NI 43-101 Technical Report
Feasibility Study for the
Altan Tsagaan Ovoo (ATO) Phase 2 Expansion Project
Mongolia

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IMPORTANT NOTICE

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This Report contains estimates, projections and conclusions that are forward-looking information within the meaning of applicable laws. Forward-looking statements are based upon the responsible Qualified Person's ("QP") opinion at the time they are made but, in most cases, involve significant risks and uncertainty. Although each of the responsible QPs has attempted to identify factors that could cause actual events or results to differ materially from those described in this Report, there may be other factors that could cause events or results to not be as anticipated, estimated or projected. There can be no assurance that forward-looking information in this Report will prove to be accurate, as actual results and future events could differ materially from those anticipated in such statements or information, Accordingly, readers should not place undue reliance on forward-looking information. Forward-looking information is made as of the effective date of this Report, and none of the QPs assume any obligation to update or revise it to reflect new events or circumstances, unless otherwise required by applicable laws.

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1 SUMMARY

1.1 Introduction

Following the successful completion of its Altan Tsagaan Ovoo (ATO) Phase 1 development and on-going crusher upgrades located in eastern Mongolia, Steppe Gold LLC (Steppe Gold or the Company) initiated studies for the ATO Phase 2 Expansion Project (the "Project"). The Phase 2 Expansion which is the subject of this Technical Report ("Report"), will expand gold production and produce saleable concentrates of lead, zinc, and pyrite from the development of underlying fresh rock ores and the construction of a new and larger conventional processing facility.

The ATO Project is 100% owned by Steppe Gold, an international mineral resource company headquartered in Toronto, with exploration, development and production properties located in Mongolia. Steppe Gold is listed on the Toronto Stock Exchange under the symbol STGO.

In 2017, an NI 43-101 Technical Report was prepared for Steppe Gold and Centerra Gold (Centerra) (the previous ATO Project owner) by the Mongolian company GSTATS Consulting LLC for Phase 1 of the ATO Project. In 2020, Steppe Gold completed the construction and commissioning for Phase 1, which consisted of an integrated oxide ore heap leach production facility which now produces 60,000 oz of gold annually.

Steppe Gold commissioned a team of consultants to complete Feasibility Study (FS) work and a Technical Report in accordance with National Instrument 43-101 (NI 43-101) guidelines for the Project. The FS work was led by DRA Global Limited (DRA) who was responsible for the mine planning, mineral reserve estimate, metallurgy, market study, capital and operating cost estimating activities, and economic analysis. The Mineral Resource estimation was provided by GeoRes. Knight Piésold provided the tailings design and Ulzii Environmental (Mongolia), LLC (Ulzii) provided the hydrogeology, water quality, and environmental resource management.

1.2 Property Description and Location

The property is located in the Tsagaan Ovoo soum territory of the Dornod province in eastern Mongolia, 660 km east of the Mongolian capital of Ulaanbaatar and 120 km northwest of the provincial capital of Choibalsan. It is located in the Davkhariin Aryn valley, at the junction of Bayan and Duruu rivers, and the foot of various regional mountains (Delger Ulziit, Bayan, Namkhai Hill and Yaruu).

The geographic zone of the ATO Project is in datum WGS-84 Zone 49N of the UTM coordinate system.

The ATO property is covered under a single mining license MV-017111 over an area of 5,493 ha.

1.3 History

Regional geological field parties working for COGEGOBI (a wholly owned subsidiary of AREVA) were the first to recognize mineralised rocks cropping out at ATO. In 1997, COGEGOBI began exploration in eastern Mongolia.

The ATO gold mine property is covered under a single mining license MV-017111 over an area of 5,493 ha.

Initially (December 30, 2003) exploration license number 6727X was issued to Coge Gobi LLC, representing 109,118 ha. In 2007, most (~85,500 ha) of the area was turned over to the Cadaster office, leaving about 23,600 ha to the license.

On May 4, 2010, Centerra acquired this reduced license by Order #513 of Head of the Cadaster Office. Centerra then conducted intense exploration programs on the acquired license area, and in 2012, prepared a feasibility study to transfer the exploration license to a mining license with an applicable area of about 11,600 ha surrounded by 19 location points.

Mining license MV-017111 was issued to Centerra on August 31, 2012 with an applicable 30-year term expiring on August 31, 2042. The license boundary has since been simplified to 8 points with area of about 5,500 ha.

On January 31, 2017, Steppe Gold entered into a definitive agreement to purchase 100% interest in the ATO Project. The mining license and other assets were transferred to Steppe Gold on September 5, 2017, and Steppe Gold is now the 100% owner of the ATO mining license.

1.4 Geological Setting and Mineralisation

1.4.1 Regional Geology

The ATO Project sits regionally within the Devonian through Late Jurassic Mongol-Okhotsk tectonic collage that has been emplaced along a transform-continental margin of the North Asian Craton (NAC). A number of Late Jurassic- early Cretaceous broad, gold-bearing mineral belts have been recognized in eastern Mongolia.

The ATO Project is located north of the Main Mongolian Lineament (MML), and midway along the NNE trending 600 km long Onon base and precious-metal province that crosses eastern Mongolia. Though the ATO Project currently represents the only well-explored gold deposit in this part of Mongolia, a large number of minor gold occurrences have been recognized throughout the region.

1.4.2 Project Geology

The geology of the ATO Project region consists of metamorphosed Devonian sedimentary rock overlain by a volcanic and sedimentary sequence of Permian age and remnant scraps of probable

Jurassic volcanoclastic units, intruded by Jurassic plutons ranging from diorite to granite in composition and including rhyolitic phases mainly as dykes.

1.4.3 MINERALISATION

Mineralisation at the ATO Project is summarised as:

- An intermediate sulfidation (IS) system.
  - Neutral low temperature, near paleo surface fluids, bladed silica after calcite indicates some local boiling. Related to Jurassic magmatic event.
  - Banded silica, broken sinter, repeatedly recrystallized at paleo top.
- Confined to pipe bodies.
  - Multiple collapse and upward transport in the pipes, repeated brecciation followed by continued ingress of steep and shallow veins, veinlets and flooding.
- Magnesium chlorite and silica dominant system.
  - Quartz, clinochlore (high Mg, Al silicate), kaolinite, gypsum (peripheral after anhydrite), adularia is absent.
- Dominated by free gold, base metal and Ag sulphides.
  - Lead-phosphates (near surface) – Pb-carbonates – galena (at depth).
  - Zinc-carbonate (near surface) – low FeS sphalerite (at depth).
  - Au in quartz & in sulphides; Ag mostly in tetrahedrite & miargyrite.

1.5 Exploration Work and Drilling

1.5.1 EXPLORATION

The field investigation that was conducted in the license area can generally be subdivided into two stages: prospecting and exploration.

In 2003-2009, geologists of COGEGOBI, who had been specialized in prospecting and exploration of uranium projects, conducted geological mapping and prospecting traverses and collected geochemical samples in the exploration license area, supplemented by a magnetic survey. This work resulted in the discovery of an epithermal gold occurrence.

In 2010, CGM carried out a prospecting stage consisting of geological mapping and prospecting traverses, surface and other sampling tasks, variety of geophysical surveys, and some trenching and drilling. As a result of the prospecting, ATO occurrence was chosen to a detailed study, and an intensive drilling program began in late 2010 to advance the ATO Project to an exploration stage.

1.5.2 DRILLING

An exploration drilling program was completed by CGM between 2010 to 2014. Diamond core drill holes (DDH) were the principal source of geological and grade data for the ATO Project. Some reverse circulation (RC) drilling was completed between 2012 and 2014 through cover to map bedrock geochemical patterns as a method of exploring for blind ATO-style mineralisation on the project area. CGM carried out a hydrogeology and geotechnical drilling program in 2011.

Commencing in 2018, Steppe commenced a three-phase exploration drilling program comprising the following:

- Phase 1 exploration program focused on the ATO4, Mungu, Tsagaan Temeet, Bayanmunkh and Bayangol targets at the ATO Project. A total of 66 holes were drilled at the Mungu Deposit, 3 holes at the ATO4 Deposit, 4 drill holes at the Tsagaan Temeet prospect, 1 drill hole at the Bayanmunkh prospect and 16 shallow drill holes at the Bayangol prospect for a total of 8,821 m. The drilling program was successful in outlining and extending known gold and silver mineralisation. In addition, new high-grade zones in deeper parts of the deposit were discovered.
- Phase 2 drilling program focused on the ATO4 end of the ATO4-Mungu trend at the ATO Project and at the Uudam Khundii Project. The drilling program was completed with three diamond core drilling rigs completing a total of 36 drill holes for 9,006 m. The completion of the Phase 2 drilling program saw the identification of the first ever visible gold seen at ATO Project, with super high gold grades being returned in ATO299 and ATO317.
- Phase 3 drilling program targeted at the ATO4-Mungu trend has commenced with 8 drill holes being completed for 2,228 m of drilling[1].

In 2019, Steppe completed a drilling program with two diamond core drilling rigs focused on updating resources and reserves for the ATO1, ATO2 and ATO4 deposits in addition to a maiden Resource and Reserve delineation for the Mungu Discovery. Steppe drilled 1,840 m at ATO1, 1,662 m at ATO2, 14,760 m at ATO4 and over 26,573 m at the Mungu Discovery[2].

Commencing 2020, Steppe drilled an additional 55 drill holes for a total of 18,200 m. A total of 53,000 m has been drilled since 2018. The drilling information was used to update the interpretation of the geological model, geometry of the mineralised zones and domains[3].

1 2019 AIF and 2018 news release.
2 2019 AIF and 29 July 2020 news release.
3 21 February 2021 news release. Information adopted from.

1.6 Mineral Processing and Metallurgical Testing

The overall ATO Project consists of two (2) processing facilities: an existing heap leach operation (Phase 1), and a proposed concentrator plant (Phase 2). The oxide portion of the ATO Project (Phase 1) employs a conventional oxide heap leach flowsheet including crushing, heap leaching, and gold recovery facilities. Phase 1 has been operational since July 2020 and focuses on the production of gold and silver doré. A subsequent expansion to Phase 1 included new three-stage crushing.

Phase 2 will consist of milling, flotation, and dewatering unit operations to produce concentrates of lead (Pb), zinc (Zn), and pyrite (Py). DRA supervised and provided input during the development and execution of the testwork program performed by the laboratory in 2021, which provided the basis for the establishment of the Phase 2 flowsheet. The interpretation and analysis of the testwork results was carried out by DRA. This analysis was then used to determine the process design basis and flowsheet of the Project.

1.6.1 Historical Testwork (2010-2018)

Several metallurgical testwork programs were undertaken on samples selected from the ATO Project. These metallurgical tests for processing of ATO ore samples were conducted at the Central Laboratory of Xstrata Process Support (XPS) in Canada, ALS Metallurgy-Ammtec laboratory in Australia, Boroo Au LLC processing plant in Mongolia and SGS Lakefield (SGS) in Canada.

Metallurgical test samples were selected from the drill core and bulk samples from ATO Deposit's oxidized zone in Pipes 1, 2, and 4. These tests for ore samples included a step-by-step leaching test carried out by the bottle roll test and granular ore test.

Various testing programs were completed, including:

- Mineralogy and elemental analysis;
- Comminution;
- Column Leach;
- Gravity recoverable gold (GRG);
- Flotation;
- Leaching and Cyanidation.

1.6.2 Testwork (2021)

The 2021 metallurgical testwork program was completed by Base Metallurgical Laboratories (BML) in Kamloops, British Columbia, Canada. The samples for the metallurgical program were selected from the ATO Deposit. BML and DRA performed a comprehensive analysis of the ore types within the deposit and concluded that the samples tested were representative of the overall deposit. This testwork program focused on creating saleable lead, zinc, and pyrite concentrates.

1.6.2.1 Head Assays and Mineralogy Characterisation

Head assays and mineralogical analysis were carried out on subsamples of the master composite and variability samples. Head assays for Au ranged between 0.86 and 1.79 g/t. The head sample assays of the precious and base metals are shown in Table 1.1.

Table 1.1 – Head Sample Assays

	Element (Average)					
	Pb	**Zn**	**Fe**	**S**	**Ag**	**Au**
Method	FAAS	FAAS	FAAS	LECO	FAAS	FAAS
Units	%	%	%	%	g/t	g/t
ATO-62	0.79	2.45	2.70	3.56	12	1.79
ATO-71	0.97	1.87	2.49	3.75	14	1.64
ATO-97	1.54	1.61	2.95	3.01	10	1.60
ATO-137	0.75	1.30	1.77	2.82	7	1.71
ATO-139	0.80	1.83	3.16	3.55	4	1.01
ATO-149	1.05	2.51	3.73	4.06	5	0.86
ATO-Master	1.05	1.99	2.80	3.47	9	1.45

1.6.3 COMMINUTION

As part of XPS's Phase 2 Program, the grindability characterisation study also included the J-K drop-weight as well as the Bond ball mill grindability tests. The three samples were labelled as Master, Pipe 2, and Pipe 4 Composites.

Based on the resistance to impact breakage (A x b), resistance to abrasion breakage (ta) and its BWi value; of the three composite samples, the Master Comp was the hardest, whereas Pipe 2 Comp and Pipe 4 Comp are considered soft to moderately soft. The results are summarised in Table 1.2.

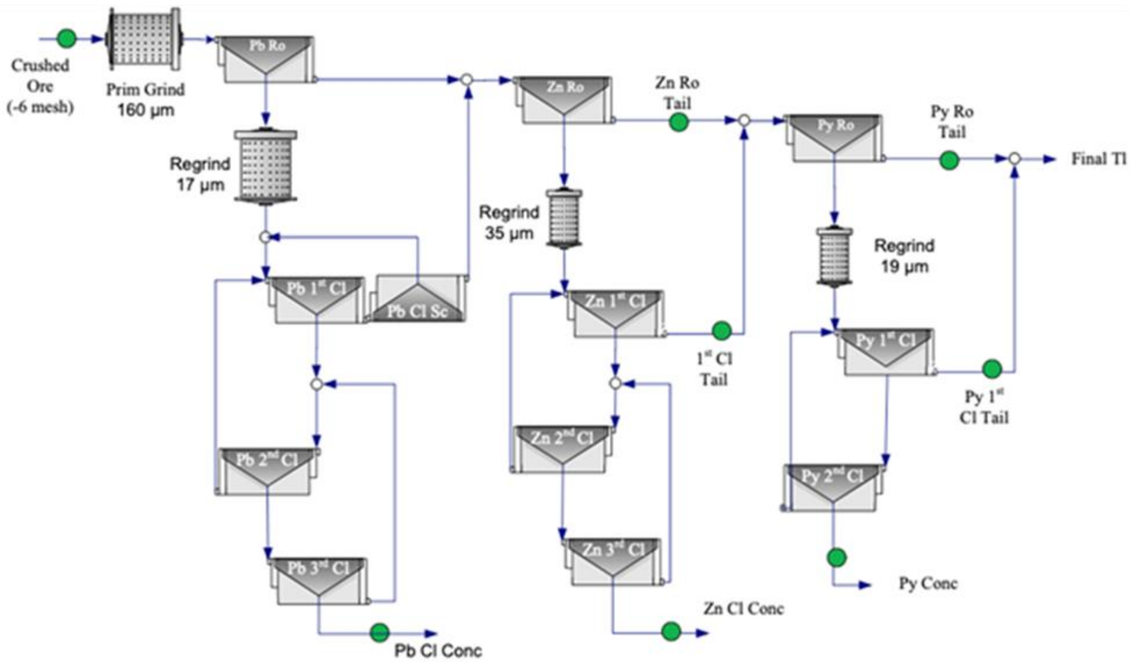
Table 1.2 - Grindability Test Summary

Sample Name	**Relative Density**	**JK Parameter A x b**	**JK Parameter ta**	**BWi (kWh/t)**
Master Comp	2.75	50.9	0.39	15.5
Pipe 2 Comp	2.75	62.2	0.66	15.6
Pipe 4 Comp	2.67	95.3	0.56	14.6

1.6.4 Flotation

LCT testwork focused on testing the amenability of the ATO ore based on the flowsheet presented in Figure 1.1, where pyrite flotation was added to obtain separate Pb, Zn, and Py concentrates.

Figure 1.1 – Updated ATO Phase 2 Flowsheet - Pb-Zn-Py Concentrate Products



Base Metallurgical Laboratories, June 2021

The testwork confirmed high recoveries of Pb and Zn, and reasonable recoveries for Au and Ag.

1.6.5 Recovery Estimates

1.6.5.1 Lead (Pb) Recovery

After analysing the flotation results, a Pb recovery relationship could not be determined and therefore a fixed value of 82.5% was used. This was the average of all the lead recovery results from the Locked Cycle Tests (LCTs) conducted. This fixed value was estimated from the average between the master composite and variability samples. For the variability samples the average was calculated by using the masses of samples based on the master composite mass splits.

The fixed Pb, Au, and Ag recovery values are shown as follows:

$$Pb\ Recovery\ \% = 82.5\ ;\ Fixed\ Value$$

$$Pb\ Conc\ Gold\ Rec\ \% = 41.2\ ;\ Fixed\ Value$$

$$Pb\ Conc\ Silver\ Rec\ \% = 45.6\ ;\ Fixed\ Value$$

1.6.5.2 *Zinc (Zn) Recovery*

A Zn recovery relationship was also unable to be determined and therefore a fixed value was used. This fixed value was estimated from the average between the master composite and variability samples. For the variability samples, the average was calculated by using the masses of samples based on the master composite mass splits.

The fixed Zn, Au, and Ag recovery values are shown as follows:

$$Zn\ Recovery\ \% = 85.9\ ;Fixed\ Value$$

$$Zn\ Conc\ Gold\ Rec\ \% =\ 14.1\ ;Fixed\ Value$$

$$Zn\ Conc\ Silver\ Rec\ \% =\ 18.2\ ;Fixed\ Value$$

1.6.5.3 *Pyrite (Py) Recovery*

Regarding Au and Ag recoveries in the Py concentrate, average values between the variability and master composite samples were used. These are shown as follows:

$$Py\ Conc\ Gold\ Rec\ \% = 23.9\ ;Fixed\ Value$$

$$Py\ Conc\ Silver\ Rec\ \% = 8.8\ ;Fixed\ Value$$

1.6.6 METALLURGICAL VARIABILITY

The metallurgical testwork completed to date is based on samples which adequately represent the variability of the ATO deposit; however, the selection of the samples was made prior the establishment of the latest mine plan.

Mineralogical analysis of the various composite and variability samples has shown that the ATO deposit is reasonably homogenous with respect to mineralogy. The exception is sample ATO-97 which showed high contents of dolomite which appear to impact detrimentally on flotation performance.

1.6.7 DELETERIOUS ELEMENTS

Pb, Zn, and Py concentrates will be subject to penalty conditions should significant grades of Zn, Pb, Hg, Sb, Bi, and As be present in high levels in the concentrates. Section 19 explores the impact of these elements which are present in the concentrates. The concentrates produced are shown to be very clean concentrates with no presence of detrimental elements leading to penalties.

1.7 Mineral Resource Estimate

The resource estimate was prepared by Mr. Robin A. Rankin, MSc DIC MAusIMM CP(Geo), Principal Consulting Geologist and operator of GeoRes. The author states the CIM equivalence of JORC (accepted as a foreign Code by NI 43-101) reporting terms used in the Resource classification.

The resource estimate is summarised in Table 1.3 and Table 1.4 by class and deposit. The effective date of resource estimate is 18 February 2021.

Table 1.3 – ATO 2021 Measured & Indicated In-Situ Mineral Resources - By Class

ATO - JORC Classified Resources by Deposit. Reported 18 February 2021 (V3).												
CLASS BY DEPOSIT	Deposit	Tonnes		Grades					Metal		
				Au	Ag	Pb	Zn	AuEq	Au	Ag	AuEq
		(M t)		(g/t)	(g/t)	(%)	(%)	(g/t)	(k oz)	(k oz)	(k oz)
MEASURED	ATO1	9.8		1.13	6.76	0.70	1.08	1.95	357	2,133	616
	ATO2	1.7		0.42	3.84	0.51	0.76	1.00	23	205	53
	ATO4	7.5		1.32	12.50	0.29	0.52	1.83	319	3,024	443
	Mungu	4.9		1.31	43.47	0.01	0.03	1.93	208	6,912	308
	TOTAL	**23.9**	**58%**	**1.18**	**15.95**	**0.41**	**0.66**	**1.84**	**907**	**12,274**	**1,419**
INDICATED	ATO1	5.4		0.79	5.77	0.64	1.11	1.60	138	1,003	278
	ATO2	1.5		0.45	4.02	0.48	0.76	1.02	22	193	49
	ATO4	8.2		0.95	15.28	0.22	0.41	1.44	250	4,006	376
	Mungu	2.6		0.88	35.63	0.01	0.03	1.39	74	3,004	117
	TOTAL	**17.7**	**42%**	**0.85**	**14.44**	**0.34**	**0.60**	**1.44**	**483**	**8,206**	**819**
MEAS + IND	ATO1	15.2	37%	1.01	6.41	0.68	1.09	1.83	495	3,137	893
	ATO2	3.1	8%	0.44	3.93	0.49	0.76	1.01	44	398	102
	ATO4	15.7	38%	1.13	13.95	0.26	0.46	1.62	569	7,029	819
	Mungu	7.6	18%	1.16	40.75	0.01	0.03	1.74	282	9,916	424
MEAS+IND	**TOTAL**	**41.6**		**1.04**	**15.31**	**0.38**	**0.63**	**1.67**	**1,390**	**20,479**	**2,238**

Table 1.4 ATO 2021 Inferred In-Situ Mineral Resources - By Class

ATO - JORC Classified Resources by Deposit. Reported 18 February 2021 (V3).											
CLASS BY DEPOSIT	Deposit	Tonnes	Grades					Metal		
			Au	Ag	Pb	Zn	AuEq	Au	Ag	AuEq
		(M t)	(g/t)	(g/t)	(%)	(%)	(g/t)	(k oz)	(k oz)	(k oz)
INFERRED	ATO1	1.1	0.51	4.44	0.54	1.19	1.30	19	164	48
	ATO2	0.5	0.28	5.70	0.70	1.34	1.21	4	84	18
	ATO4	2.3	0.61	14.91	0.19	0.35	1.03	45	1,098	76
	Mungu	1.7	0.82	25.05	0.01	0.02	1.18	45	1,386	65
INFERRED	**TOTAL**	**5.6**	**0.62**	**15.13**	**0.25**	**0.50**	**1.15**	**113**	**2,732**	**208**

The mineral resources are derived from a block model, and a basic block size of 5 m was chosen to suit the typical 30 * 30 * 2 m sampling. A differentiating parameter of the block models was the choice of primary block size (without any further sub-blocking) to accommodate both the data spacing and the shapes of the deposits:

- Pipe 1, 2, 4 model: 5 * 5 * 5 m;
- Mungu: 2 * 5 * 5 m.

Lower grade cut-offs used here were applied to the AuEq variable, were stipulated by Steppe, and were:

- Oxide 0.15 g/t AuEq;
- Transitional 0.40 g/t AuEq;
- Fresh 0.40 g/t AuEq.

Bulk densities were applied by oxidation level, were described above, and were:

- Oxide 2.46 t/m³;
- Transitional 2.59 t/m³;
- Fresh 2.64 t/m³.

The Author QP was not aware of any other factors (excluding those specifically mentioned below here), including environmental, title, economic, market or political, which could generally or in-particularly influence the Resources reported here for the Project.

The mineral resource estimate may be impacted by several factors, including but not limited to:

Grade cut-off:

- In the Author QP's experience the cut-offs used here are comparatively low.
- Raising cut-offs would reduce the Resources. The Author QP has not studied the relationship between cut-off and Resources – but does not believe that raising the cut-off slightly (say to 0.5 g/t AuEq) would reduce Resources significantly.
- However, the Author QP accepts the lower grade cut-offs supplied by Steppe Gold believing that the down-stream mining and extraction analyses performed by Steppe Gold justify the values economically.

Bulk density:

- Actual bulk densities could prove to be different and could thus alter Resources.
- However, the Author QP does not consider that density could be significantly different to that used here, and therefore would not have a significant influence on Resources.

Gold equivalent:

- The gold equivalent calculation was based on international metals prices to mid-January 2021.
- The calculation is most susceptible to changes in the price of gold.
- The Author QP has not studied the relationship between prices, gold equivalent and Resources, but does not believe that the scale of price changes normal within the recent past (say a year) would have a significant effect on Resources.

1.8 Mineral Reserve Estimate

The mineral reserves estimate with an effective date of June 30, 2021 for the Project is based on the parameters and steps outlined in this section as well as the resource estimate presented in Section 14. The mineral reserves for the ATO and Mungu pits are estimated at 26.4 Mt of Proven and Probable Reserves at a grade of 1.86 g/t AuEq, based on the marginal cut-off grades described in Section 15.3. To access the ore, a total of 69.2 Mt of waste rock will need to be extracted, resulting in a 2.62 stripping ratio detailed in Section 16.3.

Table 1.5 – Mineral Reserve Estimate, Effective June 30, 2021

Category	Material	Ore	Grades					Contained Metal		
			AuEq	Au	Ag	Pb	Zn	Au	Ag	AuEq
		(kt)	(g/t)	(g/t)	(g/t)	(%)	(%)	(k oz)	(k oz)	(k oz)
ATO										
Proven	Oxide	1,618	1.54	1.45	12.81	0.54	0.40	75	666	80
	Transition	6,604	2.16	1.34	10.35	0.51	0.86	285	2,198	459
	Fresh	6,673	2.04	1.17	6.80	0.55	1.02	251	1,459	438
Probable	Oxide	1,035	1.19	1.07	16.36	0.33	0.26	36	544	40
	Transition	3,721	1.81	1.10	14.23	0.36	0.67	132	1,702	217
	Fresh	5,669	1.67	0.92	9.03	0.44	0.83	168	1,646	304
Proven & Probable	*Oxide*	*2,652*	*1.40*	*1.30*	*14.19*	*0.46*	*0.35*	*111*	*1,210*	*119*
	Transition	*10,324*	*2.03*	*1.25*	*11.75*	*0.46*	*0.80*	*415*	*3,900*	*674*
	Fresh	*12,342*	*1.87*	*1.06*	*7.82*	*0.50*	*0.93*	*421*	*3,103*	*742*
	Subtotal	*25,318*	*1.89*	*1.16*	*10.09*	*0.48*	*0.81*	*944*	*8,213*	*1,538*
Mungu										
Proven	Oxide									
	Transition									
	Fresh									
Probable	Oxide	289	1.06	0.83	30.52	0	0	8	284	10
	Transition	385	1.22	0.68	38.18	0	0.01	8	473	15
	Fresh	412	1.10	0.53	39.62	0	0.02	7	525	15
Proven & Probable	*Oxide*	*289*	*1.06*	*0.83*	*30.52*	*0*	*0*	*8*	*284*	*10*
	Transition	*385*	*1.22*	*0.68*	*38.18*	*0*	*0.01*	*8*	*473*	*15*
	Fresh	*412*	*1.10*	*0.53*	*39.62*	*0*	*0.02*	*7*	*525*	*15*
	Subtotal	*1,086*	*1.13*	*0.66*	*36.68*	*0*	*0.01*	*23*	*1,281*	*39*

Category	Material	Ore	Grades					Contained Metal		
			AuEq	Au	Ag	Pb	Zn	Au	Ag	AuEq
		(kt)	(g/t)	(g/t)	(g/t)	(%)	(%)	(k oz)	(k oz)	(k oz)
Combined (ATO and Mungu)										
Proven	Oxide	1,618	1.54	1.45	12.81	0.54	0.40	75	666	80
	Transition	6,604	2.16	1.34	10.35	0.51	0.86	285	2,198	459
	Fresh	6,673	2.04	1.17	6.80	0.55	1.02	251	1,459	438
Probable	Oxide	1,324	1.16	1.01	19.45	0.26	0.20	43	828	49
	Transition	4,105	1.75	1.06	16.47	0.33	0.61	140	2,174	231
	Fresh	6,081	1.63	0.90	11.10	0.41	0.77	176	2,170	319
Proven & Probable	*Oxide*	*2,942*	*1.37*	*1.25*	*15.80*	*0.41*	*0.31*	*118*	*1,494*	*130*
	Transition	*10,709*	*2.00*	*1.23*	*12.70*	*0.44*	*0.77*	*423*	*4,373*	*689*
	Fresh	*12,753*	*1.85*	*1.04*	*8.85*	*0.48*	*0.90*	*426*	*3,629*	*759*
	Total	**26,404**	**1.86**	**1.14**	**11.18**	**0.46**	**0.78**	**968**	**9,491**	**1,579**

Notes

1. Mineral Resource Estimate was estimated by the Resources QP
2. ATO and Mungu Mineral Reserves are effective as of June 30, 2021
3. Mineral Reserves are included in Mineral Resources
4. Mineral Reserves are reported in accordance with CIM and NI 43-101 guidelines
5. Ore dilution is 3% and ore loss is 2%
6. Contained metal estimates have not been adjusted for metallurgical recoveries
7. The open pit mineral reserves are estimated using a cut-off grade of 0.42 g/t AuEq for oxide material and 0.45 g/t AuEq for transition and fresh material
8. Mineral Reserves are contained within an optimised pit shell based on a gold price of $1,610 USD per ounce
9. A conversion factor of 31.103477 grams per troy ounce and a conversion factor of 453.59237 grams per pound are used in the resource and reserves estimates
10. AuEq has been calculated using the following metal prices: $1,610/oz gold, $21/oz silver, $1,970/t lead, $2,515/t zinc
11. Oxide AuEq calculation: $AUEQ_{(g/t)} = Au_{(g/t)} + \frac{Ag_{(g/t)}\times 21\times 0.4}{1,610\times 0.7}$
12. Transition and fresh AuEq calculation: $AuEq_{(g/t)} = Au_{(g/t)} + \frac{Ag_{(g/t)}\times 21\times 0.858}{1,610\times 0.8} + \frac{Pb_{(\%)}\times 1,970\times 0.88}{1,610\times 0.8} + \frac{Zn_{(g/t)}\times 2515\times 0.88}{1,610\times 0.8}$
13. Totals may not match due to rounding
14. The Mineral Reserves are stated as dry tonnes processed at the crusher
15. The QP is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially impact the Mineral Reserves Estimate

Table 1.6 summarises the mineral reserves and reconciles with the resources estimate.

Table 1.6 – Resource and Reserve Reconciliation

Reserves Reported 30 June 2021							**Resources** Reported 18 February 2021							**Reserves vs Resources**
Category	**Material**	**Cut-off AuEq**	**Ore**	**Grades AuEq**	**Grades Au**	**Grades Ag**	**Category**	**Material**	**Cut-off AuEq**	**Ore**	**Grades AuEq**	**Grades Au**	**Grades Ag**	**Ore**
		(g/t)	(kt)	(g/t)	(g/t)	(g/t)			(g/t)	(kt)	(g/t)	(g/t)	(g/t)	(kt)
ATO														
Proven + Probable	*Oxide*	*0.42*	*2,652*	*1.4*	*1.3*	*14.19*	*Measured + Indicated*	*Oxide*	*0.15*	*7,400*	*1.29*	*0.88*	*10.01*	*-4,748*
	Transition	*0.45*	*10,324*	*2.03*	*1.25*	*11.75*		*Transition*	*0.40*	*9,700*	*1.96*	*1.27*	*11.91*	*624*
	Fresh	*0.45*	*12,342*	*1.87*	*1.06*	*7.82*		*Fresh*	*0.40*	*16,700*	*1.64*	*0.92*	*8.15*	*-4,358*
ATO Subtotal			***25,318***	***1.89***	***1.16***	***10.09***	***ATO Subtotal***			***34,000***	***1.66***	***1.01***	***9.64***	***-8,682***
Mungu														
Proven + Probable	*Oxide*	*0.42*	*289*	*1.06*	*0.83*	*30.52*	*Measured + Indicated*	*Oxide*	*0.15*	*500*	*0.87*	*0.57*	*21.25*	*-211*
	Transition	*0.45*	*385*	*1.22*	*0.68*	*38.18*		*Transition*	*0.40*	*500*	*1.18*	*0.66*	*36.50*	*-115*
	Fresh	*0.45*	*412*	*1.1*	*0.53*	*39.62*		*Fresh*	*0.40*	*6,600*	*1.85*	*1.24*	*42.61*	*-6,188*
Mungu Subtotal			***1,086***	***1.13***	***0.66***	***36.68***	***Mungu Subtotal***			***7,600***	***1.75***	***1.16***	***40.80***	***-6,514***
Combined														
Proven + Probable	*Oxide*	*0.42*	*2,942*	*1.37*	*1.25*	*15.8*	*Measured + Indicated*	*Oxide*	*0.15*	*7,900*	*1.26*	*0.86*	*10.72*	*-4,958*
	Transition	*0.45*	*10,709*	*2*	*1.23*	*12.7*		*Transition*	*0.40*	*10,200*	*1.92*	*1.24*	*13.11*	*509*
	Fresh	*0.45*	*12,753*	*1.85*	*1.04*	*8.85*		*Fresh*	*0.40*	*23,300*	*1.70*	*1.01*	*17.92*	*-10,547*
Total			**26,404**	**1.86**	**1.14**	**11.18**	**Total**			**41,600**	**1.67**	**1.04**	**15.31**	***-15,196***

The variation in tonnage (for all ores) is due to lower cut-off grades used in the Resources estimate and gold equivalent grades calculation based on different metal prices.

For the oxides, the Resources and Reserves estimates are reported at different time periods, and the oxide ore mined between the reporting periods is excluded from the Reserves estimate.

For the transition ore, there is slightly different interpretation of boundary surfaces for each material.

In terms of fresh ore, the variation in tonnage is due to the Resources estimate not being constrained by a pit shell. The Resources estimate QP believes the low cut-off grades used reflected relatively shallow mining of predominantly oxidised material and a bulk low-cost extractive process (assumptions may not apply to deeper mining of fresh rock, i.e. Mungu).

The Resources QP is of the opinion that reporting of deep mineralisation should use a higher cut-off grade to reflect potential underground mining.

An opportunity exists to gain up to 1.5 M tonnes of reserve inside the existing pit design at ATO by drilling exploration holes to bring the inferred ore to minimum indicated resource level.

1.9 Mining Method

The Project mineral reserves were estimated for the ATO and Mungu Pits based on the economic and pit design parameters detailed in Section 15. The total tonnage to be mined from these pits is estimated at 96.5 million tonnes, ore and waste combined. The material will be mined over a period of approximately 12.5 years.

The mining method selected for the Project is a conventional open pit operation with rigid body mining trucks, hydraulic excavators, and wheel loaders. The Project consists of two separated mining areas, namely ATO and Mungu.

A mine plan (or schedule) was prepared to estimate a probable production schedule for the Project and assess the mine equipment fleet requirements, as well as the mine capital and operating costs for the Project's financial model. The mine plan was based on a production rate of 1.2 Mtpa of oxide ore at the existing leach pad and 2.20 Mtpa of transition and fresh ores at the new mill.

Waste material mined from each of the Project pits will be stored in two waste stockpiles. The ATO stockpile is located West of the ATO Pit, and the Mungu stockpile is located West of the Mungu Pit.

The total material movement is presented in Figure 1.2 and the mine production schedule is presented in Table 1.7.

Figure 1.2 – Total Material Movement

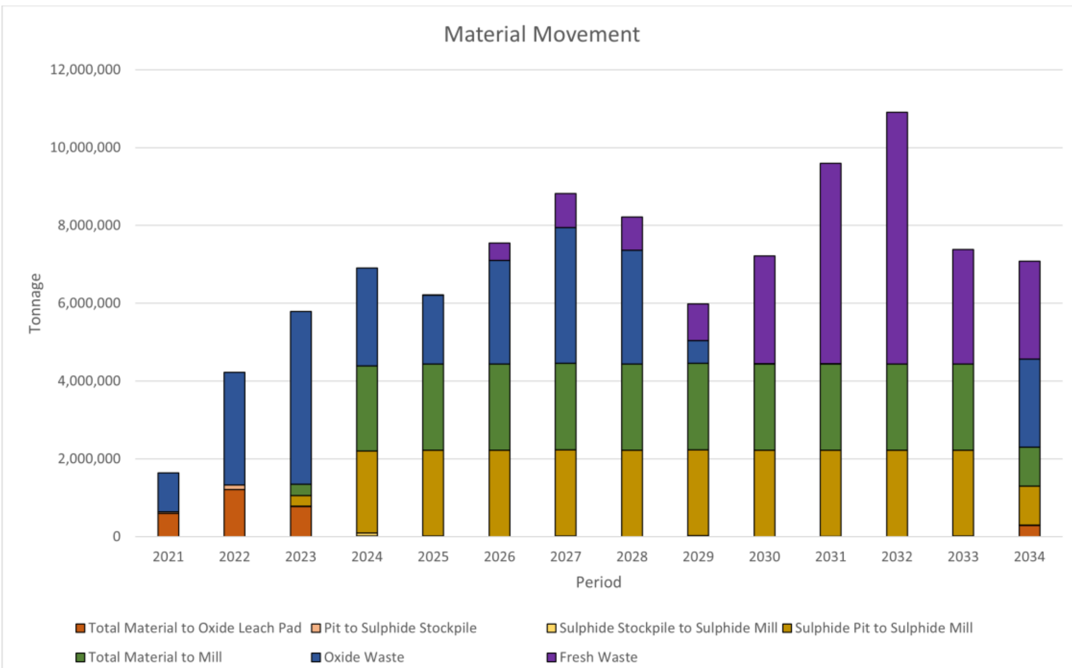


Table 1.7 – Mine Production Schedule (by Year)

Year	Ore						Waste	Total Mined	Stripping Ratio
	Tonnage (kt)	AuEq (g/t)	Au (g/t)	Ag (g/t)	Zn (%)	Pb (%)	(kt)	(kt)	(w/o)
2021[1]	643	1.38	1.22	14.75	0.35	0.40	994	1,637	1.55
2022	1,325	1.55	1.40	14.67	0.41	0.39	2,988	4,313	2.25
2023	1,047	1.61	1.28	11.31	0.46	0.52	4,814	5,861	4.60
2024	2,134	2.87	1.81	11.69	1.11	0.73	4,787	6,921	2.24
2025	2,201	2.21	1.35	14.27	0.86	0.47	5,037	7,239	2.29
2026	2,221	2.33	1.82	10.12	0.49	0.26	5,979	8,200	2.69
2027	2,236	1.97	1.40	13.62	0.50	0.28	5,979	8,215	2.67
2028	2,221	2.01	0.97	6.64	1.23	0.72	5,979	8,200	2.69
2029	2,224	1.89	0.76	5.57	1.41	0.71	5,979	8,203	2.69
2030	2,221	1.70	1.11	8.84	0.60	0.33	5,979	8,200	2.69
2031	2,208	1.60	0.87	9.54	0.80	0.42	5,979	8,188	2.71
2032	2,221	1.60	0.86	10.25	0.83	0.39	6,479	8,700	2.92
2033	2,204	1.46	0.63	6.30	0.98	0.53	2,987	5,192	1.36
2034[2]	1,290	1.12	0.62	31.92	0.13	0.06	4,746	6,036	3.68

1 Year 2021 represents the period of July 2021 to December 2021
2 Year 2034 represents approximately 5 months of production at the end of the mine life

1.10 Recovery Methods

In general, the overall Project comprises two distinct phases:

- Phase 1 – Heap Leach (Oxide Ore) - Completed and In Operation

  The oxide portion of the ATO Project process employs a conventional oxide heap leach flowsheet including crushing, heap leaching, and gold recovery facilities.

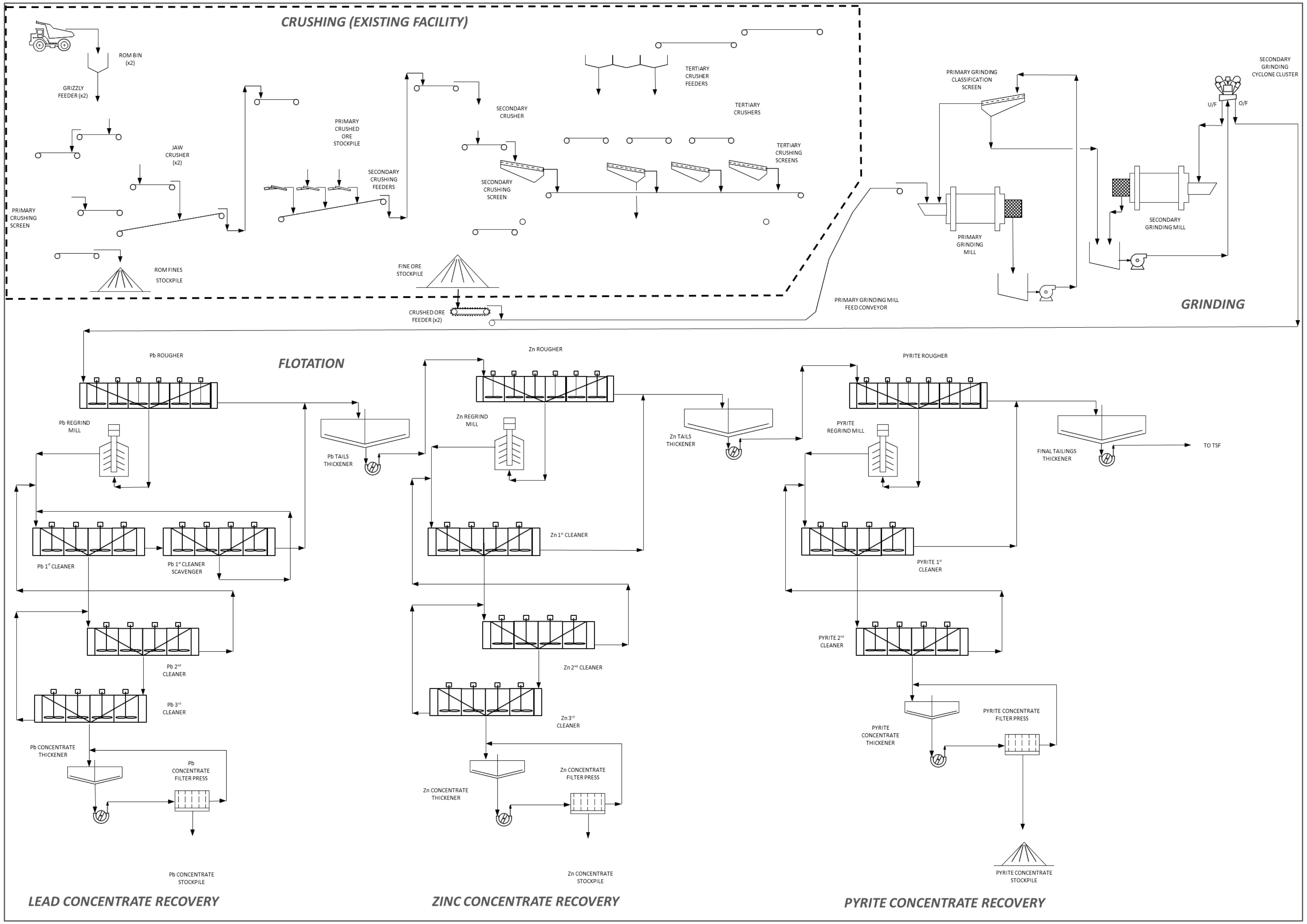
  Phase 1 of the Project has been operational since 2020 and remains operational as of the Effective Date of this Technical Report. The upgraded three-stage crushing system and ore storage facility (purchased by Steppe Gold and currently being installed) is part of Phase 1.

- Phase 2 – Concentrator (Fresh and Transition Ores) – Design in Progress

  The Phase 2 Concentrator will consist of collecting the crushed ore beneath the ore storage building, conveying to the concentrator, milling, flotation, and dewatering unit operations to produce saleable concentrates of lead, zinc, and pyrite. Tailings will be disposed of in the new Tailings Storage Facility (TSF).

An overall flow diagram summarising the Phase 2 concentrator plant and process flows is shown in Figure 1.3.

Figure 1.3 – ATO Phase 2 Project Overall Flowsheet



Source: DRA 2021

The existing crushing circuit is designed for a capacity of 2.2 Mtpa. The three-stage circuit reduces run-of-mine (ROM) material from an F_{100} of 800 mm to a P_{80} of 10 mm. The primary crushing circuit is utilised for an annual operating time of 5,694 h/a (65% utilisation) and operates in open circuit.

ROM material is dump-fed into ROM hoppers, installed in parallel. The primary crusher feed will be drawn from the ROM hoppers by vibrating grizzly feeders to feed primary jaw crushers, installed in parallel. Grizzly feeder undersize (U/S) is bypassed and conveyed to a primary crushing screen allowing for U/S material to be stockpiled.

For Phase 2, the crushed ore product will be reclaimed via one of two new apron feeders installed underneath the fine ore stockpile. Fresh feed is collected at a controlled rate to feed the concentrator feed conveyor. The concentrator is utilised for an annual operating time of 90% utilisation.

The grinding circuit consists of two-stage sequential grinding with a primary ball mill in closed circuit with a classification screen followed by a secondary ball mill in closed circuit with hydrocyclones. Hydrocyclone underflow is fed to the flotation process.

The flotation process is separated into Pb concentrate, Zn concentrate, and Py concentrate circuits to target each of the materials individually and maximize their recoveries. Process water is kept separate for the Pb concentrate and Zn concentrate circuits.

Grinding product is combined with process water and reagents and mixed thoroughly. The slurry is conditioned and fed to the Pb rougher flotation cells. The circuit consists of six (6) tank cells to provide sufficient flotation residence time.

Each product's flotation process has its own dedicated thickener; underflow from the final cleaner stage reports to this concentrate thickener, the underflow is pumped to a stock tank before compressed air filtration. Concentrate filter cake is stockpiled in product sheds, one each for Pb, Zn and Py concentrate, and fed to transport trucks via front end loader. Trucks are weighed via a truck scale prior to shipment.

The tailings thickener receives the following feed streams:

- Py rougher tailings, and
- Py cleaner tailings.

These streams are combined in the thickener feed well where flocculant is added to facilitate solids settling. Final tailings thickener overflow is recycled to the reclaim process water pond. Thickener underflow is pumped to the final tailings tank where the tailings are pumped to the TSF. Water from the TSF is reclaimed back to the reclaim process water pond to minimise fresh water make-up.

1.11 Project Infrastructure

1.11.1 Existing Infrastructure

The ATO mine has been in production since 2020 and has the necessary infrastructure required to support the open pit mining operation. This includes, but is not limited to, ADR plant, laboratory, fuel storage, chemical storage, power supply, water supply, heap leach facilities and ponds, camp, open pit mining fleet, waste facility, and necessary offices, warehouses, and workshops to sustain the current operation.

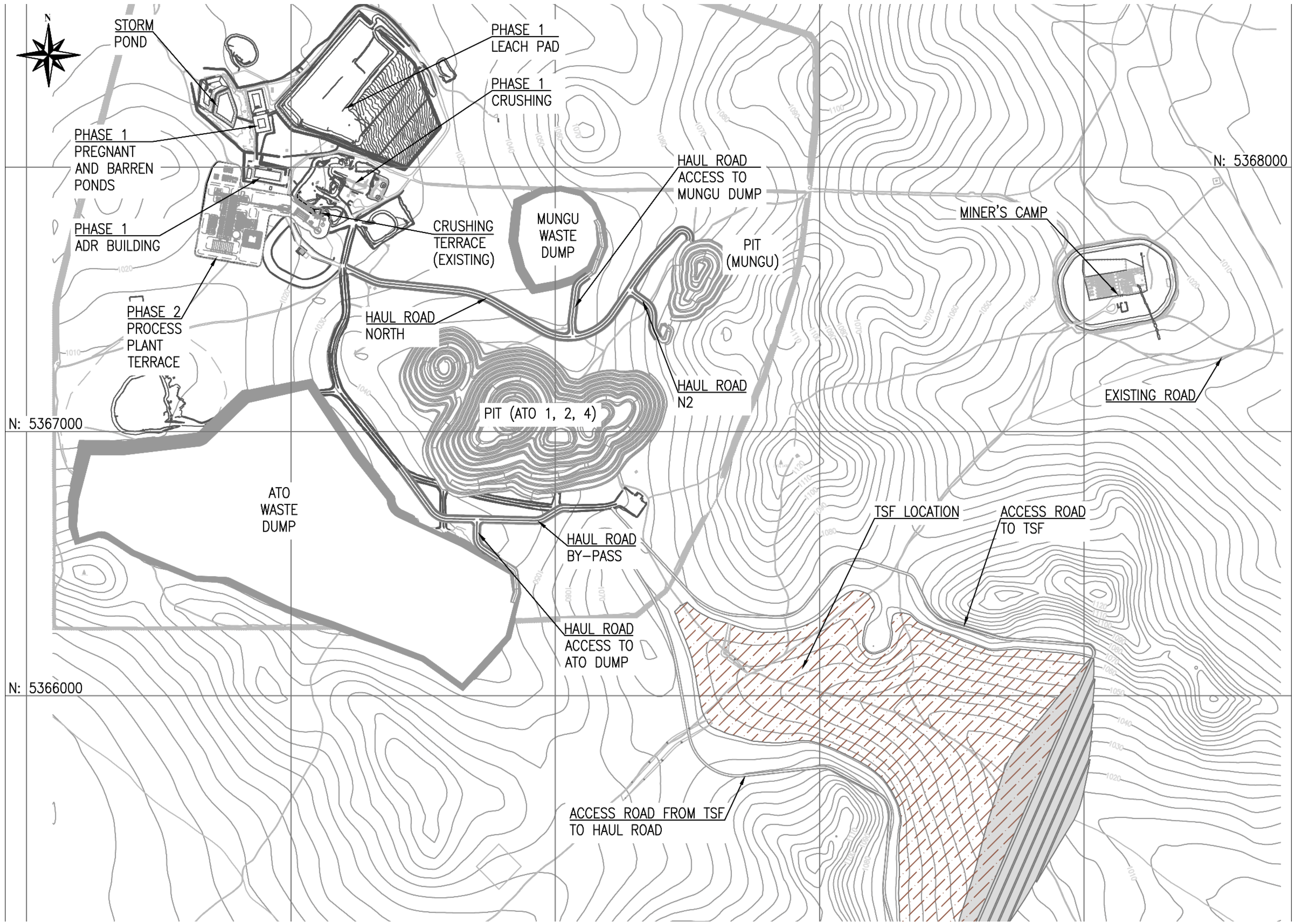
1.11.2 Overview

For the Phase 2 Expansion Project, this section describes the main Project elements related to process, followed by support infrastructure. Figure 1.4 illustrates all existing and planned infrastructure and locations of the plant and mines and Figure 1.5 depicts the process plant area.

It should be noted that no geotechnical investigations have been performed to characterise the ground condition for foundation design nor for any borrow materials for any of the facilities presented in this Report. As no geotechnical information was available at the time of developing the design, a field and laboratory investigation program will need to be carried out as part of the next project phase to confirm the assumptions made, or if changes to the design need to be made. Further details are geotechnical requirements are provided in this Report.

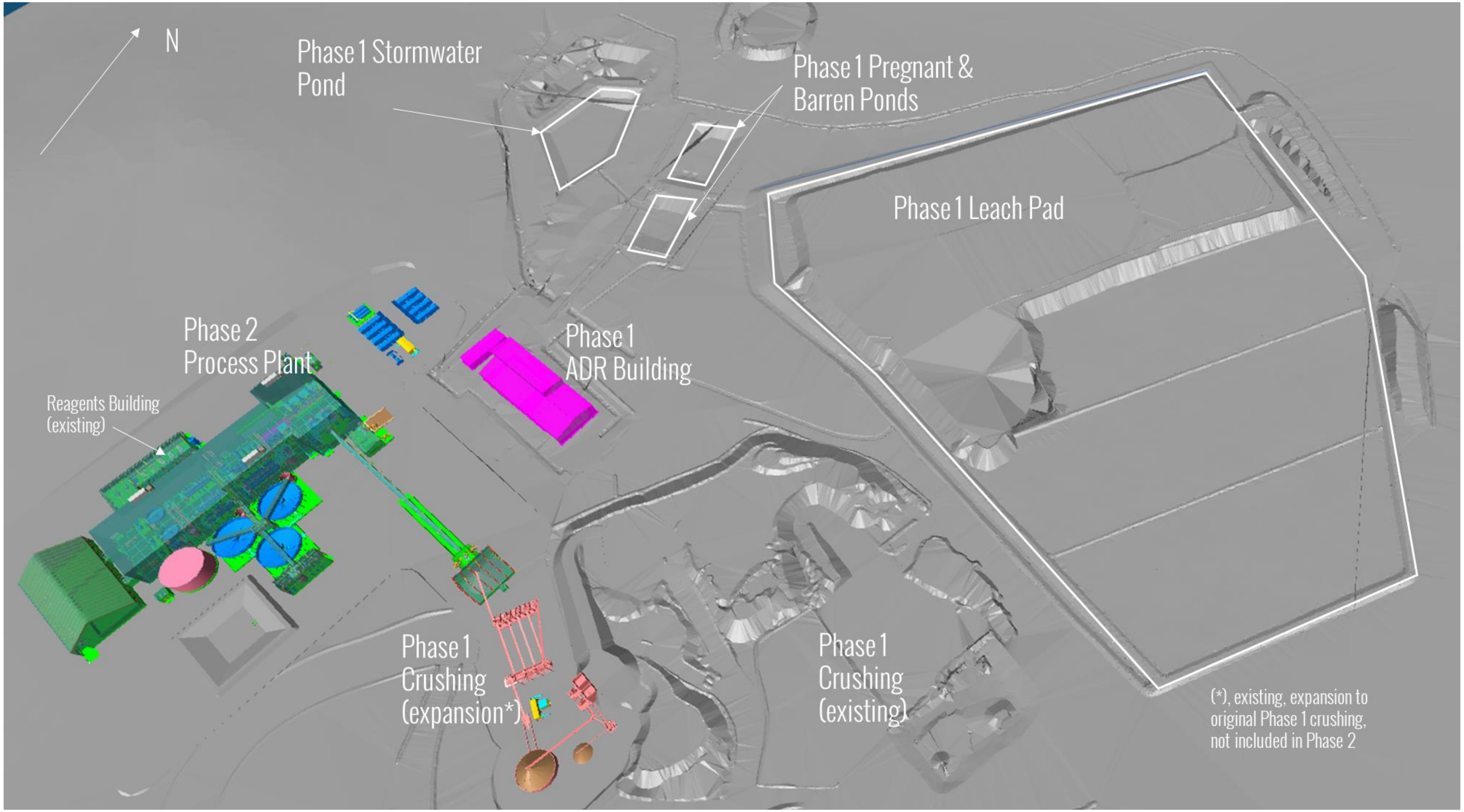
It is understood that certain elements of the site infrastructure which are currently under construction for use in Phase 1 will also be utilised in the Phase 2 operation, specifically the crushing circuit and the reagents building.

Figure 1.4 - General Overall Site Plan



Source: DRA 2021

Figure 1.5 – Process Plant Layout



Source: DRA 2021

Other ancillary areas and buildings include a fuel tank farm (existing 4 x 50,000 L tanks plus 8 new), camp for 300 staff, and explosives storage.

In terms of water, five water circuits (Raw, Potable, Fire, Gland, and Process) have been developed to support the requirements of the plant and surrounding infrastructure.

The mine access road connects the Project site to Choibalsan city. The road is constructed with gravel as its base and it is assumed to be constructed to carry normal loads able to sustain delivery of materials and equipment and transport outgoing products. The new process plant site will have internal gravel roads to allow access to the different buildings. Approximately 3 km of new gravel haul roads will connect new pits with existing pits A new 6 km long gravel road will provide access around the TSF.

The TSF is located in a south-east facing valley approximately 2 km south-east of the pit. Its location is indicated in Figure 1.4.

The TSF will be a high-density polyethylene (HDPE) - lined cross-valley storage facility formed by multi-zoned earth fill embankment, encompassing a total footprint area (including basin area) of approximately 47 ha for Stage 1, and increasing to 112 ha for the final TSF. Downstream raise construction methods will be utilised for all TSF embankment lifts. The TSF embankment construction materials will be principally sourced from local borrow material within the basin area and mine waste.

1.12 Market Studies and Contracts

The ATO Project is an operating site producing a readily saleable commodity in the form of gold doré bars. Doré is sent via secure transportation to a refinery for further refining.

Steppe Gold sells its gold production directly to the Mongolian government at spot price. Two types of doré are produced:

- First doré contains approximately 70% Au by weight and the remaining 30% is a mixture of Ag, base metals and Fe.
- Second doré is Ag produced and sold separately.

All the doré is transported to the Central Bank of Mongolia (Mongolbank). The Bank of Mongolia announces the official Au and Ag rates for the day using the London Metal Exchange (LME) closing rate from the previous day.

For the Phase 2 Expansion Project, Pb and Zn metals are prime indicator of Pb and Zn concentrates. Steppe Gold will produce and sell its concentrates (Pb, Zn, and Py) for the Project.

The research group (CRU) expects global lead consumption to grow at a compounding average growth rate (CAGR) of 2.09% between 2020 and 2025, reaching 13.3 Mt in 2025. Europe and China are expected to account for about ~50% of growth in global demand by 2025. Thailand, Vietnam, and Indonesia are set to drive lead demand in Southeast Asia, which is forecast to increase from 331 kt in 2020 to 414 kt in 2025.

Zn prices, traded on the London Metal Exchange (LME), have recovered to above US $3,000/t in August 2021, up 66% from the multi-year lows reached in March 2020. The price expectations for the remainder of 2021 are expected to average of US $2,875/t for the year.

According to S&P, Zn price forecasts are set to average of US $2,885/t in 2022 and $2,858/t in 2023 with a medium-term average price of US $2,935/t in 2025.

Due to the stricter enforcement of environmental standards in China, CRU estimates that Py concentrate demand will decline to 9.6 Mt in 2025.

Although Zn and Pb concentrates are the main source of revenue for the Phase 2 Expansion Project, Py concentrate is forecasted to contribute additional revenue.

Steppe Gold entered into a metals purchase and sale agreement (the “Stream Agreement”) dated August 11, 2017 with Triple Flag International to sell Au and Ag produced from the ATO Project and was amended on September 30, 2019. Steppe Gold also has a number of contracts, agreements and/or purchase orders in place for supply and services that are material to the operation.

1.13 Environmental Studies, Permitting and Social or Community Impact

Steppe Gold has conducted stakeholder and community participatory regular/routine environmental monitoring program at the ATO Project site and surrounding areas, and reporting to relevant authorities and local communities addressing the monitoring and control impacts on air, water, land/soil and biodiversity.

The General Environmental Impact Assessment (GEIA) was completed and approved by Ministry of Environment and Tourism of Mongolia (MMET). The environmental and social impacts are summarised in the report, and include changes to topography from mining operations, impacts on vegetation from mine clearing, impacts on fauna from land clearing, surface water hydrology impacts from interrupted natural drainage and soil and water contamination from mine development.

Steppe Gold has conducted water resource studies from 2017 to 2019 and received water resource statements from the relevant authorities and received land use permits for mining, construction, other infrastructures sites from local authorities.

The mine minerals waste handling plan has been developed to ensure that the management of mining activities and the implementation of environmental and social management plans and mine closure at the ATO Project will be conducted according to best practice methodologies to eliminate the potential for contamination.

The management of the ATO Project's significant environmental and social aspects and impacts is achieved through a suite of Management Plans that have been developed and is maintained such as Air Quality Management Plan and Water Resources Management Plan.

1.14 Capital and Operating Costs

1.14.1 Capital Cost Estimate

The Capital Cost Estimate (CAPEX) consists of direct and indirect capital costs as well as contingency. Provisions for sustaining capital are also included. Amounts for mine closure, rehabilitation of the site, and other specific items are excluded and further detailed in Section 21. The CAPEX is reported in United States Dollars ($, $ USD).

Table 1.8 presents a summary of the initial CAPEX by Major Area. Sustaining CAPEX is distributed over the LOM, separately indicated from the initial CAPEX. Certain Owner's Costs and contingency amounts are included in this CAPEX.

Table 1.8 – Initial CAPEX Summary

WBS	Major Area	Total Cost ($ USD)
2000	Mining - Open Pit	1,870,684
5000	Process Plant	75,185,111
6000	Tailings/ Reclaim Water and Water Treatment Facilities	13,485,178
7000	Power Plant & Distribution	1,701,307
9000	Indirect Costs	23,130,353
10000	Owner's Costs	1,150,307
20000	Project Contingency	11,477,060
	Total Costs	**128,000,000**
	Totals may not add up due to rounding.	

1.14.1.1 Sustaining CAPEX Summary

The sustaining capital requirements for the process plant include the purchase of spare parts for equipment, and replacement of equipment when required. The tailings area sustaining costs cover the expansion of the TSF as the tailings storage increases in area.

The sustaining capital costs are tabulated in Table 1.9, but are not included in the initial CAPEX.

Table 1.9 – Sustaining CAPEX Summary

WBS	Major Area	Total Cost ($ USD)
9320	Capital Spares (First year of operation only)	997,822
9330	Operational Spares (First year of operation only)	756,186
15100	Sustaining Capital	16,000,000
	Total Sustaining CAPEX	**17,754,008**
	Totals may not add up due to rounding.	

1.14.2 OPERATING COST ESTIMATE

The Operating Cost Estimate (OPEX) is presented in $ USD. DRA developed these operating costs in conjunction with Steppe Gold, with specific inputs provided by external consultants. The estimate includes mining, processing, and general and administration (G&A). The estimate has an accuracy of +30% -15%.

The OPEX is estimated at $668.6 M over the life of mine or $25.64/t of ore processed, with two years of operation for Phase 1 and 10.5 years of operation in Phase 2. The major project area over the LOM OPEX for the entire project for both Phases is summarised in Table 1.10.

Table 1.10 – Phases 1 and 2 - OPEX Summary by Major Area

Description by Area	Average Annual Costs (M USD/a)	Cost / t of Ore Processed (USD/t)	Total Cost LOM (M USD)
Mining	14.54	6.97	181.72
Process	27.47	13.17	343.35
G&A	11.49	5.51	143.58
Total[1]	**53.49**	**25.64**	**668.64**

1 Figures may not add due to rounding

1.15 Economic Analysis

The Project has been evaluated using discounted cash flow (DCF) analysis. Cash inflows were estimated based on annual revenue projections. Cash outflows consist of operating costs, capital expenditures, royalties, and taxes. The analysis considers two years of production in Phase 1, (existing operation) and 10.5 years of production through Phase 2.

The Net Present Value (NPV) of the Project was calculated by discounting back cash flow projections throughout the LOM to the Project's valuation date using three different discount rates

(5%, 8%, and 10%). The base case used a discount rate of 5%. The internal rate of return (IRR) and the payback period were also calculated.

Tables 1.11 and 1.12 summarise the economic/financial results of the Project for the base case for Phase 1 and Phase 2 as well as for Phase 2 respectively. All figures are in USD. For this Project, the Phase 1 and Phase 2 base case used a discount rate of 5%. After-Tax NPV is $232.08 M USD at a discount rate of 5%. The After-Tax IRR is 66.6% and the After-Tax payback on initial investment is 3.0 years.

Table 1.11 – Phase 1 and Phase 2 - Base Case Financial Results

Financial Results	Unit	Pre-Tax	After-Tax
NPV @ 5%	M USD	319.97	232.08
IRR	%	108.8	66.6
Payback Period	years	2.5	3.0

Table 1.12 – Phase 2 - Financial Results

Financial Results	Unit	Pre-Tax	After-Tax
NPV @ 5%	M USD	261.21	187.82
IRR	%	49.1	37.2
Payback Period	years	3.4	3.8

The sensitivities of the after-tax NPV and IRR were tested using alternate metal price assumptions and increases and decreases to CAPEX and OPEX.

Figure 1.6 indicates that the Project's after-tax viability is mostly vulnerable to a price forecast reduction, while being less affected by the under-estimation of capital and operating costs.

Figure 1.7 indicates that the IRR is more sensitive to variations in prices than CAPEX and OPEX.

Figure 1.6 – After-Tax $NPV_{5\%}$: Sensitivity to CAPEX, OPEX, and Prices

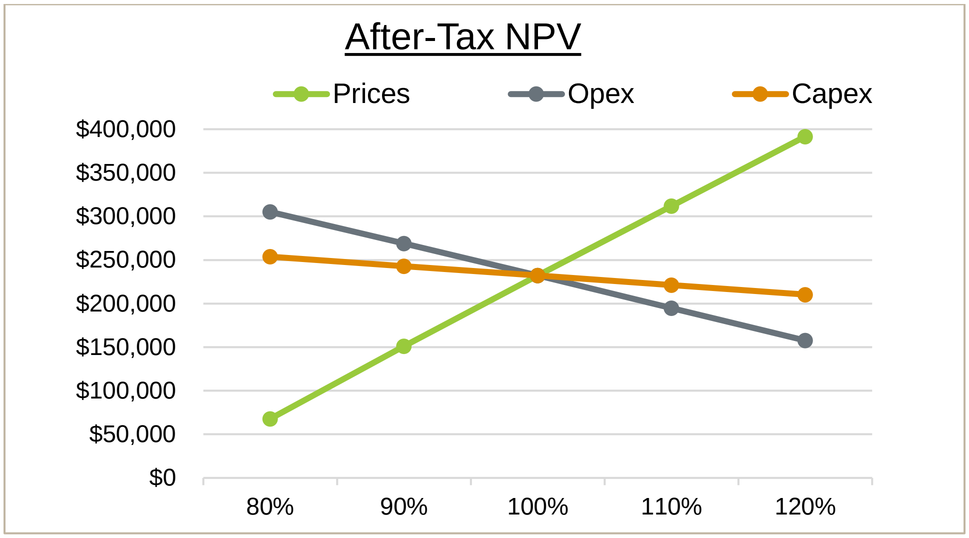
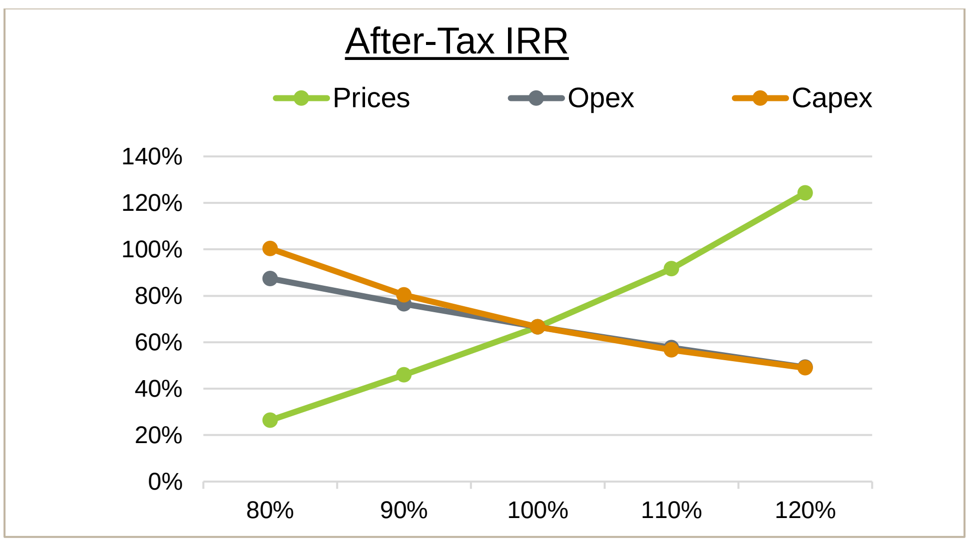


Figure 1.7 – After-Tax IRR: Sensitivity to CAPEX, OPEX, and Prices



1.16 Interpretation and Conclusions

1.16.1 Mineral Resource & Reserves

There are 22 Mt of ore reserves, between the ATO transition and fresh ores. An opportunity exists to gain up to 1.5 Mt of reserve inside the existing pit design at ATO by drilling exploration holes to bring the inferred ore to minimum indicated resource level.

There is also potential to expand the current ATO open pit by performing additional exploration drilling on inferred resources outside the pit shell, more particularly the eastern part of ATO-4 pipe.

As Mungu is a probable reserve, further metallurgical work is required to process this ore.

Mungu is open at depth and only a small portion can be mined economically by open pit. Therefore, there is an opportunity for the Mungu Pit to be expanded to an underground mine. Further studies should be carried out to confirm the potential for an underground mine.

1.16.2 Mining Methods

The Phase 2 Expansion Project, based on a mineral reserve estimate and associated mine plan for an open pit operation, has a mine life of approximately 12.5 years. This includes 26.4 Mt of ore at an average grade of 1.86 g/t AuEq and an average stripping ratio of 2.62. The ore material is contained within two major areas, namely ATO and Mungu. There are three ore material types, namely: oxide, transition, and fresh. Material is mined to achieve leach pad and mill targets of 1.20 Mtpa and 2.20 Mtpa respectively, while reducing waste mining requirements.

The mine will operate seven days a week, 24 hours a day (two 12-hour shifts per day) for a total of 330 operating days per year after adjusting for adverse weather as well as holidays. The mine haulage will be performed by a Contractor using 32-t trucks.

1.16.3 Mineral Processing and Recovery Methods

A comprehensive metallurgical testwork program was conducted on representative samples from the ATO deposit. The results from the program were used to define and optimize the process flowsheet for the economic extraction of Pb, Zn, and Au contained in the ATO ores. The flowsheet developed is considered a conventional sequential flotation flowsheet.

The Phase 1 process flowsheet continues to operate successfully with respect to extraction of Au from the ATO oxide deposit. A new crushing and screening plant will be installed to increase Au production rates by increasing oxide ore stacked production rates.

The same crushing and screening plant will be used for the Phase 2 plant for processing ATO fresh and transition ores. The sequential flotation flowsheet developed for production of separate Pb, Zn, and Py concentrates is considered robust.

1.16.4 INFRASTRUCTURE

The Phase 1 site layout has been expanded to accommodate the new deposit (Mungu) and process building. The Mungu ore and waste pits are located just north of the Phase 1 ATO deposits, and the new Phase 2 process plant just southwest of the Phase 1 leach pad.

The site consists of several process buildings at the process plant, a worker camp, explosives storage, water supply, and TSF. New roads would provide access to the new pits and buildings.

The power demand of the Steppe Gold site was assumed as peak load of 15 MW and average load of 12.5 MW. A hybrid solution Diesel-RES power plant (30 MW solar PV, 20 MW diesel, 4 MW/4 MWh BESS) was demonstrated to be the optimal low-cost solution for the Project.

The site will be supplied at 11 kV, 3 phase, 50 Hz from a power plant installed in the vicinity of the site. The power plant will consist of eight diesel generators each using LFO (diesel) fuel, in an N+2 configuration (6 in operation, one stand-by, one maintenance or repair).

The TSF is located in a south-east facing valley approximately 2 km south-east of the pit. It will be a high-density polyethylene (HDPE) - lined cross-valley storage facility formed by multi-zoned earth fill embankment, encompassing a total footprint area (including basin area) of approximately 47 ha for Phase 1, and increasing to 112 ha for the final TSF.

1.17 Recommendations

1.17.1 MINING AND GEOLOGY

Additional exploration drilling at ATO is recommended to bring the inferred resources to minimum indicated level. This has the potential to increase reserves up to 1.5 Mt within the existing pit envelope, and expend the pit limit boundary by drilling inferred resources outside the current pit design. The best potential option to expend the pit will be to drill the inferred resource of the East end of ATO-4.

1.17.2 PROCESS

As overall Au recovery in Phase 2 is relatively low, there is an opportunity to perform additional testwork aimed at improving overall Au recovery. It is also recommended to investigate how recovery and/or payability may be improved in the next phase of the Project.

Specifically, recommendations with respect to the fresh and transitional ore types include:

- Additional testwork to better understand fresh and transitional resources and development of a geometallurgical model including base metals and Au recovery relationships for the Pb, Zn, and Py concentrates produced, if they exist.
- Geo-metallurgical work to better understand the lithology and mineralogy of the ATO deposit particularly with respect to the proportion of ore which will be processed containing "high dolomite" material which produced variable flotation results;
- Further testwork targeting optimized Au and base metals recoveries through improved flotation performance or cyanidation of the flotation tailings stream.

1.17.3 Environment

Several existing environmental and social permits are already in place, and must be maintained and updated continuously.

The Environmental and Social Management Plan consists of several sub-level workplans, components of which are described in the Report. These plans should be prepared to minimise any potential project delays.

1.17.4 Infrastructure

As no geotechnical information was available at the time of developing the design, a field and laboratory investigation program will need to be carried out (as part of the next project phase) to characterise the ground condition for foundation design and any borrow materials for any of the facilities presented in this Report. These results will confirm the assumptions made, or determine if changes to the design need to be made. This program will include geophysics, drilling and test pitting in the designated area, as well as taking samples for geotechnical laboratory testing.

Phase 2 will require make-up water of approximately 4,800 m^3 per day. The source of this water should be confirmed, and that is has sufficient capacity for the Project.

2 INTRODUCTION

Following the successful completion of its Altan Tsagaan Ovoo (ATO) Phase 1 development and on-going crusher upgrades located in eastern Mongolia, Steppe Gold Limited (Steppe Gold or the Company) has initiated studies for the ATO Phase 2 Expansion Project (the “ATO Project” or the "project"). The Phase 2 Expansion, which is the subject of this Technical Report ("Report"), will expand gold production and produce saleable concentrates of lead, zinc and pyrite from the development of underlying fresh rock ores and the construction of a new and larger conventional processing facility.

DRA Global Limited (DRA) was mandated by Steppe Gold to lead the preparation of Feasibility Study - level engineering for the Altan Tsagaan Ovoo (ATO) Phase 2 Expansion Project, situated in eastern Mongolia, as shown in Figure 2.1. The property is located in the territory of Tsagaan Ovoo soum, in the Mongolian province of Dornod, 660 km east of the Mongolian capital of Ulaanbaatar and 120 km northwest of the provincial capital of Choibalsan.

Figure 2.1 – Map of Asia showing Location of ATO Mine Site

Source: DRA

DRA led the mine planning, mineral reserve estimate, metallurgy, and capital and operating cost estimating activities. The work was supported by additional leading consultants with expertise in various fields, including: GeoRes for mineral resource estimation, Ulzii Environmental LLC for hydrogeology, water quality, environmental resource management for social and environmental, Knight Piésold for tailings storage facility (TSF), and Base Metallurgical Laboratories Ltd. (BML) for metallurgical test work.

Steppe Gold is an international mineral resource company headquartered in Toronto, with exploration, development and production properties located in Mongolia. Steppe Gold is listed on the Toronto Stock Exchange under the symbol STGO.

In 2017, Steppe Gold acquired the ATO Project, and is currently the 100% owner of the mining license for the Project. That same year, a NI 43-101 Technical Report was prepared for Steppe Gold and Centerra Gold (the previous ATO Project owner) by the Mongolian company GSTATS Consulting LLC for Phase 1 of the ATO Project. In 2020, Steppe Gold completed the construction and commissioning for Phase 1, which consisted of an integrated oxide ore heap leach production facility which now produces 60,000 oz of gold annually.

A Techno-Economic Feasibility Study for this larger project was previously prepared in 2012 by Glogex LLC, a mine design and research company in Mongolia, for the previous owner (Centerra Gold).

Aspects of this previous report are referenced in various places in this Technical Report for Steppe Gold.

In general, the overall project comprises two (2) distinct phases:

Phase 1 – Heap Leach (Oxide Ore) - Completed

- The oxide portion of the ATO Project process employs a conventional oxide heap leach flowsheet including crushing, heap leaching, and gold recovery facilities.
- Phase 1 of the Project has been operational since 2020 and remains operational as of the Effective Date of this Technical Report. The upgraded three-stage crushing system and ore storage facility as well as the Reagent building (purchased by Steppe Gold and currently being installed) are considered part of Phase 1 of the Project.

Phase 2 - Expansion – Concentrator for Fresh and Transition Ores – Design in Progress

- Phase 2 will consist of collecting crushed ore from beneath the ore storage building, conveying to concentrator, milling, flotation, and dewatering unit operations to produce saleable concentrates of lead, zinc, and pyrite. Tailings will be disposed of in a new Tailings Storage Facility (TSF).

2.1 Effective Date

This Technical Report has the following effective dates:

- Date of Mineral Resource Estimate: February 18, 2021;
- Date of Mineral Reserve Estimate: June 30, 2021;
- Date of Capital and Operating Costs / Economic Analysis: 3rd Quarter 2021.

2.2 Qualified Persons

Table 2.1 provides a detailed list of Qualified Persons (QPs) as defined in Section 1.5 of NI 43 101 and their respective sections of responsibility.

The following QPs have completed property site visits:

- Ochirkhuyag Baatar[4] in October 2020;
- Dave Frost in August 2018;
- Ulziibayar Dagdandorj in April 2021 as well as June 2021.

Other QPs were generally unable to visit the site due to international travel limitations caused by the COVID-19 pandemic.

Table 2.1 – Qualified Persons – Sections of Responsibilities

Name	Company	Responsible for Section
Ulziibayar Dagdandorj, MAusIMM	Ulzii Environmental LLC	20
Tim Fletcher, P. Eng.	DRA Global Limited	2 to 6, 23, 24, portions of 18, 21, 25 to 27, and overall report compilation
Dave Frost, FAusIMM	DRA Global Limited	13, 17, 19, and portions of 1, 18, 21, and 25 to 27
Daniel Gagnon, P. Eng.	DRA Global Limited	22 and portions of 1, and 25 to 27
Dan Michaelsen, FAusIMM	Ulzii Environmental LLC	20
David Morgan, MIE Aust CPEng	Knight Piésold Consulting	Portions of 1, 18, 25, and 26
Ghislain Prévost, P. Eng.	DRA Global Limited	15 and 16, and portions of 1, 21, and 25 to 27
Robin A Rankin, MAusIMM CP(Geo)	GeoRes	7 to 12, 14, 23, and portions of 1 and 25 to 27

2.3 Units and Currency

In this Report, all currency amounts are US Dollars ("$USD", "**USD**" or "**$**") unless otherwise stated. Quantities are generally stated in Système international ("**SI**") metric units, as per standard Canadian and international practices, including metric tonne ("**tonne**", "**t**") for weight, and kilometre ("**km**") or metre ("**m**") for distances. Abbreviations used in this Report are listed in Section 28.

[4] Visited site on behalf of QP Robin Rankin

3 RELIANCE ON OTHER EXPERTS

The QPs prepared this Report using reports and documents as noted in Section 27. The authors wish to make clear that they are QPs only with respect to sections of this Technical Report attributed to them in Table 2.1 above as well as in their individual Certificates of Qualified Person.

The QPs of this Report are not qualified to provide extensive commentary on legal issues associated with Steppe Gold's Mongolian operations, or the legal rights to the mineral properties. Steppe Gold has provided certain information, reports, and data to DRA and others in preparing this Technical Report which, to the best of DRA's knowledge and understanding, is complete, accurate, and true.

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

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

DRA has relied upon market studies provided by Steppe Gold. The market study was prepared by CRU, an independent research and market consultancy firm. Section 19 summarises the key information regarding the Pyrite market overview and outlook. CRU was mandated to prepare a market study to evaluate potential target pyrite markets. DRA has reviewed the content of the market study presentation and believes that it provides a reasonable overview of the past and current market as well as projections according to various recognised sources.

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

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Introduction

Much of the information of this chapter is similar to and extracted from the previous Technical Report for ATO Phase I (October 2017). Further details as documented therein remain correct and valid, while other more recent details are addressed herein.

4.2 Property Location

The property is located in the Tsagaan Ovoo soum territory of the Dornod province in eastern Mongolia, 660 km east of the Mongolian capital of Ulaanbaatar and 120 km northwest of the provincial capital of Choibalsan. It is located in the Davkhariin Aryn valley, at the junction of Bayan and Duruu rivers, and the foot of various regional mountains (Delger Ulziit, Bayan, Namkhai Hill and Yaruu). The geographic zone of ATO project is in datum WGS-84 Zone 49N of the UTM coordinate system. In mining terms, the Property is defined by Mining License MV-017111.

Figure 4.1 - Map of Mongolia showing Location of ATO Mine Site

4.3 Property Ownership

The ATO gold mine property is covered under a single mining license MV-017111 over an area of 5,493 ha.

Initially (December 30, 2003) exploration license number 6727X was issued to Coge Gobi LLC, representing 109,118 ha. In 2007, most (~85,500 ha) of the area was turned over to the Cadaster office, leaving about 23,600 ha to the license.

On May 4, 2010, Centerra Gold Mongolia LLC (Centerra) acquired this reduced license by Order #513 of Head of the Cadaster Office. Subsequent to this, Centerra conducted intense exploration programs on the reduced license area. In 2012, Centerra prepared a Feasibility Study to transfer the exploration license to a mining license with an applicable area of about 11,600 ha surrounded by 19 location points. Mining license MV-017111 was issued to Centerra on August 31, 2012 with an applicable 30-year term expiring on August 31, 2042. The license boundary has since been simplified to 8 points with an area of about 5,500 ha.

On January 31, 2017, Steppe Gold entered into a definitive agreement to purchase 100% interest in the ATO Project. The mining license and other assets were transferred to Steppe Gold on September 5, 2017, and Steppe Gold is now the 100% owner of the ATO mining license.

4.4 Property Description

The Project will be located on land currently owned by Steppe Gold. The Project will be integrated into the existing operating plant site, which is illustrated in Figure 4.2. As seen in the figure, Steppe Gold completed construction and commissioning of Phase 1 of the ATO Project in 2020, which consists of:

- Heap leach pad;
- Pregnant and barren solution ponds;
- Water pond;
- Adsorption-Desorption-Recovery (ADR) plant;
- Ore crushing facilities;
- Various site infrastructure.

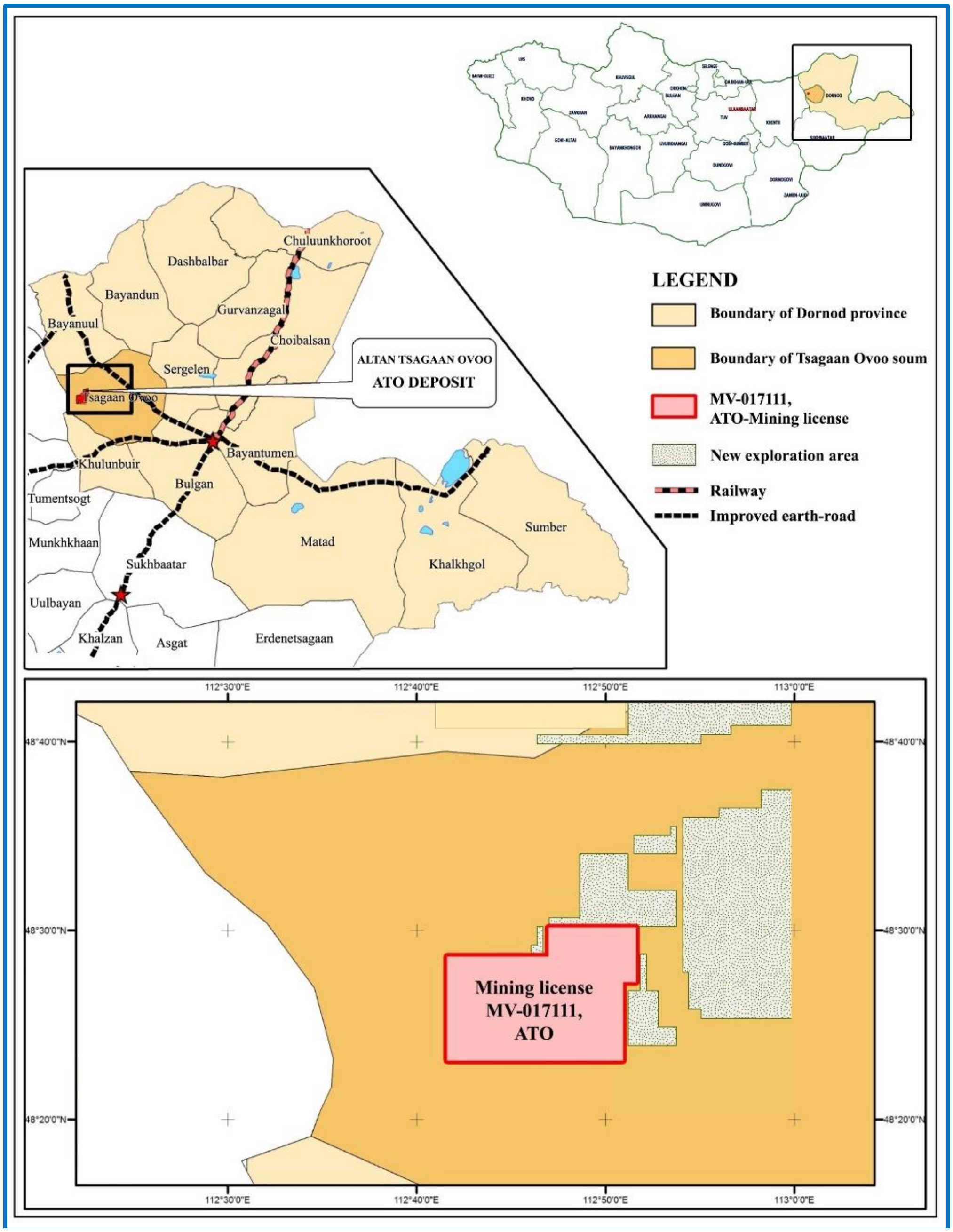
Figure 4.2 - Image of Existing ATO Project Site as of December 2020

4.5 Location and Coordinates

Location: Figure 4.3 illustrates the ATO Project regional location in Eastern Mongolia. ATO is in the border Dornod Province (light orange shading) adjacent to China in the east and south and Russia in the north. ATO is in the Territory of Tsagaan Ovoo Soum (darker orange shading on the western side of Dornod Province). Regionally ATO is 660 km east of Mongolia's capital Ulaanbaatar, 120 km west-north-west of Dornod's provincial capital Choibalsan (centre of radial black roads), and 38 km west of the closest town Tsagaan Ovoo Soum.

Coordinates: The coordinate datum used is WGS84, Zone 49 (108°E to 114°E in northern hemisphere) in the UTM system. A point just to the south of the deposits would have lats and longs of 112°47'00"E;48°26'30"N and metric coordinates of 632,000E;5,367,000N.

Figure 4.3 - ATO Project Regional Location



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

5.1 Introduction

Much of the information of this chapter is similar to and extracted from the previous Technical Report for ATO Phase I (October 2017). Further details as documented therein remain correct and valid, while other more recent details are addressed herein.

The ATO mine will operate all year around.

5.2 Accessibility

As illustrated in the previous chapter, the ATO property is located in eastern Mongolia. It is 660 km east of the capital city of Ulaanbaatar, 120 km northwest of Choibalsan city and 38 km west of Tsagaan Ovoo soum. The property is accessible from Ulaanbaatar by highway to Choibalsan, which is in the centre of the Dornod province. From Choibalsan, an upgraded and unpaved road continues to Tsagaan Ovoo soum. The property is connected to other settlements by dirt roads. In addition, it is possible to fly to Choibalsan from Ulaanbaatar on domestic Mongolian airlines.

5.3 Climate

Like the Boroo mine site, which is also located in northeastern Mongolia, climate of the ATO mine site region is characterized by extreme cold weather in the winter and hot weather in the summer. The Dornod steppe typically exhibits humid and cold weather or dry and cold weather, as most of the rainfalls are forced to Khentii Mountain by north and northwest wind (neargov.org). Average wind speed is 4 to 8 m/s, and maximum wind speed reaches about 20 m/s.

Spring begins in late March and continues until early June, and is characterized by fluctuating atmospheric temperature, air dryness and strong wind. Summer is shorter than other seasons, as well as dry and chilly.

With respect to annual precipitation, 60% to 80% falls as rain during July and August. The annual average number of days with precipitation is 59 in a year. The average annual rainfall is 150 mm to 300 mm.

Daily, monthly, and yearly fluctuations of temperature are common. The number of sunny days per year is 251 to 260. Winters can be harsh and cold, and last from November to March during which time stable snow cover exists. Soil freezing typically starts in mid-September and continues until late May, with the freezing depth reaching 2.5 m. Average and extreme seasonal temperatures are summarised in Table 5.1.

Table 5.1 - Average and Extreme Seasonal Temperatures at ATO Mine Site

	Period	Temperature (°C)	
		Average	Max / Min
Summer	June to September	21	40 (max)
Winter	December to March	-27	-46 (min)

5.4 Local Resources and Infrastructure

The closest settlement to the property is the central village of the Tsagaan Ovoo soum, (population 3,800) located beside Khuuvur Lake, which has some infrastructure. Nationality of the local population consists of 80% indigenous Buryats, and the balance being Khalkha Mongols. The central village includes administrative offices, a cultural centre, secondary schools, a hospital, a kindergarten, a communications centre, cellphone stations, a gas station, and high-voltage substations.

The community is mainly active in animal breeding and farming / plantations. The land surrounding the property is mainly used for nomadic herding of goats, cows, horses and sheep. Use is based on informal traditional Mongolian principles of shared grazing rights with limited land tenure for semi-permanent winter shelters and other improvements.

The natural water network of the area belongs to the Pacific Ocean basin. Small local rivers and streams fed from the mountains of the Khentii Range flow into small lakes. The sizes of these rivers vary depending on precipitation. Drinking water is only from wells due to the low water network density. Regional lakes (Duut, Tsagaan, Ovoot, Eregtseg, Ukhaagiin Tsagaan, Davkhariin Tsagaan, and Khaichiin) and many other small salt lakes are also fed by rainfall. In recent years, small rivers and streams have dried up due to global warming and reduced precipitation. In the summer, seasonal springs form from melting of small patchy permafrost in intermountain valleys and from seasonal thawing of frozen ground.

5.5 Physiography

Geographically, the area of the site is located in the low mountain zone at the northeastern end of the Khentii Mountain Range and at the southwest portion of the Dornod high steppe. The topography of the area generally consists of small rounded mountain complexes with small hillocks in a steppe. Vegetation and grass cover the entire area and includes pasture plants such as khazaar grass, wormwood, stipa, brome-grass, and couch grass. There are few trees.

The average elevation of the region is 980 m to 1,050 m above sea level, with the lowest point being Deliin Well (979.3 m) and the highest point being Mount Temdegt (1144.7 m). The relative elevation variation is 60 m to 120 m. Predominant ground features are brown and black-brown gravel, sandy loam, and gravel-mild clay of the steppe zone.

Wildlife of the region are summarised as follows:

- Hoofed animals include white gazelle.
- Carnivores include wolves and foxes / corsacs.
- Rodents include marmots, gophers, shrew-mice, and stoats.
- Birds include larks, red noses, cranes, bustards, scoters, and brown noses.
- Crawlers, locusts, grasshoppers, mosquitoes and midges are abundant.

The Law on Environmental Impact Assessment (2012) and the guidelines require the inclusion of a risk assessment in project documentation. This means identification and prediction of the possible emergencies and accidents that could occur during the production process or natural disasters, and elimination and mitigation of their consequences.

There are mining licenses for mining operations and related permits, the availability of power source, water, potential areas for waste material and heap leach pad area, mining personnel and local working power in the project area.

6 HISTORY

6.1 Introduction

Much of the information of this chapter is similar to and extracted from the previous Technical Report for ATO Phase I (October 2017). Further details as documented therein remain correct and valid, while other more recent details are addressed herein.

6.2 History and Land Holdings

6.2.1 Initial Discovery

The area of the ATO Project is focussed on three low (< 200 m high) gently rounded hills surrounded by widespread unconsolidated deposits. The surrounding ranges consist of low gently rounded slopes of somewhat steeper inclines with moderate slopes.

Regional geological field parties working for COGEGOBI (a wholly owned subsidiary of AREVA) were the first to recognize mineralised rocks cropping out at ATO. In 1997 COGEGOBI began exploration in eastern Mongolia. In 2003, after a six-year reconnaissance effort, COGEGOBI settled on a selected region, and obtained eight exploration licenses in eastern and southeastern Mongolia. COGEGOBI then embarked upon a four-year exploration effort for viable gold and uranium deposits in all eight of their licensed areas. Two of their licenses (3,425.5 km² in all) were in the general area of ATO. As part of that effort, COGEGOBI geologists collected 52 grab samples from outcrops and subcrops at ATO, and identified anomalous gold concentrations in vein quartz-rich rock (0.06 to 27.8 g/t Au). COGEGOBI proceeded to describe the occurrence at ATO as “intense hydrothermal alteration associated with volcanoplutonic structures” (Hocquet, 2005).

At the end of a four-year long exploration cycle in the eight licensed areas (which was focused on gold but also included uranium), uranium prices increased in mid-2007 US$30 per lb U_3O_8 to about US$140 per lb U_3O_8. AREVA subsequently:

- Acquired East Asia Minerals Energy Company in September 2007;
- Became AREVA Mongol in 2008; and
- Gained 100% interest in COGEGOBI in 2017.

Consequently, AREVA Mongol’s interest shifted from precious metals to the energy sector, and one of their two exploration licenses surrounding ATO was dropped.

6.2.2 Initial Field Examination by Centerra and Discovery

In late 2009, Centerra geologists, accompanied during their field examination by AREVA Mongol geologists, were invited by AREVA to visit the ATO site. Subsequently, initial agreement to explore the remaining licensed area (Tsagaan Ovoo) encompassing ATO was obtained from AREVA Mongol by CGM in May 2010, and all property rights were obtained by Centerra in late 2010. AREVA retains a 1.75% NSR in the Tsagaan Ovoo license.

Even during Centerra's first brief visit to the property, it became clear that exposures of the mineralised system at ATO include highly disrupted, near paleosurface epithermal, silica-dominated rocks that had the potential to host a significant tonnage of precious and base metal mineralised rock. Four grab samples collected by Centerra (1.3 to 3.3 g/t Au), during its initial visit to the property confirmed presence of anomalous Au as originally reported.

More recent exploration history at ATO (still prior to 2017), comprises three (3) stages:

- Initial work by COGEGOBI/AREVA;
- Due diligence by Centerra that also included soil geochemistry and limited IP; and
- Post May 2010 comprehensive exploration by Centerra.

From May 2010 through December 2014, Centerra designed significant exploration work in the area, incorporating geologic mapping (including ASTER imaging), widespread grab sampling, additional grid soil sampling, stream-sediment sampling, geophysical surveys (air mag, ground mag, IP, gravity), trenching, and extensive core drilling. In addition, a wide-ranging district-wide grab sampling program was conducted in association with regional geologic mapping.

As a result, it became readily apparent that most of the presently known precious and base metal mineralisation at ATO is in three carrot-shaped vertically downward plunging, presumably Jurassic breccia pipes.

6.2.3 Previous Mineral Reserves and Mineral Resources

Centerra reported an ATO Mineral Resource summary in its 2016 Annual Information Form (AIF) on SEDAR in May 31, 2017. The information has been subsequently superseded by more recent Mineral Resource estimates as further described in Section 14.

6.2.4 Acquisition by Steppe Gold

On September 15, 2017, the Company completed the acquisition of the Altan Tsagaan Ovoo Property (the "ATO Project" or "ATO Mine"), from Centerra Gold Mongolia LLC, for aggregate consideration of $19.8 million plus $1.98 million in value added tax (the "ATO Acquisition"). The transaction has been accounted for as an asset acquisition.

6.2.5 Equity Financing

On May 22, 2018, Steppe Gold announced the closing of its initial public offering ("the Offering") of units of the Company ("Units"). Under the Offering, the Company issued 10,569,185 Units at a price of CAD$2.00 per Unit ("the Issue Price") for gross proceeds of $16,532,434. Each Unit comprises one common share of the Company and one common share purchase warrant ("a Warrant"). Each Warrant is exercisable for one common share at an exercise price equal to CAD$2.34 for a period of 24 months after the closing date of the Offering. The distribution of the Units was qualified by way of prospectus dated May 2, 2018 filed with the securities regulatory authorities in each of the provinces and territories of Canada, other than Quebec. Haywood Securities Inc. and PI Financial Corp. ("Agents") acted as co-lead agents on the Offering. Total cash costs for the initial public offering amounted $1,630,152, allocated as follows: $1,473,752 to common shares; and $156,400 to Warrants.

On December 23, 2019, the Company issued 2,222,222 common shares for CAD$2,000,000.

On August 5, 2020, the Company issued 6,976,944 units at a price of CAD$2.15 per unit for gross proceeds of CAD$15,000,000. Each unit comprises one common share of the Company and one common share purchase warrant, with each warrant entitling the holder to acquire one additional common share of the Company at a price of CAD$3.00 per share for a period of 24 months from the closing date. The Company incurred finder's fees of CAD$600,000 and legal fees of CAD$7,000 in relation to equity financing.

6.2.6 Convertible Debenture

On July 2, 2019 and on August 27, 2019, the Company closed the private placements issuing $5.4 million and $3.04 million principal amount of two-year unsecured convertible debentures ("Debentures") respectively. $600,000 of the proceeds from the debentures was allocated from unsettled accounts payable.

On January 30, 2020, the Company received funding from the Mongolian National Investment Fund PIF SPV (the "Fund"). The Fund has subscribed for a 12% two-year secured convertible debenture of the Company in the principal amount of $3 million. The debt is secured against all the shares of Steppe West owned by the Company.

6.2.7 Debt Financing

In connection with the ATO Acquisition, the Company's subsidiaries, Steppe Gold LLC ("Steppe Mongolia") and Steppe Investments LLC ("Steppe BVI") entered into a metals purchase and sale agreement (the "Stream Agreement") dated August 11, 2017 with Triple Flag International to sell gold and silver produced from the ATO Project. Under the terms of the Stream Agreement, Triple Flag International advanced an upfront deposit of $23,000,000 to Steppe Gold and Steppe BVI is obligated to sell to Triple Flag international 25% of the gold and 50% of the silver produced from the ATO Project until such time as Steppe BVI has sold an aggregate of 46,000 ounces of gold and 375,000 ounces of silver, respectively. Thereafter the annual amounts that Steppe BVI is obligated to sell to Triple Flag International is capped at 5,500 ounces for gold (plus 250 ounces of gold for each three month period in which the commercial production date follows September 30, 2018) and 45,000 ounces for silver (plus 2,045 ounces of silver for each three month period in which the commercial production date follows September 30, 2018). On September 30, 2019, the Company entered into an agreement to amend the terms of its existing gold stream with Triple Flag International. Under the terms of the amendment, Triple Flag International advanced an additional deposit of $5,000,000 to Steppe Gold, bringing the total amount advanced to Steppe Gold by Triple Flag International under the gold stream to $28,000,000. The proceeds received from Triple Flag International were used to repay the final $5,000,000 promissory note issued as part of the purchase price for the acquisition by the Company of the ATO Project.

On September 18, 2020, the Company entered into a loan agreement with the Trade and Development Bank of Mongolia ("TDBM") for 30 billion Mongolian Tugriks (US$10,510,000). The loan is financed by the Bank of Mongolia for a period of 24 months secured by a cash deposit held by TDBM totaling 35.4 billion Mongolian Tugriks (US$12,386,000 as of June 30, 2021).

6.3 ATO Phase 1

In 2017, an NI 43-101 Technical Report was prepared for Steppe Gold and Centerra Gold (Centerra) (the previous ATO Project owner) by the Mongolian company GSTATS Consulting LLC for Phase 1 of the ATO Project.

6.3.1 Further Details on Completion of Phase 1

Phase 1 construction of the ATO project was established in 2018. Between 2018 and 2019, Steppe Gold LLC completed the plant construction in stages to process oxidized ore of the ATO deposit by heap leaching technology. Steppe Gold has obtained the necessary permits, the processing plant and open pit mine have been commissioned and the project is currently running with under normal operation.

6.3.2 Financing of Project

Steppe Gold has invested total of US$160 million for the full development of Phase 1 of the ATO Project.

6.3.3 Detailed Engineering and Construction

The Company has built required facilities for heap leaching of oxidized ore as: water supply system, crusher with a capacity of 1.2 million t/a, heap leaching area with capacity of 5.37 million t of ore, processing plant with a capacity of 300 m^3 of solution per hour, pregnant and barren ponds with a capacity of 47,000 m^3, chemical storage, fuel station and boiler house has been commissioned recently.

6.3.4 Start Up of Phase 1

In 2020, Steppe Gold completed the construction and commissioning for Phase 1, which consisted of an integrated oxide ore heap leach production facility which now produces 60,000 oz of gold annually.

In the second quarter of 2020 the Company achieved commercial production at the ATO Mine. Prior to the commencement of commercial production, production costs were capitalised within construction in progress.

7 GEOLOGICAL SETTING AND MINERALISATION

The information in this section is largely drawn and/or summarised from the Report available on SEDAR entitled: "Altan Tsagaan Ovoo Project (ATO), 2021 Mineral Resources, Technical Report (Amended NI 43-101)", prepared by GeoRes, with an effective date March 30, 2021, issued on March 30, 2021 and amended on June 9, 2021. Further details as documented therein remain correct and valid.

The geological setting and mineralisation of the Project is described in terms of:

- Regional geology.
- District geology and magmatism.
- District mineralisation.
- Geology of the mineralised ATO deposits.
- Mineralisation at ATO.

7.1 Regional Geological Setting

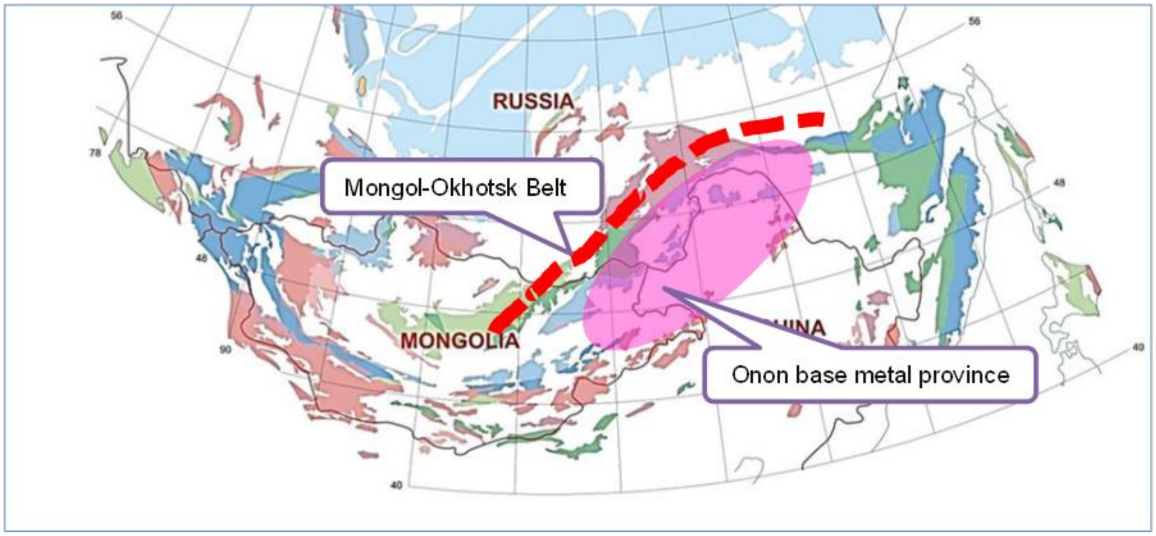
Relative to its continental-scale geologic framework, ATO is situated within the Devonian through Late Jurassic Mongol-Okhotsk tectonic collage (Figure 7.1[5]) that has been emplaced along a transform-continental margin of the North Asian Craton (NAC) as shown by Parfenov and others (2010). In addition, the Transbaikalian-Daxinganling transpressional magmatic arc that is present south of ATO along an ENE, 2,000 km long trend was thought to range in age from 175 to 96 Ma (Middle Jurassic to Early Cretaceous) (Parfenov and others, 2010).

Regional metallogenic setting of ATO is important from an exploration perspective. Mineral deposits range widely in age throughout Eastern Mongolia and neighbouring regions of Russia and China. For example, the China Altay hosts 380–360 Ma siliciclastic VMS deposits with bimodal geochemistry in a major magmatic arc (Goldfarb and others, 2003). Further, as summarised by Xiao and others (2009), end-Permian to mid-Triassic docking of the Tarim and North China cratons against the Siberian craton resulted in (1) closure of the Paleoasian Ocean and led to (2) formation of a number of world-class metal deposits, some of which are Triassic in age.

A number of Late Jurassic-early Cretaceous (175–96 Ma) broad, gold-bearing mineral belts also have been recognized in eastern Mongolia and in the surrounding region (Rodionov and others, 2004). Their Middle Jurassic-Early Cretaceous (175–96 Ma) time slice yields 31 gold-bearing mineral belts among 56 belts in all (55%) – the most mineral belts outlined for the various time slices established in the above-cited report.

[5] CGM 2012 Exploration Report

Figure 7.1 - Location of Mongol-Okhotsk Belt and Onon Precious Base Metal Province



Source: ATO 2021 Mineral Resources, Technical Report (Amended NI 43-101), 2021

Most gold-bearing belts during the Middle Jurassic-Early Cretaceous have moved decidedly “inboard” towards the Siberian craton relative to older belts, and they are present in China and Mongolia, as well as eastern Siberia. Areal distribution of the gold-bearing belts generally follows the tectonic grain or trend of the various geologic terranes and their overlap and “stitch” assemblages throughout the region.

ATO is located north of the Main Mongolian Lineament and midway along the NNE trending 600km long Onon base and precious-metal province that crosses eastern Mongolia.

The overwhelming bulk of the Pb–Zn occurrences and deposits in eastern Mongolia are located north of the MML and east of the Onon trend. The Novo and Lugiin polymetallic deposits in the Russian Federation were used to anchor the northern terminus of the Onon province as depicted in the Russian Federation. The two major mineralised trends or metallotects in this part of Mongolia (Onon and Yeroogol, red dashed line) parallel Lake Baikal, and must represent deep-seated splays possibly dating from zones of crustal weakness first developed at the time of Devonian-age accretion and dislocations along the MML. However, rifting in Lake Baikal is much younger as it began about 30 m.y. ago; i.e., during the Middle Oligocene.

Therefore, the two mineralised trends (Onon and Yeroogol) must mark zones of rifting in the earth’s crust, somewhat deeper seated than that at Lake Baikal, and whose regional least principal stress direction must have been oriented NW–SE (present day coordinates). However, this orientation of the regional least principal stress must represent a clockwise rotation from its essentially EW orientation that prevailed during final stages of rifting in the Cretaceous following mineralisation and associated magmatism at ATO.

A number of Hg–Sb occurrences are aligned closely along the trace of the Onon trend, as are numerous clusters of gold occurrences that are associated somewhat more broadly in the immediate region.

Further, as defined herein, the Onon base and precious-metal province coincides with relatively thick crust, roughly greater than 125 km in thickness, which partly explains abundance of base metals (especially Pb) throughout the province. Rocks south of the MML are considered to be largely Silurian to Carboniferous island-arc volcano-stratigraphic packages of rock.

Though ATO presently represents the only well-explored gold deposit in this part of Mongolia, a large number of minor gold occurrences have been recognized throughout the region. Most of these gold occurrences are located outside the Onon province, as are the overwhelming number of recognized porphyry systems. The Onon province also includes a number of primarily Ag occurrences, as well as a few porphyry systems, including the Avdartolgoi porphyry, about 200 km NE of ATO, which only is mentioned briefly in passing by Dejidmaa and others (1999) without any further details.

A number of major base and precious-metal deposits also are present in the region with metal associations similar to ATO. These include the Novo and Lugiin polymetallic deposits at the northern distal end of the Onon province.

7.2 District Geology and Magmatism

The term “district” is used here in its broadest sense (“sensu lato”) because no unifying genetic model or linked group of models has been established confidently for all mineralised occurrences close to ATO other than a presumed association with Jurassic magmatism. Furthermore, the age of gold-mineralised rock has not been determined radiometrically, either at ATO or at mineralised rock in most all its surrounding occurrences.

Figure 7.2 presents the district geology around the ATO Project (red oval). The area shown has approximate dimensions 25 km E/W and 35 km N/S.

The oldest layered rocks in the ATO district are Devonian. Relatively small, isolated areas of outcrop of Early Devonian trachyrhyolite, trachyrhyolite porphyry, ignimbrite (welded tuff), and minor limestone are present in the northern part of the ATO district (Figure 7.2). The Devonian rocks are in tectonic contact with Early and Late Permian strata along pre-Lower Cretaceous NW–striking, high angle faults near the NW corner of the area, and Lower Cretaceous rocks overlie unconformably the Devonian rocks. Devonian rocks are intruded by Early Permian leucogranite near the NE corner of the district (Figure 7.2).

Though the ATO district includes limited exposures of Devonian and Triassic rocks, the most widespread rocks in the district are Early Permian volcaniclastics including tuff breccias, as well as high K andesite, and rhyolite exposed in the cores of broad uplifts. Zircons from rhyolite have been dated at 285.9 Ma (Early Permian) by a major ongoing collaborative program (to date magmatism and mineralisation in the district) by CGM with Jim Mortensen at the University of British Columbia. The Early Permian volcaniclastics in the ATO district are further intruded by early Permian leucogranite, plagio-granite, and diorite; zircons from plagio-granite have been dated at 279.5 Ma.

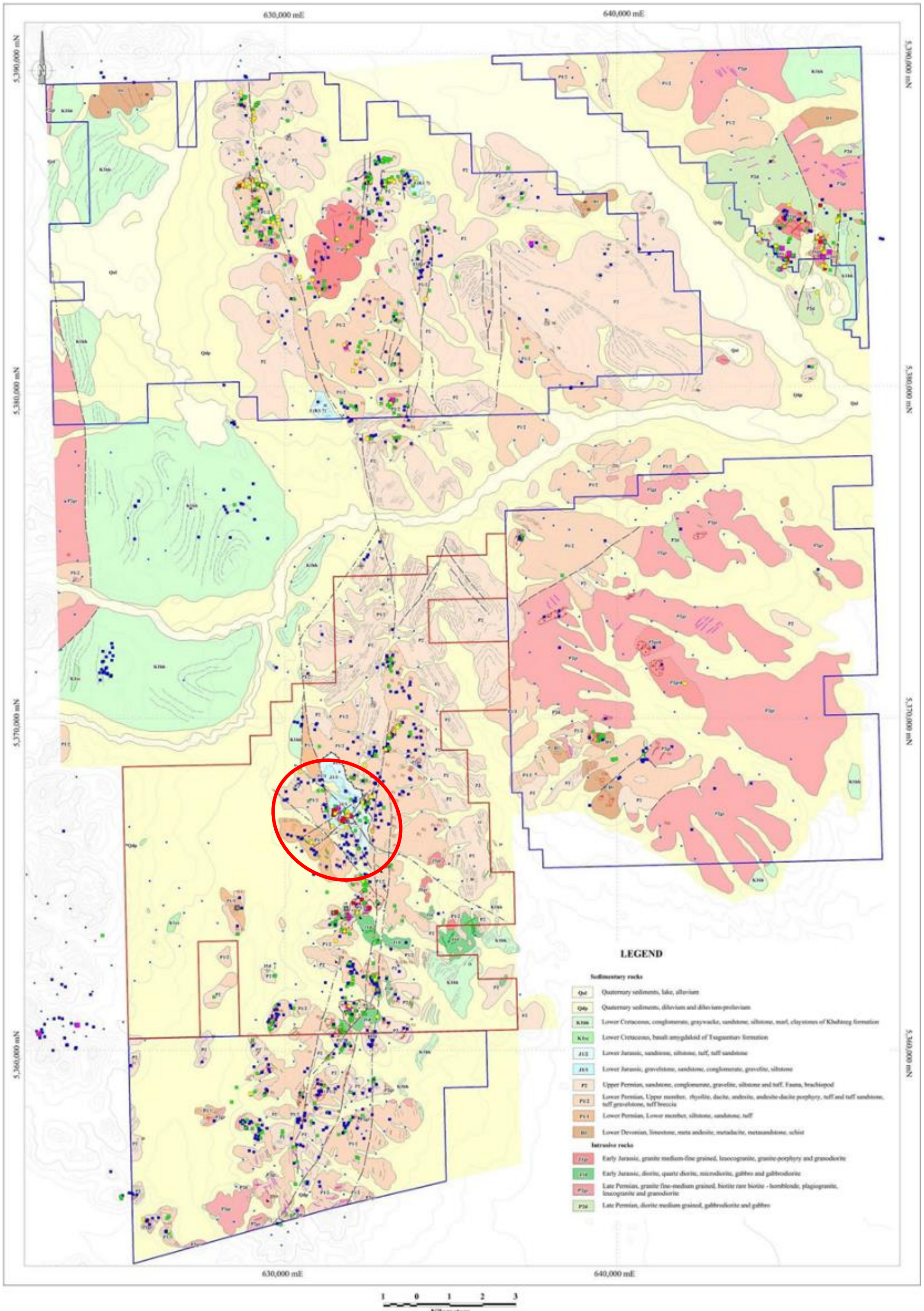
Nonetheless, Permian strata largely form the cores of broad horsts in three areas: (1) a relatively small area of outcrop (2 km in long dimension) near the NE corner of the district; (2) an approximately 25 km long, NW elongated expanse that extends from the east central part of the area to the north edge of the district; and (3) a 16 km long, NS elongated belt that extends from about 5 km N of ATO to the south edge of the district. Rocks shown as Early Permian strata are mostly volcanic affiliated (rhyolite, dacite, ignimbrite, andesite-dacite porphyry, tuff, and tuffaceous sandstone), whereas strata assigned provisionally to the Late Permian are mostly sandstone, conglomerate, siltstone, and tuff.

Early Permian diorite crops out near the NE corner of the district. The age of this unit is inferred from the age of rocks that intrude it. The diorite in this area is intruded by Late Early Permian leucogranite, plagio-granite, and granodiorite, as well as Early Jurassic granite and granodiorite at the Bayan Munkh prospect. Late Early Permian leucogranite, plagio-granite, and granodiorite crop out mainly in four areas: (1) near the NE corner of the area; (2) near the west-central edge of the area, (3) NW of ATO, and (4) near the south edge of the mapped area. Early Permian leucogranite intrudes Devonian rocks as well as rocks assigned to both Permian units (P1 and P2).

At ATO, gravel and coarse pebbly sandstone fill a broad shallow depression on the flanks of a Permian-cored uplift. Mesozoic intrusive rocks also crop out in various locales in the district. Some are mineralised as at Bayan Munkh.

Cretaceous sedimentary rock unconformably rests on all of the older units in the district (green and light green Figure 7.2). Cretaceous siltstone, sandstone, conglomerate, and basalt fill a narrow NS Cretaceous graben NW of ATO. Preliminary PIMA examination of four samples of laminated siltstone in a drill hole into the graben indicate presence of chlorite (D. John, written commun., 2012), as opposed to presence of mixed layer clays now known to form the host mineral for lithium in similar basins elsewhere. A number of prominent NS striking faults pass just to the east of ATO, and NE-striking, high-angle faults also are present at ATO.

Figure 7.2 - Geology Map of ATO District



Source: NI 43-101 ATO Gold Project, 2017

The Cretaceous graben reaches its maximum width of about 6 km approximately 22 km N of ATO. Coarse-grained Paleozoic granite also is well exposed and widespread west of the graben that probably opened in response to crustal EW (present coordinates) regional extension beginning in the Late Cretaceous.

7.3 District Mineralisation

Though the ATO Project is the most important discovery to date (and is present in the south-central part of the district), a number of other mineralised occurrences also are present nearby including from N to S the Bayan Munkh, the High Land, Duut Nuur, Bayan Gol, Mungu, Apricot and Davkhar Tolgoi prospects. CGM discovered in the exploration area a hard-rock gold-lead-zinc deposit and other similar occurrences and mineralisation points as well as two gold occurrences, four gold mineralised points, three lead mineralised points, three lead-zinc mineralised points, and several secondary dispersion halos of gold, lead, zinc and silver.

7.4 Geology of the Mineralised ATO Deposits

The geology of the mineralised ATO deposits is described in terms of:

- Deposit geology.
- Deposit mineralisation controlling factors.
- Silicate alteration in the mineralised pipes.
- Silica cap to Pipe 1.
- Styles of sulphide mineralised rock at ATO.

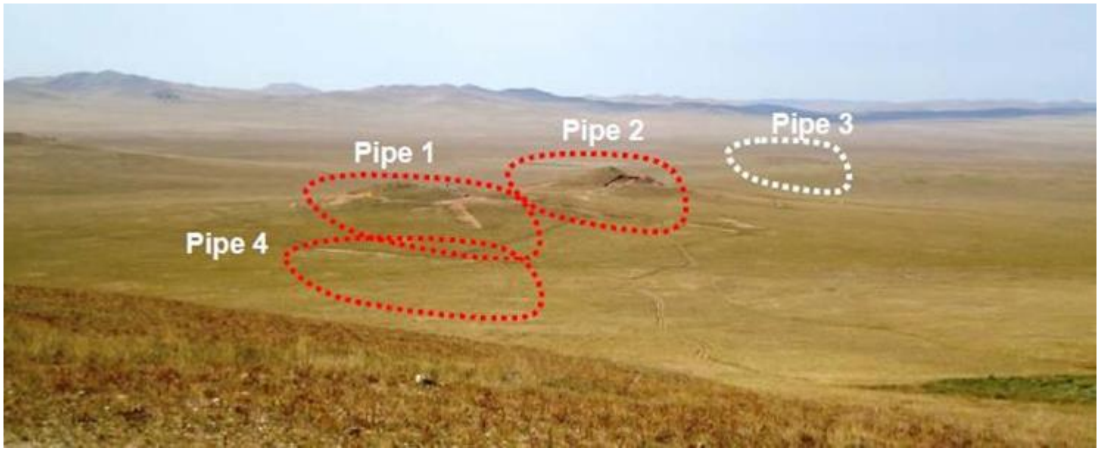
7.5 ATO Deposits Geology

Pipes 1, 2, and 4: Up to 2017 exploration focussed on three mineralised pipes at ATO. These were named Pipe 1, 2, and 4 (now ATO 1, 2, and 4) and are illustrated with red ovals in Figure 7.3 and in plan in Figure 7.4. Pipe 3 to the west (white oval) is only poorly mineralised and has not been evaluated further so far.

Subsequently a fourth well mineralised deposit (Mungu) was discovered slightly to the north east of Pipe 4.

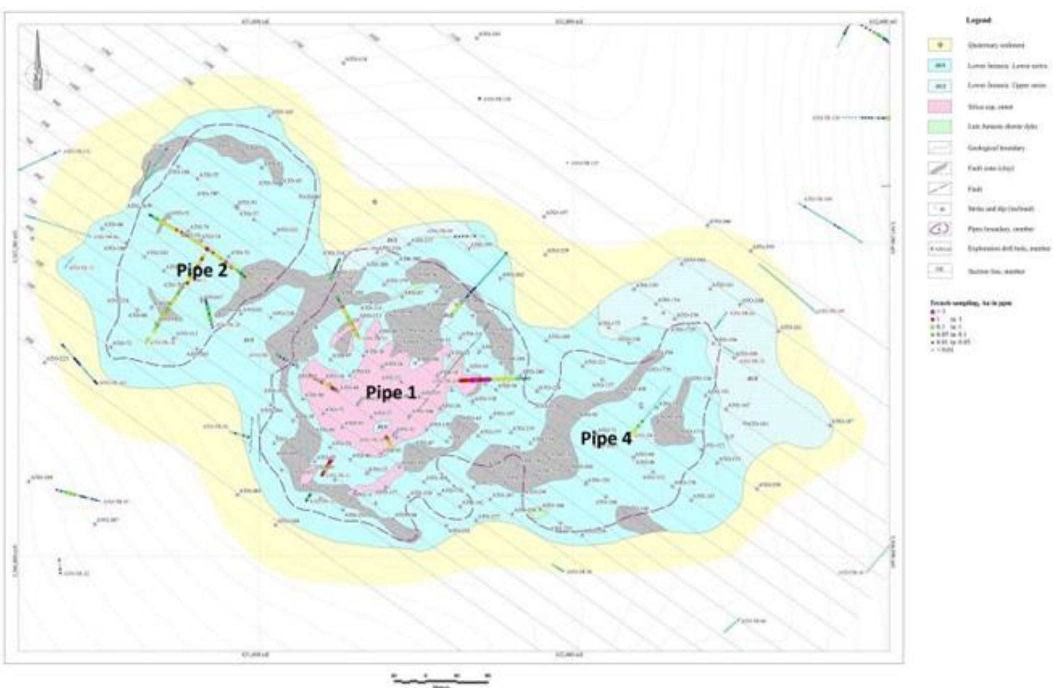
Adjacent pipes ATO 1, 2 and 4 were emplaced into stratified rocks as young as presumably Early to Middle Jurassic. Pipe 4 is mainly concealed. Pipe 3 to the west contains abundant pyrite, but no significant amounts of Au, Ag, Pb, and Zn – and hence has not so far been explored further. There is a strong Au anomaly in soil at ATO, as well as other accompanying metals, particularly Pb. In the aeromagnetic field, only post-mineral young dikes have a prominent positive response; much weaker responses outline some ring-shaped features. ATO resides near the centre of the latter. Sinter and silicified rocks are reflected as shallow resistivity anomalies, but broad clay and chlorite-altered rocks are characterised by low resistivity. The pipes coincide with chargeability anomalies that overall are quite weak.

Figure 7.3 - Perspective View of ATO Pipes 1, 2 and 4, looking SW



Source: NI 43-101 ATO Gold Project, 2017

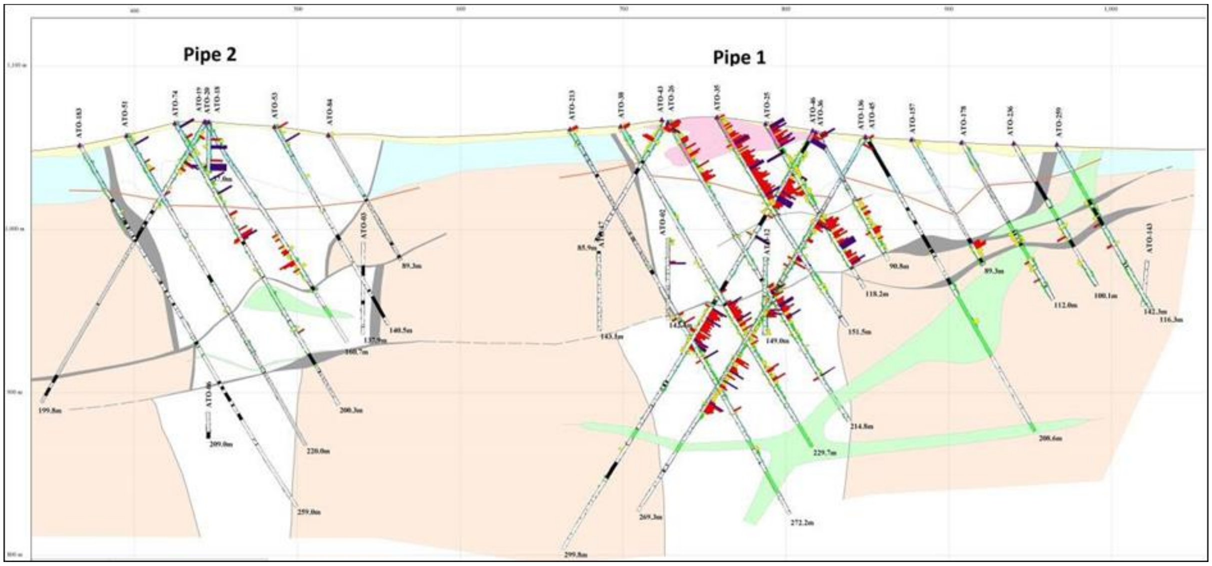
Figure 7.4 - Geology Map of ATO Pipes 1, 2, and 4



Source: NI 43-101 ATO Gold Project, 2017

Surface projection of the morphology of Pipes 1, 2, and 4 is shown in Figure 7.4. An upper zone of Au-Pb-Zn-Ag mineralised rock at Pipe 1 is approximately oval in shape and is ~320 m wide. Pipe 2 is elongate to the NE and is ~320 m by ~160 m in maximum dimension. Pipe 4, which is completely concealed, also is elongate to the NE and is ~400 m by ~200 m. The NW/SE cross section through Pipes 1 and 2 (Figure 7.5) shows that the pipes taper slightly with depth and have a carrot-shaped 3D configuration, narrowing gradually at depth to ~200 m wide.

Figure 7.5 - Cross-Section Through Pipes 1 and 2



Source: NI 43-101 ATO Gold Project, 2017

The deepest hole into Pipe 1, inclined 60°, is ~700 m long. Silica cap rock (pink) has a variable thickness in Pipe 1, generally tapering from a maximum thickness of about 40 m under the topographic high point of the pipe to less than 1 m near its margins. However, bottom surface of the cap rock is highly irregular, showing sharp undulations with underlying quartz-veined Middle-Late Jurassic gravel and coarse pebbly sandstone, some blocks of which are totally engulfed by massive silica.

The pipes also are cut by a number of minor faults, both steeply dipping and shallow dipping. Some narrow flat-lying post-mineral diorite dikes also have been emplaced along faults that offset margins of the pipes.

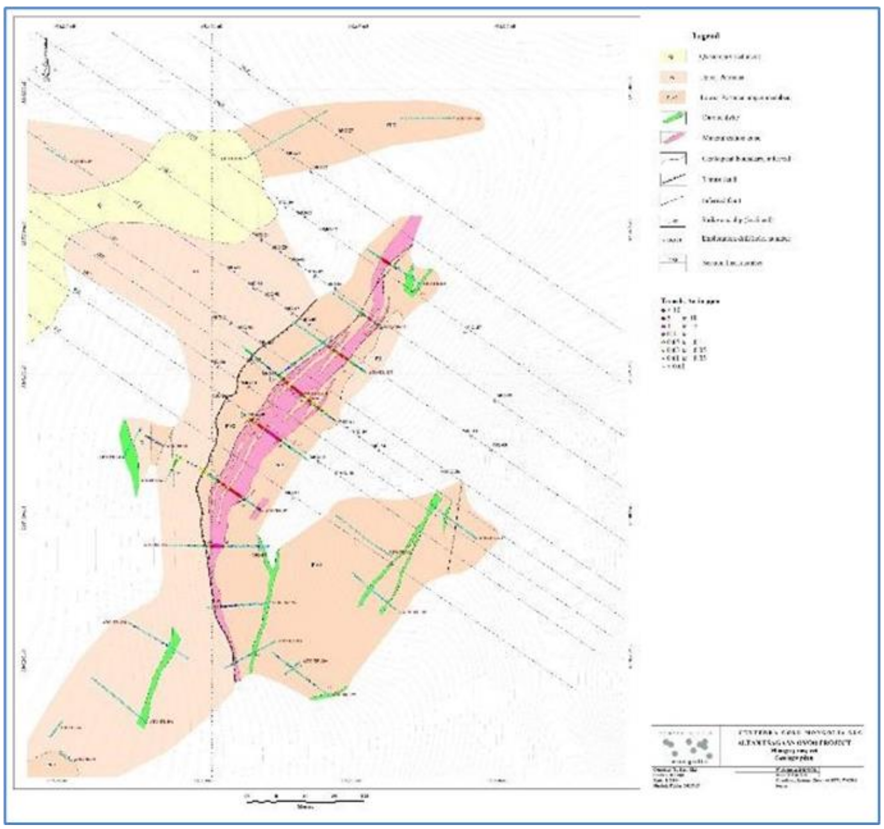
Pebbly conglomerate and pebbly sandstone were being shed from both nearby mostly Early Permian highlands elevated during emplacement of Early Jurassic magmatic rocks, as well as apparent high walls of an enclosing oval collapse feature. Continued deposition of Jurassic strata then covered the pipes after cessation of mineralisation.

Despite the three mineralised pipes being so geographically close to one another, there are distinct differences in their metal geochemistry. Pipe 2 is notably base-metal enriched; Pipe 1 contains less base metals, and Pipe 4 contains further decreases in base metals, particularly near its margins where extremely high Ag contents (locally 100’s of ppm Ag across narrow intercepts) are present in association with base metal concentrations of a few hundred ppm in all. Mungu is particularly enriched in silver in comparison to the others.

7.6 Mungu Deposit Geology

Mungu: Mungu deposit is hosted in Lower Permian age volcano-sedimentary rocks and overlying Upper Permian sedimentary rocks and is itself cut by late diorite dykes (Figure 7.6). Post-mineralisation diorite porphyrite dykes are abundant in the area. Weakly to moderately chloritized, black green coloured diorite porphyrite dykes with rare pyrite dissemination occur at depths of 150-180 m. They are in massive structure, have undergone little fracturing, and are consistently continuous along dip with average thickness of 10-15 m in almost horizontal position. They are branched out in some parts in varying directions. Also, light green coloured, weakly sericitized, strongly fractured and deformed diorite dyke have been found that undergone clay alteration and have a small thickness (up to 1 m). This post-mineral dyke is found close to a fault that displaced the orebody along a horizontal plane. These dyke plays destructive role in deposit settings.

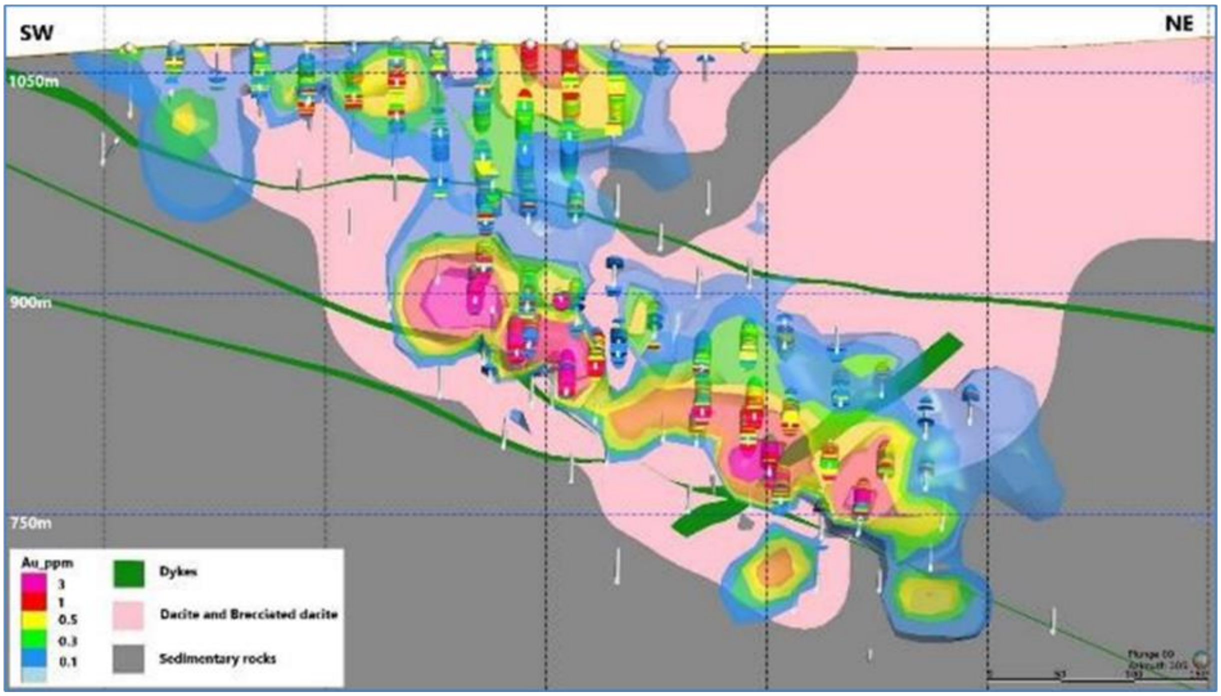
Figure 7.6 - Geology Map of Mungu Deposit



Source: ATO 2021 Mineral Resources, Technical Report (Amended NI 43-101), 2021

Mungu is a structurally controlled epithermal gold-silver system with localized bonanza grades, and an Ag:Au ratio approximating 10:1. Mineralization occurs in brecciated zones controlled by NE trending structure and tiny dark coloured quartz-sulphide veinlets developed along the big fault zones. Overall the ore body has an almost linear vertical shape striking, to the NE at 035°. Trend of mineralisation is plunging to NE by azimuth 035 and dip angle 40°. The mineralised bodies separated by late post-mineralised dykes into parts and most significant dyke occur in 150 m depth below the surface, lies almost horizontally. Surface projection of the morphology Mungu orebody shown in Figure 7.6. Mungu orebody is elongate to the NE direction and continued about 700 m along the structure and orebody thickness varies up to 200 m in plan. The SW/NE cross section through Mungu orebody along the strike of mineralisation is show in Figure 7.7. In detail the ore body consists of multiple close spaced sub-parallel sub-vertical lenses.

Figure 7.7 - Mungu Deposit in Cross-Section Looking NW



Source: ATO 2021 Mineral Resources, Technical Report (Amended NI 43-101), 2021

Mineralisation consists of variable concentrations of mainly pyrite with arsenical rims, plus minor amounts of base metal sulphides and rare silver sulfosalts as veins, disseminations, and breccia fill in a relatively steep, narrow structure which has been traced over a lateral distance of about 700 m and vertical extension about 350 m depth from surface, mainly within dacitic rocks. Quartz veining is a minor component of the mineralisation, and banding and other typical epithermal textures are essentially absent. Argillic alteration forms as envelope with tens of meters wide. About 5-40 m wide zone of low-grade mineralisation with localized narrow zones of high-grade mineralisation at depth has yielded bonanza grades in both gold (to 172.88 g/t) and silver (to 1,500.0 g/t).

7.7 Deposit Mineralisation Controlling Factors

Important geologic controlling factors for the mineralised pipes at ATO include:

1. Presence in a major base metal (Pb-Zn) province (thick crust) known to include large tonnage Au-Pb-Zn deposits.
2. Near paleo surface, epithermal (hot spring) emplacement of the upper parts of mineralised pipes associated with Jurassic magmatism into a near surface area.
3. Shallow depression where Middle-Late Jurassic pebbly conglomerate and pebbly sandstone were being deposited prior to mineralisation and continuing after mineralisation.

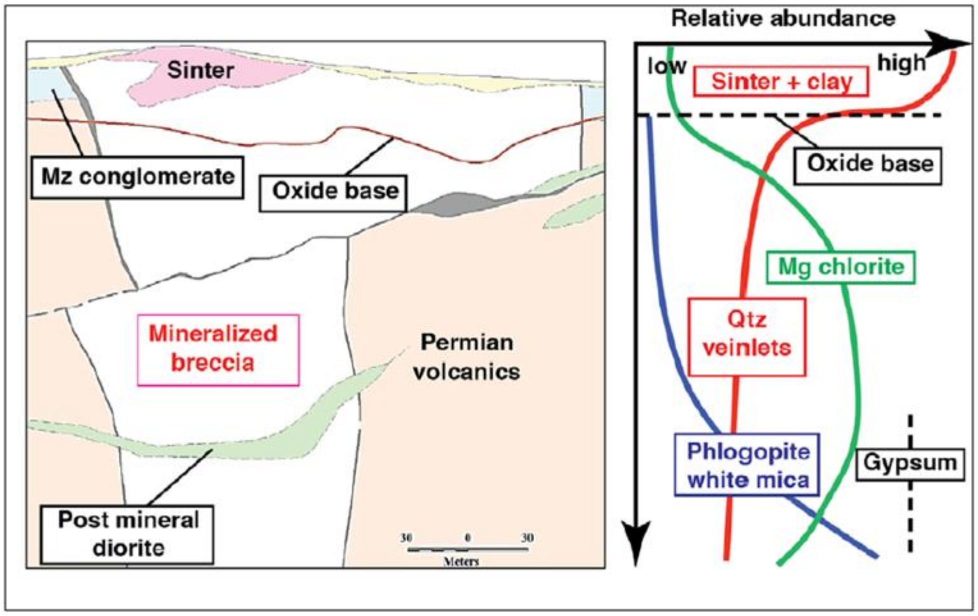
Origin of brecciation in the mineralised pipes remains unclear however but shows features of both magmatic and hydromagmatic breccias (see Sillitoe, 1985). Some fine-grained, matrix-supported breccias with abundant rock flour are encountered at depth at ATO and are typical of a magmatic breccia-style diatremes (magmatic hydrothermal systems that extend to surface).

7.8 Silicate Alteration in the Mineralised Pipes

ATO is characterized by an absence of adularia that is relatively abundant elsewhere in epithermal Au-Ag deposits. As discussed below, at ATO this primarily is a reflection of high $Mg/_2H+$ ratios and a correspondingly low K/H+ ratio in mineralizing fluids associated with an underlying largely dioritic magmatic complex. At ATO siliceous sinter forms a cap rock at the top of Pipe 1 (left side Figure 7.8) and includes barite (in narrow micro veins as well as tabular crystals in open cavity fillings), clinochlore (Mg chlorite), and less abundant Mg-Fe chlorite. With increasing depth (right side Figure 7.8), silica is increasingly present in the pipes as relatively late paragenetic stage vein quartz filling central parts of sulfide mineral-rich veins. Though clays (mostly kaolinite) occur with sinter at the top of the mineralised column at Pipe 1, clinochlore (Mg chl) is the dominant alteration hyrosilicate mineral at depth throughout mineralised breccia. At depths near 200 m, however, clinochlore begins to give way to phlogopitic white mica. White mica also increases near margins of Pipe 4. Gypsum (after anhydrite) is concentrated near margins of the pipes.

Pipe 2 at the surface, in place of a massive silica cap, instead is marked by variable concentrations of quartz veins and veinlets in networks that cut pebbly conglomerate and pebbly sandstone. The veined pebbly conglomerate and pebbly sandstone at this pipe extends outward to where it eventually is covered by unconsolidated Quaternary deposits.

Figure 7.8 - Schematic Geological Cross-Section Through Pipe 1 (Left), Showing Alteration Mineral Changes with Depth (Right)



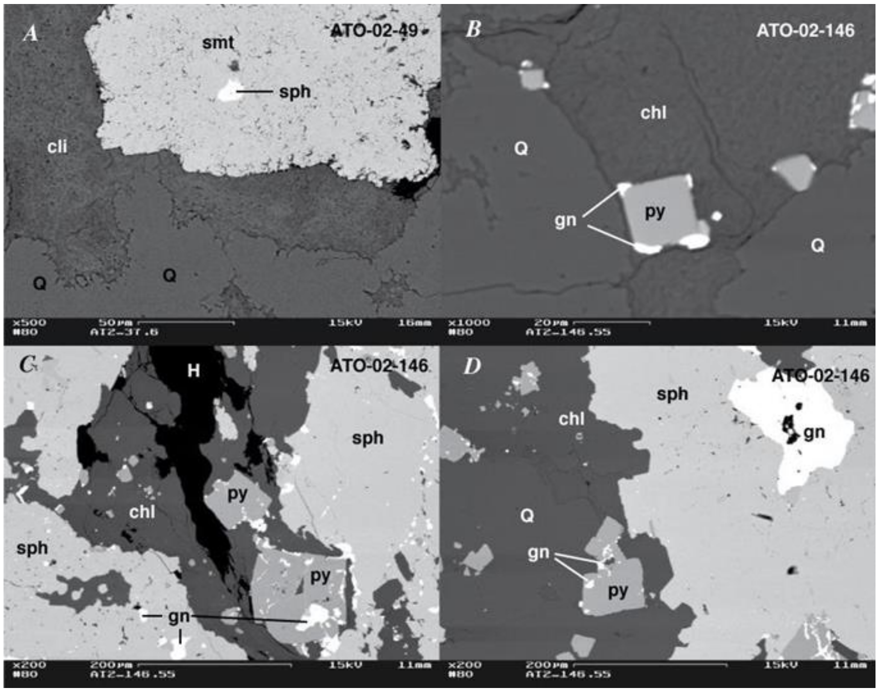
Source: NI 43-101 ATO Gold Project, 2017

Back scattered electron micrographs (Figure 7.9) of polished thin sections show clinochlore (cli) and chlorite (chl) compatibility with various sulphide and carbonate minerals in drill core at ATO Pipe 1.

- A (top left): Sample DDH ATO-02-49; sph, sphalerite; Q, quartz; smt, smithsonite.
- B (top right): Sample ATO-02-146. Chlorite associated with pyrite (py) and galena (gn).
- C (bottom left): Sample ATO-02-146. Same as B. H, hole in polished thin section.
- D (bottom right): Sample ATO-02-146. Galena inclusions in pyrite and sphalerite mantled by chlorite.

Though quartz is the dominant alteration mineral at the surface, magnesium minerals also are widespread in the pipes. They, in essence, replace rock flour during pipe development and eventually comprise a fluidized matrix together with iron sulphide and base-metal sulphide minerals. Their importance is well indicated by mineralised core contents of many 10s of thousands ppm Mg to greater than 100,000 ppm Mg throughout the mineralised pipes. The dominant Mg alteration mineral in the pipes is clinochlore (hydrous Mg Al silicate member of the chlorite group). With increasing depth in the pipes, clinochlore becomes progressively enriched in Fe and assumes petrographic characteristics of typical chlorite and, in turn, becomes associated with increased abundances of phlogopitic white mica.

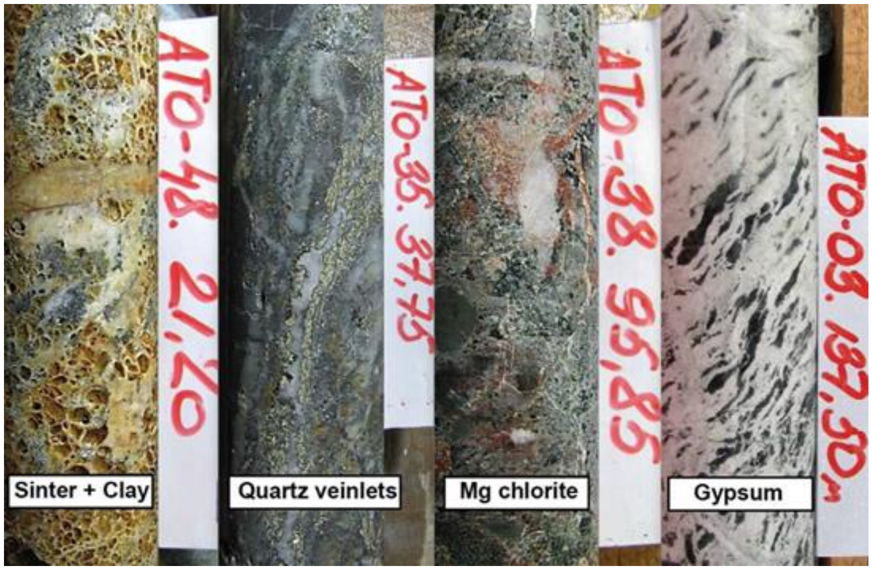
Figure 7.9 - Polished Thin Section Micrograph



Source: NI 43-101 ATO Gold Project, 2017

Alteration assemblages typically appearing in drill core at ATO (Figure 7.10, left to right) include (1) oxidized siliceous sinter; (2) steeply dipping quartz veinlets with associated pyrite and base metals; (3) matrix supported mineralised breccia where magnesian chlorite is the dominant hydrosilicate in the matrix; and lastly (4) gypsum after anhydrite especially concentrated near pipe margins. As depicted in Figure 7.10 (ATO-35-37.75), many quartz veinlets containing abundant sulfide minerals can be steeply dipping in the pipes (i.e., essentially parallel to core axes). In addition, Mg chlorite (clinochlore) plus sulfide minerals also may be disseminated widely throughout heavily mineralised core where the rocks in effect are matrix supported by alteration silicates and sulfide minerals (ATO-38-95.85). These relations, together with presence of well-defined sulfide-silicate banding in many veins (see below), suggest veining in the pipes continued well after initial replacement of rock flour during the earliest stages of mineralisation.

Figure 7.10 - Alteration Assemblages Typically Seen in ATO Drill Core



Source: NI 43-101 ATO Gold Project, 2017

7.9 Silica Cap in Pipe 1

Because of the importance that the silica cap at Pipe 1 played in the discovery of the mineralised pipes at ATO it was described in detail in the 2017 NI 43-101 Report[6] (but is not repeated in full here for brevity).

Silica cap, though presently recrystallized multiple times during repeated passage of fluids streaming upwards through the pipe, has a number of characteristics that indicate it was originally deposited as colloform-banded sinter near the original paleosurface of an intermediate sulfidation system. Silica cap rock is highly resistant and readily recognizable even from moderate distances because of its stark white colour against a dark landscape. Figure 7.11 shows (clockwise from top left):

a. Ribs of silica marking margins of empty cavities formerly occupied by sulphide minerals, mostly pyrite but also a number of Pb minerals.

b. Feeder vein of quartz (outlined in red) into basal part of banded quartz veins.

c. Angular blocks of colloform-banded quartz veins engulfed by additional vein quartz. Note matchbox near centre top of photo for scale.

d. Close up view of recrystallized banded silica showing cavities formerly filled by mostly pyrite.

[6] 2017 NI 43-101, Section 7.3.3.1, pp40

Figure 7.11 - Silica Cap Outcrops at Pipe 1

Source: NI 43-101 ATO Gold Project, 2017

Highly disrupted banded quartz veins comprise a silica cap rock in Pipe 1 at ATO. These silica outcrops are dominated by quartz which, under the microscope show effects of repeated recrystallization, commonly marked by newly grown quartz transecting growth zones in previously crystallized colloform or banded quartz. Potassium feldspar is not present in these rocks.

Almost all silica cap rock encountered by drilling is oxidized and includes various abundances of secondary minerals and traces of primary sulphide minerals. Silica cap is made up predominantly of quartz (sparse chalcedonic fabrics are preserved) and variable concentrations of iron oxide minerals, as well as less abundant kaolinite, illite, numerous primary and a number of secondary Pb minerals (rare galena mostly preserved in a surrounding mantle of a Pb–Al phosphate (plumbogummite); pyromorphite (Pb phosphate); and Pb–Mn oxide minerals), sphalerite (rarely encapsulated in quartz), arsenopyrite (also in quartz), argentite, barite (in narrow micro veins as well as tabular crystals in open cavity fillings), clinochlore, chlorite, and rare prehnite. It is the secondary Pb minerals at ATO that provide the source for the strong Pb anomaly in soils at ATO.

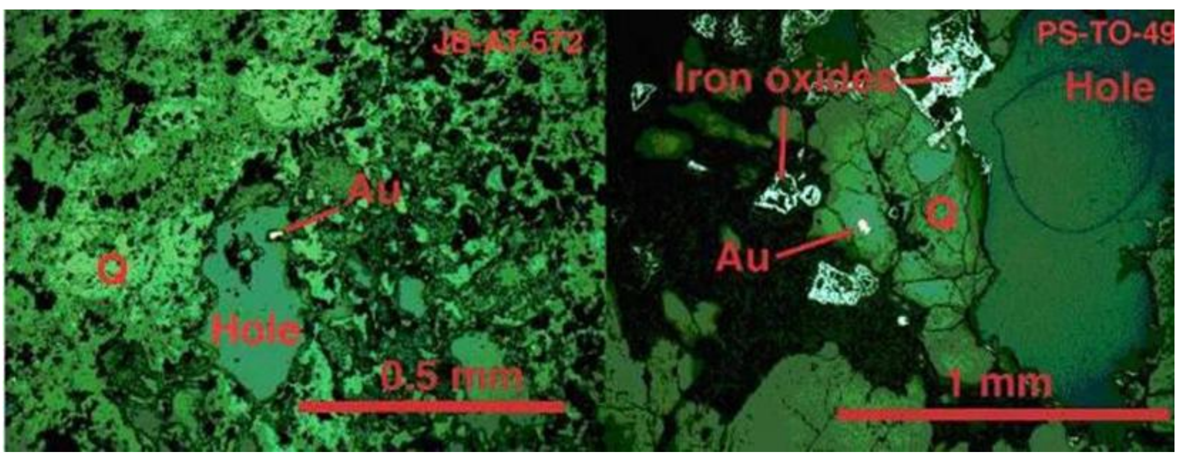
Outcrops on Pipe 1 indicate overall that its silica records a complex geologic history. The rocks display highly disturbed almost chaotic orientations even within individual outcrops of about 3-5 m wide. Banded and crustified silica has orientations that are extremely variable. Further, presence of now bladed silica that replaced earlier deposited calcite indicates that boiling had to have occurred near the top of the ATO system, wherein removal of CO_2 after breakdown of bicarbonate led to deposition of early paragenetic stage, bladed calcite at the paleo uppermost levels of the silica cap. A boiling environment must have contributed to further disruption of the rocks, as well as a number of other geologic events. This boiling environment must have been below the water table underlying sinter at the actual paleosurface of Pipe 1.

Some surface outcrops of silica cap initially must have crystallized during mineralisation at ATO. Presence of reticulated mats of silicified reeds both in longitudinal section and in cross section are well preserved in some grab samples. Typically, phosphorous contents of drill core through silica cap are in excess of 1,500 ppm. The fossil reeds at ATO are inferred to be somewhat analogous to reeds present today in thermal ponds surrounding the geysers at Yellowstone National Park (USA). In fact, the micro plumose or feathery outlines of the reeds preserved in thin section at ATO are quite similar to those of present-day reeds at Yellowstone.

All introduced silica, especially below the surface, shows complex recrystallization textures indicating repeated passage of fluids associated with base and precious metal introduction. Undoubtedly near-surface siliceous sinter and banded silica must have largely recrystallized as the sinter was broken and disrupted during repeated passage of mineralizing fluids. From the present day extent of the silica cap at Pipe 1 the paleo thermal field must not have had that wide a footprint.

Most importantly, particles of free gold containing variable amounts of silver are present in surface samples of silica cap rock. Figure 7.12 shows photomicrographs in reflected light of particles of free gold (Au) in surface grab samples at ATO. Sample JB-AT-572 (left) from silica cap at Pipe 1; Sample PS-TO-493 (right) at Pipe 2.

Figure 7.12 - Photomicrograph of Free Gold Particles in Cap Rock Surface Grab Samples



Source: NI 43-101 ATO Gold Project, 2017

Presence of free gold in surface samples of silica cap at ATO was further verified using the SEM. Some of this free gold can have about a 60/40 ratio in its Au/Ag content, and gold has been shown by drilling to be especially concentrated throughout the silica cap portion of the underlying pipe.

Though recognition of Au in high-grade grab samples from surface outcrops proved to be relatively straightforward by both SEM and standard petrographic methods, this turned out to become increasingly difficult at depth as the mineralised system becomes highly enriched in galena.

7.10 Styles of Sulphide Mineralised Rock at ATO

A number of styles of sulphide-mineralised rock are present below the oxide zone in the pipes at ATO, ranging from disseminated flooding by sulphide minerals in matrix of breccia to multiply banded veins. The latter may be either flat lying or steeply dipping.

Figure 7.13 shows general fabric of mineralised breccia in diamond holes ATO-19 and ATO-20, Pipe 2.

a. Interval contains 1,783 ppm Cu and 33.9 ppm Ag.

b. Amythestine quartz associated with introduction of galena and sphalerite. Interval contains 21.1 ppm Ag, 47,200 ppm Pb, and 52,900 ppm Zn.

c. Abundant calcite–impregnated breccia near pipe margin where breccia fragments are sugary textured, chilled margin of post-mineral dike. Interval contains 0.6 ppm Ag, 1,164 ppm Pb, and 2,196 ppm Zn.

d. Sphalerite (pale brown) and galena (blue gray). Interval contains 1.94 ppm Au, 5,738 ppm Cu, 146,900 ppm Pb, and 106,300 ppm Zn.

e. Quartz-sphalerite banded veins. Interval contains 19.2 ppm Ag, 79,400 ppm Pb, and 133,400 ppm Zn.

f. Breccia including angular fragments of pale brown meta-siltite.

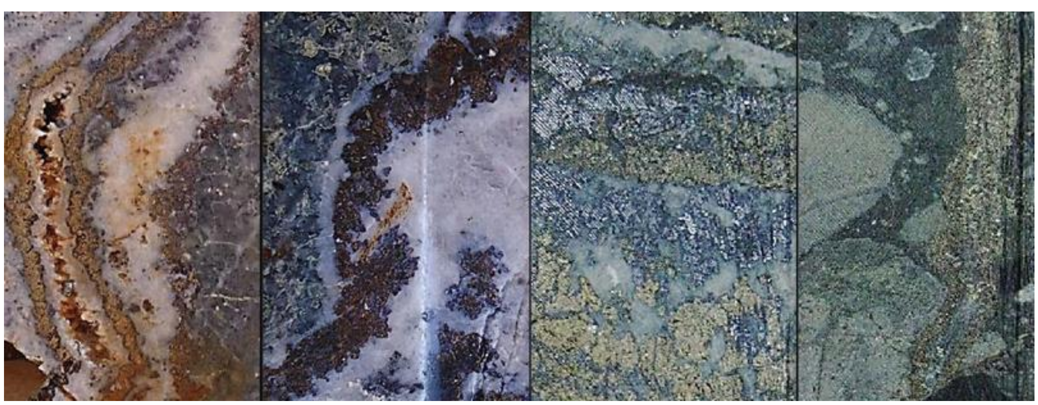
Generally, sphalerite becomes more Fe rich with depth in the system. Compare honey brown sphalerite at 82.4 m in hole ATO-19 in Pipe 2 (A) with brown sphalerite at 97.80 m associated with weakly amethystine quartz (B). Or compare honey brown sphalerite at 85.9 m in hole ATO-20 (E) versus dark brown sphalerite at 193.9 m (F). The latter is associated with amethystine quartz.

Figure 7.13 - Mineralised Breccia Fabrics from Pipe 2

Source: NI 43-101 ATO Gold Project, 2017

Figure 7.14 shows general styles of mineralised rock at ATO (from left: ATO-20 85.9 m; ATO 142.7 m; ATO-111 266.5 m; ATO-111 275.1m).

Figure 7.14 - Mineralised Rock Styles at ATO



Source: NI 43-101 ATO Gold Project, 2017

Flat-lying galena-sphalerite flooding in diamond hole ATO-111 at 266.5 m (second from right in Figure 7.14) can be followed down hole within a few meters by steep veins at 275.1 m (right). The latter veins also cut paragenetically earlier disseminated sulphide minerals that form a matrix support to mineralised breccia. In addition, late paragenetic stage manganiferous calcite, in places actual rhodocrocite, rarely cuts across locally layered breccia. In addition, the multiple bands of sulphide minerals and quartz in many veins suggest a protracted period of vein emplacement after initial onset of brecciation associated with pipe emplacement.

Overall, the mineralised system at ATO really does not contain that much manganiferous calcite, though some is present rarely in cm wide veins.

7.11 Mineralisation Genesis and Classification at ATO

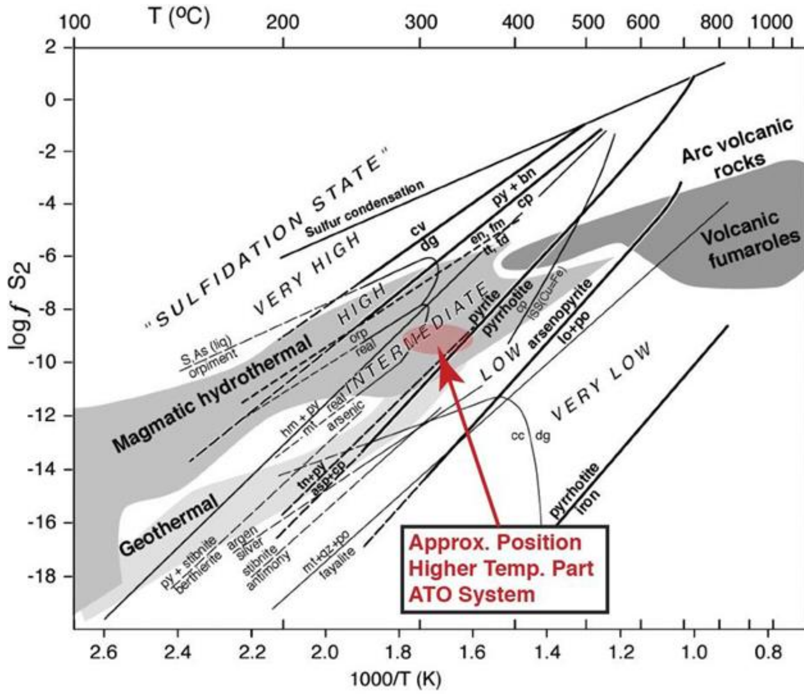
Summary: Mineralization at ATO is summarised as:

- An intermediate sulfidation system (IS).
  - Neutral low temperature, near paleo surface fluids, bladed silica after calcite indicates some local boiling. Related to Jurassic magmatic event.
  - Banded silica, broken sinter, repeatedly recrystallized at paleo top.
- Confined to pipe bodies.
  - Multiple collapse and upward transport in the pipes, repeated brecciation followed by continued ingress of steep and shallow veins, veinlets and flooding.
- Magnesium chlorite and silica dominant system.
  - Quartz, clinochlore (high Mg, Al silicate), kaolinite, gypsum (peripheral after anhydrite), adularia is absent.
- Dominated by free gold, base metal and Ag sulphides.
  - Lead-phosphates (near surface) – Pb-carbonates – galena (at depth).
  - Zinc-carbonate (near surface) – low FeS sphalerite (at depth).
  - Au in quartz & in sulphides; Ag mostly in tetrahedrite & miargyrite.

Figure 7.15[7] shows a diagram of sulphur fugacity versus temperature and various sulphide mineral assemblages in epithermal deposits that reflect sulfidation states varying from low through intermediate to very high sulfidation states. Approximate compositional fields of geothermal, magmatic hydrothermal, and volcanic fumaroles are shown in varying shades of grey.

[7] From Sillitoe and Hedenquist, 2003. *Linkages between volcanotectonic settings, ore-fluid compositions, and epithermal precious-metal deposits*. In Simmons, S.F., ed., Giggenbach Volume, Society of Economic Geologists and Geochemical Society, Special Publication 10.

Figure 7.15 - Diagram of Sulphur Fugacity Versus Temperature



Source: ATO 2021 Mineral Resources, Technical Report (Amended NI 43-101), 2021

The approximate position of higher temperature parts of the ATO mineralised system is shown in pink on the basis of sulphide mineral assemblages stable in pipes.

Details: The mineralised pipes at ATO are unusual from a variety of standpoints. Nonetheless, a number of general statements can be made concerning their genesis and classification. As noted above, the ATO base (Pb-Zn-(Cu)) metal and Au-Ag mineralised pipes are intermediate sulfidation (IS) epithermal deposits characterized by an absence of adularia. They are apparently magma affiliated on the basis of their sulphur isotopic data and must thus be associated with emplacement of Jurassic igneous rock at depth. The predominance of low FeS sphalerite, galena, Ag-bearing tetrahedrite-miargyrite, and chalcopyrite at ATO unquestionably are all compatible with an IS state (Figure 7.15).

However, as opposed to a strong correspondence of IS deposits worldwide (Western US, Peru, central and Eastern Europe) with andesitic magmas (Sillitoe and Hedenquist, 2003), ATO appears instead to be associated with more mafic, dioritic Jurassic arc terrane magmas. These magmas thereby must have contributed to a correspondingly high $Mg/_2H+$ component in fluids associated with mineralisation that in turn inevitably led to widespread presence of the Mg chlorite clinochlore as a gangue mineral in the pipes as opposed to adularia. In addition, Jurassic volcanic rocks are not present in the region of ATO even though the pipes appear to have been formed near surface (banded silica, siliceous reed-bearing sinter at Pipe 1). This in turn suggests rapid emplacement of the pipes occurred as a temporally transient or fleeting event. Regardless, well-formed sulphide-silicate banded veins in the pipes suggest protracted veining continued after formation of the brecciated columns of rock. However, the question still remains as to whether or not the mineralised pipes at ATO represent distal parts of a deep-seated porphyry environment.

Thus, upward flaring brecciation in Pipes 1, 2, and 4 at ATO provided fluid pathways for ingress of fluids associated with an open-space filling mineralisation event that occurred over a protracted time interval. Mineralization in the pipes does not represent an instantaneous event essentially contemporaneous or immediately following the pipes' piercing through their surrounding host rocks. Fist-sized veins, well banded with successive layers of early low FeS sphalerite and somewhat later galena, together with precious metals, in the pipes attest to a relatively protracted time for individual vein emplacement within the confines of the pipes.

Origin of brecciation remains enigmatic, however, and shows features of both magmatic and hydromagmatic breccias, as well as tectonic breccias (see Sillitoe, 1985). Additional complications hampering straight forward interpretation results from superposition of brecciation associated with mineralisation onto detrital fragment-rich Early Permian volcaniclastic rock and Jurassic pebbly conglomerate, as well as protracted passage of fluids associated with mineralisation in the pipes. Nonetheless, some fine-grained, matrix-supported breccias with abundant rock flour encountered at depth at ATO north-west of Pipe 2 are typical of magmatic breccia-style diatremes (magmatic hydrothermal systems that extend to surface). Rock flour where present at ATO, though important from a pipe genesis standpoint, is itself not mineralised, and represents those brecciated rocks that did not encounter mineralizing fluids and thus were spared alteration during subsequent passage of the fluids.

After near-surface deposition of banded siliceous sinter in an essentially horizontal orientation, sinter was disturbed into jumbled blocks as the process of underlying mineralisation continued to evolve. This may account for the highly discordant attitudes of sinter layering among nearby outcrops at Pipe 1, though tectonic disturbance after pipe emplacement also may have contributed to the discordant attitudes. Banded and crustified silica has orientations that are extremely variable in outcrop at Pipe 1. Further, presence of now bladed silica that replaced earlier deposited calcite indicates that some boiling had to have occurred near the top of the ATO system, wherein removal of CO2 after breakdown of bicarbonate led to deposition of early paragenetic stage, bladed calcite at the paleo uppermost levels of the silica cap. A boiling environment also must have contributed to disruption of the rocks, as well as a number of other geologic events. This boiling environment must have been below the water table underlying the well-developed sinter at Pipe 1.

The pipes at ATO were cut by a number of post-mineralisation fairly flat-lying faults and dikes. Norton and Cathles (1973) note that the primary question concerning any hypothesis of breccia development is how the void amongst the fragments was created. Voids in breccias typically provide sites that are filled partially to completely by subsequently introduced gangue (quartz, magnesian chlorite, phlogopitic white mica at ATO) and ore minerals (sphalerite, galena, chalcopyrite, pyrite, tetrahedrite /tennantite, miargyrite, and particles of free gold at ATO). As further noted by Sawkins and Sillitoe (1985), the largest variety worldwide of breccia types resides in magmatic arc terranes.

Sillitoe (1985) lists the following six fundamental mechanisms for formation of breccias:

- Release of fluids from high-level magma chambers during second boiling;
- Magmatic heating and expansion of meteoric pore fluids;
- Cool ground waters interacting with magma;
- Magmatic-hydrothermal brecciation, including pre- and post-mineralisation diatremes;
- Mechanical disruption of a magma's wall rocks by subsurface movement;
- Fault displacements.

For a variety of reasons, at ATO the most likely mechanism associated with emplacement of Pipe 1, 2, and 4 is the fourth one listed above (i.e., magmatic hydrothermal brecciation). Decompression of volatiles associated with second boiling in a dioritic complex at depth led to formation of the permeable channel ways that subsequently were filled by mineralised rock. Another characteristic of these types of breccias is that in many districts they tend to be present in clusters (Sillitoe, 1985), a characteristic that applies to ATO. Yet, what is observed at ATO, and not at many other magmatic-hydrothermal breccia occurrences, is upward pipe termination where at Pipe 1 now highly disturbed siliceous sinter (and highly Au-mineralised as well) is present very close to the original paleosurface.

Nonetheless, some differences from the norm for many well-described, mineralised magmatic-hydrothermal breccias elsewhere characterize the breccias at ATO. These differences include typically diffuse or poorly marked pipe margins at ATO attributable largely to a relatively prolonged mineralisation event that continued well after cessation of the brecciation that first created the voids.

8 DEPOSIT TYPE

The information in this section is largely drawn and/or summarised from the Report available on SEDAR entitled: "Altan Tsagaan Ovoo Project (ATO), 2021 Mineral Resources, Technical Report (Amended NI 43-101)", prepared by GeoRes, with an effective date March 30, 2021, issued on March 30, 2021 and amended on June 9, 2021. That information itself was largely extracted from the 2017 NI 43-101 Report. *Further details as documented therein remain correct and valid.*

The ATO deposit type is described in terms of:

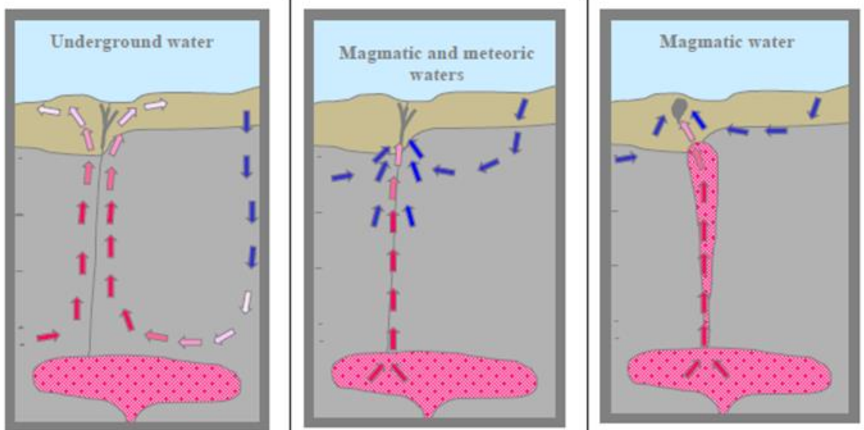
- Mineral deposit type – surface epithermal deposits with intermediate sulphidation pipes below.
- Geological model – the shape of the deposits and the metal zonation expected.
- Exploration model – essentially the area drilling.

8.1 Mineral Deposit Type

This Project's mineral deposit type (from the viewpoint of exploration and subsequent geological modelling) is that of multiple surface epithermal deposits with intermediate sulphidation (feeder) pipes below. This implies a specific shape where the top part (near or at current surface) would represent a wide thinnish roughly circular accumulation of mineralisation in country rock around an original surface ground-water-interacting hydro-thermal or fumarole vent system. Below that would be a tall root-shaped breccia pipe, flared at the top and narrowing downwards, through which the magmatic or meteoric fluids rose above a lower hot igneous body. The pipe would be vertically veined and/or brecciated.

This deposit classification is based on the ATO mineralisation texture, geochemical associations, and mineral composition. The classification process involved comparison of the ATO deposit with natures of the high, intermediate and low sulfidations of epithermal deposits (as illustrated in Figure 8.1).

Figure 8.1 - Schematic Epithermal System Types

EPITHERMAL MINERALIZATION		
LOW SULPHIDATION (LS)	INTERMETIATE SULPHIDATION (IS)	HIGH SULPHIDATION (HS)
		
Au-Ag	Zn-Pb-Ag- (Au) or Zn-Pb-Ag- (Cu-Sn)	Cu-Au-Ag or Zn-Pb-Ag
Neatral pH (increases in gas and acidity)	Neutral pH	Very acidic pH
Sinter	Sinter possible	No sinter
Quartz, chalcedony, adularia, calcite, illite	Quartz, calcite, minor chalcedony, chlorite, Mn carbonate, anhydrite (gypsum), illite	Quartz, alunite, argillic alteration minerals
Reticulate, banded, colloform	Banded veins, brecciated, colloform, cockade, crustification, and, reticulate	Vuggy, massive sulfide

Source: Altan Tsagaan Ovoo Project (ATO), 2021 Mineral Resources, Technical Report (Amended NI 43-101)", prepared by GeoRes

NB: This model applies well for Pipes 1, 2 and 4. The QP is not sure how directly it applies to the Mungu deposit – but assumes it represents the pipe root system without the development (or remains) of the surface deposit (see below also).

8.2 Geological Model for Estimation and Exploration

The geological deposit style model is described in terms of:

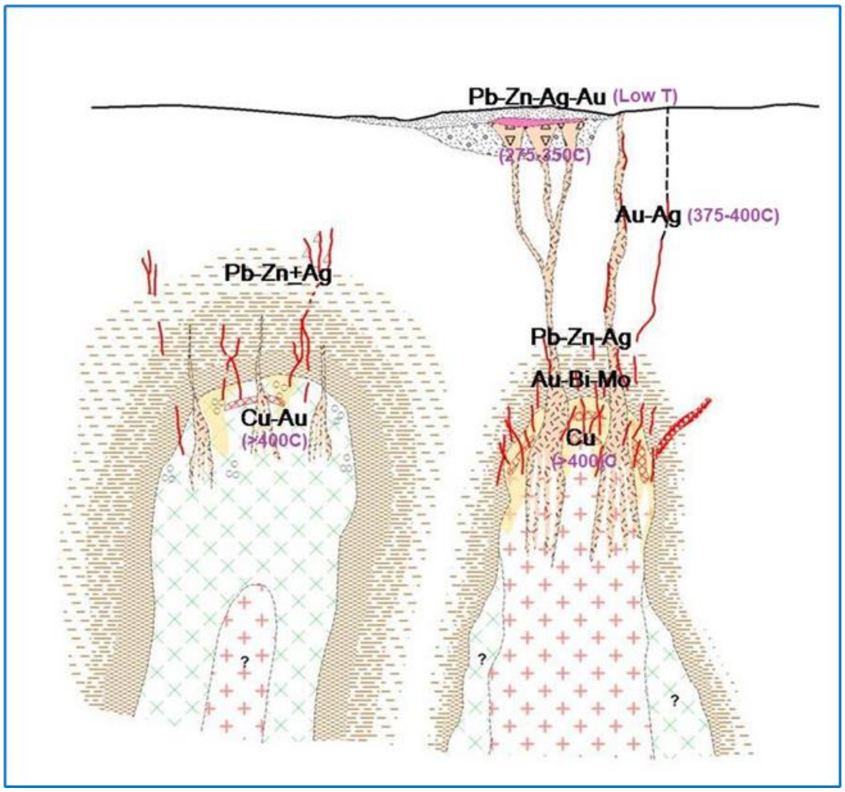
- Geological and mineralisation model – effectively defining the shape.
- Genesis of the ATO deposits.
- Geochemical zonation.

8.3 Geological and Mineralisation Model

The geology of the property consists of metamorphosed Devonian sedimentary rock overlain by a volcanic and sedimentary sequence of Permian age and remnant scraps of probable Jurassic volcaniclastic units, intruded by Jurassic plutons ranging from diorite to granite in composition and including rhyolitic phases mainly as dikes. Petrographic study suggests simple, single-pulse injections of the intrusions, with late-stage generation/expulsion of felsic phases and contemporaneous concentration of metal-rich fluids.

Vertical metal zonations and mineralisation temperatures expected to be associated with Jurassic intrusives at ATO are shown in Figure 8.2.

Figure 8.2 - Cartoon at ATO of Jurassic Intrusives with Streaming Mineralisation Emanations Above



Source: Altan Tsagaan Ovoo Project (ATO), 2021 Mineral Resources, Technical Report (Amended NI 43-101)", prepared by GeoRes

The rock types are:

- Diorite-granodiorite intrusions – green crosses;
- Granite intrusions – pink plusses;
- Rhyolite upwardly extrusive dykes – tan with flecks and triangular breccia symbols;

- Deuteric alteration above intrusions – yellow;
- Hornfels contact metamorphosed wall rocks – brown shading adjacent to intrusions, decreasing outwards;
- Quartz veins – red;
- Sinter cap at surface – dark pink.

Mineralization at ATO is probably related to the Jurassic intrusive magmatic rocks as sources of heat, metals, and fluids (Figure 8.2). The main ATO system (Pipes 1, 2 and 4, represented by the intrusive and surface deposit on the right in Figure 8.2) is associated with a stratigraphic unit that appears to be localized in a graben or collapse feature that is possibly but not certainly Jurassic in age; any stratigraphy could potentially be prospective for this style of mineralisation. The ATO mineralisation appears to have occurred as a protracted single-stage process in separately upward-flaring pipe-shaped bodies, with temperatures ranging from ~300-350°C at depth to possibly ambient temperature in surficial sinter.

The nearby Mungu mineralisation (possibly represented by the intrusive on the left in Figure 8.2) also appears to have occurred in a single pulse, but at higher temperatures of ~375-400°C intimately associated with emplacement of rhyolite.

Metal zonation: In a broad sense, metal zoning shows a clear, classic pattern of intrusion-centred copper (plus or minus molybdenum, tungsten, gold, and other elements) outward to country-rock hosted lead, zinc, and silver (plus or minus gold, arsenic, antimony, mercury, and other elements). It is presumed that this lateral zonation also occurs vertically, as evidenced by increasing copper values at depth in ATO.

Intrusions: The three main Jurassic intrusive phases are:

- diorite-granodiorite;
- granite; and
- rhyolite.

The diorite-granodiorite plutons are highly magnetic and typically develop large aureoles of magnetic hornfels. They show little or no quartz veining and exhibit no significant alteration apart from weak chloritization, which may be a regional metamorphic effect. On their upper and outer contacts, they locally have patchy albite-sericite (muscovite) zones which are considered to be simple deuteric alteration related to final crystallization processes. In some cases, they appear to be zoned inward to more felsic phases. Pegmatite-aplite phases show diffuse margins, suggesting segregation and streaming of more felsic fractions during late-stage crystallization. Miarolytic cavities are common and locally abundant, containing coarse euhedral biotite, magnetite, pyrite, chalcopyrite, and local bornite.

The granite plutons are moderately magnetic and produce smaller, patchy hornfels aureoles. They may locally be zoned outward to more mafic composition. The upper and outer contacts typically show patchy to pervasive sericite (muscovite) alteration and common to ubiquitous grey quartz-sulphide veins which are often druzy. Pegmatite-aplite phases are common and locally show possible unidirectional solidification textures, and rhyolitic phases with diffuse margins are present, all suggesting segregation and streaming of volatile-rich phases during late-stage crystallization.

The rhyolitic phases are typically emplaced as dikes and small plugs. They are moderately magnetic but do not produce contact metamorphic aureoles. The rhyolites are typically pyritic, and locally highly so.

Intrusion metal associations: Metal patterns appear to vary systematically with intrusive composition.

The diorite-granodiorite bodies have a copper-gold-tungsten signature related to visually obvious disseminated sulphide and sulphide-filled miarolytic cavities. Flanking hornfelsed country rocks generally show annular halos of geochemically anomalous lead and zinc, and locally contain percent-level concentrations of base metals plus or minus silver.

The granite bodies show essentially the same patterns, with some differences. Granite intrusions show copper anomalies, but typically at only geochemical levels, and the anomalies in the flanking country rocks have a more distinctly silver-rich character. The largest apparent difference however is the local development of a gold-bismuth-molybdenum zone in an intrusion-proximal setting within hornfels.

The rhyolites have different metal patterns depending on the level of emplacement. At deeper levels, where the rhyolites were confined under lithostatic load, the geochemical signature is gold-silver-copper. In this setting it appears that sulphides may have been admixed with the rhyolite magma as it was being emplaced, and it is likely that metal deposition was caused by cooling and fluid mixing. At shallower levels under hydrostatic conditions the geochemical signature is lead-zinc-silver-gold, with metal deposition related to cooling, possible boiling, and possible wall rock interactions.

Mineralisation temperatures: There is relatively little hard data available on temperatures and isotopes for mineralisation in the ATO district. Petrographic relationships would indicate that the disseminated and miarolytic cavity-filling sulphides in the diorite-granodiorite intrusives were deposited at magmatic temperatures of 400°C or more, and the temperatures would be roughly equivalent or slightly lower for the granites. Mineral geothermometry using paired arsenopyrite and sphalerite in drill core from ATO gives ~275-350°C. The lower end of the possible temperature range is suggested by siliceous material at the outcrop of ATO Pipes 1 and 2, which has been interpreted as sinter based on textures and high phosphorus contents.

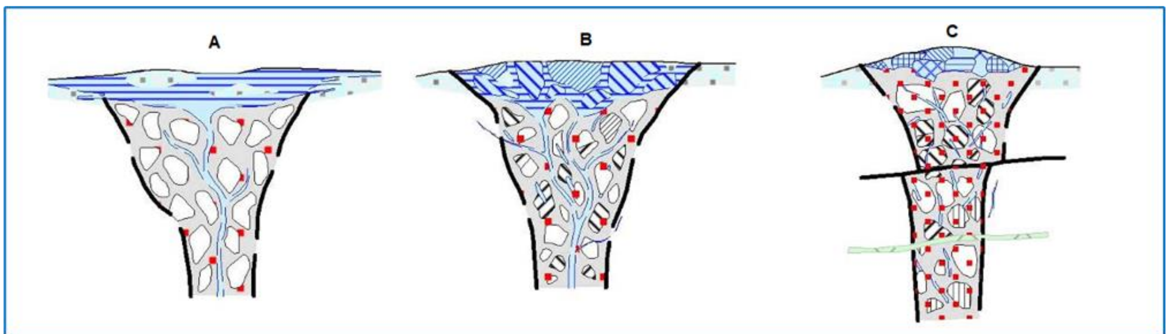
8.4 Genesis of the ATO Deposits

It is believed that the ATO deposit is an epithermal gold and polymetallic deposit of transitional sulphides in breccia pipes in a Mesozoic continental rift zone in eastern Mongolia. Below are accounts on similarity of ATO to other deposits in terms of formation of breccia pipe, its development stages, geochemical zonation, and the type of deposit genesis.

Formation of breccia pipe: Tectonic-magmatic activities began to take place in the area around ATO deposit in Early Jurassic when hydrothermal and metasomatic alterations and mineralisation were formed in relation to intrusions and dykes. Absolute age of an intrusive massive located outside and east of ATO prospect area was determined to 189 million years. n addition, an Early Jurassic massive was formed in the southern part of the area comprises two phases of rocks of sub-alkaline series distributed in small, separate outcrops of small intrusions. Criteria or signs for copper porphyry mineralisation have been noticed in these two intrusions. It is believed that the pipe bodies of ATO deposit were formed in a structure that resulted from partial melting and upheaval of magmatic fluids. This can be explained in more detail in a model proposed by Noel White.

In the area of ATO deposit, a small basin was deposited with sediments in Lower Jurassic, which include sandstone, siltstone, tuff sandstone, and gravelite, conglomerate, with coarser rocks at the bottom and finer rocks at the top. Quartz sinter (blue layers in Figure 8.3) was accumulated in pipe structures at ATO deposit as a result of hot spring activities (A in Figure 8.3) at the same time as the formation of these sedimentary rocks. This can be explained by the facts that sinter materials and fragments of low temperature chalcedonic quartz vein are found in the gravelite, and that lenticular bodies of gravelite are found in the sinter.

Figure 8.3 - Model of Upper Breccia Pipe Formation At ATO



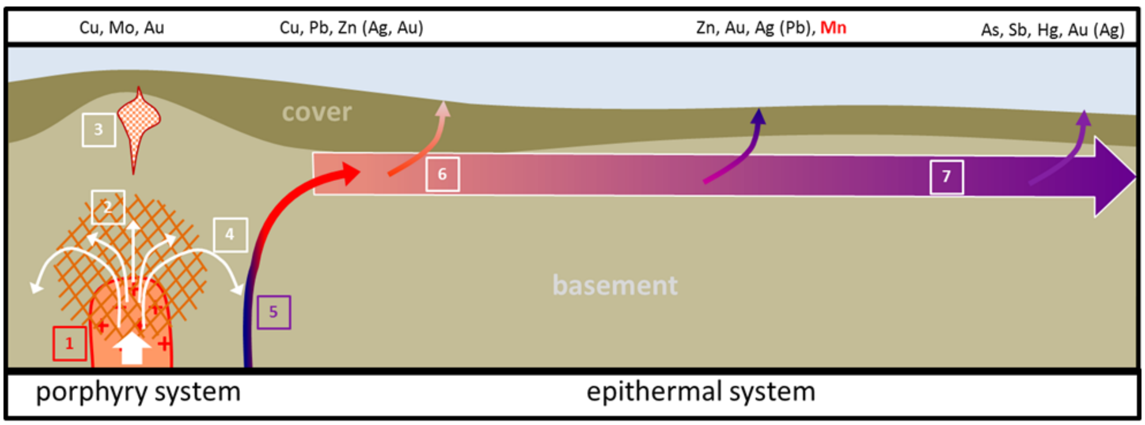
Source: Altan Tsagaan Ovoo Project (ATO), 2021 Mineral Resources, Technical Report (Amended NI 43-101)", prepared by GeoRes

After the formation of layered quartz accumulation, or sinters, on the surface of the ground, the sinters were broken apart into blocks of varying sizes (individual upper blue layered blocks in B, Figure 8.3). This explains the outcrop called Pipe 1 where the layered quartz sinter is broken into blocks and the layering in these blocks has become disordered and oriented in random directions. Sedimentary rocks in the pipe structures become brecciated allowing flow of fluids and providing a domain for mineralisation. Lower pipe-form mineralisation (grey parts) were subjected to post-mineralisation horizontal faulting and the pipes appear to be displaced along the fault. Small bodies of diorite have been found in these faults.

8.5 Geochemical Zonation

The pipes that have been identified in the deposit are aligned in a row in a structure with a trend from south east to north west and form mineralisation with differing horizontal geochemical zonation with certain metallic concentrations. Noel White (2001) devised a pattern of flow of fluids based on this geochemical zonation similarity with that found adjacent to porphyry copper intrusions (presented schematically in Figure 8.4). White's model postulates that the horizontal mineral zonation between the ATO pipes could originate from a large intrusion (such as a porphyry copper) located tens of kilometres away from ATO deposit.

Figure 8.4 - Schematic Relationship of ATO Mineral Zonation to a Porphyry Copper System



Source: Altan Tsagaan Ovoo Project (ATO), 2021 Mineral Resources, Technical Report (Amended NI 43-101)", prepared by GeoRes

Of the ore minerals sphalerite is the most common mineral in the ATO pipes with light yellow sphalerite as the dominant variety. In addition, abundance of veins and veinlets of gypsum indicate the oxidizing environment of the fluids and original magma. It is very likely that such kind of magma may be connected to porphyry mineralisation (1 in Figure 8.4). A gold and copper molybdenum stockwork would surround the intrusion (2 in Figure 8.4), with a high sulfidation epithermal mineralisation on top of them (3 in Figure 8.4). Magmatic fluids with high metallic content (5 in Figure 8.4) would rise upward, halt at certain level, and drift horizontally away along open spaces to get mixed with underground water and change its composition. After that the fluids would break the cover burden at some points to travel out to the surface through several different routes (6 in Figure 8.4).

It is postulated that this process could explain the differing metal zonation between the ATO pipes. Concentration of metallic components varies from pipe to pipe due to differences between pH levels, variations in pressure and temperature, and the differing composition of the fluids that were injected to the pipes of ATO deposit. It ranges from copper rich to lead and zinc rich and this explains the geochemical zonation in the deposit. As for ATO deposit, the burden that kept the fluids from traveling upward is thought to be a unit of very dense and massive, black coloured siltstone. The siltstone unit occurs at the bottom of some of the deeper drill holes having thickness ranging from a dozen meters to 30-40 m. A break in the black siltstone unit caused hydrothermal explosion and rapid upheaval of CO_2, which resulted in formation of breccia pipes. Fluids rose through open spaces between breccia fragments where pressure and temperatures were lower and gangue and ore minerals filled them completely or partially. As a result, mineralisation took place.

8.6 Exploration Model

The exploration model for recent exploration (since the general appreciation of the location and extents of Pipes 1, 2 and 4 and Mungu) has been to drill and sample holes on an increasingly close regular pattern across the deposits. Drilling commenced with relatively short vertical holes at wide spacing. This could be characterised as mostly drilling the upper oxidised material. More recent drilling has mostly been with closer and longer holes predominantly driller towards the south east. This could be characterised as drilling the lower transitional and fresh material.

The drilling model is essentially to traverse the deposits fully in plan and to a reasonable depth. Hole spacing and short sampling intervals would be tight enough to ensure grade continuity between holes.

9 SURFACE EXPLORATION (EXCLUDING DRILLING)

The information in this section is largely drawn and/or summarised from the Report available on SEDAR entitled: "Altan Tsagaan Ovoo Project (ATO), 2021 Mineral Resources, Technical Report (Amended NI 43-101)", prepared by GeoRes, with an effective date March 30, 2021, issued on March 30, 2021 and amended on June 9, 2021. Further details as documented therein remain correct and valid.

Exploration (other than drilling) at ATO is described in terms of:

- Surface exploration methods and parameters.
- Sampling methods, quality and representation.
- Sample data details.
- Surface exploration results and interpretations

9.1 Preparation for Exploration

At this stage, reports and related materials, including maps, sketches, and logs, of historical mapping and prospecting works conducted by earlier researchers were collected, compiled and studied. Interpretation of aerial and satellite images were also performed. Coordinates used by CGM for exploration on the Project is UTM coordinates with the datum set to WGS-84, Zone 49N. The boundary coordinates of the exploration and mining licenses are defined by latitude and longitude coordinates.

9.2 Field Work Programs

The field investigation that was conducted in the license area can generally be subdivided into two stages: prospecting and exploration.

In 2003-2009, geologists of COGEGOBI, who had been specialized in prospecting and exploration of uranium projects, conducted geological mapping and prospecting traverses and collected geochemical samples in the exploration license area, supplemented by a magnetic survey. The result was they discovered an epithermal gold occurrence.

In 2010, CGM carried out a prospecting stage consisting of geological mapping and prospecting traverses, surface and other sampling tasks, variety of geophysical surveys, and some trenching and drilling. As a result of the prospecting, ATO occurrence was chosen to a detailed study, and an intensive drilling program began in late 2010 to advance the project to exploration stage.

9.3 Geological Mapping

Systematic mapping and prospecting traverses carried out in the entire ATO exploration license area. Traverses were placed 100 m apart. During traverses, grab samples were collected from alteration zones and rocks with possible mineralisation. As a result, a 1:25,000 scale geological map was produced covering the entire license area.

A total of 397 grab samples were collected in 2010 during mapping. Systematic mapping continued until 2014 including detail mapping at the ATO prospect, as well as recon scale mapping with some grab samples.

Mapping and prospecting work was assisted by maps such as 1:32,000 scale aerial image, a satellite image, and topographic maps at scales of 1:25,000 and 1:50,000.

As a result of this work the area now contains the ATO gold-polymetallic occurrence chosen as a potential target. Detailed prospecting work was aimed to draw a shape and size of a mineralised body, boundaries of lithology and alteration zones, and study the sources of secondary dispersion halos and geophysical anomalies. Based on the result of the detailed prospecting traverses, detailed 1:10,000 scale geological map of the ATO deposit area was interpreted.

9.4 Stream Sediment Sampling

The ATO district is characterized by limited and weak systems of streams and galleys, wherein down stream sediment transport is practically non-existent. Thus, minimal stream-sediment sampling was carried out to obtain a general geochemical characteristic of the area in 2010.

As an initial task of the field work, stream sediment samples were taken from Holocene ditches and ravines, where outcrops of Late Paleozoic to Early Mesozoic rocks distributed, to detect stream sediment anomalies. Samples were taken using hand augers and shovels to dig 0.2-0.6 m deep holes and extract 1.5-2.0 kg samples from alluvial gravel and sand materials. To avoid contaminating the samples, stainless steel drills and sifters were used. Samples were air dried before sieved in a 20 mesh sifter. The samples were reduced to 150-200 g after sieving and the reduced samples were submitted to the laboratory for analyses. A total of 509 stream sediment samples were collected counting 1 sample for per square km. Coordinates of sample location were recorded by a field GPS devices, and a simple field illustration was drawn including catchments width, depth, rank, and direction; and the sand ratio, gravel and clay size, colour, degree of gravel rounding of samples; and information on nearby country rocks. A stream sediment map was produced as a result and used for further studies.

9.5 Soil Geochemical Sampling

Initial soil sampling (20 * 100 m) was completed at the ATO prospect in the fall of 2009 by CGM. During 2010, an additional 4,256 samples were collected at 100 * 100 m and 200 * 200 m in areas showing potential, and, during 2012, an additional 18,471 soil samples were collected from broad areas across the entire ATO district (gold shown in Figure 9.1). These 18,471 soil samples are from seven domains within the ATO district and include 7,290 samples collected from the Davkhar Tolgoi area south of the mineralised pipes at ATO, and 4,860 soil samples are from areas close to the pipes. Thus, broad expanses of the licensed areas to the north-north-east of the ATO deposit finally were covered in 2012 by systematically gathered soil grids, areas of the seven exploration licenses held by CGM at the end of 2012 that previously had not been sampled.

The best indicator of presence of significant, underlying gold-mineralised rock at ATO was a gold anomaly in soil at ATO. This gold anomaly in soil at ATO was as much as 600 m wide in its longest dimension and includes concentrations of 50–500 ppb Au. By far, this is the strongest gold anomaly in the immediate area of the pipes within ATO district.

In addition, a strong lead anomaly in soil is mostly coincident with the soil gold anomaly and contains generally 50–300 ppm Pb. Most lead in soil is derived from secondary lead minerals, including plumbogummite (a Pb-Al phosphate), pyromorphite (a Pb phosphate), and Pb–Mn oxide minerals in the uppermost, oxidized parts of the underlying mineralised pipes. In addition, anomalous lead in soil E–SE of the Bayangol prospect appears to be defined, in part, by the trace of an inward-dipping low-angle thrust fault exposed in some trenches, though Early Jurassic intrusive rocks emplaced into Early Permian volcanic rock, largely rhyolite, also may be contributing to the lead anomaly.

Further, anomalous >500 ppm Zn in soil is present in soil on top of Quaternary-covered mineralised rock in Pipe 4, as well as on the northern fringe of the surface projection of Pipe 1 and on the SW margin of Pipe 2.

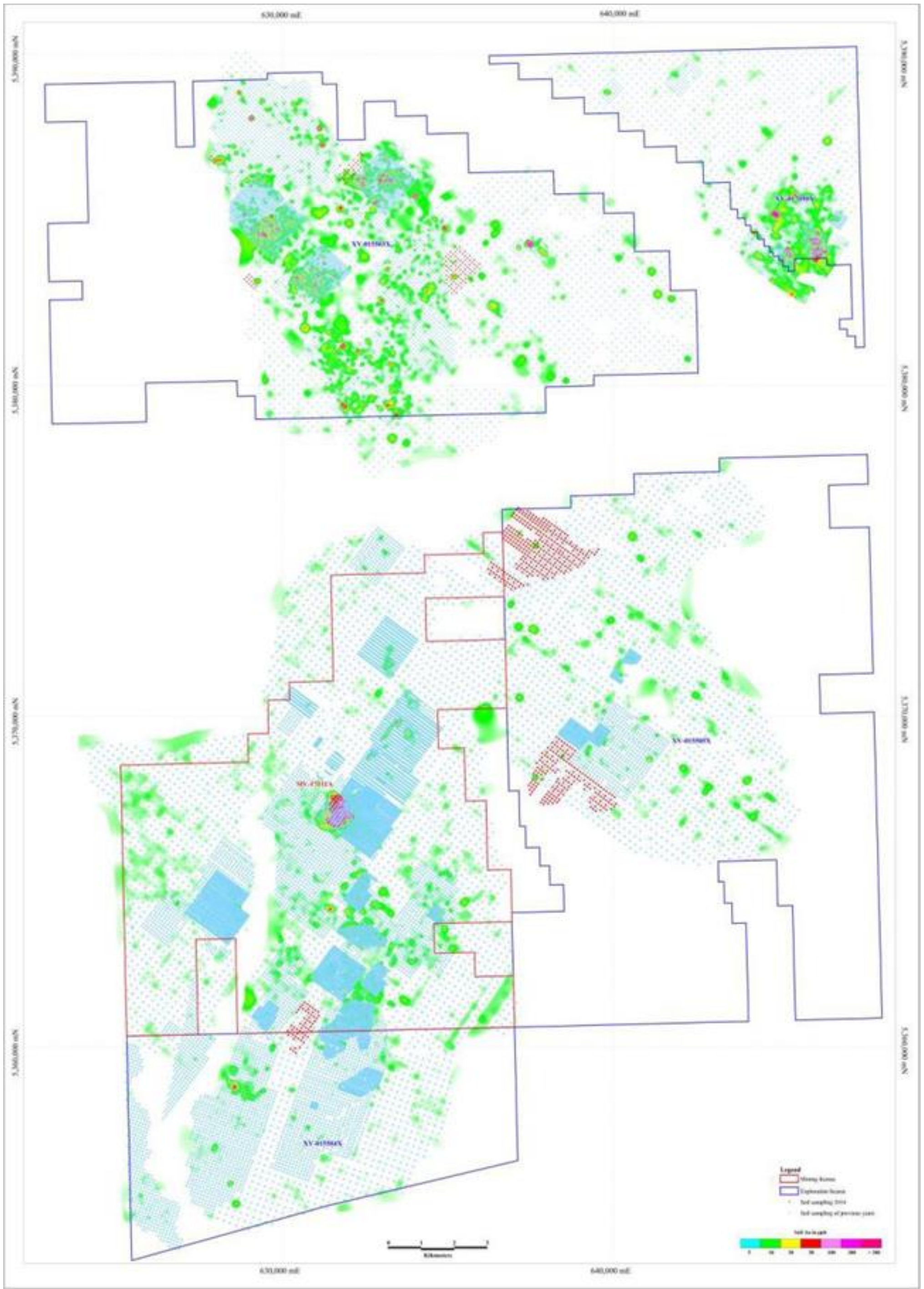
The combined Au, Ag, Pb and Zn anomaly over the pipes is illustrated in Figure 9.2.

Extensive soil sampling was undertaken in 2013 and a minor amount of soil sampling was undertaken in 2014 to infill the grid spacing in various areas on the property, and to slightly expand the grids beyond their previous coverage in a few places. Most areas were infilled from a pre-existing 200 * 200 m grid spacing to 100 * 100 m and 100 * 50 m and 50 * 50 m on target areas. The infill sampling enhanced some anomalies detected in the wider spaced grids, but did not identify any new significant anomalies.

Occurrence of rock outcrops was sparse in the license area; most of it was covered by loose sediments. Soil samples were extracted from a depth of 30-60 cm (B horizon), or form underneath the brown soil with plant roots, sometimes from 1 to 2-meter depths drilled by hand augers when the cover was significantly thick. Each weighing 1.5-2 kg, the samples were dried and sieved in 80 mesh steel sifters to produce 150-200 gr samples ready for analyses. Coordinates of sampling sites were recorded in GPS devices, and a journal was kept about the composition, colour, and structure of soil, the depth of extraction, and information on nearby rock bodies.

The red boundary in Figure 9.1 illustrates Mining License MV-17111 (the previous boundary to the current one).

Figure 9.1[8] Gold Soil Sampling in the ATO District – 2010 to 2014



[8] CGM 2014 ER.

9.6 Grab Sampling

To evaluate the potentiality of occurrences, mineralised points and altered rocks identified by the prospecting and exploration work, and to determine elemental grades, size and shape, and boundaries of mineralisation, samples such as grab samples, channel samples were collected.

Grab samples were collected from outcrops and fragments of altered and/or mineralised rocks during prospecting and mapping traverses in order to determine and evaluate the grades of valuable elements. Sometimes a composite sample was taken with certain intervals from sites where there is extensive alteration and mineralisation. A grab sample weighed 1-2 kg. Each of the samples was placed in a proper bag and its coordinate was recorded. A total of 422 such samples were collected during the program.

9.7 Channel Sampling

A significant channel sampling (trenching) program was carried out in 2010 and 2011 to test geochemical anomalies and the geological environment mainly in the ATO prospect but also in the surrounding Davkhar Tolgoi, Bayan Munkh, Bayan Gol and Duut Nuur occurrences. In some cases, trenching led to new target discoveries. Trenching was concentrated mainly in ATO deposit area from 2012 to 2014. In all 244 trenches were excavated in ATO district and surrounding areas including 168 trenches in ATO prospect. A total of 28,809 m of trenching was done.

Trenching was performed to confirm the results of surface geochemical and grab sampling and the geophysical anomalies that had been surveyed in the area where precious and base metal mineralisation found from previous and year 2014.

When excavating trenches, the black-coloured topsoil was stripped carefully and piled separately on one side of the trench and 1 m away from the edge of the trench, and the materials below topsoil were dumped on the other side of the trench. Depth of the trench varied depending on the thickness of loose sediments present and it averaged of 2 m. Excavation did not reach the hard rock in some of the trenches where cover sediment was more than 6 m deep. Width of the trenches varied from 1 to 1.2 m and the walls were slightly sloped to ensure safety. The trenching was aimed at defining alteration zones and soil profiles and evaluating the geochemical and geophysical anomalies that cover large areas. Length of the trenches varied depending on the purpose of each trench, with a minimum of 15 m and a maximum of 276 m. Surveying of trenches were made using differential GPS records.

After making a description and documentation of a trench, the trench was cleaned, and a channel sample was taken from the walls and floor of the trench using a chisel and a sledge hammer. Mineralized bodies and alteration zones found in trenches were sampled over 1 to 2 m. Unaltered rocks were sampled with intervals up to 5 m. With cross-section measured 10 by 5 cm, the samples weighed 8 to 15 kg. Locations of the samples were recorded using a differential GPS device. A total of 7,689 channel samples were taken.

Documentation of a trench was performed after thorough cleaning of the walls and floor of the trench and it involved mapping at the scale of 1:100 and taking of photos. After that, channel samples and rock chip samples were taken.

After documenting the trench and taking samples, the trench was filled with rock material that was initially taken from the bottom of the trench and covered it with black topsoil material. 100% of the excavated material was put back in the trench and the local authority and local environmental office approved a fact of reclamation.

9.8 Geophysical Surveying

Geophysical data gathered during 2009-2012 was acquired by CGM during its exploration efforts, and included magnetic, gravity, and IP (induced polarization) surveys. Geophysical work included a ground magnetic survey carried out by Monkarotaj, as well as a D-D IP and gravimetric survey completed by Geomaster.

Magnetic surveys: Ground magnetic surveys were conducted at a grid 25 * 100 m over an area of 79.3 km2 south and north of the initial completed grid and also at other prospects in the CGM licensed areas in 2010. Air-magnetic and spectrometry surveys were conducted over ~1,000 km2 (four licenses of CGM). Surveys was carried along profiles 100 m apart – 11,021 km in 2011.

IP dipole-dipole (D-D IP) survey: A D-D IP survey was initially completed across the ATO prospect in the fall of 2009 on recommendations of CGM and then carried on during 2010-2011 using two modifications – 50 and 100 m measurement spacing. D-D IP Sections were placed 100 or 200 m apart over 324.9 km.

Gravity survey: A gravity survey (200 * 200 m, 1,704 stations) was completed in 2010 over the ATO prospect and its vicinity. A detailed 50 * 50 m grid was used in the immediate area of Pipes 1, 2 and 3. In 2011, 3,318 stations were completed.

9.9 Metallurgical Sampling

A minimum of 500 g sample from each of selected diamond drill holes was submitted to the laboratory of Actlabs Asia for Bottle-roll test for gold. When selecting samples, they were sorted with regard to their grades of gold and Pb-Zn, degree of oxidation (oxidized, intermediate, and unoxidized), and the type of host rock, and they were selected for their relative consistency of distribution and ability to represent their respective mineralised bodies (Table 9.1). Sampling was aimed to test the metal recovery of mineralised bodies. A total of 93 metallurgical samples were prepared.

Table 9.1 - Metallurgical Sample Schedule

Grade	Very high			High			Medium			Low		
Degree of oxidation	Oxidized	Transition	Unoxidized	Oxidized	Transition	Unoxidized	Oxidized	Transition	Unoxidized	Oxidized	Transition	Unoxidized
Au	+	+	+	+	+	+	+	+	+	+	+	+
Au-Pb-Zn	+	+	+	+	+	+	+	+	+	+	+	+
Pb-Zn	+	+	+	+	+	+	+	+	+	+	+	+

In addition, a contract was established with the laboratory of Xstrata Process Support Centre to perform metallurgical test, and pursuant to it, initial test samples were sent to the laboratory in April 2011 followed by the second test samples in July 2011. The test work involved using of gravity method to produce low-grade concentrates and then flotation method for separating lead and zinc. The purpose of the test work was to maximize metal recovery from the ore and solving the issue of process plant design. Below are accounts on the two stages of test work.

1. The initial stage of the test work was performed on five sets of samples representing oxidized, intermediate, and unoxidized zones of the upper and lower parts Pipe 1 and 2 (Table 9.2). Eleven drill holes were selected and the samples from them were sorted according to their high, medium or grades of gold and lead-zinc.

Table 9.2 - Sample Sets for Metallurgical Testing

Zone	Oxidation	Drill Hole ID	Intervals (m)	Width (m)	Set No	Weight (kg)	
Upper	Oxidized	ATO-12	2.00	44.30	45.6	ATO - 1	128.8
		ATO-14	2.60	27.30	14.0		
		ATO-15	0.90	13.00	4.0		
	Transition	ATO-07	32.70	54.90	22.2	ATO - 2	127.7
		ATO-11	38.90	75.90	37.0		
		ATO-14	66.95	78.05	11.1		
	Unoxidized	ATO-11	75.90	97.00	21.1	ATO - 3	127.8
		ATO-11	101.40	120.20	18.8		
		ATO-27	74.35	88.35	14.0		
		ATO-28	61.15	74.65	13.5		

Zone	Oxidation	Drill Hole ID	Intervals (m)	Width (m)	Set No	Weight (kg)	
Lower	Unoxidized (Fresh)	ATO-15	108.70	154.25	45.6	ATO - 4	127.3
		ATO-32	140.20	154.20	14.0		
		ATO-38	128.05	132.05	4.0		
		ATO-07	147.40	150.55	3.2	ATO - 5	128.8
		ATO-07	162.30	170.10	7.8		
		ATO-12	125.00	127.00	2.0		
		ATO-12	130.00	136.00	6.0		
		ATO-14	94.10	100.35	6.3		
		ATO-24	124.40	144.40	20.0		
		ATO-34	189.50	209.20	19.7		
						Total	**640.4**

2. The second stage of test work was performed on nine sets of samples. First seven of the nine sets of samples were taken from the following locations, respectively (Table 9.3).
    - Weakly mineralised intervals with the minimum grades of Au and Pb-Zn.
    - Intervals with certain grades but not included in resource blocks.
    - Intervals with extreme gold grades and moderate Pb-Zn grades.
    - Intervals with high gold grades and high to moderate Pb-Zn grades.
    - Intervals with moderate gold grades and extreme Pb-Zn grades.
    - Intervals with weak gold grades and moderate to weak Pb-Zn grades.
    - Intervals with mixed grades of gold and Pb-Zn.

In addition, one of the two remaining sets was chosen in order to evaluate the metal recovery of the newly discovered Pipe 4. The other set was taken with an aim to increase the metal recovery of Pipe 2.

Table 9.3 - Sample Sets for Second Metallurgical Test

	Drill Hole ID	Intervals (m)	Width (m)	Set No	Weight (kg)
Weakly mineralised	ATO-128	70.00	82.00	Set-1	75.98
	ATO-162	50.70	62.60		
Outside of pipes	ATO-55	114.00	122.00	Set-2	74.58
	ATO-160	30.00	45.80		

	Drill Hole ID	Intervals (m)	Width (m)	Set No	Weight (kg)
Mixed-1	ATO-64	115.80	121.80	Set-3	36.00
	ATO-92	108.30	111.30		
Mixed-2	ATO-41	60.10	69.10	Set-4	30.88
Mixed-3	ATO-60	110.95	119.10	Set-5	35.61
Mixed-4	ATO-60	81.05	89.40	Set-6	31.72
Master	ATO-19	85.70	91.70	Set-7	249.95
	ATO-40	50.25	58.40		
	ATO-49	101.00	106.00		
	ATO-71	74.60	84.60		
	ATO-87	172.10	177.30		
	ATO-110	55.30	62.55		
	ATO-135	50.50	69.00		
	ATO-136	160.90	174.90		
Pipe 4	ATO-96	109.80	140.00	Set-8	200.29
	ATO-116	25.00	50.00		
Pipe 2	ATO-20	60.10	88.40	Set-9	168.27
	ATO-55	135.05	145.50		
	ATO-60	119.10	141.80		
			Total	**9.00**	**903.29**

To prepare these samples, all of the half cores from the selected drill holes were retrieved from storage. Each of the sets was packed in a 60 L barrels, and all necessary documents were obtained for customs clearance. The samples were shipped via freight forwarder DHL. The samples weighed a total of 1,543.7 kg.

9.10 Geotechnical Sampling

Geotechnical samples were selected to have consistent distribution in the pipes of the deposit taking into account the types of rocks present, degree of oxidation of pies, and grades of gold and other metals. The samples were prepared in two different forms before submitting them to the Actlabs laboratory (ACTLABS).

- 10 cm long sample accounting for ¼ of the core sample. 117 such samples were tested for bulk density by coating them with paraffin and dipping them in distilled water to measure displaced water. The volume obtained was then compared with density of the distilled water to determine the bulk density of rocks.

- 124 samples each weighing 50 g were taken from remnants of the pulverized samples that had been prepared for assays. The pulverized samples were put into fluids in a standard condition, the volume of displaced water was measured, and it was compared with the density of the distilled water to determine the ultimate relative density.

9.11 Petrographic Analysis Sampling

To study the lithological composition and mineral composition of rocks found in the deposit area, a total of 45 samples were taken from all types of rocks and alterations. Their petrographic descriptions have been made by Professor Bal-Ulzii (PhD) of University of Science and Technology of Mongolia and PhD A.B. Ted Theodore of USGS, an adviser to CGM. Ted Theodore used in his petrographic analyses Zeiss Axioskop 40 Pol microscope with zooms at 2.5X, 5X, 10X, 20X, and 50X and a lens 10X-20 Pol capable of zooming at 500X. The microscope is able to work with reflected and absorbed lights. The microscope was equipped with a digital camera MicroPublisher 3.3 RTV, which was connected to an iMac computer with 2.4 GHz Intel Core 2 Duo processor. Photos taken with this camera was then processed in Adobe Illustrator CS5 and Adobe Photoshop CS5 to jpeg format. The results were presented in the previous sections describing ATO deposit styles and geological settings.

9.12 Mineralogical Sampling

To study the mineral composition of the deposit's mineralisation, possible sequence of concentrating of minerals, and the nature of structure and texture of the mineralisation, a total of 59 samples were taken representing all of mineralised assemblages at the deposit, and they were analysed by senior teacher Myagmarsuren of the University of Science and Technology and Ph.D A.B. Ted Theodore of USGS, an adviser to CGM. Ted Theodore conducted his analyses in Menlo Park of USGS in California. He used a LEO 982 electronic microscope to determine the sequence of crystallization of minerals and their textures. He also determined chemical compositions at some random points in the samples. Necessary photos were taken and have been included in the report in jpeg format and results are presented in previous sections.

9.13 Absolute Age Determination Sampling

In order to determine the ages of extrusive and intrusive rocks mapped in the area, seven samples were taken and sent for analyses by U-Pb dating on zircon crystals to Ph.D. J.K. Mortensen of the Pacific Centre for Isotopic and Geochemical Research of the University of British Columbia.

15 to 20 zircon crystals were picked from each of the samples were analysed by the Laser Ablation ICP-MS (described by Tafti, 2009) using a New Wave UP-213 laser. Zircon crystals that measure no less than 74 microns were selected and mounted in an epoxy puck along with several crystals of internationally accepted standard zircon (Plesovice, FC1) that dates 197 million years and a couple of internal quality control samples, and they were fed into the instrument in a linear form. Crystals

selected were washed by reduced nitric acid for about 10 minutes and then rinsed in distilled water to make them high quality crystals with no alterations and no prints. Laser beam level was taken at 45% to allow ablation crater size to be 15 microns. Laser beam was off for 10 seconds and of for 35 seconds. Data collected was reduced in GLITTER software. Biases were corrected using the Plesovice standard. Adjustment of the instrument was controlled by zircons of the internal quality control. The analytical regime included 4 measurements of Plesovice's standard zircon, two measurements of internal quality control, and 5 measurements of prepared zircons. Interpretation of the processed results was performed using ISOPLOT software of Ludwig.

9.14 Paleontological Sampling

Six (6) samples were taken from the paleontological faunal and floral relics found in the exploration area and they were analysed to determine stratigraphic ages. The task was performed by PhD Minjin of the University of Science and Technology of Mongolia, who is a member of the Stratigraphic Commission of Mongolia.

9.15 Ground-Water Exploration

A comprehensive preliminary hydrogeological exploration program was successfully completed across the Davkhar basin at ATO prospect area through a 2011 and 2012 work program to support a C2 groundwater reserve estimate under Mongolian classification systems. The exploration program comprised surface geophysics (VES and MRS), core exploration bores completed as observation bores and test bores for aquifer testing.

9.16 Surface Exploration Results and Interpretation

9.16.1 Soil Geochemical Result Analysis

Results of the soil sampling at the ATO license area were produced to analyse dispersion halos for elements of Au, Ag, Pb, and Zn. Analysis was attempted to determine the vertical and horizontal zones of elements associated geochemically to the ATO deposit and figure out the geochemical features of the deposit.

In order to determine the horizontal zonation of the deposit, 959 samples were analysed (analysed for a suite of 49 elements by ICP in the Stuart Global Laboratory) that belong only to the ATO deposit area. In addition, certain representative drill holes were selected at each of the pipes and their samples were analysed for a suite of 45 elements by IPC in the ACTLABS laboratory. A total of 2,651 core samples from a total of 19 drill holes were included in the study.

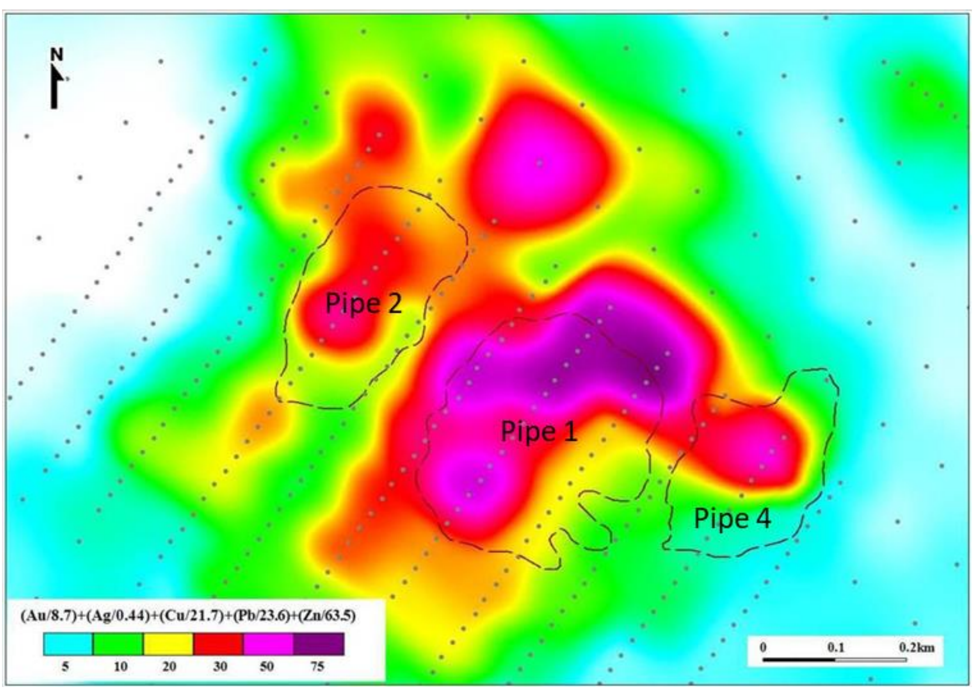
Elemental associations differ with verticality and horizontality for the mineralised pipes of the ATO deposit and their grades vary.

Horizontal geochemical zonation of the deposit: At the ATO deposit elements such as Au, Ag, Pb and Zn have spatially coincident anomalous grades. Therefore, anomalous grades of these elements were combined to make an integrated dispersion halo map, shown in Figure 9.2. The background grades of elements are 8.7 ppb Au, 0.44 ppm Ag, 21.7 ppm Cu, 23.6 ppm Pb, and 63.5 ppm Zn.

A geochemical anomalous halo of combined elements was outlined with a size of 750 * 900 m, slightly elongated to the north. The pipe-shaped mineralisation is coincident with and clearly distinguished by the contour of combined value of 30 (red shading). This contour is generally irregular in shape. Pipe 1 has the highest concentration of metals in soil whereas Pipe 2 has low intensity. Anomalous contours have gradual weakening to the southwest and south of Pipe 1, which reflects the transportation of elements along the features of relief. Very high intensity is attained over a small distance at the northeastern part of the pipe.

The degree of correlation between any two of important elements at the ATO deposit can be seen for each of the pipes in the following Tables.

Figure 9.2 - Soil Combined Element Halo Map Over ATO Pipes



Source: ATO Technical Report, 2017

Table 9.4 - Element Correlation in Pipe 1 Soil Geochemical Halo

Correlation	Au	Ag	As	Cu	Mn	Mo	Pb	Sb	Zn
Au	1.00	0.66	0.53	0.44	-0.40	0.14	0.46	0.52	-0.04
Ag	0.68	1.00	0.56	0.60	-0.41	0.44	0.59	0.54	0.01
As	0.53	0.56	1.00	0.85	-0.33	0.56	0.76	0.82	0.30
Cu	0.43	0.60	0.85	1.00	-0.44	0.48	0.92	0.64	0.15
Mn	-0.40	-0.41	-0.33	-0.44	1.00	-0.33	-0.30	-0.31	0.12
Mo	0.14	0.44	0.56	0.48	-0.33	1.00	0.37	0.49	0.23
Pb	0.46	0.59	0.76	0.92	-0.30	0.37	1.00	0.54	0.06
Sb	0.52	0.54	0.82	0.64	-0.31	0.49	0.54	1.00	0.06
Zn	-0.04	-0.01	0.30	0.15	0.12	0.23	0.06	0.06	1.00

Based on analyses of 62 samples

Table 9.5 - Element Correlation in Pipe 2 Soil Geochemical Halo

Correlation	Au	Ag	As	Cu	Mn	Mo	Pb	Sb	Zn
Au	1.00	0.63	0.46	0.76	-0.14	0.07	0.73	-0.14	0.08
Ag	0.63	1.00	0.59	0.50	-0.09	0.38	0.51	-0.02	0.12
As	0.46	0.59	1.00	0.39	-0.06	0.60	0.33	0.34	0.35
Cu	0.76	0.50	0.39	1.00	-0.16	-0.01	0.64	-0.08	0.33
Mn	-0.14	0.09	0.06	-0.16	1.00	-0.04	0.09	0.19	0.54
Mo	0.07	0.38	0.60	-0.01	0.04	1.00	0.05	0.48	0.07
Pb	0.73	0.51	0.33	0.64	0.09	0.05	1.00	-0.06	0.31
Sb	-0.14	0.02	0.34	-0.08	0.19	0.48	0.06	1.00	0.09
Zn	0.08	0.12	0.35	0.33	0.54	0.07	0.31	0.09	1.00

Based on analyses of 46 samples

Table 9.6 - Element correlation in Pipe 4 Soil Geochemical Halo

Correlation	Au	Ag	As	Cu	Mn	Mo	Pb	Sb	Zn
Au	1.00	0.59	0.88	0.81	-0.42	0.68	0.86	0.61	0.71
Ag	0.59	1.00	0.79	0.82	-0.08	0.84	0.72	0.89	0.89
As	0.88	0.79	1.00	0.97	-0.19	0.88	0.98	0.81	0.89
Cu	0.81	0.82	0.97	1.00	-0.08	0.94	0.97	0.88	0.95
Mn	-0.42	-0.08	-0.19	-0.08	1.00	0.11	-0.20	0.05	0.00
Mo	0.68	0.84	0.88	0.94	0.11	1.00	0.84	0.90	0.96

Correlation	Au	Ag	As	Cu	Mn	Mo	Pb	Sb	Zn
Pb	0.86	0.72	0.98	0.97	-0.20	0.84	1.00	0.77	0.86
Sb	0.61	0.89	0.81	0.88	0.05	0.90	0.77	1.00	0.95
Zn	0.71	0.89	0.89	0.95	0.00	0.96	0.86	0.95	1.00
Based on analyses of 25 samples									

Gold in Pipe 1 has moderate correlations with silver, arsenic, antimony, and lead, and a weak correlation with copper, which in turn has a very good correlation with lead. At Pipe 2, gold gives moderate correlations to copper, lead, silver, and arsenic. At Pipe 4, it has high correlations with lead, zinc, and copper, and moderate correlations with arsenic, antimony, and molybdenum.

Vertical geochemical zonation of the deposit: These results are presented in Section 10 with the drilling results.

9.16.2 Channel Sampling Results

A selection of significant results from the channel sampling programs is given in Table 9.7.

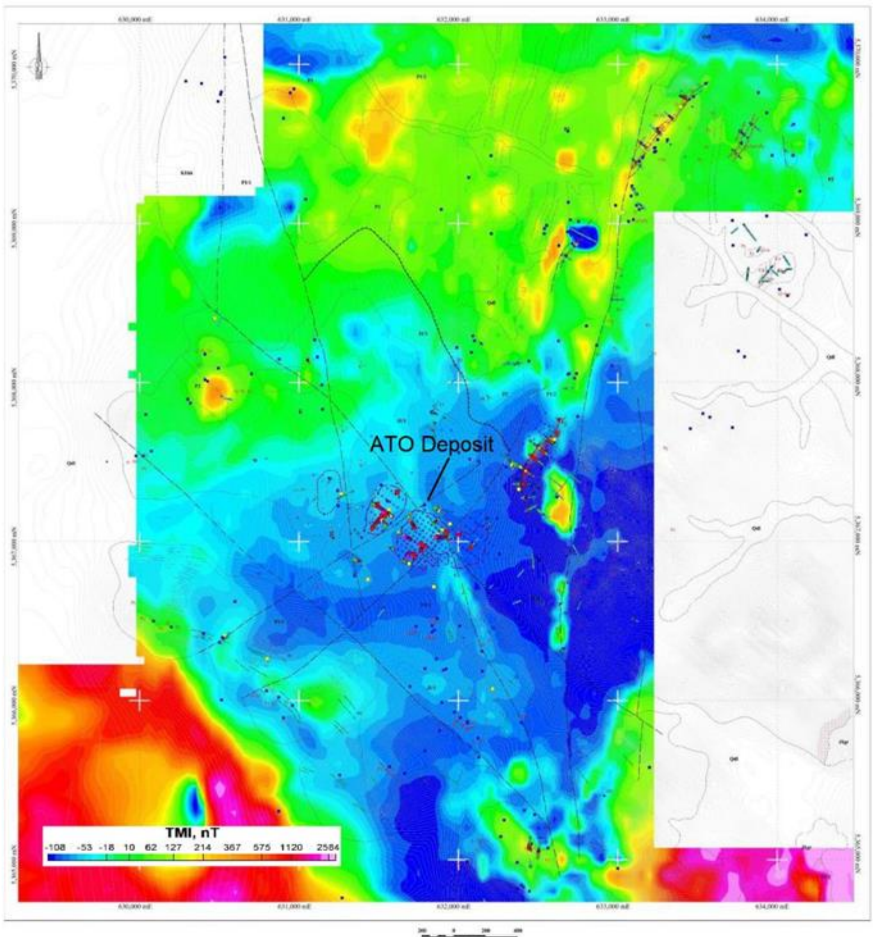
Table 9.7 - Channel Sample Significant Results

Trench ID	Length (m)	From (m)	To (m)	Lithology	Au ppm	Ag ppm	As ppm	Cu ppm	Pb ppm	Zn ppm
ATO-TR-117	2.2	2.8	5.0	Tuff dacite	0.13	<	201	11	15	89
	1.2	19.5	20.7	Tuff dacite	0.10	<	124	12	43	59
ATO-TR-238	1.7	70.6	72.3	Fault	0.38	6.0	3980	184	583	12400
ATO-TR-239	0.5	19.6	20.1	Andesite	1.99	<	8420	256	6	97
ATO-TR-240	1.0	19.5	20.5	Andesite	3.16	<	459	332	10	127
	1.0	21.5	22.5	Andesite	1.97	<	423	86	6	79
	1.2	22.5	23.7	Andesite	1.98	<	1610	218	8	91
ATO-TR-241	1.0	22.5	23.5	Crush zone	0.14	<	685	88	9	258
	1.0	41.0	42.0	Andesite	0.33	<	257	81	6	395
ATO-TR-184	1.0	2.0	1.0	Fracture zone	0.11	2.0	218	18	37	222
	2.0	4.8	2.8	Rhyolite	0.13	<	321	31	43	112
	27.1	28.8	1.7	Rhyolite	0.16	2.0	604	22	27	35
	52.8	55.0	2.2	Siltstone	0.12	2.0	152	48	91	80
ATO-TR-187	6.0	6.7	0.7	Silica	0.24	3.0	160	75	107	126
ATO-TR-218	1.0	3.0	2.0	Siltstone	0.13	<	714	194	29	336

9.16.3 Geophysical Survey Results

Magnetic survey results: Magnetic data were instrumental in identifying major structures, mafic dykes, and ring intrusion-related anomalies. The deposit at ATO is in a broad generally low magnetic area that contains linear, low level positive features that coincide with NW and N striking faults (Figure 9.3).

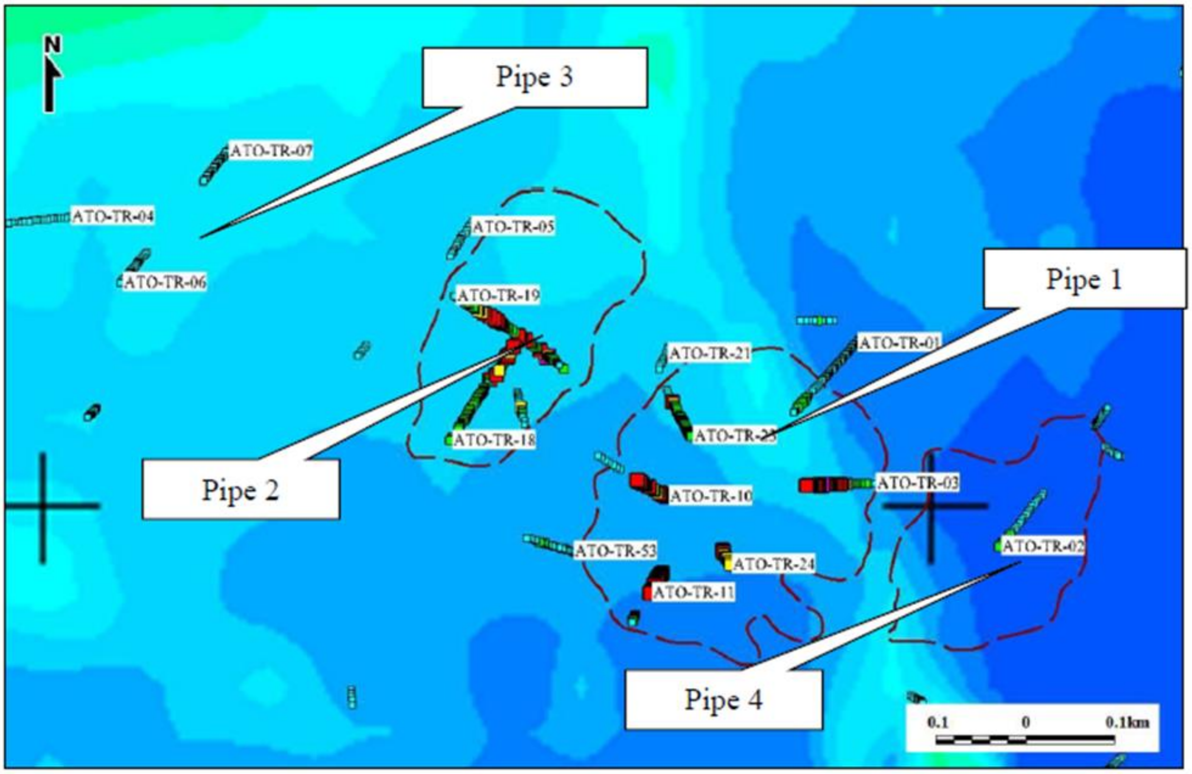
Figure 9.3 - Magnetic Survey Map Over ATO District



Source: ATO Technical Report, 2017

The result map of the magnetic survey shows that the pipes of the ATO deposit are in an area which generally has weak magnetic intensity with magnetic field values slightly higher to the northwest than to the southeast. In another word, Pipe 4 is characterized by very low magnetic values whereas Pipe 2 has relatively higher magnetic response. A strip of higher magnetic values ~100 m wide is observed extending in a northwest direction from the southwest of Pipe 4 to the north of Pipe 1, and this strip spatially coincides with a diorite dyke that is found on the geology map of the area (Figure 9.4).

Figure 9.4 - Magnetic Survey Map Over ATO Deposit

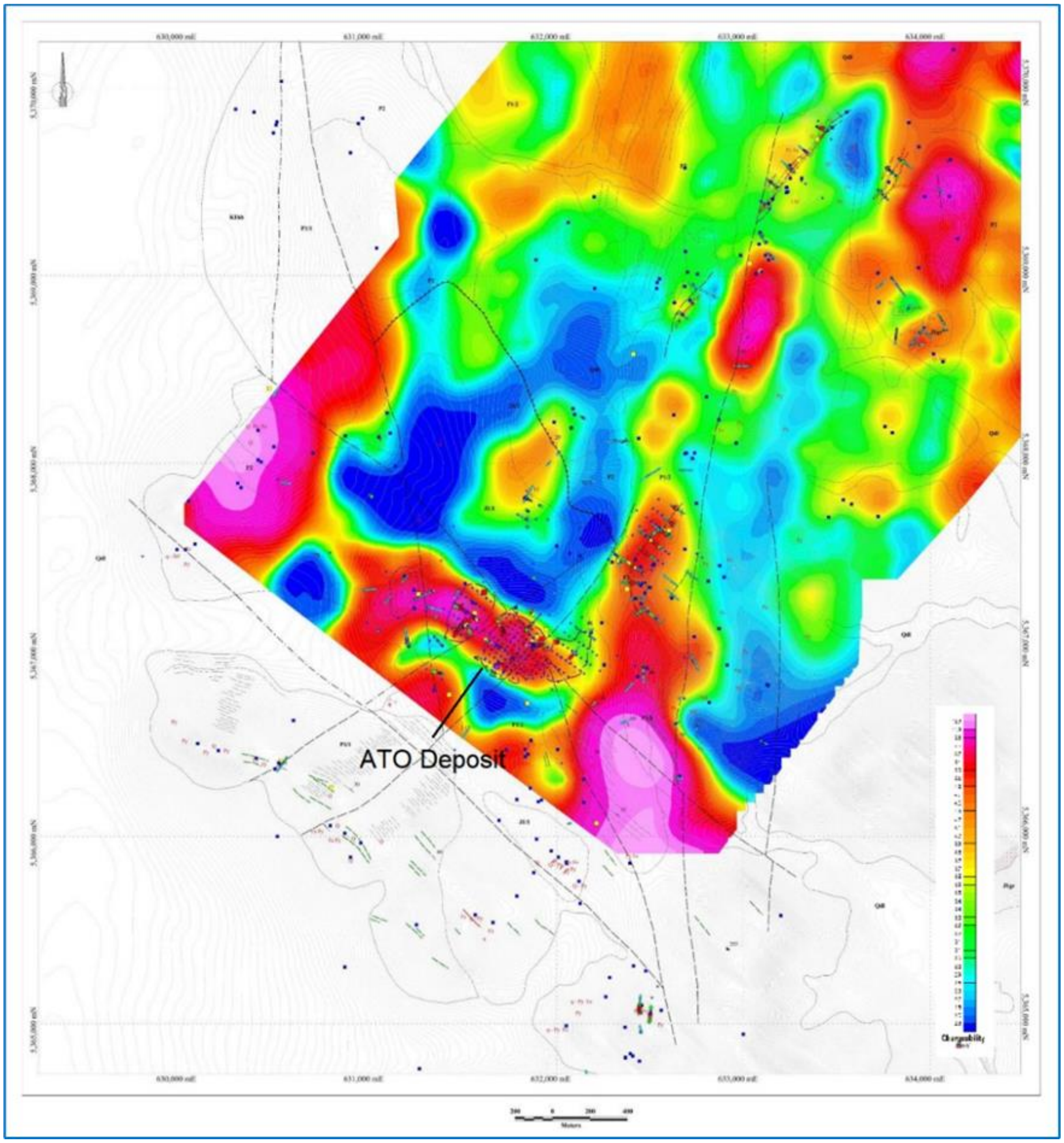


Source: ATO Technical Report, 2017

IP dipole-dipole (D-D IP) survey results: D-D IP surveys play critical roles in detecting silica and clay alteration (respectively high and low resistivity) and sulphide mineralisation (high chargeability), even though a direct 100% relationship is not always present.

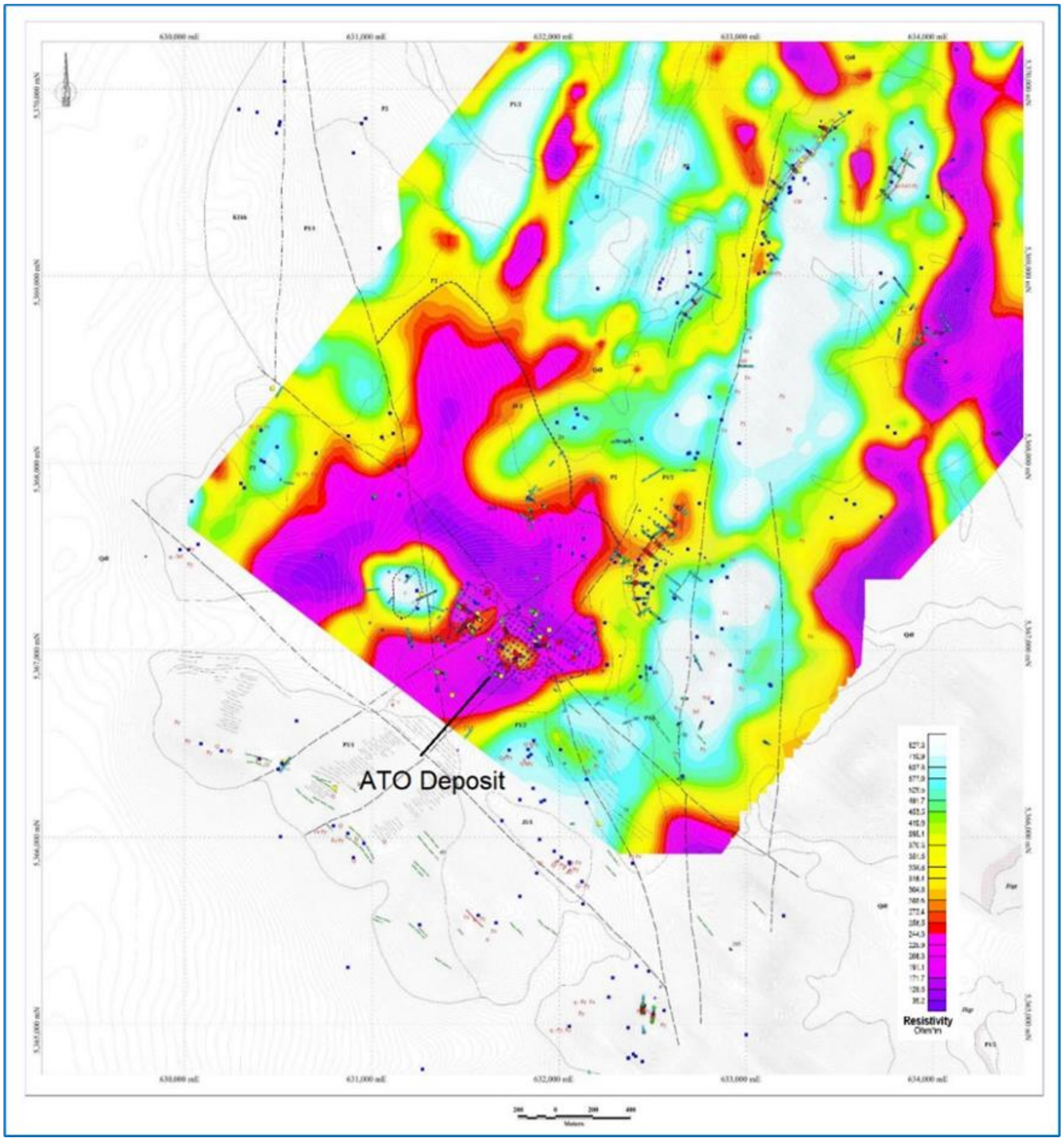
The mineralised pipes at ATO coincide with D-D IP 100-m chargeability and resistivity highs, as shown in Figure 9.5 and Figure 9.6.

Figure 9.5 - D-D IP 100 M Chargeability Survey Map Over ATO District



Source: ATO Technical Report, 2017

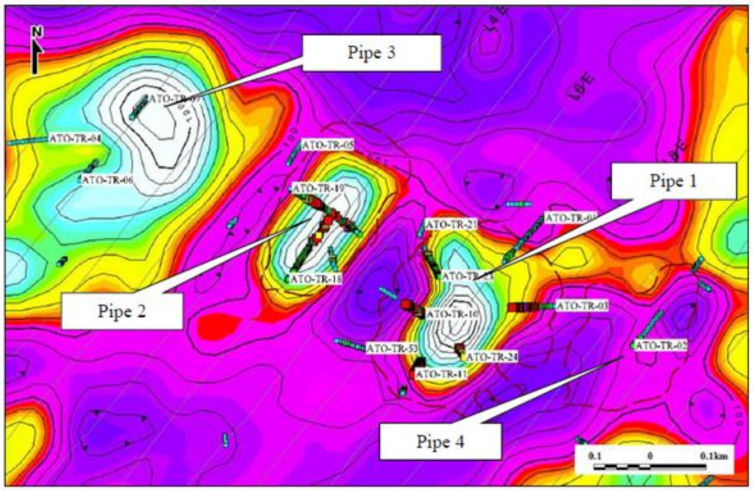
Figure 9.6 - D-D IP 100 M Resistivity Survey Map Over ATO District



Source: ATO Technical Report, 2017

Based on the results in sections, 10 m below the ground surface, several separate areas of very distinct resistivity anomalies were identified and they spatially coincide with a north west-trending series of small hills that are formed of pipe bodies. Therefore, boundaries of pipes and the locations of trenches are plotted on a map of Level N1, where the anomalies are named after their corresponding pipes (Figure 9.7).

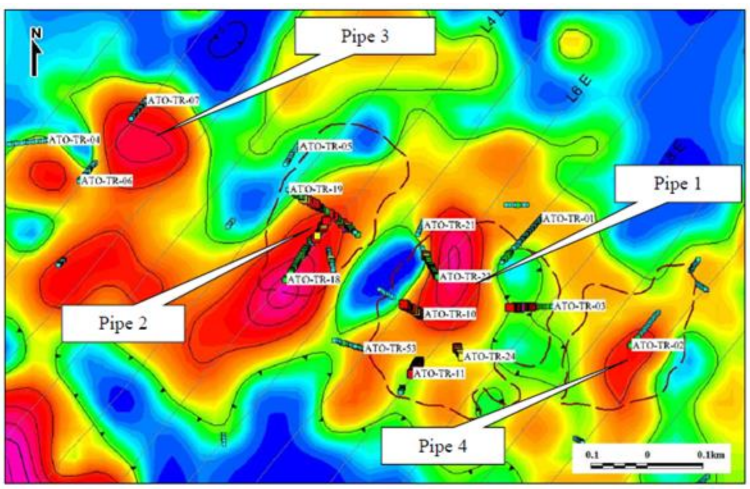
Figure 9.7 - D-D IP Resistivity Map Over ATO Deposit



Source: ATO Technical Report, 2017

The chargeability map (Figure 9.8) shows the pipes not so clearly but as some small anomalous areas of medium to weak responses.

Figure 9.8 - D-D IP Chargeability Map Over ATO Deposit

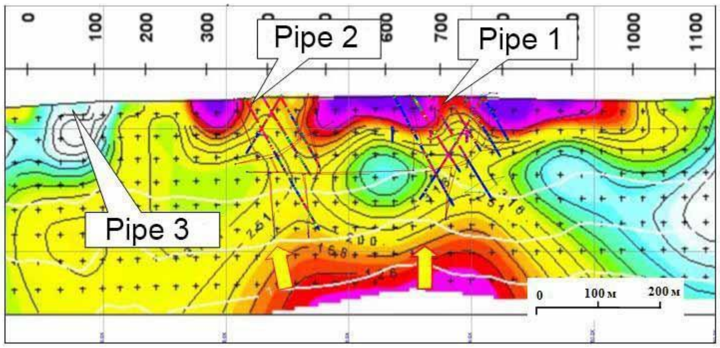


Source: ATO Technical Report, 2017

Pipe 1 area anomaly: This anomalous area is ~220 m long and 140 m wide and its size is smaller than the outline of the pipe. Resistivity values range from 100 to 1000 ohm. It has been observed that the resistivity values decreases with depth. On the chargeability map, there is a small anomaly that coincides with the northern half of the resistivity anomaly of Pipe 1. With sub-longitudinal strike, this chargeability anomaly is 200 m long and 80 m wide.

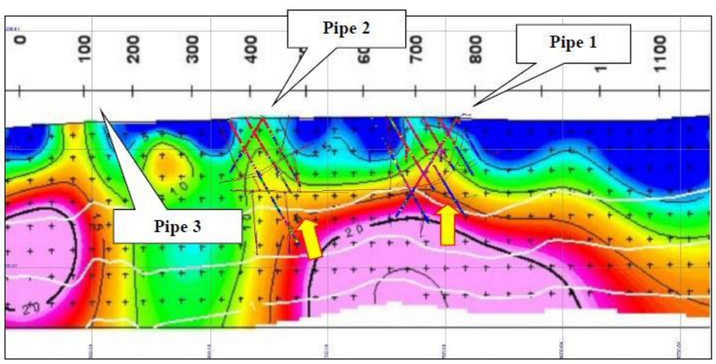
The pipe bodies are well distinguished in the dipole-dipole sections of chargeability and resistivity. They are sourced from the large, high response anomalies identified at depth, have moderate values and show columnar and tunnel shapes (Figure 9.9 and Figure 9.10)

Figure 9.9 - D-D IP Resistivity Cross-Section Through ATO pipes 1, 2, and 3



Source: ATO Technical Report, 2017

Figure 9.10 – D-D IP Chargeability Cross-Section Through ATO Pipes 1, 2, and 3



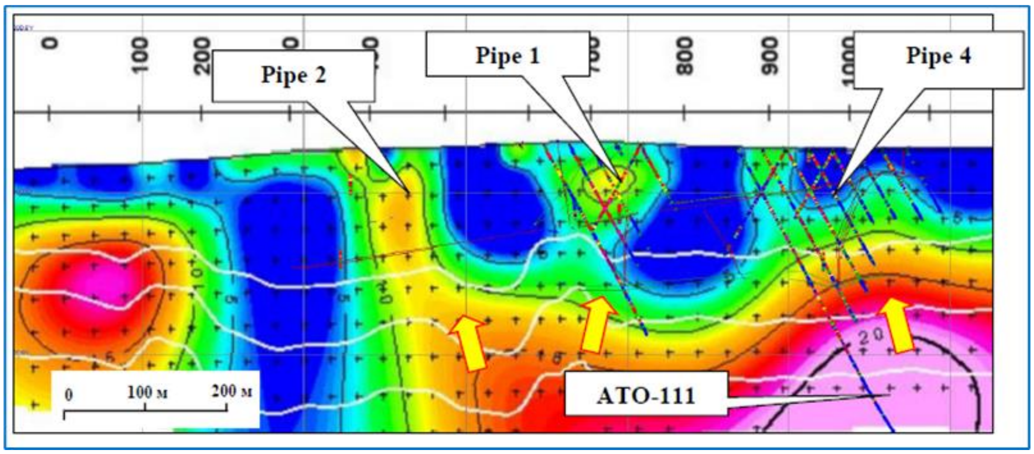
Source: ATO Technical Report, 2017

Pipe 2 area anomaly: Anomalous area measures 280 m long and 130 m wide. Responses of the area are similar to Pipe 1 in terms of values. Chargeability map shows a 350 m long, elongated anomalous area that coincides with the southern part of the resistivity anomaly. They are distinguished clearly in sections (Figure 9.9 and Figure 9.10).

Pipe 3 area anomaly: This is similar to Pipes 1 and 2 and features an area of high resistivity values. Size of the anomalous area is 300 * 200 m, relatively larger than the two pipes discussed above. The chargeability map shows three separate anomalies for this anomalous area, but they join at depth and form one big anomaly in the dipole-dipole chargeability section. However, on the dipole-dipole resistivity section the anomalous values disappear at depth and this is how the anomalous area of Pipe 2 differs from the other two. Barren rhyolite with pyrite alteration intersected by drill holes serves as the explanation of this anomalous area.

Pipe 4 area anomaly: No resistivity anomaly is noticed near the ground surface on the resistivity map but only a small area of weak chargeability anomaly is observed in the chargeability map, attracting no interest at this level. However, an anomalous area similar to those of other pipes is manifested in lines of dipole-dipole sections in this part. It was assumed that it may be an underground mineralisation after making a comparison of it with the geophysical anomalous areas of the other known mineralisation. The small area of weak chargeability anomaly in the dipole-dipole chargeability section has a consistent continuance to depth, is similar to those of Pipes 1 and 2, and is a columnar body sourced from a large anomaly of higher values at depth (Figure 9.11).

Figure 9.11 - D-D IP Chargeability Cross-Section Through ATO Pipes 1, 2, and 4

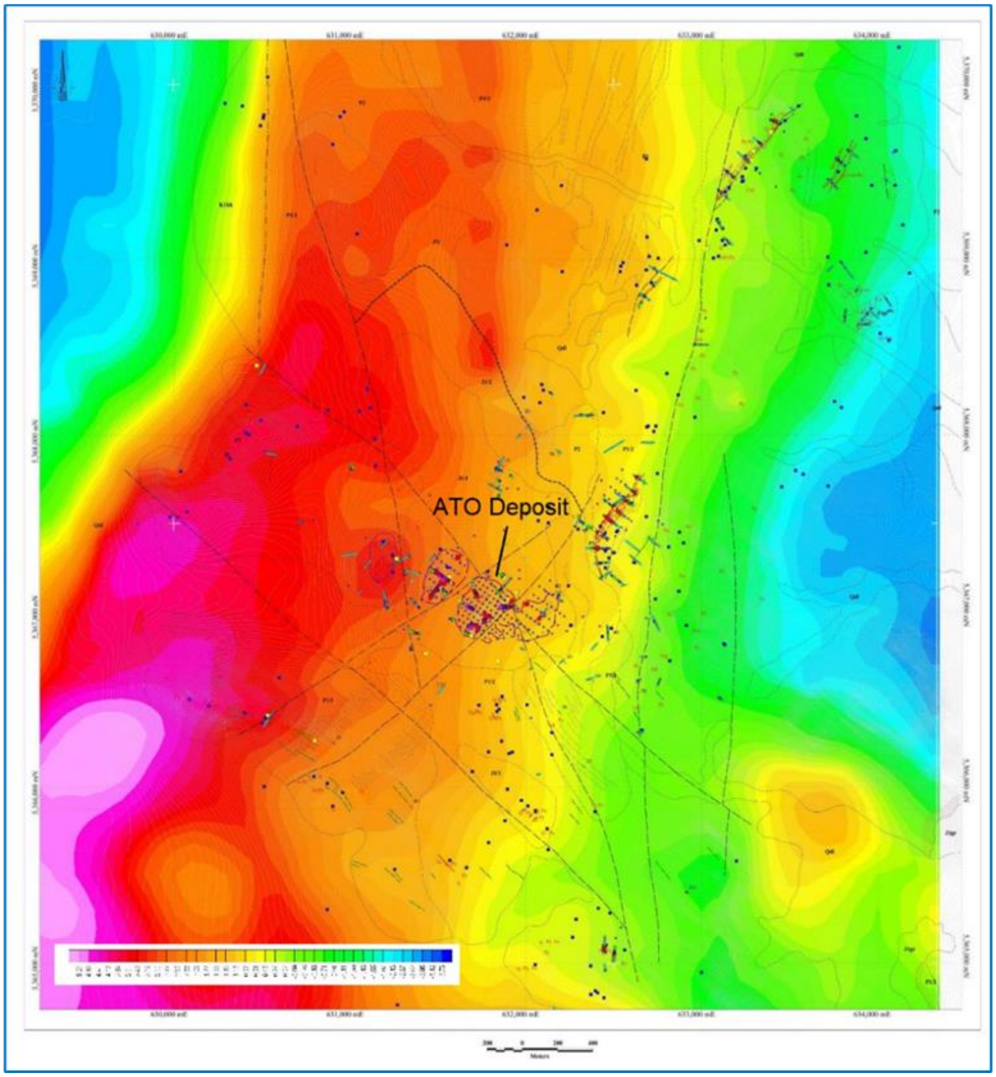


Source: ATO Technical Report, 2017

Drill hole ATO-111 was drilled to a depth of 500 m in order to check the more extensive and higher geophysical values but it did not yield any mineralisation. Therefore, the columnar resistivity and chargeability anomalies of weak to moderate values sourced from the larger anomaly of higher values at more depth serve as a signature for pipe-shaped mineralisation.

Gravity survey results: The gravity survey of the ATO deposit and its surrounding area shows that the deposit is on the flank of a broad NNE-trending gravity high (Figure 9.12).

Figure 9.12 - Gravity Survey Map Over ATO District

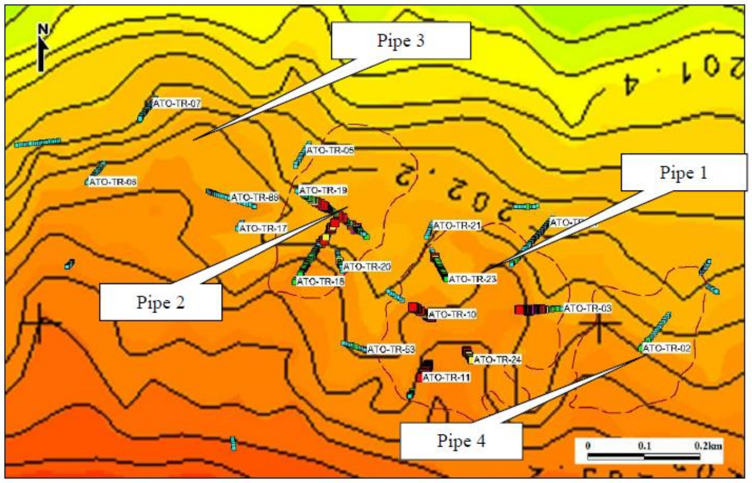


Source: ATO Technical Report, 2017

As seen from the gravity map the pipes are located within the limit of values of 202.2-203.2 mgal that extend sub-latitudinally. The gravity contours are bent and folded around the ATO deposit itself and it was interesting that the mineralised pipes are located in these folds (Figure 9.13).

Pipe 1 sits 100 to 150 m wide and is at the center of an area of relatively high gravity response intruding from south to north while Pipe 2 locates at the northern end of a 100 m wide area of relatively high gravity response that slightly pushes from southwest to north east. Pipe 4 is at the eastern part of an area of relatively high gravity response protruding from south to north.

Figure 9.13 - Gravity Survey Map Over ATO Deposit



Source: ATO Technical Report, 2017

9.16.4 GROUND-WATER EXPLORATION RESULTS

Exploration works have identified two potential groundwater flow systems extending over an area up to 140 km² that can be characterised as:

- Partial coverage of a shallow system extending from surface to between 0 (absent) to 100 mbgl comprising unconsolidated Quaternary and/or weathered Cretaceous sediments.
- A deeper system extending from 0 m to over 250 mbgl comprising semi consolidated Cretaceous sediments.

No continuous low permeability strata have been identified separating the deep and shallow systems across the basin and aquifer testing has demonstrated hydraulic continuity across the systems indicating a leaky or unconfined aquifer system prevails across the exploration area. As such it is assumed the groundwater systems will act as one system through long term supply development. A total saturated sedimentary sequence of at least 250 m has been identified.

Aquifer test interpretation indicates variable transmissivity and aquifer responses from the two test bores with higher transmissivity in the north (>150 m²/d) and lower transmissivity conditions (50 m²/d) in the south. An aquifer storage range of 2.0E-04 – 1.5E-03 has been established from the preliminary aquifer testing. Local barrier boundary conditions are apparent in the south and vertical leakage effects were observed in the north. Individual bore yields of 16 and 18 l/s with pumping bore drawdown limited to 37 and 39 m support the supply potential of the basin. Insufficient data currently exists to firmly characterize permeability distribution which will require assessment as part of future work programs.

Groundwater quality analysis has established various water types across the area which can be generally grouped into ion exchanged waters and areas subject to increased mixing and recharge influences. Groundwater quality is generally brackish across the exploration area; this water should be suitable for industrial purposes, but treatment would be required should drinking water use be required.

A numerical groundwater model has been developed to support the groundwater reserve estimate. The model was developed to simulate a conceptual bore-field in the southern basin area in proximity to the mine site operating over a 20-year period assuming conservative aquifer parameters. Model predictions indicate a maximum drawdown of 113 m (45% of the initial model saturated aquifer thickness) under assumed worst case reduced hydraulic conductivity and aquifer storage conditions.

A C2 reserve estimate of 417 l/s is presented based on traditional analytical methodology and supported by a preliminary groundwater numerical model with adoption of conservative parameters for each of aquifer area (140 km^2), saturated aquifer thickness (150 m) and specific yield (2.5%).

10 DRILLING

The information in this section is largely drawn and/or summarised from the Report available on SEDAR entitled: "Altan Tsagaan Ovoo Project (ATO), 2021 Mineral Resources, Technical Report (Amended NI 43-101)", prepared by GeoRes, with an effective date March 30, 2021, issued on March 30, 2021 and amended on June 9, 2021. Further details as documented therein remain correct and valid.

Drilling and sampling at ATO are described in terms of:

- Drilling details – the incremental programs and drilling contractors.
- Drill holes – the numbers, types and lengths of holes drilled. Trenches were treated as pseudo drill holes.
- Collar and down-hole surveying – methods.
- Locations – of drill holes in plan and on cross-section line.
- Spacing and orientation of drill holes – most commonly on the section lines at 30 * 30 m spacing and dipping 60° towards 125°.
- Sampling – method of cutting and sampling the diamond drill core.
- Drill core recovery – good average 97%.
- Geological modelling – methods.
- Sample intervals for assaying – continuous over full hole length, and typically 1 m long in mineralisation and 2 to 3 m in the remainder.
- Sample attitude to mineralisation – effectively across strike and as close to across true width as practicable.
- Anomalous mineralisation – effectively none due to the style of the deposit.
- Factors that could impact accuracy – effectively none.
- Summary – the Author QP's opinion of the drilling (albeit without the benefit of a site visit to observe it) was that it was well performed and very adequate for the task of interpretation and Resource estimation.

10.1 Drilling Details

10.1.1 Drilling up to 2017

Programs: An exploration drilling program was completed by CGM between 2010 to 2014. Diamond core drill holes (DDH) were the principal source of geological and grade data for ATO. Some reverse circulation (RC) drilling was completed between 2012 and 2014 through cover to map bedrock geochemical patterns as a method of exploring for blind ATO-style mineralisation on the project area. CGM carried out a hydrogeology and geotechnical drilling program in 2011.

Drilling contractors: Diamond and RC drilling on the project were done by Falcon Drilling Mongolia, based out of Canada, using a BBS-56 rig. Core diameter was HQ-size (63.5 mm nominal core diameter).

10.1.2 DRILLING 2017 TO 2021

Programs:

- Commencing in 2018, the Company (Steppe) commenced a three-phase exploration drill program:
  - Phase 1 exploration program focussed on the ATO4, Mungu, Tsagaan Temeet, Bayanmunkh and Bayangol targets at the ATO Project. A total of 66 holes were drilled at the Mungu Deposit, 3 holes at the ATO4 Deposit, 4 drill holes at the Tsagaan Temeet prospect, 1 drill hole at the Bayanmunkh prospect and 16 shallow drill holes at the Bayangol prospect for a total of 8,821 m. The drilling program was successful in outlining and extending known gold and silver mineralisation. In addition, new high grade zones in deeper parts of the deposit were discovered.
  - Phase 2 drilling program focused on the ATO4 end of the ATO4-Mungu trend at the ATO Project and at the Uudam Khundii Project. The drilling program was completed with three diamond core drilling rigs completing a total of 36 drill holes for 9,006 m. The completion of the Phase 2 drilling program saw the identification of the first ever visible gold seen at ATO project, with super high gold grades being returned in ATO299 and ATO317.
  - Phase 3 drilling program targeted at the ATO4-Mungu trend has commenced with 8 drill holes being complete for 2,228 m of drilling[9].
- In 2019, the Company completed a drilling program with two diamond core drilling rigs focused on updating resources and reserves for the ATO1, ATO2 and ATO4 deposits in addition to a maiden Resource and Reserve delineation for the Mungu Discovery. The Company drilled 1,840 m at ATO1, 1,662 m at ATO2, 14,760 m at ATO4 and over 26,573 m at the Mungu Discovery[10].
- Commencing 2020, the Company drilled an additional 55 drill holes for a total of 18,200 m. A total of 53,000 m has been drilled since 2018. The drilling information was used to update the interpretation of the geologic model, geometry of the mineralised zones and domains[11].

[9] 2019 AIF and 2018 news release.
[10] 2019 AIF and 29 July 2020 news release.
[11] 21 February 2021 news release. Information adopted from.

10.2 Drill Holes

10.2.1 Holes up to 2017

All holes: At the end of the 2014, a total of 597 drill holes for ~63,866 m had been drilled over the whole Project area (Table 10.1). Of these 54,425 m was core drilling in 370 holes and 9,441 m was in 227 RC holes.

Table 10.1[12] - Exploration Drill Hole Summary – to 2017

Exploration Drilling Program		Core Holes	RC Holes	TOTAL
2010	Number	62		62
	Length (m)	11,606		11,606
2011	Number	141		141
	Length (m)	24,874		24,874
2012	Number	52	90	142
	Length (m)	10,444	2,259	12,703
2013	Number	7	137	144
	Length (m)	1,539	7,182	8,721
2014	Number	108		108
	Length (m)	5,962		5,962
TOTAL	Number	**370**	**227**	**597**
	Length (m)	**54,425**	**9,441**	**63,866**

Deposit holes: The 2017 drill hole data base for the deposit Resource estimation contained 265 diamond drill holes[13]. In the Pipe 1, 2 and 4 deposit area there were in 238 diamond drill holes for a total of 44,284.2 m. That data included 32,791 assays.

10.2.2 Holes up to 2021

All holes: At the end of 2020 the Author's database contained 767 drill holes for 120,320.3 m. These were over the wider Project area as well as over the deposits. A break-down of these is given in Table 10.2. Older hole names, and those principally over the southern Pipe 1, 2 and 4 deposits, carry the prefix "AT". New hole names over the Mungu deposit carry the prefix "MG".

[12] 2017 NI 43-101. Section 10.1. Table 10.1, pp 74.
[13] 2017 NI 43-101. Section 14.1.1, pp118.

Table 10.2 - Exploration Drill Hole Summary – to 2021

Holes	Number	Length (m)	Avg Length (m)
AT diamond	385	73,133.8	190.0
AT RC	227	9,441.0	41.6
MG diamond	155	37,745.5	243.5
All holes	**767**	**120,320.3**	**158.4**

This newer diamond core drilling total was 540 holes for 110,879.3 m and increase of 170 holes for 56,036 m. These holes were predominantly from the Mungu deposit, and although the number of diamond holes added was ~50% of before the increase in metres was ~100% (as holes at Mungu were considerably longer).

Holes and trenches: Drill holes were augmented by a considerable number of channel samples taken from trenches. The trenches represent pseudo-holes. Trench names also carried the prefix "AT" (similar to the older holes) but with the addition of "TR". A summary of the Project's holes and trenches is given in Table 10.3.

Table 10.3 - Exploration Drill Hole & Trench Summary - to 2021

Holes	Number	Length (m)	Avg Length (m)
All holes	767	120,320.3	156.9
Trenches	167	10,184.3	61.0
All	**934**	**130,504.5**	

10.3 Collar and down-Hole Surveying

Proposed drill hole collars were surveyed by a hand-held GPS unit for preliminary interpretations.

Completed drill holes had PVC pipes inserted and the hole collar was marked by a cement block inscribed with the drill hole number (e.g., ATO-99). A differential GPS was used for a final survey pickup.

The two collar readings were compared, and if any significant differences were noted the collar was re-surveyed; otherwise, the final survey was adopted as the final collar reading.

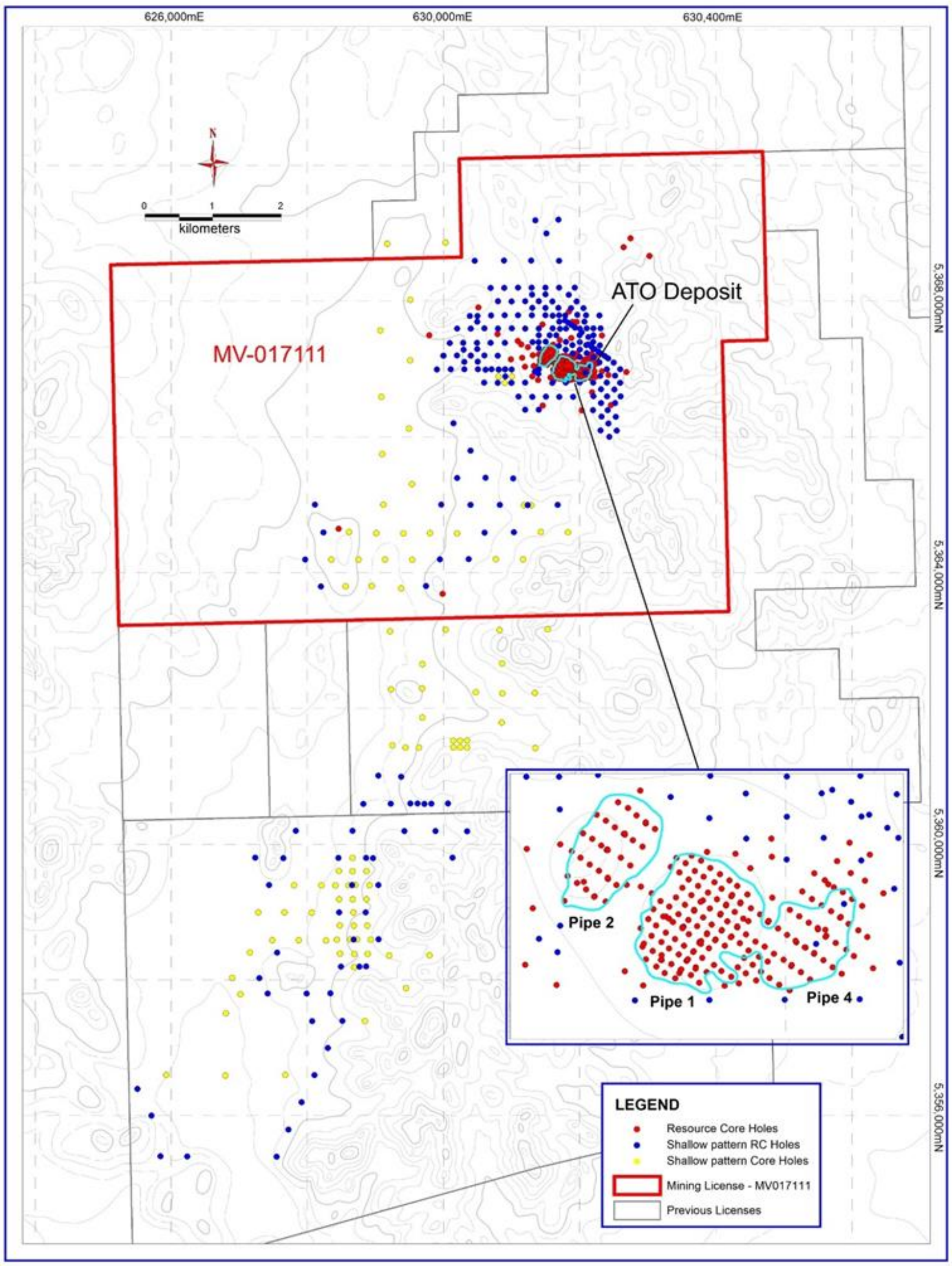
CGM used down-hole survey instrument, Reflex Instrument AB, to collect the azimuth and inclination at each 50 m depth increment in most of the diamond drill holes.

A Reflex ACT II Rapid Descent tool was used for core orientation.

10.4 Drill Hole Locations

10.4.1 LOCATION OF HOLES UP TO 2017

Figure 10.1 - Drill Holes Locations – Holes to 2017[14]



14 2017 NI 43-101. Section 10.2, Fig 10.1, pp75.

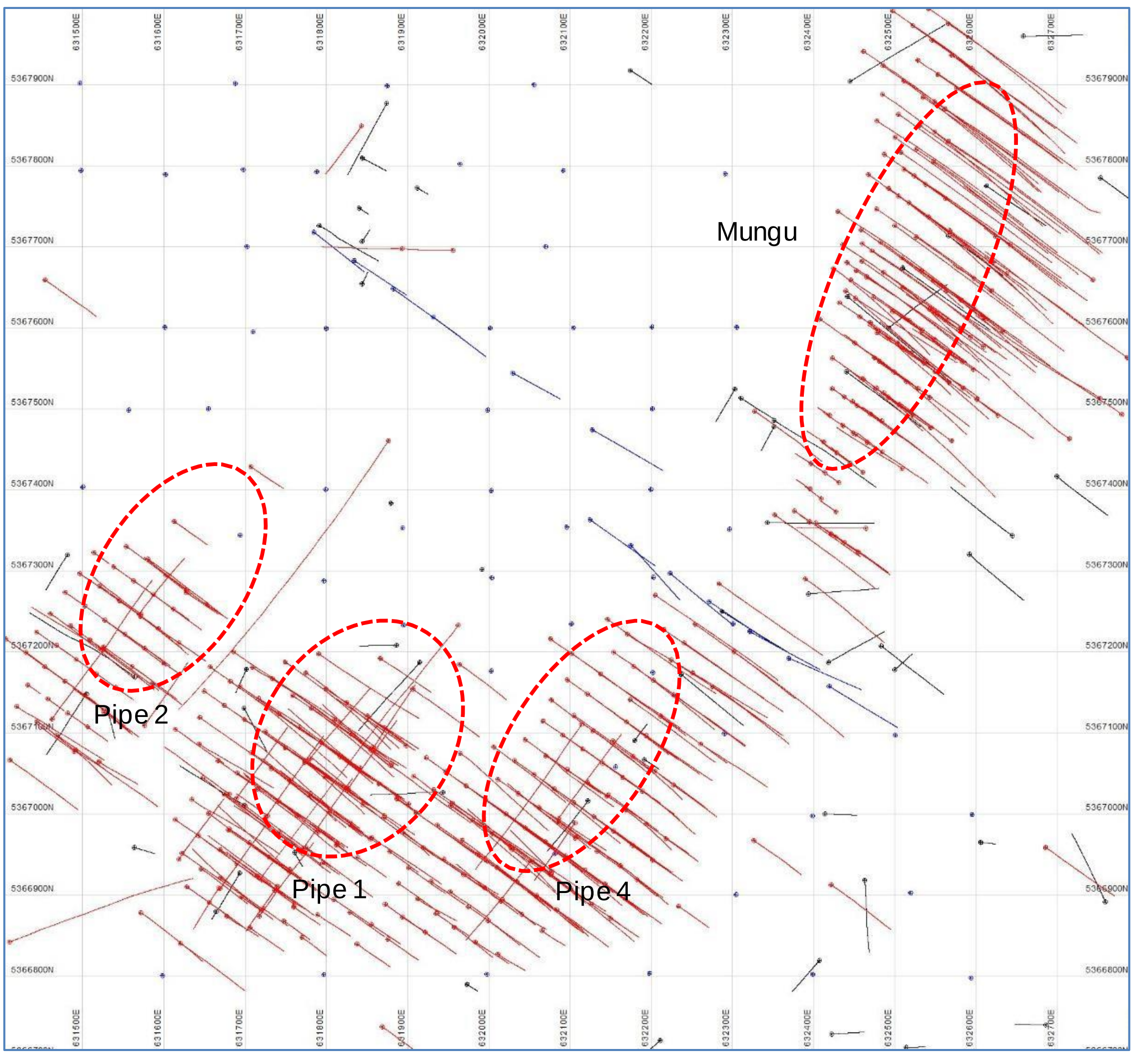
The drilling prior to 2017 had been spread over the ATO mining license as well as a south exploration area (Figure 10.1).

Drilling at the ATO deposit was conducted along 18 cross-section lines oriented WNW to ESE (125°) traversing Pipes 2, 1, and 4 respectively in order west to east. The sections were spaced 30 m apart.

10.4.2 Location of Holes to 2021

Diamond drilling and trenching in the period 2017 to 2021 was all in the area of the ATO deposits. The diamond drilling was predominantly at the new Mungu deposit in the north (Figure 10.2). In the Figure the older RC holes are shown in blue, the diamond holes in red, and the trenches in black. The four deposits locations are approximately shown with labelled red dashed ovals.

Figure 10.2 - Drill Hole and Trench Locations – Deposit Area 2021



Source: NI 43-101 ATO Gold Project, 2017

Drilling since 2017 by Steppe continued through to Mungu on the 30 m spaced 125° oriented cross-section lines.

10.5 Hole Spacing and Orientation

The older RC drill holes (blue dots in Figure 35, drilling through cover exploring for blind deposits) were drilled vertically and on a square 100 * 100 m pattern. These short holes averaged ~40 m in depth. A 125° oriented line of longer parallel inclined RC holes was also drilled across the centre of the deposit area (and turned out to between the Pipe 1, 2 and 4 deposits and Mungu).

The bulk of diamond core (DDH) holes were located on drilling cross-sections oriented at 125° and 30 m apart. These holes were drilled dipping at 60° below vertical and oriented parallel to the cross-sections on 125° azimuths, with a few also drilled the other way on the sections towards 315°. On section the collars were either 30 or 60 m apart (and typically wider at the edges of the deposits). These hole orientations and spacings are illustrated well in Figure 6. A limited number of diamond holes were also vertical, and a limited number were inclined holes and drilled at random azimuths.

The AT diamond holes drilled at the Pipe 1, 2 and 4 deposits averaged ~190 m in length and the MG holes drilled at Mungu averaged ~240 m in length.

All core holes were down-hole survey to determine hole orientation.

10.6 Core Sampling Method

Core samples were taken from diamond drill holes to examine mineralisation at depth. Before sampling, core sample was placed in wooden boxes in a proper order, geotechnical measurements were made, and core recovery was estimated. After this, geological documentation and descriptions were recorded, and samples were taken with intervals of 1 to 2 m at mineralised zones and 2 to 3 m at unaltered host rocks. The samples were weighed 8-12 kg each.

At the head of the core boxes, notes were put down as to drill hole ID, box number, length of core in the box, and depth intervals in metres. Relevant notes were put down on aluminum plates nailed next to sampled intervals, and the cores in the boxes were then sliced into two by diamond saw. Photo documentation was performed both before and after slicing a sample.

Saw blades were cleaned by working 5 cm deep into a barren rock before slicing a sample. Coolant water was applied in a continuous flow to prevent contamination of the sample during the core slicing process.

After slicing the core, half of every sample was placed in a special plastic bag, which was then tied with cable ties, to be sent to a laboratory. The other half was put back in the box, which was then sealed and sent to Ulaanbaatar for storage.

10.7 Core Sample Recovery (%)

The methodology used for measuring core recovery was standard industry practice. In general, core recoveries averaged 97% for the deposit. In localized areas of faulting and/or fracturing the recoveries decreased; however, this occurred in a very small percentage of the overall mineralised zones.

10.8 Geological Logging

Core logging was done at core shed at the ATO site camp. The geological logging procedure was:

- Quick review.
- Box labelling check: The core boxes are checked to ensure they are appropriately identified with the drill hole number, meters from–to, and box number written with a permanent marker on the front.
- Core re-building: Core was usually rotated to fit the ends of the adjoining broken pieces.
- Core photography.
- Geotechnical logging: Used pre-established codes and logging forms, includes length of core run, recovered, drilled ratio, rock quality designation (RQD), and maximum length, structural data, and oriented core data.
- Geological logging: Logging was completed on a paper logging forms. Thereafter information was entered into MS Excel software, which used standardized templates and validated logging codes that must be filled out prior to log completion. The template included header information, lithology description and lithology code, graphic log, coded mineralisation, and alteration.
- Core cutting: The geologist marks a single, unbiased cutting line along the entire length of the core for further processing.

RC logging involved capture of geological, alteration, and mineralisation data on paper logging forms using samples collected in plastic chip trays.

10.9 Sample Intervals for Assaying

All holes were sampled continuously over their full length.

Sample intervals were predominantly 1.0 m long in mineralised zones. Other interval lengths of 1.5 m, 2.0 m and 3.0 m were generally used in un-altered non-mineralised rock and sporadically.

Drill core was geologically logged on-site before the core was transported to core sheds in Ulaanbaatar for storage, cutting and sampling.

10.10 Sample Attitude to Mineralisation

The drilling of 60° inclined holes on 125° oriented cross-sections was aimed at drilling across the perceived strike of the deposits (and so normal to the mineralisation strike). It was also as close to across true width as practicable (see below also).

In terms of the small scale (say 20 to 50 m) continuity and width of mineralisation (at approximately a NE/SW strike) the Author QP considers the drilling orientation to be roughly normal in all holes. And this would also be so in terms of the general NE/SW elongation of the deposits. This situation would be strongest at Mungu with the more linear lenticular interpretations of the Author QP.

However, the generally massive shape of the mineralisation in the deposits (particularly for the southern Pipe 1, 2 and 4 deposits) meant that drill hole orientation was not particularly important. This should be seen in the context of the deposits generally being much wider than several adjacent holes.

And being massive in shape the drill hole lengths did not measure any “true width” as the dimensions of the deposits would be described from the interpreted shapes.

10.11 Anomalous Mineralisation Intervals

The Author QP would take the view that mineralised zones were fairly repeatable although variably sized and that whilst the quantum of their mineralisation was considerably above background it was still relatively “restrained” in variability. This is demonstrated by the geochemical zonation described in Section 0 below.

No wildly anomalous values are seen, and the grade variability that does exist horizontally would be typical of the style of the deposit where the bulk would be typified by vertical streaming of vein fluids upwards.

10.12 Drilling and Sampling Accuracy Factors

The Author QP considers that errors in collar surveys and down-hole surveying could be a source of inaccuracy but would not have made a material difference for the Resource estimation at Pipes 1, 2, and 4 (trusting particularly that collar errors seem very unlikely given consistent drilling results over a long time). At Mungu down-hole survey errors would have a greater impact as the deposit is taller, thinner and deeper. However, any error would seem mitigated by the fact that virtually all holes were drilled the same direction at the same dip, with potentially the same drift if there was one.

The Author QP would similarly consider that potential sampling inaccuracy would be minimal given that the vast bulk of drilling was diamond core, with minimal recovery loss, and sampling intervals were consistent lengths.

10.13 Summary of Drilling Results and Interpretation

10.13.1 Overall Opinion of Drilling

The Author QP's overall pinion of the drilling, sampling and subsequent assaying (albeit without the benefit of a site visit to observe it) was that it was well performed, comprehensive, consistent (and extensive) and very adequate from a point of view of allowing a straight-forward interpretation of mineralisation at the deposits and of estimation of their Resources.

This opinion was reinforced by the vertical geochemical analysis (Section 0 immediately below) previously performed which effectively indicated good data.

10.13.2 Vertical Geochemical Zonation at ATO

In order to investigate the deposit to depth, some drilling sections were selected to represent the general characteristics and features of the deposit and used the analytical data of core samples from the mineralised pipes to establish the general pattern of vertical geochemical zonation by converting the intervals of core samples to elevation levels and comparing the grades of gold and associated polymetallic elements.

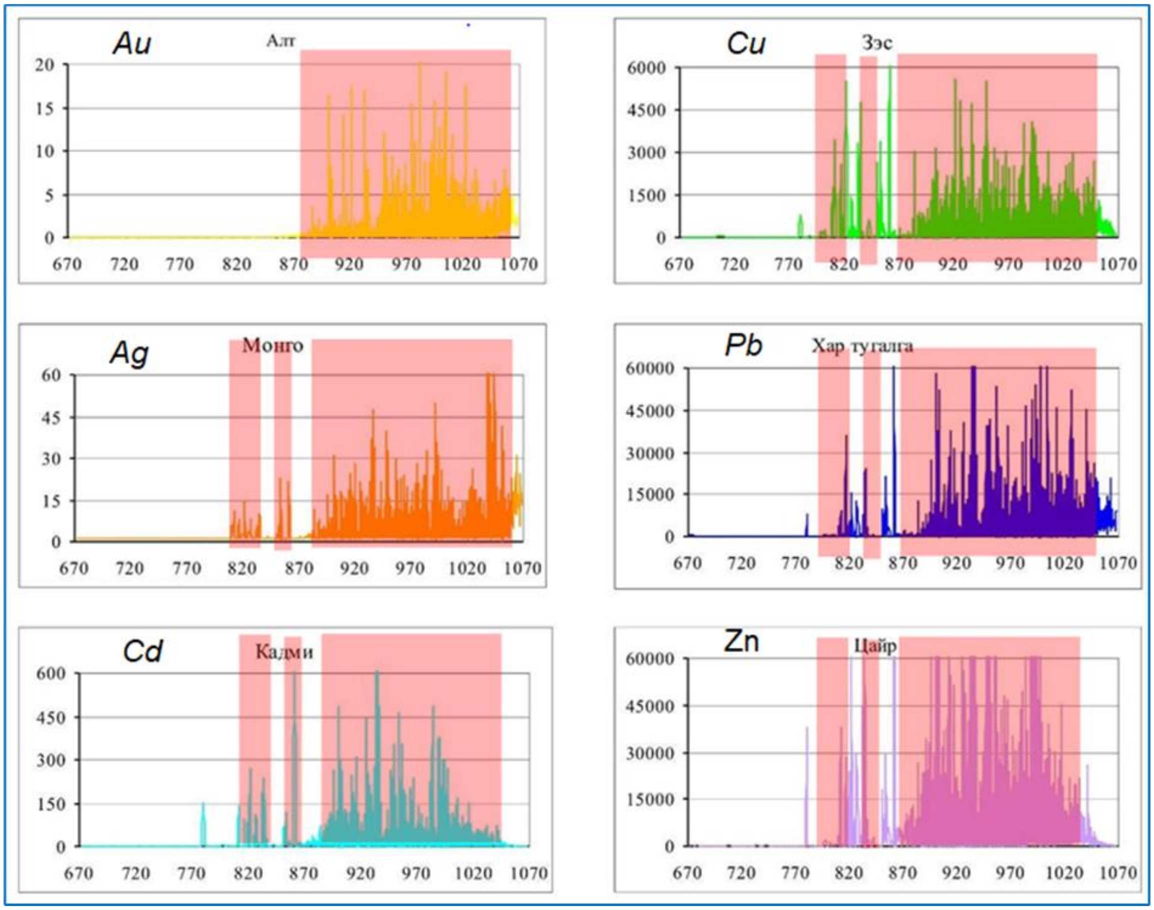
Pipe 1: A total of 1,329 samples from eight drill holes were assayed for 45 elements for Pipe 1. However, the table below shows correlations of only the following elements, which have been selected because of their importance in hydrothermal explosion deposits (Table 10.4). Gold in Pipe 1 has moderate correlations with silver, cadmium, copper, and zinc, and copper has strong correlations with cadmium, lead, and zinc, and moderate correlations with silver and antimony. Lead has strong correlations with cadmium, copper, and zinc, a moderate correlation with silver, and a weak correlation with antimony. Zinc has a very strong correlation with cadmium, strong correlations with copper and lead, a moderate correlation with silver, and a weak correlation with antimony.

Table 10.4 - Pipe 1 Core Sample Element Correlations

Correlation	Au	Ag	As	Cd	Cu	Mo	Pb	Sb	Zn
Au	1.00	0.37	0.16	0.38	0.46	-0.05	0.52	0.05	0.38
Ag	0.37	1.00	0.47	0.49	0.56	0.28	0.57	0.34	0.49
As	0.16	0.47	1.00	0.08	0.23	0.33	0.18	0.46	0.08
Cd	0.38	0.49	0.08	1.00	0.68	0.08	0.78	0.25	0.98
Cu	0.46	0.56	0.23	0.68	1.00	0.04	0.68	0.43	0.70
Mo	-0.05	0.28	0.33	0.08	0.04	1.00	0.02	0.17	0.07
Pb	0.52	0.57	0.18	0.78	0.68	0.02	1.00	0.22	0.77
Sb	0.05	0.34	0.46	0.25	0.43	0.17	0.22	1.00	0.27
Zn	0.38	0.49	0.08	0.98	0.70	0.07	0.77	0.27	1.00

An element correlation with depth analysis (Figure 10.3) shows the gold mineralisation in Pipe 1 continues from the 880 m level to the 1070 m level. As per polymetallic elements, their mineralisation occurs at two separate depth ranges such as from level 810 m to level 860 m and from level 880 m to level 1070 m. The lower range of polymetallic mineralisation has no gold and discontinuous while the mineralisation at upper levels coincides with gold mineralisation and feature discontinuous high grades. Grades of cadmium, zinc, and copper decrease dramatically near surface, or at the 1060 m level, while those of gold, silver, and lead remain relatively stable.

Figure 10.3 - Pipe 1 Element Correlation with Depth



Pipe 2: A total of 374 samples from three drill holes were assayed by multi-element analyses for Pipe 2. The results were then processed and displayed in Table 10.5 showing correlations of gold and polymetallic elements. Gold (Table 10.5) has moderate correlations with cadmium, copper, tin, and zinc, and weak correlations with silver, bismuth, and lead. Copper has strong correlations with silver, cadmium, and zinc, a moderate correlation with antimony, and weak correlations with bismuth and molybdenum. Lead has a very strong correlation with silver, strong correlations with cadmium, copper, antimony, tin, and zinc, and weak correlations with arsenic, bismuth, and molybdenum. Zinc

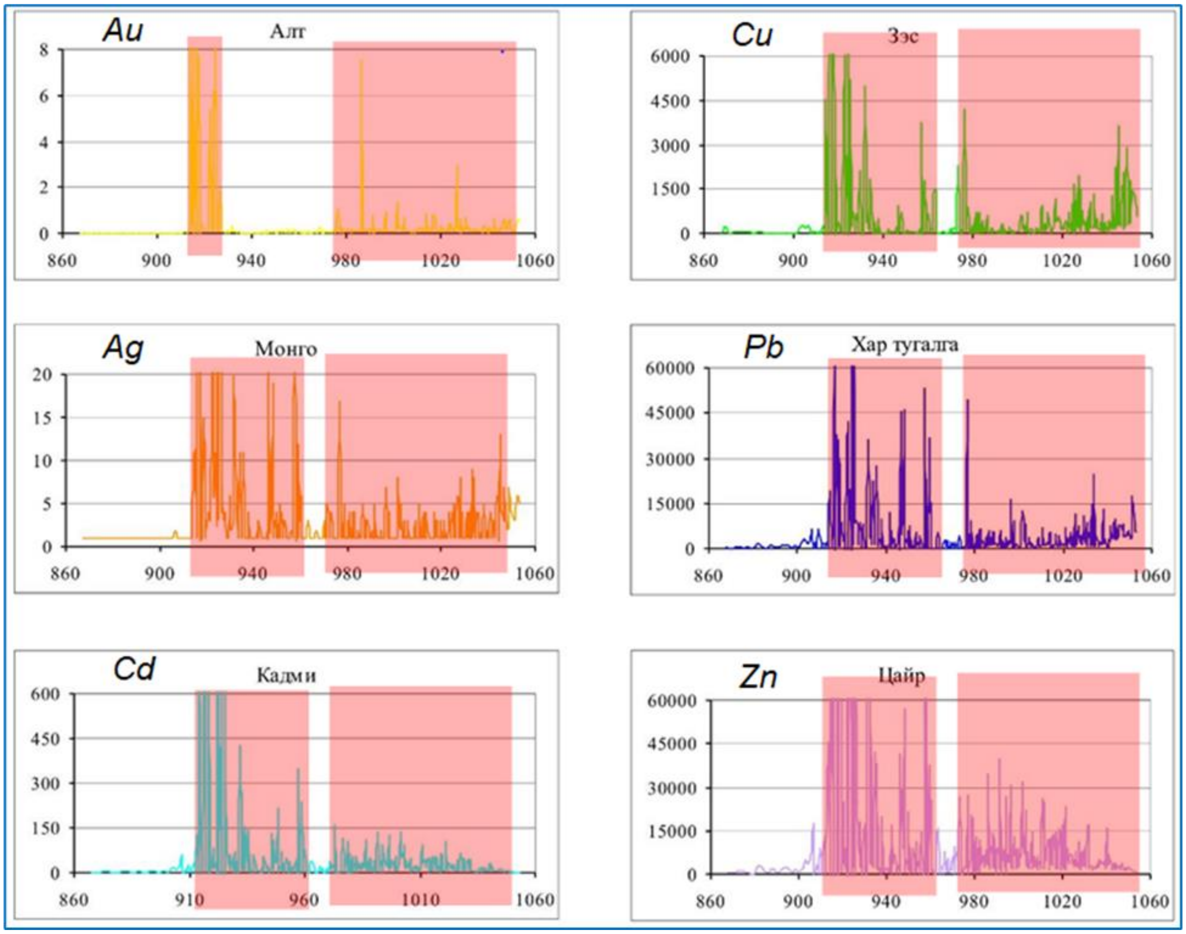
has a perfect correlation with cadmium, strong correlation with silver, copper, lead, and tin, a moderate correlation with antimony, and a weak correlation with bismuth.

Table 10.5 - Pipe 2 Core Sample Element Correlations

Correlation	Au	Ag	As	Bi	Cd	Cu	Mo	Pb	Sb	Sn	Zn
Au	1.00	0.28	-0.01	0.11	0.43	0.38	-0.03	0.18	0.08	0.30	0.43
Ag	0.28	1.00	0.18	0.25	0.79	0.87	0.15	0.96	0.66	0.72	0.79
As	-0.01	0.18	1.00	-0.02	0.07	0.05	0.09	0.16	0.37	0.03	0.06
Bi	0.11	0.25	-0.02	1.00	0.29	0.24	0.00	0.25	0.16	0.32	0.29
Cd	0.43	0.79	0.07	0.29	1.00	0.86	0.09	0.73	0.47	0.87	1.00
Cu	0.38	0.87	0.05	0.24	0.86	1.00	0.13	0.79	0.59	0.79	0.86
Mo	-0.03	0.15	0.09	0.00	0.09	0.13	1.00	0.14	0.22	0.15	0.09
Pb	0.18	0.96	0.16	0.25	0.73	0.79	0.14	1.00	0.63	0.66	0.72
Sb	0.08	0.66	0.37	0.16	0.47	0.59	0.22	0.63	1.00	0.44	0.46
Sn	0.30	0.72	0.03	0.32	0.87	0.79	0.15	0.66	0.44	1.00	0.88
Zn	0.43	0.79	0.06	0.29	1.00	0.86	0.09	0.72	0.46	0.88	1.00

According to the depth analysis in Pipe 2 (Figure 10.4), gold occurs at a zone from 910 m level to 930 m level with very high grades and at a zone from 975 m level to 1,050 m level with irregular lower grades. As for other metals, there is a zone of irregular high grades from 910 m level to 960 m level, followed by a zone of lower grades from 960 m level to 1,050 m level. The graphs show that the grades of copper, silver, and lead increase slightly near surface whereas grades of zinc and cadmium start to drop at 1,040 m level to very low grades.

Figure 10.4 - Pipe 2 Element Correlation with Depth



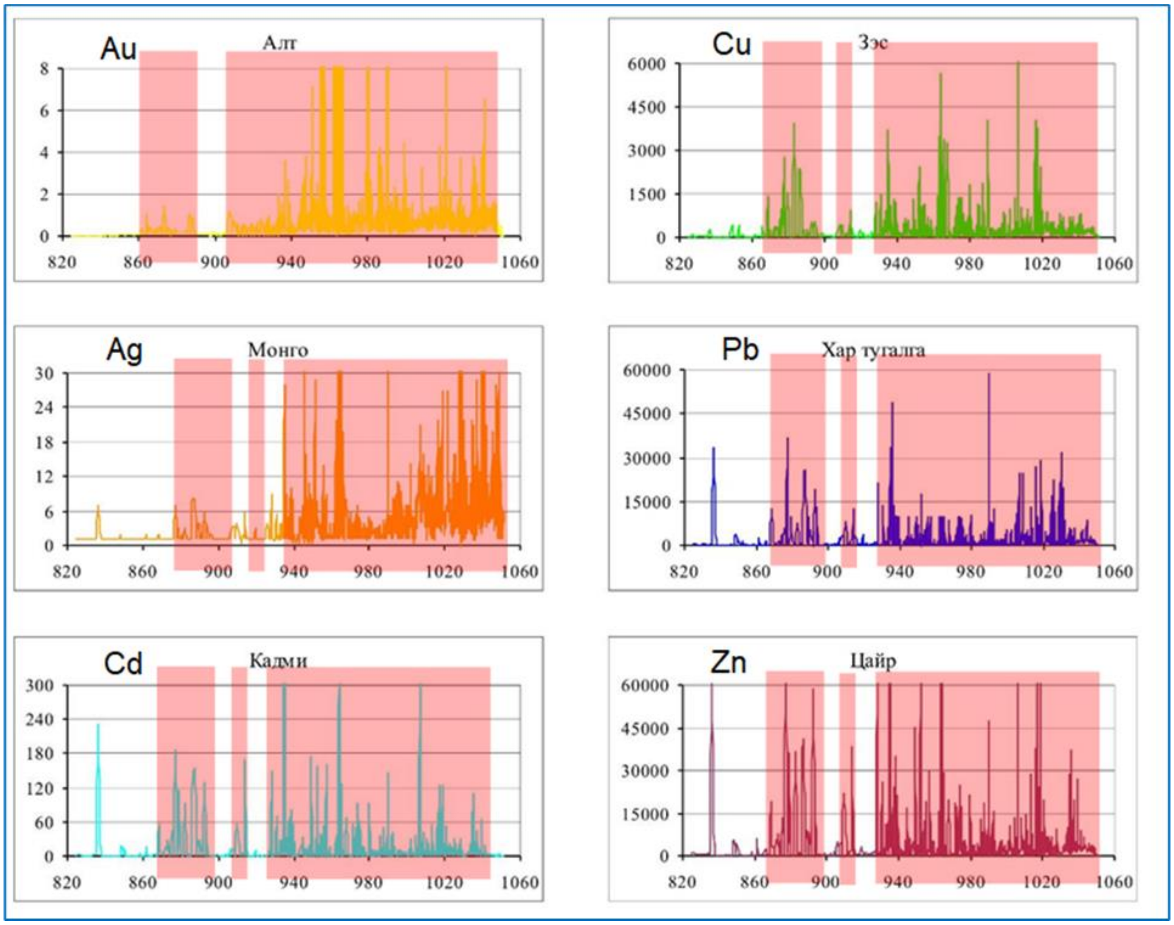
Pipe 4: Correlations of elements in Pipe 4 are provided in Table 10.6. In Pipe 4, gold has a moderate correlation with copper and weak correlations with silver, cadmium, lead, antimony, and zinc. Copper has strong correlations with cadmium, lead, and zinc, a moderate correlation with antimony, and weak correlations with silver and molybdenum. Zinc has very a strong correlation with cadmium, strong correlations with copper and lead, and a moderate correlation with antimony.

Table 10.6 - Pipe 4 Core Sample Element Correlations

Correlation	Au	Ag	As	Cd	Cu	Mo	Pb	Sb	Zn
Au	1.00	0.11	0.06	0.26	0.36	0.03	0.19	0.12	0.23
Ag	0.11	1.00	0.05	0.07	0.11	0.17	0.09	0.06	0.09
As	0.06	0.05	1.00	-0.02	0.07	0.40	0.04	0.50	0.01
Bi	0.01	0.00	0.00	0.04	0.02	-0.01	0.04	-0.01	0.03
Cd	0.26	0.07	-0.02	1.00	0.75	0.07	0.63	0.33	0.97
Co	-0.12	-0.08	-0.05	-0.15	-0.17	-0.05	-0.11	-0.12	-0.16
Cu	0.36	0.11	0.07	0.75	1.00	0.10	0.61	0.49	0.77
Mo	0.03	0.17	0.40	0.07	0.10	1.00	0.17	0.37	0.09
Pb	0.19	0.09	0.04	0.63	0.61	0.17	1.00	0.33	0.67
Sb	0.12	0.06	0.50	0.33	0.49	0.37	0.33	1.00	0.36
Zn	0.23	0.09	0.01	0.97	0.77	0.09	0.67	0.36	1.00

These results suggest that polymetallic elements have better correlations with each other than with gold for Pipe 4. This is also seen clearly on the graphs of grades of gold and polymetallic elements displayed in Figure 10.5. Elements such as copper, lead, cadmium, and zinc have irregular pattern of grades at depth ranges of 860-900 m level, 910-925 m level and 925-1050 m level. As per gold, its mineralisation is established at depth ranges of 860-885 m level and 910-1050 m level. In general, gold and polymetallic mineralisation has been identified having stable and relatively thick distribution below surface. This near-surface zone is marked by good correlations of gold, silver, cadmium, copper, lead and zinc, while the zones of high grades of certain metals with irregular distribution and low thickness that are found below it do not demonstrate correlations with gold; only the polymetallic elements have good correlations with one another.

Figure 10.5 - Pipe 4 Element Correlation With Depth



11 SAMPLE PREPARATION, ANALYSIS AND SECURITY

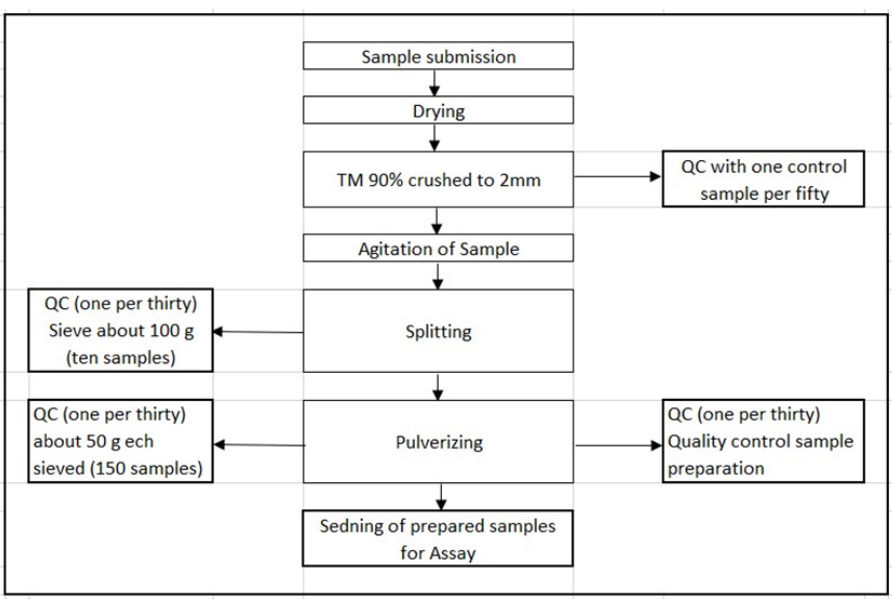
The information in this section is largely drawn and/or summarised from the Report available on SEDAR entitled: "Altan Tsagaan Ovoo Project (ATO), 2021 Mineral Resources, Technical Report (Amended NI 43-101)", prepared by GeoRes, with an effective date March 30, 2021, issued on March 30, 2021 and amended on June 9, 2021. Further details as documented therein remain correct and valid.

11.1 Sample Preparation Before Lab Analysis

Sample preparation is a process that requires such delicate and meticulous procedures as implemented in laboratory analyses. Rock samples (drill core) had been prepared from drill cores using blade saw cutting the drill core on site and sampling 50% of every metre interval from mineralised zones and remaining 50% of drill core kept in storage. Samples bagging in cotton samples bags and numbered to be ready for submission to a laboratory.

The sample preparation for analysis to be done at ALS laboratory workshop in Ulaanbaatar, and the remainders of the samples are kept in ALS storage. The actual preparation starts with a jaw crusher Rhino-Terminator, which produces an average of 2 to 3 mm fractions. About 750-1500 g of material from the jaw crusher is then fed into a Lab Tech Essa 2018 type rotary mill, and after that, into bowl and ring pulverizers like Lab tech LM1 and LM2, reducing 90% of the material to 75 µm. After preparation of each sample, all crusher and mills are cleaned by blowing with high-pressure air. Samples each weighing 300 g were sent for assay at ALS Ulaanbaatar for precious and base metals. Figure 11.1 is a diagram showing the sample preparation procedure at ALS laboratory, inclusive of an internal quality control regime.

Figure 11.1 - Sample Preparation Procedure



Sample preparation Quality Control (QC) regimes:

- Control samples of preparatory stage:
  - Duplicate sample was submitted 1 in 50 samples;
  - It must be duplicated from the preceding sample;
  - This will serve to control the preparation process and must be prepared the same way as primary samples.
- Control samples of pulverizing process:
  - A control sample was prepared 1 in 30 samples;
  - It must the duplicated from the preceding sample;
  - This will serve to control the pulverizing process and must be prepared the same way as primary samples.

11.2 Sample Security

At the drill rig, the drillers removed core from the core barrel and place it directly in wooden core boxes. Individual drill runs were identified with small wooden blocks, where the depth (m) and drill hole numbers are recorded. Unsampled core was never left unattended at the rig; boxes were transported to the core logging facility at the ATO camp under a geologist's supervision.

Core was logged on-site before cut for sampling. Remaining core after sampling was transported in sealed boxes by truck to the core shed in Ulaanbaatar.

All core was stored in a secure location in Ulaanbaatar. Core storing workshop was facilitated with the stable shelfs and logging and sample cutting areas. A full-time assistant was working for the core workshop to ensure core and samples security and tidiness.

11.3 Sample Analysis

All of the core samples, some of the channel and grab samples were assayed at the ALS Ulaanbaatar laboratory used an analytical suite as Multi elements by Aqua-Regia digestion with ICP-AES finish (ME-ICP41). This analytical suite detects 35 analytes, but some analytes have incomplete digestion. These analytes and detection limits are:

Table 11.1 - Analytes and Detection Limits

Analytes and Detection Limits (Ppm)							
Ag	0.2-100	Co	1-10,000	Mn	5-50,000	Sr*	1-10,000
Al*	0.01-25%	Cr*	1-10,000	Mo*	1-10,000	Th*	20-10,000
As	2-10,000	Cu	1-10,000	Na*	0.01-10%	Ti*	0.01-10%
B	10-10,000	Fe*	0.01-50%	Ni*	1-10,000	Tl*	10-10,000

Analytes and Detection Limits (Ppm)							
Ba*	10-10,000	Ga*	10-10,000	P	10-10,000	U*	10-10,000
Be	0.5-1000	Hg	1-10,000	Pb	2-10,000	V	1-10,000
Bi	2-10,000	K*	0.01-10%	S*	0.01-10%	W*	10-10,000
Ca*	0.01-25%	La*	10-10,000	Sb*	2-10,000	Zn	2-10,000
Cd	0.5-1000	Mg*	0.01-25%	Sc*	1-10,000		
Reportable Analytes Upon Request							
Ce*	10-10,000	Nb*	10-10,000	Sn*	10-10,000	Y*	10-10,000
Hf	10-10,000	Rb*	10-10,000	Ta*	10-10,000	Zr*	5-10,000
Li*	10-10,000	Se*	10-10,000	Te*	10-10,000		

* Indicates Possible Incomplete Digestion

For gold analysis, an analytical suite as Fire assay with AAS finish (Au-AA24) was used.

Table 11.2- Analytes and Detection Limits - Gold

Analytes and Detection limits (ppm)	
Au	0.005-10

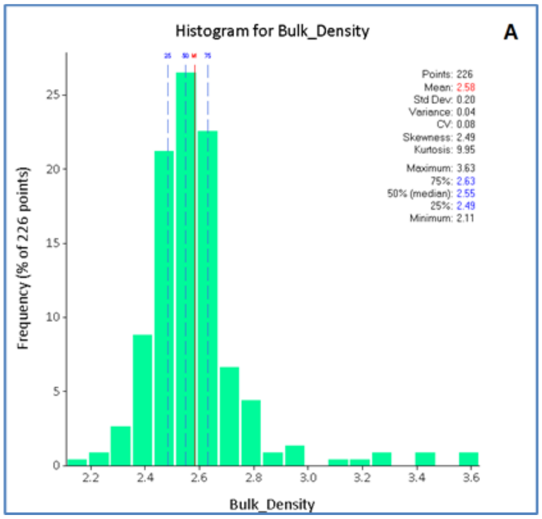
Duration for one submission of laboratory analyses was two weeks turnaround during the field exploration and the results were returned periodically. The dispatch of analysis consists of about 1000 samples each, including standard, blank and duplicate samples.

11.4 Bulk Density Measurements

A total of 226 bulk density determinations were made in 2010 and 2011 from some of the 238 diamond drill holes.

Bulk densities for all samples ranged from 2.11 t/m^3 to 3.63 t/m^3 with a mean of 2.58 t/m^3 (Figure 11.2 (A)).

Figure 11.2 - Density Histogram - All Samples



NB: The Author QP has moved to using t/m³ units of density (specific gravity) here rather than the g/cm³ used previously.

When sorted by oxidation level the densities were:

- Oxide average density 2.46 t/m³ (Figure 11.3 (B))
- Transition average density 2.59 t/m³ (Figure 11.4 (C))
- Fresh average density 2.64 t/m³ (Figure 11.5 (D))

Oxide material densities (62 samples) were the lowest and had a 0.15 t/m³ spread in the 25 to 75% range.

Transitional material densities (42 samples) were markedly higher than for oxide, and the small number of high values were probably from mis-sorted fresh samples. They also had a lower 0.11 t/m³ spread in the 25 to 75% range (were more consistent than oxide).

Fresh material densities (116 samples, more than twice the others) were the highest and had a 0.12 t/m³ spread in the 25 to 75% range, similar to the transitional material.

Figure 11.3 - Density Histogram - Oxide

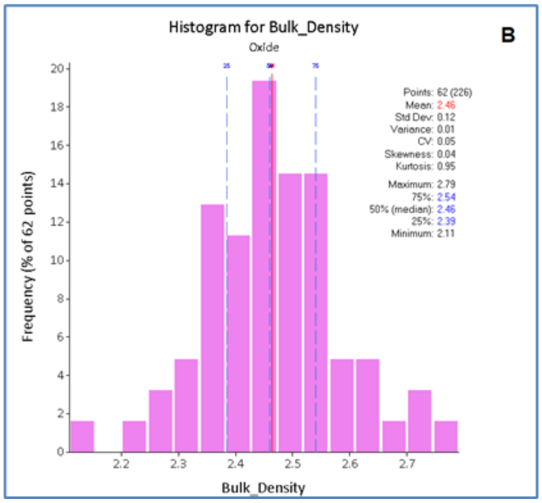


Figure 11.4 - Density Histogram - Transition

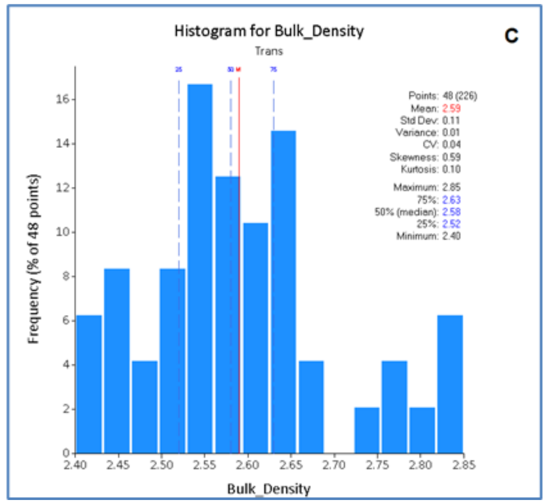
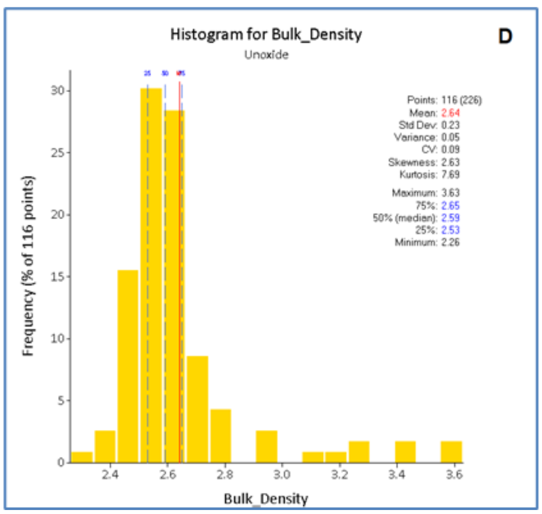


Figure 11.5 - Density Histogram - Fresh



Samples were considered to have had a regular spatial distribution and the 2017 QP considered the averages to represent the deposits.

These density averages for the three oxide levels were used for the 2021 estimation reporting here.

11.5 QA/QC Procedures Behind Sample Confidence

Quality assurance and quality control of samples are some of the procedures that are mandatory in mineral exploration projects from the discovery of a deposit to a pre-production feasibility study stage. These have many advantages, and most important of these advantages are, firstly, actual existence of a mineral deposit and the assurance to the investors as to quality of work performed by geologists, secondly, controlling of laboratories that perform analyses of samples and materials by the exploration company and geologists, and thirdly, confirmation to the professional regulatory organizations as to the reality of information and data collected by exploration activities.

A quality control procedure was maintained before exploration program. Standards for quality control of sampling and assaying have been well maintained during the exploration work of the company. The following types of control samples have been implemented and these constitute the primary actions of Steppe Gold designed to control the works of laboratories.

Steppe Gold's QA/QC protocols for diamond drilling comprise three standards, two blanks and one core (field) duplicate inserted randomly in batches of 100 samples. Field duplicates are prepared by initially cutting the core in half down the long axis with a diamond circular saw. One half is subsequently cut it again so that quarter core samples were submitted to the laboratory as field duplicates.

Standard Reference Material (SRM): SRMs purchased from Ore Research & Exploration P/L, have been used for ATO project since 2018. Certified Value and standard deviation data for the SRMs is shown in Table 11.3.

Table 11.3 - Standard Reference Material Samples Used at ATO

Description	Standard_Id	Element	Nominal Value	Std_Deviation
Lateritic soil Lithogeochem	OREAS 45e	Cu (ppm)	709	52
		Pb (ppm)	14.3	2.4
		Zn (ppm)	30.6	4.8
		Au (ppb)	53	3
Porphyry Copper-Gold-Molybdenum	OREAS 504b	Ag (ppm)	2.98	0.21
		Cu (%)	1.1	0.022
		Mo (ppm)	476	19.4
		Pb (ppm)	20.1	0.92
		Zn (ppm)	96	5.2
		Au (ppm)	1.61	0.04

Description	Standard_Id	Element	Nominal Value	Std_Deviation
High Sulphidation Epithermal Ag-Cu-Au Ore	OREAS 600	Cu (ppm)	488	19.4
		Mo (ppm)	1.92	0.31
		Pb (ppm)	157	5
		Zn (ppm)	598	35.3
		Au (ppm)	0.2	0.006
	OREAS 601	Ag (ppm)	49.4	1.47
		Cu (%)	0.101	0.003
		Mo (ppm)	3.8	0.64
		Pb (ppm)	283	9.5
		Zn (ppm)	1293	78.6
		Au (ppm)	0.78	0.031
Gold-Silver Ore	OREAS 60d	Ag (ppm)	4.45	0.237
		Cu (ppm)	72	2.8
		Pb (ppm)	8.67	0.799
		Zn (ppm)	32.9	2.14
		Au (ppm)	2.47	0.079
	OREAS 61f	Ag (ppm)	3.61	0.171
		Cu (ppm)	39.2	2.2
		Pb (ppm)	7.95	0.529
		Zn (ppm)	44.4	2.59
		Au (ppm)	4.6	0.134
Volcanic Hosted Massive Sulphide Zn-Pb-Cu-Ag-Au Ore	OREAS 620	Ag (ppm)	38.4	1.31
		Cu (%)	0.175	0.005
		Mo (ppm)	8.97	0.71
		Pb (%)	0.774	0.024
		Zn (%)	3.12	0.086
		Au (ppm)	0.685	0.021
	OREAS 621	Ag (ppm)	68	2.41
		Cu (%)	0.366	0.011
		Pb (%)	1.36	0.03
		Zn (%)	5.17	0.148
		Au (ppm)	1.25	0.042
	OREAS 623	Ag (ppm)	20.4	1.15
		Cu (%)	1.72	0.066
		Pb (%)	0.252	0.01
		Zn (%)	1.01	0.038
		Au (ppm)	0.827	0.039

Blanks: These are also submitted to the lab in purpose to detect contamination and sequencing errors. For analysis, blanks purchased from Ore Research & Exploration was utilised as blank material to check for contamination. Those blanks are shown in Table 11.4.

Table 11.4 - Standard Blank Samples Used At ATO

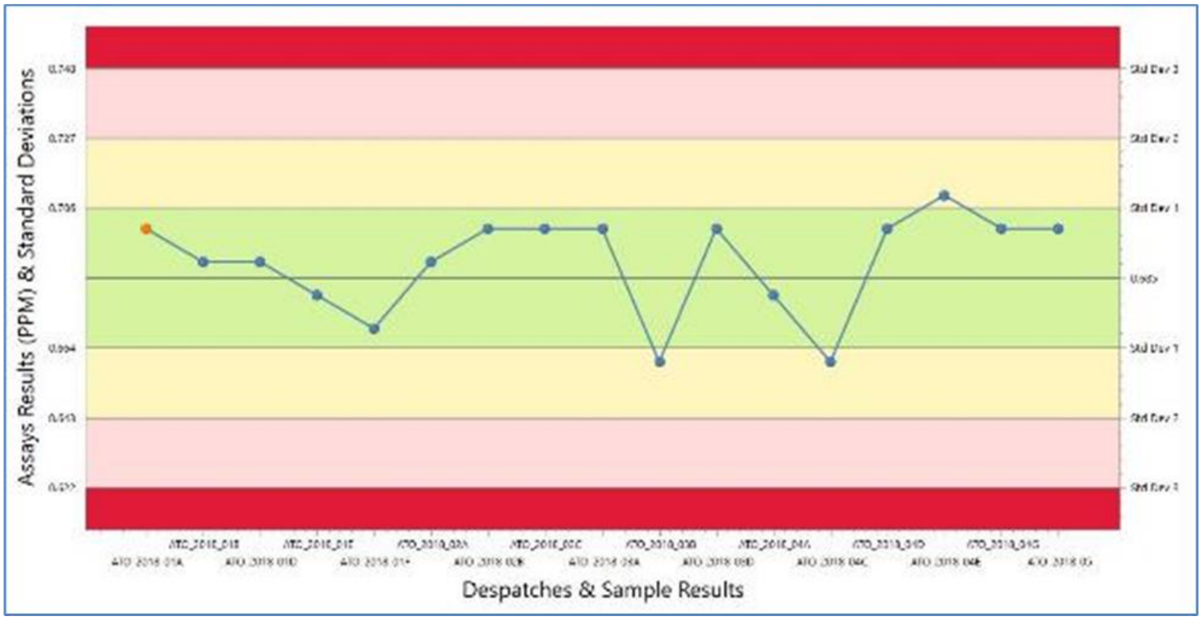
Standard_Id	Description	Element	Nominal Value	Std_Deviation
OREAS 21e	Oxide quartz blank	Cu (ppm)	5.68	0.81
		Pb (ppm)	<1	0
		Zn (ppm)	2.91	0.56
		Au (ppb)	<1	0
		Ag (ppm)	<0.05	0
OREAS 24d	Basalt Blank pulp	Cu (ppm)	43.2	2.29
		Mo (ppm)	4.46	0.350
		Pb (ppm)	3.56	0.44
		Zn (ppm)	104	7
		Au (ppb)	<1	0
		Ag (ppm)	<0.2	0

SRM and blank evaluation: For the purpose of Monitoring QC/QA procedure, there have been set a batch failure criterion using the Standard reference materials and Blanks as follows:

- Individual standards assayed greater than +3 SD of round robin mean;
- Two or more consecutive standards assayed greater +2 SD of round robin mean;
- Individual blanks assayed greater than a cut-off limit of about 0.05 to 0.10 g/t.

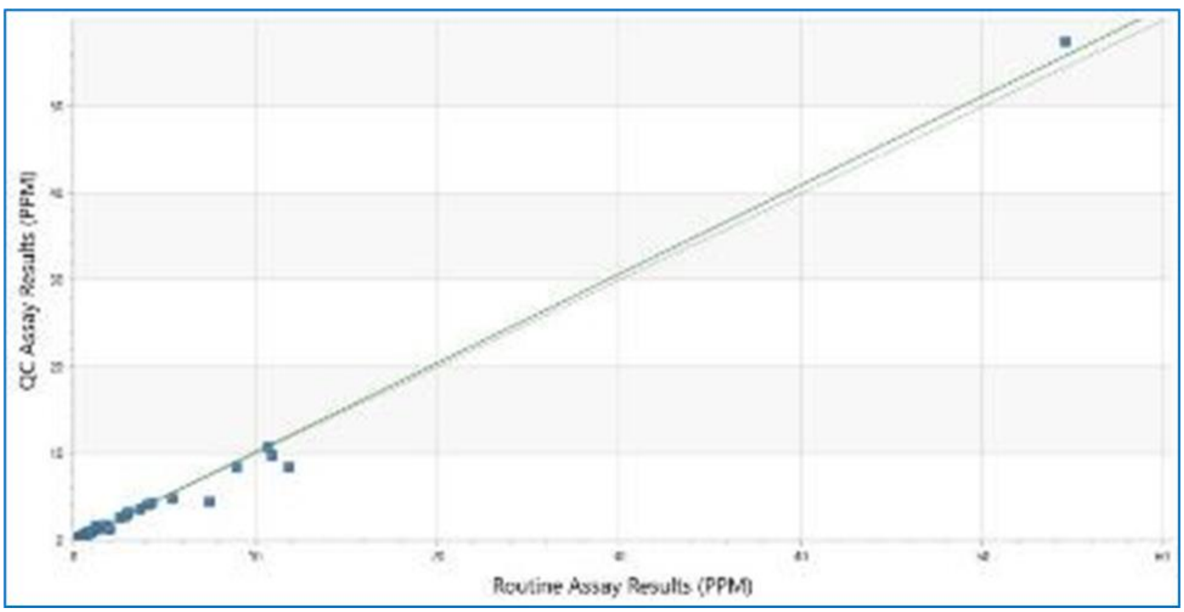
Shewhart plots (Figure 11.6) is constructed from SRM assay results to show the assay mean and distribution compared to the expected mean and distribution as defined by the certificates.

Figure 11.6 - Standard – ID:OREAS 620, Element: Au|Au-AA26



Field Duplicate evaluation: The precision of a measurement system is the degree to which repeated measurements under unchanged conditions show the same results. Scatter plots (Figure 11.7) are created to pair each duplicate with the original assay.

Figure 11.7 - XY chart - routine VS duplicate (F&L): Element: Au|Au-AA26



During the exploration program since 2017, a total of 20,640 samples were submitted from drill cores to the laboratories for analyses. 995 QAQC samples were analysed as part of QAQC procedures, and they account for approximately 4.8% of all core samples.

Laboratories that assayed the samples from the ATO deposit implemented their own internal quality control measures by assaying all of the standard samples, gold-blank samples, and other control samples. In particular, they conducted a control assay on remainder of one pulverized sample out of ten. Quality control of assays of all samples have been monitored and graphed. It is considered a warning sign when standard deviation reaches a point two times greater than actual, and control measures are taken when it is three times greater. Re-assays were conducted on selected samples when standards exceeded the warning level, and when the results of re-assays exceeded it again, all samples in the batch in question were internally re-assayed.

11.6 QP's Opinion on Adequacy of Sampling

The Author QP's overall opinion[15] of ATO's sample preparation, sample security, and sample analysis procedures (including their QA/QC) is positive. The procedures appear sound, thorough, and likely to result in truthful and accurate analyses. They also follow typical industry standards.

The fact that all drilling is now by diamond coring lends further overall confidence in sampling as geology may be inspected more thoroughly than with other methods and storage of core allows future duplicate sampling if issues are found.

[15] This opinion is qualified by the fact that the Author has not physically been able to sight any of the sampling himself (largely due to his current unavoidable inability to visit the site). Although all information has been remotely supplied to the Author (by Steppe, Steppe's Alternate QP and past reporting) he nevertheless finds no reasons or evidence to doubt it. He would comment that ATO's sample security procedures are most difficult to evaluate, and he is simply forced to trust Steppe's word that it is adequate.

12 DATA VERIFICATION

The information in this section is largely drawn and/or summarised from the Report available on SEDAR entitled: "Altan Tsagaan Ovoo Project (ATO), 2021 Mineral Resources, Technical Report (Amended NI 43-101)", prepared by GeoRes, with an effective date March 30, 2021, issued on March 30, 2021 and amended on June 9, 2021. Further details as documented therein remain correct and valid.

Relevant data verification here applied to that data used in the Resource estimation work - namely drilling data.

12.1 Data Verification

Data verification by the Author QP was on the drill hole data supplied by Steppe. It was necessarily only undertaken in an overall viewpoint, on checking locations, and in a statistical manner.

Overall view: The viewpoint approach was to evaluate whether all data, and particularly the new (since 2017) data, "hung together" well. This was evaluated in terms of the described drill hole locations, their orientation and spacing, and the down-hole sampling – from the viewpoint of adequately exploring the geology (particularly as it was appreciated at the time of collection) and the style of mineralisation and deposit.

The drilling was found to be well laid out, spaced and sampled in order to explore the pipe-like deposits. At Mungu, the drilling has not yet determined the full extent of the deposit, a situation simply pertaining because sufficient time has not yet been spent exploring it.

This overall view on data adequacy and accuracy also took into account the similar views expressed in the 2017 Report[16].

Hole locations: Databased drill holes were checked against hard copy plots. All holes were found to match.

Drill hole database: A variety of semi-automatic data checks were made whilst loading raw data into the drill hole database. Very few data issues were found, and those that were (such as the trench surveys) were all and easily corrected.

Statistical approach: Raw drill hole data assays were inspected using simple statistics to evaluate if the data was inside normal bounds and if the later data conformed to earlier results. All data was found to be within reasonable limits and comparable with earlier data.

16 2017 NI 43-101. Section 12.1, pp 99.

12.2 Limitations on Data Verification

The principal limitations to the Author QP's data verification (and understanding of the Project generally) were a site visit and geological rock type logging analysis.

Site visit: The initial limitation was simply the inability to physically check drill hole locations, inspect local geology, inspect drill core, and view sampling procedures. This limitation was solely due to the lack of a site visit, itself caused by the Covid 19 pandemic interrupting international travel. As of the issue data Australia (the Author QP's country of residence) was still barring all citizen exit from the country.

Rock type logging: The overly complex geological logging data prevented a serious analysis of the influence of primary rock types on mineralisation and the geological deposit interpretation. However, the Author expects that the logging would not make a material difference to the mineralisation modelling as the mineralisation observed is too continuous in many areas.

12.3 QP's Opinion on Data Adequacy for Task

The Author QP's overall opinion was that ATO's drilling data was completely adequate* for Resource estimation (the purpose of the Consulting and this Report).

*As before this opinion is qualified by the fact that the Author has not physically been able to sight any of the Project's geology or drilling himself (largely due to his current un-avoidable inability to visit the site).

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

The overall Project consists of two (2) processing facilities: an existing heap leach operation (Phase 1), and a proposed concentrator plant (Phase 2). The oxide portion of the ATO Project (Phase 1) employs a conventional oxide heap leach flowsheet including crushing, heap leaching, and gold recovery facilities. Phase 1 has been operational since July 2020 and focuses on the production of gold and silver doré.

Phase 2 will comprise three-stage crushing (existing, as part of expansion to Phase 1 crushing), milling, flotation, and dewatering unit operations to produce concentrates of lead (Pb), zinc (Zn), and pyrite (Py). This section covers the testing programs conducted on representative samples from the ATO deposit for Phase 2 and it will also reference some of the Phase 1 historical testwork.

This section includes summaries of past and recent testwork that were used to develop the process flowsheet and plant design for treating the oxide, transitional ore and sulphide ore based on the mineral resources estimation process. These testwork programs will be referenced throughout this section of the Report.

DRA supervised and provided input during the development of the testwork conducted by the laboratory in 2021. The interpretation and analysis of the testwork results were carried out by DRA. This analysis was then used to determine the process design basis and flowsheet of the ATO Phase 2 Project.

13.2 Historical Testwork (2010-2018)

The information presented in this section is, for the most part, largely drawn or summarised from the reports entitled “Xstrata Process Support - Draft Report - 401824.00 – Mineralogical and Metallurgical Test Program – Final Report submitted, June 30, 2011”, “Centerra Gold Inc. Altan Tsagaan Ovoo (“ATO”) Pilot Plant, rev. Fina, Project # 4011907.00, November 9, 2012”, and “Mineralogical and Metallurgical Test, Program - Phase 2, Master Composite, Pipe 2, Pipe 4 and Variability Samples 1 to 4 based on Optimized Results, rev. Final, Project # 4011903.00, November 21, 2012 “.

Several metallurgical testwork programs have been undertaken on samples selected from the Project. These metallurgical tests for processing of ATO ore samples were conducted at the Central Laboratory of Xstrata Process Support (XPS) in Canada, ALS Metallurgy-Ammtec laboratory in Australia, Boroo Au LLC processing plant in Mongolia and SGS Lakefield (SGS) in Canada.

Metallurgical test samples were selected from the drill core and bulk samples from ATO Deposit’s oxidized zone in Pipes 1, 2, and 4. These tests for ore samples included a step-by-step leaching test carried out using bottle roll testing.

The following testwork programs were completed:

1. Grindability Characteristics Pt. 1 and Pt. 2 (from March 11, 2011, to October 24, 2011), SGS Lakefield (SGS).
2. Mineralogical and Metallurgical Test Program, Phase 1 and Phase 2 (from June 30, 2011 to November 21, 2011), XPS.
3. ATO Pilot Plant Testing Pt. 1 and Pt. 2 (September 2012 to November 2012), XPS.

Table 13.1 is a matrix summarising the different tests completed within each program.

Table 13.1 – Historical Testwork Matrix

Reports	Laboratory	Mineralogy	Comminution	Cyanidation		Flotation
				Column Leach	Bottle Roll	
Grindability Characteristics Pts. 1 & 2	SGS		x			
Mineralogical and Metallurgical Test Program. Pts.1 & 2	XPS	x			x	x
ATO Pilot Plant Pts.1 & 2	XPS	x	x			

13.2.1 Mineralogy and Elemental Analysis

13.2.1.1 Mineralogy

In 2011, XPS undertook metallurgical testwork on composites from five (5) zones from the ATO deposit known as Upper Oxide Zone, Upper Transition Zone, Lower Transition Zone, Upper Sulphide Zone and Lower Sulphide Zone composites.

During the Mineralogical and Metallurgical test program, a basic mineralogical characterisation was completed. Mineralogical analysis via QEMSCAN was completed for three (3) composites (Oxide, Upper Transition, and Lower Sulphide) and is summarised in Table 13.2.

Table 13.2 – Minerals Distribution Within the Three Composites

Mineral	Oxide	Upper Transition	Lower Sulphide
Chalcopyrite (%)	0.03	0.13	0.14
Sphalerite (%)	0.03	3.84	2.48
Pyrite (%)	0.23	3.2	4.13
Galena (%)	0.14	1.81	1.13
Other Sulphides (%)	0.05	0.02	0.02
Quartz (%)	81.13	67.86	61.37
Chlorite (%)	8.83	15.51	13.35
Muscovite (%)	1.58	1.26	6.08
Biotite (%)	1.82	3.18	6.35
Other Silicate Gangue (%)	0.30	0.13	0.71
Zn Carbonate (%)	0	0.20	0
Pb Carbonate (%)	0	0.02	0
Pb Sulphates/Phosphates (%)	1.78	0.29	0
Fe Sulphates (%)	1.07	0	0
Carbonates (%)	1.40	2.20	3.80
Other (%)	1.59	0.37	0.43

Zinc Deportment data is presented in Table 13.3.

Table 13.3 – Zn Deportment in Oxide, Transition and Sulphide Zones

Mineral	Oxide	Transition	Sulphide
Chalcopyrite (%)	0.02	0.03	0.01
Sphalerite (%)	15.1	93.0	98.0
Galena (%)		0.10	0.01
Chlorite (%)	32.2	1.9	1.3
Fe Clay (%)	10.2		
Muscovite (%)	10.3	0.04	
Biotite (%)	3.0	0.12	0.5
Zn Carbonate (%)	0.5	4.7	
Pb Sulphates/Phosphates (%)	20.6	0.06	
Pb bearing Siderite (%)	8.0		
Carbonates (%)		0.06	0.25

Zn deportment data indicated that the maximum achievable recovery of Zn to Sphalerite in the sulphide zone is 98%. This assumes 100% recovery of sphalerite, and 100% rejection of other mineral species. In reality, this degree of separation is not possible. Maximum achievable recovery dropped to 93% in the Transition Zone due to the Zn that is present in non recoverable carbonate and silicate forms. Zn in the Oxide Zone occurred primarily in silicate, phosphate/sulphate and carbonate forms. Only 15.1% of the Zn in the Oxide Zone occurred as sphalerite. Zn in the Transition and Sulphide Zones occurred primarily as sphalerite, at 95 and 98% respectively.

Mineralogical analysis of the Master Composite rougher tailing analysis indicated that Zn losses were primarily in the form of Zn carbonate and Zn that occurred as solid solution within silicate gangue. Just 20% of the total Zn in the rougher tailing was in the form of sphalerite. A total of 90% of the sphalerite occurred as locked particles within silicates or carbonate gangue. The rougher flotation circuit therefore produced very good sphalerite recoveries.

Liberation of sphalerite (defined as particles containing >90% by area sphalerite) was good to very good in the samples.

Galena liberation was lower than sphalerite in all of the samples analysed. In most cases, liberation improved with increasing grades. However, in this case, the Pipe 2 sample had the highest grade of galena, while the liberation is better in the Master Composite.

Mineralogical analysis of unsized Master Composite Pb rougher and Zn rougher flotation concentrates were completed to assess dilution and understand limitations to selectivity.

- The Pb rougher concentrate contained 14.3% non-sulphide gangue, 10.7% sphalerite and 19.2% pyrite dilution. Liberation of galena was low at 57.5%. Of the locked and middling galena, 10% was with sphalerite and the remainder was with pyrite and non- sulphide gangue (NSG).
- Sphalerite dilution in the Pb rougher concentrate was 48.2% liberated. 20% of the middling and locked sphalerite was associated with galena.
- Pyrite dilution in the Pb rougher concentrate was 75% liberated.
- NSG in the Pb rougher concentrate was 57.4% free or liberated (>90% NSG by area in particle) and may have been recovered through entrainment.
- Galena losses in the Zn rougher concentrate were 82% locked. The grain size of the locked galena was close to 5 µm and associated with sphalerite and pyrite.
- NSG in this stream was 7.7% free, 52% liberated and 39% in the middling category. 75% of the NSG middling locks were associated with sphalerite only. 15% of the middling locks were associated with pyrite only.

13.2.1.2 Elemental Analysis

XPS - 2011

During the first part of the Mineralogical and Metallurgical Test Program developed by XPS in 2011, an elemental analysis was conducted where approximately 130 kg of drill core each from Upper Oxide, Upper Trans, Upper Sulphide, Lower Trans Zone, and Lower Sulphide Zone composites were tested, and results are summarised in Table 13.4.

Table 13.4 – Sample Composites

Sample	Cu	Zn	Pb	Fe	S	Au	Ag	Cd
	(%)	(%)	(%)	(%)	(%)	(ppm)	(ppm)	(%)
ATO-01 (Upper Oxide Composite)	0.09	0.27	1.45	2.61	0.69	2.35	6.37	0.001
ATO-02 (Upper Trans Composite)	0.11	2.64	1.24	2.58	2.78	4.14	10.63	0.009
ATO-03 (Upper Sulphide Composite)	0.11	1.21	0.95	2.78	2.45	6.39	14.50	0.004
ATO-04 (Lower Trans Composite)	0.26	4.90	4.14	3.88	5.99	4.07	23.50	0.019
ATO-05 (Lower Sulphide Composite)	0.11	2.69	1.67	3.21	3.76	2.10	7.31	0.009

XPS - 2012

During the Mineralogical and Metallurgical Test Program Part 2 developed by XPS in 2012, the lab received drill core samples of the ATO deposit. From this core, a Master Composite, Pipe 2 Composite, Pipe 4 Composite, and Variability Samples 1 to 4 were created. The samples were prepared and assayed using XPS external reference distribution method to ensure quality, and the results are reported in Table 13.5.

Table 13.5 – Average Grades of Samples Received (XPS, 2012)

Sample	Au	Ag	Pb	Zn	Cu	Fe	S
	(ppm)	(ppm)	(%)	(%)	(%)	(%)	(%)
Master Composite	1.69	8.40	0.84	1.44	0.06	2.98	2.64
Pipe 2 Composite	0.68	6.69	1.30	2.85	0.07	4.27	3.01
Pipe 4 Composite	2.74	8.30	0.47	0.84	0.06	3.18	2.64
Variability #1	11.7	40.2	5.98	5.91	0.713		7.14
Variability #2	6.0	35.0	1.70	2.65	0.102		1.52
Variability #3	0.21	27.3	3.46	13.56	0.152	3.99	10.19
Variability #4	0.43	12.4	1.40	4.79	0.093	3.87	4.14

A Mini Pilot Plant (MPP) Sample was created, made up of specific masses of variability samples and blended to achieve the Pb and Zn grades that are representative for the orebody. The MPP Sample along with representative subsamples were sent to determine the assay of the composite and an external reference distribution was performed to ensure quality of blending. The results are presented in Table 13.6.

Table 13.6 – External Reference Distribution for MPP Sample

Sample	Cu	Fe	Pb	Zn	S	Ag	Au	Pd	Pt
	(%)	(%)	(%)	(%)	(%)	(g/t)	(g/t)	(g/t)	(g/t)
Average	0.055	2.91	0.68	1.13	2.39	5.90	1.24	<0.01	<0.01
Std. Dev (%)	0.003	0.12	0.027	0.04	0.04	0.32	0.03	N/A	N/A
RSD (%)	5.77	4.27	3.93	3.49	1.66	5.36	2.68	N/A	N/A

N/A: Not Applicable

13.2.2 COMMINUTION

The information presented in this section is, for the most part, largely drawn or summarised from the report entitled "SGS Minerals Services Lakefield; An Investigation into the Grindability Characteristics of Three Samples from Centerra Gold", submitted by XPS, Project 12076-014 – Final Report, October 26, 2011. XPS mandated SGS to carry out the grindability testing on three (3) samples.

13.2.2.1 Ore Hardness

As part of XPS's Phase 2 Program, the grindability characterisation study also included the J-K drop-weight as well as the Bond ball mill grindability tests. The three samples were labelled as Master Comp, Pipe 2 Comp, and Pipe 4 Comp.

Based on the resistance to impact breakage (A x b), resistance to abrasion breakage (t_a) and its BWi value; of the three composite samples, the Master Comp was the hardest, whereas Pipe 2 Comp and Pipe 4 Comp are considered soft to moderately soft. The results are summarised in Table 13.7.

Table 13.7 – Grindability Test Summary

Sample Name	Relative Density	JK Parameter A x b	JK Parameter t_a	BWi (kWh/t)
Master Comp	2.75	50.9	0.39	15.5
Pipe 2 Comp	2.75	62.2	0.66	15.6
Pipe 4 Comp	2.67	95.3	0.56	14.6

13.2.3 Column Leach

The results of the column leach cyanidation for the Master Comp reveal the following:

- Column leach cyanidation testwork resulted in an excellent Au extraction level of 81.9 % after 40 days of leaching. The extraction level of Ag at 61.5%, would be considered good for a column leach cyanidation test.
- Au dissolution kinetics were relatively rapid with the bulk of exposed cyanidable Au being solubilised within ten days from the start of column leaching.
- Sodium cyanide consumption was relatively high, being 4.6 kg/t; this is believed to be due to the open environment to which the column leach is exposed as compared to the closed environment for the earlier bottle roll leach tests. Also, actual operational heap leach cyanide consumption would be in the order of 30% of column leach test results due to the hydrolysis associated with the exposure of the cyanide in solution to air and sunlight.

Table 13.8 – Column Leach Test Conditions

Sample	Test	Crush Size	Leach Duration	Wash Duration	Percolation Rate*	
		mm	days	days	L/min	L/m²/hr
MC1	J51059	<12.5	40	7	0.216	413.2

Note: * This is on final day prior to column being dumped.

Table 13.9 – Column Leach Extraction Results

Sample	Test	Element	Au and Ag Extraction (% after day)								Consumption (kg/t)	
			1	2	5	10	20	30	40	Final*	NaCN	Lime
MC 1	JS1059	Au	30.9	44.3	59.2	68.6	76	80.3	81.9	82.5	4.6	1.6
		Ag	3.2	7.7	19.3	33.4	48	57.4	61.5	62.5		

Note: * Forty days of leaching plus seven days of water wash.

The column leach tests were terminated at day 57 with maximum Au recovery achieved after 35 days of leaching. Solution to ore ratios between 2.4 and 6.1 were achieved at the completion of the tests. Au and Ag analysis in each size fraction were conducted on column residues following rinsing for four days to remove residual Au in solution.

Figure 13.1 – Large Scale Column Test Flowsheet

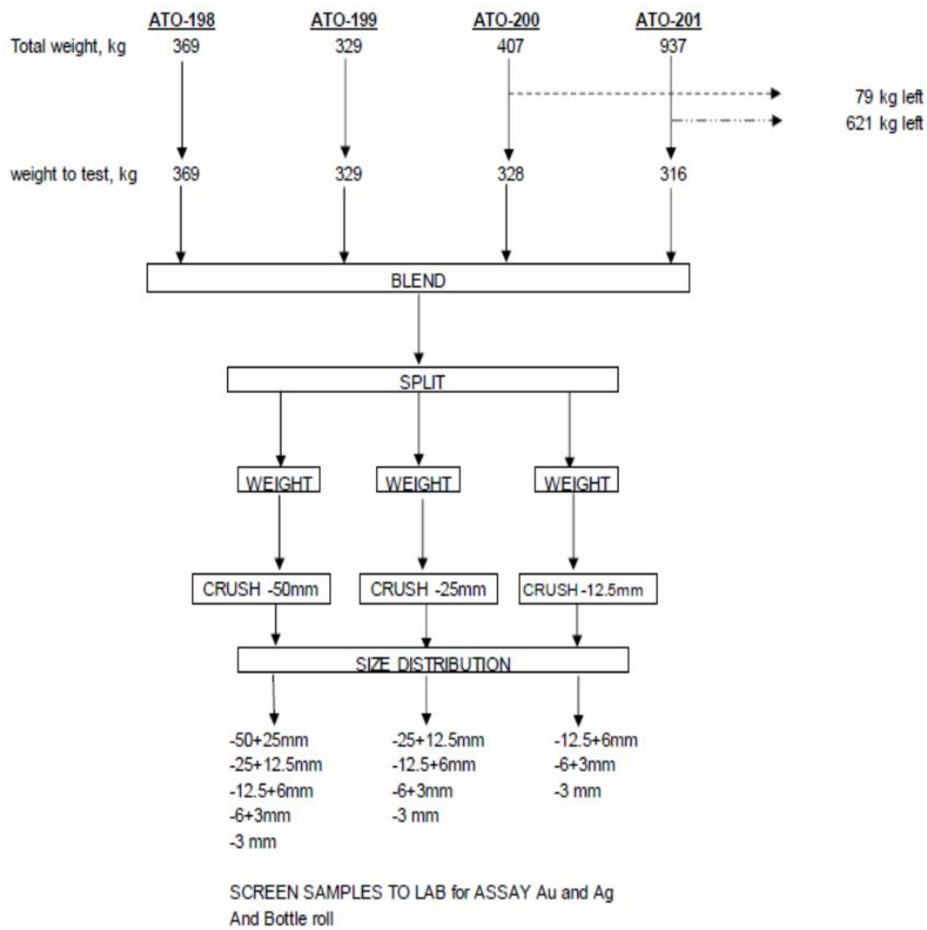


Table 13.10 – Large Column Test Results

Test	Feed Grade	Recovery (%)		Ratio	Cyanide Solution	pH
	(g/t)	Bottle Roll (24 h)	Column Test		(mg/L)	
-12.5 mm	1.25	36.8	72.65	2.5	370	10.53
-25 mm	1.34	58.1	72.23	2.6	488	10.75
-50 mm	1.12	42.0	58.47	2.38	628	10.87

Table 13.11 – Recovery of Large Column Test

Test	Recovery (% after day)						Average Recovery
	5	15	25	35	45	57	(%)
-12.5 mm	39.83	66.69	70.61	71.73	72.58	72.65	10.53
-25 mm	54.01	66.78	69.80	71.15	71.91	72.23	10.75
-50 mm	39.44	51.93	55.32	56.71	57.99	58.47	58.47

Table 13.12 – Recovery of Large Column Test at Different Ratios

Test	Solution/Solid Ratio 1:1			Solution/Solid Ratio 2:1		
	Recovery (%)	Time (day)	Percolation Rate (L/h/m²)	Recovery (%)	Time (day)	Percolation Rate (L/h/m²)
-12.5 mm	70.59	23	0.21	72.58	46	0.2
-25 mm	69.80	24	0.2	71.91	45	0.22
-50 mm	55.32	24	0.2	58.23	47	0.2

Table 13.13 – Au and Ag Recovery in Ore Classification

Class	Before Testing	After Testing	Recovery % (Ore)
	Au (g/t)	Au (g/t)	Au (%)
Test: -50 mm			
-50 mm +25 mm	0.6	0.2	68.3
-25 mm +12.5 mm	1.1	0.5	52.3
-12.5 mm +6aim	15	0.5	69.3
-6 mm +3 mm	1.9	0.7	62.4
-3 mm	5.0	0.9	83.0
Test: -25 mm			
-25 mm +12.5 mm	1.2	0.3	73.5
-12.5 mm +6 mm	1.7	0.3	80.4
-6 mm +3 mm	1.8	0.6	68.0
-3 mm	3.1	0.4	88.3
Test -12.5 mm			
-12.5 mm +6 mm	1.7	0.3	80.1
-6 mm +3 mm	2.1	0.4	80.8
-3 mm	5	0.2	95.6

13.2.4 Gravity Recoverable Gold (GRG) Testing

During the Mineralogical and Metallurgical Test Program developed by XPS in 2011, sample ATO-01 was submitted to the Knelson Research and Technology Centre to determine its Gravity Recoverable Gold (GRG) value. The GRG test is based on progressive particle size reduction followed by precious metals recovery using a Knelson concentrator at each stage. The progressive size reduction stages allow for precious metals recovery as they are liberated while minimising over grinding and smearing. The GRG for sample ATO-01 was only 4.8% with a tailings grade of 1.92 g/t Au from a head grade of 2.66 g/t Au. This sample is not economically amenable to gravity separation.

As part of the Mineralogical and Metallurgical Test Program developed by XPS in 2011, the master composite was also submitted for GRG testing.

A Knelson single pass test resulted in 24% Au recovery to a concentrate with 3.5% mass recovery and demonstrates the potential for gravity recovery with this sample. Results are presented in Table 13.14.

Table 13.14 – Single Pass Knelson Test on Master Composite Feed

Product & Mass			Assay (%, g/t)				Recovery (%)			
Product	Mass (g)	Mass (%)	Pb	Zn	S	Au	Pb	Zn	S	Au
ERD Head Assay	4,000.00		0.84	1.44	2.64	1.69				
Calc Head	3,987.65	99.54				1.74				
Panner Conc	59.76	1.50				21.30				18.35
Panner Tail	81.38	2.04				4.94				5.80
Knelson Conc	141.14	3.54	6.76	2.85	9.17	10.40	28.53	7.03	12.31	24.15
Knelson Tail	3,840.51	96.46	0.62	1.39	2.40	1.37	71.47	92.97	87.69	75.85

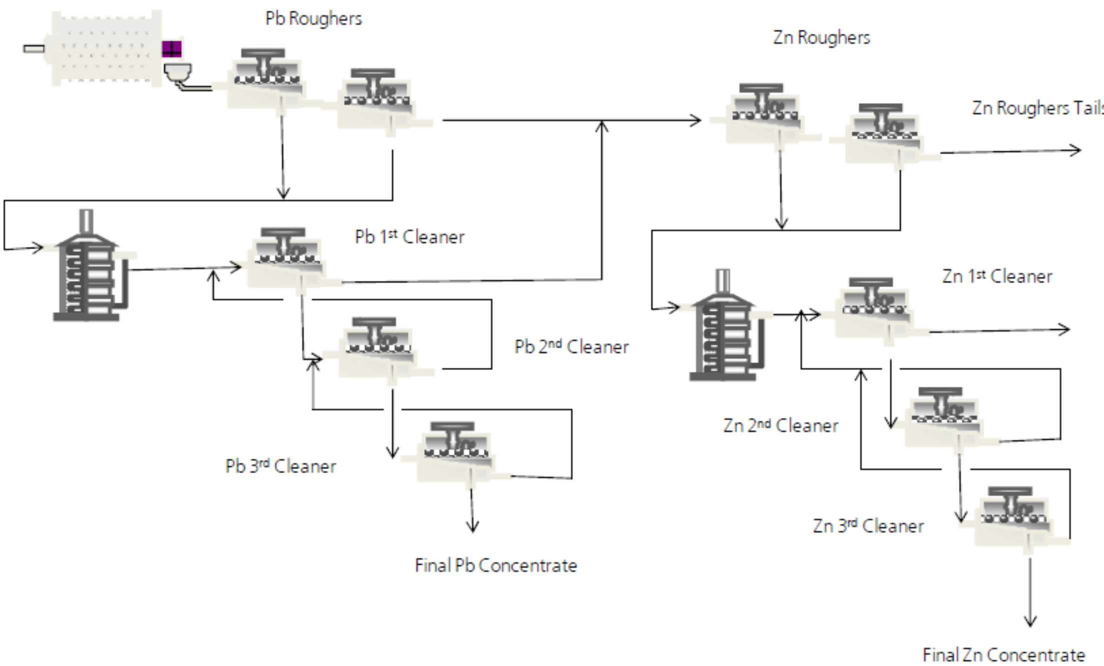
13.2.5 Flotation

As part of the Mineralogical and Metallurgical Test Program developed by XPS in 2011, various Locked Cycle Tests were performed, results are shown as follows:

13.2.5.1 Locked Cycle Tests (LCTs)

Locked Cycle Tests (LCTs) were conducted to confirm the open test results, the flowsheet considered one Pb rougher stage followed by regrinding and three cleaner stages. Tails from the Pb circuit fed the Zn rougher cells which were followed by regrinding and three cleaner stages. The flowsheet is shown in Figure 13.2. The LCT produced Pb and Zn grades >50%. The Pb grade was shown to exceed the 50% target at the 3rd cleaner stage. The Mini Pilot Plant (MPP) flowsheet was also developed based on the flowsheet shown in Figure 13.2.

Figure 13.2 - Standard Process Flowsheet for MPP Run



Generally, the MPP performed well. Au and Ag recovery to the Pb concentrate averaged 28% and 42% respectively and decreased with increasing Pb concentrate grades.

The MPP circuit depicted in Figure 13.2 was judged to be successful in upgrading the ATO ore tested and does not require any further revision for Pb and Zn upgrading. Au and Ag recovery units reporting to combined tailings amounted to 67% and 51% respectively, opening the possibility of cyanidation of mill tailings.

13.2.6 LEACHING AND CYANIDATION

During the Mineralogical and Metallurgical test program Phase 2 developed by XPS, leaching tests were completed on Zn 1st cleaner tails, Zn 2nd cleaner tails and Zn rougher tails from LCT #1. The pH was adjusted using lime. For all tests Au leach recovery levelled off at about two hours of leaching which implied that there was no further leaching occurred after this time.

Cyanidation tests were completed on the MPP products; one on the Zn rougher tails and a second on the Zn cleaner 1 tails. An overall additional 30% of the Au and 19% of the Ag can be extracted form this ore via direct cyanidation of the tailings streams.

The distribution or deportment of Au in various minerals is determined by a series of selective leaches, usually by increasingly stronger oxidative acid leaches. Between each stage, cyanide leaching is used to extract the released Au. In this study, a total of ten analysis stages were carried out for the composite at a grind size of 150 µm. A summary of the results can be seen in Table 13.15. This test work was conducted at ALS Metallurgy Ammtec in Perth, Australia.

Table 13.15 – Multi-Stage Sequential Diagnostic Au Leach Summary

Description	Au (g/t)	Distribution (%)
Gravity/Free Cyanidable Au	1.28	79.3
Carbonate Locked Au Content	0.14	8.7
Iron Oxide Locked Au Content	0.04	2.2
Arsenical Mineral Locked Au Content	0.11	7.1
Pyritic Sulphide Mineral Locked Au Content	0.007	0.5
Silicate (Gangue) Locked Au Content	0.036	2.2

The diagnostic results indicate that the bulk of the Au content occurs as free (gravity/cyanidable) gold, being 79.3% of the total Au content. There was a moderate amount of Au associated with carbonate minerals and arsenical minerals, being 8.7% and 7.1%, respectively.

The preliminary cyanidation test program included tests aimed at investigating the effect that crush and grind size has on Au and Ag extraction via cyanidation. Conventional cyanidation was subsequently investigated as a potential process flowsheet. The following subsections discuss the results of these tests.

A schematic of the test flowsheet, along with the test results, are displayed in Figure 13.3 and Table 13.16. The results of these tests reveal the following points of interest:

- Au extraction appears to be insensitive to grind size, with recovery being approximately 80% for a range of grind sizes; from a K80 of 78 µm to a crush size of K100 3,350 µm.
- Ag extraction increased markedly (from ~50% to 69%), when feed size to cyanidation was decreased from a K100 of 3,350 µm to a K80 of 299 µm. Below this sizing, there was no significant improvement in Ag extraction.
- Sodium cyanide consumption levels are moderate (<2.0 kg/t). Lime consumption decreases as grind size increases (4.3 kg/t at K80 of 78 µm K80 and 1.6 kg/t at K100 of 3,350 µm).

Figure 13.3 – Flowsheet - Master Composite 1

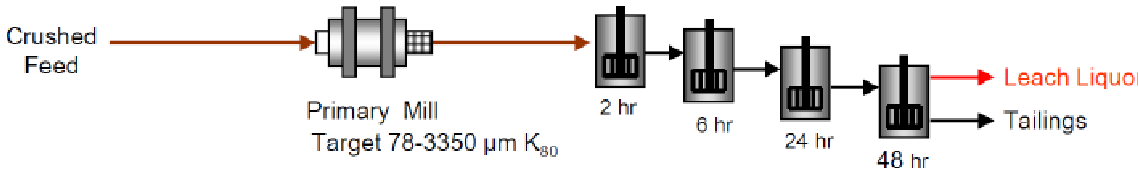


Table 13.16 – Test Results – Master Composite 1

Test	Grind Size	Extraction at Time Hours (Au %)				Extraction at Time Hours (Ag %)				Consumption (kg/t)	
#	µm	2	6	24	48	2	6	24	48	NaCN	Lime
1	78	44.6	62.6	71.5	77	39.6	64.3	69.5	71.7	1.9	4.3
2	102	62.7	72.1	75.5	80.1	43.9	57.4	71.5	75.0	1.7	2.5
4	153	65.0	71.6	76.1	79.4	46.3	59.1	70.2	74.5	1.4	2.0
5	299	57.5	66.5	66.5	82.2	43.2	54.6	64.7	68.9	1.5	1.5
3	3,350	53.8	63.2	63.2	80.9	28.1	37.3	47.2	50.4	1.8	1.6

Note: 3,350 µm was K100 crush size.

13.3 Testwork (2021)

The information presented in this section is, for the most part, largely drawn or summarised from the report entitled "Feasibility Level Metallurgical Testing of the ATO Project", prepared by Base Metallurgical Laboratories (BML), Project BL656 – Draft Report, August 30, 2021.

13.3.1 Sample Selection

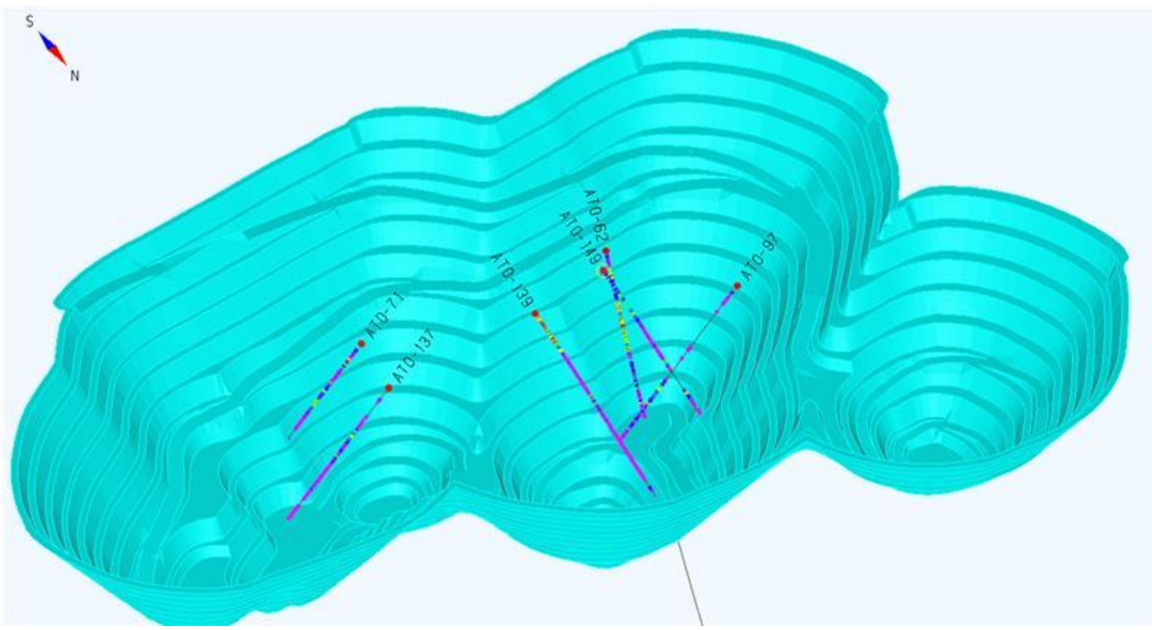
The 2021 metallurgical testwork program was completed by Base Metallurgical Laboratories (BML) in Kamloops, British Columbia, Canada. The samples for the metallurgical program were selected from the ATO Deposit. BML and DRA performed a comprehensive analysis of the ore types within the deposit and concluded that the samples tested are representative of the overall ATO deposit. At the time of selection, a mine production schedule and LOM plan generated during the Feasibility Study work was not available. Table 13.17 lists the ATO samples collected for the metallurgical testwork. The samples were chosen with a focus on the main lithologies within the deposit which are tuff gravelstone and breccia.

Table 13.17 –ATO Samples for Metallurgical Testwork

HOLE_ID	Depth From (m)	Depth To (m)	Au g/t	Ag g/t	Pb % w/w	Zn % w/w	Lithology	Weight Quarter Core
ATO-139	102.85	127.95	1.07	4.87	0.80	1.89	Tuff gravelstone	52.46
ATO-149	150.40	175.4	0.86	5.16	0.99	2.19	Tuff breccia	52.25
ATO-137	60.00	90.1	1.97	7.76	0.87	1.41	Tuff gravelstone	62.91
ATO-62	127.20	169.4	1.45	11.49	0.84	2.53	Tuff breccia	88.21
ATO-71	63.20	106.6	1.49	14.98	0.86	1.75	Tuff breccia	90.71
ATO-97	135.40	182.4	1.33	9.51	1.43	1.70	Tuff gravelstone	98.24

Figure 13.4 depicts the latest mine pit shell and location of the six (6) ATO samples selected for the testwork.

Figure 13.4 - Pit Shells with the ATO Samples



Once the sample preparation had been completed, head assays, mineralogical analysis, physical characterisation, gravity gold analysis, flotation, diagnostic leach and settling tests were conducted on both the Master Composite and Variability 1-6 samples. Table 13.18 shows the composition of the Master Composite.

Table 13.18 – Master Composition Make-Up Sample

Var #	Description	Hole ID	Depth From (m)	Depth To (m)	Mass (kg)	Split Master[1] (kg)
1	Tuff Gravelstone	ATO-139	102.85	127.95	91	45.5
2	Tuff Breccia	ATO-149	150.4	175.4	86	43
3	Tuff Gravestone	ATO-137	60	90.1	110	55
4	Tuff Breccia	ATO-62	127.2	169.4	150	75
5	Tuff Breccia	ATO-71	63.2	106.6	108	54
6	Tuff Gravelstone	ATO-97	135.4	182.4	162	81

1: Mass used to form the Master Composite.

13.3.2 Head Assays and Mineralogy Characterisation

Head assays and mineralogical analysis were carried out on subsamples of the master composite and variability samples. Head assays for Au ranged between 0.86 and 1.79 g/t. The head sample assays of the precious and base metals are shown below in Table 13.19.

Table 13.19 – Head Sample Assays

Products	Element (Average)					
	Pb	Zn	Fe	S	Ag	Au
Method	FAAS	FAAS	FAAS	LECO	FAAS	FAAS
Units	%	%	%	%	g/t	g/t
ATO-62	0.79	2.45	2.70	3.56	12	1.79
ATO-71	0.97	1.87	2.49	3.75	14	1.64
ATO-97	1.54	1.61	2.95	3.01	10	1.60
ATO-137	0.75	1.30	1.77	2.82	7	1.71
ATO-139	0.80	1.83	3.16	3.55	4	1.01
ATO-149	1.05	2.51	3.73	4.06	5	0.86
ATO-Master	1.05	1.99	2.80	3.47	9	1.45

Mineralogical analysis using QEMSCAN was conducted on the Master Composite and Variability Samples. The primary grind is required to expose mineral surfaces for recovery to each rougher concentrate followed by regrinding to liberate galena (Pb) or sphalerite (Zn) ahead of their respective cleaning circuits. Table 13.20 presents the Modal mineral abundance analysed for the Master Composite and Variability samples.

Table 13.20 – Mineral Abundance in Master Composite and ATO Variability Samples

Mineral Abundance (wt%)	Samples						
	Master Composite	ATO-62	ATO-71	ATO-97	ATO-137	ATO-139	ATO-149
Pyrite	3.83	3.17	3.32	3.34	3.01	3.84	3.72
Chalcopyrite	0.18	0.11	0.13	0.34	0.17	0.19	0.18
Bornite	<0.01	-	-	-	-	-	-
Chalcocite/Covellite	<0.01	-	-	-	-	-	-
Tetrahedrite/Tennantite/Enargite	0.05	0.01	0.13	0.01	0.05	0.01	0.00
Sphalerite	2.91	3.69	3.34	2.64	2.84	3.08	3.18
Galena	1.11	0.67	1.30	1.50	0.93	1.08	0.87
Other Sulphides	-	0.02	0.02	0.03	0.03	0.02	0.02
Quartz	62.9	57.8	52.3	51.1	62.9	64.4	58.2
Plagioclase	0.13	0.05	0.03	0.08	0.03	0.37	0.02
K-Feldspar	0.82	0.32	0.29	0.26	0.24	0.44	0.11
Sericite/Muscovite/Illite	4.87	2.72	3.32	2.33	2.19	4.49	1.36
Biotite/Phlogopite	0.54	1.00	2.64	1.30	1.90	1.41	0.68
Chlorite	12.6	21.9	24.2	11.4	18.2	18.3	25.4
Clays	-	0.17	0.13	0.13	0.11	0.34	0.07
Other Silicates	-	1.00	0.78	1.55	1.42	0.56	1.00
Pyrrhotite	0.02	-	-	-	-	-	-
Arsenopyrite	<0.01	-	-	-	-	-	-
Oxides/Iron Oxides	0.12	0.07	0.09	0.12	0.05	0.10	0.08
Calcite	2.16	2.84	4.58	3.01	2.51	0.86	1.31
Dolomite/Ankerite	6.84	2.60	2.12	20.5	1.77	0.15	3.64
Barite	0.54	1.64	1.15	0.33	1.45	0.25	0.07
Rutile/Anatase	0.18	-	-	-	-	-	-
Apatite	0.06	0.12	0.09	0.06	0.09	0.10	0.09
Other	0.13	0.08	0.04	0.07	0.05	0.03	0.05
Total	**100.00**	**100.00**	**100.00**	**100.00**	**100.00**	**100.00**	**100.00**

13.3.3 Flotation

13.3.3.1 Rougher Kinetic Tests

Rougher Kinetic tests were completed on the Master Composite sample to establish optimal flotation conditions for Pb and Zn recovery. A total of seven rougher kinetic tests were completed on the ATO Master Composite. Various grind sizes were used for the tests conducted with final test and chosen grind size being a P80 of154 µm and a decision to further increase the grind size to 160 µm for the following test. Final rougher test results under selected conditions are presented below in Table 13.21.

Table 13.21 - Rougher Kinetic Test Results

Product	Mass		Assay					Distribution				
	%	grams	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	S (%)	Pb (%)	Zn (%)	Au (%)	Ag (%)	S (%)
Pb Ro Conc 1	3.4	67.5	18.3	8.40	11.1	83	11.8	56.3	14.1	26.0	38.3	12.4
Pb Ro Conc 2	2.0	40	15.5	12.5	13.1	82	14.2	28.3	12.4	18.3	22.4	8.9
Pb Ro Conc 3	1.7	33.2	3.67	10.7	5.36	33	10.2	5.6	8.8	6.2	7.5	5.3
Pb Ro Conc 4	1.1	21.5	0.98	5.70	2.71	12	5.86	1.0	3.0	2.0	1.8	2.0
Zn Ro Conc 1	3.8	75.9	0.33	27.0	3.73	17	19.2	1.1	50.8	9.9	8.8	22.8
Zn Ro Conc 2	2.3	44.9	0.29	3.20	3.84	13	15.1	0.6	3.6	6.0	4.0	10.6
Zn Ro Conc 3	2.4	47.6	0.26	0.89	3.07	10	11.6	0.6	1.1	5.1	3.2	8.6
Zn Ro Conc 4	3.3	66.3	0.27	0.68	2.08	7	7.36	0.8	1.1	4.8	3.2	7.6
Zn Ro Tail	80.1	1,596.6	0.08	0.13	0.39	1	0.88	5.8	5.1	21.7	10.9	21.8
Feed (calc.)	100	1,993.5	1.10	2.02	1.44	7.3	3.21	100.0	100.0	100.0	100.0	100.0
Feed (dir.)			1.09	2.12	1.19	9.0	3.47					

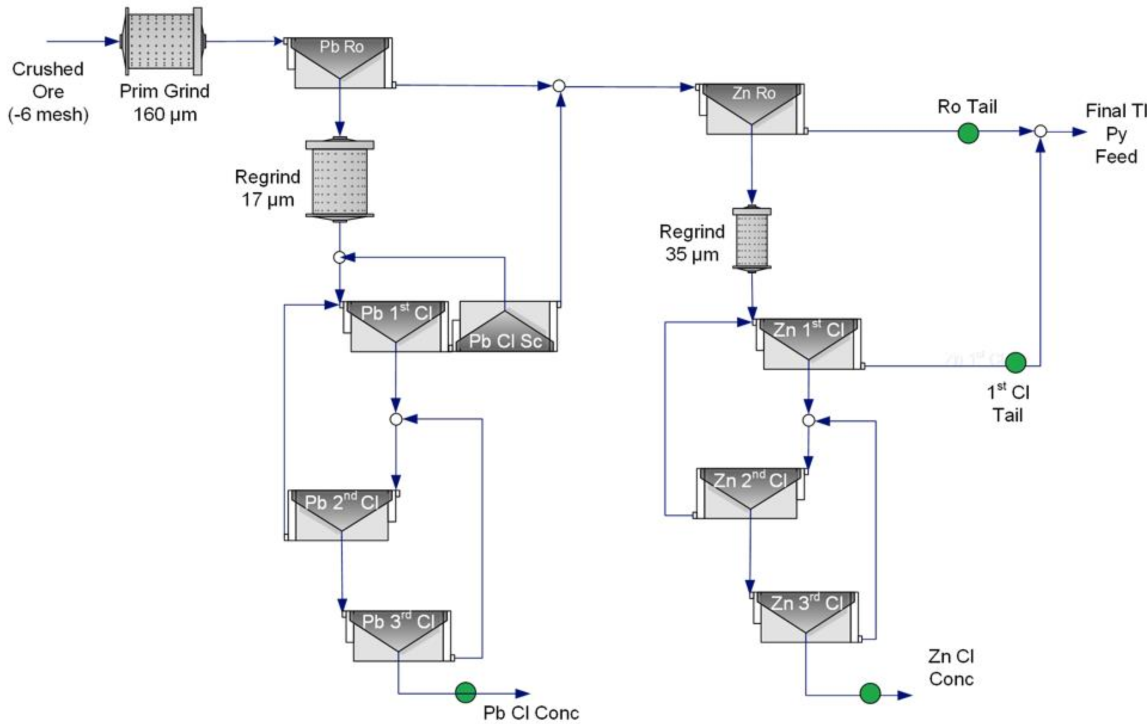
13.3.3.2 Cleaner Tests

Three cleaner tests were completed on the ATO Master Composite to optimise the process cleaning stages. The final cleaner test was conducted at an initial grind size of 160 µm and optimal Pb and Zn regrind sizes of 16 µm and 30 µm, respectively.

13.3.3.3 Preliminary LCTs – Pb-Zn Products

LCT testwork focussed on testing the amenability of the ATO ore to a sequential flotation flowsheet presented below in Figure 13.5.

Figure 13.5 – Selected ATO Phase 2 Flowsheet - Pb-Zn Products



Source: Base Metallurgical Laboratories, 2021

The first LCT was completed based on the previous cleaner flotation test conditions. The test provided reasonably good results which confirmed the potential of the initial flowsheet shown in Figure 13.5. A repeat of this test was completed without the Zn 1st Cleaner Scavenger stage and a reduced Pb 1st cleaner collection stage to mimic plant operational conditions. Feed material for the test was the “Master Composite” sample. The target grind sizes, pulp pH reagent additions, and actual test conditions achieved were as follows:

- Primary grind target 80%, passing 160 µm;
- Pb concentrate regrind target 80% passing 17 µm;
- Zn concentrate regrind target 80% passing 35 µm;
- The first Pb flotation stage included three minutes of conditioning time with the remaining two stages including one minute of conditioning time;
- Pb flotation included one rougher flotation stage followed by regrind, 1st cleaner, 1st cleaner scavenger, 2nd cleaner and 3rd cleaner flotation stages.
- Zn flotation included one rougher flotation stage followed by regrind, 1st cleaner, 2nd cleaner and 3rd cleaner flotation stages.

The test conditions used on the LCT of the flowsheet in Figure 13.5 are presented in Table 13.22.

Table 13.22 - Optimised LCT Test Conditions

Stage	Reagents - g/t							Time (min)		Pulp	
	Soda Ash	ZnCN	3418A	Lime	$CuSO_4$	SIPX	MIBC/H57	Cond.	Froth	pH	Eh
Primary Grind	1000	500						15		8.6	266
Pb Rougher 1			10					1	1	8.6	263
Pb Rougher 2			10				7/0	1	1	8.7	254
Pb Rougher 3			2.5					1	1.5	8.7	249
Pb Regrind (Stirred)		150						5		8.5	255
Pb 1st Cleaner			2.5				28/0	1	5	8.5	260
Pb 1st Cleaner Scavenger			2.5				14/0	1	1	8.5	260
Pb 2nd Cleaner		25					14/0	1	3	8.5	244
Pb 3rd Cleaner		25					7/0	1	3	8.5	247
Zn Feed (Pb RT & Pb 1CT)											
Zn Cond. 1				1090				2		11.0	154
Zn Cond. 2				250	500			5		11.0	119
Zn Rougher 1						15	0/30	1	1	11.0	93
Zn Rougher 2						10	0/10	1	2	10.9	104
Zn Regrind (SS Rod)				200	120			10		10.9	108
Zn 1st Cleaner						5	14/40	1	3	11.0	40
Zn 2nd Cleaner							0/10	1	2	11.5	12
Zn 3rd Cleaner							7/0	1	2	11.5	8
TOTAL	**1,000**	**700**	**28**	**1,540**	**620**	**30**	**0**		**26**		

Results of all LCTs conducted on the Pb-Zn flotation flowsheet, (Figure 13.5), are presented in Table 13.23.

Table 13.23 - Stream Assay and Overall Recoveries of all LCT Pb-Zn Flowsheet

	Product	Mass		Assays					Distribution (%)				
		Dry	%	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	S (%)	Pb	Zn	Au	Ag	S
LCT-19 Master Comp	Pb Cleaner Concentrate	27.4	1.4	59.3	8.5	43.7	357.9	19.6	86.2	6.5	45.2	55.9	9.4
	Zn Cleaner Concentrate	56.25	2.8	1.1	54.6	6.8	49.5	32.8	3.3	85.5	14.4	15.9	32.4
	Zn 1st Cleaner Tail	286.8	14.5	0.2	0.2	1.2	6.0	4.8	2.4	2.0	13.4	9.8	24.3
	Zn Rougher Tail	1608.6	81.3	0.1	0.1	0.4	2.0	1.2	8.1	6.0	27.0	18.4	33.8
	Head (calc.)	1979.05	100.0	0.9	1.8	1.3	8.8	2.9	100.0	100.0	100.0	100.0	100.0
LCT-20 Master Comp	Pb Cleaner Concentrate	24.1	1.2	62.3	5.9	48.6	387.9	18.6	85.1	4.1	41.8	57.2	7.2
	Zn Cleaner Concentrate	56.4	2.8	1.1	52.9	6.4	48.0	33.1	3.4	86.7	12.9	16.6	30.0
	Zn 1st Cleaner Tail	293.15	14.8	0.2	0.3	1.7	7.0	6.6	2.8	2.9	18.0	12.5	31.2
	Zn Rougher Tail	1611.65	81.2	0.1	0.1	0.5	1.4	1.2	8.7	6.3	27.3	13.8	31.5
	Head (calc.)	1985.3	100.0	0.9	1.7	1.4	8.2	3.1	100.0	100.0	100.0	100.0	100.0
LCT-21 ATO-137	Pb Cleaner Concentrate	21.95	1.1	60.3	5.0	118.6	440.6	18.6	91.1	4.0	56.4	62.3	8.0
	Zn Cleaner Concentrate	41.4	2.1	0.6	59.5	10.0	40.9	34.0	1.8	90.3	9.0	10.9	27.6
	Zn 1st Cleaner Tail	196.7	9.9	0.1	0.1	1.6	4.0	4.7	1.1	1.0	6.9	5.1	18.1
	Zn Rougher Tail	1723.75	86.9	0.1	0.1	0.7	1.9	1.4	5.9	4.7	27.7	21.7	46.2
	Head (calc.)	1983.8	100.0	0.7	1.4	2.3	7.8	2.6	100.0	100.0	100.0	100.0	100.0
LCT-24 ATO-139	Pb Cleaner Concentrate	18.7	0.9	71.3	3.4	27.4	252.5	16.7	91.1	1.7	24.8	52.3	4.8
	Zn Cleaner Concentrate	57.6	2.9	1.0	59.7	5.4	26.0	32.0	4.0	93.2	15.2	16.6	28.5
	Zn 1st Cleaner Tail	143.45	7.2	0.1	0.4	3.0	6.0	10.7	1.0	1.5	20.6	9.6	23.7
	Zn Rougher Tail	1766.95	88.9	0.0	0.1	0.5	1.1	1.6	3.9	3.6	39.4	21.5	43.0
	Head (calc.)	1986.7	100.0	0.7	1.9	1.0	4.5	3.2	100.0	100.0	100.0	100.0	100.0

	Product	Mass		Assays					Distribution (%)				
		Dry	%	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	S (%)	Pb	Zn	Au	Ag	S
LCT-25 ATO-71	Pb Cleaner Concentrate	22.8	1.1	64.1	5.5	39.1	583.1	17.9	85.7	3.6	31.9	63.5	5.9
	Zn Cleaner Concentrate	57.35	2.9	1.9	56.7	4.9	39.7	32.7	6.5	91.5	10.0	10.9	27.1
	Zn 1st Cleaner Tail	192.9	9.7	0.2	0.3	2.5	3.8	6.9	2.3	1.8	17.1	3.5	19.3
	Zn Rougher Tail	1715.65	86.3	0.1	0.1	0.7	2.7	1.9	5.5	3.1	41.1	22.1	47.7
	Head (calc.)	1988.7	100.0	0.9	1.8	1.4	10.5	3.5	100.0	100.0	100.0	100.0	100.0
LCT-26 ATO-62	Pb Cleaner Concentrate	17.05	0.9	65.1	6.5	88.8	539.1	17.7	84.8	2.4	51.2	50.5	4.7
	Zn Cleaner Concentrate	72.75	3.7	1.7	55.9	8.5	76.0	31.8	9.2	87.8	21.0	30.4	36.0
	Zn 1st Cleaner Tail	281.1	14.2	0.1	0.4	0.9	4.0	4.6	1.7	2.5	8.7	6.2	20.3
	Zn Rougher Tail	1609.4	81.3	0.0	0.2	0.4	1.5	1.6	4.3	7.2	19.1	12.9	39.0
	Head (calc.)	1980.3	100.0	0.7	2.3	1.5	9.1	3.2	100.0	100.0	100.0	100.0	100.0
LCT-27 ATO-149	Pb Cleaner Concentrate	24.8	1.2	66.2	7.8	27.8	199.0	18.4	87.7	4.2	40.2	49.3	5.6
	Zn Cleaner Concentrate	72	3.6	1.1	56.4	3.1	24.2	32.5	4.1	88.8	12.8	17.4	28.7
	Zn 1st Cleaner Tail	262.5	13.2	0.1	0.3	0.6	2.5	4.1	1.8	1.7	9.6	6.5	13.2
	Zn Rougher Tail	1629.1	81.9	0.1	0.1	0.4	1.6	2.6	6.4	5.3	37.5	26.8	52.5
	Head (calc.)	1988.4	100.0	0.9	2.3	0.9	5.0	4.1	100.0	100.0	100.0	100.0	100.0
LCT-29 ATO-97	Pb Cleaner Concentrate	32.35	1.6	65.3	6.6	26.8	310.3	16.9	73.5	6.9	32.5	49.2	10.2
	Zn Cleaner Concentrate	45	2.3	6.3	51.4	9.6	89.6	30.3	9.9	74.4	16.1	19.8	25.5
	Zn 1st Cleaner Tail	120.05	6.0	0.5	0.7	2.1	10.2	5.5	2.3	2.8	9.5	6.0	12.4
	Zn Rougher Tail	1791	90.1	0.2	0.3	0.6	2.9	1.6	14.3	15.9	41.9	25.0	51.9
	Head (calc.)	1988.4	100.0	1.4	1.6	1.3	10.2	2.7	100.0	100.0	100.0	100.0	100.0

	Product	Mass		Assays					Distribution (%)				
		Dry	%	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	S (%)	Pb	Zn	Au	Ag	S
LCT-30 Master Comp	Pb Cleaner Concentrate	26.35	1.3	62.9	5.2	48.5	339.1	17.9	86.9	3.8	43.9	56.4	7.4
	Zn Cleaner Concentrate	59	3.0	1.2	53.0	6.4	42.0	33.2	3.6	87.4	13.0	15.6	30.7
	Zn 1st Cleaner Tail	441	22.2	0.1	0.3	1.4	5.4	5.6	3.2	3.5	21.3	15.1	38.6
	Zn Rougher Tail	1459.8	73.5	0.1	0.1	0.4	1.4	1.0	6.3	5.4	21.8	12.9	23.3
	Head (calc.)	1986.15	100.0	1.0	1.8	1.5	7.9	3.2	100.0	100.0	100.0	100.0	100.0
LCT-35 Master Comp	Pb Cleaner Concentrate	24.85	1.3	58.9	7.4	45.9	358.6	19.1	83.8	5.1	43.6	54.9	7.8
	Zn Cleaner Concentrate	58.2	3.0	1.4	54.4	6.0	51.2	32.0	4.6	86.8	13.4	18.4	30.4
	Zn Rougher Tail	1842.55	95.7	0.1	0.2	0.6	2.4	2.1	11.6	8.1	43.0	26.7	61.8
	Head (calc.)	1925.6	100.0	0.9	1.9	1.4	8.6	3.3	100.0	100.0	100.0	100.0	100.0

Note: Figures may not add due to rounding

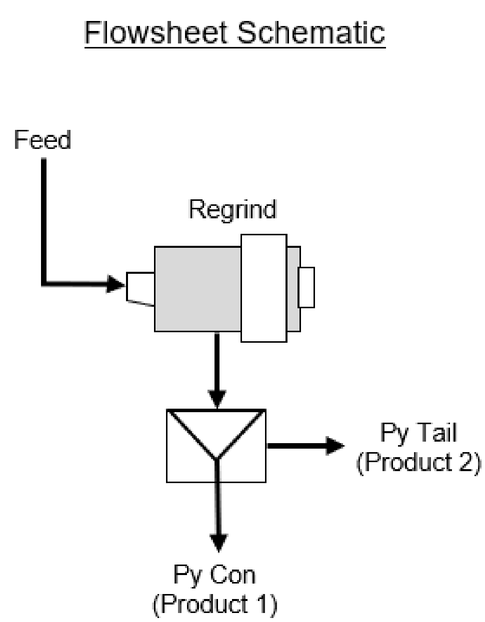
13.3.3.4 Pyrite Flotation Circuit

The flowsheet for Pb and Zn products shown in Figure 13.5 was modified to add a pyrite flotation stage after Zn flotation. The pyrite flotation will be fed with the Zn 1st cleaner and Zn rougher tails and will produce a pyrite concentrate containing saleable levels of gold and silver. Rougher kinetic tests were completed on the LCT Zn 1st cleaner and on the rougher tails. Both tests were conducted at a grind size of 31 µm, with both returning good results.

Following the rougher tests, six cleaner tests were completed on each of the Variability Samples. Regrind sizes averaged 10 and 20 µm for Pb and Zn respectively.

Once the rougher and cleaner tests were optimised, pyrite flotation circuit tests were conducted. The following flowsheet was followed for pyrite flotation tests.

Figure 13.6 - Flowsheet Schematic for Pyrite Flotation Circuits



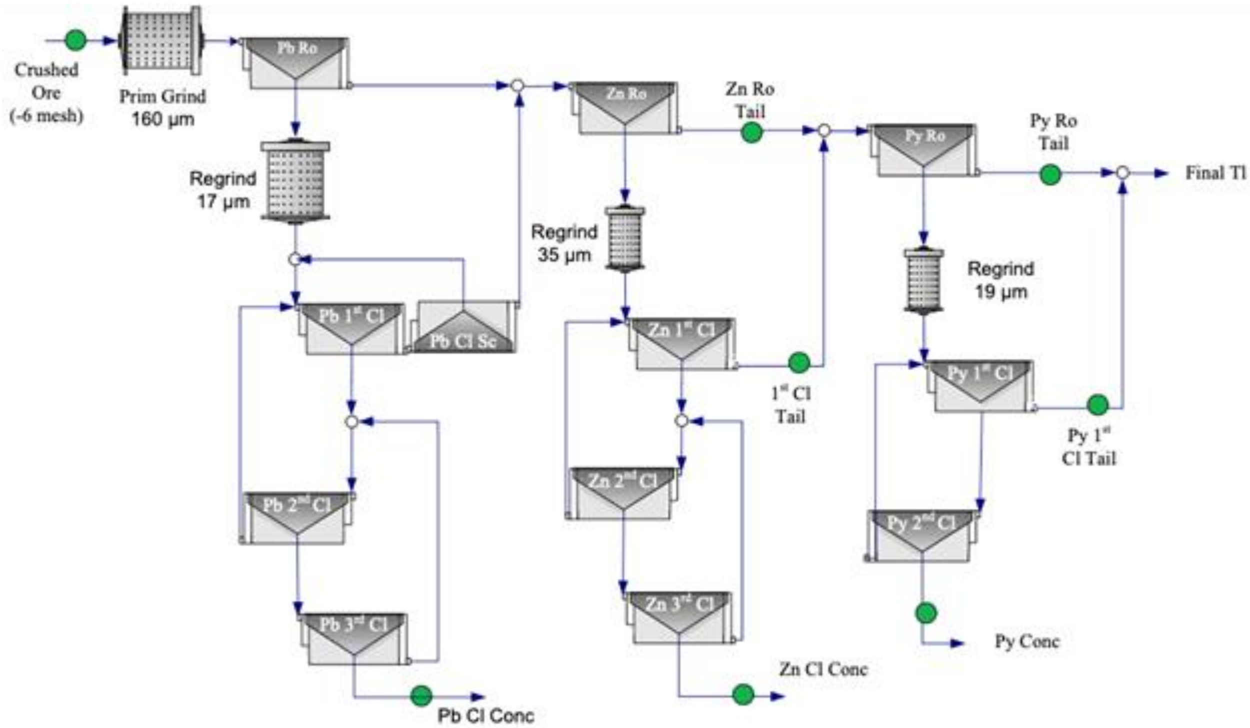
Base Metallurgical Laboratories, June 2021

A total of 12 pyrite LCT tests were completed. The addition of the pyrite flotation circuit increased combined flotation Au recovery (Pb + Zn + Pyrite concentrates) levels to between 62.5% and 88.9%.

13.3.3.5 Final LCTs – Pb-Zn-Pyrite Concentrate Products

LCT testwork focussed on testing the amenability of the ATO ore based on the flowsheet presented in Figure 1.1, where pyrite flotation was added to obtain separate Pb, Zn, and pyrite concentrates.

Figure 13.7 – Updated ATO Phase 2 Flowsheet - Pb-Zn-Pyrite Concentrate Products



Base Metallurgical Laboratories, June 2021

The additional Pyrite flotation section included one rougher flotation stage followed by regrind, 1st cleaner and 2nd cleaner flotation stages. The optimised grind size after pyrite regrind was estimated at between a P80 of 17 µm and 19 µm.

The test conditions used on the LCT for the Master Composite are presented in Table 13.24. Seven LCT tests were conducted on the Pb-Zn-Pyrite Concentrate Flowsheet, one for every variability sample and one for the master composite shown in Table 13.18. Results of all these LCTs are presented in Table 13.25.

Results in Table 13.25 showed Pb and Zn concentrate grades averaging 62.7% and 56.0% respectively, with Pb recoveries from 63.4% to 87.4% and Zn recoveries from 73.7% to 93.6%. Total gold recovery considering Pb, Zn and Pyrite concentrates was 70.3% to 82.4%, while total silver recovery for the same concentrates was 68.6% to 86.0%.

Table 13.24 - Optimised LCT Test Conditions

Stage	Reagents (g/t)										Time (min)		Pulp	
	Soda Ash	ZnCN	3418A	Lime	$CuSO_4$	SIPX	H57	MIBC	CMC	PAX	Cond.	Froth	pH	Eh
Primary Grind	1,000	500									15		8.5	252
Pb Rougher 1			10					21			1	1	8.6	184
Pb Rougher 2			10								1	1	8.6	192
Pb Rougher 3			2.5								1	1.5	8.5	198
Pb Regrind (stirred)		150									5		8.9	244
Pb 1st Cleaner			2.5					14			1	5	8.9	240
Pb 1st Cleaner Scavenger			2.5					7			1	1	8.4	260
Pb 2nd Cleaner		25									1	3	8.4	257
Pb 3rd Cleaner		25						7			1	3	8.3	254
Zn Feed														
(Pb RT & Pb 1CT)													8.9	132
Zn Cond. 1				1050							2		11.0	80
Zn Cond. 2				190	500						5		11.0	61
Zn Rougher 1				-		15	30				1	1	11.0	42
Zn Rougher 2						10	10				1	2	11.0	38
Zn Regrind (SS Rod)				200	120						10		10.8	53
Zn 1st Cleaner				150		5	30				1	3	11.5	4
Zn 2nd Cleaner							20				1	2	11.5	13
Zn 3rd Cleaner							20				1	2	11.5	35
Py Ro (Zn RT+Cl)								14		75		8	10.6	11
Py Rg'd (~19 µm)											22		8.6	237
Py 1st Cleaner									100			6	8.6	221
Py 2nd Cleaner												5	8.5	201
Total	**1,000**	**700**	**28**	**1,590**	**620**	**30**	**90**				**49**	**24**		

Table 13.25 – Stream Assay and Overall Recoveries of all LCT Pb-Zn-Pyrite Flowsheet

	Product	Weight		Assays					Distribution (%)				
		Dry	%	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	S (%)	Pb	Zn	Au	Ag	S
LCT-37 ATO-137	Pb Cl Conc	16.5	0.8	61.3	2.3	112.5	290.4	18.7	69.6	1.4	40.0	40.0	5.7
	Zn Cl Conc	36.4	1.9	6.8	54.1	17.1	53.0	31.8	17.1	73.7	13.5	16.1	21.5
	Py Conc	46.7	2.4	2.1	12.2	18.6	32.0	42.6	6.6	21.3	18.8	12.5	36.8
	Py 1st Cl Tail	218.4	11.2	0.1	0.1	0.7	2.2	1.2	1.0	0.8	3.1	4.1	5.0
	Py Ro Tail	1636.2	83.7	0.0	0.0	0.7	2.0	1.0	5.6	2.7	24.7	27.3	31.0
	Head (calc.)	1954.1	100.0	0.7	1.4	2.4	6.1	2.8	100.0	100.0	100.0	100.0	100.0
LCT-38 ATO-139	Pb Cl Conc	17.2	0.9	69.1	3.8	26.9	268.1	17.7	83.0	1.8	20.5	54.8	4.6
	Zn Cl Conc	59.1	3.0	2.8	58.9	6.2	30.1	33.1	11.6	93.6	16.4	21.2	29.6
	Pyrite Conc	63.6	3.2	0.7	1.0	14.3	10.2	48.6	3.0	1.7	40.3	7.7	46.7
	Zn 1st Cl Tail	169.4	8.5	0.0	0.1	0.6	1.6	1.6	0.6	0.4	4.2	3.2	4.2
	Zn Ro Tail	1693.6	84.6	0.0	0.1	0.2	0.7	0.6	1.8	2.5	18.5	13.2	15.0
	Head (calc.)	2002.8	100.0	0.7	1.9	1.1	4.2	3.3	100.0	100.0	100.0	100.0	100.0
LCT-39 ATO-71	Pb Cl Conc	25.6	1.3	58.8	8.5	43.9	576.3	19.6	87.4	5.9	37.3	62.6	7.4
	Zn Cl Conc	54.6	2.7	1.3	61.2	5.2	44.8	33.4	4.1	89.7	9.3	10.4	26.7
	Pyrite Conc	59.5	3.0	0.6	0.8	12.4	46.9	40.9	1.9	1.2	24.4	11.8	35.6
	Zn 1st Cl Tail	173.0	8.7	0.1	0.1	1.3	4.1	3.0	1.2	0.5	7.6	3.0	7.5
	Zn Ro Tail	1685.4	84.4	0.1	0.1	0.4	1.7	0.9	5.4	2.7	21.4	12.2	22.8
	Head (calc.)	1998.1	100.0	0.9	1.9	1.5	11.8	3.4	100.0	100.0	100.0	100.0	100.0

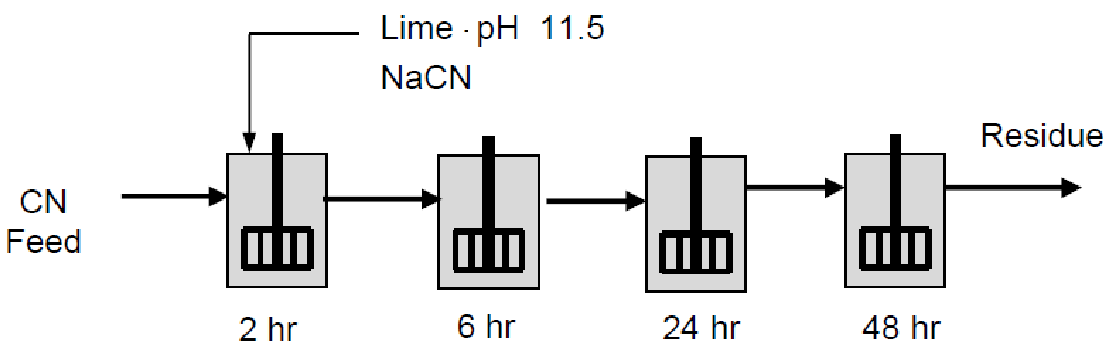
	Product	Weight		Assays					Distribution (%)				
		Dry	%	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	S (%)	Pb	Zn	Au	Ag	S
LCT-40 ATO-62	Pb Cl Conc	17.4	0.9	62.4	7.5	80.1	535.7	17.0	83.7	2.8	50.1	51.6	4.4
	Zn Cl Conc	71.4	3.6	1.6	56.9	7.5	74.2	32.5	8.6	85.9	19.2	29.3	34.2
	Pyrite Conc	45.2	2.3	0.4	3.1	6.8	20.8	46.5	1.5	2.9	11.0	5.2	31.0
	Zn 1st Cl Tail	216.9	10.8	0.1	0.2	0.5	2.8	1.9	1.1	1.1	3.6	3.4	6.0
	Zn Ro Tail	1651.4	82.5	0.0	0.2	0.3	1.2	1.0	5.1	7.4	16.1	10.5	24.4
	Head (calc.)	2002.2	100.0	0.6	2.4	1.4	9.0	3.4	100.0	100.0	100.0	100.0	100.0
LCT-41 ATO-149	Pb Cl Conc	16.3	0.8	62.7	5.5	36.0	231.5	16.2	63.4	1.9	34.6	38.3	3.5
	Zn Cl Conc	78.5	3.8	5.4	55.3	3.3	42.9	32.1	26.5	90.8	15.2	34.2	33.1
	Pyrite Conc	59.3	2.9	0.8	2.0	5.9	9.3	42.8	3.0	2.5	20.6	5.6	33.3
	Zn 1st Cl Tail	337.4	16.4	0.1	0.1	0.2	1.5	0.9	1.8	0.8	4.2	5.1	4.1
	Zn Ro Tail	1569.1	76.2	0.1	0.1	0.3	1.1	1.3	5.4	3.9	25.4	16.7	26.1
	Head (calc.)	2060.5	100.0	0.8	2.4	0.8	4.9	3.8	100.0	100.0	100.0	100.0	100.0
LCT-42 ATO-97	Pb Cl Conc	25.9	1.3	66.1	2.7	32.9	341.0	16.0	64.8	2.4	35.8	44.4	8.5
	Zn Cl Conc	46.2	2.3	8.7	49.6	7.1	93.7	28.9	15.2	76.9	13.7	21.8	27.4
	Pyrite Conc	42.1	2.1	2.6	3.0	15.9	50.5	41.2	4.2	4.3	28.1	10.7	35.6
	Zn 1st Cl Tail	237.0	11.9	0.2	0.3	0.9	5.8	2.5	2.1	2.1	8.5	6.9	12.2
	Zn Ro Tail	1644.7	82.4	0.2	0.3	0.2	1.9	0.5	13.7	14.3	13.8	16.1	16.2
	Head (calc.)	1995.7	100.0	1.3	1.5	1.2	9.9	2.4	100.0	100.0	100.0	100.0	100.0
LCT-43 Master Comp	Pb Cl Conc	25.3	1.3	58.2	7.8	39.6	348.4	19.3	83.5	5.3	43.1	54.3	7.9
	Zn Cl Conc	55.6	2.8	1.5	57.0	6.1	53.2	31.4	4.8	85.8	14.6	18.2	28.4
	Pyrite Conc	66.8	3.4	0.7	1.4	8.6	28.7	33.2	2.7	2.5	24.7	11.8	36.1
	Zn 1st Cl Tail	274.4	13.8	0.1	0.1	0.2	3.5	2.0	1.6	0.9	2.1	6.0	9.0
	Zn Ro Tail	1560.2	78.7	0.1	0.1	0.2	1.0	0.7	7.5	5.5	15.5	9.6	18.6
	Head (calc.)	1982.2	100.0	0.9	1.9	1.2	8.2	3.1	100.0	100.0	100.0	100.0	100.0

13.3.4 CYANIDATION OF PB-ZN PRODUCTS

13.3.4.1 Cyanidation of Pb-Zn Tails

To evaluate Au recovery using conventional cyanidation, tests were completed on the Zn rougher tails and Zn first cleaner tails produced from LCT 19. Both tests were completed based on the flowsheet shown in Figure 13.8.

Figure 13.8 - Flowsheet Schematic for Cyanidation of Pb-Zn Products



Base Metallurgical Laboratories, March 2021

Test conditions and results on the Zn rougher tails are shown in Table 13.26 and Table 13.27 respectively.

Table 13.26 – Cyanidation Test Conditions on Zn Rougher Tails LCT 19

Parameter	Time	Added		Residual	Consumed	pH		Dissolved
	Cum (h)	NaCN (g)	Lime	NaCN (g)	NaCN (g)	Measured	Adjusted	O_2 (mg/L)
Natural	-	-	-	-	-	7.9	-	8.3
Pre-Ox	2		1.81			7.9	11.5	8.3
Leach 1	0	3.75	0.90	-	-	11.0	11.5	>20
Leach 2	2	0.21	-	3.84	-0.09	11.7	-	>20
Leach 3	6	0.06	-	3.69	0.36	11.7	-	>20
Leach 4	24	0.00	-	3.75	0.00	12.2	-	>20
Leach 5	48	-	-	3.63	0.12	12.6	-	>20
Total	**48**	**4.02**	**2.71**	**3.63**	**0.39**	**-**	**-**	**-**

Table 13.27 – Cyanidation Results of Zn Rougher Tails LCT 19

Product	Cumulative Time (h)	Vol / Mass	Assay Au (g/t)	Distribution Au (%)
Cyanide Liquor	2	1,500 mL	0.05	19.5
Cyanide Liquor	6	1,500 mL	0.06	23.5
Cyanide Liquor	24	1,500 mL	0.07	27.5
Cyanide Liquor	48	1,500 mL	0.07	27.7
Cyanidation Residue	-	995 g	0.28	72.3
Calculated Feed		**995 g**	**0.39**	**100.0**

Test conditions and results on Zn first cleaner tails are shown in Table 13.28 and Table 13.29 respectively.

Table 13.28 – Cyanidation Test Conditions on Zn First Cleaner Tails LCT 19

Parameter	Time	Added		Residual	Consumed	pH		Dissolved
	Cum (h)	NaCN (g)	Lime	NaCN (g)	NaCN (g)	Measured	Adjusted	O_2 (mg/L)
Natural	-	-	-	-	-	7.8	-	8.0
Pre-Ox	2		1.78			7.8	11.5	8.0
Leach 1	0	0.75	-	-	-	11.7	-	>20
Leach 2	2	0.25	-	0.61	0.14	11.8	-	>20
Leach 3	6	0.20	-	0.80	0.06	11.8	-	>20
Leach 4	24	0.12	-	0.88	0.12	12.3	-	>20
Leach 5	48	-	-	0.90	0.10	12.7	-	>20
Total	**48**	**1.32**	**1.78**	**0.90**	**0.42**	**-**	**-**	**-**

Table 13.29 – Cyanidation Results of Zn First Cleaner Tails LCT 19

Product	Cumulative Time (h)	Vol / Mass	Assay Au (g/t)	Distribution Au (%)
Cyanide Liquor	2	400 mL	0.15	25.2
Cyanide Liquor	6	400 mL	0.11	19.7
Cyanide Liquor	24	400 mL	0.11	20.7
Cyanide Liquor	48	400 mL	0.11	21.6
Cyanidation Residue	-	196 g	0.96	78.4
Calculated Feed		**196 g**	**1.22**	**100.0**

The cyanidation tests completed on the Pb-Zn tails returned poor recovery results and therefore further testwork was not conducted. Thus, an alternative to increasing gold and silver recoveries from processing of the Pb-Zn flotation tails products was considered. The option of an additional pyrite flotation circuit to create a saleable pyrite concentrate containing gold and silver was evaluated.

13.3.4.2 Cyanidation of Pb-Zn- Pyrite Rougher Tails

To further increase Au recovery, additional cyanide testing was performed on the pyrite flotation tails. The test followed the same schematic as shown above in Figure 13.8.

Test conditions and results on the pyrite tails are shown in Table 13.30 and Table 13.31 respectively.

Table 13.30 – Cyanidation Test Conditions on Pyrite Rougher Tails

Parameter	Time	Added		Residual	Consumed	pH		Dissolved
	Cum (h)	NaCN (g)	Lime	NaCN (g)	NaCN (g)	Measured	Adjusted	O_2 (mg/L)
Natural	-	-	-	-	-	9.6	11.5	7.5
Leach 1	0	1.50	0.55	-	-	9.6	11.5	7.5
Leach 2	2	0.15	-	1.35	0.15	11.7	-	>20
Leach 3	6	0.21	-	1.29	0.21	11.5	-	>20
Leach 4	24	0.15	-	1.35	0.15	11.6	-	>20
Leach 5	48	-	-	1.50	-	11.3	-	>20
Leach 6	72	-	-	1.40	0.10	10.9	-	>20
Total	**72**	**2.01**	**0.55**	**1.40**	**0.61**	**-**	**-**	**-**

Table 13.31 – Cyanidation Results of Pyrite Rougher Tails

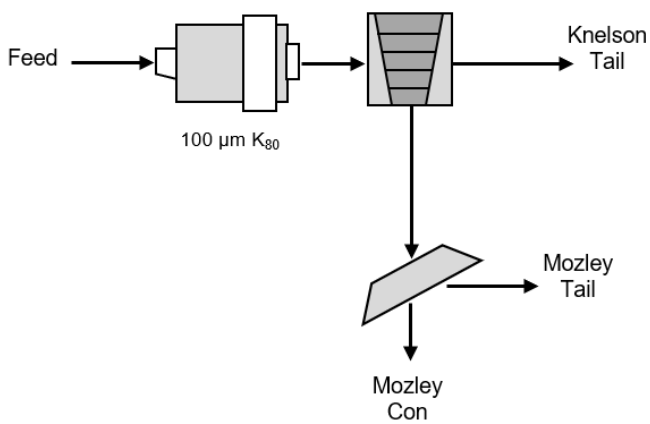
Product	Cumulative	Vol /	Assay	Distribution
	Time (h)	Mass	Au (g/t)	Au (%)
Cyanide Liquor	2	1500 mL	0.08	29.8
Cyanide Liquor	6	1500 mL	0.09	33.9
Cyanide Liquor	24	1500 mL	0.11	41.8
Cyanide Liquor	48	1500 mL	0.11	42.4
Cyanide Liquor	72	1500 mL	0.11	42.9
Cyanidation Residue	-	998 g	0.23	57.1
Calculated Feed		998 g	0.40	100.0

Although this additional step of leaching on the pyrite tail products showed a small recovery of Au from the pyrite tails, the leaching circuit was not included in the final design of the process as it was not deemed to be a financially viable option.

13.3.5 GRAVITY TESTS

Following the rougher tests, a gravity test with a combination of Knelson and Mozley gravity concentration was completed to determine the gravity separation performance. The flowsheet utilized to conduct the gravity test is presented in Figure 13.9 below.

Figure 13.9 - Gravity Test Flowsheet



Base Metallurgical Laboratories, February 2021

The test returned poor results, which can be seen in Table 13.32 with only a 5% Au recovery to final gravity concentrate. It was therefore determined that gravity separation would not be implemented into the final process flowsheet.

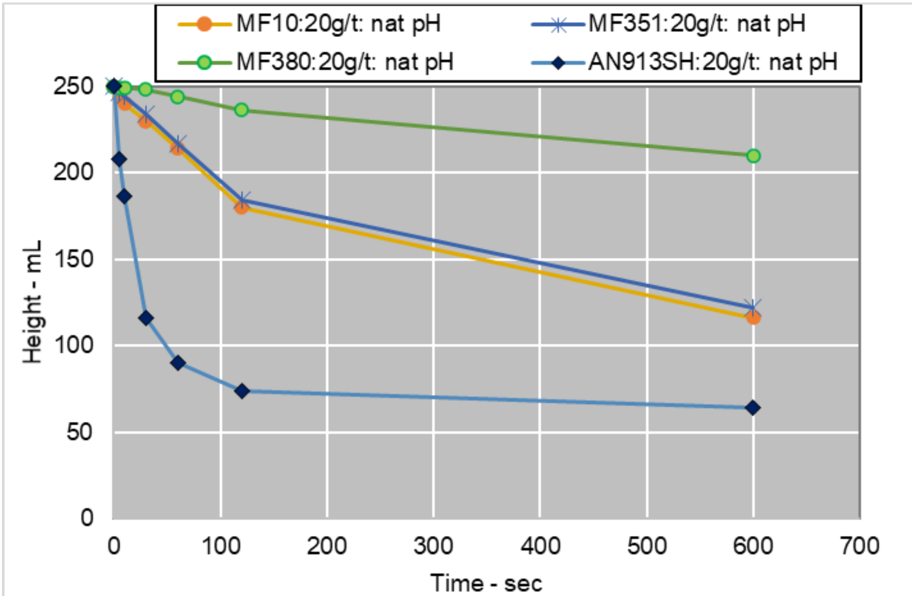
Table 13.32 - Gravity Test Results

Product	Mass		Assay (g/t)		Distribution (%)	
	%	g	Au	Pb	Au	Pb
Moz Conc	0.1	6.0	130	45.0	5.8	2.7
Knelson Conc	1.0	100.3	20.0	24.0	15.0	23.8
Zn Ro Tail	99.0	9,899.3	1.15	0.78	85.0	76.2
Feed (calc.)	99.0	9,899.3	1.15	0.78	0	0

13.3.6 FLOCCULANT SCREENING

A flocculant screening test was conducted to determine a suitable flocculant to be added to the process to aid with solids settling. The test compared four different types of flocculant (two anionic and two non-ionic / cationic). The same dosage of 20 g/t was used for each of the flocculants and the test was conducted at natural pH. Results are presented in Figure 13.10.

Figure 13.10 – Flocculant Scoping Test Results



Base Metallurgical Laboratories, April 2021

Results were compared for slurry samples at a constant volume of 250 mL and feed densities of 15% solids; scoping tests suggested AN913SH followed by MF10 or MF351 were the most promising and formed the basis for static settling testwork. Flocculant AN913SH was selected and included in the operational expenses estimation.

13.4 Recovery Estimates

13.4.1 Lead (Pb) Recovery

After analysing all of the flotation results (Table 13.25), a Pb recovery relationship could not be determined and therefore a fixed value of 82.5% was used. This was the average of all the lead recovery results from the LCTs conducted. This fixed value was estimated from the average between the master composite and variability samples. For the variability samples the average was calculated by using the masses of samples based on the master composite mass splits. The fixed Pb recovery value is shown as follows:

$$Pb\ Recovery\ \% = 82.5\ ;\ Fixed\ Value$$

The same methodology was applied to the Au recovery in the Pb concentrate. The fixed Au recovery value is shown as follows:

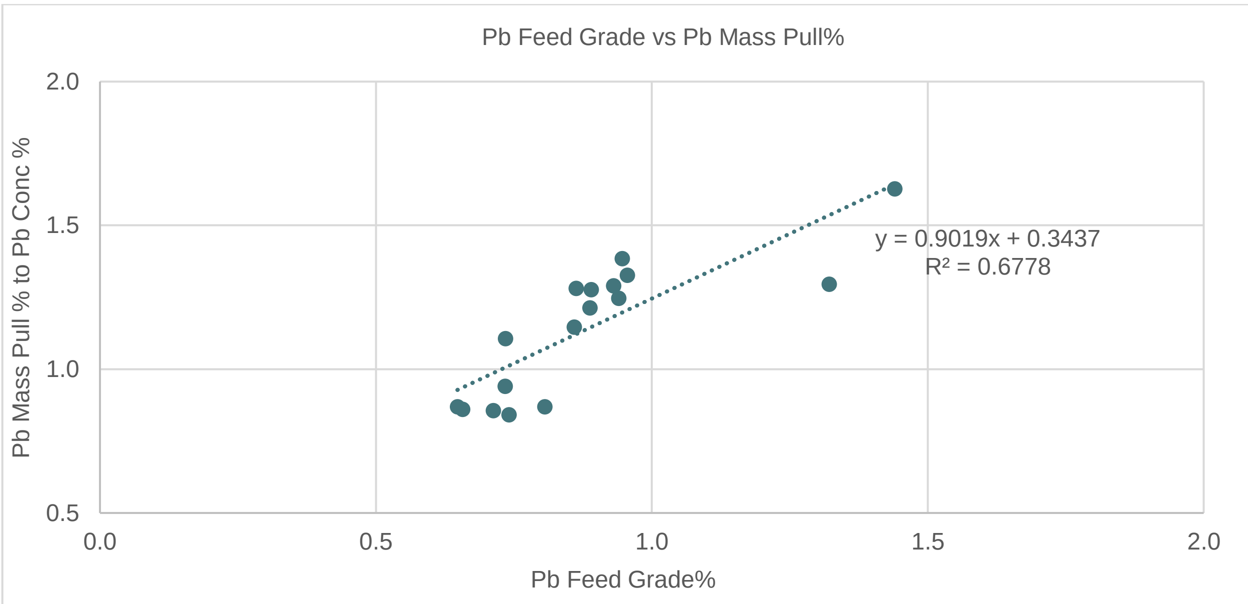
$$Pb\ Conc\ Gold\ Rec\ \% =\ 41.2\ ; Fixed\ Value$$

The Pb mass pull % to the lead concentrate is based on the relationship shown below.

$$Pb\ Mass\ Pull\ \% = 0.9019\ x\ Pb\ Feed\ Grade + 0.3437$$

Figure 13.11 illustrates the relationship between Pb feed grade versus Pb mass pull:

Figure 13.11 – Pb Feed Grade vs. Pb Mass Pull



Base Metallurgical Laboratories, June2021

13.4.2 ZINC (ZN) RECOVERY

A Zn recovery relationship was also unable to be determined and therefore a fixed value was used. This fixed value was estimated from the average between the master composite and variability samples. For the variability samples, the average was calculated by using the masses of samples based on the master composite mass splits. A fixed Pb recovery value was determined as follows:

$$Zn\ Recovery\ \% = 85.9\ ; Fixed\ Value$$

The same methodology was applied to the Au recovery in the Zn concentrate. The fixed Au recovery value is as follows:

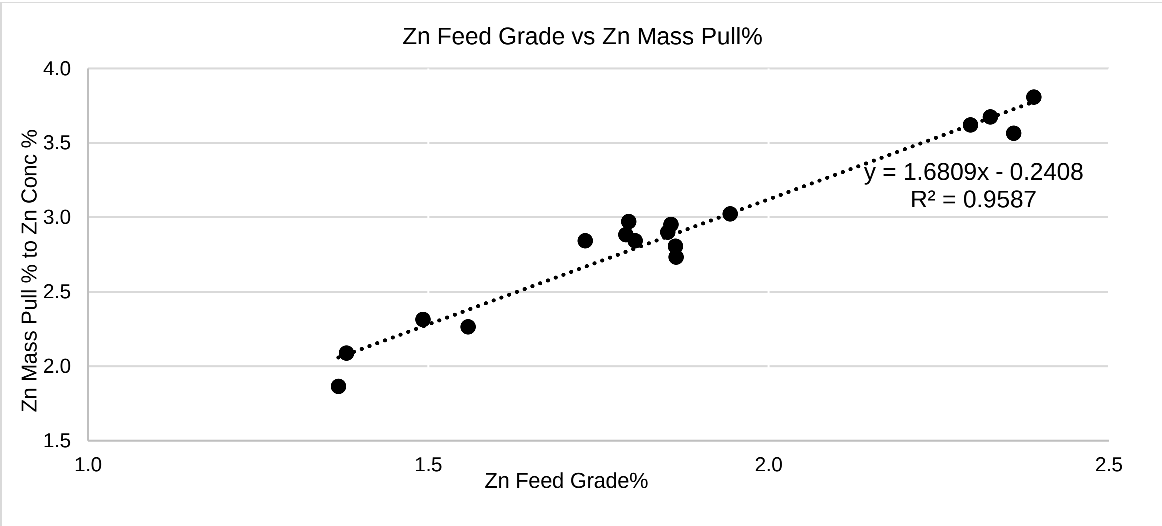
$$Zn\ Conc\ Gold\ Rec\ \% =\ 14.1\ ; Fixed\ Value$$

The Zn mass pull % to the zinc flotation is based on the relationship below.

$$Zn\ Mass\ Pull\ \% = 1.6809\ x\ Zn\ Fd\ Grade - 0.2408$$

Figure 13.12 illustrates the relationship between Zn feed grade and Zn mass pull % to Zn concentrate.

Figure 13.12 – Zn Feed Grade vs. Zn Mass Pull



Base Metallurgical Laboratories, June2021

13.4.3 Pyrite Recovery

Regarding Au and Ag recoveries in the pyrite concentrate, average values between the variability and master composite samples were used. These are shown as follows:

$$Au\ Recovery\ in\ Pyrite\ Concentrate\ \% = 23.9\ ;\ Fixed\ Value$$

$$Ag\ Recovery\ in\ Pyrite\ Concentrate\ \% = 8.8\ ;\ Fixed\ Value$$

Regarding the Au grade in the pyrite concentrate, a similar methodology was applied, averaged between the variability and master composite samples was used, and is shown as follows:

$$Pyrite\ Conc\ Au\ Grade\ (g/t) = 10.5\ ;\ Fixed\ Value$$

13.4.4 SUMMARY OF GOLD, SILVER, LEAD AND ZINC RECOVERIES

Table 13.33 summarises the results of the weighted variability samples and master composite samples.

Table 13.33 - Consolidated Results for Variability and Master Composite Samples

Item	Variability[1]	Master
Pb Head Grade %	0.90	0.92
Zn Head Grade %	1.86	1.83
S Head Grade %	3.18	3.11
Au head Grade g/t	1.43	1.35
Ag head Grade g/t	8.20	8.36
Ag/Au Ratio	5.75	6.18
Pb Concentrate Pb Recovery %	79.8	85.1
Pb Concentrate Pb Grade %	64.3	60.3
Pb Concentrate Au Recovery %	38.8	43.5
Pb Concentrate Au Grade g/t	56.5	45.3
Pb Concentrate Ag Recovery %	51.4	55.8
Pb Concentrate Ag Grade g/t	393.7	358.4
Pb Concentrate Ag/Au Ratio	7.7	6.4
Pb Concentrate Mass Pull %	1.10	1.30
Zn Concentrate Zn Recovery %	85.3	86.4
Zn Concentrate Zn Grade %	55.8	54.4
Zn Concentrate Au Recovery %	14.6	13.6
Zn Concentrate Au Grade g/t	7.6	6.3
Zn Concentrate Ag Recovery %	20.4	16.9
Zn Concentrate Ag Grade g/t	58.4	48.8
Zn Concentrate Ag/Au Ratio	7.7	7.7
Zn Concentrate Mass Pull %	2.86	2.90
Py Concentrate Au Recovery %	23.1	24.7
Py Concentrate Au Grade g/t	12.4	8.6
Py Concentrate Ag Recovery %	9.0	8.6
Py Concentrate Ag Grade g/t	30.6	28.7
Py Concentrate Ag/Au Ratio	2.5	3.3
Py Concentrate Mass Pull %	2.55	3.37

Item	Variability[1]	Master
Total Concentrate Mass Pull %	6.5	7.6
Total Au Recovery %	76.6	81.8
Total Ag Recovery %	80.7	81.3

1 Results have been estimated as a weighted average from all variability samples.

As shown in Table 13.33, the average of the variability sample LCT results reconciles well with the Master Composite LCT result.

13.5 Metallurgical Variability

The metallurgical testwork completed to date is based on samples which adequately represent the variability of the ATO deposit; however, the selection of the samples was made prior the establishment of the latest mine plan.

Mineralogical analysis of the various composite and variability samples has shown that the ATO deposit is reasonably homogenous with respect to mineralogy. The exception is sample ATO-97 which showed high contents of dolomite which appear to impact detrimentally on flotation performance.

13.6 Deleterious Elements

Pb, Zn, and Pyrite concentrates will be subject to penalty conditions should significant grades of zinc, lead, mercury, antimony, bismuth and arsenic be present in high levels in the concentrates. Section 19 explores the impact of these elements which are present in the concentrates. The concentrates produced are shown to be very clean concentrates with no presence of detrimental elements leading to penalties.

14 MINERAL RESOURCE ESTIMATE

The information in this section is largely drawn and/or summarised from the Report available on SEDAR entitled: "Altan Tsagaan Ovoo Project (ATO), 2021 Mineral Resources, Technical Report (Amended NI 43-101)", prepared by GeoRes, with an effective date March 30, 2021, issued on March 30, 2021 and amended on June 9, 2021. Further details as documented therein remain correct and valid.

The Section discusses and details the methodology and results of the Consultant/QP's 2020-2021 estimation of gold and related precious and base metal Mineral Resources in ATO's four deposits for this Report. These new Resources supersede the previous ones published for the three deposits in the 2017 NI 43-101 Report (Pipe 1, 2 and 4) and provide the initial Resources on the fourth (Mungu).

Resource estimation is described in terms of:

- Introductory statements and certifications – qualifying the Author QP, reporting codes and source of data.
- Background – emphasizing that this is a re-estimate off the Pipe 1, 2, and 4 deposits and an initial estimate of the Mungu deposit.
- Raw data supplied – listing the old and new raw data used in the estimate.
- Software used.
- Methodology – the data manipulation, analysis, interpretation, modelling and reporting process.
- Data pre-processing – the steps in treating the data from raw to fully modelled.
- Drill hole databasing.
- Geological interpretation – of deposit shapes and oxidation levels.
- Wire-frame modelling – of individual deposits.
- Surface modelling – of topography and oxidation levels.
- Simple statistics – of sample grades, and the derived data limits to use in geo-stats and grade estimation.
- Geo-statistics – 3D analysis, and the derived continuity.
- Resource block model – details.
- Block grade estimation – parameters and typical cross-sections.
- Block gold equivalent grade calculation.
- Bulk density.
- Resource classification.
- ATO 2021 Resources.

- Reconciliation of Resources with other estimates.

14.1 Introductory Statements

This 2021 Resource estimation was independently undertaken by the Author QP / CP, who makes the following Statements as to competency under the JORC and NI 43-101 Codes. The Author also states the CIM equivalence of JORC (accepted as a foreign Code by NI 43-101) reporting terms used in the Resource classification (as set out previously in the Terms of Reference).

14.1.1 JORC (2012) Competent Person (CP) Statement

The Consultant's Competent Person (CP) Statement accompanying these Mineral Resources is given below to meet the JORC code requirements.

- **Source data:** All source data was supplied by the Client and was taken at face value by the Consultant. The Consultant performed validation of the drill hole data to the extent thought possible and believes that validation to at least be to the level required for JORC Mineral Resource estimation and reporting. Although the Consultant validated the data to his satisfaction, he nevertheless provides this Mineral Resource estimate and the following Competent Person Statement for it on the basis that the Client takes responsibility to a Competent Persons level for the integrity of the source data.
- **Statement:** The information in this report that relates to ATO 2021 Mineral Resources is based on information compiled by Robin Rankin, a Competent Person who is a Member (#110551) of the Australasian Institute of Mining and Metallurgy (MAusIMM) and accredited since 2000 as a Chartered Professional by the AusIMM in the Geology discipline (CP(Geo)). Robin Rankin provided this information to his Client Steppe Gold Limited as paid consulting work in his capacity as Principal Consulting Geologist and operator of independent geological consultancy GeoRes. He and GeoRes are professionally and financially independent in the general sense and specifically of their Client and of the Client's Project. This consulting was provided on a paid basis, governed by a scope of work and a fee and expenses schedule, and the results or conclusions reported were not contingent on payments. Robin Rankin has sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity being undertaken to qualify as a Competent Person as defined in the 2012 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves' (the JORC Code). Robin Rankin consents to the inclusion in the report of the matters based on his information in the form and context in which it appears.

14.1.2 NI 43-101 Qualified Person (QP) Certification:

Certification: Robin A Rankin certifies, that in relation to his reporting of ATO 2021 Mineral Resources in this Technical Report, he is and a "Qualified Person (QP)" for the purposes of and as defined in Canadian National Instrument 43-101 (NI 43-101) of 24th June 2011. He fulfils that definition by reason of his education (geoscientist with BSc and MSc degrees), professional association affiliation (MAusIMM), association membership designation (Chartered Professional in the Geology discipline (CP(Geo)), and experience. He also certifies that this Technical Report has been prepared in compliance with that instrument and Form 43-101F1.

14.1.3 CIM Definition Standards Reconciliation

Statement: In compliance with Canadian NI 43-101 the Consultant states that the JORC Code (an acceptable foreign code in terms of NI 43-101) Mineral Resource categorisation used in this Technical Report is directly equivalent to the CIM categorisation.

14.2 Estimation Background

The background to this estimate is the previous 2017 estimate of Mineral Resources in Pipe 1, 2 and 4 by GSTATS. At that time the Mungu deposit was either not appreciated or had too little drilling for estimation. These current 2021 estimates therefore represent a re-estimation of Pipe 1, 2 and 4 (incorporating more drilling) and an initial estimate of Mungu. In practice the Pipe 1, 2 and 4 deposits were modelled together because of their close lateral proximity to each other. And the Mungu deposit area was modelled on its own as it existed as a distinctly isolated deposit.

14.3 Raw Data Supplied

Raw data supplied is described in terms of:

- 2017 data – used for the previous estimate.
- 2021 data – new data added to the 2017 data for this estimate.

14.3.1 2017 Data

Drill hole data: The 2017 GSTATS drill hole data base for Resource estimation contained 265 diamond drill holes[17].. In the Pipe 1, 2 and 4 deposit area there were in 238 diamond drill holes for a total of 44,284.2 m. That data included 32,791 assays. Thus, 27 holes were outside the estimated deposits.

Density data: A total of 226 bulk density determinations were made in 2010 and 2011. That data was used to determine average density for oxide, transitional and fresh material. Those density averages were used for the 2021 estimation reporting here.

17 2017 NI 43-101. Section 14.1.1, pp 118

14.3.2 2021 Raw Data

Project reports: The 2017 NI-43-101 report prepared by GSTATS was supplied to describe the Project and all previous work on it. Aspects of that report are included in this (particularly concerning background, history and geology).

Drill hole data: All drill hole data collected on the Project up to 2021 was supplied in MS Excel format. Data was supplied separately for collar, down-hole survey, assay and lithology data types. Later 2020 assaying was supplied incrementally as it became available. Collar data dip not contain hole collar azimuth and dips – they were loaded from the down-hole survey data.

Channel/trench data: Data from channel sampling from trenches was supplied in MS Excel format in a format for them to be treated as drill holes just below topography surface and following along the surface slope. This data was simply supplied along with the drill dole data.

Map data: Data on topography was supplied as 1 m interval surface contours in a DXF file.

Interpretations: No geological interpretations were supplied. All interpretations mentioned here were done by the Author QP.

Reporting parameters: During the course of the Consulting various parameters were sought from and supplied by Steppe. The principal ones were the lower grade cut-offs to use in Resource reporting.

14.4 Software

Drill hole data manipulation was mostly performed using MS Excel spreadsheet software. Mapping data was manipulated using Global Mapper. All geological interpretation, modelling, analysis and estimation was done with Minex geological and mining software. Reporting was done in MS Word.

14.5 Estimation Methodology

The 2021 Resource estimation process described in following sub-sections follows the flow of the processing, interpretation and estimation – the estimation methodology.

The estimation methodology used was:

- Data processing and consolidation.
- Drill hole databasing – storage of drill hole and trench data.
- Map databasing – storage of CAD data and deposit outline interpretations.
- Geological interpretation.
- Topography surface modelling.
- Oxidation surface interpretation and modelling.

- Statistics – simple analysis of sample assays; and data limit determination.
- Geo-statistics – 3D analysis; and interpretation of grade continuity directions and distances.
- Block modelling – creating a geological block Resource model.
- Block grade – estimation and validation.
- Resource classification – of grade blocks into JORC reporting classes.
- Resource reporting.
- Reconciliation – of Resources with past estimates.

14.6 Data Pre-Processing

Raw data was pre-processed to some degree to prepare it for use in the geological interpretation and grade estimation.

Drill hole data pre-processing: Raw data in the MS Excel spreadsheets was essentially formatted into flat column and row tabulations for export to ASCII and loading into the Minex geological software. Each data type (collar, survey, assay and lithology) was treated individually.

A primary edit to the Steppe data was to remove the "-" characters (a mathematical operator) from drill hole names (e.g. ATO-01 was changed to ATO01). Otherwise, the principal editing was formatting most numerical data to two decimal places.

Part of the processing included iteratively and retroactively incorporating the geological interpretations into the spreadsheets so that they could be seen alongside assays or lithology. This essentially marked the down-hole intercepts of the mineralised deposit intersections against assay intervals. This process was also used to create a new data type – population domains. A similar process happened with the oxidation level interpretations.

Topography data pre-processing: The raw 1 m interval surface contour data strings were all supplied as closed polygons (see Figure 14.10). These were edited in Global Mapper software to remove the closures.

14.7 Drill Hole Database

Data source: All drill hole and trench data were sourced from Steppe (see above) and pre-processed (see above for formatting and export to flat ASII files) before being loaded into a Minex drill hole database.

Databasing: A Minex drill hole database was loaded with the collar, survey, and assay data types ASCII extracts from the raw data. The latest version of the Minex database for estimation was ATO_Gold_20210205_GR2104.B3*. The load process included gross error checking. Only trivial errors were found, and they were rectified in the raw data before being reloaded.

Subsequent interpretations of the deposits (and therefore data populations) and oxidation levels were entered as new raw data in the spreadsheet and then loaded into Minex.

Holes (and trenches): Number and lengths of drill holes and trenches (pseudo drill holes) were given above.

Collar surveys: Collar data loaded included hole names, location, depth and drilling dip and azimuth. As the hole collar dip and azimuth was missing from the raw data all holes were loaded as vertical by default. This would be updated by the correct data in the down-hole survey data.

A hole type variable was added to enable holes selection on drilling method (diamond – DDH, reverse circulation – RC, and trench – TR). This type could also have been set on drilling year – but the Author QP was not familiar enough with the eras of drilling or aware if this would be useful.

Down-hole surveys: Down-hole surveys were loaded for all holes, and these included the azimuth and dip at the collar. Surveys were generally at 50 m intervals down-hole.

Some trenches had the sign of their dips obviously incorrect as they were seen in cross-section not running parallel to surface. This generally happened to trenches which ran up-hill from the start/collar end. These were corrected and reloaded. An example of a trench (ATOTR130) is shown on cross-section 2,270N at Mungu (Figure 14.4). The collar starts on surface and then the trench follows a few metres below topography.

Assays: Raw assay data was available for a wide range of elements. After a review of the data the Author QP loaded a limited selection of elements though to be best reflective of the expected gold and associated base mental mineralisation. Those elements included:

- Gold (Au);
- Silver (Ag);
- Lead (Pb);
- Zinc (Zn);
- Copper (Cu);
- Arsenic (As);
- Iron (Fe);
- Phosphorus (P);
- Sulphur (S).

Assays were provided in ppm units. The elements lead, zinc, copper, phosphorus, iron and sulphur were also loaded in % units (by dividing ppm by 10,000).

Geological logging: Lithology logging data was not loaded into the database as the raw data had not segregated the different aspects of the descriptive logging (mineralogy, alteration, fractures, recovery etc.) apart from the basic rock type. These variables were simply presented as long strings.

Consequently, lithology was not used in the mineralisation interpretation. It was however used, in the raw spreadsheet, for the oxidation level interpretation.

Density: No density raw data was supplied. Default bulk density was applied by oxidation level during the Resource reporting.

Domains: Population domains (whole numbers) were loaded from the mineralisation interpretations. This was done on a deposit basis as shown in Table 14.1.

Table 14.1 - Deposit Domains

Domain	Deposit
number	
1	Pipe 1
2	Pipe 2
4	Pipe 4
5 - 11 and 15	Mungu layers

At Mungu, the individual lenses were segregated by domain, hence the multiple domains. Other domains were interpreted (12,13 etc.) but were too sparse to model.

Oxidation: Interpretations of the oxidation levels were loaded as layer intercepts from the raw lithology data. The codes for the three intercepts (from surface down) were:

- OX – oxidized material.
- TR – transitional material.
- FR – fresh material.

14.8 Map Databasing

A map database was created to store CAD type data such as geological deposit outline interpretation strings and topography contour strings. This would also be used to store cross-section definitions. The latest version of the Minex map database was ATO_Gold_20210105_GR2104.GM3.

14.9 Geological Interpretation

Geological interpretation was carried out to:

- Define the outlines (shape) of the mineralised deposits.
- Determine the base of oxidation and the top of fresh rock.

14.9.1 Deposit Shape Interpretation

Basis: The basis for the geological deposit interpretation here was taken solely as mineralisation. A number of factors influenced this decision:

- The previous 2017 estimation was based on “grade shells” (see below).
- Lithological logging appeared too complex to deal with.
- Mineralization appeared reasonably continuous over long sections of holes, and correlated with similar sections in adjacent holes.
- These mineralised intersections were clearly concentrated in contiguous shapes.
- The shapes could represent practical mineable deposits.

Previous 2017 deposit model: The previous 2017 GSTATS estimation was based on “grade shells” created in Leapfrog software using a lower 0.1 g/t gold threshold (Figure 14.1). The three differently coloured Pipes are shown looking NNE. The green tabular shapes are diorite dykes.

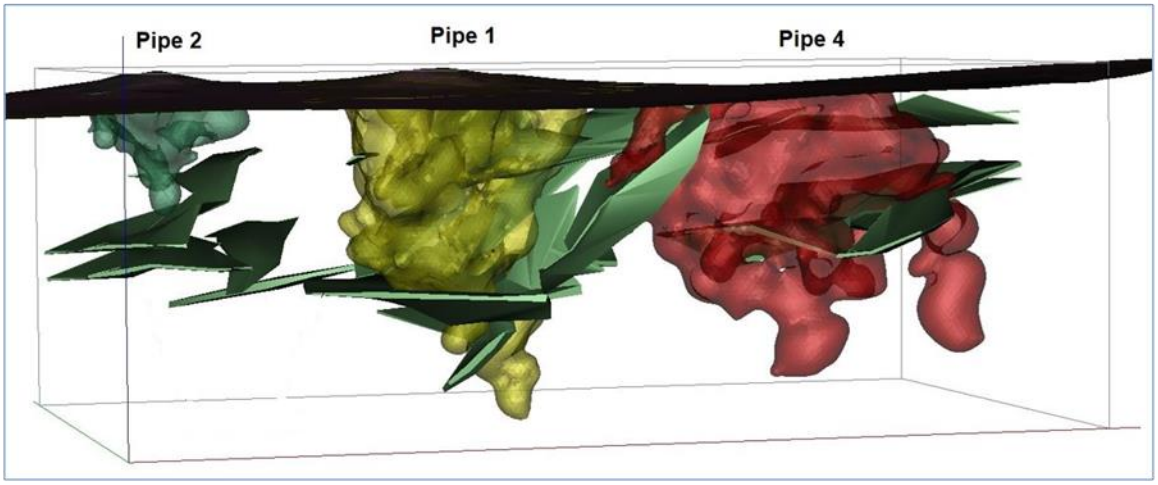
GSTATS justification of this was similar to that here – that the logged lithology was overly complex (24 different codes, and even when rationalised there were 10). They also commented that the “ore body was not significantly controlled by lithology”[18].

Although GSTATS also modelled the base of the surface sediments, cross-cutting diorite dykes, and two types of faults (low angle thrusts and semi-vertical faults bounding the Pipes) they still used the grade shells as the basis for their block model.

The grade shell models in Figure 14.1 seen to be fairly complex and tortuous in shape.

18 2017 NI 43-101. Section 14.2.1, 2nd paragraph, pp119.

Figure 14.1 - Previous 2017 Grade Shell Models of Pipes [19]



Source: ATO Mineral Resources Technical Report, 2021

Sectional interpretation method: The Author QP chose not to continue with a grade shell approach to modelling but to base 2021 deposit modelling on manual deposit outline interpretation on cross-sections. Those outlines would then be connected together into a standard “wire-frame” model.

Advantages of this approach were:

- Ability to consider multiple elements when interpreting an outline (see below).
- Ensure the ultimate outlines were reasonable contiguous and practical for mining.
- Allow the lower cut-off (see below) to be relatively dynamic and not reliant on only one element.

The method involved:

1. Creating a series of parallel vertical cross-sections through the drill holes as shown in Figure 14.2. These followed the drilling directions and so were oriented at 125° and were 30 m apart.
2. Plotting the drill holes projected onto the sections from 15 m either side (example in Figure 14.3). Gold and silver assays were annotated, and colour coded to help identification of mineralised zones.
3. Digitising a deposit outline (or several) around mineralised intersections (Figure 14.3 and Figure 14.5) using a “dynamic” lower grade cut-off (see further below) based initially on gold and silver. Outlines were named for the Domain number for the deposit (Table 14.1).
4. Identifying the mineralised intersections in the holes in the raw assay spreadsheet as depicted in Figure 14.6. This step was iteratively combined with the outline digitising as more practical

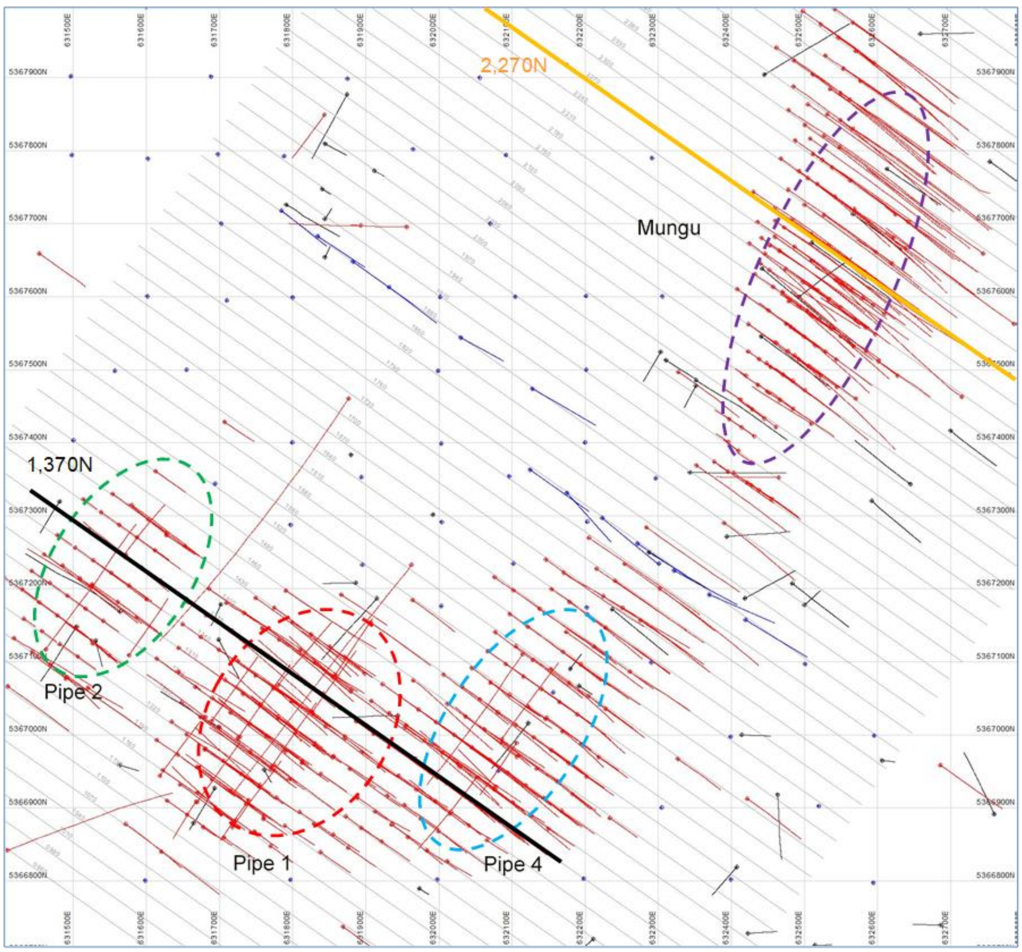
19 2017 NI 43-101. Section 14.2.1, Fig 14.2, pp120.

shapes emerged. It was also done to ensure the deposit assays were flagged with the domain number for the deposit (to be used during block grade estimation).

Cross-sections: The vertical cross-section lines at 125° are shown in grey and labelled in Figure 14.2. They are 30 m apart and each is seen to be close to the original drilling cross-sections (i.e. they line up with the red diamond drill hole traces). Naming of the cross-section was based on arbitrarily starting a southern one at 1,000N.

The locations of the deposits (dashed ovals) in the Figure are approximate. Pipe 2 (left oval) is shown green, Pipe 1 is red, Pipe 4 is blue (to match the outlines in Figure 14.3), and Mungu is shown purple. The coordinate grid is at 100 * 100 m spacing. The thicker black line traversing Pipe 1, 2, and 4 mark cross-section 1,370N as shown in Figure 14.2.

Figure 14.2 - Drill Hole & Trench Locations – Deposit Area 2021

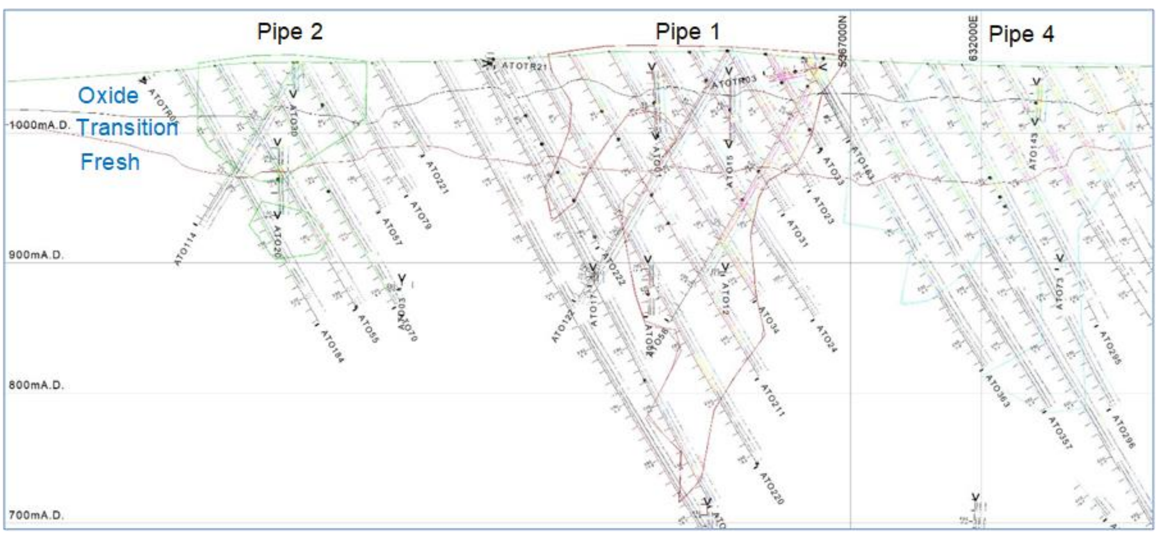


Source: ATO Mineral Resources Technical Report, 2021

Outline interpretation: Interpretation involved digitising an outline around mineralisation on a particular cross-section. The position of the line was based on enclosing material above lower gold and silver grade cut-offs (see below). This approximate cut-off was not applied in an absolute way as lower grade material would be included (internal dilution) if a more realistic body was created (see examples below). The outlines aimed to create large singular bodies where possible. Interpretation was performed on ~50 cross-sections.

Figure 14.3, of vertical cross-section 1,370N (location shown by the thick black line in Figure 14.2), shows outlines through Pipe 2 (left, green), 1 (centre, red) and 4 (right, cyan). Pipe 1 and 4 have single outlines; Pipe 2 has two outlines. The vertical levels are at 100 m spacing. Pipe 1 and 4 outlines here are 350 m and 250 m deep respectively. Figure 14.3 also shows the oxidation level surface models. Surface topography is marked by a green line, base of oxide by a black line, and top of fresh (or base of transition) by a red line.

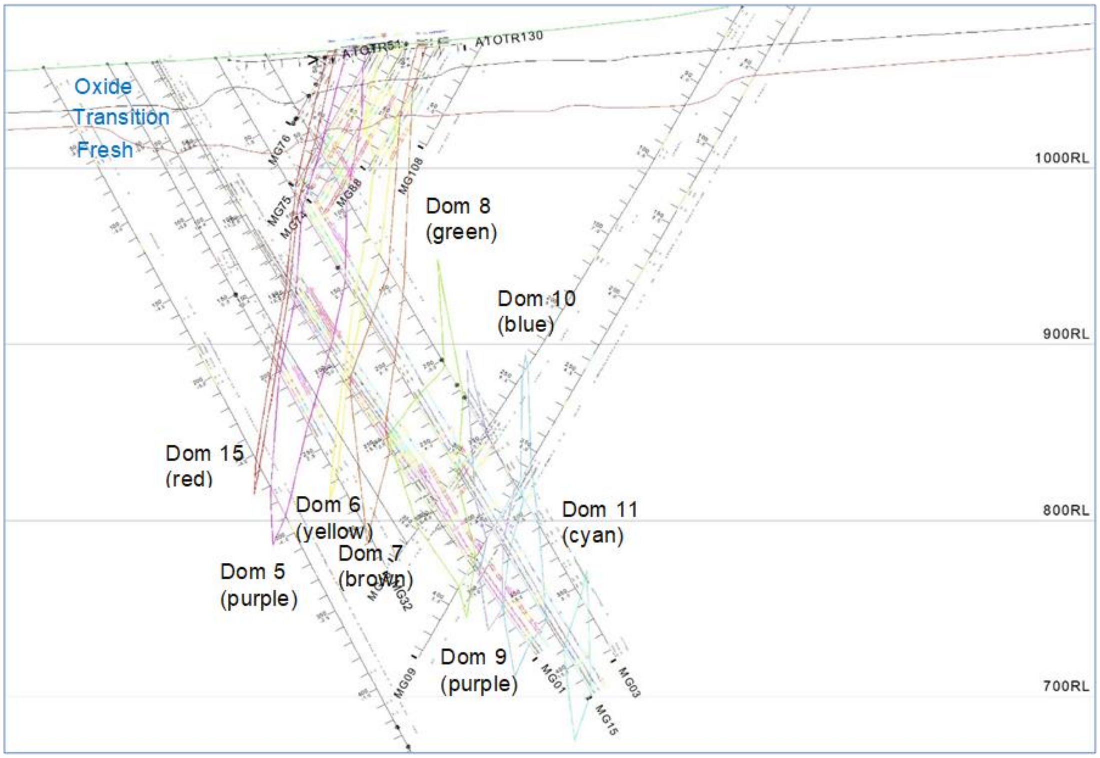
Figure 14.3 - Pipe 1, 2 and 4 Outlines Interpreted on Cross-Section 1,370N



Source: ATO Mineral Resources Technical Report, 2021

Figure 14.4 shows Mungu outlines on central vertical cross-section 2,270N (location shown by the thick orange line in Figure 14.2). This cross-section shows all 8 domain outlines (domain 5 to 11, and 15) interpreted at Mungu. The domains are tall and thin, nearly vertical, sub-parallel and strike 035° (see Figure 14.9) normal to the cross-sections. Figure 14.4 also shows the oxidation level surface models near surface.

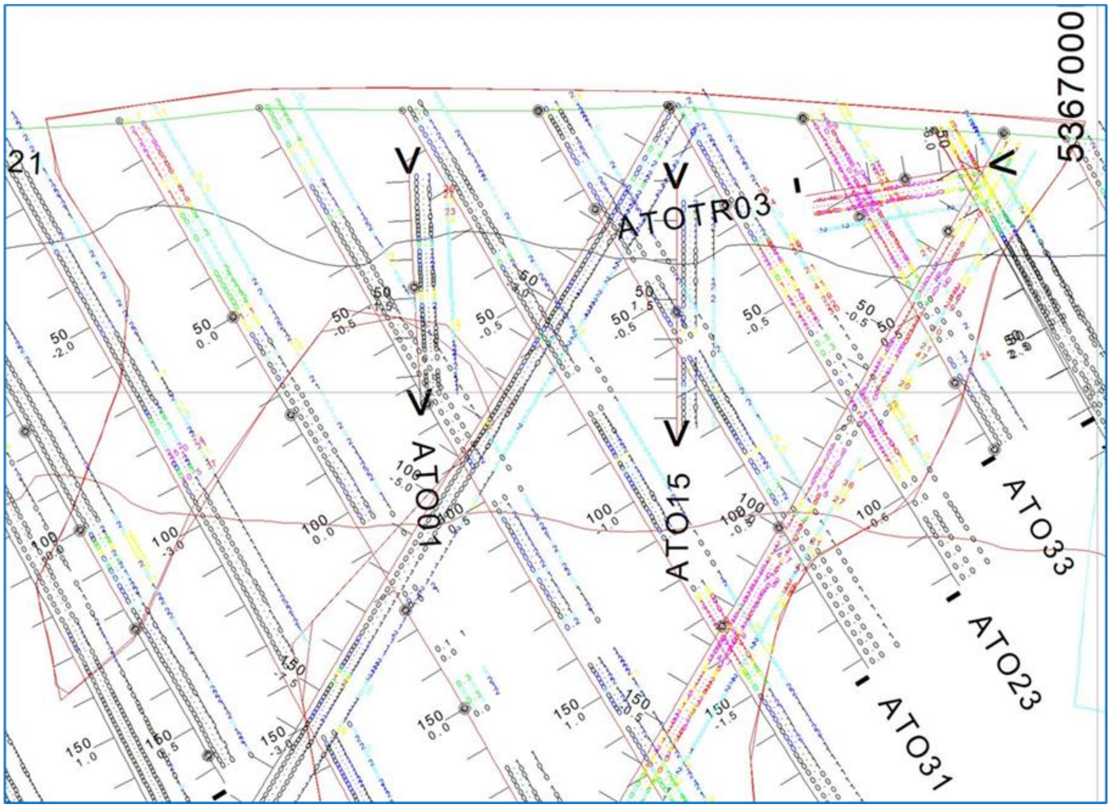
Figure 14.4 - Mungu Outlines Interpreted on Cross-Section 2,270N



Source: ATO Mineral Resources Technical Report, 2021

Deposit outline grade cut-offs: Figure 14.5 shows the central Pipe 1 on cross-section 1,370 in close-up with the red outline marking the upper part of the boundary of the interpreted deposit. It aims to illustrate the boundary in relation to the colour-coded gold and silver grades in the drill holes.

Figure 14.5 - Close-Up of Hole ATO23 (Pipe 1) on Cross-Section 1,370N



Source: ATO Mineral Resources Technical Report, 2021

After inspecting grades on multiple cross-sections the Author QP settled on a lower grade cut-offs at approximately >0.15 g/t gold and >1.0 g/t silver. These were chosen because of their observed coincidence in most mineralised intersections. Increasing grades above these values in Figure 14.5 are plotted cyan/green/yellow/red/purple (and are mostly inside the outline). Grades below are dark blue or black (and are mostly outside the outline).

However, the cut-off was also decided based on a decision as to whether the hole intercept was "generally" mineralised. This brought other elements into play – and necessitated analysis of the assay spreadsheet data (Figure 14.6) during the cross-sectional interpretation. The Author QP observed that mineralised zones also often (or usually) carried elevated values of the base metals lead, zinc and copper as well as of arsenic (typically associated with gold) and iron. This is well illustrated in Figure 14.6's snap-shot of spreadsheet colour-coded assays in hole ATO23. The two zones (<49.2 to 57.6 m and 75.3 to 96.9 m) of elevated gold assays (5th column from left, colours yellow/red/purple) are adjacent to similarly elevated values of lead and zinc and to a lesser extent copper (columns right to gold).

Figure 14.6 – Hole ATO23 Colour Coded Assay Spreadsheet Data

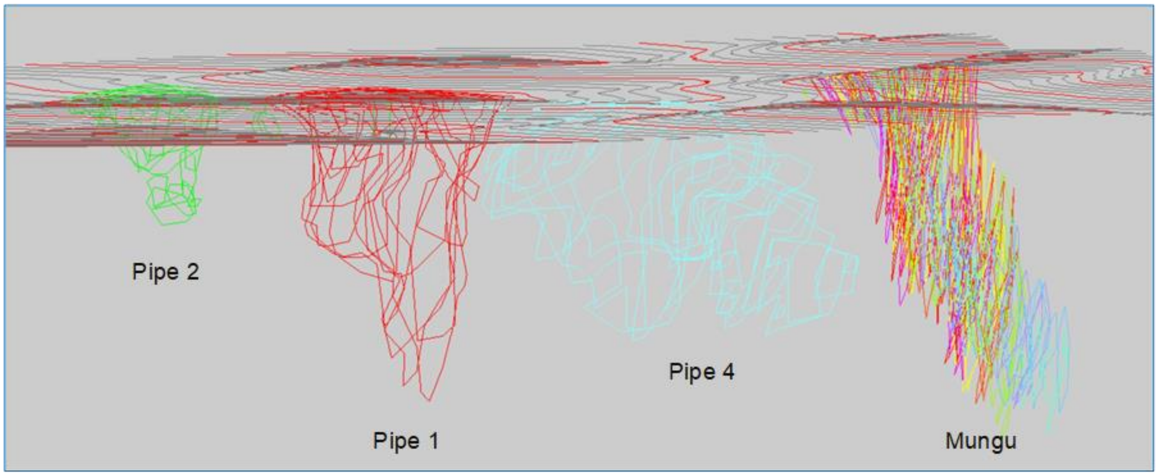
BOREID	FROM	TOASSAY	DOM	AU	AG	PB_PCT	ZN_PCT	CU_PCT	AS	FE_PCT
ATO23	49.20	50.25	1	1.69	14.7	2.37	4.71	0.16		
ATO23	50.25	51.30	1	2.03	41.0	6.39	11.86	0.74		
ATO23	51.30	52.35	1	1.22	17.1	2.39	4.79	0.30		
ATO23	52.35	53.40	1	0.96	17.0	3.25	7.64	0.29		
ATO23	53.40	54.45	1	1.99	28.7	7.43	10.69	0.34		
ATO23	54.45	55.50	1	1.06	7.1	0.93	1.39	0.09		
ATO23	55.50	56.55	1	2.18	21.8	4.12	4.85	0.30		
ATO23	56.55	57.60	1	2.55	29.6	6.52	8.11	0.32		
ATO23	57.60	60.10	1	0.08	0.6	0.04	0.79	0.00		
ATO23	60.10	62.60	1	0.46	0.7	0.04	0.07	0.01		
ATO23	62.60	65.10	1	0.34	0.5	0.04	0.06	0.01		
ATO23	65.10	67.60	1	0.05	0.2	0.01	0.08	0.00		
ATO23	67.60	70.10	1	0.09	0.3	0.00	0.08	0.00		
ATO23	70.10	72.70	1	0.15	0.3	0.00	0.02	0.00		
ATO23	72.70	75.30	1	0.09	0.4	0.01	0.33	0.00		
ATO23	75.30	76.30	1	2.63	7.2	1.22	1.52	0.12		
ATO23	76.30	77.30	1	2.88	6.7	0.58	0.77	0.07		
ATO23	77.30	78.30	1	6.96	11.3	1.25	1.23	0.18		
ATO23	78.30	79.30	1	0.61	3.0	0.27	0.34	0.05		
ATO23	79.30	80.30	1	2.73	7.0	0.55	0.56	0.07		
ATO23	80.30	81.30	1	1.67	3.3	0.26	0.23	0.03		
ATO23	81.30	82.30	1	8.13	21.9	3.31	2.21	0.30		
ATO23	82.30	83.30	1	6.13	16.8	1.63	1.24	0.37		
ATO23	83.30	84.30	1	6.10	15.4	3.33	3.08	0.21		
ATO23	84.30	85.30	1	2.78	10.3	0.94	1.38	0.26		
ATO23	85.30	86.30	1	2.36	6.5	0.63	0.87	0.06		
ATO23	86.30	87.30	1	1.70	5.6	0.38	0.47	0.12		
ATO23	87.30	88.30	1	2.30	12.9	0.42	0.48	0.27		
ATO23	88.30	89.30	1	2.03	4.9	0.16	0.10	0.08		
ATO23	89.30	90.30	1	1.71	11.0	0.19	0.28	0.31		
ATO23	90.30	91.40	1	1.96	26.8	4.22	4.93	0.54		
ATO23	91.40	92.50	1	1.12	4.6	0.14	0.10	0.06		
ATO23	92.50	93.60	1	0.82	3.2	0.24	0.33	0.04		
ATO23	93.60	94.70	1	0.63	3.4	0.15	0.09	0.03		
ATO23	94.70	95.80	1	0.74	10.4	1.55	2.13	0.21		
ATO23	95.80	96.90	1	0.67	3.5	0.21	0.11	0.05		
ATO23	96.90	99.20		0.04	0.2	0.01	0.02	0.00		
ATO23	99.20	101.50			0.3	0.00	0.01	0.00		
ATO23	101.50	103.80		0.01		0.00	0.01	0.00		
ATO23	103.80	106.10			0.3	0.00	0.01	0.00		
ATO23	106.10	107.70		0.03	0.4	0.00	0.01	0.00		
ATO23	107.70	109.30		0.02	0.4	0.00	0.01	0.00		
ATO23	109.30	110.70		0.03	0.4	0.00	0.01	0.00		
ATO23	110.70	111.70		0.05	0.3	0.00	0.01	0.00		
ATO23	111.70	114.70		0.03	0.3	0.00	0.01	0.00		

Source: ATO Mineral Resources Technical Report, 2021

The inclusion or exclusion of areas based on the gold and silver cut-offs (and the other elements) was decided with the aim of creating realistic and practical shapes. As shown in Figure 14.5, there are clearly areas below the cut-off included within the outline. Some appear trivial, others are because grades on adjacent cross-sections are better. Hole ATO23 as shown in Figure 14.6 appears on the right in cross-section in Figure 14.5. It seemed logical to include the low-grade intersection 57.6 to 75.3 m (in Figure 14.6) because it was generally surrounded by high grades in adjacent holes.

Pipe 1, 2 and 4 and Mungu outlines: Figure 14.7 shows all outline interpretations of all deposits below contoured topography. The view is looking 012° and down 2° (approximately the same way as the 2017 model in Figure 14.1).

Figure 14.7 – Outline Interpretations of All Deposits



Source: ATO Mineral Resources Technical Report, 2021

14.9.2 OXIDATION LEVEL INTERPRETATION

Interpretation of the degree or level of oxidation at the deposits was done in all drill holes from the lithological logs. A column for an oxidation code was added to the lithological data spreadsheet. From surface the hole interval was interpreted as oxidised (code OX), then as partly oxidised or transitional (code TR), then as un-oxidised or fresh (code FR).

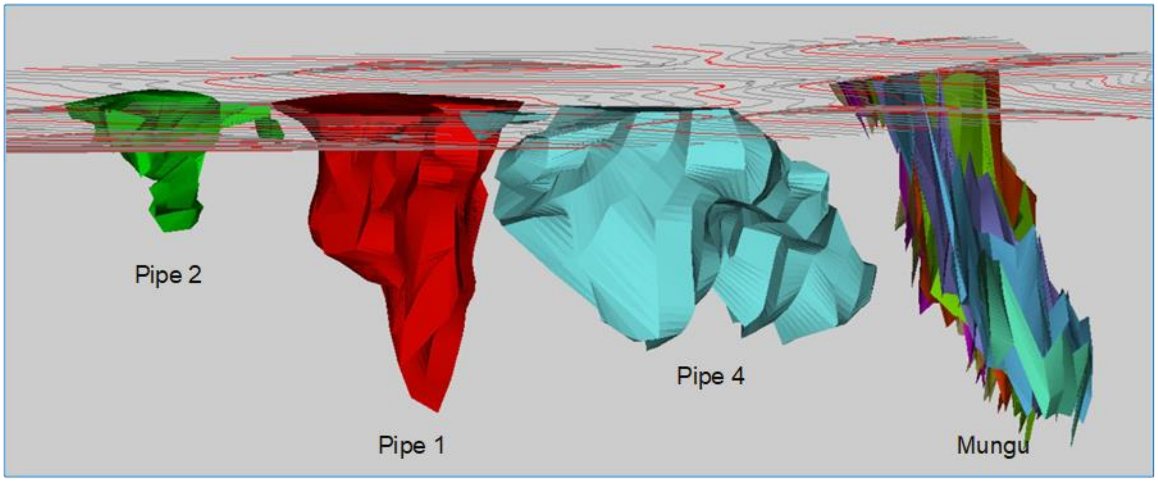
Interpretation relied upon the logging descriptions. Logging over the period of drilling was variable and hence not all oxidation had been logged. Where missing the Author QP used other clues, such as the rock type summary, fracturing and weathering comments.

Interpretations were modified and improved iteratively after the interval data was loaded, modelled and viewed in cross-section.

14.10 Wire-Frame Modelling of Deposits

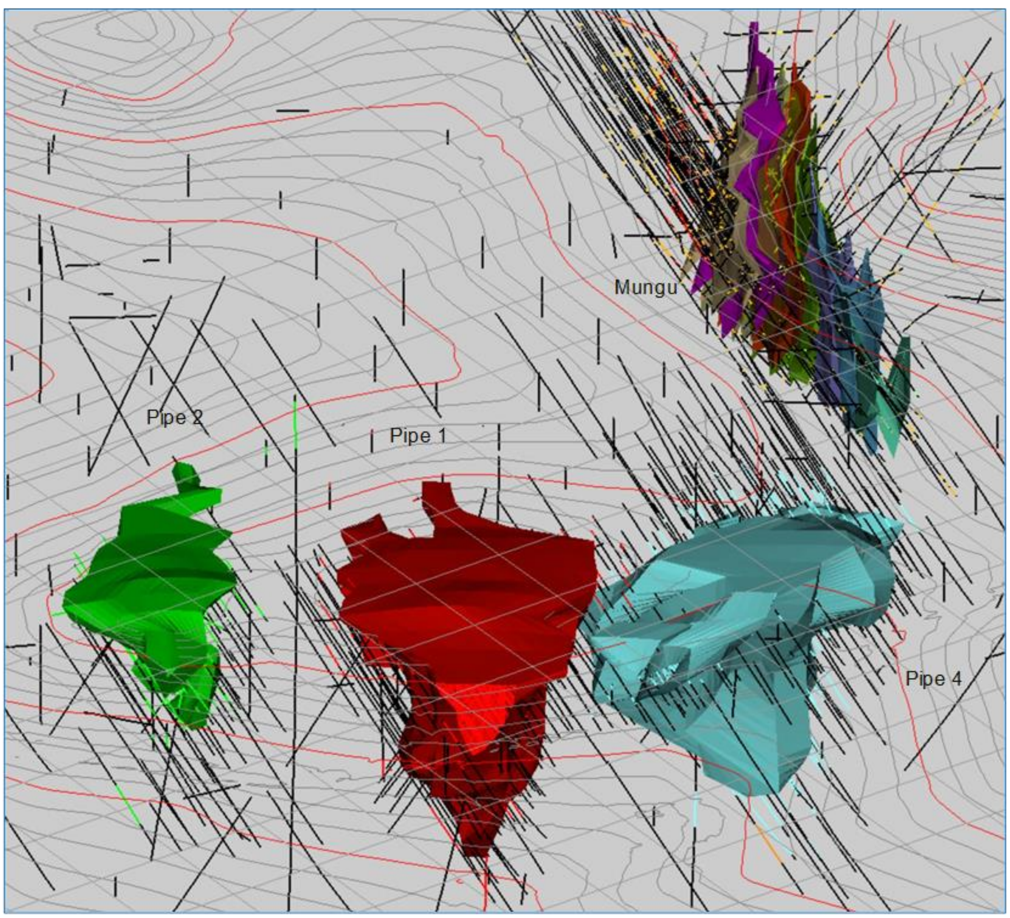
Once fully interpreted the cross-sectional outlines for each deposit (domain) were connected together with wires to form solid wire-frame models. Figure 66 shows the wire-frame models of all deposits below contoured topography. The view is looking towards 012° and down 2° (approximately the same way as the 2017 model in Figure 14.1 and the 2021 outlines in Figure 14.7).

Figure 14.8 - Wire-Frame Models of All Deposits – Looking ~North



Source: ATO Mineral Resources Technical Report, 2021

Figure 14.9 - Wire-Frame Models of All Deposits – Looking 035°



Source: ATO Mineral Resources Technical Report, 2021

Figure 14.9 shows all of the deposits looking towards 035° and down 35° - the view normal to the cross-sections and along strike. The Figure also shows the drill holes in 3D. The deposits are seen to be thinnest in this view as they are elongated along strike. And the multiple sub-parallel and sub-vertical Mungu lodes are very clearly parallel to this strike view direction.

14.11 Surface Modelling

Surface modelling was undertaken to produce digital terrain model (DTM) surfaces for topography over the area and for the two oxidation levels (base of oxidised rock and top of fresh rock) below surface.

14.11.1 Topography Surface

Topography data was supplied at 1 m interval horizontal contour strings as shown in Figure 14.10.

As all strings were supplied as closed polygons with their ends joined the data required pre-processing to break these connections.

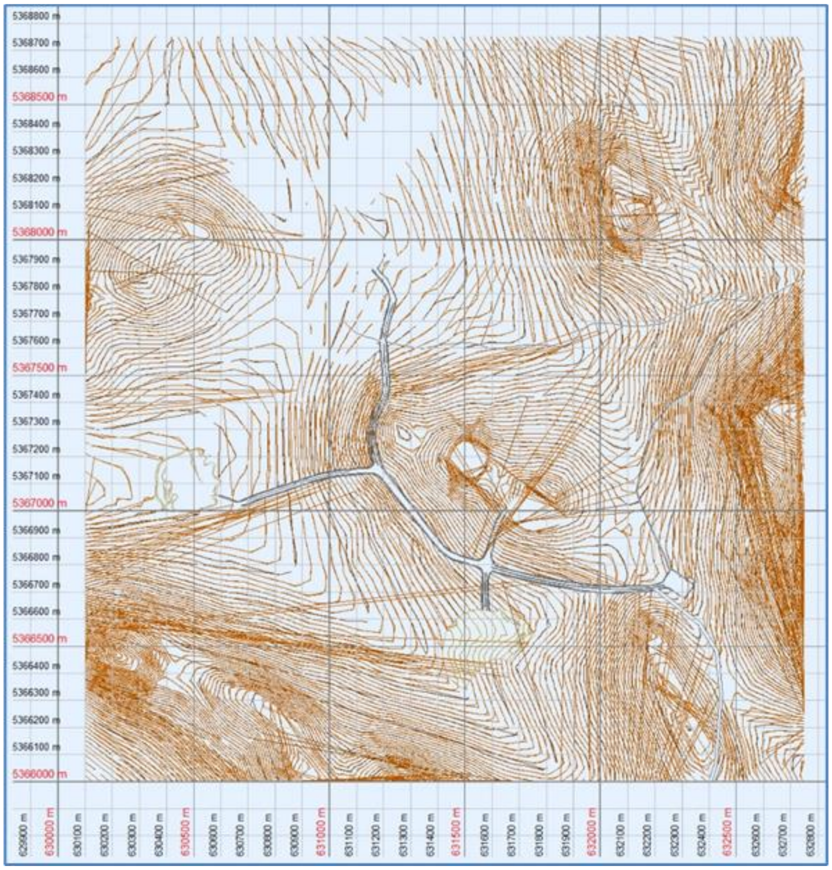
Strings were loaded into the Minex map database.

Modelling the topography surface was done using triangulation (creating triangle file TOPO.tr5).

For subsequent use in Resource reporting and display the topography triangle surface was converted to a regular 5 * 5 m gridded surface (grid TOPO).

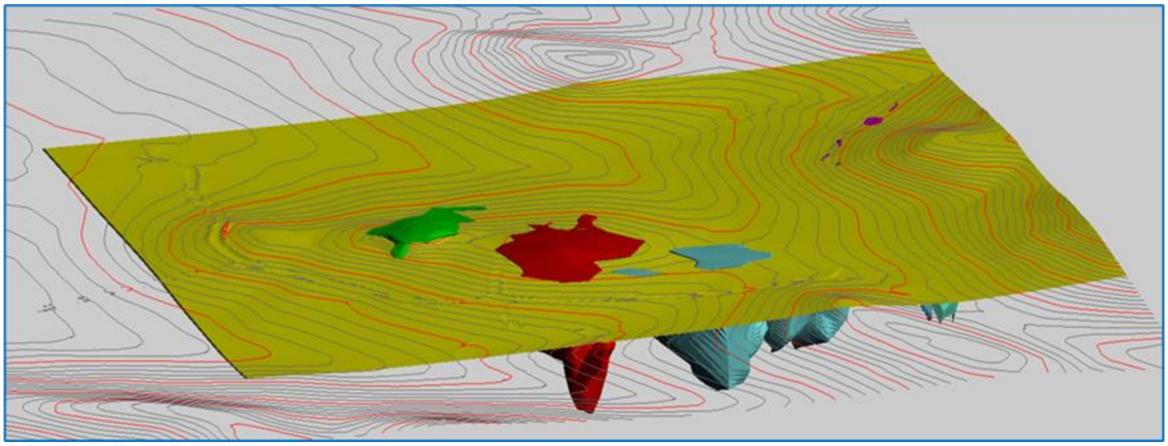
Figure 14.11 shows the topography gridded surface as a yellow solid above the wire-frame deposit models. The surface is also contoured at 2 m intervals in grey and 10 m intervals in red. The view is towards 015° and down at 20°. Lighting is from the south-east.

Figure 14.10 - Topography Raw 1 M Contour String Data



Source: ATO Mineral Resources Technical Report, 2021

Figure 14.11 - Topography Surface Model



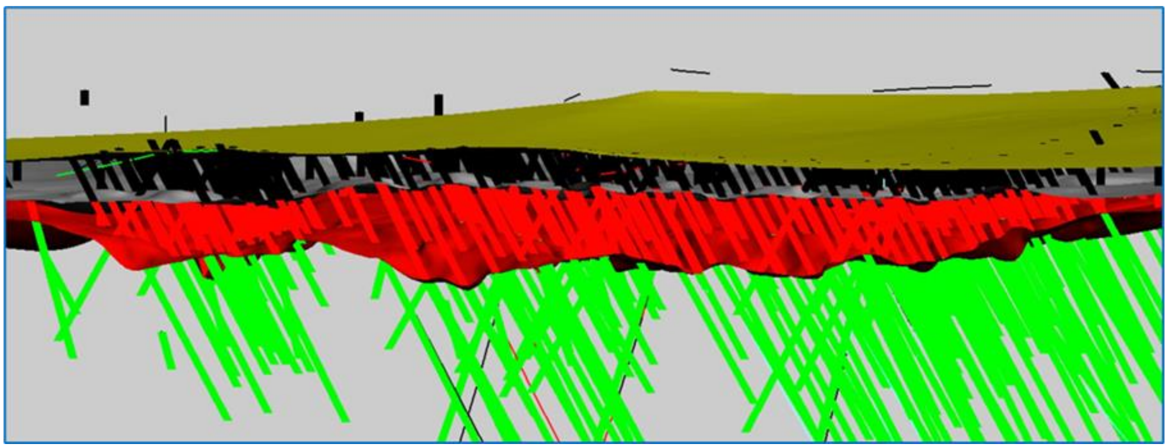
Source: ATO Mineral Resources Technical Report, 2021

14.11.2 Oxidation Level Surfaces

Oxidation level surfaces were interpolated in 3D from the interpretations (OX, TR and FR) loaded in the drill hole database. Interpolation used a DTM growth algorithm. Surfaces were interpolated for the base of oxidation (grid OX_SF) and for the base of transition (grid TR_SF). The base of transition was equivalent to the top of fresh.

Figure 14.12 illustrates the oxidation (grey) and fresh (red) surfaces below topography (yellow). The view is through the middle of Pipe 1, 2 and 4, is looking ~north and slightly downwards, and a clipping plane has cut off the southern half of the lodes. The drill holes are shown, with the upper oxided parts in black, the transitional parts red and the lower fresh parts green.

Figure 14.12 - Oxidation Surface Models



Source: ATO Mineral Resources Technical Report, 2021

Oxidation levels were also shown on the cross-sections above (Figure 14.3 to Figure 14.5).

14.12 Simple Sample Grade Statistics

Simple statistical analysis was performed briefly (as they had been studied in some detail for the 2017 estimation) to determine the variation and character of values for the different elements. In particular, it looked at anomalous upper and/or lower data values to evaluate what clipping or cutting of grade values might be required during further statistical analysis and block grade estimation. Only the statistics for gold (the dominant mineralisation) are given here.

Gold: Table 14.2 tabulates simple raw (un-composited) gold statistics for all samples and then for Pipe 1, 2 and 4 (domains 1, 2 and 4). Very few gold samples were shown to be highly anomalous, which in itself is unusual for gold deposits. Of all 52,515 samples (which included samples inside and outside the deposits) only 128 (0.2%) were >20 g/t. For Pipe 1 and 4 values >10 g/t accounted for ~1% of samples, and with Pipe 2 that proportion applied to values >5 g/t.

Table 14.2 - Gold statistics

Element	Domain	Limits	Samples number	diff	%	Length (m)	Max (g/t)	Min (g/t)	Av (g/t)	Med (g/t)	SD	Variance	CV
Au	All		52,515			65,591.9	382.00	0.00	0.62	0.09	3.72	13.84	6.03
	All	<20	52,387	128	0.2%	65,460.6	19.59	0.00	0.49	0.09	1.31	1.71	2.65
	All	0.01<<20	44,782	7,733	14.7%	54,275.8	19.59	0.02	0.58	0.14	1.40	1.95	2.43
Au	1		12,050			14,840.0	71.08	0.00	1.02	0.31	2.25	5.05	2.21
	1	<10	11,938	112	0.9%	14,725.3	9.90	0.00	0.87	0.30	1.36	1.76	1.53
	1	0.01<<10	11,774	276	2.3%	14,457.8	9.90	0.02	0.88	0.31	1.33	1.77	1.52
Au	2		3,663			4,566.0	183.00	0.01	0.35	0.07	3.56	12.65	10.09
	2	<5	3,635	28	0.8%	4,532.9	4.88	0.01	0.19	0.07	0.42	0.17	2.16
	2	0.01<<5	3,487	176	4.8%	4,332.7	4.88	0.02	0.20	0.07	0.42	0.18	2.11
Au	4		12,599			14,652.1	382.00	0.01	0.82	0.19	5.73	32.87	6.96
	4	<10	12,467	132	1.0%	14,514.0	9.92	0.01	0.49	0.08	0.95	0.90	1.93
	4	0.01<<10	12,059	540	4.3%	13,981.3	9.92	0.02	0.51	0.20	0.96	0.92	1.89

From these results (minimal anomalous values) the Author QP chose not to cut input grades during block grade estimation. This decision was at odds with the 2017 estimation where grades were cut.

Figure 14.13 to Figure 14.17 show gold histograms for Pipe 1, 2 and 4. Normal histograms are on the left, log histograms on the right. Gold is seen to be log normal in distribution. The log histogram for Pipe 1 indicates several populations are present.

Figure 14.13 - Gold Histogram Pipe 1 - Normal

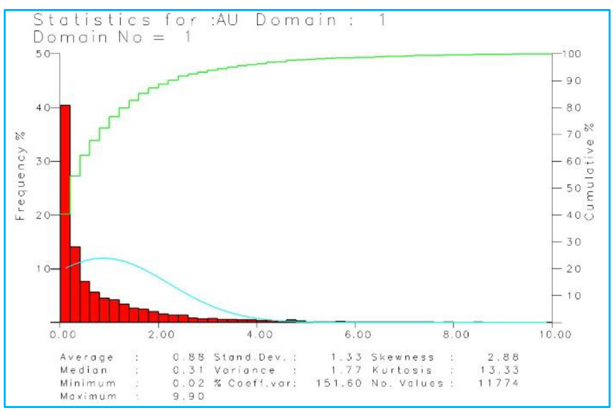


Figure 14.14 - Gold histogram Pipe 1 - Log

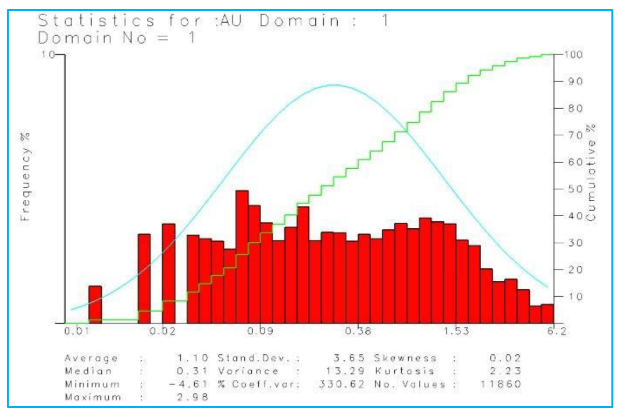


Figure 14.15 - Gold Histogram Pipe 2 - Normal

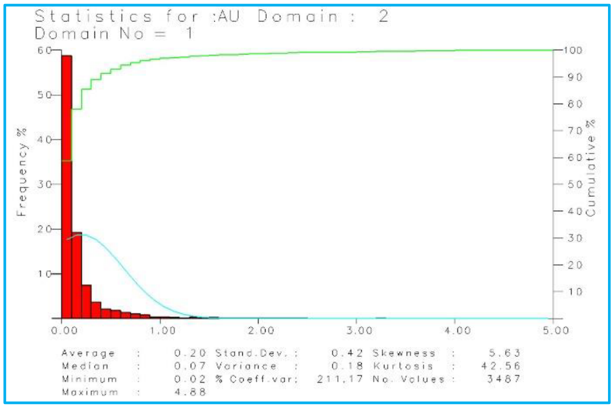


Figure 14.16 - Gold Histogram Pipe 4 - Normal

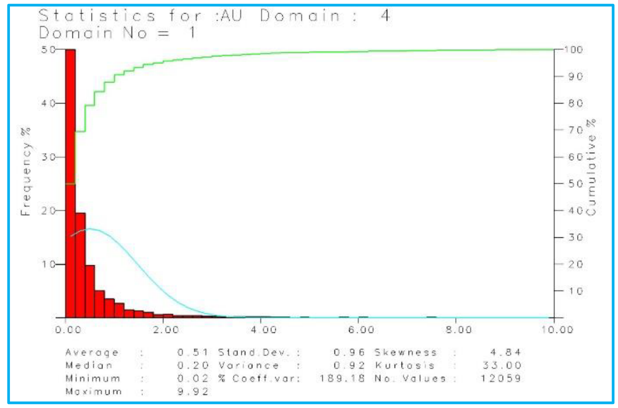


Figure 14.17 - Gold Histogram Pipe 4 - Log

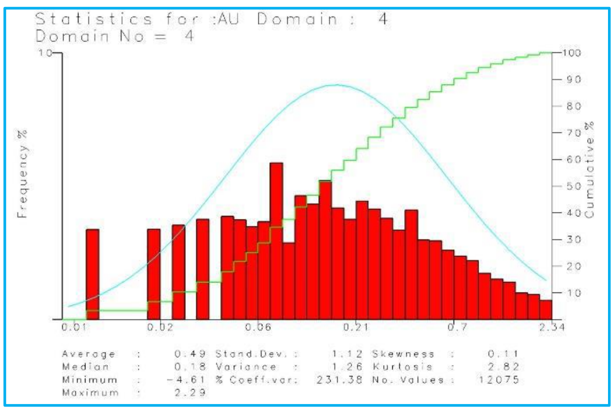
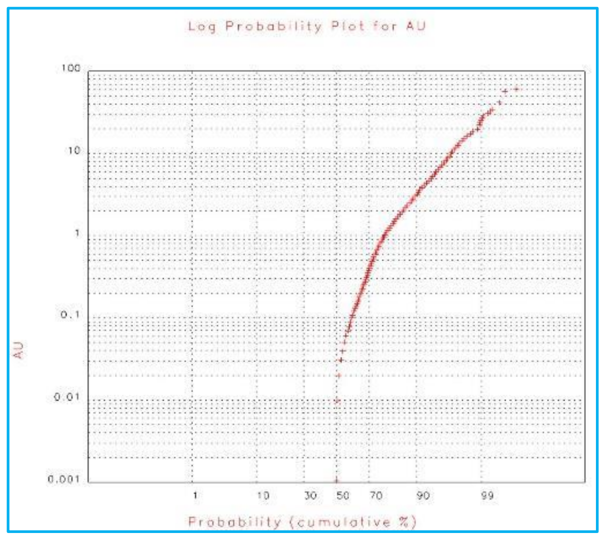


Figure 14.18 - Gold Lop Prob Pipe 1



The gold log probability plot for Pipe 1 (Figure 14.18) indicates several populations are present, with a change in slope at ~1.0 g/t.

14.13 Geo-Statistical Grade Analysis

Geo-statistical analysis was only performed very briefly to attempt to determine grade continuity directions and distances. This was because most variograms studied initially gave ranges of at least ~25 m. In other words, the same order of magnitude or longer than the typical 30 * 30 m drill hole spacing. This implied that the ranges were approaching the same dimensions (50-100 m) observed of the well mineralised parts of the interpreted deposits – and that it was less necessary to perform a detailed analysis as drill hole samples essentially fully filled the interpreted shapes.

Variograms: The >25 m ranges determined are illustrated in several variograms for gold in Pipe 4 with directions fairly randomly chosen. Figure 14.19 is in the horizontal E/W direction – and has a maximum range of ~25 m. Figure 14.20 is dipping 45° up towards the east (or 45° down towards the west) and has a maximum range of 45 m.

Figure 14.19 - Gold Pipe 4 Variogram 0°@090°

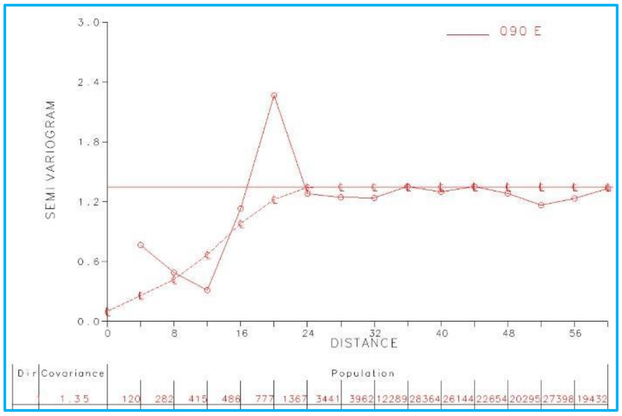
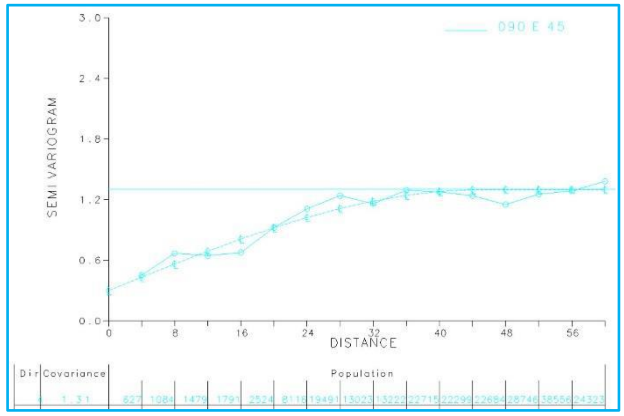


Figure 14.20 - Gold Pipe 4 variogram +45°@090°



A fairly detailed geo-statistical analysis was performed for the 2017 estimate and those findings were partly used to confirm the approach taken here. Gold ranges then ranged from ~20 m to ~60 m[20].

Continuity: The Author QP's approach to selecting data continuity distances took his brief results, with confirmation of reasonable ranges from the 2017 work, to assume that the sample density was sufficient to cover all expected distances within the deposit models (and well short of the selected 75 m estimation scan distance). This was eventually proved accurate when the average sample estimation distances (D, used in the Resource classification) proved to be only ~28 m in Pipe 1,2 and 4 and ~31 m at Mungu.

The Author QP's approach to selecting continuity directions was not to use Variography but to base it on the clear mineralisation directions evident during the deposit cross-sectional interpretation. At Pipe 1 this was a steep 80°W dip. At Pipe 2 and 4 it was an intermediate 45°W dip. And at Mungu it was a vertical dip with the lodes striking 033°. Those directions are tabulated in Table 14.5.

14.14 Resource Block Model

Block models: Separate block models were built for the southern Pipe 1, 2 and 4 deposits (model ATO_D124_V1_555_20210108_M5.G3*) and for the northern Mungu deposit (model ATO_MUNGU_V2_255_20210201_GRADE_STR_033_M5.G3*). The reason for the separate

20 2017 NI 43-101. Section 14.5, Table 14.11, pp135.

block models, apart from practicality, was that the Mungu model would use tall thin blocks to better represent the deposit shapes.

Build: Each block model was built (the blocks created) from the deposit wire-frame models. Any blocks above topography would be excluded during reporting by the use of the topography surface model as a vertical limit.

Block size: A basic block size of 5 m was chosen to suit the typical 30 * 30 * 2 m sampling. Drill holes were ~30 m apart on cross-section (X direction; cross-sections were 30 m apart (Y direction); and sampling down-hole was ~1-2 m (Z direction). Taking into account also the typical 60° dip of the drill holes the choice of 5 m blocks was ~20% of the data spacing.

A differentiating parameter of the block models was the choice of primary block size (without any further sub-blocking) to accommodate both the data spacing and the shapes of the deposits:

- Pipe 1, 2, 4 model: 5 * 5 * 5 m;
- Mungu: 2 * 5 * 5 m.

The tall (in Z direction) thin (in E/W or X direction) strike (Y direction) aligned lodes at Mungu was better modelled with smaller blocks in the X direction (hence 2 m).

Block model dimensions: Tables 14.3 and 14.4 and give the block model dimensions for the Pipe 1, 2 and 4 and Mungu deposits respectively. The origin and extents of each cover the full volume of the geological models. Both bock models are orthogonal to the coordinate system as neither were rotated (in contrast to the 2017 model which was rotated 55° to align the cross-sections parallel to an axis).

Table 14.3 - Pipe 1, 2 and 4 Block Model Dimensions

Parameter		Direction		
		X	Y	Z
Origin (m)	From	631,420	5,366,800	660
(UTM, WGS 84, Zone 49N)	To	632,400	5,367,450	1,080
Extent (m)		980	650	420
Rotation (°)		0	0	0
Primary block size (m)		5.0	5.0	5.0
Primary block numbers		196	130	84
Sub-block number		1	1	1
Total Block Number			**36,628**	

Table 14.4 - Mungu Block Model Dimensions

Parameter		Direction		
		X	Y	Z
Origin (m)	From	632,350	5,367,250	600
(UTM, WGS 84, Zone 49N)	To	632,770	5,367,920	1,095
Extent (m)		420	670	495
Rotation (°)		0	0	0
Primary block size (m)		2.0	5.0	5.0
Primary block numbers		210	134	99
Sub-block number		1	1	1
Total Block Number		**96,259**		

Block domains: The build process also tagged the blocks with the respective domain numbers (Table 14.1).

Block variables: Variables were created for:

- Grades: Au (g/t), AuEg (g/t), Ag (g/t), Cu (%), Pb (%), Zn (%);
- Classification: Au_D (average distance (m)), Au_P (number points), Au_CAT (class number).

Oxidation levels were not loaded into the blocks as this was accounted for dynamically in Resource reporting using the oxidation surface models as vertical limits.

14.15 Block Grades

Block grades in each deposit block model were estimated individually from assays in the drill hole database.

Block grades are described in terms of:

- Grade estimation;
- Resource classification parameters;
- Validation;
- Grade plotting – on cross-section.

14.15.1 Block Grade Estimation

Domain control: Data population control within each deposit (or lode) was ensured by matching the block domains (loaded from the wire-frame models) with the sample domains (interpreted to match the deposit outlines).

Sample compositing: Sample compositing is done within domains. For the more massively shaped Pipe 1, 2 and 4 deposits the drill hole samples were composited down-hole to exactly 2.0 m with residuals included >1.0 m (50%). For the taller thinner lodes at Mungu the drill hole samples were composited down-hole to exactly 1.0 m with residuals included >0.5 m (50%).

Cutting / clipping input grades: No cutting or clipping was done of input or output grade data as none was considered necessary. The Author QP's justification of this was twofold. In the first place the consulting time precluded detailed statistical analysis (and therefore fine-tuning grade estimation). And secondly and more importantly he considered that the input mineralised data did not contain extremely anomalous values, and in fact was very mildly distributed between highs and lows. Given the large numbers of potential samples available to estimate each grade block single outlier grades would have very little influence. And the Author QP here wanted to allow those limited highly anomalous samples to have some limited influence.

Estimation algorithm & parameters: Block grade estimation was performed using a standard inverse distance squared algorithm (ID2). Parameters were applied slightly differently between the deposits to adapt to their orientations. The same parameters were applied to all elements estimated (gold, silver, copper, lead, zinc). Grade estimation parameters are given in Table 14.5.

Table 14.5- Grade Estimation Parameters

Parameter			Pipe 1		Pipe 2 & 4		Mungu	
Data limits			-		-		-	
Scan (m)			75		75		75	
Points	Min sectors		1		1		1	
	Max pts/sector		3		3		3	
Axes	Rotation (°)	X	0		0		0	
		Y	+10	Dip 80°W	+45	Dip 45°W	0	
		Z	0		0		+33	Strike 033°
	Weighting	X	1.5	Weaker E/W	1.5	Weaker E/W	1.5	Weaker E/W
		Y	1.5	Weaker N/S	1.5	Weaker N/S	1.0	Stronger N/S
		Z	1.0	Stronger vert	1.0	Stronger vert	1.0	Stronger vert

Pipe 1 parameters: These were to apply a steep 80° westerly down-dip continuity.

Pipe 2 and 4 parameters: These were to apply a 45° westerly dip and a stronger continuity down that dip.

Mungu parameters: These were to apply a 033° strike direction and stronger continuity in the vertical strike plane.

Grade estimation statistics: Table 14.6 and Table 14.7 give the raw block estimation statistics for all elements by deposit.

Table 14.6 - Block Estimation Statistics - Pipe 1, 2 And 4

Domain	Element	Accessed				Interpolated							
		Pts	Max	Min	Av	Pts	Max	Min	Av	Med	SD	Variance	CV
1	Au (g/t)	24,717	195.95	0.00	0.54	62,279	17.46	0.01	0.86	1.74	1.04	1.07	1.20
2	Au (g/t)					13,286	30.54	0.01	0.43				
4	Au (g/t)					72,131	41.29	0.01	0.86				
1	Ag (g/t)	26,497	1,256.98	0.20	5.49	62,279	518.55	0.22	5.79				
2	Ag (g/t)					13,286	108.35	0.36	4.01				
4	Ag (g/t)					72,131	571.64	0.26	11.81				
1	Pb (%)	32,315	23.19	0.00	0.23	62,279	7.02	0.00	0.58				
2	Pb (%)					13,286	6.19	0.00	0.48				
4	Pb (%)					72,131	5.54	0.00	0.20				
1	Zn (%)	32,395	33.37	0.00	0.40	62,276	20.92	0.00	0.93				
2	Zn (%)					13,286	19.57	0.01	0.74				
4	Zn (%)					72,131	10.41	0.00	0.37				

Table 14.7 - Block Estimation Statistics – Mungu

Domain	Element	Accessed				Interpolated							
		Pts	Max	Min	Av	Pts/Blocks	Max	Min	Av	Med	SD	Variance	CV
All	Au (g/t)	15,265	172.00	0.01	0.53	116,629	82.15	0.01	0.71	0.43	1.75	3.07	2.47
15	Au (g/t)	"	"	"	"	5,733	3.27	0.01	0.23	0.26	0.33	0.11	1.43
5	Au (g/t)	"	"	"	"	24,434	3.96	0.01	0.43	0.96	0.35	0.12	0.81
6	Au (g/t)	"	"	"	"	21,203	5.75	0.01	0.53	0.52	0.46	0.21	0.85
7	Au (g/t)	"	"	"	"	20,189	82.15	0.01	1.31	0.79	3.02	9.12	2.31
8	Au (g/t)	"	"	"	"	22,765	48.92	0.01	1.13	0.03	2.52	6.35	2.24
9	Au (g/t)	"	"	"	"	10,920	4.04	0.01	0.40	0.16	0.46	0.22	1.17
10	Au (g/t)	"	"	"	"	8,766	3.91	0.01	0.32	0.26	0.35	0.12	1.10
11	Au (g/t)	"	"	"	"	2,619	0.82	0.01	0.22	0.19	0.15	0.02	0.71
All	Ag (g/t)	17,137	1,500.00	0.20	16.33	116,626	684.97	0.20	24.64	16.34	37.06	1373.37	1.50
All	Pb (%)	32,971	1.82	0.00	0.00	116,631	0.936	0.000	0.007	0.004	0.019	0.00	2.82
All	Zn (%)	33,130	2.62	0.00	0.01	116,631	1.661	0.000	0.020	0.016	0.031	0.00	1.56
All	Au Eq (g/t)					116,631	84.50	0.01	1.07	0.53	1.94	3.74	1.81

14.15.2 Resource Classification Parameters

As part of the block gold grade estimation, two other variables were also estimated for each block – which the Author QP would use as the basis to JORC classification. These variables were:

- AU_D – the average distance of samples used.
  - With the Pipe 1, 2 and 4 block model the resulting statistics were:
    - Average distance 28.3 m (mean 25.1 m);
    - Minimum distance 3.4 m;
    - Maximum distance 68.4 m.
  - With the Mungu block model the resulting statistics were:
    - Average distance 31.2 m (mean 25.0 m);
    - Minimum distance 1.9 m;
    - Maximum distance 106.4 m.
- AU_P – the number of samples used. In the ATO case this would range between 1 and 18.

14.15.3 Block Grade Validation

Estimated block grades were initially validated through comparison of the statistics of the source drill hole data and of the interpolated blocks (see estimation statistics Tables above). Thereafter they were checked with cross-sectional plots and through 3D visualisation.

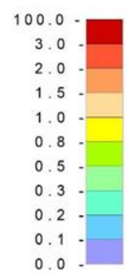
The Author QP did not consider the raw data to have extreme statistics in the first place. Consequently, the interpolated blocks were considered to be acceptable as the reflected the raw data fairly well.

14.15.4 Block Grade Cross-Sections

The following Figures illustrate typical vertical E/W cross-sections through the ATO deposits and the block models.

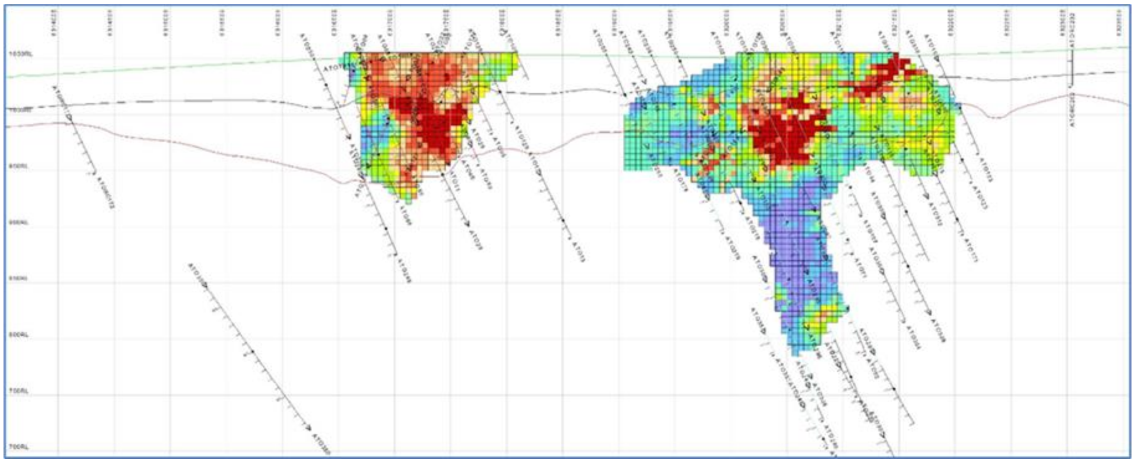
Blocks are colour-coded on gold grades according to the ranges and colours as shown in Figure 14.21. Drill holes are shown projected from up to 10 m either side of the sections. Surface intersection are shown as coloured lines – topography in green, base of oxidation in black, top of fresh (or base of transition) in red.

Figure 14.21 – Gold



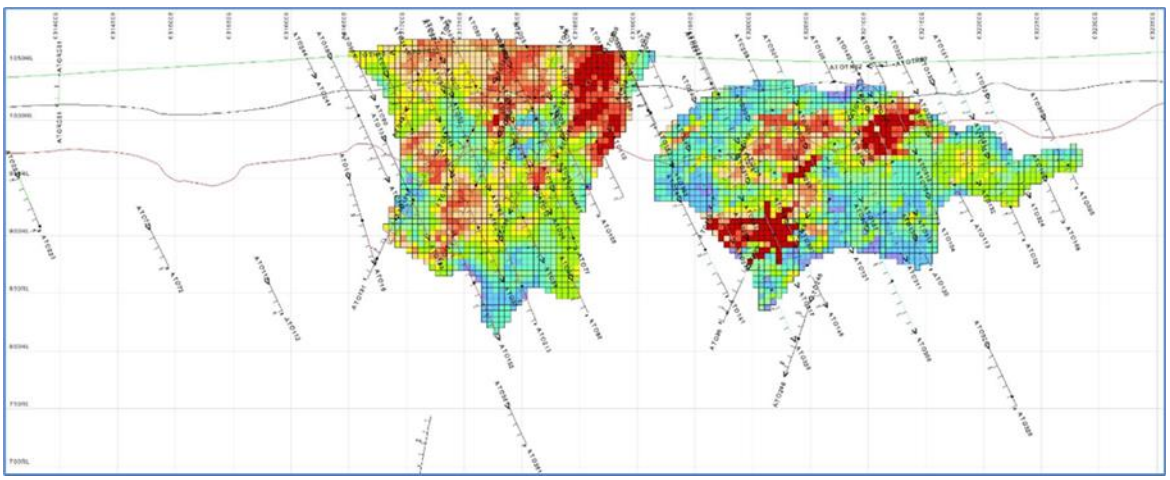
Pipe 1, 2 and 4: Figure 14.22 to Figure 14.26 illustrate a series of east/west cross-sections, looking north, though Pipe 1, 2 and 4 (Pipe 2 on the left (e.g. Figure 14.26, west), Pipe 1 in the middle (e.g. Figure 14.24), and Pipe 4 on the right (e.g. Figure 14.22, east)). The sections are ordered from south to north.

Figure 14.22 - Pipe 1, 2 and 4 Gold Block Cross-Section 5,366,902N



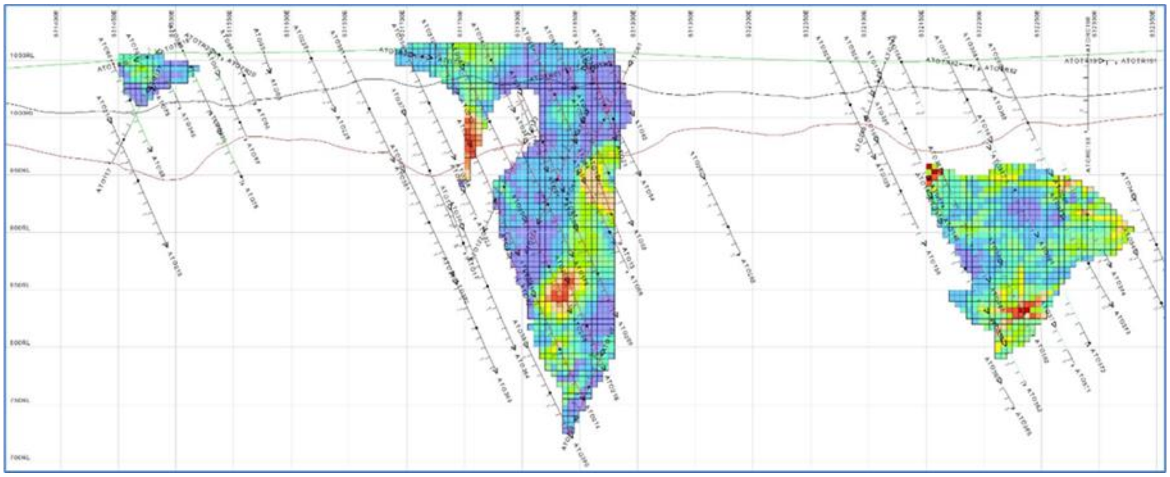
Source: ATO Mineral Resources Technical Report, 2021

Figure 14.23 - Pipe 1, 2 and 4 Gold Block Cross-Section 5,367,002N



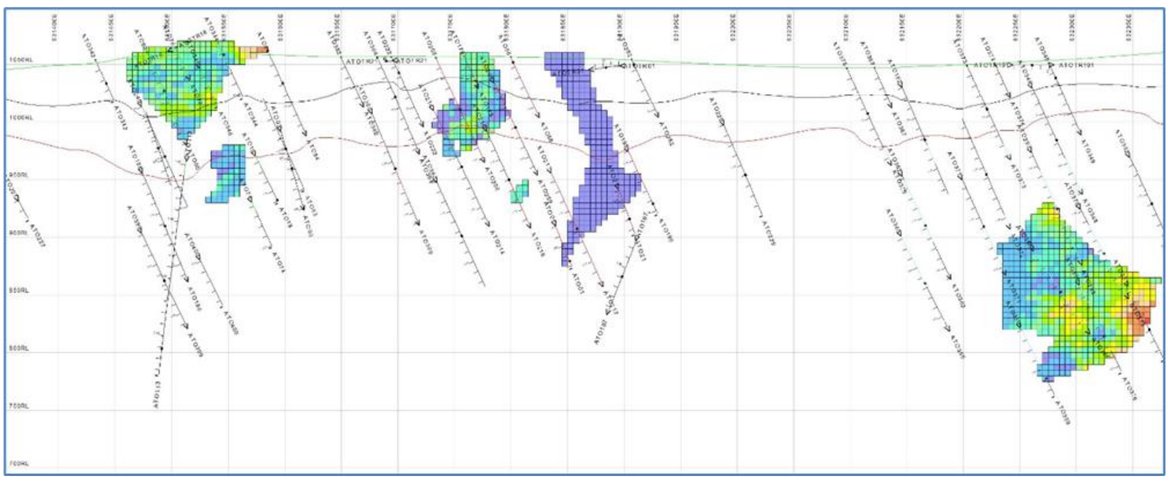
Source: ATO Mineral Resources Technical Report, 2021

Figure 14.24 - Pipe 1, 2, and 4 Gold Block Cross-Section 5,367,102N



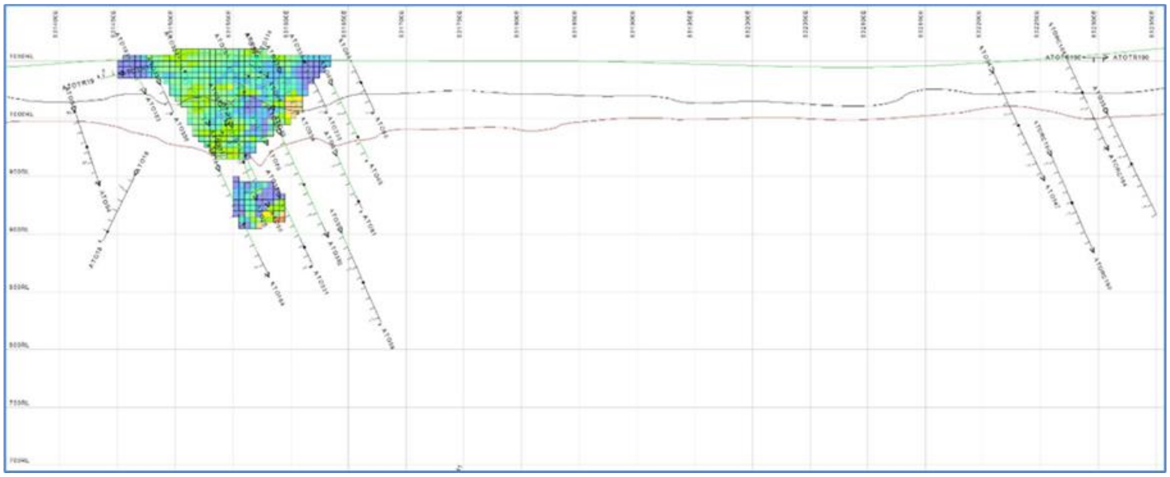
Source: ATO Mineral Resources Technical Report, 2021

Figure 14.25 - Pipes 1, 2, and 4 Gold Block Cross-Section 5,367,152N



Source: ATO Mineral Resources Technical Report, 2021

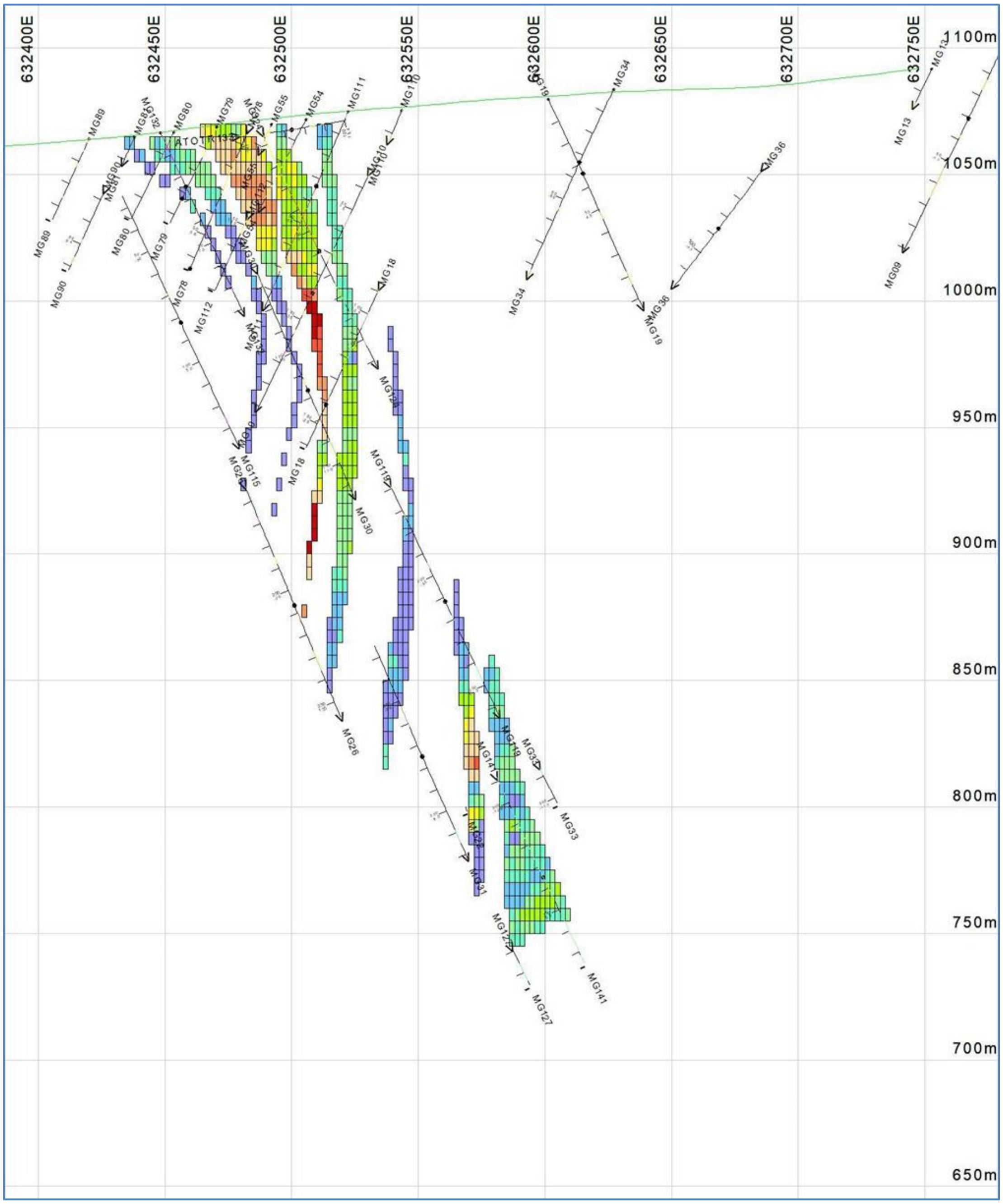
Figure 14.26 - Pipe 1, 2, and 4 Gold Block Cross-Section 5,367,252N



Source: ATO Mineral Resources Technical Report, 2021

Mungu: Figure 14.27 to Figure 14.30 illustrate a series of east/west cross-sections, looking north, though Mungu, showing colour-coded gold blocks. The sections are from south to north.

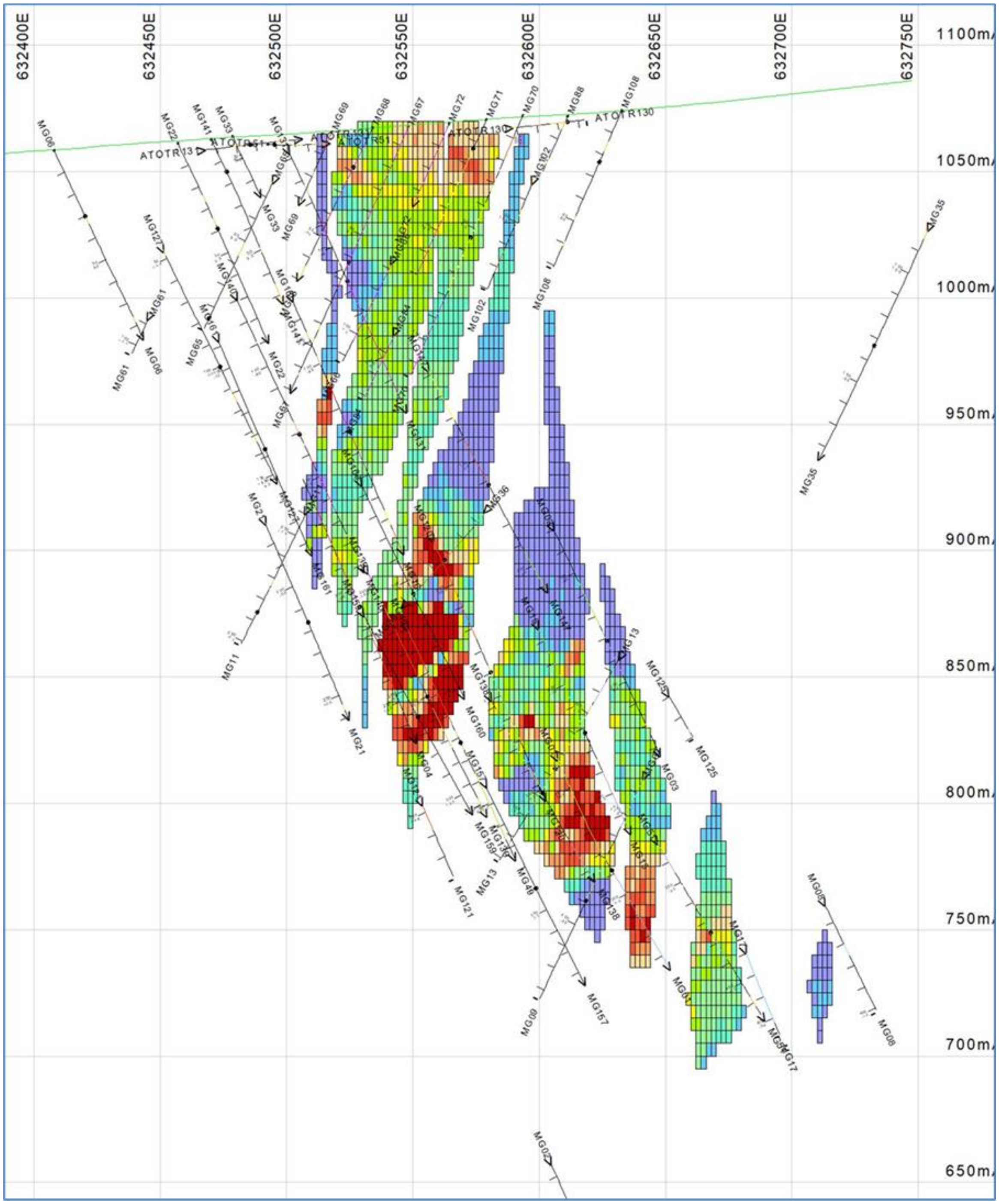
Figure 14.27 - Mungu Gold Block Cross-Section 5,367,502N



Source: ATO Mineral Resources Technical Report, 2021

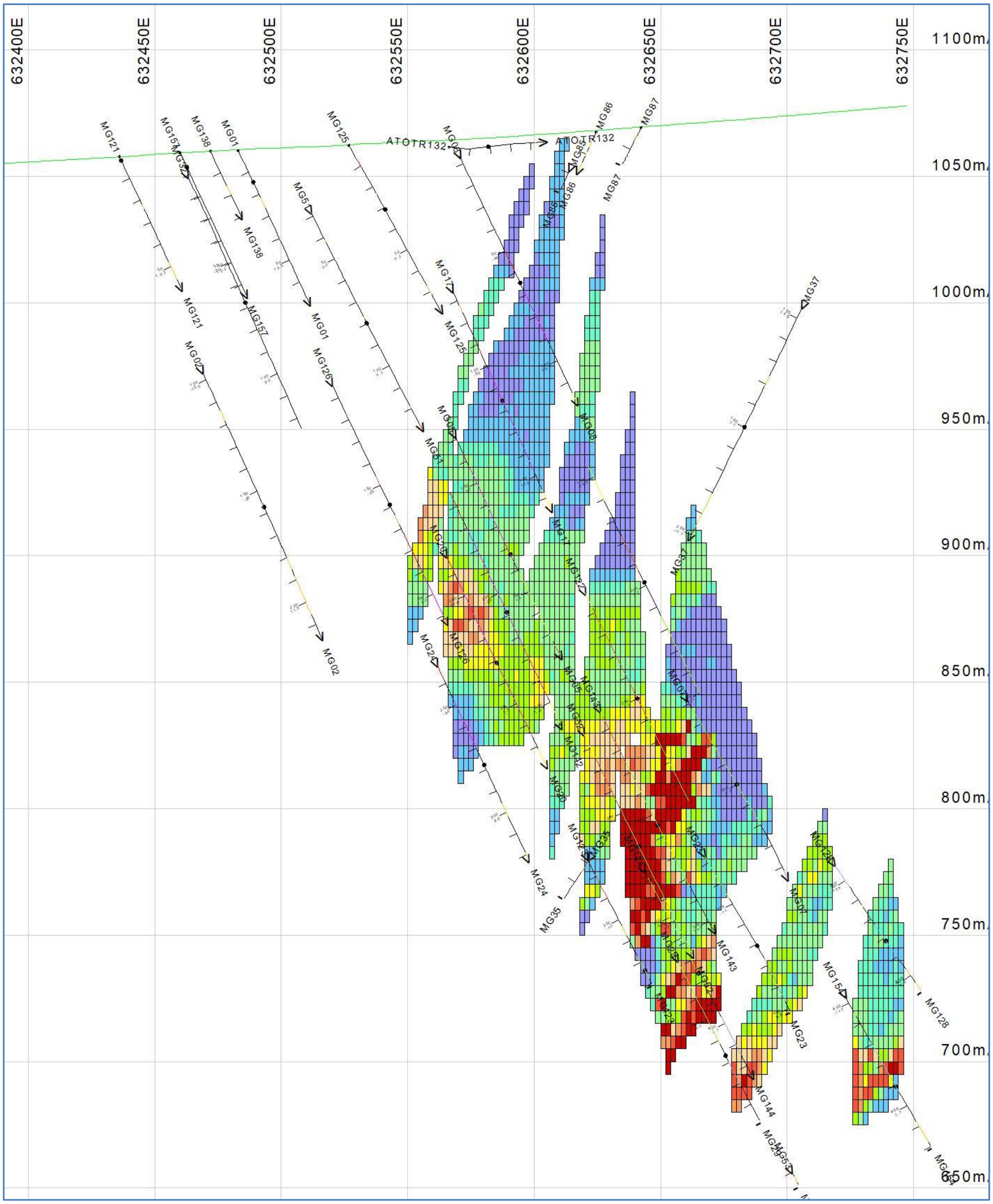
Figure 14.28 of cross-section 5,367,602N illustrates colour-coded block gold grades and the ~375 vertical extent of the deposit and the ~100 m width east-west.

Figure 14.28 - Mungu Gold Block Cross-Section 5,367,602N



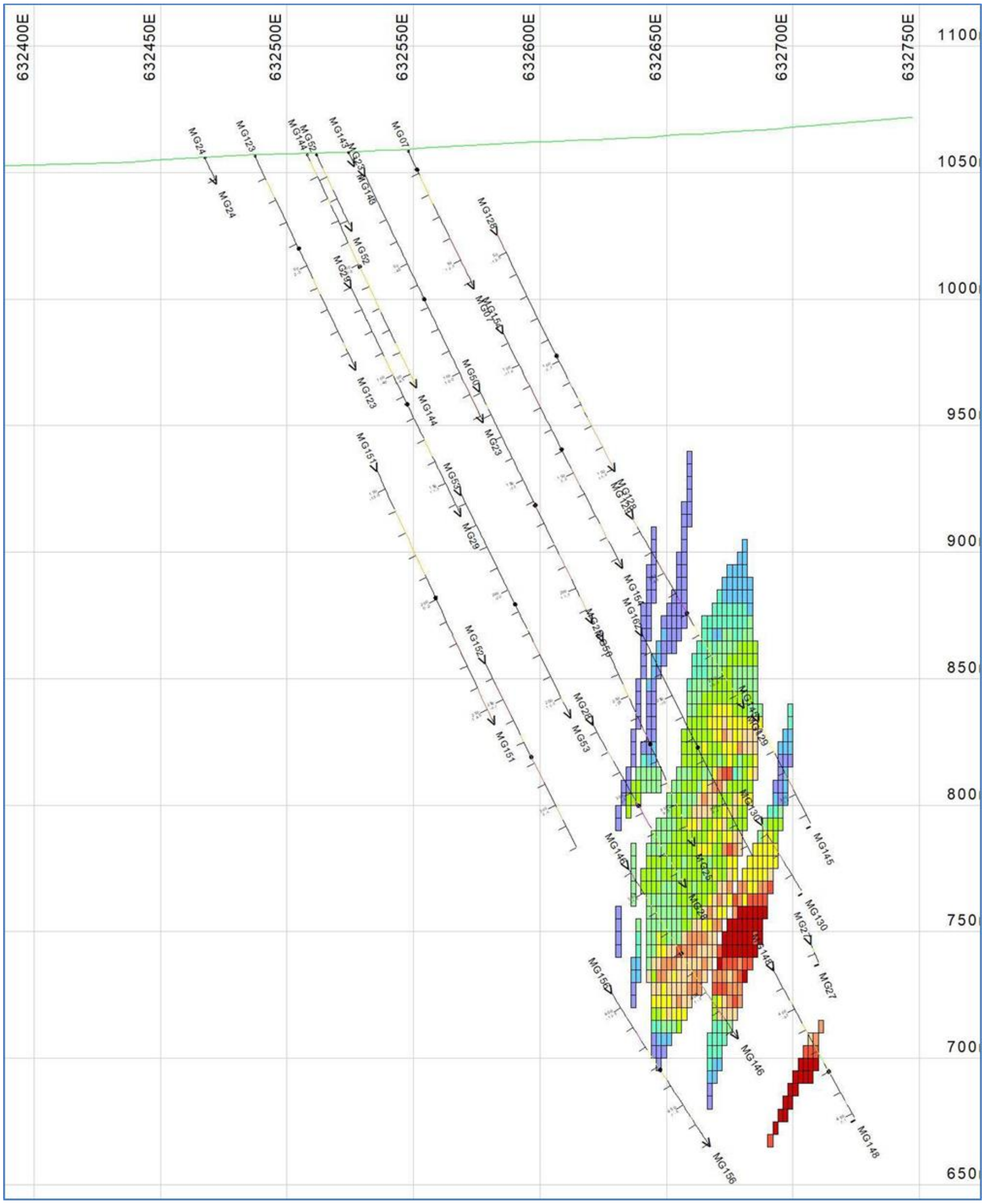
Source: ATO Mineral Resources Technical Report, 2021

Figure 14.29 - Mungu Gold Block Cross-Section 5,367,702N



Source: ATO Mineral Resources Technical Report, 2021

Figure 14.30 - Mungu Gold Block Cross-Section 5,367,802N



Source: ATO Mineral Resources Technical Report, 2021

Mungu: Figure 14.31 to Figure 14.33 illustrate a series of typical level cross-sections, looking vertically down, though Mungu, showing colour-coded gold blocks. The sections are top downward.

Figure 14.31 - Mungu Level 900RL

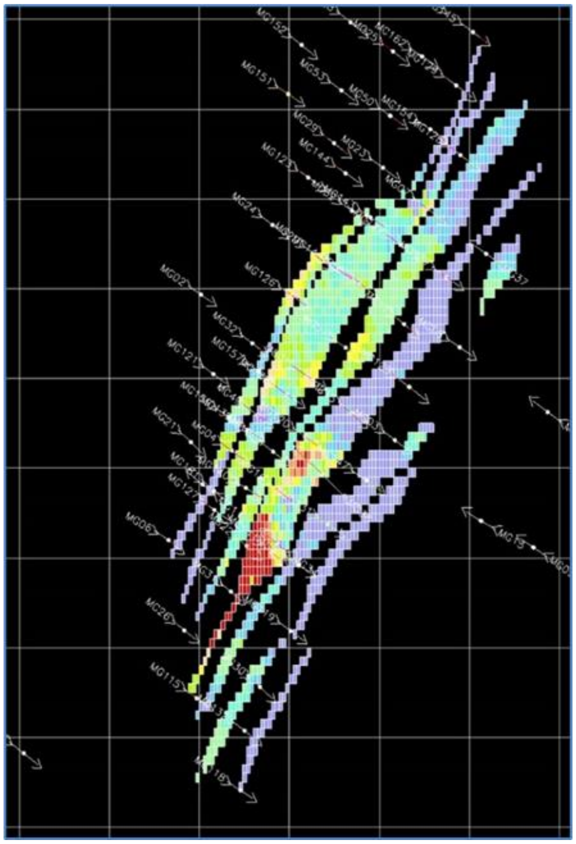


Figure 14.32 - Mungu Level 850RL

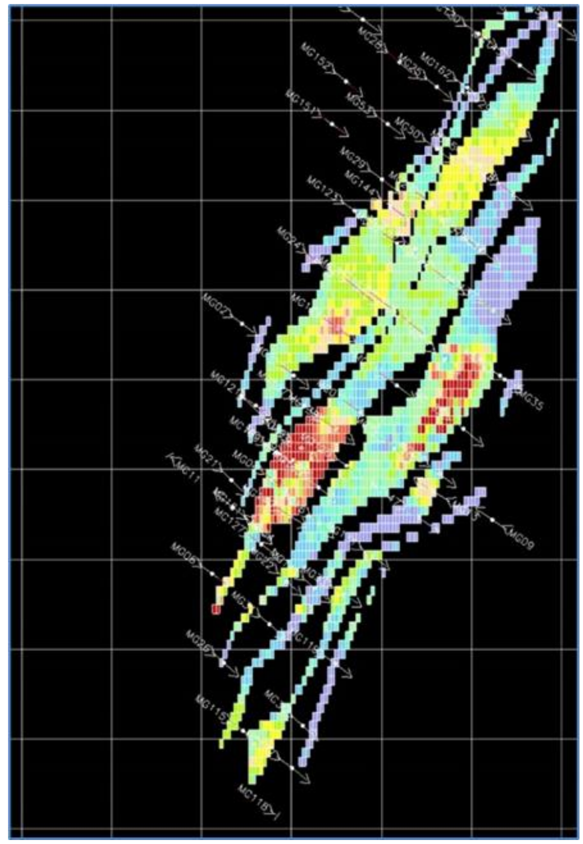
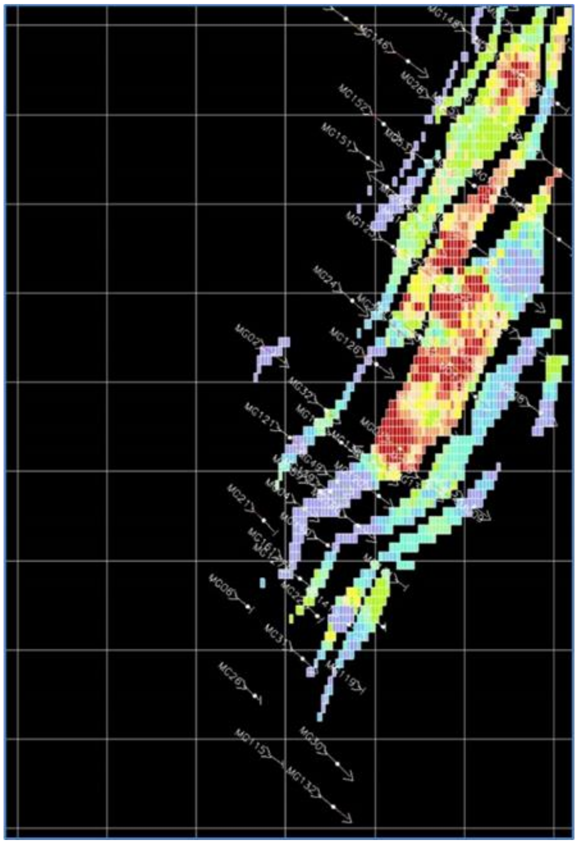


Figure 14.33 - Mungu Level 800RL



14.16 Block 'Gold Equivalent' Grade Calculation

A "gold equivalent" block grade (AuEq in g/t) was calculated to account for and add in the value of mineralisation other than gold at the Project. This variable was used as the lower grade cut-off in the Resource reporting.

The gold equivalent was calculated in each block from the individually estimated grades of the elements in the block. The formula was based on ~30 day averages of metal sales prices published on Kitco for the month before 11 January 2021. So, the formula was effective mid-January 2021.

The AuEq formula was:

Au_Eq = Au + ((Ag * AgF) + (Pb * PbF) + (Zn * ZnF))

where the "F" for each element was the Price Factor (P) of the element relative to (divided by) gold (e.g. AgF = PAg/PAu).

So, where a unit price of gold was reduced to 1.0 the factors for the elements were:

- Gold (Au) 1.000;
- Silver (Ag) 0.014;
- Lead (Pb %) 0.332;
- Zinc (Zn %) 0.463.

Copper (Cu %) was excluded because the grades were so trivial.

The factors (F) were based on the logic as shown in Table 14.8.

Table 14.8 - Price Factors for Gold Equivalent Calculation (Effective Mid-January 2021)

International price (Kitco)			Metric price			Convert to price for a single unit of grade (eg 1% or 1 g/t)				AuEq
Element x	Listed price	Units	Unit conversion	Metric price	Units	Grade units	Grade conversion	Unit price Px	Units	Ratio factor Fx (Px/PAu)
Zn	0.88	US$/lb	2204.62	1,940	US$/t	%	100	19.40	US$/1%	0.462
Cu	3.55	US$/lb	2204.62	7,826	US$/t	%	100	78.26	US$/1%	1.864
Pb	0.84	US$/lb	2204.62	1,852	US$/t	%	100	18.52	US$/1%	0.441
Au	1,306	US$/oz	31.1035	41.9888	US$/g	g/t	1	41.99	US$/g	1.000
Ag	21.6	US$/oz	31.1035	0.6945	US$/g	g/t	1	0.69	US$/g	0.017
Sn ???		US$/lb	2204.62	0	US$/t	%	100	0.00	US$/1%	0.000

This gold equivalent calculation was effectively the same as the 2017 Resource estimation[21] – with the exception that no metal recoveries were applied here and the formula was applied in the same way to all blocks (ie not differently to oxide and fresh rock).

14.17 Bulk Density

Bulk density was studied for the 2017 Resource estimation by determining values for 226 samples. Those same results were used here also. Average values were determined for the three oxidation levels as:

- Oxide 2.46 t/m^3;
- Transitional 2.59 t/m^3;
- Fresh 2.64 t/m^3.

14.18 Resource Classification (CIM/JORC)

JORC Resource classification required distilling geological and data factors into a decision on the potential classification level and then developing a scheme to implement that classification. To some extent this decision would require consideration of the past classification used for the 2017 estimate.

Classification is described in terms of:

21 2017 NI 43-101. Section 14.8, pp140-141.

- Level and justification – what classes the Author QP decided.
- Method – how to implement classification based on estimation distances and numbers of samples.
- Criteria – the values used to differentiate Resource classes.
- Cross-sections – through the classes to illustrate distribution.

14.18.1 Classification Level and Thinking

In the previous 2017 Resources, a high 93% of estimated blocks by tonnage in the Pipes 1, 2 and 4 deposits were classified as Measured and Indicated, with Measured representing 64% and Indicated representing 36%. The Author QP would consider those proportions to have been relatively too high given:

- the mid-range exploration status of the Property at the time (not particularly developed and exposed);
- the arguably maiden status of the Resource reporting.
- and the Author's view that the geological model (based as it was on grade shells) took comparatively little account of geology.

Notwithstanding the Author's QP slightly negative view of the past classification his considered opinion here is that the bulk of material should still be partly classified as Measured[22] and partly as Indicated[23]. Resources but with a slightly higher proportion (than in the past) of Indicated (42%, Table 60) given the rigour required to meet the Measured status. Peripheral material (surrounding the other classes where drilling information declines) should be classified as Inferred[24] Resources.

With regard to classification decisions, he considers overall that:

- The deposits are well, closely (~30 m), and fairly uniformly drilled.
- In-fill drilling since the 2017 estimate has very largely confirmed the previous results, thus raising confidence.
- Mineralised zones are very clearly continuous over multiple adjacent drill holes, thus giving confidence in the drill hole spacing and geological deposit interpretation.
- The good continuity and compact nature (shape) of the deposit shapes (particularly at Pipe 1, 2 and 4) lends great support to allow detailed down-stream mine planning.
- The mine planning mentioned above would no doubt be optimised after further exploration (particularly of the Mungu deposit with its deeper aspects) to fully 'close-out' the deposits.

22 JORC Code (2012 Edition), point 23, pp13.
23 JORC Code (2012 Edition), point 22, pp13.
24 JORC Code (2012 Edition), point 21, pp12.

- The lack of bulk sampling from wide openings and/or physical access to the deposits at surface and depth are mitigated by the high-quality core drilling method overwhelmingly employed for exploration (which allows a reasonable visual appreciation of the rock as well as facilitating geotechnical analysis).

All points justify the Indicated Resource classification and points 4 to 6 further justify the Measured Resource classification.

14.18.2 Classification Method

The JORC classification used here differed from the method used in the 2017 estimate. That classification was based on reporting Measured Resources from a first pass grade estimate, Indicated Resources from a second and Inferred Resources from a third[25]. The passes differed in estimation parameter, essentially increasing scan distances relating to different components of the geostatistical variograms.

For JORC classification here the Author QP used combinations of the average sample distances (D) and number of samples/points (P) stored for each block during the single-pass block gold grade estimation. Ranges of these D and P values would then be decided based on statistics, distribution and concepts for the Resource. These ranges would then be combined to compute vales into a block categorisation (CAT) using an SQL macro.

The CAT block value would be set to 3 for Measured, 2 for Indicated or 1 for Inferred. This value would be used to subdivide the Resource reporting into classes.

14.18.3 Classification Criteria

Based on drill hole sample statistics, average drill hole spacing and inspection of the D and P values on cross-section, the criteria in Table 14.9 were developed. This process was adapted iteratively by seeing where the resultant classifications were distributed on cross-section – with the aim of developing combinations which would have a large degree of spatial continuity (and avoiding the 'spotted dog' pattern).

With the dense and fairly equi-spaced drill hole data by far the principal component of the classification was the distance (D) variable.

25 2017 NI 43-101, Section 14.9.1, pp 141.

Table 14.9 - JORC Classification Criteria

	Average	Number	Class
Resource	distance	points	
class	AU_D	AU_P	AU_CAT
	(m)	(#)	
Measured	≤27.5	≥12	3
Indicated	≤35.0	≥6	2
Inferred	>35.0	≥1	1

It was decided not to refine the classifications further, based on physical location (such as by elevation of actual digitised areas), because the existing classification was considered adequate.

14.18.4 Block Classification Cross-Sections

Pipes 1 and 4: Figure 14.34 to Figure 14.36 illustrate the distance (D), points (P) and eventual classification (CAT) on a typical block cross-section at the Pipes 1 and 4 deposits (Pipe 2 to the west is completely north of this cross-section).

In these plots the block colouring scheme is:

- Distance:

  ≤27.5 m orange (≈Measured) ≤35.0 m yellow (≈Indicated) >35m blue (≈Inferred).

- Points:

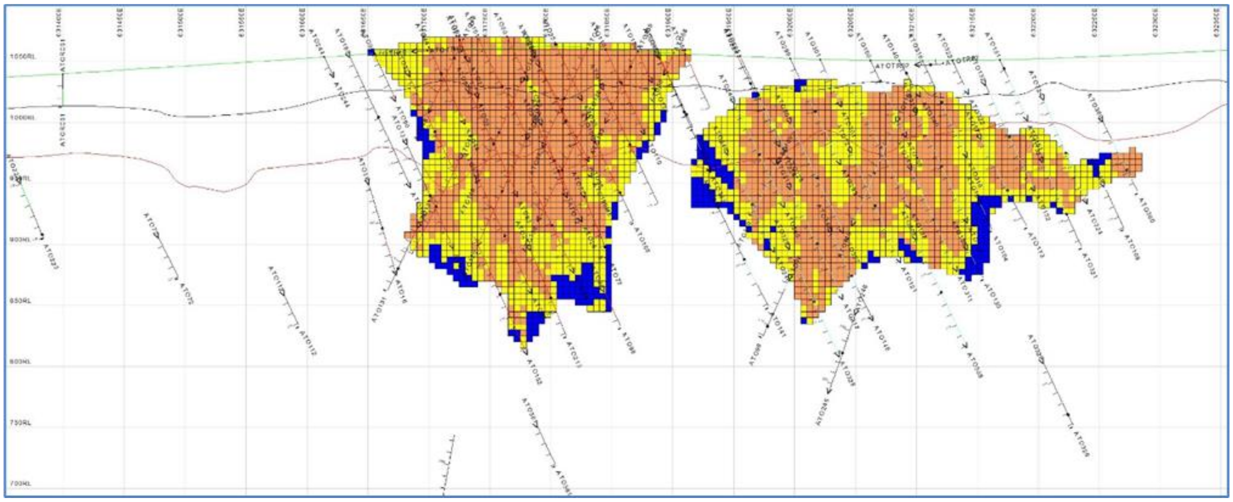
  ≥12 orange (≈Measured) ≥6 yellow (≈Indicated) ≥ blue (≈Inferred).

- Classification:

  3 orange (Measured) 2 yellow (Indicated) 1 blue (Inferred).

The distance plot as shown in Figure 14.34 clearly illustrates the preponderance of areas of close-spaced (<27.5 m) drilling (orange). It also illustrates that there are proportionately very few blocks where the average estimation distance is >35 m (blue).

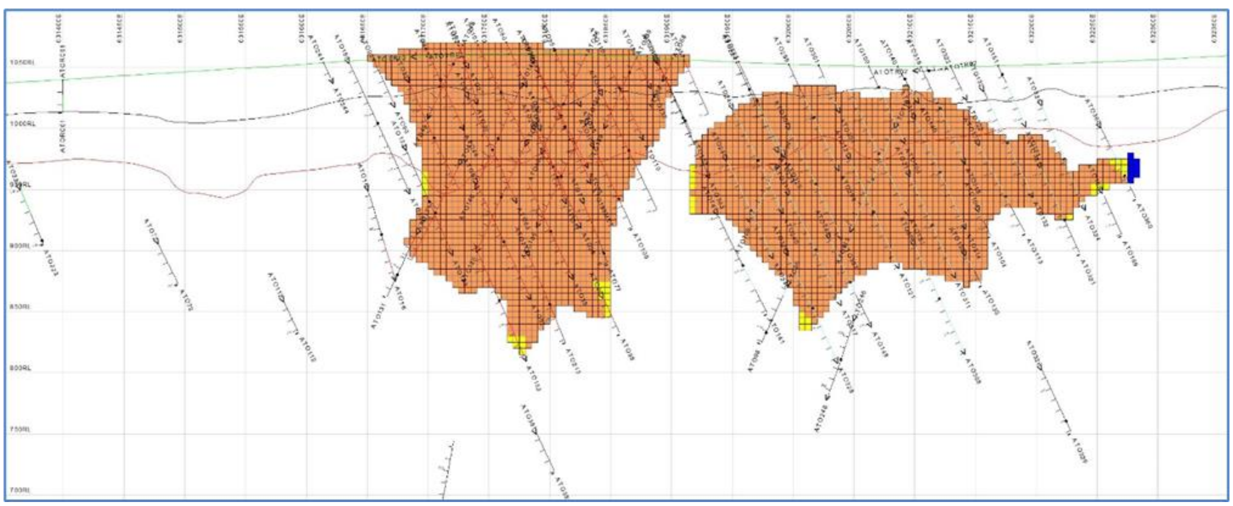
Figure 14.34 - Pipes 1 and 4 Distance (D) Block Cross-Section 5,367,000N



Source: ATO Mineral Resources Technical Report, 2021

The points plot as shown in Figure 14.35 illustrates that virtually all blocks had at least 12 points (orange) in their estimation, even the blocks at the edge of modelled deposits.

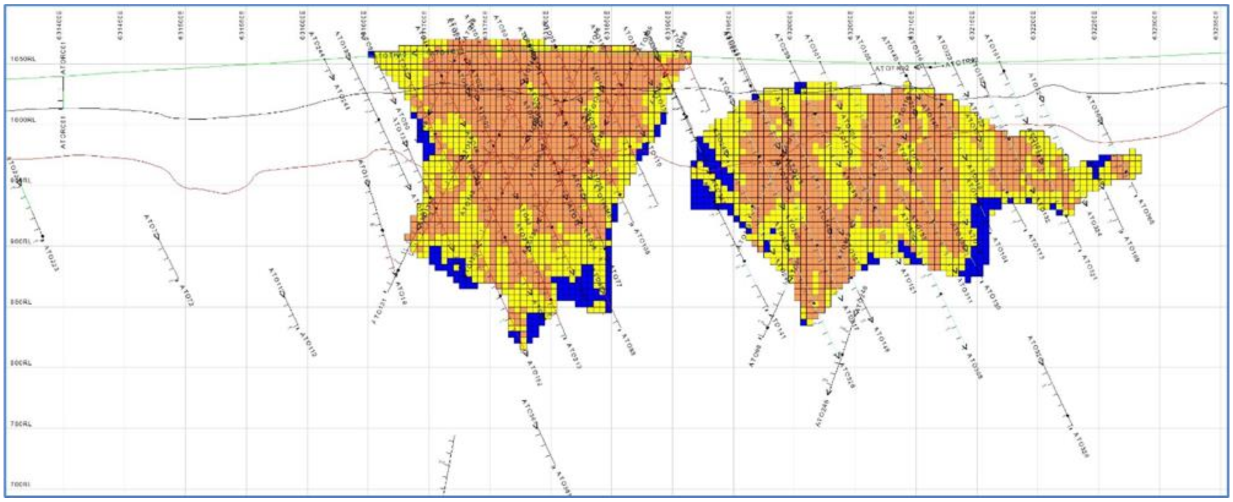
Figure 14.35 - Pipes 1 and 4 Points (P) Block Cross-Section 5,367,000N



Source: ATO Mineral Resources Technical Report, 2021

The eventual classification as shown in Figure 14.36 is seen to virtually mirror the distance plot on this central cross-section to Pipes 1 and 4.

Figure 14.36 - Pipes 1 and 4 Classification (CAT) Block Cross-Section 5,367,000N



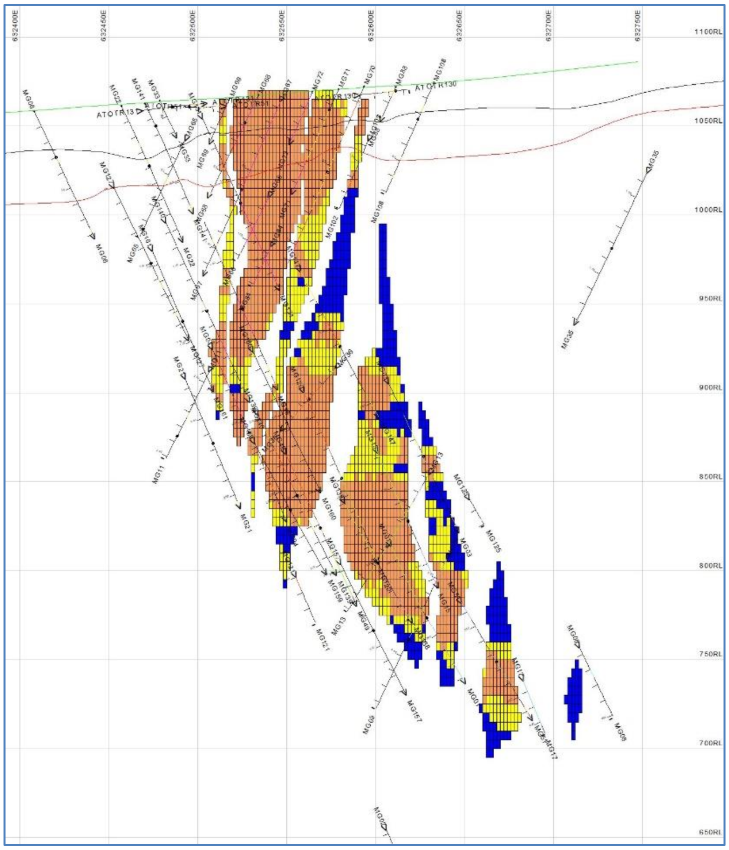
Source: ATO Mineral Resources Technical Report, 2021
Mungu: Figure 14.37 to Source: ATO Mineral Resources Technical Report, 2021

Figure 14.39 illustrate the distance (D), points (P) and eventual classification (CAT) on a typical block cross-section at the Mungu deposit.

The comments made above for the plots for distance, points and classification in Pipes 1 and 4 also very largely apply to the Mungu plots below.

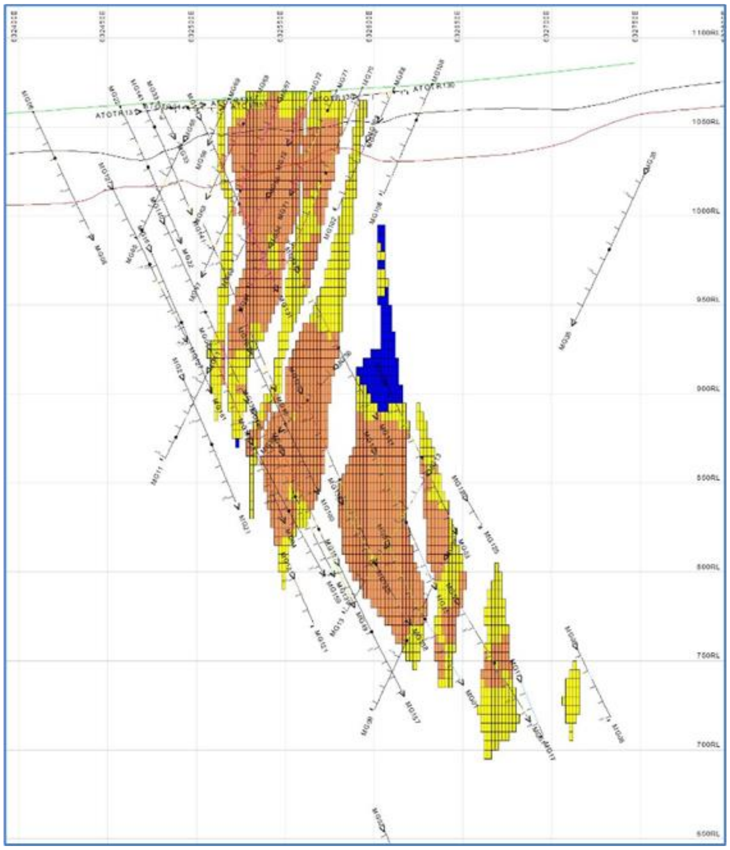
In terms of distance the lower deeper parts of Mungu were only drilled in a steeply dipping zone ~100 m wide and hence the deposit has zones with longer distances.

Figure 14.37 - Mungu Distance (D) Block Cross-Section 5,367,600N



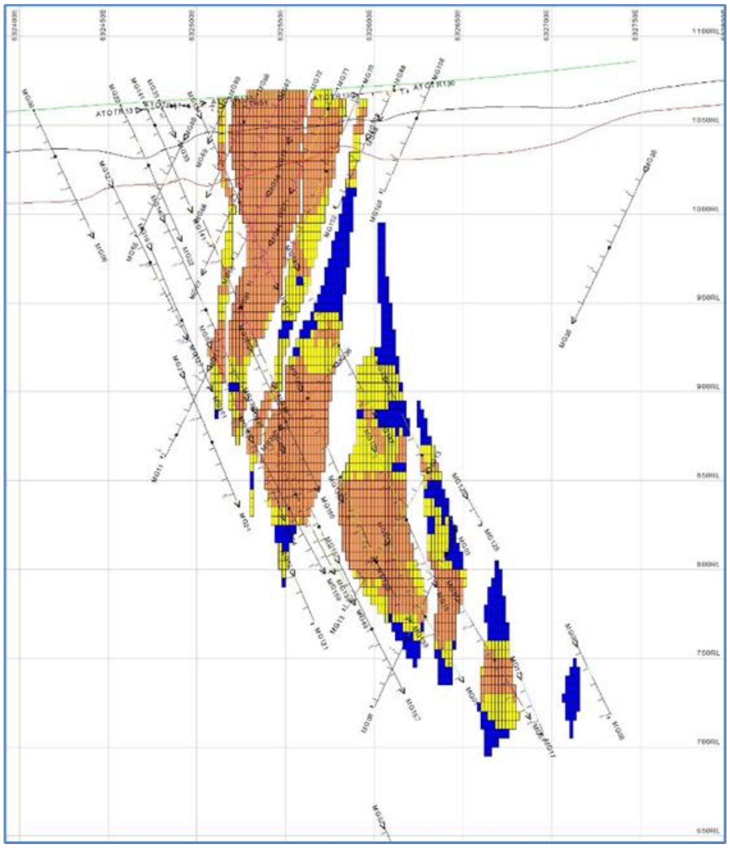
Source: ATO Mineral Resources Technical Report, 2021

Figure 14.38 - Mungu Points (P) Block Cross-Section 5,367,600N



Source: ATO Mineral Resources Technical Report, 2021

Figure 14.39 - Mungu Classification (CAT) Block Cross-Section 5,367,600N



Source: ATO Mineral Resources Technical Report, 2021

14.19 ATO 2021 JORC Mineral Resources

The Author QP reports here the Minerals Resources estimated in late 2021 and early 2021 for the ATO Project. The effective date of these reports is February 2021 when Steppe issued a press release[26].

Requisite statements, certifications and declarations by the Author QP are made above to satisfy the reporting codes governing these Resources. This JORC classification is directly equivalent to CIM categorisation.

The basis for the JORC classification here is given above. The disclosure of Resource categories is specifically governed by NI 43-101 (in contrast to straight JORC reporting) and precludes the addition of Inferred Resources to Measured and Indicated Resources – hence they are reported separately below.

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[26] Steppe, 24 February 2021. Press release through Newsfile Corp.

Cut-off grades: Lower cut-off grades to use in reporting were applied by oxidation level. Values used in the 2017 reporting[27] were 0.3 g/t AuEq for oxide material and 1.1 g/t AuEq for fresh material (it is not clear which level the transitional material was grouped with), considerably different to those used here.

Lower grade cut-offs used here were applied to the AuEq variable, were stipulated by Steppe, and were:

- Oxide 0.15 g/t AuEq;
- Transitional 0.40 g/t AuEq;
- Fresh 0.40 g/t AuEq.

Bulk density: Bulk densities were applied by oxidation level, were described above, and were:

- Oxide 2.46 t/m^3;
- Transitional 2.59 t/m^3;
- Fresh 2.64 t/m^3.

Reporting by oxidation level: JORC (2012 Edition) classified Measured and Indicated classes of in-situ Global Mineral Resources of gold and related precious and base metals are reported by oxidation level for the ATO Project, as of February 18, 2021, in Table 14.10. These Resources supersede any previously reported. Here, the deposit names ATO1, ATO2 and ATO4 are interchangeable with names Pipes 1, Pipe 2 and Pipe 4 mentioned in the Report and in the past. Bulk densities were applied by oxidation level, as were lower gold equivalent grade cut-offs (both given above). Numbers have been rounded and may not sum exactly.

27 2017 NI 43-101. Section 14.9.4, pp145.

Table 14.10 - ATO 2021 Measured & Indicated In-Situ Mineral Resources - By Oxidation Level

ATO - JORC Classified Resources by oxidation surface level. Reported 18 February 2021 (V3).													
CLASS BY OXIDATION LEVEL	Deposit	Cut-off AuEq (g/t)	Bulk density (t/m^3)	Tonnes (M t)		Grades Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	AuEq (g/t)	Metal Au (k oz)	Ag (k oz)	AuEq (k oz)
MEASURED													
Oxide (above OX_SF)	ATO1	0.15	2.46	2.6		1.22	9.34	0.59	0.33	1.70	103	786	143
	ATO2	0.15	2.46	1.1		0.46	3.58	0.40	0.33	0.79	17	130	29
	ATO4	0.15	2.46	0.7		0.99	20.49	0.17	0.28	1.46	23	471	34
	Mungu	0.15	2.46	0.3		0.66	25.21			1.01	7	258	10
	TOTAL	**0.15**	**2.46**	**4.8**	**20%**	**0.97**	**10.71**	**0.44**	**0.30**	**1.40**	**149**	**1,645**	**216**
Transition (between OX_SF and TR_SF)	ATO1	0.40	2.59	3.4		1.45	6.46	0.72	1.16	2.31	160	712	255
	ATO2	0.40	2.59	0.4		0.33	4.07	0.64	1.40	1.25	4	55	17
	ATO4	0.40	2.59	2.4		1.41	17.24	0.21	0.39	1.90	111	1,356	149
	Mungu	0.40	2.59	0.4		0.69	39.49		0.01	1.25	8	448	14
	TOTAL	**0.40**	**2.59**	**6.6**	**28%**	**1.32**	**12.03**	**0.49**	**0.83**	**2.04**	**283**	**2,570**	**435**
Fresh (below TR_SF)	ATO1	0.40	2.64	3.8		0.78	5.25	0.75	1.52	1.80	94	635	218
	ATO2	0.40	2.64	0.1		0.42	5.64	1.07	2.73	2.12	2	21	8
	ATO4	0.40	2.64	4.4		1.32	8.53	0.36	0.63	1.85	185	1,197	260
	Mungu	0.40	2.64	4.3		1.41	45.16	0.01	0.03	2.06	194	6,206	283
	TOTAL	**0.40**	**2.64**	**12.5**	**52%**	**1.18**	**20.03**	**0.36**	**0.71**	**1.91**	**475**	**8,058**	**768**
MEASURED	**TOTAL**			**23.9**	**58%**	**1.18**	**15.95**	**0.41**	**0.66**	**1.84**	**907**	**12,274**	**1,419**
INDICATED													
Oxide (above OX_SF)	ATO1	0.15	2.46	1.1		0.84	8.80	0.50	0.25	1.25	30	314	45
	ATO2	0.15	2.46	1.0		0.44	3.55	0.33	0.26	0.72	15	118	24
	ATO4	0.15	2.46	0.9		0.85	20.28	0.13	0.22	1.28	25	587	37
	Mungu	0.15	2.46	0.2		0.43	15.30			0.65	3	99	4
	TOTAL	**0.15**	**2.46**	**3.2**	**18%**	**0.69**	**10.72**	**0.31**	**0.23**	**1.05**	**72**	**1,118**	**110**
Transition (between OX_SF and TR_SF)	ATO1	0.40	2.59	1.3		1.04	5.96	0.64	1.08	1.83	45	256	79
	ATO2	0.40	2.59	0.3		0.44	4.67	0.64	1.35	1.34	4	46	13
	ATO4	0.40	2.59	1.9		1.26	21.78	0.13	0.29	1.74	79	1,359	109
	Mungu	0.40	2.59	0.1		0.55	24.55		0.01	0.90	2	75	3
	TOTAL	**0.40**	**2.59**	**3.7**	**21%**	**1.09**	**14.67**	**0.35**	**0.66**	**1.72**	**129**	**1,737**	**203**
Fresh (below TR_SF)	ATO1	0.40	2.64	3.0		0.66	4.54	0.69	1.44	1.62	63	433	154
	ATO2	0.40	2.64	0.1		0.54	5.96	1.14	2.98	2.38	3	29	11
	ATO4	0.40	2.64	5.3		0.86	12.06	0.27	0.49	1.35	147	2,060	231
	Mungu	0.40	2.64	2.3		0.93	37.85	0.01	0.03	1.47	70	2,830	110
	TOTAL	**0.40**	**2.64**	**10.8**	**61%**	**0.82**	**15.48**	**0.34**	**0.69**	**1.46**	**282**	**5,351**	**506**
INDICATED	**TOTAL**			**17.7**	**42%**	**0.85**	**14.44**	**0.34**	**0.60**	**1.44**	**483**	**8,206**	**819**
MEASURED + INDICATED													
Oxide	TOTAL	0.15	2.46	8.0	19%	0.86	10.72	0.39	0.27	1.26	221	2,763	325
Transition	TOTAL	0.40	2.59	10.3	25%	1.24	12.97	0.44	0.77	1.92	412	4,307	638
Fresh	TOTAL	0.40	2.64	23.3	56%	1.01	17.93	0.35	0.70	1.70	757	13,409	1,274
MEAS + IND	**TOTAL**			**41.6**		**1.04**	**15.31**	**0.38**	**0.63**	**1.67**	**1,390**	**20,479**	**2,238**

JORC classified Inferred Resources of in-situ Global Mineral Resources of gold and related precious and base metals are reported by oxidation level for the ATO Project, as of February 18, 2021, in Table 14.11.

Table 14.11 - ATO 2021 Inferred In-Situ Mineral Resources- By Oxidation Level

ATO - JORC Classified Resources by oxidation surface level. Reported 18 February 2021 (V3).													
CLASS BY OXIDATION LEVEL	Deposit	Cut-off AuEq (g/t)	Bulk density (t/m^3)	Tonnes (M t)		Grades					Metal		
						Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	AuEq (g/t)	Au (k oz)	Ag (k oz)	AuEq (k oz)
INFERRED													
Oxide	ATO1	0.15	2.46	0.2		0.45	6.10	0.28	0.22	0.73	2	32	4
(above OX_SF)	ATO2	0.15	2.46	0.3		0.28	3.26	0.21	0.24	0.51	2	27	4
	ATO4	0.15	2.46	0.2		0.73	10.83	0.10	0.16	0.99	5	74	7
	Mungu	0.15	2.46	0.1		0.37	9.41			0.50	1	22	1
	TOTAL	**0.15**	**2.46**	**0.7**	**13%**	**0.46**	**6.83**	**0.17**	**0.19**	**0.70**	**11**	**155**	**16**
Transition	ATO1	0.40	2.59	0.1		1.16	6.49	0.63	1.24	2.03	4	23	7
(between OX_SF	ATO2	0.40	2.59	0.1		0.39	8.12	1.17	2.21	1.91	1	26	6
and TR_SF)	ATO4	0.40	2.59	0.1		0.66	16.15	0.08	0.23	1.02	1	36	2
	Mungu	0.40	2.59	0.1		0.45	14.84		0.01	0.66	1	25	1
	TOTAL	**0.40**	**2.59**	**0.3**	**6%**	**0.71**	**10.34**	**0.58**	**1.12**	**1.56**	**8**	**110**	**17**
Fresh	ATO1	0.40	2.64	0.9		0.44	3.88	0.58	1.36	1.32	12	109	37
(below TR_SF)	ATO2	0.40	2.64	0.1		0.17	9.45	1.45	3.27	2.29	1	31	8
	ATO4	0.40	2.64	2.0		0.59	15.30	0.20	0.37	1.04	38	989	67
	Mungu	0.40	2.64	1.6		0.85	26.11	0.01	0.02	1.23	44	1,339	63
	TOTAL	**0.40**	**2.64**	**4.6**	**82%**	**0.64**	**16.75**	**0.23**	**0.50**	**1.19**	**95**	**2,468**	**175**
INFERRED	**TOTAL**			**5.6**		**0.62**	**15.13**	**0.25**	**0.50**	**1.15**	**113**	**2,732**	**208**

Reporting by class: Table 14.12 reports similar Measured and Indicated Resources to those in Table 14.10 – except summarised by class.

Table 14.12 - ATO 2021 Measured & Indicated In-Situ Mineral Resources - By Class

ATO - JORC Classified Resources by Deposit. Reported 18 February 2021 (V3).											
CLASS BY DEPOSIT	Deposit	Tonnes (M t)		Grades					Metal		
				Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	AuEq (g/t)	Au (k oz)	Ag (k oz)	AuEq (k oz)
MEASURED	ATO1	9.8		1.13	6.76	0.70	1.08	1.95	357	2,133	616
	ATO2	1.7		0.42	3.84	0.51	0.76	1.00	23	205	53
	ATO4	7.5		1.32	12.50	0.29	0.52	1.83	319	3,024	443
	Mungu	4.9		1.31	43.47	0.01	0.03	1.93	208	6,912	308
	TOTAL	**23.9**	**58%**	**1.18**	**15.95**	**0.41**	**0.66**	**1.84**	**907**	**12,274**	**1,419**
INDICATED	ATO1	5.4		0.79	5.77	0.64	1.11	1.60	138	1,003	278
	ATO2	1.5		0.45	4.02	0.48	0.76	1.02	22	193	49
	ATO4	8.2		0.95	15.28	0.22	0.41	1.44	250	4,006	376
	Mungu	2.6		0.88	35.63	0.01	0.03	1.39	74	3,004	117
	TOTAL	**17.7**	**42%**	**0.85**	**14.44**	**0.34**	**0.60**	**1.44**	**483**	**8,206**	**819**
MEAS + IND	ATO1	15.2	37%	1.01	6.41	0.68	1.09	1.83	495	3,137	893
	ATO2	3.1	8%	0.44	3.93	0.49	0.76	1.01	44	398	102
	ATO4	15.7	38%	1.13	13.95	0.26	0.46	1.62	569	7,029	819
	Mungu	7.6	18%	1.16	40.75	0.01	0.03	1.74	282	9,916	424
MEAS+IND	**TOTAL**	**41.6**		**1.04**	**15.31**	**0.38**	**0.63**	**1.67**	**1,390**	**20,479**	**2,238**

Table 14.13 reports similar Inferred Resources to those in Table 14.11 – except summarised by class.

Table 14.13 - ATO 2021 Inferred In-Situ Mineral Resources - By Class

ATO - JORC Classified Resources by Deposit. Reported 18 February 2021 (V3).										
CLASS BY DEPOSIT	Deposit	Tonnes	Grades					Metal		
			Au	Ag	Pb	Zn	AuEq	Au	Ag	AuEq
		(M t)	(g/t)	(g/t)	(%)	(%)	(g/t)	(k oz)	(k oz)	(k oz)
INFERRED	ATO1	1.1	0.51	4.44	0.54	1.19	1.30	19	164	48
	ATO2	0.5	0.28	5.70	0.70	1.34	1.21	4	84	18
	ATO4	2.3	0.61	14.91	0.19	0.35	1.03	45	1,098	76
	Mungu	1.7	0.82	25.05	0.01	0.02	1.18	45	1,386	65
INFERRED	**TOTAL**	**5.6**	**0.62**	**15.13**	**0.25**	**0.50**	**1.15**	**113**	**2,732**	**208**

14.20 Reconciliation – of Resources with Other Estimates

Reconciliation of these 2021 Resources could only be done for the three deposits al reported in the 2017 estimate (Pipes 1, 2 and 4). No data existed to reconcile the Mungu deposit against. Reconciliation was approximated to account for differences in estimate reporting parameters (principally cut-off grade) between 2017 and 2021.

The 2017 report used lower AuEq cut-off grades of 0.30 g/t in Oxide and 1.10 g/t in Fresh as opposed to the 0.15 g/t in Oxide and 0.40 g/t in Transition and Fresh here. And the actual Oxide/Fresh interface surface models would have differed to some degree in interpretation. However, a reasonable approximation was made by excluding material reported below 0.30 g/t from the 2017 Resources reported with a lower cut-ff of 0.1 g/t (by using the 2017 Resources broken-down by Grade Group[28]). As the 2017 reporting omitted some elements (AuEq, Pb, and Zn) grade comparisons were only possible for Au and Ag.

Comparable Measured and Indicated Resources for equivalent deposits (Pipes 1, 2 and 4) and similar lower AuEg cut-offs (~0.3 g/t) are given in Table 14.14 for 2017 reporting (light blue) and 2021 reporting (light green).

Table 14.14 - Comparable Resource Reconciliation 2017 / 2021

Estimate	Resource Class	Tonnes (M t)		Au (g/t)		Ag (g/t)		Au metal (k oz)		Ag metal (k oz)	
GSTATS 2017	Measured	16.2	60%	1.14		7.35		592		3,835	
GSTATS 2017	Indicated	11.0	40%	0.91		9.76		319		3,431	
GSTATS 2017	**Total**	**27.2**		**1.04**		**8.32**		**911**		**7,266**	
GeoRes 2021	Measured	19.0	56%	1.14		8.78		698		5,362	
GeoRes 2021	Indicated	15.1	44%	0.85		10.75		409		5,201	
GeoRes 2021	**Total**	**34.0**		**1.01**		**9.65**		**1,108**		**10,564**	
2021 difference	Measured	2.8	17%	0.01	1%	1.43	19%	107	18%	1,527	40%
2021 difference	Indicated	4.1	37%	-0.06	-7%	0.99	10%	90	28%	1,771	52%
2021 diff	**Total**	**6.9**	**25%**	**-0.03**	**-3%**	**1.33**	**16%**	**197**	**22%**	**3,298**	**45%**

28 2017 NI 43-101. Section 14.9.3, Tables 14.13 and 14.14, pp143-144.

This 2021 Resource contained 25% more tonnes (34.0 Mt vs 27.2 Mt) at a 3% lower Au grade (1.01 g/t vs 1.04 g/t) and a 16% higher Ag grade (9.65 g/t vs 8.32 g/t). These combined to give the 2021 Resource 22% more contained Au metal (1.11 M oz vs 0.91 M oz) and 45% more contained Ag metal (10.56 M oz vs 7.27 M oz).

The QP considers that the 2017 and 2021 Resources can be well reconciled. Whilst the tonnage differences are notable, they are considered to be almost wholly due to the different deposit modelling approaches of the two estimates. And further drilling at the deposit since 2017 was also thought to have increased its volume.

The 2017 estimate used Leapfrog's gold grade shells to define the Resources; this 2021 estimate used detailed section-by-section multi-element mineralisation interpretation. This 2021 estimate better integrates geological deposit shape interpretation with practical mining shapes (essentially more contiguous shapes). They include greater internal low-grade dilution than in 2017 – which increases the volume and reduces the gold grade (albeit by a trivial amount which is almost fully within the Indicated sections). The greater consideration of elements other than gold during the deposit shape interpretation, particularly of silver, had the consequence of including more silver mineralisation and raising the silver grade.

14.21 Potential Impact on Resources by Other Factors

The Author QP was not aware of any other factors (excluding those specifically mentioned below here), including environmental, title, economic, market or political, which could generally or in-particularly influence the Resources reported here for the ATO Project.

Grade Cut-Off:

- In the Author QP's experience the cut-offs used here are comparatively low.
- Raising cut-offs would reduce the Resources. The Author QP has not studied the relationship between cut-off and Resources – but does not believe that raising the cut-off slightly (say to 0.5 g/t AuEq) would reduce Resources significantly.
- However, the Author QP accepts the lower grade cut-offs supplied by Steppe believing that the down-stream mining and extraction analyses performed by Steppe justify the values economically.

Bulk Density:

- The Author QP is satisfied with the number (226) of bulk density determinations carried out for the 2017 Report and assumes that they were taken correctly.
- He notes that this number (226) is considerably more than many other advanced Projects achieve.
- However actual densities could prove to be different and could thus alter Resources.

- However, the Author QP does not consider that density could be significantly different to that used here, and therefore would not have a significant influence on Resources.

Gold Equivalent:

- The gold equivalent calculation was based on international metals prices to mid-January 2021.
- The calculation is most susceptible to changes in the price of gold.
- Metals prices vary with time, and therefore the gold equivalent value would change – which would alter the Resources very slightly because the lower grade cut-off was based on gold equivalent.
- The Author QP has not studied the relationship between prices, gold equivalent and Resources – but does not believe that the scale of price changes normal within the recent past (say a year) would have a significant effect on Resources.

Geological Model:

- The volume of the geological deposit model obviously directly influences the Resource tonnage.
- So, a reduction in model volume would reduce the Resources.
- However, the Author QP does not consider that the current model is over optimistic in size.

JORC Classification:

- The Author QP has previously expressed the opinion that the 2017 reporting of Measured and Indicated Resources as 93% of the total estimated blocks was relatively too high, given the mid-range exploration status of the Project at that time and the comparatively little account of geology in the interpretation.
- Furthermore the 2017 report had Measured as 64% of the total Measured and Indicated Resources.
- Here the Author QP's classification has the Measured and Indicated Resources as being a lower (albeit slightly) 88% of all blocks.
- And here the Measured is a lower 58% of the total Measured and Indicated Resources.
- The Author QP regards the latest class proportions to be more realistic than the 2017 proportions.
- However, the Author QP also considers that the proportion of Measured and Indicated is well supported by the compact and clos drill hole and sample spacing.

Mining Method and Depth:

- The Author QP believes the low cut-off grades used reflected relatively shallow mining of predominantly oxidised material and a bulk low-cost extractive process.
- These assumptions may not apply to deeper mining of fresh rock (say at Mungu).
- The Author QP's opinion would be that reporting of deep mineralisation should use a higher cut-off grade (with attendant lower Resources in those deep areas) to reflect potential underground mining and a different more costly extractive process.
- The Author QP does not know at what depth this consideration should apply from.

15 MINERAL RESERVE ESTIMATES

15.1 Introduction

The mineral reserve estimate with an effective date of June 30, 2021, for the Project is based on the parameters and steps outlined in this section as well as the resource estimate presented in Section 14.

The terminology used to classify the reserves in this report follows the National Instruments (NI) 43-101, the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (2014), and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practices Guideline (2019).

15.1.1 MINERAL RESERVES AS DEFINED BY THE CIM

Mineral reserves are sub-divided in order of increasing confidence into Probable Mineral Reserves and Proven Mineral Reserves. A Probable Mineral Reserve has a lower level of confidence than a Proven Mineral Reserve.

A Mineral Reserve is the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined.

Mineral Reserves are those parts of Mineral Resources which, after the application of all mining factors, result in an estimated tonnage and grade which, in the opinion of the Qualified Person(s) making the estimates, are the basis of an economically viable Project after taking account of all relevant processing, metallurgical, economic, marketing, legal, environmental, socio-economic and government factors. Mineral Reserves are inclusive of diluting material that will be mined in conjunction with the Mineral Reserves and delivered to the treatment plant or equivalent facility. The term 'Mineral Reserve' need not necessarily signify that extraction facilities are in place or operative or that all governmental approvals have been received. It does signify that there are reasonable expectations of such approvals.

Probable Mineral Reserve

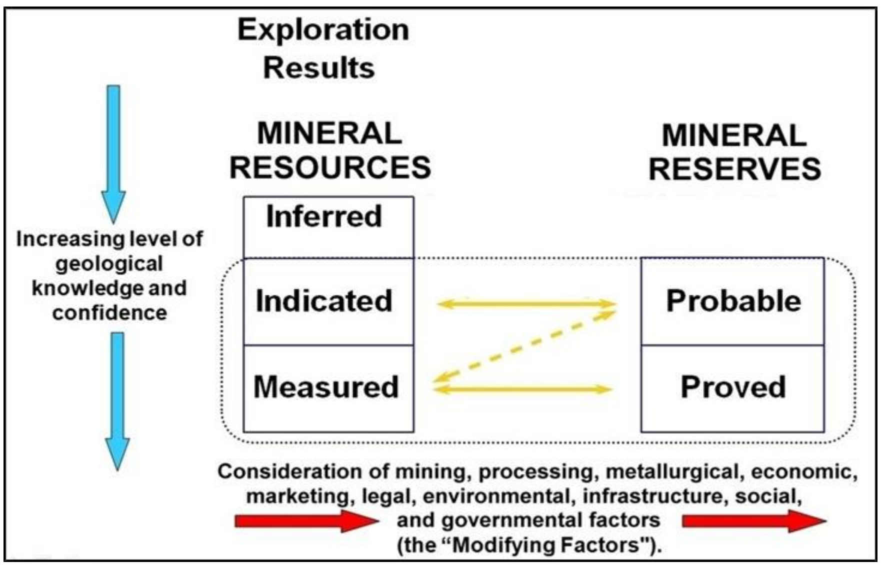
A 'Probable Mineral Reserve' is the economically mineable part of an Indicated and, in some circumstances, a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified.

Proven Mineral Reserve

A 'Proven Mineral Reserve' is the economically mineable part of a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Report must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified.

Figure 15.1 displays the relationship between the Mineral Resource and Mineral Reserve categories.

Figure 15.1 – Relationship Between Mineral Resources and Mineral Reserves



Source: CRIRSCO International Reporting Template, October 2019

Mining and geotechnical factors have been considered in the estimation of the Mineral Reserves, including the application of dilution and ore recovery factors, where appropriate.

15.2 Pit Optimisation

The open pit optimisation was conducted on the deposits to determine the economic pit limits for the ATO and Mungu pits. The optimisation was performed using the initial cost, product sales prices (for gold, silver, lead and zinc), and pit and plant operating parameters. The pit optimisation was completed using Hexagon MinePlan Project Evaluator (MPPE) module. The optimiser operates on a net value calculation for the blocks (i.e. revenue from product sales minus the operating cost). The formulas for this net value calculation are presented below:

$$\boldsymbol{Product\ Tonnage} = Ore\ Tonnage \times Recovery \times Feed\ Grade$$

$$\boldsymbol{Revenue} = \sum\nolimits_{products} Product\ Tonnage \times Sales\ Price$$

$$\boldsymbol{Net\ Value} = Revenue - (Mining\ Cost + Processing\ Cost + G\&A\ Cost)$$

Only Measured and Indicated ore have been considered in the optimisation and mine plan. Table 15.1 presents the pit optimisation parameters used to determine the ultimate pit limit. The parameters were developed assuming a standard open pit truck and shovel operation, a heap leach pad and a mill.

Table 15.1 – Pit Optimisation Parameters

Description	Unit	Oxide	Fresh/ Transition
Ore mining cost	$/t ore mined	2.27	2.23
Waste mining cost	$/t waste mined	1.77	1.79
Milling cost	$/t milled	6.59	14.96
G&A	$/t milled	7.11	2.16
Processing throughput rate (leach pad)	Mtpa	1.20	-
Processing throughput rate (mill)	Mtpa	-	2.20
Gold recovery	%	70.0	80.0
Silver recovery	%	40.0	85.8
Lead recovery	%	-	88.0
Zinc recovery	%	-	88.0
Gold sales price	$/oz	1,610	
Silver sales price	$/oz	21.00	
Lead sales price	$/t metal	1,970	
Zinc sales price	$/t metal	2,515	
Gold refining charge	$/oz	2.00	
Silver refining charge	$/oz	1.00	
Lead refining charge	$/t metal	520	
Zinc refining charge	$/t metal	525	
Royalties (Au, Ag, Pb, Zn)	%	5.0	
Ore loss	%	2.0	
Ore dilution	%	3.0	
Slope angles	Refer to Section 16.2		

15.3 Cut-Off Grade

The cut-off grade (COG) for the oxides as well as for the transition and fresh ores were calculated according to the following formula:

$$COG_{(AUEQ)} = \frac{Mining\ Cost + Processing\ Cost + G\&A}{\sum_{metals}(Metal\ Price_{metal} \times (1 - Royalty_{metal}) - Refining\ Cost_{metal}) \times Process\ Recovery_{metal}}$$

Where the metals for the oxide ore are gold and silver, and the metals for the transition and fresh ores are gold, silver, lead and zinc.

The marginal COG is calculated by dividing the total ore cost, excluding mining costs, but including a rehandling cost, by the new recovered gold price. Marginal ore is used throughout the LOM to supplement the leach pad and mill feeds.

The COGs calculated as described above and using the economic parameters listed in Table 15.1 are presented in Table 15.2.

Table 15.2 – COG Results

Material Type	Unit	COG Grade	Marginal COG
Oxide	AuEq (g/t)	0.46	0.42
Transition and Fresh	AuEq (g/t)	0.48	0.45

15.4 Ore Recovery and Dilution

The ore zones for the ATO and Mungu deposits are defined by grade rather than a clear ore/waste geological contact. Therefore, an ore recovery of 98% and an ore dilution of 3% were applied to compensate for the impossibility of perfectly differentiating between ore and waste at the contacts.

15.5 Pit Optimisation Results

The optimal pit mining limits for the ATO and Mungu deposits were determined using MPPE, using the Pseudoflow algorithm. The Pseudoflow algorithm is a network-flow algorithm similar to the Lersch-Grossman algorithm. However, Pseudoflow is computationally more efficient, able to achieve the same results as Lersch-Grossman in a shorter time.

Table 15.3 and Figure 15.2 present the results of the optimisation at the ATO pit. When varying gold price, the best and worst case NPVs increase gradually until it reaches Pits 20 and 18, for the best-case and worst-case scenarios, respectively. From these points, the NPV decreases slightly due to the costs associated with waste mining exceeding revenues. On this basis, Pit 20 was chosen as the ultimate pit limit.

Table 15.3 – ATO MPPE Pit Optimisation Results

Pit Shell	NPV – Best	NPV – Worst	Ore	Waste	Grade AuEq	Stripping Ratio
	$ USD ('000)	$ USD ('000)	(t)	(t)	(g/t)	(w/o)
1[1]	179,057	179,057	2,828,769	811,146	2.168	0.29
2	294,730	292,634	4,981,429	2,500,897	2.143	0.50
3	407,366	401,945	7,185,924	4,233,487	2.150	0.59
4	511,235	501,501	9,217,651	6,164,444	2.182	0.67
5	594,793	578,656	11,140,610	8,225,134	2.173	0.74
6	666,005	642,989	12,902,591	10,603,444	2.166	0.82
7	723,530	696,099	14,378,054	13,262,632	2.170	0.92
8	772,034	740,925	16,045,186	15,682,332	2.136	0.98
9	811,193	775,530	17,657,683	18,174,487	2.094	1.03
10	843,542	802,642	19,136,314	20,788,408	2.059	1.09
11[2]	879,911	836,367	20,358,303	23,675,841	2.065	1.16
12	906,775	857,846	22,040,165	30,062,388	2.023	1.36
13	918,762	866,119	22,925,560	33,737,450	2.001	1.47
14	928,708	873,346	23,626,492	37,251,379	1.987	1.58
15	934,817	875,673	24,245,465	40,824,699	1.970	1.68
16	940,318	876,895	24,882,314	44,230,870	1.952	1.74
17	943,825	875,882	25,212,557	47,854,019	1.947	1.90
18	949,136	879,117	25,677,548	51,479,283	1.940	2.00
19	951,789	878,279	26,130,651	55,158,206	1.927	2.11
20[3]	953,984	876,588	26,550,718	58,811,605	1.916	2.22
21	955,341	874,360	27,154,486	65,261,172	1.898	2.40

[1] Pit shell for Pushback 1 pit
[2] Pit shell for Pushback 2 pit
[3] Pit shell for ultimate pit limit

Figure 15.2 – ATO MPPE Pit Optimisation Results

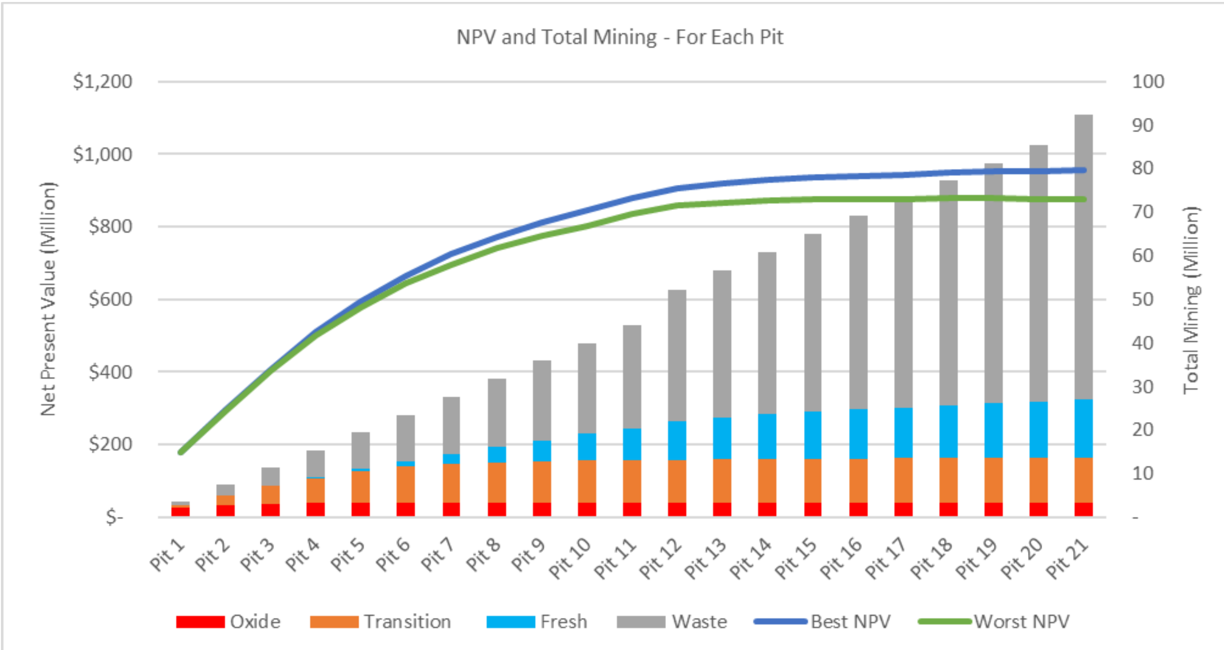


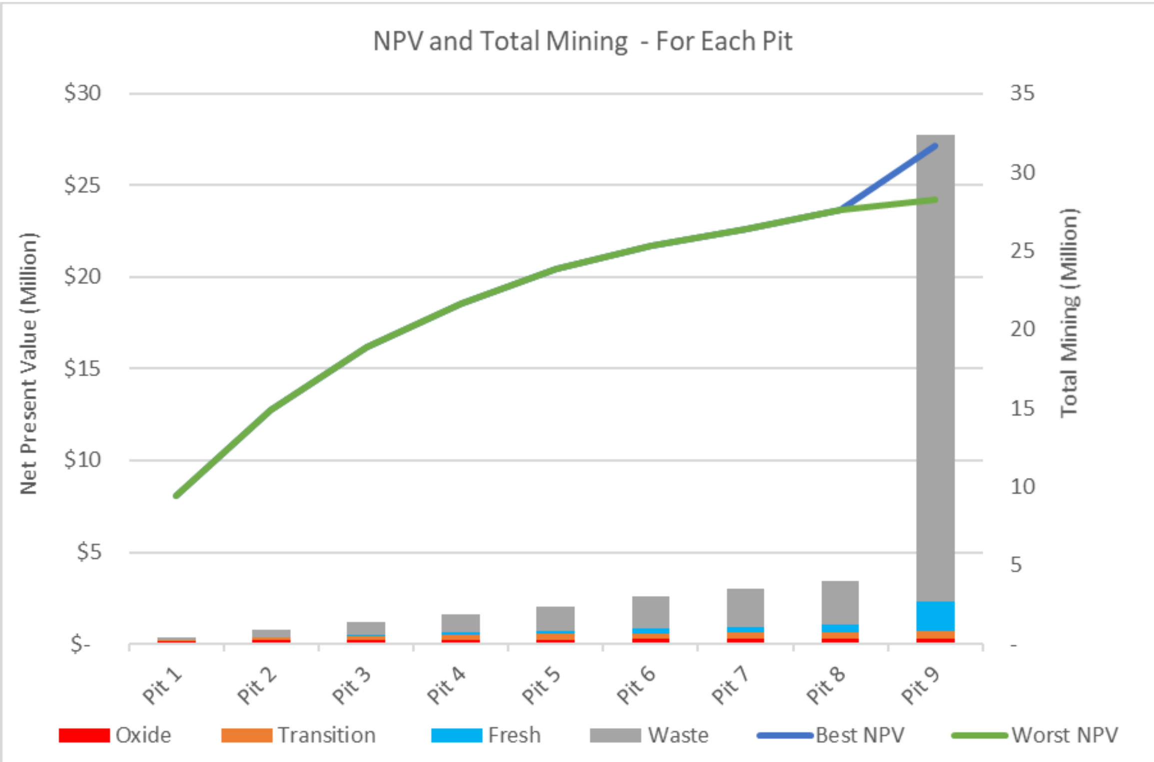
Table 15.4 and Figure 15.3 present the results of the optimisation at the Mungu pit. When varying gold price, the best case NPV increases gradually while the stripping ratio also steadily increases. Between Pits 8 and 9, the stripping ratio increases significantly, indicating a large quantity of waste is required to obtain relatively little ore. Thus, Pit 8 was chosen as Mungu's ultimate pit limit.

Table 15.4 – Mungu MPPE Pit Optimisation Results

Pit Shell	NPV – Best	NPV – Worst	Ore	Waste	Grade AuEq	Stripping Ratio
	$ USD ('000)	$ USD ('000)	(t)	(t)	(g/t)	(w/o)
1	8,085	8,085	231,787	191,866	1.448	0.83
2	12,774	12,774	415,718	461,579	1.322	1.11
3	16,179	16,179	587,471	826,879	1.264	1.41
4	18,520	18,520	731,874	1,159,680	1.223	1.58
5	20,420	20,420	858,886	1,510,676	1.196	1.76
6	21,673	21,673	979,069	2,020,024	1.174	2.06
7	22,605	22,605	1,083,348	2,401,166	1.151	2.22
8[1]	23,629	23,629	1,192,184	2,833,918	1.137	2.38
9	27,135	24,218	2,670,359	29,694,072	1.481	11.12

[1] Pit shell for ultimate pit limit

Figure 15.3 – Mungu MPPE Pit Optimisation Results



15.6 Pit Design

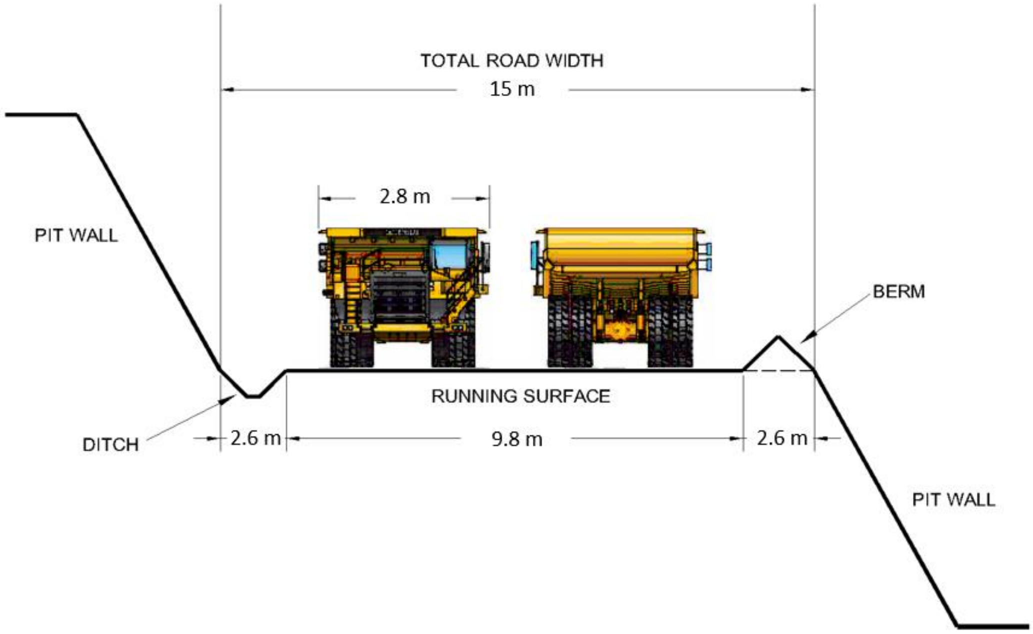
After determining the ultimate pit limit, an operational pit must be designed. This pit forms the basis of the production plan and the mineral reserve estimate. The operational pit uses the selected ultimate pit limit as a guide and smooths the pit walls, adds ramps to access the pit bottom, and ensures that the pit can be mined with the selected equipment.

15.6.1 Haul Road Design

The ramps and haul roads were designed for an overall width of 15 m. For double lane traffic, industry standard practice indicates the running surface width of a road should be a minimum of 3.5 times the width of the largest truck travelling on it. The overall width of a 32-T haulage truck is 2.8 m, which results in a running surface of 9.8 m. The allowance for berms and ditches increases the overall haul road width to 15 m. A typical ramp cross section is presented in Figure 15.4.

A maximum centreline grade of 10% was considered for the Report.

Figure 15.4 – Haul Road Design



Source: DRA, 2021

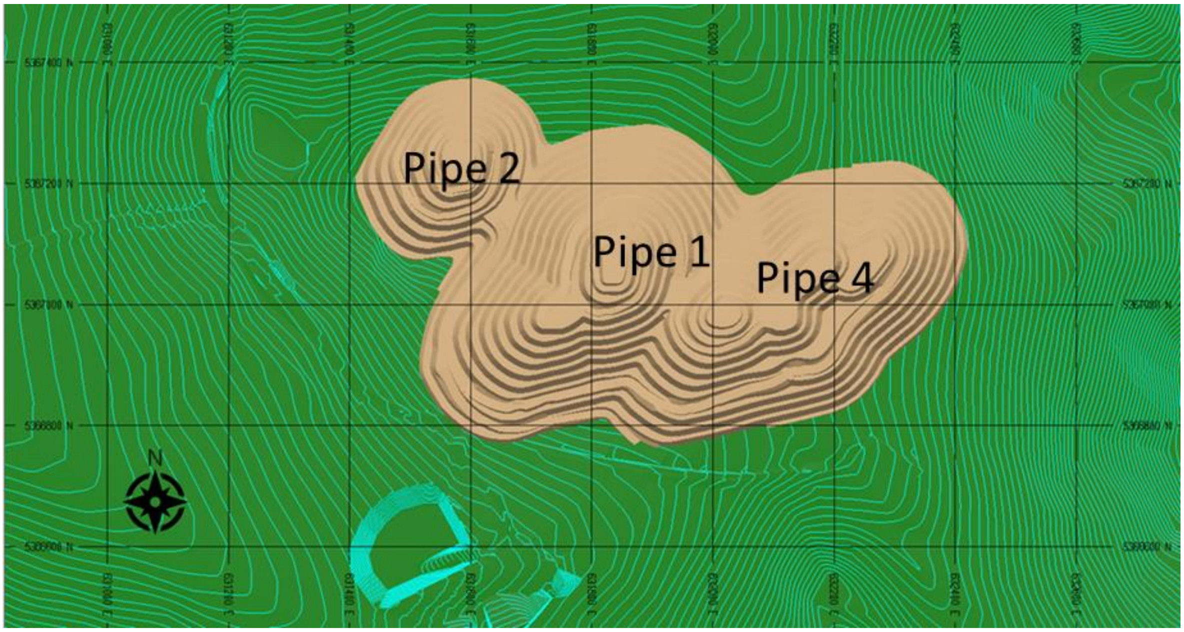
15.6.2 Pit Slopes

The pit designs followed the recommended geotechnical slopes for the ATO and Mungu Pits, as described in Section 16.2.

15.6.3 Open Pit Design Results

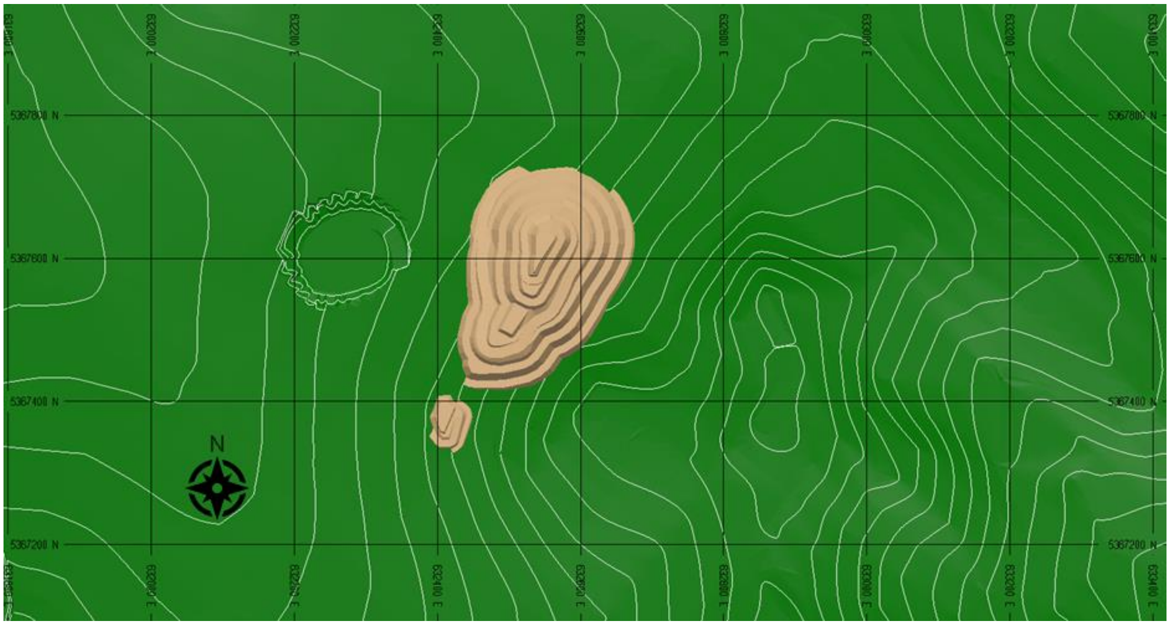
The final pit designs for the ATO and Mungu pits were used for the mineral reserves estimate. Figure 15.5 presents the ultimate pit design for ATO and Figure 15.6 presents the ultimate pit design for Mungu.

Figure 15.5 – ATO Ultimate Pit Design



Source: DRA 2021

Figure 15.6 – Mungu Ultimate Pit Design



Source: DRA 2021

15.7 Mineral Reserve Statement

The mineral reserves for the ATO and Mungu Pits are estimated at 26.4 Mt of Proven and Probable Reserves at a grade of 1.86 g/t AuEq, based on the marginal cut-off grades described in Section 15.3. To access the ore, a total of 69.2 Mt of waste rock will need to be extracted, resulting in a 2.62 stripping ratio detailed in Section 16.3.

Table 15.5 – Mineral Reserve Estimate, Effective June 30, 2021

Category	Material	Ore	Grades					Contained Metal		
			AuEq	Au	Ag	Pb	Zn	Au	Ag	AuEq
		(kt)	(g/t)	(g/t)	(g/t)	(%)	(%)	(k oz)	(k oz)	(k oz)
ATO										
Proven	Oxide	1,618	1.54	1.45	12.81	0.54	0.40	75	666	80
	Transition	6,604	2.16	1.34	10.35	0.51	0.86	285	2,198	459
	Fresh	6,673	2.04	1.17	6.80	0.55	1.02	251	1,459	438
Probable	Oxide	1,035	1.19	1.07	16.36	0.33	0.26	36	544	40
	Transition	3,721	1.81	1.10	14.23	0.36	0.67	132	1,702	217
	Fresh	5,669	1.67	0.92	9.03	0.44	0.83	168	1,646	304
Proven & Probable	*Oxide*	*2,652*	*1.40*	*1.30*	*14.19*	*0.46*	*0.35*	*111*	*1,210*	*119*
	Transition	*10,324*	*2.03*	*1.25*	*11.75*	*0.46*	*0.80*	*415*	*3,900*	*674*
	Fresh	*12,342*	*1.87*	*1.06*	*7.82*	*0.50*	*0.93*	*421*	*3,103*	*742*
	Subtotal	*25,318*	*1.89*	*1.16*	*10.09*	*0.48*	*0.81*	*944*	*8,213*	*1,538*
Mungu										
Proven	Oxide									
	Transition									
	Fresh									
Probable	Oxide	289	1.06	0.83	30.52	0	0	8	284	10
	Transition	385	1.22	0.68	38.18	0	0.01	8	473	15
	Fresh	412	1.10	0.53	39.62	0	0.02	7	525	15
Proven & Probable	*Oxide*	*289*	*1.06*	*0.83*	*30.52*	*0*	*0*	*8*	*284*	*10*
	Transition	*385*	*1.22*	*0.68*	*38.18*	*0*	*0.01*	*8*	*473*	*15*
	Fresh	*412*	*1.10*	*0.53*	*39.62*	*0*	*0.02*	*7*	*525*	*15*
	Subtotal	*1,086*	*1.13*	*0.66*	*36.68*	*0*	*0.01*	*23*	*1,281*	*39*

Category	Material	Ore	Grades					Contained Metal		
			AuEq	Au	Ag	Pb	Zn	Au	Ag	AuEq
		(kt)	(g/t)	(g/t)	(g/t)	(%)	(%)	(k oz)	(k oz)	(k oz)
Combined (ATO and Mungu)										
Proven	Oxide	1,618	1.54	1.45	12.81	0.54	0.40	75	666	80
	Transition	6,604	2.16	1.34	10.35	0.51	0.86	285	2,198	459
	Fresh	6,673	2.04	1.17	6.80	0.55	1.02	251	1,459	438
Probable	Oxide	1,324	1.16	1.01	19.45	0.26	0.20	43	828	49
	Transition	4,105	1.75	1.06	16.47	0.33	0.61	140	2,174	231
	Fresh	6,081	1.63	0.90	11.10	0.41	0.77	176	2,170	319
Proven & Probable	*Oxide*	*2,942*	*1.37*	*1.25*	*15.80*	*0.41*	*0.31*	*118*	*1,494*	*130*
	Transition	*10,709*	*2.00*	*1.23*	*12.70*	*0.44*	*0.77*	*423*	*4,373*	*689*
	Fresh	*12,753*	*1.85*	*1.04*	*8.85*	*0.48*	*0.90*	*426*	*3,629*	*759*
	Total	**26,404**	**1.86**	**1.14**	**11.18**	**0.46**	**0.78**	**968**	**9,491**	**1,579**

Notes

1. Mineral Resource Estimate was estimated by the Resources QP
2. ATO and Mungu Mineral Reserves are effective as of June 30, 2021
3. Mineral Reserves are included in Mineral Resources
4. Mineral Reserves are reported in accordance with CIM and NI 43-101 guidelines
5. Ore dilution is 3% and ore loss is 2%
6. Contained metal estimates have not been adjusted for metallurgical recoveries
7. The open pit mineral reserves are estimated using a cut-off grade of 0.42 g/t AuEq for oxide material and 0.45 g/t AuEq for transition and fresh material
8. Mineral Reserves are contained within an optimised pit shell based on a gold price of $1,610 USD per ounce
9. A conversion factor of 31.103477 grams per troy ounce and a conversion factor of 453.59237 grams per pound are used in the resource and reserves estimates
10. AuEq has been calculated using the following metal prices: $1,610/oz gold, $21/oz silver, $1,970/t lead, $2,515/t zinc
11. Oxide AuEq calculation: $AUEQ_{(g/t)} = Au_{(g/t)} + \frac{Ag_{(g/t)} \times 21 \times 0.4}{1{,}610 \times 0.7}$
12. Transition and fresh AuEq calculation: $AuEq_{(g/t)} = Au_{(g/t)} + \frac{Ag_{(g/t)} \times 21 \times 0.858}{1{,}610 \times 0.8} + \frac{Pb_{(\%)} \times 1{,}970 \times 0.88}{1{,}610 \times 0.8} + \frac{Zn_{(g/t)} \times 2515 \times 0.88}{1{,}610 \times 0.8}$
13. Totals may not match due to rounding
14. The Mineral Reserves are stated as dry tonnes processed at the crusher
15. The QP is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially impact the Mineral Reserves Estimate

A Mineral Reserve is the economically mineable part of a Measured and/or Indicated Mineral Resource, including diluting materials. A Probable Mineral Reserve is the economically mineable part of an Indicated and, in some circumstances, a Measured Mineral Resource. The confidence in Modifying Factors applied to a Probable Mineral Reserve is lower than that applied to the Modifying Factors of a Proven Mineral Reserve. A Proven Mineral Reserve is the economically mineable part of a Measured Mineral Resource and implies a high degree of confidence in the Modifying Factors.

16 MINING METHODS

16.1 Mining Operation

The mining method selected for the Project is a conventional open pit operation with rigid body mining trucks, hydraulic excavators, and wheel loaders. The Project consists of two (2) separated mining areas: the ATO and the Mungu Pits. Only open pit mining is considered in this Report; however, the Mungu Pit is open at depth and only a small portion can be mined economically. Therefore, there is an opportunity for the Mungu Pit to be expanded to an underground mine once the open pit reserves are depleted.

The oxide ore will be transported by truck from the pit to the leach pad, and the transition and fresh ore will be transported by truck to either a stockpile or the mill. Waste material will be deposited in waste stockpiles. The location of the pits, ore stockpile, leach pad, mill, tailings storage facility, and waste stockpiles is illustrated in Figure 18.1.

Ore sent to the mill is a combination of transition and fresh material; however, there is no need for a particular blend of these ores. Marginal ore (above the marginal COG, but below the mining COG) will be mined and processed throughout the LOM to supplement leach pad and mill feeds. The mine will operate year-round, seven (7) days a week, twenty-four (24) hours a day (two 12-hour shifts). 35 days of downtime are considered due to weather delays as well as holidays, for a total of 330 operating days a year. Table 16.1 presents the general work schedule parameters used for the Project.

Table 16.1 – Mine Working Schedule

Work Schedule	Unit	Value
Weeks per Year	Weeks	52
Days per Week	Days	7
Day per Year	Days	365
Holidays, Weather Delays	Days	35
Operating Days per Year	Days	330
Shifts per Day	Shifts	2
Shifts per Year	Shifts	660
Hours per Shift	Hours	12
Effective Hours per Shift	Hours	10

16.2 Geotechnical

In order to develop pit slope parameters for ATO, DRA relied mostly on the 2011 and 2012 ATO slopes study reports from Victor Vdovin, Corporate Geotechnical Engineer for Centerra Gold Inc. (Centerra), Technical Development Group. Both reports were translated from Mongolian to English and the geotechnical data was summarised in table format to extract all available information. The data from the reports was of very high quality and was fully used.

16.2.1 Pit Slope Parameters

Pit slope parameters were developed by reviewing a combination of factors such as the 2011 and 2012 geotechnical recommendations by Centerra, a review of the geology and structures in recent infill drilling and a review of the as-built pit wall slope behaviours.

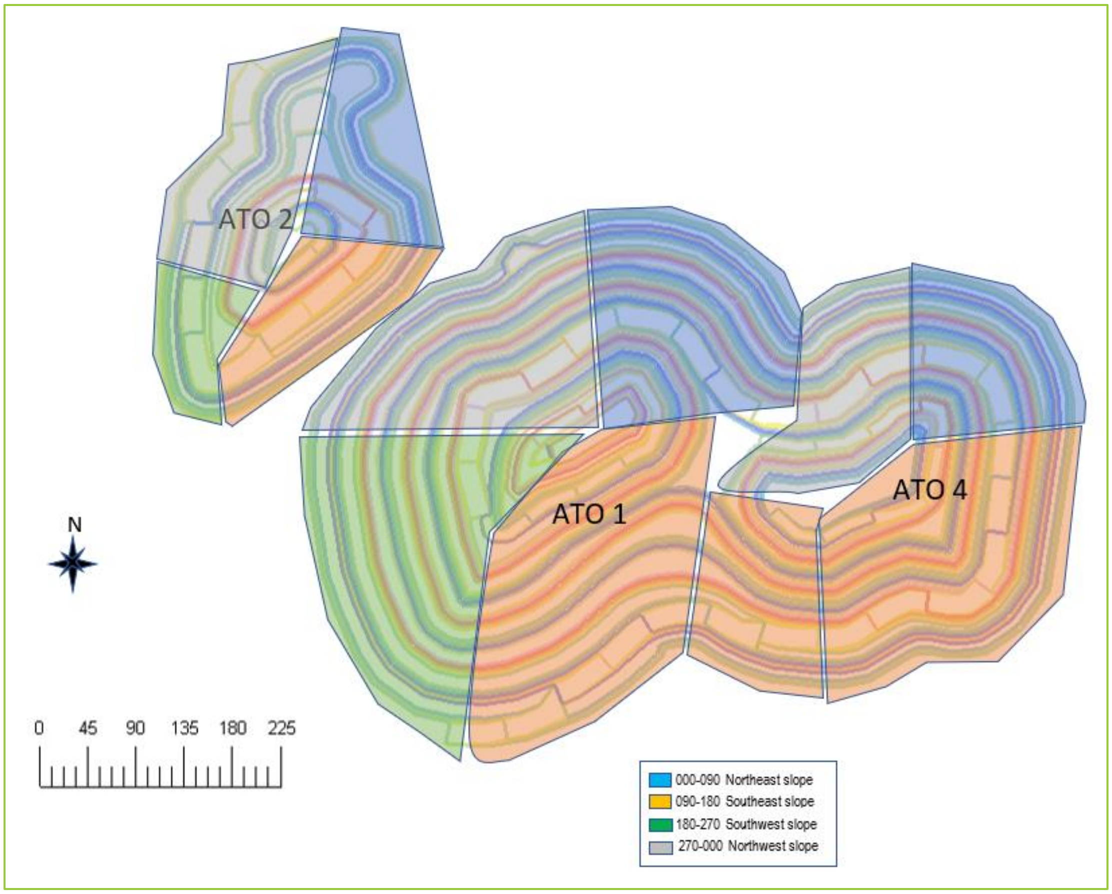
Based on these reviews, DRA developed the following geotechnical parameters for pit design and pit optimisation, as presented in Table 16.2 and Figure 16.1. The geotechnical parameters developed by DRA effectively split the ATO Pit into four (4) geotechnical domains based on the location of the pit wall. Since the Mungu Pit has no available geotechnical information, it was decided to use conservative slope parameter numbers. As the Mungu Pit is developed and geotechnical data is made available, Steppe Gold should revisit the slope parameters used to consider whether steeper angles could be applied safely.

Table 16.2 – Pit Slope Parameters

Geotechnical Domain[1]	Material Type	BFA[2]	Bench Height	Planned Berm Width	Stack Height	Design IRA[3]	Overall Slope Angle
		(°)	(m)	(m)	(m)	(toe to toe)	(°)
000-090 Northeast slope (blue)	Oxide and Transition	65	10	6.8	20	51.1	50
	Fresh						
090-180 Southeast slope (orange)	Oxide and Transition	65	10	4.7	20	55	52
	Fresh						
180-270 Southwest slope (green)	Oxide and Transition	65	10	10.5	20	45	42
	Fresh						
270-360 Northwest slope (grey)	Oxide and Transition	65	10	4.7	20	55	52
	Fresh						
Mungu 0-360 Northwest slope	Oxide and Transition	65	10	7.5	20	50	45
	Fresh						

1 The colours in parentheses correspond to the colours used to differentiate zones in Figure 16.1
2 Bench face angle
3 Inter-ramp angle

Figure 16.1 – ATO Plan View – Geotechnical Domains



Source: DRA 2021

The recommended slope through the overburden formation is 26.6° with a 10 m wide catch bench at the contact between the overburden and the bedrock. The recommended slopes assume pre-shearing blasting techniques will be used.

16.3 Mining Design

The Project mineral reserves were estimated for the ATO and Mungu Pits based on the economic and pit design parameters detailed in Section 15. The total tonnage to be mined from these pits is estimated at 96.6 Mt (ore and waste combined). The material will be mined over a period of approximately 13 years.

16.3.1 Pit Designs

The final pit designs for the ATO and Mungu follow the recommended geotechnical parameters and domains outlined in Section 16.2. Detailed tonnages by ore type and pit are presented in Table 16.3.

The ATO and Mungu final pit designs were previously presented in Figures 15.5 and 15.6, respectively.

Table 16.3 – Project Reserves by Pit

Pit	Total Ore		Total Waste	Stripping Ratio
	kt	AuEq (g/t)	kt	w/o
ATO	25,318	1.89	65,328	2.58
Mungu	1,086	1.13	3,893	3.59
Total	**26,404**	**1.86**	**69,221**	**2.62**

16.3.2 Waste Stockpile Designs

Waste material mined from each of the Project pits will be stored in two (2) waste stockpiles. The ATO stockpile is located East of the ATO Pit and has a capacity of 31.8 Mm^3 and a maximum height of 50 m. The Mungu stockpile is located West of the Mungu Pit and has a capacity of 2.6 Mm^3 and a maximum height of 33 m.

The stockpiles were designed using the following parameters:

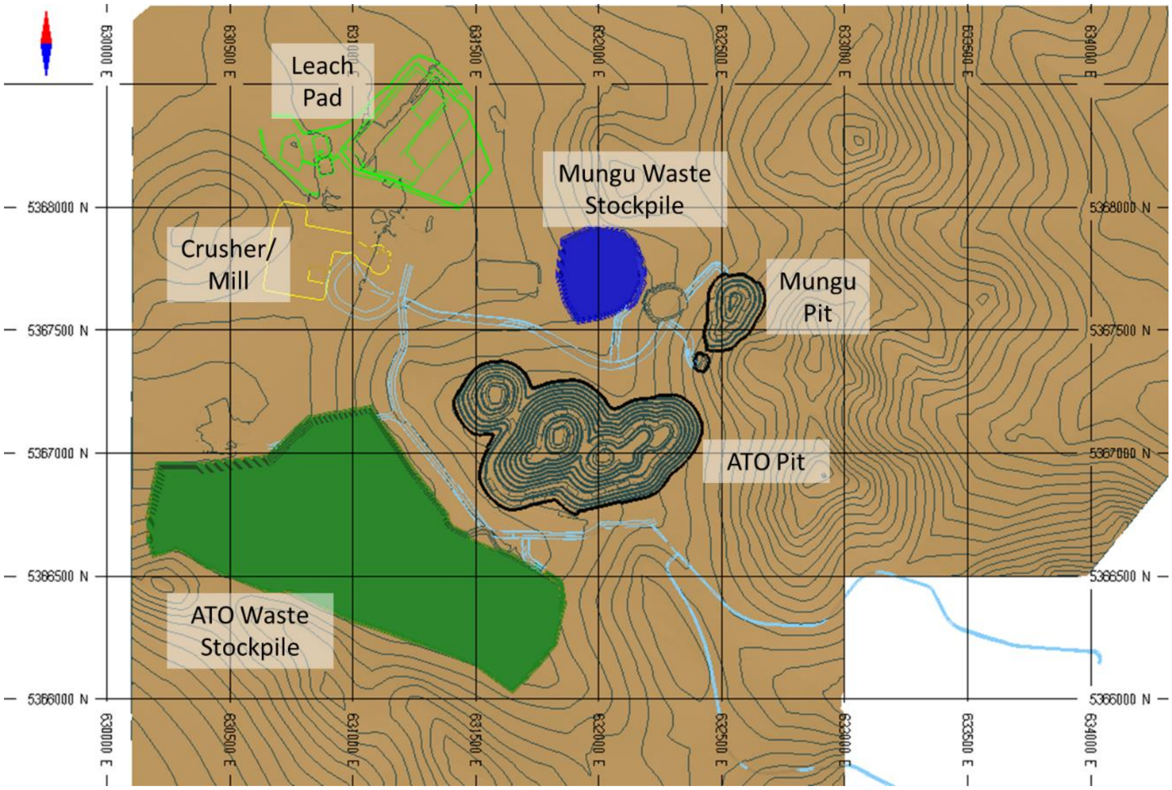
- 5 m high lifts;
- 70° lift face angles;
- 0 m berms at every lift;
- 45° overall slope;
- 30% swell factor from bank in-situ m^3 to in-place m^3.

The waste stockpile locations relative to the pits are presented in Figure 16.2.

16.4 Pit Dewatering

The pits will require two (2) diesel-powered pit dewatering pumps, with each having a pumping capacity of 200 m^3/h to 300 m^3/h.

Figure 16.2 – Waste Stockpile Locations



Source: DRA 2021

16.5 Mine Planning

A mine plan (or schedule) was prepared to estimate a probable production schedule for the Project and assess the mine equipment fleet requirements, as well as mine capital and operating costs for the Project's financial model. The mine plan was based on a production rate of 1.2 Mtpa of oxide ore at the leach pad and 2.20 Mtpa of transition and fresh ore at the mill.

Mine planning was performed using Hexagon MinePlan Schedule Optimiser (MPSO) module based on the final pit designs, the intermediate pushbacks described in the following section, and the mineral resources block model. The mine plan was estimated on a monthly basis for the first three years of mill use, and then annually for the LOM.

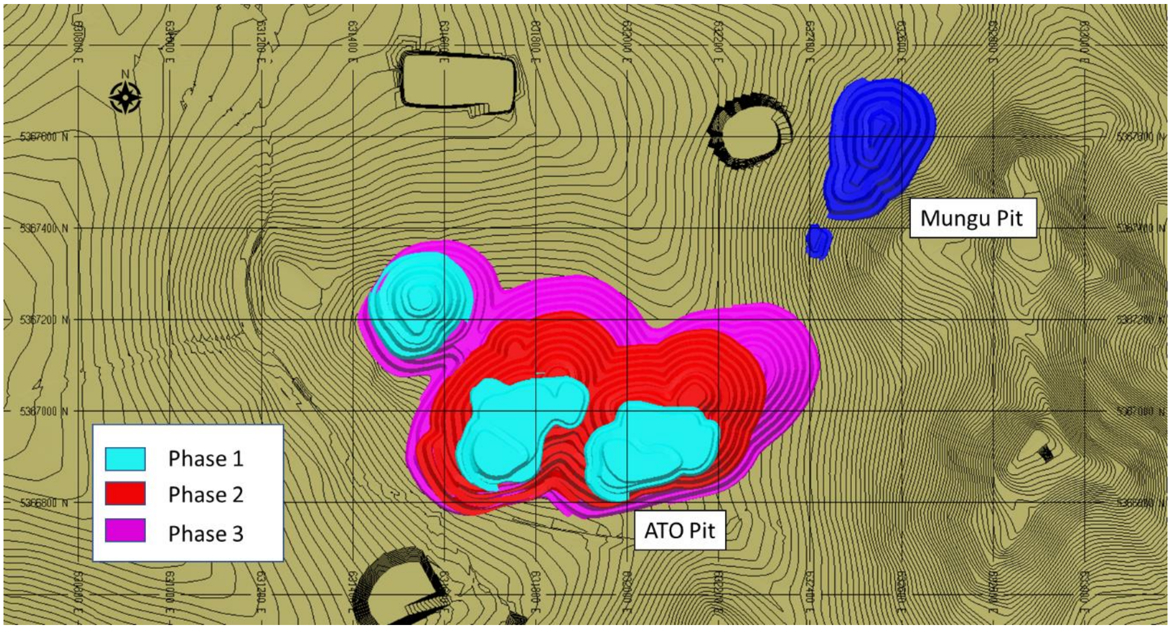
16.5.1 Pushback Design

The ATO pit was separated into three (3) pushbacks while the Mungu pit, due to its small size, was considered to be a single pushback. The reserves contained in each pushback are presented in Table 16.4. Figure 16.3 presents an overview of the ATO and Mungu pushbacks, and Figure 16.4 to Figure 16.6 show the ATO pushbacks individually.

Table 16.4 – Reserves by Pushback

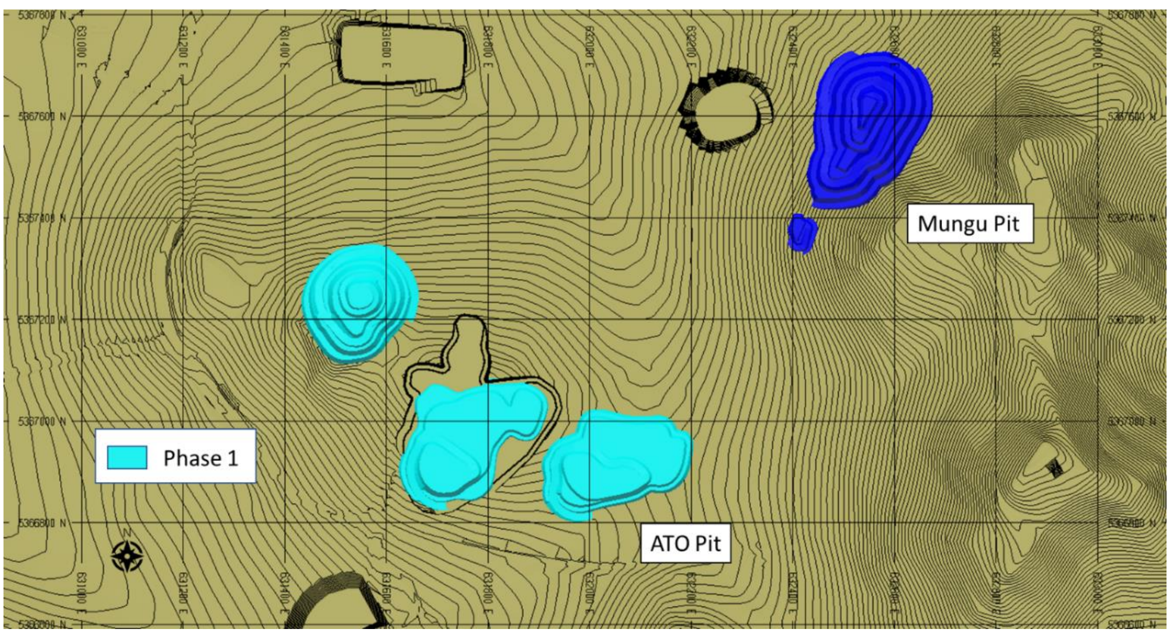
Material Type	Ore						Waste	Total Mined	Stripping Ratio
	kt	AuEq (g/t)	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	kt	kt	
ATO Pushback 1									
Oxide	2,252	1.41	1.31	14.16	0.47	0.33	3,003	5,255	1.07
Transition	658	1.51	0.58	5.57	0.61	1.12	118	777	
Fresh	-	-	-	-	-	-	-	-	
Total	***2,911***	***1.43***	***1.17***	***12.52***	***0.49***	***0.48***	***3,121***	***6,032***	
ATO Pushback 2									
Oxide	362	1.43	1.32	15.37	0.40	0.45	10,012	10,374	1.53
Transition	9,272	2.09	1.33	12.31	0.44	0.75	11,732	21,003	
Fresh	7,474	2.09	1.27	7.17	0.52	0.95	4,349	11,823	
Total	***17,108***	***2.08***	***1.30***	***10.13***	***0.47***	***0.83***	***26,093***	***43,201***	
ATO Pushback 3									
Oxide	36	0.65	0.61	5.60	0.45	0.16	9,126	9,162	6.81
Transition	395	1.56	0.54	8.96	0.59	1.22	10,080	10,475	
Fresh	4,867	1.53	0.74	8.82	0.46	0.90	16,908	21,776	
Total	***5,299***	***1.52***	***0.72***	***8.77***	***0.47***	***0.92***	***36,114***	***41,413***	
Mungu									
Oxide	289	1.06	0.83	30.52	0.00	0.00	2,268	2,557	3.59
Transition	385	1.22	0.68	38.18	0.00	0.01	1,566	1,566	
Fresh	412	1.10	0.53	39.62	0.00	0.02	856	856	
Total	***1,086***	***1.13***	***0.66***	***36.68***	***0.00***	***0.01***	**3,893**	**4,979**	

Figure 16.3 – Overview Pit Pushbacks



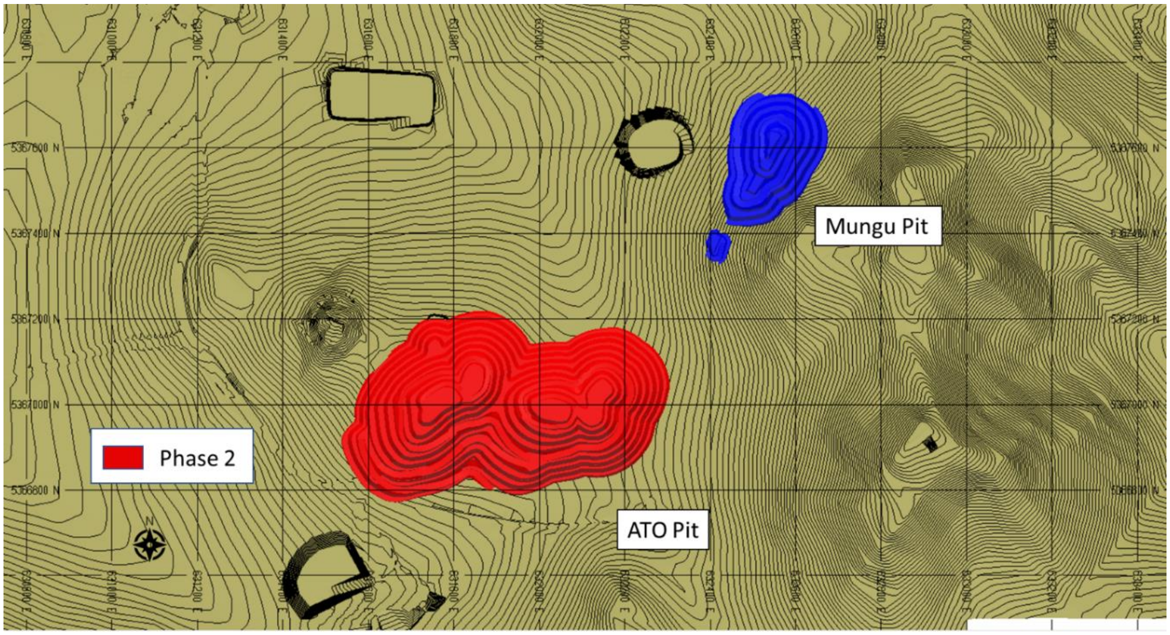
Source: DRA 2021

Figure 16.4 – ATO Pushback 1 Design



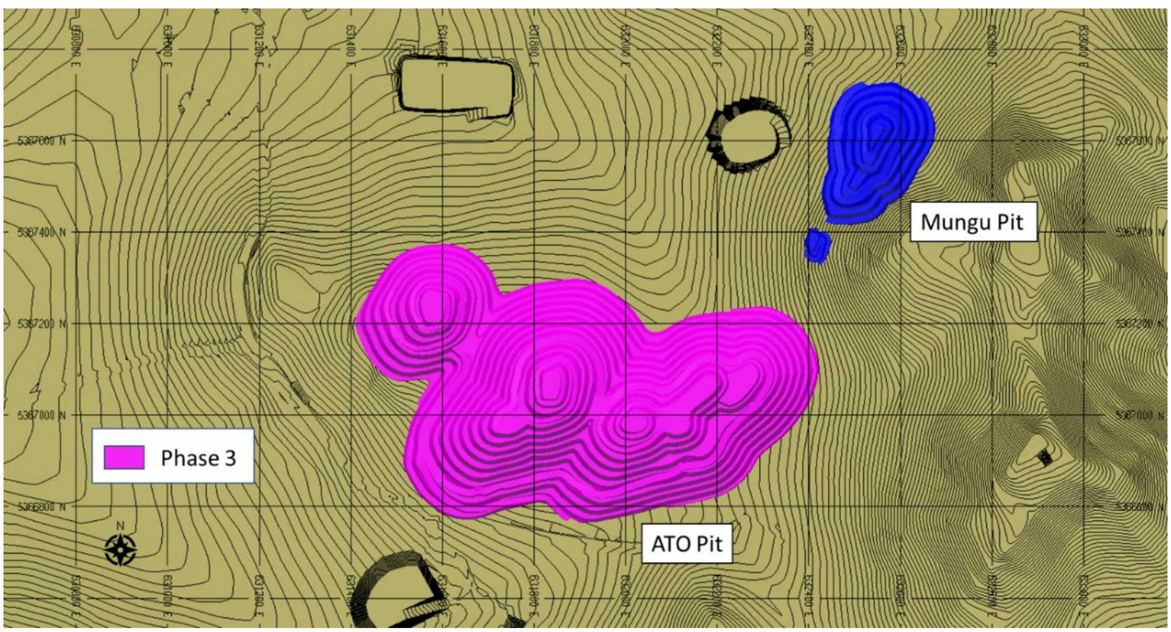
Source: DRA 2021

Figure 16.5 – ATO Pushback 2 Design



Source: DRA 2021

Figure 16.6 – ATO Pushback 3 Design



Source: DRA 2021

16.5.2 Mine Production Schedule

Mine planning was performed using the MPSO module to determine the most productive mining sequence while maximising metal production and minimising material movement. The total material movement is presented in Figure 1.2 and the mine production schedule is presented in Table 1.7. Figure 16.8 presents the material sent to leach pad over the LOM and Figure 16.9 presents the material sent to the mill over the LOM. Table 16.6 presents the monthly mill feed production during its first years of operation. Figure 16.10 to Figure 16.16 present the yearly end of period maps.

Figure 16.7 – Total Material Movement

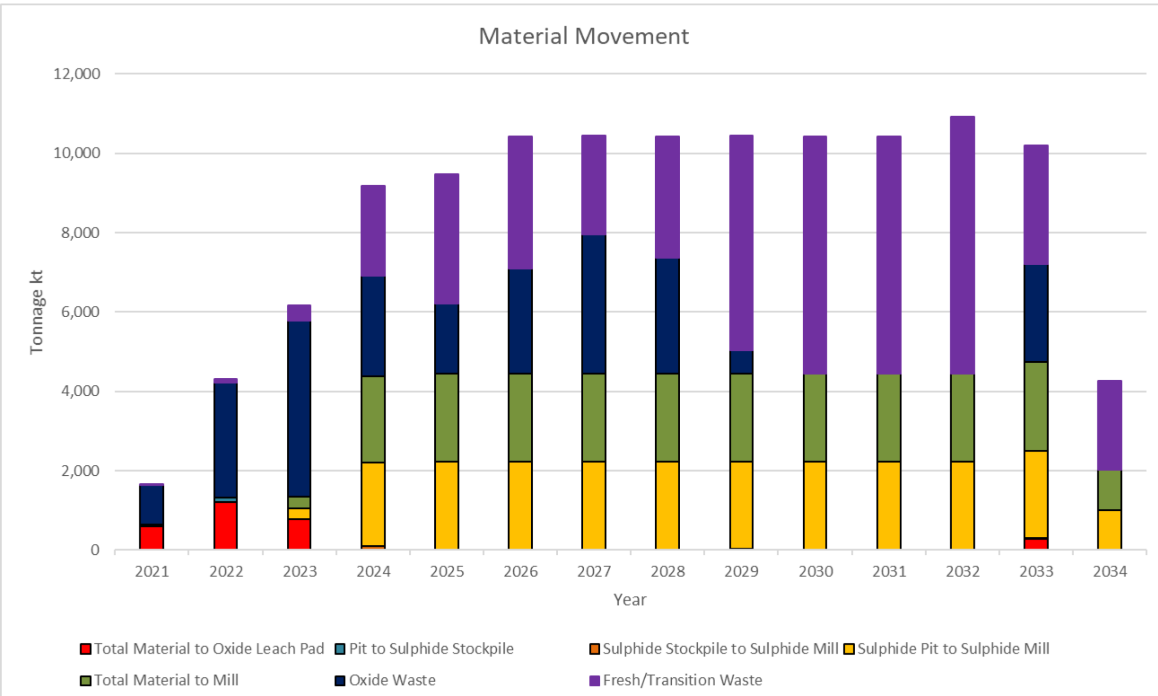


Table 16.5 – Mine Production Schedule (by Year)

Year	Ore						Waste	Total Mined	Stripping Ratio
	Tonnage (kt)	AuEq (g/t)	Au (g/t)	Ag (g/t)	Zn (%)	Pb (%)	(kt)	(kt)	(w/o)
2021[1]	606	1.79	1.68	15.20	0.35	0.32	994	1,637	1.55
2022	1,211	1.51	1.40	14.50	0.39	0.41	2,988	4,313	2.25
2023	1,464	1.48	1.24	8.19	0.45	0.27	4,814	5,861	4.60
2024	2,204	2.87	1.82	11.70	1.10	0.74	4,787	6,921	2.24
2025	2,221	2.21	1.35	14.40	0.86	0.47	5,037	7,239	2.29
2026	2,221	2.33	1.82	10.12	0.49	0.26	5,979	8,200	2.69
2027	2,236	1.97	1.40	13.62	0.50	0.27	5,979	8,215	2.67
2028	2,221	2.01	0.97	6.64	1.23	0.72	5,979	8,200	2.69
2029	2,235	1.89	0.77	5.66	1.41	0.71	5,979	8,203	2.69
2030	2,221	1.70	1.11	8.84	0.60	0.33	5,979	8,200	2.69
2031	2,221	1.61	0.87	9.58	0.80	0.42	5,979	8,188	2.71
2032	2,221	1.61	0.86	10.25	0.83	0.39	6,479	8,700	2.92
2033	2,510	1.42	0.65	9.13	0.87	0.47	5,476	7,986	2.18
2034[2]	1,008	1.13	0.55	32.06	0.17	0.08	2,258	3,266	2.24

1 Year 2021 represents the period of July 2021 to December 2021
2 Year 2034 represents approximately 5 months of production at the end of the mine life

Figure 16.8 – Leach Pad Schedule

Figure 16.9 – Mill Schedule

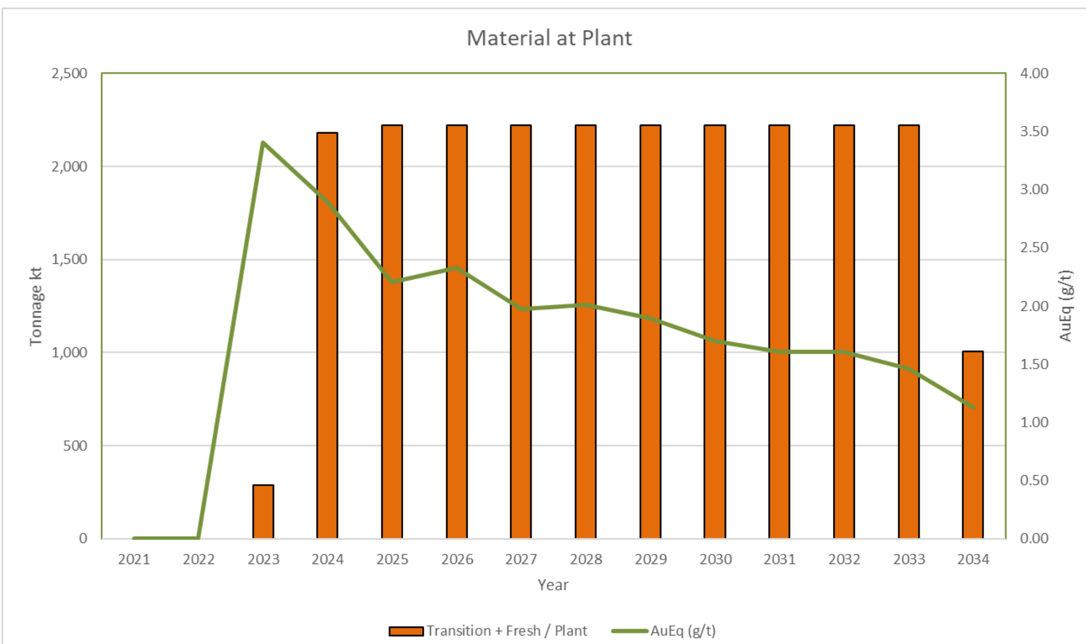
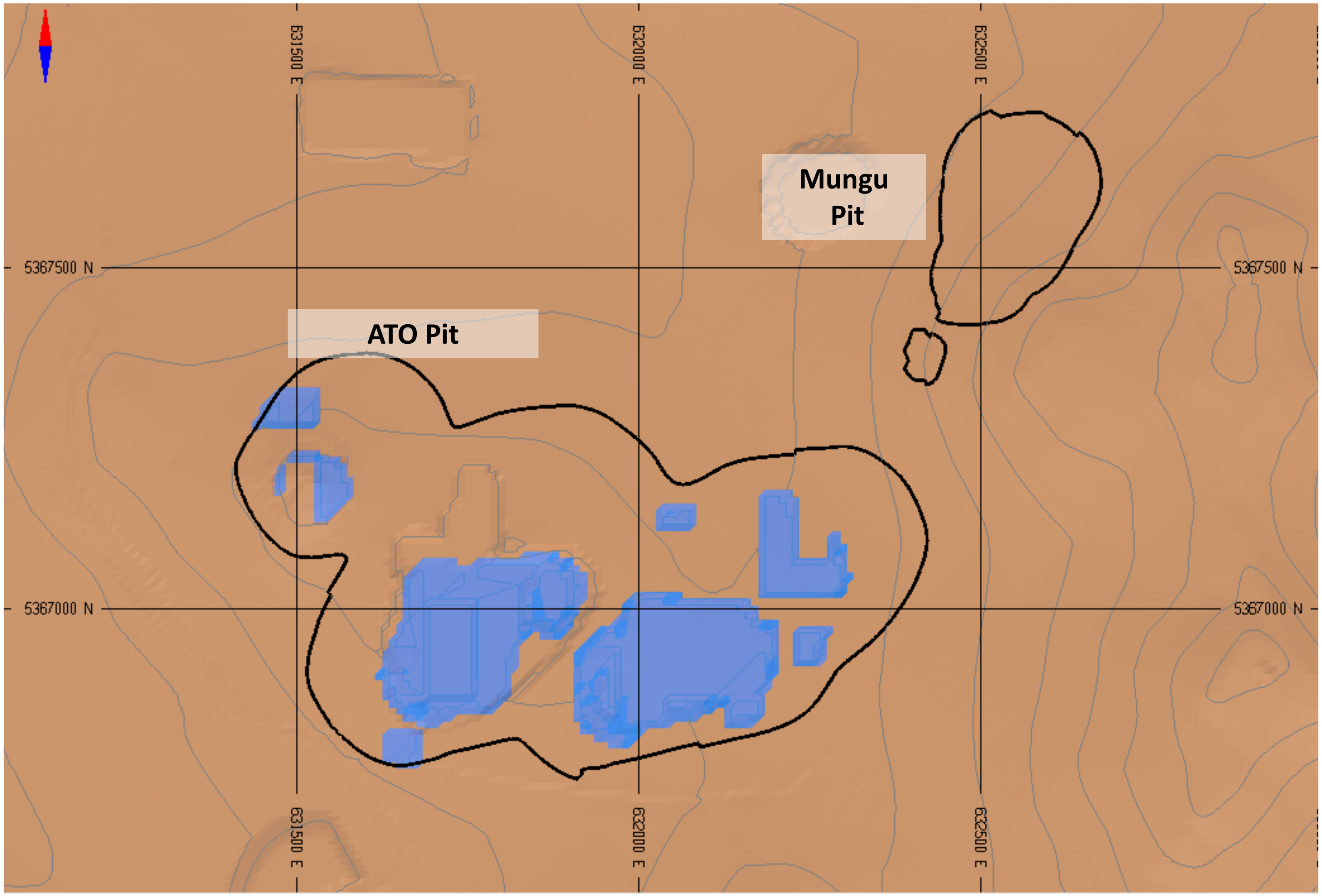


Table 16.6 – Mill Feed Breakdown During Initial Operation

Year	Month	Ore					
		Tonnage (kt)	AuEq (g/t)	Au (g/t)	Ag (g/t)	Zn (%)	Pb (%)
2023	11	139	3.62	2.52	21.03	1.10	0.52
	12	148	3.20	2.18	11.10	0.97	0.83
2024	1	157	3.36	2.20	7.36	1.19	1.03
	2	176	3.83	2.49	9.71	1.44	1.05
	3	183	3.41	2.17	10.48	1.21	1.08
	4	185	2.93	1.57	7.95	1.49	1.07
	5	185	2.72	1.69	6.61	1.20	0.73
	6	185	2.62	1.73	46.64	0.31	0.18
	7	185	2.89	2.23	15.09	0.61	0.30
	8	185	2.52	1.61	9.93	0.98	0.60
	9	185	2.50	1.55	5.74	1.08	0.72
	10	185	2.83	1.70	6.70	1.34	0.77
	11	185	2.75	1.70	6.13	1.22	0.75
	12	185	2.48	1.41	8.01	1.26	0.68
2025	1	185	2.46	1. 60	22.45	0.71	0.40
	2	185	2.19	1.39	29.26	0.51	0.30
	3	185	2.09	1.44	18.42	0.51	0.30
	4	185	2.04	1.32	33.95	0.33	0.16
	5	185	1.98	0.95	7.08	1.22	0.69
	6	185	2.19	1.19	9.84	1.16	0.56
	7	185	2.03	1.25	7.11	0.92	0.46
	8	185	2.11	1.15	4.74	1.21	0.60
	9	185	2.41	1.43	5.28	1.17	0.67
	10	185	2.48	1.62	13.83	0.86	0.49
	11	185	2.43	1.67	11.00	0.78	0.46
	12	185	2.05	1.20	9.84	0.92	0.53

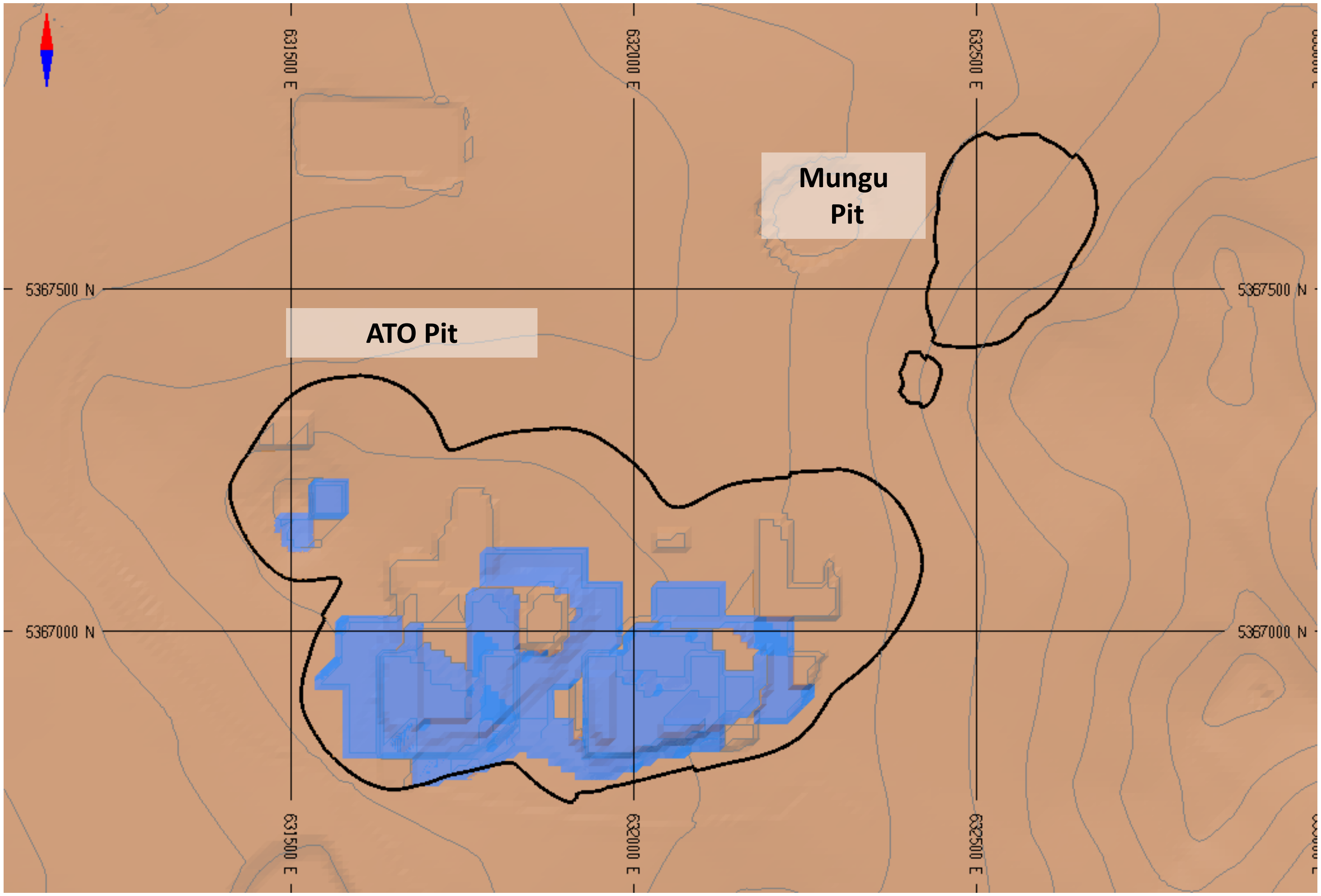
Note: The mill begins to accept ore in November 2023

Figure 16.10 – End of Period Map 2021



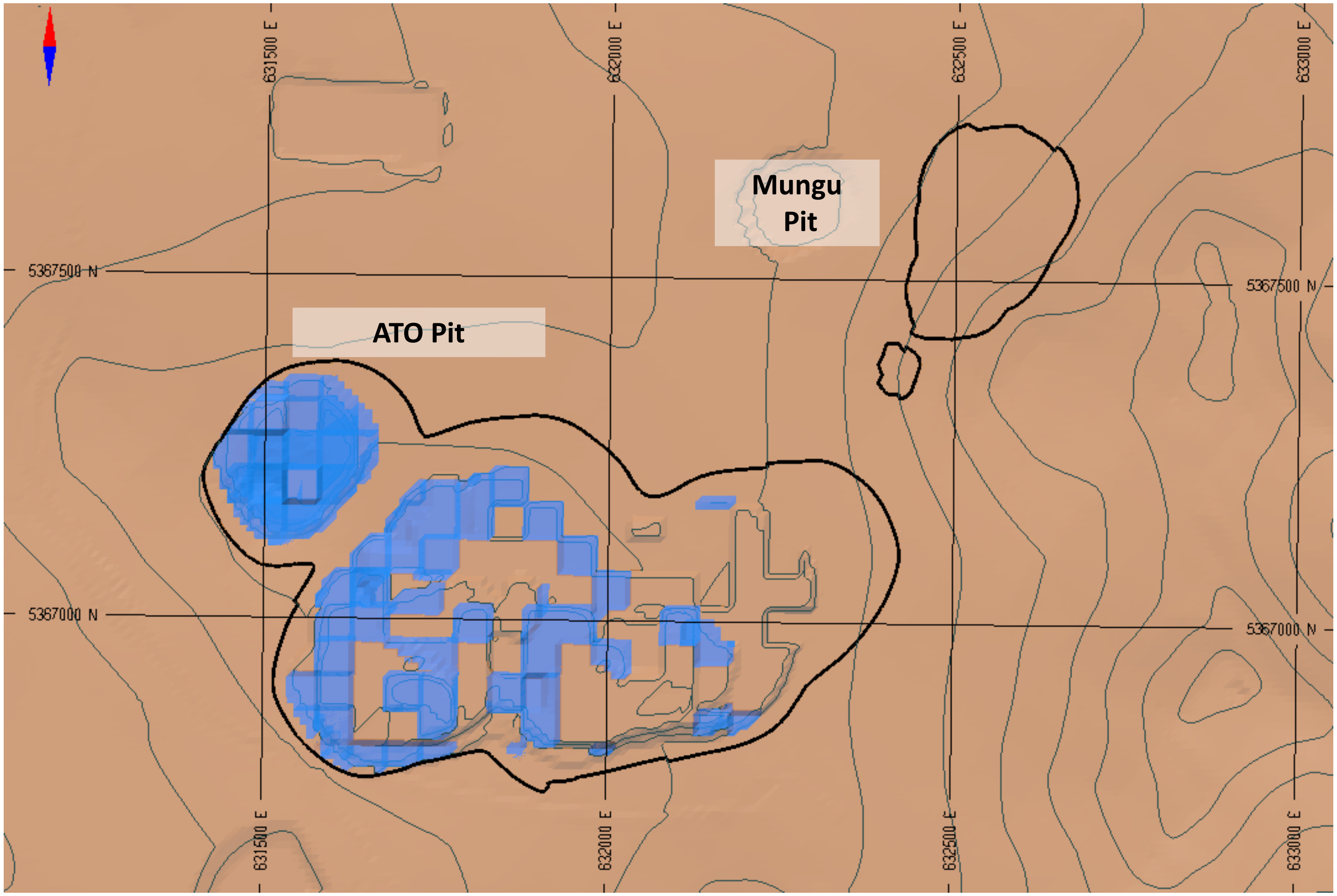
Source: DRA 2021

Figure 16.11 – End of Period Map 2022



Source: DRA 2021

Figure 16.12 – End of Period Map 2023

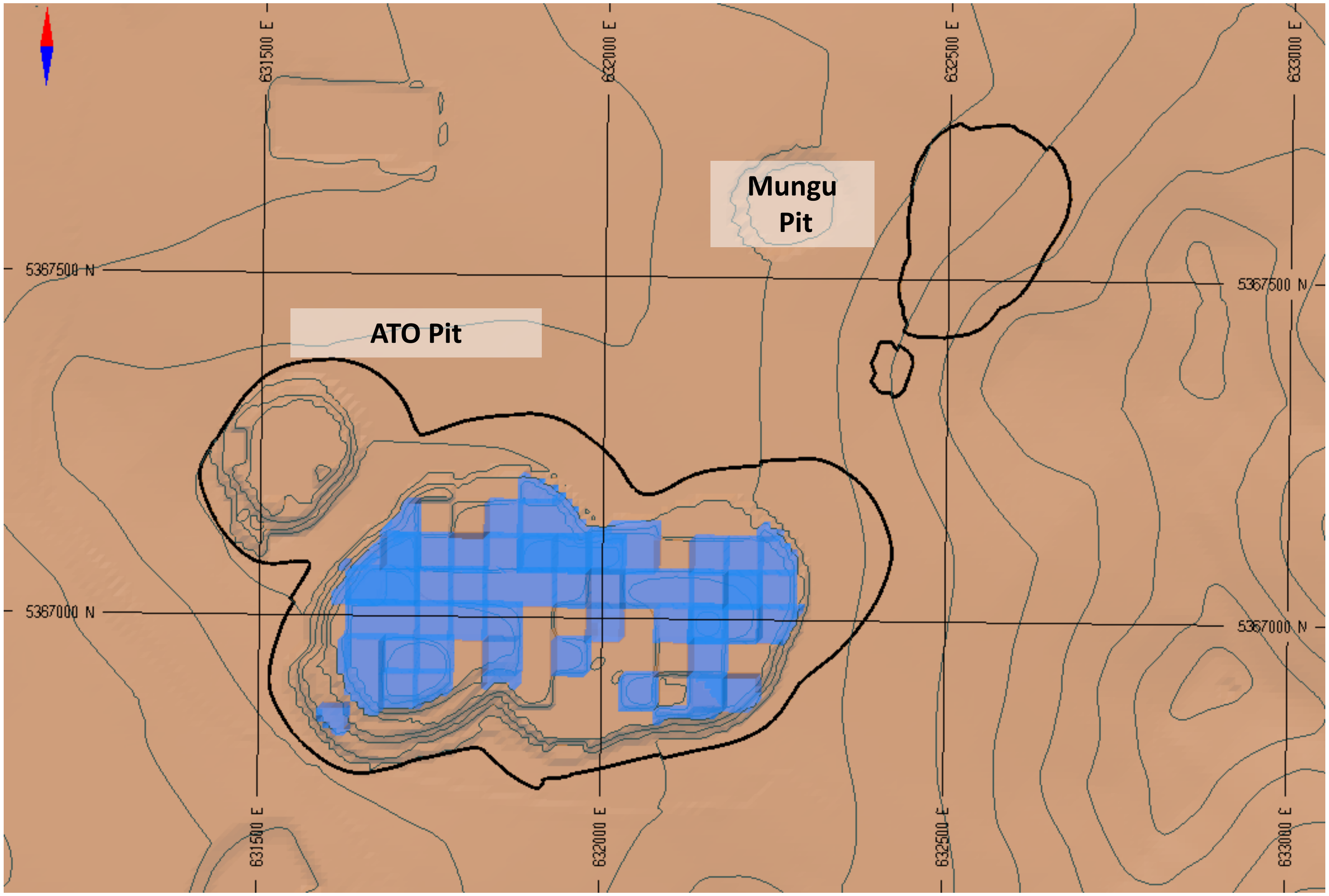


Source: DRA 2021

Figure 16.13 – End of Period Map 2024

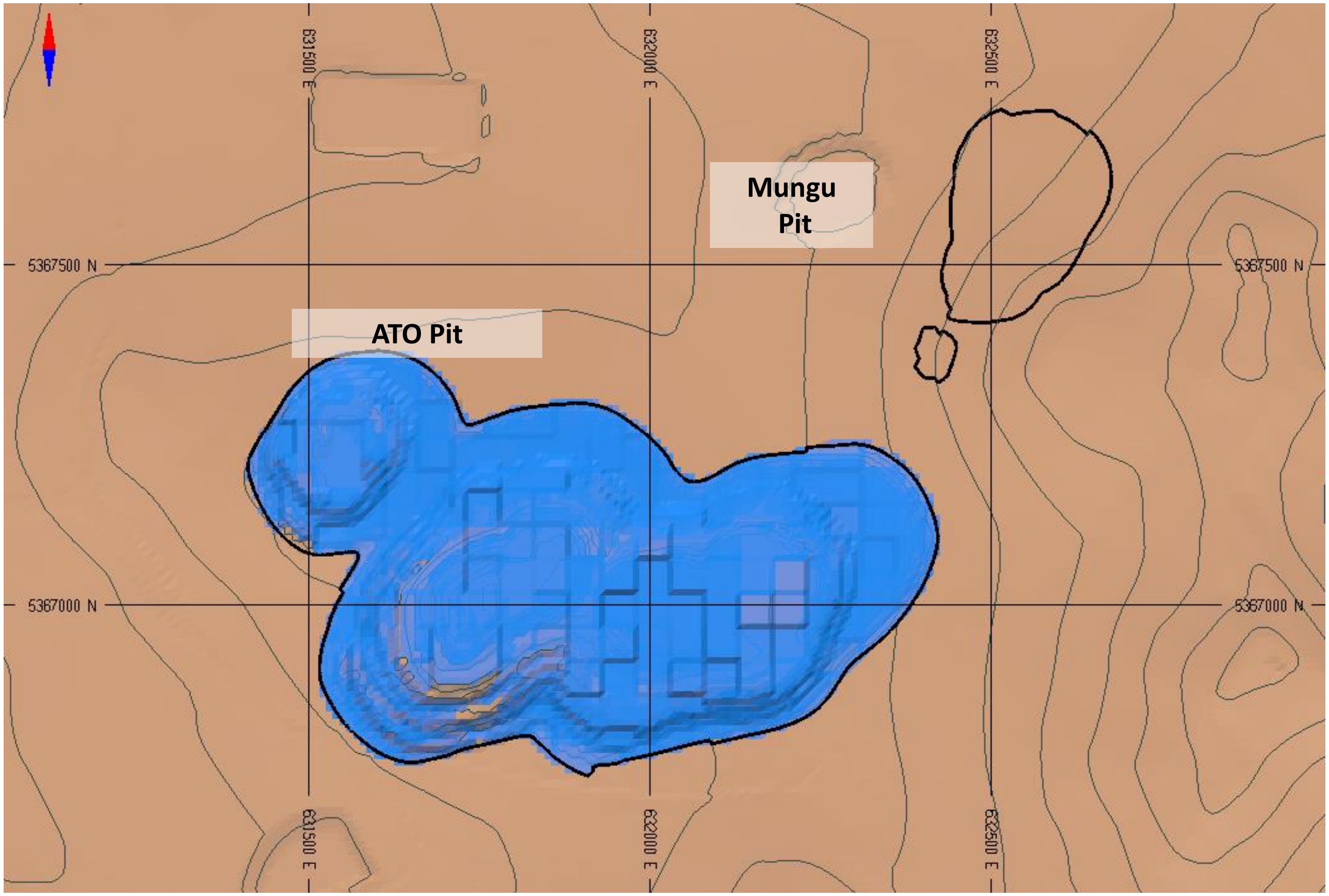
Source: DRA 2021

Figure 16.14 – End of Period Map 2025



Source: DRA 2021

Figure 16.15 – End of Period 2026 - 2030



Source: DRA 2021

Figure 16.16 – End of Period 2031-2034

Source: DRA 2021

Table 16.7 – Proposed Contractor Mining Equipment Fleet

Equipment Type	Equipment Specification	2021[1]	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034[2]
Dump truck	32 t	4	9	12	15	15	19	20	23	22	21	25	28	29	21
Hydraulic excavator	2.3 m^3	3	3	4	5	5	6	6	6	6	6	6	6	6	5
Dozer	264 kW	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Front-end loader	1.5 m^3	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Rotary drill	172 mm	2	2	3	3	3	3	3	3	3	3	3	3	3	2
Grader	115 kW	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Water truck	9500 L	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Fuel truck		1	1	1	1	1	1	1	1	1	1	1	1	1	1
Light tower	7.8 kW	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Light vehicle		3	3	4	4	4	4	4	4	4	4	4	4	4	4
Mini van		1	1	1	1	1	1	1	1	1	1	1	1	1	1
Bus		1	1	1	1	1	1	1	1	1	1	1	1	1	1

1 Year 2021 represents the period of July 2021 to December 2021
2 Year 2034 represents approximately 5 months of production at the end of the mine life

16.6 Mine Equipment

The following section discusses the fleet requirements to carry out the proposed mine plan. The mine is operated by a contractor with their own fleet. Table 16.7 presents the proposed equipment fleet.

16.6.1 Haulage Trucks

Haul truck requirements were estimated for the ore transport from the mine to either the leach pad, ore stockpile and/or mill, as well as for the waste from the pit to the waste stockpiles. The Project's Contractor currently uses 32-tonne trucks. The following parameters were used to calculate the number of trucks required to carry out the mine plan:

- Mechanical availability: 85%;
- Utilisation: 90%;
- Shift schedule: two 12-hour shifts per day, 330 days a year;
- Operational delays: 100 min/shift;
- Rolling resistance: 3%;
- Maximum speed: 45 km/h overall; 20 km/h downhill.

Haul routes were designed for each period of the mine plan to calculate the truck cycle times from the pits to the different destinations. The cycle times are a function of the haulage time, queuing time, spot time, load time and dump time. The TALPAC 3D © software was used to calculate the haulage times for each period. The haulage times are presented in Table 16.8.

Table 16.8 – Haulage Times

Period	To Waste	To Processing
2021	9.00	8.84
2022	8.47	9.86
2023	9.89	10.18
2024	9.96	10.47
2025	12.13	12.79
2026	11.60	12.20
2027	13.11	13.77
2028	15.40	16.15
2029	14.30	15.02
2030	12.50	15.50
2031	18.00	18.70
2032	22.00	19.20
2033	21.30	23.50

16.6.2 EXCAVATORS

The excavator requirements were estimated based on the number of mining faces and tonnages to be extracted. The Contractor currently uses 2.3 m^3 hydraulic shovels and 1.5 m^3 front end loaders. The following parameters were used to calculate the number of excavators and loaders required to carry out the mine plan:

- Mechanical availability: 85%;
- Utilisation: 90%;
- Shift schedule: two 12-hour shifts per day, 330 days a year;
- Operational delays: 100 min/shift.

16.6.3 DRILLING AND BLASTING

Production drilling is carried out by the Contractor while the blasting is undertaken by a sub-contractor. Blasthole drilling is carried out using rotary drills producing 127 mm diameter, 5 m blast holes in both ore and waste. The burden and spacing are 3.5 m and 4.5 m, respectively, and both emulsion and ANFO are used in the blasts. Steppe Gold is responsible for providing diesel fuel as it is applied to ANFO blasting agents.

The following parameters were used to calculate the number of drills required to carry out the mine plan:

- Mechanical availability: 85%;
- Utilisation: 80%;
- Shift schedule: two 12-hour shifts per day, 330 days a year;
- Operational delays: 100 min/shift.

16.7 Manpower Requirements

Manpower requirements for the Project include both Owner and Contractor needs. Owner manpower requirements are presented in Table 16.9 and the proposed Contractor requirements are presented in Table 16.10.

Table 16.9 – Owner Manpower Requirements

Position	Quantity
Mine manager	1
Superintendent	1
Foreman	3
Mine planner	1
Senior geologist	1
Mining geologist	1
Resource geologist	1
Assistant geologist	4
Total	**13**

Table 16.10 – Proposed Contractor Manpower Requirements

Position	2021[1]	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034[2]
Driller	4	4	4	4	4	6	6	6	6	6	6	6	6	4
Excavator	6	6	8	10	10	12	12	12	12	12	12	12	12	10
Truck driver	12	18	24	30	30	38	40	46	44	42	50	56	58	42
Heavy equipment operator	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Utility operator	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Blaster	2	2	3	3	3	4	4	4	4	4	4	4	4	3
Mechanic	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Electrician	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance worker	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Labourer	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Total Work Force	**41**	**47**	**56**	**64**	**64**	**77**	**79**	**85**	**83**	**81**	**89**	**95**	**97**	**76**
Workers per Shift	**21**	**24**	**28**	**32**	**32**	**39**	**40**	**43**	**42**	**41**	**45**	**48**	**49**	**38**

1 Year 2021 represents the period of July 2021 to December 2021
2 Year 2034 represents approximately 5 months of production at the end of the mine life

17 RECOVERY METHODS

The process described below covers two (2) phases of the overall process design as described in Section 17.1. Phase 1 is deemed to be existing and in place and is described in Section 17.2. Phase 1 work is described to show a complete process from the crusher to the loadout.

The current mandate (the subject of this report) is to provide the design for Phase 2 work only which is described in Section 17.3.

In general, the overall Project comprises two (2) distinct phases:

- Phase 1 – Heap Leach (Oxide Ore) - Completed

  The oxide portion of the ATO Project process employs a conventional oxide heap leach flowsheet including crushing, heap leaching, and gold recovery facilities.

  Phase 1 of the Project has been operational since 2020 and remains operational as of the Effective Date of this Technical Report. The upgraded three-stage crushing system and ore storage facility (purchased by Steppe Gold and currently being installed) is part of Phase 1.

- Phase 2 – Concentrator (Fresh and Transition Ores) – Design in Progress

  The Phase 2 Concentrator will consist of collecting the crushed ore beneath the ore storage building, conveying to the concentrator, milling, flotation, and dewatering unit operations to produce saleable concentrates of lead, zinc, and pyrite.

  Tailings will be disposed of in the new Tailings Storage Facility (TSF).

17.1 Overall Process Design

17.1.1 Phase 1 – Heap Leach (Oxide Ore)

The existing oxide ore processing facilities include the following unit operations:

Crushing and Ore Handling

- **Primary Crusher**: a vibrating grizzly screen and jaw crusher in open circuit producing a product P_{80} of approximately 190 mm;
- **Secondary and Tertiary Crusher**s: a vibrating screen and cone crushers operating in closed circuit producing a final product P_{80} of 25 mm; and
- **Heap Placement**: crushed ore stacked to a 3,000 t capacity stockpile.

Heap Leach Pad

- **Ore Heap Leaching**; and
- **Barren Solution Delivery and Pregnant Solution Recovery Piping Systems**.

ADR Plant

- **Carbon-in-Column (CIC) Adsorption**: adsorption of solution gold onto carbon particles;
- **Desorption**: acid wash of carbon to remove inorganic foulants, elution of carbon to produce a gold-rich solution, thermal regeneration of carbon to remove organic foulants; and
- **Recovery and Refining**: gold electrowinning (sludge production), filtration, drying, mercury retorting, and smelting to produce gold doré.

17.1.2 Phase 2 – Concentrator (Fresh and Transition Ores)

The Phase 2 Concentrator facilities will include the following:

- The crushed ore feeders located under the ore storage facility and conveyors to the concentrator; grinding, and classification;
- Sequential Flotation Circuits for Concentrates of Lead, Zinc and Pyrite;
- Dewatering of Concentrates of Lead, Zinc, and Pyrite; and
- Tailings Thickening, Handling, and Disposal.

17.2 Phase 1 – Heap Leach Process Description

The information in this section is largely drawn and/or summarised from the Report available on SEDAR entitled: "Altan Tsagaan Ovoo Project (ATO), 2021 Mineral Resources, Technical Report (Amended NI 43-101), prepared by GeoRes, with an effective date March 30, 2021, issued on March 30, 2021 and amended on June 9, 2021. Further details as documented therein remain correct and valid.

The existing oxide ore process flowsheet was designed on the basis of leaching 1.2 Mtpa of ore at an average gold grade of 1.13 g/t at 70% gold recovery and average silver grade of 9.25 g/t at 40% silver recovery. The current operation has been operating since 2020 and will continue to operate intermittently until the completion of operations according to the mine plan which is Q4 2023.

The existing crushing plant was designed to operate at a nominal throughput of 5,860 tpd for 275 days per year (75% utilisation). There were some supply chain and parts issues with the main crusher in the first and second quarters of 2020 which were resolved in Q3 2020 and higher crushing plant throughout are presently being achieved.

A new three-stage crushing plant was purchased in Q3 2020 which will increase crushing rates above design capacity. The crushing circuit will consist of primary, secondary, and tertiary stages and will feed the Heap Leach Facility (HLF) at a rate of 1,000 tph with a product size (P_{100}) of approximately 10 mm.

The existing ADR process plant is located near and down-gradient from the HLF which minimises the pumping and pipeline requirements for pregnant and barren solutions and operates 365 days per year. The pregnant solution is designed to flow to the plant at a nominal rate of 200 m^3/h and design flowrate of 250 m^3/h.

The plant processes 5 t of carbon per strip using the ADR process: CIC Adsorption, Desorption (elution) and Recovery (Electrowinning and Gold room) to extract gold from the pregnant solution to produce gold doré.

17.2.1 Plant Design Criteria

The key process design criteria for Phase 1 are listed in Table 17.1.

Table 17.1– Parameter – Key Process Design Criteria for Phase 1

Description	Unit	Value
Design Plant Throughput (Dry)	Mtpa	1.2
ATO Oxide Gold Grade	g/t	1.13
Ore Moisture Content	%	2.5
Crushing and Stacking Availability	%	75
Crushing and Stacking Operating	h/a	4,950
Crushing and Stacking Ore Processing Rate	t/h	242
Crushing Feed Top Size (F_{100})	mm	800
Crushing Feed Product Size (P_{80})	mm	25
Heap Leach Pad Stacked Density	t/m^3	1.6
Leach Cycle Time	days	60

17.2.2 Production Summary

For Phase 1, the ATO Gold Mine achieved commercial production in 2020. By Q4 2020, 1,531,790 t of ore had been mined and 1,068,462 t of crushed and stacked ore were under irrigation at a gold grade of 2.03 g/t Au on the heap leach pad (Cells 1 and 2) for 69,734 oz of gold stacked. Steppe Gold commenced stacking of Cell 1 in Q4 2019 and Cell 2 in June 2020. Gold produced as of December 31, 2020 was 33,154 oz or 47.5% gold recovered from stacked ore. An additional 170,130 t were mined in Q1 2021 for a total of 1,701,920 t and 189,283 crushed, stacked and under irrigation at a grade of 1.91 g/t Au for a total of 1,257,745 t at 81,357 oz. In Q1 2020, no additional gold was recovered giving an overall reconciled gold recovery of 40.8%.

Ultimate gold recoveries above 70% are expected upon completion of irrigation. Table 17.2 presents the production figures for 2019, 2020 andQ1 2021.

Table 17.2 – Production Summary – Oxide Plant

Description	Unit	2019	2020	Q1 2021
Waste Mined	bcm	8,999	318,591	99,910
Ore Mined	t	393,581	1,138,209	170,130
Ore Processed	t	369,258	699,204	189,283
Gold Grade (1)	g/t	-	2.03	1.91
Gold under Irrigation				
Gold Produced	oz	-	33,154	-
Gold Sold	oz	-	31,733	945
Silver Produced	oz	-	35,563	-
Silver Sold	oz	-	13,710	861

(1) Grade is in respect of the gold grade of ore fed through the heap leach pad.

17.2.3 Production Rate and Products

Proven and Probable reserves were used to schedule mine production, and inferred resources onside of the pit were considered as waste. The final production schedule uses trucks and shovels as required to produce the ore to be fed into the process plant and maintain stripping requirements for each case. Table 17.3 shows the mine-production schedule for the ATO oxide ore during years 2021 to 2023.

Table 17.3 – Oxide Plant Production Forecast

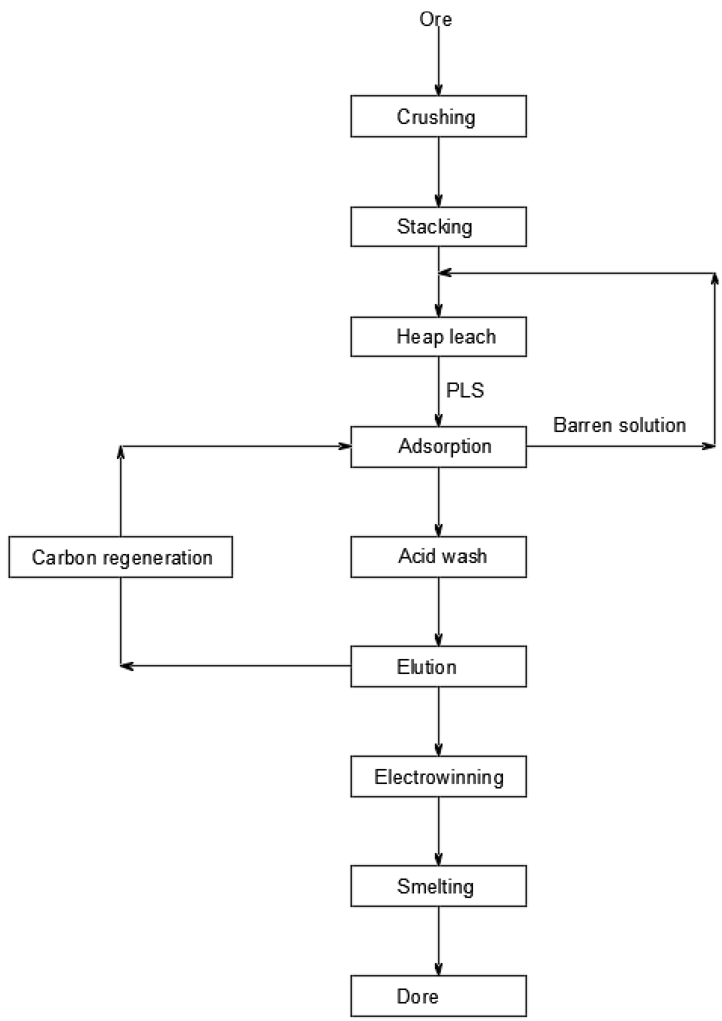
Year	Total Rock (t)	Oxide Ore ROM (t)	Waste (t)	Au (g/t)	Ag (g/t)	Au Eq (g/t)	Contained Metal	
							Au (k oz)	Ag (k oz)
2021	2,372,660	1,160,303	1,212,357	1.55	8.56	1.63	57.7	319.2
2022	3,104,549	1,236,000	1,868,549	1.23	19.78	1.42	48.7	785.9
2023	201,988	184,819	17,170	0.37	7.14	0.44	2.2	42.4
Total	**5,679,197**	**2,581,121**	**3,098,075**	**1.31**	**13.83**	**1.44**	**108.6**	**1,147**

Note: Figures may not add due to rounding

17.2.4 Process Description

A block flow diagram summarising the existing plant and process flows is shown in Figure 17.1.

Figure 17.1 – Oxide Plant and Process Flows



17.2.4.1 Crushing

The crushing plant operates 275 days per year. If the crushing plant is down, the mine haul trucks dump onto the Run-of-mine (ROM) stockpile. A Front end loader (FEL) is used to reclaim the ROM material and deliver the material to the dump pocket. The ROM stockpile is also used to feed the crusher if the mining operations are suspended.

A single crushing circuit is utilised for preparation of the heap leach feed. The circuit consists of a primary jaw crusher, a secondary crusher and an optional tertiary cone crusher operating in closed circuit with a final product screen.

The ROM ore is trucked from the open pits and dumped directly into a primary feed hopper. Primary crusher feed is drawn from the feed hopper by a feeder discharging onto a vibrating grizzly screen.

The grizzly screen oversize feeds the primary jaw crusher. The grizzly undersize and jaw crusher product are transported to the secondary screen by a secondary screen feed conveyor.

Ore from the secondary screen feed conveyor is transported to the secondary vibrating screen. Screen oversize material is conveyed to the secondary cone crusher. The secondary cone crusher discharge and jaw crusher product are collected on the secondary screen feed conveyor back to the secondary screen. Screen undersize material is conveyed to the 3,000 t heap leach feed stockpile. Lime is added to the stockpile feed conveyor from the 200 t lime silo by screw conveyor for pH control.

17.2.4.2 Heap Leach Facility (HLF)

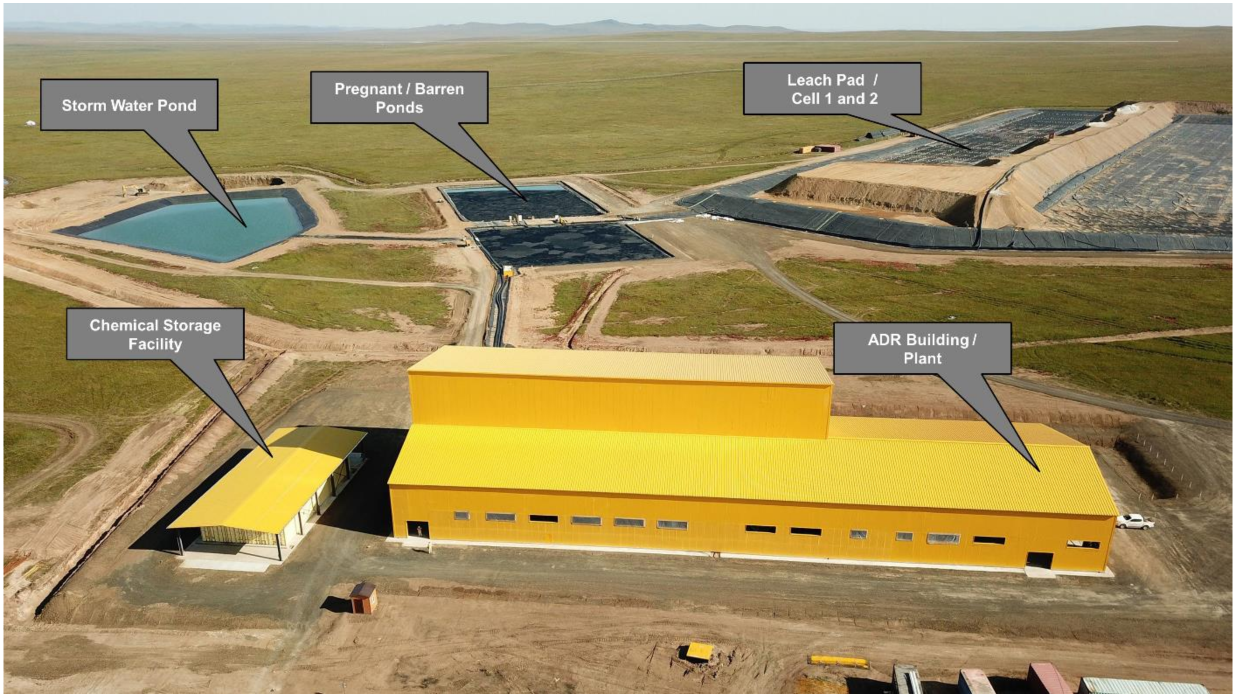
The existing HLF is designed to allow crushed ore stacking to a maximum height of approximately 24 m (measured vertically over the liner system), which results in a design capacity of 5.6 Mt. The HLF comprises:

- Conventional, three lift stages (nominally 8 m per lift), free-draining heap over a gently sloping heap leach pad along the axis of the ridgeline west of the ADR plant.
- Leach Pad constructed in a nominally balanced cut-and-fill manner using locally borrowed (within heap boundary) rock for structural fill, supplemented as needed by mine waste including waste rock and, if available, thaw-stable soil for lining the pad subgrade before placement of the liner system.
- Permanent and interim perimeter diversion channels and berms to manage surface water flows.
- Perimeter access and ore haulage roads.
- Leach pad liner system constructed in steps as described below:
  - Graded subgrade to provide a non-puncturing surface for the geosynthetic liner;
  - Leak detection using horizontal wick drains to operate as large-scale lysimeters;
  - Primary geomembrane liner, 1.5 mm thick linear low-density polyethylene (LLDPE), bottom side aggressively textured;
  - Drainage pipes installed to remove solution and minimise hydraulic head directly over geomembrane; and
  - Gravity drainage from the leach pad to the pregnant tank at the ADR plant or (in the case of an upset) events ponds in double-contained and buried pipes.

Originally a single emergency pond was to be constructed for storm water collection. This will be replaced with two separate storm water ponds, the first of which is now constructed. The second one will be constructed later in the mine life.

Figure 17.2 depicts the ADR Building / Plant / Chemical Storage Facility / 2 Ponds as well as the Leach Pad Cells 1 and 2; Figure 17.3 shows an interior view of the ADR Building / Plant.

Figure 17.2 – Existing Phase I Site Processing Facilities



Source: Steppe Gold Photo

Figure 17.3 – Existing Phase I Site Processing Facilities

Source: Steppe Gold Photo

17.2.4.3 ADR Gold Recovery Plant

1. Carbon Adsorption

The carbon adsorption circuit consists of a train of five cascading carbon columns. The pregnant (gold-enriched) solution is pumped to the carbon adsorption circuit across a stationary trash screen for the removal of any debris from the heap leach pad. The solution flows counter-current to the movement of carbon from column 1 to column 5. The solution overflow from the final column discharges onto a screen in order to recover any carbon. The barren solution, which at this stage has adsorbed most of the gold in solution, discharges from the final carbon column and is pumped to the barren tank.

Cyanide solution, caustic solution, anti-scalant and make-up water are added to the barren tank as needed. Barren solution is heated to increase solution temperature by 8°C before being pumped back to the leach pad in order to maintain the thermal integrity of the heap leach pad. On average, 5 t of loaded carbon from the first carbon column are pumped to the acid wash and stripping circuits each day. The carbon in the second column is advanced to the first tank and the process is continued down the train. The carbon from the sixth column advances to the fifth column and then freshly reactivated carbon is added.

2. Desorption and Gold Refining

The loaded carbon is transferred to the acid wash vessel and treated with 3% hydrochloric acid solution to remove calcium, magnesium, sodium salts, silica, and fine iron particles. Organic foulants such as oils and fats are unaffected by the acid and are removed after the stripping or elution step by thermal reactivation utilising a kiln. The dilute acid solution is pumped into the bottom of the acid wash vessel, exiting through the top of the vessel back to the dilute acid tank. At the conclusion of the acid wash cycle, a dilute caustic solution is used to wash the carbon and neutralize the acidity.

A recessed impeller pump transfers acid washed carbon from the acid wash tank into the strip or elution vessel. Carbon slurry discharges directly into the top of the elution vessel. Under normal operation, only one elution takes place each day.

3. Carbon Stripping (Elution)

After acid washing, the loaded carbon is stripped of the adsorbed gold using a modified Zadra process. The strip vessel holds approximately 5t of carbon. During elution a solution containing approximately 1 % sodium hydroxide and 0.1 % sodium cyanide is heated to a temperature and pressure of 140°C and 450 kPa respectively and circulated through the strip vessel. Heat from the outgoing pregnant solution tank is transferred to the incoming cold barren solution. A diesel-powered boiler is used as the primary solution heater to maintain the barren solution at 140°C. The cooled pregnant solution flows by gravity to the electrowinning cells. At the conclusion of the strip cycle, the stripped carbon is rinsed and then pumped to the carbon regeneration circuit.

4. Carbon Regeneration

The stripped or eluted carbon from the strip vessel is pumped to the vibrating carbon-sizing screen. The kiln-feed screen doubles as a dewatering screen and a carbon-sizing screen, where fine carbon particles are removed. Oversize carbon from the screen discharges by gravity to the 7.5 t carbon-regeneration kiln-feed hopper. Screen undersize carbon drains into the carbon-fines tank and then can be filtered and bagged for disposal. A 250 kg/h diesel-fired horizontal kiln treats 5.0 tpd of carbon at 650°C, equivalent to 100% regeneration of carbon. The regeneration kiln discharge is transferred to the carbon quench tank by gravity, cooled by fresh water or with carbon-fines water, prior to being pumped back into the CIC circuit.

To compensate for carbon losses by attrition, new carbon is added to the carbon attrition tank.

New carbon and fresh water are mixed to break off any loose pieces of carbon prior to being combined with the reactivated carbon in the carbon holding tank.

5. Refining

Pregnant solution flows by gravity to a secure gold room. The solution flows through one of two 3.54 m^3 electrowinning cells. Gold is plated onto knitted-mesh steel wool cathodes in the electrowinning cell. Loaded cathodes are power washed to remove the gold-bearing sludge and any remaining steel wool. The gold-bearing sludge and steel wool are filtered to remove excess moisture and then retorted to remove any mercury. The retort residue is mixed with fluxes consisting of borax, silica and soda ash before being smelted in an induction furnace to produce gold doré and slag. The doré is then transported to an off-site refiner for further purification. Slag is processed to remove prills for re-melting in the furnace. The gold bars are stored in a vault located in the gold room prior to secure off-site transportation by aircraft.

6. Reagents

Sodium cyanide (NaCN) briquettes are delivered to site in containers and in one tonne super sacks contained in a wood frame. The briquettes are mixed in the cyanide mix tank and subsequently transferred to the cyanide solution storage tank. The concentrated cyanide solution is added to the barren tank at a rate of 0.2 kg/t of ore. Cyanide is used in the carbon strip circuit at a concentration of 0.1%. The principles and standards of practice for the transport to site and handling of cyanide on site are in accordance with the guidelines set out in the International Cyanide Management Code (ICMC).

Sodium Hydroxide (caustic) is supplied to site in one tonne totes. The caustic is mixed and stored for distribution to the acid wash and strip circuits. The caustic is used to neutralise the acid in the acid wash circuit. A solution of 1.0% caustic is mixed with barren solution in the carbon strip circuit.

Hydrochloric acid and anti-scalant solutions are supplied to site in one tonne totes. The solutions are metered directly from the totes for distribution in the plant.

Hydrated lime is delivered to the site in bulk by trucks and stored in a 200 t lime silo. The lime is delivered at a rate of approximately 2.7 kg/t of ore by screw feeder onto the heap leach feed conveyor during heap loading operations.

7. Laboratory

An on-site assay and metallurgical laboratory is equipped to perform sample preparation and assays by atomic absorption (AAS technology). The laboratory facility supports minor environmental sampling, Total Suspended Solids (TSS) monitoring and processing. The majority of the environmental samples are sent off-site to an accredited laboratory for third-party reporting. The laboratory has space available for process optimisation and test program. All exploration drill samples, mine grade control and some process samples are sent off site.

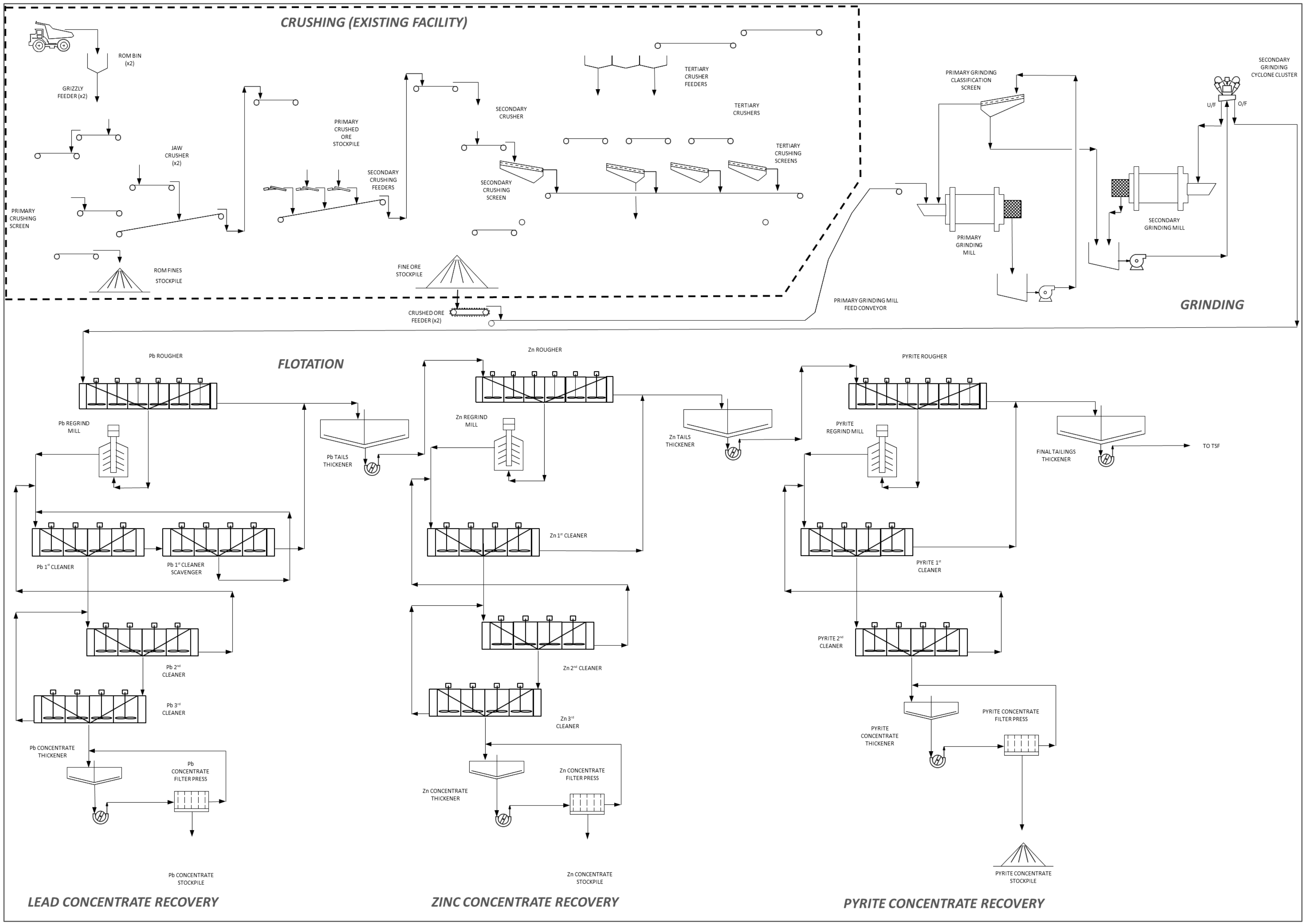
17.3 Phase 2 – Concentrator Process Description

Phase 2 consists of the following unit operations:

- Grinding and Classification;
- Lead Concentrate Flotation;
- Zinc Concentrate Flotation;
- Pyrite Concentrate Flotation;
- Lead Concentrate Dewatering;
- Zinc Concentrate Dewatering;
- Pyrite Concentrate Dewatering;
- Tailings Thickening and Handling;
- Water Services;
- Air Services;
- Electrical Power; and
- Process Consumables.

An overall flow diagram summarising the concentrator plant and process flows is provided in Figure 1.3.

Figure 17.4 – ATO Phase 2 Project Overall Flowsheet



17.3.1 Plant Design Criteria

The key process design criteria for Phase 2 are outlined in Table 17.4.

Table 17.4 – Key Process Design Criteria for Phase 2

Parameter	Units	Value
Plant Throughput	Mtpa	2.2
Pb Head Grade	%	0.47
Zn Head Grade	%	0.85
Au Head Grade	g/t	1.15
Ag Head Grade	g/t	10.75
Bond Ball Milling Work Index	kWh/t	17.1
Primary Crushing Plant Availability	%	65
Secondary / Tertiary Crushing Plant Availability	%	75
Concentrator Availability	%	90
Material Specific Gravity	-	2.6
Material Moisture Content	%	3.0
ROM Feed Size (F_{100})	mm	800
Crushing Plant Product Size (P_{80})	mm	10
Flotation Feed Size (F_{80})	µm	160
Pb Recovery in Pb Concentrate	%	82.5
Au Recovery in Pb Concentrate	%	41.2
Ag Recovery in Pb Concentrate	%	45.6
Pb Grade in Pb Concentrate	% Pb	64.5
Au Grade in Pb Concentrate	g/t	49.1
Ag Grade in Pb Concentrate	g/t	376.1
Zn Recovery in Zn Concentrate	%	85.9
Au Recovery in Zn Concentrate	g/t	14.1
Ag Recovery in Zn Concentrate	g/t	18.2
Zn Grade in Zn Concentrate	% Zn	55.4
Au Grade in Zn Concentrate	g/t	6.9
Ag Grade in Zn Concentrate	g/t	52.8
Au Recovery in Pyrite Concentrate	%	20.9
Ag Recovery in Pyrite Concentrate	%	10.2

Parameter	Units	Value
Au Grade in Pyrite Concentrate	g/t	11.3
Ag Grade in Pyrite Concentrate	g/t	32.8
Total Au Recovery	%	76.2
Total Ag Recovery	%	74.0

The production forecast of the Phase 2 concentrator is outlined in Table 17.5.

Table 17.5 – Phase 2 Production Forecast

Year	Feed					Pb Concentrate				Zn Concentrate				Pyrite Concentrate		
	Tonnes	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	Tonnes	Au (g/t)	Ag (g/t)	Pb (%)	Tonnes	Au (g/t)	Ag (g/t)	Zn (%)	Tonnes	Au (g/t)	Ag (g/t)
1	286,838	2.35	15.91	0.68	1.04	2,747	100.92	756.67	58.62	4,300	22.13	193.43	59.29	13,376	10.50	34.70
2	2,181,815	1.83	11.76	0.74	1.11	22,092	74.35	529.18	60.40	35,410	15.93	132.11	58.66	79,245	10.50	32.94
3	2,220,676	1.35	14.40	0.47	0.86	17,020	72.55	856.09	50.43	26,739	15.85	218.05	61.30	59,575	10.50	54.60
4	2,220,680	1.82	10.12	0.26	0.49	12,927	128.52	792.09	37.45	13,054	43.70	313.88	72.01	80,157	10.50	28.52
5	2,220,680	1.40	13.69	0.28	0.50	13,157	97.50	1052.50	38.39	13,366	32.95	414.57	71.52	61,889	10.50	49.96
6	2,220,680	0.97	6.64	0.72	1.23	21,957	40.30	306.05	59.65	40,460	7.51	66.46	57.84	42,685	10.50	35.15
7	2,220,680	0.77	5.66	0.71	1.42	21,877	32.02	261.80	59.53	47,525	5.06	48.23	56.83	33,792	10.50	37.84
8	2,220,680	1.11	8.84	0.33	0.60	14,272	71.44	626.73	42.53	17,052	20.53	209.90	67.10	49,187	10.50	40.60
9	2,220,680	0.87	9.58	0.42	0.80	16,025	49.52	604.60	47.88	24,604	11.07	157.57	62.19	38,285	10.50	56.50
10	2,220,680	0.86	10.25	0.39	0.83	15,518	50.42	668.26	46.46	25,576	10.50	162.24	61.76	37,747	10.50	61.34
11	2,220,680	0.63	6.29	0.53	0.98	18,269	31.45	348.66	53.23	31,188	6.32	81.72	59.84	27,720	10.50	51.30
12	1,007,581	0.55	32.06	0.08	0.17	8,034	28.60	1831.92	8.30	14,151	5.57	416.19	10.44	11,084	10.50	296.46
LOM	**23,462,35**	**1.15**	**10.75**	**0.47%**	**0.85%**	**183,894**	**60.27**	**625.17**	**49.43%**	**293,425**	**12.97**	**156.78**	**58.54%**	**612,348**	**10.50**	**36.20**

17.3.2 Primary, Secondary, and Tertiary Crushing and Stockpile (Existing from Phase 1 Expansion)

The following crushing description is for information purposes only. The crushing system is purchased by Steppe Gold and currently under construction, and for the purpose of this Project is considered to be existing.

The crushing circuit is designed for a capacity of 2.2 Mtpa and will be installed as part of the Phase 1 expansion.

A three-stage crushing circuit will reduce run-of-mine (ROM) material from an F_{100} of 800 mm to a P_{80} of 10 mm. The primary crushing circuit is utilised for an annual operating time of 5,694 h/a or 65% utilisation and operates in open circuit.

ROM material is dump-fed into ROM hoppers, installed in parallel. The primary crusher feed will be drawn from the ROM hoppers by vibrating grizzly feeders to feed primary jaw crushers, installed in parallel. Grizzly feeder undersize (U/S) is bypassed and conveyed to a primary crushing screen allowing for U/S material to be stockpiled.

Jaw crusher product from both streams is combined with primary crushing screen oversize (O/S) and conveyed to the primary crushed ore stockpile. The primary crushed ore stockpile has a feed rate of 361 tph with a full capacity of 5,953 t. Primary crushed material is reclaimed via three vibrating pan feeders at a controlled rate of 313 t/h to feed the secondary and tertiary crushing circuit via a conveyor.

The secondary and tertiary crushing circuits are utilised for an annual operating time of 6,570 h or 75% utilisation and operates in closed circuit with screens. Secondary crusher product is screened with U/S reporting to the fine ore (mill feed) stockpile and O/S from the secondary crushing screen is conveyed to one (1) of three (3) tertiary crushing bins and distributed evenly via fixed trippers. Tertiary crushing is completed using three cone crushers installed in parallel in a closed circuit. The cone crusher product is conveyed to dry screens for classification. O/S material is combined with secondary crushing screen O/S and recirculated to the tertiary crushing bins. Tertiary screen U/S is combined with the secondary screen U/S and conveyed to the fine ore stockpile which has a live capacity of 1,400 t.

17.3.3 Concentrator

17.3.3.1 Grinding

Crushed ore product is reclaimed via one of two apron feeders installed underneath the fine ore stockpile. Fresh feed is collected at a controlled rate to feed the concentrator feed conveyor. The concentrator is utilised for an annual operating time of 90% utilisation.

The grinding circuit consists of two-stage sequential grinding with a primary ball mill in closed circuit with a classification screen followed by a secondary ball mill in closed circuit with hydrocyclones. Fresh feed is weighed prior to discharging into the primary ball mill feed trolley and combined with process water. The product is discharged through a trommel with O/S screened out and discharged to a scats bunker. U/S is discharged into a pump box and pumped to a classification screen. The O/S is gravitated back to the primary grinding mill and combined with fresh feed while U/S material with a transfer size (T_{80}) of 720 µm is gravity-fed to the secondary grinding feed pump box for further size reduction.

The secondary grinding circuit operates in a reverse feed arrangement, with feed classified by a hydrocyclone cluster to bypass fines. The ball mill product is discharged through a trommel with O/S screened out and discharged to a scats bunker. U/S is discharged into the secondary grinding feed pump box and recirculated for further classification and size reduction. The cyclone underflow is fed into the secondary grinding mill feed trolley, along with process water. The overflow is sampled, and gravity fed to the lead rougher flotation conditioning tank via a trash screen to remove foreign material.

17.3.3.2 Lead Concentrate Flotation

The flotation process is separated into lead concentrate, zinc concentrate, and pyrite concentrate circuits to target each of the materials individually and maximize their recoveries. Process water is kept separate for the lead concentrate and zinc concentrate circuits.

Grinding product is combined with process water and reagents and mixed thoroughly. The slurry is conditioned and fed to the lead rougher flotation cells. The circuit consists of six (6) tank cells to provide sufficient flotation residence time.

The lead rougher flotation concentrate is pumped to the lead regrind circuit to liberate the base metal sulphide minerals present to allow for cleaning and separation. The regrind circuit consists of a tower mill operating in open circuit with a scalping cyclone. The overflow material at a P_{80} of 17 µm combines with lead rougher regrind mill discharge and is sent to the first lead cleaner flotation stage. Lead rougher flotation tailings are sent to the lead tails thickener.

The lead first cleaner flotation circuit cells operate in a cleaner/scavenger mode with the cleaner scavenger concentrate circulated back to the lead first cleaner feed box while the cleaner scavenger tailings are directed to the lead tailings thickener. The lead first concentrate is upgraded by second and third cleaner flotation stages to produce the final lead concentrate which is pumped to concentrate dewatering. The overflow from the lead tailings thickener is reused as lead flotation circuit process water.

17.3.3.3 Zinc Concentrate Flotation

The lead tailings thickener underflow is combined with raw water and reagents and mixed thoroughly. The slurry is conditioned and fed to the zinc rougher flotation cells. The circuit consists of six (6) tank cells to provide sufficient flotation residence time.

The zinc rougher flotation concentrate is pumped to the zinc regrind circuit consisting of a vertical mill operating in open circuit with a scalping cyclone. The overflow material of P_{80} of 35 µm is combined with zinc rougher regrind mill discharge and sent to the first zinc cleaner flotation. Zinc rougher flotation tailings are sent to the zinc tailings thickener.

Three (3) zinc cleaner flotation circuits are used to recover respective saleable zinc concentrates with the zinc first cleaner flotation tailings sent to the thickener. The zinc first cleaner flotation concentrate is upgraded by the second and third cleaner flotation stages to produce the final zinc concentrate which is pumped to concentrate dewatering. The overflow from the zinc tailings thickener is re-used as zinc flotation circuit process water.

17.3.3.4 Pyrite Concentrate Flotation

Zinc tails thickener underflow is combined with process water and reagents and mixed thoroughly. The slurry is conditioned and fed to the pyrite rougher flotation cells. The circuit consists of eight (8) tank cells to provide sufficient flotation residence time.

The pyrite rougher flotation concentrate is pumped to the pyrite regrind circuit consisting of a vertical mill operating in open circuit with a scalping cyclone. The overflow material of P_{80} 19 µm combines with pyrite rougher regrind mill discharge and is sent to the first pyrite cleaner flotation. Pyrite rougher flotation tailings are sent to the final tailings thickener.

Two (2) pyrite cleaner flotation circuits are used to recover respective saleable pyrite concentrates with the pyrite tailings sent to final tailings thickener. The pyrite first cleaner flotation concentrate is upgraded by a second cleaner flotation stage to produce the final pyrite concentrate which is pumped to concentrate dewatering. The overflow from the final tailings thickener is sent to the reclaim process water pond.

17.3.3.5 Lead Concentrate Dewatering

Lead concentrate is mixed with flocculant in the thickener feed box. Thickener overflow is discharged into the lead circuit process water tank. Thickener underflow is pumped to a stock tank prior to compressed air filtration. A dewatered concentrate filter cake is stockpiled in a shed and fed to transport trucks via a FEL. Trucks are weighed via a truck scale prior to shipment. Filtrate is returned to the lead concentrate thickener.

17.3.3.6 Zinc Concentrate Dewatering

Similarly, to lead concentrate dewatering, zinc concentrate is mixed with flocculant in the thickener feed box. Thickener overflow is discharged into the zinc circuit process water tank. Thickener underflow is pumped to a stock tank prior to compressed air filtration. Zinc concentrate filter cake stockpiled in a shed and fed to transport trucks via a FEL. Trucks are weighed via a truck scale prior to shipment. Filtrate is returned to the zinc concentrate thickener.

17.3.3.7 Pyrite Concentrate Dewatering

Pyrite concentrate is mixed with flocculant in the thickener feed box. Thickener overflow is discharged to the reclaim process water pond. Thickener underflow is pumped to a stock tank before compressed air filtration. Pyrite concentrate filter cake is stockpiled in a shed and fed to transport trucks via a FEL. Trucks are weighed via a truck scale prior to shipment. Filtrate is returned to the pyrite concentrate thickener.

17.3.4 FINAL TAILINGS THICKENING AND HANDLING

The tailings thickener receives the following feed streams:

- Pyrite rougher tailings, and
- Pyrite cleaner tailings.

These streams are combined in the thickener feed well where flocculant is added to facilitate solids settling. Final tailings thickener overflow is recycled to the reclaim process water pond. Thickener underflow is pumped to the final tailings tank where the tailings are pumped to the TSF. Water from the TSF is reclaimed back to the reclaim process water pond to minimise fresh water make-up.

17.3.5 REAGENTS

17.3.5.1 Lead Concentrate Flotation Reagents

The lead flotation process will use several reagents; a description of each is provided below.

The storage silo will receive dry sodium carbonate (soda ash) from a truck. The mixing system will receive soda ash from a silo. A feeder on the bottom of the silo meters the soda ash in a heated insulated mixing tank. The mix is transferred via pump to the heated and insulated distribution tank. The distribution tank is connected to distribution pumps and will deliver to the primary grinding. The soda ash is used for pH modification.

Zinc sulphate will be delivered dry in bulk bags. A bag breaker on top of the tank will split the bag and its contents will drop into the mixing tank. Three (3) bags per mix will be used. Zinc cyanide is made with 3:1 $ZnSO_4$:NaCN (cyanide existing), pH is controlled with addition of caustic (existing). The mix is transferred via pump to the distribution tank. The distribution tank is connected to distribution pumps and will deliver to the primary grind, lead regrind, and the second and third lead cleaner stages.

Aerophine 3418 is delivered in intermediate bulk containers (IBCs). The tote containers will be hooked up in pairs, both on a decanting frame and connected to distribution pumps. The distribution pumps will distribute it to the to the lead rougher flotation and regrind as well as the first lead cleaner flotation stage.

Liquid Methyl Isobutyl Carbinol (MIBC) is delivered in IBC containers. The tote containers will be hooked up in pairs, both on a decanting frame and connected to distribution pumps. The distribution pumps will distribute it to the lead rougher stage, all the lead cleaner stages, and to the pyrite rougher stage.

17.3.5.2 Zinc Concentrate Flotation Reagents

The zinc flotation process will use several reagents, a description of each is provided below.

Hydrated lime will be delivered to the storage silo from a truck. A feeder on the bottom of the silo meters the hydrated lime in a mixing tank. Lime is transferred via pump to the distribution tank. The distribution tank is connected to distribution pumps and will distribute it to the zinc rougher flotation and regrind stages.

Copper sulphate is delivered dry in bulk bags. A bag breaker on top of the tank will split the bag and its contents will drop into the mixing tank. Three (3) bags are used per mix. The mix is transferred via a pump to the distribution tank. The distribution tank is connected to distribution pumps where it will be delivered to the second zinc rougher flotation stage and the zinc regrind stage.

The system will receive dry Sodium Isopropyl Xanthate (SIPX) in bulk bags. A bag breaker on top of the tank will split the bag and its contents will drop into the mixing tank. One bag per mix will be added. The mix is transferred via pump to the distribution tank. The distribution tank is connected to distribution pumps which will distribute it to both the first and second zinc rougher flotation stages as well as the first zinc cleaner stage.

Polyfroth H57 is delivered in IBCs. The tote containers will be hooked up in pairs, both on a decanting frame and connected to distribution pumps. The distribution pumps will distribute it to the first zinc rougher flotation stage and regrind and both the first and second zinc cleaner flotations.

17.3.5.3 Pyrite Concentrate Flotation Reagents

The pyrite flotation process will use several reagents, a description of each is provided below.

The system will receive dry Carboxymethyl Cellulose (CMC) from bags and keep it in the storage hopper. A feeder underneath the hopper will meter the CMC into a cone and transfer it to the wetting element which will discharge into a mixing tank with an agitator at a mix of 1%. The mix is transferred via pump to the distribution tank which is connected to metering pumps. After the metring pumps, the solution is diluted using in-line mixers to 0.1% and distributed.

The system will receive dry Potassium Amyl Xanthate (PAX) in bulk bags. A bag breaker on top of the tank will split the bag and its contents will drop into the mixing tank which will contain three (3) bags per mix. The mix is transferred via pump to the distribution tank. The distribution tank is connected to distribution pumps and will deliver to the pyrite flotation conditioning tank.

The pyrite process will also use MIBC, which was described previously in Section 17.3.5.1.

17.3.6 UTILITIES AND SERVICES

17.3.6.1 Water Services

The five (5) water circuits (Raw Water, Potable Water Fire Water, Gland Water, and Process Water) have been developed to support the requirements of the plant, and in the case of the potable water circuit, the surrounding infrastructure. Approximately 4,800 m^3 of daily make-up water will be required to support the requirements mentioned above and the ratio is 0.79 $m^3$of fresh water per tonne of dry ore fed to the plant.

a. Raw Water

Raw water demand for the process is minimised with reticulation of process water to meet most process water demands. Raw water is used for reagent preparation and filter press wash water.

b. Potable Water

Filtered raw water is further treated by a UV sterilisation unit for potable water requirements. Potable water is used in the safety showers and eyewash stations, and for potable water reticulation to the non process infrastructure.

c. Fire Water

A dedicated fire water volume has been allocated within the raw water tank. The fire water circuit is powered by a typical fire water skid consisting of a primary electric fire water pump, and a pressure maintaining jockey pump, plus an emergency diesel powered fire water pump to ensure there is still fire water available in the event of loss of power to the process plant.

d. Gland Water

A gland water circuit has been incorporated, supplying gland water to all slurry pumps within the process. This water is fed from the raw water tank with particle filtration on the suction side of the gland water distribution pumps.

e. Process Water

The process water settling pond collects the various thickener overflow streams, reclaim water, and fresh makeup water to enclose the water balance. The process water is then distributed within the concentrator plant through the process water pumps and the delivery piping.

17.3.6.2 Air Services

Compressed air of the required quality, flow, and pressure will be provided by two (2) air compressors, one operating and one stand-by, located in a centralised compressor station, and delivered to the plant users through a piping network. Individual air receivers are in service for concentrate pressure filters.

17.3.6.3 Electrical Power

Power will be provided by diesel gensets, as further detailed in Section 18.

The total process installed electrical power is estimated at 15 MW, while the total process absorbed electrical power is estimated at 11 MW.

Electrical consumption was calculated using expected absorbed power draw as determined for individual equipment items after applying use and electrical correction factors.

17.3.6.4 Process Consumables

Process consumables such as mill grinding media and liners, filter cloth, screen panels, and any lubricants; are delivered from the plant warehouses as per the operational demand.

18 PROJECT INFRASTRUCTURE

18.1 Existing Infrastructure

The ATO mine has been in production since 2020 and has the necessary infrastructure required to support the current open pit mining operation. This includes, but is not limited to, ADR plant, laboratory, fuel storage, chemical storage, power supply, water supply, heap leach facilities and ponds, camp, open pit mining fleet, waste facility, and all the necessary offices, warehouses, and workshops to sustain the current operation.

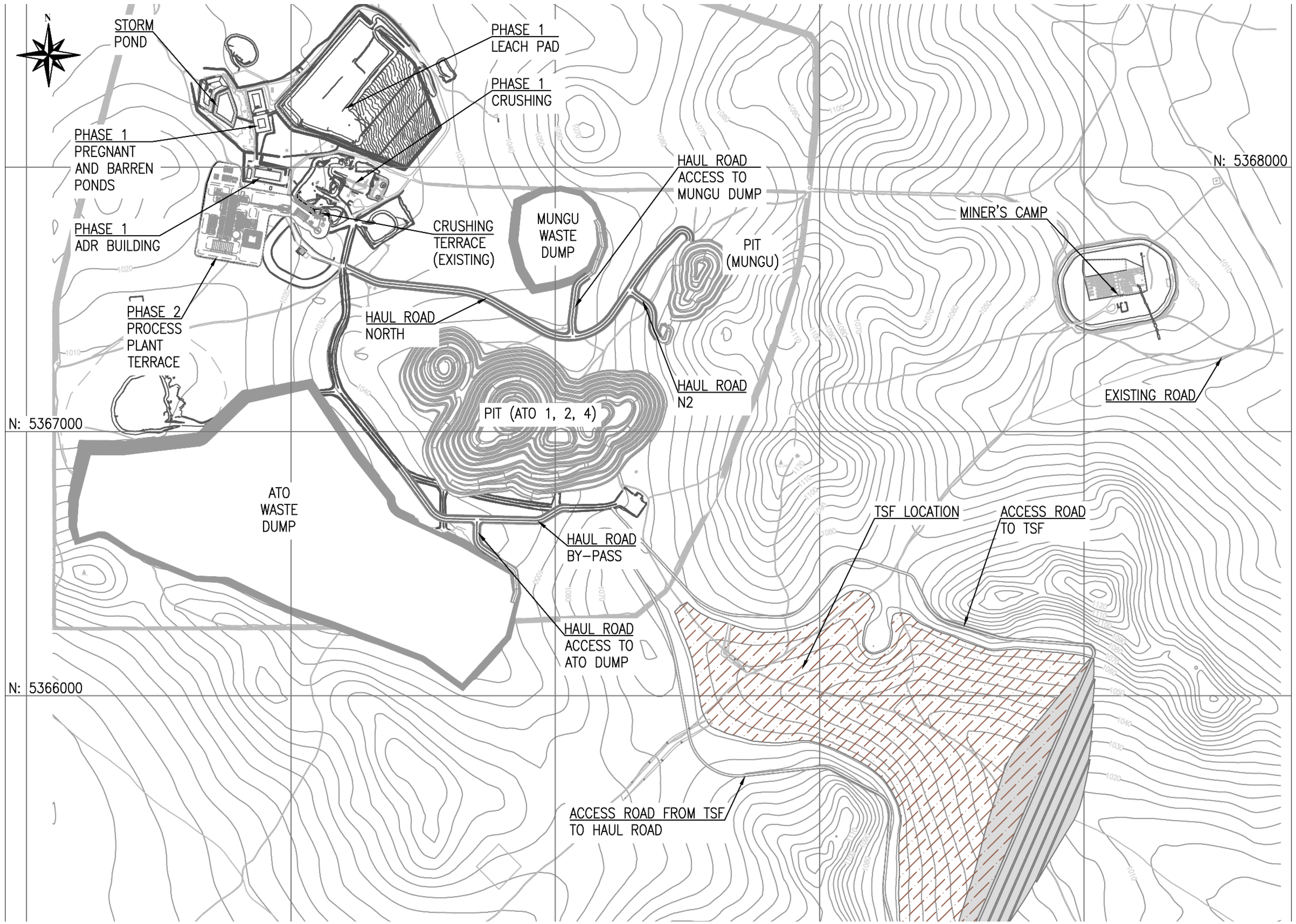
18.2 Overview

For the Phase 2 Expansion Project, this section describes the main Project elements related to process, followed by support infrastructure. Figure 1.4 shows all existing and planned infrastructure and locations of the plant and mines and Figure 1.5 depicts the process plant area.

It should be noted that no geotechnical investigations have been performed to characterise the ground condition for foundation design nor for any borrow materials for any of the facilities presented in this Report. As no geotechnical information was available at the time of developing the design, a field and laboratory investigation program will need to be carried out as part of the next project phase to confirm the assumptions made, or if changes to the design need to be made. This program will include geophysics, drilling and test pitting in the designated area, as well as taking samples for geotechnical laboratory testing.

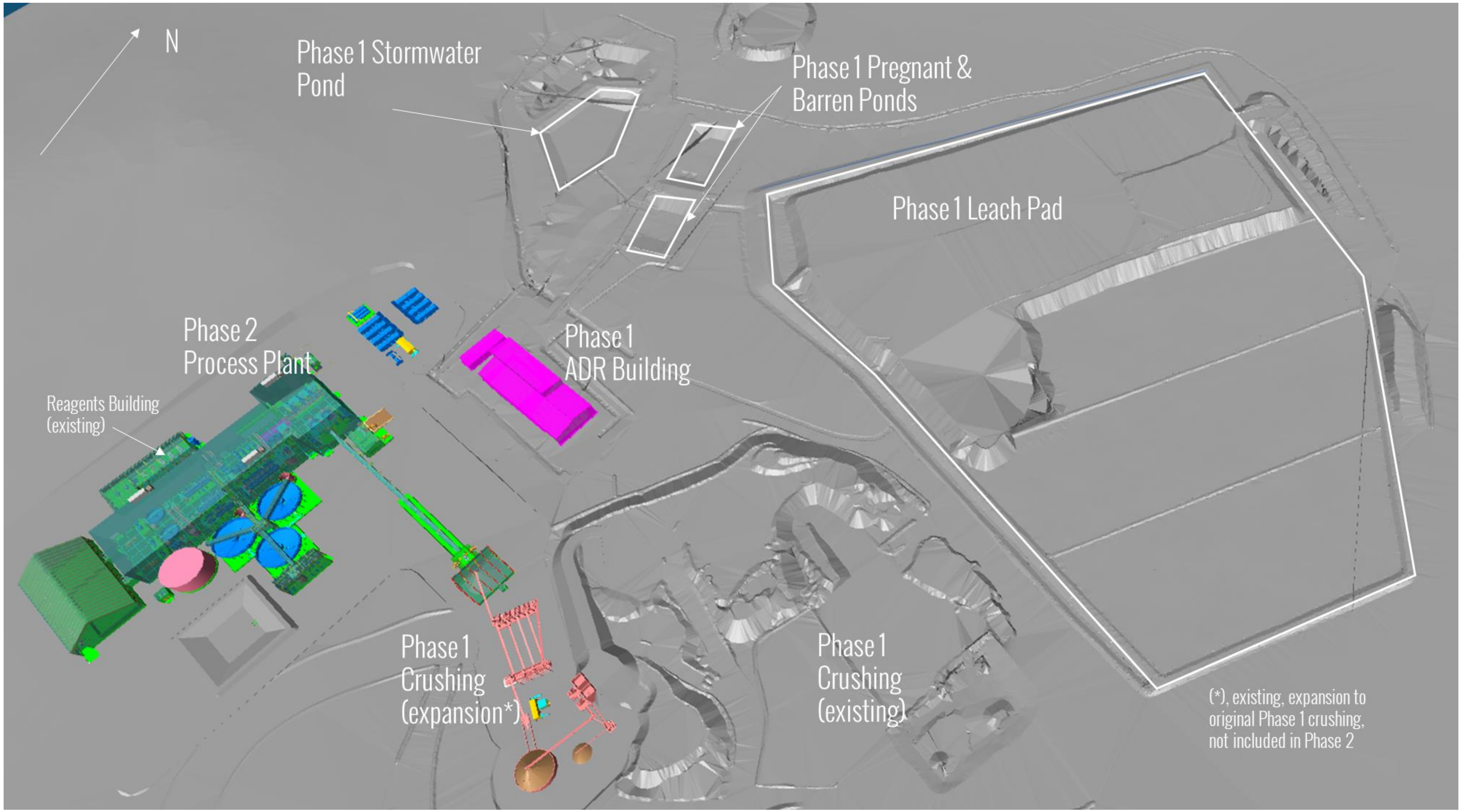
Certain elements of the Phase 1 site process plant infrastructure which are currently under construction will also be utilised in the Phase 2 operation, specifically the crushing circuit and the reagents building.

Figure 18.1 - General Overall Site Plan



Source: DRA 2021

Figure 18.2 – Process Plant Site Layout



Source: DRA 2021

18.3 Facilities

Major facilities, which are included as part of the Project, are described in the following sub-sections:

18.3.1 Processing Plant Buildings

The processing plant area includes various buildings of steel frame and insulated metal panel cladding construction, as shown in Figure 18.3.

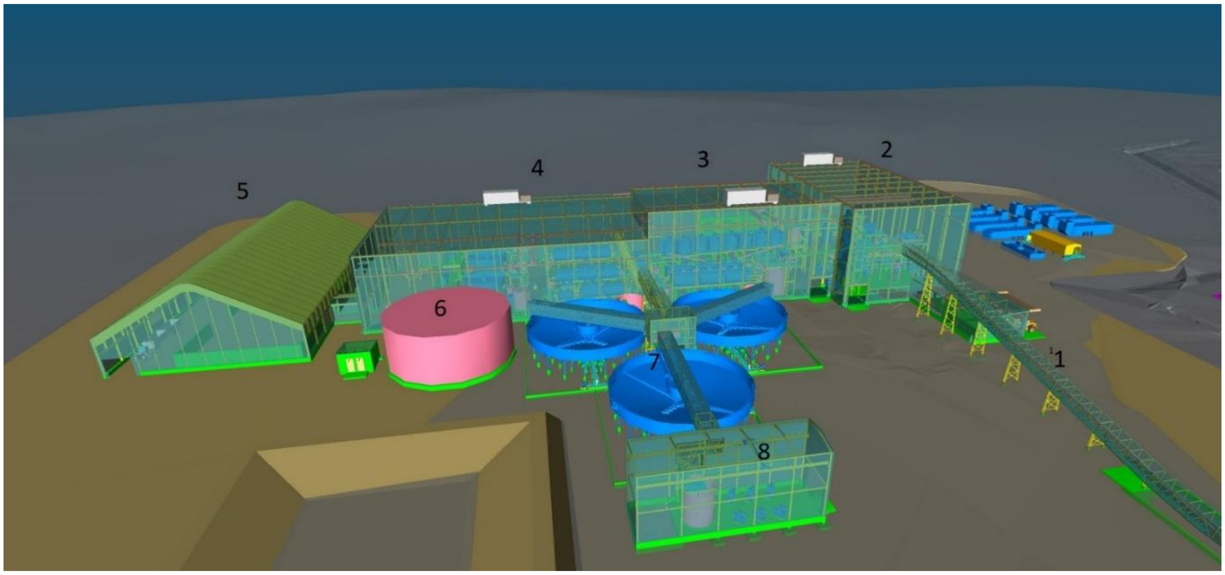
The main process building footprint is approximately 9,180 m² and includes the milling circuit, flotation cells and concentrate thickeners. The concentrate load-out building footprint is approximately 3,500 m², and the footprint of the existing main building housing the reagents is approximately 1,840 m². In addition, there are two smaller buildings, one for zinc cyanide preparation and dosing and one for the soda ash preparation and dosing.

Figures 18.4 and 18.5 illustrate the Process Plant West and East elevations, respectively.

The labels in all three figures correspond to the following:

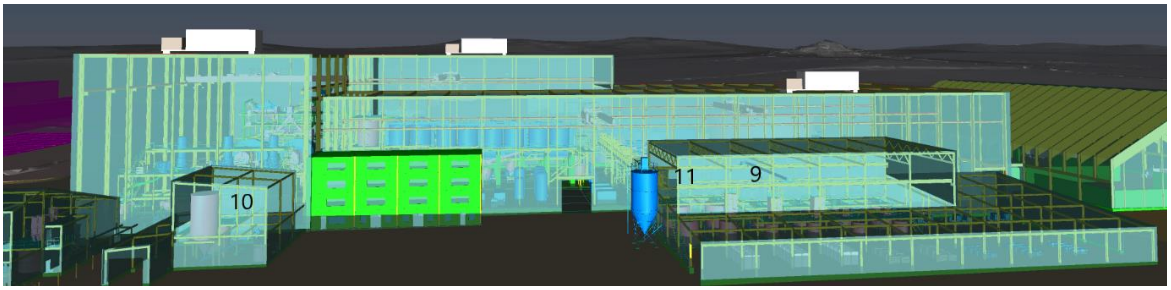
1. Mill Feed Conveyor;
2. Milling and Regrind Area;
3. Rougher Flotation Area;
4. Cleaner Flotation, Concentrate Thickening and Filtration Area;
5. Concentrate Loadout Area;
6. Combined Water Tank;
7. Tailings Thickening Area;
8. Final Tailings Pumping Area;
9. Existing Reagents Building;
10. Zinc Cyanide Mixing and Dosing Building;
11. Soda Ash Mixing and Dosing Building.

Figure 18.3 – Process Plant Buildings



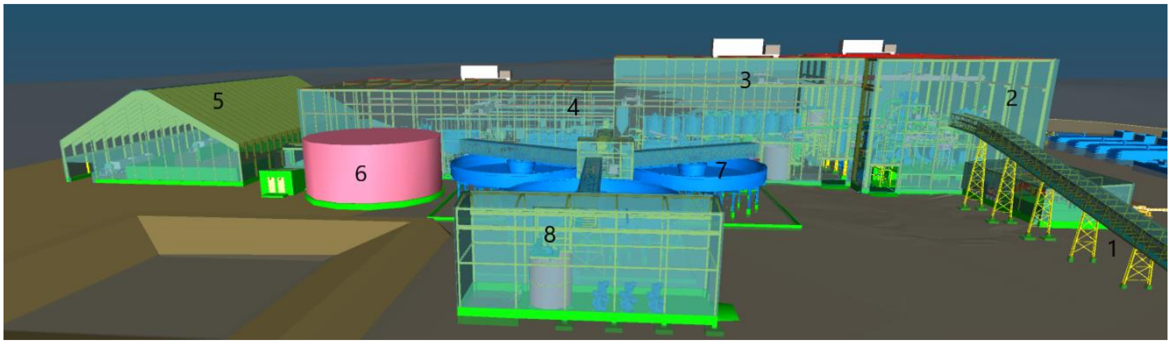
Source: DRA 2021

Figure 18.4 – Process Plant Building West Elevation



Source: DRA 2021

Figure 18.5 – Process Plant Buildings East Elevation



Source: DRA 2021

18.3.2 Tank Farm

Currently, diesel and gasoline are purchased in bulk and stored on-site in four (4) existing refuelling tanks. Each tank has a capacity of 50,000 L and is constructed with full containment systems in the event of tank rupture. Mining and on-site diesel-powered mobile equipment are fuelled at the storage tanks.

The new power demand for the site at full capacity is estimated at 15 MW. As a result of the Phase 2 Expansion Project, additional fuelling capacity is required to supplement the existing tank farm. An additional eight (8) fuel tanks will be required, each having 50,000 L of fuel capacity. These 12 tanks in total will provide a minimum of 6 days of fuel storage for all Project operations.

18.3.3 Explosives

Since the start of mining operations in 2019, ANFO explosive is delivered straight to the hole for blasting. As part of prior work, an explosive storage has now been installed at the site which is located approximately 3 km from open pit. All explosive, detonators and transfer wires are in separate containers within an enclosed fenced area.

18.3.4 Camp

The existing camp constructed at the ATO site has capacity for 300 staff, and includes:

- Kitchen capacity for 250 people. Building space is also allocated for restroom, cold storage area and staff room;
- Laundry building;
- Heating plant; and
- Septic system.

There is no requirement to modify or expand the camp for the Phase 2 Project.

18.4 Water Supply

Based upon the completed water resource study prepared in 2019 by Mongol Us (Water Resource Department) State owned enterprise of the MMET, an official “Possible water use conclusion of Altan Tsagaan Ovoo gold mine operations and processing plant operations” dated on 10 July 2019 was issued.

It is assumed that there is sufficient local water supply to provide the Project’s water requirements which is outlined in Table 18.1. It is recommended to confirm the availability of water in the future.

Table 18.1 – Water Usage

Area	Total Usage	
	L/s	m³/h
Camp	0.6	2.16
Open Pit	2.65	9.54
Phase 1-Process Plant	7.77	27.97
Phase 2-Process Plant	55.55	199.29
Total	**66.57**	**238.96**

Figures may not add due to rounding

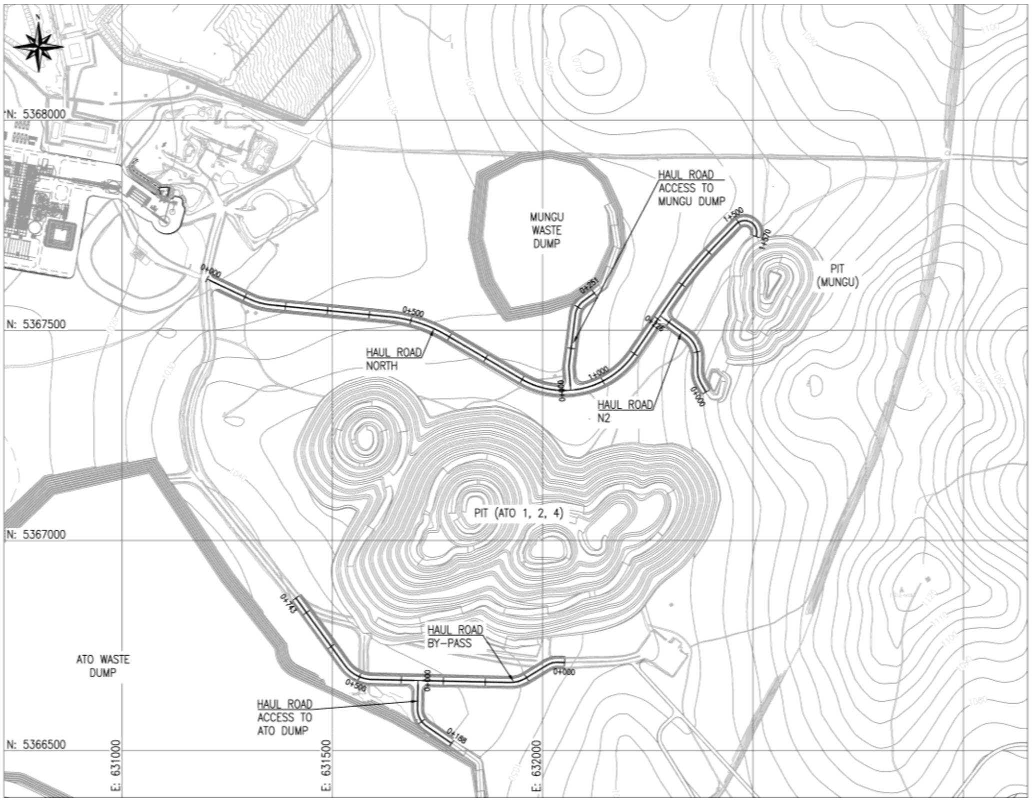
With regard to the process plant in Phase 2, the five (5) water circuits (Raw Water, Potable Water, Fire Water, Gland Water, and Process Water) have been developed to support the requirements of the plant, and in the case of the potable water circuit, the surrounding infrastructure. A daily make-up water amount of approximately 4,800 m^3 will be needed to support the requirements mentioned above and the ratio is 0.79 m^3 of fresh water per tonne of dry ore fed to the plant.

18.5 Roads

The mine access road connects the Project site to Choibalsan City, a distance of 120 km. The road is constructed with gravel as its base and it is assumed to be constructed to carry normal loads able to sustain delivery of materials and equipment and transport outgoing products.

The new process plant site will have internal gravel roads to allow access to the different buildings. Approximately 3 km of new gravel haul roads will connect the new pits with the existing pits, and a new 6 km long gravel road will provide access around the TSF, as depicted in Figures 18.6 and 18.7.

Figure 18.6 – Mine Access and Haul Roads



Source: DRA 2021

Figure 18.7 – TSF – Access Roads

Source: DRA 2021

18.6 Tailings Storage Facility (TSF)

18.6.1 Design Summary

The TSF will be located in a south-east facing valley approximately 2 km south-east of the pit and is shown in Figure 1.4 - General Overall Site Plan

The TSF is designed with an initial starter cell capacity of 3.6 Mt (first 18 months) and a final capacity of 14.8 Mt. Subsequent to Stage 1, the TSF will be constructed in annual raises to suit storage requirements. However, this may be adjusted to biennial raises to suit mine scheduling during operation.

18.6.2 Dam Break Assessment

A dam break assessment was conducted for the TSF main embankment based on a hypothetical failure flowing to the east. A runout assessment was conducted over site topography taking into account the slope and gradient of the downstream area and the liquefied tailings slurry profile. The area downstream is defined by a confined valley for approximately 2 km before opening up to a wide open plain. The initial level assessment indicated the inundation extents could extend past the local valley and report off lease. There is the potential to inundate nomadic settlements however no other infrastructure or population centres were identified.

The TSF is designed to ANCOLD guidelines (refs. 1, 2 and 3) and the Global Industry Standard on Tailings Management (GISTM) (Ref, 4). An ANCOLD Dam Failure Consequence Category of 'High C' was determined on the basis of a potential PAR (Population at Risk) in the range of '≥1 to <10' and a Severity Level of 'Major'. An ANCOLD Environmental Spill Consequence Category of 'Low' was determined on the basis of a potential PAR being '<1' and a severity level of 'Medium' in the event of a spillway discharge.

A GISTM Dam Failure Consequence Classification of 'Significant' was determined from the PAR and environmental, health, social, cultural, infrastructure and economic losses.

18.6.3 TSF Design

The TSF will be a high-density polyethylene (HDPE) - lined cross-valley storage facility formed by multi-zoned earth fill embankment, encompassing a total footprint area (including basin area) of approximately 47 ha for Stage 1, and increasing to 112 ha for the final TSF. Downstream raise construction methods will be utilised for all TSF embankment lifts. The TSF embankment construction materials will be principally sourced from local borrow material (assuming that it is available and sufficient, see Section 18.2) within the basin area and mine waste. A typical section through the embankment is presented in Figure 18.8.

Figure 18.8 – Typical TSF Sections

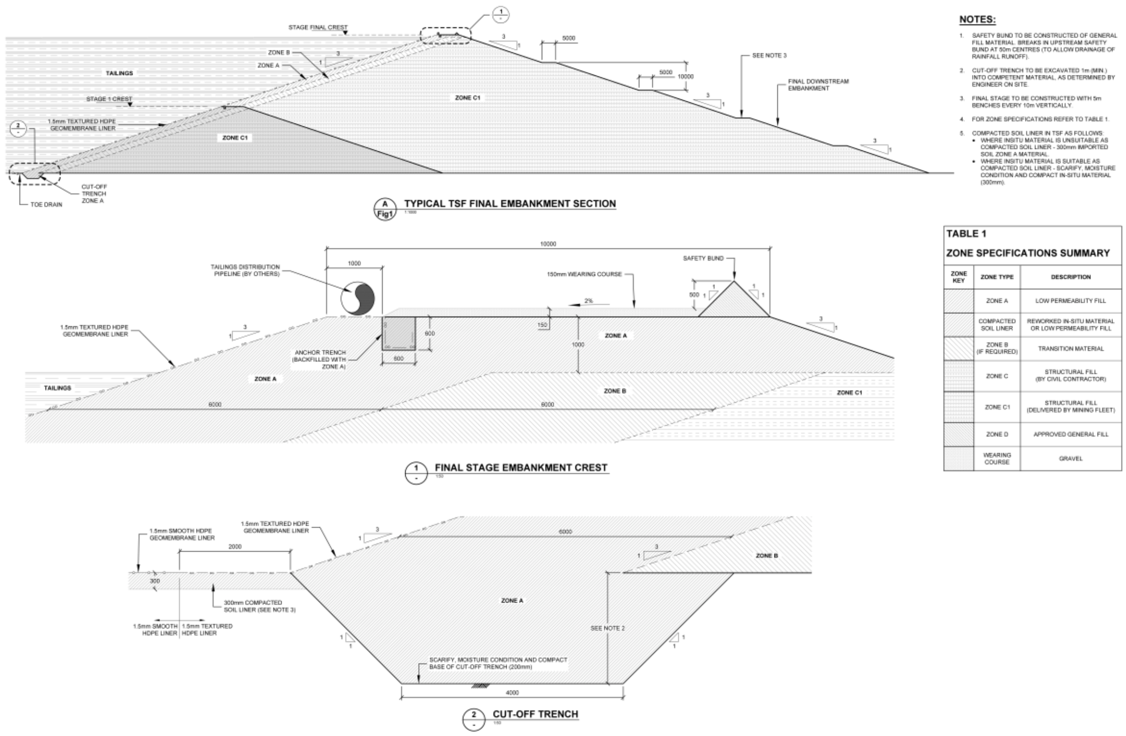


TABLE 1

ZONE SPECIFICATIONS SUMMARY

ZONE KEY	ZONE TYPE	DESCRIPTION
	ZONE A	LOW PERMEABILITY FILL
	COMPACTED SOIL LINER	REWORKED IN-SITU MATERIAL OR LOW PERMEABILITY FILL
	ZONE B (IF REQUIRED)	TRANSITION MATERIAL
	ZONE C	STRUCTURAL FILL (BY CIVIL CONTRACTOR)
	ZONE C1	STRUCTURAL FILL (DELIVERED BY MINING FLEET)
	ZONE D	APPROVED GENERAL FILL
	WEARING COURSE	GRAVEL

Source: Knight Piésold Consulting, 2021

Underdrainage will be installed above the HDPE liner and a leakage collection and recovery system (LCRS) will be installed beneath the basin composite liner to reduce water pressure build-up on the HDPE liner. Solution recovered from the underdrainage system and LCRS will be returned to the TSF via submersible pumps.

Supernatant water will be removed from the TSF via submersible pumps located within a series of decant towers located in the centre of the valley within the TSF basin. Solution recovered from the decant system will be pumped back to the plant for re-use in the process circuit.

The TSF design criteria are summarised in Table 18.2.

Table 18.2 – TSF Design Criteria and Specifications

Design Criteria	Specification
TSF Consequence Category	High C
Dam Spill Consequence Category	Low
TSF Stormwater Storage Capacity	Average supernatant pond superimposed with greater of: 100-year ARI, 72 hr flood and 5-year ARI wet season runoff (assuming 100% runoff and no evaporation).
TSF Emergency Spillway: Spillway capacity Erosion protection	 PMF (probable maximum flood) / critical duration 100-year ARI / critical duration
TSF closure spillway: Spillway capacity Erosion protection	 PMF / critical duration 100-year ARI / critical duration
Contingency Freeboard Wave run-up Additional freeboard	 Nil Nil
Earthquake Loading Operating Final	 Operating Basis Earthquake (OBE) 1:475 AEP Safety Evaluation Earthquake (SEE) 1:2,000 AEP
Operating criteria	
Capacity Final Starter	 14.82 Mt of dry tails 3.6 Mt of dry tails (18 months initial capacity)
Production Rate	Approximately 2.1 MTPA (95% of ore processed)
Slurry Characteristics Percent solids Beach slope Density Stage 1 Final	 50% 120H:1V 1.20 t/m^3 1.35 t/m^3

18.6.4 SITE CONDITIONS

18.6.4.1 Climate

Average annual precipitation is approximately 227 mm with approximately 75% of this being rainfall occurring between June and August (summer months). Average temperatures are below freezing for five (5) months of the year (November to March).

18.6.4.2 Seismicity

Central Asia is characterised by large intracontinental basins that include the Tarim and Junggar basins, separated by intracontinental orogens such as the Tien Shan and north of it the Altai Sayan, Mongolian Altai, Gobi-Altai, and intracontinental rift zones such as the Baikal rift system. Current active tectonic deformation in Mongolia results from the far afield effects of the India-Eurasia collision zone.

This active deformation in Western and Central Mongolia leads to high seismicity throughout the region and the occurrence of strong earthquakes.

A design earthquake event of magnitude 6.5 was determined for the maximum credible earthquake (MCE), occurring at a distance of 50 km from the site, and at a depth of 15 km and with a peak ground acceleration of 0.11g.

18.6.4.3 Ground Conditions

The geology of the area comprises fractured and weathered sedimentary rocks (sandstone, siltstone and mudstone) of the Upper Jurassic-Lower Cretaceous which is covered by elluvium (residual soil) and diluvium-proluvium (deposited during flood conditions) quaternary soil.

A site investigation specific to the TSF location will be undertaken. The following information is based on investigation of the heap leach pad that was designed and the construction supervised by Knight Piésold and is currently in operation. The heap leach pad is located approximately 3 km to the north-west of the TSF.

78 boreholes were drilled in the vicinity of the heap leach pad. The site investigation logs identify:

- A topsoil layer of between 300 mm and 700 mm thick and averaging 480 mm.
- The diluvium-proluvium (deposited during flood conditions) is a highly variable material that ranges from sandy clay to gravelly and clayey sand and is typically present to depths between approximately 1.5 m and 10 m depth, but were absent in some places.
- Residual soil (elluvium) originating from Quaternary sedimentary rock underlies the topsoil and (where present) diluvium-proluvium. This material is described as poorly graded gravel with clay (GP-GC).

The heap leach pad is located on lower lying and more gently sloping ground than the TSF site, which is located within a distinct valley. It is likely that the thickness of the diluvium-proluvium (deposited during flood conditions) and any hill wash material will be thinner.

Based on laboratory test results, the diluvium-proluvium comprised sand, gravelly and silty/clayey. The fines content was highly variable and was typically low plasticity and on the boundary between silt and clay.

It is expected that the near surface soils will not be sufficiently consistent or low permeability for use as a low permeability liner for the TSF and an HDPE membrane in conjunction with a compacted soil layer has been allowed for in the design.

The boreholes undertaken at the heap leach pad extended to a maximum depth of approximately 20 m and no groundwater was encountered. The depth to groundwater at the TSF site is expected to be at greater than 20 m depth.

18.6.5 Tailings Testing

Physical and geochemical testing of combined tailings samples derived from the different ore bodies was conducted. The tailings stream is composed of pyrite cleaner tailings (12%) and flotation rougher tailings (88%). The sample was a combination of cleaner tailings from the pyrite circuit (around 12% by dry mass) and rougher tailings from the pyrite circuit (around 88% by dry mass).

The tailings tested and can be described as low plasticity sandy silt having a particle size of approximately 80% passing 150 µm sieve. The sample indicated a settled dry density of between 1.0 t/m^3 and 1.2 t/m^3 with air drying increasing the dry density to 1.6 t/m^3. Supernatant water release when tested at 50% w/w solids ranged from 29 to 36% of water in slurry before accounting for rainfall or evaporation but incorporating the loss of water to re-saturate lower waste fines layers. Assuming that the facility is efficiently operated, it is estimated that the average settled density for the tailings mass would be approximately 1.40 t/m^3.

The TSF incorporates a low permeability HDPE liner to prevent the loss of tailings water and seepage from the facility based on the expected tailings geochemistry.

18.6.6 Tailings Deposition

A tailings delivery pipeline will run from the plant site to the TSF in a low permeability trench. The deposition of tailings into the TSF cells will be sub-aerial from the perimeter of the TSF.

Tailings will be discharged into the TSF by sub-aerial deposition methods, using a combination of spigots at regularly spaced intervals from the TSF embankment and other specified locations, to locate the supernatant pond at the decant towers. The supernatant pond will gradually migrate up the valley to the north-west during operation towards the upstream end of the facility and distant from the embankment.

The deposition sequence will encourage the formation of beaches over which the slurry will flow in a laminar non-turbulent manner. The solids will settle as deposition continues and water will be released to form a thin film on the surface of the tailings. This water will flow to the supernatant pond from where it will be removed from the facility by means of decant pumps. Tailings deposition will then be moved to an adjacent part of the storage to allow the deposited layer to dry and consolidate.

After deposition in a particular area of beach ceases, and settling of the tailings has been completed, further de-watering will take place through evaporation with some assistance from drainage into the drainage system. As water evaporates and the moisture content drops the volume of tailings will reduce to maintain a condition of full saturation within the tailings. This process will continue until interaction between the tailings particles hinders volume reduction.

18.6.7 MONITORING

A monitoring program for the TSF will be developed to monitor for any potential problems which may arise during operations. The monitoring will include:

- Monitoring bores and surface water sampling stations downstream of the TSF.
- Standpipes piezometers in the TSF embankment.
- Settlement pins to check embankment movement.

The piezometers and monitoring bores will be checked water levels and water quality.

If the monitoring program indicates that potential problems are developing, an increase in monitoring frequency will be implemented and a response plan developed.

18.6.7.1 Monitoring Installations

Groundwater monitoring stations (one shallow and one deep) will be installed downstream of the TSF to facilitate early detection of changes in groundwater level and/or quality, both during the operating life and following decommissioning.

Standpipe piezometers will be installed in the TSF embankment to monitor pore water pressures at several locations within the embankment to inform stability assessments.

Survey pins will be installed at regular intervals along the TSF embankment crest in order to monitor embankment movements and assess effects of any such movement on the embankment.

18.6.7.2 Monitoring Program

As part of the operation of the TSF, extensive monitoring of all aspects of the operation should be undertaken. This monitoring falls into three basic categories:

- **Short-term operation monitoring** – includes items such as offtake location (whether pipe joints are leaking, etc.), which are part of ensuring that the TSF is operating smoothly.
- **Compliance monitoring** – includes items such as checking settlement pins for movement and monitoring bores for contamination, etc., which are used to ensure that the project is meeting all of its commitments in regard to a safe, secure operation.

- **Long-term performance monitoring** – includes such items as tailings level surveys and tailings and water flow measurements (using flow meters installed at designated locations), etc., which are used to monitor long term performance of the facility and refine future embankment lift levels and final tailings (and low permeability soil liner) extents.

In addition, the TSF will undergo annual audits by a suitably qualified geotechnical engineer to ensure that the facilities are operating in a safe and efficient manner.

18.6.8 WATER MANAGEMENT

The management of water is a critical aspect of the design. In order to understand the flow of water around the site, a water balance model was developed. The water balance modelling includes the TSF, process plant and borefield water supply with a view to determining site water storage requirements. Modelling includes for design wet conditions to ensure that the TSF is designed with sufficient storage capacities to comply with design criteria.

Key findings from the water balance modelling are as follows:

- TSF is designed to hold tailings plus design precipitation, and thus has sufficient storm water storage capacity for all design storm events and rainfall sequences.
- Supernatant pond volume peaks at the end of winter as ice melts and during periods of summer rainfall, before returning to the minimum operating pond volume during subsequent periods.
- During winter months water in the decant pond may need to be agitated at the decant to reduce freezing.
- Decant return / process water shortfall is expected to occur under average and design dry climatic conditions.
- All make-up water requirements will be provided by groundwater abstraction from the borefield.

18.6.9 SURFACE WATER MANAGEMENT

The site experiences low rainfall. The TSF is located in a single catchment and any sediment arising from disturbed areas in the vicinity of the TSF will be directed to a sediment control structure located downstream of the embankment toe of the TSF. The sediment control structure will comprise a low height water dam which reduces flow velocities facilitating sediment settling.

18.6.10 CLOSURE SUMMARY

At the end of the TSF operation, the downstream faces of the embankment will have a slope of 3H:1V, with 5 m wide benches located at 10 m height intervals, for an overall slope profile of approximately 3.5H:1V. The profile will be inherently stable under both normal and seismic loading conditions and will provide a stable surface water drainage system and will allow for revegetation.

Rehabilitation of the tailings surface will commence upon termination of deposition into the TSF. The closure spillway will be constructed at the upstream end of the facility in such a manner as to allow rainfall runoff from the surface of the rehabilitated TSF to discharge via the closure spillway.

The closure cap to the TSF is planned to comprise a 500 mm layer of mine waste to form a capillary break, 300 mm compacted low permeability layer to limit water ingress and a 200 mm topsoil layer to promote surface revegetation.

18.7 Electrical Installation

18.7.1 General

The design of the electrical installation is based on the IEC standards related to the equipment and material specific for the Project.

The electrical installation and the major electrical equipment for distribution (switchgears, transformers) is designed without redundancy.

18.7.2 Power Demand

The power demand of the Steppe Gold site was assumed as peak load of 15 MW and average load of 12.5 MW.

18.7.3 Power Plant

Euro Khan and Wood were appointed by Steppe Gold to provide a technical evaluation for the potential development of a hybrid Renewable Energy System (RES) to power the Steppe Gold mine.

In January 2021, Euro Khan and Wood provided the document "Power Optimisation Study for Steppe Gold Mine", Technical Report, Document Ref: SAL_EKRFQ_281119_01_ Rev03.

The solutions proposed in the Study were as follows:

- wind turbine generators;
- solar photovoltaic (PV) modules completed with Battery Energy Storage System (BESS);
- diesel engine generator sets;
- coal power plant.

Based on the modelling work and analysis, a hybrid solution Diesel-RES power plant (30 MW solar PV, 20 MW diesel, 4 MW/4 MWh BESS) was demonstrated to be the optimal low-cost solution for the Project.

Key factors that make this system the preferred choice are as follows:

- short development time and lead time for EPC of diesel and solar plant;
- diesel fuel availability, transportability, and low cost;
- strength of solar irradiation resource;
- significantly reduced operating cost through inclusion of RES.

The site will be supplied at 11 kV, 3 phase, 50 Hz from a power plant installed in the vicinity of the site. The power plant will consist of:

- Eight (8) diesel generators (DGs) each using LFO (diesel) fuel, 2500 kWe @ 0.8 PF, 11 kV, 3 Ph;
- Individual control panel, local monitoring, battery and charger, silencer;
- Individual walk-in enclosure complete with fuel tanks and transfer pumps;
- One 11 kV switchgear, 1250 A, 20 kA, 3 sec, equipped with 8 incoming breakers and 6 output feeders and with protection and synchronizing relays;
- One auxiliary transformer 11 kV / 415 V;
- One prefabricated Electrical Room.

The operating mode for the diesel generators will be N+2 (6 in operation, one in stand-by, and one under maintenance or repair). It is probable that the diesel generators could be purchased from China, which offers a lower cost than other options.

18.7.4 Plant Reticulation Network

The reticulation network consists of an 11 kV buried cable network and 11 kV pole lines.

The 11 kV buried cable network starts from the Power Plant 11 kV MV Switchgear output feeders to the step-down transformer 11 kV – 6 kV installed in the Power Plant yard and to transformers 11 kV – 0.4 kV installed in the vicinity of the plant Electrical Room ER-200.

The 11 kV pole line supplies also the Crusher Electrical Room ER-100.

Electrical supply was not requested for the mine site.

18.7.5 MV and LV Distribution Levels, Systems Earthing and Load Ranges

The proposed distribution voltage levels for equipment and the type of motors are defined as indicated in the follow table:

Table 18.3 – Design Voltage Levels

Voltage	Earthing	Loads
11 kV, 3Φ, 3W	IT	MV Plant Distribution
6 kV, 3Φ, 3W	IT	Very large loads - mills
415 V, 3Φ, 4W	TT	Fixed speed and variable speed motors 415V Non process loads larger than 6 kW
380/220 V, 3Φ, 4W	TT	Large HVAC Lighting in Process Area Welding receptacles
380/220 V, 3Φ, 4W or 220 V, 1Φ	TT	Small motors 220 V Lighting in Buildings and Small HVAC Small loads up to 6 kW

18.7.6 MAIN ELECTRICAL EQUIPMENT

The main electrical equipment is related to the process areas and is installed in Electrical Rooms.

18.7.6.1 Crushing and Conveying

The crushing system, purchased by Steppe Gold, is currently under construction, and for the purpose of this Project is considered to be existing.

It is assumed that the crushing area will be supplied from the Power Plant by a dedicated 11 kV feeder. Electrical equipment will be installed in an Electrical Room, with a LV MCC equipped with starters and VFDs to control equipment in this area, and auxiliary transformers (lighting, services) and Control Panels for control and instrumentation. Step-down transformers are installed outside of the electrical room.

18.7.6.2 Process Plant

The main areas in the process plant are:

- 5200 Grinding / Milling / Classification;
- 5300 Rougher Flotation & Regrind;
- 5400 Cleaner Flotation;
- 5500 Tailings Thickening & Concentrate Handling;
- 5700 ADR Plant;
- 5800 Reagent Preparation (existing) & Grinding Media;
- 5900 Process Plant Services & Utilities;

Electrical equipment will be installed in ER-200, adjacent to the process plant building. The following equipment will be installed inside ER-200:

- 11 kV switchgears 7900-MVSW-12, -13: 11 kV, 1000 A, 20 kA @ 3 s, 50 Hz
- 6 kV MCC, 5200-MVMCC-21: 6 kV, 1000 A, 20 kA @ 3s, 50 Hz (this MCC controls the grinding and regrinding mills).
- A step-down 11 kV – 6 kV, 10 MVA transformer supplying the 5200-MVMCC-21 will be installed in the Power Plant yard, in the vicinity of ER-200.
- Step-down distribution transformers complete with Load Break Switch: 7900-XTR-31, -32, -33: 2000 kVA, 11 kV – 415 V, AN.
- Auxiliary transformers complete with Load Break Switch: 7900-XTR-41, -42: 800 kVA, 11 kV – 415 V, AN.
- 415 V switchgears 7900-LVSW-31, -32, -33: 415 V, 3200 A, 50 kA, 50 Hz.
- 415 V LV-MCC equipped with starters and VFDs to control all process plant area equipment.

For Areas 5300, 5400, and 5500, MCC loads are split by concentrate type (Pb, Zn, pyrite).

18.7.7 Electrical Equipment Specification

The characteristics of major electrical equipment are based on design criteria from information received from suppliers.

The 10 MVA, 11 kV – 6 kV mill transformer will be outdoor, liquid fill type. Winding connections will be delta (primary) and wye (secondary).

The 2 MVA, 11 kV – 415 V distribution transformers will be indoor, dry type. Winding connections will be delta (primary) and wye (secondary).

The MV Switchgear 11 kV, 1600 A, 20 kA / 3, type IAC-AFLR, LSC-2B will consist of a line-up of withdrawable type; the circuit breakers electrically will be operated at 220 VAC supplied from an external UPS. The unit will have metering, monitoring and protective relays (Arc Flash Detection).

The mill MV MCC 6 kV, 1600 A, 20 kA / 3, type IAC-AFLR, LSC-2B will consist of a line-up of withdrawable type; contractors (DOL starters) will be operated from a 220 VAC. The unit will have metering, monitoring and protective relays (Arc Flash Detection).

The 415 V LV switchgears will be metal-enclosed type with draw-out circuit breakers, rated for 3200A, 50 kA.

LV MCCs will be freestanding, factory assembled and wired, single faced or back-to-back mounted. LV MCCs will have draw-out type feeder units housing circuit breakers and rated for 1600A, 50 kA.

Each starter will be equipped with microprocessor-based relays ("intelligent relay") provided with Modbus TCP/IP communication.

LV VFDs will be installed separately or within the MCC. The VFDs will have input filters to limit the total harmonic distortion and will be 6 pulse type.

The settings of the protective devices for the 11 kV MV switchgear incoming breakers will be fully coordinated with the utility system protection.

In general, protective relays shall be micro-processor based, multi-function type complete with built-in Modbus TCP/IP communication.

All protection relays of a switchgear or MCC will have at least one port. Relays will be compatible with IEC 61850 protocol and will communicate over a Modbus TCP/IP network.

For each switchgear or MV-MCC line-up, the supplier will supply, install and power the required number of switches. The supplier will provide and install a Modbus TCP/IP cable between measuring/protective equipment and the switch.

Protective relays and lockout relays will have external test switches to allow for functional testing of protective relaying and their associated circuits.

18.7.8 Earthing

The earthing system for 11 kV and 6 kV is IT.

The earthing system, consisting of a grid of copper conductors, will be provided for each process and ER building. Earth conductors will run externally around each building with taps thermo-welded to every other column. Individual earth grids will be tied together with interconnecting earth cables.

All major electrical equipment such as transformers, switchgears, large motors, motor controllers, cable tray systems, water and fuel tanks, substation fencing, etc. will be individually connected to the earth network from two points.

The earthing system will be designed to limit overall resistance to earth to four ohms (4 Ω) or less.

A separate earth bus in electrical rooms and/or control room will be dedicated to instrumentation cables and equipment earthing. This earth bus will be connected to an isolated earthing system and insulated from the main plant earth. An insulated green earth wire will run to the instrumentation equipment earth studs to ensure instrument earthing system integrity. The instrument earth bus will be connected to the main plant earthing system.

18.7.9 Lightning Protection

An appropriate level of lightning protection will be installed to protect property, personnel and equipment. Subject to results of an evaluation, the complexity of design will depend on the severity or level of incidence of lightning strikes in the area of the site as well as the type of plant and risks in the event of lightning strikes.

Conventional lightning protection uses passive air terminal rods installed at the highest points of the stacks or roof with copper lightning conductors of natural air-cooled braid design or copper strips at the exterior of the buildings directly to dedicated earth rods. These earth rods are connected underground to the main earthing loop.

The lightning protection for the DG Power Plant is provided by lightning masts installed around the DG Power Plant yard.

18.7.10 CABLES AND CABLE TRAYS

The power cables will consist of single conductor or three conductors, copper, XLPE-insulated, with aluminium or steel armour and PVC sheath, rated to 90°C.

Cable trays will be ladder type, galvanised steel. Cable trays for instrument cables will have a separated section. Separate trays will be provided for cables of different voltage ratings, or if installed in the same tray, separating barriers will be provided.

18.7.11 LIGHTING AND SMALL POWER

The necessary illumination levels will be provided for all areas. The following types of lighting fixtures will be used in each area:

- Process areas with high headroom (3 m+) will be lit by metal halide industrial (or LED equivalent) high or low bay lighting fixtures with integral ballast. Other internal areas of the plant (process areas less than 3 m high, offices, electrical and control rooms, etc.) will be lit by energy saving fluorescent lamps.
- Outdoor areas (process yards, roads, parking, etc.) will be lit by high-pressure sodium (or LED equivalent) roadway lighting fixtures and floodlights installed on steel poles.

Emergency Lighting will be provided as follows:

- Process working areas, control and electrical rooms, etc. will be fitted with rapid restarting fixtures to provide partial or full illumination after voltage dips or normal power failure.
- To permit movement of personnel during a power failure or emergency situation, all areas will be fitted with individual battery pack units located near passages, stairwells and exits. Exit lights will have built-in batteries and energy efficient lights; the modules will be located near the exits.
- The lighting system and receptacle power will be fed by 220/380 V dry type transformers and panel boards located in electrical rooms.

- Lighting in process and production areas will be switched from panel boards. Outdoor lighting will be controlled by photocells or timers.
- Welding/power outlets will be installed at appropriate locations for supplying power to portable welders and similar loads.

18.8 Control System

18.8.1 Automation Process Network

The Process Control System ("**PCS**") will be based on an Ethernet Modbus TCP/IP backbone network in a ring type topology. The network links all the main automation equipment, such as Supervisory Control and Data Acquisition ("**SCADA**") system, Historian, Human Machine Interface ("**HMI**") and PCS processor.

The network includes fibre optic linking of the following main areas of the plant:

- Central Control Room;
- Electrical Room for DG Power Plant;
- Electrical Rooms (ER-100 for the existing Crushing system and ER-200 for Process Plant);
- Fine Ore Stockpile Area;
- Grinding and Regrinding Mills Area;
- Rougher Flotation Area;
- Cleaner Flotation and Filter Press & Concentrate Thickener Area;
- Tailings Thickener Area;
- Reclaim Process Water Pond and Pumps Area;
- Raw Water, Potable Water and Fire Water Pump House Area;
- Soda Ash and Cyanide Area;
- ADR Plant;
- Fuel Station;

Network automation communication services are:

- SCADA stations located in the Central Control Room and HMI screens located in the process and associated areas;
- PCS processors inter-communication;
- PCS/Remote Input/Output ("I/O") communication;
- PCS direct interface to the Motor Control Centres (MCCs);
- IEC61850 interface to the power distribution equipment;

- Field device communication including communication with 3rd party PLCs supplied with mechanical equipment;
- Process camera system installed in the plant for process control viewing purposes;
- Security camera system installed in the plant for security viewing purposes.

18.8.2 PROCESS CONTROL SYSTEM (“**PCS**”)

A PCS system will be supplied to control each strategic areas of the process plant with remote I/O racks located in the main process plant areas.

The PCS will be of PLC type.

The PCS hierarchy will be designed in accordance with the concept known as Computer Integrated Manufacturing (CIM) and will be designed as a multi-level integrated structure.

This multi-level structure contains the following three (3) levels which concur with the levels described as the International Standards Organisation (ISO) Open Systems Interconnection (OSI), model for degree of integration.

- Level 0 - Field Equipment and Instruments;
- Level 1 - Process Control and Measurements;
- Level 2 - Process Control and Supervision.

The major equipment like crushers, mills, filter press and thickeners could come with their own PLC and with a Local Control Panel.

The central SCADA system has the capacity to control and supervise all the remote PCS equipment. In a communication outage situation, the critical equipment will be controlled locally.

18.8.3 PCS CONTROL PHILOSOPHY

The general control philosophy for the Project will be one with a high level of automation and central control facilities to allow critical process functions to be carried out with minimal operator intervention. Instrumentation will be provided within the plant to measure and control key process parameters.

The architecture system is designed in such a way that it is fully automated.

A Local Control Station will be installed for each motor. Each motor starter shall have remote control and local control.

The MCC starters (6 kV and 415 V) will be equipped with microprocessor-based relay ("intelligent relay") provided with Modbus TCP/IP communication.

The VFDs will be equipped with Modbus TCP/IP communication card.

18.8.4 SCADA

The SCADA system will be based on client/server technology and will include:

- two (2) SCADA servers for redundancy;
- one (1) historian server;
- two (2) HMI operator stations;
- one (1) engineering station.

The system will be installed in the Central Control Room.

18.8.5 SCADA and PLC Power Sources

In case of plant power outage, the PCS, switches, main servers, phone system, and security systems will be fed by Uninterruptible Power Supply ("UPS"). UPS status will be monitored.

18.8.6 Redundancy

For the automation network, the redundant ring topology design insures a second route in case of a communication outage on one (1) segment.

18.8.7 Process Analog Instruments

The supply is at 24 Vdc and 4-20 mA loop cabling with enabled HART protocol will be utilized.

18.8.8 Wiring and Junction Boxes

All the field instruments and switches will be wired to the PCS through junction boxes up to remote I/O racks situated in the various areas of process plant.

The wiring system will include field junction boxes for instrument power supply, for digital signals and for analog signals.

The motor temperature input will be wired directly to the related motor protection relays while equipment RTD signals will be connected directly to the PCS remote I/Os.

The junction boxes will be located and installed in all process areas of the plant. The junction boxes will be wired to the PCS I/O racks via multi-conductor cable.

18.9 Communication System (Local and External)

18.9.1 Telecommunication Local System

The telecommunication system will be based on Ethernet links throughout the plant buildings and administrative buildings following generally the electrical reticulation network (buried and/or installed on the pole lines).

Single-mode fiber optic backbone will be deployed through the plant to accommodate both automation and corporate services on the same fiber cable on different fiber.

18.9.2 Telecommunication and Mobile Radio Systems

The telecom service includes the tower located in a high elevation zone of the plant; it will be supplied by a third-party provider and will communicate with the plant communication interface.

The telecommunication systems will include:

- IP Phones;
- Process Camera System;
- Security Camera System;
- Fire Detection System;
- Access Control System (gate, door);
- Mobile Radio System.

The mobile radio system will be provided for the construction phase and the operation of the mine and plant site.

18.9.3 Telecommunication Services

The site will be connected to a local Internet Service Provider ("**ISP**"). A backup system will use a cellular modem or satellite technology. The IP phone system will be connected to an Internet Telephone Service Provider ("**ITSP**").

18.9.4 Telecommunications Distribution

During the construction phase, all communication services, such as Internet and phone, will be distributed via Wi-Fi, Wimax and Microwave point-to-point radios to reach all areas of the plant site.

All mine trucks and pick-ups will be equipped with a Wimax/Wi-Fi antenna that shall also act as a Wi-Fi local access point.

The telecommunication distribution will be through the plant fiber optic network covering the processing plant area, Administration Office, Camp and Cafeteria.

If necessary, wireless communication will be provided for the other auxiliary outside of the plant.

18.9.5 Corporate Network

The automation Ethernet backbone network, in a ring type topology described in the previous section will be used for the automation, the camera and security video, the IP phone system and the corporate network applications.

All the major network equipment will be located in dedicated server rooms located in the administrative office, the telecom shelter, the control room and electrical rooms.

Corporate services will include:

- Wired/Wireless Phones and System Servers;
- Process and Security Camera System;
- Access Control System (gate, door);
- Fire Detection.

18.9.6 Process Camera System

A process camera system, with a recorder and a viewer, will be installed in the control room. Cameras will be installed in the plant for process control purposes.

18.9.7 Security Camera System

A security camera system, with a recorder and a viewer, will be installed in the main gate office. Cameras will also be installed in the plant for security purposes.

19 MARKET STUDIES AND CONTRACTS

19.1 Phases 1 and 2 – Gold and Silver

The ATO Project is an operating site producing a readily saleable commodity in the form of gold doré bars. Doré is sent via secure transportation to a refinery for further refining.

Steppe Gold sells its gold production directly to the Mongolian government at spot price. Two (2) types of doré are produced:

- First doré contains approximately 70% gold by weight and the remaining 30% is a mixture of silver, base metal and iron.
- Second doré is silver produced and sold separately.

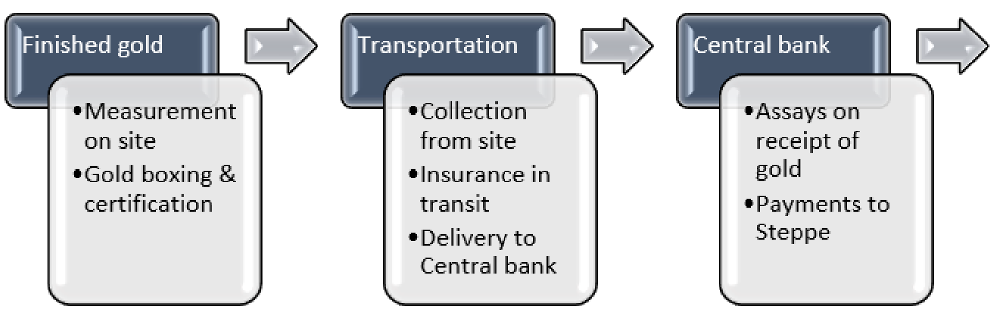
All the doré is transported to the Central Bank of Mongolia (Mongolbank). The Bank of Mongolia announces the official gold and silver rates for the day using the London Metal Exchange (LME) closing rate from the previous day.

The following deductions apply on the LME doré closing date which is the Bank of Mongolia official rate for the day:

- Refining charge of $2 USD per gold oz sold;
- Refining charge of $1 USD per silver oz sold;
- 5% royalty on gross gold revenue; and
- 5% royalty on gross silver revenue (up to $25).

The doré is sent to a refinery for smelting and sampling to determine gold and silver content. Once these assay grades are determined, the proceeds from the sale are credited to the Steppe Gold account. A flowsheet for the receipt of income is shown in Figure 19.1.

Figure 19.1 – Flowsheet for Receipt of Income



19.2 Phase 2 –Lead, Zinc, and Concentrates

This section is largely summarised key information from the Report entitled "Steppe Gold's ATO Phase 2 Expansion Project: Lead and Zinc Market Overviews", dated August 12, 2021, Draft Report, Reference # ST2260-21, prepared by CRU International Ltd (CRU) and presentation entitled "Commodity Quarterly: Zinc Q2 2021", dated July 26, 2021, prepared by S&P Global Market Intelligence Inc (S&P).

Steppe Gold will continue to pursue its effort in exploring zinc, lead as well as selling their concentrates (lead, zinc, and pyrite) for its Phase 2 Project. This section highlights the global market outlook, including demand and supply of key commodities such as lead, zinc, and concentrates that are all products of the ATO Phase 2 Project.

19.2.1 Lead Market

19.2.1.1 Lead Supply and Demand

Lead is used for vehicle and industrial batteries, submarine cables, radiation shielding, paint, petrol (gasoline), solders, and galvanising alloys. It's often used to store corrosive liquids. Like most other minerals, China is the world's top consumer of lead and accounts for more than 40% of the global demand last year, as per CRU data.

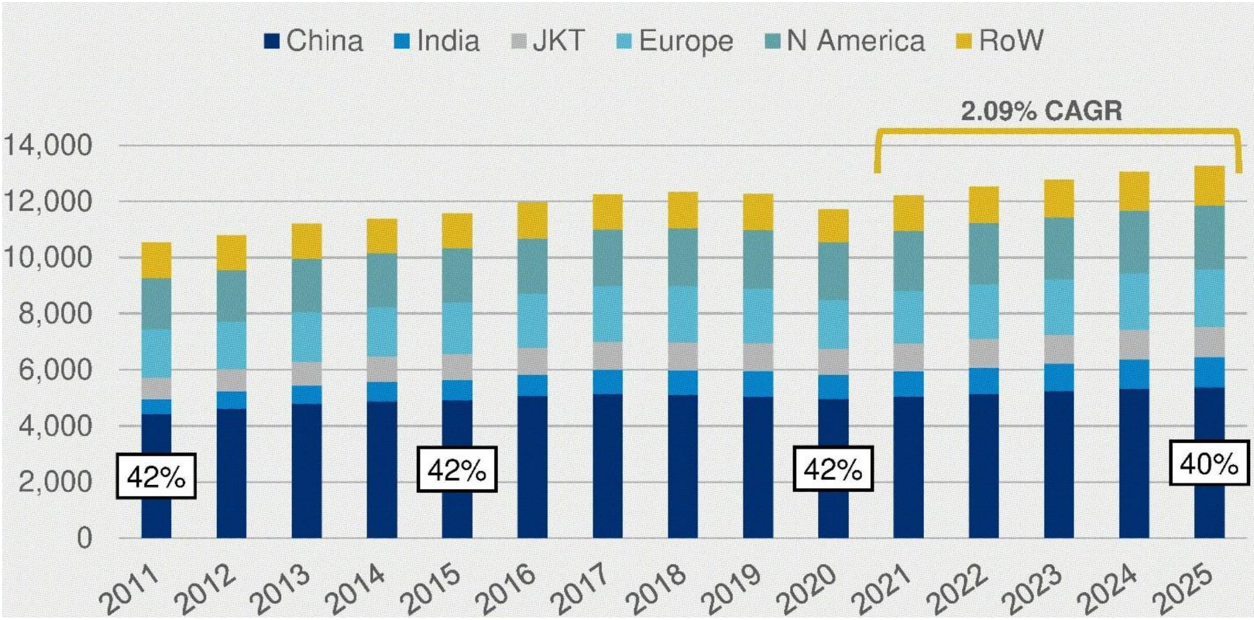
CRU estimates that global lead demand totalled 11.7 Mt in 2020, falling from 12.3 Mt in 2019. Global lead demand was still materially higher than the 10.5 Mt total in 2011, despite vehicle production directly impacting lead used in original equipment (OE) and batteries during the COVID-19 pandemic in 2020.

CRU expects global lead demand to rebound to 12.2 Mt in 2021, up by 4.5% year-on-year, thanks to the recovery in vehicle production and OE automotive battery sales which is set to drive demand back to 2019 levels.

Interestingly, underpinning demand, CRU estimates the battery industry represented 86% of total lead demand in 2020. The majority (54%) of lead consumed in the battery sector was from Starting, Lighting and Ignition (SLI) batteries, which are mostly found in cars and motorcycles. Industrial batteries accounted for an estimated 32% of lead demand, according to CRU.

The research group expects global lead consumption to grow at a compounding average growth rate (CAGR) of 2.09% between 2020 and 2025, reaching 13.3 Mt in 2025. Europe and China are expected to account for about ~50% of growth in global demand by 2025. Thailand, Vietnam, and Indonesia are set to drive lead demand in Southeast Asia, which is forecast to increase from 331 kt in 2020 to 414 kt in 2025.

Figure 19.2 – Global Lead Demand by Region from 2011 to 2025 in kt



Source: CRU 2021

Looking at the supply side, as per CRU data, China – the world's largest producer – accounted for an estimated 46.7% of global lead mine supply in 2020. CRU forecasts that global lead mine production will have a CAGR of 1.54% between 2021 and 2025.

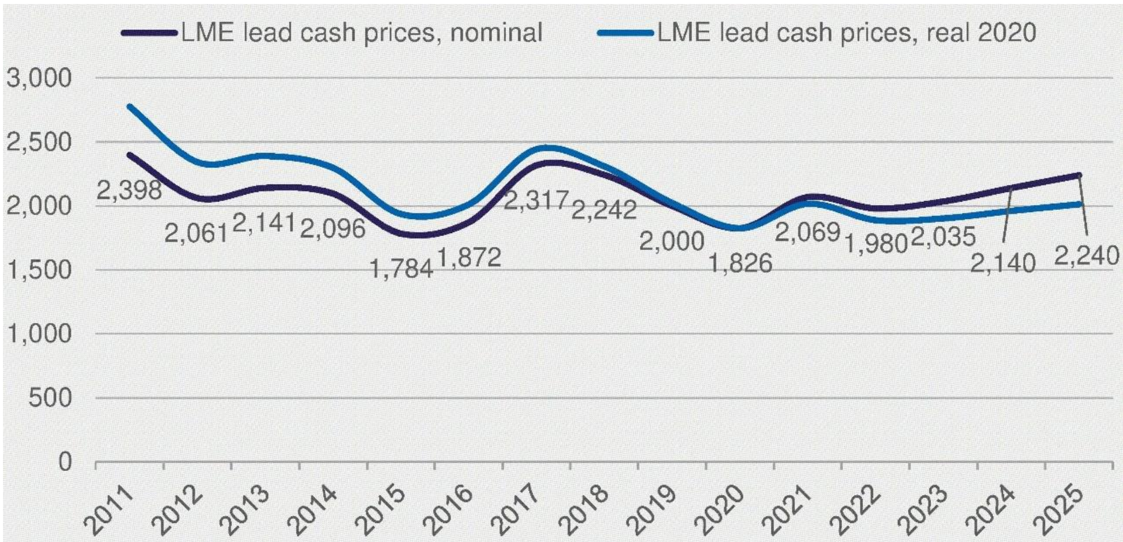
19.2.1.2 Lead Price Forecast

The COVID-19 pandemic caused a large surplus in the global refined lead market, resulting in the average LME lead cash price falling to US $1,826/t in 2020. The average LME lead cash prices are estimated to rise to US $2,069/t in 2021, as per CRU forecasts.

Unfortunately, unlike other 'battery' metals – i.e., lithium, copper, and nickel – in the vehicle electrification story, CRU believes lead will continue to be weighed down in investors' eyes by a lack of a compelling positive narrative in the 2020s. In that way, the research group forecasts a modest LME lead cash prices recovery from US $1,980/t in 2022 to US $2,240/t in 2025.

That said, considering its current and future dominant role in most battery sectors and impressive 'green' recycling record, CRU believes that lead's tarnished image among the investment community is somewhat misplaced. That's why it's forecasting moderate growth over medium term.

Figure 19.3 – LME Lead Cash Prices from 2011 to 2025



Source: CRU 2021 (DATA: LME and CRU).
Note: US$/t, Nominal and Real 2020
Real prices are calculated using the US GDP deflator based to 2020)

19.2.2 ZINC MARKET

At the time of writing of this Report, base metals are trading near multi-year highs following the aftereffects of the 3.6 trillion yuan (~US$500 billion) Chinese government stimulus package in May 2020. Commodity markets have reacted positively, with most metals' prices rising because of increased demand from China and the ongoing supply-chain disruptions following the COVID-19 restrictions put in place by governments world-wide. This combined effect has seen the zinc price rise to above the US$3,000/t mark in September 2021 and trading above 2019-2020 price levels.

The positive sentiment fuelled by the Chinese stimulus package, alongside other flagged global infrastructure stimulus packages (i.e., US), has led to robust zinc fundamentals. With economic activity healthy across Europe and the US (PMIs in expansionary territory), zinc stock levels have declined by roughly 40% since March 2021. This is generally indicative of a tight metal market and corresponds to higher prices. Moreover, the Chinese government has forecast an impressive GDP growth rate of 8.3 to 8.4% for 2021, as it embarks on a massive nationwide infrastructure building program – especially for projects under the country's 14th Five-year plan (2021-2025).

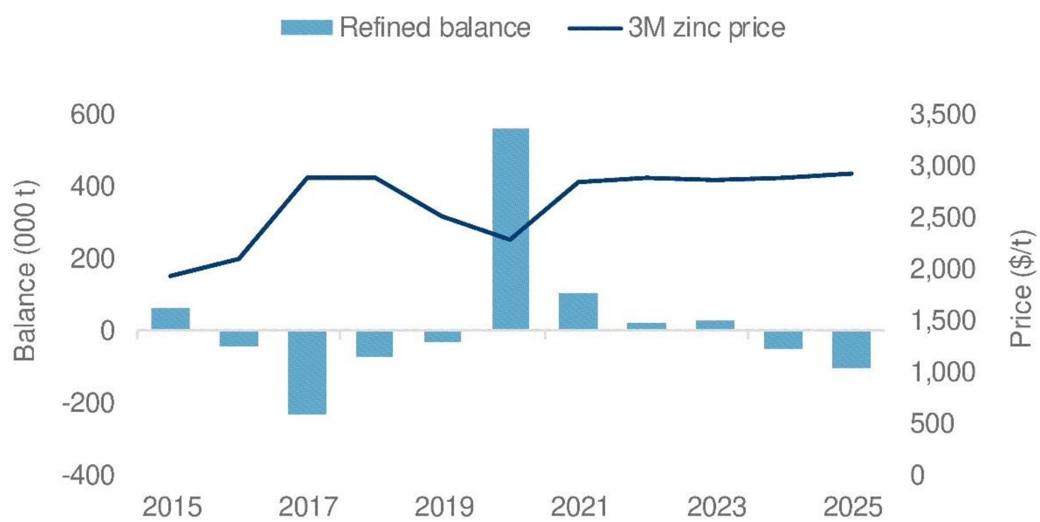
19.2.2.1 Zinc Supply and Demand

Zinc demand is closely correlated with steel production, as it is primarily used in galvanisation. On this basis, S&P Global Market Intelligence (S&P) forecasts global refined zinc demand to grow from ~13.2 Mt in 2020 to ~14.1 Mt in 2021. This is primarily due to Chinese consumption growth driven by the country's stimulus measures, as stated previously. China is expected to account for 51% of global zinc consumption in 2021, according to S&P.

Global mined zinc production is forecast to grow by 5.4 % from ~13.2 Mt in 2020 to ~14.1 Mt in 2021, with zinc mine production rising in key producing regions, according to S&P. The largest project set to enter production in the mid-term is the Nexa Resources' Aripuana asset in Brazil, due to come on stream in 2022.

Looking at the project pipeline for zinc, a sudden wave of new supply looks unlikely. S&P forecasts production from existing mines and probable new projects will peak at ~14.6 Mt in 2025, before falling to ~11 Mt by 2030. The commodity research group is predicting the refined zinc market to remain in slight surplus until 2023, turning to a deficit from 2024, as per Figure 19.4.

Figure 19.4 - Forecast Global Zinc Refined Balance from 2015 to 2024



Source: S&P Market Intelligence 2021

19.2.2.2 Zinc Price Forecast

Zinc prices, traded on the London Metal Exchange (LME), have recovered to above US $3,000/t in August 2021, up 66% from the multi-year lows reached in March 2020. The price expectations for the remainder of 2021 are expected to average of US $2,875/t for the year.

According to S&P, zinc price forecasts are set to average of US $2,885/t in 2022 and $2,858/t in 2023 with a medium-term average price of US $2,935/t in 2025. Figure 19.5 shows that S&P's zinc metal price forecast is above consensus estimates. But some research groups believe that zinc's use in solar power may climb in the years ahead, as the United Nations has delivered another stark warning on climate change. Goldman Sachs is the most bullish observer, forecasting US $ 3,000/t by June 2023. This suggests zinc prices will remain elevated over the next few years.

Figure 19.5 - Global Zinc Price Forecast from 2015 to 2025

Source: S&P Global Market Intelligence, 2021

19.2.3 Lead, Zinc, and Pyrite Concentrate Market

In Steppe Gold's Phase 2 Project, as stated above, lead and zinc metals are prime indicator of lead and zinc concentrates. Steppe Gold will produce and sell its concentrates (lead, zinc, and pyrite) for its Phase 2 Project.

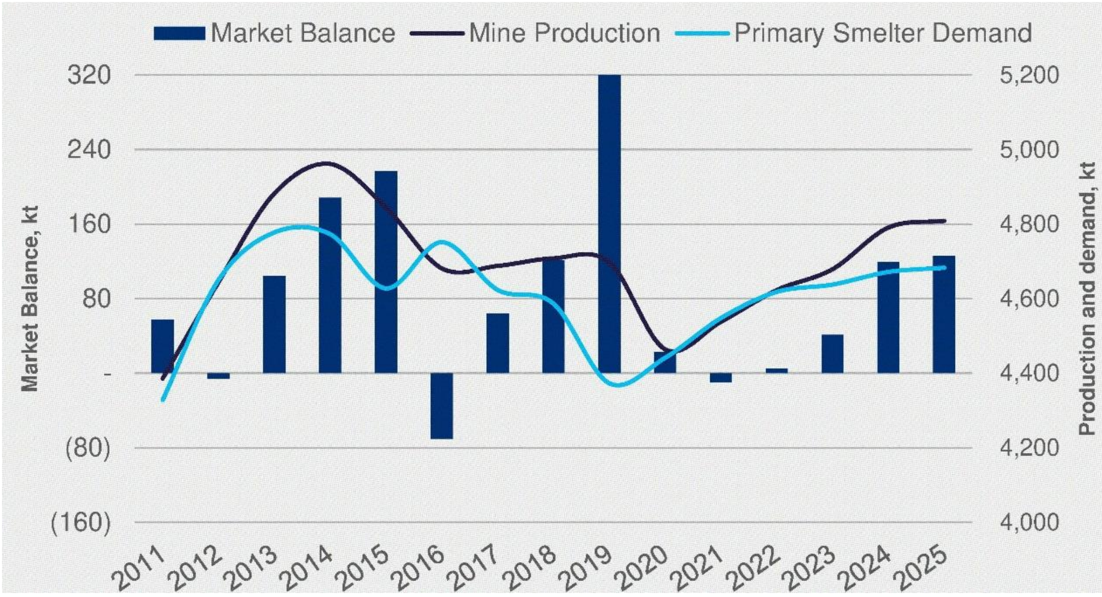
19.2.3.1 Lead Concentrate Market Summary

The lead concentrate market moved from surplus (2011 – 2015) to deficit in 2016 on the back of some production declines. But it was short-lived. The market returned to surplus by 2019, thanks to a sharp reduction in lead concentrate demand and modest growth in mine production. There was a slight surplus in 2020, thanks to the Covid-19 pandemic.

China accounted for 45.4% of global concentrate output in 2020, with ~30% of the demand met via imports. Russia and Peru were the country's two (2) largest suppliers of imported lead concentrates.

Looking ahead, CRU forecasts a slight deficit in the Chinese lead concentrate market in 2021 and 2022. The market is forecast to move back to surplus from 2023-2025, with mine production growth running ahead of slowing smelter demand.

Figure 19.6 – Global Lead Concentrate Market Balance from 2011 to 2025



Note: DATA: ILZSG, CRU. Note: Balance calculated assuming average 97% smelter recoveries
Source: CRU 2021

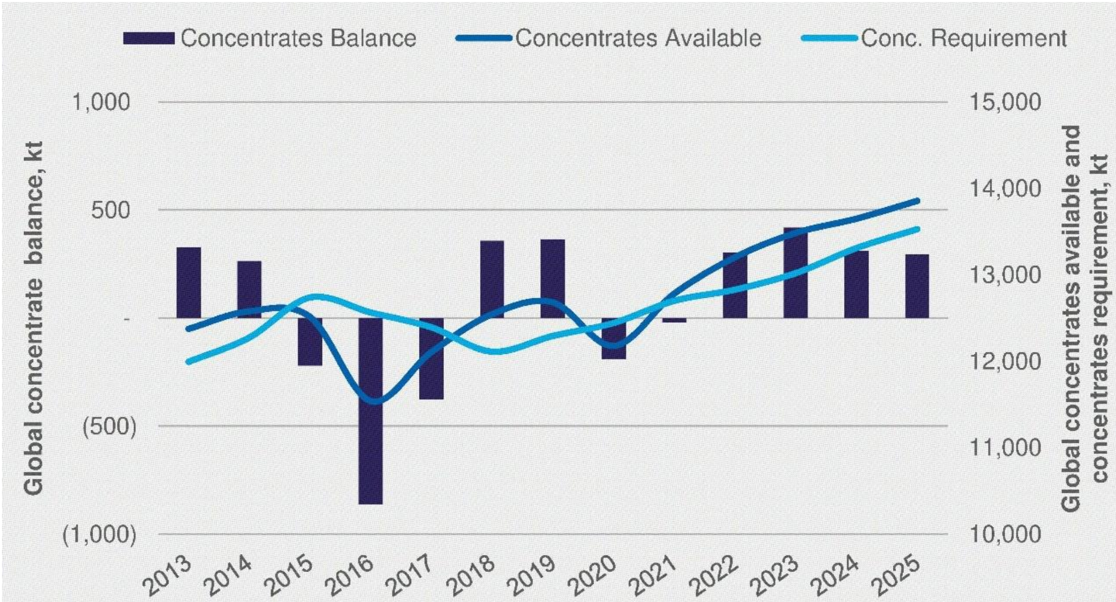
In the medium term, the market will remain moderately balanced according to CRU. Chinese net imports are forecasted to decrease slightly, from 765 kt in 2020 to 675 kt in 2025.

Similar to the lead market, China is both the largest zinc concentrate producer and consumer. Domestic zinc mine output cannot fully satisfy the primary demand from zinc smelters (usually ~90% of total smelter output) and, as a result, approximately 25% of the zinc concentrate requirement is met via imports.

19.2.3.2 Zinc Concentrate Market Summary

CRU forecasts lower concentrate imports in 2021 due to high inventory levels. That said, the research firm expects higher than normal inventory level due to residual concerns over Covid-19 mine disruptions in 2021. From 2022 onwards, considering low demand growth and high refined zinc stocks, CRU forecasts annual surpluses of zinc concentrate into 2025.

Figure 19.7 – Global Zinc Concentrate Market Balance from 2013 to 2025



Note: Concentrates available are mine output after adjustment for Shaimerden.
Source: CRU 2021

Regarding pricing, zinc concentrate is normally bought and sold based on its contained metal value less a processing fee. The contained metal value is determined by reference to some published quotation, such as the refined zinc prices on the LME.

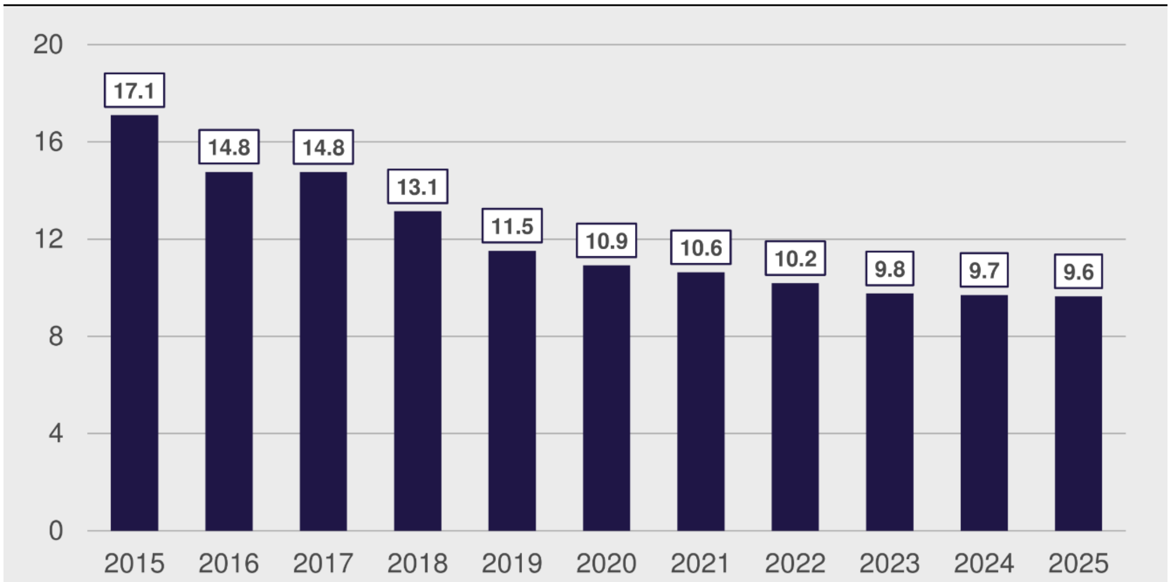
19.2.3.3 Pyrite Concentrate Market Summary

Pyrite concentrate is used in the production of metal and alloys.

China accounts for more than 90% of global pyrite concentrate demand. In 2021, CRU forecast that Chinese demand (production plus net trade) for pyrite concentrates will be 10.6 Mt – 38% lower than the 17.1 Mt total in 2015 as depicted in Figure 19.8. The decline is due to the continued growth in involuntary smelter acid output, which competes against pyrite-based sulphuric acid in China.

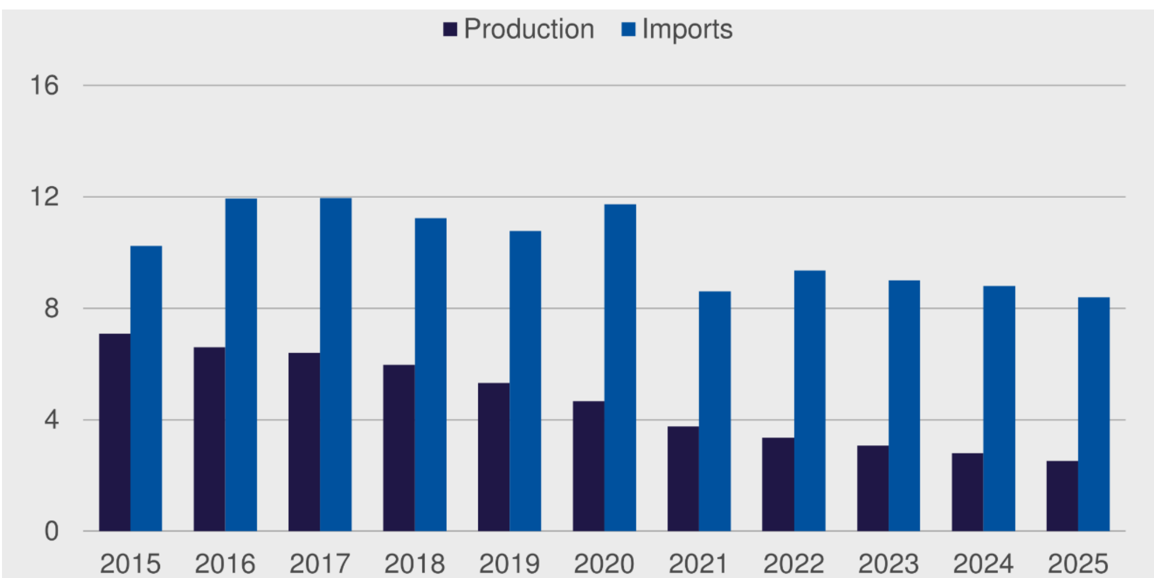
Unfortunately, due to the stricter enforcement of environmental standards in China, CRU estimates that pyrite concentrate demand will decline to 9.6 Mt in 2025.

Figure 19.8 – Demand - Chinese Pyrite Concentrates from 2015 to 2025 (Mt)



Considering the lower demand outlook, CRU forecasts an overall downtrend for both production and imports to 10.9 Mt in total in 2025 from 16.4 Mt in 2020, as shown Figure 19.9. Chinese pyrite concentrate production has seen a sustained decline since 2020.

Figure 19.9 – Chinese Pyrite Concentrate Import and Production Volume from 2015 to 2025



Source: CRU Data 2021

19.3 Comments on Market Studies

Reviewing the opportunity for Steppe Gold, considering the apparent market balance over the coming years, it could service Liaoning, Shandong, Henan and Qinghai as markets of interest. Other provincial markets with deficits, such as Hunan and Yunnan would be too far from Steppe Gold and hence incur high inland transportation costs.

Although Inner Mongolia is the largest lead concentrate supplying province in China, accounting for over 29.1% of Chinese total concentrate output in 2020, Inner Mongolia is also a key port and imports lead concentrates from Russia and Mongolia.

Like the lead concentrate market, Inner Mongolia – China’s largest zinc concentrate producer – is also a key importer. China imports around one quarter of its total zinc concentrates demand. According to available trade data in 2019 and 2020, Guangxi, Shanghai and Inner Mongolia were the three (3) main regions to receive import zinc concentrates.

This data suggests that the Liaoning and Shaanxi in North Central, Henan in East Central, should be targeted as potential markets for Steppe Gold’s zinc concentrate. Hunan, Yunnan and Guangxi provinces are not suitable because of their long distance from Steppe Gold; they also benefit from proximity to coastal ports for seaborne imports.

Although zinc and lead concentrates are the main source of revenue for Steppe Gold Phase 2 Project. Pyrite concentrate is forecasted to contribute additional revenue.

19.4 Contracts

With the ATO Acquisition, Steppe Gold’s subsidiaries, Steppe Gold LLC (“Steppe Mongolia”) and Steppe Investments LLC (“Steppe BVI”) entered into a metals purchase and sale agreement (the “Stream Agreement”) dated August 11, 2017 with Triple Flag International to sell gold and silver produced from the ATO Project and was amended on September 30, 2019. For more details, Readers can refer to Section 22.3.4 of the Report.

19.4.1 Supply and Services

Steppe Gold has a number of contracts, agreements and/or purchase orders in place for supply and services that are material to the operation. All contracts or agreements are negotiated with local vendors and have a contractual scope, terms and conditions. Contracts are negotiated and renewed as needed. Contract terms are considered to be within industry norms. A list of the material contracts currently in place is shown in Table 19.1.

Table 19.1 – Material Contracts Currently in Place

Material contracts currently in place			
#	**Product**	**Supplier**	**Status of contract**
1	Heavy equipment (CAT) Parts and maintenance	Barloworld Mongolia LLC	Supply agreement
2	Diesel	PEC Mongolia LLC	Supply agreement
3	Plant reagents	Hebei Chengxin Co Ltd (Cyanide) Dow Chemical Mongolia LLC (other reagents)	Sales and purchase agreement
4	Power	Barloworld Mongolia LLC	Supply and service
5	Grinding media		
7	Open pit explosives	Special Mining LLC Ord Geo LLC	Service agreement
8	TMA construction	RES LLC Bazura Design LLC Erkhet Construction LLC Sukhjin LLC Gom Ilch LLC	Construction agreement
9	Camp and catering	Penin LLC	Service agreement
10	Heavy fabrication	Professional Supply Services LLC Ukhaa Sar LLC	
11	Heavy equipment (drills) parts and maintenance	Unitra LLC Professional Supply Services LLC Ukhaa Sar LLC Euro Khan LLC Wurth Mongolia LLC	Supply agreement
12	Tire supply	Bridgestone, Tavanbogd Mongolia LLC	Supply agreement
13	Laboratory service	ALS Group LLC	Service agreement
14	Hospital service	Biomedpharm LLC	Service agreement
15	Processing plant spare parts	Shangdong Xinhai Mining Technology &Equipment Inc	Supply agreement
16	Crusher, spare parts	Ikher Bar LLC	Supply agreement

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL IMPACT

20.1 Introduction

Information presented in this section is based on publicly available information, supplemented with environmental baseline studies conducted within the Project's property boundary and surrounding area. Information included herein may require review and reassessment should changes to the scope, area, or design of the project occur as project planning and design progress.

20.1.1 Site Visit

Ulzii Environmental LLC (licensed environmental consulting firm to conduct Environmental Impact Assessment #0000211) Principal Environmental Consultant Mr. Ulziibayar Dagdandorj MAusIMM (Member) Env (#335969) discipline of Environment), conducted an initial reconnaissance of the ATO gold mine Project site from June 25 to June 27, 2021.

Senior Associate Dan V. Michaelsen, FAusIMM (CP) Env (#200991) has supervised the preparation of ITEM 20 – as 20. Environmental Studies, Permitting, and Social Impact with that instrument and form.

20.1.2 Project

There was, however, considerable evidence of ATO gold mine project development and operations Phase 1 footprints. During Phase 1 of the Project, Steppe Gold have been managing the Environmental and Social activities compliant with requirements of Mongolian Mining and Environmental Legislations.

In addition to information gathered during the site visit, ATO's Environmental Permits and all relevant environmental and social plans and other documents were reviewed. In the past, Steppe Gold has conducted detailed Environmental Impact Assessments (EIA) and Environmental Management Plan (EMP) (for 5 years) in 2019 under the requirement of the Mongolian Environmental Legislation.

20.2 Environmental and Social Baseline

20.2.1 Environmental Setting

The Project site, under mining license (MV-017111), is located in the territory of Tsagaan Ovoo soum, Dornod province of Eastern Mongolia. Size of the mining licensed area is 5,492.63 ha.

The Project site is located 660 km east of Ulaanbaatar city, 120 km northwest of Choibalsan, the provincial capital of Dornod province, and 38 km west of Tsagaan Ovoo soum. It borders mountains such as Delger Ulziit, Bayan, Namkhai Hill and Yaruu. The geographic zone of the Project is in datum WGS-84 Zone 49N of UTM coordinate system and the geographic center of the property is 48°26'N latitude and 112°46'E longitude.

Geographically, the license area is in the low mountain zone at the north-east end of the Khentii Mountain Range and at the south-west part of the Dornod high steppe. The topography of the Project area generally consists of small rounded, separate mountain complexes with small hillocks in a steppe. The average elevation is 980-1050 m above sea level, with the lowest point being Deliin Well (979.3 m) and the highest point being Mount Temdegt (1144.7 m). The relative elevation is 60-120 m. Mostly brown and black, brown gravel, sandy loam, and gravel-mild clay of steppe zone are predominant.

The Project site is situated in "Myangan Khonit", at the west side of Davkhar hill, stretching from north to the south. Geological formation consists of sediment accumulations from Lower Permian to the Quarternary period and site area is located in the zone, with two examples of magma and tectnic activity.

At the Project site, it snows rarely in winter, air moisture is low, windy in spring and its hot and cold mode can fully typify an extreme climate condition of the region. The lowest registered air temperature was -45.5°C, while the hottest air temperature was +39.8°C. Annual average air temperature is -0.4°C. In January which is the coldest month of the year, the average temperature is -21.8°C and in July, the hottest month of the year, the average temperature is +19.1°C.

The annual average precipitation at Tsagaan-Ovoo soum is 250 mm. The wind speed average at Tsagaan-Ovoo soum is not high, around 2.5-3.0 m/sec. Windless or calmness frequency, one main indicator of wind mode is 21%. In most months, wind speed reaches more than 15 m/sec, while in springtime it has reached 28 m/sec.

20.2.2 ENVIRONMENTAL AND SOCIAL BASELINE STUDIES

An Independent Environmental Baseline Assessment (EBA) for the Project site and surrounding areas was conducted in 2019 by a Mongolian entity with professional certification for EIA services granted by the government.

The EBA acts as the Project's statutory study of relevant environmental and social baseline research.

Steppe Gold has routinely engaged stakeholders and community members on matters of environmental monitoring and management since 2019, and has reported results for impacts on air, water, soil, and biodiversity to relevant authorities as required by law.

20.2.2.1 Air Quality

Air quality in the Project area is generally good due to the lack of any emission sources. The only man-made emission source affecting the area is dust from the use of local roads, including public traffic, and natural wind-blown dust because of local weather conditions. In the past 4 years, Steppe Gold periodically monitored air quality within the Project area which has provided an indication of the prevailing dust levels and emissions. Data collected during that period, showed relatively consistent conditions for rainfall, air temperature, humidity, wind speed and wind direction.

20.2.2.2 Water Resources

The water network of the area belongs to the Pacific Ocean basin. Small local rivers with short-lived streams fed from eastern branch mountains of the Khentii Range flow into small lakes. Size of these rivers vary depending on their main source of water collection which is precipitation. Drinking water can only be obtained from wells due to low density of water network. Lakes in the region such as Duut, Tsagaan, Ovoot, Eregtseg, Ukhaagiin Tsagaan, Davkhariin Tsagaan, and Khaichiin and many other small salt lakes are also fed by rainfall. In recent years, small rivers and streams have dried up due to global warming and the decrease of precipitation. In the summertime, seasonal springs can be found from the melting of small patchy permafrost in small intermountain valleys and from seasonal thawing of frozen ground.

Drainage from the Project area follows the gradient to the west into the northerly draining basin towards the Bayangol River. The basin to the east of the Project area drains to the south. Regional drainage in the Project area is to the east, interrupted by the 22 km north-south trending elevated ridge where the Project is located.

20.2.2.3 Land and Soil

The land surrounding the property is predominantly used for nomadic herding of goats, cows, horses, and sheep by small family units. Use of the land is based on informal traditional Mongolian principles of shared grazing rights with limited land tenure for semi-permanent winter shelters and other improvements. The site area can be included in brown soil zone of steppe in terms of soil classification of Mongolia. Mainly dark brown and brown soil at the hillside looking north and brown and light brown soil at the hillside looking south can be found. Saltmarsh is accumulated in lowlands and salty soils prevail around them. Depending on ecological condition fauna, sunken or raised land, moisture temperature mode and soil rocks etc. of the soil layers are dominant in the Project area.

20.2.3 BIODIVERSITY

The Project site is included in Mongolian Daurian landscape in terms of zoogeographic regions of Mongolia. A total of 32 bird species, 1 reptile species, 1 lagomorpha species, 1 Artiodactyla species, 6 rodent species and 3 carnivora species were registered during the environmental baseline studies at the site in 2017 and 2019. There was no indication of any dangered or endangered animals in the area.

Hoofed animals of steppe in the region include white gazelle. Carnivores are wolf, fox and corsac. Rodents include marmot, gopher, shrew-mouse, and stoat. Birds include lark, red nose, crane, bustard, scoter, and brown nose. Also, crawlers, locust, grasshoppers, mosquitoes and midges are abundant.

The Project site is located at the border of forest-steppe and steppe zones. Constant warm periods for plant growth is 150-170 days. Fertile and powdered carbonate, pale sandy, light muddy soil is common in the soil covers, and salty pale muddy soils and brown muddy soils can be found near rivers, streams, and lakes, while rocky, gravel sandy and pale brown soils are common at the top or sidehill of mountains and hills. A total of 214 weed plants of 133 species of 41 families, 8 landscapes and 21 sub-groups of 12 groups have been identified during environmental baseline studies at the site in 2017 and 2019. Vegetation and grass cover the entire area and include pasture plants such as khazaar grass, wormwood, stipa, brome-grass, and couch grass.

Currently, Steppe Gold is conducting biodiversity study and biodiversity offset management plan, started in September 2020, and was completed in July 2021.

20.2.4 Cultural Heritage/Archaeology

Archeologists from the Archeology and Antropology department of the School of Arts and Sciences of the National University of Mongolia carried out archeological exploration under the requirements of Article 17.10 of the Law on Protection of Cultural Heritage of Mongolia.

As a result of the exploration, a total of 51 monuments were found including ancient tomb burials and structures. Furthermore, archeological sites, including Bayan 1, Tumendelger 6, Salkhit 19, Naiman Khanat 23 and Maikhant 1 were found at the border of the licensed area. Archeological monuments are evenly distributed in the area and they are mostly found in hillside and places warmed by the sun. As evidenced by the archeological exploration, this area was inhabited by nomadic people since the Bronze Age.

Archeologists have concluded that these monuments would not be impacted by the Project's mining activity, but advised to inform and cooperate with professional organizations, if any historical or archeological monuments were found at the site, in conformity with corporate responsibility.

20.2.5 Population and Demography

The nearest settlement to the property is the central village of the Tsagaan Ovoo soum, which is settled at side of the Khuuvur Lake, with moderately developed infrastructure. The Tsagaan Ovoo soum consists of six subsections and has total population of 3,800. Nationality consists of 80% Buryats and the rest of Khalkh people.

The community is mainly engaged in domestic animal husbandry, with some plantation agriculture, and growth of vegetables for household use. The central village comprises administrative offices, a cultural center, secondary schools, a hospital, a kindergarten, a communications center, cell phone stations, a gas station, and high-voltage sub-stations.

Tsagaan-Ovoo soum, where the Project is situated, is sparsely populated. Traditional and nomadic cattle-breeding is dominant at the area and there is only one licensed area, where mining exploration is carried out. Infrastructure of the soum is less developed compared to other soums of the province. Tsagaan-Ovoo soum is divided into six administrative units. Bag governor is chosen from the poll among the bag residents and his or her term is four years.

20.3 Environmental Impact Assessment

Based upon completed Environmental baseline research report and approved Mongolian Feasibility Report of the Project, a General Environmental Impact Assessment (GEIA) was completed and approved by the Mongolian Ministry of Environment and Tourism (MMET).

Under the requirements of an approved GEIA, an Independent Detailed Environmental Impact Assessment (DEIA) including a 5 year EMP has been developed by Mongolian entity with professional certification for EIA services granted by the government. The DEIA report was approved by Mongolian Ministry of Environment and Tourism in 2019, which acts as the Project's potential environmental and social impact assessments, and included a 5 year Environmental Management (mitigation) Plan acting as environmental and social impacts mitigation measures.

In addition, several technical environmental studies have been conducted as part of the Project's Phase 1 development and operations. These studies were intended to provide direction for the environmental assessment process and guide the environmental authorities with the information required to determine the range of information and degree of detail needed in the formal EIA.

20.3.1 Environmental and Social Impact and Alterations

20.3.1.1 Topography and Landform:

Temporary changes to the existing topography from mining operations include access and haul roads, laydown and hardstand area, topsoil stockpiles, process plant site, and support infrastructure. Permanent changes include the open pit void, waste rock dumps, and tailings storage facilities.

20.3.1.2 Flora and Vegetation:

Direct impact on the 297 ha of flora and vegetation will mainly occur through clearing for the mine, waste rock dumps, processing plant, tailings storage facility and associated infrastructure.

20.3.1.3 Fauna:

The impact of mining on fauna can generally be described as either primary or secondary. The primary impact of mining on fauna is the direct destruction of habitats through land clearing and earthmoving activities. Secondary impacts relate to activities with varying degrees of disturbance beyond the immediate point where mining is taking place, such as access and haul roads, powerlines, pipeline corridors and other infrastructure, feral animals and general workforce activities.

20.3.1.4 Surface Water Hydrology and Groundwater:

The development of the open pits, stockpiles, waste rock dumps, tailings storage facilities, processing plant and infrastructure often interrupt some of the natural drainage paths. Interference with drainage patterns may result in deprivation of water to drainage systems downstream of the mining developments or localized ‘shadowing’ effects on some vegetation which may be reliant on intermittent flows.

20.3.1.5 Soil and Water Contamination:

Direct impact on the 297 ha on soil cover will mainly occur through clearing for the mine, waste rock dumps, processing plant, tailings storage facility and associated infrastructure. Chemical reactions in waste rock and tailings have the potential to be detrimental to plant growth and could result in contamination of both surface and groundwater. In addition, mining and processing operations transport, store and use a range of hazardous materials including fuels, process reagents, lubricants, detergents, explosives, solvents and paints. If these materials are not properly managed, they may have the potential to cause atmospheric, soil or water contamination and could potentially pose ongoing risks to human health and the environment.

20.3.1.6 The Closure Phase Impacts:

The closure phase is expected to involve a decline in direct, indirect and induced employment, and potential contraction of the local economy. The closure of the Project will negatively impact direct and indirect employment as job losses will occur along the supply chain as well as in induced employment because of the reduced demand for services.

20.3.2 Environmental and Social Management Plans

Management of the Project’s significant environmental and social aspects and impact is achieved through a suite of Management Plans. The following Management Plans will be developed and will outline the expected Project performance.

- Air Quality Management Plan
  - Dust and Emission Monitoring
  - Dust and Emission Control Plan

- Water Resources Management Plan
  - Surface Water Management and Monitoring
  - Ground Water Management and Monitoring
  - Water Recycling and Efficient Use
- Land and Soil Management Plan
  - Soil Cover and Quality Monitoring
  - Erosion Management Plan
- Biodiversity Management Plan
  - Biodiversity Study and Monitoring
  - Biodiversity Offset Management Plan
- Waste Management Plan
- Hazardous Materials Management Plan
  - Chemicals (Cyanide) Management Plan
  - Waste Rock Management Plan
  - TSF Management Plan
  - AMD/ARD Management Plan
- Occupational Health, Safety and Security Management Plan
  - Transport Management Plan
  - HSE Plans and Internal Procedures
- Stakeholder and Communications Management Plan
  - Community Grievance Management Plan
  - Communications Plan
- Local Cooperation and Development Management Plan
  - Local Cooperation and Development Agreement
  - Community Development Programs
- Actual/Detailed and Integrated Mine Closure Plan
  - Progressive Rehabilitation Management Plan
    - Rehabilitation Trails and Revegetation
    - Integrated Progressive Rehabilitation Plan
    - Topsoil management Plan
  - Final Closure Management Plan
    - Decommissioning Plan
    - Mine Pits & Waste Rock Dumps Closure Plan

- Heap Leach Pad Closure Plan.
- TSF closure Plan
- Post-Closure Maintenance and Monitoring Plan
  - Rehabilitation handling and maintenance plan
  - Environmental Monitoring plan
  - Land relinquishment

20.3.3 ANNUAL ENVIRONMENTAL PLAN AND REPORT

Mining License holders are required (under the requirements of mining and environmental laws of Mongolia) to earn approval of EMPs for operations planned each year. Performance is reported annually to the government. Steppe Gold remains in compliance and in good standing with its annual environmental reporting requirements since 2018. The Project's Annual EMP's implementation and performances have resulted more than 90% and passed in 2018, 2019, and 2020. Steppe Gold has earned approval of its Annual EMP from the MMET for its 2021 Project operations and production. The Mongolian Annual EMP covers following listed items/activities.

- Mitigation Measures Plan;
- Rehabilitation Plan;
- Biodiversity Offsetting Plan;
- Community And Resettlement Plan;
- Heritage Management Plan;
- Risk Management Plan;
- Waste Management Plan;
- Environmental Monitoring Program.

20.4 Environmental Permits and Agreements

20.4.1 ENVIRONMENTAL IMPACT ASSESSMENTS

The Project's DEIA studies and reports were approved by the MMETin 2019, which acts as the Project's potential ESIA and includes the 5 year EMP which acts as an environmental and social management plan.

In addition, the Project's DEIA report the Project,s operational use of chemicals was approved by MMET in 2019, which acts as the Project's NaCN and other chemical's importing, transporting, storing and usage risk assessments and chemicals management plans.

Based on all completed environmental and social studies and assessments, Steppe Gold has been developing is Annual EMPs, and implementing the planned environmental and social activities under the requirements of Mongolian Environmental Legislations since 2018.

The current approved EBA, GEIA, DEIA and EMP cover the Project's impact on environmental and the community. A DEIA is an obligatory document for all mining and minerals processing projects in Mongolia.

20.4.2 Hazardous Materials

20.4.2.1 Chemicals Storage Facility

The DEIA on the mine operational use of chemicals was approved by MMET in 2019, which acts as the Project's NaCN and other chemical's importing, transporting, storing and usage risk assessments (under MNS 6458:2014 standard) and chemicals management plans. Steppe Gold has granted a special license to import and use of NaCN.

Steppe Gold built an appropriate and secured chemicals storage facility on-site. All chemical usage is managed and maintained under the relevant regulations and requirements.

The current chemicals storage consists of a flat roofed shelter fixed to sea containers sitting on a concrete floor slab. Sea containers provide additional storage space for reagents. The structure integrates lighting, perimeter fencing, and access control for security. Safety measures include strict access control, security gates with controlled access to keys, eyewash, compliant Personal Protective Equipment, fire suppression, and detailed emergency management plans. Material safety data sheets (MSDS) will be available, spill response and disposal measures in place, and the storage facilities will be bunded to prevent unintentional release to the environment.

20.4.2.2 Heap Leach Pad Cells and Ponds

The Project's heap leach pad cells 1, 2, 3, 4, and 5 were fully commissioned in Phase 1, under operations and productions with the ore crusher and ADR plant. Environmental monitoring and management activities are continuing.

The Heap Leach Facility (HLF) is designed to allow crushed ore stacking to a maximum height of approximately 24 m (measured vertically over the liner system), which results in a design capacity of 5.6 Mt. The HLF comprises the following:

- Conventional, three stage lift (nominally 8 m per lift), free-draining heap over a gently sloping Heap Leach Pad (HLP) along the axis of the ridgeline west of the ADR plant.
- The leach pad graded and constructed in a nominally balanced cut-and-fill manner using locally borrowed (within the heap boundary) rock for structural fill, and placement of the certified HDPE geomembrane liner;
- Permanent and interim perimeter diversion channels and berms manage surface water flows;
- Leach pad certified HDPE liner system was constructed.

20.4.2.3 Waste Rock Piles

Based on acid rock drainage (ARD) testing to date, the current (low grade ore zone) waste rock has low percentage of potentially acid generating (PAG) samples and samples with uncertain ARD potential from the ATO's Pipe 1 and Pipe 4 pits are less than 3%, respectively. Therefore, the current waste rock piles (stockpiles) are expected to be non-acid generating (NAG). Metal leaching (ML) potential of waste rock is currently considered to be low, based on chemistry of leachates. Further test work is required to confirm the initial results.

Based on ARD testing to date, the proposed (high grade ore zone), waste rock has high percentage of PAG samples. Further test work and ARD management works are required to confirm the initial results and to manage PAG rocks from the mine pits.

20.4.3 Environmental and Social Permits

Steppe Gold conducted water resource studies from 2017 to 2019 and received water resource statements (possible usage amount of water resource) from the relevant authorities and received land use permits for mining, construction, other infrastructures sites from local authorities.

The list of completed and received environmental and social permits, and agreements that are already in place (under the requirements of Mongolian Environmental Legislations) documents are shown in Table 20.1.

The listed permits and agreements are being maintained and updated continuously on required deadlines.

Table 20.1 – List of Environmental and Social Permits and Agreements

	Environmental Permit, Approval or Authorisation Activity	Issuing Authority and Approval Agency	Status and Comment
1	**Land**		
1	Land quality assessment	Authority of Land Affairs and Geodesy8 Ministry of Construction and urban Development	Completed and approved in 2018
2	Land use certificate and land use agreement 1 -Mining area	Governor of Tsagaan Ovoo soum, Dornod province	Issued and approved in 2018
3	Land use certificate and land use agreement 2-Accomodation's camp area	Governor of Tsagaan Ovoo soum, Dornod province	Issued and approved in 2019
4	Land use certificate and land use agreement 3 - Heap Leach Pad area	Governor of Tsagaan Ovoo soum, Dornod province	Issued and approved in 2019
5	Land use certificate and land use agreement 4-Water ponds and Drainage's area	Governor of Tsagaan Ovoo soum, Dornod province	Issued and approved in 2018

	Environmental Permit, Approval or Authorisation Activity	Issuing Authority and Approval Agency	Status and Comment
6	Land use certificate and land use agreement 6-Fuel Farm area	Governor of Tsagaan Ovoo soum, Dornod province	Issued and approved in 2018
7	Land use certificate and land use agreement 7-Explosives Storage area	Governor of Tsagaan Ovoo soum, Dornod province	Issued and approved in 2019
2	**Heritages**		
1	Archaeological study report and conclusion of ATO gold mining project	Institute of History and Archeology (authorised/licensed entity)	Completed and submitted in 2019
2	Paleontological study report and conclusion of ATO gold mining project	Institute of Paleontology (authorised/licensed entity)	Completed and submitted in 2019
3	**Water**		
1	Water resource statement for possible water use	Mongol Us state owned enterprise (Water Resource Department) MMET	Completed and approved in 2019
2	**Possible water resource statement** "Possible water resource for the industrial water supply of Altan Tsagaan Ovoo gold project is 34.0 L/s (691.2 m³/day)"	(Water Resource Department) MMET	Dated on 30 May 2013.
3	**Possible water use conclusion** "Possible water use conclusion of Altan Tsagaan Ovoo gold project's processing plant is 9.9 L/s	(Water Resource Department) MMET	Dated on 9 October 2017.
4	**Possible water use conclusion** "Possible water use conclusion of Altan Tsagaan Ovoo gold mine operations and processing plant operations"	Mongol Us state owned enterprise (Water Resource Department) MMET	Dated on 10 July 2019.
5	Hydrogeology study report	Water Management LLC	Completed and submitted in 2019
6	Water use permit (Annual)	Kherlen River Basin Authority	Issued and approved in 2019, 2020
7	Water use conslusion (Annual)	Mongol Us state owned enterprise	Issued and approved in 2018, 2019, 2020
8	Water use agreement (Annual)	Kherlen River Basin Authority	Issued and approved in 2018, 2019, 2020
4	**Waste and Chemicals**		
1	Special License for the importation, transport and use of NaCN	MMET	Issued and approved in 2019

	Environmental Permit, Approval or Authorisation Activity	Issuing Authority and Approval Agency	Status and Comment
2	Domestic solid waste transport and disposal agreement (valid for 2 years)	Dornod Public Service state owned enterprise (authorised/licensed entity)	Signed in 2020
3	Domestic liquid waste transport and disposal agreement (valid for 2 years)	Dornod Public Service state owned enterprise (authorised/licensed entity)	Signed in 2020
4	Medical/Clinic waste transport and disposal agreement (valid for 2 years)	Tsagaan Ovoo soum Health Clinic (authorised/licensed entity)	Signed in 2020
5	Plastic waste transport and disposal agreement (valid for 2 years)	Mog Plastic LLC (authorised/licensed entity)	Signed in 2020
6	Chemical's waste/packages transportation and disposal agreement (valid for 2 years)	Tsetsuukh Trade LLC (authorised/licensed entity)	Signed in 2020
7	Waste oil transportation and disposal agreement (valid for 2 years)	Hi B Oil LLC (authorised/licensed entity)	Signed in 2020
5	**EIAs and EMPs**		
1	DEIA & EMP-ATO gold mine operations project (valid for 5 years)	Completed by Make Green LLC (authorised/licensed entity) and approved by MMET	Completed and approved in 2019
2	DEIA & EMP-ATO Chemicals Facility (valid for 5 years)	Completed by Ekhmongolyn Baigal LLC (authorised/licensed entity) and approved by MMET	Completed and approved in 2019
3	Emergency Response Plan of ATO gold mining project	National Emergency Authority of Mongolia	Completed and approved in 2018, 2019, 2020.
4	Disaster Risk Assessment of ATO gold mining project	National Emergency Authority of Mongolia	Completed and approved in 2019
5	Environmental Audit report of ATO gold mining project (valid for 2 years)	Environmental Compliance LLC (authorised/licensed entity)	Completed and submitted in 2019
6	Regular/Routine Environmental Monitoring Report of ATO gold mining project (Annual)	Ulzii Environmental LLC (authorized/licensed entity)	Completed and submitted in 2020
7	Annual EMPs of ATO gold mining project (Annual)	MMET	Developed and approved in 2018, 2019, 2020, and 2021
8	Annual EMPs execution results (Annual)	Environment and Tourism Agency and Inspection Agency of Dornod Province	EMPs implements resulted more than 90% and passed in 2018, 2019, and 2020
6	**Social & Community**		
1	Local Development and Cooperation Agreement 1	Governor of Dornod Province and Governor of Tsagaan Ovoo soum of Dornod Province	Consulted and signed in 2019
2	Community Resettlement Agreements (confidential)	ATO project indirect impact zone community (13 households)	Consulted and signed in 2018, 2019, 2020

20.4.4 Legal Aspects and Compliance

The Project's key environmental and social requirements under Mongolian laws are regulated through the application of the Law on Environmental Impact Assessment (2012) and the Minerals Law (2006). Steppe Gold received most of the required Project environmental approvals under this legislation through the development of the DEIA completed by independent, licensed Mongolian companies. The DEIA reports contain enforceable commitments for protection of the environment, monitoring and the avoidance and mitigation of Project related impacts.

Steppe Gold has been working to ensure through its environmental and social appraisal and monitoring processes that the projects it finances are:

a. Socially and environmentally sustainable.
b. Respect the rights of affected workers and communities; and
c. Designed and operated in compliance with applicable regulatory requirements and good international practice.

In addition, EBRD and IFC Performance Requirements, Statements of Financial Accounting Standards (FAS 143 - Asset Retirement Obligation-ARO), International Cyanide Management Code (2009) requirements shall be met to develop, modify, and implement environmental and social management plans and mine rehabilitation and closure plans of the Project.

20.5 Community Relations and Stakeholder Engagement

Steppe Gold has strong support for mining industry at all levels of government. The Company has been building strong local support and relationships for many years, prior to commencing its exploration and production efforts.

20.5.1 Government and Industry Relations

Steppe Gold has consulted and continues to consult with provincial and central regulatory departments to discuss project development, operations and project extension plans and activities occurring on-site since initiating exploration activities in the Project area since 2018.

Steppe Gold has been an active member of Mongolian National Mining Association for 4 years and active partner with provincial (local) service providers and supplier companies. During this time, Steppe Gold has fostered relationships with supply and service companies in support of ongoing exploration and operations activities.

The activities mentioned above will be continued in during the Project's Phase 2 as extension construction and operations.

20.5.2 LOCAL COOPERATION AND DEVELOPMENT AGREEMENT

Pursuant to Article 42 of the Law on Minerals of Mongolia, minerals license holders are required to enter into a Local Cooperation and Development Agreement with the local government of the jurisdiction within which a given minerals license is located. In 2016, the Government of Mongolia approved model Local Cooperation Agreements for minerals license holders that commit companies to undertake environmental management in the course of operations and encourage public information sharing about the license holders activities locally.

Steppe Gold has successfully consulted with Dornod Province level government officials and Tsagaan Ovoo soum level governmental officials in 2019, then Steppe Gold has in place a Local Cooperation Agreement with local government through the end of 2019. The current signed Local Cooperation and Development Agreement acts to support province level and sub province as Tsagaan Ovoo soum level social development activities. Key clauses of the local cooperation and development agreement are:

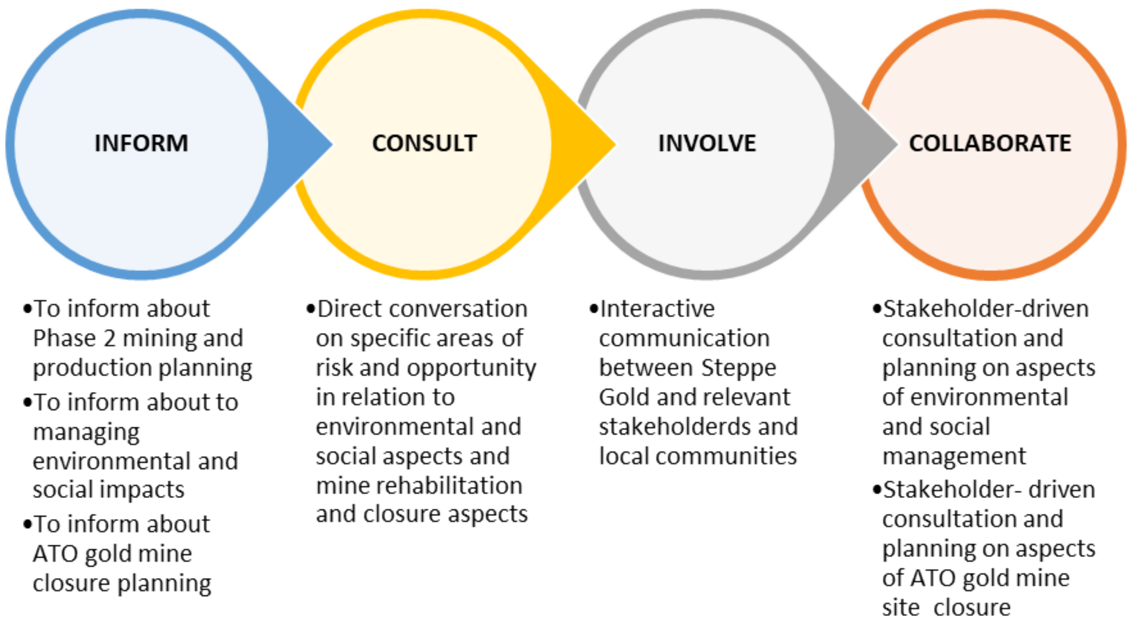
- Mining local work force shall be no less than 75 percent;
- Company shall openly announce and procure from local areas and it shall be no less than 80 percent;
- Prioritise environmental protection and shall not create any non-standard pollution;
- Support local development fund.

20.5.3 STAKEHOLDER ENGAGEMENT

- Stakeholder engagement is an important tool in social impacts and its management. Environmental and social management plans only have relevance in the community and context to which they are designed and communicated. By measuring and monitoring local community and other stakeholder engagement and development prior to production, during, and post-closure, Steppe Gold will be provided with an opportunity to:
- Gain feedback from all stakeholders regarding options and alternatives
- Build relationships of trust or repair fractured relationships
- Maintain the social license to operate
- Help mitigate dependency
- Benchmark environmental and social management plans effectiveness
- Benchmark rehabilitation and closure plan effectiveness
- Enhance the potential for responsible and sustainable operations and productions

The involvement of stakeholders is key to the success of the environmental and social activities of the Phase 2 development and operations of the Project. Steppe Gold is focused to do cooperative and collaborative planning for environmental and social impacts and sustainable land use after mining.

Figure 20.1 – Continuum of Stakeholder Engagement



20.6 Occupational and Community Health, Safety, and Security

The Project HSEC Management Plan will align the EPCM HSEC Management Systems with Steppe Gold's specific HSEC Management Systems, Policies, Plans and Procedures for the Project. The HSEC Management Plan and associated contracts, legislation and codes of practice, identify and encompass the standards, working behaviours, and safe work practices that will be expected of all employees including contractors.

The key potential impacts in relation to occupational and community health, safety and security are linked to the remote nature of the site and associated potential occupational health and safety impacts, including emergency events for workers. The overall impact on the community health, safety, and security is considered to be minor.

20.6.1 Fire Suppression System

All facilities of Project site will have a fire suppression system in accordance with the structure's function. For the most part, fire water will be supplied with a ring main network around the facilities. All buildings will have hose cabinets and handheld fire extinguishers. Electrical and control rooms will be equipped with dry-type fire extinguishers. Ancillary buildings will be provided with automatic sprinkler systems. For the reagents, appropriate fire suppression systems will be included according to their MSDS.

20.6.2 Safe and Healthy Workplace

The Steppe Gold team believes that all accidents/incidents are preventable. They aim to operate a safe and healthy workplace that is injury and fatality free. It is Steppe Gold's intention to provide a zero accident zone in the workplace and enhance the well-being of employees, contractors and communities. To achieve this, Steppe Gold team will:

- Design and operate its facilities to ensure that effective controls and technologies are in place to minimise and mitigate identified risks;
- Maintain occupational health and industrial hygiene programs;
- Promote overall health and wellness and establish programs to protect them;
- Maintain a high degree of emergency preparedness to effectively respond to emergencies.

20.7 Acid Rock Drainage

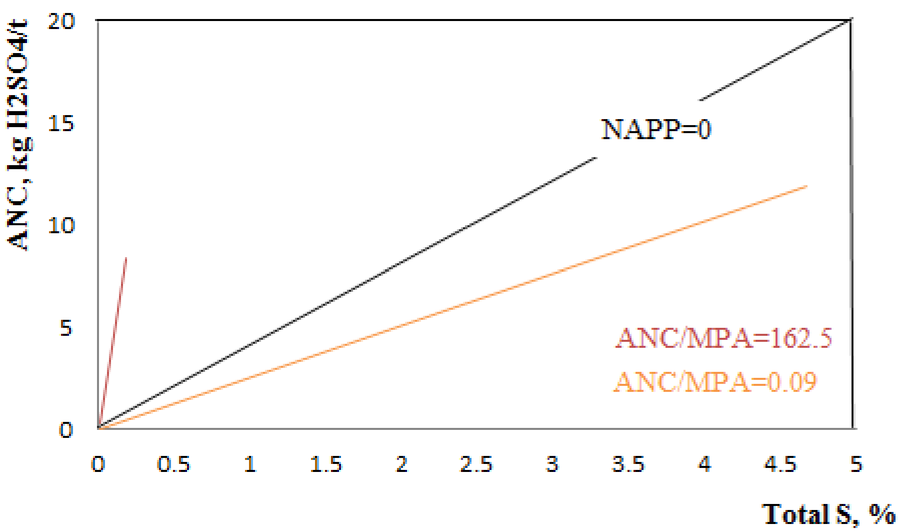
Ulzii Environmental LLC experts have worked at the Project site in March 2021 and collected 2 packages of randomly selected and blended rocks samples from the mine's current waste rock dump and exploration drilling core samples. L-5436 and ATO-WRD-AMD01/21 samples were taken from the current waste rock dump and L-5437 and ATO-PIT-AMD01/21 samples were taken from exploration core samples.

Acid forming potential testing was completed in accredited minerals laboratory analyses from March 2021 to April 2021. Below is the summary of the report on testing for acid production.

20.7.1 Acid-Base Account Plot

Sulphur and ANC data are often presented graphically in a format similar to that is shown in Figure 20.2. This figure includes a line indicating the division between NAPP positive samples from NAPP negative samples.

Figure 20.2 – Acid-Base Account (ABA) Plot



1. NAG rock is material which is highly unlikely to produce acid when exposed to atmospheric oxygen and water and where the Acid Potential Ratio (APR) (defined as ANC/MPA) is > 3. So, L-5436 and ATO-WRD-AMD01/21 sample is classified NAG.
2. PAG rock is material which is highly unlikely to produce acid when exposed to atmospheric oxygen and water and where the APR (defined as ANC/MPA) is < 3. So, L-5437 and ATO-PIT-AMD01/21 sample is classified PAG.

The acid forming characteristics of the two (2) rock samples.

Table 20.2 – Rock Samples Geochemical Classification

No.	pH	EC-(1:2)	Total	Pyrite -	SO_4 -	ANC	NAPP	NAG	NAG-pH	ANC/ MPA ratio	ARD classification
	(1:2)	[μS/cm]	S (wt %)	S (%)	S (%)	kg H_2SO_4/t					
5436	9.22	183.9	<0.05	<0.10	<0.10	8.94	-8.88	0.69	6.35	162.5	NAF
5437	7.71	1356	4.84	4.84	<0.10	12.69	135.41	3.33	2.39	0.09	PAF

Notes:

EC = Electrical Conductivity; ANC = Acid-Neutralisation-Capacity; NAPP = Net-Acid-Producing-Potential; AFP = Acid-Formation-Potential; NAG = Net-Acid Generation; nc = not calculated; NAF = Non-Acid-Forming; PAF = Potentially-Acid Forming.

pH-(1:2) and EC-(1:2) values correspond to pH and EC measured on sample slurries prepared with deionised-water, and a solid:solution ratio of c. 1:2 (w/w).

All results expressed on a dry-weight basis, except for pH-(1:2), EC-(1:2), and NAG-pH.

20.7.2 Key Results of Preliminary Acid Forming Potential Test

- Rock samples have prepared for the geochemical analyses from March 2021 to April 2021.
- The pH1:2 for the composite rock L-5436 and ATO-WRD-AMD01/21 is moderately alkaline and showed limited variability with pH=9.22, EC1:2=183.9 µ/cm. This data suggest that a low potential for immediately release of acidity and salinity from this sample.
- The pH1:2 for the composite rock L-5437 and ATO-PIT-AMD01/21 is moderately alkaline and showed limited variability with pH=7.71, EC1:2=1356 µ/cm. This data suggest that a high potential for immediately release of acidity and salinity from this sample.
- The sample L-5436 and ATO-WRD-AMD01/21 showed total S concentration <0.05 S wt%, while ANC=8.94 kg H_2SO_4/t, NAPP value is -8.88 kg H_2SO_4/t. This sample is classified as NAG.
- The sample L-5437 and ATO-PIT-AMD01/21 showed total S concentration of 4.84 S wt%, while ANC=12.69 kg H_2SO_4/t, NAPP value is 135.41 kg H_2SO_4/t. This sample is classified as PAG.

20.8 Minerals Waste Management

The intent of proposed minerals waste handling plan is to ensure that the management of mining activities and the implementation of environmental and social management plans, mine closure at the ATO and it will be conducted according to best practice methodologies to eliminate the potential for contamination of the surrounding soil and water resources from the generation of Acid Rock Drainage (ARD).

Steppe Gold will implement a predictive ARD operational management plan based on a data developed from a comprehensive waste rock and ore type characterization program that will commence prior to Phase 2 development and operations. The characterization of waste rock and stockpiled ore during operations will be used to verify the predictive ARD model and will be accomplished using on-site testing methods combined with periodic laboratory analysis at a suitable facility.

20.8.1 Waste Rock Management

The intent of this ARD and waste rock handling management plan is to ensure that the management of mining activities and the implementation of mine closure at the Project site and it will be conducted according to best practice methodologies to eliminate the potential for contamination of the surrounding soil and water resources from the generation of Acid Rock Drainage (ARD).

This will be achieved through ensuring:

- A high level of understanding of the ARD characteristics and potential of the various rock types encountered during mining to allow effective operational identification and management of ARD hazards.
- The appropriate design and construction of waste rock landforms and ore stockpiles.
- ARD management controls will be integrated with mine planning and operational grade control and mitigation measures will be clearly communicated and understood by mine management, supervisors, and operators.
- Monitoring programs which will be designed and implemented to allow for the performance of ARD controls to be measured, and corrective actions applied in a timely manner when monitoring indicates ineffective ARD control.
- Installation of appropriate water management features to ensure water captured from waste rock landform does not interact with natural water resources; and
- The effective development and implementation of mine reclamation and closure plans that ensure ARD potential is minimised including the use of backfilling and flooding of mine pits upon cessation of mining activities.

20.8.2 Tailing's Storage Facility Management

Initial static tests have shown a potential for some high-grade ore to be potentially acid generating. However, based on the geology, further metallurgical testing and ARD/ML testing on source rock, and lab-scale process tailings, the combined tailings are not expected to generate ARD.

Humidity cells show that metal concentrations in runoff from tailings beaches will be below MDMER limits. Aging tests of process water indicate that the TSF pond might have excesses to the MDMER limits for CN total, unionized NH_3 (product of CN decomposition) and Cu (added as catalysis during CN destruction or leached from the ore) during operation.

The same parameters might exceed MDMER limits in seepage according to chemistry of leachates from subaqueous columns. Excess water produced by the TSF will be reclaimed to the process plant to offset process water demand and limit volumes of discharge from the TSF pond. TSF excess water that is not reused in ore processing will be treated via a water treatment plant and directed to a polishing pond prior to discharge to the environment. Effluent discharged to the environment will meet MDMER discharge criteria. Ongoing testing and water balance and quality modelling will support future water management plans.

20.9 Environmental and Social Management Plan

20.9.1 Environmental and Social Management Plans

Management of the Project's significant environmental and social aspects and impacts is achieved through a suite of Management Plans. The following Management Plans will be developed and outline the expected Project performance.

20.9.1.1 Air Quality (including noise and vibration) Management Plan:

The following mitigation and management measures are proposed:

- An Air Quality Management Plan will be prepared and implemented.
- Management of mining and transport activities to minimise dust emissions will include limiting vehicle speeds; dust suppression on stockpiles, haul roads and wind erosion sources; using sprays on conveyors and plant equipment; and enclosing plant equipment where possible.
- A Traffic Management Plan will be prepared and implemented (see also Transport impacts).
- A Noise and Vibration Management Plan will be prepared and implemented.
- Optimising, if possible, the number of heavy earth moving equipment to reduce the total noise levels.
- Selecting equipment with noise and vibration abatement technology where possible.
- Regular maintenance of equipment to the manufacturer's specifications.
- Erecting, where required and feasible, noise shields around high noise generating equipment;
- Notifying community member and workers of the blasting schedule

20.9.1.2 Water Resources Management Plan:

The following mitigation and management measures are proposed:

Infrastructure will be designed to minimise interference with natural flow regimes.

- Where necessary, roads will be constructed with gutters to accommodate any stormwater runoff and maintain local hydrology.
- Stormwater will be diverted from operational areas at the Site and directed to natural downstream drainage in a way that prevents increased rates of sedimentation and erosion.
- Water Resources Management Plan (WRMP) will be prepared and implemented.

- Water Monitoring Plan (WMP) will be prepared and implemented.

20.9.1.3 Land and Soil Management Plan:

The following mitigation and management measures are proposed:

- A Land Disturbance and Rehabilitation Management Plan will be prepared and implemented.
- Soil Disturbance will be managed in accordance with a Topsoil Handling Procedure and Land Disturbance Procedure (LDP), including the planning, stripping, storage and use of topsoil.
- A Transport Management Plan will be prepared and implemented to mitigate impacts from vehicle movements on roads, including dust suppression and road maintenance. Off-road driving will be prohibited, and drivers will be required to adhere to sign-posted speed limits.
- In relation to soil contamination, a Hazardous Materials Management Plan and Waste Management Plan will be prepared and implemented, including measures such as spill kits, protective equipment, and other necessary equipment will be available on-site.
- All hazardous materials will only be transported by licensed operators. Wastes will not be sent off-site for disposal or treatment other than to licensed Contractors.

20.9.1.4 Biodiversity Management Plan:

The following mitigation and management measures are proposed:

- Biodiversity Management Plan will be prepared and implemented.
- In all cases, the Project will implement a Land Disturbance Permit Procedure for areas to be disturbed.
- Rehabilitation and revegetation will occur for areas of temporary disturbance due to Project construction activities to the extent possible.
- Transport Management Plan will be prepared and implemented to mitigate impacts from vehicle movements along roads, including dust suppression and road maintenance. Off-road driving will be prohibited, and drivers will be required to adhere to sign posted speed limits.
- Traffic and potential areas with wildlife crossings are monitored, and records are kept of any collisions.

20.9.1.5 Waste Management Plan:

The following mitigation and management measures are proposed:

- Hazardous Materials and Waste Management Plan will be prepared and implemented.
- No hazardous waste will be landfilled at the site - all hazardous wastes will be transported to licensed waste facilities.

- Disposal of wastes will primarily be managed through minimizing waste generation, recycling of waste and appropriate disposal.
- Waste will be segregated to allow appropriate handling, storage, treatment, and disposal by waste stream. These will be classified using national standards and international guidelines.
- Waste inventory will be maintained including of quantities of waste generated per month.
- Where possible, recyclable materials will be made available for reuse, such as wood, tires, scrap metal, or cardboard, as per consultation with stakeholders.
- Wastewater treatment plant will be installed and commissioned for use.
- Wastewater will be treated in accordance with national standards prior to re-use for dust suppression or discharge to environment.

20.9.1.6 Hazardous Materials Management Plan:

The following mitigation and management plans are proposed:

- Chemicals (Cyanide) Management Plan;
- Waste Rock Management Plan;
- TSF Management Plan;
- AMD/ARD Management Plan.

20.9.1.7 Occupational and Community Health, Safety and Security Management Plan:

The following mitigation and management measures are proposed:

- An Occupational Health, Safety and Security Management Plan, Community Health, Safety and Security Management Plan, and Crisis and Emergency Response Management Plan will be developed and implemented.
- Dedicated measures for emergency response will deal with the potential for off-site incidents that may affect local communities and will include arrangements for prompt notification, communication, and evacuation as well as collaboration with the local authorities and communities to build capacity for emergency preparedness.
- The ATO will apply the principles of the International Cyanide Management Code for the manufacture, transport and use of cyanide to ensure good international industry practice.
- There will be barriers to public and livestock access to the mine site through use of stockproof fencing, and security personnel, armed with lethal and/or non-lethal weapons, will be on-site to ensure that there is no unauthorised public access.
- Health screening will be conducted for employees and contractors, in addition to ongoing health-related awareness training. The workforce will be housed on-site to minimise

interactions with the community while working, in addition to the implementation of strict camp rules for employees and contractors.

- Consultation on traffic awareness with community and herder groups affected by the ATO and related traffic generation. Consultation with police, border and emergency services agencies is recommended to coordinate emergency response and preparedness.
- Ensure measures in place to mitigate any transport through community centers include speed restrictions and bypass routes where appropriate. Drivers should follow pre-determined routes that have been subjected to risk assessments.
- Maintaining or improving road sections, where feasible.
- Stipulations that all driving by mine and service personnel is to occur during day-time hours, where possible to improve safety.

20.9.1.8 Stakeholder and Communications Management Plan:

The following mitigation and management plans are proposed:

- Community Grievance Management Plan;
- Communications Plan.

20.9.1.9 Local Cooperation and Development Management Plan:

The following mitigation and management measures are proposed:

- Local Cooperation Management Plan, and Stakeholder and Communications Management Plan will be developed and implemented, and include the following provisions to enhance beneficial impacts:
- Seek opportunities to leverage funds and build partnerships with Government, civil society groups, and other private companies working in the region.
- Target community development activities based on impacts, and specifically for those community members who are potentially adversely affected by the Project.
- Priorities of investment into human capital (rather than capital expenditure and infrastructure) for more sustainable outcomes. Steppe Gold's approach to community development will prioritize needs-based, participatory planning and implementation, which are critical for the success and sustainability of community-based development plans.
- Conduct community consultation to further define potential community development activities for support by Steppe Gold prior to finalization of the Local cooperation and development agreement.
- Steppe Gold will be obtaining feedback on how existing successful community development activities and programs may be further scaled-up consistent with local needs and participation (e.g., scholarship program, water stewardship activities).

20.9.2 Environmental and Social Management Costs

The estimated cost to complete the Environmental and Social activities for the Project included in the financial analysis sections of this report are based on the Project's Annual EMPs (from 2018 to 2021), DEIA report (Make Green 2019), Annual Environmental Monitoring Report (from 2018 to 2021) completing the environmental and social activities during the Project operational life described above. These costs are based on the current level of detail for the Project and is equivalent to a Class 4 Estimate (±20%).

Table 20.3 – Cost Summary ATO Environmental and Social

Items	Description	Total Cost ($USD)
1	ATO Environmental Impacts Management Activities (10 years)	
1	Air Quality MP	402,000
2	Water Resource MP	955,000
3	Land and Soil MP	339,000
4	Biodiversity MP	546,000
5	Waste MP	532,000
6	Hazardous Materials MP	1,020,000
7	Occupational Health, Safety, Security MP	3,911,500
8	Other	1,516,155
	Sub-Total Environmental Impacts Management	*9,221,655*
2	ATO Social Impacts Management Activities (10 years)	
1	Local Community Relations	659,000
2	Local Cooperation and Development Agreement	3,900,000
3	Corporate Social Responsibility & Sustainability activities and support (students' scholarships and donations)	700,000
4	Other	285,390
	Sub-Total Social Impacts Management l	*5,544,390*
	Grand Total	14,766,045

20.10 Rehabilitation and Mine Closure Plan

A complete actual and detailed or integrated Mine Closure Plan has not been developed for the Project at the time of writing. However, Steppe Gold have a Conceptual Mine Closure Plan developed by Polaris Engineering Consulting LLC in in 2019. Closure costs estimated for this Conceptual Closure Plan is based on the current project description and are considered part of a Class 4 estimate with an accuracy of ±20%. The Conceptual Mine Closure Plan describes the general rehabilitation and closure philosophies that will be used for the financial reporting.

In addition, Steppe Gold's financial department have been working with Polaris Engineering Consulting LLC experts to prepare quarterly and year end ARO estimate reports as Unplanned Mine Closure plan to date under the Financial Provision cost estimate (as per the statutory requirement of the jurisdiction in which the company operates and the Statement of Financial Accounting Standards No. 143 Accounting for Asset Retirement Obligations FAS143, IFRIC1, IAS 37 requirements. Closure costs estimated for this Unplanned Closure Plan to date is based on the current project footprints and are considered part of a Class 3 estimate with an accuracy of ±10%.

The current Project's Conceptual Mine Closure Plan and Asset Retirement Obligation (ARO) estimate reports are important baseline documents to develop further Integrated Mine Closure Plans for the Project's Phase 2 construction, operation, and production.

- Mine Closure costs were estimated up to $10.1 M on the Conceptual Closure Plan which is included operational stage rehabilitation, environmental and social management and final closure, and post closure activities costs.
- ARO cost estimate was based upon current facilities on-site and footprints up to date as Unplanned Mine Closure plan to date of 31 March 2021 cost, were estimated up to $1.9 M, which included only decommissioning, unplanned closure by date and post-closure activities costs.

20.10.1 Mine Closure and Rehabilitation Framework

Mine closure and reclamation will be performed in accordance with Mongolian regulations and guidelines. All buildings and facilities not identified for a post-mining use will be removed from the site during the salvage and site demolition phase.

The conceptual mine closure plan ("CMCP") for the Project will be reviewed and continually improved as Actual or Integrated Mine Closure Plan during the Project's Phase 2 as development and operations of the Project. A statutory final mine closure plan must be filed with the government three years prior to the planned completion of mine operations. Consideration will be given to the following statutory and voluntary Project standards for mine closure and reclamation planning;

- Mongolian Minerals Law (2006),
- Law on Environmental Impact Assessment (2012)
- Law of Mongolia on Toxic and hazardous chemicals importing, transporting and disposals (2006)

- Regulations, Guidelines, and MNS standards of Mongolia on Mine Reclamation and Closure (2015-2019)
- Integrated Mine Closure Framework (ICMM-2018)
- International Cyanide Management Code (2009)
- EBRD and IFC applicable performance standards
- Statement of Financial Accounting Standards No. 143
- Accounting for Asset Retirement Obligations FAS143, IFRIC1, IAS 37

20.10.2 Mine Closure Planning

The Actual as Integrated Mine Closure Plan for the Project will be developed and implemented under the set of principles grouped under four key areas as detailed in this Section.

20.10.2.1 Physical Stability:

Steppe Gold remains after closure should be constructed or modified at closure to be physically stable, ensuring it does not erode, subside, or move from its intended location under natural extreme events or disruptive forces to which it may be subjected.

20.10.2.2 Chemical Stability:

Steppe Gold remains after closure should be chemically stable; chemical constituents released from the project components should not endanger human, wildlife, or environmental health and safety, should not result in the inability to achieve the water quality objectives in the receiving environment, and should not adversely affect soil or air quality into the long-term.

20.10.2.3 Not Long-Term Active Care Requirements:

Steppe Gold remains after closure should not require long-term active care and maintenance. Thus, post-closure environmental monitoring would be required for a defined period by third party contractor.

20.10.2.4 Sustainable Future Land Use:

The Project site should be compatible with the surrounding lands and water bodies once closure activities have been completed. The selection of closure objectives at the mine site should consider: The rehabilitation and the cost estimate is based on previous plans and experience on similar projects and complies with the current guidelines and accepted international practices.

The Actual as Integrated Mine Closure Plan for the Project will be described as the process of rehabilitation for the Project at any stage, up to and including closure. Rehabilitation is defined as measures taken to restore a property as close to its former use or condition as practicable, or to an alternate use or condition that is deemed appropriate and acceptable by Mongolian Ministry of Mine and MMET. There are three key stages of rehabilitation activities that occur over the life span of a mine, which include:

1. Progressive Rehabilitation;
2. Closure Rehabilitation;
3. Post-Closure Monitoring and Treatment.

Progressive rehabilitation involves rehabilitation that is completed throughout the mine operation prior to closure wherever possible or practicable to do so. This will include activities that contribute to the overall rehabilitation effort and that would otherwise be carried out as part of the closure rehabilitation at the end of mining life.

Closure rehabilitation involves activities that are completed after mining operations cease to restore and/or return the project to as close to its pre-mining condition as practicable. Such activities include demolition and removal of site infrastructure, re-vegetation of disturbed areas, and other activities.

Once closure rehabilitation activities have been completed, a period of post-closure monitoring is required to ensure that the rehabilitation has been successful. The post-closure monitoring will continue until it has been demonstrated that the rehabilitation of the site has been successful. The site can then be closed out or released by Ministry of Mine and MMET and the Local Government, and an application made to relinquish the property back to the Local Government and Community.

20.10.3 Proposed Approach to Rehabilitation & Closure

As the planning and design stages of the Project continue, consideration for the future closure issues and requirements will be incorporated into final plans. In efforts to be proactive with rehabilitation activities, the following steps will be implemented:

- Disturbances of terrain, soil, and vegetation will be limited to the areas necessary to complete the required work.
- Organic soils, mineral soils, glacial till, and excavated rock will be stockpiled separately wherever possible, and protected for future use.
- Stabilisation of disturbances will be completed to reduce erosion and promote natural re-vegetation.
- Natural and man-made re-vegetation will be encouraged throughout the Project.

20.10.3.1 Progressive Rehabilitation

As the mine advances from development to operational stages, opportunities for progressive rehabilitation are possible. Such opportunities include but are not limited to the following:

1) Progressive/Continuous Rehabilitation (rehabilitation trials and topsoil management) activities as Rehabilitation trials, nursery establishment and its operations and topsoil storage management as re-vegetation works for long term

2) Demolishing and rehabilitation of construction or exploration-related buildings, roads, laydown areas, etc.

3) Grading and revegetation of completed tailings areas, if possible.

4) Barricades and signage long the high walls of the open pits.

5) Erosion stabilization and re-vegetation of completed overburden and/or waste rock pile areas.

6) Infilling or flooding of exhausted mining areas.

20.10.3.2 Closure and Rehabilitation

The closure activities associated with the major components of the Project are described in the following sections. In general, the closure activities that will be completed for the site include, but are not limited to, the following:

1) All hazardous chemicals, reagents, and similar materials will be removed for re-sale or disposal at an approved facility as per regulations.

2) All equipment will be disconnected, drained, and cleaned, disassembled, and where possible, sold for re-use to a licensed scrap dealer. If this is not achievable, equipment will be removed from site for disposal or recycled at an approved facility.

3) All site buildings and surface infrastructure will be dismantled and removed for disposal or recycling at approved facilities.

4) All concrete foundations will be demolished to a minimum of 0.3 m below the surface grade and covered with natural overburden materials to promote re-vegetation. Demolished concrete will be used as fill material for re-grading or removed from site for disposal in an appropriate facility.

5) Fuel and explosive storage and dispensing facilities will be removed, and these areas rehabilitated.

6) Sedimentation, stormwater, and effluent management ponds will be breached to allow drainage to the surrounding areas for natural filtration. Prior to release to the environment, water quality testing will be completed on the pond waters. These features will subsequently be graded and contoured to re-establish drainage patterns and revegetated as required.

7) All wells on-site will be decommissioned. This includes dewatering wells, groundwater monitoring wells, potable drinking water wells and/or industrial water wells. The decommissioning will comply with Mongolian MNS standards.

8) Pre-mining site drainage patterns will be re-established to the extent possible.

9) Disturbed areas will be graded and/or scarified, covered with overburden and organic materials, where required, and seeded to promote natural re-vegetation.

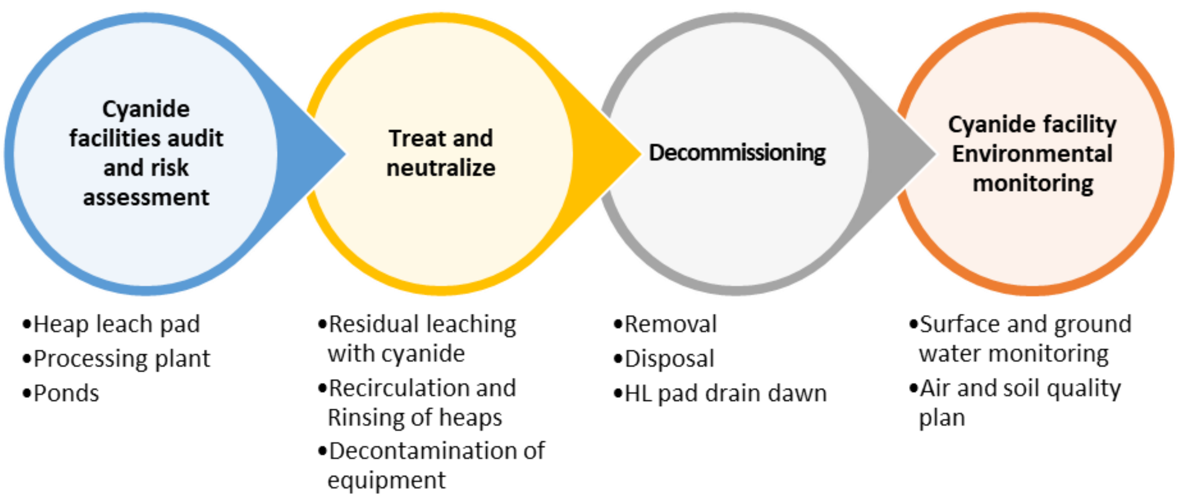
10) Phase I and potentially a Phase II of the Project, Environmental and Social Assessments (ESIA) may be required to evaluate for potentially impacted soils and groundwater.

11) Phase I and potentially a Phase II of the Project, Integrated Mine Closure Plan (IMCP) may be required to evaluate for social license to operate.

20.10.3.3 Neutralisation and Remediation

Decommissioning and closure of cyanide facilities will entail the use in process or detoxification of unused sodium cyanide and the clean-up of cyanide-containing residues in process tanks and equipment.

Figure 20.3 – Cyanide Facility Neutralisation and Remediation Steps



Cyanide facilities decommissioning plan and implementation cost estimates are included in this unplanned/sudden closure plan cost estimates of the Project. Closure cost estimates that specifically includes an estimate of third-party costs to close all cyanide facilities.

20.10.3.4 Open Pits/Pipes Closure

Upon closure, all equipment and dewatering infrastructure will be removed, and the open pit(s) will be allowed to naturally fill with surface water runoff, precipitation, and groundwater seepage. Rock or soil barricades and signage will be constructed along the crest of the open pit(s), as well as across any access roads or ramps, barricading access to the open pit(s). Warning signs will be erected at regular intervals along the berm, notifying the public of the open pit. Areas of sloped access, above and below the final high-water mark, will be constructed to permit ingress and egress for people or animals.

20.10.3.5 Waste Rock Stockpiles Closure

Two waste rock piles (main waste rock and low-grade ore stockpile) will be created throughout the operational life of the Project. These dumps will be constructed from the existing ground surface and will be sloped and benched as they are developed, creating overall safe slopes for final closure of three horizontal to one vertical (3H:1V). The waste rock piles will be progressively rehabilitated via placement of overburden on benches and slopes and subsequently revegetated. At final closure, the remaining areas of the waste rock piles will be rehabilitated in the same manner.

20.10.3.6 Heap Leach Pad Closure

Five cells of the Heap Leach Dam will be created throughout the operational life of the Project. These dumps will be constructed from the existing ground surface and will be sloped and benched as they are developed, creating overall safe slopes for final closure of three horizontal to one vertical (3H:1V). The Heap leach pad will be rehabilitated fifth year of the Project life via placement of overburden and topsoil on benches and slopes and subsequently revegetated.

20.10.3.7 Tailings Storage Facility Closure

The tailings that are produced as part of the mineral extraction process will be stored on-site in the TSF for the first 9 years of operations. The tailings dam embankment will be constructed with a downstream slope of 2 horizontal to one vertical (2H:1V) and will not require further grading at closure.

Upon closure, the tailings solids within the impoundment will be capped with overburden and revegetated. A larger, closure spillway will be constructed to convey water from within the impoundment.

20.10.4 Post Closure Maintenance and Monitoring

The post-closure monitoring program will continue after final closure activities are completed for an estimated 3 to 5 years. However, the monitoring period could be shortened or extended based on the satisfaction of the regulatory bodies that all physical and chemical characteristics are acceptable and stable. When the Project is deemed physically and chemically stable, the site will be relinquished to the central and local government.

The post-closure and long-term monitoring plans are pre-developed in Conceptual Closure Plan of the Project. It is anticipated that the post closure monitoring plans will mirror the operational monitoring program to provide continuity of data and a historical baseline. It is also anticipated that as post-closure time increases the monitoring requirements will decrease until ultimately, they will no longer be required.

Post-closure environmental monitoring, which shall be executed after mine closure is to observe and register geotechnical transformation, generated by dimensional effects of open pit, monitor pit slope stability, determine whether structures at the Project area have been changed and need a rehabilitation work, whether mining activity negative impacts to the humans, animals and environment have reduced, eliminated or activated, whether a new impact has been generated, monitor, check and report result of mine rehabilitation works by using particular methods and implement necessary measures.

20.10.5 Mine Closure Cost Estimate

The estimated cost to complete the closure activities for the Project, included in the financial analysis sections of this report, are based on ATO gold project's Conceptual Closure Plan (Polaris Engineering 2019), the Project's ARO estimate report (Polaris Engineering 2019 Q1 2021), Feasibility Study Report (Midas Mining 2019) Detailed Environmental Impact Assessment report (Make Green 2019), completing the closure activities described above. These costs are based on the current level of detail for the Project and is equivalent to a Class 4 Estimate (±20%).

Table 20.4 - Cost Summary ATO Mine Rehabilitation and Closure

Item	Description	Total Cost (USD)
1	**Progressive/Continuous Rehabilitation (Rehabilitation Trails and Topsoil Management)**	
	Rehabilitation trails nursery establishment	80,800
	Rehabilitation trails nursery operations	247,100
	Progressive rehabilitation	429,850
	Other	159,127
	Sub-Total Progressive/Continuous Rehabilitation	*916,877*

Item	Description	Total Cost (USD)
2	**Final Closure Rehabilitation A (Demobilizing and Decommissioning)**	
	Equipment Demobilization (transportation ATO to and from UB 1,200 km)	213,000
	Demolition/Decommissioning (Equipment and buildings removal)	1,189,170
	Cyanide facility decommissioning	863,500
	Other	431,060
	Sub-Total Final Closure Rehabilitation A	*2,696,730*
3	**Final Closure Rehabilitation B (Earth Works, Re-Vegetation and Handling)**	
	Topsoil movement and placement (earth works)	
	Re-vegetation	1,386,000
	Maintenance and handling (labour and operations)	499,200
	Other	395,892
	Sub-Total Final Closure Rehabilitation B	*2,281,092*
4	**Post Closure Monitoring and Maintenance (Land Relinquishment) for 5 Years**	
	Workforce	72,200
	Field sampling and Laboratory analyzing	657,800
	Other	251,310
	Sub-Total Post Closure Monitoring and Maintenance	*981,310*
	Grand Total	**6,876,010**

21 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimate

Steppe Gold recently completed construction of an open pit heap leach operation at the ATO Project, which is now in operation. The current work, the subject of this Technical Report, assesses the economic processing of sulphide deposits within and beneath the identified ATO orebodies, as well as the Mungu deposit.

The capital cost and operating cost for the Project, as presented herein, have been developed by DRA or consolidated from external sources as listed below:

- DRA prepare the capital and operating cost estimates for the mining, preproduction, and mine infrastructure, the process areas;
- Knight Piésold prepared the estimate for the TSF;
- The geotechnical and environmental information, if any, was provided by external consultants;
- The Owner Costs were provided by Steppe Gold.

All costs provided by external sources were free of contingency and escalation.

21.1.1 Introduction

The Capital Cost Estimate (CAPEX) prepared for this Report is based on the scope of work as presented in earlier sections of this Report.

The CAPEX consists of direct and indirect capital costs as well as contingency. Provisions for sustaining capital and closure have been completed and are depicted separately from initial CAPEX requirements.

The CAPEX prepared for this Project is based on a Class 3 type estimate as per the Association for the Advancement of Cost Engineering (AACE) Recommended Practice 47R-11 with a target accuracy of +30% -15% at 75% probability based on the information available for the estimate. Although some individual elements of the CAPEX may not achieve the target level of accuracy, the overall estimate falls within the parameters of the intended accuracy.

The CAPEX is reported in United States Dollars ($, USD).

The reference period for the cost estimate is 3rd Quarter 2021.

21.1.1.1 Purpose of this NI43-101 Report

The purpose of this Report is to support Steppe Gold in further developing the project definition as well as to assist Steppe Gold in making a decision to further pursue the Project and to help build stakeholder confidence in the Project.

21.1.2 Estimate Coding

All estimate line items were coded using the developed Work Breakdown Structure (WBS); some adjustments were made to better encompass the scope of work. The CAPEX is organised in accordance with the Project WBS defined for both direct and indirect areas as presented in Table 21.1.

Table 21.1 – WBS – Level 2

Area	Discipline Description
2000	**Mining-Open Pit**
2100	Open Pit Mine Development
2200	Open Pit Mining Equipment
2300	Open Pit Mine Infrastructure
3000	**Mining Site -Surface Infrastructure**
3300	Mine Site Electrical Substation
5000	**Process Plant**
5100	Crushing and Conveyor (Already Existing)
5200	Grinding / Milling / Classification
5300	Rougher Flotation & Regrind
5400	Cleaner Flotation
5500	Tailings Thickening and Concentrate Handling
5600	Leaching and Adsorption (Already Existing)
5700	ADR Plant (Already Existing)
5800	Reagent Preparation (Already Existing) & Grinding Media
5900	Process Plant Services & Utilities
6000	**Tailings /Reclaim Water and Water Treatment Facilities**
6100	Tailings Disposal
6200	Reclaim Water
6300	Water Management
6400	ARD Treatment
6500	Effluent Treatment
6600	Cyanide Destruction / Cyanide Recovery (Already Existing) – For Phase 1
7000	**Power Plant and Distribution**
7100	Power Plant and Distribution

Area	Discipline Description
9000	**Indirect**
9100	Contractor Indirects
9200	Construction Accommodation & Catering
9300	Spares, Fills & Inventory
9400	Engineering Procurement Construction Management
9500	Commissioning
9600	Freight / Traffic Warehouse Services & Logistics
10000	**Owner's Costs**
11000	Owner's Costs
12000	Taxes & Duties
13000	Health Safety and Security
14000	Information Technology
15000	Sustaining Capital, Closure and Rehabilitation
20000	**Project Contingency**
21000	Contingency for Mining
22000	Contingency for Bulk Earthworks
23000	Contingency for Civils & Concrete
24000	Contingency for SMPP
25000	Contingency for Architectural Building
26000	Contingency for EC&I
27000	Contingency for Contractor Indirects
28000	Contingency for Spares
29000	Contingency for External Consultants

21.1.3 Assumptions

The CAPEX is based on the following assumptions:

- Engineering, Procurement, Construction, and Management (EPCM) wherein the in-country EPCM contractor will provide detailed engineering, procurement, and construction management activities for all aspects of the Project. All sub-contracts would be managed by the Owner and/or the EPCM contractor;
- Minimal requirement or limitation with respect to local content in terms of labour, materials, equipment and economic impact;
- No restriction to site at any time during execution of the Project;

- No delays in execution from time of contract award to the selected EPCM contractor as a result of either of the following:
  - Owner's financing charges;
  - Owner's permitting delays.
- No delays as a consequence of labour disputes;
- No underground obstructions for all excavation activities to be performed during the construction;
- Milestone schedule presented in the Report.
- Quotes from vendors for equipment and materials are valid for budget purposes only.
- Suitable backfill material is available locally. Soil conditions are adequate for foundation bearing pressures.
- Engineering and Construction activities are to be carried out in a continuous program with full funding available including contingency.
- Labour productivities are established with input from experienced local contractors.
- Bulk materials such as cement, rebar, structural steel and plate, cable, cable tray, and piping are all readily available in the scheduled timeframe.
- Capital equipment is available in the timeframe shown.

21.1.4 EXCLUSIONS

Unless specifically included in the Owner's Cost (Section 21.1.10), the CAPEX excludes allowances for the following:

- The entire crushing circuit and all related equipment (including civils, structural steel, plateworks, piping, electrical & instrumentations), which is already purchased, under construction on site by Steppe Gold.
- The entire Reagent Building, which is already purchased and under construction on site by Steppe Gold.
- Escalation during construction;
- Interest during construction;
- Schedule delays exceeding two (2) weeks and associated costs;
- Scope changes;
- Unidentified ground conditions;
- Extraordinary climatic events;
- Force majeure;
- Labour disputes;

- Insurance, bonding, permits and legal costs;
- Handover from the current Engineering service provider to the in-country EPCM service provider;
- Receipt of information beyond the control of EPCM;
- Schedule recovery or acceleration;
- Cost of financing;
- All duties and taxes (including VAT);
- Salvage values;
- Expenditures to date (sunk costs);
- Research and exploration drilling;
- Owner's project administration and management team;
- Owner's pre-production and mine development;
- Health, safety, security, and emergency response;
- Process rights, royalties, license fees, and technology fees;
- Training and recruiting of plant operating personnel;
- Working and venture capital;
- Start up and commissioning costs (Owner's commissioning team);
- Environmental / permitting / government / community relations; and
- Clinics.

21.1.5 Currency Exchange Rates

For information purposes, the following is a preliminary list of currencies that could potentially be used, along with the conversion exchange rates based on an average of the previous 90 days prior to finalisation of the CAPEX.

The currency exchange rates shown in Table 21.2 will form the basis for any currency exchanges used in the CAPEX. The rates are based on averages to September 2021.

Table 21.2 – Currency Exchange Rates

Source Currency	Description	Base Currency	Exchange Rate	US Dollar
USD	US Dollar	USD	1.000	1.000
CAD	Canadian Dollar	USD	1.300	0.769
EUR	Euro	USD	0.846	1.150
MNT	Mongolian Tugrik	USD	0.00035	2,855.51
CNY	Chinese Yuan	USD	0.147	6.791

21.1.6 Scope of the Estimate

The initial CAPEX estimate includes all Project direct and indirect costs to be expended during the implementation of the Project.

The CAPEX is deemed to cover the period starting at the approval by Steppe Gold of this Report and finishing after commissioning is achieved. It should hence be understood that this CAPEX excludes transfer to Steppe Gold operations, performance test, start-up, ramp up and operations.

21.1.7 CAPEX Summary

Table 21.3 presents a summary of the initial CAPEX by Major Area. Sustaining CAPEX is distributed over the LOM, separately indicated from the initial CAPEX. Owner's costs and contingencies amounts have been included in this CAPEX.

Table 21.3 – Initial CAPEX Summary

WBS	Major Area	Total Cost ($ USD)
2000	Mining - Open Pit	1,870,684
5000	Process Plant	75,185,111
6000	Tailings/ Reclaim Water and Water Treatment Facilities	13,485,178
7000	Power Plant & Distribution	1,701,307
9000	Indirect Costs	23,130,353
10000	Owner's Costs	1,150,307
20000	Project Contingency	11,477,060
	Total Costs	**128,000,000**

Totals may not add up due to rounding.

21.1.8 Direct CAPEX

The Direct CAPEX includes the material, equipment, labour and freight required for the mine, process facilities, infrastructure and services necessary to support the operation.

21.1.8.1 Mining - Open Pit Surface Infrastructure Direct CAPEX

Table 21.4 depicts the initial capital costs for the Haul Road from Pit to Process Plant and Haul Road from Process Plant to TSF.

The estimated costs for those haul roads were developed from quantities prepared from topography drawings and following logical land contours to minimise cut and fill quantities.

Table 21.4 – CAPEX Summary: Mining - Open Pit

WBS	Major Area	Total Cost ($ USD)
2310	Haul Road from Pit to Process Plant	698,486
2315	Haul Road from Process Plant to TSF	1,172,198
2000	**Total Mining - Open Pit**	**1,870,684**

Totals may not add up due to rounding.

21.1.8.2 *Process Plant Direct CAPEX*

Table 21.5 depicts the initial capital costs for the process plant facilities. This area covers the crushed ore stockpile, process plant including grinding, flotation, dewatering, and utilities. Plant terracing refers to the initial cut and fill and site preparation activities prior to detailed excavation for the facilities.

Table 21.5 – CAPEX Summary: Process Plant Facilities

WBS	Major Area	Total Cost ($ USD)
5100	Crushing and Conveying (crushed ore stockpile only)	3,650,086
5200	Grinding / Milling / Classification	30,561,091
5300	Rougher Flotation & Regrind	13,470,616
5400	Cleaner Flotation	4,379,205
5500	Tailings Thickening & Concentrate Handling	12,550,322
5600	Leaching & Adsorption	-
5800	Reagent Preparation & Grinding Media	4,303,933
5900	Process Plant Services & Utilities	6,269,858
5000	**Total Process Plant**	**75,185,111**

Totals may not add up due to rounding.

21.1.8.3 Tailings Storage Facility

Table 21.6 presents the summary for Tailings Storage Facility CAPEX.

Table 21.6 – CAPEX Summary: Tailings Storage Facility

WBS	Major Area	Total Cost ($ USD)
6110	Tailings Storage Facility	13,152,245
6140	Tailings Facility and Services	332,933
6000	**Total Tailings Storage Facility**	**13,485,178**

Totals may not add up due to rounding.

21.1.8.4 Power Plant and Distribution

Table 21.7 presents the summary for Power Plant & Distribution CAPEX.

Table 21.7 – CAPEX Summary: Power Plant and Distribution

WBS	Major Area	Total Cost ($ USD)
7110	Site Power Generation and Distribution	1,587,790
7120	On-Site Power Supply and Transmission	113,517
7000	**Total Power Plant and Distribution**	**1,701,307**

Totals may not add up due to rounding.

21.1.9 INDIRECT CAPEX

The indirect costs include design, procurement and construction management activities, vendor representatives, spare parts and first fills, and contractor's indirect costs. Table 21.8 depicts the indirect costs.

Table 21.8 – CAPEX Summary: Indirect Costs

WBS	Major Area	Total Cost ($ USD)
9100	Contractor Indirect Costs	9,304,536
9200	Construction Accommodation & Catering	2,816,627
9300	Spares, Fills and Inventory	1,164,551
9400	Engineering Procurement Construction Management	5,354,183
9500	Commissioning	480,000

WBS	Major Area	Total Cost ($ USD)
9600	Freight / Traffic Warehouse Services & Logistics	4,010,455
	Total Indirect Costs	**23,130,353**

Totals may not add up due to rounding.

21.1.10 OWNER'S COSTS

The Owner's Costs were provided by Steppe Gold was compiled by Owner's Team, and consist of costs related to:

- Operational Readiness;
- Environmental;
- Safety;
- Health;
- Security; and
- Information Technology.

These costs are summarised in Table 21.9.

Table 21.9 – CAPEX Summary: Owner's Costs

WBS	Major Area	Total Cost ($ USD)
11000	Operational Readiness	300,000
12000	Duties and Taxes	Not included
13000	Environmental, Safety, Health and Security	723,991
14000	Information Technology	126,316
	Total Owner's Costs	**1,150,307**

Totals may not add up due to rounding.

21.1.11 SUSTAINING COSTS

The sustaining capital requirements for the process plant includes for the purchase of spare parts for equipment, and for replacement of equipment when required. The tailings area sustaining costs cover the expansion of the tailings storage facility as the tailings storage increases in area.

The sustaining capital costs are tabulated in Table 21.10 but not included in the initial CAPEX.

Table 21.10 – Sustaining Capital Cost Estimate

WBS	Major Area	Total Cost ($ USD)
9320	Capital Spares (first year of operation only)	997,822
9330	Operational Spares (first year of operation only)	756,186
15100	Sustaining Capital	16,000,000
	Total Sustaining CAPEX	**17,754,008**

Totals may not add up due to rounding.

21.1.12 Closure and Rehabilitation Costs

At the end of the Project life, it is required that all disturbed areas are rehabilitated, and equipment and buildings are disposed of. The closure and rehabilitation costs of $10,100,000 include for the dismantling and removal of all facilities and services and re-vegetation of the area plus an allowance for contingency. Additionally, an Asset Retirement Obligation (ARO) has been estimated at $1,900,000, which includes decommissioning, unplanned closure by date and post-closure activities costs.

Any revenue generated from the sale of used equipment and materials was not considered in the rehabilitation costs.

21.1.13 Contingency

Contingency was included in this CAPEX to cover items which are included in the scope of work as described in this Report, but which cannot be adequately defined at this time due to lack of accurate detailed design information. Contingency covers uncertainty in the estimated quantities and unit prices for labour, equipment and materials contained within the scope of work. Contingency, as defined herein, is not intended to cover items such items as:

- Major scope changes, such as changes in product specification, capacities, building sizes, or location of the asset or project;
- Extraordinary events, such as major strikes and natural disasters;
- Labour disputes; and
- Price escalation and currency effects.

An average allowance of 9.85%, excluding mining costs and Owner's Costs, was applied to all areas of the Project to cover the unforeseen occurrences that could arise during the Project duration.

21.2 Operating Cost Estimate

This Section describes the basis of estimate and approach taken in calculating the operating costs for Phases 1 and 2 of the Project. Phase 1 includes the existing heap leach operation that will produce gold doré bars during 2022 and 2023 and Phase 2 which includes the new processing plant that will produce lead, zinc and pyrite concentrates from Q4 2023 to Q2 2034.

The Operating Cost Estimate (OPEX) is presented in United States Dollars ($ USD). DRA developed these operating costs in conjunction with Steppe Gold, with specific inputs provided by external consultants. The estimate includes mining, processing, and general and administration (G&A). The estimate has an accuracy of +30% -15%.

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

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

The OPEX is estimated at $668.6 M over the life of mine or $25.64/t of ore processed, with two (2) years of operation for Phase 1 and 10.5 years of operation in Phase 2. Table 21.11 summarises the OPEX by area over the LOM for Phase 1 and Phase 2.

Table 21.11 – Phases 1 and 2 - OPEX Summary by Area

Description by Area	Average Annual Costs (M USD/a)	Cost / t ore processed (USD/t)	Total Cost LOM (M USD)
Mining	14.54	6.97	181.72
Process	27.47	13.17	343.35
General & Administration	11.49	5.51	143.58
Total[1]	**53.49**	**25.64**	**668.64**

1 Figures may not add due to rounding

21.2.1 Mining Operating Costs

Table 21.12 denotes the summary of the Open Pit Mining OPEX over the LOM with an average annual cost and a cost per tonne of material. Mine OPEX was calculated based on the mine schedule over the LOM. Contractor mining rates were provided by Steppe Gold. DRA provided the stockpile rehandling cost by benchmarking similar projects.

Table 21.12 – Open Pit OPEX

Description	Total Cost LOM (M USD)	$/t material (USD/t)
Contract Mining Waste (Oxide)	43.83	1.77
Contract Mining Ore (Oxide)	6.66	2.27
Contract Mining Waste (Fresh)	78.67	1.79
Contract Mining Ore (Fresh)	52.32	2.23
Ore Rehandling	0.25	0.45
Total[1]	**181.72**	**1.91**

1 Figures may not add due to rounding.

21.2.2 PROCESS PLANT OPERATING COSTS

Based on oxides that is leached and the plant design nameplate feed throughput capacities of 2.2 Mt/a for Phase 2, the estimated process operating costs for are divided into five (5) main components: manpower, electrical power, reagents, grinding media and liners, and maintenance.

The breakdown of these costs is summarised in Table 21.13 and is detailed in the following sections.

Table 21.13 – Phases 1 and 2 - Summary of Estimated Annual Process Plant OPEX

Operating Cost	Average Annual Costs	Total Cost LOM	Cost / Ore Processed	Total
	(M $ USD/a)	(M $ USD)	(USD/t)	(%)
Manpower	1.11	13.86	0.53	4.0
Power	13.58	169.76	6.51	49.4
Reagent Consumption	7.45	93.18	3.57	27.1
Grinding Media and Liners	3.64	45.48	1.74	13.2
Maintenance	1.69	21.06	0.81	6.1
Total Operating Costs[1]	**27.47**	**343.35**	**13.17**	**100.0**

1. Figures may not add due to rounding.

21.2.2.1 Manpower Costs - Phase 1 and Phase 2

The total operational manpower averages 92 employees for Phase 2 with 15 salaried employees and 77 hourly employees. Personnel incorporates requirements for plant operation, management, laboratory, and maintenance. The labour rates were provided by Steppe Gold. Table 21.14 depicts the manpower for the process facility. The total annual cost for process plant manpower is estimated at $1,108,776.

Table 21.14 – Plant Manpower OPEX - Phase 2

Description	Number of Employees	Cost (M $ USD/a)
Salaried	15	0.51
Operators	46	0.41
Technicians	4	0.02
Laboratory	7	0.05
Maintenance	20	0.12
Total	**92**	**1.11**

21.2.2.2 Power Costs - Phase 1 and Phase 2

Power is required to operate equipment in the processing plant such as conveyors, crushers, mills, screens, pumps, agitators, plant services (compressed air and water), etc. The unit cost of on-site generated electricity was established at $0.20/kWh. The unit cost was provided by Steppe Gold based on a power study developed in January 2021 by a third-party.

Power consumption was determined based on the total installed power (excluding standby equipment) derived from the Mechanical Equipment List. The power draw was based on the average power utilisation of each motor. The estimated electrical operating costs is based on the plant operating 24 hours per day, 7 days per week, with a run time of 65% for the primary crushing area, 75% for the secondary and tertiary crushing area and 92% for the grinding and flotation areas as an operating percentage. The total average annual cost for process plant electrical power is estimated at $13,580,467.

21.2.2.3 Reagent Consumption Costs – Phase 1 and Phase 2

Reagent consumption rates were estimated based on metallurgical testwork results. Reagent costs were obtained through benchmarking for similar projects performed by DRA and/or provided by Steppe Gold. Details about reagents consumptions are explained in Section 13. The total average annual cost for process plant reagent consumption is estimated at $7,454,620.

21.2.2.4 Grinding Media and Liners - Phase 2

Ball mills will need the addition of steel balls to replace the worn media to maintain the steel load in the mills and to perform proper size reduction on the material. Regrind mills will require addition of media for replacement. Consumption of the grinding media is based on abrasion index, power consumption and experience.

The consumption of crusher liners, grinding mill liners, regrind mill liners was obtained from the equipment suppliers and from experience with similar operations. The average annual cost for process plant grinding media and liners is estimated at $3,638,727.

21.2.2.5 Maintenance Costs - Phase 2

Annual maintenance costs were factored based on the total installed mechanical equipment capital cost using an average factor of 5% The total annual cost for process plant maintenance cost is estimated at $1,685,030. This value includes an annual cost of $300,000 for lease of equipment during the operation of Phase 2.

21.2.2.6 General and Administration Costs

The General and Administration (G&A) costs include the following categories:

- Corporate;
- Site services.

The overall G&A average cost per year for Phase 1 and Phase 2 is estimated at $ 7.59 M per annum. Given the nature of G&A costs, plant operations and throughput have little to no impact on these costs. Table 21.15 summarises the G&A breakdown.

Table 21.15 – G&A Costs - Phase 1 and Phase 2

Description	Average Annual Costs	Total Cost LOM
	(M $ USD/a)	(M $ USD)
Corporate	6.68	83.51
Site Services	4.81	60.07
Total[1]	**11.49**	**143.58**

1 Figures may not add due to rounding

22 ECONOMIC ANALYSIS

The Project has been evaluated using discounted cash flow (DCF) analysis. Cash inflows were estimated based on annual revenue projections. Cash outflows consist of operating costs, capital expenditures, royalties, and taxes. The analysis considers 2 years of production in Phase 1, (existing operation) and 10.5 years of production through Phase 2.

The Net Present Value (NPV) of the Project was calculated by discounting back cash flow projections throughout the life-of-mine (LOM) to the Project's valuation date using three (3) different discount rates, 5%, 8%, and 10%. The base case used a discount rate of 5%. The internal rate of return (IRR) and the payback period were also calculated.

Table 22.1 and Table 22.2 summarise the economic/financial results of the Project for the base case (Phase 1 and Phase 2) and for the Phase 2 respectively. All figures are in USD currency.

Table 22.1 – Base Case Financial Results (Phase 1 and Phase 2)

Financial Results	Unit	Pre-Tax	After-Tax
NPV @ 5%	M USD	319.97	232.08
IRR	%	108.8	66.6
Payback Period	Year	2.5	3.0

Table 22.2 –Financial Results (Phase 2 Standalone)

Financial Results	Unit	Pre-Tax	After-Tax
NPV @ 5%	M USD	259.35	185.96
IRR	%	48.2	36.5
Payback Period	Year	3.4	3.8

22.1 Financial Assumptions

The economic analysis is based on several technical and economic input assumptions, as presented in Table 22.3.

Table 22.3 – Financial Analysis Assumptions

Description	Unit	Assumption
Discount Rate	%	5
Depreciation		
Mechanical Equipment	Annum	9%
Plant Infrastructure	Annum	9%
Surface Infrastructure	Annum	9%
Transport Charges[1]		
Transport Charges (Pb, Zn, Pyrite Concentrates)	$/dmt of conc	72.72
Treatment Charges[2]		
Treatment Charges Pb Concentrate	$/dmt of conc	151.67
Treatment Charges Zn Concentrate	$/dmt of conc	240.00
Treatment Charges Pyrite Concentrate	$/dmt of conc	0.0
Average Payabilities[3]		
Gold in Lead Concentrate	%	95.0%
Silver in Lead Concentrate	%	95.0%
Lead in Lead Concentrate	%	93.0%
Gold in Zinc Concentrate	%	75.0%
Silver in Zinc Concentrate	%	70.0%
Zinc in Zinc Concentrate	%	82.4%
Gold in Pyrite Concentrate[4]	%	70.0%
Silver in Pyrite Concentrate	%	0%
Exchange Rates	Provided in Chapter 21	

1.Transport costs have been estimated as an average of all buyers from a Logistic study developed by Steppe Gold
2. Treatment charges taken from Section 19. Average from years 2023, 2024 2025 has been used.
3. Details on payabilities of metals are found in Section 19 of this report.
4. Steppe Gold believes payability of gold in pyrite concentrate will be improved through further negotiations. For this study 70% is used.

22.2 Financial Evaluation

DRA has prepared the assessment of the Project on the basis of a discounted cash-flow model, from which NPV, IRR, payback, and other measures of the Project's economic viability can be determined. Assessments of NPV are generally accepted within the mining industry as representing the economic value of a project after allowing for the cost of capital invested.

The objective of the Technical Report was to determine the potential economic viability of developing the Project, which consists of a current heap leach process to produce doré and a concentrator to produce three (3) different concentrates: Lead, Zinc, and Pyrite.

The cash flow arising from the Report has been forecast, enabling a computation of the NPV to be made. The sensitivity of the NPV and IRR to changes in the base case assumptions is also examined. NPV results presented are based on an evaluation and start date of January 2022.

The results of the economic analysis represent forward-looking information that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented.

22.2.1 Commodity Prices

Commodities prices used are based on the review of median consensus price forecasts, supply and demand forecasts over the next decade. A sensitivity analysis was performed to address the impact of various financial and operating variables on the overall project economic results.

For the purposes of the economic analysis, the following commodity prices were assumed:

Phase 1:

- $1,750/oz Au;
- $25/oz Ag;

Phase 2:

- $1,610/oz Au;
- $21/oz Ag;
- $1,970/t Pb;
- $2,515/t Zn.

For the sensitivity analysis prices of metals were evaluated at 80%, 90%, 110% and 120%, prices per metal are shown as follows:

Phase 1:

- Gold price sensitives are tabulated at $1,400, $1,575, $1,925, and $2,100/oz.
- Silver price sensitives are tabulated at $20.0, $22.5, $27.5, and $30.0/oz.

Phase 2:

- Gold price sensitives are tabulated at $1,288, $1,449, $1,771, and $1,932/oz.

- Silver price sensitives are tabulated at $16.8, $18.9, $23.1, and $25.2/oz.
- Lead price sensitives are tabulated at $1,576, $1,773, $2,167, and $2,364/t.
- Zinc price sensitives are tabulated at $2,012, $2,264, $2,767, and $3,018/t.

22.2.2 ESCALATION AND INFLATION

There is no adjustment for inflation and escalation in the financial model; all cash flows are in US dollars.

22.2.3 DEPRECIATION

Depreciation using a straight-line balance method at 9% per annum will be applied to all mechanical equipment part of the concentrator plant.

Depreciation of sustaining CAPEX has also been applied based on units of production.

No depreciation on property assets will be applied.

Depreciation is included to determine Gross and Net Profits indications and has no bearing on the free cashflow NPV calculations.

22.2.4 WORKING CAPITAL

Working capital assumptions were not included in the economic analysis as the facility is currently operating with adequate working capital.

22.3 Taxation and Royalties

The following outlines the main taxation considerations applied in the financial model:

- 10% applies to the first 2.1 Million USD of annual taxable income.
- If annual taxable income exceeds 2.1 Million USD, the tax shall be 0.21 Million USD plus 25% of income exceeding 2.1 Million USD.

22.3.1 VALUE ADDED TAXES

No Value Added Tax (“VAT”) has been advised by Steppe Gold to be included in the estimate for services and goods.

22.3.2 DUTIES

No import duty has been included for process equipment, materials and consumables sourced internationally, per the advisement of Steppe Gold.

22.3.3 Income Tax

Steppe Gold expects to pay minimal income tax in respect of the 2020 financial year at a rate of 25% for the LOM of the Project.

22.3.4 Royalties

Mineral production in Mongolia is subject to fixed and sliding scale government royalties. The production is subject to a flat rate of 5% royalty apart from silver which is subject to a sliding scale royalty starting at 5%, up to $25 USD/oz.

Table 22.4 – Silver Royalties

Product Type	Unit	Market price level	Percentage to be added to the base rate
Silver	$USD/oz	0 - 25	0.00
		25 - 30	1.00
		30 - 35	2.00
		35 - 40	3.00
		40 - 45	4.00
		45 and above	5.00

The ATO Project is also subject to a 1.75% net smelter return royalty in favour of CogeGobi.

In connection with the ATO acquisition, Steppe Mongolia and Steppe BVI entered into a metals purchase and sale agreement, known as a "Stream Agreement", dated August 11, 2017 with Triple Flag Mining Finance International Inc ("Triple Flag").

Under the original terms of the Stream Agreement, Triple Flag advanced Steppe Gold $23 MUSD, obligating Steppe BVI to sell Triple Flag 25% of the gold ounces and 50% of the silver ounces produced from the Project until such time that Steppe BVI has sold an aggregate of 46,000 ounces of gold and 375,000 ounces of silver respectively, known as the "Delivery Milestones". On these terms, the parties agreed that Triple Flag will pay for the delivered metal ounces at 30% of the relevant market price on the delivered date.

Once these milestones have been achieved, Steppe BVI has agreed to sell Triple Flag 5,500 ounces for gold (plus 250 ounces of gold for each three month period in which the commercial production date follows September 30, 2018) and 45,000 ounces for silver (plus 2,045 ounces of silver for each three month period in which the commercial production date follows September 30, 2018) annually, known as the "Annual Cap Amounts". This obligation is in effect for the life of the mine and includes all gold or silver produced by Steppe Mongolia within an agreed area of 20 km from the original mineral licenses comprising the Project. Triple Flag has now determined the Annual Cap Amounts upon the achievement of the Commercial Production Date as the Gold Cap Amount to be 7,125 oz. of Produced Gold annually and the Silver Cap Amount to be 59,315 oz. of Produced Silver annually. On September 30, 2019, Steppe BVI and Triple Flag agreed to amend the terms of the existing Stream Agreement. Under the new terms, Triple Flag advanced an additional $5 MUSD to Steppe that was used to repay the final $5 million payment on a promissory note issued as part of the purchase price for the Project.

In consideration of this additional advance, the parties agreed to reduce Triple Flag's agreed upon purchase price of gold and silver from 30% to 17% of the relevant market price for delivered metals. All other terms of the agreement as noted above remain the same.

22.4 Cash Flow Analysis and Economic Results

22.4.1 Summary of Financial Results

The financial analysis results are summarised in Table 22.5Table 22.3.

Table 22.5 – Economic Summary

Description	Unit	Value
Phase 1		
Tonnage Ore Processed Phase 1	kt	2,614
LOM Recovery - Au Phase 1	%	70.0%
LOM Recovery - Ag Phase 1	%	40.0%
Au recovered in Heap Leach Phase 1 - (2021 & 2022)	oz	98,600
Ag recovered in Heap Leach Phase 1 - (2021 & 2022)	oz	490,501
Production Period Phase 1	Annum	2
Phase 2		
LOM Tonnage Ore Processed Phase 2	kt	23,462
LOM Feed Grade Processed - Au Phase 2	g/t	1.15
LOM Feed Grade Processed - Ag Phase 2	g/t	10.75
LOM Feed Grade Processed - Pb Phase 2	%	0.47%
LOM Feed Grade Processed - Zn Phase 2	%	0.85%
LOM Recovery - Au Phase 2	%	79.2%
LOM Recovery - Ag Phase 2	%	72.6%
LOM Recovery - Pb Phase 2	%	82.5%
LOM Recovery - Zn Phase 2	%	85.9%
Production Period Phase 2	Annum	10.5
LOM Pb Concentrate Production	t	183,894
LOM Zn Concentrate Production	t	293,425

Description	Unit	Value
LOM Pyrite Concentrate Production	t	612,348
Phase 1 + Phase 2		
Total Net Revenue (Phase 1 + Phase 2)	$USD million	$ 1,546,143
LOM Operating Costs (Phase 1 + Phase 2)	$USD million	$ 668,642
	$USD/t ore processed	$ 25.6
All-in Sustaining Costs (Phase 1 + Phase 2)	$USD/oz Au eq)	$ 928

Table 22.6 and Table 22.7summarise the financial results. NPV is calculated at three (3) different discount rates, 5%, 8%, and 10% for the base case (Phase 1 and Phase 2) and for Phase 2 respectively.

Table 22.6 – Project Financial Results Base Case (Phase 1 + Phase 2)

Financial Results	Unit	Pre-Tax	After-Tax
NPV @ 5%	M USD	319.97	232.08
NPV @ 8%	M USD	268.60	191.09
NPV @ 10%	M USD	239.97	168.36
IRR	%	108.8	66.6
Payback Period	Year	2.5	3.0

Table 22.7 – Project Financial Results (Phase 2 Standalone)

Financial Results	Unit	Pre-Tax	After-Tax
NPV @ 5%	M USD	259.35	185.96
NPV @ 8%	M USD	209.68	146.12
NPV @ 10%	M USD	182.20	124.21
IRR	%	48.2	36.5
Payback Period	Year	3.4	3.8

For this Project, the base case (Phase 1 and Phase 2) used a discount rate of 5%. After-Tax NPV is $232.08 M at a discount rate of 5%. The After-Tax IRR is 66.6% and the after-tax payback on initial investment is 3.0 years.

Figure 22.1 depicts the annual cash flow projections.

Figure 22.1 – Cash Flow Statement – Base Case (Phase 1 +Phase 2)

YEAR				*FY2022*	*FY2023*	*FY2024*	*FY2025*	*FY2026*	*FY2027*	*FY2028*	*FY2029*	*FY2030*	*FY2031*	*FY2032*	*FY2033*	*FY2034*
REVENUE SCHEDULE			*(Tot. / Avg.)*													
Total Net Revenue	*($'000s)*		**$1,546,143**	**$113,607**	**$101,248**	**$191,789**	**$147,628**	**$161,700**	**$135,163**	**$130,814**	**$120,447**	**$114,552**	**$105,417**	**$105,057**	**$93,479**	**$25,243**
STREAM SCHEDULE			*(Tot. / Avg.)*													
Stream Schedule	*($'000s)*		**$178,247**	**$25,359**	**$20,995**	**$26,342**	**$10,555**	**$10,555**	**$10,555**	**$10,555**	**$10,555**	**$10,555**	**$10,555**	**$10,555**	**$10,555**	**$10,555**
PRE-TAX CASH FLOW																
Total Net Revenue (After Stream Schedule)	*($'000s)*		**$1,367,896**	$88,247	$80,252	$165,447	$137,073	$151,145	$124,608	$120,259	$109,892	$103,997	$94,862	$94,502	$82,924	$14,688
Less: Production Royalties	*($'000s)*		**($86,185)**	($5,680)	($5,216)	($10,721)	($8,239)	($8,768)	($7,372)	($7,597)	($7,152)	($6,359)	($5,995)	($5,984)	($5,482)	($1,620)
Less:Net smelter Royalties	*($'000s)*		**($28,656)**	($1,889)	($1,734)	($3,565)	($2,739)	($2,915)	($2,451)	($2,526)	($2,378)	($2,114)	($1,993)	($1,990)	($1,823)	($539)
Gross Income	*($'000s)*		**$1,253,055**	$80,678	$73,302	$151,162	$126,096	$139,462	$114,784	$110,135	$100,362	$95,523	$86,874	$86,529	$75,619	$12,530
Less: Total Operating Costs	*($'000s)*		**($668,642)**	($40,094)	($34,082)	($53,811)	($55,161)	($57,014)	($57,181)	($57,313)	($57,529)	($57,687)	($57,828)	($59,320)	($55,540)	($26,081)
EBITDA	*($'000s)*		**$584,414**	$40,584	$39,220	$97,350	$70,934	$82,448	$57,603	$52,822	$42,833	$37,836	$29,046	$27,208	$20,079	($13,551)
Capital Expenditures																
Development Capital	*($'000s)*		**($128,000)**	($64,000)	($64,000)	-	-	-	-	-	-	-	-	-	-	-
Sustaining Capital	*($'000s)*		**($16,000)**	($1,000)	($1,000)	-	-	($2,000)	($2,000)	($2,000)	($2,000)	($2,000)	($2,000)	($2,000)	-	-
Closure Capital	*($'000s)*		**($4,464)**	-	-	($19)	($19)	($19)	($31)	($16)	($16)	($31)	($93)	($93)	($5,901)	$1,774
Total Capital Expenditures	*($'000s)*		**($148,464)**	($65,000)	($65,000)	($19)	($19)	($2,019)	($2,031)	($2,016)	($2,016)	($2,031)	($2,093)	($2,093)	($5,901)	$1,774
Changes in Working Capital	*($'000s)*		**-**													
Interest expense	*($'000s)*		**-**	-	-	-	-	-	-		-	-	-	-	-	-
Pre-Tax Cash Flow	*($'000s)*		**$435,950**	**($24,416)**	**($25,780)**	**$97,331**	**$70,915**	**$80,429**	**$55,572**	**$50,807**	**$40,818**	**$35,805**	**$26,953**	**$25,115**	**$14,178**	**($11,777)**
Adj. Cumulative Pre-Tax Cash Flow	*($'000s)*			*($24,416)*	*($50,196)*	*$47,135*	*$118,050*	*$198,479*	*$254,051*	*$304,858*	*$345,676*	*$381,481*	*$408,434*	*$433,549*	*$447,727*	*$435,950*
Discounted Payback Calculation	*(years)*		**2.5**	*n/a*	*n/a*	*2.5*	*2.34*	*2.53*	*1.4*	*1.0*	*-0.5*	*-1.7*	*-5.2*	*-6.3*	*-19.6*	*50.0*
Mid-Year Adjustment	*(x)*	1		*1*	*2*	*3*	*4*	*5*	*6*	*7*	*8*	*9*	*10*	*11*	*12*	*13*
Discount Factor	*(x)*	5%		*0.952*	*0.907*	*0.864*	*0.823*	*0.784*	*0.746*	*0.711*	*0.677*	*0.645*	*0.614*	*0.585*	*0.557*	*0.530*
Discounted Pre-Tax Cash Flow	*($'000s)*		**$319,966**	**($23,253)**	**($23,383)**	**$84,078**	**$58,342**	**$63,018**	**$41,469**	**$36,108**	**$27,627**	**$23,080**	**$16,547**	**$14,684**	**$7,895**	**($6,246)**
Pre-Tax IRR	*(%)*		**108.8%**													
AFTER-TAX CASH FLOW																
Pre-Tax Cash Flow	*($'000s)*		**$435,950**	($24,416)	($25,780)	$97,331	$70,915	$80,429	$55,572	$50,807	$40,818	$35,805	$26,953	$25,115	$14,178	($11,777)
Less: Corporate Income Taxes (25%)	*($'000s)*		**($110,491)**	($9,831)	($9,093)	($20,995)	($14,338)	($17,157)	($10,875)	($9,607)	($7,018)	($5,652)	($3,295)	($2,631)	-	-
After-Tax Cash Flow	*($'000s)*		**$325,458**	**($34,247)**	**($34,872)**	**$76,336**	**$56,578**	**$63,272**	**$44,696**	**$41,200**	**$33,800**	**$30,153**	**$23,658**	**$22,484**	**$14,178**	**($11,777)**
Adj. Cumulative After-Tax Cash Flow	*($'000s)*			*($34,247)*	*($69,119)*	*$7,216*	*$63,794*	*$127,066*	*$171,763*	*$212,963*	*$246,763*	*$276,915*	*$300,574*	*$323,057*	*$337,235*	*$325,458*
Discounted Payback Calculation	*(years)*		**2.99**	*n/a*	*n/a*	*2.9*	*2.9*	*3.0*	*2.2*	*1.8*	*0.7*	*-0.2*	*-2.7*	*-3.4*	*-11.8*	*40.6*
Discount Factor	*(x)*	5%		0.952	0.907	0.864	0.823	0.784	0.746	0.711	0.677	0.645	0.614	0.585	0.557	0.530
Discounted After-Tax Cash Flow	*($'000s)*		**$232,084**	**($32,616)**	**($31,630)**	**$65,942**	**$46,547**	**$49,575**	**$33,353**	**$29,280**	**$22,877**	**$19,437**	**$14,524**	**$13,146**	**$7,895**	**($6,246)**
After-Tax IRR	*(%)*		**66.6%**													

Figure 22.2 – Cash Flow Statement – Phase 2 Standalone

YEAR						FY2022	FY2023	FY2024	FY2025	FY2026	FY2027	FY2028	FY2029	FY2030	FY2031	FY2032	FY2033	FY2034
REVENUE SCHEDULE			*(Tot. / Avg.)*															
Total Net Revenue	*($'000s)*		**$1,361,331**			**-**	**$30,042**	**$191,789**	**$147,628**	**$161,700**	**$135,163**	**$130,814**	**$120,447**	**$114,552**	**$105,417**	**$105,057**	**$93,479**	**$25,243**
STREAM SCHEDULE			*(Tot. / Avg.)*															
Stream Schedule	*($'000s)*		**$137,354**			**-**	**$5,462**	**$26,342**	**$10,555**	**$10,555**	**$10,555**	**$10,555**	**$10,555**	**$10,555**	**$10,555**	**$10,555**	**$10,555**	**$10,555**
PRE-TAX CASH FLOW																		
Total Net Revenue	*($'000s)*		**$1,223,977**			-	$24,580	$165,447	$137,073	$151,145	$124,608	$120,259	$109,892	$103,997	$94,862	$94,502	$82,924	$14,688
Less: Production Royalties	*($'000s)*		**($76,790)**			-	($1,502)	($10,721)	($8,239)	($8,768)	($7,372)	($7,597)	($7,152)	($6,359)	($5,995)	($5,984)	($5,482)	($1,620)
Less:Net smelter Royalties	*($'000s)*		**($25,533)**			-	($499)	($3,565)	($2,739)	($2,915)	($2,451)	($2,526)	($2,378)	($2,114)	($1,993)	($1,990)	($1,823)	($539)
Gross Income	*($'000s)*		**$1,121,654**			-	$22,579	$151,162	$126,096	$139,462	$114,784	$110,135	$100,362	$95,523	$86,874	$86,529	$75,619	$12,530
Less: Total Operating Costs	*($'000s)*		**($600,449)**			-	($5,983)	($53,811)	($55,161)	($57,014)	($57,181)	($57,313)	($57,529)	($57,687)	($57,828)	($59,320)	($55,540)	($26,081)
EBITDA	*($'000s)*		**$521,205**			-	$16,595	$97,350	$70,934	$82,448	$57,603	$52,822	$42,833	$37,836	$29,046	$27,208	$20,079	($13,551)
Capital Expenditures																		
Development Capital	*($'000s)*		**($128,000)**			($64,000)	($64,000)	-	-	-	-	-	-	-	-	-	-	-
Sustaining Capital	*($'000s)*		**($16,000)**			($1,000)	($1,000)	-	-	($2,000)	($2,000)	($2,000)	($2,000)	($2,000)	($2,000)	($2,000)	-	-
Closure Capital	*($'000s)*		**($4,464)**			-	-	($19)	($19)	($19)	($31)	($16)	($16)	($31)	($93)	($93)	($5,901)	$1,774
Total Capital Expenditures	*($'000s)*		**($148,464)**			($65,000)	($65,000)	($19)	($19)	($2,019)	($2,031)	($2,016)	($2,016)	($2,031)	($2,093)	($2,093)	($5,901)	$1,774
Changes in Working Capital	*($'000s)*		-			-	($1,756)	($7,500)	$3,849	($941)	$2,232	$379	$891	$511	$779	$217	$492	$847
Interest expense	*($'000s)*		-															
Pre-Tax Cash Flow	*($'000s)*		**$372,741**			**($65,000)**	**($50,160)**	**$89,831**	**$74,764**	**$79,488**	**$57,804**	**$51,186**	**$41,709**	**$36,316**	**$27,732**	**$25,332**	**$14,670**	**($10,930)**
Adj. Cumulative Pre-Tax Cash Flow	*($'000s)*					*($65,000)*	*($115,160)*	*($25,330)*	*$49,434*	*$128,922*	*$186,726*	*$237,912*	*$279,621*	*$315,937*	*$343,668*	*$369,000*	*$383,670*	*$372,741*
Discounted Payback Calculation	*(years)*		**3.4**			*n/a*	*n/a*	*n/a*	*3.3*	*3.4*	*2.8*	*2.4*	*1.3*	*0.3*	*-2.4*	*-3.6*	*-14.2*	*47.1*
Mid-Year Adjustment	*(x)*	1				*1*	*2*	*3*	*4*	*5*	*6*	*7*	*8*	*9*	*10*	*11*	*12*	*13*
Discount Factor	*(x)*	5%				*0.952*	*0.907*	*0.864*	*0.823*	*0.784*	*0.746*	*0.711*	*0.677*	*0.645*	*0.614*	*0.585*	*0.557*	*0.530*
Discounted Pre-Tax Cash Flow	*($'000s)*		$259,346	-	-	**($61,905)**	**($45,497)**	**$77,599**	**$61,509**	**$62,281**	**$43,134**	**$36,377**	**$28,230**	**$23,410**	**$17,025**	**$14,811**	**$8,169**	**($5,796)**
Pre-Tax IRR	*(%)*		**48.2%**															
AFTER-TAX CASH FLOW																		
Pre-Tax Cash Flow	*($'000s)*		**$372,741**			($65,000)	($50,160)	$89,831	$74,764	$79,488	$57,804	$51,186	$41,709	$36,316	$27,732	$25,332	$14,670	($10,930)
Less: Corporate Income Taxes (25%)	*($'000s)*		**($95,004)**			-	($3,436)	($20,995)	($14,338)	($17,157)	($10,875)	($9,607)	($7,018)	($5,652)	($3,295)	($2,631)	-	-
After-Tax Cash Flow	*($'000s)*		**$277,737**	-	-	**($65,000)**	**($53,597)**	**$68,835**	**$60,426**	**$62,331**	**$46,929**	**$41,579**	**$34,691**	**$30,664**	**$24,437**	**$22,700**	**$14,670**	**($10,930)**
Adj. Cumulative After-Tax Cash Flow	*($'000s)*					*($65,000)*	*($118,597)*	*($49,762)*	*$10,665*	*$72,996*	*$119,925*	*$161,504*	*$196,195*	*$226,858*	*$251,295*	*$273,996*	*$288,666*	*$277,737*
Discounted Payback Calculation	*(years)*		**3.83**			*n/a*	*n/a*	*n/a*	*3.8*	*3.8*	*3.4*	*3.1*	*2.3*	*1.6*	*-0.3*	*-1.1*	*-7.7*	*38.4*
Discount Factor	*(x)*	5%				0.952	0.907	0.864	0.823	0.784	0.746	0.711	0.677	0.645	0.614	0.585	0.557	0.530
Discounted After-Tax Cash Flow	*($'000s)*		$185,957	-	-	**($61,905)**	**($48,614)**	**$59,462**	**$49,713**	**$48,838**	**$35,019**	**$29,549**	**$23,480**	**$19,766**	**$15,002**	**$13,272**	**$8,169**	**($5,796)**
After-Tax IRR	*(%)*		**36.5%**															

22.5 Sensitivity Analysis

A sensitivity analysis has been carried out, with the base case described above as a starting point, to assess the impact of changes in total pre-production (initial) capital expenditure ("CAPEX"), operating costs ("OPEX") and product prices ("PRICE") on the Project NPV @ 5 % and IRR. Impact of prices of gold, silver, lead and zinc were analyzed.

Each variable was examined one-at-a-time (price forecasts of the different concentrate products are varied together). An interval of ±20% with increments of 10% was used.

The pre-tax results of the sensitivity analysis, as shown in Figure 22.3 and Figure 22.4, indicate that, within the limits of accuracy of the cost estimates in this Report, the Project pre-tax viability does not seem significantly vulnerable to under-estimation of OPEX and CAPEX, taken one at-a-time. In Figure 22.3, the NPV is more sensitive to variations in Prices than OPEX and CAPEX, as shown by the steeper slope of the prices curve. As expected, the NPV is most sensitive to variations in price. The NPV remains positive in a price interval that can fall up to -20%.

Figure 22.3 – Pre-tax NPV5%: Sensitivity to CAPEX, OPEX and Prices (Phase 1 and Phase 2)

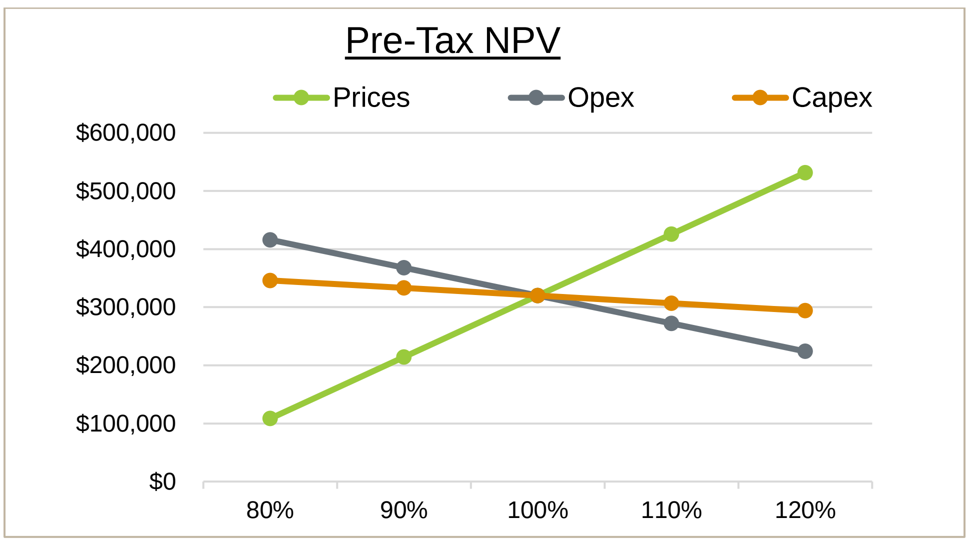
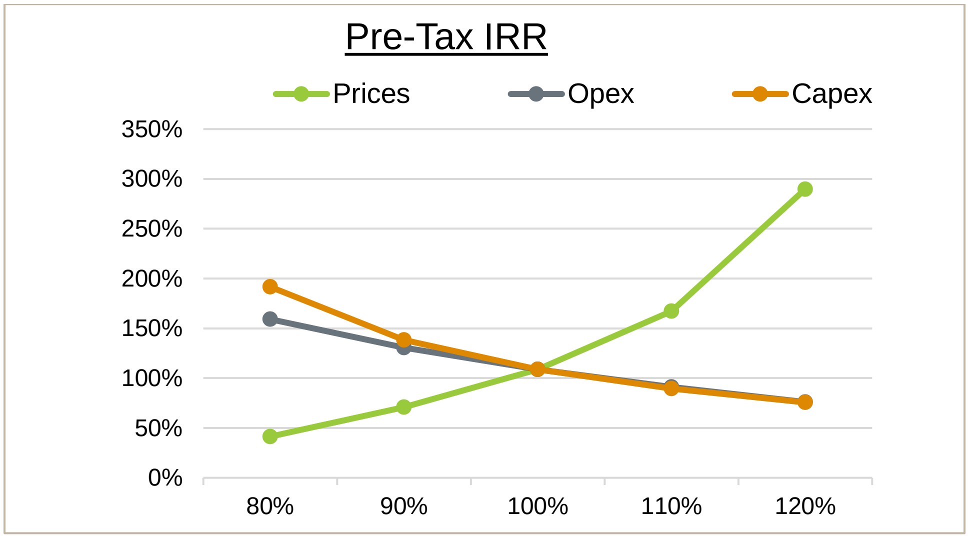


Figure 22.4 indicates that the IRR is more sensitive to variations in Prices than CAPEX and OPEX.

Figure 22.4 – Pre-tax IRR: Sensitivity to CAPEX, OPEX, and Prices (Phase 1 and Phase 2)



The same conclusions can be made from the after-tax results of the sensitivity analysis as shown in Figure 1.6 and Figure 1.7. Figure 1.6 indicates that the Project after-tax viability is mostly vulnerable to a price forecast reduction, while being less affected by the under-estimation of capital and operating costs.

Figure 22.5 – After-tax NPV5%: Sensitivity to CAPEX, OPEX, and Prices (Phase 1 and Phase 2)

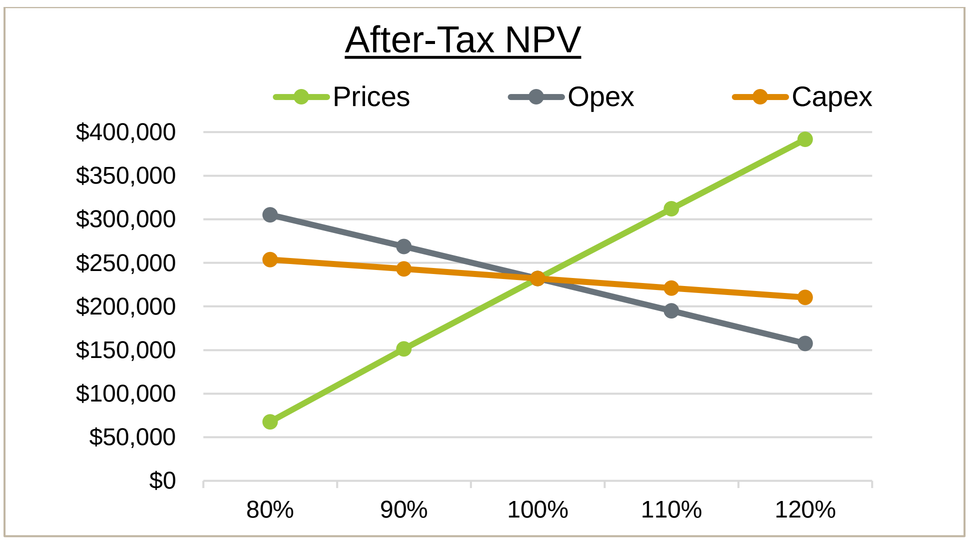
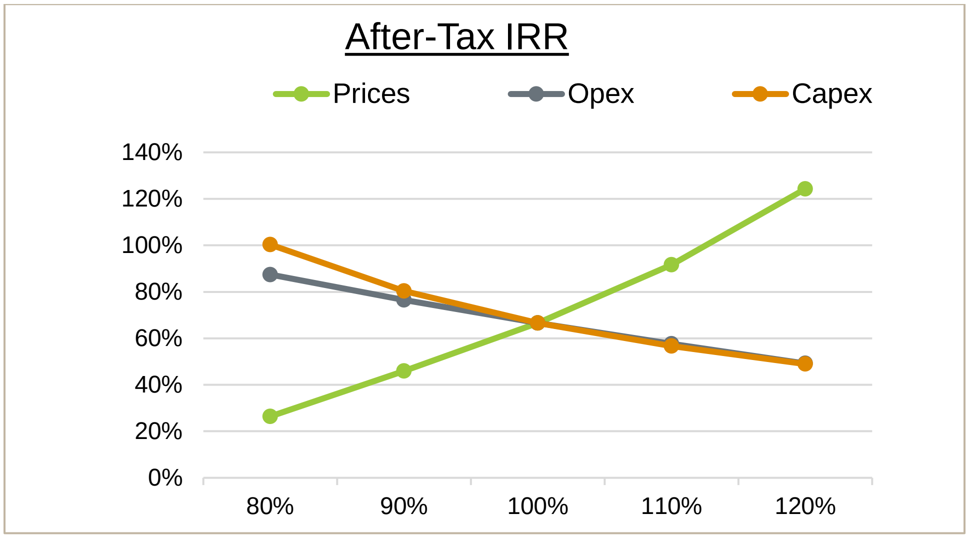


Figure 1.7 indicates that the IRR is more sensitive to variations in Prices than CAPEX and OPEX.

Figure 22.6 – After-tax IRR: Sensitivity to CAPEX, OPEX and Prices (Phase 1 and Phase 2)



23 ADJACENT PROPERTIES

The Author QPs have not been supplied with any information by Steppe Gold concerning any adjacent properties that Steppe may have interests in.

24 OTHER RELEVANT INFORMATION

24.1 Project Execution Schedule

A master schedule was developed for the ATO Project to sequence the main activities associated with the Project. The master schedule includes activities such as completion of the Feasibility Study phase, engineering, procurement, construction, commissioning and ramp-up. It is assumed that all applicable permits are obtained in a timely manner so as not to impact the schedule.

24.1.1 Schedule Assumptions

The master schedule was developed considering the following assumptions:

- **Feasibility Study Phase**: complete and Project go-ahead is approved.
- **Design**: based on preliminary vendor information. Once certified information is received, verification of the design will be done. No allowance is made for significant design changes.
- **Procurement**: duration of 12 weeks is assumed for the total procurement cycle for each package which includes:
  - Specifications and enquiry development; period (4 weeks);
  - Tender period (3 weeks);
  - Tender adjudication period (2 weeks);
  - Contract compilation period (3 weeks).
- **Fabrication, Manufacturing, and Delivery**: where current equipment vendor manufacturing durations are not available, manufacturing durations from similar DRA projects were utilised.
- **Construction**: durations are estimated based on similar recent DRA projects.
- **Commissioning**: durations are estimated based on similar recent DRA projects.

Table 24.1 and Figure 24.1 highlight key milestones and phases of Project development.

Table 24.1 – Key Project Milestones by Month

Milestone	Month
Project Execution Start	0
EP Consultant Appointed	0
Construction & Site Establishment Start	6
Plant Terrace Start	6
Primary Mill Order Placed	6
Process Building Construction Complete	16
Primary Mill Installed	23

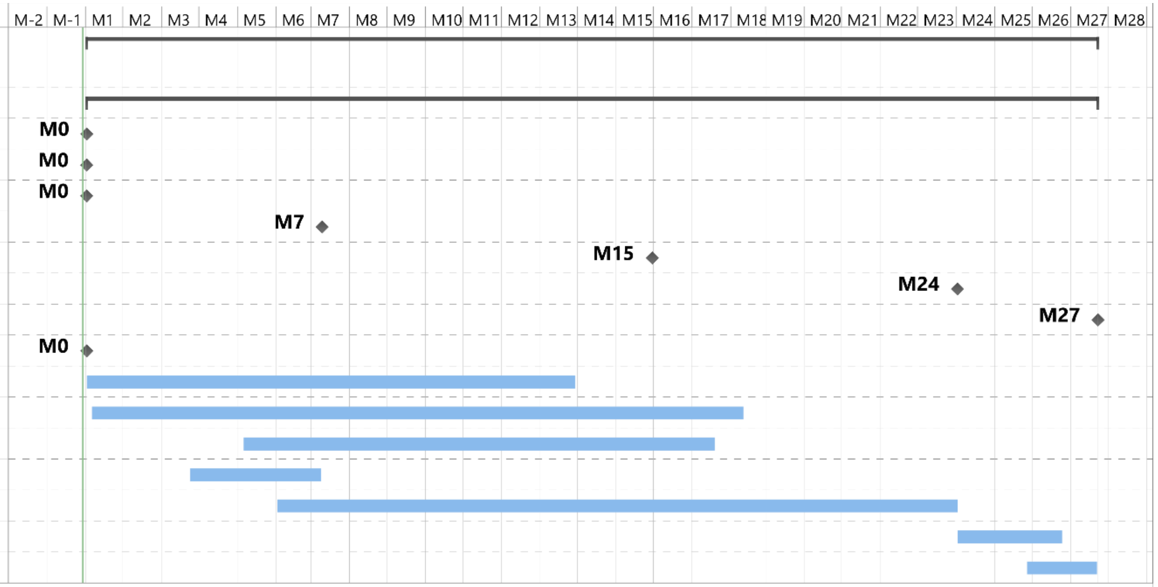
Milestone	Month
Overall C5 Commissioning Complete	25
Project Close-Out	27

24.1.2 Critical Path and Long Lead Items and Activities

With current available information, the Project critical path runs through the procurement of the primary mill and enclosing the process building before the start of winter. The installation of the primary mill and commissioning thereof will then complete the critical path.

Figure 24.1 – High Level Execution Schedule

ID	Task Name	Duration
1	**J3075 - Steppe Gold ATO FS - Execution Schedule_FINAL (28 Jul 2021)**	**116 w**
2	**Project Milestones**	**116 w**
3	Project Start	0 w
4	FEED/FINAL Stage Gate Approvals	0 w
5	Process Freeze	0 w
6	Plant Terrace Start	0 w
7	Process Building Complete	0 w
8	Primary Mill SMPP Installation Complete	0 w
9	Project Complete	0 w
10	EPCM Appointed	0 w
11	**Engineering & Design**	**56 w**
12	**Procurement**	**75 w**
13	**Fabrication & Delivery**	**54 w**
14	**Site Establishments**	**15 w**
15	**Construction**	**78 w**
16	**Commissioning**	**12 w**
17	**Close- Out**	**8 w**



24.2 Opportunities

There are numerous opportunities to further consider and/or develop in the next stage of the project lifecycle. Proposed opportunities and potential impacts are described in Table 24.2.

Table 24.2 – Proposed Project Opportunities

Opportunity	Potential Benefit to Project
Further optimise mining method with respect to associated mining equipment.	Reduce OPEX
Further optimise resource model via increased drilling	Increase LOM, reduce costs, increase revenue
Further optimise mine designs / dilution control via increased drilling	Increase production, increase LOM, reduce costs, increase revenue
Optimise construction and commissioning timeline with respect to cost and ore availability (mine plan)	Increase NPV
Optimise production ramp up with respect to cost and ore availability (mine plan)	Increase NPV
Optimise plant equipment selection and sizing via further variability test work. Further test work may improve confidence surrounding equipment sizing in comminution and filtration circuits.	Lower CAPEX and OPEX, reduce process risk
Optimise milling and crushing circuits in tandem	Lower CAPEX, improve maintainability and operability
Further optimise flotation circuit reagent regime	Increase grade and recovery, increase revenue
Further optimise plant layout and position	Reduce CAPEX, improve maintainability and operability
Investigate alternative equipment and fabricated items suppliers	Reduce CAPEX and/or delivery times

25 INTERPRETATIONS & CONCLUSIONS

25.1 Mineral Reserves

There are 22Mt of ore reserves, between the ATO transition and fresh ores. An opportunity exists to gain up to 1.5M of reserve inside the existing pit design at ATO by drilling exploration holes to bring the inferred ore to minimum indicated resource level.

There is also the potential to expand the current ATO open pit by performing additional exploration drilling on the inferred resources outside the pit shell, more particularly the eastern part of ATO-4 pipe.

As Mungu is a probable reserve, work would be required on the metallurgical side to process this ore.

Mungu is open at depth and only a small portion can be mined economically by open pit. Therefore, there is an opportunity for the Mungu Pit to be expanded to an underground mine. Further studies should be carried out to confirm the potential for an underground mine.

25.2 Mining Methods

This Technical Report presents a mineral reserve estimate and associated mine plan for an open pit operation with an LOM of approximately 13 years (including oxide). This includes 26.4 Mt of ore at an average grade of 1.86 g/t AuEq and an average stripping ratio of 2.62. The ore material is contained within two (2) major areas: ATO and Mungu. There are three (3) ore material types, namely: oxide, transition, and fresh. Material is mined to achieve leach pad and mill targets of 1.20 Mtpa and 2.20 Mtpa respectively, while reducing waste mining requirements.

The mine will operate seven (7) days a week, 24 hours a day (two 12-hour shifts per day) for a total of 330 operating days per year after adjusting for adverse weather as well as holidays. The mine haulage will be performed by a Contractor using 32-tonne trucks.

25.3 Metallurgical Testing, Mineral Processing & Recovery

A comprehensive metallurgical testwork program was conducted on representative samples from the ATO deposit. The results from the program were used to define and optimize the process flowsheet for the economic extraction of lead, zinc and gold contained in the ATO ores. The flowsheet developed is considered a conventional sequential flotation flowsheet

It should also be noted that the design criteria was such that cyanide was not used in the new flowsheet.

For a plant of this size, the OPEX reported is reasonable.

The potential for additional recovery (5% to 10%) through increases in CAPEX and OPEX was investigated, however it was determined that this increase would not result in the desired recovery improvement.

25.4 Infrastructure

The Phase 1 site layout has been expanded to accommodate the new deposit (Mungu) and process building. The Mungu ore and waste pits are located just north of the Phase 1 ATO deposits, and the new Phase 2 process plant just southwest of the Phase 1 leach pad.

The site consists of several process buildings at the process plant, a worker camp, explosives storage, water supply, and TSF. New roads would provide access to the new pits and buildings.

The power demand of the site was assumed as peak load of 15 MW and average load of 12.5 MW. A hybrid solution Diesel-RES power plant (30 MW solar PV, 20 MW diesel, 4 MW/4 MWh BESS) was demonstrated to be the optimal low-cost solution for the Project.

The site will be supplied at 11 kV, 3 phase, 50 Hz from a power plant installed in the vicinity of the site. The power plant will consist of eight (8) diesel generators each using LFO (diesel) fuel, in an N+2 configuration (6 in operation, one in stand-by, one maintenance or repair).

The TSF will be located in a south-east facing valley approximately 2 km south-east of the pit. It will be a high-density polyethylene (HDPE) - lined cross-valley storage facility formed by multi-zoned earth fill embankment, encompassing a total footprint area (including basin area) of approximately 47 ha for Phase 1, and increasing to 112 ha for the final TSF.

25.5 Environmental Considerations and Permitting

Steppe Gold has conducted stakeholder and community participatory regular/routine environmental monitoring program at ATO gold project site and surrounding areas, and reporting to relevant authorities and local communities addressing the monitoring and control impacts on air, water, land/soil and biodiversity.

The General Environmental Impact Assessment (GEIA) was completed and approved by Ministry of Environment and Tourism of Mongolia. The environmental and social impacts are summarised in the report, and include changes to topography from mining operations, impacts on vegetation from mine clearing, impacts on fauna from land clearing, surface water hydrology impacts from interrupted natural drainage and soil and water contamination from mine development.

Steppe Gold has conducted water resource studies from 2017 to 2019 and received water resource statements from the relevant authorities and received land use permits for mining, construction, other infrastructures sites from local authorities.

The mine minerals waste handling plan has been developed to ensure that the management of mining activities and the implementation of environmental and social management plans and mine closure at the ATO will be conducted according to best practice methodologies to eliminate the potential for contamination.

The management of the Project's significant environmental and social aspects and impacts is achieved through a suite of Management Plans that have been developed and maintained, such as Air Quality Management Plan and Water Resources Management Plan.

25.6 Economics

At the assumed base case metal prices, key metrics of the Technical Report for the Project are summarised in the following tables, for both Phases 1 & 2 combined and for Phase 2 only.

Table 25.1 Project Financial Results Base Case (Phase 1 + Phase 2)

Financial Results	Unit	Pre-Tax	After-Tax
NPV @ 5%	M USD	319.97	232.08
NPV @ 8%	M USD	268.60	191.09
NPV @ 10%	M USD	239.97	168.36
IRR	%	108.8	66.6
Payback Period	Year	2.5	3.0

Table 25.2 Project Financial Results (Phase 2)

Financial Results	Unit	Pre-Tax	After-Tax
NPV @ 5%	M USD	259.35	185.96
NPV @ 8%	M USD	209.68	146.12
NPV @ 10%	M USD	182.20	124.21
IRR	%	48.2	36.5
Payback Period	Year	3.4	3.8

26 RECOMMENDATIONS

26.1 Mining & Reserves

Additional exploration drilling at ATO is recommended to bring the inferred resources to minimum indicated level. This has the potential to increase reserves up to 1.5Mt within the existing pit envelope, and expand the pit limit boundary by drilling inferred resources outside the current pit design. The best potential option to expand the pit will be to drill the inferred resource of the East end of ATO-4 pipe.

26.2 Testwork and Recovery

As overall gold recovery in Phase 2 is relatively low, there is an opportunity to perform additional testwork aimed at improving overall gold recovery.

It should also be investigated how to improve recovery and/or payability on the Project.

Specifically, recommendations with respect to the fresh and transitional ore types include:

- Additional test work is conducted to better understand the fresh and transitional resource and the development of a geometallurgical model including base metals and gold recovery relationships for the lead, zinc and pyrite concentrates produced, if they exist.
- Geo-metallurgical work should be completed to better understand the lithology and mineralogy of the ATO deposit particularly with respect to the proportion of ore which will be processed containing "high dolomite" material which produced variable flotation results.
- Further testwork targeting optimized gold and base metals recoveries through improved flotation performance or cyanidation of the flotation tailings stream.
- Production of sufficient separate lead, zinc and pyrite concentrates to be used in discussions with potential buyers.

26.3 Infrastructure

As no geotechnical information was available at the time of developing the design, a field and laboratory investigation program will need to be carried out (as part of the next project phase) to characterise the ground condition for foundation design and any borrow materials for any of the facilities presented in this Report. These results will confirm the assumptions made, or determine if changes to the design need to be made. This program will include geophysics, drilling and test pitting in the designated area, as well as taking samples for geotechnical laboratory testing.

A geochemical assessment of the mine waste needs to be undertaken to determine the suitability of the material for use for TSF embankment construction.

Phase 2 will require make-up water of approximately 4,800 m^3 per day. The source of this water should be confirmed, and that is has sufficient capacity for the project.

27 REFERENCES

Altan Tsagaan Ovoo Project (ATO) – 2021 Mineral Resources Technical Report (Amended NI 43-101), Project # GR2104", with an effective date 30th March, 2021 and report amended date 9th June, 2021, GeoRes.

CIM (The Canadian Institute of Mining, Metallurgy and Petroleum), 10th May 2014. CIM Definition Standards – for Mineral Resources and Mineral Reserves. Prepared for the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council on 10th May 2014. These definitions are incorporated into the NI 43-101 by reference.

JORC, 2012. Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (The JORC Code, 2012 Edition). Prepared by the Joint Committee of the Australasian Institute of Mining and Metallurgy, Australasian Institute of Geoscientists and Minerals Council of Australia (JORC).

NI43-101 Technical Report of Altan Tsagaan Ovoo Gold Project, Tsagaan Ovoo, Dornod, Mongolia, effective date September 6, 2017 and issued date October 4, 2017, GSTATS CONSULTING.

Steppe Gold Ltd., 24 February 2021. Steppe Gold doubles the ATO Gold Mine Resource to 2.45 Moz gold equivalent. Press release through Newsfile Corp., release 75306.

Techno-Economic Feasibility Study (TEFS) for mining the Altan Tsagaan Ovoo (ATO) gold-polymetal hard rock deposit located in the territory of Tsagaan Ovoo soum, Dornod province, Mongolia, Capacity - 2,000,000 tonnes/year, "Centerra Gold Mongolia" LLC, 2012, prepared by Glogex LLC, Geology and Mining Consulting Services in Mongolia, Draft Revision, translated from Mongolian to English.

27.1 Mining

Vdovin, Viktor. Geotechnical Slope Parameters Assessment of ATO Deposit. Report in Mongolian prepared by Centerra Gold Inc., Technical Development Group, 17 October 2011

Vdovin, Viktor, 17 September 2012. Update on the Geotechnical Slope Parameters Assessment of ATO Deposit. Report in Mongolian prepared by Centerra Gold Inc., Technical Development Group.

Wood, Matthew (Executive Chairman), 27 August 2020. Geological Consultant engagement letter. Emailed letter to GeoRes from Steppe Gold Ltd.

27.2 Process

Base Metallurgical Laboratories (BML), Feasibility Level Metallurgical Testing of the ATO Project, Project BL656 – Draft Report, August 30, 2021

T.Di Feo, L. Kormos, D. Fragomeni, M. Hoffman, S. Wojtowicz and Y. Boudreau, Xstrata Process Support, Mineralogical and Metallurgical Test Program for Centerra Gold Inc., Project 4010824.00, Final Report, June 30, 2011.

Xstrata Process Support, An Investigation into The Grindability Characteristics of Three Samples from Centerra Gold, (ATO-02, ATO-03, and ATO-05 Samples) Project 12076-006, Final Report, March 18, 2011

Xstrata Process Support, An Investigation into The Grindability Characteristics of Three Samples from Centerra Gold, (Master, Pipe 2, Pipe 4 Composites) Project 12076-014, Final Report, October 26, 2011

Xstrata Process Support, Centerra Pilot Plant, Flotation Mini Pilot Plant Technology (MPP), Centerra Gold Inc., Project 4011907.00, Final Report, September 07, 2012

Xstrata Process Support, Centerra Gold Inc., Altan Tsagaan Ovoo (“ATO”) Pilot Plant, ATO Deposit Samples for the Mini Pilot Plant Technology (MPP), Centerra Gold Inc., Project 4011907.00, Final Report, November 09, 2012

Xstrata Process Support, Mineralogical and Metallurgical Test, Program - Phase 2, Master Composite, Pipe 2, Pipe 4 and Variability Samples 1 to 4 based on Optimized Results, Project 4011903.00, Final Report, November 21, 2012

27.3 Tailings Storage Facility

Australian National Committee on Large Dams (ANCOLD), “Guidelines on Tailings Dams, Planning, Design, Construction, Operation and Closure”, Rev. 1, July 2019.

Australian National Committee on Large Dams (ANCOLD), “Guidelines on the Consequence Categories for Dams”, October 2012.

Australian National Committee on Large Dams (ANCOLD), “Guidelines for Design of Dams and Appurtenant Structures for Earthquake”, May 2019.

“Global Industry Standard on Tailings Management”, August 2020.

27.4 Marketing

CRU Consulting, August 12, 2021, Steppe Gold’s ATO Phase 2 Expansion Project: Lead and Zinc Market Overviews, CRU Reference # ST2260-21

Nickels, Luke, July 26, 2021, Commodity Quarterly: Zinc Q2 2021, S&P Global Market Intelligence

27.5 Environment

Annual EMPs and reports (STEPPE GOLD-2018, 2019, 2020)

Annual Environmental Monitoring reports (EHSM LLC- 2018, 2019, Ulzii Environmental -2020)

Conceptual Mine Closure Plan of Altan Tsagaan Ovoo gold project (Polaris Engineering Consulting LLC-2019)

Detailed Environmental Impacts Assessment report of Altan Tsagaan Ovoo project's Chemicals usage (EMB-2019)

Detailed Environmental Impacts Assessment report of Altan Tsagaan Ovoo project's mining and processing (MAKE GREEN-2019)

Feasibility Study Report of Altan Tsgaan Ovoo mining project (MIDAS MINING-2019)

Feasibility Study Report of Altan Tsgaan Ovoo processing project (MIDAS MINING-2019)

Water resource and hydrogeology study report of Altan Tsagaan Ovoo gold project site (Water Management LLC-2019)

28 ABBREVIATIONS

The following abbreviations may be used in this Report.

Abbreviation	Meaning or Units
'	Feet
"	Inch
$	Dollar Sign
$/m²	Dollar per Square Metre
$/m³	Dollar per Cubic Metre
$/t	Dollar per Tonne
%	Percent
% w/w	Percent Solid by Weight
¢/kWh	Cent per Kilowatt hour
°	Degree
°C	Degree Celsius
2D	Two Dimensions
3D	Three Dimensions
µm	Microns, Micrometre
µg	Microgram(s)
µg/m³	Micrograms per cubic metre
µPa	Micropascal
ADR	Adsorption, Desorption, Recovery
Ag	Silver
AP	Acid Potential
ARD	Acid Rock Drainage
As	Arsenic
AISC	All-In-Sustaining Costs
As	Arsenic
ASL	Above Sea Level
Au	Gold
AuEq	Equivalent Gold
AWG	American Wire Gauge
Az	Azimuth
bcm	Bank Cubic Metre

Abbreviation	Meaning or Units
BFA	Bench Face Angle
Bi	Bismuth
BML	Base Metal Laboratories
BoQ	Bill of Quantities
BSG	Bulk Specify Gravity
BSTP	Biological Sewerage Treatment Plant
BTU	British Thermal Unit
BWI	Bond Ball Mill Work Index
Ca	Calcium
$CaCO_3$	Calcium Carbonate
CAD	Canadian Dollar
CAGR	Compound Annual Growth Rate
CAPEX	Capital Expenditure or Capital Cost Estimate
Cd	Cadmium
Ce	Cesium
CEC	Cation Exchange Capacity
cfm	Cubic Feet per Minute
CFR	Cost and Freight
CIC	Carbon-in-Column
CIF	Cost Insurance and Freight
CIL	Carbon-in-Leach
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIP	Carbon-in-Pulp
cm	Centimetre
CM	Construction Management
Co	Cobalt
CO	Carbon Monoxide
CO_2	Carbon Dioxide
COG	Cut-Off Grade
COV	Coefficient of Variation
Cu	Copper
CWI	Crusher Work Index
dB	Decibel

Abbreviation	Meaning or Units
dBA	Decibel with an A Filter
DEM	Digital Elevation Model
DCF	Discounted Cash Flow
DRA	DRA Global Limited
DWI	Drop Weight Index
DWT	Drop Weight Test
DXF	Drawing Interchange Format
E	East
EA	Environmental Assessment
EHS	Environmental, Health and Safety
EIA	Environmental Impact Assessment
ELVs	Emission Limit Values
EMP	Environmental Management Plan
EP	Engineering and Procurement
EPA	Environmental Protection Agency
EPCM	Engineering, Procurement and Construction Management
ER	Electrical Room
Fe	Iron
FEED	Front End Engineering Design
FEL	Front End Loader
FeS_2 or Py	Pyrite
FOB	Free on Board
FS	Feasibility Study
ft	Feet
g	Gram
G&A	General and Administration
g/L	Grams per Litre
g/t	Grams per Tonne
gal	Gallons
GCW	Gross Combined Weight
GDP	Gross Domestic Product
GEIA	General Environmental Impact Assessment
GPS	Global Positioning System

Abbreviation	Meaning or Units
H	Horizontal
h	Hour
h/d	Hours per Day
h/a	Hour per Annum
H_2	Hydrogen
H_2O	Water
H_2SO_4	Sulphuric Acid
ha	Hectare
HCl	Hydrochloric Acid
HDPE	High Density PolyEthylene
HF	Hydrofluoric Acid
HFO	Heavy Fuel Oil
Hg	Mercury
HVAC	Heating Ventilation and Air Conditioning
HW	Hanging Wall
Hz	Hertz
I/O	Input / Output
ICMC	International Cyanide Management Code
IEC	International Electro-Technical Commission
in	Inches
IRR	Internal Rate of Return
ISO	International Standards Organisation
IT	Information Technology
JORC	Joint Ore Reserves Committee
JV	Joint Venture
K	Kelvin
KCl	Potassium Chloride
kg	Kilogram
kg/L	Kilogram per Litre
kg/t	Kilogram per Tonne
kL	Kilolitre
km	Kilometre
km/h	Kilometre per Hour

Abbreviation	Meaning or Units
koz	Kilo ounce (troy)
kPa	Kilopascal
kt	Kilotonne
kV	Kilovolt
kVA	Kilovolt Ampere
kW	Kilowatt
kWh	Kilowatt-Hour
kWh/t	Kilowatt-Hour per Tonne
L	Litre
L/h	Litre per Hour
lbs	Pounds
LCT	Locked Cycle Test
LFO	Light Fuel Oil
LG	Low Grade
Li	Lithium
LOM	Life of Mine
Ltd	Limited
LV	Low Voltage
m	Metre
m/h	Metre per Hour
m/s	Metre per Second
m^2	Square Metre
m^3	Cubic Metre
m^3/d	Cubic Metre per Day
m^3/h	Cubic Metre per Hour
m^3/y	Cubic Metre per Year
mA	Milliampere
Masl	Meters Above Sea Level
MCC	Motor Control Centre
mg	Milligram(s)
Mg	Magnesium
mg/kg	Miligram per Kilogram
mg/L	Milligram per Litre

Abbreviation	Meaning or Units
mg/m²/day	Milligram per Square Metre per Day
min	Minute
min/shift	Minute per Shift
mL	Millilitre
ML	Metal Leaching
mm	Millimetre
mm/d	Millimetre per Day
Mm³	Million Cubic Metres
MMET	Ministry of Environment and Tourism of Mongolia
MML	Main Mongolian Linament
Mn	Manganese
Mt	Million Tonnes
Mtpa or Mt/a	Million Tonne per Annum
M USD	Million United States Dollars
MV	Medium Voltage
MVA	Mega Volt-Ampere
MW	Megawatts
MWh/d	Megawatt Hour per Day
My	Million Years
N	Nitrogen
N	North
NAAQS	National Air Quality Standards
NaCN	Sodium Cyanide
NAG	Non-Acid Generating
NaHS	Sodium Hydrosulfide
NE	Northeast
NFPA	National Fire Protection Association
NGO	Non-Governmental Organisation
NGR	Neutral Grounding Resistor
Ni	Nickel
NI 43-101	National Instrument 43-101
Nm³/h	Normal Cubic Metre per Hour
NNE	North - Northeast

Abbreviation	Meaning or Units
NNP	Net Neutralisation Potential
NP	Neutralisation Potential
NPV	Net Present Value
NSR	Net Smelter Return
NTP	Normal Temperature and Pressure
NTS	National Topographic System
NW	North West
O/F	Overflow
OPEX	Operating expenditure / Operating cost estimate
oz	Troy ounce (31.1034768 grams)
p	Pressure
P&ID	Piping and Instrumentation Diagram
Pa	Pascal
PAG	Potential Acid Generating
Pb	Lead
Pd	Palladium
PF	Power Factor
PFC	Power Factor Correction
PGE	Platinum-Group Element
Ph	Phase (electrical)
pH	Potential of Hydrogen
PLC	Programmable Logic Controller
ppm	Parts per Million
psi	Pounds per Square Inch
Pt	Platinum
P-T	Pressure-Temperature
Py	Pyrite
PVC	Polyvinyl Chloride
QA/QC	Quality Assurance / Quality Control
QP	Qualified Person
RF	Revenue Factor
RFQ	Request for Quotation
ROM	Run of Mine

Abbreviation	Meaning or Units
rpm	Revolutions per Minute
S	South
S	Sulphur
S/R or SR	Stripping Ratio
SABC	SAG and Ball (Milling) Circuit
SAG	Semi Autogenous Grinding
Sb	Antimony
scfm	Standard Cubic Feet per Minute
SCIM	Squirrel Cage Induction Motors
SE	South East
s	Second
SG	Specific Gravity
SO_2	Sulphur Dioxide
SoW	Scope of Work
SPI	SAG Power Index
SW	South West
t	Tonnes
t/d or tpd	Tonne per Day
t/h or tph	Tonne per Hour
t/h/m	Tonne per Hour per Metre
t/h/m²	Tonne per Hour per Square Metre
t/m or tpm	Tonne per Month
t/m²	Tonne per Square Metre
t/m³	Tonne per Cubic Metre
Ta	Tantalum
ton	Short Ton
tonne or t	Metric Tonne
ToR	Terms of Reference
TOS	Trade-Off Study
tpa or t/a	Tonne per Annum
TSF	Tailings Storage Facility
TSP	Total Suspended Particulates
TSS	Total Suspended Solids

Abbreviation	Meaning or Units
TSX	Toronto Stock Exchange
U	Uranium
U/F	Under Flow
U/S	Undersize
ULC	Underwriters Laboratories of Canada
US, USA	United States (of America)
USD, $ USD, US$	United States Dollar
USGPM	US Gallons per Minute
USGS	United States Geological Survey
UTM	Universal Transverse Mercator
V	Vertical
V	Volt
VFD	Variable Frequency Drive
VLF	Very Low Frequency
W	Watt
W	West
w/o	Waste / Ore
X	X Coordinate (E-W)
XPS	Xstrata Process Support
XRD	X-Ray Diffraction
XRF	X-Ray Fluorescence
y	Year
Y	Y coordinate (N-S)
Z	Z coordinate (depth or elevation)
Zn	Zinc

29 CERTIFICATE OF QUALIFIED PERSON

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CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*NI 43-101 Technical Report – Feasibility Study for the Altan Tsagaan Ovoo (ATO) Phase 2 Expansion Project, Mongolia*" with an effective date of October 27, 2021 and issued on November 30, 2021 (the "Technical Report") prepared for Steppe Gold LLC ("Steppe Gold" or the "Company").

I, *Ulziibayar Dagdandorj, MAusIMM* do hereby certify:

1. I am Principal Environmental Consultant with Ulzii Environmental (Mongolia), LLC., Orient Plaza office 2nd floor, Room 206 Sukhbaatar District-1 Chagdarjav Street-9 Ulaanbaatar 14210, Mongolia.
2. I graduated with Bachelor's of Science degree in Forestry from the National University of Mongolia in 2002 and a Master's of Business Administration degree in CITI University of Mongolia in 2021.
3. I am a registered member of Australasian Institute of Mining and Metallurgy (MAusIMM) (#335969) since 2020, Head of Environmental Professionals Association of Mongolia.
4. My relevant work experience includes: Brief summary of relevant experience.
    - Experienced Environmental Specialist with 18 years of demonstrated history of working in the Environmental Consulting sector. Skilled in ISO 14001, Environmental Impact Assessment, Environmental Compliance, Natural Resource Management, and Sustainability. Specializing on mine rehabilitation, remediation and closure.
    - Leads and manage the environmental consulting services as mine closure planning and its performances, mine site routine environmental monitoring, environmental and social impact assessments for mining, processing and infrastructures development projects in Mongolia.
    - Participation and author of several NI 43-101 Technical Reports.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
6. I am independent of the Company applying all the tests in Section 1.5 of NI 43-101.
7. I am responsible for the preparation of Section 20 of the Technical Report.

8. I personally visited property that is the subject to the Technical Report to conduct a site visit and an initial reconnaissance of the ATO Project site from June 25 to June 27, 2021.
9. I have had prior involvement with the property that is the subject of the Technical Report.
   - *"Altan Tsagaan Ovoo Project (ATO), 2021 Mineral Resources, Technical Report (Amended NI 43-101)"*, prepared by GeoRes, with an effective date March 30, 2021, issued on March 30, 2021 and amended on June 9, 2021
   - Unplanned mine closure plan for asset retirements obligations (ARO) estimate report - *Altan Tsagaan Ovoo Gold Project, Tsagaan Ovoo, Dornod, Mongolia"* prepared by Polaris Engineering Consulting LLC with an effective date September 30th, 2021 and issued on October 5th, 2021 prepared for Steppe Gold LLC".
   - Biodiversity offset study report - *Altan Tsagaan Ovoo Gold Project, Tsagaan Ovoo, Dornod, Mongolia"* prepared by Ulzii Environmental LLC with an effective date July 30th, 2021 and issued on August 9th, 2021 prepared for Steppe Gold LLC".
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible is not misleading.

Dated this 30th day of November 2021

"Original Signed and sealed"

Ulziibayar Dagdandorj MAusIMM #335969
Principal Environmental Consultant, Ulzii Environmental (Mongolia), LLC.

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CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*NI 43-101 Technical Report – Feasibility Study for the Altan Tsagaan Ovoo (ATO) Phase 2 Expansion Project, Mongolia*" with an effective date of October 27, 2021 and issued on November 30, 2021 (the "Technical Report") prepared for Steppe Gold LLC ("Steppe Gold" or the "Company").

I, *Tim Fletcher, P. Eng.*, Toronto, Ontario, do hereby certify that:

1. I am Senior Project Manager, DRA Global Limited with an office at 20 Queen Street West, 29th Floor, Toronto, Ontario, Canada M5H 3R3;
2. I am a graduate from University of Toronto, with a B.A.Sc. in Mechanical Engineering in 1992 and a M.A.Sc. in Metallurgical Engineering in 1995;
3. I am a Professional Engineer licensed by Professional Engineers Ontario (Membership Number 90451964).
4. I have worked as an Engineer in the Mining and Metals industry continuously since my graduation from university.
5. My relevant work experience includes:
    - Over 25 years of metallurgical project development experience, for numerous commodities and clients, in the capacities of Mechanical Engineer, Project Engineer, Engineering Manager, and Project Manager;
    - Management of numerous studies and projects of varying complexity, involving multi-disciplinary engineering teams for projects in gold, base metals, and other commodities;
    - Participant and author of various NI 43-101 Technical Reports.
6. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.

7. I am independent of the Company applying all the tests in Section 1.5 of NI 43-101.
8. I am responsible for the preparation of Sections 2 to 6, 23, 24, and portions of Sections 1, 18, 21, and 25 to 27, and for overall report compilation.
9. I did not visit the property on that is the subject to the Technical Report.
10. I have had prior involvement with the property that is the subject of the Technical Report.
    - *Reviewed "NI 43-101 Technical Report - Altan Tsagaan Ovoo Gold Project, Tsagaan Ovoo, Dornod, Mongolia*" prepared by GSTATS Consulting with an effective date September 6th, 2017 and issued on October 4th, 2017 prepared for Steppe Gold LLC".
11. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
12. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 30th day of November 2021

"Original Signed and sealed" Tim Fletcher

Tim Fletcher, P. Eng
Senior Project Manager
DRA Global Limited

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CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*NI 43-101 Technical Report – Feasibility Study for the Altan Tsagaan Ovoo (ATO) Phase 2 Expansion Project, Mongolia*" with an effective date of October 27, 2021 and issued on November 30, 2021 (the "Technical Report") prepared for Steppe Gold LLC ("Steppe Gold" or the "Company").

I, David Frost, FAusIMM, B. Met Eng, Toronto, Ontario, do hereby certify that:

1. I am Vice President Process Engineering, DRA Global Limited with an office at 20 Queen Street West, 29th Floor, Toronto, Ontario, Canada M5H 3R3;
2. I graduated from the Royal Melbourne Institute of Technology (RMIT), Melbourne, Australia with a Bachelor of Metallurgical Engineering in Metallurgy in 1993;
3. I am a registered Fellow Member of the Australian Institute of Mining and Metallurgy (FAusIMM) membership #110899.
4. I have worked as a Metallurgist and Process Engineer in various capacities since my graduation from university in 1993.
5. My relevant work experience includes:
   - 30 years of experience, 15 years in process plant operations including the operation of complex flotation flowsheets and 15 years in process plant flowsheet design;
   - Supervision, operations, maintenance and designs for heap leach operation for several projects in Europe, South America, Asia, and Australia;
   - Polymetallic flotation testwork supervision, management and flowsheet design for several projects; and
   - Participant and author of several NI 43-101 Technical Reports inclusive of polymetallic flowsheets.
6. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.

7. I am independent of the Company applying all the tests in Section 1.5 of NI 43-101.

8. I am responsible for the preparation of Sections 13, 17, 19, and portions of Sections 1, 18, 21, and 25 to 27.

9. I visited the Altan Tsagaan Ovoo Project (ATO) location in August 2018 and personally viewed the drill cores both at the site and in storage in Ulaanbataar which were used for metallurgical test work.

10. I have had prior involvement with the property that is the subject of the Technical Report.
    - QP for "*Altan Tsagaan Ovoo Project (ATO), 2021 Mineral Resources, Technical Report (Amended NI 43-101)*", prepared by GeoRes, with an effective date March 30, 2021, issued on March 30, 2021 and amended on June 9, 2021
    - *Reviewed "NI 43-101 Technical Report - Altan Tsagaan Ovoo Gold Project, Tsagaan Ovoo, Dornod, Mongolia*" prepared by GSTATS Consulting with an effective date September 6th, 2017 and issued on October 4th, 2017 prepared for Steppe Gold LLC".

11. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.

12. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 30th day of November 2021

Original Signed and sealed" David Frost

David Frost, FAusIMM, B. Met Eng
Vice President – Process Engineering
DRA Global Limited

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CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*NI 43-101 Technical Report – Feasibility Study for the Altan Tsagaan Ovoo (ATO) Phase 2 Expansion Project, Mongolia*" with an effective date of October 27, 2021 and issued on November 30, 2021 (the "Technical Report") prepared for Steppe Gold LLC ("Steppe Gold" or the "Company").

I, *Daniel M. Gagnon, P. Eng.,* do hereby certify:

1. I am Vice President Mining, Geology and Met-Chem Operations, with DRA Global Limited located at 555 René Lévesque West, 6th Floor, Montreal, Quebec Canada H2Z 1B1.
2. I am a graduate of École Polytechnique de Montréal, Montreal, Quebec, Canada in 1995 with a bachelor degree in Mining Engineering.
3. I am registered as a Professional Engineer in the Province of Quebec (Reg. #118521).
4. I have worked as a Mining Engineer for a total of 25 years continuously since my graduation.
5. My relevant work experience includes:
    - Design, scheduling, economic analysis cost estimation and Mineral Reserve estimation for several open pit studies similar to Steppe Gold in Canada, the US, South America, West Africa and Morocco.
    - Technical assistance in mine design and scheduling for mine operations in Canada, the US and Morocco.
    - Participant and author of several NI 43-101 Technical Reports.
6. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
7. I am independent of the Company applying all the tests in Section 1.5 of NI 43-101.
8. I am responsible for the preparation of Section 22, and for portions of Sections 1 and 25 to 27.

9. I did not visit the property that is the subject to the Technical Report.

10. I have had no prior involvement with the property that is the subject of the Technical Report.

11. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.

12. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 30th day of November 2021

"Original Signed and sealed" Daniel Gagnon

Daniel M. Gagnon, P. Eng.
VP Mining, Geology and Met-Chem Operations
DRA Global Limited

CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*NI 43-101 Technical Report – Feasibility Study for the Altan Tsagaan Ovoo (ATO) Phase 2 Expansion Project, Mongolia*" with an effective date of October 27, 2021 and issued on November 30, 2021 (the "Technical Report") prepared for Steppe Gold LLC ("Steppe Gold" or the "Company").

I, *Dan V. Michaelsen, FAusIMM (CP) Env*, do hereby certify:

1. I am Principal Adviser Sustainability with Beaumont Consulting in Ulaanbaatar, Mongolia, and Senior Associate of Ulzii Environmental (Mongolia), LLC., Orient Plaza office 2nd floor, Room 206 Sukhbaatar District-1 Chagdarjav Street-9 Ulaanbaatar 14210, Mongolia.
2. I graduated with an MBA from University of New England in 2007, a Master of Environmental and Business Management from University of Newcastle, and a BSc (Agriculture) Hons. from University of Western Australia.
3. I am a member of the Australasian Institute of Mining and Metallurgy (MAusIMM) where I am a Fellow and Chartered Professional (Env) (#20099) since 1996.
4. I have more than 40 years' experience in Environmental and Sustainability Management in the international mining industry in Australia, Indonesia, Papua New Guinea, Mongolia and Ghana for recognized multinational mining companies.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
6. I am independent of the Company applying all the tests in Section 1.5 of NI 43-101.
7. I am responsible for the preparation of Section 20. I am also responsible for portions of Sections 1 and 25 to 27 of the Technical Report.
8. I personally did not visit the property that is the subject to the Technical Report. Ulzii Environmental LLC, Principal Environmental Consultant, Mr. Ulziibayar Dagdandorj MAusIMM (#335969) conducted a site visit and an initial reconnaissance of the ATO Project site from June 25 to June 27, 2021.
9. I have had no prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible

have been prepared in compliance with NI 43-101.

11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible is not misleading.

Dated this 30th day of November 2021

"Original Signed and sealed" Dan Michaelsen

Dan V. Michaelsen, FAusIMM (CP) Env
Senior Associate, Ulzii Environmental (Mongolia), LLC.

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CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*NI 43-101 Technical Report – Feasibility Study for the Altan Tsagaan Ovoo (ATO) Phase 2 Expansion Project, Mongolia*" with an effective date of October 27, 2021, and issued on November 30, 2021 (the "Technical Report") prepared for Steppe Gold LLC ("Steppe Gold" or the "Company").

I, David Morgan, MIE Aust CPEng, do hereby certify:

1. I am the Managing Director and Project Director of Knight Piésold Pty Ltd located at Level 1, 184 Adelaide Terrace East Perth, Western Australia 6004.
2. I am a graduate of University of Manchester, (BSc, Civil Engineering, 1980), and the University of Southampton (MSc, Irrigation Engineering, 1981).
3. I am a registered member in good standing of the Australasian Institute of Mining and Metallurgy (Australasia, 202216) and a Chartered Professional Engineer and member of the Institution of Engineers Australia (Australia, 974219).
4. I am a Chartered Civil Engineer with over 39 years' experience in the civil and tailings management sector.
5. I have worked as a Project Manager, Senior Engineer or Project Director continuously since my graduation from university. I have gained relevant experience on projects similar to the Altan Tsaagan Ovoo project, including:
   - Work on gold mining operations and projects located throughout Asia and West Africa.
   - I have supervised and contributed to many tailings management engineering studies for different projects at various stages of development. Hands-on experience for gold in Ivory Coast, Mali, Burkina Faso, Mongolia, Australia and Canada;
   - Design, supervision and implementation of construction programs;
   - Review, audits of tailings management systems; and
   - Participation in the preparation of parts of NI 43-101 compliant Technical Reports.
6. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.

7. I am independent of the Company applying all the tests in Section 1.5 of NI 43-101.
8. I am responsible for the relevant portions of Sections 1, 18, 25 and 26 of the Technical Report.
9. I personally visited the property that is the subject to the Technical Report between April 06 – 12, 2018.
10. I have had prior involvement with the property that is the subject of the Technical Report. Knight Piesold was involved with the heap leach design and construction in 2018.
11. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
12. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 30th day of November 2021

"Original Signed and sealed" David Morgan

David Morgan, MIE Aust CPEng
Managing Director
Knight Piesold Pty Limited

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CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*NI 43-101 Technical Report – Feasibility Study for the Altan Tsagaan Ovoo (ATO) Phase 2 Expansion Project, Mongolia*" with an effective date of October 27, 2021 and issued on November 30, 2021 (the "Technical Report") prepared for Steppe Gold LLC ("Steppe Gold" or the "Company").

I, *Ghislain Prévost, P. Eng., B. Mining Eng, M.Sc.A. Mineral Eng*, Montreal, Quebec, do hereby certify that:

1. I am Senior Mining Engineer with DRA Global Limited with an office at suite 600, 555 René-Lévesque Blvd. West, Montreal, Quebec, Canada;
2. I am a graduate from "*École Polytechnique de Montréal"* with Bachelor of Mining Engineer in 1996 and a Master degree Applied Science in Mineral Engineering in 1999.
3. I am a registered member of "*Ordre des Ingénieurs du Québec*" (# 119054).
4. I have practiced my profession continuously since 1999 with over 20 years of experience in mining exploration in gold, silver, base metals, and other projects across Canada and worldwide.
5. My relevant work experience includes:
    - Design, scheduling, cost estimation and Mineral Reserve estimation for several open pit studies.
    - Technical assistance in mine design and scheduling for mine operations in Canada, Brazil, and Guinea.
    - Participation and author of several NI 43-101 Technical Reports.
6. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
7. I am independent of the Company applying all the tests in Section 1.5 of NI 43-101.

8. I am responsible for the preparation of Sections 15 and 16, and portions of Sections 1, 21, and 25 to 27 of the Technical Report.

9. I did not visit the property on that is the subject to the Technical Report.

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

    - QP for "*Altan Tsagaan Ovoo Project (ATO), 2021 Mineral Resources, Technical Report (Amended NI 43-101)*", prepared by GeoRes, with an effective date March 30, 2021, issued on March 30, 2021 and amended on June 9, 2021.

11. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.

12. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 30th day of November 2021

"Original Signed and sealed" Ghislain Prévost

Ghislain Prévost, P. Eng., B. Mining Eng, M.Sc.A.
Principal Mining Engineer
DRA Global Limited

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CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*NI 43-101 Technical Report – Feasibility Study for the Altan Tsagaan Ovoo (ATO) Phase 2 Expansion Project, Mongolia*" with an effective date of October 27, 2021 and issued on November 30, 2021 (the "Technical Report") prepared for Steppe Gold LLC ("Steppe Gold" or the "Company").

I, ***Robin A. Rankin****, MSc DIC MAusIMM CP(Geo)*, do hereby certify:

1. I am Principal Consulting Geologist and operator of GeoRes, with business address at 4 Warenda Street, Bowral, New South Wales, Australia, 2576.
2. I graduated with a Bachelor of Science (BSc) degree in Geology from the University of Cape Town, South Africa, in1980. In addition, I have obtained a Master of Science (MSc) degree in Mineral Production Management from the University of London (Royal School of Mines at Imperial College London), United Kingdom, in 1988 and a Diploma of the Imperial College (DIC) from Imperial College London in 1988.
3. I am a member (#110551) of the Australasian Institute of Mining and Metallurgy (MAusIMM). Furthermore, I have been registered as a Chartered Professional (CP) in the Geology discipline (CP(Geo)) by the AusIMM since 2000.
4. I have practiced my profession (geology) virtually continuously since 1981 (+40 years). A summary of my relevant experience follows:
   - 1981 – 1987: Mineral exploration geologist employed or contracted by several mining, exploration and consulting companies including Goldfields of South Africa, Australian Groundwater Consultants Union Oil Development Corporation, and BHP (under contract).
   - 1989 – 2003: Metals geologist, employed by Exploration Computer Services (ECS) and then Surpac Minex Group (SMG), specialising in mining software and resource modelling and estimation.
   - 2003 – 2006: Principal metals geologist employed by SMG Consultants.
   - 2006 – Present: Principal consulting geologist and proprietor of GeoRes.

5. I have read the definition of “Qualified Person” (QP) set out in the National Instrument 43-101 – Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a Qualified Person for the purposes of NI 43-101.

6. I am independent of the Company applying all the tests in Section 1.5 of NI 43-101 and the Companion Policy 43-101CP.

7. I am responsible for the preparation of Sections 7 to 12, 14, and 23, and portions of 1 and 25 to 27.

8. Due to the COVID-19 Pandemic I personally did not visit the property that is the subject to the Technical Report.

9. I have had prior involvement with the property that is the subject of this Technical Report having been principal author and QP of the following previous report:

   - *“Altan Tsagaan Ovoo Project (ATO), 2021 Mineral Resources, Technical Report (Amended NI 43-101)”*, prepared by GeoRes, with an effective date March 30, 2021, issued on March 30, 2021 and amended on June 9, 2021.

10. I have read NI 43-101 and the Sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.

11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 30th day of November 2021

“Original Signed and sealed”

***Robin A Rankin**, MSc DIC MAusIMM CP(Geo)*
Principal Consulting Geologist
GeoRes