Tasiast Mine
Mauritania
National Instrument 43-101 Technical Report

Prepared for:
Kinross Gold Corporation

Prepared by:
Kinross Gold Corporation

Effective Date: **December 31, 2024**

Signature Date: **March 27, 2025**

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1. SUMMARY

1.1 Executive Summary

Kinross Gold Corporation (Kinross) has prepared a Technical Report for the wholly-owned Tasiast Mine (Tasiast or the Complex) located in the Islamic Republic of Mauritania (Mauritania), Africa. The purpose of this Technical Report is to support disclosure of Mineral Resources and Mineral Reserves for the Complex, inclusive of the West Branch, Piment, Prolongation, and Fennec deposits (Mineral Resources) and West Branch, Piment, and Fennec deposits (Mineral Reserves), with an effective date of December 31, 2024. The Technical Report conforms to National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) and has an effective date of December 31, 2024.

The Complex and the exploitation permit are owned by Tasiast Mauritanie Limited S.A. (TMLSA). SENISA (Société d'Extraction du Nord de l'Inchiri S.A., a sister company of TMLSA) holds two mining permits (for the Tmeimichat and Imkebden areas)that are contiguous with the Tasiast mining permit land (collectively, the Tasiast Lands). As part of the conversion process, Kinross has undertaken to transfer to the Government of Mauritania a 10% carried interest in SENISA. Kinross acquired TMLSA, including the Tasiast Lands, through its acquisition of Red Back Mining Inc (Red Back) in September 2010. There are exploration prospects in all three exploitation permit locations.

The current configuration of the Tasiast carbon-in-leach (CIL) plant has proven capable of processing approximately 24 thousand tonnes per day (kt/d). In September 2019, Kinross completed a feasibility study (Tasiast '24K' Project) to incrementally increase throughput capacity at Tasiast from approximately 15 kt/d to 24 kt/d – which included the intermediary step to 21 kt/d ('21K' Project) The project ('24K' Project) was completed in 2023 with throughput rates increased to 24 kt/d on a daily basis. Additionally, the '24K New CIL' Project was executed in 2024, replacing the previous CIL circuit with new tanks of similar design with the goal of extending their reliability and operation through the life of mine (LOM). The capital-efficient projects have increased production, lowered costs, and generated significant cash flow and attractive returns.

In late 2023, Kinross completed the development of its photovoltaic solar power plant at Tasiast with power generation capacity of 34 MW and a battery system of 18 MW. The Complex is part of Kinross' efforts to reduce its greenhouse gas (GHG) emissions and provides approximately 20% of the site's power. The Tasiast solar project is expected to generate positive returns and to reduce GHG emissions by approximately 530 kt over the LOM, which could save approximately 180 million litres of fuel over the same period. The project is expected to contribute to the Government of Mauritania's GHG reduction targets in the country.

To date, 15,862 drill holes (14,763 RC, 869 diamond core (DD), and 230 RC pre-collar with DD tail (RC-DD)) for an aggregate total of 1,713,081 m have been completed within the three exploitation licences that constitute the Tasiast Lands: Guelb El Ghaîcha, Imkebden, and Tmeimichat. Drilling activities were conducted by various drilling contractors and supervised by geological staff of the mine operator.

Commercial production of gold at Tasiast began in January 2008 under Red Back, and as of December 31, 2024, approximately five million ounces have been produced by Tasiast.

Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves dated May 10, 2014 (CIM (2014) Definitions) were followed for Mineral Resources and Mineral Reserves.

Mineral Reserves were estimated for three deposits at the Complex: West Branch, Piment, and Fennec, and are summarized in Table 1-1.

Table 1-1: Summary of Project Mineral Reserves – December 31, 2024

Classification	Tonnes (kt)	Grade (g/t Au)	Gold Ounces (koz)
Proven	14,819	1.34	640
Proven Stockpiles	42,542	0.99	1,361
Subtotal	**57,361**	**1.08**	**2,000**
Probable	45,471	1.85	2,705
TOTAL P&P	**102,831**	**1.42**	**4,705**

Notes:

1. CIM (2014) Definitions were followed for Mineral Reserves.
2. Mineral Reserves are limited to blocks within the reserve pit design and within mineable panels/polygons within which blocks meet an average cut-off grade of 0.6 g/t Au and consider ore loss, dilution, and mining selectivity.
3. Mineral Reserves are estimated using an average long-term gold price of US$1,600 per ounce.
4. Bulk density is assigned by oxidation state and lithology.
5. Numbers may not add due to rounding.

Mineral Resources were estimated for four areas at the Mine: West Branch, Piment, Prolongation, and Fennec, and are summarized in Table 1-2 with an effective date of December 31, 2024.

Table 1-2: Summary of Project Mineral Resources as at December 31, 2024

Classification	Tonnes (kt)	Grade (g/t Au)	Gold Ounces (koz)
Measured	21,296	0.70	478
Indicated	53,255	1.10	1,887
TOTAL M&I	**74,550**	**0.99**	**2,365**
Inferred	21,047	2.41	1,632

Notes:

1. CIM (2014) Definitions were followed for Mineral Resources.
2. Mineral Resources are estimated using a long-term gold price of US$2,000 per ounce.
3. Open pit Mineral Resources are constrained within an optimized pit shell reported to cut-off grades ranging from 0.39 g/t Au to 0.50 g/t Au.
4. Underground Mineral Resources are constrained within resource panels below the optimized pit shell which consider a minimum thickness of 2.5 m and a cut-off grade of 1.8 g/t Au. Crown pillar resource panels are factored to represent a 100% extraction limit at West Branch.
5. Bulk density is assigned by oxidation state and lithology.
6. Mineral Resources are exclusive of Mineral Reserves.
7. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
8. Numbers may not add due to rounding.

Conclusions

- Tasiast is viewed as a long-term strategic asset for Kinross, located in a district that is believed to have significant future potential.
- The Tasiast gold deposits fall into the broad category of orogenic gold deposits. Tasiast gold deposits are hosted in Archaean volcanic-sedimentary sequences that have been deformed and metamorphosed to lower amphibolite peak metamorphic grade. Mineralization is both structurally and lithologically controlled and is epigenetic in style.
- The Complex is currently operated as an open pit. Mineral Resources include both open pit and underground conceptual scenarios.
- There is a good understanding of the geology and the nature of gold mineralization at the Complex. The lithology model represents the support data well and it is developed using appropriate resolution.
- The Mineral Resource estimate is of sufficient quality to support public disclosure and is supported by best practice guidelines.

Recommendations

1. Foster the already-strong Continuous Improvement culture, looking for further opportunities to improve operating and cost performance in the mine, plant, and administration. In particular, focus on opportunities to add production to the 2025 to 2027 window while stripping is ongoing at West Branch 5.

2. Continue to explore the underground potential at Tasiast. High potential exists to continue to expand upon underground resources at Tasiast at West Branch and Piment but also to establish underground higher-grade resources at Prolongation.

3. Evaluate further push backs at West Branch and other open pits given the elevated gold price environment.

4. Evaluate opportunities to strategically stockpile lower grade material, which could be processed at the end of the mine life in a higher metal price environment. Furthermore, this lower grade material could be used to keep the mill full, if successful in converting the current underground resource. At YE 2024, Tasiast added 152 koz of low grade which will be stockpiled for the end of mine life. Future mine plans should look to add this to the back end of mine life, as well as evaluate other opportunities.

5. Continue to explore the addition of other satellite ore bodies to the mine plan such as C67 and C68. The recent addition of Fennec to the mine plan highlights the success that can come from detailed technical review of some of these higher-grade satellite opportunities.

6. Given the positive reconciliation of 106% on F3 ounces seen in 2024, continue to drill targeting higher grade plunge controlled mineralization in the West Branch 5 push back.

1.2 Technical Summary

Property Description

The Complex is located in northwestern Mauritania, approximately 300 km north of the capital Nouakchott and 250 km southeast of the major city of Nouadhibou. The Tasiast Lands are accessed from Nouakchott by using the paved Nouakchott to Nouadhibou highway for 370 km and then via 66 km of graded mine access road, which is maintained by TMLSA. There is an airstrip at the Complex that is used for light aircraft travelling to and from Nouakchott.

Mining operations commenced in 2007, with commercial production reached in January 2008. Infrastructure on site supports an open pit mining operation and associated processing facilities consisting of a CIL mill and a run of mine (ROM) dump leach.

Land Tenure

TMLSA holds a valid exploitation permit, 229C2 (Guelb El Ghaîcha), covering 312 km^2, granted in January 2004 and valid for a period of 30 years. The mining operations and infrastructure lie entirely within the lands subject to the exploitation permit. The exploitation permit is located centrally within two additional contiguous mining permits (totalling 1,597 km^2), each of which is in good standing. The Tasiast Lands fall within the Inchiri and Dakhlet Nouadhibou districts, with 229C2 within the Inchiri District only.

Surface rights are granted along with permit 229C2 and are paid annually as determined by decree under the Mauritanian Mining Code. Surface rights for the permits are in good standing.

The operation's water supply is located 64 km west of the Complex and consists of a bore field in a semi-saline aquifer. Water is pumped from the bore field to the Complex. The Tasiast permit, issued May 7, 2017 by the Ministry of Hydraulics and Sanitation, allows abstraction at a maximum rate of 30,000 m^3/d through to December 31, 2034.

History

Exploration programs have included geological and regolith mapping, satellite image interpretation, airborne and ground magnetic geophysical surveys, soil, rock chip, and grab geochemical sampling, trenching, RC and core drilling, engineering studies, metallurgical test work, and specialist geological studies such as ore and alteration petrography. Work was completed by the Office Mauritanien de Recherches Géologiques (OMRG), Normandy LaSource Development Ltd. (NLSD), Midas Gold plc. (Midas), Geomaque Explorations Inc. (Geomaque), Defiance Mining Corporation (Defiance), Rio Narcea Gold Mines Ltd. (Rio Narcea), Red Back, and Kinross.

Geological Setting and Mineralization

The Tasiast district is situated in the southwestern corner of the Reguibat Shield, which is a northeast-trending crustal block of the West African Craton. The Reguibat Shield contains the oldest rocks in Mauritania and consists of two major subdivisions separated by a crustal-scale shear zone representing a major accretionary boundary. The southwestern part (which hosts the Tasiast deposits) consists of Mesoarchean to Paleoproterozoic rocks that include high-grade granite-gneiss and greenstone belt assemblages. The northeastern part of the shield consists of younger Paleoproterozoic to Neoproterozoic successions, which hosts many orogenic gold occurrences in the West African Craton. This region is characterized by a series of volcano-sedimentary belts and associated batholithic-scale granitic intrusive suites of different ages cut by major shear zones.

The district scale geology is characterized by basement rocks, largely composed of orthogneiss, overlain by deformed north-striking metavolcanic and metasedimentary successions intruded by stocks and plutons of mafic to intermediate composition (granite-greenstone belts). All the rock units are cut by unfoliated and post-mineral mafic (gabbroic) dykes.

The Tasiast Mine consists of two deposit areas hosted within distinctly different rock types, both situated within the hanging wall of the Tasiast thrust. The Piment deposits are hosted within metasedimentary rocks including metaturbidites and banded iron formation. The West Branch geology succession comprises mafic to felsic volcanic sequences, iron-rich formations and clastic units that have undergone mid greenschist to lower amphibolite facies metamorphism and multiple deformation events.

The Tasiast gold deposits fall into the broad category of orogenic gold deposits. The regional geological setting and deposit features at Tasiast are similar to other well known Archaean lode gold deposits hosted along greenstone belts in granitoid- greenstone terranes.

Exploration

To date, 15,953 drill holes (14,849 RC, 874 diamond core (DD), and 230 RC pre-collar with DD tail (RC-DD)) for an aggregate total of 1,725,424 mhave been completed within the three mining licences of Guelb El Ghaîcha, Imkebden, and Tmeimichat. Drilling activities were conducted by various drilling contractors and supervised by geological staff of the mine operator.

The Complex area has significant exploration potential to delineate additional resources both around the Tasiast mine pits (near mine exploration) and within the wider district (generative exploration).

Exploration efforts to date have discovered additional prospects, gold deposits, and mineral resources along strike to the north and to the south of the main Tasiast Complex area (West Branch and Piment-Prolongation), and generally along the Aouéouat (Tasiast) belt. The deformed greenstone rocks to the west (Imkebden-Kneiffissat) of the Aouéouat belt are notable in that they host quartz-carbonate veins with anomalous gold values, however, to date no significant deposits or mineral resources have been defined.

Beyond 25 km from the Tasiast operation, within the northern extents of the Imkebden and Tmeimichat exploitation permits, there are several gold exploration prospects that are pending follow-up exploration and drilling; of note are C23, Kneiffissat, and

Grindstone. These prospects have significant surface geochemical footprints and are considered highly prospective.

Mineral Resources

The Mineral Resource estimate (MRE) is defined by five mineralized domains for West Branch and twelve mineralized domains for Piment and Prolongation. Each domain comprises an outer shear zone or mineralized envelope modelled as tabular veins, and an inner mineralized sub-domain modelled as an indicator interpolant. The mineralized zones were modelled using a grade threshold of 0.15 g/t Au as a guide, with logging data used in the absence of mineralization.

Samples were composited to two metres within each domain with capping done, per domain, thereafter. Capped values were estimated into sub-blocked models using a three-pass ordinary kriging (OK) approach for the mineralized zones and inverse distance squared (ID^2) for the waste zone. The model was validated using a combination of methods including visual comparison of block estimates and composites, swath plots and change-of-support checks, using the nearest neighbour (NN) de-clustered distribution, as well as visual and statistical validation against the short-term model estimated using grade-control data.

Classification of Mineral Resources considered the confidence in geological continuity, drill hole spacing, proximity to the current mining areas, and grade control drilling. Areas characterized by drill hole spacings of approximately 30 m define areas classified as Measured for West Branch, Piment, and Prolongation. Spacings of approximately 60 m to 70 m support areas classified as Indicated for West Branch and 70 m for Piment and Prolongation. For the Inferred classification, drill hole spacings of approximately 150 m were used for West Branch and 120 m for Piment and Prolongation. An additional small deposit, Fennec, was estimated in 2020 and forms a small portion of the Mineral Resources and Mineral Reserves.

Mineral Resources for the Tasiast Mine, including the West Branch, Piment, Prolongation, and Fennec deposits are presented in Table 1-3.

Table 1-3: Tasiast Mineral Resource estimate as at December 31, 2024

Deposit	Open Pit			Underground			Stockpiles			Combined		
Class	Tonnes (kt)	Grade (g/t Au)	Gold Ounces (koz)	Tonnes (kt)	Grade (g/t Au)	Gold Ounces (koz)	Tonnes (kt)	Grade (g/t Au)	Gold Ounces (koz)	Tonnes (kt)	Grade (g/t Au)	Gold Ounces (koz)
West Branch												
Measured	1,181	0.75	28				13,339	0.51	220	14,520	0.53	248
Indicated	43,584	1.09	1,528							43,584	1.09	1,528
Meas + Ind	44,765	1.08	1,556				13,339	0.51	220	58,104	0.95	1,776
Inferred	3,462	2.17	242	13,825	2.52	1,119				17,288	2.45	1,360
Piment												
Measured	4,668	0.77	116				33	0.60	1	4,702	0.77	117
Indicated	7,423	1.17	280							7,423	1.17	280
Meas + Ind	12,092	1.02	396				33	0.60	1	12,125	1.02	396
Inferred	432	1.30	18	2,769	2.57	228				3,202	2.39	247
Prolongation												
Measured	2,073	1.69	113							2,073	1.69	113
Indicated	1,136	1.33	48							1,136	1.33	48
Meas + Ind	3,210	1.56	161							3,210	1.56	161
Inferred	190	1.16	7							190	1.16	7
Fennec												
Measured												
Indicated	1,112	0.88	32							1,112	0.88	32
Meas + Ind	1,112	0.88	32							1,112	0.88	32
Inferred	367	1.50	18							367	1.50	18

Deposit	Open Pit			Underground			Stockpiles			Combined		
Class	Tonnes (kt)	Grade (g/t Au)	Gold Ounces (koz)	Tonnes (kt)	Grade (g/t Au)	Gold Ounces (koz)	Tonnes (kt)	Grade (g/t Au)	Gold Ounces (koz)	Tonnes (kt)	Grade (g/t Au)	Gold Ounces (koz)
Combined Total												
Measured	7,923	1.01	257				13,373	0.51	221	21,296	0.70	478
Indicated	53,255	1.10	1,887							53,255	1.10	1,887
Meas + Ind	61,178	1.09	2,144				13,373	0.51	221	74,550	0.99	2,365
Inferred	4,452	1.99	284	16,595	2.53	1,347				21,047	2.41	1,632

Notes:

1. CIM (2014) Definitions were followed for Mineral Resources.
2. Mineral Resources are estimated using a long-term gold price of US$2,000 per ounce.
3. Open pit Mineral Resources are constrained within an optimized pit shell reported to cut-off grades ranging from 0.39 g/t Au to 0.50 g/t Au.
4. Underground Mineral Resources are constrained within resource panels below the optimized pit shell which consider a minimum thickness of 2.5 m and a cut-off grade of 1.8 g/t Au. At West Branch, crown pillar resource panels are factored to represent a 100% extraction limit.
5. Bulk density is assigned by oxidation state and lithology.
6. Mineral Resources are exclusive of Mineral Reserves.
7. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
8. Numbers may not add due to rounding.

The QP is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the MRE.

Mineral Reserves

Mineral Reserves were estimated for three areas at the Mine: West Branch, Piment, and Fennec, and are summarized in Table 1-4.

Table 1-4: Summary of Project Mineral Reserves – December 31, 2024

Classification	Tonnes (kt)	Grade (g/t Au)	Gold Ounces (koz)
Proven	14,819	1.34	640
Proven Stockpiles	42,542	0.99	1,361
Subtotal	**57,361**	**1.08**	**2,000**
Probable	45,471	1.85	2,705
TOTAL P&P	**102,831**	**1.42**	**4,705**

Notes:

1. CIM (2014) Definitions were followed for Mineral Reserves.
2. Mineral Reserves are limited to blocks within the reserve pit design and within mineable panels/polygons within which blocks meet an average cut-off grade of 0.6 g/t Au and consider ore loss, dilution, and mining selectivity.
3. Mineral Reserves are estimated using an average long-term gold price of US$1,600 per ounce.
4. Bulk density is assigned by oxidation state and lithology.
5. Numbers may not add due to rounding.

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

Mining

Ore and waste rock is mined in 10 m benches by conventional open pit methods from the West Branch and Piment pits. Tasiast currently operates a haulage fleet of 45 Caterpillar 793D (220 t), five Hitachi EH-4000 (220 t), and two Komatsu HD785 (92 t). The haulage fleet is primarily loaded by six Caterpillar 6060 shovels and two Bucyrus RH340B excavators, with three Caterpillar 994 front-end loaders utilized for rehandle purposes and four Komatsu PC1250s for auxiliary loading of the smaller Komatsu trucks. Blasting techniques, including presplit and buffer hole blasting, are employed to protect the pit walls. The current mill operates at approximately 24 kt/d. Ore is fed directly from the mine and stockpiles to the primary crusher.

Mineral Processing and Metallurgical Testing

The current configuration of the Tasiast CIL plant has proven capable of processing approximately 24 kt/d. In September 2019, Kinross completed a feasibility study (Tasiast ‘24K’ Project) to incrementally increase throughput capacity at Tasiast from approximately 15 kt/d to 24 kt/d – which included the intermediary step to 21 kt/d (‘21K’ Project) The project (‘24K’ Project) was completed in 2023 with throughput rates increased to 24 kt/d on a daily basis. Additionally, the ‘24K New CIL’ Project was executed in 2024, replacing the previous CIL circuit with new tanks of similar design with the goal of extending their reliability and operation through the LOM. The capital-efficient projects have increased production, lowered costs, and generated significant cash flow and attractive returns.

Environment and Permitting

Current mine operations and the ‘24K’ Project are based on the formal approval of a number of Environmental Impact Assessment (EIA) studies completed before and since mine commissioning in 2007.

A review of waste rock geochemistry to determine the potential for acid rock drainage concluded that the rock has excess neutralizing capacity. Given the excess neutralizing capacity and the very low precipitation at Tasiast, acid rock drainage is not anticipated.

The Tasiast facilities operate under an environmental management system (EMS) that specifies activities to be planned and implemented by the mine’s environmental management team. The EMS incorporates the project design and management, mitigation strategies and performance monitoring commitments outlined in the environmental assessments, applicable legislation, and specific permit requirements.

An element of each EIA prepared for the Tasiast site is a preliminary reclamation and closure plan and associated cost estimate. The preliminary reclamation and closure plan outlines the measures that will be taken to reclaim and close the proposed activities assessed in each EIA. The preliminary reclamation and closure cost estimate forms the basis of the financial assurance. Tasiast, with the support of SRK Consulting (SRK), updated its financial assurance in 2024, including the 24 kt/d ‘24K’ Project . The estimated closure cost is approximately US$64 million. This financial assurance will be submitted to the Mauritanian regulators for their review and approval in February 2025. Once the government of Mauritania validates the assurance, Tasiast will issue a new letter of credit that reflects the remaining amount, taking into account the existing financial assurance of US$6.2 million. At least two years before entering closure, a detailed reclamation and closure plan must be submitted to the appropriate ministries for approval.

Current environmental liabilities are those that would be expected from a mining operation, and include the mine, crushing and CIL processing plant, dump leach facilities, power plant, tailings and waste rock facilities, power grids, roads, accommodation camp, ancillary facilities, and drill pads established to support mining and exploration activities.

Capital and Operating Costs

Capital costs for the Tasiast mine are split into the following categories:

- Non-sustaining capital
    - Capitalized Waste Stripping at West Branch pit
    - Mobile fleet replacement
- Sustaining Capital
    - Capitalized Waste Stripping at Piment pit
    - Mining Sustaining Capital (incl. fleet replacement)
    - Mill Sustaining Capital
    - Tailings Sustaining Capital
    - Other

Total capital for the life-of-mine is estimated at:

- Non-sustaining:
- Sustaining:

Operating cost estimates are shown in Table 1-5. The operating costs for each area include allocations for power plant operating costs.

Table 1-5: Operating cost estimates (January 1, 2025)

Operating Cost	Unit	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	LOM 2025-2035
Mining	US$/t mined	3.9	3.9	3.9	3.6	3.5	4.2	0.0	0.0	0.0	0.0	0.0	3.8
Processing (Mill)	US$/t processed	16.2	15.7	15.9	15.1	15.0	14.7	13.6	13.2	13.4	12.8	12.7	14.4
Site Admin	million US$/a	112	106	101	98	96	75	41	41	34	30	17	754
Royalties	US$/oz sold	186	187	188	187	187	187	187	187	187	187	187	187
Other	US$/oz sold	2.2	2.3	2.3	1.7	1.7	2.2	4.4	4.6	5.0	5.1	2.2	2.6

2. INTRODUCTION

Kinross Gold Corporation (Kinross) has prepared a Technical Report for the wholly-owned Tasiast Mine (Tasiast or the Complex) located in the Islamic Republic of Mauritania (Mauritania), Africa. The purpose of this Technical Report is to support disclosure of Mineral Resources and Mineral Reserves for the Complex, inclusive of the West Branch, Piment, Prolongation, and Fennec deposits (Mineral Resources) and West Branch, Piment, and Fennec deposits (Mineral Reserves), with an effective date of December 31, 2024. The Technical Report conforms to National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) and has an effective date of December 31, 2024.

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All measurement units used in this Technical Report are metric, and currency is expressed in US dollars unless stated otherwise. Mauritania uses the Ouguiya (MRU) as its currency.

Information used to support this Technical Report has been derived from the reports and documents listed in the References section of this Technical Report.

The use of the terms "we", "us", "our", or "Kinross" in this Technical Report refer to Kinross Gold Corporation.

2.1 Qualified Persons

The Qualified Persons (QP) for this Technical Report are summarized in Table 2-1:

Table 2-1: Qualified Persons and their responsibilities

QP Name, Designation, Title	Site Visit	Responsible for Sections
Nicos Pfeiffer, P.Geo., VP Geology & Technical Evaluations	25 – 28 Nov 2024	3-6, 20, 23, 24, and relevant portions of 1, 2, 25, 26, 27
Agung Prawasono, P.Eng, Sr. Director, Mine Planning	23 Feb – 2 Mar 2025	15, 16, and relevant portions of 1, 2, 25, 26, 27
Yves Breau, P. Eng, VP Metallurgy & Engineering	4 – 10 Aug 2024	13, 17, 18, 19, relevant portions of 1, 2, 25, 26, 27
Graham Long, P.Geo., VP Exploration	17 – 26 Nov 2024	7, 8, 9 ,10, relevant portions of 1, 2, 25, 26, 27
Jacob Brown, SME (RM), Dir. Resource and Mine Geology	--	11, 12, 14, relevant portions of 1, 2, 25, 26, 27
Kevin van Warmerdam, P.Eng, Sr. Dir. Engineering & Energy	17 – 21 Oct 2024	21, 22, relevant portions of 1, 2, 25, 26, 27

Mr. Pfeiffer visited the site most recently in November 2024. During the site visit, he inspected core and surface outcrops, drill platforms and sample cutting and logging areas; discussed geology and mineralization with project staff; reviewed geological interpretations with staff; and inspected the major infrastructure and current mining operations. There have been no material changes in site conditions since this most recent site visit. All sections in this Technical Report have been prepared under the supervision of Mr. Pfeiffer.

***Mineral Resources*:** The MREs included in this Technical Report were prepared under the supervision Jacob Brown, SME, Kinross Director Resource and Mine Geology, Kinross Technical Services. Mr. Brown is a Professional Member of the Society for Mining, Metallurgy & Exploration.

Mineral Reserves / Mining: The Mineral Reserve estimate and economic analysis included in this Technical Report was prepared under the supervision of Agung Prawasono, Senior Director Mine Planning, Kinross Technical Services. Mr. Prawasono is a Registered Professional Engineer in the Province of Ontario. Mr. Prawasono visited the site most recently in November 2024.

Mineral Processing: Mineral processing aspects of this report were prepared under the supervision of Yves Breau, Vice President, Metallurgy and Engineering, Kinross

Technical Services. Mr. Breau is a Registered Professional Engineer in the Province of Ontario. Mr. Breau visited the site most recently in August 2024.

2.2 Sources of Information

Information used to support this Technical Report was derived from previous technical reports on the property, and from the reports and documents listed in Section 27 References of this Technical Report.

2.3 Effective Date

The effective date of this Technical Report is December 31, 2024.

There were no material changes to the information on the Tasiast Mine between the effective date and the signature date of the Technical Report.

2.4 List of Abbreviations

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

°C	degree Celsius
a	annum
Au	gold
bbl	barrels
BV/h	bed volumes per hour
C$	Canadian dollars
cm	centimetre
cm^2	square centimetre
d	day
dia.	diameter
ft	foot
ft/s	foot per second
ft^2	square foot
ft^3	cubic foot
g	gram
G	giga (billion)
g/L	gram per liter
g/t	gram per tonne
Ga	Giga-annum (billions of years before present)
Ha	hectare
in.	inch
in^2	square inch
J	joule
k	thousand (kilo)
kg	kilogram
km	kilometre
km^2	square kilometre
km/h	kilometres per hour
koz	thousand ounces
kt	thousand tonnes
kt/d	thousand tonnes per day
kPa	kilopascal
kWh/t	kilowatt-hour per tonne
kW	kilowatt
kWh	kilowatt-hour
L	litre
L/s	liters per second
m	metre
M	mega (million)
m^2	square metre
m^3	cubic metre
Ma	Mega-annum (millions of years before present)
min	minute
masl	metres above sea level
mm	millimetre
Mt/a	million tonne per year
MW	megawatt
MWe	megawatt-electrical
m^3/h	cubic metres per hour
oz	Troy ounce (31.1035 g)
ppm	part per million
s	second
t	metric tonne
t/a	metric tonne per year
t/d	metric tonne per day
US$	United States dollar
V	volt
yr	year

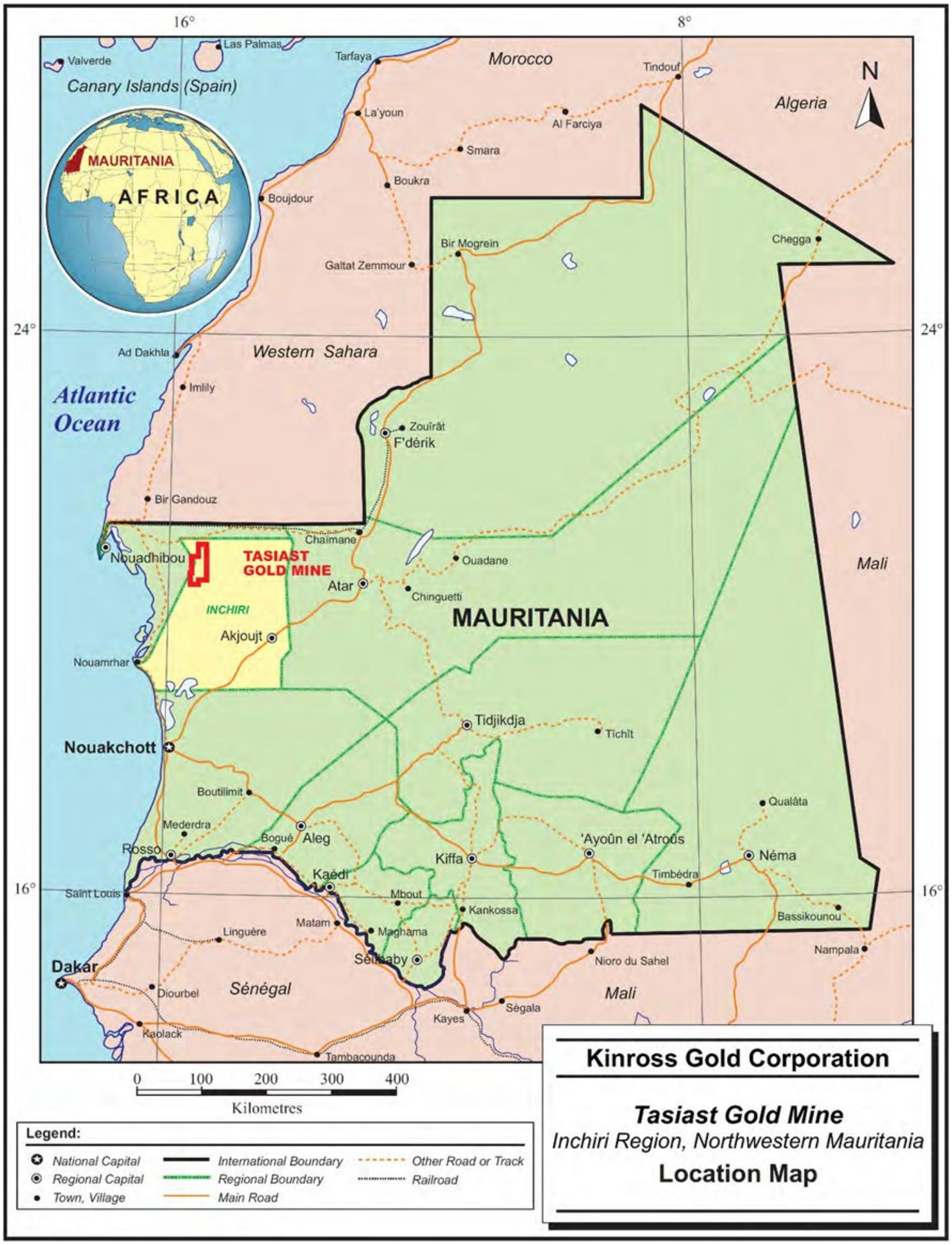
3. RELIANCE ON OTHER EXPERTS

In the preparation of the Technical Report, the QPs relied on information provided by internal Kinross legal counsel for the discussion of claim numbers, title types, anniversary dates, and confirmation that the claims are in good standing as of the date of this Technical Report, as summarized in Section 4.2.

4. PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Tasiast Lands are located in northwestern Mauritania, approximately 300 km north of the capital Nouakchott and 250 km southeast of the major city of Nouadhibou. The Tasiast Lands fall within the Inchiri and Dakhlet Nouadhibou districts. The Complex is located at 446600E and 2275600N (UTM, WGS84, Zone 28N) and is presented in Figure 4-1 and Figure 4-2.



Source: Kinross 2019.

Figure 4-1: Location map

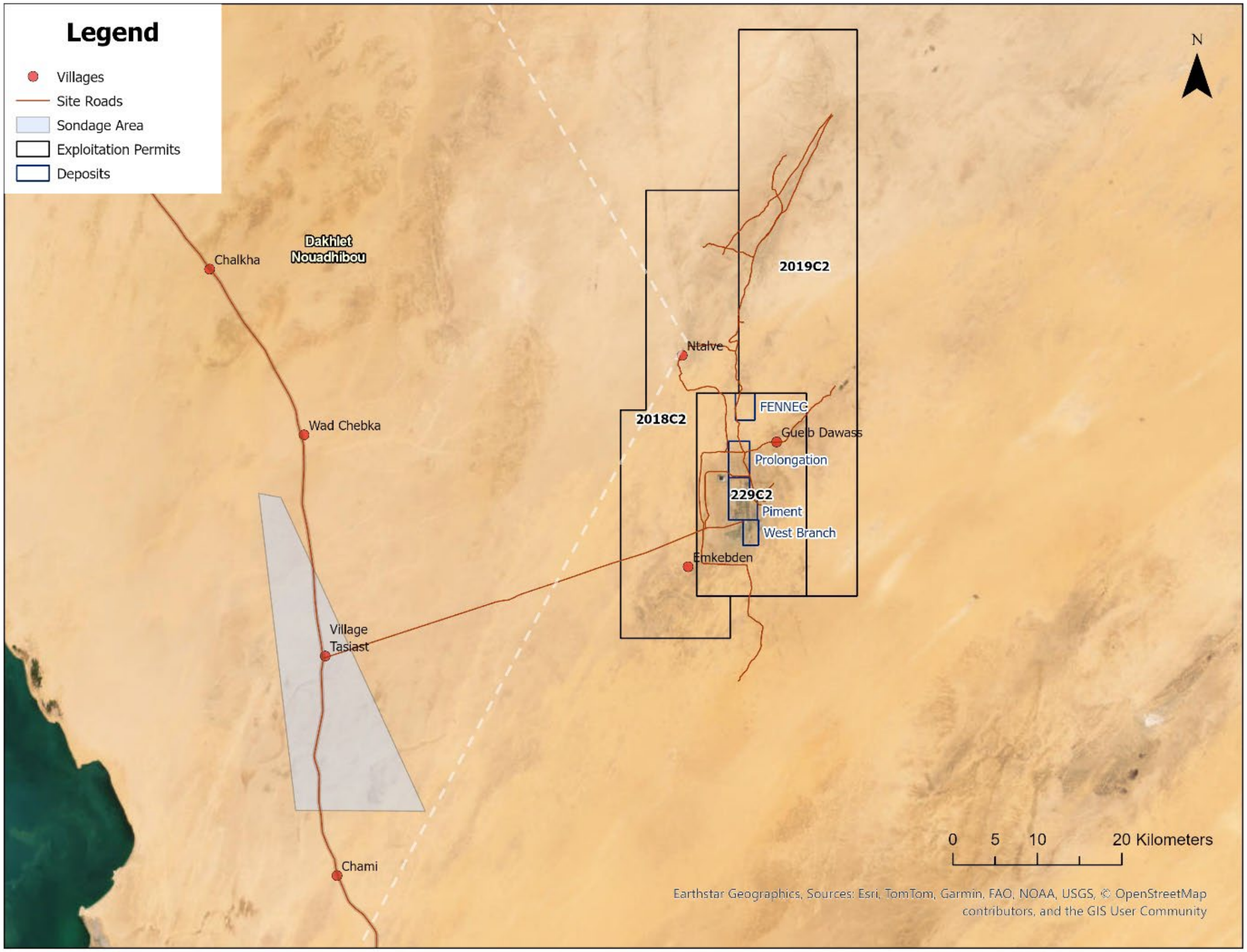


Figure 4-2: Property Map

4.2 Mineral Tenure

Mauritania's mining sector is regulated by the Mining Code 2008 (Law No. 2008 – 011, dated). The regulatory framework for the mining sector also includes Law No. 2009-026 (amendment to Mining Code); Law No. 2012-014 (amendment to Mining Code); and Law No. 2014-008 (amendment to Mining Code). The law is available on the official website of the government regulatory entity (https://mmi.gov.mr/fr/legislations-mines/).

The Tasiast Mine is owned and operated by TMLSA, a wholly owned subsidiary of Kinross, within the 312 km^2 exploitation permit (permis d'exploitation, or PE) of Ghelb El Ghaîcha (No. 229C2 or "PE No. 229"). The Tasiast Mine and the exploitation permit are owned by TMLSA.

Exploitation permit No. 229C2 is located centrally within a surrounding permit block of two contiguous exploitation permits, totalling 1,597 km^2, as listed in Table 4-1 and shown in Figure 4-2. All these permits are in good standing.

The adjacent two permits are held by the sister company of TMLSA, namely SENISA. These two mining permits cover the Tmeimichat and Imkebden areas. In July 2022, as part of the December 2014 conversion process of the two exploration permits, SENISA transferred to the Government of Mauritania a 10% carried interest in the shareholding of the company. Other than the 10% carried interest in SENISA held by the Government of Mauritania, all permit-holding affiliates, including TMLSA, are wholly-owned indirect subsidiaries of Kinross. Kinross acquired TMLSA, including the Tasiast operation and exploration permits and lands, through its acquisition Red Back in September 2010.

Tenure coordinates are shown in Table 4-2. A permit boundary is defined by a list of the coordinates of its corners or pillar points. The boundaries are not physically marked on the ground and have not been surveyed, however, extensive surveying has been conducted within both the exploitation permit No. 229C2 and adjoining permits. All the known gold deposits are well inside the boundaries, and the size and shape of the exploitation permit are adequate for the intended exploration, mining, and processing activities.

Table 4-1: Mineral tenure summary – Tasiast Property

Name	District	Type	No.	Area (km^2)	Granted	Expiry
Tasiast (Guelb El Ghaîcha)	Inchiri	Exploitation Permit	229C2	312	January 19, 2004	January 9, 2034
Imkebden	Dakhlet Nouadhibou and Inchiri	Exploitation Permit	2018C2	539	December 1, 2014	December 1, 2044
Tmeimichat	Inchiri	Exploitation Permit	2019C2	746	December 1, 2014	December 1, 2044

Table 4-2: Permit boundary coordinates

Permit Name	Permit Number	Point	Coordinates	
			UTM (E)	UTM (N)
Guelb El Ghaîcha	229C2	A	441000	2287000
		B	454000	2287000
		C	454000	2263000
		D	441000	2263000
Imkebden	2018C2	A	435000	2311000
		B	446000	2311000
		C	446000	2287000
		D	441000	2287000
		E	441000	2263000
		F	445000	2263000
		G	445000	2258000
		H	432000	2258000
		I	432000	2285000
		J	435000	2285000
Tmeimichat	2019C2	A	446000	2330000
		B	460000	2330000
		C	460000	2263000
		D	454000	2263000
		E	454000	2287000
		F	446000	2287000

Surface rights are granted along with the three permits, and applicable fees are paid annually, as determined by decree under the Mining Code. Surface rights for the permit are in good standing, and there are no competing mining rights in the area.

Exploration permits (permis de recherche minière, or PRM) grant exclusive exploration rights over a specific block (maximum 500 km^2, in accordance with Mining Code 2008, amendment enactment date April 29, 2014) and are granted for a three-year period, renewable twice for up to three years at each renewal. Exploitation permits are granted for 30 years and are renewable for periods of 10 years each. A condition of each permit is that the holder is required to hire Mauritanian tradespersons to provide services and to contract with national suppliers and businesses in preference to foreign service providers, where the national suppliers and businesses can offer at least the same terms, quality, and pricing. Table 4-3 summarizes the durations of exploration and mining permits in Mauritania. Operating permits are discussed in Section 20.2.

Current environmental liabilities are those that would be expected from a mining operation, and include the mine, crushing and CIL processing plant, dump leach facilities, power plant, tailings and waste rock facilities, power grids, roads, accommodation camp, ancillary facilities, and drill pads established to support mining and exploration activities.

Table 4-3: Permit durations under the 1999 Code applicable to Tasiast

Licence Type	Duration	Renewal Period	Number of Permissible Renewals	Rules and Mechanisms
Exploration Permit	3 years	3 years	Two After the two renewal periods lapse, the permit expires unless it is converted (in whole or in part) into an exploitation permit.	• Surface Area:1,000 km^2 blocks • Confers right to explore for resources to any depth within permit area • Number is limited to 20 exploration permits per holder; a holder must have the technical and financial capability to conduct the work • Permits taken under a joint venture are not taken into consideration for the calculation of the above limit, if the holder is not the controlling partner or the operator • Transferable under conditions established by the Decree on Mining Titles
Mining Permit (Exploitation Permit)	30 years	10 years	Several	• Necessary for operating a mine • Within an area initially covered by an exploration permit, for the same commodities, and on the basis of a feasibility study • Granted only to a legal entity incorporated under Mauritanian law and created by the holder of the exploration permit • Transferable under conditions established by the Decree on Mining Titles • Personnel health and safety reports to be lodged with the Ministry every six months, and environmental and activity reports every year • Land needs to be rehabilitated after mining

4.3 Fees, Royalties, Duties, and Taxes

Mining activities in Mauritania are mainly governed by the Mining Code and its regulations, and by the Model Mining Convention Law, which provides the legal and tax framework for all mineral exploration and extraction activities. TMLSA is governed by the Mining Code.

The Mining Code establishes conditions and rules governing all phases of mining activity. The Model Mining Convention Law provides that each exploration permit is subject to a mining convention with the State of Mauritania, which outlines the framework of customs, economic, financial, legal, and tax terms and conditions under which the permit holder proceeds with its exploration or mining activities inside the perimeter of its permit. A mining convention is attached to a given permit. Table 4-4 summarizes provisions of the TMLSA Mining Convention relating to fees, royalties, duties, and taxes.

The Mining Code is also complemented by the Decree on Mining Titles, which provides more details on the process governing the grant, renewal, expansion or reduction, division or merger, transfer, termination, suspension, and cancellation of a permit for exploration or exploitation. It also governs the conversion of an exploration permit into an exploitation permit.

The conditions embodied in the Model Mining Convention (Law No. 2002/02, subsequently replaced by Law 2012-012) are designed to stimulate and encourage investment in both exploration and mining. The mining industry is seen as one of the main growth industries for the improvement of the country’s economy.

In addition to the royalty payable to the government, Franco-Nevada Corporation (Franco-Nevada) holds a 2% net smelter return (NSR) royalty on gold production at the Tasiast mine in excess of 600,000 cumulative ounces produced. Production at the Tasiast mine reached 600,000 ounces in July 2011, and the first royalty payment to Franco-Nevada was made in October 2011. This 2% royalty will also apply to any eventual production from SENISA from its first ounce produced.

On July 15, 2021, TMLSA signed an agreement with the Government of Mauritania, which, among other things, solidified the additional sliding scale royalty, which the Company has been paying since June 15, 2020, that ranges from 1% (at a gold price less than $1,000 per ounce) up to 3.5% (at a gold price of $1,800 per ounce or more). This amount is in addition to the 3% royalty payable under the Mining Convention.

Table 4-4: Applicable fees, royalties, duties and taxes

Applicable Obligation	Exploration Permit	Mining Permit
Compensatory fees (for the issuance, extension or reduction, renewal, early termination and transfer of a permit)	MRU 200,000	MRU 250,000
Annual surface fee	Initial period: MRU 200/km^2-600/km^2 First renewal period: MRU 1,000/km^2-1,400/km^2 Second renewal period: MRU 2,000/km^2-2,400/km^2	MRU 5,000/km^2
Royalty (payable to the Government)		3% gross + additional sliding scale royalty (based upon the sale price per ounce) Less than 1,000 USD = 1% 1,000 to 1,199 USD = 1.5% 1,200 to 1,399 USD = 2% 1,400 to 1,599 USD = 2.5% 1,600 to 1,799 USD = 3% 1,800+ USD = 3.5%
Customs duties and other taxes		Complete exemption on all imported equipment and supplies, for five years after the start of production (ended August 2012). Customs duties of 5% thereafter on equipment and supplies imported. Exemption on imported fuel is applicable for the duration of the Mining Convention.
Tax on business profits		TMLSA was exempt from this tax for the first three years after its first production. After three years, the rate was fixed and stabilized at 25%. Articles 8 to 17 inclusively of the Mining Convention establish the relevant deductions to determine taxable profits.
Fixed minimum tax		Exemption until the end of the third year following the year the exploitation permit was granted. Since the end of this period, the standard rate of 2.5% applies at a reduced rate (1.25%) for TMLSA.
Tax on salaries, wages and annuities of		The standard rate of 40% reduced to half (20%) for TMLSA.

Applicable Obligation	Exploration Permit	Mining Permit
expatriates employed by TMLSA		
Income tax from capital		Customary applicable rate is 10%. Dividends reinvested on Mauritanian territory are exempt from this tax.
General income tax		Exemption for the duration of the Mining Convention
Value-added tax (VAT)		Customary applicable rate was increased from 14% to 16% in January 2015, except for exports by TMLSA, which is 0% (contingent on export of at least 80% of production).
Tax on sales - consumption tax		Exemption for the duration of the Mining Convention
Housing tax		Applies according to the CGI (General Tax Code), as from the first exploitation permit
Land income tax (on properties built)		Rules of application are those of the CGI. The rate, as voted by the town council, is 3% to 10% of the taxable value, which is the rental value of the property minus 20%.
Trading tax		Applicable from the date the first exploitation permit was granted according to the CGI. This tax is a fixed duty based on turnover
Registration and stamp duties		Exemption for the duration of the Mining Convention
Tax on motor vehicles		Applies according to the CGI as from the first exploitation permit. The rate of the tax is based on the use of the vehicle and its tax power
Apprenticeship tax		Exemption for the duration of the Mining Convention

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

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

5.1 Accessibility

The Tasiast Lands are accessed from Nouakchott by using the paved Nouakchott to Nouadhibou highway for 370 km and then via 66 km of graded mine access road which is maintained by TMLSA. An airstrip at the Complex is used for light aircraft, primarily travelling to and from Nouakchott.

The principal ports of entry for goods and consumables are either Nouakchott or Nouadhibou. Materials are transported by road to the Complex site.

Access to the major urban centres of Mauritania is also possible via air. Nouakchott is accessible via international flights operated by numerous West and North African carriers; Air France also provides a direct connection to Paris.

5.2 Climate

Mauritania has an arid desert climate, with an average annual high temperature of above 45°C between May and August. Low temperatures may drop below 10°C in December and January. From January to March, sandstorms frequently occur in the country causing sand build up and dune formation. Sandstorms vary in intensity, and visibility can be reduced to several metres.

Average annual precipitation at Tasiast is approximately 90 mm and usually occurs from July to September. The average recorded monthly evaporation is approximately 320 mm/month (3,840 mm/a).

Mauritania is located along the northwestern coast of Africa and is bordered by the Atlantic Ocean to the west. The country's land mass covers the western portion of the Sahara Desert. Mauritania's land mass consists mainly of flat and barren desert landscape surfaces that are crosscut by three large northeast-southwest trending longitudinal dune fields. In the desert regions, vegetation is sparse, consisting of various species of trees (e.g., acacia) and grasses.

5.3 Local Resources and Infrastructure

The Complex is located in a remote area where there is no electric power grid. On-site power generation is discussed in Section 18.

The source of mine water supply is located 64 km west of the Complex and consists of a semi-saline underground aquifer, with wells for water production. Water is pumped from the bore field to the Mine (see Section 18).

In 2024, the Tasiast Complex had approximately 1,726 employees, of whom approximately 1,680 are Mauritanian nationals. Staff accommodation is provided at the Complex (see Section 18).

The terrain surrounding the Complex is flat and is adequate for construction and operation of the camp, mine, plant, tailings, and waste rock disposal facilities.

5.4 Physiography and Environment

The topography of the Tasiast Lands consists mainly of flat, barren plains which are primarily covered by regolith and locally by sand dunes or eroded paleo-lateritic profiles. Elevation ranges from approximately 130 masl to 150 masl.

The drainage pattern around the Tasiast Lands consists of several intermittent dendritic first- and second-order streams that generally flow in a southwesterly direction. There are no permanent watercourses in the area. However, there are numerous, intermittent watercourses, known as "*wadis*", which flow for only a few days per year. The largest *wadi* is the Khatt Ataoui *wadi*, which is located approximately six kilometres from the mine site.

The Complex is located in the arid Saharan zone, where plant life is very scarce, consisting mainly of the low shrubs *Zygophyllum album*, the small tree *Maerua* crassifolia (atil) and the grass *Aristida pungens (sbot)*. Acacias are also present along many of the *wadis*.

Hares, hamsters, and gerbils are the most common mammals at the mine site, and jackals, fennec fox, and polecat can also be found in the area. There are no protected species in the Tasiast area. The eastern boundary of the Banc D'Arguin National Park is located approximately two kilometres west of the bore field area and approximately 60 km from the mine site.

6. HISTORY

6.1 Tenure History

In 1996, the Office Mauritanien de Recherches Géologiques (OMRG) completed a regional reconnaissance exploration program within and around the lands hosting the Tasiast deposit and made the results available to third parties. As a result, Normandy LaSource Development Ltd. (NLSD), a subsidiary of Normandy Mining Ltd. of Australia, acquired the exploration rights to the Tasiast deposit.

In 2001, NLSD was acquired by Newmont Mining Corporation, creating Newmont LaSource. Midas Gold PLC (Midas) was incorporated in England and Wales in 2002 for the purpose of acquiring Newmont LaSource's assets in Mauritania, including exploration permits over lands hosting the Tasiast deposit, as well as other permit areas. Midas completed its acquisition of the Tasiast deposit from Newmont LaSource on April 1, 2003, and, in April 2003, Geomaque Explorations Inc. (Geomaque) announced the acquisition of Midas. The merger of Geomaque and Midas ultimately created a new entity - Defiance Mining Corporation (Defiance). In June 2004, Rio Narcea Gold Mines, Ltd. (Rio Narcea) acquired Defiance and took ownership of the Tasiast deposit.

Red Back acquired the Tasiast deposit from Lundin Mining Corporation (Lundin) in August 2007 following Lundin's acquisition of Rio Narcea. In September 2010, Kinross completed the acquisition of Red Back. Kinross, through TMLSA, holds 100% of the Project.

6.2 Project History

From 1962 to 1993, the Tasiast region was the subject of three regional exploration programs for pegmatites, iron ore, and nickel sulphides which were carried out by the Bureau de Recherches Géologiques et Minières (BRGM) and Societe Nationale Industrielle et Minière (SNIM).

Three exploration programs were carried out in the Tasiast region between 1993 and 1996 as a European Development Fund project. Work completed included regional-scale reconnaissance geological mapping and geochemical sampling. Traverse lines for the mapping and geochemical sampling programs were oriented east-west with samples collected at 500 m centres. This work identified the Tasiast area as being anomalous in gold and was followed by more detailed soil sampling on 250 m spaced centres, and trenching.

NLSD, in the period from 1996 to 2001, completed geological and regolith mapping, interpretation of satellite imagery, airborne and ground magnetic geophysical surveys, specialist petrographical, mineralogical, and geological studies, metallurgical test work, and auger, reverse circulation (RC), and core drilling.

Midas undertook a full review of all existing information in 2003, and prepared MREs for the West Branch and Piment areas. From 2003 to 2004, Defiance completed mineralogical and metallurgical test work, hydrogeological studies, a preliminary pit slope design study, RC and core drilling, a MRE, and a feasibility study.

Rio Narcea completed additional RC and core drilling from 2005 to 2006. Red Back also undertook RC and core drilling, re-estimated mineral resources, and updated engineering studies.

Between August 2007 and September 2010, Red Back completed several large exploration campaigns in the Piment and West Branch areas, as well as at several district targets. Early drilling campaigns were directed at testing the lateral and vertical extents of the mineralization at Piment and drilling oxide resources at West Branch. In October 2009, Red Back discovered the Greenschist Zone at West Branch and commenced drilling the deposit.

From September 2010 to date, TMLSA has ramped up exploration with the majority of activities directed towards delineating the extents of the Greenschist Zone.

Mining at Tasiast commenced in April 2007 and the mine was officially opened by the President of Mauritania, His Excellency Sidi Mohamed Ould Cheikh Abdallahi, on July 18, 2007. A summary of gold production at Tasiast is included in Table 6-1 There has been no historical gold production from other deposits in the Tasiast area.

Table 6-1: Production summary

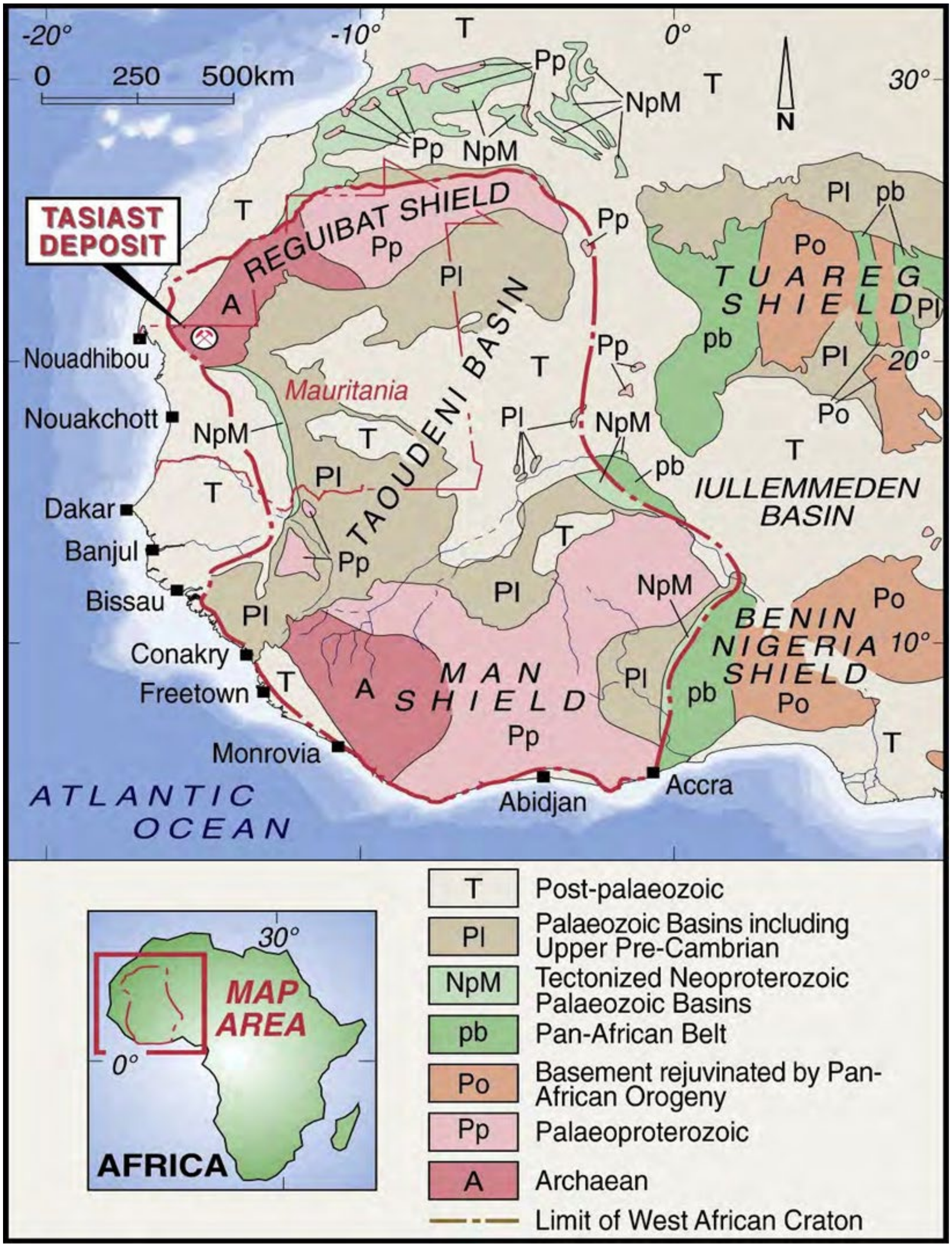
Year	Carbon-In-Leach				Dump Leach	Total
	Tonnes Milled	Grade	Gold Produced	Recovery	Gold Produced	Gold Produced
	(Mt)	(g/t)	(koz)	(%)	(koz)	(koz)
2007	0.22	4.36	21	68.6%	0	21
2008	1.49	3.07	140	95.4%	0	140
2009	1.68	2.88	142	91.4%	16	159
2010	2.14	2.52	150	86.8%	36	186
2011	2.60	2.04	153	89.4%	48	201
2012	2.55	1.54	114	90.2%	71	185
2013	2.50	1.96	144	91.3%	104	248
2014	2.56	2.16	161	93.1%	94	255
2015	2.54	2.17	160	90.1%	60	220
2016	2.46	1.81	131	90.6%	45	176
2017	3.04	2.36	214	92.3%	30	244
2018	3.73	2.02	225	92.6%	30	255
2019	5.23	2.33	379	96.6%	0	379
2020	5.35	2.49	404	94.4%	0	404
2021	3.73	1.69	193	94.9%	0	193
2022	6.57	2.75	526	90.5%	0	526
2023	6.72	3.19	636	92.3%	0	636
2024	8.64	2.44	623	92.1%	0	623
Total/ Average	**63.72**	**2.39**	**4,513**	**92.0%**	**554**	**5,066**

7. GEOLOGICAL SETTING

7.1 Regional Geology

The Tasiast district is situated in the southwestern corner of the Reguibat Shield, which is a northeast-trending crustal block of the West African Craton (Figure 7-1). The Reguibat Shield contains the oldest rocks in Mauritania and consists of two major subdivisions separated by a crustal-scale shear zone representing a major accretionary boundary (Lahondère et al. 2003; Pitfield et al. 2004; Schofield et al. 2006). The southwestern part (which hosts the Tasiast deposits) consists of Mesoarchean to Paleoproterozoic rocks that include high-grade granite-gneiss and greenstone belt assemblages. The northeastern part of the shield consists of younger Paleoproterozoic to Neoproterozoic successions, which host many orogenic gold occurrences in the West African Craton. This region is characterized by a series of volcano-sedimentary belts and associated batholithic-scale granitic intrusive suites of different ages cut by major shear zones.

The Reguibat Shield is bound on all sides by Pan African orogenic belts and covered in the south by the extensive intra-cratonic sediments of the Taoudeni Basin. The Taoudeni Basin represents one of the largest Mesoproterozoic to Paleozoic cratonic sedimentary basins in Africa. It consists of many thousands of metres of continental sandstones, platform carbonate rocks, and lesser shales.



Modified from Fabre (2005)

Figure 7-1: Geology of the West African Craton

7.2 District Geology

The district scale geology is characterized by basement rocks, largely composed of orthogneiss, overlain by deformed north-striking metavolcanic and metasedimentary successions intruded by stocks and plutons of mafic to intermediate composition (granite-greenstone belts). All the rock units are cut by unfoliated and post-mineral mafic (gabbroic) dykes. Two significant Archaean greenstone belts are exposed within the Tasiast District (Figure 7-2):

- Aouéouat (+75 km long x 8 km wide); and
- Imkebden-Kneiffissat (+60 km long x 9 km wide).

The greenstone belts are wholly enclosed by granitic to gabbroic intrusive rocks and gneissic domes that comprise the bulk of the rocks within the district and the Reguibat Shield overall. The greenstone belts comprise ultramafic to felsic volcanic and volcano-sedimentary packages with variably preserved ferruginous quartzite, locally termed banded magnetite. Rock units within the belts have undergone mid greenschist to lower amphibolite facies metamorphism and multiple deformation events. Swarms of non-foliated mafic (basaltic) dykes striking north-northeast/south-southwest and roughly east-west crosscut all other rocks in the district, including undeformed pegmatite units.

A Precambrian lithostratigraphy was established by Kinross for the Aouéouat belt (Figure 7-3) including several U-Pb dates for rocks of the Aouéouat and Tasiast assemblages and for granodiorite intrusions. The mafic to felsic volcanic and intrusive units that host the West Branch deposit belong to the Aouéouat assemblage that crystallized between 2,990 Ma and 3,000 Ma. Metasedimentary rocks of the Tasiast assemblage that overlay the mineralized West Branch units contain detrital zircons of similar ages and older populations derived from approximate 3,200 Ma orthogneiss basement. Granodiorites that crosscut the metavolcanic rocks are dated 2,960 Ma to 2,970 Ma. An age of 2,839 ±36 Ma obtained from the hydrothermal overgrowth on zircons from a quartz vein at the Fennec deposit is interpreted to represent the age of mineralization at Tasiast (Heron et al., 2016).

The principal north-south structural fabric in the Tasiast granite-greenstone belts is evident in satellite images (Worldview-2), airborne geophysics, and regional geological maps. Steeply dipping foliations and isoclinals folds with north-south to northwest-southeast axial surface traces are common across the Aouéouat belt. Those structures formed through east-west transpressive shortening that occurred as a result of basin inversion. Strain was partitioned between tightly folded domains and north-south striking shear zones. Several families of late-stage crosscutting faults with northeast and southwest strikes are occupied by fresh mafic dyke material.

All the significant mineralized bodies defined to date dip moderately to steeply (45° to 70°) to the east and have a south–southeasterly plunge. Gold deposits on the Tasiast trend are associated with second order shear zones and splays cutting the hanging wall block of an inferred thrust. The volcano-sedimentary stratigraphy has been tightly to isoclinally folded and is cut longitudinally by sub-parallel shears that are sub-parallel to the predominant foliation.

The main Tasiast gold trend includes the West Branch and Piment-Prolongation deposits. Gold mineralization is spatially associated with the west vergent Tasiast shear system that places mafic to felsic volcanic and intrusive rocks of the Aouéouat assemblage, including the host rocks of the West Branch deposit, on top of the younger metasedimentary rocks of the Tasiast assemblage. The Tasiast trend passes north-south through the Guelb El Ghaîcha mining permit and extends to the north and south onto adjacent licences.

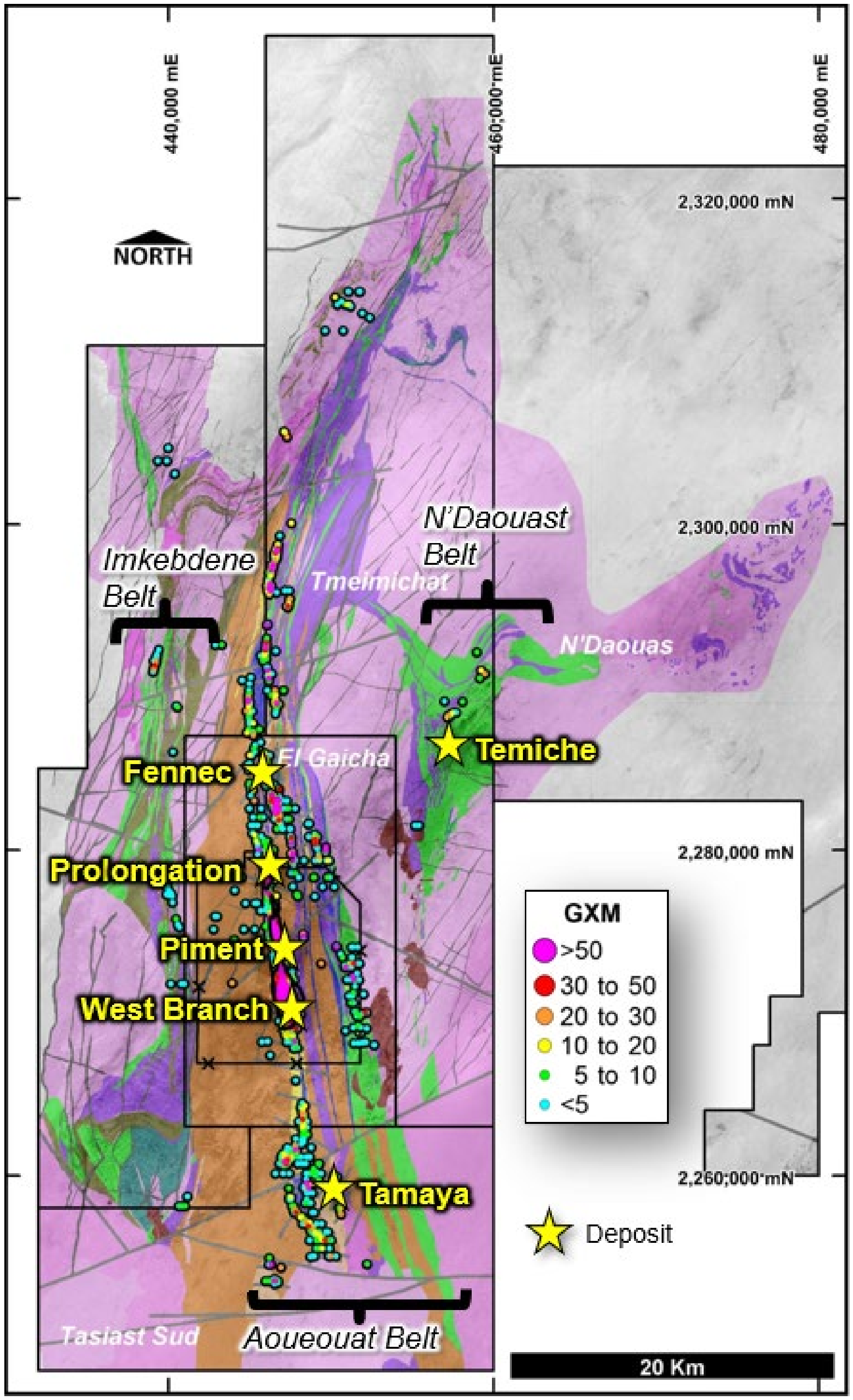


Figure 7-2: Tasiast District tectonic map - Granite-Greenstone Belts

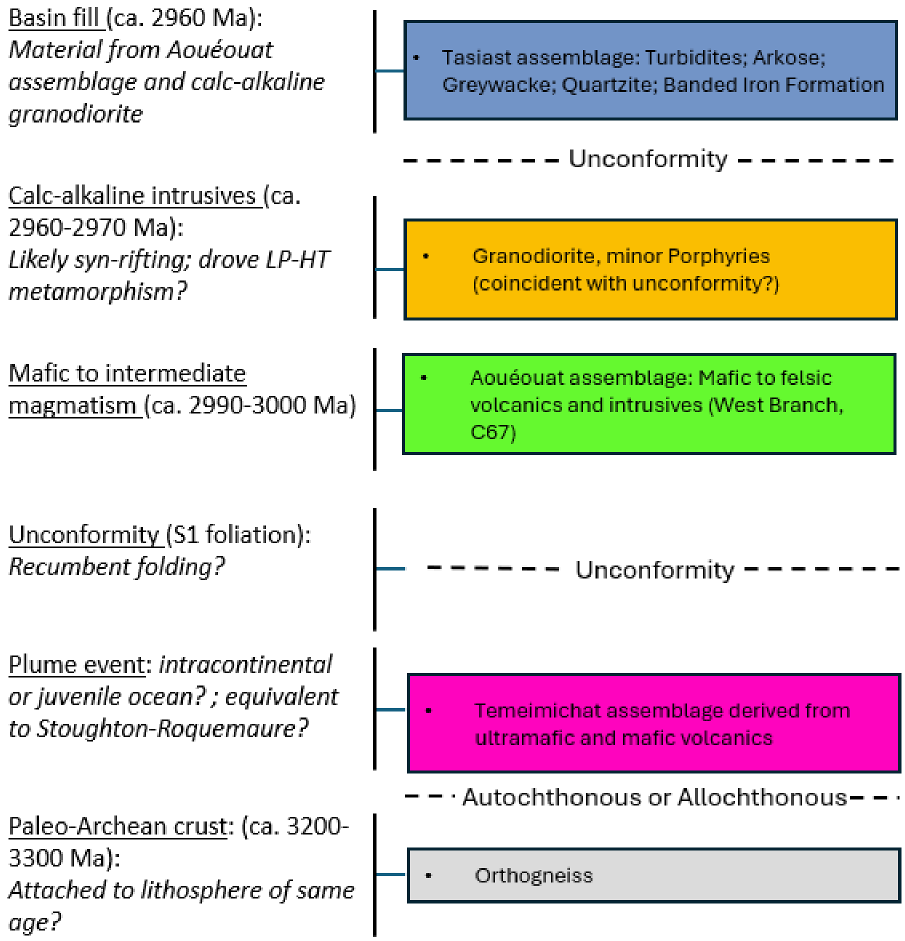


Figure 7-3: Precambrian stratigraphy of the Aouéouat Greenstone Belt

7.3 Deposit Geology

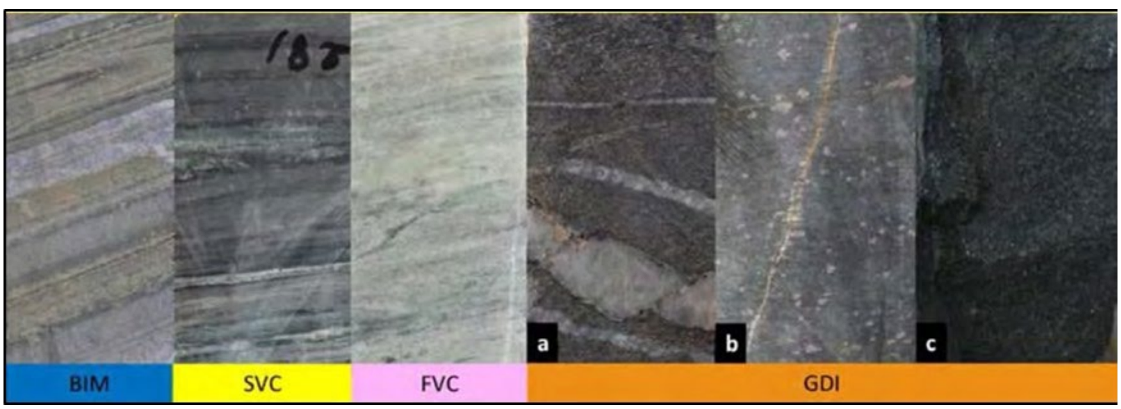
The Tasiast Mine consists of two deposit sets hosted within distinctly different rock types, both situated within the hanging wall of the west-vergent Tasiast thrust (Figure 7-4 and Figure 7-5).

The Piment deposits (Piment and Prolongation) are hosted within metasedimentary rocks including metaturbidites and banded iron formation where the main mineral association consists of magnetite - quartz pyrrhotite ± actinolite ± garnet ± biotite. Gold is associated with silica flooding and sulphide replacement of magnetite in the turbidites and in the banded iron formation units.

The West Branch deposit geological succession comprises mafic to felsic volcanic sequences, iron- rich formations and clastic units that have undergone mid greenschist to lower amphibolite facies metamorphism and multiple deformation events.

The main units recognized at West Branch are:

- **GDI** – A diorite to quartz diorite intrusion(s) of the Aouéouat Assemblage. Historically, the GDI was referred to as the Greenschist Zone (GST).
- **FVC** – Felsite of the Aouéouat Assemblage.
- **SVC** and **BIM** – Siliciclastic metasedimentary rocks of the Tasiast Assemblage.
- **MDO** – Mafic dykes, post schistosity and post mineralization



Notes:

1. Biotite rich, 2% to 5% sulphides (pyrrhotite>>pyrite) intense quartz- carbonate veining (5% to 10%) folded and boudinaged => biotite schist (BST) zone
2. Actinolite+ garnet >biotite => GST 1 zone
3. biotite rich => BST

Figure 7-4: Main lithologies identified in the West Branch area

Most of the gold mineralization at West Branch is hosted by quartz–carbonate veins within the sheared and hydrothermally altered meta-diorites that constitute the Greenschist Zone. All the significant mineralized bodies defined to date dip moderately to steeply (45° to 70°) to the east and have a south–southeasterly plunge (Figure 7-6 and Figure 7-7).

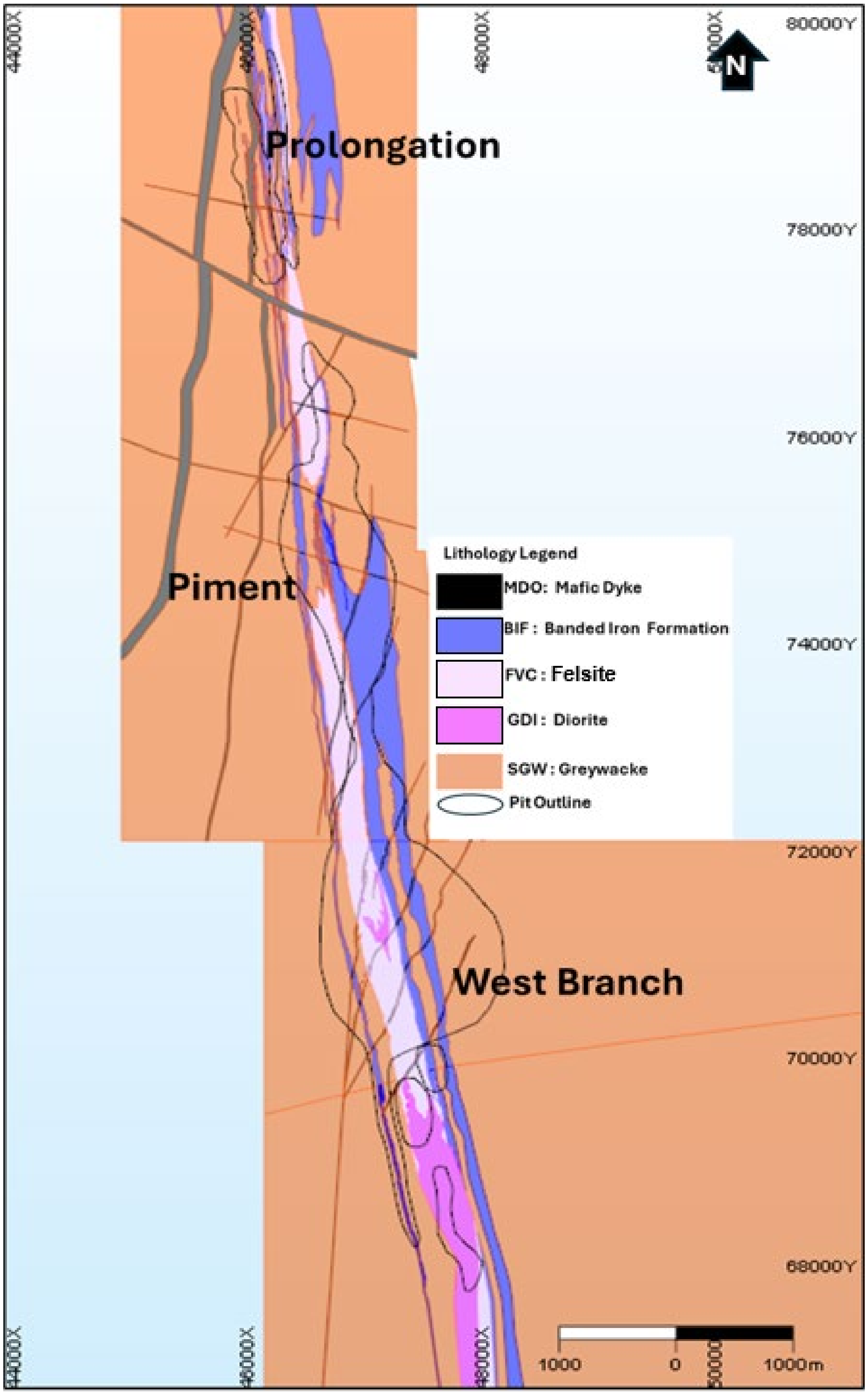


Figure 7-5: Near mine geology model (looking down at 4,950 m elev)

Diorites and Basalts of the Aouéouat Assemblage

The majority of the economic mineralization at West Branch is hosted by a diorite to quartz diorite intrusion (termed the GDI) which has intruded into FVC or felsite (clastic sediments). The GDI is light to medium grey, medium- to fine-grained, and composed of plagioclase, quartz, and biotite with some potassium feldspar (up to half of the plagioclase content). It typically shows a zonation from a barren garnet-amphibole assemblage at its margins to an auriferous quartz-biotite-ankerite-pyrite-pyrrhotite assemblage and back into the barren garnet-amphibole assemblage. This unit is also characterized by a distinctive penetrative foliation that is most strongly expressed by the alignment of biotite crystals. It was previously referred to as plagioclase-biotite schist and was logged as schist (SHT) and biotite schist (BST). At West Branch, the GDI consistently averages 50 m to 100 m in thickness over a strike length that exceeds two kilometres.

Felsite of the Aouéouat Assemblage

The FVC unit is a clastic sedimentary sequence of predominantly quartzite with minor locally polymict felsic conglomerate layers. Previously, this unit had been interpreted to be of volcanic origin and hence its confusing name (FVC = Felsic Volcanic). At West Branch, the FVC unit is present both structurally above and below the GDI. The unit is intensely sheared and preserves a well-developed phyllosilicate foliation. Commonly this unit is albitized and is called ‘felsite’, which is not a rock type. Within the FVC, near to its contacts with other units, a cream coloured rock locally occurs that hosts fuchsitic (chromium rich) mica.

Siliciclastic Metasedimentary Rocks of the Tasiast Assemblage

The rocks on the eastern and western sides of the Project area are primarily sediments. Most of these units are greywacke, siltstone, arenite, and iron formation (locally termed BIM or banded-iron-magnetite).

Greywacke, Siltstone & Conglomerate

The SVC or SGW (volcaniclastic – clastic sedimentary rock) is mainly a greywacke-turbiditic clastic sedimentary unit. The beds are typically 1-10 cm thick, well sorted and often graded. The logging code ‘SVC’ has been inherited from early days of exploration when the rocks were interpreted to be volcaniclastic sediments.

Banded Iron and Magnetite

The banded iron mineralization (BIM) is composed of alternating layers of dark greenish magnetite-grunerite and light gray quartzofeldspathic compositions, typical of Algoma style iron formation. The aluminum component is detrital and has a composition-mixing trend to the clastic sedimentary rocks (SVC, SGW). The iron formation units can vary from centimetre to decimetre scale in thickness, with millimetre to centimetre beds common. Although the units are locally tightly folded, attenuated, or boundinaged, individual units can in some cases be traced for hundreds to thousands of metres along strike. Three BIM units have been logged and mapped in the West Branch project area; two in the hanging wall, and one in the footwall of the Tasiast thrust system. The hanging wall BIM units are generally unmineralized and the footwall BIM is variably mineralized (dependent upon proximity to structure). The contact between the hanging wall BIM and the FVC units is locally defined by the presence of a discontinuous conglomerate that contains abundant clasts derived from the FVC unit and a subordinate proportion of clasts derived from mafic metavolcanic units.

Mafic Dykes

Mafic dykes that are post schistosity and post mineralization are dark olive green, fine to medium-grained and are locally plagioclase phyric. Dykes are typically less than five metres wide, weakly magnetic and have locally developed hornfels and brecciated margins with a carbonate-chlorite assemblage. The dykes are dominantly barren and crosscut mineralized units.

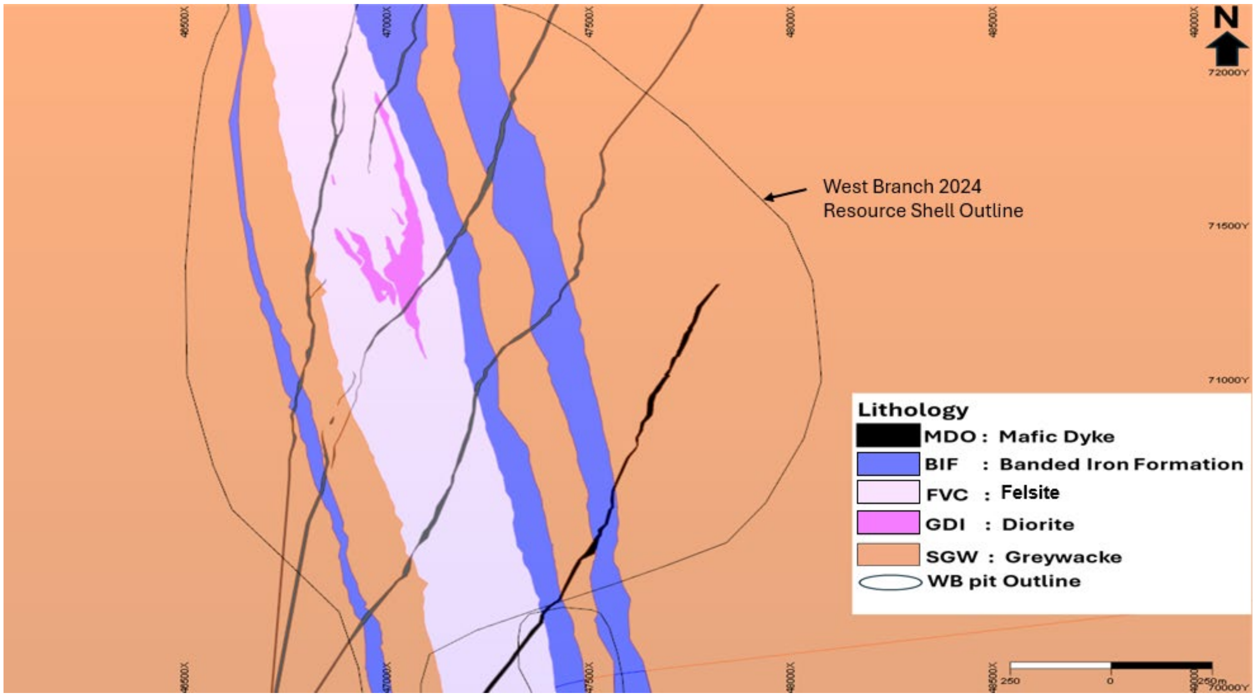


Figure 7-6: Lithology level plan at West Branch – looking down at 4,900 m elev

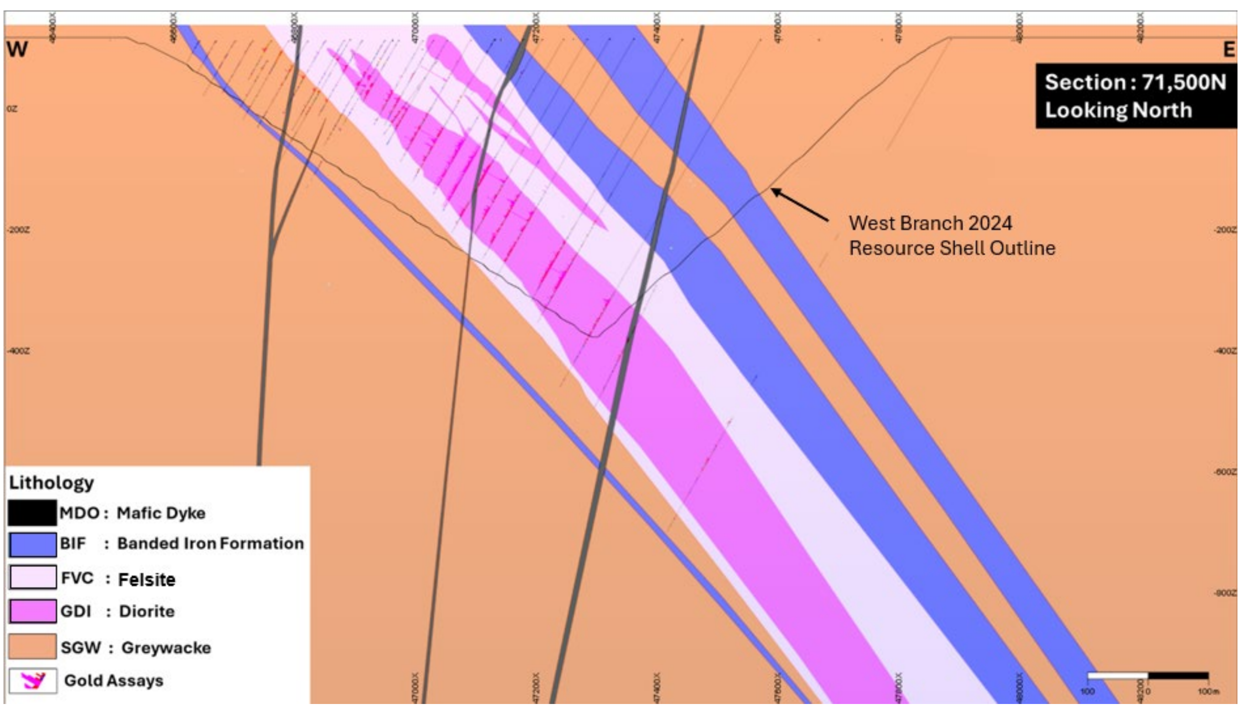


Figure 7-7: Lithology cross section at West Branch – looking north at 71,500 N

7.4 Structural Geology

The Tasiast deposits are hosted within a package of strongly foliated and folded rocks in the hanging wall block of an assumed thrust fault or thrust fault system referred to as the Tasiast thrust system (Figure 7-8). Modelling and interpretation of high-resolution gravity data (Figure 9-1) shows deep geometry suggestive of a thrust system underlying the Aouéouat belt. The Tasiast thrust system displays zones of strong deformation typically 0.5 m to 10 m wide and characterized by laminated foliation with locally preserved mylonitic textures. Hydrothermal alteration assemblages, sulphides, and quartz veins are commonly spatially associated with the zones of intense deformation.

All the Tasiast deposits host an intense, generally north-south striking, variably dipping, penetrative foliation, S1, which is axial planar to tight isoclinal folds in the host sequence (Davis 2018). The foliation fabric within the main mine sequence, at West Branch, dips moderately to the East at 40° to 50°, steepens to the north to 55° to 65° at Piment, and becomes sub-vertical, at the north end of the mine sequence, near Prolongation.

Pit mapping at Tasiast includes the collection of structural fabric measurements for structural geology and geotechnical application. Numerous consultants have assisted Kinross in developing a structural model for the Tasiast deposits along with developing pit mapping procedures. Most recently, in 2018, Dominique Chardon worked with the Mine Geology department to review and analyze the significant database of structural data collected from pit mapping activities. A synthesis of this review is presented in Figure 7-9 as detailed pit-scale maps.

Quartz-carbonate vein sets occur sub-parallel and oblique to foliation and range in style from boudinaged, buckled, folded to planar. The veins clearly formed in extensional and/or Riedel shear orientations and were progressively folded, rotated, locally boudinaged, and partially or wholly transposed parallel to the foliation. In the core of the West Branch Greenschist Zone vein, densities are typically higher in the meta-intrusive dioritic unit (averaging between 2% to 5%) than in the meta-basalt (<2%). This higher density suggests the coarser-grained feldspar-rich dioritic facies focused stresses and readily developed brittle-ductile shears, as expected for quartzofeldspathic rocks under retrograde Greenschist metamorphic conditions. Along the margins of the West Branch deposit, both the dioritic and meta-mafic volcanic units have a low vein density (<1%). Quartz-carbonate veins also developed locally within FVC that envelops the Greenschist Zone and within the footwall meta-sedimentary units.

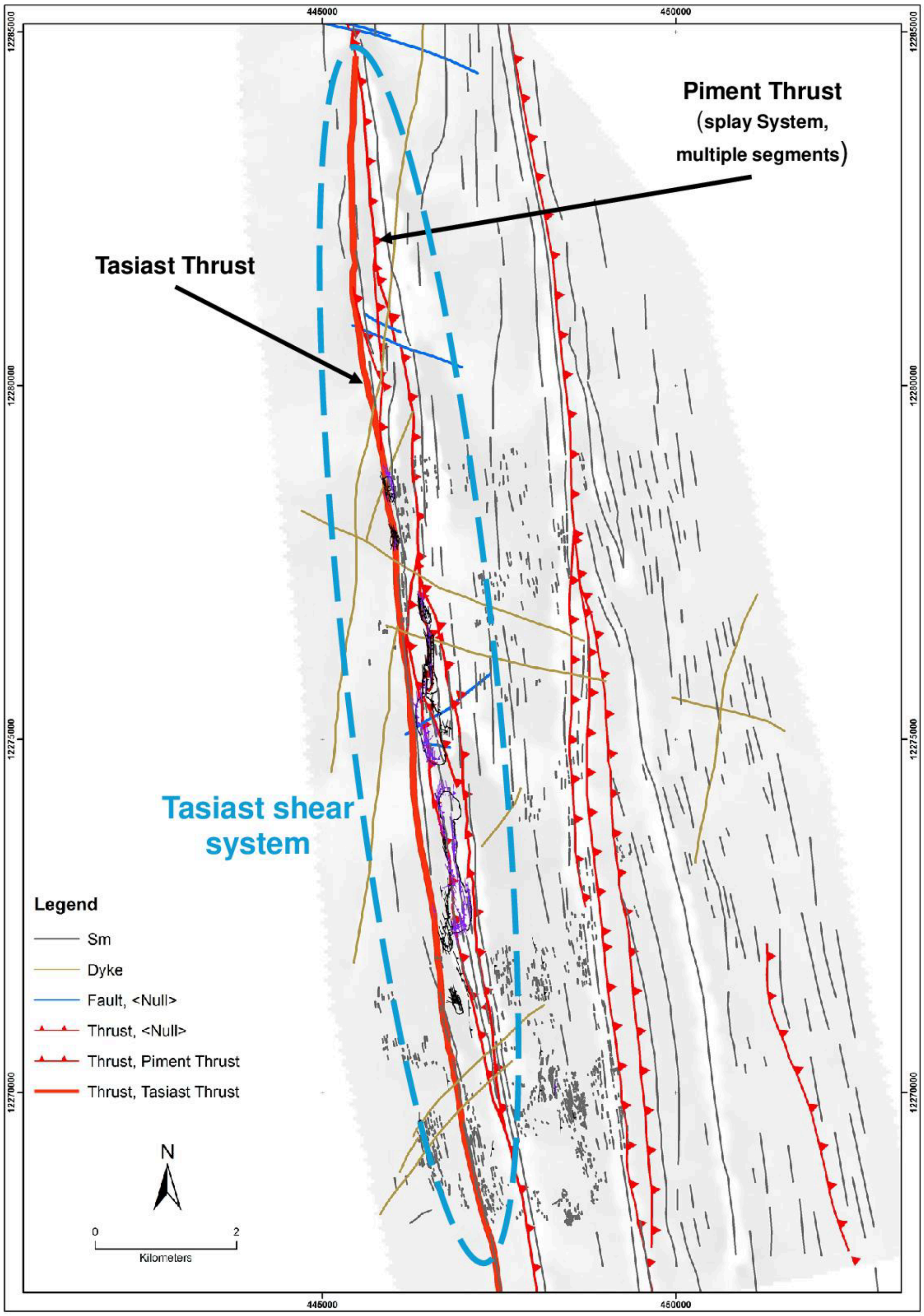
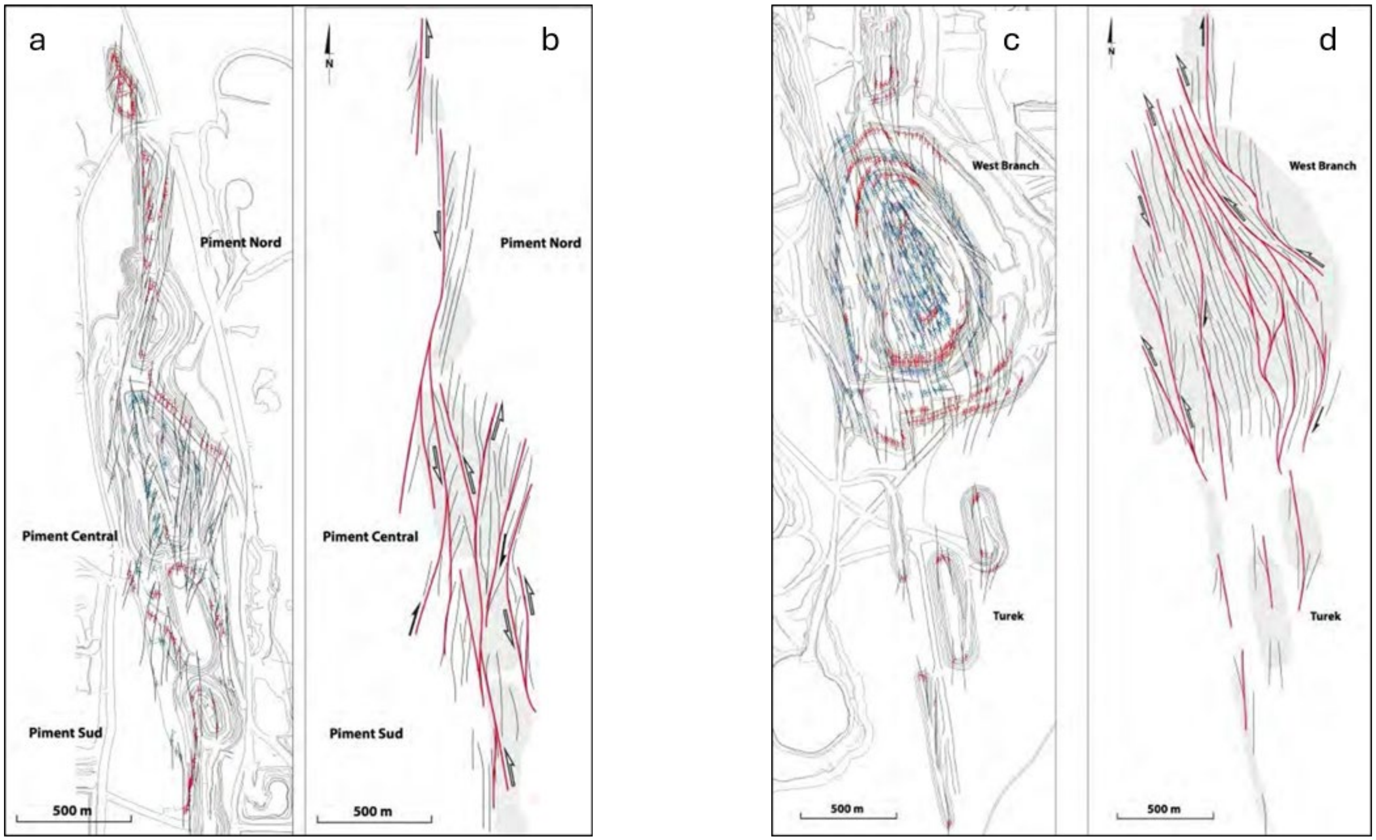


Figure 7-8: Local scale shear framework and form surface map



a. Fabric measurements (in red and blue) and fabric trajectories (in black) at Piment
b. Fabric trajectories and interpretative shear zone pattern (in red) at Piment
c. Fabric measurements (in red and blue) and fabric trajectories (in black) at West Branch and Turek
d. Fabric trajectories and interpretative shear zone pattern (in red) at West Branch and Turek

Figure 7-9: Structural maps of the northern pits (Piment; a, b) and southern pits (West Branch, Turek; c, d)

7.5 Mineralization and Alteration

All the rocks in the property area have undergone lower amphibolite facies metamorphism. Given the metamorphic grade, it is challenging to identify the rocks' alteration as it has been largely, if not totally, overprinted by the metamorphism. A description of the commonly observed types of alteration are presented in Table 7-1.

Table 7-1: West Branch alteration codes and descriptions

Code	Meaning	Occurrence
ALB	Albite-Biotite Albite > Biotite	SVC unit.
BIO	Biotite rich	SVC unit in both FW and HW and in the upper lens of the GDI
BST	Biotite >>> Actinolite, pyrrhotite» pyrite, Quartz veins rich but garnet is absent	GDI and is a good proxy for the 2 git Au grade shell
GST1	Actinolite-Garnet > Biotite	GDI, outer envelope around the mineralization.
GST2	Biotite-Garnet > Biotite	Transitional unit between GST 1 and BST
GST3	Like GST1 with abundant magnetite (very magnetic)	Typically found in BIM

The bulk of the mineralization at the West Branch deposit is hosted within the GDI. The GDI typically shows a zonation from a barren garnet-amphibole assemblage at its margins to an auriferous quartz-biotite-ankerite-pyrite-pyrrhotite assemblage and back into the barren garnet-amphibole assemblage. This zonation likely reflects a metamorphosed alteration assemblage with the garnet-amphibole assemblage representing a chlorite alteration precursor and the biotite-quartz-sulfides representing a quartz-sericite precursor.

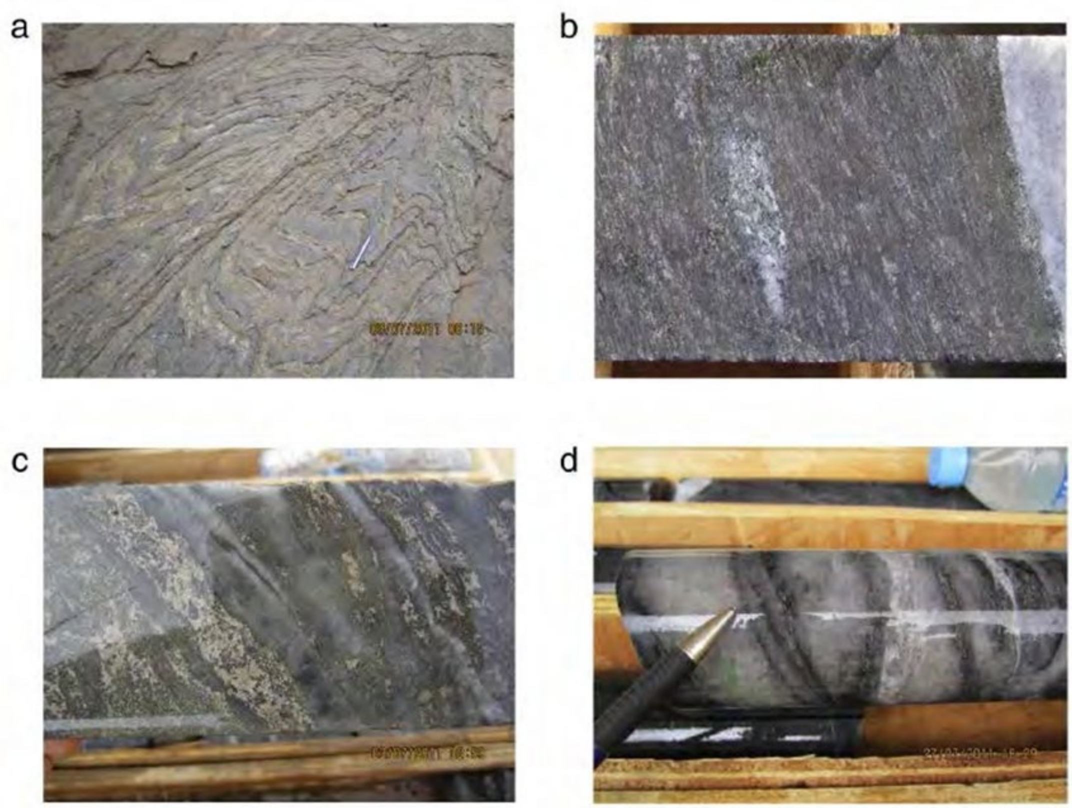
Gold mineralization is also present in the other main units (FVC, SVC & BIM) (see Figure 7-10).

In the FVC, two different styles of mineralization are observed:

- High-grade quartz veins (approximately 5 g/t to 10 g/t) stockwork directly above the "apex" of the GDI. This is often referred to as Pluto style mineralization. The veins are highly deformed, in particular folded. The strike and down dip continuity is limited to a few hundreds of metres.

- Lower grade quartz veins (averaging approximately one gram per tonne) stockwork preferentially in the hanging wall and central part. The footwall is usually barren.

In the SVC unit, gold is associated with quartz veins and garnet - actinolite metamorphosed assemblage. In the BIM unit, gold is associated with sulphidic replacement of the magnetite by pyrrhotite. Locally quartz veining is also developed.



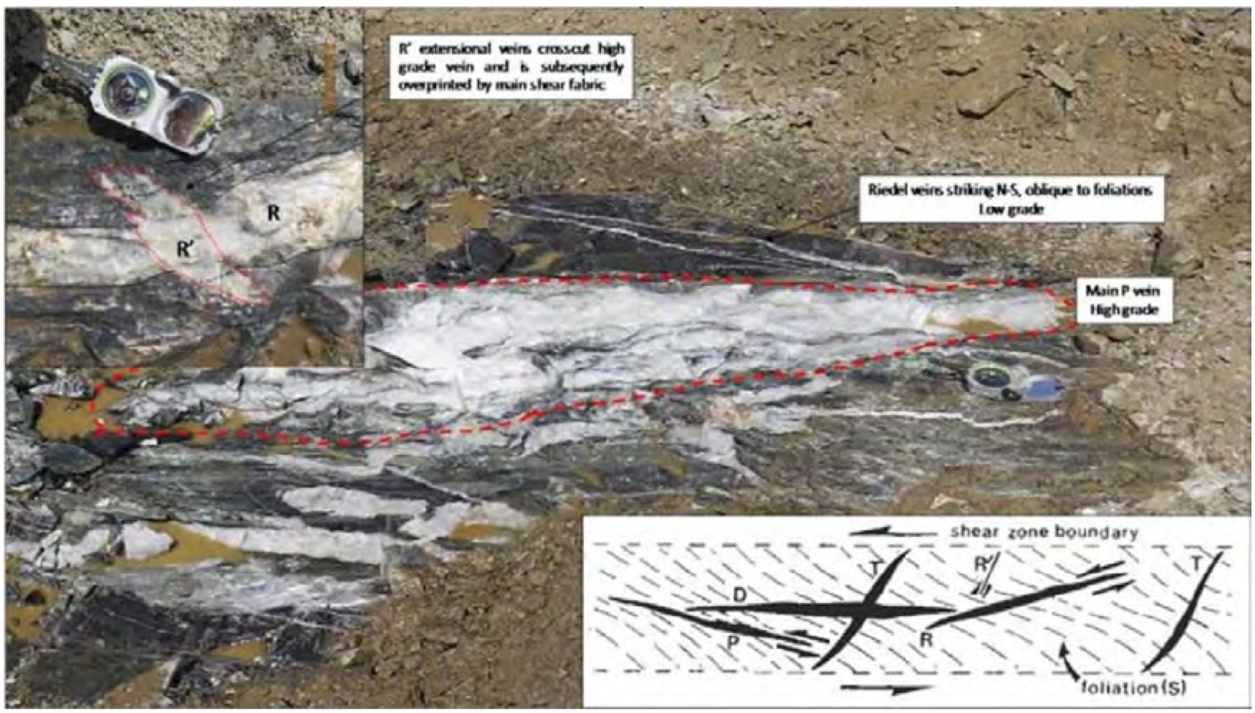
Notes:

a. Isoclinally folded sediments that are representative of the host rocks for all the Piment deposits, Tasiast Mine, pen for scale.
b. Strongly foliated meta-diorite from West Branch typical of the Greenschist Zone.
c. Silica flooding and extensive sulphide mineralization, mainly pyrrhotite, parallel to bedding in the metaturbidites at Piment.
d. Visible gold hosted in a quartz–carbonate vein, in drill core taken from the West Branch deposit.

Figure 7-10: Photographs illustrating mineralization styles

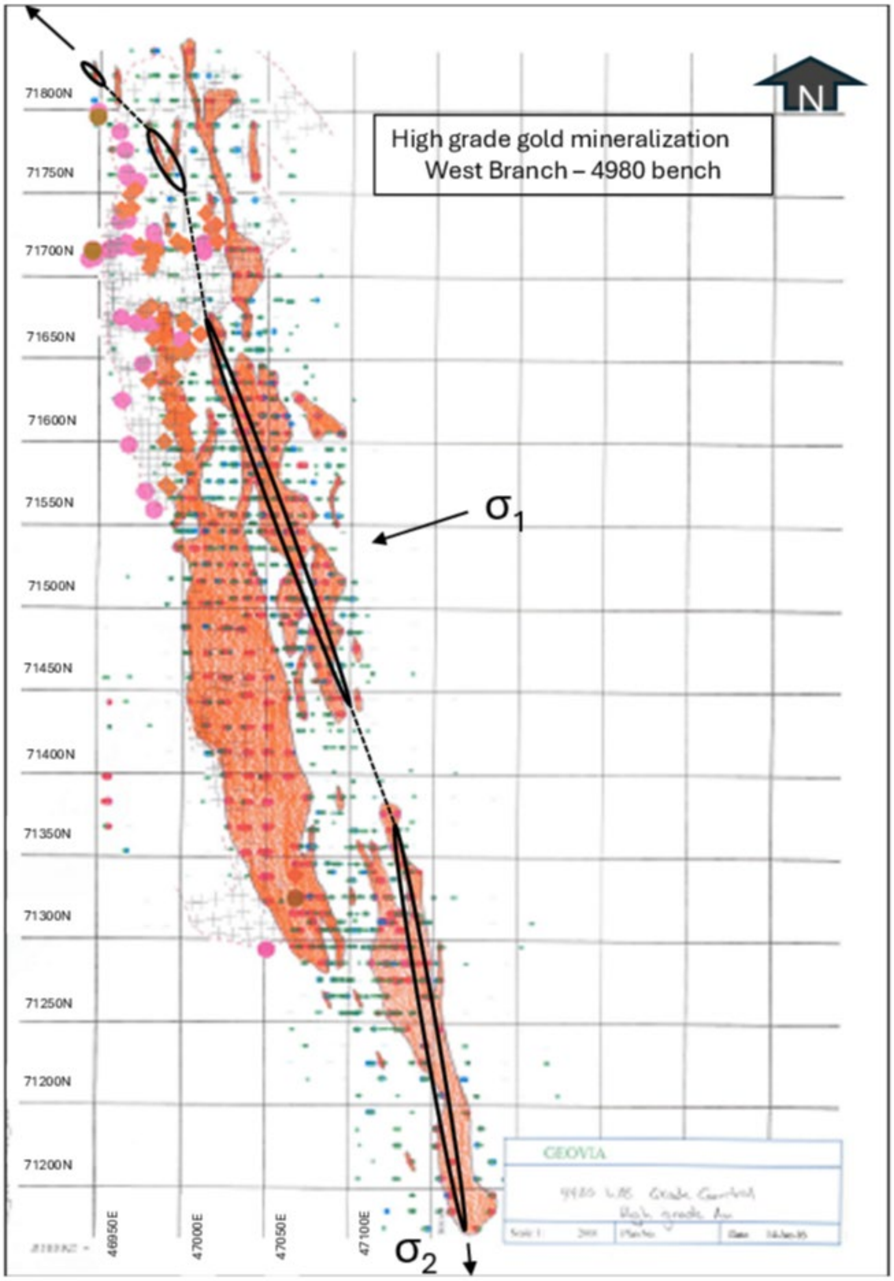
Ore control and pit mapping identified a high-grade quartz-carbonate-chlorite-tourmaline-gold vein which is coincident with an interpreted Tasiast splay (locally termed the central fault). The vein has been mapped over several benches striking 330° to 340° and dipping at approximately 55° to 60°to the east (Figure 7-11 and Figure 7-12).

Quartz-carbonate vein sets occur sub-parallel and oblique to foliation and range in style from boudinaged, buckled, folded to planar. The veins formed in extensional and/or Riedel shear orientations and were progressively folded, rotated, locally boudinaged, and partially or wholly transposed parallel to the foliation. Density of veining is typically higher in the GDI (averaging between 2% to 5%) than in the meta-basalt (<2%). Quartz-carbonate veins also observed locally within FVC that envelops the GDI and within the footwall meta- sedimentary units.



Note: Multiple vein sets are observed (Riedel, Riedel'/extensional).

Figure 7-11: West Branch 4980 m bench high-grade vein



Note: Vein is completely boudinaged at extremities to the north and south.

Figure 7-12: West Branch 4980 m bench showing mapped GDI (orange) and high-grade vein (black)

Gold occurs as both microscopic grains and coarse visible gold. When observed in hand specimen, grains are commonly spatially associated with hairline fractures in quartz veins and margins of sulphide minerals. The majority of the quartz veins containing coarse visible gold cut the foliation at a slightly oblique angle and mainly dip gently to the east. Most gold grains occur along the margin of the gangue and ore minerals, with 98% of the calculated volume/mass of the grains occurring in liberated and partially liberated forms. By volume/mass calculations, the majority (greater than 70%) of the volume is associated with the coarser (plus 75 μm) size fraction. Encapsulated gold grains are rarely observed. When present they are predominantly a very fine grain size. Semi-quantitative scanning electron microscopy analysis of gold grains indicated low silver (less than 15%) and trace iron (less than 3%) content.

8. DEPOSIT TYPES

Tasiast gold deposits are hosted in Archaean volcanic-sedimentary sequences that have been deformed and metamorphosed to lower amphibolite peak metamorphic grade. Mineralization is both structurally and lithologically controlled, epigenetic in style and was coincident with early stages of post-peak metamorphic retrograde Greenschist P-T conditions.

The Tasiast gold deposits fall into the broad category of orogenic gold deposits. The regional geological setting and deposit features at Tasiast are similar to other well-known Archaean lode gold deposits hosted along greenstone belts in granitoid-greenstone terranes.

Examples of analogue terranes of similar ages to the Aouéouat belt include the Kaapvaal craton in South Africa, the Pilbara craton in Australia and the Wyoming craton in the USA. The Aouéouat belt also shares many similarities with gold-rich Archaean terranes, such as the Yilgarn in Australia and the Abitibi in Canada.

9. EXPLORATION

Exploration activities have been undertaken by TMLSA, its precursor companies, consultants, and contractors.

9.1 Grids and Surveys

The coordinate system used on site is a mine grid, a truncated UTM Zone 28 North grid system; the UTM Easting is shifted by -400,000 m and UTM Northing is shifted by -2,200,000 m. The Original Control has been set out by IPH Engineering and 10 control points are set out across the Complex. Surveyors use a differential global positioning system (GPS) for surveying at the Complex (Trimble DGPS TSC3, Scanner MAPTEK I-Site 8820, and Trimble Drone UX5HP).

9.2 Geological and Regolith Mapping

Numerous phases of geological and regolith mapping have been undertaken during the life of the project and range from regional (1:100,000) to prospect (1:1,000) scale. Work was completed by the BRGM, SNIM, NLSD, Defiance, Red Back, and Kinross. Mapping was facilitated by good outcrop, RC and diamond drilling (DD), high resolution satellite imagery, and detailed airborne geophysical data. Results were used to identify areas of alteration, structural complexity, quartz-carbonate veining, and sulphide outcrop that warranted additional work.

9.3 Geochemistry Sampling

A total of 22,294 surface samples have been collected by Kinross since it started operations, including soil samples (40%) and rock samples (60%) that cover a surface area of approximately 1,000 km^2. In addition, 299 auger drill samples were collected during 2016. Soil samples were collected by a contractor and supervised by Kinross staff. The sample grid was generally west-east with lines spacing at 800 m and sample spacing at 200 m. The geochemical sampling includes exposed geology as well as areas covered by sand, in which the bedrock was sampled with auger drilling. Accordingly, the geochemical dataset has the potential to identify areas of prospective mineralization otherwise blind from surface mapping. Surface exploration geochemistry samples were analyzed for gold and multi-element geochemistry.

9.4 Geophysics

Airborne magnetic-radiometric surveys were completed by NLSD (2000-2001) and Red Back (2007). These surveys were mainly used to map out lithological formations and major structures. In 2008-2009, Red Back completed a helicopter-borne versatile time domain electromagnetic (VTEM) survey. In 2011, Kinross completed airborne magnetic and radiometric surveys over the complete licence package. This survey overlapped the previous survey and generated a higher resolution version. In 2013, Kinross completed ground based gravimetric and induced polarization (PDIP/IP) surveys. The gravity survey covered the complete licence package (Figure 9-1) while the IP surveys covered only specific prospect areas; South West Branch South, Fennec, C67, C68, and Morris. In 2014, a comprehensive review was completed by a consulting geophysicist, along with some reprocessing of the magnetic data. In December 2023, a 3D IP survey was completed by geoscience consulting firm Quantec Geoscience Ltd under contract with Kinross to help define the nature and setting of gold mineralization on the SENISA exploitation permits, and to gain a better understanding of the complex structural geology controls around spatial distribution and gold ore genesis. The survey covered target areas at GRD, C23, KNF, and Morris (N1-N2), as shown in Figure 9-2. Results of the IP chargeability for the different areas are shown in Figure 9-3 to Figure 9-6.

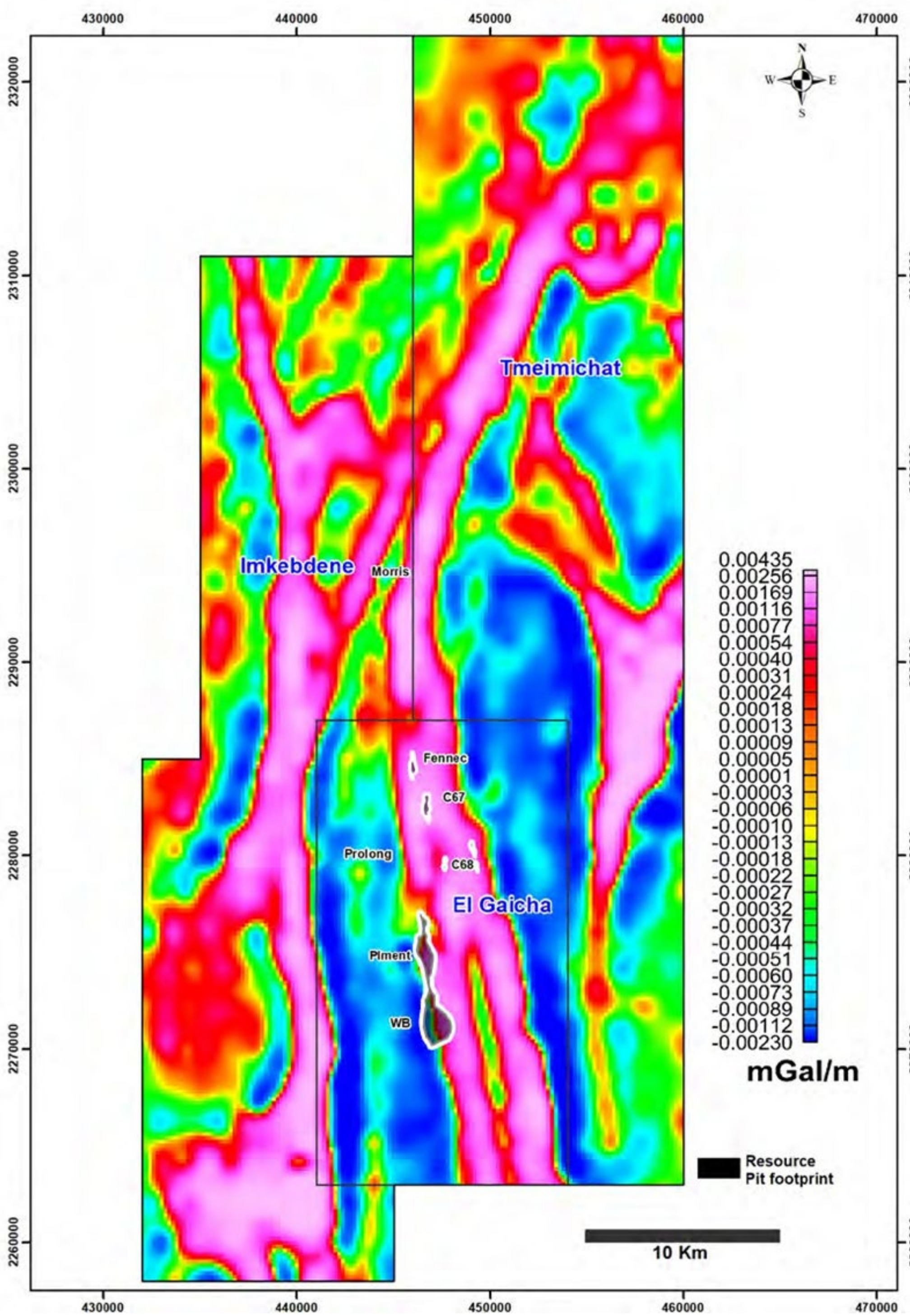


Figure 9-1: Depth slice (880 m) of the Bouguer gravity 1VD

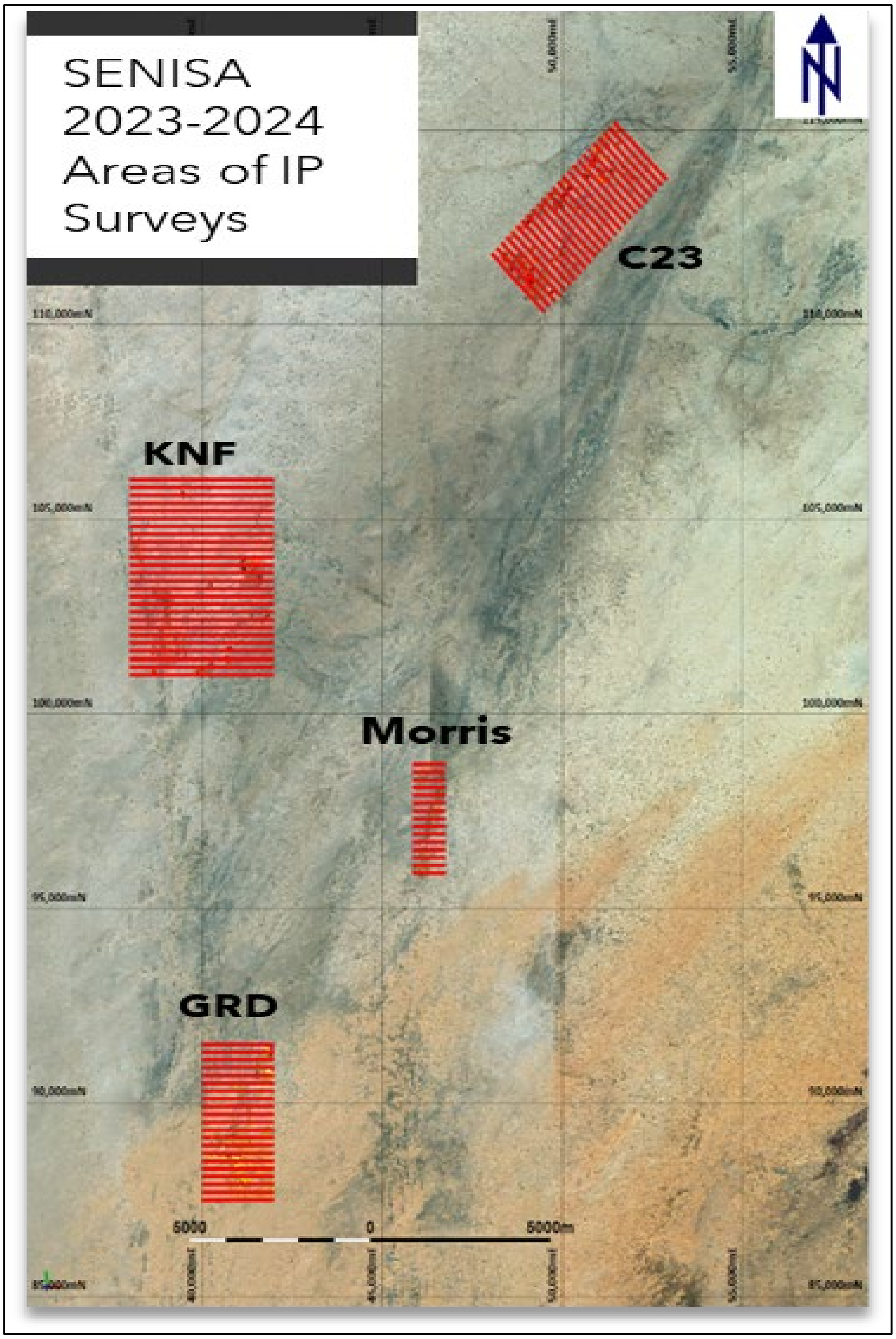


Figure 9-2: Senisa target area of IP surveys

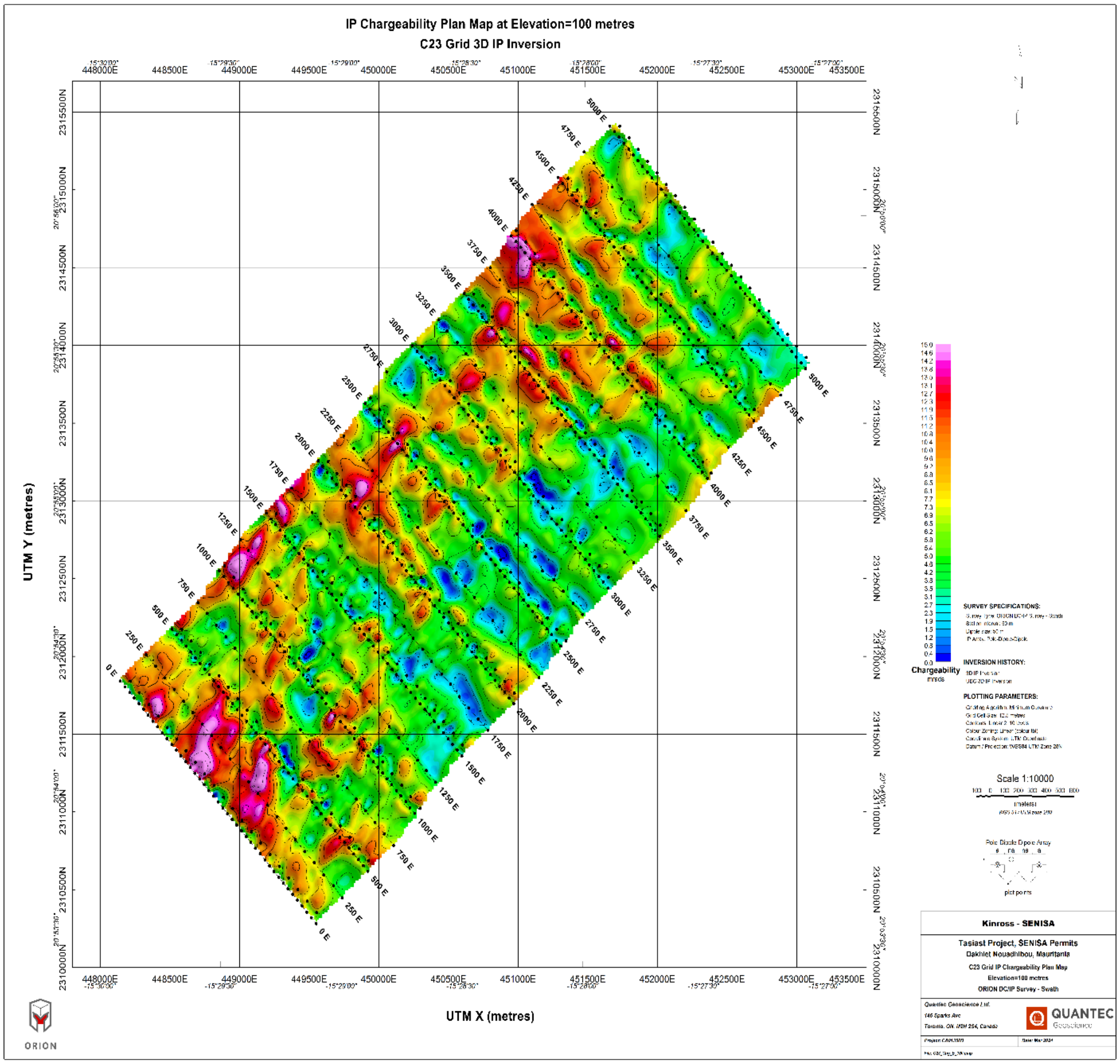


Figure 9-3: Depth slice (100 m) of IP chargeability at C23

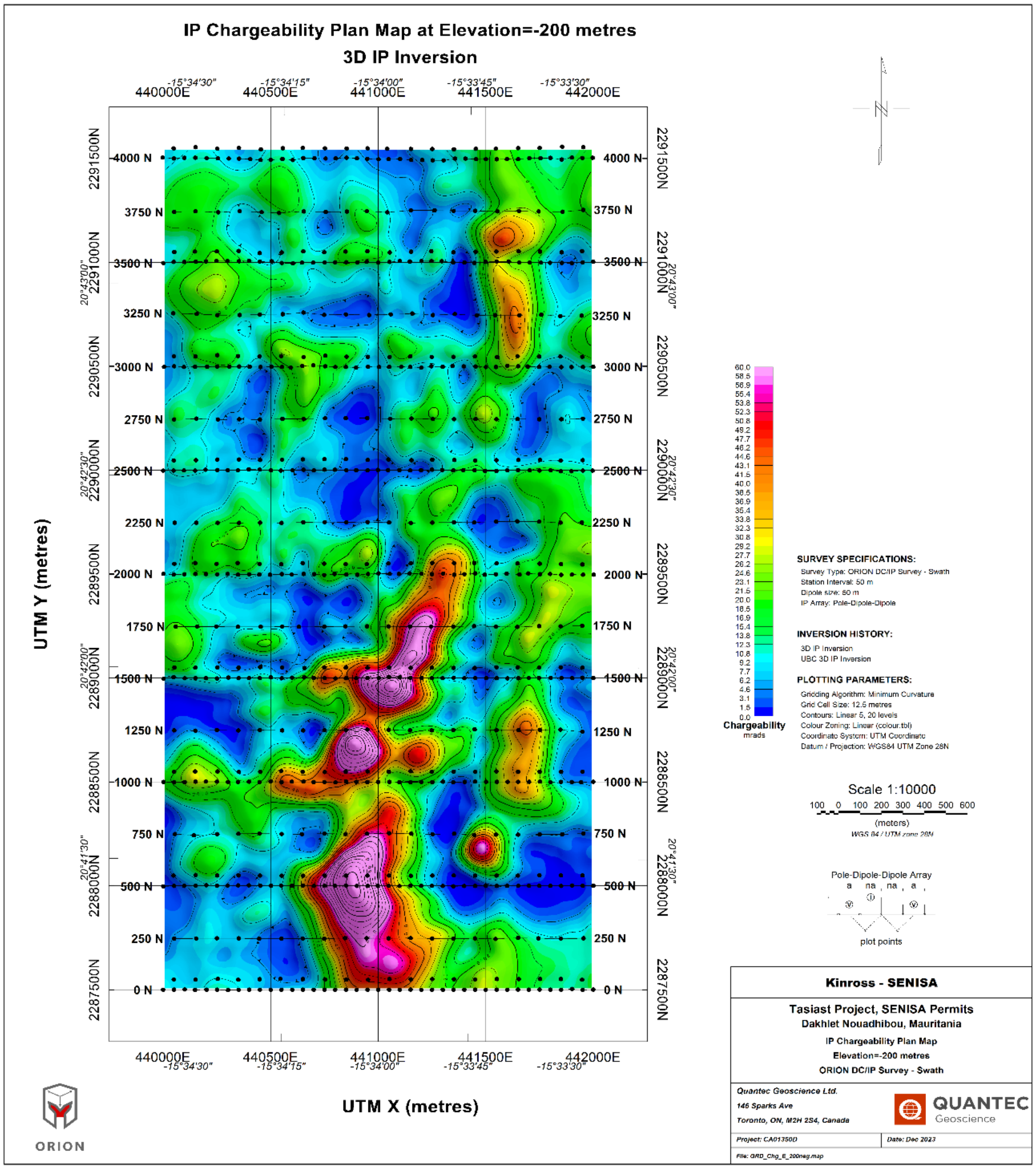


Figure 9-4: Depth slice (200 m) of IP chargeability at GRD

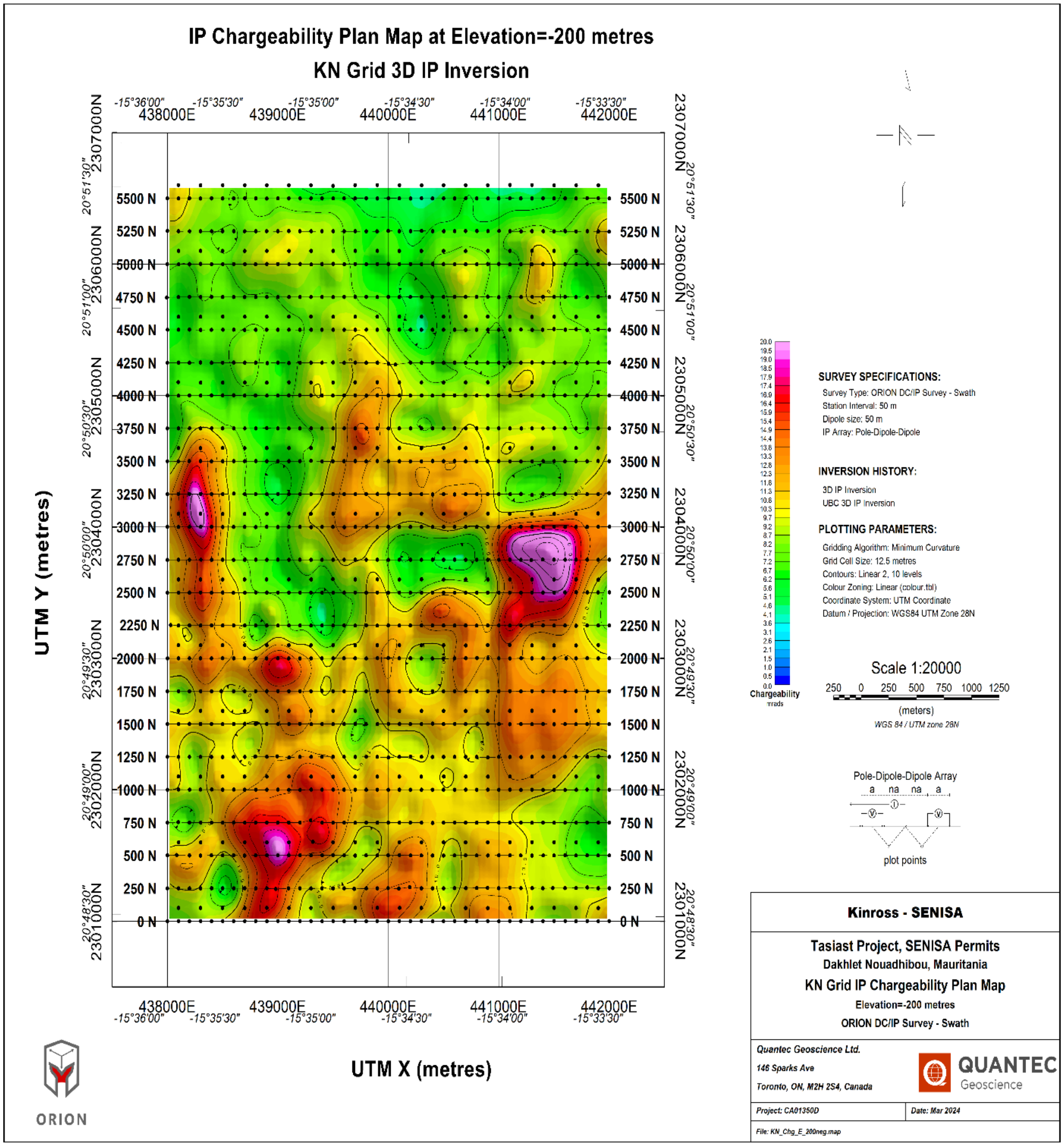


Figure 9-5: Depth Slice (200 m) of IP chargeability at KHN

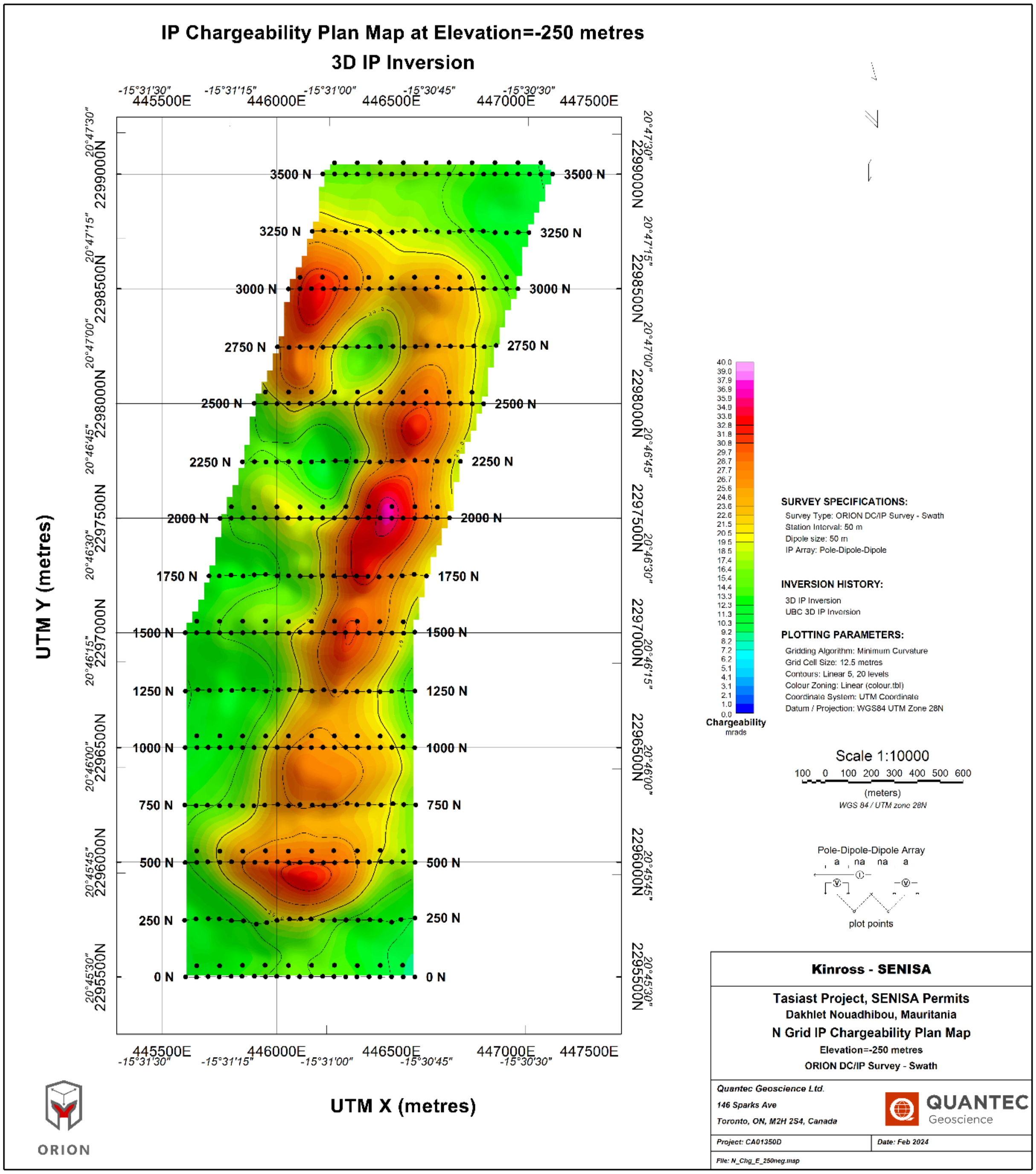


Figure 9-6: Depth slice (250 m) of IP chargeability at Morris

9.5 Pits and Trenches

Excavation of trenches as an exploration technique has been very successful at Tasiast. In total, 485 trenches for 111,068 m have been completed across the Tasiast lands. Historically, trenches were completed manually, and since the 2000s, trenches are completed using an excavator. The standard excavated trench dimension is approximately two metres wide and not more than 1.5 m deep and typically sampled every two metres along the full length of the trench.

9.6 Petrology, Mineralogy, and Other Research Studies

Numerous petrographic and gold deportment studies have been completed by TMLSA and predecessors on Tasiast in 2006, 2010, 2011, 2012, and 2017. In 2010, Red Back submitted 10 core samples from West Branch for a petrological and mineralogical study. Results from the work indicated significant pyrrhotite mineralization developed along foliation planes and associated with accessory magnetite, chalcopyrite, and pyrite (Strashimirov, 2010). Further petrological studies were carried out for Kinross in 2010-2017, including work by Larson (2011), Pollard (2011), and Panterra Geoservices (2012 and 2017), which concluded that quartz veins are pre- and/or syn- tectonic and folded or transposed into the dominant foliation and pyrrhotite is the dominant sulphide mineral in many samples. A mineralogical (gold) characterization study of five samples was completed by Blake (2011a, b). The main conclusions were that coarse gold forms a significant component of the total gold content in the samples and that native gold grains encapsulated within their host (gangue/ore minerals) are relatively uncommon and often exhibit a very fine grain size.

9.7 Exploration Potential

The Tasiast area has significant exploration potential to delineate additional resources both around the Tasiast mine pits (near mine exploration) and within the wider Complex (generative exploration). Exploration targeting and target ranking at Tasiast incorporates all available datasets including; satellite imagery (Worldview-2), airborne geophysical data (high resolution aeromagnetics and VTEM), reconnaissance scale geological prospecting, regional-, district-, and target-scale geological mapping, surface soil and auger sampling (gold and multi-element geochemical data), trenching, reconnaissance style shallow RC drilling on fences and conventional reverse circulation/diamond drilling to define resources.

Exploration efforts to date have discovered additional prospects, gold deposits, and mineral resources along strike to the north and to the south of the main Tasiast area (West Branch and Piment-Prolongation), and generally along the Aouéouat (Tasiast) belt. The deformed greenstone rocks to the west (Imkebden-Kneiffissat) of the Aouéouat belt are notable in that they host quartz-carbonate veins with anomalous gold values, however, to date no significant deposits or mineral resource have been defined. To the immediate north of the Tasiast operation (5 km to 12 km) and within the Guelb El Ghaîcha mining permit, a cluster of deposits referred to as “northern satellites” have been outlined, namely Fennec, C67, and C68 (W and Central). These gold deposits currently host approximately 0.5 Moz Au and are part of the near-mine resource growth strategy. Further north of the Tasiast operation (12 km to 25 km) and within the Imkebden and Tmeimichat mining permits are another cluster of gold deposits referred to simply as “Morris”, namely Tef, Askaf, Central, NE, N1, and N2. This large area saw extensive exploration drilling from 2012 to 2014, which resulted in the discovery of several small deposits best described as narrow, high-grade vein systems. Most of these deposits are open to depth down plunge.

Beyond 25 km from the Tasiast operation, within the northern extents of the Imkebden and Tmeimichat exploitation permits, are several gold exploration prospects that are pending follow-up exploration and drilling, of note are C23, Khnefissat, and Grindstone (Figure 9-7). These prospects have significant surface geochemical footprints and are considered highly prospective.

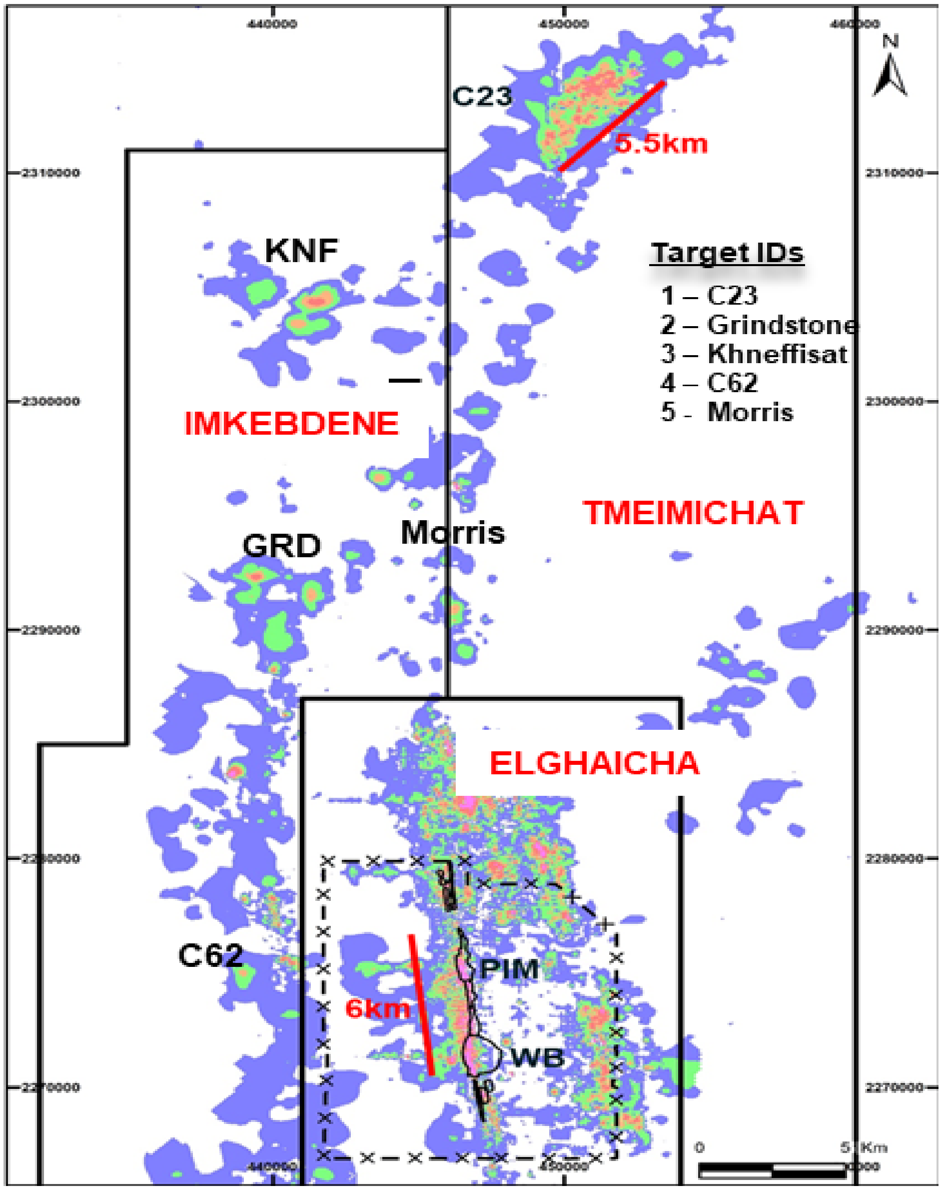


Figure 9-7: C23, KHN, and GRD targets and soils anomalies (Red Back data)

10. DRILLING

10.1 Summary

To date, 15,953 drill holes (14,849 RC, 874 diamond core (DD), and 230 RC pre-collar with DD tail (RC-DD)) for an aggregate total of 1,725,424 m have been completed within the three mining licences that constitute the Tasiast Lands (Table 10-1, Figure 10-1). Drilling activities peaked in 2011 during the West Branch resource definition program. Drilling activities were conducted by various drilling contractors and supervised by geological staff of the project operator. Where programs are referred to by company name, that company was the project manager at the time of drilling and was responsible for the collection of data. Since 2012, most drilling has concentrated on exploration targets north and south of the main mining areas of West Branch, Piment, and Prolongation.

Table 10-1: Kinross regional drill hole summary

Company	Year	RC		RC-DD		Diamond		Total Drilling	
		Count	Metres	Count	Metres	Count	Metres	Count	Metres
Historical Operators									
Normandy LaSource	1999	289	24,437			10	525	299	24,962
Normandy LaSource	2000	50	3,603			36	4,871	86	8,474
Midas	2003	84	7,898	4	1,417	29	2,908	117	12,224
Defiance	2003	219	17,914					219	17,914
Defiance	2004	6	1,207					6	1,207
Rio Narcea	2004	106	7,740					106	7,740
Rio Narcea	2006	9	1,435					9	1,435
Rio Narcea	2007	70	7,238	1	236	60	7,375	131	14,849
Red Back	2007	173	18,007	1	173	11	316	185	18,496
Red Back	2008	1,019	112,337			23	2,716	1,042	115,053
Red Back	2009	2,857	200,482	1	300	28	3,492	2,886	204,274
Red Back	2010	1,662	159,801	18	12,766	64	12,454	1,744	185,021
Sub-total		**6,544**	**562,099**	**25**	**14,892**	**261**	**34,657**	**6,830**	**611,649**
Current Operator									
Kinross	2010	895	73,922	65	45,581	1	683	961	120,186
Kinross	2011	2,691	281,796	95	65,839	185	104,185	2,971	451,820
Kinross	2012	1,562	152,604	2	251	216	50,569	1,780	203,424
Kinross	2013	709	68,434	14	2,757	102	16,367	825	87,558
Kinross	2014	253	20,308	1	383	19	3,959	273	24,650
Kinross	2015	353	33,592	2	1,243	24	2,151	379	36,986
Kinross	2016	194	19,258	14	4,764	2	371	210	24,393
Kinross	2017	504	44,604	2	290	31	2,404	537	47,298
Kinross	2018	226	28,187	10	4,683	10	887	246	33,757
Kinross	2019								
Kinross	2020	61	4,062					61	4,062

Company	Year	RC		RC-DD		Diamond		Total Drilling	
		Count	Metres	Count	Metres	Count	Metres	Count	Metres
Kinross	2021	220	15,356			1	441	221	15,797
Kinross	2022	4	429			8	3,444	12	3,873
Kinross	2023	68	5,440					68	5,440
Kinross	2024	565	48,078			14	6,453	579	54,531
Sub-total		**8,305**	**796,070**	**205**	**125,791**	**613**	**191,914**	**9,123**	**1,113,775**
Total		**14,849**	**1,358,168**	**230**	**140,684**	**874**	**226,572**	**15,953**	**1,725,424**

Notes:

1. Excludes 13 rotary air blast (RAB) holes completed in 2014
2. Data as at December 31, 2024

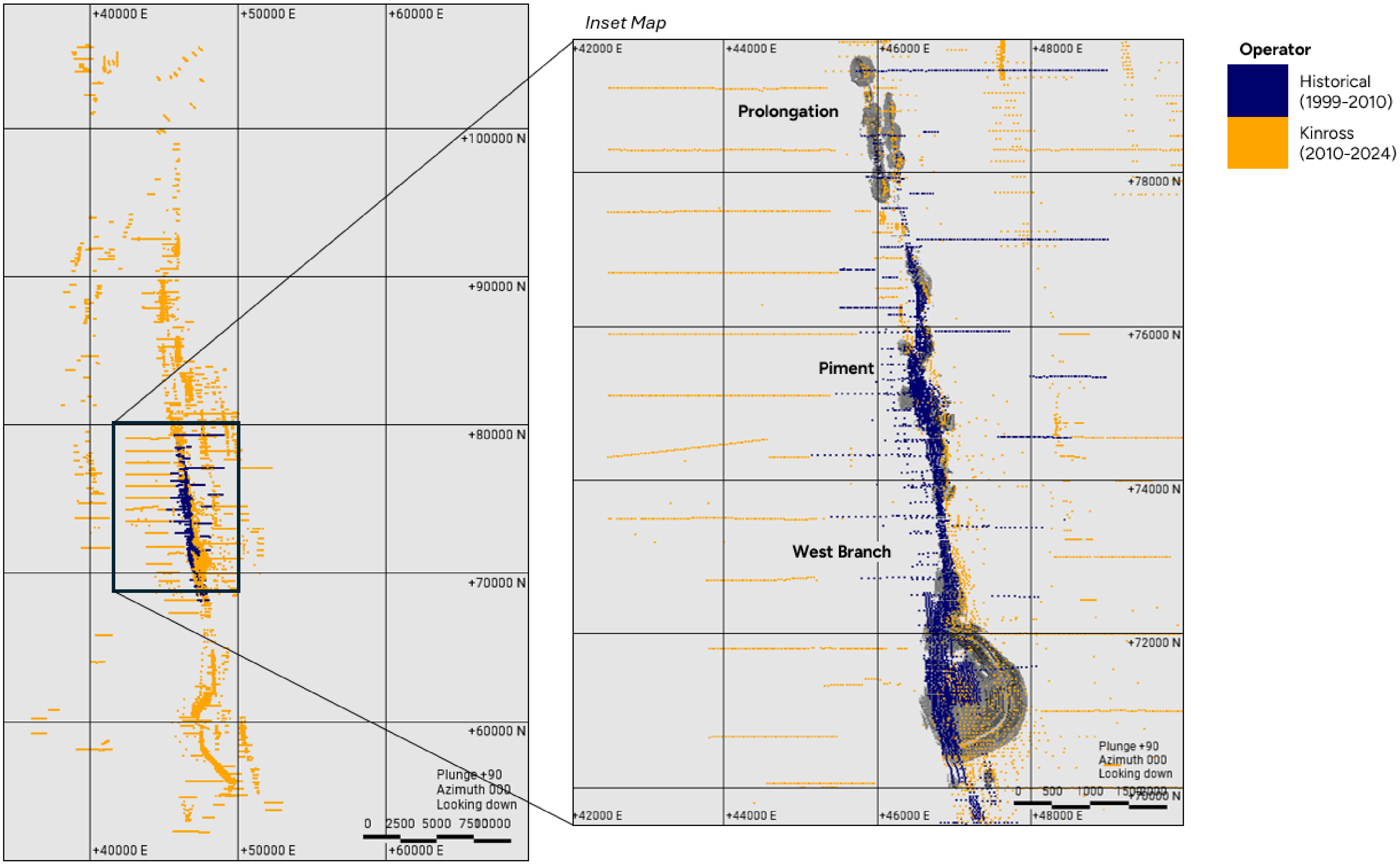


Figure 10-1: Drill hole collar map

10.2 Historical Drilling

Normandy LaSource Development Ltd. (1999-2000)

Between 1999 and 2000, NLSD completed 339 RC holes for 28,039.69 m (including nine RAB holes) and 46 diamond core holes for 5,396.27 m at the Piment deposit area. Drilling was initially undertaken along 200 m spaced east-west sections at 50 m hole spacing, to 50 m to 100 m depth. Drilling methods were predominantly RC with lesser core drilling (HQ, 63.5 mm core diameter) that included RC pre-collars with diamond core tails (NQ, 47.6 mm diameter core).

Newmont Mining Corporation (2001-2002)

No drilling was completed by Newmont during the period that it held the Tasiast property (as part of its acquisition of Normandy).

Midas Gold PLC (2003)

From March to April 2003, Midas drilled 84 RC holes for 7,898 m and 29 diamond drill holes for 2,908.4 m at the Piment deposit area. Diamond drilling was completed mainly with HQ3 core diameter (61.1 mm) and PQ3 core diameter (83 mm) for seven geotechnical holes and three metallurgical holes. In addition, four RC pre-collars with diamond core tails for a total of 1,417.2 m were completed to test down-dip extensions.

Defiance Mining Corporation (2003-2004)

Defiance completed 225 RC drill holes for 19,121 m at the Piment deposit area. The drilling program primarily focused on infilling existing NLSD drill holes along 25 m spaced, east-west drill fences.

Rio Narcea Gold Mines (2004-2007)

Between 2004 and 2007, Rio Narcea completed 246 holes for 24,024 m mostly aimed at extending the Piment deposit northwards towards what is now referred to as Prolongation. Additionally, Ro Narcea completed sterilization drilling over planned waste dumps and tailings storage facility (TSF).

Red Back Mining Inc (2007-2010)

Following the acquisition of Tasiast in 2007, Red Back commenced an infill program of RC drilling to fully define and grow the mineral resources at and around the Piment deposit. In early 2009, step back drilling to the south of Piment discovered what is now known as the West Branch deposit. For the remainder of 2009 and into 2010, RC drilling was ramped up to test the resource potential of West Branch. In late 2009, diamond drilling was increased added to the continuation of West Branch mineralization beyond the depth penetration limits of the RC rigs. A small RC rig was used to conduct shallow (40 m) RC drilling on district targets along reconnaissance style fences. In total, Red Back completed 5,857 drill holes for 522,844 m.

10.3 Recent Drilling (2010–2024)

Upon closing of the Red Back acquisition in 2010, Kinross further accelerated drilling activities and by 2011 a total of 23 drill rigs were operating on site. From 2010 to 2012, drilling was primarily aimed at resource and reserve growth of the West Branch deposit. In addition, drilling activities to support geotechnical, hydrogeological, and metallurgical studies were completed . From 2013 to 2015 drilling shifted focus to the northern licenses (Tmeimichat and Imkebden exploration permits) with an aim to define resources that could be used in a study to support the conversion of both permit types from exploration to exploitation. From 2016 to 20124 drilling was refocused back on the Guelb El Ghaîcha mining permit and continued to test near-mine exploration targets. In total, Kinross has completed 9,123 drill holes for 1,113,775 m (by length this equates to 71% RC holes, 17% DD holes, and 12% combination RC-DD holes).

10.4 Logging Procedures

For the Red Back and Kinross RC drill programs, a geologist first described (logged) the drill cuttings (chips) and then placed a representative sample into pre-labelled plastic RC chip boxes. The logging data was recorded directly in digital format at the rigs into the database system. Data recorded included drill hole ID; sample number and depth; oxidation state; colour; presence of water; lithology; texture; structure; alteration; presence and type of quartz carbonate veining; and presence, type, and abundance of sulphide minerals. Prior to 2009, diamond core logging geologists recorded geological and geotechnical descriptions on separate, hard copy log sheets and then input to Microsoft Excel files.

In 2009, the diamond core logging methodology was converted to the current system of digitally recording geological information via a notebook or tough book computer into a Fusion database which was replaced with AcQuire in 2018. Diamond core logging collected rock quality designation (RQD), lithology, oxidation, alteration, sulphide mineralogy, structure, and veining. All diamond core holes have been photographed in the entirety, with digital camera and are stored in site server.

10.5 Collar Surveys

Pre-Red Back, drill collars were surveyed upon completion, using a Geodimeter 510 total station instrument. During the Red Back and Kinross drilling programs between 2006 to 2012, drill hole collars were surveyed immediately after completing the holes or later, initially with electronic distance measurement (EDM) and differential GPS. Once completed, the Cartesian coordinates were digitally recorded and emailed to the database manager to be imported into the database. Starting in 2013, the survey data was imported directly into the database. From 2013 to date, exploration drill collars are surveyed exclusively with a differential GPS, operated by trained staff with oversight by the Tasiast Survey department. The drill collar locations are collected in the local grid system (Section 9.1) and includes a comprehensive array of permanent and semi-permanent survey stations, which have been checked for internal consistency by numerous EDM traverse closures and numerous comparisons with differential GPS data. Kinross completed an internal audit (re-survey) in 2013 using a differential GPS with 87% of all the project holes identified and validated.

10.6 Downhole Surveys

Prior to 2010, most of the drilling was completed by shallow RC and did not include downhole survey due to the complexities of surveying RC drill holes. Where diamond core drilling was completed (typically in deeper drill holes) and in selective cases of RC drilling, Humphries gyroscope, Maxibore, and Reflex single shot downhole survey tools were used. From 2010 to 2013 Kinross used three different contractors to complete downhole surveys: ABIM solution in 2010 using SPT004 NS GYRO (measurements were taken every five metres), WELL FORCE International from 2010 to 2012 using Gyro and MEMS (measurements were taken every 10 m), and SEMM Logging from 2011 to 2013 using SPT Gyro 07 and SPT Gyro 109 (measurements were taken every five metres). From 2014 to 2017, the downhole surveys were completed by trained Kinross staff using MEMS and North Seeking Gyro 103 (measurements were taken every 10 m). In 2018, downhole surveys were completed with Reflex EZ Gyro, operated by drilling companies (measurements were taken every 10 m for core holes and 24 m for RC holes). Considering the complete project database; 60% of all drill holes have downhole survey data (58% of RC, 99% of RC-DD combination, and 77% of DD holes).

10.7 Recovery

Prior to 2013 total RC sample weights were not collected routinely, however, based on selective available data, RC recoveries were determined to be acceptable. From 2013 onwards, total sample weights were routinely collected and confirm that recoveries are good. Recovery data for diamond core holes was collected from all Red Back and Kinross drill programs. Based on 17,718 measurements, the average total recovery from core runs (in both oxide and fresh) is 98% and the RQD is greater than 93%. Measurements from downhole depths below 50 m (approximate oxide-fresh boundary) returned values of 99% and 95% for total recovery and RQD, respectively, in comparison to shallower depths where total recovery is 87% and RQD averages 43%.

10.8 Drill Hole Orientation

Both the Piment and West Branch deposits dip eastward at moderate angles (approximately 40° to 60°). In consideration of the deposit geometry, Exploration and Resource definition drilling at Piment and West Branch was inclined at approximately 60° towards azimuth 270° (drilling east to west). The Piment and shallow portion of

West Branch deposits were initially drilled along 50 m spaced sections at approximately 50 m drill hole spacing. Infill drilling was later completed along 25 m sections to approximately 25m drill hole spacing. Deeper drilling (down dip and down plunge at West Branch) was completed along 50 m spaced sections with approximately 75 m hole spacing.

10.9 Geotechnical, Hydrogeological, and Metallurgical Drilling

Geomechanical drilling campaigns for collection of rock mass characteristics and groundwater conditions were conducted across El Ghaicha to identify potentially suitable locations for mine infrastructure. The various geotechnical studies completed, based upon the progress of pit development, include the following:

- Golder Associates (2004): This study provided preliminary pit slope design parameters for the oxide zone and fresh zone. Results were based on the analysis of geotechnical data collected in 2003 and were used in the preliminary mine planning and development of the project. The study focused on the BIF lithology and four open pits were proposed within this lithological unit. Rock testing was confined to this unit, and the drill hole data was subject to a strong directional bias with almost all boreholes drilled with 60° dip and 270° azimuths.

- Scott Wilson (2008a,b,c; 2009): This study provided slope stability analysis and ultimate pit slope design parameters of the four open pits (Piment North, Central, South-North, and South-South) for the oxide zone and fresh zone. Results were based on the analysis of the geotechnical data collected in 2008, which was based on an orthogonally oriented drill hole program. Geotechnical and discontinuity data were collected and processed to form eleven pit sectors based on geotechnical characteristics. The study also included a review of the seismicity and hydrogeology of the Piment site. The 2008 investigation also focused on logging the wider variety of lithologies and structural features encountered within the four open pits. Laboratory and field rock strength testing was also undertaken on representative samples to establish base design values. Each pit sector was compared to the discontinuity sets to identify kinematically feasible modes of failure. Slope designs were done for a base pit, as defined by the “$700 (ultimate) pit shell” provided by TMLSA. A series of recommendations included overall slope angle, bench stack angles, inter-ramp angles, and the structural and bedding controls based on operational assumptions adopted for safe operation.

- Scott Wilson (2011): This study provided slope stability analysis and ultimate pit slope design parameters for the West Branch area. Results were based on the analysis of geotechnical data collected in 2010, which was based on an orthogonal oriented drill hole program. Geotechnical and discontinuity data were collected and processed to form four broad pit sectors based on geotechnical characteristics. The collected data was based on the configuration of drill holes of the West Branch proposed pit shell. The focus of the study was on logging a wider variety of lithologies and structural features encountered within the open pit footprint, based on experiences from the study conducted in 2008. Laboratory and field rock strength testing was also undertaken on representative samples to establish base design values. Each pit sector was compared to the collected discontinuity sets to identify kinematically feasible modes of failure. Slope designs were performed for a base of pit, as defined by a 30 kt/d CIL pit shell. A series of recommendations included overall slope angle, bench stack angles, inter-ramp angles, and the structural and bedding controls based on operational assumptions adopted for safe operation.

- Scott Wilson (2011): This study provided a slope stability analysis and refined the ultimate pit slope design parameters for the West Branch area. Results were based on analysis of the geotechnical data collected in 2010 and 2011. A total of 23 geotechnical and six hydrogeological drill holes were completed. Two pits at West Branch were designed (North and South pits). The northern pit assumes a 700 m pit depth, whereas the southern pit assumes a shallower depth.
- Golder Associates (2014) and Schlumberger (2014): A geotechnical and hydrogeological drilling program for the West Branch pit was completed from March to June of 2013 (eight holes at 4,612 m). The purpose of the drilling and investigation program was to provide additional geotechnical and hydrogeological data where data was lacking and to complement those data that currently exist from previous investigation campaigns.

Hydrogeological drilling was conducted in the area of the water bore field, to identify sufficient water for processing. Large diameter core holes (typically PQ) were completed to collect samples for metallurgical test work.

10.10 Comments on Drill Programs

In the opinion of the QP, the quantity and quality of the lithological, geotechnical, collar, and downhole survey data collected in exploration and infill drill programs are sufficient to support mineral resource and mineral reserve estimation as follows:

- Core logging meets industry standards for gold exploration.
- Collar surveys have been performed using industry-standard instrumentation.
- Downhole surveys have been performed using industry-standard instrumentation.
- Recovery data from core drill programs are acceptable.
- Geotechnical logging of drill core meets industry standards for open pit operations.

- Drilling is normally perpendicular to the strike of the mineralization. Depending on the dip of the drill hole and the dip of the mineralization, drill intercept widths are typically greater than true widths.
- Drill orientations for Tasiast are appropriate for the mineralization style and have been drilled at orientations that are optimal for the orientation of mineralization for the bulk of the deposit area.

11. SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Sampling Method and Approach

Project staff of the operator at the time were typically responsible for the following:

- Sample collection
- Core splitting
- Delivery of samples to the analytical laboratory
- Density (specific gravity) determinations
- Sample storage
- Sample security

Geochemical and Trench Sampling

As the geochemical and trench data have been superseded by information from drilling and mining operations, these sample types are not discussed further in the Technical Report. This information is not relied on for use in geological modelling or resource estimations.

Normandy LaSource Development Ltd.

Little information has been kept or is available regarding drilling procedures used in drilling completed by NLSD.

Defiance Mining Corporation and Rio Narcea Gold Mines

<u>RC Drill Holes</u>

All RC holes were sampled at one-metre intervals and each sample was collected in a large plastic sample bag that was held below the cyclone spigot by a drill helper. All samples were sent for assay except those that originated from the non-mineralized hanging wall at the start of each hole. To avoid sample contamination, after a drill run was completed, blow-backs were carried out at the end of each 6.0 m run by the driller whereby the percussion bit was lifted off the bottom of the hole and the hole was blown clean. When water was encountered in the hole, the driller would dry out the hole by increasing air pressure into the hole and lifting and lowering the rods prior to continuing the drilling.

Throughout the Defiance RC drill program, logging of all RC drill holes was conducted by the field geologist at the drill site. After each drilled 1.0 m interval, the sample was weighed, sieved, and split to give a two kilogram to three kilogram sample for analysis.

A representative subsample for geological logging was collected from the large sample bag by spearing a small diameter PVC pipe into the bag and emptying the contents of the PVC pipe into a hand sieve.

At the end of each day or at the completion of a RC hole, calico sample bags for RC drill holes completed that day were loaded onto a 4x4 pick-up truck by the field geologist and then delivered directly to the on-site sample preparation laboratory. Once the samples were unloaded from the pick-up truck and both the field geologist and laboratory technician confirmed receipt of all calico sample bags, the field geologist then registered the sample number sequence in the database.

Diamond Drill Holes

Upon completion of geological and geotechnical core logging of a diamond drill hole, Defiance's core logging geologist identified the sections of core to be sampled and analyzed for gold. Once identified, the core logging geologist measured and marked out the sample intervals onto the uncut core down the right-hand side of the orientation line. Individual sample intervals were recorded onto a core sampling sheet. The core was sampled according to lithological boundaries and vein widths, and the maximum sample interval did not exceed 1.50 m in length.

At the core cutting facility, the drill core boxes were stacked in ascending order to avoid sampling mix-ups. The core was cut on the orientation line marked by the geologist and the right-hand side of the core (looking down hole) was placed in a numbered calico bag.

Once the core for a drill hole was cut and sampled, the core cutter and the core logging geologist then delivered the samples, with the core sampling sheet, to the preparation laboratory technician for sample preparation.

Red Back Mining Inc and TMLSA

RC Drill Holes

To minimize down-the-hole deviation, RC drilling is conducted with contract single and multi-purpose rigs using a standard 5½" face sampling hammer leading a 4½" rod string.

The entire sample is collected in a large plastic bag tightly clamped onto the cyclone base. The entire length of each RC hole is sampled. A one-metre sample length is used in all holes. Dry samples, of nominal 20 kg to 25 kg weight, are reduced in size by riffle splitting using a three stage Jones riffle splitter to approximately three kilograms to four kilograms and then placed in pre-numbered sample bags for dispatch to the assay laboratory. A record is made at the drill site of the sample identity numbers and corresponding intervals, and this is also recorded in the geological log.

After September 2013, RC samples with a nominal weight from 36 kg to 40 kg (each one metre) were collected in a large plastic bag, tightly clamped onto the cyclone base and reduced in the field by 50/50 manual riffle splitters. Approximately six kilogram to eight kilogram weight samples were placed into pre-numbered sample bags to dispatch to the laboratory. Every 20 samples, a field duplicate was collected as part of the quality assurance/quality control (QA/QC) procedure.

Diamond Drill Holes

For diamond drilling, core was transported from the drill rigs to the core facility where geological and geotechnical core logging was completed. The geologist marked one-metre intervals and orientation lines (bottom of hole) along the core axis for core cutting. A record was made at the core facility of the sample identity numbers and corresponding intervals. At the core cutting facility the drill core boxes were stacked in ascending order to avoid sampling mix-ups. The core was cut on the metre and orientation lines and the left-hand side of the core looking downhole was placed in a numbered plastic bag with sample ticket.

Once the core for a drill hole was cut, sampled, and bag sealed, the core was then stored in a secure area (either locked 40 ft shipping container or fenced off area) for sample dispatch.

All the sampling processes for RC and diamond drilling were handled under TMLSA's chain of custody.

Density/Specific Gravity Measurement

The results from 1,699 bulk density determinations completed by NLSD at Tasiast during previous drilling programs are available. The origin of the sample, its borehole number, and sample depth were entered as an individual MS Access database file into NLSD's project database. Information on the sample size/length, lithology, and oxidation state was not recorded in the NLSD database. The bulk density measurement for each NLSD sample was derived by using the Weight in Air/Weight in Water (Archimedes) method. The oxidized core samples were sealed with molten wax and reweighed to determine the weight of the paraffin coating, prior to weighing in water. The bulk density determinations were done on short (five centimetre), half core specimens, taken at close intervals. The NLSD bulk density data were collected from one core hole in the Piment South area and from 13 core holes from the Piment Central area.

A total of 131 bulk density measurements were carried out on lengths of complete drill core by Defiance during their programs. Density determinations were undertaken prior to core sawing on 131 samples of approximately eight centimetres to 15 cm in length and of both HQ and HQ3 diameter. The water displacement method was used.

From 2008 to December 2011, Red Back and TMLSA completed 24,702 specific gravity determinations of bulk density using the Archimedes method. The samples were selected to provide a representative suite of densities covering all major lithology types and from all oxidation levels.

Initial Red Back and TMLSA density determinations were done using wax-coated samples for both oxide and primary material. This procedure was changed by using uncoated core samples for only primary material to speed up the test work. Duplicate tests with one-wax coated samples for every lithology per hole were done to evaluate bias between the data pairs. Approximately 650 duplicate tests were done up to December 2011. Initial analysis of the check samples showed very good correlation between the uncoated and coated density values. A total of 90% of the dataset shows a difference of 1% variability between the sample pairs (coated and uncoated).

11.2 Analytical Laboratories

Sample preparation was undertaken on site by NLSD staff during their drill programs. Analytical laboratories used were the BRGM laboratory in Orleans, France and OMAC Laboratories Ltd. (OMAC) in Ireland. QA/QC was undertaken by Genalysis Laboratories (Genalysis) in Perth, Australia, and SGS Laboratories in France. Laboratory accreditations at the time are not known; all analytical laboratories were independent of NLSD.

During Defiance's RC and diamond drill hole programs, the analytical work was carried out by SGS Analab in Kayes, Mali and by Abilab located in Bamako, Mali. Analab is an ISO accredited laboratory whereas Abilab is not ISO accredited. The laboratories were independent of Defiance.

Following Red Back's acquisition of the Tasiast deposit in August 2007, an on-site SGS Analab sample preparation facility became operational. Prior to that time, samples had been prepared on site by the previous owners' technical crew members under supervision of senior geological staff. All drill samples since 2007 have been prepared and analyzed under contract by SGS on site and by SGS Analab in Kayes, Mali, SGS Analab in Morila, Mali, and SGS in Ouagadougou, Burkina Faso. Laboratories were independent of Red Back. The two SGS laboratories hold ISO9000 accreditations.

In December 2010, SGS constructed and commissioned a mobile sample preparation facility in Nouakchott, Mauritania, and selected samples were submitted to the facility for preparation. In late 2011, a new on-site SGS preparation and assay laboratory was commissioned at Tasiast, with a capacity of up to 2,000 samples per day. In mid-2012, TMLSA stopped sending exploration samples to the SGS Tasiast preparation laboratory due to quality control concerns. Due to the large volumes of samples and turnaround time issues, TMLSA started sending samples to nine different accredited laboratories outside the country for sample preparation and assays. These were Actlabs Burkina Faso (ISO 9001), ALS Johannesburg (ISO 9001), ALS Kumasi (ISO 9001), ALS Loughrea (ISO 9001), ALS Romania

(ISO 9001), ALS Vancouver (ISO 9001), SGS Kayes (ISO 9001), SGS Morila (ISO 9001), and SGS Ouaga (ISO 9001). In April 2013, ALS Chemex took over the Tasiast laboratory facilities and undertook the sample preparation and analytical services. All drill hole samples have been prepared and assayed by the ALS Tasiast laboratory (ISO 9001).

11.3 Sample Preparation

Midas, Defiance, and Rio Narcea RC drill hole sample preparation included using the entire RC calico sample bag, which was oven dried for 24 hours and then weighed before pulverizing the entire two kilogram to three kilogram subsample using a Labtecnics LM5 mill. Each core sample was crushed to -10 mm in a jaw crusher, and the entire sample was pulverized to P_{90} (90% passing) at 75 µm using a Labtecnics LM5 mill. Barren dune sand was used to clean the bowls after every sample. The pulverized material was sampled using a spatula, and two 120 g pulp sub-splits were taken; one packet was prepared for shipment to the assay laboratory and one packet remained on site for future reference. Blanks of dune sand and certified reference standard were then inserted with the field samples.

Sample pulp shipments were conducted on a weekly basis. The samples were transported in secured wood boxes to Nouakchott, where Mauritanian Customs inspected the shipment and released the proper documentation for exportation. The boxes included a sample submission sheet prepared by the laboratory manager. Samples were then shipped by airfreight to SGS Analab, in Kayes.

At SGS Tasiast and SGS Nouakchott, the entire RC and core sample was oven-dried for 24 hours in a cleaned metal dish, weighed and then crushed to 75% passing at two millimetres. At SGS Tasiast, a 1.5 kg subsample was split using a Jones riffle splitter and pulverized in a Labtech Essa LM2 ring pulverizer using a two kilogram bowl to 85% passing at 75 µm. At SGS Nouakchott, the sample was split once using a Jones riffle splitter and pulverized in a Labtech Essa LM2 ring pulverizer using a two kilogram bowl to 85% passing at 75 µm. Both laboratories took a 200 g subsample for gold (Au) fire assay.

At the SGS Tasiast laboratory and relocated mobile sample preparation facility, the procedure for sample analysis remained unchanged, however, subsample size at ALS Tasiast increased to two kilograms, to improve the precision of results.

For RC and core samples processed by SGS Analab in Kayes, samples were stockpiled in a secure area within the Tasiast core facility and collected by a truck contracted by either Analab or TMLSA for shipment to Kayes. The samples were enclosed and secured in a large tarpaulin and transported directly from the site to the laboratory. The entire core or RC sample was oven dried for 24 hours and then weighed before pulverization. Samples were crushed to 75% passing two millimetres, and two 1.5 kg subsamples were split using a Jones riffle splitter and pulverized in a Labtech Essa LM2 ring pulverized using a two kilogram bowl to

85% passing at 75 µm. These two pulps were recombined before being subsampled (200 g) for an Au fire assay.

After ALS Chemex took over the laboratory facilities at Tasiast in early 2013, several changes were introduced at the sampling preparation stage, including:

- Drying the entire RC or core sample for three to four hours at 105 ºC.
- Registering the dry weight.
- Crushing the entire sample to 80% passing two millimetres. In November 2013, it was decided to increase the passing to 85% passing two millimetres. Every 20 samples must generate a preparation duplicate.
- Splitting the samples using a Jones riffle splitter to obtain one kilogram and pulverizing the samples to 85% passing 75 µm.
- Taking 200 g for analysis. Every 20 samples must generate a pulp duplicate.

11.4 Sample Analysis

For the samples processed by the BRGM laboratory on behalf of NLSD, the following methods were used:

- Roasting (77 Phase 1 samples and all Phase 2 samples).
- Total attack (hydrofluoric acid and aqua regia).
- Atomic absorption (AA) analysis, detection limit: 20 ppb Au (Phase 1) and 100 ppb Au (Phase 2).

Check analysis of 74 Phase 1 samples showed no significant variations between roasted and non-roasted samples (Guibal et al., 2003).

OMAC used the following methods on samples processed for NLSD:

- Ignition / Aqua Regia Digest / MIBK Extraction / AA on 30 g sample; detection limit: 10 ppb Au; 10% repeats.
- Fire assay (30 g sample): re-analysis of 903 mineralized samples (Phases 1 & 2) + all samples >1 g/t Au and those <1 g/t Au which were included in mineralized intersections (Phases 3 & 4); detection limit: 10 ppb Au.

All the sample pulps from the Midas, Defiance, and Rio Narcea drill programs were analyzed for gold using a 50 g fire assay with an atomic absorption spectroscopy (AAS) finish at both laboratories. The Analab 50 g fire assay/AAS method (FA50) has a lower detection limit of 0.005 g/t Au; Abilab's lower detection limit is 0.010 g/t Au.

Analab routinely ran random check assays in all batches, however, when the laboratory was notified of possible samples containing high values of gold for the core samples, Analab carried out a fire assay / AAS method, with repeats in some cases, as well as fire assay / gravimetric analysis for samples grading greater than 5.00 g/t Au. Analab also provided Defiance with its internal QA/QC data during the analysis period.

For Red Back and TMLSA samples, sample pulps were analyzed for gold using a 50 g fire assay with an AAS finish with a detection limit of 0.01 g/t Au. Results higher than five grams per tonne gold were re-analyzed by fire assay technique and gravimetric finish. In 2012, TMLSA began gravimetric finishes for gold above five grams per tonne and began screened metallic fire assays.

11.5 Independent Review Work - Quality Assurance and Quality Control

SRK Consulting (2003)

The following is summarized from (Guibal et al., 2003). Kinross has not sourced a copy of the original data (field duplicates, blanks, and standards) supporting the analysis and findings of this study, however, this work has been retained in this document for completeness, and it does indicate that historical drill programs were supported by QA/QC samples and monitoring. Kinross notes that in 2018 the Tasiast drill hole database was completely rebuilt from laboratory certificates (where available, otherwise migrated from the previous database and denoted as such) following several integrity issues encountered during migration to a new database management software. In this context, some findings in this section may not be completely accurate.

Most of the documented QA/QC cited by SRK Consulting (SRK) in 2003 on NLSD samples are related to measurements of the analytical errors through pulp duplicates, where two analytical methods (AAS and fire assaying [FA]) are compared (Guibal et al., 2003). No significant problem was detected. In early 2003, a total of 429 pulp samples, collected by staff from Midas and representing close to 10% of the mineralized samples within the wireframed resources, along with 54 standards (of values 0.5 g/t, 1.66 g/t, and 3.22 g/t Au) and 18 blanks were re-assayed by Genalysis. SRK noted that the Genalysis results compared well with the database and standards and blanks were assayed within acceptable limits.

For the Defiance and Rio Narcea drill programs, a total of 21,686 RC sample pulps, including field duplicates, blanks, and standards, and 904 diamond drill hole core sample pulps, including field duplicates, blanks, and standards, were shipped in 16 batches, of which 14 went to SGS Analab Kayes and two went to Abilab. Included within these sample batches were a total of 774 field duplicate samples, each one being a second split from a one metre interval field sample bag, and 1,136 preparation duplicates, each one being a second split from the pulverized RC and core sample at the preparation laboratory.

The analytical QA/QC program implemented by Defiance was monitored by the routine submission of commercial Certified Reference Materials (CRMs) purchased from Gannet Holdings Pty. Ltd. (Gannet) of South Perth, Western Australia. CRMs were inserted at every 20th sample and an internally prepared coarse blank sand inserted at every 10th sample within the RC and core sample stream. Field duplicates were collected by the field geologist after the completion of each RC hole and the number of field duplicates on a per RC hole basis was dependent on the length of the hole or equivalent to every 20th sample. Preparation duplicates were selected for every 20th sample number in a sequence and submitted as a separate sample number series on a per batch basis.

Heberlein Geoconsulting (2011–2013)

The following is summarized from (Kinross, 2019). Kinross has not sourced a copy of the original data (field duplicates, blanks, and standards) supporting the analysis and findings of this work, however, this work has been retained in this document for completeness, and it does provide an indication that the historical drill programs were supported by QA/QC samples and monitoring. Kinross notes that in 2018 the Tasiast drill hole database was completely rebuilt from laboratory certificates (where available, otherwise migrated from the previous database and denoted as such) following several integrity issues encountered during migration to a new database management software. In this context, some findings in this section may not be completely accurate.

In 2011, TMLSA engaged an independent consultant to provide a regular review of the QA/QC data. Issues identified during the early review in September 2011 (Heberlein, 2011), such as switched standards and standard identification, have been corrected and control actions implemented.

Additional actions implemented to address other recognized issues, such as duplicate precision, include the following:

1. Conducted routine crushing and analyzed pulverized duplicate samples at the majority of the laboratories.
2. Conducted gradual replacement of three tier-riffle splitters and cone splitters on RC rigs by 50/50 manual riffle splitters.
3. Established a dedicated group to control and monitor sampling, dispatch, and quality control analytical results.
4. Assigned a Tasiast technician permanently to the on-site Tasiast exploration preparation facility to monitor and control the workflow.
5. Conducted a routine independent-consultant review of data and laboratory audits.

6. Reviewed various sample volume and preparation methods that have resulted in larger samples (between five kilograms and 10 kg) collected from RC rigs in 2013.

TMLSA's QA/QC process before 2012 was as follows:

- A routine analytical sample 'field duplicates' were collected every 20th sample and submitted in blind sequence.
- For RC samples, a further representative triplicate sample was routinely collected every 60th original sample in the sequence, and retained for later submission to a third-party, independent referee laboratory.
- Analytical 'blanks' were inserted every 20th original sample and were taken from barren dune sand collected from a source distant from the mine.
- CRMs, in pulp form, from Gannet, Rocklabs, and Geostats Pty Ltd (Geostats) were selected based on certain resource thresholds, and inserted as standards every 20th sample.
- All QA/QC samples were inserted by the rig geologist at the rig.
- Grades of standards to be used were selected by the senior geologist and provided to the rig geologist in the rig box.
- TMLSA submitted 16% routine QC samples within the sample string.
- Holes were submitted by the rig geologist directly to the on-site laboratory as individual batch jobs, or dispatched from the site to Mali, Burkina Faso and South African laboratories.
- Started using the certified standard deviation (SD) of the mean value for the CRM (previously ±10% of mean value), rules for batch pass and failures.

In 2012, a QA/QC team was established and managed the sampling protocols, sample transport, sample tracking, and reporting.

In 2012, the quality control processes were modified as follows:

- Analytical 'blanks' were inserted after every 20th original sample and were taken from gold barren material (not sand) collected from a source distant from the mine. This material was submitted blind with the samples dispatched, as with other samples. RC blanks were crushed to simulate the RC sample grain size. Diamond drill blanks were used that resembled the diamond drill core size.
- Canadian Centre for Mineral and Energy Technology (CANMET), Rocklabs, and Geostats CRMs in pulp form were selected based on certain resource thresholds and inserted as standards every 20th sample.

- All QA/QC samples were inserted by the QA/QC team in the exploration yard, except the field duplicates that were generated at the drill. Grades of standards to be used were selected by the geologist that logged the RC or diamond drill holes, and they were provided by the QA/QC team. For every batch, TMLSA inserted 16% QC samples within the sample string. Holes were submitted by the QA/QC team directly to the on-site laboratory as individual batch jobs, or dispatched from the site to Mali, Burkina Faso, and South African laboratories.

Twice in 2013, 5% of the samples (pulp) were sent to a third laboratory to be checked.

SRK (2013)

In April 2013, SRK conducted a review of the analytical quality control procedures and results for Tasiast (Chartier, 2013). The objective of the review was to provide an independent analysis of the sampling procedures and a review of the analytical quality control results for the data to be used in resource estimation.

SRK visited the Complex site from October 11 to 15, 2013. SRK also visited a third-party preparation laboratory operated by SGS Minerals, in Nouakchott, Mauritania. The purpose of the site visit was to audit the technical data collection and processing and to collect all relevant information for the compilation of the Sample Preparation, Analyses, and Security and Data Verification sections of a technical report. SRK was given full access to relevant data and conducted interviews with Kinross personnel to obtain information on past exploration work and understand the procedures used to collect, record, and analyze historical and current exploration data.

SRK reviewed the field procedures and analytical quality control measures at Tasiast. In SRK's opinion, Kinross personnel used care in collecting and managing field and assay exploration data. The sample preparation, security, and analytical procedures used by Kinross were consistent with generally accepted industry best practices and are therefore adequate to support the mineral resource estimation.

A summary of SRK's main conclusions were:

- The sampling procedures used meet industry best practices. All borehole sampling was conducted by appropriately qualified personnel under the direct supervision of appropriately qualified geologists. The sample preparation, security, and analytical procedures were consistent with generally accepted industry best practices and are therefore adequate to support Mineral Resource estimation.

- The paired data results are consistent with results expected from gold mineralization in an epigenetic deposit that is structurally and lithologically controlled.

- Results from the CRMs are acceptable.

- The non-certified field blanks consistently returned values at or below the detection limit at most primary laboratories.
- SRK concurred with Heberlein (2013) that sample preparation procedures are failing to properly homogenize the samples. SRK attributes part of that failure to a prominent nugget effect and notes that control charts display no apparent bias between original and duplicate samples.
- Much of the analytical data informing the MRE was derived from several different unaccredited laboratories, including the mine laboratory operated by SGS.

TMLSA (2017)

As part of database migration from Fusion to acQuire, the QA/QC data from 2007 to 2017 for the Guelb El Ghaîcha area, which contains West Branch pit, all Piment and Prolongation pits, and the northern satellite deposits (C67, C68, and Fennec), were reviewed in preparation for the 2019 model update. The purpose was to validate all quality control assays available in the database since Red Back's acquisition of the project in 2007, and to ensure that only assays with acceptable quality control results are being used for resource estimation. A total of 16 different assay laboratories were involved for the sample analysis in this period. Table 11-1 presents the summary of quality control samples reviewed during this time. A total of 44,003 standard assays and a total of 12,261 blank assays were exported from the Tasiast acQuire database for review. Duplicate assays were not included because the purpose of the review was to select the assays with acceptable quality control results for the resource estimation; duplicate assays present only precision information on each sampling/subsampling stage, which does not provide the pass/fail criteria.

Table 11-1: QA/QC samples by laboratory

Laboratory	Standard	Blank	Total
Actlabs Burkina Faso	501	486	987
ALS Chemex Ouaga	675	28	703
ALS Johannesburg	1,951	1,502	3,453
ALS Kumasi	744	669	1,413
ALS Loughrea	439	387	826
ALS Nouakchott	82	84	166
ALS Romania	987	846	1,833
ALS Tasiast	2,703	2,815	5,518
ALS Vancouver	437	287	724
SGS Kayes	17,801	1,368	19,169
SGS Lakefield	5	0	5
SGS Morila	7,966	263	8,229
SGS Ouaga	1,783	99	1,882
SGS TML	7,929	3,427	11,356
Total	**44,003**	**12,261**	**56,264**

The exported standard and blank assays were grouped and assessed by Lab Job number, which is equivalent to sample shipment number. The Lab Job number may contain multiple assay batches. The pass/fail assessment was performed for 5,302 Lab Jobs. The pass/fail criteria applied were plus/minus three standard deviations (3SD) for standards and less than or equal to 0.05 g/t Au for blanks. A total of 48 different standards and two types of blanks (barren sand and barren pegmatite) were used as QC samples. Among these 48 standards, 16 historical standards that were used from 2007 to 2011 do not have any information on the actual standard code or their standard deviations. A constant standard deviation of 3.33% had been used for these historical standards in quality control checks and this caused a significant number of standards failures due to the challenging control limit of 10.0% (3 x 3.33%). A decision was made to re-assign a constant standard deviation of 6.67% for these historical standards, considering the acceptable industry practice standard deviation level of 5% to 7% for gold standard. This resulted in reducing the number of unnecessary quality control failures. The average certified standard deviation of all standards, including the historical standards, was 5.6%, which is well within the acceptable industry practice level mentioned above.

Numerous occasions of quality control sample swaps were identified, especially during 2008-2012 drilling campaigns when there were a high number of drills running simultaneously with limited trained work force. The total number of swaps identified was 906, which is equivalent to 1.6% of all quality control samples reviewed. Most of the swaps were easily identifiable in the quality control chart by the presence of cluster(s) of noticeably different grade value(s) compared to the certified value (Figure 11-1).

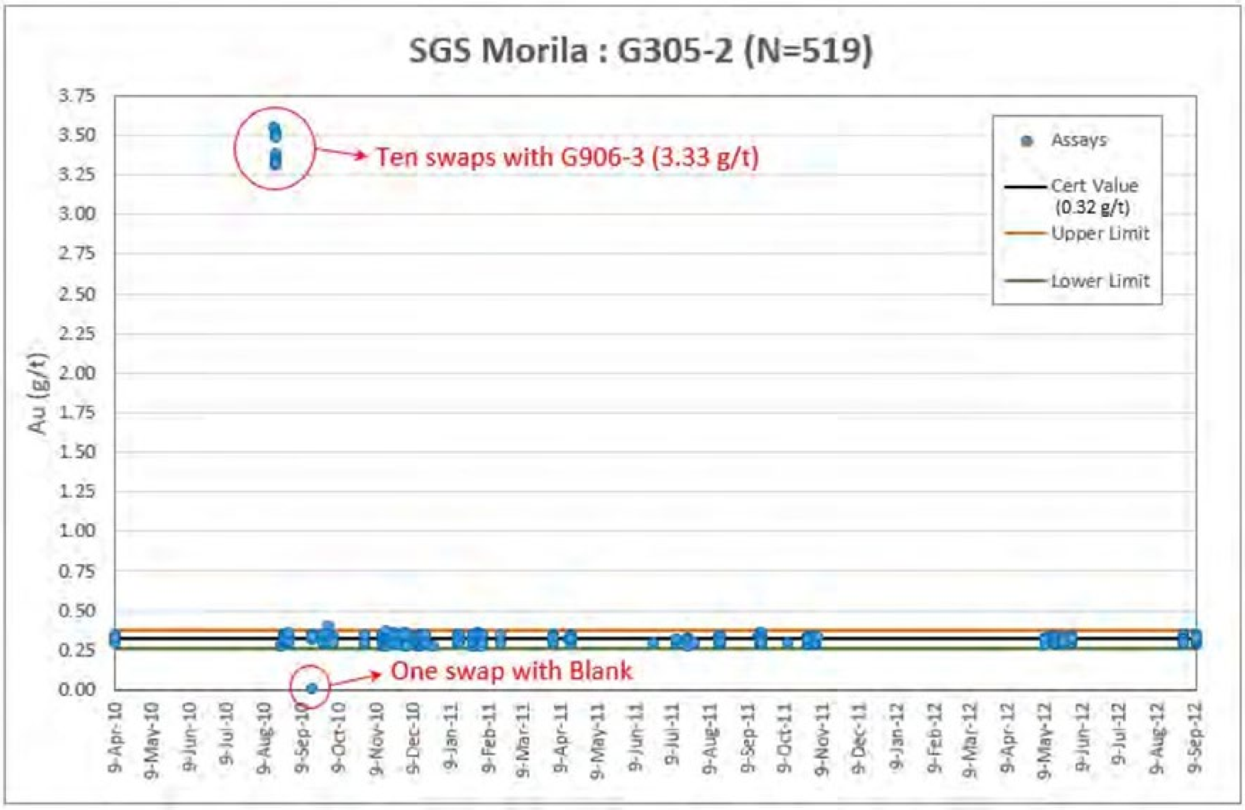


Figure 11-1: Standard control plot for Standard G305-2 from SGS Morila

The number of quality control sample swaps by year is summarized in Figure 11-2. The number of swaps has decreased to an acceptable level since the start of the 2013 drilling campaign. The identification of swaps resulted in lowering the quality control failure rate significantly.

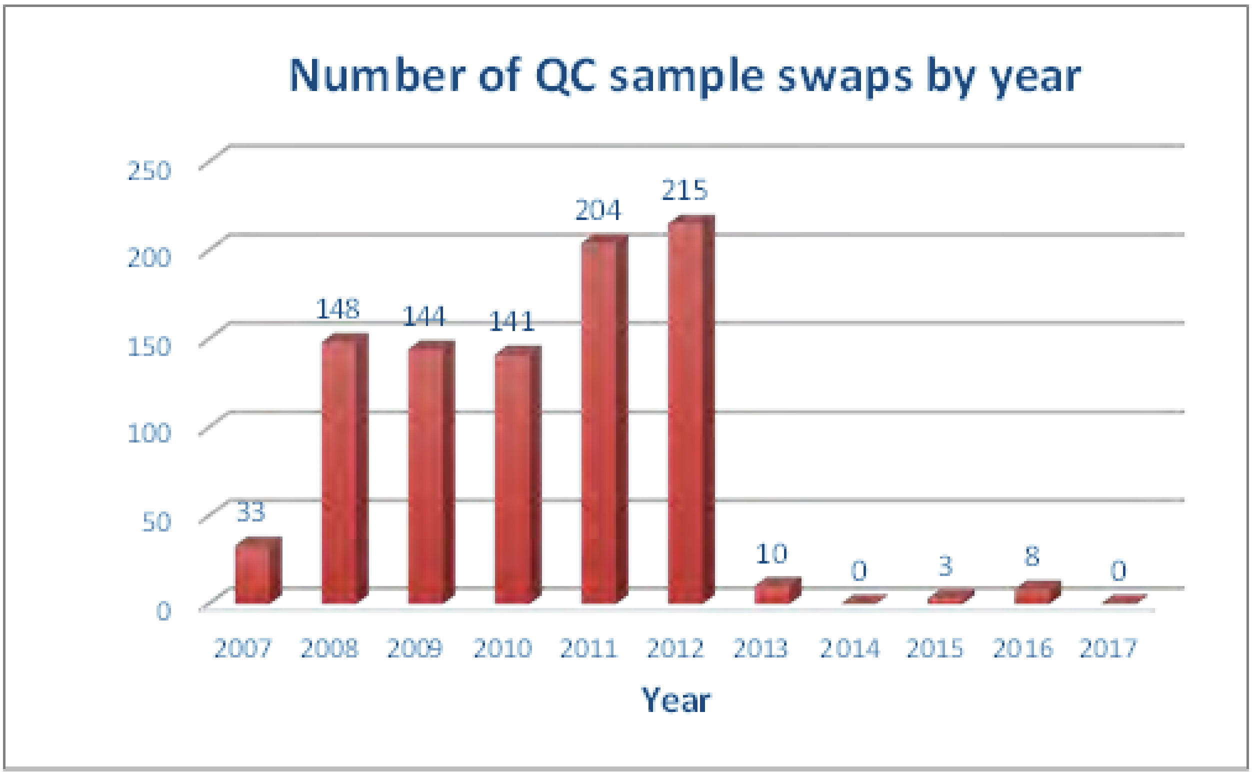


Figure 11-2: Summary of QC sample swaps by year

The standards and the blanks that failed quality control were requested to be re-analyzed where a material impact on the reported results was considered. In this event that the original pulps were unavailable, the failed assays were simply excluded from the resource estimation.

An average accuracy of 98% was achieved for the internationally accredited standards (Figure 11-3). This implies that 98% of the total standard samples submitted to the various laboratories reported within an acceptable limit of ±3SD. The blank samples submitted to the various laboratories reported similar results with 98% of the samples below 0.05 g/t Au.

Heberlein (2013) noted measurable improvement in the QA/QC procedures over the course of his involvement in the project, especially in duplicate assays (Table 11-2). The improvement has resulted in a measurable increase in the overall precision of the analytical results. Early precision estimates of field duplicate results (containing the total error of sampling, preparation, and analysis) showed unacceptably high values for both core and RC duplicate samples. The initial analysis of duplicate results in 2011 determined precisions above 85% range for drill core and above 80% range for RC chips. Improvements to sampling and sample preparation procedures, particularly at the ALS Tasiast Laboratory, have brought the duplicate precision down to the 45% range, which is reasonable for the nugget style of mineralization at Tasiast.

Table 11-2: 2007-2017 Resource QA/QC results

Laboratory	Number of Standards	Number of Blanks	% Standards Within 3SD	% Banks ≤0.05 g/t
Actlab Burkina Faso	501	486	98%	98%
ALS Chemex Ouaga	675	28	97%	100%
ALS Johannesburg	1,951	1,502	95%	94%
ALS Kumasi	744	669	90%	89%
ALS Loughrea	439	387	100%	100%
ALS Nouakchott	82	84	100%	100%
ALS Romania	987	846	100%	99%
ALS Tasiast	2,703	2,815	100%	100%
ALS Vancouver	437	287	98%	99%
SGS Kayes	17,801	1,368	99%	100%
SGS Lakefield	5	0	100%	-
SGS Morila	7,966	263	98%	89%
SGS Ouaga	1,783	99	100%	100%
SGS TML	7,929	3,427	94%	97%
Total	**44,003**	**12,261**	**98%**	**98%**

Notes:

1. SD = Standard Deviation

SLR Consulting (Canada) Ltd. (SLR) (2024)

In 2024, as part of an external audit of the Complex's Mineral Resources and Mineral Reserves, SLR received raw QA/QC data collected from 2013 to 2024 at the site. Their work and findings (SLR, 2024) are summarised below.

Blanks and CRMs were used as control samples during this period, with a total of 24,681 control samples submitted to ALS Tasiast . No duplicates were included in the drilling sample streams. Observations from SLR's review of the Kinross Tasiast QA/QC database are detailed in the following discussion.

<u>Certified Reference Materials</u>

Results of the regular submission of CRMs are used to identify potential issues with specific sample batches and long-term biases associated with the primary assay laboratory. SLR reviewed the results from 22 different types of CRM used.

A total of 12,345 commercial CRMs, sourced from either Geostats or OREAS, were inserted into streams of drilling samples and submitted to ALS Tasiast. The CRM samples were inserted across different project areas according to the distribution shown in Table 11-3. The upper and lower control limits were determined using 3SD above and below the expected value (EV).

No significant failures were observed following this criterion, and overall, good accuracy was noted for the laboratory, with biases ranging between -5.0% and 3.3%, as presented in Table 11-4. CRMs with a count smaller than five were disregarded as they are not representative.

Table 11-3: Distribution of CRMs by deposit: 2013–2024

Year	PROJECT					Grand Total
	Elgaicha	Imke	Ndaouase	Tasouth	Teme	
2013	587	238	7	138	127	**1,097**
2014	655	182	-	1,091	1,013	**2,941**
2015	864	948	-	660	1,453	**3,925**
2016	911	-	-	250	-	**1,161**
2017	284	-	-	1,850	-	**2,134**
2018	216	-	-	122	-	**338**
2020	143	-	-	2	-	**145**
2021	440	4	-	-	-	**444**
2023	79	-	-	-	-	**79**
2024	81	-	-	-	-	**81**
Grand Total	**4,260**	**1,372**	**7**	**4,113**	**2,593**	**12,345**

The CRMs cover a good range of gold grades analyzed by the FA method, however, SLR noted that seven types of CRMs were inserted in 2024 alone. SLR recommended reducing this number to a maximum of three or four material types: one approximating the cut-off grade, one or two close to the average grade, and a high-grade CRM. This reduction will be sufficient to monitor laboratory performance and track potential emerging biases or systematic failures over extended timeframes.

Table 11-4: Summary of CRM samples used in the 2013 to 2024 QA/QC programs

CRM	Year Range	Num Samples	Bias (%)	Mean	EV	SD	Num Outliers	Outliers (%)
OREAS-15f	2013-2015	2153	-1.28	0.33	0.33	0.02	8	0.37
G912-7	2014-2024	1804	-0.75	0.42	0.42	0.02	10	0.55
G910-2	2014-2024	1766	1.15	0.91	0.9	0.05	20	1.13
OREAS-16a	2014-2024	345	0.03	1.81	1.81	0.06	8	2.32
G908-3	2014-2018	1302	0.9	1.04	1.03	0.05	10	0.77
G307-6	2013-2014	304	-2.34	1.04	1.07	0.05	2	0.66
G300-9	2013-2021	783	-1.04	1.51	1.53	0.06	11	1.4
G911-4	2013-2024	288	-0.44	2.42	2.43	0.09	7	2.43
G910-7	2013-2015	1100	-1.25	0.5	0.51	0.03	7	0.64
OREAS-66a	2014-2018	445	-0.01	1.24	1.24	0.05	4	0.9
GLG914-4	2015-2021	766	3.32	0.38	0.37	0.02	11	1.44
G910-3	2013-2024	156	-3.09	3.9	4.02	0.17	7	4.49
G308-4	2013-2021	14	-4.97	6.43	6.77	0.29	1	7.14
G302-7	2014-2014	1	8.88	2.33	2.14	0.09	0	0
OREAS-251	2015-2016	115	2.81	0.52	0.5	0.02	19	16.52
OREAS-203	2015-2016	85	1.53	0.88	0.87	0.03	1	1.18
G305-2	2014-2014	6	-1.56	0.32	0.32	0.02	0	0
G305-4	2014-2014	9	-3.38	4.04	4.18	0.15	1	11.11
G310-6	2015-2024	814	-2.57	0.63	0.65	0.04	7	0.86
G316-2	2018-2024	82	-1.17	1.03	1.04	0.04	1	1.22
OREAS 67a	2021-2021	3	2.62	2.3	2.24	0.1	0	0
OREAS-19a	2021-2021	4	5.83	5.81	5.49	0.1	2	50

Notes:

1. Au in ppm
2. EV = Expected Value
3. SD = Standard Deviation

Overall, the CRM results indicate good performance, consistent with the expected values. All CRMs were initially reviewed for overall performance using z-score plots, which included all CRM series (Geostats and OREAS). Occasional mislabeling was observed. As shown in Figure 11-3, there appears to be a reduction in the number of outliers and an improvement in results from 2023 onwards, with a slight negative bias observed in the CRMs during 2024. The number of outliers, however, does not significantly affect the overall assessment of the CRMs.

Figure 11-3: Tasiast CRM Z-Score

SLR selected three CRMs for an in-depth review, representing the low, average, and high gold grade ranges. These were selected based on their sample quantity and extended periods of use.

Figure 11-4 to Figure 11-6 present the results for 1,804 samples of G912-7, 345 samples of OREAS-16a, and 156 samples of G910-3. The CRM G912-7 exhibits an acceptable bias of -0.75%, with 10 samples falling outside the mean ±3SD threshold. Four of these samples are likely mislabeled, with three showing values near 2.9 g/t Au and one at 0.18 g/t Au. The CRM OREAS-16a demonstrates a bias of 0.0%, with eight failures, all close to the threshold. The CRM G910-3 shows good scatter levels with a negative bias of -3.1%, including seven failures slightly below the lower limit and one potential mislabeling at 2.57 g/t Au.

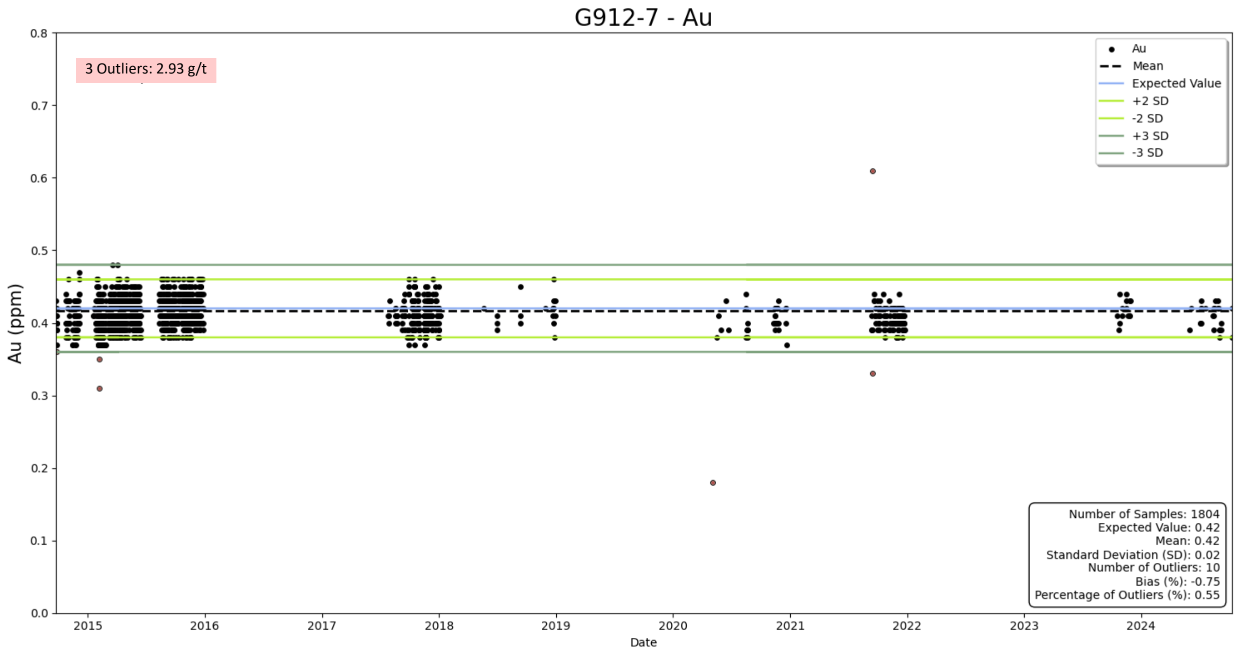


Figure 11-4: Control chart of CRM G912-7 for gold in ALS: 2015–2024

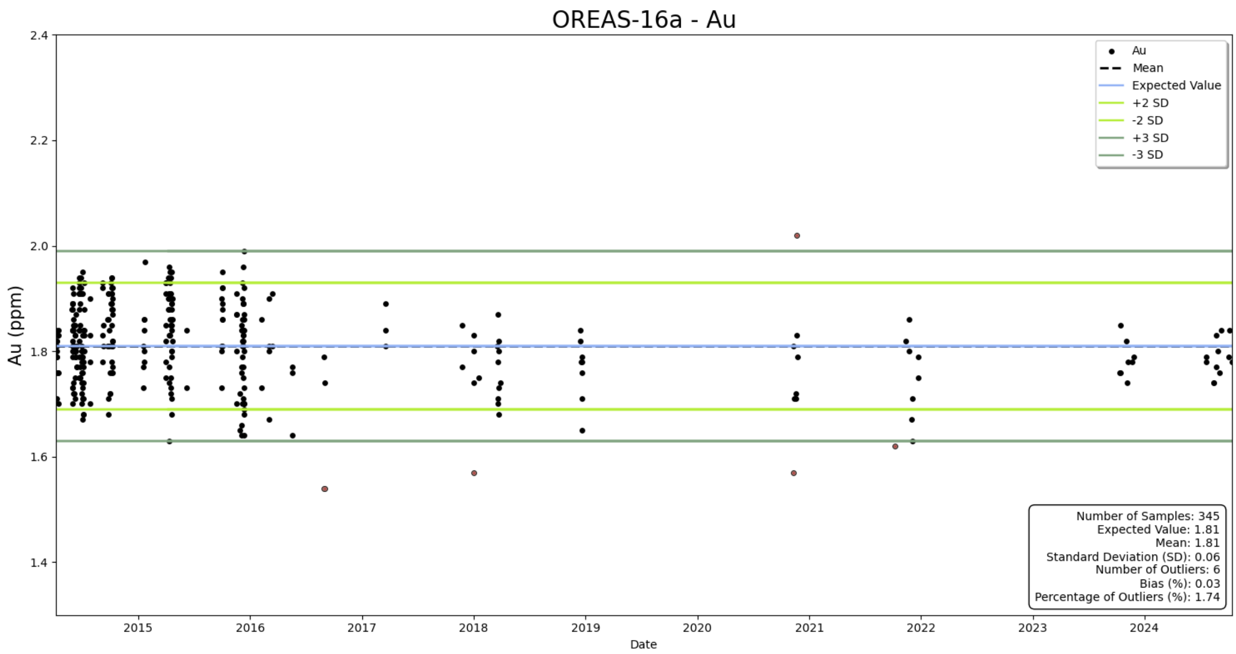


Figure 11-5: Control chart of CRM OREAS-16a for gold in ALS: 201 –2024

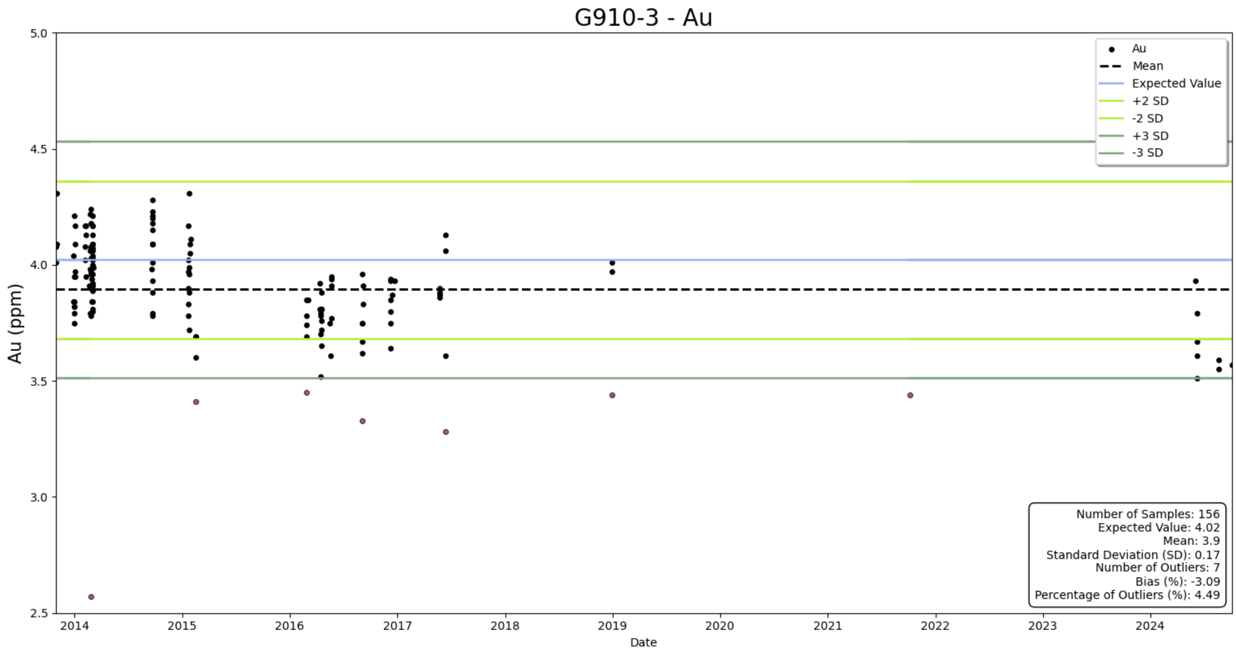


Figure 11-6: Control chart of CRM G910-3 for gold in ALS: 2014–2024

SLR recommended continuous monitoring of the CRM data to ensure early detection of potential emerging biases that may require re-analysis, to promptly identify and rectify any biases that could affect the reliability of the results, and to check and correct any mislabels in the dataset.

<u>Blank Material</u>

The regular submission of blank material is used to assess contamination, either during sample preparation or analyses, and to identify sample numbering errors. Coarse blanks consisted of barren rock. Each blank sample was placed into a plastic sample bag and assigned a unique identification number. These blanks were inserted into the sample stream the and underwent the same sample preparation and analytical procedures as the rest of the drill samples.

A total of 12,336 samples were submitted to ALS Tasiast between 2013 and 2024. These samples originate from different deposit areas, as shown in Table 11-5. Blank assay results exceeding 10 times the detection limit (0.005 g/t Au) are classified as failures. A review of the coarse blanks submitted to ALS Tasiast indicates no significant contamination during the preparation stage. A total of 40 blank samples, or 0.3% of the total samples, exceeded the acceptable limit. Among these, the highest gold grades were reported in 2014 but did not exceed 0.5 g/t Au (see Figure 11-7).

Table 11-5: Insertion of blank samples by deposit: 2013–2024

Year	PROJECT					Grand Total
	Elgaicha	Imke	Ndaouase	Tasouth	Teme	
2013	647	246	9	143	127	**1,172**
2014	696	186	-	1,172	1,083	**3,137**
2015	8	924	-	480	1,348	**2,760**
2016	946	-		268	-	**1,214**
2017	328	-	-	2,023	-	**2,351**
2018	237	-	-	188	-	**425**
2020	154	-	-	-	-	**154**
2021	960	-	-	-	-	**960**
2023	81	-	-	-	-	**81**
2024	82	-	-	-	-	**82**
Grand Total	**4,139**	**1,356**	**9**	**4,274**	**2,558**	**12,336**

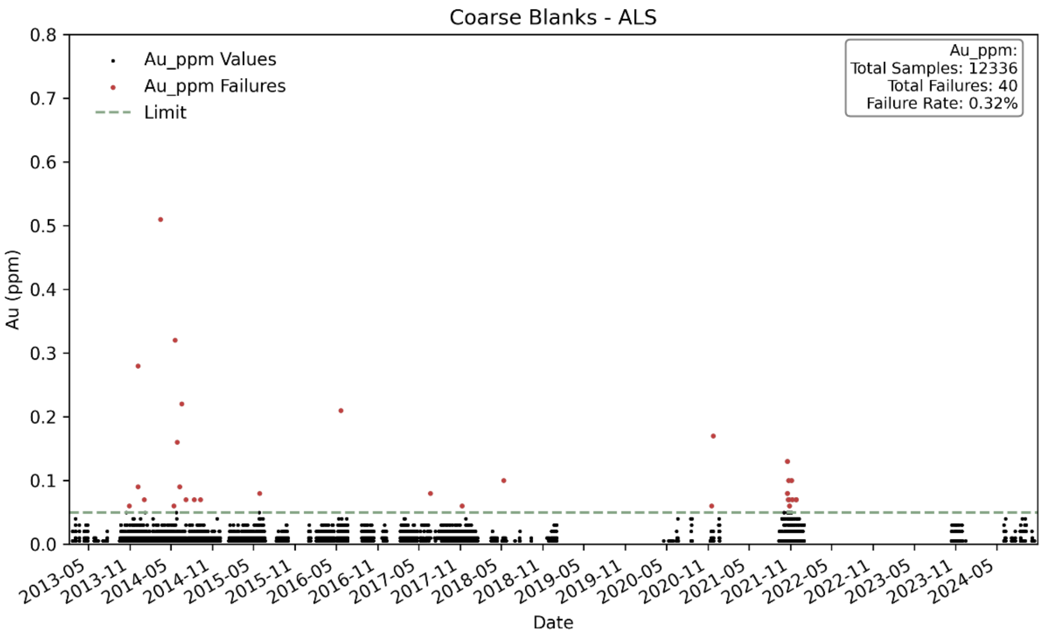


Figure 11-7: 2013 - 2024 results of coarse blank samples in ALS

Conclusions and Recommendations

Based on the review of data spanning from 2013 to 2024, the conclusions and recommendations from SLR are as follows:

- No significant contamination occurrences were identified during sample preparation at ALS Tasiast. Review the highest grade samples to determine if they may be mislabeled and make the necessary corrections in the database, always maintaining a record of the changes made.
- CRMs demonstrated good performance, maintaining a bias within ±5% and control limits set at ±3SD from the expected value. Only a few isolated mislabeling cases were observed. Review and correct these samples in the database.
- Reduce the number of CRM types to three or four: one close to the cut-off grade, one or two close to the average grade, and a high-grade CRM. This approach is sufficient to effectively monitor laboratory performance and track potential emerging biases or systematic failures over extended periods.
- Implement a duplicate sample program to enhance monitoring of grade variability during sampling, preparation, and assaying.
- Implement a check assay program to a tertiary laboratory to validate the reproducibility of gold values from the primary laboratory (ALS).
- Based on the Tasiast QA/QC program results, the overall precision and accuracy of the current gold assays are acceptable and sufficient for inclusion in an MRE.

11.6 Data and Sample Security

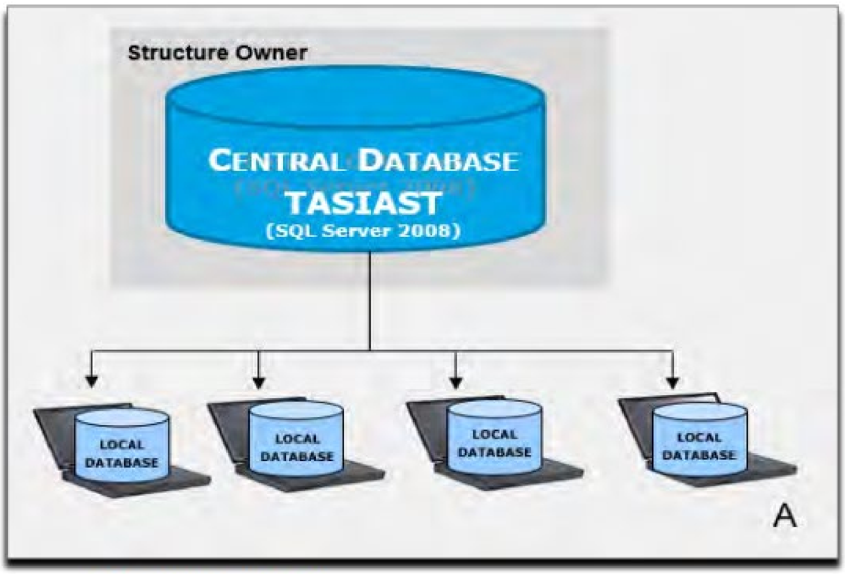
Database

All drill hole and geological data at Tasiast have been collected and stored using a variety of software and systems. Limited information exists regarding the way the data was stored prior to Red Back, however, data analysis suggests that the data was collected using spreadsheets and then imported into the 3D software that was used at the time. During this process much of the information regarding assays, such as laboratories and methods used, was lost.

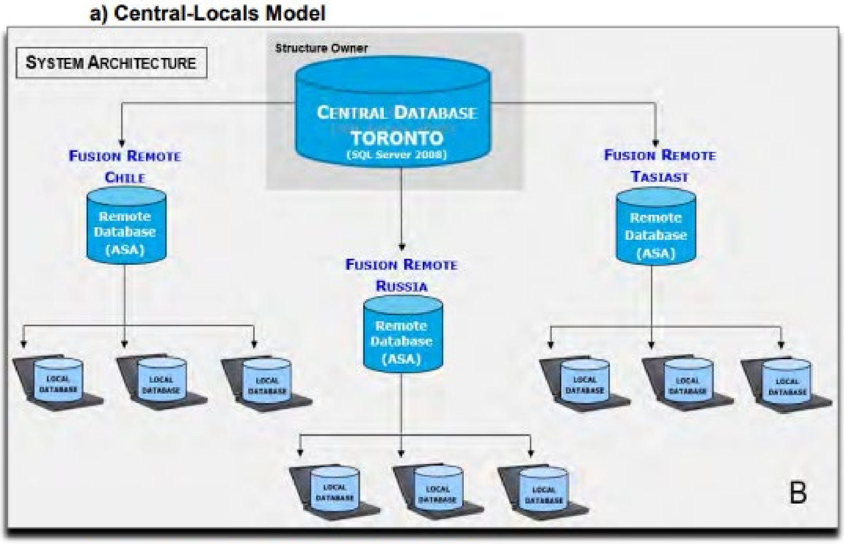
In 2007, Red Back implemented a Relational Database Management System called Fusion, which was developed by Century Systems and later sold to Datamine.

After Kinross acquired Red Back Mining in September 2010, Kinross continued using Fusion. In 2012, Kinross migrated the Tasiast System Architecture to the Central – Fusion Remote – Locals Model, which had already been implemented by Kinross globally. The Central Database (structure owner) was located in Toronto, Canada.

The sites (Brazil, Chile, Tasiast, Russia) hosted in their local server the Remotes, which were the data owners. In Tasiast, Exploration and Grade Control shared the same Remote database. In 2014, Technical Services Tasiast separated Exploration and Grade Control data into two different databases: The exploration team continued to utilize the Fusion Remote architecture as outlined as in Figure 11-8b and the grade control team went back to utilizing Central Database (which was used prior to 2012) architecture as outlined in Figure 11-8a, with the Central being located in Tasiast.



a) Central-Locals Model



b) Central-Fusion Remote-Locals Model

Source: Sims 2019.

Figure 11-8: Representation of different system architectures for Fusion

In January 2018, Kinross replaced Fusion with acQuire, another Relational Database Management System, based in structured query language (SQL). The Exploration Fusion Remote Database was disconnected from the Central database in Toronto and both Fusion databases at Tasiast (Grade Control and Exploration) were migrated and combined into a single database in acQuire. Since acQuire has more robust business rules than Fusion, when there were conflicting data (e.g., orphan samples, two records in the assay table for the same interval and method), the conflicting records were not migrated.

Assay Database Rebuild

In October 2018, during a validation process of the assay records in acQuire, some issues that compromised the integrity of the data were encountered. The causes of the issues had different origins that had not been previously detected. Some of the issues found are listed below:

- Laboratory reports used the wrong Lab Job number in the import file. Since this information was used for QA/QC purposes, as explained in Section 12.2, the process could be jeopardized.
- Some laboratories reported various samples using a different sample ID than the one in the database. Previous database administrators had manually corrected the laboratory report to match the sample ID in the database instead of requesting the laboratory to reissue the import file. As many of the reports were some years old and it would be difficult to request the laboratory to re-issue, the sample ID in the database was changed to match the laboratory report and the re-import could be done properly using the original laboratory report.
- A few samples and assay results had never been imported into the database.
- There were some examples of incorrect gold values.

These issues triggered the decision to completely rebuild the assay database.

The rebuild of the assay database was done by re-importing all the laboratory certificates that were available and then performing the QA/QC to pass or reject assays using the procedure described in the previous subsections. The re-import of certificates for holes in the West Branch area was completed in March 2019, then the reimport continued with the Piment and Prolongation areas, and finished with the northern satellite deposits. For records where no laboratory certificates were found, the data was migrated from the previous database using a qualifier to denote that the data was not imported from certificates during this phase.

Sample Storage

Sample pulps are returned from the laboratory in plastic vials or sealed paper envelopes, and these are stored in sealed containers at site. The majority of historical coarse reject samples were not stored, however, TMLSA has commenced storing selected mineralized coarse reject material. The remaining half of the drill core is securely stored in stacked wooden trays referenced by hole identification number and interval length. Some core intervals have been totally sampled for metallurgical or check (umpire) sampling.

Sample Security

Following TMLSA's acquisition of the Tasiast Mine in September 2010, all drill samples collected are under direct supervision of TMLSA staff, starting at the drill rig, and remain within the custody of staff until they are delivered to ALS Tasiast or placed on contracted trucks for delivery to the Mali laboratory. Samples, including duplicates, blanks, and CRMs, are delivered daily from the drill rig to a secure storage area within the fenced Tasiast core facility.

Chain of custody procedures consist of filling out sample submittal forms that are sent to the laboratory with sample shipments to make certain that all samples are received by the laboratory.

11.7 Comment on Sample Collection, Preparation, Analysis, and Security

The QP considers the sampling methods acceptable, consistent with industry standards, and suitable for Mineral Resource and Reserve estimation as well as mine planning. This conclusion is based on the following key observations:

- **Sampling Protocols**: Data were collected in accordance with industry-standard procedures, ensuring reliable sample integrity.
- **Density/Specific Gravity**: Density/specific gravity determinations followed established methods, with sufficient data to support tonnage interpolations for both mineralized zones and waste.
- **Sample Preparation**: Both RC and core samples were prepared using industry-standard procedures suitable for coarse gold deposits hosted in greenstone belts and banded iron formations.
- **QA/QC Program**: The current program included CRM and blank samples, which demonstrated no significant contamination during sample preparation at ALS Tasiast. CRMs performed within a ±5% bias, with an acceptable number of outliers, although some mislabeled samples require verification. In the QP's opinion, the QA/QC program as designed and implemented by TMLSA and Kinross is adequate and the assay results within the database are suitable for use in a Mineral Resource estimate.The QP recommends enhancing the

program by including duplicates and implementing external checks to improve grade variability monitoring and validate the reproducibility of gold assays from the primary laboratory.

- **Sample Storage and Security**: Samples were securely stored or monitored prior to dispatch, with chain-of-custody forms ensuring proper tracking and receipt at the laboratory.
- Based on these findings, the QP concludes that the sample collection, preparation, and analysis are robust, reliable, and appropriate for Mineral Resource and Reserve estimation.

12. DATA VERIFICATION

Numerous verification checks have been performed on data collected from the Complex, either in support of technical reports, or as part of the Tasiast Mine feasibility study. The QP has reviewed this work and in many cases, contributed to or overseen, these verification activities.

12.1 Verification in Support of Technical Reports

This section summarizes the verification activities completed at the Complex by external consultants.

SRK (2003)

SRK (Guibal, 2003) reviewed the data available in 2003, as part of supporting documentation for the acquisition of Midas by Geomaque, and commented:

- Although SRK was not involved in the selection and collection of the check samples, the documentation supplied by [Midas] which includes all the analytical results show that the general quality of the sampling/assaying is acceptable and to industry standards.
- Although the density data originates from a relatively small number of drill holes, there are indications that a single tonnage factor for oxide and for primary, as used in the resource estimates, is a simplification of the true variation with depth.

ACA Howe (2003 and 2007)

ACA Howe inspected Defiance's sample preparation facility and considered the facility to be reasonably well equipped and maintained, in accordance with acceptable industry standards (Leroux and Puritch, 2003; Leroux et al., 2007).

Midas collected a total of 429 pulp samples of known NLSD drilled mineralized zones in early 2003 (Hyde, 2003). Midas inserted blanks and standards and submitted this sample batch to Genalysis. The Genalysis results compared well with the NLSD assays, and the standards and blanks inserted by Midas assayed within acceptable limits.

A comparison of RC and core duplicate samples indicated no major bias across all grade ranges. ACA Howe considered that the degree of scatter shown in graphed data was acceptable for resource estimation purposes. No bias occurred towards the higher grade original or repeat assays.

A total of 134 one-metre interval RC samples from six of Defiance's RC drill holes and 27 core pulp samples from two Defiance core drill holes were submitted by ACA Howe to ALS Chemex Laboratories in Mississauga, Ontario for check analysis.

Upon review of the results, ACA Howe was of the opinion that its independent check assay results confirmed the presence of gold mineralization at Tasiast.

SNC-Lavalin (2004)

Defiance selected mineralized intersections from 30 RC holes covering the four mineralized areas of the Piment deposit, which were sent to Canada for metallurgical test work. SNC-Lavalin (SNC) (Demers et al., 2004) reviewed the drill hole information on the geological sections prepared by ACA Howe and combined the sampled intersections of several drill holes to obtain nine samples considered to be representative for the various mineralized zones and their high and low gold grades. These samples were sent to SGS Lakefield in Ontario, Canada; an ISO/IEC 17025 accredited laboratory for assay. The comparison of the assay results of the initial samples and those from Lakefield was acceptable and showed a reasonable correlation.

SNC representatives collected eight samples of RC drilling chips that had previously been assayed by SGS Analab. These samples were sent to Lakefield for assay. Results showed that gold was present in the indicated mineralized zones even though the correlation was rather erratic due to the statistically low number of samples.

Red Back (2008-2010)

Red Back conducted an analysis of the available, historical QA/QC data from Defiance and Rio Narcea as part of the February 2008 resource update comparing all historical data with data generated by Red Back as at February 2008. The following conclusions were noted (Stuart, 2008):

- Globally, all the laboratories used to compile the Tasiast resource database have reported the ore grade standards well. On average, 85% of the six +1.5 g/t Au internationally accredited CRM samples submitted reported to within an accuracy of ±10%. The historical pre-Red Back database reported 86%. The Red Back database reported 84%.
- A minor negative bias was repeated in each of the standards tested at each of the laboratories. The negative direction of the standards bias, however, results in a degree of conservatism in the assays reported.
- The less than 1.0 g/t Au standards did not perform as well, with a range of 67% to 75% of the standards submitted reporting to within ±10%. The poorer precision and accuracy of the QA/QC data below 1.0 g/t Au is evident across the whole of the resource timeframe.
- Both routine Red Back and pre-Red Back blank submissions performed well, exhibiting only a minor low level <50 ppb Au cross-contamination. Evidence

suggests a component of poor blank selection may have contributed to the Red Back higher bias and was noted for further attention.

- The total operational precision (TOP) achieved by Red Back, demonstrated by a percent Median Absolute Half Difference (%MAHD) = ±14% of resource grade assays >0.2 g/t Au, is generally within acceptable limits of a coarse gold deposit, such as Tasiast. The coarse gold nature of the deposit is apparent within the range of errors expressed by the 90th percentile Absolute Half Difference (AHD) = ±58%.

- While analyzing the historical pre-Red Back duplicate data, it was observed that the historical resource data reported similar "nuggetty" duplicate assaying, closely comparative to the Red Back data with %MAHD = ±14% and a P_{90} AHD = ±50% >0.2 g/t Au. The datasets have equivalent coarse gold features and equivalent assaying precision.

- The imprecision consequent of coarse gold is evident across the entire Tasiast grade profile from 0.2 g/t Au to 10 g/t Au. Clustering of "nuggetty assaying" is often observed in mesothermal greenschist facies, epigenetic, structurally controlled deposits at the high-grade end of the profile due to the coarse gold being hosted dominantly in the quartz vein materials, the gold being finer, and closer to sulphide lattice within the disseminated selvages.

- Red Back considered the extent of the assay data included in the Tasiast resource data to be accurate and precise to within the inherent, natural coarse grade variation observed in the grade profile of this structurally controlled, sub-amphibolite, BIM-hosted style of gold mineralization.

Review of the blank, duplicate, and CRM submissions in 2009 and 2010 (Stuart, 2009; Stuart, 2010) indicated no significant errors or biases in the analytical data. Prior to late 2009 the majority of the field duplicate analyses completed were from non-Greenschist mineralization styles, e.g., Piment iron-formation and West Branch footwall. A total of 16,907 (2009) and 15,929 (2010) QA/QC samples were blindly inserted as part of the routine sample preparation and were submitted for analysis. Red Back concluded that the QA/QC data reported was of industry accepted standards and the assay data was considered reliable for inclusion in the December 2008, 2009, and 2010 resource estimations.

TMSLA (2017)

Verification tests performed on the data at Tasiast by TMSLA included:

1. Collar coordinates, by comparing the data in the database with the data obtained by the surveyor.
2. Downhole survey, by plotting the holes in 3D software and checking for any anomalies in the deviations of the holes. Many different instruments for

measuring hole deviation in downhole surveying were used at Tasiast. As a result, a ranking system of the instrument was implemented, giving priority to the instrument with better precision when multiple instruments were used to survey the same hole. Instruments giving the more accurate reading preceded all other surveys.

3. Lithology, alteration, mineralization, by comparing the data in the database with the data captured in the logs and correcting a few cases of gaps and/or overlaps.
4. Assays, by comparing 10% of the results in the database to the results in the original certificates provided by the laboratories.
5. The validation process was completed by importing the data into Micromine (mining software) and Leapfrog Geo (Leapfrog) to visually check the validity of the data and to generate a report.

12.2 Comment on Data Verification

The process of data verification for the Complex has been performed by TMLSA, Red Back, and personnel of precursor companies, and external consultancies contracted by those companies.

The QP has reviewed the reports and is of the opinion that the data verification programs for the Complex adequately support the geological interpretations, the analytical and database quality, and comply with industry standards, and therefore support the use of the data in Mineral Resource and Mineral Reserve estimation.

Data used to support Mineral Resource and Mineral Reserve estimates have been subjected to validation, using built-in software program triggers that automatically check data for a range of data entry errors. Verification checks on surveys, collar coordinates, lithology, and assay data have also been conducted. The checks are appropriate and consistent with industry standards.

Ongoing sample preparation and analytical work is recommended by the QP to obtain more acceptable precision from the duplicate samples.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Mineralogy

The Tasiast mineralization is free-milling and amenable to gold extraction by simple gravity and cyanide leaching. The existing mill has been operating since 2008, initially treating oxide BIM hosted ore yielding a typical gold recovery of 93%. Gold recovery from fresh ore, which forms an increasing portion of the mill feed since 2010, varies between 91% and 95%. A proportion of the gold is coarse and responds well to gravity concentration. Gold mineralization is associated with structurally controlled faults and shears, quartz-veining, and silica-flooding. Gold grains observed in the exploration core holes are seen in isolated grains in quartz veins and are closely associated with pyrrhotite. The mineralization has relatively low levels of sulphides, approximately 1% to 5% S, predominantly represented by pyrrhotite and to lesser extents pyrite, arsenopyrite, and chalcopyrite. Other metal contents are low, such as silver at approximately 1 ppm to 2 ppm, copper at approximately 100 ppm, arsenic at approximately 10 ppm, and very low levels of mercury at less than 0.3 ppm.

13.2 Metallurgical Test Work

The bulk of the metallurgical test work has been done to evaluate the optimum process for the West Branch ore which has become the major source to the processing plant.

Four major metallurgical sampling campaigns were conducted on the West Branch mineralized zone as follows:

- Campaign 1: Ammtec Pty. Ltd. (Ammtec) (2010)
- Campaign 2: SGS Canada Inc. (2012)
- Campaign 3: SGS Canada Inc., additional boreholes to test for variability (2014)
- Campaign 4: KHD Humboldt Wedag (KDH) and JKTech Pty. Ltd. (JKTech) (2012), high pressure grinding roll (HPGR) and comminution tests

A program of waste rock sampling and characterization was also undertaken with core samples selected to represent all rock lithologies and depths.

In 2024, test work was carried out on composites collected from drill holes of the satellite pits Piment and Fennec, to assess their metallurgical performance with the existing flowsheet.

Test Work Program

Test work was conducted by multiple laboratories and the results from the different laboratories were comparable.

The information generated in the test work programs was sufficient for:

- Preparation of ore characterization
- Process selection
- Process Flowsheet development
- Production scheduling
- Expenditure estimates

Summary of Comminution Characteristics

Work was carried out by Ammtec, SGS, and JKTech to determine the comminution characteristics primarily of West Branch, as well as Piment samples. Tests were performed to assess the variation in comminution parameters and confirm grinding energy requirements for the deeper ore. Sample locations are shown in Figure 13-1.

All of the samples were checked for their correct lithologies and split into separate lithologies for analysis. The mine plan by lithology is shown in Table 13-1 and comminution parameters obtained from laboratory test work are in Table 13-2. The majority of the ore will be from the fresh granodiorite intrusives (GDI) and fresh banded iron (BIM) lithologies.

Table 13-1: Mill feed plan by lithology

	Lithology	2025 (kt)	2026 (kt)	2027 (kt)	2028 (kt)	2029 (kt)	2030 (kt)	2031 (kt)	2032 (kt)	2033 (kt)	2034 (kt)	2035 (kt)
Oxides												
Trans.												
Fresh	BIM	79	775		31	1,637	3,059					
	GDI	575	28	1,762	4,757	2,645	440					
	SVC		623	942	14							
	FVC					13	1,514					
	SGW		68	361		157	61					
Stockpiles		8,154	7,315	5,714	3,982	4,308	3,687	8,760	8,784	8,760	8,760	8,360
CIL Plant		8,808	8,808	8,760	8,784	8,760	8,760	8,760	8,784	8,760	8,760	8,360

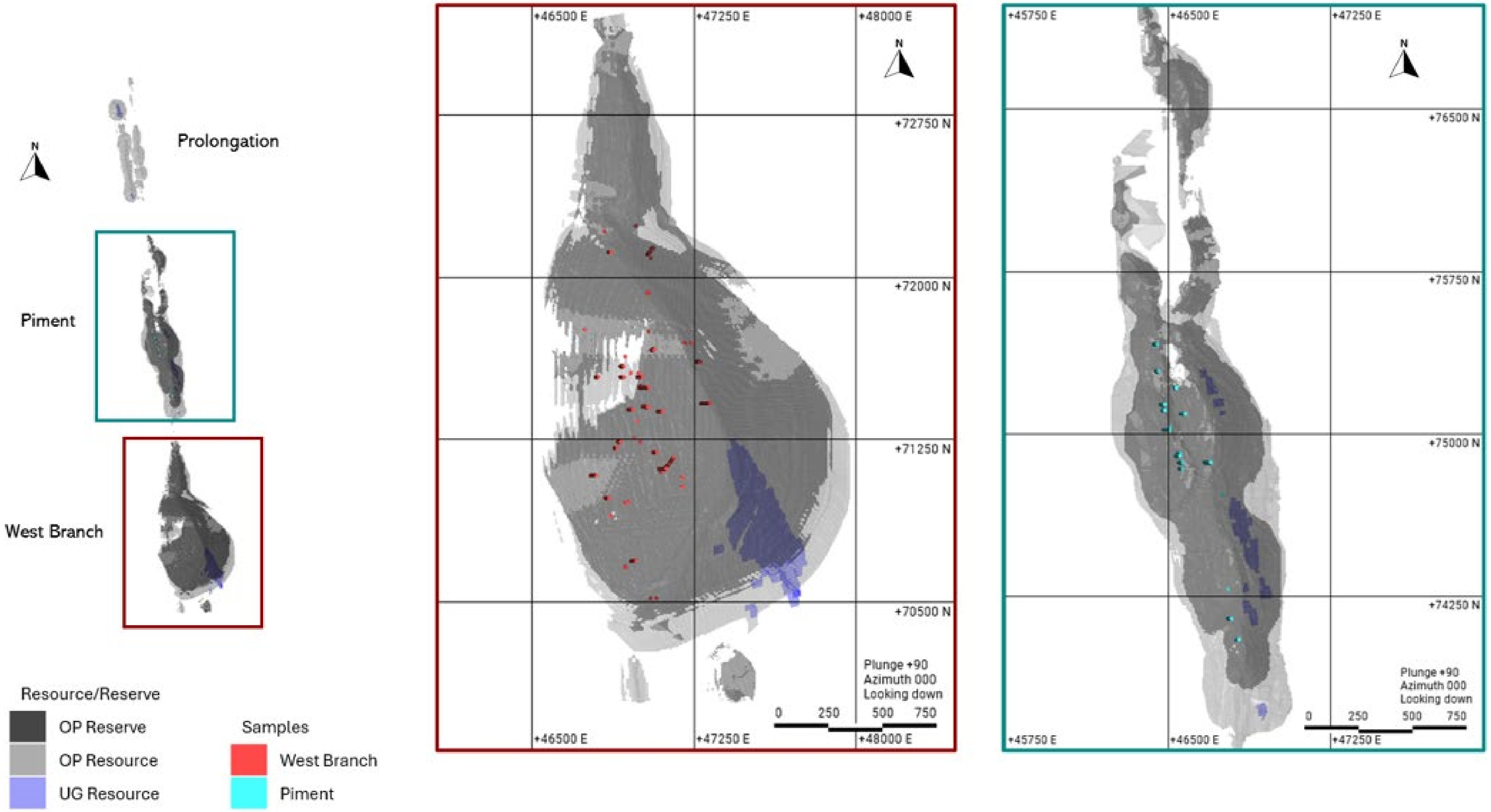


Figure 13-1: Comminution sample locations

Table 13-2: Comminution characteristics by lithology

	Lithology	SG	JK Parameters2		Work Indices (kWh/t)				AI
			t_a	a x b	DWI	CWI	RWI	BWI	
75th Percentile	WB BIM Fr3	3.27	0.26	32.9	10.0	-	-	-	-
	WB FVC Fr	2.68	0.25	26.0	10.4	15.9	25.3	17.1	0.713
	WB GDI Fr4	2.90	0.25	27.6	10.4	17.6	17.2	13.9	0.450
	WB SVC Fr	2.85	0.30	32.4	8.8	16.4	19.2	14.8	0.444
	PIM SVC Fr	2.98	0.31	33.3	8.4	14.6	19.5	15.5	0.265
	PIM BIM Fr	3.31	0.28	34.1	9.6	14.6	19.2	13.3	0.379
Average	WB BIM Fr	3.19	0.30	37.1	8.8	-	-	-	-
	WB FVC Fr	2.68	0.28	29.0	9.4	14.3	23.4	15.9	0.587
	WB GDI Fr	2.86	0.27	29.8	9.7	15.4	15.8	13.4	0.376
	WB SVC Fr	2.82	0.33	35.7	8.1	14.2	18.7	14.4	0.373
	PIM SVC Fr	2.90	0.33	36.4	8.0	14.6	19.5	15.5	0.265
	PIM BIM Fr	3.12	0.30	35.2	8.9	12.7	18.0	12.7	0.312

Notes:

1. a x b and t_a are parameters in the JKTech Drop Weight Test model.
2. BIM lithology combines samples previously tested as BIM_FW (footwall) and BIM_HW (hanging wall).
3. Granodiorite intrusives (GDI) were previously referred to as the greenschist (GST) lithology.
4. SG = Specific gravity.
5. DWI = Drop Weight Index.
6. CWI = Crushing Work Index.
7. RWI = Rod Mill Work Index.
8. BWI = Bond Work Index.
9. AI = Abrasion Index.

The semi-autogenous grinding (SAG) mill test work indicates that the ore becomes progressively harder at depth. A typical relationship indicating the trend of increasing SAG mill grinding energy (SAG power index [SPI]) requirement with depth is shown in Figure 13-2.

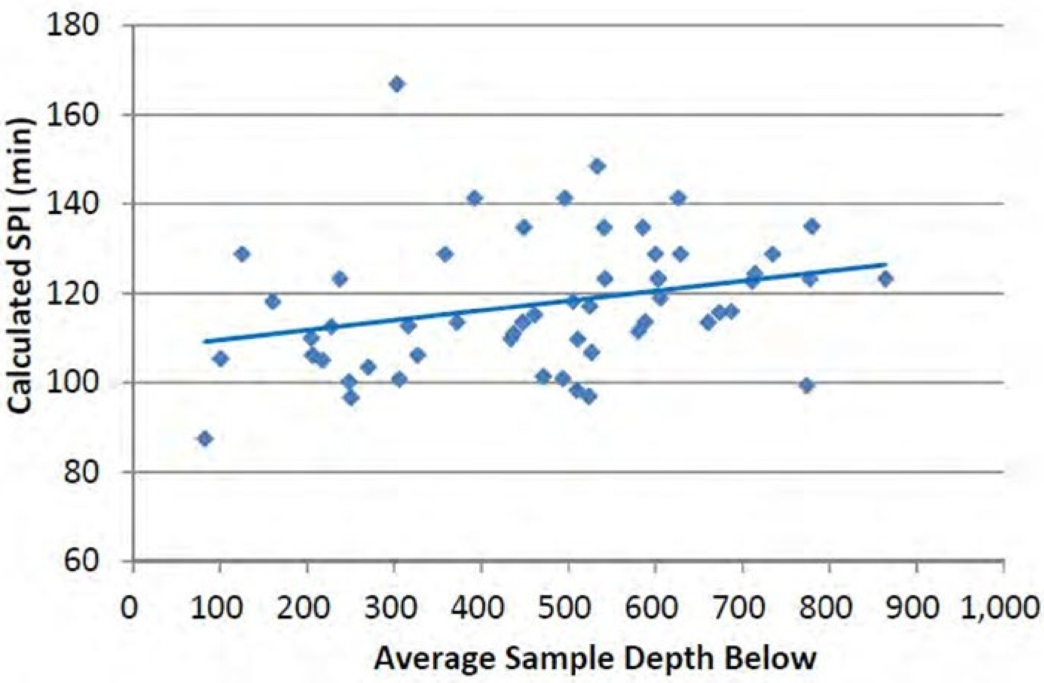


Figure 13-2: Variation of SAG milling power index with vertical depth

The current mine plan does not include mining to below a depth of 500 m, so the SAG energy requirement increase is very low.

Cyanidation Test Work – West Branch

Metallurgical Test Work Parameters

Extensive metallurgical testing was completed on West Branch samples, twinned hole samples, and deeper level variability samples. In general, test work indicated that the ore was amenable to gravity recovery and cyanide leaching, resulting in selection of a flowsheet like that of the existing plant. Some of the key parameters that resulted from the test work are:

- Grind size: 90 µm
- Gravity recovery: variable, approximately 30% to 50% of gold in feed
- Leach retention: approximately 24 hours
- Ore is not preg-robbing
- Cyanide consumption: 0.7 kg/t

Cyanide Concentrations and Grinding Test Work

The grinding test work results show that gold extraction increases with a finer grind size. Gold dissolution kinetics were enhanced at the finer 80% passing (P_{80}) grind sizes of 90 µm and 75 µm. At the selected grind of 90 µm, test work indicates that most leaching is complete at approximately 24 hours, as shown in Figure 13-3. Historically, the operating plant has shown 18 hours to be the optimal leach time, which was selected for design. The improved kinetics relative to the test work are likely the result of grinding in process water containing cyanide recycled from the tailings thickener and gravity recovery circuit, which removes coarse slower leaching gold.

The cyanide addition rate has been optimized to a low addition rate. Test work results indicate that cyanide consumption rate as low as 0.5 g/L is possible. Operationally, a cyanide addition rate of 0.7 kg/t is used for CIL.

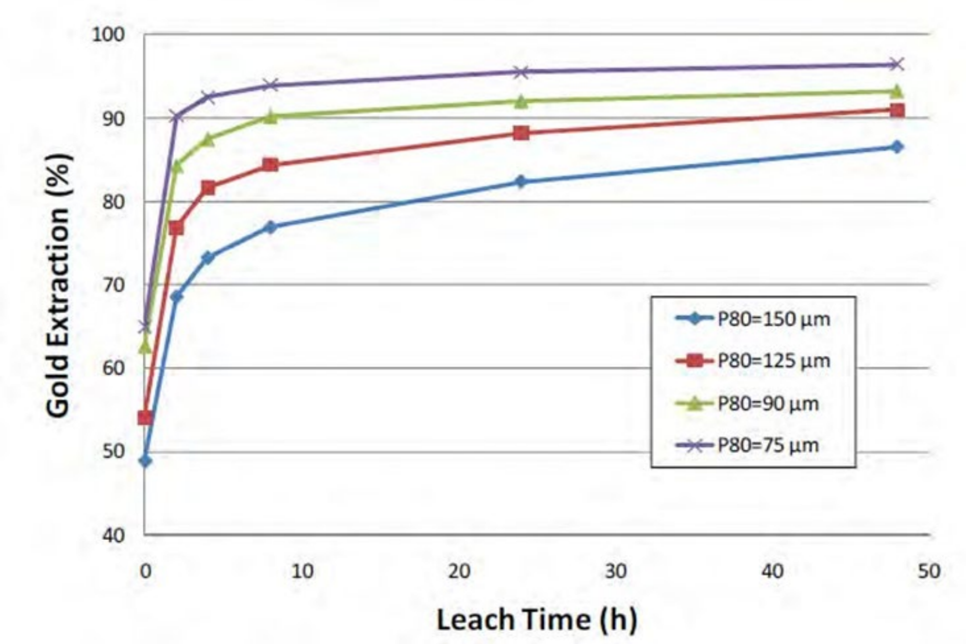


Figure 13-3: Gold recovery as a function of grind and leach retention

Summary of Tests and Recoveries – West Branch

A compilation of all the relevant tests done, limited to those samples within the currently defined resource, produced the recoveries shown in Figure 13-4.

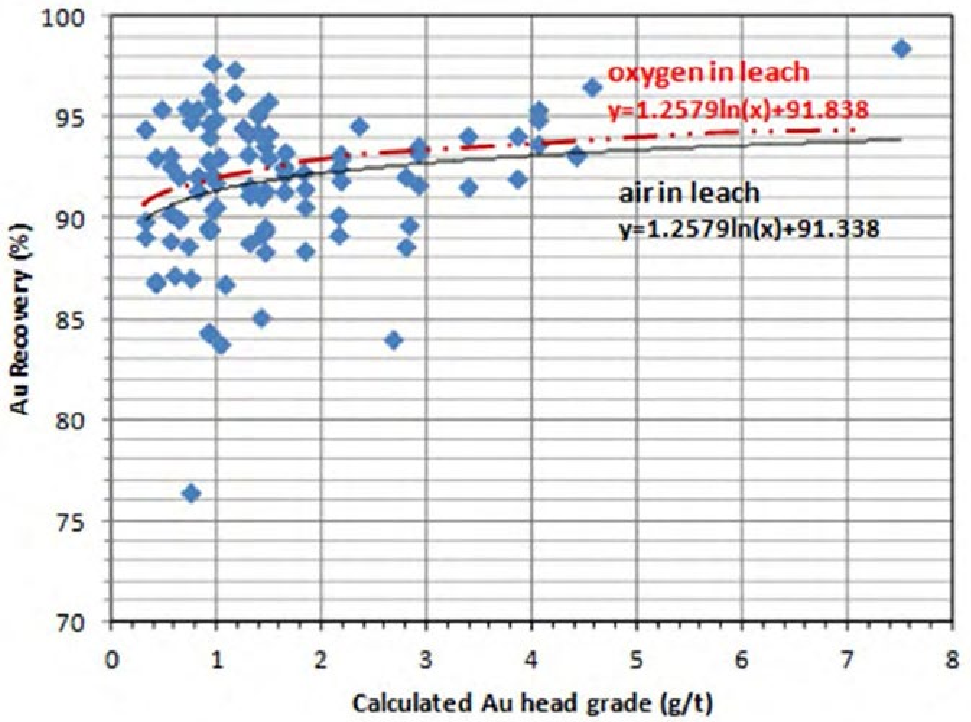


Figure 13-4: Gold grade vs. recovery relationship

The figure shows that all of the selected samples leached well, that oxygen enhancement improved leach rate, and that recoveries are between 84% and 94% at a grade of approximately two grams per tonne. The recoveries are predominantly above 86%, with a few exceptions that, from a metallurgical perspective, gives high confidence that all the sampled parts of the orebody are amenable to gravity and cyanidation. Comparative tests using oxygen (dissolved oxygen (DO) = 15 mg/L to 20 mg/L) vs. air (DO = 6 mg/L to mg/L) indicate that oxygen increases gold extraction in the range of 0.5% to 1.4%, depending on cyanide concentration.

The trend line made up of representative samples indicates a relationship between head grade and gold recovery, with higher recoveries achieved at higher gold head grades, as expected. The mathematical relationship developed was used to estimate recovery based on the ore grade obtained from the mine plan.

Metallurgical Test Work – Piment and Fennec

Test work was carried out by SGS Lakefield in 2024 for the Piment and Fennec satellite pits. Four composites were collected for each pit from full and half HQ core drilled by the Exploration team. The composites were collected with main lithology and alterations represented as well as zones that were expected to be metallurgically challenging to process.

As Fennec is a different rock type (Greywacke) than West Branch and Piment, Bond ball mill work index (BWi) testing we completed at the end of 2024 for indication on comminution performance – the results were not yet available at the time of writing this report. Thickening test work is planned and expected to be completed in early 2025.

The cyanidation tests for Piment and Fennec were run with parameters similar to those in the plant's current operation. The tests included gravity concentration, followed by cyanide leach via bottle roll of the gravity tails:

- At a particle size of approximately 90 µm
- At 250 ppm to 350 ppm free cyanide concentration maintained
- 1.5 hours to 4.0 hours of pre-aeration at natural pH
- 24 hours total leach
- 0 g/t to 250 g/t of lead nitrate

When comparing the averages of the Piment and Fennec tests to test work done at SGS on West Branch samples with similar parameters, the overall recoveries (gravity + leach) are similar, ranging from 94.8% to 95.7%. A summary of results is shown in Table 13-3.

A comparison of cyanidation test work results from Piment and Fennec bottle rolls and the 2022 West Branch tests are shown in Figure 13-5.

Table 13-3: Test work results (SGS 2022 and SGS 2024)

Test Work Program	Sample	CN Test	CN maintain (ppm)	Grind Size P_{80}, (µm)	$Pb(NO_3)_2$ dose, (g/t)	*Pre-aeration time (hr)	Grav Sep	Grav + CIL	Direct SGS	Modelled
			Conditions							
2022 SGS – West Branch mine composites	C. 1 & 2	CN1	350	94	0	0	43.7	93.1	3.12	3.31
	C. 1 & 2	CN2	350	94	100	4	43.7	93.1	3.12	3.31
	C. 1 & 2	CN3	350	94	250	4	43.7	93.2	3.12	3.31
	C. 3	CN4	350	90	0	0	40.0	96.6	2.30	1.99
	C. 3	CN5	350	90	100	4	40.0	96.2	2.30	1.99
	C. 3	CN6	350	90	250	4	40.0	97.1	2.30	1.99
	C. 5 & 6	CN7	350	94	0	0	30.2	93.8	2.89	2.41
	C. 5 & 6	CN8	350	94	100	4	30.2	95.1	2.89	2.41
	C. 5 & 6	CN9	350	94	250	4	30.2	95.1	2.89	2.41
	C. 10	CN10	350	89	0	0	39.9	94.6	3.31	2.91
	C. 10	CN11	350	89	100	4	39.9	93.9	3.31	2.91
	C. 10	CN12	350	89	250	4	39.9	94.6	3.31	2.91
	C. 7 & 8	CN13	350	91	0	0	29.3	95.4	4.02	3.92
	C. 7 & 8	CN14	350	91	100	4	29.3	95.4	4.02	3.92
	C. 7 & 8	CN15	350	91	250	4	29.3	95.3	4.02	3.92
Average			350	92	117	3	36.6	94.8	3.13	2.91
2024 SGS – Piment drill hole composites	S. 1	CN1	250	121	100	1.5	40.0	93.9	1.73	1.89
	S. 1	CN3	250	90	100	1.5	40.0	95.5	1.73	1.89
	S. 1	CN13	250	90	0	1.5	40.0	95.6	1.73	1.89
	S. 2	CN2	250	119	100	1.5	26.2	92.9	0.48	0.71

Test Work Program	Sample	CN Test	CN maintain (ppm)	Grind Size P_{80}, (µm)	$Pb(NO_3)_2$ dose, (g/t)	*Pre-aeration time (hr)	Grav Sep	Grav + CIL	Direct SGS	Modelled
			Conditions							
	S. 2	CN6	250	90	100	1.5	26.2	96.3	0.48	0.71
	S. 2	CN14	250	88	0	1.5	26.2	93.2	0.48	0.71
	S. 3	CN5	250	129	100	1.5	28.0	92.6	0.93	1.68
	S. 3	CN7	250	101	100	1.5	28.0	95.1	0.93	1.68
	S. 3	CN15	250	87	100	1.5	28.0	91.9	0.93	1.68
	S. 4	CN4	250	77	100	1.5	20.1	95.8	2.05	3.10
	S. 4	CN8	250	73	100	1.5	20.1	96.3	2.05	3.10
	S. 4	CN16	250	83	100	1.5	20.1	95.9	2.05	3.10
Average of finer grind tests			250	88	75	2	28.6	95.0	1.30	1.85
2024 SGS – Fennec drill hole composites	S. 1	CN1	250	94	100	1.5	22.3	95.7	0.73	0.80
	S. 2	CN2	250	91	100	1.5	22.1	95.4	1.75	1.91
	S. 3	CN5	250	93	100	1.5	6.6	95.6	4.86	3.53
	S. 4	CN4	250	89	100	1.5	15.0	96.0	9.45	8.99
Average			250	92	100	2	16.5	95.7	4.20	3.81

Notes:

C. = Composite sample; S. = Sample

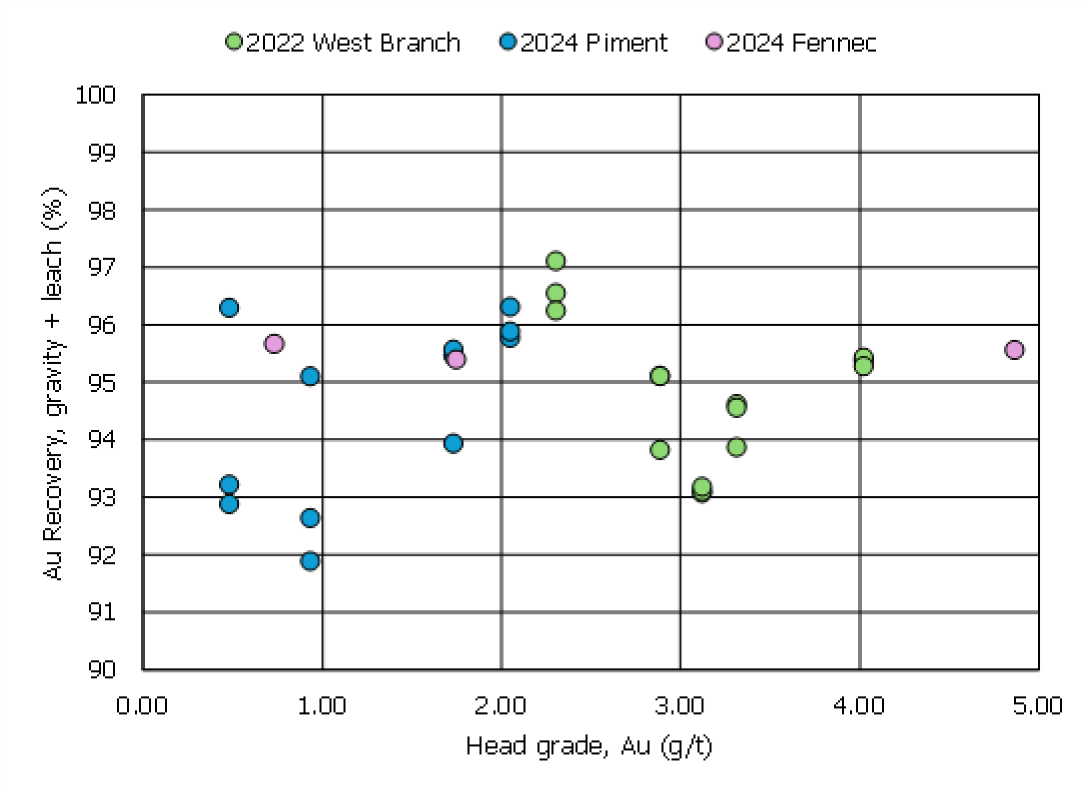


Figure 13-5: Grade-recovery comparison of Piment, Fennec, and West Branch test work

Summary of Thickening Characteristics – West Branch

Ammtec performed flocculent screening tests on ground composite samples of West Branch ore using seawater obtained near Perth. Magnafloc MF336 flocculent was selected for subsequent settling tests to optimize flocculent consumption and develop thickener sizing criteria.

Thickening characteristics of deeper level variability samples were determined through Outotec test work in 2010, FLSmidth test work in 2011, SGS Lakefield test work in 2013, and FLSmidth test work in 2013. Outotec investigated the dynamic settling characteristics and determined the thickener sizing criteria. In 2011, FLSmidth conducted sedimentation and rheology testing. SGS conducted dynamic settling tests on a number of composite samples that had been prepared for leaching test work in 2012 and 2013. Based on test work a unit rate of 0.45 m^2/t/d was selected for design.

Acid Rock Drainage Characteristics of West Branch Samples

Acid rock drainage (ARD) testing was completed on leach residue generated from the GDI samples in the Ammtec 2011 follow-up test work program to simulate plant tailings. Results indicated that the leach residues do not have potential acid generating characteristics, however, have significant acid consuming capacity (likely due to the carbonate content of each ore composite).

In 2011, a waste rock material characterization program was conducted by URS Scott Wilson and supported by Kinross Tasiast and SRK Consulting. During the study, 154 samples were collected from exploration drilling core of different lithologies to assess the ARD potential. Study results showed that waste rock typically exhibits a significant residual neutralization potential for all the lithologies investigated.

The study results, coupled with the favorable arid climate, lack of surface water, and very limited groundwater (no viable groundwater aquifer exists), as well as a Materials Management Plan indicate low potential for ARD or metal leaching to develop.

14. MINERAL RESOURCE ESTIMATE

14.1 Summary

The previous Technical Report (Kinross, 2020) for the property included resource models for West Branch, Piment, and Prolongation deposits, with an effective date of October 31, 2019. An updated MRE was prepared by the Kinross team in October to December 2021 for Piment and Prolongation and November 2022 to February 2023 for West Branch, using available drill hole data as of July 26, 2021. An additional small deposit, Fennec, was estimated in 2020, however, as it's contribution to the total Mineral Resources is so small, it is not discussed in detail in this section. There was no additional exploration drilling completed since the 2019 update and no significant changes to the exploration database. The update was prompted by several structural studies on mineralization control, advances in mining, and the need to use more robust modelling techniques.

Leapfrog Geo software (version 2021.1.3 and 2022.2.1) was used to construct the geological model and Mineral Resource domains. Datamine Studio software (version 1.7.1), Leapfrog Geo, and Snowden Supervisor software were used in the preparation of assay data for geostatistical analysis, block model construction, and gold grade estimation.

The MRE is defined by five mineralized domains for West Branch and twelve mineralized domains for Piment and Prolongation. Each domain comprises an outer shear zone or mineralized envelope modelled as tabular veins, and an inner mineralized sub-domain modelled as an indicator interpolant. The mineralized zones were modelled using a grade threshold of 0.15 g/t Au as a guide, with logging data used in the absence of mineralization.

Samples were composited to two metres within each domain with capping done, per domain, thereafter. Capped values were estimated into sub-blocked models using a three-pass ordinary kriging (OK) approach for the mineralized zones and inverse distance squared (ID^2) for the waste zone. The model was validated using a combination of methods including visual comparison of block estimates and composites, swath plots and change-of-support checks, using the nearest neighbour (NN) de-clustered distribution, as well as visual and statistical validation against the short-term model estimated using grade-control data. An external audit was conducted on the Mineral Resources for West Branch, Piment, and Prolongation in 2024 by SLR.

Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (CIM (2014) definitions) were used for Mineral Resource classification. Classification of Mineral Resources considered the confidence in geological continuity, drill hole spacing, proximity to the current mining areas, and grade control drilling. Areas characterized by drill hole spacings of approximately 30 m define areas classified as Measured for West Branch, Piment, and Prolongation. Spacings of approximately 60 m to 70 m support areas classified as Indicated for West Branch and 70 m for Piment and Prolongation. For the Inferred classification, drill hole spacings of approximately 150 m were used for West Branch and 120 m for Piment and Prolongation.

Mineral Resources for the Tasiast Mine, including the West Branch, Piment, Prolongation, and Fennec deposits are presented in Table 14-1.

Table 14-1: Tasiast Mineral Resource estimate as at December 31, 2024

Deposit	Open Pit			Underground			Stockpiles			Combined		
Class	Tonnage (kt)	Grade (g/t Au)	Contained Gold (koz)	Tonnage (kt)	Grade (g/t Au)	Contained Gold (koz)	Tonnage (kt)	Grade (g/t Au)	Contained Gold (koz)	Tonnage (kt)	Grade (g/t Au)	Contained Gold (koz)
West Branch												
Measured	1,181	0.75	28				13,339	0.51	220	14,520	0.53	248
Indicated	43,584	1.09	1,528							43,584	1.09	1,528
Meas + Ind	44,765	1.08	1,556				13,339	0.51	220	58,104	0.95	1,776
Inferred	3,462	2.17	242	13,825	2.52	1,119				17,288	2.45	1,360
Piment												
Measured	4,668	0.77	116				33	0.60	1	4,702	0.77	117
Indicated	7,423	1.17	280							7,423	1.17	280
Meas + Ind	12,092	1.02	396				33	0.60	1	12,125	1.02	396
Inferred	432	1.30	18	2,769	2.57	228				3,202	2.39	247
Prolongation												
Measured	2,073	1.69	113							2,073	1.69	113
Indicated	1,136	1.33	48							1,136	1.33	48
Meas + Ind	3,210	1.56	161							3,210	1.56	161
Inferred	190	1.16	7							190	1.16	7
Fennec												
Measured												
Indicated	1,112	0.88	32							1,112	0.88	32
Meas + Ind	1,112	0.88	32							1,112	0.88	32
Inferred	367	1.50	18							367	1.50	18

Deposit	Open Pit			Underground			Stockpiles			Combined		
Class	Tonnage (kt)	Grade (g/t Au)	Contained Gold (koz)	Tonnage (kt)	Grade (g/t Au)	Contained Gold (koz)	Tonnage (kt)	Grade (g/t Au)	Contained Gold (koz)	Tonnage (kt)	Grade (g/t Au)	Contained Gold (koz)
Combined Total												
Measured	7,923	1.01	257				13,373	0.51	221	21,296	0.70	478
Indicated	53,255	1.10	1,887							53,255	1.10	1,887
Meas + Ind	61,178	1.09	2,144				13,373	0.51	221	74,550	0.99	2,365
Inferred	4,452	1.99	284	16,595	2.53	1,347				21,047	2.41	1,632

Notes:

1. CIM (2014) Definitions were followed for Mineral Resources.
2. Mineral Resources are estimated using a long-term gold price of US$2,000 per ounce.
3. Open pit Mineral Resources are constrained within an optimized pit shell and reported to cut-off grades ranging from 0.39 g/t Au to 0.50 g/t Au.
4. Underground Mineral Resources are constrained within resource panels below the optimized pit shell which consider a minimum thickness of 2.5 m and a cut-off grade of 1.8 g/t Au. At West Branch, crown pillar resource panels are factored to represent a 100% extraction limit.
5. Bulk density was assigned considering oxidation and lithology.
6. Mineral Resources are exclusive of Mineral Reserves.
7. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
8. Numbers may not add due to rounding.

14.2 Comparison to Previous Estimate

Year over year changes to the exclusive MRE are presented in Table 14-2. The principal reasons for the year over year changes can be attributed to:

1. Reduction in Measured gold ounces due to the use of a lower reserve cut-off grade which has converted a portion of the Resource into Reserve.
2. Increase in Indicated and Inferred ounces due to a slight increase in resource shell size due to an increase in the resource gold price (US$2,000 vs. US$1,700).

Differences due to changes in the estimation approach (2019 vs. 2021/2022) are captured in Table 14-3.

Table 14-2: Year over year changes to the Mineral Resources

	Opening Balance (EOY 2023)			Production Depletion			Geology Change			Engineering Change			Closing Balance (EOY 2024)		
Classification	**Tonnage (kt)**	**Grade (g/t Au)**	**Ounces (koz)**	**Tonnage (kt)**	**Grade (g/t Au)**	**Ounces (koz)**	**Tonnage (kt)**	**Grade (g/t Au)**	**Ounces (koz)**	**Tonnage (kt)**	**Grade (g/t Au)**	**Ounces (koz)**	**Tonnage (kt)**	**Grade (g/t Au)**	**Ounces (koz)**
Measured	9,615.1	0.92	284.0	(434.5)	0.5	(7.4)	5.0	2.8	0.4	(1,262.9)	0.5	(19.7)	7,922.7	1.01	257.3
Indicated	48,936.3	1.05	1,646.0	(1,970.4)	0.6	(35.4)	(486.0)	0.5	(8.2)	6,775.0	1.3	284.7	53,255.0	1.10	1,887.1
Measured (stockpile)	-	-	-	13,372.7	0.5	220.5	-	-	-	-		-	13,372.7	0.51	220.5
Meas + Ind	**58,551.4**	**1.03**	**1,930.0**	**10,967.9**	**0.5**	**177.7**	**(481.0)**	**0.5**	**(7.7)**	**5,512.1**	**1.5**	**265.0**	**74,550.4**	**0.99**	**2,364.9**
Inferred	19,551.0	2.39	1,504.1	-	-	-	-	-	-	1,495.5	2.7	127.7	21,046.5	2.41	1,631.8

Notes:

1. Engineering change includes changes to the cut-off grade and pit designs.

When originally developed, the block models supporting the updated MREs at West Branch, Piment, and Prolongation were compared to the previous (2019) work. Changes to Mineral Resources were mainly attributed to the improved modelling methodology resulting in more constrained mineralized domains compared to the previous model. This change allowed for the preservation of more mineralized material, an increase of the reported tonnage and prevented the high-grade intersections affecting the larger volumes and creating the clusters of disseminated high-grade.

Table 14-3 presents a comparison between the current MRE and the 2019 MRE for West Branch and Piment and Prolongation, with both models upscaled to the same block support and the same representative areas selected. The area within the 2022 final design pit shell was selected for West Branch and the area within the optimized resource shell was selected for Piment and Prolongation. The material already mined out was excluded from the comparison.

The updated MRE shows a 9% increase in total tonnage, a 3% decrease in grade, and a 5% increase in total metal for West Branch. Similarly, for Piment and Prolongation, the updated MRE shows a 13% increase in total tonnage, a 7% decrease in grade, and a 5% increase in total metal. The changes are mainly attributed to the update to the modelling methodology resulting in more constrained mineralized domains compared to the previous model. This change allowed for the preservation of more mineralized material, an increase of the reported tonnage, and prevented the high-grade intersections affecting the larger volumes and creating the clusters of disseminated high-grade.

Table 14-3: Comparison between the current and previous MRE

Deposit	Classification	Model Year	Tonnage (kt)	Grade (g/t Au)	Gold Ounces (koz)
West Branch	Measured + Indicated	2019	46,538	2.16	3,239
		2022	50,593	2.10	3,416
		Difference	9%	-3%	5%
Piment and Prolongation	Measured + Indicated	2020	21,144	1.74	1,186
		2021	23,904	1.63	1,249
		Difference	13%	-7%	5%

Notes:

1. The cut-off-grade of 0.70 g/t Au was applied.
2. This table does not represent Mineral Resources and is only for comparative purposes.

14.3 Mineral Resource Cut-off Grades

Assumptions and inputs into the cut-off grade calculations for open pit (OP) and underground (UG) gold mining at Tasiast are presented in Table 14-4.

Table 14-4: Cut-off grade inputs and assumptions

Inputs	Units	OP	UG
Resource Metal Price	USD/oz	2,000	2,000
Payable Metal	%	99.95%	99.95%
Treatment and Refining Charges	USD/oz	5.47	10.3
Reclamation Cost	USD/oz	11.06	
Royalty	%	8.72%	8.2%
Average Head Grade	g/t	1.04	2.5
Processing Recovery	%	92.8	93.9
Mining Cost	USD/t mined	3.38 – 3.42	47.6
Rehandle Cost	USD/t milled	0.91	
Haulage to Mill	USD/t milled	0 – 3.01	0
Processing Cost	USD/t milled	15.61 – 15.79	14.6
G&A Cost	USD/t milled	8.23	13.4
Mining Sustaining Capital	USD/t mined	0.24 – 0.33	
Processing Sustaining Capital	USD/t milled	1.48 – 1.55	
Cut-off Grade	**g/t Au**	**0.4 – 0.5**	**1.8**

Notes:

1. Ranges in OP Resource inputs due to differences in main and satellite pit treatment.
2. OP mining cost includes base cost and incremental bench cost (increasing at depth).

14.4 Resource Database

The combined West Branch, Piment, and Prolongation exploration database is constrained between Y>68,000 and Y<80,000 in the local mine grid. The overall database includes 7,236 exploration drill holes with a total length of 1,078,840 m. This includes 601 diamond holes, 6,424 RC holes, and 211 RC pre-collar (diamond tail) holes. The database contains 787,976 assayed gold intervals with a total length of 850,075 m.

14.5 Geological Interpretation

Mineralized Domains

The MRE is defined by five mineralized domains for West Branch and twelve mineralized domains for Piment and Prolongation, each comprising an outer shear zone/mineralized envelope modelled as tabular veins, and an inner mineralized sub-domain modelled as an indicator interpolant. The mineralized envelopes modelled broad zones of potential mineralization and did not apply additional constraints on the amount of internal dilution. The mineralized sub-domains were modelled within the envelopes to avoid excessive dilution and further refine each domain. These domains were interpolated along the structural trends consistent with the orientation of the respective envelopes, following the gentle south plunge. Smaller domains supported by two or fewer boreholes were removed from the final domains for West Branch. For Piment and Prolongation, an additional disseminated domain was created to constrain many discontinuous lenses.

The mineralized zones were modelled using a grade threshold of 0.15 g/t gold as a guide, with lithology, mineralization, and alteration logging used in the absence of mineralization. The grade modelling threshold was selected based on the apparent inflection of the mineralized population on the gold probability plot (Figure 14-1).

Final mineralized domains are presented in Table 14-5 for West Branch and Table 14-6 for Piment and Prolongation. Plan and cross-sectional views of the mineralized zones and envelopes are presented for each deposit in Figure 14-2 to Figure 14-4.

Table 14-5: Mineralized domains – West Branch

Domain Type	Domain Code	Description	Volume (m^3)
Mineralized Zone	3100	FVC mineralization between GDI and HW BIF and disseminated in upper GDI	33,735,000
	3200	Disseminated in HW BIF	11,655,000
	3300	Major GDI Mineralization	140,900,000
	3400	SVC and FVC contact in the south and SVC Hosted	55,307,000
	3500	South part of SVC hosted mineralization	39,337,000
Envelope	13100	FVC mineralization between GDI and HW BIF and disseminated in upper GDI	82,887,000
	13200	Disseminated in HW BIF	31,887,000
	13300	Major GDI Mineralization	207,900,000
	13400	SVC and FVC contact in the south + SVC Hosted	78,032,000
	13500	South part of SVC hosted mineralization	52,208,000
Waste	3999		11,995,000,000

Notes:

1. Envelope volumes are inclusive of mineralized subdomain volumes

Table 14-6: Mineralized domains – Piment and Prolongation

Domain Type	Domain Code	Description	Volume (m^3)
Mineralized Zone	100	Piment main zone	45,880,000
	200	Piment hanging wall zone	5,229,500
	300	Piment and Prolongation FW zone - transition to GDI at West Branch	35,274,000
	400	Piment small HW lens	171,040
	500	Prolongation zone main north	2,641,800
	600	Prolongation zone south 1	1,099,600
	700	Prolongation north lens	569,880
	800	Prolongation zone south 2	377,450
	900	Prolongation zone south 3	318,010
	1000	Piment FW BIF zone	2,996,200
	1100	C68 and NE Satellites	1,506,500
	1200	East small lens - ARL zone	76,095
Envelope	10100	Piment main zone	93,927,000
	10200	Piment hanging wall zone	16,390,000
	10300	Piment and Prolongation FW zone - transition to GDI at West Branch	73,489,000
	10400	Piment small HW lens	302,400
	10500	Prolongation zone main north	4,996,800
	1,600	Prolongation zone south 1	2,306,700
	10700	Prolongation north lens	1,104,300
	10800	Prolongation zone south 2	755,390
	10900	Prolongation zone south 3	480,670
	11000	Piment FW BIF zone	14,268,000
	11100	C68 and NE Satellites	6,130,200
	11200	East small lens	548,390
Disseminated	1900	Disseminated zone	4,125,600
Waste	1999	Waste	58,089,000,000

Notes:

1. Envelope volumes are inclusive of mineralized subdomain volumes

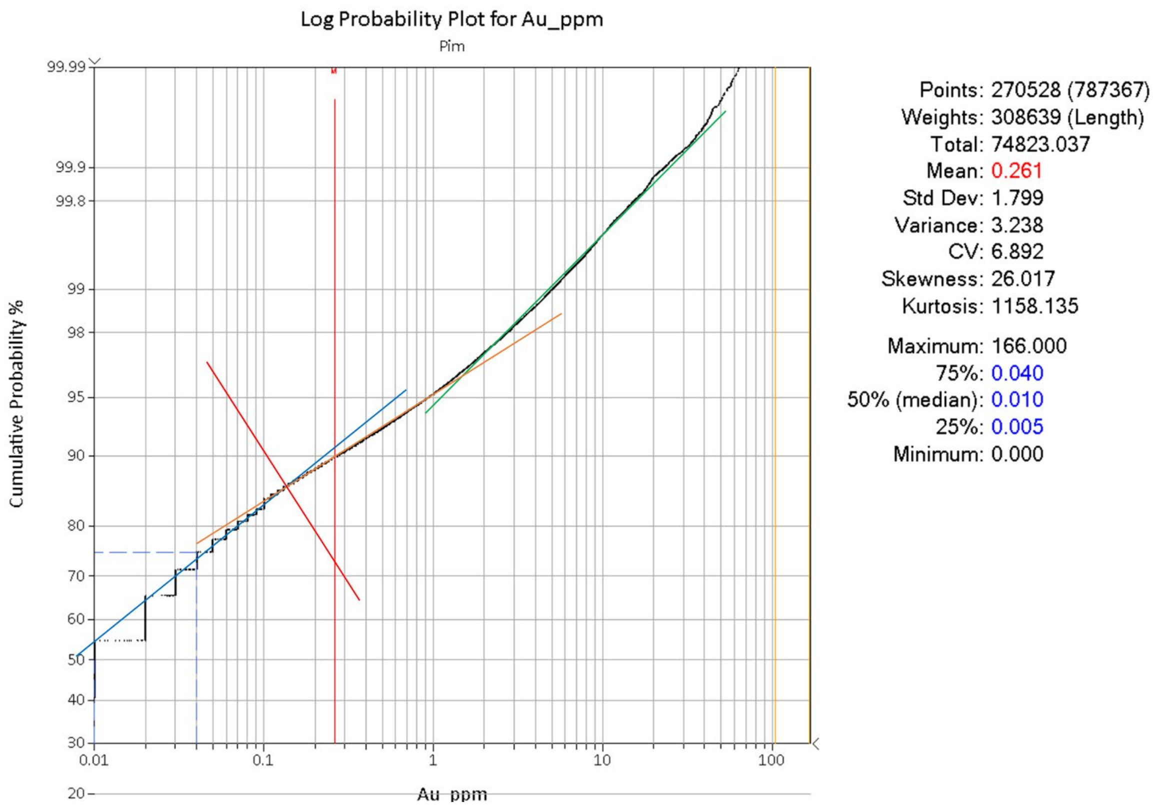
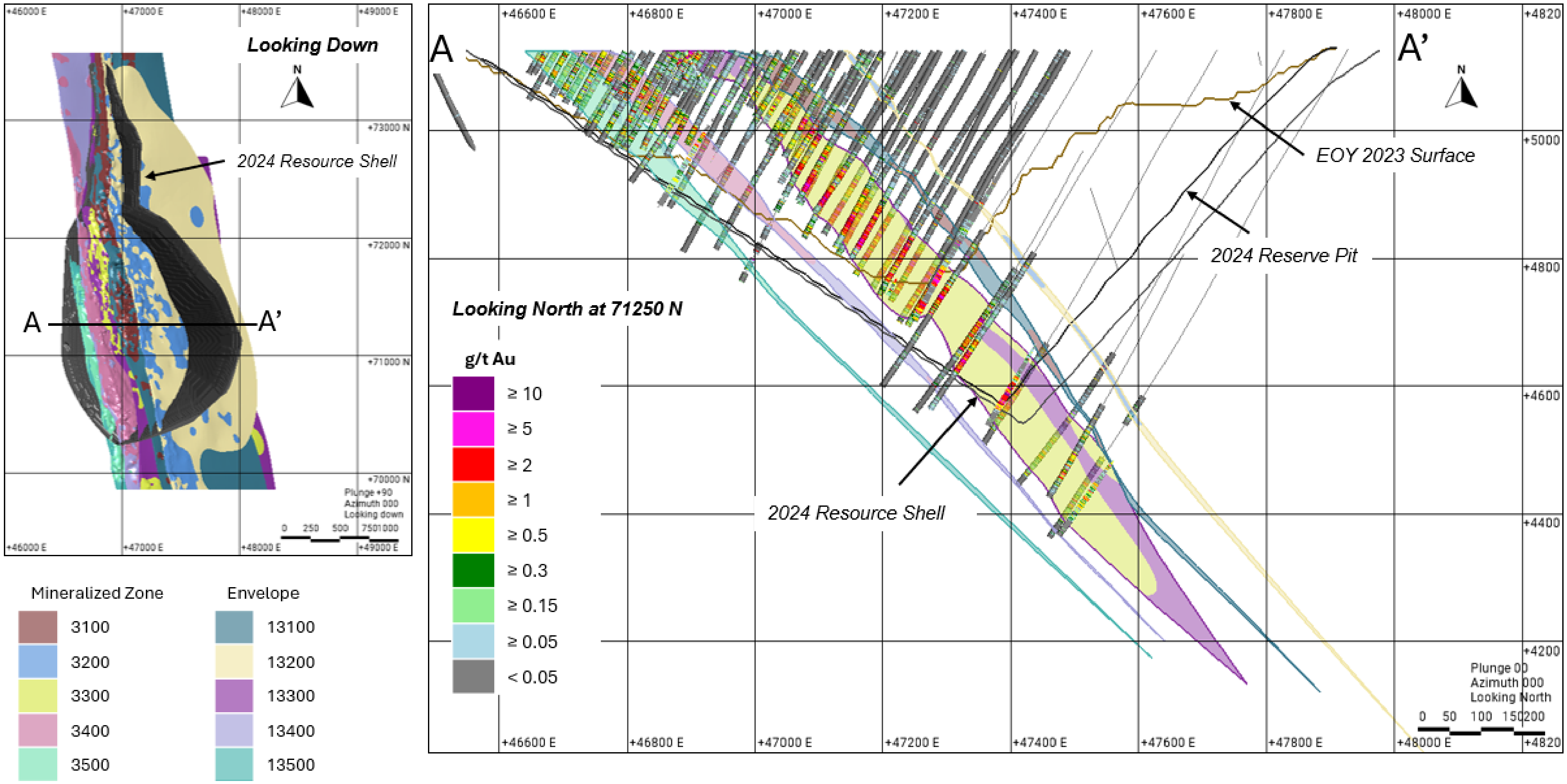
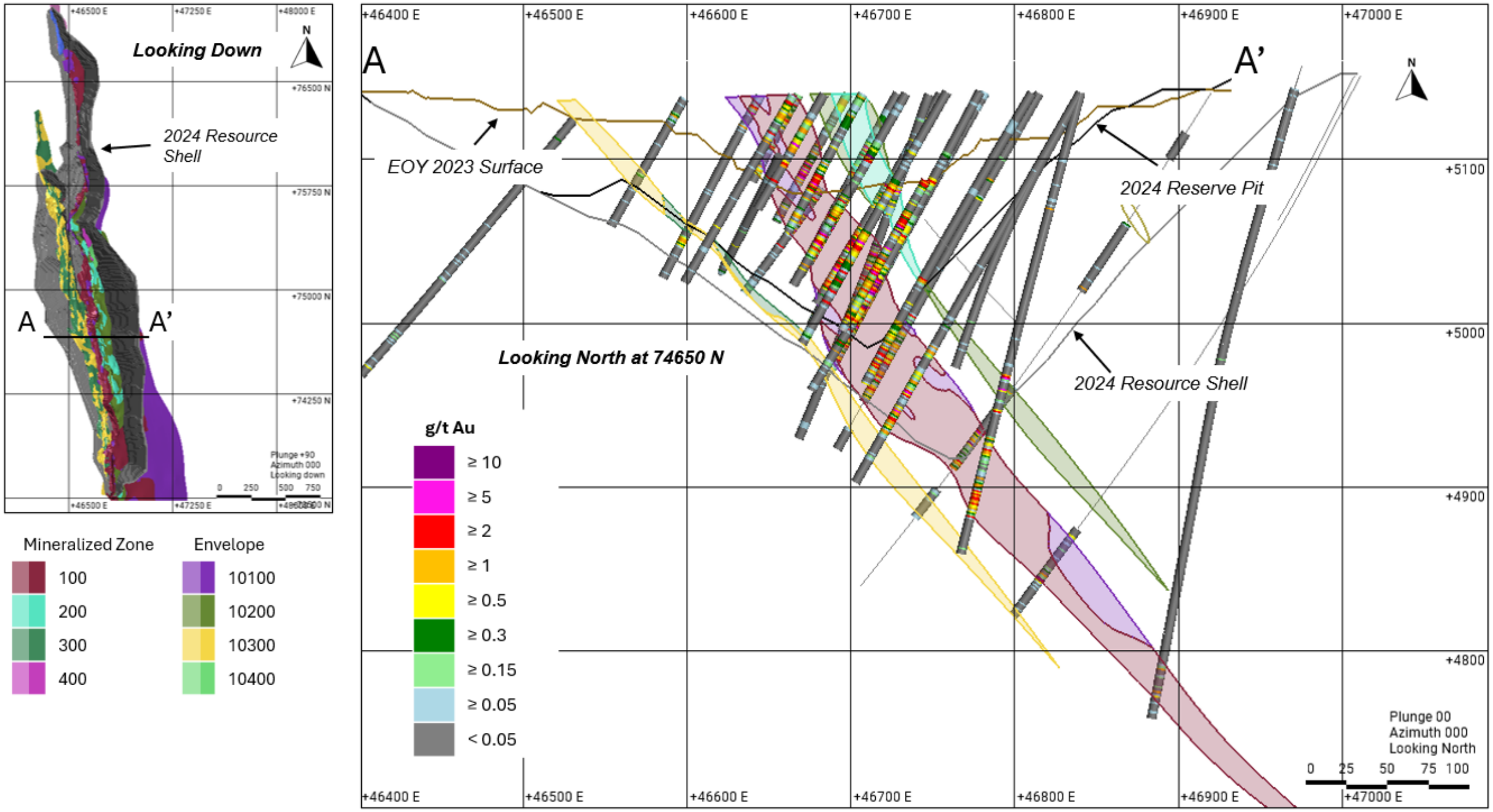


Figure 14-1: Length-weighted gold distribution indicating the 0.15 g/t modelling threshold at West Branch



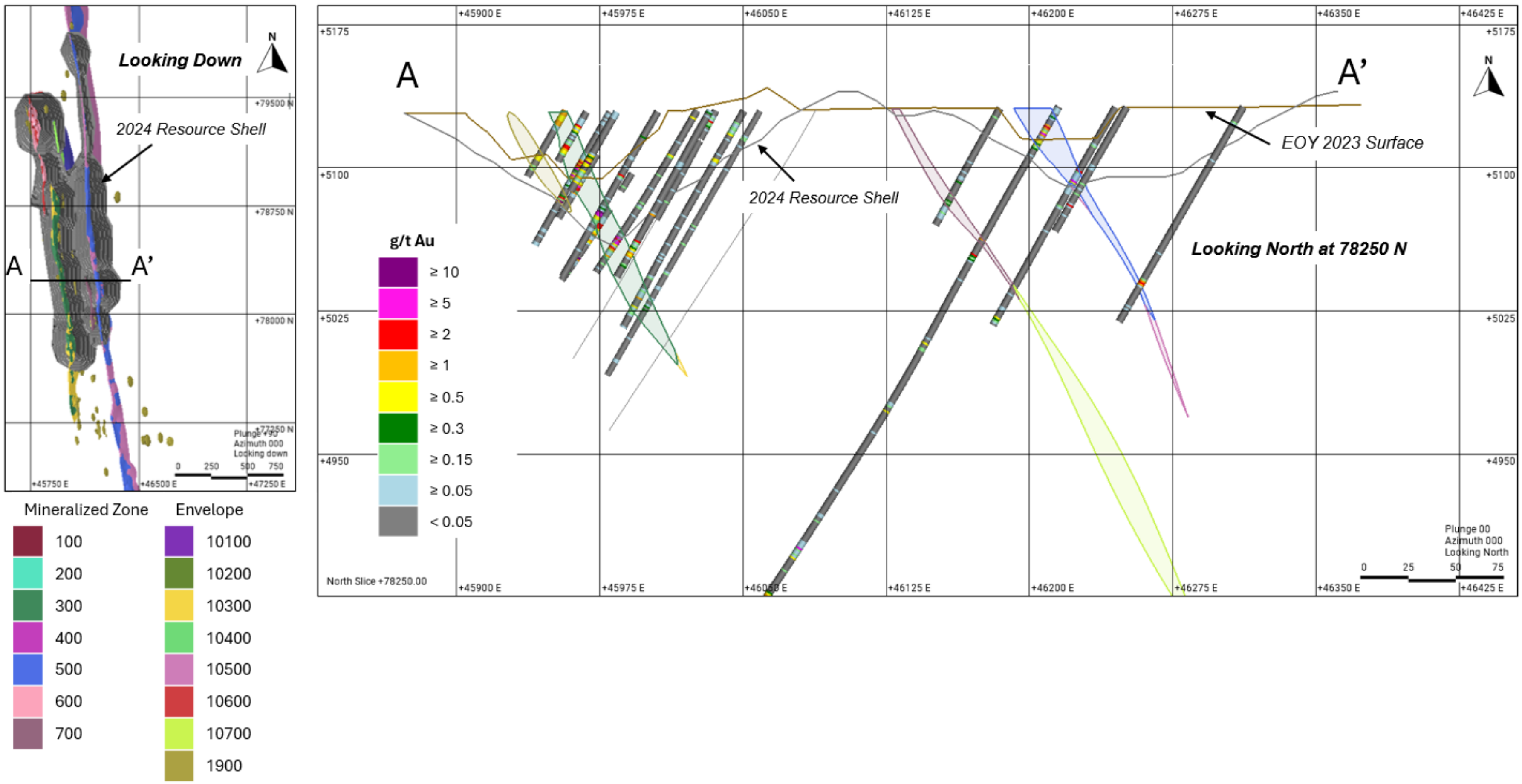
Source: SLR 2024.

Figure 14-2: Mineralized zones and envelopes – West Branch



Source: SLR 2024.

Figure 14-3: Mineralized zones and envelopes – Piment



Source: SLR 2024.

Figure 14-4: Mineralized domains and envelopes – Prolongation

14.6 Lithology and Redox Domains

Since the last MRE update in 2019 the lithology model for West Branch was extended to deeper levels (3,900 m) with some updates to the interpretation of the dykes. The redox model was also updated using the logging information to better reflect the drilling data, however, these changes are considered immaterial for the results of the current MRE as the oxidized material is mined out. For Piment and Prolongation, the lithology and redox models were not changed since the last MRE update in 2019.

The summary of the lithology and redox codes is presented in Table 14-7. A lithology model is shown in Figure 7-5.

Table 14-7: Lithology and redox codes

Deposit	Data	Domain	Codes
West Branch	Lithology	GDI	30
		BIF	41 (FW) and 42 (HW)
		FVC	50
		Dykes	10
		SGW	Merged as SVC Unit with a code of 61 (FW) and 62 (HW)
		SSL	
	Redox	Oxide	1
		Transitional	2
		Fresh	3
Piment and Prolongation	Lithology	GDI	NA
		BIF	1
		FVC	2
		Dykes	3
		SGW	4
		SSL	5
	Redox	Oxide	1
		Transitional	2
		Fresh	3

Notes:

1. HW – hanging wall; FW – footwall

14.7 Resource Assays

Compositing

The majority (approximately 96%) of the assay intervals in the database are sampled at 2.0 m or less with the vast majority (94%) having a length of 1.0 m. Therefore, samples were composited to 2.0 m, within each domain, with residual intervals less than 1.0 m added to the previous composite. Table 14-8 and Table 14-9 summarize the original and composted gold assay statistics per domain for West Branch and Piment and Prolongation, respectively.

Table 14-8: Gold assay statistics before and after compositing – West Branch

Domain Code	Assays Length-Weighted						Composites 2 m					
	Count	Length (m)	Mean (g/t)	Min. (g/t)	Max. (g/t)	CV	Count	Length (m)	Mean (g/t)	Min. (g/t)	Max. (g/t)	CV
Mineralized Zone												
3100	29,496	29,780	0.75	0.0001	88.90	3.12	15,229	29,780	0.75	0.0001	48.30	2.47
3200	6,042	6,059	0.56	0.0001	83.50	2.73	3,185	6,059	0.56	0.0050	41.84	2.00
3300	62,310	61,614	1.49	0.0001	176.00	2.24	31,196	61,613	1.48	0.0001	169.50	1.84
3400	32,999	33,374	0.77	0.0001	170.00	3.11	16,983	33,374	0.76	0.0001	86.66	2.35
3500	23,559	22,894	0.55	0.0001	496.00	6.60	11,661	22,894	0.55	0.0001	248.08	4.63
Envelopes												
13100	17,463	17,140	0.09	0.0001	50.00	7.39	8,776	17,140	0.09	0.0001	34.50	6.20
13200	2,716	2,564	0.09	0.0001	17.30	4.84	1,349	2,564	0.10	0.0001	17.30	5.15
13300	12,107	11,775	0.08	0.0001	63.00	8.49	6,076	11,775	0.08	0.0001	31.52	5.93
13400	4,466	4,352	0.08	0.0001	4.17	2.22	2,271	4,351	0.08	0.0001	3.63	1.77
13500	2,702	2,645	0.08	0.0001	3.48	2.16	1,398	2,644	0.08	0.0001	2.69	1.69
Dykes												
5555	11,386	11,154	0.14	0.0001	50.00	6.39	5,663	11,154	0.14	0.0001	34.48	5.18
Waste												
3999	263,304	268,736	0.05	0.0001	129.00	12.08	135,680	268,734	0.05	0.0001	90.40	10.42
Total Mineralized												
	154,406	153,721	1.01				78,254	153,720	1.01			
Total Envelope												
	39,454	38,476	0.09				19,870	38,474	0.09			

Notes:

1. Min = Minimum; Max = Maximum; CV = Coefficient of Variation.
2. Null vales were replaced with a background value of 0.00013 g/t.
3. Intervals longer than two metres with no assay value were considered not sampled and excluded from the estimation.
4. Intervals shorter than two metres with no assay value were considered barren and assigned the background value 0.00011 g/t.

Table 14-9: Gold assay statistics before and after compositing – Piment and Prolongation

Domain Code	Assays Length-Weighted						Composites 2 m					
	Count	Length (m)	Mean (g/t)	Min. (g/t)	Max. (g/t)	CV	Count	Length (m)	Mean (g/t)	Min. (g/t)	Max. (g/t)	CV
Mineralized Zone												
100	38,792	40,531	1.40	0.0001	119.00	2.66	20,676	40,514	1.39	0.0001	69.70	2.20
200	4,987	5,535	1.12	0.0001	83.50	2.50	2,885	5,531	1.11	0.0001	41.84	2.00
300	21,745	22,394	1.05	0.0001	154.00	3.66	11,561	22,375	1.04	0.0001	106.80	2.93
400	137	175	1.32	0.0050	66.40	4.00	92	175	1.34	0.0300	26.46	2.41
500	1,299	1,430	1.24	0.0001	30.90	2.27	756	1,427	1.21	0.0050	21.18	1.86
600	823	950	2.32	0.0050	49.30	2.18	500	950	2.26	0.0050	34.40	1.84
700	283	288	1.07	0.0001	34.90	3.37	150	287	1.05	0.0050	29.85	2.99
800	122	136	0.53	0.0001	14.10	2.52	71	136	0.53	0.0001	7.14	1.79
900	142	145	0.48	0.0050	11.40	2.50	75	144	0.47	0.0050	5.81	1.88
1000	1,274	1,320	0.65	0.0050	18.90	2.01	700	1,318	0.66	0.0050	11.10	1.64
1100	1,260	1,287	1.45	0.0050	68.00	3.69	681	1,284	1.41	0.0050	46.75	2.90
1200	118	119	1.89	0.0100	21.00	1.89	63	119	1.85	0.0150	18.60	1.63
Envelopes												
10100	9,675	9,929	0.08	0.0001	33.50	6.00	5,165	9,920	0.08	0.0001	17.43	4.49
10200	5,105	5,331	0.07	0.0001	10.80	4.53	2,738	5,320	0.08	0.0001	4.28	3.39
10300	12,469	12,646	0.08	0.0001	166.00	18.61	6,543	12,632	0.09	0.0001	94.30	14.72
10400	106	160	0.11	0.0050	2.97	3.74	82	160	0.12	0.0050	2.97	3.34
10500	462	489	0.07	0.0001	2.44	3.00	252	488	0.07	0.0050	1.85	2.62
10600	537	581	0.07	0.0050	5.83	4.16	298	580	0.08	0.0050	2.94	3.20
10700	71	73	0.09	0.0050	1.83	3.11	38	72	0.07	0.0050	1.21	2.83
10800	25	25	0.06	0.0025	0.69	2.41	14	25	0.06	0.0025	0.39	1.75
10900	53	45	0.05	0.0050	0.33	1.50	25	45	0.06	0.0050	0.24	1.12
11000	999	992	0.06	0.0001	1.76	1.72	512	990	0.06	0.0050	1.25	1.50
11100	1,915	1,924	0.05	0.0001	3.97	3.95	993	1,924	0.05	0.0050	2.60	3.03

Domain Code	Assays Length-Weighted						Composites 2 m					
	Count	Length (m)	Mean (g/t)	Min. (g/t)	Max. (g/t)	CV	Count	Length (m)	Mean (g/t)	Min. (g/t)	Max. (g/t)	CV
11200	184	184	0.05	0.0050	2.07	3.42	95	184	0.06	0.0050	1.13	2.52
Disseminated												
1900	2,232	2,408	0.89	0.0001	124.00	4.81	1,280	2,408	0.86	0.0050	62.32	3.68
Waste												
1999	277,318	314,363	0.03	0.0001	73.80	11.31	158,251	314,288	0.03	0.0001	57.75	8.94
Total Mineralized												
	70,982	74,310	1.27	0.0001	154.00	2.93	38,210	74,259	1.26			
Total Envelope												
	31,601	32,379	0.08	0.0001	166.00	13.09	16,755	32,339	0.08			

Capping

Capping was applied to composites within each estimation domain. The capping strategy considered a combination of probability plots, decile analysis, and capping sensitivity plots. Separation of grade populations characterized by inflections in the probability plot or gaps in the high tail of the grade distribution were indicators of potential capping values. Decile analysis was then used to confirm the reasonableness of capped threshold. A visual review of the spatial distribution of these potential capped values was also performed.

Table 14-10 and Table 14-11 summarize the composited and capped gold assay statistics for West Branch and Piment and Prolongation, respectively. Figure 14-5 and Figure 14-6 show examples of capping analysis for domain 3300 (West Branch) and domain 100 (Piment and Prolongation). The results of the analysis (visual and statistical) demonstrate well-behaved lognormally distributed grades in the capped composites dataset. The coefficient of variation (CV) values close to 2.0 or less suggest a statistically homogeneous population which is mostly the result of additional sub-domaining of the mineralized envelopes.

Table 14-10: Composited and capped gold assay statistics – West Branch

Domain Code	Count	Composites 2 m				Capped Composites 2 m				Difference	
		Mean (g/t)	Min (g/t)	Max (g/t)	CV	Mean (g/t)	Min (g/t)	Max (g/t)	CV	Mean (g/t)	CV
Mineralized Zone											
3100	15,229	0.75	0.0001	48.30	2.47	0.74	0.0001	30.00	2.34	-1%	-6%
3200	3,185	0.56	0.0050	41.84	2.00	0.54	0.0050	7.00	1.52	-2%	-24%
3300	31,196	1.48	0.0001	169.50	1.84	1.47	0.0001	40.00	1.62	-1%	-12%
3400	16,983	0.76	0.0001	86.66	2.35	0.74	0.0001	18.00	1.78	-3%	-24%
3500	11,661	0.55	0.0001	248.08	4.63	0.51	0.0001	10.00	1.48	-7%	-68%
Envelope											
13100	8,776	0.09	0.0001	34.50	6.20	0.08	0.0001	3.50	2.64	-12%	-58%
13200	1,349	0.10	0.0001	17.30	5.15	0.08	0.0001	1.20	2.01	-20%	-61%
13300	6,076	0.08	0.0001	31.52	5.93	0.07	0.0001	1.20	1.70	-14%	-71%
13400	2,271	0.08	0.0001	3.63	1.77	0.08	0.0001	1.00	1.20	-4%	-32%
13500	1,398	0.08	0.0001	2.69	1.69	0.08	0.0001	1.20	1.39	-3%	-18%
Dykes											
5555	5,663	0.14	0.0001	34.48	5.18	0.12	0.0001	5.00	3.51	-8%	-32%
Waste											
3999	135,680	0.05	0.0001	90.40	10.42	0.04	0.0001	1.00	2.28	-19%	-78%

Table 14-11: Composited and capped gold assay statistics – Piment and Prolongation

Domain Code	Count	Composites 2 m				Capped Composites				Difference	
		Mean (g/t)	Min. (g/t)	Max. (g/t)	CV	Mean (g/t)	Min. (g/t)	Max. (g/t)	CV	Mean (g/t)	CV
Mineralized Zone											
100	20,676	1.39	0.0001	69.70	2.20	1.38	0.0001	37.00	2.06	-1%	-6%
200	2,885	1.11	0.0001	41.84	2.00	1.04	0.0001	9.00	1.56	-6%	-22%
300	11,561	1.04	0.0001	106.80	2.93	1.00	0.0001	27.00	2.36	-4%	-19%
400	92	1.34	0.0300	26.46	2.41	0.95	0.0300	4.50	1.39	-29%	-42%
500	756	1.21	0.0050	21.18	1.86	1.15	0.0050	10.00	1.66	-5%	-11%
600	500	2.26	0.0050	34.40	1.84	2.05	0.0050	14.00	1.55	-9%	-16%
700	150	1.05	0.0050	29.85	2.99	0.75	0.0050	6.00	1.63	-28%	-45%
800	71	0.53	0.0001	7.14	1.79	0.53	0.0001	7.14	1.79	0%	0%
900	75	0.47	0.0050	5.81	1.88	0.47	0.0050	5.81	1.88	0%	0%
Envelope											
1000	700	0.66	0.0050	11.10	1.64	0.63	0.0050	5.00	1.39	-5%	-15%
1100	681	1.41	0.0050	46.75	2.90	1.24	0.0050	17.00	2.31	-12%	-20%
1200	63	1.85	0.0150	18.60	1.63	1.53	0.0150	6.00	1.19	-17%	-27%
10100	5,165	0.08	0.0001	17.43	4.49	0.08	0.0001	3.00	3.04	-7%	-32%
10200	2,738	0.08	0.0001	4.28	3.39	0.08	0.0001	3.00	3.17	-2%	-6%
10300	6,543	0.09	0.0001	94.30	14.72	0.06	0.0001	3.00	2.87	-25%	-81%
10400	82	0.12	0.0050	2.97	3.34	0.12	0.0050	2.97	3.34	0%	0%
10500	252	0.07	0.0050	1.85	2.62	0.07	0.0050	1.85	2.62	0%	0%
10600	298	0.08	0.0050	2.94	3.20	0.08	0.0050	2.94	3.20	0%	0%
10700	38	0.07	0.0050	1.21	2.83	0.07	0.0050	1.21	2.83	0%	0%
10800	14	0.06	0.0025	0.39	1.75	0.06	0.0025	0.39	1.75	0%	0%
10900	25	0.06	0.0050	0.24	1.12	0.06	0.0050	0.24	1.12	0%	0%
11000	512	0.06	0.0050	1.25	1.50	0.06	0.0050	1.25	1.50	0%	0%
11100	993	0.05	0.0050	2.60	3.03	0.05	0.0050	1.00	2.14	-9%	-29%
11200	95	0.06	0.0050	1.13	2.52	0.06	0.0050	1.13	2.52	0%	0%
Disseminated											
1900	1,280	0.86	0.0050	62.32	3.68	0.76	0.0050	17.00	2.47	-12%	-33%
Waste											
1999	158,251	0.03	0.0001	57.75	8.94	0.03	0.0001	3.00	4.04	-11%	-55%

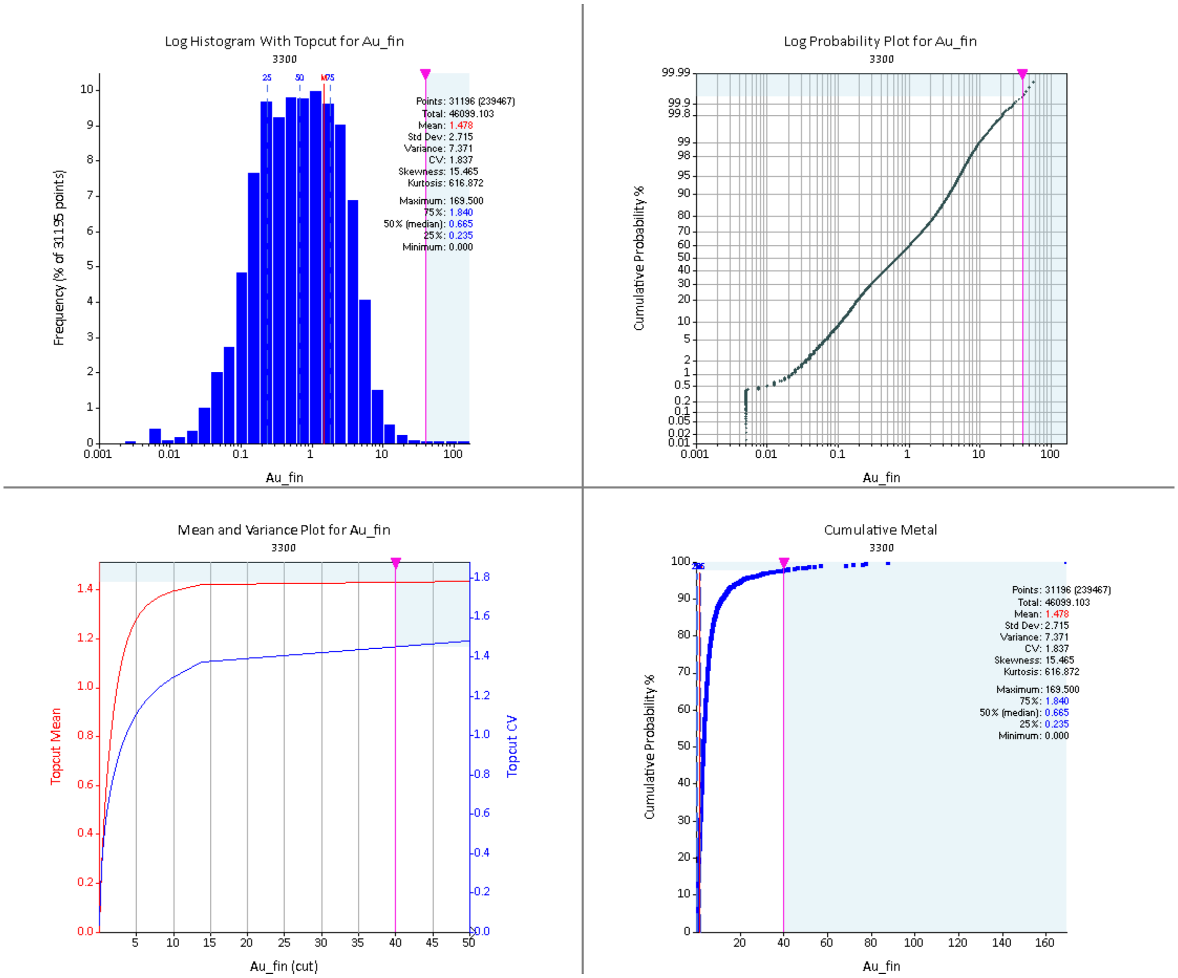


Figure 14-5: Capping analysis for Domain 3300 – West Branch

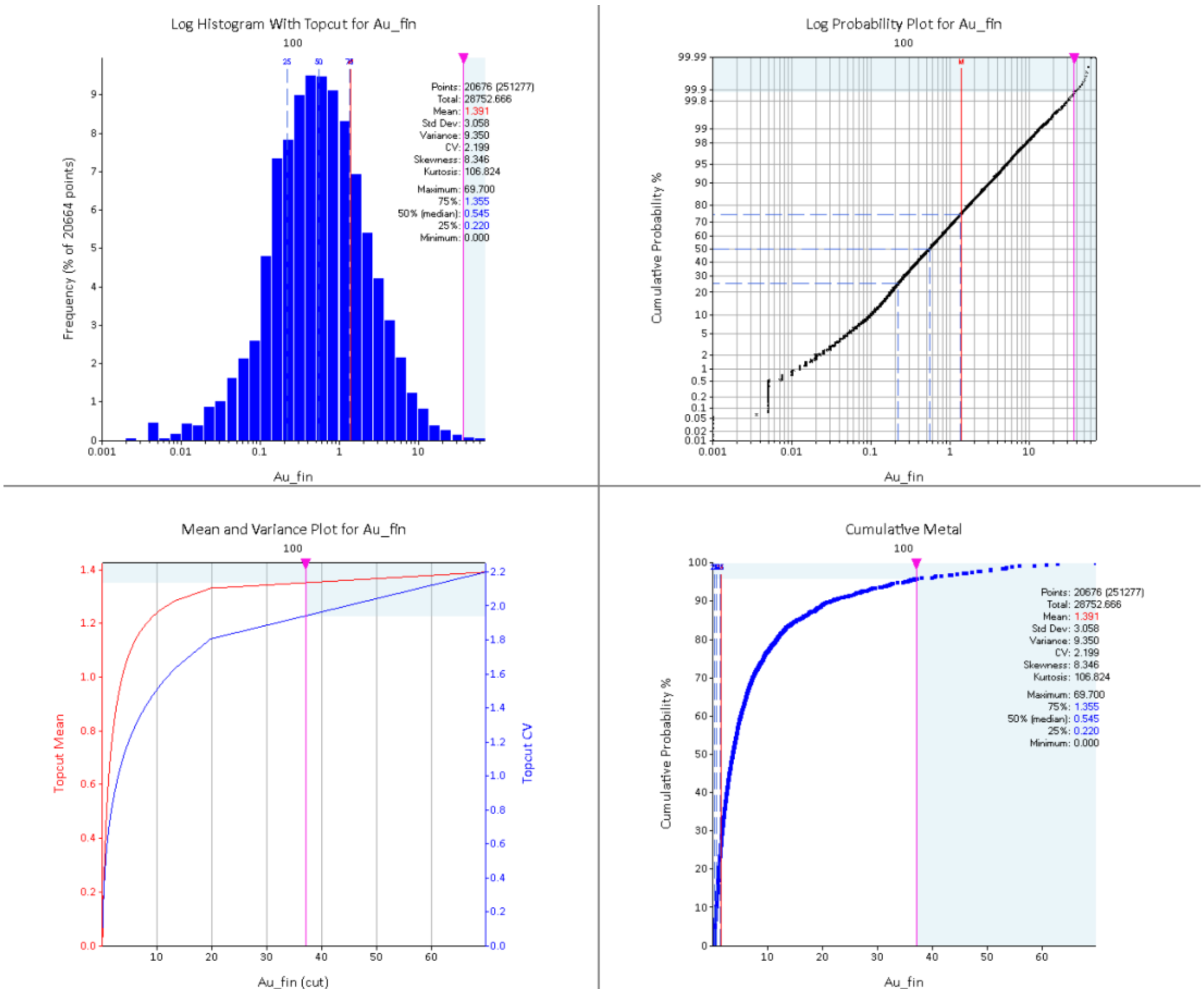


Figure 14-6: Capping analysis for Domain 100 – Piment and Prolongation

14.8 Variography

The author used Snowden Supervisor software (version 8.13) to calculate and model gold variograms for the mineralized domains. Variograms were computed on composited capped grade values producing variograms with a normalized sill value of 1.0. For each domain, two different spatial metrics were assessed: (1) traditional semi-variogram of gold, and (2) correlogram of gold and the most appropriate fit was selected. Downhole variograms were calculated to determine the nugget effect. The nugget effect is generally between 30% and 40% for all domains, with the nugget effect for domain 3300 (West Branch) and domain 100 (Piment and Prolongation) interpreted at 30%. The composite data was grouped for smaller domains with the details of the grouping presented in Table 14-12 and the variogram models for West Branch in Table 14-13. The variogram model for domain 3300 (West Branch) is presented in Figure 14-7 and shows continuity of 300 m x 100 m in the major and semi-major directions.

Table 14-12: Domain groupings for variogram analysis

Deposit	Variogram Code	Source of Data: Domains	Used for Estimation: Domains
West Branch	WB_3100	3100	3100
	WB_3200	3200	3200
	WB_3300	3300	3300
	WB_3400	3400	3400
	WB_3500	3500	3500
	WB_ENV	13100-13200, 13400-13500	13100-13200, 13400-13500
	WB_13300	13300	13300
Piment and Prolongation	PP_100	100	100
	PP_300	300	300
	PP_SMALL	200 and 400-1000	200, 400-1000 and 1900
	SAT	1100-1200	1100-1200
	PP_ENV	10100-11000	10100-11000
	SAT_ENV	11100-11200	11100-11200

Table 14-13: Gold variogram model results per domain – West Branch

Domain	Rotation Leapfrog	Structure	Type	Nugget	Major	Intermediate	Minor	Sill	Model
3100	50-80-150	1	Sph	0.4	25	10	4	0.4	Cor + Var
		2	Sph		200	100	8	0.2	
3200	50-80-150	1	Sph	0.4	30	10	4	0.5	Cor + Var
		2	Sph		150	80	8	0.1	
3300	50-80-150	1	Sph	0.3	35	30	5	0.35	Var
		2	Sph		300	100	25	0.35	
3400	50-80-150	1	Sph	0.4	25	10	4	0.4	Cor + Var
		2	Sph		150	110	8	0.2	
3500	50-80-150	1	Sph	0.4	17	17	4	0.5	Cor + Var
		2	Sph		120	120	8	0.1	
13100+13200+13400+13500	50-80-150	1	Sph	0.5	70	20	5	0.4	Var
		2	Sph		100	100	7	0.1	
13300	50-80-150	1	Sph	0.5	60	35	4	0.4	Var
		2	Sph		250	100	5	0.1	

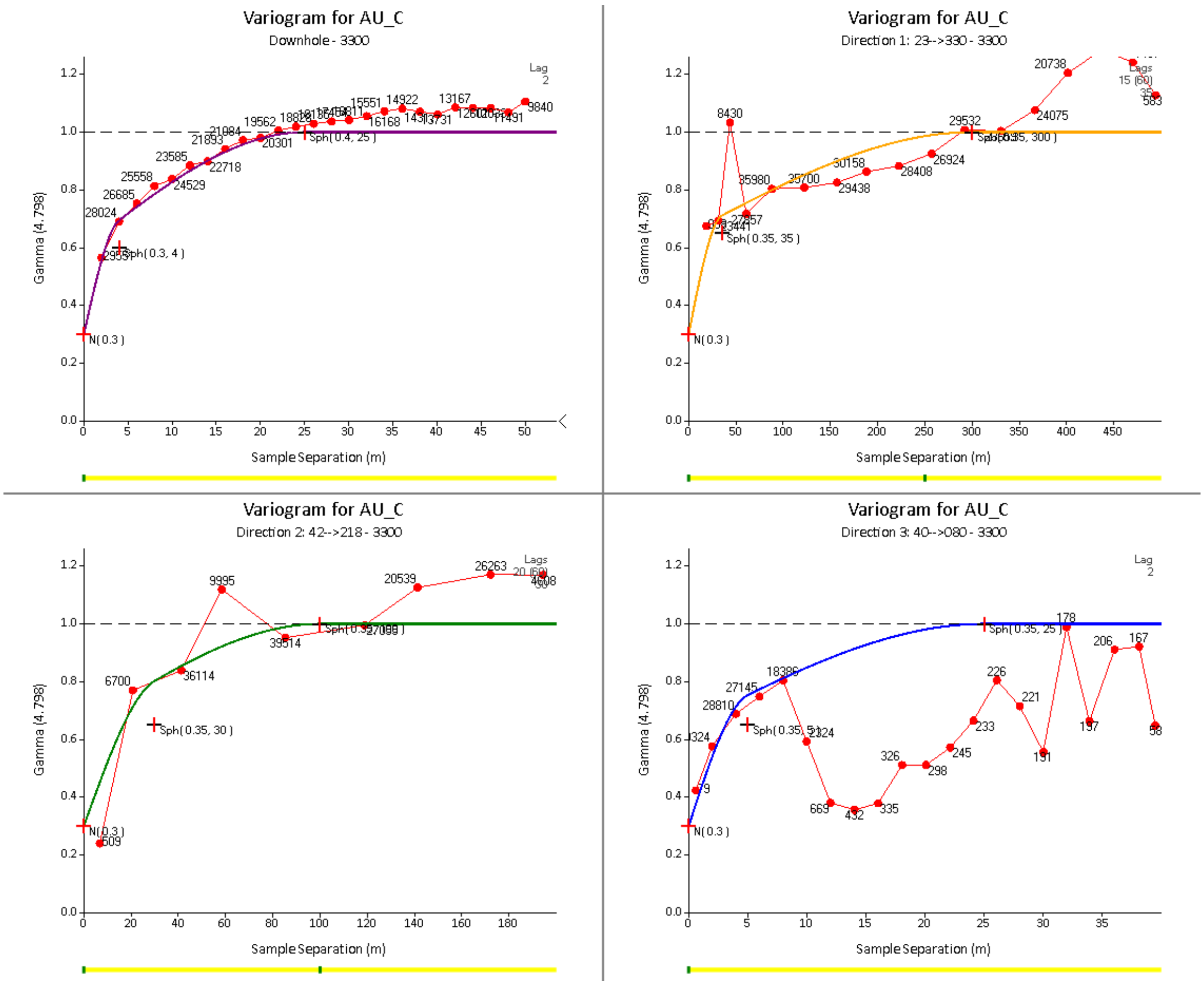


Figure 14-7: Gold variogram for Domain 3300 – West Branch

14.9 Density

Consistent with the previous modelling rationale, the combination of lithology and redox was used to assign the density values to the block model. The values assigned are based on an extensive study conducted on the exploration data and were not changed since the previous estimation. The quality of the density database was found to be suboptimal, however, and the direct reproduction of the assigned density values from the drilling data was not possible. This does not appear to be a material risk to tonnage estimates as the deposit was in operation for many years with satisfactory results for tonnage reconciliation with the previous models. The assigned density values are presented in Table 14-14 and Table 14-15.

Table 14-14: Assigned density values (t/m^3) – West Branch

Oxidation	Lithology						
	Dykes	GDI	BIF HW	BIF FW	FVC	SVC HW	SVC FW
Oxide	2.81	2.38	2.80	2.53	2.15	2.46	2.35
Transitional	2.89	2.72	2.89	2.85	2.64	2.65	2.64
Fresh	2.98	2.87	3.27	3.18	2.68	2.90	2.76

Table 14-15: Assigned density values (t/m^3) – Piment and Prolongation

Oxidation	Lithology				
	BIF	FVC	Dykes	SGW	SSL
Oxide	3.00	2.39	3.00	2.57	2.89
Transitional	3.05	2.66	3.00	2.66	2.89
Fresh	3.14	2.72	3.00	2.80	2.89

14.10 Search Strategy and Estimation Parameters

The block model was populated with estimated gold grades using OK in the mineralized envelope and sub-domains, applying up to three estimation runs with progressively relaxed search ellipsoids and data requirements. The unmineralized domain (3999) was estimated using ID^2. All passes use an ellipsoidal search based on the variogram model approach and for domains 100, 300, and 500 (Piment and Prolongation) dynamic anisotropy, based on a structural trend, was used to conform to the varying orientation of the modelled zones. In all cases, gold was estimated using a hard boundary. Furthermore, an additional limited search radii estimation was used after detailed calibration of the model performance to the short-term model. Table 14-16 and Table 14-17 summarizes the search ellipse dimensions and sample selection plan per domain.

Table 14-16: Search ellipse dimensions

Deposit	Estimation Domain	Variogram Range				Search Ellipse				
		X	Y	Z	Logic Run 1	X	Y	Z	Logic Run 2	Logic Run 3
Piment and Prolongation	PP_100	180	70	25	0.9 sill	100	40	30	2x	5x
	PP_300	100	65	25	0.95 sill	60	35	30	2x	5x
	PP_ENV	70	70	20	1 sill	70	70	30	2x	10x
	PP_SMALL	115	65	25	1 sill	115	65	30	2x	10x
	SAT	65	65	15	1 sill	65	65	30	2x	5x
	SAT_ENV	40	40	10	1 sill	40	40	30	2x	5x
West Branch	WB_3100	200	100	8	0.95 sill	100	50	40	2x	5x
	WB_3200	150	80	8	0.95 sill	60	40	40	2x	5x
	WB_3300	300	100	25	0.90 sill	150	60	40	2x	5x
	WB_3400	150	110	8	0.95 sill	100	65	40	2x	5x
	WB_3500	120	120	8	0.95 sill	60	60	40	2x	5x
	WB_ENV	100	100	7	0.95 sill	60	60	40	2x	10x
	WB_13300	250	100	5	0.95 sill	120	60	40	2x	10x
	3999					50	50	30		

Table 14-17: Sample selection plan

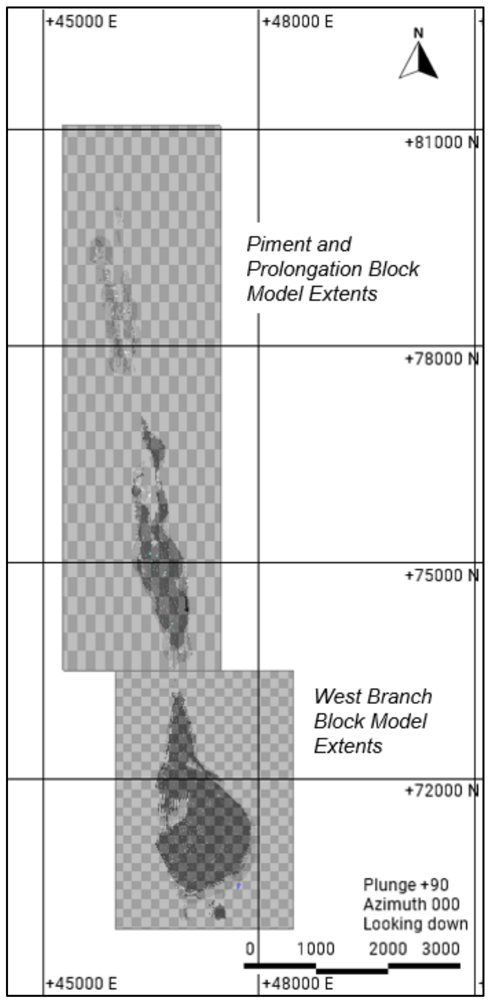
Deposit	Estimation Domain	Min. – Max. Samples			Max. Comps/Hole	HG Limited Radii
		Run 1	Run 2	Run 3		
Piment and Prolongation	PP_100	7-15	4-18	2-24	3	15 m @8 g/t limited search
	PP_300	7-15	4-18	2-24	3	
	PP_ENV	7-15	4-18	2-24	3	
	PP_SMALL	7-15	4-18	2-24	3	
	SAT	7-15	4-18	2-24	3	
	SAT_ENV	7-15	4-18	2-24	3	
West Branch	WB_3100	11-25	6-30	2-40	5	12 m @10 g/t limited search
	WB_3200	11-25	6-30	2-40	5	
	WB_3300	11-25	6-30	2-40	5	
	WB_3400	11-25	6-30	2-40	5	
	WB_3500	11-25	6-30	2-40	5	
	WB_ENV	11-25	6-30	2-40	5	
	WB_13300	11-25	6-30	2-40	5	
	3999	2-25			5	

14.11 Block Model

The block models for both deposits were developed without the overlap to avoid a double accounting for the mineralized tonnage. The West Branch model covers a 500 m deeper extent than the 2019 model and the updated Piment and Prolongation model. The block dimensions reflect the corresponding drilling grid with sub-blocking parameters selected to provide improved filling of the estimation domains and a good resolution for potential underground planning. No rotation was applied to the block model. Table 14-18 summarizes the block model and Figure 14-8 shows the block model extents for West Branch and Piment and Prolongation.

Table 14-18: Block model dimensions

Deposit	Origin			Blocks			Block Size (m)			Min. Sub-block (m)		
	X	Y	Z	X	Y	Z	X	Y	Z	X	Y	Z
Piment and Prolongation	45,280	73,510	4400	440	756	150	5	10	5	1.25	2.50	1.25
West Branch	46,000	69,930	3,900	248	358	125	10	10	10	2.50	2.50	2.50



Source: SLR 2024.

Figure 14-8: Block model extents

14.12 Block Model Validation

The estimated block model was validated using a combination of methods including visual comparison of block estimates and informing composites (Figure 14-9 to Figure 14-11), swath plots, and change-of-support checks using the nearest neighbour (NN) de-clustered distribution as well as visual and statistical validation against the short-term grade control model.

The along strike swath plot for domain 3300 and domain 100, showing the ordinary kriged block model with the NN de-clustered composites, is presented in Figure 14-12. The swath plot shows generally good agreement between the kriged estimation and the NN de-clustered data. The quantile-quantile (QQ) plot, presented in Figure 14-13, shows the change-of-support verification for domain 3300. The plot shows the ordinary kriged estimate (Y-axis) corresponds well to the de-clustered change-of-support corrected distributions (X-axis). Mineral Resources at West Branch, Piment, and Prolongation were audited by SLR in 2024.

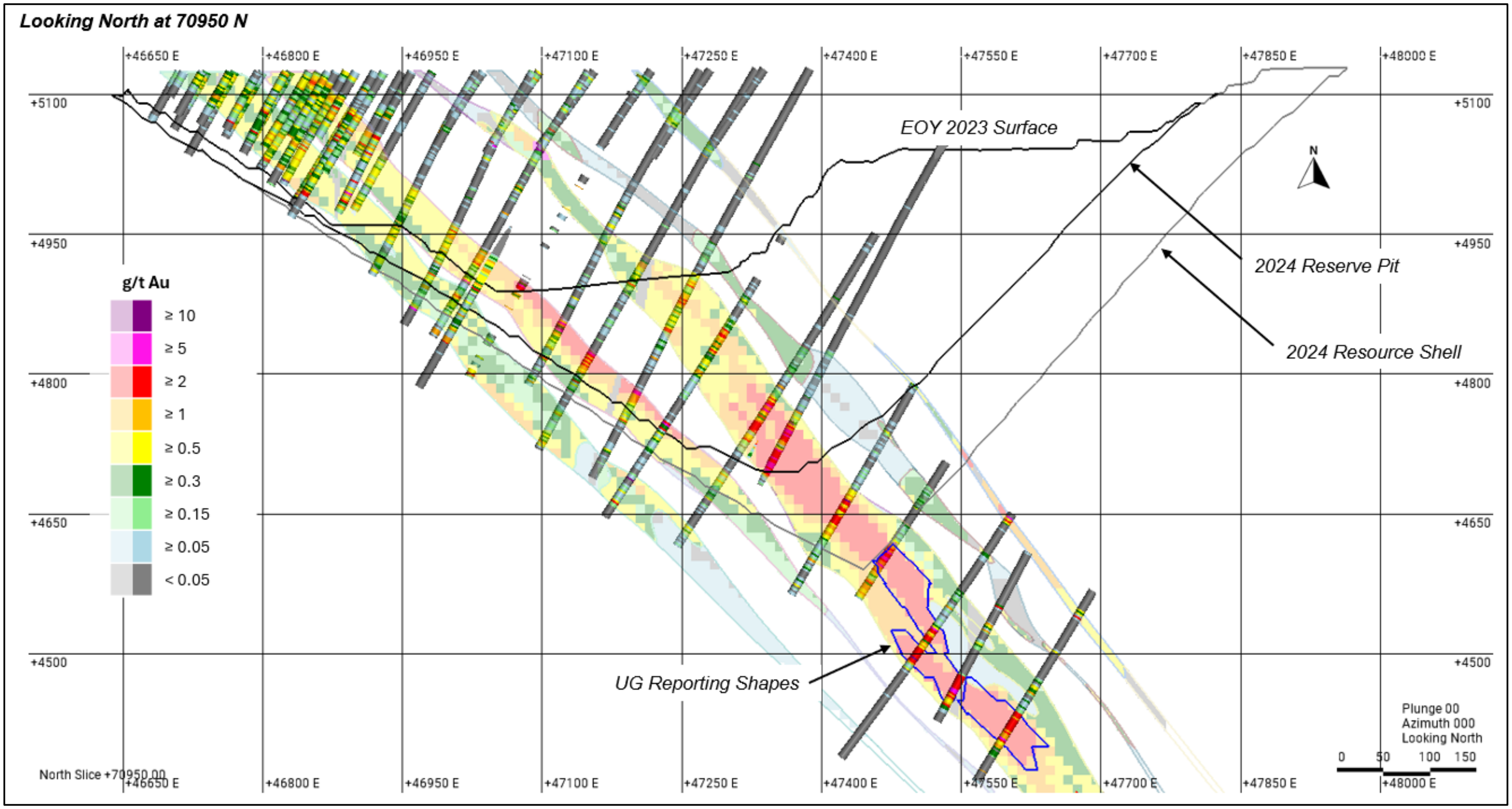


Figure 14-9: Visual comparison of West Branch composite and block model gold grades

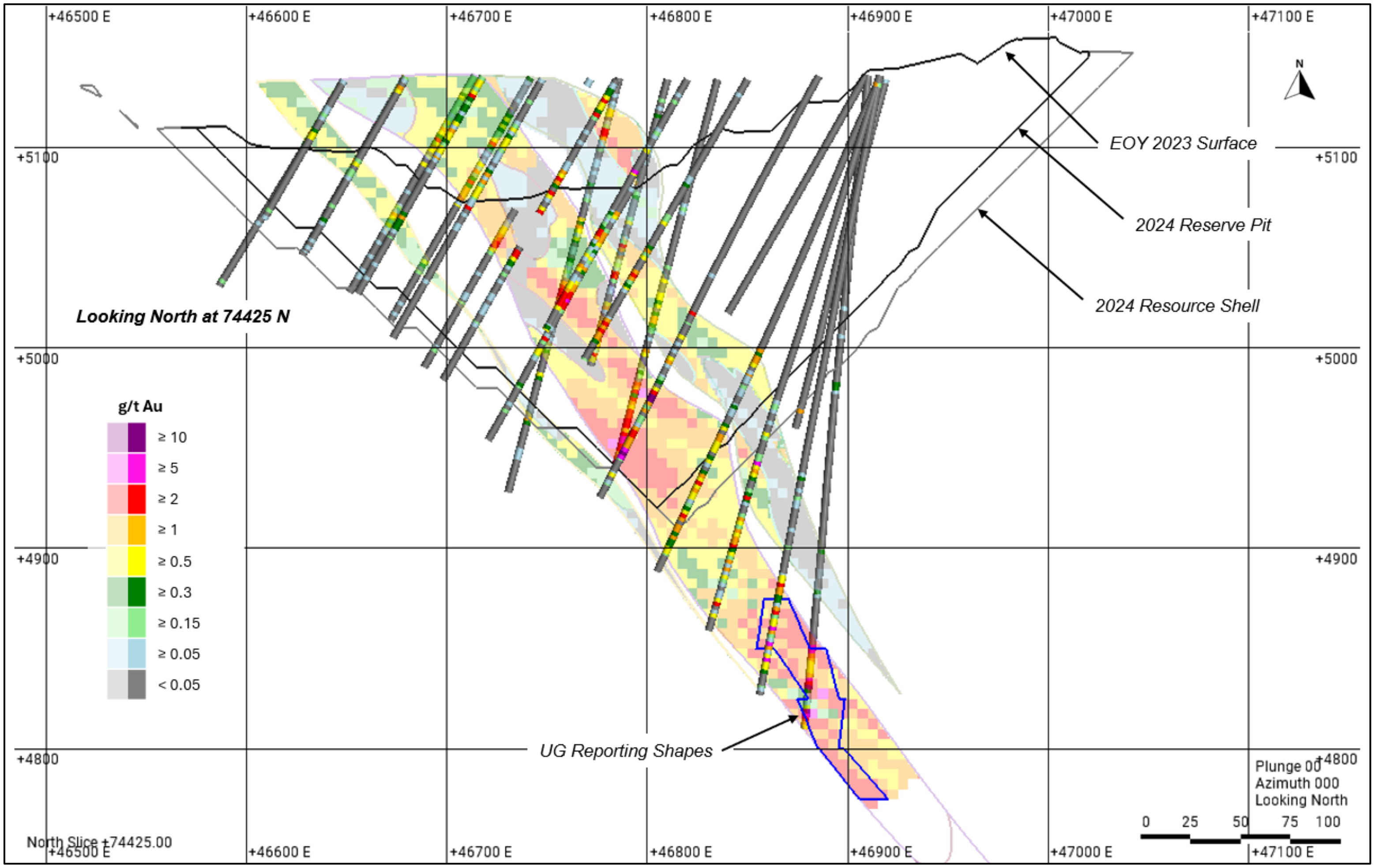


Figure 14-10: Visual comparison of Piment composite and block model gold grades

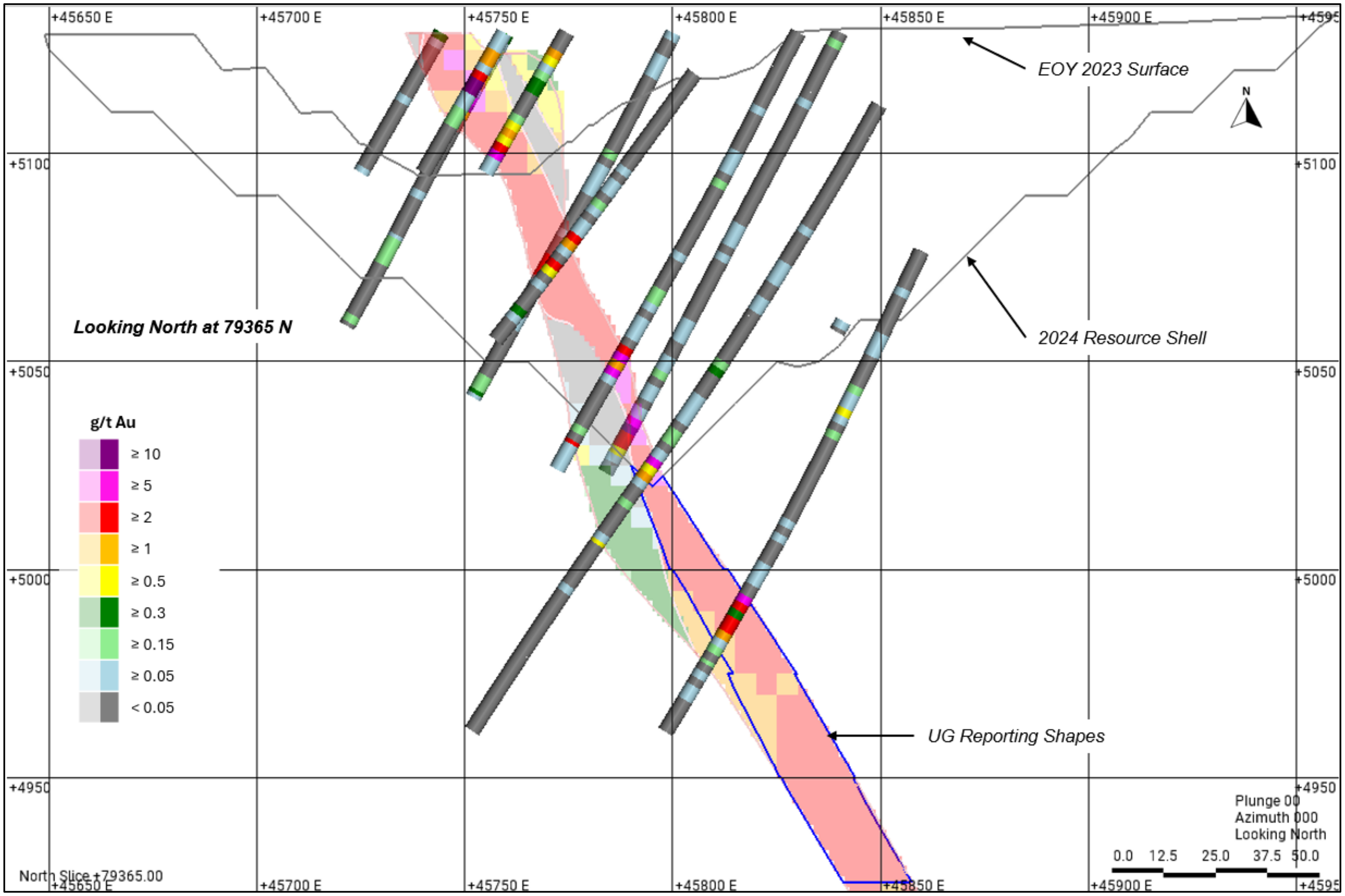
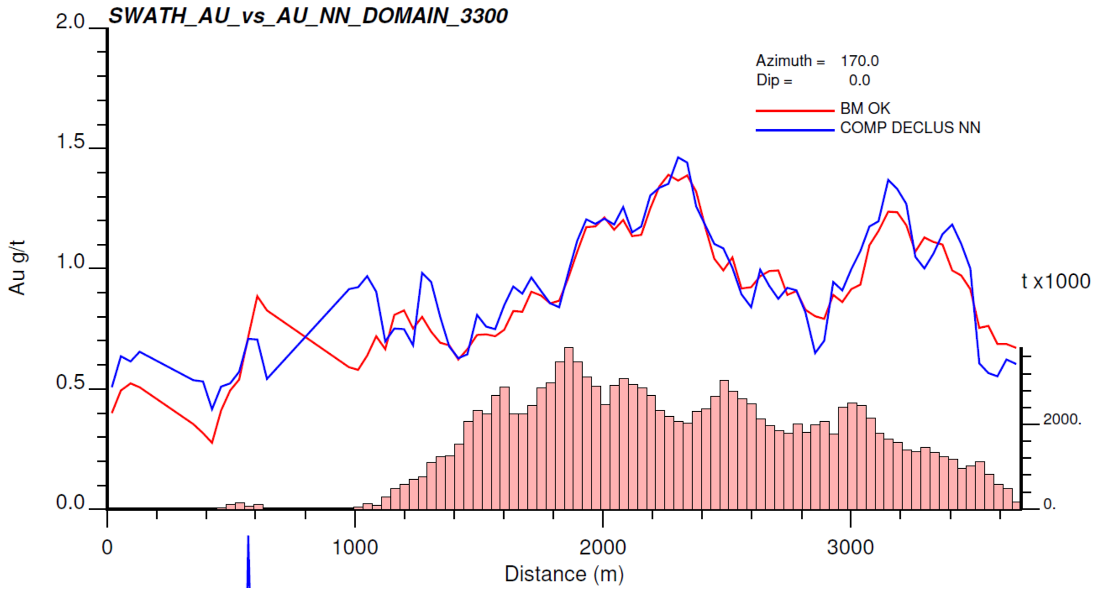
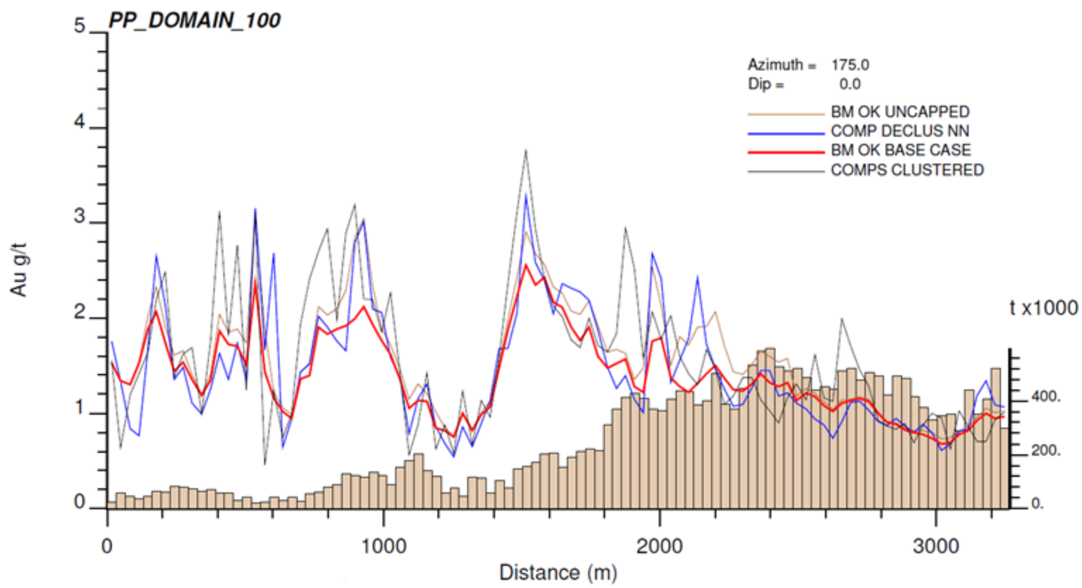


Figure 14-11: Visual comparison of Prolongation composite and block model gold grades





Notes:

1. Histogram corresponds to block model volume along the swath

Figure 14-12: Swath plot for Domain 3300 and Domain 100

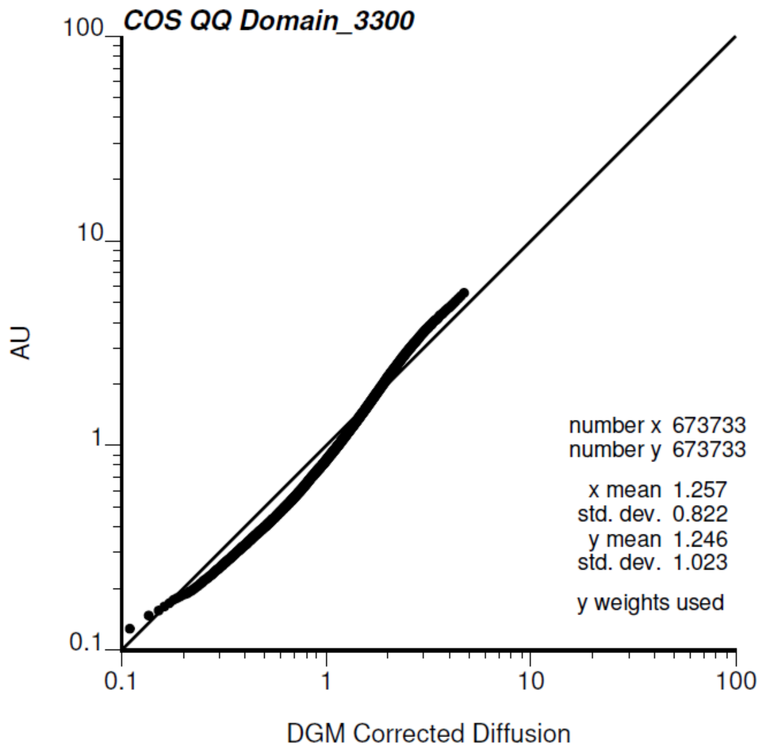


Figure 14-13: Change of support QQ plot for Domain 3300 – West Branch

The block models for both deposits were also compared with the short-term models estimated using the grade control data. The comparison between the 2022 and 2019 resource block models and the short-term grade control model for West Branch, at a cut-off of 0.7 g/t and covering the mining area between 2019 and 2022, is presented in Figure 14-14 and Figure 14-15. For both deposits the 2022 resource model demonstrates reasonable comparison with the grade control model, with metal difference at the reporting cut-off of 0.70 g/t gold, not exceeding 1% in contained ounces.

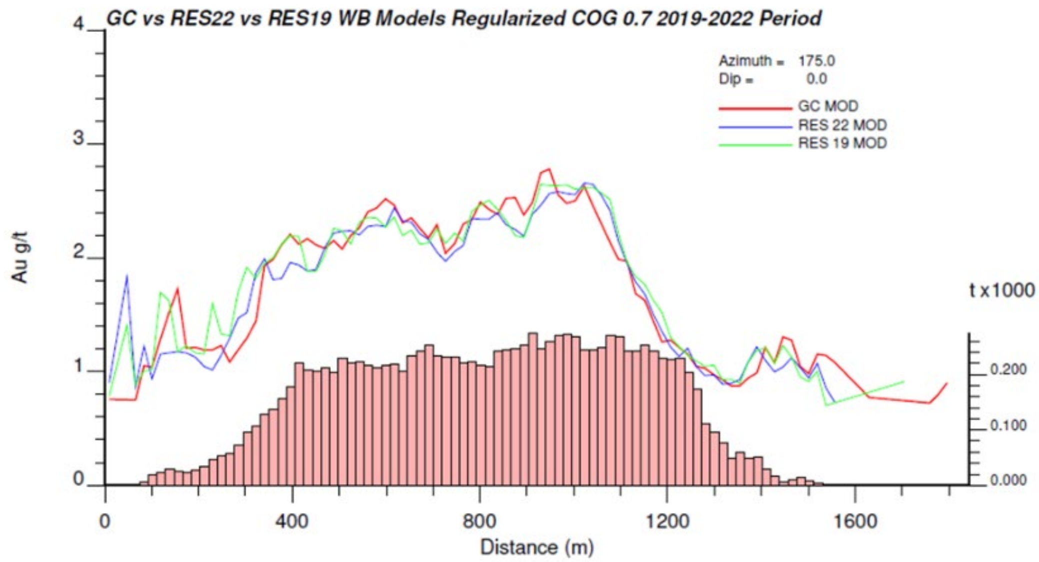


Figure 14-14: Swath plot comparing the resource models and short-term model – West Branch

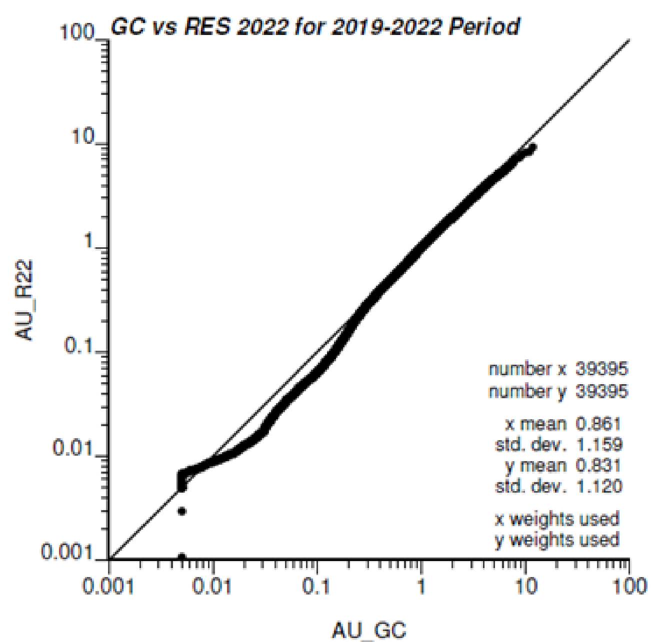


Figure 14-15: QQ plot comparing the 2022 resource model and short-term model – West Branch

14.13 Classification

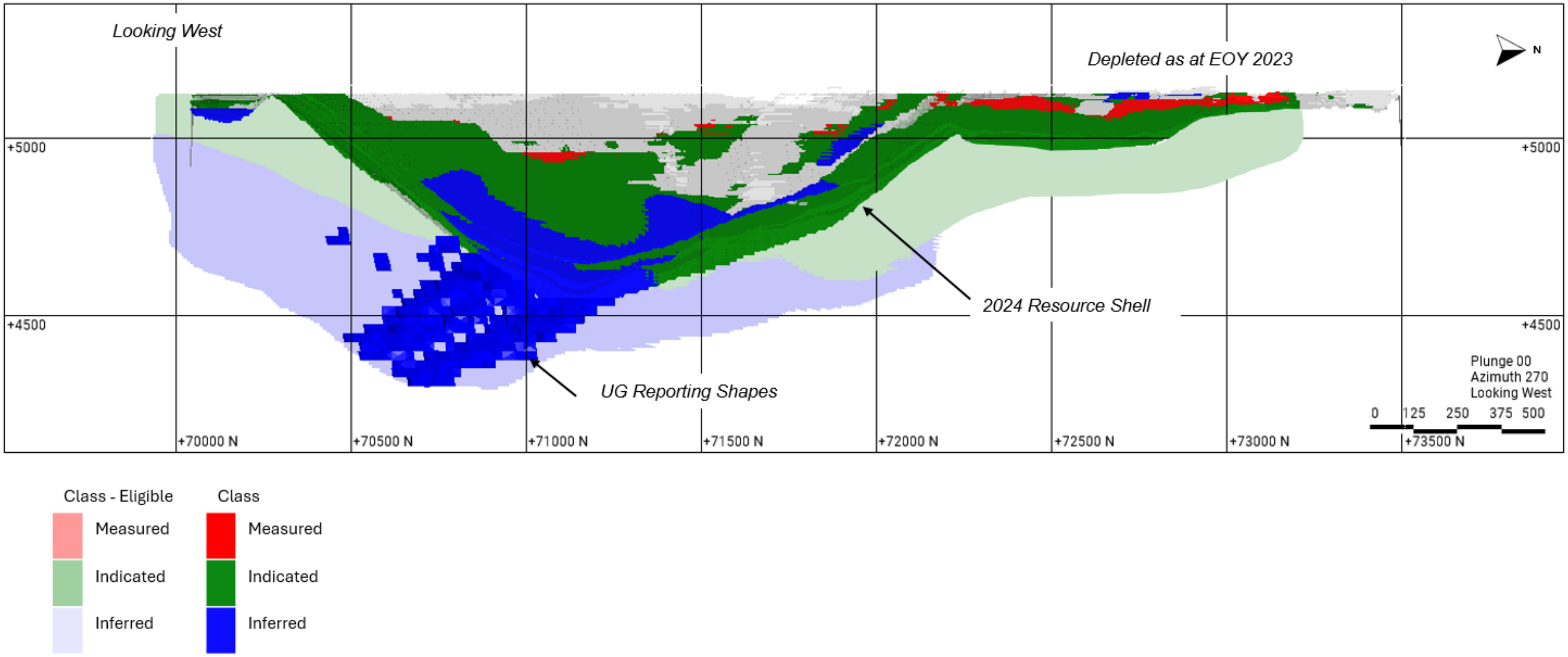
Definitions for Mineral Resource categories used in this Technical Report are consistent with those defined by CIM (2014) and adopted by NI 43-101. In the CIM classification, a Mineral Resource is defined as “a concentration or occurrence of solid material of economic interest in or on the Earth’s crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction”.

Mineral Resources are classified into Measured, Indicated, and Inferred categories. A Mineral Reserve is defined as the “economically mineable part of a Measured and/or Indicated Mineral Resource” demonstrated by studies at Pre-Feasibility or Feasibility level as appropriate. Mineral Reserves are classified into Proven and Probable categories.

The classification strategy considers the confidence in geology continuity, drill hole spacing, and proximity to the current mining areas and grade control drilling. To assist in categorization, the separate block model was created and then post-smoothed to ensure continuity within the classification categories. The classified block model was then examined visually on plans and sections. A summary of the classification criteria is presented in Table 14-19 and the West Branch classification wireframes together with the block model, filtered at one gram per tonne of gold, are shown in Figure 14-16. Piment and Prolongation are shown in Figure 14-17.

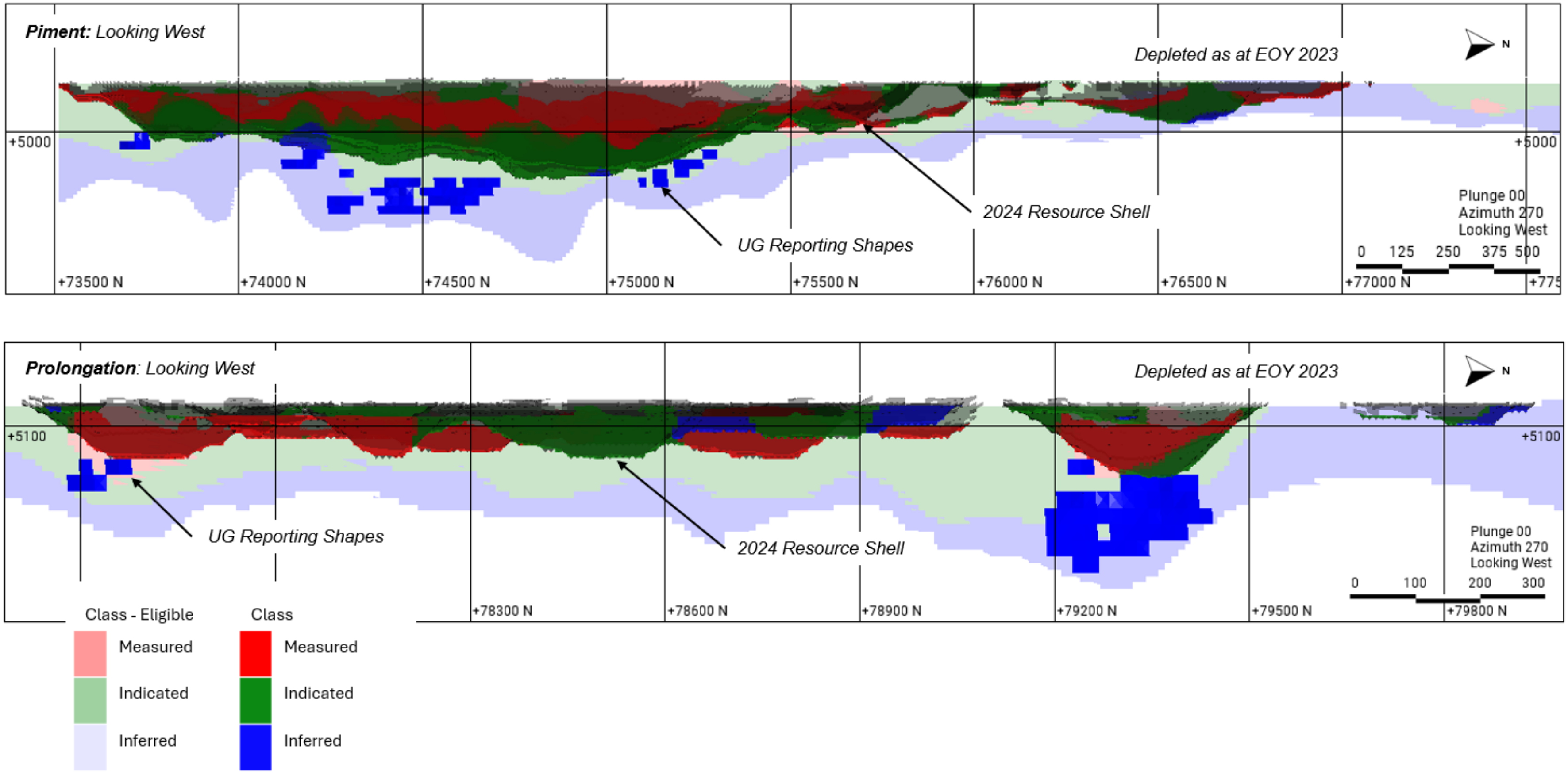
Table 14-19: Classification criteria

Deposit	Criteria	Measured	Indicated	Inferred	Unclassified
Piment and Prolongation	Geology Continuity	Domains 100-800 and 10100-10800	Domains 100-900 and 10100-10900	All domains except 1900 and 1999	All domains
	Geometry Criteria	3 holes in 25 m buffer Equivalent to 30-35 m drill spacing	3 holes in 50 m buffer Equivalent to 70 m drill spacing	2 holes in 60 m buffer Equivalent to 120 m drill spacing	The rest of the mineralized material
	Production Data	Approx. 30 m buffer from GC drilling	NA	NA	NA
West Branch	Geology Continuity	All domains except 3999			
	Geometry Criteria	3 holes in 25 m buffer Equivalent to 30-35 m drill spacing	3 holes in 45-50 m buffer Equivalent to 70 m drill spacing and 60 m for disseminated domains (not 3300)	2 holes in 75 m buffer Equivalent to 150 m drill spacing	The rest of the mineralized material
	Production Data	Was not explicitly used because already corresponds well with the geometry criteria			



Source: SLR 2024.

Figure 14-16: Classification – West Branch



Source: SLR 2024.

Figure 14-17: Classification – Piment and Prolongation

15. MINERAL RESERVE ESTIMATE

The Mineral Reserve for the Tasiast open pit mine was estimated using a planning model derived from the resource models for West Branch (WB), for Piment (PM), and Fennec (FN) models, as discussed in Section 14.

The 24 kt/d pre-feasibility Mineral Reserves, effective December 31, 2024, were estimated using the mine planning block model and using parameters as shown in Table 15-2. The Mineral Reserve estimate includes material contained within the final pit design that can be extracted and processed economically. Reported reserves are solely based on the Measured and Indicated mineral resource classifications, which correspond to Proven and Probable reserves classifications.

Mineral Reserves for the Tasiast Mine, including the West Branch, Piment, and Fennec deposits, are presented in Table 15-1.

Table 15-1: Tasiast Mineral Reserve Estimate as at December 31, 2024

Deposit Classification	Tonnage (kt)	Grade (g/t Au)	Contained Gold (koz Au)
West Branch			
Proven	1,889	1.12	68
Probable	34,683	1.93	2,147
West Branch Total	**36,571**	**1.88**	**2,216**
Piment			
Proven	12,930	1.38	572
Probable	8,875	1.57	447
Piment Total	**21,804**	**1.45**	**1,019**
Fennec			
Proven	0	0.00	0
Probable	1,913	1.79	110
Fennec Total	**1,913**	**1.79**	**110**
Stockpile(s)			
Proven	42,542	0.99	1,361
Stockpiles Total	**42,542**	**0.99**	**1,361**
Combined			
Proven Subtotal	57,361	1.08	2,000
Probable Subtotal	45,471	1.85	2,705
Combined Total	**102,831**	**1.42**	**4,705**

Notes:

1. CIM (2014) definitions were followed for Mineral Reserves.
2. Mineral Reserves are limited to blocks within the reserve pit design and within mineable panels/polygons within which blocks meet an average cut-off grade of 0.6 g/t Au and consider ore loss, dilution, and mining selectivity.
3. Mineral Reserves are estimated using an average long-term gold price of US$1,600 per ounce.
4. Bulk density is estimated and averages 2.91 t/m^3.
5. Numbers may not add due to rounding.

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

15.1 Basis of Reserve Estimate and Pit Optimization

An economic pit shell generated at a gold price of US$1,600/oz, with cost criteria, metallurgical recoveries, and geologic and geotechnical considerations guides the final pit design. The economic pit shell used to define the final pit limits was created using Datamine's NPV Scheduler software (NPVS). NPVS utilizes the Lerchs-Grossman (LG) algorithm to define blocks that can be mined at a profit. The program then creates an economic shell based on the following information:

- Starting topography
- Overall slope angles by rock type
- Metallurgical recoveries by mineralization and rock type and oxidation state
- Geologic grade model with gold grades, density, lithology, and mineral types
- Process (including rehandle) and mining costs
- Incremental vertical bench mining cost
- Downstream costs, such as gold refining, freight, and marketing
- Sustaining capital for future equipment replacements

The Mineral Reserve Estimate was prepared using the December 31, 2024 topography and the parameters detailed in Table 15-2.

Table 15-2: Pit optimization parameters

LG Parameter	Cost / Assumption	Unit
Piment (PM) Mining Unit Cost	2.58	US$/t
PM Haulage Increment per Bench	0.045	US$/t/bench
West Branch (WB) Mining Unit Cost	2.58	US$/t
WB Haulage Increment per Bench	0.045	US$/t/bench
Total Mining Unit Cost	3.48	US$/t
Mining Sustaining Capital	0.24	US$/t
Processing Unit Cost	15.61	US$/t ore
Site Admin Unit Cost	8.23	US$/t ore
Processing Sustaining Capital	1.55	US$/t ore
Stockpile Re-handle	0.90	US$/t ore
Gold Price	1,600	US$/oz
Total Selling Cost (including below)	148.89	US$/oz
Royalties	8.22	%
Payable	99.95	%
Refining Charges and Fees	5.47	US$/oz
Reclamation Cost	11.06	US$/oz
Discount Rate	5.0	%

Mineral Reserves are stated within an ultimate pit design at cut-off grades that are based on the process type, operating costs, and metallurgical recovery.

Slope parameters based on geotechnical considerations were applied to the pit design along with ramps and geotechnical catch benches, and were subsequently used to generate overall slope angles. The overall slope angles used in pit optimization are shown in Table 16-4 and Table 16-5.

Gold recovery is determined by ore type and process method. The gold recovery is calculated from the information in Table 15-3 where gold grade is expressed in grams per tonne (g/t).

Table 15-3: Process Recoveries

Process	Au Recovery
CIL Recovery	(Au-MAX(Au*(1-0.25)*(1-(1.6772*LN(Au*(1-0.25))+90.8)/100),0.04+0.015*(1-0.6)/0.6))/Au [1]

Notes:

1. Head grade vs. recovery relationship develop from test work.
2. Au = head grade, g/t.

The mine operating costs used for pit optimization include ongoing major mine equipment capital costs. The mine equipment sustaining capital was used in the economic model to simulate mine capital expenditures when generating the economic pit.

The top-down discount method was used during pit optimization. This is a procedure based on multiplying the block value by a discount factor that is a function of the annual cost of capital, an estimate of the average annual vertical advance rate of mining, and the relative depth of the block. This method simulates the actual mine plan discounted cash flow that is burdened with up front stripping costs and aids in the selection of a higher value pit.

16. MINING METHODS

16.1 Mining Operations

The Complex is located in northwestern Mauritania, approximately 300 km north of the capital Nouakchott and 250 km east-southeast of the port city of Nouâdhibou. The Tasiast Permit Area is in the Inchiri and Dakhlet Nouâdhibou districts.

The Complex is located in a remote part of the Sahara Desert, consisting largely of flat barren plains covered by stony surface with some sand and soil, interspersed with occasional sand dunes and upstanding outcrops of bedrock. The average elevation is approximately 130 masl. Vegetation is sparse and consists primarily of grasses and occasional acacia trees. The climate is Saharan with an average rainfall of 90 mm, most of which falls from July to September. The climate is hot most of the year ranging from 10°C to 45°C and experiences strong prevailing NE-SW winds from the Sahara and occasional reverse SW-NE winds from the Atlantic.

The main ore hosting lithology is a Granodiorite Intrusive (GDI) with lesser contributions from the FVC (Felsic Volcaniclastic) and BIM (Banded Iron Magnetite). The orebody strikes at approximately 350° and dips easterly at approximately 50°, true width is approximately 40 m.

Ore and waste rock is mined in 10 m benches by conventional open pit methods from the West Branch, Piment, and Fennec deposits. A view of the current West Branch pit is presented in Figure 16-1. Tasiast currently operates a haulage fleet of 45 Caterpillar 793D (220 t), five Hitachi EH-4000 (220 t), and two Komatsu HD785 (92 t). The haulage fleet is primarily loaded by six Caterpillar 6060 shovels and two Bucyrus RH340B excavators, with three Caterpillar 994 front-end loaders utilized for rehandle purposes and four Komatsu PC1250s for auxiliary loading of the smaller Komatsu trucks. Blasting techniques, including presplit and buffer hole blasting, are employed to protect the pit walls. The grinding circuit produces a product size of 80% passing 90 µm which is processed in a conventional CIL circuit to produce gold bullion. Gold recovery averages 93%. Tailings slurry from the CIL process is pumped to the tailings storage facility (TSF).

Commercial production of gold at Tasiast began in January 2008 under Red Back. As of December 31, 2024, approximately five million ounces have been produced by Tasiast. Prior mining has taken place in West Branch, Piment, and several smaller pits at Tasiast. From late 2010 when Kinross acquired the property to the end of 2024, a total of 997 Mt of material have been mined from various pits, including 61 Mt in 2022, 63 Mt in 2023 and 71 Mt in 2024.

Figure 16-1: West Branch Pit - looking southwest

The current mill operates at approximately 24 kt/d. Ore is fed directly from the mines and stockpiles to the primary crusher.

Cut-off grades are based on the net block value and cut-over grades applied during scheduling. Applying cut-over grades during scheduling ensures that the highest value materials are routed to the CIL process over time. Lower grade materials are routed to stockpiles. All materials below the cut-off grade are sent to waste destinations. The grades and potential destinations used for strategic planning are shown in Table 16-1.

Table 16-1: Material routing

Grade Bin (g/t)	Identification	Potential Stockpile
2.0+	Super High Grade (SHG)	“Purple” Stockpiles
1.5-2.0	High Grade (HG)	“Red” Stockpiles
1.1-1.5	Medium Grade (MG)	“Blue” Stockpiles
0.7-1.1[1]	Low Grade (LG)	“Green” Stockpiles
0.4-0.7	Mineralized Waste	“Sub-Green” Stockpiles

Notes:

1. Mineral Reserves cut-off grade has since been updated to 0.6g/t Au (as described in Section 15).

During the expansion of the processing plant to 24 kt/d, one Caterpillar 6060 shovel and five Hitachi EH-4000 haul trucks were added to the primary mobile fleet to support production. Further to the plant expansion, satellite pit mining at Piment and Fennec includes an additional eight Caterpillar 777 haul trucks (arriving in 2025), one Caterpillar 6030 shovel (2025), and 1 Caterpillar 992 front-end loader (2026). Estimates of future equipment utilization are based on current operating practices, general and site experience, and the following considerations:

- Operation of several mining faces simultaneously to meet the long-range schedule and requirements.
- Use of large-scale mining equipment to lower operating costs.
- Use of well-proven and advanced mine equipment technologies to improve performance.
- Use of component replacement and preventive maintenance practices to minimize major equipment failures.
- Design of constraints to accommodate the selected fleets in both direct production and support roles.

The existing primary production equipment and satellite pit fleet will be used for the duration of mining and no replacement of this equipment is anticipated, however, there will be an addition of two Komatsu PC1250 excavators, one Sandvick D650i drill, and one Caterpillar MD6250 drill. Equipment life has been projected from actual operating hours, with estimates of future usage based on the mine plan. The current mining fleet at Tasiast is shown in Table 16-2, as well as the LOM peak unit usage (including future purchases).

Table 16-2: 24 kt/d total mobile equipment fleet

Mobile Equipment	Units Available in 2025	LOM Peak
Hauling		
Caterpillar 793D	45	45
Hitachi EH-4000	5	5
Komatsu HD785	2	2
Caterpillar 777	8	8
Loading		
Caterpillar 6060	6	6
Bucyrus RH340	2	2
Komatsu PC1250	4	6
Caterpillar 994	3	3
Caterpillar 6030	1	1
Caterpillar 992	0	1
Drilling		
DR580/D560i	7	8
Caterpillar MD6250	6	7
Bucyrus SKFX	3	3
Atlas PV 235	1	1

16.2 Mine Design

The Tasiast WB and PM final pit designs consist of a series of pits that extend along a strike length of approximately eight kilometres. The configuration of the mining area is shown in Figure 16-2. Both WB and PM deposits are actively being mined in 2025. The Fennec satellite deposit is located approximately 10 kilometres north of the CIL plant.

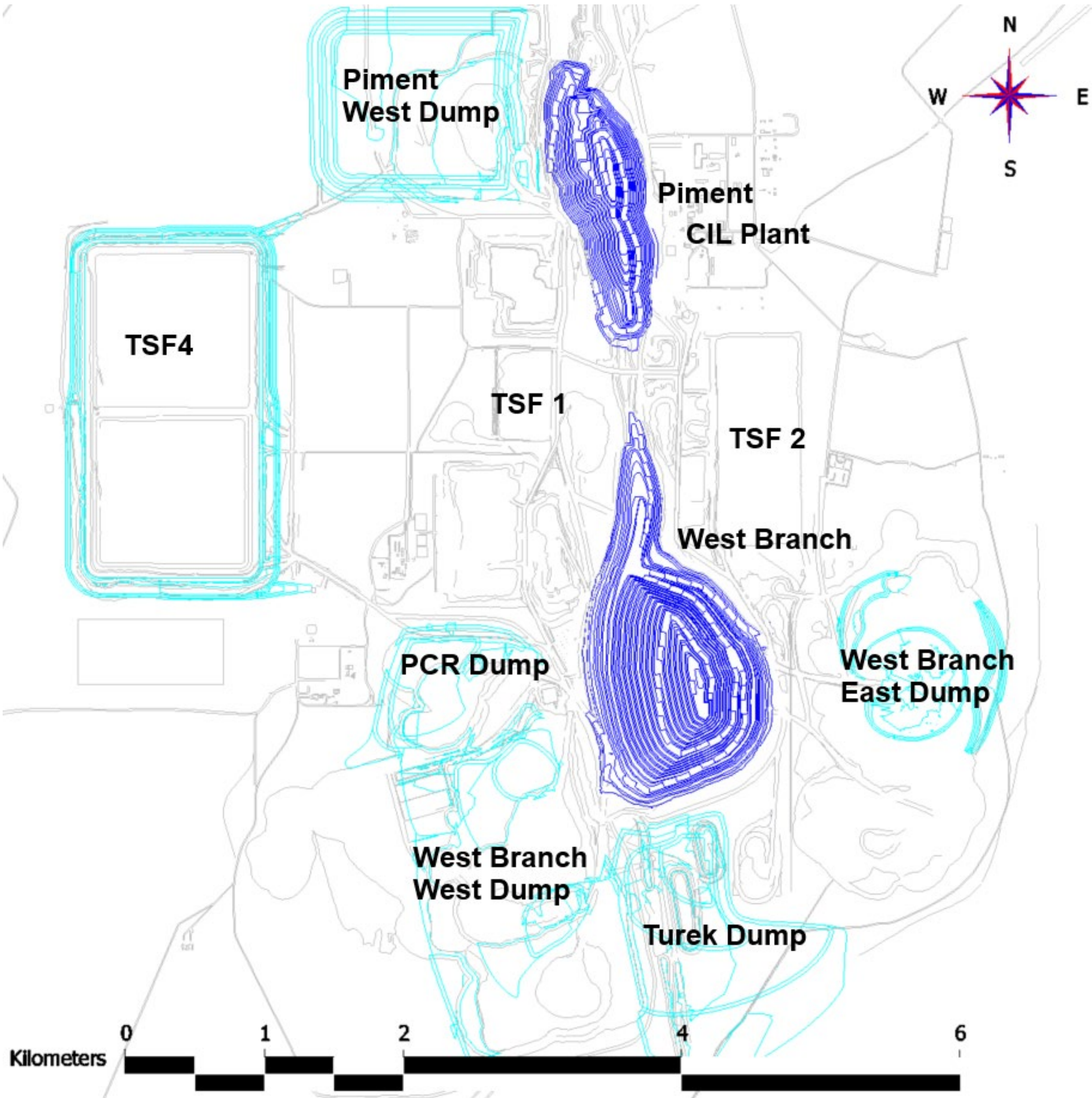


Figure 16-2: Plan view of Tasiast site

Geotechnical Considerations

Historically, the Complex has been divided into two geotechnical zones where the Piment Zone is north of approximately 72,000 N and the West Branch Zone is south of 72,000 N. Overall pit slope angles and inter-ramp angles for the Piment Zone were initially determined by Scott Wilson Mining UK (Scott Wilson) in 2009 and subsequently optimized by Stacey Mining Geotechnical Ltd (Stacey) in 2011. The slope angles that are applied at Piment are shown in Table 16-4.

Golder Associates, the engineer of record for Tasiast, generated the slope design parameters for West Branch as summarized in Table 16-5. These slope parameters are based on the assumption that appropriate practices in dewatering or depressurization, drilling and blasting, and movement monitoring are effectively implemented and carried out. The design criteria that have been used in the study of the open pit stability are based on the best practice guidelines presented in Table 16-3. For overall slope failure, a target factor of safety (FoS) of 1.3 was used in the slope stability analysis.

Table 16-3: Slope FoS Criteria

Slope Scale	Consequence of Failure	Static Acceptance Criteria (FoS)	Dynamic Acceptance Criteria (FoS)
Bench	Low - High	1.1	NA
	Low	1.15 - 1.2	1.0
Inter-ramp	Medium	1.2	1.0
	High	1.2 - 1.3	1.1
	Low	1.2 - 1.3	1.0
Overall	Medium	1.3	1.05
	High	1.3 - 1.5	1.1

Slope stability analysis has been carried out to provide the FoS mentioned above for slopes within the pit. Two methods of analysis have been employed incorporating both Limit Equilibrium and Finite Element numerical modelling methods. The conclusions of the slope stability assessment are as follows:

- All slopes meet the target design criteria;
- Good correlation has been seen between the two methods of analysis;
- Safety Factors on the Footwall indicate there may be potential to undercut the foliation, however on balance this is not recommended because of the possible risk of slope failure if the incipient foliation planes open;
- Safety Factors for the Hanging Wall indicate there may be potential for an increase in the Inter Ramp Angle (IRA); and

- Tabular bench scale failure is to be expected on the Footwall where the Bench Face Angle (BFA) is greater than the foliation angle.

It was concluded from the slope stability analysis that the rock mass is of sufficient quality to incorporate the 75° bench face angle geometry proposed for the Hanging Wall.

Geotechnical berms of 20 m at the boundary of transition and fresh rock material and 15 m for every 150 m of vertical wall (with a ramp passing through) are also required in the pit designs.

Although the Fennec pit design continues to evolve, the current design is representative for ore, grade, and overall tonnage. Geotechnical design parameters are based on Piment parameters in Oxide and Transition (all walls), and Piment HW in fresh rock (single bench).

Kinross Technical Services reviewed and recompiled rock mass rating (RMR) data in 2017. The data indicated that the rock mass quality in the Lower Transition zone is in the same range as that in the Fresh zone, as shown in Figure 16-3. After consultation with Stacey, a modification to the bench face angle (BFA) for the hanging wall was recommended and the proposed BFA was 70° with a calculated inter-ramp angle (IRA) of 52°; the Footwall specifications did not change.

Table 16-4: Geotechnical design parameters - Piment Zone

Zone	Depth (m)	Wall(s)	Bench Face Angle (°)	Inter-Ramp Angle (°)	Total Bench Height (m)	Berm Width (m)
Oxide	0 - 10	All walls	60	39	10	6.5
Transition	10 - 50	West wall (FW)	50	34	10	6.5
		All other walls	65	42		
Fresh	> 50	West walls (FW)	50	38.5	20	8.5
		East walls (HW) and end walls	75	55		

Table 16-5: Geotechnical design parameters - West Branch Zone

Zone	Depth (m)	Azimuth	Bench Face Angle (°)	Inter-Ramp Angle (°)	Total Bench Height (m)	Berm Width (m)
Oxide	0 - 30	330° to 210°	60	39.2	10	6.5
		210° to 330°	45	31.2		
Upper Transition	30 - 60	330° to 210°	65	41.9	10	6.5
		210° to 330°	45	31.2		
Lower Transition	60 - 100	330° to 210°	70	52	20	8.5
		210° to 330°	45	35.1		
Fresh	>100	330° to 210°	75	58.3	30	10.5
		210° to 330°	45	36.5		

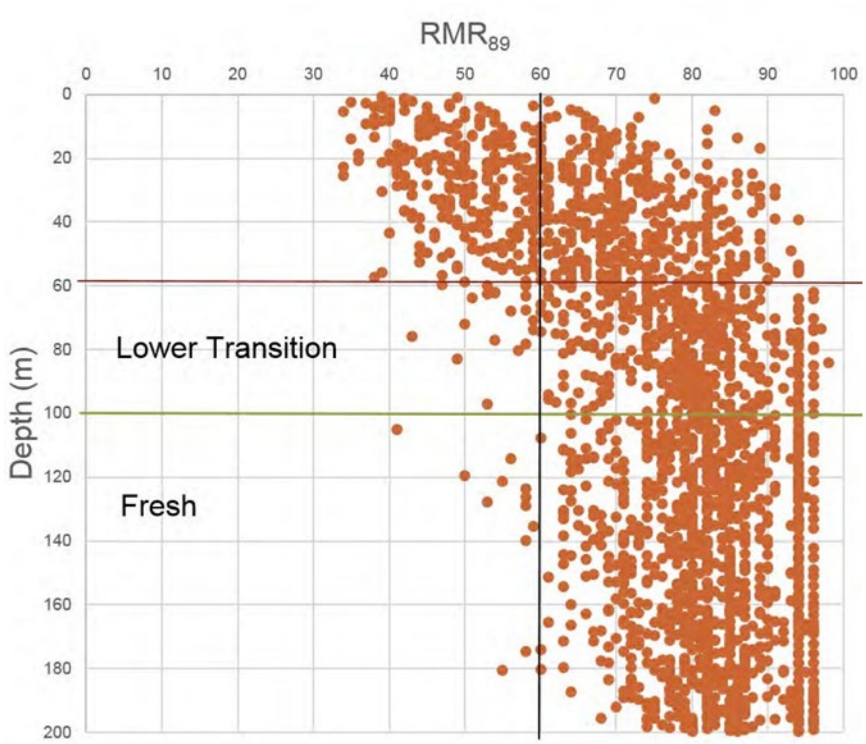


Figure 16-3: Rock mass rating vs. depth (all West Branch data)

Pit Design

The basis for the final pit design is an optimized economic shell generated using Datamine's NPVS software package. NPVS uses the LG algorithm to define blocks that can be mined at a profit. Cut-off grade and pit limits developed in NPVS were defined using the criteria outlined in Section 15. Adjusted overall slope angles were used to define the slopes in NPVS. These adjustments were used to address the placement of ramps within the mined area. Only Measured and Indicated resources were used to define this limit. Sensitivity analyses were carried out comparing the effects of high and low ranges for various inputs on ore tonnes and contain gold ounces mined. Inputs tested were gold price, mining cost, processing cost, CIL recovery, and pit slopes. High- and low- end ranges for the sensitivity analyses are as follows:

- Gold price: -25% to +25%
- Mining cost: -20% to +20%
- Processing cost: -20% to +20%
- CIL recovery: -3.5% to +3.5%
- Pit slopes: -4° to +4°

See Figure 16-4 and Figure 16-5 for the sensitivity analysis.

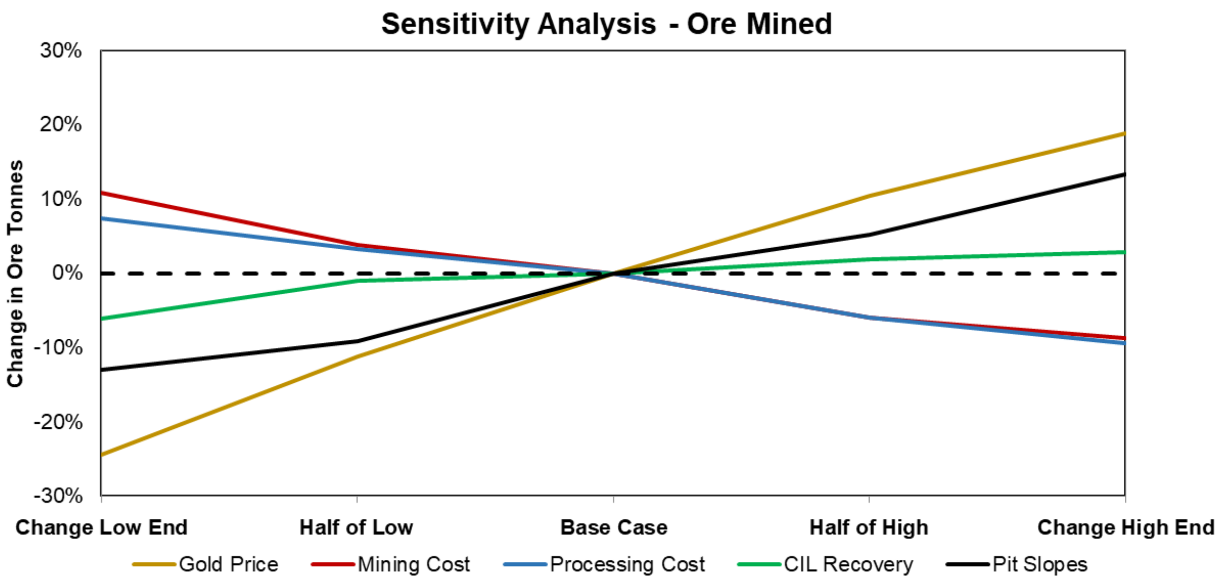


Figure 16-4: Sensitivity analysis – ore tonnes

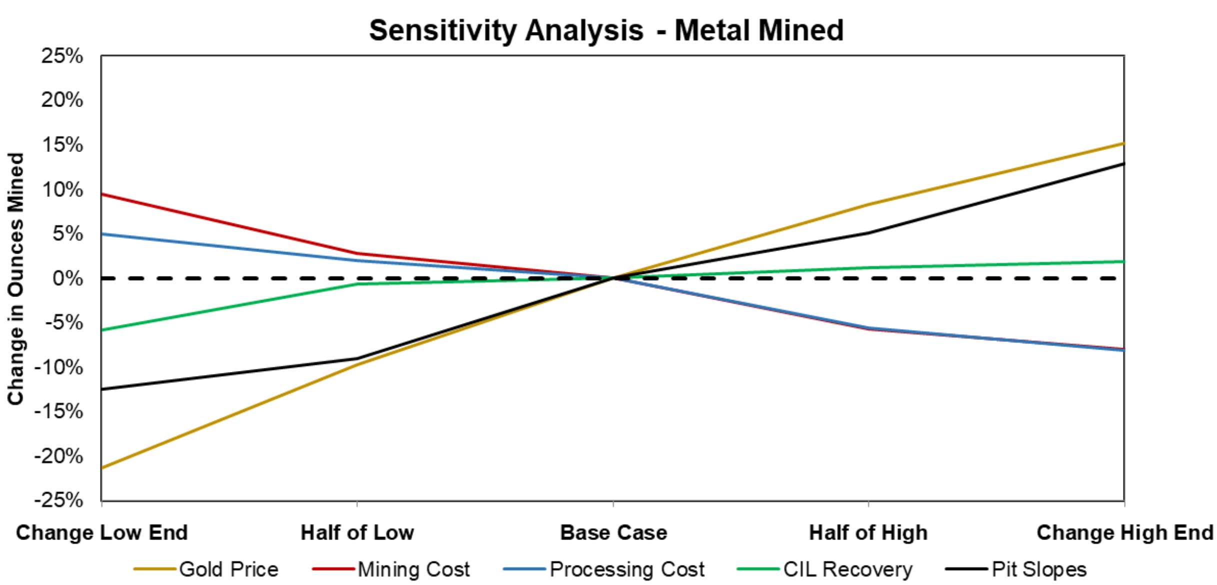


Figure 16-5: Sensitivity analysis – contained gold

The detailed mine design used to define reserves was based on key considerations that include:

- Compliance with the geotechnical recommendations for slope angle set out by the geotechnical studies for haul road widths and maximum effective grades for operation with the existing fleet.
- Bench height that is safely manageable with the existing fleet of Caterpillar 6060 face shovels and RH340 excavators.
- Minimum allowable mining width for practical mining with the existing shovel fleet.
- Pit exits near material destinations (i.e., stockpiles, waste destinations, and primary crusher).
- Options that provide for two or more operational ramps to increase the flexibility and viability of the mining layout.

Inter-ramp slopes – Inter-ramp slopes are based on geotechnical recommendations outlined in the previous subsection and vary according to the slope sector involved. Inter-ramp slope angles range from 31.2° to 58.3° based on the criteria. The inter-ramp angles and the bench face angles were adhered to for the pit and phase designs. Catch benches for the design vary based on bench face angles and ultimate bench height. The overall design slopes include access ramps and follow the same criteria used in the LG cone calculation.

Bench height – The design operating bench height is 10 m. The final pit walls in West Branch will be triple benched where it is permissible, resulting in a bench height of 30 m with intervening catch benches. Piment will be double benched, resulting in a 20 m bench height with the appropriate catch benches. Satellite pits will be single benched (at 10 m).

Minimum mining width – A phased approach was taken as an optimization strategy to improve the mining sequence. Efforts were made to maximize mining width where possible. Where mining widths indicated by the selected LG shells were too narrow (<45 m) to safely or effectively mine, the phase walls were pushed out to the final pit design.

Access – Dual lane haul roads for the current designs are 32.5 m wide. Ramps are designed at a maximum gradient of 10%. Intersections and switchback curves are designed without grade (flat) wherever possible. Two main haul roads exit the West Branch pit, with a third ramp exiting the pit through the WB extension at the northern end of the pit. These haul roads split to minimize haul times between different destinations. Piment will also incorporate two main haul road exits. For the satellite pits, dual lane haul roads in the current design are 23.2 m with similar design parameters (grade and corners) to the West Branch and Piment pits. Haul road profiles are shown in Figure 16-6. Haul road dimensions are shown in Table 16-6.

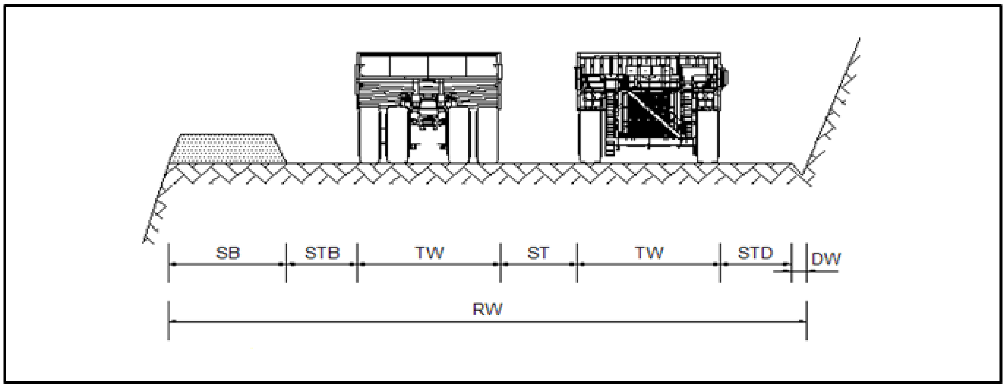


Figure 16-6: Haul road profile

Table 16-6: Double-lane haul road dimensions

Road Attribute	West Branch and Piment	Satellite Pits
Safety berm (SB)	4.6 m	3.6 m
Space between truck and safety berm (STB)	3.8 m	2.65 m
Truck width (TW)	7.6 m	5.3 m
Space between trucks (ST)	3.8 m	2.65 m
Space between truck and ditch (STD)	3.8 m	2.65 m
Ditch width (DW)	1.0 m	1.0 m
Ramp width	32.5 m	23.2 m

Detailed pit design work was completed using Deswik CAD software. The Deswik CAD software Design & Solids Modeling tool uses design parameters contained within the 3D block model to ensure compliance to available geotechnical parameters. Isometric views of the West Branch and Piment main pits, and the Piment and Fennec satellite pits are shown in Figure 16-7 through Figure 16-10.

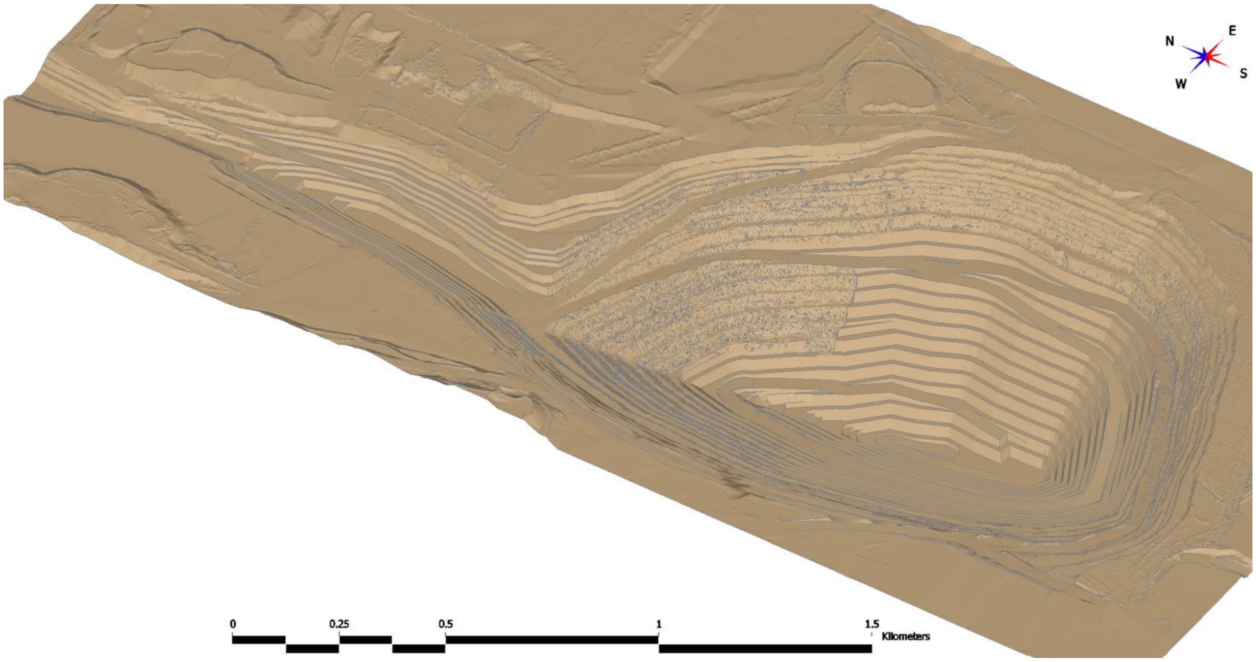


Figure 16-7: West Branch pit isometric view

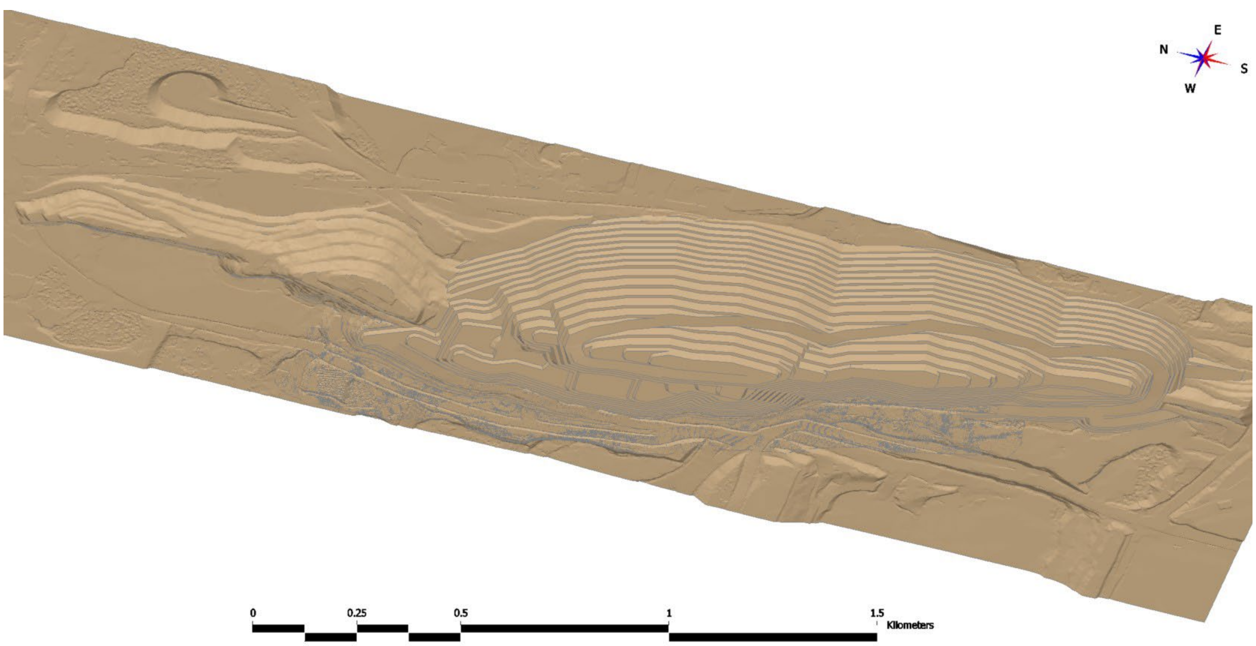


Figure 16-8: Piment pit isometric view

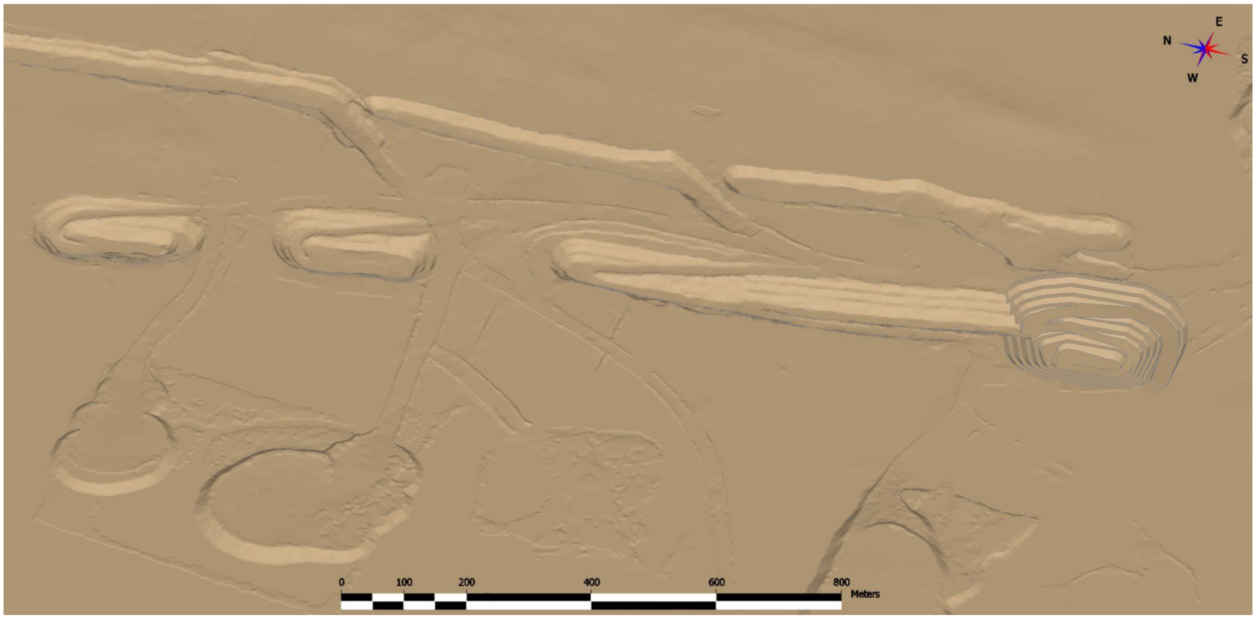


Figure 16-9: Piment satellite pit isometric view

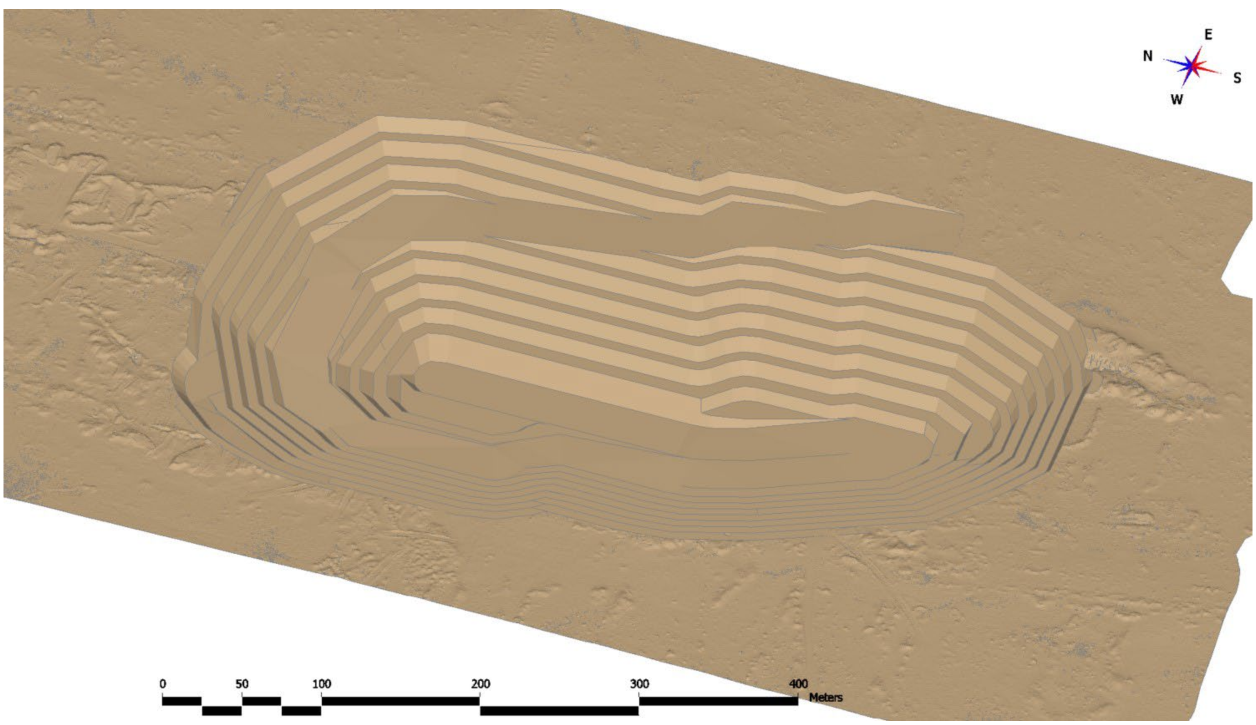


Figure 16-10: Fennec satellite pit isometric view

Waste Dumps

Waste rock is used for haul road and tailings dam construction as needed. The existing road network is well developed and requires continued maintenance. Additional roads will also be required throughout the LOM. These roads will be constructed using the current mining and support mining fleets.

Dumped waste material comprises weathered and fresh rock. Blasted weathered rock is finely graded, including clay-silt fines. The fresh rock is strong and massive and, when blasted, is coarsely graded, including boulders that can require secondary blasting prior to loading.

As the climate is arid and there is no permanent surface water and very limited groundwater, there is low potential for ARD, however, any potential ARD issue will be mitigated by ensuring that material that is identified as potentially acid forming (PAF) will not be dumped on the outer shell of the waste dumps.

The dump design is based on 40 m high lifts with a maximum overall effective slope of 2H:1V (27° overall slope angle). The maximum dump height is currently limited to 100 m total vertical height. The dumps will be accessed by 35 m wide dual-lane ramps at a maximum gradient of 10%. Track bulldozers will be used to assist the haulage fleet to facilitate proper dump construction, including grading the top of each lift away from the pit to direct any rainwater run-off and placing coarsely graded, fresh rock on final dump faces.

The waste dumps are located within the footprints of previous studies and require no modification to the current permitting.

There is one active waste dump along the west side of the Piment pit and four active waste dumps along the south, east, and west sides of the West Branch pit (Figure 16-2). An additional waste dump is planned next to the Fennec satellite pit. Based on the current mine plan, the waste dumps will require 325 Mt of capacity after 2024, and an additional 16 Mt to the Fennec dump. The current permitted waste dumps have excess capacity relative to the current mining requirement.

16.3 Schedules

Pushback Sequences

Phased pushbacks were developed to optimize the mining sequence for both West Branch and Piment (Figure 16-11 and Figure 16-12, respectively). Five pushbacks were developed for the West Branch pit: WB1, WB2, and WB3 have been completed, and

WB4 will be completed in 2025. Piment consists of two phases located around the current Piment Central pit.

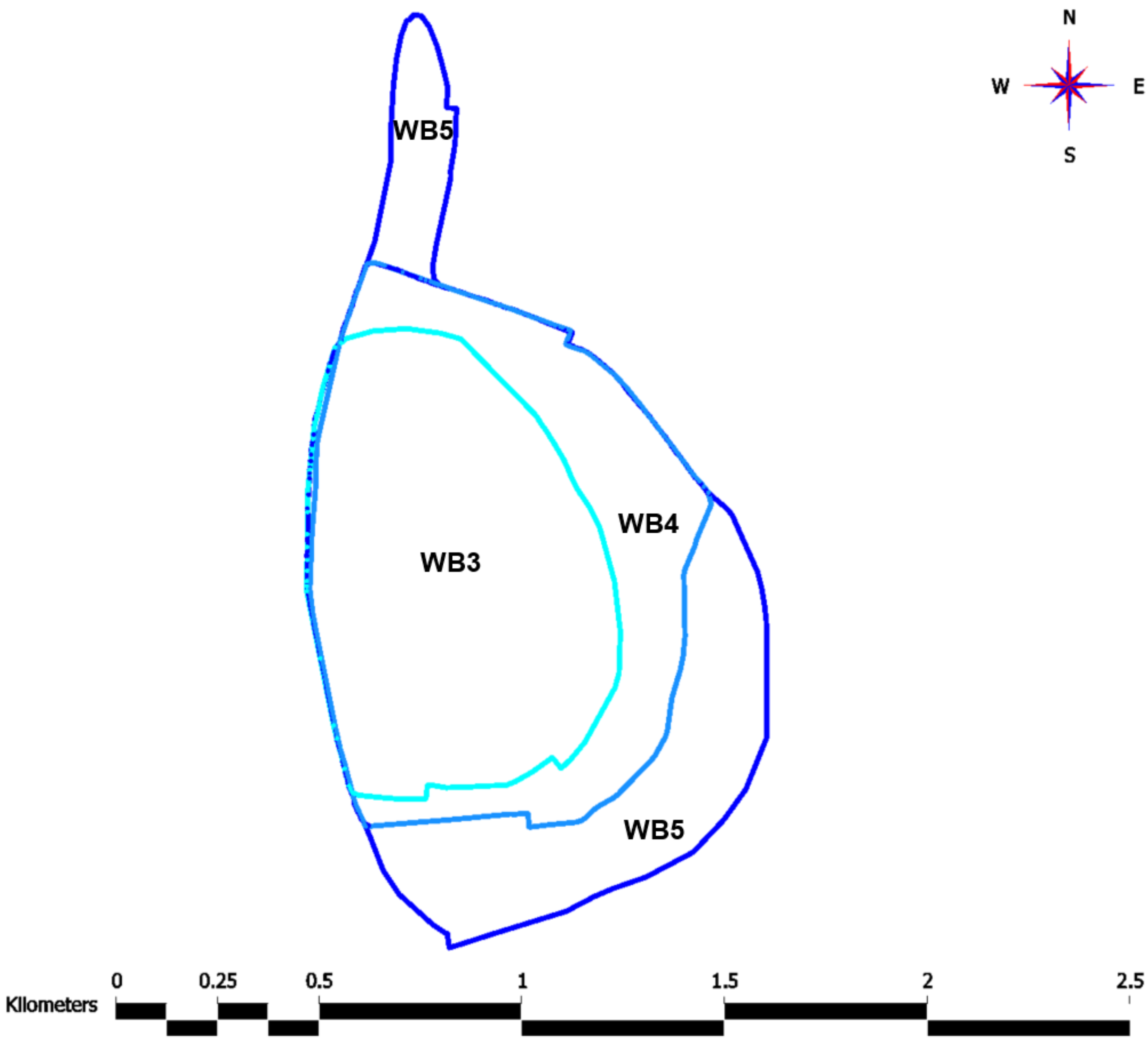


Figure 16-11: West Branch (WB) pit phasing

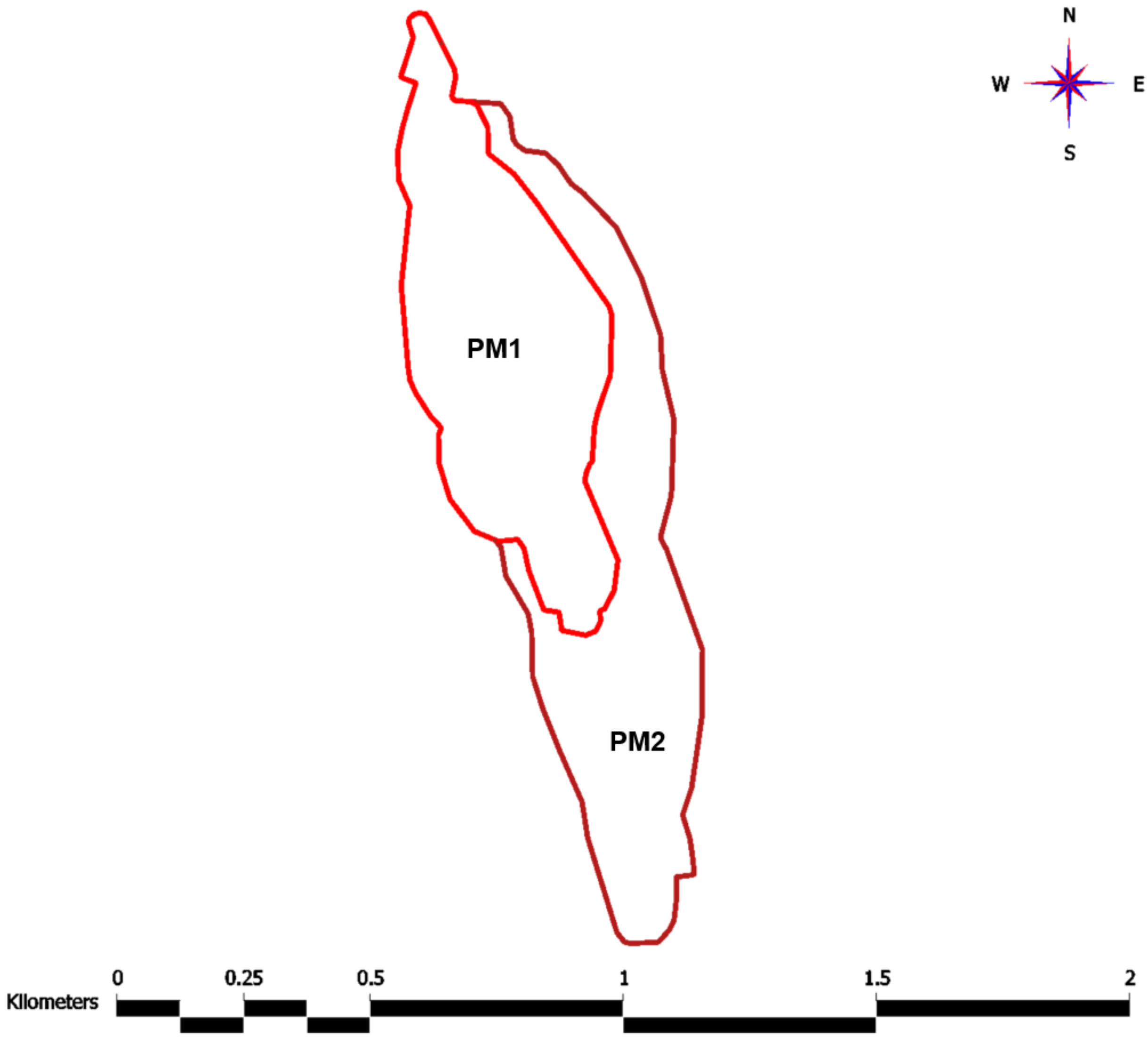


Figure 16-12: Piment (PM) pit phasing

A cross-section of the West Branch pit is shown in Figure 16-13. This section illustrates how the pushback sequencing targeted high ore zones while minimizing the waste stripping requirements during the early pushbacks. As of the end of 2024, the early pushbacks are largely depleted, with the pit advancing into the higher strip WB5 phase.

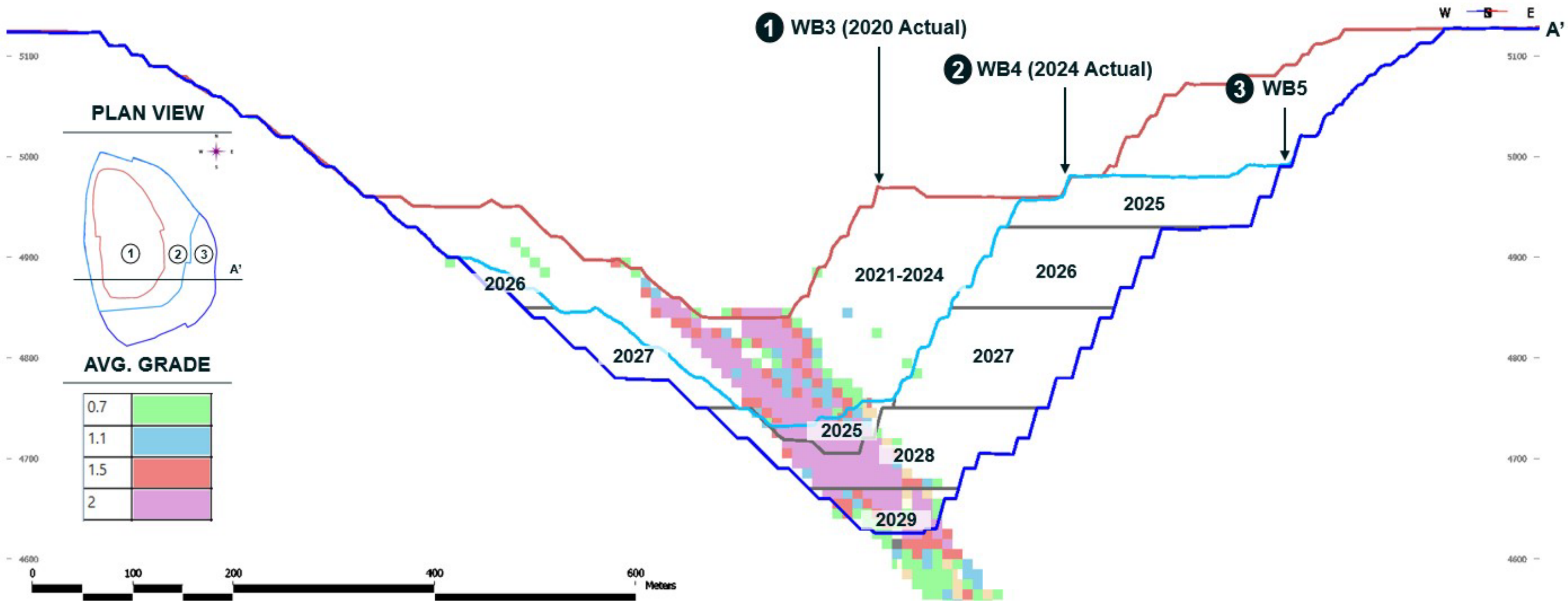


Figure 16-13: West Branch production advance section view

Mine Production Rate

The mine production schedule is optimized to maximize value and gold grade of feed material at the CIL plant, which operates at a throughput rate of 24 kt/d. The planned mining rate peaks from 2025 to 2027 at 75 Mtpa to 80 Mtpa (Figure 16-14) and then ramps down as the pits are depleted. Over the next few years, the majority of mining will take place in higher strip ratio pushbacks and, therefore, mill feed will be supplemented by high-grade stockpiles. This mining profile leverages fleet capacity and maximum vertical advance throughout.

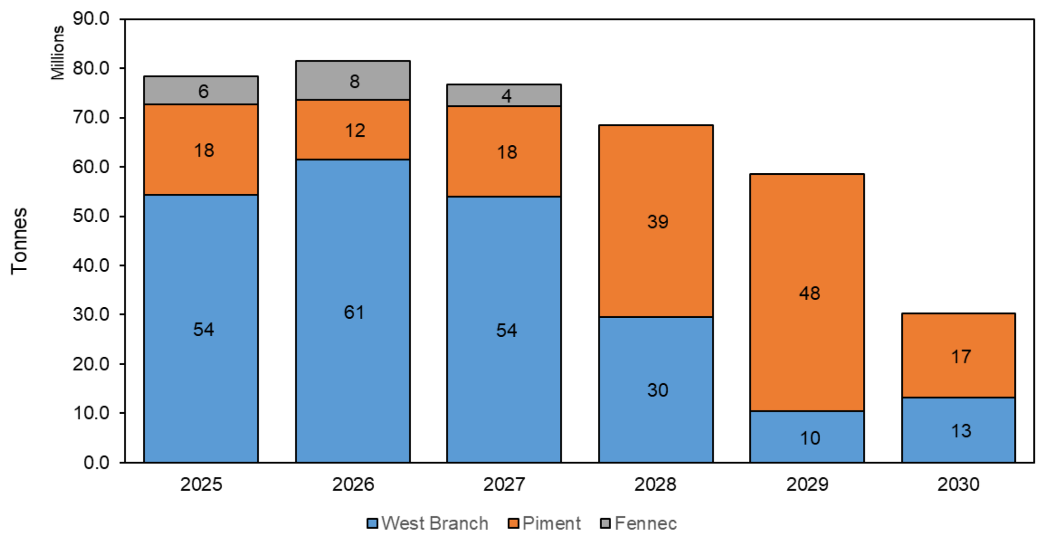


Figure 16-14: Annual mining rate by pit

Scheduling Constraints

The constraints used to generate the schedule are presented in Table 16-7.

Table 16-7: Scheduling constraints

Constraint	Description
Mining Capacity	In most periods, the maximum mining capacity allowed in the schedule was calculated based upon the existing mining equipment fleet. Each fleet's available operating hours are based on a calculation which includes units available, historical and projected mechanical availability and use of availability, and utilization efficiencies. In all periods, it is assumed that only 20% of mill feed from the pit(s) will be direct-fed, with the remainder subject to rehandling at the primary crusher.
Sinking Rate	Depending on the pit phase, a maximum sinking rate of nine 10 m benches per year was applied to planned mining areas. Generally, maximum sinking rates are decreased in congested areas such as the bottom of the pit.
Stockpiles	Four stockpiles were used for mine scheduling: Purple (>2.0 g/t), Red (>1.5g/t), Blue (>1.1g/t), Green (>0.7g/t), and Subgreen (>0.4g/t).

Mine Production Schedule

The distribution of equipment across the pits, phases, and benches results in variable production rates using the same fleet. As such, the total material moved varies year over year (Table 16-8). The mining schedule was optimized with a maximum of 45 Caterpillar 793D and five Hitachi EH-4000 haul trucks and a maximum of six Caterpillar 6060 shovels.

The mine production schedule (split by deposit) is shown in Table 16-8. WB4 is depleted in 2025, and stripping of the WB5 pushback is well under-way. The bulk of the WB5 orebody is reached in 2027, with significant quantities of very high-grade material coming out of the pit in 2028. A small section of the West Branch pit, called WB5 Extension, is planned in 2029-2030. Piment main and its satellite pits as well as Fennec satellite pit are continuously planned for mining throughout the schedule for supplementary plant feed to West Branch.

Table 16-8: Mine Production Schedule

Description	LOM	2025	2026	2027	2028	2029	2030
Ore Mined (kt)							
West Branch	31,428	3,239	4,400	6,431	9,656	4,356	3,346
Piment	19,254	3,191	2,946	1,271	910	5,914	5,021
Fennec	1,910	-	1,507	403	-	-	-
Total	52,592	6,431	8,853	8,104	10,566	10,271	8,368
Au Grade (g/t)							
West Branch	2.00	1.83	1.20	1.67	2.71	2.22	1.50
Piment	1.59	1.61	1.79	2.02	0.94	1.34	1.75
Fennec	1.91	-	1.88	2.00	-	-	-
Total	1.84	1.72	1.51	1.74	2.56	1.71	1.65
Waste Mined (kt)							
West Branch	191,369	51,056	57,058	47,551	19,891	6,046	9,768
Piment	133,981	15,166	9,210	17,089	38,090	42,246	12,179
Fennec	16,094	5,728	6,297	4,069	-	-	-
Total	341,444	71,950	72,565	68,709	57,981	48,292	21,947
Total Mined (kt)							
West Branch	222,797	54,295	61,458	53,981	29,547	10,402	13,114
Piment	153,235	18,357	12,156	18,360	39,000	48,161	17,201
Fennec	18,004	5,728	7,804	4,472	-	-	-
Total	394,036	78,381	81,417	76,814	68,547	58,563	30,315

Notes:

1. Mine production schedule does not include the 0.6-0.7 g/t Au material that is included in the Reserves Statement. In future iterations, this material will be planned at the end of the plant life, extending processing for an additional year.
2. Excludes ore tonnes and gold ounces in starting stockpile mined before the start of 2025.

The process plant currently operates at 24 kt/d, with the plant quickly ramping down in 2035 once all stockpile balances have been diminished. Table 16-9 shows the process plant feed schedule by year.

Table 16-9: CIL process plant feed schedule

Description	LOM	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035
CIL Process Plant												
Plant Feed (kt)	96,104	8,808	8,808	8,760	8,784	8,760	8,760	8,760	8,784	8,760	8,760	8,360
Head Grade (g/t)	1.48	1.88	1.81	1.75	2.38	2.38	1.88	0.91	0.88	0.81	0.79	0.78
Contained Au (koz)	4,573	531	512	493	672	672	528	256	249	228	222	209
Recovery	93%	93%	93%	93%	94%	94%	93%	93%	93%	92%	92%	92%
Produced Au (koz)	4,266	500	476	460	630	630	493	238	230	211	205	197

Notes:

1. Process plant feed schedule includes ore tonnes and gold ounces mined before 2025, processed from stockpile.
2. 2025 includes 7 koz of gold in circuit (GIC) drawdown to bring GIC levels in line with historical operating levels.

Once mining operations have been completed in 2031, the CIL plant will continue processing the low-grade stockpiles that will have developed over the mine life.

Stockpile Evolution

The stockpiles will be used in conjunction with the operating cut-off grade strategy to add value to the project and aid in sequencing the pit. Generally, the stockpiles will be reclaimed during periods where the material mined in the pit is of lower grade than the stockpiled material, however, the stockpiles are also planned to be reclaimed during the 2025-2026 periods due to the lower quantities of ore planned to be mined from the pits (during WB5 and PM stripping). Stockpile evolution over the LOM is presented in Figure 16-15.

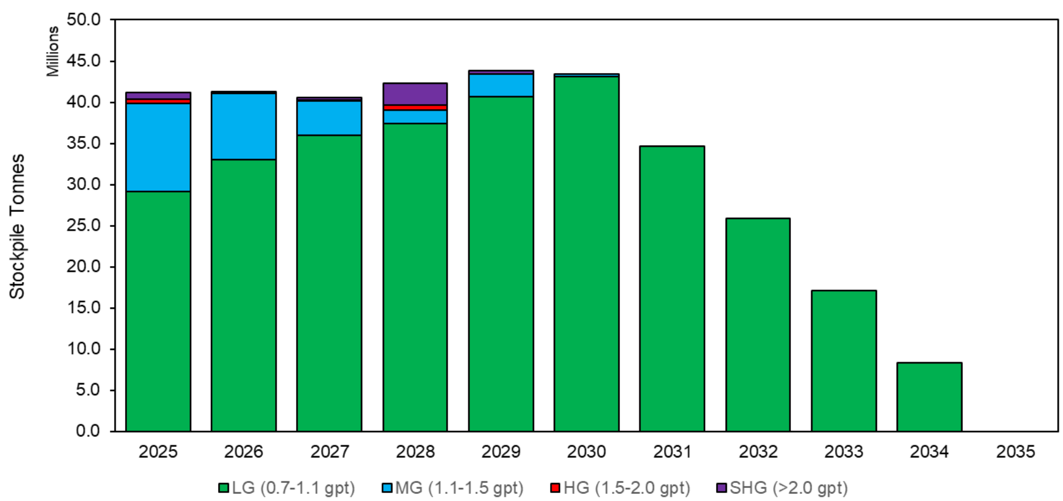


Figure 16-15: LOM stockpile balances and depletion

16.4 Mine Equipment

Haulage Estimates

Selection of the mining equipment at Tasiast is based on the current mining fleet on site. The primary equipment fleet includes Caterpillar 6060 and Bucyrus RH340 excavators paired with Caterpillar 793D and Hitachi EH4000 haul trucks for the main pits (West Branch and Piment) and a Caterpillar 6030 excavator paired with Caterpillar 777 haul trucks for the satellite pits.

Payload sizes were estimated for the study, based on analysis of actual payloads. Payloads estimates were not varied by rock type or time and represent the average payload size expected, (Table 16-10).

Table 16-10: Payload sizes for planning

Truck Type	Payload 2025+ (tonnes)
Caterpillar 793D	220
Hitchi EH-4000	220
Caterpillar 777	90

The LOM schedules provide details regarding the tonnages moved by material type and destination (i.e., waste dumps, ore stockpiles, primary crusher, TSF construction, etc.). During mine scheduling, haulage analyses were performed using Deswik software to estimate the required number of truck hours.

Haulage speed estimates used the speeds displayed in Table 16-11; these speeds have been benchmarked to reflect operating conditions at Tasiast. These speeds were applied for all haulage fleets.

Table 16-11: Haul truck speeds

Road	Loaded Speed (km/h)	Empty Speed (km/h)
Flat	30	35
Down Ramp	17	28
Up Ramp	12	22
Pit Bench Area	19	19
Dump Area	35	35

Complete cycle time estimates were calculated for each discrete pit cut to each required material destination. This cycle time estimate includes travel time, loading time, shovel spotting time, and dumping time. Loading time estimates are calculated using dig-rate estimates for the different primary loading units used in the mine plan. Variable dig rates are applied in oxide, transition, and fresh materials. These dig rates are used to calculate load times for paired trucks. Dig rate estimates are derived from current rates and are consistent with the site’s budget estimates. Dig rate estimates in tonnes per operating hour (t/h) are shown in Table 16-12 through Table 16-13.

Table 16-12: Loading unit planned productivities

Loading Unit Productivity (t/hr)	CAT6060 Shovel	RH340 Excavator	CAT6030 Shovel
Oxide	3,000	2,500	950
Transition	2,800	2,200	885
Fresh	2,800	2,200	885

Table 16-13: Front-end Loader (FEL) planned productivities

FEL Productivities (t/hr)	CAT994	CAT992
Stockpile/ROM Rehandle	1,000	800
Production	1,000	N/A

A fixed time allocation for dump time is applied to each haul cycle. The fixed non-travel time is based on current site data. Truck requirements and productivity by period were calculated based on the quantity of material moved and the cycle times associated with each material. The breakdown of the fixed cycle times is shown in Table 16-14.

Table 16-14: Breakdown of cycle time components

Activity (minutes)	CAT 793D	EH-4000	CAT 777
Loading	Variable	Variable	Variable
Dumping	2.16	2.16	2.16
Total	Variable	Variable	Variable

As the mine sequence progresses, the pit becomes deeper resulting in longer travel times to reach the pit exit point. Similarly, the primary waste dump destinations will be filled, and longer routes will be required to reach their higher dump elevations. The crusher location will remain fixed during the LOM. The net of these effects over time is that the average cycle time generally increases over the mine life. Figure 16-16 shows the average truck fleet requirement over the LOM (including stockpile reclaim). As this is the average annual figure, in some periods (months), peak required trucks could be different than these average figures. For instance, in Q1 2025 Tasiast will require 44 CAT-793 and four EH4000 trucks compared to 45 total required trucks in full year 2025.

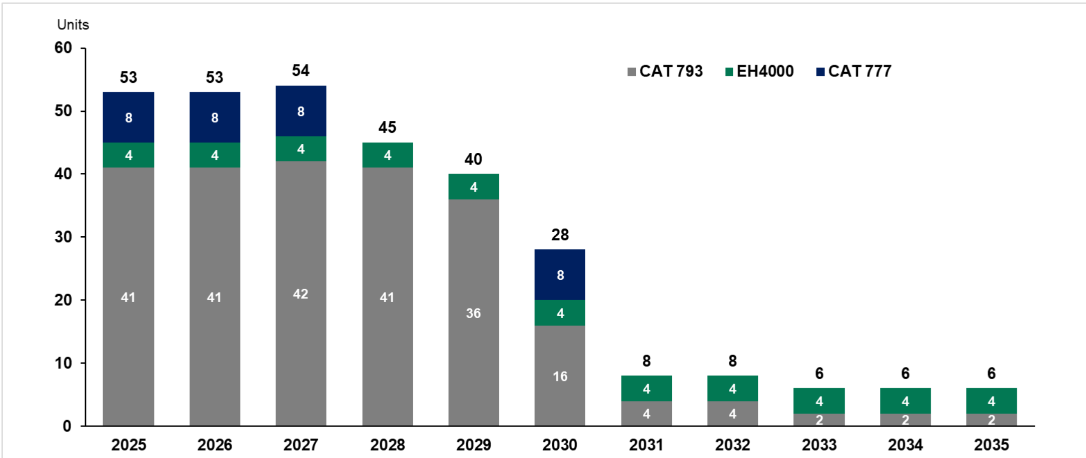


Figure 16-16: Annual hauling requirements

Loading Equipment Estimate

Estimation for the loading equipment requirements is based on the material movement specified in the LOM schedule and the estimated productivity rates. The productivity rates for the loading units are based on loading cycle times, bucket capacity, bucket fill factors, and historical productivity rates.

The fleet size was determined based on anticipated operating hours for the loading equipment and machine life estimates provided by equipment manufacturers, along with benchmarking data. The total fleet requirement was estimated by applying the percentage of mechanical availability and usage to the operating fleet requirements. Loading fleet requirements by year are shown in Figure 16-17.

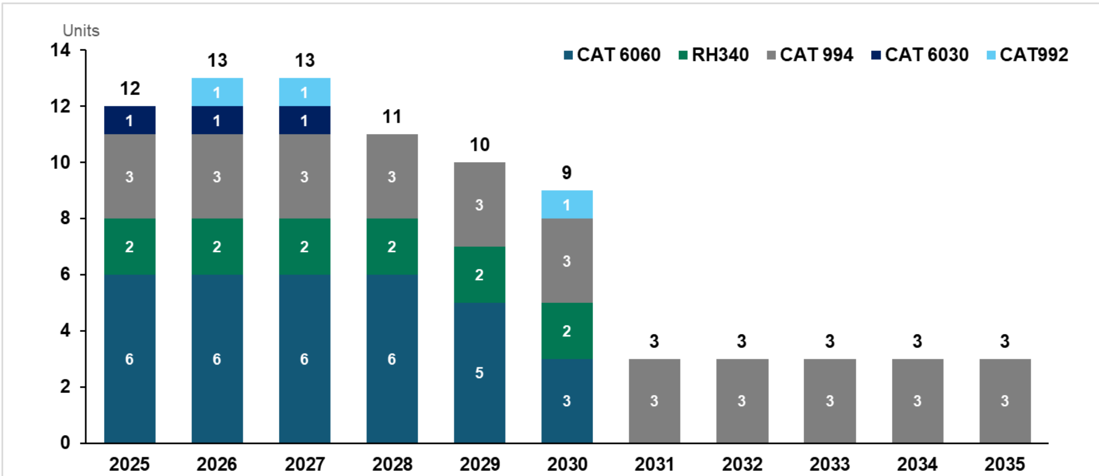


Figure 16-17: Loader requirements by year

Drilling Equipment Estimate

Blasting requirements determine the drill hole pattern size (i.e., burden and spacing) which provides the information to estimate the metres of drilling and hours required to achieve planned production. The plan assumes that the Sandvik DR580 and DR650i drills will be used predominantly for pre-split drilling, wall control holes in trim shots, and re-drill holes. The larger Bucyrus SKFX and Caterpillar MD6250drills will be used for ore and waste production drilling.

The blast design parameters and pattern designs match current practices at the site. Penetration rates for each drill by material type are illustrated in Table 16-15.

Table 16-15: Planned drill productivities

Drilling Rates	Oxide (m/hr)	Fresh (m/hr)	Presplit (m/hr)
DR580 & DR650i	30	24	21
SKFX	50	30	
MD6250	42	38	

The fleet size was determined based on estimated drilling hours, machine-life estimates provided by equipment manufacturers and benchmarking data. The total fleet requirements were estimated by applying percent mechanical availability and utilization to the operating fleet requirements.

Support Equipment Estimate

The mine equipment fleet will need various support equipment for constructing and maintaining roads, waste storage dumps ore stockpiles, and run-of-mine (ROM) pads. Support equipment fleet requirements depend on infrastructure maintenance requirements, and the number of shovels and excavators operating in the pit. Typically, these estimates were based on operating experience, and benchmarking data.

Tasks performed by auxiliary equipment include:

- Road maintenance and construction
- Waste dump maintenance
- Cleanup around excavators
- Cleanup of ore contact
- Batter trimming
- Drainage construction
- Dewatering
- Dust suppression
- Lighting
- Servicing
- Bulldozing waste dump batters to final profile angle
- Clearing and topsoil stripping

Costs and equipment hours for support tasks were not calculated in detail. Annual auxiliary equipment hours were estimated based on historical performance and benchmarking of operations with comparable material movement rates and mining equipment fleets.

A modular mining dispatch system with high-precision GPS is used for haul truck dispatching. Slope monitoring is carried out with portable slope radar and survey equipment.

16.5 Personnel Requirements

The current mine operation at Tasiast is owner-operated and applies conventional open-pit operational practices, with drilling, blasting, loading, hauling, support, and administrative functions. The mine operates 365 scheduled days per year and 24 hours per day, primarily divided into two 12-hour shifts per day for mine operations and maintenance.

The mine organization includes functional groups for mine operations (drilling and blasting, loading and hauling), maintenance, mine technology, and technical services (Table 16-16). The mine is staffed to support all operational, safety, and environmental requirements. Mining-related functional groups are organized under the mine manager or technical services manager. The mine manager is allocated functional groups for mine operations, maintenance, and technology. Among the functional groups responsible for mine operations, drilling and blasting are managed together, as are loading and hauling. The technical services manager oversees functional groups for technical services, including mine planning, survey, geology, and geotechnical services. The mine manager and technical services manager collaborate to manage mine operations.

Table 16-16 shows staffing levels for the mining area by functional group. Staffing levels remain relatively steady until the completion of WB5 in 2029 and then reduce significantly once mining is completed in 2030. A portion of the workforce remains operational during rehandling (and depletion) of lower grade stockpiles at the end of the mine life. Table 16-17 presents the levels of national and expatriate personnel at site.

Table 16-16: Mining personnel by function by year

Department	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035
Mine Operations	558	558	557	530	509	233	51	51	51	51	53
Mine Maintenance	365	365	364	359	358	165	23	23	23	23	21
Mine Technology	11	11	11	10	10	10	2	2	2	2	2
Technical Services	53	53	53	50	50	43	8	8	8	8	8
Geology	52	52	52	51	50	32	4	4	4	4	4
Mining Total	**1039**	**1039**	**1037**	**1000**	**977**	**483**	**88**	**88**	**88**	**88**	**88**

Table 16-17: National and expatriate mining personnel by year

Type	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035
Expatriates	13	13	11	5	3	2	0	0	0	0	0
Nationals (Admin)	138	138	138	137	137	121	33	33	33	33	39
Operators	572	572	572	545	524	232	38	38	38	38	35
Mechanics	316	316	316	313	313	128	17	17	17	17	14
Total	**1039**	**1039**	**1037**	**1000**	**977**	**483**	**88**	**88**	**88**	**88**	**88**

17. RECOVERY METHODS

The Tasiast CIL plant has undergone multiple upgrades through the last decade. Since 2019, modifications have been made to increase the previous throughput of 15 kt/d (with the ‘12K’ flowsheet) up to the new standard of 24 kt/d, on average. The current flowsheet is detailed below.

17.1 Water Management

A brackish water bore field (known as “Sondage") provides raw water for the mine, CIL plant, and camp. The potable water requirements are met by three reverse osmosis plants.

For the initial 21 kt/d expansion, pumps at the existing Sondage pump station P1 will be upgraded. Pump stations, P2 and P4, will also be added along the existing 400DN PVC pipeline. For the 24 kt/d expansion, two additional pump stations, P3 and P5, will be added on the existing 400DN PVC pipeline as well. Each pump station will consist of a tank, operational and standby pumps, and ancillaries. Several additional wells will be added at the Sondage in order to provide additional capacity and flexibility for the operation.

The pumps supplying raw water to the CIL plant have been upgraded to improve reliability and meet the increased raw water consumption for the new 24 kt/d throughput.

Internal process water recovery from the thickener overflows, reclaimed water from the TSF, and recycled grey water from the camp are used to reduce the overall plant raw water requirements.

17.2 Gold Recovery

The CIL plant produces all the gold shipped from Tasiast. A Dump Leach with a dedicated adsorption, desorption, recovery (ADR) plant previously supplemented gold production but has since been shutdown. All produced gold is in the form of bullion and is transported regularly to a refinery for final refining and sale.

Unit gold recovery estimates for the CIL plant are based on metallurgical test work and a review of historical CIL plant performance.

17.3 Current CIL Processing Plant – The ‘24K New CIL’ Project

The Tasiast CIL plant was recently expanded in 2023 from a design throughput of 21 kt/d to the new design of 24 kt/d. In 2024, a new series of seven CIL tanks were commissioned to replace the historical ‘12K’ CIL tanks. The simplified process flowsheet for the current CIL plant is shown in Figure 17-1 where bolded equipment are modifications made for the ‘24K New CIL’ Project (‘24K’ Project) and grey equipment exists from the ‘12K’ and ‘21K’ projects. A simplified plant layout is shown in Figure 17-2.

Design Criteria

The design capacities for the crushing plant and process plant use 70% and 92% effective operating time, respectively. Based on test work and operating experience, the key nominal design criteria for the major process circuits are summarized in Table 17-1. It should be noted that design factors accounting for higher gold grades were applied when required to match the mine plan. The plant design life is 15 years.

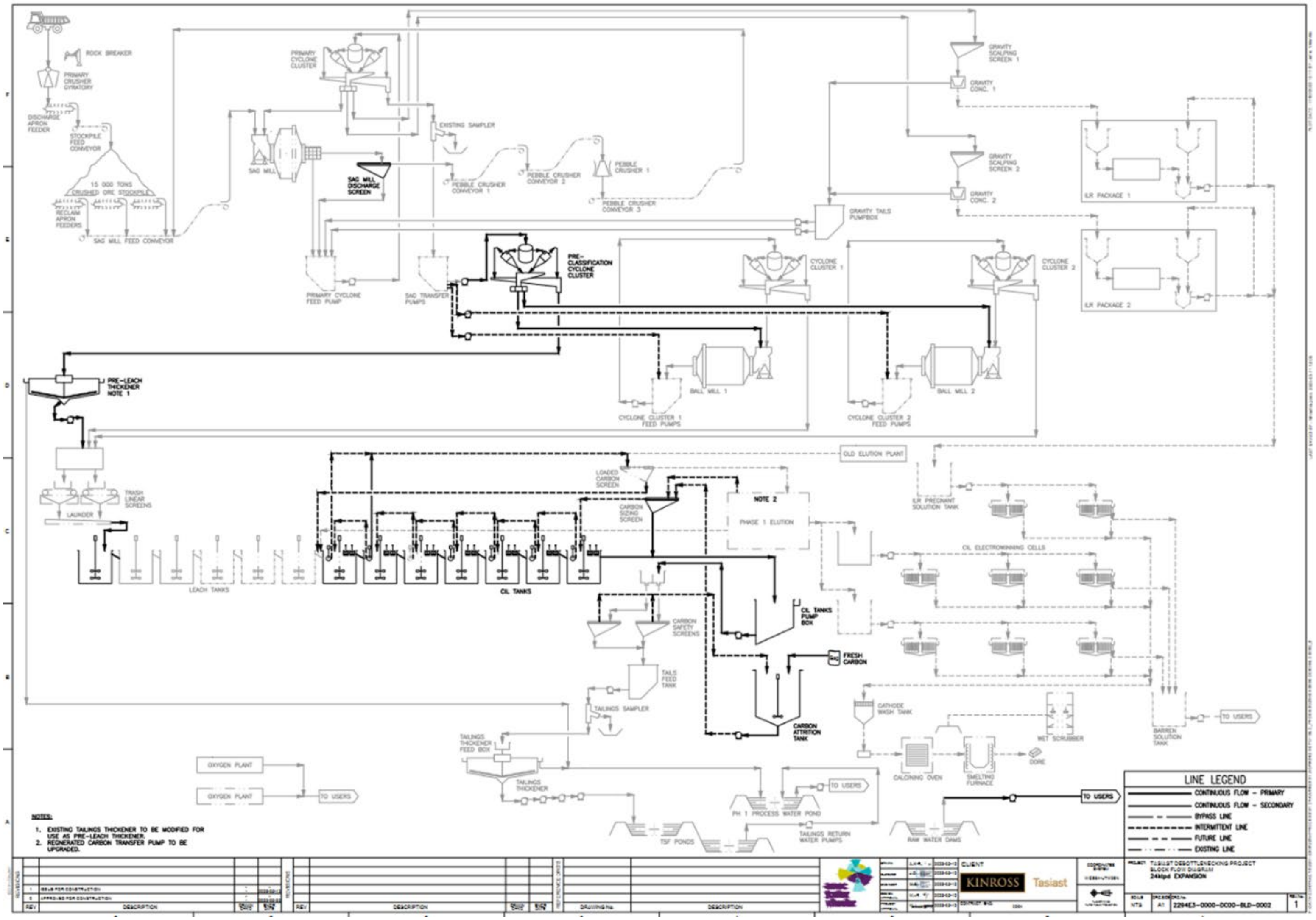


Figure 17-1: Simplified process flowsheet

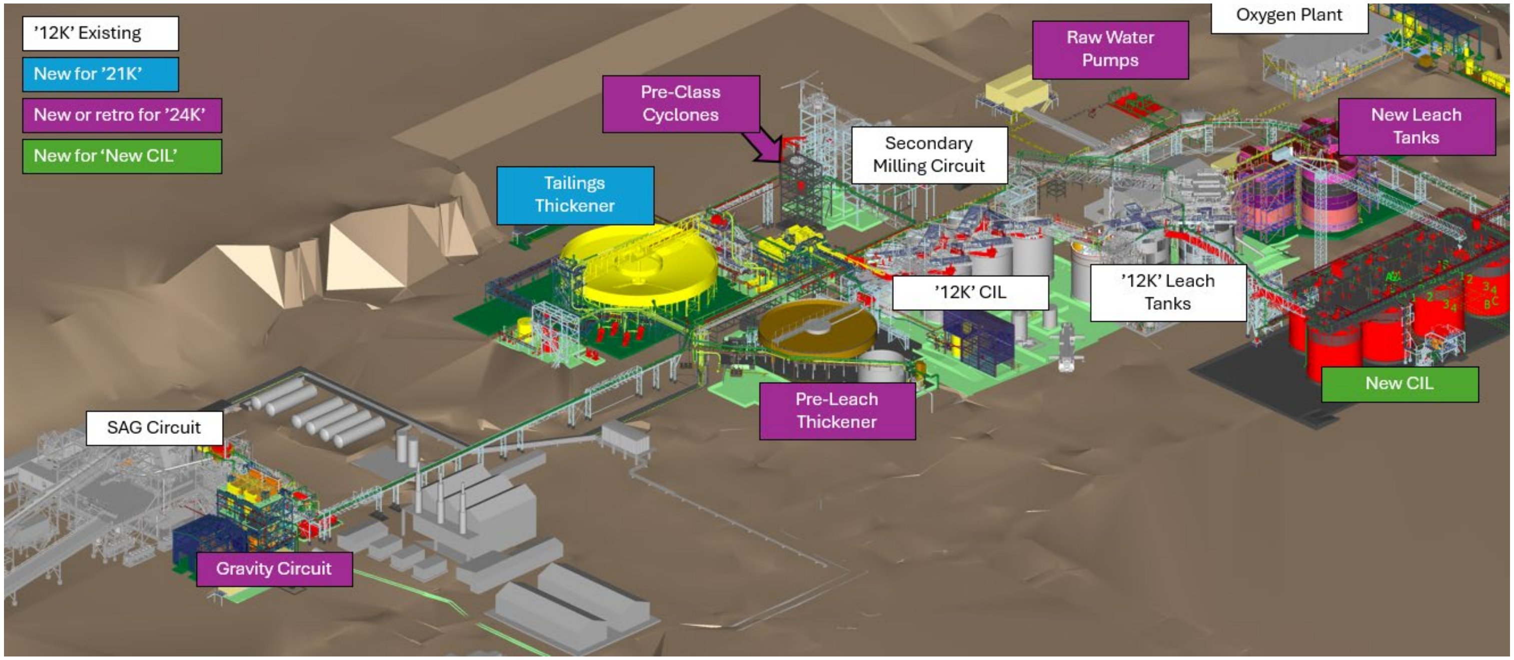


Figure 17-2: '24K New CIL' Project plant layout

Table 17-1: Key process design criteria

Area	Criteria	Unit	Value
General	Daily throughput	t/d	24,000
	Process plant availability	%	92.0
	Gold feed grade	g/t	1-3
Primary crusher	Availability and utilities	%	70
	Crusher work index	kWh/t	14.6
	ROM top size	mm	1,100
	Crusher product size (P_{80})	mm	150
Ore storage	Capacity (live)	t	15,000
	Capacity (live)	h	15
	JKTech Axb	-	31.4
Grinding and pebble crushing	SAG mill product size (P_{80})	µm	160
	Nominal fresh feed to SAG	t/h	1,087
	Bond Ball Mill work index	kWh/t	13.3
	Ball mill product size (P_{80})	µm	90
Gravity	Recovery	%Au feed	20 to 25
Pre-leach thickening	Thickener diameter	m	30
	Underflow density	% w/w	50
Leaching and CIL	Leach residence time	h	9
	CIL residence time	h	9
	Total leach + CIL	h	18
Tailings thickening	Thickener diameter	m	40
	Underflow density	% w/w	60
Carbon treatment	Stripping solution flow rate	BV/h	2
	Operating temperature	°C	140
	Operating pressure	kPa	450
	Number of elution vessels	-	2
	Elution batch size	t	7.5
Electrowinning & refining	EW recovery	%	98

Recent Plant Performance

Following throughput and recovery challenges during the expansion phases and ramp-ups, the CIL process plant's performance improved through 2024 (Table 17-2) with daily tonnage reaching the plant's new nominal daily throughput and recovery re-attaining the test work-derived recovery curve in Q4. The CIL gold recovery equation is presented in Table 17-3.

The start-up of the new CIL and realization of recovery-based improvement initiatives in Q4 contributed to the significant recovery increase.

Table 17-2: 2024 CIL plant quarterly performance and recovery curve details

2024 Quarter	Tonnes per day (t/d)	Process plant overall gold recovery (%)
Q1	22,782	91.3
Q2	23,742	91.8
Q3	23,953	91.2
Q4	23,968	94.3

Table 17-3: CIL gold recovery equation

Process	Au Recovery
CIL Recovery	(Au-MAX(Au*(1-0.25)*(1-(1.6772*LN(Au*(1-0.25))+90.8)/100),0.04+0.015*(1-0.6)/0.6))/Au [1]

Notes:

1. Head grade vs. recovery relationship develop from test work.
2. Au = head grade, g/t.

Primary Crushing

Mined ore above plant feed cut-off grade is transported from the open pits to the plant by truck and either deposited onto the ROM pad or directly dumped into a Gyratory Crusher. The primary crushing facility is shown in Figure 17-3.The material can be blended according to grade and competency.

Crushing of the mineralized material takes place in a single stage; a primary gyratory crusher that reduces rock to less than 210 mm, targeting a P_{80} of 125 mm.

The rock is conveyed to a coarse ore stockpile shown in Figure 17-4 that uses three apron feeders to feed the 40 ft SAG mill.

Figure 17-3: Primary Crushing

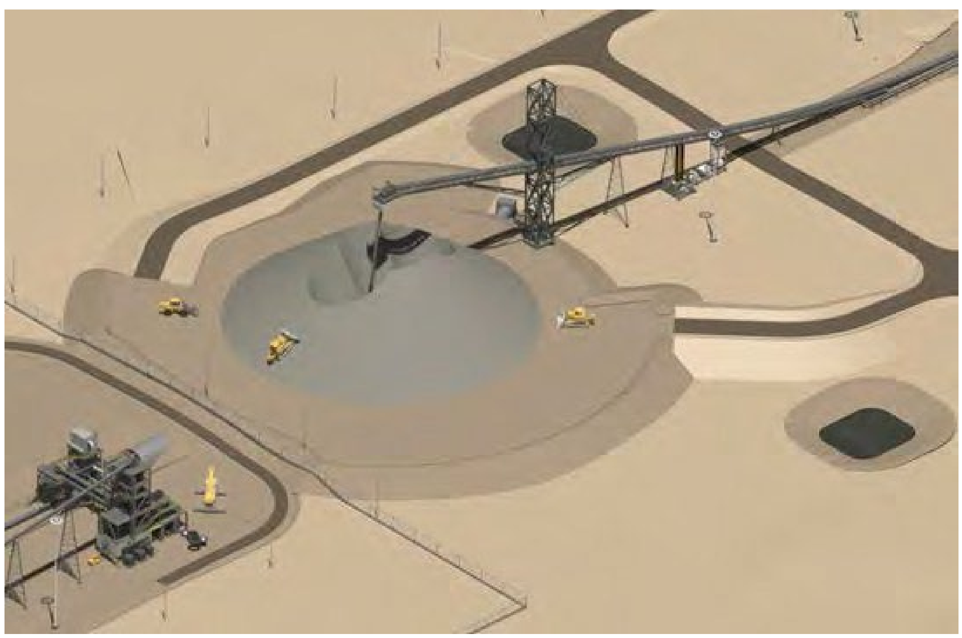


Figure 17-4: Ore stockpile

Grinding Circuit

The SAG mill operates in closed circuit with a pebble crusher and a primary cyclone cluster (Figure 17-5). A SAG discharge vibrating screen was added as part of the '24K' Project. Pebble circulating load varies from 5% to 20%, and cyclone circulating load from 150% to 250%. A portion of the primary cyclone underflow reports to the gravity circuit. The primary cyclone overflow pipe features an autosampler that contributes to the metallurgical accounting of the CIL plant.

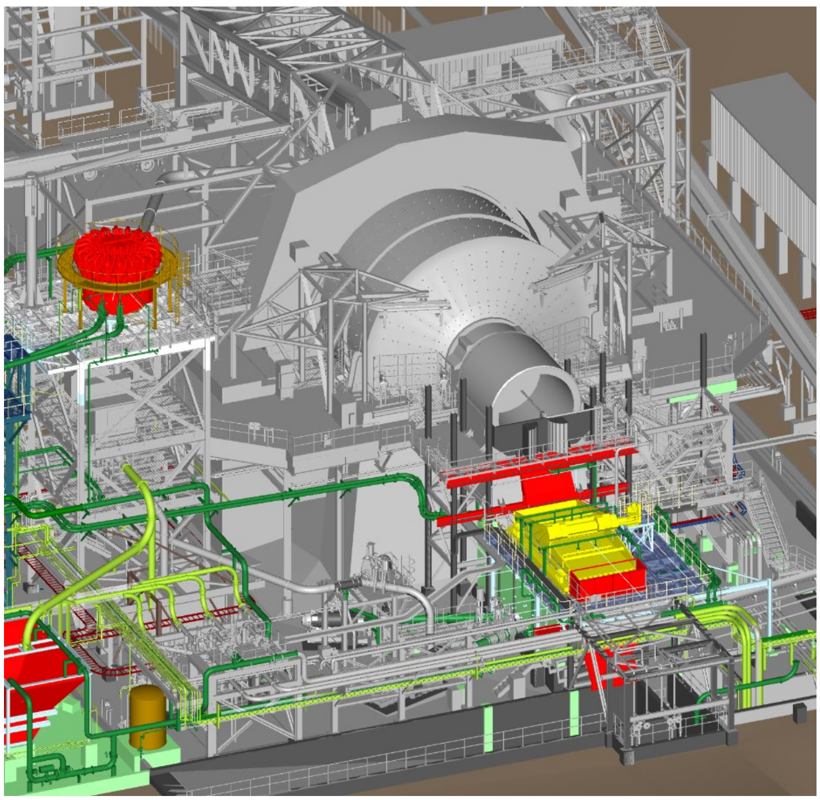


Figure 17-5: SAG circuit – with new vibrating screen and cyclones installed in '24K' Project

The Primary cyclone overflow, at a particle size P_{80} (80% passing) of 200 µm, is pumped to a pre-classification cyclone cluster. The cyclone overflow redirects the 'slimes', or finer particles at a size of P_{80} of 90 µm to a pre-leach thickener (previously the '12K' tailings thickener). The pre-leach thickener underflow is routed to linear trash screens before feeding the leaching circuit, and the underflow reports to parallel ball mills. The ball mills are in closed circuit with cyclones. The cyclone overflow at target particle size P_{80} of 90 µm reports to trash screens that then feed the leaching circuit.

Gravity Recovery and Intensive Leach Circuits

Two to six primary cyclone underflows report to two scalping screens that feed two Knelson concentrators for gravity recovery of coarse recoverable gold. The recovery from the gravity circuit ranges from 10% to 15% of the fresh feed to the SAG Mill. The equipment will be located on the east side of the grinding area (Figure 17-6).

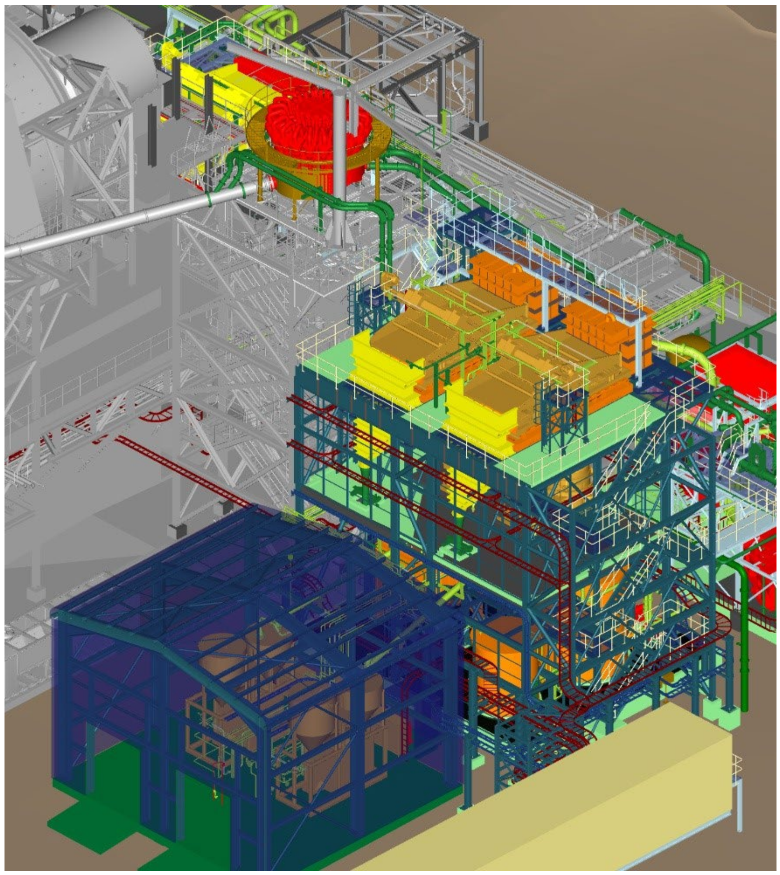


Figure 17-6: Gravity Concentration and ILR

The gravity concentrate is collected and treated separately in a batch process by intensive leach reactors (ILR) where the gold is dissolved, and the resulting pregnant solution is pumped to the gold room for electrowinning and smelting. Most of the gravity circuit is relocated equipment from its former location at the ball mills.

Leaching Circuit

The leaching circuit at Tasiast includes six leach tanks and seven CIL tanks (Figure 17-7). Three tanks from the ‘12K’ Project are still in operation, with two additional leach tanks constructed and commissioned during the ’21’ Project, and a final new leach tank added with the ‘24K’ Project. The leach circuit is fed at a slurry density target of 50% solids by weight and has a residence time of 12 hours at nominal flow.

The slurry flows by gravity through the leach tanks. The first leach tank can be used as a pre-aeration step with no cyanide addition and operate at the ‘natural’ pH of 8-8.5 produced by the grinding circuits. Lime is added to increase the slurry pH to 10.5 and sodium cyanide solution is added to maintain a target free cyanide concentration of approximately 250 ppm. High-pressure oxygen is sparged into the bottom of the leach tanks to enhance gold leaching kinetics. The leaching circuit’s purpose is to increase gold concentration in the solution before contact with activated carbon in the CIL circuit.

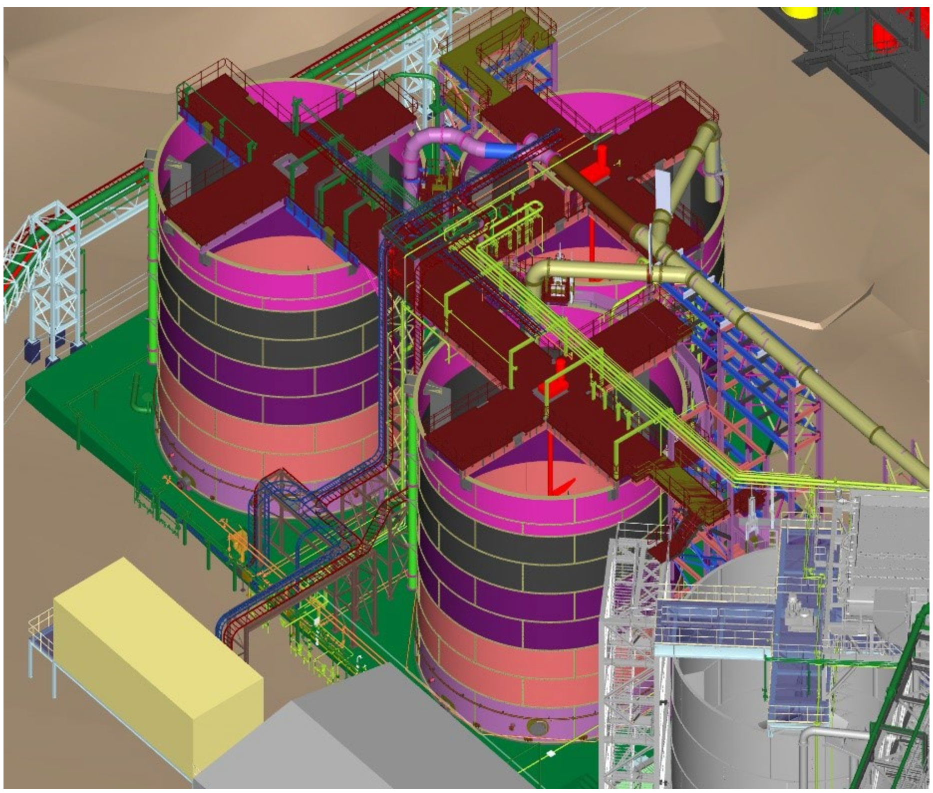


Figure 17-7: New Leach Tanks installed as part of the ‘21K’ and ‘24K’ Projects

The CIL tanks operate in a carbon-in-pulp (CIP) configuration but have retained the CIL naming convention due to historical operation as a CIL. As part of the ‘24K’ Project, seven new CIL tanks were built and commissioned in 2024. The seven new CIL tanks have a residence time of nine hours at nominal flow, bring the leach and CIL circuit total to 18 hours at nominal flow. In the CIL tanks, the dissolved gold in solution is recovered on activated carbon. Activated carbon granules contact slurry in the CIL tanks to adsorb the dissolved gold from solution. Carbon that has achieved the target gold content, ranging from 3,000 g/t to 5,000 g/t, is considered "loaded" and is transferred daily to the elution circuit for recovery of the gold on a batch basis. After each such transfer, carbon in the remaining tanks is advanced counter-current to the slurry flow on a sequential basis, and fresh or “barren” carbon is added to the last CIL tank to maintain the carbon inventory.

After maximizing gold recovery from the solution and ore particles in the leaching circuit, the resulting slurry flows via carbon safety screens to the tailings thickener, where the solids settle to a target slurry density of 60% solids by weight, and thickener overflow is returned to the plant as process water.

Carbon Elution, Electrowinning, and Refining

Loaded carbon recovered from the slurry in the CIL circuit by screening is first water washed to remove entrained ore particles and then washed with hydrochloric acid solution in a dedicated acid wash vessel to remove inorganics from the carbon surfaces. After being neutralized, the acid-washed carbon is transferred to the elution pressure vessel. To recover gold from the loaded carbon, batches of approximately 7.5 t of carbon are subjected to a high pressure and temperature, stripping process, called elution. Tasiast uses the Anglo American Research Laboratories (AARL) strip process. A hot caustic and cyanide rinse under pressure removes the gold from the carbon and into solution. After gold removal, the "barren" carbon is transferred to a regeneration kiln for thermal reactivation of the carbon. Reactivated carbon is returned to the last CIL tank.

Gold is recovered from the caustic solution by electrowinning onto stainless steel wire wool cathodes in electrowinning cells, located within the Gold Room. Two additional electrowinning cells were added as part of the ‘21K’ Project. The gold is removed as a sludge by pressure washing the cathodes at intervals. The sludge is dried and mixed with fluxing materials and charged to a diesel-fired crucible furnace. After melting, the slag is poured and followed by pouring the gold into bullion moulds. Bullion, or doré, contains approximately 94% or higher gold content together with a minor content of silver. The doré bars are transported by a security firm to a commercial refinery for further purification and sale.

Tailings Thickening

Slurry from the last CIL tank is pumped through carbon safety screens for the recovery of fine carbon, which feed a tailings thickener. The 40 m diameter tailings thickener increases the slurry density of the CIL circuit tails from 50% to approximately 60% solids by weight and recovers process water via its overflow. The safety screens and thickener were added in the ‘21K’ Project. The transfer line from the carbon safety screens underflow to the tailings thickener feed features an autosampler that contributes to the metallurgical accounting of the CIL plant. The layout of the tailings thickener is shown in Figure 17-8.

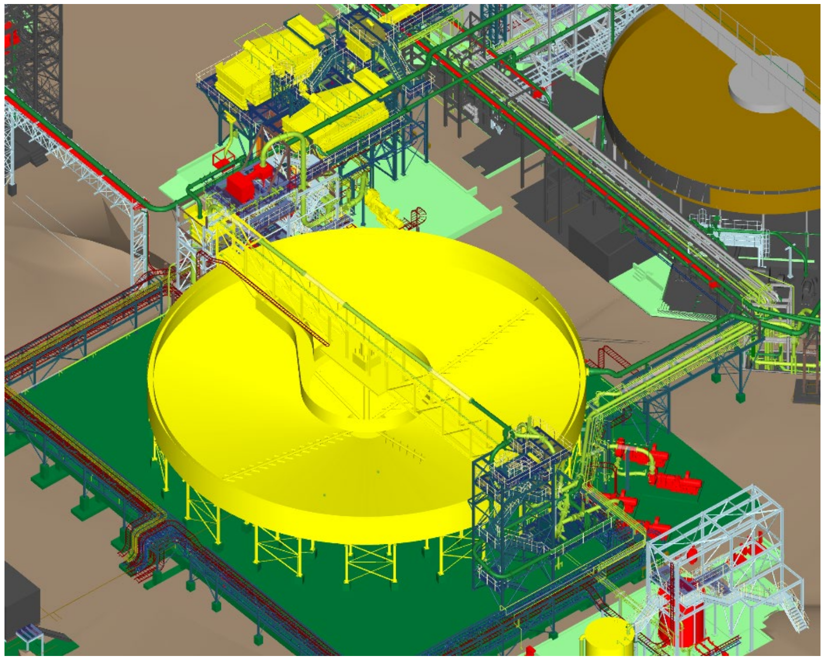


Figure 17-8: New Tailings thickener

Tailings Storage Facility

Slurry at the tailings thickener underflow is treated with a ferrous sulfate solution to detoxify the residual cyanide before being pumped to the TSF via a tailings pipeline. An intermediate tailings pumping booster station (TP3) allows transport of the slurry to different deposition points in the TSF. After settling of the solids, a portion of the contained water drains to a collection area within the storage basin, from there it is returned to the plant process water system as reclaim water. Solids are retained in the TSF.

The tailings pipeline and the reclaim water line are contained in a trench connected to five existing emergency ponds. In case of emergency or a prolonged shut down period, tailings slurry can be discharged into these ponds by operating the manual valves provided to prevent line blockage. The tailings pipeline design includes flushing points at regular interval along the length of the line to the TSF for manual line flushing. This is achieved by connecting a flushing hose between the slurry line and the duty tailings return water line.

Solution and Slurry Containment

All wet areas of the process plant will be bunded with a containment volume equal to 110% of the volume of the largest tank in the containment area, or 25% of the total combined tank volume in the case of hazardous materials.

Areas with specific requirements (design code requirements or incompatibility of solutions, such as hydrochloric acid and sodium cyanide) will be provided with separate containment.

The CIL plant is designed and operated to International Cyanide Management Code (ICMC) standards.

Utilities

Water Distribution

- **Process water**: The overflow of the thickeners (tailings and pre-classification thickeners) returns process water to the process water ponds, which is then repumped to the plant. Three process water pumps supply the process water circuit of the CIL plant.

- **Raw water**: Raw water is stored in two raw water ponds within the plant boundary. A new pumping station of two pumps (one operating and one standby) was installed for the '24K' Project. The pumps feed a raw water distribution network throughout the plant. Piping allows flushing of the unused lines back to the ponds.

Oxygen

The CIL plant features four vacuum swing adsorption (VSA) oxygen generation modules – three installed during the '12K' Project with a fourth installed for '21K' – and a recently installed portable pressure swing adsorption (PSA) containerized unit of five 'mini' modules. The VSAs and PSA are designed for an oxygen supply of 200 Nm^3/hr to 250 Nm^3/hr per unit at 93% purity. Oxygen is supplied at low or high pressure via dedicated booster compressors, with high pressure oxygen supplied to the bottom of the leach tanks via SlamJet spargers.

Reagent Mixing and Storage

Sodium Hydroxide (Caustic)

The '12K' sodium cyanide mixing and distribution system was converted to become the sodium hydroxide mixing and distribution system during the '21K' Project. The pumps from the previous system were relocated, and a previous cyanide transfer pump was retained for supplying caustic solution to the gravity recovery circuit.

Sodium Cyanide

A new sodium cyanide facility was constructed during ‘21K’ (Figure 17-9). Sodium cyanide is supplied in one tonne boxed bags as solid briquettes and dissolved in treated water to make a 20% w/w solution in the 50 m^3 mixing tank. Sodium hydroxide is added to safely dissolve the solid sodium cyanide in a high pH solution. During dissolution, the solution is maintained at a pH greater than 12 to avoid volatilization of hydrogen cyanide gas. The cyanide is pumped into a ring-main, and a dedicated pump services the gravity recovery circuit.

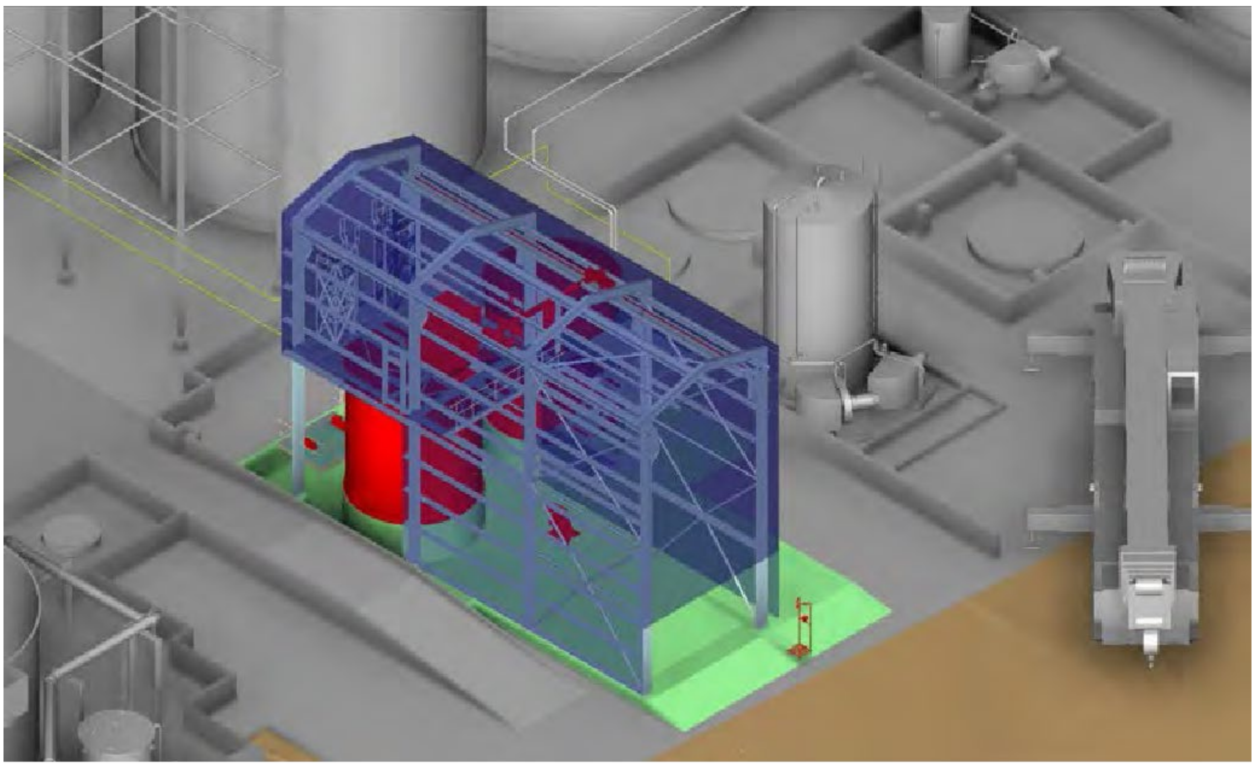


Figure 17-9: Sodium cyanide mixing layout

18. PROJECT INFRASTRUCTURE

18.1 Water Supply

Raw water for the Tasiast site is supplied from a bore field which is located 64 km west of the Complex, and draws from a brackish aquifer using a system of 47 boreholes. Individual well yields range from 265 m^3/d to 1,080 m^3/d based on 2024 production pumping. Three separate well areas are combined in a manifold for each area and fed to a primary pumping station located at a facility referred as the Sondage. Water from the Sondage is transported to site via pipelines with intermediate booster stations. The Sondage can supply the site with up to 19,800 m^3 of raw water per day.

The Tasiast permit, issued May 7, 2017 by the Ministry of Hydraulics and Sanitation, allows abstraction at a maximum rate of 30,000 m^3/d through to December 31, 2034. Modelling done by Piteau Associates, and calibrated against monitoring data, confirms the Sondage can be operated at this maximum rate and continue to meet the permit conditions.

Reverse osmosis (RO) water treatment plants and storage basins/tanks are located at the Complex. Saline water produced from the RO plant is used to water the haul roads or used in processing. Potable water is produced from RO water following additional disinfection steps and is used in certain processing applications as well as domestic purposes.

18.2 Power Supply

Electric power is provided by the following installed generation equipment:

- The solar plant consisting of 78,000 550W JA Solar Bifacial photovoltaic panels capable of generating up to 34 MWe and a battery system of 18 MWe.
- The Phase 2 plant consisting of four Wärtsilä W20V32TS simple cycle, medium speed, reciprocating engines (heavy fuel oil as primary fuel and light fuel oil as back-up) with a total capacity of approximately 40 MWe.
- The Phase 1B plant consisting of four Wärtsilä 12V32 medium speed generator sets (heavy fuel oil as primary fuel and light fuel oil as back-up), with a total capacity of approximately 19 MWe.
- The Phase 1 plant consisting of seven light fuel oil Caterpillar 3512 MUI high-speed generator sets and three heavy fuel oil Caterpillar MaK 6CM32C medium-speed generator sets, with a total capacity of approximately 14 MWe.

- The Tasiast Team Village (TTV) plant consisting of seven light fuel oil MTU Model 16V40000G23 high-speed generator sets, with a total capacity of 9 MWe.

All of Tasiast's power plants are interconnected and able to feed onto the 33 kV islanded distribution grid to supply the required site loads. The average demand is 40 MWe to 45 MWe, which is supplied by the combination of the approximately 82 MWe of thermal generation capacity and supplemented by an approximate 20% contribution from the solar plant based on daily and seasonal radiation availability.

The combined plants have the ability to meet the required net peak power demand at any expected ambient condition, while accounting for equipment fouling, ageing, power plant parasitic loads, as well as spinning reserve requirements. The combined facility has sufficient redundancy to be able to meet the maximum site power demand with the loss of availability of generating capability considering both scheduled and forced maintenance.

The power plants operate 24 hours per day, seven days per week and 365 days per year as an islanded operation.

18.3 Waste Management

Waste from plant and equipment maintenance, construction, offices, kitchens, and accommodation are processed at the waste management facility where materials are sorted for reuse, recycling, or incineration. Composters are also used in the camp to process food waste into compost for use in tree planting initiatives.

Sewage is collected and pumped to the wastewater treatment plant with treated effluent recycled back into the process or reused in road watering or vegetation projects. In remote locations septic tanks and leach beds are used.

18.4 Tasiast Team Village

The TTV is sized to accommodate a workforce of 3,540 personnel. It includes various facilities, such as clinic, laundry, kitchen and dining areas, gymnasiums, recreational rooms, and various sports fields.

18.5 Service and Administration Buildings

Service and administration buildings include:

- Guardhouse
- Warehouses

- Kitchen and dining hall
- Plant office building
- Security, Human Resources, General Management, IT, health, safety, and environment (HSE), and Site Services offices

18.6 Accessibility

The Tasiast Lands are accessed from Nouakchott by using the paved Nouakchott to Nouadhibou highway for 370 km and then via 66 km of graded mine access road which is maintained by TMLSA. An airstrip at the Complex is used for light aircraft, primarily travelling to and from Nouakchott.

The principal ports of entry for goods and consumables are either Nouakchott or Nouadhibou. Materials are transported by road to the Complex site.

Access to the major urban centres of Mauritania is also possible via air. Nouakchott is accessible via international flights operated by numerous West and North African carriers; Air France also provides a direct connection to Paris.

19. MARKET STUDIES AND CONTRACTS

Kinross typically establishes refining agreements with third parties for refining of doré. Kinross's bullion is sold on the spot market or as doré, by marketing experts retained in-house by Kinross. The QP is of the opinion that the terms contained within the refining contracts and sales contracts are typical and consistent with standard industry practice and are similar to contracts for the supply of bullion and doré elsewhere in the world.

20. ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Studies

Current mine operations and the ‘24K’ Project are based on the formal approval of a number of Environmental Impact Assessment (EIA) studies completed before and since mine commissioning in 2007.

For all deposit areas, environmental baseline conditions have been determined by reviewing existing published data, previous EIAs, satellite imagery, and environmental reporting undertaken for the Complex. Where appropriate, existing data for deposit areas was supplemented by primary data collected through environmental baseline surveys. Field-level baseline surveys were completed for deposit areas, including air quality, archaeology, flora, fauna, marine, water quality, traffic, and socioeconomics.

The baseline conditions formed the basis to assess the project through a series of EIAs and Environmental Impact Notices (EINs). The environmental assessments used applicable Mauritania legislation, the International Finance Corporation Performance Standards, the International Cyanide Management Code, and Kinross Health, Safety, Environment and Social Management Systems for project design and management, mitigation strategies, and performance monitoring. The environmental assessments determined appropriate mitigation and management where impacts could not be avoided through project design.

A geochemical review of waste rock, based on historical laboratory data, indicates excess neutralizing capacity, suggesting a low potential for ARD. Given this capacity and the minimal precipitation at Tasiast, ARD is unlikely. Additional test work is planned to corroborate these findings at depth, as the initial studies were based on samples collected before 2015.

The Tasiast facilities operate under an environmental management system (EMS) that specifies activities to be planned and implemented by the Complex’s environmental management team. The EMS incorporates the project design and management, mitigation strategies, and performance monitoring commitments outlined in the environmental assessments, applicable legislation, and specific permit requirements.

An element of each EIA prepared for the Tasiast Mine site is a preliminary reclamation and closure plan and associated cost estimate. The preliminary reclamation and closure plan outlines the measures that will be taken to reclaim and close the proposed activities assessed in each EIA. The preliminary reclamation and closure cost estimate forms the basis of the financial assurance. Tasiast, with the support of the SRK, updated its financial assurance in 2024 including the 24,000 kt/day expansion. The estimated closure cost is approximately US$64 million. This financial assurance will be submitted to the Mauritanian regulators for their review and approval in February 2025. Once the government of Mauritania validates the assurance, Tasiast will issue a new letter of credit that reflects the remaining amount, taking into account the existing financial assurance of US$6.2 million. At least two years before entering closure, a detailed reclamation and closure plan must be submitted to the appropriate ministries for approval.

20.2 Permitting

In addition to the exploitation permit No. 229C2 (Section 4.2) and the adjacent exploitation permits, all other necessary permits for exploiting the Tasiast Complex have been granted by the relevant Mauritanian authorities. A Phase 3 EIA for “off-site” sea water supply was approved in 2014 following submission of a Phase 3 addendum. A subsequent EIA was approved to allow receipt of pre-assembled equipment at a beach landing and transportation to site. In addition, following discussion with the Government, an addendum to the Phase 2 EIA was submitted and approved that described the project optimization through incremental increases in production and relocation of certain infrastructure. This addendum was approved by the Ministry of Environment in February 2016 and the Ministry of Mines in March 2016. An authorization for the installation of 15 additional wells at the Sondage was received from the Mauritanian government on September 21, 2020. Meanwhile, Tasiast is working on a new Environmental Impact Assessment (EIA) for the northern satellite pit, known as the Fennec project, intending to complete the process within the second quarter of 2025. The key permits are shown in Table 20-1.

Table 20-1: Key operating permits and environmental assessments

Brief Name	Issue Date
Authorization to construct a water pipeline route to the mine	MMI Letter 090 – 23 May 2006
Authorization of water extraction (12 boreholes)	MHE Letter 560 – 24 July 2008
Original EIA permit for Tasiast Mine	MEDD Letter 407 – 27 August 2009 MIM Letter 264 – 27 August 2009
New developments EIA permit (dump leach, TSF2)	MEDD Letter 408 – 27 August 2009 MIM Letter 264 – 27 August 2009
Groundwater abstraction permit for 30,000 m^3 per day	MHA Letter 125 – 25 April 2011
Environmental authorization for Phase 1ai and Phase 1aii environmental impact notices	MEDD Letter 151 – 16 June 2011 MEDD Letter 166 – 10 July 2011
Environmental authorization for West Branch development (EIA)	MEDD Letter 665 – 10 October 2011 MPEM Letter 1209 – 25 October 2011
Environmental authorization for Phase 1b development (EIA)	MEDD Letter 713 – 18 October 2011 MPEM Letter 1210 – 25 October 2011
Environmental authorization for Phase 2 development (EIA)	MEDD Letter 556 – 19 July 2012 MPEM Letter 1049 – 25 July 2012
Conditional environmental authorization for Phase 3 development (EIA)	MEDD Letter 600 – 22 September 2013 MPEM Letter 844 – 6 October 2013
Extension of groundwater abstraction permit for 30,000 m^3 per day until 2020	MHA Letter 6 – 7 January 2014
Addendum Phase 3 EIA, full authorization for Phase 3	MEDD Letter 605 – 21 June 2014 MPEM Letter 751 – 31 July 2014
Beach Landing EIA	AZFN Letter 2 – 13 January 2014
Addendum Phase 2 and Environmental Authorization	MEDD Letter 59 – 4 February 2016 MPEM Letter 276 – 30 March 2016
Extension of groundwater abstraction permit for 30,000 m^3 per day until 2034	MHA Letter 94 – 9 May 2017
Authorization for drilling of 15 new wells for Tasiast Sondage	MHA letter 149 – 21 September 2020
Construction of Tasiast solar PV plant permit	MEDD/DECE Lettre 150 – 14 April 2022

Note:

1. MMI = Ministère des Mines et de l'Industrie
2. MHE = Ministère de l'Hydrqulique et de l'Energie
3. MEDD = Ministère de l'Environnement et du Développement Durable

4. MIM = Ministere de l'Industrie et des Mines
5. MHA = Ministère de l'Hydraulique et de l'Assainissement
6. MPEM = Ministère du Pétrole, de l'Energie et des Mines
7. AZFN = Autorité de la Zone Franche de Nouadhibou

20.3 Socio-Economics

Mauritania is divided into 12 wilayahs (regions), one district (Nouakchott), 53 moughataas (counties), and 208 communes (municipalities).

The Complex is located in the Inchiri wilayah, which has a very low population density. The wilayah includes the Akjoujt moughataa and two main municipalities, Akjoujt and Bennichab, Akjoujt being the administrative capital with a population of approximately 8,500. The wilayah is administered by a council, directed by a governor (wali) who reports to the Minister of Interior. The basic administrative unit, the moughataa, is directed by a prefect (Hakem) who exercises his power under the authority of the governor.

Inchiri is the least populated wilayah in the country, with the nomadic way of life being a key feature making up 20% of the total population. There tends to be a small number of nomadic people in the vicinity of the Tasiast Complex. The Complex is located 80 km northeast from the nearest permanent community of Chami.

The nearest industries to the site are in the towns of Chami, Boulanour, Akjoujt, and Bennichab, which are respectively 80 km southwest, 120 km northwest, 150 km east-southeast and 130 km southeast from the site.

There are no permanent settlements within the vicinity of the Complex, however, a number of isolated families have set up structures within 30 km of the site and reside, predominantly within three communities. Residents practice animal husbandry and other subsistence forms of livelihood. There are also nomadic groups that occasionally transit the area.

21. CAPITAL AND OPERATING COSTS

This section covers Capital and Operating Costs for the Tasiast Mine.

21.1 Capital Costs

Capital costs for Tasiast are summarized in Table 21-1 and Table 21-2 and show sustaining capital and non-sustaining capital, respectively.

After the completion of the '24K' Project in 2023, go-forward capital costs are largely comprised of:

- Non-sustaining capitalized development for West Branch 5 stripping and non-sustaining equipment maintenance (Table 21-1).
- Sustaining capital requirements across the Complex site (maintenance, tailings, mill, etc.) (Table 21-2).

Capital costs are higher earlier in the remaining mine life, primarily due to the up-front investment from West Branch 5 [and Piment] waste stripping, maintenance strategies to optimize maintenance capital, and tailings capital being spent ahead of deposition.

Capital costs are estimated on a real basis, using January 1, 2025 dollars to estimate, and reflect the delivered cost of any equipment.

Table 21-1: Annual sustaining capital

Year	Sustaining Capitalized Stripping (US$ million)	Mining Sustaining Capital (US$ million)	Mill Sustaining Capital (US$ million)	Tailings Sustaining Capital (US$ million)	Other Sustaining Capital (US$ million)
2025	$21.1	$42.2	$16.3	$8.3	$6.0
2026	$3.2	$5.6	$6.3	$9.6	$1.5
2027	$12.2	$0.8	$4.4	$7.9	$5.2
2028	-	$0.6	$4.2	$7.9	$4.3
2029	-	$0.6	$3.9	$7.9	$3.3
2030	-	-	$3.9	$7.9	$2.8
2031	-	-	$3.9	$7.9	$3.0
2032	-	-	$2.6	$7.9	$1.7
2033	-	-	$2.6	-	$1.7
2034	-	-	-	-	-
2035	-	-	-	-	-
2036	-	-	-	-	-
2037	-	-	-	-	-
Total	36.5	49.8	48.3	65.3	29.3

Table 21-2: Annual non-sustaining capital

Year	Non- Sustaining Capitalized Stripping (US$ million)	Mobile Equipment Maintenance Non-Sustaining Capital (US$ million)	Other Non-Sustaining Capital (US$ million)
2025	$238.6	$18.6	$2.5
2026	$172.9	$16.9	-
2027	$47.3	$19.9	-
2028	$131.5	$11.0	-
2029	$26.5	$5.7	-
2030	$11.4	-	-
2031	-	$0.2	-
2032	-	-	-
2033	-	-	-
2034	-	$0.9	-
2035	-	-	-
2036	-	-	-
2037	-	-	-
Total	**628.1**	**73.1**	**2.5**

21.2 Estimate Basis – Non-sustaining and Sustaining Capitalized Stripping

Stripping costs, which provide future economic benefits and identifiable improved access to the orebody, can be capitalized. Estimates for capitalized stripping are the pro-rated mining cost within each period, spread based on operating vs. capitalized tonnes mined.

Capitalized mining cost is built up using a zero-based operating cost model, constructed in XERAS software platform. This uses forward-looking estimates for the key cost drivers and are supported by contractual agreements, 2025 labour rates, actual consumable prices and rates, and a detailed mine plan schedule to build up the cost drivers, such as tonnes mined, equipment hours, etc.

Per World Gold Council guidance, the capitalized portion of West Branch waste movement is considered non-sustaining LOM, as it exposes long-term ore sources. Piment and Fennec waste stripping is considered sustaining capital, due to the smaller pit sizing and shorter duration of stripping.

21.3 Estimate Basis – Non-sustaining and Sustaining Mobile Equipment Maintenance

Mobile equipment maintenance costs include fleet purchases, planned component replacements of major fleets, and the capitalized portion of maintenance spend for the mobile fleet of equipment.

Planned fleet purchases include some replacement capital over the LOM, including two CAT 793 haul trucks and other support equipment. Furthermore, to support mining at the Fennec and Prolongation satellite pits, a small fleet of equipment is planned for purchase in early 2025 , including:

- eight CAT 777 haul trucks
- one CAT 992 loader
- one CAT 6030 shovel
- two CAT D10 dozers
- two CAT 16M graders
- Other miscellaneous support equipment

Capitalized maintenance spend reflects a detailed maintenance cost forecast. This involves zero-based logic to calculate the timing of the key maintenance events over the life-of-mine. Key inputs include:

- 2025 opening balance component hours
- Expected component life
- Delivered component pricing

The maintenance model is predicated on a run-to-failure maintenance strategy, where work is assumed to reduce towards the end of the mine life, not conducting major capital component replacements in the final stages of the mine life. This is to ensure a more capital efficient approach to the mine.

21.4 Estimate Basis – Mill Sustaining Capital

Capital spending in the mill is largely routine in nature without any major upgrades or replacements planned in the LOM. Routine spending includes planned equipment and component replacement to maintain the plant and deliver the planned availability.

21.5 Estimate Basis – Tailings Sustaining Capital

The active TSF is a combined facility, TSF 4/5. Based on planned dam raises to the facility, the TSF can meet the required LOM tailings capacity. The TSF is a rectangular-shaped ring embankment with a downstream construction method.

The tailings sustaining capital estimate is primarily for dam raises constructed at approximate two to three year intervals through the remaining mine life. The estimate is based on a combination of contractor quotes and site experience with construction of past dam raises. The main cost components are zoned earthfill placement and geomembrane installation (by contractors) and construction of the downstream rock fill dam shell, which is self-performed using mine mobile equipment.

21.6 Operating Costs

Basis of Estimate – Operating Costs

The Tasiast LOM operating costs are split into five primary categories: Mining, Processing, Site Administration, Royalties, and Other. See Table 21-3 for a summary of the basis of estimate for these categories.

Operating costs are estimated on a real basis, using January 1, 2025 dollars. Approximately 50% of LOM operating costs are based on the Mauritanian Ouguiya, with the remaining 50% exposed to US Dollars.

Labour rates are estimated using existing role profiles and labour rates at Tasiast, adjusted for 2025 dollars moving forward. Consumables prices generally reflect a combination of current pricing, and near-term projections for some major global indices (i.e., caustic, ammonia, etc.), which have been provided by specialized third party advisors.

Operating costs have been adjusted downwards beyond 2030 to reflect the reduced scale of operation that would come from just stockpile rehandling and processing. This reflects assumptions around limited load/haul requirements from stockpile and associated reductions in other cost drivers like headcount, processing fixed costs, etc.

Table 21-3: Basis of estimate – operating costs

Operating Cost Category	Estimate Basis

Mining	Developed from first principles by:
	• Developing a detailed mine plan schedule.
	• Defining a haulage network (specific to the detailed mine plan) and generating truck hours based on travel distance, speed, and fixed non-travel time.
	• Applying key cost parameter inputs such as:
	  o Input prices (diesel, blasting consumables and tires) from existing site contracts.
	  o Productivity – rates have been baselined to existing productivity rates on-site.
	  o Headcounts – fitted to the scale of the mine (i.e., fewer operator and non-operator positions would be required as mining rate decreases).
	  o Fuel burn rates – based on expected LOM fuel burn rates, baselined to actual levels.
	  o Maintenance costs, calculated from a zero-based maintenance model (tracks and schedules maintenance events for each piece of equipment at site by operating hours).
	  o Other inputs, such as tire life and drill consumption rates – based on existing site strategy and experience.
	The following costs are allocated by department and based on actuals adjusted for changes in mining headcount and rates:
	• Tasiast Team Village (camp).
	• People mobility.
	• Nouakchott accommodations.
	• Power.
	• Water.

Processing	Estimation methodology varied by cost component, but primarily built from first principles, relying on a combination of: • Knowledge from existing operations. • Laboratory testing. • Energy consumption estimates per motor. • Mass and water balance. Major categories include the following: • Power. • Consumables (i.e., liner, grinding balls). • Reagents. • Labour. • Maintenance. • Water.

Operating Cost Category	Estimate Basis
	• Laboratory. • Plant administration. The following costs are allocated by department and based zero-based models, which calculate power supply cost, camp costs, etc.: • Tasiast Team Village (camp). • People mobility. • Nouakchott accommodations. • Power. • Water.
Site Administrative	Bottom-up approach applying labour, and other costs to various areas including: • Administration (finance, supply chain, security, IT, HR, etc.) • Insurance. • Heath, safety and environment. • Training. • Site services. • Refining and shipping costs. The following costs are allocated by department and based on zero-based models, which calculate power supply cost, camp costs, etc.: • Tasiast Team Village (camp). • People mobility. • Nouakchott accommodations. • Power. • Water.
Royalties	This category captures royalties paid by the mine: • Franco Nevada Royalty (2.222% of production). • Price dependent government royalty as follows: Au Price < $1,000/oz 4.0% of Revenue Au Price $1,000/oz ≤ Au Price < $1,200 4.5% of Revenue Au Price $1,200/oz ≤ Au Price < $1,400 5.0% of Revenue Au Price $1,400/oz ≤ Au Price < $1,600 5.5% of Revenue Au Price $1,600/oz ≤ Au Price < $1,800 6.0% of Revenue Au Price > $1,800 6.5% of Revenue
Other	This category captures all operating costs not considered in the three categories above, including: • World Gold Council fee. • Training and development spend

Operating cost estimates are not escalated over time and all estimates are on a real basis. LOM operating cost estimates are shown in Table 21-4.

Table 21-4: Operating cost estimates (January 1, 2025 forward)

Operating Cost	Unit	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	LOM 2025-2035
Mining	US$/t mined	3.9	3.9	3.9	3.6	3.5	4.2	0.0	0.0	0.0	0.0	0.0	3.8
Processing (Mill)	US$/t processed	16.2	15.7	15.9	15.1	15.0	14.7	13.6	13.2	13.4	12.8	12.7	14.4
Site Admin	million US$/a	112	106	101	98	96	75	41	41	34	30	17	754
Royalties	US$/oz sold	186	187	188	187	187	187	187	187	187	187	187	187
Other	US$/oz sold	2.2	2.3	2.3	1.7	1.7	2.2	4.4	4.6	5.0	5.1	2.2	2.6

Operating Cost	Unit	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	LOM 2025-2035
Mining (incl. capitalized stripping)	million US$/a	304	316	296	247	205	126	21	19	25	25	19	1,602
Processing (Mill)	million US$/a	142	138	140	133	132	129	119	116	117	112	106	1,384
Site Admin	million US$/a	112	106	101	98	96	75	41	41	34	30	17	754
Royalties	million US$/a	92	87	85	115	115	90	44	42	39	38	35	782
Other	million US$/a	1	1	1	1	1	1	1	1	1	1	0	11

22. ECONOMIC ANALYSIS

Under NI 43-101 guidelines, a producing issuer may exclude the information required for Item 22 – Economic Analysis on properties currently in production for a case that does not include a material expansion of current production. Kinross is a producing issuer, the Tasiast mine is currently in production, and the current LOM does not consider a material expansion. Kinross has carried out an economic analysis of the LOM case consistent with technical information in this Technical Report and confirms that the outcome is a positive cash flow.

23. ADJACENT PROPERTIES

To the north of the Tasiast property, SENISA, a sister company of TMSLA, holds two mining permits that are contiguous wit the Tasiast mining permit land. There are exploration prospects in all three exploitation permit locations.

24. OTHER RELEVANT DATA AND INFORMATION

No additional information is necessary for this Technical Report.

25. INTERPRETATION AND CONCLUSIONS

- Tasiast is viewed as a long-term strategic asset for Kinross, located in a district that is believed to have significant future potential.
- The Tasiast gold deposits fall into the broad category of orogenic gold deposits. Tasiast gold deposits are hosted in Archaean volcanic-sedimentary sequences that have been deformed and metamorphosed to lower amphibolite peak metamorphic grade. Mineralization is both structurally and lithologically controlled and is epigenetic in style.
- The Complex is currently operated as an open pit. Mineral Resources include both open pit and underground conceptual scenarios.
- There is a good understanding of the geology and the nature of gold mineralization at the Complex. The lithology model represents the support data well and it is developed using appropriate resolution.
- The Mineral Resource estimate is of sufficient quality to support public disclosure and is supported by best practice guidelines.

26. RECOMMENDATIONS

1. Foster the already-strong Continuous Improvement culture, looking for further opportunities to improve operating and cost performance in the mine, plant, and administration. In particular, focus on opportunities to add production to the 2025 to 2027 window while stripping is ongoing at West Branch 5.

2. Continue to explore the underground potential at Tasiast. High potential exists to continue to expand upon underground resources at Tasiast at West Branch and Piment but also to establish underground higher-grade resources at Prolongation.

3. Evaluate further push backs at West Branch and other open pits given the elevated gold price environment.

4. Evaluate opportunities to strategically stockpile lower grade material, which could be processed at the end of the mine life in a higher metal price environment. Furthermore, this lower grade material could be used to keep the mill full, if successful in converting the current underground resource. At YE 2024, Tasiast added 152 koz of low grade which will be stockpiled for the end of mine life. Future mine plans should look to add this to the back end of mine life, as well as evaluate other opportunities.

5. Continue to explore the addition of other satellite ore bodies to the mine plan such as C67 and C68. The recent addition of Fennec to the mine plan highlights the success that can come from detailed technical review of some of these higher-grade satellite opportunities.

6. Given the positive reconciliation of 106% on F3 ounces seen in 2024, continue to drill targeting higher grade plunge controlled mineralization in the West Branch 5 push back.

27. REFERENCES

Blake, C., 2011a. Mineralogical characterisation of five gold-bearing samples from the Tasiast mine, Mauritania for Kinross Gold Corporation. Internal unpublished report, February 2011.

Blake, C., 2011b. Mineralogical characterisation of seven gold-bearing composite samples from the Tasiast Mine, Mauritania for Kinross Gold Corporation. Internal unpublished report, September 2011.

Canadian Institute of Mining, Metallurgy and Petroleum (CIM), 2014. CIM Standards for Mineral Resources and Mineral Reserves, Definitions and Guidelines: Canadian Institute of Mining, Metallurgy and Petroleum, May 10, 2014 (CIM (2014) Definitions).

Chartier, D., 2013. Draft of Sampling and Data Verification Sections for Tasiast Feasibility Study, memo dated December 4, 2013.

CIM, 2019. CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines, adopted by the CIM Council on November 29, 2019 (MRMR Best Practice Guidelines).

Davis, B., 2018. Tasiast gold deposit, Mauritania – A structural geological review. Kinross Gold Corporation.

Demers, P., Gauthier, D., Kroon, A.S., and Lafleur P-J., 2004. Technical Report on the Tasiast Gold Project, Islamic Republic of Mauritania: unpublished technical report prepared by SNC Lavelin for Defiance Mining Corporation Limited, effective date May 27, 2004.

Fabre, J., 2005. Géologie du Sahara occidental et central. Série/Reeks: Tervuren African Geosciences Collection, MRAC Tervuren, Belgique, p. 572.

Golder Associates, 2004. Preliminary Pit Slope Design, Tasiast Project, Mauritania, Report 03- 117-073. Unpublished Technical Report, June 2004.

Golder Associates, 2014. Geotechnical Study in Support of West Branch Expansion, Kinross Tasiast, Report 12514150422.502/A.0. Unpublished Technical Report, June 2014.

Guibal, D., Uttley, P., de Visser, J., and Warren, M., 2003. Independent Technical Assessment Report on the Tasiast Project, Mauritania; Report Prepared for Midas Gold plc. and Geomaque Explorations Ltd: unpublished technical report prepared by SRK Consulting for Midas Gold plc. and Geomaque Explorations Ltd, effective date March 4, 2003.

Heberlein, D., 2011. QAQC update – laboratory duplicate results (Oct-Dec 2011). Internal unpublished report (October 2011).

Heberlein, D., 2013. QAQC Update (August 2012). Internal report prepared for Tasiast Mauritanie Ltd S. A.; a division of Kinross Gold Corp, September 4, 2013.

Heron, K., Jessell, M., Benn, K., Harris, E., and Crowley, Q.G., 2016. The Tasiast deposit, Mauritania. Ore Geology Reviews 78: 564–572.

Hyde, R., 2003. Tasiast Gold Project, Review of Sampling Procedures and QAQC Analysis for Midas Gold plc.: RSG Global Report No. TAS003, 42 pages (unpublished).

JKTech, undated. JK Drop Weight Test, "In Detail" white paper by JKTech Laboratory Services. Leitch, C., 2010. Petrographic report on 18 samples from Archean greenstone au deposit. Internal unpublished report, October 2010.

Lahondere, D., Thieblemont, D., Goujou, J.C., Roger, J., Moussine-Pouchkine, A., Le Metour, J., Cocherie, A., and Guerrot, C., 2003. Notice explicative des cartes géologiques à 1/200,000 et 1/500,000 de l'extrême sud de la Mauritanie: DMG, Ministère des Mines et de l'Industrie, Nouakchott, Rapport BRGM/RC-54273-FR, p. 610.

Larson, T., 2011. Petrography: Tasiast Project (WestBranch_Petrography). Unpublished report.

Leroux, D.C. and Puritch, E., 2003. Technical Report and Resource Estimation on the Tasiast Gold Project Islamic Republic of Mauritania for Defiance Mining Corporation: unpublished technical report prepared by ACA Howe for Defiance Mining Corporation, effective date October 10, 2003.

Leroux, D.C., Roy, W.D., and Orava D., 2007. Technical Report on the Tasiast Gold Project Islamic Republic of Mauritania for Red Back Mining Inc.: unpublished technical report prepared by ACA Howe for Red Back Mining Inc., effective date July 20, 2007.

Panterra, 2012. Petrographic Study of the Tasiast Deposit, Mauritania, West Africa, November 7, 2012.

Panterra, 2017. Petrographic Report on the Tasiast West Branch Pit, Mauritania, West Africa, Feb 18, 2017.

Pitfield, P.E.J., Key, R.M., Waters, C.N., Hawkins, M.P.H., Schofield, D.I., Loughlini, S., and Barnes, R P., 2004. Notice explicative des cartes géologiques et gîtologiques à 1/200 000 et 1/500 000 du Sud de la Mauritanie, Volume 1 Géologie: DMG Ministère des Mines et de l'Industrie, Nouakchott, p. 314.

Pollard, P., 2011. Hydrothermal Alteration and Mineralization in the Lower West Branch Zone, Tasiast Gold Mine, Mauritania: unpublished technical report prepared for Kinross Gold.

Schlumberger, 2014. Hydrogeological Characterisation of the West Branch Pit, Tasiast, Mauritania. Technical Report Ref 52116/R1, January 2014.

Schofield, D., Horstwood. M.S.A., Pitfield, P.E.J., Crowley, Q.G., Wilkinson, A.F. & Sidaty, H.C.O., 2006. Timing and kinematics of Eburnean tectonics in the central Reguibat Shield, Mauritania. Journal Geological Society London, 163, 549-560.

Scott Wilson, 2008a. Tasiast Gold Project, Environmental Impact Study, Addendum II of IV, Environmental Impact Review of Tailings Storage Facility: Report prepared by Scott Wilson Limited for Tasiast Mauritania Limited S.A. for submission to Mauritanian Government, February 2008.

Scott Wilson, 2008b. Tasiast Gold Project, Environmental Impact Study, Addendum III of IV, Environmental Management Plan: Report prepared by Scott Wilson Limited for Tasiast Mauritania Limited S.A. for submission to Mauritanian Government, February 2008.

Scott Wilson, 2008c. Tasiast Gold Project, Environmental Impact Study, Addendum IV of IV Preliminary Rehabilitation and Closure Plan: Report prepared by Scott Wilson Limited for Tasiast Mauritania Limited S.A. for submission to Mauritanian Government, February 2008.

Scott Wilson, 2009. Tasiast Gold Project, Environmental Impact Study for New Developments. Scott Wilson, 2010. Tasiast Gold Mine West Branch Development: Environmental Impact Assessment.

Scott Wilson, 2011. Tasiast West Branch Slope Stability Analysis.

Sims, J., 2019. Tasiast Project, National Instrument 43-101 Technical Report: prepared for Kinross Gold Corporation, effective date October 31, 2019.

SLR, 2024. Tasiast Mine. Audit Review of YE 2023 MRMR.

Strashimirov, S., 2010. Petrological and mineralogical studies of 10 samples from Mauritania. Internal unpublished report, August 2010.

Stuart, H., 2008. Technical Report on the Tasiast Gold Mine Islamic Republic of Mauritania for Red Back Mining Inc.: unpublished technical report prepared for Red Back Mining Inc., effective date May 24, 2008.

Stuart, H., 2009. Technical Report on the Tasiast Gold Mine Islamic Republic of Mauritania for Red Back Mining Inc.: unpublished technical report prepared for Red Back Mining Inc., effective date May 8, 2009.

Stuart, H., 2010. Technical Report on the Tasiast Gold Mine Islamic Republic of Mauritania for Red Back Mining Inc.: unpublished technical report prepared for Red Back Mining Inc., effective date September 6, 2010.

28. DATE AND SIGNATURE PAGE

This Technical Report entitled "Tasiast Mine, Mauritania - National Instrument 43-101 Technical Report" with an effective date of December 31, 2024 was prepared by the following authors:

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(Signed and Sealed) ***Agung Prawasono***

Agung Prawasono , P.Eng.
Sr. Director, Mine Planning
March 27, 2025

(Signed and Sealed) ***Yves Breau***

Yves Breau, P.Eng.
Vice President, Metallurgy & Engineering
March 27, 2025

(Signed and Sealed) ***Graham Long***

Graham Long, P.Geo.
Vice President, Exploration
March 27, 2025

(Signed and Sealed) ***Jacob Brown***

Jacob Brown, SME (RM)
Director, Resource & Mine Geology
March 27, 2025

(Signed and Sealed) ***Kevin van Warmerdam***

Kevin van Warmerdam, P. Eng.
Sr. Director, Engineering & Energy
March 27, 2025

29. CERTIFICATE OF QUALIFIED PERSON

29.1 Nicos Pfeiffer

I, Nicos Pfeiffer, P.Geo., as an author of this report entitled "Tasiast Mine, Mauritania - National Instrument 43-101 Technical Report" with an effective date of December 31, 2024, prepared for Kinross Gold Corporation, do hereby certify that:

1) I am Vice President, Geology & Technical Evaluations and Company QP with Kinross Gold Corp. of 25 York Street, 17th floor, Toronto, Ontario.
2) I am a graduate of Carleton University, Ottawa, Ontario in 2009 with an Honours B.Sc. Earth Science.
3) I am registered as a Professional Geologist in the Province of Ontario (Reg# 2354). I have over 15 years of mining industry experience. My relevant experience for the purpose of the Technical Report is:
    - Domestic and international experience in both underground and open pit operational geology roles as well as exploration and resource estimation.
    - Experience leading multi-disciplinary technical teams in both a corporate and operational capacity.
4) I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5) I last visited the Tasiast Mine on 25 – 28 November, 2024.
6) I am responsible for Sections 3-6, 20, 23, 24, and relevant portions of 1, 2, 25, 26, 27 of the Technical Report.
7) I am not independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
8) I have had prior involvement with the property that is the subject of the Technical Report.
9) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the 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 27th day of March, 2025

(Signed and Sealed) ***Nicos Pfeiffer***

Nicos Pfeiffer, P.Geo.

29.2 Agung Prawasono

I, Agung Prawasono, P.Eng., as an author of this report entitled “Tasiast Mine, Mauritania - National Instrument 43-101 Technical Report” with an effective date of December 31, 2024, prepared for Kinross Gold Corporation, do hereby certify that:

1) I am Senior Director, Mine Planning with Kinross Gold Corp. of 25 York Street, 17th floor, Toronto, Ontario.
2) I am a graduate of UPN “Veteran” Yogyakarta, Indonesia in 1999 with a Mining Engineer Degree.
3) I am a Professional Engineer in Professional Engineers Ontario (No. 1005533117). I have worked as a mining engineer for a total of 25 years since my graduation. My relevant experience for the purpose of the Technical Report is:
    - A total of 25 years experience in resource optimization related works that includes mine designs and mine planning for precious and base metal operations and projects in Indonesia, India, Africa, North America, and South America.
4) I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5) I last visited the Tasiast Mine on 23 February – 2 March, 2025.
6) I am responsible for Sections 15 and 16 and relevant portions of 1, 2, 25, 26, 27 of the Technical Report.
7) I am not independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
8) I have had prior involvement with the property that is the subject of the Technical Report.
9) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the 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 27th day of March, 2025

(Signed and Sealed) ***Agung Prawasono***

Agung Prawasono, P.Eng.

29.3 Yves Breau

I, Yves Breau, P.Eng., as an author of this report entitled “Tasiast Mine, Mauritania - National Instrument 43-101 Technical Report” with an effective date of December 31 2024, prepared for Kinross Gold Corporation, do hereby certify that:

1) I am Vice President, Metallurgy & Engineering with Kinross Gold Corporation, of 25 York Street, 17th Floor, Toronto, Ontario, M5J 2V5.
2) I am a graduate of University of Laval, Québec City in 1997 with a B.Sc. in Materials and Metallurgy Engineering.
3) I am registered as a Professional Engineer in the Province of Ontario (Reg.# 100194755). I have worked as an engineer for a total of 26 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - My work experience has included multiple operations roles from metallurgist to process manager and multiple mining company corporate roles from manager to Vice-President.
   - In my roles in operations and corporate, I have completed many studies related to gold mineral processing.
4) I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.
5) I last visited the Tasiast Mine on 4 – 10 August, 2024.
6) I am responsible for Sections 13, 17, 18, 19, and relevant portions of 1, 2, 25, 26, 27 of the Technical Report.
7) I am not independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
8) I have had prior involvement with the property that is the subject of the Technical Report.
9) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the 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 27th day of March, 2025

(Signed and Sealed) ***Yves Breau***

Yves Breau, P.Eng.

29.4 Graham Long

I, Graham Long, P.Geo., as an author of this report entitled "Tasiast Mine, Mauritania - National Instrument 43-101 Technical Report" with an effective date of December 31, 2024, prepared for Kinross Gold Corporation, do hereby certify that:

1) I am Vice President, Exploration with Kinross Gold Corp. of 25 York Street, 17th floor, Toronto, Ontario.
2) I am a graduate of Concordia University, Montreal in 1988 with a B.Sc. Specialization in Geology.
3) I am a Professional Geologist registered with the Ordre des Géologues du Québec (OGQ, [No. 01030]) and the Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (NAPEG, [No. L2076]). I have worked as a geologist for a total of 36 years since my graduation. My relevant experience for the purpose of the Technical Report is:
    - Experience in domestic and international work in exploring orebodies from surface and underground. I have experience in both open pit and underground mining.
4) I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5) I last visited the Tasiast Mine on 17 – 26 November 2024.
6) I am responsible for Sections 7, 8, 9 ,10, and relevant portions of 1, 2, 25, 26, 27 of the Technical Report.
7) I am not independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
8) I have had prior involvement with the property that is the subject of the Technical Report.
9) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the 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 27th day of March, 2025

(Signed and Sealed) ***Graham***

Long Graham Long, P.Geo.

29.5 Jacob Brown

I, Jacob Brown, SME (RM), as an author of this report entitled “Tasiast Mine, Mauritania - National Instrument 43-101 Technical Report” with an effective date of December 31, 2024, prepared for Kinross Gold Corporation, do hereby certify that:

1) I am Director, Resource and Mine Geology with Kinross Gold Corp. of 25 York Street, 17th floor, Toronto, Ontario.

2) I am a graduate of University of Northern Colorado, Colorado in 2012 with a Geology Degree, and an MBA in 2024 from the same university.

3) I am a Registered Member of the Society for Mining , Metallurgy & Exploration (No. 04293143). I have worked as a geologist for a total of 13 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Domestic and international experience across North America, South America and Africa. Experienced in performing and being directly responsible for multidisciplinary teams including open pit mine geology, underground mine geology, resource estimation, exploration, short- and long-range mine planning.

4) I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5) I have not visited the Tasiast Mine.

6) I am responsible for Sections 11, 12, 14, and relevant portions of 1, 2, 25, 26, 27 of the Technical Report.

7) I am not independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.

8) I have had prior involvement with the property that is the subject of the Technical Report.

9) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the 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 27th day of March, 2025

(Signed and Sealed) ***Jacob Brown***

Jacob Brown, SME (RM)

29.6 Kevin van Warmerdam

I, Kevin van Warmerdam, P.Eng, as an author of this report entitled "Tasiast Mine, Mauritania - National Instrument 43-101 Technical Report" with an effective date of December 31, 2024, prepared for Kinross Gold Corporation, do hereby certify that:

1) I am Senior Director, Engineering and Energy with Kinross Gold Corp. of 25 York Street, 17th floor, Toronto, Ontario.
2) I am a graduate of Queen's, Ontario in 2008 with a Degree of Mechanical Engineering. I am a graduate of the Schulich School of Business, Toronto, Ontario in 2016 with an MBA.
3) I am a Professional Engineer in Professional Engineer in the Province of Ontario (No. 100133956). I have worked as an engineer for a total of 16 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - My work experience has included involvement in and leadership of many gold projects ranging from early-stage studies to detailed execution including detailed design, construction, commissioning, and ramp-up.
   - I have developed and owned detailed financial models for gold project valuations as well as led or peer reviewed project economic analysis work by others.
4) I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5) I last visited the Tasiast Mine on 17 – 21 October, 2024.
6) I am responsible for Sections 21, 22, and relevant portions of 1, 2, 25, 26, 27 of the Technical Report.
7) I am not independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
8) I have had prior involvement with the property that is the subject of the Technical Report.
9) I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the 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 27th day of March, 2025

(Signed and Sealed) ***Kevin van Warmerdam***

Kevin van Warmerdam, P.Eng.