, Nyam and Kwakyekrom, as disclosed in Section 9. The surveys were completed in two phases, with smaller sections in the northeast (Nyam), south-central (Sewum) and northwest (part of Boin) initially surveyed by photogrammetry and followed by a LiDAR survey to cover and connect the central portion of the property.

Subsequent to the completion of the Mineral Resource Update, it was determined that three separate geoid models were employed during the compilation of the topographic surveys. Boin and Kwakywkrom are consistent with project datum WGS 84 Zone 30N Orthometric, while topographic surfaces and all derived information at Sewum and Nyam require -43.2 and -24.5m adjustments to elevation.

14.3.3 Bulk Density

Specific gravity (SG) data was collected on site in a dedicated room within the Newcore field camp. A total of 5,395 samples were measured for specific gravity. A summary of the SG results is presented in Table 14.3.1.

Drill logs recorded the weathering profile from W1 (fresh) to W6 (strongly weathered). Surface models were generated in MinePlan 3D using data from drill logs for the base of oxidized zones (W6, W5 and W4) and base of transition (W2 and W3) to fresh zones (W1) for the four deposits.

Diamond drill core samples were selected for SG measurement from the Sewum, Boin, Nyam and Kwakyekrom deposits for inclusion in the mineral resource estimation. SG measurements were also collected from the Eradi prospect. A summary of the drill holes and number of SG samples collected is presented in Table 14.3.1.

A total of 5,395 density values were supplied. DRA requested 160 rechecks and 33 new samples. After reviewing and removing outliers, the final dataset consists of 5,419 samples across all zones.

Table 14.3.1 Summary of Bulk Density Data by Target Area

Deposit	Holes	Samples
Sewum	85	2,532
Boin	78	1,755
Nyam	40	715
Kwakyekrom	3	60
Eradi	20	357

SG samples were selected at regular intervals through the weathered, transition and fresh zones of diamond drill holes. Seventy-five percent (75%) of the samples are spaced between 0 and 7.5 m apart.

Low porosity high competence samples were measured using direct immersion. These samples were weighed in air and then weighed while submersed within a bucket of water. The measurements were completed using a Mettler – Toledo Gold Balance JE3002GE. SG values were calculated using the following formula:

$$\rho_b = \frac{m_d}{\frac{m_d - m_s}{\rho_{water}}}$$

ρ_b = bulk density [g/cm^3]
ρ_{water} = set to 1 g/cm^3
m_d = dry weight
m_s = submersed weight

High porosity or low competence samples were measured by adding a film layer to the sample before immersion. These samples were weighed in air before file, in air after film applied and during submersion within a bucket of water.

$$\rho_b = \frac{m_d}{\frac{m_{dc} - m_{sc}}{\rho_{water}} - \frac{m_{dc} - m_d}{\rho_{coating}}}$$

ρ_b = bulk density [g/cm^3]
ρ_{water} = set to 1 g/cm^3
$\rho_{coating}$ = set to 0.91 g/cm^3
m_d = dry weight
m_{dc} = dry weight coated
m_{sc} = submersed weight coated

Due to the differences in the different deposits at Enchi – densities were assigned by zone. Where there was less information approximates were used based on similarities to surrounding zones.

Table 14.3.2 Summary of Selected Densities Used for Mineralization by Deposit

Zone	Domain	Type	Oxidation	Density g/cm^3
Boin	All	Mineralized Zone	Fresh	2.63
Boin	All	Mineralized Zone	Transition	1.93
Boin	All	Mineralized Zone	Oxide	1.64

Zone	Domain	Type	Oxidation	Density g/cm^3
Kwakyekrom	All	Mineralized Zone	Fresh	2.63
Kwakyekrom	All	Mineralized Zone	Transition	2.38
Kwakyekrom	All	Mineralized Zone	Oxide	1.85
Nyam	All	Mineralized Zone	Fresh	2.63
Nyam	All	Mineralized Zone	Transition	2.38
Nyam	All	Mineralized Zone	Oxide	1.85
Sewum	Checkerboard	Mineralized Zone	Fresh	2.69
Sewum	Checkerboard	Mineralized Zone	Transition	1.86
Sewum	Checkerboard	Mineralized Zone	Oxide	1.53
Sewum	Sewum South	Mineralized Zone	Fresh	2.48
Sewum	Sewum South	Mineralized Zone	Transition	2.27
Sewum	Sewum South	Mineralized Zone	Oxide	2.15
Sewum	Sewum North	Mineralized Zone	Fresh	2.74
Sewum	Sewum North	Mineralized Zone	Transition	2.07
Sewum	Sewum North	Mineralized Zone	Oxide	1.54

Table 14.3.3 Summary of Selected Densities Used for Waste Material by Deposit

Zone	Domain	Type	Oxidation	Density (g/cm^3)
Boin	All	Waste	Fresh	2.63
Boin	All	Waste	Transition	1.93
Boin	All	Waste	Oxide	1.59
Kwakyekrom	All	Waste	Fresh	2.56
Kwakyekrom	All	Waste	Transition	2.00
Kwakyekrom	All	Waste	Oxide	1.77
Nyam	All	Waste	Fresh	2.56
Nyam	All	Waste	Transition	2.00
Nyam	All	Waste	Oxide	1.77
Sewum	Checkerboard	Waste	Fresh	2.57
Sewum	Checkerboard	Waste	Transition	1.81
Sewum	Checkerboard	Waste	Oxide	1.46
Sewum	Sewum South	Waste	Fresh	2.45
Sewum	Sewum South	Waste	Transition	2.15
Sewum	Sewum South	Waste	Oxide	1.63

Zone	Domain	Type	Oxidation	Density (g/cm^3)
Sewum	Sewum North	Waste	Fresh	2.74
Sewum	Sewum North	Waste	Transition	2.05
Sewum	Sewum North	Waste	Oxide	1.45

14.4 Boin Deposit

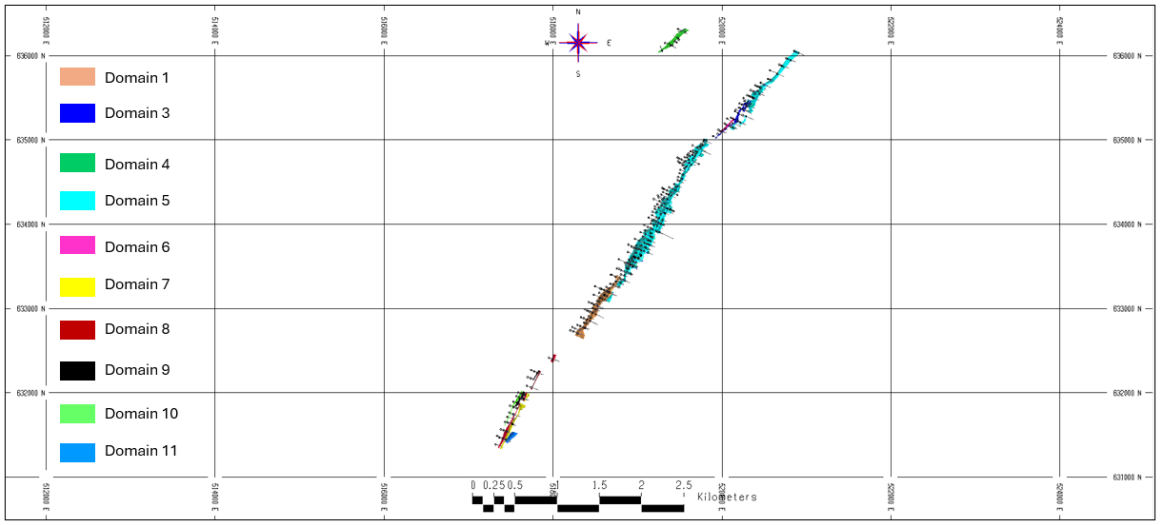
14.4.1 Three-Dimensional (3D) Modelling

Newcore provided DRA with an initial set of wireframes for a total of 12 zones of mineralized domains at the Project. These mineralized domains were all generated in Leapfrog Geo. Following review of the approach and methodology used to produce these wireframes, DRA conducted an independent review of the interpreted zones in both 2D and 3D.

A new set of wireframes was generated by DRA using an explicit approach in MinePlan 3D software. The modelling was dominantly driven by lithological and grade controls with additional considerations for grade continuity, specifically in areas with intersecting mineralization types (i.e., shallow-dipping, low-grade mantos and high-grade subvertical structures).

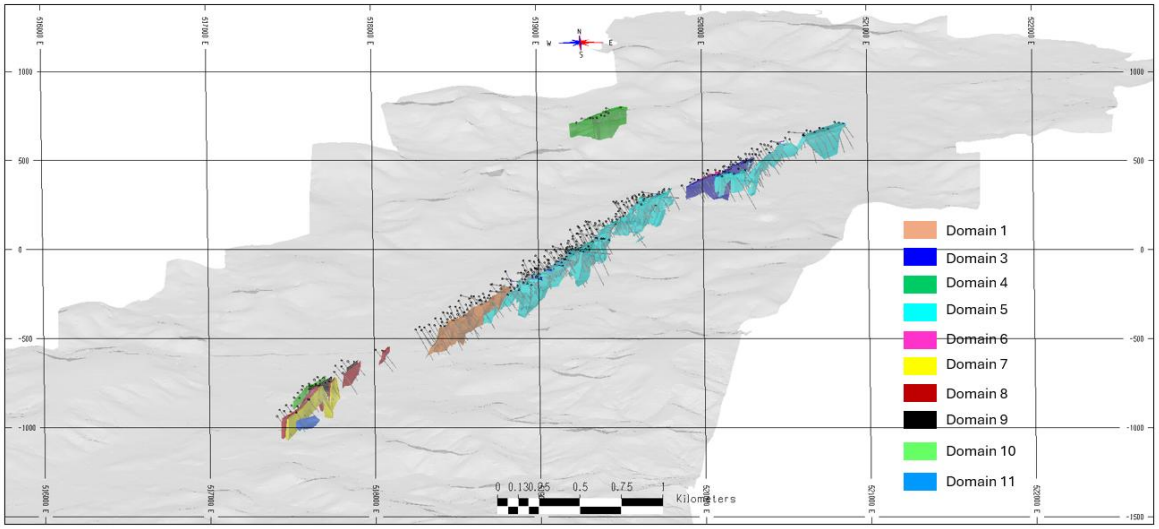
The new mineralized domains present similar strikes compared to the initial set with better volumetric controls with respect to the drilling distribution across the different mineralized domains. Changes were discussed with Newcore in order to finalize the set of wireframes used in the subsequent mineral resource estimation (Figure 14.4.1 and Figure 14.4.2)

Figure 14.4.1 3D Plan View (Looking Down) of Boin Mineralized Domains



Source: DRA, 2026

Figure 14.4.2 3D Orthographic View of Boin Mineralized Domains



Source: DRA, 2026

14.4.2 Descriptive Statistics

Data was flagged according to the modelled wireframes, then statistically analyzed to determine the underlying data distributions and to help with eventual comparisons to composited and estimated block grade data for reconciliation purposes. The data was imported into the MSDA software package for statistical and geostatistical computations.

It is noted that relatively rare instances of unsampled intervals within the solids, mostly related to internal waste, were replaced with zero values as grades. Moreover, missing values for certain intervals (partial analyses, different drill campaigns, etc.) were assigned a value using the average of the attribute as correlation coefficients between elements were too weak to consider linear regression.

Basic descriptive statistics were then calculated for the raw data samples contained within each of the mineralized envelopes (i.e., zone intercepts) at the Boin Deposit. These results are summarized by zone below in Table 14.4.1.

Table 14.4.1 Summary of Basic Descriptive Statistics for Raw Data Samples by Mineralized Domain, Boin Deposit

Domain	Variable	Count	Arithmetic Mean	Weighted Mean	Standard Error	Median	Mode	Standard Deviation	Variance	CoV	Kurtosis	Skewness	Minimum	Maximum
1	Au (g/t)	1,194	0.59	0.58	0.03	0.26	0.10	0.95	0.90	1.61	33.74	4.63	0.01	11.98
3	Au (g/t)	421	0.50	0.49	0.06	0.17	0.01	1.24	1.53	2.50	76.63	7.66	0.01	16.00
4	Au (g/t)	482	0.32	0.34	0.04	0.06	0.01	0.82	0.68	2.54	52.68	6.33	0.00	9.14
5	Au (g/t)	9,593	0.66	0.66	0.02	0.22	0.01	1.98	3.91	3.00	2192.73	35.61	0.00	133.47
6	Au (g/t)	99	0.21	0.21	0.03	0.11	0.02	0.31	0.09	1.47	9.88	2.93	0.01	1.84
7	Au (g/t)	315	0.42	0.42	0.04	0.24	0.01	0.63	0.4	1.49	47.50	5.43	0.01	7.30
8	Au (g/t)	288	0.36	0.35	0.03	0.17	0.10	0.56	0.31	1.56	26.29	4.46	0.01	4.67
9	Au (g/t)	134	0.48	0.48	0.06	0.24	0.12	0.68	0.47	1.41	11.63	3.19	0.02	4.38
10	Au (g/t)	51	0.29	0.28	0.04	0.17	0.11	0.28	0.08	0.98	3.26	1.82	0.02	1.32
11	Au (g/t)	93	0.22	0.22	0.02	0.16	0.14	0.16	0.03	0.73	3.20	1.65	0.02	0.87

14.4.3 Grade Capping

Grade capping is used to limit the spatial extrapolation of occasional isolated high grades in the resource model estimates. Capping analyses undertaken include the use of histograms, log probability plots and ranked composites (outlier analysis).

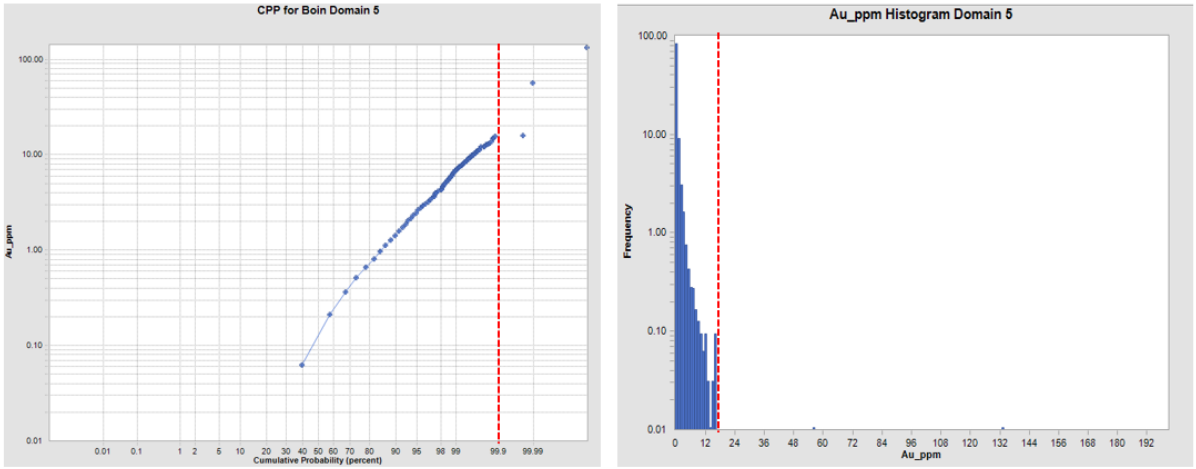
Histograms were used to search for distinct breaks in grade distributions in tandem with log probability plots, which generally show clear inflection points at the selected capping value. Outliers were also examined by means of ranked composites and observing the effect of sequential capping on the coefficient of variation (CoV) of the remaining data.

The thorough analysis of the sample dataset broken down by mineralized domain led to the conclusion that grade capping is only required for Domain 5 where outlier gold grade values were reported.

Table 14.4.2 Summary of Capping Grades for Domain 5, Boin

Domain	Variable	Uncapped Mean	Uncapped CoV	Capping Grade	Capped Comps #	Capped Comps %	Capped Mean	Capped CoV
5	Au (g/t)	0.66	3.00	16	2	0.02	0.64	2.08

Figure 14.4.3 Histogram and Log Probability Plots, Domain 5



Source: DRA, 2026

14.4.4 Compositing

Drill hole intercepts through the interpreted mineralized domains at the Project were composited to 2 m fixed length intervals. Intervals shorter than 0.7 m were removed from the final composites set used to run the resource interpolation. The selected composite length was based on the nuggety behaviour of the Boin mineralization. Although the sampling length histogram shows 1 m as the mode of the sampling length, at Boin it was found more appropriate to use 2 m as a compositing length to help mitigate the impact of isolated high-grade gold clusters in the estimation process. It is also supported by the difficulty encountered to define stable variograms with 1 m composites in the 3D-space during the geostatistical analysis. Basic descriptive statistics for the composited data contained within wireframes (i.e., zone intercepts) are summarized by zone below in Table 14.4.3.

Table 14.4.3 Summary of Basic Descriptive Statistics for 2 m Composited Data at the Boin Deposit

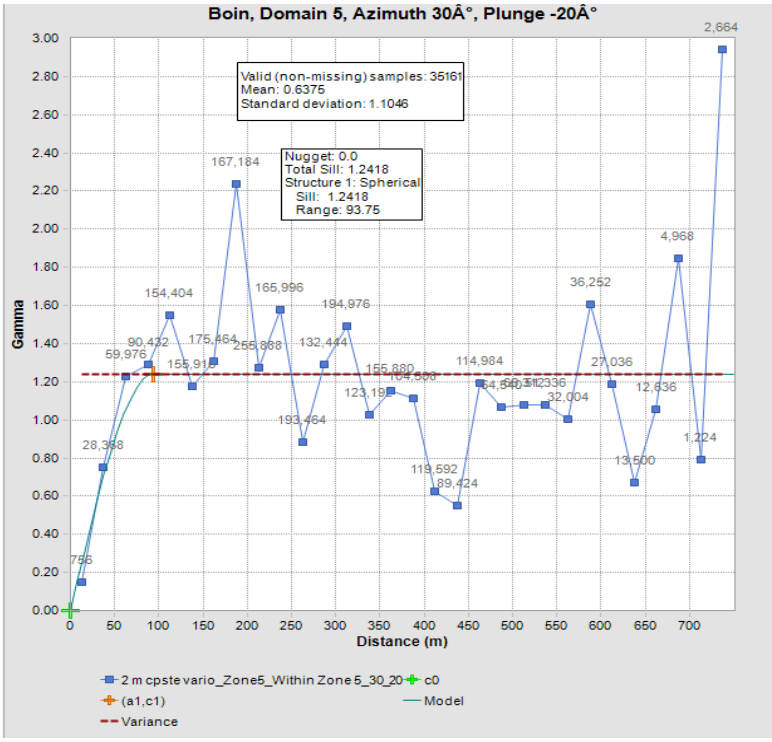
Domain	Variable	Count	Arithmetic Mean	Standard Error	Median	Mode	Standard Deviation	Variance	CoV	Kurtosis	Skewness	Minimum	Maximum
1	Au (g/t)	622	0.58	0.03	0.29	0.11	0.83	0.68	1.43	38.05	4.72	0.01	9.47
3	Au (g/t)	215	0.49	0.07	0.20	0.09	1.01	1.02	2.05	44.30	5.97	0.01	9.61
4	Au (g/t)	262	0.34	0.04	0.12	0.01	0.66	0.43	1.95	19.62	4.10	0.01	4.60
5	Au (g/t)	5,018	0.64	0.02	0.25	0.03	1.11	1.22	1.73	35.81	4.81	0.01	16.00
6	Au (g/t)	52	0.24	0.05	0.11	0.11	0.36	0.13	1.46	8.87	2.88	0.01	1.84
7	Au (g/t)	177	0.42	0.04	0.25	0.23	0.51	0.26	1.21	17.46	3.49	0.01	4.05
8	Au (g/t)	161	0.35	0.04	0.19	0.11	0.46	0.21	1.32	11.78	3.20	0.01	2.91
9	Au (g/t)	70	0.48	0.07	0.27	0.27	0.60	0.36	1.25	8.84	2.85	0.06	3.12
10	Au (g/t)	28	0.28	0.04	0.22	0.08	0.22	0.05	0.80	3.58	1.64	0.02	1.04
11	Au (g/t)	47	0.23	0.02	0.20	0.30	0.13	0.02	0.57	1.18	1.16	0.05	0.63

14.4.5 Variography

Variography aims to assess the spatial continuity of grade for an element of interest and ultimately helps guide the definition of parameters for the interpolation of Mineral Resources. The selected approach, inverse distance cubed weighting (IDW^3), is a linear geostatistical estimation method that requires input parameters to limit the size of the search neighbourhood (via a defined search ellipsoid) for each point to be interpolated within the block model.

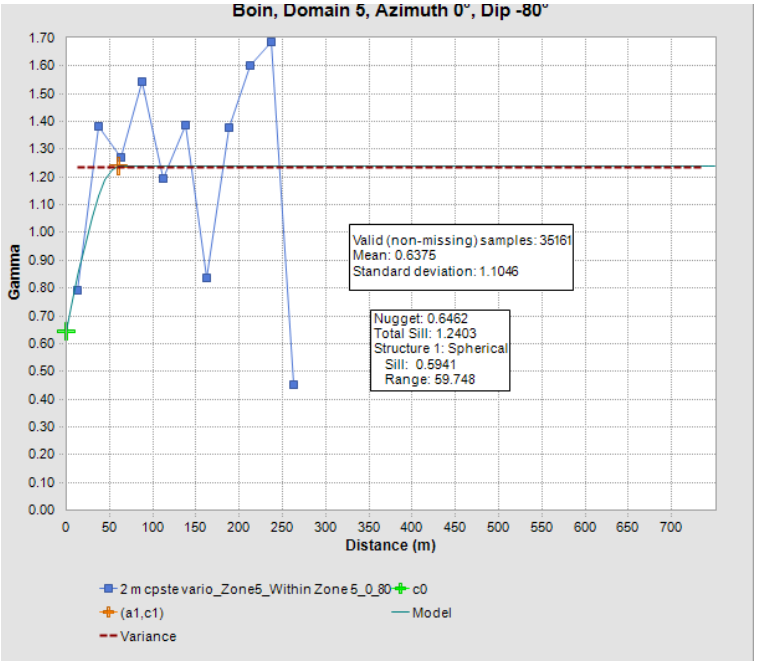
Downhole and directional variography for the Project were run also using MSDA software. Relevant structure with stable variograms were only found for Domain 5 being the larger deposit at Boin. Parameters defined in the variography analysis were used in conjunction with geological information, the QP experience for similar deposits and other statistical factors such as average drill spacing to select the interpolation parameters. Figure 14.4.4 and Figure 14.4.5 present the best variograms defined at Boin for Domain 5 for the major and vertical axis. Combined Downhole variogram was generated and used as a proxy for the minor axis (Figure 14.4.6).

Figure 14.4.4 Variogram on the Major Axis (Domain 5)



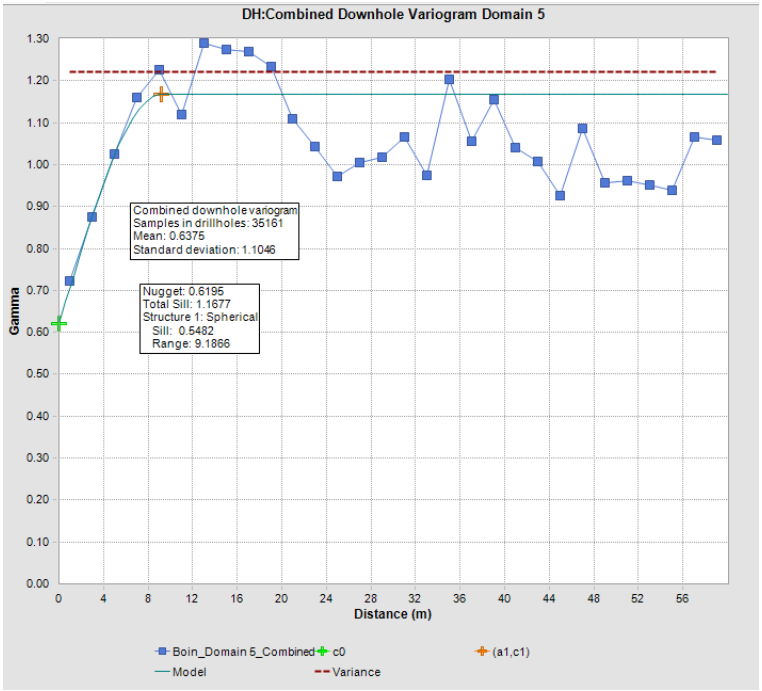
Source: DRA, 2026

Figure 14.4.5 Variogram on the Vertical Axis (Domain 5)



Source: DRA, 2026

Figure 14.4.6 Variogram on the Vertical Axis (Domain 5)



Source: DRA, 2026

14.4.6 Mineral Resource Estimate

Following wireframing to constrain the resource estimation domains with an explicit modelling approach, exploratory data analysis and geostatistical assessments were performed using MSDA software. The block model was then constructed with subsequent grade and tonnage estimates computed in MinePlan 3D.

Block Model

A single block model was constructed for the mineralized domains at Boin to capture a variety of data types including the relevant mineralized domain codes, estimated block grades, density values, percentage of material beneath the topographic surface, the closest / furthest / average distances between informing composites, initial resource category as determined by multiple-pass interpolation and finalized resource category as determined by the resource modeller.

Sub-blocking was used to better define narrow portions of the block model and to respect the volumetrics of the modelled mineralized domains and triangulations. The model was rotated at 30° to align with the strike of the gold mineralization. Relevant block model definition parameters are summarized in Table 14.4.4. On the resource interpolation completed the parent sub-blocked model was standardized into a regularized block model of 5 m x 5 m x 5 m for mine engineering purposes.

Table 14.4.4 Block Model Definition Parameters, Boin

Description	Value
Model Dimension X (m)	8,000
Model Dimension Y (m)	2,500
Model Dimension Z (m)	340
Origin X (Easting)	517,360.31
Origin Y (Northing)	630,558.43
Origin Z (Lower Elev.)	-170
Rotation (º)	30
Block Size X (m)	10
Sub-block Size X (m)	5
Block Size Y (m)	5
Sub-block Size Y (m)	2.5
Block Size Z (m)	10
Sub-block Size Z (m)	5

Search Strategy and Interpolation

Block values were estimated for each individual mineralized domain using the generated composites and the inverse distance cubed weighting (IDW^3) method. The set of search parameters used in the multi-pass interpolations, derived mainly from geological information and interpreted continuity with support from variography and statistical factors such as average drill hole spacing, are summarized by estimation domain in Table 14.4.5.

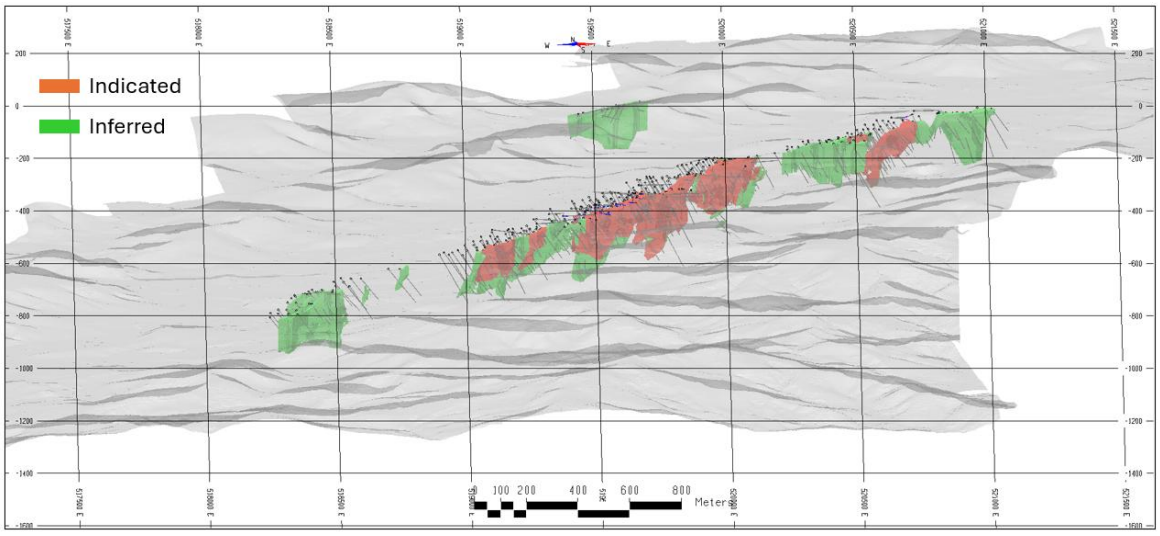
Table 14.4.5 Inverse Distance Weighted (IDW^3) Interpolation Parameters Summary for the Boin Deposit Block Model

Domain	Pass	Estimation Method	Min Samples	Max Samples	Max. Samples / DDH	Major Axis	Semi Axis	Vert Axis
Domain 1	1	IDW^3	7	15	3	115	10	60
	2	IDW^3	5	15	2	150	15	80
	3	IDW^3	2	15	2	275	30	275
	Strike= 35°, Plunge = 0°, Dip = -80°							
Domain 3 and 4	1	IDW^3	7	15	3	115	10	60
	2	IDW^3	5	15	2	150	15	80
	3	IDW^3	2	15	2	300	40	300
	Strike= 50°, Plunge = 0°, Dip = -90°							
Domain 5	1	IDW^3	7	15	3	115	10	60
	2	IDW^3	5	15	2	150	15	80
	3	IDW^3	2	15	2	300	40	300
	Strike= 30°, Plunge = -20°, Dip = -70°							
Domain 6	1	IDW^3	7	15	3	115	10	60
	2	IDW^3	5	15	2	150	15	80
	3	IDW^3	2	15	2	275	40	275
	Strike= 35°, Plunge = 0°, Dip = -90°							
Domain 7 to 10	1	IDW^3	7	15	3	115	10	60
	2	IDW^3	5	15	2	150	15	80
	3	IDW^3	2	15	2	250	40	250
	Strike= 20°, Plunge = 0°, Dip = -85°							
Domain 11	1	IDW^3	7	15	3	115	10	60
	2	IDW^3	5	15	2	150	15	80
	3	IDW^3	2	15	2	200	40	200
	Strike= 30°, Plunge = -20°, Dip = -70°							

Mineral Resource Classification

The Mineral Resources reported herein for the Boin Deposit have been classified into Indicated and Inferred categories. This classification is based on the interpreted geological and grade continuity of the observed mineralization. Primary categorization was based on multiple-pass IDW^3 interpolation which employed increasing search ellipsoid ranges (refer back to Table 14.4.5). Categorical smoothing was manually employed to create a more coherent and geological consistent classification scheme. A 3D orthographic view of the final block classification is provided in Figure 14.4.7.

Figure 14.4.7 3D Orthographic View of the Final Block Classification for Boin



Source: DRA, 2026

14.4.7 Mineral Resource Statement

The Mineral Resource Estimate statement for the Boin deposit prepared by DRA is summarized in Table 4.4.6. Additional details on mining and processing modifying factors are also provided in the corresponding footnotes. The Resources have been reported using a constraining resource pit at a gold price of $3,200/oz.

Table 14.4.6 Mineral Resource Estimate – Effective Date of October 6, 2025

Zone	Classification	Tonnes '000	Au Grade g/t	Contained Au ounces
Boin	Indicated	23,477	0.73	550,000
	Inferred	9,237	0.60	178,000

Notes for Mineral Resource Estimate:

1. Canadian Institute of Mining Metallurgy and Petroleum (CIM) definition standards were followed for the resource estimate.

2. The effective date of the Resource is 6 October 2025.
3. All figures are rounded to reflect the relative accuracy of the estimate and numbers may not add due to rounding.
4. The resource model used Inverse Distance Weighting cubed (IDW^3) grade estimation within a three-dimensional block model with mineralized zones defined by wireframed solids and constrained by a pit shell. Validations were completed using Ordinary Kriging (OK).
5. Open pit cut-off grades varied from 0.1 to 0.2 g/t Au based on mining and processing costs as well as the recoveries in different weathered material.
6. A $3,200/ounce gold price was used to determine the cut-off grade.
7. Metallurgical recovery of 85% was applied to oxide and transition mineralization, and 91.7% for fresh mineralization.
8. The pit optimization considered the following costs: mining cost based on mineralization type of $1.97/tonne for oxide, $2.62/tonne for transition, and $3.15/tonne for fresh; waste mining costs of $1.64/tonne for oxide, $2.34/tonne for transition, and $2.87/tonne for fresh; processing and G&A costs assumed of $8.74/tonne for oxide, $8.49/tonne for transition, and $19.29/tonne for fresh.
9. Average densities of mineralized material varied between 1.53 and 2.15 g/cm^3 for oxide, 1.86 and 2.38 g/cm^3 for transition, and 2.48 and 2.74 g/cm^3 for fresh rock. Average densities of waste rock varied between 1.45 and 1.77 g/cm^3 for oxide, 1.81 and 2.15 g/cm^3 for transition, and 2.45 and 2.74 g/cm^3 for fresh rock
10. Optimization pit slope angles varied by deposit and mineralized area, with an overall strip ratio including all pits of 3.35.
11. Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have economic viability.
12. The resource estimate was prepared by Schadrac Ibrango, P.Geo of DRA Americas Inc. in accordance with NI 43-101. This individual is an independent qualified person (QP) as defined by NI 43-101.
13. As of the Report's date, the QPs, to the best of their knowledge, are not aware of any metallurgical, environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other risk factors that might materially affect the estimate of Mineral Resources.

14.4.8 Block Model Validation

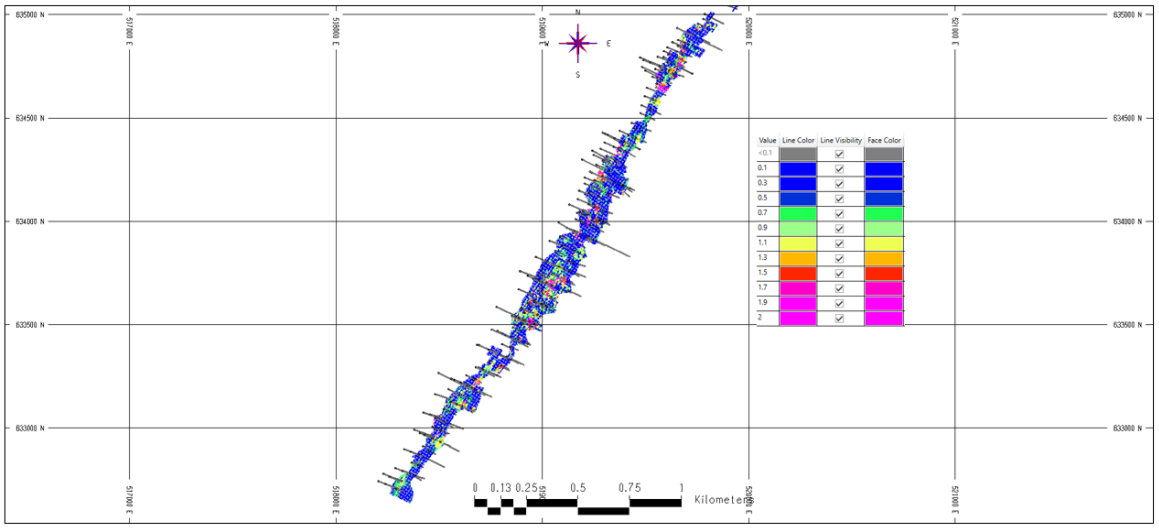
The current Project block model has been validated by DRA using a combination of visual inspection and statistical comparisons, including:

- Visual comparison of assays and block grades.
- Inspection of generated swath plots.
- Alternative interpolation methods.

Visual Inspection

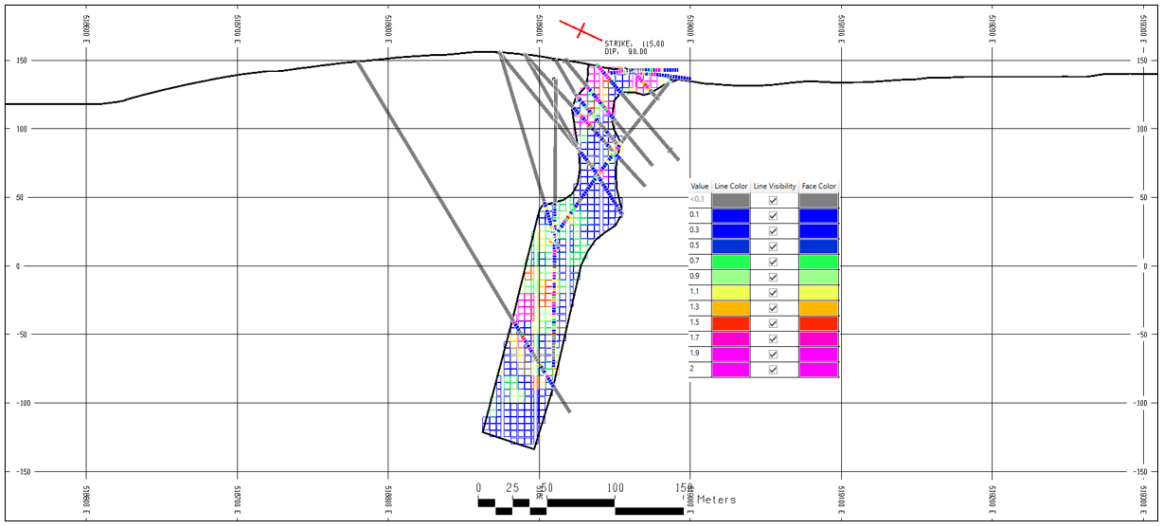
Estimated blocks and drill hole intercepts at Enchi were reviewed both on 2D vertical sections and level plans, as well as interactively within the MinePlan 3D software environment. The block grades suitably respect assay grades throughout the deposit. A plan view of Au (g/t) with assay and block grades superimposed is presented in Figure 14.4.8. A representative north-south vertical section through the core of the deposit is shown in Figure 14.4.9.

Figure 14.4.8 Plan View of Au (g/t) at Boin (Domain 5) with Assay and Block Grades



Source: DRA, 2026

Figure 14.4.9 Comparison of Assay and Block Grades for the Boin Deposit on Representative Vertical Section (2610N)

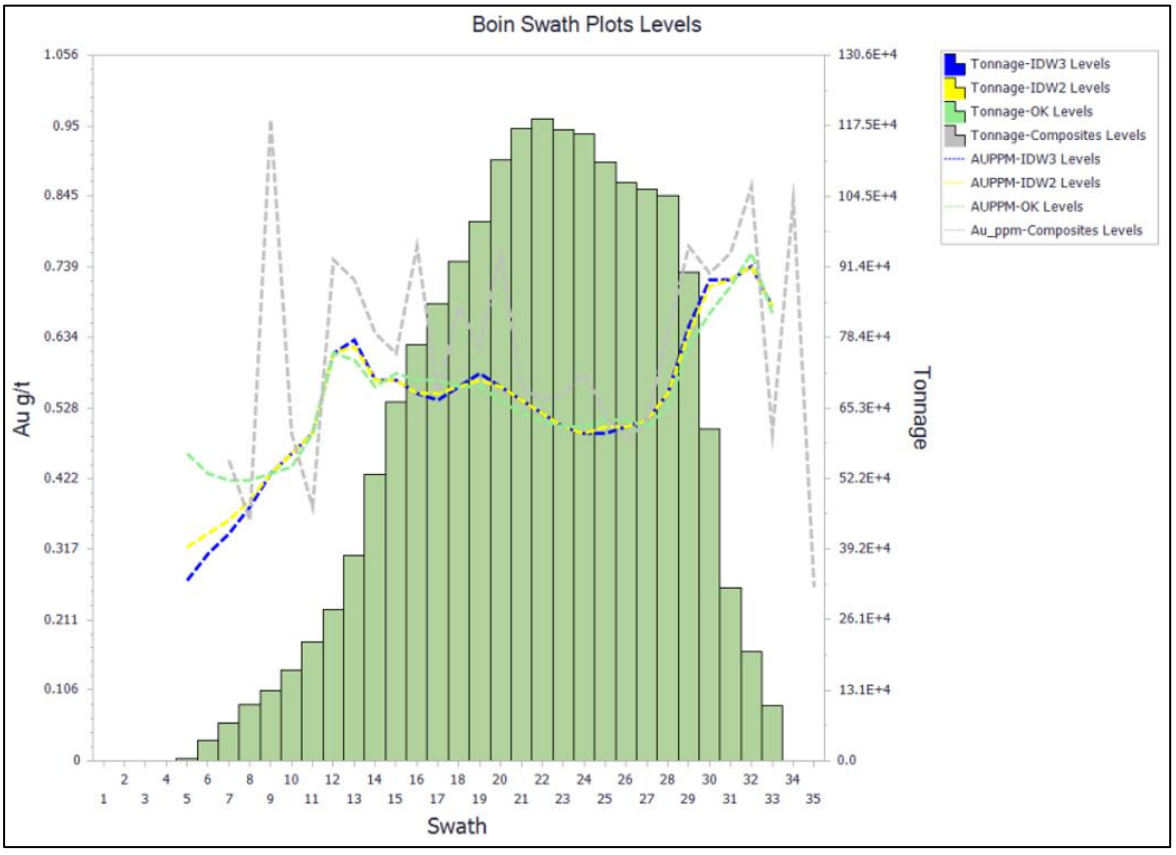


Source: DRA, 2026

Swath Plots

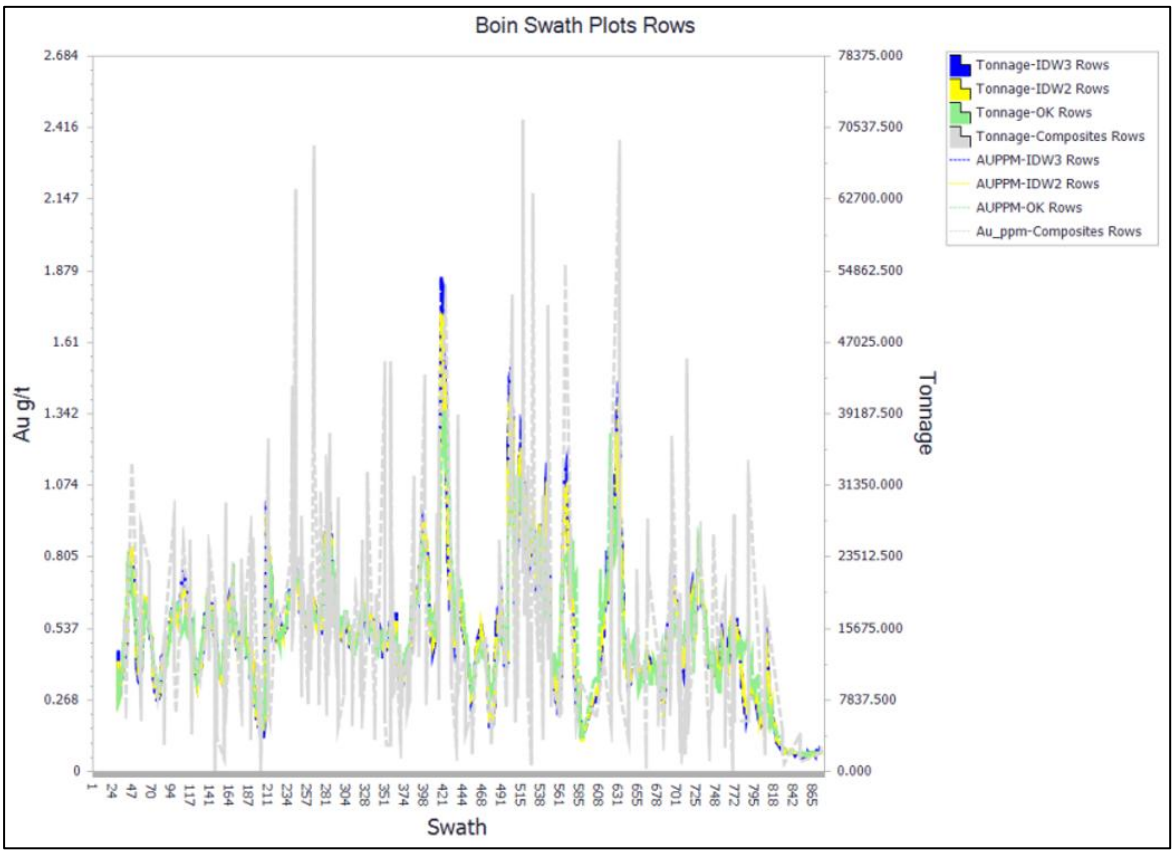
Average composite and block grades were compared by means of swath plots generated on vertical and horizontal planes spaced throughout the core of the Project. The generated comparative sections for the levels (elevations), rows (long sections) and columns (cross-sections) are provided in Figure 14.4.10 to Figure 14.4.12, respectively. It is clear from these plots that estimated block grades closely match those of the 2 m composite data throughout the deposit, with a minor amount of smoothing (as expected). Additionally, there is also good agreement along each direction between alternative interpolation methods including inverse distance cubed (IDW^3), inverse distance squared (IDW^2) and ordinary kriging (OK) model types.

Figure 14.4.10 Swath Plot for Au (g/t) – By Level – 2 m Composites vs. Block Grades by IDW^2, IDW^3 and OK Interpolation Methods, Boin Deposit



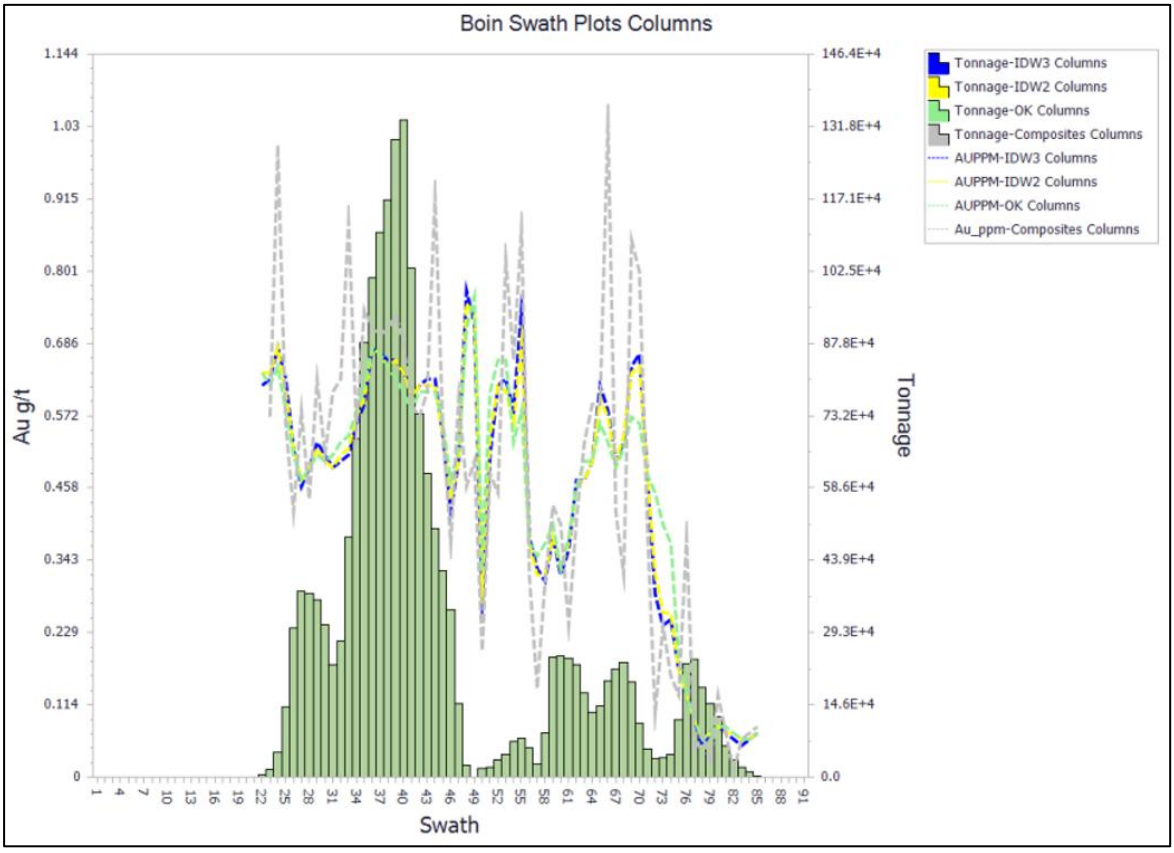
Source: DRA, 2026

Figure 14.4.11 Swath Plot for Au (g/t) – By Row – 2 m Composites vs. Block Grades by IDW², IDW³ and OK Interpolation Methods, Boin Deposit



Source: DRA, 2026

Figure 14.4.12 Swath Plot for Au (g/t) – By Column – 2 m Composites vs. Block Grades by IDW², IDW³ and OK Interpolation Methods, Boin Deposit



Source: DRA, 2026

Alternative Interpolation Methods

As shown in Figure 14.4.10 to Figure 14.4.12 above, IDW² and ordinary kriging (OK) models were also run as alternative interpolation methods in order to compare against the selected inverse distance weighting (IDW³) method used for the reported resource estimate. The global results are quite comparable. IDW³ was selected to report the resource estimate as it was deemed superior at reproducing the local distribution and variability of the gold grades.

Table 14.4.7 Comparison of IDW³, IDW² and OK Interpolation Methods, Enchi Block Model

Domain	Tonnes	Volume	AuPPM_IDW³	AuPPM_IDW²	AuPPM_OK
Domain 1	6,219,044	2,606,250	0.52	0.52	0.51
Domain 3	2,293,819	1,003,250	0.38	0.38	0.37
Domain 4	3,616,813	1,546,750	0.31	0.32	0.32
Domain 5	33,635,470	14,522,375	0.57	0.57	0.56

Domain	Tonnes	Volume	AuPPM_IDW^3	AuPPM_IDW^2	AuPPM_OK
Domain 6	440,820	198,375	0.42	0.42	0.39
Domain 7	2,399,086	976,250	0.47	0.45	0.4
Domain 8	2,146,788	898,750	0.33	0.33	0.34
Domain 9	801,873	340,250	0.44	0.44	0.46
Domain 10	702,621	314,125	0.3	0.3	0.31
Domain 11	819,245	311,500	0.21	0.21	0.21
Total	**53,075,578**	**22,717,875**	**0.51**	**0.51**	**0.50**

14.5 Sewum Deposit

14.5.1 Three-Dimensional (3D) Modelling

Newcore provided DRA with an initial set of wireframes for a total of four domain wireframes for Sewum:

- Domain 1: Checkerboard.
- Domain 2: Sewum North.
- Domain 3: Sewum South.
- Domain 4: Sewum Southeast.

The initial wireframes were based on broad geology and wide mineralization.

The four broad domains were divided into 15 mineralized domains. Following review of the approach and methodology used to produce these wireframes, DRA conducted an independent review of the interpreted zones in both 2D and 3D.

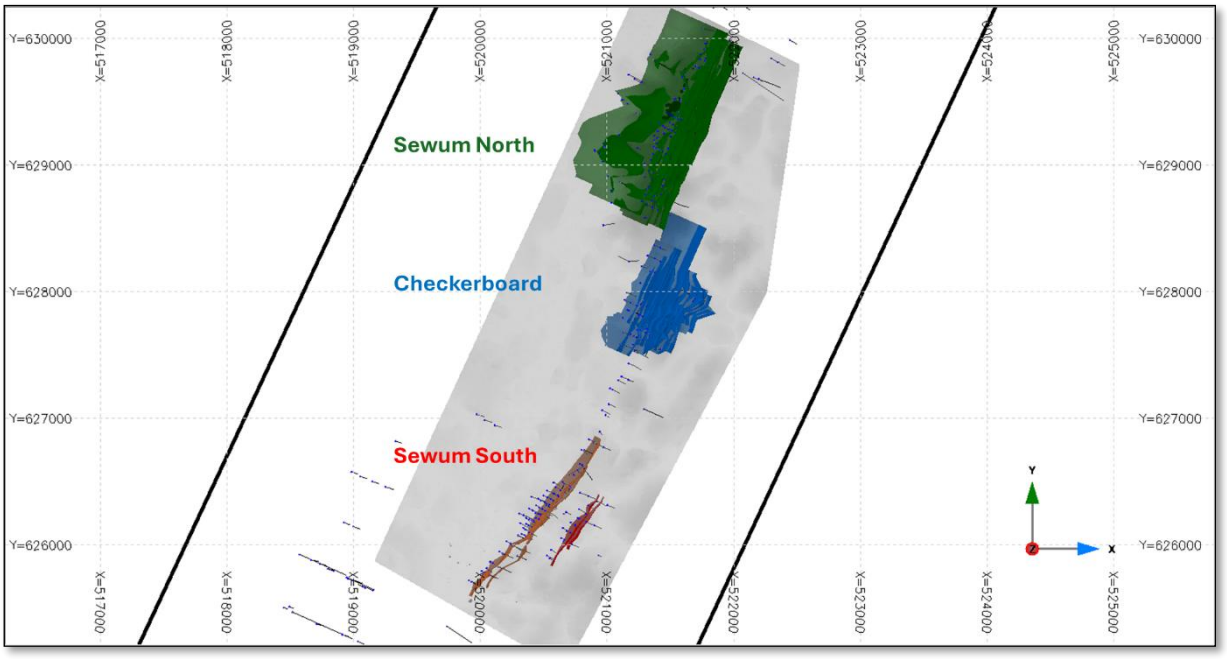
A new set of wireframes was generated by DRA using explicit approaches in Genesis software packages. The modelling was dominantly driven by grade continuity with additional considerations for lithological controls.

DRA adapted the wireframes using explicit modeling to separate out mineral domains that are distinct from low grade mineralization along strike of the deposit. The result was 27 wireframes for Sewum, but with only three estimation domains.

- Fourteen wireframes in checkerboard.
- Nine wireframes in Sewum North.
- Four wireframes in Sewum South.

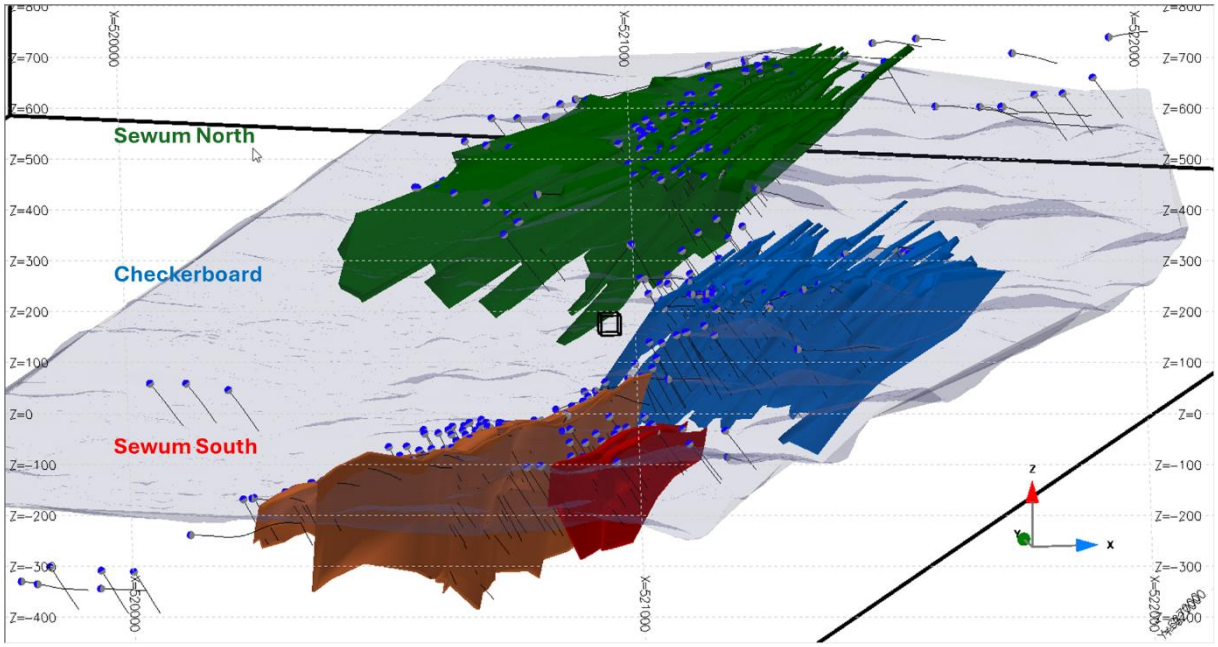
The new mineralized domains were comparable to the initial set, with adjustments based on interpreted continuity and some domains split to remove dilution. Changes were then discussed with Newcore in order to finalize the set of wireframes used in the subsequent mineral resource estimation (Figure 14.5.1 and Figure 14.5.2).

Figure 14.5.1 3D Plan View (Looking Down) of Sewum – Mineralized Domains



Source: DRA, 2026

Figure 14.5.2 3D Orthographic View of Sewum – Mineralized Domains



Source: DRA, 2026

14.5.2 Descriptive Statistics

Data was flagged according to the modelled wireframes, then statistically analyzed to determine the underlying data distributions and help with eventual comparisons to composited and estimated block grade data for reconciliation purposes. The data was imported into the Isatis.neo software package for statistical and geostatistical computations.

It is noted that relatively rare instances of unsampled intervals within the solids, mostly related to internal waste, were replaced with zero values as grades. Moreover, missing values for certain intervals (partial analyses, different drill campaigns, etc.) were assigned a value using the average of the attribute as correlation coefficients between elements were too weak to consider linear regression.

Basic descriptive statistics were then calculated for the raw data samples contained within each of the mineralized envelopes (i.e., zone intercepts) at the Project. These results are summarized by zone below in Table 14.5.1

Table 14.5.1 Summary of Basic Descriptive Statistics for Raw Data Samples at the Sewum Deposit

Au (g/t) Statistics	Min Value	Max Value	Average	Length Weighted Average	Sum of Length	Variance	Standard Deviation	% Variation	Median	Count
Assays	0.0001	62.61	0.32	0.32	22,954	0.90	0.95	2.92	0.12	19,511

14.5.3 Grade Capping

Grade capping is used to limit the spatial extrapolation of occasional isolated high grades in the resource model estimates. Capping analyses undertaken include the use of histograms, log probability plots and ranked composites (outlier analysis).

Histograms were used to search for distinct breaks in grade distributions in tandem with log probability plots, which generally show clear inflection points at the selected capping value. Outliers were also examined by means of ranked composites and observing the effect of sequential capping on the coefficient of variation (CoV) of the remaining data.

Table 14.5.2 Capping of Au (g/t) Sewum – Capped Proportion and Statistics for Various Cutoff Scenarios

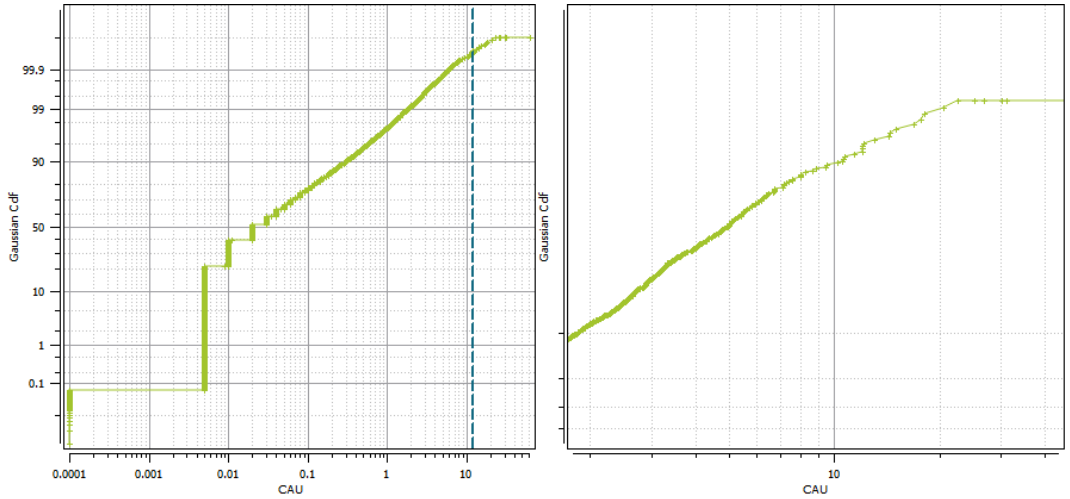
Cutoff	Percentile %	Mean	Std. Dev	COV	Metal Loss %	Capped Count	Capped Proportion %
0.30	90.00	0.07	0.10	1.339	48.83	5400	9.87
0.34	91.00	0.08	0.10	1.395	46.13	4826	8.82
0.38	92.00	0.08	0.11	1.447	43.70	4343	7.94
0.43	93.00	0.08	0.12	1.506	40.99	3794	6.94
0.50	94.00	0.09	0.14	1.579	37.74	3234	5.91
0.59	95.00	0.09	0.15	1.662	34.23	2692	4.92
0.70	95.96	0.10	0.17	1.751	30.68	2208	4.04
0.71	96.00	0.10	0.17	1.759	30.39	2178	3.98
0.89	97.00	0.10	0.19	1.882	25.92	1626	2.97
1.19	98.00	0.11	0.23	2.049	20.60	1091	1.99
1.85	99.00	0.12	0.28	2.309	13.96	544	0.99
5.81	99.90	0.13	0.40	2.962	4.42	54	0.10
12.00	99.97	0.14	0.46	3.348	2.10	17	0.03
12.42	99.97	0.14	0.46	3.366	2.02	14	0.03
62.61	100.00	0.14	0.59	4.244	0.00	0	0.00

The final selected capping grades used in the resource estimate are summarized along with a subset of basic descriptive statistics in Table 14.5.3. Representative histograms and log probability plots for the various mineralized zones are also provided in Figure 14.5.3 and Figure 14.5.4, respectively.

Table 14.5.3 Summary of Capping Grades by Mineralized Domain, Sewum Deposit

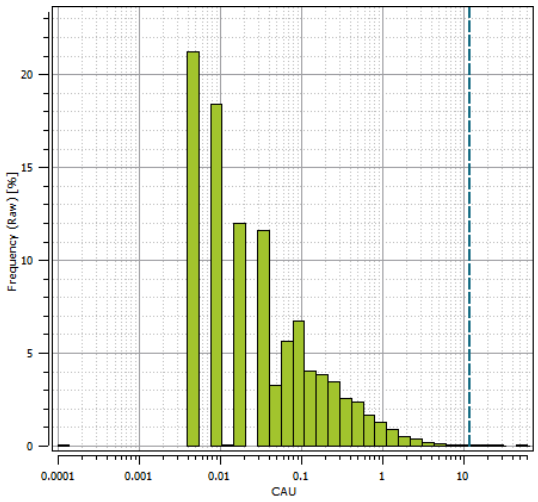
Domain	Variable	Uncapped Mean	Uncapped CoV	Capping Grade	Capped Comps #	Capped Comps %	Capped Mean	Capped CoV	Metal Loss %
Checkerboard	Au (g/t)	0.33	3.344	12	5	0.1%	0.31	2.284	
Sewum North	Au (g/t)	0.29	2.655	12	6	0.1%	0.28	2.133	2
Sewum South	Au (g/t)	0.53	2.227	12	3	0.1%	0.51	1.959	

Figure 14.5.3 Representative Log Probability Plots, Sewum Deposit



Source: DRA, 2026

Figure 14.5.4 Representative Histograms, Sewum Deposit



Source: DRA, 2026

14.5.4 Compositing

Drill hole intercepts through the interpreted mineralized domains at the Sewum deposit were composited to 1 m fixed length intervals, with a 0.25 m tolerance to merge shorter intervals resulting from intersection with wireframe edges or unsampled / missing intervals. The selected composite length was based on the systematic sampling approach used historically by geologists to focus on 1 m intervals through the target mineralized zones; this is also supported by statistical analysis with mean, median and mode lengths of 0.998 m, 1.000 m and 1.000 m, respectively. Drill hole intercepts include both RC and DDH intervals.

Trench samples were composited at 2 m fixed length intervals, with a 0.25 m tolerance. The selected composite length was based again on the systematic sampling approach used historically. Because of the location and quantity of trench samples, it was decided to use two composite sizes, and it is not believed to cause material difference to the estimation. The trench samples have an average length of 1.886 with a median and mode both of 2.000.

Basic descriptive statistics for the composited data within wireframes (i.e., zone intercepts) are summarized by zone below in Table 14.5.4.

Table 14.5.4 Summary of Basic Descriptive Statistics for 2 m Composited Data

Domain	Variable	Count	Mean	Variance	Std. Dev	COV	Minimum	Maximum	Skewness	Kurtosis
Checkerboard	Au (g/t)	7,288	0.31	0.4868	0.7	2.284	0	12	8.482	111.2
Sewum North	Au (g/t)	8,958	0.28	0.3632	0.6	2.133	0	12	9.915	156.7
Sewum South	Au (g/t)	2,315	0.51	1	1	1.959	0	12	4.7	36.8

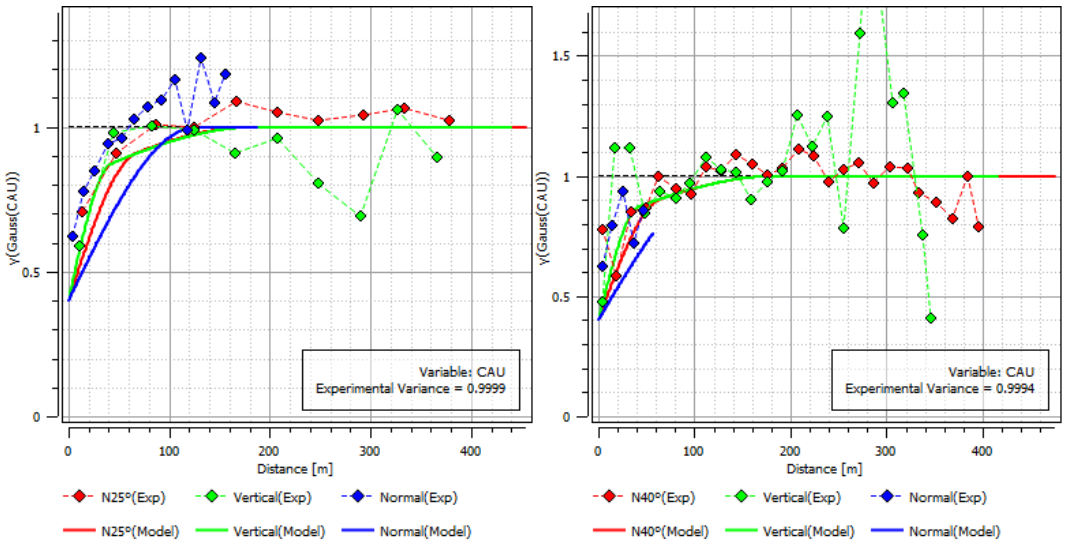
14.5.5 Variography

Variography aims to assess the spatial continuity of grade for an element of interest and ultimately helps guide the definition of parameters for the interpolation of Mineral Resources. The selected approach, Ordinary Kriging (OK), is a linear geostatistical estimation method that requires input parameters to limit the size of the search neighbourhood (via a defined search ellipsoid) for each point to be interpolated within the block model. Downhole and directional variography for the Sewum deposit were run also using Isatis.neo software.

Envelopes and containing composites grouped into broader zones Sewum south, checkerboard and Sewum North. Checkerboard and Sewum north share a dominant orientation for strike, Sewum south has a similar strike but differs by approximately 15 degrees. Each zone has similar dip and dip directions but seem to shallow to the west. The primary direction and dominant dip were modeled in Isatis VMAP and any additional trend plunges where modelled based on variance map. Sewum North produced the best variogram based on sample density and continuity. Checkerboard has cross structures and variography may still need optimization through additional geological data, however the dominant trend is comparable to Sewum North and the same variogram can be used.

It should be noted that the indicated ranges were only used as a guide in the selection of maximum search ellipsoid distances for Measured, Indicated and Inferred Resource categories, in conjunction with geological information and other statistical factors such as average drill hole spacing.

Figure 14.5.5 Back Transformed Normal Score Variograms for Au (g/t), Sewum Deposit



Checkerboard and Sewum North **Sewum South and Sewum Southeast**

Source: DRA, 2026

Table 14.5.5 Variogram Model Parameters (raw), Sewum Deposit

Domain	Sewum North	Checkerboard	Sewum South
Nugget	0.7	0.7	0.75
Azimuth	25	25	40
Dip	0	0	0
Spin	90	90	90
Structure 1			
Sill	0.15	0.15	0.10
Range 1	30	30	40
Range 2	17	17	20
Range 3	13	13	10
Type	Spherical	Spherical	Spherical
Structure 2			
Sill	0.15	0.15	0.15
Range 1	150	150	183
Range 2	111	111	97
Range 3	64	64	10
Type	Spherical	Spherical	Spherical

14.5.6 Mineral Resource Estimate

Following exploratory data and geostatistical analyses using Isatis.neo software, the block model was built, and subsequent grade and tonnage estimates were computed in Genesis before standardizing in MinePlan 3D.

Block Model

A single block model was constructed for the mineralized domains at Sewum to capture a variety of data types including the oxidation type, relevant mineralized domain codes, estimated block grades, density values, percentage of material at wireframe boundaries, initial resource category as determined by multiple-pass interpolation and finalized resource category as determined by the resource modeller.

Percent envelope was used to better define narrow portions of the block model and to respect the volumetrics of the modelled mineralized domains and triangulations. The model was rotated at 25 degrees and aligned along the approximate strike of the deposit. Relevant block model definition parameters are summarized in Table 14.5.6.

Table 14.5.6 Block Model Definition Parameters, Sewum Deposit

Description	Value
Model Dimension X (m)	4 000
Model Dimension Y (m)	10 000*
Model Dimension Z (m)	675
Origin X (Easting)	516 500
Origin Y (Northing)	623 500
Origin Z (Lower Elev.)	-301
Rotation (º)	25
Block Size X (m)	5
Block Size Y (m)	5
Block Size Z (m)	5
Percent Envelope	TRUE
Sub-blocking	FALSE

* Model dimensions intentionally large along the Northing to accommodate the same model for new zones at Sewum (Sewum South West, and Tokosea to the northeast) in the future.

Search Strategy and Interpolation

Block values were estimated for each individual mineralized domain using the generated composites and the Ordinary Kriging (OK) method. The set of search parameters used in the multi-pass interpolations, derived mainly from geological information and interpreted continuity with support from variography and statistical factors such as average drill hole spacing, are summarized by estimation domain in Table 14.5.7.

Table 14.5.7 Ordinary Kriging (OK) Interpolation Parameters Summary for the Sewum Block Model

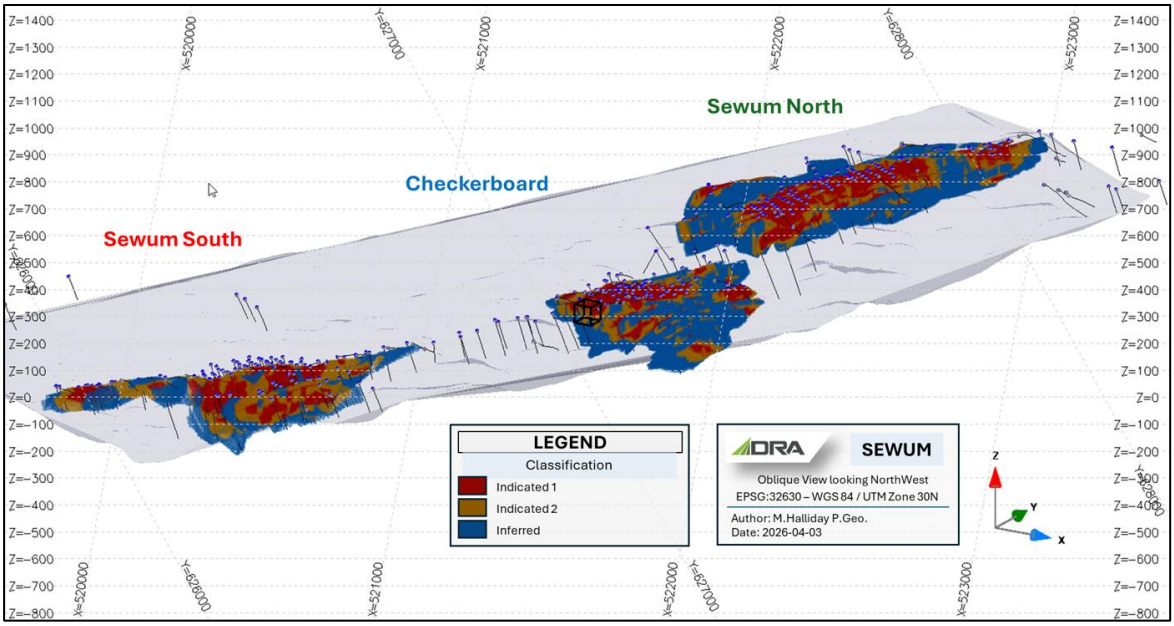
Domain	Pass	Estimation Method	Min Samples	Max Samples	Max. Samples /DDH	Major Axis	Semi Axis	Vert Axis
Sewum North / Checkerboard	1	OK	7	20	3	75	56	32
	2	OK	7	20	3	112.5	84	48
	3	OK	5	20	3	188	139	80
Sewum South	1	OK	7	20	3	93	50	10
	2	OK	7	20	3	140	75	10
	3	OK	5	20	3	231	125	13

Mineral Resource Classification

The Mineral Resources reported herein for the Sewum deposit have been classified into Measured, Indicated and Inferred categories. This classification is based on the interpreted geological and grade continuity of the observed mineralization.

Primary categorization was based on the minimum and maximum number of composites and experimental variogram distances used parameters during multiple-pass OK interpolation (refer back to Table 14.5.7). Secondary classification was carried out by creating geometric volumes of higher confidence and continuity based on drill hole density, with subsequent minor manual adjustments to remove any remaining spotted dog effects. A 3D orthographic view of the final block classification is provided in Figure 14.5.6.

Figure 14.5.6 3D Orthographic View of the Final Block Classification, Sewum Deposit



Source: DRA, 2026

14.5.7 Mineral Resource Statement

The Mineral Resource Estimate statement for the Sewum deposit prepared by DRA is summarized in Table 14.5.8. Additional details on mining and processing modifying factors are also provided in the corresponding footnotes. The Resources have been reported using a constraining resource pit at a gold price of $3,200/oz.

Table 14.5.8 Mineral Resources Estimate – Effective Date of 6 October 2025

Zone	Classification	Tonnes '000	Au Grade g/t	Contained Au ounces
Sewum	Indicated	41,233	0.43	573,000
	Inferred	24,246	0.39	308,000

Notes for Mineral Resource Estimate:

1. Canadian Institute of Mining Metallurgy and Petroleum (CIM) definition standards were followed for the resource estimate.
2. The effective date of the Resource is 6 October 2025.
3. All figures are rounded to reflect the relative accuracy of the estimate and numbers may not add due to rounding.
4. The resource model used Ordinary Kriging (OK) grade estimation within a three-dimensional block model with mineralized zones defined by wireframed solids and constrained by a pit shell. Validations were completed using Inverse Distance squared (ID^2).
5. Open pit cut-off grades varied from 0.1 to 0.2 g/t Au based on mining and processing costs as well as the recoveries in different weathered material.
6. A $3,200/ounce gold price was used to determine the cut-off grade.
7. Metallurgical recovery of 85% was applied to oxide and transition mineralization, and 91.7% for fresh mineralization.
8. The pit optimization considered the following costs: mining cost based on mineralization type of $1.97/tonne for oxide, $2.62/tonne for transition, and $3.15/tonne for fresh; waste mining costs of $1.64/tonne for oxide, $2.34/tonne for transition, and $2.87/tonne for fresh; processing and G&A costs assumed of $8.74/tonne for oxide, $8.49/tonne for transition, and $19.29/tonne for fresh.
9. Average densities of mineralized material varied between 1.53 and 2.15 g/cm^3 for oxide, 1.86 and 2.38 g/cm^3 for transition, and 2.48 and 2.74 g/cm^3 for fresh rock. Average densities of waste rock varied between 1.45 and 1.77 g/cm^3 for oxide, 1.81 and 2.15 g/cm^3 for transition, and 2.45 and 2.74 g/cm^3 for fresh rock
10. Optimization pit slope angles varied by deposit and mineralized area, with an overall strip ratio including all pits of 3.35.
11. Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have economic viability.
12. The resource estimate was prepared by Matthew Halliday, P.Geo of DRA Americas Inc. in accordance with NI 43-101. This individual is an independent qualified person (QP) as defined by NI 43-101.
13. As of the Report's date, the QPs, to the best of their knowledge, are not aware of any metallurgical, environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other risk factors that might materially affect the estimate of Mineral Resources.

14.5.8 Block Model Validation

The current Sewum deposit block model has been validated by DRA using a combination of visual inspection and statistical comparisons, including:

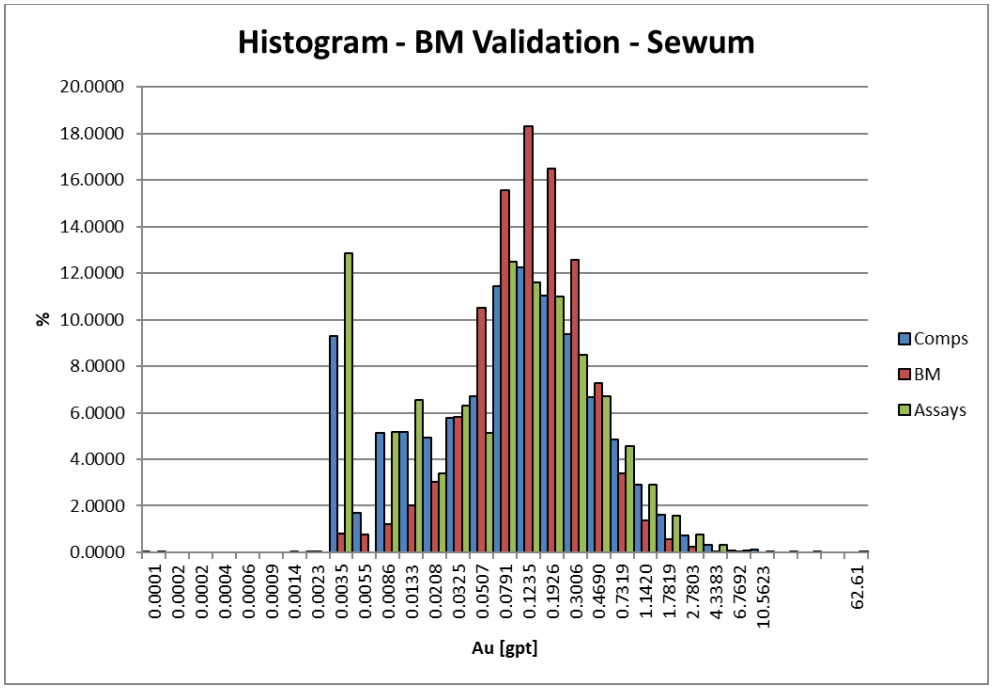
- Basic statistical review.
- Visual comparison of assays and block grades.
- Inspection of generated swath plots.

- Alternative interpolation methods.

Statistical Review

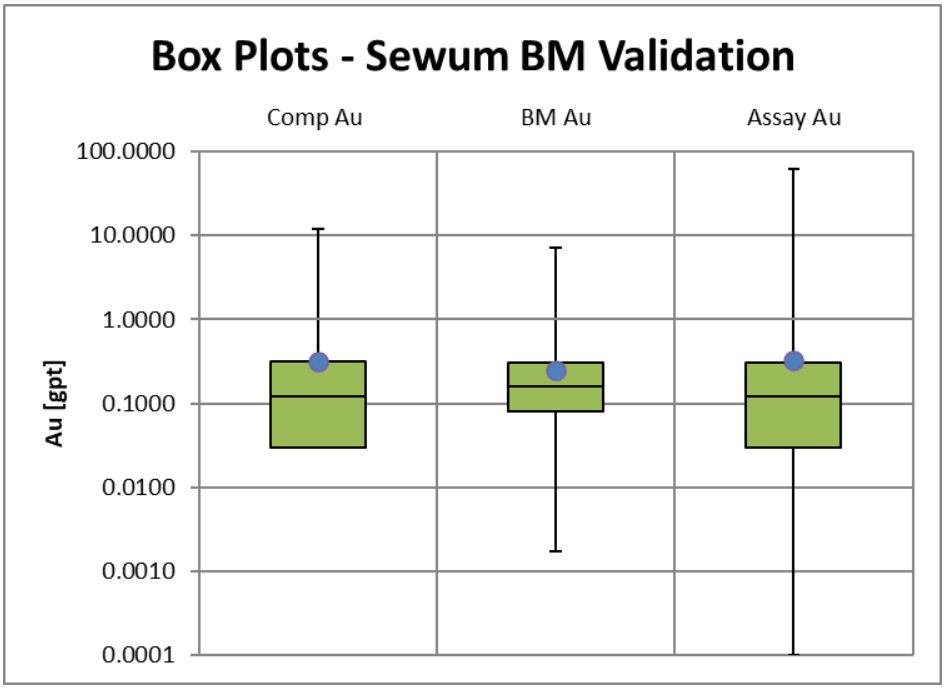
Basic review indicated that there is adequate smoothing between levels of support from assays to composites to blocks.

Figure 14.5.7 Block Model Validation – Sewum



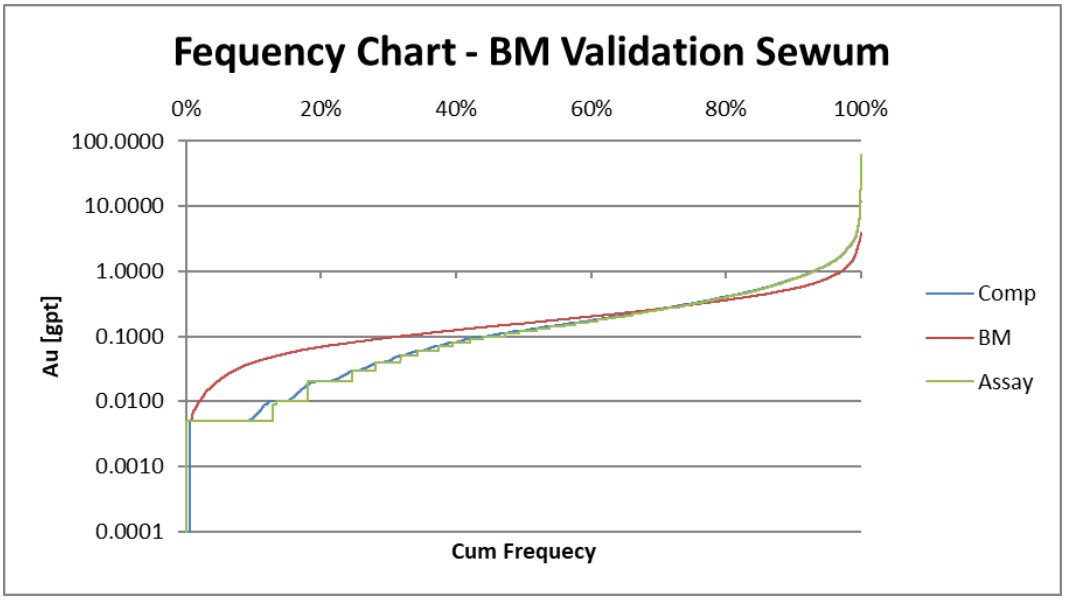
Source: DRA, 2026

Figure 14.5.8 Block Model Validation – Sewum



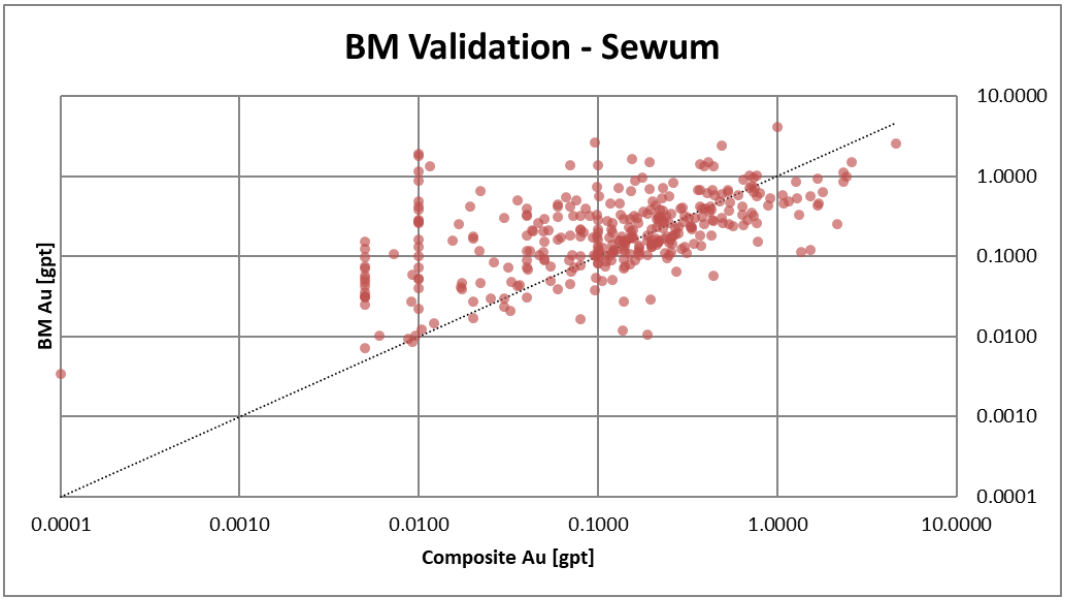
Source: DRA, 2026

Figure 14.5.9 Block Model Validation – Sewum



Source: DRA, 2026

Figure 14.5.10 Block Model Validation - Sewum



Source: DRA, 2026

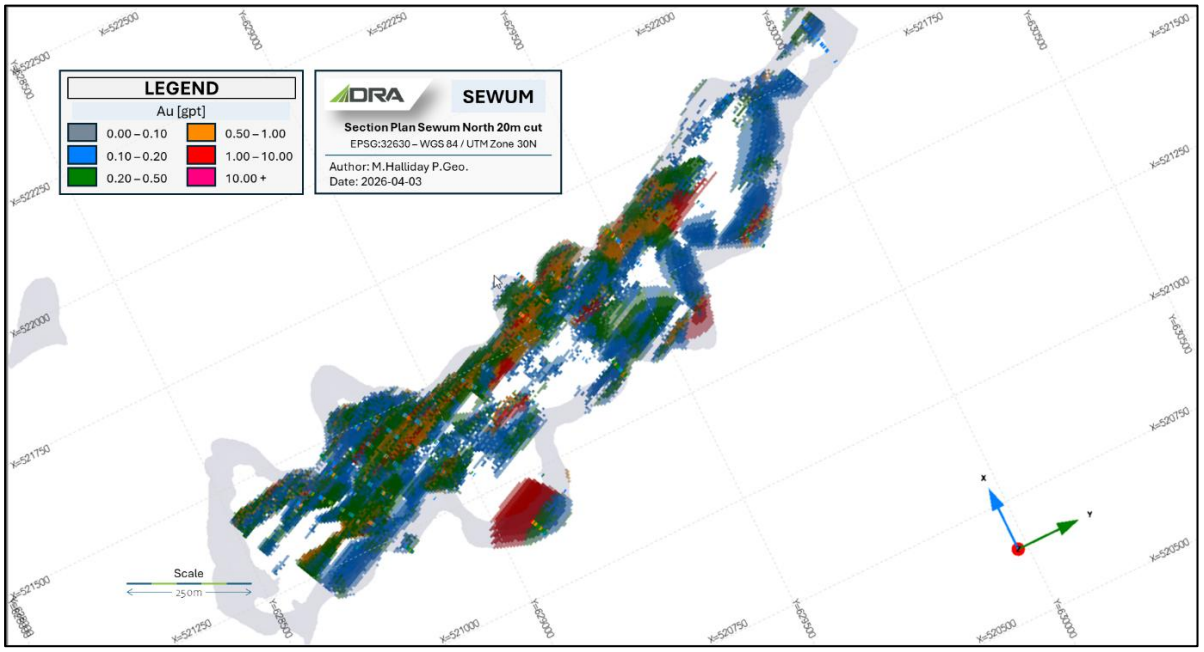
Table 14.5.9 Comparable Statistics for Different Sample Supports

Au (g/t) - Statistics	Composite	Block Model	Assays
Min Value	0.00	0.0017	0.0001
Max Value	12.00	7.07	62.61
Average	0.32	0.25	0.32
Length Weighted Average	0.33	-1.00	0.32
Sum of Length	20,102	-1.00	22,954
Variance	0.50	0.10	0.90
Standard Deviation	0.70	0.32	0.95
% Variation	2.20	1.27	2.92
Median	0.12	0.16	0.12
First Quartile	0.03	0.08	0.03
Third Quartile	0.32	0.30	0.31
Count	18,564	770,070	19,511
Count Missing	0	93480	4

Visual Inspection

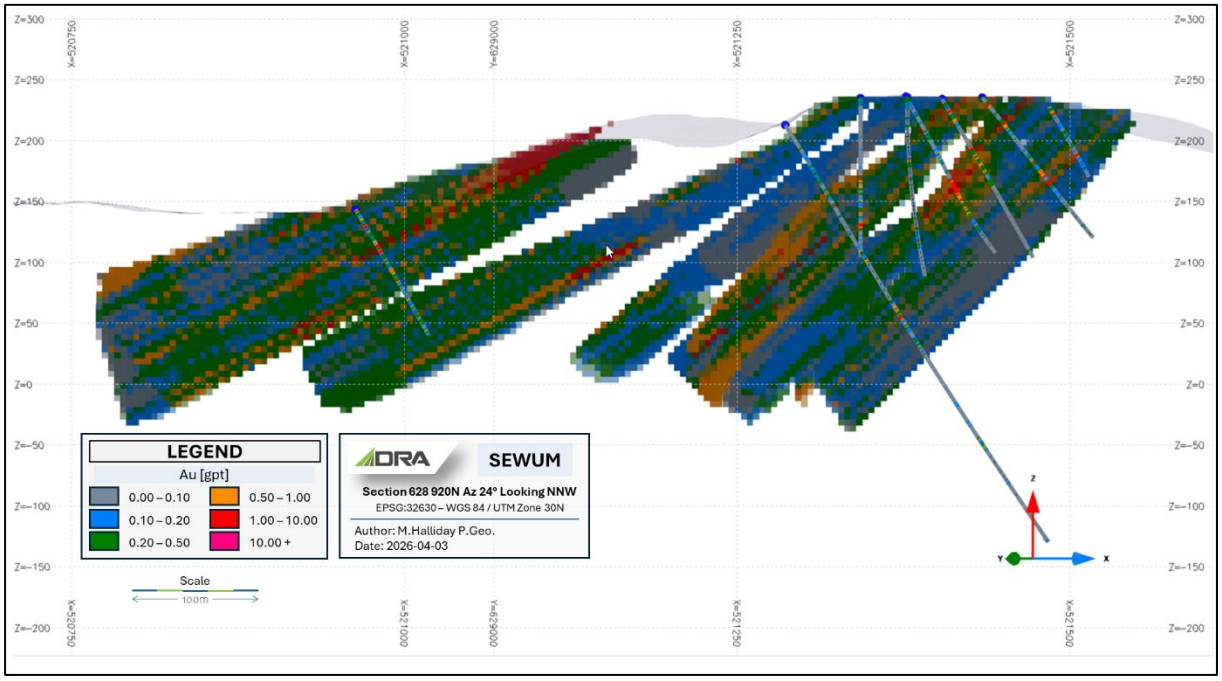
Estimated blocks and drill hole intercepts at Sewum were reviewed both on 2D vertical sections and level plans, as well as interactively within the MinePlan 3D software environment. The block grades suitably respect assay grades throughout the deposit. A plan view of Au (g/t) with assay and block grades superimposed is presented in Figure 14.5.11. A representative north-south vertical section through the core of the deposit is shown in Figure 14.5.12.

Figure 14.5.11 Plan View of Au (g/t) at Sewum North (20 m slice) with Assay and Block Grades



Source: DRA, 2026

Figure 14.5.12 Comparison of Assay and Block Grades for the Sewum Deposit on Representative Vertical Section (628 920N) Azimuth 24°

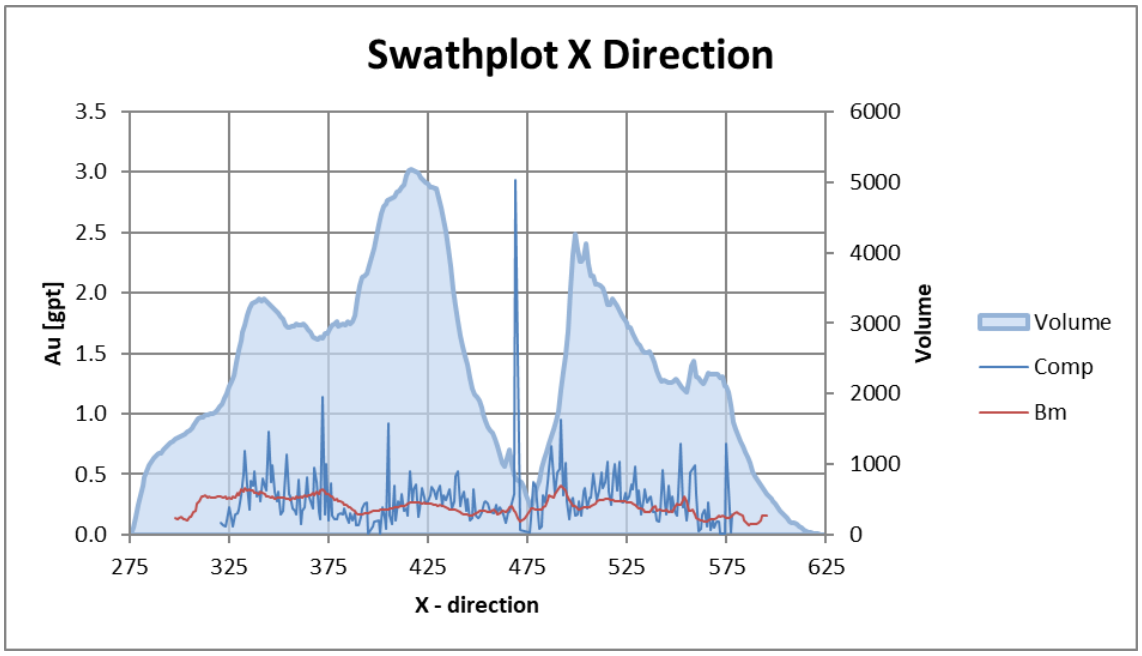


Source: DRA, 2026

Swath Plots

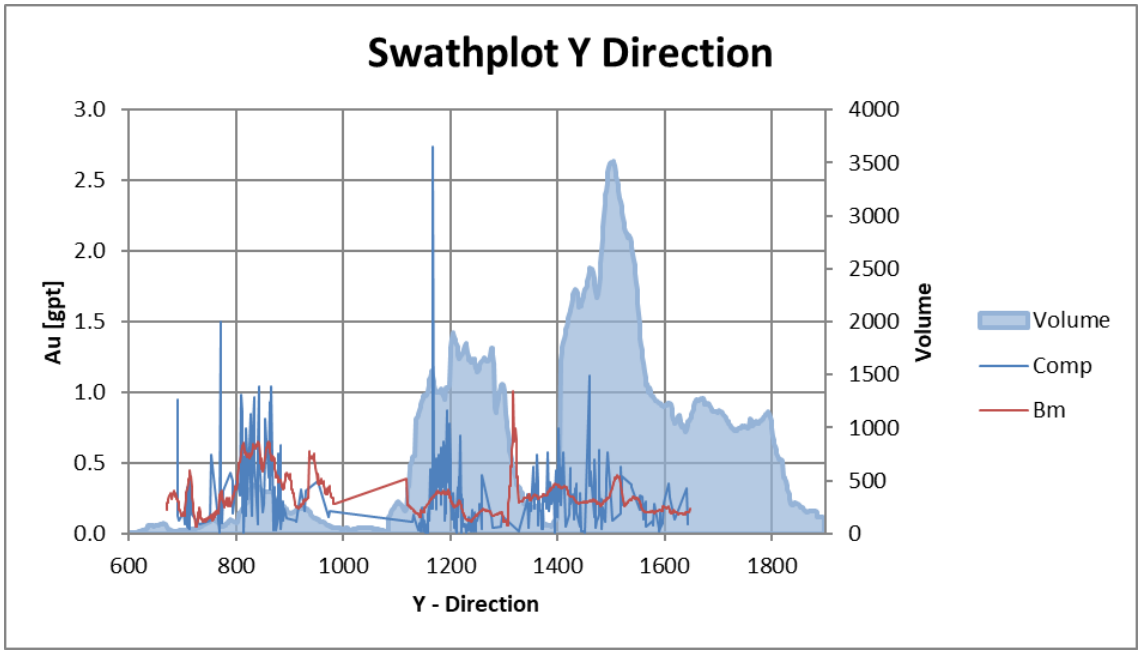
Average composite and block grades were compared by means of swath plots generated on vertical and horizontal planes spaced throughout the core of the Sewum deposit. The generated comparative sections for the levels, rows and columns are provided in Figure 14.5.13 to Figure 14.5.15, respectively. It is clear from these plots that estimated block grades closely match those of the 1 m composite data throughout the deposit, with a minor amount of smoothing (as expected). Additionally, there is also good agreement along each direction between alternative interpolation methods including inverse distance weighted (IDW^2) and ordinary kriging (OK) model types.

Figure 14.5.13 Swath Plot for Au (g/t) – X-Direction – Composites vs. Block Grades by OK Interpolation Method



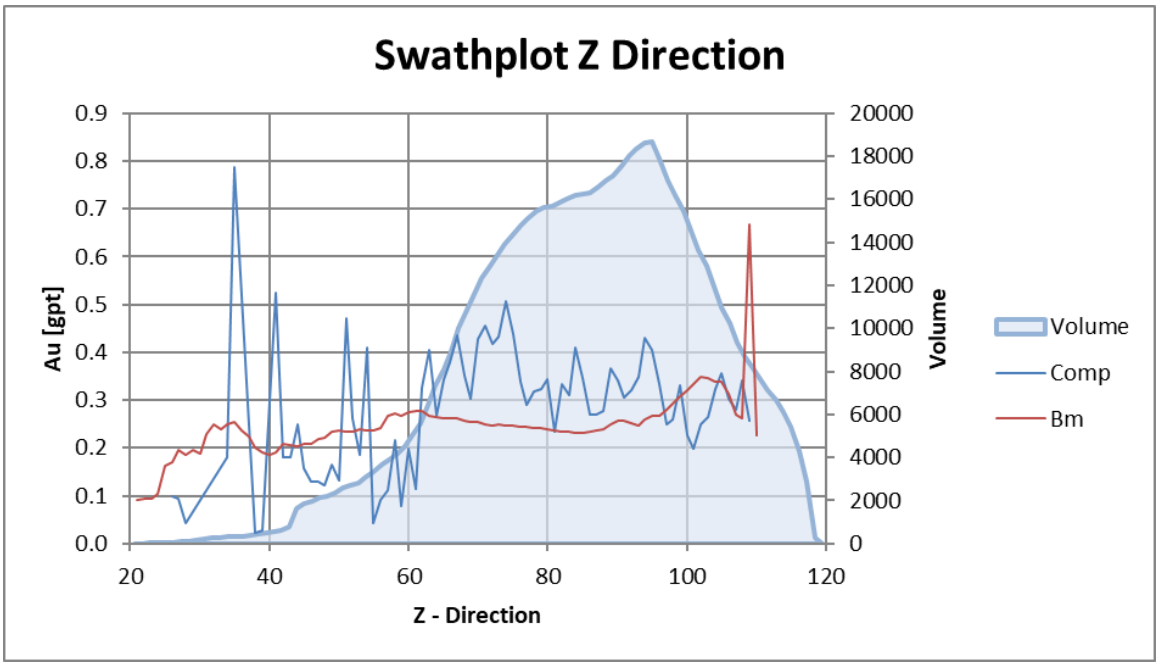
Source: DRA, 2026

Figure 14.5.14 Swath Plot for Au (g/t) – Y-Direction – Composites vs. Block Grades by OK Interpolation Method



Source: DRA, 2026

Figure 14.5.15 Swath Plot for Au (g/t) – By Z-Direction – Composites vs. Block Grades by OK Interpolation Method



Source: DRA, 2026

Alternative Interpolation Methods

As shown in Figure 14.5.13 to Figure 14.5.15 above, an inverse distance squared weighting (IDW^2) model was also run as an alternative interpolation method in order to compare against the selected OK method used for the reported resource estimate. The results of this comparison are summarized here mainly as a global bias check (Table 14.5.10). The correlation between the models is considered acceptable by DRA, with global grades showing overall percent differences of less than 5% between interpolation types.

Table 14.5.10 Comparison of IDW^2 and OK Interpolation Methods, Sewum Block Model

Category	IDW^2		OK			
	Tonnes	Grade	Tonnes	% Diff	Grade	% Diff
Indicated	42,850,000	0.43	44,300,000	4%	0.41	4%
Inferred	18,450,000	0.40	19,120,000	4%	0.38	5%

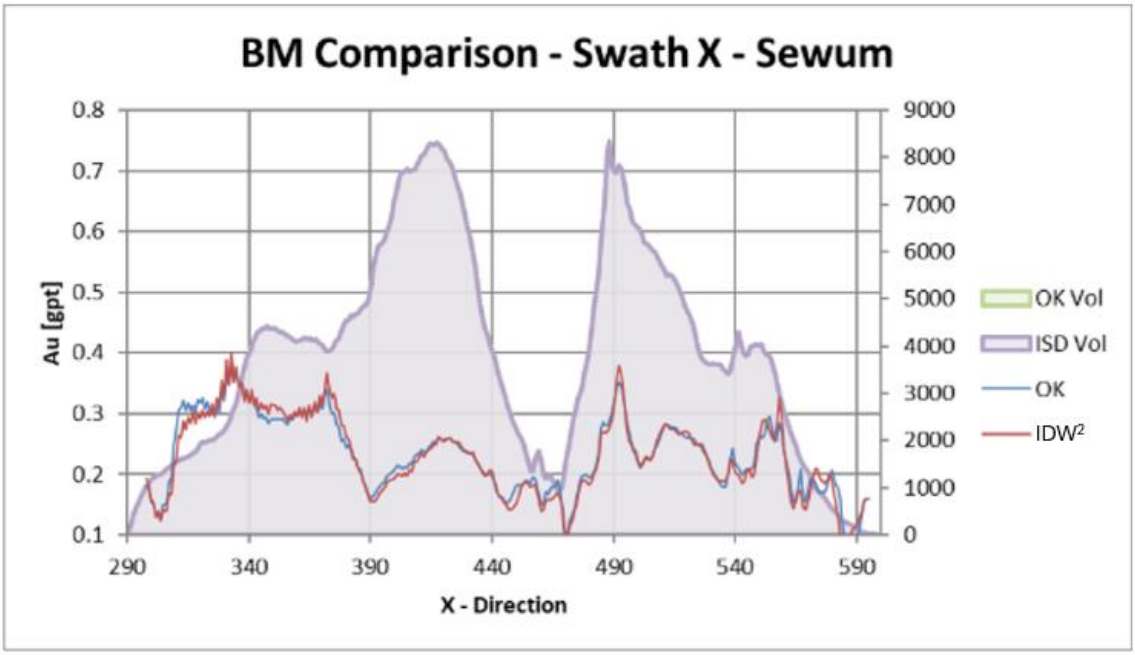
Using a 0.10 cut-off grade global on resource model in pit, percent difference is smaller when using the same cut-off (i.e., cog=0.10 for IDW^2)

Globally the means are the same, although there is slightly more variance in the IDW^2 method. See Table 14.5.11.

Table 14.5.11 Block Statistics Between Estimate Methods

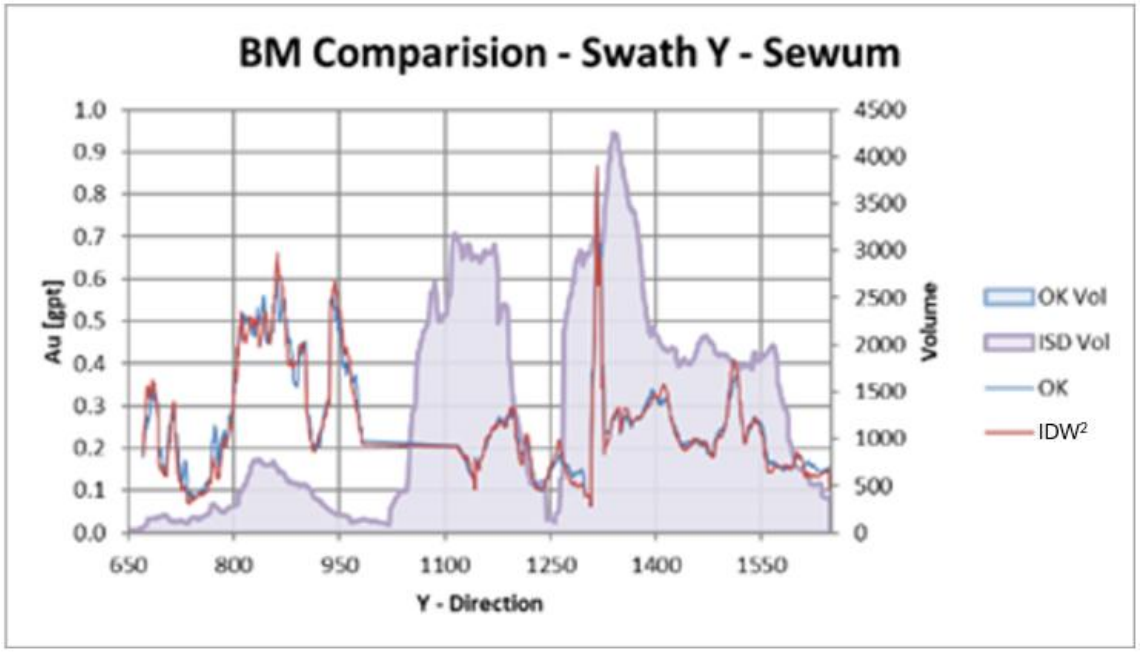
Au (g/t) - Statistics	OK	IDW²
Min Value	0.002	0.000
Max Value	7.07	10.72
Average	0.24	0.24
Variance	0.10	0.12
Standard Deviation	0.31	0.35
% Variation	1.27	1.42
Median	0.15	0.14
First Quartile	0.08	0.07
Third Quartile	0.29	0.28
Count	1,121,229	1,121,229
Count Missing	146,592	146,592

Figure 14.5.16 Block Model Validation – Swath X – Sewum



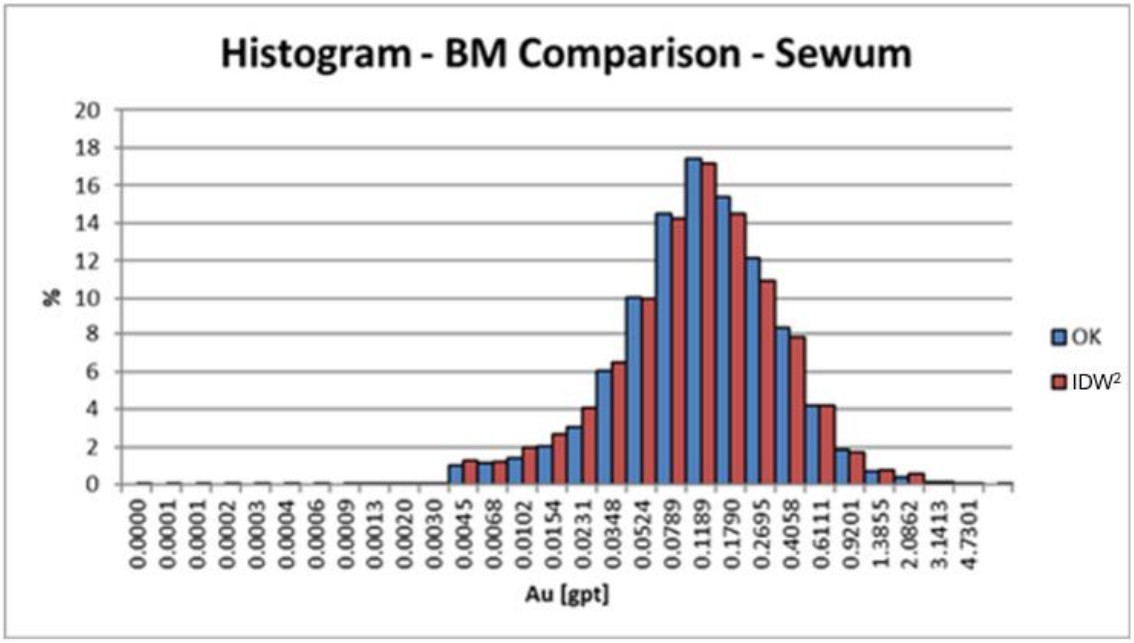
Source: DRA, 2026

Figure 14.5.17 Block Model Validation – Swath Y – Sewum



Source: DRA, 2026

Figure 14.5.18 Histogram – Block Model Validation – Sewum



Source: DRA, 2026

Reviewing the comparison of block grades in different directions and total block histograms both methods give similar results on a global scale.

14.6 Nyam Deposit

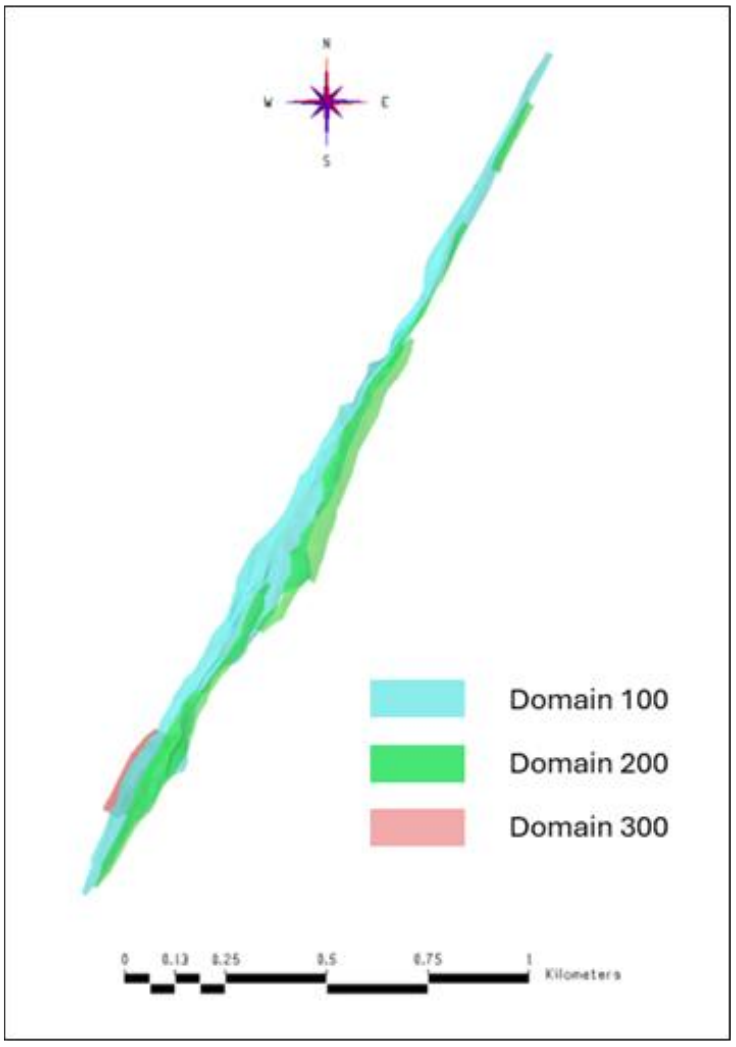
14.6.1 Three-Dimensional (3D) Modelling

Newcore provided DRA with an initial set of wireframes for a total of three mineralized domains at the Nyam deposit. Following review of the approach and methodology used to produce these wireframes, DRA conducted an independent review of the interpreted zones in 2D and 3D.

A new set of wireframes was generated by DRA using an explicit approach in MinePlan 3D software. The modelling was dominantly driven by grade continuity and supported by geological controls (e.g., known mineralized shear zones).

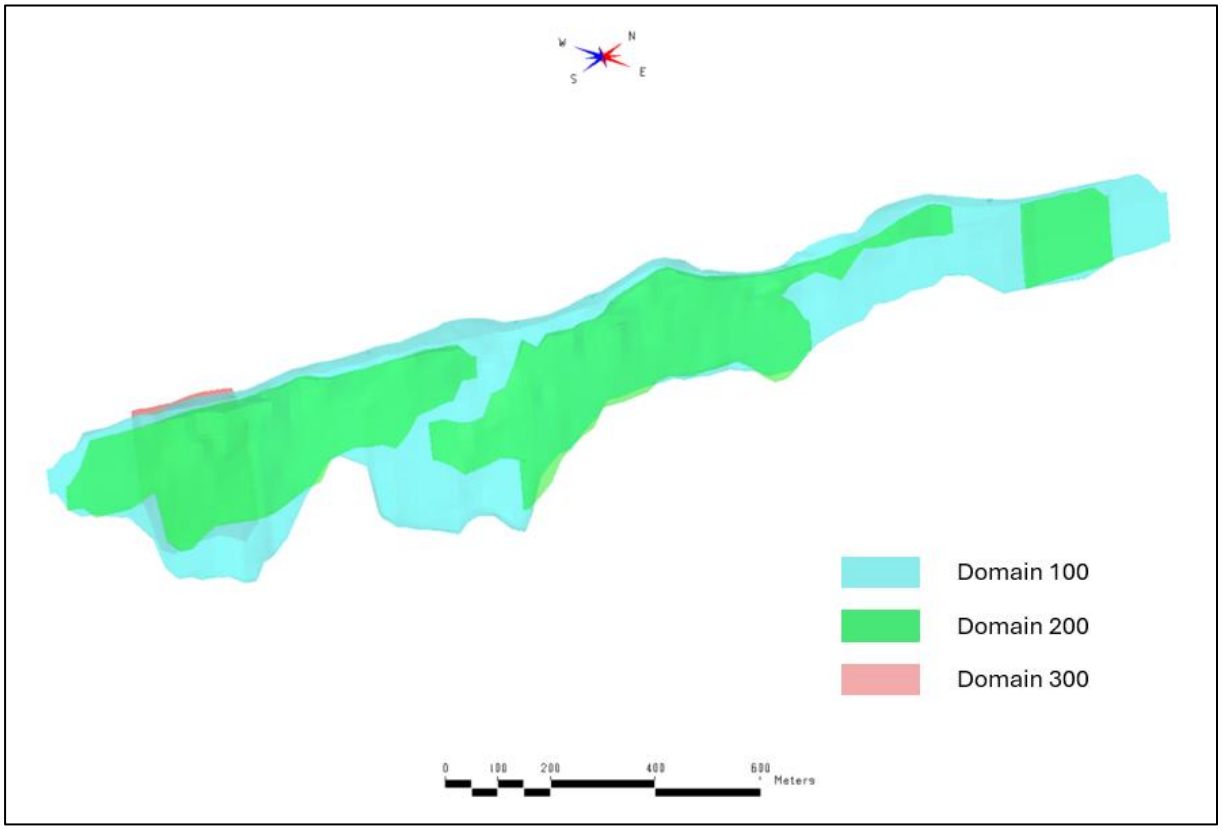
The new mineralized domains were comparable to the initial set, with small adjustments made according to interpreted continuity by the current QP. Changes were discussed with Newcore in order to finalize the set of wireframes used in the Mineral Resource Estimate (Figure 14.6.1 and Figure 14.6.2).

Figure 14.6.1 3D Plan View (Looking Down) of Nyam Mineralized Domains



Source: DRA, 2026

Figure 14.6.2 3D Orthographic View of Nyam Mineralized Domains



Source: DRA, 2026

14.6.2 Descriptive Statistics

Data was tagged according to the modelled wireframes, then analyzed statistically to determine the associated data distributions; this is important for future comparisons to composited and estimated block grade data for validation purposes. The data was imported into the Isatis.neo software package for statistical and geostatistical calculations.

It is noteworthy that uncommon instances of unsampled intervals within the solids, mostly related to internal waste, were replaced with zero values as grades.

Basic descriptive statistics were calculated for the raw data samples contained within each of the mineralized envelopes (i.e., zone intercepts) at the Nyam deposit. These results are summarized by zone below in Table 14.6.1.

Table 14.6.1 Summary of Basic Descriptive Statistics for Raw Data Samples by Mineralized Domain, Nyam Deposit

Domain	Variable	Count	Mean	Variance	Standard Deviation	CoV	Minimum	Maximum
100	Au (g/t)	7,223	0.47	0.98	0.99	2.11	0.00	41.70
200	Au (g/t)	1,837	0.48	1.33	1.16	2.40	0.00	16.70
300	Au (g/t)	126	0.26	0.19	0.43	1.64	0.00	3.37

14.6.3 Grade Capping

Grade capping is used to limit the spatial extrapolation of anomalously high grades in the resource model estimates. Capping analyses undertaken include the use of histograms, log probability plots and ranked composites (outlier analysis).

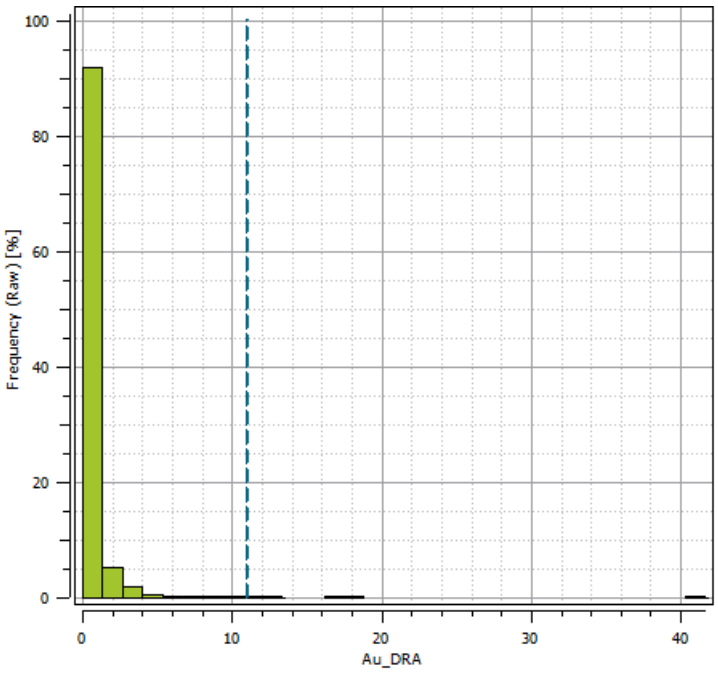
Histograms were used to search for distinct breaks in grade distributions along with log probability plots, which typically show clear inflection points at the selected capping value. Outliers were also examined by means of ranked composites and observing the effect of sequential capping on the coefficient of variation (CoV) of the remaining data.

The final selected capping grades used in the resource estimate are summarized along with a subset of basic descriptive statistics in Table 14.6.2. Representative histograms and log probability plots for the various mineralized zones are also provided in Figure 14.6.3 and Figure 14.6.4, respectively.

Table 14.6.2 Summary of Capping Grades by Mineralized Domain, Nyam Deposit

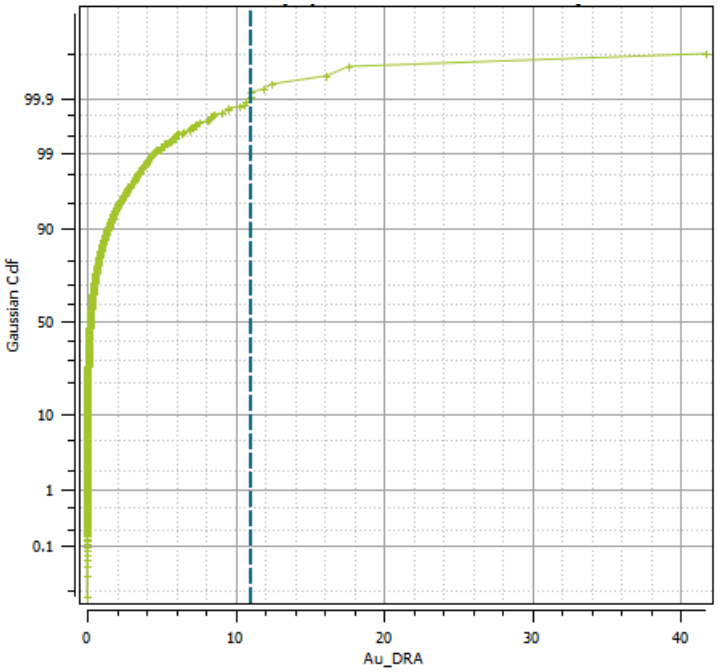
Domain	Variable	Uncapped Mean	Uncapped CoV	Capping Grade	Capped Comps #	Capped Comps %	Capped Mean	Capped CoV	Metal Loss %
100	Au (g/t)	0.47	2.11	11.00	6	0.08	0.46	1.99	0.70
200	Au (g/t)	0.48	2.40	8.40	8	0.44	0.46	2.07	3.81
300	Au (g/t)	0.26	1.64	1.00	5	3.97	0.23	1.15	13.47

Figure 14.6.3 Representative Histogram – Domain 100, Nyam Deposit



Source: DRA, 2026

Figure 14.6.4 Representative Log Probability Plot – Domain 100, Nyam Deposit



Source: DRA, 2026

14.6.4 Compositing

Drill hole intercepts through the interpreted mineralized domains at the Nyam deposit were composited to 1 m fixed length intervals, with a 0.3 m tolerance to merge shorter intervals along wireframe edges or resulting from unsampled / missing intervals. The selected composite length was based on the systematic sampling used historically by geologists to focus on 1 m intervals through the target mineralized zones; this is also supported by statistical analysis with mean and median lengths of 1.1 m and 1.0 m, respectively. Descriptive statistics for the composited data within wireframes (i.e., zone intercepts) are summarized by zone below in Table 14.6.3.

Table 14.6.3 Summary of Basic Descriptive Statistics for 1 m Composited Data, Nyam Deposit

Domain	Variable	Count	Mean	Variance	Standard Deviation	CoV	Minimum	Maximum
100	Au (g/t)	7,223	0.46	0.85	0.92	1.99	0.00	11.00
200	Au (g/t)	1,837	0.46	0.92	0.96	2.07	0.00	8.40
300	Au (g/t)	126	0.23	0.07	0.26	1.15	0.00	1.00

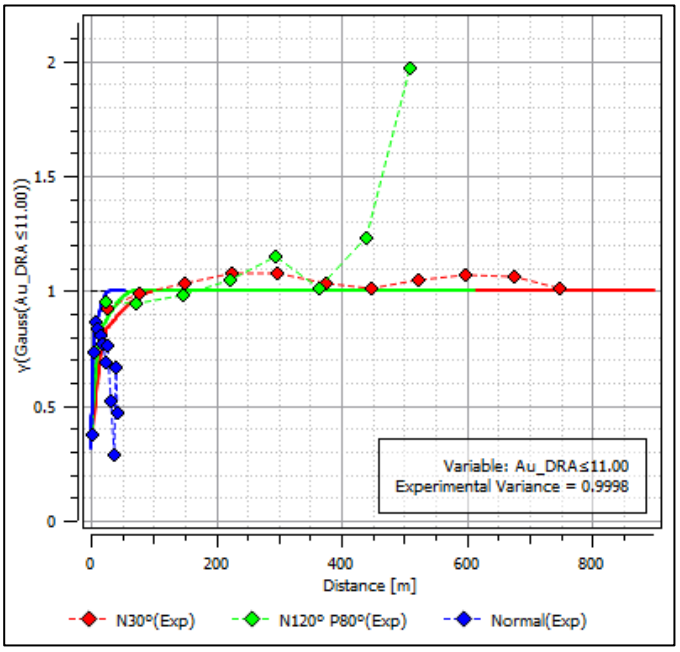
14.6.5 Variography

Variography aims to assess the spatial continuity of grade for an element of interest and ultimately helps guide the definition of parameters for the interpolation of Mineral Resources. The selected approach, inverse distance squared weighting (IDW^2), is a linear geostatistical estimation method that requires input parameters to limit the size of the search neighbourhood (via a defined search ellipsoid) for each point to be interpolated within the block model. Downhole and directional variography for the Project were run also using Isatis.neo software.

Representative normal score variograms for Domain 100 are provided in Figure 14.6.5, with final variogram model parameters also summarized in Table 14.6.4. Unfortunately, the variograms for Domains 200 and 300 were very noisy and did not produce meaningful data from which to draw geostatistical ranges and/or parameters, likely due to sparsity of data samples. As such, the parameters for Domain 100 have been applied to all three mineralized zones.

It should be noted that the indicated ranges were only used as a guide in the selection of maximum search ellipsoid distances for Indicated and Inferred Resource categories, in conjunction with geological information and other statistical factors such as average drill hole spacing.

Figure 14.6.5 Representative Normal Score Variogram for Au (g/t), Nyam Deposit



Source: DRA, 2026

Table 14.6.4 Variogram Model Parameters, Nyam Deposit

Domain	100	200	300
Nugget	0.3	0.3	0.3
Azimuth	30	30	30
Dip	80	80	80
Pitch	0	0	0
Structure 1			
Sill	0.44	0.44	0.44
Range 1	25	25	25
Range 2	15	15	15
Range 3	6	6	6
Type	Sph	Sph	Sph
Structure 2			
Sill	0.26	0.26	0.26
Range 1	105	105	105
Range 2	70	70	70
Range 3	30	30	30
Type	Sph	Sph	Sph

14.6.6 Mineral Resource Estimate

Gold is the only commodity of economic interest at the Project and hence was used to establish the relevant interpolation parameters.

Following exploratory data and geostatistical analyses using Isatis.neo software, the block model was built, and subsequent grade and tonnage estimates were computed in MinePlan 3D.

Block Model

A single block model was constructed for the mineralized domains at Nyam to capture a variety of data types including the relevant mineralized domain codes, estimated block grades, density values, percentage of material beneath the topographic surface, the closest / furthest / average distances between informing composites, initial resource category as determined by multiple-pass interpolation and finalized resource category as determined by the resource modeller.

Relevant block model definition parameters are summarized in Table 14.6.5.

Table 14.6.5 Block Model Definition Parameters, Nyam Deposit

Description	Value
Model Dimension X (m)	1,360
Model Dimension Y (m)	3,600
Model Dimension Z (m)	498
Origin X (Easting)	529,134.44
Origin Y (Northing)	636,083.81
Origin Z (Lower Elev.)	-280
Rotation (º)	30
Block Size X (m)	5
Block Size Y (m)	5
Block Size Z (m)	6

Search Strategy and Interpolation

Block values were estimated for each individual mineralized domain using the generated composites and the inverse distance weighted (IDW^2) method. The set of search parameters used in the multi-pass interpolations, derived mainly from geological information and interpreted continuity with support from variography and statistical factors such as average drill hole spacing, are summarized by estimation domain in Table 14.6.6.

Table 14.6.6 Inverse Distance Weighted (IDW^2) Interpolation Parameters Summary for the Nyam Deposit Block Model

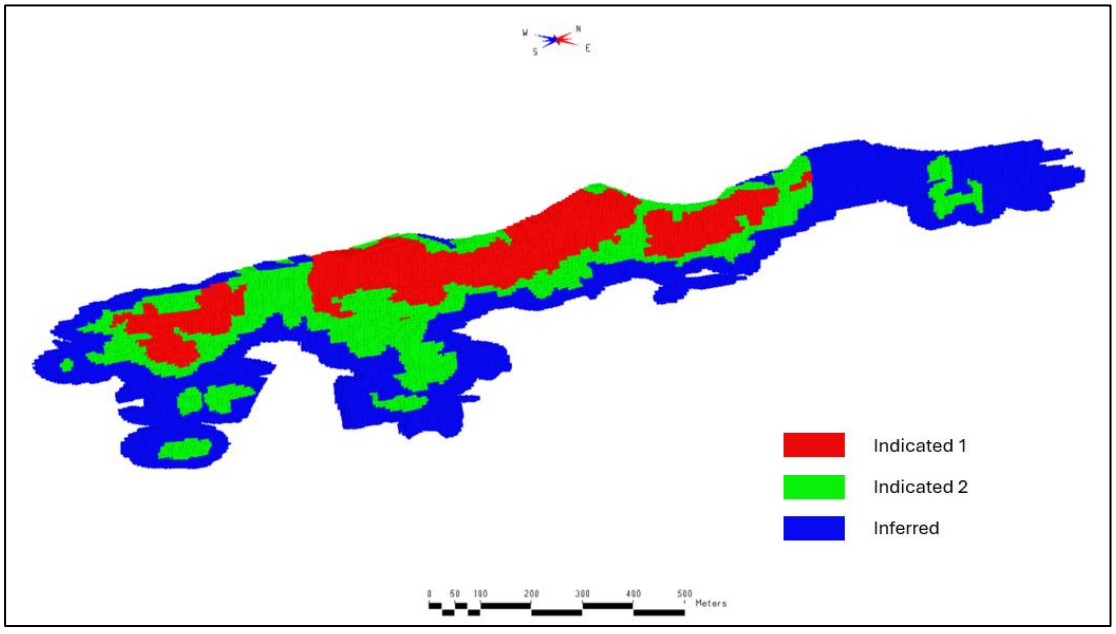
Domain	Pass	Estimation Method	Min Samples	Max Samples	Max. Samples /DDH	Major Axis	Minor Axis	Vert Axis
All Zones	1	IDW^2	6	12	2	55	15	35
	2	IDW^2	4	12	2	70	20	50
	3	IDW^2	4	12	3	130	38	88

Mineral Resource Classification

The Mineral Resources reported herein for the Nyam deposit have been classified into Indicated and Inferred categories. This classification is based on the interpreted geological and grade continuity of the observed mineralization.

Primary categorization was based on multiple-pass IDW^2 interpolation which employed increasing search ellipsoid ranges (refer back to Table 14.6.6). Category smoothing was manually employed to create a more coherent and geologically consistent classification scheme. A 3D orthographic view of the final block classification is provided in Figure 14.6.6.

Figure 14.6.6 3D Orthographic View of the Final Block Classification, Nyam Deposit



Source: DRA, 2026

14.6.7 Mineral Resource Statement

The Mineral Resource Estimate statement for the Nyam deposit prepared by DRA is summarized in Table 14.6.7. Additional details on mining and processing modifying factors are also provided in the corresponding footnotes. The Resources have been reported using a constraining resource pit at a gold price of $3,200/oz.

Table 14.6.7 Mineral Resource Estimate – Effective Date of 6 October 2025

Zone	Classification	Tonnes '000	Au Grade g/t	Contained Au ounces
Nyam	Indicated	13,458	0.66	287,000
	Inferred	5,471	0.68	120,000

Notes for Mineral Resource Estimate:

1. Canadian Institute of Mining Metallurgy and Petroleum (CIM) definition standards were followed for the resource estimate.
2. The effective date of the Resource is 6 October 2025.
3. All figures are rounded to reflect the relative accuracy of the estimate and numbers may not add due to rounding.
4. The resource model used Inverse Distance squared (ID^2) grade estimation within a three-dimensional block model with mineralized zones defined by wireframed solids and constrained by a pit shell. Validations were completed using Ordinary Kriging (OK).
5. Open pit cut-off grades varied from 0.1 to 0.2 g/t Au based on mining and processing costs as well as the recoveries in different weathered material.
6. A $3,200/ounce gold price was used to determine the cut-off grade.
7. Metallurgical recovery of 85% was applied to oxide and transition mineralization, and 91.7% for fresh mineralization.
8. The pit optimization considered the following costs: mining cost based on mineralization type of $1.97/tonne for oxide, $2.62/tonne for transition, and $3.15/tonne for fresh; waste mining costs of $1.64/tonne for oxide, $2.34/tonne for transition, and $2.87/tonne for fresh; processing and G&A costs assumed of $8.74/tonne for oxide, $8.49/tonne for transition, and $19.29/tonne for fresh.
9. Average densities of mineralized material varied between 1.53 and 2.15 g/cm^3 for oxide, 1.86 and 2.38 g/cm^3 for transition, and 2.48 and 2.74 g/cm^3 for fresh rock. Average densities of waste rock varied between 1.45 and 1.77 g/cm^3 for oxide, 1.81 and 2.15 g/cm^3 for transition, and 2.45 and 2.74 g/cm^3 for fresh rock
10. Optimization pit slope angles varied by deposit and mineralized area, with an overall strip ratio including all pits of 3.35.
11. Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have economic viability.
12. The resource estimate was prepared by Ryan Wilson, P.Geo of DRA Americas Inc. in accordance with NI 43-101. This individual is an independent qualified person (QP) as defined by NI 43-101.
13. As of the Report's date, the QPs, to the best of their knowledge, are not aware of any metallurgical, environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other risk factors that might materially affect the estimate of Mineral Resources.

14.6.8 Block Model Validation

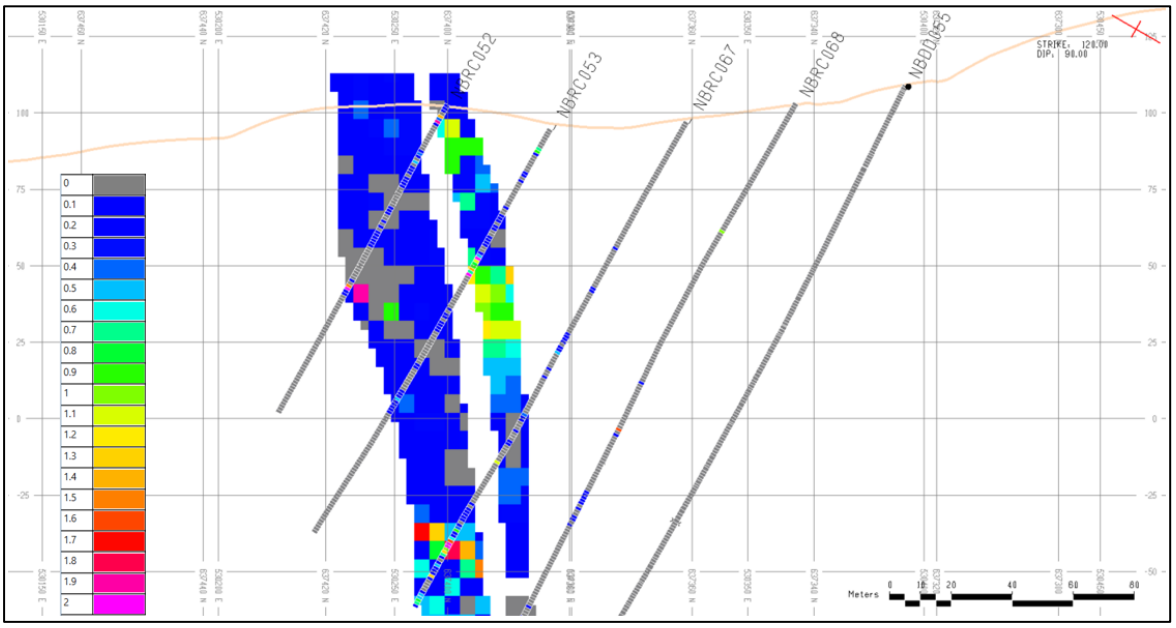
The current Nyam deposit block model has been validated by DRA using a combination of visual inspection and statistical comparisons, including:

- Visual comparison of assays and block grades.
- Inspection of generated swath plots.
- Alternative interpolation methods.

Visual Inspection

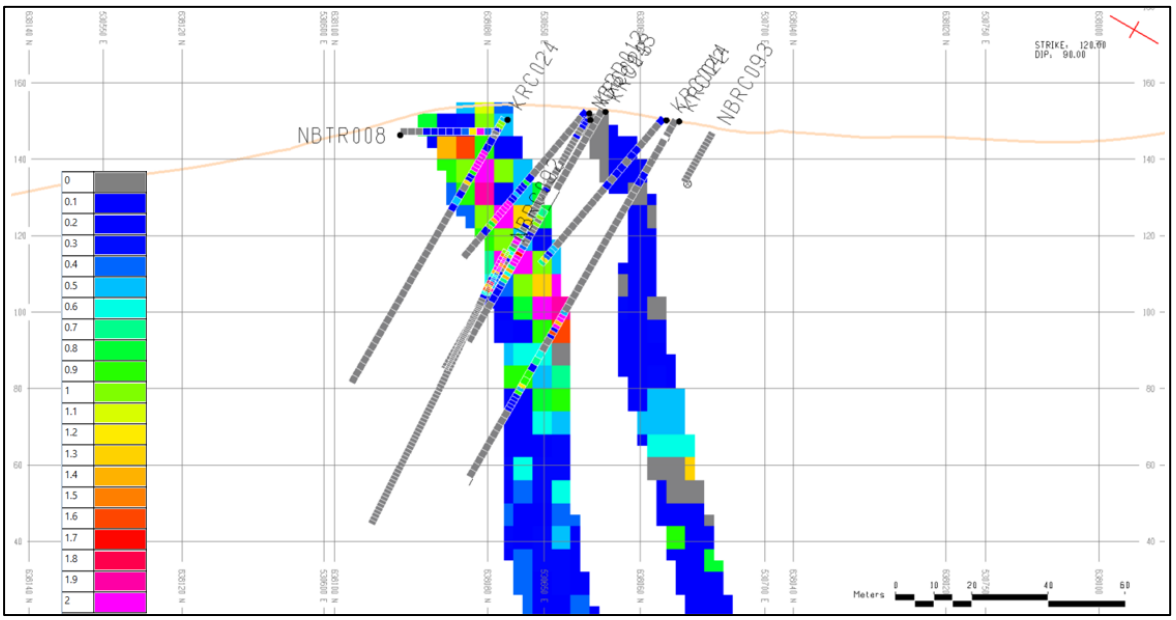
Estimated blocks and drill hole intercepts at the Nyam deposit were reviewed both on 2D vertical sections and level plans, and interactively within MinePlan 3D software. The block grades respect assay grades appropriately throughout the deposit. Representative northwest-southeast vertical sections through the heart of the deposit are shown in Figure 14.6.7 and Figure 14.6.8.

Figure 14.6.7 Comparison of Assay and Block Grades on Representative Vertical Section (675N), Nyam Deposit



Source: DRA, 2026

Figure 14.6.8 Comparison of Assay and Block Grades on Representative Vertical Section (1450N), Nyam Deposit

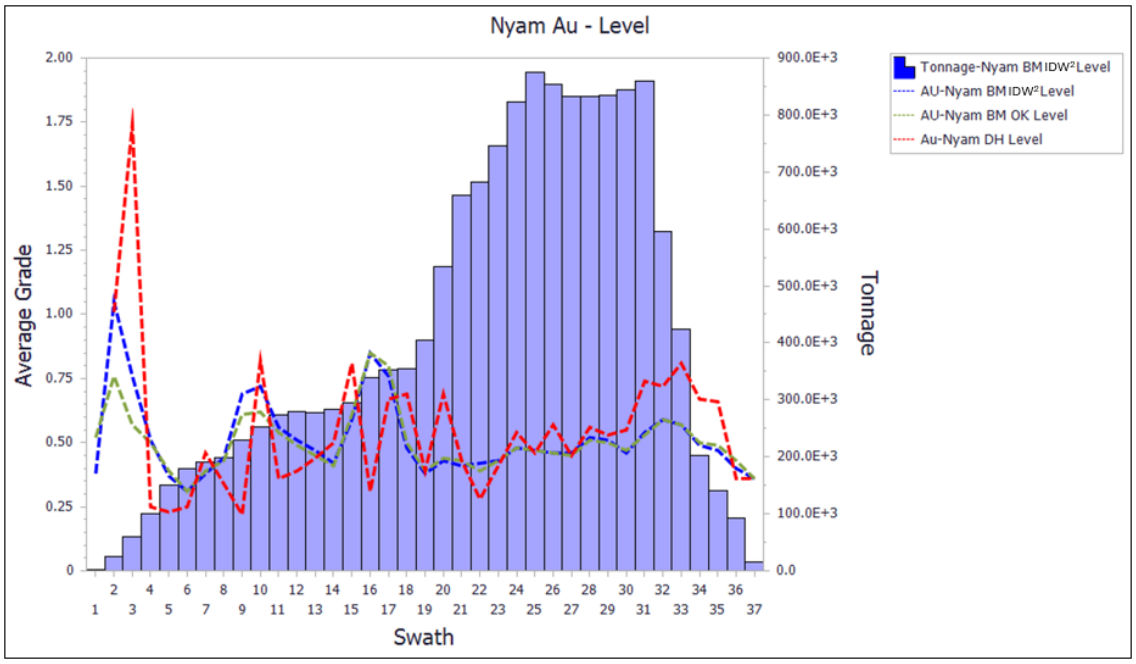


Source: DRA, 2026

Swath Plots

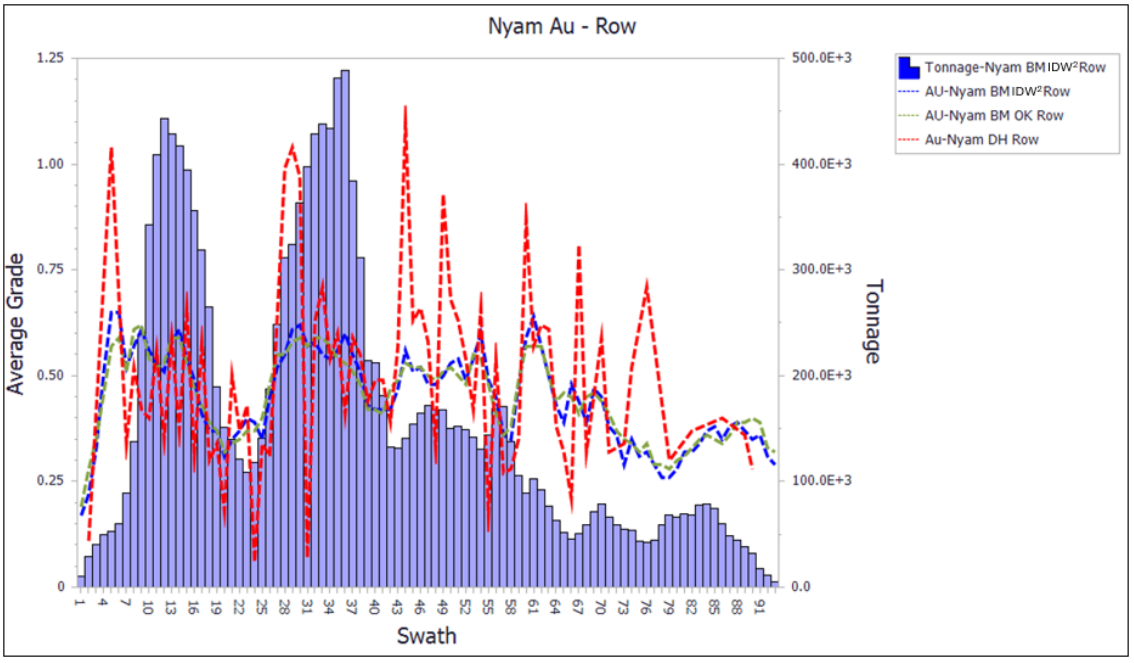
Average composite and block grades were compared by means of swath plots generated on X, Y and Z planes crossing the main trend of the block model defined in Table 14.6.8. The resultant comparative sections for the levels, rows and columns are provided in Figure 14.6.9 to Figure 14.6.11, respectively. Analysis of the plots shows that estimated block grades suitably match those of the 1 m composite data, with an expected minor amount of smoothing. Moreover, alternative interpolation methods, including inverse distance weighting (IDW^2) and ordinary kriging (OK), follow similar trends.

Figure 14.6.9 Swath Plot for Au (g/t) – By Level – 1 m Composites vs. Block Grades by IDW² and OK Interpolation Methods, Nyam Deposit



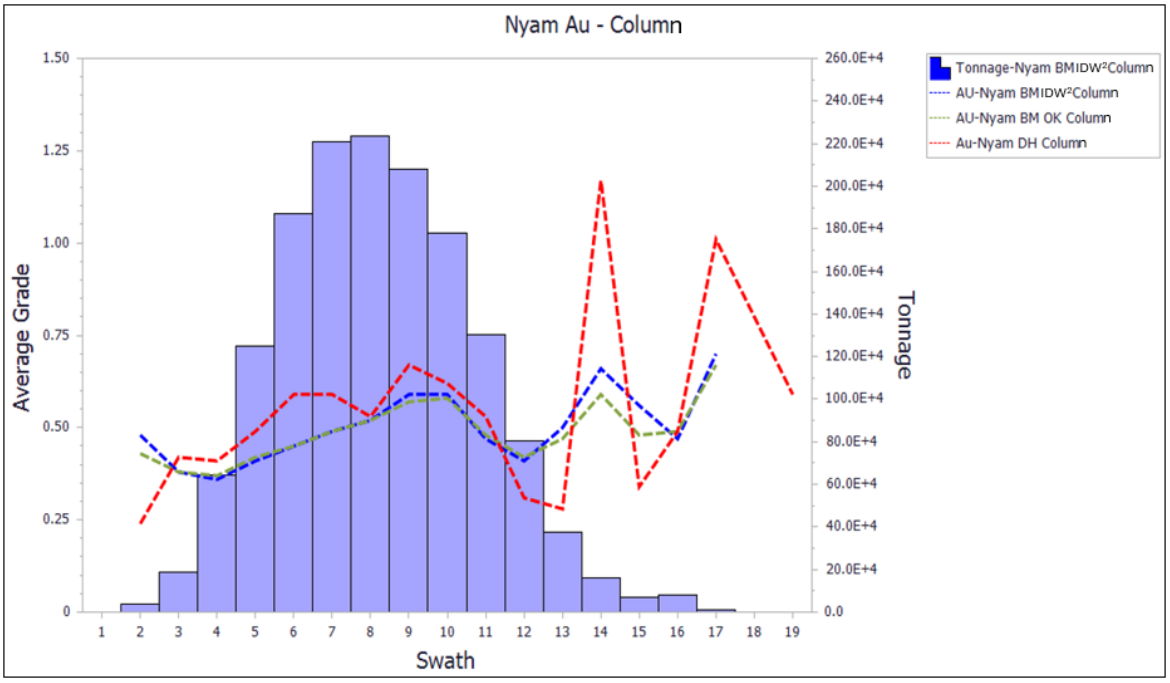
Source: DRA, 2026

Figure 14.6.10 Swath Plot for Au (g/t) – By Row – 1 m Composites vs. Block Grades by IDW² and OK Interpolation Methods, Nyam Deposit



Source: DRA, 2026

Figure 14.6.11 Swath Plot for Au (g/t) – By Column – 1 m Composites vs. Block Grades by IDW² and OK Interpolation Methods, Nyam Deposit



Source: DRA, 2026

Alternative Interpolation Methods

As shown in Figure 14.6.9 to Figure 14.6.11 above, alternative interpolation methods, including inverse distance weighting (IDW²) and ordinary kriging (OK) were run as a comparison on the global model. As such, the results of this comparison serve as a global bias check and are summarized in Table 14.6.8. The correlation between the models is considered acceptable by DRA, with global tonnes, grades and ounces yielding overall percent differences of less than 4% between interpolation types.

Table 14.6.8 Comparison of IDW² and OK Interpolation Methods, Nyam Block Model

Cutoff (g/t)	IDW²			OK					
	Tonnes	Grade	Ounces	Tonnes	% Diff	Grade	% Diff	Ounces	% Diff
0	28,463,407	0.55	503,316	29,428,930	3.4	0.53	-3.6	501,466	-0.37

14.7 Kwakyekrom Deposit

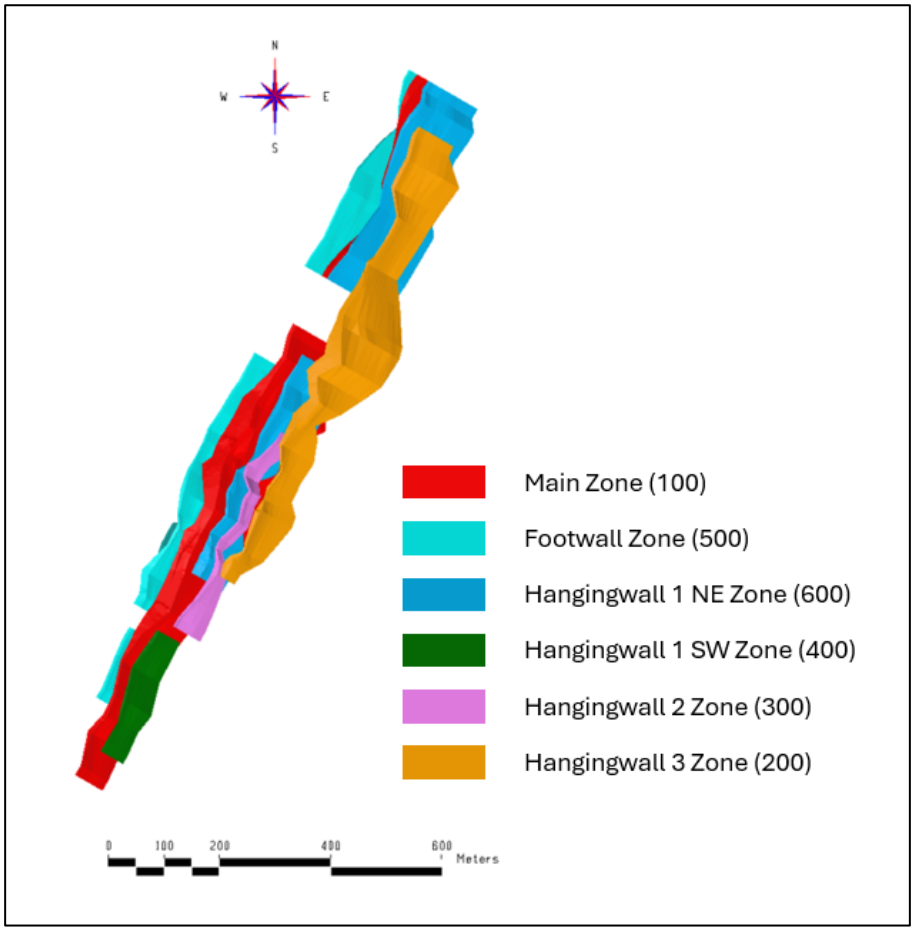
14.7.1 Three-Dimensional (3D) Modelling

Newcore provided DRA with an initial set of wireframes for a total of six mineralized domains at the Kwakyekrom deposit. Following review of the approach and methodology used to produce these wireframes, DRA conducted an independent review of the interpreted zones in 2D and 3D.

A new set of wireframes was generated by DRA using an explicit approach in MinePlan 3D software. The modelling was dominantly driven by grade continuity and supported by geological controls (e.g., known mineralized shear zones).

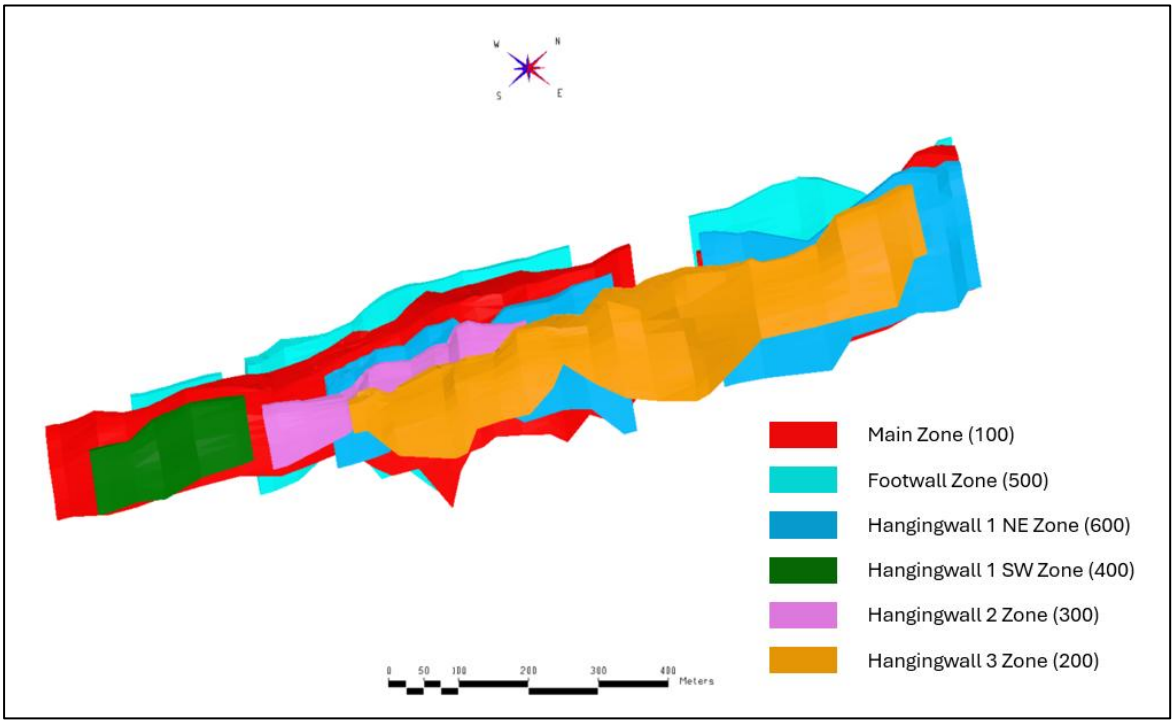
The new mineralized domains were comparable to the initial set, with small adjustments made according to interpreted continuity by the current QP. Changes were discussed with Newcore in order to finalize the set of wireframes used in the Mineral Resource Estimate (Figure 14.7.1 and Figure 14.7.2).

Figure 14.7.1 3D Plan View (Looking Down) of Kwakyekrom Mineralized Domains



Source: DRA, 2026

Figure 14.7.2 3D Orthographic View of Kwakyekrom Mineralized Domains



Source: DRA, 2026

14.7.2 Descriptive Statistics

Data was coded according to the modelled wireframes, then analyzed statistically to determine the associated data distributions; this is an important step for future comparisons to composited and estimated block grade data for validation purposes. The data was imported into the Isatis.neo software package for statistical and geostatistical calculations.

It is noteworthy that uncommon instances of unsampled intervals within the solids, mostly related to internal waste, were replaced with zero values as grades.

Basic descriptive statistics were calculated for the raw data samples contained within each of the mineralized envelopes (i.e., zone intercepts) at the Kwakyekrom deposit. These results are summarized by zone below in Table 14.7.1.

Table 14.7.1 Summary of Basic Descriptive Statistics for Raw Data Samples by Mineralized Domain, Kwakyekrom Deposit

Domain	Variable	Count	Mean	Variance	Standard Deviation	CoV	Minimum	Maximum
100	Au (g/t)	2,012	0.44	0.73	0.85	1.96	0.00	13.18
200	Au (g/t)	263	0.23	0.18	0.42	1.81	0.00	3.49
300	Au (g/t)	64	0.17	0.05	0.22	1.29	0.00	1.11
400	Au (g/t)	389	0.36	0.50	0.71	1.94	0.00	6.13
500	Au (g/t)	740	0.29	0.35	0.59	2.06	0.00	7.44
600	Au (g/t)	348	0.34	0.48	0.69	2.03	0.00	6.13

14.7.3 Grade Capping

Grade capping is used to limit the spatial extrapolation of occasional isolated high grades in the resource model estimates. Capping analyses undertaken include the use of histograms, log probability plots and ranked composites (outlier analysis).

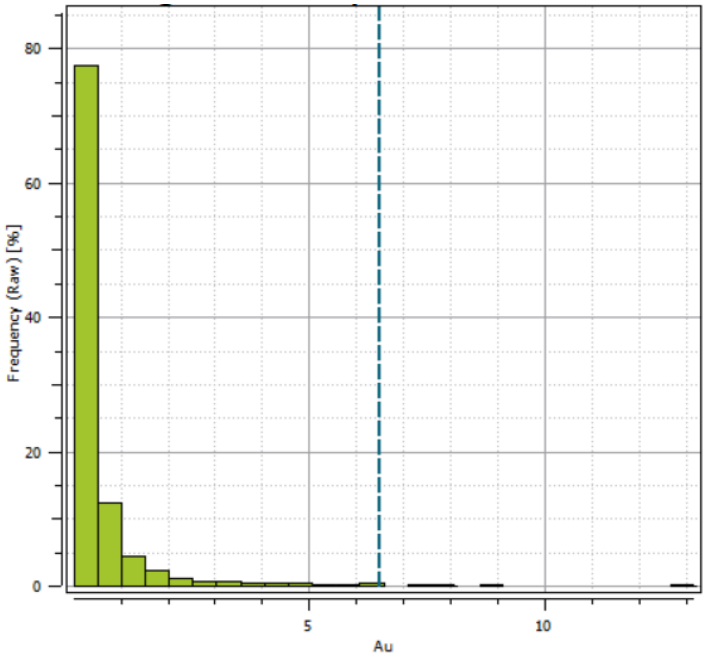
Histograms were used to search for distinct breaks in grade distributions in tandem with log probability plots, which generally show clear inflection points at the selected capping value. Outliers were also examined by means of ranked composites and observing the effect of sequential capping on the coefficient of variation (CoV) of the remaining data.

The final selected capping grades used in the resource estimate are summarized along with a subset of basic descriptive statistics in Table 14.7.2. Representative histograms and log probability plots for the various mineralized zones are also provided in Figure 14.7.3 and Figure 14.7.4, respectively.

Table 14.7.2 Summary of Capping Grades by Mineralized Domain, Kwakyekrom Deposit

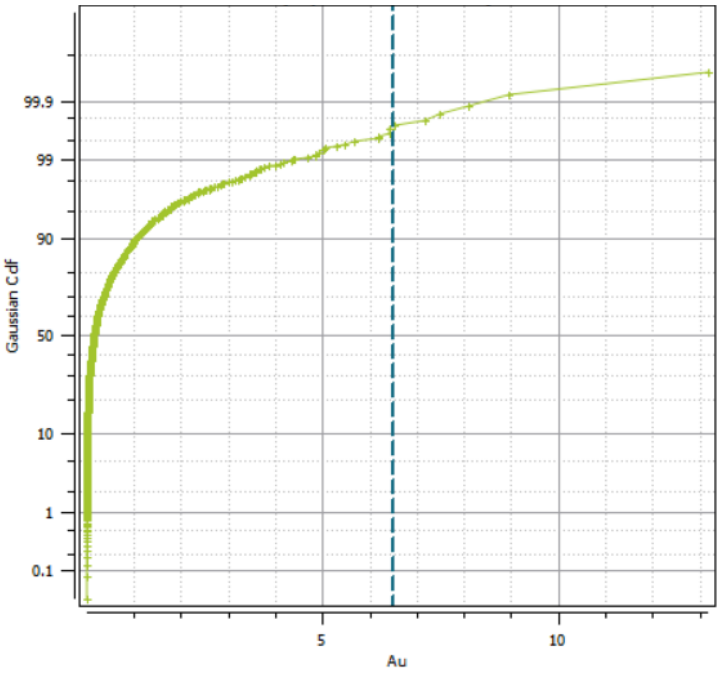
Domain	Variable	Uncapped Mean	Uncapped CoV	Capping Grade	Capped Comps #	Capped Comps %	Capped Mean	Capped CoV	Metal Loss %
100	Au (g/t)	0.44	1.96	6.50	6	0.30	0.43	1.84	1.41
200	Au (g/t)	0.23	1.81	-	-	-	-	-	-
300	Au (g/t)	0.17	1.29	-	-	-	-	-	-
400	Au (g/t)	0.36	1.94	-	-	-	-	-	-
500	Au (g/t)	0.29	2.06	4.00	3	0.41	0.28	1.87	2.58
600	Au (g/t)	0.34	2.03	-	-	-	-	-	-

Figure 14.7.3 Representative Histogram – Domain 100, Kwakyekrom Deposit



Source: DRA, 2026

Figure 14.7.4 Representative Log Probability Plot – Domain 100, Kwakyekrom Deposit



Source: DRA, 2026

14.7.4 Compositing

Drill hole intercepts through the interpreted mineralized domains at the Kwakyekrom deposit were composited to 1 m fixed length intervals, with a 0.3 m tolerance to merge shorter intervals along wireframe edges or resulting from unsampled / missing intervals. The selected composite length was based on the systematic sampling used historically by geologists to focus on 1 m intervals through the target mineralized zones; this is also supported by statistical analysis with mean and median lengths of 1.05 m and 1.0 m, respectively. Descriptive statistics for the composited data within wireframes (i.e., zone intercepts) are summarized by zone below in Table 14.7.3.

Table 14.7.3 Summary of Basic Descriptive Statistics for 1 m Composited Data at the Kwakyekrom Deposit

Domain	Variable	Count	Mean	Variance	Standard Deviation	CoV	Minimum	Maximum
100	Au (g/t)	2,012	0.43	0.63	0.79	1.84	0.00	6.50
200	Au (g/t)	277	0.23	0.17	0.42	1.80	0.00	3.49
300	Au (g/t)	73	0.16	0.04	0.21	1.31	0.00	1.11
400	Au (g/t)	41	0.36	0.33	0.57	1.57	0.00	2.89
500	Au (g/t)	740	0.28	0.27	0.52	1.87	0.00	4.00
600	Au (g/t)	381	0.34	0.48	0.69	2.03	0.00	6.13

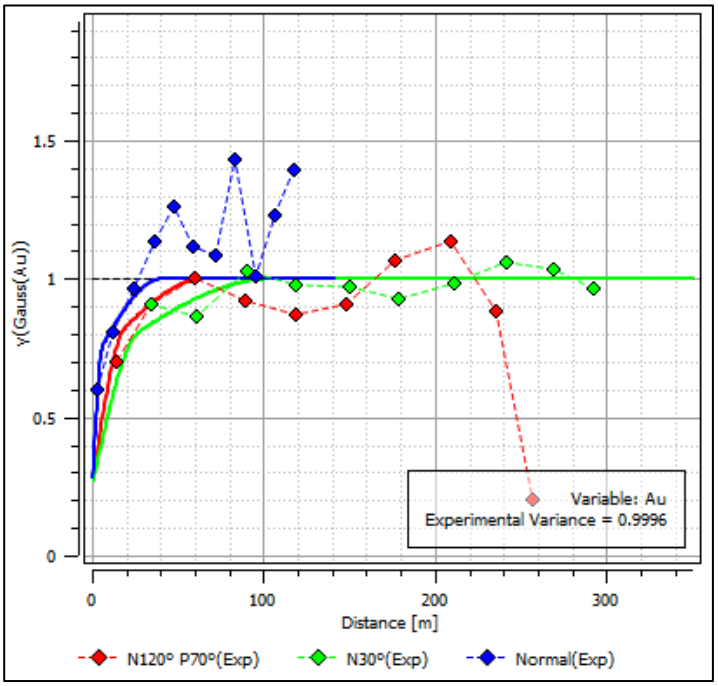
14.7.5 Variography

Variography aims to assess the spatial continuity of grade for an element of interest and ultimately helps guide the definition of parameters for the interpolation of Mineral Resources. The selected approach, inverse distance squared weighting (IDW^2), is a linear geostatistical estimation method that requires input parameters to limit the size of the search neighbourhood (via a defined search ellipsoid) for each point to be interpolated within the block model. Downhole and directional variography for the Project were run also using Isatis.neo software.

Representative normal score variograms for Domain 100 are provided in Figure 14.7.5, with final variogram model parameters also summarized in Table 14.7.4. Similar to Nyam, the variograms for the remaining smaller domains did not produce meaningful data from which to draw geostatistical ranges and/or parameters, likely due to sparsity of data samples. As such, the parameters for Domain 100 have been applied to all six mineralized zones.

It should be noted that the indicated ranges were only used as a guide in the selection of maximum search ellipsoid distances for Indicated and Inferred Resource categories, in conjunction with geological information and other statistical factors such as average drill hole spacing.

Figure 14.7.5 Representative Normal Score Variograms for Au (g/t), Kwakyekrom Deposit



Source: DRA, 2026

Table 14.7.4 Variogram Model Parameters, Kwakyekrom Deposit

Domain	All Zones
Nugget	0.25
Azimuth	30
Dip	70
Pitch	0
Sill	0.44
Range 1	27
Range 2	19
Range 3	6
Type	Sph
Sill	0.31
Range 1	105
Range 2	75
Range 3	35
Type	Sph

14.7.6 Mineral Resource Estimate

Gold is the only commodity of economic interest at the Project and hence was used to establish the relevant interpolation parameters.

Following exploratory data and geostatistical analyses using Isatis.neo software, the block model was built, and subsequent grade and tonnage estimates were computed in MinePlan 3D.

Block Model

A single block model was constructed for the mineralized domains at Kwakyekrom to capture a variety of data types including the relevant mineralized domain codes, estimated block grades, density values, percentage of material beneath the topographic surface, the closest/furthest/average distances between informing composites, initial resource category as determined by multiple-pass interpolation and finalized resource category as determined by the resource modeller.

Relevant block model definition parameters are summarized in Table 14.7.5.

Table 14.7.5 Block Model Definition Parameters, Kwakyekrom Deposit

Description	Value
Model Dimension X (m)	1,330
Model Dimension Y (m)	2,555
Model Dimension Z (m)	444
Origin X (Easting)	527,544.19
Origin Y (Northing)	633,342.19
Origin Z (Lower Elev.)	-244
Rotation (º)	25
Block Size X (m)	5
Block Size Y (m)	5
Block Size Z (m)	6

Search Strategy and Interpolation

Block values were estimated for each individual mineralized domain using the generated composites and the inverse distance weighted (IDW^2) method. The set of search parameters used in the multi-pass interpolations, derived mainly from geological information and interpreted continuity with support from variography and statistical factors such as average drill hole spacing, are summarized by estimation domain in Table 14.7.5.

Table 14.7.6 Inverse Distance Weighted (IDW^2) Interpolation Parameters Summary for the Kwakyekrom Deposit Block Model

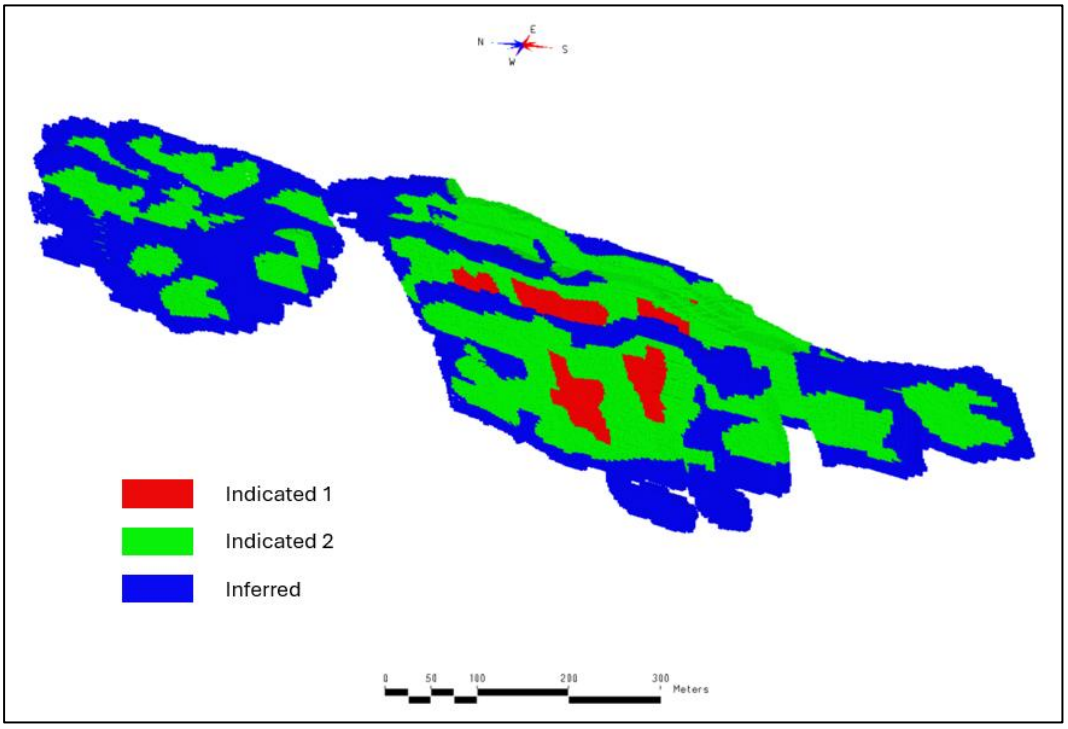
Domain	Pass	Estimation Method	Min Samples	Max Samples	Max. Samples /DDH	Major Axis	Semi Axis	Vert Axis
All Zones	1	IDW^2	6	12	2	55	15	35
	2	IDW^2	4	12	2	70	20	50
	3	IDW^2	4	12	3	130	38	88

Mineral Resource Classification

The Mineral Resources reported herein for the Project have been classified into Measured, Indicated and Inferred categories. This classification is based on the interpreted geological and grade continuity of the observed mineralization.

Primary categorization was based on multiple-pass IDW^2 interpolation which employed increasing search ellipsoid ranges (refer back to Table 14.7.7). Categorical smoothing was manually employed to create a more coherent and geologically consistent classification scheme. A 3D orthographic view of the final block classification is provided in Figure 14.7.6.

Figure 14.7.6 3D Orthographic View of the Final Block Classification, Kwakyekrom Deposit



Source: DRA, 2026

14.7.7 Mineral Resource Statement

The Mineral Resource Estimate statement for the Kwakyekrom deposit prepared by DRA is summarized in Table 14.7.7. Additional details on mining and processing modifying factors are also provided in the corresponding footnotes. The Resources have been reported using a constraining resource pit at a gold price of $3,200/oz.

Table 14.7.7 Mineral Resource Estimate - Effective date of 6 October 2025

Zone	Classification	Tonnes '000	Au Grade g/t	Contained Au ounces
Kwakyekrom	Indicated	5,447	0.52	92,000
	Inferred	1,156	0.52	19,000

Notes for Mineral Resource Estimate:

1. Canadian Institute of Mining Metallurgy and Petroleum (CIM) definition standards were followed for the resource estimate.
2. The effective date of the Resource is 6 October 2025.
3. All figures are rounded to reflect the relative accuracy of the estimate and numbers may not add due to rounding.
4. The resource model used Inverse Distance squared (ID^2) grade estimation within a three-dimensional block model with mineralized zones defined by wireframed solids and constrained by a pit shell. Validations were completed using Ordinary Kriging (OK).
5. Open pit cut-off grades varied from 0.1 to 0.2 g/t Au based on mining and processing costs as well as the recoveries in different weathered material.
6. A $3,200/ounce gold price was used to determine the cut-off grade.
7. Metallurgical recovery of 85% was applied to oxide and transition mineralization, and 91.7% for fresh mineralization.
8. The pit optimization considered the following costs: mining cost based on mineralization type of $1.97/tonne for oxide, $2.62/tonne for transition, and $3.15/tonne for fresh; waste mining costs of $1.64/tonne for oxide, $2.34/tonne for transition, and $2.87/tonne for fresh; processing and G&A costs assumed of $8.74/tonne for oxide, $8.49/tonne for transition, and $19.29/tonne for fresh.
9. Average densities of mineralized material varied between 1.53 and 2.15 g/cm^3 for oxide, 1.86 and 2.38 g/cm^3 for transition, and 2.48 and 2.74 g/cm^3 for fresh rock. Average densities of waste rock varied between 1.45 and 1.77 g/cm^3 for oxide, 1.81 and 2.15 g/cm^3 for transition, and 2.45 and 2.74 g/cm^3 for fresh rock
10. Optimization pit slope angles varied by deposit and mineralized area, with an overall strip ratio including all pits of 3.35.
11. Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have economic viability.
12. The resource estimate was prepared by Ryan Wilson, P.Geo of DRA Americas Inc. in accordance with NI 43-101. This individual is an independent qualified person (QP) as defined by NI 43-101.
13. As of the Report's date, the QPs, to the best of their knowledge, are not aware of any metallurgical, environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other risk factors that might materially affect the estimate of Mineral Resources.

14.7.8 Block Model Validation

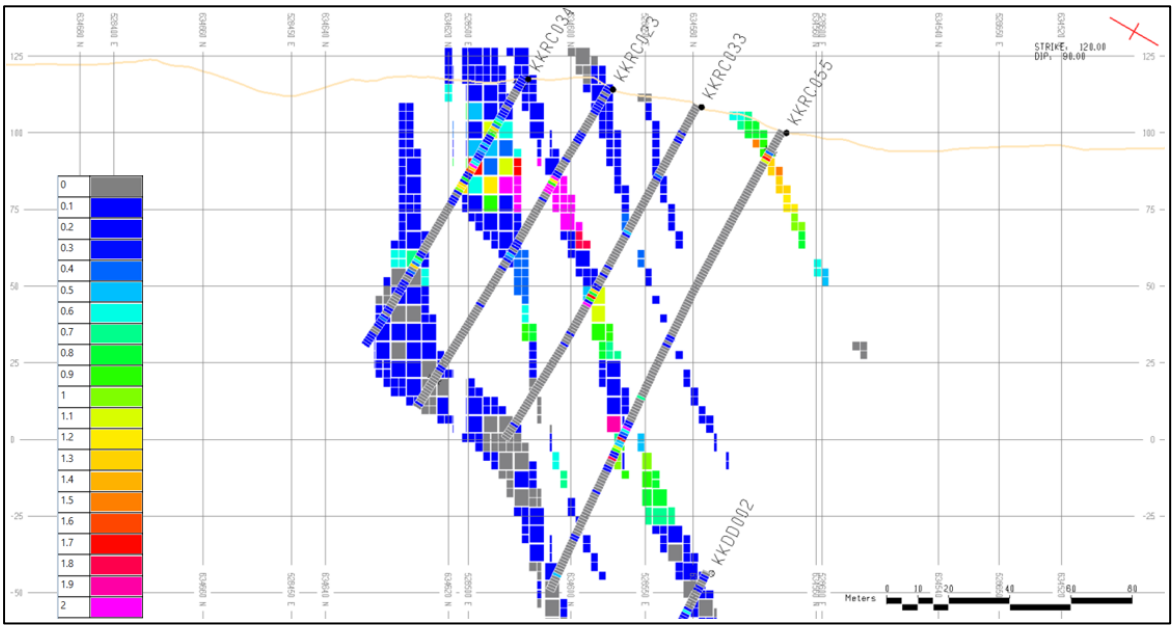
The current Kwakyekrom deposit block model has been validated by DRA using a combination of visual inspection and statistical comparisons, including:

- Visual comparison of assays and block grades.
- Inspection of generated swath plots.
- Alternative interpolation methods.

Visual Inspection

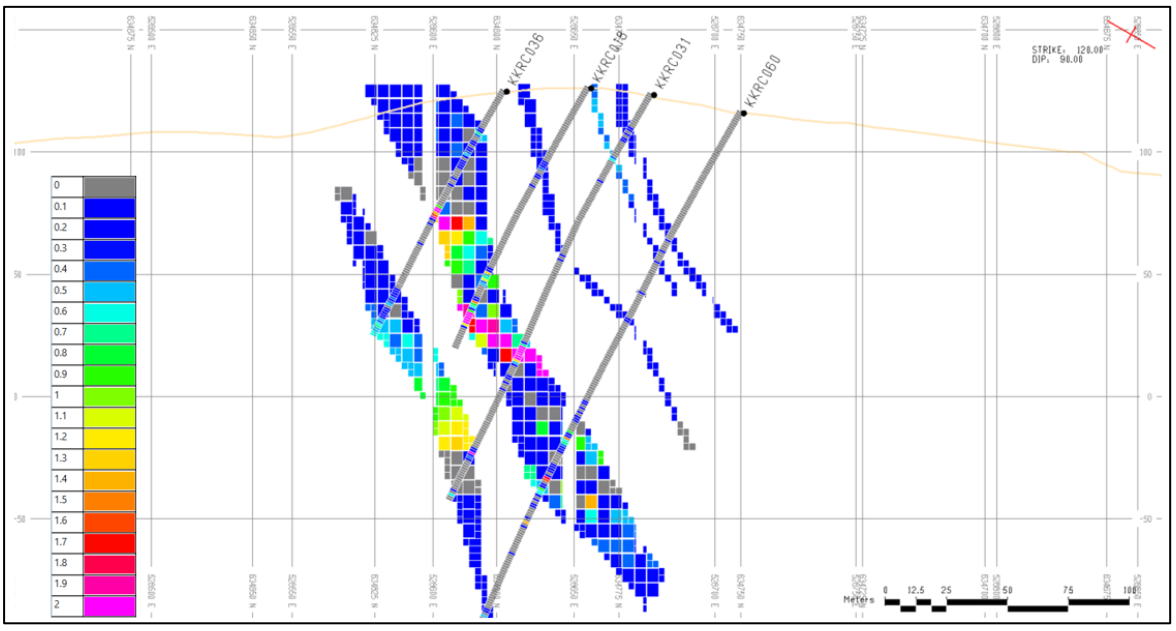
Estimated blocks and drill hole intercepts at the Kwakyekrom deposit were reviewed both on 2D vertical sections and level plans, and interactively within MinePlan 3D software. The block grades are considered to suitably respect assay grades; representative northwest-southeast vertical sections through the core of the deposit are shown in Figure 14.7.7 and Figure 14.7.8.

Figure 14.7.7 Comparison of Assay and Block Grades on Representative Vertical Section (460N), Kwakyekrom Deposit



Source: DRA, 2026

Figure 14.7.8 Comparison of Assay and Block Grades on Representative Vertical Section (680N), Kwakyekrom Deposit

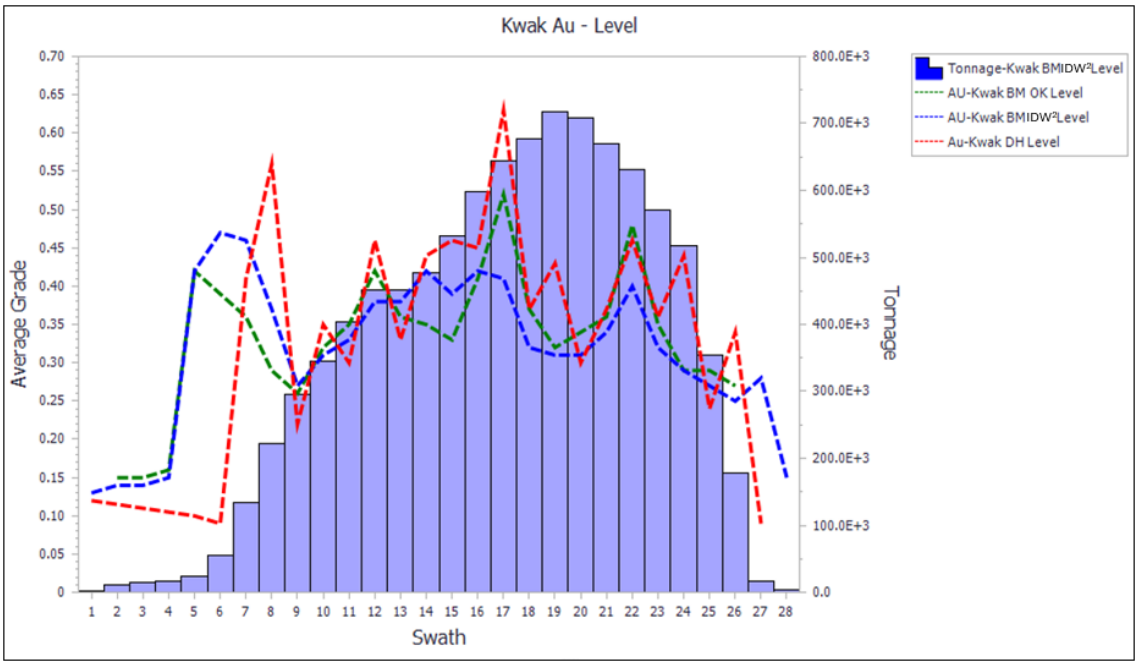


Source: DRA, 2026

Swath Plots

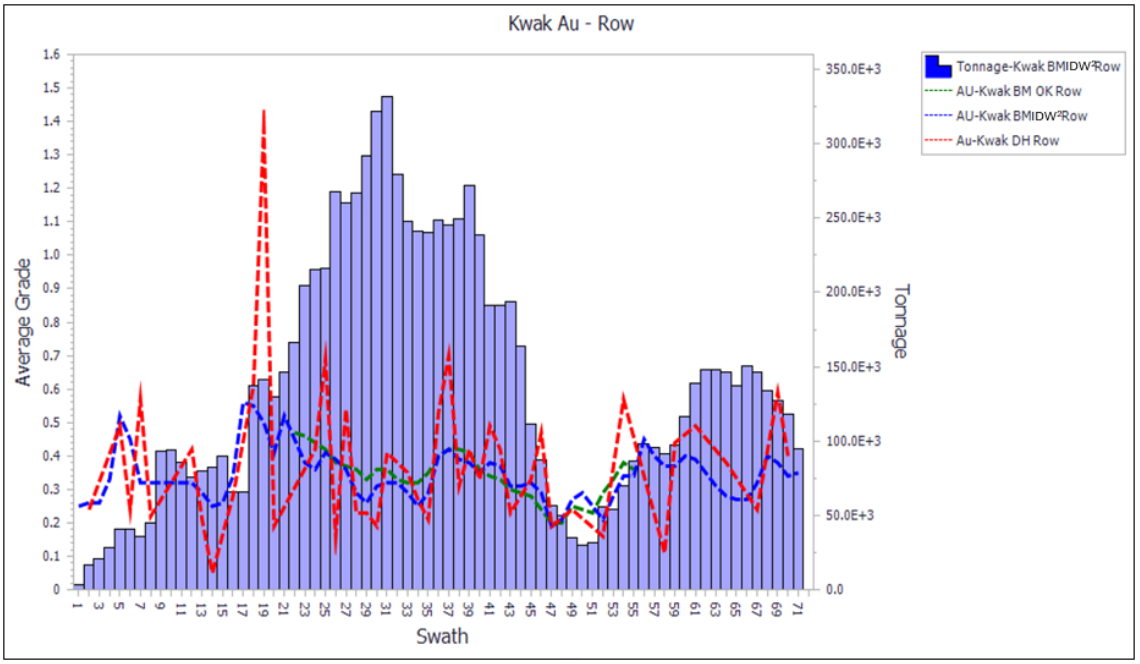
Average composite and block grades were compared by means of swath plots generated on X, Y and Z planes crossing the main trend of the block model defined in Table 14.7.8. The resultant comparative sections for the levels, rows and columns are provided in Figure 14.7.9 to Figure 14.7.11, respectively. Analysis of the plots shows that estimated block grades appropriately match those of the 1-m composite data, with an expected minor amount of smoothing. Moreover, alternative interpolation methods, including inverse distance weighting (IDW^2) and ordinary kriging (OK), follow similar trends.

Figure 14.7.9 Swath Plot for Au (g/t) – By Level – 1 m Composites vs. Block Grades by IDW² and OK Interpolation Methods, Kwakyekrom Deposit



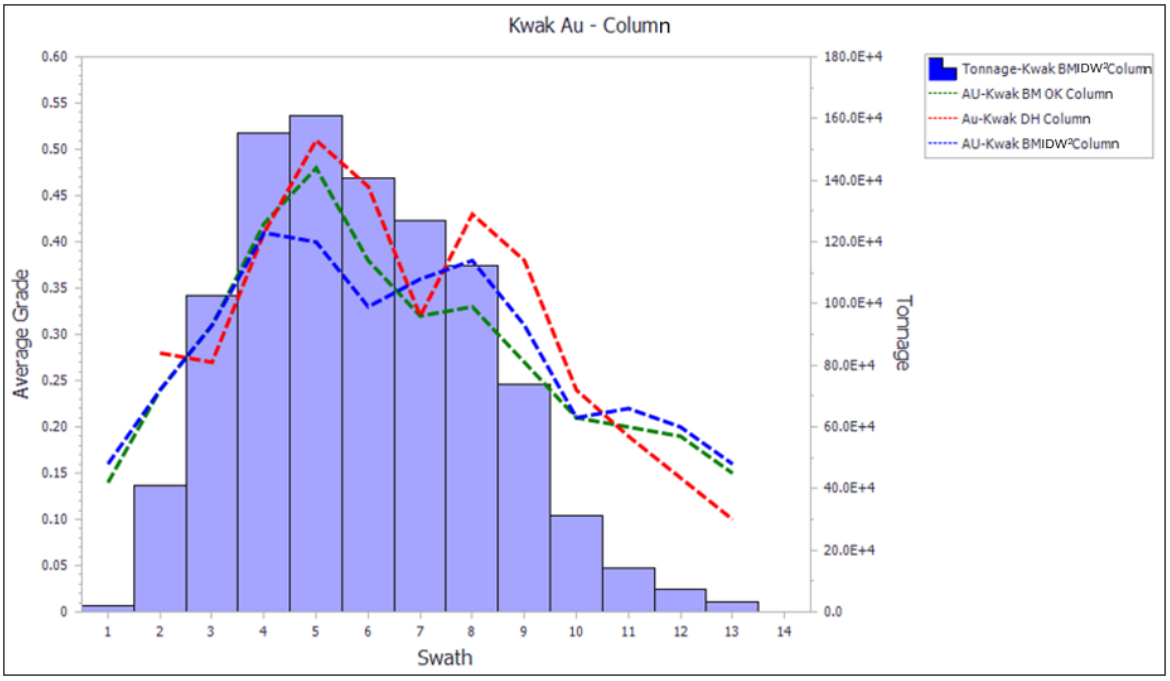
Source: DRA, 2026

Figure 14.7.10 Swath Plot for Au (g/t) – By Row – 1 m Composites vs. Block Grades by IDW² and OK Interpolation Methods, Kwakyekrom Deposit



Source: DRA, 2026

Figure 14.7.11 Swath Plot for Au (g/t) – By Column – 1 m Composites vs. Block Grades by IDW² and OK Interpolation Methods, Kwakyekrom Deposit



Source: DRA, 2026

Alternative Interpolation Methods

As shown in Figure 14.7.12 to Figure 14.7.14 above, alternative interpolation methods, including inverse distance weighting (IDW²) and ordinary kriging (OK) were run as a comparison on the global model. As such, the results of this comparison serve as a global bias check and are summarized in Table 14.7.8. The correlation between the models is considered acceptable by DRA, with global tonnes, grades and ounces yielding overall percent differences of less than 3% between interpolation types.

Table 14.7.8 Comparison of IDW² and OK Interpolation Methods, Kwakyekrom Block Model

Cutoff (g/t)	IDW²			OK					
	Tonnes	Grade	Ounces	Tonnes	% Diff	Grade	% Diff	Ounces	% Diff
0	13,403,295	0.41	176,680	13,616,197	1.59	0.40	-2.44	175,108	-0.89

14.8 Mineral Resource Statement

The Mineral Resource Estimate statement for the Project prepared by DRA is summarized in Table 14.8.1. Additional details on mining and processing modifying factors are also provided in the corresponding footnotes. The Resources for each deposit have all been reported using a constraining resource pit at a gold price of $3,200/oz. Mineral Resources are reported inclusive of Mineral Reserves; Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 14.8.1 Mineral Resource Estimate - Effective Date of 6 October 2025

Zone	Classification	Tonnes '000	Au Grade g/t	Contained Au ounces
Boin	Indicated	23,477	0.73	550,000
	Inferred	9,237	0.60	178,000
Sewum	Indicated	41,233	0.43	573,000
	Inferred	24,246	0.39	308,000
Nyam	Indicated	13,458	0.66	287,000
	Inferred	5,471	0.68	120,000
Kwakyekrom	Indicated	5,447	0.52	92,000
	Inferred	1,156	0.52	19,000
Total Indicated		**83,615**	**0.56**	**1,502,000**
Total Inferred		**40,111**	**0.49**	**626,000**

Notes for Mineral Resource Estimate:

1. Canadian Institute of Mining Metallurgy and Petroleum (CIM) definition standards were followed for the resource estimate.
2. The effective date of the Resource is 6 October 2025.
3. All figures are rounded to reflect the relative accuracy of the estimate and numbers may not add due to rounding.
4. The resource models used a combination of ordinary kriging (OK) and inverse distance weighting (IDW) grade estimation techniques within a three-dimensional block model with mineralized zones defined by wireframed solids and constrained by pits shells for Sewum, Boin, Nyam and Kwakyekrom. Validations were completed using alternative interpolation techniques for each deposit.
5. Open pit cut-off grades varied from 0.1 to 0.2 g/t Au based on mining and processing costs as well as the recoveries in different weathered material.
6. A $3,200/ounce gold price was used to determine the cut-off grade.
7. Metallurgical recovery of 85% was applied to oxide and transition mineralization, and 91.7% for fresh mineralization.
8. The pit optimization considered the following costs: mining cost based on mineralization type of $1.97/tonne for oxide, $2.62/tonne for transition, and $3.15/tonne for fresh; waste mining costs of $1.64/tonne for oxide, $2.34/tonne for transition, and $2.87/tonne for fresh; processing and G&A costs assumed of $8.74/tonne for oxide, $8.49/tonne for transition, and $19.29/tonne for fresh.
9. Average densities of mineralized material varied between 1.53 and 2.15 g/cm^3 for oxide, 1.86 and 2.38 g/cm^3 for transition, and 2.48 and 2.74 g/cm^3 for fresh rock. Average densities of waste rock varied between 1.45 and 1.77 g/cm^3 for oxide, 1.81 and 2.15 g/cm^3 for transition, and 2.45 and 2.74 g/cm^3 for fresh rock.
10. Optimization pit slope angles varied by deposit and mineralized area, with an overall strip ratio including all pits of 3.35.

11. Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
12. The mineral resource estimate was prepared by Ryan Wilson, P.Geo, Matthew Halliday, P.Geo, Schadrac Ibrango, P.Geo of DRA Americas Inc. in accordance with NI 43-101. These individuals are independent qualified persons (QP) as defined by NI 43-101.
13. As of the Report's date, the QPs, to the best of their knowledge, are not aware of any metallurgical, environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other risk factors that might materially affect the estimate of Mineral Resources.

14.9 Open-Pit Sensitivity Analysis

Cut-off grades are strongly dependent on the selected modifying factors and assumptions, which is especially important to note with recent gold price trends. As such, a sensitivity analysis was carried out for the open-pit optimization work used to report the current Resources.

For the sensitivities, all input costs (e.g., mining, processing, G&A, etc.) were held constant to those defined in the official MRE statement (Section 14.8) of this Report as a direct comparison; incremental adjustments were then made to the gold price prior to optimization. The results of this analysis, reported out of the incremental optimized pit shells are provided in Table 14.9.1 and Table 14.9.2. These pit-constrained scenarios are presented strictly for information purposes related to sensitivity of the Project to the assumed gold price. This analysis is not in addition to the current MRE and does not represent an economic analysis.

Table 14.9.1 Summary of Open-Pit Sensitivity Analysis – By Zone

	Class	Boin				Kwakyekrom				Nyam				Sewum				Total Waste MT	Total Material MT	Stripping Ratio
		Indicated		Inferred		Indicated		Inferred		Indicated		Inferred		Indicated		Inferred				
		MT	Au (g/t)	MT	Au (g/t)	MT	Au (g/t)	MT	Au (g/t)	MT	Au (g/t)	MT	Au (g/t)	MT	Au (g/t)	MT	Au (g/t)			
Gold Price ($/oz)	2,050	17	0.85	5	0.78	3	0.65	0	0.46	10	0.78	2	0.82	26	0.52	10	0.52	225	296	3.12
	2,550	21	0.78	6	0.70	4	0.59	0	0.40	11	0.72	3	0.76	32	0.48	16	0.44	295	389	3.14
	3,000	23	0.74	9	0.62	5	0.54	1	0.54	13	0.68	5	0.71	38	0.45	21	0.41	388	503	3.38
	3,050	23	0.74	9	0.61	5	0.54	1	0.53	13	0.67	5	0.70	39	0.44	22	0.41	405	523	3.43
	3,200*	23	0.73	9	0.60	5	0.52	1	0.52	13	0.66	5	0.68	41	0.43	24	0.39	414	538	3.35
	3,350	24	0.72	10	0.59	6	0.51	1	0.51	14	0.66	6	0.67	42	0.43	25	0.39	432	559	3.38
	3,500	24	0.72	10	0.59	6	0.51	2	0.51	14	0.64	6	0.65	43	0.43	27	0.38	447	580	3.36

* Selected gold price for reporting the current MRE

Table 14.9.2 Summary of Open-Pit Sensitivity Analysis – Global

	Class	Indicated			Inferred			Total Waste MT	Total Material MT	Stripping Ratio
		MT	Au (g/t)	Oz Au	MT	Au (g/t)	Oz Au			
Gold Price ($/oz)	2,050	55	0.67	1,190,000	17	0.63	340,000	225	296	3.12
	2,550	68	0.62	1,350,000	26	0.55	460,000	295	389	3.14
	3,000	79	0.58	1,460,000	36	0.51	590,000	388	503	3.38
	3,050	81	0.57	1,480,000	37	0.50	600,000	405	523	3.43
	3,200	84	0.56	1,500,000	40	0.49	630,000	414	538	3.35
	3,350	86	0.55	1,530,000	42	0.48	650,000	432	559	3.38
	3,500	88	0.55	1,560,000	44	0.47	670,000	447	580	3.36

15.0 MINERAL RESERVE ESTIMATES

15.1 Introduction

The Mineral Reserve Estimate for The Enchi Gold Project has been prepared in accordance with the disclosure requirements of National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* and the guidelines set out in the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014). The estimate defines the economically mineable portion of the Indicated Mineral Resources, incorporating all relevant modifying factors.

Only Indicated Mineral Resources have been considered for conversion to Mineral Reserves, and these have been reclassified as Probable Reserves based on the level of confidence and the application of modifying factors. Inferred Mineral Resources are considered too speculative geologically to have economic considerations applied that would enable them to be categorized as Mineral Reserves.

The reserve estimation is supported by detailed open pit mine designs, including pit optimization, phase designs, haul road layouts, and waste dump configurations. The selected mining method is conventional truck-and-shovel open pit mining, with design parameters informed by geotechnical investigations and operational considerations. The Project currently includes four deposits which contain Probable Reserves: Boin, Sewum, Nyam and Kwakyekrom. The location of the deposits and process plant location are shown in Figure 18.1.1.

15.2 Open Pit Mineral Reserves

The Mineral Reserves presented herein reflect the application of modifying factors, including mining dilution and recovery, metallurgical recoveries, infrastructure requirements, environmental and permitting considerations, and applicable legal and social factors. Economic assumptions applied in the estimation include a gold price of $3,200/oz, mining and processing costs, general and administrative costs, and metallurgical recovery factors derived from test work. A cut-off grade has been determined based on these inputs and applied to define economically mineable material. Table 15.2.1 summarizes the Mineral Reserves estimation by deposit.

Table 15.2.1 Open Pit Mineral Reserves by Deposit

Probable Mineral Reserves					
Mining Area	**Material Type**	**Tonnes**	**Au Grade g/t**	**Contained Gold oz**	**Reserve Cut-off Grade g/t Au**
Boin	Oxide	2,517,000	0.77	62,000	0.14
	Transition	5,164,000	0.77	128,000	0.18
	Fresh	8,225,000	0.90	239,000	0.27
	Total	**15,906,000**	**0.84**	**429,000**	
Sewum	Oxide	12,059,000	0.41	159,000	0.14
	Transition	8,760,000	0.49	138,000	0.18
	Fresh	5,394,000	0.63	110,000	0.27
	Total	**26,213,000**	**0.48**	**406,000**	
Nyam	Oxide	1,026,000	0.70	23,000	0.18
	Transition	2,532,000	0.67	55,000	0.22
	Fresh	3,635,000	0.83	97,000	0.30
	Total	**7,193,000**	**0.76**	**175,000**	
Kwakyekrom	Oxide	513,000	0.48	8,000	0.18
	Transition	811,000	0.61	16,000	0.22
	Fresh	654,000	0.98	21,000	0.34
	Total	**1,978,000**	**0.70**	**44,000**	
Total	Oxide	16,115,000	0.49	252,000	
	Transition	17,267,000	0.61	336,000	
	Fresh	17,908,000	0.81	467,000	
	Total	**51,290,000**	**0.64**	**1,055,000**	

Notes for Mineral Reserve Estimate:

1. CIM definition standards were followed for the reserve estimate.
2. The effective date of the Mineral Reserve Estimate is 23 June 2026.
3. All figures are rounded to reflect the relative accuracy of the estimate and numbers may not add due to rounding.
4. A $3,200 per ounce gold price was used to determine the Mineral Reserve cut-off grade.
5. Reserve cut-off grades range from 0.14 g/t Au to 0.34 g/t Au depending on the material type and deposit.
6. A mining recovery of 96.0% was assumed for all deposits.
7. Dilution has been accounted for within the block model regularization process and is considered appropriate for the deposit type, varying by deposit at approximately 6 - 7%.
8. The Mineral Reserve estimate considered the following costs: Mining cost ranged from $1.94 to $4.32/tonne mined depending on the material type and deposit; Mineral processing cost of $9.78/tonne for oxide, $13.36/tonne for transition, $20.00/tonne for fresh ore, with an additional ore transportation cost included for the satellite mining deposits of $4.09/tonne for Nyam and $3.49/tonne for Kwakyekrom; Mine site G&A cost of $1.97/tonne processed assumed for all deposits.

9. Processing recoveries are variable, dependent on material type and mining area, ranging from 80% to 95%.
10. Mineral Reserves are stated in terms of delivered tonnes; gold ounces are reported as contained and do not include allowances for processing losses.
11. The mineral reserve estimate was prepared by Grant Carlson, P. Eng of Fuse Advisors in accordance with NI 43 101. This individual is an independent qualified persons ("QP") within the meaning of NI 43-101.
12. As of the effective date of the Mineral Reserve Estimate, the QP, to the best of their knowledge, is not aware of any metallurgical, environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other risk factors that might materially affect the potential development of the Mineral Reserves.

15.3 Factors that May Affect the Mineral Reserves

The current assumptions are considered appropriate for the declaration of the Mineral Reserves; however, it is recognized that changes in these assumptions could materially impact the Mineral Reserve statement. Factors that may influence the Mineral Reserves include, but are not limited to:

- Long-term metal price assumptions.
- Foreign exchange rates.
- Geotechnical slope design assumptions for pit walls.
- Capital and operating cost estimates.
- Unforeseen dilution and/or lower-than-expected mining and processing recoveries.
- Environmental, social licence, and permitting timeline assumptions.

15.4 Block Models

Regularized block models were provided to Fuse by DRA as part of the 2026 Mineral Resource model update. The model dimensions for Boin and Sewum were 5x5x5m in size, while the model dimensions for Nyam and Kwakyekrom were 5x5x6m. The models for Boin and Kwakyekrom were received with grade fields diluted to whole block size, while the models for Sewum and Nyam contained an ore percent field and indicated grades represented only the material within the resource domains. The ore percent models were diluted to whole block size by incorporating the waste tonnage in boundary blocks into an overall diluted grade, which aligned the format across all four deposits.

These block models were reviewed and simplified for Reserve Estimation. The simplified models contained information relating to gold grade, resource category, density, lithology (oxide, transition, fresh), and topo percent indicating the volume of surface blocks below the latest topography surface.

15.5 Open Pit Optimization

Pit shells were generated utilizing Geovia Whittle's Pseudoflow algorithm. The Pseudoflow algorithm produced a series of nested pit shells based on the geotechnical parameters, mining dilution and recovery factors, operating costs and varying revenue factors. Costs and revenues were modelled into each block to define economic and non-economic blocks for processing. The optimization parameters are outlined in Section 15.5.1 and Table 15.5.1. Details of the pit optimization results are in Section 15.5.

15.5.1 Input Parameters

Geotechnical Parameters

DRA provided baseline pit Inter-Ramp Angles (IRAs) to Fuse for the Enchi Gold Project. DRA completed a geotechnical gap analysis which informed the geotechnical drilling program and additional point load and uniaxial compressive strength testing completed by Newcore. Although the geotechnical holes did not cover all deposits, the basic geotechnical information from the resource drill database, RQD measurements and targeted lab tests provides a strong basis for assessing geotechnical conditions in the various pits.

The IRAs were developed using geotechnical drilling and sampling data from the program described above, supplemented by a review of geotechnical datasets and corresponding IRAs from nearby mines and projects with similar geological and structural settings. The reference projects included:

- The Chirano Gold Mine (Asante Gold Corporation) located approximately 50 km to the north of the Enchi Gold Project and along the Bibiani Shear Zone.
- The Bibiani Gold Mine (Asante Gold Corporation) located approximately 80 km to the north of the Enchi Gold Project and along the Bibiani Shear Zone.
- The Afema Gold Project (Turaco Gold Limited) located approximately 40 kilometres south of the Enchi Gold Project on the same belt and in geologically and structurally similar terrain.

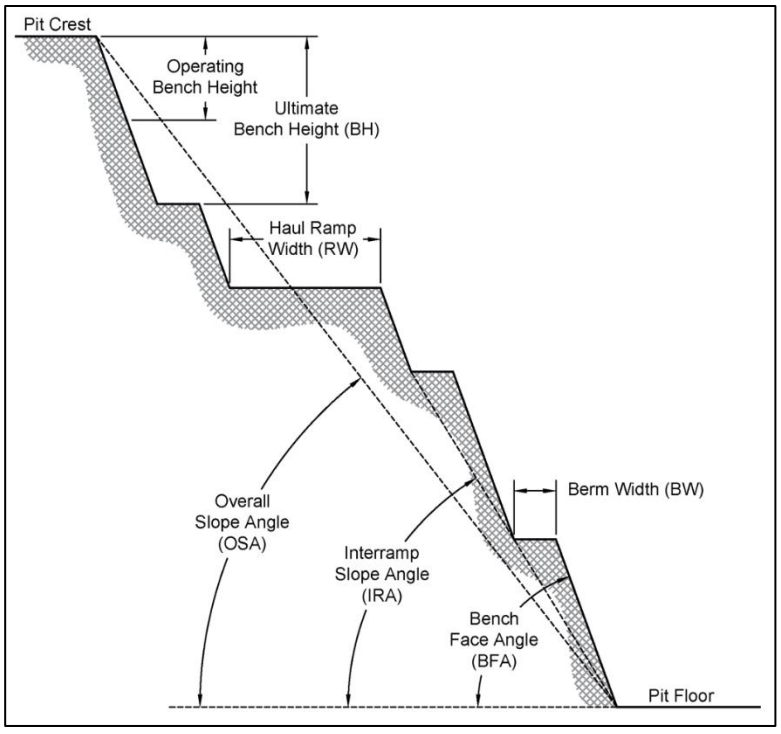
Based on the available data and comparable projects, the resulting IRAs and bench design criteria are outlined in Table 15.5.1. Figure 15.5.1 provides a schematic representation of bench configuration and terminology.

Table 15.5.1 DRA Geotechnical Pit Design Criteria

Criteria	Unit	Boin			Sewum			Nyam			Kwakyekrom		
		Oxide	Trans	Fresh	Oxide	Trans	Fresh	Oxide	Trans	Fresh	Oxide	Trans	Fresh
Bench Height	m	10	10	10	10	10	10	12	12	12	12	12	12
Bench Stacking	#	1	1	1	1	1	1	1	1	1	1	1	1
Max Bench Height	m	10	10	10	10	10	10	12	12	12	12	12	12
Bench Face Angle	deg	45	55	75	45	55	75	45	55	70	45	55	70
Bench Width	m	5	5	5	5	5	5	5	5	5	5	5	5
IRA	deg	34	40	52	34	40	52	35	42	52	35	42	52

Source: parameters provided by DRA [C8808 - Enchi - Mine Design Criteria - Rev D_260206_CB.xls] on 6 February 2026

Figure 15.5.1 Pit Slope Design Terminology



To account for operational requirements, including access to the final pit depth, Fuse modified the optimization IRAs to account for the inclusion of haul ramps along the pit walls, producing IRAs that provided a more realistic basis for pit design. Figure 15.5.2 illustrates the concept of reducing the IRA to incorporate a haul ramp. The adjusted IRAs used in the pit optimizations assume the inclusion of a single 24 m-wide haul ramp within the pit wall slopes for each of the transition and fresh rock material types.

Figure 15.5.2 Conceptual IRA Adjustment for Pit Ramps

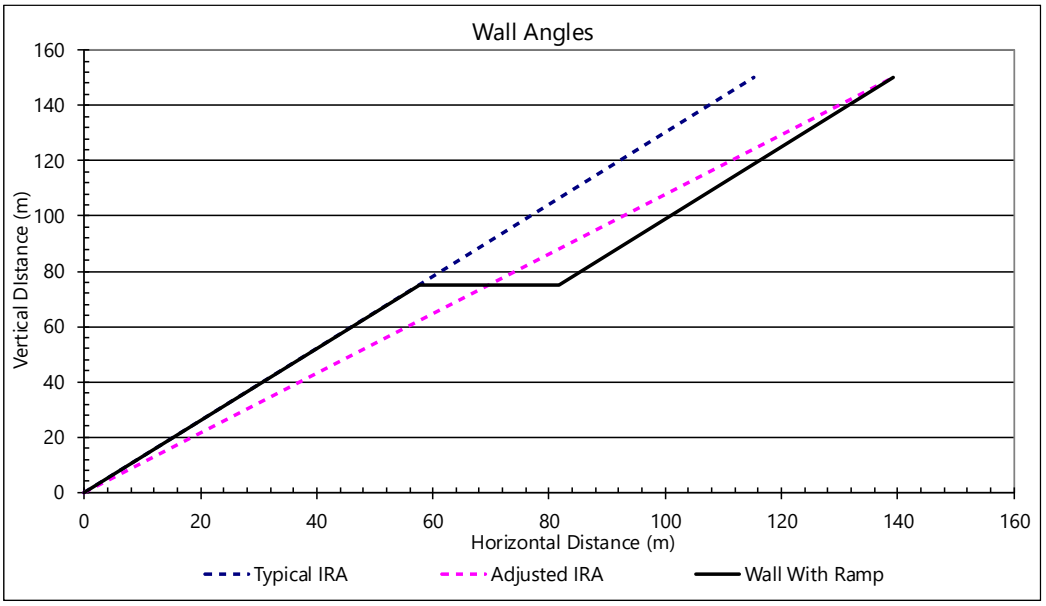


Table 15.5.2 Inter-Ramp Angles, outlines the baseline (DRA) IRAs as well as the modified (Pit Optimization) IRAs that were used in the pit optimization.

Table 15.5.2 Inter-Ramp Angles

Mining Area	Material	DRA IRA Degrees	Pit Optimization IRA Degrees
Boin	Oxide	34	34
	Transition	40	34
	Fresh	52	47
Sewum	Oxide	34	34
	Transition	40	34
	Fresh	52	47

Mining Area	Material	DRA IRA Degrees	Pit Optimization IRA Degrees
Nyam	Oxide	35	35
	Transition	42	35
	Fresh	52	46*
Kwakyekrom	Oxide	35	35
	Transition	42	35
	Fresh	52	47

*Reduced to 46 to minimize slope error within Whittle software when using 47 degrees

Mining Dilution and Recovery Parameters

A mining recovery factor of 96% was applied to all blocks above the cut-off grade to reflect the expected losses that occur during extraction. The estimation approach assumes that whole-block grade dilution is a reasonable stand-in for the dilution expected under actual mining conditions, including the effects of blast movement, material mis-routing, and mixing during loading. These assumptions are tied to the anticipated fleet configuration and the level of selectivity achievable with the loading equipment, particularly in areas where ore zones narrow.

To check that these assumptions were appropriate, each deposit was reviewed within its Revenue Factor 1 pit shell to evaluate how much waste material was being captured in blocks allocated for ore mining. Waste inclusion was estimated using the ore percent block model fields to calculate the waste tonnage present in blocks above cut-off grade. The resulting dilution values fell between 6% and 7% of total material, which aligns with expectations for mining conditions and is considered reasonable for this level of study.

Processing Recovery Parameters

Processing recovery is based on treatment through a Carbon-in-Leach (CIL) plant and was derived from laboratory test work. Test conditions to achieve these recoveries included grinding to 53 µm, lead nitrate addition, oxygen injection for all material types and kerosene addition for fresh material.

Table 15.5.3 Processing Recovery Parameters by Mining Area and Material

Mining Area	Material	Mill Recovery %
Boin	Oxide	95.0%
Boin	Transition	95.0%
Boin	Fresh	89.0%

Mining Area	Material	Mill Recovery %
Sewum	Oxide	95.0%
Sewum	Transition	95.0%
Sewum	Fresh	90.0%
Nyam	Oxide	91.0%
Nyam	Transition	91.0%
Nyam	Fresh	90.0%
Kwakyekrom	Oxide	90.0%
Kwakyekrom	Transition	90.0%
Kwakyekrom	Fresh	80.0%

Cost and Revenue Factors

A gold price of $3,200/troy oz (t.oz) was selected for the pit optimization, and a discount rate of 5% was used, consistent with standard practice in the gold mining industry.

Royalty assumptions included a 5% gross royalty to the Ghanaian Government, and a 2% NSR royalty payable to Triple Flag Precious Metals Corp (with 1% subject to a buy-back option for a lump sum payment of $3.5 million at any time held by Royal Gold Inc.). Subsequently a sliding scale royalty was implemented by the Ghanaian government and was incorporated in Section 22 Economic Analysis. A payability of 99.9% was used, with a $4.00/oz Au TC/RC/Transport cost. A summary can be found in Table 15.5.4.

Table 15.5.4 Optimization Revenue Factors

Revenue Factors	Unit	Value
Gold Price	$/oz	3,200
Royalties	$/oz	223.70
Refinery and Shipment	$/oz	4.00
Payability Reduction	$/oz	3.20
Total Selling Costs	$/oz	230.90

Mining operating costs were estimated based on an updated mine production schedule developed following the 2023 Preliminary Economic Assessment (PEA Technical Report), and costs were separated for ore and waste, with each material type further subdivided into oxide, transition, and fresh rock categories to reflect differences in material hardness, equipment productivity, and expected blasting requirements. For transition material, costs assumed that 35% of the material would require blasting.

Cost inputs were derived from multiple mining contractor quotations received in late 2025, providing current market-based rates for drilling, blasting, loading, hauling, and support activities. Secondary ore haulage costs were also included for the satellite Nyam and Kwakyekrom deposits. Table 15.5.5 summarizes the mining costs by mining location and material type.

Milling costs for each material type were derived from a trade-off study completed prior to commencing the Pre-Feasibility Study. Oxide material, being much softer, is estimated to have a lower milling cost.

Selling costs and site G&A costs were estimated based on benchmarks from comparable operations, supplemented by assessments from previous technical studies and supported by industry-standard data points. The selected inputs reflect typical cost structures observed at similar mining operations and have been cross-checked against publicly available data and historical project assumptions to ensure their reasonableness for the Project.

Table 15.5.5 Operating Costs by Mining Area and Material

Mining Area	Material	Ore Mining Cost $/t ore Mined	Waste Mining Cost $/t waste Mined	Addition Ore Haulage Cost $/t ore Mined	Mill Cost $/t ore Mined	Total Selling Costs $/oz	G&A Cost $/t
Boin	Oxide	3.04	1.95	0.00	9.78		
Boin	Transition	3.55	2.37	0.00	13.36	230.90	1.97
Boin	Fresh	4.32	3.06	0.00	20.00		
Sewum	Oxide	2.63	2.08	0.00	9.78		
Sewum	Transition	3.06	2.40	0.00	13.36	230.90	1.97
Sewum	Fresh	3.82	3.06	0.00	20.00		
Nyam	Oxide	1.94	2.19	4.09	9.78		
Nyam	Transition	2.42	2.54	4.09	13.36	230.90	1.97
Nyam	Fresh	3.05	3.14	4.09	20.00		
Kwakyekrom	Oxide	2.07	2.00	3.49	9.78		
Kwakyekrom	Transition	2.32	2.26	3.49	13.36	230.90	1.97
Kwakyekrom	Fresh	2.97	2.86	3.49	20.00		

15.6 Cut-off Grade

The break-even cut-off grade (COG) was calculated based on operating cost per tonne, metallurgical recovery, and commodity price. As described in Section 15.4.1, costs and recoveries were differentiated by mining area and material type. Accordingly, multiple cut-off grades were calculated for the Project and used for the pit optimizations. Table 15.6.1 summarizes the COGs calculated and utilized for the mining schedule and Mineral Reserve estimation.

Table 15.6.1 Cut-Off-Grades For Mineral Reserve Estimation

Mining Area	Material	Cutoff g/t
Boin	Oxide	0.14
Boin	Transition	0.18
Boin	Fresh	0.27
Sewum	Oxide	0.14
Sewum	Transition	0.18
Sewum	Fresh	0.27
Nyam	Oxide	0.18
Nyam	Transition	0.22
Nyam	Fresh	0.30
Kwakyekrom	Oxide	0.18
Kwakyekrom	Transition	0.22
Kwakyekrom	Fresh	0.34

15.7 Pit Optimization Results

Separate Whittle pit optimizations were completed for each deposit (Boin, Sewum, Nyam, Kwakyekrom). Initial runs indicated an opportunity for further subdivision, as discrete nested pit shells were identified. The geographic separation of these areas enabled independent optimization of each major pit and improved pit selection within distinct mining areas of each deposit.

This additional subdivision resulted in Boin being separated into three discrete pits (Boin Central, Boin North, Boin South), with Sewum also divided into three mining areas (Sewum North, Sewum Checkerboard, Sewum South).

Pit optimization involved evaluating the economic sensitivity of the deposits across a range of revenue factors. The final pit shell for each deposit was selected based on a review of detailed results, taking into consideration:

- Undiscounted project value.
- Mineralized material grade.
- Incremental recovered metal relative to cost.
- Incremental strip ratio.
- Practical mining widths and preliminary mine scheduling considerations.

The following sections summarizes the pit optimization results for each mining area.

15.7.1 Boin Pit Optimization Result

Boin pit optimization was divided into three discrete areas for shell selection, the results are as follows:

- Boin Central: The revenue factor 0.92 (Shell 32) was selected, as the undiscounted value curve begins to flatten, while grade declines and strip ratio increases. Shell 32 provided 358,000 contained ounces at an average grade of 0.86 g/t and a strip ratio of 5.91. These results are demonstrated in Table 15.7.1 and Figure 15.7.1

- Boin Central Interim Phase: In addition to the selection of Shell 32 for the ultimate pit limits, a lower revenue factor of 0.5 (Shell 11) was evaluated as a guiding interim phase to support improved project economics during the early years of mining. Shell 11 is characterized by a higher average grade of 0.99 g/t and a lower strip ratio of 4.52, making it a favourable target for early extraction. This shell contains approximately 64% of the total undiscounted value while comprising only 43% of the total mineralized material defined within the ultimate pit.

- Boin North: The revenue factor 0.94 (Shell 33) was selected, as the undiscounted value curve begins to flatten, while grade declines and strip ratio increases. Shell 33 provided 34,000 contained ounces at an average grade of 0.81 g/t and a strip ratio of 6.45. These results are demonstrated in Table 15.7.2 and Figure 15.7.2.

- Boin South: The revenue factor 0.94 (Shell 33) was selected. Although there was a slight increase in undiscounted project value observed at shell 35, the higher strip ratio resulted in higher mining costs and marginal incremental increases to profit margin. Shell 33 provided 36,000 contained ounces at an average grade of 0.74 g/t and a strip ratio of 7.23. These results are demonstrated in Table 15.7.3 and Figure 15.7.3.

Table 15.7.1 Boin Central Pit Optimization Results

Final Pit	RF	Undiscounted Value M$	Total Ore tonnes	Diluted Au g/t	Total Mined Metal Content Au oz	Total Waste tonnes	SR W:O	Total Material tonnes
1	0.3	$113	1,296,640	1.19	49,482	2,726,059	2.10	4,022,699
2	0.32	$120	1,414,529	1.16	52,905	3,073,090	2.17	4,487,619
3	0.34	$144	1,876,528	1.08	65,294	4,238,661	2.26	6,115,189
4	0.36	$160	2,190,233	1.05	73,829	5,341,835	2.44	7,532,068
5	0.38	$174	2,516,770	1.01	81,596	6,178,999	2.46	8,695,769
6	0.4	$182	2,674,338	1.00	85,746	6,813,894	2.55	9,488,232
7	0.42	$189	2,847,068	0.98	90,023	7,527,859	2.64	10,374,927
8	0.44	$220	3,472,810	0.98	109,174	11,917,930	3.43	15,390,740

Final Pit	RF	Undiscounted Value M$	Total Ore tonnes	Diluted Au g/t	Total Mined Metal Content Au oz	Total Waste tonnes	SR W:O	Total Material tonnes
9	0.46	$310	5,242,313	0.99	167,409	23,661,448	4.51	28,903,761
10	0.48	$317	5,390,078	0.99	171,785	24,405,412	4.53	29,795,490
11	0.5	$321	5,502,033	0.99	174,593	24,883,867	4.52	30,385,900
12	0.52	$392	7,273,993	0.97	226,895	35,823,365	4.92	43,097,358
13	0.54	$399	7,478,850	0.97	232,571	36,862,731	4.93	44,341,581
14	0.56	$408	7,722,894	0.96	239,374	38,430,724	4.98	46,153,618
15	0.58	$414	7,935,431	0.96	244,486	39,476,911	4.97	47,412,342
16	0.6	$422	8,200,409	0.95	251,752	41,040,844	5.00	49,241,253
17	0.62	$431	8,478,833	0.95	259,156	43,059,328	5.08	51,538,161
18	0.64	$433	8,578,813	0.95	261,363	43,548,817	5.08	52,127,630
19	0.66	$439	8,849,384	0.94	267,766	45,090,010	5.10	53,939,394
20	0.68	$442	8,992,249	0.94	270,799	45,720,434	5.08	54,712,683
21	0.7	$445	9,131,933	0.93	274,221	46,675,235	5.11	55,807,168
22	0.72	$451	9,511,579	0.92	281,666	48,312,727	5.08	57,824,306
23	0.74	$456	9,838,002	0.91	289,075	50,638,254	5.15	60,476,256
24	0.76	$458	9,911,338	0.91	290,713	51,161,375	5.16	61,072,713
25	0.78	$479	11,291,400	0.89	324,473	63,680,594	5.64	74,971,994
26	0.8	$481	11,434,115	0.89	327,469	64,638,383	5.65	76,072,498
27	0.82	$488	12,080,662	0.88	340,711	68,645,220	5.68	80,725,882
28	0.84	$489	12,235,655	0.87	344,013	69,795,821	5.70	82,031,476
29	0.86	$492	12,478,078	0.87	349,437	72,526,995	5.81	85,005,073
30	0.88	$493	12,613,940	0.87	352,288	73,647,548	5.84	86,261,488
31	0.9	$494	12,801,035	0.87	356,324	75,273,647	5.88	88,074,682
32	0.92	$494	12,899,064	0.86	358,428	76,172,207	5.91	89,071,271
33	0.94	$495	13,066,206	0.86	361,647	77,524,575	5.93	90,590,781
34	0.96	$495	13,112,179	0.86	362,643	77,961,502	5.95	91,073,681
35	0.98	$495	13,175,363	0.86	363,933	78,557,304	5.96	91,732,667
36	1	$495	13,599,474	0.85	372,227	83,498,049	6.14	97,097,523
37	1.02	$495	14,691,327	0.84	396,327	96,227,289	6.55	110,918,616
38	1.04	$494	14,751,291	0.84	397,363	96,653,257	6.55	111,404,548
39	1.06	$494	14,773,383	0.84	397,810	96,895,518	6.56	111,668,901
40	1.08	$494	14,820,240	0.84	398,629	97,278,258	6.56	112,098,498

Figure 15.7.1 Boin Central Pit Optimization Results

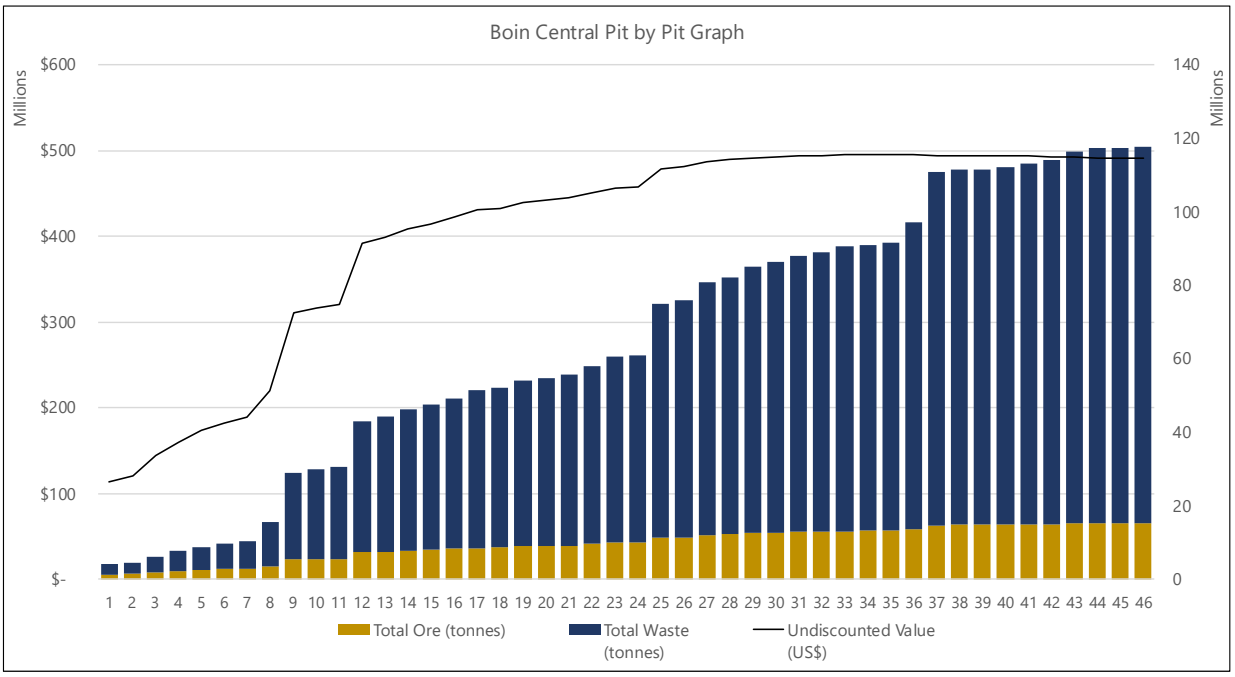


Table 15.7.2 Boin North Pit Optimization Results

Final Pit	RF	Undiscounted Value M$	Total Ore tonnes	Diluted Au g/t	Total Mined Metal Content Au oz	Total Waste tonnes	SR W:O	Total Material tonnes
1	0.3	$6	69,704	1.22	2,726	125,396	1.80	195,100
2	0.32	$6	71,379	1.21	2,767	127,076	1.78	198,455
3	0.34	$7	78,436	1.17	2,951	138,008	1.76	216,444
4	0.36	$7	86,563	1.14	3,165	152,232	1.76	238,795
5	0.38	$7	89,164	1.13	3,229	158,249	1.77	247,413
6	0.4	$7	95,926	1.10	3,393	178,769	1.86	274,695
7	0.42	$8	97,744	1.10	3,471	200,355	2.05	298,099
8	0.44	$8	100,318	1.09	3,518	203,694	2.03	304,012
9	0.46	$8	102,584	1.08	3,566	210,777	2.05	313,361
10	0.48	$8	105,574	1.07	3,618	219,186	2.08	324,760
11	0.5	$9	123,625	1.05	4,154	377,240	3.05	500,865
12	0.52	$9	128,512	1.02	4,224	386,277	3.01	514,789
13	0.54	$17	314,408	1.05	10,566	2,179,441	6.93	2,493,849
14	0.56	$26	610,977	0.92	18,045	3,833,046	6.27	4,444,023
15	0.58	$27	624,444	0.92	18,403	3,894,092	6.24	4,518,536
16	0.6	$27	639,806	0.91	18,794	3,973,894	6.21	4,613,700
17	0.62	$28	685,938	0.91	20,146	4,329,160	6.31	5,015,098
18	0.64	$30	749,862	0.90	21,784	4,696,899	6.26	5,446,761
19	0.66	$30	760,554	0.90	21,984	4,736,870	6.23	5,497,424
20	0.68	$33	893,070	0.86	24,790	5,272,028	5.90	6,165,098
21	0.7	$33	897,595	0.86	24,867	5,285,732	5.89	6,183,327
22	0.72	$33	914,078	0.86	25,204	5,356,160	5.86	6,270,238
23	0.74	$33	921,780	0.86	25,348	5,386,592	5.84	6,308,372
24	0.76	$33	937,244	0.86	25,769	5,537,511	5.91	6,474,755
25	0.78	$34	978,527	0.85	26,721	5,838,937	5.97	6,817,464
26	0.8	$34	979,621	0.85	26,737	5,842,142	5.96	6,821,763
27	0.82	$34	991,341	0.85	26,940	5,894,006	5.95	6,885,347
28	0.84	$37	1,241,557	0.82	32,628	7,873,948	6.34	9,115,505
29	0.86	$38	1,289,959	0.81	33,717	8,281,060	6.42	9,571,019
30	0.88	$38	1,312,998	0.81	34,238	8,489,167	6.47	9,802,165
31	0.9	$38	1,316,133	0.81	34,279	8,492,998	6.45	9,809,131
32	0.92	$38	1,319,289	0.81	34,343	8,518,237	6.46	9,837,526
33	0.94	$38	1,323,939	0.81	34,425	8,543,928	6.45	9,867,867

Final Pit	RF	Undiscounted Value M$	Total Ore tonnes	Diluted Au g/t	Total Mined Metal Content Au oz	Total Waste tonnes	SR W:O	Total Material tonnes
34	0.96	$38	1,362,234	0.80	35,132	8,828,208	6.48	10,190,442
35	0.98	$38	1,378,920	0.80	35,434	8,950,128	6.49	10,329,048
36	1	$38	1,382,189	0.80	35,470	8,955,071	6.48	10,337,260
37	1.02	$38	1,406,637	0.80	35,986	9,222,554	6.56	10,629,191
38	1.04	$38	1,431,908	0.79	36,473	9,477,825	6.62	10,909,733
39	1.06	$38	1,434,117	0.79	36,505	9,488,904	6.62	10,923,021
40	1.08	$38	1,435,064	0.79	36,521	9,493,072	6.62	10,928,136

Figure 15.7.2 Boin North Pit Optimization Results

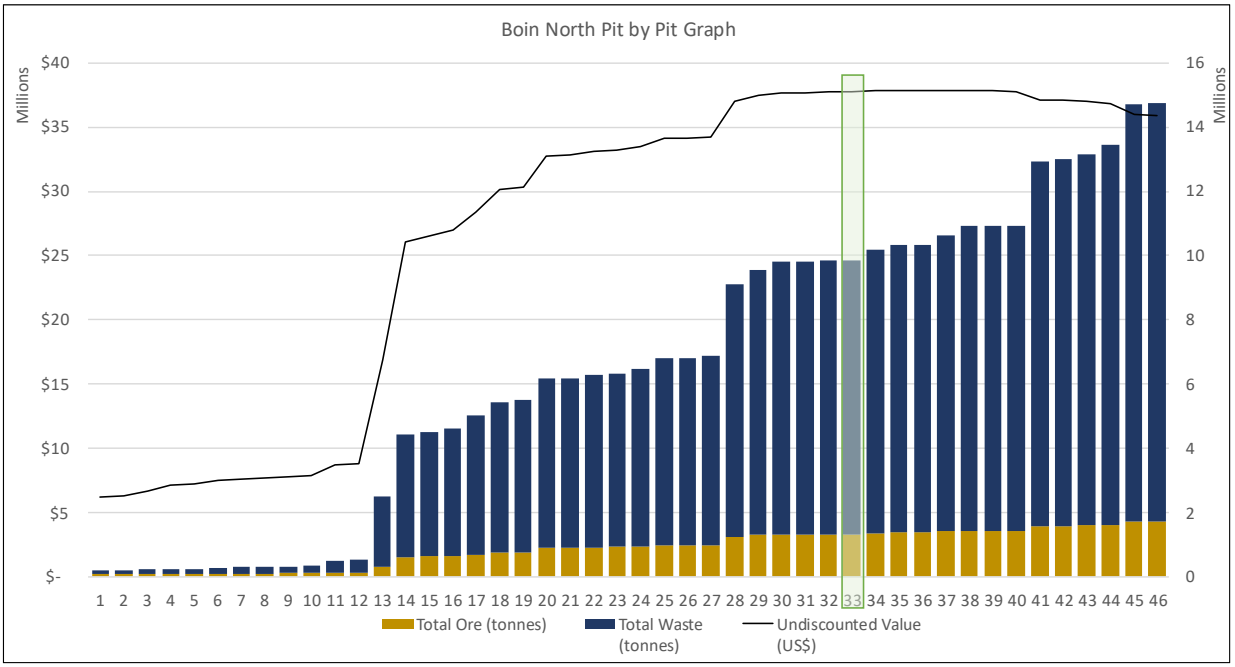
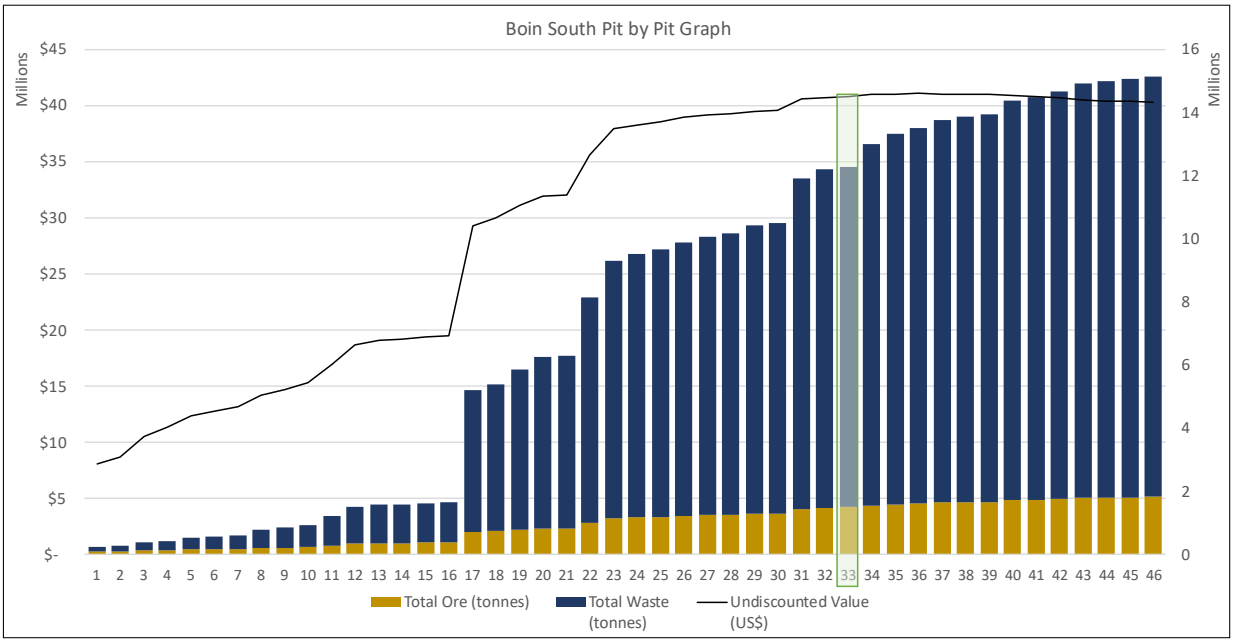


Table 15.7.3 Boin South Pit Optimization Results

Final Pit	RF	Undiscounted Value M$	Total Ore tonnes	Diluted Au g/t	Total Mined Metal Content Au oz	Total Waste tonnes	SR W:O	Total Material tonnes
1	0.3	$8	78,177	1.39	3,485	170,978	2.19	249,155
2	0.32	$9	88,515	1.32	3,744	186,892	2.11	275,407
3	0.34	$11	121,606	1.20	4,676	257,351	2.12	378,957
4	0.36	$11	136,996	1.16	5,088	297,072	2.17	434,068
5	0.38	$12	159,415	1.11	5,664	372,148	2.33	531,563
6	0.4	$13	167,646	1.09	5,893	406,540	2.42	574,186
7	0.42	$13	177,057	1.07	6,118	439,137	2.48	616,194
8	0.44	$14	199,856	1.05	6,722	565,309	2.83	765,165
9	0.46	$15	212,539	1.03	7,044	641,477	3.02	854,016
10	0.48	$15	235,751	0.98	7,441	698,337	2.96	934,088
11	0.5	$17	285,206	0.92	8,476	930,242	3.26	1,215,448
12	0.52	$19	345,722	0.87	9,711	1,147,414	3.32	1,493,136
13	0.54	$19	359,256	0.86	9,975	1,208,397	3.36	1,567,653
14	0.56	$19	363,597	0.86	10,036	1,216,091	3.34	1,579,688
15	0.58	$19	371,299	0.85	10,179	1,251,949	3.37	1,623,248
16	0.6	$19	377,205	0.85	10,273	1,271,390	3.37	1,648,595
17	0.62	$29	711,086	0.83	18,925	4,487,172	6.31	5,198,258
18	0.64	$30	734,804	0.83	19,540	4,667,487	6.35	5,402,291
19	0.66	$31	781,843	0.82	20,612	5,059,934	6.47	5,841,777
20	0.68	$32	818,286	0.82	21,482	5,423,954	6.63	6,242,240
21	0.7	$32	822,686	0.82	21,565	5,456,344	6.63	6,279,030
22	0.72	$36	1,007,684	0.80	25,798	7,148,266	7.09	8,155,950
23	0.74	$38	1,143,655	0.78	28,804	8,159,928	7.13	9,303,583
24	0.76	$38	1,165,103	0.78	29,280	8,351,863	7.17	9,516,966
25	0.78	$39	1,191,173	0.78	29,794	8,482,043	7.12	9,673,216
26	0.8	$39	1,217,602	0.77	30,326	8,650,041	7.10	9,867,643
27	0.82	$39	1,238,141	0.77	30,769	8,830,212	7.13	10,068,353
28	0.84	$39	1,256,786	0.77	31,062	8,901,317	7.08	10,158,103
29	0.86	$39	1,277,539	0.77	31,508	9,131,777	7.15	10,409,316
30	0.88	$40	1,290,901	0.76	31,733	9,198,667	7.13	10,489,568
31	0.9	$41	1,445,659	0.75	34,862	10,461,081	7.24	11,906,740
32	0.92	$41	1,481,515	0.75	35,523	10,719,643	7.24	12,201,158
33	0.94	$41	1,489,721	0.74	35,667	10,767,132	7.23	12,256,853

Final Pit	RF	Undiscounted Value M$	Total Ore tonnes	Diluted Au g/t	Total Mined Metal Content Au oz	Total Waste tonnes	SR W:O	Total Material tonnes
34	0.96	$41	1,553,434	0.74	37,047	11,436,885	7.36	12,990,319
35	0.98	$41	1,581,756	0.74	37,631	11,727,631	7.41	13,309,387
36	1	$41	1,598,844	0.74	37,960	11,916,885	7.45	13,515,729
37	1.02	$41	1,644,629	0.73	38,622	12,124,827	7.37	13,769,456
38	1.04	$41	1,654,728	0.73	38,799	12,199,854	7.37	13,854,582
39	1.06	$41	1,668,131	0.73	38,989	12,265,865	7.35	13,933,996
40	1.08	$41	1,711,052	0.72	39,763	12,644,432	7.39	14,355,484

Figure 15.7.3 Boin South Pit Optimization Results



15.7.2 Sewum Pit Optimization Result

Sewum pit optimization was divided into three discrete areas for shell selection, the results are as follows:

- Sewum North: The revenue factor 0.94 (Shell 33) was selected, as the undiscounted value curve begins to flatten and strip ratio increases. Shell 33 provided 229,000 contained ounces at an average grade of 0.43 g/t and a strip ratio of 2.04. These results are demonstrated in Table 15.7.4 and Figure 15.7.4.

- Sewum Checkerboard: The revenue factor 0.92 (Shell 32) was selected, as the undiscounted value curve begins to flatten and strip ratio increases. Shell 32 provided 95,000 contained ounces at an average grade of 0.50 g/t and a strip ratio of 2.44. These results are demonstrated in Table 15.7.5 and Figure 15.7.5.

- Sewum South: The revenue factor 0.92 (Shell 32) was selected, as the undiscounted value curve begins to flatten, grade decreases and strip ratio increases. Shell 32 provided 84,000 contained ounces at an average grade of 0.76 g/t and a strip ratio of 4.18. These results are demonstrated in Table 15.7.6 and Figure 15.7.6.

Table 15.7.4 Sewum North Pit Optimization Results

Final Pit	RF	Undiscounted Value M$	Total Ore tonnes	Diluted Au g/t	Total Mined Metal Content Au oz	Total Waste tonnes	SR W:O	Total Material tonnes
1	0.3	$16	270,104	0.81	7,035	109,985	0.41	380,089
2	0.32	$22	448,927	0.72	10,366	218,162	0.49	667,089
3	0.34	$28	629,577	0.66	13,367	293,673	0.47	923,250
4	0.36	$37	932,262	0.62	18,469	491,374	0.53	1,423,636
5	0.38	$51	1,442,617	0.57	26,525	939,210	0.65	2,381,827
6	0.4	$63	1,870,984	0.55	32,934	1,121,213	0.60	2,992,197
7	0.42	$68	2,077,279	0.54	35,864	1,214,860	0.58	3,292,139
8	0.44	$75	2,415,425	0.52	40,561	1,464,133	0.61	3,879,558
9	0.46	$97	3,329,160	0.51	54,554	3,279,915	0.99	6,609,075
10	0.48	$126	4,682,272	0.49	73,396	5,144,162	1.10	9,826,434
11	0.5	$136	5,223,523	0.48	80,624	6,007,756	1.15	11,231,279
12	0.52	$141	5,475,337	0.48	83,878	6,365,979	1.16	11,841,316
13	0.54	$150	6,055,844	0.47	91,112	7,226,055	1.19	13,281,899
14	0.56	$162	6,852,315	0.46	101,044	8,596,351	1.25	15,448,666
15	0.58	$171	7,448,285	0.45	108,630	9,699,579	1.30	17,147,864
16	0.6	$181	8,175,485	0.45	117,082	10,782,352	1.32	18,957,837
17	0.62	$191	8,977,116	0.44	126,398	11,929,617	1.33	20,906,733
18	0.64	$204	10,024,968	0.43	139,433	13,716,781	1.37	23,741,749
19	0.66	$217	11,040,358	0.43	152,795	15,514,316	1.41	26,554,674
20	0.68	$222	11,455,669	0.43	158,345	16,239,953	1.42	27,695,622
21	0.7	$227	11,815,405	0.43	163,367	16,930,818	1.43	28,746,223
22	0.72	$241	13,225,862	0.43	182,110	19,502,857	1.47	32,728,719
23	0.74	$248	13,885,685	0.43	191,226	21,704,599	1.56	35,590,284
24	0.76	$252	14,246,164	0.43	196,838	23,399,485	1.64	37,645,649

Final Pit	RF	Undiscounted Value M$	Total Ore tonnes	Diluted Au g/t	Total Mined Metal Content Au oz	Total Waste tonnes	SR W:O	Total Material tonnes
25	0.78	$254	14,474,605	0.43	199,955	24,270,131	1.68	38,744,736
26	0.8	$256	14,803,980	0.43	204,506	25,681,205	1.73	40,485,185
27	0.82	$257	14,905,903	0.43	205,779	26,013,521	1.75	40,919,424
28	0.84	$261	15,498,233	0.43	214,796	29,234,211	1.89	44,732,444
29	0.86	$262	15,586,098	0.43	215,826	29,440,552	1.89	45,026,650
30	0.88	$262	15,731,998	0.43	217,660	30,004,266	1.91	45,736,264
31	0.9	$263	15,822,924	0.43	218,755	30,218,418	1.91	46,041,342
32	0.92	$264	16,067,936	0.43	221,726	31,407,540	1.95	47,475,476
33	0.94	$265	16,572,809	0.43	228,664	33,825,836	2.04	50,398,645
34	0.96	$265	16,722,911	0.43	230,437	34,384,584	2.06	51,107,495
35	0.98	$265	16,857,682	0.43	231,953	34,795,915	2.06	51,653,597
36	1	$265	16,957,018	0.43	233,585	35,705,537	2.11	52,662,555
37	1.02	$265	17,110,362	0.43	235,893	36,882,871	2.16	53,993,233
38	1.04	$265	17,168,292	0.43	236,573	37,139,678	2.16	54,307,970
39	1.06	$265	17,293,192	0.43	238,165	37,882,203	2.19	55,175,395
40	1.08	$265	17,340,604	0.43	238,757	38,089,482	2.20	55,430,086

Figure 15.7.4 Sewum North Pit Optimization Results

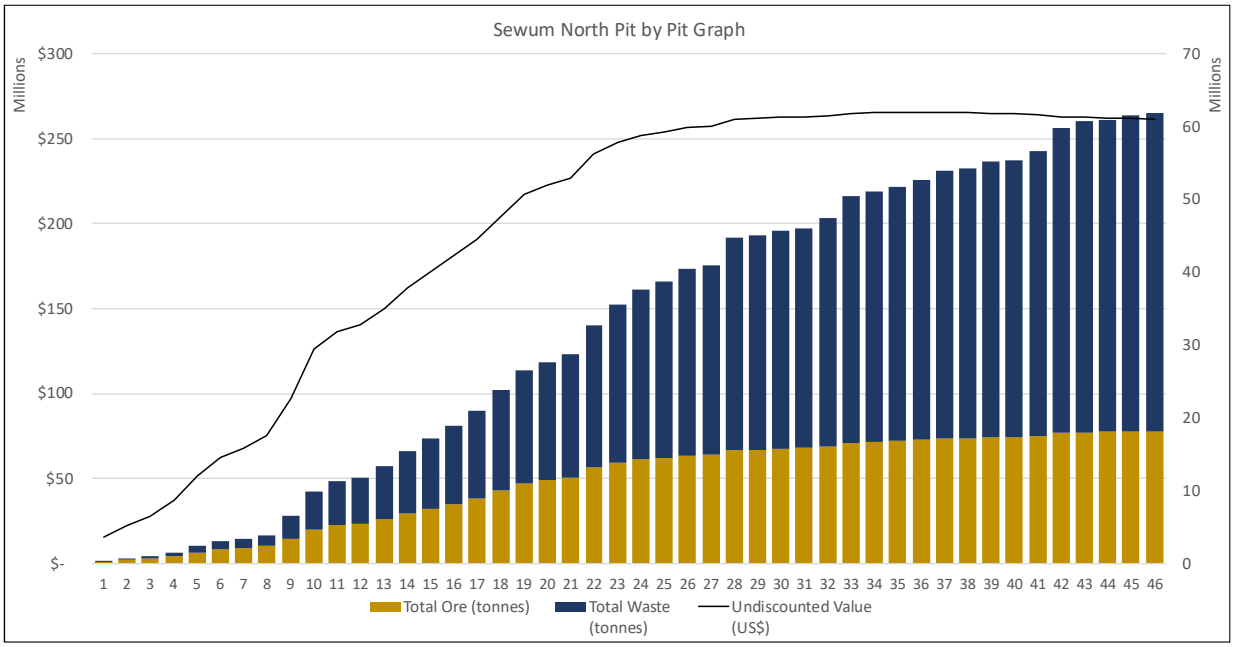


Table 15.7.5 Sewum Checkerboard Pit Optimization Results

Final Pit	RF	Undiscounted Value M$	Total Ore tonnes	Diluted Au g/t	Total Mined Metal Content Au oz	Total Waste tonnes	SR W:O	Total Material tonnes
1	0.3	$9	156,947	0.83	4,209	56,399	0.36	213,346
2	0.32	$12	221,588	0.76	5,449	90,628	0.41	312,216
3	0.34	$13	263,697	0.73	6,191	104,357	0.40	368,054
4	0.36	$16	352,809	0.67	7,644	134,075	0.38	486,884
5	0.38	$20	483,604	0.62	9,683	180,545	0.37	664,149
6	0.4	$23	589,231	0.60	11,353	250,608	0.43	839,839
7	0.42	$30	897,431	0.54	15,697	394,859	0.44	1,292,290
8	0.44	$36	1,196,062	0.52	19,808	609,738	0.51	1,805,800
9	0.46	$37	1,254,259	0.51	20,648	659,143	0.53	1,913,402
10	0.48	$59	2,288,347	0.49	36,022	1,607,101	0.70	3,895,448
11	0.5	$61	2,408,719	0.49	37,672	1,726,444	0.72	4,135,163
12	0.52	$65	2,578,628	0.49	40,511	2,035,948	0.79	4,614,576
13	0.54	$73	3,016,642	0.48	46,651	2,873,488	0.95	5,890,130
14	0.56	$77	3,211,894	0.48	49,917	3,333,660	1.04	6,545,554
15	0.58	$82	3,484,364	0.49	54,629	4,046,543	1.16	7,530,907
16	0.6	$84	3,554,471	0.49	55,628	4,201,041	1.18	7,755,512
17	0.62	$85	3,645,908	0.49	57,315	4,514,550	1.24	8,160,458
18	0.64	$86	3,698,848	0.49	57,955	4,605,278	1.25	8,304,126
19	0.66	$88	3,859,296	0.49	60,211	5,015,808	1.30	8,875,104
20	0.68	$88	3,873,522	0.49	60,402	5,038,457	1.30	8,911,979
21	0.7	$90	3,989,243	0.48	61,942	5,391,085	1.35	9,380,328
22	0.72	$93	4,232,466	0.48	65,806	6,271,207	1.48	10,503,673
23	0.74	$93	4,255,611	0.48	66,108	6,345,232	1.49	10,600,843
24	0.76	$93	4,288,766	0.48	66,517	6,429,991	1.50	10,718,757
25	0.78	$100	4,830,986	0.50	77,256	9,727,886	2.01	14,558,872
26	0.8	$101	4,894,015	0.50	78,130	9,960,175	2.04	14,854,190
27	0.82	$101	4,972,304	0.50	79,323	10,254,637	2.06	15,226,941
28	0.84	$102	5,079,463	0.50	80,956	10,793,037	2.12	15,872,500
29	0.86	$107	5,760,632	0.50	92,046	13,681,002	2.37	19,441,634
30	0.88	$107	5,860,577	0.50	93,435	14,039,621	2.40	19,900,198
31	0.9	$107	5,909,280	0.50	94,193	14,301,362	2.42	20,210,642
32	0.92	$108	5,956,738	0.50	94,947	14,534,383	2.44	20,491,121
33	0.94	$108	6,079,425	0.50	96,929	15,168,595	2.50	21,248,020

Final Pit	RF	Undiscounted Value M$	Total Ore tonnes	Diluted Au g/t	Total Mined Metal Content Au oz	Total Waste tonnes	SR W:O	Total Material tonnes
34	0.96	$108	6,104,095	0.50	97,254	15,243,919	2.50	21,348,014
35	0.98	$108	6,179,925	0.49	98,237	15,664,100	2.53	21,844,025
36	1	$108	6,437,793	0.49	102,277	17,204,468	2.67	23,642,261
37	1.02	$108	6,491,120	0.49	103,060	17,508,636	2.70	23,999,756
38	1.04	$108	6,524,945	0.49	103,548	17,695,456	2.71	24,220,401
39	1.06	$108	6,538,702	0.49	103,695	17,737,384	2.71	24,276,086
40	1.08	$108	6,599,292	0.49	104,516	18,126,069	2.75	24,725,361

Figure 15.7.5 Sewum Checkerboard Pit Optimization Results

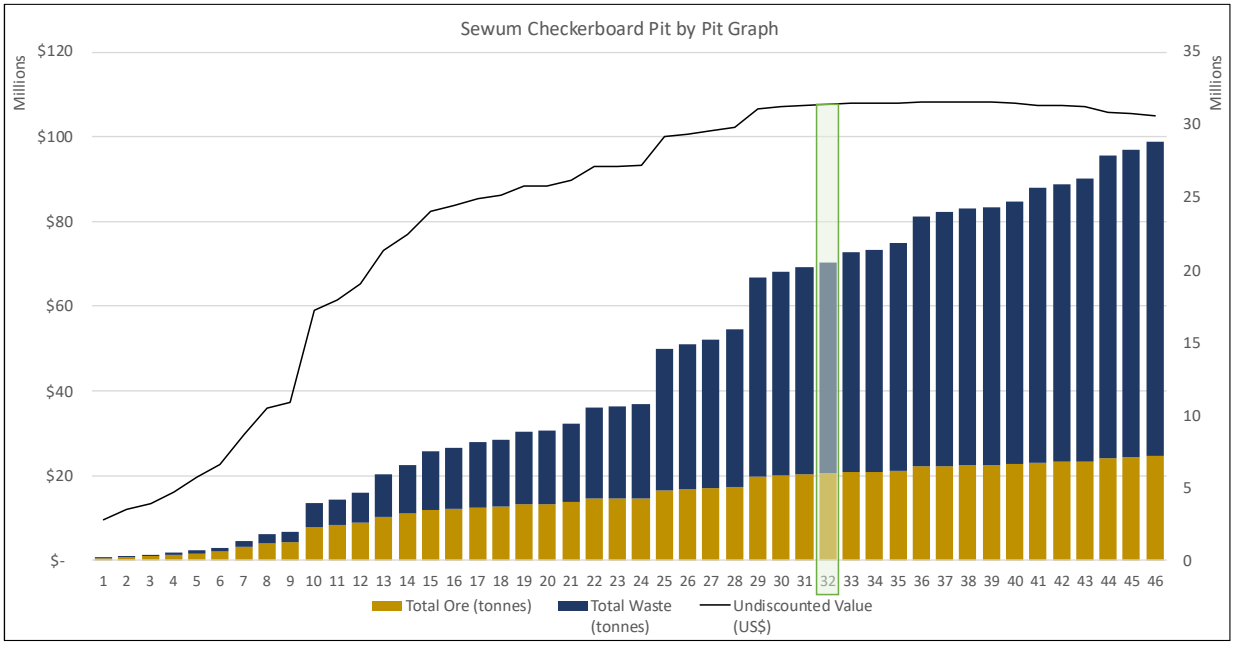
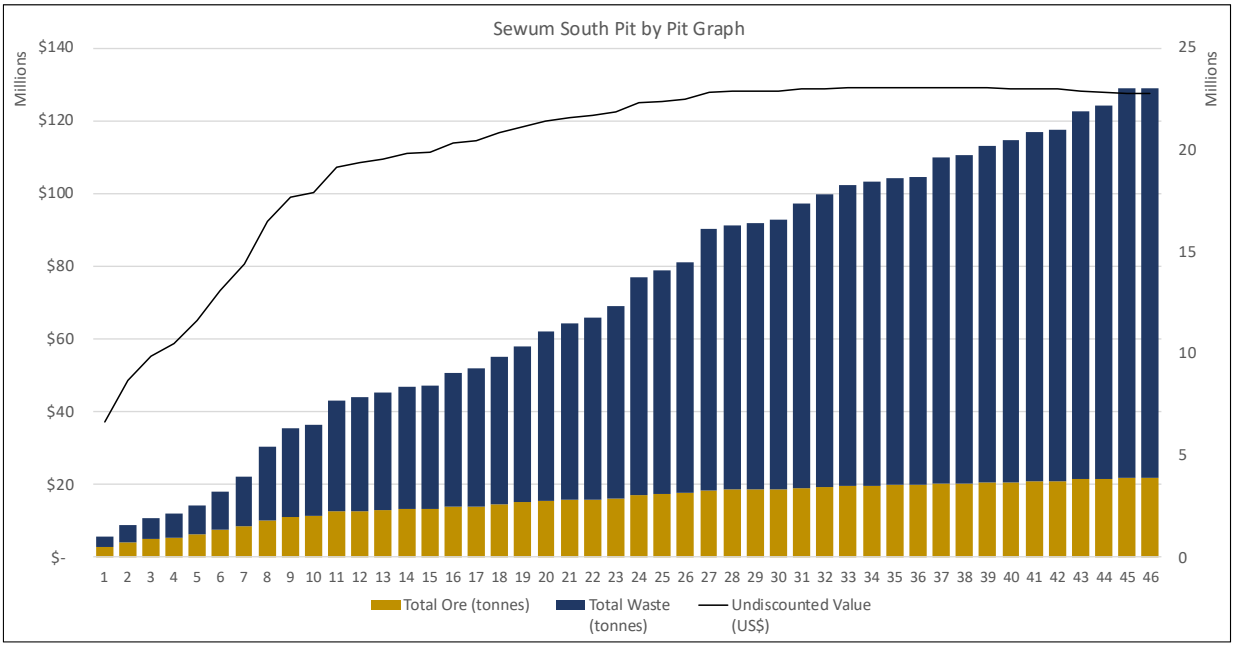


Table 15.7.6 Sewum South Pit Optimization Results

Final Pit	RF	Undiscounted Value M$	Total Ore tonnes	Diluted Au g/t	Total Mined Metal Content Au oz	Total Waste tonnes	SR W:O	Total Material tonnes
1	0.3	$37	508,966	1.01	16,471	472,061	0.93	981,027
2	0.32	$48	716,853	0.96	22,026	838,465	1.17	1,555,318
3	0.34	$55	877,009	0.91	25,578	1,011,446	1.15	1,888,455
4	0.36	$59	959,941	0.89	27,443	1,146,346	1.19	2,106,287
5	0.38	$65	1,118,344	0.86	30,916	1,422,167	1.27	2,540,511
6	0.4	$73	1,336,811	0.83	35,704	1,863,309	1.39	3,200,120
7	0.42	$80	1,520,659	0.82	40,017	2,438,904	1.60	3,959,563
8	0.44	$92	1,797,456	0.82	47,253	3,614,535	2.01	5,411,991
9	0.46	$99	1,974,092	0.81	51,562	4,329,504	2.19	6,303,596
10	0.48	$100	2,018,457	0.81	52,556	4,491,431	2.23	6,509,888
11	0.5	$107	2,216,846	0.81	57,449	5,447,020	2.46	7,663,866
12	0.52	$108	2,250,801	0.80	58,208	5,589,902	2.48	7,840,703
13	0.54	$109	2,279,575	0.80	58,946	5,767,757	2.53	8,047,332
14	0.56	$111	2,338,862	0.80	60,129	6,010,763	2.57	8,349,625
15	0.58	$111	2,358,613	0.80	60,446	6,076,553	2.58	8,435,166
16	0.6	$114	2,461,909	0.79	62,759	6,597,982	2.68	9,059,891
17	0.62	$115	2,493,461	0.79	63,399	6,747,757	2.71	9,241,218
18	0.64	$117	2,581,805	0.79	65,258	7,252,489	2.81	9,834,294
19	0.66	$118	2,665,690	0.78	66,840	7,677,746	2.88	10,343,436
20	0.68	$120	2,753,143	0.78	68,848	8,333,523	3.03	11,086,666
21	0.7	$121	2,799,240	0.77	69,732	8,642,808	3.09	11,442,048
22	0.72	$121	2,833,133	0.77	70,478	8,928,331	3.15	11,761,464
23	0.74	$122	2,886,135	0.77	71,740	9,404,507	3.26	12,290,642
24	0.76	$125	3,040,090	0.77	75,325	10,709,052	3.52	13,749,142
25	0.78	$125	3,083,931	0.77	76,042	11,014,749	3.57	14,098,680
26	0.8	$126	3,140,060	0.76	76,967	11,353,556	3.62	14,493,616
27	0.82	$128	3,267,507	0.76	80,229	12,825,210	3.93	16,092,717
28	0.84	$128	3,292,337	0.76	80,670	12,990,383	3.95	16,282,720
29	0.86	$128	3,306,065	0.76	80,893	13,088,247	3.96	16,394,312
30	0.88	$128	3,331,262	0.76	81,339	13,252,289	3.98	16,583,551
31	0.9	$129	3,398,215	0.76	82,750	13,934,502	4.10	17,332,717
32	0.92	$129	3,434,484	0.76	83,542	14,369,811	4.18	17,804,295
33	0.94	$129	3,482,418	0.75	84,355	14,788,561	4.25	18,270,979

Final Pit	RF	Undiscounted Value M$	Total Ore tonnes	Diluted Au g/t	Total Mined Metal Content Au oz	Total Waste tonnes	SR W:O	Total Material tonnes
34	0.96	$129	3,503,867	0.75	84,710	14,951,026	4.27	18,454,893
35	0.98	$129	3,527,603	0.75	84,993	15,061,212	4.27	18,588,815
36	1	$129	3,534,823	0.75	85,081	15,098,645	4.27	18,633,468
37	1.02	$129	3,600,076	0.75	86,596	16,045,760	4.46	19,645,836
38	1.04	$129	3,607,645	0.75	86,747	16,130,797	4.47	19,738,442
39	1.06	$129	3,644,456	0.75	87,389	16,523,616	4.53	20,168,072
40	1.08	$129	3,673,671	0.74	87,859	16,778,673	4.57	20,452,344

Figure 15.7.6 Sewum South Pit Optimization Results



15.7.3 Nyam Pit Optimization Result

Nyam pit optimization revenue factor 0.90 (Shell 31) was selected, as the undiscounted value curve begins to flatten and a notable increase in strip ratio is observed with Shell 32. Shell 31 provided 172,000 contained ounces at an average grade of 0.77 g/t and strip ratio of 4.41. These results are demonstrated in Table 15.7.7 and Figure 15.7.7.

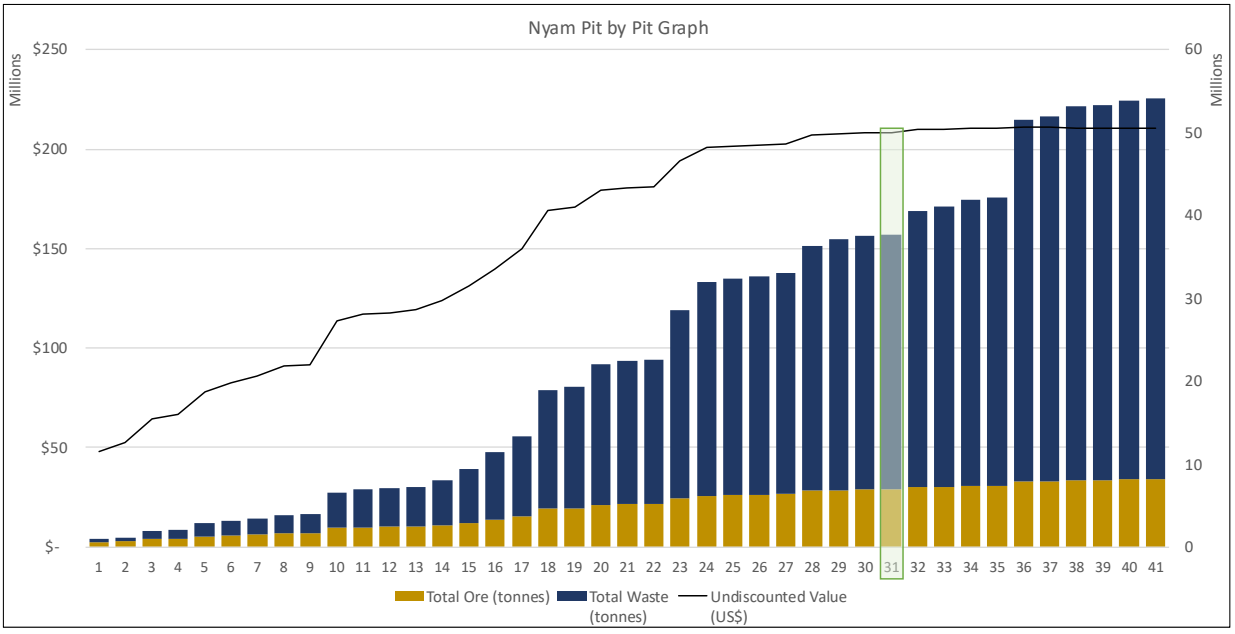
In addition to the selection of Shell 31 for the ultimate pit limits, a lower revenue factor of 0.62 (Shell 17) was evaluated as a guiding interim phase to support improved project economics during the early years of mining. Shell 17 is characterized by a higher average grade of 0.80 g/t and a lower strip ratio of 2.57, making it a favorable target for early extraction. This shell contains approximately 72% of the total undiscounted value while comprising only 54% of the total mineralized material defined within the ultimate pit.

Table 15.7.7 Nyam Pit Optimization Results

Final Pit	RF	Undiscounted Value M$	Total Ore tonnes	Diluted Au g/t	Total Mined Metal Content Au oz	Total Waste tonnes	SR W:O	Total Material tonnes
1	0.3	$48	624,192	1.11	22,366	357,621	0.57	981,813
2	0.32	$52	715,696	1.08	24,838	471,546	0.66	1,187,242
3	0.34	$64	984,085	1.00	31,521	895,535	0.91	1,879,620
4	0.36	$67	1,036,402	0.99	32,827	984,678	0.95	2,021,080
5	0.38	$78	1,308,664	0.95	39,874	1,564,649	1.20	2,873,313
6	0.4	$82	1,406,538	0.94	42,440	1,802,426	1.28	3,208,964
7	0.42	$86	1,503,700	0.92	44,595	1,952,465	1.30	3,456,165
8	0.44	$91	1,649,067	0.90	47,938	2,220,436	1.35	3,869,503
9	0.46	$92	1,674,412	0.90	48,472	2,261,119	1.35	3,935,531
10	0.48	$114	2,300,549	0.86	63,887	4,312,408	1.87	6,612,957
11	0.5	$117	2,404,355	0.86	66,297	4,557,562	1.90	6,961,917
12	0.52	$118	2,445,987	0.85	67,088	4,642,882	1.90	7,088,869
13	0.54	$119	2,503,154	0.85	68,322	4,798,051	1.92	7,301,205
14	0.56	$124	2,647,602	0.84	71,821	5,365,292	2.03	8,012,894
15	0.58	$131	2,930,198	0.83	78,083	6,460,565	2.20	9,390,763
16	0.6	$140	3,268,559	0.82	86,454	8,118,799	2.48	11,387,358
17	0.62	$150	3,729,441	0.80	95,911	9,594,668	2.57	13,324,109
18	0.64	$169	4,631,704	0.78	115,857	14,352,616	3.10	18,984,320
19	0.66	$171	4,712,309	0.78	117,613	14,676,030	3.11	19,388,339
20	0.68	$179	5,078,701	0.78	126,778	16,968,686	3.34	22,047,387
21	0.7	$180	5,138,139	0.78	128,123	17,264,971	3.36	22,403,110
22	0.72	$181	5,177,425	0.77	128,898	17,417,440	3.36	22,594,865
23	0.74	$194	5,836,641	0.78	146,258	22,694,806	3.89	28,531,447
24	0.76	$201	6,212,687	0.78	156,092	25,774,710	4.15	31,987,397
25	0.78	$202	6,267,813	0.78	157,227	26,104,691	4.16	32,372,504
26	0.8	$202	6,315,143	0.78	158,143	26,335,983	4.17	32,651,126
27	0.82	$203	6,386,522	0.78	159,501	26,696,253	4.18	33,082,775

Final Pit	RF	Undiscounted Value M$	Total Ore tonnes	Diluted Au g/t	Total Mined Metal Content Au oz	Total Waste tonnes	SR W:O	Total Material tonnes
28	0.84	$207	6,763,012	0.77	168,052	29,530,591	4.37	36,293,603
29	0.86	$208	6,863,321	0.77	170,128	30,204,897	4.40	37,068,218
30	0.88	$208	6,931,257	0.77	171,436	30,574,023	4.41	37,505,280
31	0.9	$208	6,950,464	0.77	171,794	30,685,696	4.41	37,636,160
32	0.92	$210	7,180,818	0.77	177,463	33,288,945	4.64	40,469,763
33	0.94	$210	7,247,635	0.77	178,925	33,882,718	4.68	41,130,353
34	0.96	$210	7,307,636	0.77	180,391	34,602,592	4.74	41,910,228
35	0.98	$210	7,333,890	0.77	180,871	34,774,008	4.74	42,107,898
36	1	$211	7,903,449	0.77	196,848	43,643,975	5.52	51,547,424
37	1.02	$211	7,941,970	0.77	197,571	43,940,665	5.53	51,882,635
38	1.04	$211	8,083,251	0.77	200,274	45,127,088	5.58	53,210,339
39	1.06	$211	8,093,855	0.77	200,447	45,191,252	5.58	53,285,107
40	1.08	$210	8,142,332	0.77	201,474	45,724,527	5.62	53,866,859

Figure 15.7.7 Nyam Pit Optimization Results



15.7.4 Kwakyekrom Pit Optimization Result

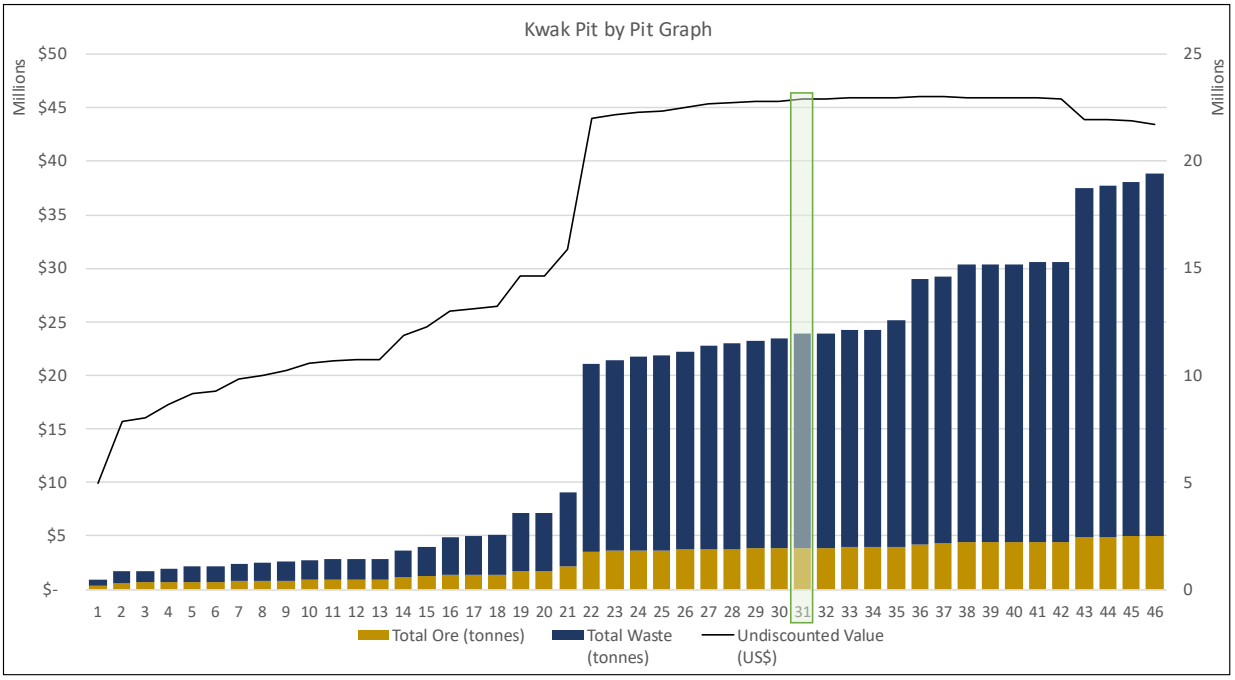
Kwakyekrom pit optimization revenue factor 0.90 (Shell 31) was selected, as the undiscounted value curve begins to flatten and incremental profit showed a negligible increase with the higher revenue factor shells. Shell 31 provided 44,000 contained ounces at an average grade of 0.71 g/t and a strip ratio of 5.11. These results are demonstrated in Table 15.7.8 and Figure 15.7.8.

Table 15.7.8 Kwak Pit Optimization Results

Final Pit	RF	Undiscounted Value M$	Total Ore tonnes	Diluted Au g/t	Total Mined Metal Content Au oz	Total Waste tonnes	SR W:O	Total Material tonnes
1	0.3	$10	198,726	0.82	5,268	260,884	1.31	459,610
2	0.32	$16	311,711	0.85	8,486	549,141	1.76	860,852
3	0.34	$16	319,960	0.84	8,692	559,078	1.75	879,038
4	0.36	$17	347,161	0.84	9,419	629,376	1.81	976,537
5	0.38	$18	370,087	0.84	10,052	704,762	1.90	1,074,849
6	0.4	$19	374,148	0.85	10,165	715,898	1.91	1,090,046
7	0.42	$20	398,250	0.85	10,847	819,023	2.06	1,217,273
8	0.44	$20	410,170	0.84	11,101	843,192	2.06	1,253,362
9	0.46	$20	421,839	0.84	11,361	877,741	2.08	1,299,580
10	0.48	$21	448,243	0.82	11,847	929,926	2.07	1,378,169
11	0.5	$21	456,361	0.82	11,994	947,964	2.08	1,404,325
12	0.52	$21	462,647	0.81	12,092	958,542	2.07	1,421,189
13	0.54	$22	467,167	0.81	12,145	959,975	2.05	1,427,142
14	0.56	$24	575,069	0.76	14,033	1,244,790	2.16	1,819,859
15	0.58	$25	604,272	0.76	14,709	1,402,241	2.32	2,006,513
16	0.6	$26	667,427	0.75	16,051	1,763,532	2.64	2,430,959
17	0.62	$26	681,526	0.74	16,309	1,840,816	2.70	2,522,342
18	0.64	$26	695,489	0.74	16,536	1,888,110	2.71	2,583,599
19	0.66	$29	850,159	0.73	19,831	2,743,919	3.23	3,594,078
20	0.68	$29	855,871	0.72	19,890	2,745,961	3.21	3,601,832
21	0.7	$32	1,057,183	0.67	22,901	3,481,885	3.29	4,539,068
22	0.72	$44	1,789,101	0.70	40,546	8,767,518	4.90	10,556,619
23	0.74	$44	1,809,602	0.71	41,088	8,913,910	4.93	10,723,512
24	0.76	$45	1,831,749	0.71	41,573	9,036,544	4.93	10,868,293
25	0.78	$45	1,840,524	0.71	41,760	9,100,524	4.94	10,941,048
26	0.8	$45	1,865,387	0.71	42,348	9,261,038	4.96	11,126,425
27	0.82	$45	1,884,289	0.71	42,905	9,525,406	5.06	11,409,695

Final Pit	RF	Undiscounted Value M$	Total Ore tonnes	Diluted Au g/t	Total Mined Metal Content Au oz	Total Waste tonnes	SR W:O	Total Material tonnes
28	0.84	$45	1,896,526	0.71	43,206	9,624,425	5.07	11,520,951
29	0.86	$46	1,913,207	0.71	43,499	9,725,386	5.08	11,638,593
30	0.88	$46	1,922,347	0.71	43,710	9,807,667	5.10	11,730,014
31	0.9	$46	1,955,482	0.71	44,336	9,994,417	5.11	11,949,899
32	0.92	$46	1,959,969	0.70	44,392	10,007,546	5.11	11,967,515
33	0.94	$46	1,973,757	0.71	44,741	10,161,459	5.15	12,135,216
34	0.96	$46	1,976,260	0.70	44,784	10,173,044	5.15	12,149,304
35	0.98	$46	2,009,333	0.71	45,595	10,545,796	5.25	12,555,129
36	1	$46	2,129,610	0.71	48,780	12,373,610	5.81	14,503,220
37	1.02	$46	2,144,171	0.71	49,012	12,451,115	5.81	14,595,286
38	1.04	$46	2,199,354	0.71	50,078	12,996,181	5.91	15,195,535
39	1.06	$46	2,200,490	0.71	50,093	12,998,553	5.91	15,199,043
40	1.08	$46	2,202,690	0.71	50,116	12,999,423	5.90	15,202,113

Figure 15.7.8 Kwakyekrom Pit Optimization Results



15.8 Pit Design and Reconciliation

Pit designs were developed using Deswik, with the objective of achieving close conformity with the selected pit optimization shells and their respective mineralized domains. The designs incorporate applicable geotechnical design criteria, ramp geometry requirements, and haulage considerations, while maintaining a focus on operational executability. The following sections describe the parameters and assumptions used to inform the pit design process and present a comparison between the optimization shells and the resulting ultimate pit designs.

15.8.1 Geotechnical Parameters

The geotechnical criteria provided by DRA was incorporated into the pit designs. Given the three distinct material types (oxide, transition, and fresh) and four discrete mining areas, the recommended bench geometries were defined accordingly, as described in Section 15.5.1 and can be found in Table 15.5.1.

15.8.2 Haul Road Design Criteria

The main in-pit ramps were classified into three categories based on the type of mining equipment local contractors utilized and expected fleet size required. These classifications were applied according to bench size and operational requirements, as outlined below:

- Two-way traffic (91-tonne haul trucks): This road configuration was applied to the upper and mid-level pit benches, where larger working areas allowed for higher equipment density and increased haulage requirements.
- Two-way traffic (41-tonne haul trucks): This road configuration was implemented on the lower benches of the pit, where bench widths could no longer accommodate 91-tonne haul fleet. Smaller, 41-tonne trucks, were therefore planned for, typically within the bottom 4 to 6 benches, depending on the specific pit configuration.
- One-way traffic (41-tonne haul trucks): This road configuration was used in the deepest portions of the pit, where bench widths could only accommodate a small fleet of 41-tonne trucks and respective loading unit. This configuration was applied within the bottom 1 to 2 benches, depending on the specific pit configuration.

The haul road widths incorporated into the mine design were developed in accordance with Minerals and Mining (Health, Safety and Technical) Regulations, 2012 (Ghana), specifically Regulation 84, which governs the design of haulage roads. Road width allowances were established based on accommodating 2.5 times the width of the largest haul truck for two-way traffic and 2.0 times the width for one-way traffic. These allowances also include additional width for safety berms on the outer edge and drainage ditches, where required.

Haul roads were classified into the three categories described above and the respective configurations are summarized in Table 15.8.1.

Table 15.8.1 Haul Road Design Criteria

Item	Two-way 91-Tonne m	Two-way 41-Tonne m	One-way 41-Tonne m
Truck Operating Width	6.7	3.4	3.4
Truck Tire Diameter	2.6	1.9	1.9
Berm Height	2.0	1.4	1.4
Berm Width (45º Slope)	4.0	3.4	3.4
Running Surface	16.8	8.6	6.9
Drainage Ditch	2.7	2.6	0.0
Total Road Allowance	**23.5**	**14.5**	**10.2**
Max Road Grade	**10%**	**12%**	**12%**

15.8.3 Pit Designs and Reconciliation

Pit designs were developed based on the inputs and considerations described above. The resulting pit designs are demonstrated in Figures 15.8.1 through Figure 15.8.17.

A reconciliation between the selected ultimate pit shells and the final pit designs was completed. As discussed in Section 15.4.1, the pit optimization IRA was reduced to account for ramps to be incorporated into the highwalls. This adjustment improved alignment between the optimization shells and the final pit designs; however, some variances remain. These differences are attributed to factors such as additional ramp requirements to reach the pit bottom, smoothing of highwall configurations to ensure operational practicality, and the inclusion of low strip ratio mineralized material where opportunities were identified.

Table 15.8.2 summarizes the reconciliation between the pit shells and the final pit designs for all deposits. Strong alignment in mineralized tonnage (1%) and grade (-1%) is observed, indicating good correspondence with the targeted ore zones identified during pit optimization. However, slightly increased waste volumes (13%) were encountered and are expected, as a result of the design considerations described above.

Table 15.8.2 Total Project Pit Shell and Design Reconciliation

All Deposits	Pit Shells	Pit Designs	% Difference
Mineralized Material (kt)	50,580	51,290	1%
Mineralized Material Grade (g/t)	0.65	0.64	-1%
Contained Gold (koz)	1,054	1,055	0%

All Deposits	Pit Shells	Pit Designs	% Difference
Waste Material (kt)	198,890	225,550	13%
Total Material (kt)	249,480	276,830	11%
Strip Ratio (W:O)	3.9	4.4[1]	12%

Note:
1 Strip ratio excludes pre-strip tonnes mined (included in initial capex)

Boin Pit Design and Reconciliation

The Boin Deposit was divided into three pits: Boin Central, Boin North, Boin South. Summaries of each pit area are provided below. All Boin pits were designed using 10 m benches, 5 m safety berms and variable bench face angles dependent on material type, described in Table 15.5.1.

The Boin Central ultimate pit design is shown in Figure 15.8.1. A cross-sectional comparison between the pit design and the corresponding pit shell is presented in Figure 15.8.2, and Table 15.8.3 summarizes the reconciliation results between the two.

Figure 15.8.1 Boin Central Ultimate Pit Design

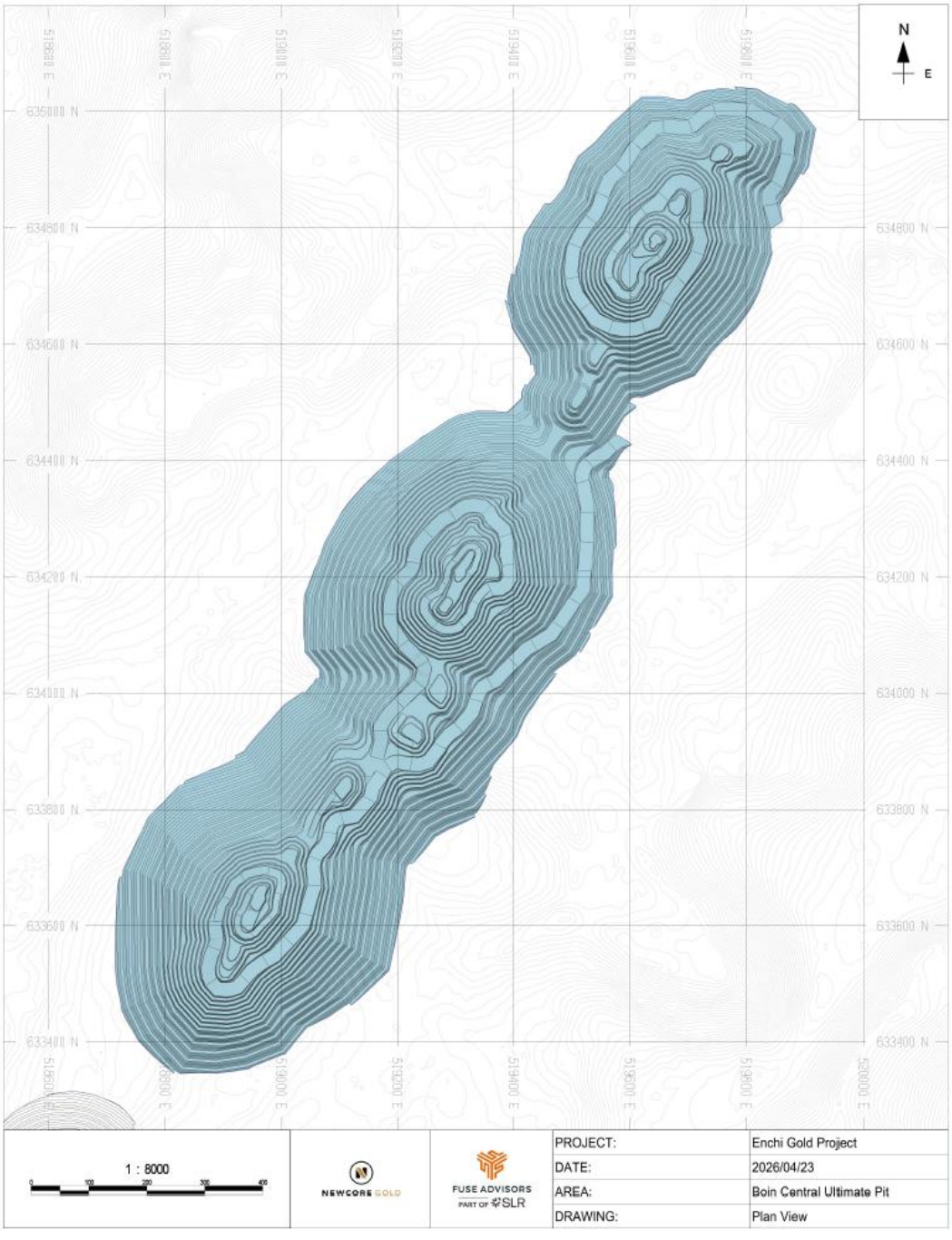


Figure 15.8.2 Boin Central Ultimate Pit Design versus Shell

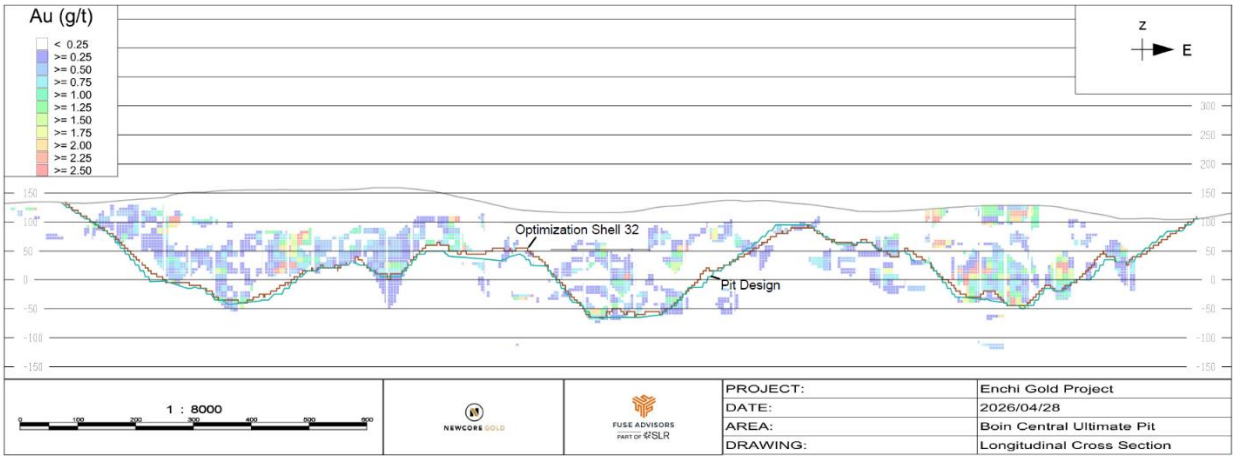


Table 15.8.3 Boin Central Pit Shell and Design Reconciliation

Boin Central	Pit Shell	Pit Design	% Difference
Mineralized Material (kt)	12,900	13,010	1%
Mineralized Material Grade (g/t)	0.86	0.86	-1%
Contained Gold (koz)	360	359	0%
Waste Material (kt)	76,170	83,230	9%
Total Material (kt)	89,070	96,250	8%
Strip Ratio (W:O)	5.9	6.4	8%

The Boin North ultimate pit design is shown in Figure 15.8.3. A cross-sectional comparison between the pit design and the corresponding pit shell is presented in Figure 15.8.4, and Table 15.8.4 summarizes the reconciliation results. Because the Boin North pit design was relatively deep compared with its overall pit footprint, the pit walls needed to be laid back to accommodate two to three highwall ramps, as shown in the cross section. The overall pit exceeded the targeted mineralized material and contained ounces, with a corresponding increase in strip ratio.

Figure 15.8.3 Boin North Ultimate Pit Design



Figure 15.8.4 Boin North Pit Design versus Shell

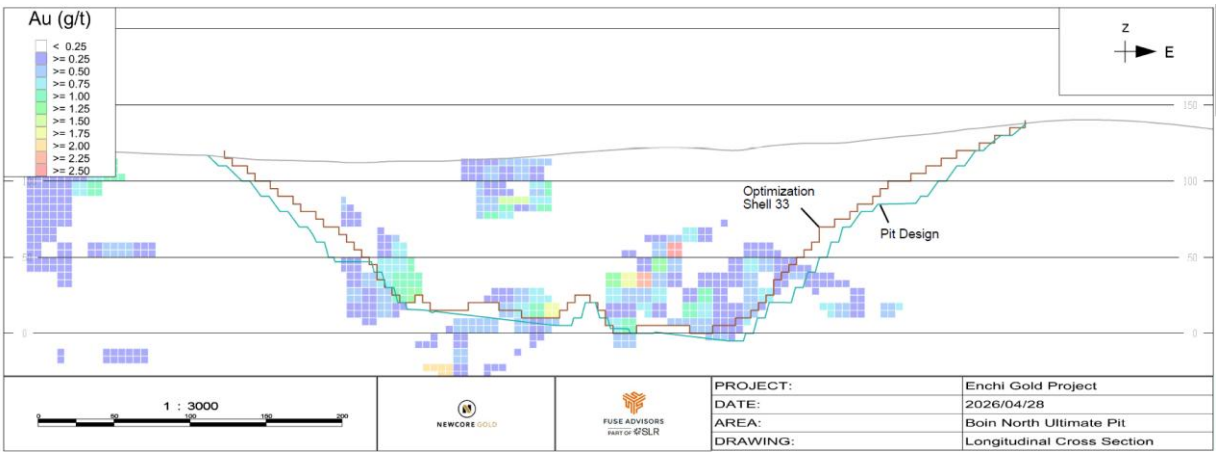


Table 15.8.4 Boin North Pit Shell and Design Reconciliation

Boin North	Pit Shell	Pit Design	% Difference
Mineralized Material (kt)	1,320	1,390	5%
Mineralized Material Grade (g/t)	0.81	0.79	-2%
Contained Gold (koz)	30	36	18%
Waste Material (kt)	8,540	12,100	42%
Total Material (kt)	9,870	13,490	37%
Strip Ratio (W:O)	6.5	8.7	35%

The Boin South ultimate pit design is shown in Figure 15.8.5. A cross-sectional comparison between the pit design and the corresponding pit shell is presented in Figure 15.8.6, and Table 15.8.5 summarizes the reconciliation results between the two. Because the Boin South pit design was relatively deep compared with its overall pit footprint, the pit walls were laid back to accommodate two to three highwall ramps. As a result, the overall pit achieved the targeted mineralized material, however, at a higher strip ratio.

Figure 15.8.5 Boin South Ultimate Pit Design

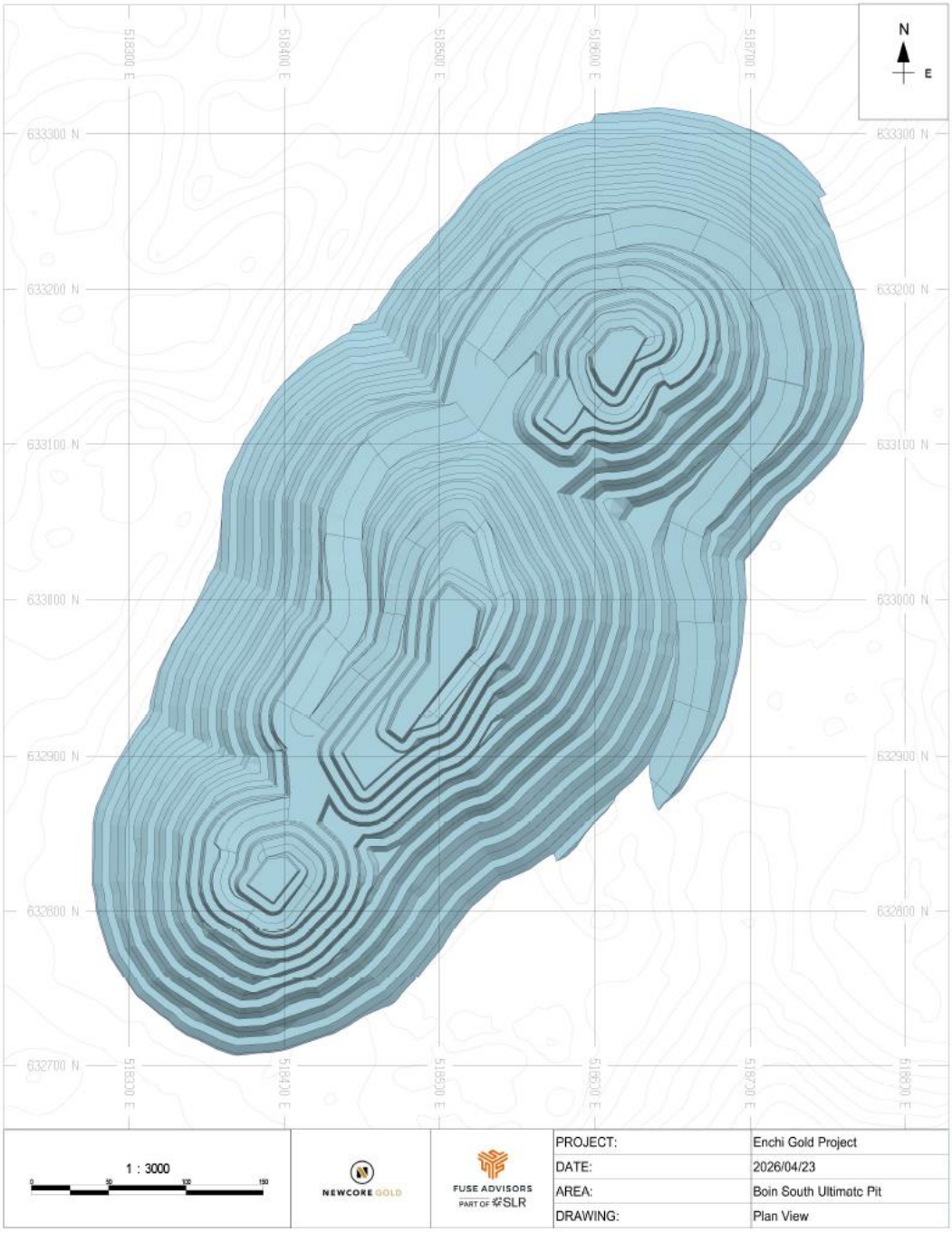


Figure 15.8.6 Boin South Pit Design versus Shell

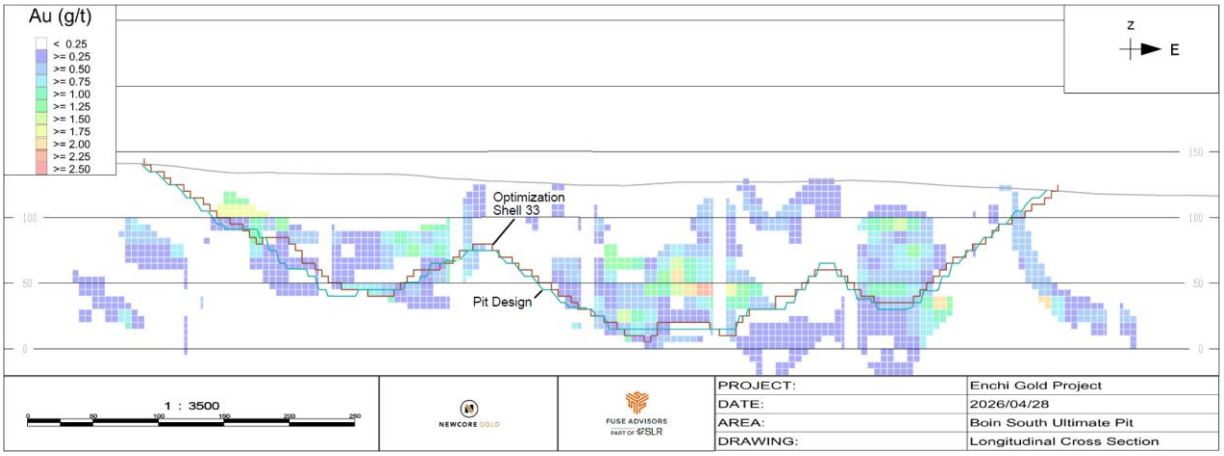


Table 15.8.5 Boin South Pit Shell and Design Reconciliation

Boin South	Pit Shell	Pit Design	% Difference
Mineralized Material (kt)	1,490	1,500	1%
Mineralized Material Grade (g/t)	0.75	0.73	-3%
Contained Gold (koz)	40	35	-13%
Waste Material (kt)	10,770	12,910	20%
Total Material (kt)	12,260	14,410	18%
Strip Ratio (W:O)	7.2	8.6	19%

Sewum Pit Design and Reconciliation

The Sewum Deposit was divided into three primary areas: Sewum North, Sewum Checkerboard, Sewum South. All Sewum pits were designed using 10 m benches, 5 m safety berms and variable bench face angles dependent on material type, described in Table 15.5.1.

The Sewum North ultimate pit design is shown in Figure 15.8.7. A cross-sectional comparison between the pit design and the corresponding pit shell is presented in Figure 15.8.8, and Table 15.8.6 summarizes the reconciliation results between the two. Additional waste was brought into the design to support more efficient haul routes providing operational flexibility and reduced haul distances to the mill and selected waste dump locations.

Figure 15.8.7 Sewum North Ultimate Pit Design

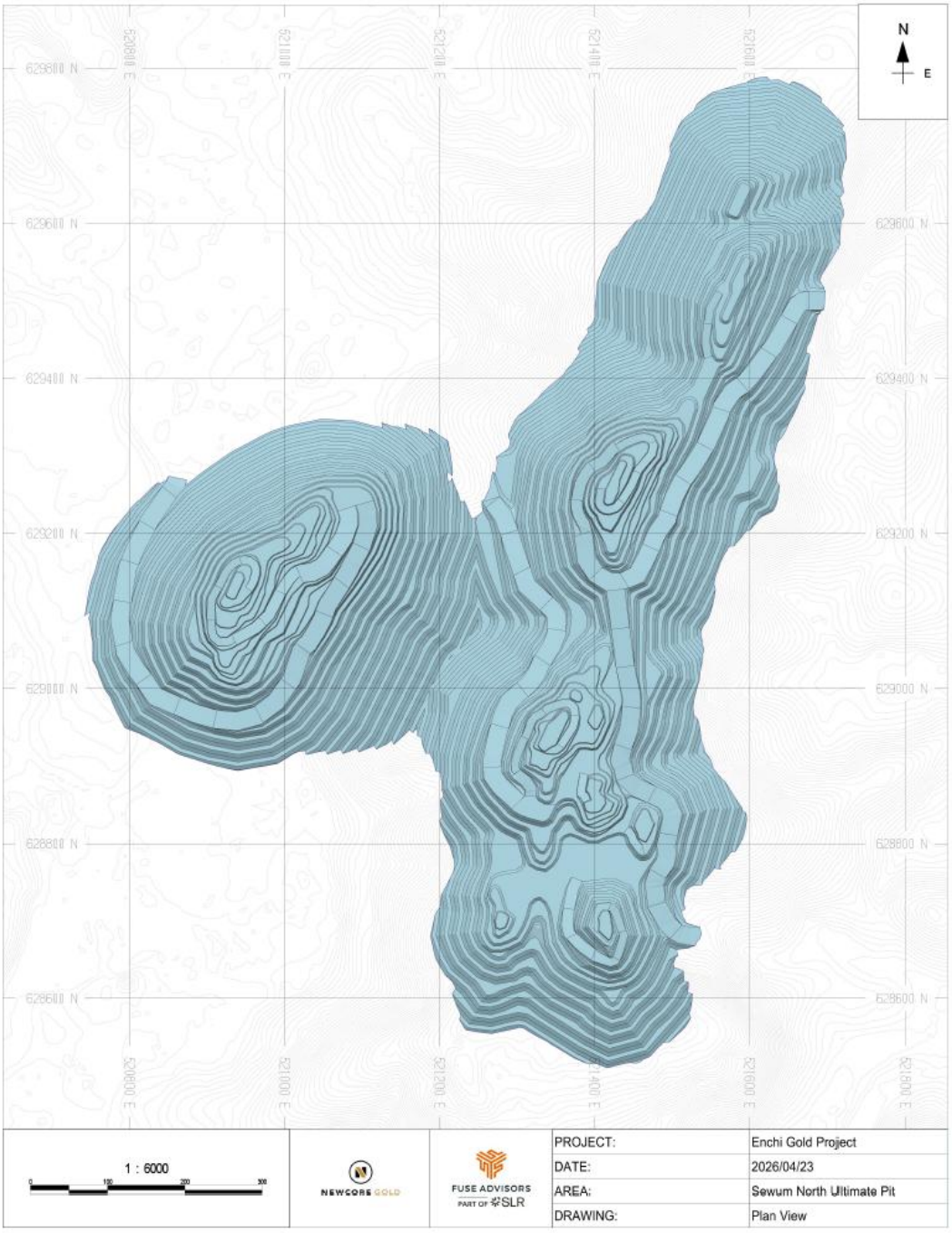


Figure 15.8.8 Sewum North Pit Design versus Shell

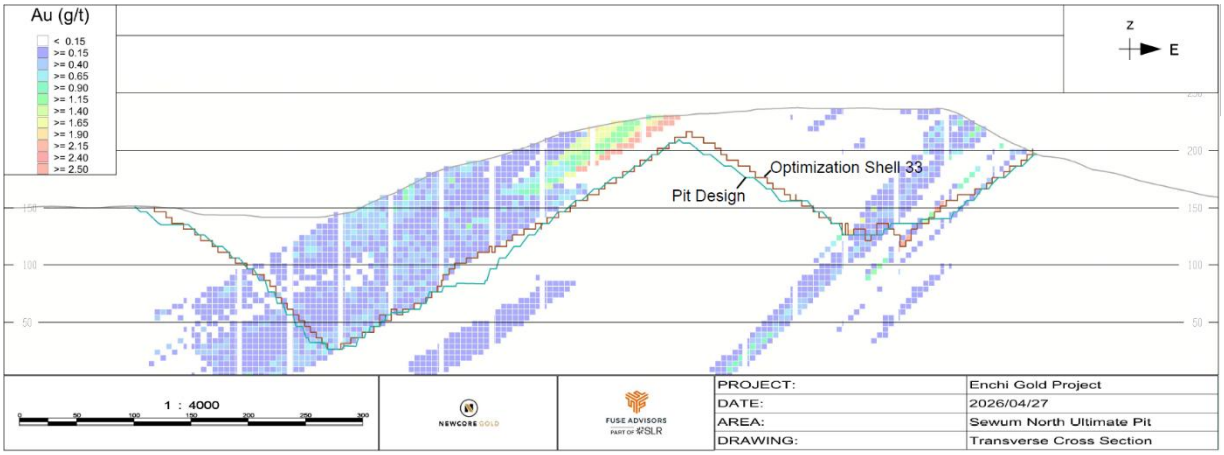


Table 15.8.6 Sewum North Ultimate Shell versus Design Reconciliation

Sewum North	Pit Shell	Pit Design	% Difference
Mineralized Material (kt)	16,570	16,690	1%
Mineralized Material Grade (g/t)	0.43	0.43	-1%
Contained Gold (koz)	229	228	0%
Waste Material (kt)	33,830	38,010	12%
Total Material (kt)	50,400	54,690	9%
Strip Ratio (W:O)	2.0	2.3	12%

The Sewum Checkerboard ultimate pit design is shown in Figure 15.8.9. A cross-sectional comparison between the pit design and the corresponding pit shell is presented in Figure 15.8.10, and Table 15.8.7 summarizes the reconciliation results between the two. The pit design achieves close alignment with the optimization shell.

Figure 15.8.9 Sewum Checkerboard Ultimate Pit Design



Figure 15.8.10 Sewum Checkerboard Pit Design versus Shell

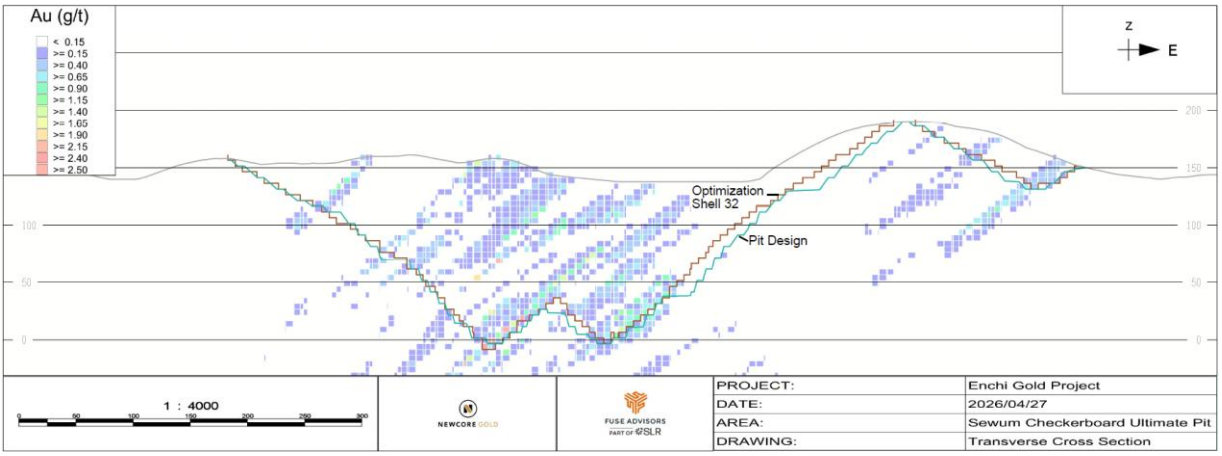


Table 15.8.7 Sewum Checkerboard Ultimate Shell versus Design Reconciliation

Sewum Checkerboard	Pit Shell	Pit Design	% Difference
Mineralized Material (kt)	5,960	6,110	3%
Mineralized Material Grade (g/t)	0.50	0.49	-1%
Contained Gold (koz)	95	96	1%
Waste Material (kt)	14,530	17,330	19%
Total Material (kt)	20,490	23,440	14%
Strip Ratio (W:O)	2.4	2.4	0%

The Sewum South ultimate pit designs are shown in Figure 15.8.11. Two cross-sections comparing between the pit designs and the corresponding pit shells is presented in Figure 15.8.12 and Figure 15.8.13, and Table 15.8.8 summarizes the reconciliation results between the two. The pit design achieves close alignment with the optimization shell.

Figure 15.8.11 Sewum South Ultimate Pit Design

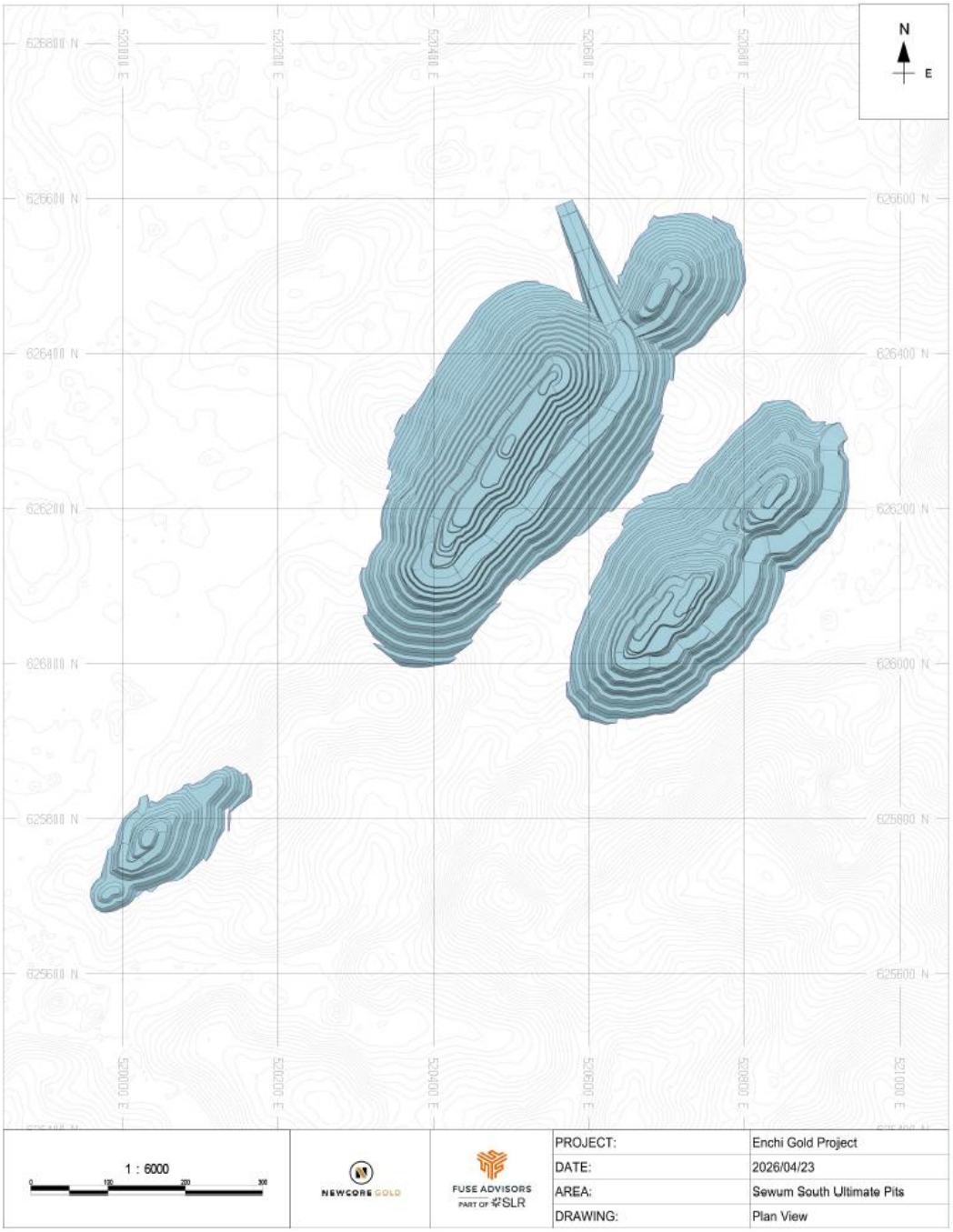


Figure 15.8.12 Sewum South Pit 3 Design versus Shell

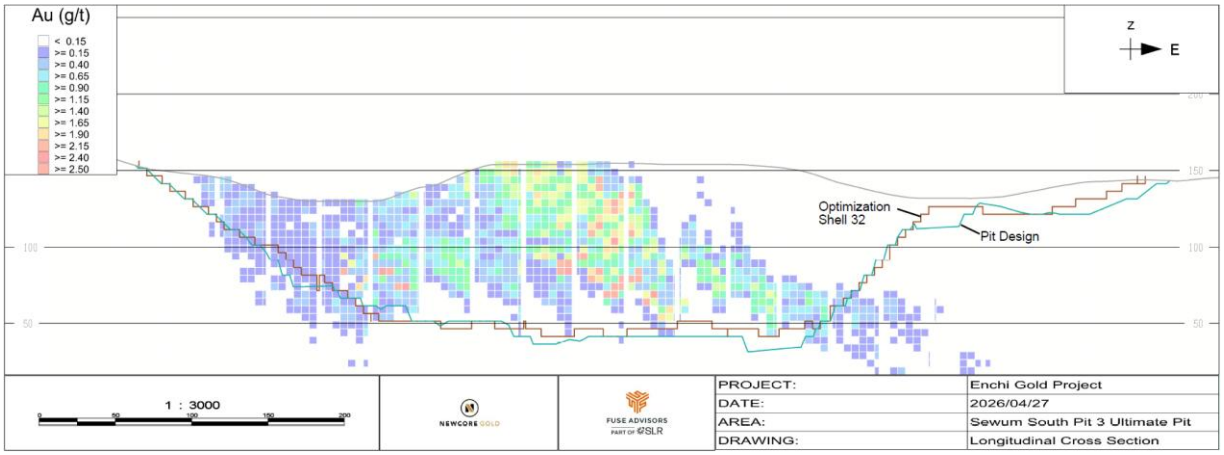


Figure 15.8.13 Sewum South Pit 4 Pit Design versus Shell

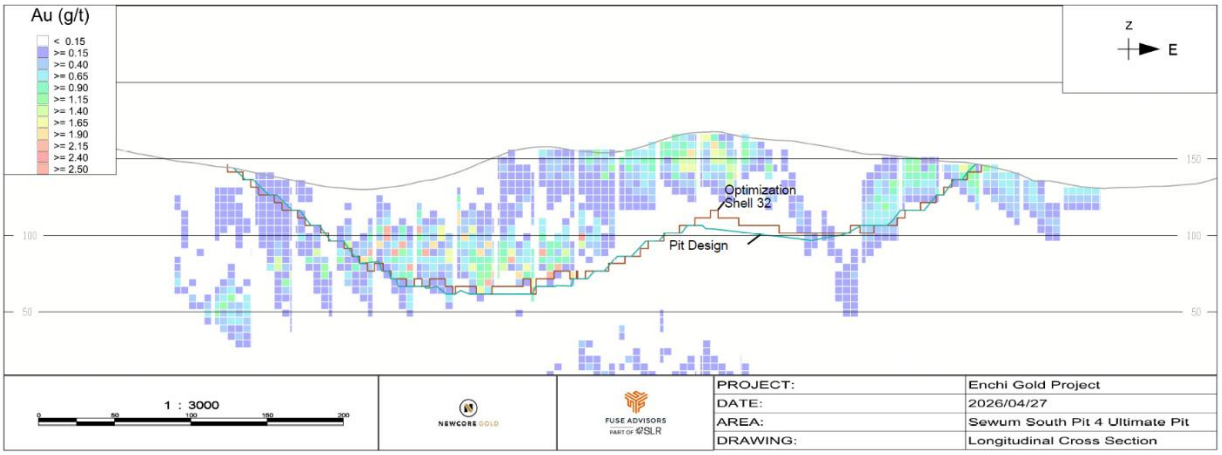


Table 15.8.8 Sewum South Ultimate Shells versus Design Reconciliation

Sewum South	Pit Shell	Pit Design	% Difference
Mineralized Material (kt)	3,430	3,420	0%
Mineralized Material Grade (g/t)	0.76	0.75	-1%
Contained Gold (koz)	84	82	-2%
Waste Material (kt)	14,370	15,720	9%
Total Material (kt)	17,800	19,130	7%
Strip Ratio (W:O)	4.2	4.6	10%

Nyam Pit Design and Reconciliation

The Nyam Deposit was designed as one pit, incorporating 12 m benches, 5 m safety berms and variable bench face angles dependent on material type, described in Table 15.5.1.

The Nyam pit design is shown in Figure 15.8.14. A cross-sectional comparison between the pit design and the corresponding pit shell is presented in Figure 15.8.15, and Table 15.8.9 summarizes the reconciliation results between the two. The overall reconciliation is well aligned, however, access to narrow pit bottoms resulted in some additional material in the pit design.

Figure 15.8.14 Nyam Pit Design

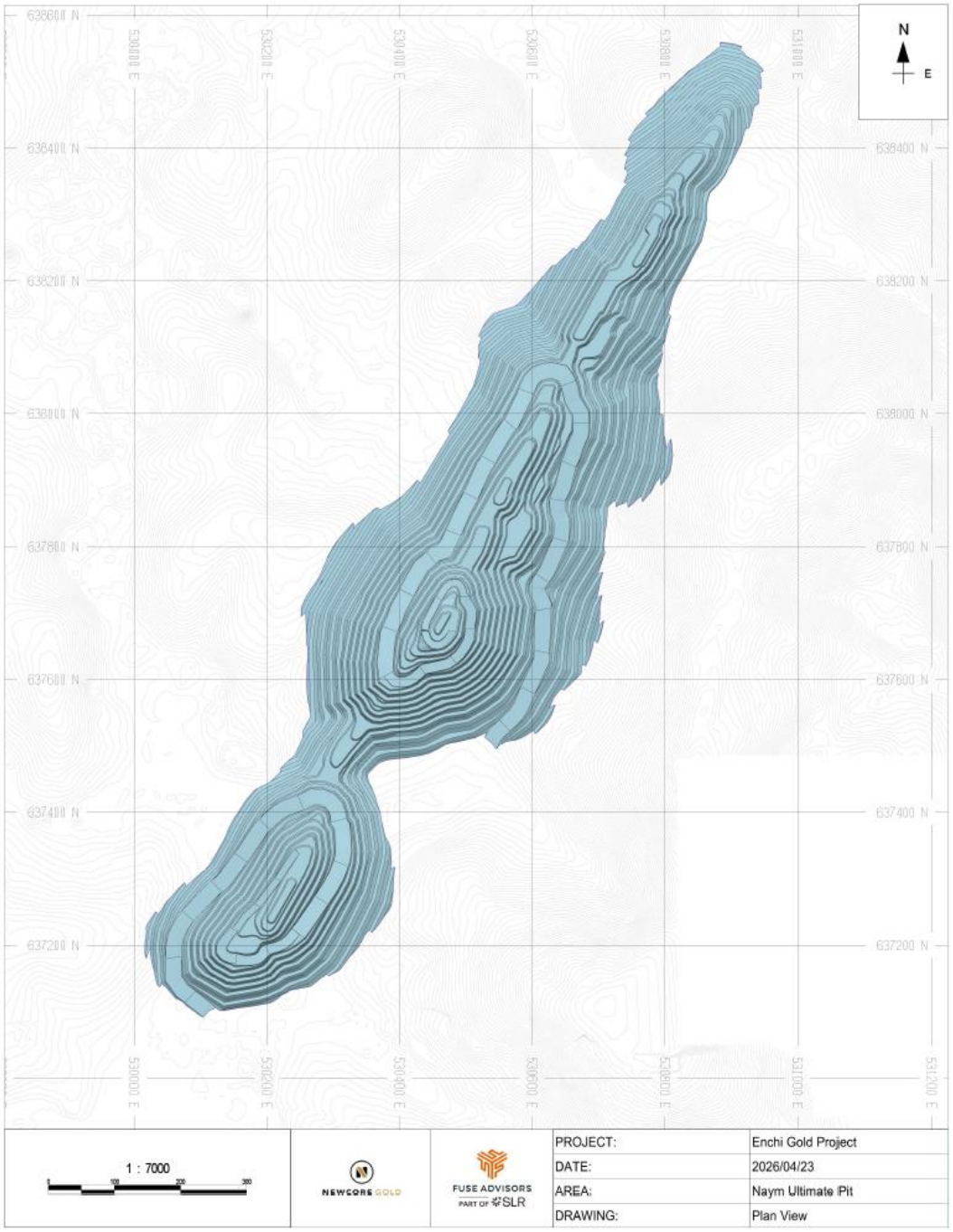


Figure 15.8.15 Nyam Pit Design versus Shell

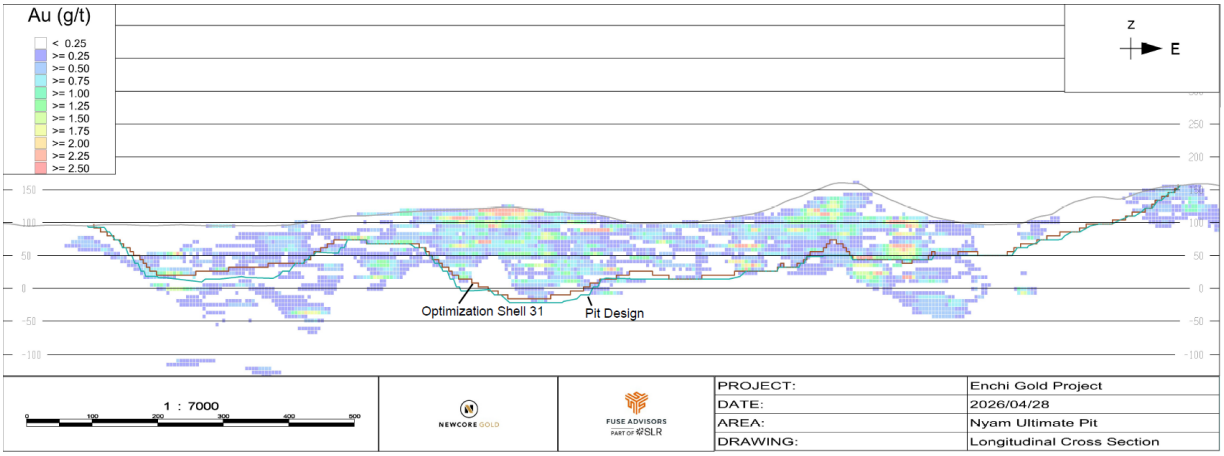


Table 15.8.9 Nyam Pit Shell versus Design Reconciliation

Nyam	Pit Shell	Pit Design	% Difference
Mineralized Material (kt)	6,950	7,190	3%
Mineralized Material Grade (g/t)	0.77	0.76	-1%
Contained Gold (koz)	172	175	2%
Waste Material (kt)	30,690	35,150	15%
Total Material (kt)	37,640	42,350	13%
Strip Ratio (W:O)	4.4	4.9	11%

Kwakyekrom Pit Design and Reconciliation

The Kwakyekrom Deposit was designed as one pit, incorporating 12 m benches, 5 m safety berms and variable bench face angles dependent on material type, described in Table 15.5.1.

The Kwakyekrom pit design is shown in Figure 15.8.16. A cross-sectional comparison between the pit design and the corresponding pit shell is presented in Figure 15.8.17, and Table 15.8.10 summarizes the reconciliation results between the two. Some additional waste was brought into the design to support more efficient haul routes; however, the overall reconciliation is relatively close to the optimization shell.

Figure 15.8.16 Kwakyekrom Pit Design

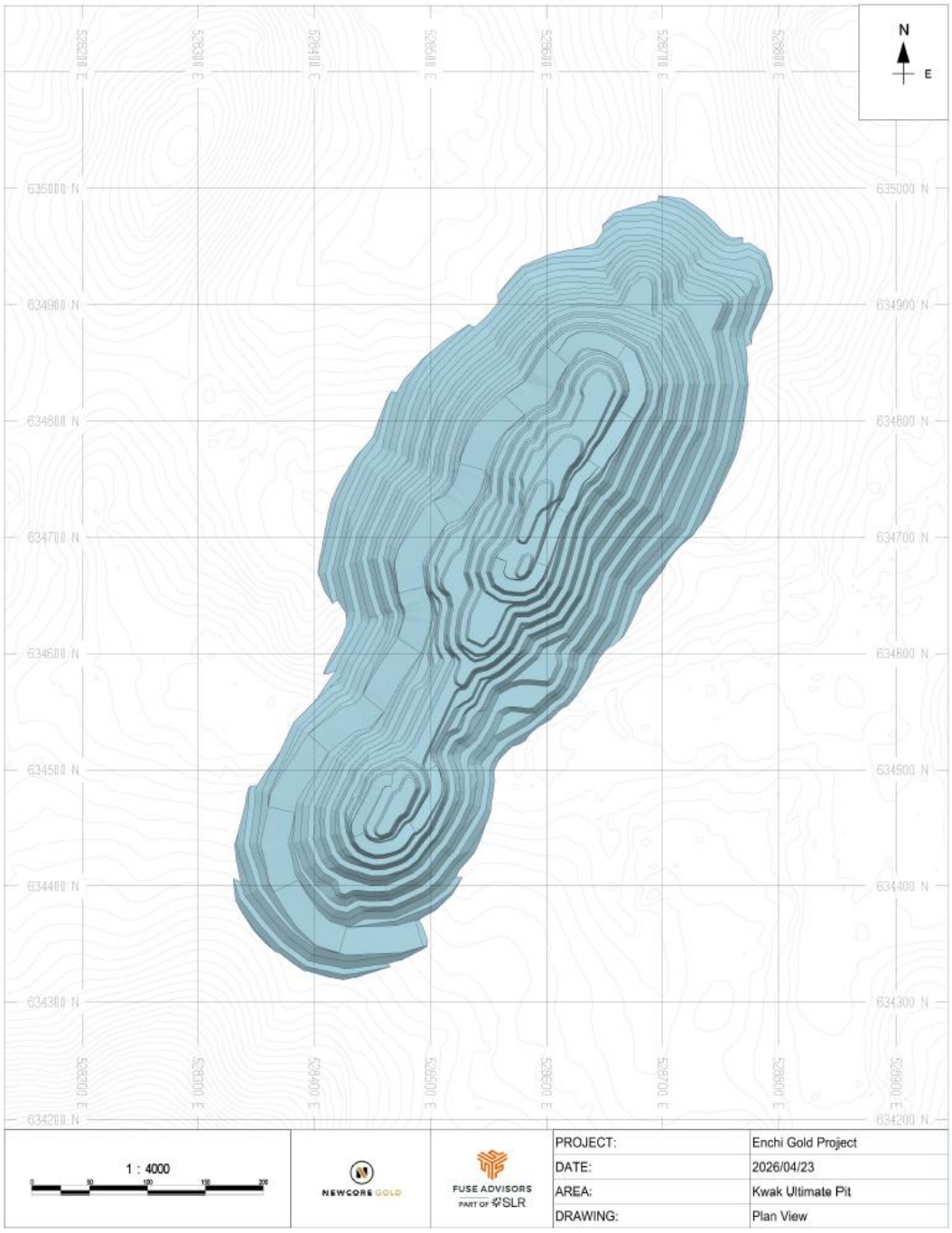


Figure 15.8.17 Kwakyekrom Pit Design versus Shell

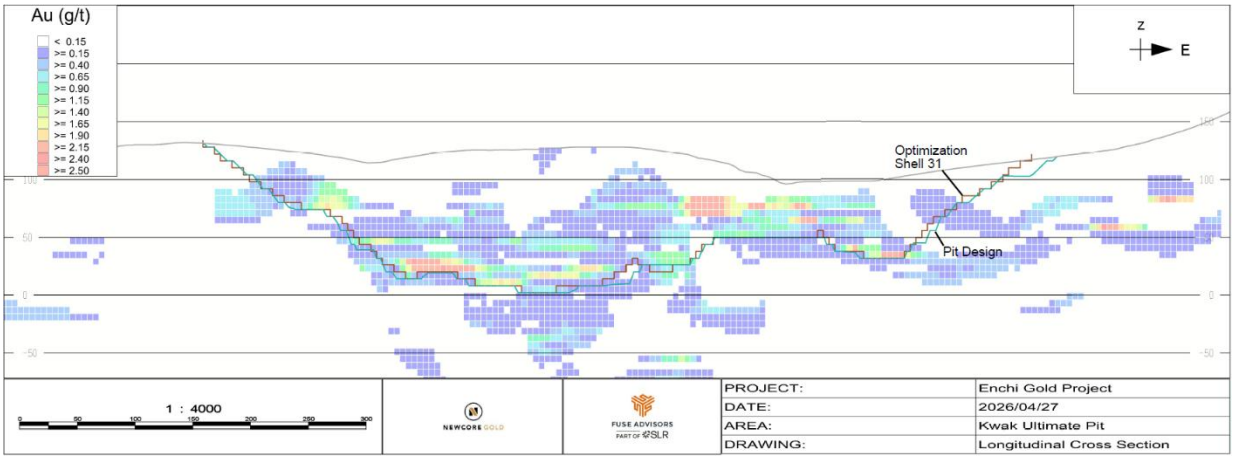


Table 15.8.10 Kwakyekrom Pit Shell versus Design Reconciliation

Kwakyekrom	Pit Shell	Pit Design	% Difference
Mineralized Material (kt)	1,960	1,980	1%
Mineralized Material Grade (g/t)	0.71	0.70	-2%
Contained Gold (koz)	45	45	-1%
Waste Material (kt)	9,990	11,100	11%
Total Material (kt)	11,950	13,070	9%
Strip Ratio (W:O)	5.1	5.6	10%

16.0 MINING METHODS

16.1 Introduction

The Enchi Gold Project is planned as an open pit mine with a conventional truck and shovel method of operations involving drilling, blasting, loading and hauling with diesel equipment to a centrally located Carbon-in-Leach (CIL) processing plant, and waste rock storage facilities located proximally to each mining area. The open pit mining operation is planned to be undertaken by a mining contractor.

The Project has four deposits which have been evaluated for the open pit Mineral Resources: Boin, Sewum, Nyam, and Kwakyekrom. The location of these deposits are illustrated in Figure 18.1.1. All four deposits were included in the Mineral Reserve Estimate and form the basis of the open pit life-of-mine (LOM) plan described below.

Mining will be conducted across four deposits and include 11 open pits divided into 17 phases. The pits will be mined in either 10 m or 12 m benches, completed in two 5.0 m or 6.0 m flitches, respectively. Oxide material will be free dug by hydraulic excavators, thus not requiring any drilling or blasting. A portion of the transition material is also expected to be sufficiently broken to allow for free digging, while the lower portion is expected to require drilling and blasting along with the fresh material. The boundary between free-dig and drill-and-blast material within the transition zone was defined using weathering profile data derived from core logging and reverse circulation (RC) chip observations. This data was then interpreted into a modeled surface used to separate material types during mine scheduling.

16.2 Pit Design

16.2.1 Geotechnical Design Criteria

The geotechnical criteria provided to Fuse by DRA was incorporated into the pit designs as detailed in Section 15.8. Table 16.2.1 summarizes the pit wall configuration by mining area and material type.

Table 16.2.1 Geotechnical Pit Design Criteria

Criteria	Unit	Boin			Sewum			Nyam			Kwakyekrom		
		Oxide	Trans	Fresh	Oxide	Trans	Fresh	Oxide	Trans	Fresh	Oxide	Trans	Fresh
Bench height	m	10	10	10	10	10	10	12	12	12	12	12	12
Bench stacking	#	1	1	1	1	1	1	1	1	1	1	1	1
Max bench height	m	10	10	10	10	10	10	12	12	12	12	12	12
Bench face angle	deg	45	55	75	45	55	75	45	55	70	45	55	70
Bench width	m	5	5	5	5	5	5	5	5	5	5	5	5

Criteria	Unit	Boin			Sewum			Nyam			Kwakyekrom		
		Oxide	Trans	Fresh	Oxide	Trans	Fresh	Oxide	Trans	Fresh	Oxide	Trans	Fresh
IRA	deg	34	40	52	34	40	52	35	42	52	35	42	52

Source: Parameters Provided by DRA [C8808 - Enchi - Mine Design Criteria - Rev D_260206_CB.xls] on 6 February 2026

16.2.2 Haul Road Design Criteria

The design criteria, assumptions, and detailed considerations applied in the development of haul road geometry, including width allowances, adherence to local regulations, and operational requirements, are detailed in Section 15.8.2, which provides the governing basis for the haul road design that was adopted for the Project.

16.3 Waste Rock Storage Facility (WRSF) Design

The waste rock storage facilities were designed at an angle of repose of 37° with 20 m lifts and 15 m berms. Upon closure, the WRSF will be contoured to an overall slope flatter than 3H:1V for benches exceeding 10 m in height, in accordance with Minerals and Mining (Health, Safety and Technical) Regulations, 2012 (Ghana), specifically Regulation 286 governing the closure of waste dumps.

Design capacity requirements for the WRSF were based on swell factors specific to each material weathering type, using the following assumptions:

- Oxide: swell of 20%.
- Transition: swell of 30%.
- Fresh: swell of 30%.

The following table summarizes the Loose Cubic Metres (LCM) design capacity and maximum overall WRSF height for each facility as well as the volume requirements based on the LOM production schedule. The LOM plan requires 147 M LCM of dumping capacity, with the current designs offering 152 M LCM with an opportunity to add additional lifts on top, as needed. WRSF designs are shown in Figure 16.4.1 to 16.4.3 in Section 16.4 below.

Geotechnical investigations to date have been limited. As a result, the proposed WRSF designs and locations will require further analysis and confirmation of appropriate assumptions from a geotechnical stability perspective in future studies. Knight Piésold Consulting reviewed the dumps included in the LOM mine plan and recommended further evaluation of additional technical, environmental and social factors including:

- Foundation bearing capacity.

- Waste rock characterization in relation to acid rock drainage.
- Visual impact of the dumps to neighboring communities, prevailing wind direction and its contribution to air quality, aquifer locations and proximity to surface water sources.

Given the predominantly low-lying topography and the high precipitation conditions anticipated at site, it is expected that foundation preparation will be required prior to dump construction. Such preparation may include stripping of unsuitable materials, subgrade improvement, and implementation of appropriate drainage controls to ensure stability and long-term performance.

Table 16.3.1 Waste Rock Storage Facility Design Capacities

Dump	Design Volume LCM	Required Volume LCM	Max Dump Height m
Boin Central	54,060,000	70,339,000	140
Boin North	7,730,000		80
Boin South	8,860,000		80
Sewum Northeast	35,670,000	49,233,000	120
Sewum Checkerboard	14,350,000		60
Sewum Northwest	1,450,000		40
Nyam North	1,800,000	20,822,000	80
Nyam South	20,750,000		100
Kwakyekrom	7,200,000	6,973,000	80
Total Dump Capacity	**151,870,000**	**147,367,000**	

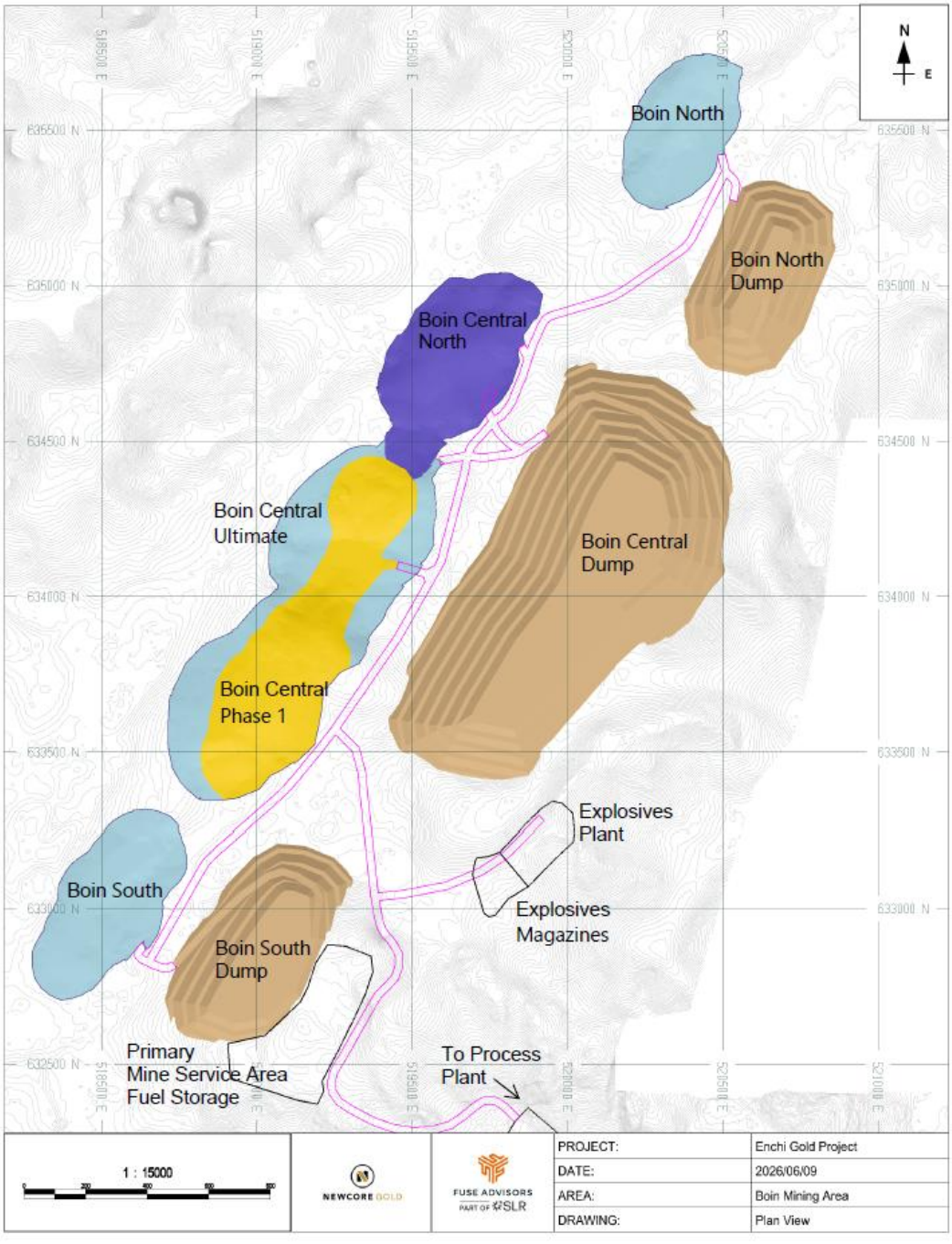
16.4 Mine Layout

The current LOM plan includes mining from 11 pits across four deposits: Boin, Sewum, Nyam and Kwakyekrom. Each mining area will include adjacent waste rock storage facilities and a temporary run-of-mine (ROM) stockpile. Ore will either be loaded from stockpile or direct from pit to a common CIL Processing Plant location. An overview of each mining area is described below and illustrated in their respective figures.

Boin Mining Area

Boin is located approximately 3.6 km west of the selected process plant location and is the first priority area for mining operations. Boin is comprised of three main pits: Boin Central, Boin South, Boin North. The Boin Central pit is further sub-phased into Boin Central North, Boin Central Phase 1 and Boin Central Ultimate. The current LOM plan includes three separate WRSF, located adjacent to each of the three main pits (Boin Central WRSF, Boin South WRSF, Boin North WRSF). A temporary ROM stockpile is also incorporated into the plan, however, most of the mineralize material is scheduled to directly report to the process plant site.

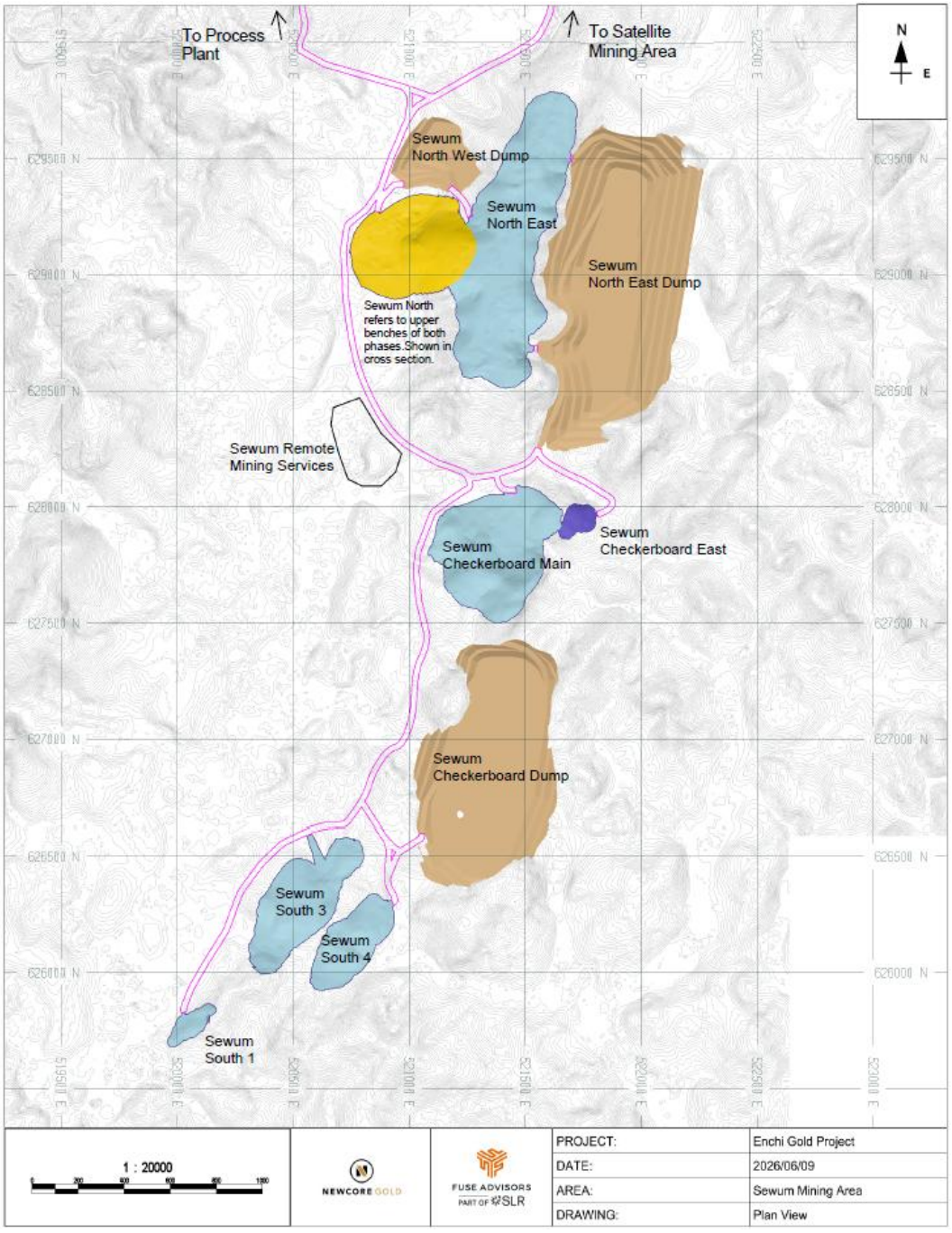
Figure 16.4.1 Boin Mining Area



Sewum Mining Area

Sewum is located approximately 7.8 km southeast of the selected process plant location, as shown in Figure 16.4.2. Sewum is comprised of three pit areas: Sewum North, Sewum Checkerboard, Sewum South. Sewum North begins as a single pit and then separates into an east and a west pit as the upper benches are mined. Sewum Checkerboard is divided into a main pit and a small satellite pit, named Sewum Checkerboard East. Sewum South consists of three individual pits: Sewum South Pit 1, Sewum South Pit 3, Sewum South Pit 4. Sewum South Pit 2 was excluded from the mine schedule following the pit design stage due to its low overall value contribution. The current LOM plan includes three WRSFs for Sewum (Sewum Northwest WRSF, Sewum Northeast WRSF, Sewum Checkerboard WRSF). A temporary ROM stockpile is also incorporated into the plan, however, most of the mineralize material is scheduled to directly report to the process plant site.

Figure 16.4.2 Sewum Mining Area

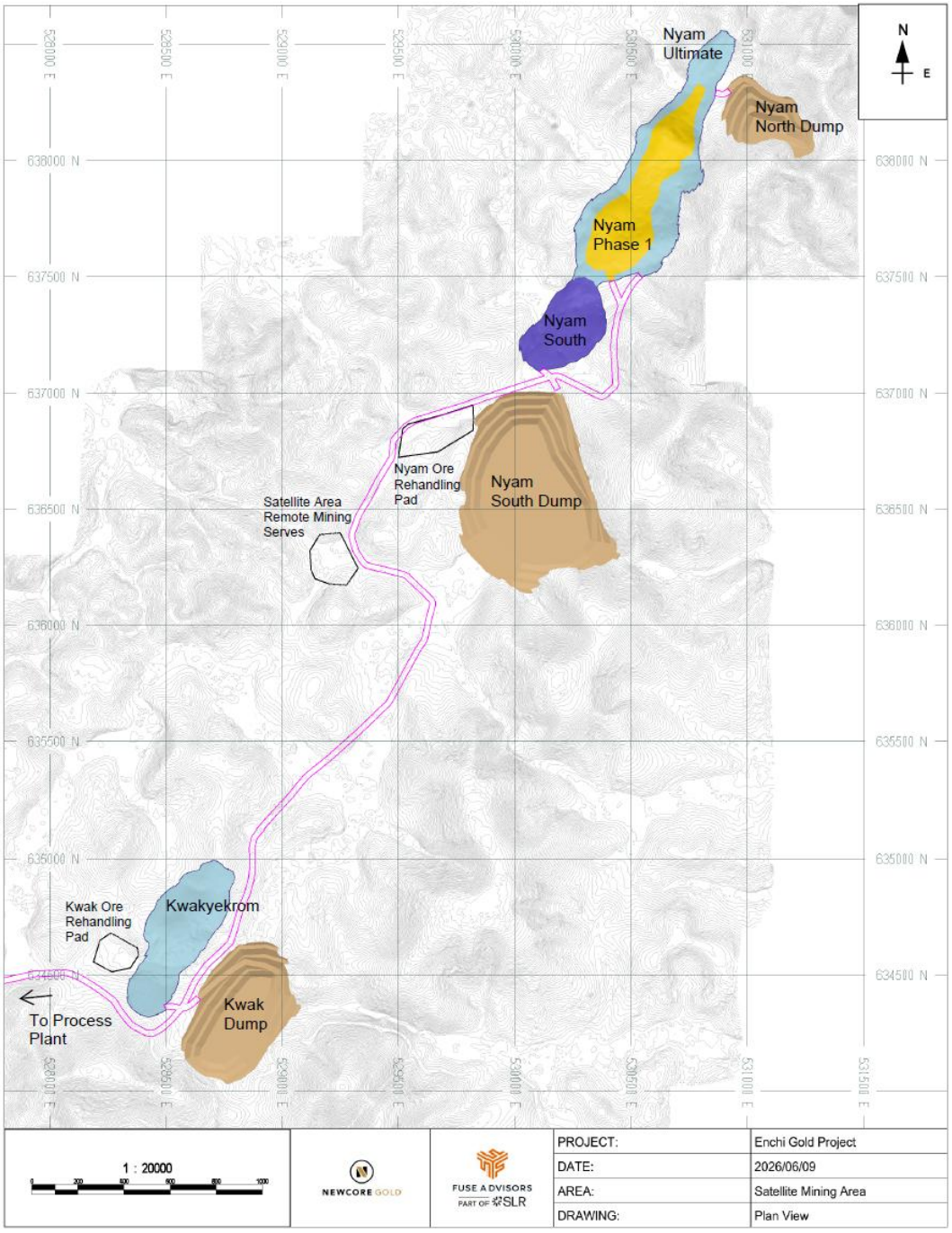


Satellite Mining Area

Nyam is located approximately 19.5 km northeast of the selected process plant site. The deposit is comprised of a single pit, subdivided into three phases: Nyam South, Nyam Main Phase 1, Nyam Ultimate. The current LOM plan includes two WRSFs (Nyam North WRSF, Nyam South WRSF). A temporary ROM stockpile is also incorporated into the plan. All ore from Nyam will initially be placed in a ROM stockpile, then loaded and hauled using a dedicated surface fleet designed for transport along the public highway connecting the Nyam mining area to the process plant site.

Kwakyekrom is located approximately 15.1 km northeast of the selected process plant location and 4 km south of the Nyam mining area. Kwakyekrom is comprised of one pit scheduled to be mined in one phase. The current LOM plan includes one WRSF (Kwakyekrom WRSF). A temporary ROM stockpile is also incorporated into the plan. All ore from Kwakyekrom will initially be placed in a ROM stockpile, then loaded and hauled using a dedicated surface fleet designed for transport along the public highway connecting the Kwakyekrom mining area to the process plant site. The layout of the satellite mining area is shown in Figure 16.4.3.

Figure 16.4.3 Satellite Mining Areas



16.4.2 Mining Infrastructure

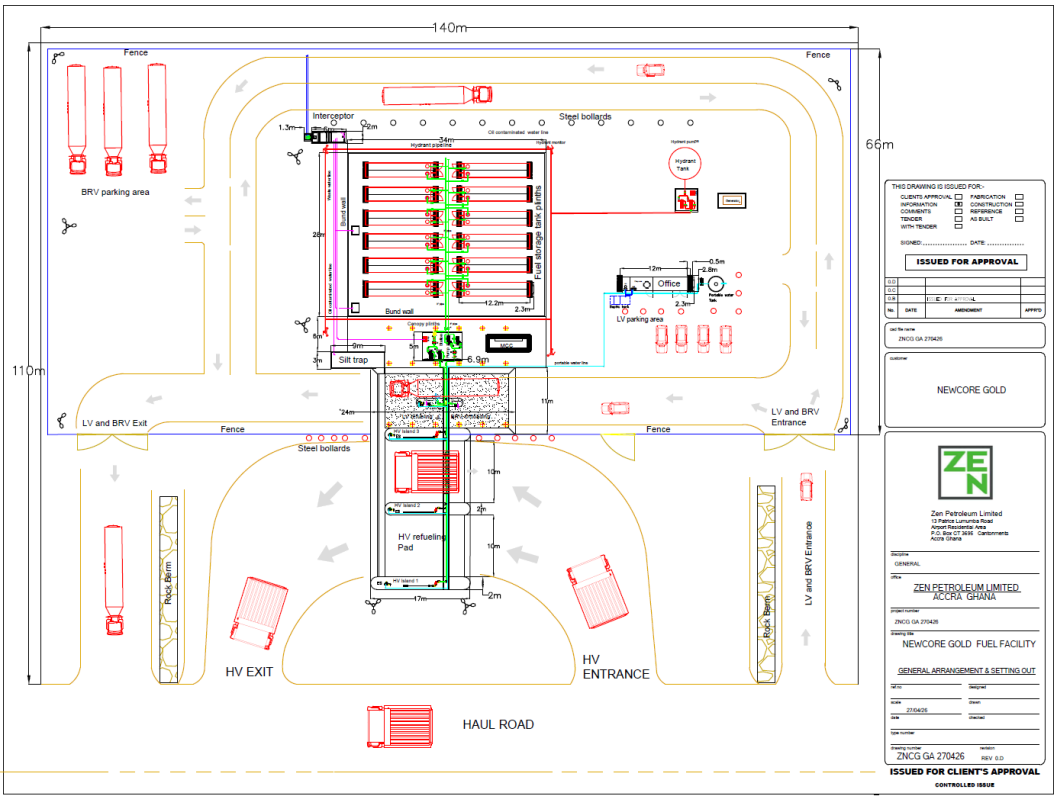
Primary Mine Service Area

A primary Mining Service Area (MSA) will be established adjacent to the process plant near the Boin deposit, occupying approximately 11.5 hectares. This facility will serve as the central hub for mining support activities and is planned to include a Heavy Mobile Equipment (HME) workshop and warehouse, lubrication building, wash bay, tire bay, and a main mining office. Personnel support infrastructure will also consist of a kitchen and dry mess, training room, bathroom facilities, and change rooms.

An integrated fuel storage and distribution center will be located within the primary MSA (refer to Figure 16.4.4). The fuel facility will comprise 12 storage tanks, each with a capacity of 60,000 L, and will be operated and maintained by a dedicated fuel distribution contractor. Associated infrastructure will include a fuel operations office, light vehicle (LV) parking, fuel truck parking, a light vehicle fueling station, and a heavy vehicle fueling pad.

The primary MSA will be supplied with up to 1 MW of electrical power via connection to the site-wide power network.

Figure 16.4.4 Fuel Storage and Distribution Facility



Explosive Management Area

Explosives management infrastructure is planned to consist of two discrete locations separated by engineered berms, including a magazine storage area and an explosives manufacturing facility. These installations will be spatially segregated from other operational areas and designed in compliance with the Ghanaian Minerals and Mining (Explosives) Regulations, 2012 (LI 2177). The total Net Explosive Quantity (NEQ) will be maintained below 50,000 kg.

Secondary Mine Service Areas

Secondary MSAs will be developed to support mining operations at Sewum, as well as the satellite deposits at Nyam and Kwakyekrom. These facilities will be diesel generator-powered and will consist of temporary maintenance shelters, laydown areas, and outdoor working zones. Their primary function is to facilitate minor maintenance activities and reduce the need for equipment transfer to the primary MSA.

In addition, small ore rehandling pads will be constructed proximal to the Nyam and Kwakyekrom pits to support ore loading and haulage operations. Ore will be transported from these locations to the process plant via an access road approximately 16 km in length that will connect with the Sewum ore haulage route.

Sewerage Handling

All mine operations ablution facilities on the mine site will be serviced by septic tanks and soak aways for sewage disposal. A local contractor will empty the septic tanks at the required intervals and dispose of the solid waste at a licensed sewage treatment site.

16.5 Mine Phases

Phasing within the ultimate pit designs was intended to prioritize higher-value areas of each deposit, typically characterized by higher grades, lower strip ratios, and/or reduced mining and processing costs. This approach commonly results in the development of smaller starter pits, which enable earlier access to higher-margin material and generate early revenue to help offset initial project development costs.

For the Enchi Gold Project, phased development was guided by the selection of lower revenue factor LG shells that captured the highest-value portions of the Mineral Reserve early in the mine life. Concurrently, phase sizes were carefully evaluated to ensure they provided adequate mining widths, practical access, and operational flexibility to support efficient mining. The following sections outline the phases by mining area along with the associated inventory by phase.

16.5.1 Boin Pit Phasing

The Boin Central pit extends approximately 2,000 m in length and 400 m in width, providing sufficient scale to support staged development through interim phasing. As a result, the pit was subdivided into three phases: Boin Central Phase 1, Boin Central North, Boin Central Ultimate pit. This phased approach enables earlier access to higher-value material while maintaining practical mining geometries and access. In contrast, phasing of the Boin North and Boin South pits was not considered beneficial to the Project, as it did not result in meaningful improvements to project economics or executability. Table 16.5.1 details the inventory of all Boin phases. Figure 16.5.1 illustrates a plan view and the associated pit and phase names.

Table 16.5.1 Boin Phase Inventories

Phase	Ore kt	Ore Au Grade g/t	Contained Gold koz	Waste Material kt	Total Material kt	Strip Ratio W:O
Central North	2,850	1.01	92.7	21,280	24,120	7.5
Central Phase 1	3,700	0.89	106.2	16,980	20,690	4.6
Central Ultimate	6,460	0.77	159.7	44,970	51,440	7.0
Boin North	1,390	0.79	35.5	12,100	13,490	8.7
Boin South	1,500	0.73	35.0	12,910	14,410	8.6
Boin Total	15,910	0.84	429.0	108,240	124,150	6.8

Table data has been rounded; totals may not equal the sum of individual values

Figure 16.5.1 Boin Deposit Phasing

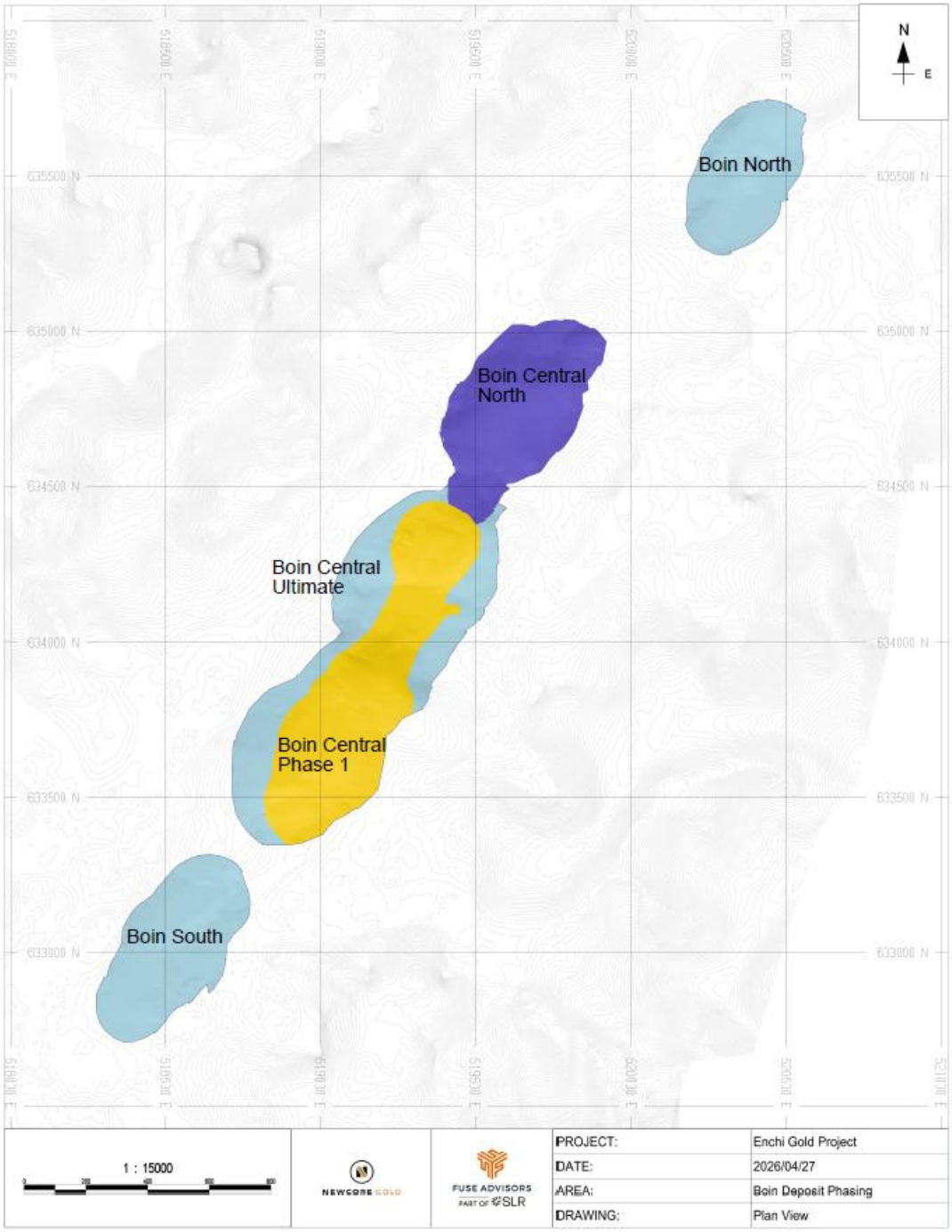
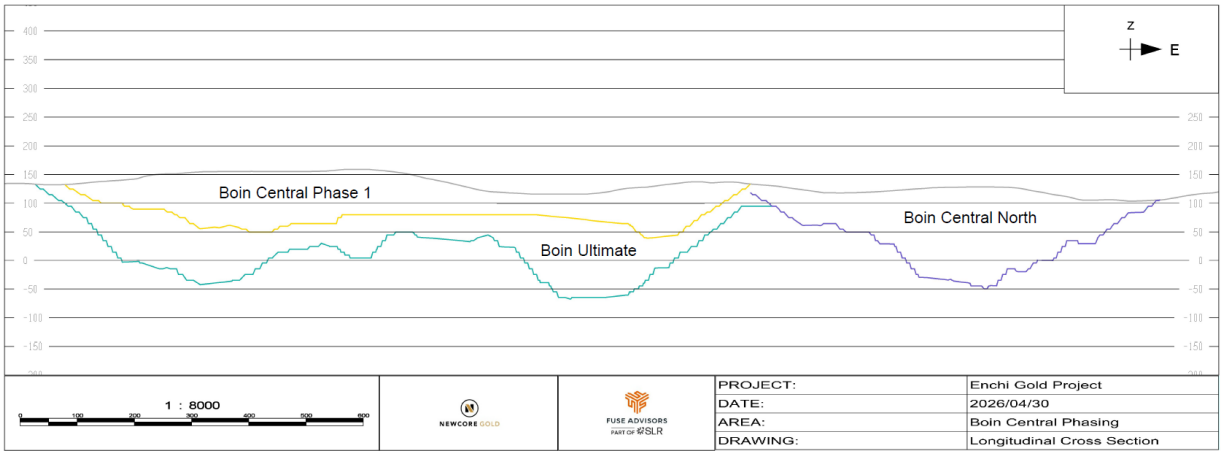


Figure 16.5.2 Boin Central Phasing Cross Section



16.5.2 Sewum Pit Phasing

The Sewum South pits and the Sewum Checkerboard pits did not present viable opportunities for phased development. This was primarily due to their small pit sizes and the generally homogeneous nature of the mineralization, which did not provide distinct zones of higher-margin material that could be preferentially targeted in early phases. Sewum North was subdivided into three phases. This approach was largely driven by the pit optimization results, where the revenue factor shells naturally separated into distinct, spatially discrete mining areas, as illustrated in Figure 16.5.3 and Figure 16.5.4. The primary purpose in phasing the pit allowed for scheduling flexibility when selecting priority benches. The pit inventory by phase is summarized in Table 16.5.2.

Table 16.5.2 Sewum Phase Inventories

Phase	Ore kt	Ore Au Grade g/t	Contained Gold koz	Waste Material kt	Total Material kt	Strip Ratio W:O
Checkerboard East	110	0.37	1.4	340	450	2.9
Checkerboard Main	6,000	0.49	94.9	16,990	22,990	2.8
North	8,900	0.40	115.5	26,640	35,540	3.0
North East	2,030	0.52	34.2	4,280	6,310	2.1
North West	5,760	0.42	78.3	7,090	12,840	1.2
South Pit 1	160	0.54	2.7	280	440	1.8
South Pit 3	2,040	0.74	48.5	10,440	12,480	5.1
South Pit 4	1,220	0.79	30.8	5,000	6,210	4.1
Sewum Total	**26,210**	**0.48**	**406.3**	**71,050**	**97,260**	**2.7**

Table data has been rounded; totals may not equal the sum of individual values

Figure 16.5.3 Sewum Deposit Phasing

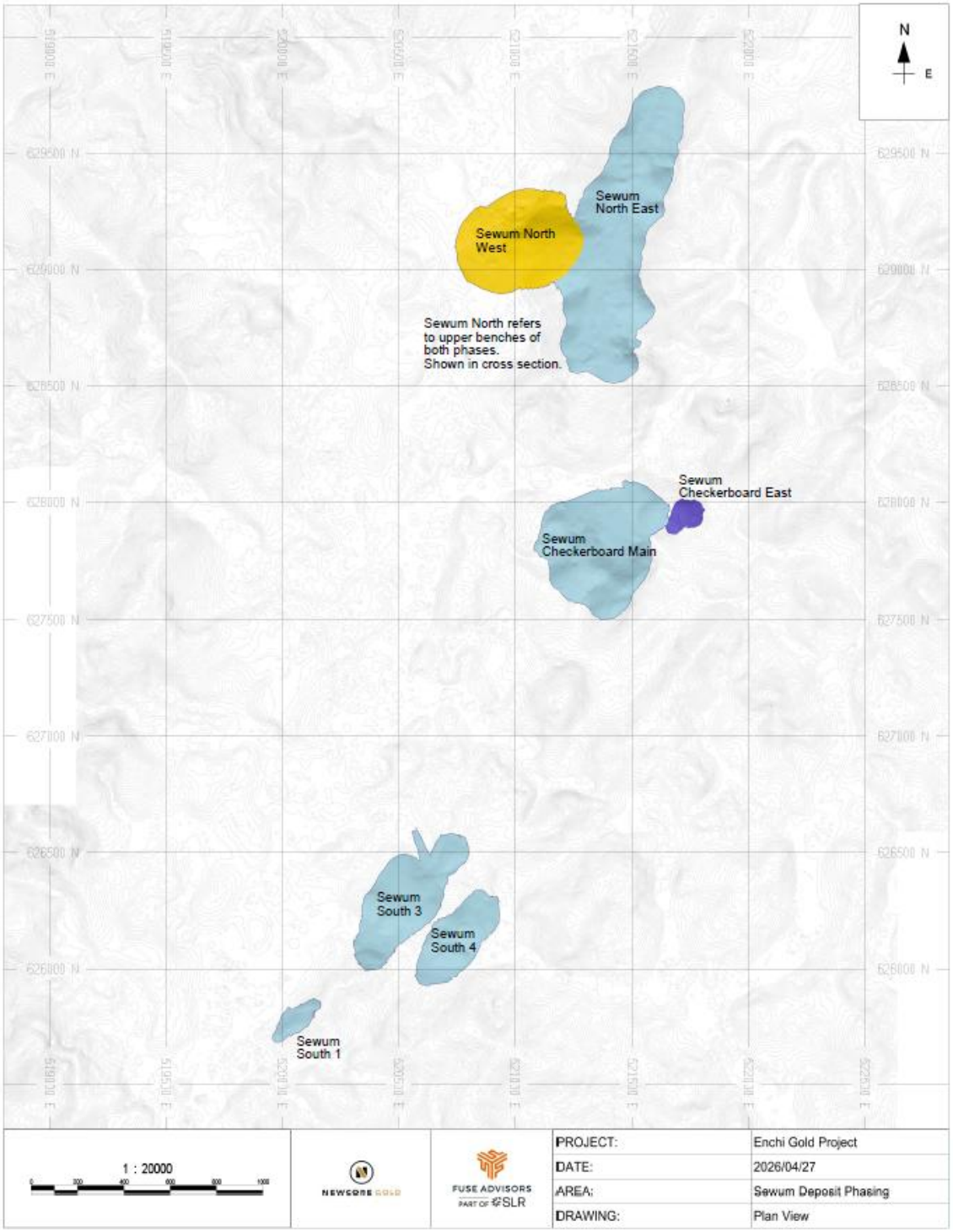
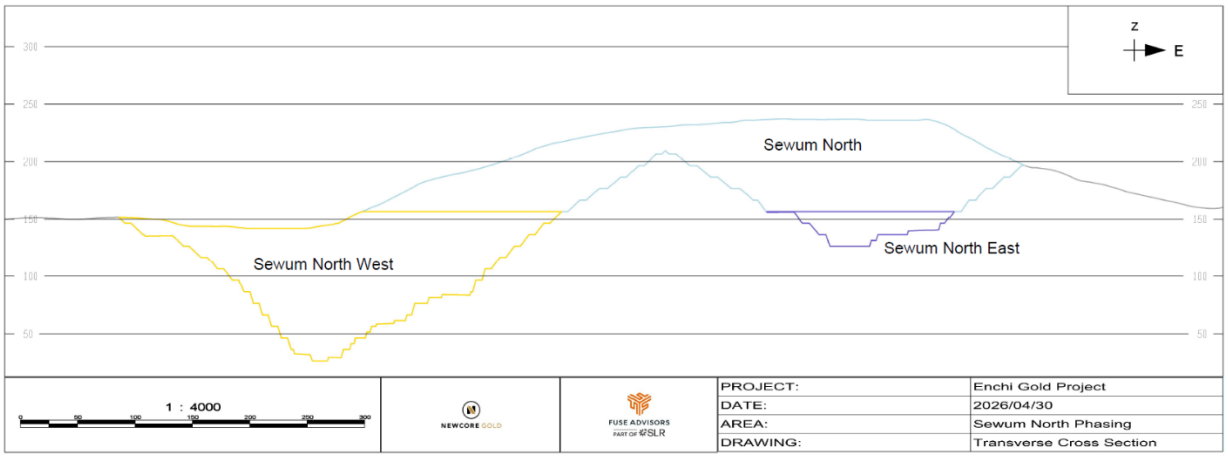


Figure 16.5.4 Sewum North Phasing Cross Section



16.5.3 Nyam Pit Phasing

The Nyam pit extends approximately 1,600 m in length and 300 m in width, providing sufficient scale to support staged development through interim phasing. Nyam was hence divided into three phases; Nyam Main Phase 1 which was defined by a low strip ratio area, the remaining Main Ultimate pit, and Nyam South which was driven by revenue factor shells naturally separated into distinct, spatially discrete mining areas. These phases can be demonstrated in the plan view shown in Figure 16.5.5 and cross section in Figure 16.5.6. The phase inventories are summarized in Table 16.5.3.

Table 16.5.3 Nyam Phase Inventories

Phase	Ore kt	Ore Au Grade g/t	Contained Gold koz	Waste Material kt	Total Material kt	Strip Ratio W:O
Main Phase 1	2,710	0.78	67.7	7,050	9,750	2.6
South	1,480	0.77	36.7	7,080	8,560	4.8
Main Ultimate	3,010	0.73	70.7	21,030	24,030	7.0
Nyam Total	**7,190**	**0.76**	**175.1**	**35,150**	**42,350**	**4.9**

Table data has been rounded; totals may not equal the sum of individual values

Figure 16.5.5 Nyam Deposit Phasing

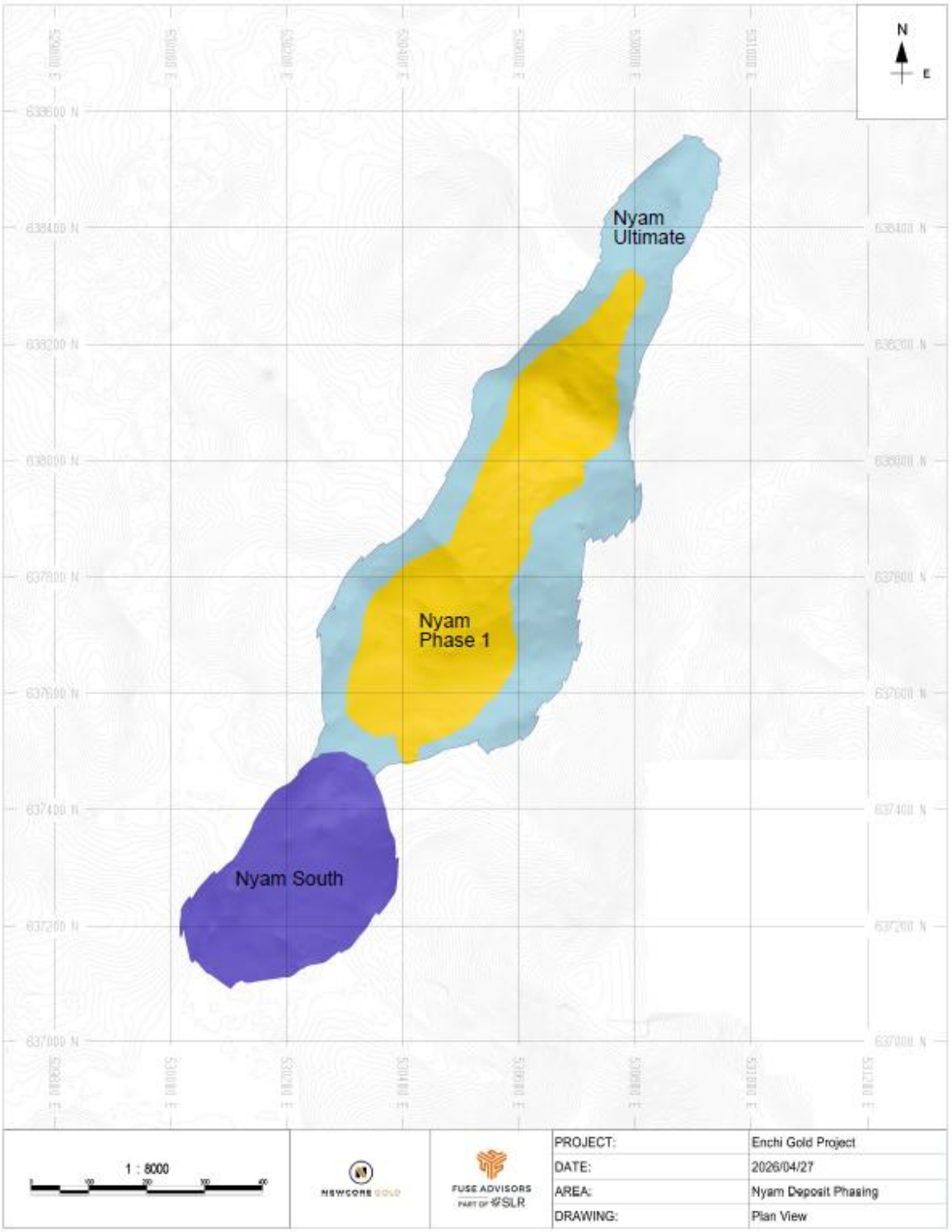
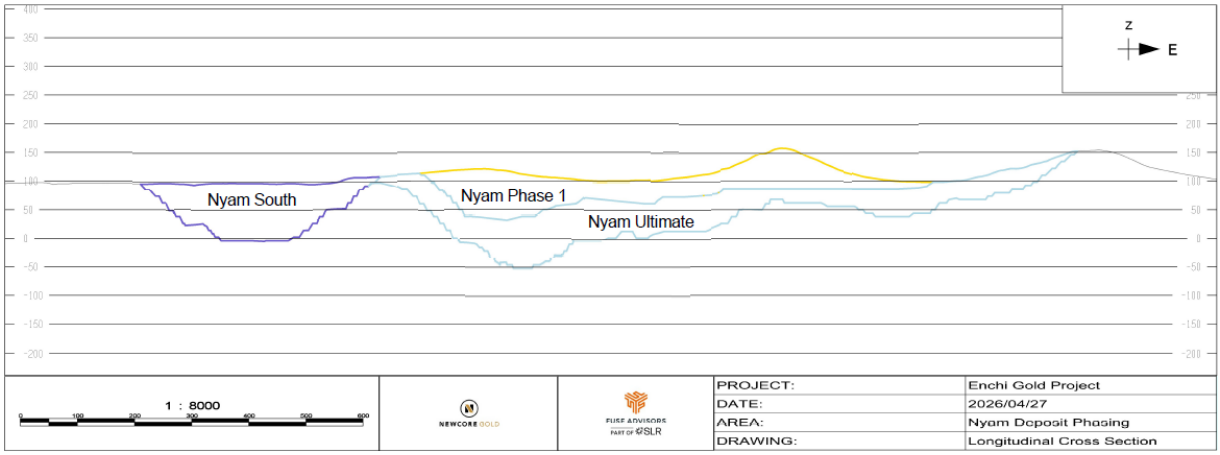


Figure 16.5.6 Nyam Pit Phasing Cross Section



16.5.4 Kwakyekrom Pit Phasing

The Kwakyekrom ultimate pit did not present viable opportunities for phased development. Table 16.5.4 outlines the pit inventory.

Table 16.5.4 Kwakyekrom Phase Inventories

Phase	Ore kt	Ore Au Grade g/t	Contained Gold koz	Waste Material kt	Total Material kt	Strip Ratio W:O
Kwakyekrom	1,980	0.70	44.5	11,100	13,070	5.6

Table data has been rounded; totals may not equal the sum of individual values

16.6 Mine Schedule

The mine sequencing was developed using the Hexagon's MinePlan Schedule Optimizer (MPSO). The production was scheduled in quarterly increments to ensure a practical and adaptable plan. MPSO optimizes based on project NPV, as such costs and revenue factors were incorporated into the scheduler to optimize the Plan, while using a series of constraints to produce an executable schedule. The mining inventory was separated into 10 m or 12 m benches for scheduling, depending on the pit geometry of the mining area. The discrete phases described in Section 16.5 were used for scheduling to provide more optionality and optimization.

16.6.1 Schedule Parameters

Within each mining area, material was separated into bins based on weathering (oxide, transition, fresh) as plant recoveries vary by weathering type as well as material swell factors. Table 16.6.1 summarizes the pit and mill recoveries by material type by mining area, respective cut-off values, and assumed mining costs for MPSO. The mining costs were based on an aggregate of multiple contractor estimates received between August and October of 2025. The processing costs were based on the results of a desktop options assessment comparing heap leach and CIL processing methods using existing datasets and cost benchmarks from comparable operations. Reserve cutoffs were an output of the pit optimization work described in Section 15.5.

Table 16.6.1 MPSO Assumptions by Mining Area and Material Type

Mining Area	Material	Pit Recovery %	Mill Recovery %	Mill Cost $/t ore mined	Mining Cost $/t ore mined	Cutoff g/t	Swell Factor %
Boin	Oxide	96.0%	95.0%	9.78	1.95	0.142	20%
Boin	Transition	96.0%	95.0%	13.36	2.37	0.182	30%
Boin	Fresh	96.0%	89.0%	20.00	3.06	0.273	30%
Sewum	Oxide	96.0%	95.0%	9.78	2.08	0.136	20%
Sewum	Transition	96.0%	95.0%	13.36	2.40	0.176	30%
Sewum	Fresh	96.0%	90.0%	20.00	3.06	0.265	30%
Nyam	Oxide	96.0%	91.0%	9.78	2.19	0.180	20%
Nyam	Transition	96.0%	91.0%	13.36	2.54	0.222	30%
Nyam	Fresh	96.0%	90.0%	20.00	3.14	0.302	30%
Kwakyekrom	Oxide	96.0%	90.0%	9.78	2.00	0.178	20%
Kwakyekrom	Transition	96.0%	90.0%	13.36	2.26	0.220	30%
Kwakyekrom	Fresh	96.0%	80.0%	20.00	2.86	0.335	30%

Multiple mining areas were advanced in parallel to maintain operational flexibility, with constraints applied in MPSO to limit the number of simultaneously mined phases to four, reducing overall mining complexity and equipment inefficiencies. Primary schedule constraints included:

- Annual mill capacity limited to 5.5 M tonnes, with an allowance for up to 6 MT in years 5+ where ore blasting requirements were assumed to be below 50% of the total mill feed.
- Total annual material movement was limited to 35 M tonnes.
- Vertical advance rate was constrained to a maximum of three benches per quarter and nine benches per year, to maintain safe and sustainable mining progression.

- Ratio of fresh rock feed to the mill was monitored, allowing for a more manageable crusher design, smoothing of drill and blast needs, and improved control over waste dump composition.

- Fresh ore was restricted in year 1, allowing for a staged ramp up of the milling circuits, and delaying the requirements for the construction of the explosive manufacturing facility during the pre-production construction years. The small amount of fresh ore that was mined in year 1 was stockpiled for processing in year 2.

- Nyam and Kwakyekrom were constrained to start no earlier than year 2, allowing for a suitable amount of time to construct the necessary access roads.

A key objective during the initial years of operation was to achieve a minimum annual production target of 100,000 ounces of gold, while maintaining a consistent total tonnage mined to minimize fluctuations in fleet size.

16.6.2 Transition Material Definition

The separation of transition material into free dig versus drilling and blasting categories enabled a more detailed assessment of explosive requirements and provided a key input into the mine operation schedule, mine contractor cost estimation, and the process plant comminution design. This classification was based on a weathering profile derived from core logging and RC chip observations, where material is assigned a standardized weathering code ranging from W6 (residual soil) to W1 (fresh rock), as outlined in Table 16.6.2.

Table 16.6.2 Transition Material – Weathering Codes

Code	Description
W6	RESIDUAL SOIL Rock is completely changed to soil, and all original rock fabric has been destroyed.
W5	COMPLETELY WEATHERED - Rock is discoloured and changed to soil but original fabric is mainly preserved. The properties of the soil depend in part on the nature of the parent rock.
W4	HIGHLY WEATHERED - Rock is discoloured; defects may be open, clay rich and have and the original fabric of the rock near the defects may be altered. Alteration penetrates deeply inwards, but core stones are still present.
W3	MODERATELY WEATHERED - Rock is discoloured. Defects may be open and will have discoloured surfaces with alteration starting to penetrate inwards; the intact rock is noticeably weaker than the fresh rock.
W2	SLIGHTLY WEATHERED - Rock may be slightly discoloured, particularly adjacent to defects, which may be open and will have slightly discoloured surfaces, the intact rock is not noticeably weaker than the fresh rock.
W1	FRESH ROCK - Parent rock shows no discolouration, loss of strength or any other weathering effects.

An engineering assessment of the weathering profile and its comminution characteristics determined that free digging is generally achievable to the base of the moderately weathered (W3) zone. Accordingly, the contact between slightly weathered (W2) and moderately weathered (W3) material was interpreted as the boundary between free dig and drill-and-blast zones. This interpreted W2–W3 surface is used in mine scheduling and drill-and-blast planning to delineate material handling requirements within the transition zone.

16.6.3 Pre-production

The priority mining areas for the first year of operations was located within the Boin and Sewum deposits, specifically Boin Central and Sewum South Pit 4. While some access infrastructure has already been developed in these areas, additional work will be required to ensure they are fully suitable for mining activities.

Pre-production stripping will entail mining of approximately 5 million tonnes of oxide material. The primary objective of this phase was to establish access to the upper benches and clear suitably sized working areas to support full-scale mining operations commencing in year 1. Small volumes of ore encountered during the pre-strip would be stockpiled at the ROM pad for initial break-in of the process plant following completion of construction.

During pre-production, topsoil will be stripped and stockpiled separately to preserve material for future reclamation activities, in accordance with environmental management and closure requirements. This stripping will initially occur in the full extent of Boin Central Phase 1 and Sewum South pits, which are the primary mining locations in year 1. The remaining site preparation work occurs in stages, starting approximately 6 months in advance of a new working area becoming active.

Pre strip and site preparation locations for Boin and Sewum are outlined in Figure 16.6.1and Figure 16.6.2.

Figure 16.6.1 Boin Waste Pre-Strip and Site Preparation

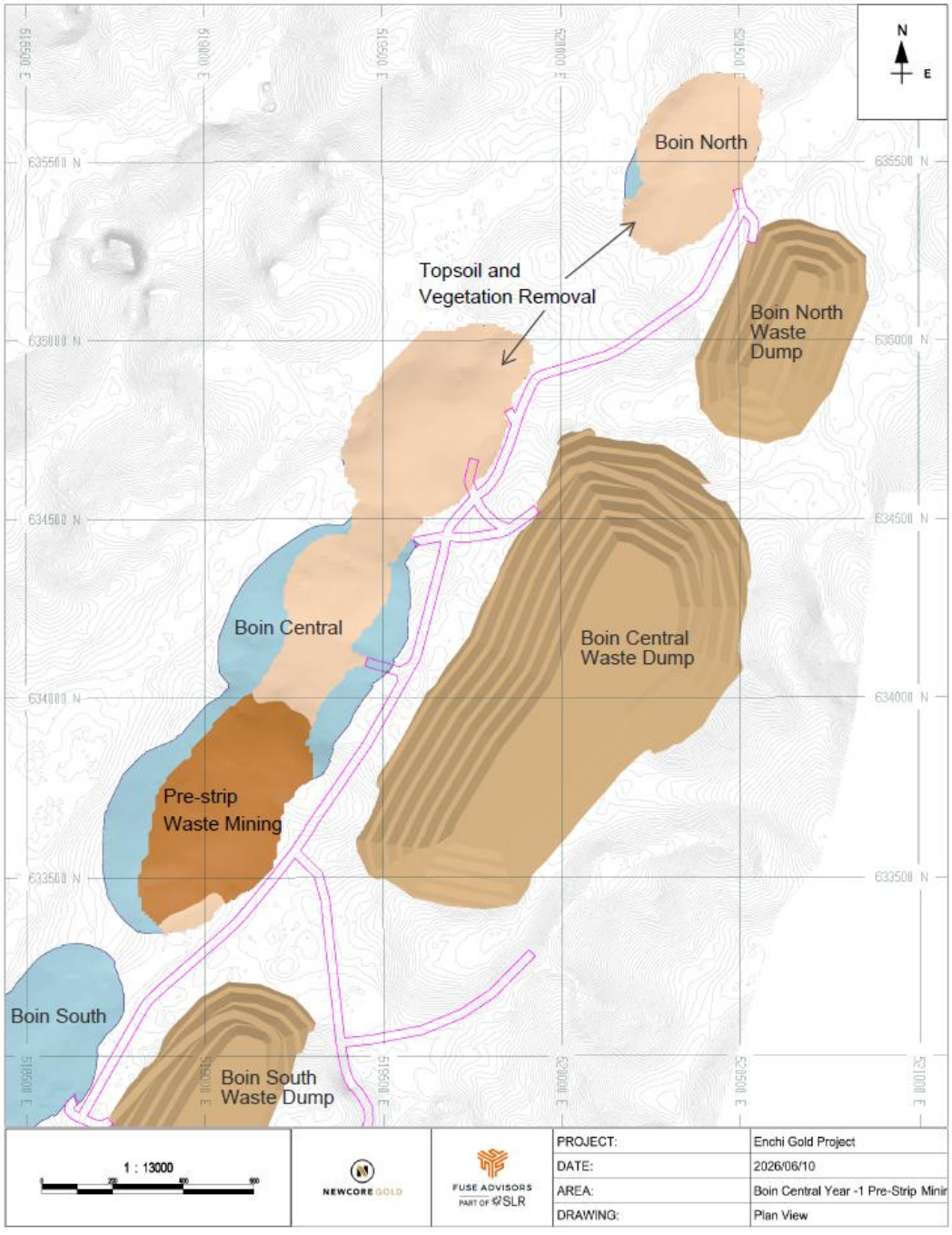
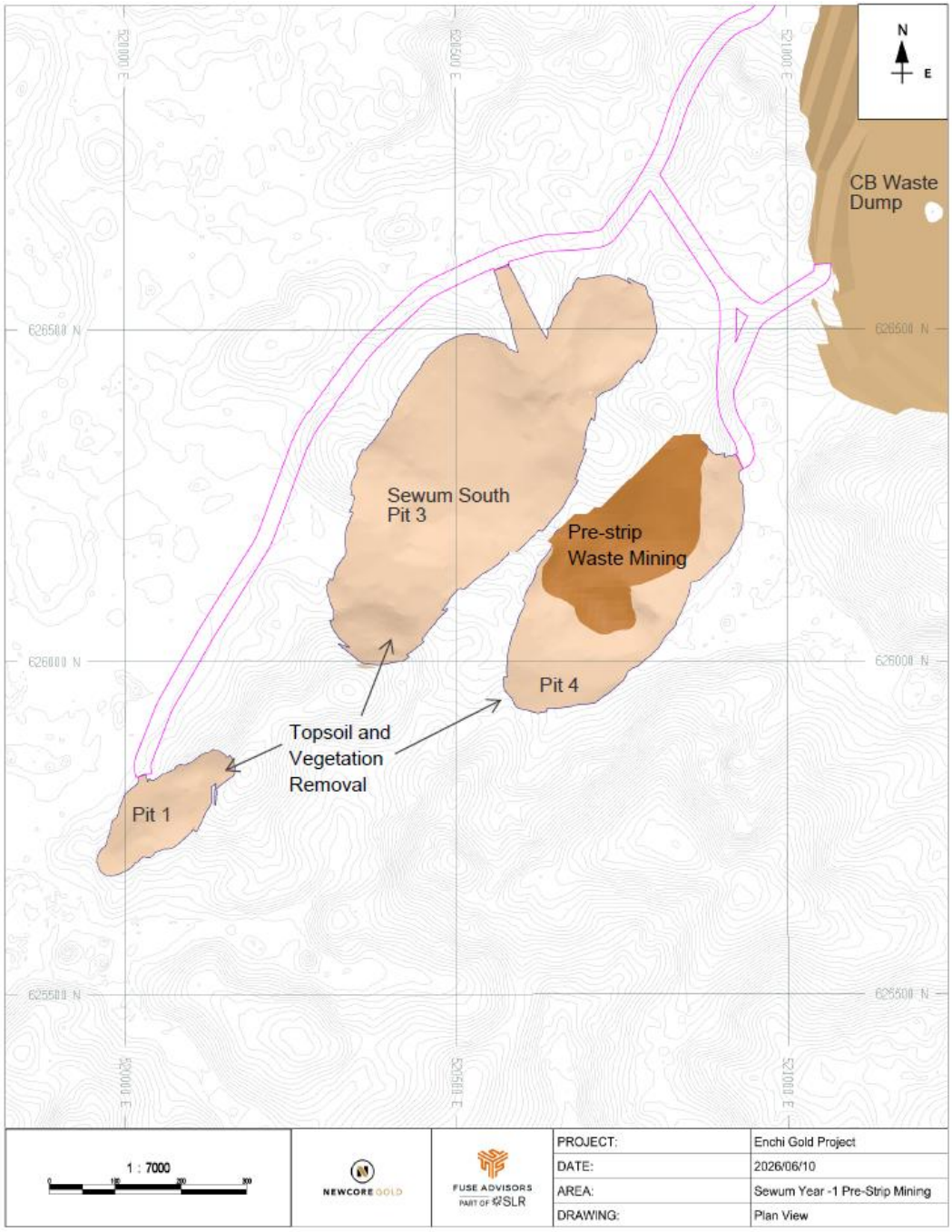


Figure 16.6.2 Sewum Waste Pre-Strip and Site Preparation



16.6.4 Mine Production

The Study forecasts 272 Mt to be mined over a 9.3 year mine life at an average strip ratio of 4.3 (waste: ore), in addition to 5 Mt of pre-stripping tonnage mined in the year prior to production. Processing of ore material through a CIL process plant is forecasted to be 51.3 Mt at an average gold grade of 0.64 g/t with an expected 1,055 Koz of contained gold. Based on the mine sequencing strategy detailed above, Table 16.6.3 summarizes the total Enchi Gold Project LOM annual material movement, plant throughput and associated contained gold ounces.

Table 16.6.3 LOM Mining Schedule

Material	Metric	Units	Totals	0Y	01Y	02Y	03Y	04Y	05Y	06Y	07Y	08Y	09Y	10Y
Oxide - Ore	Total	Kt	16,114	-	2,430	1,358	323	1,808	3,397	2,145	1,265	1,235	1,064	90
	Grade	g/t	0.49	-	0.80	0.67	0.42	0.42	0.40	0.43	0.41	0.40	0.36	0.32
	Contained	Koz	252	-	62	29	18	24	44	29	16	16	12	1
Transition - Ore	Total	Kt	17,267	-	3,082	2,622	1,468	1,709	400	3,320	1,494	1,490	1,652	30
	Grade	g/t	0.61	-	0.85	0.73	0.66	0.62	0.57	0.45	0.52	0.40	0.47	0.32
	Contained	Koz	336	-	85	62	31	34	7	48	25	19	25	0
Fresh - Ore	Total	Kt	17,908	-	-	1,551	2,706	1,933	2,017	535	2,741	2,800	2,784	841
	Grade	g/t	0.81	-	-	1.24	0.91	0.87	0.80	0.59	0.75	0.85	0.56	0.62
	Contained	Koz	467	-	-	62	79	54	52	10	66	77	50	17
Total - Ore	**Total**	**Kt**	**51,289**	**-**	**5,511**	**5,532**	**5,497**	**5,450**	**5,814**	**6,000**	**5,500**	**5,525**	**5,500**	**961**
	Grade	**g/t**	**0.64**	**-**	**0.83**	**0.86**	**0.73**	**0.64**	**0.55**	**0.45**	**0.61**	**0.63**	**0.49**	**0.58**
	Contained	**Koz**	**1,055**	**-**	**147**	**152**	**128**	**113**	**103**	**87**	**108**	**111**	**87**	**18**
Oxide - Waste	Total	Kt	84,232	5,000	14,881	8,256	8,553	12,167	17,051	9,179	2,959	3,729	2,458	-
Transition - Waste	Total	Kt	83,550	-	14,149	13,908	10,549	12,345	1,893	17,033	7,317	1,510	4,846	-
Fresh - Waste	Total	Kt	57,761	-	487	6,113	10,764	5,158	9,092	2,658	12,686	4,794	4,632	1,376
Total - Waste	**Total Waste**	**Kt**	**225,544**	**5,000**	**29,518**	**28,276**	**29,866**	**29,670**	**28,036**	**28,870**	**22,962**	**10,033**	**11,935**	**1,376**
Total	**Strip Ratio[1]**	**W:O**	**4.3**		**5.4**	**5.1**	**5.4**	**5.4**	**4.8**	**4.8**	**4.2**	**1.8**	**2.2**	**1.4**
	Total Mined	**Kt**	**276,833**	**5,000**	**35,029**	**33,808**	**35,363**	**35,120**	**33,850**	**34,870**	**28,462**	**15,558**	**17,435**	**2,337**
	Total – Free Dig	**Kt**	**166,468**	**5,000**	**32,193**	**21,073**	**18,245**	**23,325**	**22,628**	**24,218**	**6,720**	**9,730**	**3,226**	**110**
	Total - Blasted	**Kt**	**110,365**	**-**	**2,836**	**12,728**	**17,117**	**11,795**	**11,954**	**9,675**	**21,342**	**9,497**	**11,035**	**2,386**

1 Strip ratio calculation excludes tonnage mined during pre-production years

Total material movement throughout the mine life is targeted at 35 Mtpa for the first six years, after which it decreases in line with the declining strip ratio's in Year 7 onward. Figure 16.6.3 illustrates the total material movement profile.

Figure 16.6.3 LOM Total Material Movement (TMM)

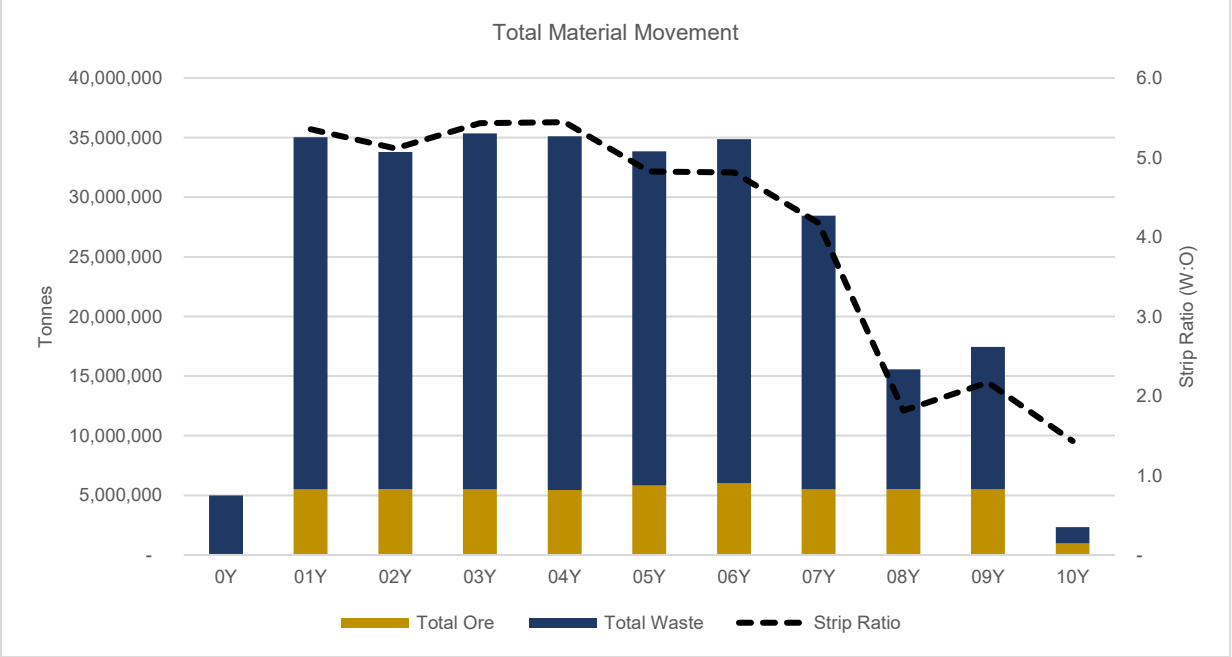
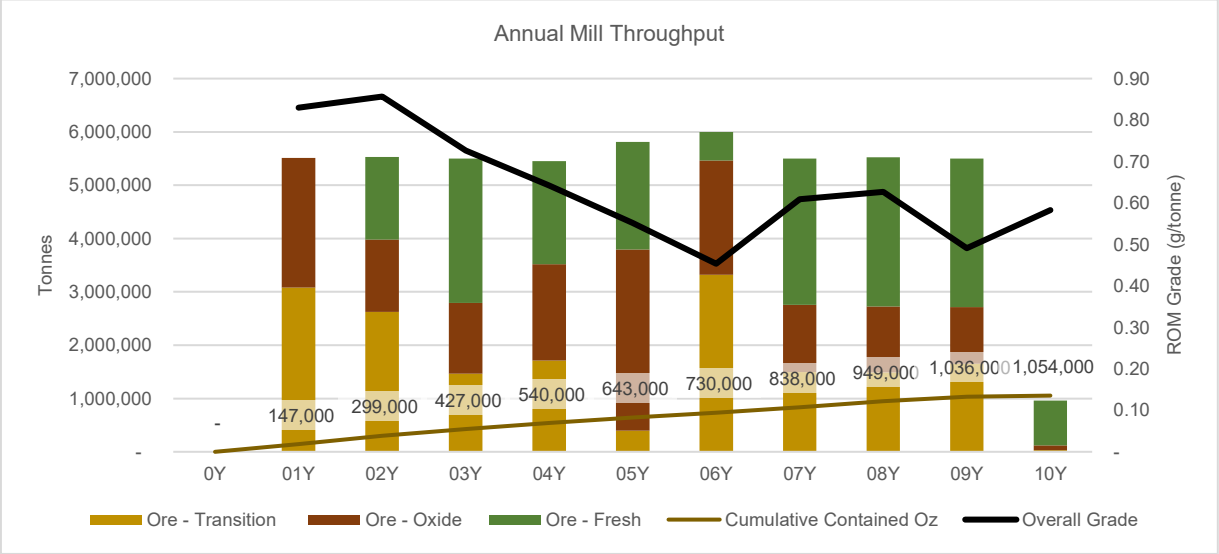


Figure 16.6.4 demonstrates the ore type breakdown of the annual mill throughput. The priority mining areas targeted in years 1 to 4 offer the highest average ROM grade, to support the highest Project NPV.

Figure 16.6.4 LOM Annual Mill Throughput



Managing the balance between free dig and drill-and-blast material throughout the mine schedule was a key metric to optimize mill sizing and manage operational complexity and dump composition. Figure 16.6.5 illustrates the annual split between material requiring blasting and free-dig material, together with the percentage of the total plant feed that is anticipated to require crushing. Excluding the partial year 10, which consists predominantly of pit-bottom mining in fresh rock, the maximum quantity of plant feed requiring crushing occurs in year 9, at 69% of plant feed (3.8 Mt). A breakdown of LOM totals by material type can be found in Table 16.6.4.

Figure 16.6.5 LOM Annual Free-Dig versus Drill-and-Blast Material

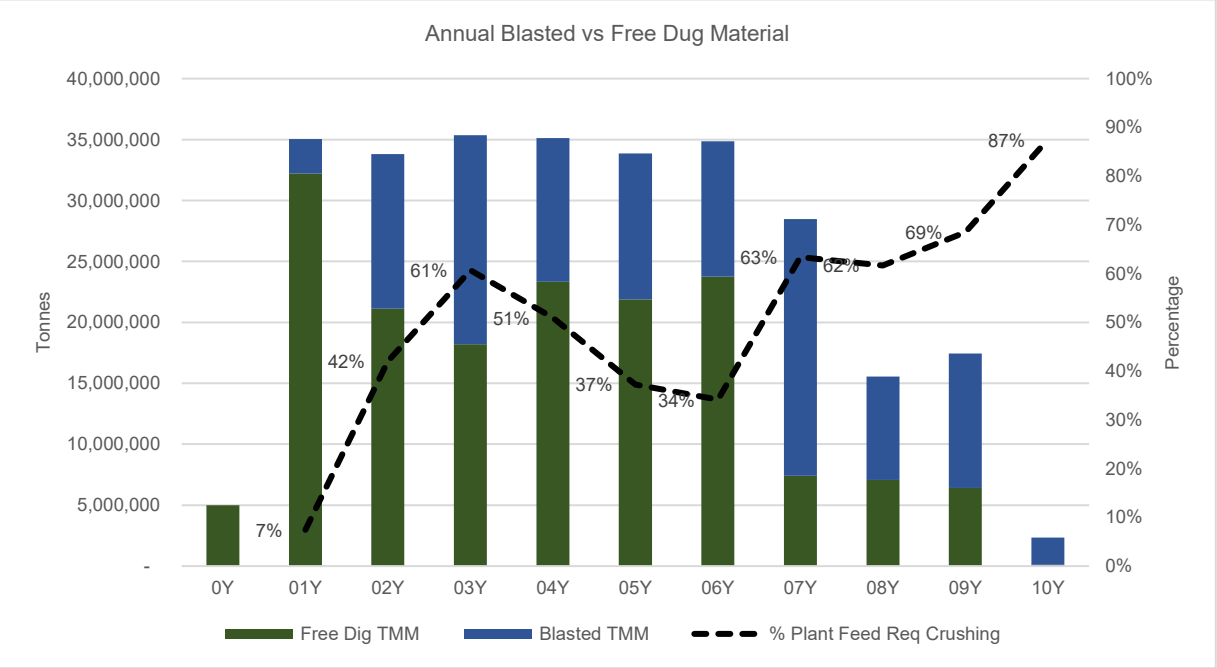


Table 16.6.4 LOM Totals by Material Type - Free-Dig versus Drill-Blast

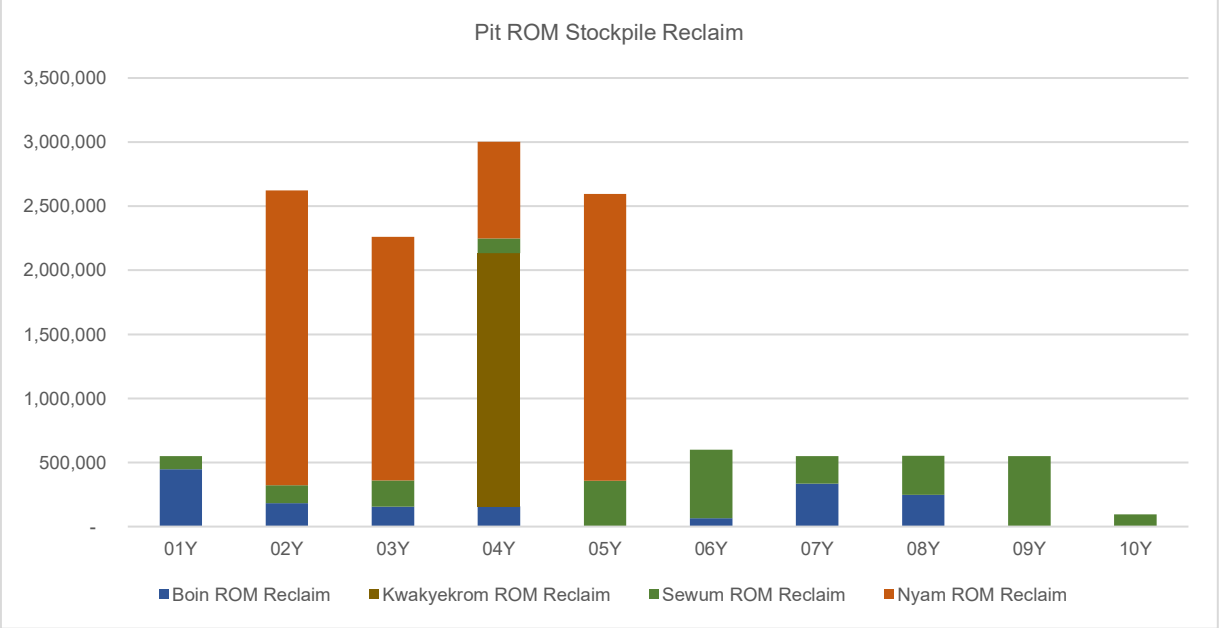
Material Type	Ore Tonnes Mt	Au Grade g/t	Contained Gold Koz	Waste Tonnes Mt	Total Tonnes Mt
Oxide	16	0.49	252	84	100
Transition – Free-Dig	11	0.61	208	56	66
Transition – Drill-and-Blast	7	0.60	128	28	35
Fresh	18	0.81	467	58	76
Total	**51**	**0.64**	**1,055**	**226**	**277**

Material grading below the cut-off grade will report to the WRSF. Material graded above the cut-off grade will be categorized ore. The primary strategy for movement of ore was direct haulage to the process plant site. Trucks were assumed to have the option to dump directly into the plant hopper or to a plant ROM stockpile for subsequent reclaim and feeding, depending on blending needs.

Due to the haul distance from the Nyam and Kwakyekrom deposits to the process plant site, and the requirement to utilize a public highway, a rehandle and reclaim strategy was incorporated for these deposits. All ore from Nyam and Kwakyekrom was assumed to be initially placed on ROM stockpiles located proximal to the respective pits. The material is subsequently reclaimed and rehandled using a dedicated, smaller truck fleet better suited for highway haulage to the process plant site.

The LOM schedule estimated that approximately 13 Mt of ore would be reclaimed over the life of mine. Most of this volume is attributable to the required fleet transition and rehandling associated with material sourced from Nyam and Kwakyekrom. As the project advances and blending requirements are further defined, this estimate is expected to be refined. Figure 16.6.6 illustrates the expected reclaim tonnes per year based on the LOM plan.

Figure 16.6.6 LOM ROM Stockpile Reclaim



Mining at Boin is assumed to commence in year 1 and is completed in year 8, after which a minor volume of stockpiled ore is processed. Mining at Sewum is assumed to ramp up in year 1 and year 2 and is completed in year 10. The Sewum pits exhibit the lowest proportion of material requiring blasting, making them well suited to supplement mill throughput during periods when higher volumes of competent, blast-dependent material are scheduled. Mining at Nyam commences in the second quarter of year 2 and is completed in year 5, allowing sufficient time for access development. Mining at Kwakyekrom is completed within year 4, leveraging equipment already mobilized in the area to support Nyam operations.

Table 16.6.5 summarizes the annual production from each mining area and Figure 16.6.7 illustrates the production profile.

Figure 16.6.7 LOM Production Profile by Mining Area

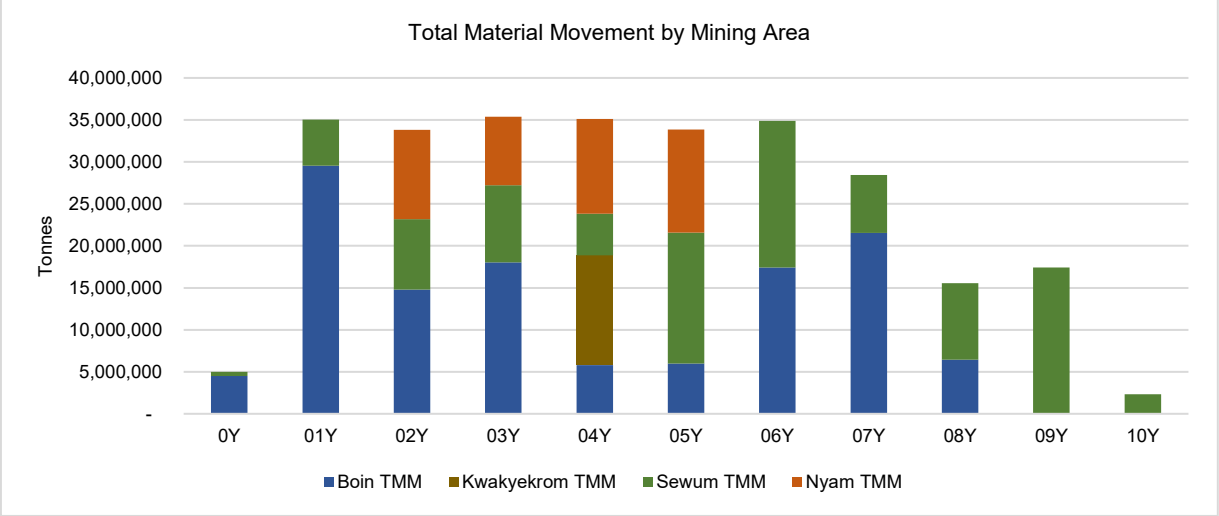
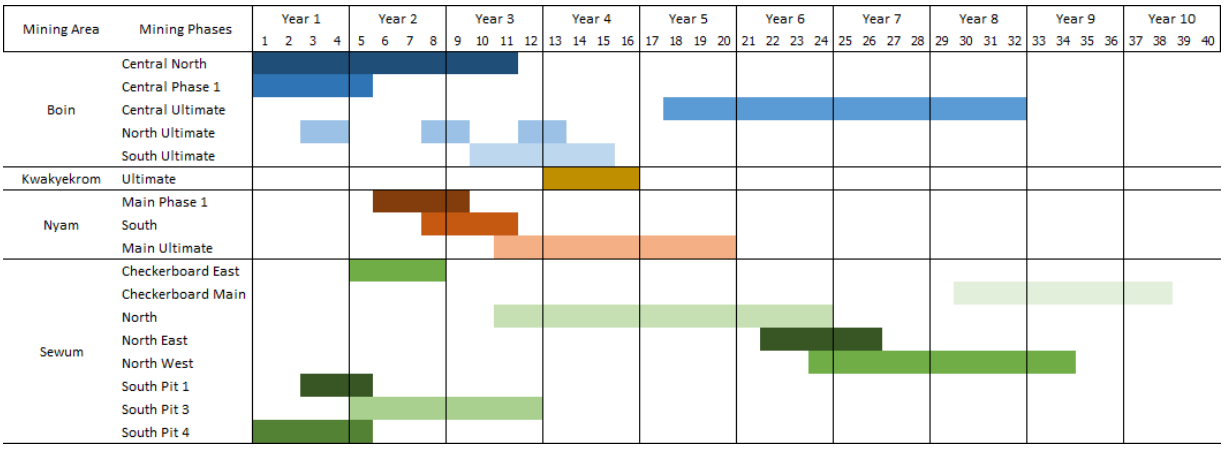


Table 16.6.5 LOM Mining Schedule by Mining Area

LOM Mining Schedule By Mining Area														
Mining Area	**Metric**	**Units**	**Totals**	**0Y**	**01Y**	**02Y**	**03Y**	**04Y**	**05Y**	**06Y**	**07Y**	**08Y**	**09Y**	**10Y**
Boin	Total Ore	Kt	15,906	-	4,477	1,832	1,563	1,572	7	652	3,339	2,464	-	-
Boin	Overall Grade	g/t	0.84	-	0.84	1.14	0.80	0.80	0.47	0.58	0.71	0.90	-	-
Boin	All Contained Oz	Koz	429	-	121	67	40	40	0	12	76	72	-	-
Boin	Total Waste	Kt	108,245	4,500	25,069	12,977	16,488	4,237	5,995	16,769	18,214	3,996	-	-
Boin	Total Material	Kt	124,151	4,500	29,546	14,809	18,051	5,810	6,002	17,421	21,553	6,461	-	-
Sewum	Total Ore	Kt	26,213	-	1,035	1,398	2,034	1,147	3,568	5,348	2,161	3,060	5,500	961
Sewum	Overall Grade	g/t	0.48	-	0.77	0.63	0.62	0.38	0.40	0.44	0.46	0.40	0.49	0.58
Sewum	All Contained Oz	Koz	406	-	26	28	40	14	46	75	32	40	87	18
Sewum	Total Waste	Kt	71,049	500	4,449	6,980	7,128	3,781	12,013	12,101	4,748	6,037	11,935	1,376
Sewum	Total Material	Kt	97,262	500	5,484	8,378	9,163	4,928	15,582	17,449	6,909	9,097	17,435	2,337
Nyam	Total Ore	Kt	7,193	-	-	2,301	1,901	753	2,238	-	-	-	-	-
Nyam	Overall Grade	g/t	0.76	-	-	0.77	0.78	0.56	0.79	-	-	-	-	-
Nyam	All Contained Oz	Koz	175	-	-	57	48	13	57	-	-	-	-	-
Nyam	Total Waste	Kt	35,153	-	-	8,320	6,249	10,556	10,028	-	-	-	-	-
Nyam	Total Material	Kt	42,346	-	-	10,621	8,150	11,309	12,267	-	-	-	-	-
Kwakyekrom	Total Ore	Kt	1,977	-	-	-	-	1,977	-	-	-	-	-	-
Kwakyekrom	Overall Grade	g/t	0.70	-	-	-	-	0.70	-	-	-	-	-	-
Kwakyekrom	All Contained Oz	Koz	44	-	-	-	-	44	-	-	-	-	-	-
Kwakyekrom	Total Waste	Kt	11,096	-	-	-	-	11,096	-	-	-	-	-	-
Kwakyekrom	Total Material	Kt	13,074	-	-	-	-	13,074	-	-	-	-	-	-
All Mining Areas	**Total Material**	**Kt**	**276,833**	**5,000**	**35,029**	**33,808**	**35,363**	**35,120**	**33,850**	**34,870**	**28,462**	**15,558**	**17,435**	**2,337**

The LOM plan was developed using quarterly scheduling periods to support the generation of an executable mining sequence and to ensure mill feed requirements met an appropriate level of granularity. Figure 16.6.8 illustrates the quarterly mining schedule by mining area and associated discrete phases. A summary of key mining periods is described below.

Figure 16.6.8 Quarterly LOM Gantt



Year 1:

Mining activities were assumed to primarily focus on Boin Central North and Phase 1. Oxide material from the Boin North area is also mined to supplement plant feed requirements and minimize the need for blasting and the release of fresh ore. Similarly, at Sewum, mining was assumed to concentrate on Sewum South Pit 4 as the primary production area, with Sewum South Pit 1 providing supplementary material to reduce reliance on blasted rock and support consistent mill feed.

Year 2:

Mining was assumed to continue at Boin Central North, with mining of Boin Central Phase 1 completed early in Q2 of year 2. The mining fleet was assumed to transition to Nyam in Q2 of year 2 with mining commencing at the Nyam Main Pit. Mining of Sewum Pit 1 and Pit 4 is assumed to be completed early in Q1 of year 2, transitioning to Sewum Checkerboard East Pit and Sewum South Pit 3.

Year 3:

Mining in Boin Central North is assumed to ramp down significantly, with completion in Q3. As mining ramps down in Boin Central North, the fleet is transitioned to Boin South. Boin North is assumed to continue to be mined intermittently to support mill feed. Mining rates remain consistent at Nyam. Mining at Nyam Main and Nyam South is completed, with mining commenced at the Nyam Ultimate Pit. Sewum South Pit 3 mining is ongoing throughout year 3, while Sewum North commences in Q3.

Year 5

Equipment transitions from the Nyam Ultimate pit to the Boin Ultimate pit throughout year 5. Mining in Sewum North is ongoing.

Year 6

Mining of the Boin Ultimate pit is ongoing. Mining of Sewum North is completed in year 6, and the fleet is utilized for the Sewum East and West pits.

Year 10

The final year of mining is focused on Sewum Checkerboard Main. Mining is assumed to be completed by the second quarter of year 10.

16.7 Mining Operations

The Enchi Gold Project is planned as a contractor-based mining operation. While a mining contractor has not yet been selected at this stage, equipment selections and fleet estimates have been developed based on vendor-supplied information specific to the PFS LOM plan.

For the purposes of this report, vendor estimates were used to guide assumptions regarding equipment types and fleet sizing. The general mining approach is to use a smaller fleet during initial stripping activities, and for the development of access to establish wider, more productive mining benches. Once sufficient working widths have been developed, a larger fleet, including 91-tonne haul trucks, will be deployed to improve productivity and reduce unit mining costs.

Pit bottom conditions are typically defined as the lowest 4 to 6 benches, where reduced bench widths and limited operational space constrain the use of larger equipment. In these areas, a smaller mining fleet is expected to be reintroduced to maintain safe and efficient operations.

Mining operations are planned to operate year-round on a continuous basis (365 days per year), with allowances made for unplanned delays due to weather and other potential operational challenges. The operations workforce will be organized into three rotating panels working 12-hour shifts on a cycle of three consecutive night shifts, followed by three consecutive day shifts, and then three days off. This rotation supports continuous production, maintenance, and essential services coverage, including during public holidays.

Administrative and support personnel will operate on a separate rotation schedule consisting of five days on and two days off. This structure ensures consistent operational oversight and support while aligning with standard work practices for non-operational staff.

16.7.1 Load and Haul

The mining fleets will be responsible for hauling ore from the pits to the process plant where the material will either be directly dumped into a hopper bin or placed on a ROM stockpile adjacent to the process plant. Equipment selections and fleet estimates have been developed based on vendor-supplied information aligned with the PFS LOM plan. The fleet size represents the maximum total fleet requirements across all four deposits, depending on which areas are scheduled to be mined during which period. These assumptions will be refined as the Project advances and a mining contractor is selected. Fleet requirements are categorized into three primary functions:

- Small-scale mining fleet - utilized to establish the mining areas and perform selective or narrow pit bottom mining where access and bench widths are limited.
- Primary mining fleet - utilized for most of the pit life once wide, productive benches are developed. At Boin and Sewum, this fleet will also be responsible for hauling ore to the process plant.
- Ore haulage fleet - dedicated to transporting ore from the Nyam and Kwakyekrom ROM stockpiles to the process plant, given the longer haul distances and highway travel requirements.

Small Scale Mining Fleet

The initial mining fleet required to support access and bench development was assumed to consist of 45-ton articulated trucks and Komatsu PC2000 excavators and Komatsu 336D excavators. This small-scale fleet will also be utilized for pit bottom mining as bench sizes become unproductive for a larger scale fleet. The Komatsu PC2000 fleet is suitable to load the articulated trucks or the rigid frame HD785 and therefore is included on both fleet lists. The equipment specifications and maximum expected fleet requirements are included in Table 16.7.1. The small-scale fleet will also be used during initial pre-strip to prepare the first mining bench for the larger equipment.

Table 16.7.1 Selective Mining Fleet

Equipment	Size	Maximum Fleet Size
Komatsu PC2000 Excavator	12 m^3	2
Komatsu 336D Excavator	2 m^3	3
Volvo A45 Haul Truck	41 tonne	6

Primary Mining Fleet

It was assumed that once productive benches are established, the primary mining fleet would consist of Komatsu PC3000 excavator and Komatsu PC2000 excavator units, supported by Komatsu HD785 haul trucks or equivalent equipment. The Komatsu PC2000 fleet is suitable to load the articulated trucks or the rigid frame HD785, therefore is included on both fleet lists. The equipment specifications and maximum expected fleet requirements are included in Table 16.7.2.

Table 16.7.2 Primary Mining Fleet

Equipment	Size	Maximum Fleet Size
Komatsu PC3000 Excavator	16 m^3	2
Komatsu PC2000 Excavator	12 m^3	3
Komatsu HD785 Haul Truck	91 tonne	40

Secondary Ore Haul Fleet

Given the highway travel requirements and longer-haul route from Nyam (20 km) and Kwakyekrom (16 km) to the process plant, it was determined that a smaller, dedicated, ore haul fleet would be implemented to haul ore from the ROM stockpiles adjacent to the Nyam and Kwakyekrom pits to the process plant. The equipment specifications and maximum expected fleet requirements are included in Table 16.7.3.

Table 16.7.3 Secondary Ore Haulage Fleet

Equipment	Size	Maximum Fleet Size
CAT 988 Front End Loader	2 m^3	2
Tipper (8X4) Dump Truck	25 tonne	26

16.7.2 Drill and Blast

The oxide material and upper portion of the transition material is expected to be free-dug and hence does not require drill and blasting. Through the lower portion of the transition zone and through the entirety of the fresh zone, material is expected to become more competent and require conventional drill and blast to fragment the material for efficient loading and hauling.

The drill and blasting operation will be completed by a selected contractor. For the purposes of the PFS, a conceptual drill and blast design was supplied to vendors for fleet sizing and pricing. The proposed drill and blast parameters were separated into two design parameters based on the bench heights in the specific mining areas. For Boin and Sewum, 10 m benches were assumed, and the associated parameters are listed in Table 16.7.4. For Nyam and Kwakyekrom, 12 m benches were assumed, and the associated parameters are listed in Table 16.7.5.

Table 16.7.4 10 m Bench Drilling Design Criteria

Parameter	Unit	Value
Bench Height	m	10.00
Burden	m	3.90
Spacing	m	4.50
Production Hole Size	mm	140
Pre-split Hole Size	mm	102
Subdrill	m	1.00
Redrill	%	5.00
Material Density	t/m^3	2.34-2.37
Collar	m	3.00

Table 16.7.5 12 m Bench Drilling Design Criteria

Parameter	Unit	Value
Bench Height	m	12.00
Burden	m	4.00
Spacing	m	4.60
Production Hole Size	mm	140
Pre-split Hole Size	mm	102
Subdrill	m	1.20
Redrill	%	5.00
Material Density	t/m^3	2.36-2.48
Collar	m	3.20

Blast hole drilling is expected to be carried out using Sandvik DP1500 drill rigs, or equivalent, capable of drilling 102 mm to 152 mm diameter holes in both vertical and inclined orientations. Supporting equipment will include an explosives delivery truck and purpose-built light vehicles for transporting personnel and accessories. Stemming material will be placed in charged holes to ensure proper confinement.

The pit configuration, bench height, and anticipated waste material characteristics support the use of drill rigs capable of drilling 140mm diameter blastholes. The burden, spacing and sub-drill design are dependent on the varying material types of the deposit and may change as rock characteristics are better understood.

An emulsion-based explosive with water-resistant properties and a high velocity of detonation is assumed to be utilized to optimize fragmentation. Pre-split blasting will be conducted using packaged emulsion products loaded into planned 102 mm diameter blastholes. The charges will be decoupled and detonated to promote controlled shearing between holes, resulting in improved wall definition and the formation of a more stable and competent final pit wall.

Explosives supply and management will be the responsibility of the mining contractor and included within their scope of costs. An existing on-site explosives magazine complies with the Ghanaian Minerals and Mining (Explosives) Regulations (LI 2177), including appropriate design, security, and permitting. Explosives and accessories will be delivered by the contractor in accordance with production requirements and regulatory storage limits.

On-site emulsion storage capacity will be established to meet operational and compliance needs, with a permanent emulsion plant scheduled for installation in Q1 of year 2. During year 1, blasting requirements are expected to be limited and will be managed using packaged explosives for localized applications.

16.7.3 Grade Control

Grade control will be achieved through systematic drilling, sampling, and geological interpretation, as mineralized zones are not expected to be visually distinguishable in the field. The following steps will support maximizing recovery, minimizing dilution, informing plant throughput and verifying model assumptions:

- Grade control drilling will be completed using dedicated reverse-circulation (RC) drill rigs. A nominal 5 m × 10 m pattern will be used to define ore–waste boundaries. Holes will typically be drilled to 10 m or 12 m depth (two benches), depending on bench configuration. In areas of competent ground, deeper holes (three benches) may be drilled to support medium-term planning and improve geological confidence ahead of mining. Blast holes will be used to supplement drilling requirements or infill areas requiring additional definition. As mining progresses, the drill density may be refined to better fit operational needs.

- Samples will be collected at 1.5 m intervals. Field technicians will manage sample collection, handling, and preparation in accordance with site procedures and QA/QC requirements. Samples will be processed within the on-site gold analysis laboratory located within the process plant facilities.

- A short-term grade control block model will be updated on a regular basis using the latest drilling results and geological interpretation. Economic mineralization will be defined using applicable cut-off grades that reflect processing recoveries, operating costs, and gold price assumptions.

- Ore and waste dig polygons will be generated for each mining bench based on geological interpretation and model outputs. Polygons will balance selectivity with practical mining constraints (e.g., minimum mining width and equipment performance). Final polygons will be issued to survey for pickup and field marking, and supporting plans will be communicated to operations to facilitate execution.

16.7.4 Ancillary Equipment

The ancillary mining equipment will include dozers, motor graders, fuel bowsers, water trucks, and a vibratory compactor for road construction. The function of this equipment will be to support the primary mining equipment by maintaining the pit floor and haul roads, provide clean-up around the excavators to prevent excessive tire damage, mobile refueling services, and to water-down road surfaces to supress dust.

Given the distances between mining deposits, a 300 tonne Sleipner dolly has also been included in the equipment selection for supporting transport of heavy machinery between pits and different mining areas. Overall costing of ancillary machinery is included in the mining rate quotes provided by the mining contractor groups.

16.7.5 In-Pit Water Management

Pit water management will be implemented using a combination of preventative and active control measures. As a first line of defense, ex-pit diversion channels will be constructed to minimize surface water inflows into active mining areas. Surface runoff and groundwater reporting to the pit will be collected via roadside ditches and culverts and directed to in-pit sumps. From these sumps, water will be pumped using diesel-powered pumps through HDPE pipelines to ex-pit sedimentation ponds for appropriate management.

Sump systems will be equipped with float-activated controls to enable automatic pump operation in response to water levels. In areas where pit depths result in pumping lifts exceeding 100 m, booster pumps or high-lift pumping systems will be installed to ensure efficient water removal. In extreme, short duration, weather events water may be directed to the lowest operating bench, and additional temporary pumping capacity will be installed to quickly and efficiently recover the working area with minimal disruption to the mining cycle.

The Enchi Gold Project is planned as a contractor-based mining operation; therefore, day-to-day water management activities will be the responsibility of the selected mining contractor. For the purposes of scoping and cost estimation, vendors were provided with both average wet season and peak flow rates for each mining area. These flow estimates, supplied by DRA, are based on a combination of outputs from prior studies, historical precipitation data, and data from other shear-hosted deposits in Southwest Ghana. This information was then combined with updated parameters for the Enchi pit sizes, including strike length, widths, and final pit depths.

17.0 RECOVERY METHODS

17.1 Overall Process Design

The plant design for the Enchi Gold Project Pre-Feasibility Study (PFS) was based on a 9.3 year mine plan and production schedule supplied from Newcore at the start of the study. The design basis had a focus on year 1 and year 4 as the representative extreme cases. Year 1 predominantly involves the treatment of oxide or free-dig transition ('Dig') ore (89%), while year 4 primarily processes fresh or drilled and blasted transition ('Blast') ore (84%). In both cases, the milling circuit was designed to achieve a target grind size of P_{80} 53 µm.

A revised mine plan was subsequently provided by Newcore toward the end of the study, indicating that Year 1 will comprise 93% 'Dig' ore. Additional metallurgical testwork conducted on oxide ore also demonstrated that a coarser grind size (P_{80} 75 µm) could achieve similar gold recovery to that obtained at P_{80} 53 µm. Based on these findings, a coarser grinding scenario was considered for the existing milling circuit, whereby a product size of P_{80} 75 µm would be targeted during the first year of operation.

The study employs a robust flowsheet for gold recovery, consisting of the following:

- A mineral sizing operation for 'Dig' ore.
- A primary crushing operation for 'Blast' ore.
- A Semi-Autogenous Grinding (SAG) mill operation, followed by a ball mill operating in closed circuit with the classification cyclones.
- Gravity concentration and Intensive Cyanidation (IC).
- Pre-leach thickening.
- Carbon-In-Leach (CIL) circuit.
- Carbon elution and electrowinning.
- A secure goldroom for smelting and producing final doré bars.
- Tails handling and decant water return.

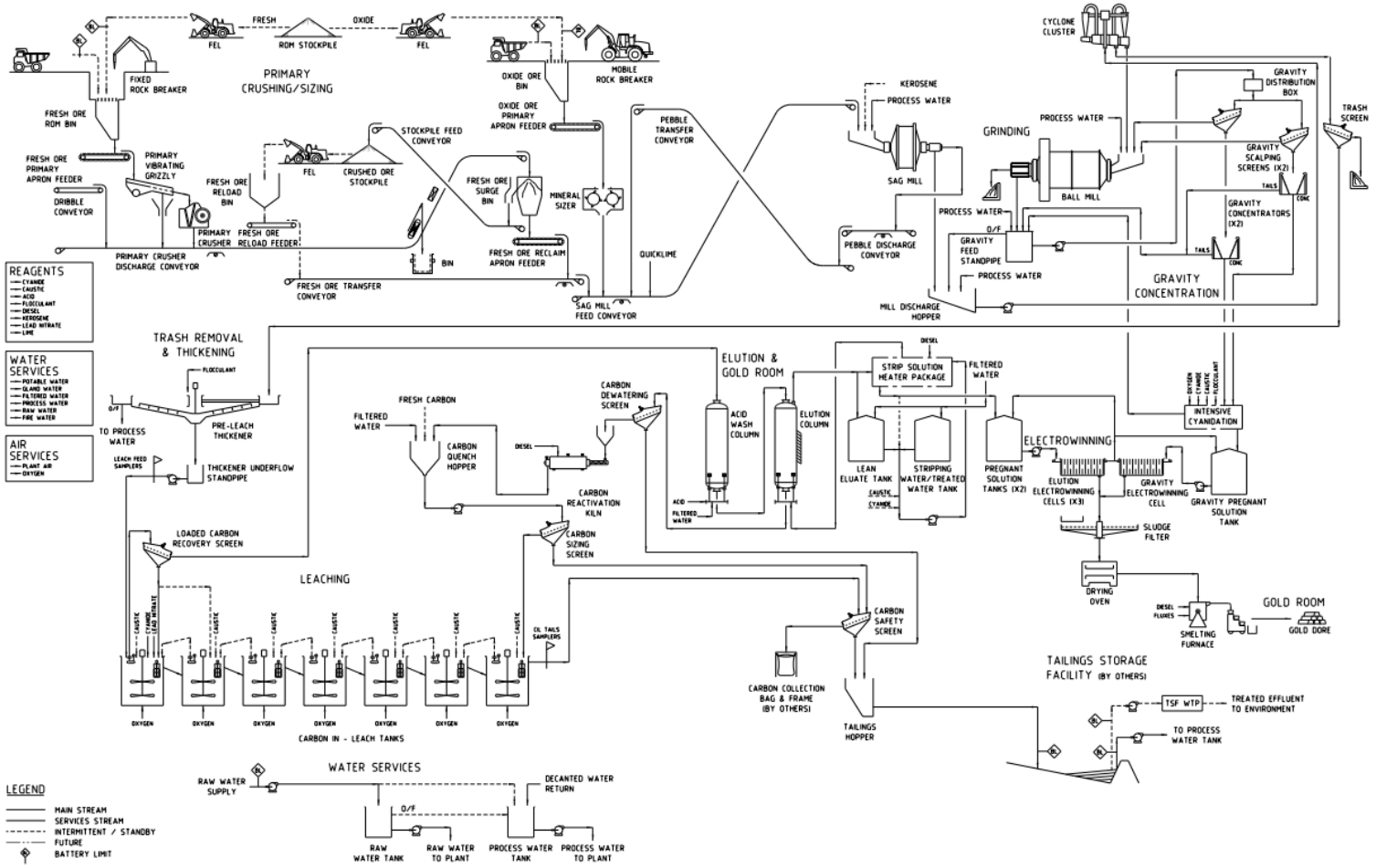
The key criteria for equipment selection in this study were suitability for duty, reliability, and provision for an optimized, low-cost design. The plant layout has been designed to also provide easy access to all equipment for operation and maintenance, while also maintaining a configuration that facilitates construction across multiple areas concurrently.

The key project design criteria for the plant were:

- Nominal plant feed rate at 5.5 Mtpa.
- A mineral sizing circuit operating at 91.3% utilization, for 'Dig' ore.
- A primary crushing circuit operating at 80% utilization, for 'Blast' ore.
- A milling circuit operating at 91.3% utilization, for both 'Dig' or 'Blast' ore.

An overall process flow diagram depicting the unit operations incorporated in the selected process flowsheet is presented in Figure 17.1.1.

Figure 17.1.1 Overall Process Flow Diagram



17.1.2 Key Process Design Criteria

The key process design criteria are listed in Table 17.1.1.

Table 17.1.1 Summary of Key Process Design Criteria

Process Design Parameters	Units	'Dig' Ore P_{80} 75 µm	'Dig' Ore P_{80} 53 µm	'Blast' Ore P_{80} 53 µm	Source
Plant Throughput	dry tpa		5,500,000		Client
Head Grade - Design	g/t Au		0.93		Agreed
	g/t Ag		1.00		Agreed
Sizing Plant Availability	%		91.3		Cons-OMC
Crushing Plant Availability	%		80.0		Cons-OMC
Milling Plant Availability	%		91.3		Cons-OMC
Crushing Work Index (CWi) - Design	kWh/t	11.9	11.9	17.1	Cons-OMC
Bond Ball Mill Work Index (BWi) - Design	kWh/t	11.0	11.0	16.7	Cons-OMC
Abrasion Index (Ai) - Design	g	N/A	N/A	0.316	Cons-OMC
A x b - Design	-	150	150	47.2	Cons-OMC
Run-Of-Mine (ROM) Material Size, F_{100}	mm	100	100	700	Agreed
Sized / Crushed Product Size, P_{80}	mm	45	45	152	Cons-OMC
Grinding Circuit Product Size, P_{80}	µm	75	53	53	Testwork/Agreed
Gravity Gold Recovery – Design	%		25.4		Testwork/Agreed
Overall (Grav+CIL) Gold Recovery – Design	%		93.6[1]		Calculated
Pre-Leach Thickener Solids Loading	$t/m^2 \cdot h$	0.21	0.21	0.48	Testwork/Agreed
Leach Circuit Residence Time	h	24	24	48	Testwork/Agreed
Leach Slurry (Solids) Density	% (w/w)	45	45	50	Testwork/Agreed
Leach Circuit Configuration	-		7 CIL Tanks		Agreed
Elution Circuit Type	-		Split-AARL		Agreed
Elution Circuit Column Size	t, carbon		18		Calculated/Agreed
Frequency of Elution	strips/wk		6		Lycopodium
Sodium Cyanide Addition[2]	kg/t feed	0.83	1.00	1.49	Testwork/Agreed
Lime Addition	kg/t feed	2.61	2.54	1.41	Testwork

Notes:
1 Calculated recovery after accounting for solution and carbon losses.
2 50% discount has been applied on the testwork cyanide consumption per agreement with Client that laboratory was overdosing cyanide.

17.2 Process and Plant Description

The proposed process design includes the following unit operations:

- A mineral sizer to provide sized product at P_{80} of 45 mm for 'Dig' ore.
- A single stage primary jaw crusher to produce a crushed product at a P_{80} of 152 mm for 'Blast' ore.
- A dead stockpile to store the crushed materials, with a live capacity of ~16,500 tonnes (24 hours).
- A milling circuit consisting of a SAG mill and a ball mill (SAB), operating in closed circuit with classification cyclones to produce a target grind size P_{80} of 53 µm. Note that in year 1 where ore is mainly oxide, the milling circuit can be operated to produce a coarser grind size of P_{80} 75 µm to save on operating costs.
- A trash screen to remove foreign materials (woodchips and plastic trash) contained in the cyclone overflow stream and prevent them from reporting to the downstream processes.
- A pre-leach thickener to increase the cyclone overflow pulp density from approximately 35% to 45% w/w solids for 'Dig' ore, and 50% w/w solids for 'Blast' ore.
- A leach circuit comprising seven CIL tanks to leach gold from the gold bearing ore.
- An elution circuit based on the split Anglo-American-Research-Laboratories (AARL) method for gold recovery from the loaded carbon. In this configuration, the elution and electrowinning steps are decoupled. The circuit consists of an acid wash column to remove inorganic foulants and an elution column for carbon stripping. The elution column, together with the stripping water, lean eluate tanks and the strip solution heating system forms a closed-loop for circulating the heated strip solution to a pregnant solution tank. The solution in the duty pregnant solution tank will then be pumped in a closed-loop with the electrowinning cells, in which gold is plated out as sludge, and the resulting barren solution is returned to the CIL circuit.
- A carbon regeneration kiln to remove organic foulants from the carbon and regenerate the carbon adsorption sites with the application of heat.
- An on-site secured goldroom facility to produce final doré bars from the pregnant solution, comprising of electrowinning cells, a sludge-pan filter, a drying oven, and a smelting furnace.
- A tails handling system to dispose the plant tailings slurry by gravity to the Tailings Storage Facility (TSF).

- Different reagents, such as quicklime (CaO), flocculant, sodium cyanide (NaCN), kerosene, activated carbon, sodium hydroxide (NaOH), hydrochloric acid (HCl), lead nitrate ($Pb(NO_3)_2$), anti-scalant, sulphamic acid, smelting fluxes (borax, soda ash, silica sand and sodium nitrate), and other consumables such as crucible, Leach-Aid, stainless steel stocking, grinding media, and diesel) will be used in the process plant, at various circuits.

17.2.1 Feed Preparation

'Dig' Ore

A mineral sizer will be deployed to process softer oxide and diggable transition ('Dig') ore and will reduce the ROM ore to a P_{80} of 45 mm.

ROM ore will be trucked from the pits to the ROM pad and will either be directly tipped into the oxide ore bin or reclaimed by Front-End Loader (FEL) into the bin. Ore drawn from the bin will feed the sizer. Product from the sizer will be transferred to the SAG mill feed conveyor.

A mobile rock breaker will be utilized to break any oversized material or rock lumps that cannot pass through the bin or the sizer opening.

Extraction of the dust from the discharge points below the sizer will be done using a dedicated dust collector (or extraction) system on the SAG mill feed conveyor.

'Blast' Ore

A primary jaw crusher will be deployed to process fresh and blasted transition ore and will reduce the ROM ore to a P_{80} of 152 mm.

ROM ore will be trucked from the pits to the ROM pad, and either be directly tipped or reclaimed by FEL to the fresh ore ROM bin. An apron feeder will deliver materials to a primary vibrating grizzly screen. The grizzly oversize materials will be crushed using the primary crusher. The grizzly undersize materials along with the apron feeder fines and the crushed materials will be delivered onto the primary crusher discharge conveyor.

A magnet will be installed above the primary crusher discharge conveyor for removal of tramp metal. A metal detector will be installed after the magnet to stop the conveyor belt if metallic materials get detected.

A fixed rock breaker will be utilized to break any oversized material or rock lumps that cannot pass through the bin or the crusher opening.

Raw water will be used for dust suppression, and the water sprays will be automatically activated during material tipping. Extraction of the dust from the discharge points below the crusher will be from a dedicated dust collector (or extraction) system on the primary crusher discharge conveyor.

Primary crushed ore will be withdrawn from the surge bin by another apron feeder, which will discharge onto a fresh ore transfer conveyor feeding the SAG mill feed conveyor.

Excess crushed ore will overflow from the surge bin and be conveyed by a stockpile feed conveyor onto a dead stockpile with a live capacity of 16,500 tonnes and will be reclaimed by FEL into a reload bin and then be transferred to the SAG mill feed conveyor via a reload feeder and a fresh ore transfer conveyor, when required.

Dust suppression (using raw water) will be automatically activated on the surge bin when the primary crusher discharge conveyor is operational and, on the stockpile, when the stockpile feed conveyor is operational. Extraction of the dust from the discharge points below the reload feeder will be done using a dedicated dust collector (or extraction) system on fresh ore transfer conveyor.

Thus, based on the original mine plan, a blend of both 'Dig' (89% in year 1) and 'Blast' (84% in year 4) materials will be conveyed to the milling circuit through the SAG mill feed conveyor. Based on the revised mine plan, mainly 'Dig' (93% in year 1) materials will be conveyed to the milling circuit through the SAG mill feed conveyor.

The feed preparation step has been designed to allow separate processing of 'Dig' and 'Blast' ore prior to feeding the milling circuit.

Quicklime for leaching area (or the CIL circuit) pH control will be added onto the SAG mill feed conveyor, via a feeder from a lime silo.

17.2.2 Milling and Classification

The milling circuit will be an SAB circuit (SAG and Ball mill), operating in closed circuit with classification cyclones.

The SAG mill feed conveyor discharge will report to the SAG mill. The SAG mill discharge will be screened using trommel screen for pebbles removal. The screen oversize material (pebbles) will be conveyed by the pebble discharge and transfer conveyors, and it will be recycled back to the SAG mill via SAG mill feed conveyor. The screen undersize will flow by gravity to the mill discharge hopper, where it will mix with the overflow slurry from the gravity feed standpipe and the milling area sump pumps' discharge.

The slurry from the mill discharge hopper will be pumped to classification cyclones operating in closed circuit with the ball mill. Cyclone underflow will be returned to the ball mill, while cyclone overflow will flow by gravity to a trash screen for removal of coarse debris ahead of the pre-leach thickener. The product exiting the SAB circuit (i.e., cyclone overflow) will have a targeted P_{80} of 53 µm, with the circuit providing flexibility to operate at a coarser grind size of P_{80} 75 µm when required.

Oversize from the ball mill trommel screen will be directed to a scats bunker, while the undersize will gravitate to the gravity feed standpipe.

The gravity feed standpipe will be used to collect slurry from the ball mill (undersize), gravity concentrator tails, and IC system leach residue. A portion of this slurry will be pumped to the gravity circuit, while the standpipe overflow will be transferred (by gravity) to the mill discharge hopper.

For both of the mills, the grinding media will be delivered to site in either bulk bags or drums. The grinding media will be unloaded by a forklift or other carrier tools into a bunker. The SAG mill grinding media will be transferred via a ball charger to be discharged directly onto the SAG mill feed conveyor. The ball mill grinding media will be transferred via a ball charger into a kibble. A hoist will lift the kibble and feed the media into the mill feed box.

Kerosene will be dosed into the SAG mill feed box to initiate the passivation of preg-robbing carbonaceous material. This will help reduce the adsorption of dissolved gold by the carbonaceous matter before it can be adsorbed onto the activated carbon in the downstream CIL circuit.

17.2.3 Gravity Concentration

The gravity feed standpipe discharge will be pumped to the gravity concentration circuit, comprising two scalping screens for feed preparation and two gravity concentrators for coarse gold recovery in the milling area. The nominal solids feed rate (dry) to the gravity circuit will be 847 t/h.

The scalping screens' undersize will feed the concentrators, while the oversize will gravitate to the ball mill.

The operation of the gravity concentrators will be in semi-batch mode. Gravity concentrators' tails will report to the gravity feed standpipe, and the gravity concentrate will be processed and subsequently leached by the IC system. There will be an option to bypass the scalping screens' undersize to the gravity feed standpipe.

17.2.4 Intensive Cyanidation

Concentrate from the gravity concentrator will gravitate to a vendor-supplied IC package / system to recover the contained gold by intensive cyanide leaching. The operation of the IC system will be in semi-batch mode.

The concentrate will gravitate to the feed storage hopper and then transferred to the reactor drum after the pre-determined feed weight (or tonnage) is reached at the hopper; at this point the batch processing will be initiated.

Filtered water will be added to this system. IC leach solution (water dosed with diluted NaCN and NaOH) will be stored in the solution hopper and transferred to the reactor drum to enhance the leaching of gold. Sodium hydroxide solution will be added mainly to maintain the pH above 10.5. Oxygen (with Leach-Aid as a back-up) will be added to the system as an oxidant during intensive leaching for a pre-determined period. Flocculant will also be added to the IC system to allow fine flocculated solids to settle in the reactor drum.

After completion of the intensive leaching, the reactor drum will be stopped, and the solution will be transferred to the solution storage to settle the solids. Sodium hydroxide will be added to the solution hopper, and the decanted clarified pregnant solution will be drained to a hopper to be pumped to the IC pregnant solution tank for gold sludge recovery using a dedicated IC electrowinning cell. The sludge from the IC electrowinning cell will be combined with the sludge from the elution plant electrowinning cells and will be filtered and smelted in the goldroom.

Once the pregnant solution transfer is complete, the reactor drum and the solution storage hopper will be washed, and the slurry will be collected in the pump hopper. This slurry (leach residue) will then be pumped to the gravity feed standpipe, with a bypass option to the CIL collection box.

17.2.5 Trash Screening and Pre-Leach Thickening

Cyclone overflow will gravitate to a trash screen. Trash screen will remove foreign materials (woodchips, plastic trash, etc.) as the screen's oversize and will report them to the trash bunker which will be periodically emptied.

The trash screen's undersize will gravitate to the pre-leach thickener to be thickened to 45% w/w solids for 'Dig' ore or 50% w/w solids for 'Blast' ore. Flocculant will be further diluted in a static mixer prior to dosing into the pre-leach thickener to aid with particle settling. The thickener overflow will be reused as process water, while the pre-leach thickener underflow will be pumped to the CIL circuit as leach feed.

An automatic two-stage slurry sampler, installed on the leach feed pumps' discharge line will collect representative samples from this stream for plant control and metallurgical accounting purposes.

17.2.6 Leaching

The leaching circuit will consist of a single train of seven CIL tanks. Lead nitrate solution will be added to improve the leach kinetics in this circuit. Cyanide solution will be added to selected CIL tanks as required, with control based on free cyanide and pH monitoring from an online free cyanide analyzer. Cyanide solution will be added to dissolve gold and silver, while oxygen will be added to maintain adequate aeration within the CIL tanks. pH will be controlled and maintained above 10.5 using quicklime, which will be added to the SAG mill feed conveyor. pH measurement and control will also be implemented in selected CIL tanks to ensure that pH is sufficiently high prior to cyanide addition to prevent HCN gas generation.

Fresh carbon or regenerated barren carbon will be fed via the carbon sizing screen to the last CIL tank with an alternative option to feed to the second last CIL tank if the last tank is down. Carbon will be advanced in a counter-current direction to the slurry flow using carbon advance pumps (i.e., recessed impeller type). The intertank screen in each tank will retain the carbon whilst allowing the slurry to flow by gravity to the downstream tanks. This counter-current process will be repeated until the gold loaded carbon reaches the first operating CIL tank.

A carbon recovery pump will be used to transfer the carbon slurry from the first operating CIL tank (CIL tank 1 or 2) to a vibrating loaded carbon recovery screen. Undersize slurry from the screen will gravitate to CIL tank 1 or 2 (bypass / operating). Oversize material (loaded carbon) from the screen will report to the acid wash column.

Slurry from the last operating CIL tank (CIL tank 6 or 7) will gravitate to a vibrating carbon safety screen to recover any carbon leaking from worn screens.

17.2.7 Carbon Acid Wash and Elution

Loaded carbon will gravitate from the loaded carbon recovery screen to the acid wash column where it will be treated with an injected HCl solution (3% w/v) to remove calcium, magnesium and other acid-soluble foulants that would otherwise render the elution process less efficient or be ‘baked on’ in the subsequent elution and carbon regeneration steps and ultimately fouling the carbon.

Entrained water will be drained from the acid wash column before being refilled with the diluted HCl solution. Once the column is filled with the carbon, it will be left to soak in the acid for 30 minutes which will include acid filling time.

Following acid injection and soaking, the spent acid will be rinsed from the carbon and drained to the acid area sump pump, located in an acid wash area bund. The acid-washed carbon will subsequently be hydraulically transferred to the elution column, where gold stripping process will take place.

Carbon elution will utilize a split-AARL elution method with a carbon capacity of 18 tonnes. Elution will commence with the injection of a fixed volume of strip solution into the bottom of the elution column. The strip solution will be made up of weak cyanide and caustic solutions. The strip solution will be re-circulated through the solution heater and heat exchanger to bring the solution temperature up to 140°C before entering the elution column.

Eluate leaving the elution column will pass through the recovery heat exchanger to either the stripping water tank discharge, lean eluate tank or one of two pregnant solution tanks depending on the sequence step. Lean eluate tank solution will be used in the elution step and refilled in the rinse and cooling steps in preparation for the next elution sequence.

Upon completion of the cool down sequence, the barren carbon will be hydraulically transferred to the carbon regeneration kiln feed hopper via a carbon dewatering screen. A bypass option will be provided to bypass the carbon regeneration process (to the carbon sizing screen), if required.

17.2.8 Electrowinning and Goldroom

Gold will be recovered from the pregnant eluate by electrowinning, sludge filtering and smelting to produce doré bars.

The Split-AARL process uses high temperature stripping solution to remove gold from carbon. The hot pregnant solution will be collected in the operating pregnant solution tank(s). The pregnant eluate from the pregnant solution tank(s) will be transferred to an electrowinning cell feed distribution box. During an electrowinning cycle, the pregnant eluate will pass through three electrowinning cells, operating in parallel, with stainless steel mesh cathodes. Gold will be electroplated on the cathodes, and the solution will be returned to the duty pregnant solution tank for recirculation until it is deemed barren. The barren solution will be returned to the CIL circuit.

The pregnant solution from IC pregnant solution tank will be pumped through a dedicated IC electrowinning cell to process pregnant solution from the IC system. Gold will be electroplated on the cathodes, and the barren solution will report back to the IC pregnant solution tank. The barren solution in the tank will be pumped to the CIL circuit (to the CIL collection box).

The gold-rich sludge will be washed off the stainless-steel mesh cathodes in all the electrowinning cells using high-pressure water sprays and will gravitate to the sludge filter hopper. The sludge will then be filtered in a sludge pan filter, dried in an oven, mixed with fluxes and smelted in a diesel-fired smelting furnace to produce gold doré bars.

The electrowinning and smelting process will take place inside a secured and supervised goldroom, equipped with access control and security systems. The electrowinning and gold room area will be serviced by a dedicated area sump pump. Any spillage within this area will be pumped back to the CIL circuit (to the CIL collection box).

17.2.9 Carbon Regeneration

Carbon will be regenerated in a diesel-fired, vendor-supplied carbon regeneration kiln with a nominal design capacity of 900 kg/h. Dewatered barren carbon from the elution column will be held in a kiln feed hopper. A screw feeder will meter the carbon into the kiln, where it will be heated to 750°C (maximum) in an atmosphere of superheated steam and held for at least 20 minutes to restore the activity of the carbon. This process will regenerate the pore structures and active sites of spent carbon primarily by removing foulants and restoring its adsorption capacity.

Carbon discharging from the kiln will be quenched with filtered water and pumped to the carbon sizing screen at the discharge end of the CIL train to remove undersized carbon fragments. The undersized fine carbon will be discharged to the carbon safety screen, and the oversize will be directed to either of the last two CIL tanks.

As carbon is lost by attrition, new carbon will be added to the circuit via the carbon quench hopper. The new carbon will be supplied in bulk bags and transferred via the carbon sizing screen into the circuit the same way as regenerated carbon.

17.2.10 Tails Handling

CIL circuit tailings slurry will report to a carbon safety screen. An automatic two-stage slurry sampler will be installed on the CIL tails line to the carbon safety screen and will collect representative sample for plant control and metallurgical accounting purposes.

Fine carbon from the screen oversize will be collected in a carbon collection bag. The screen undersize materials will gravitate to a tailings hopper, prior to being gravitated to the TSF.

A Weak Acid Dissociable (WAD) cyanide analyser will be installed to measure the concentration of WAD cyanide in the tails discharge solution. This will enable continuous monitoring of cyanide concentrations in tailings slurry to ensure that cyanide levels in the slurry remain within specified limits for discharge to the TSF.

TSF Decant Water Treatment Plant

A vendor supplied TSF Decant Water Treatment Plant will be installed after year 1 to treat approximately 150 m^3/hr of decant water recovered from the TSF. The system will collect decant water from the TSF using a floating turret and pump arrangement and treat the water to meet applicable discharge criteria prior to any release to the environment. Natural degradation of residual cyanide within the TSF decant pond is anticipated through exposure to sunlight and atmospheric conditions. To further reduce cyanide concentration in the tailings solution, a decant water return line will be tied to a location near the tailings spigots to provide additional dilution to the tailings stream.

17.3 Reagents Handling and Storage

The following reagents will be required in the process plant: quicklime, flocculant, sodium cyanide, kerosene, activated carbon, sodium hydroxide, hydrochloric acid, lead nitrate, anti-scalant, sulphamic acid, and smelting fluxes (borax, soda ash, silica sand and sodium nitrate).

For the management of unexpected reagent spills, the reagent preparation and storage facilities will be located within containment areas designed to accommodate more than the content of the largest tank.

Where required, each reagent system will be located within its containment area to facilitate its return to its respective storage vessel and to avoid the mixing of incompatible reagents.

Storage tanks will be equipped with level indicators, instrumentation, and alarms to ensure spills do not occur during normal operation.

Sumps and sump pumps will be provided for spillage control. Necessary ventilation, fire and safety protection, eyewash stations, and Material Safety Data Sheet (MSDS) stations will be located throughout the facilities, to minimize hazard and enhance safety practices.

17.3.1 Quicklime

Quicklime (90% CaO) will be used for pH control in the leach circuit. Lime will be delivered to site in lime tanker and then transferred to the lime silo as required, using a lime transfer blower. Quicklime will be extracted from the lime silo via a variable-speed rotary valve or feeder and fed to the SAG mill feed conveyor.

As an emergency / contingency system, lime will be delivered in 1 t bulk bags and transferred to the lime transfer hopper as required. Bags will be lifted into the dedicated bag breaker located on top of the lime transfer hopper using hoist and lifting frame, and lime will be loaded into the transfer hopper. Quicklime will be extracted from the lime transfer hopper via a variable-speed rotary valve or feeder and fed to SAG mill feed conveyor.

17.3.2 Sodium Cyanide

Sodium cyanide (NaCN) will be used mainly for gold dissolution and desorption. Cyanide briquettes will be delivered in 1 t bulk bags which will be lifted into the bag breaker located on top of the cyanide mixing tank, by the bag lifting frame and hoist. Raw water will be added to the mixing tank to the level required for achieving the stock solution concentration. Sodium hydroxide will also be added to the mixing tank to raise the pH to avoid the generation of HCN gas. An agitator will be used to dissolve cyanide in the mixing tank.

After the mixing period is complete, cyanide solution will be transferred to the cyanide storage tank using the cyanide transfer pump. Cyanide will be delivered to the CIL circuit via a ring main. Separate cyanide dosing pumps will be used to deliver cyanide solution to the IC system and elution circuit.

17.3.3 Sodium Hydroxide

Sodium hydroxide (NaOH) will be used mainly in the IC system and elution circuit, for neutralization and desorption processes.

Sodium hydroxide beads or flakes will be delivered in 1 t bulk bags. Bags will be lifted into the bag breaker located on top of the mixing and storage tank by the lifting frame and hoist (shared with the hoist dedicated for the cyanide system).

Sodium hydroxide will then be transferred to the mixing and storage tank by a screw feeder. Raw water will be added to the tank to the level required for achieving the stock solution concentration before the addition of sodium hydroxide beads. The sodium hydroxide solution will then be pumped to the elution circuit, IC system and cyanide mixing tank. When required, sodium hydroxide will be used in the CIL circuit for pH correction, to avoid the generation of HCN gas.

17.3.4 Hydrochloric Acid

Hydrochloric acid (HCl) will be used for carbon acid washing. Concentrated acid will be delivered in 1 m^3 IBC. An unloading pump will pump the concentrated acid from the IBCs to the mixing and storage tank where raw water will be added to dilute the acid to the desired stock solution concentration. Diluted hydrogen chloride solution will be pumped to the acid wash column.

17.3.5 Flocculant

Flocculant will be used mainly for the IC system and thickening processes. Powdered flocculant will be delivered to the site in 1 t bulk bags.

A vendor supplied flocculant mixing and transfer system will be installed, which will primarily include a hopper, a blower, a wetting head, a mixing tank, a bag breaker, and a flocculant transfer pump.

Powdered flocculant will be loaded manually into the hopper located at the ground level. Dry flocculant will be pneumatically transferred into the wetting head, where it will be contacted with raw water. Flocculant solution, at 0.25% (w/v) concentration will be agitated in the mixing tank for a pre-set period for proper hydration. After a pre-set time, the flocculant will be transferred to the storage tank using the flocculant transfer pump.

Flocculant will be dosed to the pre-leach thickener and the IC system, using dedicated dosing pumps. Flocculant will be further diluted (using HP process water) in the static mixer to a concentration of 0.025% (w/v), before dosing into the thickener.

17.3.6 Lead Nitrate

Lead Nitrate ($Pb(NO_3)_2$) will be delivered in powder form in 1 t bulk bags to the site. Raw water will be added to the agitated mixing / storage tank. Bags will be lifted into the bag breaker, located on top of the tank, using the lifting frame and hoist. The reagent will fall into the tank and be dissolved in water to achieve the required dosing concentration. Lead Nitrate solution will be then delivered to CIL circuit using the dosing pump(s). Reagent solution will be transferred to the buffer tank before the next batch preparation, so there is a continuous supply of solution available for the CIL circuit.

17.3.7 Activated Carbon

Activated carbon will be delivered in a solid granular form in 500 kg bulk bags. Fresh carbon will be introduced into the carbon quench hopper where it will be contacted with filtered water and pumped to the carbon sizing screen. The screen undersize or the carbon fines will discharge to the carbon quench hopper, and the screen oversize or carbon will be transferred to the last tank of the CIL circuit (or tank 6 if tank 7 is not operating).

17.3.8 Kerosene

Kerosene will be used to mitigate preg-robbing effects, thereby enhancing overall CIL circuit performance. It will be dosed in the SAG mill feed box, when 'Blast' ore is being processed. Kerosene will be delivered to site in 1 m^3 IBCs. Dosing pump(s) will transfer kerosene from the storage tank to the milling circuit.

17.3.9 Anti-Scalant

Anti-scalant will be used mainly for reducing scaling in the stripping water and process water tanks. Anti-scalant will be delivered in solution form in IBCs and stored in the reagent shed until required. IBCs will be placed locally to the dosing points, which will be the stripping water and process water tanks. It will be dosed neat without any dilution. A dedicated dosing pump will deliver the anti-scalant to the dosing points. Top up of the IBCs will be carried out manually as required.

17.3.10 Sulphamic Acid

Sulphamic acid will be delivered in IBCs to the site and stored in the reagent shed. Solution will be dosed as required, to the heat exchanger in the elution area.

17.3.11 Smelting Fluxes

Smelting Fluxes (borax, soda ash, silica sand and sodium nitrate) will be used mainly for smelting charge preparation. The fluxes will be delivered as solid crystals in 25 kg bags and stored in the reagent shed until needed. Smelting fluxes will be used to remove impurities, enhance gold separation, and improve molten gold fluidity, enabling efficient recovery and production of high-purity gold doré bars.

17.3.12 Annual Consumptions

A summary of the annual reagents consumption is shown in Table 17.3.1.

Table 17.3.1 Annual Reagents Consumption

Reagents	Annual Consumption (t/y)		
	'Dig' Ore at P_{80} 75 µm	'Dig' Ore at P_{80} 53 µm	'Blast' Ore at P_{80} 53 µm
Quicklime	19,381	18,213	10,464
Sodium Cyanide	3,807	4,278	5,425
Sodium Hydroxide	198	199	201
Hydrochloric Acid	687	687	687
Flocculant	740	740	960
Lead Nitrate	1,403	1,403	1,403
Activated Carbon	165	165	165
Kerosene	-	151	1,157
Anti-Scalant	3.07	3.07	3.07
Sulphamic Acid	2.08	2.08	2.08
Smelting Fluxes	2.30	2.30	2.30

17.4 Water Supply

The plant will be equipped with raw and process water supply systems. Raw water, sourced from rainwater, will be stored and treated for use in reagent mixing, dust suppression in the crushing area, and as a feed source for the filtered and potable water treatment plants. Initially, rainwater collected in the TSF will be used to supply water for plant start-up. Thickener overflow and TSF decant water will serve as the primary source of process water. Due to the rainy climate in Ghana, raw water make-up to the process water tank is not anticipated. After the first year of operation, any excess water in the TSF decant pond will be treated and discharged to the environment to manage the positive water balance (if any).

17.4.1 Raw Water Supply

Raw water will be stored in the raw water tank at the plant, and will be supplied to all users, including:

- Water for crushing area dust suppression.
- Water for reagents' mixing.
- Supply water to the potable water treatment plant.

- Supply water to the filtered water treatment plant.

The raw water tank will also reserve a portion of the volume for fire water. Fire water will be supplied and distributed to fire hydrants and hose reels within the plant via a ring main distribution system.

17.4.2 Process Water Supply

The raw water tank overflow will report to the process water tank. Pre-leach thickener's overflow water stream will also be sent to the suction of the high-pressure (HP) process water pumps.

The HP process water pumps will supply water to flocculant static mixers, and screen sprays in the milling, CIL and elution circuits, mainly to carbon sizing, carbon safety, gravity scalping, loaded carbon recovery, ball mill trommel, SAG mill trommel and trash screens.

The low-pressure (LP) process water pumps will deliver dilution and flushing water to different plant areas, mainly to the mill discharge hopper, SAG and ball mill feeding, and different slurry pumps.

17.4.3 Potable Water Supply

Raw water will be fed to a potable water treatment plant to be further treated and sterilized via filtration, chlorination and Ultra-Violet (UV) sterilisation.

Plant potable water will be supplied to the power station, administrative/plant offices, laboratories, ablutions and safety and WAD cyanide analysers. The treatment plant backwash effluent will be sent to the tailings hopper.

17.4.4 Filtered Water Supply

Raw water will be fed to a filtered water treatment plant to be further treated via multi-media filtration.

Filtered water will be used as gland seal water for slurry pumps (cyclone feed, gravity feed, leach feed, and HP and LP process water pumps), and used in the IC system, the elution circuit, and the goldroom.

The treatment plant backwash effluent will be sent to the tailings hopper.

17.4.5 Cooling Water Supply

Filtered water will be fed to a vendor supplied cooling tower package to reduce water temperature to a suitable point for supply to the mills. The cooling tower will remove heat from the circulating water and return cooled water to the system. Water will be supplied to vendor package equipment including SAG mill and ball mill drives, where heat is generated during operation. After absorbing heat, the water will return back to the cooling tower, forming a closed-loop cooling cycle that will maintain equipment temperature effectively within the desired control range.

17.5 Services and Utilities

This plant will be equipped with a vendor supplied rotary screw air compressor and dryer system to meet plant air demands. An oxygen plant will be used to supply oxygen via spargers to each of the CIL tanks. A diesel-supply system will also be installed to support fuel requirements.

17.5.1 Air Services

Compressed HP plant air at 750 kPa(g) will be provided by rotary screw-type plant air compressors, operating in a lead-lag configuration.

Compressed air will be dried and fed to plant air receiver, from which it will be distributed to the required plant areas via dedicated air receivers servicing the feed preparation, milling, reclaim, elution and tailings areas, as required.

17.5.2 Oxygen Plant

The oxygen plant will be a vendor supplied system. Oxygen will be supplied to the plant at a required pressure, flow rate, and purity. Oxygen will be primarily utilized in the CIL circuit and the IC system.

17.5.3 Fuel

Diesel will be supplied from a dedicated diesel day tank to the strip solution heater, carbon regeneration kiln and smelting furnace systems.

17.5.4 Energy Requirements

The estimated annual power consumption is summarized in Table 17.5.1.

Table 17.5.1 Annual Power Usage

Area	Power Usage kWh/y		
	'Dig' Ore at P_{80} 75 µm	'Dig' Ore at P_{80} 53 µm	'Blast' Ore at P_{80} 53 µm
Feed Preparation (Sizing)	3,659,867	3,659,867	
Feed Preparation (Crushing)			3,425,071
Stockpiling & Reclaim			604,532
Milling Circuit	85,885,553	97,039,956	178,632,116
Trash Removal & Thickening Circuit	1,899,630	1,899,630	1,899,630
Leaching Circuit	14,904,431	14,904,431	14,904,431

Area	Power Usage kWh/y		
	'Dig' Ore at P_{80} 75 µm	**'Dig' Ore at P_{80} 53 µm**	**'Blast' Ore at P_{80} 53 µm**
Elution & Gold Room Circuit	1,189,390	1,189,390	1,189,390
Tails Handling Circuit	4,467,162	4,467,162	5,746,823
Reagents	670,297	670,297	670,297
Water Services	5,423,912	5,423,912	5,423,912
Plant Services	3,034,028	3,034,028	3,034,028
Air Services	8,241,048	8,241,048	8,241,048
Fuels	32,991	32,991	32,991
Process Plant Buildings (Lighting, Switching, Miscellaneous)	3,572,519	3,572,519	3,572,519
TOTAL	**132,980,828**	**144,135,231**	**227,376,788**

18.0 PROJECT INFRASTRUCTURE

18.1 Introduction

The PFS details the key infrastructure requirements for the Enchi Gold Project. Key Infrastructure is detailed in the upcoming sections and includes the following scope:

- Four open pit mining areas.
- Site access roads.
- Site haul roads.
- Tailing Storage Facility (TSF).
- Site water management system.
- Mine services area.
- Administration and plant buildings.
- Process plant, including plant site, warehouses and ROM pad foundation.
- Stockpiles.
- Power generation plant.
- Fuel supply.
- Communications.
- Plant security.

The Overall Site Layout plan is shown below in Figure 18.1.1 and Process Plant Layout is shown in Figure 18.1.2.

Figure 18.1.1 Overall Site Plan

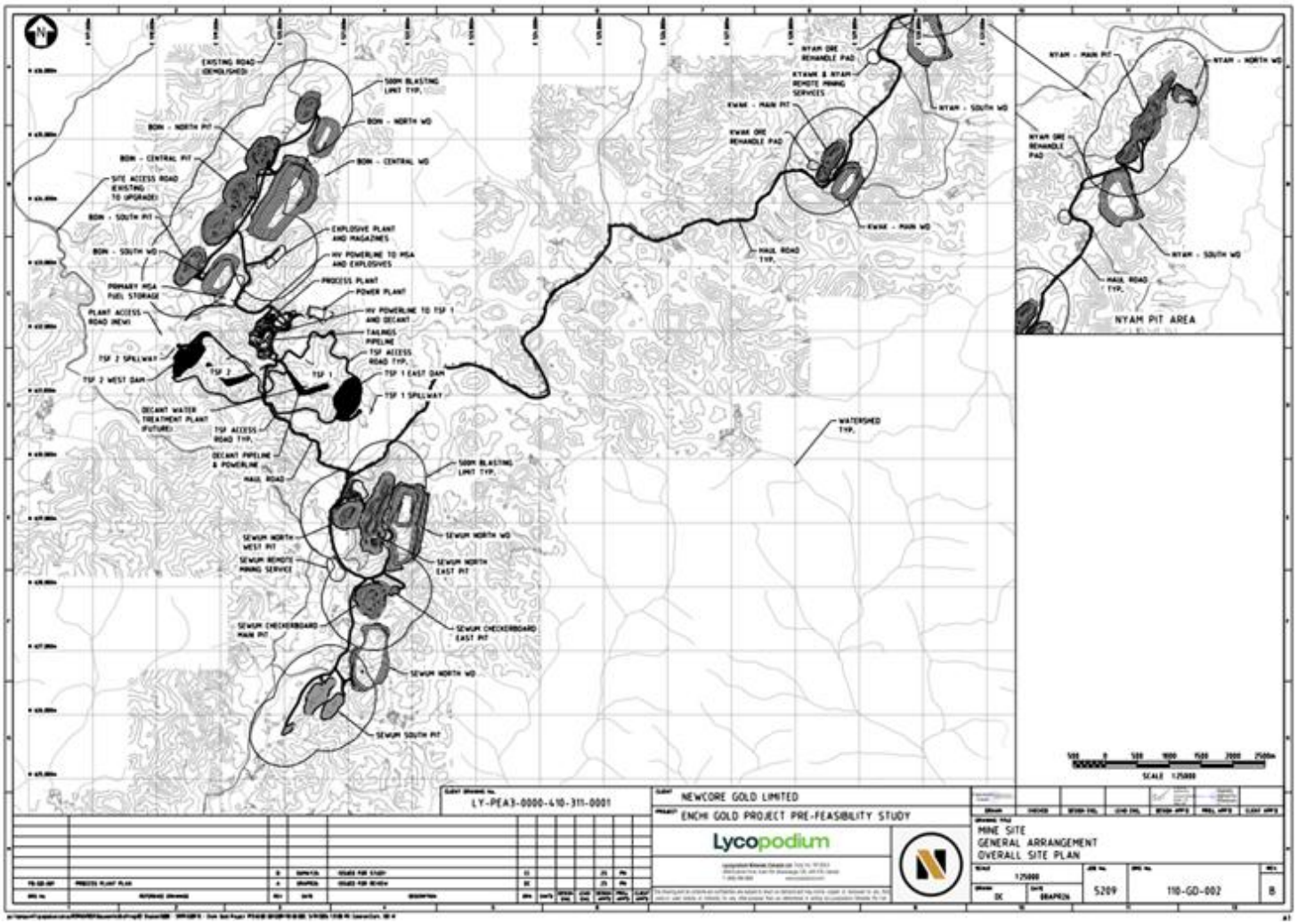
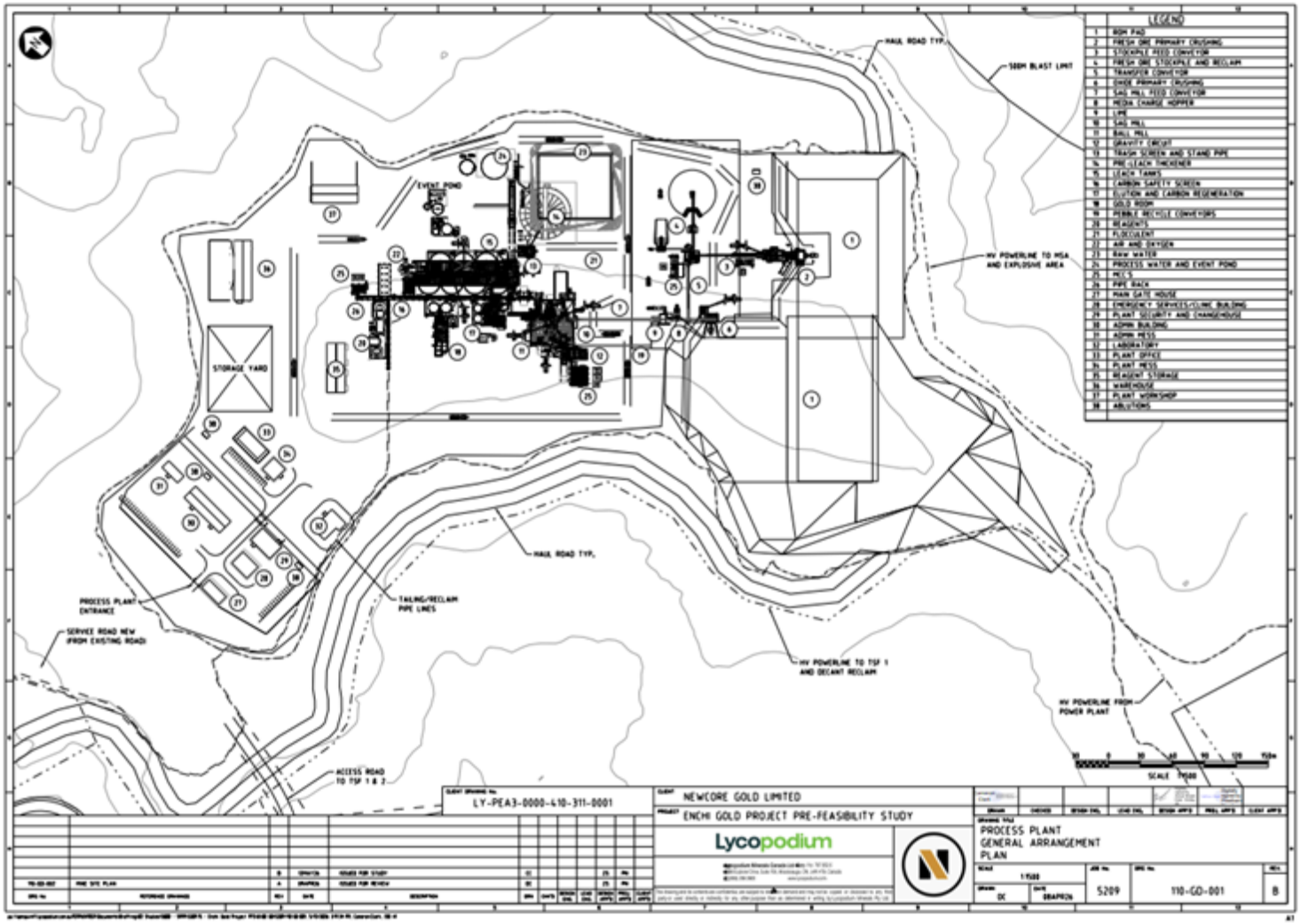


Figure 18.1.2 Process Plant Layout



18.2 Roads

Access to the Project site is provided by an existing gravel road that connects the village of Akunso to Enchi. The national highway N12 passes through Enchi and the village of Kramokrom, which is located northeast of the plant site. When travelling from Enchi, the access road to the Project site and Akunso branches off the N12 just before entering the village of Kramokrom.

The nearest seaport is located in Takoradi, approximately 205 km southeast of the property via the N1 and N12 paved highways. Accra, the capital of Ghana and the country's principal gateway for international air and sea travel, is situated approximately 427 km east of the project site by road.

18.2.1 Project Site Roads

Internal roads are planned to provide light vehicle access between crushing, process plant and mine services areas including the mine pits, mine dry complex, explosives magazine and fuel and lubricant storage areas. These roads will be 6 metres wide and constructed flush with bulk earthworks pads to facilitate stormwater management.

18.2.2 Mining Haul Roads

Mining haul road design for the Enchi Gold Project has been developed in accordance with the criteria outlined in Section 15.8.2, reflecting anticipated equipment selection and operational requirements. Road width allowances assume the use of 91-tonne haul trucks across all mining areas, with double-lane configurations designed to comply with applicable regulatory standards, including berm heights set at approximately 75% of the height of the largest haul truck tire operating on the roadway. These designs incorporate appropriate provisions for safe vehicle passing, safety berms, and surface water drainage.

18.3 Waste Rock Storage Facilities

Design parameters for the waste rock storage facilities are outlined in Section 16.3, including swell factors, bench heights and slopes, and a summary of all dumps with their respective storage capacities relative to requirements. Section 16.4 presents the mine layouts, illustrating the location, extent, and configuration of each dump area in relation to adjacent surface mining operations.

18.4 Process Plant Infrastructure

The infrastructure will support mining, processing, and construction activities. Located centrally between the Sewum and Boin Mines, the process plant will host crushing and feed preparation facilities, processing plant operations, and all site buildings. Open pit mines will operate independently but connect to the process plant via haul roads and mine access roads.

Key process plant infrastructure includes:

- ROM pad, ROM stockpiling area, crushing and feed preparation facilities.

- Processing / Extraction plant (Includes SAGB milling circuit, gravity circuit, pre-leach thickening, CIL leach tanks, carbon safety, elution and carbon regeneration facilities).
- Utilities (air and oxygen, raw water, process water, fire water, potable water).
- Reagents storage.
- Pipe racks.
- Event pond.

Services provided to the process plant cover power generation and distribution, water supply and distribution, surface water management, sewage and waste management. Electrical switchrooms (MCCs) will be located near areas of use.

18.4.1 Buildings

The buildings onsite consist of administrative and operational facilities for personnel involved in day-to-day site activities such as mining, construction and process plant operation.

Most buildings considered for this study are to be pre-fabricated, modular style buildings built with semi-permanent building materials such as cladding, corrugated steel roofing, etc.

The compound just after the entrance gate is dedicated for administrative and operational facilities and is located east of the process plant. Key buildings included in this compound are:

- Main entrance gate house.
- Emergency services / clinic building.
- Plant security and changehouse.
- Main admin building.
- Admin mess hall.

Inside the process plant fenced area, just after the entrance gate and to the east of the process plant, the following buildings are included:

- Laboratory.
- Plant office.
- Plant mess.
- Plant warehouse and workshop.

- Reagent storage.
- Storage yard.

There are four ablutions facilities in the project site for both male and female ablutions. Two in the admin compound, one alongside the plant office, and one to the west of the process plant close to the primary crushing facility.

The gold room is situated south of the process leach tank area. This is envisioned to be a steel-clad building. The building will house the leach reactor, calcine oven, electrowinning cells, smelting furnace, safe (enclosed within a concrete vault), and associated equipment. A supervisor workstation will be installed in the gold room; this workstation will be equipped with a telephone and data connection. A secure area with inner and outer doors will ensure that the gold room remains sealed during bullion transfer to the transport vehicle. All operations within the gold room will be subject to full-time closed-circuit television (CCTV) surveillance with security alarms provided to the security coordinator.

18.5 Power Supply and Distribution

The Plant is estimated to have a connected load of approximately 37MW with a maximum demand (running) of an estimated 33MW. This is significantly higher than the previous study due to additional loads and process plant infrastructure requirements. Reference the mechanical equipment list.

There are currently two options for power supply for mining operations: Both options remain technically feasible and capable of supplying the forecast project electrical demand with option 2 being the preferred based around:

- Lower Net Present Cost over the life of the project.
- The provision of dedicated onsite generation capacity.
- Integrated fuel and power supply responsibility under a single counterparty.
- The avoidance of significant upfront capital expenditure for power infrastructure.

Option 1

Supply from the grid via the GRIDCo 161 kV Network. This would involve construction of a dedicated 161 kV transmission line and step-down substation that would be connected to the Ghana GRIDCo 161kV network.

Option 2

Supply from an onsite gas-fired generation via a build-own-operate (BOO) arrangement with Genser Energy. This involves development of a dedicated onsite gas-fired power station and natural gas pipeline infrastructure owned and operated by Genser.

Based on preliminary information, the Genser Option 2 (onsite gas-fired generation) is the preferred power supply solution. The implementation schedule for this option will be influenced by long-lead engine procurement and gas infrastructure development requirements. The grid supply option is anticipated to have an implementation timeframe of approximately 24 months, subject to transmission infrastructure approvals and construction requirements.

The plant power supply originates from the 33kV power station site switchboard via options 1 or 2 above and follows the haul road via underground conduits and pits, crossing under the haul road and entering the plant switchboard.

Plant process areas are planning to have the following switchrooms and transformers which would be fed from the plant switchboard by a network of underground conduits and pits.

Table 18.5.1 Summary of Switchrooms - Transformers and Switchboards

Area	Switchroom	Transformer	Switchboard
Primary Crushing Fresh Ore	120-SR-001	120-TX-001	120-MC-001
Primary Crushing Oxide Ore	120-SR-001	120-TX-002	120-MC-002
Grinding Area	130-SR-001	130-TX-001	130-MC-001
Grinding Area – SAG Mill	340-SR-002	130-TX-002	340-SB-002
Grinding Area – Ball Mill	340-SR-002	130-TX-003 and 004	340-SB-002
CIL and Thickeners	160-SR-001	160-TX-001 and 002	160-MC-001 & 002
Air-Reagents-Gold Recovery	240-SR-001	240-TX-001 and 002	240-MC-001 & 002

A 33kV overhead powerline network originates from the 33kV power station site switchboard where it splits and feeds two separate areas as listed below:

1 Supply to the mine services and explosives area (by others).

2 Supply to a plant RMU (Ring Main Unit for electrical power distribution and used for switching, protection and isolation) within the plant compound, which in turn feeds the site and plant infrastructure (e.g. site and plant building distribution) then continues out to the tailings storage and decant / return area transformers and switchboards.

18.6 Tailings Storage Facility

18.6.1 Design Basis and Criteria

The TSFs design was performed within current regulatory compliance requirements and international best practice requirements adopted for the PFS. The following regulations and guidelines were applied:

National Regulations

- Minerals and Mining (Health, Safety and Technical) Regulations, 2012 (L.I. 2182).

- Environmental Protection (Environmental Assessment) Regulations,2025 (L.I. 2504).
- Water Resources Commission Act, 1996 (Act 522).

International Guidelines

- ICMM (revised Good Practice Guide on Integrated Mine Closure 2nd edition).
- Global Industry Standard on Tailings Management.
- ANCOLD Guidelines on Tailings Management (2019).
- CDA Tailings Dam Breach Bulletin (2021).

A design criteria has been developed for the proposed TSFs to enable evaluation of the available storage capacity within each TSF site and the combination of sites that can meet the Project requirements. It incorporates storage, regulatory, and project-specific requirements. Table 18.6.1 presents a summary of the design basis.

Table 18.6.1 Design Basis Summary

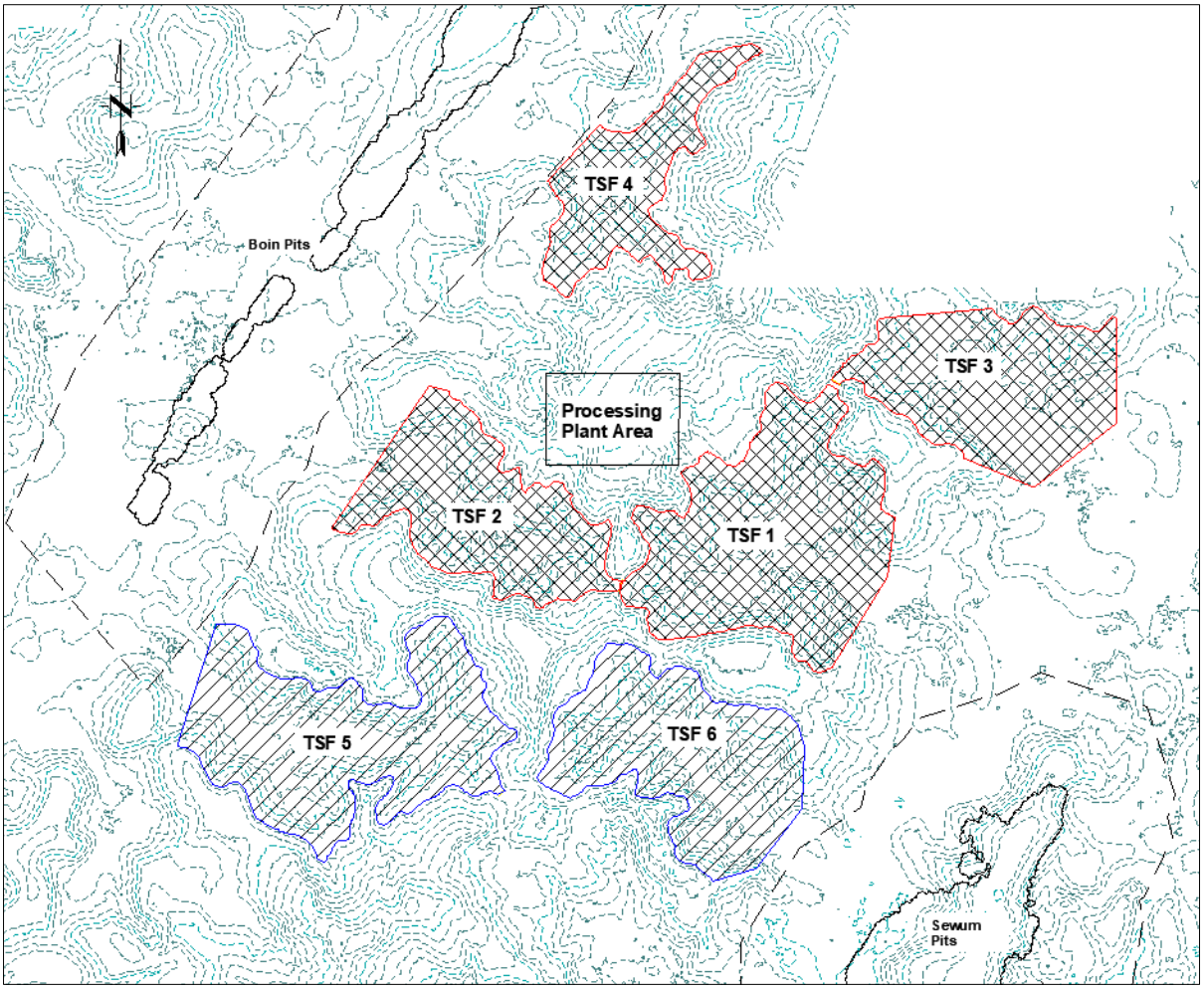
Design Criteria	Value	Source
Storage	Tailings and Process Water	Client
Annual throughput	5.5 Mtpa	Lycopodium
LoM	9.3 years	Lycopodium
Tailings Characteristics		
Average Sub-Aerial Dry Density	1.4 t/m^3	KP
Average Sub-Aqueous Dry Density	1.1 t/m^3	KP
Tailings Deposition		
Beach Slopes	1V:200H	KP
Subaqueous Slopes	1V:50H	KP
Design Rainfall Depths		
1 in 10 year 24 hour	149 mm	KP
1 in 100 year 24 hour	223 mm	KP
1 in 1000 year 24 hour	308 mm	KP
Probable Maximum Precipitation (PMP)/24 hr	692 mm	KP
Stability Analysis		
Static Drained Factor of Safety	≥ 1.7	L.I. 2182
Static Undrained Factor of Safety	≥ 1.5 (1.3 for no loss of containment)	ICOLD / ANCOLD
Post Liquefaction Factor of Safety	≥ 1.2	CDA

18.6.2 TSF Sitting Assessment

The study assessed six potential tailings storage facility (TSF) sites to identify the most suitable options for managing the Project's projected 49 Mt life-of-mine tailings throughput. A Multi-Criteria Accounts Assessment (MAA) was used to evaluate each option systematically against technical, environmental, economic, and social / governance criteria. This method provided an objective basis for comparing the sites and selecting options that could be advanced to conceptual design level as practical and provide cost-effective tailings management.

The assessment identified TSF 1 as the preferred primary option, recommended TSF 2 to meet current life-of-mine storage requirements and indicated TSF 4 as a potential option for future expansion. Figure 18.6.1 presents the layout of the sites considered in the assessment.

Figure 18.6.1 TSF Location Options



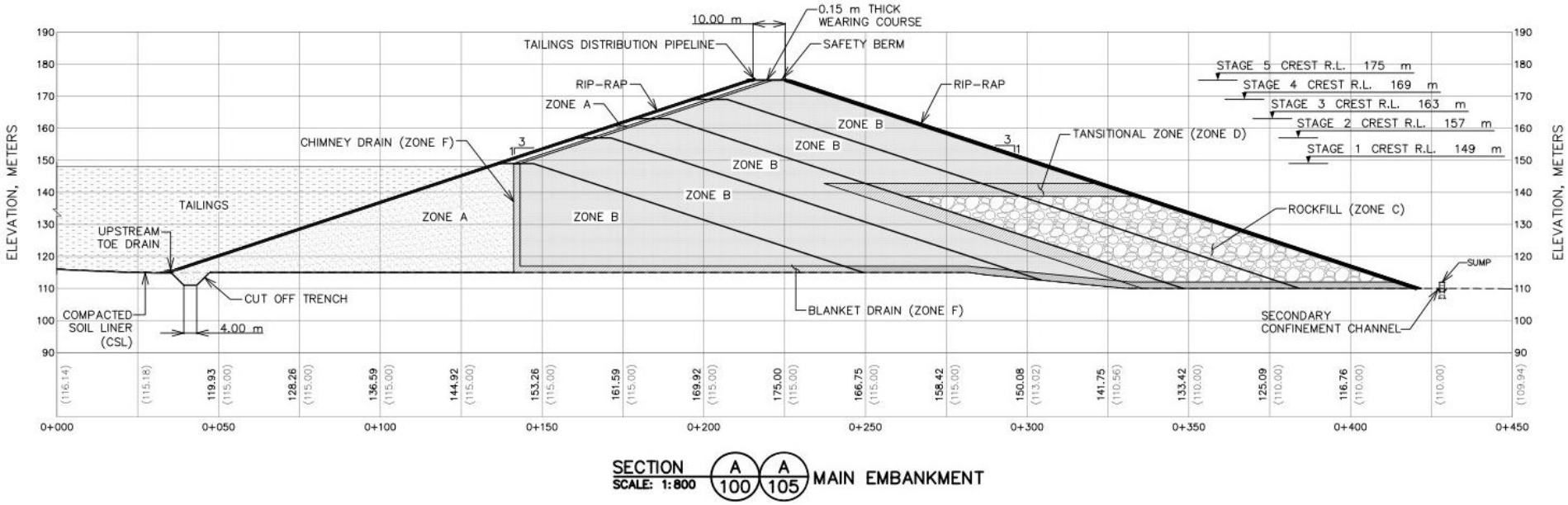
The subsequent sections present the civil design for TSF 1 and 2.

18.6.3 Embankment Geometric Design

The embankments for the TSF 1 and TSF 2 are designed to be constructed using the downstream method. The final crest elevation for the two TSFs is set at 175 mRL and 173 mRL respectively. The crest will be 10 m wide for single-lane access for light vehicles. Both upstream and downstream slopes are designed at a 1:3 (horizontal to vertical) gradient respectively for both TSFs.

The embankments are generally designed in a trapezoidal profile, incorporating a sequence of engineered fill zones to provide hydraulic containment, filter stility and erosion resistance. Typical embankment configuration is presented in Figure 18.6.2.

Figure 18.6.2 Typical TFS Embankment Configuration



18.6.4 Storage Capacity Analysis

The tailings storage capacity analysis was carried out in Muck3D (3D modelling software) considering an anticipated annual tailings throughput of 5.5 Mtpa. Average in-store tailings dry densities adopted were 1.4 t/m^3 and 1.1 t/m^3 for sub-aerial and sub-aqueous tailings, respectively. Sub-aerial deposition was modelled to the maximum beach elevation of 174 mRL, for TSF 1 and 172 mRL for TSF 2 considering the various input parameters and assumptions.

From the storage capacity analysis, the proposed TSF 1 and TSF 2 configuration will provide estimated storage capacities of 34.7Mt and 16.5 Mt respectively.

The results from the analysis were used to plot the cumulative storage, which is presented in Figure 18.6.3 and Figure 18.6.5.

An overview of TSF1 and TSF 2 at the end of deposition to tailings elevation of 174 mRL and 172 mRL are presented in Figure 18.6.4 and Figure 18.6.6.

Figure 18.6.3 TSF 1 Storage Capacity Curves

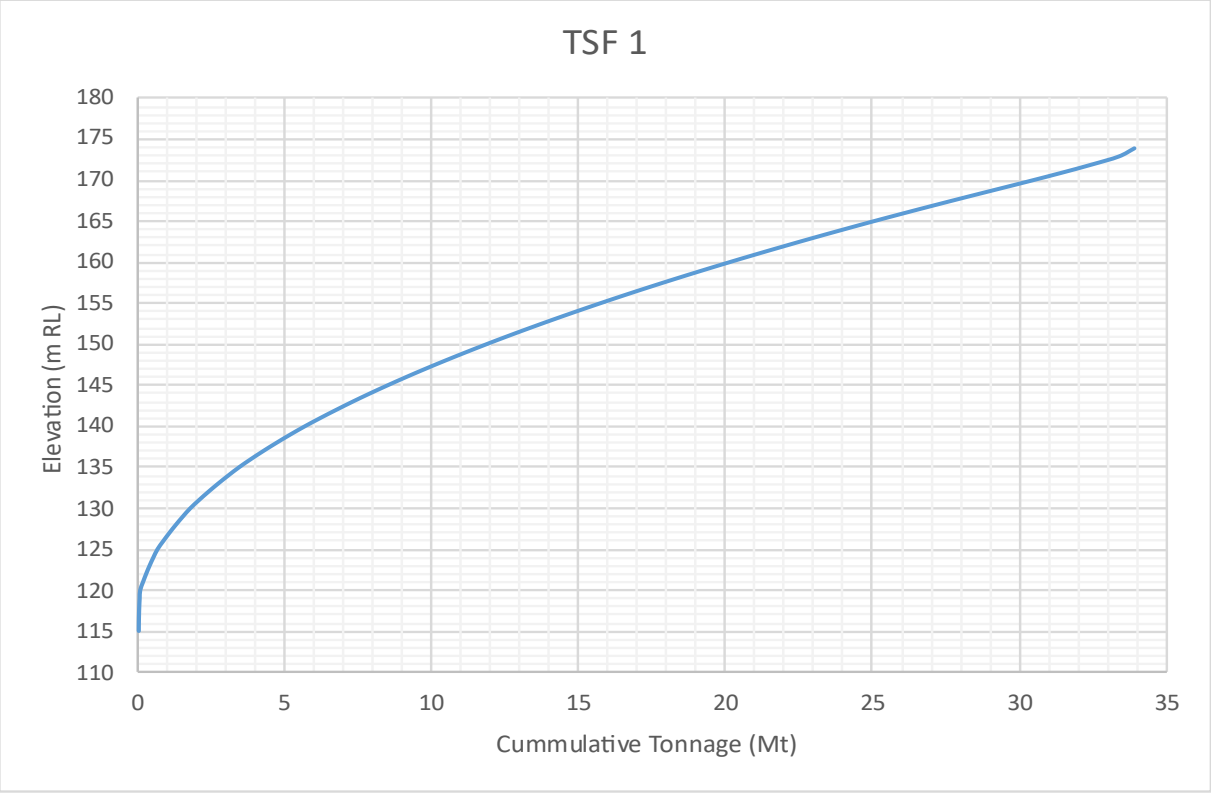


Figure 18.6.4 TSF 1 Post Deposition



Figure 18.6.5 TSF 2 Storage Capacity Curves

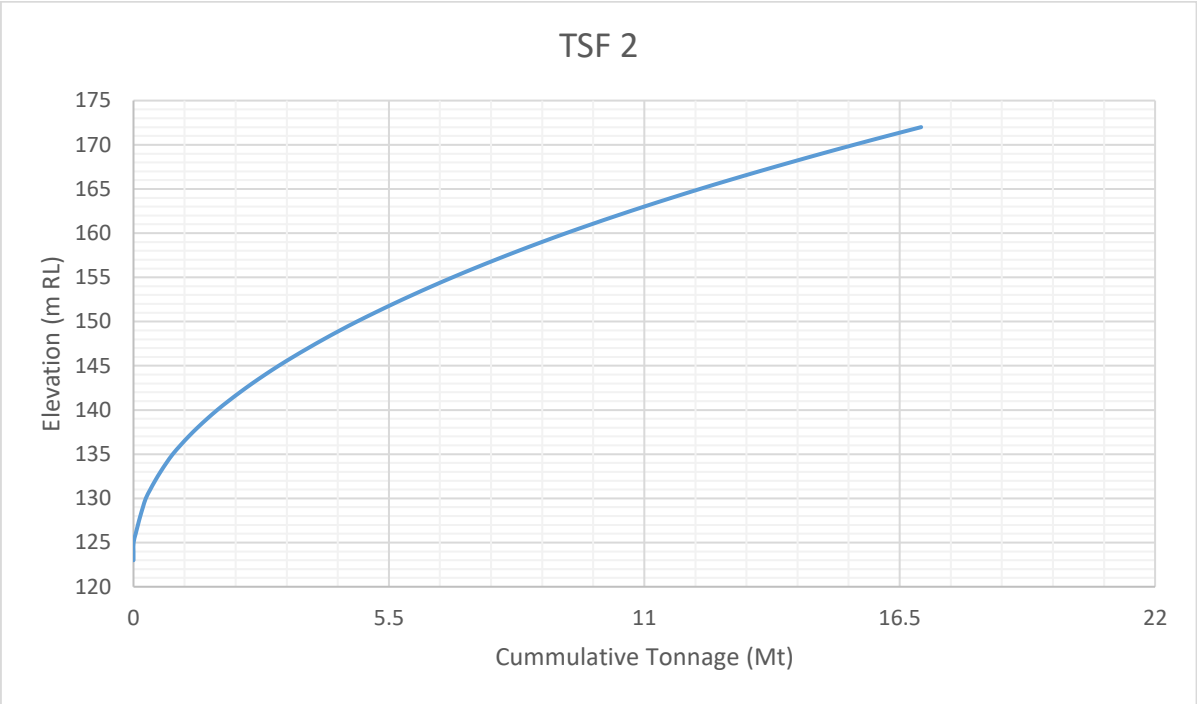
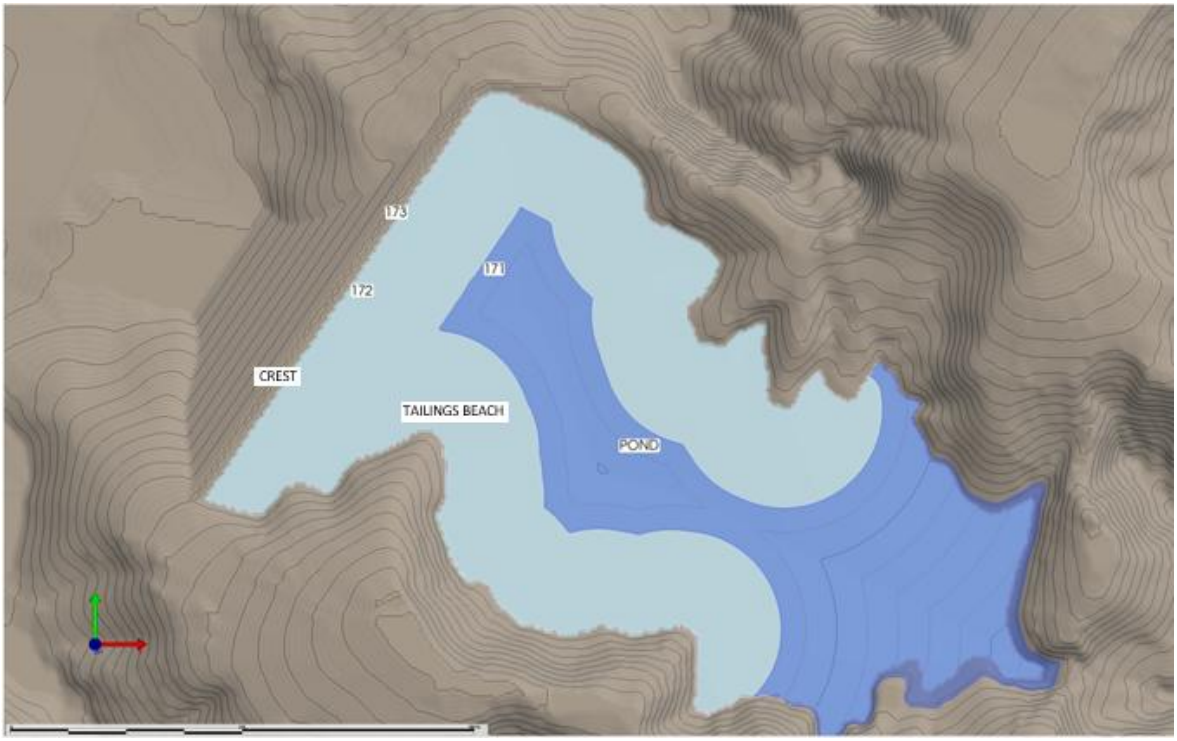


Figure 18.6.6 TSF 2 Post Deposition



18.6.5 TSFs Construction Schedule

Table 18.6.2 shows the respective periods (years) for construction of TSF 1 and TSF 2. TSF 1 will be required before start of processing plant operations.

Table 18.6.2 TSF Construction Stages and Cres Elevations

Facility ID	Stage	Year of Construction	Crest Elevation m RL
TSF 1	1	0	141
	2	1	149
	3	2	157
	4	3	163
	5	4	169
	6	5	175
TSF 2	1	6	153
	2	7	164
	3	8	173

18.6.6 Slope Stability Analysis

The TSF 1 and 2 main embankment Stage 1 and ultimate configurations were analysed under static drained, static undrained and post- earthquake loading conditions and the results were compared with the minimum recommended factor of safety requirements as detailed in the design criteria.

Factor of safety (FoS) is defined as the ratio of resisting forces to acting forces. An FoS less than unity (< 1.0) indicates that acting forces exceed resisting forces, resulting in an unstable condition. Conversely, an FoS greater than unity (> 1.0) indicates that resisting forces exceed acting forces, resulting in a stable condition.

The slope stability analyses were completed using the two-dimensional computer software (GeoStudio 2021) which allows the user to conduct limit-equilibrium slope stability calculations by a variety of calculation methods.

The material properties used in the analysis were derived from foundation characterization considering the limited site-specific subsurface data and experience working with similar materials within the project vicinity. The results typically satisfy or surpass the established minimum design criteria, demonstrating the adequacy of the embankment configuration.

18.6.7 Basin Liner

The TSF 1 and 2 basin is assumed to be constructed with a low permeability compacted clay liner (CCL) system to minimize seepage losses and prevent potential contamination of underlying soil and groundwater. The liner system provides a robust, proven barrier technology suitable for the site's geological conditions, regulatory and environmental protection requirements.

18.6.8 Surface Water Management

Effective water management is fundamental to the safe operation and long-term stability of TSFs.

The aim of the water management strategy is the segregation of contact and non-contact water streams. Contact Water is water that has been in direct contact with tailings material or has the potential to contain elevated concentrations of process chemicals or dissolved metals.

Contact Water Segregation.

Within the TSF 1 and 2 system, contact water comprises four distinct sources: Supernatant Pond, Underdrainage Seepage, Embankment Toe Seepage, Groundwater Drainage System. The following systems are designed to manage contact water.

Supernatant Pond Decant System.

The supernatant pond represents the largest volume of contact water requiring active management.

The decant system will consist of a diesel-powered pump mounted on a floating platform or pontoon, accessible via a retracting rockfill causeway extending from the natural high ground towards the lowest point within the basin. This configuration provides several operational advantages: the pump intake can be positioned to draw from optimal depths below the water surface to minimize sediment uptake while avoiding floating debris, the causeway can be progressively extended as the facility expands, and the diesel power source ensures pumping capability independent of electrical infrastructure that may be compromised during extreme weather events. Due consideration has been made of the head requirements to deliver water to the process plant and water treatment plant.

The retracting causeway will be constructed using free-draining rockfill material placed in progressive lifts as the tailings surface rises.

Underdrainage Seepage Collection System

The underdrainage system serves the critical function of removing water that percolates through the tailings mass, thereby reducing pore pressures in the tailings and lowering the phreatic surface within the embankment structure. This system will be installed directly above the basin clay liner.

The drainage network employs a herringbone configuration. This system comprises:

- ***Branch Drains***: Perforated high-density polyethylene (HDPE) pipe installed in parallel arrays across the basin floor. The perforations will allow seepage water to enter while providing sufficient structural integrity to resist crushing under tailings loads. The pipe will be embedded in clean, free-draining granular material and a geotextile wrap that serves as a filter to prevent fine tailings particles from entering and potentially clogging the drainage system.

- ***Collector Drains***: Solid HDPE pipe positioned at an angle to the branch drains, receiving flow from multiple branches and conveying it to collection sumps. Solid pipes are specified for collectors to maximize flow capacity and structural integrity. The collector gradient will be adequate to maintain self-cleansing velocities under anticipated flow conditions.

Drainage pipe connections will be properly sealed using appropriate HDPE fittings and fusion welding techniques to prevent separation. The drainage network will be installed with appropriate cover material to protect against damage during tailings deposition operations.

Seepage collected by the underdrainage network will discharge into a dug-out sump constructed at a low point of the facility basin along the Main embankments. The sump will be equipped with submersible pumps designed for continuous operation in sediment-laden water. Collected water will be pumped to the supernatant pond via steel riser pipes running along the upstream slope of the Main embankment.

Groundwater Drainage and Leak Detection System

A secondary drainage network will be installed beneath the basin liner to serve the dual purpose of foundation drainage and leak detection. This system will monitor liner performance and detect any seepage through the liner system that might indicate damage or degradation.

The groundwater drainage network mirrors the underdrainage design, employing a similar herringbone pattern of branch and collector drains. However, this system will be positioned in the foundation material below the clay liner and be isolated from the underdrainage system.

Flow from the groundwater drainage system will be directed to a separate dug-out sump along the Main embankments.

The groundwater drainage sump will be equipped with submersible pumps capable of handling anticipated groundwater inflows. Collected water will be pumped to the supernatant pond via steel riser pipes running along the upstream slope of the main embankment.

Secondary Confinement System

Seepage emerging at the downstream toe of the embankment will be captured by a secondary confinement system designed to prevent uncontrolled release to the environment. This system comprises a trapezoidal channel constructed parallel to the embankment toe, lined with HDPE geomembrane to prevent infiltration.

The channel is designed with sufficient capacity to convey seepage flows plus surface runoff from the embankment face during rainfall events. The channel gradient will direct flows to a centralized concrete sump. The sump will be equipped with a submersible pump system to return captured seepage to the facility supernatant pond.

Non- Contact Water Segregation

Non-Contact Water consists of precipitation falling on areas outside the active tailings deposition zone that can be diverted around the facility without contacting tailings material. This includes rainfall runoff from undisturbed and disturbed catchment areas upstream of the facility, precipitation on perimeter access roads and berms, and natural drainage from adjacent slopes.

The physical separation of these water streams is achieved through strategically positioned diversion channels, liner systems, and controlled collection points. The following systems are designed to manage contact water:

- Perimeter Diversion Channels with Discharge Provision and Erosion Control.
  - Perimeter diversion channels will be constructed along the upstream limits of the facility to intercept natural runoff from contributing catchments and direct it around the TSF.
  - These channels are trapezoidal in cross-section, designed to convey the peak flow from the design storm event (1 in 1000year 24 hour storm event) with adequate freeboard. The channels will follow the natural topographic contours where practical to minimize earthwork requirements while maintaining sufficient gradient for self-cleansing velocities.

18.6.9 Emergency Spillway

The emergency spillway for both the TSF1 and TSF 2 will be located at the southwestern abutment and the northeastern abutment respectively of both main embankments.

The emergency spillways are designed as broad-crested rectangular weir out-falling into a trapezoidal chute and into an emergency pond. Details of the TSF 1 and TSF 2 emergency spillway design are presented in Table 18.6.3. The emergency spillway for both TSF1 and TSF 2 are designed to be lined with 1.5 mm thick HDPE geomembrane liner to control the effect of erosion. The design for the section of the spillway crest to be used for vehicular access has been modified to be lined with geotextile fabric and a 450 mm thick lateritic fill to enhance vehicular movement along this section of the spillway crest.

Table 18.6.3 Emergency Spillway Design

Characteristic	Values	
Spillway Crest	**TSF 1**	**TSF 2**
Spillway depth (m)	1.0	1
Inlet elevation (mRL)	174	172
Spillway crest length (m)	10.0	10
Spillway crest width (m)	39.0	10
Spillway crest side slopes (V:H)	1:6	1:6
Longitudinal slope for the spillway crest (%)	2	2
Spillway Chute 1		
Spillway chute inlet elevation (mRL)	173	172
Spillway chute width (m)	10	10
Spillway chute depth (m)	1	1
Spillway chute side slopes (V:H)	1:2	1:2
Spillway chute 1 length (m)	172	357
Spillway chute 1 outlet elevation (mRL)	126	155
Longitudinal slope for the spillway chute 1 (%)	27.59	5.32
Spillway Chute 2		
Spillway chute inlet elevation (mRL)	126	155
Spillway chute width (m)	10	10
Spillway chute depth (m)	1	1
Spillway chute side slopes (V:H)	1:2	1:2
Spillway chute 2 length (m)	189	272
Spillway chute 2 outlet elevation (mRL)	110	120
Longitudinal slope for the spillway chute 2 (%)	8.42	12.80

An emergency pond that collects flows from the spillway will also be constructed during Stage 1. The emergency ponds would provide nominal retention of overflow from the TSFs in compliance with L.I. 2182. In addition, it would also help reduce the potential for erosion at the spillway outlet.

Details of the emergency ponds design are presented in Table 18.6.4.

The emergency ponds for both TSFs will comprise dug-out sumps lined with geotextile fabric and subsequently lined with HDPE geomembrane liner. A 1 m high safety berm is to be constructed around both ponds. This is to provide additional freeboard for the ponds. The ponds are to be equipped with a 10m wide spillway to allow release of direct rainfall runoff (non-contact water).

Table 18.6.4 Emergency Pond Characteristics

Characteristic	Value
Base Dimension (width x length)	20m x 20m
Top Dimension (width x length)	20m x 20m
Depth	3.0m
Emergency pond spillway width	10m
Side slopes	1V:2H
Safety berm height	1.0m

18.6.10 Tailings Delivery and Distribution System

The tailings delivery and distribution system is designed to convey process plant tailings slurry from the mill discharge point to designated deposition locations within the TSF. The system is designed to provide operational flexibility, reliable performance, and controlled placement of tailings to maintain facility stability and optimize storage capacity.

The delivery pipeline will run along the access road linking the process plant to the TSFs and tee-off at the entry to the TSFs to a distribution manifold system to run clockwise and counterclockwise along the TSF perimeter roads and embankment crests.

The distribution manifold system will direct flow to multiple discharge points along the HDPE pipeline distribution system along the perimeter embankment. The distribution network will consist of:

- Valves allowing flow direction control between multiple spigot locations.
- Spigot discharge points spaced at approximately 25-meter intervals around the active deposition area.
- Isolation valves enabling maintenance and operational flexibility.

The distribution system is designed for progressive sub aerial upstream beach development.

Tailings will be discharged via spigotting methods from the perimeter embankment, promoting sub aerial beach formation and natural segregation of particles. Spigot locations will be rotated systematically to achieve:

- Uniform beach profile development.
- Optimal pool management and freeboard maintenance.
- Controlled rate of rise.
- Maximum particle size retention near discharge points.

18.7 TSF to Plant Site Access Road

18.7.1 Design Standard and Criteria

The Ghana Highway Authority (GHA) and Ministry of Roads and Highways design manuals were adopted for the design. Reference was also made to the Minerals and Mining (Health, Safety and Technical Regulations, 2012) (L.I. 2182).

The designs comply with the aforementioned design standards and aim at ensuring that adequate levels of safety are achieved whilst taking into consideration driver safety and cost efficiency.

The following main criteria were used in the design:

- Terrain and functional characteristics.
- Vehicle characteristics.
- Design speed.

18.7.2 Vehicle Characteristics

The design vehicles and equipment adopted for the designs are:

- Crane truck
- CAT D9 dozer with the following key dimensions (for cross section, alignment and pavement designs):
  - Overall length (with SU-Blade and Ripper): 8.22 meters
  - Width (over trunnions): 3.31 meters
  - Width (with SU blade): 4.35 meters

- Height (to top of cab / ROPS): 4.00 meters
- Ground clearance: 0.46 meters
- Operating weight: 49,900 - 50,500 kg (49.9 - 50.5 tonnes).

- 20 m^3 tipper truck with design height of 3.5 m, width of 2.5 m and length of 10.0 m (for alignment designs).

18.7.3 Design Speed

According to the Road Design Guide (GHA, 1991b), the design speed for this terrain and functional classification is 40 km/h.

18.7.4 Design Elements

Road Cross Sections and Road Prism

Details of the typical cross-sections for the access roads are presented in Table 18.7.1 in compliance with the Road Design Guide (Ghana Highway Authority, 1991), Minerals and Mining (Health, Safety and Technical Regulations, 2012) (L.I. 2182) and best practices.

Table 18.7.1 Cross Sectional Dimensions

Sections	Road Width Main Access Road m	Road Width On Top of TSF	Safety Berms m	Cross Slope of Carriageway %
Access 1 – TSF 1	10.00	6.658	2.00	4.00
Access 2 – TSF 2	10.00	6.658	2.00	4.00

The desirable lane width according to GHA Road Design Guide is 3.0 m.

According to the Minerals and Mining (Health, Safety and Technical Regulations, 2012) (L.I. 2182), for haulage roads, the desired travel width is 2.5 x the width of the design vehicle. For a CAT D9 Bulldozer it is 2.5 x 4.35m = 10.875 m i.e. lane width of 5.44 m.

Considering the nature of the terrain, low level of traffic and the drop-offs greater than three meters, a lane width of five meters have been adopted. Berms (safety bunds) have been proposed where there is a drop-off greater than three metres from the road. In accordance with Minerals and Mining (Health, Safety and Technical Regulations, 2012) (L.I. 2182), the height of the berm have been proposed to be 75% of the height of the largest tyre of the vehicle and equipment used on the road.

The road width proposed for the access on top of the TSF is narrower due to its special usage.

Road signs have been provided to enhance safe usage of the access roads.

Horizontal and vertical alignment

The horizontal and vertical alignments for the Project's roads have been designed with guidance from the Road Design Guide (GHA, 1991b) and shown in Table 18.7.2.

Table 18.7.2 Horizontal and Vertical Alignment Criteria

Design criteria	Service Road
Design speed (km/h)	40
Minimum curve radius – desirable	100
Minimum curve radius – absolute	50
Minimum curve length	70
Maximum superelevation (e) (%)	
Absolute	9
Desirable	5
Curve radius(m) where superelevation is not required	≥ 800
Min longitudinal gradient (%)	0.4
Limit length (m) for non-standard gradients	400 (8%)
	300 (9%)
	200 (10%)

The design sort to minimize sharp bends for safety and vehicle maneuverability, balance cut-and-fill as much as possible to reduce earthworks and ensure gradients are manageable for vehicles and equipment.

In situations where the terrain does not support horizontal and vertical designs that comply with road design standards and desirable criteria, permissible deviations from design standards while focusing on maintaining safety were adopted.

Pavement Structure

The Caterpillar D9 bulldozer, with an operating weight of approximately 50 tonnes, track gauge of 2.25 m, and blade width up to 4.35 m, represents a heavy tracked vehicle that imposes significant stresses on roads. The pavement structure must therefore be designed to resist rutting, shear failure, and settlement, while ensuring safe and efficient operations.

This proposal outlines a pavement structure for the TSF access road intended to support the design vehicle and construction traffic. In the absence of detailed geotechnical investigation, the design adopts conservative assumptions and relies on standard practices, ensuring durability, safety, and cost-effectiveness while allowing for future adjustments once soil data becomes available during detailed design phase.

- Design Basis:
  - Design Vehicle: CAT D9 Bulldozer (≈ 50 tonnes operating weight)
  - Assumptions: Subgrade soil is of moderate strength and requires compaction but no stabilization
  - Soil Data: No geotechnical investigation available; conservative assumptions applied
  - Traffic: Frequent movement of tracked equipment and heavy trucks
  - Materials: Locally available aggregates (laterite, crushed stone, gravel) to minimize cost and ensure timely construction
  - Proposed pavement structure: main access road.
- Pavement Structure – Main Access Road:
  - Subgrade
    - Compacted native soil
    - Minimum CBR assumed: 5–8%
    - Stabilization with lime / cement if weak zones are encountered
  - Sub-base
    - Granular material (laterite)
    - Thickness: 450 mm (conservative to account for unknown soil strength)
  - Base Course
    - Granular material (laterite)
    - Thickness: 300 mm (conservative to account for unknown soil strength)
    - Provides main structural strength against bulldozer track pressures

Granular base allows easier maintenance and reworking compared to stabilized bases

- Wearing Course

Crushed stone

Thickness: 150 mm

Designed for abrasion resistance and dust control track movement

Proposed pavement structure: Access road on top of TSF

- Pavement Structure – Access Road on Top of TSF:
  - Subgrade / Sub-base / Base Layer (Single Layer)

Compacted suitable material used for the embankment of TSFs

  - Wearing Course

Crushed stone

Thickness: 150 mm

Designed for abrasion resistance and dust control track movement

- Design Considerations:
  - Track Pressure: The D9's wide track gauge distributes load, but concentrated stresses still require robust granular base and sub-base layers
  - Drainage: Essential to prevent softening of subgrade under heavy tracked loads
  - Maintenance: Granular base facilitates re-grading and compaction, making it suitable for haul and similar roads with frequent bulldozer and other heavy equipment traffic
  - Safety: Adequate width to accommodate blade width (≈ 4.35 m) and manoeuvring space.

Although geotechnical data is unavailable, this conservative pavement structure provides a robust foundation for haul road operations. The design prioritizes safety, durability, and adaptability, ensuring reliable performance under heavy traffic while allowing for future refinements once soil investigations are conducted.

18.7.5 Road Drainage Design

- Design Basis:

- Engineering Application: Rational Method using rainfall intensity (i) from IDF curve corresponding to time of concentration (Tc) and desired return period
- IDF Curve Data for project region: Short duration and intensive storms were assumed since the Project is located in a wet region of Ghana, receiving periods of heavy rainfall throughout the year.

  Return Period: 10-year storm

  Rainfall Intensity (i): 90 mm/hr
- Runoff Coefficient (C): 0.25 (for unpaved rural area)
- Freeboard: 0.3 m
- Velocity control: Keep flow velocity between 0.6 - 3.0 m/s to avoid erosion or sedimentation as much as possible
- Adopted Threshold for catchment:

  Small area (from <500 m2 to 1000 m^2): To be handled by proposed side drains or surface flow, not a culvert.

  Typical design range: Culverts were considered for catchments greater than or equal to 1,000 m^2 (0.001 km^2) only.

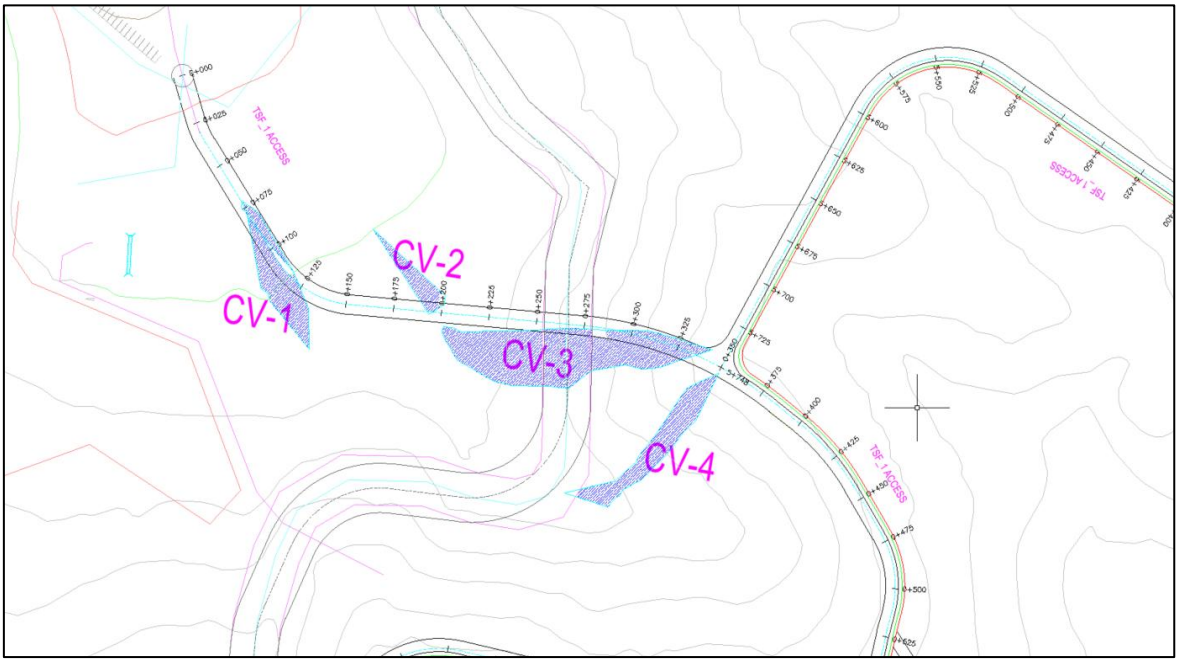
• Catchments

Table 18.7.3 below includes the identified catchments along the main access road. Culverts were considered for only catchments with area greater than or equal to 1,000 m^2 (0.001 km^2).

Table 18.7.3 Identified Catchments Along Main Access Road

Road	Chainage	Catchment ID	Area m^2	Area km^2
TSF 1_Access	0+115	CV-1	919.355	0.00092
TSF 1_Access	0+193	CV-2	336.944	0.00034
TSF 1_Access	0+340	CV-3	2778.083	0.00278
TSF 1_Access	0+350	CV-4	932.058	0.00093

Figure 18.7.1 Identified Catchment Areas



- Design Outputs

Table 18.7.4 and Table 18.7.5 include the design input and outputs for each catchment area. Even though 600 mm pipe culvert has the capacity for the catchment area, 900 mm pipe culvert is recommended for the following reasons:

- Safety Factor in Design Standards
  - Ghana Highway Authority specifies minimum culvert sizes of 900 mm for durability and maintenance access, regardless of hydraulic calculations.
- Blockage and Maintenance Margin
  - Smaller pipes clog more easily with debris, silt, or vegetation.
  - A 900 mm pipe provides extra clearance, reducing maintenance frequency.
- Future Capacity and Climate Resilience
  - Rainfall intensities are increasing in Ghana and globally.
  - Upsizing provides resilience against larger-than-expected storms.

- Construction Tolerances
  - Field conditions (poor bedding, misalignment, sedimentation) reduce effective capacity.
  - Larger pipes compensate for these uncertainties.
- Hydraulic Headwater Control
  - A bigger pipe lowers upstream ponding levels, protecting the road embankment.
- Ease of Inspection and Cleaning
  - Workers can physically enter a 900 mm pipe for inspection and cleaning, while a smaller 600 mm pipe is too small for safe access.

Table 18.7.4 Estimation of Peak Runoff for each Catchment

	Catchment ID	CV-3
1	Runoff coefficient (dimensionless, depends on land use / cover) [C]	0.25
2	Rainfall intensity (mm/hr) for a duration equal to the time of concentration [I]	90
3	Catchment Area in Km^2 [A]	0.0028
4	Unit conversion factor (from mm/hr·km^2 to m^3/s) [0.278]	0.278
5	Return Period (10, 25, 50 years)	10
6	Peak Runoff Q (m^3/s) = 0.278 C*I*A	0.0174

Table 18.7.5 Culvert Size Selection and Capacity Check for each Catchment

	Catchment ID	CV-3
1	Peak Runoff (calculated Q)	0.0174
2	Drain Type	Concrete
3	Manning's roughness coefficient (0.015 for concrete) [n]	0.015
4	Pipe Culvert Size (mm)	600
5	Cross-sectional area of flow (m^2) [A]	0.2827
6	Hydraulic radius = A / wetted perimeter (m) [R]	0.15
7	Slope of channel (m/m) [S]	0.001
8	Capacity check [Q]	0.1683

18.8 Mine-Wide Water Balance

The water balance model considers all TSFs, Waste Dumps, Open Pits, and the Process Plant. These facilities are interconnected as part of the mine waste and water management strategy and will collectively support ongoing and future tailings deposition. While each facility operates at different stages of the mine life, the water balance reflects their dynamic and coordinated roles within the overall management system. The following objectives were addressed as part of the study:

- Estimate whether the water demands of the site can be reliably met through the TSF system under normal and stressed conditions.
- Evaluate the likelihood and magnitude of water surpluses and deficits under a range of historical climatic conditions.
- Identify operational or design-level solutions to manage water surpluses and/or deficits.
- Provide guidelines, such as pumping rates and required storage volumes, to allow stable and efficient system operations.
- Verify whether the site's water storage systems meet the design criteria for storm events.

The water balance estimated water quantities during one year of construction and the operations phase only. Water quality and closure as well as post-closure analyses were not performed as part of the PFS. This mine water balance considers the process plant water flows and volumes, as well as the available make-up water sources. Also, it considers two stages (TSF1 and TSF2) of development of the project; if changes or different stages are considered, it is recommended to develop a site-wide water balance that includes additional site facilities such as new dams, open pits, and ponds, to improve overall water management. The battery limits of the model were limited to the reclaim pipelines from the TSFs, open pits, and waste dumps to the process plant, and the tailings slurry delivery pipelines to the TSFs.

18.8.1 Water Circuit Description and Water Management Strategy

The analyses for the PFS focused on the following key components and operational phases:

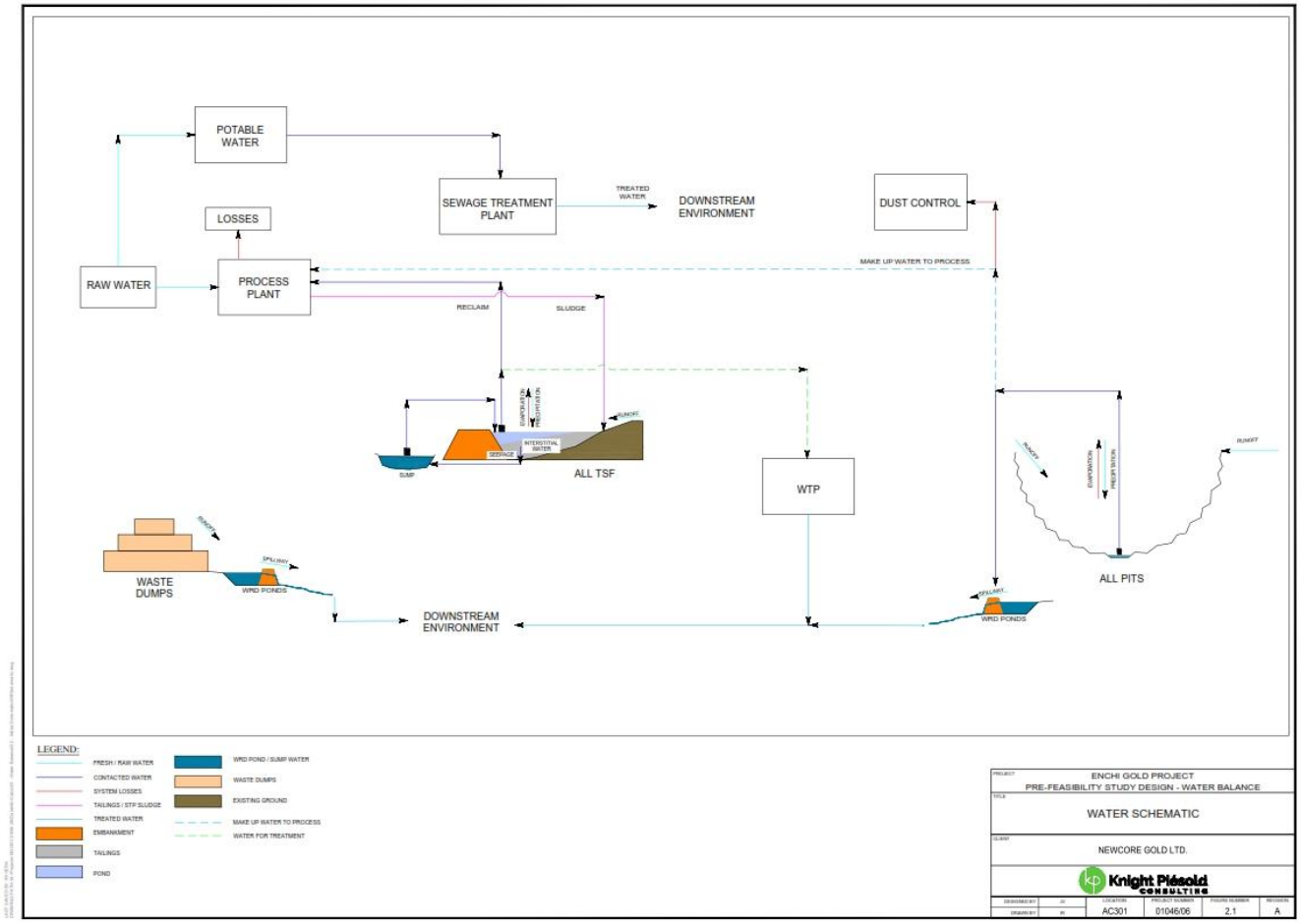
- Tailings slurry from the process plant is discharged to the active TSF initially the TSF1 from month 1 to month 60, and subsequently to the TSF2 from month 61 to month 120. Each TSF operates with a single spillway elevation throughout its operational period.
- Reclaim water for the process plant is managed through two independent systems. The system that retrieves reclaim water from the active TSF's Pond; and in parallel, a separate reclaim system to pump water directly from the open pits if needed.

- All TSFs and Open Pits account for natural hydrologic inputs and outputs, including direct precipitation, evaporation, and surface runoff. Make-up water from open pits is used only as a supplemental source to cover shortfalls during operations. The specific inflows to Water Treatment Pant are not defined at this stage, and a constant treatment rate of 150 m^3/hr is assumed. A site-wide water balance is recommended for future project stages to properly account for all such site water management interactions.

The operational water balance schematic is shown in Figure 18.7.1. Overall, the water management strategy is designed to maximize water recirculation within the project boundaries, minimize reliance on external make-up water, and maintain appropriate dam safety conditions. The project water circuit integrates multiple sources, storage components, and reuse streams to support the process plant while minimizing freshwater intake and environmental discharge.

Fresh and raw water is supplied to the system and used primarily for reagent preparation and dilution in the process plant, and as process plant make-up water. Additional water inputs include direct precipitation over all site facilities, such as pits, waste rock dumps (WRDs), the tailings storage facility (TSF), and other infrastructure areas.

Figure 18.8.1 Operational Water Balance Schematic



Runoff generated from precipitation is collected across the site. All runoff and direct precipitation captured within the TSF is collected and recirculated to the process plant as reclaim water. Runoff from the WRDs is routed to sedimentation ponds for controlled discharge to the environment. Similarly, direct precipitation and runoff generated within the open pits, as well as dewatering flows, are pumped to sedimentation ponds for controlled discharge. A dedicated pipeline is also available to pump this water to the process plant when a water deficit requires additional supply.

The TSF receives tailings slurry from the process plant, where solids settle and water is stored in the tailings pond. Supernatant water is reclaimed and returned to the process plant, reducing the demand for fresh water. Seepage within the TSF is also accounted for in the water balance. Seepage is assumed to be fully collected and pumped back to the system; therefore, at the model time step it is treated as a net-zero component.

Treated water from the WTP and the sewage treatment plant (STP) can be discharged to the downstream environment in compliance with regulatory standards or reused for site activities such as dust control.

System losses occur through evaporation from exposed water surfaces (e.g., the TSF pond and sedimentation ponds), through operational uses such as dust suppression, and through interstitial water retention within the TSF (i.e., water stored in the tailings mass), which are represented in the water balance model.

Overall, the water circuit is designed to maximize water recovery and reuse within the system, minimize freshwater consumption, and ensure that any discharge to the downstream environment meets environmental requirements.

The Project has several exploration targets identified across the property, four of which have a Mineral Resource Estimate and as such have been evaluated for the open pit mineral resources, pit optimization and production scheduling. The deposits with mineral resource estimates (MRE) for the open pit mining are Boin, Sewum, Nyam and Kwakyekrom.

Based on the FS production estimates, the Boin, Sewum, Nyam, and Kwakyekrom open pits have been included in the water management circuit, as illustrated in Figure 18.6.1. Reclaimed water from the open pits may be utilized for dust suppression, pumped to the Process Plant, or discharged to the downstream receiving environment.

Waste Rock Facilities (waste dumps)

Waste rock will be transported to one of the Waste Rock Facilities (WRFs) located in close proximity to the open pits. The development of the WRFs has been sequenced to optimize haul distances throughout the different stages of the Project's mine life.

The WRF water management consists of collecting runoff from the WRFs and conveying it to sedimentation sumps (or ponds) prior to controlled discharge to the environment.

18.8.2 Design Criteria, Inputs and Assumptions

The design criteria, input parameters, and assumptions that have been applied to the water balance model include:

- Climatic / Hydrologic Parameters: Key parameters include rainfall, evaporation, and runoff characteristics. The Enchi dataset provides a 30 year continuous monthly rainfall record from 1992 through 2022. The 100 year/24-hr storm event depth was estimated using the site design rainfall record and applied to the water balance. The 24 hour PMP storm event depth was obtained using the methodology developed by Hershfield (WMO, 2009) with a value of 692 mm. This is 2.25 higher than the 1,000 yr/24-hr storm for reference

- Stations and Associated Data: A combination of on-site and off-site climate stations was reviewed to develop the climatic dataset for the water balance model. Climate data were primarily provided by Newcore and supplemented with publicly available sources. In total, seven stations were assessed, including three on-site rain gauges (Boin, Sewum, Nyam) and four regional / reference datasets or stations. The available records vary substantially in period of record and completeness: the on-site datasets cover 2023–2025 with ~74–77% daily completeness, whereas key off-site measured stations provide longer and more complete records (e.g., Tarkwa: 1988–2025, 97% complete; Enchi: 1992–2022, 94.2% complete). In addition, gridded / satellite products (CHIRPS and MERRA; 1981–2025, 100% complete) were considered as supplementary reference datasets.

- Operational parameters: Operational input parameters used in the water balance model were primarily based on data provided by Newcore, Lycopodium, and Knight Piésold Consulting (KP) through a collaborative data review. The model assumes continuous mill operation (24 hours per day, 365 days per year) and a total process plant water demand ranging from 633 to 1,154 cubic meters per hour (m^3/h), as provided by Lycopodium. Over the modelling period, the demand is assumed to be 1,154 m^3/h during Months 1–12 and 633 m^3/h from Month 13 through Month 120.

A constant water treatment rate of 150 m^3/h was assumed throughout the simulation, even when additional treatment capacity may be available, to limit water accumulation within the system.

To represent changing basin geometries over time, dynamic elevation–volume relationships for both TSF1 and TSF2 were utilized based on tailings deposition modeling. These relationships were incorporated to simulate variations in pond capacity and geometry throughout the project life. The water storage design criterion for TSF1 and TSF2 (i.e., the required water storage volume) is defined as the storage volume corresponding to the simulated monthly average pond volume from the water balance plus the 100 year, 24 hour design storm event volume, while maintaining the pond water level at least 1 m below the dam crest.

The following assumptions were considered and incorporated during the model development:

- The off-site Enchi climate station was selected as the representative rainfall dataset, as per agreement with NGL, based on their confidence in the station's data reliability and accuracy.

- Missing rainfall values were infilled using a double-gamma Monte Carlo synthetic generation approach. The infilling procedure was parameterized using the statistics of the measured record to preserve the observed rainfall distribution and maintain consistency with site conditions.
- TSF1 was assumed to be operational one year prior to the start of tailings deposition to allow accumulation of water required for process plant commissioning and start-up.
- A constant water treatment rate of 150 m^3/h was assumed from Month 13 (start of TSF1 deposition) onward.
- Once tailings deposition commenced in TSF2, deposition in TSF1 was assumed to cease. All water accumulating in TSF1 due to climatic inputs and consolidation (porewater release) was assumed to be pumped to TSF2 as an interim measure while progressive closure options for TSF1 are developed and implemented.
- A tailings dry density of 1.3 t/m^3 was assumed.
- A spillway was assumed to be constructed only in the final stage of each TSF; however, compliance with Ghanaian freeboard requirements was maintained throughout operations.
- The model assumes constant production and tailings parameters over defined time periods (step changes), consistent with the project production schedule.
- The water balance was simulated using a monthly time step; therefore, daily variability and short-duration operational fluctuations are not explicitly captured.
- Tailings slurry solids content was assumed to be 32.8% in the early years, increasing to 46.7% in later years.
- Runoff from waste rock dumps (WRDs) was assumed not to enter the TSF water circuit; routing WRD runoff to sedimentation facilities was assumed to provide sufficient treatment prior to controlled discharge to the environment.

18.8.3 Results

The results of the monthly water balance analyses performed for the TSFs based on the following:

- Timeframe of analysis: Month 1 (start of TSF1 construction) through Month 86 (end of operations of TSF1), then from Month 87 (start TSF2 tailings deposition) through Month 132 (end of operations of TSF 2).
- The operations water balance schematic presented on Figure 18.6.1.

- Deterministic results: Presented as minimums (Det. Min.), averages (Det. Avg.), and maximums (Det. Max.). These results provide an indication of how the water circuit will be affected by the historical climatic / hydrologic records.

- Tailings Storage Facility 1 (TSF1) Pond.

The results indicate that TSF1 pond volumes will vary according to seasonal fluctuations due to climatic effects. The average pond volumes during the construction and stage 1 of tailings deposition are predicted to range from 0.0 million cubic meters (Mm^3) to 1.25 Mm^3 with an overall average of 0.62 Mm^3 from Month 1 through Month 34. For stages 2 through 5, average pond volumes range from 0.0 Mm^3 (no pond) to 0.43 Mm^3 with an overall average of 0.12 Mm^3 from Month 35 through 86. Finally, when TSF1 becomes inactive the average pond volumes range from 0.0 Mm^3 (no pond) to 0.47 Mm^3 with an overall average of 0.06 Mm^3 from Month 87 through 132.

The predicted TSF1 water reclaim flowrates, and the pumping capacity indicate an average reclaim rates ranging from 1,150 m^3/h to 600 m^3/h during operational conditions (Month 13 through Month 86). For the majority of the TSF1 stages there is exceedance of reclaim that goes to the water treatment plant. When TSF1 becomes inactive the reclaim water is transferred to TSF2.

Predicted deficit flowrates from TSF1 to Plant indicate that the average maximum deficit flowrate is 189 m^3/hr and that the maximum deficit after dewatering is 50.6 m^3/hr with an overall average of 6.0 m^3/hr.

Tailings Storage Facility 2 (TSF2) Pond

The results indicate the TSF2 pond volumes will vary according to seasonal fluctuations due to climatic effects. The average pond volumes during stage 1 of tailings deposition are predicted to range from 0.0 million cubic meters (Mm^3) to 0.69 Mm^3 with an overall average of 0.33 Mm^3 from Month 87 through Month 132.

The predicted TSF2 water reclaim flowrates, and the pumping capacity indicate the average reclaim rates ranges around 600 m^3/h during operational conditions (Month 87 through Month 132). For all TSF2 stages there is exceedance of reclaim that goes to treatment.

Predicted deficit flowrates from TSF2 to Plant indicate that the average maximum deficit flowrate is 195 m^3/hr and that the maximum deficit after dewatering is 83.2 m^3/hr with an overall average of 16.0 m^3/hr.

Overall, the predicted pit dewatering flowrates including the construction year and subsequent operational stages of TSF1 and TSF2 exhibits a cyclical pattern, reflecting seasonal variability, with recurring peaks and troughs throughout the years. During the early operational phase (post-construction and TSF1 Stage 1), total flowrates increase rapidly, reaching peak values of approximately 300–350 m^3/hr, primarily driven by contributions from the Boin Pit.

As the Project progresses through TSF1 Stages 2 to 4, peak dewatering rates continue to rise, reaching maximum values close to 500 m^3/hr around the transition between TSF1 Stage 3 and Stage 4. This period represents the highest overall dewatering demand in the model. Individual pit contributions show distinct temporal patterns. The Boin Pit dominates the early and mid-stage flows, while the Nyam Pit becomes a consistent secondary contributor during TSF1 operations, with moderate but stable flowrates. The Kwakyekrom Pit contributes intermittently with relatively lower flowrates. As the system transitions from TSF1 to TSF2, a noticeable decline in total dewatering flowrates occurs. This reduction is accompanied by a shift in contributing sources, with the Sewum North and Sewum Checkerboard pits becoming more prominent.

During TSF2 stages, total flowrates stabilize at lower levels, generally ranging between 60 and 130 m^3/hr, while maintaining the same seasonal variability pattern. Toward the end of the simulation period, dewatering flows gradually decrease and eventually approach zero, indicating the depletion or closure of active dewatering sources.

Water quality constraints were not considered within the scope of this study, and Knight Piésold Consulting. acknowledges that the assumption that, all available water sources are suitable for reclaim may not hold in practice. This should be evaluated in future project phases.

18.9 Mine-Wide Stormwater Management Plan

18.9.1 Structure Description

The proposed surface water management structures are designed to collect and safely discharge stormwater runoff downstream within the Boin pit basin area in a controlled manner. During the operational phase of the Project, these structures will serve to divert surface water away from the pits and waste dumps, mitigate sediment loads in discharged flows to ensure regulatory compliance, and safely convey stormwater at intersections with main access roads.

The following structures are proposed:

- Boin Pit TSF 1 and TSF 2 Area.
  - Boin pit diversions channels (secondary): trapezoidal section and riprap lining
  - Sediment ponds for pit dewatering flows: rectangular ponds near each pit and with side slopes 2H:1V and an overflow structure to discharge to downstream drainages
  - Sediment ponds for waste dumps runoff collection and solids removal before discharging at downstream drainage
  - Culverts at main access road and streams crossings: concrete box culvert to route the design storm event peak flow.

- Sewum Pit Area.
  - Sewum pit diversion channel (main): trapezoidal section and riprap lining
  - Sewum pit and water dump diversion channel (secondary): trapezoidal section and riprap lining
  - Sediment ponds for pit dewatering flows: rectangular ponds near north, central and south pits with side slopes 2H:1V and an overflow structure to discharge to downstream drainage
  - Sediment ponds for waste dumps runoff collection and solids removal before discharging at downstream drainage
  - Culvert at access road and stream crossings: concrete box culvert to route the design storm event peak flow.
- Nyam and Kwakyekrom Area.
  - Nyam waste dump and pit diversion channel (main): trapezoidal section with riprap lining
  - Diversion berms in the east side perimeter of the Nyam pit, Nyam waste dump, Kwakyekrom waste dump and south and west side perimeter of Kwakyekrom pit: trapezoidal cross section berm to divert runoff from the upstream catchment area to the downstream drainage at the north / northwest direction
  - Sediment ponds for pit dewatering flows: rectangular ponds near north, central and south pits with side slopes 2H:1V and an overflow structure to discharge to downstream drainage
  - Sediment ponds for waste dumps runoff collection and solids removal before discharging at downstream drainage
  - Culvert at access road and stream crossings: concrete box culvert to route the design storm event peak flow.

18.9.2 Design Criteria, Inputs and Assumptions

Heading Analyses

The contributing-area basin delineations for the project site surface stormwater management were estimated and applied to the hydrologic analyses for the structures:

- Main non-contact water diversion channels.

- Secondary non-contact water diversion channels.
- Culverts at possible crossings of the streams with the access road.
- Diversion berms around the Nyam and Kwakyekrom pits and waste dumps.

All channels (non-contact) are sized to convey (capacity) and withstand (erosion protection) the peak flows generated from the 100 year/24 hour storm event according to Ghana regulations: Minerals and Mining (Health, Safety and Technical Regulations) (L.I. 2182).

Site-specific hyetographs (i.e. temporal rainfall distributions) nor less than 24 hour precipitation records were not available at the time of the study. The storm depths were conservatively distributed over 24 hours according to the Huff 1st quartile hyetographs, which includes a peak intensity of 17% of the 24 hour rainfall depth occurring in 1.2 hours. Huff 1st quartile storm distribution represents heaviest rainfalls occurring in the first 25% of the duration being common in short-duration storms.

NRCS Curve Number and Time of Concentration were assumed based on NRCS (2004), and experience with similar projects in similar climates.

The time of concentration (T_c), for each representative subbasin was estimated according to equation 15-4b (NRCS, 2010).

Hydraulic Analyses

The following design criteria, inputs and assumptions were applied to the hydraulic analyses for the surface stormwater management structures:

- Design phase: operations / closure-post-closure.
- Construction / erosion protection materials.
  - Non-contact water diversion channels: Riprap

    Manning's roughness (n) coefficient values for the riprap were calculated based on the median riprap size (D50) and the normal flow depths in an iterative process

    Minimum factor of safety (FoS): 1.2
  - Diversion berms: unlined disturbed ground with riprap in the outer face
  - Sediment basins: geomembrane lining system.
- Bottom widths.
  - Non-contact diversion channels 1 and 7 (main): 3.0 m

  - Non-contact diversion channels 2, 3, 5 and 6 (secondary): 2.0 m
  - Non-contact diversion channels 4, 8 and 9 (secondary): 1.0 m.
- Side slopes: 2.0 horizontal to 1.0 vertical (2.0H:1.0V).
- Freeboard depth (between the design normal flow depth and channel crest).
  - Larger of 0.3 meters (m) or 25% of the design normal flow depth, per the United States Department of the Interior Office of Surface Mining (1982).
- Diversion berms.
  - A trapezoidal ridge geometry of compacted soil was proposed
  - Designed to collect and covey around the pits and waste dumps perimeter the runoff from upstream catchment area
  - Berm height estimation based on normal flow depth for peak flow associated with a 100-yr/24-hr storm event plus a freeboard with above stated criterion.
- Culverts.
  - Reach - access roads crossings 1-7

    Culverts length (approximated): 6 m to 31 m

    Material and type: concrete box

    Number of barrels: 1 and 2 depending on the peak flow from the 100-yr/24-hr storm event.

    Culverts slopes: based on existing ground elevations points before and after each crossing
- Sediment ponds.
  - Length / width ratio: 2
  - Basin side slopes: 2.0 horizontal to 1.0 vertical
  - Basin overflow weir

    Invert elevation: 0.5 meter below crest elevation

    Geometry: trapezoidal section with side slopes 2.0 horizontal to 1.0 vertical (2.0H:1.0V)

    Weir's bottom width: 0.5 m for waste dumps basins and pits basins.

Weir outflow design criteria: Peak average dewatering flow for pits sediment basins and 90th percentile stormwater depth for waste dumps sediment basin plus PMP / 24-hours storm event over the sediment basin surface area.

- Basins geometrical design was performed purely with a sediment removal rate criterion. This mean that surface area required to achieve a 90%+ rate is set as target but required volume (dead sediment storage plus water storage volume) will need to be refined where available data on sediment inflow concentration is estimated.

- Sediment size 0.125 mm and a density of 2.65 t/m^3 was assumed as initial values for the calculations on velocity settling required to estimate removal rates within sediment ponds.

- Sediment ponds inflows coming from the Site-Wide Water Balance accounts for direct precipitation, runoff, groundwater and evaporation.

- The 90th percentile rainfall depth of all rainy days (P>0 mm) in the complete precipitation record was chosen as the criteria for sizing (surface area estimation) the waste dumps sediment ponds.

18.9.3 Results

Hydrologic Analyses

The results of peak flows for each structure were estimated from representative subbasins modelled in HEC-HMS and unit area peak flow methodology:

- Non-contact diversion channel 1 (main): 20.3 m^3/s.
- Non-contact diversion channel 2 (secondary): 3.5 m^3/s.
- Non-contact diversion channel 3 (secondary): 2.2 m^3/s.
- Non-contact diversion channel 4 (secondary): 1.7 m^3/s.
- Non-contact diversion channel 5 (secondary): 5.4 m^3/s.
- Non-contact diversion channel 6 (secondary): 5.1 m^3/s.
- Non-contact diversion channel 7 (main): 12.1 m^3/s.
- Non-contact diversion channel 8 (secondary): 0.5 m^3/s.
- Non-contact diversion channel 9 (secondary): 0.4 m^3/s.
- Stream – access road culvert crossings: 1.67 m^3/s to 29.22 m^3/s.

- Open pits and Waste Dumps diversion berms: 0.40 m^3/s to 1.47 m^3/s.
- Open pit sediment basins: 0.012 m^3/s to 0.055 m^3/s.
- Waste dump typical sediment basins: 0.009 m^3/s.
- Culvert crossing 1-3: 29.22 m^3/s.
- Culvert crossing 4: 1.67 m^3/s.
- Culvert crossing 5: 4.46 m^3/s.
- Culvert crossing 6: 5.63 m^3/s.
- Culvert crossing 7: 18.27 m^3/s.
- Hydraulic Analyses.
- Diversion Channels, Diversion Berms and Culverts Crossing.

The following minimum channel depths (normal flow depth plus freeboard) were estimated per structure:

- Non-contact diversion channels 1 and 7 (main): from 1.7 m to 1.9 m.
- Non-contact diversion channel 2, 3, 5 and 6 (secondary): from 1.0 m to 1.3 m.
- Non-contact diversion channel 4, 8 and 9 (secondary): from 0.7 m to 1.0 m.
- Diversion berm: 0.8 m to 1.2 m.
- Culverts:
  - Headwater: 0.92 to 2.44 m
  - Normal depth: 0.4 m to 1.5 m.
- Sediment Ponds.

Pits sediment ponds were classified in three different sizes (small, medium and large) based on the peak design inflow and a typical sediment pond for waste dumps was designed (based on the one with greater inflow). The following dimensions were proposed for each one to achieve the required area for the target sediment and its desired removal rate:

- Pit small sediment ponds:

  - Peak design inflow: 0.012 m^3/s to 0.017 m^3/s
  - Large: 22 m
  - Width: 11 m
  - Surface area: 242 m.
- Pit medium sediment ponds:
  - Peak design inflow: 0.026 m^3/s to 0.036 m^3/s
  - Large: 18 m
  - Width: 9 m
  - Surface area: 162 m.
- Pit large sediment ponds:
  - Peak design inflow: 0.055 m^3/s
  - Large: 16 m
  - Width: 8 m
  - Surface area: 128 m.
- Waste dump typical sediment ponds:
  - Peak design inflow: 0.009 m^3/s
  - Large: 10 m
  - Width: 5 m
  - Surface area: 50.

18.10 Mine-Wide Hydrogeology

18.10.1 Hydrostratigraphy

The following hydrostratigraphic units exist across the Project site: Weathered Overburden (soil / regolith), Volcaniclastic / turbiditic sequence, fractured volcanic and intrusive rocks, graphitic phyllites and pelitic turbidites quartz veins, breccia and shear zones.

The site is therefore best classified as a fractured crystalline basement (fractured rock aquifer) and consolidated weathered material aquifer system, both of which have an overall low to moderate productivity, with the fractured rock aquifer being characterized by strong structural control and pronounced heterogeneity in hydraulic conductivity.

18.10.2 Groundwater Levels and Flow Patterns

A total of 271 boreholes from the Sewum, Boin, Kwakyekrom, Enchi, and Tokosea areas were compiled into a single groundwater level dataset. The raw dataset indicated some anomalies:

- Reported standing water levels >100 m below collar in boreholes where the maximum reach of the dipmenter (100 m) was exceeded; and
- Time series records for some boreholes showing water level variations over more than two orders of magnitude, far exceeding plausible piezometric responses in fractured bedrock systems.

For hydrogeological interpretation, analysis of datasets was restricted to the April 2024 data subset. This subset provides the most spatially extensive and internally consistent coverage and is considered to best approximate groundwater conditions at the end of the wet season.

The mapped west / southwest–to–northeast head gradient suggests that the aquifer system is not characterised by a single, laterally uniform water table. Instead, the groundwater reflects:

- Strong variations in groundwater levels associated with the weathered material aquifer, the transition zone, and the fractured rock aquifers (saprolite vs saprock vs fresh bedrock). The fracture network connectivity also impacts the groundwater level distribution;
- Groundwater flow patterns mimic topography, albeit at lesser gradients. This is expected in a greenfield project area such as this, where there are no impacts on groundwater levels and flow patterns due to large-scale mine dewatering; and
- Possible compartmentalization by major shear zones and lineaments, which may act either as preferential flow conduits or partial hydraulic barriers.

18.10.3 Hydrochemistry and Geochemistry

Hydrochemistry

- The groundwater analytical analysis was done to characterise baseline groundwater quality and assess its relevance to TSF operations and environmental compliance. Results were evaluated against international and national drinking-water benchmarks, specifically World Health Organization (WHO) drinking water guideline values and Ghanaian DGS 175 standards, to place the data in an appropriate regulatory and environmental context. On the basis of the December 2025 dataset, the following key observations are made:

- Generally, *pH* all five samples were within the range of approximately 6.5–8.5, fully compliant with both WHO and DGS 175 guideline ranges. This indicates circumneutral groundwater conditions with no evidence of acidification.
- *Electrical Conductivity (EC)*: EC values ranged from about 10 to 60 mS/m, indicating low mineralization and a fresh groundwater type.
- Corresponding TDS values are consistent with dilute, low-salinity water suitable for most operational uses, subject to site-specific process requirements.
- The summed concentrations of major anions (e.g. chloride, sulphate, bicarbonate / alkalinity) in all samples were below WHO and DGS 175 guideline limits.

- Overall, exceedances are confined to a small subset of parameters and wells and are best explained by natural hydrochemical reactions namely water–rock interaction, mineral weathering and local redox conditions rather than anthropogenic contamination, as expected in a green fields site such as this. Slightly elevated salinity and hardness (TDS, Ca and sum of cations) are restricted to HLF003 and HLF004, while elevated Fe, Al and turbidity occur at HLF002 and HLF005, consistent with groundwater evolving within mafic / intermediate volcanic and volcaniclastic units and locally reducing, mineralized zones.
- From a mine-water and environmental-management perspective, the groundwater is generally suitable for process use, subject to relatively simple treatment principally settling / filtration and, where required, iron and aluminium control. For potable or other sensitive end uses, standard treatment would be necessary at wells exhibiting elevated Fe, Al and turbidity. Importantly, these conclusions are based on a limited dataset focused on the TSF investigation area; at later study stages, the hydrochemical monitoring network should be expanded spatially across the broader project area to confirm these baseline patterns, capture any lateral variability, and strengthen the regional groundwater quality assessment.

From Figure 18.9.1, the Piper diagram shows that all five samples plot within a relatively narrow compositional range and define broadly similar hydrochemical facies, with modest but systematic variation between wells. A Ca–Mg–HCO_3 water type is typical of groundwater circulating through mafic to intermediate volcanic and volcaniclastic rocks under relatively low-salinity conditions, as expected at Enchi. The dominance of bicarbonate and divalent cations, together with low sulphate and modest chloride, indicates that groundwater chemistry is controlled primarily by natural water–rock interaction weathering of plagioclase, mafic minerals and minor carbonates within the weathered and fractured Birimian units.

Figure 18.10.1 Piper Diagram of the Boreholes

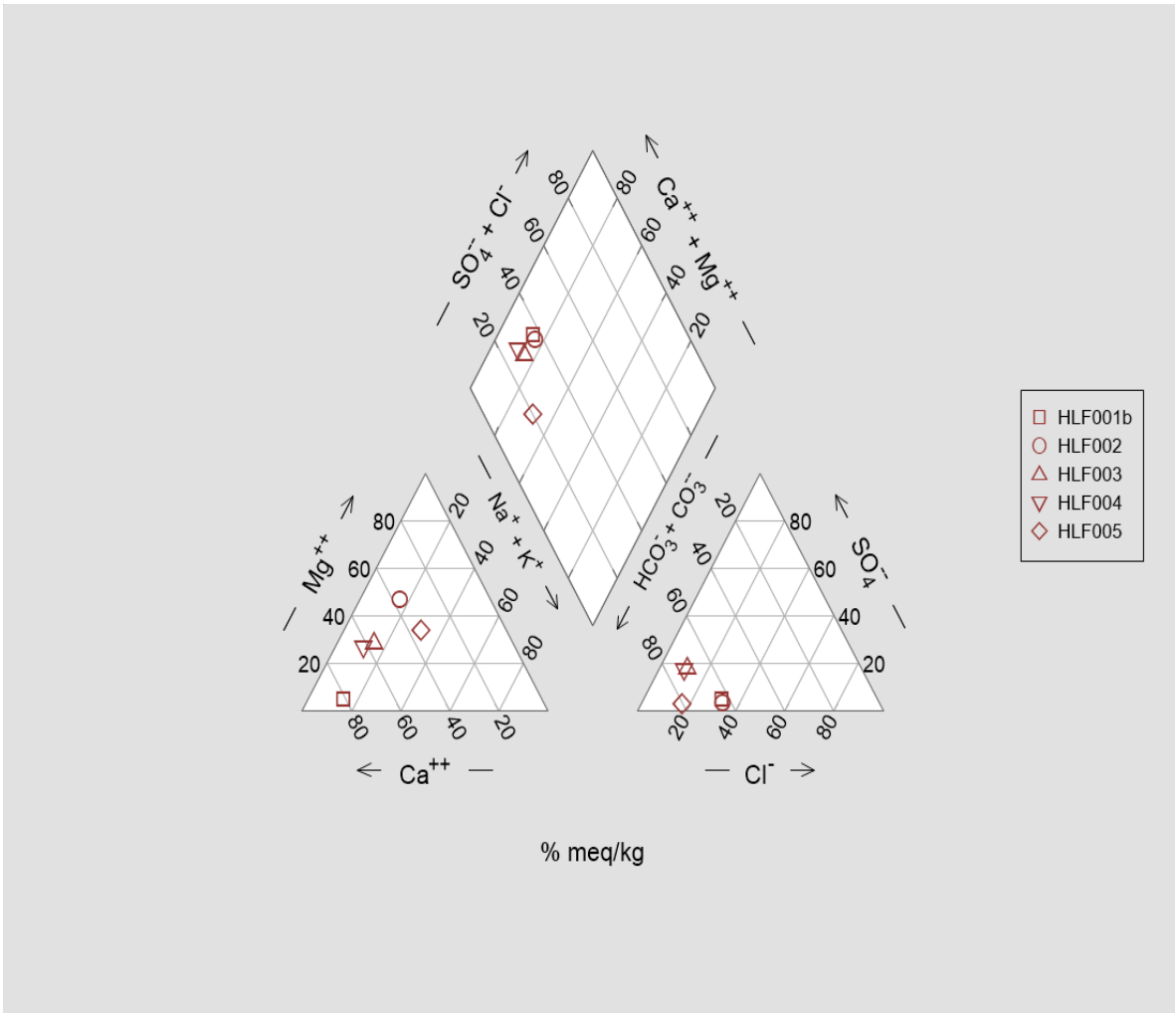


Table 18.10.1 summarizes the Langelier Saturation Index (LSI) values calculated for water samples collected in each sampled borehole studied.

Table 18.10.1 Langelier Saturation Index Results

Sample ID	pH	Temp ˚C	TDS mg/L	Ca mg/L	Hardness ($CaCO_3$)	Alkalinity	pHs	pH	LSI
					mg/L	mg/L			
HLF001b	6.6	25	74	20	50	21.78	8.77	6.6	-2.17
HLF002	7.4	25	138	16	40	31.29	8.69	7.4	-1.29
HLF003	8.2	25	305	60	150	91.28	7.53	8.2	0.67
HLF004	8.	25	315	70	175	99.08	7.53	8	0.47
HLF005	7.3	25	203	17	42.50	47.87	8.53	7.3	-1.23

Sample ID	pH	Temp ˚C	TDS mg/L	Ca mg/L	Hardness ($CaCO_3$) mg/L	Alkalinity mg/L	pHs	pH	LSI
LSI (Carrier) Indications:									
-2 to -0.5		-0.5 to 0		0		0 to 0.5	0.5 to 2		
Serious corrosion		Slightly corrosion but non-scale forming		Balanced but pitting corrosion possible		Slightly scale forming and corrosive	Scale forming but non-corrosive		

From a hydrogeological perspective, the LSI pattern reflects the natural geochemical evolution of groundwater along different flow paths and lithological domains:

- The corrosive, undersaturated waters (HLF001b, HLF002, HLF005) likely represent relatively younger or less mineralised groundwater with lower carbonate buffering capacity.
- The scale-forming, supersaturated waters (HLF003, HLF004) are consistent with more evolved groundwater in stronger contact with Ca-rich mafic / intermediate rocks, resulting in higher hardness and alkalinity.

For mine-water management and infrastructure design, this means that:

- At locations such as HLF001b, HLF002 and HLF005, corrosion control (e.g. material selection, potential pH / alkalinity adjustment) should be considered.
- At HLF003 and HLF004, scale control (periodic cleaning, use of scale-resistant materials, or chemical conditioning if used in closed-circuit systems) may be required, particularly for pipelines, heat exchangers or process equipment exposed to long residence times.

Overall, the LSI results confirm a natural, lithology-controlled range from corrosive to moderately scale-forming waters within the project area and provide a useful basis for selecting appropriate materials and water-treatment strategies for future mine development.

Geochemistry

The geochemical assessment is based on a review of available metallurgical test results. Although there is a wide range of test results available from test work done by Intertek, the University of Mines and Technology (UMaT), and BWI laboratories during the periods 2012 - 2013, and 2020 - 2025, most refer to the extractable gold content at various grades and from the different lithologies present. Metallurgical test results relevant to the environmental geochemical assessment include:

- Acid-base- accounting studies on Newcore Ore Samples: (Amankwah, 2024)

- Acid-base- accounting studies on Newcore Ore Samples (Batch 2): (Owusu (2025), Acid Mine Drainage Studies of Rock Samples (Batch 2) Acid-base- accounting studies on Newcore Ore Samples (Batch #3): (Owusu (2025), Acid Mine Drainage Studies of Rock Samples Batch 3) Excel spreadsheet summarising the results from the acid-base-accounting tests.

- Leaching of Batch 3 Samples with Rainwater: (Owusu (2025), Acid Mine Drainage Studies of Rock Samples (Batch 2) (Owusu, 2025 (Leaching of Batch 3 Samples with Rainwater (Owusu, Leaching of Batch 3 Samples with Rainwater, 2025)

- Minerals Metallurgy Report: (Intertek, 2023) (Intertek, 2023)

Acid-Base-Accounting Testing

In total three series of acid-base-accounting (ABA) tests were performed by UMaT:

- Batch 1: A total of 38 samples were submitted by Newcore for testing.

- Batch 2: A total of 24 rock samples were submitted by Newcore for testing.

- Batch 3: A total of 50 samples were submitted by Newcore for testing.

From the summarized results it can be seen that:

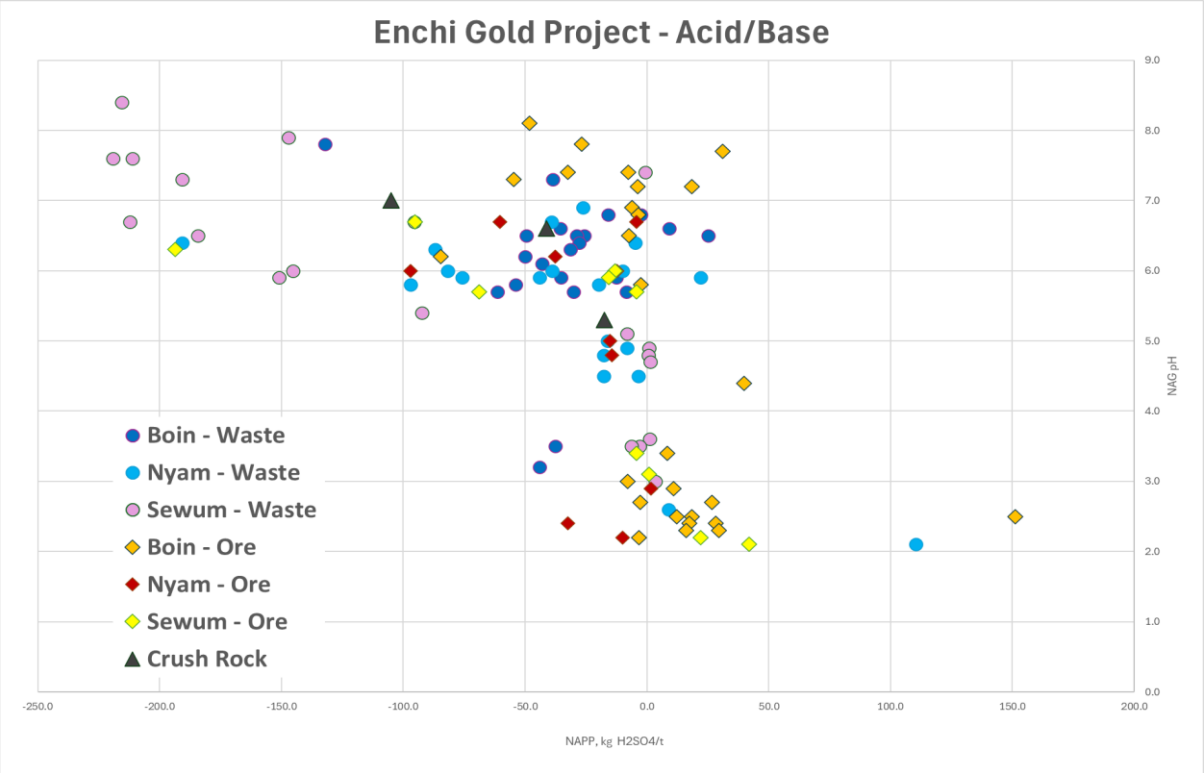
- Between 80 and 82% of the waste material from all three deposits (Nyam, Boin, Sewum) are classified as non-acid forming, with between 10 and 18% being classified as uncertain, and between 0 (Boin) and 10% being classified as potentially acid forming.

- Between 40 and 80% of the ore material is classified and non-acid forming, between 20 and 22% is classified as uncertain, and between 10 and 37% (Boin) is classified as potentially acid forming.

From the results it is concluded that both the waste material and the ore from the three different deposits could be acid forming, with the Boin deposit material being slightly anomalous to that from Nyam and Sewum. The ore material is interpreted to be more likely to form acid mine drainage (AMD) conditions than the waste material.

The NAPP and NAG pH as obtained from the geochemical testing for the different material (waste and ore) and the three deposits (Nyam, Boin, Sewum) are shown in Figure 18.10.3. The figure shows that, generally the ore material clusters at a NAG pH of less than 3.5 and with a NAPP between -50 and +50 H_2SO_4/t, while the waste material shows a NAG pH between 4.5 and 8.0 with a NAPP of between -225 and 0 H_2SO_4/t. There are some exceptions to this, where the NAG pH of some ore material is between 4.5 and 8.0, or the NAG pH of waste material is <3.5, or the NAPP of the waste material is >0 kg H_2SO_4/t.

Figure 18.10.2 Plot of NAPP and NAG pH



It should be noted that the ore material, which shows the highest acid forming potential, will be processed to extract the gold. It is assumed that all the ore will be stored for only a short period on the run-of-mine (ROM) pad, before being processed. This assumption implies that there won't be a low-grade stockpile where the material will be stored for an undetermined time before processing under favourable market conditions. A short turn-around from the ROM pad will reduce oxidation reactions and the production of poor-quality leachate from the ROM pad. It is expected that the gold ore is associated with sulphide minerals and that the sulphate minerals will be leached from the produced tailings during processing, thereby reducing the acid forming potential of the plant waste material (tailings) that will be stored on the TSF.

Where the ore body is exposed in the pit walls, it is expected that oxidation of the sulphide minerals will lead to acid forming reactions. This will impact groundwater seeping from the surrounding aquifers, through the pit wall, and into the pit.

Total Sulphur and Carbonate Percentages

The total sulphate percentage for the non-acid forming (NAF) waste range between 0 and 1.54%. From the 62 non-acid forming waste samples, 15 (24%) has a total sulphur percentage higher than 0.3%. Two of the nine waste material samples that were classified as 'uncertain' have a total sulphur concentration of higher than 0.3%. This equates to 22% of the 'uncertain' samples. Only three of the waste samples across all three deposits were classified as potentially acid forming (PAF). Two of these, both at the Nyam deposit, had total sulphur percentages higher than 0.3%.

Kinetic Leach Testing

Based on the results from the static testing that was done, kinetic leach testing was performed on the 50 Batch 3 samples described and discussed above. A report was published detailing the findings (Owusu, Leaching of Batch 3 Samples with Rainwater, 2025).

The leach testing was perform using rainwater that was collected over a period of one week in open pans and then filtered to remove particulate matter. The rainwater had a pH of 5.5 and showed traces of some elements possibly from its interaction with particles in the atmosphere. Five hundred grams (500 g) of pulverized samples were leached for 24 h with rainwater at 50% solids by weight. The filtrate was subjected to multielement analysis by ICP.

The analysed concentrations are compared to the International Finance Corporation (IFC) Mine Effluent Quality Guidelines (IFC, 2007), as well as the World Health Organisation (WHO) Drinking Water Standards of 2022 (World Health Organisation, 2022). From the table it can be seen that:

- In general, the expected leach qualities comply with the IFC Mine Effluent Guidelines, as well as the WHO Drinking Water Standards.
- Arsenic concentrations exceed the IFC Mine Effluent Guideline (0.1 mg/L) in 11 out of 50 samples, and the WHO Drinking Water Standard (0.01 mg/L) in 13 samples. The arsenic concentrations range up to 0.54 mg/L. At concentrations between 0.3 and 0.6 mg/L possible adverse, chronic effects can occur in sensitive individuals. Brief exposure has no effect; skin lesions, including hyperpigmentation, will begin to appear on long-term exposure.
- Cadmium concentrations exceed the WHO Drinking Water Standard (0.003 mg/L) but comply with the IFC Mine Effluent Guideline in all samples. The cadmium concentrations that exceed the WHO standard all range between 0.015 and 0.019 mg/L. Concentrations between 0.01 and 0.02 mg/L are the threshold for health damage with continuous exposure. Single incidence of exposure will not have observable effects.
- The copper concentration in one sample (2.152 mg/L) exceed both the IFC Mine Effluent Guideline (0.3 mg/L), and the WHO Drinking Water Standard (2.0 mg/L). No health effects are expected at concentrations below 30 mg/L.
- Manganese concentrations exceed the WHO Drinking Water Standard (0.08 mg/L) in five samples. The manganese concentrations range up to 0.404 mg/L. There is no IFC Mine Effluent Guideline specified. At concentrations up to 1 mg/L severe staining and taste impacts can be expected. No health impacts are expected.

The lead concentrations exceed the WHO Drinking Water Standard (0.01 mg/L) in 15 samples. The lead concentrations range up to 0.09 mg/L. All samples comply with the IFC Mine Effluent Guideline of 0.2 mg/L. At concentrations between 0.01 and 0.05 mg/L there is no danger of any adverse health effects except for a slight risk of behavioural changes and possibility of neurological impairment, where the exposure to lead from other sources, such as food, is not minimised. At concentrations between 0.05 and 0.1 mg/L possible neurological damage can be expected where the nerve and brain tissues are developing, that is, in foetuses and young children. Young children and pregnant women should avoid exposure.

18.10.4 Three-Dimensional Conceptual Hydro-Stratigraphic Model

Conceptual Hydrogeological Model

Three-dimensional modelling was adopted to; (i) reveal geological and structural framework for the Birimian and associated units; (ii) delineate the principal hydrostratigraphic units, including the weathered overburden and fractured bedrock aquifers; and (iii) integrate borehole lithology and monitoring data, pumping-test interpretations, geophysical information and hydrochemical results into a single coherent model. Recharge and discharge zones, fracture networks and aquifer connectivity were represented explicitly in three dimensions.

Limitations and Assumptions

As with any conceptual model developed at PFS level, the 3D hydrostratigraphic model for the Project is based on a series of simplifying assumptions and is subject to important data-related limitations. These need to be explicitly recognised when interpreting results or using the model to support design and planning:

- Borehole control is spatially constrained.
- The regional geological mapping used to define the broader framework (Regional Geological Map of Ghana) is stratigraphically generalized, representing broad lithological domains rather than detailed formations.
- The south-western model boundary, which extends into Côte d'Ivoire, is effectively unconstrained by site-specific data.
- Hydraulic conductivity values were primarily obtained from a pumping test campaign conducted in November 2025 at the planned TSF and process plant area, involving five boreholes.
- Climatic inputs and recharge were necessarily simplified. Rainfall input is based on available long-term average climatic data for the Enchi area, assumed to be representative of conditions over the mine life. A recharge coefficient of 10% of mean annual rainfall was adopted.

- The hydrogeological model is explicitly tied to the 2025 mine design and preliminary schedule.

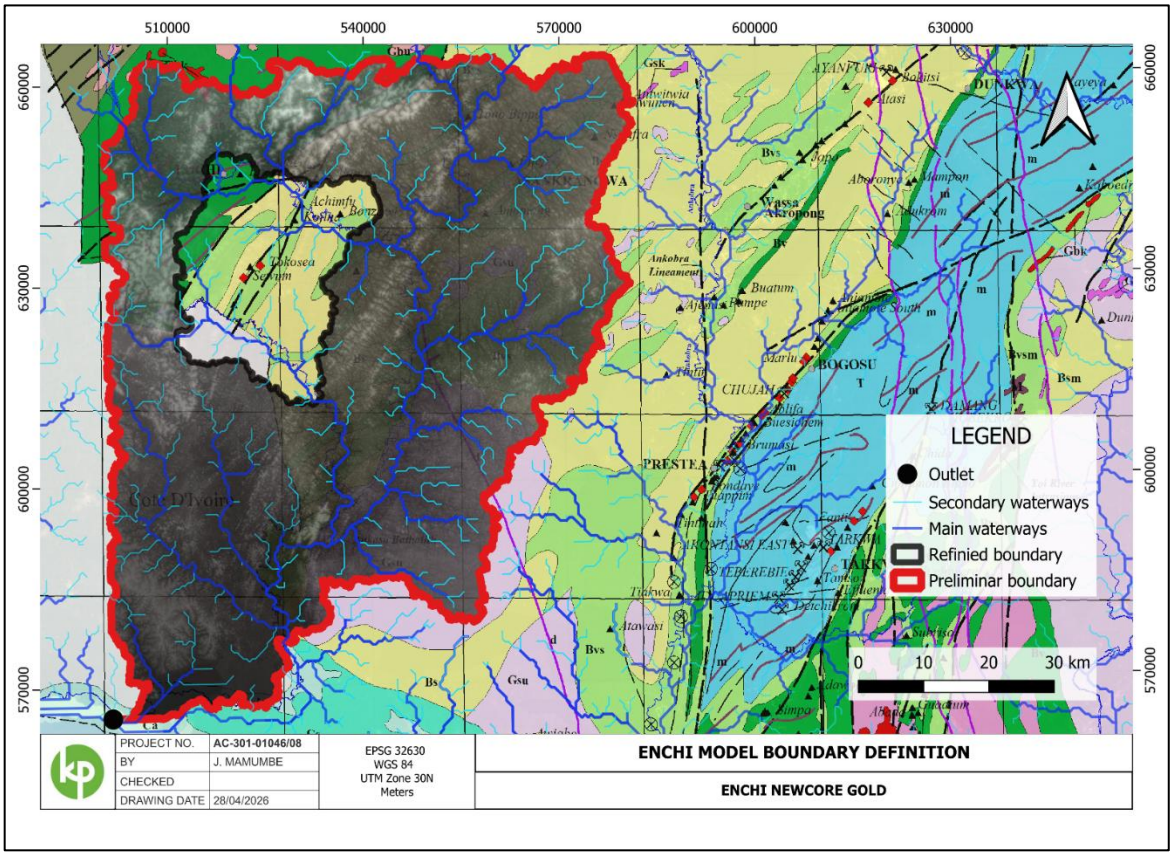
Model Development

The 3D hydrostratigraphic model was developed by integrating multiple spatial and temporal datasets within the Leapfrog Hydro™ environment. Data inputs comprised of: regional topography, regional geology, borehole data, structural geology and hydrological network**.**

Model Boundary

The lateral extent of the hydrogeological model was first defined by the inferred groundwater catchment, derived from the Tano river hydrological basin and constrained by regional geology and topography. However, to keep the model computationally tractable while still hydrologically robust, the initial domain was reduced by applying a smaller buffer around the hydrogeological catchment, so to achieve a fit for purpose model area for PFS level assessments (Figure 18.10.3).

Figure 18.10.3 Numerical Model Boundary



Model Construction

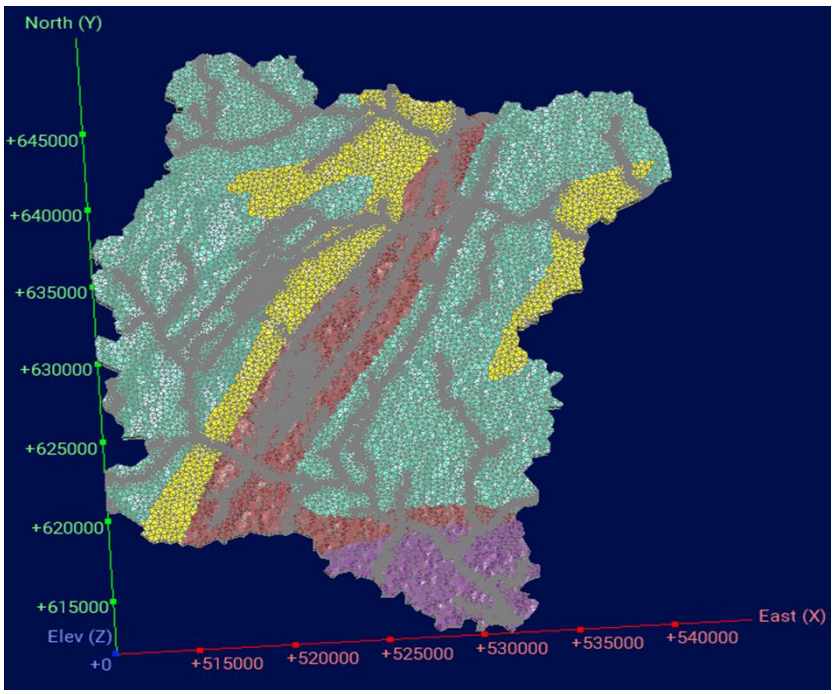
At the regional scale, a first-pass 3D framework was built from the Regional Geological Map of Ghana, which was generalized into three principal lithological domains: volcaniclastic units, marine clastic / sediments units, and basaltic–andesitic units. This model provided the broader structural and stratigraphic context within which the Project is situated.

At the mine scale, a more detailed 3D model was developed from geological borehole logs. The original logs contained numerous, variably coded lithostratigraphic descriptions. These were systematically standardised and grouped into a consistent set of modelling units to ensure robust correlation and interpretation. The final grouped units comprise: porphyroblastic siltstone (SSL), sandstone (SST), shale (SSH), breccia (BC), dolerite (MD), felsic units (F), graphitic phyllite (SPG), greywacke (SGW), mafic units (M), mylonites (MY), phyllite (SPH), siltstone (SSL), volcaniclastic sediments (SVC), and soil (SU). Planned mine layouts and their associated surface and subsurface meshes were integrated into the modelling environment to ensure that geological interpretations were spatially aligned with the mine design.

Finite Element Model Creation

The finite element model (FEM) was developed in Leapfrog Hydro using a structured, spatially adaptive mesh to capture key hydrogeological processes while maintaining computational efficiency. Mesh resolution was selectively refined in zones of elevated hydrogeological sensitivity and operational importance, with element sizes typically ranging from 20 to 50 m. Additional refinement was applied around critical mining infrastructure, including the tailings storage facility (TSF), to better resolve hydraulic gradients, seepage pathways, and potential water-balance impacts.

Figure 18.10.4 FEM in Leapfrog



18.10.5 Groundwater Numerical Model

The numerical model objectives were to characterise the site hydrogeological system and to predict mine-related groundwater impacts. The model provides a decision-support tool for groundwater management and underpins the assessment of mine-water balance, dewatering requirements, and environmental compliance at a PFS level.

The model was constructed in FEFLOW, a three-dimensional (3D) finite element approach to simulate groundwater flow and solute transport within the Enchi Gold concession and surrounding communities. The model domain and hydro stratigraphic structure were based on a conceptual model developed in Leapfrog, incorporating geological mapping, cross-sections, and extensive borehole data.

The simulation represents a sophisticated hydrogeological model designed to address combined groundwater flow and contaminant transport processes in a three-dimensional, saturated, confined aquifer system. This is achieved using FEFLOW's capability for handling transient flow coupled with transient transport, enabling the model to simulate changes in hydraulic head and solute concentrations over time in response to variable stresses and boundary conditions.

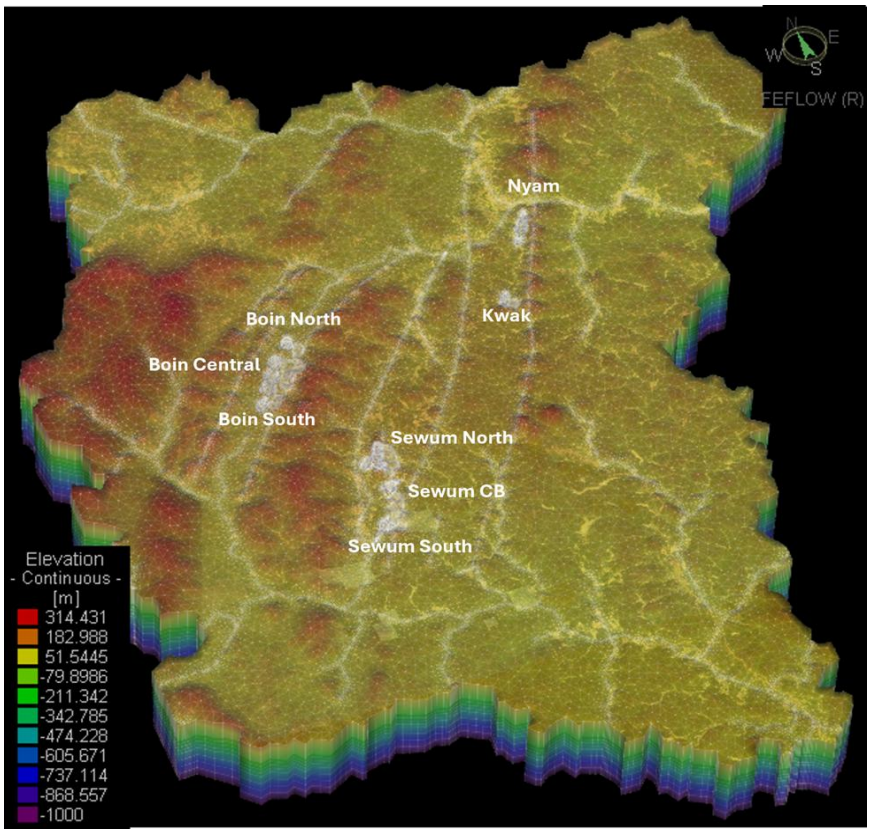
Mesh and Geometry

The model domain encompasses the broader hydrogeological setting of the proposed Enchi Gold Mine, extending approximately 25 km in the east–west direction and 30 km in the north–south direction. Vertically, the domain extends to a depth of about 1,314 m below digital elevation model (DEM) defined land surface, which forms the upper boundary of the mesh. The resulting three-dimensional domain has a total area of about 7.40×10^8 m^2 and a total volume of approximately 8.28×10^{11} m^3. The domain limits were selected to ensure that model boundaries are sufficiently distant from the main mining areas so that they do not unduly influence simulated heads and flows around the mine.

Spatial discretization is achieved using a structured finite element mesh composed of six node triangular prism elements. The mesh contains 2,372,568 elements and 1,244,975 nodes in total. In the vertical direction, the model is divided into 25 computational slices, giving 24 layers. Each layer comprises roughly 98,857 elements, and each slice contains about 49,799 nodes. The layering scheme provides adequate vertical resolution to represent the principal hydrostratigraphic units and to capture vertical gradients associated with dewatering, recharge, and potential contaminant migration. Both groundwater flow and solute transport are simulated under transient conditions, with constant time steps (30 days) throughout the simulation period.

Figure 18.10.5 illustrates the three-dimensional finite-element mesh and model domain developed for the numerical groundwater model.

Figure 18.10.5 Numerical Model Domain and 3D Finite Element Mesh (Elevation in mamsl)

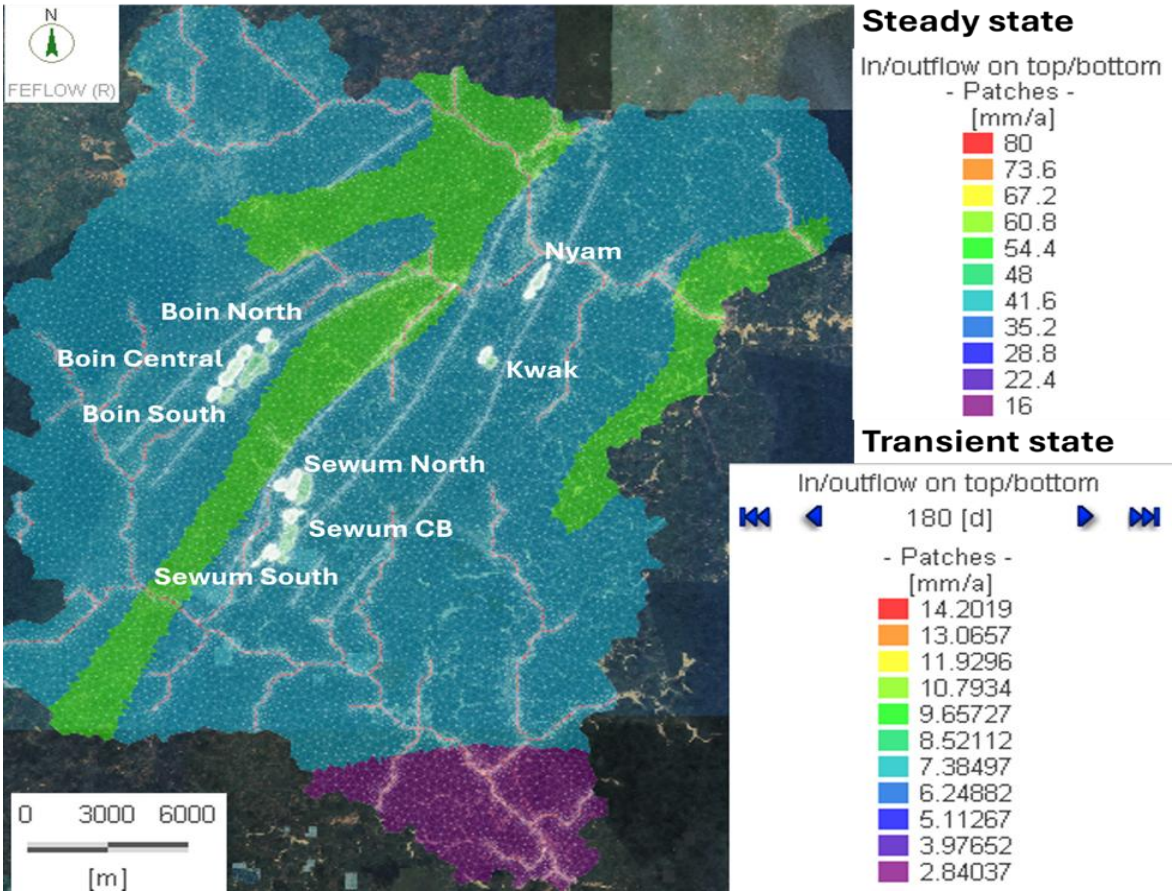


Boundary Conditions

Recharge is represented in the model as a top surface Neumann (flux) boundary condition, applied to the uppermost slice of the mesh as distributed areal inflow. Spatial variability is captured through recharge coefficients that reflect the differing infiltration capacities of the main surface geological units. For the steady state calibration, four recharge zones were defined: basaltic / andesitic terrains, marine clastic sediments, volcaniclastic units, and active stream corridors. The calibrated recharge coefficients for these zones are 0.011 for Basaltic / Andesitic, 0.029 for Marine clastic, 0.041 for volcaniclastic, and 0.057 for Quaternary sediments, expressed as fractions of the long-term mean annual rainfall. These fluxes were derived during steady state calibration to obtain a good match between simulated and observed grouvolcaniclastics, resulting in a recharge pattern that is highest along stream channels and lowest over the relatively less permeable basaltic / andesitic areas (Figure 18.10.6).

For the transient simulations, an additional flux boundary zone representing mine facilities was introduced, with a calibrated recharge coefficient of 0.036. This mine facilities recharge zone accounts for enhanced or modified infiltration associated with disturbed ground and engineered surfaces within the mine footprint and allows temporal changes in recharge from mine infrastructure to be represented explicitly.

Figure 18.10.6 Spatial Distribution of Recharege Flux on the Top Model Surface (Steady State and Transient)



Mine excavations were represented explicitly by modifying both the active model domain and the boundary conditions in accordance with the mine plans and production schedule. For each pit and excavation stage, elements located within the planned excavation envelope were progressively deactivated (mined elements removed from the flow domain) at the scheduled time, thereby mimicking the removal of rock and the loss of storage and transmissive capacity in those volumes.

Table 18.10.2 summarizes the planned mining schedule for the Enchi deposits adopted for the PFS groundwater model. It presents the years of operation for each pit (Boin, Nyam, Kwakyekrom, Sewum South, Sewum CB and Sewum North) over the 10 year mine life, and forms the basis for staging of dewatering and hydrogeological impact assessments in the numerical model.

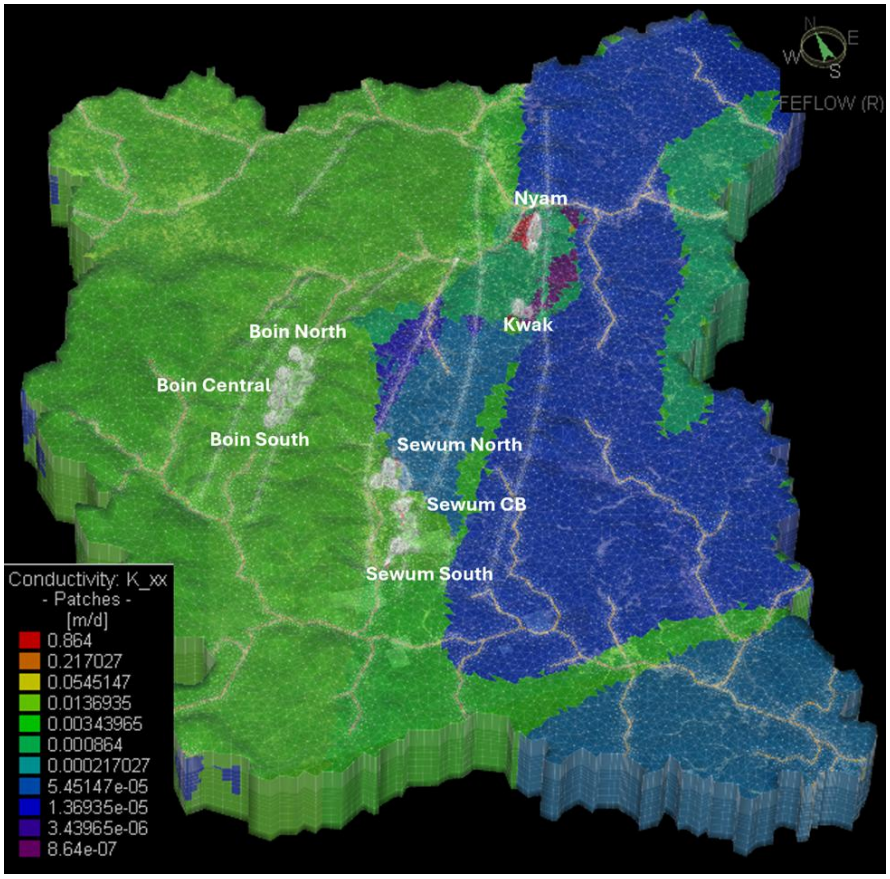
Table 18.10.2 Mine Schedule

MINE	PERIOD									
	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9	Y 10
Boin										
Nyam										
Kwakyekrom										
Sewum South										
Sewum Cb										
Sewum North										

Aquifer Parameters

Hydraulic conductivity in the model is defined with separate values for the principal directions Kxx, Kyy and Kzz (principal hydraulic conductivity components). For the Enchi model, horizontal conductivity is assumed to be isotropic, such that Kxx = Kyy in all material zones. These horizontal conductivities vary spatially in accordance with the mapped hydrostratigraphic units (Figure 18.10.7), ranging from about 8.6×10^{-7} m/d in the lowest permeability domains to approximately 0.864 m/d in the most transmissive zones. To account for the typically lower permeability in the vertical direction due to bedding, layering, and secondary structures, a vertical anisotropy ratio of 0.1 was adopted, i.e. Kzz = 0.1 × Kxx. This implies that vertical flow is constrained to be an order of magnitude less conductive than horizontal flow, which is consistent with the conceptual understanding of the fractured and layered geology at Enchi and helps to reproduce observed vertical gradients between model layers while maintaining realistic lateral groundwater flow patterns. The hydraulic conductivities were first assigned to the regional model based on the regional geological map, then adjusted based on the detailed information from the boreholes logs in the mine areas.

Figure 18.10.7 Spatial Distribution of Horizontal Hydraulic Conductivity Kxx (Kyy; Kzz = 0.1 Kxx)



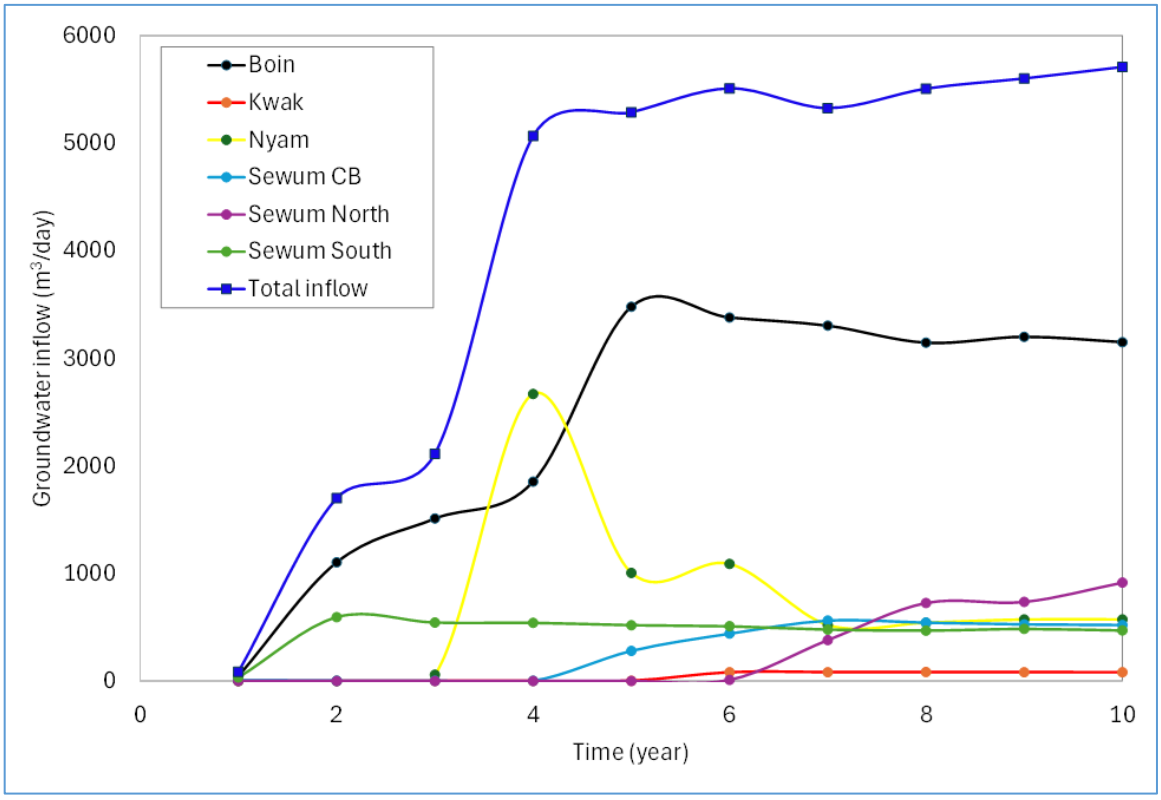
18.10.6 Predictive Simulations and Model Results

Groundwater Inflows

Given that the pits are hydraulically discrete and are scheduled to operate over different time periods, groundwater inflows are most appropriately quantified on a pit specific basis for each pit's respective period of activity. Figure 18.10.8 shows the modelled evolution of groundwater inflow volumes into each pit and the combined total over the 10 year mine life.

Boin operates from Years 1–5 and dominates the groundwater budget and exhibits the largest and most sustained inflows, increasing from about 46 m^3/day in Year 1 to 1,510 m^3/day in Year 3 and peaking at approximately 3,480 m^3/day in Year 5. Nyam is active from Years 3–6; inflows are modest initially (58 m^3/day in Year 3), rise rapidly to a peak of about 2,670 m^3/day in Year 4, and then decline to roughly 1,090 m^3/day by Year 6. Kwakyekrom, which is by far the smallest pit. is only mined in Years 5–6 and remains a minor contributor, with inflows of around 80 m^3/day.

Figure 18.10.8 Predicted Groundwater Inflow to Individual Pits and Total Site Inflow Over the Life-of-Mine



The Sewum pits show moderate but still material inflows over shorter operating periods. Sewum South is active in Years 1 and 2, with inflows increasing from about 33 m^3/day to nearly 600 m^3/day. Sewum Central Block (CB) operates in Year 1 and again in Years 5 to 7; inflows are negligible in Year 1 when the pit is still shallow, rising to about 280 m^3/day in Year 5, and reach around 560 m^3/day in Year 7. Sewum North is mined from Years 6 to 9, with inflows increasing from about 10 m^3/day in Year 6 to roughly 915 m^3/day by Year 10 (post-operation residual inflow).

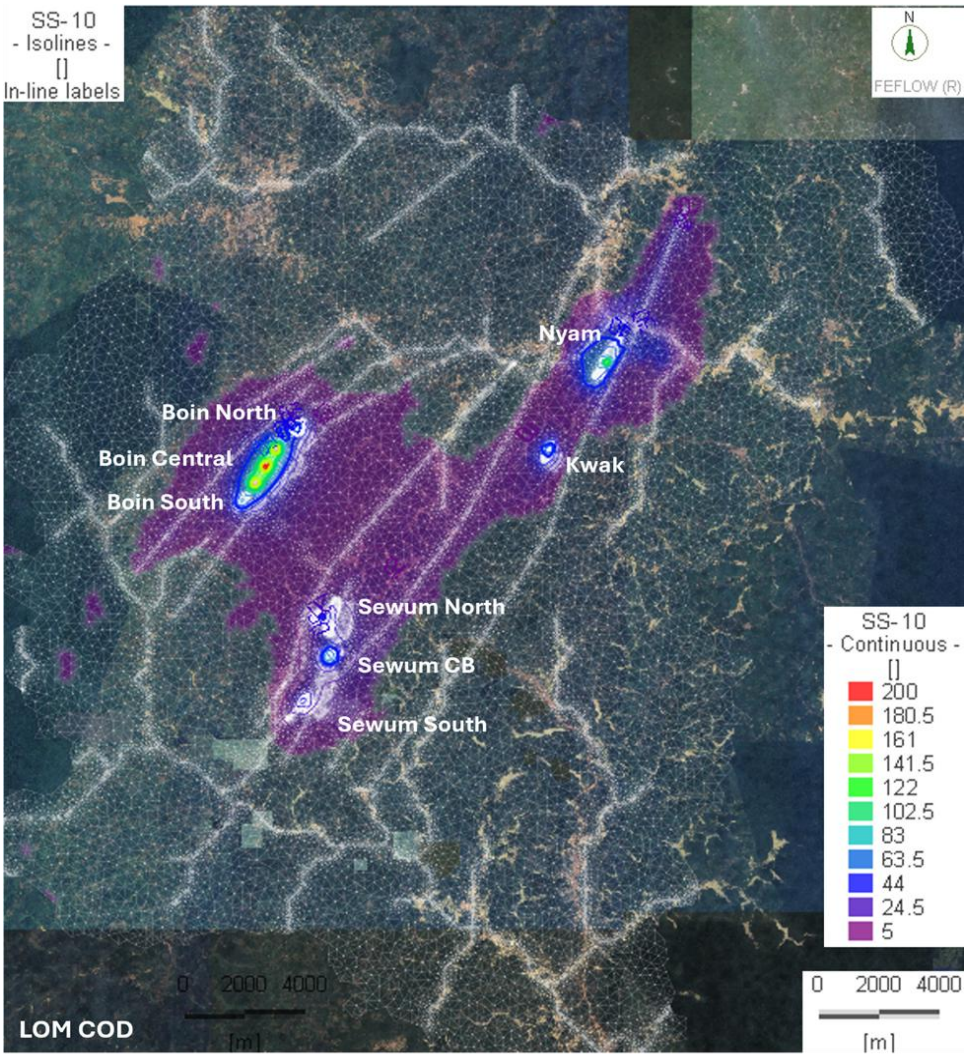
These results demonstrate that dewatering must be designed and managed on a pit specific basis. High, sustained capacities are required at Boin (Years 2 to 5) and at Nyam (particularly Years 3 to 4), with progressively increasing capacity at the Sewum pits during their operating years, while only limited, short-lived dewatering is needed at Kwakyekrom and Sewum South.

Groundwater Level Drawdown Cone

The cone of depression (COD) map for the life of mine (LOM) scenario shows the cumulative drawdown in groundwater levels produced by dewatering of the planned open pits (Figure 18.10.9). Distinct, localized cones develop around each pit, with the vertical drawdown magnitude decreasing radially away from the pit centres. The most pronounced COD occurs at Boin, where combined dewatering from the North, Central, and South pits generates drawdowns exceeding 180–200 m in the pit areas and more than 100 m over a broader footprint with a 5 m cutoff ~2.5 km to the west. Nyam also exhibits a considerable cone, with maximum drawdowns typically between about 160 and 180 m, with a 5 m cutoff of ~ 5.5 km along a regional fault zone to the north. These larger cones reflect the higher inflow rates and deeper excavations projected for these pits.

In contrast, the Sewum pits (North, Central Block and South) generate more moderate drawdowns, generally below about 100 m at the pit centres and attenuating rapidly over relatively short distances, while a smaller but clearly defined COD is associated with Kwakyekrom. Outside the immediate vicinity of the pits, regional drawdown is limited, and the water table remains close to pre mining levels, as indicated by the extensive areas with 5 m or less of vertical drawdown. The spatial separation of the Boin, Nyam–Kwakyekrom and Sewum clusters means that the cones of depression remain largely discrete, with limited overlap between mining areas. Overall, the LOM COD distribution suggests that dewatering impacts will be strongly focused around the individual pit complexes, with only modest cumulative effects on groundwater levels at the margins of the project area.

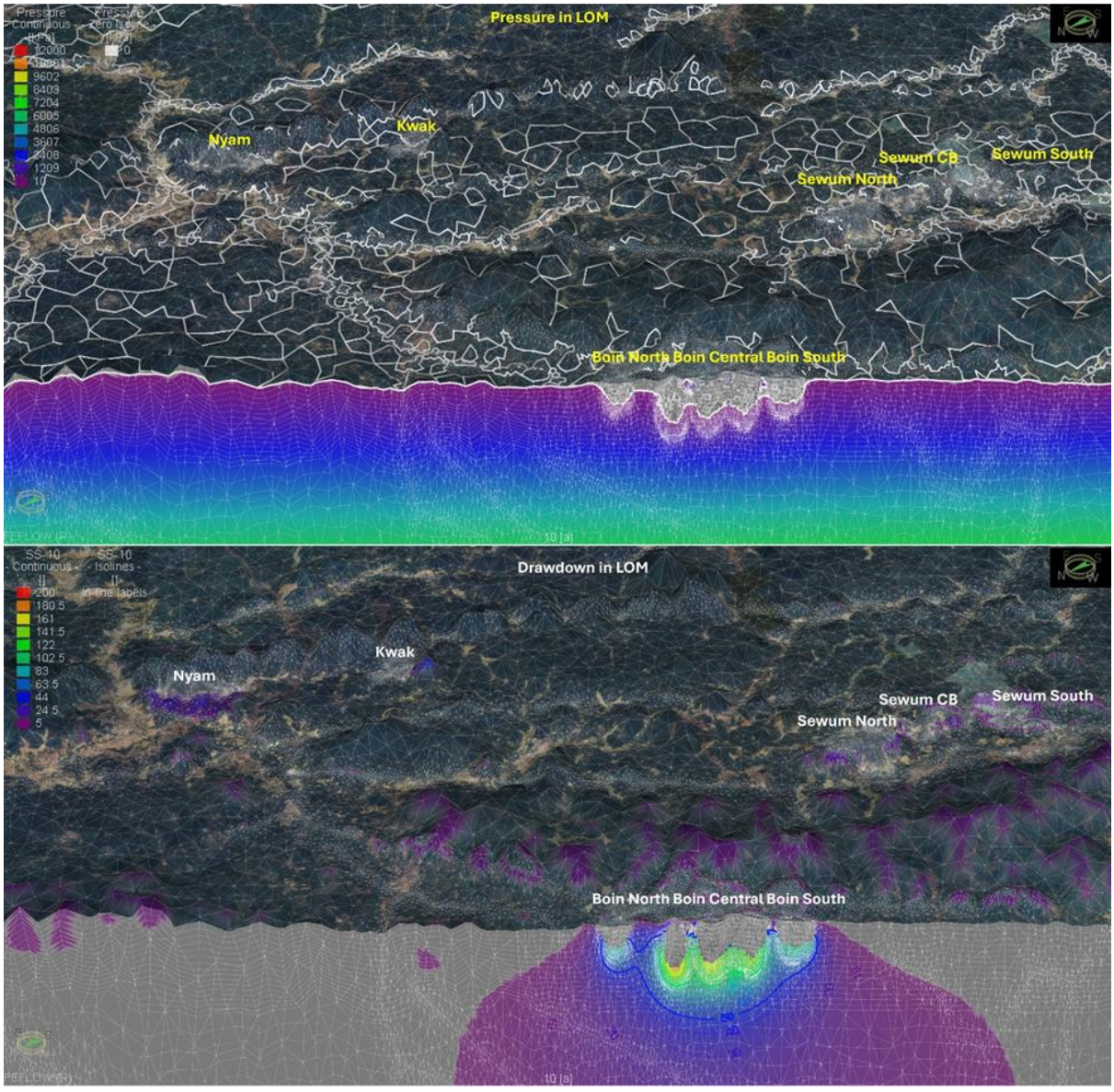
Figure 18.10.9 Life of Mine Cones of Depression Around Open Pits (Cumulative Drawdown in m)



The 3D cross-section for the life of mine (LOM) scenario illustrates both pore pressure distribution and associated drawdown beneath the open pits. The upper panel in Figure 18.10.11 shows pore pressure, with higher pressures at depth and a pronounced reduction directly beneath the Boin pit complex where dewatering is most intensive; the pressure field there is strongly depressed relative to the surrounding rock mass, forming a deep, laterally extensive low-pressure zone. Nyam, Kwakyekrom and the Sewum pits also generate localized pressure reductions, but these are less pronounced than at Boin and decay rapidly away from the excavations.

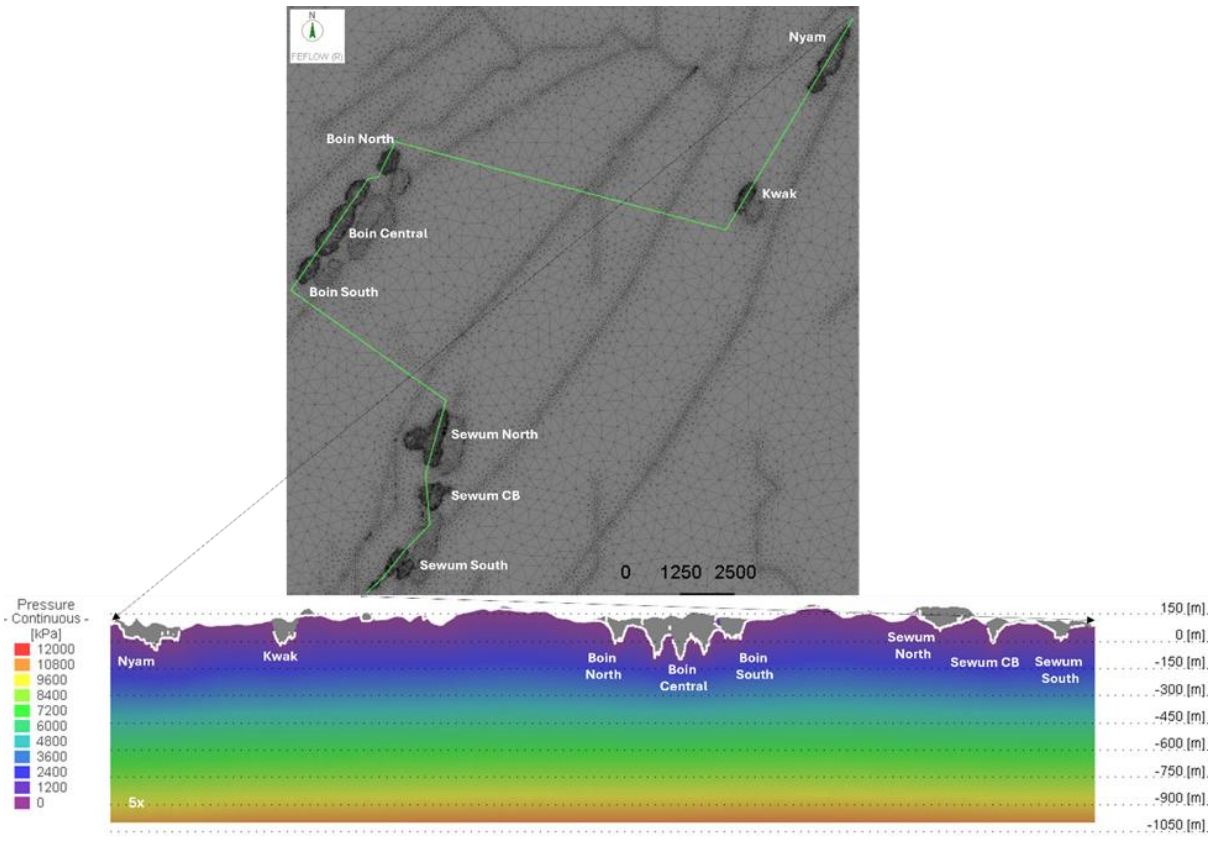
The lower panel in Figure 18.10.10 presents the corresponding drawdown field, clearly showing the vertical and lateral extent of the cones of depression. Groundwater level and pressure changes occur beneath Boin North, Central, and South, where the zone of influence of the pressure release due to excavation of the pit and the associated upwelling of groundwater through the pit floor extends several hundred metres below pit floor level, while more moderate cones, less visible in the cross section, are under Nyam, Kwakyekrom and the Sewum pits. Together, the two views confirm that the most significant hydraulic disturbance is focused beneath the main pit complexes, with limited impact on pore pressures and heads at greater distances from the mine.

Figure 18.10.10 3D Cross Section of Pore Pressure and Drawdown for the Life of Mine Scenario



The non-straight cross-section (see green line in Figure 18.10.11) traces a continuous line through all pits (Nyam, Kwakyekrom, Boin North–Central–South, Sewum North, Sewum Central Block, and Sewum South) to present their hydraulic conditions on a single profile at the end of the mine life. Along this section, the zero-pressure isoline clearly delineates the water table, showing how it is locally drawn down at each pit relative to the regional groundwater surface. Beneath the water table, the colour-coded pressure field reveals the increase in pore pressure with depth and the downwards curvature in the pressure, which is most clearly seen at the Boin complex, and to a lesser extent at Nyam and the Sewum pits, highlighting the strong pressure depressions developed due to mine dewatering. This composite view emphasises the localized nature of the hydraulic perturbations caused by dewatering at each mining area.

Figure 18.10.11 Composite Curvilinear Cross Section Through All Pits Showing Water Table (Zero Pressure Isoline) and Pore Pressure Distribution in LOM



Solute Transport

The composite map and 2D cross sections of the solute transport simulation for the waste rock dumps (WRDs) associated with the open pits and the two TSFs indicate that potential leachate impacts are highly localized over the life of mine, even under the conservative assumption of unlined facilities. Using a 250 mg/L concentration cut off, the average lateral migration of the plume from each WRD / TSF is limited to approximately 20–30 m, demonstrating that horizontal transport is strongly constrained by the low regional groundwater velocities and the relatively small driving heads beneath the facilities. Vertically, the simulated downward movement of solutes from the Nyam WRD to the Sewum South WRD ranges between about 13 and 16 m over the life of mine (individual annual increments of 15, 13, 13, 14, 16, 16, 14, 15, 13 and 13 m), indicating that leachate remains confined to the shallow groundwater system and does not penetrate deeply into the underlying aquifer sequence.

Overall, these results suggest that, with appropriate operational controls and seepage management measures, the risk of far field groundwater quality impacts from WRDs and TSFs is low; contaminant plumes are predicted to remain close to the facilities and within the immediate mine footprint over the projected mine life. With lining in place, solute migration is expected to be reduced by approximately 13–23 m in the horizontal direction and vertically contained within the lined layer over the life of mine.

Figure 18.10.12 Map Showing Simulated Solute Transport from WRDs and TSFs Over Life of Mine (250 mg/L Cut off)

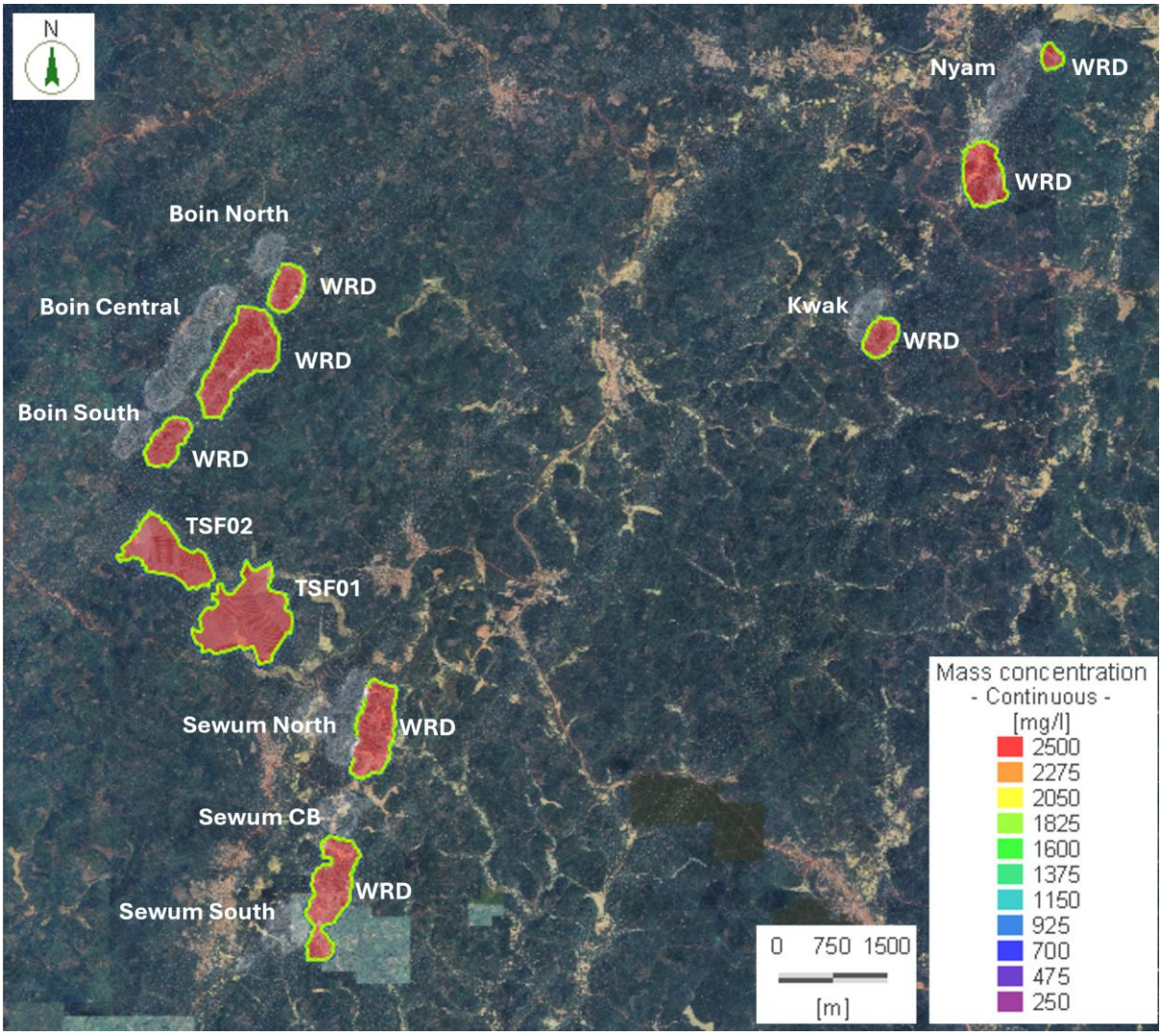
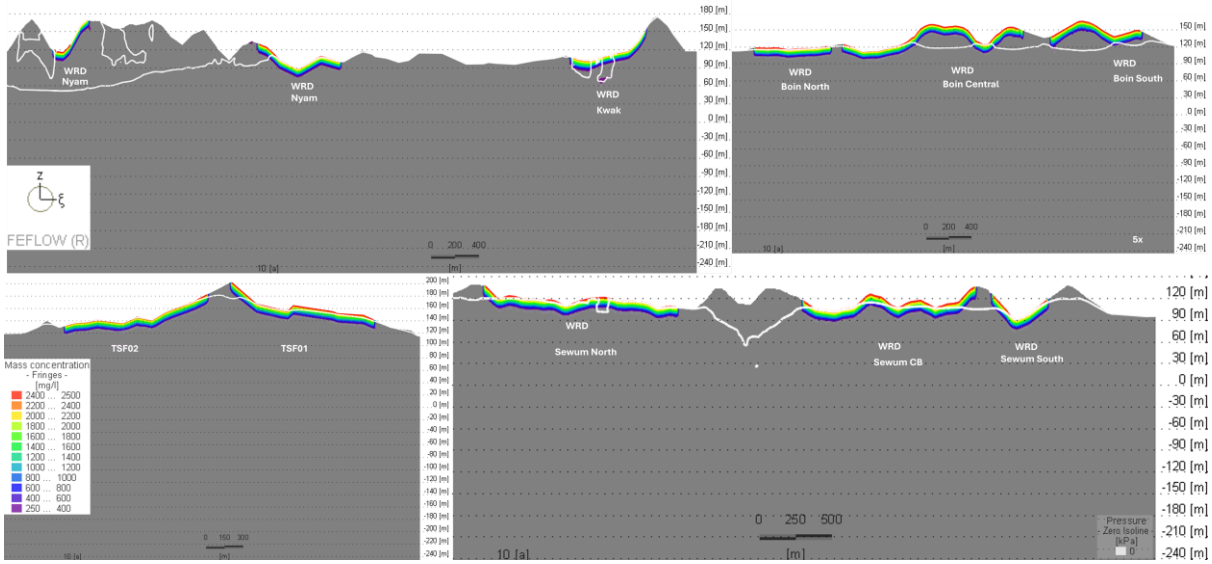


Figure 18.10.13 Composite Cross-Sections of Simulated Solute Concentrations Beneath WRDs and TSFs for Life-of-Mine Conditions



Top left – Nyam WRD and Kwakyekrom WRD, top right – Boin North, Boin Central and Boin South WRDs, bottom left – TSF01 and TSF02, bottom right – Sewum North, Sewum CB and Sewum South WRDs

However, the groundwater table in the project area is generally shallow, and it is therefore likely that any leachate generated in the WRDs and the TSFs will reach the saturated zone. The current numerical modelling simulations cover the operational period; it is essential to extend this work with a post-closure simulation to evaluate long term solute transport and potential recirculation between pit lakes, WRDs and TSFs. In addition, the present simulations address groundwater only and do not explicitly consider transport to surface water receptors; a complementary assessment of solute mobilisation via runoff, erosion and baseflow to streams and other surface water bodies is recommended.

To manage and verify the modelling predictions, a comprehensive monitoring network of groundwater boreholes and surface water sites should be designed and implemented from pre mining through operations and into post closure, allowing early detection of any leachate migration and providing data for ongoing model refinement and risk management.

18.10.7 Conceptual Dewatering and Water Supply

The conceptual dewatering strategy for Enchi is based on controlling groundwater inflows to individual, spatially separated open pits while limiting the lateral extent of drawdown beyond the mine footprint. Numerical modelling indicates that each pit (Boin, Sewum, Nyam, Kwakyekrom) generates its own discrete cone of depression, with the largest and most sustained inflows occurring at Boin and, for a shorter period, at Nyam.

Dewatering will therefore be implemented as a combination of in pit sumps and strategically located perimeter / depressurisation boreholes specific to each pit or pit cluster, rather than a single, centralized system. Pumping rates and well layouts will be tailored to the predicted inflow profiles and operational periods of each pit, with capacity ramped up as excavations deepen and then reduced or redeployed as mining ceases and inflows decline. This pit focused approach maximises hydraulic control where it is needed, reduces unnecessary regional drawdown, and provides flexibility to adapt to changing mine plans.

18.11 Water Systems

18.11.1 Raw Water Supply System

It is assumed that raw water will be obtained by way of rain catchment. This water will be pumped to an open top raw water tank. Raw water will then 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.

18.11.2 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 576 m^3. In addition, all buildings will be equipped with hose cabinets and supplemented with handheld fire extinguishers of 2 types—general purpose extinguishers for inside plant areas, and dry type extinguishers for inside electrical and control rooms.

18.11.3 Potable Water Supply

The potable water treatment plant will be designed to local drinking water guidelines. The water treatment plant is expected to be a containerised unit with 80 m^3/h processing capacity and 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 (600 m^3 capacity) 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.

18.12 Sewage Treatment

Sanitary sewage will be collected via underground networks and collected in a holding tank. Sludge disposal will be managed by licensed contractors.

Sewerage handling for the mine operations specific areas is outlined in Section 16.4.

18.13 Fuel Storage and Distribution

The primary Mining Service Area (MSA), located adjacent to the process plant near the Boin deposit, will support key mining operations and site-wide fuel requirements through an integrated fuel storage and distribution facility (see Section 16.4.1 for additional detail). This facility will include bulk fuel storage tanks, dedicated fueling infrastructure for both light and heavy vehicles, and associated operational areas. Operated by a specialized contractor, it will provide reliable on-site fuel management and distribution, supported by connection to the site-wide power network to ensure continuous and efficient operation.

18.14 Communication

The site communication strategy integrates wireless networks and two-way radio systems to provide reliable, site-wide coverage. Contractor quotations for mining operations and services include the supply, installation, and maintenance of communication equipment required to support all mining areas. Core communication infrastructure will be centralized within the process plant area, serving as the primary hub for network management and distribution. This approach ensures dependable communications to support both operational efficiency and safety requirements across the site

18.15 Accommodation Camp

No on-site accommodations have been accounted for in this study given the proximity of the Project to the town of Enchi. Senior staff and expatriate employees will be accommodated in rental units in the town of Enchi.

19.0 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

The Enchi Gold Project will produce doré bars containing gold. These doré bars will be sold for further processing into refined products for commercial sale.

The market for doré is well established, and Enchi's doré bars could be readily sold on the spot market. Gold is traded globally across numerous active markets, making pricing information easily accessible at any time. Gold benefits from highly liquid markets, supported by a large number of well-informed buyers and sellers.

Over the past 12-months, the gold price has seen a strong increase and has traded in a range between $3,280/oz and $5,595/oz. Table 19.1.1 presents the spot prices for gold as of 23 June 2026, based on the London Bullion Market Association (LBMA) spot rates.

Table 19.1.1 Gold Price

Metal	Unit	23 June 2026
Gold	US$/oz	4,100

Table 19.1.2 presents the gold price and payability forecast utilized in the economic analysis in this study.

Table 19.1.2 Payability Forecast

Metal	Unit	Value
Gold Price	US$/oz	3,800
Refining Payability	Gold %	99.9

19.2 Contracts

At present, the Project does not have any sales contracts in place. Any future agreements would align with standard industry practices and be comparable to typical contracts governing the sale of doré globally.

In addition to product sales, the Project's major contractual commitments are expected to relate to mine equipment supply, drill-and-blast services, bulk materials, and provision of specialized technical services.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

This section of the report summarizes the current status of permitting as well as environmental and social studies undertaken for the Enchi Gold Project. It provides an overview of the environmental and social context, and identifies preliminary risks and impacts, together with recommendations for management. It also presents high-level information on closure planning. Information presented in this section is based on secondary data, primary baseline data completed to date, and photographs from site visits conducted as part of the Pre-Feasibility Study.

20.1 Environmental and Social Assessment

20.1.1 Legal and Regulatory Framework

Exploration and mining activity in Ghana is governed by the Ministry of Lands and Natural Resources (MLNR) and is administered by the Minerals Commission (MINCOM). Key legislation governing mining in Ghana is comprised of the following:

- 1992 Constitution of Ghana.
- Minerals and Mining Act, 2006 (Act 703).
- Minerals and Mining Act (Amendment) 2015 (Act 900) and 2019 (Act 995).
- Minerals and Mining (General) Regulations, 2012 (L.I. 2173).
- Minerals and Mining (Licensing) Regulations, 2012 (L.I. 2176).
- Minerals and Mining (Health, Safety and Technical) Regulations, 2012 (L.I. 2182).
- Minerals and Mining (Compensation and Resettlement) Regulations, 2012 (L.I. 2175).
- Minerals and Mining (Explosives) Regulations, 2012 (L.I. 2177).
- Minerals and Mining (Local Content and Local Participation) Regulations, 2020 (L.I. 2431).
- Minerals and Mining (Support Services) Regulations, 2012 (L.I. 2174).
- Minerals and Mining (Mineral Operations – Tracking of Earth Moving and Mining Equipment) Regulations, 2020 (L.I. 2404).

- Minerals and Mining (Ground Rent) Regulations, 2018 (L.I. 2357).
- Additional supporting legislation for environmental protection, water, land use, forestry and wildlife includes the following:
- Environmental Protection Act, 2025 (Act 1124).
- Minerals Commission Act, 1993 (Act 450).
- Water Resources Commission Act, 1996 (Act 522).
- Forestry Commission Act, 1999 (Act 571).
- Ghana Geological Survey Authority Act, 2016 (Act 928).
- Forests Protection Act, 1974 (NRCD 243), amended in 2002 (Act 624).
- Land Act, 2020 (Act 1036).
- Land Use and Spatial Planning Act, 2016 (Act 925).
- Wildlife Resources Management Act, 2023 (Act 1115).
- Environmental Protection (Environmental Assessment) Regulations, 2025 (L.I. 2504).
- Environmental Protection (Air Quality Management), 2025 (L.I. 2507).
- Environmental Protection (Effluent), 2025 (L.I. 2506).
- Water Use Regulations, 2001 (L.I. 1692).
- Drilling License and Groundwater Development Regulations, 2006 (L.I. 1827).
- Hazardous, Electronic and Other Wastes (Classification) Control and Management Regulations, 2016 (L.I. 2250).
- Land Use and Spatial Planning Regulations, 2019 (L.I. 2384).

Relevant national standards for environmental protection that apply to mining operations include the following:

- Environment and Health Protection – Requirements for Ambient Air Quality and Point Source / Stack Emissions (GS 1236:2019).

- Health Protection – Requirements for Ambient Noise Control (GS 1222:2018).
- Environmental Protection – Requirements for Effluent Discharge (GS 1212:2019).
- Water Quality – Specification for Drinking Water (GS 175:2017).

As the Project advances beyond the PFS stage, a more detailed analysis of the applicable regulatory framework requirements will be undertaken.

20.1.2 Mineral Rights and Licensing Requirements

The Minerals and Mining Act, 2006 (Act 703) defines mineral operations as an activity that involves reconnaissance, prospecting, or mining for or of minerals. The Act provides for three types of mineral rights (licences) for large-scale mining operations, namely: Reconnaissance Licence, Prospecting Licence, and Mining Lease. Key features of these mineral rights are summarized in Table 20.1.1.

The Reconnaissance Licence and Prospecting Licence are exploration mineral rights as they only allow the holder to prospect or search for minerals and conduct limited activities to determine the mineral potential of area. The Mining Lease, however, is an exploitation mineral right which permits the holder to mine the mineral / ore for processing.

Table 20.1.1 Mineral Rights / Licences and Requirements

Mineral Right / Licence	Duration / Validity	Permitted Activities / Requirements	Applicability to Project
Reconnaissance Licence	Up to 1 year, but renewable.	Allows reconnaissance activities like preliminary survey works in support of exploration. No drilling (boreholes) or excavations.	Not applicable (concluded phase of Project)
Prospecting Licence	Up to 3 years, but renewable with holder shedding off 50% of the exploration area during each renewal.	Allows prospecting activities, including drilling (boreholes) and excavations, erection of camps and temporary structures (buildings). Holder must obtain an environmental permit for exploration from Environmental Protection Authority (EPA). Holder must obtain a Prospecting Operating Permit from MINCOM.	Applicable to current phase of the Project

Mineral Right / Licence	Duration / Validity	Permitted Activities / Requirements	Applicability to Project
Mining Lease	Up to 30 years	Allows for mineral operations (i.e., mining and prospecting), including erection of buildings, processing plants, other supporting infrastructure, and storage / disposal of mineral waste. Holder must obtain parliamentary ratification of mining lease agreement in line with Article 268 of the 1992 Constitution of Ghana. Holder must obtain an environmental permit for mining from EPA based on an approved Environmental Impact Statement (EIS). Holder must obtain a Mining Operating Permit from MINCOM.	Applicable to next phase of the Project

The Project is currently at the prospecting phase with valid Prospecting Licences. Prospecting Licences are valid for an initial period of three years, and a renewal application can be made at least three months prior to the licence expiry date and subject to a one-time reduction in the size of the exploration area. A successful renewal is valid for a further three years. In practice, the Prospecting Licences can be extended multiple times so long as a minimum of one hundred and twenty-five contiguous blocks remain subject to the licence. As is the case for the Enchi Gold Project, the Prospecting Licences have been renewed multiple times.

The licensing procedure for Prospecting Licences involves review of the application by MINCOM, gazetting and notification of key stakeholders (Chiefs, District Assembly, etc.), and review of the proposed work programme by an Institutional Technical Committee (ITC). The ITC is composed of representatives from Environmental Protection Authority (EPA), Ghana Geological Survey Authority (GGSA), Lands Commission (LC), Water Resources Commission (WRC), and Forestry Commission (FC). Following this review, a technical recommendation is submitted by MINCOM to the Minister for Lands and Natural Resources who then approves or rejects the licence application. Once a Prospecting Licence has been issued, the licence holder must obtain an Environmental Permit for exploration from EPA, and a Prospecting Operating Permit from MINCOM prior to commencing exploration activities. The validity periods of the permits would be specified on the permits, including conditions for renewal prior to their expiration.

Mining Lease applications require undertaking a full Environmental Impact Assessment (EIA) and presenting findings in an Environmental Impact Statement (EIS) for review by EPA for the grant of an Environmental Permit for mining. A Feasibility Study Committee is also constituted to review the Feasibility Study (FS) report of the applicant, then a technical recommendation is submitted by MINCOM to the Minister for Lands and Natural Resources. The Minister upon approval will submit a request for ratification to Cabinet for review and thereafter to the Parliament of Ghana to ratify the lease agreement. The Government of Ghana (GoG), subject to Section 43 of Act 703, will acquire a ten percent (10%) free carried interest in the mining rights and obligations of the mineral operations. Subject to Section 17 and 18 of Act 703, the holder of a mineral right shall also obtain requisite approvals, licenses, or permits from other institutions such as the FC and WRC before commencing activities.

20.1.3 Project Permitting Status

The Project currently comprises nine Prospecting Licences, details of which are summarized in Table 20.1.2. All required documentation to extend the licences for the Enchi Gold Project have been submitted to MINCOM. Official stamped and signed mineral title maps were obtained from MINCOM on March 9, 2026, and the maps include all the Prospecting Licences and applications related to the Enchi Gold Project.

Sewum, Enkye, Nyamebekyere and Yehikwawkrom

The Prospecting Licences for Sewum, Enkye, Nyamebekyere, and Yehikwawkrom with expiry dates of 31 May 2023 received approval letters from MINCOM to the Minister recommending extension to 2026, pending the grant of the licence. Confirmation letters from the Minister of Lands and Natural Resources ("Minister") confirming the extensions were received for Enkye and Yehikwawkrom. Exploration can legally continue on the Licences while the renewal process proceeds (including renewal of operating permits). Annual Mineral Right (AMR) fees for 2026 have been paid for all the Licences. Processes for renewal of environmental permits beyond 2026 are progressing, with permit fees paid for Enkye and Yehikwawkrom, and the others pending.

Section 35 (4) and Section 38 (2) of Act 703 makes provision for continuation of exploration activity in the event that a decision by MINCOM or other government agency is delayed:

35 (4) "Where a holder of a prospecting licence has made an application for an extension of the term of the licence and the term of the prospecting licence would, but for this subsection expire, the prospecting licence shall continue in force in respect of the land the subject of the application until the application is determined."

38 (2) "Where, at least one month before the end of the term or within a shorter period that the Minister may allow, the holder of a prospecting licence satisfies the Minister that delay by a government institution or agency in the issuance of permits or in carrying out a lawful activity have resulted in delay by the holder in the discharge of an obligation under the prospecting licence, the holder may apply in writing to the Minister for relief from the obligation to surrender land and the Minister may exempt that holder from the requirements of this section, either wholly or in part, for a period, not more than twelve months and on other conditions that the Minister thinks fit."

Abotia and Omanpe

The Prospecting Licences for Abotia and Omanpe are currently valid. Application for an extension beyond 2026 has been filed with MINCOM, and operating permit renewal is in progress. Annual Mineral Right (AMR) fees for 2026 have been paid and prospecting activity is allowed to continue under Ghanaian mining law pending the receipt of extension approval documents. Environmental permits remain valid until November 2027.

Nyame Esa, Nkwanta and Anguzu

The Prospecting Licences for Nyame Esa, Nkwanta, and Anguzu are under the name of Boin Resources Limited and controlled 100% by Newcore. These licences cover previously 'lost' portions of other Prospecting Licences due to the mandatory requirement of Act 703 (Section 38) for reduction in spatial area for licence extensions. Notice (letter) of grant of licence for these areas was received on 11 September 2024 and all requested fees paid. Final Prospecting Licences are in the process of being issued and are expected. At the time of this report, no exploration activities were being undertaken in these areas.

20.1.4 Good International Industry Practice

In addition to compliance with all applicable Ghanaian legal requirements, Newcore will develop the Project in alignment with Good International Industry Practices (GIIP). This approach demonstrates Newcore's responsible business ethics and commitment to environment, social and governance (ESG) principles. This will further facilitate meeting potential financial lender requirements in the future. Newcore will refine its approach to GIIP and continue to build on its current ESG commitments as the Project advances.

Some relevant international standards and guidelines which constitute GIIP and may apply to the Project include the following:

- **International Finance Corporation (IFC) Performance Standards (PS) on Environmental and Social Sustainability (IFC, 2012) and IFC Health, Safety and Environment Guidelines for Mining** – these are part of the IFC's Sustainability Framework launched in January 2012. The IFC PS which are eight in total provide a baseline of environmental and social good practice and form an important assessment reference for managing project risks and impacts in a sustainable way.

- **Equator Principles (EP)** – the EP4 (Equator Principles Association, 2020) is the latest iteration of the EP, published in July 2020. The EP are a risk management framework, adopted by international financial institutions for determining, assessing, and managing environmental and social risk in projects. The EP framework is based on the IFC PS and the World Bank Group (WBG) Environmental, Health and Safety (EHS) Guidelines on environmental and social sustainability.

- **World Bank Environmental, Health and Safety Guidelines –** the EHS guidelines, developed with IFC, include General EHS Guidelines (WBG, 2007a) and other sector-specific guidelines, as well as various supporting Good Practice Notes and Handbooks. They provide a source of technical information during project appraisal and are globally accepted as technical reference documents presenting general and industry specific examples of GIIP. The EHS Guidelines for Mining (WBG, 2007b) specifically addresses EHS issues across the mining industry, including open-pit mining operations.

- **International Council on Mining and Metals (ICMM) Mining Principles (ICMM, 2020)** – these have been developed in response to evolving societal expectations of the mining industry, and include a comprehensive set of 10 Mining Principles, 39 Performance Expectations, and 8 Position Statements. ICMM has also developed various Good Practice Guides which include widely recognised guidelines for integrated mine closure (ICMM, 2025a), catchment-based water management (ICMM, 2015), water reporting (ICMM, 2021a), health and safety (ICMM, 2021b), emissions accounting and reporting (ICMM, 2023), tailings management (ICMM, 2025b), and acid rock drainage (ICMM, 2025c).

- **Global Industry Standard for Tailings Management (GISTM) –** the Standard which was launched in August 2020 was developed by an independent review process in response to a number of catastrophic tailings dam failures. It was initiated by ICMM, United Nations Environment Programme (UNEP) and Principles for Responsible Investment (PRI). GISTM provides a global benchmark to achieve strong social, environmental, and technical outcomes in tailings management. The Standard strives to achieve the goal of zero harm to people and the environment with zero tolerance for human fatality (Global Tailings Review, 2020).

- **The International Cyanide Management Code (ICMC) for the Manufacture, Transport and Use of Cyanide in the Production of Gold (Cyanide Code)** – the Cyanide Code is a voluntary, performance driven, certification program of best practices for gold and silver mining companies, and companies producing and transporting cyanide (International Cyanide Management Institute, 2021).

Table 20.1.2 Summary and Status of Enchi Gold Project Licences and Permits

Deposit Name	Prospecting Licence (PL) Number	PL Area (km^2)	Holding Company	PL Expiry Date / Status	EPA Permit Number	EPA Permit Expiry	Operating Permit Number	Operating Permit Expiry	Notes
Sewum	PL2/424	32.55	Cape Coast Resources Ltd	31 May 2023 (Extension in progress; awaiting Minister's approval / signature)	EPA/HQ/MEI/ 0029290/24/SE	16 January 2026 (renewal in progress)	012390/25	31 December 2025 (renewal in progress)	Letter from MINCOM to Minister recommending extension to 2026 received. Operating permit renewal in progress and contingent on extension of PL. AMR and stool land fees for 2026 paid.
Enkye	PL2/404	34.65	Cape Coast Resources Ltd	31 May 2026 (Extension granted on 30 April 2024 for a further 3 years)	EPA/HQ/MEI/ 0029288/24/SE	16 January 2026 (permit renewal fees paid on 11 February 2026)	012391/25	31 December 2025 (renewal in progress)	Confirmation letter from Minister approving extension to 2026 received. Operating permit renewal in progress. AMR and stool land fees for 2026 paid.
Nyamebekyere	PL2/406	35.91	Cape Coast Resources Ltd	31 May2023 Extension in progress; awaiting Minister's approval / signature)	EPA/HQ/MEI/ 0029289/24/SE	16 January 2026 (renewal in progress)	012387/25	31 December 2025 (renewal in progress)	Letter from MINCOM to Minister recommending extension to 2026 received. Operating permit renewal in progress and contingent on extension of PL. AMR and stool land fees for 2026 paid.

Deposit Name	Prospecting Licence (PL) Number	PL Area (km^2)	Holding Company	PL Expiry Date / Status	EPA Permit Number	EPA Permit Expiry	Operating Permit Number	Operating Permit Expiry	Notes
Yehikwawkrom	PL2/405	29.82	Cape Coast Resources Ltd	31 May 2026 (Extension granted on 30 April 2024 for a further 3 years)	EPA/HQ/MEI/ 0029287/24/SE	16 January 2026 (permit renewal fees paid on 11 February 2026)	012389/25	31 December 2025 (renewal in progress)	Confirmation letter from Minister approving extension to 2026 received. Operating permit renewal in progress. AMR and stool land fees for 2026 paid.
Abotia	PL2/119	25.83	Cape Coast Resources Ltd	10 April 2026 (Extension application submitted)	EPA/HQ/ME1/0 63650/25/SE	19 November 2027	012388/25	31 December 2025 (renewal in progress)	PL Extension application in progress and approval expected. Operating permit renewal in progress. AMR and stool land fees for 2026 paid.
Omanpe	PL2/436	32.13	Cape Coast Resources Ltd	10 April 2026 (Extension application submitted)	EPA/HQ/ME1/0 63639/25/SE	19 November 2027	012386/25	31 December 2025 (renewal in progress)	PL Extension application in progress and approval expected. Operating permit renewal in progress. AMR and stool land fees for 2026 paid.
Nyame Esa	APL-1895	24.36	Boin Resources Ltd	Notice of grant of PL received on 11 September 2024	Pending	Pending	Pending	Pending	Former portion of Nyamebekyere PL. Confirmation letter from Minister approving Licence received but Licence not yet issued.

Deposit Name	Prospecting Licence (PL) Number	PL Area (km^2)	Holding Company	PL Expiry Date / Status	EPA Permit Number	EPA Permit Expiry	Operating Permit Number	Operating Permit Expiry	Notes
									EPA and operating permits contingent on receipt of PL
Nkwanta	APL-1892	30.87	Boin Resources Ltd	Notice of grant of PL received on 11 September 2024	Pending	Pending	Pending	Pending	Former portion of Sewum PL. Confirmation letter from Minister approving Licence received but Licence not yet issued. EPA and operating permits contingent on receipt of PL
Anguzu	APL-P-510	1.89	Boin Resources Ltd	Notice of grant of PL received on 11 September 2024	Pending	Pending	Pending	Pending	Former portion of Sewum PL. Confirmation letter from Minister approving Licence received but Licence not yet issued. EPA and operating permits contingent on receipt of PL.

20.1.5 Status of Environmental and Social Studies

The Project is currently at the PFS stage. Prior to this stage, preliminary environmental and social baseline studies were undertaken in 2015 by Kings Environmental Resource Management Consultancy (KERMC) and, in August 2022 by Abbakus Geosocial Consult (AGC) Ltd. Updated baseline studies were further undertaken by AGC in August 2024 and April 2025 with expanded monitoring locations. Site visits undertaken as part of the 2026 PFS and earlier baseline studies were used to gain a general understanding of field conditions, identify the Project area of influence, and establish the physical, biological, socio-economic and cultural setting.

Additional studies undertaken as part of the PFS to update the baseline information and knowledge base of the Project include:

- Socio-economic studies to support an Abbreviated Resettlement Action Plan (ARAP) comprising of:
  - Community Asset and Project Affected Persons (PAPs) Survey
  - Community Engagement and Stakeholder Consultation.
- Hydrogeological study involving a hydro census of the project area and conceptual modelling.

The most recent baseline studies covered the following environmental and social aspects:

- **Background air quality and noise levels** – Air quality and noise levels were measured at 8 locations in April 2025, including locations monitored in previous studies. The locations include Kangaboin (Boin) Basic School (AN1), Sewum Hospital (AN2), Kwakyekrom (AN3), Newcore Residence at Enchi (AN4), Main Gate of Ndaa Akua Hotel at Enchi (AN5), St Michael Anglican Basic School at Morcherkrom (AN6), Amonie Primary and JHS (AN7), and Tokosue M/A Primary (AN8).
- **Soil characteristics** – Soil samples were taken from seven locations in April 2025, including locations where samples were previously collected. The sampling locations include Amonie (AMO), Tokosue (TK), Boin (BOI), Kwakyekrom (KWA), Mocherkrom (MO), Nyamebekyere (NYA), and Sewum (SW). For each location, eight samples were taken (split between topsoil and subsoil), except at NYA where six samples were taken. A total of 54 samples were sent for laboratory analysis of quality parameters.

- **Water and sediment quality** – Surface water and sediment samples were collected from 10 locations in April 2025, including locations sampled in previous studies. The locations include Gyanae at Nyamebekyere (S1), Asuo Kofi at Kwakyekrom (S2), Edwe Downstream at Abokyea-Kogyina (S3), Boin at Boinso (S4), Kangaboin Stream (S5), Edwe Upstream near Kangaboin (S6), Disue Downstream at Achimfo (S7), Disue Upstream at Yakasi (S8), Subri at Amonie (S9), and Nyamesa Stream (S10). Groundwater samples were also taken from boreholes at Kwakyekrom (BHW1), Edawanim near Kangaboin (BHW2), Nkwanta (BHW3), Alatakrom (BHW4), Sewum (BHW5a & BHW5b), and Mokyekrom (BHW6). All samples were sent for laboratory analysis of quality parameters.

- **Aquatic and terrestrial biodiversity** – An ecological study was undertaken in April - May 2025, focusing on aquatic and terrestrial flora and fauna distribution in the Project area and establishing trends against previous studies. Sixteen vegetation sampling plots of 20 m radius were selected from a mix of vegetation types. The fauna survey employed various methods, including opportunistic observations, walking along transects, live-trapping of small mammals, and interviewing hunters and farmers in communities within the Project area. The aquatic ecology study was carried out at the 10 water quality sampling sites and included confirming catchment characteristics (e.g., stream flow, riparian vegetation, and anthropogenic activities), experimental fishing, informal interviews with local fishermen, and laboratory analysis of field specimen.

- **Socio-economics and cultural heritage** – The studies, conducted in April 2025, built on the information collected in previous studies. The study focused on eight communities within the Project catchment: Sewum, Achimfo, Kwakyekrom, Kangaboin, Amonie, Tokosue, Morcherkrom, and Nyamebekyere, as well as eight hamlets identified within the Project area. The study employed mixed method approach, including focus group discussions and key informant interviews. Consultations were held with relevant government institutions and some community members and opinion leaders to obtain their input on the Project and gather other relevant information. Two shrines are located in Kangaboin.

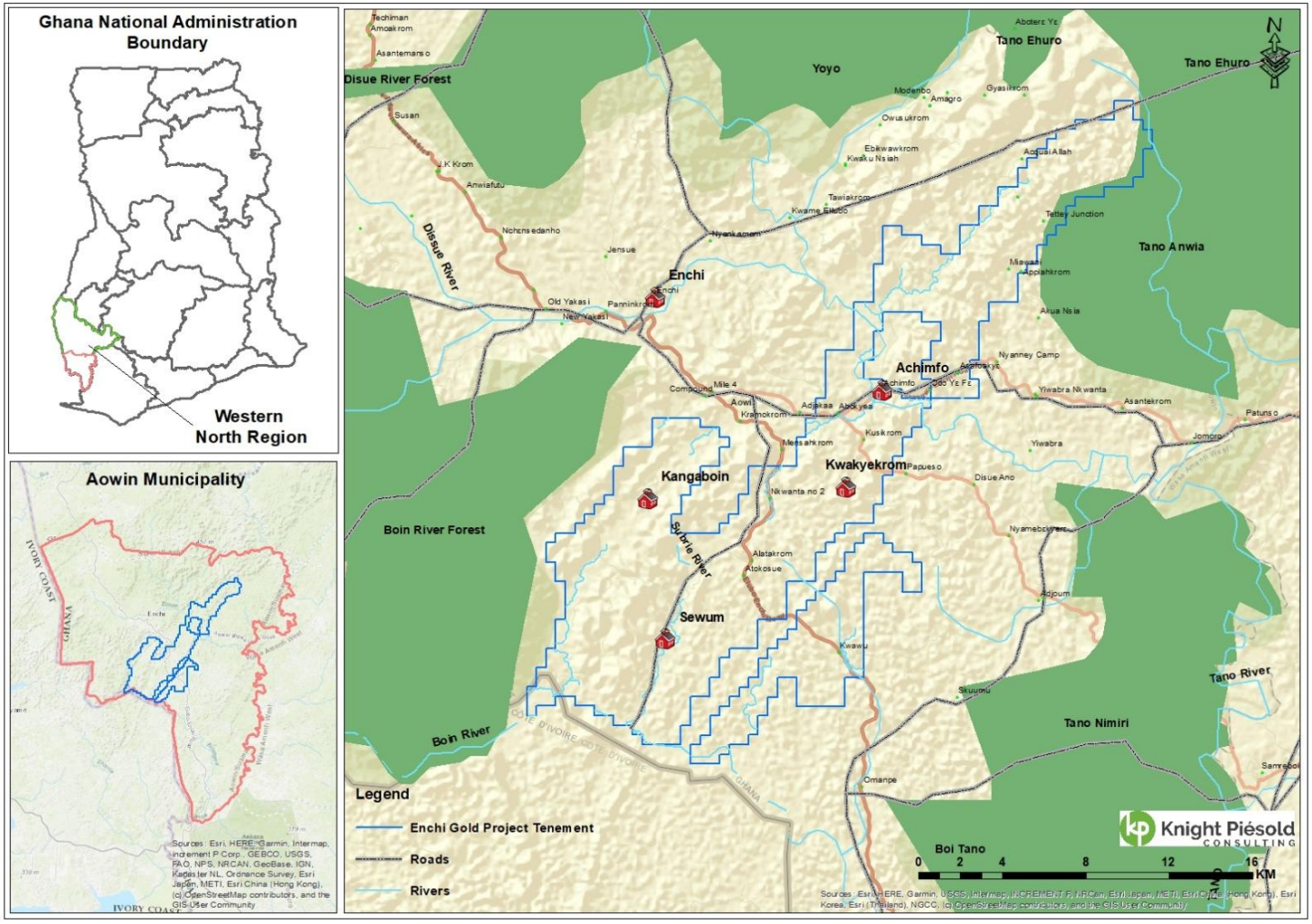
- **Abbreviated Resettlement Action Plan (ARAP)** – A site visit and social survey of Kangaboin (Boin) community was undertaken in February 2026. The community, along with the hamlet settlement are located in proximity of the earmarked location for the Boin Pit, Tailings Storage Facility (TSFs) and process plant. To provide access for development of the Boin pit, the Kangaboin (Boin) community would require relocation. The ARAP also notes that other communities (Sewum, Kwakyekrom and Nyamebekyere) may be impacted by future production. A more detailed analysis of relocating these communities will be investigated in further stages of the Project.

- **Hydrogeological Study** – A site visit and hydro census was undertaken in February 2026 to support development of a preliminary mine-wide hydrogeological model for the Project.

The baseline studies have not identified environmental compliance barriers to Project development, however, recommendations for additional studies have been made to determine seasonal differences or longer-term trends in environmental and social conditions. The baseline studies noted some species of conservation importance within the wider Project area, as well as increased illegal and unregulated artisanal and small-scale mining (ASM) activities within the area that is negatively impacting waterbodies and riparian vegetation. These would necessitate biodiversity and ASM management plans to be developed for effective management during construction and operation of the mine.

A detailed EIA is expected to be undertaken at a later stage of the Project and an EIS submitted to the Ghanaian EPA for decision making on the grant of Environmental Permit to cover the Project.

Figure 20.1.1 Environmental and Social Context of the Enchi Gold Project



20.1.6 Environmental and Social Context

Location and Landscape

The Project covers a total geographical area of 248 km^2 along a 40 km strike length and is located in southwestern Ghana, specifically the Aowin Municipality in the Western North Region. It is bordered by five Districts: Amenfi West Municipal to the east, Jomoro Municipal to the south, Suaman District to north-west, and to north-east by Sefwi Akontombra District and Sefwi Wiawso Municipal (AGC, 2025b). The operating Chirano Gold Mine is located approximately 50 km to the north of the Project, and there are numerous prospecting licences held by other exploration companies in the vicinity.

The Project landscape is characterized by undulating topography with an average elevation of around 300 metres above mean sea level (masl). The Project site exhibits lush, green hills and gently rolling terrain. Its mountainous topography is defined by steep slopes, prominent ridges, and intervening valleys (Figure 20.1.2). The drier upper and middle slopes are mostly cultivated and there is an extensive drainage network. Secondary forests or thickets intersperse the landscape, giving it a mosaic / fragmented appearance. The existing vegetation of the Project area is a mixture of farmlands, farm re-growths / secondary thickets, secondary forest with broken canopy, and freshwater swamp / wetland (AGC, 2025b).

Artisanal mining is predominant in the Project area which has transformed the natural topography / landscape (Figure 20.1.2). These workings vary in size and depth, and as a result, the land is left with uneven surfaces, voids, and weakened structural integrity that could pose hazards for future development.

Figure 20.1.2 Typical Natural Landscape of the Project Area



Figure 20.1.3 Typical Artisanal Mining Activity in the Project Area



Climate

The Project area has a wet semi-equatorial (tropical savannah) climate, with high rainfall, high humidity, and medium temperature. It experiences two rainy seasons, with the main season from March to July and a minor season from September to November. Rainfall events are often intense, torrential, and stormy, with high runoff and associated erosion. Strong harmattan winds affect the Project area in the dry season during November to March (AGC, 2025b) but winds are otherwise generally light (KERMC, 2015).

Annual rainfall for the Project area ranges between 1,217 mm - 2,076 mm, based on a 20-year rainfall record (2003 to 2023) from the synoptic weather station at Sefwi-Bekwai, which is within 70 km (bird's eye) of the Project area. Average annual rainfall for the period is 1,517 mm. Mean monthly relative humidity ranges from 66.5% to 82.1%, and average monthly temperature ranges from 26.4°C to 29.8°C (AGC, 2025b). Exploration and mining operations can be conducted on the Project area year-round.

Water Resources

Surface Water

The Project area is primarily drained by the Disue River to the north and the Boin river to the south (Figure 20.1.4). Both rivers are tributaries of the transboundary Tano River Basin, which has a total catchment area of around 15,000 km^2 (shared with Côte d'Ivoire). Several smaller tributaries also exist within the Project area, including the Dare, Edwe, Yoyo, Subiri, Anyaha, Ajunsuo (Figure 20.1.4). Some of these waterbodies are not well defined or mapped (KERMC, 2015), and several bodies of standing water (wetlands) are also present across the Project area (Figure 20.1.5). Artisanal mining is a prominent observation across the Project area and has created some disruptions to the local hydrology of the area (Figure 20.1.3).

Preliminary baseline studies found that watercourses in the Project area have been historically impacted by erosion, riverbank instability, and sediment deposition. Water quality has been affected by runoff from farms, artisanal mining and contamination from domestic waste and lack of appropriate sanitation (AGC, 2023).

Preliminary water quality sampling found the majority of parameters within acceptable Ghanaian standards, with the exception of widespread turbidity (high sediment load) in surface water. Elevated concentrations of some heavy metals (iron, manganese, cadmium, lead, mercury) were observed in sediment samples which could be attributed to the geological formation of the project area and other anthropogenic activities (AGC, 2023, 2025b). Similar water quality results were obtained during previous analysis (KERMC, 2015).

Figure 20.1.4 Boin River (Left) and Ajunsuo River (Right)



Figure 20.1.5 Range of Wetlands in the Project Area



Groundwater

A hydrogeological study was commissioned as part of the PFS. The work scope involved a hydro census, examination of borehole cores, conceptual hydrogeological model, groundwater numerical flow model, simulation and prediction of groundwater level and geochemistry analysis. The hydro census identified boreholes, rivers, streams, wetlands, and water features across the Project area, with water sampled from monitoring wells and the Subiri River for quality testing. The hydro census identified a complex drainage system with strong surface-groundwater interaction across the Project area. The river systems were found to be largely perennial with groundwater-fed baseflow, with some streams being ephemeral. The Project area is characterized largely by a confined aquifer system and varying groundwater heads with low-permeability confining layer maintaining hydraulic pressure. Preliminary assessments indicate low groundwater heads around the Boin pit and community area.

Preliminary groundwater quality testing found the groundwater of acceptable drinking quality with the majority of parameters within recommended limits per Ghanaian standards. However, iron concentrations were elevated in the single available groundwater monitoring borehole (AGC, 2023). The monitoring scope was expanded in April 2025 with a total of six boreholes sampled. The results confirmed a generally good groundwater quality, with elevated iron concentration in five out of the six boreholes (AGC, 2025b).

Most communities in the Project area depend on groundwater sourced from community boreholes (Figure 20.1.6), however, water is also sourced from streams and rivers.

Figure 20.1.6 Community Boreholes at Kangaboin



Biodiversity

The Project is situated in a region of rich natural biodiversity and is classified as Moist Evergreen Forest type of the High Forest Zone of Ghana (Hall and Swaine, 1981). Figure 20.1.7 shows a typical forest landscape within the Project area.

Terrestrial Flora

Three habitat types were identified in the Project area: cocoa farms with isolated trees, secondary thickets or secondary forests, and wetlands. Much of the forested areas outside of the Project footprint are managed for commercial timber, with typical hardwood species including red ironwood (*Lophira alata*), African whitewood (*Triplochiton scleroxylon*), African mahogany (*Khaya*), and African teak (*Milicia excelsa*). Habitat loss has occurred as a result of timber harvesting and illegal mining activities (AGC, 2023, 2025b).

The Project site is dominated by cocoa farms intercropped with food crops such as plantain and cassava. There is minimal mature forest cover or canopy tree layer, rather, the cocoa farms have isolated trees that provide shade, notably *Triplochiton scleroxylon*, *Terminalia superba*, *Terminalia ivorensis*, *Morinda lucida*, *Persea americana*, *Milicia excelsa*, *Myrianthus libericus*, and *Albizia zygia*. Rubber is also cultivated. Forbes and herbaceous species which constitute the undergrowth of the cocoa farms include *Nephrolepis biserrata*, *Sida acuta*, *Schranckia leptocarpa*, *Mimosa pudica*, *Stachytarpheta cayennensis*, *Passiflora foetida*, and *Asystasia gangetica*. The farm re-growths and secondary thickets and secondary forest have species such as *Chromolaena odorata* (invasive plant), *Ficus sur*, *Funtumia africana*, *Anthocleista djalonensis*, *Solanum torvum*, *Cecropia peltata*, *Musanga cecropioides*, *Albizia zygia*, *Sterculia tragacantha*, and *Griffonia simplicifolia*. Mined-out areas have patchy re-growth or secondary thicket vegetation. The swamp and wetland vegetation has species such as *Raphia hookeri*, *Scleria verrucosa*, *Ludwigia decurrens*, and *Alchornea cordifolia* (AGC, 2023, 2025b).

Preliminary baseline studies conducted have recorded more than 100 flora species across the Project site, dominated by trees from the cocoa farms and secondary forest patches. Other life forms include herbs, climbers, and shrubs. Many of the local trees and plants are used for traditional medicinal purposes and supplementary food (AGC, 2023, 2025b).

Fourteen plant species of national conservation concern were recorded across the Project site, out of which six are of global conservation concern per the 2025 International Union for Conservation of Nature (IUCN) Red List of Threatened Species. The species of global conservation concern include three Near Threatened (NT) species and three Vulnerable (VU) species. The NT species comprise Budongo mahogany (*Entandrophragma angolense*), African teak (*Milicia excelsa*), and Rubiaceae (*Mitragyna stipulosa*), while the VU species comprise *Cedrela odorata*, *Khaya ivorensis*, and *Terminalia ivorensis*. Siam weed (*Chromolaena odorata*) is the most common invasive species in the area (AGC, 2023, 2025b).

Figure 20.1.7 Forest Cover Surrounding the Project Area



Terrestrial Fauna

The natural ecology of the region has been impacted by various human activities, including agriculture, hunting, artisanal mining, firewood collection, and commercial forestry. Consequently, the presence of fauna, and in particular large mammals, is limited in the Project area (KERMC, 2015; AGC, 2023, 2025b).

Various species of mammals, birds, butterflies, amphibians, and reptiles were recorded during the preliminary baseline surveys. More than 100 fauna species were observed across the project site, which includes five species of amphibians, nine species of reptiles, 65 species of birds and 22 species of mammals. Four fauna species are of global conservation significance per the 2025 IUCN Red List of Threatened Species which are the Black-bellied Pangolin (*Phataginus tetradactyla*) that is Vulnerable, and the African Rock Python (*Python sebae*), Bay Duiker (*Cephalophus dorsalis*), and Bongo (*Tragelaphus eurycerus*) that are Near-Threatened. The remaining species were of Least Concern status per IUCN status (AGC, 2025b).

Regarding national conservation status, 29 species are protected by the Ghana Wildlife Conservation Regulation. These include seven species that are completely protected (Schedule I), comprising of one species of reptile (Nile Monitor Lizard), three species of birds (Black Sparrowhawk, Red-necked Buzzard and Yellow-billed Kite) and three species of mammals (Black-bellied Pangolin, Bongo, Western Palm Squirrel). The other 22 species are partially protected (Schedule II), comprising two species of reptile, nine species of birds, and 11 species of mammals (AGC, 2025b).

Bird species common in the project area include the Village Weaver (*Ploceus cucullatus*), Bronze Mannikin (*Permestes cucullata*), Barn Swallow (*Hirundo rustica*), Orange-Cheeked Waxbill (*Estrilda melpoda*), Pied Crow (*Corvus albus*), Red-Eyed Dove (*Streptopelia semitorquata*), and Common Bulbul (*Pycnonotus barbatus*). Amphibians that are common include Crowned Bullfrog (*Hoplobatrachus occipitalis*), Common Frog (*Amnirana galamensis*), Variable Reedfrog (*Hyperolius concolor*), Flat-Backed Toad (*Sclerophrys maculata*), and Common Toad (*Sclerophrys regularis*). For reptiles, the Agama Lizard (*Agama agama*), Nile Monitor Lizard (*Varanus niloticus*), Orange-Flanked Skink (*Trachylepis sp.*), Forest Hinge-Back Tortoise (*Kinixys erosa*), and Western Green Mamba (*Dendroaspis viridis*) are common species in the area (AGC, 2023, 2025b).

Mammals such as the African Pygmy Mouse (*Mus musculoides*), Rusty-Bellied Brush-Furred Rat (*Lophuromys sikapusi*), Grasscutter (*Thryonomys swinderianus*), Marsh Mongoose (*Atilax paludinosus*), Maxwell's Duiker (*Philantomba maxwellii*), Royal Antelope (*Neotragus pygmaeus*), Striped Ground Squirrel (*Euxerus erythropus*), and Western Palm Squirrel (*Epixerus ebii*) are common in the area. Interviews with local hunters suggested that some large mammals such as the Bay Duiker (*Cephalophus dorsalis*), Bongo (*Tragelaphus eurycerus*), Common Warthog (*Phacochoerus africanus*), and Black-Bellied Pangolin (*Phataginus tetradactyla*), which were previously common in the area, have moved further away due to pressures from deforestation and hunting (AGC, 2023, 2025b).

Aquatic Ecology

Generally, streams and rivers in the Project area, including the riverine forest ecology, have been disturbed by agricultural and illegal artisanal mining activities. Aquatic flora and fauna were sampled at the same locations as the water quality sampling. Macroinvertebrates, zooplankton, and phytoplankton diversity and abundance were studied. Experimental fishing was also undertaken, and local fishermen were interviewed using a picture fish identification guide (Holden and Reed, 1972) to obtain additional information on ichthyofauna (AGC, 2023, 2025b). Fish species recorded were of IUCN Least Concern. Species of relatively high abundance comprise *Brycinus nurse*, *Oreochromis niloticus*, *Pellonula sp.*, *Mormyrid spp.*, *Barbus occidentalis*, *Tilapia niloticus*, *Tilapia aurea*, *Tilapia zillii*, *Malapterurus electricus*, *Citharinus citherus*, *Chrysichthys nigrodigitatus*, *Lates niloticus*, *Hemichromis fasciatus*, *Labeo coubie*, and *Clarias lazera* (AGC, 2023, 2025b).

The most abundant phytoplankton species recorded comprise *Chaetophora sp.*, *Phacus sp.*, *Euglena mutabilis*, *Trachelomona sp.*, *Hydrodicton sp.*, *Asterionella sp.*, *Chaetoceros sp.*, *Oscillatoria sp.*, *Favella sp.,* and *Trachelomonas hispida*. The observed zooplankton groups are primarily from the taxa Copepoda, Amphipoda, Cladocera, and Ostracoda. The most common or abundant taxa was Copepoda which was recorded at almost all sampling locations. Macroinvertebrates also showed low diversity in the most recent studies compared to previous studies with a complete loss of taxa and specimens at several sampling locations, indicating ecological disturbance (AGC, 2023, 2025b).

The baseline studies conducted over the years reveal a general declining trend in diversity and abundance of all species (macroinvertebrate, zooplankton, phytoplankton, and ichthyofauna). The results indicate prevailing ecological stress or imbalances potentially due to pollution, altered nutrient levels, predation, and negative impacts from artisanal mining activity.

Land Use and Protected Areas

The Project is located in between Boin National Park and Boin Tano Forest Reserve (Figure 20.3.1). Adjacent to the Boin Tano Forest Reserve is the Jema-Asemkrom Forest Reserve, which has been designated as a Key Biodiversity Area (KBA) of international significance. Additional nearby protected areas include the Tano Anwia and Tano Nimiri Forest Reserves, and the Nini-Suhien National Park (part of the Ankasa Conservation Area KBA). There are several designated International Bird Areas (IBAs) in the vicinity of the Project which correspond with forest habitat.

Land use in the Project area is predominantly small-scale farming with cocoa being the widely cultivated cash crop. Other crops include plantain, cassava, yams, pineapple, tomatoes, pepper, and garden eggs. Artisanal mining is also active in the Project area leading to negative impacts on vegetation and waterbodies, as well as incidents of farmers giving lending out their farmlands for mining. It is noted that a colonial mining shaft runs through Sewum, Morcherkrom, Tokosue and Achimfo which been mined over the years by local people (AGC, 2025b). Site visit observations indicate that artisanal mining has, in some instances, followed areas where Newcore completed drilling.

Commercial timber production is also undertaken in nearby forested areas and the wider region as the Municipality is host to eight forest reserves out of which six are productive reserves with active timber harvesting. However, the forest reserves are threatened by increasing exploitation, illegal mining, sand winning activities, and general land encroachment (AGC, 2025b).

Hunting is also practiced to a limited extent as most animals have reportedly moved out due to increasing deforestation, hunting pressure, and illegal mining activities (Adomako, 2025; AGC, 2025b).

Geology and Soil

The geological profile of the Project area is defined by an extensive oxide layer overlying zones of transitional material and bedrock. Oxide and transition layers vary in thickness from approximately 30 metres to depths exceeding 100 metres. These conditions indicate that the near-surface soils are compressible and may experience settlement when subjected to loading. Valley areas may also have layers of alluvium deposits which are known to be compressible.

The main soil types in the Project area are Acrisols and Ferralsols, interspersed with Fluvisols. These soils are mainly loamy, rich in humus, and suitable for a variety of crop production. However, soil fertility is reported to be declining in recent years from continuous farming without adequate fallow periods or soil improvement practices. Baseline soil sampling determined that metal loading was generally low and within acceptable limits. Soil pH was generally acidic and in the range of 4.97 to 6.22, with considerable organic matter content, although a depletion of plant nutrient in the upper soil profile and layers was apparent (AGC, 2023, 2025b). Illegal mining activities could be contributing to declining soil fertility in the Project area.

Air Quality, Noise and Vibration

There is minimal industrial activity in or near the Project area that is likely to affect local air quality, and noise (KERMC, 2015). However, there are existing exploration activities and several residential settlements which are sources of air and noise emission. Localized dust, noxious emissions, and noise generation associated with vehicular traffic and drilling activities occur on a periodic basis.

Preliminary air quality baseline studies indicate that air quality in the Project area is generally acceptable and complies with national standards (GS 1236:2019) and international guidelines (WHO and IFC). However, some occasional instances exceeding recommended standards were noted in dust (particulate matter) levels which are likely attributable to dusty conditions on unpaved roads in the Project area (AGC, 2023, 2025b), particularly during the dry seasons.

Preliminary noise baseline studies indicate acceptable daytime and nighttime noise levels in the Project area, with occasional high noise levels attributable to existing community life or human activities in the Project area (AGC, 2023, 2025b). Vibration monitoring has not been undertaken, but baseline levels are presumed to be low, except in the immediate vicinity of exploration drilling.

Socio-Economic Setting

The Project area is adjacent to the border with Côte d'Ivoire and is situated 290 km west of Ghana's capital Accra, 170 km south of the city of Kumasi (the commercial centre for Ghana's mining industry), and approximately 200 km northwest from Takoradi port. The Project is located in the Aowin Municipality with the town of Enchi as the administrative capital. Several small villages are within the Project area, with the closest town being Enchi which is located approximately 10 km to the northwest. Other communities include Sewum, Achimfo, Kwakyekrom, Kangaboin, Atokosue, Abokyia, Nyamkamam, Mocherkrom, Amonie, and Nyamebekyere. Widespread artisanal mining activities in the project area have also given rise to informal settlements and hamlets.

The total population of Aowin Municipality, according to the 2021 Population and Housing Census (PHC), is 129,721, with more males 68,236 (52.6%) than females 61,485 (47.4%), and a young population of about 68.7% within an age range of 15 to 39 years old. In terms of population dynamics, Sewum's population is approximately 6,100 (60% male, 40% female), Achimfo's population is approximately 4,050 (65% female, 35% male), Kwakyekrom's population is approximately 860 (65% female, 35% male), Kangaboin's population is approximately 664 (60% male, 40% female), and Amonie's population is approximately 980 (60% male, 40% female). Tokosue has an estimated population of 750 (60% male, 40% female), Morcherkrom's population is approximately 1,170 (60% male, 40% female), and Nyamebekyere has a population of approximately 302 (60% male, 40% female) (AGC, 2025b).

Several communities, namely (Kangaboin, Sewum, Kwakyekrom, Nyamebekyere) are located within the Project footprint and may be impacted by infrastructure including mining areas and pits, waste dumps, TSFs, and the process plant. The Kangaboin community is located within proximity of the Boin pit and waste dump and would be impacted by development, resulting in the need for relocation.

Kangaboin (Boin) community residents are mainly migrants of the Ewe tribe, with common household sizes of 5 to 10 people. There are approximately 300 buildings in the Boin community, mainly mud and brick houses roofed with corrugated aluminum sheets. The Boin community has electricity, two community boreholes, three churches, and a school with enrolment of approximately 135 students. There are also approximately 21 hamlet settlements and farmsteads currently anticipated to be impacted by construction and Project development. The hamlet dwellers, largely migrants from the Northern tribes of Ghana, have typical household sizes of 2 to 21 people (Figure 20.1.8.). The hamlets have no electricity and source water from rivers and streams within the tenement.

The Project area is rural, and the predominant economic activity is subsistence and small-scale agriculture, with land areas cleared for farming by slash and burn methods. Crops grown include cocoa, maize, cassava, plantain, cocoyam and vegetables. Individual farmland sizes range from 5 acres to 30 acres. An estimated 90% of land is under traditional ownership with land acquired through the chiefs. Farmlands are mostly owned by individuals while others engage in the traditional sharecropping system ("Do ma y3n ky3" in the local parlance). Small scale livestock rearing also takes place, with fish farming also being gradually introduced. Small-scale artisanal mining (known locally as 'Galamsey') is a growing source of income for some households and is widespread throughout the region, including within the communities of Sewum, Atokosue, Abokyia and Nyamkamam. Trading is also an important economic activity in the municipality with special days set aside as market days – Enchi on Wednesdays, Boinso on Tuesdays, Omanpe and Achimfo on Thursdays, and Sewum on Fridays (AGC, 2023).

There are five schools in the Project area; two schools in Sewum two at Achimfo, and one at Kangaboin. Large healthcare facilities are located in the town of Enchi, with two mobile clinics also located in the communities of Sewum and Achimfo. The project area lacks healthcare facilities, with some community members relying on traditional healers for healthcare as they are believed to be less expensive and more easily accessible. There is a high prevalence of malaria and diarrhoea due to poor sanitary conditions. There are few public toilets and pit latrines (often shared), making open defecation and disposal of refuse prevalent (AGC, 2025b).

There is an available workforce in the region comprised of both skilled and unskilled labour, due to the long history of exploration and mining. Newcore maintains good relations with the local communities. Resettlement of the Kangaboin community and PAPs will be guided by the ARAP and applicable national regulations (Knight Piésold, 2026).

Figure 20.1.8 Typical Hamlet Structures



Cultural Heritage

The Project area is governed by the Aowin Municipal Assembly, which includes a Traditional Council at Enchi. Traditional leaders work with the administrative authorities to represent the various communities and indigenous people (Brusas). Communities in the Aowin Municipality are all under the Paramount Chief of the Aowin Traditional Council who superintends over the chiefs and odikros (sub-chiefs) of the various communities. The dominant ethnic group in the Municipality are the Brusas / Brossas with cross mixture of Sefwi, Bono, Ahantas, Asantes and Nzemas. There are also other ethnic groups from the northern and other parts of the country (AGC, 2025b).

Christianity is the dominant religion (77.8%), with the Islamic religion also prevalent (13.9%). Less than two percent are either traditionalists (0.4%) or belong to other religions (1.0%), with the remainder agnostic (6.9%). There are approximately 20 church buildings in Sewum, 15 in Achimfo, four in Kangaboin and two in Kwakyekrom (AGC, 2025b).

Two shrines are located in the Kangaboin community (Figure 20.1.9). It is likely that scattered individual burial sites may exist, in addition to more formal cemeteries (KERMC, 2015).

Newcore has implemented a Chance Find Procedure during exploration activity and will facilitate a more detailed cultural heritage study with an archaeologist in the future.

Figure 20.1.9 Shrines at Kangaboin



Traffic, Transport and Supporting Infrastructure

The Project is located in a rural area with limited infrastructure. Services and supplies, including accommodation, food and fuel, are available in the nearby town of Enchi. There is a health clinic at Sewum and a hospital in the town of Enchi. Groundwater boreholes serve as the main source of potable water, complemented by water tankers.

Roads in the Project area are mostly unpaved, hence erosion and dust generation are notable, the latter causing hazy driving conditions and damage to vegetation. Access roads will require regular maintenance due to seasonal heavy rainfall and runoff (Figure 20.1.10).

Figure 20.1.10. Condition of Access Roads to the Project Sites



The N12 highway runs north-south past Enchi and bisects the Project area, with villages or communities located on either side. The N12 connects to the N1 paved highway in the south and provides access to seaports at Takoradi and Tema and the international airport in Accra.

There is electricity supply available via the national grid, though blackouts are common. Newcore plans to establish a more reliable and sustainable energy supply for mine operations. There appears to be a lack of municipal facilities for domestic waste management (garbage and recycling) in the area. The area also benefits from a fixed telephone line and mobile phone service tower. Mobile cell service exists over much of the Project area.

20.1.7 Environmental and Social Risk and Impacts

Impacts and Risks

The Project will be designed to minimize negative environmental and social impacts and enhance positive impacts and opportunities. A detailed assessment of potential environmental and social impacts will be undertaken as part of the future ESIA process. Based on the current Project design, findings from preliminary baseline studies, and gold mining operations in similar environments, the main potential risks and impacts of the Project are anticipated to include the following:

- **Natural Hazards** – The Project is located in a heavily vegetated area interspersed with areas of exposed ground, steep slopes, ridges, and intervening valleys. The area also experiences heavy rainfall with illegal and unregulated ASM activities also prominent. The area is likely to be vulnerable to hazards such as forest fires, flash flooding, and landslips. The existing state of local infrastructure, for example overhead power lines surrounded by overgrown vegetation and poorly maintained roads, add to these risks.

- **Air Quality, Noise, Vibration and Artificial Lighting** – The Project will generate air emissions in connection with mineral processing activities and the operation of vehicles and equipment. Local noise and vibration will be influenced by routine mining activities, including drilling, blasting, and crushing. Artificial lighting will also be required for work at night. These activities will cause disturbance to local communities and wildlife, particularly birds, and will need to be monitored.

- **Water Resources** – Project operations have the potential to negatively impact water quality via stormwater runoff and drainage that comes into contact with processing areas, potential seepage (from waste material and TSF) and accidental spills and leaks. The risk of acid rock drainage and metal leaching (ARDML) and potential upward groundwater flux through the TSF base may be present and will need to be verified through geochemical and hydrogeological studies and modelling.

- **Biodiversity** – The Project is located in close proximity to a number of forest reserves, national parks, key biodiversity areas (KBAs) and important biodiversity areas (IBAs). Some species of national and global conservation importance have been identified during preliminary baseline studies. As a greenfield project, vegetation clearing is anticipated during construction of the Project infrastructure and access roads, including safety clearances for overhead power lines. Surface water abstraction and stream diversions may also be required. There will also be disturbance from artificial lighting, noise, and the presence of vehicles and workers during construction. Various project activities will cause disturbance to terrestrial and aquatic ecosystems and life forms. Although the area is a modified habitat, the secondary thickets and farm re-growths provide habitat for various fauna species, including migratory birds. There will be habitat loss from vegetation removal, including loss of any ecosystem service benefits derived from the Project site.

- **Soil and Land Use** – The dominant land use in the project area is agriculture. The area is host to large cocoa farms intercropped with various food crops. The Project will result in the loss of vast fertile lands that support agriculture and other non-mining uses. The transition to mining land use could result in soil contamination and loss of fertility through various mining-related activities, including potential seepages and hydrocarbon spills.

- **Socio-Economic Impacts** – Overall, the Project is expected to have a positive impact on the local and regional economy, through creation of direct and indirect jobs and associated training opportunities. Details of job opportunities will be refined as the Project progresses beyond the PFS stage and priority will be given to hiring and procurement from local communities.

Some negative socio-economic impacts may also occur which includes the following:

- Some local community members or Non-Governmental Organizations (NGOs), particularly those linked to international biodiversity conservation, may oppose a new mining Project in the area.

- There is potential for damage to or loss of cultural heritage sites during exploration and construction, especially where defined accesses and tracks are not used or there is inadequate community consultations. This has the potential to escalate opposition to the Project. The shrines at Kangaboin are likely to be impacted by the Project.

- The Project will result in involuntary resettlement of the most impacted communities and settlements by key infrastructure (i.e., open pits, waste dumps, etc.).

- There is potential for conflict with artisanal miners if they are not considered as relevant stakeholders and adequately consulted, regardless of their perceived or actual legal status.

- Multiple exploration companies have been involved in the Project area over the years, and this may generate unrealistic long-term expectations from the local communities and an overreliance on donations from Newcore.

- The Project is located at the border with Côte d'Ivoire and may be affected by transnational and transboundary issues such as political tensions, birds and wildlife migration routes, and transboundary rivers and tributaries (e.g., River Tano) which drain within the Project area.

- There are old underground mine workings and historical tailings deposits located in the vicinity of the Project, which pose safety and environmental risks. Whilst not directly related to the Project, this historical infrastructure could cause issues with ground stability and water movement and quality.

- The Project is located in close proximity to several community settlements, with many local residences and businesses situated along key access roads. There is also high pedestrian traffic, including schoolchildren and livestock movement. The lack of defined pavements and sidewalks, and the potential need to upgrade and widen these roads, presents a safety risk to these communities. Additional risks will be associated with heavy goods vehicles regularly delivering Project supplies, in particular processing reagents.

Mitigation Measures

The potential environmental and social risks and impacts associated with the Project are considered typical of similar exploration and mining projects in Sub-Saharan Africa. Newcore will incorporate sufficient environmental and social protection measures into the project design to minimize or mitigate negative impacts from the Project. Some mitigation measures that will be implemented include the following:

- Develop and implement a comprehensive Environmental and Social Management System (ESMS) for the Project.

- Implement proactive and transparent stakeholder engagement on a continuous basis throughout the life of the Project.

- Establish or maintain a proactive and supportive senior management team and allocate sufficient resources to specialists and staff for environmental monitoring and community relations.

- The Project design will carefully consider natural hazard risks and incorporate climate-resilient systems.

- In addition to embedded engineering controls, operational controls such as dust suppression and good driving practices will be used to minimize dust generation and greenhouse gas emissions.
- The Project will operate as a closed loop water system, and the Project design will incorporate sufficient water storage and management facilities to promote safe operation during rainy seasons.
- The Project design will take a holistic, site-wide approach to water management and ensure sufficient seasonal and emergency storage is incorporated into the Project design, as well as provision for water treatment.
- Maintain close collaboration with relevant regulatory authorities and conservation bodies to agree on measures for any necessary biodiversity offsets and ensure disturbance to flora and fauna of conservation importance is minimized or adequately compensated for.
- The design of the TSF will incorporate international guidelines and include appropriate liners, drainage, and monitoring systems to minimize the risk of seepage to groundwater.
- The Project design will incorporate birds and wildlife deterrents at key infrastructure areas such as the TSFs, and water dams to minimize fauna deaths.
- Implement appropriate Chance Find Procedures and a Cultural Heritage Management Plan to mitigate potential loss of cultural heritage.
- Develop a strategic and collaborative approach to community investment, with a focus on partnership funding, capacity building, local empowerment and business diversification that will be sustainable beyond the life of the Project.
- Monitor historical mine workings and tailings deposits in the vicinity of the Project and collaborate with local authorities and regulatory agencies to identify and implement measures to responsibly decommission the mine and mitigate any cumulative environmental impacts.
- Implement appropriate speed limits and restrict the use of community roads by heavy mobile equipment during construction and operation to limit safety and traffic risks within local communities.
- Implement a RAP to manage potential resettlement and compensation of people who will be involuntarily displaced by the Project (e.g., the Kangaboin community and hamlets).
- Allow a buffer zone of 100 metres from roads, forest reserves, streams, rivers, and railway lines, as well as 300 metres buffer from pylons.

20.1.8 Environmental and Social Management

Environmental permits issued in association with Prospecting Licences for all exploration areas under the Project are subject to various conditions, as listed in each permit. These include:

- Implementing site-specific management measures for air quality, noise levels, protection of water resources, minimizing land disturbance, and promoting health and safety.
- Avoiding environmental and culturally sensitive areas.
- Provision of a reclamation bond for rehabilitation of disturbed areas.
- Compensation payments for any damage to buildings and infrastructure.
- Following specific criteria for community consultation and land access.

In addition to Ghanaian legislative requirements and permit conditions, Newcore will adopt a proactive approach to environmental and social management for development of the Project. This approach includes:

- Regular review of all aspects of Project design to ensure environmental protection is prioritized.
- Adapting the Project design to incorporate new and more sustainable technologies as they become available.
- Protecting and conserving water resources.
- Minimizing and responsibly managing non-mining waste products.
- Incorporating renewable energy and decarbonization options as they become viable.

Newcore has installed a groundwater monitoring borehole and intends to expand its long-term monitoring network as the Project develops, in addition to supporting additional third-party monitoring studies.

Newcore has undertaken community investment in the Project area to date, including the donation of medical equipment and educational supplies, practical assistance with road improvements and potable water supply, and refurbishment of a local school and sports ground. Newcore will manage community expectations as the Project advances and continue with the strategic and collaborative approach to community investment.

At the conclusion of exploration and during the Mining Lease application, a provisional Environmental Management Plan (EMP), which includes social considerations, will be submitted with an EIS to EPA for the grant of an environmental permit for mining. Once mining operations commence, a comprehensive EMP will be submitted and updated every three years, in a format prescribed by the EPA. Newcore will also submit periodic (monthly and annual) environmental reports and operational records to EPA and MINCOM during mining operations.

Prior to commencing mining operations, Newcore shall implement an ESMS and Occupational Health and Safety Management System (OHSMS) that aligns with International Organization for Standardization (ISO) requirements, such as ISO 14001 and ISO 45001. Newcore shall develop standard operating procedures (SOPs) and provide adequate training to staff and contractors in their respective functions or responsibilities.

20.1.9 Provisional Closure Planning

Responsible closure planning should be integrated into all phases of the Project and undertaken in compliance with Ghanaian legislative requirements and GIIP.

Legal Requirements

The legal and regulatory framework for Project closure includes the requirements of both national and international legislations that govern the Project. Conditions set out in the environmental permits associated with the Prospecting Licences for the Project, as well as closure-related requirements of the various Ghanaian legislation discussed in Section 20.1.1, will apply to the Project.

Additional requirements for mine closure prescribed by various international standards and guidelines will apply to the Project. This includes the following:

- EHS Guidelines for Mining (WBG, 2007b).
- Integrated Mine Closure: Good Practice Guide, 2nd Edition (ICMM, 2025a).
- Key Performance Indicators: Tool for Closure (ICMM, 2020b).
- ISO 21795:2021 Mine Closure and Reclamation Planning.
- ISO 18504:2017 Soil Quality – Sustainable Remediation.

Closure Vision and Objectives

The closure vision is to leave the mine site in a state that is safe, physically stable, non-polluting, and capable of supporting pre-mining or next land uses to the benefit of catchment communities.

High-level closure objectives for the Project that will help achieve the closure vision include:

- Achieve compliance with national and international legislative requirements.
- Implement GIIP and latest recommendations in mine rehabilitation science as much as practicable.
- Consult local communities and all other relevant stakeholders and obtain buy-in on proposed end land uses.
- Establish a reliable closure knowledge base throughout the life of the Project to support closure planning and guide implementation of closure activities.
- Ensure safety of humans and animals.
- Return disturbed areas to pre-mining land use to the extent possible or to other beneficial land uses.
- Minimize negative environmental and social impacts associated with closure of the mine.
- Enhance positive environmental and social benefits associated with closure of the mine.

Newcore will identify specific closure objectives for the various infrastructure types that will be developed across the mine as the Project advances to detailed design and mining phase.

Closure Options and Measures

Closure options and measures that will be considered during the prospecting and mining phases of the Project are summarized in the following sections.

Exploration and Prospecting Phase

Closure options and measures for the prospecting phase include requirements under Act 703 (Section 37, Subsection 2f - 2g) for the holder of a prospecting licence to:

- Fill back or otherwise make safe to the satisfaction of the Commission, a borehole or excavation made during the course of prospecting operations.
- Unless the Commission otherwise stipulates, remove within 60 days from the date of the expiration of the prospecting licence, a camp, temporary building or machinery erected or installed and make good, to the satisfaction of the Commission, damage to the surface of the ground occasioned by the removal.

- The environmental permits associated with the Prospecting Licences for the Project require that Newcore implement the following measures:
  - Rehabilitate the site to a condition consistent with the pre-existing characteristics and utility of the area
  - Inspect all excavations made during the course of the exploration and backfill and/or seal them
  - Appropriately dispose of all refuse from exploration areas
  - Complete exploration site rehabilitation within three months of abandonment
  - Post a reclamation bond (financial assurance or security deposit).

Other closure conditions may apply once the Project obtains a Mining Lease and operational permit for full-scale mining. Closure of exploration drill holes is particularly important in locations where future processing infrastructure may be located, such as the TSF, where there is a potential risk of seepage to groundwater.

Mining Phase

At the conclusion of exploration and during the Mining Lease application, a provisional Closure and Reclamation Plan will be submitted with an EIS to the EPA. The Reclamation Security Agreement (RSA) or reclamation bond with the EPA or bank will also be updated based on the approved reclamation work plan and cost estimate.

Following the grant of a Mining Lease and supporting permits, Newcore is required to commission mining operations and implement final closure measures at the end of the life of mine before relinquishing the mine. The estimated LOM is currently 9.3 years, and Newcore will implement progressive reclamation as much as practicable during the life of the mine. Newcore will update the Closure and Reclamation Plan regularly throughout the mine life. Newcore will submit a detailed Closure and Reclamation Plan to MINCOM and the EPA detailing plans for decommissioning and relinquishment not later than 60 days before commencing closure operations in line with Regulation 275 of L.I. 2182 and Regulation 41 of L.I. 2504.

Should mining operations be temporarily closed or suspended during the mine life for any reason, Newcore shall, in accordance with Section 51 of Act 703 and Regulation 21 and 22 of L.I. 2173, notify MINCOM and the Minister three months in advance or within 14 days of the suspension and accordingly submit a Suspension Plan.

General closure options and measures for the Project, at the end of mining, will include the following:

- Investigate feasible closure design options for various infrastructure areas (closure domains).
- Conduct early stakeholder consultation to obtain inputs and reach agreements on end land uses for the closure domains.
- Re-contour and stabilize slopes.
- Install safety berms in areas where needed to promote safety and slope stabilization.
- Decontaminate (where needed) and dismantle all mining and processing infrastructure and remove from site.
- Dismantle supporting infrastructure (camp, offices, utilities, etc.) unless handover process to local community is agreed.
- Re-vegetate disturbed areas by applying appropriate soil cover and establishing native plants (including tree species) to promote a self-sustaining vegetation cover and potential return of native fauna.

Closure Monitoring and Maintenance

Newcore will implement post-closure environmental monitoring and maintenance (aftercare) for at least five years or until closure success criteria are met and confirmed by EPA. Monitoring will cover, but will not be limited to, surface and groundwater quality, air quality, soil quality, erosion, slope stability, re-vegetation progress, and social issues and grievances. Care and maintenance activities will cover stabilisation of reclaimed areas, caring for established vegetation, erosion and sediment control, etc.

Closure Cost Estimate

A PFS-level closure cost estimate of approximately $18.5 M was budgeted for physical closure, and reclamation works involving the removal of all infrastructure through dismantling of key components, demolition of concrete and brick structures, and rehabilitating the footprint with native tree species. A summary cost estimate is presented in Table 20.1.3.

The costs are based on professional judgment and high-level estimates of quantities, along with average unit rates of suppliers and contractors from typical works within the region. The cost estimate is highly conceptual and should be reviewed and updated as the Project advances and closure knowledge base, design elements and field survey data become more refined. The closure cost assumes revegetating all decommissioned infrastructure and disturbed areas with native tree species into a secondary forest and plantation land use. Newcore will firm up post-mining land use agreements for the various facilities and infrastructure through stakeholder engagements as the Project advances. It is assumed that all closure and reclamation works will be undertaken directly by Newcore or by a third-party contractor established on site.

Table 20.1.3 Closure Cost Summary

No.	Key Infrastructure / Domain	Estimated Cost US$
1	**Mining**	**2,400,500**
1.1	Primary Mining Demobilization	2,400,500
2	**Decommissioning**	**2,570,000**
2.1	Process Plant, Crushers, and Conveyors	800,000
2.2	Demolition of Reinforced Concrete, Brick Buildings and Structures	665,000
2.3	Tailings Storage Facility Pipelines	500,000
2.4	Power Plant and Powerlines	200,000
2.5	Explosive Plant and Magazines	250,000
2.6	Disposal of Demolition Waste	155,000
3	**Reclamation and Revegetation**	**12,572,000**
3.1	Open Pits	2,200,000
3.2	Waste Rock Dumps	1,800,000
3.3	ROM Pads and Low Grade Stockpiles	800,000
3.4	Tailings Storage Facilities	3,400,000
3.5	Process Plant, Crushers, and Conveyors	1,220,000
3.6	Access and Haul Roads	600,000
3.7	Built Infrastructure Areas	1,452,000
3.8	General Surface and Disturbed Areas	600,000
3.9	Process and Event Ponds	500,000
4	**Post-Closure Monitoring and Maintenance (5 Years)**	**950,000**
4.1	Care and Maintenance of Rehabilitated Areas	200,000
4.2	Environmental Monitoring	750,000
	TOTAL (1 + 2 + 3 + 4)	**18,492,500**
	15% Contingency	**2,773,875**
	GRAND TOTAL	**21,266,375**

20.1.10 Recommendations

Ongoing development of the Project will require a full EIA, supported by additional baseline studies, a public consultation process, and other terms and conditions that may be outlined by regulatory authorities. Recommendations for the next phase of the Project include the following:

- Undertake additional baseline studies to support EIA and engineering designs. Seasonal baseline studies will provide comprehensive pre-project baseline information or knowledge base against which to assess potential impacts and evaluate the Project against national and international requirements. Future studies should consider an Ecosystem Services Benefit (ESB) study and a Biodiversity Management Plan to preserve and conserve species of conservation importance.

- Social baseline studies should include detailed mapping of catchment communities, settlements and hamlets, as well develop a full RAP for potential displacement or resettlement of people.

- Continue with regular monitoring with installed monitoring infrastructure such as a weather and air quality stations at relevant locations.

- Undertake a regional hydro census and water quality testing program to generate a comprehensive dataset to improve confidence in modelling outcomes (e.g., hydrogeological, water balance, solute transport models, etc.)

- Undertake a comprehensive stakeholder engagement program and document all stakeholder interactions to assist the Community Relations team as reference points for future engagements.

- Ensure ongoing community investment is done in a strategic and sustainable way so that any supported Project infrastructure can be independently managed and maintained.

- Integrate sensitive and protected areas into the Global Information System (GIS) used by the exploration team, to minimize the risk for damage to cultural heritage sites and known wildlife habitats.

- Continue to ensure exploration drill holes and trenches are properly sealed, to minimize land disturbance and avoid future problems with water connectivity – establish a formal procedure for this and ensure the closure of all drill sites is properly documented, in particular near the proposed TSF location.

- Update engineering designs, models, analyses, and cost estimates as necessary, and incorporate environmental and social risks.

20.2 Abbreviated Resettlement Action Plan for the Kangaboin Community

Scope of Abbreviated Resettlement Action Plan (ARAP)

The geographical scope of this study is confined to the Kangaboin (Boin) community, located in the Western Region of Ghana. The ARAP completed in advance of the Pre-Feasibility Study assessed the potential relocation requirements for the Kangaboin community due to its location and proximity to the Boin pit. The community will need to be relocated and resettled prior to commencing development at Boin. Conceptually, the ARAP proposes a schedule for conducting the RAP studies and implementation, develop a framework for public consultation, develop a framework for development planning, and develop a grievance redress mechanism procedure. Also, a census of all the PAPs, valuation of community and PAPs assets, as well as the provision of a resettlement budget and development of a resettlement implementation framework and schedule.

Limitations

Although a full census will be undertaken during the RAP stages, it may not capture the full size of the community since PAPs might not be informed about the RAP and the valuation exercise. To mitigate this limitation, a pre-planning announcement will need to be delivered through community information centres, opinion leaders, elders, and chiefs to ensure maximum community involvement.

20.2.1 Legal, Policy and Institutional Framework

Establishing a legal, policy and institutional framework ensures that all Project-related acquisition and compensation activities are conducted in accordance with national legislation and aligned with internationally recognized resettlement best practices. The objective is to guarantee fairness, legality, transparency, and restoration of affected persons' livelihoods. The following national regulatory frameworks apply:

- 1992 Constitution of Ghana.
- Land Use and Spatial Planning Act, 2016 (Act 925).
- Land Act 2020 (Act 10360).
- Administration of Lands Act, 1962 (Act 123).
- Minerals and Mining (Compensation and Resettlement) Regulations, 2012 (L.I. 2175).

Other applicable international regulations include:

- International Financial Corporation Performance Standards.

- Guiding Principles on Resettlement, 2026.
- Voluntary Guidelines on Responsible Governance of Tenure, 2012.

Valuation Methodology Valuation constitutes the technical and financial foundation of the compensation and resettlement process. In the context of involuntary land acquisition, valuation is not merely an exercise in assigning monetary figures to affected areas; rather, it defines a structured professional undertaking, aimed at restoring livelihoods, protecting property rights, and ensuring that PAPs are not rendered economically worse off as a result of Project implementation.

The methodology adopts the Replacement Cost Method (RCM) for residential structures, community facilities, and other physical improvements; and the Investment-Income Method (IIM) for crops and other income-generating agricultural assets. The approaches ensure that compensation reflects the full economic value of lost assets, enabling PAPs to reconstruct or restore their livelihoods without financial disadvantage. The entire process will follow a logical and chronological sequence, beginning with methodology, detailed asset identification and descriptions, culminating in approximate compensation value for affected crops, land, and structures.

20.2.2 Basis of Valuation

The basis of valuation adopted for the PFS ARAP is a full RCM. Under this principle, compensation is assessed based on the total cost required to replace affected assets with new assets of equivalent size, quality, and functionality, using prevailing market prices at the time of valuation. Replacement cost includes the cost of materials, labour, transportation, contractor overheads, and any associated transaction costs necessary to restore the asset to its pre-displacement condition. No deductions are made for depreciation, age, physical deterioration, salvage value, or obsolescence. Three main methods were used to establish the RCM. These comprised, Direct Comparison (DC), Investment Capitalization Approach (ICA) and Depreciated Replacement Cost (DRC).

20.2.3 Valuation Results and Compensation

The valuation of affected assets was conducted using established professional valuation methods appropriate to each asset category. The Replacement Cost Method was applied to residential structures and community infrastructure to ensure that affected persons can reconstruct equivalent structures using prevailing market prices for materials and labour. The Investment Method was applied to agricultural crops to determine the present value of expected income streams from perennial crops such as cocoa, while the Direct Comparison Method was used to estimate land values based on comparable land transactions within the Project area.

The valuation results indicate that the total compensation value for affected assets amounts to approximately GH¢43.1 M equivalent to approximately $4.0 M. This estimate includes compensation for crops, land, residential structures, and community assets, reflecting the cost required to replace affected assets at full replacement value. It is anticipated that the total compensation value may increase once a comprehensive RAP is undertaken, as additional affected persons and assets are identified that were not captured during the ARAP stage. It is assumed that the resettlement would occur prior to construction, and as such costs related to the relocation and resettlement of the community are not included in the Pre-Feasibility Study.

21.0 CAPITAL AND OPERATING COSTS

21.1 Introduction

The economic evaluation of the Enchi Gold Project is based on the capital and operating cost estimates developed in this Pre-Feasibility Study. The assessment assumes an open pit mining operation incorporating crushing, milling, carbon-in-leach (CIL) processing, elution, gold room operations, tailings management, supporting infrastructure, and owner's costs.

The capital and operating cost estimates assume the use of contract mining for all mining-related activities, while key functional areas such as processing operations and environmental management remain under the responsibility of the Owner, Newcore. Contractor costs have been derived from budgetary quotations and unit rates provided by experienced Ghana-based contractors.

All costs are presented in United States Dollars (USD), unless otherwise noted, and are based on Q1 2026 pricing. The overall estimate is classified with an accuracy range of -20% to +30%. Where input costs were originally provided in other currencies, the following exchange rates have been applied:

Table 21.1.1 Exchange Rates

Currency	Exchange Rate to USD
USD	1.00
CAD	0.72
EUR	1.16
AUD	0.65
XOF	0.0018
ZAR	0.05
GBP	1.33
TZS	0.00039

Although the estimate is sensitive to changes in exchange rates, the impact of exchange rate fluctuations was not evaluated through a sensitivity analysis in this study.

21.2 Overall Project Cost

The initial capital cost is estimated at $351 million, with sustaining capital costs of $135 million and closure costs of $21 million. The total capital cost estimate for the Project was prepared in support of the Enchi Gold Project Pre-Feasibility Study (PFS) and is based on a nominal process plant throughput of 5.5 million tonnes per annum (Mtpa).

The total capital cost estimate reflects the process plant design to support a 9.3-year mine plan and production schedule, with Year-1 and Year-4 adopted as representative design cases for the two primary ore types that will be encountered across the mine life. A summary of the total Project cost estimate is presented in Table 21.2.1.

Table 21.2.1 Total Project Capital Cost Summary

Description	Initial $M	Sustaining $M	Closure $M	LOM $M
Construction Distributables	$34.3	-	-	$34.3
Treatment Plant Costs	$113.5	-	-	$113.5
Reagents and Plant Services	$18.7	-	-	$18.7
Infrastructure - Tailings Dam	$25.8	$108.4	-	$134.3
Infrastructure - Power Supply	$4.9	-	-	$4.9
Other Site Infrastructure and Services	$16.9	-	-	$16.9
Mining (Incl. Pre-Production)	$23.3	$8.6	-	$32.0
EPCM Costs	$38.6	-	-	$38.6
Owners Projects Costs	$37.4	-	-	$37.4
Closure Capital	-	-	$18.5	$18.5
Contingency	$37.7	$17.7	$2.8	$58.2
Total Capital Costs	**$351.2**	**$134.8**	**$21.3**	**$507.2**

21.3 Capital Cost Estimate

21.3.1 Introduction

The initial capital cost estimate has been developed to define the capital required to construct, commission, and bring the Project to operational readiness. The estimate is based on inputs obtained from open pit mining and development contractors, third-party specialists, and Lycopodium.

The cost basis incorporates budget quotations for major equipment, vendor and contractor pricing obtained from the local market, and relevant historical cost data from comparable projects.

The following sections provide a detailed breakdown of the total initial capital cost estimate including contingencies by project area and cost category.

21.3.2 Area 000 – Construction Distributables

Construction distributables cover the general site-wide costs necessary to support construction activities that are not directly attributable to a specific work package. This includes the provision of a heavy lift crane for major equipment installation activities, the installation of plant security fencing to secure the construction site perimeter and the provision of essential services, including water and power, to the EPCM construction offices throughout the construction period.

The construction distributables capital cost estimate is itemized into the following categories, with corresponding cost allocations presented in Table 21.3.1:

Table 21.3.1 Construction Distributables Capital Cost Estimate

Plant Area	Project Total Cost $M
010 Construction Distributables - Contractors	$29.9
020 Site Construction Distributables General	$3.3
030 Site Construction Facilities	$0.7
050 Construction Operations	$4.9
060 Construction Accommodation	$0.5
Total	**$39.3**

Notes:
The project total cost includes the allocated project contingency.

21.3.3 Area 100 – Treatment Plant Cost

The treatment plant capital cost includes expenditures associated with general facilities required to support plant operations, as well as all process areas spanning from feed preparation through to the gold room and tailings handling systems.

The treatment plant capital cost estimate is further itemized into the following categories, with corresponding cost allocations presented in Table 21.3.2:

Table 21.3.2 Treatment Plant Capital Cost Estimate

Plant Area	Project Total Cost $M
110 Treatment Plant - General	$14.5
120 Feed Preparation	$18.2
130 Milling	$51.9
140 Trash Removal and Thickening	$5.2
160 Leaching	$25.1
170 Elution and Gold Room	$11.1
180 Tails Handling	$1.4
Total	**$127.3**

Notes:
The total project cost includes the allocated project contingency.
Values may not sum due to rounding.

21.3.4 Area 200 – Reagent and Plant Services

The reagents and plant services scope include all infrastructure, utilities, and systems necessary to support the process plant operations. Key reagents for the project comprise sodium cyanide, flocculant, acid, and sodium hydroxide.

The reagent and plant services capital cost estimate is itemized into the following categories, with corresponding cost allocations presented in Table 21.3.3:

Table 21.3.3 Reagent and Plant Services Capital Cost Estimate

Plant Area	Project Total Cost $M
210 Reagents	$2.4
220 Water Services	$6.3
230 Plant Services	$0.4
240 Air Services	$4.1
250 Fuels	$0.0
260 Electrical Services	$7.9
Total	**$21.1**

Notes:
The total project cost includes the allocated project contingency.

21.3.5 Area 300 – Infrastructure

The Enchi Gold Project infrastructure will encompass all facilities and systems required to support mining operations, process plant operation, and construction activities. Key project infrastructure and processing facilities will be located between the Sewum and Boin deposits. The estimate assumes that power will be supplied by a Genser on a unit power consumption basis; therefore, no upfront capital cost for power supply infrastructure has been included.

The infrastructure capital cost estimate is itemized into the following categories, with corresponding cost allocations presented in Table 21.3.4:

Table 21.3.4 Infrastructure Capital Cost Estimate

Plant Area	Project Total Cost $M
310 Infrastructure - General	$6.1
320 Environmental	$5.0
330 Water and Sewerage	$1.0
340 Power Supply	$5.5
350 Tailings Dam	$29.6
360 Buildings - Admin & Security	$1.9
370 Buildings - Plant	$4.9
Total	**$54.1**

Notes:
The total project cost includes the allocated project contingency.
Values may not sum due to rounding.

21.3.6 Area 400 – Mining

Mining contractor budget pricing was obtained to support the PFS and indicate contractor interest in the Project. The pricing is non-binding and reflects assumptions current at the time of the study. The mining scope includes load, haul and placement, drilling and blasting, mine operations and technical services, site establishment, secondary ore haulage, and contractor mobilisation and demobilisation, aligned with the 5.5 Mtpa PFS mine plan.

Mining capital costs are summarized in Table 21.3.5 and relate entirely to pre-production activities, totalling $25.2 million.

Table 21.3.5 Mining Capital Cost Estimate

Plant Area	Project Total Cost $M
430 Mining Pre-Production	$25.2
Total	**$25.2**

Notes:
The total project cost includes the allocated project contingency.

Initial capital represents expenditures incurred in the pre-production period (Year -1) to establish the site and enable full production in Year 1. Under the contract mining model, the contractor is responsible for mobilisation, infrastructure development, and operational readiness. As a result, initial capital primarily comprises mobilisation, site establishment, and pre-production services, and does not include fleet acquisition as this is not required with contract mining.

Mining Infrastructure Establishment

Contractor establishment initial capital costs cover all infrastructure and work required to support mining operations, including laydown site preparation, road development, and construction of operational facilities such as maintenance workshops and offices. These operational facilities include:

- Heavy mining equipment workshop.
- Lube building, wash bay, and tire bay.
- main mining office and training room.
- Mine service area kitchen and dry mess facilities.
- Changing rooms and ablution facilities.
- Explosive facility, magazine storage, and blasting team office space.
- Mining area site access control and security.
- Construction of the fuel storage and distribution facility.

Fuel storage and distribution facility construction costs are planned to be amortized over a 10 year period and therefore only 10% of the total construction costs are attributable to initial capex. Explosive facility construction is planned to commence in Year -1 and complete in Year 1 in advance of blasting activities ramping up in Year 2 of the mining schedule. As a result, the explosive facility costs have been distributed evenly across Year -1 and Year 1. Initial haul road development is included in the mining infrastructure development costs and requires the following:

- 2.5 km of roads establishing a connection between Boin, primary MSA, explosive facility, and the process plant.
- 3.0 km road network connecting all Boin pits and waste dumps.
- 5.0 km road network connecting each individual Sewum deposit.
- 3.6 km connection between the Sewum mining area and the process plant.

Mining Equipment Mobilization

Mobilization of the primary and secondary mining fleets has been split between initial and sustaining capital costs, with 80% of the fleet delivered and prepared for full production in Year -1, with the remaining 20% of costs following in Year 1. Year 1 equipment mobilization is expected to consist of additional secondary fleet equipment such as dozers, water trucks, drilling equipment, and dewatering pumps.

Mine Area Development and Site Preparation

Pre-production mine development includes initial pit area site preparation, including clearing and grubbing of organic material and surface topsoil removal and stockpiling with 1 km of the source across areas required for Year -1 and Year 1 operations. Following site preparation works an initial waste stripping of 5 Mt is planned, with 4.5 Mt being removed from Boin and 0.5 Mt from Sewum South.

Contractor Management and Supporting Systems

Supporting infrastructure established during the pre-production period includes the initial setup of pit dewatering systems to ensure safe and efficient mining conditions ahead of full production. In parallel, pre-production technical work is undertaken, comprising geological services and ore control drilling and sampling programs. These activities support grade control and short-term mine planning, helping to reduce uncertainty and improve operational performance during the early stages of mining.

In addition, monthly contractor support fees are incurred during Year -1 to ensure overall site readiness. These costs include management and administrative services, technical personnel and equipment, non-direct costs, personnel transport, Health, Safety, and Environmental (HSE) programs, accommodation, and insurance. Collectively, these expenditures support workforce mobilization, site functionality, and the establishment of operational systems required for a smooth transition into full-scale production.

Table 21.3.6 Mining Initial Capital Cost Summary

Category	Project Total Cost $M
Mining Infrastructure Establishment	$5.3
Mining Equipment Mobilization	$1.9
Mine Area Development and Site Preparation	$13.0
Contractor Management and Supporting Systems	$3.2
Total Initial Mining Capex (Excluding Contingency)	**$23.3**

21.3.7 Area 500 – Management Costs

Management costs include costs associated with EPCM services, specialist consultants and vendor representatives.

The EPCM services cost estimate includes head office support and site staffing, sub-consultants, office consumables, equipment and associated project travel. The cost of a fully equipped home design office and all project computing requirements are included under Management costs.

The engineering and construction supervision costs for the Tailings Storage Facility (TSF) were provided by Knight Piésold Consulting (KP) as a lump sum allowance, developed in accordance with their scope of services for the TSF design and reflected their anticipated level of engineering effort and site supervision requirements through the construction period.

Vendor representatives are typically required during the installation, commissioning, and start-up phases of major equipment items to ensure compliance with manufacturer specifications, warranty obligations, and operational performance guarantees. The factored allowance has been assessed as reasonable.

The management capital cost estimate is itemized into the following categories, with corresponding cost allocations presented in Table 21.3.7.

Table 21.3.7 Management Capital Cost Estimate

Plant Area	Project Total Cost $M
510 EPCM - Home Office	$21.2
520 EPCM - Site	$19.1
530 Specialist Consultants - Design	$0.7
550 Vendor Representatives	$1.5
Total	**$42.5**

Notes:
The total project cost includes the allocated project contingency.

21.3.8 Area 600 – Owner Project Costs

Owner's costs represent the internal and external costs incurred directly by the Owner in the management, administration, and oversight of the project, independent of the EPCM services scope. Owner's costs cover the pre-production expenditures incurred by Newcore Gold during the 24-month construction period (Year -1 and Year -2) as follows:

- Owner's Project Team: Owner's project team costs cover salaries, burden, and associated expenses for Newcore's construction-period personnel and support functions. Supporting costs include consultant fees, Accra office costs, travel, and accommodation.
- Legal and Corporate: Legal fees cover legal counsel, contract review and negotiation support, permitting and regulatory compliance with legal support.
- Commissioning and Ramp-Up: Commissioning and ramp-up costs cover the organizational readiness activities required to transition from construction to operations, including training, operations readiness, maintenance readiness and site security.

The Owner project capital cost estimate is itemized into the following categories, with corresponding cost allocations as shown in Table 21.3.8:

Table 21.3.8 Owner Project Capital Cost Estimate

Plant Area	Project Total Cost $M
610 Owners Costs - General	$15.2
620 Plant and Admin Pre-Production	$13.4
640 Spare Parts	$8.3
670 Plant Mobile Equipment	$4.7
Total	**$41.6**

Notes:
The total project cost includes the allocated project contingency.

21.4 Operating Cost Estimate

21.4.1 Introduction

The project operating cost estimate is built-up from three components:

- The mine operating costs developed by Fuse Advisors.
- The process plant operating costs developed by Lycopodium.
- The on-site general and administration (G&A) operating costs developed by Lycopodium with input from Newcore.

The estimated life of mine operating cost per tonne of ore processed is summarized in Table 21.4.1.

Table 21.4.1 Life of Mine Operations Costs

Cost Category	Total Cost ($M) from First Gold Pour	$/t Processed
Mining	$880	$17.15/t
Processing	$647	$12.61/t
Mine Site G&A	$84	$1.63/t
Total Cash Cost	**$1,610**	**$31.39/t**

Note:
Numbers may not add due to rounding.

The key basis of estimate assumptions for the operating costs include:

- All currencies are stated in U.S. dollars unless otherwise indicated.

- All costs have an overall accuracy of -20/+30% based on Q1 2026 pricing.
- Consumable data is based on the process design criteria.
- Average plant throughput is 5.5 Mtpa.
- Labour requirements as developed by Lycopodium with agreement from Newcore.
- Labour unit costs were developed based on benchmark national survey for mining companies operating in Ghana.
- Power unit cost of $0.12/kWh.
- Diesel fuel cost of $1.03/L.
- Refining costs, NSR royalties and tax are excluded from the operating costs but are applied to the financial model.
- Mobile equipment costs include fuel and maintenance only.

21.4.2 Mining Operating Cost Estimate

Mining operating costs include all expenditures associated with the extraction of ore and waste, including drilling, blasting, loading, hauling, and mine support services. Mining operations are assumed to be conducted under a contractor mining model, with costs expressed on a unit basis ($/t mined) and on a Life-of-Mine (LoM) basis, excluding pre-production activities.

The estimate is based primarily on contractor quotations, supplemented by first-principles calculations and vendor pricing where required. Table 21.4.2 below outlines the total mining operating costs.

Table 21.4.2 Mining Operating Cost Summary

Category	LOM Mining OPEX $M	Unit Mining Cost $ / tonne Mined
Primary Mining Costs	$837.1	$3.08
Secondary Ore Haulage Costs	$42.0	$4.58
Total Mining Operating Costs	**$879.1**	**$3.24**

Note:
Numbers may not add due to rounding.

Primary Mining Costs

The mining contractor scope includes all primary production and support activities required to sustain mining operations. This includes production drilling, blasting, loading and hauling, as well as the operation of mine support and auxiliary equipment and mobile equipment maintenance.

Specialized services are subcontracted under the contractor's responsibility, including explosives facility management, manufacture and supply of explosives, and provision of down-the-hole blasting services. Accommodation and daily per diems for non-local mining personnel are included within the contractor rates.

Mine support services provided by the contractor include pit dewatering, mining management and administrative functions, and technical personnel such as mine planning and survey. The contractor is also responsible for health and safety services and personnel transport, with monthly contractor management fees covering on-site supervision, coordination, and overall operational management.

Fuel supply and related infrastructure are managed by a separate fuel supply contractor, including operation and maintenance of on-site storage and distribution facilities. Fuel consumption is incorporated within the mining contractor's unit rates, based on a diesel price assumption of $1.03 per liter.

Costs not included in contractor rates were estimated separately and incorporated into the mining cost model. These include mine geology and ore control personnel, ore control drilling and sampling, with assaying costs accounted for under process plant laboratory costs.

A breakdown of the primary mining costs is shown below in Table 21.4.3.

Table 21.4.3 Primary Mining Cost Breakdown

Category	LOM Cost $M	Unit Mining Cost $ / tonne Mined
Load and Haul	$529.9	$1.95
Drilling[1]	$77.2	$0.28
Blasting[1]	$68.7	$0.25
Pit Dewatering	$19.4	$0.07
Geology and Ore Control	$32.4	$0.12
Monthly Management Fees[2]	$109.5	$0.40
Total Primary Mining Costs	**$837.1**	**$3.08**

Notes:
1 Drill and Blast unit costs in this table are represented against all mined tonnage, including material that will not require drilling and blasting; Average unit mining costs for material requiring drilling and blasting are $0.70 and $0.62/tonne respectively
2 Monthly management fees include both mine operations and fuel facility fees

Secondary Ore Haulage

Secondary ore haulage from satellite deposits is included in the mining cost estimate based on contractor quotations. Costs comprise unit haulage rates ($/t) for the Nyam and Kwakyekrom deposits, together with monthly contractor management fees, outlined in Table 21.4.4. These rates include full operation of the haulage fleet, labour and supervision, fuel consumption and maintenance, and logistics personnel required to support continuous ore transport.

Additional costs include labour associated with site access control and security at the satellite mining deposits.

Table 21.4.4 Secondary Ore Haulage Cost Breakdown

Category	LOM Cost $M	Unit Mining Cost $ / tonne Mined
Nyam Ore Haul[1]	$28.7	$3.99
Kwakyekrom Ore Haul[1]	$7.2	$3.64
Secondary Ore Haul Management[2]	$6.1	$0.66
Total Secondary Ore Haul Costs	**$42.0**	**$4.58**

Notes:
1 Ore haul unit costs are attributed to each respective mining area's ore tonnage only
2 Secondary ore haul costs are represented against the total satellite mining ore tonnage

Total Mining Cost by Material and Deposit

Mining cost per tonne varies by deposit and material type. Oxide material and the upper transition zone can be mined without drilling or blasting, whereas lower transition material and fresh rock require drill-and-blast activities. Variations in haulage distances, pit depths, and material composition drive cost differences between mining areas. Cost differences between oxide and upper transition material also reflect early mining works undertaken in the oxide horizon. Additionally, Nyam and Kwakyekrom also include all secondary ore haulage costs, allocated to each individual deposit accordingly. A breakdown of each material type, and by deposit, is shown in Table 21.4.5.

Table 21.4.5 Total Mining Cost - Detailed

Mining Cost by Material and Deposit - $ / tonne mined					
Material Type	**Boin**	**Sewum**	**Nyam**	**Kwakyekrom**	**Total By Material**
Ore	**$3.86**	**$5.35**	**$6.93**	**$6.39**	**$5.15**
Oxide	$2.86	$4.84	$6.29	$5.78	$4.65
Trans - Dig	$2.80	$4.70	$6.19	$5.67	$4.38

Mining Cost by Material and Deposit - $ / tonne mined					
Material Type	**Boin**	**Sewum**	**Nyam**	**Kwakyekrom**	**Total By Material**
Trans - Blast	$4.20	$6.04	$7.39	$6.96	$5.75
Fresh	$4.54	$6.56	$7.44	$6.96	$5.83
Waste	**$2.84**	**$2.82**	**$2.74**	**$2.29**	**$2.79**
Oxide	$2.25	$2.41	$2.28	$1.97	$2.31
Trans - Dig	$2.17	$2.30	$2.21	$1.79	$2.19
Trans - Blast	$3.60	$3.68	$3.46	$3.08	$3.58
Fresh	$3.72	$3.99	$3.19	$3.30	$3.63
Total by Deposit	**$2.98**	**$3.50**	**$3.45**	**$2.91**	**$3.24**

21.4.3 Process Plant Operating Cost Estimate

This section describes the processing operating costs compiled by Lycopodium for the Project, which includes all direct costs to allow production of gold bullion at the plant site.

Processing operating costs have been estimated for the plant with an ore feed rate of 5.5 Mtpa, based continuous operation of 24 hours per day and 365 days per year, for the following operating scenarios:

- Oxide ore (or 'Dig' ore) at a P_{80} grind size of 75 µm;
- Oxide ore (or 'Dig' ore) at P_{80} grind size of 53 µm;
- Fresh ore ('Blast' ore) at P_{80} grind size of 53 µm.

The plant will operate at the following availabilities:

- 91.3% (8,000 h/y) – mineral sizing operation while processing oxide or 'Dig' ore;
- 80.0% (7,008 h/y) – crushing operation while processing fresh or 'Blast' ore;
- 91.3% (8,000 h/y) – milling operation and the downstream processing plant.

The LOM processing operating cost will be calculated using a combination of the operating costs from the three scenarios.

The operating costs were developed based on the following:

- Lycopodium's database of consumables prices from:

  - Similar operations in Africa.
  - Vendor provided budget quotations and proposals.
- Benchmarking within Ghana and with other similar operations, mainly in Senegal, Burkina Faso, and Namibia.
- Reagent consumptions and gold extractions based on laboratory testwork results.
- Crushing, sizing and milling energy consumption and consumables based on OMC modelling of parameters measured from comminution testwork.
- First principle estimates based on typical operating data.

The following exchange rates have been used for the preparation of the operating cost estimate:

- US$1.00 = A$1.39 (Canadian Dollar).
- US$1.00 = 12.0 GHS (Ghanaian Cedi).
- US$1.00 = 0.86 € (Euro).

The operating cost estimates are summarized at a high level in this section. The estimates are considered accurate to within -20% / +30%, are expressed in US dollars (US$) and are based on the estimates (unless otherwise noted) from the first quarter of 2026 (1Q26).

All costs associated with mining are excluded from this estimate (and are included in the section 21.4) except for laboratory grade control related to mine exploration.

The battery limits for the Enchi process plant operating costs are:

- Fresh and oxide ore ROM bins.
- Gold bullion in plant gold room safe.
- Raw water supply source.
- Fuel supply.
- TSF discharge to environment (if any).

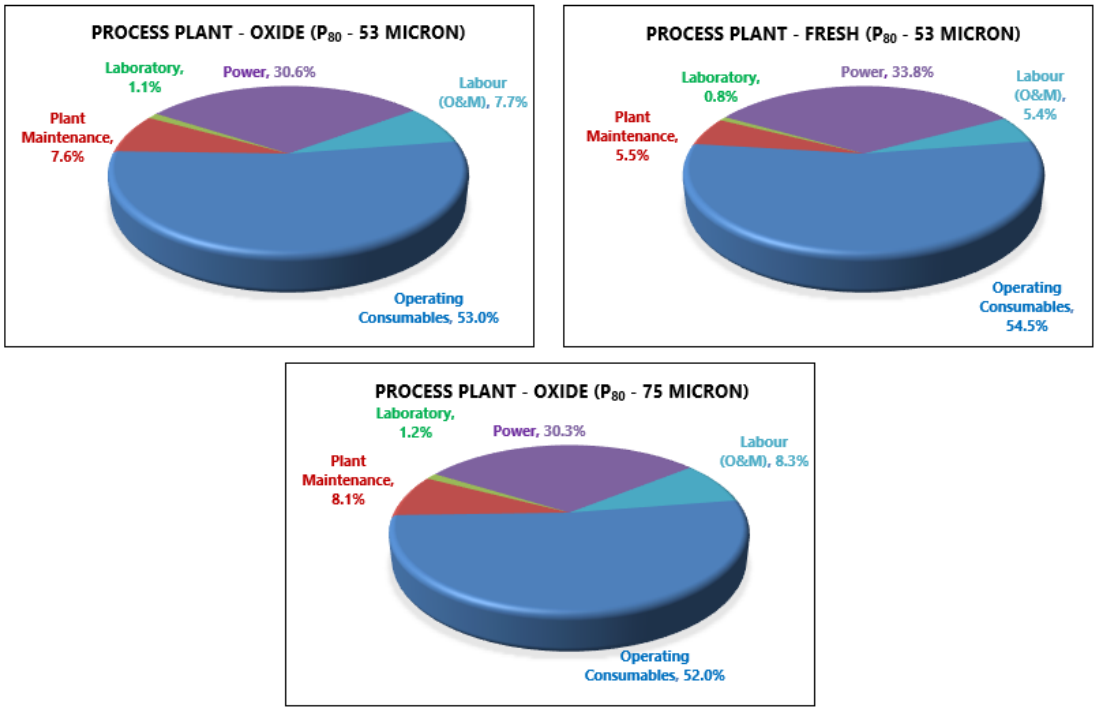
A summary of the processing operating costs is presented in Table 21.4.6 with a breakdown of the costs in Figure 21.4.1.

Table 21.4.6 Operating Cost Summary

Plant Throughput	Oxide (@ P_{80} - 75 µm)		Oxide (@ P_{80} - 53 µm)		Fresh (@ P_{80} - 53 µm)	
	5,500,000 tpa Ore		5,500,000 tpa Ore		5,500,000 tpa Ore	
Cost Centre	**Total Operating Cost**		**Total Operating Cost**		**Total Operating Cost**	
	$/year	**$/t Ore**	**$/year**	**$/t Ore**	**$/year**	**$/t Ore**
Process Plant						
Operating Consumables	$27,775,107	$5.05	$30,443,154	$5.54	$44,621,233	$8.11
Power	$16,203,714	$2.95	$17,562,878	$3.19	$27,705,862	$5.04
Plant Maintenance	$4,331,696	$0.79	$4,364,818	$0.79	$4,516,975	$0.82
Laboratory	$658,666	$0.12	$658,666	$0.12	$658,666	$0.12
Labour (O&M)	$4,433,134	$0.81	$4,433,134	$0.81	$4,433,134	$0.81
Total - Process Plant	**$53,402,316**	**$9.71**	**$57,462,649**	**$10.45**	**$81,935,869**	**$14.90**

The Oxide (75 µm) processing scenario applies to both oxide ore and diggable upper transition ore during Year 1. After Year 1, the Oxide (53 µm) processing scenario applies to the remaining diggable upper transition ore. Fresh (53 µm) processing scenario also applies to lower transition ore that requires drilling and blasting. The finer grind size reflects the need to blend this material with fresh ore or drilled-and-blasted lower transition ore to achieve the required processing characteristics.

Figure 21.4.1 Operating Cost Breakdown



Qualifications

The process operating cost estimate presented in this section excludes the following:

- All sunk costs.
- Government monitoring and compliance costs.
- Gold refining costs.
- Bullion transport costs including security staff for the transport of bullion.
- Bullion marketing costs.
- Bullion insurance in transit costs.
- External Government required monitoring and compliance costs.
- Rehabilitation or closure costs.
- Contract labour other than maintenance.
- First fill and opening stocks are captured in capital cost estimate.
- Taxes.
- Global geopolitical risks.

The following form the basis of the operating cost estimate:

- Costs have been based on SAB and gravity concentrator circuits followed by pre-leach thickening, CIL, elution, EW, and tails handling.
- Consumables unit costs have been estimated based on equipment suppliers and Lycopodium database.
- Grinding media consumption rates have been calculated based on OMC supplied data.
- Key reagent consumption rates have been based on available testwork.
- Reagent transportation costs have been included in the reagent costs.
- Crushing and milling power consumptions have been calculated by OMC.

- Other power consumptions have been estimated based on the electrical load list.
- Power unit cost was based on ECG's Power Study, and the fuel unit cost was estimated based on similar projects with agreement from Client.
- Maintenance costs were factored from the Pre-Feasibility Study capital cost estimate.
- Labour roster and unit costs have been based on input from Newcore, Knight Piésold Consulting, Fuse Advisors, Lycopodium and a mining industry salary survey for Ghana.
- Mobile equipment cost provides for fuel and maintenance, not for purchase or vehicle lease.
- Travel and worker transportation costs by others.
- TSF, environmental and regulatory costs have been estimated based on input from Knight Piésold Consulting.
- Mining costs have been provided by Fuse Advisors Surface Mining Group.

Operating Consumables

Costs for processing operating consumables, including reagents, liners, fuels, and process supplies have been estimated and are summarized separately for the oxide and fresh materials, by plant area in Table 21.4.7.

In general, reagents and consumable usage have been determined from laboratory testwork and comminution circuit modelling at average mineralised material properties, calculated from first principles, or assumed based on experience with other operations. No additional allowance for process upsets or reagent wastage were considered.

Reagents, fluxes and consumables (grinding media) costs were derived from vendor budget quotations or similar projects in comparable geographic region from Lycopodium's database and escalated to reflect inflation. Transport and freight to site, as well as import duties and taxes, have been added.

OMC modelled comminution testwork data to quantify wear rates for the crusher jaws (mobile and fixed) and wear rates of cheek plates (upper and lower) were obtained from vendor proposal, for both material types. For the mineral sizer, wear rates for different consumables (scraper comb, bearing assemblies, sizing segment, etc.) were obtained from vendor proposal.

For the milling circuit (SAG and ball mills), OMC also determined different key operating parameters, including liner thickness, liner consumption rates, and grinding media consumption rates.

For both oxide and fresh materials, reagent consumption rates for the milling, gravity concentration, CIL, and elution circuits were estimated from metallurgical testwork data. It is assumed that kerosene will be added in the milling circuit when treating fresh ore only, to improve oxygen transfer and reduce preg-robbing. The pre-leach thickener will be utilized for both ore types; therefore, the consumption of flocculant has been considered in both cases.

All reagents' supply, dosing, and consumption associated with the TSF treatment plant will be managed by the plant vendor. Reagent requirements and operating consumptions have been included within the vendor's scope of supply and associated operating cost estimates, ensuring compliance with process discharge requirements.

A diesel price of $1.03/L was utilized based on Pre-Disruption Scenario Proposal by Zen Petroleum Limited. Diesel will be used in the plant mobile equipment, the strip solution heater package, the carbon regeneration kiln, and the smelting furnace. Diesel consumption for the mobile equipment is based on industry standard vehicle consumption rates and estimated equipment utilisation rate, referencing Lycopodium's in-house database for comparable-capacity projects within the similar geographic regions.

Allowances have also been made for mill lubricants and water treatment (plant potable and filtered) plants.

Table 21.4.7 Consumables Cost Summary

Plant Areas	Oxide (@ P_{80} - 75 µm)		Oxide (@ P_{80} - 53 µm)		Fresh (@ P_{80} - 53 µm)	
	Total Consumables Cost		Total Consumables Cost		Total Consumables Cost	
	$/year	$/t Ore	$/year	$/t Ore	$/year	$/t Ore
Feed Preparation	$153,197	$0.03	$153,197	$0.03	$793,533	$0.14
Milling	$12,899,415	$2.35	$14,633,706	$2.66	$20,513,234	$3.73
Trash Removal & Thickening	$278,857	$0.051	$278,857	$0.051	$531,967	$0.097
Leaching	$9,663,173	$1.76	$9,663,173	$1.76	$9,663,173	$3.10
Elution & Gold Room	$3,891,551	$0.71	$3,891,551	$0.71	$3,891,551	$0.71
Tails Handling	$25,747	$0.005	$25,747	$0.005	$25,747	$0.005
Miscellaneous	$863,168	$0.16	$1,796,923	$0.33	$1,796,923	$0.33
Total:	**$27,775,107**	**$5.05**	**$30,443,154**	**$5.54**	**$44,621,233**	**$8.11**

Power

The average annual power draw and total annual power costs by ore type is summarized by plant areas in Table 21.4.8 for the oxide and fresh.

The proposed solution for power generation is based on an on-site power generation facility that will use natural gas supplied via existing pipeline infrastructure. The facility has been designed to match the operational demand and scaled over the LOM. It will provide reliable baseload power, bypass grid dependence, and improve energy security. The process plant has been projected to operate at an estimated power cost of $0.12/kWh.

Overall power consumption for the process plant has been estimated based on the installed power ratings of various equipment supplied by electrical load list for both plant infrastructure and camp facilities. Typical drive efficiencies and load factors were applied to the connected power ratings to calculate the peak continuous power demand across different equipment and plant areas.

For all the ore types, average annual power consumption of the SAG and ball mills was estimated by scaling the duty power predicted by OMC at 85th percentile ore hardness to the corresponding average ore hardness.

The milling circuit is the major power consumer across all operating plant areas and contributes the highest proportion of annual operating power cost. The fresh ore scenario shows a significantly higher milling power requirement due to the finer grind size and harder ore characteristics, resulting in an annual average milling power consumption of approximately ~227 million kWh and an annual milling power cost of about $28 million.

Table 21.4.8 Power Consumptions Cost Summary

Plant Throughput	Oxide (@ P_{80} - 75 µm)			Oxide (@ P_{80} - 53 µm)			Fresh (@ P_{80} - 53 µm)		
	5,500,000 tpa			5,500,000 tpa			5,500,000 tpa		
Plant Area	**Annual Average Power kWh**	**Total Annual Power Cost $/year**	**Cost, $/t**	**Annual Average Power kWh**	**Total Annual Power Cost $/year**	**Cost, $/t**	**Annual Average Power kWh**	**Total Annual Power Cost $/year**	**Cost, $/t**
Feed Preparation (Sizing)	3,659,867	$445,955	$0.08	3,659,867	$445,955	$0.08			
Feed Preparation (Crushing)							3,425,071	$417,345	$0.08
Stockpiling and Reclaim							604,532	$73,662	$0.01
Milling Circuit	85,885,553	$10,465,155	$1.90	97,039,956	$11,824,319	$2.15	178,632,116	$21,766,323	$3.96
Trash Removal & Thickening Circuit	1,899,630	$231,470	$0.04	1,899,630	$231,470	$0.04	1,899,630	$231,470	$0.04
Leaching Circuit	14,904,431	$1,816,105	$0.33	14,904,431	$1,816,105	$0.33	14,904,431	$1,816,105	$0.33
Elution & Gold Room Circuit	1,189,390	$144,927	$0.03	1,189,390	$144,927	$0.03	1,189,390	$144,927	$0.03
Tails Handling Circuit	4,467,162	$544,324	$0.10	4,467,162	$544,324	$0.10	5,746,823	$700,250	$0.13
Reagents	670,297	$81,676	$0.01	670,297	$81,676	$0.01	670,297	$81,676	$0.01
Water Services	5,423,912	$660,904	$0.12	5,423,912	$660,904	$0.12	5,423,912	$660,904	$0.12
Plant Services	3,034,028	$369,696	$0.07	3,034,028	$369,696	$0.07	3,034,028	$369,696	$0.07
Air Services	8,241,048	$1,004,172	$0.18	8,241,048	$1,004,172	$0.18	8,241,048	$1,004,172	$0.18
Fuels	32,991	$4,020	$0.001	32,991	$4,020	$0.001	32,991	$4,020	$0.001
Process Plant Buildings (Lighting, Switching, Miscellaneous)	3,572,519	$435,311	$0.08	3,572,519	$435,311	$0.08	3,572,519	$435,311	$0.08
TOTAL	**132,980,828**	**$16,203,714**	**$2.95**	**144,135,231**	**$17,562,878**	**$3.19**	**227,376,788**	**$27,705,862**	**$5.04**

Maintenance

The maintenance cost has been derived from the capital supply cost, by deducting the costs attributed to non-maintenance capital components, such as earthworks, structural steel, and concrete from the total supply cost.

A maintenance factor of 5.0% of equipment supply cost has been applied across the plant areas. Fixed maintenance costs generally account for approximately 70% to 100% of the total maintenance allocation, depending on the equipment category and critical factor considerations. The fresh ore scenario shows a marginal increase in the plant maintenance cost due to the higher installed equipment value in the crushing circuit and increased duty associated with harder ore processing.

Provision has been made for general maintenance expenses, such as specialist maintenance software, maintenance manuals and control systems-related expenditures (maintenance and licence fees). The plant infrastructure allowance primarily includes provisions for general infrastructure, utilities / services, plant and ancillary buildings, electrical services, construction-related indirect expenses, EPCM services, and maintenance requirements associated with the process plant and TSF.

The maintenance cost also excludes payroll maintenance labour which has been included in the labour cost. Contract labour costs have been incorporated for the SAG and ball mills, plant shutdowns (and maintenance) and statutory inspections.

The costs related to maintenance has been estimated to be \$0.79/t for oxide ore at P_{80} grind size of 75 µm and \$0.79/t for oxide ore at P_{80} grind size of 53 µm, and \$0.82/t for fresh ore at P_{80} grind size of 53 µm. The slight variation is attributed mainly to the differences in the milling consumables requirements and associated operational activities, including equipment change-outs and maintenance events managed by contract labour on an annual basis.

Labour

The labour rates, labour levels and rosters used to determine the labour operating cost estimate have been developed based on benchmark national survey for mining companies operating in Ghana.

Labour costs and labour levels have been developed and reported for the following:

- Plant administration – \$0.52/t.
- Process plant operation and maintenance –\$0.81/t.
- TSF operation –\$0.10/t.

The estimate of the labour contingent has been based on two shifts of operation, working 12 hours per shift, to provide continuous coverage for the plant operations with allowance for leave and absenteeism coverage. Provision has been made in the labour numbers to accommodate annual and sick leave requirements. The overhead cost includes allowances (50% for the junior staff and 40% for the seniors and managers) based on benchmarking against national survey data for mining companies operating in Ghana.

The Process Plant operation and maintenance accounts for the largest share; within this category, milling operations, laboratory personnel, and maintenance shift supervision represent the key cost drivers, primarily due to higher staffing levels and technical skill requirements. Maintenance activities, particularly mechanical, electrical, and welding roles, also contribute significantly to sustaining plant availability. TSF operations and related costs are mainly driven by skilled labour and supervisory roles required for tailings deposition and operational control.

Laboratory

The laboratory and assay cost allowances have been provided for periodic plant solids and solution samples and a minimum of 150 grade control samples' assessment per day, related to mine explorations. Provisions have been made for both internal and external testing procedures.

External testing procedures and associated costs are based on Lycopodium's benchmarking across comparable gold mining operations in similar geographic regions, ensuring alignment with industry standards and regional operating conditions. Internal testing procedures and costs are derived from the proposal provided by Intertek Minerals, that will deliver the required sample preparation and analytical services at the Project site.

The laboratory cost estimate also includes allowances for specialist contract maintenance on the laboratory equipment, replacement glassware and environmental follow-up samples. There is no provision in the laboratory and assay cost estimate for the purchase or lease of equipment.

Laboratory cost is estimated to be $0.12/t for the process plant.

Mobile Equipment

Plant mobile equipment requirements, including mobile rock breaker for mineral sizing application, have been agreed with Newcore, and based on similar reference projects in the same geographic regions. Mobile equipment costs provide for the fuel and maintenance of the mobile equipment fleet in the process plant area (excluding the mining fleet and mining light vehicles). The procurement costs of the equipment have been included in the capital cost estimate.

The front-end loaders are the highest diesel consumers within the medium and heavy equipment fleet, accounting for ~236,520 litres per year due to high utilization.

The fuel and maintenance costs for the mobile equipment are included in the consumables and maintenance cost centres, respectively.

Water

The water usage costs have been included for the operation and maintenance of potable, filtered and TSF decant water treatment plant operations and included in the consumables cost centre.

Water supply costs have not been estimated separately as they have been included in the other cost centres:

- Water supply pumping power is included in the power cost.
- Maintenance costs associated with water services area supply are included in the maintenance cost.
- Labour required for water services area supply has been included in the labour cost.

21.4.4 TSF Operating Cost Estimate

The operational expenditure (OPEX) costs for the Tailings Storage Facility (TSF) are comprised of three primary cost components. The first component relates to TSF Operations Personnel, which includes a team of staff members across four roles: a Tailings Superintendent, a Tailings Supervisor, Task Leader, and Skilled Labourers, collectively forming the core operational workforce required for the day-to-day running of the facility. The second component covers yearly oversight costs, which encompass Quarterly Facility Audits with Reporting, Annual Dam Safety Audits, and Ad Hoc Services, ensuring that the facility remains compliant with regulatory and safety requirements on an ongoing basis. The third component addresses Design and Construction Supervision Costs, which include periodic Site Investigations, TSF Detailed Design per Stage Raise, and TSF Construction Quality Assurance costs, reflecting the technical expertise required to support the staged development and construction of the facility.

Table 21.4.9 TSF Operating Cost Summary

Description	Estimated Yearly Cost $/year
TSF Operational Personnel	$555,800
KP Yearly Oversight Cost	$115,000
Total	**$670,800**

21.4.5 General and Administration

The General and Administration (G&A) costs are based on information and cost data provided by Newcore and Lycopodium's in-house database, based on an agreement with Newcore and referencing similar projects within comparable geographic regions.

The G&A expenses include the following major components:

- Site office expenses including telecommunications costs and maintenance.
- Expenses of software licenses, stationery items and computer supplies.
- Insurance expenses covering third party liability, light vehicles, bullion security (such as transport) and other industrial risks.
- Financial charges including banking fees, legal fees, auditing costs, and tenement fees.
- Bullion refining and royalties are excluded as they are deducted directly from revenue in a separate cost category.
- Consulting fees covering environmental consultants, vendors, and compliance testing.
- Personnel-related expenses including first aid, medical items, visas, passports, safety supplies, recruitment, relocation, transportation, training, and accommodation.
- Other operational expenses covering camp catering, camp management fees, security contracts, and miscellaneous site expenses related to camp, community and environment.

21.5 Sustaining Capital

Sustaining capital costs represent the ongoing capital expenditure required to maintain the asset's productive capacity and operational integrity over the life of mine. These costs have been estimated and classified in accordance with the same criteria and methodology applied to the initial capital expenditure (CAPEX), ensuring consistency in scope definition, cost structuring, and reporting standards across the estimate. Closure costs are not included in this estimate and fall outside the scope of sustaining capital.

The sustaining capital cost estimate is itemized into the following categories, with corresponding cost allocations as shown in Table 21.5.1:

Table 21.5.1 Sustaining Cost Summary

Description	Sustaining $M
Infrastructure - Tailings Dam	$108.4
Mining	$8.6
Contingency	$17.7
Sustaining Capital Costs	**$134.8**

A detailed year-by-year breakdown of the sustaining capital cost estimate is provided in Table 21.5.2.

Table 21.5.2 Sustaining Cost Expenditures by Year

Expenditure Year	Infrastructure – Tailings Dams $M	Mining $M	Contingency $M	Total Estimated Cost $M
Year 1	$10.5	$3.0	$2.1	$15.6
Year 2	$8.9	$1.9	$1.6	$12.5
Year 3	$10.2	$1.5	$1.8	$13.5
Year 4	$11.3	$0.6	$1.8	$13.6
Year 5	$12.1	$0.3	$1.8	$14.2
Year 6	$27.6	$0.3	$4.2	$32.1
Year 7	$10.8	$0.3	$1.7	$12.7
Year 8	$11.7	$0.6	$1.9	$14.2
Year 9	$5.3	$0.3	$0.8	$6.4
Total	**$108.4**	**$8.8**	**$17.7**	**$134.8**

Note:
Numbers may not add due to rounding

21.5.1 Mining

Sustaining capital in mining covers open pit works for equipment mobilization and fuel facility costs, satellite site establishment and site preparation across all mining areas, and secondary ore haulage provisions such as equipment mobilization and site access control infrastructure. A summary of the mining sustaining capital costs is shown in Table 21.5.3.

Table 21.5.3 Mining Sustaining Costs Summary

Category	Sustaining Mining Capex $M
Mining Infrastructure Establishment	$4.6
Mining Equipment Mobilization	$1.1
Mine Area Development and Site Preparation	$3.0
Total Sustaining Mining Capex	**$8.6**

Mining Infrastructure Establishment

Contractor establishment sustaining capital costs include the remaining construction expenditures for explosives and fuel storage and distribution facilities, as well as additional infrastructure required as mining expands throughout the mine life and as satellite mining areas are mined. These items include:

- Preparation and construction of the Sewum remote MSA.
- Completion of a remote MSA for the satellite mining area.
- Site access control and security infrastructure for the satellite mining area.
- Construction of a 16.0 km mine haulage road connecting the primary mining area with the satellite mining areas.

These works are scheduled for completion during Years 1 and 2, in preparation for the commencement of mining at Nyam in Q2 of Year 2.

Mining Equipment Mobilization

The remaining mining equipment fleet is expected to be mobilized in Year 1, along with the deployment of secondary ore haulage equipment required for transportation of ore from Nyam and Kwakyekrom to the processing plant.

Mine Area Development and Site Preparation

To support mining in additional areas, clearing (grubbing) of organic material, along with removal and stockpiling of surface topsoil, is planned approximately six months prior to the commencement of mining activities.

Sustaining capital costs for these activities have been distributed throughout the schedule to align with the development of new mining pits. Key activities include:

- Nyam: Preparation activities in Years 1 and 2, supporting staggered pit development phases.
- Kwakyekrom: Development begins in Year 3, enabling mining in Year 4.
- Sewum North: Pre-stripping conducted in Year 3.
- Sewum Checkerboard: Pre-stripping activities occurring in Year 2 (East Pit) and Year 8 (Main Pit).

Contractor Management and Supporting Systems

Contractor management costs and associated supporting systems beyond Year -1 are classified as operating expenses and have therefore been addressed in Section 21.4.

21.5.2 Process Plant

The process plant sustaining capital incorporates costs associated with the following:

- 350-PP-010 - Tailings Storage Facility 2 (TSF 2) Secondary Confinement Sump Pump - 8.5kW.
- 350-PP-011 - Tailings Storage Facility 2 (TSF 2) Underdrainage Pump 1 and 2 - 30kW.
- 350-PP-013 - Tailings Storage Facility 2 (TSF 2) Emergency Spillway Pond Pump.
- Decant Water Treatment Plant -150 m^3/h capacity.

21.5.3 Tailings Storage Facilities

The tailings storage facilities are phased across the life of mine to accommodate the project's tailings deposition requirements. The table below summarizes the construction schedule and corresponding crest elevations for each facility stage.

Table 21.5.4 Tailings Storage Facilities

Facility	Year of Construction	Crest Elevation (m RL)	Stage
TSF 1	-1	141	1
TSF 1	1	149	2
TSF 1	2	157	3
TSF 1	3	163	4
TSF 1	4	169	5
TSF 1	5	175	6
TSF 2	6	153	1

Facility	Year of Construction	Crest Elevation (m RL)	Stage
TSF 2	7	164	2
TSF 2	8	173	3
TSF 2	9		4

21.6 Closure Costs

Closure planning should be integrated across all phases of the Project lifecycle and executed in accordance with applicable legislative requirements and Good International Industry Practice (GIIP).

The closure vision is to transition the mine site to a condition that is safe, physically stable, non-polluting, and capable of supporting pre-mining land uses or alternative sustainable land uses that provide long-term value to local communities and the surrounding catchment.

A PFS-level closure cost estimate of approximately $18.5 million was prepared by Knight Piésold Consulting (KP) and Fuse Advisors, covering physical closure and reclamation activities. A 15% contingency was incorporated into the cash flow model, resulting in a total estimated closure cost of $21.3 million. These activities encompass the dismantling and removal of key infrastructure, demolition of concrete and masonry structures, and rehabilitation of disturbed areas through the re-establishment of native vegetation. Closure costs assume $40,000 per year over five years for care and maintenance of rehabilitated areas, as well as $150,000 per year over five years for environmental monitoring. These costs first occur in Year 10.

A summary of the mine closure cost estimate is presented in Table 21.6.1.

Table 21.6.1 Closure Costs Estimate Summary

Description	Closure Costs $M
Mining Demobilization	$2.4
Site Demobilization and Reclamation	$15.1
Post-Closure Monitoring and Maintenance	$1.0
Salvage Value	-
Total Pre-Contingency	**$18.5**
Contingency (15%)	$2.8
Total Closure Capital Costs	**$21.3**

22.0 ECONOMIC ANALYSIS

22.1 Forward Looking Information and Cautionary Statements

The economic analysis results presented in this section are forward-looking in nature. They rely on assumptions and inputs that are subject to known and unknown risks, uncertainties, and other factors, any of which could cause actual Project outcomes to differ materially from those projected herein.

Forward-looking information includes:

- Mineral Resource and Mineral Reserve Estimates.
- Assumed commodity prices and exchange rates.
- Mine production plans.
- Projected recovery rates.
- Sustaining and operating cost estimates.
- Assumptions as to closure costs and closure requirements.
- Assumptions as to environmental, permitting, and social risks.

Additional risks to the forward-looking information include:

- Changes to costs of production.
- Changes to construction execution strategy.
- Unrecognized environmental risks.
- Unanticipated reclamation expenses.
- Unexpected variations in quantity of mineralized material, grade, or recovery rates.
- Changes to geotechnical and hydrogeological considerations during mining.
- Failure of plant, equipment, or processes to operate as anticipated.
- Accidents, labour disputes, and other risks of the mining industry.

22.2 Methodologies Used

A high-level illustrative economic analysis of the potential of the Enchi Gold Project has been carried out using a cash flow model. The financial model is based on annual cash flows, incorporating yearly processed tonnages and grades in the process plant feed. It also accounts for process recoveries, metal prices, operating costs, refining costs, royalties, and capital expenditures, including both initial and sustaining capital.

The model has been developed on a 100% project basis. A royalty on revenues is payable to the Government of Ghana based on a sliding-scale structure ranging from 5% to 12%, with a rate of 12% when the gold price is at or above $4,500/oz. At the base case gold price of $3,800/oz a government royalty rate of 10% is applicable. Additionally, a 2% net smelter return (NSR) royalty is payable to Triple Flag Precious Metals Corp (with 1% subject to a buy-back option for a lump sum payment of $3.5 million at any time held by Royal Gold Inc.). Consistent with standard practice for projects of this nature, the financial evaluation is conducted on a "100% equity" basis, with no consideration given to debt and equity financing structures. Inflation has not been incorporated into the analysis. Results are presented on both a pre-tax and after-tax basis, with current Ghanaian tax regulations applied to determine corporate and mining tax obligations. The Government of Ghana retains a 10% free carried interest in the Project.

All monetary values are expressed in United States dollars (US$) and, unless otherwise noted, are denoted as "$". The model reflects the base case and technical assumptions outlined in the preceding sections of this report. A mid-year discounting is applied from the second year of mine construction (Year -1 onwards).

22.3 Fiscal and Economic Parameters

22.3.1 Project Exploitation Permits

Upon issuance of the exploitation permit, and in accordance with the Government of Ghana mining regulations, the Government of Ghana is entitled to retain a 10% free carried equity interest in the Project, as stipulated under Section 8 of the Ghanaian Mining Act. This interest cannot be diluted by any future increases in share capital. The government receives its share of this equity interest through dividend payments when dividends are declared.

22.3.2 Royalties and Duties

The economic analysis incorporates royalties due to the Government of Ghana as well as a royalty payable to Triple Flag Precious Metals Corp. The Government of Ghana also collects various taxes and duties on the importation of fuels, supplies, equipment, and outside services as specified in the Mining Code. A royalty on revenues is payable to the Government of Ghana based on a sliding-scale structure ranging from 5% to 12%, with a rate of 12% when the gold price is at or above $4,500/oz. At the base case gold price of $3,800/oz a government royalty rate of 10% is applicable. A 2% NSR royalty on production from the Project is held by Triple Flag Precious Metals Corp. (with 1% subject to a buy-back option for a lump sum payment of $3.5 million at any time held by Royal Gold Inc.). Ghana requires the payment of annual fees for mining rights and ground rents (occupation of land).

The mining rights apply on the entire claims with mining activity at $32.96 per hectares for the first three years, then $47.08 per hectares from the third year onward. The ground rent applies to the total surface of disturbed areas for a fee of $6.12 per hectare.

22.3.3 Taxes

Newcore's understanding of current Ghana tax regulations are applied to assess the tax liabilities. Taxation calculations are based on a 35% mining corporate tax rate. Depreciation allowance is considered at a rate of 20% over five years and accumulates in the capital expenditures tax pool if it is not used within the qualifying year of assessment.

22.3.4 Economic Parameters

All costs are expressed in United States dollars (US$), with capital and operating costs presented in Q1 2026 real terms. The financial evaluation has been conducted on a constant dollar basis, with no allowance made for inflation.

No price escalation has been applied in the model, and cost input parameters are assumed to remain constant over the life of mine for the purposes of this valuation.

No salvage value has been assigned to Project equipment or infrastructure, including buildings, earthworks, or remediation works.

The financial model has been completed using a base case gold price of $3,800/oz.

22.4 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 the two years prior to 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 five months of commissioning, in line with other similar projects that have been constructed by Lycopodium elsewhere in Africa during the last few years.

22.5 Financial Model

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 22.5.1.

Table 22.5.1 Key PFS Assumptions and Economic Parameters

Key Assumptions	
Base Case Gold Price	$3,800/oz
Production Profile	
Total Tonnes Processed (mt)	51.3
Total Tonnes Waste (mt)	220.5
Strip Ratio [1]	4.3
Gold Grade Processed (g/t Au)	0.64
Mine Life (years)	9.3
Throughput (mtpa)	5.5
Gold Recovery	90.5%
LOM Payable Gold Production (ozs Au)	953,350
LOM Average Annual Gold Production (ozs Au)	104,162
Peak Gold Production (ozs Au)	136,709
Unit Operating Costs	
LOM Average Operating Cost [2] ($/oz Au)	$1,689
LOM Average Cash Cost [3] ($/oz Au)	$2,149
LOM AISC (Cash Cost plus Sustaining Cost) (4) ($/oz Au)	$2,290
Initial Production – Years 1 to 3	
Average Annual Gold Production (ounces)	128,924
Mill Head Feed Grade (g/t Au)	0.80
Average Operating Costs [2] (US$/oz Au)	$1,399
Average Cash Costs [2] ($/oz Au)	$1,859
AISC (Cash Costs plus Sustaining Capital) [4] ($/oz Au)	$1,967

Key Assumptions	
Capital Costs	
Initial Capital Cost ($M)	$351
LOM Sustaining Capital Cost ($M)	$135
Closure Cost ($M)	$21

Notes:

1. Strip ratio excludes pre-strip tonnes mined (included in initial capex).
2. Cash costs consist of mining costs, processing costs, and mine-site G&A.
3. Cash costs consist of operating costs plus treatment and refining charges, and royalties.
4. AISC consists of cash costs plus sustaining capital (excluding closure costs and taxes.)

A complete and detailed description of the capital expenditure (CAPEX), operating expenditure (OPEX), and sustaining expenditure (SUSEX) assumptions and estimates is provided in Section 21.

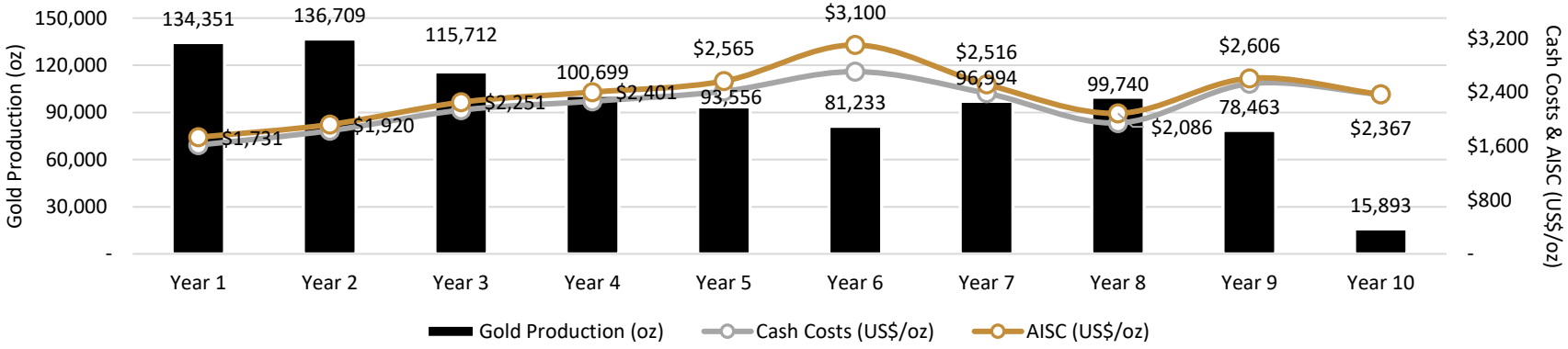
Table 22.5.2 Cash Flow Model

	Units	Total/Average	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14
REVENUES (US$3,800/oz Gold Price)																		
Gold Production (Payable)	oz	**953,350**	-	-	134,351	136,709	115,712	100,699	93,556	81,233	96,994	99,740	78,463	15,893	-	-	-	-
Gold Dore (Gross Revenue)	**US$ '000s**	**$3,622,729**	**-**	**-**	**$510,533**	**$519,494**	**$439,707**	**$382,657**	**$355,514**	**$308,684**	**$368,577**	**$379,011**	**$298,158**	**$60,393**	**-**	**-**	**-**	**-**
Treatment & Refining Charges	US$ '000s	**$3,813**	-	-	$537	$547	$463	$403	$374	$325	$388	$399	$314	$64	-	-	-	-
Royalties	US$ '000s	**$434,651**	-	-	$61,253	$62,328	$52,756	$45,911	$42,654	$37,036	$44,221	$45,473	$35,773	$7,246	-	-	-	-
Net Revenues	**US$ '000s**	**$3,618,915**	**-**	**-**	**$509,995**	**$518,948**	**$439,245**	**$382,254**	**$355,140**	**$308,359**	**$368,189**	**$378,612**	**$297,844**	**$60,329**	**-**	**-**	**-**	**-**
OPERATING COSTS																		
Mining	US$ '000s	**$879,542**	-	-	$89,732	$109,212	$111,702	$102,715	$102,864	$101,276	$103,922	$65,284	$78,011	$14,823	-	-	-	-
Processing	US$ '000s	**$646,773**	-	-	$56,303	$68,784	$72,945	$70,019	$70,755	$72,005	$73,641	$73,529	$74,898	$13,895	-	-	-	-
Mine Site G&A	US$ '000s	**$83,724**	-	-	$9,126	$9,126	$9,126	$9,126	$9,126	$9,126	$9,126	$9,126	$9,126	$1,594	-	-	-	-
Total Operating Costs	**US$ '000s**	**$1,610,039**	**-**	**-**	**$155,160**	**$187,122**	**$193,772**	**$181,859**	**$182,745**	**$182,407**	**$186,688**	**$147,939**	**$162,035**	**$30,312**	**-**	**-**	**-**	**-**
Total Operating Costs	**US$/oz**	**$1,689**	**-**	**-**	**$1,155**	**$1,369**	**$1,675**	**$1,806**	**$1,953**	**$2,245**	**$1,925**	**$1,483**	**$2,065**	**$1,907**	**-**	**-**	**-**	**-**
Total Cash Cost (Operating + Refining + Royalties)	**US$/oz**	**$2,149**	**-**	**-**	**$1,615**	**$1,829**	**$2,135**	**$2,266**	**$2,413**	**$2,705**	**$2,385**	**$1,943**	**$2,525**	**$2,367**	**-**	**-**	**-**	**-**
CAPITAL COSTS																		
Initial Capital Costs																		
Construction Distributables	US$ '000s	**$34,315**	$13,726	$20,589	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Treatment Plant Costs	US$ '000s	**$113,455**	$45,382	$68,073	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Reagents & Plant Services	US$ '000s	**$18,721**	$7,488	$11,233	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Infrastructure - Tailings Dam	US$ '000s	**$25,844**	-	$25,844	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Infrastructure - Power Supply	US$ '000s	**$4,925**	$1,970	$2,955	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Infrastructure - Other	US$ '000s	**$16,853**	$6,741	$10,112	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mining Pre-Production	US$ '000s	**$23,335**	-	$23,335	-	-	-	-	-	-	-	-	-	-	-	-	-	-
EPCM Costs	US$ '000s	**$38,618**	$15,447	$23,171	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Owners Projects Costs	US$ '000s	**$37,408**	$14,963	$22,445	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Contingency	US$ '000s	**$37,750**	$12,833	$24,917	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Initial Capital Costs	**US$ '000s**	**$351,224**	**$118,551**	**$232,673**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Sustaining Capital Costs																		
Tailings Dam	US$ '000s	**$108,449**	-	-	$10,519	$8,899	$10,234	$11,256	$12,058	$27,637	$10,807	$11,747	$5,292	-	-	-	-	-
Open Pit Mining Works	US$ '000s	**$1,454**	-	-	$1,454	-	-	-	-	-	-	-	-	-	-	-	-	-
Secondary Ore Haulage	US$ '000s	**$1,563**	-	-	$858	$705	-	-	-	-	-	-	-	-	-	-	-	-
Site Preparation	US$ '000s	**$3,335**	-	-	$413	$970	$1,253	$323	-	-	-	$376	-	-	-	-	-	-
Fuel Storage and Distribution Facility	US$ '000s	**$2,297**	-	-	$255	$255	$255	$255	$255	$255	$255	$255	$255	-	-	-	-	-
Contingency	US$ '000s	**$17,655**	-	-	$2,116	$1,624	$1,761	$1,775	$1,847	$4,184	$1,659	$1,857	$832	-	-	-	-	-
Total Sustaining Capital Costs	**US$ '000s**	**$134,753**	**-**	**-**	**$15,615**	**$12,454**	**$13,504**	**$13,609**	**$14,160**	**$32,076**	**$12,721**	**$14,234**	**$6,379**	**-**	**-**	**-**	**-**	**-**
Closure Costs (Incl. Environmental Monitoring)	US$ '000s	**$21,266**	-	-	-	-	-	-	-	-	-	-	-	$11,686	$8,925	$219	$219	$219
Total Capital Costs	**US$ '000s**	**$507,243**	**$118,551**	**$232,673**	**$15,615**	**$12,454**	**$13,504**	**$13,609**	**$14,160**	**$32,076**	**$12,721**	**$14,234**	**$6,379**	**$11,686**	**$8,925**	**$219**	**$219**	**$219**
Total AISC (Cash Cost + Sustaining Capital) [1]	**US$/oz**	**$2,290**	**-**	**-**	**$1,731**	**$1,920**	**$2,251**	**$2,401**	**$2,565**	**$3,100**	**$2,516**	**$2,086**	**$2,606**	**$2,367**	**-**	**-**	**-**	**-**
(1) AISC consists of cash costs plus sustaining capital costs (excludes closure costs and taxes)																		
CASH FLOW																		
Pre-Tax Cash Flow	US$ '000s	**$1,066,982**	($118,551)	($232,673)	$277,967	$257,043	$179,213	$140,875	$115,581	$56,841	$124,558	$170,966	$93,658	$11,086	($8,925)	($219)	($219)	($219)
Corporate Income Tax	US$ '000s	**$381,718**	-	-	$44,191	$67,774	$39,955	$33,920	$40,555	$25,114	$42,023	$58,744	$29,443	-	-	-	-	-
Post-Tax Cash Flow	**US$ '000s**	**$685,263**	**($118,551)**	**($232,673)**	**$233,776**	**$189,270**	**$139,258**	**$106,955**	**$75,025**	**$31,726**	**$82,535**	**$112,222**	**$64,215**	**$11,086**	**($8,925)**	**($219)**	**($219)**	**($219)**

FINANCIAL SUMMARY	$3,800 Gold Price	
	Pre-Tax	After-Tax
Net Present Value (5% Discount Rate) ($M)	$794	**$496**
Internal Rate of Return (%)	51%	**37%**
Payback Period (years)	1.3	**1.6**

Gold Price Sensitivity Analysis	$3,400/oz	$3,600/oz	**$3,800/oz**	$4,000/oz	$4,200/oz	$4,500/oz	$5,000/oz	$5,500/oz
Pre-Tax NPV5% ($M)	$556	$662	**$794**	$925	$1,025	$1,220	$1,508	$1,829
Pre-Tax IRR	40%	45%	**51%**	57%	61%	68%	79%	90%
After-Tax NPV5% ($M)	$340	$410	**$496**	$582	$647	$775	$963	$1,172
After-Tax IRR	29%	33%	**37%**	42%	45%	50%	58%	67%

Figure 22.5.1 Enchi Gold Project Production and Cost Profile



22.6 Results

22.6.1 Key Economic Results

On a pre-tax basis, the Project has a Net Present Value (NPV) at a discount rate of 5% of $794M, with an Internal Rate of Return (IRR) of 51% and a payback period of 1.3 years. On an after-tax basis, the $NPV_{5\%}$ is $496M, the IRR is 37%, and the payback period is 1.6 years. The payback period corresponds to the time at which the cumulative cash flow becomes positive.

The key economic results of the Pre-Feasibility Study for the Enchi Gold Project are presented in the table below.

The key assumptions and economic parameters used in the PFS are as follows:

Table 22.6.1 Key Economic Results

Summary	Life-of-Mine
Gold Price ($/oz)	$3,800
Discount Rate	5%
Production	
Mine Life	9.3 years
Processing Throughput (tonnes/yr)	5.5
Average Strip Ratio (waste:ore) [1]	4.3
Avg. Annual Gold Production (ozs)	104,162
Max Annual Payable Gold Production (ozs)	136,709
LOM Payable Gold Production (ozs)	953,350
Average Gold Grade (g/t)	0.64
Average Gold Recovery (%)	90.5%
Avg. LOM Operating Costs ($/oz)	$1,689
Avg. LOM Cash Costs ($/oz)	$2,149
Includes Royalty Costs ($/oz)	$456
Avg. LOM Mine AISC Costs ($/oz)[2]	$2,290
Operating Costs	
Mining Cost ($/tonne mined)	$3.24
Mining Cost ($/tonne ore)	$17.15
Processing Cost ($/tonne ore)	$12.61
G&A Cost (Mine Site) ($/tonne ore)	$1.63
Total Operating Cost ($/tonne ore)	$31.39

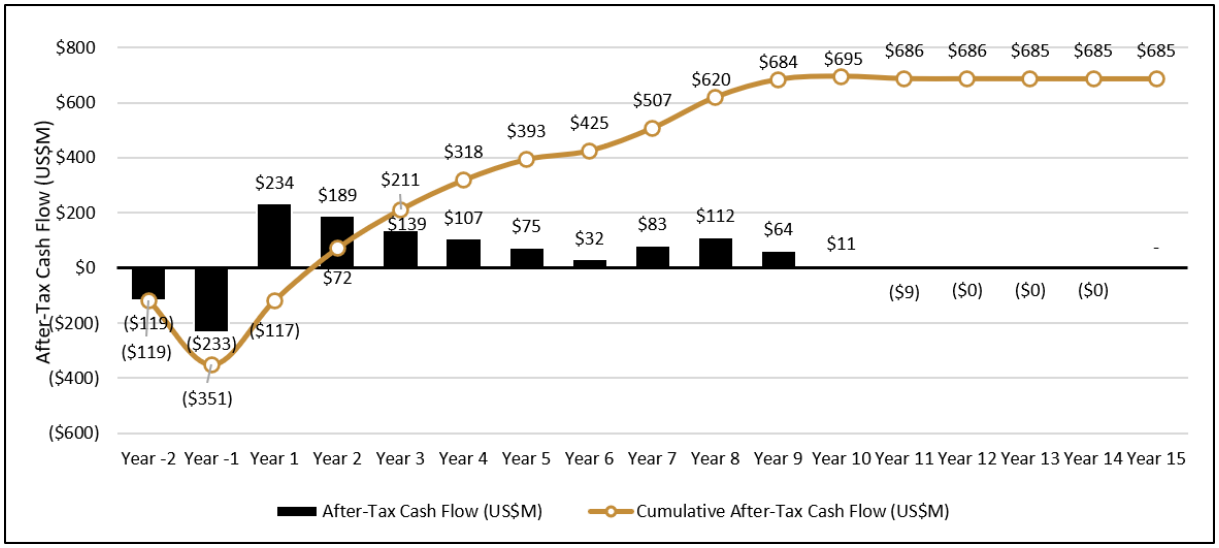
Summary	Life-of-Mine
Capital Costs	
Initial Capital Costs ($M)	$351
Sustaining Capital ($M)	$135
Closure Capital ($M)	$21
Total Capital Expenditures ($M)	$507
Mineral Resource	
Total Indicated Mineral Resource (ozs Au)	1,502,000
Total Mineral Reserve (ozs Au)	1,055,000
Pre-Tax Valuation Summary	
Pre-Tax NPV at 5.0% ($M)	$793.7
Pre-Tax IRR (%)	50.9%
Pre-Tax Payback (years)	1.3 years
Pre-Tax Cash Flow ($M)	$1,067
After Tax Valuation Summary	
After-Tax NPV at 5.0% ($M)	$495.9
After-Tax IRR (%)	37.4%
After-Tax Payback (years)	1.6 years
After-Tax Cash Flow ($M)	$685

Notes:
1. Strip ratio excludes pre-strip tonnes mined (included in initial capex)
2. Pre-Tax and Corporate G&A

22.6.2 Life of Mine Cash Flow Analysis

Figure 22.6.1 shows the cumulative and annual after-tax cash flow for the project at a gold price of $3,800/oz. The cash flows starting for years 10 through 15 relate to closure cost and environmental monitoring.

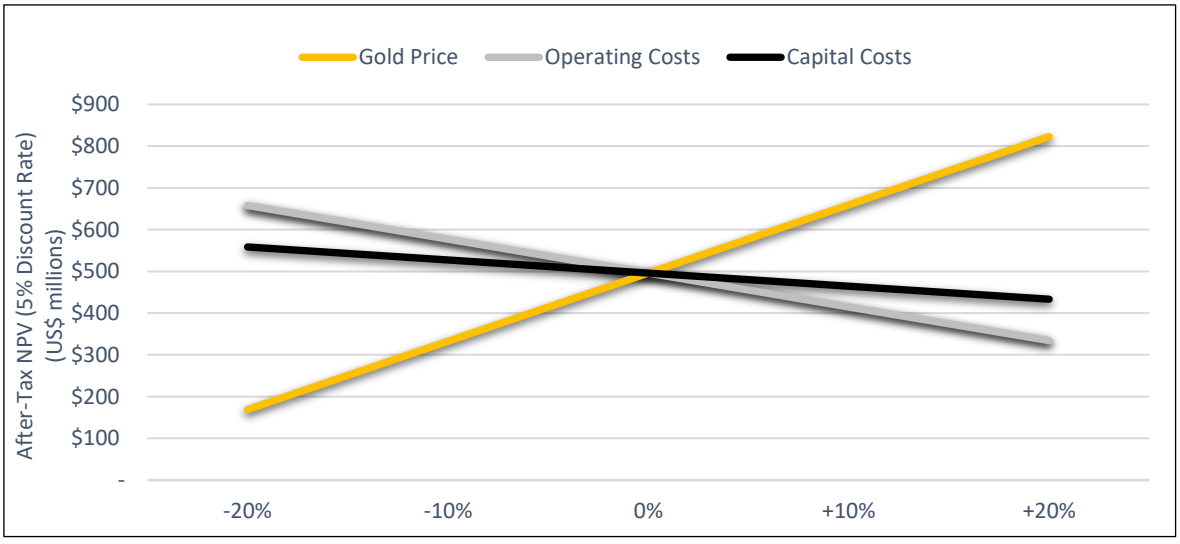
Figure 22.6.1 After-Tax Cash Flow Analysis



22.6.3 Project Parameter Sensitivities

Sensitivity analysis at a 5% discount rate 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.

Figure 22.6.2 After-Tax NPV Sensitivity Analysis Curves



The sensitivity of the Project NPV and IRR to gold price, recovery, OPEX and CAPEX are shown in the Tables 22.6.2, 22.6.3, 22.6.4, and 22.6.5 below, with the base case shown using bold font.

Table 22.6.2 Project Economic Sensitivity – Gold Price (Percentage Change)

Gold Price	-20%	-10%	0%	+10%	+20%
Pre-Tax $NPV_{5\%}$ (M)	$294	$544	**$794**	$1,043	$1,293
Pre-Tax IRR (%)	26%	39%	**51%**	61%	71%
Pre-Tax Payback (years)	2.0	1.6	**1.3**	1.1	1.0
Pre-Tax Cash Flow	$429	$748	**$1,067**	$1,386	$1,705
After-Tax $NPV_{5\%}$ (M)	$169	$332	**$496**	$659	$823
After-Tax IRR (%)	19%	29%	**37%**	45%	52%
After-Tax Payback (years)	2.5	1.9	**1.6**	1.4	1.2
After-Tax Cash Flow	$267	$476	**$685**	$894	$1,103
Gold Price	$3,040	$3,420	**$3,800**	$4,180	$4,560

Table 22.6.3 Project Economic Sensitivity – Gold Price

Gold Price	$3400	$3600	$3800	$4000	$4200	$4500	$5000	$5500
Pre-Tax $NPV_{5\%}$ (M)	$556	$662	$794	$925	$1,025	$1,220	$1,508	$1,829
Pre-Tax IRR (%)	40%	45%	51%	57%	61%	68%	79%	90%
Pre-Tax Payback (years)	1.5	1.4	1.3	1.2	1.1	1.0	0.9	0.8
After-Tax $NPV_{5\%}$ (M)	$340	$410	$496	$582	$647	$775	$963	$1,172
After-Tax IRR (%)	29%	33%	37%	42%	45%	50%	58%	67%
After-Tax Payback (years)	1.9	1.8	1.6	1.5	1.4	1.3	1.1	1.0

Table 22.6.4 Project Economic Sensitivity – Gold Recovery Sensitivity

Gold Recovery	-10%	-5%	0%	+5%	+10%
Pre-Tax $NPV_{5\%}$ (M)	$544	$669	**$794**	$918	$1,009
Pre-Tax IRR (%)	39%	45%	**51%**	56%	60%
Pre-Tax Payback (years)	1.6	1.4	**1.3**	1.2	1.1
Pre-Tax Cash Flow	$749	$908	**$1,067**	$1,226	$1,343
After-Tax $NPV_{5\%}$ (M)	$333	$414	**$496**	$578	$637
After-Tax IRR (%)	29%	33%	**37%**	41%	44%
After-Tax Payback (years)	1.9	1.8	**1.6**	1.5	1.4
After-Tax Cash Flow	$476	$581	**$685**	$790	$866
Avg. Gold Recovery	81.4%	85.9%	**90.5%**	95.0%	98.3%

Table 22.6.5 Project Economic Sensitivity – Operating Cost Sensitivity

OPEX	-20%	-10%	0%	+10%	+20%
Pre-Tax $NPV_{5\%}$ (M)	$1,041	$917	**$794**	$670	$546
Pre-Tax IRR (%)	60%	55%	**51%**	46%	41%
Pre-Tax Payback (years)	1.1	1.2	**1.3**	1.4	1.5
Pre-Tax Cash Flow	$1,388	$1,228	**$1,067**	$906	$746
After-Tax $NPV_{5\%}$ (M)	$658	$577	**$496**	$415	$334
After-Tax IRR (%)	44%	41%	**37%**	34%	30%
After-Tax Payback (years)	1.5	1.5	**1.6**	1.7	1.8
After-Tax Cash Flow	$896	$791	**$685**	$580	$475
Total OPEX ($/t)	**$25.13**	**$28.26**	**$31.39**	**$34.52**	**$37.65**

Table 22.6.6 Project Economic Sensitivity – Capital Cost Sensitivity

CAPEX	-20%	-10%	0%	+10%	+20%
Pre-Tax $NPV_{5\%}$ (M)	$884	$839	**$794**	$749	$704
Pre-Tax IRR (%)	66%	58%	**51%**	45%	40%
Pre-Tax Payback (years)	1.0	1.1	**1.3**	1.4	1.6
Pre-Tax Cash Flow	$1,164	$1,116	**$1,067**	$1,018	$970
After-Tax $NPV_{5\%}$ (M)	$558	$527	**$496**	$465	$433
After-Tax IRR (%)	48%	42%	**37%**	33%	30%
After-Tax Payback (years)	1.3	1.5	**1.6**	1.8	1.9
After-Tax Cash Flow	$750	$718	**$685**	$653	$620
Total Capex ($M)	**$410**	**$459**	**$507**	**$556**	**$604**

22.7 Conclusion

An economic evaluation of the mine schedule, based on a gold price of $3,800/oz, indicates that the project is financially viable. The analysis shows that project economics are most sensitive to changes in gold price, gold recovery, and operating costs, while exhibiting lower sensitivity to variations in capital costs.

23.0 ADJACENT PROPERTIES

23.1 Introduction

Several exploration licences are active or in the application phase immediately adjacent to the Project (Figure 23.1.1). These exploration licences are all held by individuals and there is no public disclosure on the activities related to these licences.

The Afema Gold Project (Afema), located in Côte d'Ivoire on the border of Ghana, is owned by Turaco Gold Limited (Turaco) and is located on the southern extension of the Enchi shear system. Afema includes a historical near-surface oxide and sulphide resource and lies within an area hosting several gold mineralized structures on extensions from the Sefwi Belt in Ghana, including the Woulo Woulo deposit. Turaco announced the results of a Pre-Feasibility Study for the development of Afema on 17 June 2026, and an independent JORC Mineral Resource Estimate for Afema of 115.3 Mt at 1.3 g/t gold for 4.65 Moz on 18 March 2026.

Newcore's Enchi Gold Project is located 50 km south of Asante's Chirano Gold Mine (previously held by Kinross). The Chirano Mine lies within the Proterozoic terrain of southwest Ghana, along a major structure separating the Sefwi Belt to the west from the Kumasi Basin to the east known as the Bibiani Shear Zone. The Project covers a 40 km segment of the Bibiani Shear Zone where known gold mineralization is associated with major structures and subsidiary splays.

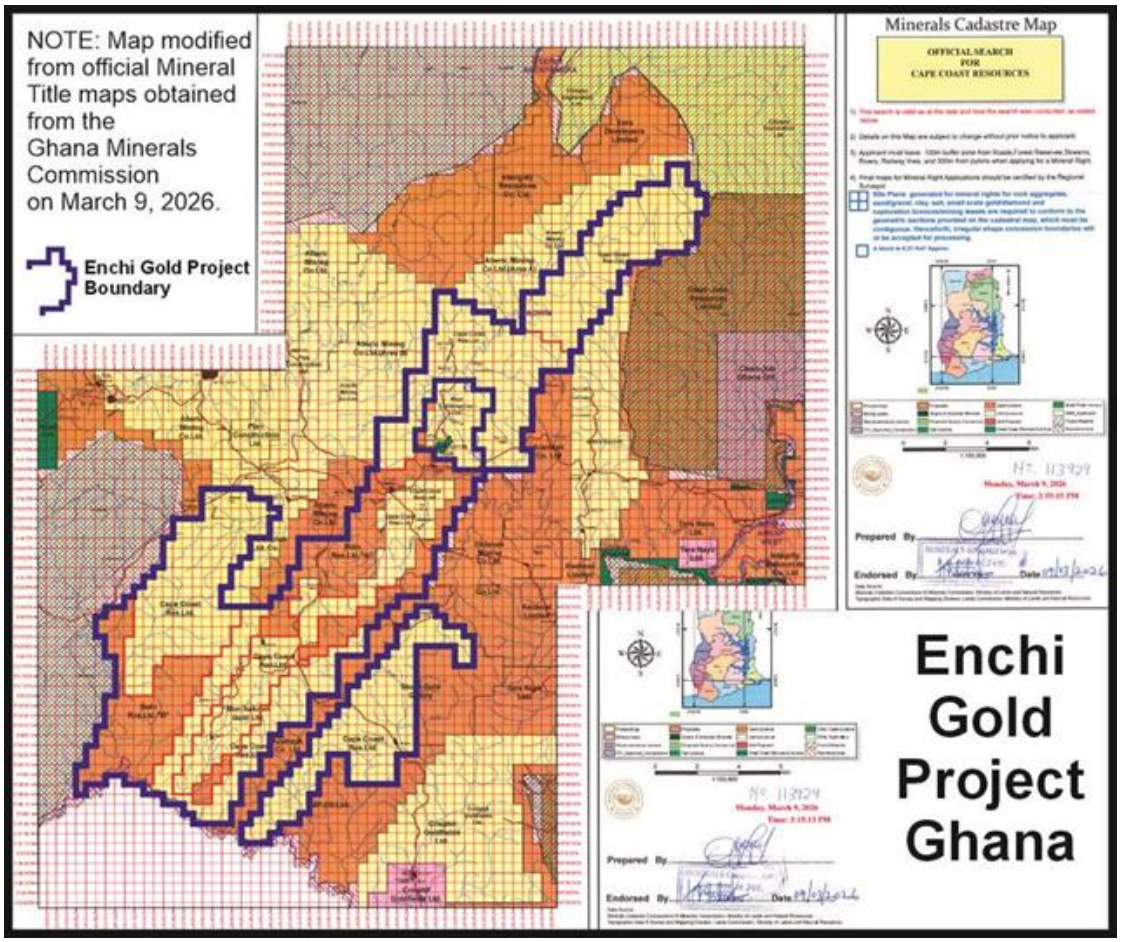
The Chirano Gold Mine is a well-established mine with both open pit and underground operations. Extensive existing infrastructure includes its own installed power supply, access to the national power grid, a 3.6 Mtpa milling and CIL processing plant, tailings storage facilities, residential housing and clinic (Begg et al., 2024). Chirano has been in production since 2005 and has produced over 3 Moz of gold since then, with 2025 gold production of approximately 130,000 oz. The deposits are hosted by fractured and altered mafic volcanics and granite and include stacked arrays of parallel veinlets, veinlet stockworks and mineralized cataclasites. The geometry and shape of the deposits range from tabular (Obra), or pipe-like (Tano) to multiple parallel lodes (Paboase). The mineralized zone thickness ranges from a few metres to over 70 m. Most deposits dip very steeply towards the west or southwest and plunge steeply. Generally, the tenor of mineralization is related to intensity of hydrothermal alteration (silica, ankerite, albite, sericite, pyrite), veining and brecciation. The gold is fine-grained and is associated with 1% - 5% pyrite (Begg et al., 2024).

The Chirano Mine has 14 known gold deposits that occur along a mineralized zone over 11 km long. Regional exploration continues to identify extensions both to the north and south. The deposits range in strike length from 150 m to 700 m, and range in thickness from a few metres to over 70 m (Begg et al., 2024). As of 31 December 2023, Proven and Probable Reserves were 18.7 Mt grading 1.76 g/t gold for 1.059 Moz (Begg et al., 2024). Within the constrained pits, the Measured and Indicated Mineral Resources totalled 14.9 Mt grading 1.05 g/t gold for 0.503 Moz, and the Inferred Mineral Resource totalled 1.71 Mt grading 1.23 g/t gold for 0.068 Moz. Within constrained underground shapes, the Measured and Indicated Resources totalled 25 Mt grading 1.97 g/t gold for 1.585 Moz, and the Inferred Resource totalled 18.3 Mt grading 1.64 g/t gold for 0.963 Moz (Begg et. al., 2024).

The QP has not verified the technical data on Chirano or Afema and the gold mineralization at Chirano or Afema is not necessarily indicative of the mineralization on the Project.

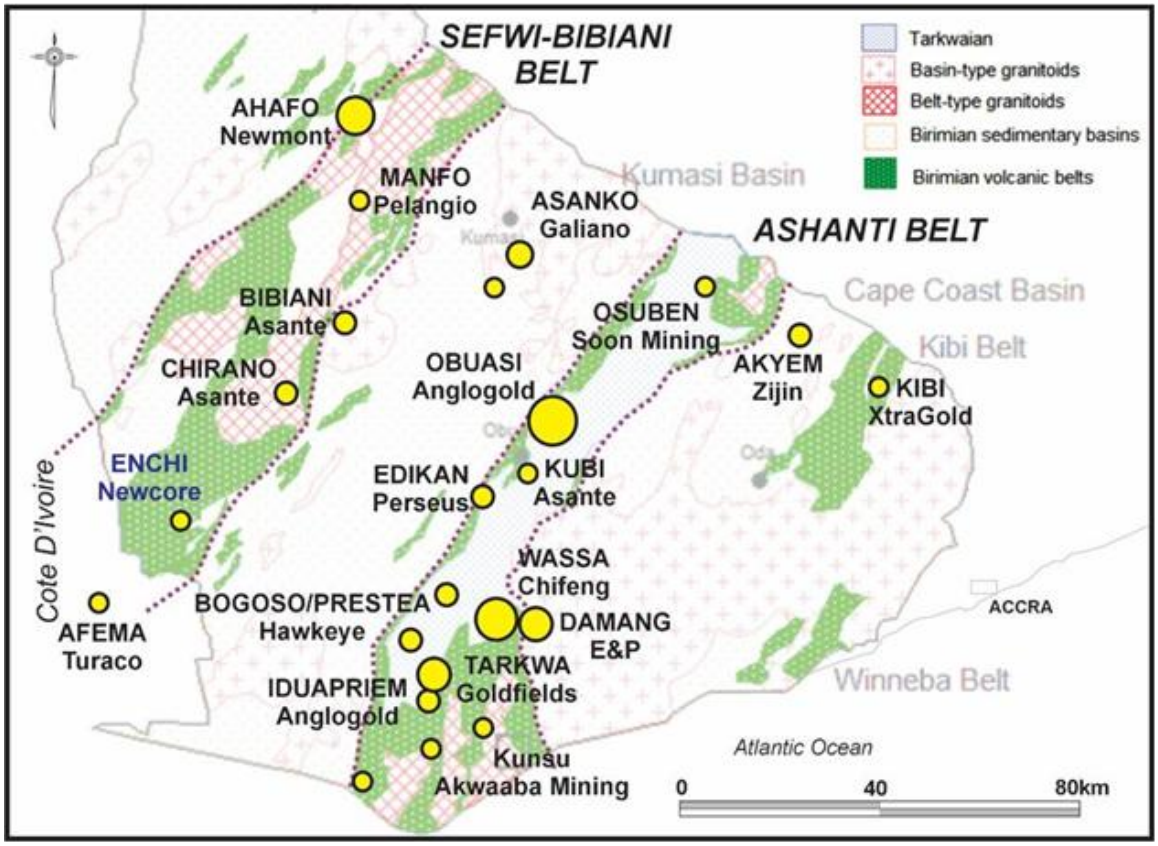
A map of proximal mines in the region surrounding the Project is provided in Figure 23.1.2.

Figure 23.1.1 Map of Adjacent Properties Surrounding the Enchi Gold Project



Source: Newcore, 2026

Figure 23.1.2 Map of Proximal Mines



Source: Newcore, 2026 (modified after "Goldfarb et al, 2017 Lipson et al, 2018").

24.0 OTHER RELEVANT DATA AND INFORMATION

This section is not relevant to this report.

25.0 INTERPRETATION AND CONCLUSIONS

25.1 Introduction

The QPs note the following interpretations and conclusions in their respective areas of expertise, based on the review of data available for this Report.

25.2 Geology and Mineral Resource Estimate

25.2.1 Geology

Based upon available information, the QP concludes the following:

- The Project is analogous to shear-hosted gold mineralization with quartz veining. This style of mineralization is typical of Birimian lode gold deposits in Ghana and West Africa.
- The Property is associated with mineralization related to the Bibiani Shear Zone that also hosts large lode-gold deposits at Chirano and Bibiani.
- Gold mineralization on the Property is associated with secondary and tertiary order shears that splay off the Bibiani Shear Zone.
- Newcore has a good understanding of regional and district geology which supports the identification and interpretation of mineralized zones within the Property.
- Mineralization is currently defined in multiple zones along defined structures at various stages of exploration. Four of these zones contain drill-defined mineral resources that are considered current.
- Drilling and sampling procedures, sample preparation, and assay protocols have generally been conducted in compliance with industry best practices.
- Verification of drillhole collars, downhole surveys, drill sample assays and drillhole logs indicates that historic data collected by Redback and Edgewater is generally considered reliable and can be integrated with Newcore data to support the current Mineral Resources.
- Sufficient Quality Assurance / Quality Control (QA / QC) protocols demonstrate that drill data is reasonably reliable to support the current Mineral Resources.
- Geological understanding of the Property is sufficient to support the mineral resource estimation.

- An oxide domain, a transition domain, and a fresh domain have been identified in the drill logs.
- Bulk density values used to determine tonnage were derived from testing selected sections of drill core from within the mineral resource area.
- Other exploration targets within the Property do not have enough data to support mineral resource estimations at this time. Exploration activities on these targets will not guarantee the delineation of additional mineral resources.

25.2.2 Mineral Resources

An updated Mineral Resource Estimate has been completed for the Project using new interpretations and modelling work completed since the effective date of the last technical report. The resources reported herein used a final database handover date of 6 October 2025.

It is the opinion of the QPs that the interpretations and related data remain valid for the Mineral Resource Estimated presented in this Report. Assumptions and methodologies used are considered both appropriate and representative of typical shear-hosted gold vein mineralized systems.

Pit-constrained resources for the Project include Indicated resources totalling 83.6 Mt at 0.56 g/t Au for 1.5 Moz of contained gold and Inferred resources of 40.1 Mt at 0.49 g/t Au for 626 koz of contained gold.

The resources have been prepared using variable cutoff grades between 0.1 and 0.2 g/t Au and a gold price of $3,200/oz.

The reported Mineral Resources are inclusive of Mineral Reserves; Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. Moreover, it is not guaranteed that all or part of the Mineral Resources will be converted into Mineral Reserves.

The QPs consider the Mineral Resources stated in this Report to have been prepared in accordance with current CIM standards, definitions and guidelines for Mineral Resource Estimation.

The QPs, to the best of their knowledge, are also not aware of any legal, title, environmental, permitting, taxation, geopolitical, socioeconomic or other factors that may materially affect the Mineral Resources reported for the Project.

A total of 16,649.6 m of drilling has been completed and publicly reported on the Project since the database handover date and has not been incorporated into the current Resource; this drilling was focused on potential depth extensions and is not anticipated to have a material effect on the reported Resource.

25.3 Mineral Processing and Metallurgical Testwork

The following conclusions can be drawn from the metallurgical and comminution testwork during the PFS phase:

Comminution Circuit:

- Comminution circuit design parameters were developed using testwork results supplemented by benchmark data where necessary. Design values were based on the 85th percentile of BWi and CWi results to provide a conservative basis for sizing and power estimation.
- The milling circuit design is based on the representative mine plan blend containing the highest proportion of fresh rock, resulting in a design ore with a BWi of 16.7 kWh/t and CWi of 17.1 kWh/t.
- Oxide and transition ores exhibit significantly softer grinding characteristics than fresh ores and were used as the basis for mineral sizer design, while fresh ores governed the primary crushing and grinding circuit design.

Leach Circuit:

- CIL testwork demonstrated that oxide ore gold recovery is relatively insensitive to grind size between P_{80} 53 µm and 75 µm, whereas fresh ore recovery improves substantially at finer grind sizes.
- Fresh ore gold recoveries increased by approximately 3% to 9% when grind size was reduced from P80 75 µm to P_{80} 53 µm. Accordingly, the processing plant was designed to achieve a target grind size of P_{80} 53 µm to maximize overall gold recovery for the fresh ore.

Thickener Sizing:

- Thickener sizing was based on tailings static settling testwork, with oxide material identified as the more challenging ore type for settling performance.
- A solids loading rate of 0.21 t/m^2/h was selected for pre-leach thickener design to achieve a target underflow density of approximately 45% solids.

Major Reagent Consumptions:

- Cyanide consumption is expected to range from 0.83 to 1.49 kg/t, with higher consumption associated with transition and fresh ores treated at the finer P_{80} 53 µm grind size.

- Cyanide consumption values used for design were reduced by 50% from laboratory testwork results based on benchmarking and the assessment that cyanide may have been overdosed during testwork.

- Lime consumption is expected to range from 1.41 to 2.61 kg/t, with lower consumption observed for fresh ore relative to oxide and transition ores.

Estimated Gold Recoveries:

- Estimated overall gold recoveries after accounting for solution and carbon losses range from 86.9% to 95.0%, depending on ore type and deposit.

The metallurgical testwork confirms that all ore types are amenable to processing by a conventional milling and CIL flowsheet, with high gold recoveries expected across the deposits under the appropriate leach conditions.

25.4 Mining and Mineral Reserves

This section presents the QP's interpretations and conclusions regarding the Mineral Reserve estimate and mining methods developed for the Enchi Gold Project. The conclusions are based on the technical studies, engineering evaluations, and mine planning work completed as part of the Pre-Feasibility Study and summarized in Chapters 15 and 16.

25.4.1 Mineral Reserves

The Mineral Reserve Estimate for the Enchi Gold Project has been prepared in accordance with National Instrument 43-101 and is supported by detailed pit optimization, mine design, production scheduling, metallurgical test work, and economic evaluations. The Mineral Reserves comprise the economically mineable portions of the Indicated Mineral Resources within the Boin, Sewum, Nyam, and Kwakyekrom deposits. The estimate totals 51.3 Mt at an average grade of 0.64 g/t Au containing approximately 1.06 Moz of gold.

The QP is of the opinion that the modifying factors applied to the Mineral Reserve Estimate are reasonable and appropriate for a PFS level assessment. Mining dilution and recovery factors have been applied based on anticipated mining selectivity, equipment size, and operating practices, while metallurgical recovery assumptions have been supported by available testwork and process design criteria. Operating costs, general and administrative costs, royalties, and commodity price assumptions were considered appropriate for the purposes of reserve estimation.

The reserve estimate is supported by detailed pit optimizations utilizing appropriate geotechnical, operating cost, processing cost, and revenue assumptions. The selected pit shells were evaluated with consideration of economic value, strip ratio, grade, and practical mineability, and were subsequently converted into detailed pit designs that demonstrate good reconciliation with the optimization results.

The QP recognizes that additional geotechnical drilling, testing, and data collection would improve confidence in pit slope design parameters and may provide opportunities for future optimization of pit wall angles and reserve inventories. Furthermore, economically significant Inferred Mineral Resources remain outside the current reserve estimate and may provide opportunities for future reserve growth through additional drilling and resource conversion.

The QP is not aware of any mining, metallurgical, infrastructure, environmental, permitting, legal, marketing, or social factors that would materially affect the Mineral Reserve Estimate beyond the risks and uncertainties typically associated with the advancement of a project from Pre-Feasibility Study to Feasibility Study level. The QP is therefore of the opinion that the Mineral Reserve Estimate represents a reasonable basis for longer-term mine planning and economic evaluation of the Enchi Gold Project.

25.4.2 Mining Methods

The proposed mining method for the Enchi Gold Project consists of conventional open pit truck-and-shovel mining utilizing contractor-operated mining fleets. The mining approach incorporates drilling, blasting, loading, and hauling activities and is considered appropriate for the geometry, size, grade distribution, and near-surface nature of the mineralization present within the Boin, Sewum, Nyam, and Kwakyekrom deposits.

Mine designs have been developed using geotechnical design criteria informed by available drilling, laboratory testing, and operating experience from comparable deposits within the region. The resulting pit designs incorporate appropriate bench geometries, inter-ramp angles, haul road configurations, and operational access requirements. The QP considers the adopted pit designs suitable for the current level of study.

The Life-of-Mine plan incorporates mining from 11 surface pits and 17 mining phases over an approximately 9.3-year operating period. The mine schedule has been developed using practical mining constraints, including maximum mining rates, plant throughput limitations, bench advance restrictions, material blending requirements, and satellite deposit haulage considerations. The resulting schedule provides an achievable basis for project evaluation and supports a consistent long-term ore supply to the processing facility.

The proposed waste rock storage facilities provide sufficient capacity to support the current Life-of-Mine production schedule. While the locations and conceptual designs are considered suitable for a Pre-Feasibility Study, additional geotechnical, environmental, hydrogeological, and waste characterization investigations are recommended to further support facility design and permitting at the Feasibility Study stage.

The satellite mining operations at Nyam and Kwakyekrom introduce additional haulage complexity due to ore transport requirements between the mining areas and the central processing facility. However, the planned use of dedicated rehandling and ore haulage fleets is considered a reasonable approach and has been incorporated into the mine planning and operating cost assumptions.

Opportunities exist to further optimize haulage networks, waste rock storage facility locations, drill-and-blast practices, fleet selection, and ore transport strategies as part of future engineering studies. Additional geotechnical and operational data may also provide opportunities to improve pit designs, reduce mining costs, and enhance overall project economics.

Based on the information reviewed, the QP is of the opinion that the proposed mining method, mine infrastructure, production schedule, and supporting mining assumptions are appropriate for the Enchi Gold Project and provide a reasonable basis for the declaration of Mineral Reserves and the advancement of the Project to the next level of study.

25.5 Recovery Methods

Based on the metallurgical testwork completed to date, the following process flowsheet has been selected for the Pre-Feasibility Study (PFS):

25.5.1 Ore Preparation

- A mineral sizer will be used for 'Dig' ore, producing a sized product with a target P_{80} of 45 mm.
- A single-stage primary jaw crusher will be used for 'Blast' ore, producing a crushed product with a P_{80} of 152 mm.
- Crushed ore will be stored in a dead stockpile providing approximately 16,500 tonnes of live storage capacity, equivalent to about 24 hours of mill feed.

25.5.2 Milling

- The milling circuit will comprise a Semi-Autogenous Grinding (SAG) mill followed by a ball mill (SAB configuration) operating in closed circuit with classification cyclones.
- The circuit is designed to achieve a target grind size of P_{80} 53 µm. During Year 1, when oxide ore predominates, the circuit can operate at a coarser grind size of P_{80} 75 µm, reducing power consumption and operating costs.

25.5.3 Pre-Leach

- A trash screen will remove debris such as wood chips and plastic materials from the cyclone overflow to prevent contamination of downstream processing circuits.
- A pre-leach thickener will increase pulp density from approximately 35% solids by weight to 45% solids for 'Dig' ore and 50% solids for 'Blast' ore.

25.5.4 Gold Recovery

- Gold leaching will be undertaken in a seven-stage Carbon-in-Leach (CIL) circuit.
- Gold-loaded carbon will be processed through an elution circuit based on the split Anglo American Research Laboratories (AARL) process, where elution and electrowinning operate as separate stages.
- In the AARL configuration, the elution and electrowinning steps are decoupled. The circuit consists of an acid wash column to remove inorganic foulants and an elution column for carbon stripping. The elution column, together with the stripping water, lean eluate tanks and the strip solution heating system form a closed loop for circulating the heated strip solution to a pregnant solution tank. The solution in the duty pregnant solution tank will then be pumped in a closed loop with the electrowinning cells, in which gold is plated out as sludge, and the resulting barren solution is returned to the CIL circuit.
- Final gold recovery will occur within a secure on-site goldroom incorporating electrowinning cells, a sludge pan filter, drying oven, and smelting furnace to produce doré bars.

25.5.5 Carbon Regeneration

- A carbon regeneration kiln to remove organic foulants from the carbon and regenerate the carbon adsorption sites with the application of heat.

25.5.6 Tailings Management

- Process tailings slurry will be discharged by gravity through a dedicated tailings handling system to the Tailings Storage Facility (TSF).

25.5.7 Process Reagents and Consumables

- The process plant will utilize a range of reagents, including quicklime (CaO), flocculant, sodium cyanide (NaCN), kerosene, activated carbon, sodium hydroxide (NaOH), hydrochloric acid (HCl), lead nitrate, anti-scalant, sulphamic acid, and smelting fluxes such as borax, soda ash, silica sand, and sodium nitrate.
- Additional consumables will include crucibles, Leach-Aid, stainless steel stocking, grinding media, and diesel fuel to support plant operations across the various processing circuits.

25.6 Infrastructure

The Enchi Gold Project infrastructure plan provides the essential facilities and services required to support construction, mining and process operations. Key conclusions include:

- Comprehensive road network: The Project includes a well-developed internal access and haul road system designed to safely and efficiently connect all major operational areas, including the process plant, open pits, mine services facilities, explosives storage, and fuel storage areas. The haul roads are sized to accommodate the selected mining fleet and support efficient movement of personnel, equipment, and materials.

- Waste rock management infrastructure: The Waste Rock Storage Facilities are designed at an angle of repose of 37° with a 20 m lifts and 15 m berms. Upon closure, the WRSF will be contoured to an overall slope flatter than 3H:1V for benches exceeding 10 metres in height, in accordance with Minerals and Mining (Health, Safety and Technical) Regulations, 2012 (Ghana), specifically Regulation 286 governing the closure of waste dumps.

- Modular and functional building infrastructure: The site will utilize primarily prefabricated modular buildings to minimize construction complexity and support operational flexibility. Facilities include administration, security, emergency response, laboratory, maintenance, warehouse, reagent storage, and accommodation camp required for both mining and processing operations.

- Secure gold recovery facilities: A dedicated goldroom will support gold recovery, refining, and bullion storage activities. The facility will incorporate robust security measures, including controlled access, surveillance systems and alarms.

- Power supply: The operation is expected to require approximately 37 MW connected load and 33 MW maximum operating demand, highlighting the importance of a reliable long-term power solution.

- Two viable power supply options under evaluation:

  - A grid connection (GRIDCo), involving a dedicated 161 kV transmission line and substation with an estimated implementation period of 24 months.

  - An onsite gas-fired power plant developed by Genser Energy under a BOO arrangement, providing an independent power source but with a longer estimated implementation timeline of 42 months, which was the chosen option for this PFS.

- Strategic infrastructure supports project readiness: Overall, the proposed infrastructure provides a practical and scalable framework to support mine development and operations, with power supply selection representing a key decision given it is a long-lead item that may impact the Project start-up schedule.

25.7 Environmental, Permitting and Social Considerations

- Continuous baseline studies and data gathering by Abbakus Geosocial Consult is adequate. Data should continue to include seasonal studies.
- Identified and predicted environmental and social impacts are manageable through implementation of recommended mitigation measures.
- Kangaboin (Boin) community within the immediate project footprint for Boin pit development will require relocation and resettlement. Given the small scale of the village, this is currently planned to be completed ahead of construction and hence is not included in the Project economics. Additional communities for relocation will be confirmed as the Project advances and final mine plan is determined.
- A full Environmental Impact Assessment (EIA) and Resettlement Action Plan (RAP) must be conducted.
- Outcome of RAP will inform relocation planning and knowledge of number of affected persons (PAPs) who will seek financial compensation and physical compensation.

25.8 Capital Cost Estimate

The total project capital cost estimate reflects the process plant design to support a 9.3 year mine plan and production schedule, with Year-1 and Year-4 adopted as representative design cases for the two primary ore types that will be encountered across the mine life.

The capital cost estimate is classified with an accuracy range of -20% to +30%. The Process Plant initial capital cost is estimated at $351.2 million.

The sustaining capital cost is estimated at $134.8 million. These costs have been estimated using the same methodology, scope definition, and reporting criteria applied to the initial capital cost estimate to ensure consistency.

A PFS-level closure cost estimate of approximately $18.5 million was prepared by Knight Piésold Consulting (KP) and Fuse Advisors, covering physical closure and reclamation activities. A 15% contingency has been incorporated into the cash flow model, resulting in a total estimated closure cost of $21.3 million.

25.9 Operating Cost Estimate

Processing operating costs have been estimated for the plant with an ore feed rate of 5.5 Mtpa, based continuous operation of 24 hours per day and 365 days per year. Cost estimates are based on first quarter 2026 pricing and have been prepared to an accuracy range of -20% to +30%. The operating cost estimate was prepared for three different plant feed scenarios, reflecting the different ore types and processing conditions anticipated over the life of the operation.

- Oxide ore at P_{80} 75 µm: $53.4 million per year, $9.71 per tonne of ore.
- Oxide ore at P_{80} 53 µm: $57.5 million per year, $10.45 per tonne of ore.
- Fresh ore at P_{80} 53 µm: $81.9 million per year, $14.90 per tonne of ore.

25.10 Economic Analysis

On a pre-tax basis, the Project is estimated to generate a Net Present Value (NPV) of $794 million at a 5% discount rate, an Internal Rate of Return (IRR) of 51%, and a payback period of 1.3 years. On an after-tax basis, the Project is expected to deliver an NPV of $496 million at a 5% discount rate, an IRR of 37%, and a payback period of 1.6 years.

A summary of the project economics is listed in Table 25.10.1, and after-tax free cash flow is shown graphically in Figure 25.10.1.

Table 25.10.1 Economic Analysis Summary

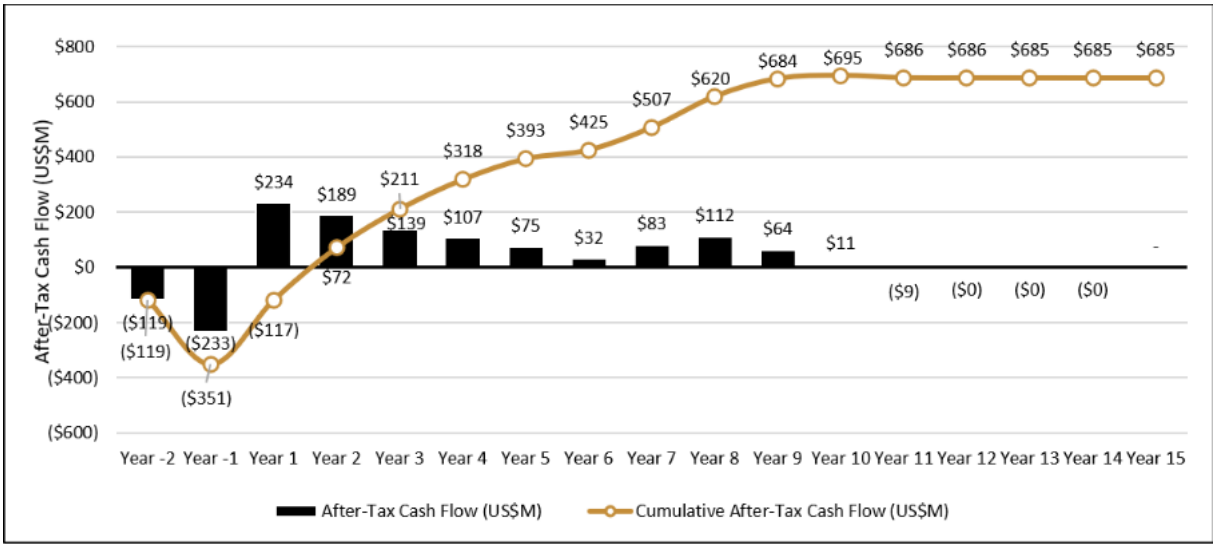
Key Assumptions	
Base Case Gold Price	**$3,800/oz**
Production Profile	
Total Tonnes Processed (mt)	51.3
Total Tonnes Waste (mt)	220.5
Strip Ratio (1)	4.30
Gold Grade Processed (g/t Au)	0.64
Mine Life (years)	9.3
Throughput (mtpa)	5.5
Gold Recovery	90.5%
LOM Payable Gold Production (ozs Au)	953,350
LOM Average Annual Gold Production (ozs Au)	104,162
Peak Gold Production (ozs Au)	136,709

Key Assumptions	
Unit Operating Costs	
LOM Average Operating Cost (2) ($/oz Au)	$1,689
LOM Average Cash Cost (3) ($/oz Au)	$2,149
LOM AISC (Cash Cost plus Sustaining Cost) (4) ($/oz Au)	$2,290
Initial Production – Years 1 to 3	
Average Annual Gold Production (ounces)	128,924
Mill Head Feed Grade (g/t Au)	0.80
Average Operating Costs (1) ($/oz Au)	$1,399
Average Cash Costs (2) ($/oz Au)	$1,859
AISC (Cash Costs plus Sustaining Capital) (3) ($/oz Au)	$1,967
Capital Costs	
Initial Capital Cost ($M)	$351
LOM Sustaining Capital Cost ($M)	$135
Closure Cost ($M)	$21

Notes:

1. Strip ratio excludes pre-strip tonnes mined (included in initial capex).
2. Cash costs consist of mining costs, processing costs, and mine-site G&A.
3. Cash costs consist of operating costs plus treatment and refining charges, and royalties.
4. AISC consists of cash costs plus sustaining capital (excluding closure costs and taxes.)

Figure 25.10.1 After-Tax Cashflow Analysis



25.11 Risks and Opportunities

25.11.1 Geology and Resource Risks

The QPs note the following key potential risks:

- Licensing and permitting issues and/or delays from the Ghanaian authorities; while these areas are always a risk to any project, they are important to identify, nonetheless.
- Further characterization of the weathering profiles at each of the deposits could alter the proportions of oxide, transitional and fresh host-rock facies, which can affect long-term modelling and planning scenarios.
- Mineralized zone geometries could change in areas that are currently less well-defined, which may affect zone trend interpretations and require adjustments in future modelling efforts.
- Cut-off grade could represent a risk given the low-grade nature of these deposits and the current elevated gold market prices; however, a reasonable selling price for the resource open-pit optimizations was selected according to sensitivity analysis.
- Nugget effect can represent a risk given the high variability observed during the QP check assay program, which is why continued monitoring has been recommended.

25.11.2 Mineral Reserves Risks

- The Mineral Reserve estimate is dependent on a range of economic and technical assumptions and modifying factors, including gold price, operating costs, royalties, mining dilution, mining recovery, and metallurgical recovery. As additional engineering, cost, and operational information becomes available during future study phases, refinement of these assumptions may result in changes to reserve inventories and project economics.

25.11.3 Mining Methods Risks

- Development of the waste rock storage facilities requires additional geotechnical and waste characterization studies to support detailed design and permitting, including foundation analyses and confirmation of geotechnical design parameters.
- Ore haulage from the Nyam and Kwakyekrom satellite deposits introduces logistical complexity due to the need to transport ore to the central processing facility, creating potential exposure to increased haulage costs and operational disruptions.

- Geotechnical uncertainty associated with pit slope design may lead to future revisions of slope angles, bench configurations, or overall pit designs, potentially affecting mineable reserves, stripping ratios, and operating costs.

25.11.4 Process Design Risks

The following risks are associated with process plant design:

- Comminution variability:
  - High variability in fresh ore comminution characteristics, particularly Boin A×b values, introduces uncertainty in milling circuit design.
  - *Mitigation:* Conduct additional comminution and SMC testwork to improve confidence in design parameters.
- Preg-robbing uncertainty:
  - Some fresh ore exhibit preg-robbing behaviour requiring kerosene addition, which has operational and safety implications.
  - *Mitigation:* Complete further variability and leach testwork to optimize kerosene dosage, confirm its necessity, and evaluate alternative flowsheet options.
- Leach circuit performance:
  - The currently proposed CIL configuration without pre-conditioning or pre-oxidation step in the leach circuit may result in higher cyanide consumption for fresh ores.
  - *Mitigation:* Undertake additional leach optimization testwork for all ore types.
- Cyanide consumption estimates:
  - Current cyanide addition rates are based on preliminary testwork with a discount added, and do not include residual cyanide requirements.
  - *Mitigation:* Refine cyanide consumption and residual cyanide requirements through additional metallurgical testing.
- Residual cyanide in tailings:
  - Absence of a dedicated cyanide detoxification circuit may result in elevated CN_{WAD} concentrations in tailings slurry.

- *Mitigation:* Optimize cyanide usage through further testwork and rely on the planned decant water treatment system to achieve discharge criteria.

- Thickener sizing uncertainty:
  - Current thickener design is based only on static settling testwork which can potentially provide very different results from dynamic settling testwork.
  - *Mitigation:* Conduct dynamic settling and rheology testwork to confirm thickener sizing and underflow density assumptions.

- Kerosene handling and carbon regeneration:
  - Kerosene use may increase risks associated with carbon fouling, kiln off-gases, and reagent handling safety.
  - *Mitigation:* Include off-gas treatment in kiln design, provide appropriate containment and ventilation systems, and use explosion-proof equipment in reagent storage and handling areas.

25.11.5 Infrastructure Risks

Following risks are associated with Infrastructure:

- Earthworks quantities are a key project risk, as the current layout is based on a conceptual design and may require further refinement following detailed geotechnical investigations. Opportunities to reduce cut-and-fill and retaining structures will be explored during the Feasibility Study stage.

- Power supply selection is also a major project risk with both grid connection and onsite generation having different capital costs, schedules, reliability and operating cost implications. It is imperative that power supply strategy be finalized during the Feasibility Study stage.

25.11.6 Tailings Storage Facility Risks

The development of the TSF presents key risks as follows.

- The presence of communities downstream represents a risk, which will require identification of impact zones and development of an Emergency Response and Preparedness plan during future phases of development.

- Artisanal mining pits within the TSF footprint pose a risk to the structural integrity of the facility foundation, potentially creating preferential seepage pathways, subsidence, and slope instability that could compromise embankment stability. Execution of a comprehensive geotechnical investigation to locate and characterize the foundation and backfilling of identified pits, and ongoing foundation monitoring during construction and operations will be required.

- The hilly terrain has an increased susceptibility of disturbed areas experiencing surface erosion and washout, particularly during high rainfall events, potentially silting downstream receptors. Development and implementation of an erosion and sediment control plan will be required.

- The exposure of tailings and disturbed surfaces to wind action presents a dust generation risk, with potential adverse impacts on downstream receptors. A dust suppression program, progressive rehabilitation and revegetation of exposed surfaces, and the establishment of an ambient air quality monitoring program will be required.

25.11.7 Geology and Resources Opportunities

The QPs note the following key potential opportunities:

- An improved understanding of structures (e.g., Checkerboard zone at Sewum) can lead to improved characterization of mineral zones, which will only benefit reconciliation for any future potential mining scenarios in addition to exploration targeting.

- Better comprehension of the relatively high variability ranges identified from the QP check assay program can lead to more robust and precise short-term models for future planning purposes.

- Greater characterization of the oxide type mineralization (i.e., subdomains of lithologies) could potentially lead to the identification of better gold carriers/hosts for future targeting purposes.

- Continued extensional drilling is warranted as many of the deposits and areas of interest remain open along strike and to depth.

- The large land package with untested ground between dominant known mineralized trends could lend towards significant expansion potential via new discoveries.

25.11.8 Mineral Reserves Opportunities

- Conversion of Inferred Mineral Resources to higher-confidence classifications through additional drilling may support future Mineral Reserve growth, extend mine life, and enhance overall project value.

- Refinement of metallurgical, operational, and economic assumptions through future engineering studies may increase reserve recovery, improve project economics, and support growth in the economically mineable portion of the Mineral Resource inventory.

25.11.9 Mining Methods Opportunities

- Optimization of the haulage network may reduce haulage distances and trucking requirements, lowering operating costs and improving mining productivity.

- Further review of drill-and-blast parameters may enhance fragmentation, increase mining productivity, and reduce unit mining costs through better alignment with specific rock types and mining domains.

- Refinement of pit designs through additional geotechnical and operational data may reduce waste stripping requirements, increase recoverable reserves, and improve overall project economics.

- Enhancement of ore transportation and rehandling strategies for the Sewum deposit may reduce transportation costs and improve operational efficiency across the mining operation.

25.11.10 Process Design Opportunities

- Conduct SMC testwork for BWi, Ai, and A x b for diggable transition and blast transition ore to further optimize the milling circuit design and milling power requirements.

- Conduct carbon kinetics tests to generate data on carbon loading capacity and adsorption kinetics for use in CIL modelling and elution plant sizing. Depending on the results, the carbon advance rate may be reduced, potentially allowing for a smaller elution plant.

25.11.11 Infrastructure Opportunities

- Further optimization of the overall infrastructure layout during the Feasibility Study stage presents an opportunity to reduce earthworks quantities, shorten overhead line and overland piping routes and optimize road alignments to reduce both capital and operating costs.

- Advancing the infrastructure design to feasibility level, including completion of geotechnical investigations, detailed material take-offs (MTOs), and optimization of the project power supply strategy, will improve cost estimate accuracy, support procurement and construction planning, and provide greater confidence in the Project's technical and economic viability.

25.11.12 TSF Design Opportunities

The development of the TSF presents key opportunities as follows:

- Favourable storage efficiency due to surrounding high ground: The natural hilly terrain provides containment, significantly reducing the volume of engineered embankment fill required and lowering overall construction costs.

- Availability of On-Site Fill Material: There is a possibility to use locally available low permeability fill material within the facility basin for embankment construction reducing reliance on imported materials and resulting in reductions in haulage costs.

- Efficient Water Management: The natural drainage characteristics of the hilly terrain facilitates effective management of seepage and surface water, supporting the implementation of efficient water management systems.

25.11.13 Environment, Permitting and Social Considerations Opportunities

- Available long term baseline data capable of supporting future EIA studies.

- Demonstrated understanding of project objectives by PAPs which enhances the Project's ability to obtain a social licence.

- RAP studies to provide better understanding of PAPs assets and developing a plan for relocation and resettlement.

25.12 Conclusion

The Mineral Resources currently estimated for the Enchi Gold Project consist of an Indicated Mineral Resource of 1,502,000 ounces of gold (83.6 million tonnes at an average grade of 0.56 g/t gold) and an Inferred Mineral Resource of 626,000 ounces of gold (40.1 million tonnes at an average grade of 0.49 g/t gold). A $3,200/ounce gold price was used to determine the cut-off grade. The Indicated Mineral Resources are reported inclusive of the Mineral Reserves.

The Enchi Gold Project contains Probable Mineral Reserves of 51.3 Mt grading 0.64 g/t Au, containing approximately 1.055 Moz of gold, as estimated from the Boin, Sewum, Nyam, and Kwakyekrom deposits. The Mineral Reserves are based on detailed pit optimization, mine design, metallurgical test work, mining recovery assumptions, operating cost estimates, and economic parameters including a gold price of $3,200/oz.

The PFS evaluates the development of the Enchi Gold Project based on conventional open-pit mining methods and a proven gold processing flowsheet comprised of crushing, milling, carbon-in-leach (CIL), and elution / electrowinning circuits.

The PFS economic analysis shows the Enchi Gold Project is expected to deliver an after-tax NPV of $496 million at a 5% discount rate, an IRR of 37%, and a payback period of 1.6 years. The robust economic indicators demonstrate the Project's technical and financial viability and provide a strong basis for advancing the Project to the next stage of development, including completion of a feasibility study and other detailed engineering work.

26.0 RECOMMENDATIONS

26.1 Path Foward

The results presented in this report demonstrate that the Enchi Gold Project is technically viable. It is recommended that the project be advanced to the feasibility study (FS) phase. A description of the recommended additional work program, organized by functional area, is provided in the following subsections.

Table 26.1.1 Summary of Costs for Future Work

Table	Cost $
Drilling	3,500,000
Geology and Resource	100,000
Mining	700,000
Metallurgy	1,000,000
Process and Infrastructure	1,500,000
Mine Water and Water Management	250,000
Environment and Engagement	750,000
Tailings Storage Facility	1,000,000
Value Engineering and Trade-off Studies	250,000
Total	**9,050,000**

26.2 Geology and Mineral Resource

26.2.1 Drilling and Sampling

- Once drilling is completed, it is recommended that any RC samples to be used are not all left unattended at each site, but rather that a certain proportion of samples are gathered and stored in a monitored location where further sampling, re-sampling or independent check sampling can be carried out without risk of sample contamination by potential illegal gold miners.

- Once all assay results are received from the lab, Newcore can elect for disposal of chip material of samples with no significant gold but should keep witness material (chips and/or lab rejects) of mineralized intervals for future use and independent checking / sampling. Currently all pulps and only selected coarse rejects are stored on site.

- It is recommended to perform additional diamond drilling to help with better defining the oxidation contacts between oxide, transition and fresh rock.

26.2.2 Sample Preparation, Analysis and Security

- It is recommended to do an evaluation of the historical QA/QC, as well as a review of currently available standards, and choosing three standards for each of oxide and fresh mineralization types (six in total) that can be purchased in bulk with no overlap in performance, i.e., low 0.2-0.3 med 1-2 and high 4-5.

- It is recommended to develop a real-time quality detection system and use an action table to document any failures, follow up actions and resolutions.

- Paying close attention to precision and eventually developing a rigorous grade control program to enhance local precision for short term models is important.

26.2.3 Bulk Density

- There is a sufficient number of density tests at Sewum and Boin to estimate density within the block model. Additional data should be collected for both Nyam and Kwakyekrom to improve the understanding of the density of these deposits within future block models.

- Additional density in-situ density test pits should be considered for near surface oxides at all deposits.

- Implementing the use of low density-fluid in the methodology for density data measurements for specific low-density oxides may be helpful. Alternatively, recording water temperature or replacing water frequently to keep temperatures in the 4 to 10º Celsius range is recommended.

- Density measurements should be selected to be robust and reflective of the drillhole, representative measurements should be taken within each lithology and mineralization and waste type.

26.2.4 Check Assay Sampling

- Additional sampling is recommended to better understand the relatively high variability ranges observed during DRA's QP check program. It is recommended to conduct regular check assay programs (on the order of 5%) for both RC and DD campaigns that are completed across the property.

- It is important to use a secondary lab for check assay sampling to assess accuracy; precision could also be monitored via resubmission to the same primary lab (with new sample identifiers). It would be ideal to perform checks on all material types (core, rejects and pulps).

- For core check assays, ensure both preparation and analysis is conducted at the secondary lab.
- Newcore should also continue to review lab precision, along with modelling of the nugget effect.

26.2.5 Geological Modelling

- It is recommended that 3D modelling is conducted in real time; downhole surveys should be imported and reviewed during drilling, allowing erroneous surveys to be discovered in real time. Additional multishot or gyro surveys can also be conducted at hole closure if warranted.
- Sections should be supplied to logging geologists to maintain consistency of logging codes. End of hole procedures should be tightened to ensure all relevant information is readily usable.
- Logging database should be source of truth, i.e., all subprojects and changes should be reflected in the master database and commented upon.
- If real-time modelling is not available, paper sections should be supplied, and continual interpretations should be made.
- Additional attention should be made to lithological, alteration and oxidation modelling. Vein modelling data should be standardized and potentially warranted to have its own data sheet. All veins should make it to the single sources of truth, even if they are primary lithology, secondary lithology, thin structure, etc.

26.2.6 Geological Modelling for Sewum

- The checkerboard zone at Sewum has two directions of mineralization, one orthogonal to the primary orientation. It is recommended to conduct additional exploration to resolve the orientation and location, width and grade distribution of secondary structures.

26.3 Mining and Mineral Reserves

This section outlines the recommended activities required to advance the Mineral Reserve Estimate, mining methods, and pit geotechnical design from the Pre-Feasibility Study (PFS) level to Feasibility Study (FS) level. The recommended work is focused on reducing technical uncertainty, improving confidence in key mine planning assumptions, and refining the basis for Mineral Reserves, pit designs, and operating cost estimates.

26.3.1 Mineral Reserves

- Incorporation of updated pit optimization inputs based on further details that were determined during the PFS study completion, including revised G&A cost assumptions and royalty rates.

- Reassess ore loss and mining recovery factors based on anticipated equipment fleet, bench configuration, and ore-waste contact complexity.
- Validate Selective Mining Unit (SMU) dimensions against planned mining equipment sizes, ore selectivity capabilities, and grade control practices.
- Targeted infill drilling programs within economically significant Inferred Mineral Resource areas with the potential for upgrading to an Indicated or Measured classification.

26.3.2 Mining Methods

- Review and optimize secondary ore haulage requirements throughout the mine life, including identification of the economic transition point where ore should be transferred from the primary mining fleet to a dedicated secondary haulage fleet.
- Evaluate potential haul road alignments and required earthworks to identify the optimum balance between upfront road construction costs and long-term haulage cost savings.
- Refine the life-of-mine haulage network plan incorporating phased road development, pit access requirements, and integration with key mine infrastructure and material destination points.
- Evaluate WRSF locations considering geotechnical, environmental, permitting, and operational factors, including assessment of surface preparation requirements, development sequencing, and associated mining costs.
- Optimize drill-and-blast design parameters and fragmentation targets to reduce waste mining costs, improve loading and hauling productivity, and support mine-to-mill optimization of ore material.

26.3.3 Pit Geotechnical

- Complete additional geotechnical drilling across all planned pits and deposits to reduce uncertainty related to lithological variability, weathering profiles, structural controls, and geotechnical domains, supporting refinement of inter-ramp and overall slope angle designs.
- Maximize the geotechnical data collection from ongoing resource drilling programs to help refine IRA and OSA parameters and strengthen confidence in the geotechnical model.
- Complete additional geotechnical and rock mechanics testing, particularly within fresh and transition materials and the varying lithologies across the Shear Zone, to expand the geotechnical database and improve confidence in the selected geotechnical design parameters.

26.4 Metallurgy

The following testwork is recommended to progress the design in the FS phase:

Sample Requirement

- A set quantity of master composites, variability samples and some bulk samples for each mineralization type using interval composite samples from drill cores. Number of representative samples to be determined by Newcore based on the mine plan.

Comminution Testwork

- Additional comminution testwork is recommended for ore types and deposits where design parameters were inferred rather than determined through direct testing, to further improve confidence in process design and equipment sizing.
- Specifically, additional SMC testwork for BWi, Ai, A x b for diggable transition and blast transition ore is recommended to improve the comminution database and support further optimization of milling circuit design and power requirements.

Ore Characterization Testwork

- Full head assays for Au, Ag, As, Fe, Sb, Hg, Te, Cu, Cu_{CNSol}, C_{total}, C_{org}, CO_3^{2-}, S_{total}, $S_{sulphide}$, and multi-element ICP Scan for master and variability samples.
- Additional mineralogy testwork to further assess gold deportment in each mineralization type.
- Bulk materials handling testwork to confirm bulk density, angle of repose, angle of surcharge, and other bulk solids handling properties.

Gravity-Leach Testwork

- Additional extended gravity separation testwork along with gravity circuit modelling to confirm gravity concentrator selection.
- Additional leach testwork on master composites to further optimize leach conditions.
- Additional variability samples to further assess variation in gold recovery across different orebody.
- Carbon kinetic tests to generate data on carbon loading capacity and kinetics for use in the CIL modelling or elution plant sizing.

- Oxygen uptake rate tests to simulate oxygen consumption in the leaching process for proper oxygen plant sizing.
- Gravity-leach testing on bulk samples to confirm flowsheet at optimum conditions at a larger scale.

Settling Testwork

- Dynamic settling testwork on master composites and bulk samples to confirm pre-leach thickener sizing under continuous mode.
- Rheology testwork to determine critical slurry viscosity and settling rate for pump design.
- Cyanide Detoxification Testwork:
  - Conduct cyanide detox tests on both tails slurry and tails solution to generate data for a detox circuit or water treatment plant design.

Water Testwork

- Conduct water analysis for site water samples (include pH, S.G.).
- Buffer curve determination (pH vs. lime addition).

Tailings Testwork

- Geochemical and geotechnical testwork to characterize the tailings for the TSF design.
- Geochemical testing typically include:
  - Acid-base accounting (ABA)
  - Net acid generation (NAG) testing
  - Shake flask extraction testing
  - Humidity cell testing for long-term leaching and drainage characterization.
- Geotechnical testing typically include:
  - Particle size distribution (sieve and hydrometer analysis)
  - Specific gravity determination

- Atterberg limits
- Standard and Modified Proctor compaction tests
- Consolidation testing (oedometer and constant rate of strain)
- Soil-moisture characteristic testing
- Shear strength testing (triaxial and direct shear)
- Permeability testing.

The cost of a full FS testwork program is estimated to range from $1.0 million to $1.5 million, depending on the number of samples collected and the scope of testing required to support detailed engineering and design optimization.

26.5 Infrastructure

The following engineering activities are recommended to advance the site infrastructure from the Pre-Feasibility Study (PFS) to the Feasibility Study (FS) level.

- Refine the overall site layout to optimize the arrangement of the process plant, administration facilities, workshops, warehouses, stockpiles, utilities, and supporting infrastructure while maintaining operational efficiency, maintenance access and future expansion capability.
- The overall process plant layout should be reviewed and optimized, including consideration of terraced facility arrangements and revised platform elevations, to reduce cut-and-fill quantities, minimize retaining structures where practical, improve constructability, and maintain adequate site drainage.
- Undertake detailed geotechnical investigations across all major infrastructure footprints, including process plant facilities, proposed power plant location, stockpiles, and utility corridors, to confirm foundation design parameters.
- Complete detailed engineering of all site access roads, mine haul roads, and process plant roads, including pavement design, drainage structures, culverts, erosion protection measures, and optimized road alignments.
- Develop detailed material take offs (MTOs) for concrete, structural steel, platework and piping to support a Class 2 capital cost estimate.

- Complete a detailed trade-off study between grid power and onsite power generation considering capital cost, operating cost, implementation schedule, reliability and expansion capability.

26.6 Mine Water and Water Management

26.6.1 Surface Water Management

- Perform a detailed design of channel cross section to optimize material quantities.
- Final verification of existing roads and new ones to assure the feasibility of river reaches and channels at each crossing.
- Sediment load rates and properties (median diameter and density) in pits dewatering flows to better assess the sediment basins capacity configuration and location.

26.6.2 Geochemistry – coverage and representativeness

- Review existing ABA / kinetic program against BFS standards:
  - Supplement geochemical testing where gaps are identified: Additional ABA and NAG tests on under-represented units, especially (High-sulphur and high-As zones, Footwall and hanging-wall rocks likely to form significant portions of WRDs, Material likely to be used in construction (TSF shells, paddocks, road fill).
  - Additional kinetic tests: Long-term column tests on representative waste classes and tailings, under site-specific conditions (e.g., site water rather than rainwater only, including redox cycling). At least a subset including blended waste types representative of actual TSF and WRD construction practices (e.g., interbedded NAF/PAF, low-grade stockpiles if planned).

26.6.3 Hydrogeology

- Integrate all relevant datasets into a single GIS / database (this has mostly been done as part of the PFS assessment).
- Critically assess current PFS-level conceptual model (this has mostly been done as part of the PFS assessment). Evaluate reliability of, Reported weathering profile, Hydrostratigraphic unit extents and connectivity, Structural controls (shears, faults, veins, graphitic units) and recharge estimates and discharge areas (rivers, springs, baseflow).

- Gap analysis and BFS requirements (this has mostly been done as part of the PFS assessment); Areas with no or sparse groundwater level data (especially up- and down-gradient of pits, WRDs, TSFs), Pits or facilities with few or no aquifer tests, Temporal gaps, Limited multi-season or multi-year monitoring of groundwater levels and streamflows.
- Ground geophysical surveys: Implement targeted geophysics (e.g., ERT, EM and/or IP) in mine corridors and facility areas to delineate major structures (faults / shears) acting as likely high-K pathways or hydraulic barriers. Priority areas include around and between Boin, Nyam, Sewum and Kwakyekrom pits, and under and adjacent to planned TSFs and key WRDs.
- Installation and construction of dedicated groundwater monitoring boreholes. Aquifer testing of newly drilled borehole at each main pit cluster (Boin, Sewum, Nyam). Use automated loggers (1–5 min intervals early time, then longer) in pumping and observation wells.
- Update the 3D Leapfrog Hydro model to BFS level.
- Predictive groundwater flow simulations.
- Contaminant migration modelling.
- Dewatering and water-supply design support.

26.7 Tailings Storage Facility

This section outlines the recommended technical investigations, engineering studies, and design development activities required to advance the TSF from PFS level to Feasibility Study (FS). The recommendations are structured to progressively reduce technical uncertainty, refine capital and operating cost estimates, and confirm compliance with applicable local regulations and international best practice standards.

26.7.1 Foundation Characterization

- Characterize subsurface conditions across the TSF's footprint to confirm foundation competence and design parameters.
- A detailed geotechnical site investigation comprising borehole drilling with in situ testing (SPT, Falling head permeability, packer testing,) and Undisturbed sampling.
- Geotechnical and geochemical laboratory testing program on sampled materials.
- Installation of piezometers and monitoring wells within the TSF's basin.

26.7.2 Borrow Source Characterization

- Confirm availability, quality, and engineering properties of construction materials.
- Undertake test pitting program to identify low permeable earth fill material.
- Sample sources of filter and structural fill material.
- Conduct a geotechnical and geochemical laboratory testing program on sampled materials.

26.7.3 Tailings Characterization

Develop understanding of tailings properties for beach slope prediction, consolidation modeling, and long-term stability analysis by undertaking the following.

- Obtain representative tailings samples from metallurgical test work.
- Conduct settling column tests and thickening tests to validate design solids concentrations.
- Perform beach slope angle testing.
- Conduct consolidation testing to determine time-rate settlement parameters.
- Undertake rheology testing of tailings to inform Dam Breach Assessment.
- Geochemical characterization including acid-base accounting, leach testing, and mineralogical analysis.

26.7.4 Design Refinements and Engineering Analyses

- Commission site-specific Probabilistic Seismic Hazard Assessment (PSHA) to determine:
  - Operating Basis Earthquake (OBE) and Maximum Design Earthquake (MDE)
  - Site-specific design response spectra
  - Liquefaction potential of foundation soils and tailings.
- Refine slope stability analyses incorporating updated foundation and fill material characterization.
- Conduct dynamic deformation analysis (Newmark displacement or FLAC dynamic analysis).

- Complete Dam Break and Inundation Study for consequence classification with input from the FMEA.
- Refine facility water balance model to predict safe water storage volumes on the TSFs.
- Develop mixing model.

26.7.5 Closure and Reclamation Planning

- Develop conceptual closure plan including:
  - Final landform design and contouring
  - Cover system design
  - Post-closure monitoring and maintenance programs
  - Estimate closure and reclamation costs with annual provisioning schedule.

26.7.6 Infrastructure and Supporting Systems

- Undertake detailed design of Tailings distribution pipeline: hydraulic modelling, material selection, alignment optimization, air release and flushing systems.
- Pump design: detailed mechanical and electrical design, redundancy provisions, power supply requirements.
- Water management: final sizing with hydraulic model verification, energy dissipation design.
- Instrumentation: comprehensive monitoring plan including piezometers, inclinometers, settlement monuments, and real-time monitoring systems with alarm thresholds.
- Access roads and site facilities: final alignment, structural pavement design, drainage systems.

26.7.7 Risk Management

- Conduct comprehensive Failure Modes and Effects Analysis workshop identifying:
  - Potential failure modes (overtopping, slope instability, seepage, piping, seismic liquefaction)
  - Consequences and likelihood of failure scenarios

  - Critical controls and mitigation measures.

- Develop Monitoring and trigger action response plans.
- Develop Tailings Management System aligned with GISTM requirements.

26.8 Environmental, Social and Community Engagement

- Conduct full Environmental Impact Assessment (EIA) in line with IFC Performance Standards.
- Conduct full Resettlement Action Plan (RAP) for Kangaboin, Sewum and other communities within the project footprint.

26.9 Feasibility Study

To advance the Enchi Gold Project to the next stage of development, it is recommended that a feasibility-level engineering study be undertaken to increase project definition and improve the accuracy and confidence of capital and operating cost estimates. This work should include further optimization of the mine plan, refinement of the process plant design based on the results of additional metallurgical testwork, and additional engineering across all major disciplines, including mechanical, civil, structural, electrical, and instrumentation. The Feasibility Study should also generate detailed material take-offs (MTOs), equipment specifications, and construction quantities to support more robust cost estimation, and project execution planning.

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28.0 QP CERTIFICATES

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CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*NI 43-101 Technical Report, Prefeasibility Study on the Enchi Gold Project, Ghana*", dated August 7, 2026 with an effective date of June 23, 2026 (the "Technical Report"), prepared for Newcore Gold Ltd. ("Newcore" or the "Company").

I, *Ryan Wilson, P.Geo.,* do hereby certify that:

1. I am Geological Mining Specialist with DRA Americas Inc., located at 555 Blvd René-Lévesque West, 6th Floor, Montreal, Quebec, Canada H2Z 1B1.

2. I am a graduate of University of Ottawa, Ottawa, Ontario, Canada in 2007 with a B.Sc. in Earth Sciences and in 2012 with an M.Sc. in Economic Geology, and a graduate of McGill University, Montreal, Quebec, Canada in 2022 with a Ph.D. in Mining Engineering.

3. I am registered as a Professional Geologist in the Province of Ontario (PGO Reg. #2511) and in the Province of Quebec (OGQ Reg. #10435).

4. I have worked and conducted research in the geological sciences and mining sector continuously since my graduation in 2007.

   I have worked on similar projects to the Enchi Gold Project in North America, South America and Australia; my experience for the purpose of the Technical Report includes:

   - Over 15 years of experience in exploration, mining and metals split between industry and specialized research. Specifically, 8 years of experience focused on intrusion-related and orogenic gold deposits in Timmins gold camp, Timmins, Ontario, Canada.
   - Technical assistance in exploration, geology and resources for a variety of projects from greenfield exploration to active mine operations in Canada. Geostatistical assistance in project evaluation for multiple projects in Australia. Additional research and collaboration on several mine-to-plant simulation studies in Canada and Chile.
   - Participation in the preparation of multiple NI 43-101 Technical Reports.

5. I have read the definition of "qualified person" set out in the National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be an independent qualified person for the purposes of NI 43-101.

6. I am independent of the Company applying all the tests in section 1.5 of NI 43-101.

7. I have participated in the preparation of this Technical Report and am responsible for Sections 4 to 10, 14 (except for 14.4 and 14.5), and 23, and portions of Sections 1, 25, 26 and 27 of the Technical Report.

8. I have not visited the property that is the subject of the Technical Report.

9. I have had prior involvement with the property that is the subject of the Technical Report:

    - *Served as QP for "Technical Report – Mineral Resource Estimate Update – Enchi Gold Project, Ghana" prepared by DRA Americas Inc., with an effective date of March 18, 2026 (issued Report Date May 1, 2026, prepared for Newcore Gold Ltd.).*

10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.

11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 07th day of August 2026, Montreal, Quebec.

"Original Signed on file"
Ryan Wilson, P. Geo.
Geological Mining Specialist
DRA Americas Inc.

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I, *Matthew Halliday, P.Geo.,* of Haileybury, Ontario, do hereby certify:

1. I am a Senior Geologist with DRA Americas Inc., located at 555 Blvd René-Lévesque West, 6th Floor, Montreal, Quebec, Canada H2Z 1B1.
2. I am a graduate of Dalhousie University in 2008 with a B.Sc. in Earth Sciences.
3. I am a registered Member of Professional Geoscientist of Ontario (PGO) (#2605) and of *Ordre des géologues du Québec* (OGQ) (#02445).
4. I have worked as a geologist in various capacities since my graduation from university in 2008.

   My relevant experience after graduation, for the purpose of the Technical Report:

   - Over 18 years of experience in all aspects of mineral exploration and mineral resource estimations for gold and base metals projects and deposits in Canada and in the USA.
   - Participation in the preparation of several NI 43-101 Technical Reports.
5. I have read the definition of "qualified person" set out in the National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be an independent qualified person for the purposes of NI 43-101.
6. I am independent of the Company applying all the tests in section 1.5 of NI 43-101.
7. I am responsible for the preparation of Sections 11 and 14.5, and portions of Sections 1 and 25 to 27 of the Technical Report.

8. I have not visited the Property that is the subject of this Technical Report.

9. I have had prior involvement with the property that is the subject of the Technical Report:

    - *Served as QP for "Technical Report – Mineral Resource Estimate Update – Enchi Gold Project, Ghana" prepared by DRA Americas Inc., with an effective date of March 18, 2026 (issued Report Date May 1, 2026, prepared for Newcore Gold Ltd.).*

10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.

11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 7th day of August 2026, Haileybury, Ontario

"Original Signed on file"
Matthew Halliday, P.Geo.
Senior Geologist
DRA Americas Inc.

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CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*NI 43-101 Technical Report, Prefeasibility Study on the Enchi Gold Project, Ghana*", dated August 7, 2026 with an effective date of June 23, 2026 (the "Technical Report"), prepared for Newcore Gold Ltd. ("Newcore" or the "Company").

I, *Schadrac Ibrango, P.Geo., Ph.D., MBA,* do hereby certify that:

1. I am a Lead Geology and Hydrogeology Consultant with DRA Americas Inc., located at 555 Blvd René-Lévesque West, 6th Floor, Montreal, Quebec, Canada H2Z 1B1.
2. I am a graduate from University of Ouagadougou (Burkina-Faso) with a Master's Degree in Geology in 1998, a Ph.D. in Engineering of Darmstadt University of Technology (Germany) in 2005 and an executive MBA from Université du Québec à Montréal (Canada) in 2016.
3. I am a registered member of the *Ordre des Géologues du Québec (OGQ)* (Reg. # 1102) and Professional Engineers & Geoscientists of Newfoundland and Labrador (Reg. # 07633).
4. I have worked continuously as a geologist for more than 25 years in the mining industry since my graduation from university.

   I have worked on similar projects to the Enchi Gold Project in Canada and in Africa; my experience for the purpose of the Technical Report includes:

   - Hands-on experience in exploration for gold, silver, iron ore, potash, graphite, lithium, PGE and base metal deposits.
   - Acting as resource estimation QP for gold, lithium, graphite and rare earth elements in Canada.
   - Design and supervision and implementation of drilling programs.
   - Review, audits, interpretation of geoscientific data.
   - Participant and author for various NI 43-101 Technical Reports.
5. I have read the definition of "qualified person" set out in the National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be an independent qualified person for the purposes of NI 43-101.

6. I am independent of the Company applying all the tests in section 1.5 of NI 43-101.

7. I am responsible for the preparation of Sections 12 and 14.4, and portions of Sections 1 and 25 to 27 of the Technical Report.

8. I visited the property that is the subject of the Technical Report from September 23 to 25, 2025.

9. I have had prior involvement with the property that is the subject of the Technical Report:

    - *Served as QP for "Technical Report – Mineral Resource Estimate Update – Enchi Gold Project, Ghana" prepared by DRA Americas Inc., with an effective date of March 18, 2026 (issued Report Date May 1, 2026, prepared for Newcore Gold Ltd.).*

10. I have read National Instrument 43-101 and Form 43-101F and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.

11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this x07th day of August 2026.

"Original Signed on file"
Schadrac Ibrango, P.Geo., Ph.D., MBA
Lead Geology and Hydrogeology Consultant
DRA Americas Inc.

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CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*NI 43-101 Technical Report, Prefeasibility Study on the Enchi Gold Project, Ghana*", dated August 7, 2026 with an effective date of June 23, 2026 (the "Technical Report"), prepared for Newcore Gold Ltd. ("Newcore" or the "Company").

I, *Claude Bisaillon, P. Eng.,* do hereby certify that:

1. I am Senior Geotechnical Engineer with DRA Americas Inc., located at 555 Blvd René-Lévesque West, 6th Floor, Montreal, Quebec, Canada H2Z 1B1.
2. I am a graduate from Concordia University in Montreal in 1991 with a B.Sc. in Geology and from the Université Laval in Quebec City in 1996 with a B.Ing. in Geological Engineering.
3. I am a member in good standing of the *Ordre des Ingénieurs du Québec* (OIQ #116407). I am a Member of the Canadian Institute of Mining, Metallurgy and Petroleum.
4. I have worked as an engineer continuously since graduation from university in 1996.

   I have worked on similar projects to the Enchi Gold Project in North America, South America and Australia; my experience for the purpose of the Technical Report includes:

   - Over 28 years of consulting in orebody modelling, pit wall slope stability, and geotechnical engineering.
   - Technical assistance in exploration, geology and resources for a variety of projects from greenfield exploration to active mine operations in various countries.
   - Participation in the preparation of multiple NI 43-101 Technical Reports.
5. I have read the definition of "qualified person" set out in the National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be an independent qualified person for the purposes of NI 43-101.

6. I am independent of the Company applying all the tests in section 1.5 of NI 43-101.
7. I am responsible for Sections 15.5.1 and 16.2.1, and portions of Sections 1 and 25 to 27 of the Technical Report.
8. I have not visited the property that is the subject of the Technical Report.
9. I have had no prior involvement with the property that is the subject of the Technical Report.
10. I have read National Instrument 43-101 and Form 43-101F and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 07th day of August 2026, Montreal, Quebec.

"Original Signed on file"
Claude Bisaillon, P. Eng.
Senior Geotechnical Engineer
DRA Americas Inc.

Certificate of Qualified Person
Grant Carlson, P.Eng.

I, Grant Carlson, P.Eng., certify that I am employed as a Director, Technical Services with Fuse Advisors Inc., with an office address of 595 Burrard Street, Suite 3113, Three Bentall Centre, Vancouver, BC, V7X 1J1.

This certificate applies to the technical report titled “NI 43-101 Technical Report, Prefeasibility Study on the Enchi Gold Project, Ghana” that has an effective date of June 23, 2026 (PFS or Technical Report).

1. I graduated from the University of British Columbia in 2007 with a Bachelor of Applied Science in Mining Engineering.
2. I am a Professional Engineer registered with Engineers and Geoscientists of BC (No. 36395).
3. I have practiced my profession for 18 years. I have been directly involved in mining operations in Canada, in similar climatic conditions as the Enchi Gold Project, as a planning engineer, shift supervisor, mine manager, chief operating officer, and mining consultant. I have worked on, visited, and carried out due diligences on precious metal, base metal, diamond, and coal mining operations around the world including Russia, Madagascar, Australia, Eritrea, Lesotho, Ghana, Sierra Leon, Brazil, Peru, Chile, Argentina, Suriname, Guyana, Mexico, the United States, and Canada.
4. I have read the definition of “Qualified Person” set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) and certify that by virtue of my education, affiliation to a professional association, and past relevant work experience, I fulfil the requirements to be a “Qualified Person” for those sections of the Technical Report that I am responsible for preparing.
5. I have not visited the Enchi Gold Project site.
6. I am responsible for Sections 1.10, 1.11, 1.18.4, 1.18.5, 15, 16 (excluding 16.2.1), 18.2.2, 18.3, 18.13, 18.14, 21.3.6, 21.4.2, 21.5.1, 25.4, 25.11.2, 25.11.3, 25.11.8, 25.11.9, 26.3 and 26.6 of the Technical Report.
7. I am independent of Newcore Gold Ltd. as independence is defined in Section 1.5 of NI 43-101.
8. I have not been previously involved with the Enchi Gold Project.
9. I have read NI 43-101 and the sections of the Technical Report that I am responsible for have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: 7 August 2026

signed and sealed

Grant Carlson, P.Eng.

Knight Piésold CONSULTING

CERTIFICATE OF QUALIFIED PERSON

I, Jorge Velázquez Suárez, as an author of this report entitled "NI 43-101 Technical Report, Prefeasibility Study on the Enchi Gold Project, Ghana", prepared for Newcore Gold Ltd. (the "Company") and dated August 7, 2026, with an effective date of June 23, 2026 (the "Technical Report"), do hereby certify that:

1) I am a Discipline Leader with Knight Piésold Consulting S.A. de C.V., with a business address at Presidente Masaryk 29 – 11, Miguel Hidalgo, Mexico City, Mexico..

2) I hold a Civil Engineering degree from the National Autonomous University of Mexico (UNAM, 2014) and a Master of Science in Hydrology and Water Resources from the University of California, Los Angeles (UCLA, 2016).

3) My relevant experience for the purpose of the Technical Report includes:

- Over 10 years of experience in hydrology and water management projects across various industries, including mining, infrastructure, and environmental sectors.
- More than 5 years of specialized experience in water and tailings management at mining sites.
- Development and management of hydrological and hydraulic projects, including water balance modeling, surface water management, climatological and hydrological studies for mining projects.}

4) I have read the definition of 'qualified person' set out in National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* (“NI 43-101”) and certify that by virtue of my education, 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 Enchi Gold Project site.

6) I am responsible for Sections 18.8 and 18.9 of the Technical Report.

7) I am independent of the Company in accordance with section 1.5 of NI 43-101.

8) I have read NI 43-101, and the Technical Report, and confirmed that the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101 and Form 43-101F1.

9) As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 7th day of August 2026

Signed and Sealed

Jorge Velázquez Suárez.

Knight Piésold CONSULTING

CERTIFICATE OF QUALIFIED PERSON

I, Hugues Ngoy Nkulu, PMP., as an author of this report entitled "NI 43-101 Technical Report, Prefeasibility Study on the Enchi Gold Project, Ghana", prepared for Newcore Gold Ltd. (the "Company") and dated 7 August 2026, with with an effective date of 23 June 2026 (the "Technical Report"), do hereby certify that:

1) I am a Lead Hydrogeologist with Knight Piesold Consulting RDC SAS., with a business address at 16.515, Avenue Rubi, Quartier Golf Météo, Commune de Lubumbashi, Lubumbashi Haut-Katanga, Democratic Republic of Congo.

2) I hold a Master of Business Administration from UNICAF, awarded in 2026; and a Bachelor's degree in Applied Science, honours in Geology from the University of Lubumbashi, D.R. Congo, awarded in 2017.

3) I am a Member of The International Association of Hydrogeologists (144674) and registered as a Professional Member of the Ground Water Division of the Geological Society of South Africa (UID: 1675/25), Project Management Professional – Project Management Institute (PMP ID 7869192), Fellow of the Council of Geologist in the Democratic Republic of the Congo (OGRDC000672017).

4) My relevant experience for the purpose of the Technical Report includes:

   - Over 10 years of experience with hydrogeology of gold projects.
   - Lead hydrogeologist for several studies including PEA, PFS and FS.

5) I have read the definition of 'qualified person' set out in National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* ("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 Enchi Gold Project site.

7) I am responsible for Sections 18.10.1, 18.10.2,18.10.4-18.10.7 of the Technical Report.

8) I am independent of the Company in accordance with section 1.5 of NI 43-101.

9) I have read NI 43-101, and the Technical Report, and confirmed that the sections of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101 and Form 43-101F1.

10) As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 7th day of August 2026

Signed and Sealed

Hugues Ngoy Nkulu, PMP

Knight Piésold CONSULTING

CERTIFICATE OF QUALIFIED PERSON

I, George Lartey-Young, C.Env., as an author of this report entitled "NI 43-101 Technical Report, Prefeasibility Study on the Enchi Gold Project, Ghana", prepared for Newcore Gold Ltd. (the "Company") and dated 7 August 2026, with with an effective date of 23 June 2026 (the "Technical Report"), do hereby certify that:

1) I am a Senior Environmental Scientist with Knight Piésold Ghana Limited, with a business address GD-162-3251, Ntiful Avenue American House, East Legon Accra | Ghana.

2) I hold a Ph.D. in Environmental Science and Engineering from Tongji University, awarded in 2023; a Master's degree in Environmental Assessment and Management from the University of Salford, United Kingdom, awarded in 2012; and a Bachelor's degree in Environmental Science from the University of Cape Coast, Ghana, awarded in 2010.

3) I am a Chattered Environmentalist of the Society of Environment (SocEnv), United Kingdom (Number: 10857) and a registered Member of the Institution of Environmental Sciences (IES), United Kingdom (Number: 62294) 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 10 years of experience with environmental assessment and management of gold projects
- Environmental assessment study lead for several studies including PEA, PFS and FS

5) I have read the definition of 'qualified person' set out in National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* ("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 visited the Enchi Gold Project site from 9th to 13th February 2026.

7) I am responsible for Sections 1.14, 1.15.4, 1.18.9, 20, 21.6,25.7, 25.11.13, 26.8 of the Technical Report.

8) I am independent of the Company in accordance with section 1.5 of NI 43-101.

9) I have read NI 43-101, and the Technical Report, and confirmed that the sections of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101 and Form 43-101F1.

10) As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 7th day of August 2026

Signed and Sealed

George Lartey-Young, C.Env

CERTIFICATE OF QUALIFIED PERSON

I, Ama Nketiah, P.E., as an author of this report entitled "NI 43-101 Technical Report, Prefeasibility Study on the Enchi Gold Project, Ghana", prepared for Newcore Gold Ltd. (the "Company") and dated 7 August 2026, with an effective date of 23 June 2026 (the "Technical Report"), do hereby certify that:

1) I am a Principal Engineer with Knight Piésold Ghana Limited, with a business address GD-162-3251, Ntiful Avenue American House, East Legon Accra | Ghana.

2) I hold a Masters in Engineering and Management from Coventry University, United Kingdom, awarded in 2016; and a Bachelor's degree in Civil Engineering from the Kwame Nkrumah University of Science and Technology, Ghana, awarded in 2006.

3) I am a registered Professional Engineer of the Ghana Institution of Engineers (License Number: 06577). 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 15 years of experience with Tailings Storage Facility (TSF) design for gold projects.
   - Project management for several studies including PEA, PFS and FS.

5) I have read the definition of 'qualified person' set out in National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* ("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 Enchi Gold Project site.

7) I am responsible for Sections 18.6, 18.7, 1.18.8, 21.4.4, 21.5.3, 25.11.6,25.11.12, 26.7 of the Technical Report.

8) I am independent of the Company in accordance with section 1.5 of NI 43-101.

9) I have read NI 43-101, and the Technical Report, and confirmed that the sections of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101 and Form 43-101F1.

10) As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 7th day of August 2026

Signed and Sealed

Ama Nketiah, P.E (MGhIE)

Knight Piésold CONSULTING

CERTIFICATE OF QUALIFIED PERSON

I, Martiens Prinsloo, Pr.Sci.Nat., as an author of this report entitled "NI 43-101 Technical Report, Prefeasibility Study on the Enchi Gold Project, Ghana", prepared for Newcore Gold Ltd. (the "Company") and dated 7 August 2026, with an effective date of 23 June 2026 (the "Technical Report"), do hereby certify that:

1) I am Section Manager: Hydrogeology with Knight Piésold Consulting., with a business address at 1 Discovery place, Sandton, Johannesburg, South Africa, 2916.

2) I hold an MSc (Geohydrology) degree from the University of the Free State, South Africa (2005), a BSc Honours degree in Geohydrology from the University of the Free State (1997), and a BSc degree in Earth Sciences from the University of Pretoria, South Africa (1996).

3) I am registered as a Natural Scientific Professional (Pr.Sci.Nat) in South Africa (Registration Number 400248/04). 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 25 years of experience with hydrogeological and environmental impact assessments
   - Hydrogeology study lead for more than 200 studies including PEA, PFS and FS

5) I have read the definition of 'qualified person' set out in National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* ("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 Enchi Gold Project site.

7) I am responsible for Sections 18.10.3 of the Technical Report.

8) I am independent of the Company in accordance with section 1.5 of NI 43-101.

9) I have read NI 43-101, and the Technical Report, and confirmed that the sections of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101 and Form 43-101F1.

10) As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 7th day of August 2026

Signed and Sealed

Martiens Prinsloo, Pr.Sci.Nat.

CERTIFICATE OF QUALIFIED PERSON

I, Zunedbhai Shaikh, P. Eng., as an author of this report entitled “NI 43-101 Technical Report, Prefeasibility Study on the Enchi Gold Project, Ghana”, prepared for Newcore Gold Ltd. (the “Company”) and dated 7 August 2026, with an effective date of 23 June 2026 (the “Technical Report”), do hereby certify that:

1) I am a Lead Mechanical Engineer with Lycopodium Minerals Canada Ltd., with a business address at 5090 Explorer Drive, Suite 700, Mississauga, ON L4W 4T9, Canada.

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 practiced my profession in the mineral processing industry continuously since graduation.

4) My relevant experience for the purpose of the Technical Report includes:

   - Lead Mechanical Engineer on a number of studies and detailed design projects in the mineral processing industry in Africa, Latin America and North America.

   - Commissioning Engineer at a number of mines in Africa, Latin America and North America.

5) I have read the definition of 'qualified person' set out in National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* (“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 Enchi Gold Project site.

7) I am responsible for Sections 1.13, 1.18.7, 18.1, 18.2, 182.1, 18.4, 18.5, 18.11, 18.12, 18.15, 25.6, 25.11.5, 25.11.11 and 26.5 of the Technical Report.

8) I am independent of the Company in accordance with section 1.5 of NI 43-101.

9) I was involved in the previous Technical Report on the Enchi Gold Project’s preliminary economic assessment published in June 2024 and mineral resource update published in May 2026.

10) I have read NI 43-101, and the Technical Report, and confirmed that the sections of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101 and Form 43-101F1.

11) As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 7th day of August 2026

Signed and Sealed

Zunedbhai Shaikh, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

I, Preetham Nayak, P. Eng., as an author of this report entitled "NI 43-101 Technical Report, Prefeasibility Study on the Enchi Gold Project, Ghana", prepared for Newcore Gold Ltd. (the "Company") and dated 7 August 2026, with an effective date of 23 June 2026 (the "Technical Report"), do hereby certify that:

1) I am a Study Manager with Lycopodium Minerals Canada Ltd., with a business address at 789 West Pender Street, Suite 570, Vancouver, BC, V6C 1H2.

2) I am a graduate of the University of British Columbia with a Masters 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 (Licence 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 15 years of experience with design and management of gold projects
   - Study manager for several studies including PEA, PFS and FS
   - Development, execution and interpretation of capital cost estimate and cashflow modelling in gold and copper projects.

5) I have read the definition of 'qualified person' set out in National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* ("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 Enchi Gold Project site.

7) I am responsible for Sections 1.1, 1.15.1, 1.15.3, 1.16, 1.17, 1.18.1, 2, 3, 19, 21.1, 21.2, 21.3.1-21.3.5, 21.3.7, 21.3.8, 21.5.2, 22, 24, 25.1, 25.8, 25.10, 25.12, 26.1, 26.9 and 27 of the Technical Report.

8) I am independent of the Company in accordance with section 1.5 of NI 43-101.

9) I was involved in the previous Technical Report on the Enchi Gold Project's preliminary economic assessment published in June 2024 and mineral resource update published in May 2026.

10) I have read NI 43-101, and the Technical Report, and confirmed that the sections of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101 and Form 43-101F1.

11) As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 7th day of August 2026

Signed and Sealed

Preetham Nayak, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

I, Ryda Peung, P. Eng., as an author of this report entitled "NI 43-101 Technical Report, Prefeasibility Study on the Enchi Gold Project, Ghana", prepared for Newcore Gold Ltd. (the "Company") and dated 7 August 2026, with an effective date of 23 June 2026 (the "Technical Report"), do hereby certify that:

1) I am a Chief Process Engineer with Lycopodium Minerals Canada Ltd., with a business address at at 5090 Explorer Drive, Suite 700, Mississauga, ON L4W 4T9.

2) I graduated from the University of Waterloo in Waterloo, Ontario, Canada in 2008 with a Bachelor of Applied Science, Honours Chemical Engineering.

3) I am a member in good standing with the Professional Engineers of Ontario (License #100136514). I have practiced my profession in the mining and metals industry continuously since graduation.

4) My background includes over 18 years of experience in the design and engineering of mineral processing plants, with a specialization in gold processing. I have led studies and projects from conceptual through to detailed design, for developments in both Canada and abroad.

5) I have read the definition of 'qualified person' set out in National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* ("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 Enchi Gold Project site.

7) I am responsible for Sections 1.8, 1.12, 1.15.2, 1.18.2, 13.0, 17.0, 21.4.1, 21.4.3, 21.4.5, 25.3, 25.5, 25.9, 25.11.4, 25.11.10, 26.4 and 27.0 of the Technical Report.

8) I am independent of the Company in accordance with section 1.5 of NI 43-101.

9) I was involved in the previous Technical Report on the Enchi Gold Project's preliminary economic assessment published in June 2024 and mineral resource update published in May 2026.

10) I have read NI 43-101, and the Technical Report, and confirmed that the sections of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101 and Form 43-101F1.

11) As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 7th day of August 2026

Signed and sealed.

Ryda Peung, P. Eng.