Boroo and Ulaanbulag Gold Project

2024 Mineral Resources & Reserve

Technical Report (Amended NI 43-101)

Effective 01st February 2024

On Mineral Resource and Reserve estimation at the Boroo and Ulaanbulag properties in Mongolia for Boroo Gold LLC.

The Boroo Gold Project is located in the Selenge Province, North-Central Mongolia.

Report for

Boroo Gold LLC

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Project

PMN20240101

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1.0 EXECUTIVE SUMMARY

1.1 Introduction

Boroo Gold LLC ("BGC", or the "Company") holds 100% ownership of two active gold surface mining projects, namely Boroo (or Boroo Gold) and Ulaanbulag and both are located in the Selenge province of Mongolia. The Company hired Game Mine LLC, a national mining specialized consultant company, to commission a team of consultants to prepare a National Instrument 43-101 (NI 43-101) Technical Report in accordance with National Instrument 43-101 (NI 43-101) guidelines for the Boroo and Ulaanbulag projects.

On April 11, 2024, Steppe Gold Ltd. entered into a definitive agreement to acquire BGC from Centerra Netherlands BVBA, the direct parent company of BGC. This Technical Report has been prepared to support the disclosure of Steppe Gold Ltd. under applicable Canadian securities laws in connection with the transaction. The subject of this Technical Report is to disclose two project's Mineral Resource and Mineral Reserve estimates with an effective date of February 1, 2024 in accordance with National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) and.

Disclosure of Mineral Resources in this report complies with the reporting requirements of the Canadian Securities Administrators National Instrument 43-101 and have been estimated in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) 'Estimation of Mineral Resource and Mineral Reserve Best Practice Guidelines'. Mineral resources do not constitute mineral reserves and do not demonstrate economic viability. There is no assurance that all or any part of the mineral resource will ultimately be converted into mineral reserves.

1.2 Project Location, Description, Climate

The Boroo Gold Project is located in the Selenge Province, North-Central Mongolia approximately 110 km to the Northwest of the capital city of Ulaanbaatar and about 230 km to the South of the international boundary with Russia, at 48°45' N and 106°10' E (Centerra, TR2009). In the Boroo mine area, BGC has been granted the exclusive rights to all hard-rock minerals and placer deposits within a number of contiguous mining licences, which cover 6593.93 ha of land centered on and surrounding the Boroo gold deposit. The licences are located in about equal measure in the counties (soums) of Bayangol and Mandal, situated in the province (aimag) of Selenge. Both the aimag and the soums play an important role in some of the permitting and environmental management aspects of the project.

1.3 Project History and Ownership

The Boroo Gold LLC (BGC) is the current project operator and is wholly owned by OZD Group (known by Boroo).

On October 11, 2018, BOROO (formerly OZD Asia Pte. Ltd.) purchased Centerra's Mongolian business unit including the Boroo Gold Mine and processing facilities (site of this current Project), the Ulaanbulag Gold Project and the Gatsuurt Gold Project, for net cash proceeds of US$35M. BOROO has purchased all the outstanding shares and debt of Centerra Netherlands BVBA which was the 100% direct shareholder of the Mongolian subsidiaries, BGC and Centerra Gold Mongolia LLC (CGM).

BGC originally acquired the Boroo Gold Project in 1997. In 1998 AGR Limited (AGR), an unlisted public company incorporated in the British Virgin Islands, acquired an initial 85% interest in BGC with the Altai Trading LLC (Altai), a Mongolian private company, holding the remaining 15%. In 2000, Altai sold two thirds of its interest to AGR resulting in AGR holding a 95% interest in BGC and Altai holding the remaining 5% (Centerra, TR2009).

In 2002, Cameco Gold Inc. (Cameco Gold), the predecessor to Centerra Gold Inc. (Centerra), acquired a 56% interest in AGR followed later in 2004 with Cameco Gold acquiring the remaining 44% interest in AGR. At the time of Centerra's initial public offering and listing on the TSX exchange in 2004, Centerra held a 95% interest in BGC with Altai holding the remaining 5%. In 2007, Centerra acquired Altai's 5% interest in BGC, resulting in Centerra owning 100% of BGC (Centerra, TR2009).

1.4 Geology and Mineralization

The Boroo gold deposit is a low silica Au+As sulphide system associated with a zone of quartz-sericite-pyrite (QSP) alteration in the sub horizontal Boroo fault. Boroo is an intrusion-related gold deposit and hosted by a Cambrian-Ordovician sequence of highly deformed shales, siltstones and fine sandstones of the Haraa turbidite sediments, and the Paleozoic granitoids of the Boroo Complex. The bulk mineable gold mineralisation at Boroo is hosted in a strongly quartz-sericite altered and sulphidised nearly flat lying zone controlled by the Boroo fault.

Ulaanbulag deposit depends on the morphology of the mineralized region with shallow dip angle and variable thickness. Mineralization is defined by quartz-cali field cali-sericit-pirite alteration and low silica Au+As sulphide system associated. Ulaanbulag is oregon related gold deposit with its tectonic-macmic environment, its carbon dioxide components, and its geochemical properties.

1.5 Mineral Resource Statement

The Mineral Resources presented herein are reported in accordance with the Canadian Securities Administrators National Instrument 43-101 and have been estimated in conformity with generally accepted Canadian Institute of Mining and Petroleum (CIM) "Estimation of Mineral Resource and Mineral Reserves Best Practices Guidelines". Mineral resources are not mineral reserves and have not demonstrated economic viability. There is no guarantee that all or any part of the mineral resource will be converted into mineral reserves.

1.5.1 Boroo

Game Mine was commissioned by Boroo Gold LLC to generate an updated Mineral Resource Estimate for the Boroo Deposit. The update incorporates 37 additional drillholes (totaling 3,629.8 m) and structural interpretation study completed by Boroo Gold on the Property since the previously announced Mineral Resource Estimate with effective date March 01st, 2023 (GSTATS, March 2023). The

Boroo Mineral Resource Estimate is based on data from 1858 RC and diamond drill holes, totaling 167,748.5 metres of drilling.

Table 1-1. Boroo Mineral Resources as of January 01, 2024

Resource Category	Oxidation State	Tonnage (Kt)	Average grade (g/t)	Metal (oz)
Measured	Oxide	1,100	0.551	19,000
	Transition	3,000	0.586	57,000
	Fresh	22,500	0.59	427,000
	Total	**26,600**	**0.588**	**503,000**
Indicated	Oxide	500	0.518	9,000
	Transition	2,400	0.544	42,000
	Fresh	14,400	0.543	251,000
	Total	**17,300**	**0.542**	**302,000**
Meas + Ind	Oxide	1,600	0.54	28,000
	Transition	5,400	0.567	99,000
	Fresh	37,000	0.571	678,000
	Total	**44,000**	**0.57**	**805,000**
Inferred	Oxide	20	0.609	400
	Transition	160	0.455	2,400
	Fresh	1,120	0.842	30,000
	Total	**1,300**	**0.789**	**33,000**

Notes:

1. *Boroo Mineral Resources are as of January 1, 2024, based on the CIM Definition Standards (2014).*
2. *Mineral Resource were estimates has been compiled under the supervision of QP Tuvshinbayar Batbayar.*
3. *Mineral Resources that are not Mineral Reserves have no demonstrated economic viability.*
4. *Reporting cut-off grade for Boroo Mineral Resources is 0.1 g/t Au (include both heap leach and milling ore).*
5. *The Boroo mineral resources has been depleted for mining up to the mining (without backfilling) as of January 1, 2024.*
6. *The Mineral Resources are stated as in situ dry tonnes. All figures are in metric tonnes.*
7. *Figures have been rounded to the appropriate level of precision for the reporting of Mineral Resources.*
8. *Due to rounding, some columns or rows may not compute exactly as shown.*

1.5.2 Ulaanbulag

Game Mine was commissioned by Boroo Gold LLC to generate an updated Mineral Resource Estimate for the Ulaanbulag Deposit. The update incorporates 72 additional drillholes (totalling 5,920.9 m) and structural interpretation study completed by Boroo Gold on the Property since the previously updated Mineral Resource Estimate with effective date May 01st, 2018 (Centerra, May 2018). The Ulaanbulag Mineral Resource Estimate is based on data from 257 RC and diamond drill holes, totaling 24,557.25 metres of drilling.

Table 1-2. Ulaanbulag Mineral Resources as of January 01, 2024

Zone	Category	Tonnage (Kt)	Average grade (g/t)	Metal (oz)
Zone 1	Measured	-	-	-
	Indicated	2,000	0.613	40,000
	Meas + Ind	2,000	0.613	40,000
	Inferred	30	0.448	400
Zone 2	Measured	4,500	0.616	89,000
	Indicated	5,000	0.438	70,000
	Meas + Ind	9,400	0.522	159,000
	Inferred	900	0.452	14,000
Zone 3	Measured	-	-	-
	Indicated	1,000	0.458	14,000
	Meas + Ind	1,000	0.458	14,000
	Inferred	1,800	0.471	27,000
Zone	Category	Tonnage (Kt)	Average grade (g/t)	Metal (oz)
Zone 4	Measured	-	-	-
	Indicated	-	-	-
	Meas + Ind	-	-	-
	Inferred	1,500	0.228	11,000
Total	**Measured**	**4,500**	**0.616**	**89,000**
	Indicated	**8,000**	**0.485**	**124,000**
	Meas + Ind	**12,400**	**0.532**	**213,000**
	Inferred	**4,300**	**0.379**	**52,000**

Notes:

1. *Ulaanbulag Mineral Resources are as of January 1, 2024, based on the CIM Definition Standards (2014).*
2. *Mineral Resource estimates have been compiled under the supervision of QP Tuvshinbayar Batbayar.*
3. *Mineral Resources that are not Mineral Reserves have no demonstrated economic viability.*
4. *Reporting cut-off grade for Ulaanbulag Mineral Resources is 0.1 g/t Au (include both heap leach and milling ore).*
5. *The Ulaanbulag mineral resources have been depleted for mining up to the mining (without backfilling) as of January 1, 2024.*
6. *The Mineral Resources are stated as in situ dry tonnes. All figures are in metric tonnes.*
7. *Figures have been rounded to the appropriate level of precision for the reporting of Mineral Resources.*
8. *Due to rounding, some columns or rows may not compute exactly as shown.*

1.6 Mineral Reserve Statement

Mineral Reserves for the Boroo deposit are based on the Measured and Indicated Resources and use engineering designs for the pit and associated operating parameters. Reserve calculations are valid at the time of estimation and use cut-off grade assumptions which were made prior to finalization of the economic model. The Mineral Reserve estimates are based on a mine plan and pit design developed using modifying parameters including metal price, metal recovery based on performance of the processing plant, and operating cost estimates.

Mining costs of $1.77/t, milling costs of $14.99/t and general and administrative costs of $2.22/t, Heap leaching costs of 2.39$/t have been used to estimate the reserves along with the gold price stated above

The cut-off grade used to report the reserves has been chosen by Game Mine at greater than 0.10 g/t gold for heap leach ore and greater than 0.43, 0.46 and 0.52 g/t gold for milling depends on mill recovery domain.

Table 1-3: Mineral Reserve Statement for Boroo Deposit, Mongolia: Game Mine LLC., January 01st, 2024

Reserve Category	Quantity (tonnes)	Average Grade (Au g/t)	Contained Metal (oz)
CIP Ore Stockpile			
Proven	768,000	1.25	31,000
Probable	-	-	-
Proven and Probable	768,000	1.25	31,000
CIP Ore			
Proven	7,318,000	1.2	282,000
Probable	3,558,000	1.15	131,000
Proven and Probable	10,876,000	1.18	413,000
Total CIP Ore			
Proven	**8,085,000**	**1.2**	**313,000**
Probable	**3,558,000**	**1.15**	**131,000**
Proven and Probable	**11,644,000**	**1.19**	**444,000**
Heap Leach Ore Stockpile			
Proven	282,000	0.3	3,000
Probable	-	-	-
Proven and Probable	282,000	0.3	3,000
Heap Leach Ore			
Proven	8,176,000	0.3	79,000
Probable	4,246,000	0.3	41,000
Proven and Probable	12,421,100	0.3	120,000
Total Heap Leach Ore			
Proven	**8,457,000**	**0.3**	**82,000**

Probable	4,246,000	0.3	41,000
Proven and Probable	12,703,000	0.3	123,000
Total Reserve			
Proven	16,542,000	0.74	395,000
Probable	7,804,000	0.69	172,000
Proven and Probable	24,346,000	0.72	567,000

Notes:

1. The effective date of the Mineral Reserve estimate is February 01st, 2024. Mineral Reserves were estimates has been compiled under the supervision of QP Tuvshinbayar Batbayar

2. The Mineral Reserve estimates were prepared with reference to the 2014 Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards (2014 CIM Definition Standards) and the 2019 CIM Best Practice Guidelines.

3. Reserves estimated assuming open pit mining methods

4. Reserves are reported on a dry in-situ basis

5. The cut-off grade used to report the reserves has been chosen by Game Mine at greater than 0.1 g/t gold for heap leach ore and greater than 0.43, 0.46 and 0.52 g/t gold for milling depends on mill recovery domain.

6. Reserves are based on a gold price of $1,750/oz, mining cost of $1.77/tonne, milling costs of $14.99/t and general and administrative costs of $2.22/t. Heap leaching costs of 2.39$/t. Heap leaching recovery 40%.

7. In the block model, no additional provisions were introduced to account for external dilution or losses during mining, while these factors always occur to some degree during mining, it would appear from the reliability of the block model relative to the production results obtained to date that the required levels of adjustment for these factors has been adequately accounted for in the initial interpolation and later unsmoothing introduced during block model creation.

8. All figures are rounded to reflect the relative accuracy of the estimate. Numbers may not add exactly due to rounding.

Table 1-4: Mineral Reserve Statement for Ulaanbulag Deposit, Mongolia: Game Mine LLC., January 01st, 2024

Reserve Category	Quantity (tonnes)	Average Grade (Au g/t)	Contained Metal (oz)
CIP Ore Stockpile			
Proven	118,000	0.96	4,000
Probable	-	-	-
Proven and Probable	118,000	0.96	4,000
CIP Ore			
Proven	1,196,000	1.4	54,000
Probable	858,000	1.19	33,000
Proven and Probable	2,054,000	1.31	87,000
Total CIP Ore			
Proven	1,314,000	1.36	57,000
Probable	858,000	1.2	33,000
Proven and Probable	2,172,000	1.3	90,000
Heap Leach Ore Stockpile			
Proven	727,000	0.4	9,000
Probable	-	-	-
Proven and Probable	727,000	0.4	9,000
Heap Leach Ore			
Proven	1,496,000	0.29	14,000
Probable	1,778,000	0.29	16,000
Proven and Probable	3,274,000	0.29	30,000
Total Heap Leach Ore			
Proven	2,223,000	0.33	23,000

Probable	1,778,000	0.28	16,000
Proven and Probable	4,001,000	0.31	40,000
Total Reserve			
Proven	3,537,000	0.71	81,000
Probable	2,636,000	0.58	49,000
Proven and Probable	6,173,000	0.66	130,000

Notes:

1. *The effective date of the Mineral Reserve estimate is February 01st, 2024. Mineral Reserves were estimates has been compiled under the supervision of QP Tuvshinbayar Batbayar.*
2. *The Mineral Reserve estimates were prepared with reference to the 2014 Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards (2014 CIM Definition Standards) and the 2019 CIM Best Practice Guidelines.*
3. *Reserves estimated assuming open pit mining methods*
4. *Reserves are reported on a dry in-situ basis*
5. *The cut-off grade used to report the reserves has been chosen by Game Mine at greater than 0.20 g/t gold for heap leach ore and greater than 0.46, 0.50 and 0.53 g/t gold for milling depends on oxidation.*
6. *Reserves are based on a gold price of $1,750/oz, mining cost of $1.77/tonne, milling costs of $14.99/t and general and administrative costs of $2.22/t. Ore transportation costs of 1.73$/t. Heap leaching costs of 2.39$/t. Heap leaching recovery 40%.*
7. *In the block model, no additional provisions were introduced to account for external dilution or losses during mining, While these factors always occur to some degree during mining, it would appear from the reliability of the block model relative to the production results obtained to date that the required levels of adjustment for these factors has been adequately accounted for in the initial interpolation and later unsmoothing introduced during block model creation.*
8. *All figures are rounded to reflect the relative accuracy of the estimate. Numbers may not add exactly due to rounding.*

1.7 Mining method

The Boroo site is comprised of the open pit mine, mill and processing facility and tailings storage facility. A production schedule based on an annualized 5,000 t/d mill feed rate has been developed for a conventional open pit mine plan for Boroo and Ulaanbulag.

The Boroo and Ulaanbulag mining operations are based on conventional open-pit methods to mine a nominal 50,000 tonnes per day material. Operations are planned to stop during the first half of 2030. Mining is done with bench heights of five metres, with ore mined on half benches for improved grade control in the flat-lying ore.

The principal rock handling equipment is supplied by Caterpillar and includes two 5.4 m3 hydraulic excavators, one 6.4 m3 hydraulic excavator and one 12 m3 hydraulic excavator and ten 50-tonne haul trucks, six 100-tonne haul trucks. Additional haul trucks are to be temporarily added to the fleet in for tailings dam construction. The waste rock mined is deposited on waste dumps immediately adjacent to the individual pits. Grade control in mining is by manual sampling of the blasthole cuttings. Two separate samples are taken for each blasthole, one for the initial 2.5 metres, one for the second 2.5 metres, which allows ore and waste selection to occur for those short vertical intervals.

Table 1-5: Boroo and Ulaanbulag Mine Production Schedule

Year	Mined CIP Ore (tonnes)	Gold Grade (g/t)	Contained Gold (kg)	Mined Heap Leach Ore (tonnes)	Gold Grade (g/t)	Contained Gold (kg)	Total Contained Gold (kg)	Waste (tonnes)	Total Material (tonnes)	Strip Ratio, t/t*
2024	3,329,000	1.06	3,500	3,503,000	0.32	1,100	4,600	11,245,000	18,077,000	1.65
2025	1,397,000	1.60	2,200	1,133,000	0.29	300	2,500	16,247,000	18,777,000	6.42
2026	1,867,000	1.50	2,800	1,209,000	0.28	400	3,200	15,702,000	18,778,000	5.11
2027	1,662,000	1.21	2,000	2,796,000	0.29	800	2,800	13,792,000	18,250,000	3.09
2028	830,000	1.15	1000	1,790,000	0.28	500	1,500	15,680,000	18,300,000	5.99
2029	2,220,000	1.15	2,600	2,690,000	0.30	800	3,400	13,340,000	18,250,000	2.72
2030	1,625,000	0.90	1,500	2,575,000	0.30	800	2,300	7,067,000	11,267,000	1.68
Total	**12,930,000**	**1.20**	**15,600**	**15,696,000**	**0.30**	**4,700**	**20,236**	**93,073,000**	**121,699,000**	**3.25**

* Notes; *heap leach ore included*

***All figures are rounded to reflect the relative accuracy of the estimate. Numbers may not add exactly due to rounding.*

Table 1-6: Mill and Heap Leach Production Schedule

Year	CIP Ore (tonnes)	Gold Grade (g/t)	Contained Gold (kg)	Recovered Gold (oz)	Heap Leach Ore (tonnes)	Gold Grade (g/t)	Contained Gold (kg)	Recovered Gold (oz)	Total Recovered Gold (oz)
2024	1,734,000	1.61	2,800	64,000	967,000	0.40	400	5,000	69,000
2025	1,734,000	1.34	2,300	53,000	2,822,000	0.34	1000	12,000	65,000
2026	1,734,000	1.74	3,000	61,000	1,133,000	0.29	300	4,000	65,000
2027	1,734,000	1.23	2,100	47,000	2,209,000	0.25	600	7,000	54,000
2028	1,738,000	0.89	1,600	36,000	2,801,000	0.30	900	11,000	47,000
2029	1,734,000	1.13	2,000	43,000	2,307,000	0.27	600	8,000	51,000
2030	1,734,000	1.06	1,800	40,000	2,533,000	0.29	700	9,000	49,000
2031	1,674,000	0.61	1,000	23,000	1,932,000	0.32	600	8,000	31,000
Total	**13,816,000**	**1.20**	**16,600**	367,000	**16,704,000**	**0.30**	**5,100**	**64,000**	**431,000**

***All figures are rounded to reflect the relative accuracy of the estimate. Numbers may not add exactly due to rounding.*

1.8 Project Infrastructure

The existing facilities at the Boroo Gold Mine will be used to process ore from this next phase of the Project. The Boroo facilities include:

- Process plant, including facilities for unloading and feeding of ore, as well as grinding and leaching.
- Tailings management and heap leach pads, including impervious multi-layered basal linings of waste dumps

- Recycling treatment circuits to remove arsenic and cyanide from tailings
- Storage facilities for chemicals and reagents
- Dewatering facilities and return water lines
- Elevated tailings dam to accommodate additional tailings
- Warehousing
- Administration offices
- Maintenance Shop

The power demand of the existing Boroo facility is 10 MW. Power is currently supplied to the existing Boroo facility via a 110 kV electric power line that crosses the Sujigtei River Valley.

The existing Boroo Gold plant (currently on Care and Maintenance) employs a Leach/CIP and gravity concentration for gold recovery. The plant comprises crushing, grinding, gravity concentration, thickening, Leach and Adsorption, and cyanide detoxification steps as. Detoxified tailings are deposited into a zero discharge tailings management facility.

An existing potable water treatment plant will treat the fresh water prior to storage in the potable water storage tank. Sewage will be collected and chlorinated before disposal. Effluent from the sewage treatment plant will be discharged into the tailings facility at Boroo.

At the Boroo site, diesel and gasoline storage facilities will be provided at the mine services area. One diesel fuel storage tank and one gasoline storage tank will be installed above-ground. A mine maintenance / operations administration building and a security gatehouse facility will be built at the mine site.

During the operating phase of the Project, Boroo operating personnel will be housed at the existing facilities at the Boroo Mine.

The existing tailings management facility (TMF) of Boroo mine is located approximately 5 km to the east of the mine. The total capacity of the TMF is 21.2 million m3

1.9 Mineral Processing and Metallurgical Testing

The current Boroo Mill flowsheet includes crushing, grinding, gravity concentration, leaching and carbon-in-pulp (CIP), cyanide detoxification and arsenic precipitation to produce gold doré.

1.9.1 Metallurgical testing

The current Boroo ore processing flowsheet is the result of a number of past metallurgical test programs and confirmed by successful operation results. The current Boroo Mill flowsheet includes crushing, grinding, gravity concentration, leaching and carbon-in-pulp (CIP), cyanide detoxification and arsenic precipitation to produce gold doré.

During 2002, bottle roll test result was representing recovery of lower transition and fresh ore in Boroo deposit. Gold recoveries were variable, averaging 75% but ranged from 57% to 90%.

During 2006, column leaching testwork has been conducted for oxide, transitional and fresh ore by AMMTECH Australia and KCA USA based labs. AMMTECH testwork gold recoveries of 91.4% were reported for oxide ore, 70.3% for transition ore and 64.9% for fresh ore. KCA testwork gold recoveries of 90% were reported for oxide ore, 65% for transition ore and 53% for fresh ore.

During 2009-2010, bottle roll tests were undertaken on oxide, transition and fresh ore of the Ulaanbulag deposit. Gold recoveries of 87.6% were reported for oxide and 81.5% for transition ores.

In 2023, Regarding to the Amb Lab fresh ore bottle roll testwork, gold recoveries were variable, averaging 51.2% and ranged 22% to 94% in Boroo deposit. It is worth to mention that Amb Lab laboratory is not accredited by any international standard.

Overall, however, metallurgical test data on the fresh ore at Boroo and Ulaanbulag was limited.

During 2021-2023 mining, Boroo deposit and Ulaanbulag operation gold recoveries averaging 77.8% and ranged 78.1% to 77.7%.

In 2023, pit 5 mining section is mostly focused on fresh ore and gold recoveries averaging 74.5%. In additional, previously classified BIOX process ore during Centerra, gold recovery averaging 58-61% during 2013-2014.

Operation results showed that the gold recovery prediction from testwork to be accurate.

1.9.2 Processing and Recovery Methods

The selected flowsheet for Boroo and Ulaanbulag ore, based on the test results described in Section 13, is a standard layout that consists of crushing, grinding, gravity concentration, cyanide leaching and gold recovery in a carbon-in-pulp (CIP) circuit.

Milling flowsheet includes the following major units:

- Primary crushing
- Grinding and classification including gravity concentration
- Thickening and leach/carbon in pulp (CIP)
- Carbon elution and reactivation
- Gold electrowinning and refining
- Cyanide detoxification; and
- Tailing storage

The Boroo mill was designed with a capacity to process 1.8 million tonnes of ore per year (5000 tpd). Mill commissioning commenced in December 2003 and by March 1, 2004 when commercial production was achieved. In 2021, the mill was restarted, and since then throughput has steadily increased to approximately 1.6-1.7 million tonnes per year or 4500 - 5 000 t/d.

Mill recovery has steadily decreased since the depletion of oxide and transitional ore. When processing sulphide or fresh ore, mill recovery is typically in the range of 60% to 70%. A significant portion of the recovery is still achieved in gravity separation.

1.9.3 Heap leaching

Nominally 3.0 million tonnes of heap leach feed will be crushed and stacked on the leach pad. Ore will be hauled from the pit or stockpiles near the pit to a heap leach stockpile. Crushing and stacking will proceed at a nominal rate of 10 000 t/d. 50% of the total heap leach ore is crushed by primary crushers.

Gold is recovered from the PLS using a CIC plant. PLS will flow through five, 2.4 m diameter

columns loaded with activated carbon. Solution is introduced into the base of each column and will overflow via launder to the next stage. Carbon is periodically transferred upstream by recessed-impeller pumps progressively adsorbing more gold from solution. When carbon in the first column achieves gold loading of 5 000 g/t it will be transferred to the elution circuit in the existing plant for stripping.

The CIC plant will be located between the existing cyanide detoxification and mill buildings. This area will be enclosed and will become an integral part of the process plant making use of existing facilities for reagent mixing and facilitating supervision of operations. PLS will enter the CIC plant at east end of the building and will cascade toward the west. Barren solution overflowing from the last column will pass over a carbon safety screen prior to being pumped to a heated and insulated BLS Tank. Make-up water and cyanide concentration will be managed in the BLS Tank again, taking advantage of the proximity of the CIC plant to existing infrastructure.

Variable frequency drives interlocked with ultrasonic level indicators will ensure that pumps

delivering PLS to the CIC plant and returning BLS to the heap maintain solution flow within an

acceptable range. Overflow conditions will cause the BLS Tank to discharge excess solution to the final tails tank in the detoxification circuit or PLS to the storm water pond. BLS will pass through a diesel fired solution heater which will raise the temperature of the BLS 5^{o} C prior to returning to the heap.

Heap leach ore to be mined from the Boroo and Ulaanbulag pits is expected to be more refractory and has been given a lower overall recovery. Operation results showed that the heap leach gold recovery is 40%, and which is aligned with testwork result.

1.10 Capital and Operating Cost Estimation

1.10.1 Capital Cost Estimates

Since the Boroo Gold mine is operational, no additional investment will be made to increase the capacity, so no investment calculation has been completed. On the other hand, the calculations made by the company for the maintenance of mining equipment, processing plant, mine accommodation and other mine assets are presented in the sustaining capex section.

1.10.2 Sustaining Capital Cost Estimates

Sustaining capital costs include all costs from 2024 of the operation to maintain mining until the end of the planned LOM in 2031. Total sustaining costs of $68.4M have been estimated for the Boroo Gold mine.

Table 1-7: Sustaining Capital Cost Summary

Main Area		Cost (US$'000)
1000	Geology	2,603
2000	Mine	25,124
3000	Process Plant	18,719
4000	Project Services	-
5000	Project Infrastructure	-
6000	Permanent Accommodation	-
7000	Site Establishment & Early Works	-
8000	Management, Engineering, EPCM Services	737
9000	Tailings Storage Facility	21,211
Total		68,394

1.10.3 Operating Cost Estimates

The LOM average operating cost for the Project is estimated at US$30.0/t processed at the nominal process rate of 5000 t/d or 1.7 Mt/year. This operating cost is derived using total LOM operating costs divided by LOM processed tonnages and excludes pre-production mining costs.

Table 1-8: Operating Cost Distribution by Operating Area

Operating Cost Summary	Total Cost M$ (LOM)	Cost $/t Processed (LOM Average)
Mining Cost	150.9	10.9
Processing Cost	190.8	13.8
Site Admin Cos	40.1	2.9
G&A Cost	23.6	1.7
Total Operating Cost	405.4	29.3

Note: Rounding may cause some computational discrepancies

Boroo Gold mine site is connected to the central power supply, and the CIP plant has access to electricity rather than a diesel generator, so diesel fuel consumption is low.

1.11 Economic Analysis

Game Mine prepared an economic evaluation of the Boroo Gold project based on a free cash flow financial model created for this technical report. For the 8-year total life-of-mine (from 2024 to 2031) and 13.8M tonne CIP ore and 16.7M tonne Heap Leach ore Mineral Reserve, the following pre-tax and post-tax financial parameters were estimated using the base case parameters:

- Pre-tax NPV at 5% discount rate - US$191.1M
- Post-tax NPV at a 5% discount rate - US$151.7M

Table 1-9: Project Economics

Parameter	Unit	Value / average
Gold Price	US$/oz	1,750
MNT/US Exchange Rate	MNT:USD	3,450:1
Production Start	Date	Operating mine
Mining cost	US$/t mined	1.24
Off-site transportation cost (Ulaanbulag)	US$/t	1.70
Processing cost -CIP	US$/t milled	10.17
Processing cost – Heap leach	US$/t milled	2.38
Site admin cost	US$/t milled	2.90
G&A cost (UB office)	US$ Million/annual	2.90
Recovery rate – CIP	%	68.5
Recovery rate – Heap leach	%	40.0
Mining royalty for gold	%	5
Corporate income tax	%	10/25

A sensitivity analysis was conducted on the base case pre-tax and post-tax NPV (Net Present Value) of the Project, using the following variables: gold price, operating costs, and total capital cost.

Table 1-10 shows the pre-tax sensitivity analysis results; post-tax sensitivity results are shown in Table 1-11. To decrease sensitivity, the project is less sensitive to changes in the total operating cost compared to sensitivity to changes in the price of gold.

Table 1-10: Sensitivity Analysis (Pre-Tax NPV@5%)

Pre-Tax NPV sensitivity to Opex										
		Gold price (%)								
		-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%
Opex (%)	-30%	**172.1**	**201.7**	**231.3**	**260.9**	**290.5**	**320.1**	**349.7**	**379.3**	**408.9**
	-20%	**139.0**	**168.6**	**198.2**	**227.8**	**257.4**	**287.0**	**316.6**	**346.2**	**375.8**
	-10%	**105.8**	**135.4**	**165.0**	**194.6**	**224.2**	**253.8**	**283.4**	**313.0**	**342.6**
	0%	**72.7**	**102.3**	**131.9**	**161.5**	**191.1**	**220.7**	**250.3**	**279.9**	**309.5**
	10%	**39.5**	**69.1**	**98.7**	**128.3**	**157.9**	**187.5**	**217.1**	**246.7**	**276.3**
	20%	**6.4**	**36.0**	**65.6**	**95.2**	**124.8**	**154.4**	**184.0**	**213.6**	**243.2**
	30%	**(26.7)**	**2.9**	**32.5**	**62.1**	**91.7**	**121.3**	**150.9**	**180.5**	**210.1**

Pre-Tax NPV sensitivity to Capex										
		Gold price (%)								
		-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%
Capex (%)	-30%	**93.5**	**123.1**	**152.7**	**182.3**	**211.9**	**241.5**	**271.1**	**300.7**	**330.3**
	-20%	**86.6**	**116.2**	**145.8**	**175.4**	**205.0**	**234.6**	**264.2**	**293.8**	**323.4**
	-10%	**79.6**	**109.2**	**138.8**	**168.4**	**198.0**	**227.6**	**257.2**	**286.8**	**316.4**
	0%	**72.7**	**102.3**	**131.9**	**161.5**	**191.1**	**220.7**	**250.3**	**279.9**	**309.5**
	10%	**65.7**	**95.3**	**124.9**	**154.5**	**184.1**	**213.7**	**243.3**	**272.9**	**302.5**
	20%	**58.8**	**88.4**	**118.0**	**147.6**	**177.2**	**206.8**	**236.4**	**266.0**	**295.6**
	30%	**51.8**	**81.4**	**111.0**	**140.6**	**170.2**	**199.8**	**229.4**	**259.0**	**288.6**

Table 1-11: Sensitivity Analysis (Post-Tax NPV@5%)

Post-Tax NPV sensitivity to Opex										
		Gold price (%)								
		-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%
Opex (%)	-30%	137.4	159.6	181.8	204.0	226.2	248.4	270.6	292.8	315.0
	-20%	112.5	134.8	157.0	179.2	201.4	223.6	245.8	268.0	290.2
	-10%	86.1	109.5	132.1	154.3	176.5	198.7	220.9	243.1	265.3
	0%	59.0	83.0	106.4	129.4	151.7	173.9	196.1	218.3	240.5
	10%	29.8	55.5	79.9	103.3	126.5	149.0	171.2	193.4	215.6
	20%	0.3	26.1	51.9	76.6	100.2	123.5	146.3	168.5	190.7
	30%	(29.3)	(3.4)	22.5	48.3	73.3	97.1	120.4	143.4	165.9

Post-Tax NPV sensitivity to Capex										
		Gold price (%)								
		-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%
Capex (%)	-30%	79.8	103.9	127.3	150.3	172.5	194.7	216.9	239.1	261.3
	-20%	72.9	96.9	120.3	143.3	165.6	187.8	210.0	232.2	254.4
	-10%	65.9	90.0	113.4	136.4	158.6	180.8	203.0	225.2	247.4
	0%	59.0	83.0	106.4	129.4	151.7	173.9	196.1	218.3	240.5
	10%	52.0	76.1	99.5	122.5	144.7	166.9	189.1	211.3	233.5
	20%	45.1	69.1	92.5	115.5	137.8	160.0	182.2	204.4	226.6
	30%	38.1	62.2	85.6	108.6	130.8	153.0	175.2	197.4	219.6

The NPV was also estimated for varying discount rates, with Game Mine applying 5% as the base case. Table 1-12 shows the post-tax NPV at varying discount rates.

Table 1-12: NPV Estimates at Varying Discount Rates

Discount rate	Pre-tax NPV	Post-tax NPV
0%	US$ 227.2M	US$ 181.3M
5%	US$ 191.1M	US$ 151.7M
8%	US$ 173.7M	US$ 137.5M
10%	US$ 163.6M	US$ 129.2M

Table 1-13: Total depreciation (2024-2031)

Asset Class	US$ Millions	Years remaining for depreciation
Geology	2.6	8
Mine	42.9	8
Process Plant	49.0	8
Management, Engineering, EPCM Services	4.1	8
TFS	21.2	8
Total	119.8	

Game Mine performed a breakeven point analysis where the NPV is zero depending on the price of gold, mining, and processing costs. These evaluations were done independently and did not reflect the impact of the variance of multiple input parameters. Gold price and cost values where NVP is zero:

1. Gold price - US$ 1199/oz.
2. Mining cost - US$ 3.06/t ROM
3. Processing cost (CIP) - US$ 28.2/t milled

2.0 INTRODUCTION

Boroo Gold LLC ("BGC", or the "Company") holds 100% ownership of two active gold surface mining projects, namely Boroo (or Boroo Gold) and Ulaanbulag and both are located in the Selenge province of Mongolia. The Company hired Game Mine LLC, a national mining specialized consultant company, to commission a team of consultants to prepare a National Instrument 43-101 (NI 43-101) Technical Report in accordance with National Instrument 43-101 (NI 43-101) guidelines for the Boroo and Ulaanbulag projects. On April 11, 2024, Steppe Gold Ltd. entered into a definitive agreement to acquire BGC from Centerra Netherlands BVBA, the direct parent company of BGC. This Technical Report has been prepared to support the disclosure of Steppe Gold Ltd. under applicable Canadian securities laws in connection with the transaction. The subject of this Technical Report is to disclose two project's Mineral Resource and Mineral Reserve estimates with an effective date of February 1, 2024 in accordance with National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) and.

2.1 Sources of information

For the preparation of this work, the QPs have relied upon information provided by Boroo Gold LLC and their agents, including drill hole data, maps, laboratory analytical certificates, costs for contractors and fuel, and from other sources such as publicly available databases, research and academic literature, and observations made during site visits.

The primary source documents supporting the Boroo Gold Project technical report were:

- Technical Management Group (TMG) Ltd. Boroo Gold Project PFS Internal Report – June 09, 2023
- Report for Boroo Gold LLC Scoping Study for the Boroo Gold project, June 2020
- Technical Report on the Boroo Gold Mine Mongolia, Centerra Gold INC. – December 17, 2009
- Interpretation and Review of 1997 – 2008 Resistivity / IP and Ground Magnetic Data Boroo and Gatsuurt Exploration Areas for Centerra Gold (Mongolia) By Jovan Silic Ph. D. Flagstaff GeoConsultants (JSA Pty Ltd), December 2008
- Technical Report on the Boroo Gold Mine Mongolia, Centerra Gold INC. – May 13, 2004
- Drilling database – survey data, assay data, lithology data, oxidation, alteration, bulk density data, and structural data.
- Detailed topographic data were provided by Boroo and surveyed by DGPS total station in UTMWGS84, Zone48 in end of 2023.
- Drilling database – supplied in multiple spreadsheets:
- Testwork report by "Amb Lab" LLC, 2023
- Boroo Gold Company Heap Leach Project Feasibility Study Report Volume I Metallurgical Basis and Process Design, Vector Colorado LLC, August 2006
- AMMTEC Metallurgical Test Report, August 2005
- KAPPES-CASSIDAY Metallurgical test report, January 2006

2.2 Qualified Persons

Game Mine LLC was engaged by Boroo to prepare the report and Tuvshinbayar B, an independent consultant of Game Mine is the sole author of the report and Qualified Person.

2.3 Site Visits

QP Tuvshinbayar B visited the property on 15-Jun 2024, which time the Boroo and Ulaanbulag projects were observed operational, the physical site was toured, and meetings with technical site personnel were conducted including exploration, ore control, mining planning and geotechnical team.

3.0 RELIANCE ON OTHER EXPERTS

This technical report has been prepared by Game Mine for BGC, which is an indirect subsidiary of BOROO.

Any analyses conducted on behalf of BOROO have relied, and believe they have a reasonable basis to rely upon the BGC SMEs who have contributed to the legal, political, environmental, cost and tax information stated in this report.

The QP has reviewed the information provided in this Technical Report and, believe it to be reasonable and reliable and does not disclaim any responsibility for this Technical Report.

4.0 PROPERTY DESCRIPTION AND LOCATION

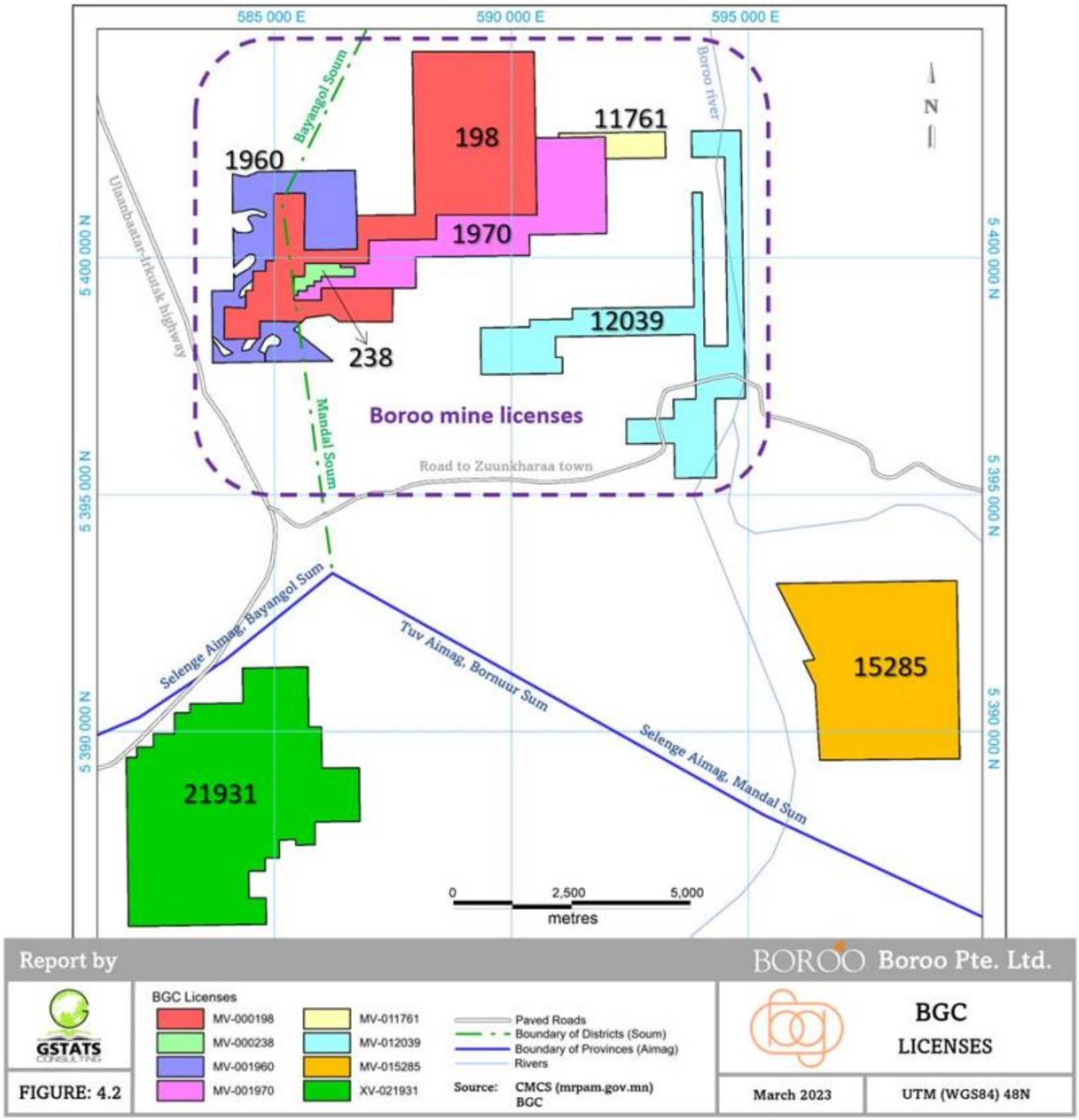
The Boroo Gold Project is located in the Selenge Province, North-Central Mongolia approximately 110 km to the Northwest of the capital city of Ulaanbaatar and about 230 km to the South of the international boundary with Russia, at 48°45' N and 106°10' E (Centerra, TR2009).

Figure 4-1 shows the location of the Boroo Gold Project in the northern part of Mongolia held by the Boroo Gold Company (BGC).

Source: Modified after UN Cartographic Section
(https://www.un.org/Depts/Cartographic/map/profile/mongolia.pdf)
Figure 4-1: Location of the Boroo Gold Project in Mongolia

In the Boroo mine area, BGC has been granted the exclusive rights to all hard-rock minerals and placer deposits within a number of contiguous mining licences, which cover 6593.93 ha of land centered on and surrounding the Boroo gold deposit. The licences are located in about equal measure in the counties (soums) of Bayangol and Mandal, situated in the province (aimag) of Selenge. Both the aimag and the soums play an important role in some of the permitting and environmental management aspects of the project.

The particulars of the individual mining licences are shown in Figure 4-2 and summarised in Table 4-1.



Source: GSTATS, TR2023

Figure 4-2: Boroo Gold Company Mining and Exploration licences

Mining License MV-000198 (original license name was A-32) covering the Boroo gold deposits was originally granted to Altai on July 4, 1996 for an initial period of 15 years. BGC was established in 1997 as a joint venture between Altai (50%) and the London-based Asia Mining Investment Corporation (AMIC, 50%). Mining Licence A-32 was transferred to BGC by the Ministry of Energy, Geology and Mining on June 21, 1997 by Special Ministerial decree (Order Al107). It was re-registered as Licence 198 A on September 20, 1997 as part of the adjustments when a new mining law was promulgated in

Mongolia (Centerra, TR2009). Since 2010, the license registering cadastral system has been using official naming format as listed in Table 4-1.

The remaining mining licences MV-001960, MV-001970, MV-011761, MV-015285 and MV-012039 were granted to BGC directly on the dates indicated in Table 4-1 below.

Table 4-1: Mining and Exploration licences held by BGC, February 2024

Licence no.	Licence name	Area (ha)	Issue date	Annual fees (MNT)	Expiry date
MV-000198	Boroo	1,398.55	June 21, 1997	30.4M	June 21, 2027
MV-000238	Ikh Dashir	40.64	July 26, 1995	0.9M	July 26, 2045
MV-001960	Boroo	530.24	November 29, 1999	11.5M	November 29, 2029
MV-001970	Boroo	642.64	December 6, 1999	13.9M	December 6, 2029
MV-011761	Ikh Dashir	79.43	May 12, 2006	1.7M	May 12, 2036
MV-012039	Ikh Mandal	910.57	September 19, 2006	19.8M	September 20, 2036
MV-015285	Unjin Uul	1204.47	November 20, 2009	26.2M	November 20, 2039
XV-021931	Bor Nuur	1787.39	September 20, 2021	0.7M	September 20, 2024
Total		6,593.93		105.1M	

BGC has advised that the property is subject to a sliding scale royalty payable to the Mongolian Government on gold sales pursuant to the Minerals Law, which starts at 5% and increases to a maximum of 10%, depending on the price per ounce of gold (maximum reached at a gold price of US$1,300 per ounce or above). However, for gold sales to Mongolbank, the central bank of Mongolia, the rate of royalty payable to the Mongolian Government is set at a flat rate of 5%. There are no other royalties, payments or other agreements or encumbrances related to the Boroo mining licences.

Surface rights are negotiated with the Mandal and Bayangol soums, providing sufficient surface area for the mill, heap leach facilities, for tailings and waste rock disposal.

The Boroo mine site includes an open pit mine with waste and ore stockpiles. Higher-grade ore was processed in a mill with a nominal capacity of 6,900 t/d until 2012 then lower-grade transition and fresh material produced until end of 2014. Lower-grade ore was processed through a heap leach facility. There is a camp / residence for employees, a warehouse, maintenance shops and offices.

A permanent tailings management facility (TMF) in the Ikh Dashir valley is connected to the process plant by a 5 km pipeline. This facility received government approval in 2003. The bottom of the TMF was sealed with a compacted clay liner and a high-density polyethylene (HDPE) liner on all embankments. The design of the TMF provides for an ultimate storage capacity of 12.0 M m3 of tailings, sufficient for the tonnage to be mined for the original life of the mine. In 2007, an extension of the original dam was constructed. Lateral dykes were constructed in 2008 for water management purposes.

The TMF has been expanding in 2020-2021 for the processing of Boroo fresh ore, Ulaanbulag ore as part of the BGC projects.

The Government Committee Act on Commissioning of the Boroo facilities into operation gives BGC the right to operate and produce gold. The Stability Agreement originally concluded between BGC and the Mongolian Government represented by the Minister of Finance on July 8, 1998 and amended on May 9, 2000, and on August 3, 2007 stipulates that there will be no nationalization, compulsory acquisition or illegal confiscation of the project by the Government of Mongolia (Centerra, TR2009).

The mining plan was submitted to the Mineral Resources and Petroleum Authority of Mongolia (MPRAM). Mining plans must be submitted in the first quarter of every year for approval of the agency noted above. All permits and licences required for the conduct of mining operations at Boroo are currently in good standing. Some of these permits are with Mongolian state agencies and some are with the other local agencies and authorities.

A general site plan is provided in Figure 4-3. Power for the operation is supplied from the main Mongolian grid via a 110 kVA line that has the capacity to supply 40 MW of power to the site. The national grid is connected to the Russian grid in the north.

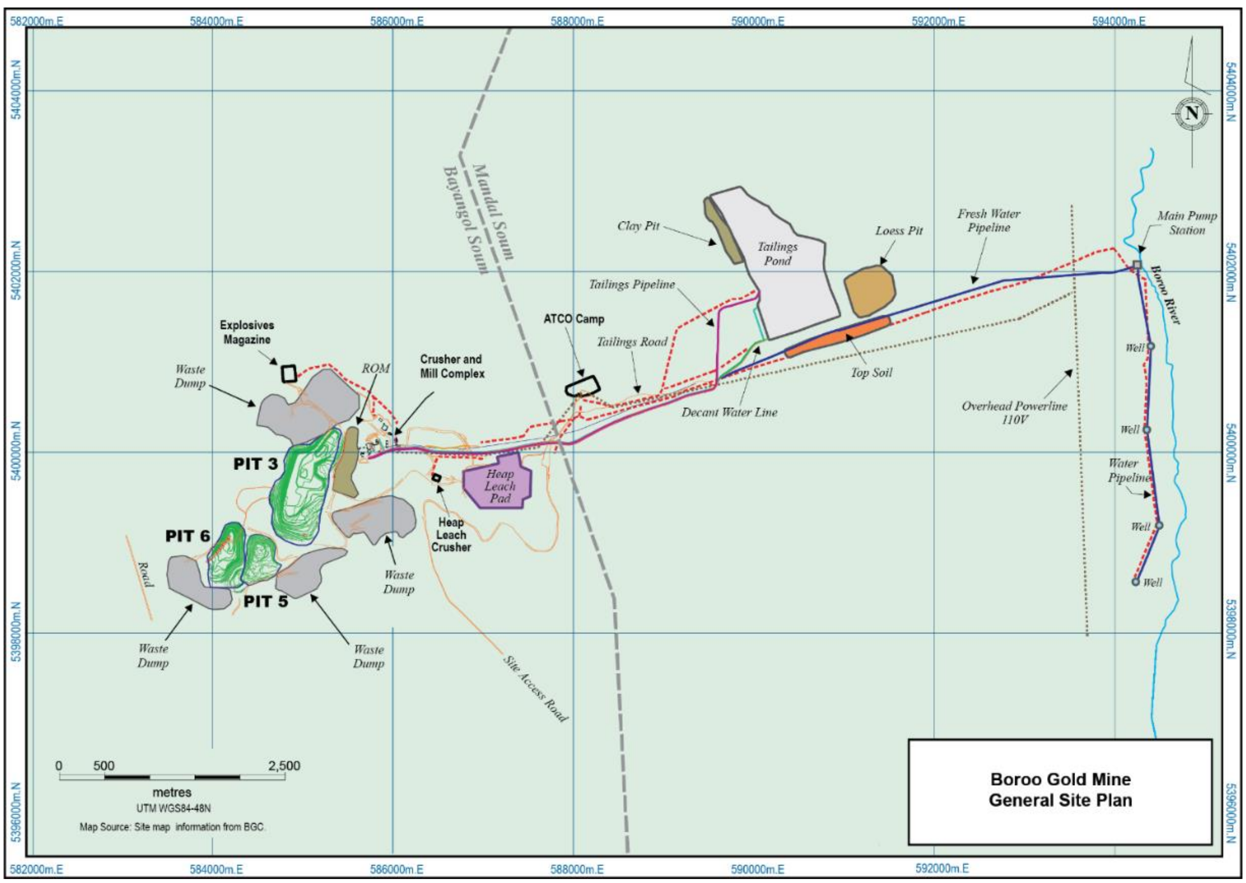


Figure 4-3: Boroo Gold Mine general site plan

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Access

The Boroo mine site is easily reached by travelling northward from Ulaanbaatar on the paved Ulaanbaatar-Irkutsk highway for about 130 km, then on an improved all-weather road east of the highway for about 10 km, the total trip taking just over two hours to complete. The railroad town of Baruunkharaa is located about 20 km north of the junction of the all-weather road with the Ulaanbaatar-Irkutsk highway (Figure 5-1). Ulaanbaatar is served by commercial aircraft connecting to national and international destinations.

Figure 5-1: Access to the Boroo Gold Project

5.2 Climate

Climate in the Project area is mid-continental, temperate to cold. Average annual temperature is approximately 0°C. Temperatures may drop to minus 40°C during the months of December through February. Summer temperatures may exceed 40°C, however, the average daytime summer temperature is approximately 21°C. The area receives about 28 cm of precipitation per year, most of which is rainfall from July through August. Winters are relatively dry, with a moderate amount of snow that may persist on the ground from early November to April due to prevailing low temperatures. Although the climate may present some challenges to heavy equipment operation, the Boroo Gold Mine and Mill have demonstrated the ability to operate effectively year-round. Field exploration is most efficient during the months of March through November, although drilling operations can continue almost year-round.

5.3 Local Resources

The Boroo area is sparsely populated and is inhabited by mainly nomadic herdsmen living in small camps and villages. Labour and other support services are available in Ulaanbaatar, the capital city, with a population of over 2,000,000 inhabitants in 2023, roughly half of the country's entire populace. Ulaanbaatar is also Mongolia's central hub for road, rail, and international air services.

Mandal soum (district) has the largest population of any soum in Mongolia at just under 27,000 people. The soum represents a prime agricultural area in Mongolia. In addition, 70% of the soum is covered by forested land, which supports forestry-based enterprises. In addition to the impact of past mining activities, the forest surrounding the Gatsuurt Valley has been, and continues to be, subjected to timber harvesting.

5.4 Infrastructure

Existing project infrastructure is described in Section 18 of this report.

5.5 Physiography

The geography of Mongolia is characterized by a great diversity of topographic landforms. From north to south, the country is categorized by the presence of four major terrains: mountain forest steppe, mountain steppe, and in the extreme south, semi-desert, and desert. North-central Mongolia includes at least two major landscape types of upland steppe, termed the "Boroogol terrain" and “Dzuun Mod”.

The Boroogol terrain is typified by gentle relief and moderately steep. rolling hills covered by grasslands with small discontinuous forests on north-facing slopes. The average elevation is 1.200 MASL. Streams are moderately or poorly developed and the higher Order streams, large part, are ephemeral. Loess (windblown silt) soil dominates the top of the soil profile.

The Dzuun Mod terrain in the Project area has rolling to steep mountains with moderately forested northern and eastern facing slopes generally devoid of outcrop. Forest cover generally consists of birch, pine, and larch species. The dryer south- and west-facing slopes are generally grass covered and, where very steep, 15% to 20% of the slopes have exposed outcrops. The average elevation is 1,300 MASL. Major river valleys are just under 1,000 MASL and the highest mountains are less than 2,500 MASL. Podzol soils dominate, and streams are moderately to well developed. Solifluction is common and evident as contour lines on grassy slopes in both the Dzuun Mod and Boroogol terrains.

Southeast of the Boroo Deposit is the Gatsuurt River Valley, which is occupied by recent alluvium flanked by older terraces. The alluvium is 10 m to 20 m thick. except where alluvium has been stripped by prior placer mining operations or buried under placer waste and tailings. Both recent alluvium and older terraces provided feed for placer sluices.

6.0 HISTORY

The Boroo deposit was reportedly discovered in 1910 and was exploited by Mongolor on an industrial scale until the 1920s when the facilities were destroyed during a civil war. Activities did not start again until about 1933, when the gold potential of the area was again investigated, followed by the installation of a gold refinery in 1942 that probably treated gold from the mining of a number of individuals, near-surface quartz veins (Cameco Gold Mongolia Inc., 2004b). There are no production records from this time. Events in the ensuing years until about 1965 remain undocumented.

6.1 Prior ownership and ownership changes

BGC is the current project operator and is wholly owned by BOROO. BGC originally acquired the Boroo Gold Project in 1997. In 1998 AGR Limited (AGR), an unlisted public company incorporated in the British Virgin Islands, acquired an initial 85% interest in BGC with the Altai, a Mongolian private company, holding the remaining 15%. In 2000, Altai sold two thirds of its interest to AGR resulting in AGR holding a 95% interest in BGC and Altai holding the remaining 5% (Centerra, TR2009).

In 2002, Cameco Gold Inc. (Cameco Gold), the predecessor to Centerra Gold Inc. (Centerra), acquired a 56% interest in AGR followed later in 2004 with Cameco acquiring the remaining 44% interest in AGR. At the time of Centerra's initial public offering and listing on the TSX exchange in 2004, Centerra held a 95% interest in BGC with Altai holding the remaining 5%. In 2007, Centerra acquired Altai's 5% interest in BGC, resulting in Centerra owning 100% of BGC (Centerra, TR2009).

On October 11, 2018, Boroo Pte. Ltd. (formerly OZD Asia Pte. Ltd.) purchased Centerra's Mongolian business unit including the Boroo Gold Mine and processing facilities, the Ulaanbulag Gold Project and the Gatsuurt Gold Project, for net cash proceeds of US$35 M. BOROO has purchased all the outstanding shares and debt of Centerra Netherlands BVBA which was the 100% direct shareholder of the Mongolian subsidiaries, BGC and Centerra Gold Mongolia LLC (CGM).

6.2 Exploration history

Following Mongolor's initial discovery and production, during the period between World War I and Civil War in Russia (1920s), their mines, equipment, and facilities were robbed or destroyed, and the company was dismantled. After that, until about 1933, there were almost no geological or mining activities in the district.

6.2.1 Soviet period

Exploration for gold in the area was re-started in 1933 (Kolokol'nikov 1933) and continued into 1941 (Alekseichik 1943). A recovery plant which used mercury to extract gold was built in 1942. Gold-bearing ore was probably mined from quartz veins in Boroo, Tsagaan Chuluut and some other small prospects in the district, but the mine had no data on reserves, and relied on empirical knowledge of day-to-day gold recovery in the plant (Cluer et al, 2005).

Prospecting between 1965 and 1969 led to recognition of Boroo's bulk-mineable potential. Detailed evaluation and reserve estimation of the Boroo gold deposit and Ikh Dashir gold placer resources was completed by the Mongolian and East German Joint Geological Expedition (MGJGE) between 1982 and 1990. Only small tonnage test mining was conducted during this period. During this time, the deposit area was drilled using vertical core holes on an approximate 80 m x 80 m grid with in-fill drilling to a density of 40 m x 40 m and even 40 m x 20 m in two areas along the eastern edge of the areas of interest. These diamond core holes of the MGJGE, generally had a comparably large diameter of 76 mm, larger than HQ core size, less frequently 52 mm (between NQ and HQ), 93 mm or 112 mm, both larger than PQ core size.

In addition, as part geological expedition, two bulk samples were obtained from small underground openings for pilot-scale metallurgical testing with initial bench scale testing conducted in Freiberg, East Germany, followed by pilot scale testing by Irgiredmet, Irkutsk, Russia, in 1987. Other studies examined the metallurgical character of both oxidized and fresh ores.

The East German-Mongolian project was terminated in 1991 because of German reunification (Centerra, TR2009).

6.2.2 Transitional Period

From 1991 to 1994 the concessions including the Boroo deposits were controlled by the Boroo Gold Mining Joint Venture consisting of Mining Bureau of the Government of Mongolia (51%), represented by Mongol Erdene, and Morrison-Knudsen Gold Company (MKG) (49%). Morrison Knudsen Exploration (MKE) estimated a resource for the deposit in 1992. MKE drilled twenty-two vertical diamond holes, all in the northern sector of the deposit, mostly to verify existing mineralization and to confirm mineralization continuity but information of these holes is not in the current database (Cluer et al, 2005).

In 1994, MKE engaged the Simons Mining Group to prepare a feasibility study (H. A. Simons, 1994) that investigated a heap leach (oxide only) and a combined heap leach / treatment plant option (oxides plus sulphides). Subsequently, MKE allowed the joint venture to lapse because of unsatisfactory project economics at the time (Centerra, TR2009).

6.2.3 Free Market Period

Mining License 187 granted to Altai, a Mongolian entity, on July 4, 1996 for an initial period of 15 years; and then, according to the Special Ministerial decree, the license was transferred to BGC by the Ministry of Energy, Geology and Mining on June 21, 1997 as license ID of A-32 (later named MV-000198). BGC was established in May 1997 as a joint venture between Altai (50%) and the London-based British Asia Mining Investment Corporation (50%) (Cluer et al, 2005).

In 1997, BGC acquired the Project and completed nine confirmatory diamond drill holes and check assaying of DDR sample splits. The new drill holes were sampled by Australian Consultants Mining and Resource Technology (MRT) personnel in Mongolia, and MRT established an electronic database for the Project (Centerra, TR2009).

In May 1998 Resolute Limited (Resolute), an Australian gold mining and exploration company, was introduced to the Project, and at the end of 1998, a tentative agreement was in place whereby AGR, an affiliate of Resolute, would indirectly acquire 85% of BGC. The remaining 15% was retained by Altai. In August 2000, AGR purchased two thirds of the Altai interest, leaving Altai with a 5% indirect interest in BGC.

As part of the preparation for a bankable feasibility document, AGR undertook a significant in-fill drill program of the deposit area in 1999 to bring the existing "reserves" to comply with the Australian JORC code for reporting of mineral resources and mineral reserves. The definition drilling programs goal was to evaluate mineralization within optimized pit shells developed by Resolute in July 1998. The main zones of mineralization were drilled on a nominal 40 m x 40 m pattern. Ulaanbaatar-based Gobi Drilling, a division of Radial Drilling of Townsville, Australia, was contracted for a combined reverse circulation (RC) and diamond drilling program undertaken in 1999.

Three drilling programs with tightly spaced RC holes were also undertaken by BGC in 1999 to establish the continuity of mineralization and to provide a geostatistical basis for the mineral resource estimation process. These were completed on what was defined as Zone 5 and on the interpreted "high grade lens" in Zone 3 (Centerra, TR2009).

The property essentially sat idle for several years during the recent gold market low cycle. Cameco Gold began negotiations to acquire a controlling interest in AGR and their principal Mongolian asset, BGC and the Boroo deposit, in 2001, culminating in an agreement announced in March 2002. Shortly thereafter, construction of the Boroo mine commenced and commercial production was achieved by BGC in March 2004. During the construction phase approximately 200 additional RC holes were drilled by CGM to confirm reserves and provide additional exploration data in the district. Cameco was renamed Centerra and became a public company in June 2004, and subsequently acquired AGR's remaining interest in BGC raising Centerra's interest in the Boroo mine to 95% (Cluer et al, 2005).

Additional RC infill drilling was undertaken to 2005 by BGC which resulted in a substantial amount of additional data being created the project. Most of this drilling was for reserve / resource delineation and definition purposes the results of which have been included in the current resource and reserve estimation model.

During the period from 2006 to October 31, 2009, the majority of drilling outlined in Table 10-1 was condemnation drilling well outside the limits of the Boroo reserve model or drilling to collect samples for additional metallurgical test-work and geotechnical studies and therefore would have no material impact on the resource model.

Once mining operations were started in 2003, on site drilling and exploration has been carried out by the Boroo mine geological staff with the assistance of CGM exploration staff. All diamond drill core, RC samples, rejects and pulps from drilling since 2004 are in locked storage at the Boroo mine site.

From 2010 to 2021, further exploration, development and infill drillings were completed throughout the mine using diamond drilling method. Drill cores from diamond drilling of all drill holes completed were sampled (GSTATS, TR2023).

6.3 Historical Mineral Resource and Mineral Reserve estimates

There have been numerous estimates of the Mineral Resources and Mineral Reserves of the bulk mineable deposits at Boroo, reflecting different methods and the database plateaus reached at various times. A partial listing is given in **Table 6-1**, and explanatory notes are offered on historical estimates.

Table 6-1: Historical Estimates

Year	Method	Author	[1]Type	Classification	Cut-off grade (g/t)	Tonnes (Mt)	Gold (g/t)	Contained ounces (Moz)
1989	Polygonal	DDR-MPR	Reserve	B & C_3	0.8	13.8	3.1	1.4
1994	BM, PO	H.A. Simons	Reserve (HL)	Unclassified	Unspecified	3.5	2.4	0.3
	BM, PO		Reserve (HL/Plant)	Unclassified	Unspecified	7.8	2.1	0.5
1997	BM	MRT	Resource	Unclassified	0.7	26.9	2.2	1.9
			Resource	Unclassified	1.5	13.0	3.4	1.4
1998	Sectional	Resolute	Resource	Unclassified	Unknown	9.1	2.8	0.8
1999	BM, PO, PD	AGR (Feasibility)	Reserve	Unclassified	1.0	11.0	2.8	1.0

Source: Centerra, TR2009

- *Historical estimates pre-date NI 43-101 and may not satisfy the current CIM reporting and classification standards and should not be relied on.*
- *COG = Cut-off grade; BM = block; PO = pit optimisation; PD = pit design; HL = heap leach.*
- *Category B in the Soviet system is broadly equivalent to the current "proven", C_1 to the current "probable" category. Only about 14% of the 1989 total was classified as category B.*

The first Mineral Reserve estimate for the bulk-mineable mineralisation at Boroo was completed at the end of the DDR-MPR geological expeditions in 1989 using the Soviet Reserve evaluation and classification criteria. The reserves of the Ikh Dashir placer deposit were also estimated in 1989.

MKE had a preliminary Mineral Resource estimation on the deposit prepared in 1992 (not shown in **Table 5-1**). After additional fieldwork, MKE in 1994 engaged Cominco Engineering Services and their successor, H.A. Simons (Simons, 1994) to prepare a feasibility study of the Boroo Project. Because of the lower metallurgical recovery achievable by heap leaching, the Mineral Reserves considered for the two approaches are significantly lower than other estimates using the same database.

In 1997, MRT completed a preliminary MRE using DDR-MPR geological expedition and BGC drilling data compiled by MRT.

In May 1998, Resolute used the MRT database to carry out a sectional interpretation of the deposit (Tchaikov, 1998). The model envisaged six individual zones numbered 1 through 6.

All of the Mineral Resource and Mineral Reserve estimates in Table 6-1 are "historical estimates" for the purpose of NI 43-101 and do not satisfy the current Canadian Institute of Mining, Metallurgy and Petroleum (CIM) reporting and classification standards, and should not be relied on.

6.4 Recent Mineral Resources and Mineral Reserve estimates

6.4.1 Centerra Estimates

Following additional drilling by BGC in 2002 and 2003, the Mineral Resources and Mineral Reserves of the Boroo mine were updated by Geostat Systems International Inc. (Geostat) in July 2003 in conjunction with the Centerra IPO and was in accordance with Canadian reporting standards as required by NI 43-101.

For preparing the Mineral Resource and Mineral Reserve estimate, Geostat used a block model approach in which a 0.8 g/t Au grade shell envelop was employed as a primary guide to define ore shapes. As BGC was in the process of developing the milling facility at the Boroo site, the requirement for the model to estimate Resources and Reserves below 1.0 g/t Au was not necessary as the expected operating cut-off at that time was estimated to be 1.2 g/t Au.

Table 6-2 :Summary of Centerra's Boroo Mineral Reserve and Mineral Resource Estimates

Estimate Date and Sources	Proven and Probable Reserves			Measured and Indicated Resources			Inferred Resources		
	Tons (kt)	Au (g/t)	Metal (koz)	Tons (kt)	Au (g/t)	Metal (koz)	Tons (kt)	Au (g/t)	Metal (koz)
Centerra TR2004, IPO	10,169	3.5	1,158	3,387	2.1	228	869	3.0	83
Centerra AR2004, YE	11,811	3.1	1,172	2,595	2.3	194	3,215	1.9	193
Centerra AR2005, YE	13,390	2.9	1,218	2,652	2.4	201	2,563	2.0	167
Centerra AR2006, YE	24,523	1.6	1,173	6,199	1.4	284	7,772	1.0	240
Centerra AR2007, YE	24,089	1.4	1,048	5,468	1.5	254	7,723	1.0	239
Centerra AR2008, YE	18,455	1.3	778	4,916	1.5	242	7,323	1.0	233
Centerra TR2009	16,349	1.2	615	4,916	1.5	242	7,323	1.0	233
Centerra AR2009, YE	15,451	1.1	567	4,916	1.5	242	7,323	1.0	233
Centerra AR2010, YE	12,660	1.0	392	4,916	1.5	242	7,323	1.0	233
Centerra AR2011, YE	9,658	1.0	298	4,916	1.5	242	7,323	1.0	233
Centerra AR2012, YE	7,196	0.8	178	4,916	1.5	242	7,323	1.0	233
Centerra AR2013, YE	2,364	0.6	49	4,916	1.5	242	7,323	1.0	233
Centerra AR2014, YE				4,916	1.5	242	7,323	1.0	233
Centerra AIF2015, YE				4,916	1.5	242	7,323	1.0	233
Centerra AIF2016, YE				4,916	1.5	242	7,323	1.0	233
Centerra AIF2017, YE				4,916	1.5	242	7,323	1.0	233

In late 2005 following another significant program of exploration drilling, the Boroo Mineral Resource and Mineral Reserve estimate was again updated utilising the block model developed by Reserva International (Reserva) with the assistance of BGC staff. The model is the estimate on which Centerra's 2009 Mineral Resources and Mineral Reserves and life-of-mine plan was based on, as well as what the current Scoping Study is based upon.

For the 2006 year-end disclosure, Centerra included both Mineral Reserves and Mineral Resources at a new lower cut-off of 0.2 g/t Au to include feed for a low-grade heap leaching operation. As a result of including significant amounts of new low-grade material into the estimate of Mineral Reserves and Mineral Resources, at 2006 year-end, there was an increase in the total tonnes in all categories and a corresponding reduction in the average grade.

Since 2006, there has been little additional exploration at the Boroo Project due to the limited number of possible areas to define new mineralisation and annual mining and milling production at Boroo has resulted in a steady depletion of the project's Mineral Reserves.

Detailed descriptions of the above are included in Centerra's IPO prospectus, technical reports and annual information forms corresponding to each of the above dates and accessible at www.sedar.com.

6.4.2 Local Estimates

Since 2018, the owner of the Boroo deposit was changed to a non-public company, Mineral Resource and Mineral Reserve estimate updates are only relied on the reports submitted to the MRPAM.

The first MRE for the Boroo deposit was completed by the DDR-MPR Expedition based on the drilling program between 1982 to 1989 using the Soviet Reserve evaluation and classification criteria. The official registration and approval by PCMR of Mongolia was completed in 1992.

The second MRE technical report was prepared by the BGC mining geology team in 2007. Due to PCMR's panel meeting overloads in that period, MRE report was approved by PCMR in the August 2008. This report was prepared based on the block models by 2005 Resolute MRE. Although, Mineral Resources and Mineral Reserves were re-classified to comply with the Mongolian classification systems and reporting regulations and guidelines.

In 2013, BGC estimated remaining Mineral Resources and Mineral Reserves of Boroo deposit based on the updated database including 2007 to 2012 drilling campaign.

BGC conducted additional drilling program on the ore zones of 2 and 5 in 2019 and 2020. Mineral Resource and Mineral Reserve estimates were completed in 2021.

In March 2023, report included Mineral Resource and Mineral Reserve estimate of the ore zone 3 based on the infill drill holes conducted until end of 2021 (Table 6-3).

Table 6-3: Summary of Mineral Resources and Mineral Reserves Registered at PCMR of Mongolia

Report year	Company	Report type	Classification category	Cut-off grade, (g/t)	Tonnages, Mt	Average grade, g/t	Contained Metals, koz
1992	DDR-MPR Expedition	Reserve	B&C1	0.8	13.9	3.1	138
		Resource	C1&C2	0.8	6.2	1.81	36
2008	BGC	Reserve	A&B	0.2	5.3	2.64	448
			C	0.2	19.2	1.28	786
2013	BGC	Resource	B	0.2	14.4	0.63	29
			C	0.2	0.2	1.39	1
2021	BGC	Resource	B	0.2	1.2	0.68	2
			C	0.2	2.12	0.79	5
2023	BGC	Resource	B	0.2	5.78	0.89	17
			C	0.2	2.2	0.67	5

6.5 Production History

The Boroo mill started processing ore in 2003 and reached commercial production on March 1, 2004. As of December 31, 2023; a total of 27.3 Mt of ore from Boroo has been milled with an average gold grade of 2.40 g/t, with 1.71 million ounces (Moz) of gold produced. The heap leach has stacked 19.4 Mt at 0.73 g/t and recovered 0.17 Moz of gold. The Boroo production history is summarized in Table 6-4.

The Ulaanbulag started processing ore in 2021. As of December 31, 2023; a total of 2.9 Mt of ore from Ulaanbulag has been milled with an average gold grade of 1.23 g/t, with 0.097 million ounces (Moz) of gold produced. The heap leach has stacked 0.376 Mt at 0.37 g/t and recovered 0.002 Moz of gold. The Ulaanbulag production history is summarized in Table 6-5.

Table 6-4: Summary of Boroo Production History to December 31, 2023

Year	Mine				Strip Ratio	Mill				Heap Leach		
	Waste (kt)	Low Grade HL Ore (kt)	Mill Ore (kt)	Mill Ore Grade (g/t)		Ore milled (kt)	Head Grade (g/t)	Gold Recovery (%)	Gold produced (oz)	Ore Stacked (kt)	Head Grade (g/t)	Gold produced (oz)
2003	3,561	32	146	3.91	20.0	113	2.94	97.0	4,326			
2004	11,057	53	1,851	4.04	5.8	1,849	4.52	93.7	217,998			
2005	15,974	369	2,516	4.07	5.5	2,232	4.23	91.5	285,788			
2006	15,890	677	2,481	4.25	5.0	2,387	4.25	87.0	282,802			
2007	15,113	3,631	2,405	3.68	1.6	2,549	3.62	85.3	254,548			
2008	15,405	3,629	2,416	2.70	2.5	2,496	2.69	77.7	167,463	3,684	0.74	25,174
2009	6,002	3,481	2,913	2.40	0.9	2,077	2.56	72.9	124,821	2,177	0.76	25,729
2010	7,265	1,694	2,399	1.70	1.7	2,466	1.38	71.8	106,197	572	0.74	4,942
2011						2,340	1.11	68.9	57,778			1,446
2012	5,288	143	907	2.10	4.4	2,382	1.32	64.0	64,352	456	0.70	7,486
2013						2,394	1.12	57.6	50,020	2,644	0.70	40,298
2014						2,083	0.66	61.2	26,685		0.52	26,443
2015									3,595		0.36	12,631
2016												5,094
2017												
2018												
2019												5,659
2020	597		102	1.36	5.8	102	1.36	90.2	4,011	9,805		9,029
2021	3,583		960	1.16	3.7	960	1.16	80.3	28,718			5,947
2022	2,645		169	1.3	15.7	137	1.49	80.1	5,250	32	0.5	262
2023	5,991		759	1.59	7.9	720	1.66	74.5	28,686	39	0.28	177
Total	**108,371**	**13,709**	**20,024**	**2.99**	**3.66**	**27,288**	**2.4**	**81.5**	**1,713,038**	**19,409**	**0.73**	**170,316**

Source: BGC

Table 6-5: Summary of Ulaanbulag Production History to December 31, 2023

Year	Mine			Strip Ratio	Mill				Heap Leach		
	Waste (kt)	Mill Ore (kt)	Mill Ore Grade (g/t)		Ore milled (kt)	Head Grade (g/t)	Gold Recovery (%)	Gold produced (oz)	Ore Stacked (kt)	Head Grade (g/t)	Gold produced (oz)
2021	6,407	1,365	0.92	4.7	571	1.04	85.0	16,190			
2022	5,535	1,826	0.98	3.0	1,636	1.01	85.0	44,933			
2023	5,144	1,314	1.44	3.9	955	1.48	81.0	36,491	376	0.37	1,961
Total	17,085	4,505			3,162	1.23	83.5	97,614	376	0.37	1,961

7.0 GEOLOGICAL SETTINGS AND MINERALIZATION

7.1 Plate-tectonic settings

Mongolia occupies a central part of the Asian continent, and an interior portion of the Eurasian Plate. Major tectonic events took place during the Palaeozoic and early Mesozoic when exotic terranes and micro-plates were repeatedly accreted to the ancient core of the Siberian plate. The age of the terranes in Mongolia thus decreases southward, with the northern Tuva Terrane consisting mostly of Proterozoic and Lower Palaeozoic rocks while the Southern Terrane contains an important component of Permian to Jurassic intrusives. A major Caledonian event cratonised northern Mongolia, while a subsequent Hercynian event affected central and southern Mongolia. Post-Permian intrusions were of anorogenic alkalic affinity, and Mesozoic volcanics were extruded in response to extensional relaxation (adapted from IFC, 2002).

The structural setting of north-central Mongolia is dominated by several north-easterly striking strike-slip faults of regional extent that are considered terrane-bounding in nature and may have tens of kilometres of cumulative sinistral displacement (Figure 7-1). The Boroo gold deposits are interpreted to be located near a second order; north-westerly striking sympathetic structure locally termed the "Highway Fault".

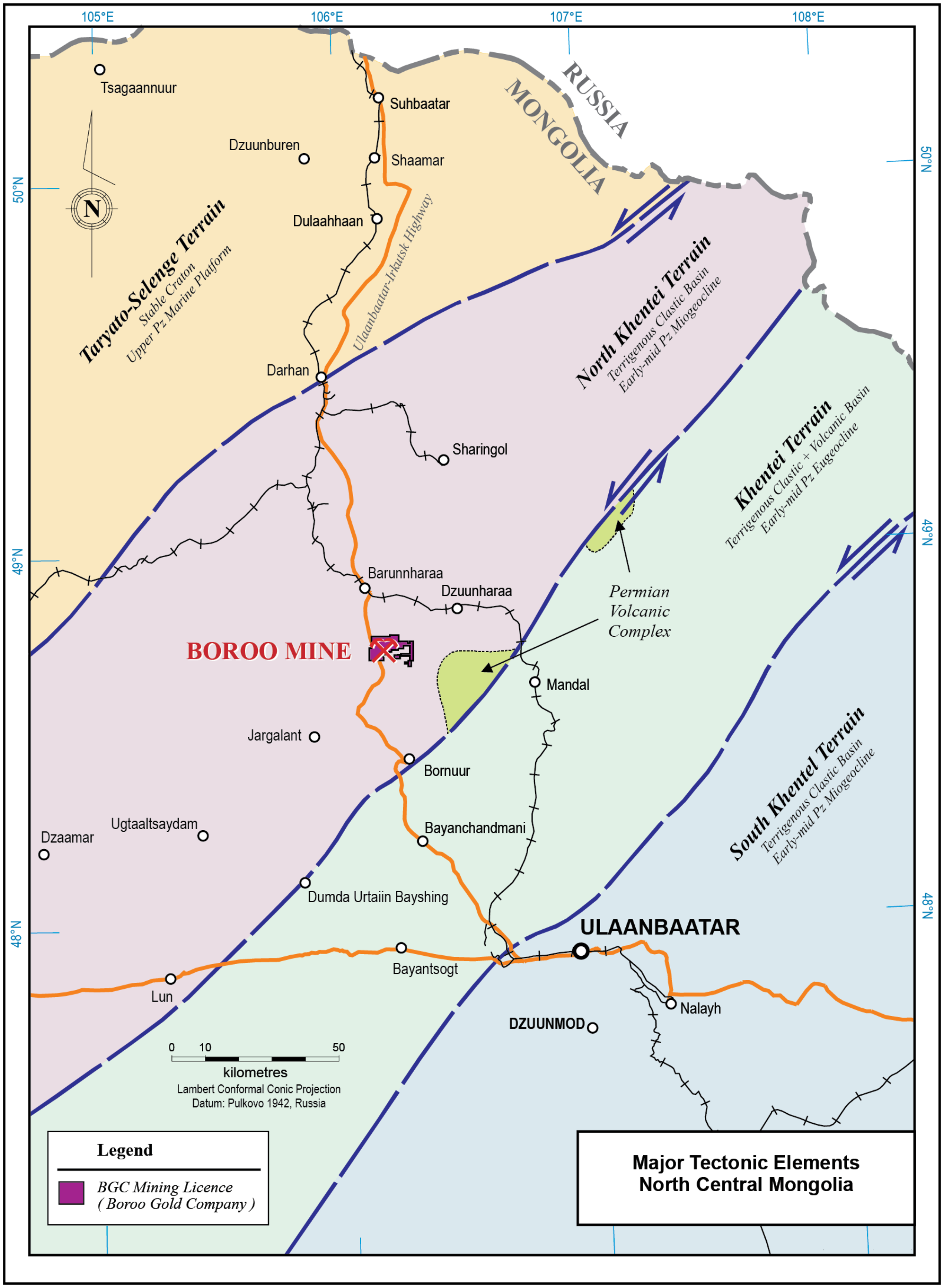


Figure 7-1: Major tectonic elements of North Central Mongolia

7.2 Geology of Boroo deposit

The geology of the Boroo area (Figure 7-2, Figure 7-3) is dominated by the folded Haraa sediments, (PZhr in Figure 7-2), a fairly monotonous sequence of flysch sediments consisting of siltstone, sandstone and greywacke. These rocks are of regional extent and are interpreted to be of Lower Palaeozoic age. Intrusive rocks of the Boroo Complex, of early Palaeozoic age (~520 to 450 Ma), have intruded the sediments. In the area, the Boroo complex is represented by leucocratic granite and granodiorite (PZgr), underlying the eastern part of Figure 7-2.

Detailed drilling around the Boroo gold deposits shows that the contact between the intrusive and the sedimentary rocks is highly irregular, with sedimentary xenoliths floating in the intrusive rocks in the border zone. A significantly younger igneous event of probably late Palaeozoic age is restricted to narrow vertical and shallow dipping dykes and fissures of granitic to dioritic composition.

The fault pattern, with the exception of the gold-bearing structures, is poorly known, but two crossing, high-angle, faults are indicated in Figure 7-2, one of them striking 70°, the other 340°, parallel to the Highway Fault mentioned above. The trace of the 340° fault, in its northern part, is directly underneath the Ikh Dashir Placer. A parallel fault is indicated on the satellite picture some 1.7 km to the east.

Much of the general area around the mine is covered by overburden that can reach several tens of metres in thickness and that consists of colluvium and loess, and minor alluvium deposited in head water drainages. The alluvial deposits can contain significant gold placer deposits. In addition, the colluvium deriving from Zone 3 also contains placer resources.

Oxidation has affected the rocks in the area to a depth of 40 m to 60 m. Oxidation is accompanied by kaolinization of the feldspar crystals in the granitic rocks, but, not having taken place under tropical conditions, has not progressed to the formation of a saprolite profile, with the rocks retaining most of their original strength even near surface.

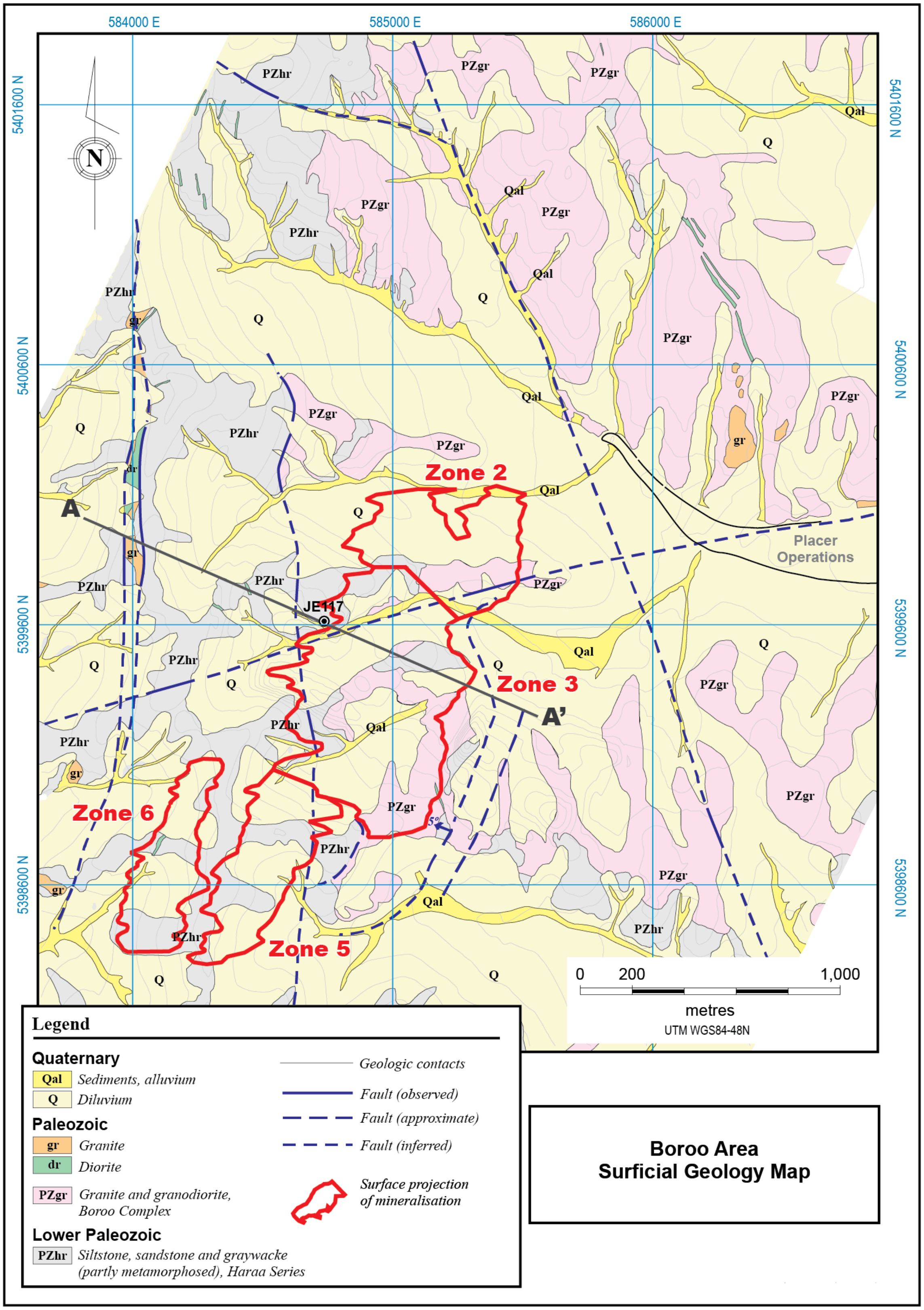


Figure 7-2: Boroo area geology map

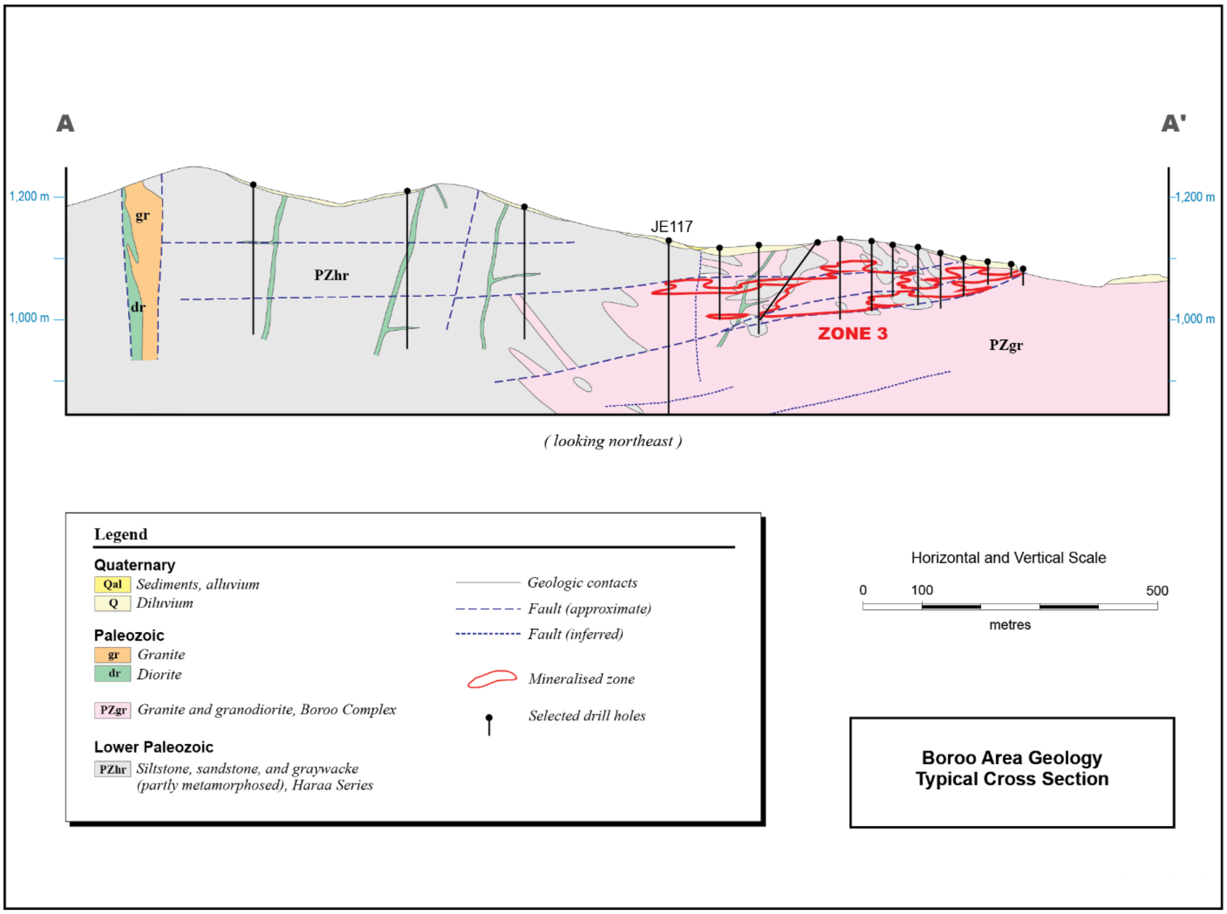


Figure 7-3: Boroo area geology typical cross section

7.2.1 Mineralization

The bulk mineable gold mineralisation at Boroo is hosted in a strongly quartz-sericite altered and sulphidised nearly flat lying zone controlled by the Boroo fault. The fault has been traced for a distance of 2.4 km and is thought to be a thrust fault that dips at an angle of 10° to the northwest and trending northeast (see Figure 7-2). It cuts across the intrusive contact between sediments and granitic rocks in the north but is entirely contained within the sediments in the south. In the cross section, the Boroo fault shows a slightly undulated shape with the structure becoming thicker to the northwest, where the alteration and mineralisation decrease. The Boroo fault is variously altered and mineralised, and where these features are strongest, individual deposits are formed. These are termed, from north to south, Zone 2, 3, 4, 5 and 6. All of the deposits are elongated in a north-easterly direction, with a length to width ratio of about two to one. In Zone 2, 4, 5 and 6, mineralisation is controlled by Boroo fault and is in the footwall. But in Zone 3, there is low grade mineralisation in both hanging wall and footwall. Grade thickness contours show the same overall elongation (Figure 7-4) probably caused more by the width than by the gold grade, with the multiple superimposed zones of alteration and mineralisation responsible for the thicker parts. The thickness of the individual deposits thus varies from a few metres at the deposit edges to several tens of metres.

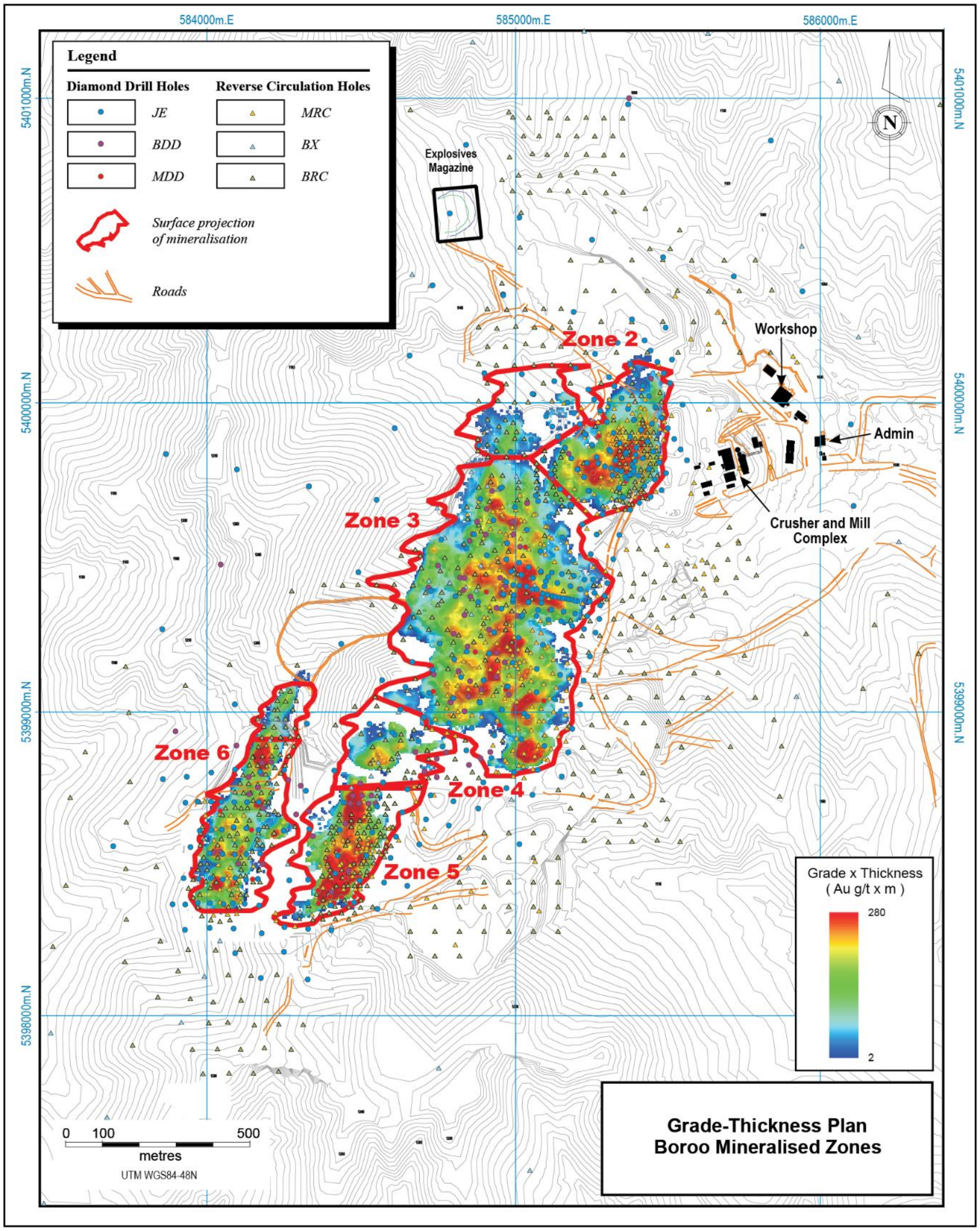


Figure 7-4: Grade-thickness plan of the Boroo mineralised zones

Two main types of mineralisation have been noted:

- Gold-sulphide zones host the largest proportion of gold mineralisation at Boroo. This type of mineralisation is strongly altered quartz-sericite sulphidised zones that occur in thin, irregular veinlets, less often in breccia zones, and disseminated within the pervasive alteration. The intensity of sulphide mineralisation depends on primary host rock and intensity of alteration being stronger in the granites than metasediments. The main sulphide minerals are pyrite, arsenopyrite and rarely chalcopyrite, tetrahedrite and galena occur. It appears that the gold in this mineralisation is relatively fine grained.

- Gold-quartz vein type. The second major gold bearing facies is massive, white quartz veins in which gold is commonly coarse-grained. The thickness of quartz veins varies from a few centimetres up to 3 m and appear as infill veins and veinlets in fractures within mostly metasediments. Veins contain small amount of sulphides and mostly coarse-grained visible gold. This type of mineralisation from a volume perspective is subordinate; however, can carry very high gold values of up to several hundred grams per tonne.

The two main types of mineralisation described above have different gold grade distribution patterns. Gold content is high in quartz. Gold values are also higher where there is quartz stockwork mineralisation associated with pyrite-arsenopyrite ore. Silver values are generally low and are not obviously correlated with gold. Silver values can be higher in the quartz veins in Zone 5 and Zone 6. Silver values, higher than 10 g/t, occur mostly in quartz veins in metasediments and are very variable. The sulphide content in both types of mineralisation is relatively low, typically a few percent. Arsenic is highly anomalous (up to 21,500 g/t) but highly variable in the different zones; 103–112 g/t in Zone 2, 3,158–3,843 g/t in Zone 3 and more than 1% in the metasediments of Zone 5. A positive correlation with gold is restricted to gold values up to about 2 g/t.

It has long been recognised that the degree of oxidation is an important economic parameter at Boroo, as the gold in the fresh ore has a refractory component that limits the metallurgical recovery. Three facies of oxidation have been defined. All sulphides are completely or predominantly oxidised in the oxide zone, and additionally, the feldspars in the granitic rocks have been partly or completely altered to kaolin. In the transition zone, kaolinization of the feldspars is partial and the original sulphides survive in the core of oxidised grains. In the fresh zone, there is no discernible oxidation in the sulphide minerals.

7.3 Geology of Ulaanbulag deposit

In the area of Ulaanbulag deposit, Lower Paleozoic metamorphic-sedimentary rocks of the Shirguu Formation in the Kharaa region, Cenozoic sediments, and Middle-Late Ordovician intrusive rocks of the Boroogol Formation are found in this region.

The Ulaanbulag deposit is complex in terms of geological formation, and the sands of the Shirguu Formation scattered in the central part of the field formed an orebody extending latitudinally in the medium grained, inlaid, feldspar granite of the III phase of the Boroogol Formation in the upper part of the Ulaanbulag fault.

However, in the southern part of the deposit, a large massive of diorite of the Boroogol Formation phase I is found.

In the northern part of the field, several small orebodies of diorite were identified within the granite, occasionally around the boundary between granite and sandstone.

At several locations, the boundary between sandstone and granite is demarcated by the Ulaanbulag inclined fault and its branch faults.

In the geological cross-section, the granite containing orebody in the upper part of the section cuts the diorite found in the lower part, and along their boundary and weakened zone, the Ulaanbulag fault is mapped with a dip of 20-400 degrees southwest.

7.3.1 Structure

The main structure of mineralization in Ulaanbulag is the Ulaanbulag fault, which has a low angle inclined fault and is 700-800 meters long in the NE direction according to the current level of exploration. This structure can be considered as equivalent to the Boroo River fault, which is the ore-controlling structure of the Boroo gold deposit.

The Ulaanbulag fault was discovered by a result of trench excavation and drilling.

The fault was interpretated in most of the drill holes, and it was found at a depth of 25.0 meters in the UB-01 hole and 108.1 meters below the surface in the UB-49 hole, which are about 550 meters apart from one another along the latitude.

Ulaanbulag the fault was discovered at a depth of 144.3 meters in the UB-53, which is drilled 600 meters away from UB-01 in the south.

Thin layered, high-grade mineralization (2 m, 19.2 g/t) was identified in channel UB–TR–22, which could be the northern continuation of the Ulaanbulag horizontal fault.

In the trench and drill core, the fault contains 0.3-5 m thick silicified and brecciated fracture zone with quartz veins, 10-30 cm thick yellow and red clay, and a 5-10 m wide schist zone.

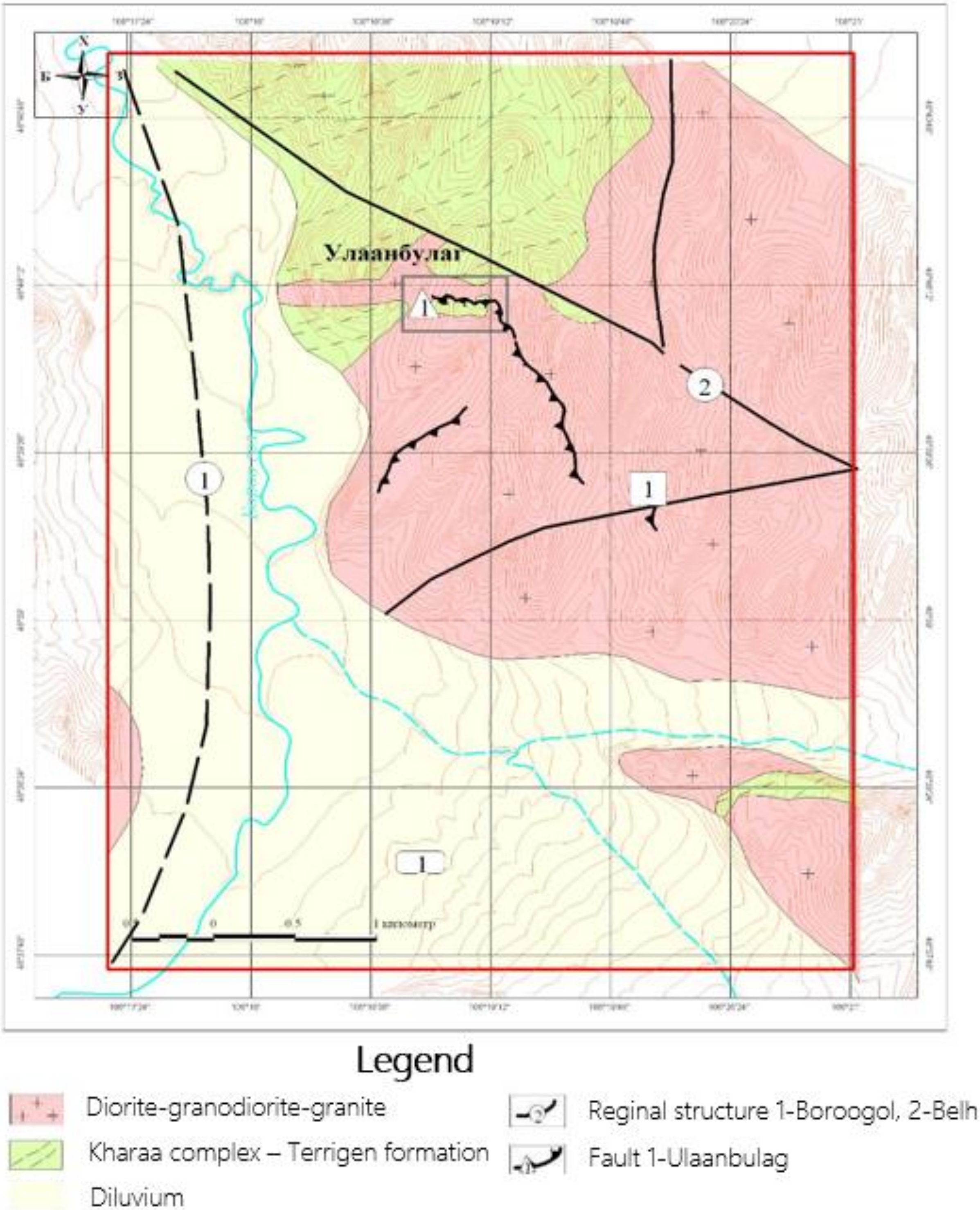


Figure 7-5: Ulaanbulag area geology map

The fact that the fault formed an angle of 70-90 degrees to the axis of the vertical drill core, indicates that the fault has a low angle slope.

7.3.2 Mineralization

There are 2 main types of mineralization in the Ulaanbulag deposit. These include:

- Gold-sulphide
- Gold-quartz vein type

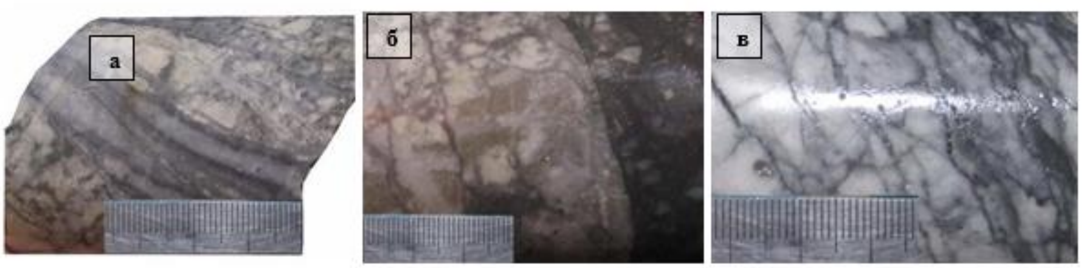
Two main types of mineralisation have been noted:

- A type of gold-sulfide. Gold-sulphide mineralization, which the majority of the deposit's gold reserves, is identified by beresite metamorphic zones. The gold grade is generally low, it has different distribution in different types of rocks, it is highly variable in small distances, and it has high variability of content. The gold content is directly proportional to the arsenopyrite or arsenic content, regardless of the density of quartz veins and veinlets, as in Boroo deposits. This type of mineralization is mainly contained in the Ulaanbulag fault and metamorphic granite rocks. In metamorphic sandstone and diorite xenoliths, the sulfide content decreases dramatically. This is why these less metamorphic rocks usually have lower gold content. The main sulfides in Ulaanbulag deposit are pyrite and arsenopyrite, and rare minerals such as chalcopyrite, galena, pyrrhotite, sphalerite, and dim ore are found.
- Gold-quartz vein type. Usually, the Ulaanbulag fault and its branches are found along the cracks and fissures, and there are fill veins, veinlets, and breccia-like veins with a thickness of several centimeters to 5 meters. The veins are low sulphide and sometimes contain visible gold inclusions up to 2 mm.

7.3.3 Alteration

Alteration In addition to boundary alteration, hydrothermal alteration zones related to mineralization are found in the deposit area. At the inner boundary of the granite, colored minerals are increased and grain size is observed, while at the outer boundary there is hornification and weak silicification, the containing sedimentary rock has become gneiss in the part cut by the granite apophysis. Along the Ulaanbulag fault and its branch faults and cracks, calcined, beresited and propylitic metamorphism zones are distinguished in the host rocks forming a belt formation. The intensity of transformation varies depending on the type of rock. Granite is highly metamorphic, while metasediments and diorite are poorly metamorphosed.

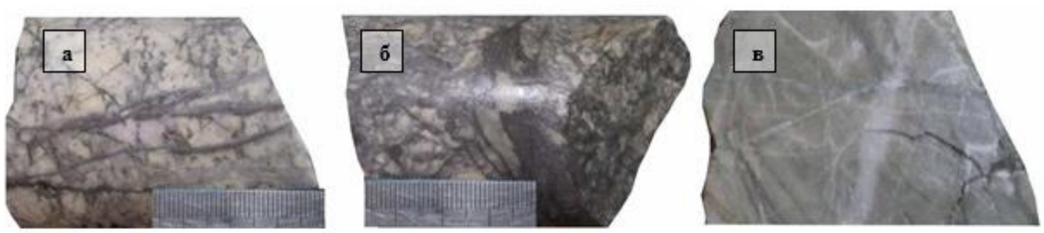
- **Strongly brecciated and silicified metamorphic zone with quartz veins, veinlets, stockwork and diffuse mineralization.**
  Along the Ulaanbulag fault, it occurs as a strongly silicified metamorphism zone with micelle-shaped quartz veins and schistose stockwork, usually 0.1-0.2 meters at the surface and increasing to 5 meters in thickness. In addition to pyrite and iron oxides, the quartz veins contain occasional visible gold. According to the results of the spot samples, the gold content in the quartz veins /0.01-3.0 g/t or more/ varies greatly, which indicates that the distribution of gold in the quartz is not uniform.



a-quartz veined granite, b-brecciated silicified rock and black quartz, c-brecciated quartz veins

Figure 7-6: Silicified Metamorphic alteration

- **The pyrite-sericite-silica metamorphism or beresite metamorphism alteration**
  Ulaanbulag fault and its branch faults and fissures, usually in granitic rocks, and the thickness ranges from 0.2 to 40 meters. The surfaces follow the Ulaanbulag fault for more than 500 meters and form a wider metamorphism zone in the granite, and along its branch faults and fissures, small mesial-like bodies (exposed surfaces) are established in the western and eastern parts of the deposit.



a, b-granite with sericite-silicified transformation, c-quartz-sericitized diorite

Figure 7-7: Berezite Metamorphic alteration

- **Propylitic Alteration**
  This carbonate-chlorite alteration contains poor mineralization at the margins of the beresited alteration and is widespread in all rock types. The facies only form a metamorphic zone 200 m long and 40-80 m wide in the western part of the deposit, mostly set in metasedimentary sediments and, to a lesser extent, granite. In this zone, the gold grade is very low and, in this sense, indicates the peripheral part of the mineralization.

8.0 DEPOSIT TYPE

The Boroo gold deposit is a low silica Au+As sulphide system associated with a zone of quartz-sericite-pyrite (QSP) alteration in the sub horizontal Boroo fault. Boroo is an intrusion-related gold deposit and hosted by a Cambrian-Ordovician sequence of highly deformed shales, siltstones and fine sandstones of the Haraa turbidite sediments, and the Paleozoic granitoids of the Boroo Complex.

Ulaanbulag deposit depends on the morphology of the mineralized region with shallow dip angle and variable thickness. Mineralization is defined by quartz-cali field cali-sericit-pirite alteration and low silica Au+As sulphide system associated. Ulaanbulag is oregon related gold deposit with its tectonic-macmic environment, its carbon dioxide components, and its geochemical properties.

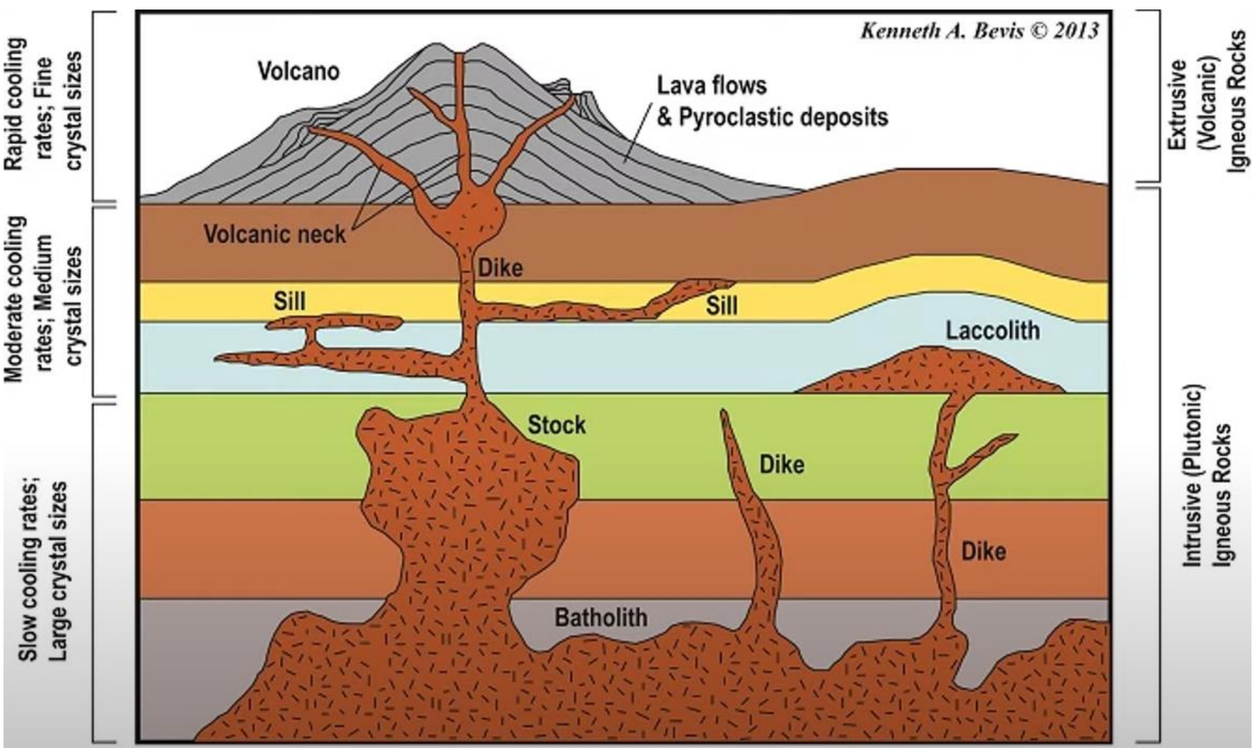


Figure 8-1: General scheme of intrusion related deposits

8.1 Structural relationship

The shape of the final orebody is largely dictated by the structural regime present in the vicinity of the geothermal system. The structures present provide a pathway and traps for the fluids, mineralization and gangue minerals.

In the case of the Boroo deposit, the mineralization and fluids are hosted in the sub horizontal Boroo fault.

In the case of the Ulaanbulag deposit, the mineralization and fluids are hosted in en-echelon style structures.

9.0 EXPLORATION

9.1 Boroo

To date, the level of exploration drilling in and around the Boroo open pits has been significant and has now tested all of the known targets and mineralised extensions that have the possibility of developing further Mineral Resources. For this reason, BGC has no plans to complete any additional exploration in close proximity to the Boroo open pit operation.

There is still however an opportunity for possible Mineral Resource additions from additional zones of mineralisation in the area immediately surrounding the larger Boroo mining licences, and ore from identified resources in the region under BGC control and within economic transport distance to the Boroo processing facilities.

Regional exploration pertinent to Boroo has been undertaken since 1999 in two different corporate settings and on two sets of mining and exploration licences. The northern Mongolia exploration licences held by BGC form the Boroo Project.

Cameco Gold has also been involved in exploration in the region since 1997. The original land position was centred on the Gatsuurt deposit and was assembled by Cascadia Chemicals and Minerals Corporation, which later became Cascadia Minerals Inc. (Cascadia) during 1996 and 1997, with Cameco Gold assuming a role in the project in 1997. Over time, Cameco Gold consolidated its interest in various corporate entities to create Centerra Gold Mongolia (CGM) that held its interests in mining and exploration licences in the region.

The CGM land position continually changed with large mature licences being reduced in size and new licenses being acquired. At 31 October 2009, CGM held seven mining licences totaling 3,565 ha and 29 exploration licences totaling 106,007 ha. A portion of the land position covered non-contiguous blocks over a 250 km strike length of the Yeroogol regional fault system. Most of the remaining exploration licences were along the trend of the Boroo deposit.

At April 30th, 2020, the CGM held no exploration licenses as previously held exploration licenses were returned. Although, there are no exploration licenses held presently, the CGM has extensive experience and resources necessary for identifying and exploring new deposits in the region.

9.1.1 Geophysical Survey

Both the ground resistivity/IP and magnetic methods can be applied in the detection of and the mapping of the Boroo gold deposits. The resistivity/IP technique can be used to directly detected sulphide mineralization associated with the emplacements of gold mineralization, whereas the magnetic data could be utilised to map the transformation of any original magnetic minerals to a non-magnetic state though the quartz-seritic alteration that commonly overprint and enriches the initial pyrite arsenopyrite mineralization.

9.1.1.1 Resistivity IP/Data over the Boroo Deposit

Much of the anomalous chargeability over the Boroo gold deposit is associated with the alteration halo extending to the north west of the Boroo fault. This IP anomaly is of a very moderate amplitude in comparison to the background within the survey area and there are number of gradient array IP anomalies within the survey grid of similar amplitude.

The lack of clear IP anomaly over the mineralized or gold bearing is zone within the shallow dipping Boroo trust zone is due to weathering or the oxidation within or close to the fault zone. This weathering process and/or fracturing along or close to Boroo fault has nevertheless resulted in the lowering of the resistivity within or close to gold mineralised zone.

Figure 9-1 which shows the gradient array chargeability data over an extensive area surrounding the Boroo deposit outlines on a number of “anomalous” chargeability zones over a background with values of about 5-6 mv/v. The amplitudes of the chargeability over most of the “anomalous” zones are only some 50 – 100% over the background values. By estimating the regional trend of the chargeability data as shown in Figure 9-2 and then displaying the difference between the regional trend values and the actual data it is apparent that the Boroo deposits are spatially correlated with although not entirely coincident with one of the “anomalous” zones (Figures 9-3).

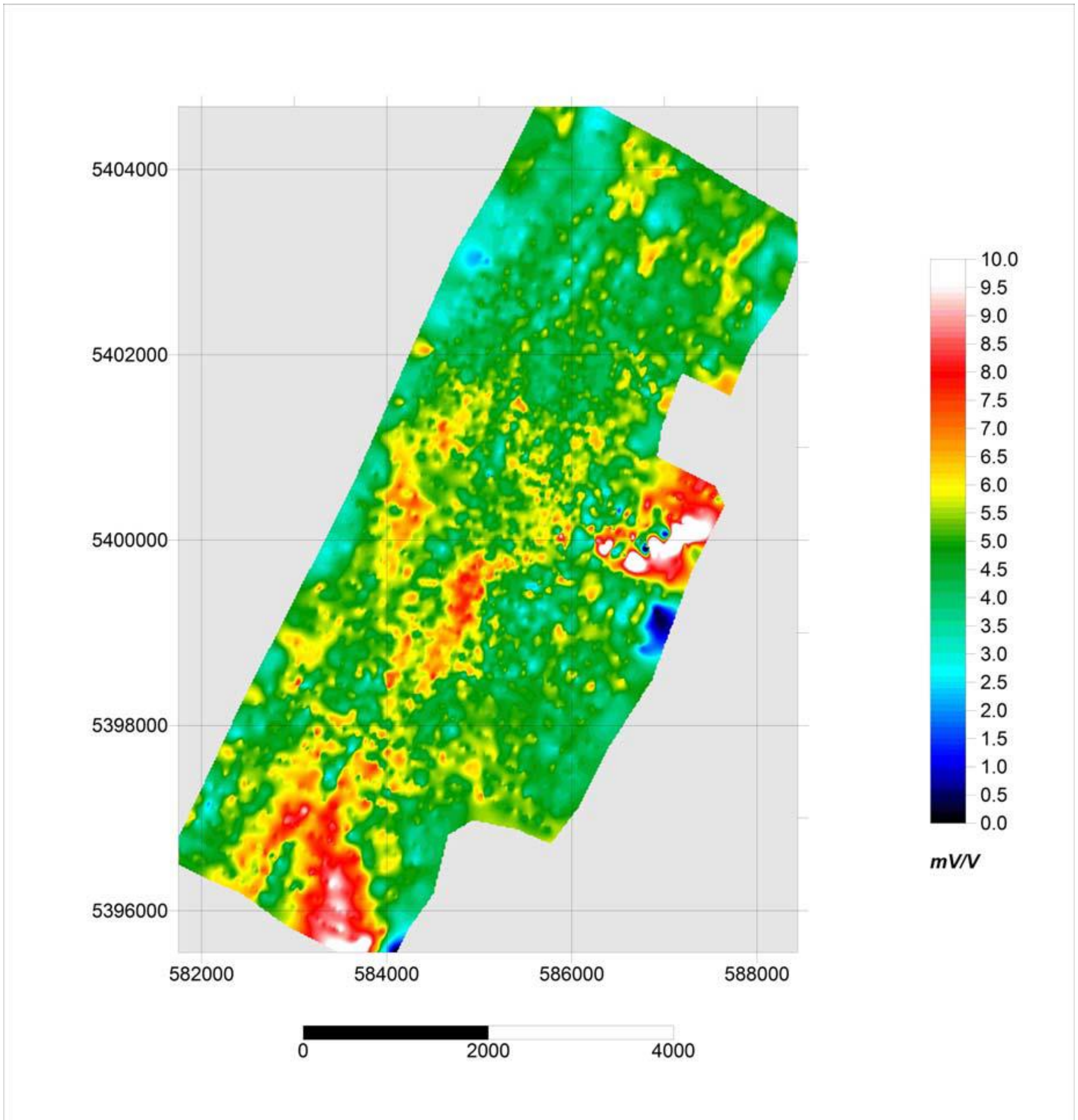


Figure 9-1: Gradient array chargeability data over the area enclosing the Boroo gold deposits.

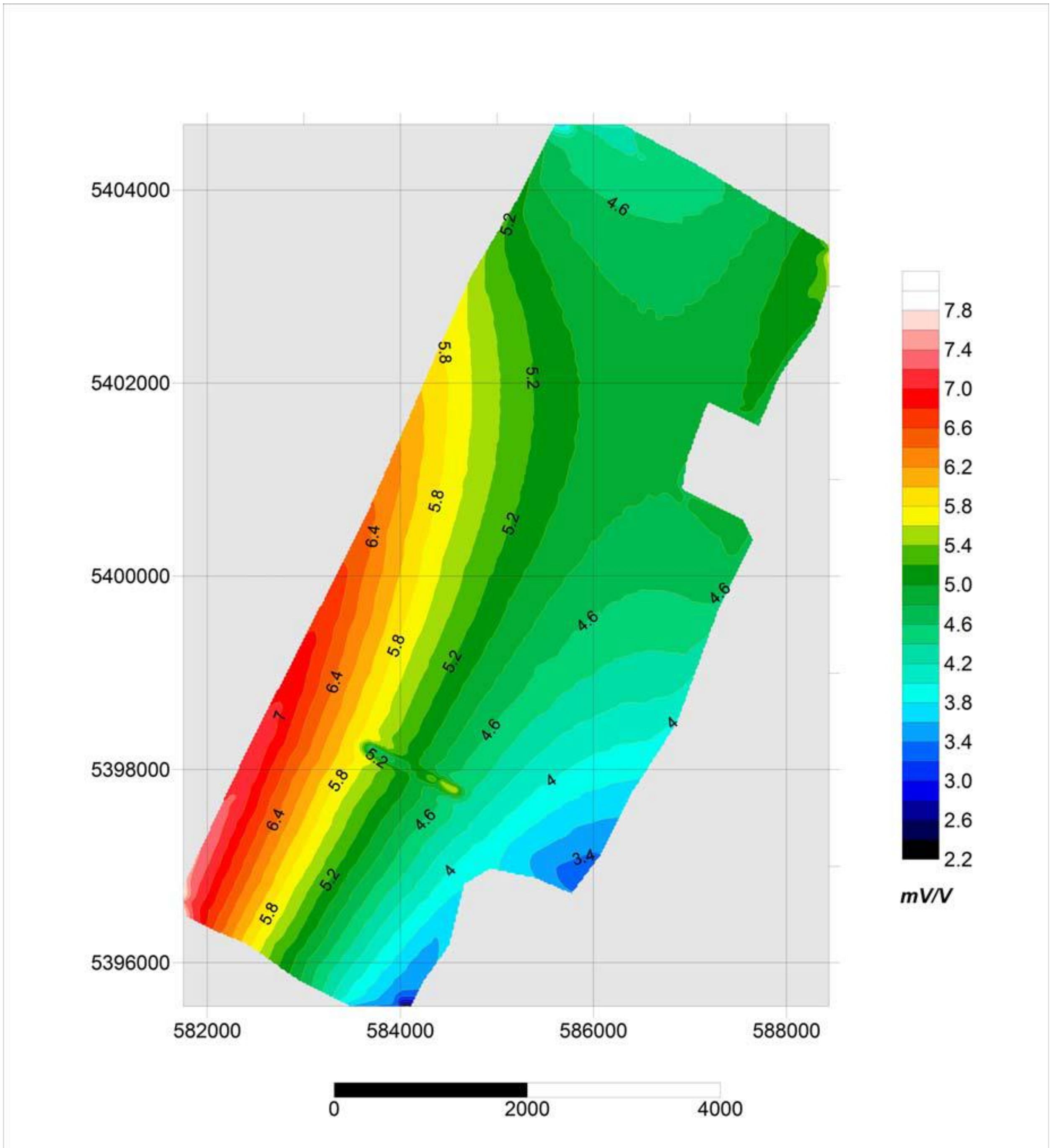


Figure 9-2: Regional trend prediction of gradient array chargeability data over the area enclosing the Boroo gold deposits

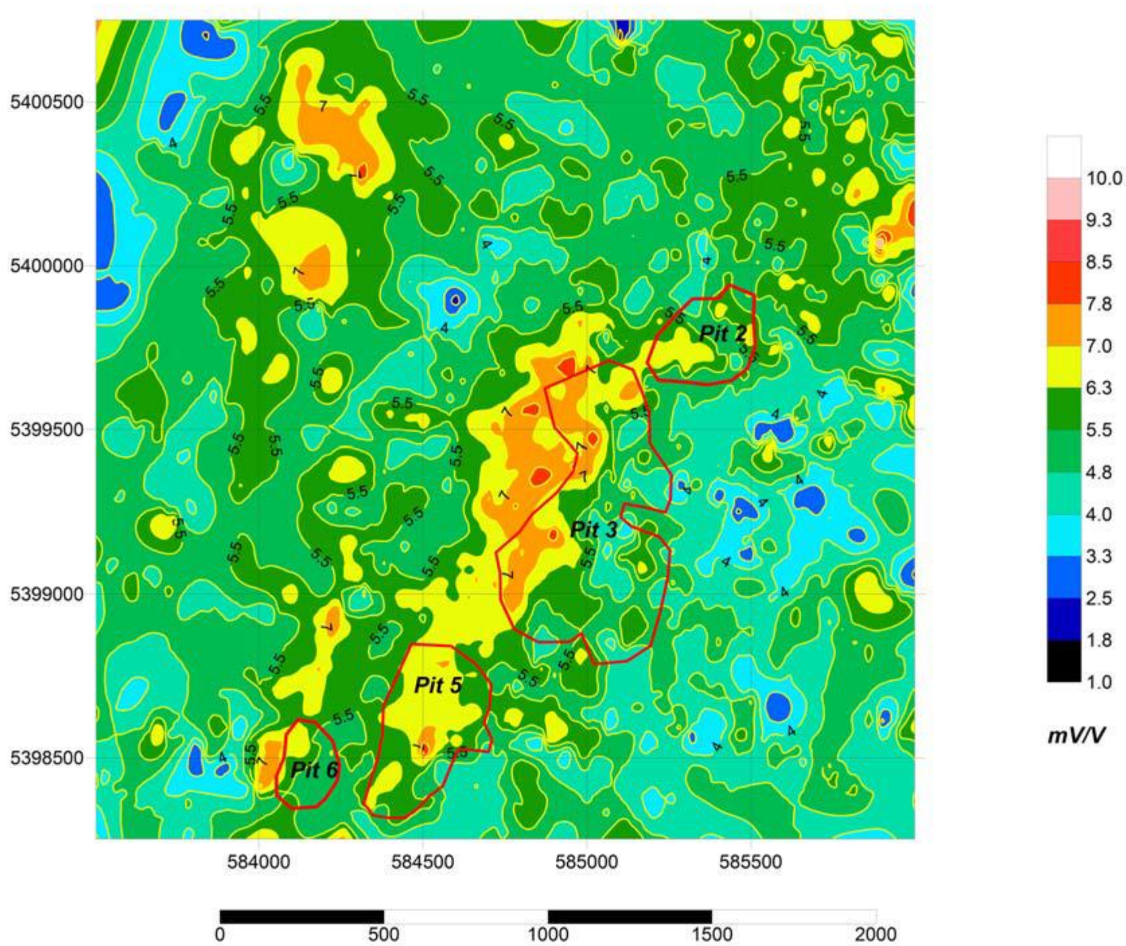


Figure 9-3: Difference between the IP gradient array regional trend and the actual data over the area enclosing the Boroo gold deposit.

Figure 9-3, it is evident that although one of the anomalous gradient array chargeability zones is spatially correlated with the extent and location of the Boroo gold deposits, it is not coincident with the Boroo gold mineralization. In fact most of the Pit 3 mineralization is outside the chargeability anomalous zone. Resistivity data shown in Figure 9-4 suggests that a localized resistivity low may be associated with a significant part of the deposit.

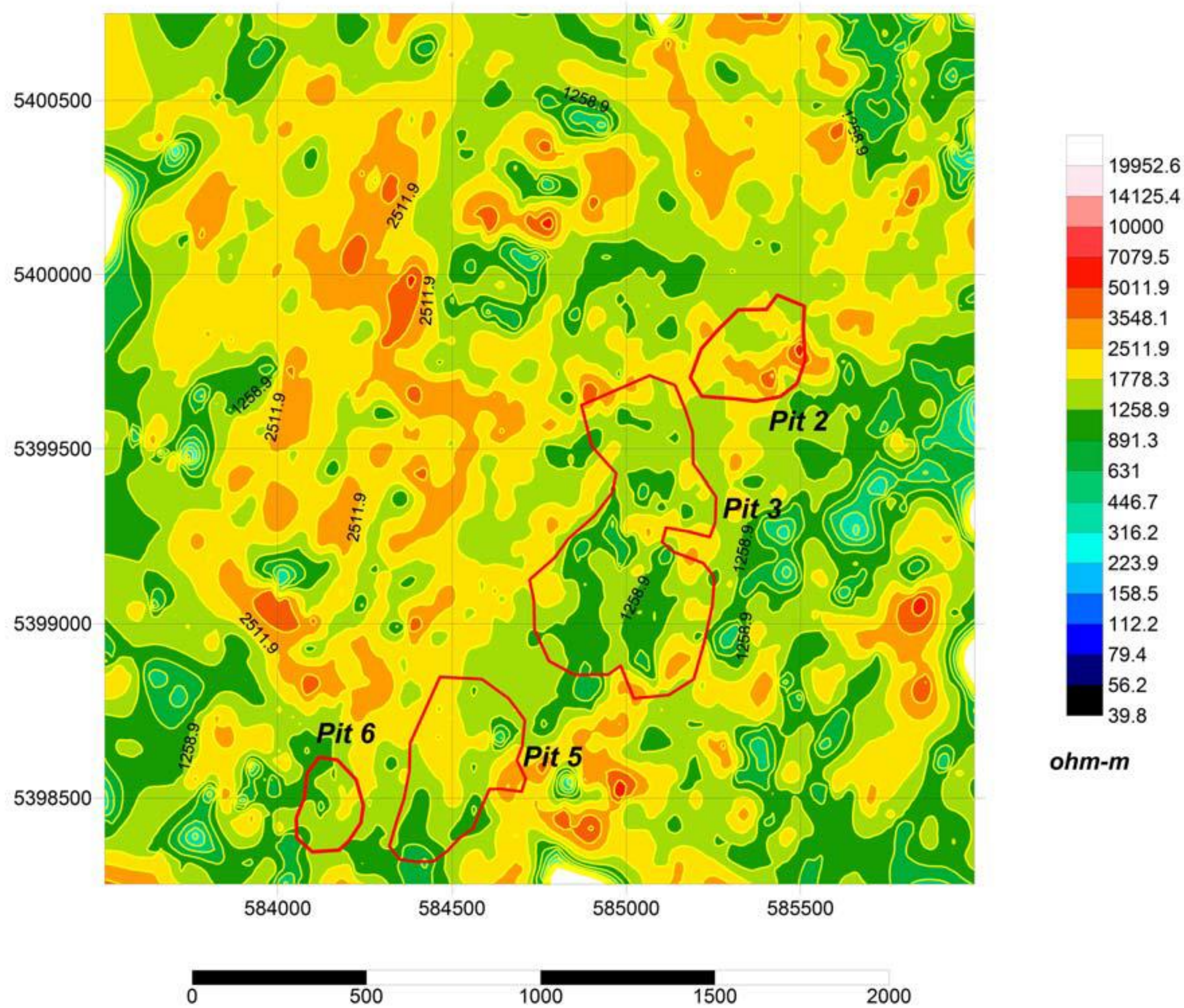


Figure 9-4: Gradient array resistivity over the Boroo gold deposit

9.1.1.2 Ground magnetic data with Boroo survey area

The ground magnetic data over the survey area enclosing the Boroo gold deposit as Figure 9-5 shows outlines a number of magnetic dikes and what appear to be very shallow magnetic sources in much of the eastern section of the survey area. The Boroo gold deposit appear to be at a "diffuse" of boundary of the magnetic data presumably reflecting the contact between the sediments to the west and the granitic rocks to the east of the deposit.

On analysing the ground magnetic data for identification of magnetically quite zones as described in the section on the application of ground geophysics in the exploration for Boroo deposits, results in no clear identification of a near surface magnetically quite zone which could be attributed to significant magnetite destruction by the alteration associated with the gold emplacement (Figure 9-6 and Figure 9-7). This is presumably reflecting the size of the alteration system, and the amount of magnetite (lack of) within the host rocks.

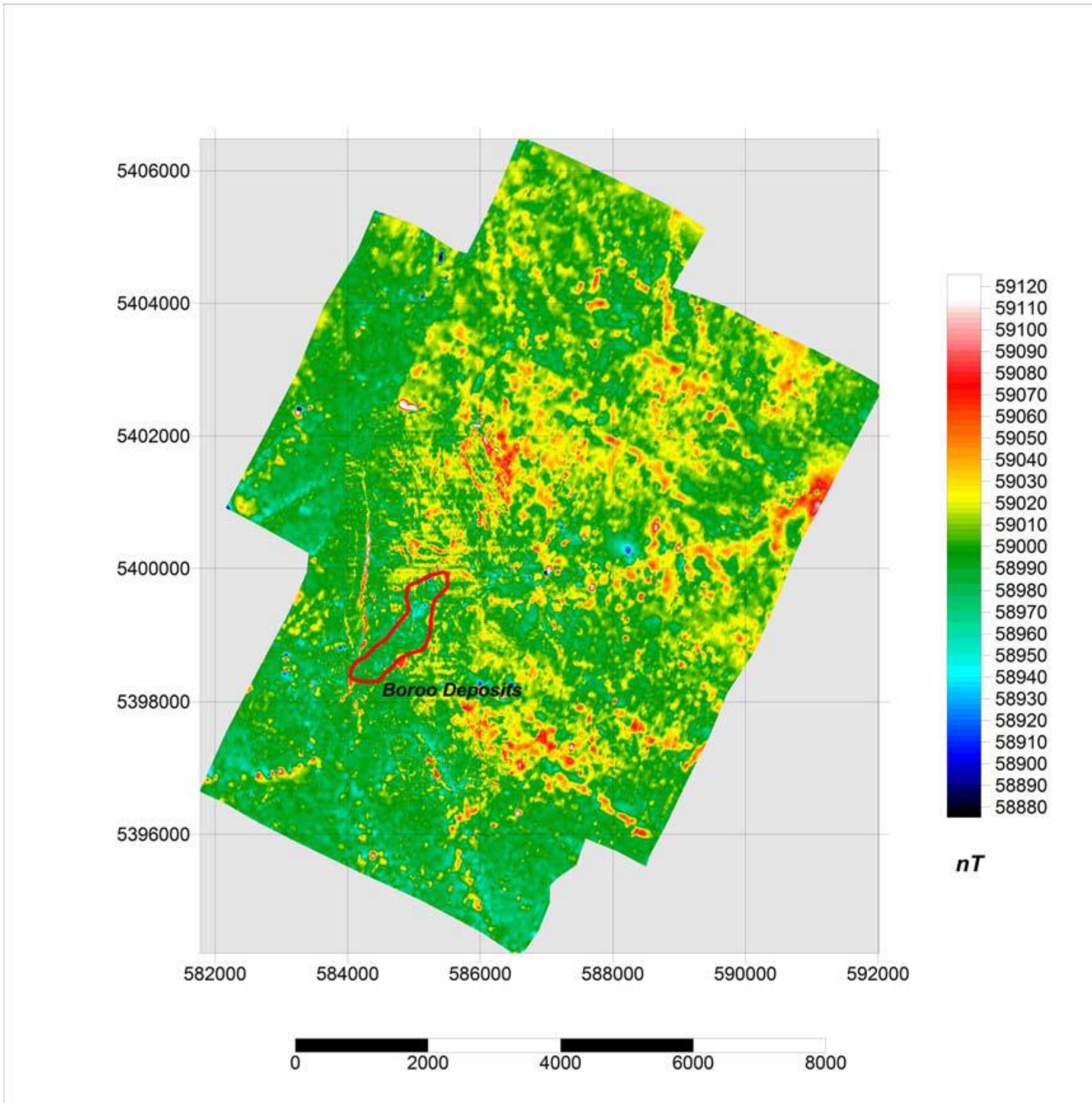


Figure 9-5: Boroo Ground magnetic data

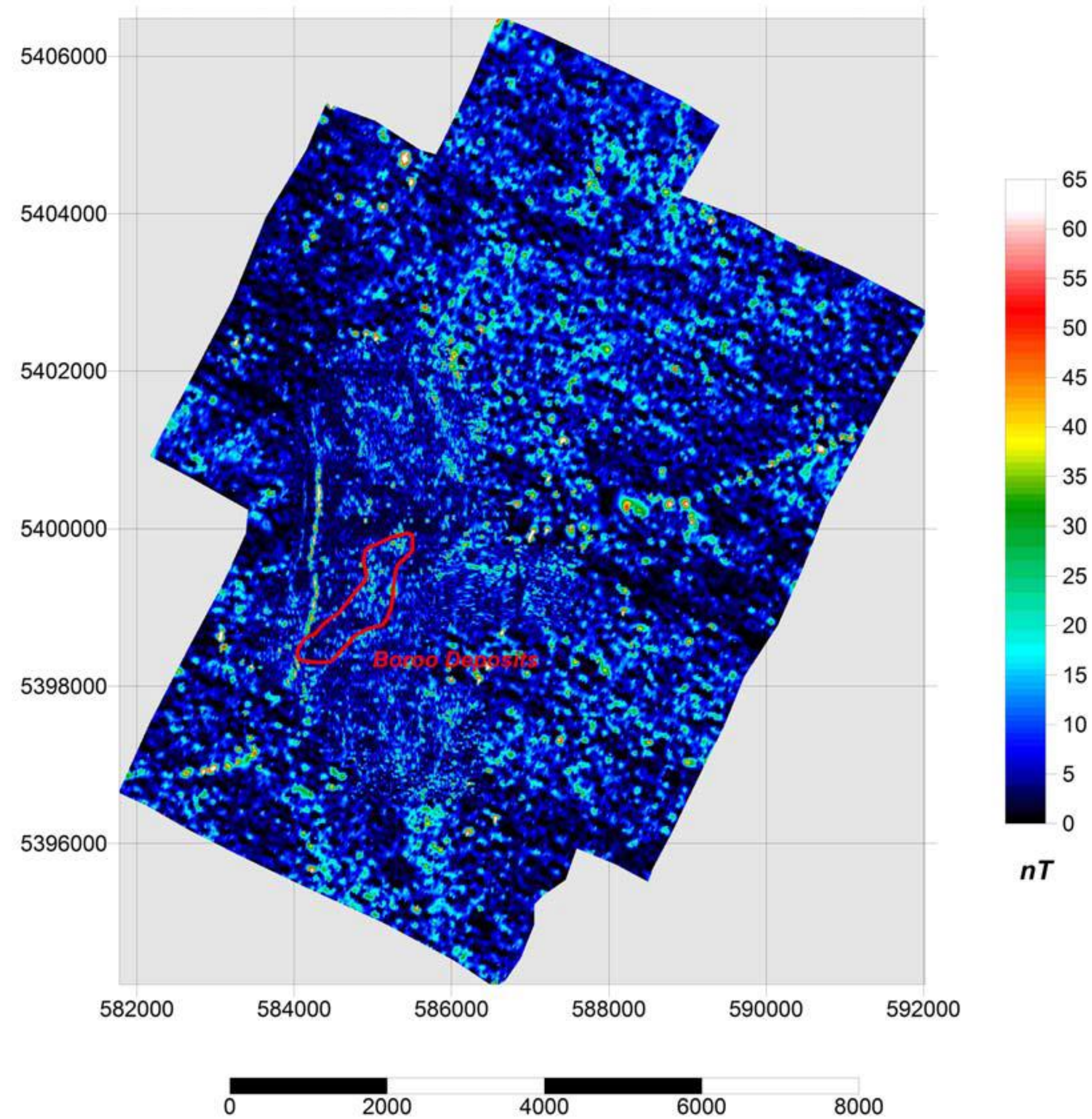


Figure 9-6: Boroo Ground magnetic Data: Difference between the actual and the filtered (low pass) data

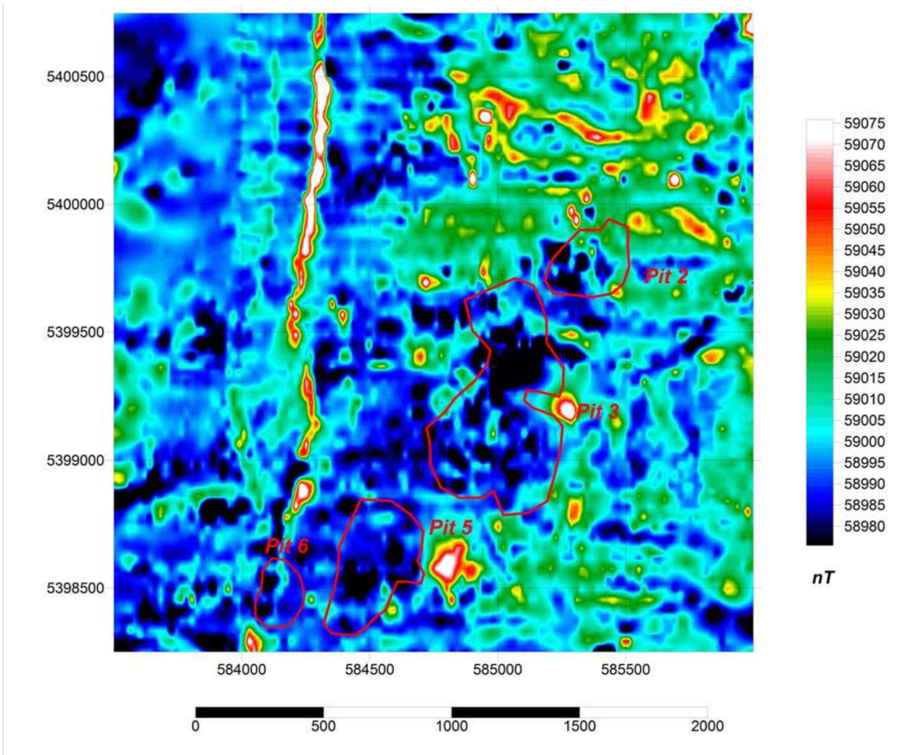


Figure 9-7: Ground magnetic Data over the Boroo deposit

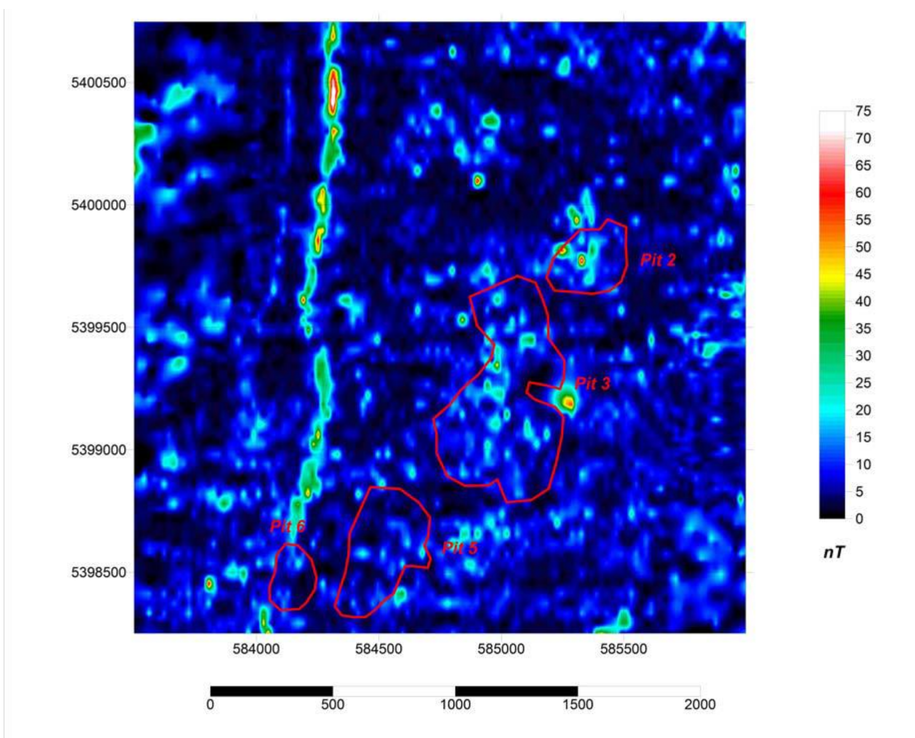


Figure 9-8: Ground magnetic Data over the Boroo deposit: Difference between the actual and the filtered (low pass) data.

9.2 Ulaanbulag

Additional exploration work, which included geophysical surveys, trenching and diamond drilling at Ulaanbulag lead to the development of a Mongolian category reserves and resources in 2008. The Mongolian category reserves and resources were confirmed by the Mongolian State Mineral Authorities and the exploration licence was transferred into mining licences on November 30, 2009.

During 2009 to 2014, additional exploration works were performed at Ulaanbulag and updated the Mongolian category reserves and resources in 2016. In the same year, a feasibility study of the Ulaanbulag was developed and approved by the Mongolian government authorities.

9.2.1 Geophysical Survey

The only chargeability (IP) body on the Ulaanbulag gold that is characterised by intrinsic chargeability higher than the 5-10 mV/V background values for the Boroo exploration areas, is located in the north-western part of the grid and strikes in on an approximate N-S direction along 597500 E.

As the data from sections on line 587100 E-587900 E (the northern parts) the high intrinsic chargeability zones are coincident with elevated resistivity in the geo-electrical section. Considering that these "targets" are covered by weathered low resistivity section of zone 25 meters thick, the estimate of the intrinsic chargeability value of the underlaying resistive body may have been over estimated.

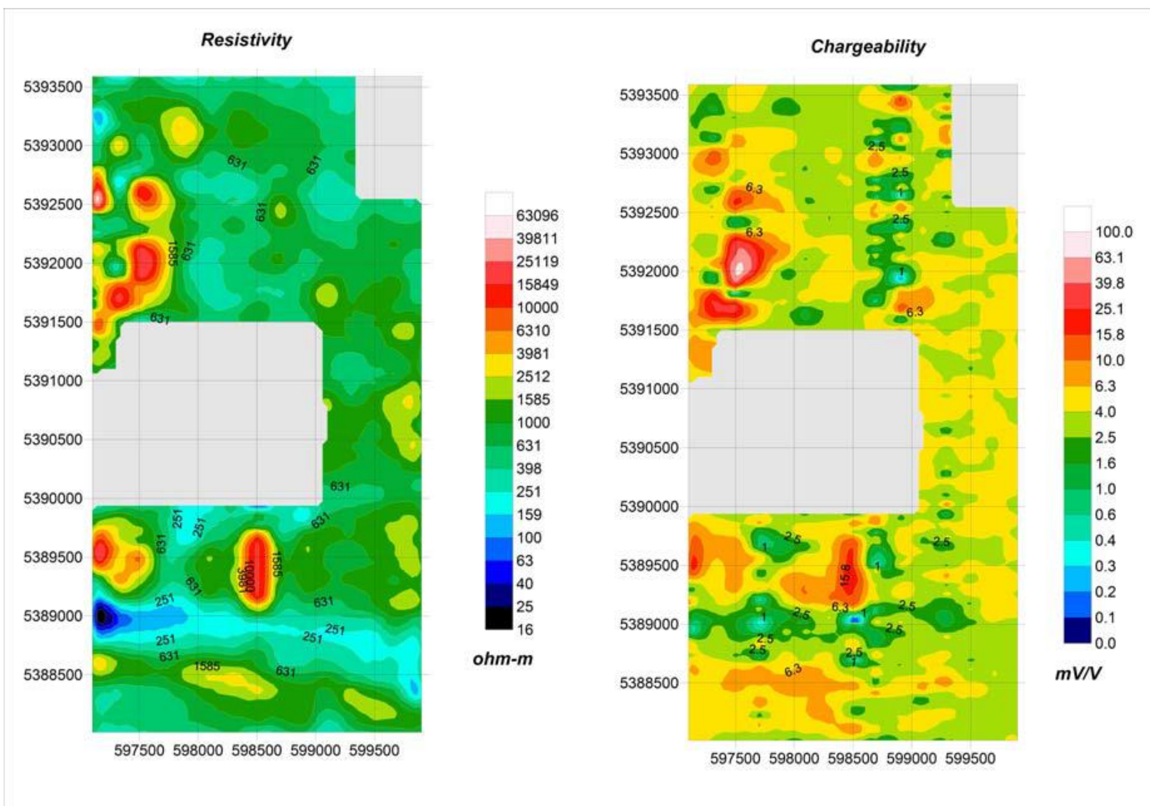


Figure 9-9: Ulaanbulag: Inverted (modelled) Resistivity/Chargeability between 75-100 meters below the surface

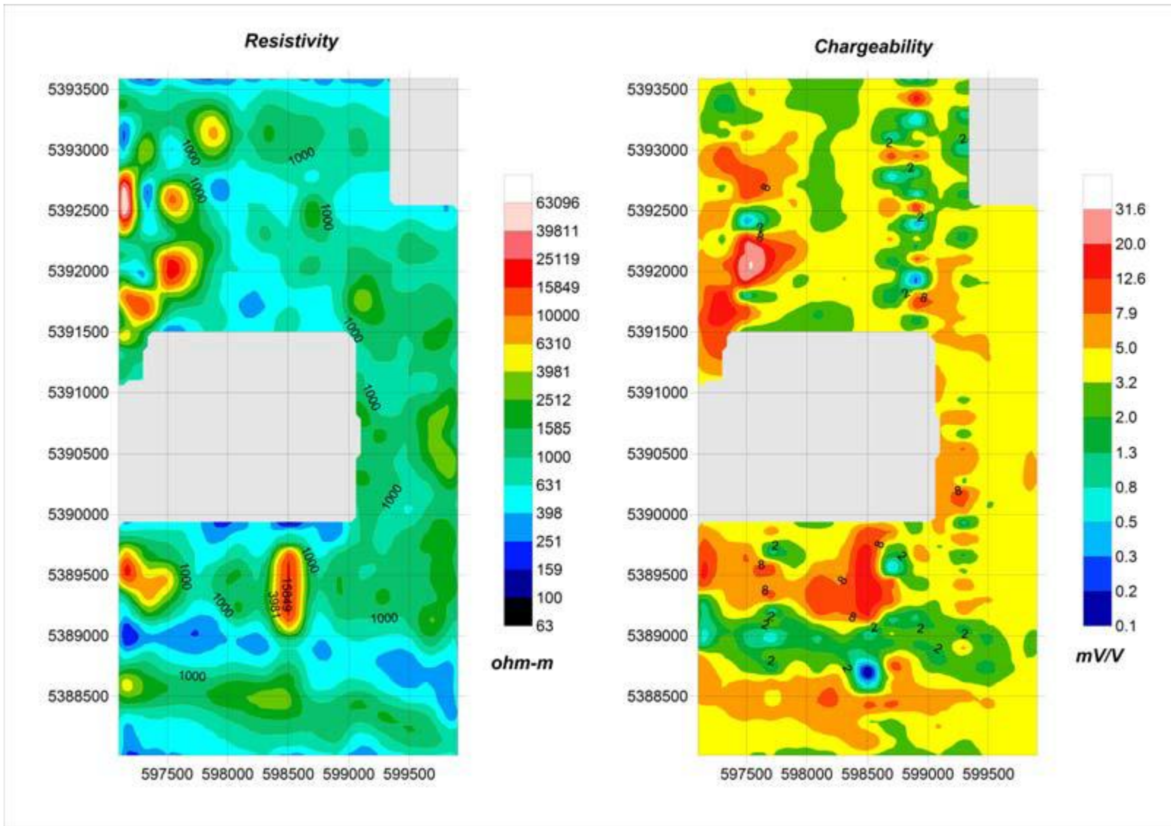


Figure 9-9: Ulaanbulag: Inverted (modelled) Resistivity/Chargeability between 125-150 meters below the surface

10.0 DRILLING

10.1 Boroo

As part of the preparation for a bankable feasibility document, AGR undertook a significant in-fill drill program of the deposit area in 1999 to bring the existing "reserves" to comply with the Australian JORC code for reporting of mineral resources and mineral reserves. The definition drilling programs goal was to evaluate mineralization within optimized pit shells developed by Resolute in July 1998. The main zones of mineralization were drilled on a nominal 40 m by 40 m pattern. Ulaanbaatar-based Gobi Drilling, a division of Radial Drilling of Townsville, Australia, was contracted for a combined reverse circulation (RC) and diamond drilling program undertaken in 1999.

Three drilling programs with tightly spaced RC holes were also undertaken by BGC in 1999 to establish the continuity of mineralization and to provide a geostatistical basis for the mineral resource estimation process. These were completed on what was defined as Zone 5 and on the interpreted "high grade lens" in Zone 3 (Centerra, TR2009).

The property essentially sat idle for several years during the recent gold market low cycle. Cameco began negotiations to acquire a controlling interest in AGR and their principal Mongolian asset, BGC and the Boroo deposit, in 2001, culminating in an agreement announced in March 2002. Shortly thereafter, construction of the Boroo mine commenced and commercial production was achieved by BGC in March 2004. During the construction phase approximately 200 additional RC holes were drilled by CGM to confirm reserves and provide additional exploration data in the district. Cameco was renamed Centerra and became a public company in June 2004, and subsequently acquired AGR's remaining interest in BGC raising Centerra's interest in the Boroo mine to 95% (Cluer et al, 2005).

Additional RC infill drilling was undertaken in 2005 by BGC which resulted in a substantial amount of additional data for the Project. Most of this drilling was for reserve / resource delineation and definition purposes the results of which have been included in the current resource and reserve estimation model.

During the period from 2006 to October 31, 2009, the majority of drilling outlined in Table 10-1 was condemnation drilling well outside the limits of the Boroo reserve model, or drilling to collect samples for additional metallurgical test-work and geotechnical studies and therefore would have no material impact on the resource model.

Once mining operations were started in 2003, on site drilling and exploration has been carried out by the Boroo mine geological staff with the assistance of CGM exploration staff. All diamond drill core, RC samples, rejects and pulps from drilling since 2004 are in locked storage at the Boroo mine site.

From 2010 to 2021, further exploration, development and infill drillings were completed throughout the mine using diamond drilling method. Drill cores from diamond drilling of all drill holes completed were sampled (GSTATS, TR2023).

In 2023, totally 3,629.8 meters infill drilling has been conducted the using diamond drilling method.

Table 10-1: Summary of drilling programs (Boroo)

Year	Company	Type of Drilling	Drill Hole Prefix	Number of Holes	Length, m
1982-1989	DDR-MPR Expedition	DDH	JE	343	28,431
1992-1994	MKG/Erdene	DDH	Unknown	22	Unknown
1997	BGC	DDH	BG	9	735
1999	BGC and AGR	DDH	MDD	31	2,033
		RC	MRC	185	11,902
2002	BGC and CGM	RC	MRC, BX	222	15,887
2003	BGC and CGM	RC	BRC, BX	233	10,115
2004	BGC	DDH	BRD	18	3,019
	BGC and CGM	RC	BRC, BX	224	23,141
2005	BGC	DDH	BRD	3	420
		RC	BRC	305	35,817
2006	BGC	DDH	BRD, BDD	20	2,944
		RC	BRC	76	10,930
2007	BGC	DDH	BDD, BGD	14	1,326
2008	BGC	DDH	BDD, BGD	20	2,051
2009	BGC	DDH	BDD	7	1,526
2010	BGC	DDH	BDD	18	1,224
2012	BGC	DDH	BDD	19	2,353
2019	BGC	DDH	BDD	6	2,000
2020	BGC	DDH	BDD	19	1,025
2021	BGC	DDH	BDD	72	9,021
2023	BGC	DDH	BDD	37	3,629.8
Totals				1,903	169,530

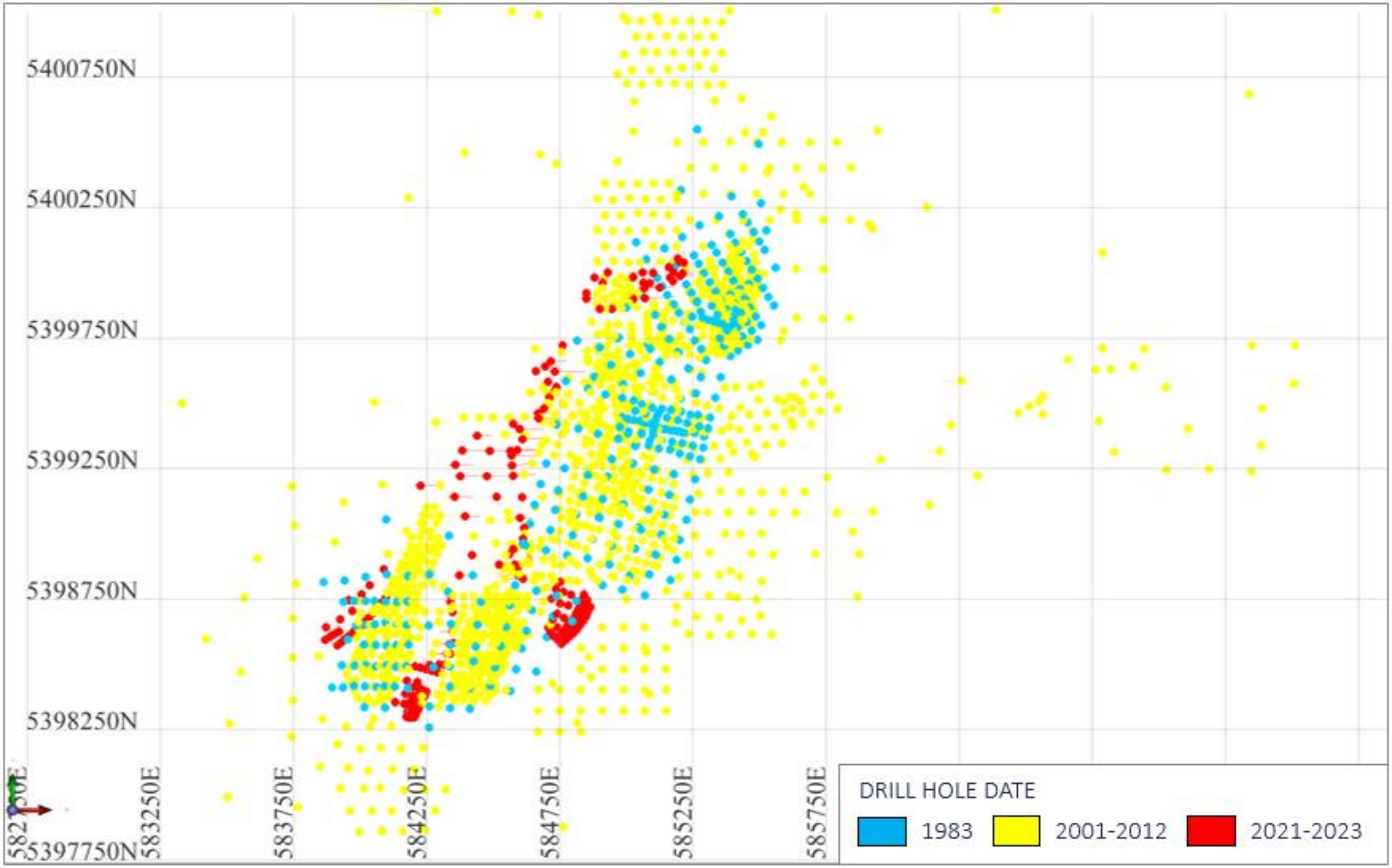


Figure 10-1: Drilling Programs on Boroo Area

10.2 Ulaanbulag

Additional exploration work, which included geophysical surveys, trenching and diamond drilling at Ulaanbulag lead to the development of a Mongolian category reserves and resources in 2008. The Mongolian category reserves and resources were confirmed by the Mongolian State Mineral Authorities and the exploration licence was transferred into mining licences on November 30, 2009.

During 2009 to 2014, additional exploration works were performed at Ulaanbulag and updated the Mongolian category reserves and resources in 2016. In the same year, a feasibility study of the Ulaanbulag was developed and approved by the Mongolian government authorities.

A staged exploration and resource delineation drilling program was carried out across the Ulaanbulag prospect between 2003 and 2023. Table 10-2 provides a breakdown of the number of holes and metres drilled over this time period. All drilling on the license area was carried out by the independent drilling contractor, such as Can-Asia drilling, Land drill international and Falcon drilling Mongolia.

Table 10-2: Summary of drilling programs (Ulaanbulag)

Year	Type of Drilling	Number of Holes	Length, m
2003	DDH	10	1,121
2004	DDH	22	2,594
2005	RC, DDH	14	1,894
2006	RC	10	1,512
2007	RC	11	1,137
2008	DDH	28	840
2009	RC	6	302
2010	RC	84	7,434
2018	DDH	9	1,801
2023	DDH	72	5,916
Totals		266	24,552

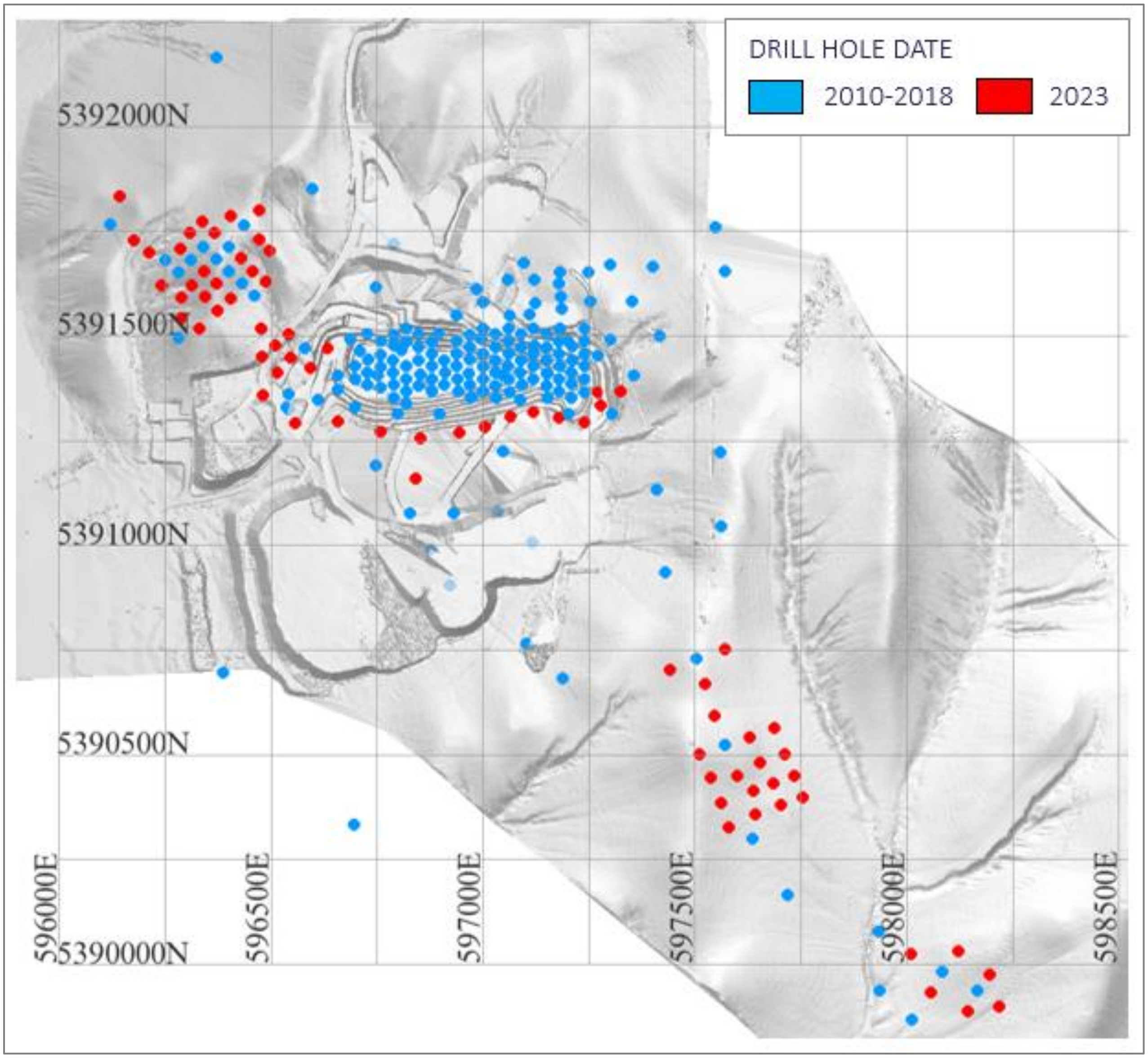


Figure 10-2: Drilling Programs on Ulaanbulag Area

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 PRE-1997 Historical Methods

When BGC conducted its due diligence drilling in 1997 under the supervision of an independent Australian consulting firm, a critical review of the DDR-MPR drilling and sampling and assaying methods was conducted, as described in Waltho (1998). Waltho also had the opportunity to inspect some of the DDR-MPR drill core from the Joint Expedition (JE) holes and reported that the drill logs and assay ledgers were of excellent quality, describing the sample protocol used that observed lithologic boundaries and identified sections of lost core.

Nearly all the drill core was split in two, with one-half submitted for assay. According to SRK (2000), most of the sampling appears to have been done by splitting rather than sawing. Full core was used for assaying in cases where the core was smaller than 76 mm and for drillholes that were spaced so tightly that their results were used to estimate Soviet category B resources. Unfortunately, most of the remaining half-core record has been lost, having been misplaced or destroyed since 1997 and was not available for more recent due diligence studies or reviews.

On the basis of the evidence given by authors who had the opportunity to observe the results of the historical methods and approach to drill core sampling and that this period of drilling has no material impact on the 2009 Reserve and Resource estimate. CGM concluded that the methods applied reflected reasonably historical and current industry standards, and that there were no obvious negative issues with the drilling.

The sample preparation and analysis protocols of the DDR-MPR campaigns is described in Waltho (1998) and Cameco Gold Mongolia LLC (2004b). The samples produced from drill core or underground openings were submitted to the Central Laboratory in Ulaanbaatar, a Soviet-era institution. Each sample was initially crushed to -1 mm and a 1 kg split was coarse pulverized to minus 315 microns (50 mesh). This material was split into two 250 g subsamples of which the first was subjected to a spectral gold analysis without further comminution. For this method, the pulverized sample is digested in aqua regia and the gold adsorbed by powdered activated carbon. The activated carbon is combusted in an optical emission spectrograph using a direct current arc, and the resulting spectrum fixed on photographic plates where the strength of the signal is compared to known gold concentrations. This method was in wide use in the former USSR, both for gold and for other metals.

Samples returning elevated gold values (the threshold was variably set at 0.1 g/t or 0.2 g/t), were re-assayed using the second 250 g split that was pulverized to minus 63 microns (230 mesh) and subjected to a 50 g fire assay with an atomic adsorption (AA) finish (Waltho, 1998, page 85). Given the secrecy surrounding any precious metals exploration and its results in the former USSR, there is reason to believe that security measures were taken to protect the samples (and the information derived from them), although no actual information is available.

A number of trace elements were determined to occur, among them silver, arsenic, antimony, chromium, nickel, and barium.

According to Waltho (1998) and Cameco Mongolia Gold LLC (2004b), a rigorous check assaying regime was maintained during the DDR-MPR programs, involving the systematic submission of duplicate samples amounting to 5% to 10% of the total sample stream to three outside laboratories, all of them in the former East Germany (in Halle, Stendal and Bismuth). Waltho has reviewed the results of 439 duplicate assays in 1997 and has concluded that they "…indicate excellent agreement between original gold assay and duplicate assay…. The duplicate assay precision exceeds ten percent for 50 percent of sample pairs" (Waltho, 1998, page 86).

As part of a detailed project review in 1999, SRK (2000a) has also commented on the fire assay results of the JE drillholes. SRK noted that the "…fire assaying of JE holes is generally of poor precision." and that "…the overall precision is not adequate to enable reliable local estimation of resources for bankable feasibility." They also note, however, that "… there is little overall bias between original fire assay and repeat fire assays. The issue relates to precision, not accuracy ". And they finally conclude "…that area solely informed by JE holes should not be considered for conversion to reserves in the feasibility study, i.e. can only have the status of Inferred Resources. Areas with substantial modern drilling can be classified as Indicated. In these areas, the JE holes can be used" (SRK, 2000a, pages 16, 17).

As part of the BGC due diligence in 1997, 254 repeat "samples collected from original DDR-MPR expedition drill core …. by BGC geologists were also analyzed at Analabs, Ulaanbaatar." Despite a problem to exactly match the original sample intervals due to lost core intervals that had moved, the "analysis of this data by MRT confirmed the existence and tenor of mineralization at Boroo. … The work completed by MRT demonstrates that original DDR-MPR expedition gold assays are both accurate and precise" (Waltho, 1998, Summary, page ii).

There is a difference of opinion between Waltho (1998) and SRK (2000a) regarding the precision of the JE drillhole fire assay database. Having reviewed the graphs in both reports, Centerra (2009) did not consider the lack of precision in the JE assay data sufficient grounds to relegate areas solely defined by JE holes to the Inferred Resource category, although the argument is now largely moot, because, as of October 31, 2009, a small and immaterial amount of the drillhole database was constituted by the JE holes, the majority of the Mineral Reserves effected by this drilling has now been mined out.

There is no indication that the historical part of the Boroo assay database is biased, but individual assay results do have a significant degree of imprecision, as a result of the inhomogeneous distribution of the gold in the Boroo mineralization ("nugget effect").

11.2 1997 AGR/BGC Methodology

In 1997, BGC conducted a due diligence drilling program consisting of nine diamond holes (**Table 10-1**). Drilling was performed by Vancouver-based Can-Asia Drilling Services Ltd. The holes were sampled by MRT personnel in Mongolia (Waltho, 1998). The core was stored at the BGC exploration camp in a locked shipping container. MRT sampled the drillholes following lithological intervals based on the BGC geological logs in a manner that provided samples with an average downhole length of about 1 m. Whole core was sampled, and 661 samples were packed in heavy plastic bags which were sealed using staples and a self-adhesive label. Bags were packed into small wooden crates that were nailed closed for shipment to Analabs in Townsville, Australia.

The 661 samples originating from the nine BGC due diligence diamond holes in 1997 (BG series) were initially analyzed at Analabs in Townsville, Australia using a 50 g fire assay aliquot with an AA finish under supervision by MRT. Analabs was subsequently certified by the Council of the National Association of Testing Authorities, Australia, as an accredited laboratory in September 2000, but did not have that certification in 1997. All samples assaying greater than 4 g/t Au were re-assayed in duplicate using gravimetric and metallics screened fire assay techniques to confirm the accuracy of fire assay data and examine the contribution of coarse gold to high grades associated with some samples.

After Cameco Gold assumed responsibility for the project in March 2002, the logging and sampling protocols were updated and improved. For RC drilling, new protocols focused on improvement of the sampling quality which included on-site chip tray filling, permanent geological control and preliminary field logging. Chip trays were sent to the CGM Ulaanbaatar office, where detailed logging took place. The geology and alteration of the cuttings were systematically logged and recorded in digital format.

This information was used to specify mineralized intervals where 1 m samples from intervals identified as mineralized or altered were submitted for assay directly. When samples appeared visually un-mineralized, 2 m composite samples were created and assayed. Where these samples returned anomalous gold values, the appropriate 1 m sample was also submitted and assayed.

Transport of the samples from the Boroo site to Analabs Ulaanbaatar, where the receiving officer at the laboratory certified the samples as received, was under the supervision of a staff geologist.

Drill core from the 28 holes of the MDD drilling program completed by AGR in 1999 were shipped to Analabs in Ulaanbaatar for sawing. SRK (2000a), who were able to physically inspect much of the JE and later BGC drill core pronounced themselves satisfied: "In summary, core sampling is considered by SRK to be acceptable for the purpose of the project" (SRK 2000a, page 11), with the "purpose of the project" referring to the AGR feasibility study. Subsequent drilling as part of the feasibility study was by RC methods, with the sample protocol used for the MRC holes in 1999 described in SRK (2000a) as follows:

- A first-pass sample taken by scooping chips from each 1 m interval and combining into 4 m composites. One-meter "re-samples" were taken at the same time, using the more conventional and more reliable Jones riffle splitter; however, these were only submitted for assay if the 4 m composite "scoop" assay returned a value of greater than 0.2 g/t Au. The samples from occasional wet intervals were also taken by scoop, which would normally be considered an unsatisfactory method.
- There are unfortunately no records from the drill logs form this period outlining how many RC holes had wet intervals, and how many such intervals are in the database.

For the 1999 program, assays were performed at Analabs in Ulaanbaatar, a commercial laboratory established by its Australian parent that had not yet received certification. The sample and assay protocol used by AGR/BGC for the 1999 MRC drillholes (Table 10-1), that represent nearly one-quarter of the drillhole database for the current resource estimate, is described and commented upon in SRK (2000a):

- Samples received are dried for three to four hours.
- A one-half to 1 kg sample is riffled out for further crushing and pulverizing.
- Crush the subsample in a jaw crusher to minus 5 mm.
- Pulverize the subsample to 95% passing 75 microns (200 mesh). The capacity of the pulverizer was the reason for the initial sample splitting after drying.
- A 50 g aliquot of the pulp is fire assayed with an AA finish at all gold concentrations.

The following observations can be made on the sample preparation and assaying protocols for the 1999 program:

- The initial splitting of the RC samples into subsamples with a mass of only 0.5 kg to 1.0 kg before further comminution will have increased the sample error attached to those assay results, due to the inhomogeneous gold distribution in this style of mineralization, continuing the poor reliability of individual assays of the earlier drill campaigns.
- The use of the metallics screen assay method is a more reliable method for such materials, and that method could have been put to good use on the higher-grade part of the Boroo sample population to reduce the variance of individual assay results. The lack of a laboratory balance at Analabs to perform a gravimetric finish for gold assays of >10 g/t is a further deficiency.
- The results of internal pulp repeat assays at Analabs are graphed in SRK (2000a), showing very good precision.
- AGR/BGC did not add any external standards to the sample stream, a QAQC measure that was standard industry practice at the time.

The sample preparation and assay protocols in place during the 1999 drill program were less than optimal for Boroo-type gold mineralization, affecting about one-quarter of the database used in the 2009 Resource estimate (current estimate). The lack of proper quality control does not mean that the resulting assay data are unreliable, but it means that reliability cannot be documented.

However, indirect evidence is available as to the performance of Analabs in 1999. While BGC did not submit any external standards together with the samples from Boroo, the exploration group of CGM (and predecessor company Cascadia), undertaking work elsewhere, did. The results of the CGM QAQC measures were reviewed by Analytical Solutions (2002), who concluded that the results "… for standards submitted to Analabs in 1999 are biased low …" and that there was "… a possibility that Analabs had technical difficulties during this time period that could affect gold assays" (Analytical Solutions, 2002, Executive Summary, first point).

- Cameco Gold (2001) report that 657 new splits were renumbered and submitted to Analabs by BGC. The results are described as follows: "The original and duplicate assay pairs show poor reproducibility, however there is not a bias …" (Cameco Gold, 2001, page 8). The poor reproducibility is at least in part the obvious result of the poor sample preparation protocol and the resultant increase in the sample error. Analytical Solutions (2002) concluded with respect to this set of 657 repeat assays as follows: "For re-splits between 5 to 10 g/t Au there is no statistical significance to the differences [i.e., there is no bias], probably due to inhomogeneity of the rejects with respect to gold. Samples with low gold grades (<5 g/t) show that there may be slight statistically significant bias towards higher assays in the original splits compared to the re-split assays" (Analytical Solutions, 2002, page 11).

- In 2002, a further 436 check assays were performed on samples from the 1999 MRC drillholes. This included 338 samples collected from the coarse reject bags stored at the Boroo site (Cameco Gold Mongolia, 2003b). The entire content of each coarse reject bag was recovered, re-bagged and sent to Analabs for sample preparation and fire assay. The original bags were rotted and open to the wind, rain and surface contamination, and it was difficult to confirm the exact sample numbers, but they all appeared to be in the original order as laid out by BGC geologists originally responsible for archiving the samples.
- The repeat assay results show no significant differences between the means and standard deviations between the two assay sets, but scatterplots demonstrate considerable differences of individual pairs, in line with earlier findings and additionally caused by the poor condition of the samples and identification problems during sample retrieval.

Finally, 98 pulps were also recovered from storage in Ulaanbaatar and re-assayed by Analabs. Comparison between original assays and re-assays of the 98 pulp samples shows a good correlation (R2 >0.9) between the two sets.

As was apparent for the early (historical) assay data, that part of the Boroo assay database created in 1999 is not biased, but, as before, individual assay results have a significant degree of imprecision, as a result of the sample preparation protocol and the small aliquot size, particularly for high-grade samples.

11.3 2003-2011 Centerra-BOROO/BGC Methodology

Since 2003, both RC and diamond drilling completed on the Boroo Project, under the direction of the Boroo exploration staff, was with the sampling method and approach described as follows:

- RC samples were taken at 1 m intervals. RC cuttings were collected from the sample collector; dry samples were mixed using the three-stage sample splitter. Two sets of samples weighing 3 kg to 5 kg were each packed in identically numbered cloth bags; one submitted for laboratory testing; the other stored in the secure sample storage area on the mine site as a coarse reject. From the primary sample, a subsample was taken, screened through a 2 mm sieve, rinsed with water and the sieve oversize packed in a special chip tray, for geological logging. From the logging results, samples identified to be mineralized, were composited to 1 m intervals and shipped for analysis, whereas samples identified without alteration or mineralization, were composited to 2 m intervals before being shipped for analysis
- Chip trays were locked in storage at CGM's Ulaanbaatar office; rejects and pulps were stored in locked storage in CGM's warehouse in Ulaanbaatar.
- Sampling of drill core was carried out at 1 m intervals in the mineralized section and at 2 m intervals in unaltered host rocks. Drill core was split or cut in half using a diamond saw, half of each sample

was used for assaying and geochemical analysis and the remaining half-core was stored for record keeping at the mine site.

From 2003 to 2008, exploration drilling was completed throughout the mine using diamond and RC drilling methods; and within the scope of the drilling program, a total of 468 holes were completed for exploration purposes; 134 holes for reserve definition; 295 holes for condemnation; and 16 for geotechnical and metallurgical purposes, respectively. Drill cores from diamond and chips from RC drilling of all drillholes completed throughout the entire deposit were sampled.

As the Project has operated without an on-site assay laboratory, all geological, mine and mill samples were bagged and logged at site for shipment to a commercial laboratory in Ulaanbaatar for sample preparation and assaying. From 2002 to 2006, SGS Mongolia Laboratories (SGS) was employed.

Since 2007, the limited number of additional exploration samples shipped for analysis have been performed by Actlabs Asia LLC in Ulaanbaatar, which commenced operations in June 2006 and achieved ISO 17025 accreditation in March 2008. Actlabs Asia LLC participates in Proficiency Testing Programs with both CANMET in Canada and Geostats in Australia.

As majority of the drilling that would impact the Mineral Reserves and Resources was done between 2003 and 2005 when Boroo was shipping exploration samples to SGS in Ulaanbaatar, the sample preparation procedures from the SGS laboratory are described as follows:

- All samples received by SGS were received with an internal sample control number and entered in the laboratory information management system, which eliminates the need for manual data entry and reduces human transcription error.
- The samples were sorted and dried prior to being crushed to 90% passing 3.0 mm in a Rhino/Terminator jaw crusher before being reduced in a Lab Tech Essa 201 eight-bin rotary splitter to a 750 g to 1.1 kg subsample.
- The subsample, which is suitable in size for single-pass comminution in a Labtech LM2 bowl and ring pulverizer was pulverized to 80% passing 75 micrometers. The crusher and pulverizer were flushed by compressed air after each sample.
- Two splits of 300 g each were taken, with one allocated for assay and the second archived.
- Assaying employed fire assay digestion on a one assay ton aliquots and an atomic absorption finish on a 10 ml volume by a Varian Spectra 50A or 55B unit. Detection limit was 0.01 g/t, or 10 ppb.
- SGS used conventional flux and crucibles which fires 50 samples per batch and dissolves the prill by HNO_3/HCl.

11.4 Internal Quality Assurance and Certification of Labs

As with most commercial labs, SGS utilize its own certified standards and blanks as part of its QAQC process. As an example, one in 10 samples is subjected to additional assaying for control purposes. All sample tests were depicted at test control monitor of the lab. Results two times higher than the standard deviation are considered to be the warning level and further monitoring tests are performed if this deviation is three times higher. Selected samples are subject to re-assaying if the deviation exceeds the warning level and all samples are re-assayed if the results exceeded monitoring level.

By October 2005, the bias variation at SGS was between -1.82% and +2.83%, which is considered satisfactory. Other deviations are between +3.8% and +4.8%, which are also below the maximum variation level of 5%.

Test results performed by the SGS lab were reviewed by Linda Bloom of Analytical Solutions Ltd. (based in Toronto) and assessed satisfactory as per request of the SGMK. Bloom has issued a number of recommendations pertaining to the operational quality assurance program of both SGS and SGMK which were adhered by these organizations.

11.5 Boroo QAQC Programs

BGC mine staff undertook an industry standard QAQC program which was designed to provide confidence in the accuracy of the assaying process of the mine exploration and infill drilling. QAQC reports were prepared and presented to Centerra on a quarterly basis. The same procedure applied to the Boroo respectively. The QAQC program consisted of the following procedures:

- One of two commercial blanks is added at a rate of one in every 20 to 40 samples. Submitted to the commercial laboratory.
- Standards are added a rate of one in every 20 to 40 samples.
- Duplicate samples amounting to 5% of the total number of samples are submitted for testing.
- Control tests are performed on approximately 200 samples at a secondary laboratory if the grade is above 0.8 g/t gold. The control tests cover approximately 8% of the mineralized samples and are performed in the second half of each year.

The number of samples, which were collected and tested at laboratories within the scope of the BGC mining exploration and development program, and the number of control samples used for quality control of the samples are shown below in Table 11-1.

Table 11-1: Summary of Quality Control Samples (Boroo)

Year	No. of samples	Quality Control Samples		
		Blank	Standard	Duplicate
2004	16,260	167	373	-
2005	25,604	338	663	89
2006	8,851	356	487	281
2007	611	75	291	80
2008	3,446	61	70	8
2009	910	36	40	-
2010	1,410	46	47	-
2012	1,392	56	64	31
2019	861	15	20	35
2020	691	16	30	30

Year	No. of samples	Quality Control Samples		
		Blank	Standard	Duplicate
2021	4,816	68	140	192
2023	1080	14	43	38
Total	**65,932**	**1,248**	**2,268**	**784**

11.5.1 Blank Samples

A total of 167 blank control samples of types of Blank A and Blank B were tested in 2004. Of these, a total of 15 samples had grades of over 0.1 g/t Au which were re-tested in bulk samples. The resulting conclusion was that the test work was successful, as the grade of all blanks was detected to be below 0.1 g/t Au. These blank samples were prepared in the Shea Clark Smith (SCS) laboratory, Nevada, USA.

Since 2005, blanks were prepared from rejects of the original splitting of samples that assay below the detection limits of the laboratory. These reject samples were returned to the mine site where they were packed and classified into oxide, transitional and fresh material according to the mine general lithology. A certain number of blank samples were sent each time when sending samples for laboratory testing. The certified blank samples from Rocklabs laboratory were utilized additionally.

Blank samples were inserted within the same lithology sample intervals in the submissions sent to assay to determine whether there was a source of any contamination during the sample preparation stage (i.e., crushing, pulverization and fire assay). Laboratory test results of blank samples assaying below 0.1 g/t Au were deemed to be within the permissible level.

Fifty-five blank samples (7% of total blank samples) returned above 0.1 g/t during 2005 to 2008. All submissions to the lab were re-analyzed and database was updated and validated by mine geology team. Three samples from 2006 and 2007 were selected from reject samples of ore intervals with the tagging errors from the lab and they were resolved with correct tagging.

Exploration samples were analyzed in SGS laboratory in 2004, 2005 and 2019 to 2021. Samples were analyzed both in SGS and Actlabs laboratories in 2006 and 2007 and since then Actlabs solely analyzed until end of the 2012.

The result summaries of blank samples are shown in Table 11-2.

Table 11-2: Blank Sample Test Results

Year	No. of Blank Samples	Results Returned, No. of Samples and %					
		< 0.1 g/t		0.1 g/t <		0.5 g/t <	
2004	167	152	91%	15	9%	5	3%
2005	338	334	99%	4	1%		
2006	356	311	87%	45	13%	8	2%
2007	75	71	95%	4	5%	2	3%
2008	61	59	97%	2	3%	1	2%
2009	36	36	100%				

Year	No. of Blank Samples	Results Returned, No. of Samples and %					
		< 0.1 g/t		0.1 g/t <		0.5 g/t <	
2010	46	46	100%				
2012	56	56	100%				
2019	15	15	100%				
2020	16	16	100%				
2021	68	68	100%				

11.5.2 Standard Samples

Standards are packaged in weights of 50 g in a pulverized form and labels with codes on the samples are taken off and numbered with new numbers prior to inclusion into the bulk assay for submission to the labs.

Bias in the quality assurance has occurred if the test results of two sequenced standard samples is two times higher or lower from the standard deviation of a given standard; if the test result of any standard is three times higher or lower than the standard deviation then these submissions shall be subject to re-assaying. **Table 11-3** demonstrates standard sample results prepared by SCS inserted to the submissions of exploration samples during 2004 to 2005, while **Table 11-4** to **Table 11-8** demonstrate test results of standard samples prepared by Rocklabs since 2005, respectively. In accordance with the tables, the highest number of samples that did not meet test requirements were recorded in 2004 (24), while lowest number was recorded in 2009 and 2010 (1 each).

Table 11-3: SCS Standard sample analysis results, 2004-2005

Standard Sample Parameters				No. of Samples within StD Limits			Results of Analysis		Samples Out of Acceptance Limits
Code	Quantity	Grade, g/t	STD	± 1	± 2	± 3	Average Grade (g/t)	STD	
ST 0.6	109	0.6		84%	100%	100%	0.53	0.11	
ST 1.0	151	1		86%	92%	97%	1.03	0.28	5
ST 3.0	112	3		23%	72%	96%	2.58	0.25	5
ST 5.5	46	5.5		53%	91%	98%	4.9	0.48	1
ST 8.0	49	8		28%	62%	76%	8.82	0.69	12
ST 12.0	39	12		62%	90%	95%	11.94	0.62	2
ST 15.0	52	15		67%	83%	98%	14.87	0.9	1
ST 19.0	52	19		50%	85%	98%	18.34	1.39	1
ST 23.0	28	23		43%	89%	100%	22.8	1.24	
	638								27

Table 11-4: Rocklabs Standard sample analysis results, 2005

Standard Sample Parameters				No. of Samples within StD Limits			Results of Analysis		Samples Out of Acceptance Limits
Code	Quantity	Grade, g/t	STD	± 1	± 2	± 3	Average Grade (g/t)	STD	
OXD27	21	0.416	0.025	67	95	100	0.442	0.01	
OXF41	11	0.815	0.024	36	64	91	0.849	0.04	1
OXH37	24	1.286	0.039	42	71	92	1.3	0.08	2
OXJ36	4	2.398	0.073	50	75	100	2.482	0.06	
OXK35	24	3.489	0.111	54	71	100	3.467	0.19	
OXL25	14	5.852	0.105	21	43	100	5.703	0.15	
OXP39	9	14.89	0.2	44	56	100	14.654	0.34	
SE19	26	0.583	0.026	73	100	100	0.596	0.02	
SF12	55	0.819	0.028	47	75	98	0.837	0.04	1
SH13	22	1.315	0.034	45	82	100	1.333	0.04	
SJ22	50	2.604	0.042	36	64	98	2.614	0.08	1
SK21	47	4.048	0.091	38	77	100	3.998	0.14	
SL20	34	5.911	0.176	53	94	100	5.776	0.16	
SP17	37	18.13	0.434	19	54	95	18.438	0.86	2
SQ18	20	30.49	0.88	50	75	100	30.695	1.39	
	398								7

Table 11-5: Rocklabs Standard sample analysis results, 2006

Standard Sample Parameters				No. of Samples within STD Limits			Results of Analysis		Samples Out of Acceptance Limits
Code	Quantity	Grade, g/t	STD	± 1	± 2	± 3	Average Grade (g/t)	STD	
OXD27	7	0.416	0.025	57	86	86	0.411	0.04	1
OXF41	8	0.815	0.024	0	50	88	0.83	0.06	1
OXH37	14	1.286	0.039	57	93	100	1.297	0.04	
OXJ36	3	2.398	0.073	0	33	67	2.34	0.24	1
OXP39	4	14.89	0.2	25	75	75	14.45	0.33	1
SE19	138	0.583	0.026	46	81	97	0.609	0.03	4
SF12	129	0.819	0.028	56	87	99	0.82	0.03	1
SH13	16	1.315	0.034	31	69	94	1.349	0.05	1
SJ22	70	2.604	0.042	34	59	99	2.593	0.08	1
SK21	37	4.048	0.091	30	62	100	3.933	0.12	
SL20	1	5.911	0.176	0	0	100	6.27		
SP17	23	18.13	0.434	39	61	87	17.597	0.83	3

Standard Sample Parameters				No. of Samples within STD limits			Results of Analysis		Samples Out of Acceptance Limits
Code	Quantity	Grade, g/t	STD	± 1	± 2	± 3	Average Grade (g/t)	STD	
OXC44	9	0.197	0.013	67	89	100	0.205	0.01	
OXD43	14	0.401	0.021	43	64	93	0.422	0.04	1
OXG46	3	1.037	0.041	67	67	100	1.063	0.05	
OXL34	11	5.758	0.173	64	73	100	5.817	0.26	1
	487								15

Table 11-6: Rocklabs Standard sample analysis results, 2007

Standard Sample Parameters				No. of Samples within STD Limits			Results of Analysis		Samples Out of Acceptance Limits
Code	Quantity	Grade, g/t	STD	± 1	± 2	± 3	Average Grade (g/t)	STD	
OXF41	24	0.815	0.024	63	83	96	0.794	0.04	1
OXH37	24	1.286	0.039	58	71	100	1.303	0.06	
OXJ36	23	2.398	0.073	39	70	96	2.433	0.14	1
OXP39	11	14.89	0.2	55	73	100	14.854	0.32	
SE19	12	0.583	0.026	67	92	100	0.597	0.03	
SF12	11	0.819	0.028	45	73	100	0.838	0.05	
SH13	13	1.315	0.034	69	92	100	1.305	0.03	
SJ22	25	2.604	0.042	32	80	100	2.613	0.07	
SK21	22	4.048	0.091	73	82	100	4.098	0.11	
SL20	10	5.911	0.176	100	100	100	5.941	0.07	
SP17	2	18.13	0.434	50	50	100	17.57		
SQ18	10	30.49	0.88	50	90	100	31.43	0.57	
OXC44	13	0.197	0.013	77	92	100	0.205	0.01	
OXD43	13	0.401	0.021	85	100	100	0.408	0.02	
OXG46	14	1.037	0.041	86	86	100	1.021	0.04	
OXL34	21	5.758	0.173	76	100	100	5.812	0.13	
OXH52	5	1.291	0.025	20	60	80	1.261	0.11	1
SE29	6	0.597	0.016	17	83	100	0.608	0.03	
SI25	11	1.801	0.044	27	55	100	1.771	0.08	
SK33	7	4.041	0.103	0	57	100	3.987	0.22	
SL34	6	5.893	0.14	50	67	100	5.865	0.28	
SN26	8	8.543	0.175	38	75	88	8.269	0.22	1
	291								4

Table 11-7: Rocklabs Standard Sample Analysis Results, 2008-2012

Standard Sample Parameters				No. of Samples within STD Limits			Results of Analysis		Samples Out of Acceptance Limits
Code	Quantity	Grade, g/t	STD	± 1	± 2	± 3	Average Grade (g/t)	STD	
OXC44	2	0.197	0.013	0%	50%	100%	0.22	0.02	
OXC72	1	0.205	0.008	100%	100%	100%	0.21	-	
OXD43	3	0.401	0.021	67%	100%	100%	0.42	0.01	
OxD73	22	0.416	0.013	41%	86%	95%	0.43	0.05	1
OXE56	8	0.611	0.015	88%	100%	100%	0.61	0.01	
OxF65	13	0.805	0.034	100%	100%	100%	0.81	0.01	
OxH52	4	1.291	0.025	25%	75%	75%	1.31	0.07	1
Oxi67	14	1.817	0.062	93%	100%	100%	1.78	0.02	
OXJ47	10	2.384	0.048	30%	60%	80%	2.12	0.74	2
OxJ68	10	2.342	0.064	80%	100%	100%	2.32	0.04	
OXN49	3	7.635	0.189	33%	33%	33%	6.69	0.8	2
SE29	5	0.597	0.016	40%	80%	100%	0.62	0.01	
SE44	31	0.606	0.017	90%	100%	100%	0.61	0.01	
SG40	11	0.976	0.022	0%	27%	64%	0.91	0.07	4
SH24	24	1.326	0.043	71%	96%	96%	1.36	0.25	1
SH35	12	1.323	0.044	92%	100%	100%	1.31	0.02	
SH41	18	1.344	0.041	67%	100%	100%	1.32	0.02	
SI42	4	1.761	0.054	100%	100%	100%	1.76	0.02	
SL20	3	5.911	0.176	100%	100%	100%	5.91	0.07	
SI25	1	1.801	0.044	100%	100%	100%	1.82	-	
SL34	20	5.893	0.14	70%	85%	100%	5.87	0.18	
SN38	1	8.573	0.158	100%	100%	100%	8.61	-	
	220								11

Table 11-8: Rocklabs Standard Sample Analysis Results, 2019-2021

Standard Sample Parameters				No. of Samples within STD Limits			Results of Analysis		Samples Out of Acceptance Limits
Code	Quantity	Grade, g/t	STD	± 1	± 2	± 3	Average Grade (g/t)	STD	
HiSilk	1	3.474	0.063	0%	100%	100%	3.35	-	
HiSilk2	42	3.474	0.087	62%	100%	100%	3.41	0.05	
OxD73	16	0.416	0.013	6%	69%	88%	0.44	0.01	2
OxH112	3	1.271	0.028	100%	100%	100%	1.28	0.01	
OxJ68	1	2.342	0.064	100%	100%	100%	2.35	-	
OxL63	7	5.868	0.141	86%	100%	100%	5.93	0.05	

Standard Sample Parameters				No. of Samples within STD Limits			Results of Analysis		Samples Out of Acceptance Limits
Code	Quantity	Grade, g/t	STD	± 1	± 2	± 3	Average Grade (g/t)	STD	
SE44	39	0.606	0.017	87%	100%	100%	0.61	0.01	
SF45	42	0.848	0.028	88%	100%	100%	0.84	0.02	
SG40	5	0.976	0.022	0%	80%	100%	1.01	0.01	
SG66	19	1.086	0.032	84%	84%	100%	1.07	0.03	
SH41	14	1.344	0.041	100%	100%	100%	1.34	0.01	
	189								2

A total of 2,035 standard samples (Table 11-3 to Table 11-8) were analyzed during 2004 to 2012 from which 64, or 3%, did not meet the acceptable limits. A total of 190 standard samples (Table 11-8) were analyzed during 2019 to 2021. One standard sample returned with failure or no result from the lab and not included in Table 11-8. Two standard sample results returned outside of the acceptance levels.

A total of 2,223 standard samples were analyzed and results falling within particular grade limits or thresholds, as follows:

- 55% of results fall within ± 1 STD of the expected grade
- 82% of results fall within ± 2 STD of the expected grade
- 97% of results fall within ± 3 STD of the expected grade

Typically, a ± 2 standard deviation limit is used as a warning limit. A ± 3 standard deviation limit is used to indicate that a standard result is statistically out of control and require a batch of samples to be re-assayed.

257 samples from 24 batch samples with bias were re-assayed and results reviewed; consequently, the database was revised. Accordingly, 108 samples were tested in 2004, 50 in 2005, 78 in 2006 and 21 in 2007. Re-assayed standard samples returned within acceptable limits. No drilling programs were conducted at Boroo in 2011 and 2013 to 2018. There is no information about the batches re-assayed during 2008 to 2021.

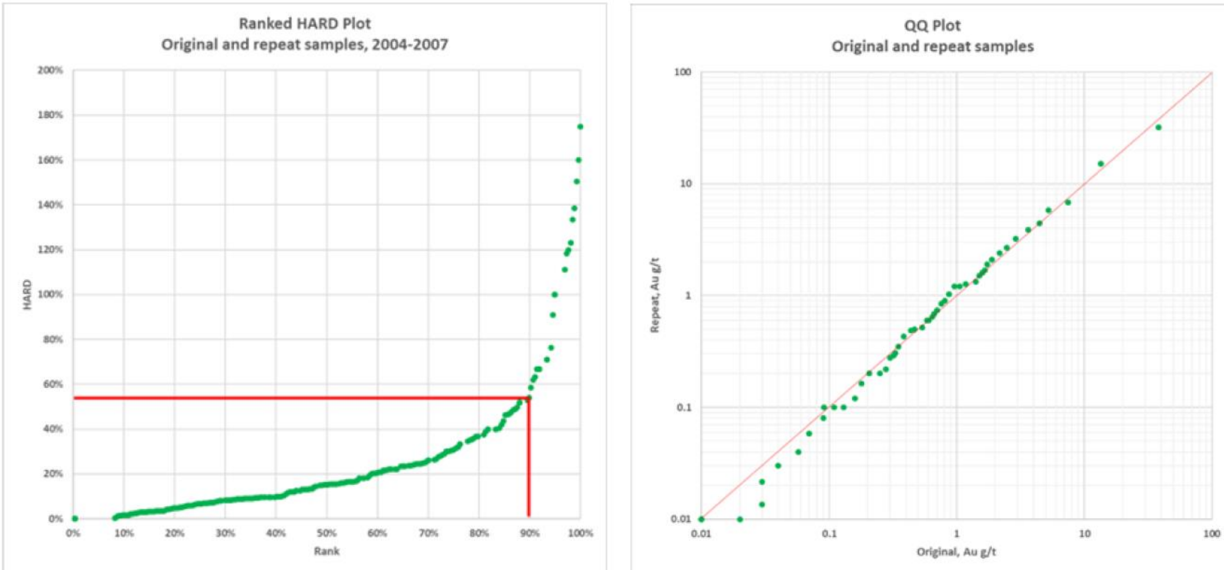


Figure 11-1: Result Plots of Repeat Against Original Samples:

(a) Ranked HARD Plot, (b) QQ Plot

Figure 11-1 shows results of repeat sample results against original sample results. The ranked HARD plot of repeats shows 90% of the samples have a precision that is less than 54% HARD, indicating that the precision of the analysis is relatively poor. The QQ plot for the repeats and original results appears to be a slight bias for most grade ranges. The original samples slightly higher average grade (1.77 g/t) than the repeats (1.71 g/t). The most likely reason for this bias is poor splitting and sub-sampling practices.

11.5.3 Duplicate Samples

Duplicate samples were obtained through the same procedure as the original samples from the RC drilling, whilst one-quarter of the core is taken as a duplicate during core sampling of the diamond drilling. The samples were assayed with one duplicate for every 20 to 40 samples or 5% of all samples were included into samples submitted for assaying. Theoretically, the grade of the duplicate samples should be the same as the original sample; however, variations occur due to gold distribution and nugget effects. Therefore, if distribution of the grade is ±20% compared to the original sample, it shall be considered as a normal indicator for audit purposes. Duplicate samples were numbered and packaged in the same way as the original sample and sent for assaying.

Since 2005, duplicate sample insertion has started for the quality control procedures. A total of 89 duplicate samples in 2005, 281 duplicate samples in 2006, and 80 duplicate samples in 2007 were assayed. All duplicate samples assayed in 2005, 96% of duplicate samples in 2006, and 26% of duplicate samples in 2007 were assayed by SGS Laboratory. The rest of the duplicate samples in 2006 and 2007; and 39 duplicate samples from 2008 and 2012 were analyzed by Actlabs laboratory. A total of 257 duplicate samples 2019 to 2021 were analyzed by SGS laboratory.

Statistics of the original and duplicate samples by sample types are shown in **Table 11-9** and drilling periods in **Table 11-10** respectively.

Table 11-9: Statistics of Original and Duplicate Samples by Sample Types

Sample Type	Chip Sample		Core Sample	
Statistics	Original Sample	Duplicate Sample	Original Sample	Duplicate Sample
Number of Samples	296	296	450	450
Mean	0.244	0.186	0.547	0.582
Sample Variance	0.732	0.675	5.648	13.17
Standard Deviation	0.856	0.822	2.377	3.629
Minimum	0.01	0.01	0.001	0.001
Maximum	7.71	12	30.7	69.42
Confidence Level (95.0%)	0.098	0.094	0.22	0.336
Correlation	0.59		0.53	

Table 11-10: Statistics of Original and Duplicate Samples by Drilling Periods

Sample Type	2005-2012		2019-2021	
Statistics	Original Sample	Duplicate Sample	Original Sample	Duplicate Sample
Number of Samples	489	489	257	257
Mean	0.539	0.399	0.213	0.475
Sample Variance	5.429	2.704	0.395	18.827
Standard Deviation	2.33	1.645	0.629	4.339
Minimum	0.001	0.001	0.005	0.005
Maximum	30.7	23.7	7.577	69.42
Confidence Level (95.0%)	0.207	0.146	0.077	0.533
Correlation	0.88		0.79	

Table 11-11: HARD Statistics of Duplicate Samples, 2005-2021

Year	No. Duplicate Samples	No. of Samples within HARD Limits							
		< ± 5%		< ± 10%		< ± 20%		< ± 50%	
2005	89	53	60%	55	62%	57	64%	77	87%
2007	80	24	30%	33	41%	42	53%	70	88%
2008	8	4	50%	5	63%	5	63%	6	75%
2012	31	10	32%	11	35%	19	61%	26	84%
2019	35	30	86%	30	86%	30	86%	33	94%
2020	30	14	47%	18	60%	21	70%	26	87%
2021	192	79	41%	102	53%	128	67%	172	90%
Total	**746**	**349**	**47%**	**409**	**55%**	**479**	**64%**	**646**	**87%**

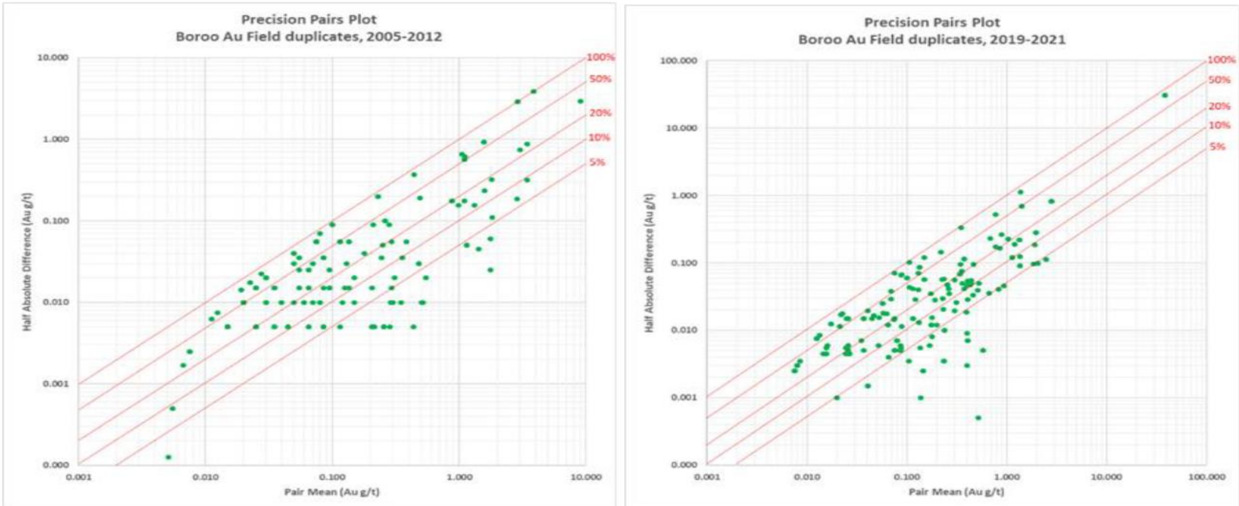


Figure 11-2: Precision Pairs Plots of Duplicates: (a) 2005-2012; (b) 2019-2021

A precision pairs plot compares the pair mean (plotted on the X-axis) to the half absolute difference (HAD, plotted on the Y-axis) (Figure 11-2). The maximum HAD value possible is the pair mean, which equates to a 100% relative difference. Precision lines equivalent to 50%, 20%, 10% and 5% (red lines in Figure 11-2) relative difference lines plotted to allow an assessment of the proportion of samples within a given precision level (Table 11-9) and Figure 11-2 indicates good precision of duplicates.

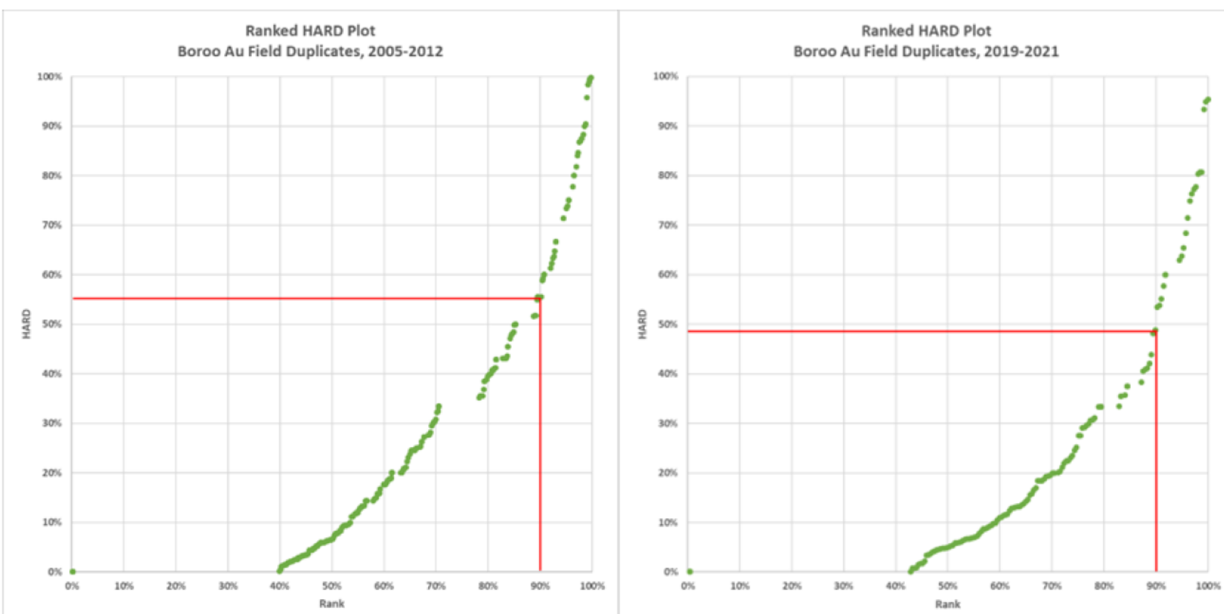


Figure 11-3: Ranked HARD Plots of Field Duplicates: (a) 2005-2012; (b) 2019-2021

The ranked HARD plots of the field duplicates are shown in Figure 11-3 by drilling periods. The ranked HARD plots of repeats show 90% of the samples have a precision that is less than 56% HARD in 2005 to 2012 and less than 46% HARD in 2019 to 2021, indicating that the precision of the analysis is relatively poor. In 2019 to 2021, the precision shows less that the precision of 2005 to 2012 drilling period suggests that the drilling was only performed by the diamond drilling method.

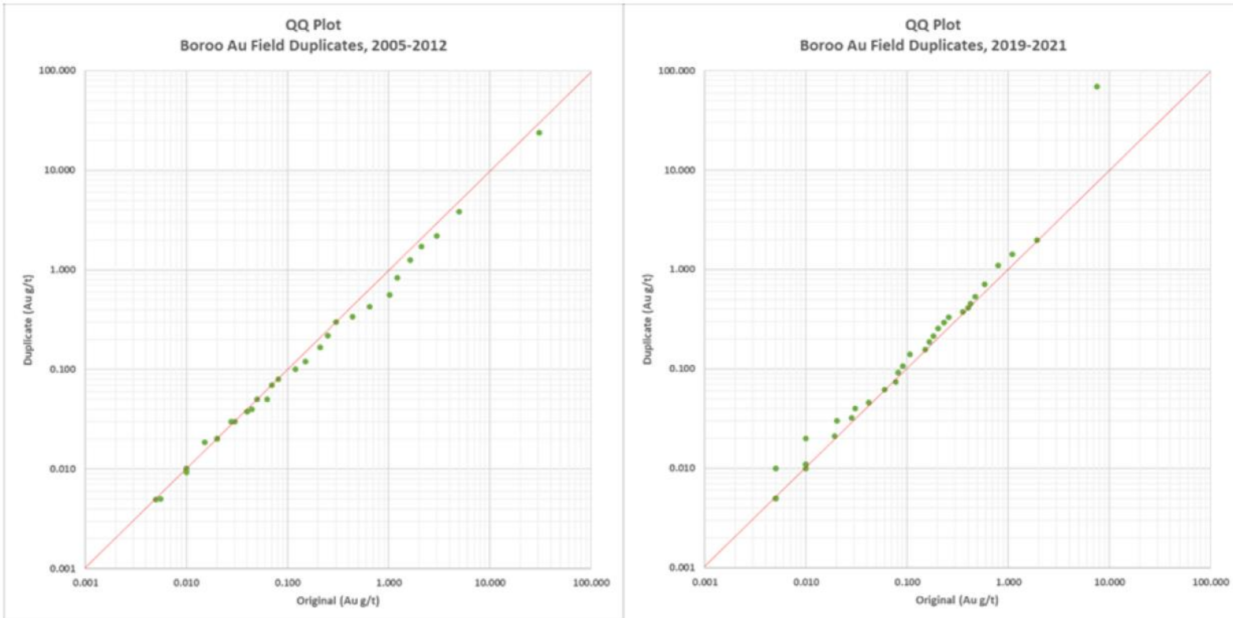


Figure 11-4: QQ Plots for Au Field Duplicates: (a) 2005-2012; (b) 2019-2021

The QQ plots (Figure 11-4) for the repeats and original results appear to be a slight bias for most grade ranges for both 2005 to 2012 and 2019 to 2021 drilling periods. The original sample mean in 2005 to 2012 is 26% higher than the duplicates where original sample mean in 2019 to 2021 is 55% lower than the duplicates. The most likely reason for this bias is poor splitting and sub-sampling practices.

11.5.4 External Laboratory Control

The external laboratory control projects were conducted by ALS laboratory in Bishkek, Kirgizstan for 2004 to 2006 drilling programs. A total of 404 duplicate samples prepared by the same sample preparation and analysis method in SGS laboratory were analyzed in this report. The drilling program of 2006 (200 samples) were controlled by ALS laboratory however information was not available.

Table 11-12: Statistics of External and Primary Laboratory Performances

Statistics	ALS	SGS
Number of samples	404	404
Mean	1.935	2.135
Sample Variance	170.631	305.586
Standard Deviation	13.063	17.481
Minimum	0.025	0.8
Maximum	136.4	230
Confidence Level (95.0%)	1.278	1.71
Correlation	0.64	

The statistics in Table 11-12 show that average grade from the primary laboratory (SGS) was 10% higher than the ALS results. BGC prepared samples above 0.8 g/t results returned from the primary laboratory, however ALS results returned with lower grades.

The HARD statistics of control samples of ALS and SGS laboratories are shown in Table 11-13.

The precision pairs plots of control samples are illustrated in Figure 11-5.

Table 11-13: HARD Statistics of Control Samples, 2004-2005

Year	No. Duplicate Samples	No. of Samples within HARD Limits							
		< ± 5%		< ± 10%		< ± 20%		< ± 50%	
2004	204	57	28%	98	48%	133	65%	182	89%
2005	200	56	28%	102	51%	147	74%	181	91%
Total	404	113	28%	200	50%	280	69%	363	90%

A precision pairs plot compares the pair mean (plotted on the X-axis) to the HARD (plotted on the Y-axis) (Figure 11-5(a)). The maximum HARD value possible is the pair mean, which equates to a 100% relative difference. Precision lines equivalent to 50%, 20%, 10% and 5% (red lines in Figure 11-5(a)) relative difference lines plotted to allow an assessment of the proportion of samples within a given precision level (Table 11-12) and Figure 11-5(a) indicates good precision of control samples.

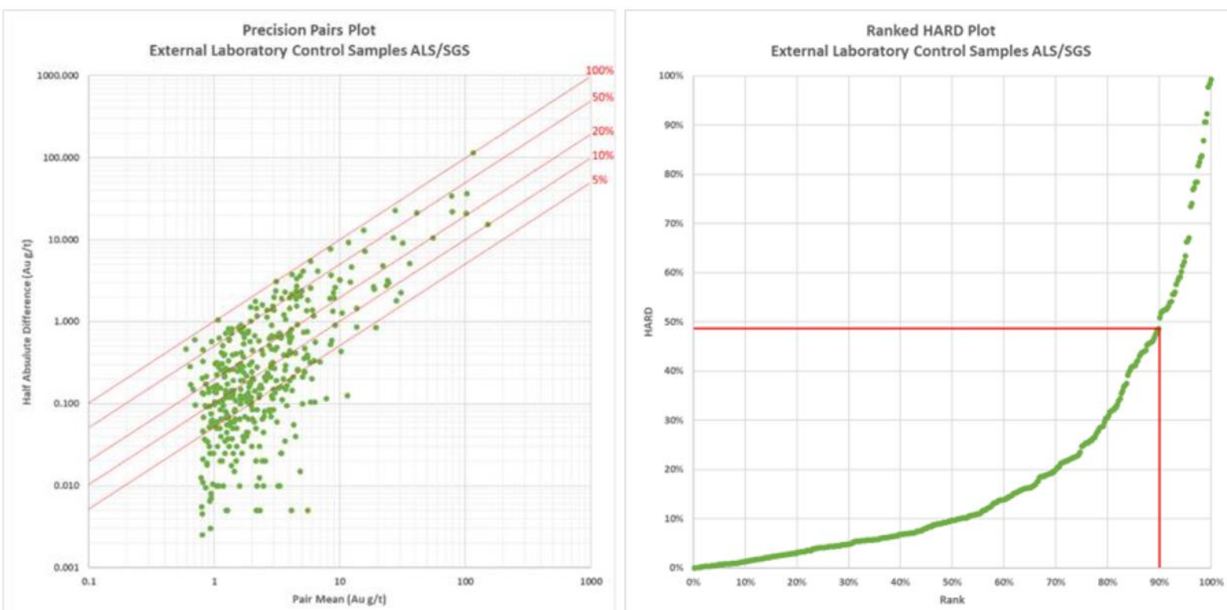


Figure 11-5: External Laboratory Control Performances:

(a) Precision Pairs Plot and (b) Ranked HARD Plot

The ranked HARD plot of the external laboratory performance is shown in Figure 11-5(b). The ranked HARD plots of repeats show 90% of the samples have a precision that is less than 49% HARD, indicating that the precision of the analysis is relatively poor.

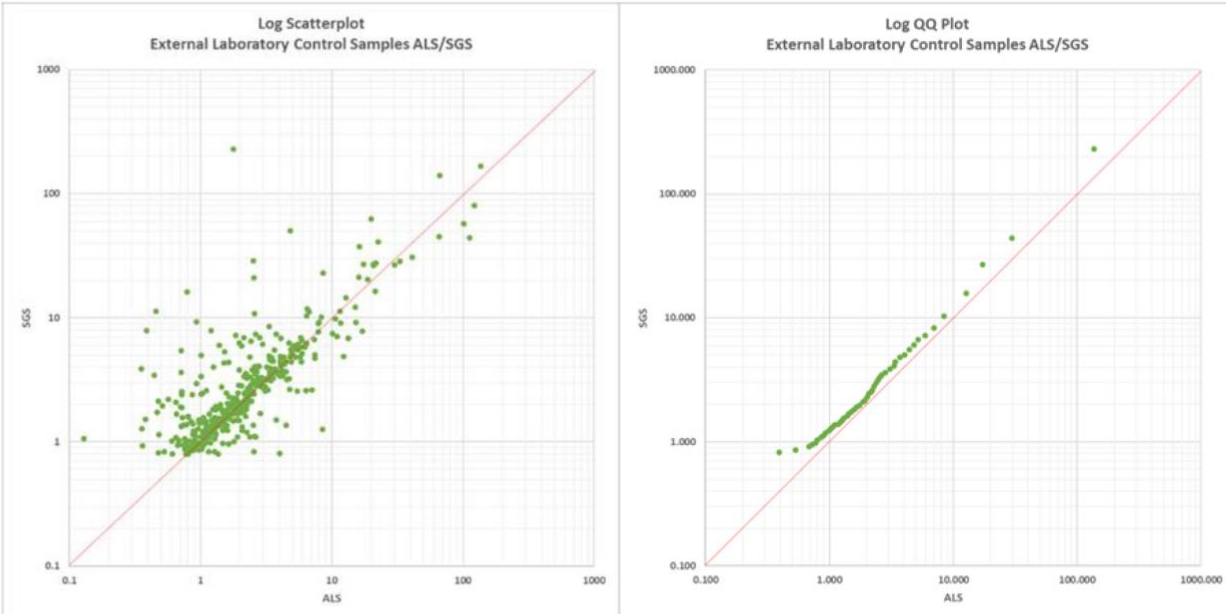


Figure 11-6: External Laboratory Control Performances:

(a) Log Scatterplot and (b) Log QQ Plot

The scatterplot and QQ plot in logarithmic scale of external laboratory control performances were illustrated in Figure 11-6, indicate primary laboratory results has higher grade than the ALS results and it shows systematic bias occurred.

The external laboratory results were not demonstrated in the previous technical report (Centerra, 2009).

11.6 Ulaanbulag QAQC Programs

The QAQC program consisted of the following procedures:

- Standards are added a rate of one in every 20 to 25 samples.
- Duplicate samples rate of in every 15-20 samples.

The number of samples, which were collected and tested at laboratories within the scope of the Ulaanbulag mining exploration and development program, and the number of control samples used for quality control of the samples are shown below in Table 11-14.

A total of 1381 samples (Table 11-14) were analyzed during 2003 to 2010 and 2023.

Table 11-14: Summary of Quality Control Samples (Ulaanbulag)

Year	No. of samples	Quality Control Samples		
		Standard	Blank	Duplicate
2003	561	30	30	
2004	1297	80	55	
2005	862	55	45	

Year	No. of samples	Quality Control Samples		
		Standard	Blank	Duplicate
2006	1054	61	48	308
2007	809	40	40	
2008	933	25	33	
2009	168	6	6	
2010	4648	178	186	144
2023	2889	92	48	96
Total	**13221**	**567**	**491**	**559**

11.6.1 Blank Samples

For blank samples testing, a totally 178 samples prepared and tested in the U.S. MEG laboratory from 2003 to 2006, and the comparison of their test results is shown in figures 11-7 to 11-10.

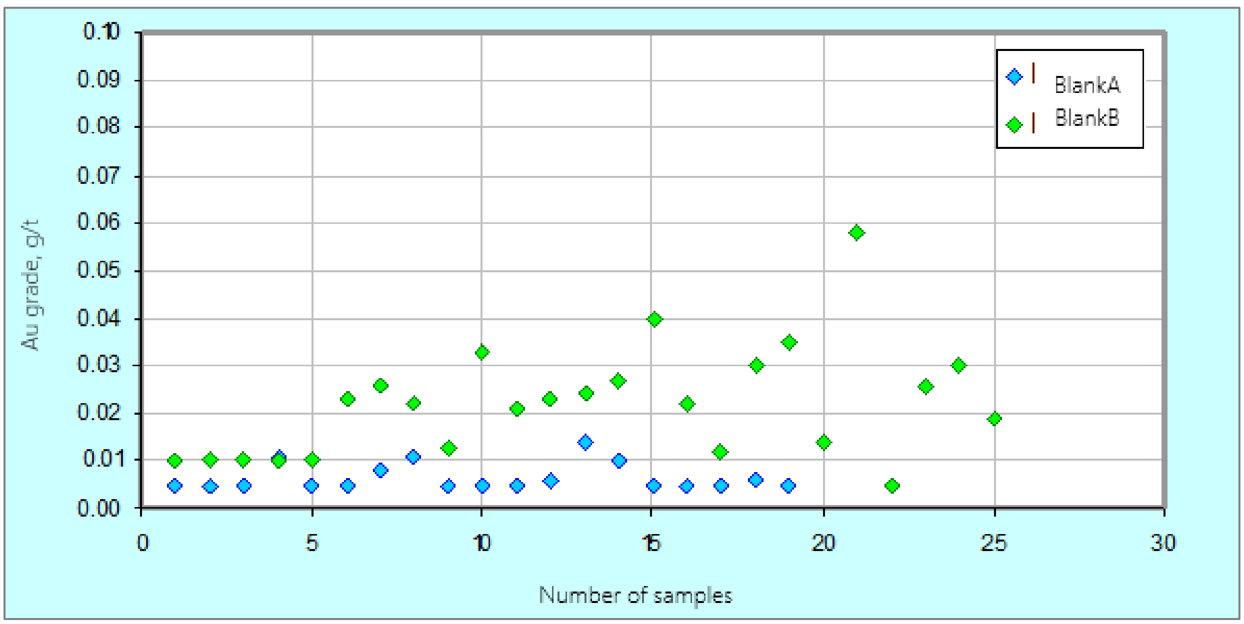


Figure 11-7: Standard sample analysis results (blank samples)

From 2007 to 2008, the West Australian company Geostats conducted blank sample test.

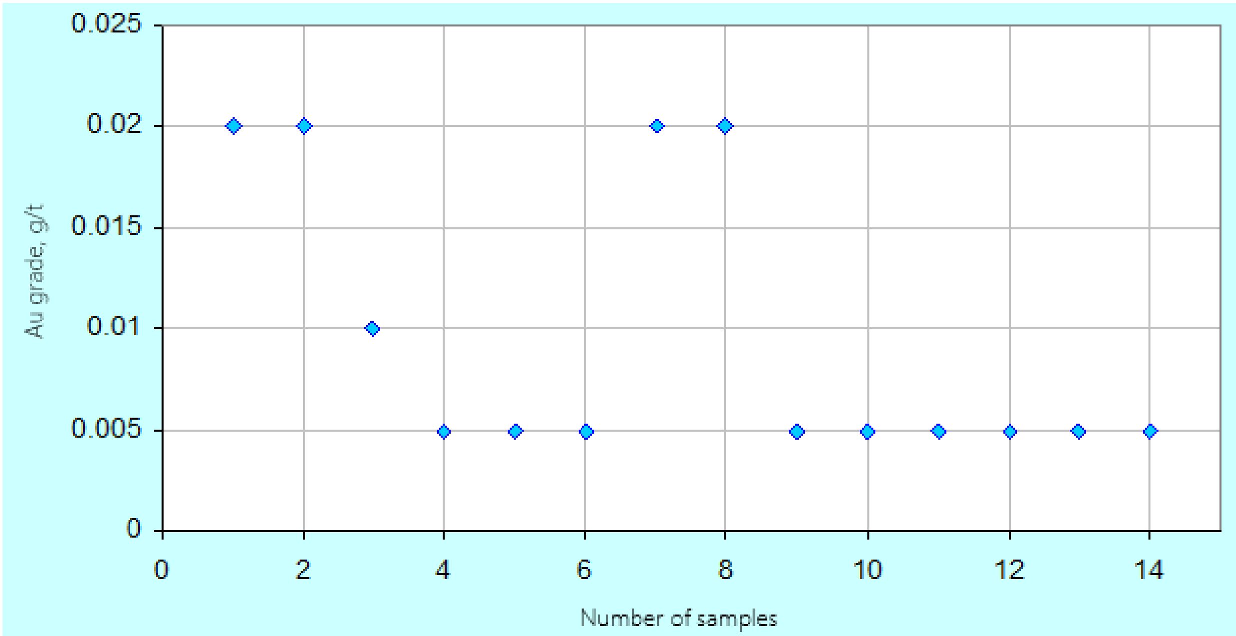
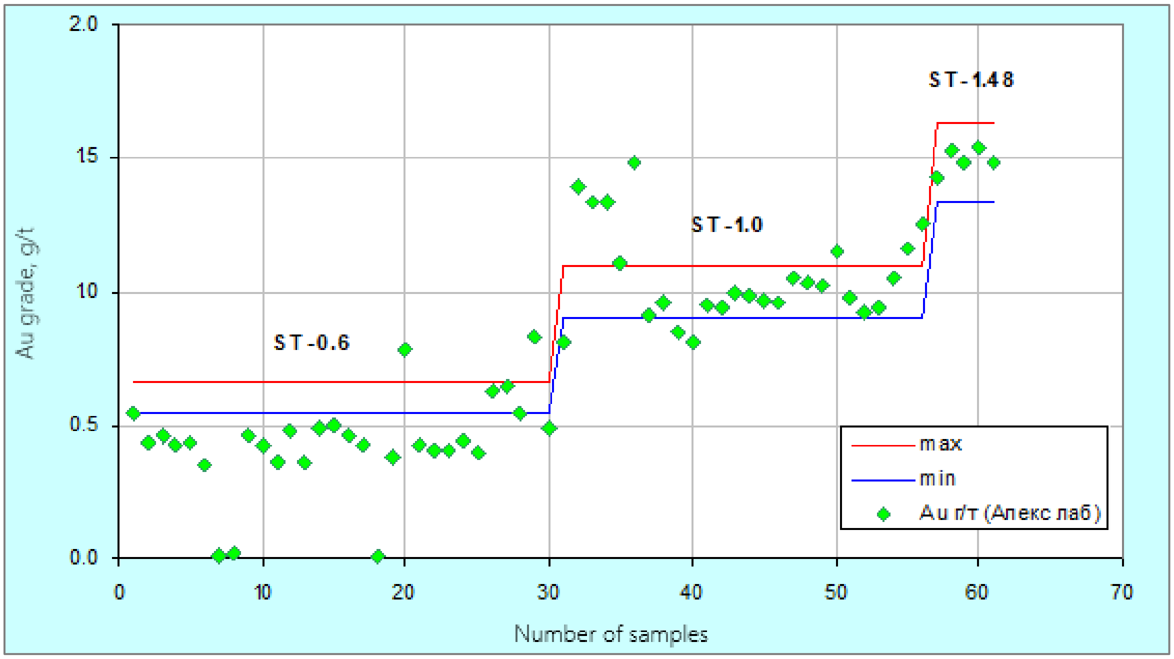


Figure 11-8: Standard sample analysis results (blank samples)

11.6.2 Standard Samples

Standards are added a rate of one in every 20 to 25 samples. Bias in the quality assurance has occurred if the test results of two sequenced standard samples is two times higher or lower from the standard deviation of a given standard; if the test result of any standard is three times higher or lower than the standard deviation then these submissions shall be subject to re-assaying.

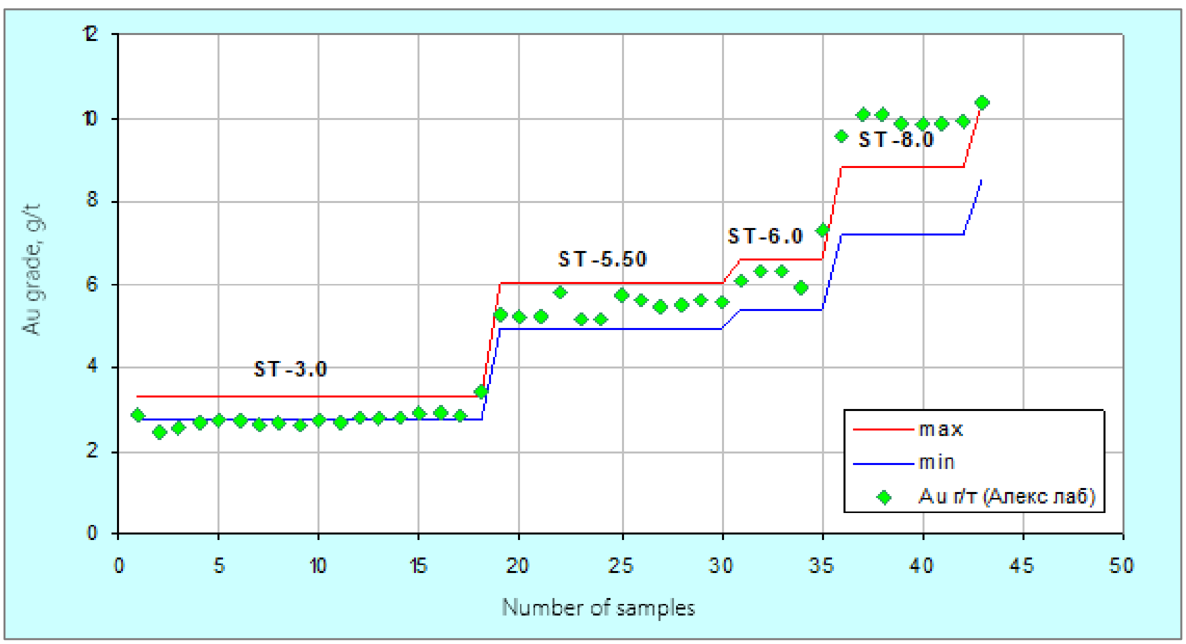
For standard samples testing, number of samples which has gold grade from 0.6 g/t to 27 g/t samples prepared in the U.S. MEG laboratory from 2003 to 2006, and the comparison of their test results is shown in Figure 11-9 to Figure 11-13.



(Au grade 0.6 g/t, 1.0 g/t, 1.48 g/t)

Figure 11-9: Standard sample analysis results

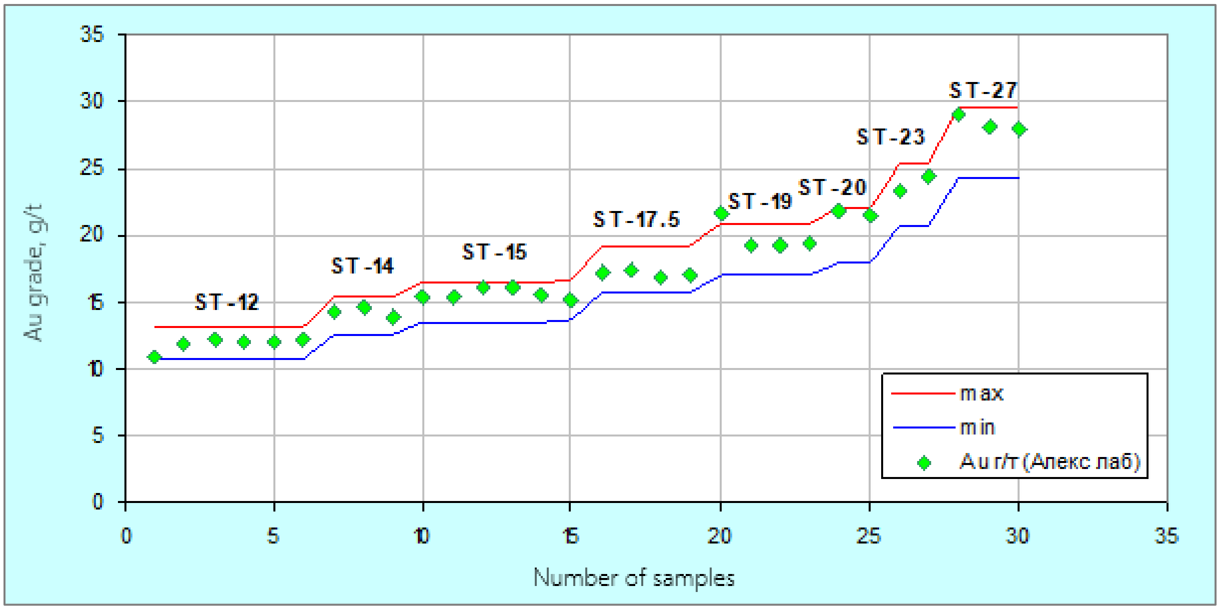
AU grade 0.6 g/t standard sample result is out of acceptance limit.



(Au grade 3.0 g/t, 5.5 g/t, 8.0 g/t)

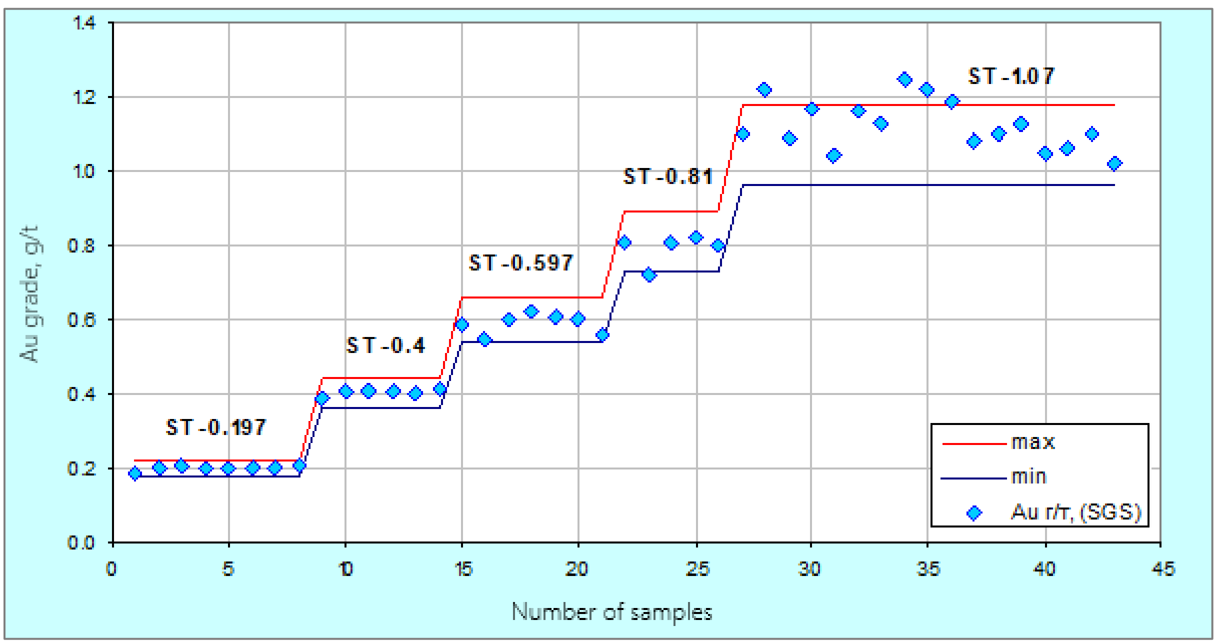
Figure 11-10: Standard sample analysis results

AU grade 3.0 g/t and grade 8.0 g/t standard sample result is out of acceptance limit.



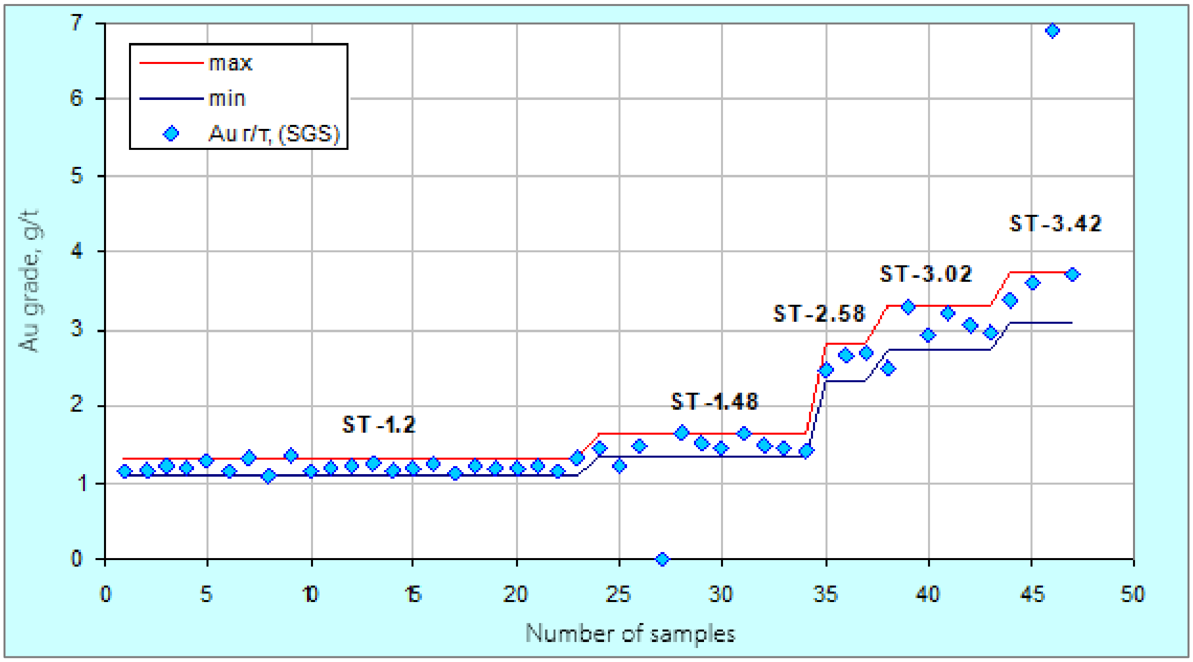
(Au grade 12.0 g/t, 14.0 g/t, 15.0 g/t, 17.5 g/t, 19.0 g/t, 20.0 g/t, 23.0 g/t, 27.0 g/t)

Figure 11-11: Standard sample analysis results



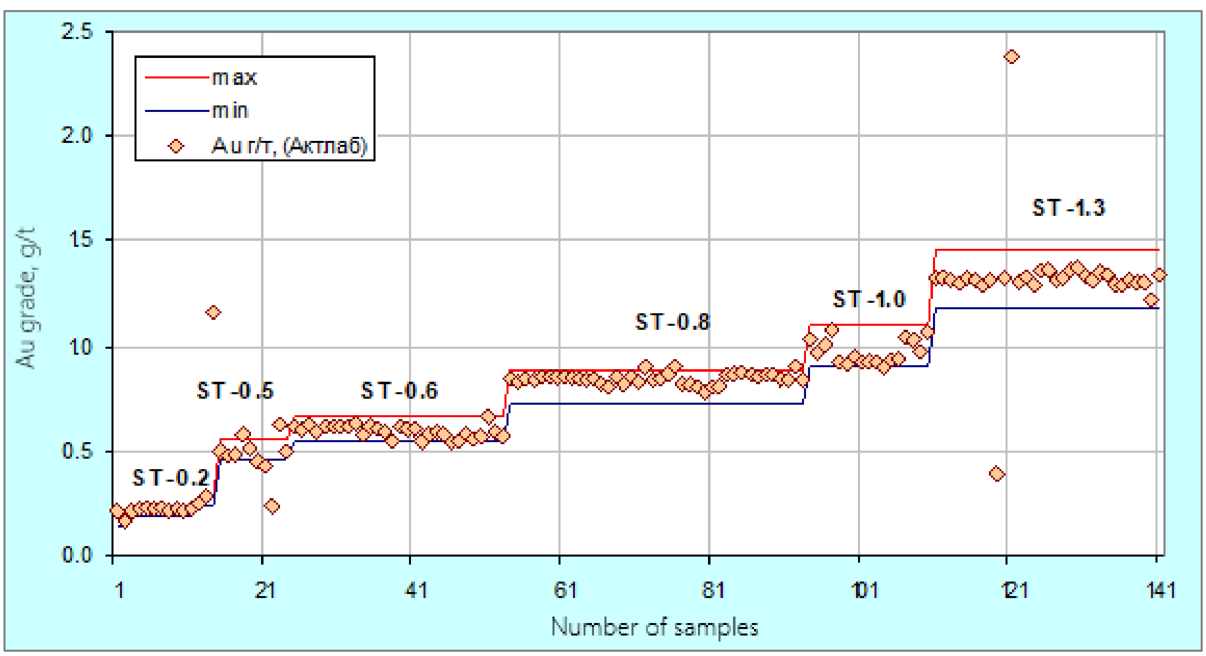
(Au grade 0.197 g/t, 0.4 g/t, 0.597 g/t, 0.81 g/t, 1.07 g/t)

Figure 11-12: Standard sample analysis results

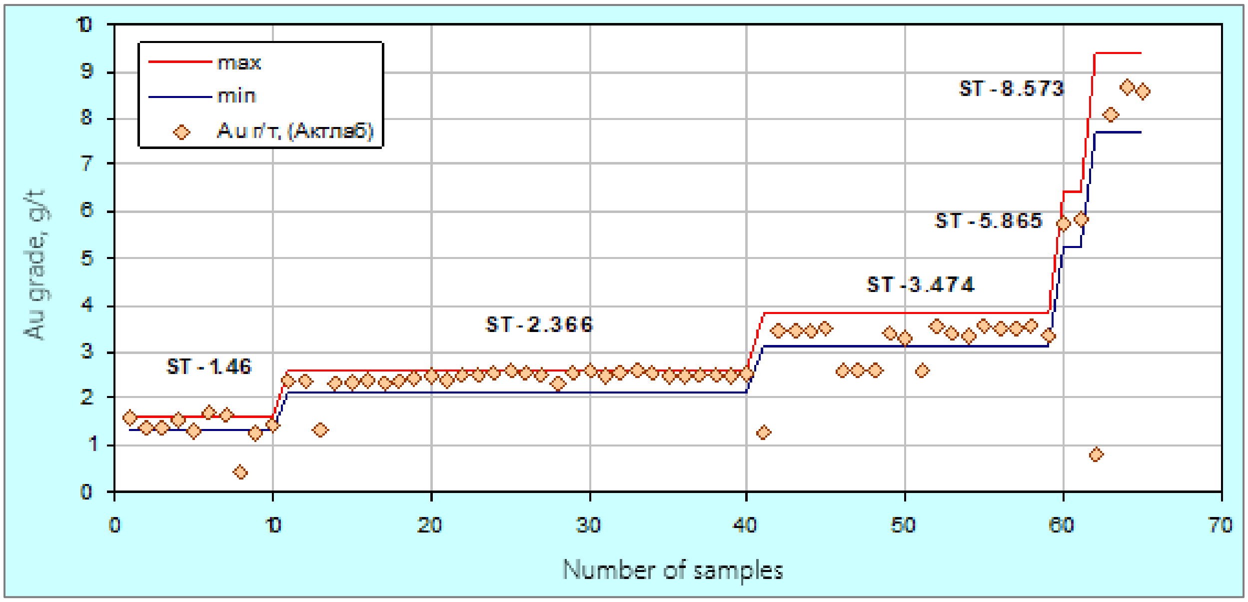


(Au grade 1.2 g/t, 1.48 g/t, 2.58 g/t, 3.02g/t, 3.42 g/t)
Figure 11-13: Standard sample analysis results

In 2009-2010, the Rocklabs company in Auckland, New Zealand, adopted gold-containing grade standards of 0.164-8.573 g/t and compared the results of tests conducted at the Actlabs Laboratory results shown in Figure 11-14 and Figure 11-15.



(Au grade 0.2 g/t, 0.5 g/t, 0.6 g/t, 0.8g/t, 1.0 g/t, 1.3 g/t)
Figure 11-14: Standard sample analysis results



(Au grade 1.46 g/t, 2.3 g/t, 3.474 g/t, 5.865 g/t, 8.573 g/t)

Figure 11-15: Standard sample analysis results

11.6.3 Duplicate Samples

A total of 443 duplicate samples were assayed which is 4.3% of total samples of Ulaanbulag deposit. The samples were assayed with one duplicate for every 15 to 20 samples.

The samples were tested in the ASA laboratory from 2003-2006, and the comparison of the results of the duplicate samples and original test results is shown in **Table 11-15** and **Figure 11-16.**

Table 11-15: Comparison of Original and Duplicate Samples

Laboratory	Original Samples > Duplicate samples		Original Samples < Duplicate samples		Original Samples = Duplicate samples	
ASA	46	33.10%	56	40.30%	37	26.62%
SGS	29	32.95%	23	26.14%	36	40.91%
Actlabs	101	44.70%	96	42.48%	29	12.83%

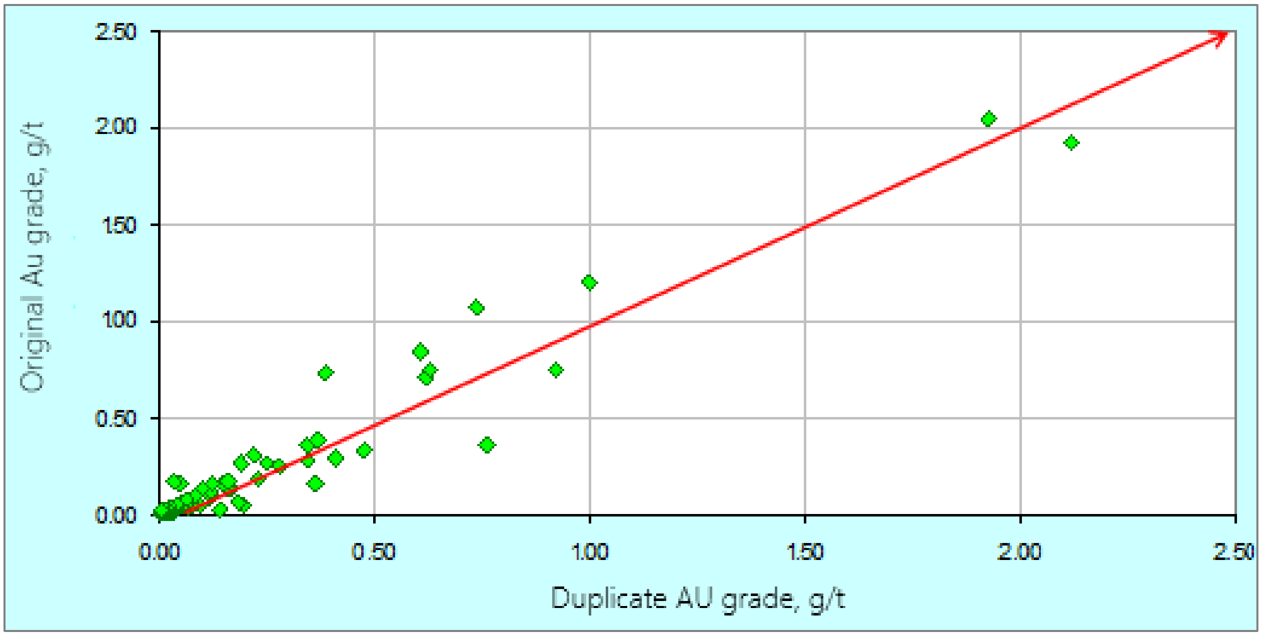


Figure 11-16: QQ Plots for Au Duplicates : 2003-2006

Table 11-16: HARD Statistics of Duplicate Samples

Laboratory	No. of Samples within HARD Limits					
	5%	10%	20%	25%	50%	>±50%
ASA	40	10	20	11	27	31
	29%	7%	14%	8%	19%	22%
SGS	37	-	11	2	13	25
	42%	0%	13%	2%	15%	28%
Actlabs	41	13	29	15	56	72
	18%	6%	13%	7%	25%	32%

The samples were tested in the SGS laboratory from 2007-2008, and the comparison of the results of the duplicate samples and original samples results is shown in **Figure 11-17**.

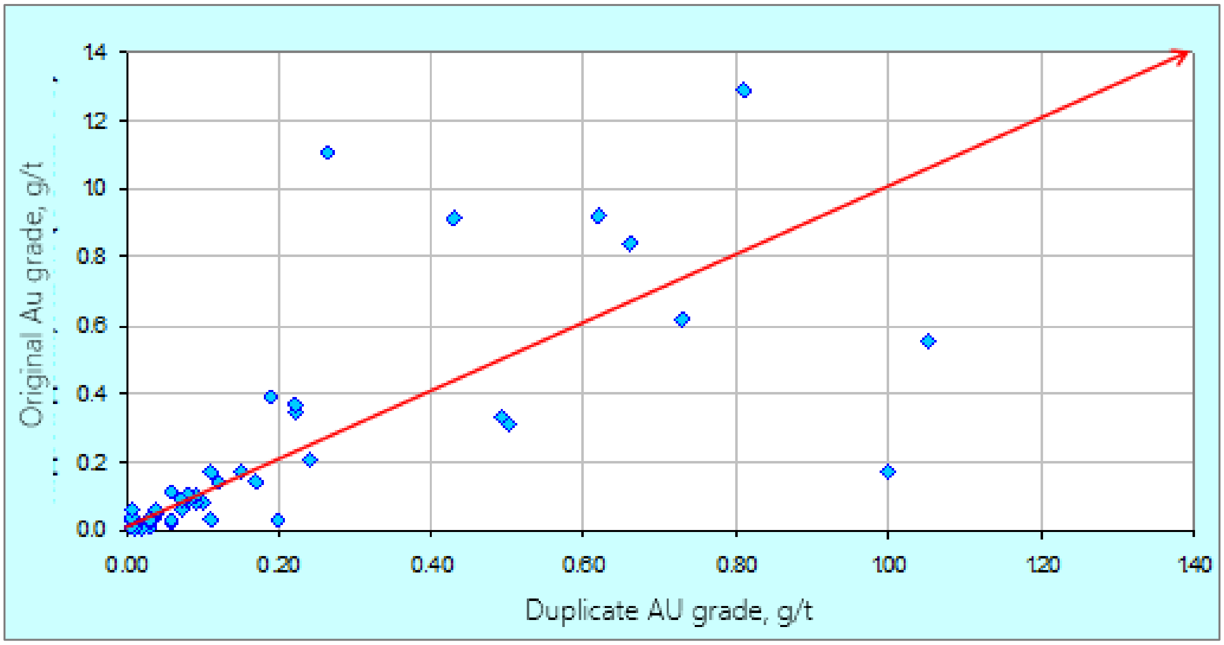


Figure 11-17: QQ Plots for Au Duplicates : 2007-2008

The samples were tested in the Actlabs laboratory from 2009-2010, and the comparison of the results of the duplicate samples and original samples results is shown Figure 11-18.

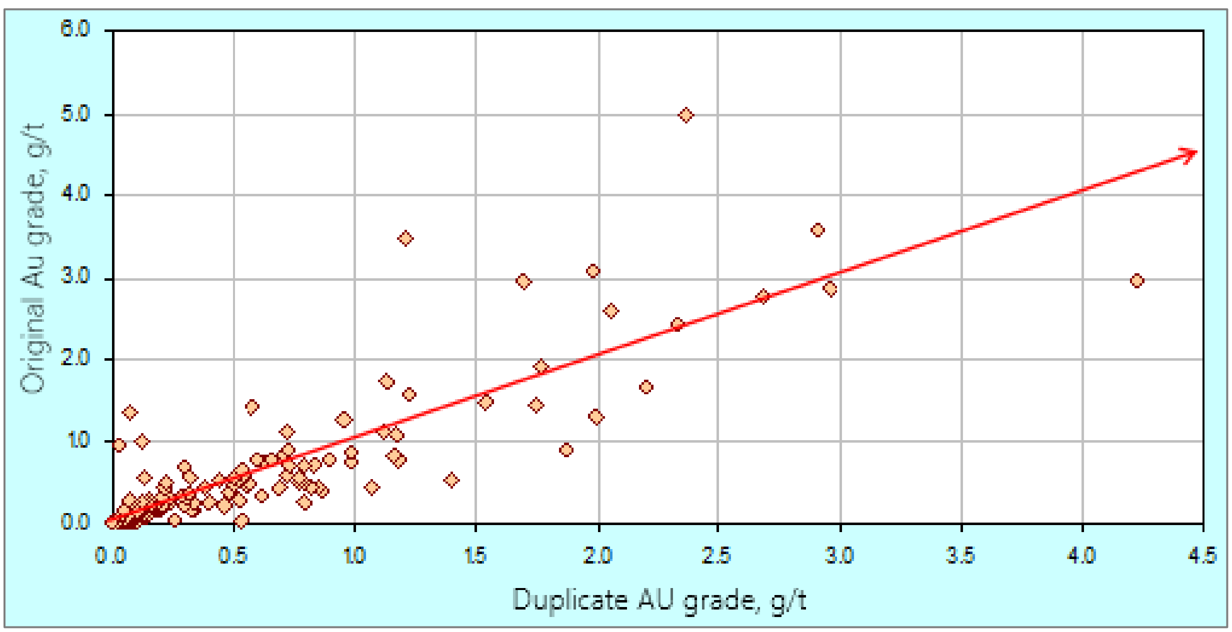


Figure 11-18: QQ Plots for Au Duplicates : 2009-2010

11.6.4 External Laboratory Control

A total of 308 samples were selected between 2003-2006, to be sent to Actlab and SGS laboratory. No sample bias is observed between Actlab and SGS (Figure 11-19).

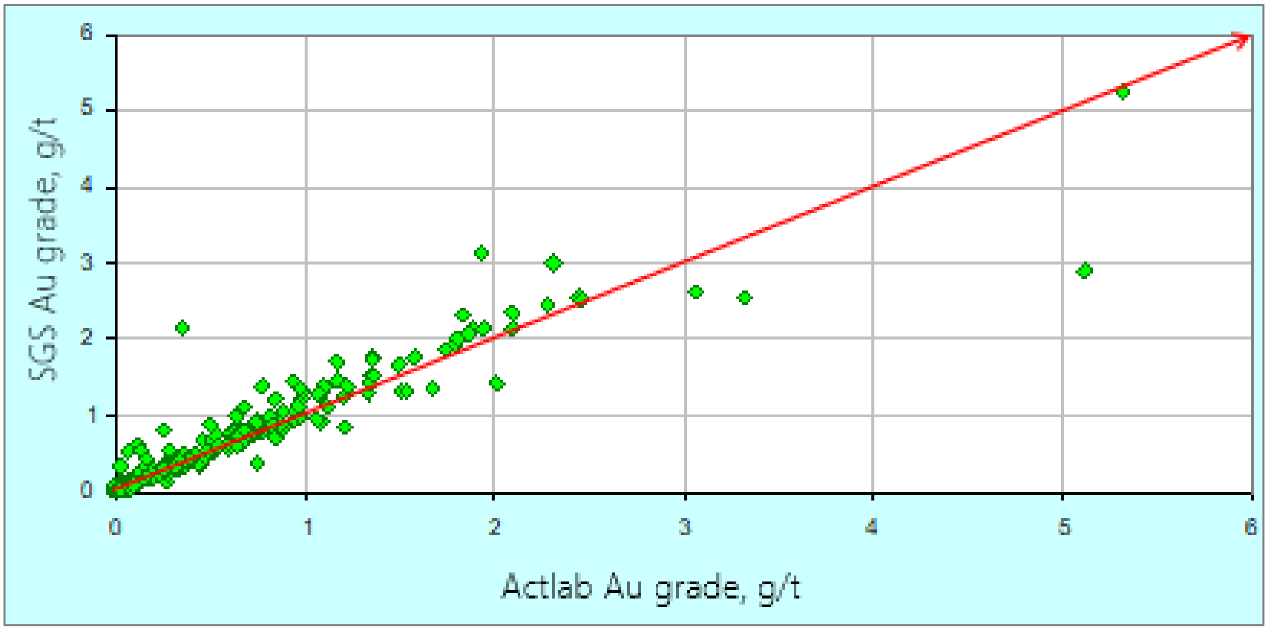


Figure 11-19: Duplicate samples results by laboratory (2003-2006)

A total of 144 samples were selected between 2007-2010, to be sent to Actlab and SGS laboratory. No sample bias is observed between Actlab and SGS (Figure 11-20).

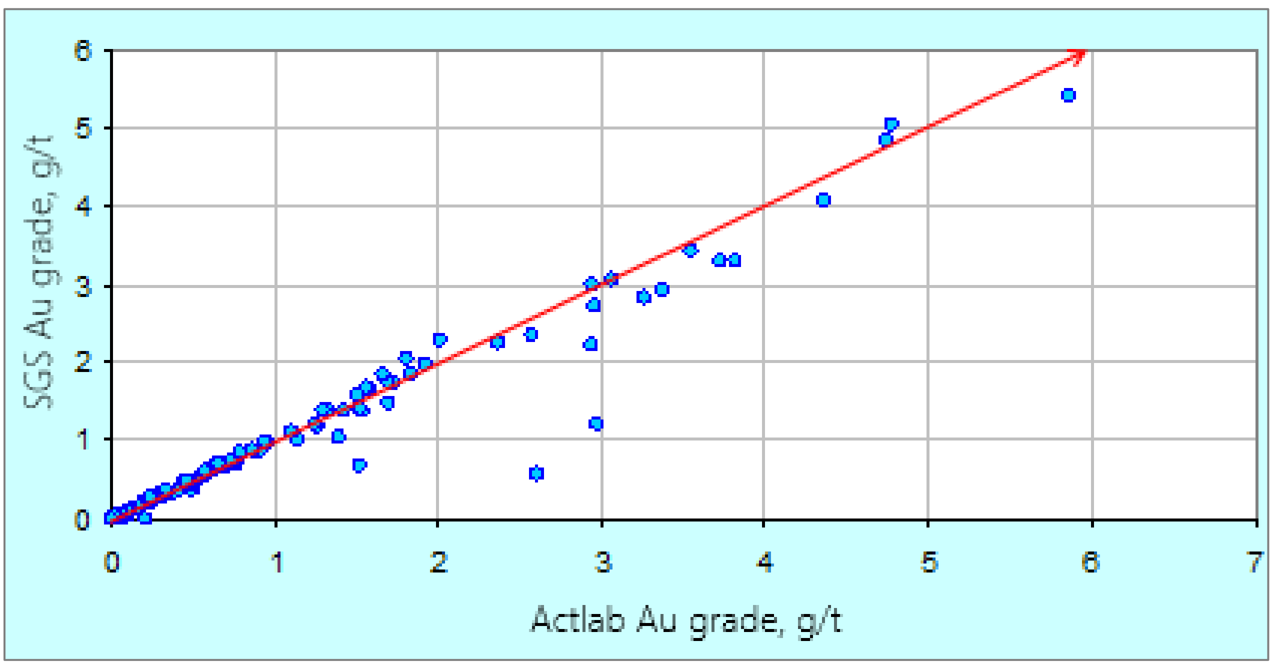


Figure 11-20: Duplicate samples result by laboratory

This shows that in terms of average gold grade, Actlab's gold grade is set slightly higher than that the SGS laboratory. But because the grade gap is low and at an acceptable rate, the laboratory test results are considered to be meeting quality requirements.

12.0 DATA VERIFICATION

During the AGR 1999 feasibility study of the Boroo Project, the information from surface trenches and drillholes was entered into a computerized database. Drill collar coordinates, assay results and information on lithology, alteration and mineralization were recorded in the mine database that is used for early Reserve and Resource estimation purposes. This data was validated at different times, initially by MRT for BGC in early 1998, later by BGC, SRK and subsequently by CGM who performed a rigorous database evaluation in 2002 to 2003.

In general, the database underlying the current Boroo MRE, assembled prior to 2002, has been verified several times in the past and while a few issues with some of the historical holes, core loss and minor clerical errors exist, the Centerra Qualified Person felt that this would not have a material effect on the outcome of the 2009 Reserve and Resource estimates, particularly as much of this historical exploration data applied has now been mined out.

Since 2003, a substantial amount of additional drill data has been added by BGC where checks to the database were performed regularly under the supervision of the BGC Chief Geologist, who was responsible for its upkeep and reliability.

During this same time, the Centerra Qualified Person had also on several occasions while at the Boroo site undertaken verification of both the exploration and production datasets by performing independent reconciliation and reviews between the exploration drilling results obtained from several years of mine and mill production.

Game Mine has completed data verification of the drilling programs until end of the 2023 and believed almost twenty years of reliable operational performance of the Boroo Project relative to the Reserve model developed from the exploration dataset had provided the most significant confidence in the reliability of both the overall exploration and production data used in the technical report. The Reserve and Resource model constructed from this exploration dataset can be considered reliable.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

This section summarizes all the relevant testwork performed on the Boroo Gold and Ulaanbulag deposit. The Project was previously operated by Centerra Gold Inc. The current Boroo ore processing flowsheet is the result of a number of past metallurgical test programs.

The current Boroo Mill flowsheet includes crushing, grinding, gravity concentration, leaching and carbon-in-pulp (CIP), cyanide detoxification and arsenic precipitation to produce gold doré.

13.1 Boroo

13.1.1 Pre-1999 Historical Testwork

Historical testing (pre-1999) included the following major programs:

- Pilot scale metallurgical supervised by DDR-MPR technical personnel with bench scale tests were conducted in Freiberg, East Germany and pilot scale testing by Irgiredmet, Irkutsk, Russia, in 1987 (USSR All Union Export and Import Association (Tsvetmetpromexport), 1987).
- McClelland Laboratories Inc. of Sparks, Nevada in 1991 recommended to Morrison Knudsen Company Inc. bench scale column leach tests, flotation and cyanidation testing.
- A comprehensive program in 1997 at AMMTEC Limited, Balcatta, Australia including grinding, leaching, column leaching and gravity concentration testing on diamond drill core and reverse circulation drilling chip samples. The program established base recoveries of 95% to 90% for oxide and upper transitional ores and 85% for lower transitional ores.
- Comminution testing was undertaken by Orway Mineral Consultants of Mt Pleasant, Western Australia in 1999. Bond ball mill work indices of 12.2 and 13.0 for oxidized and fresh “granite”, respectively, were determined while the figures for the Haara sediments (hornfels) were much higher in the range of 22.3 to 24.6. Several options for the design of a comminution circuit were evaluated.

Additional bottle roll testwork at a grind of 80% passing 75 microns on ore within the planned open pits that had not been adequately covered earlier was undertaken in 2002 (Chapman, 2003), using ore from the lower transitional or fresh material. Again, the results in the fresh ore were very variable, averaging 75% gold recovery, but ranging from 57% to 90%.

Early metallurgical data for the primary or fresh ore at Boroo, and the expected average gold recovery at 77% was based on limited testing.

13.1.2 2006 Column leach tests

Column leaching conducted by AMMTEC in Australia, Kappes-Cassiday & Associates (KCA) USA in 2006. The ore types are identified as Oxide, Transition, and Fresh with the names indicating the degree of sulphide oxidation.

13.1.2.1 AMMTEC testwork

The samples were identified as low grade (pit 2 & 5), oxide (pit 3), and sulphide (pit 2). The samples were tested in columns at sizes designated at <25mm. The ore to each column was pre-alkalized with 0.5 kg/t of lime and solution with 0.5 g/l of sodium cyanide. The columns were operated up to 29 days with solution to ore rations up to 4.5:1.

The cyanide strength for the leaching was maintained at 0.5 gram per litter of sodium cyanide. The following **Table 13-1** summarizes the results of the column tests.

Table 13-1: Summary of AMMTEC Column leaching studies

AMMTEC Test No.	Sample ID	Calc Head grade Au, g/t	Tail Assay AU, g/t	Gold Recovery, %	P80 size, mm	Days of leach	Kg/t NaCN
<25mm							
HS11864B	Low grade	0.786	0.064	91.4	23	29	0.74
HS12079	Oxide	2.05	0.6	70.3	15.6	29	0.85
HS12080	Sulfide	0.543	0.19	64.9	18.5	29	0.59

13.1.2.2 Kapper, Cassiday & Associates testwork

Three samples of ore grade material were treated in the later part of 2005. The samples were tested in column at sizes designated at As-received and minus 25 mm. The columns were operated up to 78 days with solution to ore ratios up to 4.5:1.

The following **Table 13-2** from the KCA report summarizes the results if the column tests.

Table 13-2: Summary of KCA Column leaching test results

KCA Test No	Sample ID	Calc Head Au, g/t	Tail assay Au, g/t	Au Recovery, %	P80 size, mm	Days of leach	Kg/t NaCN
As-Received							
34516	Oxide	1.57	0.19	88	40	78	0.2
34519	Transitional	1.13	0.29	74	45	78	0.1
34522	Fresh	0.71	0.4	44	70	78	0.1
Crushed to minus 25mm							
34525	Oxide	2.46	0.24	90	17	73	0.77
34528	Transitional	1.37	0.48	65	15	73	0.67
34533	Fresh	0.77	0.36	53	18	73	0.37

The only anomaly in the results was the lower extraction of gold from the transitional material when crushed to minus 25mm.

13.1.3 2007 BIOX process

In late 2007, a composite sample of drill core from Pit 6 and Pit 3 representing potential refractory ore, was sent to SGS Lakefield Africa for flotation. The sample assayed 2.26 g/t Au, 0.71% As, 1.00% S and 2.34% Fe. The purpose of the program was to generate flotation concentrate for refractory processing evaluation. The concentrate produced assayed 28.2 g/t Au, 10.9% As, and 13.7% S with a mass recovery of 7.1% and 92.9% Au recovery and 96.6% S recovery. The concentrate was produced for sulphide oxidation testing with the BIOX® process.

13.1.4 2021 Metallurgical Testing Program

The 2021 metallurgical testing program was completed at Base Metallurgy (BaseMet) Laboratories in Kamloops, BC, Canada. The scope of the program included:

- Head assays.
- Comminution testing including Bond ball mill work index, and Steve Morrell Comminution (SMC) tests.
- Bulk mineralogy.
- Extended gravity recoverable gold (E-GRG) testing.
- Flotation testing.
- Leach tests.
- Solids-liquids separation tests

Eight composite samples from nine drill holes were selected for the metallurgical testing program. The samples were selected to represent the planned open pit mining areas of the Boroo reserves, Pits 2, 3 and 4/5.

Table 13-3: Boroo PFS Metallurgical Samples Description

Sample	Pit	Oxidation Type
Composite 1	2	Transition
Composite 2	2	Sulphide
Composite 3	3	Transition
Composite 4	3	Sulphide
Composite 5	4	Sulphide
Composite 6	4	Transition
Composite 7	3	Oxide
Composite 8	4	Sulphide

Sample assays are shown in Table 13-4. Sulphur occurs primarily as sulphide sulphur (S2-) in all samples, including the oxide sample (Comp 7). Elevated As levels are found in all samples, including the oxide sample (Comp 7) and are highest in Comp 2 (Pit 2 fresh).

Table 13-4: Boroo PFS Sample Assays

Sample	Analyte					
	Au (g/t)	Ag (g/t)	ST (%)	SO42- (%)	S2- (%)	As (%)
Composite 1	0.92	0.5	0.44	0.03	0.40	0.18
Composite 2	0.47	0.4	0.51	<0.01	0.51	0.86
Composite 3	0.96	0.7	0.53	0.01	0.52	0.40
Composite 4	1.14	0.6	1.10	<0.01	1.10	0.57
Composite 5	2.12	0.6	0.63	<0.01	0.63	0.51
Composite 6	1.57	0.8	1.14	<0.01	1.14	0.74
Composite 7	1.05	0.7	0.44	0.02	0.42	0.49
Composite 8	1.86	0.6	0.78	0.02	0.75	0.45

All samples were submitted for bulk mineral analysis (BMA) using QEMSCAN (quantitative evaluation of minerals by scanning electron microscopy). BMA determines major mineral species and sulphide mineral speciation. The results are presented in **Table 13-5**.

Table 13-5: Boroo PFS Samples Bulk Mineralogy Analysis

Mineral Abundance (wt%)	Comp. 1	Comp. 2	Comp. 3	Comp. 4	Comp. 5	Comp. 6	Comp. 7	Comp. 8
Pyrite	0.67	0.25	1.01	1.78	0.77	1.37	0.76	0.59
Arsenopyrite	0.29	2.55	0.98	1.46	1.25	2.17	0.38	1.38
Other Sulphides	0.01	0.03	0.01	0.03	0.04	0.07	0.02	0.10
Quartz	41.3	52.0	44.8	55.2	34.2	57.1	55.1	50.9
Plagioclase	30.5	19.2	10.5	5.18	0.44	0.41	0.60	0.72
K-Feldspar	22.6	20.1	12.3	6.31	1.41	0.68	0.57	2.43
Sericite/Muscovite	1.55	3.11	16.7	19.1	39.5	20.6	25.7	22.1
Chlorite	0.50	0.45	1.02	0.25	1.82	0.35	2.43	1.04
Other Silicates	1.29	1.28	3.04	2.06	4.38	2.90	3.46	5.84
Fe-Oxides	0.16	0.04	0.35	0.06	0.42	0.07	1.38	0.21
Other Oxides	0.08	0.05	0.21	0.16	0.20	0.24	0.29	0.05
Calcite	0.32	0.20	0.65	0.33	0.28	0.49	1.51	0.35
Dolomite	0.02	0.01	2.07	3.38	1.12	2.82	1.08	1.21
Ankerite	0.51	0.57	5.60	4.69	9.51	10.3	6.31	12.3
Siderite	0.02	0.00	0.43	0.00	4.08	0.03	0.20	0.33
Apatite	0.08	0.06	0.28	0.03	0.52	0.33	0.12	0.37
Other	0.09	0.08	0.10	0.02	0.08	0.06	0.06	0.05
Total	100	100	100	100	100	100	100	100

13.1.4.1 Comminution Testing

Comminution test samples were selected from half HQ and PQ drill core received. Testing on the eight individual variability composites included BWI (Bond ball mill work index) and SMC (Steve Morrell Comminution) (completed at -31.5 / +26.5mm) tests. Results are summarized in **Table 13-6.**

Table 13-6: Boroo PFS Comminution Tests Results

Description	JK Parameter			BWI (metric)
	Rel. Density	A x b	DWI (kWh/m3)	
Average	2.75	72.2	72.2	15.4
Std. Dev.	0.11	39.1	3.4	1.4
Rel. Std. Dev.	4.0	46.2	48.5	8.9
75th Percentile	2.83	49.1	6.00	16.3
Median	2.72	65.4	4.21	15.6
Maximum	2.93	32.3	8.74	17.3

13.1.4.2 Gravity Separation

Gravity separation testwork program was completed on all samples to investigate the efficacy of gravity-based gold separation. The extended gravity recoverable gold procedure (E-GRG) was used for the testing.

During sample preparation, a single 20-kg charge was created for each composite. Once split, the material was stage crushed to 100% passing 1.7 mm producing an 80% passing size (k80) of approximately 1.2 mm. This sample was utilized for E-GRG testing. The E-GRG test was conducted by passing the entire crushed material through a Knelson MD-3 concentrator at a force of 60-Gs. The concentrate was retained and sized for assay; the tailings were subsampled for sizing. The tailings were grounded in a laboratory rod mill and repassed (Pass 2) at grind target k80 of 250 µm, the concentrate and tailings were sample as per the initial pass before regrinding (k80 of 75 µm) the tailings and repassing (Pass 3). The final tailings are sampled, screened and assayed by size. All assaying included Au by fire assay with concentrate fractions assayed to extinction.

Table 13-7: Boroo E-GRG Test Results

Composite	Cum. Mass Recovery (%)	Combined Conc. Assay (g/t Au)	Cum. Recovery (% Au)
1-Transition	1.39	77.1	67.1
2-Sulphide	3.32	22.6	78.1
3-Transition	1.37	117	72.7
4-Sulphide	1.66	48.0	42.5
5-Sulphide	1.71	65.0	45.9
6-Transition	1.68	47.0	40.3
7-Oxide	1.62	17.0	24.1
8-Sulphide	1.90	121	64.4

The E-GRG test results are shown in **Table 13-7**. The results indicate highly variable gravity recoveries with a range spanning from 24% to 78%, with an average gold recovery of 54%. The GRG in the samples were considered highly amenable to gravity concentration, with most of recoverable gold was captured in the combined concentrate except for Composite 7 (oxide sample) which only 24.1% of gold in total was captured by gravity concentration. The Boroo plant includes gravity concentration within the current flowsheet. The results show that the possibility to produce gold in doré for as well as in flotation concentrate will be possible.

13.1.4.3 Flotation testing

Initial tests were based on the 2007 SGS flotation tests. Initial tests included gravity concentration prior to rougher flotation targeting a mass recovery of 0.05% to gravity concentrate. Test results are presented in **Table 13-8**. Observations on the results include:

Table 13-8: Boroo Initial Flotation Tests (With Gravity Concentration)

Composite	Gravity Gold Conc.		Rougher Concentrate					Flotation Tail Grade Au (g/t)	Calc. Head Grade Au (g/t)	Overall Recovery Au (%)
					Au					
	(g/t)	(% rec.)	Mass (%)	As (%)	(g/t)	(Rec., %)	S (%)			
1	749	25.7	5.9	2.4	18.1	72.4	7.5	0.03	1.47	98.1
2	196	12.5	9.8	9.2	6.3	86.2	5.3	0.01	0.71	98.7
3	394	48.8	3.7	6.5	22.4	41.9	8.5	0.19	1.98	90.8
4	250	3.7	13.0	4.0	19.5	94.3	7.1	0.06	2.69	98.1
5	213	3.2	7.9	6.3	27.6	91.8	6.8	0.13	2.38	95.0
6	117	2.6	11.8	6.7	16.0	93.9	9.0	0.08	2.01	96.5
7	60	5.0	6.3	3.7	9.1	47.4	6.4	0.62	1.22	52.4
8	167	11.1	6.1	6.6	27.1	86.9	8.4	0.04	1.89	98.0

Overall high combined gravity plus flotation gold recoveries, ranging from 90.7% to 98.7%. Comp 7 is an oxide sample and will not respond to sulphide flotation, so the poor results are unexpected.

A second set of rougher flotation tests was completed without gravity concentration prior to flotation. The primary objective was to determine what the increase in concentrate gold grades is without removal of free gold prior to flotation. The results are shown in **Table 13-9**.

Table 13-9: Boroo Flotation Tests (Without Gravity Concentration)

Composite	Rougher Concentrate				Flotation Tail Au (g/t)	Calc. Head Grade Au (g/t)	Overall Recovery Au (%)
	Mass (%)	As (%)	Au (g/t)	S (%)			
1	4.1	3.53	30.9	9.5	0.04	1.32	97.1
2	6.0	13.7	9.08	8.2	0.04	0.59	93.6
3	6.1	5.91	20.5	7.7	0.14	1.39	90.5
4	13.4	3.89	13.9	7.3	0.05	1.91	97.7
5	10.4	4.83	20.4	5.2	0.12	2.24	95.2
6	11.2	7.44	16.8	9.0	0.04	1.92	98.2
7	11.9	2.36	6.35	4.0	0.62	1.30	58.1
8	10.3	5.08	22.4	7.0	0.07	2.36	97.3

The results of those 2 different flotation of with and without gravitation are not significantly different from the flotation tests with gravity concentration.

Overall gold recoveries, ranging from 90.7% to 98.7%, averaging 91.0% (95.7% Au excluding Composite 7 results). Comp 7 produced low recoveries.

13.1.4.4 Leach Tests

Leach tests were run on the eight composites to provide a baseline case for processing the reserves through the existing plant without modifications. The tests were run under the following conditions:

- Gravity concentration prior to leach targeting 0.05% mass recovery.
- Grind to k80 = 75 µm.
- 40% solids pulp density by weight.
- 36-hour test duration.
- Cyanide maintained to 0.25 g/L NaCN
- Air addition to maintain dissolved oxygen in slurry.

The results are shown in **Table 13-10**. Gravity recoveries were generally low, ranging from 1.4% to 11.7% for gold. These are all less than those achieved for the same samples prior to flotation. The combined gravity/leach recoveries for gold varied, with Composite 1 having the highest combined recoveries at 81.1% while Composite 6 results had the lowest combined recoveries of 5.1%. Sodium cyanide (NaCN) and lime (CaO) consumption ranged between 0.32 kg/t to 0.41 kg/t and 1.03 kg/t to 1.46 kg/t respectively.

Table 13-10: Boroo Leach Test Results

Samples	Assay Head (g/t)				Calc. Head Grade Au (g/t)	Leach Residue Grade Au (g/t)	Au Recovery (%)	
	NaCN	CaO	Au	Ag			Grav. Conc.	Total
Comp 1	0.38	1.04	1.42	0.5	1.4	0.12	10.3	81.1
Comp 2	0.41	1.1	0.56	0.4	0.58	0.13	7.6	50.3
Comp 3	0.39	1.07	1.42	0.7	1.43	0.4	11.7	61.9
Comp 4	0.38	1.09	1.62	0.6	1.78	0.42	1.5	26.4
Comp 5	0.35	1.2	1.86	0.6	2.07	0.38	4	16.6
Comp 6	0.38	1.04	1.5	0.8	1.81	0.46	1.7	5.1
Comp 7	0.32	1.46	1.25	0.7	1.39	0.42	1.4	65.2
Comp 8	0.38	1.03	1.84	0.6	1.54	0.52	2.4	18.2

13.1.5 2023 Amb Lab testwork

Boroo has been requested Amb Lab to conduct bottle roll testwork for sulphide ore. The report objective is purely for internal usage to understand sulphide ore recovery.

Sulphide ore samples are dry and crushed by -2 mm in diameter after the bags were received, then prepared for testwork by separation tool.

Table 13-11: Sample Assays

Numbers	Sample	Weight, kg	Head grade, Au g/t
1	BDD204-AMB5	11.05	1.17
2	BDD231-AMB15	12	1.37
3	BDD216-AMB11	11.7	1.61
4	BDD202-AMB2	10.75	1.69
5	BDD204-AMB4	11	1.83
6	BDD221-AMB12	8.15	2.55
7	BDD216-AMB10	10.8	2.59
8	BDD233-AMB18	11.7	2.62
9	BDD231-AMB14	11.1	2.65
10	BDD236-AMB19	10.7	2.97
11	BDD236-AMB20	11.85	3.82
12	BDD210-AMB9	11.6	3.96
13	BDD210-AMB8	11.65	4.15
14	BDD208-AMB6	11.3	4.16
15	BDD208-AMB7	11.5	4.25
16	BDD231-AMB16	9.9	4.26
17	BDD233-AMB17	9.25	5.01
18	BDD204-AMB3	10.8	6.13
19	BDD202-AMB1	9.7	8.65
20	BDD225-AMB13	11	8.79

13.1.6.2 Gravity Separation

Gravity separation testwork program was completed on all samples to investigate the efficacy of gravity-based gold separation. Testwork results show that gravity concentrate yield is 0.9-2.0%, gold grade in concentrate 23.09-552.9 grams, metal recovery 18.63-89.22 percent.

Table 13-12: Gravity separation result

Sample	Grade Au g/t	Gravity Gold Conc.		Gravity Tailing		Gravity Recovery, %	Gravity Tailing, %
		Yield, %	Au, g/t	Yield, %	Au, g/t		
BDD204-AMB5	1.17	1.9	31.81	98.1	0.57	52.08	47.92
BDD231-AMB15	1.37	1.3	23.09	98.7	1.08	21.97	78.03
BDD216-AMB11	1.61	1.3	34.15	98.7	1.17	28.42	71.58
BDD202-AMB2	1.69	1.1	47.51	98.9	1.2	29.87	70.13
BDD204-AMB4	1.83	1.4	68.55	98.6	0.86	53.76	46.24
BDD221-AMB12	2.55	2	40.74	98	1.76	32.52	67.48
BDD216-AMB10	2.59	1.4	44.92	98.6	1.97	25.01	74.99

Sample	Grade	Gravity Gold Conc.		Gravity Tailing		Gravity	Gravity
	Au g/t	Yield, %	Au, g/t	Yield, %	Au, g/t	Recovery, %	Tailing, %
BDD233-AMB18	2.62	1.3	56.55	98.7	1.91	27.87	72.13
BDD231-AMB14	2.65	1.5	43.25	98.5	2.04	23.85	76.15
BDD236-AMB19	2.97	1.4	63.55	98.6	2.09	30.59	69.41
BDD236-AMB20	3.82	1.3	133.67	98.7	2.11	45.52	54.48
BDD210-AMB9	3.96	1.4	150.41	98.6	1.88	53.09	46.91
BDD210-AMB8	4.15	1.5	51.19	98.5	3.43	18.63	81.37
BDD208-AMB6	4.16	1.4	91.18	98.6	2.93	30.47	69.53
BDD208-AMB7	4.25	1.4	64.07	98.6	3.42	20.6	79.4
BDD231-AMB16	4.26	1.6	183.23	98.4	1.29	70.32	29.68
BDD233-AMB17	5.01	1.4	267.55	98.6	1.21	76.16	23.84
BDD204-AMB3	6.13	1.3	309	98.7	2.16	65.31	34.69
BDD202-AMB1	8.65	1.4	552.9	98.6	0.95	89.22	10.78
BDD225-AMB13	8.79	0.9	454.8	99.1	4.56	48.66	51.34

13.1.6.3 Bottle roll leaching test

Bottle roll leaching test has been conducted for gravity tailings. Bottle roll leaching test after using NaOH to make PH-10.5-11, 300 ppm of NaCN prepare a solid liquid ratio of 1:1 for 72 hours and conducted experiments.

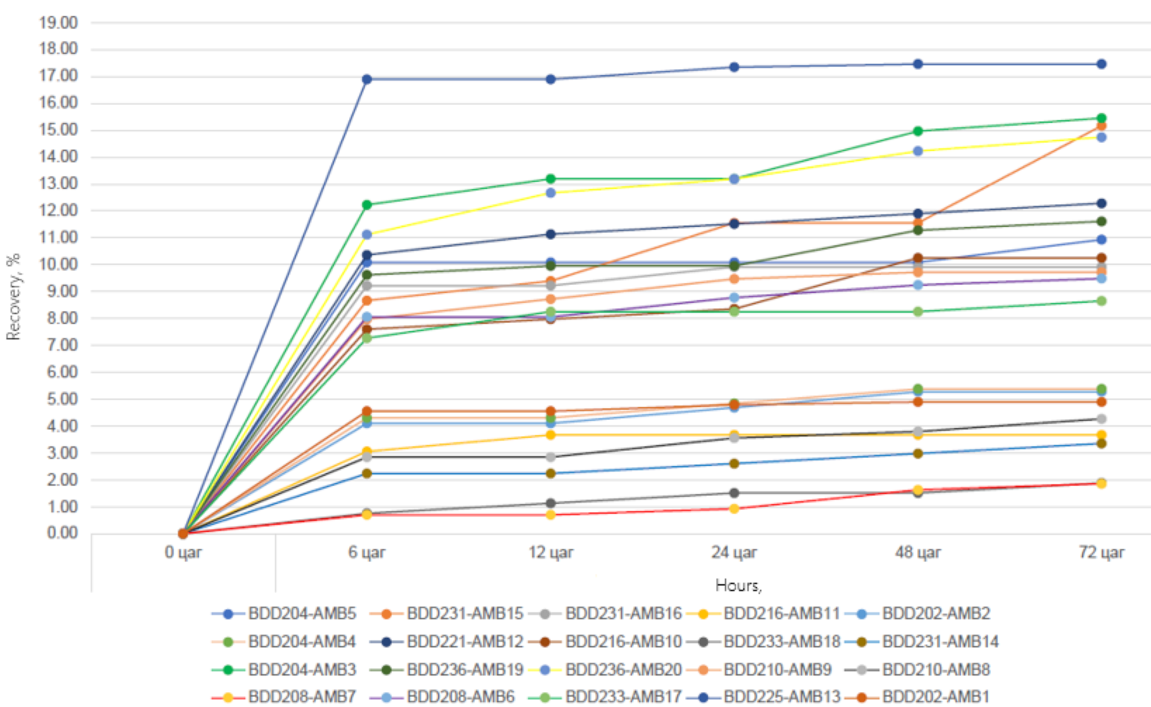


Figure 13-1: Gravity tailing leaching result

Table 13-13: Amb Lab testwork recovery result

#	Sample	Head grade, Au g/t	Gravity Gold Conc.		Gravity Tailing		Gravity recovery, %	Leaching Recovery, (Gravity tail)	Total recovery
			Weight, g	Au, g/t	Weight, g	Au, g/t			
1	BDD204-AMB5	1.17	0.211	31.81	10.84	0.57	52.08	10.93	63.02
2	BDD231-AMB15	1.37	0.156	23.09	11.84	1.08	21.97	15.17	37.14
3	BDD216-AMB11	1.61	0.157	34.15	11.54	1.17	28.42	3.67	32.09
4	BDD202-AMB2	1.69	0.114	47.51	10.64	1.2	29.87	5.28	35.14
5	BDD204-AMB4	1.83	0.158	68.55	10.84	0.86	53.76	5.38	59.14
6	BDD221-AMB12	2.55	0.166	40.74	7.98	1.76	32.52	12.29	44.8
7	BDD216-AMB10	2.59	0.156	44.92	10.64	1.97	25.01	10.26	35.27
8	BDD233-AMB18	2.62	0.151	56.55	11.55	1.91	27.87	1.88	29.75
9	BDD231-AMB14	2.65	0.162	43.25	10.94	2.04	23.85	3.35	27.2
10	BDD236-AMB19	2.97	0.153	63.55	10.55	2.09	30.59	11.61	42.21
11	BDD236-AMB20	3.82	0.154	133.67	11.7	2.11	45.52	14.74	60.27
12	BDD210-AMB9	3.96	0.162	150.41	11.44	1.88	53.09	9.72	62.81
13	BDD210-AMB8	4.15	0.176	51.19	11.47	3.43	18.63	4.27	22.9
14	BDD208-AMB6	4.16	0.157	91.18	11.14	2.93	30.47	9.49	39.95
15	BDD208-AMB7	4.25	0.157	64.07	11.34	3.42	20.6	1.86	22.46
16	BDD231-AMB16	4.26	0.162	183.23	9.74	1.29	70.32	9.92	80.24
17	BDD233-AMB17	5.01	0.132	267.55	9.12	1.21	76.16	8.65	84.81
18	BDD204-AMB3	6.13	0.14	309	10.66	2.16	65.31	15.45	80.77
19	BDD202-AMB1	8.65	0.135	552.9	9.54	0.95	89.22	4.9	94.12
20	BDD225-AMB13	8.79	0.1	454.8	10.53	4.56	48.66	17.46	66.12
Average							**42.2**	**8.8**	**51.0**

As shown in above **Table 13-13**, sulphide ore recovery is reached 51% with bottle roll leaching testwork which confirms actual mined ore recovery.

13.2 Ulaanbulag

13.2.1 2005 Bottle Roll testwork

Initial leaching testwork work has been conducted in 2005 for Ulaanbulag deposit by ACA lab. A totally 9 samples were selected and tested. In additional, 7 samples were tested in Actlabs out of those 9 samples. Below **Table 13-14** is showing comparison of those 2 labs result.

Table 13-14: Bottle Roll testwork result ACA lab VS Actlabs

Sample	ACA lab, 2005			Actlabs, 2010		
	Au grade, g/t		Recovery, %	Au grade, g/t		Recovery, %
	Head	Tail		Head	Tail	
UB-41-1	0.55	0.10	81.64%	0.66	0.05	92.42%
UB-41-2	0.75	0.66	12.27%	0.72	0.26	64.58%
UB-41-3	1.87	0.34	81.87%	0.64	0.17	73.44%
UB-41-4	2.83	1.24	56.18%	2.16	0.66	69.37%
UB-42-5	2.22	1.10	50.45%	1.59	0.10	93.71%
UB-42-6	4.79	1.96	59.08%	3.24	0.19	94.13%
UB-42-7	1.81	1.10	39.23%	1.86	0.54	71.24%

As a result of ACA and Actlabs, recovery difference was high however same laboratory method was applied. It is possible the Actlabs laboratory leaching tested well, the recovery may be considered high in all tests, depending on the tailing grades.

13.2.2 2009 Bottle Roll testwork

During 2009-2010, A totally 127 sample were tested in Actlabs and following **Figure 13-2** and **Figure 13-3** shows result of testwork.

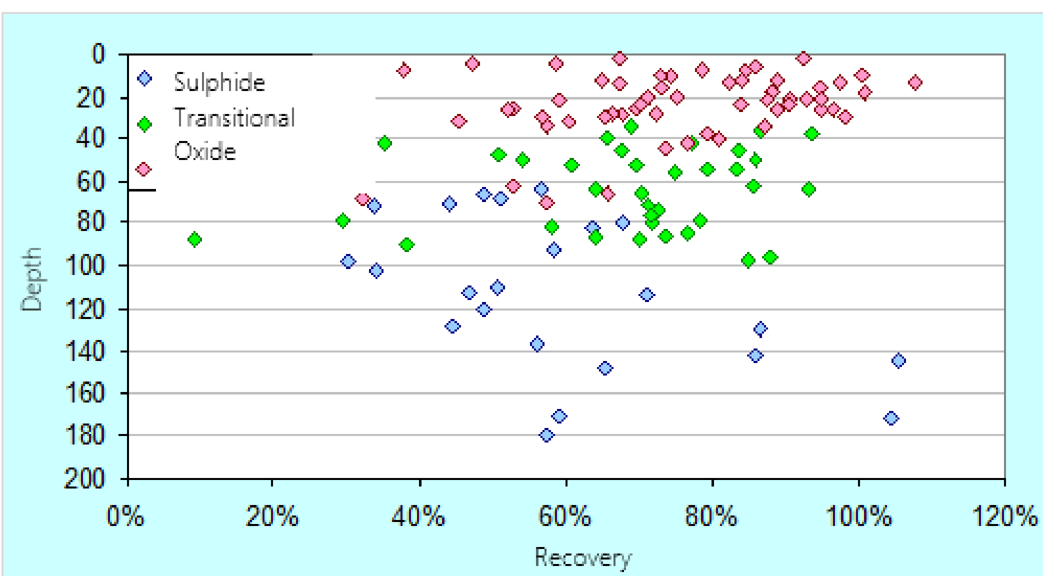


Figure 13-2: Recovery VS Pit depth

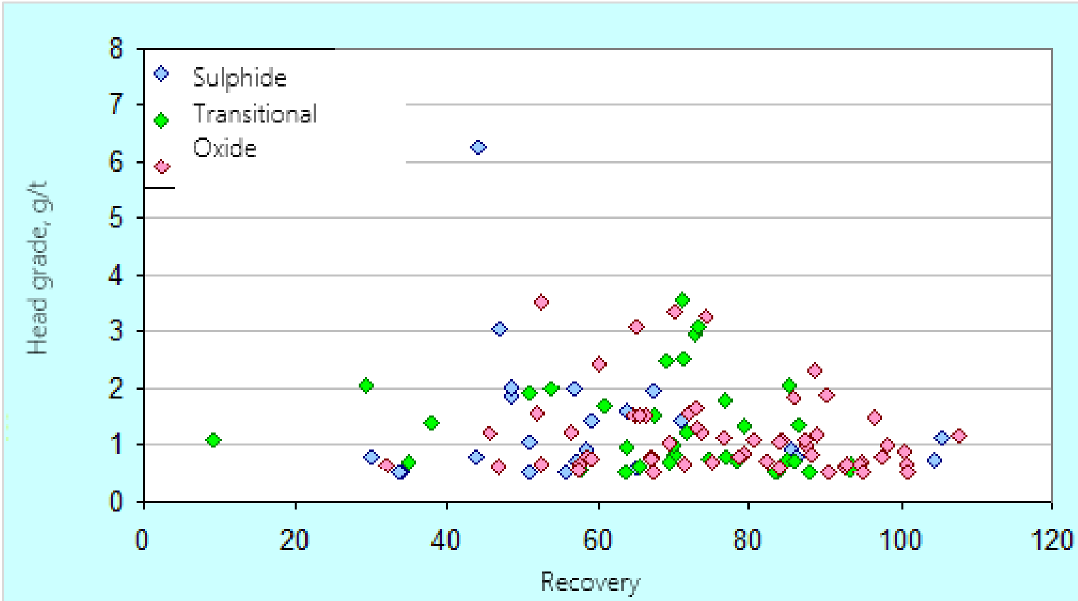


Figure 13-3: Recovery VS Head grade

As shown in the **Figure 13-2** and **Figure 13-3** above, the gold recovery is less dependent on depth and gold head grade.

Ulaanbulag deposit gold recoveries of 87.6% were reported for oxide and 81.5% for transition ores. During 2021-2023, Operation gold recoveries averaging 83.5% and ranged 81% to 85%.

13.3 Heap Leach

In 2005 preliminary testing was started to determine the amenability of low grade stockpiled ore to heap leaching.

The ore types tested were primarily oxide and transition. The availability of samples of fresh ore was limited by the state of current pit development. Metallurgical testing was conducted by AMMTEC in Australia, Kappes-Cassiday & Associates (KCA) in Colorado, and BGC.

To provide more definitive information on possible recoveries at coarse crush sizes, crib tests were carried out on three large bulk samples of low grade run of mine ore. Each test utilized 60 to 70 t of sample. The cribs consisted of two containers welded end to end, set up vertically. A solution drainage system was incorporated into the bottom of each crib prior to sample placement. Drip emitters were used to distribute leach solution on top of each sample. Gold dissolved in solution was adsorbed onto small activated carbon columns. The barren solution from the carbon columns was circulated back to the crib. Two samples were treated as received while another was crushed using the mill primary crusher. All samples were collected from Pit 3.

One sample was identified as oxide. The other samples were identified as low grade transition and the other as high grade transition. The high grade transition sample was crushed through the mill feed jaw crusher with a closed side setting of 100 mm.

The samples used for the crib testing program were found to be representative of oxide and transitional ores that would be processed by heap leaching, in the opinion of the author. The calculated head grades would represent higher grade material that would be heap leached.

Table 13-15: Results of Crib Leach Tests Conducted at Boroo Mine, 2006

Sample	Calculated Head Grade (g/t Gold)	Extraction (% Au) at Days of Leaching					Ultimate Extraction (% Au)	Total Leach Time (days)
		15.00	30.00	45.00	60.00	75.00		
Oxide	1.01	55.70	68.10	75.10	80.10	83.50	83.90	83
Low Grade Transition	1.05	44.40	57.40	61.70	64.30	66.00	66.40	83
High Grade Transition	0.92	56.70	69.50	76.00	79.70	82.00	82.20	79

Based on the crib test results, it appears that after solution to ore application ratio of 1:1 or approximately 30 days of leaching, gold recovery of 50% of ultimate recovery is achieved (ultimate recovery defined as solution to ore ratio of 4:1). The 4:1 ratio will be obtained in the later leach cycles when additional lifts are under primary leach and the lower lifts are under re-leach.

13.4 Summary

The current Boroo ore processing flowsheet is the result of a number of past metallurgical test programs and confirmed by successful operation results. The current Boroo Mill flowsheet includes crushing, grinding, gravity concentration, leaching and carbon-in-pulp (CIP), cyanide detoxification and arsenic precipitation to produce gold doré.

During 2002, bottle roll test result was representing recovery of lower transition and fresh ore in Boroo deposit. Gold recoveries were variable, averaging 75% but ranged from 57% to 90%.

During 2006, column leaching testwork has been conducted for oxide, transitional and fresh ore by AMMTECH Australia and KCA USA based labs. AMMTECH testwork gold recoveries of 91.4% were reported for oxide ore, 70.3% for transition ore and 64.9% for fresh ore. KCA testwork gold recoveries of 90% were reported for oxide ore, 65% for transition ore and 53% for fresh ore.

Diagnostic leach tests on mill tailings during the period of low recovery found that 24% to 35% of the gold was locked in sulphide.

During 2009-2010, bottle roll tests were undertaken on oxide, transition and fresh ore of the Ulaanbulag deposit. Gold recoveries of 87.6% were reported for oxide and 81.5% for transition ores.

In 2023, Regarding to the Amb Lab fresh ore bottle roll testwork, gold recoveries were variable, averaging 51.2% and ranged 22% to 94% in Boroo deposit. It is worth to mention that Amb Lab laboratory is not accredited by any international standard.

Overall, however, metallurgical test data on the fresh ore at Boroo and Ulaanbulag was limited.

During 2021-2023 mining, Boroo deposit and Ulaanbulag operation gold recoveries averaging 77.8% and ranged 78.1% to 77.7%.

In 2023, pit 5 mining section is mostly focused on fresh ore and gold recoveries averaging 74.5%. In additional, previously classified BIOX process ore during Centerra, gold recovery averaging 58-61% during 2013-2014.

Operation results showed that the gold recovery prediction from testwork to be accurate.

13.5 Recommended future work

Future testwork will be required and will involve defining the remaining ore source. As the oxide material was tested extensively and zone is going to be mined out mostly, it is recommended that future testwork focus on fresh rock for Boroo and Ulaanbulag deposit. The recommended components of the testwork may include gravity recoverable gold, bottle roll leach testing.

14.0 MINERAL RESOURCE ESTIMATES

14.1 Summary

The Boroo Gold Project includes Mineral Resource Estimates for the Boroo and Ulaanbulag deposits as outlined in the following sections.

14.2 Boroo

14.2.1 Basis of Current Mineral Resource Estimate

Game Mine was commissioned by Boroo Gold LLC to generate an updated Mineral Resource Estimate for the Boroo Deposit. The update incorporates 36 additional drillholes (totaling 3,269.8 m) and structural interpretation study completed by Boroo Gold on the Property since the previously announced Mineral Resource Estimate with effective date March 01st, 2023 (GSTATS, March 2023). The Boroo Mineral Resource Estimate is based on data from 1858 RC and diamond drill holes, totaling 167,748.5 metres of drilling. The focus of the 2023 drilling program consisting of infilling drilling was to:

- Expand the understanding of the mineralization zones;
- Build upon the previous geological interpretation;
- Improve drill spacing to show continuity of mineralization and increase overall confidence in the deposit.

Completion of the updated Boroo Resource involved the assessment of an update drill hole database and subsequent generation of an updated geological model, updated structural control model, an updated three-dimensional (3D) grade model. The QP visited the property from August 30 to August 31 2023. The effective date of the Boroo Mineral Resource Estimate is February 01st, 2024. Ordinary Kriging (OK) restricted to a mineralized domain was used to interpolate gold grades (g/t) into a block model. Measured, Indicated and Inferred Mineral resources are reported in summary tables in Section 14.2.10.

14.2.2 Previous NI 43-101 Mineral Resource Estimate

A Mineral Resource Estimate with an effective date of March 30, 2022, was developed for the Boroo deposit. This mineral resource estimate was based on 164,478.72 m of drilling from 577 diamond, and 1245 RC drillholes.

The block model was developed using wireframing method and was classified with Measured, Indicated, and Inferred Resources in accordance with CIM definitions and standards (2014).

Table 14-1 shows the previous Mineral Resource Estimate which is superseded by the current Mineral Resource Estimate stated in Section 14.2.6 and should no longer be relied upon as current.

Table 14-1: Previous Mineral Resource Estimate for the Boroo Deposit estimated by GSTATS as of January 1, 2023

Resource Category	Oxidation State	Tonnage (kt)	Average Grade (@Au g/t)	Metal (kg)
Measured	Oxide	69	0.87	60
	Transition	2,885	1.43	4,126
	Fresh	5,231	1.14	5,973
	Total	8,185	1.24	10,159
Indicated	Oxide	81	0.94	76
	Transition	2,999	1.15	3,455
	Fresh	6,484	0.93	6,051
	Total	9,564	1.00	9,582
Measured + Indicated	Oxide	150	0.91	136
	Transition	5,884	1.29	7,581
	Fresh	11,715	1.03	12,024
	Total	17,749	1.11	19,741
Inferred	Oxide	8	0.85	7
	Transition	561	1.07	598
	Fresh	785	0.81	636
	Total	1,355	0.92	1,241

Notes:

1. *Boroo Mineral Resources are as of January 1, 2023, based on the CIM Definition Standards (2014).*
2. *Mineral Resources that are not Mineral Reserves have no demonstrated economic viability.*
3. *Reporting cut-off grade for Boroo Mineral Resources is 0.2 g/t Au.*
4. *Mineral Resources were constrained by the 0.2g/t mineralized ore shells and mining pits (without backfilling) as of January 1, 2023.*
5. *The Mineral Resources are stated as in situ dry tonnes. All figures are in metric tonnes.*
6. *Inferred Mineral Resources have a great amount of uncertainty as to their existence and as to whether they can be mined economically. It cannot be assumed that all or part of the Inferred Mineral Resources will ever be upgraded to a higher category.*
7. *Figures have been rounded to the appropriate level of precision for the reporting of Mineral Resources.*
8. *Totals may not match due to rounding*

14.2.3 Comparison to Previous Mineral Resource Estimate

A comparison of the current Boroo Deposit updated Mineral Resource Estimate completed by Game Mine and the 2023 Mineral Resource Estimate completed by GSTATS is presented in Table 14-2. The 2023 mineral resource estimate for Boroo has been superseded by the 2024 mineral resource estimate, and as such, should no longer be relied upon.

The increase in gold ounces along with the higher level of confidence in the mineral resource estimate is the result of additional diamond drilling, and updated modeling approaches, whereby high-grade domains were re-modeled based upon this drilling and structural information. The 2024 model was able to utilize a significant dataset of structural measurements which has subsequently increased the understanding of the deposit, specifically the orientation and continuity of higher grade mineralized zones.

Table 14-2: Comparison of the 2023 and 2024 Boroo Deposit Mineral Resource Estimates

Resource Category	Cut-Off Grade (@Au g/t)	Tonnage (kt)	Average Grade (@Au g/t)	Metal (kg)
2023 Measured	0.20	8,185	1.240	10,159
2024 Measured	0.10	26,609	0.588	15,636
2023 Indicated	0.20	9,564	1.000	9,582
2024 Indicated	0.10	17,318	0.542	9,394
2023 Inferred	0.20	1,355	0.920	1,241
2024 Inferred	0.10	1,307	0.789	1,032

Notes:
The 2023 and 2024 mineral resource estimate have utilized different modeling techniques, utilized different input parameters, and have been reported at different cut-off grades. The 2023 mineral resource estimate for Boroo has been superseded by the 2024 mineral resource estimate, and as such, should no longer be relied upon.

14.2.4 Database

In order to complete an updated Mineral Resource Estimate for Boroo, a database comprising a series of excel spreadsheets containing drillhole information was provided by Boroo Gold to the QP. The database includes hole location information (UTM WGS 84, Zone 48N), survey data, assay data, lithology data, oxidation, alteration, bulk density data, and structural data. The data was verified (Section 12) and then imported into Geovia Surpac 2021 and Seequent Leapfrog Geo3D version 2023.2 software ("Leapfrog") for geological modeling and the development of the grade wireframes. Overall, information for 1858 drillholes was provided to Game Mine. The particulars of the information provided to Game Mine are presented below in Table 14-3.

Table 14-3: Summary of Boroo Drilling Database

Record Numbers	Drill Holes		Total
	DD	RC	
Meters drilled	60,316.52	107,792	168,108.52
Collars Records	613	1,245	1,858
Survey Records	1,821	926	2,747
Assay Records	35,548	73,293	108,841
Lithology Records	6,395	8,049	14,444
Oxidation Records	1,465	5,884	7,349
Alteration Records	2,699	320	3,019

A location plan map for the Boroo drill holes, colored by drill hole types and dates are presented in Figure 14-1.

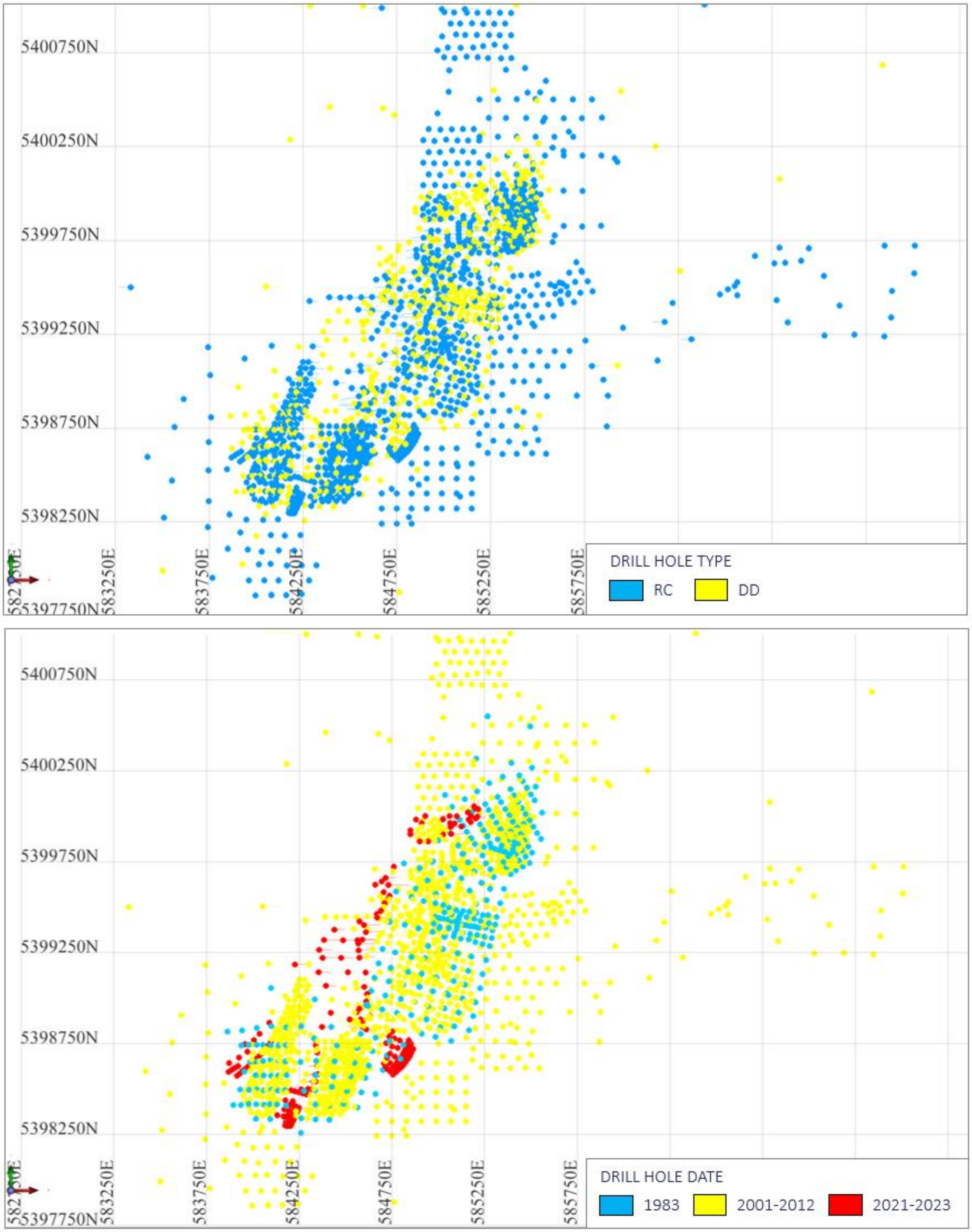


Figure 14-1: Location map of Boroo drillholes

14.2.5 Geological and Resource Interpretation

14.2.5.1 Geological Interpretation

In order to better model and geologically constrain the mineralization, a 3D geological model was constructed in Leapfrog Geo 3D prior to any resource interpretation. Geology model was created in order to control gold mineralization and rock density statistics and geotechnical domain. Five main lithology units were modelled including sandstone (SST), granite (GR), diorite (DI), quartz vein (QZ) and quaternary sediment (Q).

An isometric of the lithological model is provided below in Figure 14-2 and Figure 14-3 respectively.

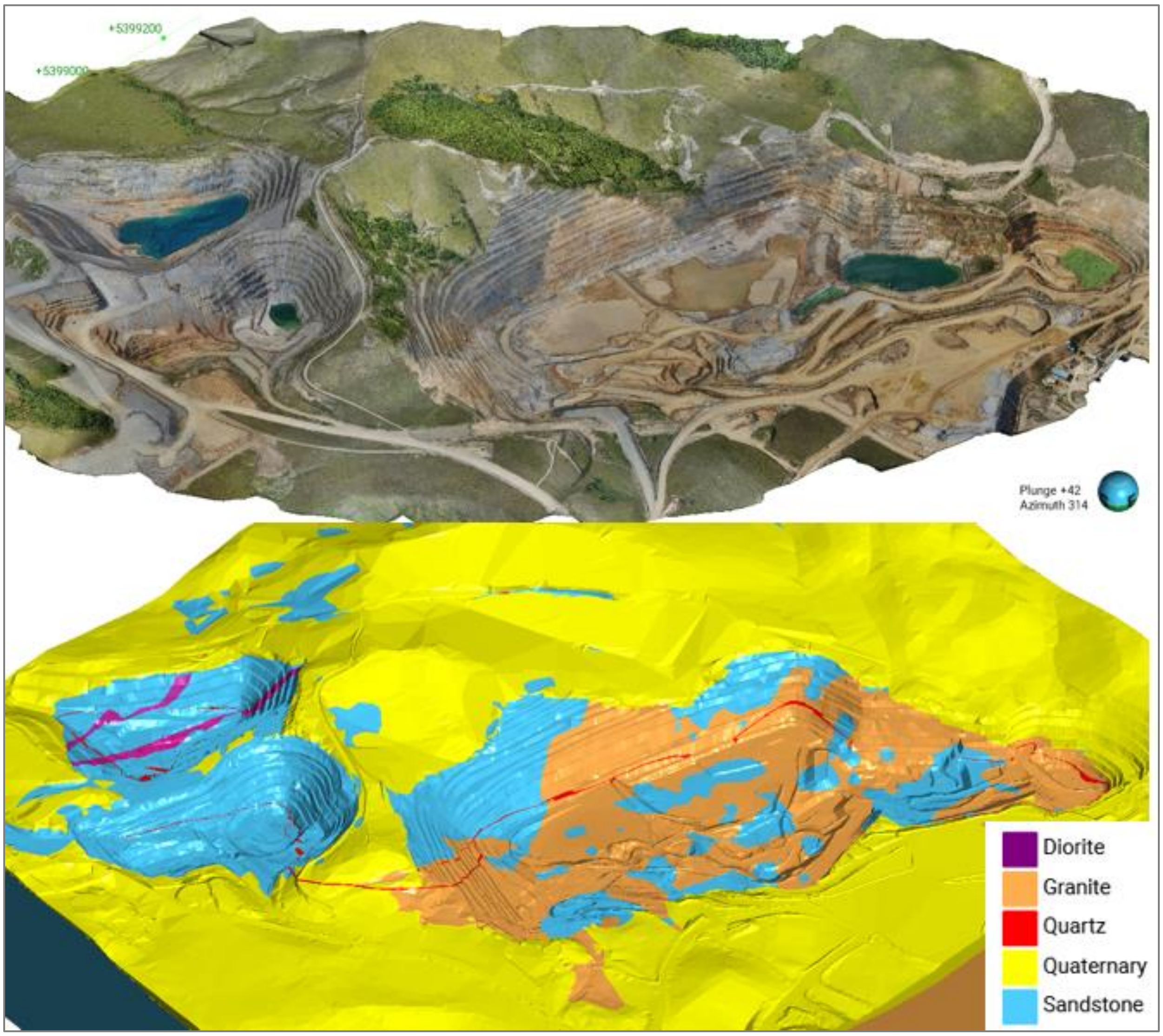


Figure 14-2: Comparison of drone survey data and lithological model

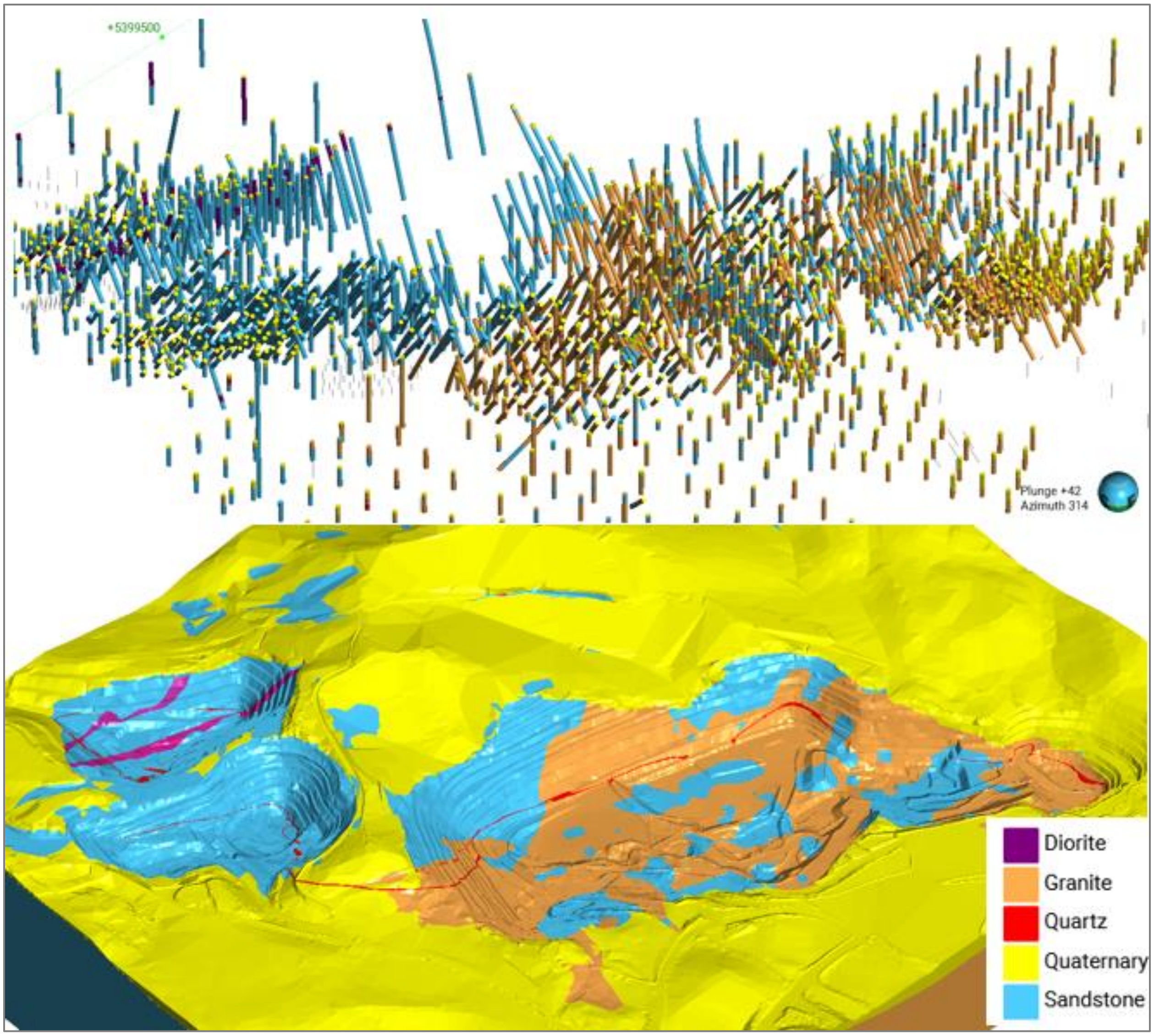


Figure 14-3: Comparison of borehole lithology data and lithological model

14.2.5.2 Oxidation Interpretation

Oxidation level of three-dimensional (3D) solids namely: oxide (Wox), transitional (Wpx) and fresh (Fr) material were created using drillhole oxidation logging information to code oxidation block model, density and gold distributions and geotechnical domain (Figure 14-4). The oxide zone, generally located near surface, has undergone the highest degree of oxidation, followed by a transitional zone, and then the underlying fresh rocks in the primary zone of mineralization.

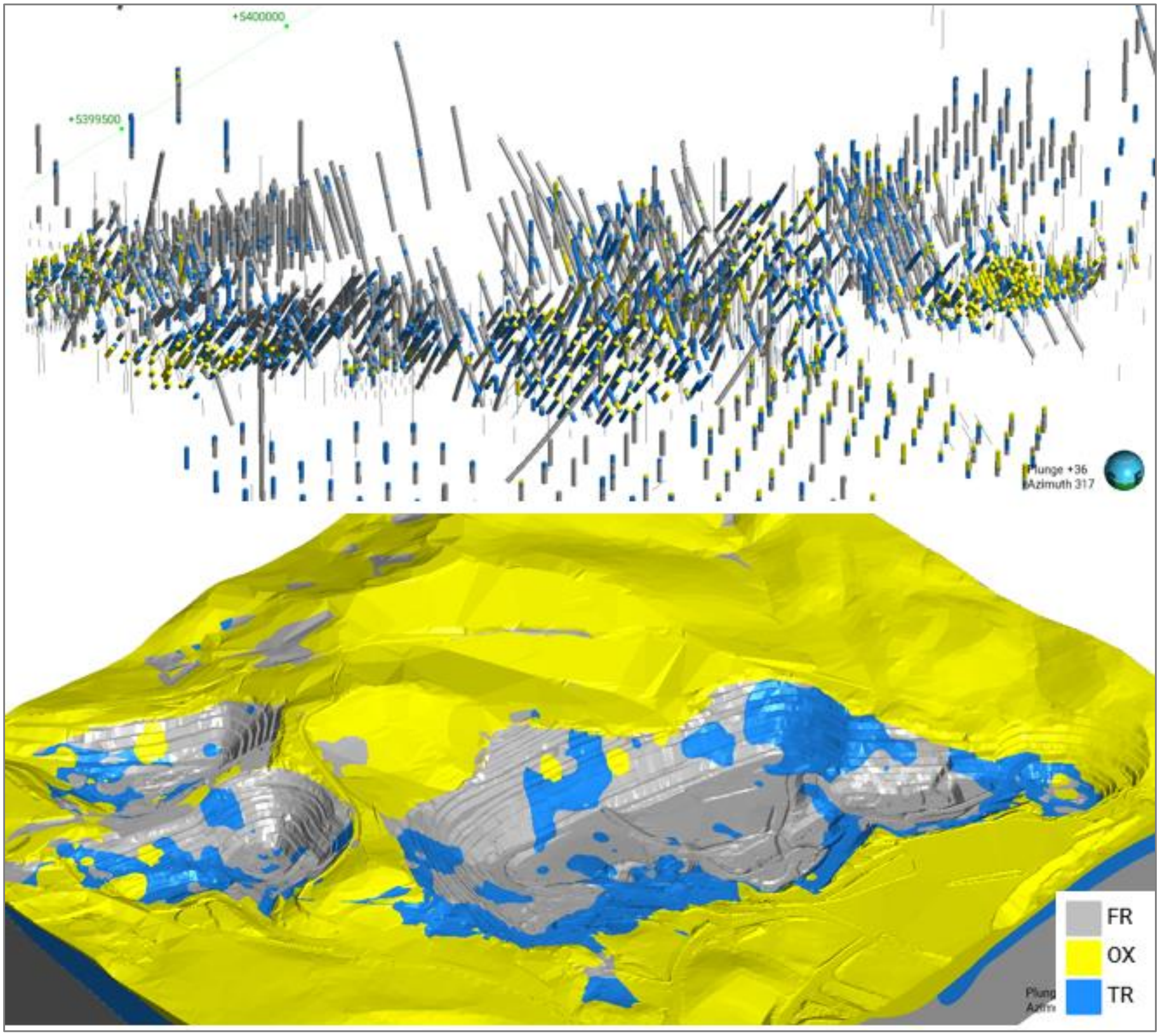


Figure 14-4: Comparison of borehole oxidation data and oxidation model

14.2.5.3 Mineralization Interpretation

The Boroo deposit has historically been divided into five separate mineralized zones or pit areas from north to south, namely Zone 2, 3, 4, 5 and 6 (Centerra, 2009, GSTATS, 2023, Figure 14-5). Based on 2023 exploration data, drone survey data and structural data interpretation, The Game mine has been confirmed that those 5 separate mineralization zones are combined as a one mineralization body. Given the strong lithological and structural influence on mineralization, lithological and structural model was developed in Leapfrog Geo3D to guide the interpretation of the mineralization solids.

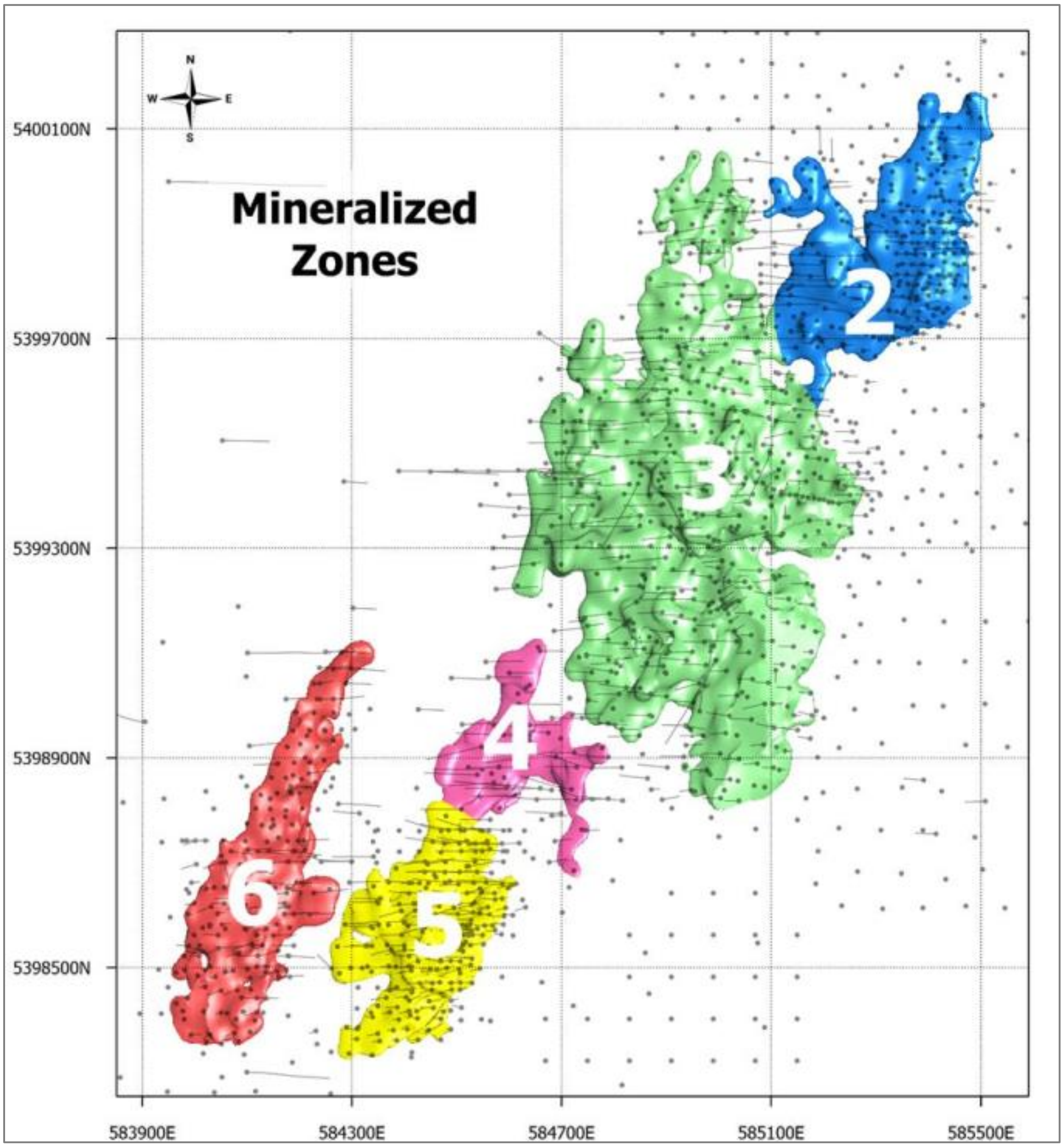


Figure 14-5: Previous five separate mineralized zones

Three-dimensional gold mineralization interpretations were generated for a low-grade and high-grade domain using Leapfrogs Geo Vein modelling and RBF interpolant function. The low-grade domain was defined by gold mineralization occurring with grades equal to or greater than 0.1 g/t Au. High grade domains were defined by hosting gold mineralization greater than 0.8 g/t. Figure 14-6 and Figure 14-7 below presents an isometric image of the mineralization model respectively for Boroo. All mineralized wireframes were trimmed to topography and pit survey data (01 January 2024).

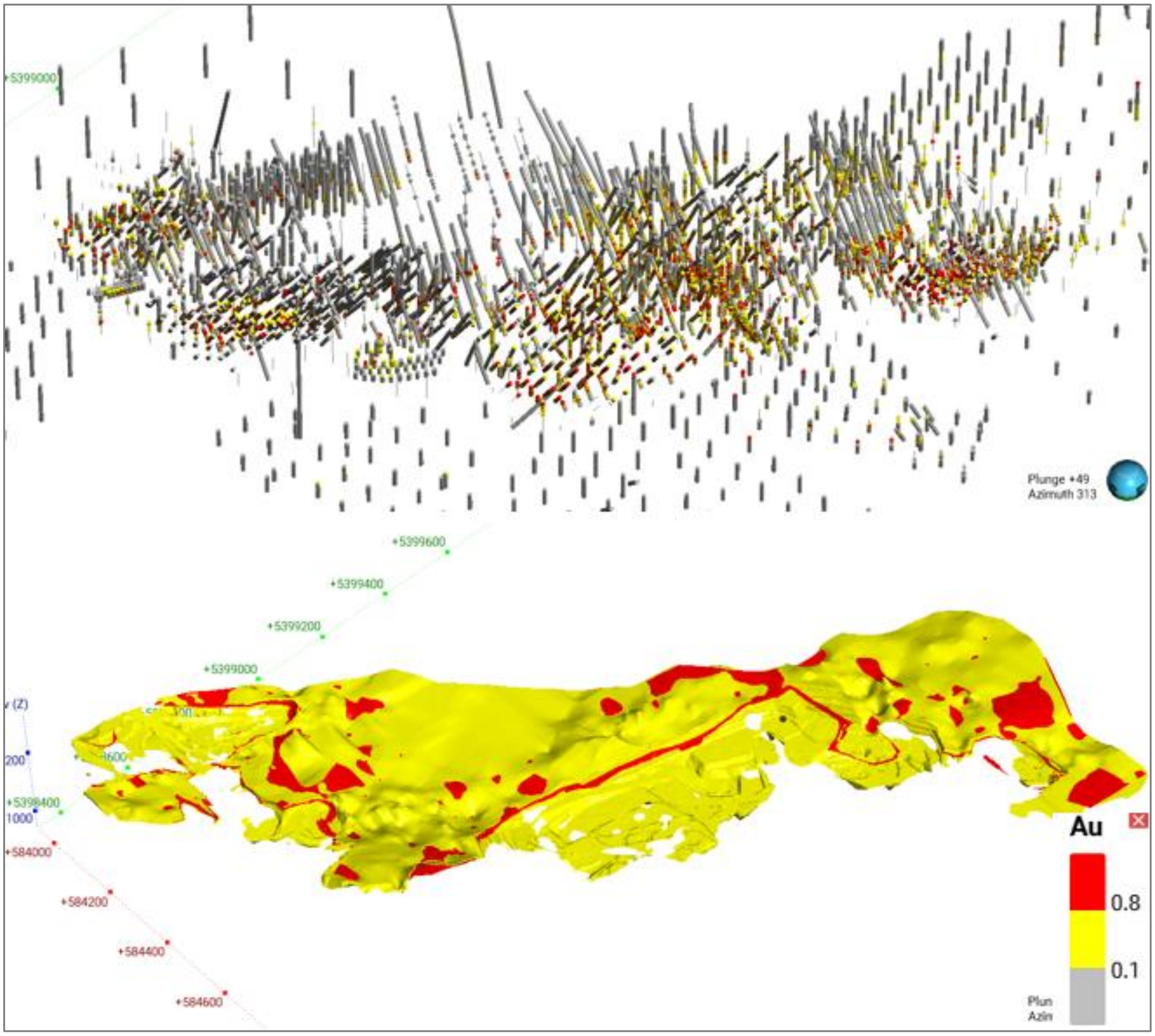


Figure 14-6: Comparison of borehole assay data and mineralization model

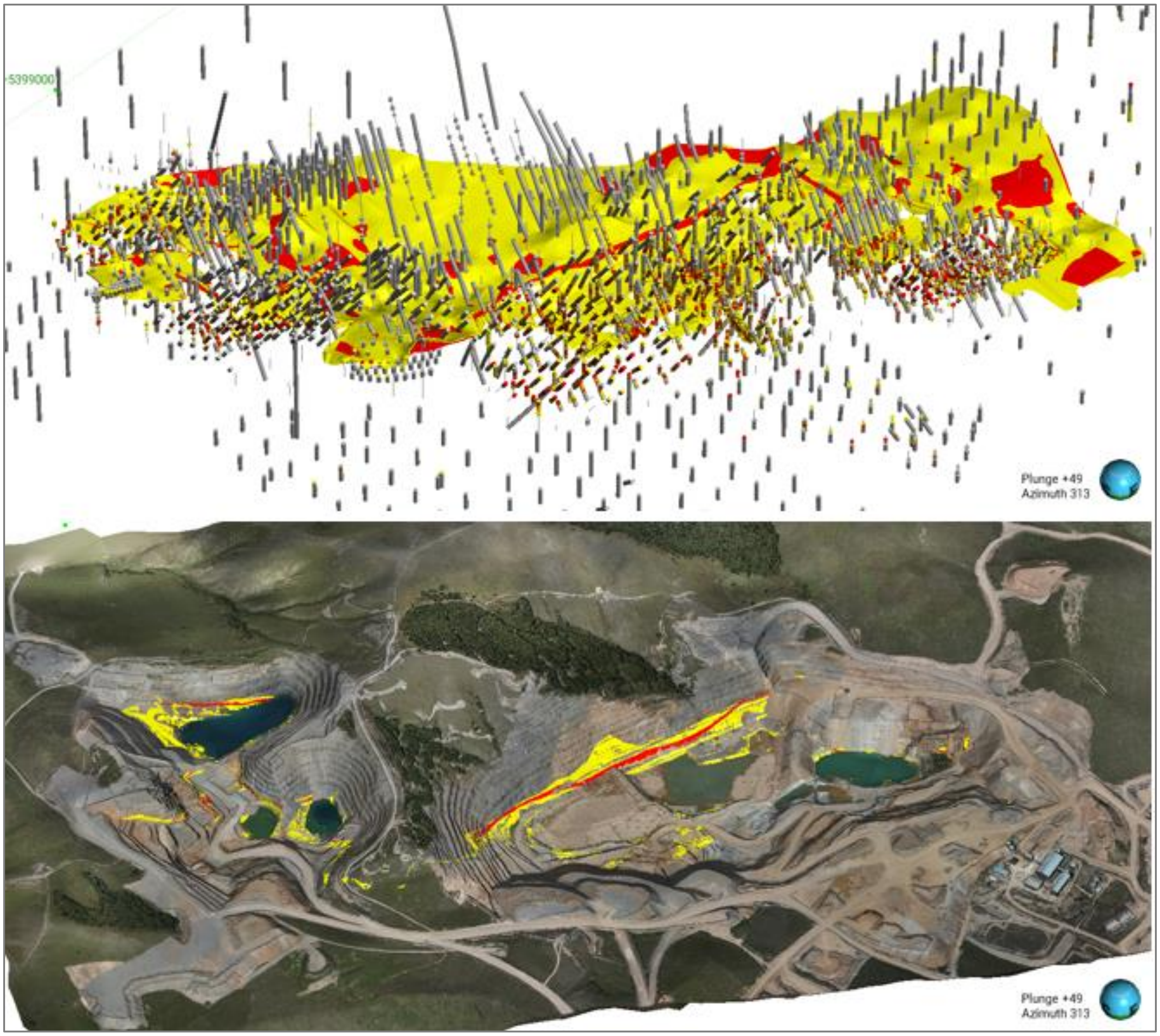


Figure 14-7: Comparison of drone survey data and mineralization model

14.2.6 Exploratory Data Analysis

Upon completion of the geological model, grade solids were imported into Leapfrog Edge version 2023.2 where numerical modeling and estimation was undertaken. The sections below summarize details associated with the various aspects of the numerical modeling and estimation process.

14.2.6.1 Assays

The assay intervals within each domain were captured using a Leapfrog Edge Estimation into individual drillhole files. These drillhole files were reviewed to ensure all the proper assay intervals were captured. The non-assayed intervals were given a zero (0) value. Gold assay statistics between raw and composite samples for each zone are summarized in Table 14-4.

14.2.6.2 Compositing

Sampling was undertaken at varying lengths within the Boroo deposit. Game Mine reviewed all sample lengths. The raw length statistics showed 1.35 m to be the mean length of sample intersect, with a median of 1.00 m and a range between 0.02 m and 19.01 m (Figure 14-8).

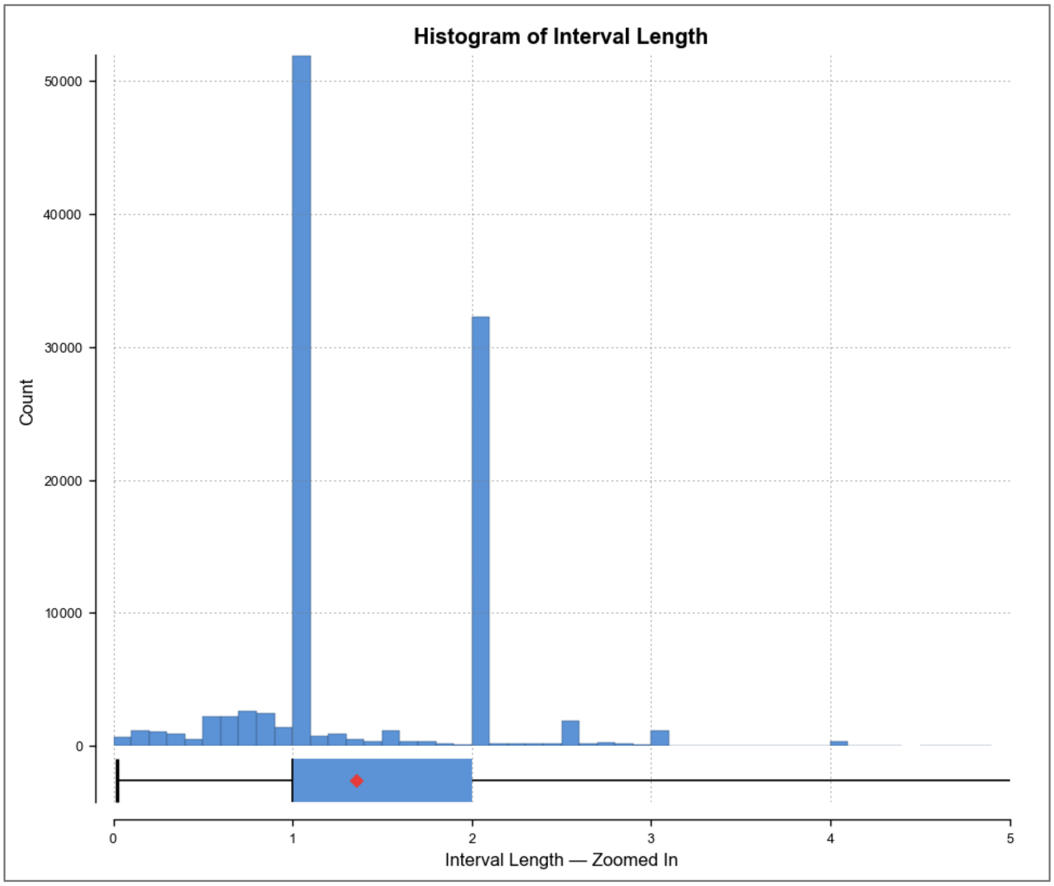


Figure 14-8: Histogram sample lengths

The technical report (Centerra, 2009) states that the samples were compositing to 2.5 m within the mineralization domains, defined by the standard bench definitions for the Boroo open pit. Missing assay data were ignored in the length weighted averaging. Residual composites <1 m was removed prior to estimation.

Game Mine compared the mean Au g/t grades for compositing to 1m and 2.5m within the modelled mineralization. Game Mine considers compositing to 2.5 m as reasonable (fits to mining benches) and removal of the residuals as good practice that limits any potential bias in the sample support during estimation.

Table 14-4: Raw and Composite Sample Statistics

Comp Length	Raw	1m	2.5m
Count	14,522	15,993	5,700
Mean	0.692	0.668	0.646
Standard deviation	2.916	2.595	1.517
Coefficient of Variation	4.210	3.770	2.348
Variance	8.505	6.735	2.302
Minimum	0.005	0.005	0.005
Q1	0.080	0.098	0.114
Q2	0.268	0.288	0.306
Q3	0.650	0.664	0.683
Maximum	132.711	114.541	50.739

14.2.6.3 Capping Analysis

Composited assay data was examined for each domain to assess the amount of metal that is at risk from high-grade assays. The Leapfrog Edge module was used to determine if grade capping was required. Capping was based on examination of the log probability plots and histograms for each metal and caps were applied where graphs showed significant outlier influence or deviation from the general trend-line (Figure 14-9). Log probability plots and Cumulative frequency plots were inspected for each domain and applied the capping grades using a combination of histograms, probability plots, and decile analyses. Coefficients of variation (CV) after applying capping were showing decreases.

Table 14-5 shows a summary of the top cuts that were applied to the domain datasets. The QP is of the opinion that the capping levels are reasonable, and suitable for the estimation of Mineral Resources.

Table 14-5: Boroo Capped Composite Statistics

Domain	High Grade		Low Grade	
	Uncapped	Capped	Uncapped	Capped
Count	975	975	4,725	4,725
Mean	1.932	1.760	0.389	0.385
Standard deviation	3.146	1.859	0.582	0.529
Coefficient of Variation	1.628	1.055	1.496	1.373
Variance	9.898	3.456	0.339	0.280
Minimum	0.005	0.005	0.005	0.005
Q1	0.722	0.722	0.092	0.092
Q2	1.184	1.184	0.235	0.235
Q3	2.038	2.038	0.492	0.492
Maximum	50.739	10.000	12.841	5.000

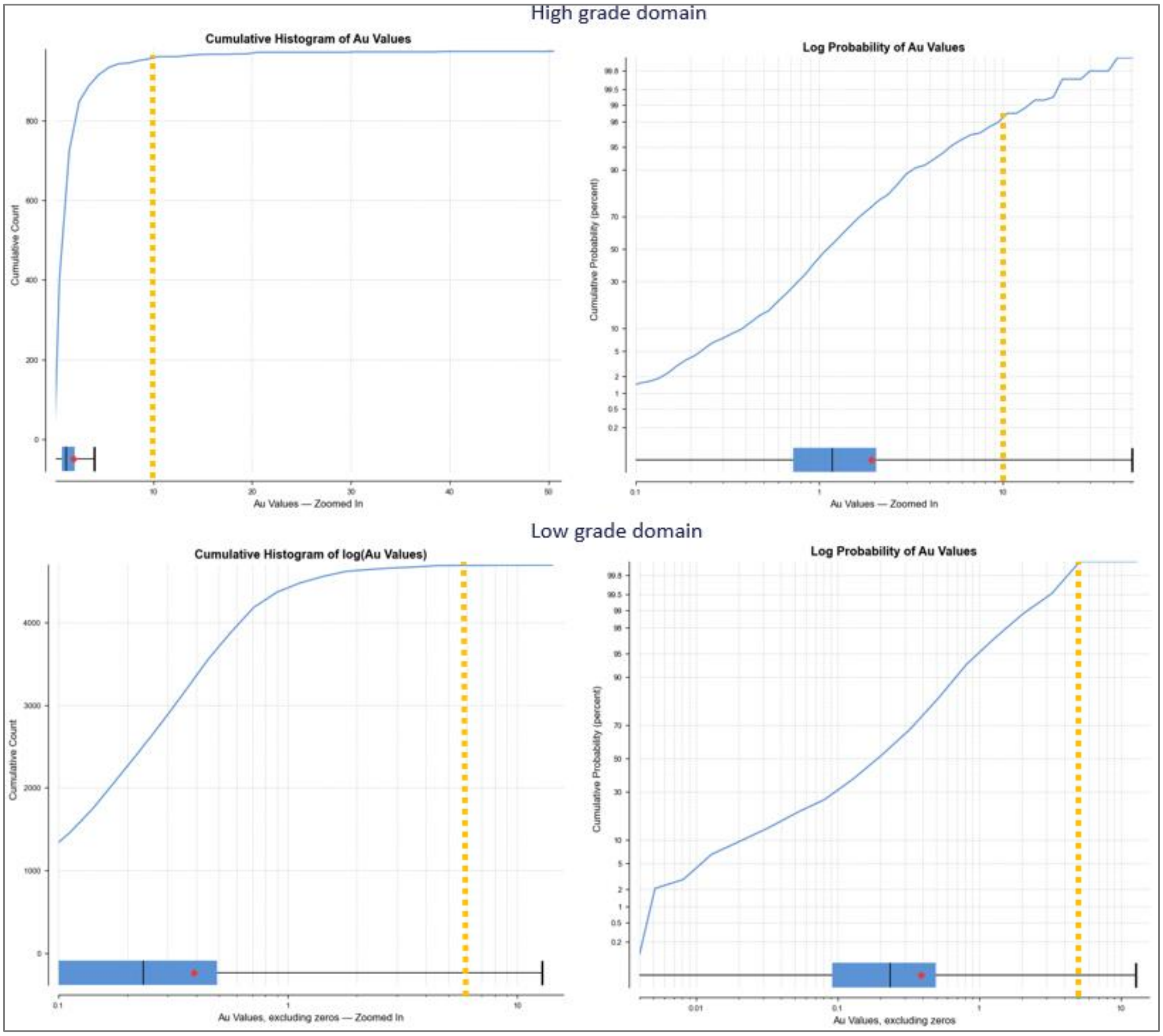


Figure 14-9: Log probability plots and Cumulative frequency plots by domain

14.2.6.4 Variogram Assessment

Downhole and directional variography was undertaken by Game Mine using Leapfrog Edge software to quantify the spatial relationship of composited data within the selected mineralized domains. Variography was used to assess grade continuity and spatial variability in the estimation domains and to determine sample search distances and kriging parameters for block grade estimation.

Downhole variograms were used to determine the nugget effect, then the variograms were modeled with two structures to determine spatial continuity in the zones. The results of the variography analysis are summarized below in Table 14-6 and Figure 14-10 present the variogram maps and variograms for each the major, semi-major, and minor axes respectively. Overall, the best variogram was achieved in the plane of the orebody.

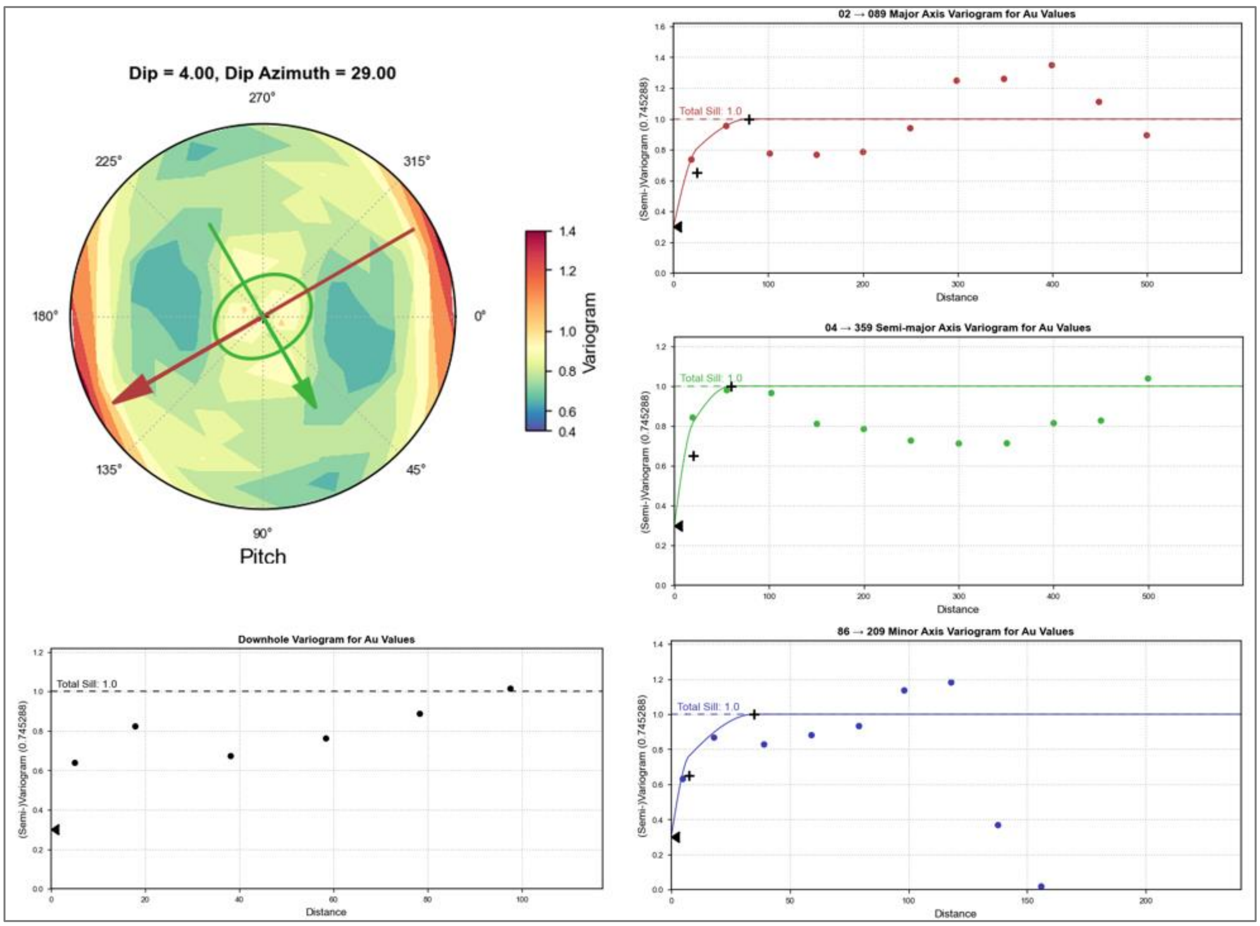


Figure 14-10: Variogram Model

Table 14-6: Variogram analysis parameters

	Norm. sill	Type	Major	Semi-Major	Minor
Nugget	0.30	-	-	-	-
Structure 1	0.35	Spherical	25	25	7.5
Structure 2	0.35	Spherical	70	55	30
Dip		Dip Direction		Pitch	
5		30		150	

14.2.6.5 Resource Block Model

Game Mine created the block models with a parent block size of 5 mN x 5 mE x 2.5 mRL, with no sub-blocking and no rotation. The block model parameters are shown in **Table 14-7**. The original block model was created for the grade interpolations and resource classifications in Leapfrog and exported to Geovia Surpac. The drill sections are nominally on 20 m x 20 m spacing, and the mineralization strikes northeast – southwest.

Table 14-7: Block model parameters

Model Name	boroo_2023_v01.mdl		
	Y	X	Z
Minimum Coordinates	5397950	583700	1270
Extent	2100	2600	330
Rotation	0	0	0
Parent Block Size (m)	5	5	2.5
Sub Block Size (m)	5	5	2.5
Attributes			
au_ok	Reportable au ppm grade (Ordinary Kriging)		
rescat	Classification (Measured, Indicated, Inferred)		
grade_shell	High grade, low grade domain		
oxidation	Oxidation state (Oxide, Transition, Fresh)		
litho	Lithology type (Granite, Diorite, Sandstone, Quartz Vein etc.)		
density	Bulk density (t/bcm)		

14.2.6.6 Dynamic Anisotropy

Due to the local changes in domain orientations and local anastomosing nature of the wireframes a single search ellipse would not truly reflect the distribution of mineralization within the domain. To accommodate the changing orientation of the domains, dynamic anisotropy was utilized to better represent continuous mineralization within the mineralized structures.

Dynamic anisotropy is a function in Leapfrog Edge Variable Orientation that permits the search ellipse orientation to be continuously adjusted to match the tangent of the average orientation of the mineralized domain to mimic curvature or folding structures (**Figure 14-11**). This allows the search volume to be oriented to follow the trend of mineralization. The azimuth of the major and semi-major axes and the search dimensions remained unchanged within the search ellipse.

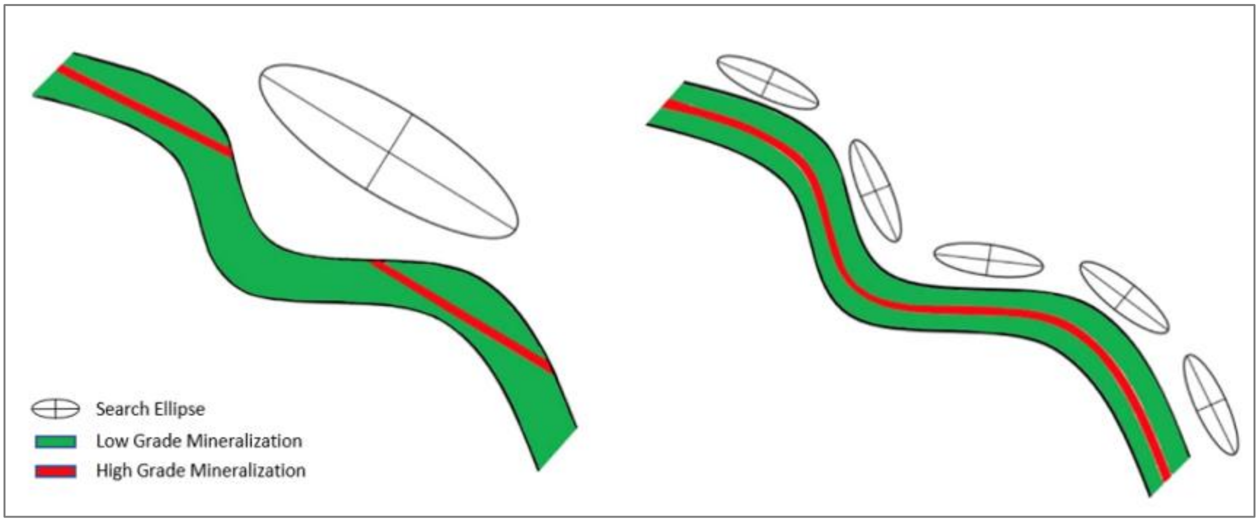


Figure 14-11: Schematic of Dynamic Anisotropy

(Red-High Grade Mineralization, Green-Low Grade Mineralization)

14.2.6.7 Estimation and Search Parameters

The Ordinary Kriging (OK) algorithm was selected for grade interpolation due to the number of samples and the interpreted geospatial analysis. The OK algorithm was utilised to minimise over smoothing within the estimate. The estimations were designed as a three-pass system which were run independently within each individual wireframe using composite data constrained within domain. This process involves the estimation being performed three times, where two expansion factors are used. During each individual estimation run this factor increases the size of the search ellipse used to select samples. This method ensures that blocks which are not estimated and populated with a grade value in the first run, are populated during one of the subsequent runs. The minimum number of samples required within the search neighbourhood for an estimate to be interpolated and a maximum number was specified. The minimum number of samples from any one drill hole was also specified. No smoothing is applied in this estimation.

The final estimated Au grade block model is illustrated in Figure 14-12 and Figure 14-13.

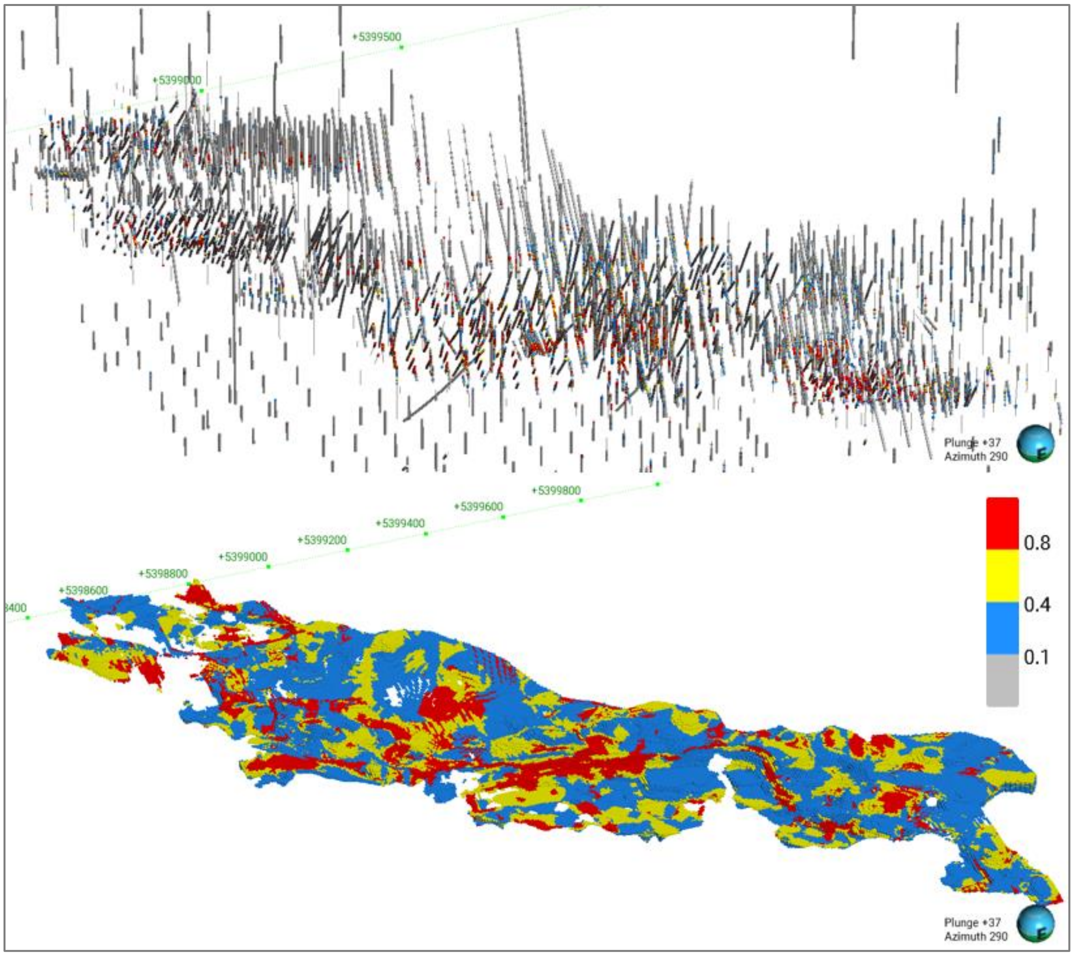


Figure 14-12: Au grade (ppm) block model illustration in 3D view

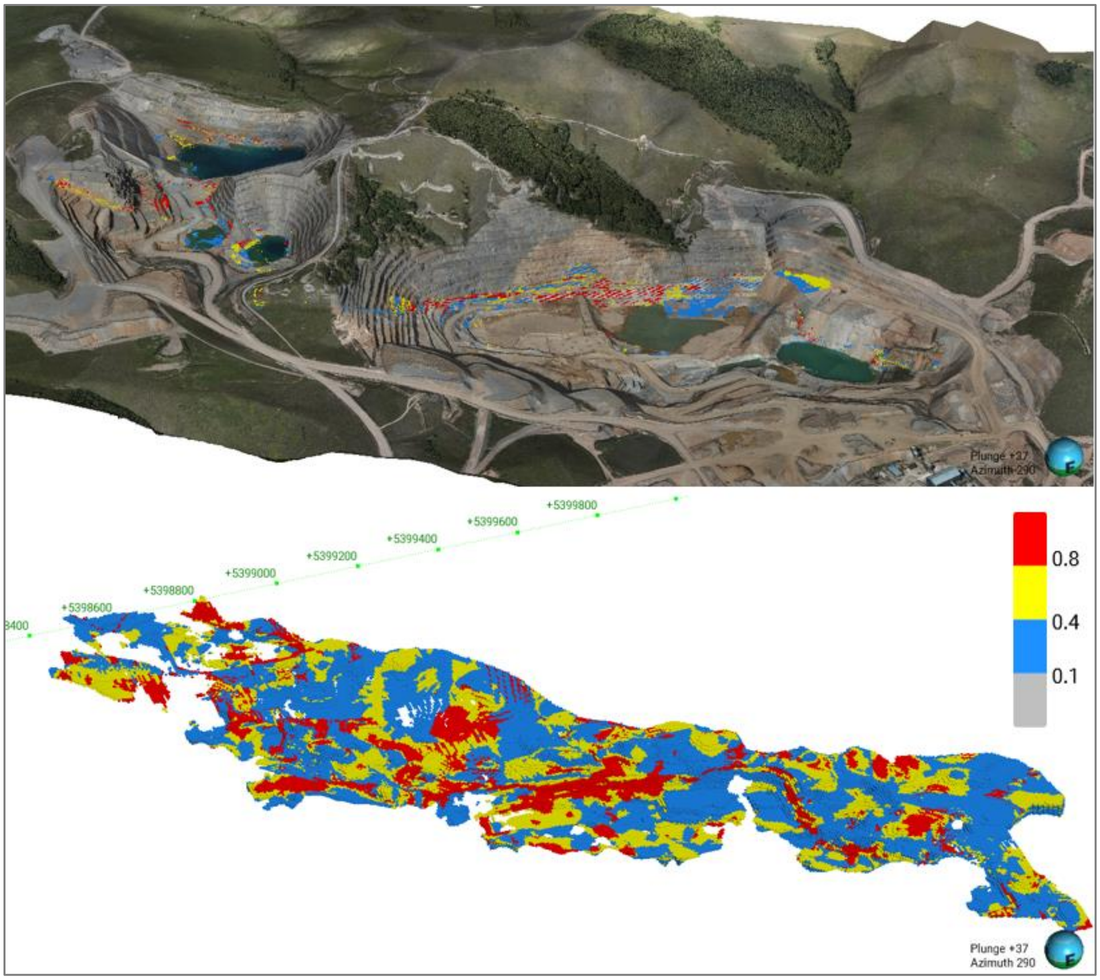


Figure 14-13: Comparison of drone survey data and Au grade (ppm) block model

14.2.7 Specific Gravity

Density assignment is a critical component in the estimation of a reliable Mineral Resource tonnage. Centerra's 2009 Technical Report states that a total of 2,319 samples were collected from drill core by the DDR-MPR expedition, and another 158 from MDD holes for the purpose of specific gravity determinations. The DDR-MPR expedition data were obtained using the method of total immersion of unwrapped and unwaxed whole drill core in water and determination of the volume displaced. This method can be reliably translated into bulk densities only for non-porous drill core. However, the specific gravity figures will be higher than the bulk density where rock porosity at the surface of the core is encountered and filled with water during the process. The specific gravity data have been directly translated into bulk density figures (Centerra, TR2009).

Game Mine received a total of 2,050 density data (GSTATS reviewed). Bulk density statistics were studied within lithological and rock type models and are summarized in Table 14-8. Kriging of density into the model is acceptable, although assigning an average bulk density by rock type (oxidation state) within lithologies is preferable and simplifies reconciliations. Thus, the density block model was assigned using average bulk densities.

Table 14-8: Average Bulk Densities of the Mineralization, by Oxidation States within Lithologies

Lith Name	Type	Count	Mean	SD	CV	Minimum	Maximum
Diorite	All	162	2.77	0.085	0.031	2.52	2.98
	Oxide	10	2.74	0.042	0.015	2.67	2.80
	Transition	97	2.77	0.091	0.033	2.52	2.97
	Fresh	54	2.79	0.080	0.029	2.59	2.98
Granite	All	1065	2.57	0.091	0.035	1.85	2.89
	Oxide	117	2.54	0.099	0.039	1.85	2.71
	Transition	516	2.56	0.090	0.035	2.05	2.85
	Fresh	419	2.59	0.079	0.030	2.06	2.89
Quartz vein	All	58	2.62	0.093	0.035	2.35	2.90
	Oxide	7	2.62	0.042	0.016	2.54	2.67
	Transition	39	2.60	0.085	0.033	2.35	2.86
	Fresh	12	2.69	0.112	0.042	2.58	2.90
Sandstone	All	617	2.68	0.117	0.044	1.94	2.91
	Oxide	43	2.62	0.147	0.056	2.02	2.87
	Transition	373	2.66	0.124	0.047	1.94	2.89
	Fresh	192	2.72	0.077	0.028	2.21	2.91

14.2.8 Block Model Validation

To check that the interpolation of the block model correctly honored the drilling data, validation was carried out using the following steps:

- Swath Plots;
- Grade Comparison;
- Visual Validation;

14.2.8.1 Swath Plots

The drill composites were compared with the block model data by easting and elevation in the swath plots, shown in **Figure 14-14, Figure 14-15** and **Figure 14-16**. The swath plots were constrained by the mineralized envelopes. These plots highlight that the estimates compare very well with the composite grades with some smoothing of the interpolation resulting from a relatively unconstrained OK estimation technique.

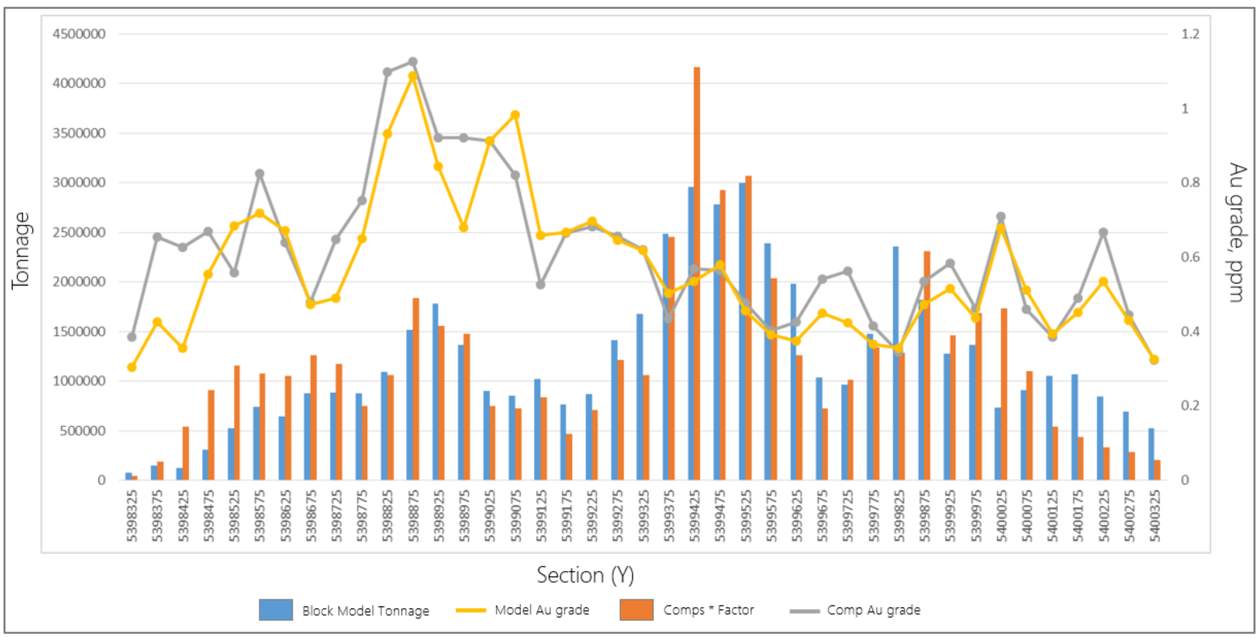


Figure 14-14: Swath plots on Northing

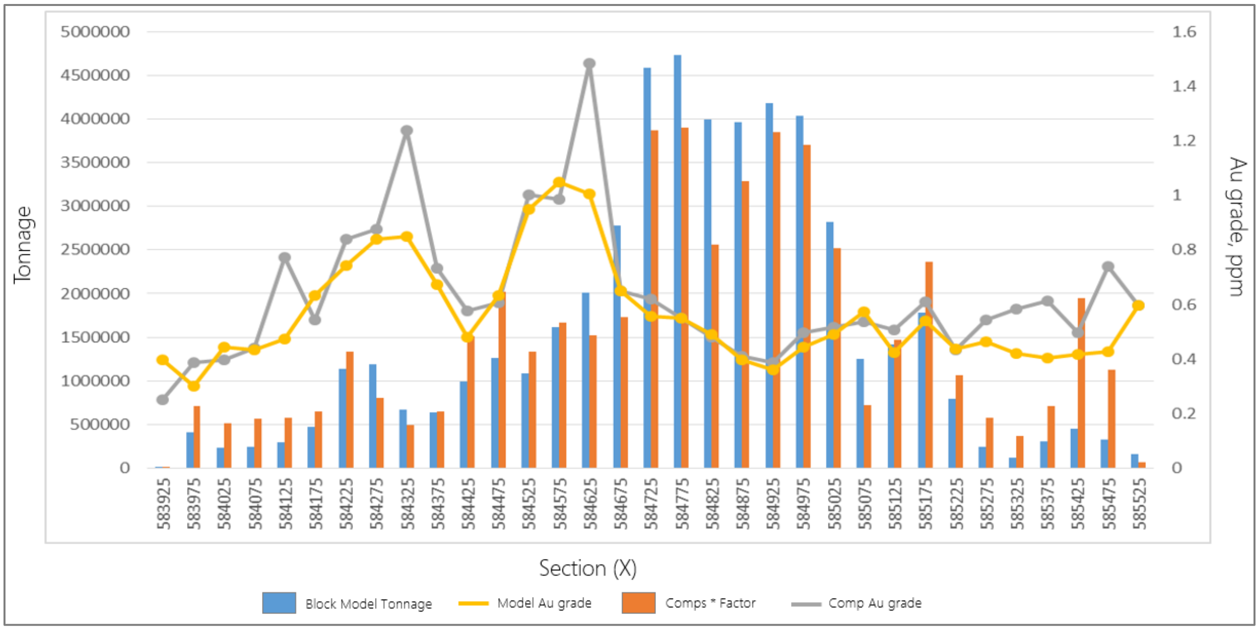


Figure 14-15: Swath plots on Easting

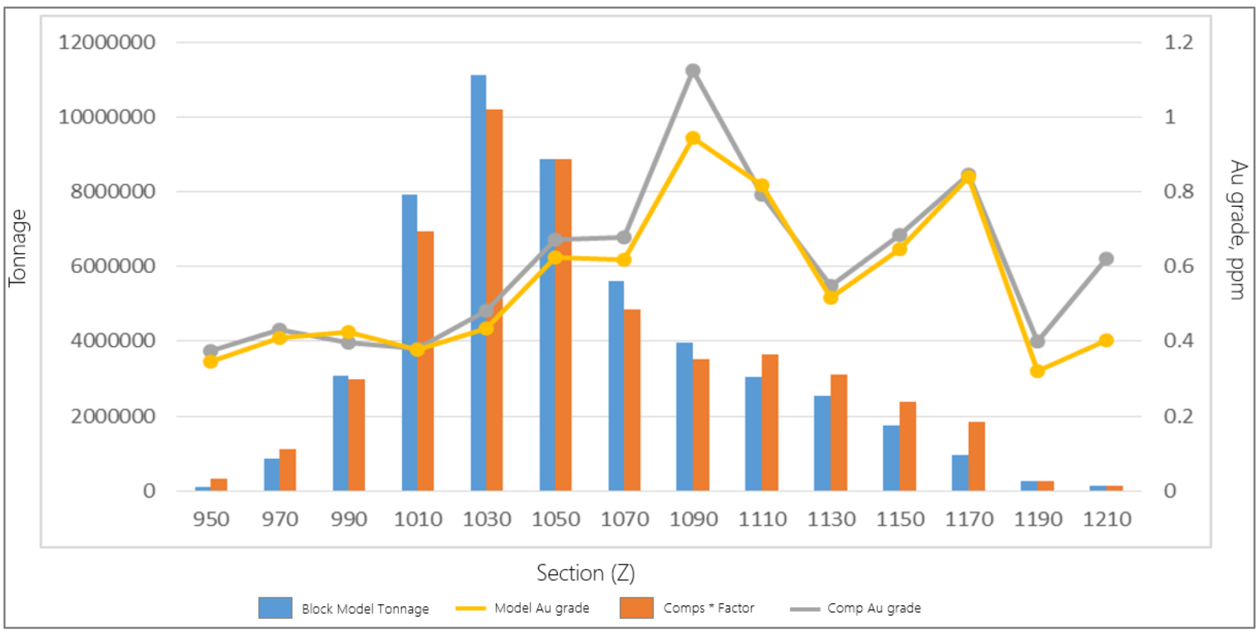


Figure 14-16: Swath plots on Elevation

14.2.8.2 Grade Comparison

Comparison of the block values and composites, results in a very good comparison (Table 14-9), with block model grade being slightly lower for than composite grades.

Table 14-9: Comparison of Block Estimates and Composites.

Domain	Block Model		Composites	
	Resource volume	Au, ppm	Number of Comps	Au, ppm
High Grade	3,602,063	1.516	975	1.760
Low Grade	14,575,563	0.352	4,725	0.385

14.2.8.3 Visual Validation

The visual comparison of block model grades against composite sample grade shows a strong correlation between the values. No significant discrepancies were apparent between the sections and the plans reviewed, yet some grade smoothing is apparent (Figure 14-17).

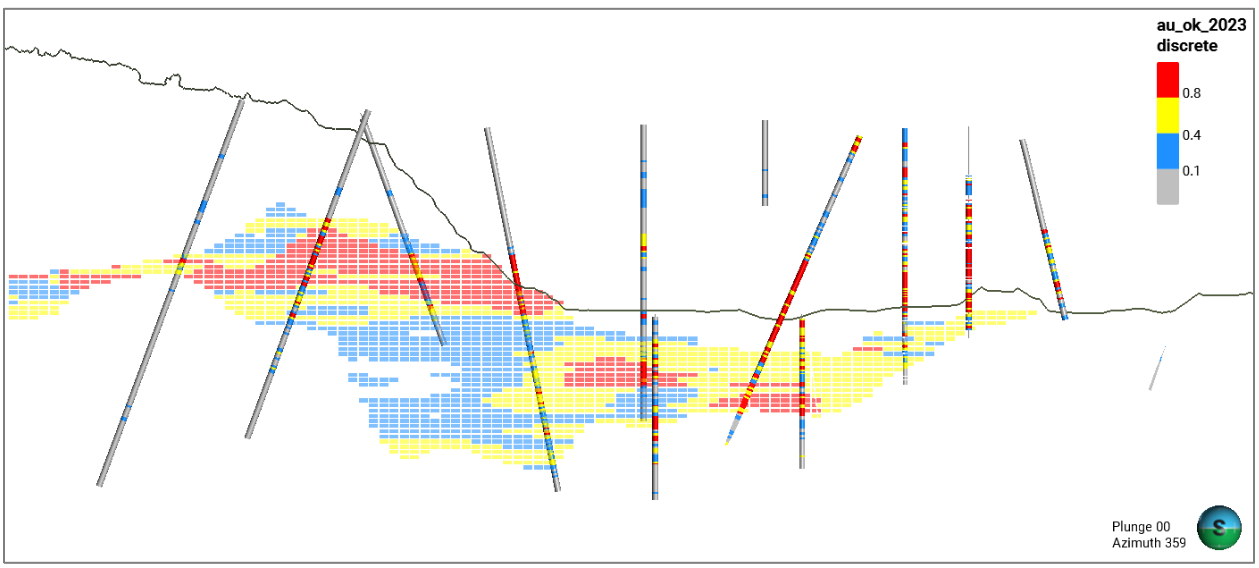


Figure 14-17: Boroo Visual Validation

14.2.8.4 Overall Validation

The review of the mathematical comparison indicates that a good correlation exists, as shown in the swath plots in Figure 14-14, Figure 14-15, Figure 14-16 and Table 14-9. This good correlation of the drillholes and interpolated block model is further supported when a visual inspection was completed. As a result of the validation completed Game Mine believes the estimate is representative of the composites, is indicative of the known controls of mineralisation and the underlying data.

14.2.9 Mineral Resource Classification

The updated mineral resource estimate presented in this Technical Report were prepared and disclosed in compliance with all disclosure requirements for mineral resources set out in the NI 43-101 Standards of Disclosure for Mineral Projects (2014).

A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.

Several factors are considered in the definition of a resource classification:

- NI 43-101 requirements;
- Canadian Institute for Mining, Metallurgy and Petroleum (CIM) Guidelines;
- Authors' experience with epithermal gold deposits;
- Spatial continuity based on variography of the assays within the drillholes;
- Drillhole spacing and estimate runs required to estimate the grades in a block;
- Observed mineralization on pit wall face;
- The confidence with the dataset based on the results of the validation; and
- The number of samples and drillholes used in each of the block estimations.

The confidence classification of the resource (Measured, Indicated, and Inferred) is based on an understanding of geological controls of the mineralization and the drillhole pierce point spacing in the resource area. Blocks were classified as Measured, Indicated, or Inferred if they were populated with grade during pass 1, pass 2, or pass 3 respectively during the interpolation process.

The classification methodology was largely based on the following:

- Measured Mineral Resources – blocks interpolated in the first search pass, where the oxidation state is well understood.
- Indicated Mineral Resources – blocks interpolated in the second pass, where the wider spaced drilling provides less confidence in the oxidation state.
- Inferred Mineral Resources – blocks interpolated in the third pass, where exploration data is wider spaced.

The Boroo Mineral Resource are classified by Game Mine in accordance with the guidelines of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources & Mineral Reserves Resource required by NI 43-101, that read in part as follows:

> *An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is*

reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation. An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit. Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation. A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

14.2.10 Mineral Resource Reporting

Table 14-10 outlines the total mineral resources of Boroo as of January 1, 2024.

Table 14-10. Boroo Mineral Resources as of January 01, 2024

Resource Category	Oxidation State	Tonnage (Kt)	Average grade (g/t)	Metal (oz)
Measured	Oxide	1,100	0.551	19,000
	Transition	3,000	0.586	57,000
	Fresh	22,500	0.59	427,000
	Total	**26,600**	**0.588**	**503,000**
Indicated	Oxide	500	0.518	9,000
	Transition	2,400	0.544	42,000
	Fresh	14,400	0.543	251,000
	Total	**17,300**	**0.542**	**302,000**
Meas + Ind	Oxide	1,600	0.54	28,000
	Transition	5,400	0.567	99,000
	Fresh	37,000	0.571	678,000
	Total	**44,000**	**0.57**	**805,000**
Inferred	Oxide	20	0.609	400
	Transition	160	0.455	2,400
	Fresh	1,120	0.842	30,000
	Total	**1,300**	**0.789**	**33,000**

Notes:

1. *Boroo Mineral Resources are as of January 1, 2024, based on the CIM Definition Standards (2014).*
2. *Mineral Resource were estimates has been compiled under the supervision of QP Tuvshinbayar Batbayar.*
3. *Mineral Resources that are not Mineral Reserves have no demonstrated economic viability.*
4. *Reporting cut-off grade for Boroo Mineral Resources is 0.1 g/t Au (include both heap leach and milling ore).*
5. *The Boroo mineral resources has been depleted for mining up to the mining (without backfilling) as of January 1, 2024.*
6. *The Mineral Resources are stated as in situ dry tonnes. All figures are in metric tonnes.*
7. *Figures have been rounded to the appropriate level of precision for the reporting of Mineral Resources.*
8. *Due to rounding, some columns or rows may not compute exactly as shown.*

14.2.11 Grade Tonnage Curves

Figures 14-18 and **Table 14-11** show the grade-tonnage curves for the Boroo block model.

Table 14-11: Tonnes and Grade

Cut Off Grade	Measured		Indicated		Inferred		Total	
	Tonnes, Mt	Au, ppm	Tonnes, Mt	Au, ppm	Tonnes, Mt	Tonnes, Mt	Au, ppm	Tonnes, Mt
0.1	26.609	0.588	17.318	0.542	1.308	0.789	45.235	0.576
0.2	22.794	0.659	14.992	0.604	1.027	0.961	38.813	0.646
0.3	17.691	0.778	11.202	0.725	0.811	1.149	29.704	0.768
0.4	13.122	0.929	7.849	0.887	0.630	1.380	21.601	0.927
0.5	9.701	1.101	5.489	1.079	0.492	1.643	15.682	1.110
0.6	7.451	1.270	4.163	1.250	0.432	1.798	12.046	1.282
0.7	6.150	1.403	3.422	1.382	0.374	1.978	9.946	1.417
0.8	5.289	1.511	2.949	1.485	0.349	2.066	8.587	1.525
0.9	4.599	1.611	2.589	1.574	0.315	2.201	7.503	1.623
1.0	4.003	1.710	2.245	1.671	0.289	2.317	6.537	1.723

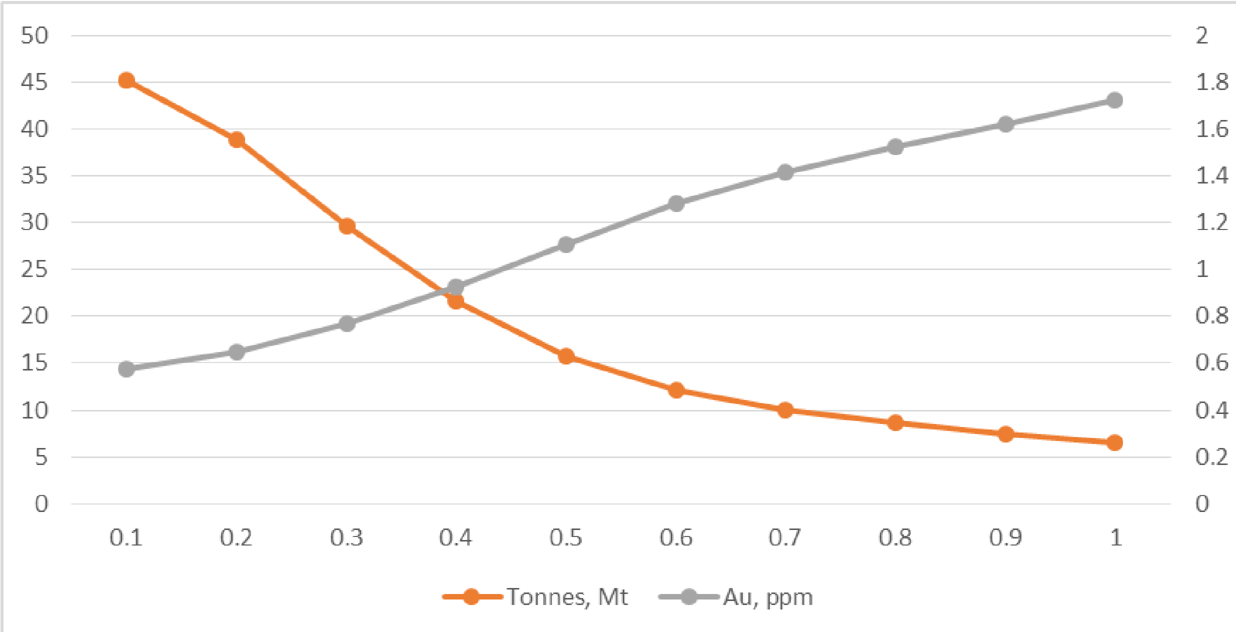


Figure 14-18: Grade Tonnage Curve

14.3 Ulaanbulag

14.3.1 Basis of Current Mineral Resource Estimate

Game Mine was commissioned by Boroo Gold LLC to generate an updated Mineral Resource Estimate for the Ulaanbulag Deposit. The update incorporates 72 additional drillholes (totalling 5,920.9 m) and structural interpretation study completed by Boroo Gold on the Property since the previously updated Mineral Resource Estimate with effective date May 01st, 2018 (Centerra, May 2018). The Ulaanbulag Mineral Resource Estimate is based on data from 257 RC and diamond drill holes, totaling 24,557.25 metres of drilling. The focus of the 2023 drilling program consisting of infilling drilling was to:

- Expand the understanding of the mineralization zones;
- Build upon the previous geological interpretation; and
- Improve drill spacing to show continuity of mineralization and increase overall confidence in the deposit.

Completion of the updated Ulaanbulag Resource involved the assessment of an update drill hole database and subsequent generation of an updated geological model, updated structural control model, an updated three-dimensional (3D) grade model. The QP visited the property from August 30 to August 31 2023. The effective date of the Ulaanbulag Mineral Resource Estimate is February 01st, 2024. Ordinary Kriging (OK) restricted to a mineralized domain was used to interpolate gold grades (g/t) into a block model. Measured, Indicated and Inferred Mineral resources are reported in summary tables in Section 14.3.10.

14.3.2 Previous NI 43-101 Mineral Resource Estimate

A Mineral Resource Estimate with an effective date of May 01, 2018, was developed for the Ulaanbulag deposit. This mineral resource estimate was based on 18,636.35 m of drilling from 146 diamond, and 39 RC drillholes.

The block model was developed using wireframing method and was classified with Measured, Indicated, and Inferred Resources in accordance with CIM definitions and standards (2014).

Table 14-12 shows the previous Mineral Resource Estimate which is superseded by the current Mineral Resource Estimate stated in Section 14.3.6 and should no longer be relied upon as current.

Table 14-12: Previous Mineral Resource Estimate for the Ulaanbulag Deposit estimated by Centerra Gold as of May 1, 2018

Resource Category	Oxidation State	Tonnage (kt)	Average Grade (@Au g/t)	Metal (koz)
Indicated	Oxide	-	-	-
	Transition	-	-	-
	Fresh	276	1.78	16
	Total	**276**	**1.78**	**16**
Inferred	Oxide	774	1.52	36
	Transition	977	1.44	45
	Fresh	6	1.53	0
	Total	**1,757**	**1.45**	**82**
Indicated + Inferred		**2,004**	**1.52**	**98**

Notes:

1. Ulaanbulag Mineral Resources are as of May 1, 2018, based on the CIM Definition Standards (2014).
2. Mineral Resources are constrained within an optimized $1,450 Au resource pit shell.
3. Mineral Resources are estimated based on a cut-off grade of 0.75 g/t Au for oxide, 0.87 g/t Au for transition and 1.07 Au g/tm for fresh.
4. Inferred Mineral Resources have a great amount of uncertainty as to their existence and as to whether they can be mined economically. It cannot be assumed that all or part of the Inferred Mineral Resources will ever be upgraded to a higher category.
5. A conversion factor of 31.10348 grams per ounce of gold is used in the reserve and resource estimates.
6. Numbers may not add up due to rounding.

14.3.3 Comparison to Previous Mineral Resource Estimate

A comparison of the current Ulaanbulag Deposit updated Mineral Resource Estimate completed by Game Mine and the 2018 Mineral Resource Estimate completed by Centerra Gold is presented in Table

14-13. The 2018 mineral resource estimate for Ulaanbulag has been superseded by the 2024 mineral resource estimate, and as such, should no longer be relied upon.

The increase in gold ounces along with the higher level of confidence in the mineral resource estimate is the result of additional diamond drilling, and updated modeling approaches, whereby high-grade domains were re-modeled based upon this drilling and structural information. The 2024 model was able to utilize a significant dataset of structural measurements which has subsequently increased the understanding of the deposit, specifically the orientation and continuity of higher grade mineralized zones.

Table 14-13: Comparison of the 2018 and 2024 Ulaanbulag Deposit Mineral Resource Estimates

Resource Category	Cut-Off Grade (@Au g/t)	Tonnage (kt)	Average Grade (@Au g/t)	Metal (kg)
2018 Measured	0.75, 0.87, 1.07	-	-	-
2024 Measured	0.10	4,472.38	0.600	2683
2018 Indicated	0.75, 0.87, 1.07	276.00	1.780	491
2024 Indicated	0.10	7,966.48	0.468	3728
2018 Inferred	0.75, 0.87, 1.07	1,757.00	1.450	2548
2024 Inferred	0.10	4,649.23	0.370	1720

Notes:
The 2018 and 2024 mineral resource estimate have utilized different modeling techniques, utilized different input parameters, and have been reported at different cut-off grades. The 2018 mineral resource estimate for Ulaanbulag has been superseded by the 2024 mineral resource estimate, and as such, should no longer be relied upon.

14.3.4 Database

In order to complete an updated Mineral Resource Estimate for Ulaanbulag, a database comprising a series of excel spreadsheets containing drillhole information was provided by Boroo Gold to the QP. The database includes hole location information (UTM WGS 84, Zone 48N), survey data, assay data, lithology data, oxidation, alteration, bulk density data, and structural data.

Table 14-14: Summary of Ulaanbulag Drilling Database

Record Numbers	Drill Holes		Total
	DD	RC	
Meters drilled	19,882.25	4,675	24,557.25
Collars Records	218	39	257
Survey Records	262	56	318
Assay Records	10,480	2,337	13,177
Lithology Records	3,047	373	3,420
Oxidation Records	1,078	192	1,270
Alteration Records	2,598	670	3,268

The data was verified (Section 12) and then imported into Geovia Surpac 2021 and Seequent Leapfrog Geo3D version 2023.2 software ("Leapfrog") for geological modeling and the development of the grade wireframes. Overall, information for 257 drillholes was provided to Game Mine. The particulars of the information provided to Game Mine are presented above in Table 14-14. A location plan map for the Ulaanbulag drill holes, colored by drill hole types and dates are presented in Figure 14-19.

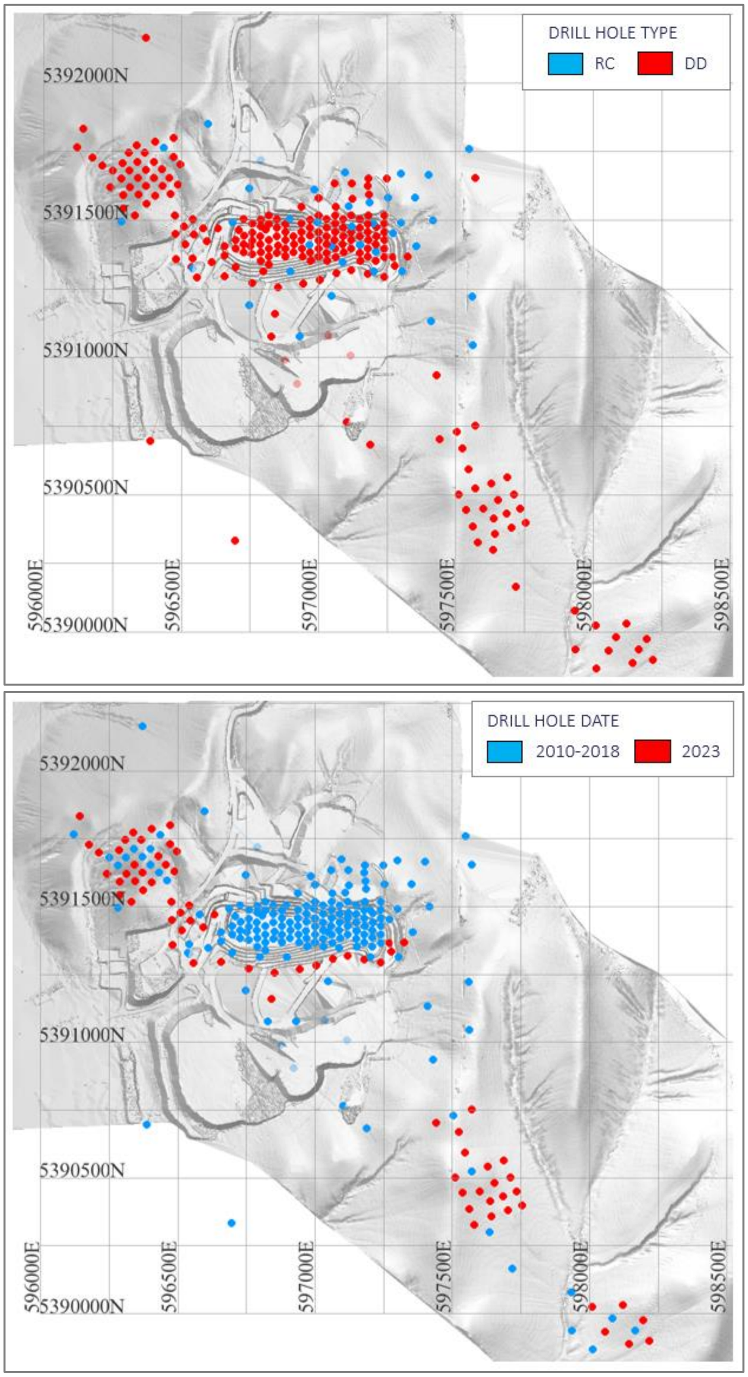


Figure 14-19: Location map of Ulaanbulag drillholes

14.3.5 Geological and Resource Interpretation

14.3.5.1 Geological Interpretation

In order to better model and geologically constrain the mineralization, a 3D geological model was constructed in Leapfrog Geo 3D prior to any resource interpretation. Geology model was created in order to control gold mineralization and rock density statistics and geotechnical domain. Four main lithology units were modelled including sandstone (ST), granite (GR), diorite (DI) and quaternary sediment (Q).

An isometric of the lithological model is provided below in **Figure 14-20** and **Figure 14-21** respectively.

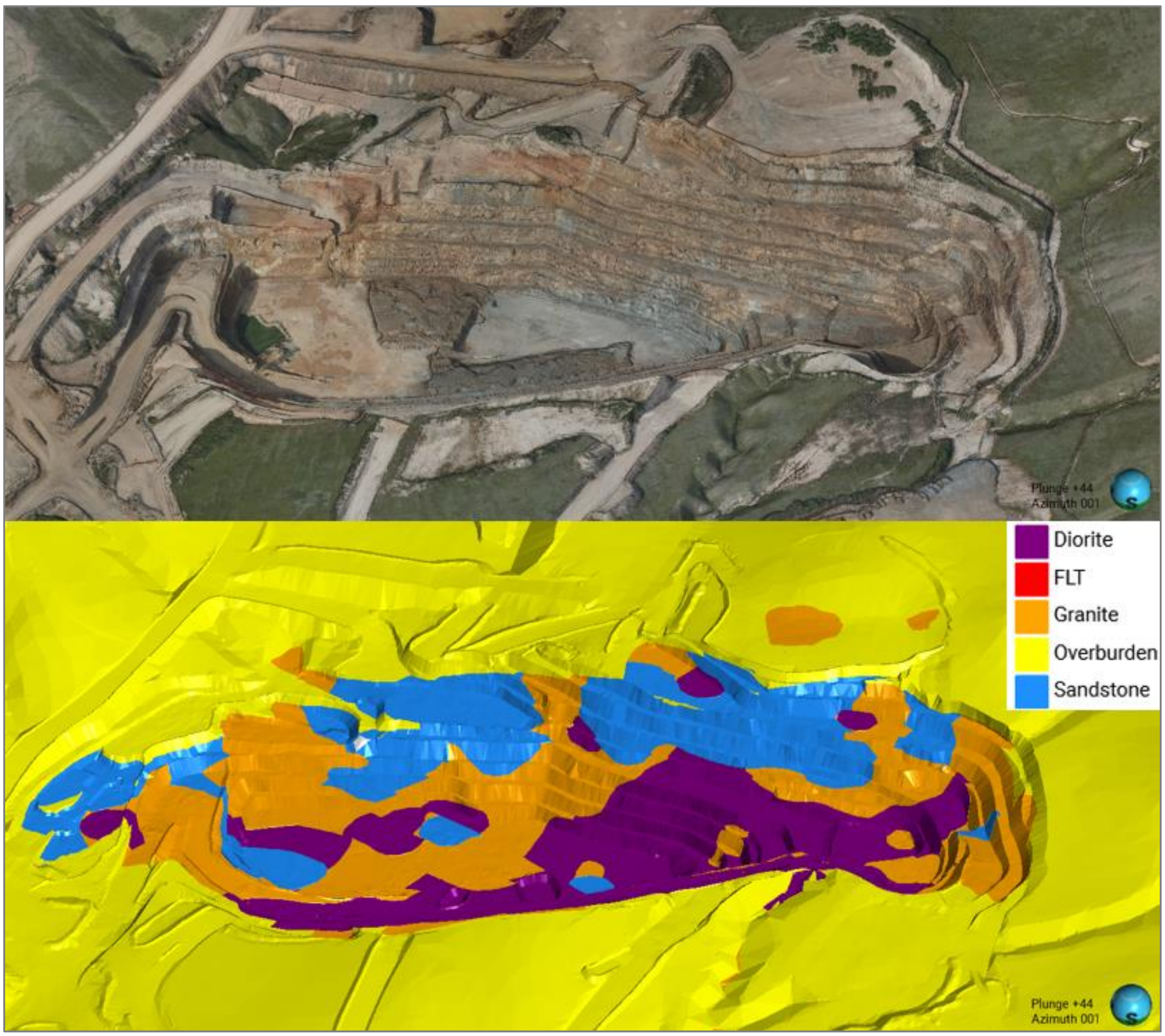


Figure 14-20: Comparison of drone survey data and lithological model

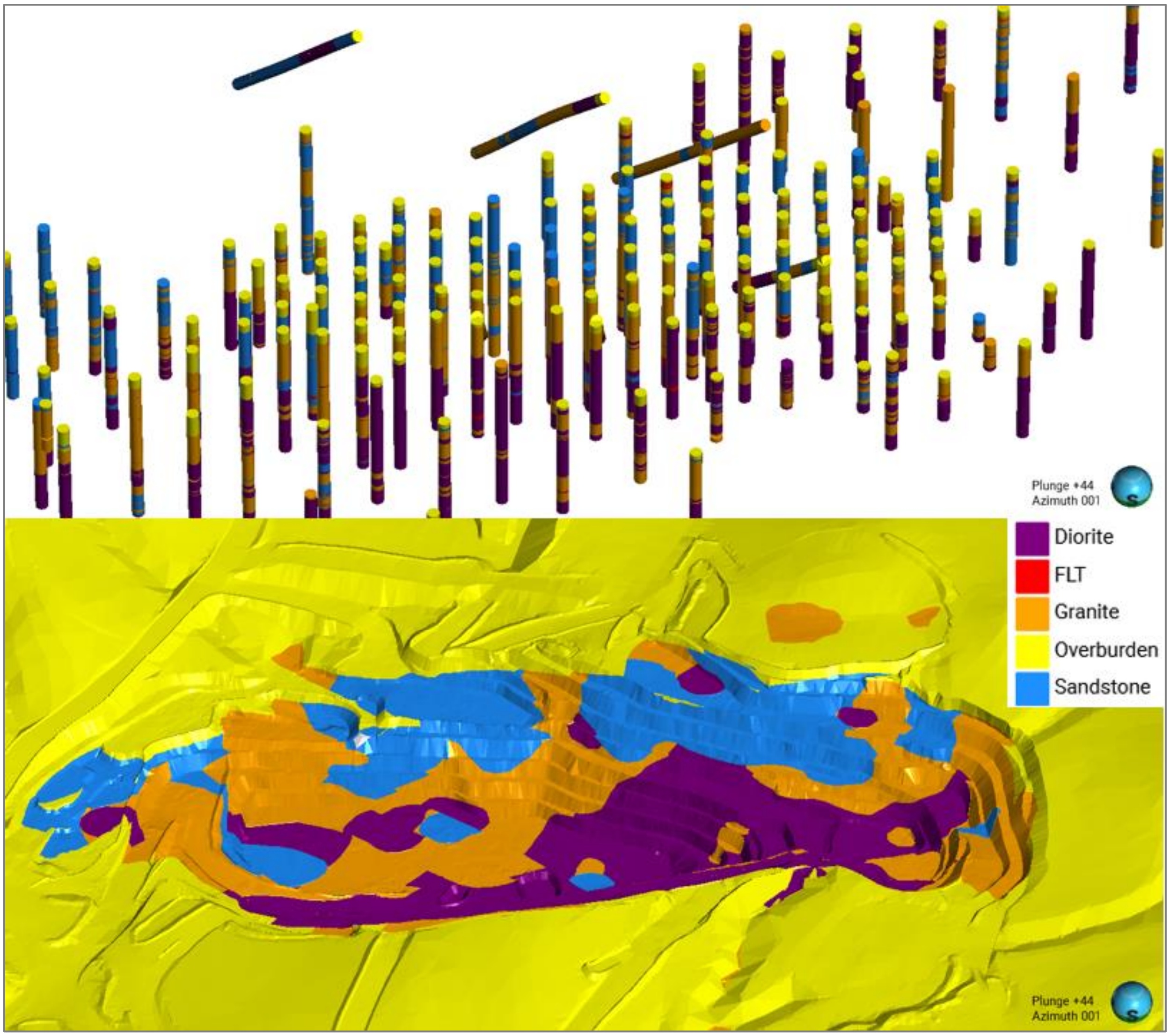


Figure 14-21: Comparison of borehole lithology data and lithological model

14.3.5.2 Oxidation Interpretation

Oxidation level of three-dimensional (3D) solids namely: oxide (Wox), transitional (Wpx) and fresh (Fr) material were created using drillhole oxidation logging information to code oxidation block model, density and gold distributions and geotechnical domain (Figure 14-22). The oxide zone, generally located near surface, has undergone the highest degree of oxidation, followed by a transitional zone, and then the underlying fresh rocks in the primary zone of mineralization.

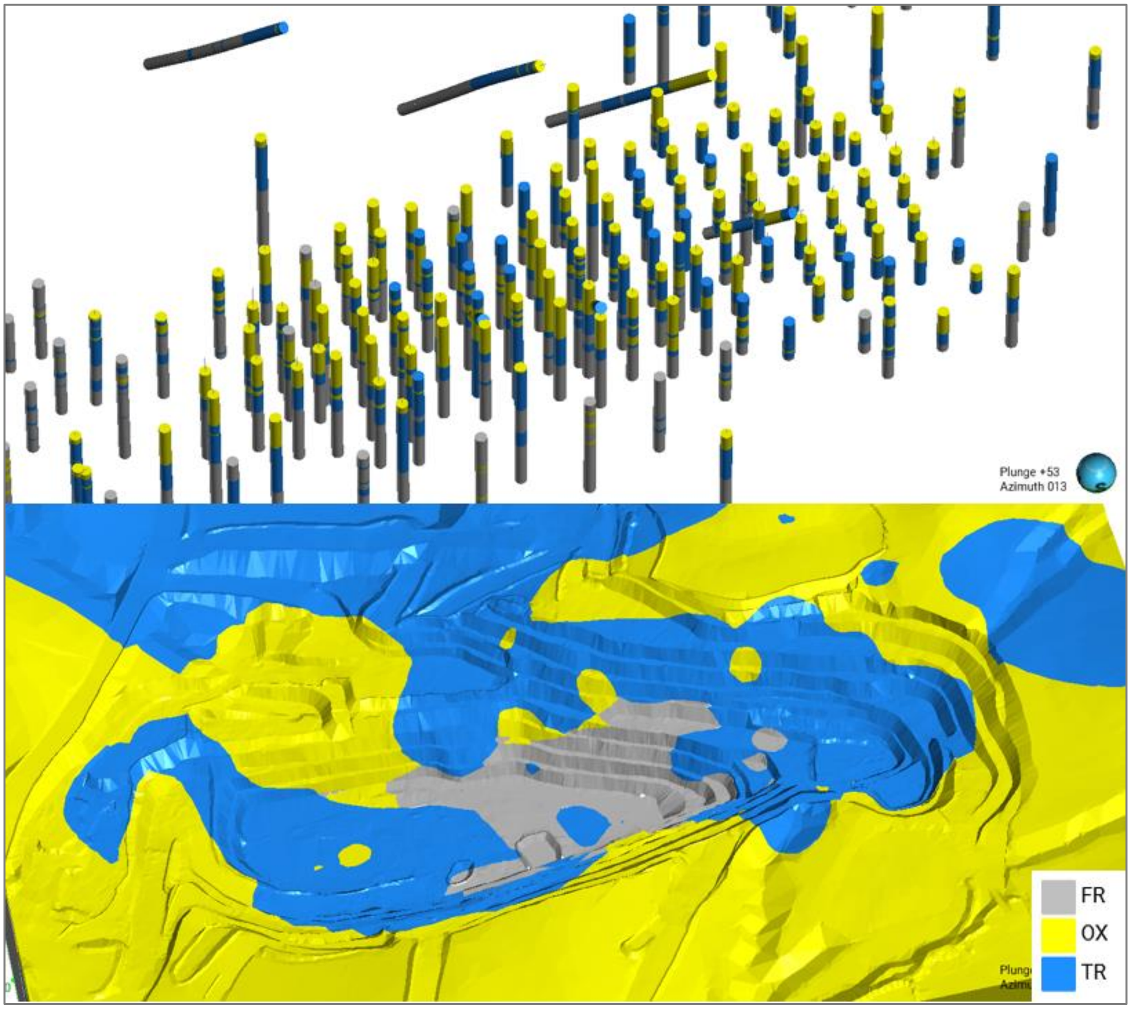


Figure 14-22: Comparison of borehole oxidation data and oxidation model

14.3.5.3 Mineralization Interpretation

The Ulaanbulag deposit has been divided into four separate mineralized zones or pit areas from North West to South East, namely Zone 1, 2, 3 and 4 (Figure 14-23 and Figure 14-24). Given the strong lithological and structural influence on mineralization, lithological and structural model was developed in Leapfrog Geo3D and Geovia Surpac to guide the interpretation of the mineralization model. All mineralized wireframes were trimmed to topography and pit survey data (01 January 2024).

Figure 14-23: Four separate mineralized zones

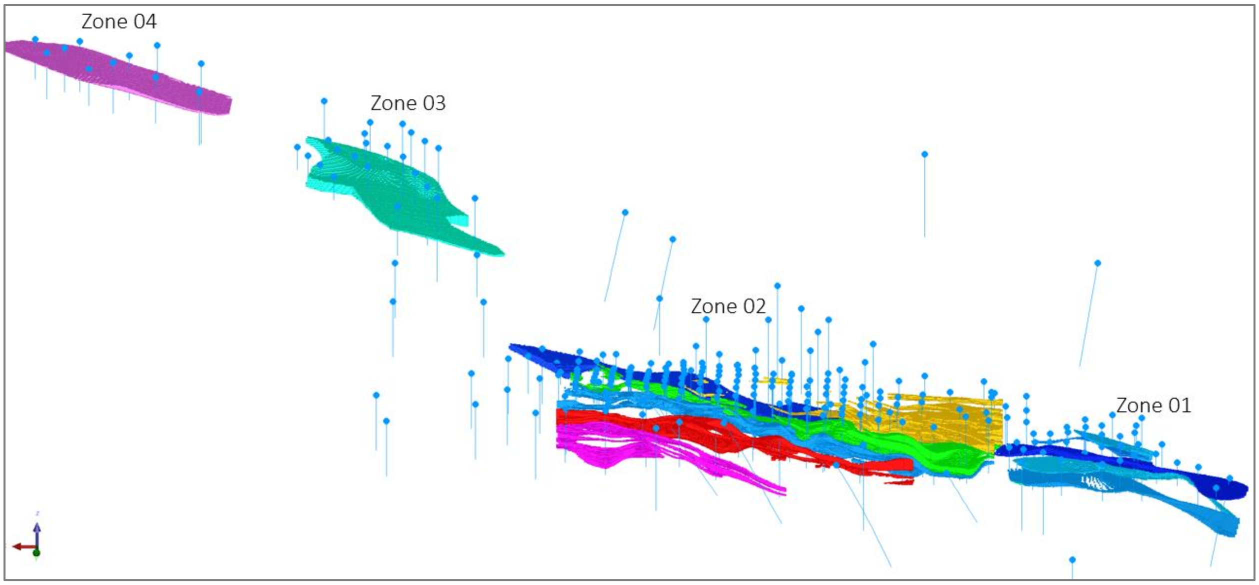


Figure 14-24: Isometric view of four separate mineralized zones

14.3.6 Exploratory Data Analysis

Upon completion of the geological model, grade solids were imported into Leapfrog Edge version 2023.2 where numerical modeling and estimation was undertaken. The sections below summarize details associated with the various aspects of the numerical modeling and estimation process.

14.3.6.1 Assays

The assay intervals within each domain were captured using a Leapfrog Edge Estimation into individual drillhole files. These drillhole files were reviewed to ensure all the proper assay intervals were captured. The non-assayed intervals were given a zero (0) value. Gold assay statistics between raw and composite samples for each zone are summarized in Table 14-15.

14.3.6.2 Compositing

Sampling was undertaken at varying lengths within the Boroo deposit. Game Mine reviewed all sample lengths. The raw length statistics showed 1.35 m to be the mean length of sample intersect, with a median of 1.00 m and a range between 0.02 m and 19.01 m (Figure 14-25).

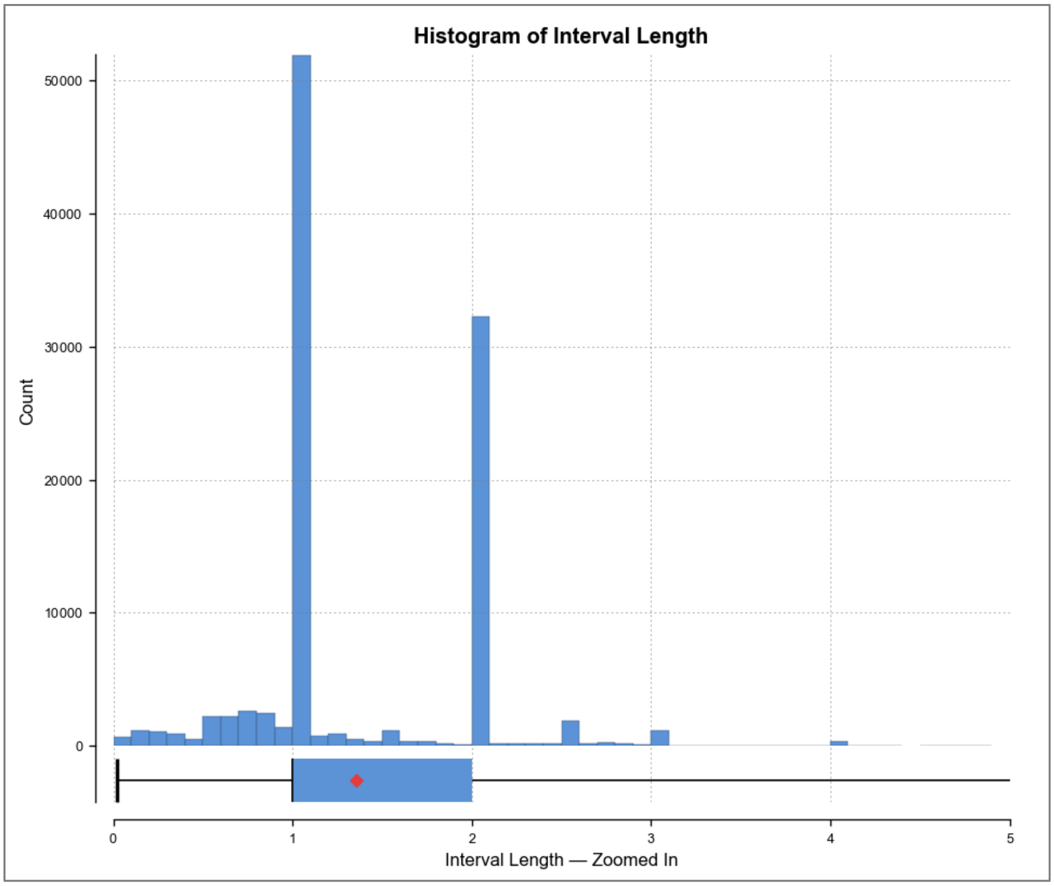


Figure 14-25: Histogram sample lengths

The technical report (Centerra, 2009) states that the samples were compositing to 2.5 m within the mineralization domains, defined by the standard bench definitions for the Boroo open pit. Missing assay data were ignored in the length weighted averaging. Residual composites <1 m was removed prior to estimation.

Table 14-15: Raw and Composite Sample Statistics

Comp Length	Raw	2.0 m	1.25 m (bench comp)
Count	3497	2492	3898
Mean	0.67	0.64	0.64
Standard deviation	1.33	1.13	1.20
Coefficient of Variation	1.98	1.78	1.87
Variance	1.79	1.27	1.45
Minimum	0.005	0.01	0.005
Median	0.33	0.34	0.35
Maximum	38.24	29.34	36.63

Game Mine compared the mean Au g/t grades for compositing to 1m and 2.5m within the modelled mineralization. Game Mine considers compositing to 2.5 m as reasonable (fits to mining benches) and removal of the residuals as good practice that limits any potential bias in the sample support during estimation.

14.3.6.3 Capping Analysis

Composited assay data was examined for each domain to assess the amount of metal that is at risk from high-grade assays. The Leapfrog Edge module was used to determine if grade capping was required. Capping was based on examination of the log probability plots and histograms for each metal and caps were applied where graphs showed significant outlier influence or deviation from the general trend-line (Figure 14-26). Log probability plots and Cumulative frequency plots were inspected for Zone 1, Zone 2 domain and applied the capping grades using a combination of histograms, probability plots, and decile analyses. Coefficients of variation (CV) after applying capping were showing decreases. Zone 3 and Zone 4 don't need any high grade capping.

Table 14-16 shows a summary of the top cuts that were applied to the domain datasets. The QP is of the opinion that the capping levels are reasonable, and suitable for the estimation of Mineral Resources.

Table 14-16: Boroo Capped Composite Statistics

Domain	Zone 1		Zone 2		Zone 3		Zone 4	
	Uncapped	Capped	Uncapped	Capped	Uncapped	Capped	Uncapped	Capped
Count	293	293	3,643	3,643	134	-	51	-
Mean	0.686	0.593	0.615	0.518	0.414	-	0.219	-
SD	1.763	0.892	0.190	0.888	0.474	-	0.110	-
CV	2.570	1.500	1.933	1.483	1.145	-	0.502	-
Variance	3.111	0.795	1.417	0.789	0.225	-	0.012	-
Min	0.000	0.000	0.005	0.005	0.005	-	0.06	-
Q1	0.108	0.131	0.151	0.151	0.133	-	0.141	-
Q2	0.264	0.259	0.320	0.320	0.236	-	0.185	-
Q3	0.637	0.673	0.671	0.671	0.441	-	0.290	-
Max	18.488	5.000	36.630	10.000	2.572	-	0.526	-

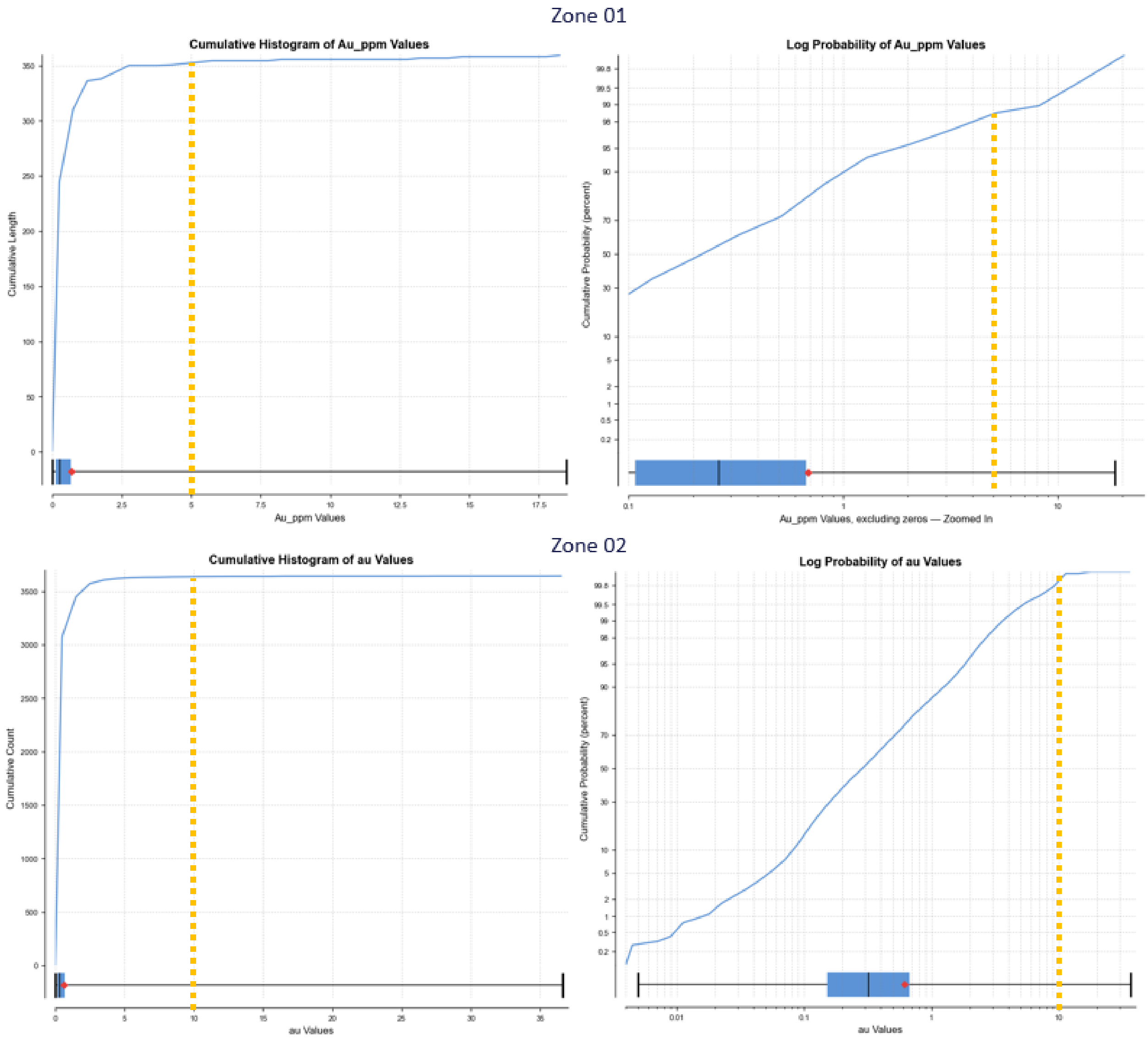


Figure 14-26: Log probability plots and Cumulative frequency plots by Zone 1 and Zone 2

14.3.6.4 Variogram Assessment

Downhole and directional variography was undertaken by Game Mine using Leapfrog Edge software to quantify the spatial relationship of composited data within the selected mineralized domains. Variography was used to assess grade continuity and spatial variability in the estimation domains and to determine sample search distances and kriging parameters for block grade estimation.

Downhole variograms were used to determine the nugget effect, then the variograms were modeled with two structures to determine spatial continuity in the zones. The results of the variography analysis are summarized below in Table 14-17 and Figure 14-27 present the variograms for each the major, semi-major, and minor axes respectively. Overall, the best variogram was achieved in the plane of the orebody.

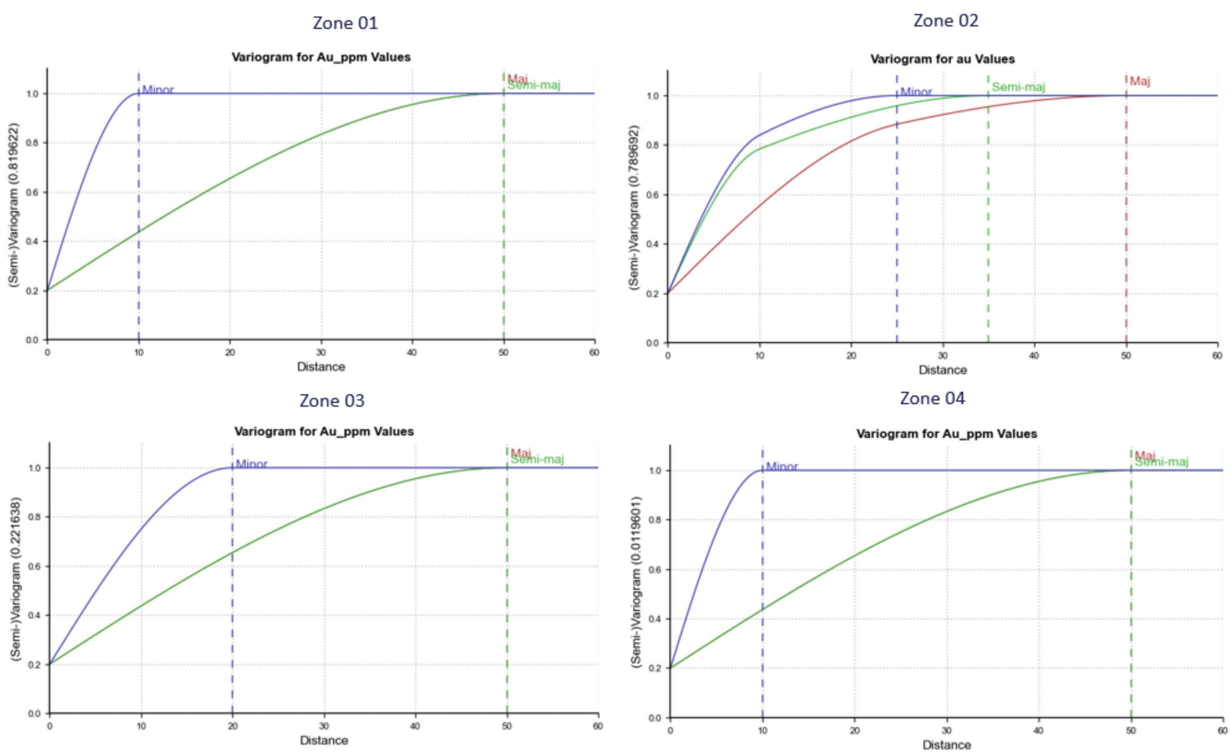


Figure 14-27: Variogram Model

Table 14-17: Variogram analysis parameters

Domain	Type	Nugget	Major	Semi-Major	Minor	Dip	Dip Azimuth	Pitch
Zone 1	Spherical	0.2	50	50	10	25	190	65
Zone 2	Spherical	0.2	50	35	20	17	270	112
Zone 3	Spherical	0.2	50	50	20	20	235	70
Zone 4	Spherical	0.2	50	50	10	20	230	10

14.3.6.5 Resource Block Model

Game Mine created the block models with a parent block size of 5 mN x 5 mE x 2.5 mRL, with no sub-blocking and no rotation. The block model parameters are shown in **Table 14-18**. The original block model was created for the grade interpolations and resource classifications in Leapfrog and exported to Geovia Surpac. The drill sections are nominally on 20 m x 20 m spacing, and the mineralization strikes northeast – southwest.

Table 14-18: Block model parameters

Model Name	ulaanbulag_2023_v01.mdl		
	Y	X	Z
Minimum Coordinates	5389700	566000	750
Extent	2200	2400	300
Rotation	0	0	0
Parent Block Size (m)	5	5	2.5
Sub Block Size (m)	2.5	2.5	1.25
Attributes			
au_ok	Reportable au ppm grade (Ordinary Kriging)		
rescat	Classification (Measured, Indicated, Inferred)		
grade_shell	High grade, low grade domain		
oxidation	Oxidation state (Oxide, Transition, Fresh)		
litho	Lithology type (Granite, Diorite, Sandstone etc.)		
density	Bulk density (t/bcm)		
veins	Ore veins		
zone	Mineralization zones		

14.3.6.6 Dynamic Anisotropy

Due to the local changes in domain orientations and local anastomosing nature of the wireframes a single search ellipse would not truly reflect the distribution of mineralization within the domain. To accommodate the changing orientation of the domains, dynamic anisotropy was utilized to better represent continuous mineralization within the mineralized structures.

Dynamic anisotropy is a function in Leapfrog Edge Variable Orientation that permits the search ellipse orientation to be continuously adjusted to match the tangent of the average orientation of the mineralized domain to mimic curvature or folding structures (Figure 14-28). This allows the search volume to be oriented to follow the trend of mineralization. The azimuth of the major and semi-major axes and the search dimensions remained unchanged within the search ellipse.

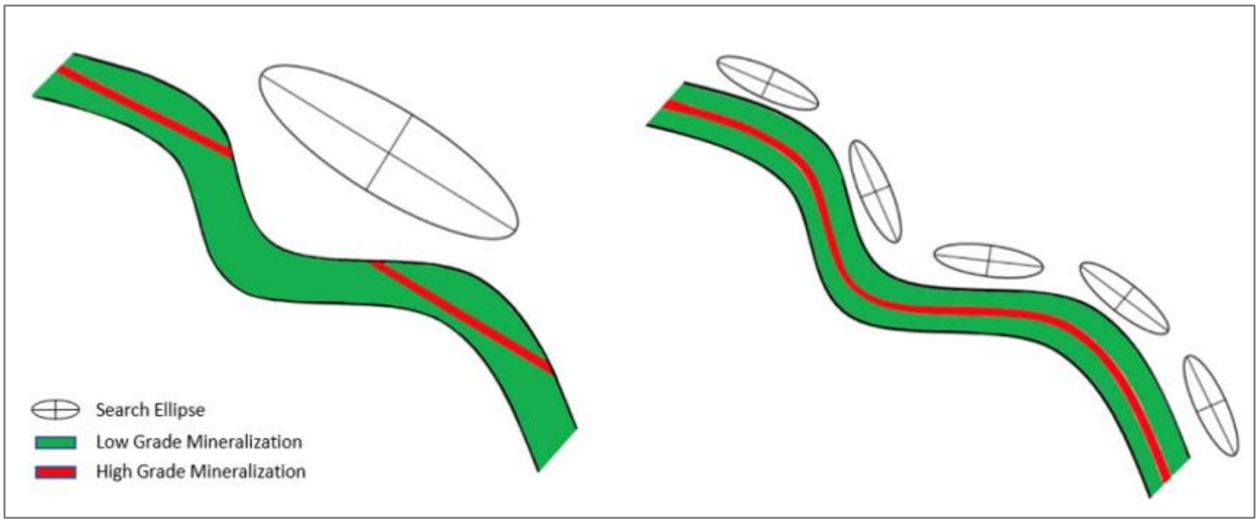


Figure 14-28: Schematic of Dynamic Anisotropy

(Red-High Grade Mineralization, Green-Low Grade Mineralization)

14.3.6.7 Estimation and Search Parameters

The Ordinary Kriging (OK) algorithm was selected for grade interpolation due to the number of samples and the interpreted geospatial analysis. The OK algorithm was utilised to minimise over smoothing within the estimate. The estimations were designed as a three-pass system which were run independently within each individual wireframe using composite data constrained within domain. This process involves the estimation being performed three times, where two expansion factors are used. During each individual estimation run this factor increases the size of the search ellipse used to select samples. This method ensures that blocks which are not estimated and populated with a grade value in the first run, are populated during one of the subsequent runs. The minimum number of samples required within the search neighbourhood for an estimate to be interpolated and a maximum number was specified. The minimum number of samples from any one drill hole was also specified. No smoothing is applied in this estimation.

The final estimated Au grade block model is illustrated in Figure 14-29 and Figure 14-30.

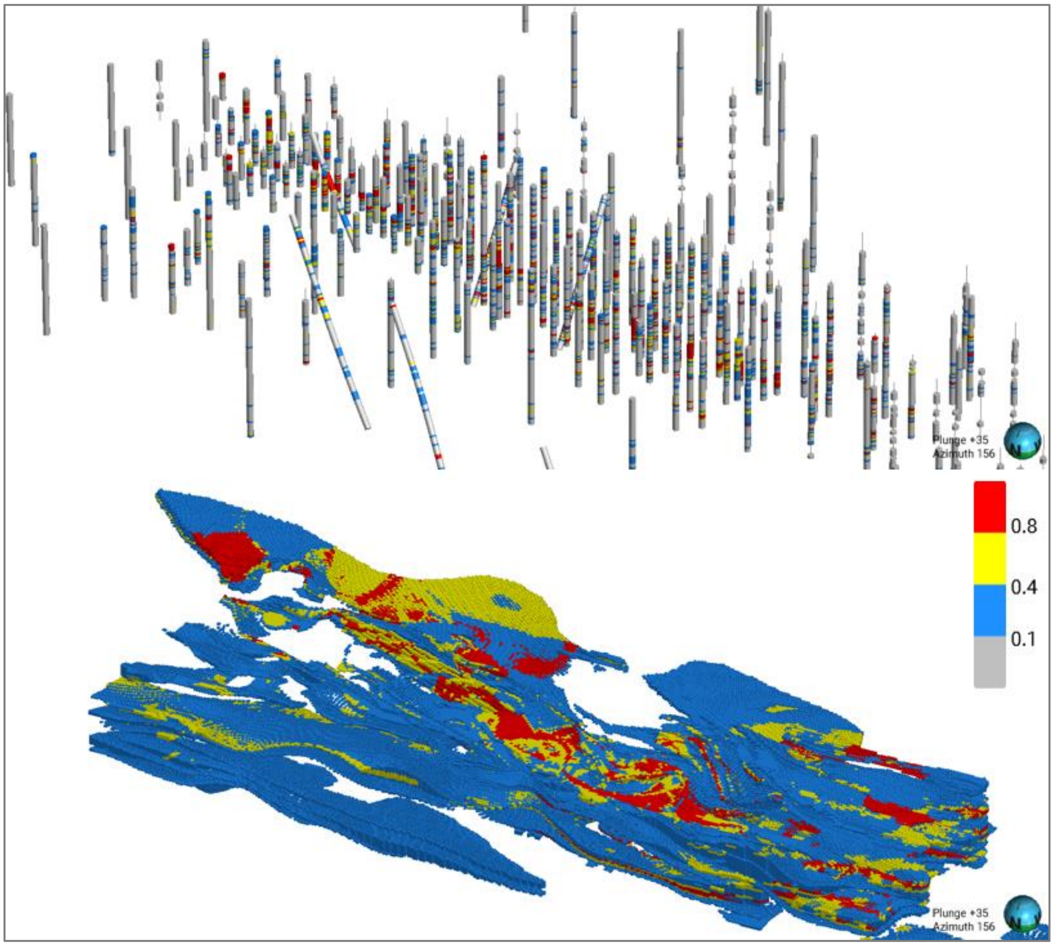


Figure 14-29: Au grade (ppm) block model illustration in 3D view

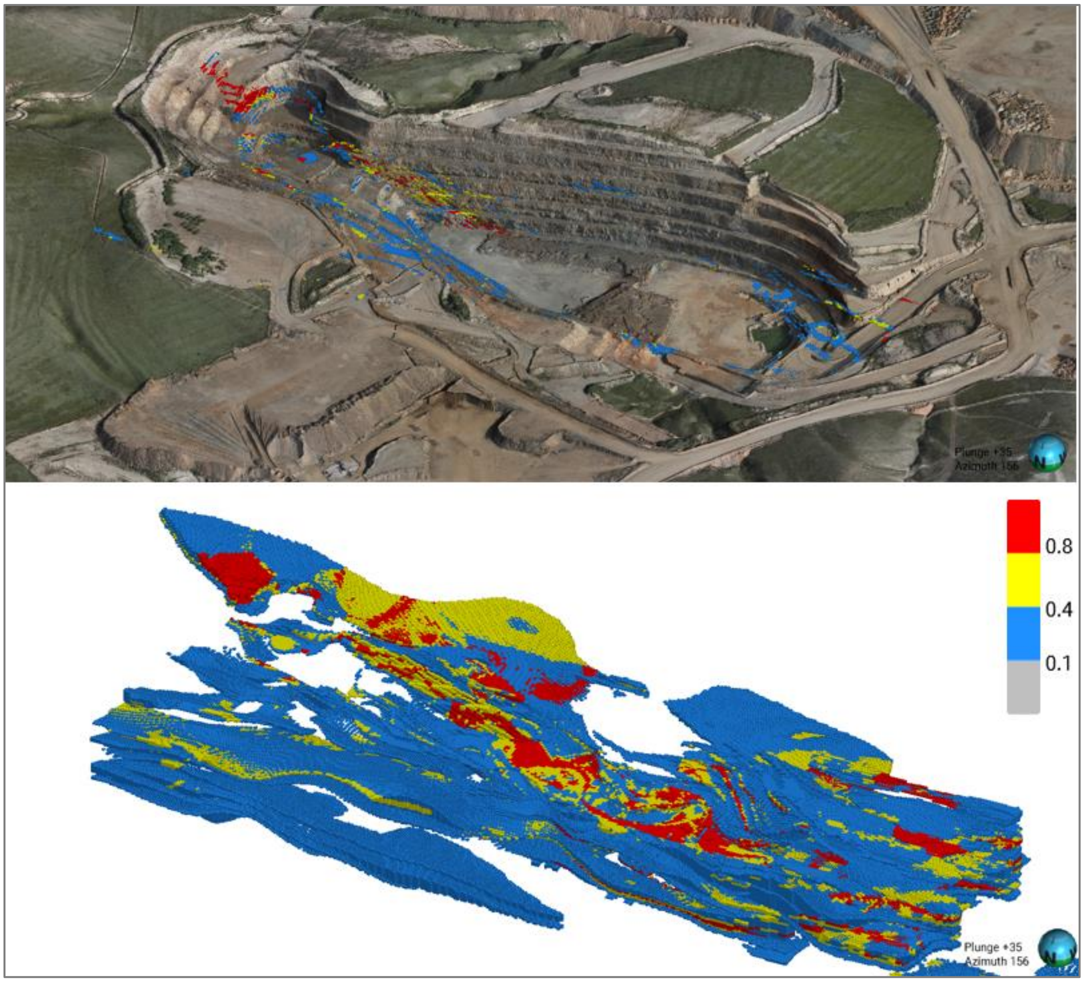


Figure 14-30: Comparison of drone survey data and Au grade (ppm) block model

14.3.7 Specific Gravity

Density assignment is a critical component in the estimation of a reliable Mineral Resource tonnage. Centerra's 2018 Technical Report states that a total of 70 samples were collected from drill core. Samples for bulk density determination were taken from the main lithology types and oxide state.

Game Mine received a total of 70 density data (Centerra Gold reviewed). The density values were assigned into the block model using the average assayed values for each oxide, transition and fresh. **Table 14-19** summarizes the specific gravity data by oxide domain.

Table 14-19: Average Bulk Densities of the Mineralization, by Oxidation States within Lithologies

Domain	Count	Minimum	Maximum	Mean	SD	CV
Oxide	17	2.21	2.61	2.47	0.11	0.05
Transition	24	2.23	2.66	2.53	0.09	0.04
Fresh	29	2.32	2.73	2.56	0.09	0.04
Total	70	2.21	2.73	2.53	0.10	0.04

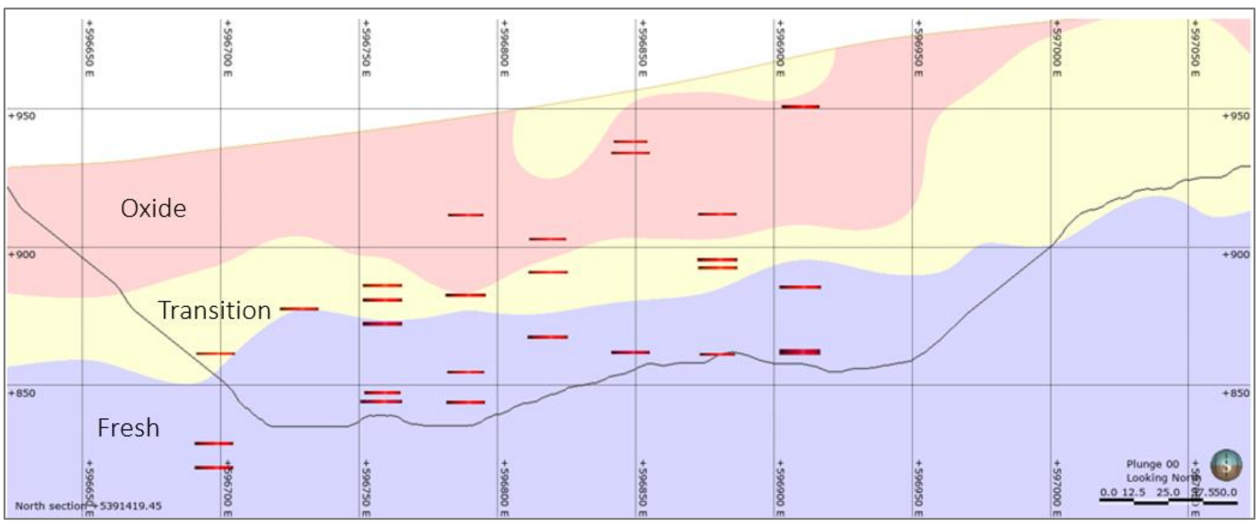


Figure 14-31: Longitudinal section looking north - bulk density sample distribution

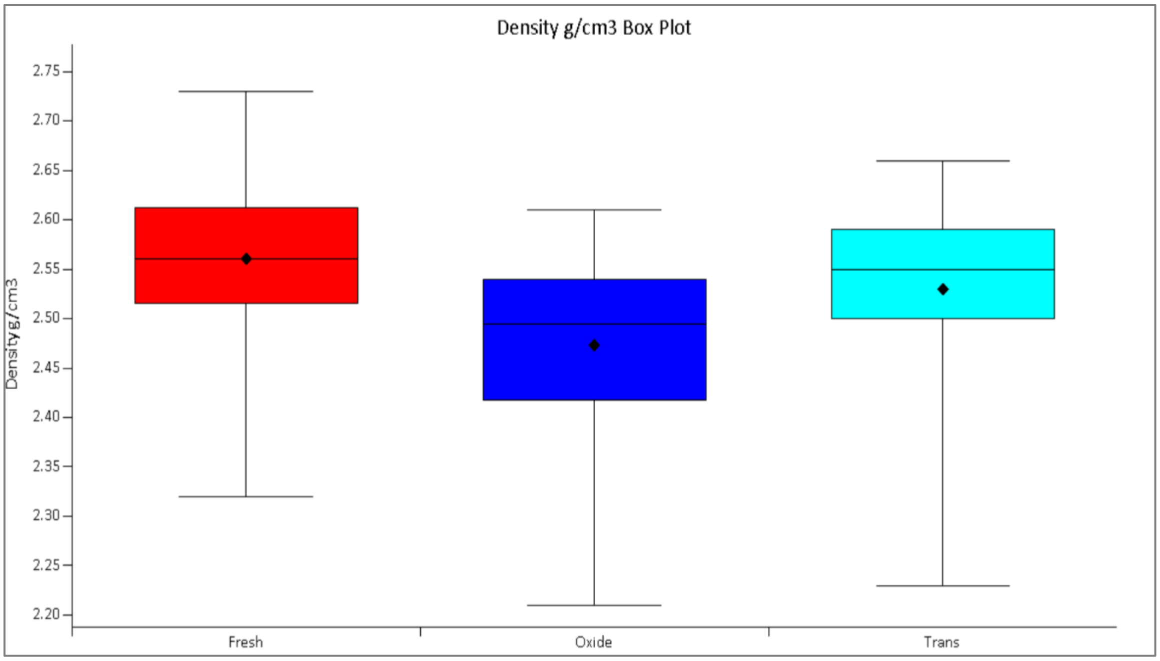


Figure 14-32: Bulk density box plot statistics by oxide state

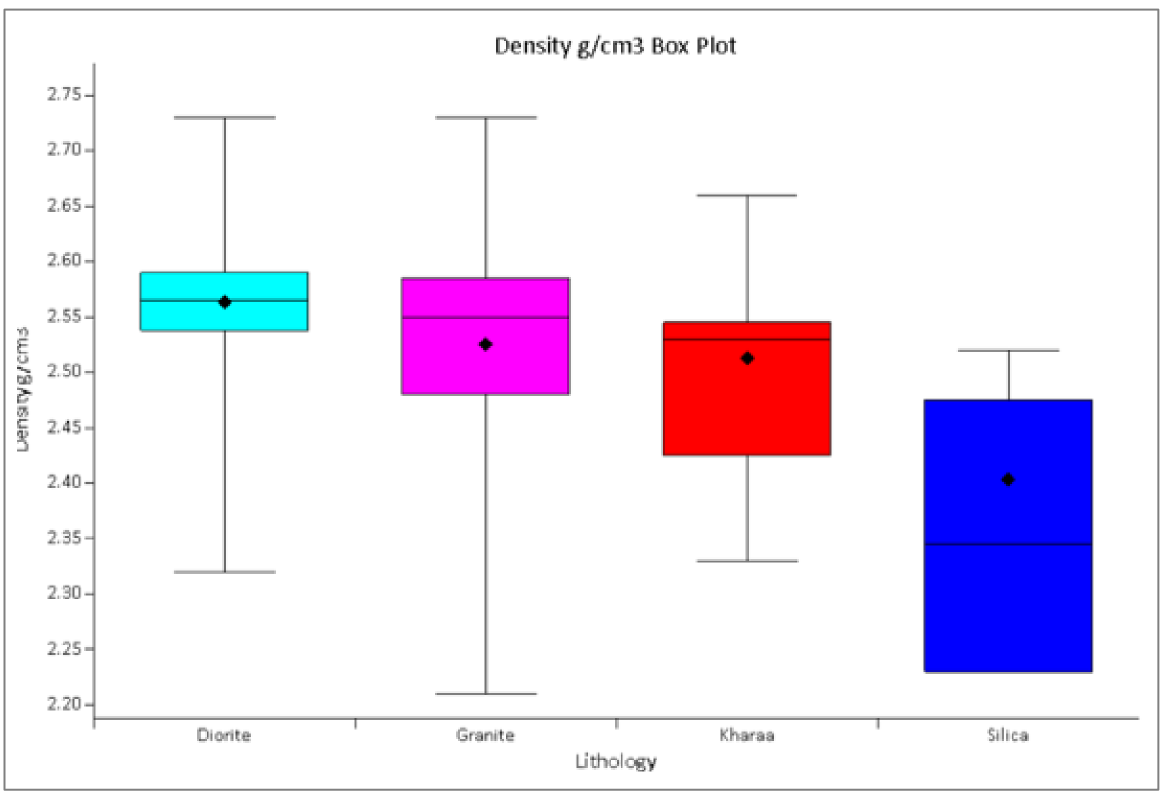


Figure 14-33: Bulk density box plot statistics by lithology

14.3.8 Block Model Validation

To check that the interpolation of the block model correctly honored the drilling data, validation was carried out using the following steps:

- Swath Plots;
- Grade Comparison;
- Visual Validation;

14.3.8.1 Swath Plots

The drill composites were compared with the block model data by easting and elevation in the swath plots, shown in Figure 14-34, Figure 14-35 and Figure 14-36. The swath plots were constrained by the mineralized envelopes. These plots highlight that the estimates compare very well with the composite grades with some smoothing of the interpolation resulting from a relatively unconstrained OK estimation technique.

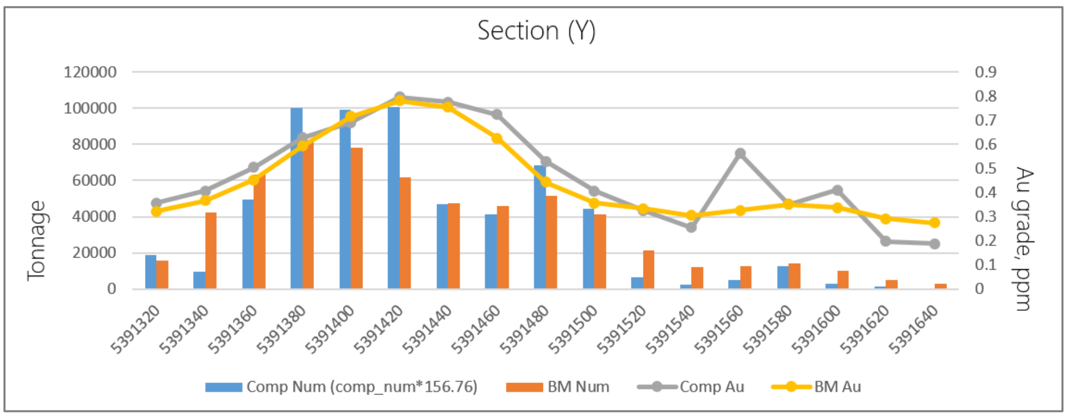


Figure 14-34: Swath plots on Northing

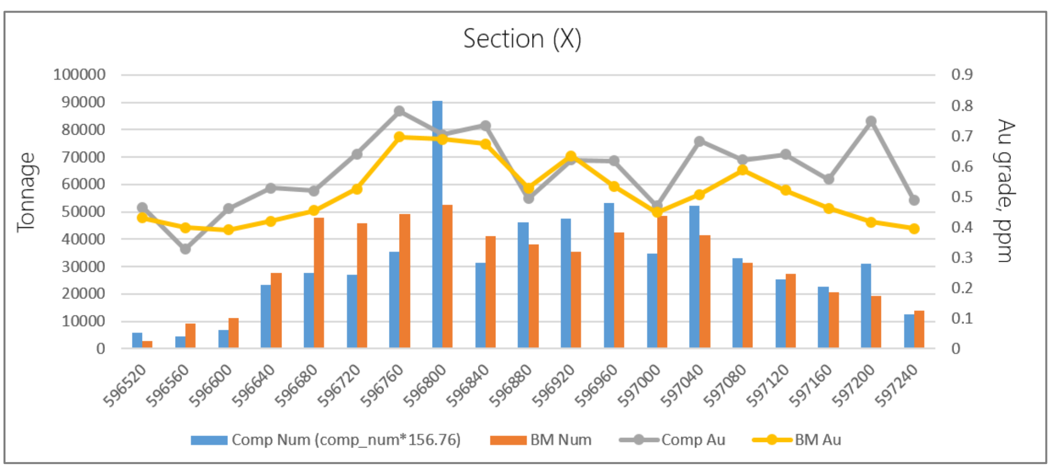


Figure 14-35: Swath plots on Easting

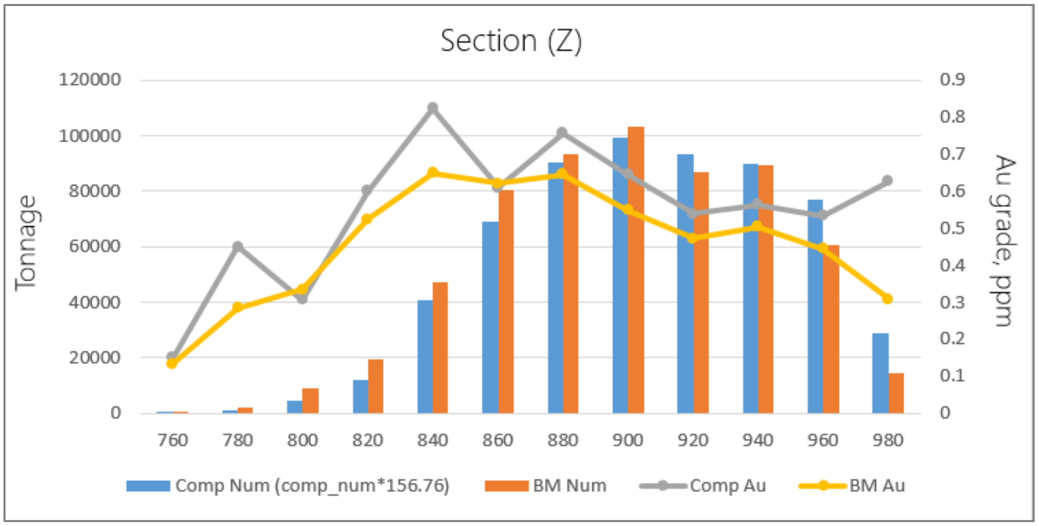


Figure 14-36: Swath plots on Elevation

14.3.8.2 Grade Comparison

Comparison of the block values and composites, results in a very good comparison (Table 14-20), with block model grade being slightly lower for than composite grades.

Table 14-20. Comparison of Block Estimates and Composites.

Zone	Block Model		Composites	
	Resource volume, Kbcm	Au, ppm	Number of Comps	Au, ppm
1	808.40	0.611	293	0.593
2	4,074.11	0.516	3643	0.518
3	1,215.99	0.420	134	0.414
4	602.69	0.228	51	0.219

14.3.8.3 Visual Validation

The visual comparison of block model grades against composite sample grade shows a strong correlation between the values. No significant discrepancies were apparent between the sections and the plans reviewed, yet some grade smoothing is apparent (Figure 14-37).

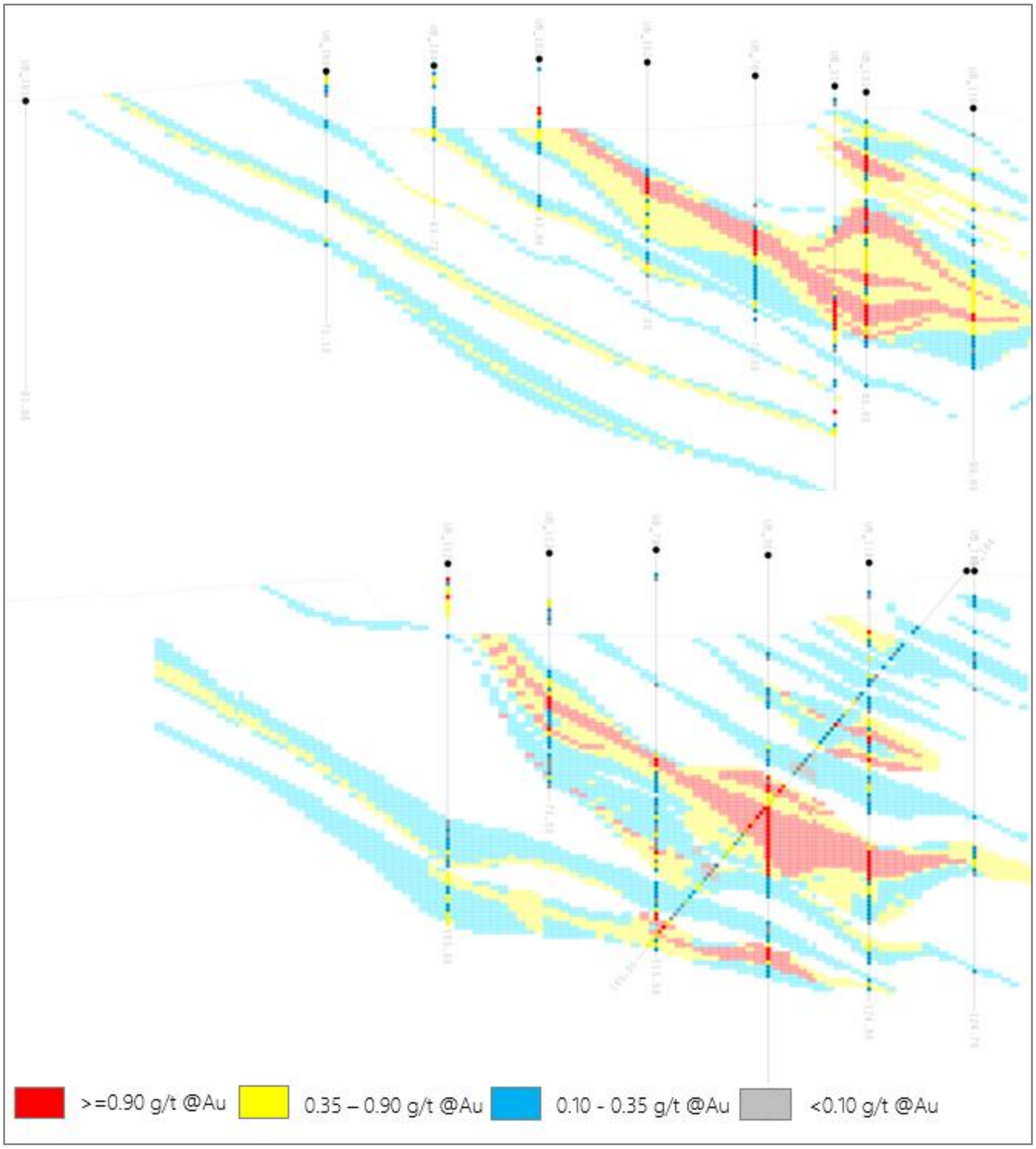


Figure 14-37: Ulaanbulag Visual Validation

14.3.8.4 Overall Validation

The review of the mathematical comparison indicates that a good correlation exists, as shown in the swath plots in Figure 14-34, Figure 14-35, Figure 14-36 and Table 14-37. This good correlation of the drillholes and interpolated block model is further supported when a visual inspection was completed. As a result of the validation completed Game Mine believes the estimate is representative of the composites, is indicative of the known controls of mineralisation and the underlying data.

14.3.9 Mineral Resource Classification

The updated mineral resource estimate presented in this Technical Report were prepared and disclosed in compliance with all disclosure requirements for mineral resources set out in the NI 43-101 Standards of Disclosure for Mineral Projects (2014).

A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.

Several factors are considered in the definition of a resource classification:

- NI 43-101 requirements;
- Canadian Institute for Mining, Metallurgy and Petroleum (CIM) Guidelines;
- Authors' experience with epithermal gold deposits;
- Spatial continuity based on variography of the assays within the drillholes;
- Drillhole spacing and estimate runs required to estimate the grades in a block;
- Observed mineralization on pit wall face;
- The confidence with the dataset based on the results of the validation; and
- The number of samples and drillholes used in each of the block estimations.

The confidence classification of the resource (Measured, Indicated, and Inferred) is based on an understanding of geological controls of the mineralization and the drillhole pierce point spacing in the resource area. Blocks were classified as Measured, Indicated, or Inferred if they were populated with grade during pass 1, pass 2, or pass 3 respectively during the interpolation process.

The classification methodology was largely based on the following:

- Measured Mineral Resources – blocks interpolated in the first search pass, where the oxidation state is well understood.
- Indicated Mineral Resources – blocks interpolated in the second pass, where the wider spaced drilling provides less confidence in the oxidation state.
- Inferred Mineral Resources – blocks interpolated in the third pass, where exploration data is wider spaced.

The Boroo Mineral Resource are classified by Game Mine in accordance with the guidelines of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources & Mineral Reserves Resource required by NI 43-101, that read in part as follows:

> *An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. An Inferred Mineral Resource has a lower level of confidence than that applying*

to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation. An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit. Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation. A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

14.3.10 Mineral Resource Reporting

Table 14-21 outlines the total mineral resources of Ulaanbulag as of January 1, 2024.

Table 14-21. Ulaanbulag Mineral Resources as of January 01, 2024

Zone	Category	Tonnage (Kt)	Average grade (g/t)	Metal (oz)
Zone 1	Measured	-	-	-
	Indicated	2,000	0.613	40,000
	Meas + Ind	2,000	0.613	40,000
	Inferred	30	0.448	400
Zone 2	Measured	4,500	0.616	89,000
	Indicated	5,000	0.438	70,000
	Meas + Ind	9,400	0.522	159,000
	Inferred	900	0.452	14,000
Zone 3	Measured	-	-	-
	Indicated	1,000	0.458	14,000
	Meas + Ind	1,000	0.458	14,000
	Inferred	1,800	0.471	27,000

Zone	Category	Tonnage (Kt)	Average grade (g/t)	Metal (oz)
Zone 4	Measured	-	-	-
	Indicated	-	-	-
	Meas + Ind	-	-	-
	Inferred	1,500	0.228	11,000
Total	**Measured**	**4,500**	**0.616**	**89,000**
	Indicated	**8,000**	**0.485**	**124,000**
	Meas + Ind	**12,400**	**0.532**	**213,000**
	Inferred	**4,300**	**0.379**	**52,000**

Notes:

1. *Ulaanbulag Mineral Resources are as of January 1, 2024, based on the CIM Definition Standards (2014).*
2. *Mineral Resource were estimates has been compiled under the supervision of QP Tuvshinbayar Batbayar.*
3. *Mineral Resources that are not Mineral Reserves have no demonstrated economic viability.*
4. *Reporting cut-off grade for Ulaanbulag Mineral Resources is 0.1 g/t Au (include both heap leach and milling ore).*
5. *The Ulaanbulag mineral resourcesi has been depleted for mining up to the mining (without backfilling) as of January 1, 2024.*
6. *The Mineral Resources are stated as in situ dry tonnes. All figures are in metric tonnes.*
7. *Figures have been rounded to the appropriate level of precision for the reporting of Mineral Resources.*
8. *Due to rounding, some columns or rows may not compute exactly as shown.*

Table 14-22 to Table 14-26 outlines the estimate of the total mineral resources by oxidation states of Ulaanbulag deposit by zones.

Table 14-22: Zone 1 Resources as of January 01, 2024

Resource Category	Oxidation State	Tonnage (Kt)	Average grade (g/t)	Metal (oz)
Measured	Oxide	-	-	-
	Transition	-	-	-
	Fresh	-	-	-
	Total	**-**	**-**	**-**
Indicated	Oxide	400	0.545	6,000
	Transition	900	0.731	21,000
	Fresh	800	0.505	12,000
	Total	**2000**	**0.613**	**40,000**
Meas + Ind	**Oxide**	**400**	**0.545**	**6,000**
	Transition	**900**	**0.731**	**21,000**
	Fresh	**800**	**0.505**	**12,000**
	Total	**2000**	**0.613**	**40,000**
Inferred	Oxide	20	0.385	200
	Transition	0	0.507	2
	Fresh	10	0.54	200
	Total	**30**	**0.448**	**400**

Table 14-23: Zone 2 Resources as of January 01, 2024

Resource Category	Oxidation State	Tonnage (Kt)	Average grade (g/t)	Metal (oz)
Measured	Oxide	100	0.347	1,000
	Transition	900	0.582	17,000
	Fresh	3,400	0.635	70,000
	Total	**4,500**	**0.616**	**89,000**
Indicated	Oxide	700	0.394	9,000
	Transition	1,700	0.382	21,000
	Fresh	2,600	0.487	40,000
	Total	**5,000**	**0.438**	**70,000**
Meas + Ind	**Oxide**	**800**	**0.388**	**11,000**
	Transition	**2600**	**0.453**	**38,000**
	Fresh	**6000**	**0.572**	**110,000**
	Total	**9400**	**0.522**	**159,000**
Inferred	Oxide	60	0.327	600
	Transition	1	0.627	20
	Fresh	890	0.46	13,000
	Total	**950**	**0.452**	**14,000**

Table 14-24: Zone 3 Resources as of January 01, 2024

Resource Category	Oxidation State	Tonnage (Kt)	Average grade (g/t)	Metal (oz)
Measured	Oxide	-	-	-
	Transition	-	-	-
	Fresh	-	-	-
	Total	-	-	-
Indicated	Oxide	-	-	-
	Transition	940	0.454	13,800
	Fresh	20	0.603	400
	Total	**960**	**0.458**	**14,200**
Meas + Ind	**Oxide**	-	-	-
	Transition	**940**	**0.454**	**13,800**
	Fresh	**20**	**0.603**	**400**
	Total	**960**	**0.458**	**14,200**
Inferred	Oxide	-	-	-
	Transition	300	0.559	5,000
	Fresh	1500	0.454	22,000
	Total	**1800**	**0.471**	**27,000**

Table 14-25: Zone 4 Resources as of January 01, 2024

Resource Category	Oxidation State	Tonnage (Kt)	Average grade (g/t)	Metal (oz)
Measured	Oxide	-	-	-
	Transition	-	-	-
	Fresh	-	-	-
	Total	-	-	-
Indicated	Oxide	-	-	-
	Transition	-	-	-
	Fresh	-	-	-
	Total	-	-	-
Meas + Ind	**Oxide**	-	-	-
	Transition	-	-	-
	Fresh	-	-	-
	Total	-	-	-
Inferred	Oxide	1,300	0.227	10,000
	Transition	40	0.201	300
	Fresh	170	0.249	1,000
	Total	**1,500**	**0.228**	**11,000**

Table 14-26: Total Resources as of January 01, 2024

Resource Category	Oxidation State	Tonnage (Kt)	Average grade (g/t)	Metal (oz)
Measured	Oxide	100	0.307	1,000
	Transition	900	0.572	17,000
	Fresh	3,400	0.62	70,000
	Total	**4,500**	**0.6**	**89,000**
Indicated	Oxide	1,100	0.424	16,000
	Transition	3,500	0.489	56,000
	Fresh	3,400	0.46	53,000
	Total	**8,000**	**0.468**	**124,000**
Meas + Ind	**Oxide**	**1,200**	**0.413**	**17,000**
	Transition	**4,500**	**0.507**	**73,000**
	Fresh	**6,800**	**0.541**	**123,000**
	Total	**12,400**	**0.515**	**213,000**
Inferred	Oxide	1,400	0.232	10,000
	Transition	300	0.511	5,000
	Fresh	2,600	0.426	37,000
	Total	**4,300**	**0.37**	**52,000**

14.3.11 Grade Tonnage Curves

Figures 14-39 and **Table 14-27** show the grade-tonnage curves for the Ulaanbulag block model.

Table 14-27: Tonnes and Grade

Cut Off Grade	Measured		Indicated		Inferred		Total	
	Tonnes, (Kt)	Au, (g/t)	Tonnes, (Kt)	Au, (g/t)	Tonnes, (Kt)	Au, (g/t)	Tonnes, (Kt)	Au, (g/t)
0.1	4,472.38	0.600	7,966.48	0.468	4,649.23	0.370	17,088.09	0.478
0.2	3,730.23	0.712	6,271.07	0.572	2,836.42	0.496	12,837.72	0.596
0.3	2,509.40	0.942	4,187.08	0.737	1,786.55	0.644	8,483.03	0.778
0.4	2,059.40	1.072	3,194.03	0.860	1,166.74	0.801	6,420.18	0.917
0.5	1,644.65	1.229	2,298.09	1.021	898.72	0.906	4,841.45	1.070
0.6	1,222.53	1.463	1,724.94	1.178	711.07	1.002	3,658.53	1.239
0.7	948.42	1.700	1,352.91	1.324	532.99	1.118	2,834.33	1.411
0.8	841.24	1.823	1,035.28	1.501	373.91	1.276	2,250.43	1.584
0.9	789.69	1.890	884.07	1.612	294.53	1.389	1,965.29	1.690
1.0	745.84	1.942	763.19	1.718	255.98	1.455	1,765.00	1.774

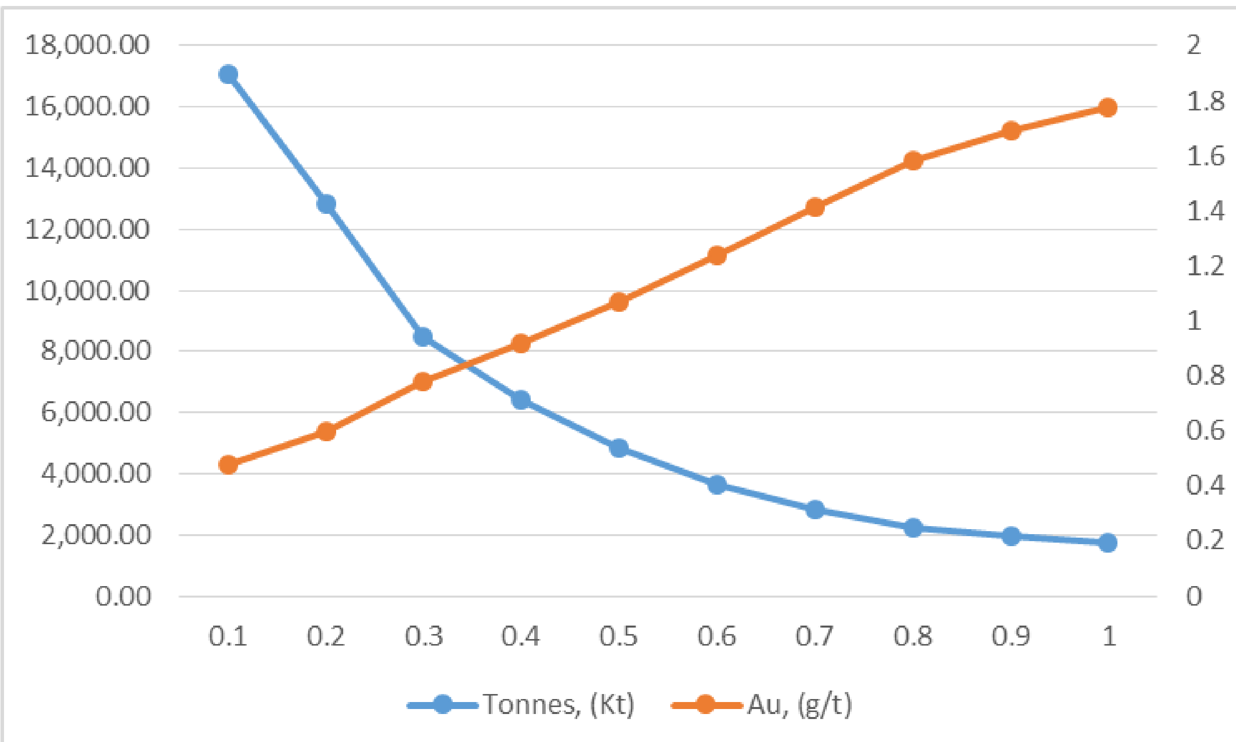


Figure 14-38: Grade Tonnage Curve

15.0 MINERAL RESERVE ESTIMATES

15.1 Summary

The Boroo Gold Project includes Mineral Reserve Estimates for the Boroo and Ulaanbulag deposits as outlined in the following sections.

15.2 Boroo

15.2.1 Introduction

Mineral Reserves for the Boroo deposit are based on the Measured and Indicated Resources presented in Chapter 14 and use engineering designs for the pit and associated operating parameters. Reserve calculations are valid at the time of estimation and use cut-off grade assumptions which were made prior to finalization of the economic model. The Mineral Reserve estimates are based on a mine plan and pit design developed using modifying parameters including metal price, metal recovery based on performance of the processing plant, and operating cost estimates.

Game Mine confirmed that there were no periods of negative cash flow following project start-up and that overall project economics are favorable at a five-year moving average gold price of $1,750/oz. Game Mine adopted standard mine planning processes to determine the Mineral Reserve estimate for the Boroo deposit. The following inputs and constraints were utilized for pit optimization and further defined in the following sections:

- Resource model with associated assay grades and densities for mineralized zones (Chapter 14)
- January 01 2024 pit survey provided by Boroo Gold
- Metallurgical recoveries (Figure 15-9)
- Geotechnical slope parameters (Section 15.2.2)
- Gold price of $1,750/oz
- Operating cost assumptions, including mine, mill, and G&A (Table 15-7)
- Dilution (Section 15.2.3.1)
- Processing rate of 5000 t/day (Section 15.2.3.4)
- Balanced total material movement per annum (Chapter 16)

15.2.2 Geotechnical Parameters

15.2.2.1 2022 Geotechnical Study

In 2021, a drill program to provide additional metallurgical information for process modelling was completed. Before the drill core was assessed for metallurgy, the core was inspected for geotechnical properties, structural data and rock samples were taken for laboratory testing. The program consisted of 69 drill holes, drilled at top west crests of the existing pits (Figure 15-1), for over 9,000 m.

Point load testing was performed on 98 suitable rock samples, consisting of 64 UCS samples, 24 Triaxial specimens (for eight tests) and 6 Brazilian samples. The minimum length for all samples collected was 2.5 to 3 times the diameter of the core in sections where no visible fractures were observed. The Triaxial samples were taken when a change continuity or lithology was seen, in groups of three every 50 m. The six Brazilian samples were collected at a sample length shorter than the diameter without cracks from the same sections as the UCS and Triaxial samples.

Geotechnical properties collected as part of the core-logging program were:

- Lithology
- Rock quality designation (RQD)
- Fracture orientation and spacing

Structural properties collected as part of the core-logging program were:

- Crack information (depth, type, infill and roughness)
- Route (subsidence, deformation, other integrity related anomalies)
- Crack monitoring (changes with, orientation and propagation)

From the geotechnical and structural data collected, the results of the laboratory testing were used to calculate strength characteristics and rock mass quality values of Q (Bartion), RMR90 (Laubscher) and RMR76 (Bieniawski). Fault structures were also estimated within the granite and sandstone formations.

RMR76 assessment yielded values of 50% to 52% (moderate) in diorite and granite and 45% (poor to moderate) in quartz and sandstone. Weathering caused degradation of the RMR76 values, in all lithologies, by ~10% in transitional zones and ~20% in weathered zones. A sample set of the drill holes displaying the RMR76 values is as shown, below:

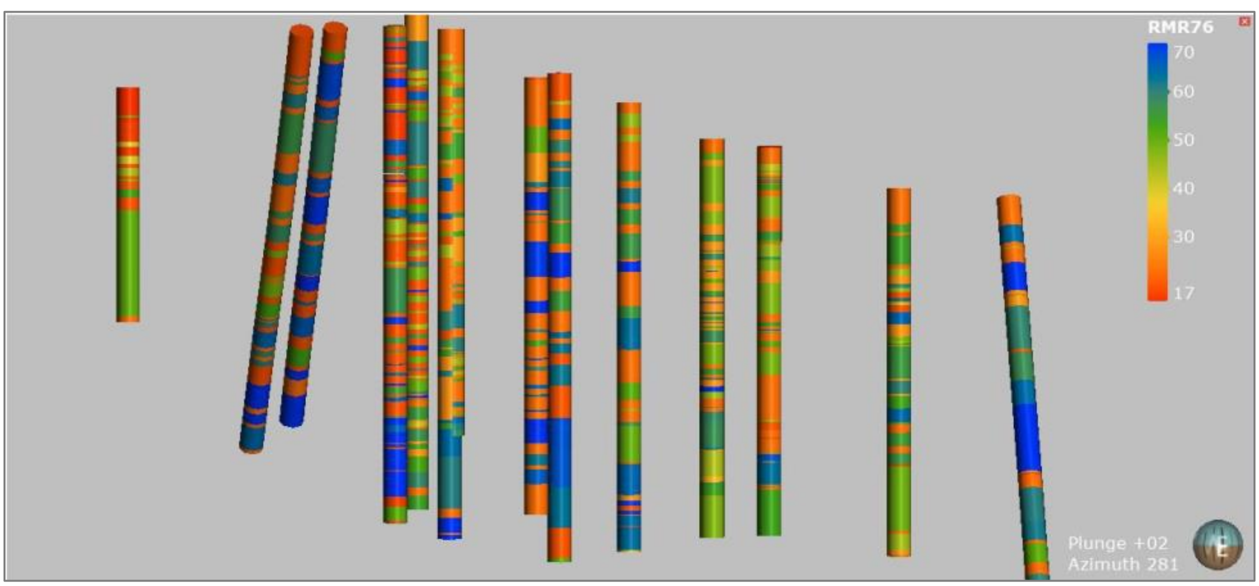


Figure 15-1: RMR76 Rock Mass in Selection of Drill Holes Sampled

Maximum indentation side angle for the rock mass was also calculated based on the Q values; 63° in sandstone, 62° in quartz, 68° in granite and 67° in diorite veins. Structural fissures and faults were measured in 66 of the 69 drill holes, however, only 19 having calculable location or measured angles (β). 2,800 measurements were observed in recovered core samples, with 1,160 of those measurements being from quality-oriented core samples.

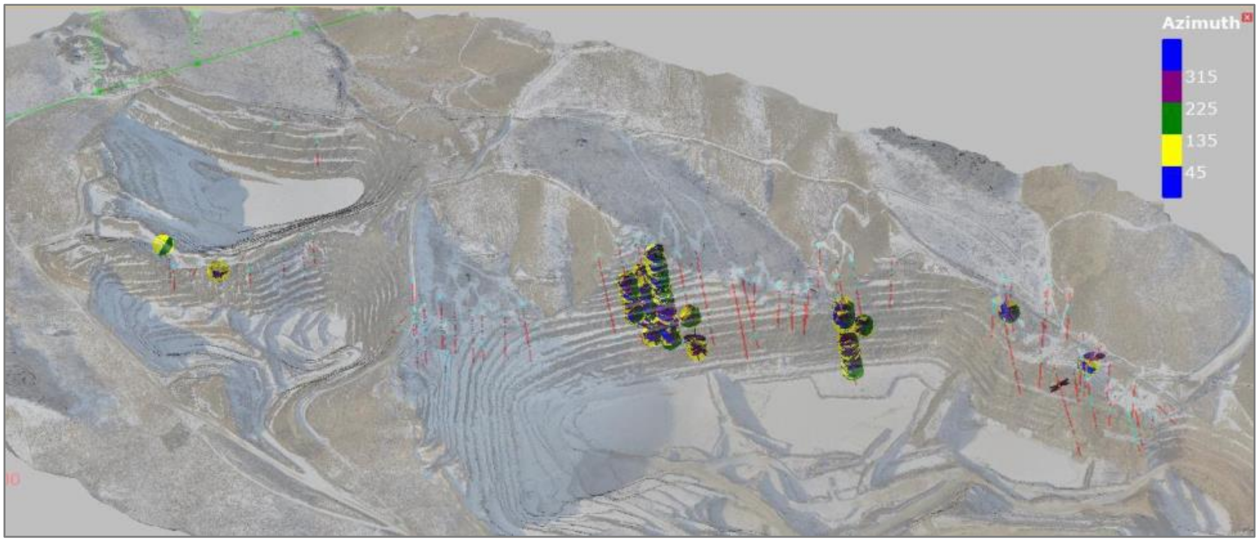


Figure 15-2: Selection of Measured Core Samples with Structure Orientations

Cluster analysis was used with groupings of fracture location elements in sandstone (Figure 15-3) and granite (Figure 15-4). Sandstone had four set types with horizontally and vertically dipping structures with a moderately south-west dipping fracture. Granite had five set types with moderately dipping structures in the same directions as the sandstone strata along with horizontally dipping faults, moderately dipping faults, steeply north-west dipping faults and a moderate north-east dipping fault.

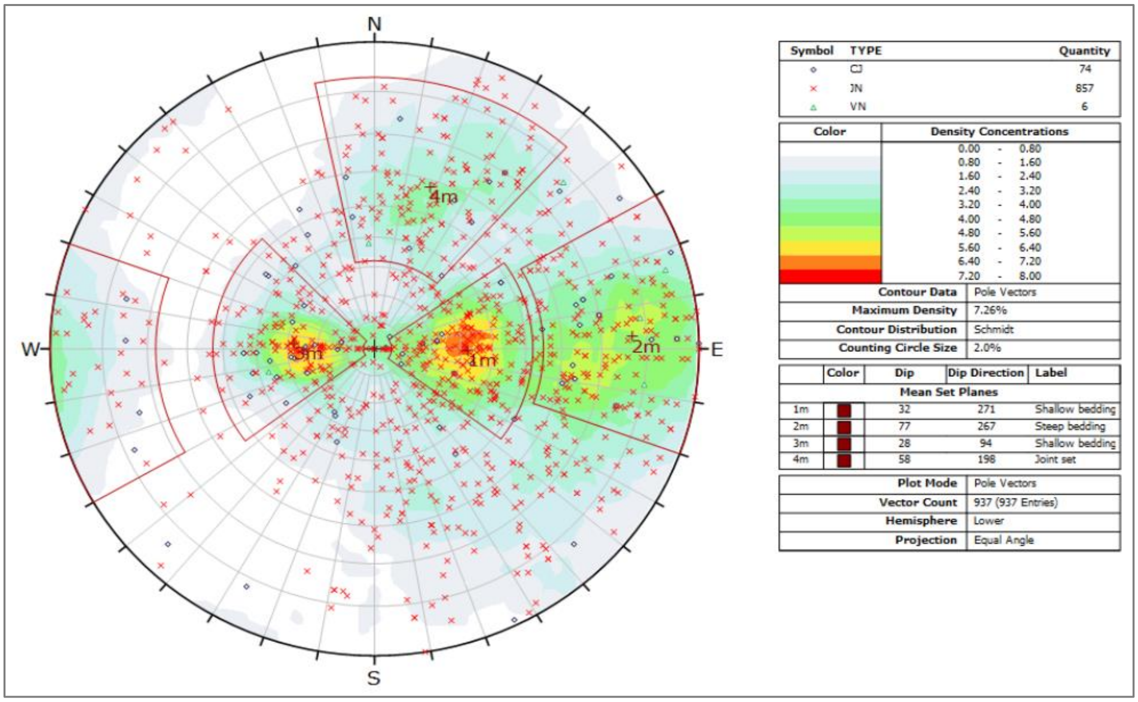


Figure 15-3: Cluster Analysis of Sandstone

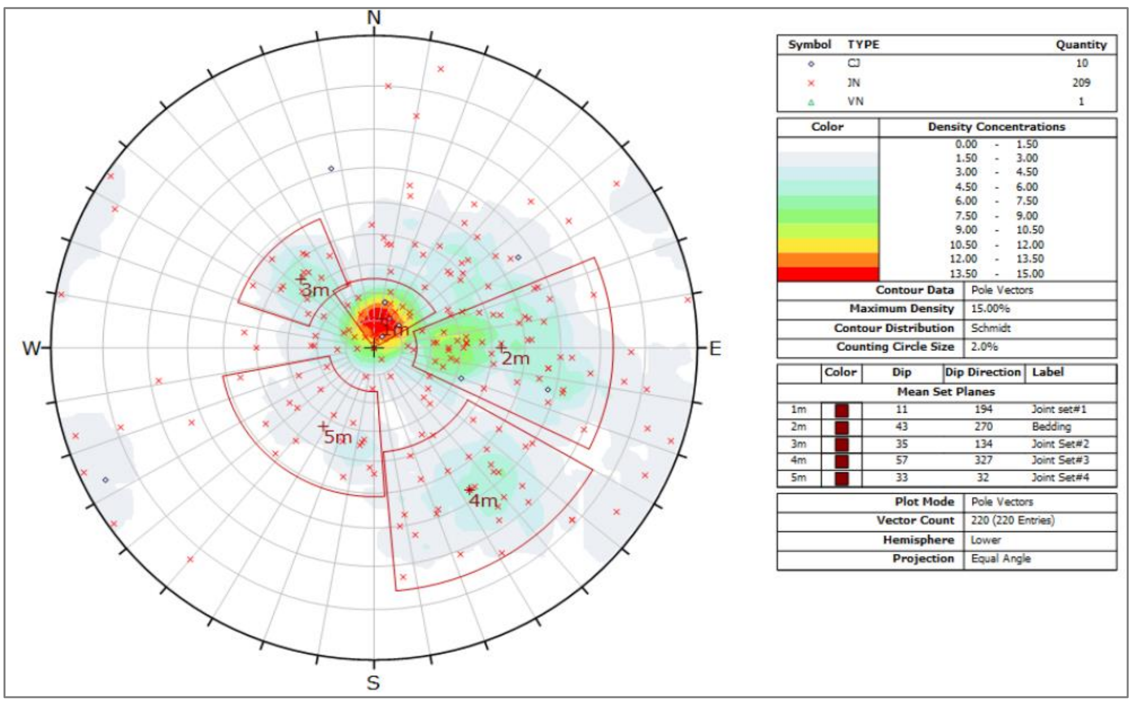


Figure 15-4: Cluster Analysis of Granite

For the sets of sandstone and granite, ITASCA KATS software calculated wedge and flat crack probabilities, collapsed material volume, and upper edge crack width using kinematic analysis. The input parameters for this analysis are as in Table 15-1:

Table 15-1: Parameters and Criteria Used in Kinematic Analysis

Parameter	Value	Units
Specific gravity of sandstone	2.5	t/m^3
Specific gravity of granite	2.6	t/m^3
Pore water pressure	-	%
Bench height	10	m
Berm width	5	M
Angle between calculation steps	30	degree
Angular limit of planar failure	30	degree
Swell factor	1.3	-
Natural slope angle	37	degree
Angular limit of wedge failure	30	degree
Fracture resistance		kPa
Friction angle of fractures	25	degree
Factor of Safety	≤ 1	
Probability	≤ 50	%

The analysis, along with the rock mass characteristics, formed the basis for the geotechnical design parameters for the mine design, as shown in **Table 15-2** and **Table 15-3**, below. The recommended geotechnical parameters for the mine are highlighted in blue.

Table 15-2: Sandstone Recommended Pit Design Parameters

Design Sector	Bench Face Angle	Bench Height	Bench Slope Azimuth	Internal Slope Angle			Berm Width		
				Design	Ritchie	Optimized	Design	Ritchie	Optimized
BG_000	70	10	345-015	49.2	44.4	70	5	6.6	0
BG_030	70	10	015-045	49.2	44.4	70	5	6.6	0
BG_060	70	10	045-075	49.2	44.4	70	5	6.6	0
BG_090	70	10	075-105	48.3	44.4	48.3	5.3	6.6	5.3
BG_120	70	10	105-135	49.2	44.4	70	5	6.6	0
BG_150	70	10	135-165	49.2	44.4	70	5	6.6	0
BG_180	70	10	165-195	47.8	44.4	47.8	5.4	6.6	5.4
BG_210	70	10	195-225	47.8	44.4	47.8	5.4	6.6	5.4
BG_240	70	10	225-255	49.2	44.4	70	5	6.6	0
BG_270	70	10	255-285	48.1	44.4	48.1	5.3	6.6	5.3
BG_300	70	10	285-315	49.2	44.4	70	5	6.6	0
BG_330	70	10	315-345	49.2	44.4	70	5	6.6	0

Table 15-3: Granite Recommended Pit Design Parameters

Design Sector	Bench Face Angle	Bench Height	Bench Slope Azimuth	Internal Slope Angle			Berm Width		
				Design	Ritchie	Optimized	Design	Ritchie	Optimized
BG_000	70	10	345-015	49.2	44.4	70	5	6.6	0
BG_030	70	10	015-045	49.2	44.4	48.2	5.3	6.6	5.3
BG_060	70	10	045-075	49.2	44.4	70	5	6.6	0
BG_090	70	10	075-105	49.2	44.4	70	5	6.6	0
BG_120	70	10	105-135	47.9	44.4	47.9	5.4	6.6	5.4
BG_150	70	10	135-165	47.8	44.4	47.8	5.4	6.6	5.4
BG_180	70	10	165-195	49.2	44.4	70	5	6.6	0
BG_210	70	10	195-225	49.2	44.4	70	5	6.6	0
BG_240	70	10	225-255	49.2	44.4	70	5	6.6	0
BG_270	70	10	255-285	47.8	44.4	47.8	5.4	6.6	5.4
BG_300	70	10	285-315	47.7	44.4	47.7	5.4	6.6	5.4
BG_330	70	10	315-345	47.8	44.4	47.8	5.4	6.6	5.4

Although kinematic analysis was not performed for quaternary sediments / weathered rock mass material (Table 15-4) and transition material (Table 15-5), mine design parameters were still estimated based on rock mass characteristics and indentation angles.

Table 15-4: Quaternary Sediments / Weathered Recommended Pit Design Parameters

Domain	Design Sector	Bench Face Angle	Bench Height	Bench Slope Azimuth	Internal Slope Angle	Berm Width
Alluvial Sediments and Weathering Zone	BG_000	55	10	345-015	36.5	6.5
	BG_030	55	10	015-045	36.5	6.5
	BG_060	55	10	045-075	36.5	6.5
	BG_090	55	10	075-105	36.5	6.5
	BG_120	55	10	105-135	36.5	6.5
	BG_150	55	10	135-165	36.5	6.5
	BG_180	55	10	165-195	36.5	6.5
	BG_210	55	10	195-225	36.5	6.5
	BG_240	55	10	225-255	36.5	6.5
	BG_270	55	10	255-285	36.5	6.5
	BG_300	55	10	285-315	36.5	6.5
	BG_330	55	10	315-345	36.5	6.5

Table 15-5: Transition Recommended Pit Design Parameters

Domain	Design Sector	Bench Face Angle	Bench Height	Bench Slope Azimuth	Internal Slope Angle	Berm Width
Transition Zone	BG_000	60	10	345-015	42.9	5
	BG _030	60	10	015-045	42.9	5
	BG _060	60	10	045-075	42.9	5
	BG _090	60	10	075-105	42.9	5
	BG _120	60	10	105-135	42.9	5
	BG _150	60	10	135-165	42.9	5
	BG _180	60	10	165-195	42.9	5
	BG _210	60	10	195-225	42.9	5
	BG _240	60	10	225-255	42.9	5
	BG _270	60	10	255-285	42.9	5
	BG _300	60	10	285-315	42.9	5
	BG _330	60	10	315-345	42.9	5

Table 15- 6: Strength Properties of Rock

Material	Unit Weight (kN/m^3)	Friction Angle	UCS	GSI	mi	D	Source
Alluvial Sediments	24.25	37	0	0	0	0	Mohr-Coulomb
Diorite	25.5	0	100000	54	20	0.7	Hoek-Brown
Granite	24.25	0	43000	55	11.1	0.7	Hoek-Brown
Sandstone	25.5	0	50000	54	7.9	0.7	Hoek-Brown

15.2.2.2 Stability Calculations

The slope stabilization limits were assessed for pit 2, 3, 5 and 6 using a three-dimensional geological model analysis performed with SLIDE3D software. Factors of safety, to meet stability criteria, were: 1.0 for a single bench, 1.25 for a slope between haul roads, and 1.5 for general slopes. The results of the analyses concluded that all cases exceeded the target Factor of Safety. The areas assessed are as shown in Figure 15-5, below.

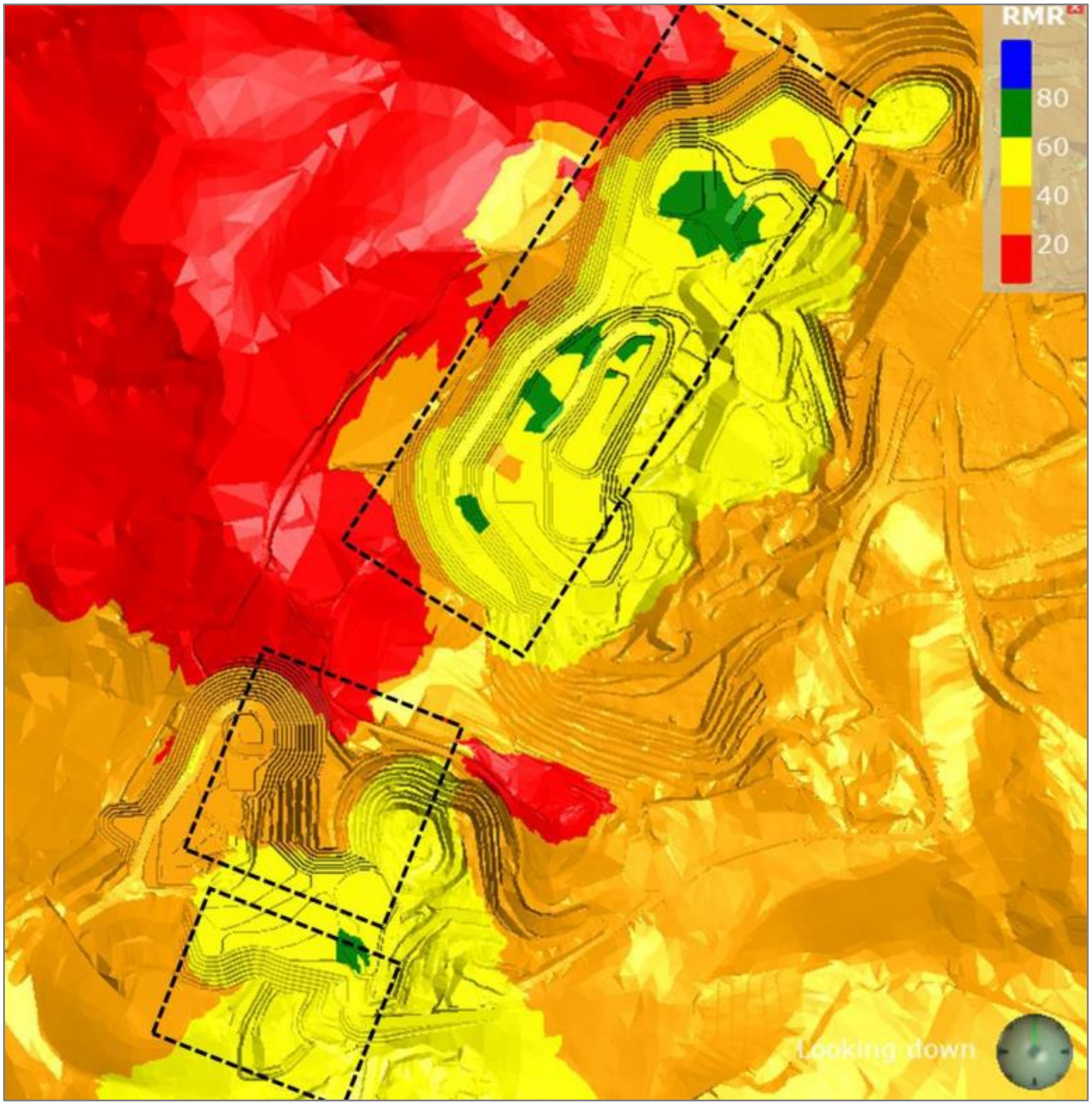
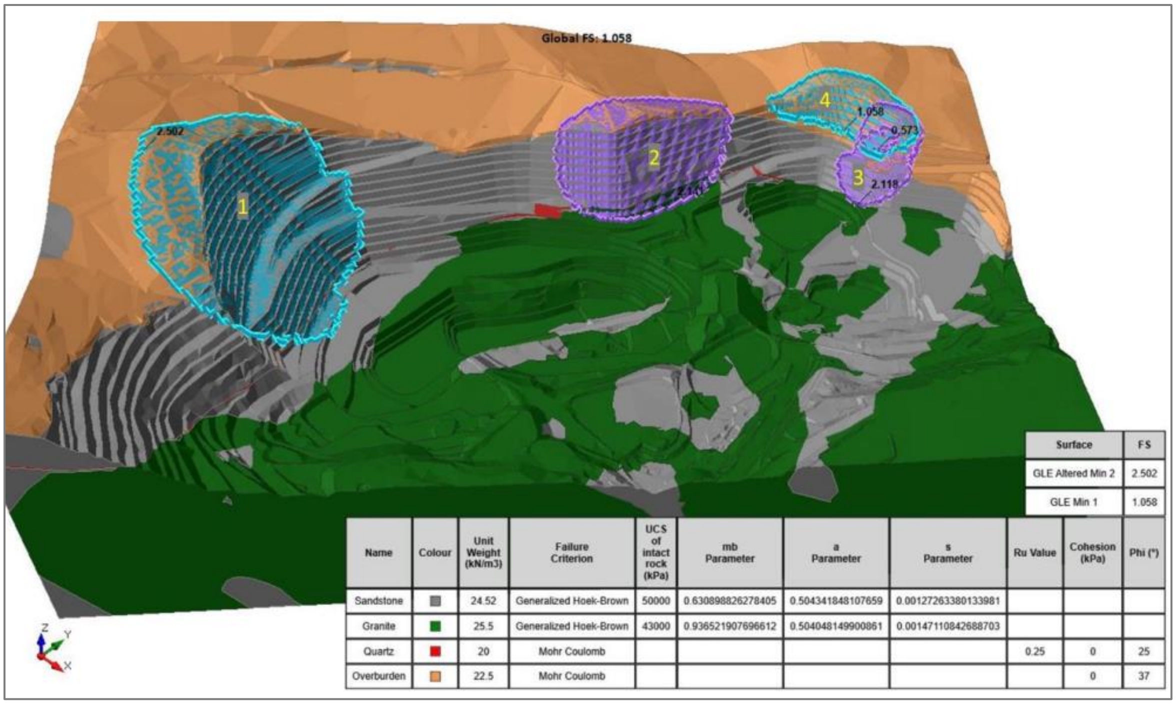


Figure 15-5: Slope Stability Calculation Area Plan View

Pit 2 and Pit 3 designs

The expansion of zones of Pit 2 and Pit 3 had four areas (Figure 15-6) assessed for the limits of stability.



Surface	FS
GLE Altered Min 2	2.502
GLE Min 1	1.058

Name	Colour	Unit Weight (kN/m3)	Failure Criterion	UCS of intact rock (kPa)	mb Parameter	a Parameter	s Parameter	Ru Value	Cohesion (kPa)	Phi (°)
Sandstone		24.52	Generalized Hoek-Brown	50000	0.630898826278405	0.504341848107659	0.00127263380133981			
Granite		25.5	Generalized Hoek-Brown	43000	0.936521907696612	0.504048149900861	0.00147110842688703			
Quartz		20	Mohr Coulomb					0.25	0	25
Overburden		22.5	Mohr Coulomb						0	37

Figure 15-6: Pit 2 and Pit 3 Stability Analysis Isometric View

The resultant Factors of Safety (FoS) calculated were as follows:

- Area 1: FoS 2.5
- Area 2: FoS 2.1
- Area 3: FoS 2.1
- Area 4: FoS <1

Area 4 contained material from a waste dump and was calculated to be unstable.

Pit 5

The expansion of zone of Pit 5 had three areas (Figure 15-7) assessed for the limits of stability.

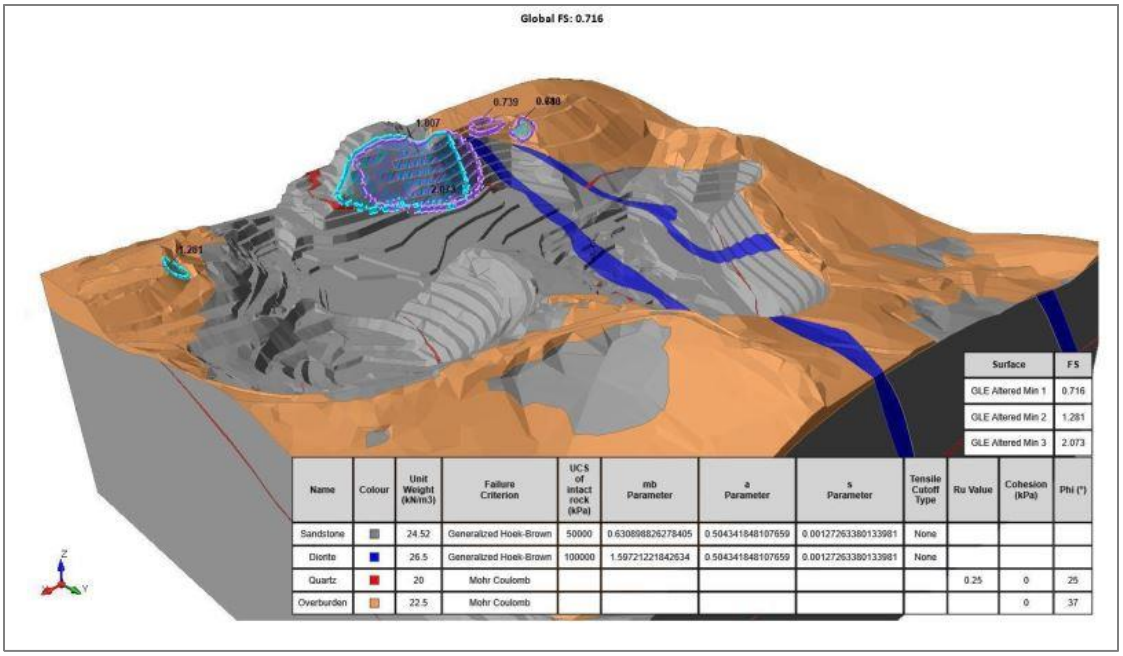


Figure 15-7: Pit 5 Stability Analysis Isometric View

The resultant Factors of Safety (FoS) calculated were averaged to be greater than 1.8, however, when the slope intersected a waste dump it was calculated to be less than 0.74 (below criteria limits).

Pit 6

The expansion of zone of Pit 6 had three areas (Figure 15-8) assessed for the limits of stability.

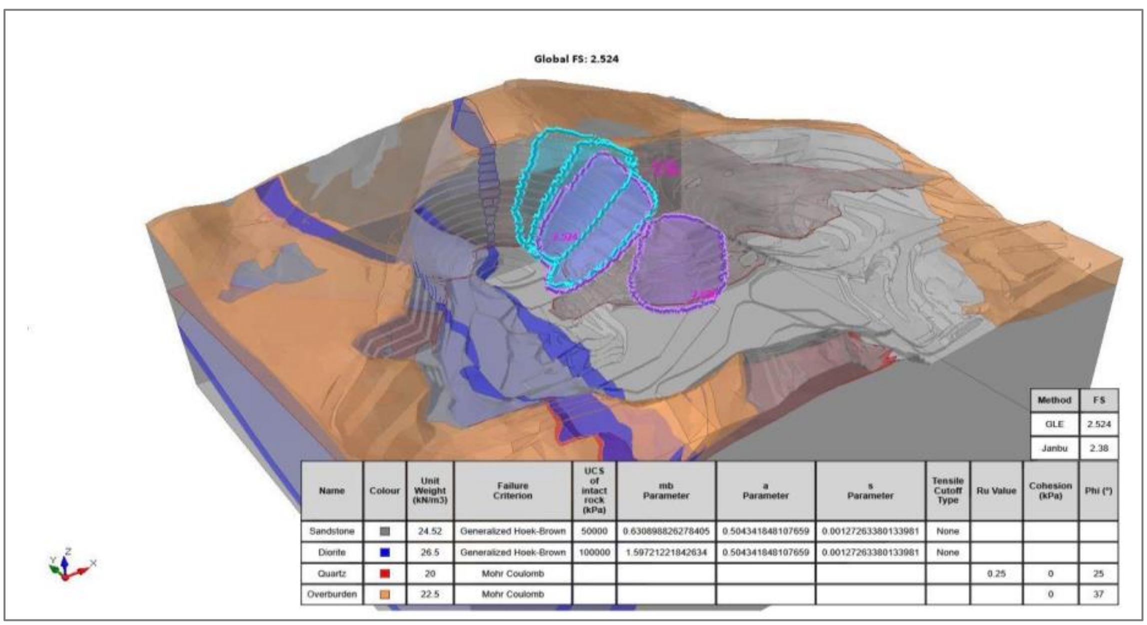


Figure 15-8: Pit 5 Stability Analysis Isometric View

The resultant Factors of Safety (FoS) calculated were averaged to be greater than 2.3.

15.2.3 Pit Limit Optimization

The terminology "pit limit optimization" refers to a process which aims to identify the best value mining pit shape for a given series of inputs and constraints. It does not imply that mining has been "optimized" in other ways; such as optimal mine sequence or optimal equipment selection.

The pit limit optimization process creates a series of pit shells based on a range of metal prices ranging from 30% to 200% of a chosen base price in increments of 10%. The lower the price the smaller the pit and the higher the potential able to be profitably extracted at this lower revenue. Hence this approach not only identifies the high value areas, but also indicates the potential pit extents should future metal price change.

The geology model created for the Mineral Resource estimate was the basis for pit limit optimization using the Whittle 4X Optimizer for gold ore deposits. The optimization process involved the following steps:

- Identify physical constraints;
- Define mining factors, such as geotechnical design parameters and ore loss and dilution;
- Define metallurgical modifying factors;
- Set mine operating cost rates;
- Set product prices; and
- Run optimizer process and report results.

15.2.3.1 Optimization Parameters

Physical Constraints

- Game Mine is not aware of any surface physical constraints to mining; such as infrastructure, rivers or environmental limits.
- A geological constraint was applied that restricted to Measured and Indicated Mineral Resources only.
- January 01 2024 pit survey.

Geotechnical Design Parameters

- Game Mine assumed a 42.8 to 44.2 degree overall pit slope. These estimates are based on a geotechnical report provided by Boroo Gold. Geotechnical Slope Parameters:
  - Bench Angle: 55° (weathered) to 70° (bedrock) (BFA)
  - Bench Height: 10 m (BH)
  - Berm Width: 5 m to 6.6 m (CB)
  - Inter-ramp Angle: 47.8° to 49.2° (IRA)

Mining Modifying Factors

- With grade control measures and smaller mining equipment, it was assumed that the mineralization will be fully recovered during mining; no Mining Recovery factor was applied. Given the size of the blocks relative to the physical mineralization, it was assumed that the blocks in the geological model are fully diluted; no Mining Dilution factor was applied

Grade/Recovery Inputs

- Recovery estimates are based on a metallurgical report provided by Boroo Gold. A summary of the recovery inputs can be seen in Figure 15-9 below.

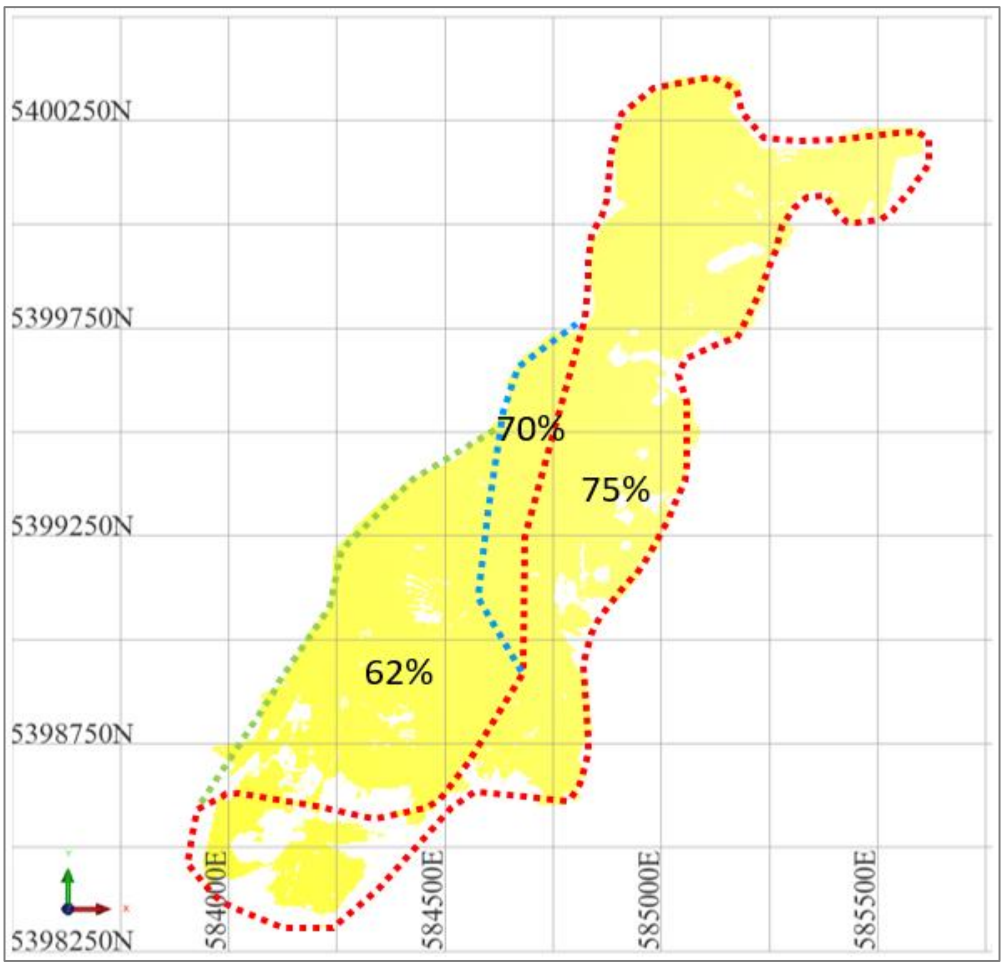


Figure 15-9: Recovery Domain

Operating Cost Rates

- Rates of operating costs were applied based on historical operating costs, escalated to 2024.
- Operating costs used for the Optimization process are given in Table 15-7.

Table 15-7: Operating Cost Parameters

Item	Units	Value
Mining Cost	US$/t	1.77
Ore processing	US$/t ore	12.14
Tailings	US$/t ore	2.26
Ore Control	US$/t ore	0.59
Overheads	US$/t ore	2.22
Royalty	% of total revenue	5.00

Product Prices

The Optimizer analysis the deposit at a range of metal prices. The "base case" metal price is called the 100% "revenue factor". The revenue factors are adjusted to effectively adjust the product price. The "base case" gold price applied was US$1750/oz. The price sensitivity analysis was on values from 30% to 200% of the base case of gold price, that is US$525/oz to US$3500/oz.

15.2.3.2 Base Case Analysis Results

The Base Case analysis utilized measured and indicated Resource only, with the inferred Mineral Resource being regards as waste as well as the unmineralised or unclassified material.

A summary of the pit optimization results for the base case parameters is set out in Table 15-8 and illustrated in Figure 15-10. A summary of the pit shell quantities by revenue factor are illustrated in Figure 15-10. Pit 8 (highlighted) is the outcome using the "Base Case" price.

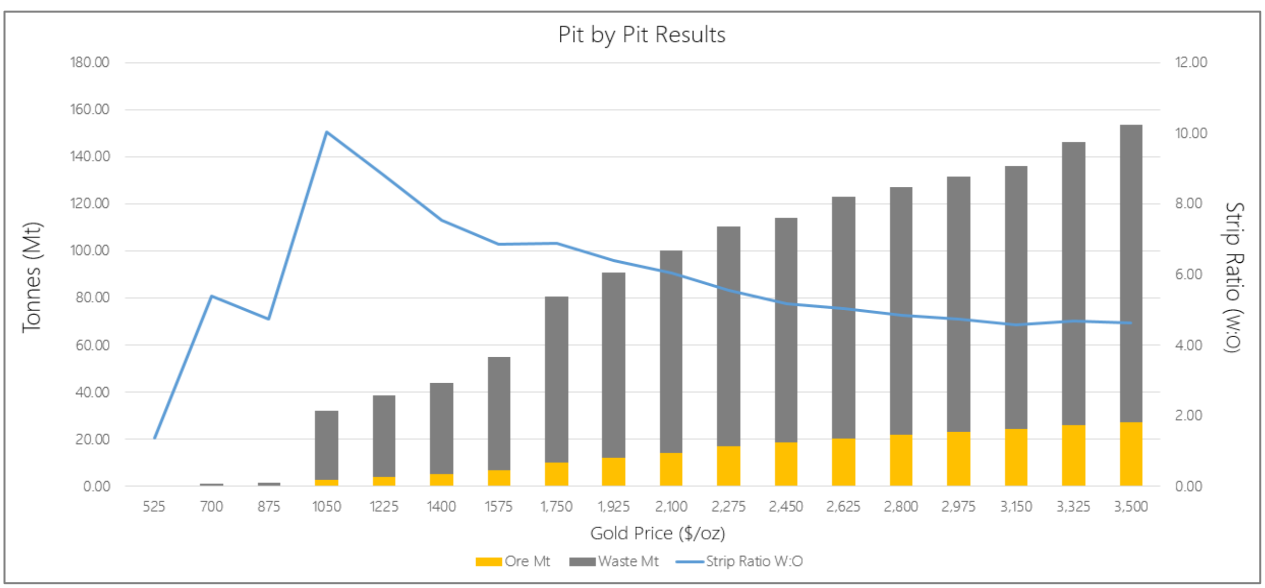


Figure 15-10: Base Case Optimization Results

Table 15-8: Base Case Optimization Results

Pit	Revenue	Au Price	Ore	Waste	Total	Strip Ratio	Au grade	Contained Au
	Factor	USD/oz	Mt	Mt	Mt	W:O	g/t	kg
1	0.3	525	0.04	0.05	0.09	1.38	2.43	93.7
2	0.4	700	0.14	0.76	0.90	5.38	2.40	337.3
3	0.5	875	0.25	1.18	1.43	4.74	2.02	505.0
4	0.6	1050	2.89	29.01	31.89	10.05	1.74	5,017.0
5	0.7	1225	3.95	34.76	38.70	8.81	1.59	6,269.4
6	0.8	1400	5.15	38.77	43.91	7.53	1.42	7,310.7
7	0.9	1575	7.00	47.96	54.96	6.85	1.28	8,929.1
8	**1.0**	**1,750**	**10.26**	**70.52**	**80.78**	**6.87**	**1.15**	**11,801.6**
9	1.1	1,925	12.30	78.67	90.97	6.39	1.07	13,165.6
10	1.2	2,100	14.23	86.06	100.29	6.05	1.01	14,318.4
11	1.3	2,275	16.85	93.75	110.60	5.56	0.93	15,665.3
12	1.4	2,450	18.48	95.65	114.13	5.18	0.88	16,342.8
13	1.5	2,625	20.39	102.66	123.06	5.03	0.85	17,234.9
14	1.6	2,800	21.74	105.25	126.98	4.84	0.82	17,751.8
15	1.7	2,975	22.95	108.84	131.79	4.74	0.79	18,228.6
16	1.8	3,150	24.46	111.85	136.31	4.57	0.77	18,749.7
17	1.9	3,325	25.79	120.47	146.26	4.67	0.75	19,330.2
18	2.0	3,500	27.29	126.24	153.53	4.63	0.73	19,860.8

Review of the optimization results indicates the total potential mineable quantity for the Project at a 100% revenue factor is estimated at approximately 10.26 Mt at a grade of 1.15 g/t Au resulting in a strip ratio of 6.87:1 (t waste: t ore). The sensitivity of the mineable quantities to metal price is reflected by the steepness of the curve graphed in Figure 15-10 at changing revenue factors (or price). These results indicate that the Project is only mildly sensitive to economic parameters such as the metal selling price near the base price. A 10% variation in the gold price above and below the base price increases and decreases the mineable ore quantity by approximately 19.8% to 32% respectively. The results suggest the deposit is highly sensitive to lower gold prices.

Base Case Cash flow Analysis

The basic Whittle pit optimizer result defines the "optimal" pit shell for fixed mining, economic and physical constraints. This outcome, however, is not necessarily the "optimal" result as it does not account for possible changes in value over the mining life. To overcome this issue, Whittle 4X software undertakes a life-of-mine (LOM) cash flow analysis to assess which pit provides the highest economic return taking into account the time value of money.

For the total cash flow of a pit to be calculated, there needs to be a LOM production schedule to allow mining costs and revenues to be determined over time. The Four-X software develops two types of schedules which it refers to as "best case" and "worst case" schedule. The best case schedule assumes mining commences at the inner-most nested pit shell and then expands to successfully larger shells until the selected pit limit is reached. As this sequence also reflects mining from the highest to lowest margin per tonne pit shell, it theoretically produces the "highest" project cash flow. The result should be considered optimistic as the pit shells often are not of practical mining width and hence the staged development represented in the schedule is not achievable in practice.

The worst case schedule assumes mining occurs in horizontal benches, starting from the highest elevation and then proceeding down to the base of the selected pit. There are no interim pit shells mined. As the upper benches typically have the highest strip ratio, this often results in the cash flow approaching the lower boundary for the selected pit.

The average of the "Best Case" and "Worst Case" cash flow schedules is calculated to reflect a more practical outcome. The preferred pit shell is often selected based on this result.

The end of a scheduling year is determined based on user-defined constraints, for example, limits to mining production. It is worth nothing that the Whittle Four-X software does not attempt to optimize a scheduling sequence to best meet all constraints and maximize cash flow, but simply follows the rules dictated by the "best" case and "worst" case scenario schedules. The best method of assessing the performance of a given pit shell and hence its cash flow is to export the data to alternative software packages where more scheduling and financial analysis flexibility and capability exist.

For the cash flow analysis, Game Mine assumed mineable quantity production of 1.7Mt ROM and a 10% discount rate. All other input parameters were as for the pit limit optimization modelling. The cash flow analysis results are set out in Table 15-9.

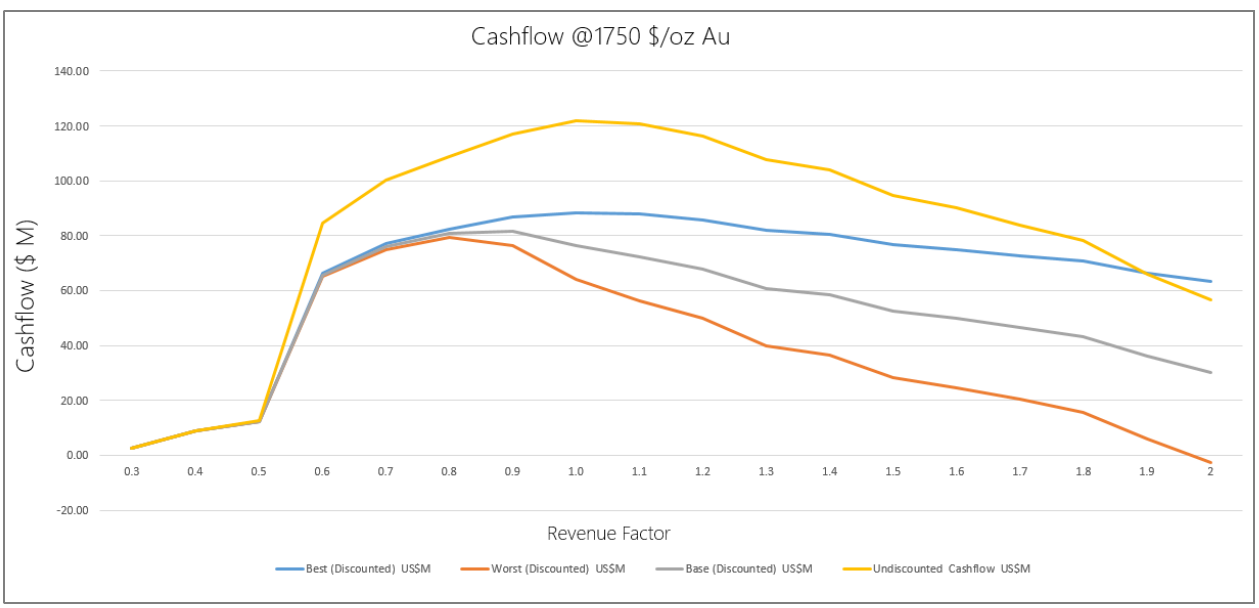


Figure 15-11: Whittle Cash Flow Analysis

Table 15-9: Base Case Optimization Cash Flow Results

Pit	RF	Best	Worst	Base	Undiscounted	Mineable Quantity	Strip Ratio	Au grade	Contained Au
		US$M	US$M	US$M	US$M	Mt	W:O	g/t	kg
1	0.3	2.73	2.73	2.73	2.74	0.04	1.38	2.43	93.7
2	0.4	8.74	8.74	8.74	8.81	0.14	5.38	2.40	337.3
3	0.5	12.35	12.35	12.35	12.56	0.25	4.74	2.02	504.9
4	0.6	66.30	65.07	65.69	84.51	2.89	10.05	1.74	5,015.6
5	0.7	77.02	74.91	75.97	100.29	3.95	8.81	1.59	6,269.9
6	0.8	82.26	79.29	80.77	108.93	5.15	7.53	1.42	7,308.4
7	0.9	87.03	76.55	81.79	116.95	7.00	6.85	1.28	8,925.9
8	**1.0**	**88.52**	**64.11**	**76.32**	**121.98**	**10.26**	**6.87**	**1.15**	**11,797.9**
9	1.1	87.88	56.43	72.16	120.71	12.30	6.39	1.07	13,163.5
10	1.2	85.74	49.76	67.75	116.23	14.23	6.05	1.01	14,316.5
11	1.3	81.98	39.85	60.91	107.63	16.85	5.56	0.93	15,671.1
12	1.4	80.54	36.42	58.48	104.16	18.48	5.18	0.88	16,334.5
13	1.5	76.78	28.43	52.60	94.87	20.39	5.03	0.85	17,232.4
14	1.6	75.08	24.76	49.92	90.37	21.74	4.84	0.82	17,758.6
15	1.7	72.77	20.43	46.60	84.05	22.95	4.74	0.79	18,222.0
16	1.8	70.65	15.45	43.05	78.40	24.46	4.57	0.77	18,758.4
17	1.9	66.25	5.85	36.05	65.83	25.79	4.67	0.75	19,317.6
18	2.0	63.19	-2.54	30.32	56.67	27.29	4.63	0.73	19,866.0

The results indicate that the Base Case pit shell, Pit 8, will not likely deliver the highest net present value (NPV). The pit shell which delivers the best NPV is Pit 7. However, the difference in cash flow between Pit shell 7 and Pit shell 8 is only US$5.4M while the variance in ore is 3.26 Mt.

15.2.3.3 Additional Analysis

The additional Whittle optimisation cases undertaken using the same revenue factor. Sensitivities were conducted on the cost and slope variables of the pit optimisation, varying input values by ±20%. The results for the options are compared to the Base Case pit shell in Table 15-10.

The results indicate that further drilling to convert the Inferred ore to Indicated or Measured will only slightly increase the quantity of mineable quantity when compared to the base case pit shell (Pit 8). Also, the steepening of pit walls has a negligible impact on the potential mineable quantity, however it does slightly decrease waste rock mined, which is as expected given the style and geometry of the mineralization within the Project.

Table 15-10: Additional Analysis

Option	Description	Mineable Quantity	Waste	Total Mined	Strip Ratio	Au grade	Contained Au	Mine Life
		Mt	Mt	Mt	W:O	g/t	kg	Years
1	Base	10.26	70.52	80.78	6.87	1.15	11,801.6	5.86
2	Inferred included	10.53	74.40	84.92	7.07	1.16	12,170.2	6.02
3	Mining Cost+20%	8.29	48.26	56.55	5.82	1.16	9,642.4	4.74
4	Mining Cost-20%	11.03	79.76	90.79	7.23	1.14	12,603.8	6.30
5	Milling Cost+20%	6.46	48.45	54.91	7.50	1.34	8,636.9	3.69
6	Milling Cost-20%	14.23	78.61	92.84	5.52	0.98	14,004.7	8.13
7	Slope Angle +2deg	10.37	68.09	78.45	6.57	1.15	11,933.1	5.92
8	Slope Angle -2deg	9.97	69.42	79.38	6.97	1.14	11,395.4	5.69

15.2.3.4 Selection of Preferred Pit Shell

The selection of the preferred pit shell was undertaken in conjunction with Boroo Gold. Key criteria used to select the pit shell included:

- High potential profitability;
- Maximise cash flow;
- Large in-pit mining inventory; and
- Potential to produce 60-65Koz gold per year;

Based on the outcomes of the pit limit optimisation and the cash flow analysis discussed above, the preferred pit shell selected is Pit 8, being the 100% revenue factor shell. This pit has higher NPV results and is estimated to have over 10.26 Mt of potential mineable quantity at 1.15 g/t Au and at a strip ratio of 6.87:1. At the preferred production rate of 1.7Mtpa the mine life would be 6 years.

15.2.4 Deposit Characterization

Deposit characterization is the process of determining and highlighting the key physical characteristics of the deposit in order to assess and understand the implications for mining and the mining strategy. It is completed by analyzing numerical data in the form of either plots or charts within the final pit design.

The key results of the deposit characterisation include the following:

- Figure 15-12 plots both mineable quantity tonnes against cut-off grade. It is a commonly used chart which demonstrates the distribution of mineable quantities relative to cut-off grade;
- Figure 15-13 plots the mineable quantity by elevation above sea level within a ultimate pit design. It illustrates how deep the various quality materials are within a pit design; and
- Figure 15-14 plots the gold grade by elevation above sea level within a pit ultimate pit design.

Deposit characterisation indicates the following:

- Approximately 10.26 Mt in situ ore resource is greater than marginal cut-off grade;
- Overall, a consistent strip ratio;

- Grade substantially high at surface; and
- Grade is consistent with depth;

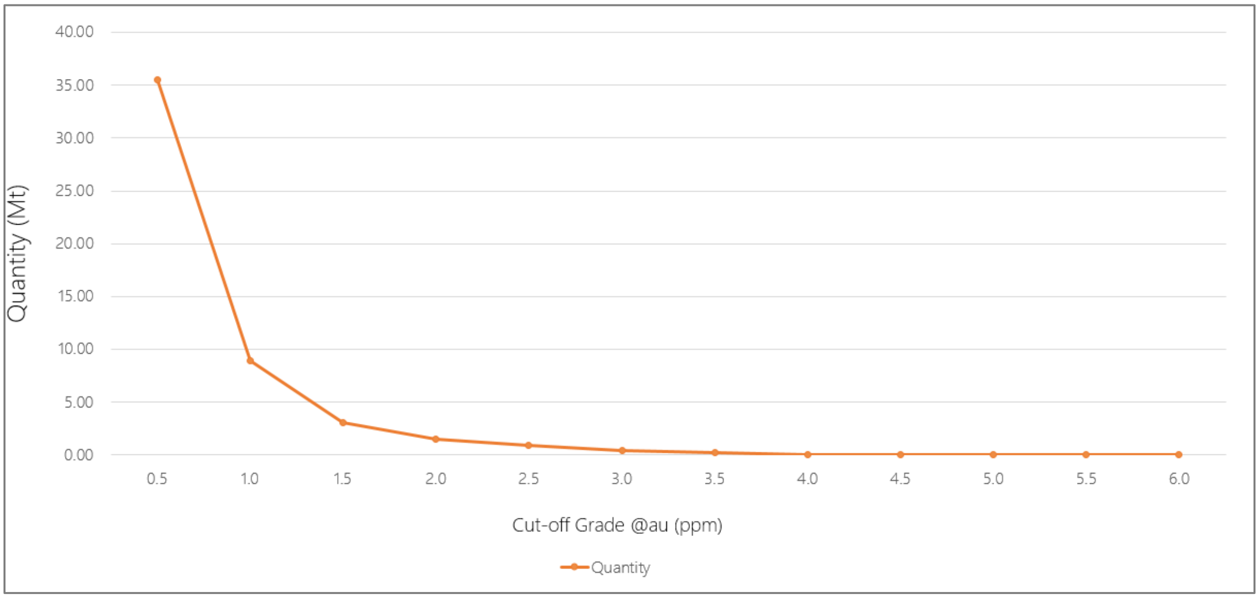


Figure 15-12: Grade-Tonnage Curve

(Measured and Indicated Quantities above Cut-Off Grade)

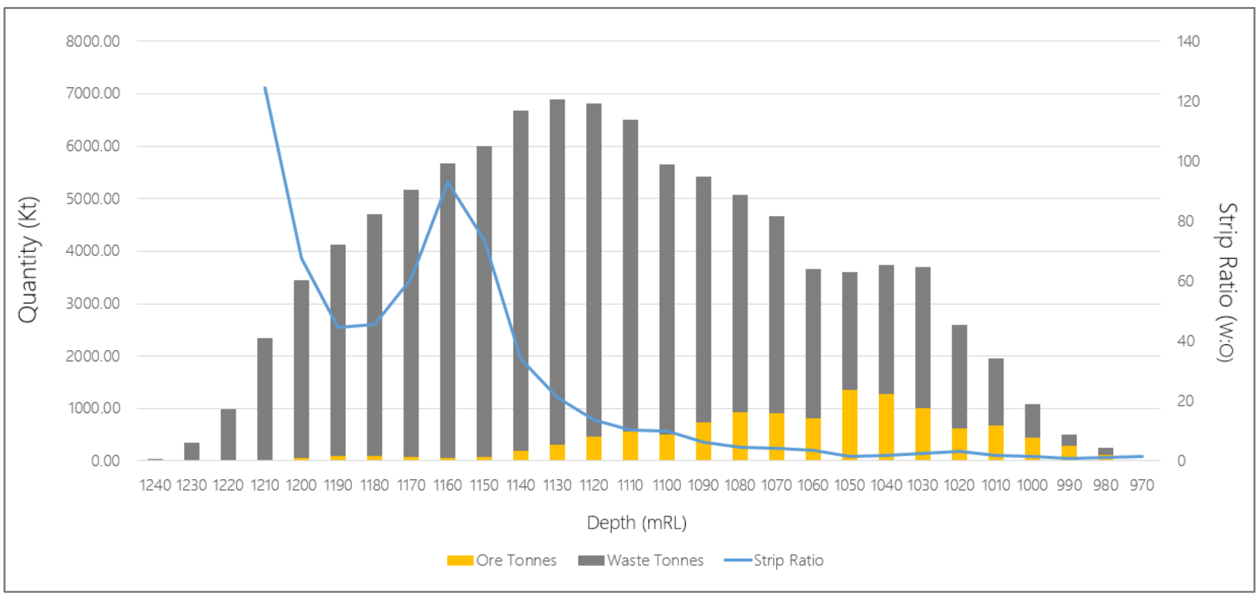


Figure 15-13: ROM Quantities by Depth
(Measured and Indicated Quantities above Cut-Off Grade)

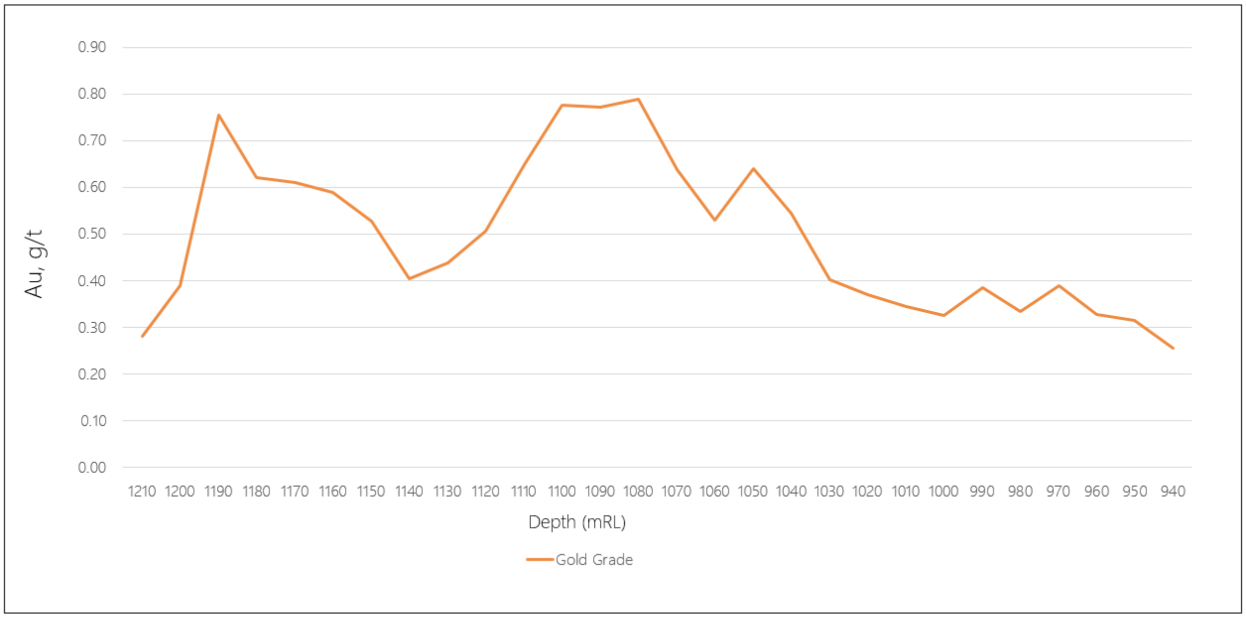


Figure 15-14: Gold Grade by Depth
(Measured and Indicated Quantities above Cut-Off Grade)

15.2.5 Mine Phase Design

A systematic approach was undertaken to identify a preferred development strategy for the mine. The first stage involved visually examining the results from the pit limit optimisation process. The changes in the shape of the optimal pit shell from low to higher revenue factors reflect potential development strategies from high value to lower value mining areas. The characteristics of the final pit shell were also examined as the shape, depth and potential mineable quantities all influence the mining rate and development. The second stage involved examining the results of the deposit characterisation analysis.

The key strategic results from the pit optimisation and the deposit characterisation were:

- Upper benches in the ultimate pit have higher strip ratios;
- Strip ratio substantially improves with depth; and

From the above results, the preferred approach was to undertake a cutback development strategy. Pit analysis indicated seven cutbacks to be preferable, which together with the ultimate pit design, results in eight stages of mining. Eight cutbacks were designed in Geovia Surpac™ based on a mining sequence from north to south. Cutbacks were designed in order to defer stripping as much as possible and to focus on targeting the high grade ore. In order to achieve this, constraints were set in Surpac™ during design and further to this during the scheduling process in MineSched™.

The incremental ore quantities and grades associated with the cutback designs are set out in **Table 15-11**.

Table 15-11: Mineable Quantities By Cutback

Phase	Total Material		CIP ORE			HEAP LEACH ORE			Waste Tonnage	Strip Ratio
	Volume	Tonnage	Tonnage	Au, g/t	Metal, kg	Tonnage	Au, g/t	Metal, kg		
P01_Pit-2	329,000	845,000	269,000	1.06	300	390,000	0.26	100	186,000	0.69
P02_Pit-5	427,000	1,110,000	175,000	1.01	200	72,000	0.31	20	863,000	4.95
P03_Pit-3	5,393,000	13,581,000	2,469,000	0.96	2,400	2,670,000	0.33	900	8,442,000	3.42
P04_Pit-4	7,291,000	18,800,000	1,440,000	1.58	2,300	1,199,000	0.29	300	16,161,000	11.22
P05_Pit-4	5,936,000	15,434,000	1,848,000	1.5	2,800	1,036,000	0.29	300	12,550,000	6.79
P06_Pit-2	4,568,000	10,910,000	830,000	1.15	1,000	1,789,000	0.28	500	8,291,000	9.98
P07_Pit-3	12,126,000	30,865,000	3,155,000	1.03	3,200	4,465,000	0.3	1,300.00	23,245,000	7.37
P08_Pit-6	3,102,000	8,117,000	690,000	1.12	800	800,000	0.31	300	6,627,000	9.61
Total	39,172,000	99,662,000	10,876,000	1.18	13,000	12,421,000	0.3	3,720	76,365,000	7.02

**All figures are rounded to reflect the relative accuracy of the estimate. Numbers may not add exactly due to rounding.*

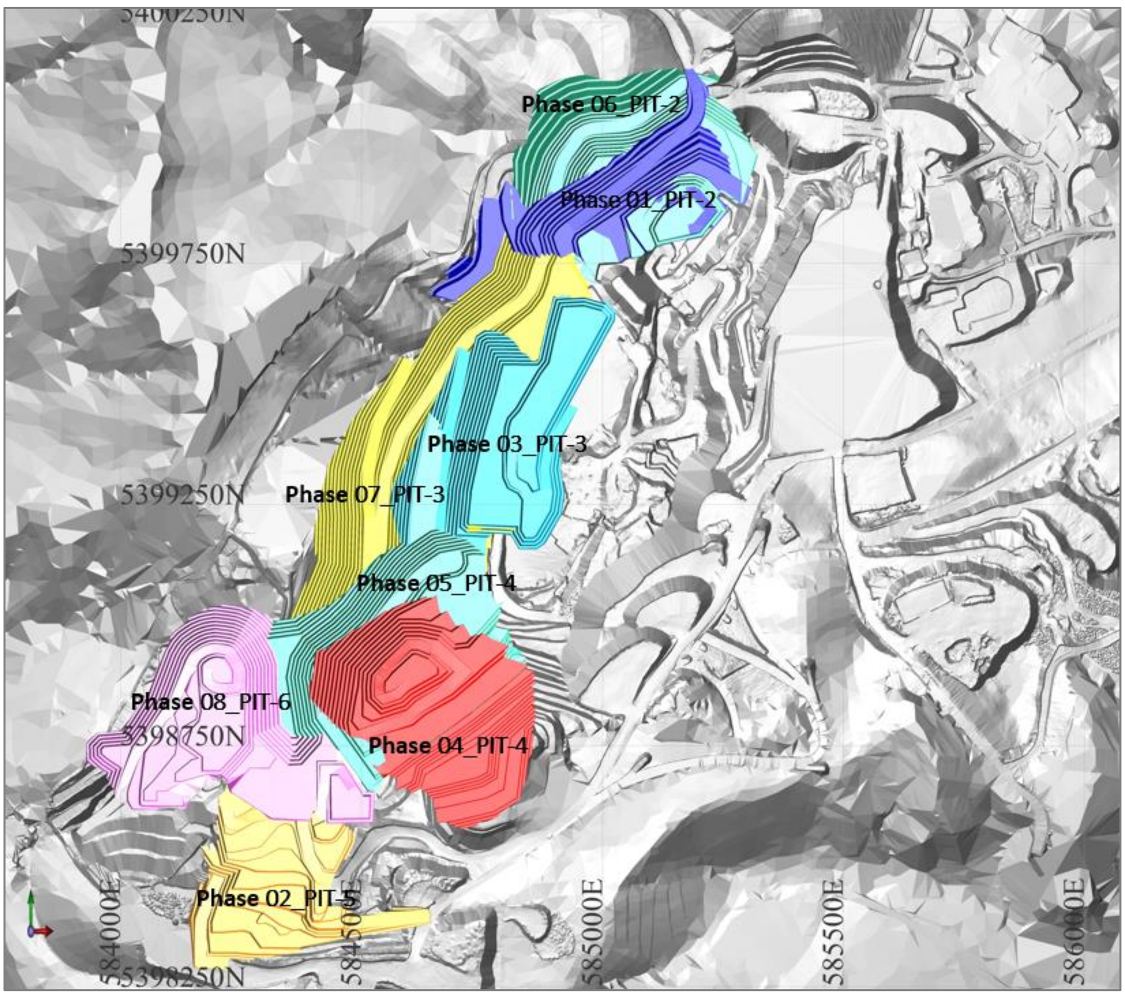


Figure 15-15: Cutback Designs for Boroo, Plan View

15.2.6 Ultimate Pit Design

The pit footprint is designed at approximately 2000 m long by 400 m wide, with the long axis of the pit aligned in a roughly N-S direction. The pit design demonstrates the viability of accessing and mining the economic reserve at Boroo. Pit design was completed using Geovia Surpac™ software and was based on the Whittle™ shell RF 1.0 outlined in Section 15.2.3. In addition, the pit design complies with the geotechnical parameters previously stated in Section 15.2 and the following additional mining parameters:

- Minimum mining width of 30 m, chosen to accommodate the highest possible ore retrieval using the equipment selected.
- Ramp width of 25m at a gradient of 10% for crest to lowest two benches, with a ramp width of 15m and gradient of 10% for access to lower sections of the orebody in lowest two benches.
- Overall pit wall height no greater than 150 m in any area.

January 01 2024 pit survey and the ultimate pit design is shown in Figure 15-16, Figure 15-17 and Figure 15-18 below. The ultimate pit design improved on the RF 1.0 shell and was able to bring more practical into the final design.

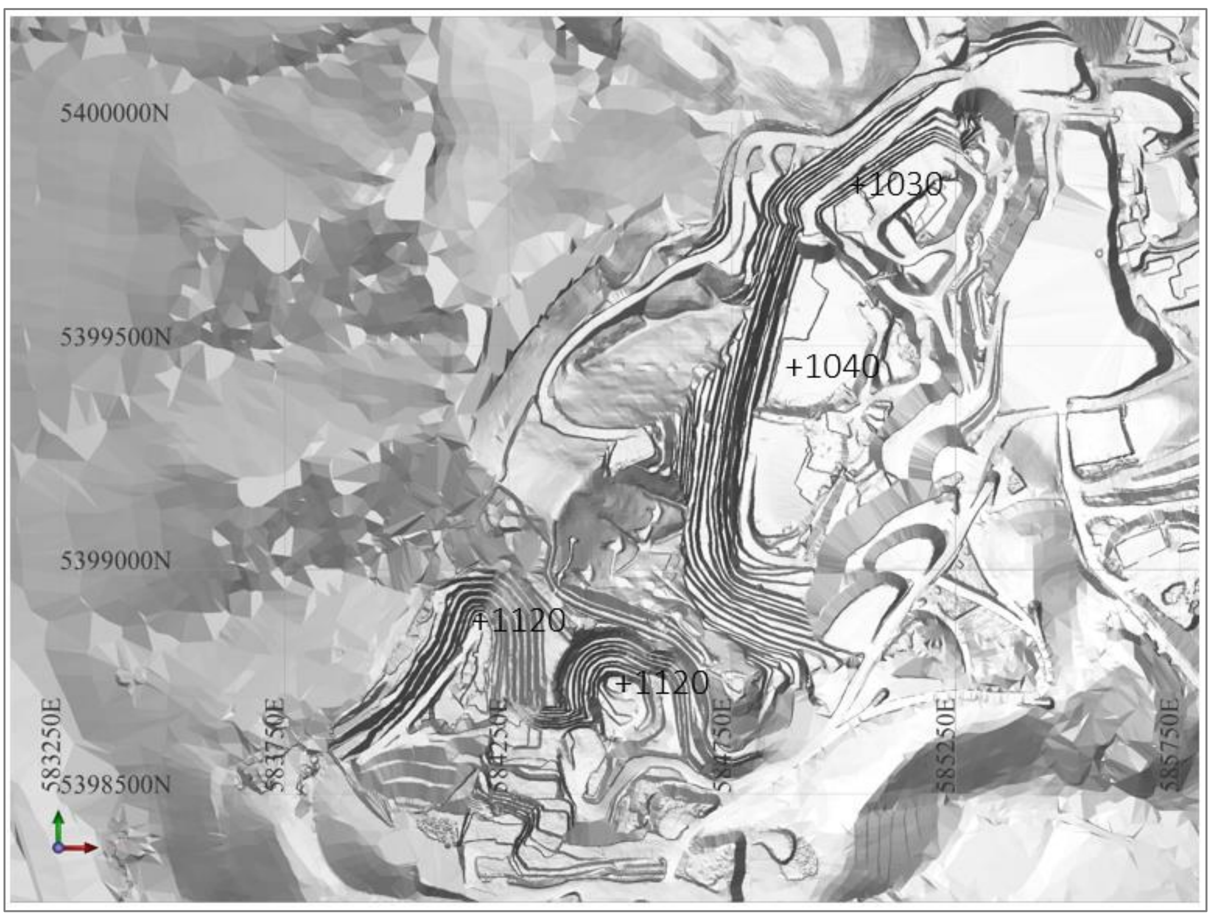


Figure 15-16: January 01 2024 pit survey, Plan View

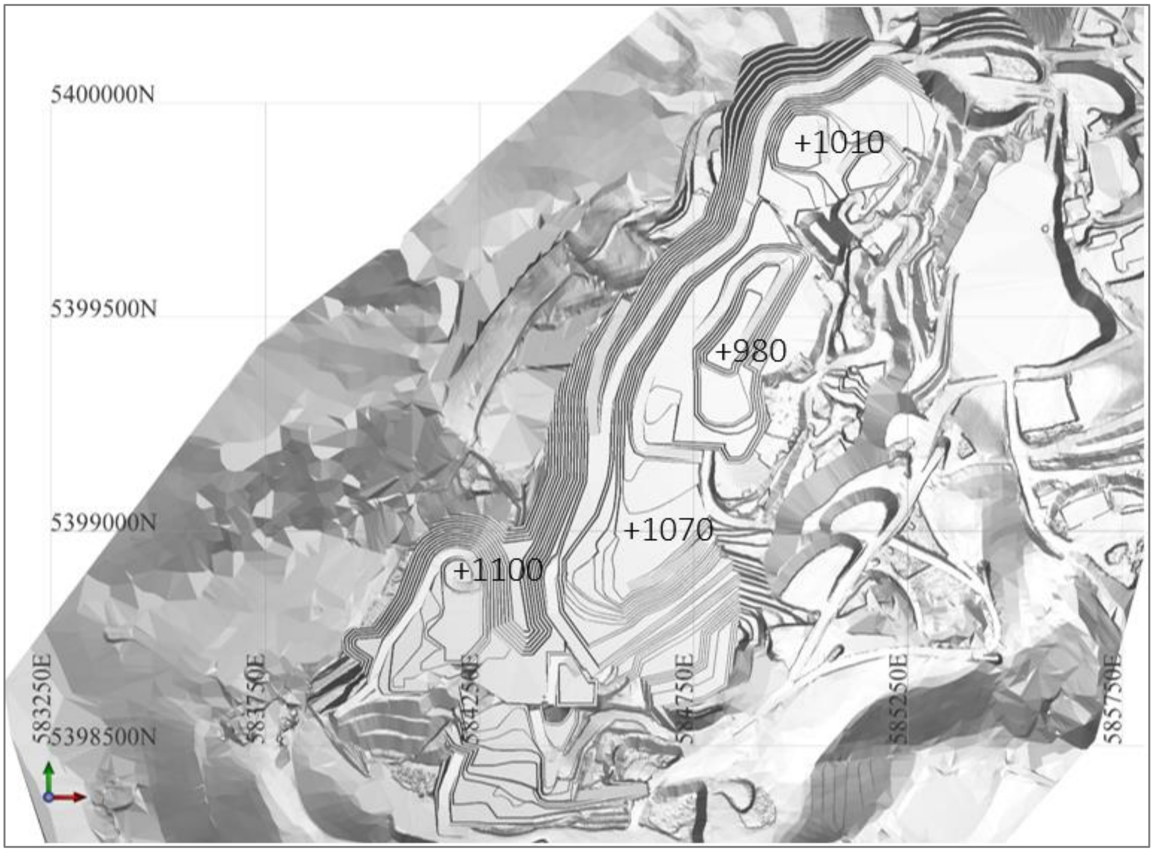


Figure 15-17: Boroo Ultimate Pit Design, Plan View

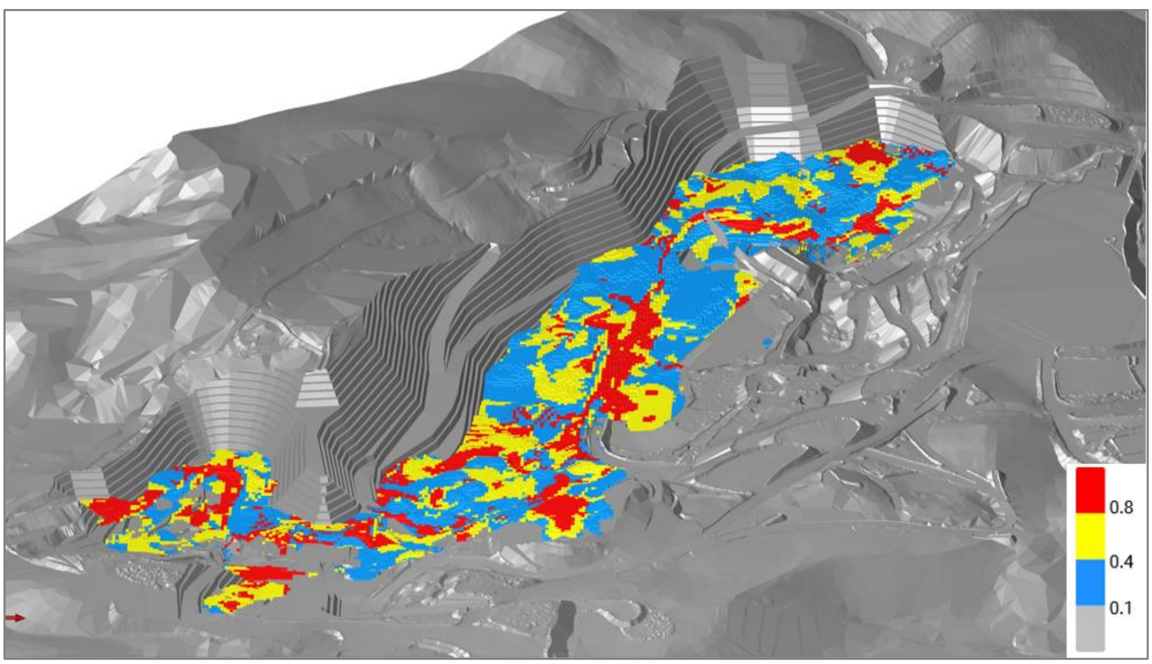


Figure 15-18: Boroo Ultimate Pit Design, Oblique View, Looking Southeast

15.2.6 Mine Haul Road Design

Haul roads in the Boroo pit are designed to provide both safe and efficient haulage routes from the base to the crest of the pit, and onwards to the processing plant and waste storage facility. All site haul roads outside of the Boroo pit are designed as two-lane roads. The use of one-lane roads is limited to the lowest two benches in the Boroo pit. Roads are designed with the following Mines Regulations specifications:

- For dual lane traffic, a minimum haul road width of no less than three times the width of the widest haulage vehicle used on the road.
- For single lane traffic, a minimum haul road width of no less than twice the width of the widest haulage vehicle used on the road.
- Provision for a safety berm with a height at least three-quarters the height of the largest tire on any vehicle operating on the road where a drop-off greater than 3 m exists.

Based on a 100 t CAT 777 model haul truck, the following parameters for haul roads have been set:

- Largest vehicle overall width: 7 m
- Double-lane haul road width: 25 m
- Single-lane haul road width: 15 m
- Roads are designed with allowances for ditches and culverts and will have a total thickness of 1 m, created with surfacing and base layers to improve durability.

15.2.6.2 Minimum Mining Width

A minimum mining width has been maintained between pit areas and at the deepest portions of the ultimate pit which is intended to allow efficient mining operations. For this study and the size of equipment chosen the minimum mining width required conforms to 30 m, which in turn maximizes the extraction of the available resources.

15.2.7 Mineral Reserve Statement

- Proven and Probable Reserves for the Boroo operation are inclusive of mineral resources and based on a five-year moving average gold price of $1,750/oz.
- Mining costs of $1.77/t, milling costs of $14.99/t and general and administrative costs of $2.22/t, Heap leaching costs of 2.39$/t have been used to estimate the reserves along with the gold price stated above.
- The pit reserves reported below assume full mine recovery and are reported on a dry in-situ basis.
- In the block model, no additional provisions were introduced to account for external dilution or losses during mining, while these factors always occur to some degree during mining, it would appear from the reliability of the block model relative to the production results obtained to date that the required levels of adjustment for these factors has been adequately accounted for in the initial interpolation and later unsmoothing introduced during block model creation.

- The cut-off grade used to report the reserves has been chosen by Game Mine at greater than 0.10 g/t gold for heap leach ore and greater than 0.43, 0.46 and 0.52 g/t gold for milling depends on mill recovery domain.
- Reserve estimates were completed using a pit-constrained resource, with an economic pit shell designed in Geovia Surpac™ and January 01 2024 pit survey.
- Mineral Reserves were estimates has been compiled under the supervision of QP Tuvshinbayar Batbayar, in accordance to National Instrument 43-101 standards and with reference to the 2014 Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards (2014 CIM Definition Standards) and the 2019 CIM Best Practice Guidelines.

Table 15-12: Mineral Reserve Statement for Boroo Deposit, Mongolia: Game Mine LLC., January 01st, 2024

Reserve Category	Quantity (tonnes)	Average Grade (Au g/t)	Contained Metal (oz)
CIP Ore Stockpile			
Proven	768,000	1.25	31,000
Probable	-	-	-
Proven and Probable	768,000	1.25	31,000
CIP Ore			
Proven	7,318,000	1.2	282,000
Probable	3,558,000	1.15	131,000
Proven and Probable	10,876,000	1.18	413,000
Total CIP Ore			
Proven	8,085,000	1.2	313,000
Probable	3,558,000	1.15	131,000
Proven and Probable	11,644,000	1.19	444,000
Heap Leach Ore Stockpile			
Proven	282,000	0.3	3,000
Probable	-	-	-
Proven and Probable	282,000	0.3	3,000
Heap Leach Ore			
Proven	8,176,000	0.3	79,000
Probable	4,246,000	0.3	41,000
Proven and Probable	12,421,100	0.3	120,000
Total Heap Leach Ore			
Proven	8,457,000	0.3	82,000
Probable	4,246,000	0.3	41,000
Proven and Probable	12,703,000	0.3	123,000
Total Reserve			
Proven	16,542,000	0.74	395,000
Probable	7,804,000	0.69	172,000
Proven and Probable	24,346,000	0.72	567,000

Notes:

1. The effective date of the Mineral Reserve estimate is February 01st, 2024. Mineral Reserves were estimates has been compiled under the supervision of QP Tuvshinbayar Batbayar.
2. The Mineral Reserve estimates were prepared with reference to the 2014 Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards (2014 CIM Definition Standards) and the 2019 CIM Best Practice Guidelines.
3. Reserves estimated assuming open pit mining methods
4. Reserves are reported on a dry in-situ basis
5. The cut-off grade used to report the reserves has been chosen by Game Mine at greater than 0.1 g/t gold for heap leach ore and greater than 0.43, 0.46 and 0.52 g/t gold for milling depends on mill recovery domain.
6. Reserves are based on a gold price of $1,750/oz, mining cost of $1.77/tonne, milling costs of $14.99/t and general and administrative costs of $2.22/t. Heap leaching costs of 2.39$/t. Heap leaching recovery 40%.
7. In the block model, no additional provisions were introduced to account for external dilution or losses during mining, While these factors always occur to some degree during mining, it would appear from the reliability of the block model relative to the production results obtained to date that the required levels of adjustment for these factors has been adequately accounted for in the initial interpolation and later unsmoothing introduced during block model creation.
8. All figures are rounded to reflect the relative accuracy of the estimate. Numbers may not add exactly due to rounding.

15.2.8 Relevant Factors

Game Mine is not aware of any existing environmental, permitting, legal, socio-economic, marketing or political factors that are likely to materially affect the mineral reserve estimate. Mineral reserves have been economically tested to ensure that they are economically viable. The project remains economic across a range of key input parameters.

If for any reason any of these project cost factors are changed such that the project capital or operating cost estimates change materially, then the mineral reserve estimates stated in this report could be materially affected.

A full sensitivity analysis of the operating and financial inputs affecting the Boroo is presented in Chapter 22.

15.3 Ulaanbulag

15.3.1 Introduction

Mineral Reserves for the Ulaanbulag deposit are based on the Measured and Indicated Resources presented in Chapter 14 and use engineering designs for the pit and associated operating parameters. Reserve calculations are valid at the time of estimation and use cut-off grade assumptions which were made prior to finalization of the economic model. The Mineral Reserve estimates are based on a mine plan and pit design developed using modifying parameters including metal price, metal recovery based on performance of the processing plant, and operating cost estimates.

Game Mine confirmed that there were no periods of negative cash flow following project start-up and that overall project economics are favorable at a five-year moving average gold price of $1,750/oz.

Game Mine adopted standard mine planning processes to determine the Mineral Reserve estimate for the Ulaanbulag deposit. The following inputs and constraints were utilized for pit optimization and further defined in the following sections:

- Resource model with associated assay grades and densities for mineralized zones (Chapter 14)
- January 01 2024 pit survey provided by Boroo Gold
- Metallurgical recoveries (Table 15-30)
- Geotechnical slope parameters (Table 15-28)
- Gold price of $1,750/oz
- Operating cost assumptions, including mine, mill, and G&A (Table 15-29)
- Dilution (Section 15.3.3.1)
- Processing rate of 5000 t/day (Section 15.3.3.4)
- Balanced total material movement per annum (Chapter 16)

15.3.2 Geotechnical Parameters

15.3.2.1 2022 Geotechnical Study

The Ulaanbulag mine design parameters identified as a result of a 2016 study revealed a total of 185 drill hole data from 2009 to 2010, 81 of which were calculated using empirical methods based on geotechnical logging data provided by geotechnical engineering.

On November 13, 2021, the "Ulaanbulag" conducted a back analysis of the failure of the western wall, using the results of the analysis to identify the geotechnical risks.

Combined with the recommendations made in the two technical memorandums mentioned above, the geotechnical analysis was carried out based on current available information.

Based on geotechnical data from 2009 to 2010, the Barton Q system developed a model for the rock mass rating. Because geotechnical logging data is directly defined the meanings of Jr and Ja, which cannot be calculated by the Laubsher assessment 1990 (RMR90) and the Bieniavsky Assessment 1976 (RMR76).

Q assessment yielded values of 1 (poor) to 10 (moderate) in diorite, granite and sandstone. Weathering caused degradation of the Q values, in all lithologies.

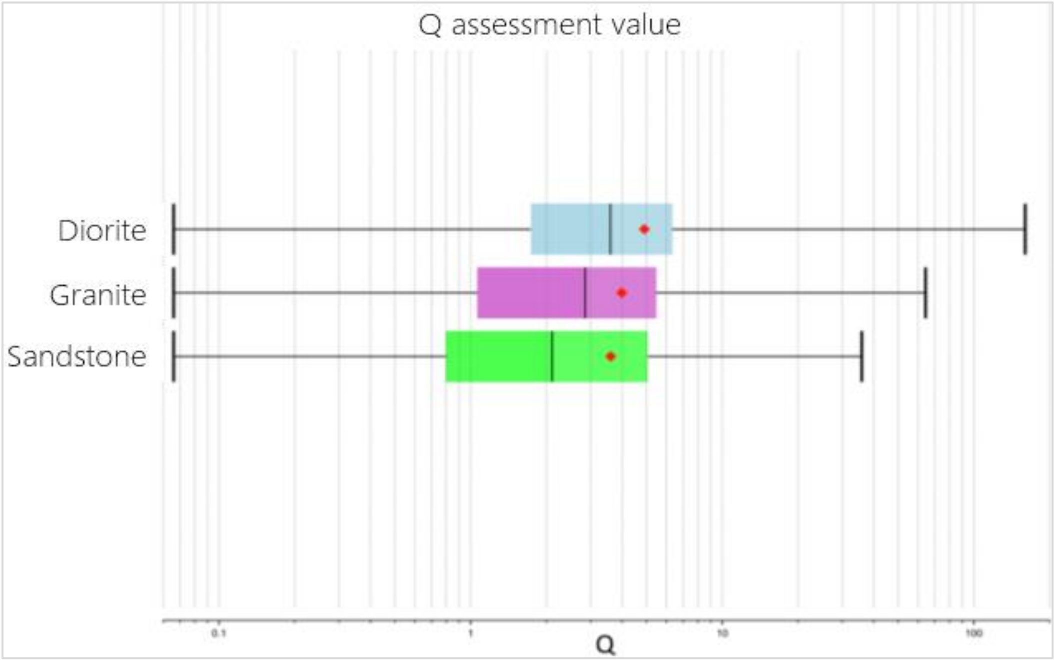


Figure 15-19: Q value assessment by lithology

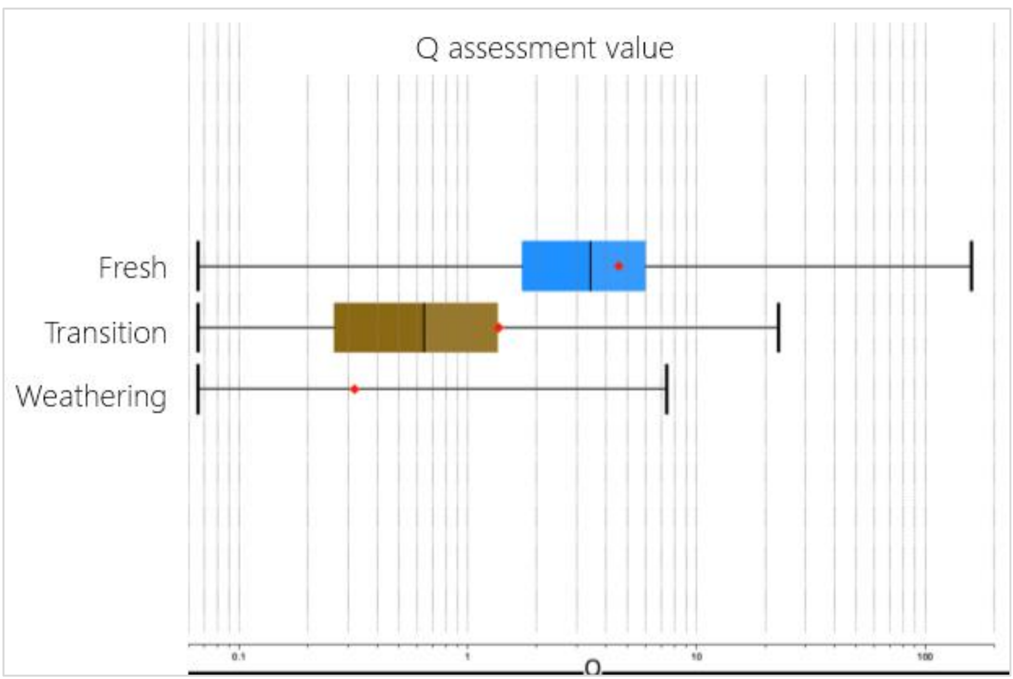


Figure 15-20: Q value assessment

Because the western walls of the mine are not deep enough, the final wall mapping was not conducted, and the geotechnical design parameters in the western part were described as empirical.

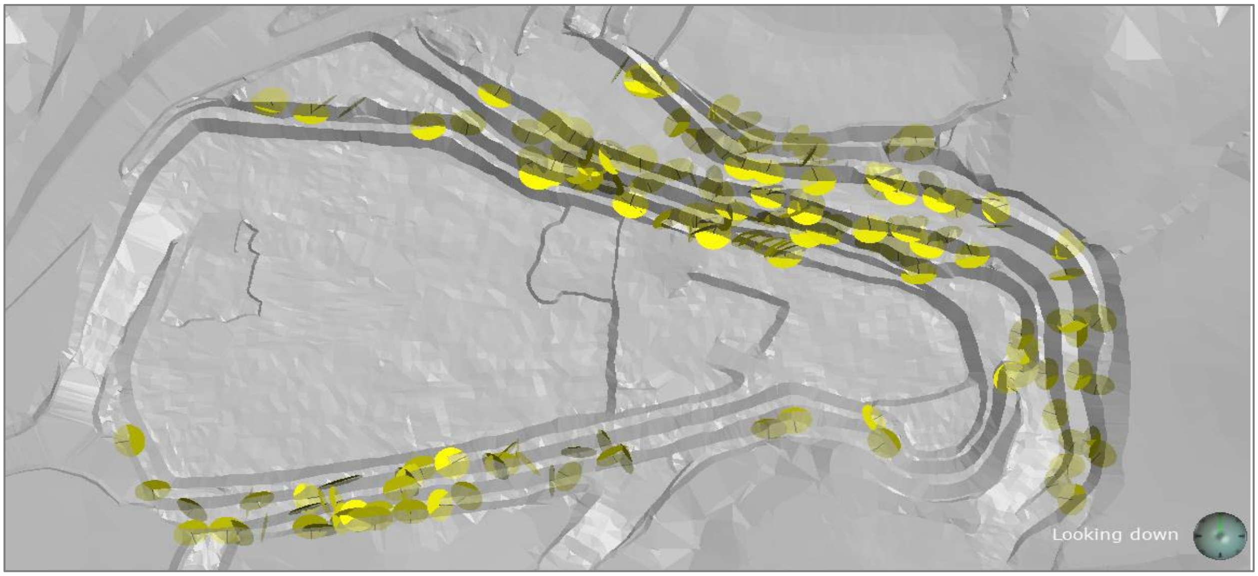


Figure 15-21: Structural measurement of wall mapping

Cluster analysis was used with groupings of structure elements in diorite (Figure 15-22), granite (Figure 15-23) and sandstone (Figure 15-24).

Diorite had four set types with horizontally and vertically dipping structures with a moderately south-north dipping fracture. For the northern wall, the possible plane failure can occur on the left-hand side of 165°-180°C, along with Seth#1 and on the left-hand side of the wall at 005°-015°C. The collapse of the rock can be formed on the slopes of the northern walls at 175°-180°C. The possible wedge collapse of the rock can be formed on the slopes of the northern walls at 175°-180°C.

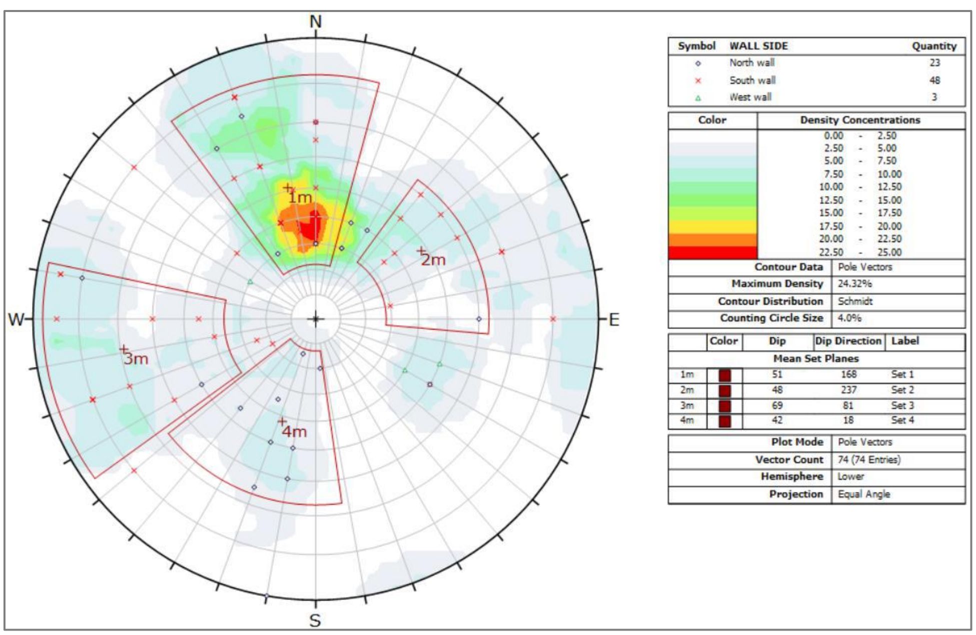


Figure 15-22: Cluster Analysis of Diorite

Granite had 4 set types with moderately dipping structures. For the northern wall, the plane collapse can be formed along Set#2 on the slopes of 205°-215° in the west. Less likely to cause wedge and toppling failure.

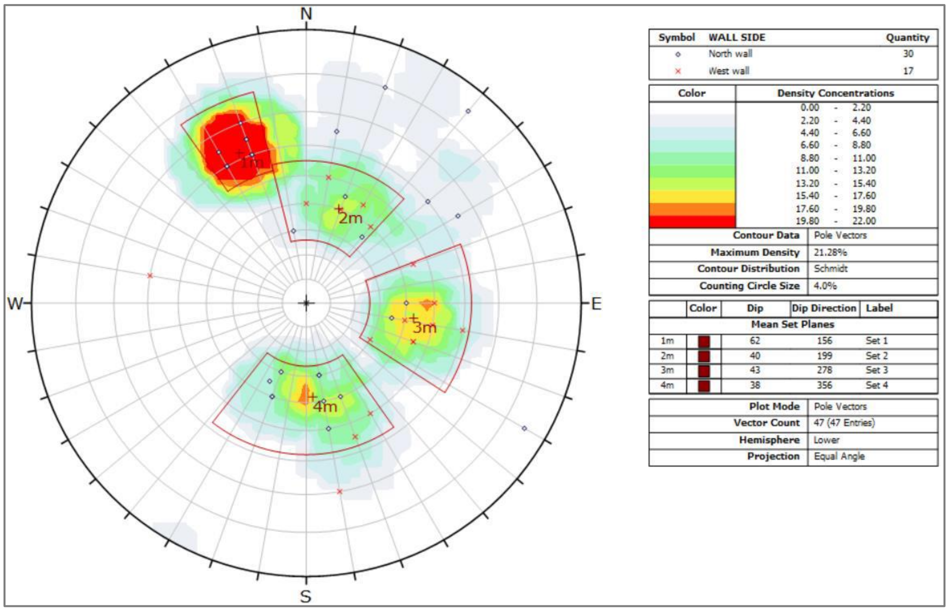


Figure 15-23: Cluster Analysis of Granite

Sandstone had 3 set types with moderately dipping structures. For the northern wall, the plane collapse can be formed along Set#2 on the slopes of 175°-185°C. Less likely to cause wedge and toppling failure.

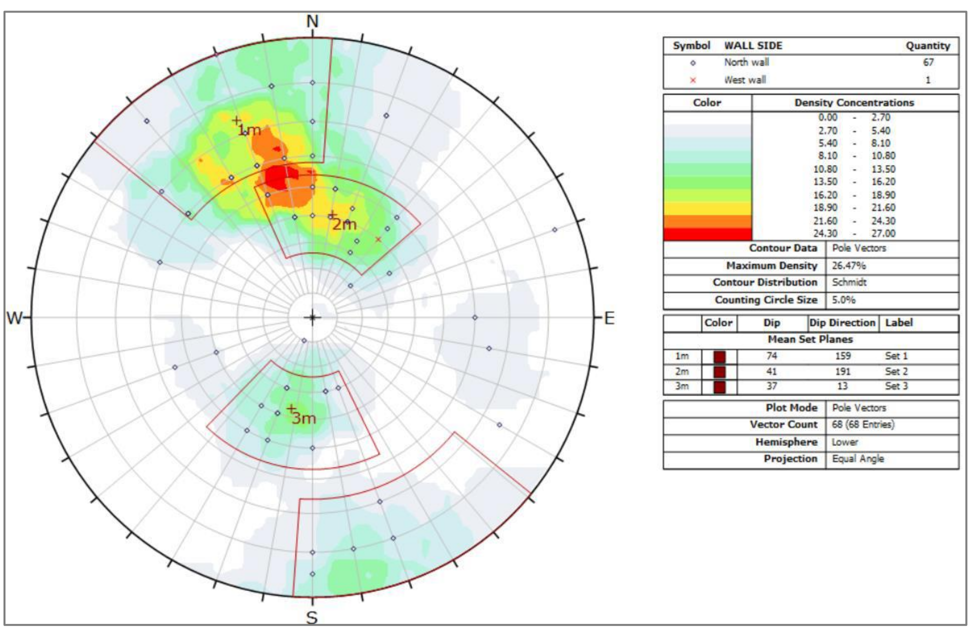


Figure 15-24: Cluster Analysis of Sandstone

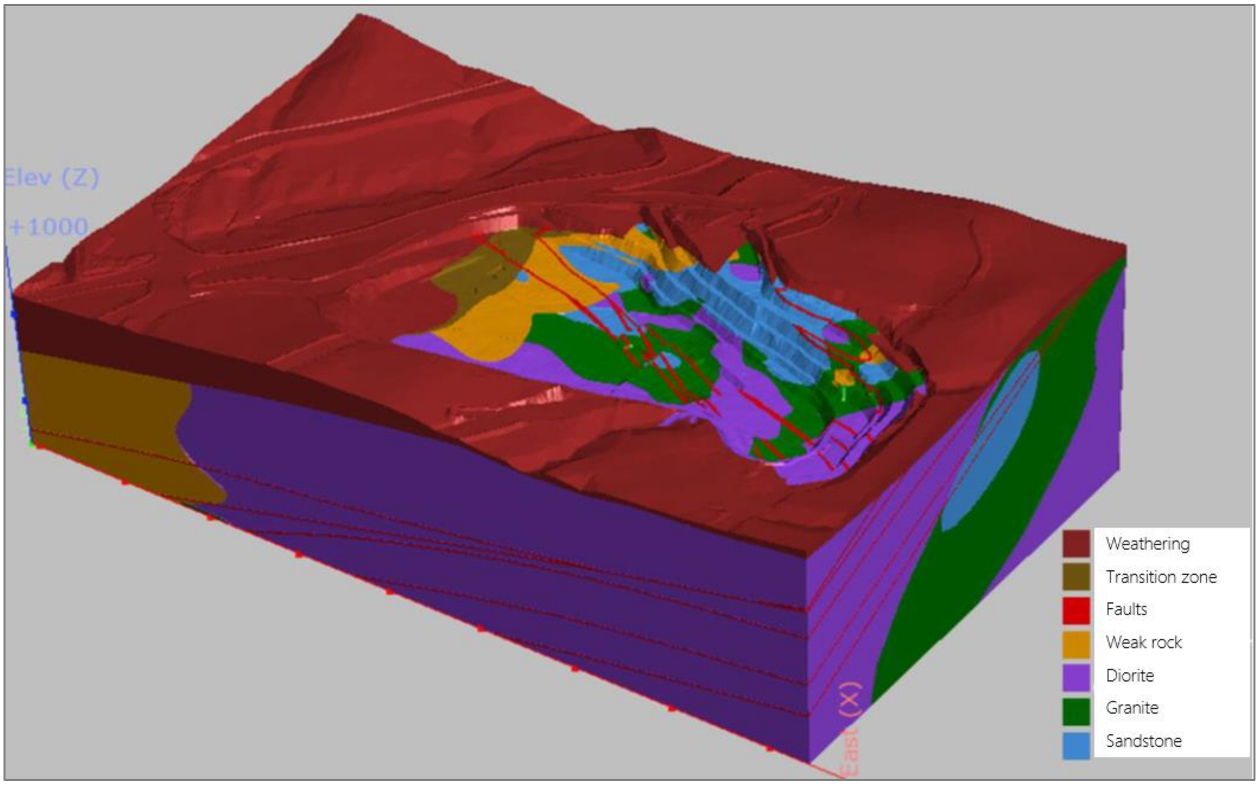


Figure 15-25: Geotechnical domain of Ulaanbulag

For the sets of diorite, sandstone and granite, ITASCA KATS software calculated wedge and plane failure modes probabilities, collapsed material volume, and upper edge crack width using kinematic analysis. The input parameters for this analysis are as in **Table 15-13:**

Table 15-13: Parameters and Criteria Used in Kinematic Analysis

Parameter	Value	Units
Specific gravity	2.53	t/m^3
Pore water pressure	-	%
Bench height	10	m
Berm width	5	m
Angle between calculation steps	30	degree
Angular limit of planar failure	30	degree
Swell factor	1.3	-
Natural slope angle	37	degree
Angular limit of wedge failure	30	degree
Fracture resistance		kPa
Friction angle of fractures	25	degree
Factor of Safety	≤ 1	
Probability	≤ 50	%

The analysis, along with the rock mass characteristics, formed the basis for the geotechnical design parameters for the mine design, as shown in **Table 15-14** and **Table 15-15**, below. The recommended geotechnical parameters for the mine are highlighted in blue.

Table 15-14: Diorite Recommended Pit Design Parameters

Design Sector	Bench Face Angle	Bench Height	Bench Slope Azimuth	Internal Slope Angle			Berm Width		
				Design	Ritchie	Optimized	Design	Ritchie	Optimized
UB_000	70	10	345-015	48	44	48	5.0	7.0	5.0
UB_030	70	10	015-045	48	44	48	5.0	7.0	5.0
UB_060	70	10	045-075	48	44	48	5.0	7.0	5.0
UB_090	70	10	075-105	48	44	48	5.0	7.0	5.0
UB_120	70	10	105-135	45	44	45	6.0	7.0	6.0
UB_150	70	10	135-165	48	44	48	5.0	7.0	5.0
UB_180	70	10	165-195	48	44	48	5.0	7.0	5.0
UB_210	70	10	195-225	49	44	70	5.0	7.0	0.0
UB_240	70	10	225-255	48	44	48	5.0	7.0	5.0
UB_270	70	10	255-285	48	44	48	5.0	7.0	5.0
UB_300	70	10	285-315	49	44	70	5.0	7.0	0.0
UB_330	70	10	315-345	49	44	70	5.0	7.0	0.0

Table 15-15: Granite Recommended Pit Design Parameters

Design Sector	Bench Face Angle	Bench Height	Bench Slope Azimuth	Internal Slope Angle			Berm Width		
				Design	Ritchie	Optimized	Design	Ritchie	Optimized
UB_000	70	10	345-015	48	44	48	5.0	7.0	5.0
UB_030	70	10	015-045	49	44	70	5.0	7.0	0.0
UB_060	70	10	045-075	49	44	70	5.0	7.0	0.0
UB_090	70	10	075-105	49	44	70	5.0	7.0	0.0
UB_120	70	10	105-135	49	44	70	5.0	7.0	0.0
UB_150	70	10	135-165	49	44	51	5.0	7.0	4.0
UB_180	70	10	165-195	48	44	48	5.0	7.0	5.0
UB_210	70	10	195-225	48	44	48	5.0	7.0	5.0
UB_240	70	10	225-255	45	44	45	6.0	7.0	6.0
UB_270	70	10	255-285	48	44	48	5.0	7.0	5.0
UB_300	70	10	285-315	48	44	48	5.0	7.0	5.0
UB_330	70	10	315-345	48	44	48	5.0	7.0	5.0

Table 15-16: Sandstone Recommended Pit Design Parameters

Design Sector	Bench Face Angle	Bench Height	Bench Slope Azimuth	Internal Slope Angle			Berm Width		
				Design	Ritchie	Optimized	Design	Ritchie	Optimized
UB_000	70	10	345-015	48	44	48	5.0	7.0	5.0
UB_030	70	10	015-045	48	44	48	5.0	7.0	5.0
UB_060	70	10	045-075	49	44	70	5.0	7.0	0.0
UB_090	70	10	075-105	49	44	70	5.0	7.0	0.0
UB_120	70	10	105-135	49	44	70	5.0	7.0	0.0
UB_150	70	10	135-165	49	44	70	5.0	7.0	0.0
UB_180	70	10	165-195	48	44	48	5.0	7.0	5.0
UB_210	70	10	195-225	48	44	48	5.0	7.0	5.0
UB_240	70	10	225-255	49	44	70	5.0	7.0	0.0
UB_270	70	10	255-285	49	44	70	5.0	7.0	0.0
UB_300	70	10	285-315	49	44	70	5.0	7.0	0.0
UB_330	70	10	315-345	49	44	70	5.0	7.0	0.0

Although kinematic analysis was not performed for quaternary sediments/weathered rock mass material (Table 15-17) and transition material (Table 15-18), mine design parameters were still estimated based on rock mass characteristics and indentation angles.

Table 15-17: Weathered Recommended Pit Design Parameters

Domain	Design Sector	Bench Face Angle	Bench Height	Bench Slope Azimuth	Internal Slope Angle	Berm Width
Weathering Zone	UG_000	55	10	345-015	36.5	6.5
	UG_030	55	10	015-045	36.5	6.5
	UG_060	55	10	045-075	36.5	6.5
	UG_090	55	10	075-105	36.5	6.5
	UG_120	55	10	105-135	36.5	6.5
	UG_150	55	10	135-165	36.5	6.5
	UG_180	55	10	165-195	36.5	6.5
	UG_210	55	10	195-225	36.5	6.5
	UG_240	55	10	225-255	36.5	6.5
	UG_270	55	10	255-285	36.5	6.5
	UG_300	55	10	285-315	36.5	6.5
	UG_330	55	10	315-345	36.5	6.5

Table 15-18: Transition zone Recommended Pit Design Parameters

Domain	Design Sector	Bench Face Angle	Bench Height	Bench Slope Azimuth	Internal Slope Angle	Berm Width
Transition zone	UG_000	65	10	345-015	42	6.5
	UG_030	65	10	015-045	42	6.5
	UG_060	65	10	045-075	42	6.5
	UG_090	65	10	075-105	42	6.5
	UG_120	65	10	105-135	42	6.5
	UG_150	65	10	135-165	42	6.5
	UG_180	65	10	165-195	42	6.5
	UG_210	65	10	195-225	42	6.5
	UG_240	65	10	225-255	42	6.5
	UG_270	65	10	255-285	42	6.5
	UG_300	65	10	285-315	42	6.5
	UG_330	65	10	315-345	42	6.5

15.3.2.2 Stability Calculations

The slope stabilization limits were assessed for Ulaanbulag pit using a three-dimensional geological model analysis performed with SLIDE3D software. Factors of safety, to meet stability criteria, were: 1.0 for a single bench, 1.25 for a slope between haul roads, and 1.5 for general slopes. Used parameters for stability analysis, shown in Table15-19.

Table 15-19: Parameters for slope stability analysis

Material Type	Specific gravity (κN/m3)	Cohesion (κPa)	Friction angle (°)	UCS2 (κPa)	GSI3	mi4	D5
Overburden	20.0	-	35	-	-	-	-
Fault	20.0	-	25	-	-	-	-
Diorite	25.1	-	-	58,000	50	25	0.70
Granite	24.7	-	-	56,000	51	32	0.70
Sandstone	24.5	-	-	49,000	40	17	0.70

As a result of the slope stability analysis, the results were 1-2 benches instability determined (FOS≤0.74) in the upper part of the western wall, which is intersected with overburden.

For the MD fault section shows the results of FoS=1.232, the lowest safety coefficient and hydrogeology is considered for analysis due to no seepage is observed.

The stability analysis with water pressure, FOS is dropped to 1 in the north wall which is shown in Figure 15-26.

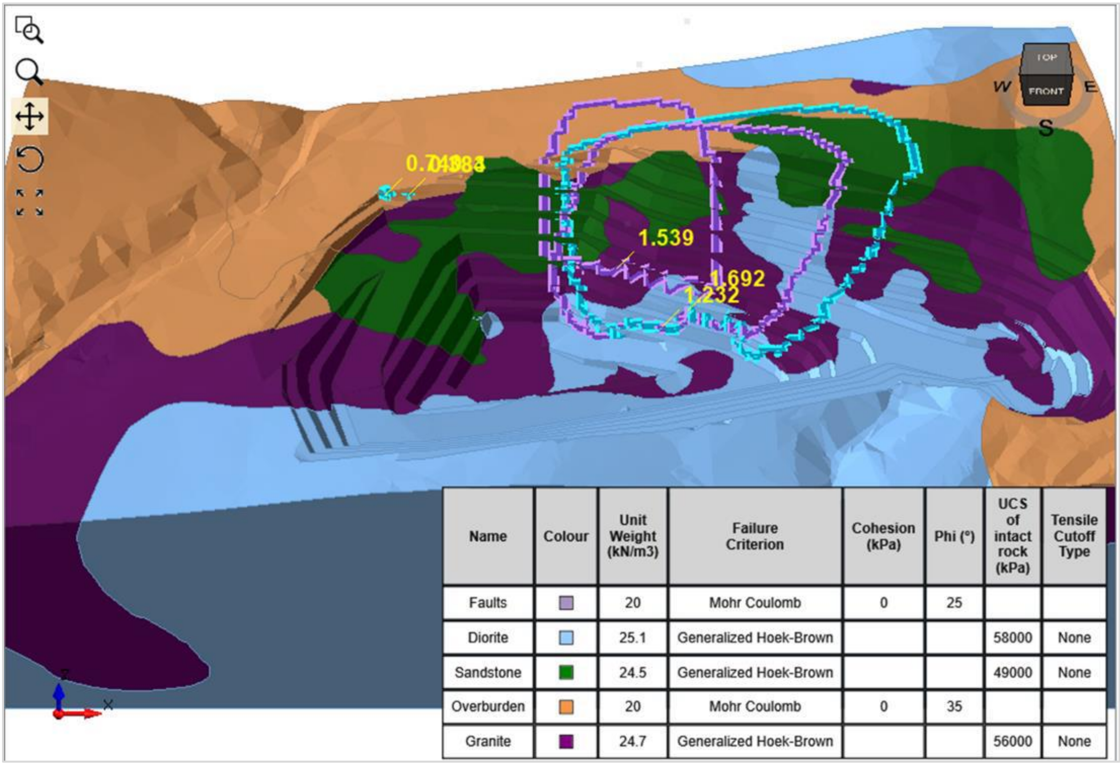


Figure 15-26: The Slope stability analysis (without water pressure)

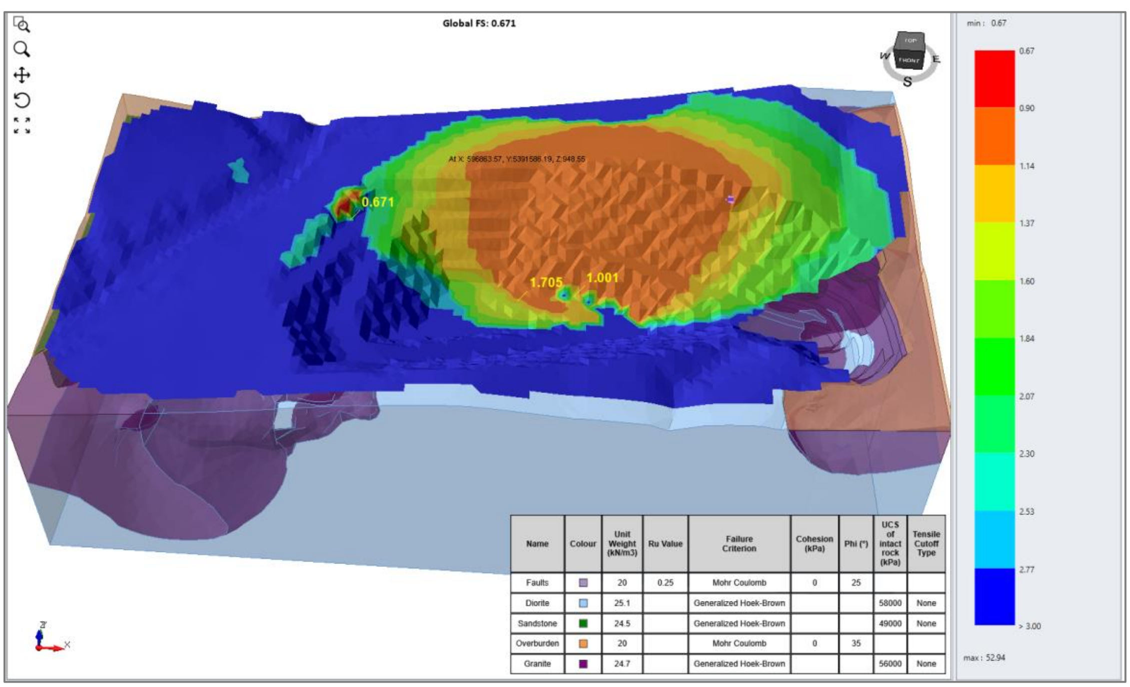


Figure 15-27: The Slope stability analysis (with water pressure)

15.3.3 Pit Limit Optimization

The terminology "pit limit optimization" refers to a process which aims to identify the best value mining pit shape for a given series of inputs and constraints. It does not imply that mining has been "optimized" in other ways; such as optimal mine sequence or optimal equipment selection.

The pit limit optimization process creates a series of pit shells based on a range of metal prices ranging from 30% to 200% of a chosen base price in increments of 10%. The lower the price the smaller the pit and the higher the potential able to be profitably extracted at this lower revenue. Hence this approach not only identifies the high value areas, but also indicates the potential pit extents should future metal price change.

The geology model created for the Mineral Resource estimate was the basis for pit limit optimization using the Whittle 4X Optimizer for gold ore deposits. The optimization process involved the following steps:

- Identify physical constraints;
- Define mining factors, such as geotechnical design parameters and ore loss and dilution;
- Define metallurgical modifying factors;
- Set mine operating cost rates;
- Set product prices; and
- Run optimizer process and report results.

15.3.3.1 Optimization Parameters

Physical Constraints

- Game Mine is not aware of any surface physical constraints to mining; such as infrastructure, rivers or environmental limits.
- A geological constraint was applied that restricted to Measured and Indicated Mineral Resources only.
- January 01 2024 pit survey.

Geotechnical Design Parameters

- Game Mine assumed a 36.5 to 47.4 degree overall pit slope. These estimates are based on a geotechnical report provided by Boroo Gold. Geotechnical Slope Parameters:

Table 15-20: Geotechnical Design Parameters

Domain	Bench Height, (m)	Bench Angle, (°)	Berm Width, m	Overall Slope Angle, (°)
Oxide	10	55	6.5	36.5
Transition	10	65	6.5	42.0
Fresh	10	75	6.5	47.4

Mining Modifying Factors

- With grade control measures and smaller mining equipment, it was assumed that the mineralization will be fully recovered during mining; no Mining Recovery factor was applied. Given the size of the blocks relative to the physical mineralization, it was assumed that the blocks in the geological model are fully diluted; no Mining Dilution factor was applied

Grade/Recovery Inputs

- Recovery estimates are based on a metallurgical report provided by Boroo Gold. A summary of the recovery inputs can be seen in Table 15-21 below.

Table 15-21: Recovery Inputs

Domain	Domain
Oxide	76.32
Transition	70.80
Fresh	66.92
Average	**71.22**

Operating Cost Rates

- Rates of operating costs were applied based on historical operating costs, escalated to 2024.
- Operating costs used for the Optimization process are given in Table 15-22.

Table 15-22: Operating Cost Parameters

Item	Units	Value
Mining Cost	US$/t	1.77
Ore transportation (Ulaanbulag to Boroo)	US$/t ore	1.73
Ore processing	US$/t ore	12.14
Tailings	US$/t ore	2.26
Ore Control	US$/t ore	0.59
Overheads	US$/t ore	2.22
Royalty	% of total revenue	5.00

Product Prices

The Optimizer analysis the deposit at a range of metal prices. The "base case" metal price is called the 100% "revenue factor". The revenue factors are adjusted to effectively adjust the product price. The "base case" gold price applied was US$1750/oz. The price sensitivity analysis was on values from 30% to 200% of the base case of gold price, that is US$525/oz to US$3500/oz.

15.3.3.2 Base Case Analysis Results

The Base Case analysis utilized measured and indicated Resource only, with the inferred Mineral Resource being regards as waste as well as the unmineralised or unclassified material.

A summary of the pit optimization results for the base case parameters is set out in Table 15-23 and illustrated in Figure 15-28. A summary of the pit shell quantities by revenue factor are illustrated in Figure 15-28. Pit 8 (highlighted) is the outcome using the "Base Case" price.

Table 15-23: Base Case Optimization Results

Pit	Revenue	Au Price	Ore	Waste	Total	Strip Ratio	Au grade	Contained Au
	Factor	USD/oz	Mt	Mt	Mt	W:O	g/t	kg
1	0.3	525	0.15	0.32	0.47	2.17	2.86	424.8
2	0.4	700	0.25	0.61	0.86	2.48	2.44	604.8
3	0.5	875	0.41	1.67	2.08	4.07	2.21	906.5
4	0.6	1050	0.56	2.72	3.27	4.87	2.03	1,131.1
5	0.7	1225	0.75	3.84	4.59	5.09	1.81	1,365.7
6	0.8	1400	1.10	6.14	7.25	5.57	1.59	1,751.6
7	0.9	1575	1.48	7.84	9.32	5.31	1.40	2,067.6
8	**1.0**	**1,750**	**1.95**	**11.43**	**13.38**	**5.86**	**1.29**	**2,507.5**
9	1.1	1,925	2.35	14.23	16.58	6.05	1.20	2,834.1
10	1.2	2,100	2.84	17.64	20.48	6.22	1.12	3,190.2
11	1.3	2,275	3.20	19.59	22.78	6.13	1.07	3,410.7
12	1.4	2,450	3.62	21.99	25.61	6.07	1.01	3,652.1
13	1.5	2,625	3.98	24.23	28.21	6.09	0.97	3,850.2
14	1.6	2,800	4.20	25.26	29.46	6.01	0.94	3,955.0
15	1.7	2,975	4.51	26.86	31.37	5.95	0.91	4,095.4
16	1.8	3,150	5.03	30.44	35.47	6.05	0.86	4,340.6
17	1.9	3,325	5.38	31.30	36.69	5.81	0.83	4,461.1
18	2.0	3,500	5.98	33.22	39.20	5.56	0.78	4,664.1

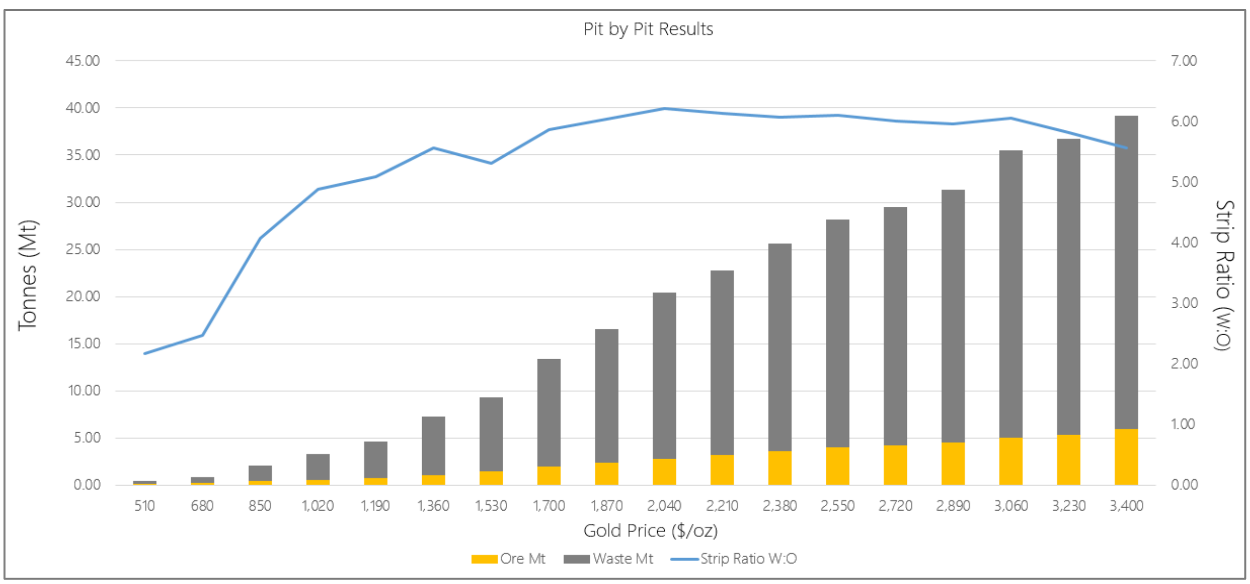


Figure15-28: Base Case Optimization Results

Review of the optimization results indicates the total potential mineable quantity for the Project at a 100% revenue factor is estimated at approximately 1.95 Mt at a grade of 1.29 g/t Au resulting in a strip ratio of 5.86:1 (t waste: t ore). The sensitivity of the mineable quantities to metal price is reflected by the steepness of the curve graphed in Figure 15-28 at changing revenue factors (or price). These results indicate that the Project is only mildly sensitive to economic parameters such as the metal selling price near the base price. A 10% variation in the gold price above and below the base price increases and decreases the mineable ore quantity by approximately 20% to 24% respectively. The results suggest the deposit is highly sensitive to lower gold prices.

Base Case Cash flow Analysis

The basic Whittle pit optimizer result defines the "optimal" pit shell for fixed mining, economic and physical constraints. This outcome, however, is not necessarily the "optimal" result as it does not account for possible changes in value over the mining life. To overcome this issue, Whittle 4X software undertakes a life-of-mine (LOM) cash flow analysis to assess which pit provides the highest economic return taking into account the time value of money.

For the total cash flow of a pit to be calculated, there needs to be a LOM production schedule to allow mining costs and revenues to be determined over time. The Four-X software develops two types of schedules which it refers to as "best case" and "worst case" schedule. The best case schedule assumes mining commences at the inner-most nested pit shell and then expands to successfully larger shells until the selected pit limit is reached. As this sequence also reflects mining from the highest to lowest margin per tonne pit shell, it theoretically produces the "highest" project cash flow. The result should be considered optimistic as the pit shells often are not of practical mining width and hence the staged development represented in the schedule is not achievable in practice.

The worst case schedule assumes mining occurs in horizontal benches, starting from the highest elevation and then proceeding down to the base of the selected pit. There are no interim pit shells mined. As the upper benches typically have the highest strip ratio, this often results in the cash flow approaching the lower boundary for the selected pit.

The average of the "Best Case" and "Worst Case" cash flow schedules is calculated to reflect a more practical outcome. The preferred pit shell is often selected based on this result.

The end of a scheduling year is determined based on user-defined constraints, for example, limits to mining production. It is worth nothing that the Whittle Four-X software does not attempt to optimize a scheduling sequence to best meet all constraints and maximize cash flow, but simply follows the rules dictated by the "best" case and "worst" case scenario schedules. The best method of assessing the performance of a given pit shell and hence its cash flow is to export the data to alternative software packages where more scheduling and financial analysis flexibility and capability exist.

For the cash flow analysis, Game Mine assumed mineable quantity production of 1.7Mt ROM and a 10% discount rate. All other input parameters were as for the pit limit optimization modelling. The cash flow analysis results are set out in Table 15-21.

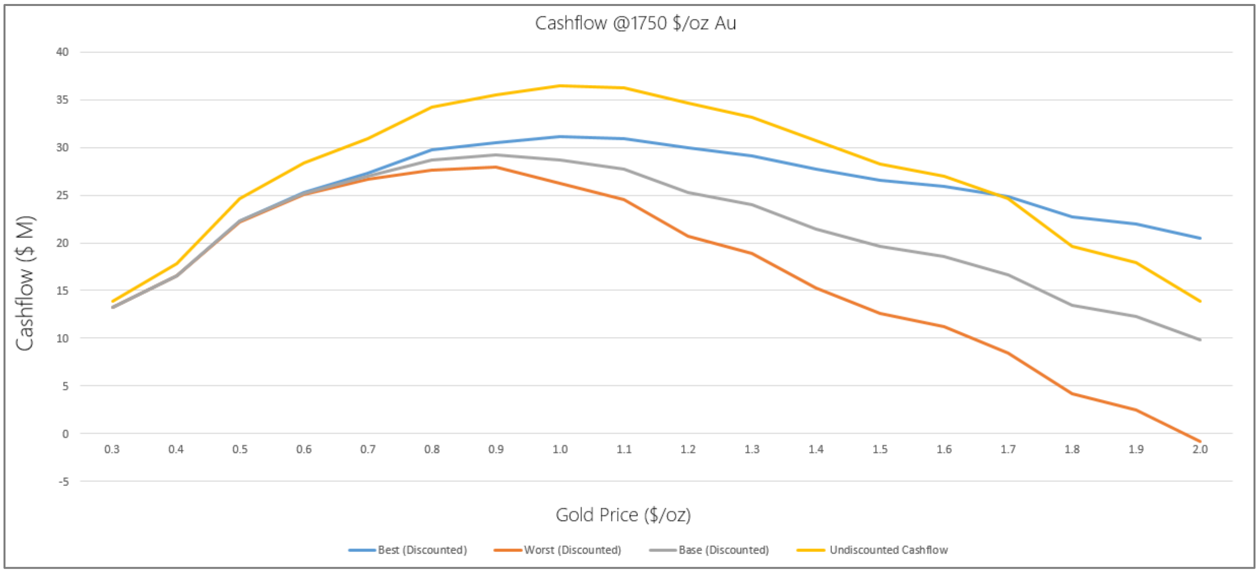


Figure 15-29: Whittle Cash Flow Analysis

Table 15-24: Base Case Optimization Cash Flow Results

Pit	RF	Best	Worst	Base	Undiscounted	Mineable Quantity	Strip Ratio	Au grade	Contained Au
		US$M	US$M	US$M	US$M	Mt	W:O	g/t	kg
1	0.3	13.23	13.23	13.23	13.88	0.15	2.17	2.86	424.8
2	0.4	16.54	16.54	16.54	17.85	0.25	2.48	2.44	604.8
3	0.5	22.27	22.23	22.25	24.62	0.41	4.07	2.21	906.5
4	0.6	25.29	25.02	25.16	28.33	0.56	4.87	2.03	1,131.1
5	0.7	27.29	26.67	26.98	30.98	0.75	5.09	1.81	1,365.7
6	0.8	29.73	27.61	28.67	34.25	1.10	5.57	1.59	1,751.6
7	0.9	30.53	27.94	29.24	35.50	1.48	5.31	1.40	2,067.6
8	**1.0**	**31.10**	**26.21**	**28.66**	**36.50**	**1.95**	**5.86**	**1.29**	**2,507.5**
9	1.1	30.91	24.50	27.71	36.22	2.35	6.05	1.20	2,834.1
10	1.2	29.94	20.65	25.30	34.66	2.84	6.22	1.12	3,190.2
11	1.3	29.09	18.88	23.99	33.17	3.20	6.13	1.07	3,410.7
12	1.4	27.77	15.21	21.49	30.69	3.62	6.07	1.01	3,652.1
13	1.5	26.57	12.60	19.59	28.29	3.98	6.09	0.97	3,850.2
14	1.6	25.95	11.20	18.58	27.00	4.20	6.01	0.94	3,955.0
15	1.7	24.87	8.42	16.65	24.64	4.51	5.95	0.91	4,095.4
16	1.8	22.70	4.20	13.45	19.60	5.03	6.05	0.86	4,340.6
17	1.9	22.00	2.50	12.25	17.88	5.38	5.81	0.83	4,461.1
18	2.0	20.44	-0.78	9.83	13.89	5.98	5.56	0.78	4,664.1

The results indicate that the Base Case pit shell, Pit 8, will not likely deliver the highest net present value (NPV). The pit shell which delivers the best NPV is Pit 7. However, the difference in cash flow between Pit shell 7 and Pit shell 8 is only US$0.58M while the variance in ore is 0.47 Mt.

15.3.3.3 Additional Analysis

The additional Whittle optimization cases undertaken using the same revenue factor. Sensitivities were conducted on the cost and slope variables of the pit optimization, varying input values by ±20%. The results for the options are compared to the Base Case pit shell in **Table 15-22.**

Table 15-25: Additional Analysis

Option	Description	Mineable Quantity	Waste	Total Mined	Strip Ratio	Au grade	Contained Au	Mine Life
		Mt	Mt	Mt	W:O	g/t	kg	Years
1	Base	1.95	11.43	13.38	5.86	1.29	2,507.5	1.11
2	Inferred included	1.95	11.46	13.42	5.87	1.29	2,512.3	1.12
3	Mining Cost+20%	1.62	7.87	9.48	4.86	1.33	2,145.2	0.92
4	Mining Cost-20%	2.29	15.83	18.12	6.92	1.25	2,866.2	1.31
5	Milling Cost+20%	1.38	8.64	10.02	6.25	1.48	2,052.3	0.79
6	Milling Cost-20%	2.54	13.98	16.52	5.51	1.14	2,903.8	1.45
7	Slope Angle +2deg	2.07	12.24	14.31	5.93	1.28	2,652.9	1.18
8	Slope Angle -2deg	1.96	12.76	14.73	6.51	1.28	2,514.8	1.12

The results indicate that further drilling to convert the Inferred ore to Indicated or Measured will only slightly increase the quantity of mineable quantity when compared to the base case pit shell (Pit 8). Also, the steepening of pit walls has a negligible impact on the potential mineable quantity, however it does slightly decrease waste rock mined, which is as expected given the style and geometry of the mineralization within the Project.

15.3.3.4 Selection of Preferred Pit Shell

The selection of the preferred pit shell was undertaken in conjunction with Boroo Gold. Key criteria used to select the pit shell included:

- High potential profitability;
- Maximise cash flow;
- Large in-pit mining inventory; and
- Potential to produce 60-65Koz gold per year;

Based on the outcomes of the pit limit optimisation and the cash flow analysis discussed above, the preferred pit shell selected is Pit 8, being the 100% revenue factor shell. This pit has higher NPV results and is estimated to have over 1.95 Mt of potential mineable quantity at 1.29 g/t Au and at a strip ratio of 5.86:1. At the preferred production rate of 1.7Mtpa the mine life would be over 1 year.

15.3.4 Deposit Characterization

Deposit characterization is the process of determining and highlighting the key physical characteristics of the deposit in order to assess and understand the implications for mining and the mining strategy. It is completed by analyzing numerical data in the form of either plots or charts within the final pit design.

The key results of the deposit characterisation include the following:

- Figure 15-30 plots both mineable quantity tonnes against cut-off grade. It is a commonly used chart which demonstrates the distribution of mineable quantities relative to cut-off grade;
- Figure 15-31 plots the mineable quantity by elevation above sea level within a ultimate pit design. It illustrates how deep the various quality materials are within a pit design; and
- Figure 15-32 plots the gold grade by elevation above sea level within a pit ultimate pit design.

Deposit characterisation indicates the following:

- Approximately 1.95 Mt in situ ore resource is greater than marginal cut-off grade;
- Overall, a consistent strip ratio;
- Grade substantially high at surface; and
- Grade is consistent with depth;

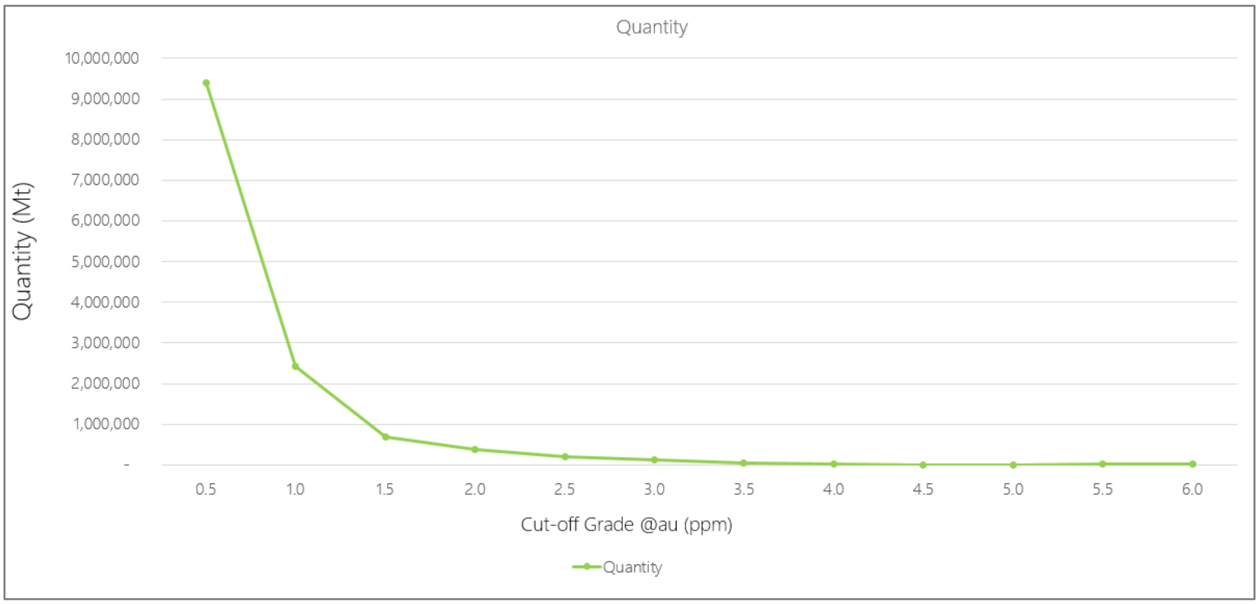


Figure 15-30: Grade-Tonnage Curve
(Measured and Indicated above Cut-Off Grade)

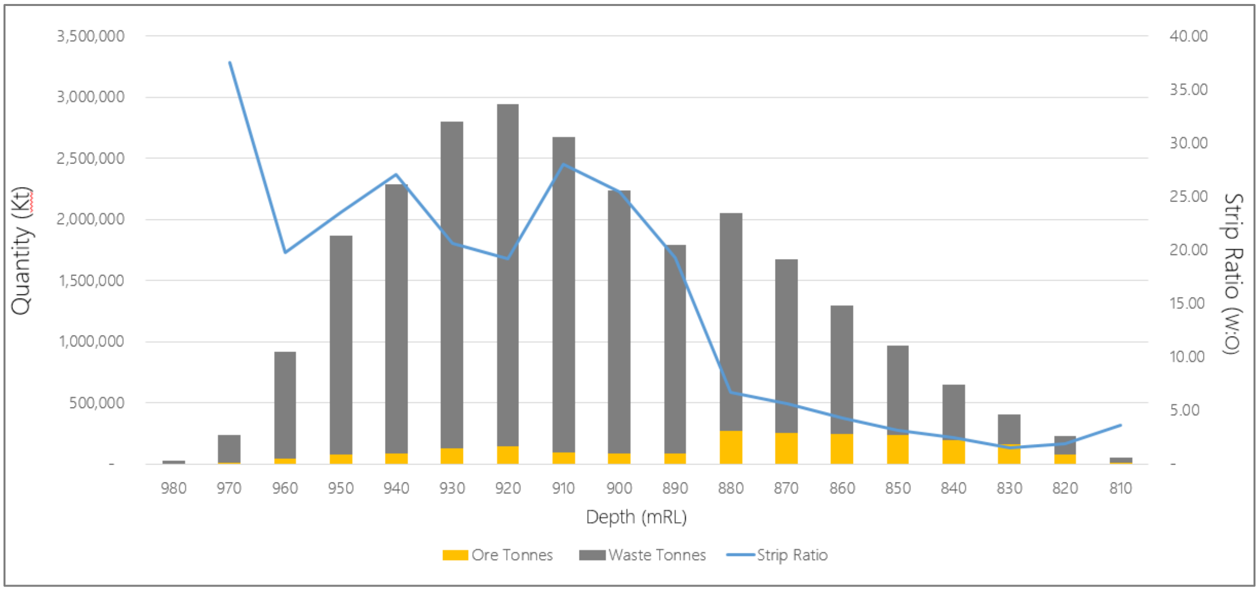


Figure 15-31: ROM Quantities by Depth

(Measured and Indicated Quantities above Cut-Off Grade)

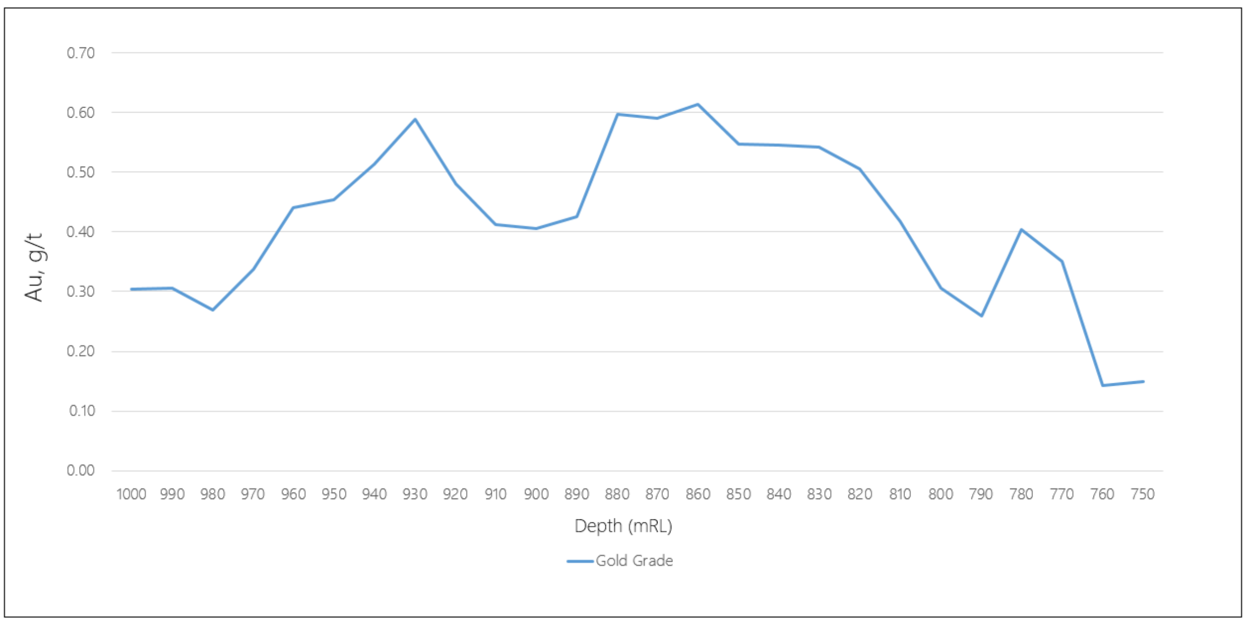


Figure15-32: Gold Grade by Depth
(Measured and Indicated Quantities above Cut-Off Grade)

15.3.5 Mine Phase Design

A systematic approach was undertaken to identify a preferred development strategy for the mine. The first stage involved visually examining the results from the pit limit optimisation process. The changes in the shape of the optimal pit shell from low to higher revenue factors reflect potential development strategies from high value to lower value mining areas. The characteristics of the final pit shell were also examined as the shape, depth and potential mineable quantities all influence the mining rate and development. The second stage involved examining the results of the deposit characterisation analysis.

The key strategic results from the pit optimisation and the deposit characterisation were:

- Upper benches in the ultimate pit have higher strip ratios;
- Strip ratio substantially improves with depth; and

From the above results, the preferred approach was to undertake a cutback development strategy. Pit analysis indicated two cutbacks to be preferable, which together with the ultimate pit design, results in three stages of mining. Three cutbacks were designed in Geovia Surpac™ based on a mining sequence from north to south. Cutbacks were designed in order to defer stripping as much as possible and to focus on targeting the high grade ore. In order to achieve this, constraints were set in Surpac™ during design and further to this during the scheduling process in MineSched™.

The incremental ore quantities and grades associated with the cutback designs are set out in Table 15-26.

Table 15-26: Mineable Quantities By Cutback

Phase	Total Material		CIP ORE			HEAP LEACH ORE			Waste Tonnage	Strip Ratio
	Volume	Tonnage	Tonnage	Au, g/t	Metal, kg	Tonnage	Au, g/t	Metal, kg		
UB_P01	375,000	957,000	374,000	1.79	700	306,000	0.35	100	278,000	0.74
UB_P02	6,139,000	15,643,000	1,240,000	1.19	1,500	2,620,000	0.27	700	11,784,000	9.50
UB_P03	2,179,000	5,437,000	440,000	1.26	500	349,000	0.34	100	4,647,000	10.55
Total	**8,693,000**	**22,037,000**	**2,054,000**	**1.31**	**2,700**	**3,275,000**	**0.29**	**900**	**16,709,000**	**8.14**

**All figures are rounded to reflect the relative accuracy of the estimate. Numbers may not add exactly due to rounding.*

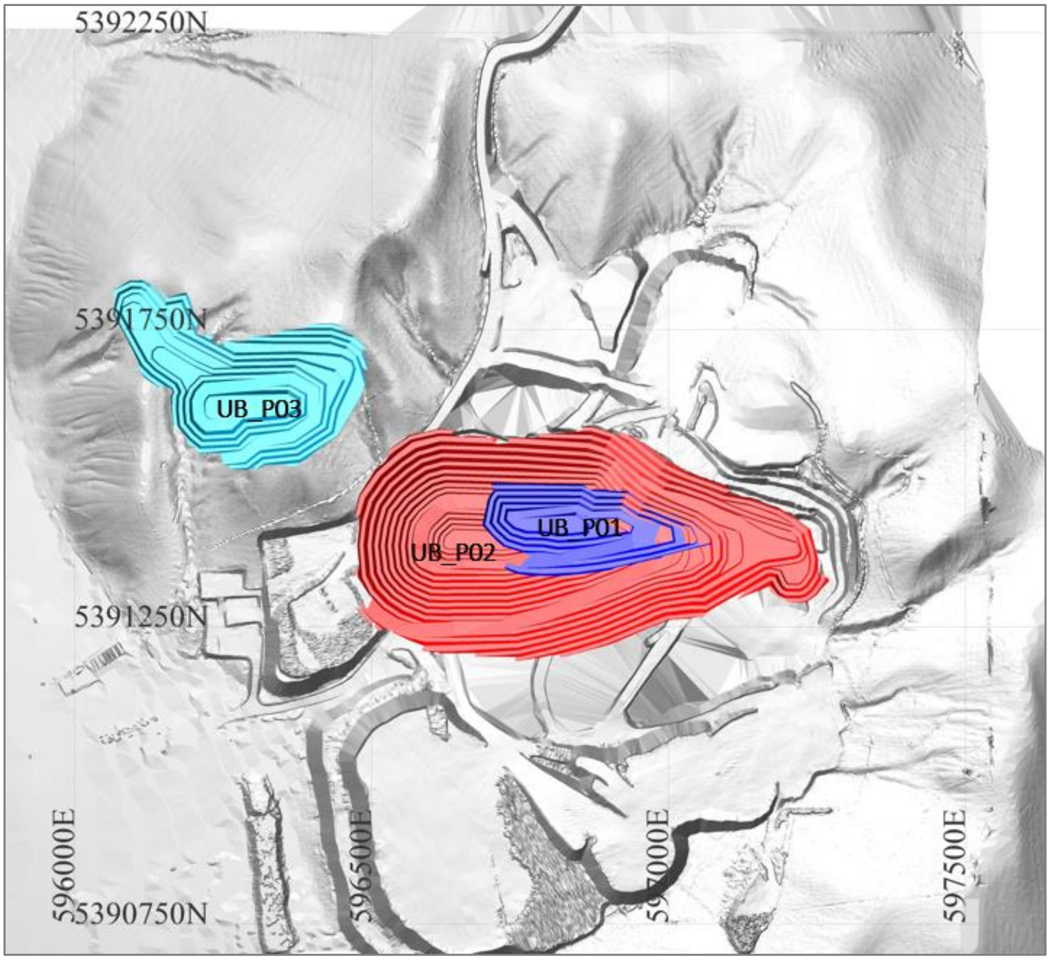


Figure 15-33: Cutback Designs for Ulaanbulag, Plan View

15.3.6 Ultimate Pit Design

The pit footprint is designed at approximately 850 m long by 400 m wide, with the long axis of the pit aligned in a roughly E-W direction. The pit design demonstrates the viability of accessing and mining the economic reserve at Ulaanbulag. Pit design was completed using Geovia Surpac™ software and was based on the Whittle™ shell RF 1.00 outlined in Section 15.3.3. In addition, the pit design complies with the geotechnical parameters previously stated in Section 15.3 and the following additional mining parameters:

- Minimum mining width of 30 m, chosen to accommodate the highest possible ore retrieval using the equipment selected.
- Ramp width of 25m at a gradient of 10% for crest to lowest two benches, with a ramp width of 15m and gradient of 10% for access to lower sections of the orebody in lowest two benches.
- Overall pit wall height no greater than 130 m in any area.

January 01 2024 pit survey and the ultimate pit design is shown in Figure 15-34, Figure 15-35 and Figure 15-36 below. The ultimate pit design improved on the RF 1.0 shell and was able to bring more practical into the final design.

Figure 15-34: January 01 2024 pit survey, Plan View

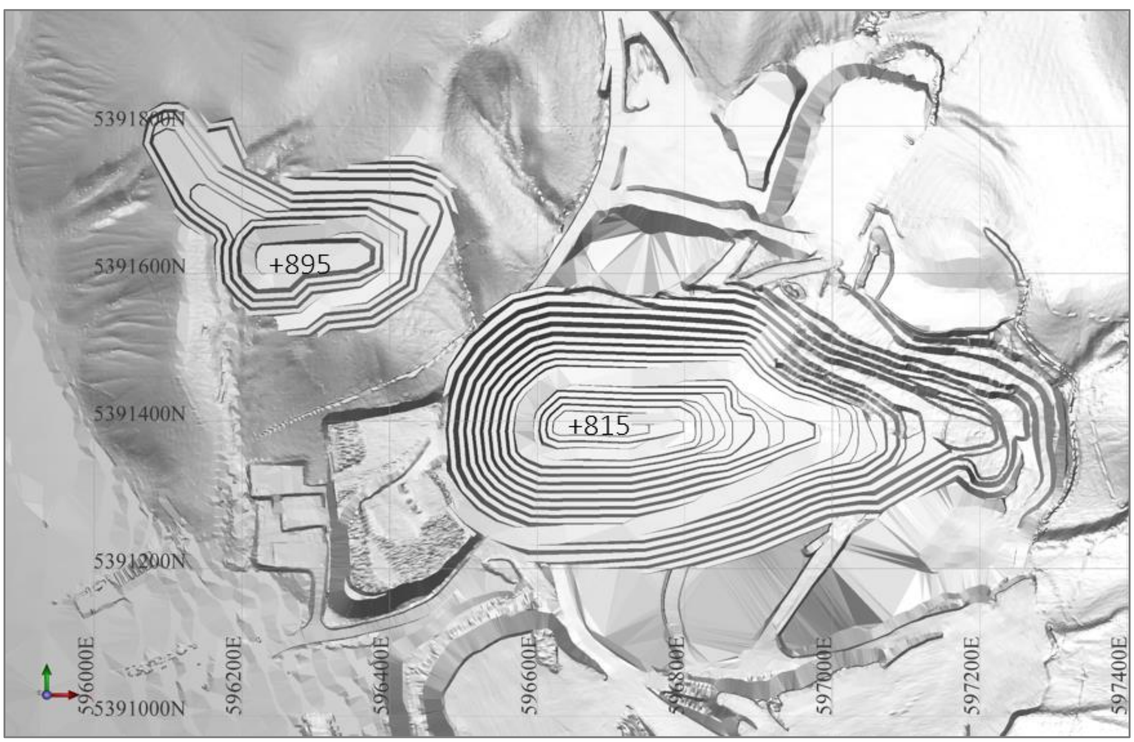


Figure 15-35: Ulaanbulag Ultimate Pit Design, Plan View

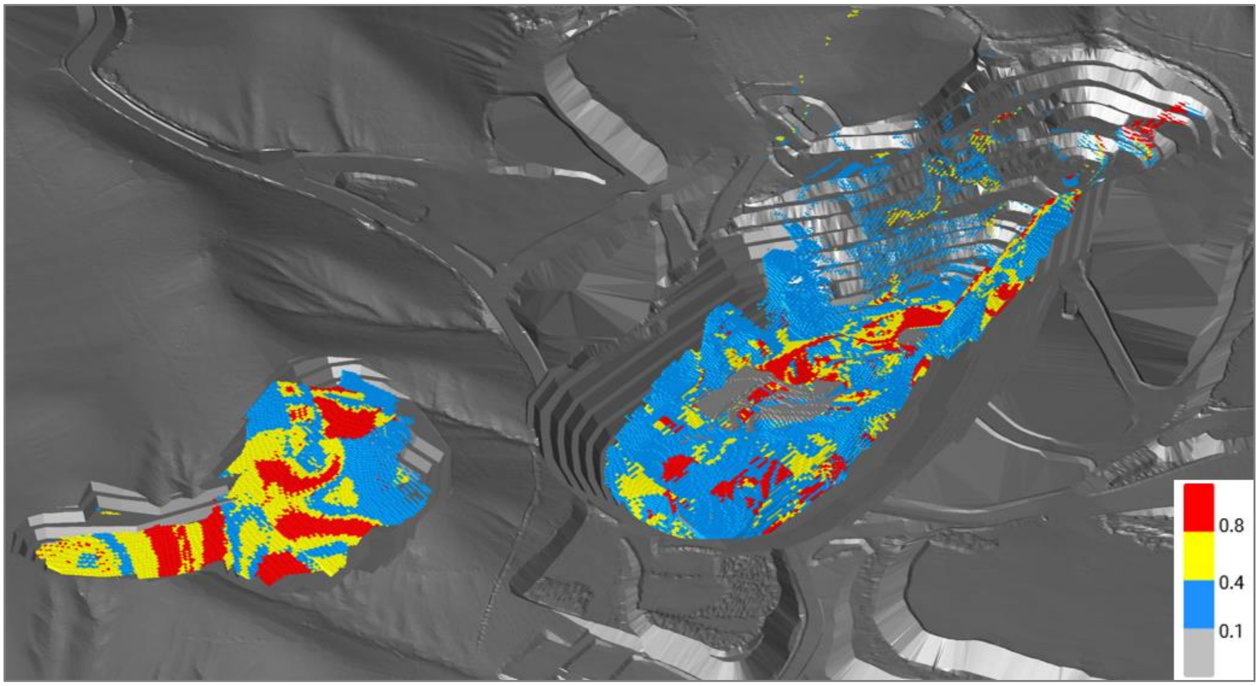


Figure 15-36: Ulaanbulag Ultimate Pit Design, Oblique View, Looking Southeast

15.3.6.1 Mine Haul Road Design

Haul roads in the Ulaanbulag pit are designed to provide both safe and efficient haulage routes from the base to the crest of the pit, and onwards to the processing plant and waste storage facility. All site haul roads outside of the Ulaanbulag pit are designed as two-lane roads. The use of one-lane roads is limited to the lowest two benches in the Ulaanbulag pit. Roads are designed with the following Mines Regulations specifications:

- For dual lane traffic, a minimum haul road width of no less than three times the width of the widest haulage vehicle used on the road.
- For single lane traffic, a minimum haul road width of no less than twice the width of the widest haulage vehicle used on the road.
- Provision for a safety berm with a height at least three-quarters the height of the largest tire on any vehicle operating on the road where a drop-off greater than 3 m exists.

Based on a 100 t CAT 777 model haul truck, the following parameters for haul roads have been set:

- Largest vehicle overall width: 7 m
- Double-lane haul road width: 25 m
- Single-lane haul road width: 15 m
- Roads are designed with allowances for ditches and culverts and will have a total thickness of 1 m, created with surfacing and base layers to improve durability.

15.3.6.2 Minimum Mining Width

A minimum mining width has been maintained between pit areas and at the deepest portions of the ultimate pit which is intended to allow efficient mining operations. For this study and the size of equipment chosen the minimum mining width required conforms to 30 m, which in turn maximizes the extraction of the available resources.

15.3.7 Mineral Reserve Statement

- Proven and Probable Reserves for the Ulaanbulag operation are inclusive of mineral resources and based on a five-year moving average gold price of $1,750/oz.
- Mining costs of $1.77/t, milling costs of $14.99/t and general and administrative costs of $2.22/t, Ore Transportation costs of $1.73/t and Heap leaching costs of 2.39$/t have been used to estimate the reserves along with the gold price stated above.
- The pit reserves reported below assume full mine recovery and are reported on a dry in-situ basis.
- In the block model, no additional provisions were introduced to account for external dilution or losses during mining, While these factors always occur to some degree during mining, it would appear from the reliability of the block model relative to the production results obtained to date that the required levels of adjustment for these factors has been adequately accounted for in the initial interpolation and later unsmoothing introduced during block model creation.
- The cut-off grade used to report the reserves has been chosen by Game Mine at greater than 0.20 g/t gold for heap leach ore and greater than 0.46, 0.50 and 0.53 g/t gold for milling depends on oxidation.
- Reserve estimates were completed using a pit-constrained resource, with an economic pit shell designed in Geovia Surpac™ and January 01 2024 pit survey.
- Mineral Reserves were estimates has been compiled under the supervision of QP Tuvshinbayar Batbayar, in accordance to National Instrument 43-101 standards and with reference to the 2014 Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards (2014 CIM Definition Standards) and the 2019 CIM Best Practice Guidelines.

Table 15-27: Mineral Reserve Statement for Ulaanbulag Deposit, Mongolia: Game Mine LLC., January 01st, 2024

Reserve Category	Quantity (tonnes)	Average Grade (Au g/t)	Contained Metal (oz)
CIP Ore Stockpile			
Proven	118,000	0.96	4,000
Probable	-	-	-
Proven and Probable	118,000	0.96	4,000
CIP Ore			
Proven	1,196,000	1.4	54,000
Probable	858,000	1.19	33,000
Proven and Probable	2,054,000	1.31	87,000
Total CIP Ore			
Proven	**1,314,000**	**1.36**	**57,000**
Probable	**858,000**	**1.2**	**33,000**
Proven and Probable	**2,172,000**	**1.3**	**90,000**
Heap Leach Ore Stockpile			
Proven	727,000	0.4	9,000
Probable	-	-	-
Proven and Probable	727,000	0.4	9,000
Heap Leach Ore			
Proven	1,496,000	0.29	14,000
Probable	1,778,000	0.29	16,000
Proven and Probable	3,274,000	0.29	30,000
Total Heap Leach Ore			
Proven	**2,223,000**	**0.33**	**23,000**
Probable	**1,778,000**	**0.28**	**16,000**
Proven and Probable	**4,001,000**	**0.31**	**40,000**
Total Reserve			
Proven	**3,537,000**	**0.71**	**81,000**
Probable	**2,636,000**	**0.58**	**49,000**
Proven and Probable	**6,173,000**	**0.66**	**130,000**

Notes:

1. The effective date of the Mineral Reserve estimate is February 01st, 2024. Mineral Reserves were estimates has been compiled under the supervision of QP Tuvshinbayar Batbayar.

2. The Mineral Reserve estimates were prepared with reference to the 2014 Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards (2014 CIM Definition Standards) and the 2019 CIM Best Practice Guidelines.

3. Reserves estimated assuming open pit mining methods

4. Reserves are reported on a dry in-situ basis

5. The cut-off grade used to report the reserves has been chosen by Game Mine at greater than 0.20 g/t gold for heap leach ore and greater than 0.46, 0.50 and 0.53 g/t gold for milling depends on oxidation.
6. Reserves are based on a gold price of $1,750/oz, mining cost of $1.77/tonne, milling costs of $14.99/t and general and administrative costs of $2.22/t. Ore transportation costs of 1.73$/t. Heap leaching costs of 2.39$/t. Heap leaching recovery 40%.
7. In the block model, no additional provisions were introduced to account for external dilution or losses during mining, While these factors always occur to some degree during mining, it would appear from the reliability of the block model relative to the production results obtained to date that the required levels of adjustment for these factors has been adequately accounted for in the initial interpolation and later unsmoothing introduced during block model creation.
8. All figures are rounded to reflect the relative accuracy of the estimate. Numbers may not add exactly due to rounding.

15.3.8 Relevant Factors

Game Mine is not aware of any existing environmental, permitting, legal, socio-economic, marketing or political factors that are likely to materially affect the mineral reserve estimate. Mineral reserves have been economically tested to ensure that they are economically viable. The project remains economic across a range of key input parameters.

If for any reason any of these project cost factors are changed such that the project capital or operating cost estimates change materially, then the mineral reserve estimates stated in this report could be materially affected.

A full sensitivity analysis of the operating and financial inputs affecting the Ulaanbulag is presented in Chapter 22.

16.0 MINING METHOD

16.1 Overview

16.1.1 Introduction

The Boroo site is comprised of the open pit mine, mill and processing facility and tailings storage facility. Additional infrastructure includes provision for offices, crib and ablutions, change and dry facilities, a heavy and light vehicle maintenance workshop, a warehouse, refuel facilities, an accommodation village, an explosive storage area, access and haul roads and all utilities as described in Chapter 18.

A production schedule based on an annualized 5,000 t/d mill feed rate has been developed for a conventional open pit mine plan for Boroo and Ulaanbulag. The mine design and schedule are based on the resources and reserves defined in Chapter 14 and Chapter 15. A life-of-mine (LOM) schedule is provided in Section 16.2.3.

Information considered and used for mine planning and design include base economic parameters, metal prices, mining cost, processing cost, throughput rate data derived from Boroo Gold and projected metallurgical recoveries.

16.1.2 Current Operations

The Boroo and Ulaanbulag mining operations are based on conventional open-pit methods to mine a nominal 50,000 tonnes per day material. Operations are planned to stop during the first half of 2030. Mining is done with bench heights of five metres, with ore mined on half benches for improved grade control in the flat-lying ore. Figure 16-1 and Figure 16-2 is a panoramic view of mining operations.

Blasthole drilling is carried out with five drill rigs, with holes of 115 mm diameter drilled on a 4.0 x 4.0-metre pattern. Bulk explosives trucks blend ammonium nitrate with fuel oil as each hole is loaded, and explosives consumption is 0.4 kg per cubic metre of ore or waste.

The principal rock handling equipment is supplied by Caterpillar and includes two 5.4 m3 hydraulic excavators, one 6.4 m3 hydraulic excavator and one 12 m3 hydraulic excavator and ten 50-tonne haul trucks, six 100-tonne haul trucks. Additional haul trucks are to be temporarily added to the fleet in for tailings dam construction. The waste rock mined is deposited on waste dumps immediately adjacent to the individual pits.

Additional mining equipment includes five large front-end loaders for ore handling and blending, four tracked dozers for the maintenance of waste dumps and benches, and three graders maintain the roads and bench floors.

Figure 16-1: Panoramic view of Boroo mining operations.

Figure 16-2: Panoramic view of Ulaanbulag mining operations.

Grade control in mining is by manual sampling of the blasthole cuttings. Two separate samples are taken for each blasthole, one for the initial 2.5 metres, one for the second 2.5 metres, which allows ore and waste selection to occur for those short vertical intervals.

Sorting of ore in the pit is done as per the following grade ranges:

Designation	Boroo Gold Grade (g/t)	Ulaanbulag Gold Grade (g/t)	Destination
Ore	> Mill cut-off grade	> Mill cut-off grade	Crusher or temporary stockpiles for blending
Heap Leach	>0.1 and < Mill cut-off grade	>0.2 and < Mill cut-off grade	Stockpile, scheduled for heap leaching, as waste for the calculation of the strip ratio.
Waste	<0.1	<0.2	Waste dumps adjacent to pits

The grade control cut-off grades currently used in mining are slightly different than those used for the Game Mine mineral reserve estimate (Section 15.2.7 and Section 15.3.7).

The blasthole assay data are determined at a SGS laboratory in Ulaanbaatar. A large proportion of each sample needs to be pulverized, and the metallics screen method is required to obtain results that are both accurate and precise (reproducible) and allow grade control decisions to be made based on few samples. The blasthole assay data are combined into an ore control model that is being used to determine the boundaries for the various grade categories and to estimate the monthly pit production.

The blasthole cuttings are logged, and a geological map is produced for each bench mined. The grade and geological maps produced as a result of the grade control work are assembled into packages that can be readily used in mining operations and are of excellent quality. Details of the mining equipment can be found in Section 16.4.

16.2 Open Pit Operations

16.2.1 Open Pit Scheduling Methodology

To generate the LOM schedule for the Boroo and Ulaanbulag pit, Geovia Whittle™ was used to develop a strategic mine schedule focused on maximizing NPV. The LOM schedule was then optimized in Geovia MineSched™ to produce a practical mine schedule, that met all scheduling constraints. As discussed in Chapter 15, waste to ore cut-offs were determined for each block in the model. Blocks were categorized as: Ore, Heap Leach. Ore blocks have greater than mill cut-off grade. Heap Leach blocks have less than milling cut-off grade but greater than 0.1 g/t Au (0.2 g/t for Ulaanbulag). By splitting the economic grade material into High, Medium and Low Grade Ore categories allowed higher grade material to be targeted in the production schedule. Only Measured and Indicated blocks could be considered Ore, and Heap Leach. Ore blocks are sent to the processing plant while Heap Leach are sent to the leach pad.

16.2.2 Mine Schedule Strategy

The objective to the Boroo and Ulaanbulag LOM schedule was to maximize the early cash flow from the pit by targeting high grade material to be fed to the processing plant while delaying costs by deferring waste stripping to later in the project life. The optimum throughput to the processing plant was determined to be 1.7 Mt/year. The primary objective of the schedule was ensuring continuous ore supply to the processing plant by delivering the highest grade ore first and meeting milling capacity constraints. Other considerations for the production schedule included:

- A maximum vertical advancement of 60 m/year; Game Mine considers that this is a reasonable vertical advancement rate and can be met by the loading equipment selected.
- Balancing the production schedule so that the annual material moved did not fluctuate greatly year to year to allow fleet depreciation. An average of 18.7 Mt of material per annum was selected as the total material movement as this tonnage met the mill process constraint and honored the 60 m/year vertical advancement constraint.
- Use of stockpiling to maximize the grade of material to the processing plant.

The initial schedule produced in Whittle™. Although this schedule had the highest NPV in Whittle™, visual inspection of the schedule revealed that the schedule was not meeting the 60 m of vertical advancement and was pre-stripping. Additional constraints were set in MineSched™ to ensure that 60 m/year of vertical advancement was met to allow for the high-grade material to be mined as early as possible, as well as delaying the stripping to later in the project life. The additional constraints were able to optimize the schedule so that the final selected schedule maximized the NPV of the project within the constraints of the equipment and resources available. Figure 16-3 shows the optimized phase sequence that was selected as it delivered the most ounces to the processing plant in the first few years of operation, while honoring process plant feed, vertical advancement and total material movement constraints.

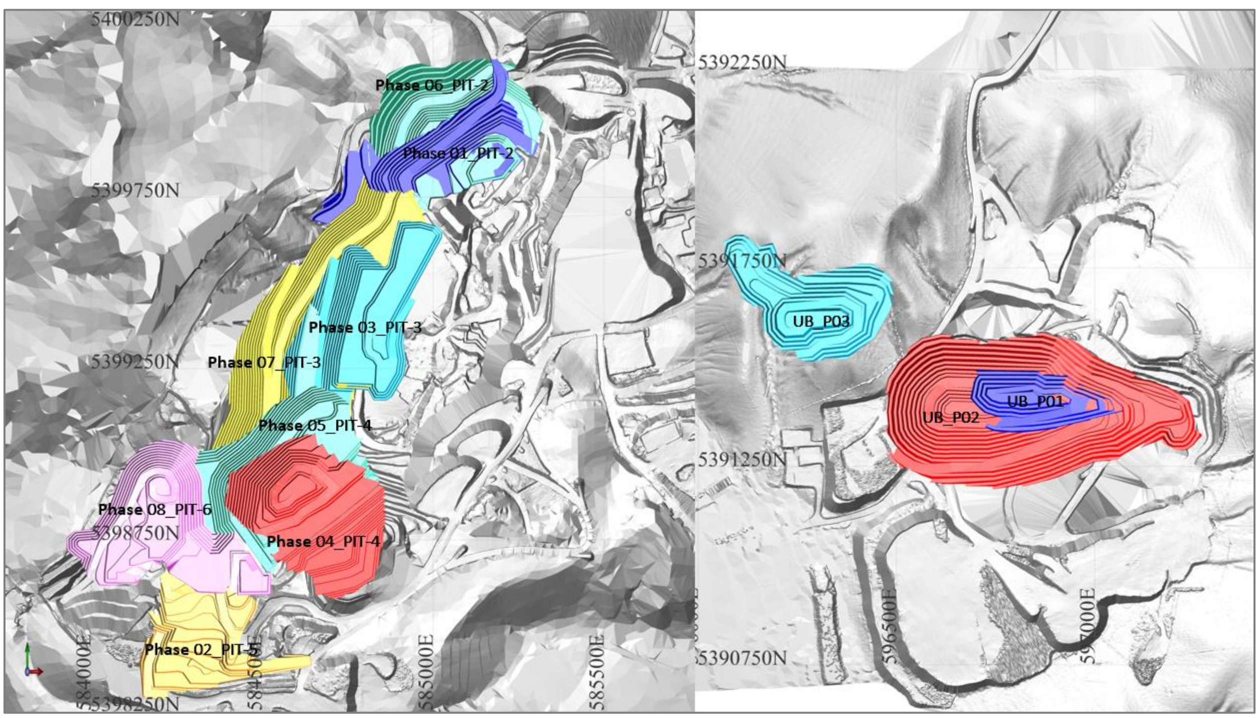


Figure 16-3: Optimized Phase Sequence

Table 16-1: Annual Tonnes Mined over the Boroo and Ulaanbulag Life of Mine by Phase

Year	2024	2025	2026	2027	2028	2029	2030
P01_Pit-2	845,000						
P02_Pit-5	1,110,000						
P03_Pit-3	13,581,000						
P04_Pit-4	1,583,000	17,217,000					
P05_Pit-4		1,561,000	13,874,000				
P06_Pit-2				2,074,000	8,836,000		
P07_Pit-3					9,464,000	18,250,000	3,151,000
P08_Pit-6							8,117,000
UB_P01	957,000						
UB_P02			4,904,000	10,739,000			
UB_P03				5,437,000			

**All figures are rounded to reflect the relative accuracy of the estimate. Numbers may not add exactly due to rounding.*

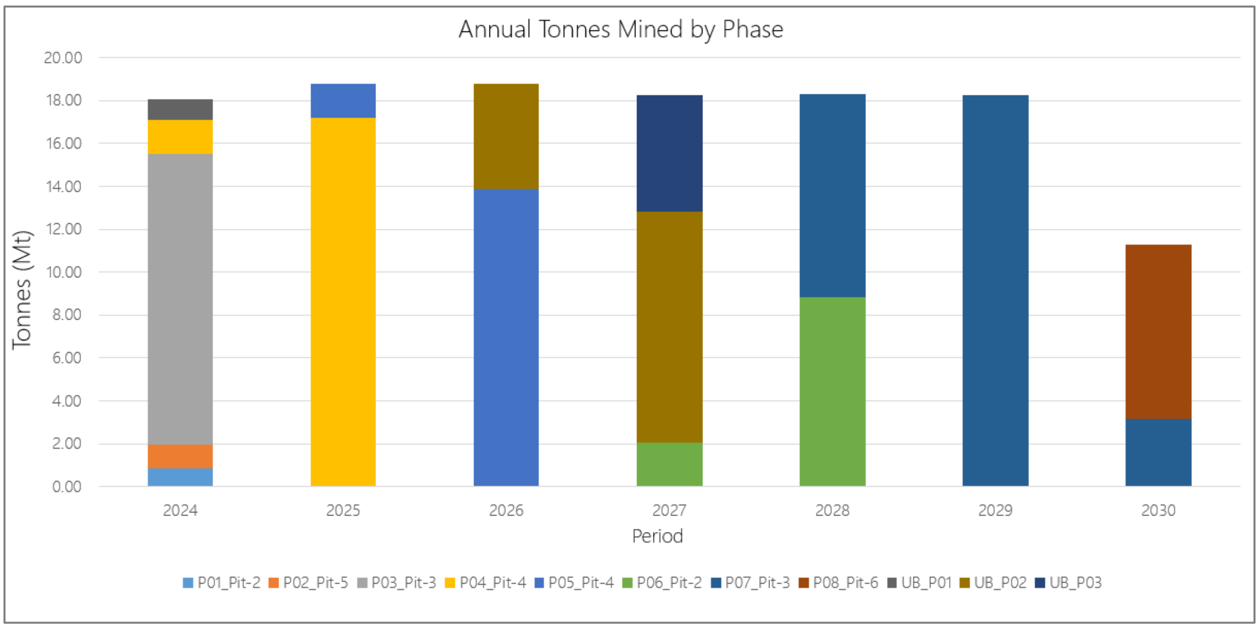


Figure 16-4: Annual Tonnes Mined over the Boroo and Ulaanbulag Life of Mine by Phase

16.2.3 Open Pit Operating Schedule

The Boroo and Ulaanbulag pits will be mined in 5 m benches with a maximum vertical advance of 60 m per year. There is no scheduled pre-stripping period, although some stripping is delayed as much as possible. The mining plan includes stockpiling on an annual basis to allow the highest-grade material to be fed to the processing plant. Mining will take place 365 days a year and no allowance has been made for climate change, labor action, or any other unscheduled shutdowns. An overall ex-pit material movement constraint of approximately 18-18.7 Mt per year was applied.

Table 16-2: Boroo and Ulaanbulag Mine Production Schedule

Year	Mined CIP Ore (tonnes)	Gold Grade (g/t)	Contained Gold (kg)	Mined Heap Leach Ore (tonnes)	Gold Grade (g/t)	Contained Gold (kg)	Total Contained Gold (kg)	Waste (tonnes)	Total Material (tonnes)	Strip Ratio, t/t*
2024	3,329,000	1.06	3,500	3,503,000	0.32	1,100	4,600	11,245,000	18,077,000	1.65
2025	1,397,000	1.60	2,200	1,133,000	0.29	300	2,500	16,247,000	18,777,000	6.42
2026	1,867,000	1.50	2,800	1,209,000	0.28	400	3,200	15,702,000	18,778,000	5.11
2027	1,662,000	1.21	2,000	2,796,000	0.29	800	2,800	13,792,000	18,250,000	3.09
2028	830,000	1.15	1000	1,790,000	0.28	500	1,500	15,680,000	18,300,000	5.99
2029	2,220,000	1.15	2,600	2,690,000	0.30	800	3,400	13,340,000	18,250,000	2.72
2030	1,625,000	0.90	1,500	2,575,000	0.30	800	2,300	7,067,000	11,267,000	1.68
Total	**12,930,000**	**1.20**	**15,600**	**15,696,000**	**0.30**	**4,700**	**20,236**	**93,073,000**	**121,699,000**	**3.25**

* Notes; heap leach ore included

***All figures are rounded to reflect the relative accuracy of the estimate. Numbers may not add exactly due to rounding.*

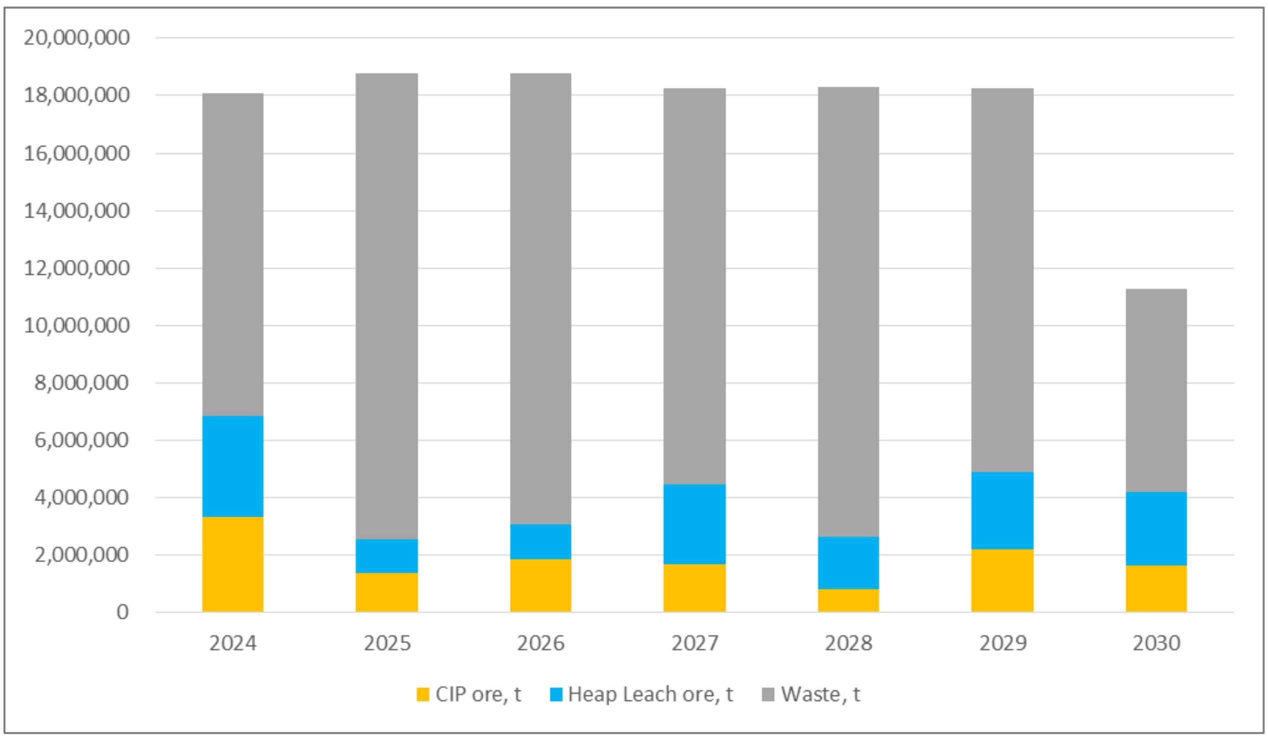


Figure 16-5: Annual Tonnes Mined over the Boroo and Ulaanbulag Life of Mine

Table 16-3: Mill and Heap Leach Production Schedule

Year	CIP Ore (tonnes)	Gold Grade (g/t)	Contained Gold (kg)	Recovered Gold (oz)	Heap Leach Ore (tonnes)	Gold Grade (g/t)	Contained Gold (kg)	Recovered Gold (oz)	Total Recovered Gold (oz)
2024	1,734,000	1.61	2,800	64,000	967,000	0.40	400	5,000	69,000
2025	1,734,000	1.34	2,300	53,000	2,822,000	0.34	1000	12,000	65,000
2026	1,734,000	1.74	3,000	61,000	1,133,000	0.29	300	4,000	65,000
2027	1,734,000	1.23	2,100	47,000	2,209,000	0.25	600	7,000	54,000
2028	1,738,000	0.89	1,600	36,000	2,801,000	0.30	900	11,000	47,000
2029	1,734,000	1.13	2,000	43,000	2,307,000	0.27	600	8,000	51,000
2030	1,734,000	1.06	1,800	40,000	2,533,000	0.29	700	9,000	49,000
2031	1,674,000	0.61	1,000	23,000	1,932,000	0.32	600	8,000	31,000
Total	13,816,000	1.20	16,600	367,000	16,704,000	0.30	5,100	64,000	431,000

**All figures are rounded to reflect the relative accuracy of the estimate. Numbers may not add exactly due to rounding.*

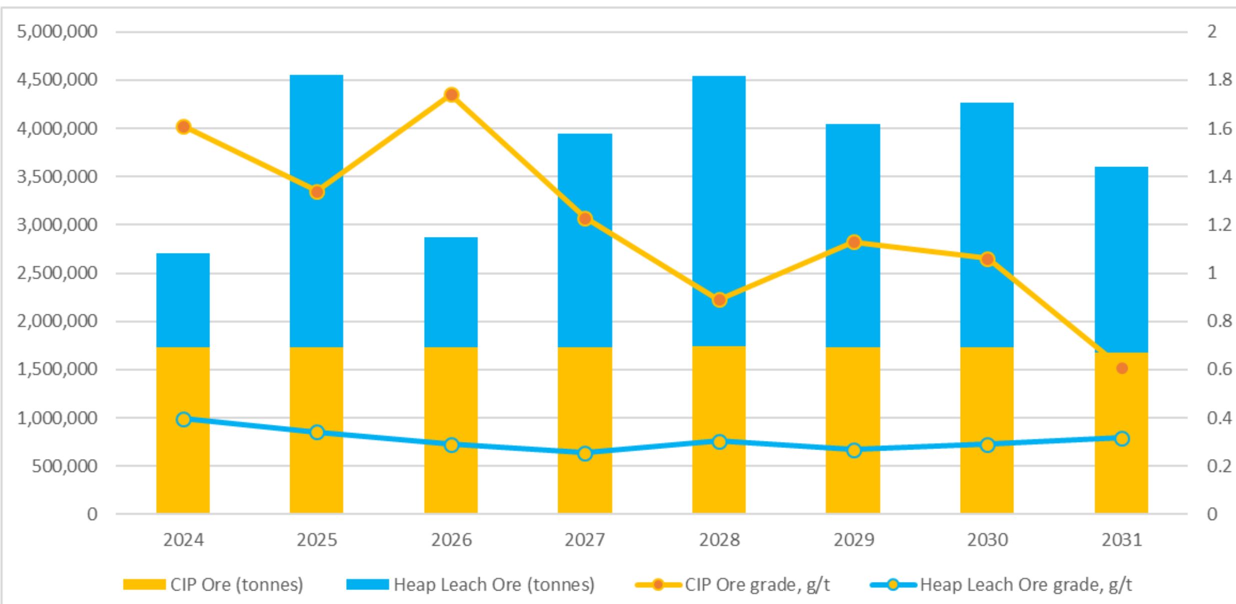


Figure 16-6: Annual Ore Feed to Plant over the Life of Mine

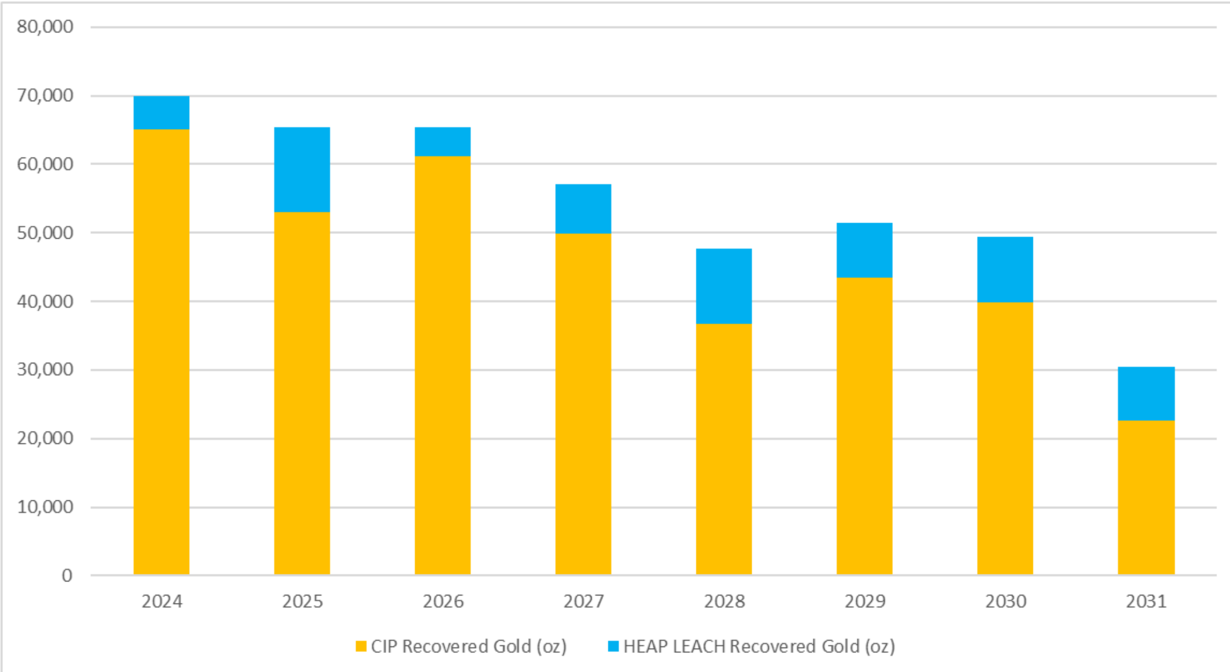


Figure 16-7: Annual Recovered Gold over the Life of Mine

Table 16-4: Mill Production Schedule By Ore Source

Period Number	Total Ore Feed to Plant, t	Average Gold Grade, g/t	Contained Gold (kg)	Recovered Gold (oz)	Boroo Ore Feed to Plant, t	Average Gold Grade, g/t	Contained Gold (kg)	Recovered Gold (oz)	Boroo Recovery	Ulaanbulag Ore Feed to Plant, t	Average Gold Grade, g/t	Contained Gold (kg)	Recovered Gold (oz)	Ulaanbulag Recovery
2024	1,734,000	1.61	2,800	64,000	1,353,000	1.50	2,000	47,000	0.72	381,000	2.00	800	17,000	0.69
2025	1,734,000	1.34	2,300	53,000	1,692,000	1.36	2,300	52,000	0.71	42,000	0.71	0	1,000	0.69
2026	1,734,000	1.74	3,000	61,000	1,593,000	1.82	2,900	59,000	0.63	141,000	0.75	100	2,000	0.71
2027	1,734,000	1.23	2,100	47,000	375,000	0.96	400	7,000	0.62	1,359,000	1.30	1,700	40,000	0.69
2028	1,738,000	0.89	1,600	36,000	1,489,000	0.94	1,400	33,000	0.74	249,000	0.60	200	3,000	0.69
2029	1,734,000	1.13	2,000	43,000	1,734,000	1.13	2,000	43,000	0.69	-	-	0	0	-
2030	1,734,000	1.06	1,800	40,000	1,734,000	1.06	1,800	40,000	0.67	-	-	0	0	-
2031	1,674,000	0.61	1,000	23,000	1,674,000	0.61	1,000	23,000	0.69	-	-	0	0	-
Total	13,816,000	1.20	16,600	367,000	11,644,000	1.19	13,800	304,000	0.68	2,172,000	1.30	2,800	63,000	0.69

**All figures are rounded to reflect the relative accuracy of the estimate. Numbers may not add exactly due to rounding.*

Table 16-5: Heap leach Production Schedule By Ore Source

Period Number	Total Heap Leach Ore (tonnes)	Gold Grade (g/t)	Total Contained Gold (kg)	Total Recovered Gold (oz)	UB Heap Leach Ore (tonnes)	Gold Grade (g/t)	Contained Gold (kg)	Recovered Gold (oz)	Boroo Heap Leach Ore (tonnes)	Gold Grade (g/t)	Contained Gold (kg)	Recovered Gold (oz)
2024	967,000	0.40	400	5,000	355,000	0.44	200	2,000	612,000	0.37	200	3,000
2025	2,822,000	0.34	1000	12,000	677,000	0.36	200	3,000	2,145,000	0.34	800	9,000
2026	1,133,000	0.29	300	4,000			-	0	1,133,000	0.29	300	4,000
2027	2,209,000	0.25	600	7,000	583,000	0.24	200	2,000	1,626,000	0.26	400	5,000
2028	2,801,000	0.30	900	11,000	1,012,000	0.34	300	4,000	1,789,000	0.28	600	7,000
2029	2,307,000	0.27	600	8,000	1,374,000	0.25	300	4,000	933,000	0.29	300	4,000
2030	2,533,000	0.29	700	9,000					2,533,000	0.29	700	9,000
2031	1,932,000	0.32	600	8,000					1,932,000	0.32	600	8,000
Total	16,704,000	0.30	5,100	64,000	4,001,000	0.31	1,200	15,000	12,703,000	0.3	3,900	49,000

**All figures are rounded to reflect the relative accuracy of the estimate. Numbers may not add exactly due to rounding.*

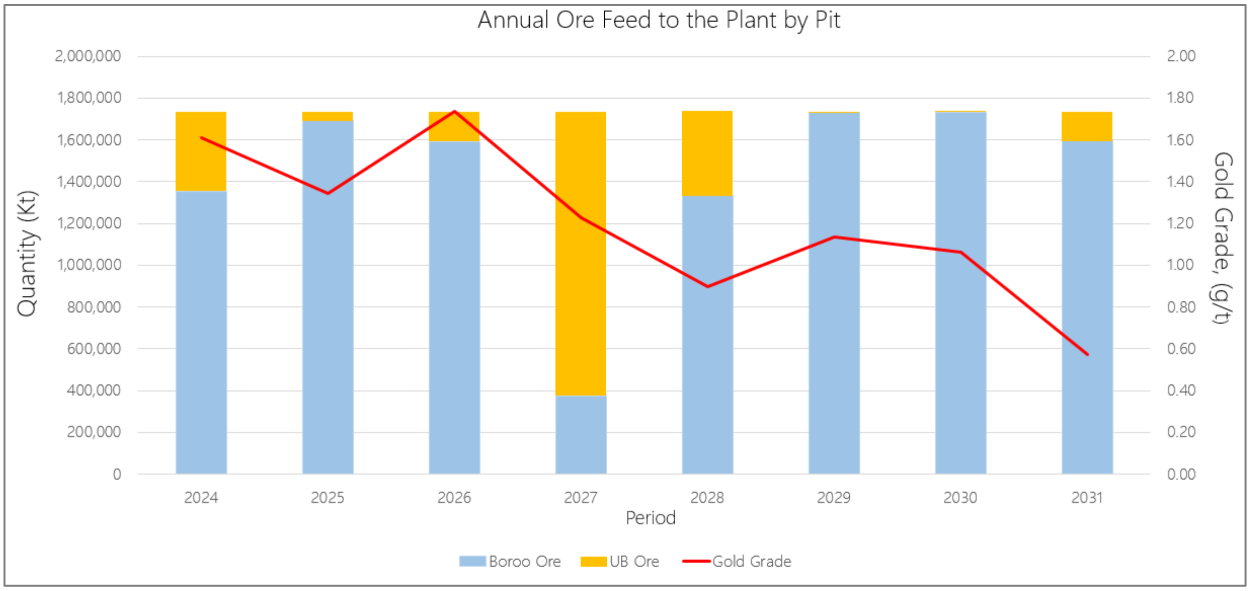


Figure 16-8: Annual Ore Feed to Mill over the Life of Mine

Table 16-6: Ulaanbulag Ore Transportation

Year	CIP Ore, t	Average Grade, g/t	Heap Leach Ore, t	Average Grade, g/t
2024	492,000	1.59	1,033,000	0.39
2025	-	-	-	-
2026	19,000	1.26	145,000	0.22
2027	1,280,000	1.34	909,000	0.29
2028	381,000	0.77	1,815,000	0.28
2029	-	-	99,000	0.22
Total	**2,172,000**	**1.30**	**4,001,000**	**0.31**

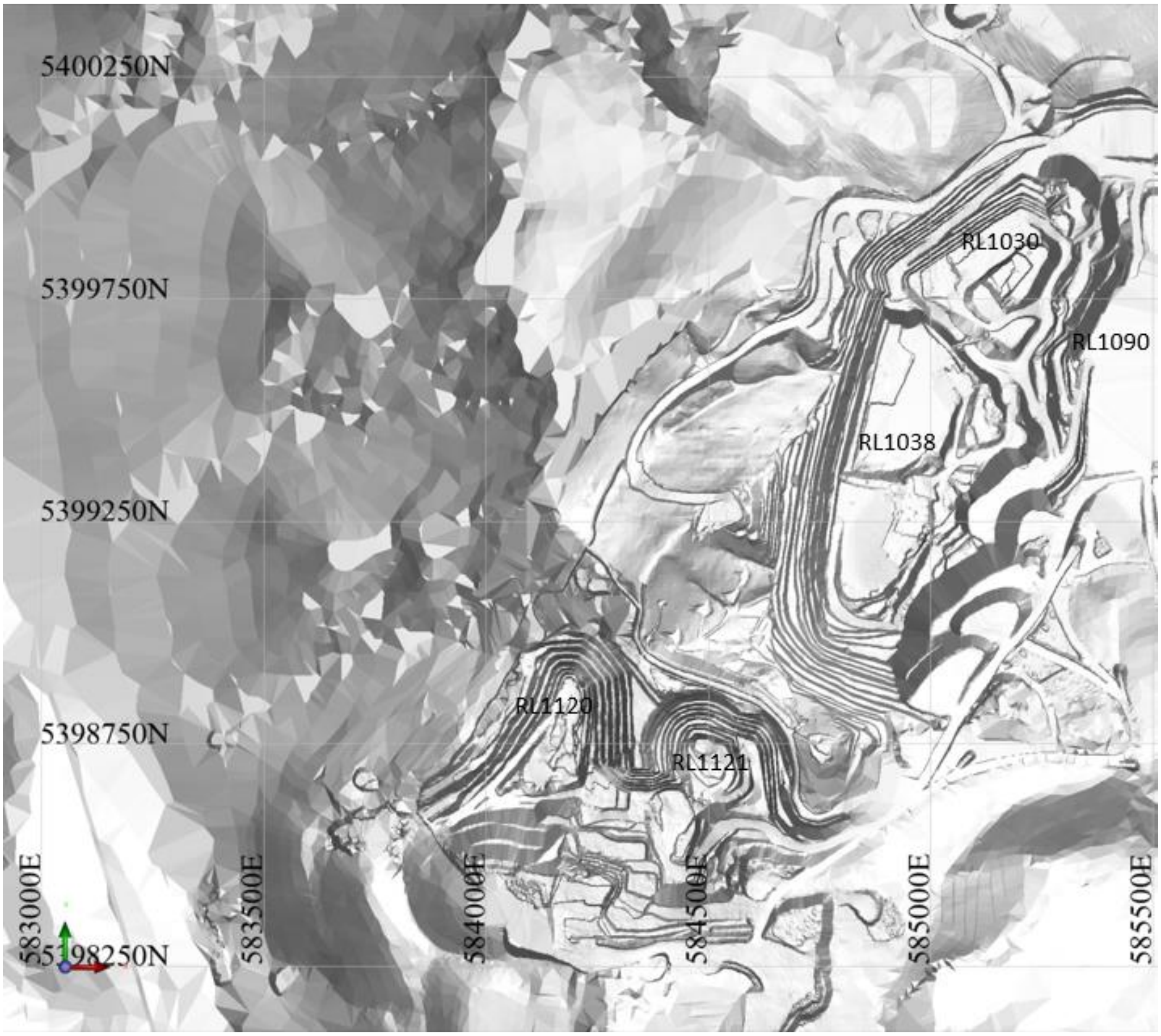


Figure 16-9: January 01 2024 pit survey

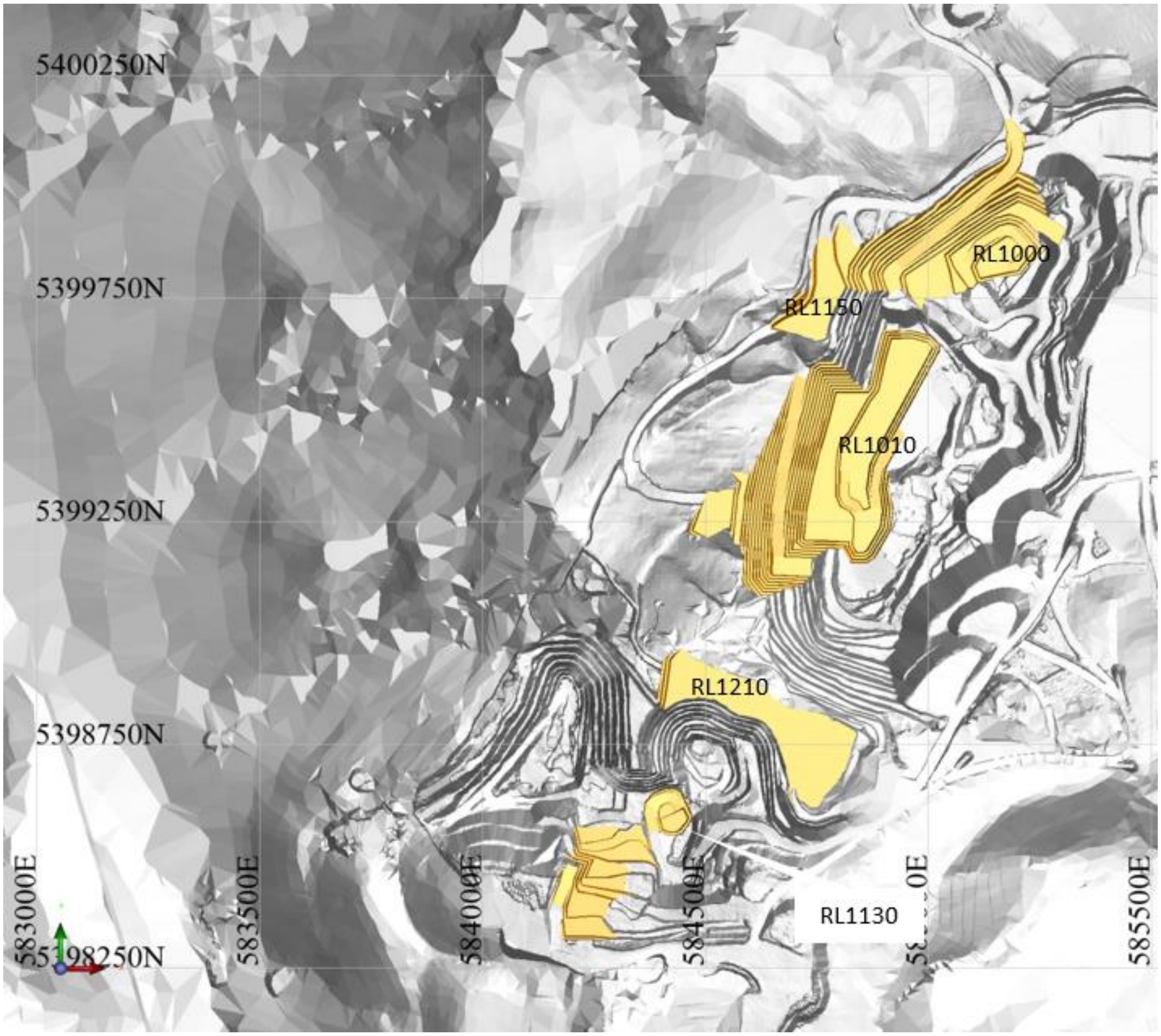


Figure 16-10: 2024 End of Pit Surface

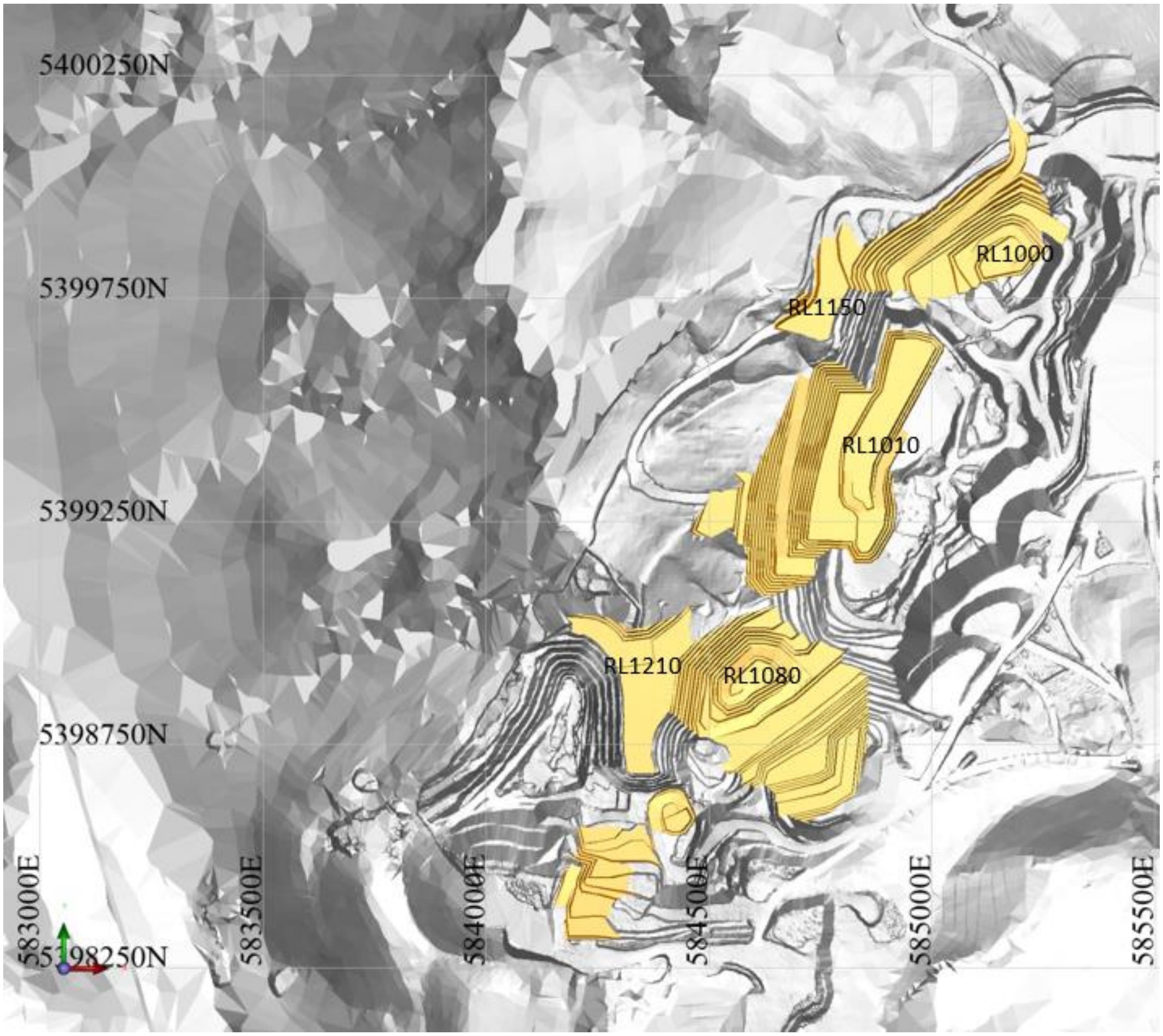


Figure 16-11: 2025 End of Pit Surface

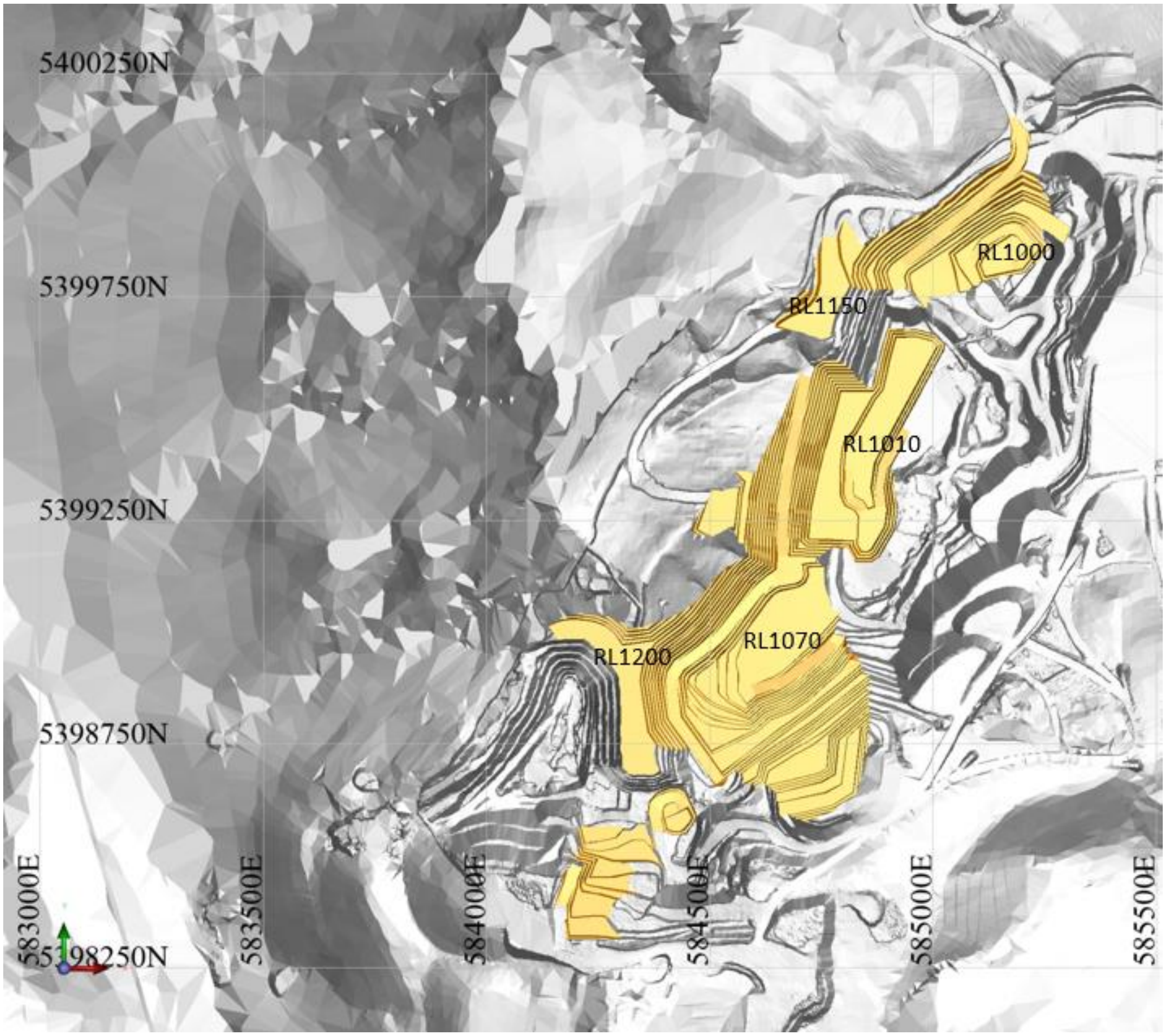


Figure 16-12: 2026 End of Pit Surface

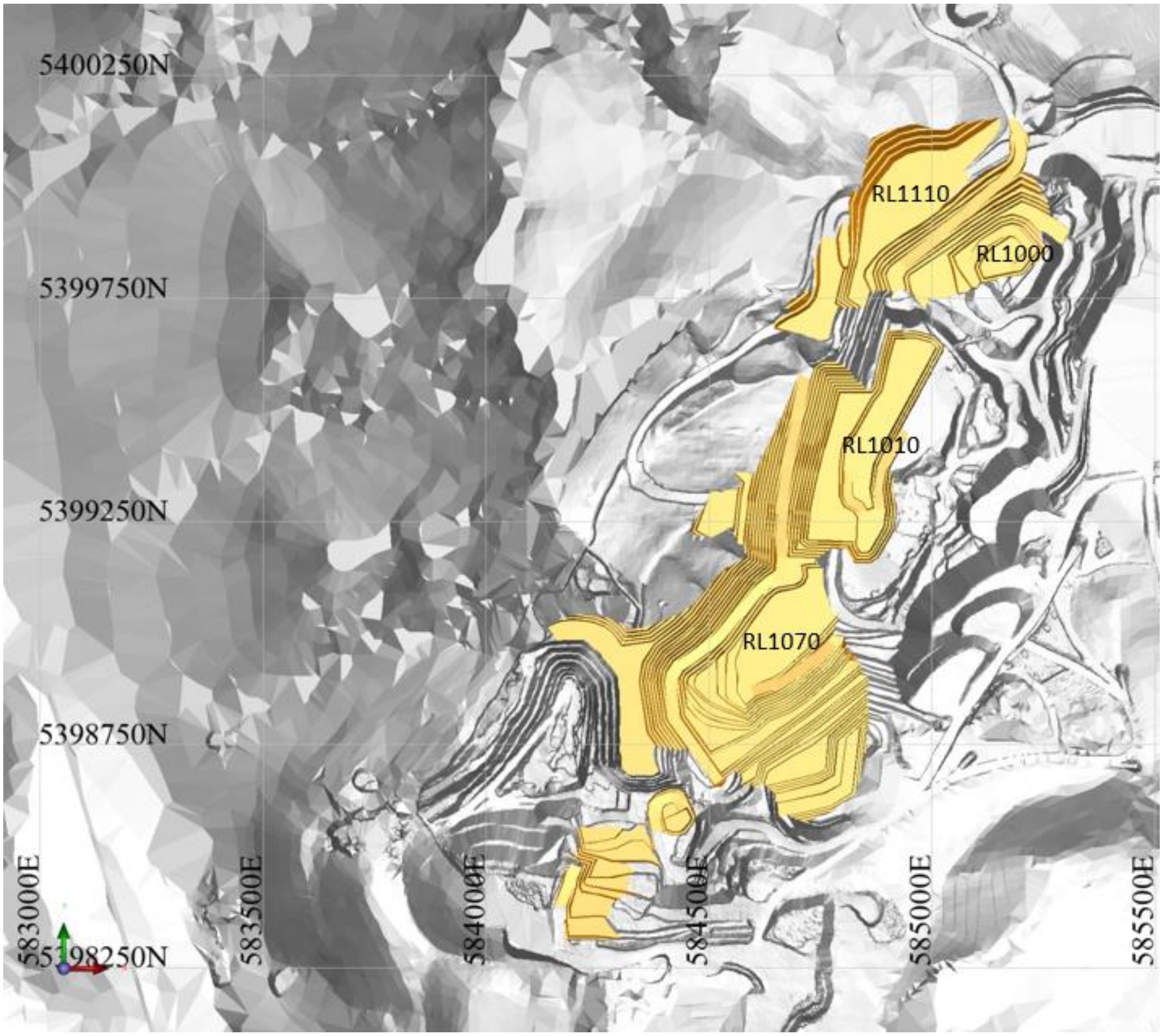


Figure 16-13: 2027 End of Pit Surface

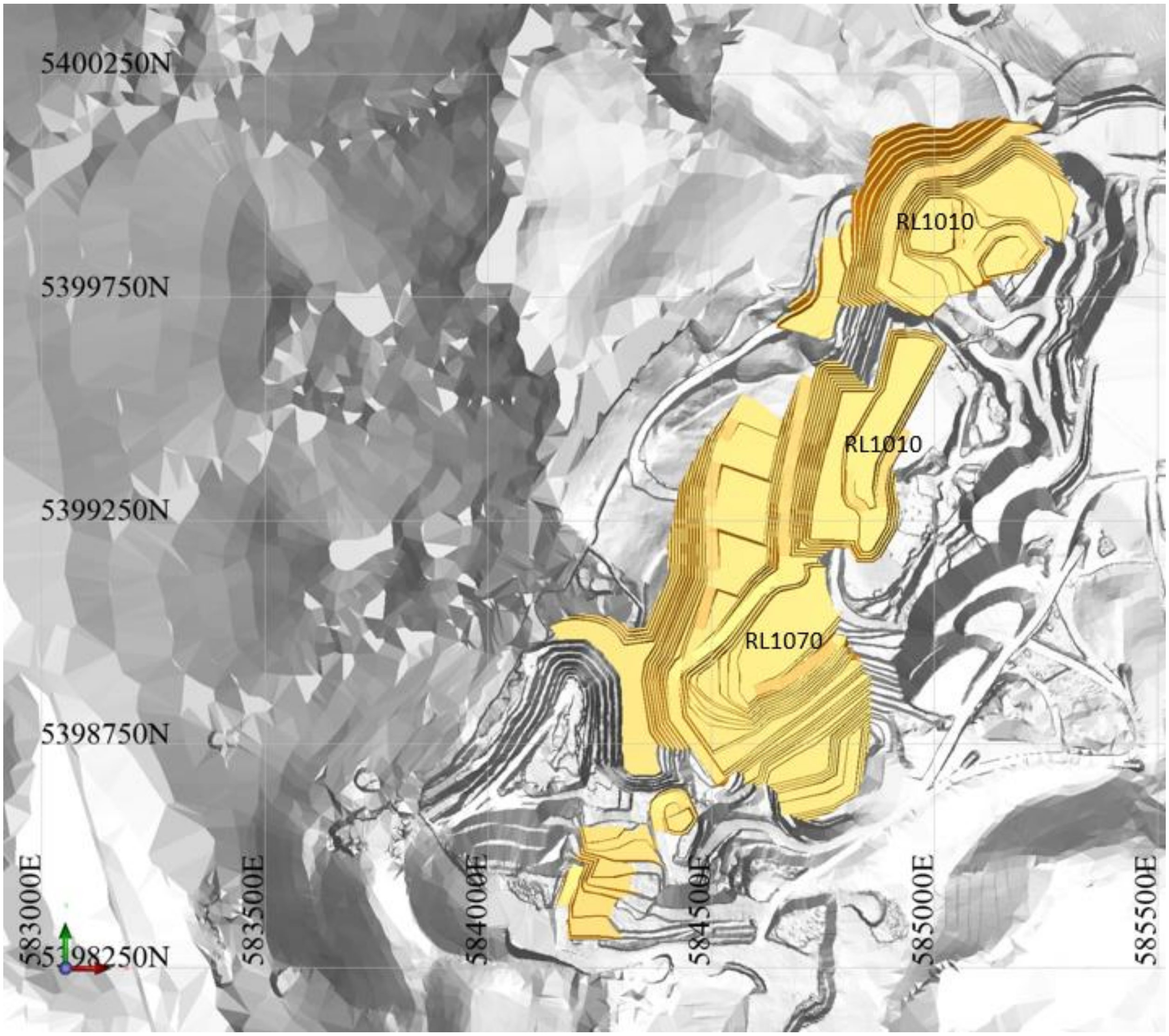


Figure 16-14 : 2028 End of Pit Surface

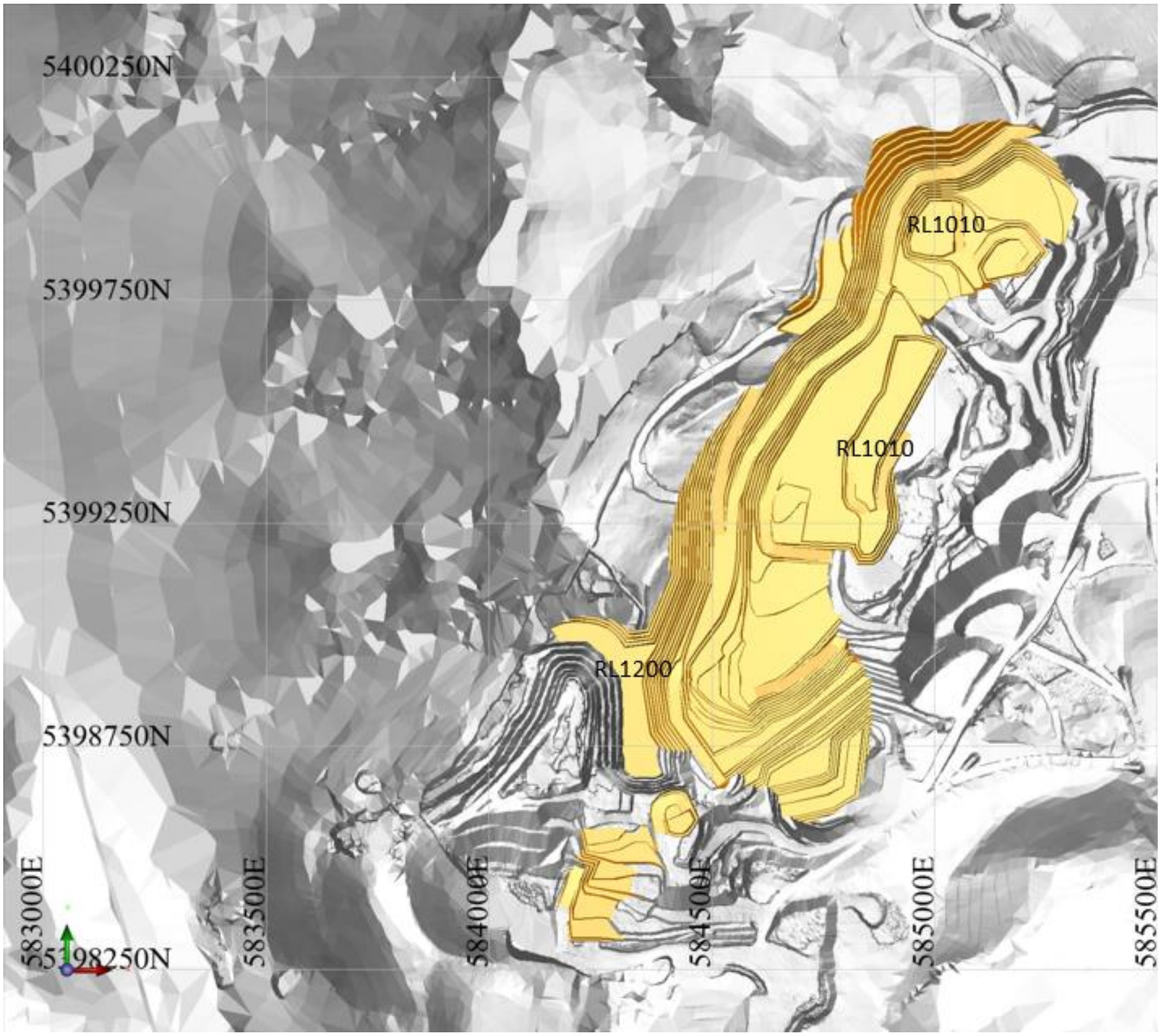


Figure 16-15 : 2029 End of Pit Surface

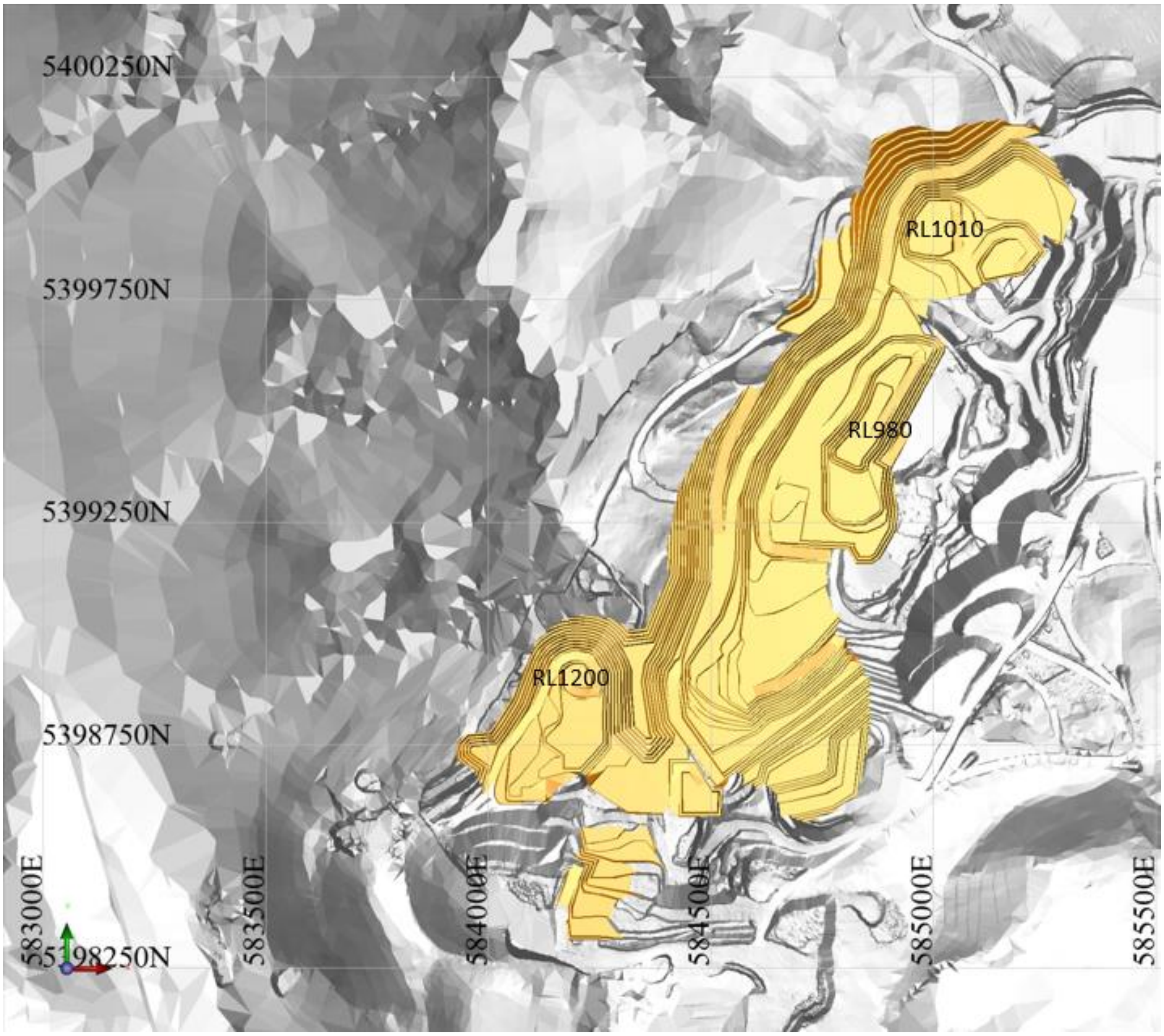


Figure 16-16 : 2030 End of Pit Surface

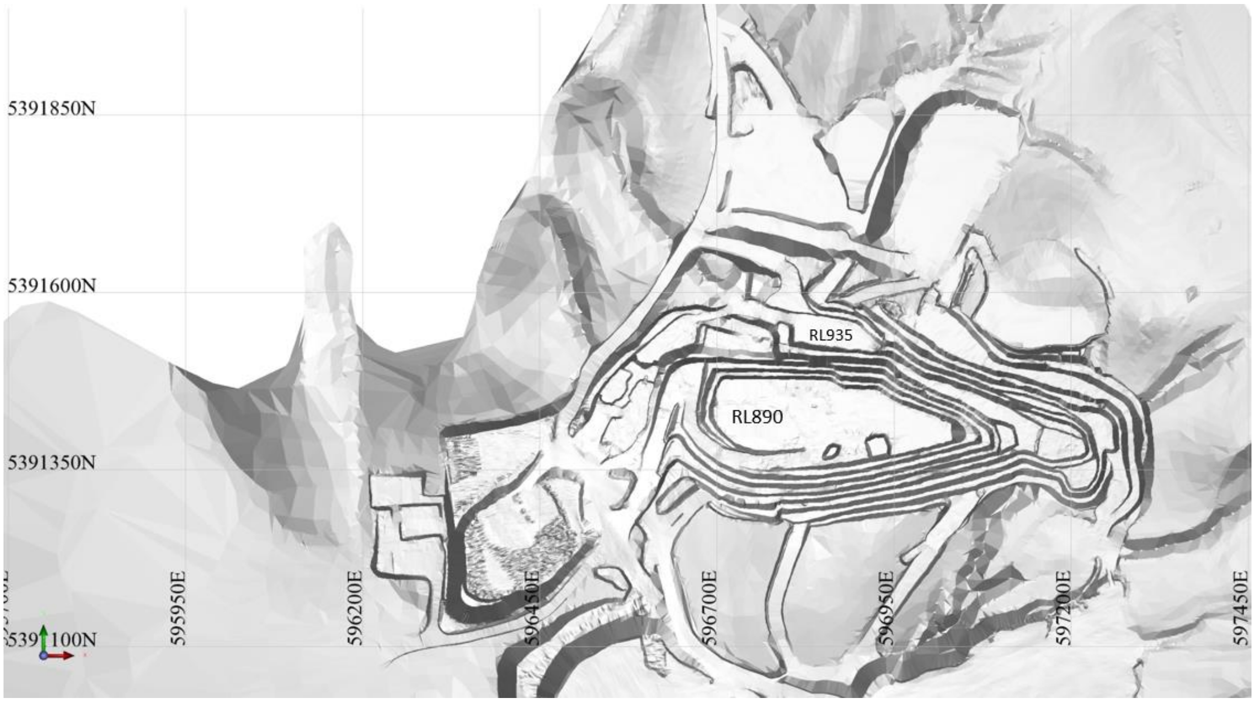


Figure 16-17: January 01 2024 pit survey

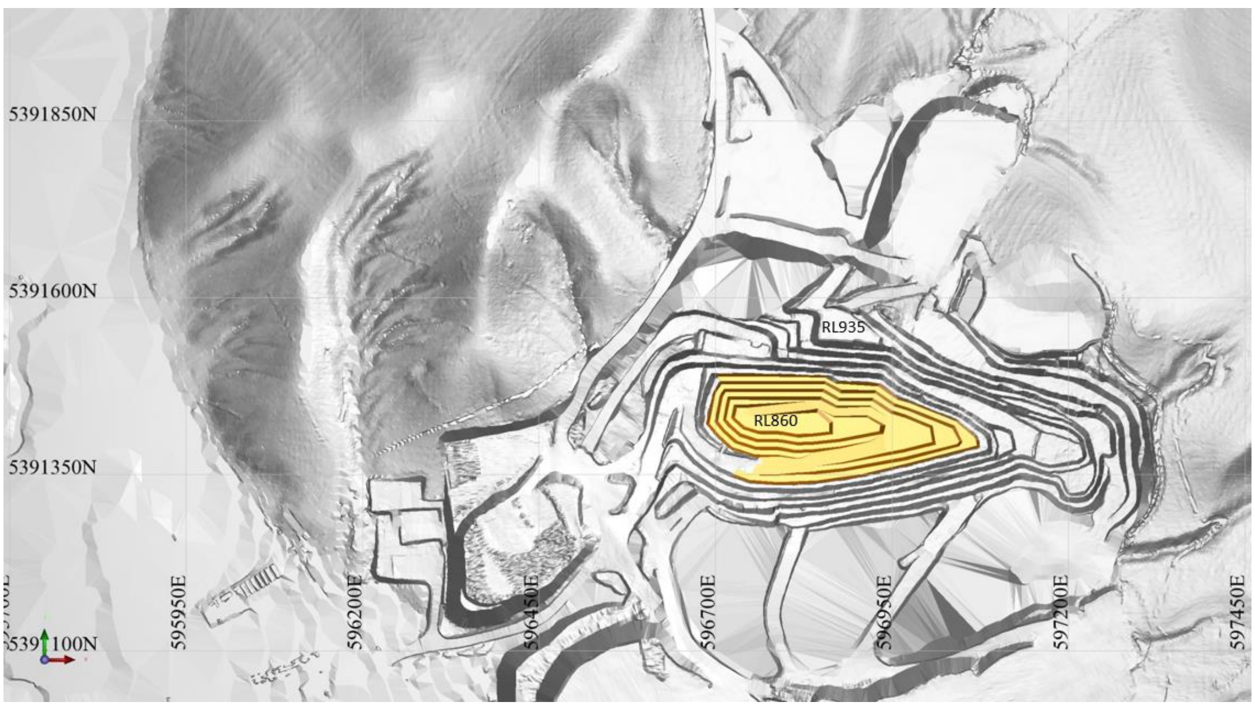


Figure 16-18: 2024 End of Pit Surface

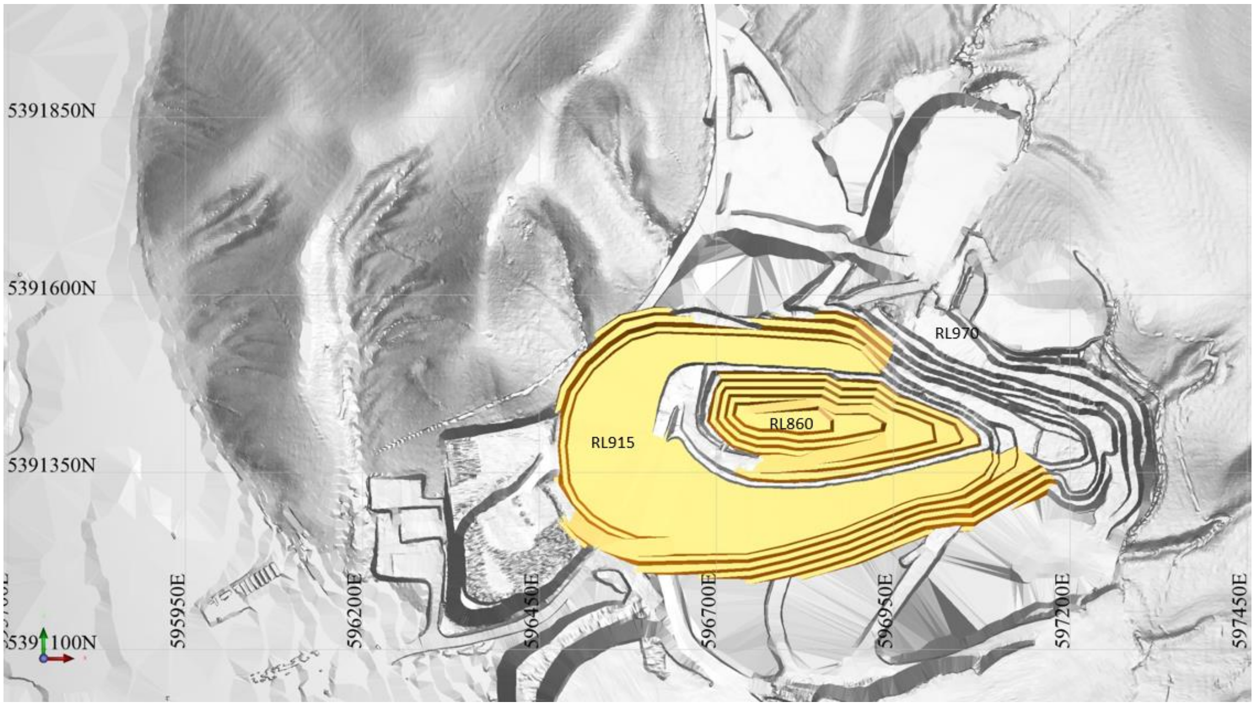


Figure 16-19: 2026 End of Pit Surface

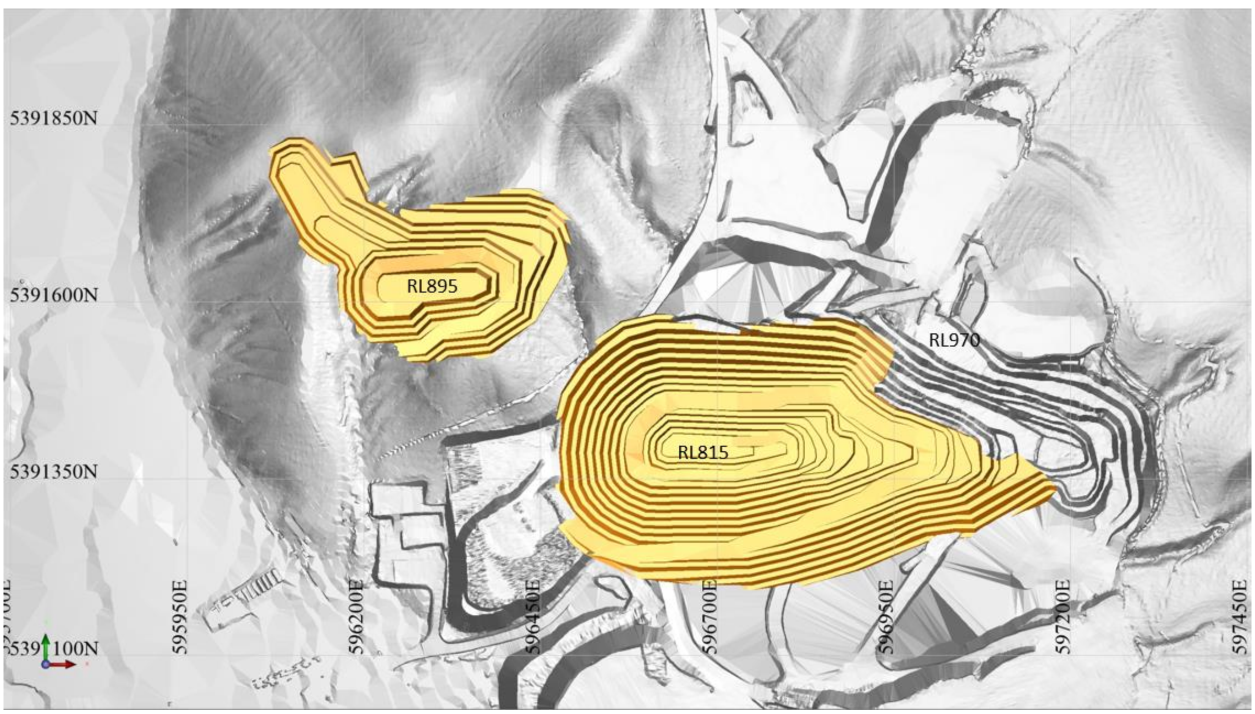


Figure 16-20: 2027 End of Pit Surface

16.2.4 Pit Dewatering

Mine dewatering has been, and will continue to be, handled by mine operations using in-pit sumps to deal with surface water run-off. It has not been observed, or is it anticipated, by site operations that any major ground source of water exists. Water will be pumped from in-pit ponds to staging ponds before being piped to water storage facilities for use in processing or treatment. Water from the staging ponds will continue to be used for water trucks to suppress dust along haul roads.

16.3 Waste Dump and Stockpile Design

16.3.1 Waste Dump Design Parameters

The waste dump parameters are as follows:

- Bench Height: 20 m (BH)
- Berm Width: 20 m (CB)
- Bench Angle: 34° (BFA)

The remaining criteria are determined by the rock characteristics of the materials to be placed as shown in the geotechnical analysis. Placement of material in the waste dumps will be done in a manner that will satisfy the minimum criteria to meet the minimum Factor of Safety for the designs.

The designs for active waste dumps assume a swell factor of 30% for the material delivered from pit benches, considering natural sorting and 10% compaction of the dumps. It is also assumed that the blasted waste rock will settle at the natural angle of repose of 34°.

Totally 93 million tonnes waste will be dumped from Boroo pit. All waste material will be dumped as a back fill.

Figure 16-21 shows the final backfill waste dump design.

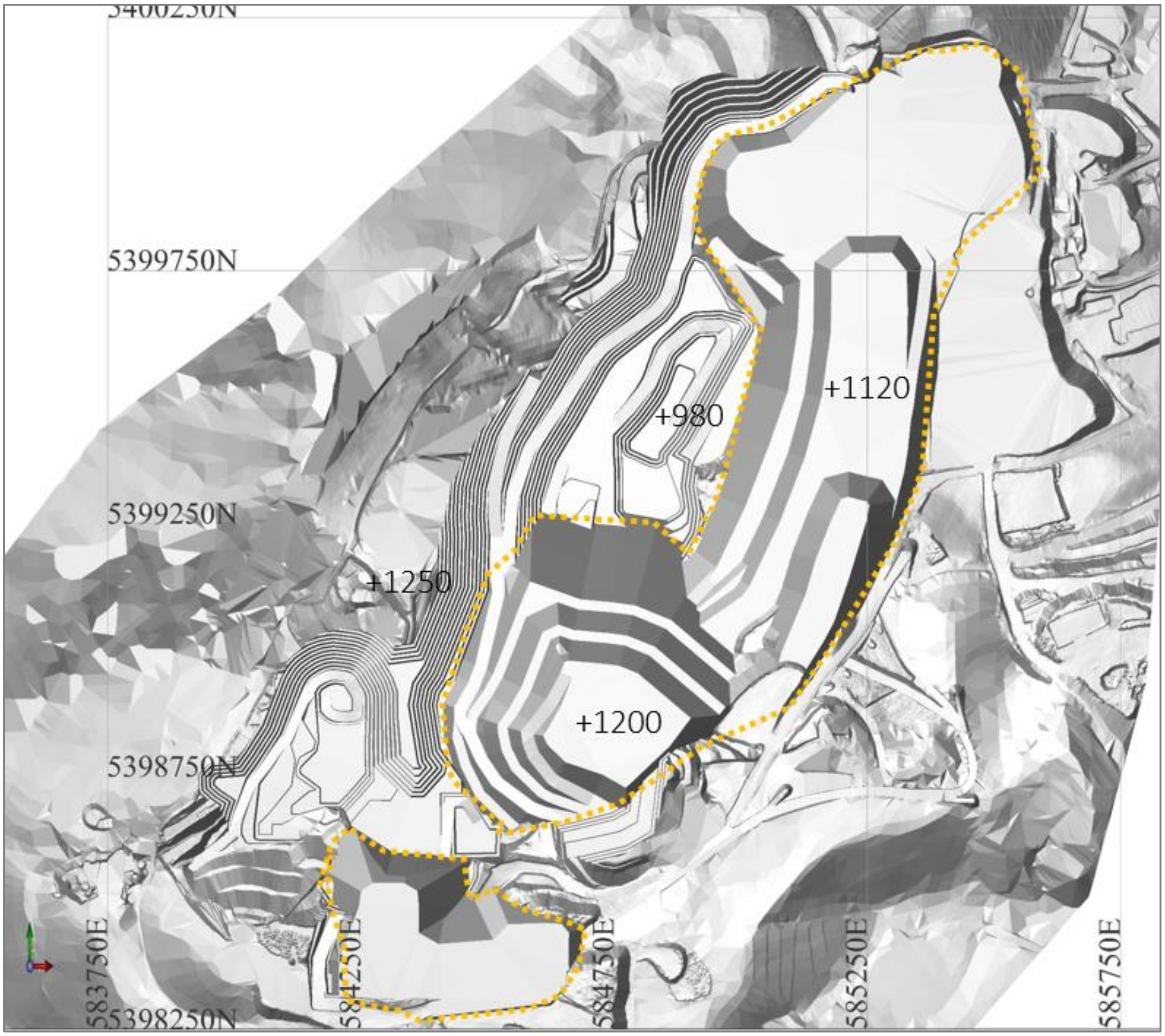


Figure 16-21: Final Backfill Waste Dump Design

Totally 8.3 million tonnes waste will be dumped from Ulaanbulag pit, and 4.1 million tonnes waste will be dumped as a back fill, rest of them to be sent waste dump as shown in Figure 16-22.

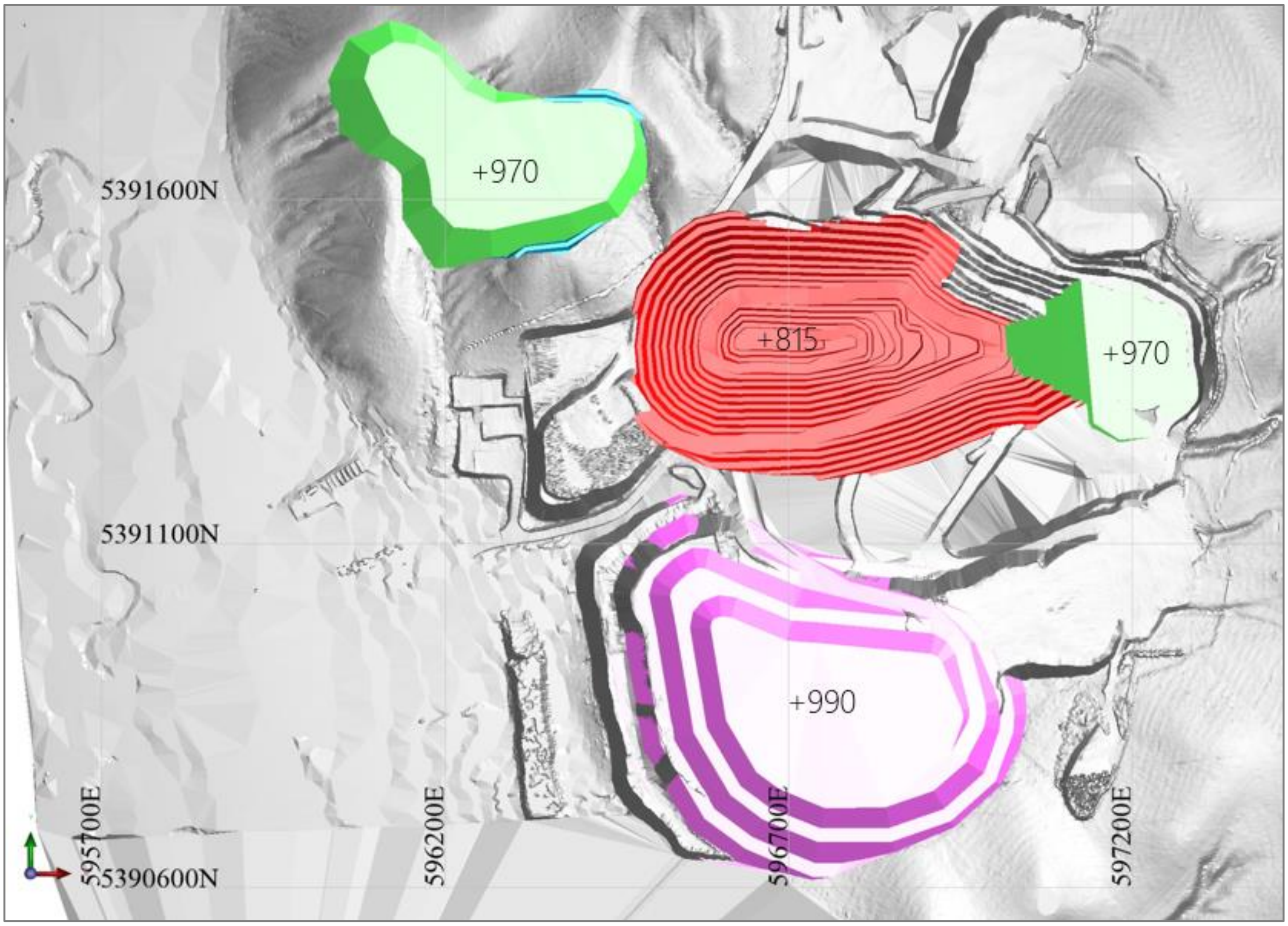


Figure 16-22: Waste Dump Design Ulaanbulag

16.3.2 Stockpile Design Parameters

16.3.2.1 Heap Leach Ore Stockpile

Totally 16.6 million tonnes heap leach ore (defined as material below the mill cut-off grade and above heap leach cut-off grade) will be crushed and stacked on the leach pad. Ore will be hauled from the pit or stockpiles near the pit to a heap leach stockpile. The largest heap leach ore stockpile will be 3.4 Mt in Year 2027 and be depleted to about 1.6 Mt for the remainder of the mine life.

16.3.2.2 Run of Mine Stockpile

The ROM stockpile is located adjacent to and west of the process plant for direct loading into the primary crusher. The ROM stockpile has been designed with this footprint to accommodate the maximum tonnage of planned stockpiled ore, with a maximum height of 5 m based on a conservatively estimated 30° angle of repose for the coarse ore material. The largest mill ore stockpile will be 2.3 Mt in Year 2026 and be depleted to about 1.6 Mt for the remainder of the mine life. As of the end of December 2023, approximately 1.8 Mt of ore have been stockpiled. The stockpile has been divided into five categories: ultra high grade (THG), high grade (HG), medium grade (MG), low grade (LG) and marginal low grade (MLG).

16.4 Mining Equipment

16.4.1 Equipment Requirements

LOM equipment requirements are based on the design parameters of the pit, production rate and haul profiles requirements. Equipment availability and utilization is based on Boroo Gold's historical data, internal asset management and maintenance plan.

For determining the number of each piece of equipment required, the following is considered:

- Annual production rate
- Haul road profile, gradient and length
- Operating speeds
- Equipment mechanical availability, utilization and overall efficiency
- Cycle times including spot, load, haul, dump and maneuvering times

A summary of the current equipment fleet is presented in Table 16-7 below.

Table 16-7: Current Core and Auxiliary Equipment

Core Equipment	Model	Class	Number
Drill	ROCD55	115 mm	5
Loading Equipment	CAT390	5.4 bcm	2
	CAT395	6.4 bcm	1
	CAT6020	12.0 bcm	1
Truck	CAT773	60 tonne	10
	CAT777	96 tonne	6
Auxiliary Equipment	Model	Class	Number
Wheel Loader	CAT988K	7.5 bcm	2
	CAT972L	4.5 bcm	1
	CAT980	6 0 bcm	1
	SEM676	3.2 bcm	1
Grader	CAT14H	-	2
	CAT16GC	-	1
Water Truck	Howo	25000L	2
	Howo	15000L	2
Fuel and Lube Truck	Fuel Truck	15000L	2
	Lube Truck	-	1
Low Bed Truck	Howo T400	-	1
Pickup Trucks	-	-	20
Light Tower	Epiroc	-	9

Table 16-8: Core Equipment Requirements by Period for Boroo and Ulaanbulag

Year	2024	2025	2026	2027	2028	2029	2030	Max
ROC D55	5	5	5	5	5	5	3	**5**
CAT390	2	2	2	2	2	2	1	**2**
CAT395	1	1	1	1	1	1	1	**1**
CAT6020	1	1	1	1	1	1	1	**1**
CAT773	10	10	10	10	10	10	10	**10**
CAT777	6	6	6	6	6	6	6	**6**
CATD9	4	4	4	4	4	4	4	**4**
CAT14H	2	2	2	2	2	2	2	**2**
CAT16G	1	1	1	1	1	1	1	**1**

Due to the relatively short life of mine at Boroo and Ulaanbulag, equipment replacement is not required for most core and auxiliary equipment. Details of capital and operating costs for mining operations can be found in Chapter 21.

16.4.2 Drilling and Blasting

Currently, drilling and blasting completed by a contractor. Primary production drilling uses a 115 mm hole for ore and waste blasting respectively. Contractor was selected due to their flexibility and ability to perform production, pre-splitting. Contractor company drills are durable and reliable and have highly efficient dust collection systems which will ensure the mechanical availability of the drills remains high while operating in a dusty environment. Blast design assumptions have been used to estimate the drilling production rate and consumables required. Totally five drills are required at peak production. The following parameters are used:

Table 16-8: Blast Design and Parameters

Parameter	Units	Ore	Waste
Bench height	m	5	5
Subdrill	m	0.5	0.5
Burden	m	4	4
Spacing	m	4	4
Hole Diameter	mm	115	115
Powder Factor	kg/bcm	0.4	0.4
Drill Penetration Rate	m/hr	25	25

16.4.3 Loading and Hauling Equipment

The current principal mining equipment is supplied by Caterpillar and includes two 5.4 m3 CAT390 hydraulic excavators, one 6.4 m^3 CAT395 hydraulic excavator and one 12 m3 CAT6020 hydraulic excavator and ten 50-tonne CAT773 haul trucks, six 100-tonne CAT777 haul trucks. Selection of loading equipment was based on the ore to waste ratio, bench sizes and material selection control, with ore excavators a smaller size for higher selectivity, particularly in the lower benches of the pit where available space is lower due to the minimum mining width. Current principal mining equipment numbers will be enough throughout the life of mine, inclusive of any time unavailable due to maintenance. Haulage profiles were developed in Talpac™ for both ore and waste cycle times. Over the life of mine, current truck numbers meet the estimated production schedule.

16.4.4 Auxiliary and Support Equipment

Auxiliary and support equipment are used for activities including creation and maintenance of the waste dump, in-pit bench preparation for drill and blast and load and haul activities, re-handling of the ROM stockpile at the process plant, haul road upkeep and refueling and maintenance of the heavy vehicle fleet. Mobile equipment maintenance trucks, a fuel and lube truck and low bed are supported heavy equipment during day-to-day operations.

16.4.5 Equipment Utilization

Mechanical availability, utilization and overall efficiency are shown in Table 16-8 below.

Table 16-9: Equipment Efficiency Factors

Equipment Efficiency Factors		
Item	Unit	Value
Availability	factor	87%
Utilization	factor	92%
Efficiency	factor	88%

To complete the equipment performance calculations, parameters based on the specifications of each piece of core equipment were included to calculate cycle times and estimate the maximum material movement possible based on the remaining available time. Annual truck hours were then derived, with excavator hours based on bucket capacity and individual loading and dumping times as described in Section 16.4.3. From this, the required equipment numbers on an annual basis are derived as described in Section 16.4.1. Haul truck parameters including load factors and travelling speeds are included in Table 16-9, with excavator parameters included in Table 16-10.

Table 16-10: Haul Truck Parameters

CAT 773			CAT 777		
Item	Value	Unit	Item	Value	Unit
Truck Life	50,000	hours	Truck Life	50,000	hours
Truck Nominal Payload	50	tonnes	Truck Nominal Payload	100	tonnes
Truck Load Factor	0.9	factor	Truck Load Factor	0.9	factor
Actual Truck Payload	45	tonnes	Actual Truck Payload	90	tonnes
Speed Loaded Ramp	12	km/hr	Speed Loaded Ramp	12	km/hr
Speed Loaded Flat	45	km/hr	Speed Loaded Flat	45	km/hr
Speed Unloaded Ramp	30	km/hr	Speed Unloaded Ramp	30	km/hr
Speed Unloaded Flat	45	km/hr	Speed Unloaded Flat	45	km/hr

Table 16-11: Excavator Parameters

CAT 390 & CAT 395			CAT 6020		
Item	Value	Unit	Item	Value	Unit
Excavator Life	50,000	hours	Excavator Life	50,000	hours
Nominal Bucket Payload	9, 11	tonnes	Nominal Bucket Payload	21	tonnes
Passes to Load	4-5	passes	Passes to Load	5	passes
Average Time Loading	33	%	Average Time Loading	79	%
Average Time Maneuvering	11	%	Average Time Maneuvering	5	%
Average Time Idling	56	%	Average Time Idling	16	%

17.0 RECOVERY

The current mill flowsheet includes crushing, grinding, gravity concentration, leaching and carbon-in-pulp (CIP), cyanide detoxification and arsenic precipitation to produce gold doré.

17.1 Process flowsheet

The selected flowsheet for Boroo and Ulaanbulag ore (Figure 17-1), based on the test results described in Section 13, is a standard layout that consists of crushing, grinding, gravity concentration, cyanide leaching and gold recovery in a carbon-in-pulp (CIP) circuit.

Milling flowsheet includes the following major units:

- Primary crushing
- Grinding and classification including gravity concentration
- Thickening and leach/carbon in pulp (CIP)
- Carbon elution and reactivation
- Gold electrowinning and refining
- Cyanide and Arsenic detoxification; and
- Tailing storage

The Boroo mill was designed with a capacity to process 1.8 million tonnes of ore per year (5000 tpd). Mill commissioning commenced in December 2003 and by March 1, 2004 when commercial production was achieved. In 2021, the mill was restarted since then throughput has steadily increased to where it is 1.6-1.7 million tonnes per year or 4500 - 5 000 t/d.

Mill recovery has steadily decreased since the depletion of oxide and transitional ore. When processing sulphide or fresh ore, mill recovery is typically in the range of 60% to 70%. A significant portion of the recovery is still achieved in gravity separation.

A jaw crusher reduces the ore size to P80 125mm. The crushed ore is fed directly to a semi-autogenous (SAG) mill (8.5-metre diameter) or to a temporary coarse ore stockpile from which it can be reclaimed during crusher maintenance.

Locally made burnt lime is added to ore on the SAG mill feed conveyor for pH control in the leach circuit.

A central dust collector is installed adjacent to the crusher building and through a series of ducts collect dust from the crusher discharge and conveyor transfer points.

Grinding is performed by an open circuit 8.5 m diameter 3 500 kW SAG mill followed by 4.8 m diameter 3 000 kW ball mill operating in closed circuit with a cyclone cluster for classification. Cyclone underflow is split between the ball mill and the gravity concentration circuit. Cyclone overflow is directed through trash screens to the leach feed thickener.

Grinding circuit product is nominally 80% passing 75 microns.

Approximately 30% of the total cyclone underflow reports to the gravity circuit, which consists of two 750-mm Knelson concentrators followed by an Acacia reactor where the gravity-recovered gold is leached in high cyanide solution. The pregnant solution is then pumped to the elution circuit for electrowinning. The range of gravity gold recoveries is 0-84%, averaging 30%. Gravity concentration tailings gravitate to the mill discharge pump box.

Pre-leach thickener overflow is pumped to the make-up process water tanks and the underflow is pumped to the leach tanks at a density of 55% solids to maximize water recovery to grinding.

A combination two-stage leach and six stage carbon in pulp (CIP) circuit recovers the remaining gold. The circuit has a nominal residence time of 18 hours to 24 hours.

In the CIP circuit the barren carbon is moved counter currently to the slurry flow to adsorb dissolved gold. Inter-stage screens in each of the six CIP tanks are used for carbon retention. Loaded carbon is pumped to the carbon elution circuit.

The 3t capacity carbon elution circuit is based on the AARL (Anglo American Research Laboratories) elution process. Loaded carbon from the heap leach is also processed through the existing elution circuit.

After the acid wash, the carbon is transferred to the elution column where it is subjected to heat of 120^{0}C and pressure of 150 kPa. Total stripped time is 6 hours. The stripped carbon is transferred to reactivation. The carbon is reactivated in a vertical kiln at 600^{o}C. After wet screening to remove undersize carbon, the reactivated carbon is pumped to the No. 6 CIP tank or the heap leach carbon in column (CIC) circuit.

The pregnant eluate from carbon elution is mixed with the pregnant solution from the Acacia Reactor.

Gold is recovered in the electrowinning cells where the gold is plated onto steel wool cathodes. The steel wool is calcined and refined to doré bullion in a tilting furnace.

Tailings slurry from the CIP circuit is pumped to the detoxification plant. The detoxification plant consists of two (2) stages of cyanide destruction where copper sulphate, sodium metabisulphite and compressed air are added to complete cyanide detoxification. Cyanide levels must be reduced to less than 1 mg/L weak acid dissociable cyanide (WAD) for disposal. This is followed by a single stage of arsenic precipitation using ferric sulphate solution. The slurry then flows by gravity to the tailings storage facility.

Detoxified slurry gravitates to a tailing storage facility approximately 5 km distant from and 150 meters below the process plant through an HDPE (high density polyethylene) pipeline.

Tailings slurry is discharged around the TMF via spigotted outlets located at regular intervals around the embankments. Decant pontoon pumps remove water from the tailings dam during non-winter months.

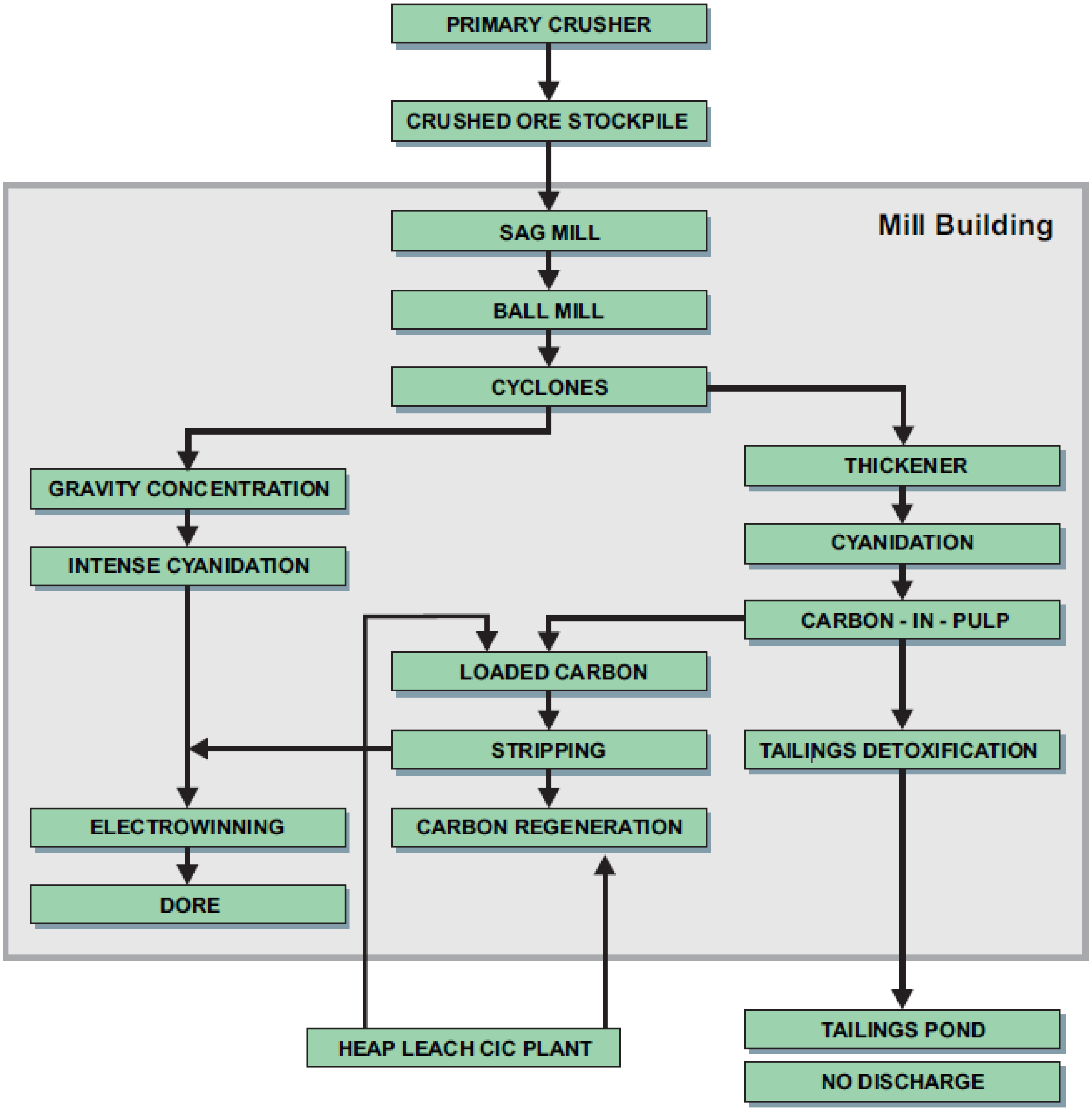


Figure 17-1: Mill Flowsheet

17.2 Boroo Recovery model

The recovery model at the Boroo Mine is modeled in three zones, and the model is based on ore body weathering, lithology, alteration and laboratory testwork results collected at the deposit scale. Also, as mentioned above, the modeled metal recovery is verified against the actual data.

The metal recovery is modeled at 75%, the zone includes oxidized and transitional ore and is lithologically dominated by granite.

The metal recovery is modeled at 70%, the area lithology is granite and sandstone.

The metal recovery is modeled 62%, the area is dominated by sandstone, and in the fresh ore zone. Based on testwork result and actual performance, the recovery averaging 65-67%.

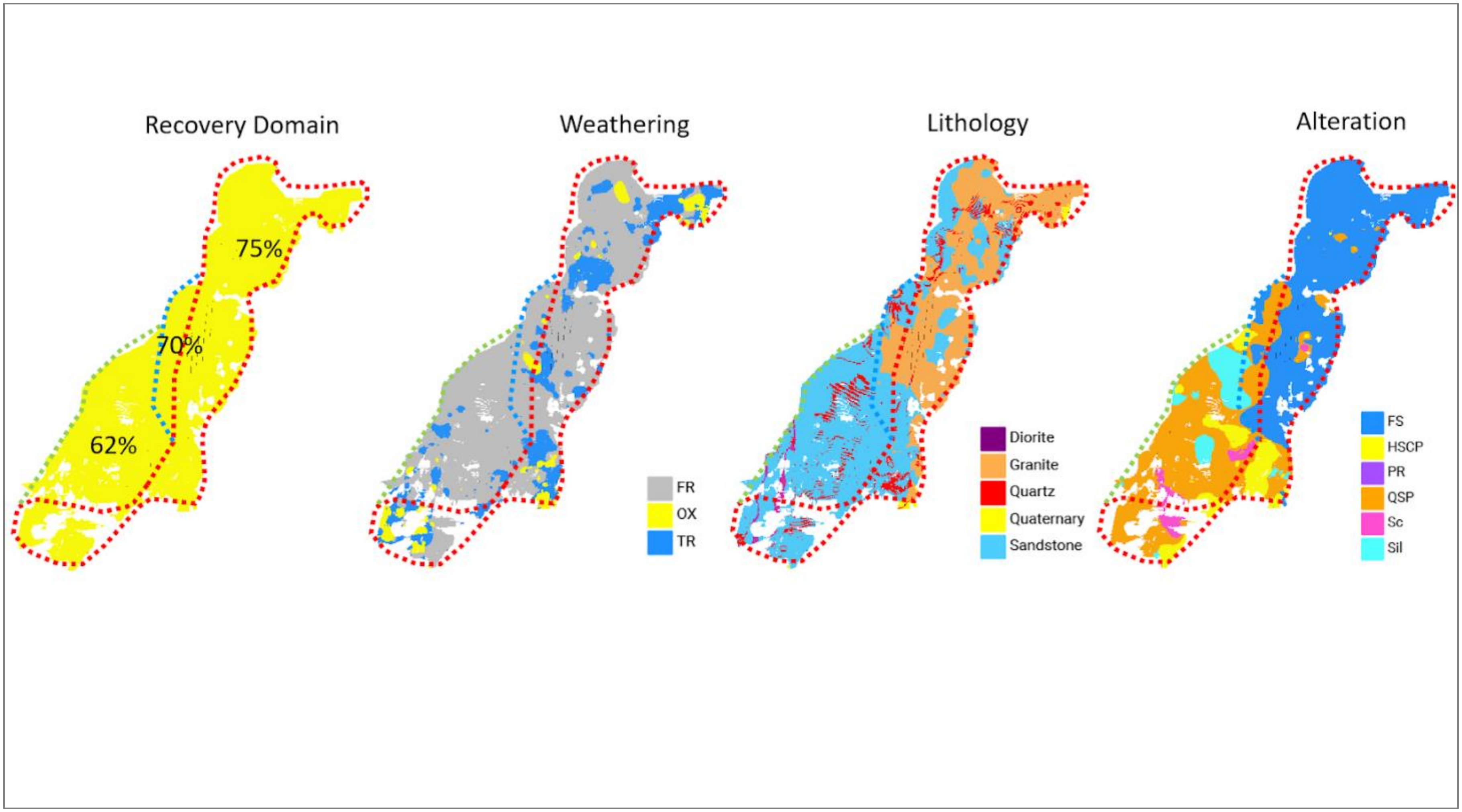


Figure 17-2: Boroo recovery model

17.3 Ulaanbulag Recovery model

Ulaanbulag recovery estimate are based on internal metallurgical testwork in 2014 and 2021. A summary of testwork result of the recovery shown in Table 17-1.

Table 17-1: Metallurgical testwork result

Oxidation	Recovery, % (2021)	Recovery, % (2014)
Oxide ore	76.3	84.7
Transitional ore	70.8	73.5
Sulphide ore	66.9	59.6
Average	71.2	71.4

During 2021-2023 mining, Ulaanbulag deposit oxide and transition ore operation gold recoveries averaging 83.5% and ranged 81% to 85%. (2023 recovery averaging 81%). Ulaanbulag deposit recovery was estimated 2021 composite testwork results, ore control results showed that the gold recovery prediction from 2021 leach well testwork to be accurate.

17.4 Heap leaching

17.4.1 Ore Handling

Nominally 3.0 million tonnes of heap leach feed will be crushed and stacked on the leach pad. Ore will be hauled from the pit or stockpiles near the pit to a heap leach stockpile. Crushing and stacking will proceed at a nominal rate of 10 000 t/d. 50% of the total heap leach ore is crushed by primary crushers.

Ore will be stockpiled by primarily by grade. Stockpiled ore will be fed from stockpile to the crusher by front end loader. The crusher is a jaw crusher operating with a closed side setting of 125 mm. Crushed ore is mixed with burnt lime on the discharge conveyor that feeds onto a surge pile. Ore will be loaded into trucks by a front-end loader, and hauled to the leach pad where the material will be stacked in 10 m lifts.

17.4.2 Heap Leach Pad

The leach pad comprises an area roughly 600 m x 500 m located on the south side of the Ikh Dashir Valley. The heap leach pad will provide an ultimate ore capacity of 18.5 million tonnes. Totally 16.6 million tons of ore will be leached until 2031, and additionally 10 million tons leach pad expansion required.

Capacity was based on an average stacked ore density of 1.7 t/m3 and a maximum heap height of 60 m over the pad liner. The leach pad is lined with a 300 mm compacted soil base overlain by 1.5 mm linear low-density polyethylene (LLDPE) synthetic liner and a 600 mm layer of crushed liner cover. The solution collection system consists of pipe placed within the liner cover layer. Primary solution collection is from 75 mm perforated pipes placed at a spacing of 10 m across the entire lined pad. The 75 mm pipes convey flow to larger collector pipes which convey pregnant solution flows to the PLS pond.

Ore will be stacked in lifts around the perimeter of the heap to provide an average overall ore slope of 2:1 (horizontal to vertical). Heap slopes have been designed employing static and dynamic factors-of-safety of 1.5 and 1.1 respectively to ensure operational and post-closure stability under both normal and earthquake conditions.

17.4.3 Solution Management

The design barren (BLS) and pregnant (PLS) leach solution flow rate of 343 m3/h was calculated based on a leaching ratio of 1:1 (tonnes of solution to tonnes of ore), derived from metallurgical test results and the annualized ore production rate of 8 220 t/d. The solution delivery systems for PLS and BLS are sized to accommodate an additional 120 m3/h temporary flow capacity to allow operating at a higher application rate for recirculation of excess solution resulting from storm events.

Solution is applied to the ore on the pad at an average rate of about 0.012 $m^3/h/m^2$. Capacity exists to increase the application rate on a short-term basis by 30%. Solution will be collected by a drainage system described previously, which directs PLS to the PLS pond located at the northeast corner of the leach pad.

The PLS pond is sized to accommodate 8 hours of operational flows plus 24 hour “drain down” of the ore heap. The PLS pond is fitted with a sump and well riser pipe to pump PLS to the plant at a normal rate of 325 m^3/h. During upset conditions, such as a power or pump outage, or a severe storm event, solution can flow by gravity from the PLS pond to a lined storm pond located adjacent to the PLS pond. The storm pond is sized to a volume of 39 200 m^3 to accommodate the 104 mm of precipitation resulting from the projected 100-year, 24-hour storm event.

The PLS pond lining system consists of a 300 mm layer of compacted soil, a 1.5 mm LLDPE liner, and a leak detection layer consisting of a geonet and a 2 millimeter primary HDPE liner. The storm pond is lined with a 2 mm HDPE primary liner placed on a 300 mm layer of compacted soil. Runoff from areas upslope of the leach pad will be routed around the facility using diversion ditches channels.

17.4.4 Leaching Operation

Normal summer operation will involve placement of drip emitter piping on the heap and connecting to one of the four dual header pipes to irrigate the active leach area. Each active leach cell will be approximately 165 m x 165 m or 28 000 m2. The stacking of each cell occurs approximately twice as fast (30 days) as leaching (60 days), permitting additional emitter piping to be placed throughout the summer. This provides sufficient material to conduct primary leaching of three cells during the winter prior to the resumption of stacking the following spring. Valves on the distribution header pipeline, accessible by manhole through insulating cover, will permit solution to be redirected as required. Various measures employed at other cold-weather heap leach operations have been incorporated into the operating plan to manage winter conditions. These include sufficient fleet for summer-only stacking, heat-traced BLS tanks and pipelines, a BLS solution heater, back-up power, and emitter lines buried under 2.5 m of crushed ore cover. A snow machine is also available to generate a blanket of artificial snow over active leaching areas to act as a layer of insulation. Provisions have also been made for ripping frozen ore prior to resuming leaching in the spring, temporary over-irrigation to melt potential ice layers in the heap, draining pipelines upon shutdown, and frost protection on the PLS pond.

17.4.5 Gold Recovery

Gold is recovered from the PLS using a CIC plant. PLS will flow through five, 2.4 m diameter columns loaded with activated carbon. Solution is introduced into the base of each column and will overflow via launder to the next stage. Carbon is periodically transferred upstream by recessed-impeller pumps progressively adsorbing more gold from solution. When carbon in the first column achieves gold loading of 5 000 g/t it will be transferred to the elution circuit in the existing plant for stripping.

The CIC plant will be located between the existing cyanide detoxification and mill buildings. This area will be enclosed and will become an integral part of the process plant making use of existing facilities for reagent mixing and facilitating supervision of operations. PLS will enter the CIC plant at east end of the building and will cascade toward the west. Barren solution overflowing from the last column will pass over a carbon safety screen prior to being pumped to a heated and insulated BLS Tank. Make-up water and cyanide concentration will be managed in the BLS Tank again, taking advantage of the proximity of the CIC plant to existing infrastructure.

Variable frequency drives interlocked with ultrasonic level indicators will ensure that pumps delivering PLS to the CIC plant and returning BLS to the heap maintain solution flow within an acceptable range. Overflow conditions will cause the BLS Tank to discharge excess solution to the final tails tank in the detoxification circuit or PLS to the storm water pond. BLS will pass through a diesel fired solution heater which will raise the temperature of the BLS 5^{o} C prior to returning to the heap.

Heap leach ore to be mined from the Boroo and Ulaanbulag pits is expected to be more refractory and has been given a lower overall recovery. Operation results showed that the heap leach gold recovery is 40%, and which is aligned with testwork result.

A simplified flowsheet for the heap leach is shown in Figure 17-3.

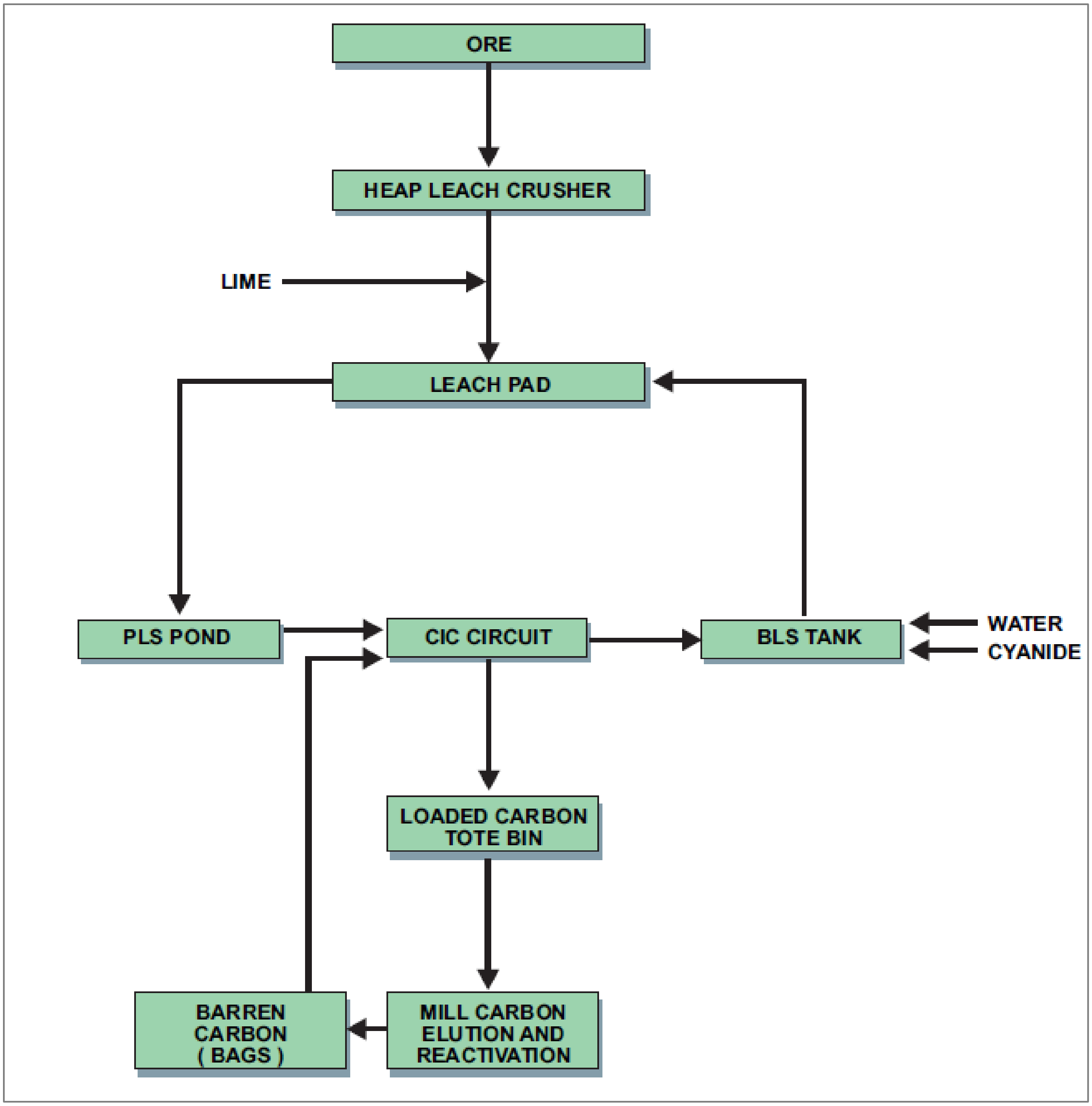


Figure 17-3: Heap Leach Flowsheet

17.5 Tailings Management Facility

The existing tailings management facility (TMF) of Boroo mine is located approximately 5 km to the east of the mine. The total capacity of the TMF is 21.3 million m3. The TMF consists of three parts: west, middle and main cell including middle cell. The total area of the TMF is 205 hectares. The ore extracted from the Boroo mine and Ulaanbulag mine will be processed at the Boroo processing plant, and the tailings from the plant will be stored in the main cell and the east and west cells.

By 2023, the main cell has been completed in a total of 10 phases, while the east cell has a total of six stages, and two phases of construction have been completed by 2023.

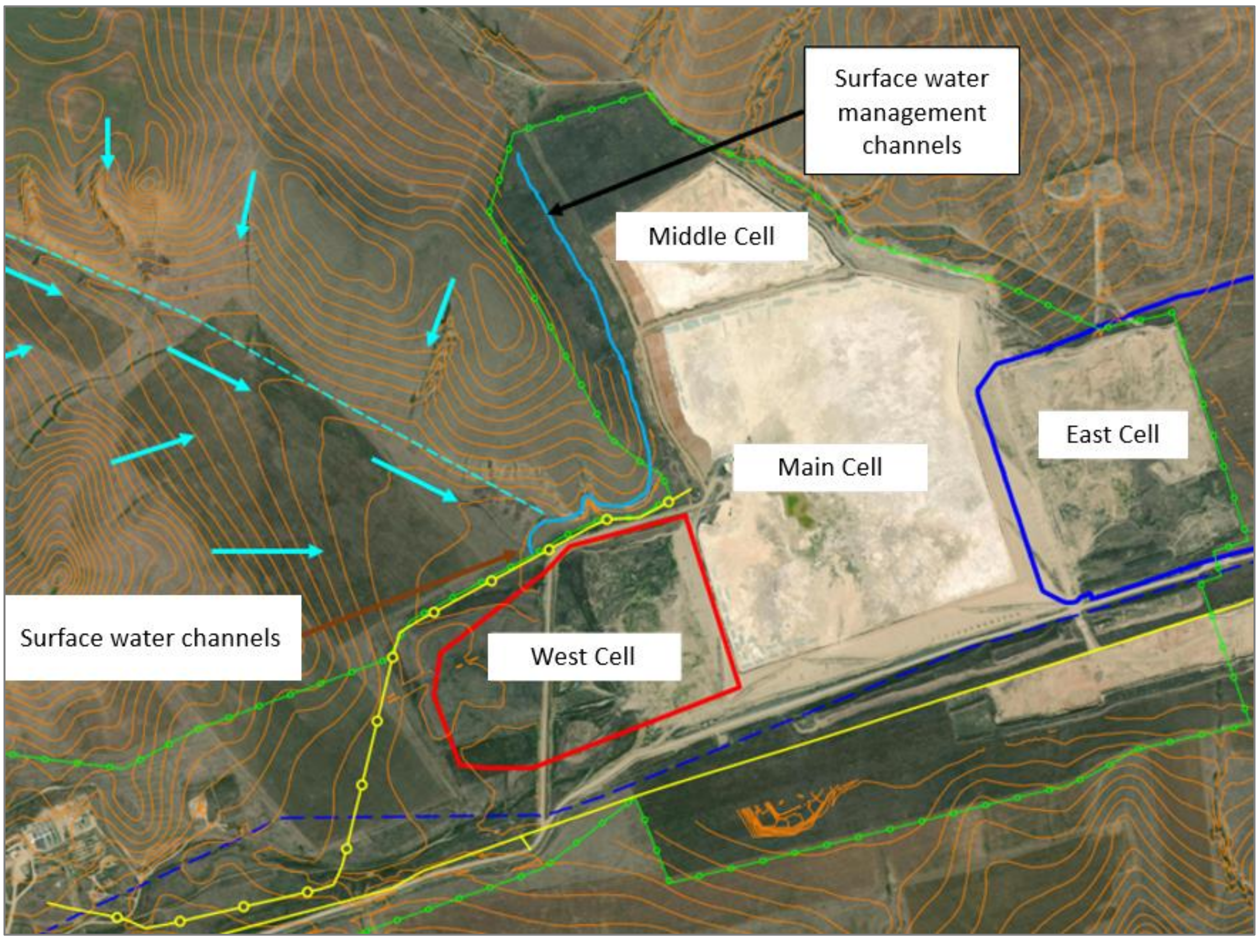


Figure 17-4: Overview of tailings management facility

The original TMF design has been made by Golder Associates Ltd. (Golder) in 2003 and additional design updates were developed in 2004 and 2006. Usny Erchim Ltd. (UEC) completed the design for TMFs further extension in 2013 and the design study has been approved by Golder.

17.5.1 East Cell

According to a blueprint approved by the “Usny Erchim” LLC in 2023, the "Usny Barilga" LLC, with a subscription from the "Boroo Gold" LLC, completed the second phase of the construction. The eastern cell was completed on November 19, 2023, subject to approval to begin construction.

The final elevation of the east cell is 928RL and the capacity is projected to be 12.0 million m^3. The following facilities include:

- Tailings cell, embankment
- A flood protection channel
- Monitoring tools and installation, piezometers,
- Piping, pumps station

The length of the embankment is 2,684 m, and the height of the embankment is 36.0 m. It is 970 m long and 800 m wide.

Table 17-2: East cell parameters

Phases	Capacity, m^3	Tailing level, m	Free board level, m	Height, m
1	2,107,475	901.0	902.5	9.5
2	2,186,947	908.0	909.5	7
3	2,149,703	913.5	915	5.5
4	2,203,763	920.5	921	6
5	1,979,020	924.0	925.5	4.5
6	1,408,663	926.5	928	2.5
Total	12,035,571			

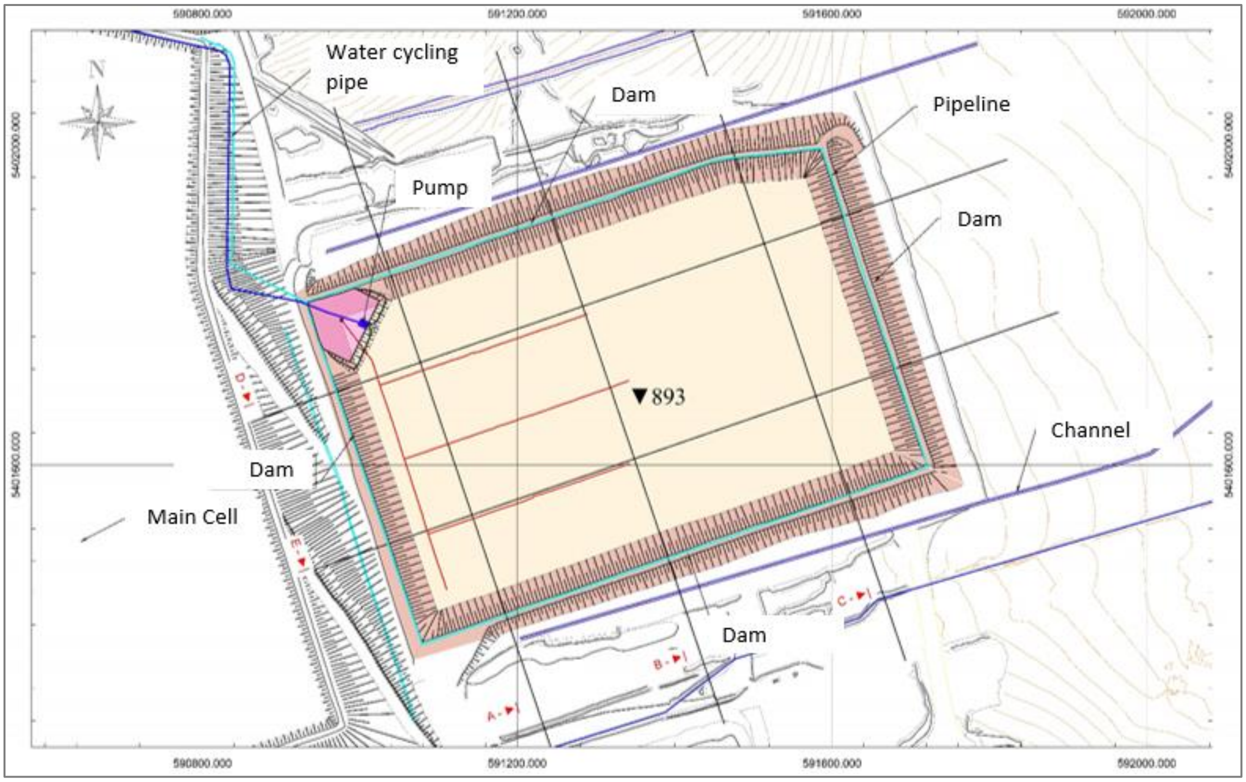


Figure 17-5: East cell tailings facility design

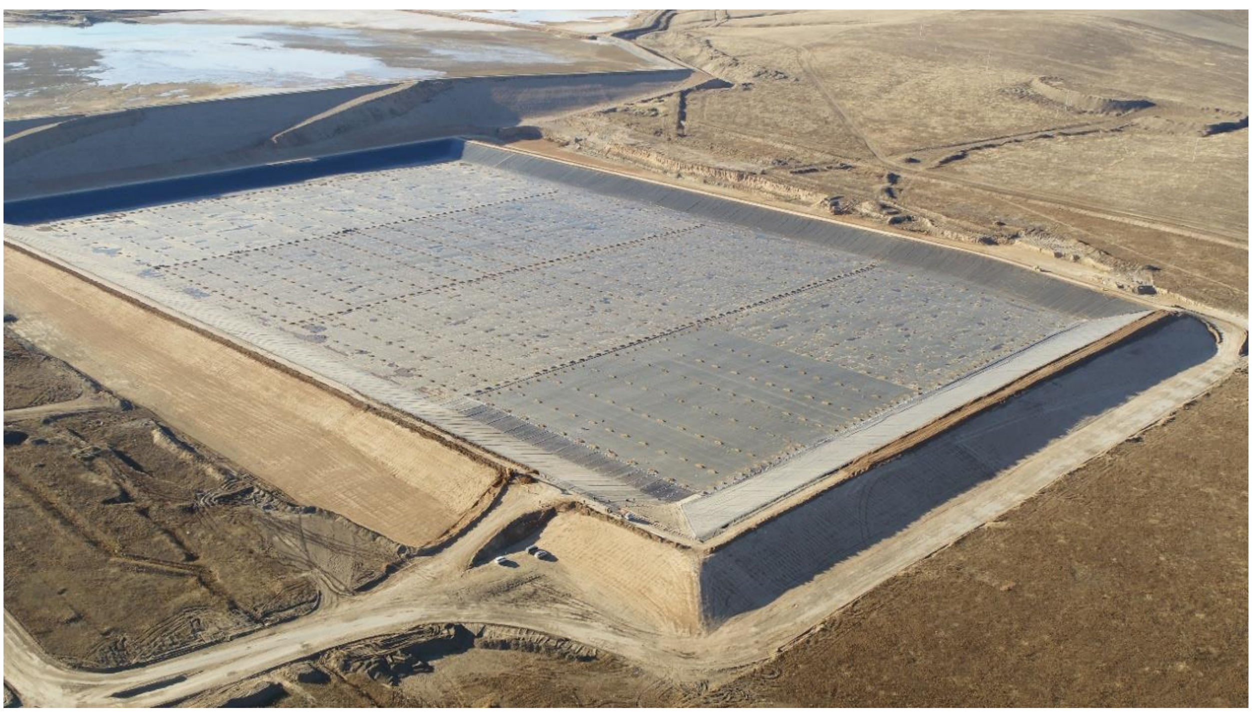


Figure 17-6: East cell tailings facility (a photo after completion)

17.5.2 West Cell

Eastern cell construction is expected to be completed in six phases until 2027. The eastern cell's capacity is capable of containing a total of 12.0 million m^3 waste, and it is necessary to build a residue of 7.6 million m^3 waste.

The final elevation of the west cell is 937.5RL and the capacity is projected to be 5.6 million m3. The following parameters of west cell

- Bottom level 920RL
- The final elevation 937.5RL
- Total area 55 hectares
- 560 m wide
- 800 m long

The water balance estimates that the west cell capacity will be available until 2029. The parameters of the western cell dams were estimated in the same way as the parameters of the main and left cell dams.

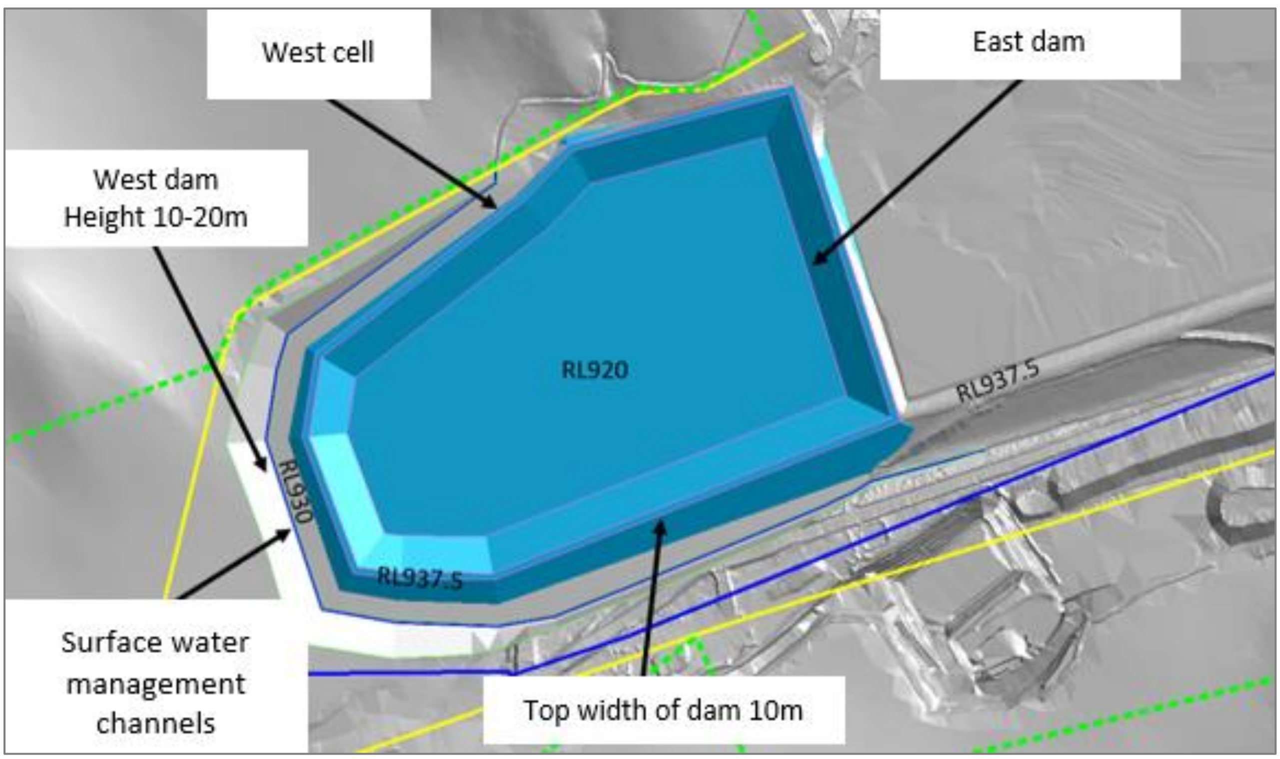


Figure 17-7: West cell tailings facility design

18.0 INFRASTRUCTURE

The existing facilities at the Boroo Gold Mine will be used to process ore from this next phase of the Project. The Boroo facilities include:

- Process plant, including facilities for unloading and feeding of ore, as well as grinding and leaching.
- Tailings management and heap leach pads, including impervious multi-layered basal linings of waste dumps
- Recycling treatment circuits to remove arsenic and cyanide from tailings
- Storage facilities for chemicals and reagents
- Dewatering facilities and return water lines
- Elevated tailings dam to accommodate additional tailings
- Warehousing
- Administration offices
- Maintenance Shop

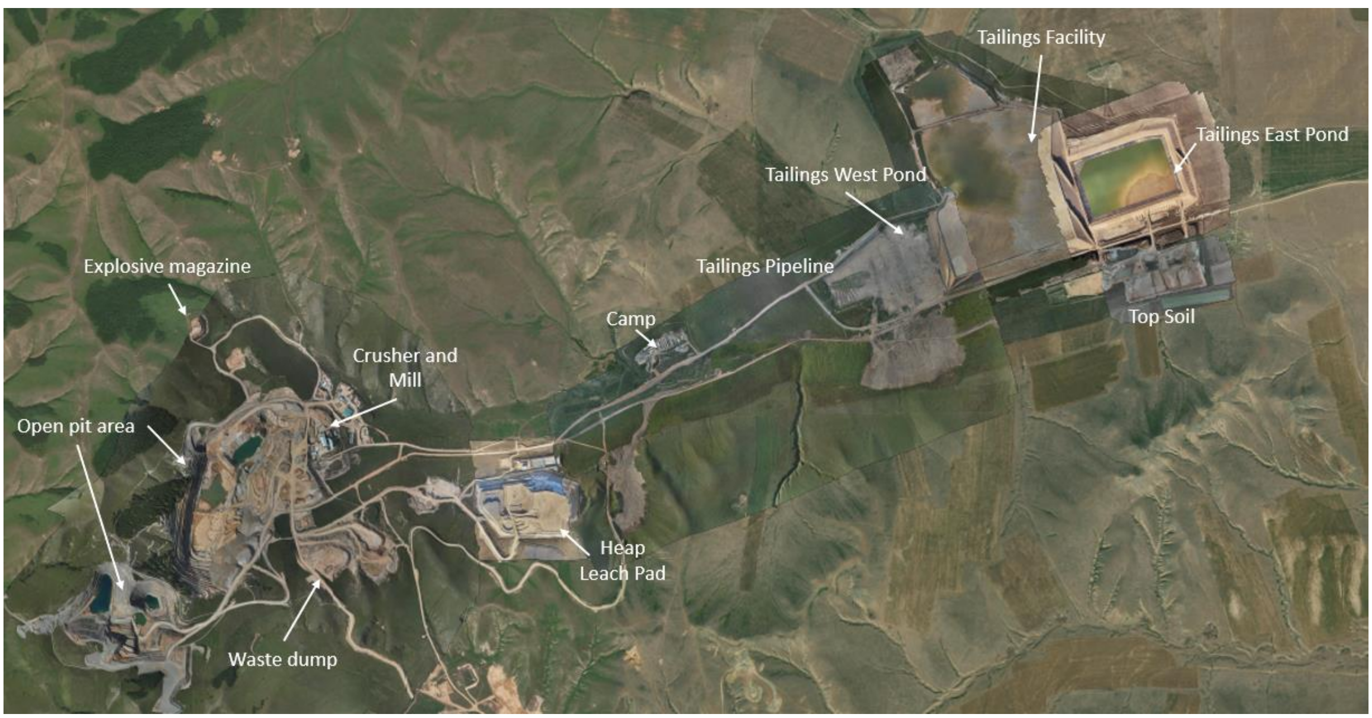


Figure 18-1: Boroo Gold site general map

18.1 Access roads

Access to the site is, in part, via the highway that connects Ulaanbaatar to Sükhbaatar (Sukhbaatar) near the north border with Russia which roughly takes one and a half hours. From the highway, there is a two-lane all-weather road through the general countryside which leads to the site, about 10 km from the highway.

Figure 18-2: Location of the Project relative to Mongolia

18.2 Off-site Transport

The Ulaanbulag Mine is located about 19 km, east of the Boroo Mine complex. Therefore, ore extracted from the mine will be transported to a Boroo plant of 40 tons of haul trucks. A totally 10 haul trucks will be operational during mining for off-site transport of ore.

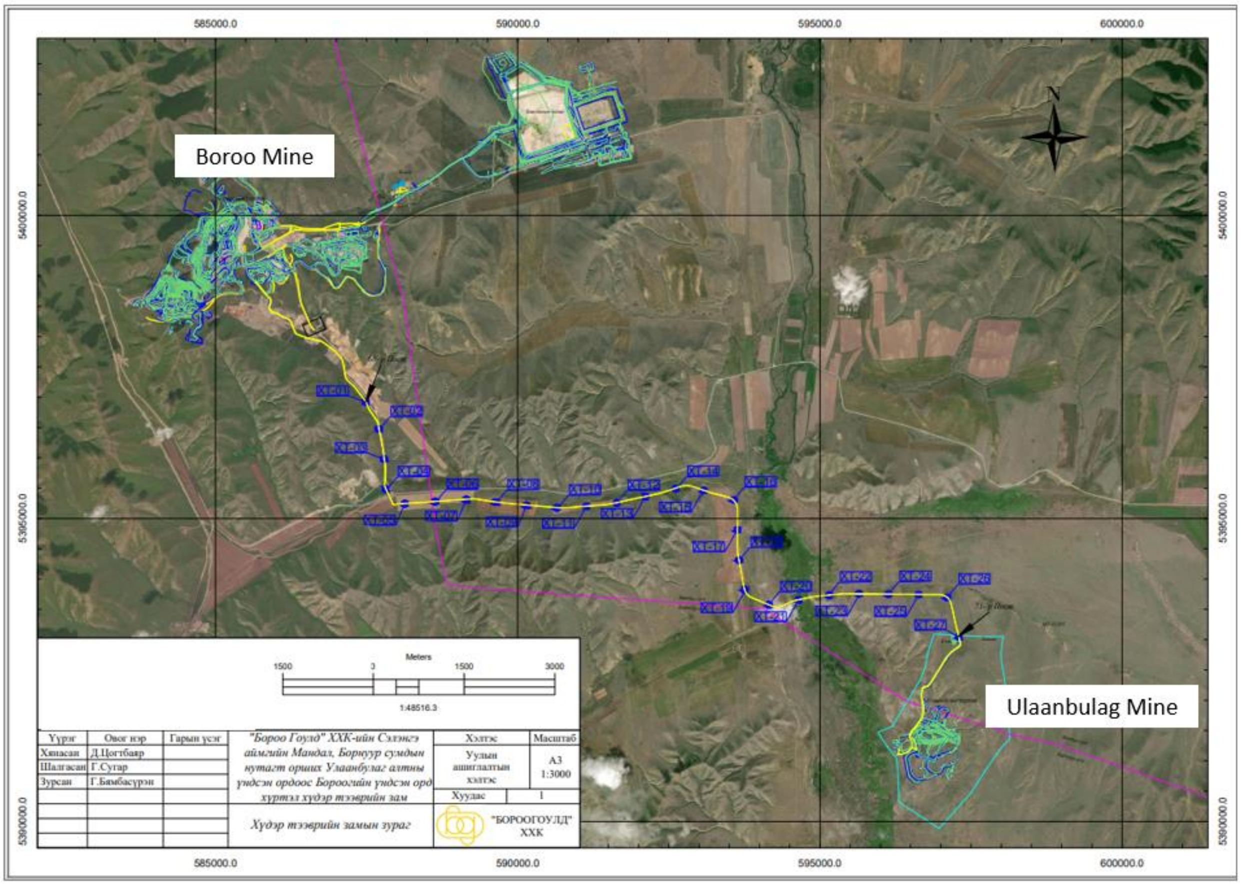


Figure 18-3: Off-site transport map for Ulaanbulag to Boroo Mine

18.3 Power Supply

The power demand of the existing Boroo facility is 10 MW. Power is currently supplied to the existing Boroo facility via a 110 kV electric power line that crosses the Sujigtei River Valley.

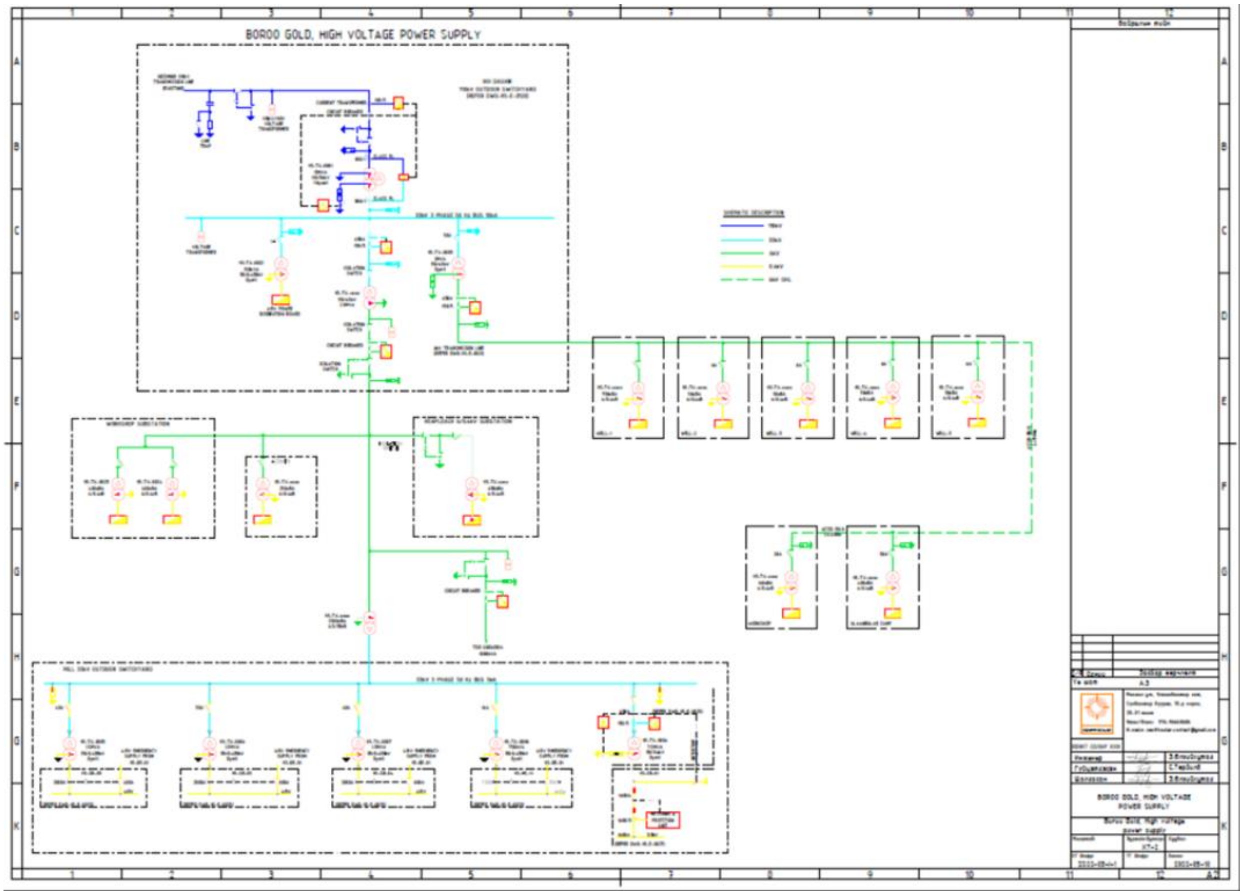


Figure 18-4: Boroo Gold High Voltage Power Supply Drawing

18.4 Process Plant Facilities

The existing Boroo Gold plant (currently on Care and Maintenance) employs a Leach/CIP and gravity concentration for gold recovery. The plant comprises crushing, grinding, gravity concentration, thickening, Leach and Adsorption, and cyanide detoxification steps as. Detoxified tailings are deposited into a zero discharge tailings management facility.

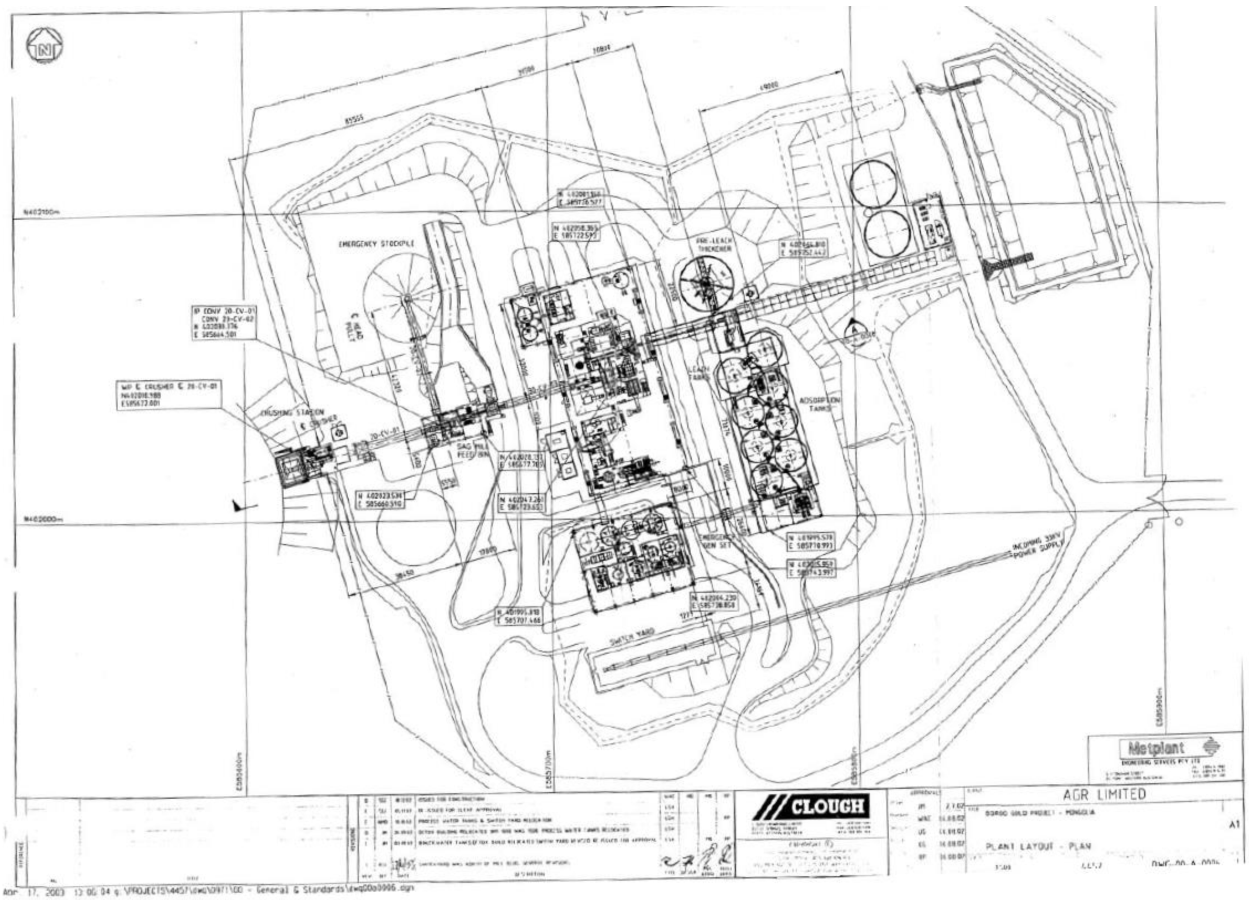


Figure 18-5: Existing Boroo Plant Layout

18.5 Site services

An existing potable water treatment plant will treat the fresh water prior to storage in the potable water storage tank. Sewage will be collected and chlorinated before disposal. Effluent from the sewage treatment plant will be discharged into the tailings facility at Boroo.

At the Boroo site, diesel and gasoline storage facilities will be provided at the mine services area. One diesel fuel storage tank and one gasoline storage tank will be utilized a fully. An existing a mine maintenance / operations administration building and a security gatehouse facility will be utilized at the mine site.

18.6 Accommodation

During the operating phase of the Project, Boroo operating personnel will be housed at the existing facilities at the Boroo Mine.

18.7 Tailing Management facility

The existing tailings management facility (TMF) of Boroo mine is located approximately 5 km to the east of the mine. The total capacity of the TMF is 21.2 million m3. The TMF consists of three parts: north, middle and main cell. The total area of the TMF is 205 hectares. The ore extracted from the Boroo mine and Ulaanbulag mine will be processed at the Boroo processing plant, and the tailings from the plant will be stored in the main cell and the east and west cells to be built in the future.

The original TMF design has been made by Golder Associates Ltd. (Golder) in 2003 and additional design updates were developed in 2004 and 2006. Usny Erchim Ltd. (UEC) completed the design for TMFs further extension in 2013 and the design study has been approved by Golder.

Figure 18-6: Overview of TMF relative to Site Infrastructure at Boroo Gold Project

East Cell

In 2022, the construction design for the East Cell was finalized by "Usny Erchim" LLC, based on the reserve estimation results and additional exploration conducted at the Ulaanbulag and Boroo mines. The East Cell will be situated adjacent to the currently operational main cell.

The approved design in 2022 enabled the successful completion of the first phase of the East Cell. The overall design consists of six stages, with a maximum elevation of 928 m. The East Cell has a total capacity of 12.0 million m3. Each phase of the East Cell is designed to store the tailings from CIL's for one year. The construction of the final phase is scheduled for completion by 2027.

West Cell

To accommodate the increased reserves resulting from further exploration at the Boroo, and Ulaanbulag deposits, the development of the West Cell is currently underway. The West Cell will be designed to store the tailings from the plant.

The initial calculations and design work for the construction of the West Cell are currently being carried out. This project is being undertaken by "Usny Erchim" LLC, an organization experienced in designing water structures, and authorized by relevant authorities. Additionally, the design and development of the West Cell will undergo a thorough evaluation and approval process by professional experts appointed by the state.

19.0 MARKET STUDIES AND CONTRACTS

Gold is a commodity that is freely traded on the world market and for which there is a steady demand from numerous buyers. It is also possible to sell gold for delivery at a fixed price at a future date (forward sale). There are a number of refiners in the world whose bars are accepted as "good delivery" through associations like the London Bullion Market Association (LBMA).

In Mongolia, the Government of Mongolia supports the purchase of gold through Mongolbank, the central bank. Sales of gold to Mongolbank have a fixed, discounted royalty, currently set at 5 percent under the Gold-2 Program. Mongolbank gold purchases are transacted according to the daily spot price on the London Metals Exchange. Exports of gold from Mongolia are subject to a graduated royalty scheme, ranging between 5 and 10 percent.

As a freely-traded commodity with clear framework for sales domestically in Mongolia, no marketing studies were considered necessary for the Boroo Gold.

19.1 Metal Selling Price

Game Mine confirmed that there were no periods of negative cash flow following project start-up and that overall project economics are favorable at a five-year moving average gold price of $1,750/oz.

19.2 Sales Contract

No sales contract is required for the sale of doré to the Mongolbank. The Company is responsible for transportation, insurance, laboratory, and other charges associated with delivering the doré to the point of sale. No precious metals refinery currently operates in Mongolia. Refinement of doré purchased by Mongolbank is managed directly by Mongolbank.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental and Social Setting

The Boroo project is located in the territories of Bayangol and Mandal counties (known as soum in Mongolia) of Selenge Province, the Ulaanbulag project is located in the territories of Bornuur counties in Tuv province, approximately 140 km to the northwest of Ulaanbaatar City, 8.5 km from the Ulaanbaatar-Darkhan-Selenge paved road, 19 km from Baruunkharaa, center of Bayangol soum, and 25 km from Zuunkharaa, center of Mandal soum, in the Ikh Dashir Valley, which is the floodplain of the Boroo River that runs between mountains Davkhar and Chandagatai in the territory of Bayangol and Mandal soums, Selenge Province, at the coordination 48° 44' 44.68649"N, 106° 10' 3.75935" E.

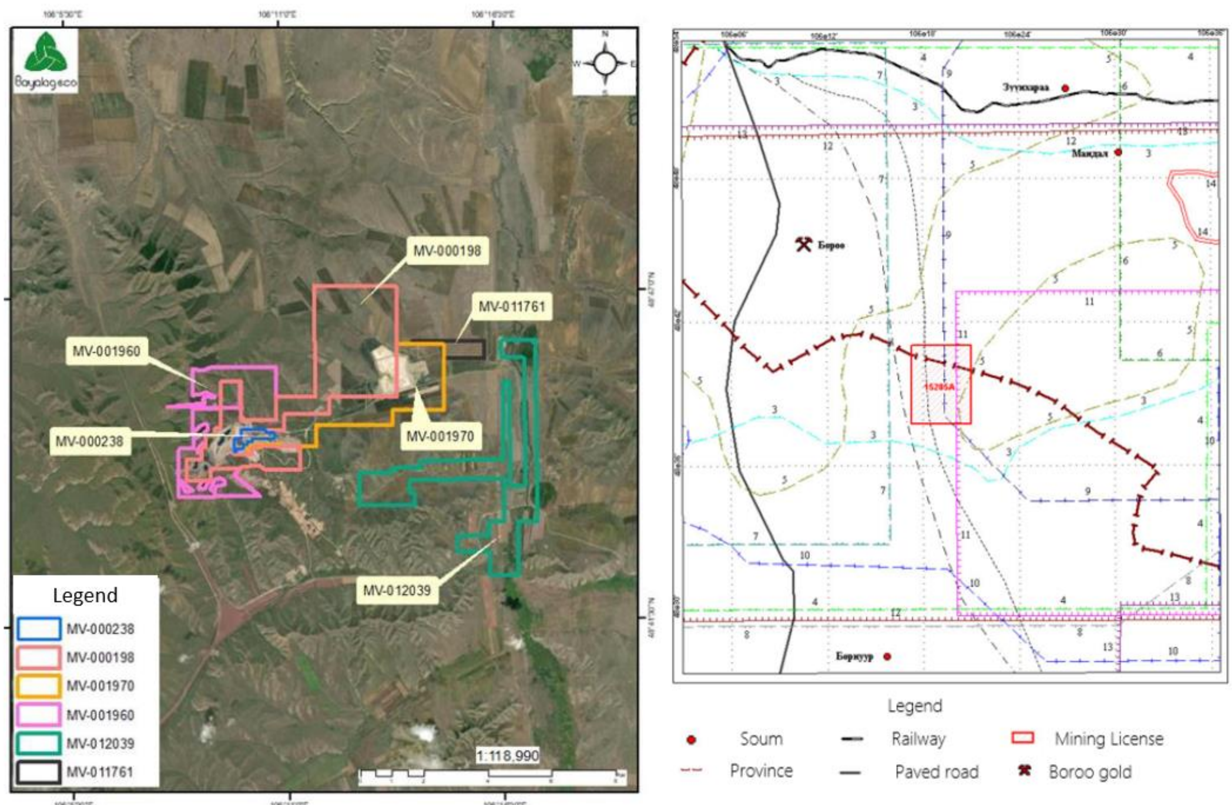


Figure 20-1: Project licences location

The Boroo gold deposit covers 3,602.07 ha in total under six mining licenses, and mining operation is conducted on area under mineral mining license MV-000198 with an area of 1,398.55 ha, while heap leach stockpile is on area under mining license MV-001970, and other infrastructure buildings and facilities are on areas under licenses MV-001960 and MV-000238.

The Ulaanbulag deposit covers 1,204.47 ha in total mining licenses, and mining operation is conducted on area under mineral mining license MV-15285.

The mine site is in an arid steppe forest zone that receives approximately 250 mm to 300 mm precipitation annually and has an average elevation of approximately 1,200 MASL. This land is described in the vegetation map of Mongolia as forb-sedge-grass steppe dominated by Poa with random birch-larch and birch-pine forests on mountain dark brown and mountain black soil. Three categories of landscape formations are observed in this region, i.e. tectonic-erosion, erosion-denudation and aggradation.

The area is rural, characterized by settlements located approximately 20 km to 30 km from the mine tenure. The predominant land use is transient herding.

Water source in the surroundings of the mine site is the Boroo River basin. Total length of Boroo River is 118.5 km. With regionalization of surface water, it is fed with spring flows from melted snow and summer flows from rain. Annual average run of variability is 0.30 to 0.40 and long term mean flow is less than 25 m/s. The Ikh Dashir Valley, where the Project is located, is the dry riverbed in the east of Boroo River. There is almost no surface water flow in the Ikh Dashir Valley and ground water is found in 40 m to 45 m depth, and soil water in the Boroo River Valley is found in 2.7 m to 3.2 m depth.

Rock distributed in the Boroo mine site are collections of the aquifers and fractures. However, it has distributed through modern and upper Quaternary perforated loose deposits; characteristics of fractured rocks and deposit thicknesses differ greatly. The Ikh Dashir Valley within the Boroo River Valley is covered with 0.5 m to 20 m thick deposit in the middle and upper parts and up to 50 m in some parts. Its percolation capacity is 0.00267-0.0000388 m/d, i.e. it has characteristics of impermeability. The ground water spread in the Ikh Dashir Valley is water of upper part of hydrogeological section and is recent age water fed by precipitation. Ground water resource is aggregated by precipitation. Ground water resource in the Ikh Dashir Valley is approximately 8.4 mm. Ground water in this area has indirect hydraulic connections with water of the Boroo River and supplies water flow in the river valley and beneath the riverbed.

The Project's water supply is provided by five drilled wells located at Boroo River valley and ore processing water use is 7,000 m3/day. Sixty to seventy percent of water used for plant operation is reused from tailings pond in summer. In 2010, hydrogeological study to re-assess groundwater reserve in Boroo River Valley used for water supply of BGC was conducted by Us Oyu LLC and was reviewed and approved by the Water Reserve Council. Ground water reserve of Boroo River Valley for BGC's water supply was accepted with A classification with 5,435 m3/day, B level with 4,165 m3/day, i.e. 9,600 m3/day or 400 m3/hr in total, and was registered in integrated registry of the 'National Water Database'.

By soil geological region, Boroo and Ikh Dashir gold deposit area belongs to the Khentiin outer rim of Khentii district of Khangai region and is covered by dark brown and mountain black soil. The soil has been divided into three types and eight categories: dark, black dirt and dark brown. Predominant soil is the dark brown soil, with thin to medium thickness.

Mine site territory is situated within boreal forested steppe zone according to vegetation regionalization of Mongolia. It is demonstrated on this vegetation map of Mongolia by patchy birch-larch and birch-pine forests with sedge- forbs and small shrubs on the mountain dark brown and dark kastanozem. Prior to mining operations this area was a pastureland dominated by grass-forb, mostly with Stipas. Twenty-five to thirty plant species are found within 100 m2 plot and the canopy cover is about 94% of which are comprised of 32% of Stipa krylovii. Dr. N.Manibazar, botanist, noted (2000) that there is very low probability for extremely rare and rare plants to be in the area of former mining operations. When main mining operations commenced in 2004, the plant cover in the mine site area was completely destroyed. However, the abandoned alluvial site and its surroundings, as well as the roads and areas disturbed due to exploration activities, have all been reclaimed concurrently with mining operations. Biological reclamation has been conducted each year at areas where mining operations are completed. Results of surveys on reclaimed areas note that local plants have been growing in addition to plants planted with reclamation. Seeds of perennials such as Agropyron, Brome grass and Couch are collected from the surrounding and other reclaimed areas within mining claims and are preserved and used for re-vegetation as the viability of native plant seeds is high. From 2010 to 2022, a vegetation and wildlife monitoring study was conducted and compared to DEIA of 1999. A total of 40 main plants currently growing at the reclaimed area, only 21 species that existed before the operation are registered. The consistency of 50% of endemic species from total vegetation community shows that the area becomes close to its natural condition.

Biodiversity study of 2022 was conducted in the areas of vegetation monitoring, soil microorganisms, insects, amphibians, birds and mammals in cooperation with professional organizations and researchers, and 161 species of insects, one species of amphibians, three species of reptiles, 84 species of birds and 17 species of mammals were recorded in the reclaimed areas of the Project. This fauna composition is a direct result of biological reclamation and is related to environmental management of BGC.

There are no special protected areas within and/or near the Project territory.

Mandal soum has the largest population of any soum in Mongolia at just under 27,000 people. The soum is a prime agricultural and cropping area in Mongolia. Furthermore, 70% of the soum is covered by forested land, which supports forestry. Similar to Mandal soum, Bayangol, Bornuur is a prime agricultural area, and is connected to Ulaanbaatar, Darkhan, Tuv and Selenge by a paved road, and by railway, on road and rail routes connecting Mongolia with Russia. Some of the key social issues in the area include the lack of vocational education opportunities, unemployment and underemployment, extensive land degradation due to overgrazing and poor agricultural practices, and inadequate availability of health care personnel.

20.2 Environmental Studies

A comprehensive environmental baseline study was prepared for Boroo in 2000 and 2003. The environmental monitoring program that was established with the 2003 baseline program has expanded since that date with the addition of more types of monitoring and the expansion of existing monitoring programs. This expanded monitoring program provides international level environmental information that facilitates operations. This monitoring data will continue to be collected for surface water, groundwater, air, soil, and meteorological data.

The main objective of the Environmental Management Plan of the given year is to plan, implement and report measures to protect the environment, avoid and prevent from main and potential impacts to be caused by project operations, eliminate and reduce impacts, control and detect potential impacts and reclaim.

A feasibility study on the Boroo Gold Mine was prepared in 1999 and its addendum reports were approved by Mongolian Mineral Council in 2013, 2020 and 2022. JEMR LLC has developed an environmental baseline study and Detailed Environmental Impact Assessment (DEIA) of the Boroo Gold mine in 2000. Changes in processing plant, renovation of equipment, extension and other changes in activities of the Boroo gold mine were clarified in DEIA and approved by Ministry of Environmental and Tourism (previously Ministry of Environment, Green Development and Tourism) on a timely basis. A brief introduction of these supplementary clarifications in DEIA is shown below.

- Gazar-Eco LLC, 2003 – detailed environmental assessment with supplementary clarifications is prepared on increased annual ore extraction, renovation of processing plant technology and installation of detoxification equipment.
- Gazar-Eco LLC, 2003 – detailed environmental assessment with supplementary clarifications is prepared on installation and use of equipment to filter and purify water discharged from employees' shower, washing room and restroom in order to reduce ground water use and eliminate risks on soil and water.
- Gazar-Eco LLC, 2004 – detailed environmental assessment with supplementary clarification is prepared on extension of tailings dam.
- Gazar-Eco LLC, 2004 – detailed environmental assessment with supplementary clarification is prepared on petroleum storage and distribution facility for project use.
- Gazar-Eco LLC, 2005 – detailed environmental assessment with supplementary clarification is prepared on increase of industrial production, use of new reagents, provide sustainable exploitation of equipment and pipelines, make changes in amount of extraction and extend duration of exploitation of water system that reuses water from tailings pond for technological purposes.
- Eco-Trade LLC, 2006 – detailed environmental assessment with supplementary clarification is prepared on operation of Pit 6 and operation of oxygen plant, as well as indicated draft mine closure plan.
- JEMR LLC, 2007 – detailed environmental assessment report is prepared on detailed environmental assessment of heap leach project extraction method for low-grade ore.
- Nemer International LLC, 2007 – detailed environmental assessment report is prepared for the Project on mining alluvial gold deposit of Ikh Dashir main valley.
- Nature Friendly LLC, 2010 - detailed environmental assessment with supplementary clarification is prepared on BIOX® bio-oxidation technology to oxidize sulfuric ore and cyanide leaching project.
- Nature Friendly LLC, 2011 – detailed environmental assessment is prepared on Boroo mine closure, reclamation plan, monitoring after closure with detailed indication of cost planned for closure, reclamation and other activities.

- Global Environ LLC, 2011 - supplementary clarification report of environmental impact caused by operation of Pit 6 and its waste rock dump that is planned to operate as indicated on BGC's mine operation plan.
- Global Environ LLC, 2011 – detailed environmental assessment with supplementary clarification is prepared on extension of tailings dam at the Project.
- Global Environ LLC, 2015 – addendum with detailed description of environmental impact caused by heap leach project.
- Natural Capital LLC, 2015 – addendum including additional activities of Boroo gold deposit in connection to update of the feasibility study.
- Ekhmongoliin Baigal LLC, 2019 – report of detailed environmental impact assessment for the project on open pit mining of Nergui valley, Ar Khundii and Ikh Dashir alluvial deposits located in Bayangol and Mandal soums of Selenge aimag.
- Ashid Ananda LLC, 2020 – report on DEIA Addendum of the Boroo gold deposit located in Bayangol and Mandal soums of Selenge aimag.
- Ashid Ananda LLC, 2020 – report on DEIA of the placer deposit between exploration lines 36-71-12 in lower part of the placer deposit of the main valley of Ikh Dashir.
- Bayalag Eco LLC, 2022 – report on DEIA Addendum of the Project to develop the Boroo gold deposit located in Bayangol and Mandal soums of Selenge aimag with open pit mining.
- Bayalag Eco LLC, 2022 – report on DEIA Addendum of the Project to develop the Boroo gold deposit located in Bayangol and Mandal soums of Selenge aimag with open pit mining.
- Bayalag Eco LLC, 2023 – Environmental management plan of the Project to develop the Ulaanbulag deposit located in Bornuur soums of Tuv aimag with open pit mining.
- Bayalag Eco LLC, 2023 – detailed environmental assessment with to develop the Ulaanbulag deposit located in Bornuur soums of Tuv aimag with open pit mining.

20.3 Project Permitting

BGC has secured many of the operating permits required to operate the Project in accordance with Mongolian regulatory requirements. Approved permits and licences include Land Use Permit for the Boroo Mine, Boroo Mineral Reserve Report Approval (2021), Boroo Mining Licences, Boroo Feasibility Study (2014), Boroo Detailed Environmental Impact Assessment (DEIA) (2022), Water Reserve Approval (2010), Hazardous Chemicals Permit (2020) Land use Permit (2007), construction work permit for the TMF extension (2022) and Water use Permit.

The status of current permits and approvals is summarized in Table 20-1.

Table 20-1: Summary of Major Permits and Approvals

Approvals and Permits	Project Activity	Statutory Basis	Approval / Permit Status (May 2023)
Company Certificate	Boroo Gold LLC	State registration agency	Established: May 05 1997 Registration No. 2094533
Mining License	MV-000198 MV-000238 MV-001960 MV-001970 MV-011761 MV-012039	Mongolian Law on Licensing	March 16 2007 February 09 2007 February 09 2007
Land Use / Possessions Certificates	Boroo Gold LLC	Mongolian Law of Land	Approved.
Boroo Mine Reserve Approval	Boroo mine	Mongolian Law of Minerals Mongolian Law of Subsoil	Obtained: September 20, 2021 No. CTP-02-02; No. н/111
Boroo Mine Feasibility Study	MV-000198 MV-000238 MV-001960 MV-001970 MV-011761 MV-012039	Mongolian Law of Minerals	Obtained: March 03, 2022
Boroo Gold Mine Detailed EIA	Boroo gold hard rock mine	Mongolian law on Environmental impact assessment	Obtained: May 2022
Hazardous Chemicals Permit	Boroo Gold LLC	Law on Hazardous chemicals	No. 0001727 No. 0001728 Obtained: October 09 2020
Historical and Cultural Heritage Exploration and Survey	Boroo Gold LLC MV-000198, MV-000238, MV-001960, MV-001970 license areas	Mongolian law of Cultural heritage conservation / 27.8 and 6.1.5 / Minerals law of Mongolia / 40.2 /	Surveys undertaken in accordance with legislation. No sites found within Project operational areas. Professional conclusion obtained in 2013.
Water Reserve Approval	Extraction water from Boroo forefields	Law on water	Approved in 2010.
Water Use Permit	Boroo Gold LLC MV-000198, MV-001970	Mongolian law of water / 28.4 /	Obtained: MV-000198 – May 17 2022 MV-001970 – May 16 2022
Permit for Hazardous wastes' Temporary Storage	Boroo Gold LLC	Waste law of Mongolia / 23.2 /	Obtained: April 13. 2023

Approvals and Permits	Project Activity	Statutory Basis	Approval / Permit Status (May 2023)
Boroo Mine Plan 2023	Boroo Gold LLC MV-000198	Minerals Law	Obtained: December 31, 2022
Annual Environmental Management Plan	Boroo Gold LLC MV-000198	Mongolian law on Environmental impact assessment	Approved in March, 2023
Boroo gold Mine Detailed EIA	Boroo Gold LLC MV-015285	Mongolian law on Environmental impact assessment	Obtained: 2023
Ulaanbulag Mine Environmental Management Plan	Boroo Gold LLC MV-015285	Mongolian law on Environmental impact assessment	Obtained: 2023

Other regular compliance and permitting documents including workplace fire safety conclusions and disaster risk assessments for chemicals warehouse, kitchen and camp for, petroleum warehouse, fuel distribution station, processing plant, heap leach facility, and maintenance shop etc., technical passports of drilled wells MB1, MB2, MB3, MB4, MB5, domestic and hazardous waste removal and transportation agreements with licensed recycling and elimination organizations, License on business activities to export, import, transport across the border, produce, use and sell toxic and hazardous chemicals, bi-annual environmental audits by licensed auditing organizations, certification of land characteristics and quality, conclusions of certification plots by professional research body, Cooperation agreements with the Bayangol and Mandal soums of Selenge provinces and the Bornuur soums of Tuv provinces were obtained or approved as required.

20.4 Social of Community Requirements

20.4.1 Demography

As of the end of 2023, the population of Mandal soum is 26,877. According to demographics, male population is 50.04%, while there are 8,154 children of 0 to 14 years of age, 16,572 persons of 15 to 59 years of age and 2,151 persons above 60, 7.8% of the total population.

As of end of 2023, the soum population was 5,702 in 1,683 households, including 219 disabled citizens, 3,470 citizens of labor age and 5 orphans.

Mandal soum, Selenge aimag has major national routes such as roads and railways and has a relatively well-developed infrastructure. Located close to the capital and other central markets, this is likely to increase in the future and create favorable conditions for future development.

Statistically, migration is declining in the Project area, although there is a perception among communities in the Project area that there is an influx of migrant herders who are viewed as a key source of pastureland degradation and land use conflict.

20.4.2 Social Structures

The levels of trust in public institutions in the Project area are generally low due to community dissatisfaction with the activities of public organizations, especially political parties. Community perceptions on corruption are also a key source of discontent with the political situation in both Mandal and Bayangol. In both soums, the soum Citizens Representative Khural (CRK) and soum Government have the greatest overall levels of trust among public organizations. Baghs face unique challenges in the Project area, including the perception among bagh citizens that their voice does not count or is not heard in decision-making.

20.4.3 Economy

Steep price inflation of goods and services is the major concern for households in the Project area, as are the poor quality of goods. Unemployment is also a problem in both soums. Moreover, livestock insurance coverage is low in the Project area, driven by a general lack of awareness of insurance (in the case of zud, fire, drought, or natural disaster).

20.4.4 Social Infrastructure

Kindergarten facilities are stretched to over-capacity in both Mandal and Bayangol, Bornuur, with children being turned away. There are limited opportunities in the Project area to access tertiary and vocational education and the facilities that do exist in Mandal soum provide vocational training for a predominantly male student population in traditionally male professions. Households in the Project area primarily receive information, news, and entertainment from social media and television. Vehicles and the railway are the most common forms of public transport, with rail transport the most popular. The construction of the Project haul road has enhanced opportunities for local small businesses.

The main sources of drinking water in the Project area are wells, rivers and springs, and mobile distribution points. Bayangol sources more water from unprotected wells than Mandal soum. Most households residing in soum centers consume energy from the central energy system, while 18.5% of surveyed households in Mandal soum and 28.8% of surveyed households in Bayangol soum use renewable energy. Solar systems are prevalent across all surveyed areas, in particular, for herders and those in Bayangol. 25% of surveyed households in Mandal soum also reported that their household energy resource is based on the local diesel system, which only works in the winter months between October and April. Heat supply is an essential basic service in Mongolia, but it is not accessible to everyone, in particular, people in rural areas. The current heating system in soums mainly consists of (i) small stoves; and (ii) centralized and decentralized coal fired boilers. Most household heating is independent of the centralized grid.

20.4.5 Land Use and Natural Resources

Overgrazing and the accompanying pastureland degradation is a central issue in the Bayangol and Mandal soums, and the Bornuur due to existing herders and livestock owners having increased the number of animals in recent years. Overgrazing problems have resulted in land use disputes among herders in some areas, and between herders and crop farmers. The primary reason behind overgrazing and the accompanying pastureland degradation is that existing herders and livestock owners have generally increased the number of animals they own.

The social baseline survey found that 60% of Mandal soum households were involved in Artisanal and Small-scale Mining (ASM) — to supplement their incomes/livelihoods, as a hobby, or as a poverty reduction strategy. Around 228 soum residents were members of the 30 officially registered ASM cooperatives (three large NGOs, with around 30 to 100 members and 27 small cooperatives with 3 to 10 members).

20.4.6 Community Health, Safety and Security

There is a lack of healthcare personnel (doctors and nurses) in the Project area of influence. Specifically, Mandal soum has a significant shortage of doctors, but a relatively better supply of nurses, while Bayangol soum lags behind both World Health Organization (WHO) indicators for doctors and the national average for nurses. Key problems households are experiencing with health services include the poor quality of health care; no time to go to the hospital; and, that hospitals are too far away.

Indicative traffic surveying shows most vehicles travelling along the public haul road of Zuunkharaa are light vehicles. The main causes of traffic accidents in the Project area are related to risky driving practices. Moreover, police statistics indicate that the major crimes in Selenge Aimag are related to traffic safety and the use of motor vehicles. Just under half of crimes registered in Mandal soum are crimes related to human life and health, while a third of all crime in Bayangol soum is related to traffic safety and the use of motor vehicles, which may be explained by the location of the soum along the international, paved road between Ulaanbaatar and the Russian border. Most households surveyed at the soum, bagh and household levels rate local crime-fighting capacity as “average and insufficient". Child labour was also reported by over 30% of respondents as a key issue in Bayangol soum, compared to slightly over 20% in Mandal soum. Child labour most often takes the forms of herding, artisanal mining or working for forestry companies.

20.4.7 Cultural Heritage

Heritage studies were commissioned by BGC to ensure compliance with the applicable Mongolian regulations, including the Constitution of Mongolia and the Procedure for the Research on Cultural Heritages of Mongolia.

Under a contract concluded by and between History and Archaeology Institute of the Academy of Sciences of Mongolia and BGC, Archaeology Institute conducted an archaeological study at licensed properties under mining licenses MV-000198, MV-000238, MV-001960, MV-001970, MV-011761, MV-012039 of the Project in 2013. As a result of the study, a Xiongnu manufacturing settlement dated back to 1 A.D. was discovered in the south of MV-012039 licensed area. This "Boroo settlement" of Xiongnu era is located in the Boroo River Valley. No artefact was discovered in other areas.

In 2014, BGC in cooperation with the History and Archaeology Institute of the Academy of Sciences of Mongolia approved the "regulation on movable and immovable historical and cultural heritage" with the purpose to ensure the found artefacts are not damaged, resolve the matter according to applicable laws and regulations, and reduce potential risks for archaeological and paleontological artefacts in the event the artefacts are found accidentally.

20.5 Environment and Social Management Plans

Management of significant Project environmental and social aspects and impacts is through a suite of Management Plans that will be implemented during the project activities. The following Environmental and Social Management Plans have been developed:

20.5.1 Environmental

- Biodiversity Management Plan
- Water Management Plan
- Acid Rock Drainage Management Plan
- Hazardous Materials Management Plan

20.5.2 Social

- Stakeholder Engagement and Grievance Management
- Social Management Plan

20.6 Mine Closure

20.6.1 Mine Closure Requirements

Long-term planning for mine closure is framed by regulatory requirements, stakeholder expectations and standards of social and environmental responsibility.

During the Feasibility Study and DEIA development stages, a preliminary Closure Management Plan is developed and approved with these documents. Therefore, the mine closure general plan is updated and included in the DEIA and its addendums. Mine closure plan's objectives, required measures and activities, such as rehabilitation of the mine site; cleaning, demolition and relocation of mine site facilities, equipment and pipelines; post-closure and rehabilitation monitoring; reporting and its approval issues; and official handover of the mine site to local and state government organizations etc. were reflected in the currently valid Boroo DEIA.

Regarding the development, approval, implementation and monitoring of the closure plans of mines, no procedures, specific regulations or instructions have been issued by the Mongolian authorities until August 28, 2019.

Following provisions demonstrate legal requirements that are critical to the development and approval of the mine closure and reclamation plan:

According to Mongolian Law on Impact Assessment (Revised on May 17, 2012), Clause 14.1.3, "the project implementer shall prepare a reclamation and closure management plan and get the opinion or recommendation by the related state central administrative body and submit the plan to the state central administrative body in charge of nature and environment prior to not less than 3 years of closure start"

Provision 10.1.14 of the Minerals Law, as amended on July 1, 2014, which states: "Regulations for reclamation and closure of mines and process plants shall be approved collectively with the state central administrative agency in charge of environmental and mining matters".

In compliance with this provision, the Ministry of Mining discussed and prepared the mine reclamation and closure regulations in cooperation with the Ministry of Environment, Green Development and Tourism. The regulation on Reclamation and closure of mines and process plants was passed and is effective as of August 28, 2019. This is the first set of closure regulations in Mongolia.

In accordance with these regulations, a detailed mine closure management plan will be developed in 2031 and approved separately from the DEIA prior to the mine closure's commencement.

The Project is assumed to achieve the closure objective in approximately seven years. This will include four years of active closure construction and reclamation activities, followed by a minimum of three years of post-closure monitoring.

Consistent with objectives for environmental and social responsibility, BGC intends to implement an environmentally and technically sound closure plan for the Project. The ultimate goal is to return the land to a condition that provides minimal risk to humans and livestock, as the lands subject to reclamation will be returned to pastures, providing a land use compatible with the surrounding hills and valleys.

The closure objectives include the following best practices:

- Develop a post-mining landform that is stable, where vegetation approximates the existing surroundings over the intermediate and long-term and soil loss is comparable.
- Route the non-contact surface water around the planned pits and dumps.
- Public health and safety.
- Chemical and geo-chemical stability.
- Reduce the flow of contact water from the waste rock dumps.

Detailed closure activities by major facility are detailed below.

20.6.2 Closure Schedule

BGC is currently undergoing additional investigations to define the exact timing of the closure schedule according to the most recent mine and mill schedule. In the interim, the closure schedule from the latest Feasibility Study should be referenced as illustrated in Table 20-2 below. Reclamation activities are planned to last seven years, including four years of active reclamation including the heap leach draindown and detoxification and three years of post-closure water management and monitoring as shown in Table 20-2. Closure of the tailing facility includes construction of an evapotranspiration pond and treating supernatant water.

Table 20-2: Closure Schedule

Schedule of Operations	2032	2033	2034	2035	2036	2037	2038
Heap Leach solution recirculating 2032							
Drain down and detoxification 2032 - 2033							
Closure of PIT							
Reclamation of Mill, Heap Leach, Tailings facility and other infrastructures							
Post closure monitoring							

20.6.3 Planned Closure Activities

Main facilities at the Project that have not been reclaimed include open pits, heap leach (HL), TMF and other buildings & infrastructures. During the closure stage, the facilities, buildings and infrastructures will be removed and the areas will be cleaned, re-contoured and re-sloped to a suitable degree for revegetation and in a manner resistant to the soil erosion.

20.6.4 Pits

The primary concerns for closure of the pits are:

- Access to the pits
- In-pit water quality
- Physical stability of the pit walls

Access roads into the pits will be blocked and a berm, with fencing, will be constructed around the pit perimeters to discourage access and warn of danger. No other reclamation activities are planned for pit closure. BGC plans to follow the internationally accepted approach by securing the open pit perimeters and not pursuing backfilling the pits. Treatment of the pit water in the existing Boroo detoxification facility and store the treated water in the tailings dam are planned.

20.6.5 Waste Rock Dumps

The current dump configuration will allow for all dump slopes to be regraded to a final slope of approximately 2.5:1 (horizontal to vertical). The top surfaces should be graded to route stormwater off the dumps as quickly as possible to limit infiltration into the dumps.

All waste rock dumps will be capped with 200 mm to 300 mm of GM and revegetated. Yard areas and ROM stockpiles will be regraded to promote drainage, scarified, and capped with at least 150 mm of GM, and revegetated.

20.6.6 Heap Leach Facility

Closure of the HL facility will be performed as per specified in the national standard MNS 6297: 2011 on closure of heap and dump leaching facilities. In this standard, wastewater from the HL facility is required to meet MNS 4943:2015 standard on effluent water quality.

20.6.6.1 Draindown and Detoxification

Recirculation of heap leach solution is planned 2032. Then draindown and detoxification of the residual solution (draindown water) will start 2032 and continue in 2033. The draindown water will be collected and delivered to the existing Boroo detoxification facility. There it will be treated to the MNS 4343 : 2015 wastewater standard criteria and later the treated water will be discharged into the tailings dam.

20.6.6.2 Physical Stability and Recontouring

Once draindown and detoxification have been completed, the external piping supporting the drip irrigation system will be removed, and the outer slopes of the heap re-graded to allow cover placement and revegetation, and to minimize erosion. Below option is considered:

- Re-grade to 2.5H:1V (22 degrees), pushing from the heap crests, out beyond the lined area.

20.6.6.3 Heap Material Geochemistry

In order to prevent from infiltration of meteoric water and causing long term seepage issues, the heap will be encapsulated by 1 m thick, impermeable clay material which will be hauled from nearby TMF area. On top of this, the clay layer will be overlain by 20 cm topsoil that will be taken from the existing topsoil stockpiles on site.

20.6.6.4 Revegetation, Erosion and Dust

With 1 m thick clay layer, no long-term chemical issues are anticipated with the heap materials. The current plan is to cover the outer heap slopes and top surface with a 20 cm growth media/topsoil. This will function as both a physical barrier to isolate the spent ore from the elements and to provide a suitable growth medium for establishment of vegetation. Various suitable seed mixtures that will assist in controlling dust from the heap surface.

20.6.7 Boroo Tailings Facility

The closure of the Boroo TMF cell will require construction of an evapotranspiration (ET) cell to collect and manage drainage from the tailings facility. This may be achieved by enhanced evaporation. Alternately, the water may be treated and discharged.

Once the TMF supernatant and initial drainage are removed, a 1.5 m rock cover will be placed on the surface. This will allow mobile equipment onto the facility and provide a foundation for placement of a soil cover between 0.5 and 1.0 m in thickness. This cover will minimize long-term infiltration into the facility.

Long-term drainage will be managed through both active and passive evaporation.

Reclamation of North, South, East and Wing dams will be conducted in the following order:

- Upper crest 50 cm, slope 1:4
- Rock fill 30 cm
- Topsoil 15 cm
- Reclamation of west dam will be same as above, but rock is not required.

Revegetation will be conducted according to the MNS 5918:2008 Standards and seed perennial plants.

20.6.8 Estimated Closure Costs

Closure costs have been estimated at $16 M for the Project and will be incurred after mining is complete.

The closure costs are updated every three years to reflect the latest changes in costs, currency exchange rates, inflation, and some changes in the legal requirements

21.0 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimates

Since the Boroo Gold mine is operational, no additional investment will be made to increase the capacity, so no investment calculation has been completed. On the other hand, the calculations made by the company for the maintenance of mining equipment, processing plant, mine accommodation and other mine assets are presented in the sustaining capex section.

21.2 Sustaining Capital Cost Estimates

Sustaining capital costs include all costs from 2024 of the operation to maintain mining until the end of the planned LOM in 2031. Total sustaining costs of $68.4M have been estimated for the Boroo Gold mine.

Table 21-1: Sustaining Capital Cost Summary

Main Area		Cost (US$'000)
1000	Geology	2,603
2000	Mine	25,124
3000	Process Plant	18,719
4000	Project Services	-
5000	Project Infrastructure	-
6000	Permanent Accommodation	-
7000	Site Establishment & Early Works	-
8000	Management, Engineering, EPCM Services	737
9000	Tailings Storage Facility	21,211
Total		68,394

The following items are excluded from the sustaining capital cost estimate:

- Financing costs
- Currency fluctuations
- Lost time due to severe weather conditions
- Lost time due to force majeure
- Additional costs for accelerated or decelerated deliveries of equipment, materials, or services resultant from a change in project schedule
- Additional months of warehouse inventories other than those supplied in the estimate
- Any project sunk costs (studies, exploration programs, etc.)
- Mine reclamation costs (included in the financial model)
- Mine closure costs (included in the financial model)
- Escalation costs
- Major scope changes
- Cost of future study
- Community relations

21.2.1 Mining

Boroo Gold purchased CAT395 excavator x 1 and CAT773 dump trucks x 4 for US$ 7.3M in January of 2024 to add the current mining fleet capacity. This is considered sustaining capex for the mine, and a total of US$ 17.7M is planned for mining equipment maintenance costs over the years to ensure smooth mining operations. A summary breakdown of the mine production equipment and maintenance cost is shown in Table 21-2.

Table 21-2: Mining Production Equipment Total

Sub Area		Cost (US$'000)
2210	Mining Equipment	7,377
2215	Mining Maintenance	17,746
Total		25,124

21.2.2 Processing

Sustaining capital included for the processing plant consists of the following:

- Processing plant and mill maintenance (US$ 13.7M)
- Leaching pad with 11Mt capacity from 2027 (US$ 5.0M)

21.2.3 Tailings Storage Facility

The TSF cost estimate includes the TSF expansion over the project's life and the TSF mobile equipment costs. The costs include clearing and grubbing, topsoil removal, dam construction, dam liner, extension of the underdrains, and developing surface water management infrastructure for the TSF. The total sustaining capital for the development of the TSF is US$ 21.2M.

21.3 Operating Cost Estimates

Operating costs are defined as the direct operating costs, including mining, ore transportation, processing, G&A costs and mining royalties. As described in Section 21.2 above, sustaining capital costs are excluded from the operating cost estimate. All costs are reported in US dollars unless otherwise stated. The operational costs are based on Boroo Gold's historical and 2024 budgeted costs.

The LOM average operating cost for the Project is estimated at US$29.3/t or US$940/oz processed at the nominal process rate of 5000 t/d or 1.7 Mt/year. This operating cost is derived using total LOM operating costs divided by LOM processed tonnages and excludes pre-production mining costs. The cost distribution for each area of the proposed mining operation is summarized in Table 21-3 and Figure 21-1 below.

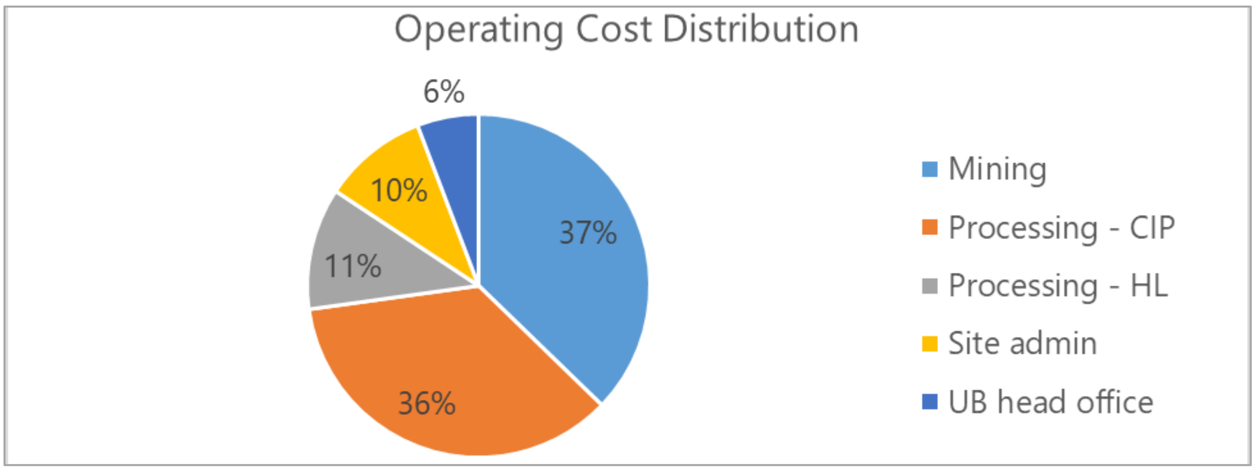
Table 21-3: Operating Cost Distribution by Operating Area

Operating Cost Summary	Total Cost M$ (LOM)	Cost $/t Processed (LOM Average)
Mining Cost	150.9	10.9
Processing Cost	190.8	13.8
Site Admin Cost	40.1	2.9
G&A Cost	23.6	1.7
Total Operating Cost	405.4	29.3

Note: Rounding may cause some computational discrepancies

Boroo Gold mine site is connected to the central power supply, and the CIP plant has access to electricity rather than a diesel generator, so diesel fuel consumption is low.

Figure 21-1: Operating Cost Distribution Pie Chart



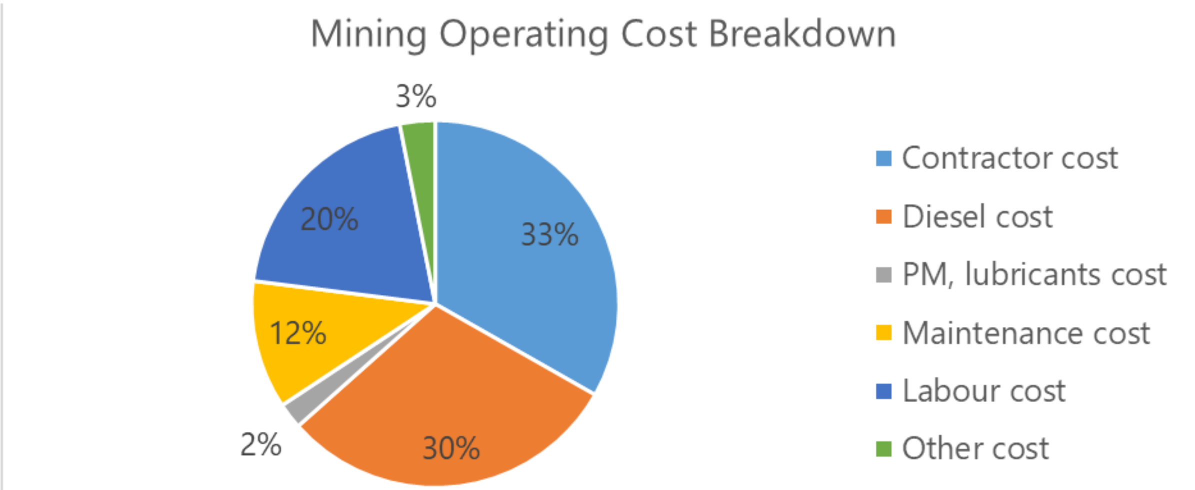
21.3.1 Mining

Mining operating costs were calculated based on the mining schedule using the unit costs estimated by the BG 2024 business plan. The cost breakdown by activity is shown in Table 21-4, and the cost breakdown by cost type is shown in Figure 21-2.

Table 21-4: Mining Cost Breakdown for Ore and Waste per Tonne Mined

Operating Cost Type	LOM costs US$ Millions	Average US$/tonne ROM
Drilling	18.7	0.15
Blasting	30.4	0.25
Loading equipment	22.0	0.18
Hauling equipment	45.2	0.37
Support equipment	23.2	0.19
Mine site admin	10.0	0.08
Other	1.5	0.01
Total costs	150.9	1.24

Figure 21-2: Mining Operating Cost Breakdown



21.3.2 Processing

Processing costs were calculated separately as CIP plant and heap leaching (HL) costs. The cost of transporting ore from the Ulaanbulag mine to the CIP plant and leaching site, as well as the costs of crushing and milling, are also included in the processing costs. The cost calculation for each area is presented in the Table 21-5 and Table 21-6.

Table 21-5: Processing Cost Summary (CIP)

Operating Cost Type	LOM costs US$ Millions	Average US$/tonne milled
Off-site transportation (UB)	3.7	0.27
Crushing	10.2	0.74
Milling	38.2	2.76
CIP cost	36.4	2.63
Processing admin	8.4	0.60
Other plant-related cost	47.4	3.43
Total costs	144.2	10.44

Table 21-6: Processing Cost Summary (heap leaching)

Operating Cost Type	LOM costs US$ Millions	Average US$/tonne milled
Off-site transportation (UB)	6.8	0.41
Crushing	13.2	0.79
Leach pad haulage	9.4	0.56
Leaching cost	13.0	0.78
Processing admin	4.2	0.25
Total costs	46.6	2.79

A breakdown of costs is provided in Figures 21-3 and Figure 21-4 below and is presented in units US$ per tonne of ore fed to the process plant.

Figure 21-3: Processing Cost Breakdown (CIP)

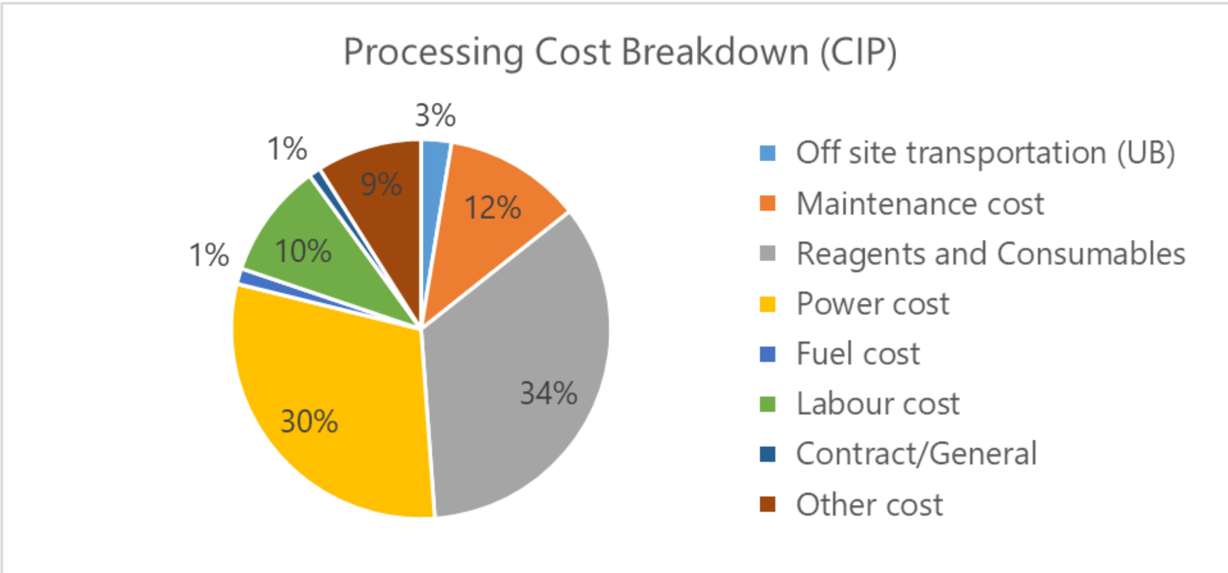
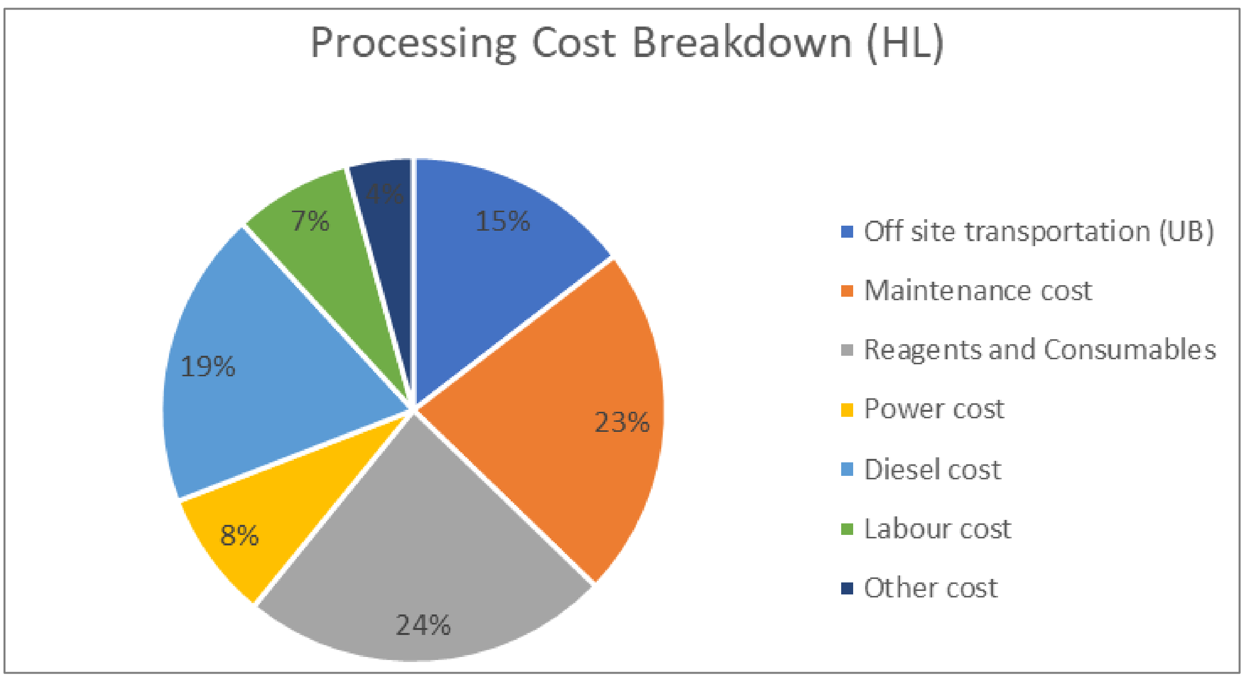


Figure 21-4: Processing Cost Breakdown (Heap Leaching)



21.3.3 General and Administrative Costs

The company provided a plan for the mine site admin cost and the general and administrative costs of the Ulaanbaatar office from 2024 to 2026, which was calculated as an average over the project period. The total cost is $63.7M for eight years, including US$ 40.3M for site admin costs and US$23.6M for UB head office costs.

21.3.4 Human Resources

The Boroo operation employs a total of 540 permanent and temporary employees as at January 1, 2024, distributed by department as shown in below.

Table 21-7: Boroo Employees by department

Department	Permanent	Temporary
Mine	176	29
Mill	53	13
Maintenance	115	5
UB administration	10	2
Site administration	11	1
Functional (legal, finance, etc.)	115	10
Total	**480**	**60**

The Mill employees include both mill and heap leach personnel.

21.4 Reclamation and Closure Costs

The company has included in its financial statements a total of $12.1 million in Asset Retirement Obligation (ARO) for the Boroo and Ulaanbulag pipeline closure plans by the end of 2023. According to this report, the duration of the project will be extended by 4-5 years, and the cost of mine closure is expected to be incurred from 2031, so the booked liabilities of ARO will increase by 4% annual inflation rate, and the fair value is calculated to be $16.0 million.

22.0 ECONOMIC ANALYSIS

22.1 Introduction

Game Mine prepared an economic evaluation of the Boroo Gold project based on a free cash flow financial model created for this technical report. For the 8-year total life-of-mine (from 2024 to 2031) and 13.8M tonne CIP ore and 16.7M tonne Heap Leach ore Mineral Reserve, the following pre-tax and post-tax financial parameters were estimated using the base case parameters (Table 22.1):

- Pre-tax NPV at 5% discount rate - US$191.1M
- Post-tax NPV at a 5% discount rate - US$151.7M

Sensitivity analyses and additional metal price scenarios were completed to assess the project economics, the details of which can be found in Section 22.4 below.

22.2 Assumptions and Parameters

The economic model included in this technical report was based on the following key assumptions:

- No additional investment is needed except sustaining capital expenditures
- Royalties of the Mongolian Government are fixed for the duration of the life of the mine at 5% of total project revenue.
- Cost estimates included in the financial model are based on the company's 2024 budget, which the Boroo Gold company provided.

Table 22-1: Parameters Used in Financial Model

Parameter	Unit	Value / average
Gold Price	US$/oz	1,750
MNT/US Exchange Rate	MNT:USD	3,450:1
Production Start	Date	Operating mine
Mining cost	US$/t mined	1.24
Off-site transportation cost (Ulaanbulag)	US$/t	1.70
Processing cost -CIP	US$/t milled	10.17
Processing cost – Heap leach	US$/t milled	2.38
Site admin cost	US$/t milled	2.90
G&A cost (UB office)	US$ Million/annual	2.90
Recovery rate – CIP	%	68.5
Recovery rate – Heap leach	%	40.0
Mining royalty for gold	%	5
Corporate income tax	%	10/25

22.3 Economic Analysis Results

The production schedule set out in Section 16.2.3 has been incorporated into the financial model to develop annual recovered metal production from tonnes processed, head grades and estimated recoveries. Revenues for gold produced are calculated based on the price shown in Table 22-1 above.

Operating costs for mining, processing, site admin and off-site charges were deducted from the revenue to derive the annual operating cash flow. This operational cash flow, sustaining capital, closure and reclamation costs were deducted to determine the net pre-tax cash flow. Sustaining capital is included to cover expenditures for mining equipment and processing plant repair and replacement.

Sustaining capital expenditure and mine closure cost included within the financial model total US$68.4M and US$16.0M, respectively.

Table 22-2: Project Economics

Financial results	Units	LOM	$/tonne milled	US$/oz sold
Tonnes CIP Processed	tonnes	13.9		
Gold Head Grade	g/t	1.20		
Tonnes Heap Leach Processed	tonnes	16.7		
Gold Head Grade	g/t	0.3		
Life of Mine	Year	8		
Dore production				
Gold Ounces Produced	Koz	431.3		
Total Project Revenue	US$ Million	754.7	54.6	1,750
Operating Costs	US$ Million	(405.4)	(29.3)	(940)
Royalties	US$ Million	(37.7)	(2.7)	(88)
EBITDA	US$ Million	311.6	22.6	722
Sustaining Capex	US$ Million	(68.4)	(5.0)	(159)
Closure cost	US$ Million	(16.0)	(1.2)	(37)
Pre-tax Free Cash Flow	US$ Million	227.2	16.4	527
Corporate income tax	US$ Million	(45.9)	(3.3)	(106)
Free Cash Flow	US$ Million	181.3	13.1	421
Pre-tax NPV @ 5%	US$ Million	191.1		
Post-tax NPV @ 5%	US$ Million	151.7		

Table 22-3: Project Cash Flow *(Dollar figures in US$M unless otherwise noted)*

BOROO GOLD	Unit	Total / Average	2024	2025	2026	2027	2028	2029	2030	2031
CASH FLOW										
Cash flow										
Metal price										
Gold price	US$/oz	1,750.0	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750
Revenue	US$M	754.7	119.7	114.3	114.2	93.9	83.0	89.9	86.4	53.3
Mining cost	US$M	(150.9)	(22.4)	(23.3)	(23.3)	(22.6)	(22.7)	(22.7)	(14.0)	-
Processing cost	US$M	(190.8)	(22.5)	(24.4)	(20.6)	(26.6)	(28.1)	(23.3)	(23.7)	(21.6)
G&A cost	US$M	(63.7)	(8.9)	(7.6)	(7.5)	(8.0)	(8.0)	(8.0)	(8.0)	(7.8)
Royalties	US$M	(37.7)	(6.0)	(5.7)	(5.7)	(4.7)	(4.1)	(4.5)	(4.3)	(2.7)
EBITDA	US$M	311.6	59.9	53.3	57.1	32.0	20.0	31.5	36.4	21.2
Sustaining Capex	US$M	(68.4)	(19.3)	(8.9)	(8.8)	(12.0)	(10.3)	(6.1)	(3.0)	-
Closure Capex	US$M	(16.0)	-	-	-	-	-	-	-	(16.0)
Pre-Tax Free Cash Flow	US$M	227.2	40.7	44.4	48.2	20.0	9.8	25.4	33.4	5.3
Corporate income tax	US$M	(45.9)	(10.8)	(9.6)	(10.9)	(4.5)	(1.1)	(3.5)	(4.4)	(1.1)
Post-Tax Free Cash Flow	US$M	181.3	29.9	34.8	37.3	15.5	8.7	22.0	29.0	4.2
Pre-Tax NPV @5%	US$M	191.1								
Post-Tax NPV @5%	US$M	151.7								

PRODUCTION

Mining and milling schedule

Physicals

	Unit	Total / Average	2024	2025	2026	2027	2028	2029	2030	2031
Production Schedule										
Ore Mined	Mt	28.6	6.8	2.6	3.1	4.5	2.6	4.9	4.2	-
Au grade	g/t	0.71	0.69	1.00	1.02	0.63	0.56	0.69	0.53	-
Strip Ratio	w:o	3.25	1.65	6.42	5.11	3.09	5.99	2.72	1.68	-
Waste Mined	Mt	93.1	11.2	16.2	15.7	13.8	15.7	13.3	7.1	-
Total Material Mined	Mt	121.7	18.0	18.8	18.8	18.2	18.3	18.3	11.3	-
Mill Schedule - CIP										
Mill Feed	Mt	13.8	1.7	1.7	1.7	1.7	1.7	1.7	1.7	1.7
Au Grade	g/t	1.20	1.61	1.34	1.74	1.23	0.89	1.13	1.06	0.61
Contained Au	kg	16,625	2,791	2,328	3,012	2,129	1,548	1,965	1,838	1,014
Recovery rate	%	69%	71%	71%	63%	68%	73%	69%	67%	69%
Recovered Au	oz	366,195	63,480	52,925	61,007	46,434	36,489	43,444	39,848	22,568
Mill Schedule - HL										
Mill Feed	Mt	16.7	1.0	2.8	1.1	2.2	2.8	2.3	2.5	1.9
Au Grade	g/t	0.30	0.40	0.34	0.29	0.25	0.30	0.27	0.29	0.32
Contained Au	kg	5,057	383	962	329	563	849	618	738	615
Recovery rate	%	40%	40%	40%	40%	40%	40%	40%	40%	40%
Recovered Au	oz	65,055	4,926	12,375	4,237	7,242	10,921	7,952	9,495	7,907

ECONOMICS

Economics

	Unit	Total / Average	2024	2025	2026	2027	2028	2029	2030	2031
Revenue Gold sales										
Gold	US$/oz	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750
Recovered Gold	oz	431,250	68,406	65,300	65,244	53,676	47,410	51,396	49,343	30,475
Royalty 5%	US$M	37.7	6.0	5.7	5.7	4.7	4.1	4.5	4.3	2.7
Net Revenue (after royalty)	US$M	717.0	113.7	108.6	108.5	89.2	78.8	85.4	82.0	50.7
Operating cost										
Mining	US$M	150.9	22.4	23.3	23.3	22.6	22.7	22.7	14.0	-
Processing - CIP	US$M	144.2	18.5	17.6	17.7	19.8	18.3	17.6	17.6	17.0
Processing - HL	US$M	46.6	4.1	6.7	2.9	6.8	9.8	5.7	6.0	4.6
Site admin	US$M	40.1	5.0	5.0	5.0	5.0	5.0	5.0	5.0	4.9
UB head office	US$M	23.6	3.8	2.5	2.5	2.9	2.9	2.9	2.9	2.9
Total	US$M	405.4	53.8	55.2	51.4	57.2	58.8	53.9	45.6	29.4
Sustaining capital expenses										
Geology	US$M	2.6	1.0	0.8	0.8	-	-	-	-	-
Mine	US$M	25.1	10.6	4.0	3.7	3.0	1.9	1.9	-	-
Process Plant	US$M	18.7	4.3	1.0	1.0	6.0	5.3	1.1	-	-
Management, Engineering, EPCM Service	US$M	0.7	0.7	0.0	0.0	0.0	0.0	0.0	-	-
TFS	US$M	21.2	2.7	3.0	3.3	3.0	3.0	3.0	3.0	-
Closure Costs	US$M	16.0	-	-	-	-	-	-	-	16.0
Total	US$M	84.4	19.3	8.9	8.8	12.0	10.3	6.1	3.0	16.0

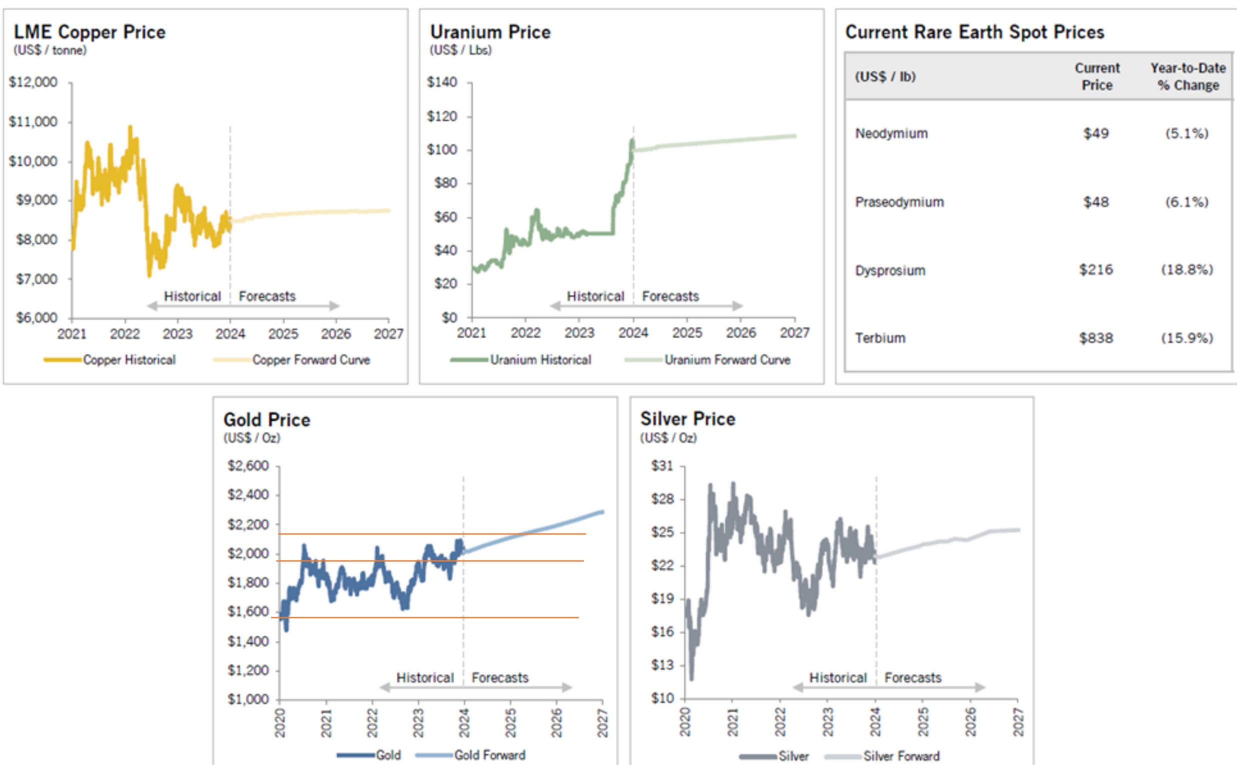
22.4 Post-Tax Scenario Analysis

Game Mine applied various metal prices to the financial model to better understand the potential economics at current and forecast metal prices.

Table 22-4: Post-Tax Scenario Analysis Results

Price case	Post-tax NPV@5% (US$ Million)
Worse case (US$1600 per ounce)	113.1
Base case prices (US$1750 per ounce)	151.7
Spot prices (US$2000 per ounce)	215.1
Best case (US$ 2200 per ounce)	265.8

Figure 22-1: Key Commodity Pricing (Jefferies Weekly Update, Jan 26th 2024)



22.5 Sensitivity Analysis

A sensitivity analysis was conducted on the base case pre-tax and post-tax NPV (Net Present Value) of the Project, using the following variables: gold price, operating costs, and total capital cost.

Table 22-5 shows the pre-tax sensitivity analysis results; post-tax sensitivity results are shown in Table 22-6. To decrease sensitivity, the project is less sensitive to changes in the total operating cost compared to sensitivity to changes in the price of gold.

Table 22-5: Sensitivity Analysis (Pre-Tax NPV@5%)

Pre-Tax NPV sensitivity to Opex										
		Gold price (%)								
		-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%
Opex (%)	-30%	172.1	201.7	231.3	260.9	290.5	320.1	349.7	379.3	408.9
	-20%	139.0	168.6	198.2	227.8	257.4	287.0	316.6	346.2	375.8
	-10%	105.8	135.4	165.0	194.6	224.2	253.8	283.4	313.0	342.6
	0%	72.7	102.3	131.9	161.5	191.1	220.7	250.3	279.9	309.5
	10%	39.5	69.1	98.7	128.3	157.9	187.5	217.1	246.7	276.3
	20%	6.4	36.0	65.6	95.2	124.8	154.4	184.0	213.6	243.2
	30%	(26.7)	2.9	32.5	62.1	91.7	121.3	150.9	180.5	210.1

Pre-Tax NPV sensitivity to Capex										
		Gold price (%)								
		-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%
Capex (%)	-30%	93.5	123.1	152.7	182.3	211.9	241.5	271.1	300.7	330.3
	-20%	86.6	116.2	145.8	175.4	205.0	234.6	264.2	293.8	323.4
	-10%	79.6	109.2	138.8	168.4	198.0	227.6	257.2	286.8	316.4
	0%	72.7	102.3	131.9	161.5	191.1	220.7	250.3	279.9	309.5
	10%	65.7	95.3	124.9	154.5	184.1	213.7	243.3	272.9	302.5
	20%	58.8	88.4	118.0	147.6	177.2	206.8	236.4	266.0	295.6
	30%	51.8	81.4	111.0	140.6	170.2	199.8	229.4	259.0	288.6

Table 22-6: Sensitivity Analysis (Post-Tax NPV@5%)

Post-Tax NPV sensitivity to Opex										
		Gold price (%)								
		-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%
Opex (%)	-30%	**137.4**	**159.6**	**181.8**	**204.0**	**226.2**	**248.4**	**270.6**	**292.8**	**315.0**
	-20%	**112.5**	**134.8**	**157.0**	**179.2**	**201.4**	**223.6**	**245.8**	**268.0**	**290.2**
	-10%	**86.1**	**109.5**	**132.1**	**154.3**	**176.5**	**198.7**	**220.9**	**243.1**	**265.3**
	0%	**59.0**	**83.0**	**106.4**	**129.4**	**151.7**	**173.9**	**196.1**	**218.3**	**240.5**
	10%	**29.8**	**55.5**	**79.9**	**103.3**	**126.5**	**149.0**	**171.2**	**193.4**	**215.6**
	20%	**0.3**	**26.1**	**51.9**	**76.6**	**100.2**	**123.5**	**146.3**	**168.5**	**190.7**
	30%	**(29.3)**	**(3.4)**	**22.5**	**48.3**	**73.3**	**97.1**	**120.4**	**143.4**	**165.9**

Post-Tax NPV sensitivity to Capex										
		Gold price (%)								
		-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%
Capex (%)	-30%	**79.8**	**103.9**	**127.3**	**150.3**	**172.5**	**194.7**	**216.9**	**239.1**	**261.3**
	-20%	**72.9**	**96.9**	**120.3**	**143.3**	**165.6**	**187.8**	**210.0**	**232.2**	**254.4**
	-10%	**65.9**	**90.0**	**113.4**	**136.4**	**158.6**	**180.8**	**203.0**	**225.2**	**247.4**
	0%	**59.0**	**83.0**	**106.4**	**129.4**	**151.7**	**173.9**	**196.1**	**218.3**	**240.5**
	10%	**52.0**	**76.1**	**99.5**	**122.5**	**144.7**	**166.9**	**189.1**	**211.3**	**233.5**
	20%	**45.1**	**69.1**	**92.5**	**115.5**	**137.8**	**160.0**	**182.2**	**204.4**	**226.6**
	30%	**38.1**	**62.2**	**85.6**	**108.6**	**130.8**	**153.0**	**175.2**	**197.4**	**219.6**

The NPV was also estimated for varying discount rates, with Game Mine applying 5% as the base case. Table 22-7 shows the post-tax NPV at varying discount rates.

Table 22-7: NPV Estimates at Varying Discount Rates

Discount rate	Pre-tax NPV	Post-tax NPV
0%	US$ 227.2M	US$ 181.3M
5%	US$ 191.1M	US$ 151.7M
8%	US$ 173.7M	US$ 137.5M
10%	US$ 163.6M	US$ 129.2M

Table 22-8: NPV Estimates at Varying Gold Price

Gold price (US$/oz)	Pre-tax NPV	Post-tax NPV
1,600	US$ 140.3M	US$ 113.1M
1,650	US$ 157.3M	US$ 126.2M
1,700	US$ 174.2M	US$ 139.0M
1,750	US$ 191.1M	US$ 151.7M
1,800	US$ 208.0M	US$ 164.3M
1,850	US$ 224.9M	US$ 177.0M
1,900	US$ 241.8M	US$ 189.7M
1,950	US$ 258.7M	US$ 202.4M
2,000	US$ 275.7M	US$ 215.1M

22.6 Break-Even Analysis

Game Mine performed a breakeven point analysis where the NPV is zero depending on the price of gold, mining, and processing costs. These evaluations were done independently and did not reflect the impact of the variance of multiple input parameters. Gold price and cost values where NVP is zero:

4. Gold price - US$ 1199/oz.
5. Mining cost - US$ 3.06/t ROM
6. Processing cost (CIP) - US$ 28.2/t milled

22.7 Tax Modelling

In the financial model, only the corporate income tax (CIT) is calculated based on the taxable income, operating costs, and depreciation expense. All operating costs are treated as deductible expenses from taxable income. Boroo Gold is considered not a VAT-paying organisation because the mined gold is delivered to an authorised commercial bank, so the VAT is not calculated.

22.8 Depreciation

To calculate the CIT, Game Mine has depreciated the fixed and mine assets over the life of the mine, and the company provides the depreciation calculation. This is shown in Table 22-8.

Table 22-8: Total depreciation (2024-2031)

Asset Class	US$ Millions	Years remaining for depreciation
Geology	2.6	8
Mine	42.9	8
Process Plant	49.0	8
Management, Engineering, EPCM Services	4.1	8
TFS	21.2	8
Total	**119.8**	

The applicable depreciation was totaled for each year of operations and deducted from years with positive cash flows.

22.9 Taxable Income Assessment

The total taxable income for the Boroo gold project is estimated to be US$ 191.8M. By corporate income tax law, CIT is calculated as 10% of the first MNT 6 billion and 25% of the taxable income over MNT 6 billion each year with positive taxable income. Net taxes were estimated at US$45.9M for the project life, resulting in post-tax free cash flow of US$ 181.3M.

23.0 INTERPRETATIONS AND CONCLUSIONS

23.1 Geology and Mineral Resources

The Boroo gold deposit is a low silica Au+As sulphide system associated with a zone of quartz-sericite-pyrite (QSP) alteration in the sub horizontal Boroo fault. Boroo is an intrusion-related gold deposit and hosted by a Cambrian-Ordovician sequence of highly deformed shales, siltstones and fine sandstones of the Haraa turbidite sediments, and the Paleozoic granitoids of the Boroo Complex. The bulk mineable gold mineralisation at Boroo is hosted in a strongly quartz-sericite altered and sulphidised nearly flat lying zone controlled by the Boroo fault.

Ulaanbulag deposit depends on the morphology of the mineralized region with shallow dip angle and variable thickness. Mineralization is defined by quartz-cali field cali-sericit-pirite alteration and low silica Au+As sulphide system associated. Ulaanbulag is oregon related gold deposit with its tectonic-macmic environment, its carbon dioxide components, and its geochemical properties.

At a cut-off grade of 0.1 g/t Au, Boroo has been estimated with a Measured Resource of 26.6 Kt at an average grade 0.588 g/t Au, an estimated Indicated Resource of 17.3 Kt at an average grade of 0.542 g/t Au, and an estimated Inferred Resource of 1.3 Kt at an average grade of 0.789 g/t Au.

At a cut-off grade of 0.1 g/t Au, Ulaanbulag has been estimated with a Measured Resource of 4.4 Kt at an average grade 0.6 g/t Au, an estimated Indicated Resource of 7.9 Kt at an average grade of 0.485 g/t Au, and an estimated Inferred Resource of 4.2 Kt at an average grade of 0.379 g/t Au.

23.2 Metallurgical Test Work

The current Boroo ore processing flowsheet is the result of a number of past metallurgical test programs and confirmed by successful operation results. The current Boroo Mill flowsheet includes crushing, grinding, gravity concentration, leaching and carbon-in-pulp (CIP), cyanide detoxification and arsenic precipitation to produce gold doré.

The metallurgical testwork has confirmed the parameters for the major aspects of the design of the processing plant.

During 2006, column leaching testwork has been conducted for oxide, transitional and fresh ore by AMMTECH Australia and KCA USA based labs. AMMTECH testwork gold recoveries of 91.4% were reported for oxide ore, 70.3% for transition ore and 64.9% for fresh ore. KCA testwork gold recoveries of 90% were reported for oxide ore, 65% for transition ore and 53% for fresh ore.

23.3 Mineral Reserve Estimate

Resources and meet the definitions of Proven and Probable Mineral Reserves as stated by NI 43-101 and defined by the CIM standards on Mineral Resources and Reserves Definitions and Guidelines (2014). The Mineral Reserve estimates are based on a mine plan and pit design developed using modifying parameters including metal price, metal recovery model based on lithology, alteration, testwork and performance of the processing plant, and operating cost estimates. The Proven and Probable Reserves are inclusive of the of Mineral Resource and based on a five-year moving average gold price of $1,750/oz.

Geotechnical investigations were conducted to assess the expected rock quality and stability at Boroo and Ulaanbulag mine. Characterization of structural domains was completed for slope stability and pit design considerations. Overall slope angles and bench parameters were provided from the geotechnical analysis as inputs to the pit optimization study.

Mining costs of $10.9 /tonne, milling costs of $13.8 /tonne and general and administrative costs of $4.6/tonne have been used to estimate the reserves along with the gold price stated above.

Proven and Probable Reserves total 30.5 Mt of ore (CIP ore 13.8 Mt, Heap leach ore 16.7), with an estimated contained 697k oz of gold.

23.4 Mining Methods

The Boroo and Ulaanbulag mining operations are based on conventional open-pit methods to mine a nominal 50,000 tonnes per day material. Operations are planned to stop during the first half of 2030. Mining is done with bench heights of five metres, with ore mined on half benches for improved grade control in the flat-lying ore.

The objective to the Boroo and Ulaanbulag LOM schedule was to maximize the early cash flow from the pit by targeting high grade material to be fed to the processing plant while delaying costs by deferring waste stripping to later in the project life. The optimum throughput to the processing plant was determined to be 1.7 Mt/year. The primary objective of the schedule was ensuring continuous ore supply to the processing plant by delivering the highest grade ore first and meeting milling capacity constraints.

The Boroo and Ulaanbulag pits will be mined in 5 m benches with a maximum vertical advance of 60 m per year. There is no scheduled pre-stripping period, although some stripping is delayed as much as possible. The mining plan includes stockpiling on an annual basis to allow the highest-grade material to be fed to the processing plant. Mining will take place 365 days a year and no allowance has been made for climate change, labor action, or any other unscheduled shutdowns. An overall ex-pit material movement constraint of approximately 18-18.7 Mt per year was applied.

Totally 93 million tonnes waste will be dumped backfill waste dump. The designs for active waste dumps assume a swell factor of 30% for the material delivered from pit benches, considering natural sorting and 10% compaction of the dumps. It is also assumed that the blasted waste rock will settle at the natural angle of repose of 34°.

Totally 16.6 million tonnes heap leach ore (defined as material below the mill cut-off grade and above heap leach cut-off grade) will be crushed and stacked on the leach pad. Ore will be hauled from the pit or stockpiles near the pit to a heap leach stockpile. The largest heap leach ore stockpile will be 3.4 Mt in Year 2027 and be depleted to about 1.6 Mt for the remainder of the mine life.

23.5 Recovery Methods

The recovery model at the Boroo Mine is modeled in three zones, and the model is based on ore body weathering, lithology, alteration and laboratory testwork results collected at the deposit scale. Also, as mentioned above, the modeled metal recovery is verified against the actual data.

Ulaanbulag recovery estimate are based on internal metallurgical testwork in 2014 and 2021.

Heap leach ore to be mined from the Boroo and Ulaanbulag pits is expected to be more refractory and has been given a lower overall recovery. Operation results showed that the heap leach gold recovery is 40%, and which is aligned with testwork result.

23.6 Environmental and Closure Planning

Management of significant Project environmental and social aspects and impacts is through a suite of Management Plans that will be implemented during the project activities.

Long-term planning for mine closure is framed by regulatory requirements, stakeholder expectations and standards of social and environmental responsibility.

Main facilities at the Project that have not been reclaimed include open pits, heap leach (HL), TMF and other buildings & infrastructures. During the closure stage, the facilities, buildings and infrastructures will be removed and the areas will be cleaned, re-contoured and re-sloped to a suitable degree for revegetation and in a manner resistant to the soil erosion.

Closure of the HL facility will be performed as per specified in the national standard MNS 6297: 2011 on closure of heap and dump leaching facilities. In this standard, wastewater from the HL facility is required to meet MNS 4943:2015 standard on effluent water quality.

The closure of the Boroo TMF cell will require construction of an evapotranspiration (ET) cell to collect and manage drainage from the tailings facility. This may be achieved by enhanced evaporation. Alternately, the water may be treated and discharged.

Once the TMF supernatant and initial drainage are removed, a 1.5 m rock cover will be placed on the surface. This will allow mobile equipment onto the facility and provide a foundation for placement of a soil cover between 0.5 and 1.0 m in thickness. This cover will minimize long-term infiltration into the facility.

23.7 Capital and Operating Costs

Since the Boroo Gold mine is operational, no additional investment will be made to increase the capacity, so no investment calculation has been completed.

Sustaining capital costs include all costs from 2024 of the operation to maintain mining until the end of the planned LOM in 2031. Total sustaining costs of $68.4M have been estimated for the Boroo Gold mine.

The LOM average operating cost for the Project is estimated at US$29.3/t processed at the nominal process rate of 5000 t/d or 1.7 Mt/year. This operating cost is derived using total LOM operating costs divided by LOM processed tonnages and excludes pre-production mining costs.

23.8 Economic Analysis

Game Mine prepared an economic evaluation of the Boroo Gold project based on a free cash flow financial model created for this technical report. For the 8-year total life-of-mine (from 2024 to 2031) and 13.8M tonne CIP ore and 16.7M tonne Heap Leach ore Mineral Reserve, the following pre-tax and post-tax financial parameters were estimated using the base case parameters:

- Pre-tax NPV at 5% discount rate - US$191.1M
- Post-tax NPV at a 5% discount rate - US$151.7M

Operating costs for mining, processing, site admin and off-site charges were deducted from the revenue to derive the annual operating cash flow. This operational cash flow, sustaining capital, closure and reclamation costs were deducted to determine the net pre-tax cash flow. Sustaining capital is included to cover expenditures for mining equipment and processing plant repair and replacement.

Sustaining capital expenditure and mine closure cost included within the financial model total US$68.4M and US$16.0M, respectively.

23.9 Opportunities

Additional mineralization opportunity for Ulaanbulag exists to the south-east and east of the currently modeled deposit.

Boroo orebody remains an opportunity for the project following initial desktop viability studies. A preliminary economic assessment can be undertaken to determine the potential for inclusion in the Boroo project life of mine plan as a smaller underground mine.

23.10 Risks

The mining assets are subject to certain inherent risks, which applies to some degree to all participants of the international mining industry. These risks are summarised as follows:

- Fluctuations in gold price – Risk of pricing regression of gold and/or US$ will increase the potential impact on the project profitability. Sensitivity analyses conducted during the economic analysis of the project confirmed that the NPV of the project is most sensitive to changes in the gold price.
- Geotechnical risk – Risk of geotechnical unknows and difficulties will increase when mining gets deeper. It requires qualified data collection, technical analysis, and assessment with ongoing operation.

- Recovery risk – Due to Boroo and Ulaanbulag mining will be mostly focusing on fresh ore in coming years, however, there is a limited testwork is conducted for fresh ore. Hence, robust reconciliation system development is recommended to understand how resource and reserve models are performing compared to actual measured performance and equally importantly how long term mine plans are being delivered via comparing medium term plans and actual mining performance.
- Asset integrity risk – Because of project is over 20 years of operation, there is a rising risk related with mobile and fixed equipment integrity.
- Tailings dam is one of the major hazards and highest risks of all mining companies and BGC has one Tailings storage facility (TSF) reached its ultimate design and started second TSF with conventional design and construction practice with limited understanding of difference in foundation of the TSF and other factors. Hence recommending adopting international standard of tailings management and have independent expert consultants for frequent review and recommendation.
- Dam break assessment for the new TSF is highly recommended to understand emergency and business response management.

24.0 RECOMMANDATIONS

24.1 Geology and Mineral Resources

The Overall geology is well understood for both Boroo and Ulaanbulag deposits. The Boroo mine structure, mineralization and alteration has been interpreted with high confidence data which is collected during the project life of mine.

To potentially expand the current resource base at Ulaanbulag, optional additional drilling can be undertaken with a specific focus on expanding and infilling the mineralization. As infill drilling is conducted, drill hole assay and lithology results should be compared against the geological and resource model in order to quantify any variation in expected and realized geology and gold grades which were intersected.

It's further recommended that Boroo insert both a higher-grade gold standard and lower grade gold standard into their data QA/QC protocols in order to better reflect the gold grades encountered at Boroo mine and Ulaanbulag.

Upon the conclusion of the additional drilling, the 3D geological and resource model should be updated in order to incorporate this data.

For Boroo mine, Additional drilling should be considered in mineralized areas along strike that have not been included in the current Mineral Resource Model in order to include mineralization in those areas in future resource updates.

It's further recommended that possible underground mining assessment for remained ore body.

24.2 Geotechnical aspects

A dedicated geotechnical site investigation is recommended to collect additional detailed geotechnical and hydrogeological information. The site investigation should consist of oriented, triple-tubed diamond drilling with associated geotechnical logging, installation of vibrating wire piezometers, and laboratory strength testing of core samples. It is recommended to keep logging final wall mapping, and collecting data for further analysis.

The data to be used to update the engineering geological model and characterization of the rock masses. Detailed slope stability analyses should be conducted to optimize pit slope design criteria.

Geotechnical monitoring tools implementation is recommended for final wall such as prism monitoring, inclinometer and TDR.

Incorporate more in-depth hydrogeological study into the next level of study to verify mine dewatering requirements required for pit expansion.

Incorporate more in-depth hydrogeological study into the next level of study to verify mine dewatering requirements required for pit expansion.

24.3 Mining

It's further recommended that backfill option for Ulaanbulag deposit once ore body knowledge is interpreted with high confidence.

Further analysis of ore loss and dilution should be carried out once operational data becomes available.

24.4 Metallurgical testing

Future testwork will be required and will involve defining the remaining ore source. As the oxide material was tested extensively and zone is going to be mined out mostly, it is recommended that future testwork focus on fresh rock for Boroo Gold deposit and Ulaanbulag. The recommended components of the testwork may include gravity recoverable gold, bottle roll leach testing.

It's recommended to focus on sulphide ore bottle roll testwork with certified laboratories due to current testwork result is mostly focused on flotation.

It's recommended to implement detailed engineering design and budget for heap leach pad.

25.0 REFERENCES

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Report for Boroo Gold LLC, Scoping Study for the Boroo Gold Project, AU10428, June 2020

CERTIFICATE OF QUALIFIED PERSON

I, Tuvshinbayar Batbayar, do hereby certify that:

1. I am an Independent Consultant for Game Mine LLC.
2. I am currently employed as a Manager Tailings and Hydrogeology at Oyu Tolgoi LLC.
3. This certificate applies to the report entitled "Technical Report on the Boroo, Ulaanbulag Gold Project" with an effective date as of February 01, 2024.
4. I am a graduate of Mongolian University of Science and Technology in Ulaanbaatar, Mongolia (Bachelors of Engineering (Mining) in 2011) and University of New South Wales (Masters of Mining Engineering in Mine Management in 2018).
5. I am a member of the Australian Institute of Mining and Metallurgy (MAusIMM) where I am a chartered professional in the disciplines of both Mining and Management. (Member No: 3000418).
6. I have practiced my profession continuously since 2010 with over 13 years of experience in mining precious metals across USA, Mongolia and in consulting in Mongolia. My relevant work experience includes:
   - Mine operation, open pit grade control, mine plan and grade reconciliation, mine designing, scheduling, cost estimation, technical open pit studies.
   - Evaluation and optimization of mine designs, mining costs and schedules.
   - Mining major hazard management, process and system development, assurance
   - Tailings management, tailings construction
7. I have read the definition of "qualified person" set out in the National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI43-101") 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 fulfil the requirements to be a qualified person for the purposes of NI 43-101.
8. I visited the Property on Jun 15, 2024.
9. I am the sole author of this report and I am responsible for its contents.
10. I am independent of Boroo Gold LLC and Steppe Gold Ltd. as described in section 1.5 of NI 43-101.
11. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI43-101 and Form 43-101F1.
12. I have not had any prior involvement with the property that is the subject of this Technical Report.
13. As of the date of this certificate, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 1st February 2024

Signing Date: 17th June 2024

"Tuvshinbayar Batbayar" [signed and sealed]