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Xenith

Steppe Gold Limited

Altan Tsagaan Ovoo Project (ATO)
2022 Mineral Resources & Reserves Report (NI 43-101)

06 NOVEMBER 2022

IMPORT NOTICE

This Report, following National Instrument 43 101 rules and guidelines, was prepared for Steppe Gold LLC (Steppe Gold or the Company) by Xenith Consulting (Xenith) and GeoRes (GeoRes). The quality of information, conclusions and estimates contained herein is consistent with the level of effort involved both Xenith's and GeoRes's services, based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this Report. This Report can be filed as a Technical Report with Canadian Securities Regulatory Authorities pursuant to National Instrument 43-101, Standards of Disclosure for Mineral Projects. Except for the purposes legislated under Canadian securities laws, any other uses of this Report by any third party are at that party's sole risk.

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.

DATE AND SIGNATURE PAGE

This Technical Report (entitled "Altan Tsagaan Ovoo Project (ATO) 2022 Mineral Resources & Reserves Report (NI 43-101)", effective date 6th November 2022) has been prepared, signed and dated by the following Qualified Persons (QP). The QP designation is within the meaning of Canadian National Instrument 43-101 of 24th June 2011.

<u>Mineral Resources have been signed by .</u>

"Signed and Sealed"

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Signed 6th March 2023

<u>Mineral Reserves have been signed by .</u>

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Signed 6th March 2023

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1 SUMMARY

This Altan Tsagaan Ovoo Project (ATO) 11/2022 NI 43-101 Mineral Resources and Reserve Technical Report (the Report) summarises the 2022 Mineral Resource and Reserve estimates of the of gold and related base and precious metals in four in-situ deposits forming Steppe Gold Ltd's (Steppe Gold) ATO Project in eastern Mongolia. The Resources (estimated by GeoRes) and Reserves (determined by Xenith) are an update to the previous March 2021 Mineral Resource estimate by GeoRes (3/2021 NI 43-101 report[3]) and a November 2021 feasibility study by DRA Global Limited (DRA) disclosed in a November 2021 NI 43-101 report (11/2021 NI 43-101)[4]. The reserves are based on an updated LOM Plan as outlined in this report.

The QP for the geological aspects of the Report is Robin Rankin (GeoRes) and the QP for the mining aspects of the Report is Grant Walker (Xenith).

1.1 Introduction

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.

The ATO Project commenced mining in 2020, initially concentrating on the near surface Oxide rock ores (Phase 1 Development). Following the successful completion of ATO Phase 1 development (including the development of the Leach Pad and on-going crusher upgrades) Steppe Gold completed studies for the ATO Phase 2 Expansion Project (Phase 2). The Phase 2 Expansion Feasibility Study was prepared by DRA and is the subject of the 11/2021 NI 43-101 Technical Report. The proposed expansion will increase 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.

Since the 11/2021 NI 43 101 Technical Report Steppe Gold commissioned GeoRes to remodel the transition/ fresh interface based on face samples and additional drill hole information. It is noted the underlying geological model has not been updated from the 3/2021 reported Resources. In addition Xenith was commissioned by Steppe Gold to revise the Life of Mine Plan and Reserves based on the new Resources and updated revenue assumptions. This report is an update to the 11/2021 NI 43 101 based on the new Resources and Reserves.

Sources of information & reliance on others: Considerable information used to support this Report was derived from the previously reported 2021 NI 43-101s and from reports and documents listed in the references section of this Report. Note that large parts of geological Sections 4 to 14 and 17 to 19 have been repeated from the 3/2121 NI 43-101 in the relevant Sections in this Report for completeness. Both

[3] Amended NI 43-101 Technical Report. Rankin, R.A., 30 March 2021. Altan Tsagaan Ovoo Project (ATO) – 2021 Mineral Resources Technical Report (Amended NI 43-101). Report for Steppe Gold Limited by GeoRes. Referenced here as the '3/2021 Resources Report'

[4] *NI 43-101 Technical Report – Feasibility Study for the Altan Tsagaan Ovoo (ATO) Phase 2 Expansion Project Mongolia*. Report for Steppe Gold LLC by DRA Global, November 29 2021, Referenced here as the '2021 NI 43-101' or '2021 Report'.

QP's has reviewed the relevant sections of 3/2021 NI 43-101 and believes the information repeated is correct. All Project data used was supplied by Steppe Gold.

Property inspection:

The principal Authors of this report have both visited the site. Robin Rankin of GeoRes visited the Property in April/May 2022 while Grant Walker of Xenith visited the site from the 3rd to the 7th of October, 2022.

1.2 Property Description and Location

Steppe Gold's ATO Property is in Eastern Mongolia. In mining terms the Property is defined by Mining License MV-017111. Surface area of the Mining Licence MV-017111 is 5,492.63 ha or ~55 km^2 (1 ha = 10,000 m^2). The immediate Project area of the four deposits is ~2 km^2, with dimensions ~1.4 km east/west * ~1.2 km north/south. Regionally ATO is ~660 km east of Mongolia's capital Ulaanbaatar, ~120 km west-north-west of provincial capital Choibalsan, and ~38 km west of the closest town Tsagaan Ovoo Soum (which it is reached from by dirt roads). The coordinate datum used is WGS84, Zone 49 (108°E to 114°E in northern hemisphere) in the UTM system.

Geography: The license area is located 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. Average elevation is 980 – 1,050 m above sea level. The area is effectively grass-covered. The land surrounding the Property is predominantly used for nomadic herding of goats, cows, horses and sheep by small family units.

Climate of the region is characterized by extreme cold and hot weather. Wide daily, monthly, and yearly fluctuations of temperature are common. Winter is harsh and very cold. Stable snow cover persists from November to March. Freezing of soil starts from mid-September and continues till late May, with the freezing depth reaching 2.5 m. Summer is shorter than other seasons, dry and chilly. The hottest temperature is up to +40˚C in summer. 60-80% of the annual precipitation falls as rain during July and August. Number of days with precipitation is 59 days per year. ATO Mine will operate all year around.

History: Modern exploration in the region commenced in 1997 when CogeGobi (a wholly owned subsidiary of the French multinational company AREVA) began their exploration efforts in eastern Mongolia looking for gold and uranium. After a six year reconnaissance effort CogeGobi settled on a selected exploration region in 2003 and then obtained eight exploration licenses in eastern and south-eastern Mongolia. CogeGobi then embarked upon a four-year concerted exploration effort. Two of the licenses (3,425.5 km^2 in all) were in the general area of ATO. Grab sampling of vein quartz lead to a stream sediment sampling program and gold anomalies were identified from two of the hills above the current deposits. CogeGobi withdrew due to falling uranium prices.

In 2010 CGM acquired the Exploration License and in 2012 acquired a Mining Licence. CGM quickly appreciated the potential for gold and commenced a significant exploration program leading to drilling ~600 holes. These discovered the three pipe-shaped deposits 1, 2 and 4. Steppe acquired the Property in 2017 and since then have more than doubled the quantity of drilling.

In 2016 CGM published an AIF with Measured and Indicated Mineral Resource of 18.6 Mt @ 1.3 g/t gold. Inferred Resources of 0.4 Mt @ 0.6 g/t gold were also reported. Reporting details were sketchy.

In 2017 Steppe published Measured and Indicated Mineral Resources of 17.6 Mt @ 1.4 g/t gold, along with Inferred Resources of 1.3 Mt @ 1.0 g/t gold. These latter Resources were in the 2017 NI 43-101. In 2017 Steppe also published Proven and Probable Mineral Reserves of 5.2 Mt @ 1.3 g/t gold. The Reserves were reported from three pits designed within the upper oxide parts of the Pipe 1, 2 and 4 deposits.

Mining commenced at ATO in 2020. Following the successful completion of ATO Phase 1 development (including the development of the Leach Pad and on-going crusher upgrades), Steppe Gold LLC (Steppe Gold) completed studies for the ATO Phase 2 Expansion Project (Phase 2). The Phase 2 Expansion Feasibility Study was prepared by DRA and is the subject of the 11/2021 NI 43 101 Technical Report. The proposed expansion will increase 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.

1.3 Geology Setting and Mineralisation

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

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.

Mineralisation: 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. It could be characterised as an intermediate sulphidation system. Up to 2017 exploration focussed on three gold, silver and base metal mineralised sub-vertical pipes (Pipes 1, 2 and 4) spaced ~300 m apart on a WNW trend. Another pipe (Pipe 3) exists just west of the others but is not mineralised. Subsequently a fourth pipe-like body (Mungu) was found ~600 m to the north east of Pipes 1, 2 and 4). The pipes have been emplaced into stratified rocks. The three pipes are elliptical in shape with the long axis oriented toward the north east. Each have approximate surface dimensions of 300 * 150 m. The pipes taper to depth vertically. Mungu is a north east plunging system of tall lenticular lodes. Pipes 1 and 2 are near paleo surface, epithermal (hot spring) emplacements and the upper parts of mineralized breccia pipes. Pipe 4 is slightly buried without the surface mineralisation.

Deposit type: ATO's mineral deposit type 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.

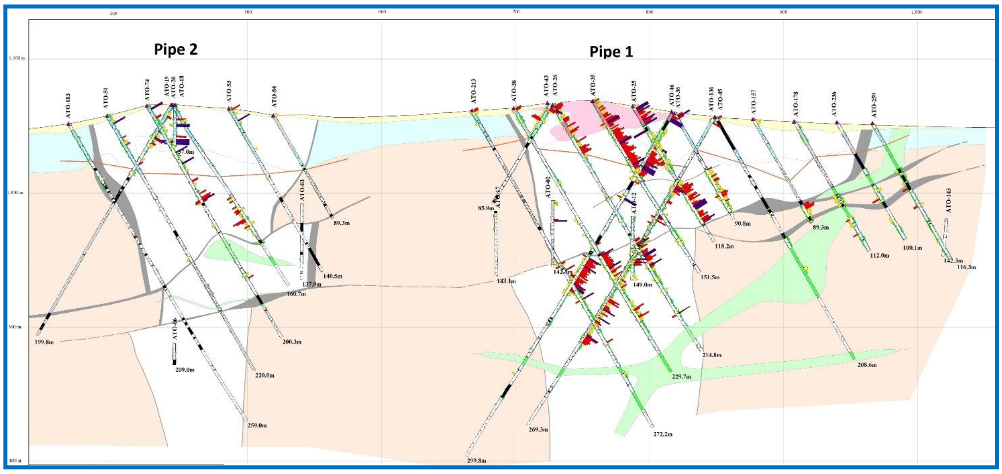
Exploration: Several companies have explored the area with regional focus shifting to the specific ATO deposit area because of its prospectivity. Various surface based programs (mapping stream and soil Geochem, geophysics, grab sampling) lead into concerted drilling commencing in 2010 on soil Geochem gold anomalies, the strongest of which were over the pipes now host deposits. Trench was initially undertaken to confirm the anomalies. In all 244 trenches were excavated in ATO district (28,809 m) and surrounding areas including 168 trenches at the ATO prospect (2012 to 2014).

Drilling: Up until the previous 2017 Resource estimate, and since acquiring the ATO Project in 2007, CGM had completed, a total of approximately 63,866 m of exploration drilling in 597 holes (to the end of 2014). Of that, 54,425 m was core drilling in 370 holes and 9,441 m was reverse circulation (RC) drilling in 227 RC holes. That drilling has been spread over the ATO mining license as well in the exploration area to the south (Figure 10.1). Drilling efforts were focused on expanding the known mineralization at the pipes and exploration drilling in several potential southern target areas.

Steppe commenced drilling in 2018 and to 2020 had added ~56,036 m in 170 diamond holes. That brought the grand total to 120,320 m in 767 drill holes. Of that diamond holes total 110,879 m in 540 holes. In the Project area trenching (pseudo surface drill holes) account for 10,184 m in 167 trenches. It is not clear if these totals include holes and trenches outside the Project area. Drilling during this period was focussed on Pipes 1,2 and 4 and increasingly on Mungu (Figure 10.2).

Initial RC discovery drill holes were relatively short (~40 m), vertical, and drilled on a 100 * 100 m square pattern. The bulk of the diamond core (DDH) holes were located on drilling cross-sections oriented at 125° and 30 m apart. This direction was perceived to be approximately across strike of the deposits. 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 7.5 - Cross-section through Pipes 1 and 2



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" drilled at Mungu averaged ~240 m in length.

Since early 2021 a further 81 (71 holes @ ATO and 10 holes @ Mungu) diamond core holes have been drilled for 16,406 m (13,337m @ ATO & 3,069 m @ Mungu). This new data has not yet been databased or used to re-estimate any new Resources.

Sample preparation, analysis and security: Most samples were of drill core which was cut and split on site before being sent away for analysis (of gold, silver and associated base metals) in the capital Ulaanbaatar. Drill hole samples were taken continuously over their full length and at 1 to 2 m intervals through mineralized zones (mostly 1 m) and at 2 to 3 m intervals through unaltered host rocks. In general core recoveries were very good and averaged 97% for the deposits.

Bulk density was determined in 2010/11 from 226 samples from diamond core holes. Bulk densities by oxidation level were 2.46 t/m^3 in oxide material, 2.59 t/m^3 in transitional material and 2.64 t/m^3 in fresh rock.

Opinion on drilling and sampling: The geological GeoRes QP's overall opinion* 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. The sample preparation, QA/QC, security and analysis procedures were considered positively.

Data verification: CGM originally implemented a series of industry standard routine verifications to ensure the collection of reliable exploration data. Documented exploration procedures exist to guide most exploration tasks to ensure the consistency and reliability of exploration data. In accordance with NI 43-101 guidelines, the Steppe in-country QP visited the ATO deposit on August 23 and October 2, 2017. The site visits were conducted to ascertain the geological setting of the ATO Project gold-lead-zinc mineralization and to witness the extent of exploration work carried out on the property.

For the 2017 estimate routine verifications were completed by the DRA QP to ensure the reliability of the drill hole and topography surface data, and analytical data provided by Steppe. In the opinion of the DRA QP then the electronic drill holes data was reliable, appropriately documented and exhaustive. The analytical results were sufficiently reliable for the purpose of resource estimation.

For the March 2021 Resource estimate the GeoRes QP's overall opinion* was that ATO's drilling data was completely adequate for Resource estimation.

*These GeoRes opinions are qualified by the fact that up to the time of the early 2021 Resource estimation the GeoRes QP had not physically been able to sight any of the Project's geology or drilling himself (due to the un-avoidable inability to visit the site because of the Covid Pandemic). Since then the QP visited site in 2022 and observed drilling operations there – which largely confirmed his previous opinions.

1.4 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 ore. A subsequent expansion to Phase 1 included new three-stage crushing (which at the time of this report had begun commission testing).

Phase 2 will consist of milling, flotation, and dewatering unit operations to produce concentrates of lead (Pb), zinc (Zn), and pyrite (Py). A testwork program performed by the laboratory in 2021 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.

Xenith and Georesreviewed the design proposed in the 2021 NS 43-101 report and consider it appropriate for the mineraology,

Historical Testwork (2010-2018)

Several metallurgical testwork programs have been 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.

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.

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 (g/t)	Au (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

Grinding

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.

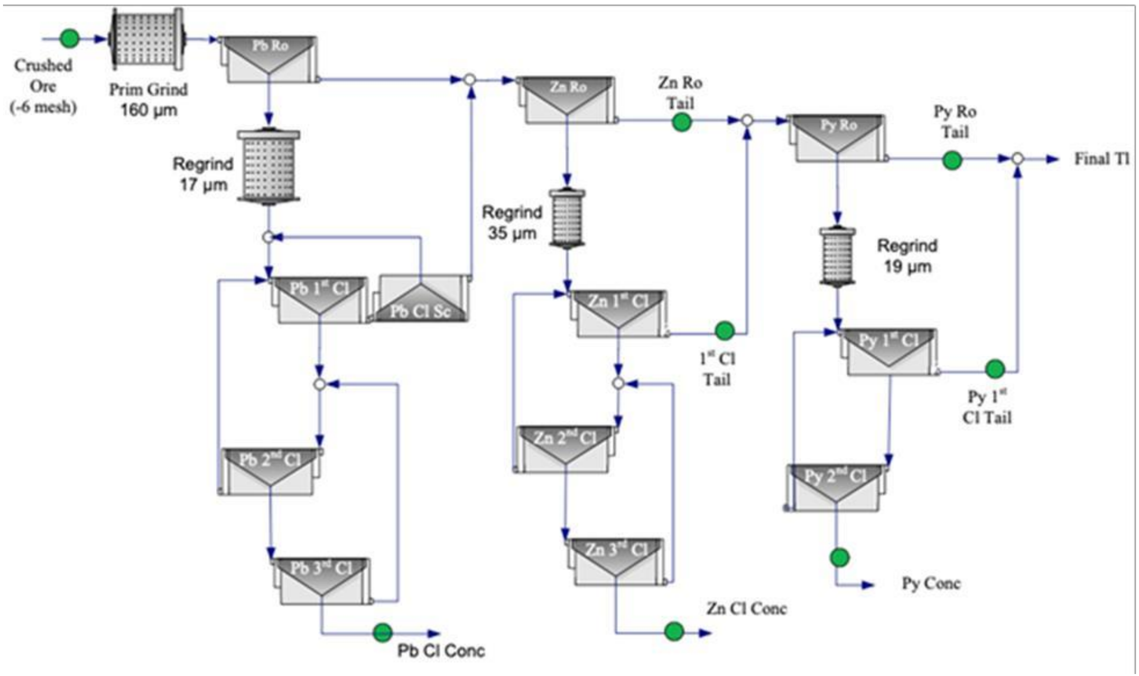
Table 1.2 Grindability Test Summary

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

Floatation

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 2 - 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.

Recovery Estimates

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 Rec % = 41.2; fixed value
- Pb Conc Silver Rec % = 45.6; fixed value

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 Rec % = 14.1; fixed value
- Zn Conc Silver Rec % = 18.2; fixed value

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

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.

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.

1.5 Mineral Resources - modelling, analysis, grade estimation and Resources:

Introductory statements: These 2022 reported Resources are based on Resource estimation independently undertaken in late 2020 and early 2021 by the GeoRes QP / CP under the CIM, JORC and NI 43-101 Codes, Instruments and definitions. The 2022 Resources have been reported from the 2021 model using new classification of the oxidation levels (Oxide/Transitional/Fresh) and exclude mining in the interim. Resources were reported according to JORC, accepted as a foreign Code by NI 43-101, and using equivalent definitions to The CIM.

Data: All data was supplied by Steppe. Data used was raw drill hole data, topography data, oxidation level data, data extracted from the 2017 Report (such as bulk density), and parameters supplied by Steppe (cut-off grades). Updated topography and oxidation level data was supplied by Steppe in 2022.

Drill hole database: A Minex software drill hole database was loaded with raw collar, down-hole survey and assay data. It was subsequently updated with interpreted data for assay population domains and oxidation surface intercepts.

Map database: A Minex map database was loaded with raw topography 1 m interval contour data. It would subsequently store deposit outline interpretations and models.

Geological interpretation and modelling of deposits: 3D inspection of the drill holes indicated that "wire-frame" modelling (joining cross-section outlines together with wires) would best suite the massively (rather than thinly) shaped deposits. With the abundance of relatively close-spaced drilling the deposit boundary outlines were interpreted around their gold (approximately >0.15 g/t), and to a lesser extent silver (approximately >1.0 g/t), whilst being also cognoscent of the lead, zinc and arsenic values, mineralisation on multiple parallel vertical cross-sections oriented at 125°. All bodies had a general north-east elongation (or strike), consequently the cross-sections were effectively across strike.

Deposit Pipes 1, 2 and 4 (now known as ATO1, 2 and 4) were wire-frame modelled (by connecting the cross-sectional outlines together to form solids) as single individual bodies; Mungu was modelled as a series of eight tightly packed, north-easterly striking sub-parallel and approximately en-echalon tall semi-vertical bodies. Samples were domain segregated by Pipe and in Mungu's case by individual body.

Geological interpretation and modelling of oxidation levels: Interpretation of the oxidation levels at the deposits was done in early 2021 in all drill holes from the lithological logs. This was necessary as bulk density would be assigned for Resource reporting by oxidation level. From surface the hole interval was interpreted as oxidised (code OX), partly oxidised or transitional (code TR), and un-oxidised or fresh (code FR). The interfaces to the intervals, representing the base of oxidation and the top of fresh rock, were modelled as DTM gridded surfaces with a 5 * 5 m grid mesh.

In July 2022 new data was supplied from site defining the base of the transitional material (top of the fresh rock). That base had previously been interpreted too deep. That base surface was re-modelled and used in this 2022 Resource reporting. The new surface was ~15 m higher than before, resulting in the transitional layer being reduced in thickness to ~4.5 m. The lower 15 m of the old transitional material was re-classified as fresh.

Topography surface model: Topography was modelled as a gridded DTM surface from the contour strings. 2021 reporting used a 2020 surface pre-mining. This 2022 reporting used a new July 2022 surface incorporating mining in Pits 1 and 4 to that date.

Grade statistics: Sample grades were briefly analysed statistically to determine data limits and block grade estimation parameters. The presence of few anomalous gold grades (<1%, unusually low for gold) prompted abandoning the use of grade cutting for the estimations. However the 1% limits derived from the simple statistics (at 10-20 g/t gold) were used to produce good variograms in the following brief geostatistical analysis. Those variograms mostly produced ranges >25 m (which agreed with results from the 2017 study where gold ranges were ~20-60 m). That distance continuity was of the same order of magnitude or longer than the typical 30 * 30 m drill hole spacing. It also implied that the grade continuity distances were approaching the same dimensions (50-100 m) observed of the well mineralised parts of the interpreted deposits. In terms of continuity directions the Author QP chose to use the clear mineralisation directions evident during the deposit cross-sectional interpretation. At Pipe 1 this was a steep 80°W dip. At Pipes 2 and 4 it was an intermediate 45°W dip. And at Mungu it was a vertical dip with the lodes striking 033°.

Resource block models: A block model was built for the Pipe 1,2 and 4 deposits (domains 1, 2 and 4 respectively) and another for the Mungu deposit (domains 5 to 11 and 15). Block models were built un-rotated within the wire-frames – deposits Pipe 1 2, and 4 with equi-dimensional 5*5*5 m blocks; Mungu with tall thin east-west 2*5*5 m blocks better representing the tall thin lodes.

Block grade estimation: Block grades were estimated individually for gold (Au), silver (Ag), copper (Cu), lead (Pb) and zinc (Zn) using an Inverse Distance squared algorithm (ID2). Drill hole sample intervals were down-hole composited by domain to 2.0 m for Pipes 1,2 and 4 and to 1.0 m for Mungu. No grade clipping or cutting was necessary. A maximum sample scan distance of 75 m was used (although in practice this wasn't needed due to the tight wire-frame model constraints and the close drill spacing).

For Pipe 1 axes were rotated 10° to give an 80°W dip and weighted to give down dip preference. For Pipes 2 and 4 axes were rotated 45° to give an 45°W dip and also weighted to give down dip preference. For Mungu axes were rotated 33° to give a 033° strike and weighted to give vertical preference.

A "gold equivalent" (AuEq) block value was computed from the individual elements by factoring them by their international metal prices averaged over the month to mid-January 2021.

Resource classification: Although the QP considered that proportions of Measured and Indicated Resources reported in 2017 were relatively too high he nevertheless considered that the bulk of estimated material in the 2021 estimate should be classified Measured or Indicated.

JORC classification was done here by block and was based on average sample distances (D) and numbers of sample points (P, minimum 1, maximum 18) in estimating each gold block grade. A Resource class was calculated for each block based on criteria combining these variables. The combinations were determined from a combination of statistics and observation of their distributions (the latter with the objective of ensuring contiguous class zones and avoiding spotting). Measured class criteria for all deposits was D ≤ 27.5 m and P ≥ 12; Inferred criteria was D ≤ 35.0 m and P ≥ 6; and Inferred was D > 35.0 m and P >1. All blocks were classified. This created Measured zones typically in the centre of deposits and in areas with highest drill hole densities. Indicated zones were in areas of sparser drilling and Inferred zones were generally restricted to the edges of deposits.

Mineral Resources: Combined Measured and Indicated JORC classified in-situ Mineral Resources (directly equivalent to CIM categorisation) were reported in July 2022 for all four deposits, using fixed densities and lower AuEq grade cut-offs. These Resources utilised the new 2022 oxidation level models which increased slightly the proportion of fresh rock over the 2021 Resources. These Resources also utilised a new July 2022 upper topographic surface incorporating mining to that point in Pits 1 and 4. Total Measured and Indicated in-situ Resources were reported at **38.0 Mt** at an average **AuEq** grade of **1.68 g/t** (for 2.1 M oz metal). In the following tabulations deposits ATO1, 2 and 4 represent Pipes 1, 2 and 4.

Further and separate Inferred JORC class in-situ Resources were reported at **5.4 Mt** at an average AuEq grade of **1.16 g/t** (for 0.2 M oz metal).

The Resource break-down into separate classes was:

Table 1-3 ATO Resource at 27/07/2022

	Cut-off	Tonnes	Grades					Metal		
	AuEq (g/t)	(Mt)	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	AuEq (g/t)	Au (k oz)	Ag (k oz)	AuEq (k oz)
Measured	0.38	21.6	1.17	16.38	0.40	0.71	1.85	811	11,370	1,287
Indicated	0.38	16.4	0.84	14.52	0.34	0.63	1.45	444	7,672	765
Meas+Ind	0.38	38.0	1.03	15.58	0.37	0.68	1.68	1,255	19,042	2,052
Inferred	0.40	5.4	0.62	15.39	0.25	0.52	1.16	108	2,655	200
Total		**43.4**	**0.98**	**15.62**	**0.37**	**0.68**	**1.68**	**1,363**	**21,787**	**2,343**

The break-down by deposit was:

Table 1-4 ATO Resource at 27/07/2022 by Deposit

	Deposit	Tonnes	Grades					Metal		
		(Mt)	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	AuEq (g/t)	Au (k oz)	Ag (k oz)	AuEq (k oz)
Measured	ATO1	7.9	1.07	5.86	0.70	1.28	1.97	272	1,493	502
	ATO2	1.7	0.43	3.85	0.50	0.77	1.01	24	212	55
	ATO4	7.0	1.35	12.11	0.30	0.53	1.86	304	2,743	422
	Mungu	4.9	1.33	43.92	0.01	0.03	1.96	209	6,922	308
	TOTAL	21.6	1.17	16.38	0.40	0.71	1.85	811	11,370	1,287
Indicated	ATO1	4.7	0.75	5.03	0.64	1.24	1.60	113	762	243
	ATO2	1.5	0.45	4.06	0.48	0.77	1.02	22	196	49
	ATO4	7.7	0.96	15.10	0.23	0.43	1.44	235	3,721	356

	Deposit	Tonnes	Grades					Metal		
	Mungu	2.5	0.90	36.53	0.01	0.03	1.43	74	2,993	117
	TOTAL	16.4	0.84	14.52	0.34	0.63	1.45	444	7,672	765
Meas + Ind	ATO1	12.6	0.95	5.55	0.68	1.27	1.83	385	2,256	745
	ATO2	3.2	0.44	3.95	0.49	0.77	1.01	45	408	105
	ATO4	14.7	1.14	13.67	0.26	0.48	1.64	540	6,463	777
	Mungu	7.5	1.18	41.39	0.01	0.03	1.77	283	9,915	425
	TOTAL	38.0	1.03	15.58	0.37	0.68	1.68	1,255	19,042	2,052
Inferred	ATO1	1.1	0.51	4.28	0.56	1.27	1.34	17	147	46
	ATO2	0.5	0.28	5.76	0.71	1.36	1.23	4	86	18
	ATO4	2.1	0.59	15.12	0.19	0.35	1.03	41	1,043	71
	Mungu	1.7	0.83	25.40	0.01	0.02	1.20	45	1,379	65
	TOTAL	**5.4**	**0.62**	**15.39**	**0.25**	**0.52**	**1.16**	**108**	**2,655**	**200**

The break-down by oxidation level (giving the AuEq lower grade cut-offs and densities used in all Resource reporting) was:

Table 1-5 ATO Resource at 27/07/2022 by Oxidation Level

Oxidation Level	Cut-off	SG	Tonnes	Grades					Metal		
	AuEq (g/t)	(t/m^3)	(Mt)	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	AuEq (g/t)	Au (k oz)	Ag (k oz)	AuEq (k oz)
MEASURED + IND + INF											
Oxide	0.15	2.46	3.9	0.49	7.57	0.31	0.26	0.84	62	951	106
Transition	0.40	2.59	1.3	1.30	9.52	0.59	0.78	2.01	55	404	85
Fresh	0.40	2.64	38.2	1.02	16.65	0.37	0.72	1.75	1,246	20,432	2,152
MEAS+IND+INF			**43.4**	**0.98**	**15.62**	**0.37**	**0.68**	**1.68**	**1,363**	**21,787**	**2,343**

Reconciliation: Reconciliation was done of th immediately previous 2/2021 Resources by the QP against the 2017 Resources reported by DRA. It could only be done for the three deposits also reported in the 2017 estimate (Pipes 1, 2 and 4). No data existed to reconcile the Mungu deposit against. Reconciliation (Table 14.14) was approximated to account for differences in estimate reporting parameters between 2017 and 2021, particularly the different cut-off grades used.

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 GeoRes QP considers that the comparable 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.

Potential impacts on Resources: The GeoRes QP was not aware of any other factors (excluding those specifically mentioned below), including environmental, title, economic, market or political, which could generally or in-particularly influence the Resources reported here for the ATO Project. Factors that could alter the Resources (but in all cases relatively insignificantly in the QP's view) were changes in grade cut-off; bulk density; gold equivalent (through variations in world metals prices); geological model; JORC classification; and mining method with depth (possibly a factor at the deeper Mungu where underground mining would be considered and which would have a considerably higher grade cut-off).

1.6 Mineral Reserve Estimate

The mineral reserves estimate with an effective date of August 30, 2022 for the Project is based on the parameters and steps outlined within this report as well as the resource estimate. The mineral reserves for the ATO gold deposit contains combined proven and probable mineral reserves totaling 29.1 million tonnes ("Mt") at 1.13 g/t gold and 12.43 g/t silver, containing 1.1 million ounces of gold and 11.7 million ounces of silver. The reserves have been classified as approximately 59% proven and 41% probable on a tonnage basis. The mineral reserve within the 2022 reserve pit shell was based on a AuEq cut-off grade of 0.43 g/t AuEq for Fresh material and 0.40 g/t AuEq for Oxide material and revenue of $1,700 per ounce gold, $20 per ounce of silver, zinc price of $2,500/t and lead price of $1,970/t. as the price assumptions. To access the ore, a total of 104 Mt of waste rock will need to be extracted at an average stripping ratio of 3.6.

Table 1.6 Mineral Reserves (as of August 2022)

		Ore	Grade					Attributable Metal		
		kt	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	AuEq (g/t)	Au (k oz)	Ag (k oz)	AuEq (k oz)
ATO										
Proven										
	Oxide	934	1.14	0.70	5.57	0.46	0.34	21	168	34
	Transition	361	1.57	0.72	10.32	0.41	0.70	8	120	18
	Fresh	13,535	2.10	1.37	8.59	0.49	0.88	597	3,749	917
	Total	**14,830**	**2.03**	**1.31**	**8.44**	**0.48**	**0.84**	**627**	**4,036**	**970**
Probable										
	Oxide	850	0.92	0.55	5.91	0.35	0.25	15	162	25
	Transition	372	1.47	0.70	11.35	0.27	0.48	8	136	18
	Fresh	9,922	1.69	1.09	11.26	0.36	0.68	350	3,603	541
	Total	**11,145**	**1.62**	**1.04**	**10.86**	**0.36**	**0.64**	**373**	**3,901**	**584**
Proven & Probable										

		Ore	Grade					Attributable Metal		
	Oxide	1,785	1.04	0.63	5.73	0.41	0.30	36	330	60
	Transition	733	1.52	0.71	10.84	0.34	0.59	17	256	36
	Fresh	23,457	1.93	1.25	9.72	0.43	0.79	947	7,352	1,458
	Total	**25,975**	**1.85**	**1.19**	**9.48**	**0.43**	**0.75**	**1,000**	**7,938**	**1,554**
Mungu										
Proved										
	Oxide	224	1.13	0.71	25.92	0.38	0.42	5	187	8
	Transition	-	-	-	-	-	-	-	-	-
	Fresh	2,193	1.27	0.64	39.67	0.08	0.09	45	2,805	90
	Total	2,417	1.26	0.65	38.39	0.11	0.12	51	2,993	98
Probable										
	Oxide	54	0.92	0.61	19.23	1.59	1.75	1	34	2
	Transition	-	-	-	-	-	-	-	-	-
	Fresh	684	1.02	0.51	32.33	0.25	0.28	11	713	22
	Total	**738**	**1.01**	**0.52**	**31.37**	**0.35**	**0.39**	**12**	**747**	**24**
Proven & Probable										
	Oxide	278	1.09	0.69	24.62	0.62	0.68	6	221	10
	Transition	-	-	-	-	-	-	-	-	-
	Fresh	2,877	1.21	0.61	37.93	0.12	0.13	57	3,518	113
	Total	**3,156**	**1.20**	**0.62**	**36.75**	**0.16**	**0.18**	**63**	**3,739**	**122**
Combine ATO & Mungu										
Proven										
	Oxide	1,159	1.14	0.70	9.50	0.44	0.36	26	355	43
	Transition	361	1.57	0.72	10.32	0.41	0.70	8	120	18
	Fresh	15,728	1.99	1.27	12.92	0.43	0.77	643	6,554	1,007
	Total	**17,247**	**1.92**	**1.22**	**12.64**	**0.43**	**0.74**	**677**	**7,029**	**1,068**
Probable										
	Oxide	905	0.92	0.56	6.71	0.43	0.34	16	196	27
	Transition	372	1.47	0.70	11.35	0.27	0.48	8	136	18
	Fresh	10,606	1.65	1.06	12.62	0.35	0.65	361	4,316	563
	Total	**11,883**	**1.59**	**1.01**	**12.13**	**0.36**	**0.62**	**385**	**4,648**	**608**
Proven & Probable										
	Oxide	2,063	1.04	0.64	8.28	0.44	0.35	42	551	69
	Transition	733	1.52	0.71	10.84	0.34	0.59	17	256	36
	Fresh	26,334	1.85	1.18	12.80	0.40	0.72	1,004	10,870	1,571
	Total	29,130	1.78	1.13	12.43	0.40	0.69	1,063	11,677	1,676

Notes

1. Mineral Reserves estimate was based on Measured and Indicated Resource Estimate by R. Rankin, QP and effective August 27 2022.
2. ATO and Mungu Mineral Reserves are effective as of August 27, 2022.
3. Mineral Reserves are included in Mineral Resources.
4. Mineral Reserves are reported in accordance with JORC and CIM and NI 43-101 guidelines.
5. Ore dilution is estimated at 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 0.40 g/t AuEq for oxide material and 0.43 g/t AuEq for transition and fresh material.
8. Mineral Reserves are contained within an optimised pit shell based on a gold price of $1,700 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: $1,700/oz gold, $20/oz silver, $1,970/t lead, $2,500/t zinc.
11. Totals may not match due to rounding.
12. The Mineral Reserves are stated as dry tonnes processed at the crusher.

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 173.1 million tonnes, ore and waste combined. The material will be mined over a period of approximately 14 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. Contractors are used to mine both the waste and ore .

A mine plan 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.

1.7 Processing

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 has essential been constructed and is undertaking commissioning) is part of Phase 1.

- Phase 2 – Concentrator (Fresh and Transition Ores)

  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

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 F100 of 800 mm to a P80 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

1.8 Project 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.

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.

1.9 Market Studies

The ATO Project is an operating site producing a readily saleable commodity in the form of gold bars. The bars are 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:

1. contains approximately 70% Au by weight and the remaining 30% is a mixture of Ag, base metals and Fe.
2. 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.

1.10 Environmental Approvals and Status

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.

1.11 Capital and Operating Costs

Operating Cost

The Operating Cost Estimate (OPEX) is presented in $ USD. The cost have been developed in conjunction with Steppe Gold. The estimate includes mining, processing, and general and administration (G&A). The estimate has an accuracy of +30% -15%.

The OPEX is estimated at $884 M over the life of mine or $30.45/t of ore processed, with 1.5 years of operation for Phase 1 and 11.5 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 1.7.

Table 1.7 Average Operating Costs

	Av. Annual Cost (USD M)	Cost / t ore processed (USD/t)	Total Cost LOM (M USD)
Mining	17.7	7.87	249
Processing	30.2	13.44	425
General Admin	12.2	5.50	174
Total	**60.1**	**26.81**	**848**

Capital Cost

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.

Table 1.8 Capital Cost Summary

	Total Capital (SD M)
Mining	1.8
Process Plant	75.2
Tailings/ reclaim water and water treatment	13.5
Power	1.7
Indirect	23.3
Owner's Cost	1.5
Contingency	11.5
	128.5

1.12 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 13 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.

Table 1.9 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 $330 M USD at a discount rate of 5%.

Table 1.9 Financial Summary

Description	Unit	Value	
LOM Tonnage Ore Processed	kt	29,103	
LOM Feed Grade Processed - Au	g/t	1.13	
LOM Feed Grade Processed - Ag	g/t	12.43	
LOM Feed Grade Processed - Pb	%	0.40	
LOM Feed Grade Processed - Zn	%	0.69	
LOM Recovery - Au	%	79.2	
LOM Recovery - Ag	%	72.6	
LOM Recovery - Pb	%	82.5	
LOM Recovery - Zn	%	85.9	
Total Net Revenue (after streaming, Payable)	USD Million	2,003	
Refining/transport costs	USD Million	229	
LOM Operating Costs	USD Million	848	
		Pre tax	Post Tax
NPV @ 5%	USD Million	364	242
NPV @ 8%	USD Million	273	176
NPV @ 10%	USD Million	226	142

1.13 Conclusions and Recommendations

1.13.1 Mineral Resource Estimate

The Overall interpretation of the estimation and resulting Resources was that it proceeded as expected, confirmed the 2017 & 2021 modelling and results of Pipes 1, 2 and 4, and produced a more accurate second generation result. Re-modelling in 2022 of the base of oxidation surface (top of fresh material) was considered to have significantly improved the accuracy of the Resource classification by oxidation level (oxide/transition/fresh) although it had minimal impact on overall Resources. Furthermore the up-dated topography took existing mining into full account.

Interpretation of mineralisation at the new Mungu deposit showed it to be more complex and lode-like than the other deposits, with greater potential for increasing the Resource with targeted drilling. Its different shape gives encouragement for further regional exploration to find other deposits of its style.

The conclusions were that:

- *The 2021/2 Measured and Indicated Resources*:
    - Pipes 1, 2 and 4:
        - The 2021/2 Resources confirmed the 2017 estimate generally.
        - It increased the Resource tonnage significantly (by 25%), decreased the gold grade slightly (by 3%), increased the silver grade reasonable (by 16%), leading to an overall significant increases in metal contents (gold by 22%, silver by 45%). This comparison uses equivalent reporting cut-offs between the two estimates.
        - The increase in deposit volume was not only because of additional drill holes but also because of more practical and geologically controlled deposit shape interpretation.
    - For Mungu:
        - The cross-sectional interpretations hung together and created a significant deposit.
        - The maiden 2021 Measured and Indicated Resource was significant at 7.6 Mt at 1.16 g/t gold (282 k oz gold metal) and 40.75 g/t silver (9,916 k oz silver metal).
        - Mungu now represents 18% by tonnage of the Resources, 20% of the gold metal and 48% of the silver metal (as the silver grade is 320% higher than for Pipes 1, 2 and 4).
    - All deposits:
        - The total Measured and Indicated Resource for all deposits in 2022 stands at 38.0 Mt at 1.03 g/t gold (1.3 Moz gold metal) and 15.58 g/t silver (19.04 Moz silver metal).
        - The absolute comparison (Table 52) of the 2017 Resource (reported at much higher cut-off grades) with the 2021 Resource showed a 136% increase in tonnage, a decrease of average gold grade of -27%, and an increase of average silver grade of 53%.
        - The absolute comparison of contained metal showed a 73% increase in gold metal (to 1.4 Moz) and a 262% increase in silver metal (to 20.5 Moz).
- *Adequacy of data*: The data supplied and used in the estimation was suitable for the purpose of Mineral Resource estimation and JORC classification.
- *Drilling data*: The drilling, sampling and subsequent assaying 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.
- *Compliance with JORC and Canadian standards*:
    - The Mineral Resource estimation Project, and the reporting of it, comply with the JORC (2012) and NI 43-101 (June 2011) standards.
    - The results of current and previous work have been successful to demonstrate the "reasonable prospects for economic extraction".
    - The fact that mining has commenced at the ATO Mine further reinforces this compliance with the Code.

Recommendations: Opportunities exist for further drilling to enlarge the defined deposits (particularly at Mungu) and for further exploration to find new ones. The latter includes exploring locally for Mungu style deposits which may have previously geologically been overlooked. Recent drilling strongly supports expansion at all deposits.

1.13.2 Mineral Reserves

ATO is an established conventional open cut Gold/Silver mine that has been in operation for several years.

Mine planning and evaluations undertaken using the latest resource models confirm that the continuation of mining and processing at ATO is both viable and economic.

The Mineral Reserve estimate has a relatively high sensitivity to revenue which is controlled by metal prices and payability. It is noted however that at current long term forecast metal prices, the mineral reserve is relatively insensitive to changes in revenue and costs. Mungu area is more sensitive to price than the ATO pit area.

Given the mine has been operational for a number of years, technical risk in relation to the Mineral Reserves estimate is deemed to be low.

2 INTRODUCTION

2.1 Purpose

This Technical Report on the Steppe Gold ATO Project has been prepared by Qualified Persons (QPs) Robin Rankin, of GeoRes, and Grant Walker of Xenith Consulting Pty Ltd.

The purpose of this Technical Report is to document the updated Mineral Resource and Mineral Reserve estimates for the Project and to provide a commentary on the status of the operations and proposed life of mine (LOM).

The specific aim of GeoRes's and Xenith's work was to a re-estimate Mineral Resources and Reserves previously reported in the 2021 NI 43-101 (JORC Resources and Reserves were to be reported in an NI 43-101 report). Resource classification under JORC was understood to be directly equivalent to the Canadian CIM standards.

2.2 Terms of Reference

GeoRes and Xenith prepared this NI 43101[5] Technical Report (the Report) for Steppe Gold Limited and Steppe will be the Issuer. Steppe is a precious metals company operating in Mongolia.

Steppe's general (and primary) objectives were understood to be exploration and development of the gold and related base and precious metals in the ATO Project in Mongolia.

The Technical Report covers the ATO and Mungu deposits and has been written to comply with the reporting requirements of the Canadian National Instrument 43-101: 'Standards of Disclosure for Mineral Projects' of the Canadian Securities Administrators (the Instrument) which in turn complies with the Standards on Mineral Resources and Reserves of the Canadian Institute of Mining, Metallurgy and Petroleum (the CIM Guidelines, 2019).

The effective date for the Mineral Resource and Mineral Reserve estimates is 30th August, 2022.

2.3 Sources – of information & data

Source: All information and data was supplied directly to GeoRes and Xenith by Steppe during 2021 and 2022.

Format: All data was supplied in digital form – predominantly in MS Excel spreadsheets. CAD data was supplied in DXF format. Reports were supplied in PDF and/or MS Word format.

[5] Canadian Securities Administrators (CSA), 24th June 2011. *National Instrument 43-101 Standards of Disclosure for Mineral Projects* (form 43-101F1). Companion Policy 43-101CP document setting out the CSA views on how they interpret and apply the provisions of the Canadian National Instrument 43-101 and Form 43-101F1.

Data & information:

- Drill hole data:
    - A series of ~7 incremental spreadsheets contained various periods of drilling data.
    - Data was supplied for drill hole collars, down-hole surveys, assays and lithology logging.
- Topography map data:
    - Mapping data was supplied as 1.0 m intervals contour strings and various road boundaries.
    - Data covered a square area 2.7 km in easting and nothing.
- Reports:
    - A 2017 NI 43-101[6] report on the ATO Project by GSTATS Consulting LLC (GSTATS) was supplied.
    - A 2021 NI 43-101[7] report on the ATO Project by DRA Global (DRA) was supplied.
- General information:
    - Project data and details continuously communicated through email.
    - Direct communication by telephone and video conferencing.

2.4 Property inspections

The principal Authors of this report have both visited the site. Robin Rankin visited the Property in April/May 2022 while Grant Walker visited the site from the 3rd to the 7th of October, 2022.

[6] Oyungeral Bayanjargal, 4 October 2017. *NI 43-101 Technical Report – Altan Tsagaan Ovoo Gold Project*. Report for Steppe Gold LLC by GSTATS Consulting LLC, Mongolia. Report reviewed by Qualified Persons Leonid Tokar (GSTATS), Tim Fletcher (DRA Americas Inc), David Frost (DRA) and Dr Martin Stapinsky (DRA). Referenced as the '2017 NI 43101' or '2017 Report'.

[7] Oyungeral Bayanjargal, 4 October 2017. *NI 43-101 Technical Report – Altan Tsagaan Ovoo Gold Project*. Report for Steppe Gold LLC by GSTATS Consulting LLC, Mongolia. Report reviewed by Qualified Persons Leonid Tokar (GSTATS), Tim Fletcher (DRA Americas Inc), David Frost (DRA) and Dr Martin Stapinsky (DRA). Referenced as the '2017 NI 43101' or '2017 Report'.

3 RELIANCE ON OTHER EXPERTS

3.1 Direct responsibility (no reliance on others)

The GeoRes QP was directly responsible here for the processing and evaluation of data to estimate and report (in this Report) new in-situ Mineral Resources at ATO. These tasks were essentially covered by:

- Geology (NI 43-101 Sections 7 and 8).
- Resource estimation (Section 14).
- Conclusions and recommendations (Sections 25 and 26).

The Xenith QP was directly responsible here for the processing and evaluation of data to estimate and report (in this Report) new in-situ Mineral Reserves at ATO. These tasks were essentially covered by:

- Reserve estimation (Section 15).
- Mining Methods (Section 16).
- Various processing and economic aspects (Sections 17 to 22).
- Conclusions and recommendations (Sections 25 and 26).

3.2 Reliance on others

As far as the other non-directly Resource/Reserve related aspects contained in this report go (listed below) the QPs relied upon:

- The 2021 Feasibility study carried out by DRA
- The 2017 NI 43-101 report by GSTAT (referenced above).
- General input from the Company.
- Specific provision of other Experts and/or Qualified Persons by the Company for those other aspects.

The non-directly Resource/Reserve related aspects included:

- Back-ground Property details (NI 43-101 Sections 4 to 6).
- Exploration (Sections 9 to 11).
- Data verification (Section 12).

Whilst not doubting the data on the non-directly Resource/Reserve related aspects relied upon (listed above), the QPs specifically disclaim responsibility for all information on legal aspects (such as mineral and land tenure, other agreements, environmental obligations and rights to operate). These aspects are beyond the QPs area of expertise and/or ability to verify.

The Resource QP also specifically disclaims responsibility for the data collection (exploration and drilling), sample preparation and data verification. Those aspects were beyond the Consultants ability to evaluate because of the limited exposure to the Company and to the Property.

Notwithstanding the statements above – of all of the external sources of Project data that the Consulting used the Resource QP was of the professional opinion that it originated from personnel who either were or would have constituted Competent Persons, Qualified Persons or Experts. This impression was either gained first hand or was assumed through the (known and assumed) professional standing and/or reputation of the organisations they represented.

4 PROPERTY DESCRIPTION, LOCATION, TENURE & ENCUMBRANCES

This Report describes Steppe's Property in Eastern Mongolia called Altan Tsagaan Ovoo (ATO). In mining terms the Property is defined by Mining License MV-017111.

The bulk of the information in this Section is extracted from the 2021 NI 43-101 Report[8] and has been reviewed and/or amended by Steppe Gold's Greg Wood.

4.1 Surface area

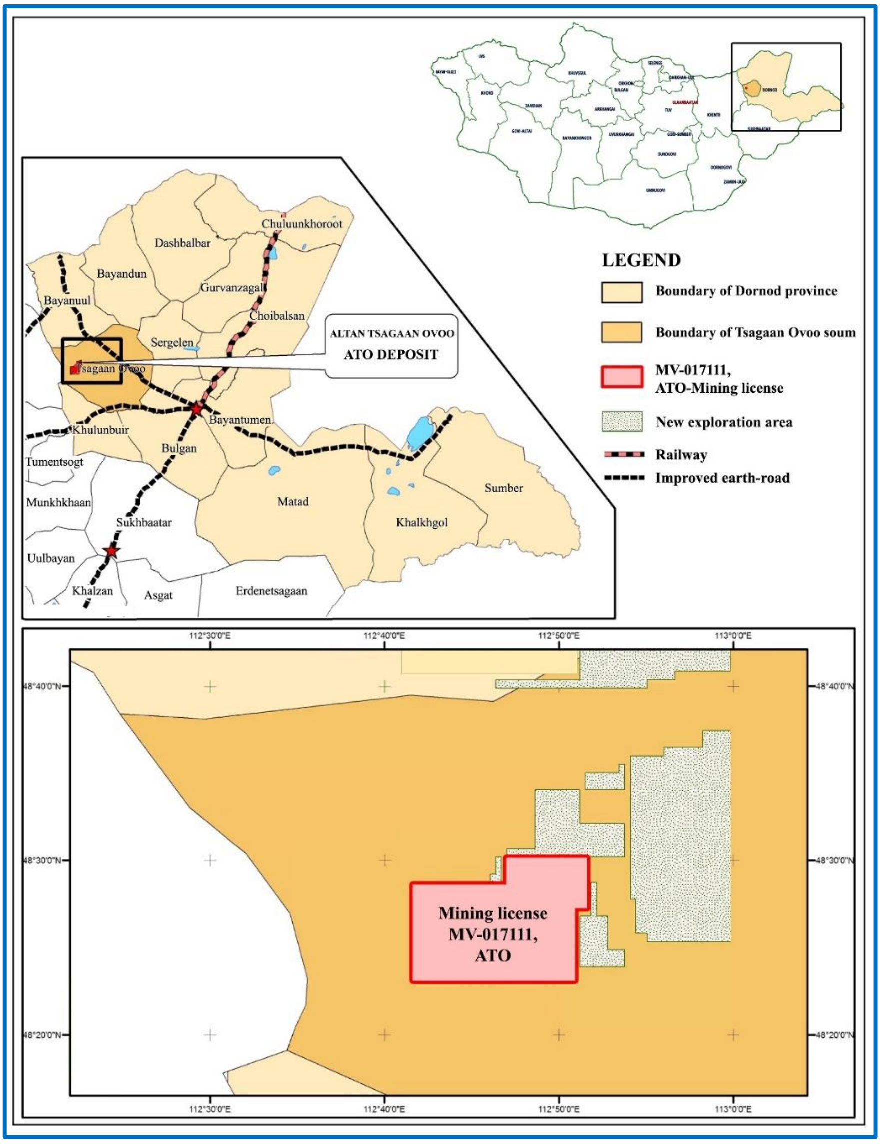
Surface area of the Mining Licence MV-017111 is 5,492.63 ha or ~55 km^2 (1 ha = 10,000 m^2). The immediate Project area of the four deposits is ~2 km^2, with dimensions ~1.4 km east/west * ~1.2 km north/south.

4.2 Location & coordinates

Figure 4.1 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 provincial capital Choibalsan (centre of radial black roads), and ~38 km west of the closest town Tsagaan Ovoo Soum.

[8] 2021 NI 43-101 Report, Section 4, pp22 onwards.

Figure 4.1 - ATO Project Regional Location



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.

4.3 Mineral Tenure

Mineral tenure has seen a steady movement from large to reduced Exploration Licenses and then similarly reducing Mining Licenses.

- Exploration License 6727X was issued to Coge Gobi LLC (CogeGobi) in 2003 of 109,118 ha.
- Subsequently 85,500 ha were relinquished in 2007, leaving 23,597 ha.
- The Exploration License was transferred to Centerra Gold Mongolia LLC (CGM) in 2010.

- CGM obtained a Mining License (MV-017111) in August 2012 of 11,614 ha. The duration of the License was 30 years, to August 2042. The boundary had 19 corner points.
- Subsequent simplification of the Mining Licence reduced it to 8 corner points and area of 5,492.63 ha.

4.4 Issuer's title – to the Property

Mining License: The ATO Project comprises one mining licence (MV-017111) over an area of 5,492.63 ha. The ATO Project is located in the territory of Tsagaan Ovoo soum, Dornod province of Eastern Mongolia, 660 km east of the Ulaanbaatar, the capital of Mongolia, 120 km northwest of Choibalsan, the provincial capital of Dornod Province, 38 km west of Tsagaan Ovoo soum. The geographic zone of ATO Project is in datum WGS-84 Zone 49N of UTM coordinate system and the geographic centre of the property is 48°26'N latitude and 112°46'E longitude.

Surface rights / ownership: Pursuant to the Licence Purchase Agreement dated January 31, 2017, Steppe Mongolia acquired the ATO Project from CGM on September 15, 2017. The ATO mining licence was registered in the name of Steppe Mongolia effective on September 5, 2017. Steppe Mongolia now has legal ownership of 100% of the ATO Project licences.

Property access: Access to the property from Ulaanbaatar is possible by highway to Choibalsan, the capital of the Dornod Province, and then by improved unpaved roads to Tsagaan Ovoo soum. The property is connected to other settlements by dirt roads. It is possible to fly to Choibalsan from Ulaanbaatar via domestic airlines.

Other Property obligations: The Author QP is unaware of the Project's status on this.

4.5 Royalties, agreements & encumbrances

Royalties: Mineral production in Mongolia is subject to fixed and sliding scale government royalties. The production of gold is subject to a flat rate 2.5% royalty and an additional sliding scale royalty up to 5% based on the market price of gold. In 2014 the Mongolian Minerals Law was amended to provide that for gold sold to the Mongolian central bank or Mongolian licenced commercial bank the royalty rate would be reduced to a flat rate of 2.5%. The amendments are in effect until January 2019. Silver production is subject to a 5% flat rate royalty and an additional sliding scale royalty of up to a maximum of 5%. The 5% maximum royalty is reached when the price of silver is US$45/oz or higher. As a result of the Company's participation in the Gold-2 Programme with the Government of Mongolia, the Company does not expect to be subject to the standard Mongolian royalty regime. Under the Gold-2 Programme, gold royalties payable to the Government of Mongolia are capped at 2.5%, and silver royalties are capped at 5%. On March 27, 2019, the Parliament of Mongolia passed certain amendments to the Minerals Law of Mongolia to increase the royalty payable on gold from 2.5% to 5% and confirmed that the sliding-scale additional royalty would not apply.

The ATO Project is also subject to a 1.75% net smelter returns royalty in favour of CogeGobi on all minerals produced from the project after the commencement of commercial production. The royalty provides that the percentage shall be adjusted inversely to any change in the rate of Mongolian government royalties after the date of the agreement, being March 26, 2010.

The Company is also obligated to sell a certain portion of the production of gold and silver from the ATO Project to Triple Flag Mining Finance Bermuda Inc. pursuant to the August 11, 2017, metals purchase and sale agreement pursuant to which Steppe agreed to sell to Triple Flag Mining Finance Bermuda Inc a portion of the gold and silver produced from the ATO Project in consideration of Triple Flag Mining Finance Bermuda Inc agreeing to provide an upfront deposit against the purchase price for the gold and silver of US$23 million in two US$11.5 million tranches.

A royalty at the rate of 5% is payable in respect of the sales value of all products extracted pursuant to a Mining Licence (other than domestically sold coal and construction minerals) that are sold, shipped for sale, or otherwise used. A portion of this 5% royalty rate goes to the central treasury, while the remaining portion goes to local authorities. The sliding scales in Clause 47.3.5 of the Minerals Law are given in Table 4.1

Table 4.1 - Mongolian Government sliding scale royalty for Au and Ag

Gold price ($/oz)	Royalty	Silver ($/oz)	Royalty
0-900	0%	0-25	0%
900-1000	1.0%	25-30	1.0%
1000-1100	2.0%	30-35	2.0%
1100-1200	3.0%	35-40	3.0%
1200-1300	4.0%	40-45	4.0%
1300 and above	5.0%	45 and above	5.0%

Table 4.2 - Mongolian Government sliding scale royalty for Pb and Zn

Lead price ($/t)	Royalty		
	Ore	Concentration	Production
0-1500	0%	0%	0%
1500-1800	1.0%	0.8%	0.4%
1800-2100	2.0%	1.6%	0.8%
2100-2400	3.0%	2.4%	1.2%
2400-2700	4.0%	3.2%	1.6%
2700 and above	5.0%	4.0%	2.0%
Zinc ($/t)	Royalty		
	Ore	Concentration	Production
1-1500	0%	0%	0%
1500-2000	1.0%	0.8%	0.4%
2000-2500	2.0%	1.6%	0.8%
2500-3000	3.0%	2.4%	1.2%
3000-3500	4.0%	3.2%	1.6%
3500 and above	5.0%	4.0%	2.0%

4.6 Environmental liabilities

Operation: Under the Environmental Protection Law of Mongolia (the EPL) business entities and organizations have the following duties, including, but not limited to, with respect to environmental protection:

A. Comply with the EPL and the decisions of the government, local self-governing organizations, local governors and Mongolian state inspectors.
B. Comply with environmental standards, limits, legislation and procedures and to supervise their implementation within their organization.
C. Keep records on toxic substances, adverse impacts, and waste discharged into the environment.
D. Report on measures taken to reduce or eliminate toxic chemicals, adverse impacts, and waste.

The Minerals Law and the Law on Environmental Impact Assessment (EIA) state that the Mining License holder shall prepare and submit an environmental impact assessment report to relevant government authority for its approval.

According to the Minerals law (Clause 35.3). the Company had engaged Midas Mining to commence updated feasibility study based on the Mongolian guidance as “Preliminary assessment of mineral mining potential, general requirements for feasibility study of mining projects and regulations for submitting feasibility study” in 2018.

In accordance with Clause 4.14 of Environmental impact assessment procedure, addendum assessment was done on Detailed environmental impact assessment and it was approved by professional commission of Ministry of Environment and Tourism.

In accordance with Law on Environmental Impact Assessment and Procedure of Environmental management plan development, revision, approval, the Company is developing and obtaining approval on Annual Environmental management plan and fulfilling its implementation.

Closure and reclamation: Mine closure and rehabilitation activities need to be planned prior to mine operation commencement. Therefore Steppe Gold has developed its mine closure conceptual plan in 2019. This conceptual plan regularly gets updated as new information becomes available. The Company develops and submits Asset Retirement Obligation /ARO/ Report quarterly basis.

ATO mine site will turn into grassland once rehabilitated and local perennial plant species will be planted as part of mine restoration activities. Since the beginning of mine operation, company conducts different studies to ensure effective closure and rehabilitation.

In accordance with this research, the company built “Tree nursery and trial plantation”, and is planting 10 types of tree and bush in 2 different timelines using 3 different planting method to research and analyse the results and choose suitable method for future rehabilitation from these varieties. Also the Company is collecting aboriginal perennial plants seeds and using them on topsoil stockpile in accordance with relevant standard to research and analyse the results.

4.7 Permits Required

Approval for commencement of mining: Since the 2017 NI 43-101 report mining has commenced.

Land possession right: The Mining License holder can request to use or possess land for the mine claim, camp and other permanent facilities for its mining operations. Only Mongolian citizens and legal entities are entitled to hold a land possession right (Clause 32.1 of Land Law). Land use or possession rights are evidenced by a Land Use or Possession Certificate issued by a local authority of the soum (district) in which the relevant land is located. The terms and conditions of land use or possession rights are governed by a land possession agreement entered into with the land division of a local authority of the relevant soum (district).

Water Use Permit: The Company has obtained the "Water reserve conclusion" from Mongol-Us SOE on 2018 in accordance with Mongolian Law on Water, therefore, obtaining Water usage conclusion from either Environment and Tourism Agency of Dornod province or Mongol-Us SOE depending on the amount of water to be used and making Water usage contract with either Tsagaan-Ovoo soum or Kherlen River Basin Administration.

Energy and Powerline permits: The project area is located in a moderately developed area in Mongolia, but in the recent years, mineral exploration has activated for gold, base metal, coal and oil prospects around this area and tends to become a major economic and infrastructure region. Currently, Steppe Gold LLC provides ATO projects reliable electricity to its customers as processing plant, crusher and camp's electricity supplied by CATERPILLER diesel generator and annual electricity consumption estimation is 6,100 kW. During the entire project operation, there will have to be three electricity consumers as:

1. Camp
2. Pit
3. Processing plant
    a. Oxidized ore - 1-5 year
    b. Fresh ore - 6-11 year

During the entire period of fresh ore mining ATO projects total the total electricity capacity will be 13,262.4 кW.

Construction permit: The Company has built required facilities for heap leaching of oxidized ore as water supply system, heap leach pad, processing plant, fuel station, boiler house and chemical storage and has been commissioned recently.

1. Mining permission
2. Processing permission
3. Permissions related to the commissioning of the construction /Camp, Processing plant, Chemical storage, crusher and heap leach pad/
4. Chemical permissions

Mine drilling and blasting performed by the blasting contractor company named as "Special Mining Service" LLC.

4.8 Other issues influencing the right or ability to operate

Socio-economic impacts: As an example of the Company's (and predecessor) successful ability to operate in Mongolia its recent past operations are detailed briefly here. Centerra Gold Inc. (CGI) is the parent company of Mongolia based CGM which has had assets in Mongolia. CGM owns a 100% interest in the Boroo Mine, a 100% interest in the Gatsuurt property, and previously owned a 100% interest in the ATO Property. Boroo is an open-pit operation and Mongolia's largest hard-rock gold mine. It began commercial production in 2004 and produced approximately 1.8 million ounces of gold from March 2004 through to December 2016. Boroo is operated through CGI's Mongolian subsidiary, Boroo Gold Company. The operations at the Gatsuurt property are pending due to suspended negotiations with the Mongolian Government. The Project will begin ore production on receipt of the final approvals and permits (Centerra

Gold Inc. 2011a). The Gatsuurt property is being currently advanced by CGM. CGM owned 100% interest in the ATO Project before it was acquired by STEPPE in 2017.

ATO project puts special attention on society-economy issues therefore, developed Sustainable development policy and implementing it in its operations. In this policy, local community relation, human resources, safety and environment was included specifically therefore, developed a good management system.

Community policy, planning, consultation and engagement: This Social Baseline Assessment has provided useful preliminary information for the planning of Project developments by CGM and it will provide Steppe with an effective pathway for conducting additional public consultations with local communities. The Alternate QP is not aware of any environmental liability or significant issues to affect development of the property. Careful Project planning, including the implementation of an international-style social and environmental impact assessment (SEIA), a Mongolian EIA, and a land use management and reclamation plan, will be important components for the success of the project.

There are several key observations drawn from this Social Baseline Assessment, based on survey work in the field, research and analysis, and also from the expressions of concern and interest by the local people. These observations include:

- There are archaeological sites on the ATO LA and sacred places located in the general area:
    - sacred places must be fully avoided and protected from any sort of Project activities;
    - archaeological sites should also be avoided. If absolutely necessary to use the area for mining and/or infrastructure, archaeological excavations should be planned and conducted prior to the mining of the site.
- New jobs will be created that will provide opportunities for local people, including possibly youths. The local people believe that:
    - use of local labour and the conditions for initial training should be investigated by Steppe;
    - a hiring policy that optimizes the opportunity for hiring a local labour force should be adopted by Steppe, and proper training programs should be developed.
- Regular communications with local authorities and local people will be vital to the success of the Project. To address this important aspect of the Project, it is recommended that Steppe should:
    - establish a local community liaison office;
    - conduct regular communications with all stakeholders with particular attention paid to vulnerable groups (nomadic communities, retired people, others);
    - develop mutually acceptable communication rules with the local people.
- Participation/sponsorship in local infrastructure development and community development projects by Steppe, as well as a safe driving program, are important issues for the local people.

5 PROPERTY GEOGRAPHY

The geography of the Property is described in terms of:

- General topography, elevation, vegetation and wildlife.
- Access to the site and nature of transport.
- Proximity to population centres.
- Climate and operating season.
- Infrastructure suitable for mining operations.
- Natural risks.

The bulk of the information in this Section is extracted from the November 2021 feasibility study by DRA Global Limited (DRA) disclosed in a November 2021 NI 43-101 report and has been reviewed and/or amended by Steppe's personnel.

5.1 Topography (general)

Topography: Geographically, the license area is located 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.

Elevation: Average elevation is 980 – 1,050 m above sea level, with the lowest point being Deliin Well (979.3 m) and the highest point being Mount Temdegt (1,144.7 m). Relative elevation is 60 – 120 m.

Vegetation: Mostly, brown and black brown gravel, sandy loam, and gravel-mild clay of steppe zone are predominant. Quaternary loose sediments are widespread in the license area where there are no trees. Mounds and hills in the area are bald and rounded. Vegetation and grass cover the entire area and includes pasture plants such as khazaar grass, wormwood, stipa, brome-grass, and couch grass.

Wildlife: 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.

Existing land use: The land surrounding the Property is predominantly used for nomadic herding of goats, cows, horses and sheep by small family units. Use is based on informal traditional Mongolian principles of shared grazing rights with limited land tenure for semi-permanent winter shelters and other improvements.

5.2 Access & nature of transport

Road: The ATO property is situated in the western side of Mongolia, 660 km east of the city of Ulaanbaatar, 120 km northwest of Choibalsan city and 38 km west of Tsagaan Ovoo soum. Access to the property from the Mongolian capital, Ulaanbaatar is possible by highway to Choibalsan, the centre of the Dornod province and improved unpaved road continue to Tsagaan Ovoo soum. The property is connected to other settlements by dirt roads.

Air: It is possible to fly to Choibalsan from Ulaanbaatar via domestic airlines.

5.3 Proximity to population centres

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 engages in domestic animal husbandry. Some of them run plantation and grow vegetable for their own household uses. The central village accommodates administrative offices, a cultural centre, secondary schools, a hospital, a kindergarten, a communications centre, cellphone stations, a gas station, and high-voltage sub-stations.

5.4 Climate & operating season

Climate summary: Climate[9] of the property region is characterized by extreme cold and hot weather. Wide daily, monthly, and yearly fluctuations of temperature are common. Winter is harsh and cold and extends from November to March. Coldest temperature reaches to -46˚C near the river valleys and flatlands in January. Stable snow cover persists from November to March. Freezing of soil starts from mid-September and continues till late May, with the freezing depth reaching 2.5 m. Spring begins in late March and continues till early June, and it is characterized by fluctuant atmospheric temperature and air dryness and strong wind. Summer is shorter than other seasons, dry and chilly. The hottest temperature is up to +40˚C in summer. 60-80% of the annual precipitation falls as rain during July and August. Number of days with precipitation is 59 days in average in a year. Average wind speed is 4-8 m/s, with dominant directions from north-west, north and north-east and maximum speed reaches 20-22 m/s.

Operating season: ATO Mine will operate all year around.

5.5 Infrastructure for mining

Water: Water network of the area belongs to the Pacific Ocean basin. Small local rivers with short-lived stream fed from eastern branch mountains of the Khentii Range flow into small lakes. Sizes of these rivers vary depending on their main source of water collection which is precipitation. Drinking water could only be taken 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 been dried up due to a global warming and decrease of precipitation. In the summer time, seasonal springs found from the melt of small patchy permafrost in small intermountain valleys and from seasonal thawing of frozen ground.

[9] 2009. *National Atlas of Mongolia*.

Land Use. The land surrounding the property is predominantly used for nomadic herding of goats, cows, horses and sheep by small family units. Use is based on informal traditional Mongolian principles of shared grazing rights with limited land tenure for semi-permanent winter shelters and other improvements

Risk Assessments. 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 license 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.

Closure and Reclamation. As part of overall project planning, a preliminary reclamation and closure plan was prepared. Certain features of the mine, such as the open pit, waste dumps, and tailings impoundment, will create permanent changes to the current landscape that cannot be completely remedied through reclamation. The closure plan will; however, ensure that these disturbed areas are seismically and chemically stable as to limit the ecological impacts to the surrounding water, air, and land.

5.6 Natural risks

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 license 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

The history of the Property is described in terms of:

- Prior ownership and initial exploration.
- Previous exploration leading to discovery.
- Historical Resource and Reserve estimates – by CGM and Steppe.
- Historical and recent mine production.

The bulk of the information in this Section is extracted from the 2017 NI 43-101 Report[10] and has been reviewed by Steppe's personnel.

6.1 Prior ownership & initial exploration

The area of immediate interest at ATO is centred on three, at most 200 m high, gently rounded hills (herein termed Hills 1, 2 and 3 from SE to NW) that are surrounded by widespread unconsolidated Quaternary deposits. Presently, the known most intensely mineralized rocks underlie Hills 1 and 2, respectively underlain by Pipes 1 and 2, as well as the completely covered Pipe 4. Low gently rounded slopes or somewhat steeper inclines with moderate slopes characterize the surrounding ranges.

Regional geologic field parties working for CogeGobi were the first to recognize mineralized rocks cropping out at ATO. In 1997 CogeGobi (a wholly owned subsidiary of the French multinational company AREVA) began their exploration efforts in eastern Mongolia. After a six year reconnaissance effort, CogeGobi settled on a selected exploration region in 2003 and then obtained eight exploration licenses in eastern and south-eastern Mongolia. CogreGobi then embarked upon a four-year-long concerted effort in the search for viable gold and uranium deposits in all eight of their licensed areas. Two of their licenses (3,425.5 km^2 in all) were in the general area of ATO. As part of that effort, CogeGobi geologists eventually collected 52 grab samples from outcrops and sub-crop at ATO and identified anomalous gold concentrations in vein quartz-rich rock there (0.06 to 27.8 g/t Au). Those sample results initially led to a stream sediment sampling program. Almost all Au–anomalous samples collected by CogeGobi were from Hills 1 and 2. CogeGobi then proceeded to describe the occurrence at ATO as "intense hydrothermal alteration associated with volcano-plutonic structures"[11].

At the end of a four-year-long exploration cycle by CogeGobi in their eight licensed areas (which was primarily focused on gold deposits but also included uranium) uranium prices skyrocketed in mid-2007 from \$30/lb U_3O_8 to approximately \$140/lb. AREVA subsequently acquired East Asia Minerals Energy Company in September 2007 and became AREVA Mongol in 2008. As of 2017 AREVA Mongol had a 100 percent interest in CogeGobi. Because of the uranium economics AREVA Mongol's interest shifted from precious metals to the energy sector, and one of their two exploration licenses surrounding ATO was dropped. At the time the CogeGobi geologists were probably discouraged by the economic potential of ATO due to their belief that all high gold-bearing rocks resided in a small volume (i.e., low tonnage) of near-

[10] 2017 NI 43-101 Report, Section 6, pp29 onwards.

[11] Hocquet, 2005.

surface mineralized quartz veins above a shallow-dipping contact with underlying low-to-nil gold-bearing rocks[12]. Subsequent exploration efforts at ATO by CGM proved this concept incorrect.

The Company acquired the ATO Project from CGM on September 15, 2017 pursuant to agreements entered into between the parties on January 31, 2017 for aggregate consideration of US$19,800,000 plus S$1,980,000 of VAT. The Company satisfied the purchase price through the payment of US$9,800,000 in cash, the issuance by Steppe Mongolia to CGM of two US$5 million principal amount non-interest bearing secured promissory notes due September 30, 2018 and September 30, 2019 (the "Purchase Notes") and a US$1 million promissory note due October 9, 2017 in respect of VAT (the "VAT Note"). The cash portion of the purchase price was satisfied with a portion of the Initial Upfront Deposit made by Triple Flag Bermuda under the Stream Agreement, which was funded concurrently with the closing of the ATO Acquisition on September 15, 2017.

The parties also entered into a guarantee agreement dated January 31, 2017 (the "Guarantee") pursuant to which the Company guaranteed all of the obligations of Steppe Mongolia under the Purchase Agreement and Steppe Mongolia agreed to assume the obligations of CGM for the 1.75% net smelter returns royalty payable to CogeGobi in respect of products extracted from the ATO mining licence.

On the closing of the ATO Acquisition, the Company, Steppe Mongolia and CGM entered into a share pledge agreement whereby the Company pledged its shares of Steppe Mongolia to secure its guarantee of the obligations of Steppe Mongolia under the Notes. As the Steppe Mongolia shares were concurrently pledged to Triple Flag Bermuda as collateral to secure the Company's obligations under the Stream Agreement, CGM and Triple Flag Bermuda entered into a security sharing agreement dated September 15, 2017 to establish their respective rights in respect of the shares in the event of default (the "Security Sharing Agreement").

The Company has repaid the VAT Note that was due October 9, 2017 and the US$5 million Purchase Note that was due September 30, 2018.

6.2 Previous exploration leading to discovery

CGM geologists were invited by AREVA Mongol (CogeGogi owner) to visit ATO in the fall of 2009. In May 2010 CGM obtained from AREVA Mongol an initial agreement to explore the one remaining licensed area (Tsagaan Ovoo) encompassing ATO. All rights to the property were obtained by CGM late in 2010. AREVA retains a 1.75 percent NSR in the Tsagaan Ovoo license. In 2017 Steppe acquired 100% of the ATO Project from CGM.

Key outcrops in the ATO property were at Hill 1 where gold-bearing silica sinter is now considered to represent a paleo hot spring environment (previously, to the best the knowledge, unknown in Mongolia). It forms a silica cap to underlying base and precious metal mineralized rock, basically at a Mesozoic paleo surface. Reticulated mats of silicified reeds, feathery in longitudinal section and oval in cross section, are

[12] Hocquet, 2005.

well preserved in this sinter at ATO (see below). The sinter is highly phosphatic, > 1,500 ppm phosphorous, all of which calls to mind features forming today at the surface at the Yellowstone geyser field.

But most importantly, siliceous sinter at ATO includes free gold. Silica cap includes mostly quartz (sparse chalcedonic fabrics are preserved; bladed silica locally replaces calcite) and variable concentrations of iron oxide minerals. Less abundant are kaolinite, illite, numerous primary and a number of secondary lead (Pb) minerals. Secondary Pb minerals include Pb–Al phosphate plumbogummite; pyromorphite (also a Pb phosphate); and Pb–Mn oxide minerals. Sparse galena in siliceous sinter was encapsulated in quartz, as is rare sphalerite, arsenopyrite, and argentite. Barite was present in narrow micro veins, as well as large tabular crystals in open cavities.

Even during CGM's first brief visit to the property, it became clear that exposures of the mineralized 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 mineralized rock. Four grab samples collected by CGM (1.3 to 3.3 g/t Au) during its initial visit to the property confirmed the presence of the anomalous Au originally reported at the ATO site by CogeGobi.

Beginning in May 2010 and through to December 2014 CGM designed significant amount of 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.

CGM's drilling comprised 597 holes for a total of 63,866 m. Of that 370 holes for 54,425 m were core holes.

As a result of CGM's exploration efforts (and up to the time of Steppe's 2017 NI 43-101 Report) it had become readily apparent that the bulk of the known precious and base metal mineralization at ATO was in **three** carrot-shaped vertically downward plunging, presumably Jurassic, breccia pipes. Two of the pipes (Pipes 1 and 2 respectively) underlie Hills 1 and 2. The third pipe (Pipe 4) is completely concealed to the SSE of Pipe 21 (Figure 7.3).

Since 2017 Steppe has undertaken considerable exploration, primarily through very extensive phased targeted drilling programs. This has confirmed the existence of a **fourth** deposit (Mungu) to the north-east of the first three. Mungu is north east trending sub-vertical structure controlled precious mineralization host in dacitic volcano-sedimentary rocks of Permian age. Mungu ore body is plunging downward to the north east and continued about 800m along the structure.

The recent exploration history at ATO, thus, comprises four stages:

1) Initial work by CogeGobi / AREVA.
2) A due diligence stage by CGM that also included soil geochemistry and limited IP.
3) Post May 2010 comprehensive exploration by CGM.
4) Post 2017 extensive drilling by Steppe.

Commencing in 2018, the Company (Steppe) commenced a three phase exploration drill program.

The 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 mineralization. In addition to this, new high grade zones in deeper parts of the deposit were discovered that increased the understanding on the controls of mineralized structures and associated feeder zones.

The Phase 2 drill program focused on the ATO4 end of the ATO4-Mungu trend at the ATO Project and of exploration activities 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.

The Phase 3 drilling program targeted at the ATO4-Mungu trend has commenced with 8 drill holes being complete for 2,228 m of drilling[13].

In 2019 the Company completed a drill program with two diamond core drilling rigs focused 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 metres at ATO1, 1,662 metres at ATO2, 14,760 metres at ATO4 and over 26,573 metres have been drilled at the Mungu Discovery[14].

Commencing 2020, the Company drilled an additional 55 drill holes for a total of 18,200 m. A total of 53,000 m have been drilled since 2018[15]. The drilling information was used to update the interpretation of the geologic model, geometry of the mineralized zones and domain.

6.3 Historical Resource and Reserve estimates

Exploration report 2012: Prior to CGM's 2016 Resource reporting (below) an exploration report on the ATO Project was prepared in 2012 for Mongolian jurisdiction. It is **not** known if that report contained Resources. That report was not prepared under compliance with NI 43-101 and the authors of the report appear not to have been qualified. Although that report was not NI compliant most of the information and studies reported were prepared similar guidelines or standards to NI 43-101.

CGM Resources 2016: Centerra Gold Inc. (CGI, owner of CGM) reported a Mineral Resource summary at ATO in its 2016 Annual Information Form on SEDAR in May 31, 2017 (Table 6.1). The combined Measured and Indicated Resource was **18.6 Mt at 1.3 g/t gold** (771 koz contained metal). An Inferred Resource of 0.4 Mt at 0.6 g/t gold was also reported. It is not know what lower grade cut-off was used, or what density was used. And it was also not stated in the Table if the reporting was JORC or NI 43-101 compliant.

[13] 2019 AIF and 2018 news releases.

[14] 2019 AIF and 29 July 2020 news release.

[15] 21 February 2021 news release.

Table 6.1 - CGM 2016 Mineral Resources[16]

Table 1 (see additional footnotes page 30)
Centerra Gold Inc. 2016 Year-End Mineral Reserve and Mineral Resource Summary – Gold [(1) (6)]
(as of December 31, 2016)

	Measured and Indicated Mineral Resources [(2)]								
	Measured			Indicated			Total Measured and Indicated		
Property	Tonnes (kt)	Grade (g/t)	Contained Gold (koz)	Tonnes (kt)	Grade (g/t)	Contained Gold (koz)	Tonnes (kt)	Grade (g/t)	Contained Gold (koz)
ATO [(5)]	9,663	1.5	465	8,920	1.1	306	**18,583**	**1.3**	**771**

Inferred Mineral Resources [(3)]			
Property	Tonnes (kt)	Grade (g/t)	Contained Gold (koz)
ATO [(5)]	388	0.6	8

1) Centerra's equity interests as of this AIF are as follows: Mount Milligan 100%, Kumtor 100%, Gatsuurt 100%, Boroo 100%, Ulaan Bulag 100%, ATO 100%, Öksüt 100% and Greenstone Gold properties (Hardrock, Brookbank, Key Lake, Kailey) 50%.
2) Mineral resources are in addition to mineral reserves. Mineral resources do not have demonstrated economic viability.
3) Inferred mineral resources have a great amount of uncertainty as to their existence and as to whether they can be mined economically. It cannot be assumed that all or part of the inferred mineral resources will ever be upgraded to a higher category.
4) Royal Gold streaming agreement entitles Royal Gold to 35% of gold sales from the Mount Milligan mine. Under the stream arrangement, Royal Gold will pay $435 per ounce of gold delivered.
5) As of January 31, 2017, Centerra Gold's Mongolian subsidiary, Centerra Gold Mongolia (CGM) entered into definitive agreements to sell the ATO Project, located in Eastern Mongolia, to Steppe Gold LLC and Steppe Gold Limited. The transaction is subject to conditions.
6) Numbers may not add up due to rounding.

Centerra Gold Inc.
2016 Annual Information Form

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The CGM Resources were superseded by the newly estimated ones reported in the 2017 NI 43-101 Report itself.

Steppe Resources 2017: Steppe reported newly estimated Resources in the 2017 NI 43-101 Report compiled by GSTATS. In the three tabulations are incorrectly labelled – the upper tabulation should read "ATO Mineral **Oxide** Resource", the lower two tabulations should read "ATO Mineral **Fresh** Resources"]. The Resources were for the three Pipes 1, 2 and 4. The Recourses were fully JORC and NI 43-101 compliant.

The total combined Measured and Indicated Resources were:

Oxide:	**5.3 Mt at 1.23 g/t gold** (208 koz contained metal). Lower cut-off 0.30 g/t AuEq.
Fresh:	**12.4 Mt at 1.50 g/t gold** (594 koz contained metal). Lower cut-off 1.1 g/t AuEq.
Total:	**17.6 Mt at 1.42 g/t gold** (803 koz contained metal)

An Inferred Resource of 1.3 Mt at 0.97 g/t gold was also reported.

[16] 2017 NI 43-101 Fig 6.1, pp30

Table 6.2 - Steppe 2017 Mineral Resources[17] – Oxide & Fresh – Pipes 1, 2 & 4

ATO Mineral Reserves (1)(3)(5)(6)(7)(8)(9)(10)(12)

	Tonnage	Au	Ag	Contained Metal		Recovered Metal	
				Au	Ag	Au	Ag
	(mt)	(g/t)	(g/t)	(koz)	(koz)	(koz)	(koz)
Proven	3.41	1.41	9.72	155	1,065	108	426
Probable	1.82	0.93	10.52	55	616	38	246
Proven and Probable	**5.23**	**1.25**	**10.00**	**210**	**1,681**	**147**	**673**

ATO Mineral Resources (1)(2)(4)(5)(8)(9)(10)(12)

Measured and Indicated Mineral Resources

	Tonnage	Au	Ag	Pb	Zn	Contained Metal			
						Au	Ag	Pb	Zn
	(mt)	(g/t)	(g/t)	(%)	(%)	(koz)	(koz)	(mlb)	(mlb)
Measured	7.77	1.51	8.16	0.86	1.54	378	2,039	148	263
Indicated	4.46	1.46	13.17	0.55	1.01	209	1,889	54	99
Measured + Indicated	**12.23**	**1.49**	**9.99**	**0.75**	**1.34**	**587**	**3,927**	**202**	**362**

Inferred Mineral Resources (11)

	Tonnage	Au	Ag	Pb	Zn	Contained Metal			
						Au	Ag	Pb	Zn
	(mt)	(g/t)	(g/t)	(%)	(%)	(koz)	(koz)	(mlb)	(mlb)
Inferred	1.05	1.03	25.18	0.52	1.11	35	848	12	26

Notes:

1. ATO Mineral Reserves and Mineral Resources are as at August 21, 2017 using the CIM Definition Standards (2014).
2. Mineral Resources are in addition to Mineral Reserves.
3. Mineral Reserves are constrained within an optimized pit shell based on a gold price of $1,300 per ounce.
4. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability for heap leaching.
5. Contained gold estimates have not been adjusted for metallurgical recoveries.
6. Recovered gold estimates have been adjusted for metallurgical recoveries based on 70% for Au and 40% for Ag.
7. Mining dilution is 3% and Ore loss is 2%.
8. Mineral Reserves and Mineral Resources are estimated using a 0.3 g/t AuEq cut-off grade for oxide material and a 1.1 g/t AuEq cut-off grade for fresh material.
9. A conversion factor of 31.103477 grams per ounce 453.59237 grams per pound are used in the reserve and resource estimates.
10. AuEq has been calculated using assumed metal prices ($1,306/oz for gold, $21.60/oz).
    Oxide ore calculation: AuEq(g/t) = Au(g/t) + (Ag(g/t) x 21.60 x 0.0321507 x 0.4) / (1,306 x 0.0321507 x 0.7)
    Fresh ore calculation: AuEq(g/t) = Au(g/t) + ((Ag(g/t) x 21.60 x 0.0321507 x 0.75) + (Pb% x 1,844 x 10^{-6} x 0.6) + (Zn% x 1,944 x 10^{-6} x 0.55)) / (1,306 x 0.0321507 x 0.71)
11. Inferred Mineral Resources have a great amount of uncertainty as to their existence and as to whether they can be mined economically. It cannot be assumed that all or part of the Inferred Mineral Resources will ever be upgraded to a higher category.
12. Totals may not match due to rounding.

The 2017 Steppe Resources are superseded by the newly estimated ones reported in this 2021 NI 43-101 Report.

Steppe Reserves 2017: Steppe designed open-cut mines on the three Pipes and reported Reserves from them in the 2017 NI 43-101 Report compiled by GSTATS (Table 6.3). The pits were essentially designed to mine only the Oxide Resources.

The total combined Proven and Probable Reserves in the three pits, before dilution, ore loss or factored gold recovery, were:

Proven: **3.4 Mt at 1.45 g/t gold** (157 koz contained metal). Lower cut-off 0.30 g/t AuEq.

[17] 2017 NI 43-101, Table 1.5 in the Summary section 1, pp 15

Probable: **1.8 Mt at 0.96 g/t gold** (55 koz contained metal). Lower cut-off 0.30 g/t AuEq.
Total: **5.2 Mt at 1.28 g/t gold** (213 koz contained metal)

Table 6.3 - Steppe Mineral Reserves[18] – Oxide – Pits 1, 2 & 4 – before dilution

Classification	Tonnage	Au	Ag	Contained Metal	
				Au	Ag
	(mt)	(g/t)	(g/t)	(koz)	(koz)
Pipe 1 (0.3 g/t AuEq Cut-off)					
Proven	2.74	1.54	9.29	136	818
Probable	0.41	1.13	9.93	15	132
Total Reserve - Pipe 1	**3.15**	**1.49**	**9.38**	**151**	**950**
Pipe 2 (0.3 g/t AuEq Cut-off)					
Proven	0.24	0.74	5.13	6	40
Probable	0.82	0.60	4.70	16	124
Total Reserve - Pipe 2	**1.06**	**0.63**	**4.80**	**22**	**164**
Pipe 4 (0.3 g/t AuEq Cut-off)					
Proven	0.39	1.25	17.22	16	219
Probable	0.57	1.33	19.98	25	367
Total Reserve - Pipe 4	**0.97**	**1.30**	**18.85**	**40**	**586**
ATO Deposit (0.3 g/t AuEq Cut-off)					
Proven	3.37	1.45	9.92	157	1,076
Probable	1.80	0.96	10.74	55	623
TOTAL RESERVE - ATO	**5.18**	**1.28**	**10.21**	**213**	**1,700**

These Reserves were also reported with 2% ore loss, 3% dilution and 70% gold recovery (Table 6.4).

Table 6.4 - Steppe Mineral Reserves[19] – Oxide – Pits 1, 2 & 4 – after dilution and recovery

Classification	Tonnage	Au	Ag	Contained Metal		Recovered Metal	
				Au	Ag	Au	Ag
	(mt)	(g/t)	(g/t)	(koz)	(koz)	(koz)	(koz)
ATO Deposit (0.3 g/t AuEq Cut-off)						**70%**	**40%**
Proven	3.41	1.41	9.72	155	1,065	108	426
Probable	1.82	0.93	10.52	55	616	38	246
TOTAL RESERVE - ATO	**5.23**	**1.25**	**10.00**	**210**	**1,681**	**147**	**673**

Notes:

1. Estimate of Mineral Resources has been estimated by the QPs.
2. ATO Mineral Reserves are as at August 21, 2017.
3. Mineral Reserves are included in Mineral Resources.
4. Mineral Reserves are constrained within an optimized pit shell based on a gold price of $1,300 per ounce.
5. Mining dilution factor is 3% and Ore loss factor is 2%.
6. Contained gold estimates have been adjusted for metallurgical recoveries.
7. The open pit Mineral Reserves are estimated using a cut-off grade of 0.3 g/t AuEq for oxide material.
8. A conversion factor of 31.103477 grams per ounce 453.59237 grams per pound are used in the reserve and resource estimates.
9. AuEq has been calculated using assumed metal prices ($1,306.6/oz for gold, $21.6/oz).
   Oxide ore calculation: AuEq(g/t) = Au(g/t) + (Ag(g/t) x 21.6 x 0.0321507 x 0.4) / (1,306.6 x 0.0321507 x 0.7)
10. Totals may not match due to rounding.

[18] 2017 NI 43-101 Report, Table 15.3 in Mineral Reserve Estimates section 15.5, pp154
[19] 2017 NI 43-101 Report, Table 15.5 in Mineral Reserve Estimates section 15.5, pp155

6.4 Historical & recent mine production

Historical mine production: It is understood that the ATO Project had seen no past mining.

Recent mine production: Steppe Gold remained busy, working non-stop and smoothened the phase one operation of Altan Tsagaan-Ovoo project and continuously produced its products. The Altan Tsagaan-Ovoo gold and polymetal deposit has completed the first stage by constructing its oxidized ore refinery infrastructure from 2018-2019 which includes a gold and silver refinery with a capacity of 300 m^3 solution per hour, a heap leach pad with a capacity of 5.37 Mt of ore, a rich and barren solution pond with a capacity of 43,000 m^3, and camp capable of containing 250 people, and thus is ready to commence operations.

7 GEOLOGICAL SETTING & MINERALISATION

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.

The bulk of the information in this Section is extracted from the November 2021 feasibility study by DRA Global Limited (DRA) disclosed in a November 2021 NI 43-101 report.

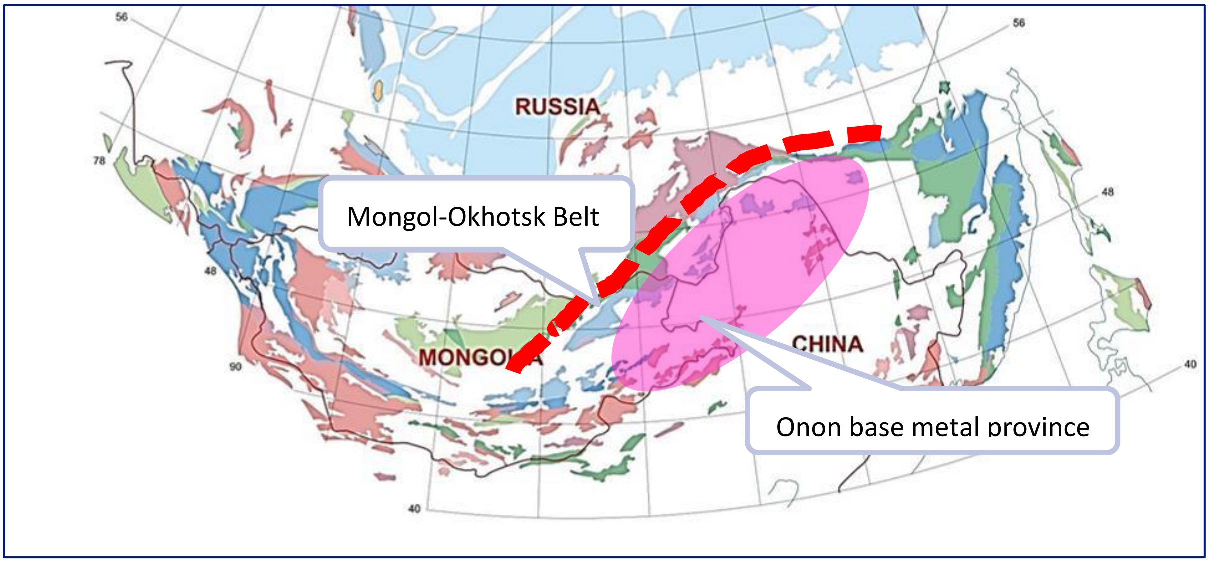
The principal author of the Report, QP Robin Rankin (geologist) has also reviewed this geological information and, notwithstanding his lack of a site visit and detailed inspection, has no argument with the geological description or mineralisation setting.

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[20]) 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).

[20] CGM 2012 Exploration Report

Figure 7.1 - Location of Mongol-Okhotsk Belt and Onon precious base metal province



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 summarized 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.

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 (MML – red dashed line in (Figure 7.1) and midway along the NNE trending 600km long Onon base and precious-metal province that crosses eastern Mongolia (pink oval in Figure 7.1).

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 mineralized 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 mineralized 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 mineralization and associated magmatism at ATO (see below).

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.

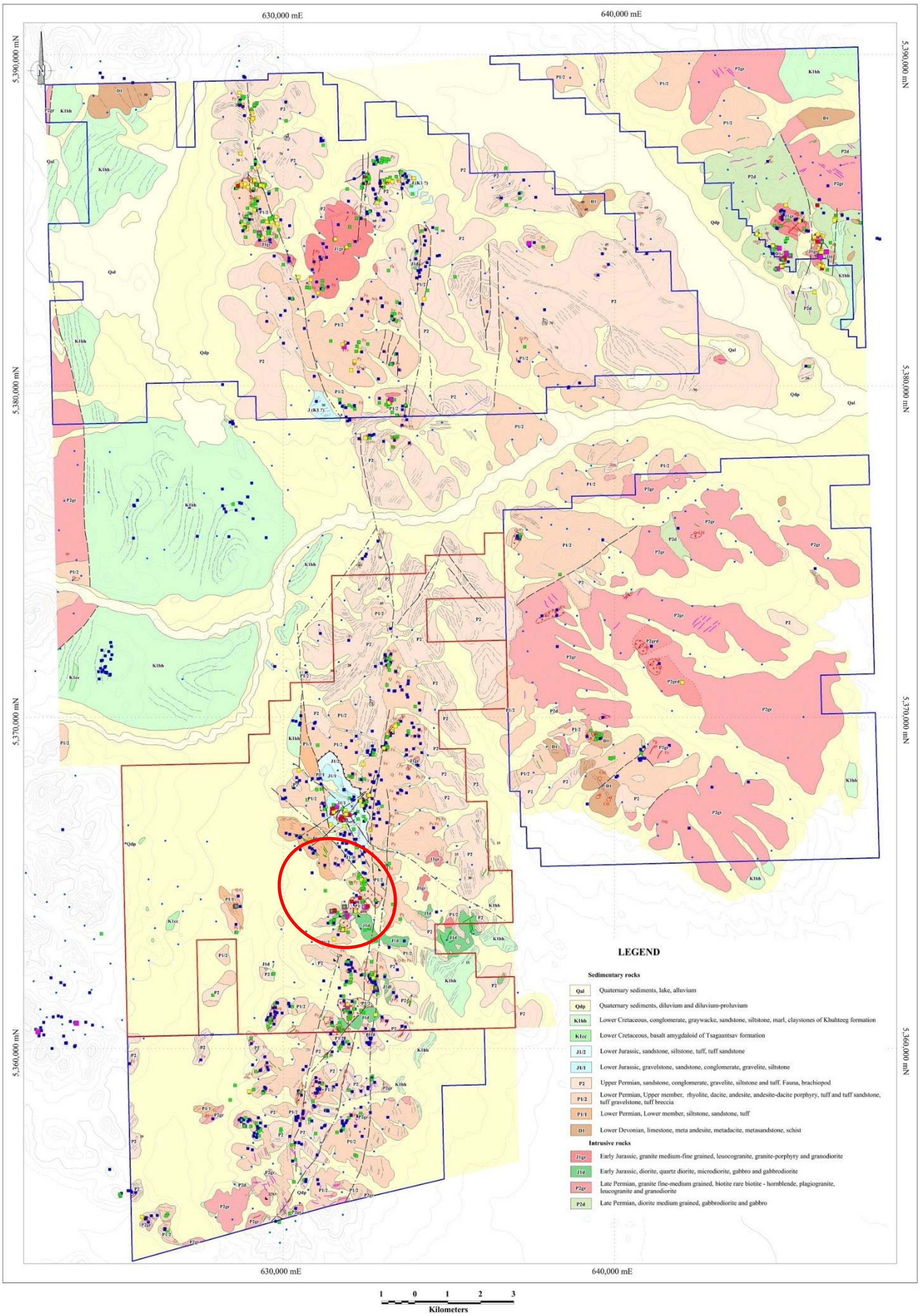
7.2 District geology and magmatism

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 mineralized occurrences close to ATO other than a presumed association with Jurassic magmatism. Furthermore the age of gold-mineralized rock has not been determined radiometrically, either at ATO or at mineralized 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).

Figure 7.2 - Geology map of ATO district



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 mineralization 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, presumably Middle-Late Jurassic gravel and coarse pebbly sandstone (as described above) 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 mineralized 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.

The Cretaceous graben reaches its maximum width of about 6 km approximately 22 km N of ATO. Coarse-grained Paleozoic granite (Early Permian) 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 mineralized 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 mineralization points as well as two gold occurrences, four gold mineralized points, three lead mineralized points, three lead-zinc mineralized 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:

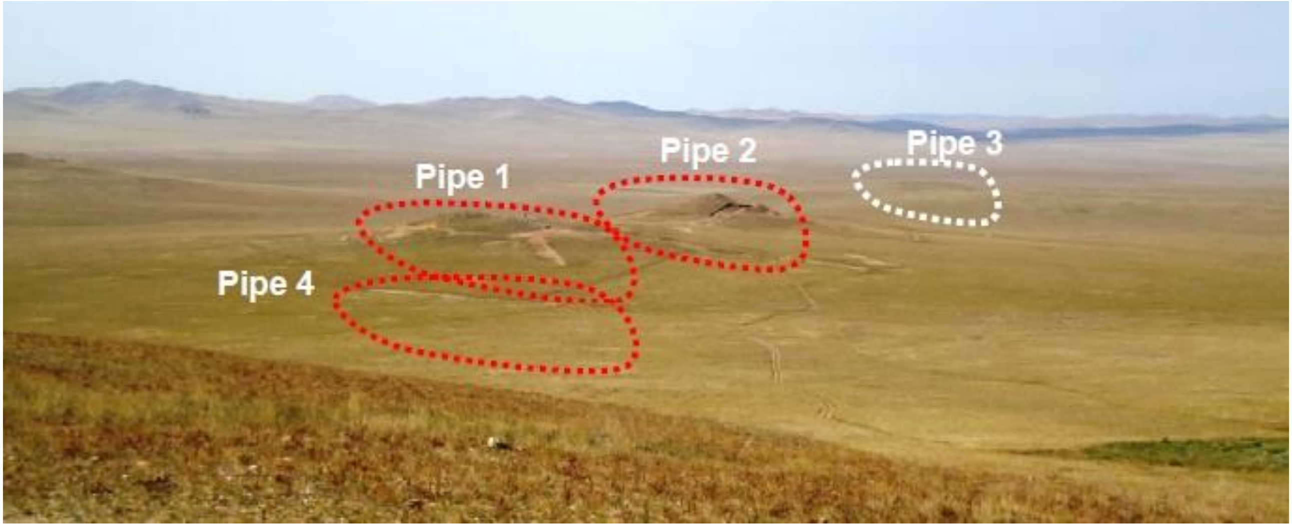
- Deposit geology.
- Deposit mineralisation controlling factors.
- Silicate alteration in the mineralised pipes.
- Silica cap to Pipe 1.
- Styles of sulphide mineralised rock at ATO.

7.4.1 ATO deposit geology

Pipes 1, 2 and 4: Up to 2017 exploration focussed on three mineralized pipes at ATO. These were named Pipe 1, 2, and 4 (now ATO1, 2 and 4) and are illustrated with red ovals in Figure 7.2 and in plan in Figure 7.3 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.

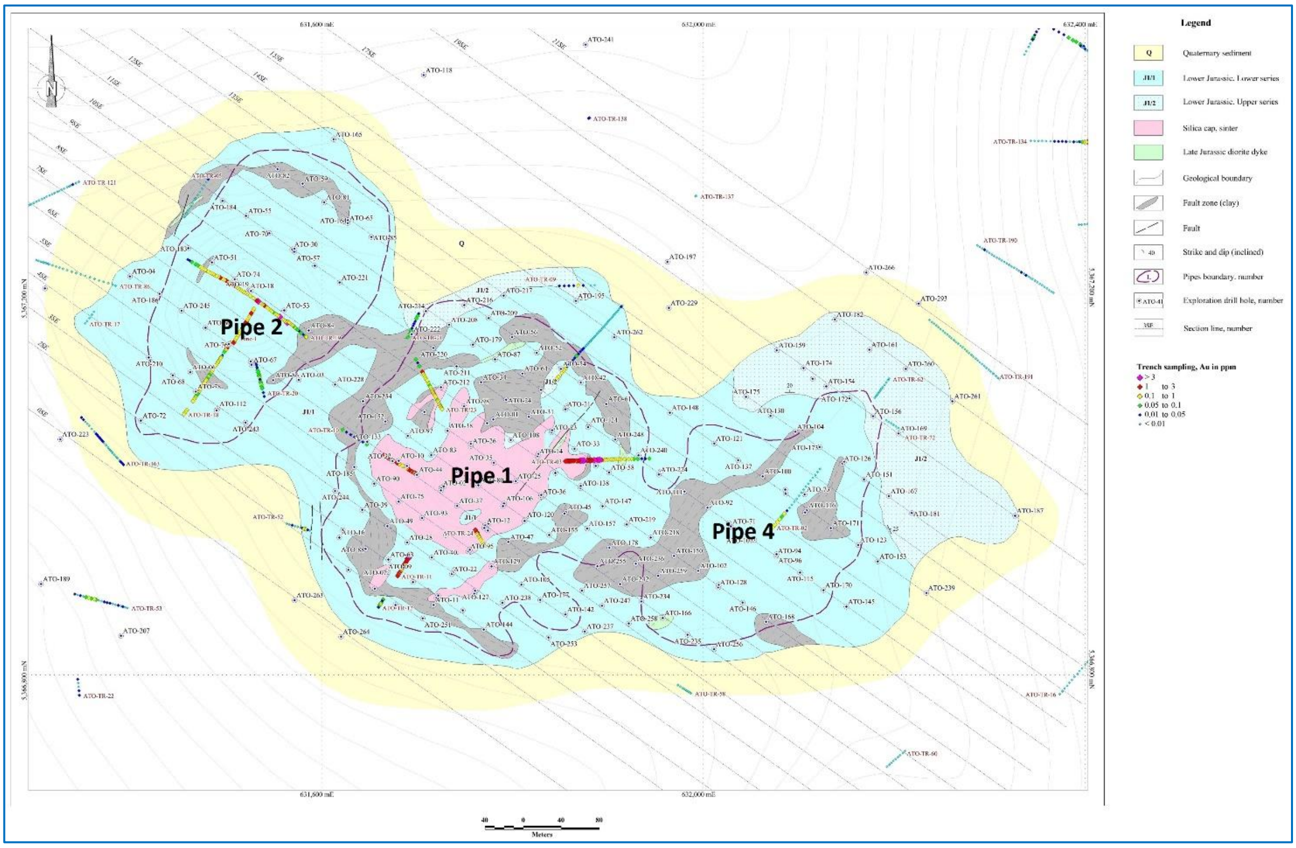
Adjacent pipes ATO 1, 2 and 4 were been emplaced into stratified rocks as young as presumably Early to Middle Jurassic. Pipe 4 is mainly concealed. Pipe 3 to the west contains abundant pyrite, but no significant amounts of Au, Ag, Pb, and Zn – and hence has not so far been explored further. There is 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 characterized by low resistivity. The pipes coincide with chargeability anomalies that overall really are quite weak.

Figure 7.3 - Perspective view of ATO Pipes 1, 2 and 4, looking WNW



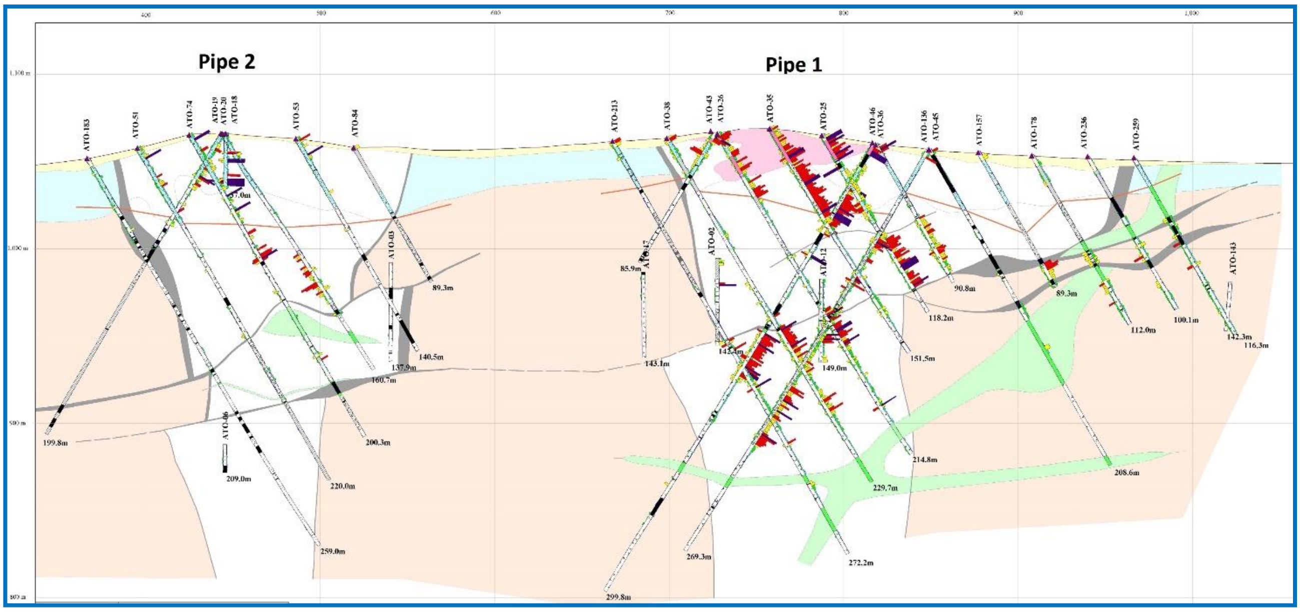
Surface projection of the morphology of Pipes 1, 2, and 4 is shown inFigure 7.4. An upper zone of Au-Pb-Zn-Ag mineralized rock at Pipe 1 is approximately oval in shape and is ~320 m wide. Pipe 2 is elongate to the NE and is ~320 m by ~160 m in maximum dimension. Pipe 4, which is completely concealed, 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 7.4 - Geology map of ATO Pipes 1, 2 and 4



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.

Figure 7.5 - Cross-section through Pipes 1 and 2



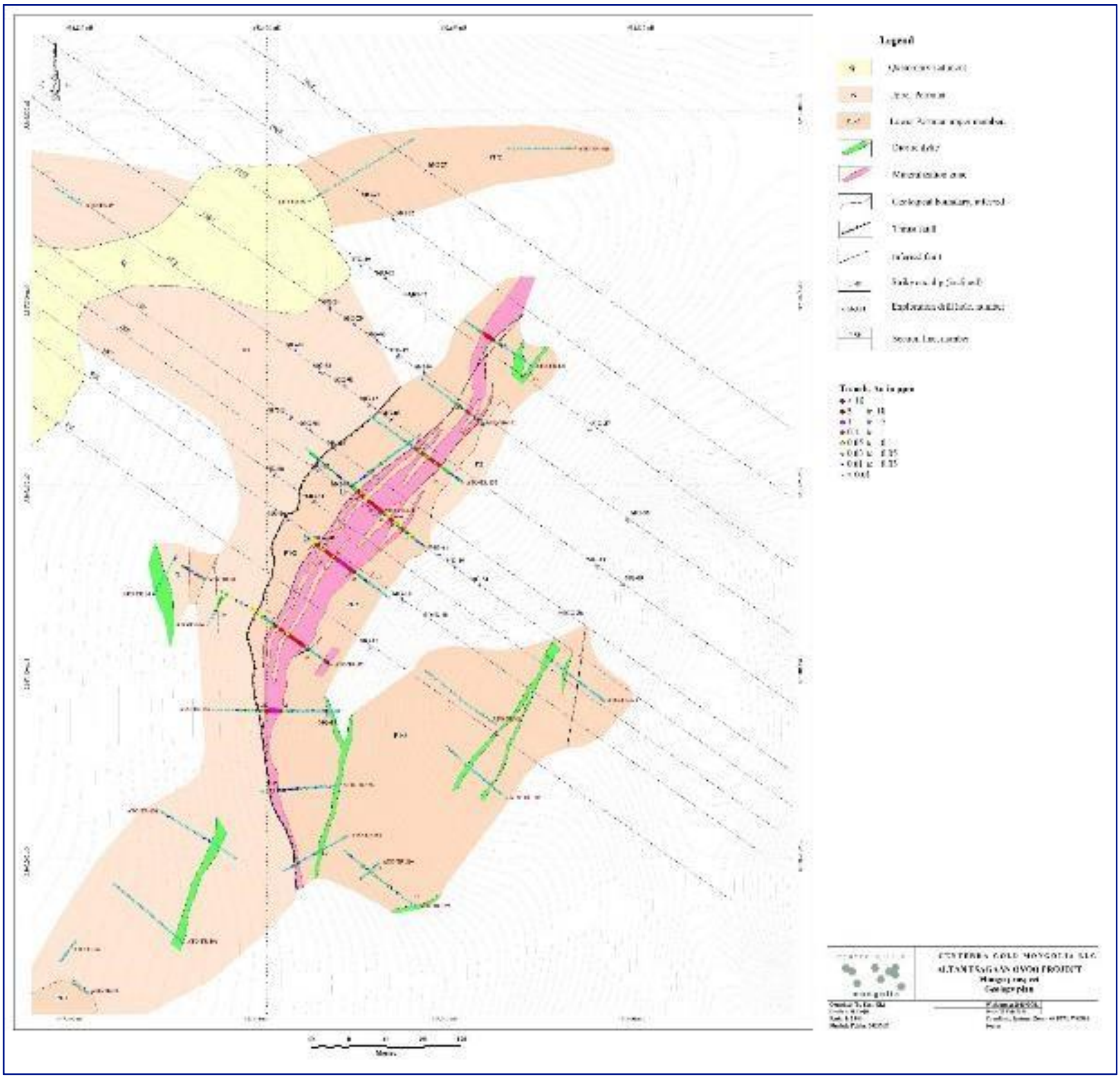
The pipes also are cut by a number of minor faults, both steeply dipping and shallow dipping. Some narrow flat-lying post-mineral diorite dikes also have been emplaced along faults that offset margins of the pipes.

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 mineralization.

Despite the three mineralized 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.

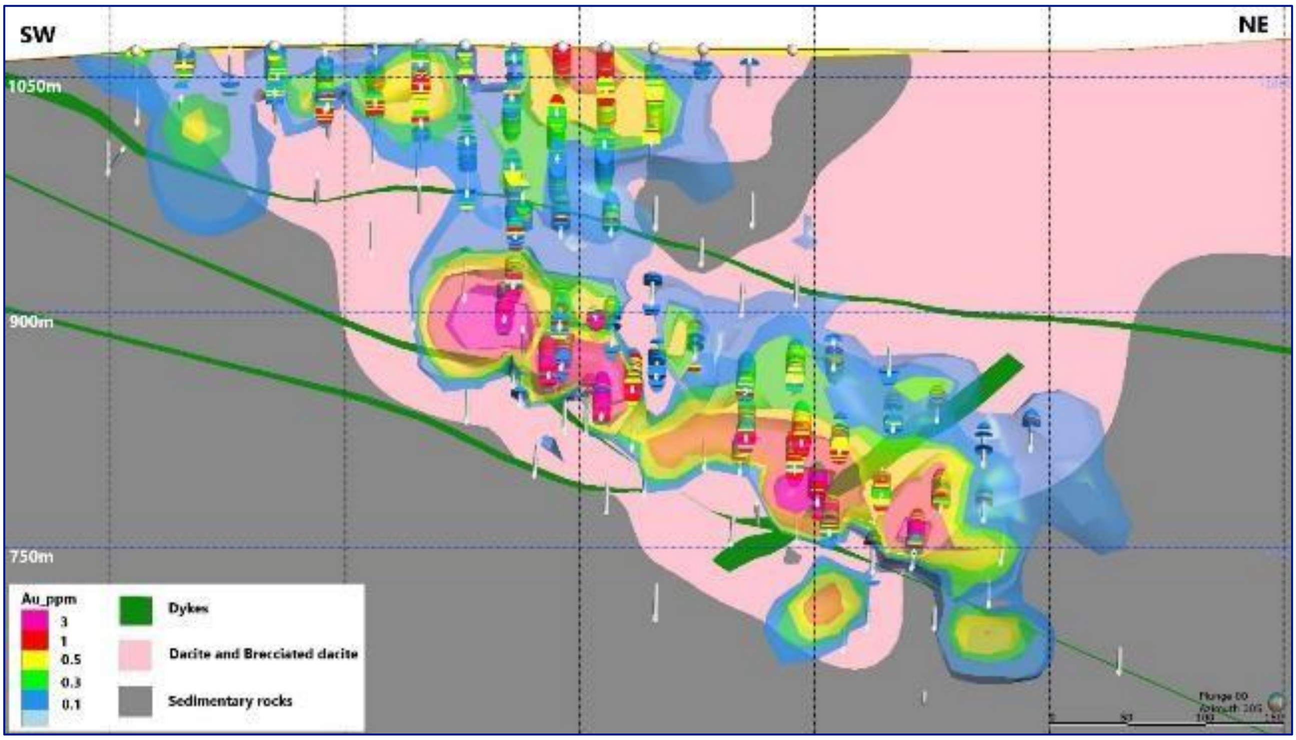
Mungu: Mungu deposit is hosted in Lower Permian age volcano-sedimentary rocks and overlying Upper Permian sedimentary rocks and is itself cut by late diorite dykes (Figure 7.6). Post-mineralization diorite porphyrite dykes are abundant in the area. Weakly to moderately chloritized, black green coloured 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



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. Ore body has almost linear shape, almost vertical and directing to NE by azimuth 035°. Trend of mineralization is plunging to NE by azimuth 035 and dip angle 40°. The mineralized bodies separated by late post-mineralized 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 7.7. 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 mineralization is show inFigure 7.7.

Figure 7.7 - Mungu deposit in cross-section looking NW



Mineralization 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 mineralization, 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 mineralization with localized narrow zones of high-grade mineralization at depth has yielded bonanza grades in both gold (to 172.88g/t) and silver (to 1500.0g/t).

7.4.2 Deposit mineralisation controlling factors

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

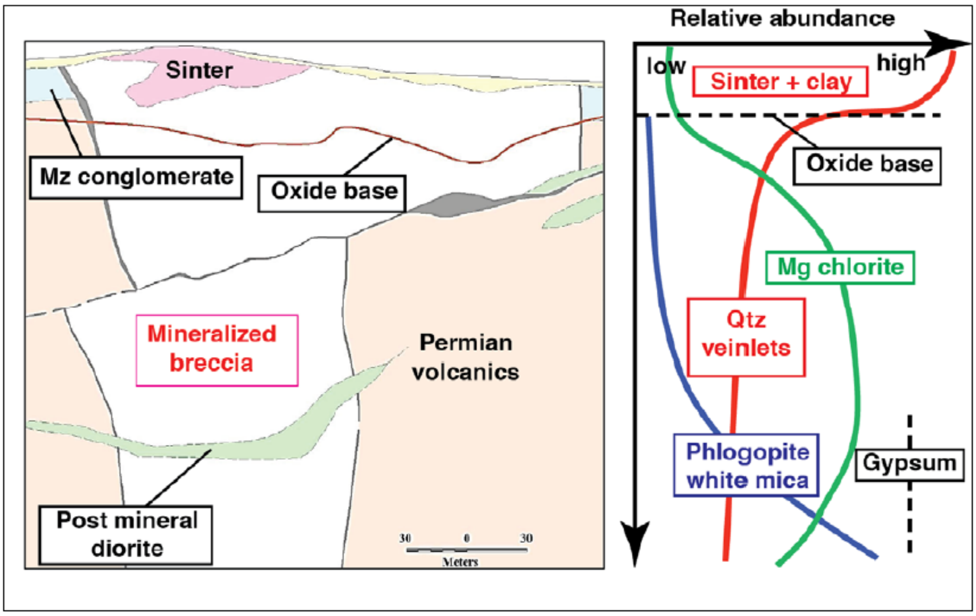
1) Presence in a major base metal (Pb-Zn) province (thick crust) known to include large tonnage Au-Pb-Zn deposits, combined with
2) Near paleo surface, epithermal (hot spring) emplacement of the upper parts of mineralized pipes associated with Jurassic magmatism into a near surface
3) Shallow depression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

Origin of brecciation in the mineralized 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).

7.4.3 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 (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 , 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 mineralized column at Pipe 1, clinochlore (Mg chl) is the dominant alteration hyrosilicate mineral at depth throughout mineralized 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.

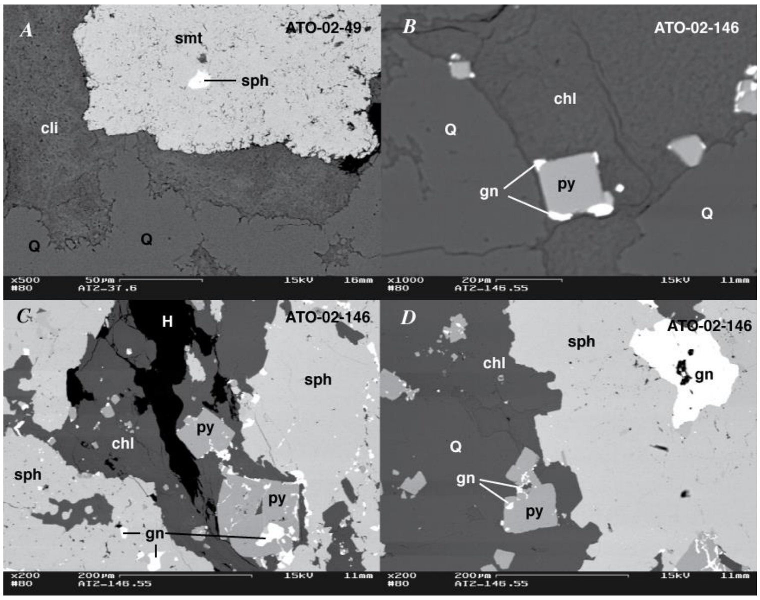
Figure 7.8 - Schematic Geological Cross-Section Through Pipe 1 (Left), Showing Alteration Mineral Changes with Depth (Right)



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 Middle-Late Jurassic 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.

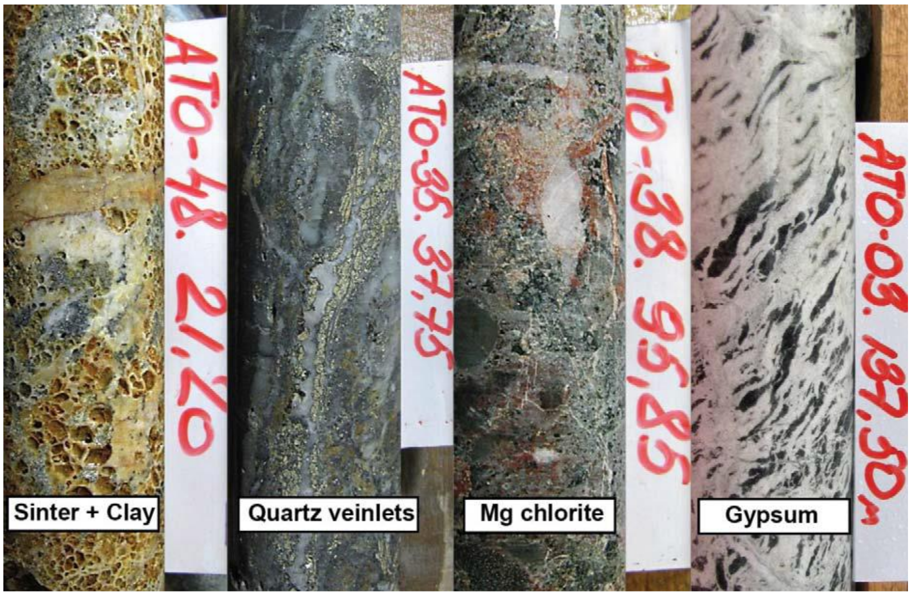
Back scattered electron micrographs (Figure 7.9) of polished thin sections show clinochlore (cli) and chlorite (chl) compatibility with various sulphide and carbonate minerals in drill core at ATO Pipe 1. A (top left, Figure 7.9): Sample DDH ATO-02-49; sph, sphalerite; Q, quartz; smt, smithsonite. B (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.

Figure 7.9 - Back scattered electron micrographs



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 mineralized core contents of many 10s of thousands ppm Mg to greater than 100,000 ppm Mg throughout the mineralized 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 7.10 - Core Photo



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 mineralized breccia where magnesian chlorite is the dominant hydrosilicate in the matrix; and lastly (4) gypsum after anhydrite especially concentrated near pipe margins. As depicted 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 mineralized 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 mineralization.

7.4.4 Silica cap in Pipe 1

Because of the importance that the silica cap at Pipe 1 played in the discovery of the mineralized pipes at ATO it was described in detail in the 2017 NI 43-101 Report (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 7.8 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.

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.

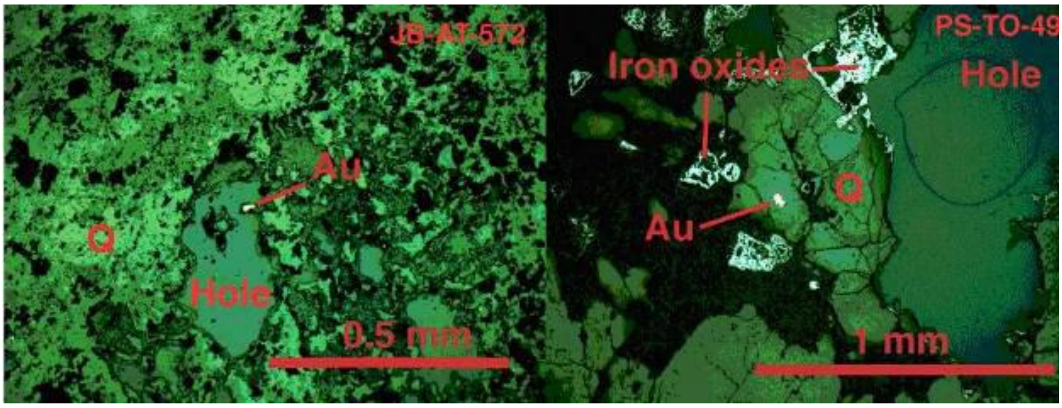
Outcrops on Pipe 1 indicate overall that its silica records a complex geologic history. The rocks display highly disturbed almost chaotic orientations even within individual outcrops of about 3-5 m wide. Banded and crustified silica has orientations that are extremely variable. Further, presence of now 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 almost at the Middle-Late Jurassic paleosurface during mineralization 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. 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 7.11 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 7.11 - Photomicrograph of free gold particles in cap rock surface grab samples



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 mineralized system becomes highly enriched in galena.

7.4.5 Styles of sulphide mineralised rock at ATO

A number of styles of sulphide-mineralized 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 7.12 - Mineralised Breccia Fabrics from Pipe 2

Figure 7.12 shows general fabric of mineralized 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 Rock Styles at ATO

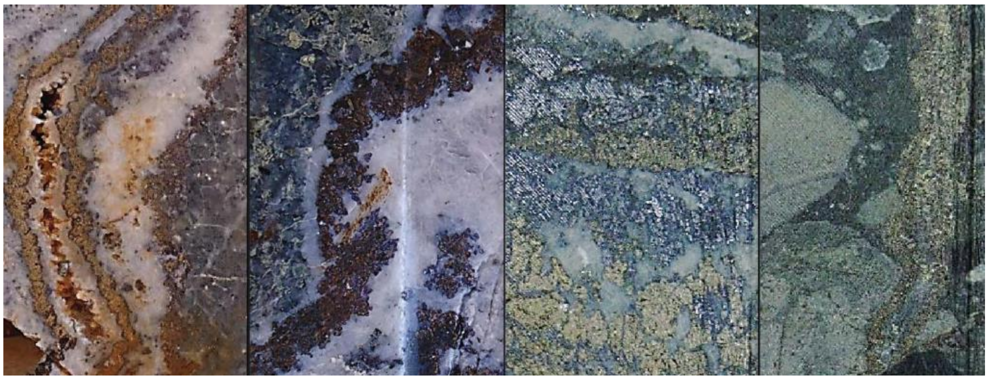


Figure 7.13 shows general styles of mineralized rock at ATO (from left: ATO-20 85.9 m; ATO 142.7 m; ATO-111 266.5 m; ATO-111 275.1m).

Flat-lying galena-sphalerite flooding in diamond hole ATO-111 at 266.5 m 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 mineralized 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 mineralized system at ATO really does not contain that much manganiferous calcite, though some is present rarely in cm wide veins.

7.5 Mineralisation 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 7.14 - Diagram of Sulphur Fugacity Versus Temperature

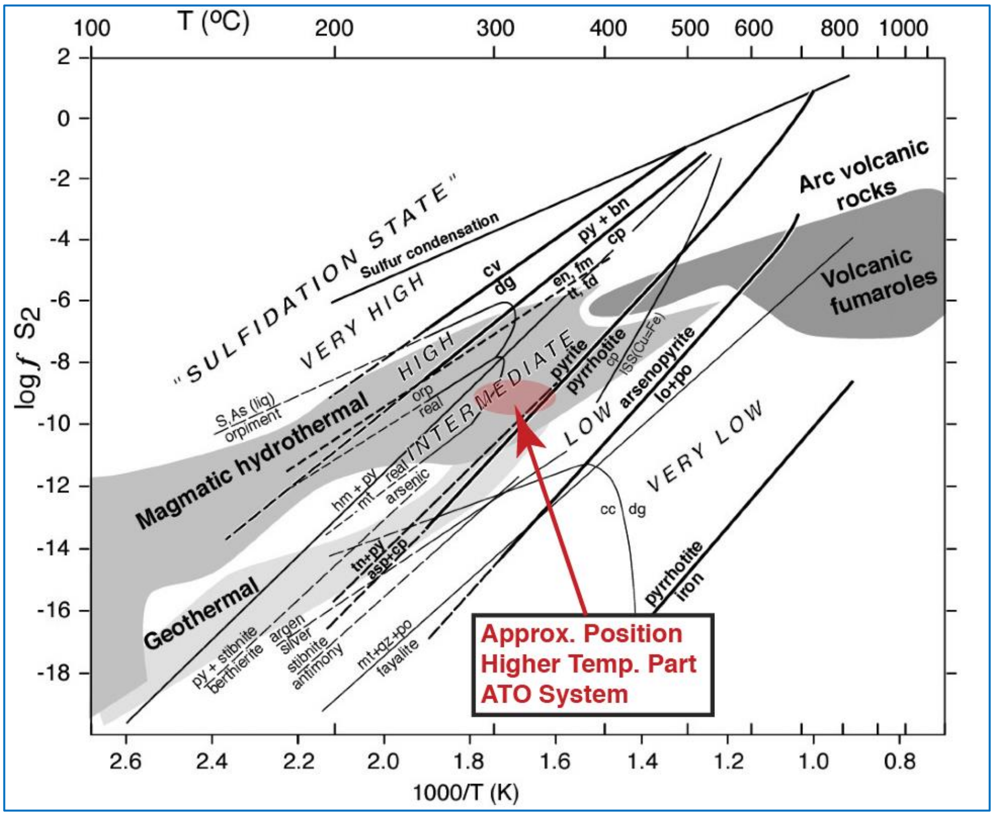


Figure 16hows a diagram of sulphur fugacity versus temperature and various sulphide mineral assemblages in epithermal deposits that reflect sulfidation states varying from low through intermediate to very high sulfidation states. Approximate compositional fields of geothermal, magmatic hydrothermal, and volcanic fumaroles are shown in varying shades of grey.

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

Details: The mineralized 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 mineralized 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.

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 mineralization 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 mineralized 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 mineralization 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 mineralization onto detrital fragment-rich Early Permian volcaniclastic rock and Jurassic pebbly conglomerate, as well as protracted passage of fluids associated with mineralization 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 mineralized, 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 mineralization 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 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 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 mineralized 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-mineralized as well) is present very close to the original paleosurface.

Nonetheless, some differences from the norm for many well-described, mineralized 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 mineralization event that continued well after cessation of the brecciation that first created the voids.

8 DEPOSIT TYPE

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.

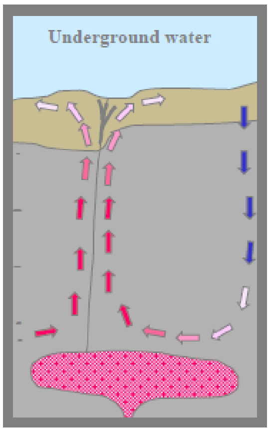
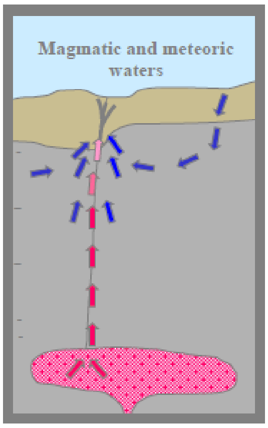
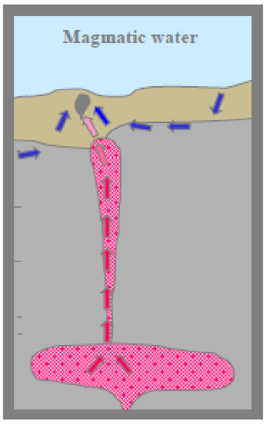
The bulk of the information in this Section is extracted from the a November 2021 feasibility study by DRA Global Limited (DRA) disclosed in a November 2021 NI 43-101 report.

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 mineralization 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

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:

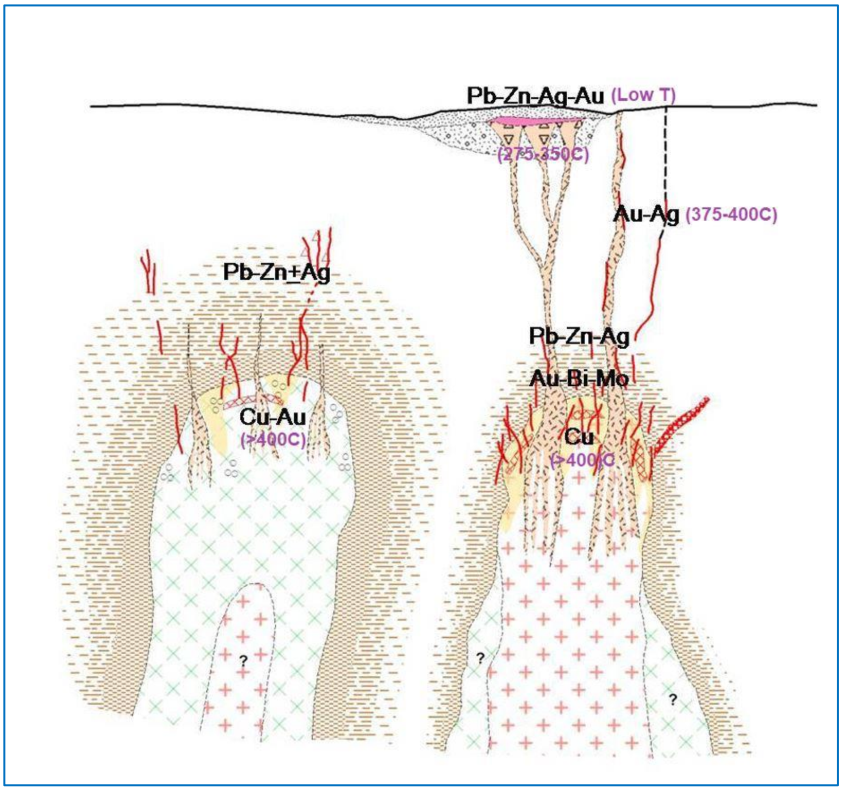
- Geological and mineralisation model – effectively defining the shape.
- Genesis of the ATO deposits.
- Geochemical zonation.

8.2.1 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



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 mineralization. The ATO mineralization 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 mineralization (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 mineralization 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.

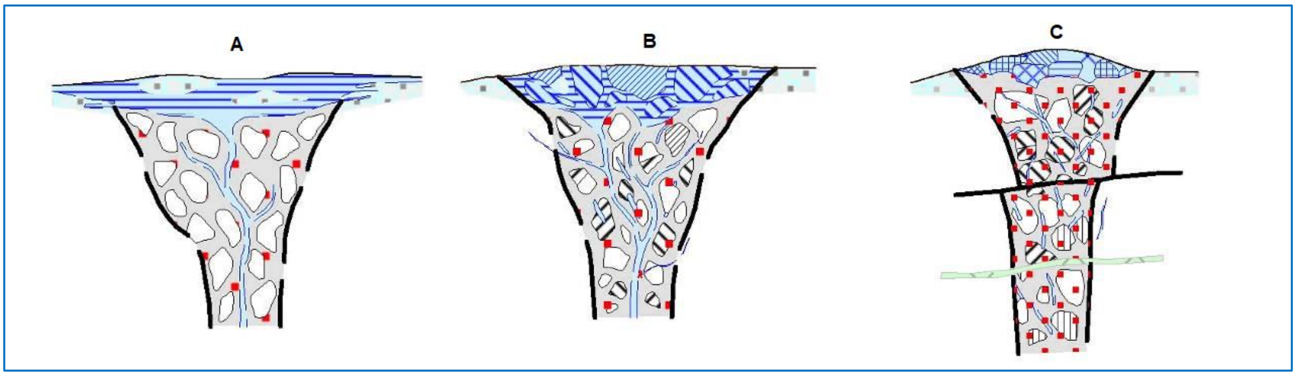
8.2.2 Genesis of the ATO deposits

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 mineralization 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. In 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 mineralization 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



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 mineralization.

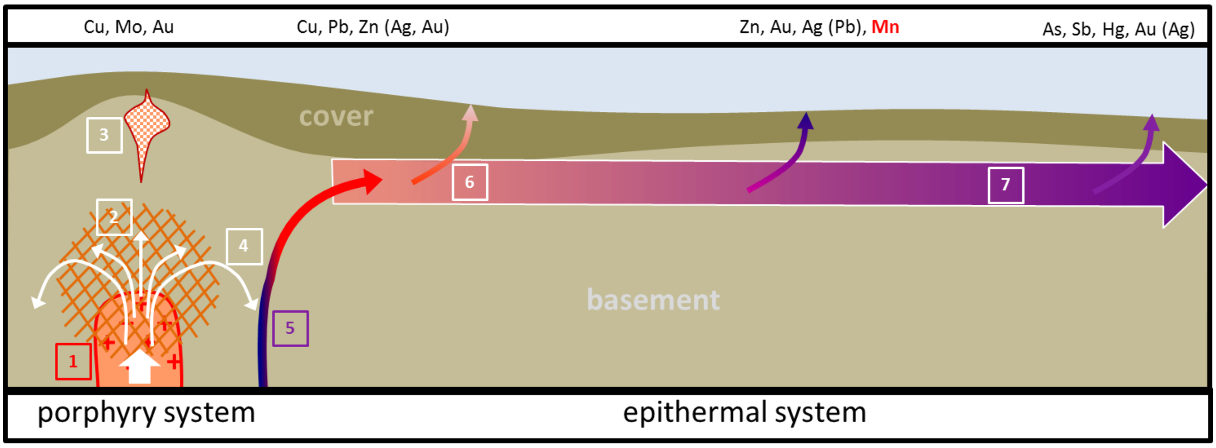
Lower pipe-form mineralization (grey parts in Figure 8.3) were subjected to post-mineralization horizontal faulting and the pipes appear to be displaced along the fault (black line in C, Figure 8.3)

Figure 8.3 Small bodies of diorite have been found in these faults.

8.2.3 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 mineralization 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



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 mineralization (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 mineralization on top of them (3 inFigure 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 colored 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 mineralization took place.

8.3 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)

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

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

The bulk of the information in this Section is extracted from the November 2021 feasibility study by DRA Global Limited (DRA) disclosed in a November 2021 NI 43-101 report.

9.1 Surface Exploration Methods and Parameters

9.1.1 Preparation

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.

9.1.2 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.

9.1.3 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 mineralization. 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 mineralized 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.

9.1.4 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.

9.1.5 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 mineralized 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-mineralized 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 mineralized 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.

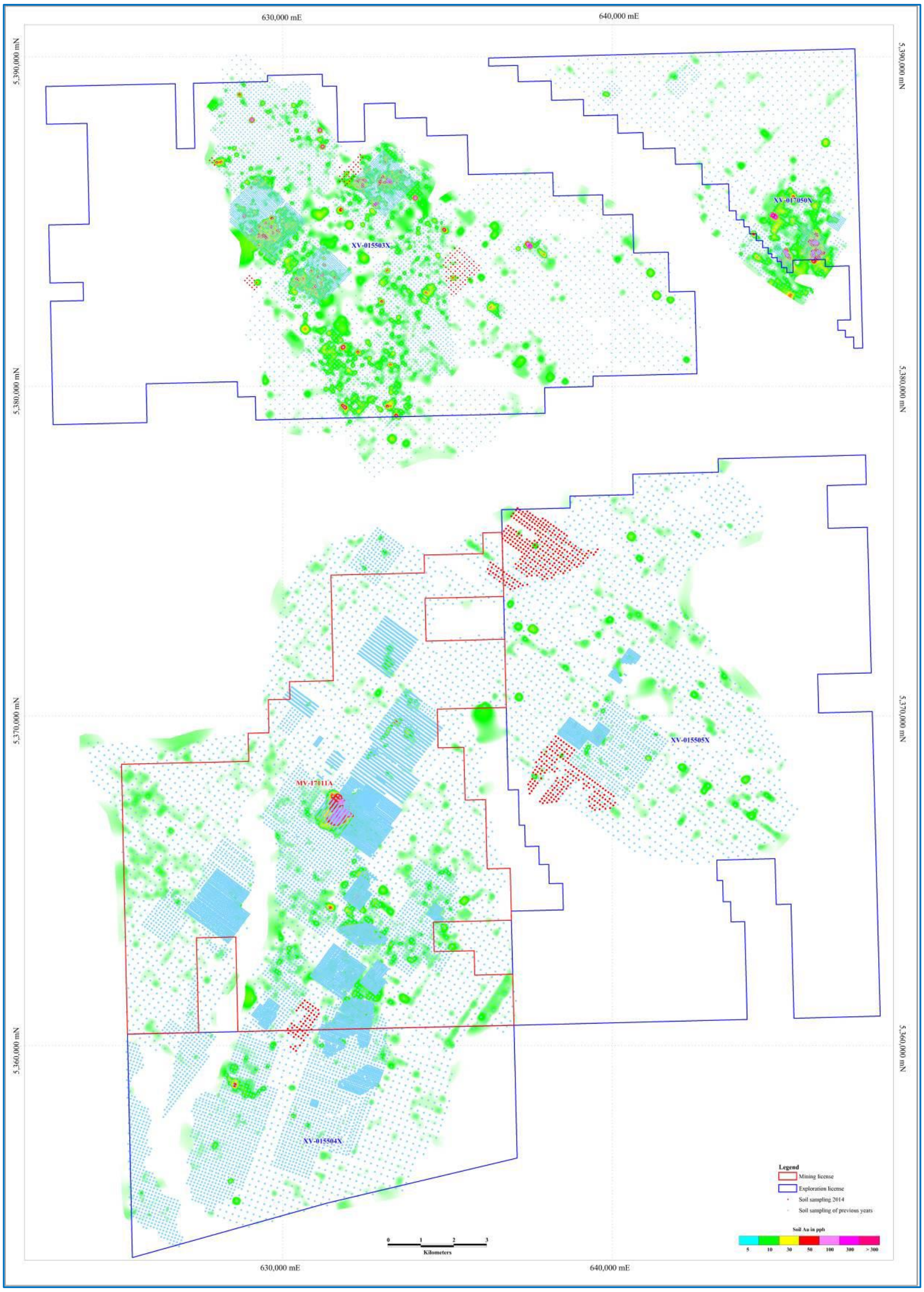
Further, anomalous >500 ppm Zn in soil is present in soil on top of Quaternary-covered mineralized rock in Pipe 4, as well as on the northern fringe of the surface projection of Pipe 1 and on the SW margin of Pipe 2.

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.

Figure 9.1 - [21] Gold soil sampling in the ATO district – 2010 to 2014



[21] CGM 2014 ER.

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

9.1.6 Grab Sampling

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

Grab samples were collected from outcrops and fragments of altered and/or mineralized 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 extensive alteration and mineralization. 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.

9.1.7 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 mineralization 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.

9.1.8 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 km^2 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 km^2 (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.

9.1.9 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 mineralized bodies (Table 9-1). Sampling was aimed to test the metal recovery of mineralized 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

Grade	Very high			High			Medium			Low		
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		
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 mineralized 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 mineralized	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		
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.

9.1.10 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.

9.1.11 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.

9.1.12 Mineralogical Sampling

To study the mineral composition of the deposit's mineralization, possible sequence of concentrating of minerals, and the nature of structure and texture of the mineralization, a total of 59 samples were taken representing all of mineralized 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.

9.1.13 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.

9.1.14 Paleontological Sampling

Six 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. He delivered his descriptions of the samples along with photographs.

9.1.15 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 C_2 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.

9.2 Surface Exploration Results & Interpretation

9.2.1 Soil Geochemical Result Analysis

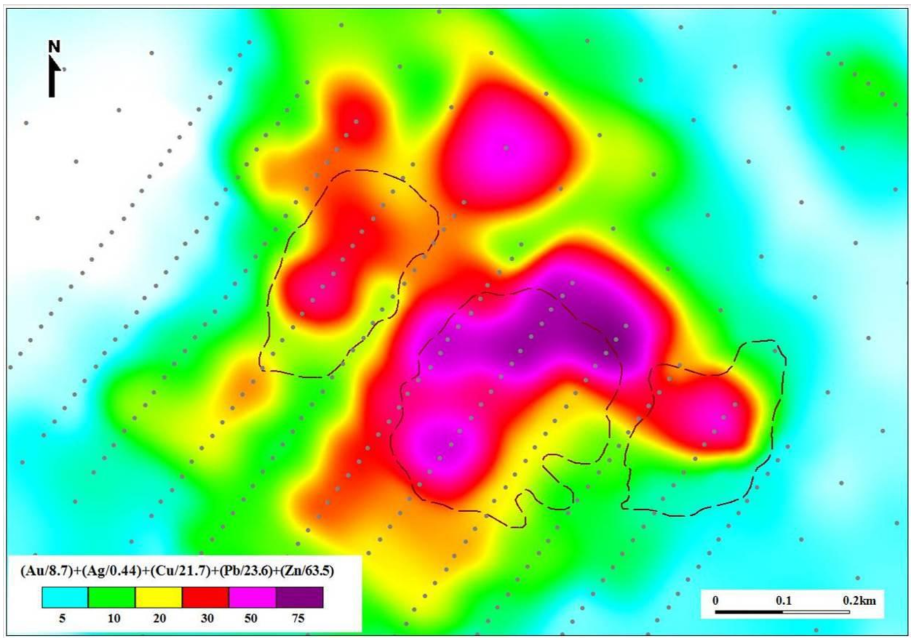
Results of the soil sampling at the ATO license area were produced to analyse dispersion halos for elements of Au, Ag, Pb, and Zn. Analysis was attempted to determine the vertical and horizontal zonations of elements associated geochemically to the ATO deposit and figure out the geochemical features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

Figure 9.2 - Soil combined element halo map over ATO pipes



A geochemical anomalous halo of combined elements was outlined with a size of 750 * 900 m, slightly elongated to the north. The pipe-shaped mineralization 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 south west 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 north eastern 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.

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
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 10.13.2 with the drilling results.

9.2.2 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

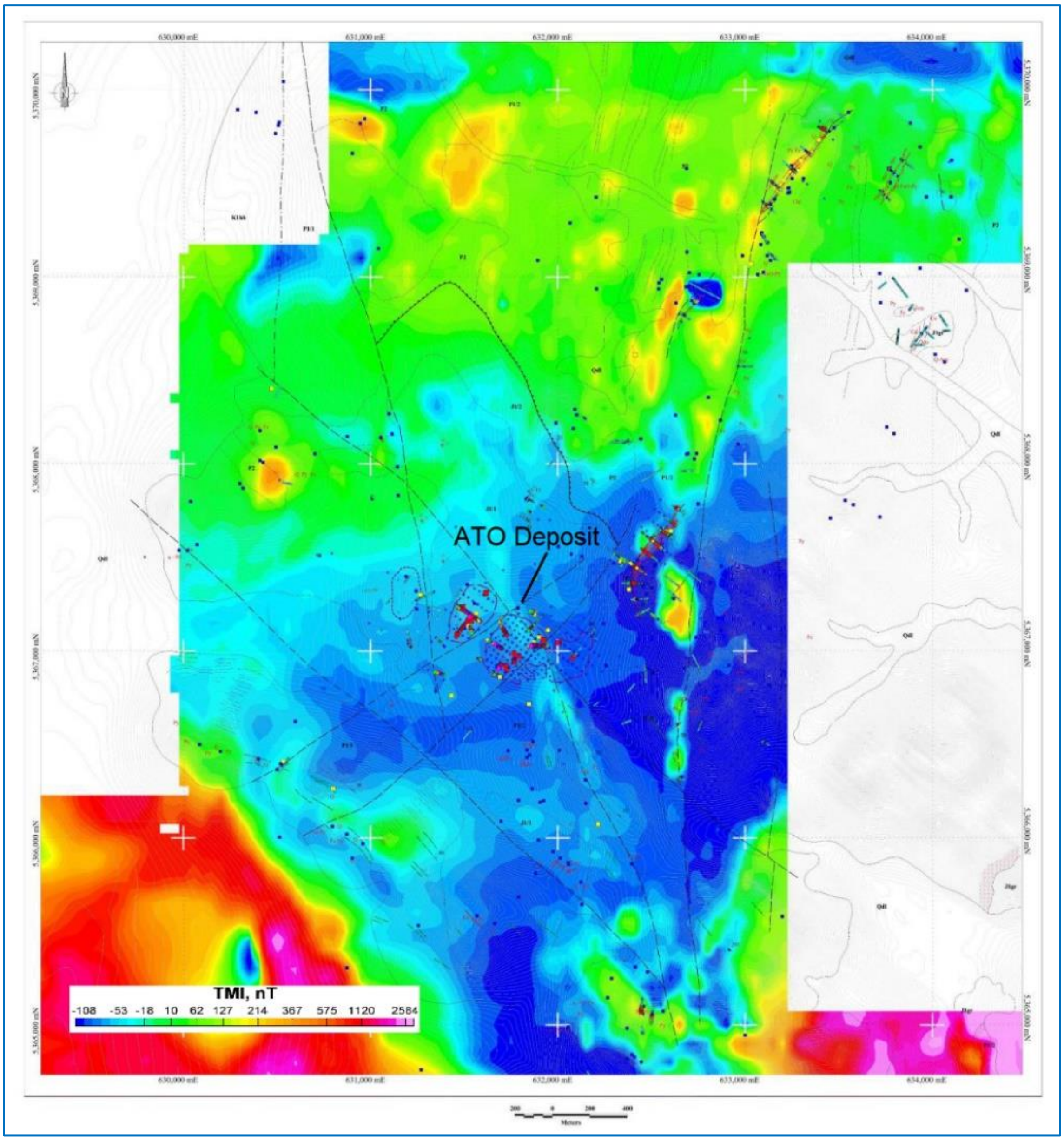
Trench ID	Length (m)	From (m)	To (m)	Lithology	Au ppm	Ag ppm	As ppm	Cu ppm	Pb ppm	Zn ppm
ATO-TR-117	2.2	2.8	5.0	Tuff dacite	0.13	<	201	11	15	89
	1.2	19.5	20.7	Tuff dacite	0.10	<	124	12	43	59
ATO-TR-238	1.7	70.6	72.3	Fault	0.38	6.0	3980	184	583	12400
ATO-TR-239	0.5	19.6	20.1	Andesite	1.99	<	8420	256	6	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

Trench ID	Length (m)	From (m)	To (m)	Lithology	Au ppm	Ag ppm	As ppm	Cu ppm	Pb ppm	Zn ppm
	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

9.2.3 Geophysical Survey Results

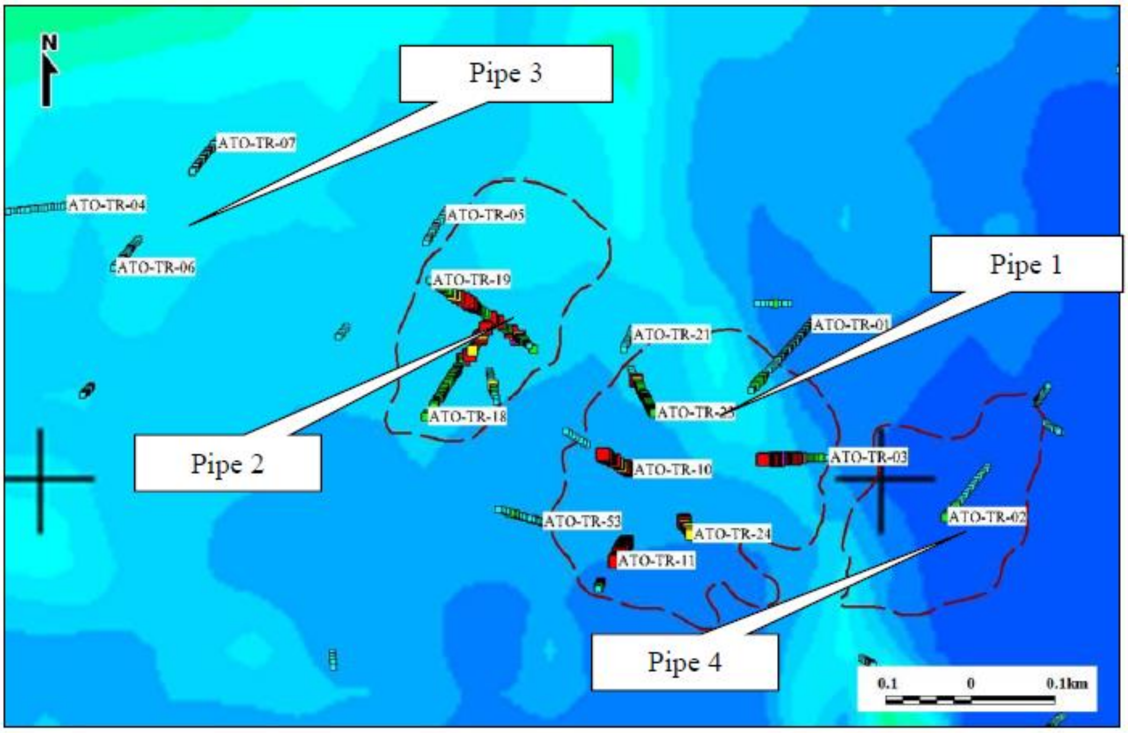
Magnetic survey results: Magnetic data were instrumental in identifying major structures, mafic dykes, and ring intrusion-related anomalies. The deposit at ATO is in a broad generally low magnetic area that contains linear, low level positive features that coincide with NW and N striking faults (Figure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

Figure 9.3 - Magnetic survey map over ATO district



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



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 mineralization (high chargeability), even though a direct 100% relationship is not always present.

The mineralized 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

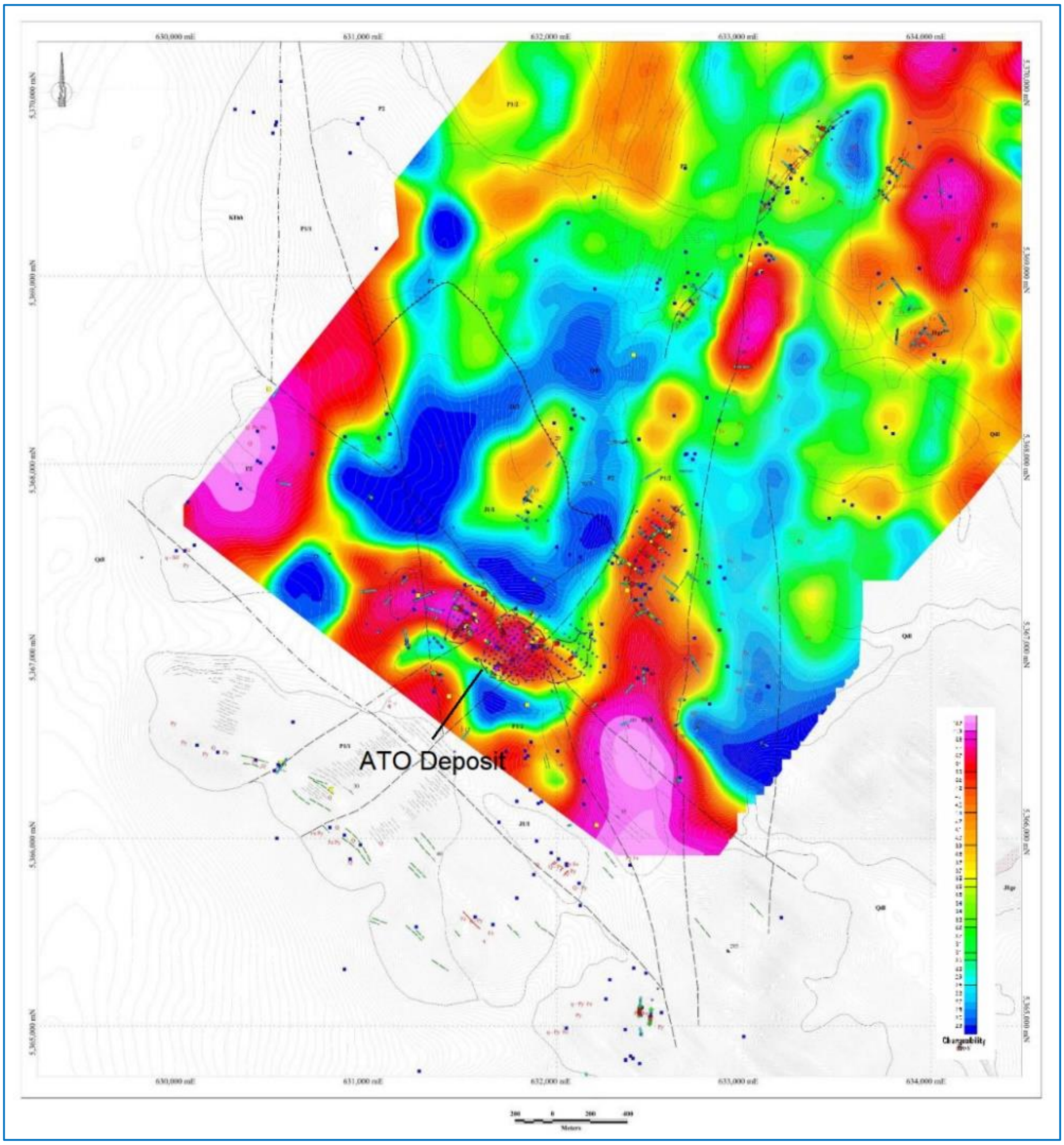
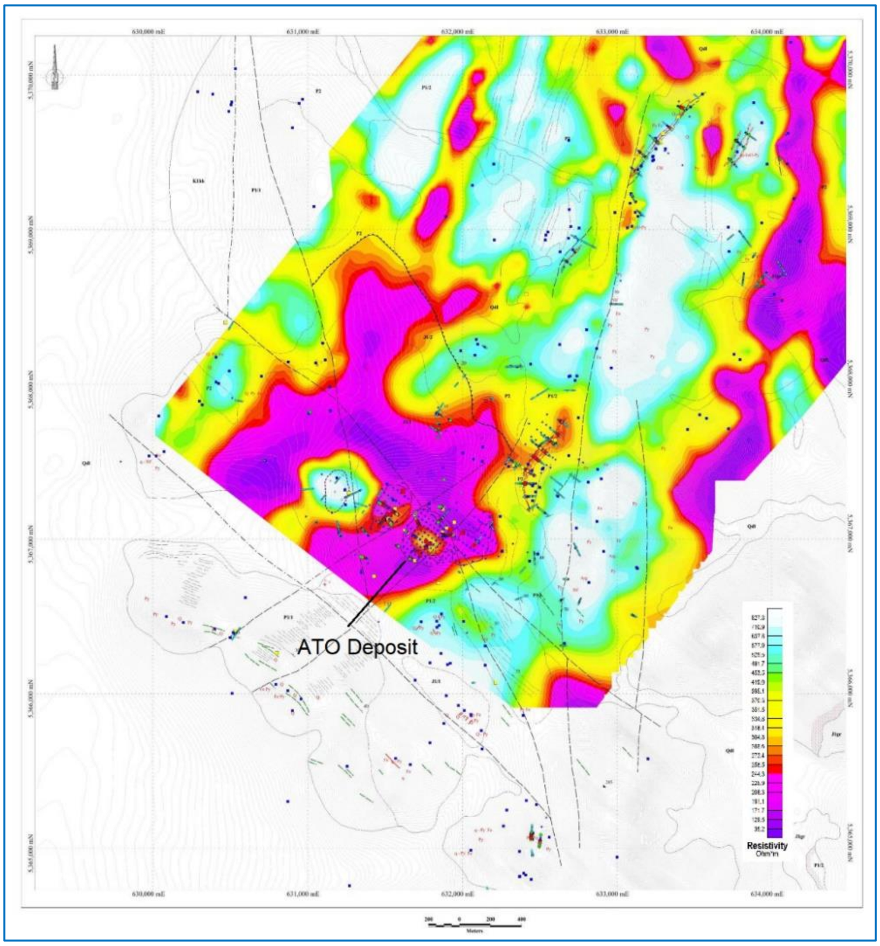
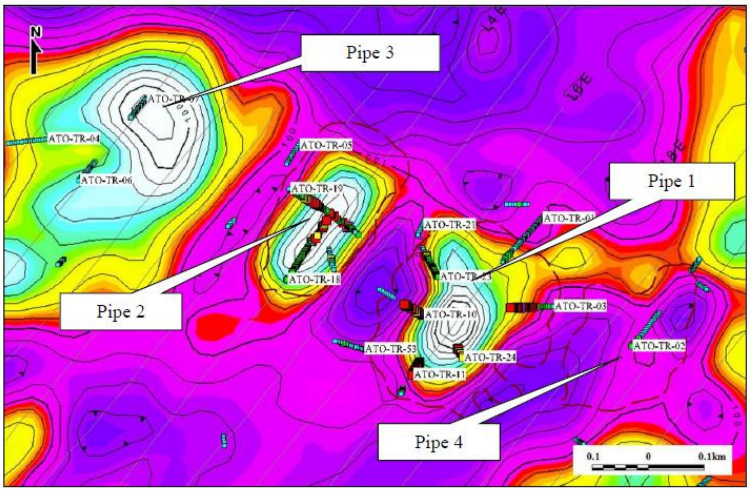


Figure 9.6 - D-D IP 100 m resistivity survey map over ATO district



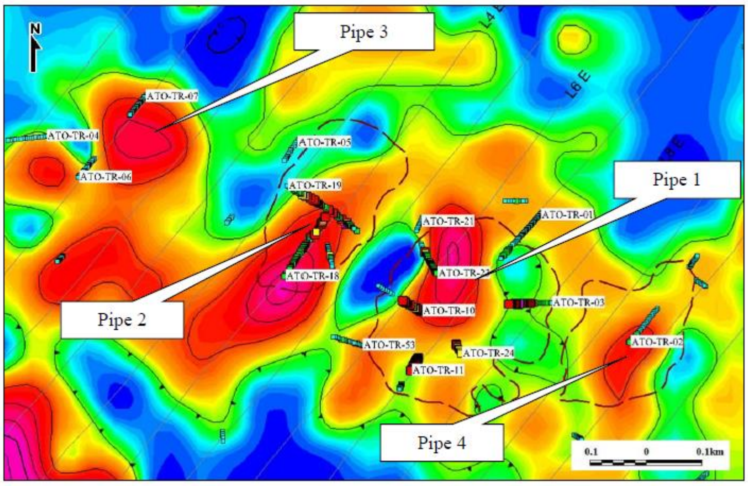
Based on the results in sections, 10 m below the ground surface, several separate areas of very distinct resistivity anomalies were identified and they spatially coincide with a north west-trending series of small hills that are formed of pipe bodies. Therefore, boundaries of pipes and the locations 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



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



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.m. 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, 3

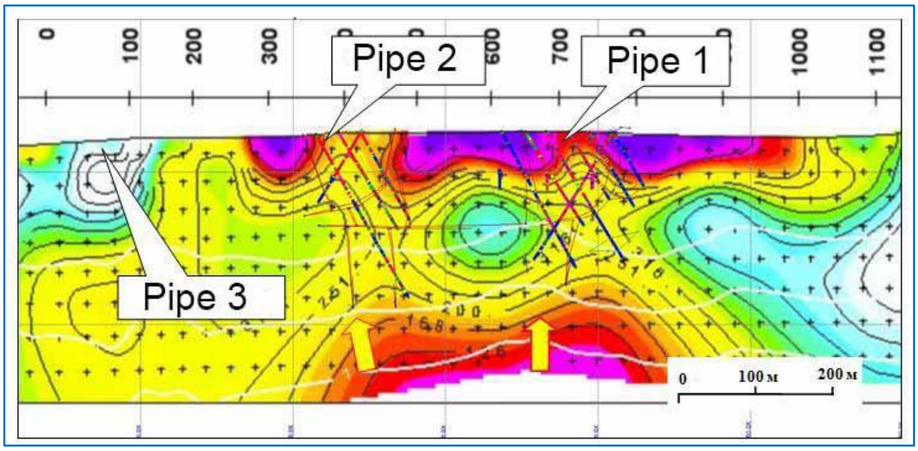
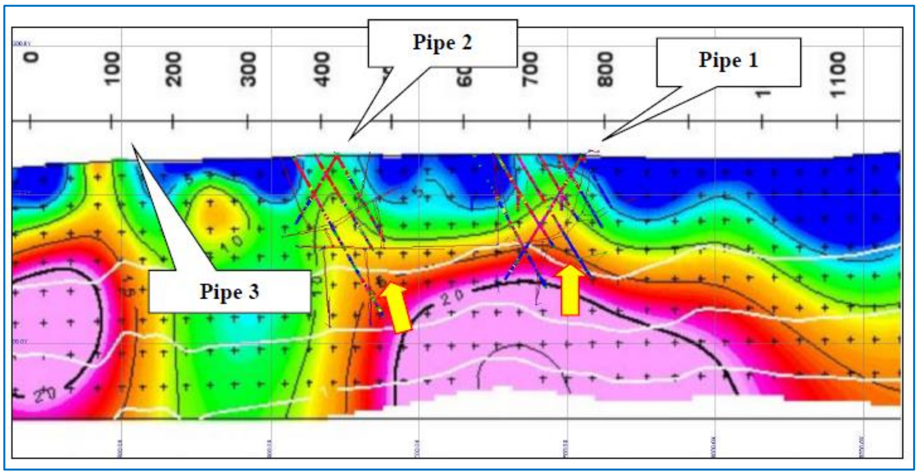


Figure 9.10 - D-D IP chargeability cross-section through ATO pipes 1, 2, 3

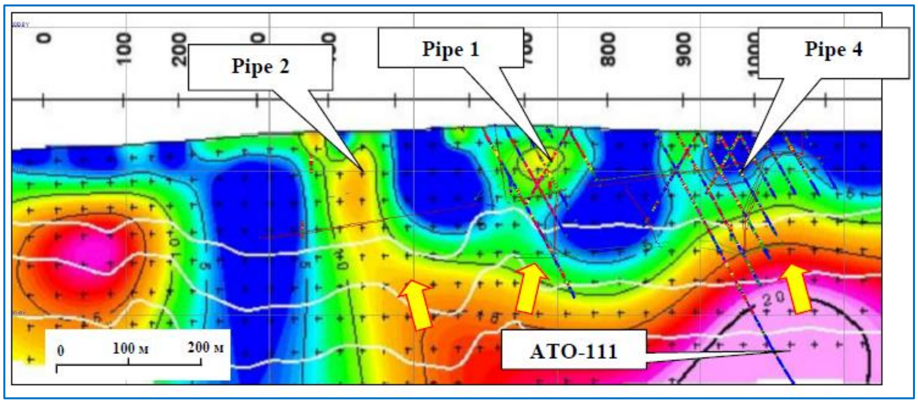


Pipe 2 area anomaly: Anomalous area measures 280 m long and 130 m wide. Responses of the area are similar to Pipe 1 in terms of values. Chargeability map shows a 350 m long, elongated anomalous area that coincides with the southern part of the resistivity anomaly. They are distinguished clearly in sections (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 mineralization after making a comparison of it with the geophysical anomalous areas of the other known mineralization. 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).

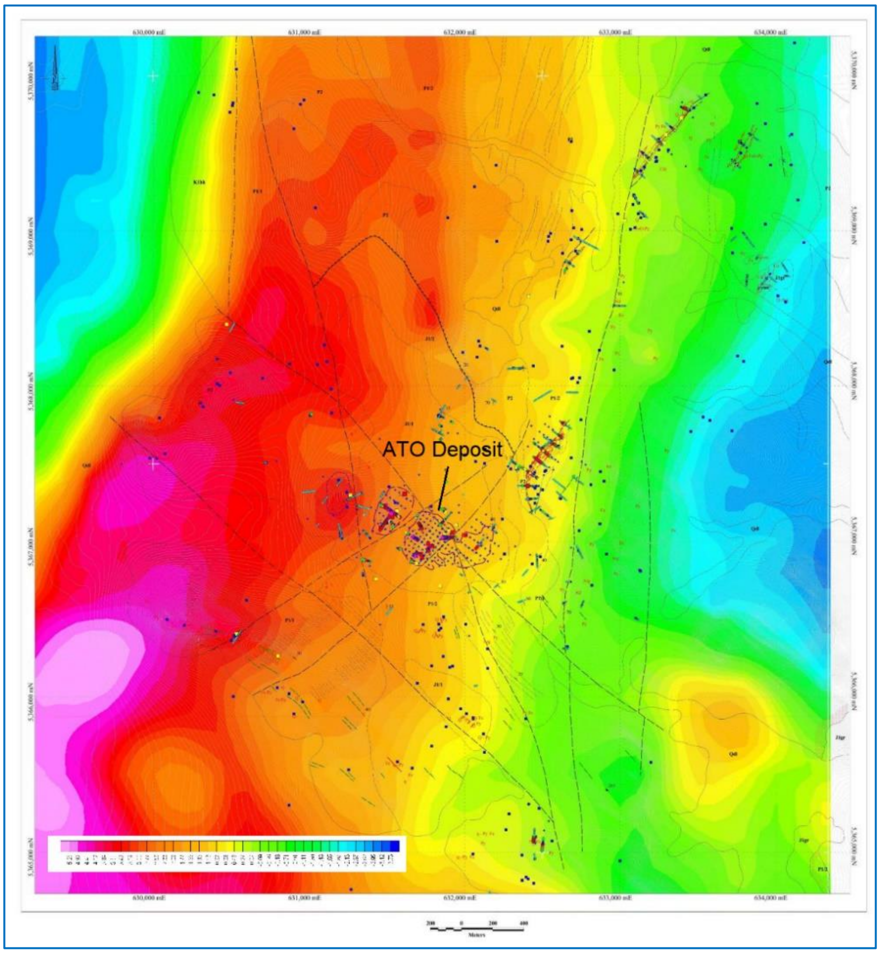
Figure 9.11 - D-D IP chargeability cross-section through ATO pipes 1, 2, 4



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 mineralization. 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 mineralization.

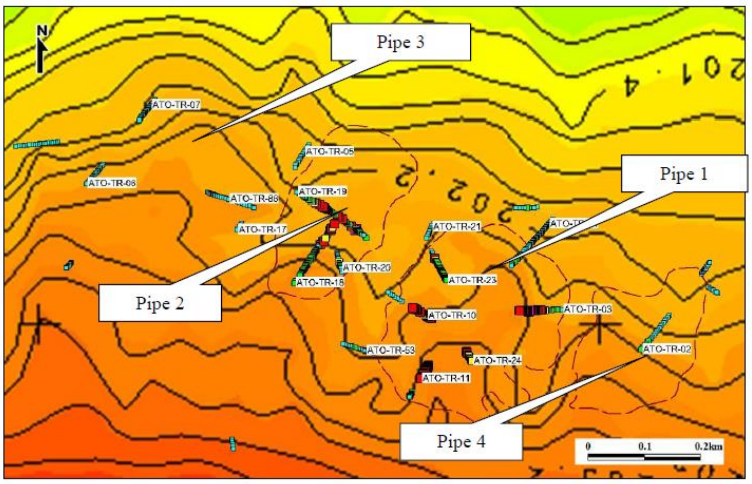
Gravity survey results: The gravity survey of the ATO deposit and its surrounding area shows that the deposit is on the flank of a broad NNE-trending gravity high (Figure 9.12).

Figure 9.12 - Gravity survey map over ATO district



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 mineralized pipes are located in these folds (Figure 9.13).

Figure 9.13 - Gravity survey map over ATO deposit



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 south west to north east. Pipe 4 is at the eastern part of an area of relatively high gravity response protruding from south to north.

9.2.4 Ground-Water Exploration Results

Exploration works have identified two potential groundwater flow systems extending over an area up to 140 km^2 that can be characterized 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 0m to over 250 mbgl comprising semi consolidated Cretaceous sediments.

No continuous low permeability strata has 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 m^2/d) and lower transmissivity conditions (50 m^2/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 C_2 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 & SAMPLING

Drilling and sampling at ATO is 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 geological GeoRes 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. Observation by GeoRes of drilling in progress in April 2022 confirmed that view.

The bulk of the information in this Section is extracted from the November 2021 feasibility study by DRA Global Limited (DRA) disclosed in a November 2021 NI 43-101 report. Additional exploration drilling since early 2021 has not yet been databased and is not described here.

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 mineralization 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 mineralization. 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[22].

- 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[23].
- Commencing 2020, the Company drilled an additional 55 drill holes for a total of 18,200 m to the end of 2020. A total of 53,000 m have been drilled since 2018. The drilling information was used to update the interpretation of the geologic model, geometry of the mineralized zones and domains[24].

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 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

[22] 2019 AIF and 2018 news release.

[23] 2019 AIF and 29 July 2020 news release.

[24] 21 February 2021 news release. Information adopted from.

Exploration Drilling Program		Core Holes	RC Holes	TOTAL
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. 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 GeoRes'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".

Table 10-2 Exploration drill hole summary – to 2021

Holes	Number	Length	Av length
		(m)	(m)
AT diamond	385	73,133.8	190.0
AT RC	227	9,441.0	41.6
Mungu diamond	155	37,745.5	243.5
All holes	767	120,320.3	156.9

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/Trenches	Number	Length	Av length
		(m)	(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.

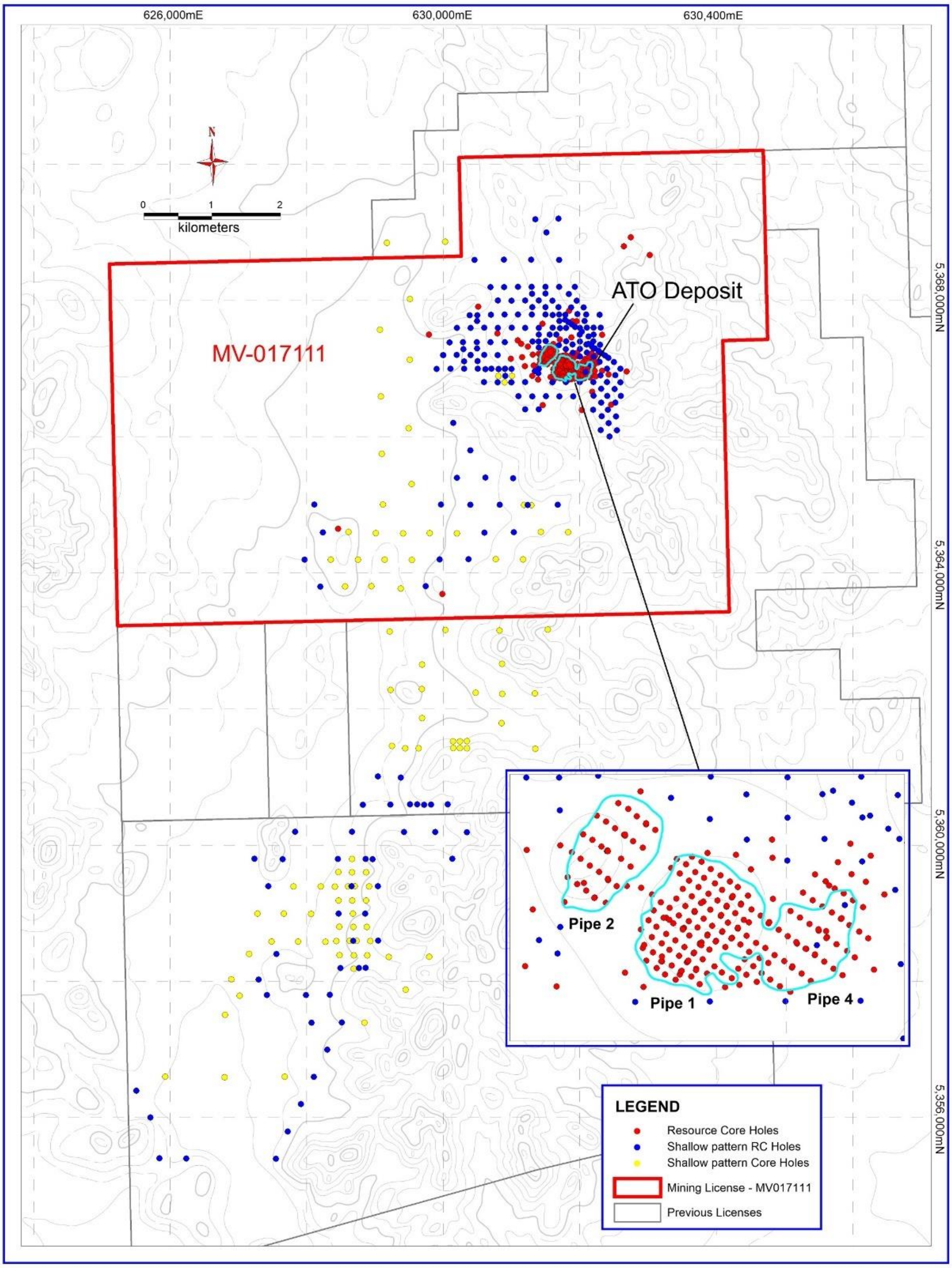
A Reflex ACT II Rapid Descent tool was used for core orientation.

10.4 Drill Hole Locations

10.4.1 Location of Holes up to 2017

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

Figure 10.1 - Drill holes locations – holes to 2017

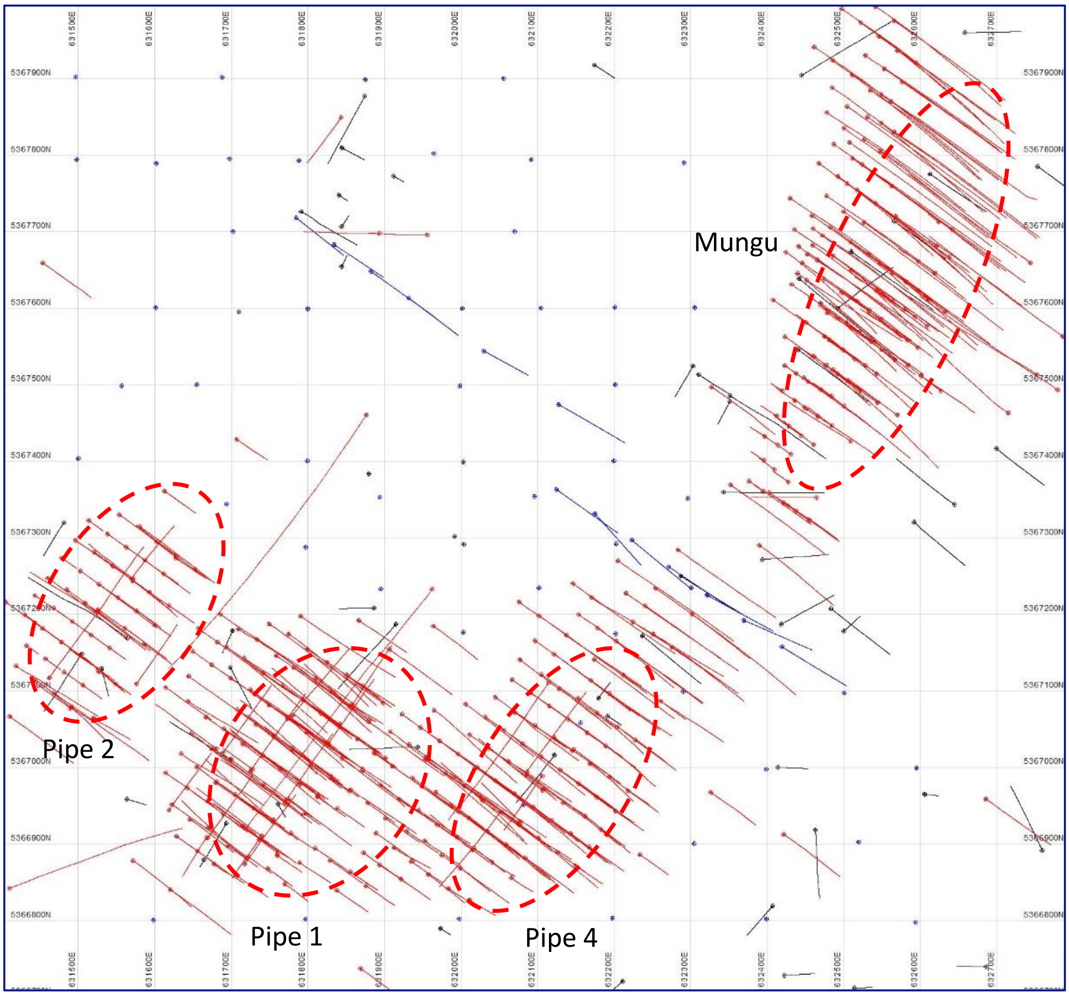


Drilling at the ATO deposit was conducted along 18 cross-section lines oriented WNW to ESE (125°) traversing Pipes 2, 1 and 4 respectively in order west to east (see lines in Figure 7.4). The sections were spaced 30 m apart. The western end of a typical cross-section was shown Figure 7.5.

10.4.2 Location of Holes to 2021

Diamond drilling and trenching in the period 2017 to end 2020/early 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 & trench locations – deposit area 2021



Drilling since 2017 by Steppe continued through to Mungu on the 30 m spaced 125° oriented cross-section lines (see Figure 14.2).

10.5 Hole Spacing and Orientation

The older RC drill holes (blue dots Figure 10.2, 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 7.5Figure 7.5 - Cross-section through Pipes 1 and 2

. 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 mineralization 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 mineralized 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 aluminium 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 mineralized 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 mineralization, 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 mineralization 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 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 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 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 10.13.2 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 & Sampling Accuracy Factors

The 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 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 & interpretation

10.13.1 Overall Opinion Of Drilling

The 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 10.13.2 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 mineralized 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.

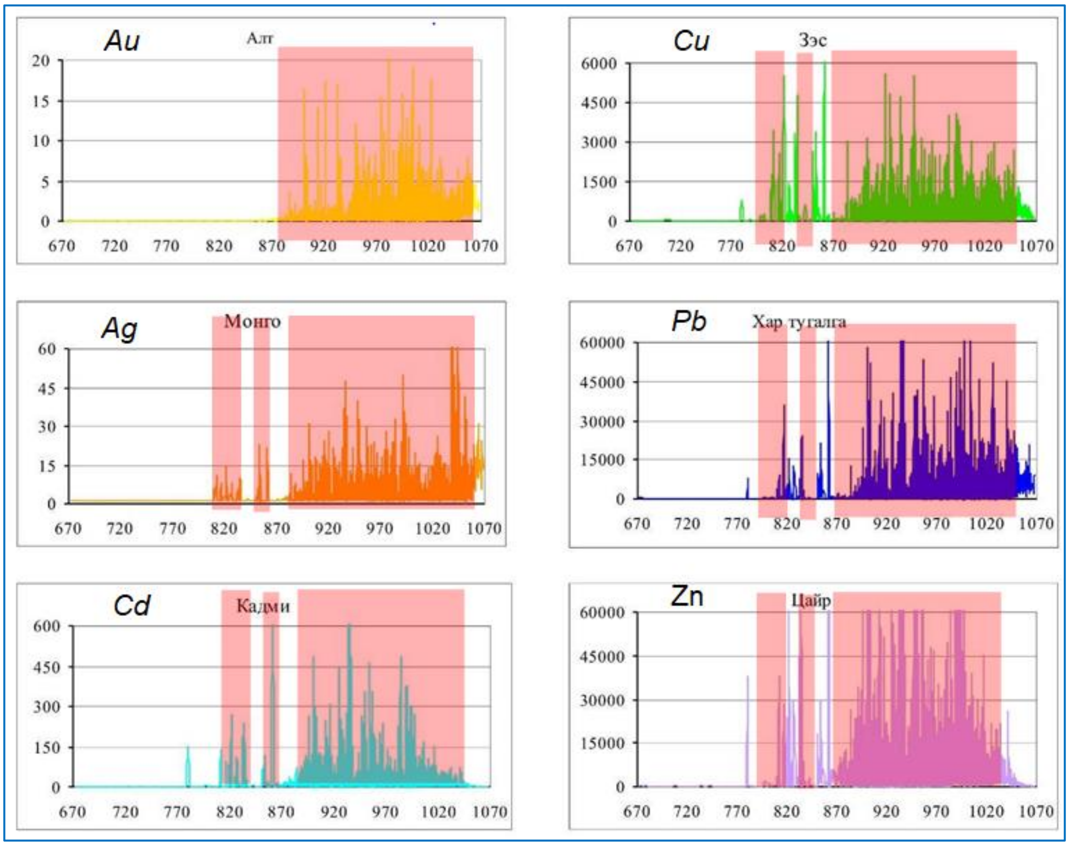
Pipe 1: A total of 1,329 samples from eight drill holes were assayed for 45 elements for Pipe 1. However, the table below shows correlations of only the following elements, which have been selected because of their importance in hydrothermal explosion deposits (Table 10-4). Gold in Pipe 1 has moderate 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 mineralization in Pipe 1 continues from the 880 m level to the 1070 m level. As per polymetallic elements, their mineralization 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 mineralization has no gold and discontinuous while the mineralization at upper levels coincides with gold mineralization 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



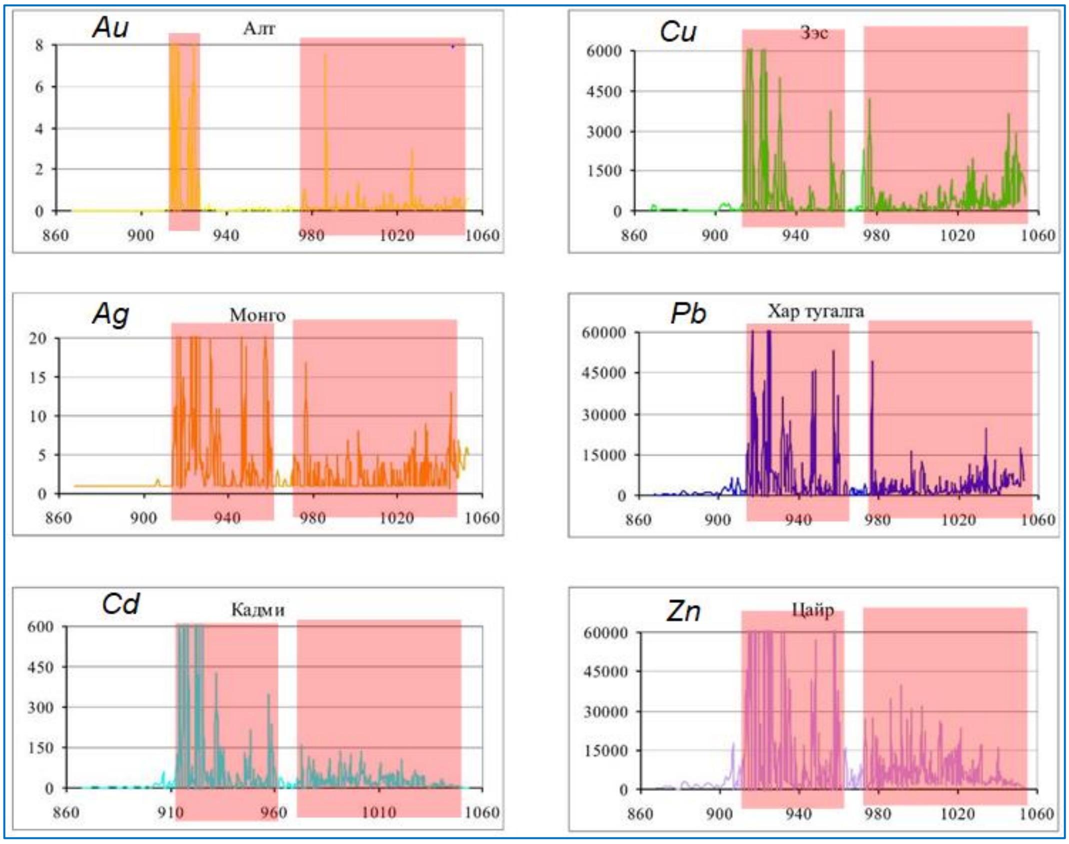
Pipe 2: A total of 374 samples from three drill holes were assayed by multi-element analyses for Pipe 2. The results were then processed and displayed Table 10-5 showing correlations of gold and polymetallic elements. Gold (Table 10-5 has moderate correlations with cadmium, copper, tin, and zinc, and 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 1050 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 1050 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 1040 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.

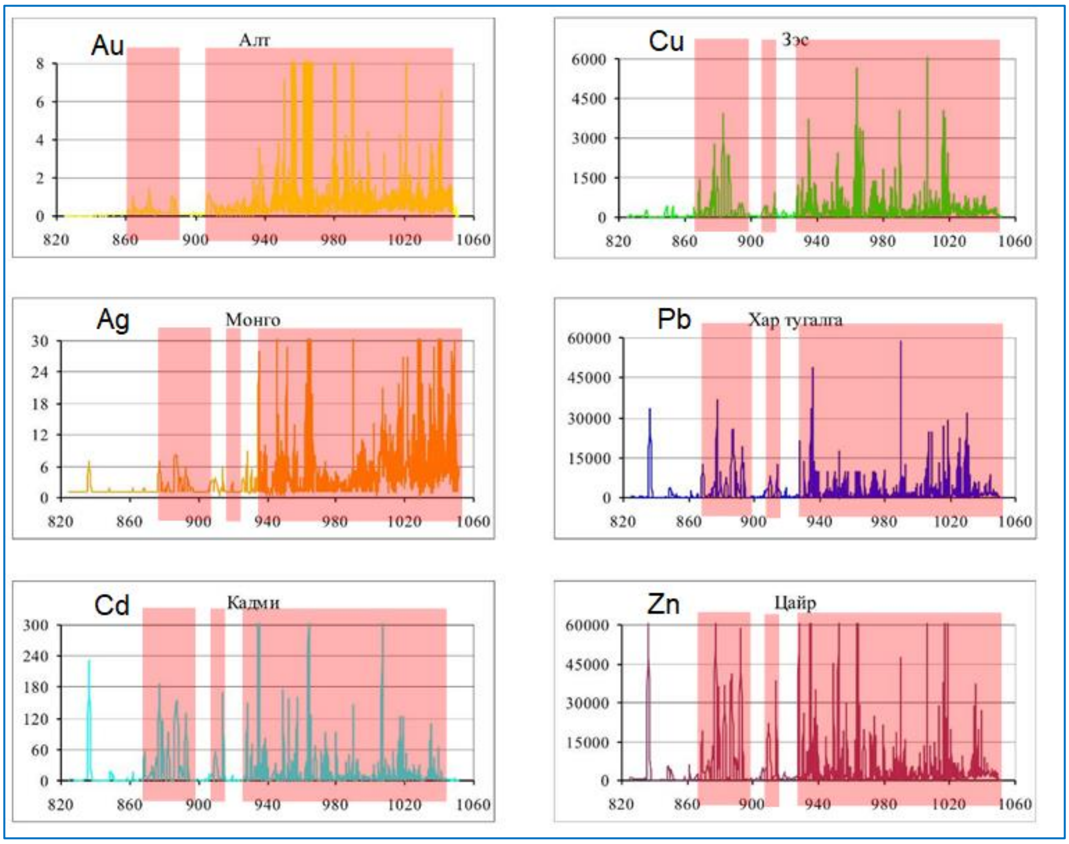
Table 10-6 Pipe 4 core sample element correlations

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

Correlation	Au	Ag	As	Cd	Cu	Mo	Pb	Sb	Zn
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 mineralization is established at depth ranges of 860-885 m level and 910-1050 m level. In general, gold and polymetallic mineralization 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



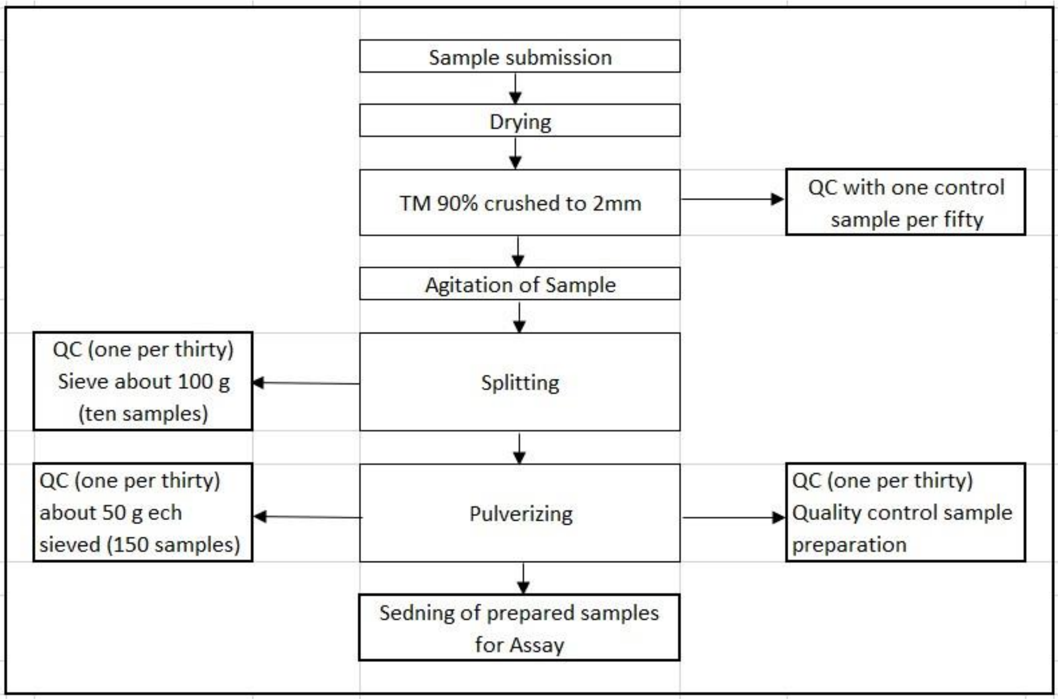
11 SAMPLE PREPARATION, ANALYSIS & SECURITY

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 meter interval from mineralized 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 below 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 CGM 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 were 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:

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
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, we used an analytical suite as Fire assay with AAS finish (Au-AA24).

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.

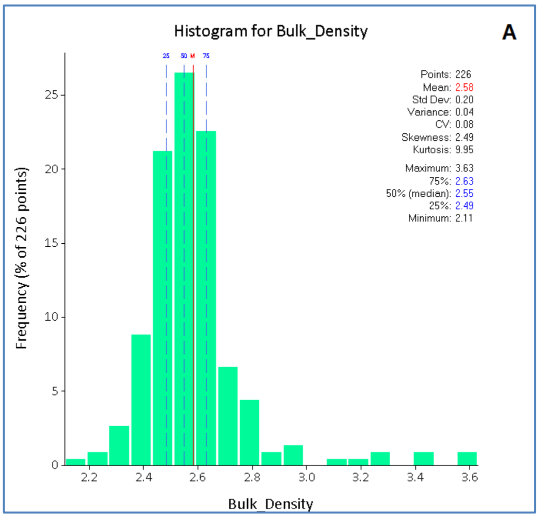
11.4 Bulk Density Measurements

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).

The Author QP has moved to using t/m^3 units of density (specific gravity) here rather than the g/cm^3 used previously.

Figure 11.2 - Density histogram - all samples



When sorted by oxidation level the densities were:

- Oxide average density 2.46 t/m^3 (Figure 11.3)
- Transition average density 2.59 t/m^3 (Figure 11.4)
- Fresh average density 2.64 t/m^3 (Figure 11.5)

Oxide material densities (62 samples) were the lowest and had a 0.15 t/m^3 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/m^3 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/m^3 spread in the 25 to 75% range, similar to the transitional material.

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

Figure 11.3 - Density histogram - Oxide

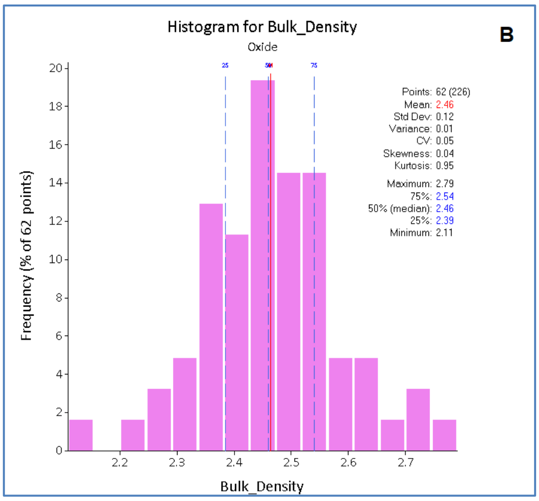


Figure 11.4 - Density histogram - Transition

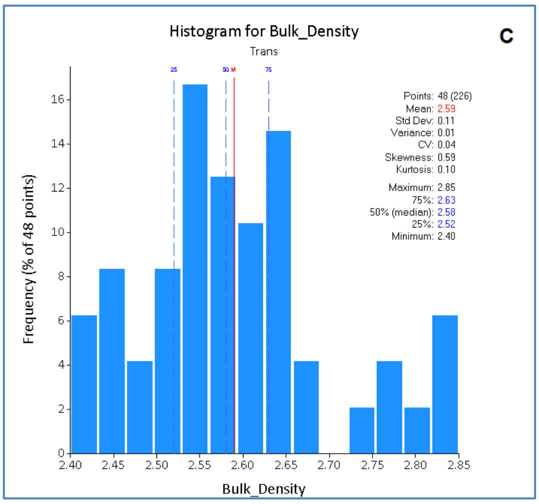
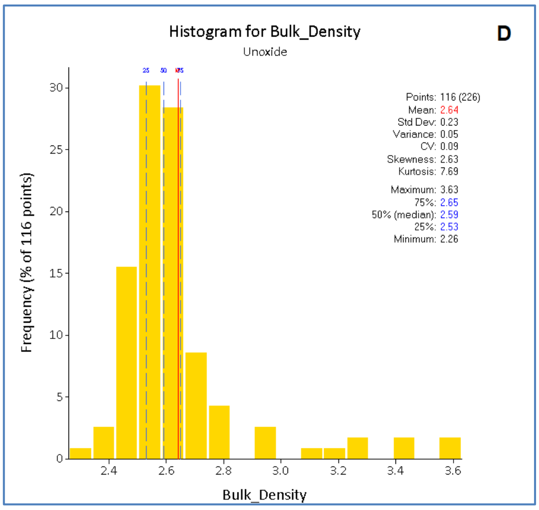


Figure 11.5 - Density histogram - Fresh



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 QAQC 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 11-1.

Table 11-1 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
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

DESCRIPTION	STANDARD_ID	ELEMENT	NOMINAL VALUE	STD_DEVIATION
		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 11-2.

Table 11-2 Standard blank samples used at ATO

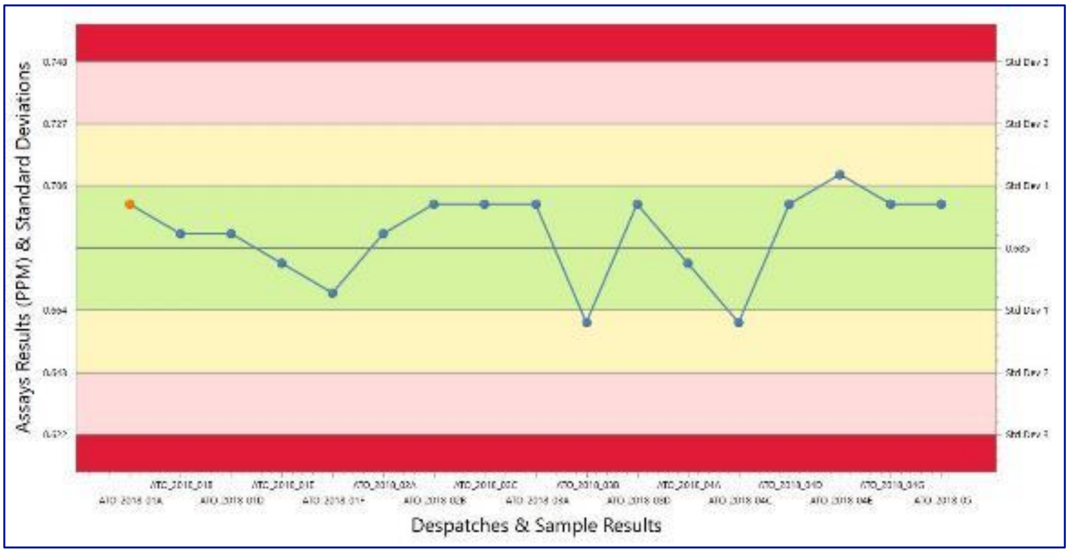
STANDARD_ID	DESCRIPTION	ELEMENT	NOMINALVALUE	STD_DEVIATION
OREAS 21e	Oxide quartz blank	Cu (ppm)	5.68	0.81
		Pb (ppm)	<1	0
		Zn (ppm)	2.91	0.56
		Au (ppb)	<1	0
		Ag (ppm)	<0.05	0
OREAS 24d	Basalt Blank pulp	Cu (ppm)	43.2	2.29
		Mo (ppm)	4.46	0.350
		Pb (ppm)	3.56	0.44
		Zn (ppm)	104	7
		Au (ppb)	<1	0
		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

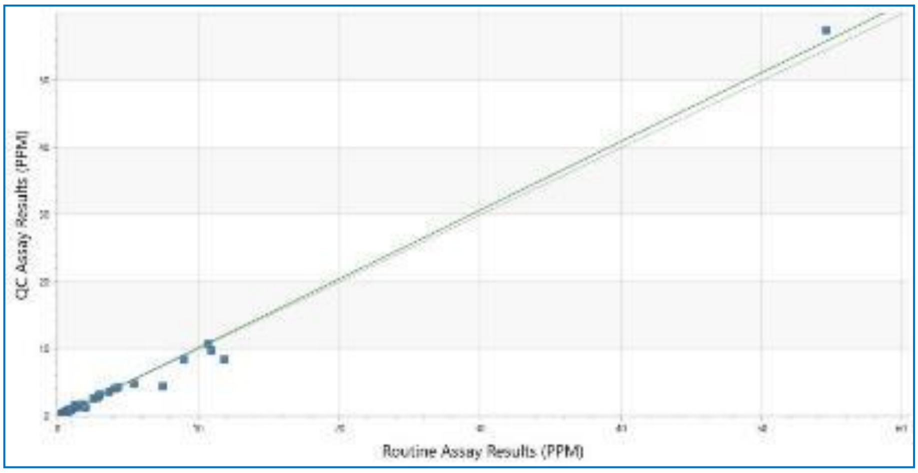
Shewhart plots (Figure 11.6) is constructed from SRM assay results to show the assay mean and distribution compared to the expected mean and distribution as defined by the certificates.

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 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 GeoRes QP's overall opinion* 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.

*This opinion is qualified by the fact that up to early 2021 the GeoRes QP had not physically been able to sight any of the Project's geology or drilling himself (sue to the travel restrictions caused by the Covid Pandemic).. Although all information had been remotely supplied to the Author (by Steppe, Steppe's Alternate QP and past reporting) he nevertheless found 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. Since 2021 the QP visited site in 2022 and observed drilling operations there – which largely confirmed his previous opinions.

12 DATA VERIFICATION (BY QP)

Relevant data verification here applied to that data used in the 2021 Resource estimation work – namely drilling data.

12.1 Data verification

Data verification by the 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[25].

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.

12.2 Limitations on data verification

The principal limitations to the QP's data verification in 2021 (and understanding of the Project generally) were lack of a site visit and geological rock type logging analysis.

Site visit: The initial 2021 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. . This limitation was removed by the QP's site visit in 2022.

[25] 2017 NI 43-101. Section 12.1, pp 99.

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

13 MINERAL PROCESSING & 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.

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.

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

14 MINERAL RESOURCE ESTIMATE

The Section discusses and details the methodology and results of the GeoRes QP's 2020/1 estimation of gold and related precious and base metal Mineral Resources in ATO's four deposits for this Report. Those new Resources were published in a 3/2021 NI 43-101 Report, superseded the previous ones published for the three deposits in the 2017 NI 43-101 Report[26] (Pipe 1, 2 and 4), and provided the initial Resources on the fourth deposit (Mungu).

These 11/2022 Resources have been reported from the 2021 model using new classification of the oxidation levels (Oxide/Transitional/Fresh) and below a new topographic surface exclude mining in the interim.

Resource estimation is described in terms of:

- Introductory statements and certifications – qualifying the Author QP, reporting codes and source of data.
- Background – emphasising 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. In 2022 both surfaces were re-estimated and used to report updated Resources.
- 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

The 2021 Resource estimation was independently undertaken by the GeoRes QP / CP. He makes the following Statements as to his competency under the JORC and NI 43-101 Codes. He also states the CIM

[26] 2017 NI 43-101. Section 14, pp118 onwards

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.

JORC (2012) Competent Person (CP) Statement: The Geological 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 and 2022 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 mineralization 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.

NI 43-101 Qualified Person (QP) certification:

Certification: Robin A Rankin certifies, that in relation to his reporting of ATO 2021 and 2022 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.

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 Pipes 1, 2 and 4 by GSTATS (see Section 6.3). 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 Pipes 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.
- 2022 data – new topography and oxidation level data.

14.3.1 2017 data

Drill hole data: The 2017 GSTATS drill hole data base (for Resource estimation) contained 265 diamond drill holes[27]. 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 (see Section11.4). Those density averages were used for the 2021 estimation reporting here.

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.

[27] 2017 NI 43-101. Section 14.1.1, pp 118

Topography 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.

14.3.3 2022 raw data

Topography data: New topographic data was supplied by Steppe in July 2022 incorporating mining in the two open pit (1 and 4) shells. The data corresponded to the surface at 1st of July 2022 and was supplied as string data.

Oxidation level data: New oxidation surface data was supplied by Steppe in July 2022. The data in tabulated csv format was of depths to the base of oxidation and base of transition (to fresh) interpreted by Steppe personnel in ATO and Mungu drill holes.

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 (eg 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 back-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 2020 raw 1 m interval surface contour data strings were all supplied as closed polygons (Figure 14.10). These were edited in Global Mapper software to remove the closures. The 7/2022 topography data was similarly pre-processed.

14.7 Drill hole database

Data source: All drill hole and trench data was 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 Table 9-3.

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 2021 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 (Table 14-1).

Table 14-1 Deposit domains

Domain Number	Deposit
1	Pipe 1
2	Pipe 2
4	Pipe 4

Domain Number	Deposit
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 2021 oxidation levels were loaded as down-hole layer intercepts from the raw lithology data.

The codes for the three intercepts (from surface down) were:

- OX – oxidised material.
- TR – transitional material.
- FR – fresh material.

In 2022 the down-hole oxidation levels intercepts were interpreted by Steppe geologists and supplied in spreadsheet form.

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:

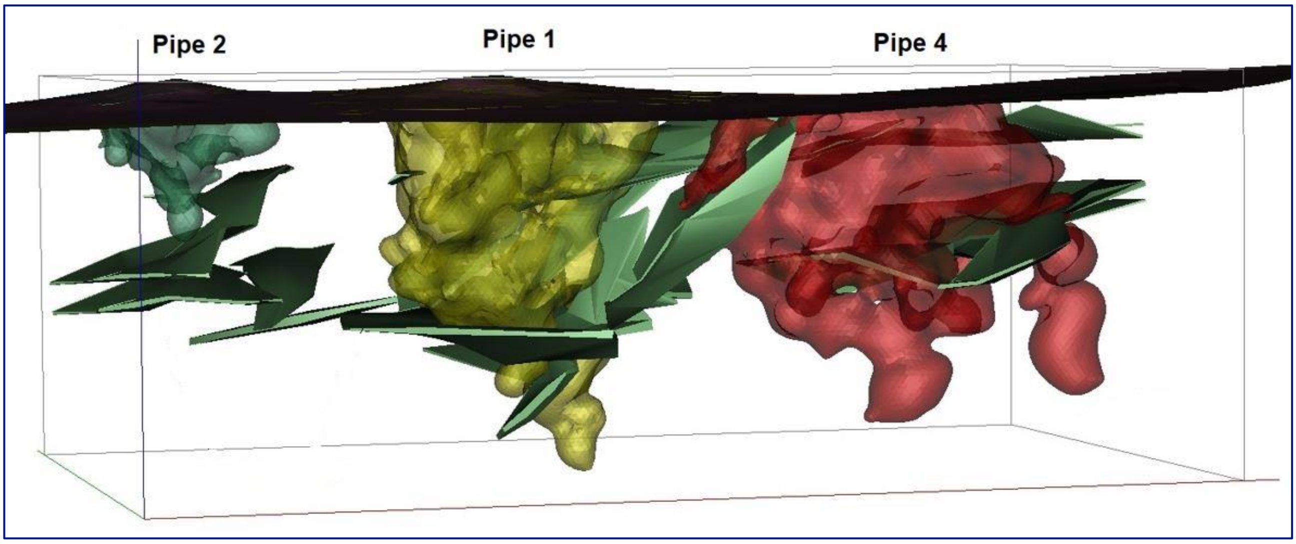
- The previous 2017 estimation was based on "grade shells" (see below).
- Lithological logging appeared too complex to deal with.
- Mineralisation appeared reasonably continuous over long sections of holes, and correlatable with similar sections in adjacent holes.
- These mineralised intersections were 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"[28].

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.

Figure 14.1 - [29] Previous 2017 grade shell models of Pipes



The grade shell models in Figure 14.1Figure 14.1 - Previous 2017 grade shell models of Pipes

are seen to be fairly complex and tortuous in shape.

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.

[28] 2017 NI 43-101. Section 14.2.1, 2nd paragraph, pp119.

[29] 2017 NI 43-101. Section 14.2.1, Fig 14.2, pp120.

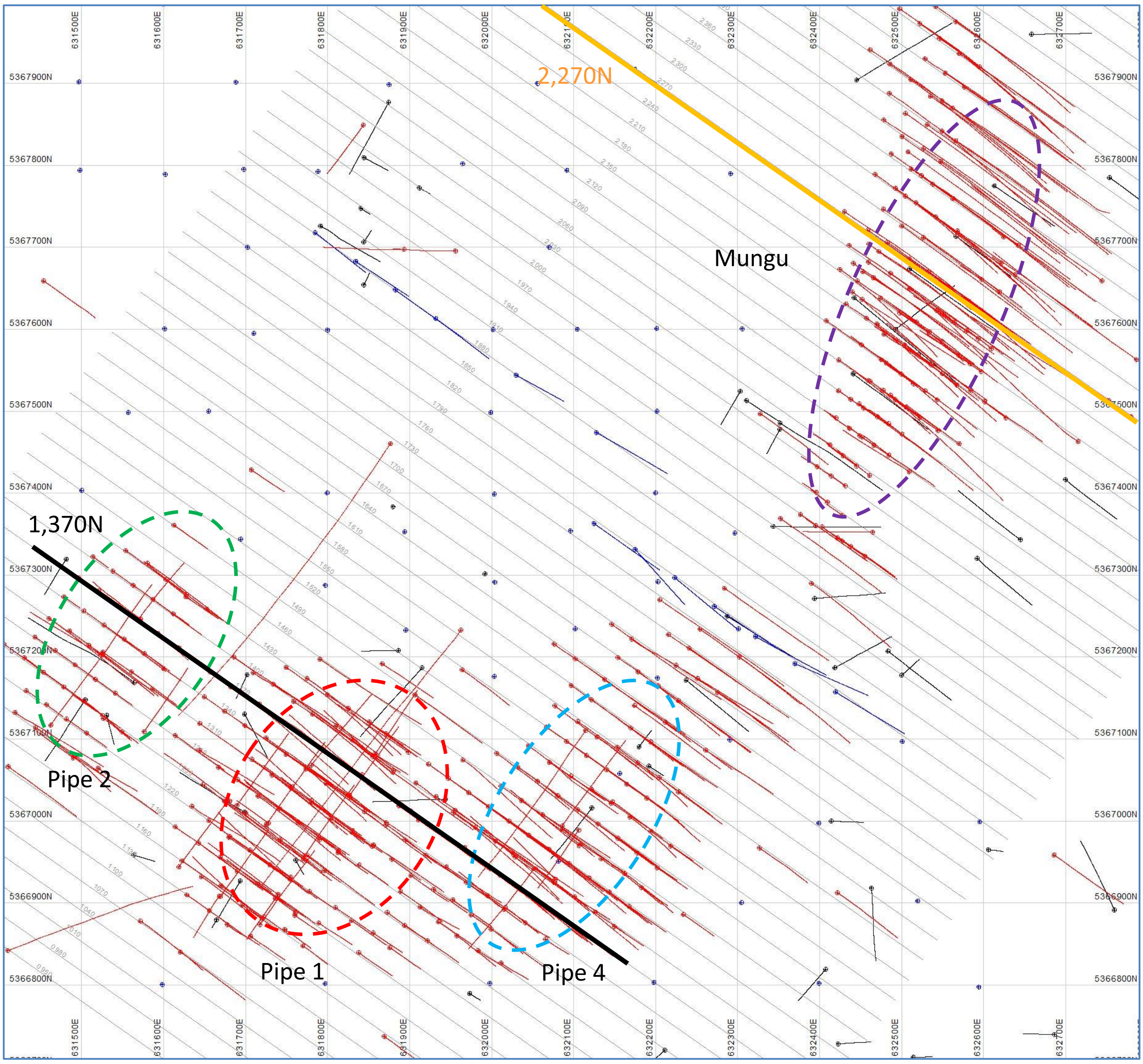
The method involved:

1. Creating a series of parallel vertical cross-sections through the drill holes (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. This step was iteratively combined with the outline digitising as more practical 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 Pipes 2, 1 and 4 marks cross-section 1,370N as shown in the following Figure 14.3.

Figure 14.2 - Drill hole & trench locations – deposit area 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 Figure 14.2), shows outlines through Pipes 2 (left, green), 1 (centre, red) and 4 (right, cyan). Pipes 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. The Figure also shows the oxidation level surface models (annotated in blue).

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

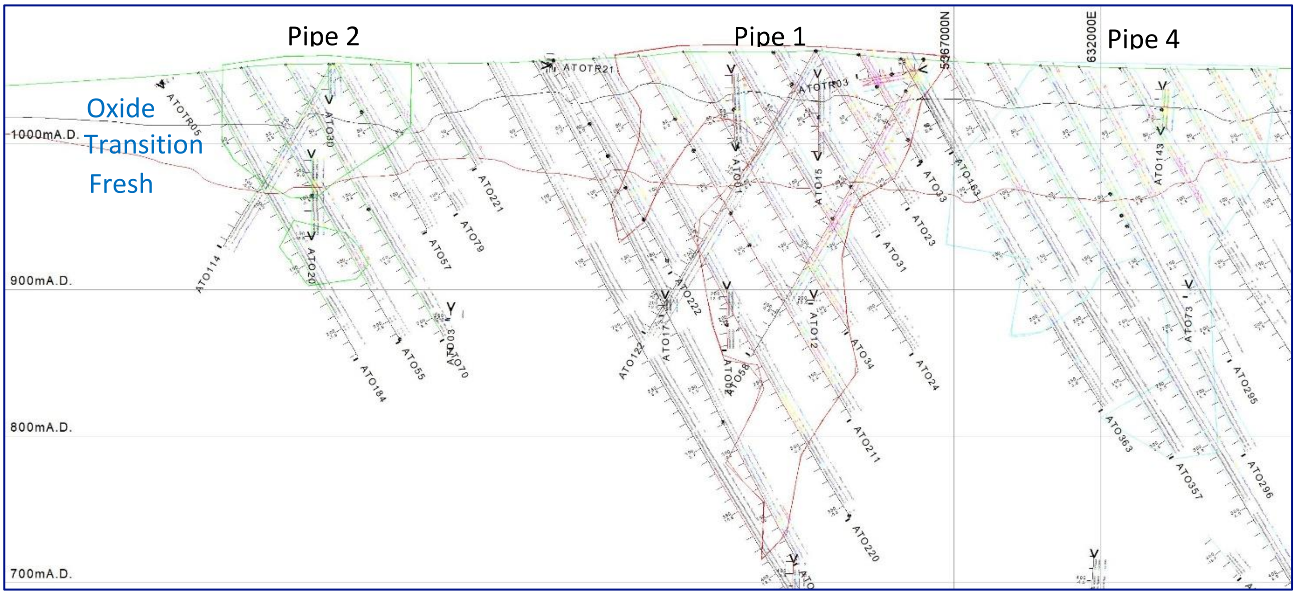
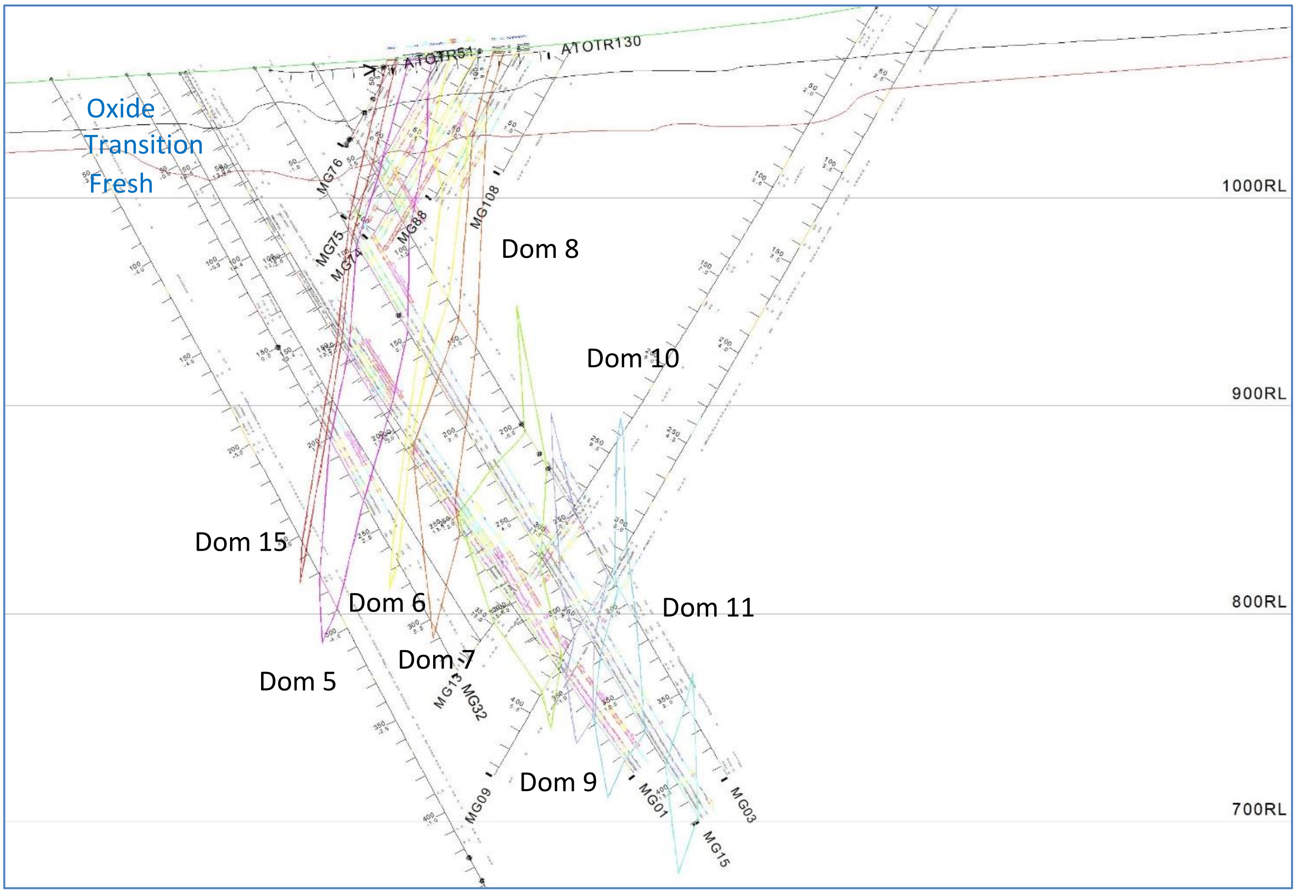


Figure 14.4 shows Mungu outlines on central vertical cross-section 2,270N (location shown by the thick orange line in Figure 14.2). This cross-section shows all 8 domain outlines (domain 5 to 11, and 15) interpreted at Mungu. The domains are tall and thin, nearly vertical, sub-parallel and strike 035 normal to the cross-sections. The Figure also shows the oxidation level surface models near surface.

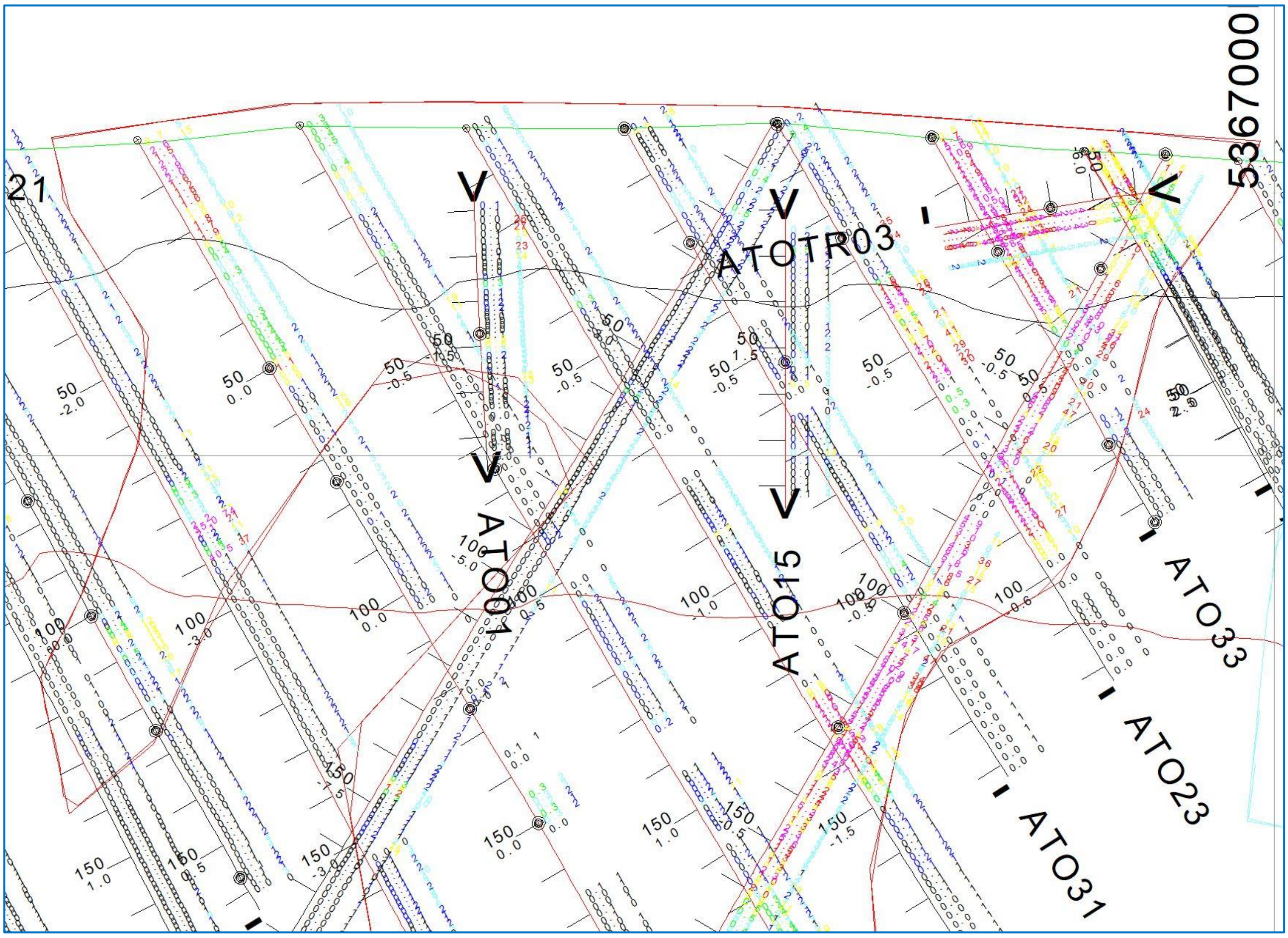
Figure 14.4 - Mungu outlines interpreted on cross-section 2,270N



Deposit outline grade cut-offs: Figure 14.5 - Close-up of hole ATO23 (Pipe 1) on cross-section 1,370N

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



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).

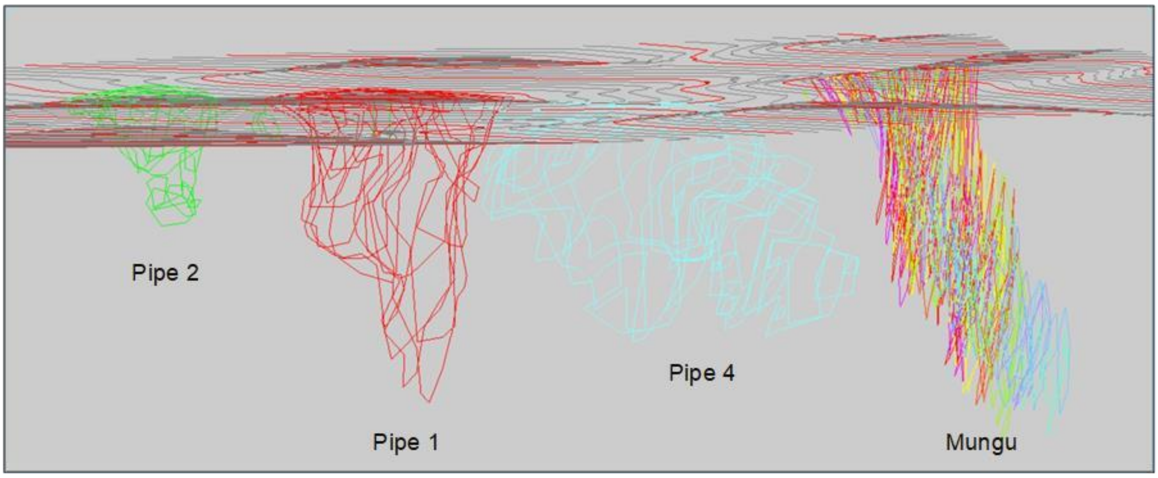
Figure 14.6 - Hole ATO23 Colour Coded Assay Spreadsheet Data

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		

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



14.9.2 Oxidation level interpretation

2021 oxidation interpretation: Interpretation of the degree or level of oxidation at the deposits was done in 2021 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 GeoRes 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 (14.11.2) and viewed in cross-section.

2022 oxidation interpretation: In 2022 oxidation levels were re-visited as mining was showing that lower parts of material classified as 'transitional' were clearly found to be 'fresh'. Previously (2020/21) the surface marking the interface between partly weathered rock and fresh rock (the base of transitional material or top of fresh material) was interpolated from the very deepest depths down-hole of any noticeable oxidation. That methodology produced a relatively thick (~21 m) transitional layer, nearly as thick as the overlying oxide layer (~23 m). In practice that transitional surface was shown by geological observation in the pits to be considerably too deep, and the transitional layer to be thinner than modelled.

Consequently more realistic down-hole depths of the base of predominant partial oxidation were interpreted in all drill holes by Steppe's geologists on-site. Those depths were provided in July 2022 and were re-modelled (as before) into a new base of transition grid surface (2022_TR). That surface was ~15 m higher than the previous surface, with the transitional layer now only ~4.5 m thick. New Resources were reported using this surface, with the volumes in the ~15 m difference between the old and new surfaces now reporting with the fresh material.

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 14.10 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

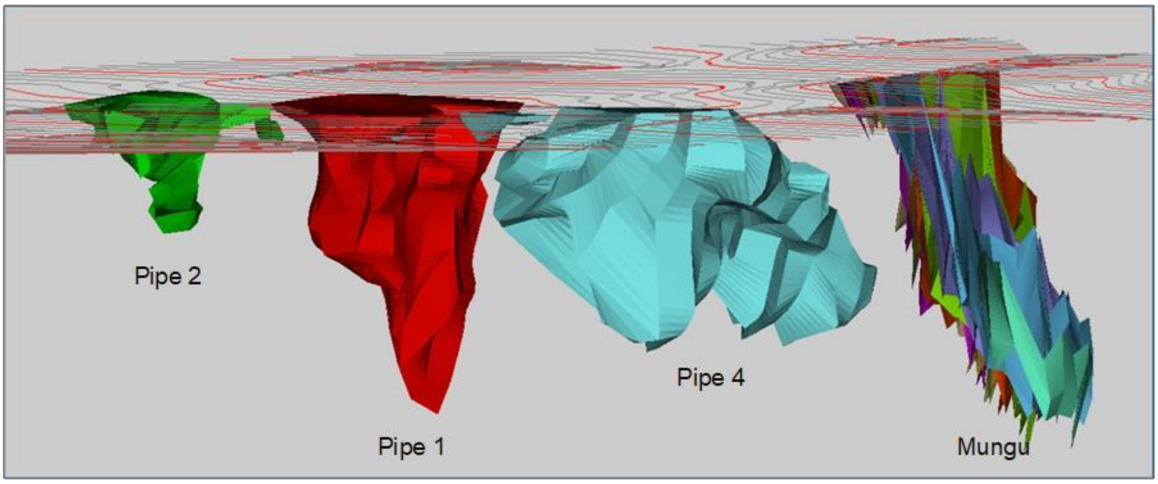
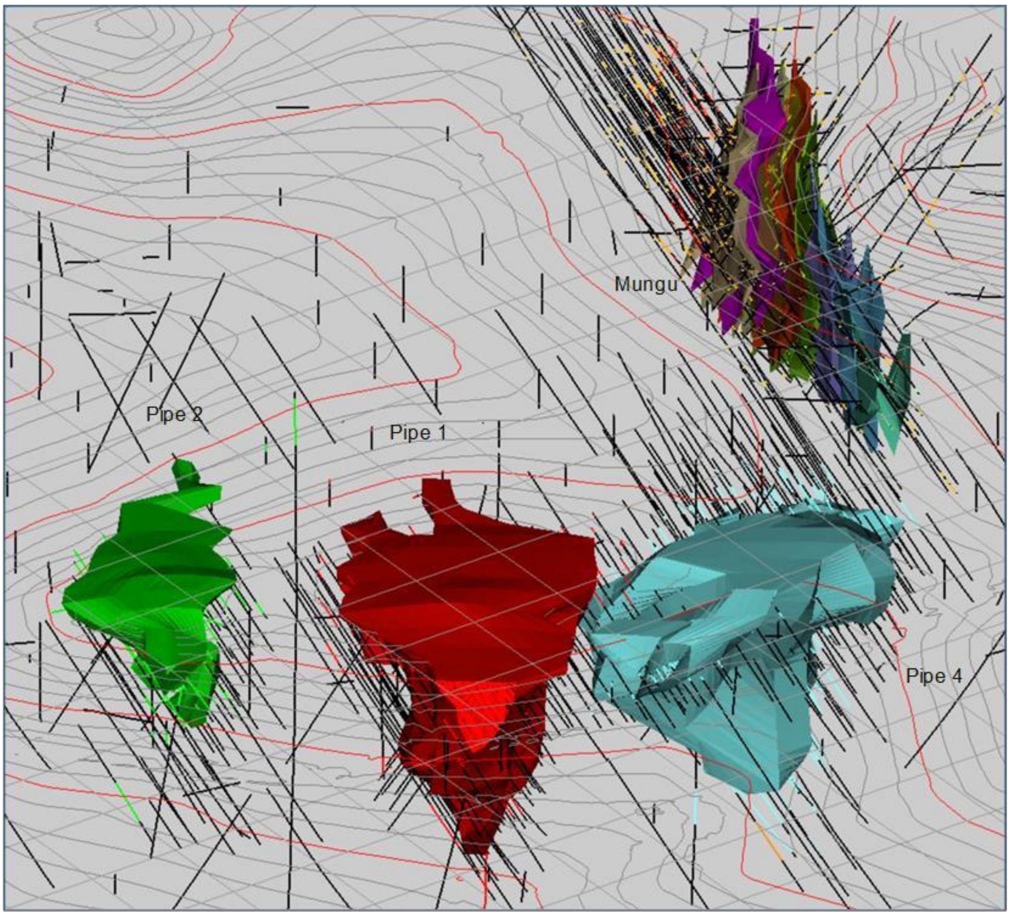


Figure 14.9. shows all of the deposits looking towards 035° and down 35° - the view normal to the cross-sections and along strike. The Figure also shows the drill holes in 3D. The deposits are seen to be thinnest in this view as they are elongated along strike. And the multiple sub-parallel and sub-vertical Mungu lodes are very clearly parallel to this strike view direction.

Figure 14.9 - Wire-Frame Models of All Deposits – Looking 035°



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 14.10 – 2021 Topography Data

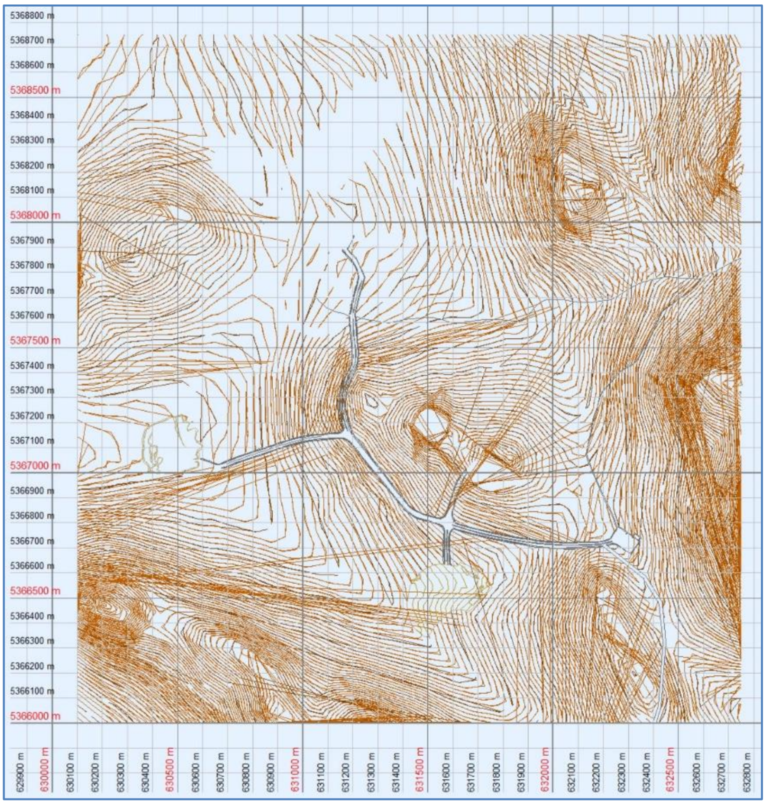
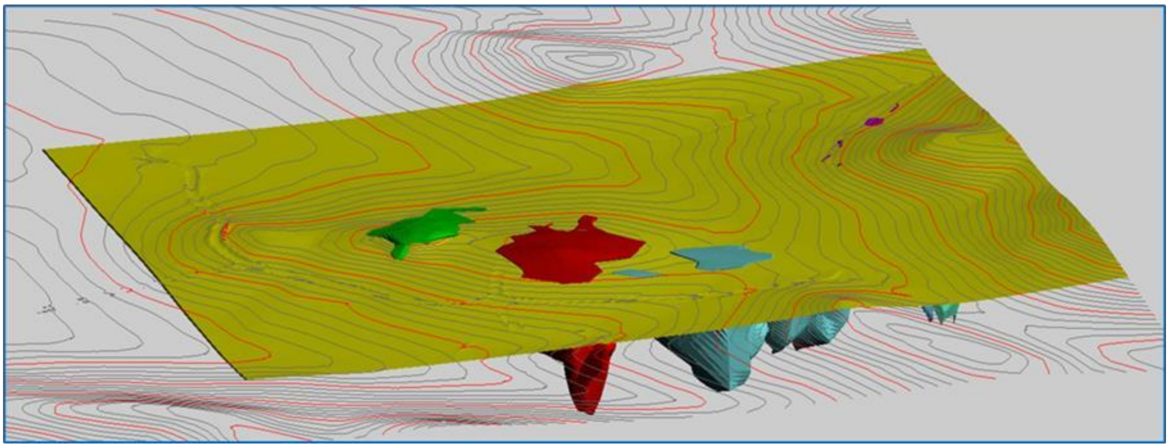


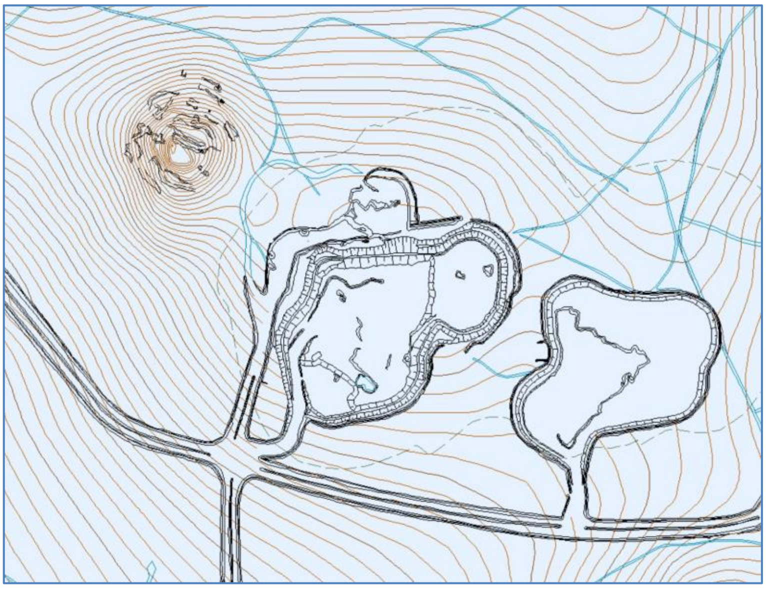
Figure 14.11 shows the topography gridded surface as a yellow solid above the wire-frame deposit models. The surface is also contoured at 2 m intervals in grey and 10 m intervals in red. The view is towards 015° and down at 20°. Lighting is from the south-east.

Figure 14.11 - Topography Surface Model



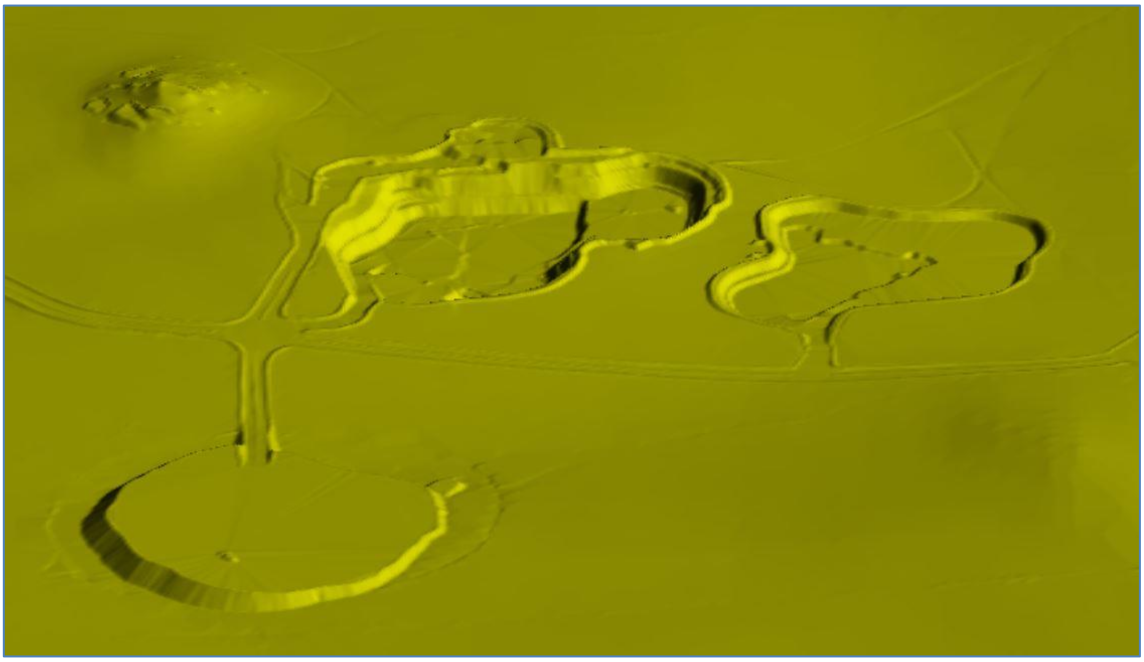
2022 topography: Updated topography data was supplied as of 1st July 2022. Figure 14.12illustrates the raw data in plan over the ATO Pipe area (Pipe 1 on the left (un-mined), Pipe 1 in the centre and Pipe 4 on the right). Data consisted of 1 m interval contour strings (brown in Figure) and well as pit bench and other feature strings (black in Figure 14.11)

Figure 14.12 – 2022 Topography Raw



After manipulation the data was again triangulated and then converted into a 2*2 m grid (20220701). Figure 14.12 shows the gridded surface in perspective view, looking down towards the north.

Figure 14.13 – July 2022 topography in perspective

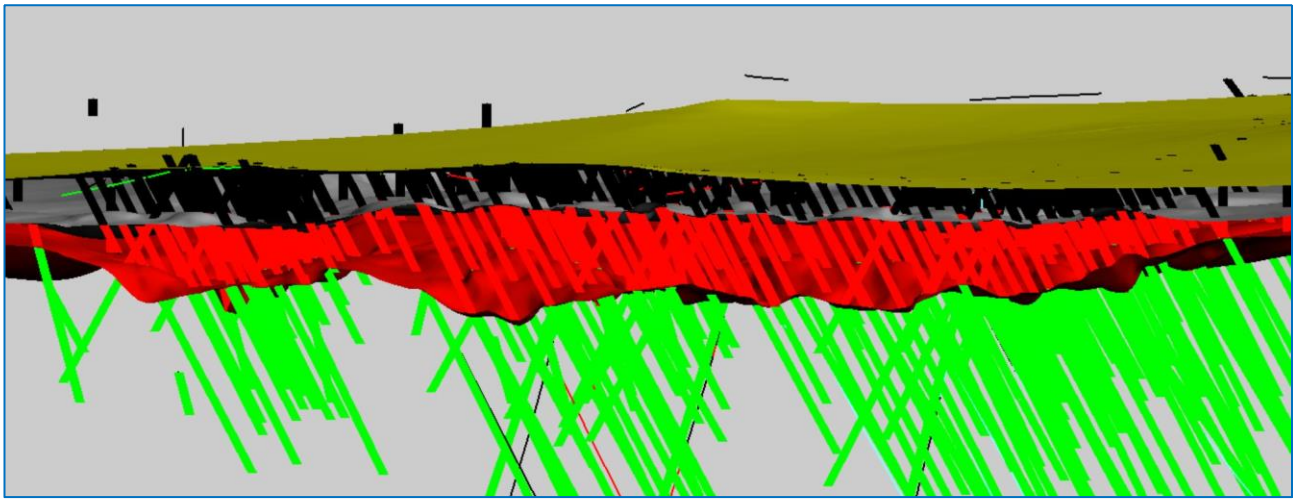


14.11.2 Oxidation level surfaces

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.

2021 oxidation surfaces: Figure 14.14 illustrates and E/W cross-section through the oxidation (grey) and fresh (red) surfaces below topography (yellow). The view is through the middle of Pipes 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 oxidised parts in black, the transitional parts red and the lower fresh parts green.

Figure 14.14 – Oxidation Surface Models

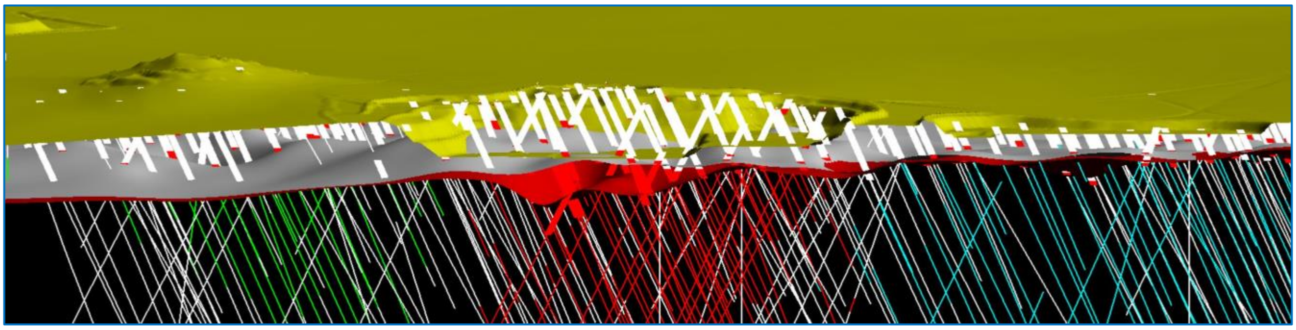


In Figure 14.14 the transitional layer (red) is seen to be considerably thicker than the overlying fully oxidised layer (grey).

2021 Oxidation levels were also shown on the cross-sections above (Figure 14.3, Figure 14.4 Figure 14.5).

2022 oxidation surfaces: Figure 14.15 illustrates the 2022 re-modelled oxidation (grey) and fresh (red) surfaces below topography (yellow). The view is similar to above, an E/W cross-section through the middle of now partially mined Pipes 1 and 4 (centre and right), 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 oxidised parts in white and the transitional parts red.

Figure 14.15 – 2022 Oxidation surface models



The 2022 oxide surface level remained little changed from the 2021 one. However the transitional surface is ~15 m higher than before and now only ~4.5 m thick on average (illustrated by the closeness of the grey and red surfaces).

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 2 tabulates simple raw (un-composited) gold statistics for all samples and then for Pipes 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 Pipes 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 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 14.16 to Figure 14.20 show gold histograms for Pipes 1, 2 and 4. Gold is seen to be log normal in distribution. The log histogram for Pipe 1 indicates several populations are present.

Figure 14.16 – Gold Histogram Pipe 1 - Normal

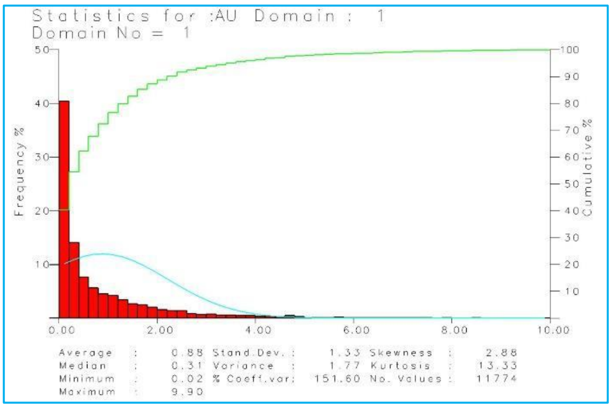


Figure 14.17 – Gold Histogram Pipe 1 - Log

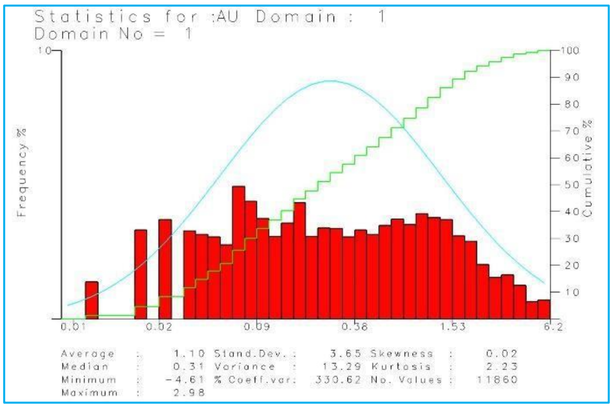


Figure 14.18 – Gold Histogram Pipe 2 - Normal

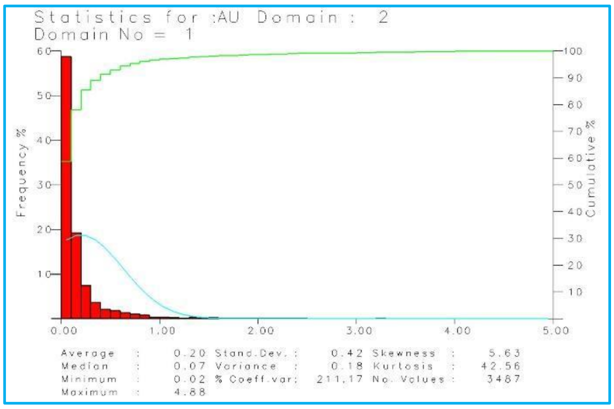


Figure 14.19 – Gold Histogram Pipe 4 - Normal

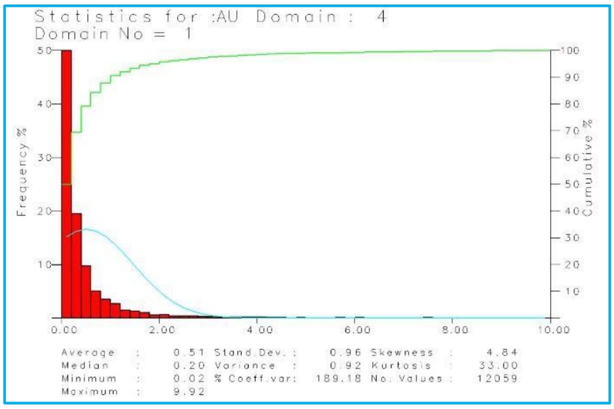


Figure 14.20 – Gold Histogram Pipe 4 - Log

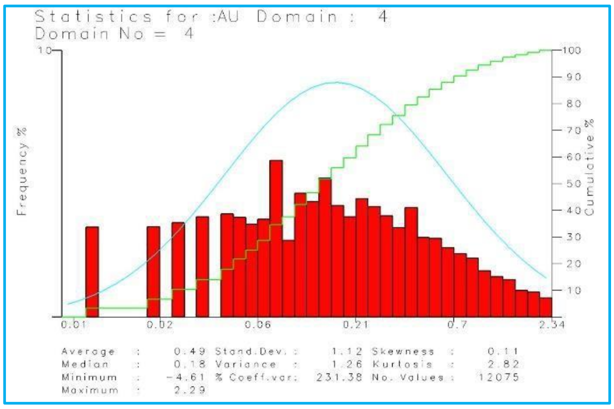
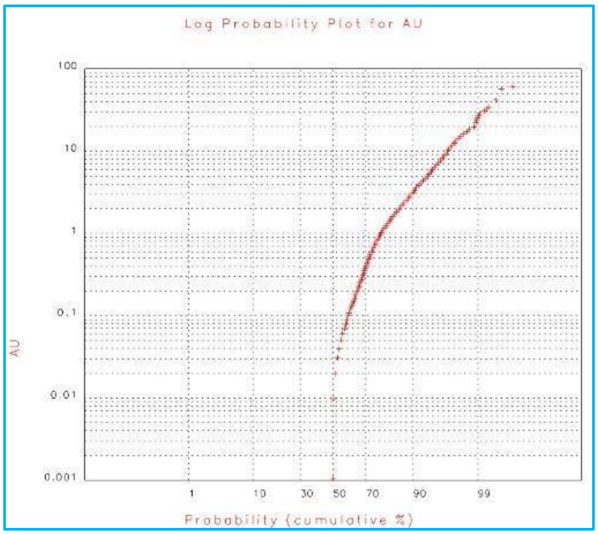


Figure 14.21 – Gold Lop Prob Pipe 1



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 14.21 is in the horizontal E/W direction – and has a maximum range of ~25 m. Figure 14.23 is dipping 45° up towards the east (or 45° down towards the west) and has a maximum range of 45 m.

Figure 14.22 – Gold Pipe 4 Variogram 0°@090°

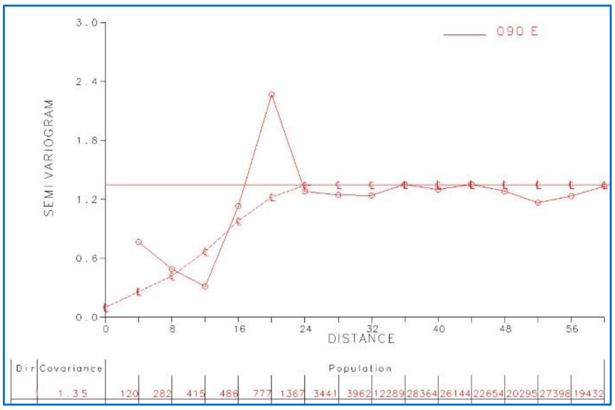
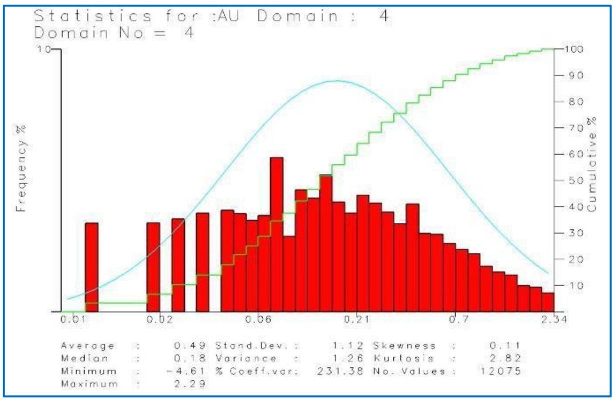


Figure 14.23 – Gold Pipe 4 Variogram 0°@090°



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 .

Continuity: The 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, see Section 14.18) proved to be only ~28 m in Pipes 1,2 and 4 and ~31 m at Mungu.

The 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 Pipes 2 and 4 it was an intermediate 45°W dip. And at Mungu it was a vertical dip with the lodes striking 033°.

1Resource 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 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: Table 14.3 and Table 14.4. 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.

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.

14.14 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.

14.14.1 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.

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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 - Pipes 1, 2 and 4

Table 14.6 - Block estimation statistics - Pipes 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

14.14.3 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 (see Section 14.18). These variables were:

- • AU_D – the average distance of samples used.
    - With the Pipes 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.

14.14.4 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 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.

14.14.5 Block grade cross-sections

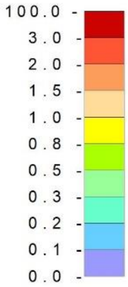
The following Figures illustrate typical vertical E/W cross-sections through the ATO deposits and the block models.

Blocks are colour-coded on gold grades according to the ranges and colours in Figure 14.24

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 14.24 – Gold



Pipes 1, 2 and 4: Figure 14.25 to Figure 14.29 illustrate a series of east/west cross-sections, looking north, though Pipes 1, 2 and 4 (Pipe 2 on the left (e.g. Figure 14.29, west), Pipe 1 in the middle (e.g. Figure 14.27), and Pipe 4 on the right (e.g. Figure 14.25, east)). The sections are ordered from south to north.

Figure 14.25 – Pipe 1, 2 and 4 gold block cross-section 5,366,902N

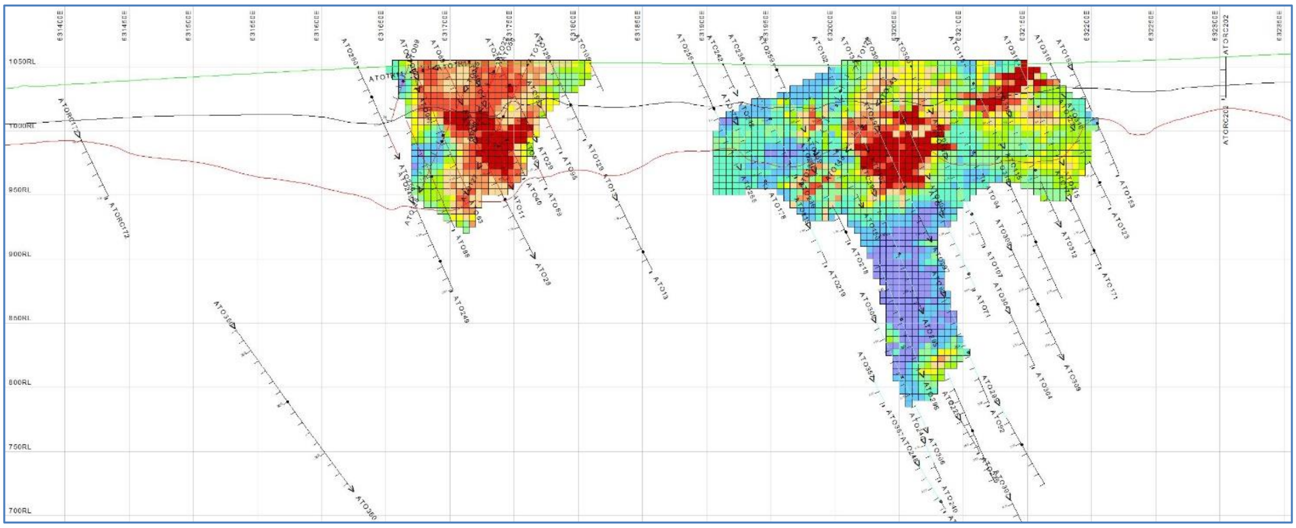


Figure 14.26 – Pipe 1, 2 and 4 gold block cross-section 5,367,002N

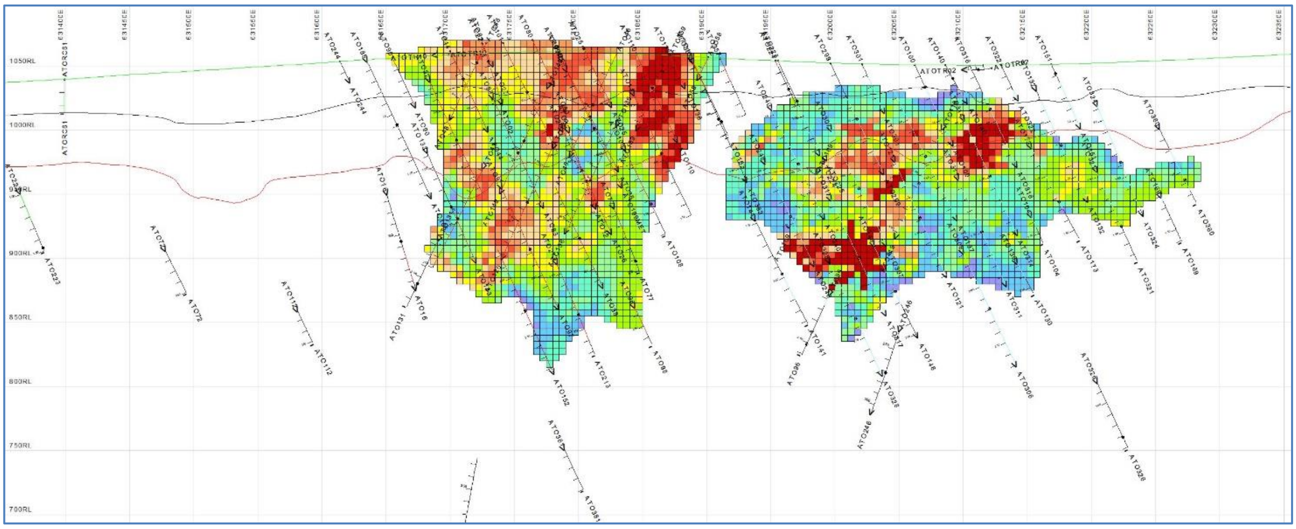


Figure 14.27 – Pipe 1, 2 and 4 gold block cross-section 5,367,102N

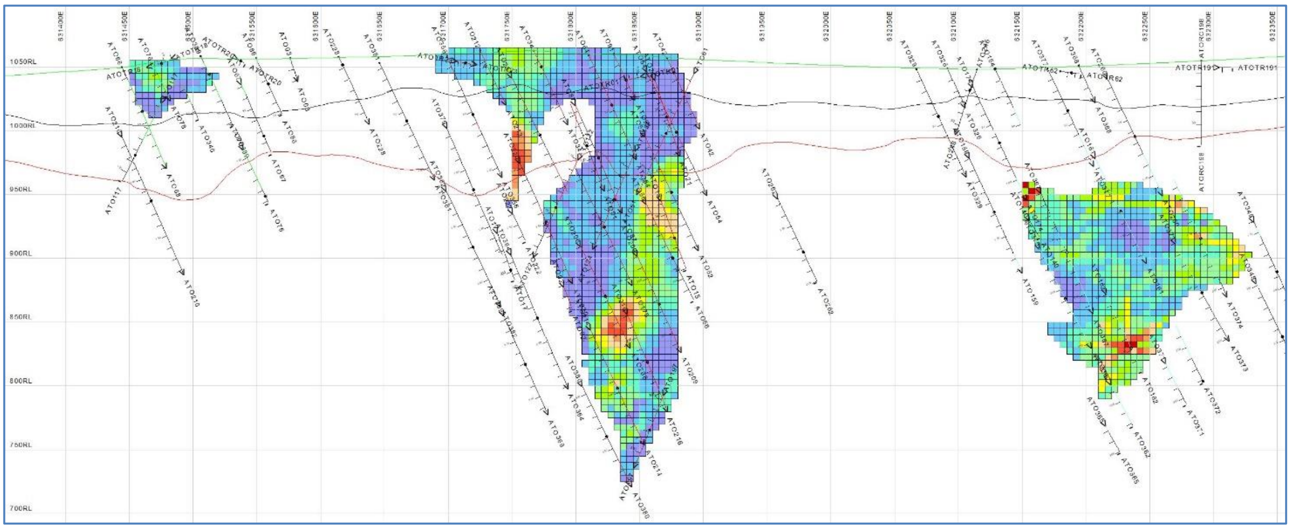


Figure 14.28 – Pipe 1, 2 and 4 gold block cross-section 5,367,152N

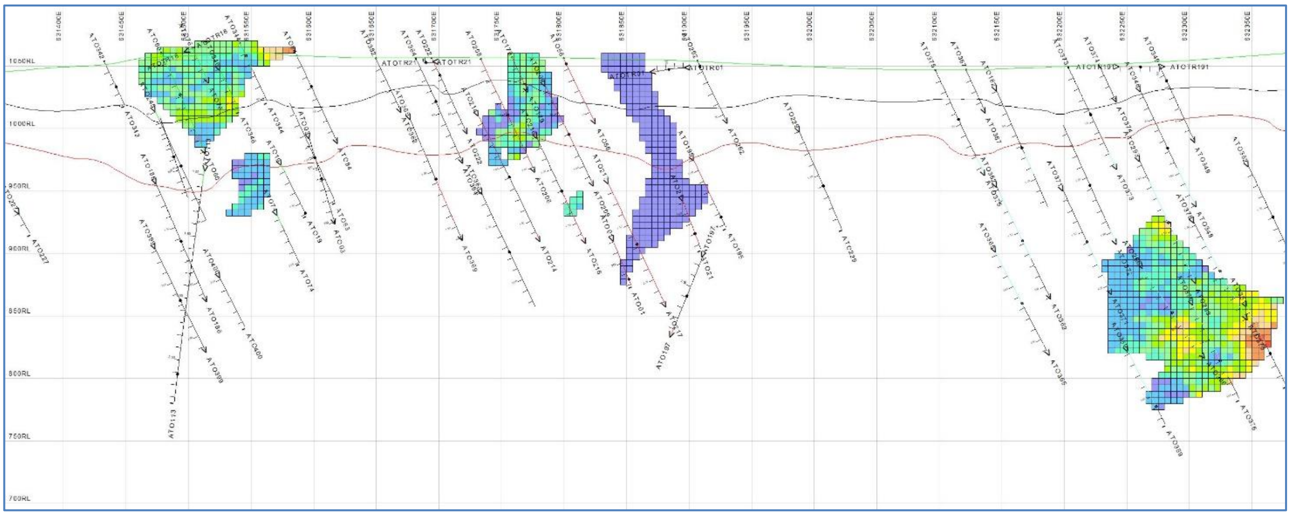
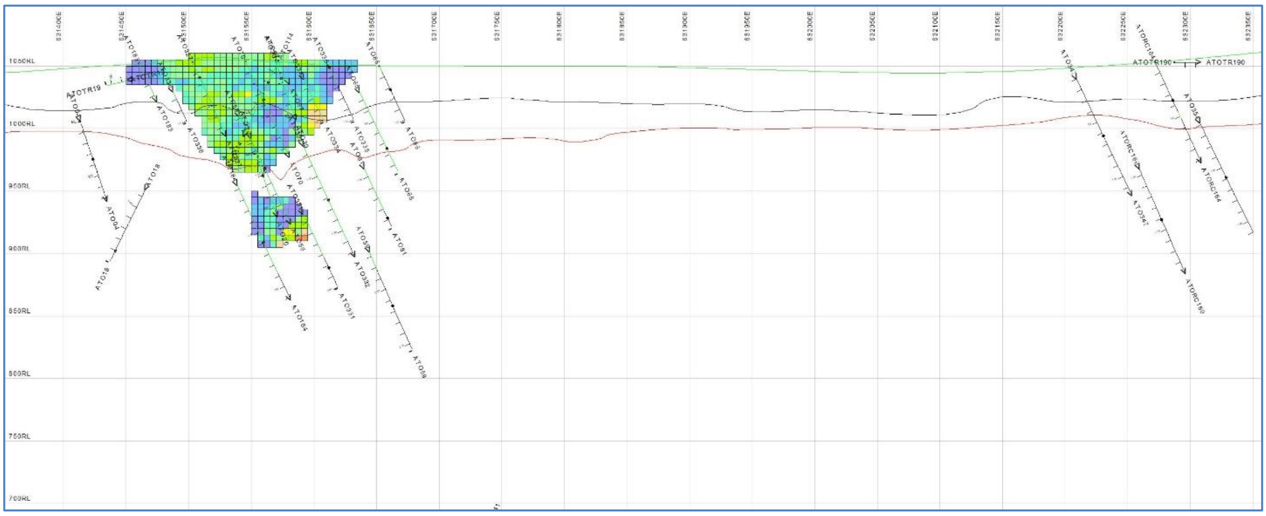


Figure 14.29 – Pipe 1, 2 and 4 gold block cross-section 5,367,252N



Mungu: Figure 14.30 to Figure 14.33 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 14.30 – Mungu gold block cross-section 5,367,502N

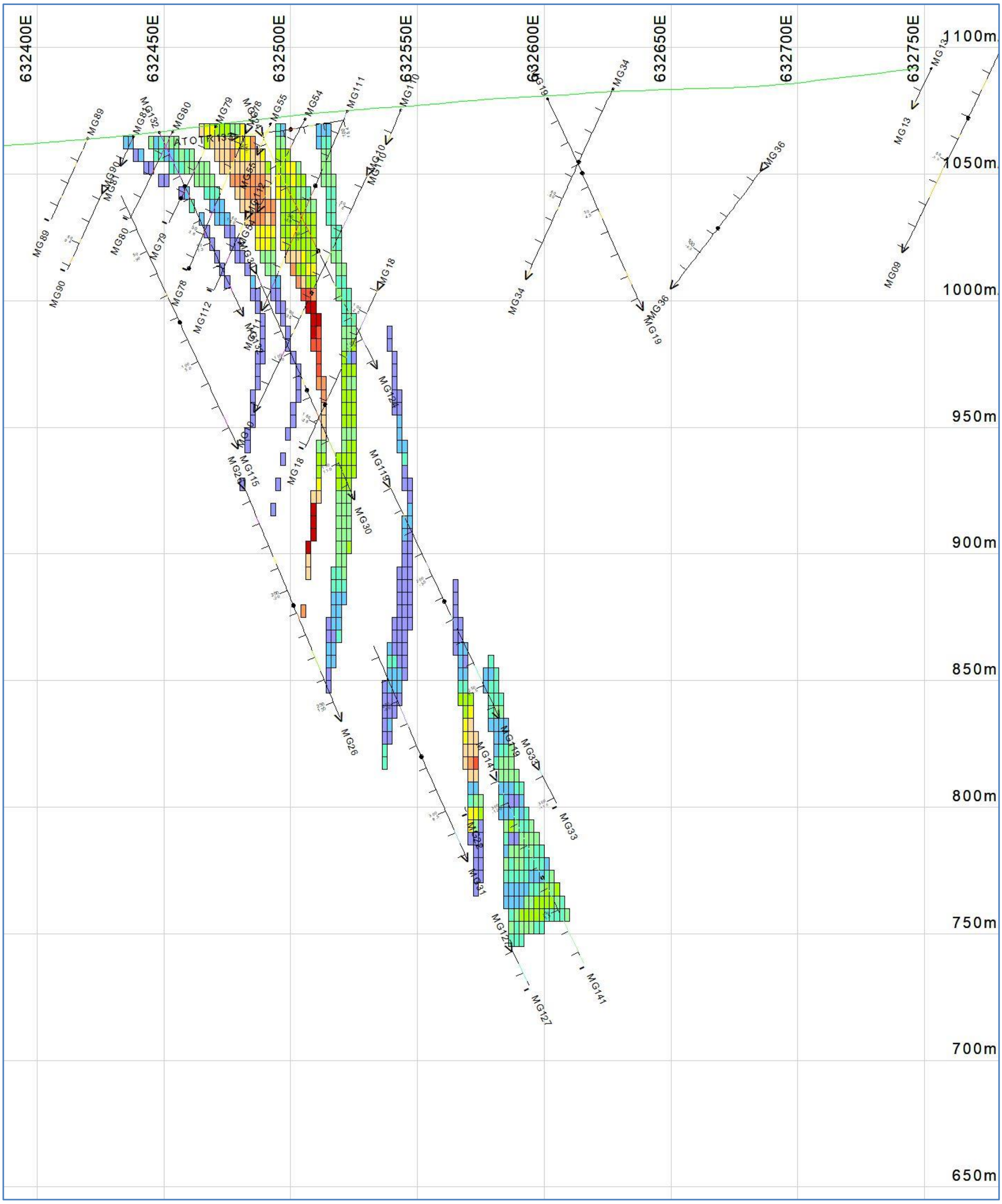


Figure 14.31 – Mungu gold block cross-section 5,367,602N

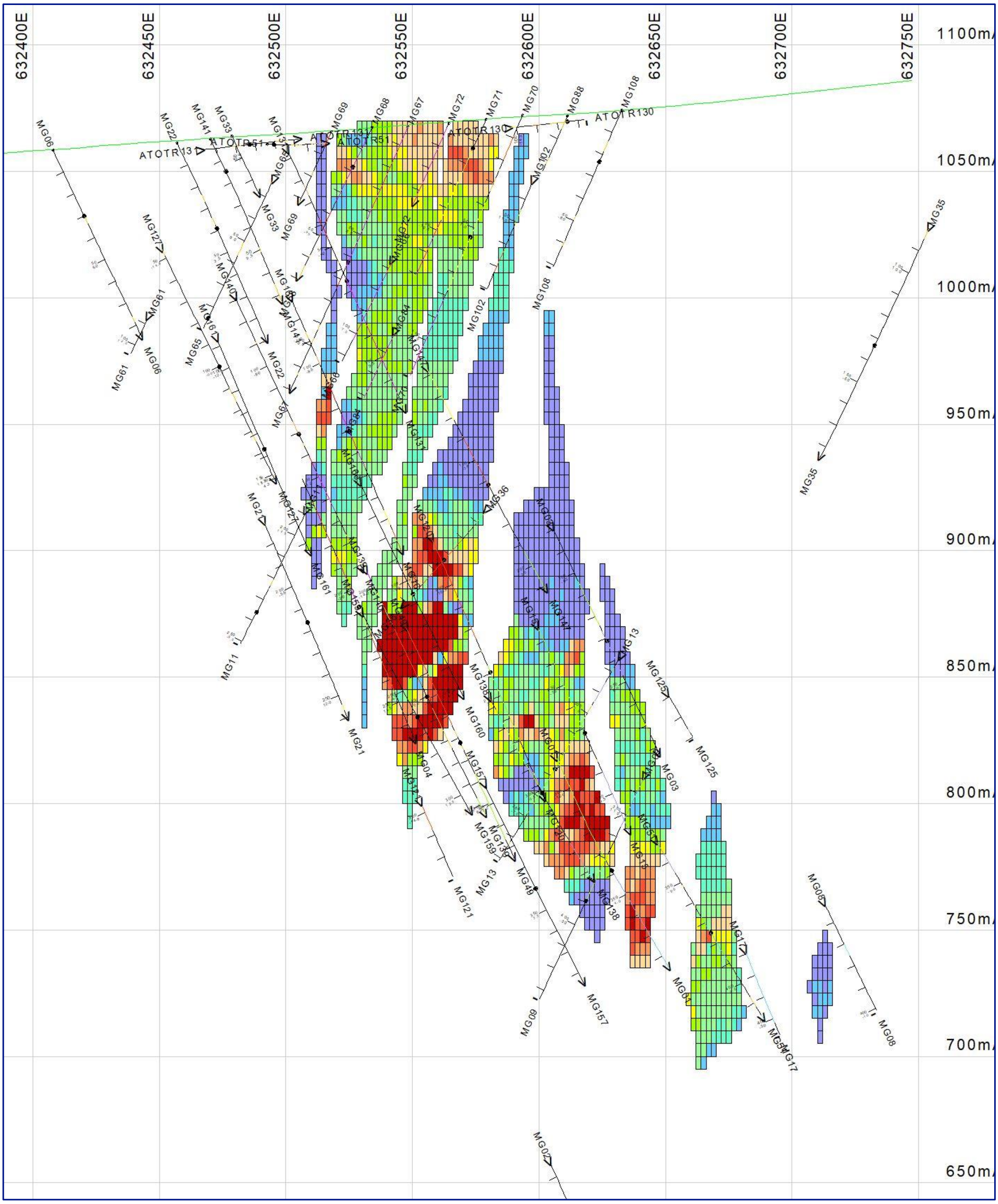


Figure 14.32 – Mungu gold block cross-section 5,367,702N

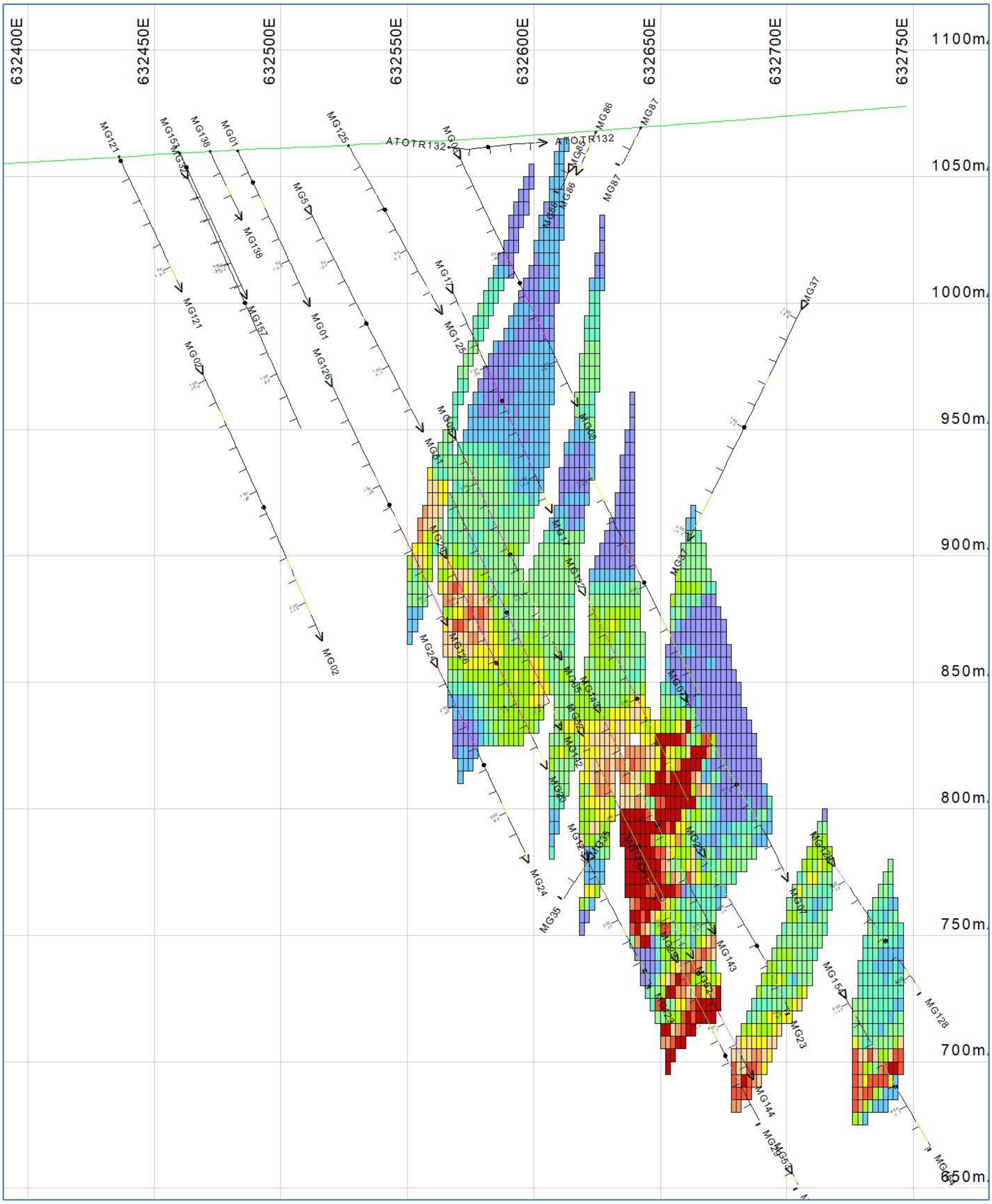
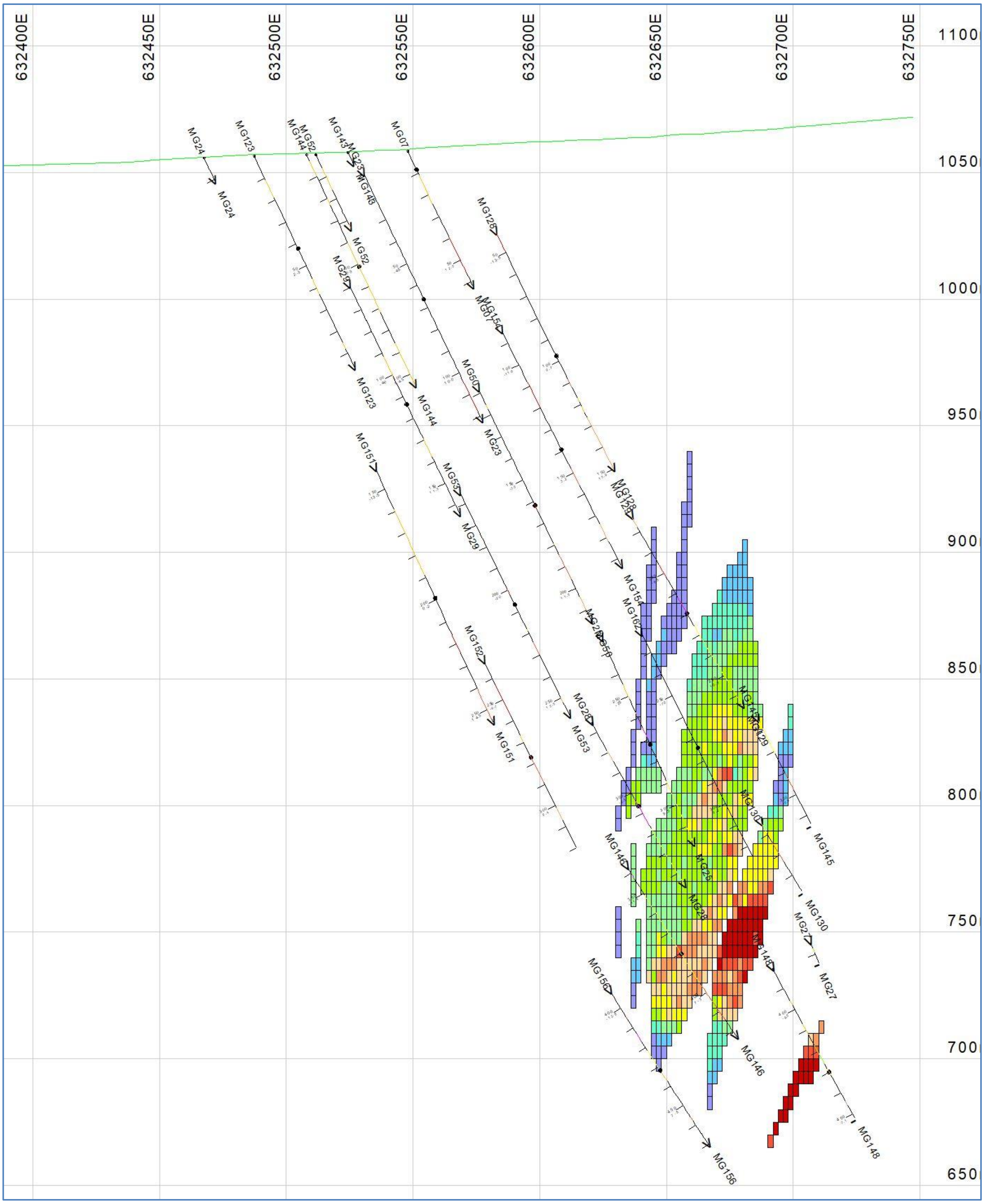


Figure 14.33 – Mungu gold block cross-section 5,367,802N



Mungu: Figure 14.34 to Figure 14.36 illustrate a series of typical level cross-sections, looking vertically down, though Mungu, showing colour-coded gold blocks. The sections are top downward

Figure 14.34 – Mungu gold block cross-section 5,367,802N

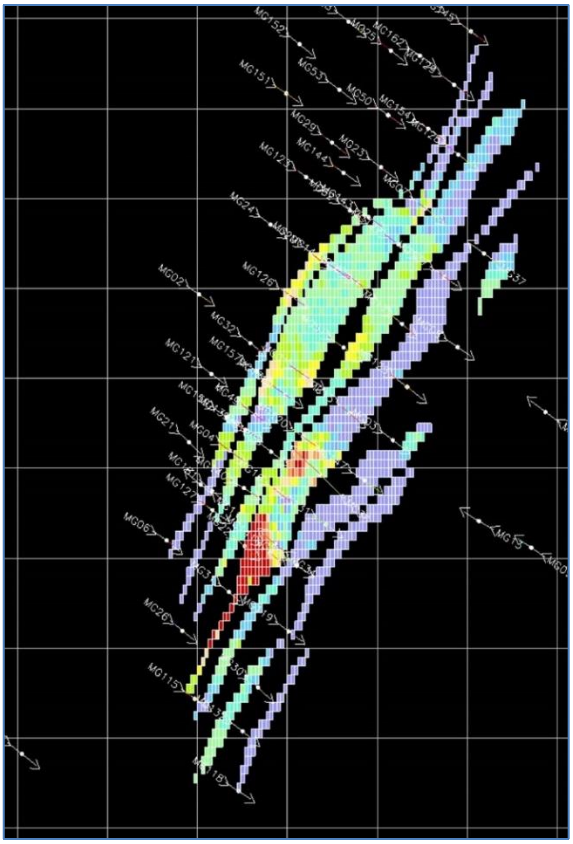


Figure 14.35 – Mungu gold block cross-section 5,367,802N

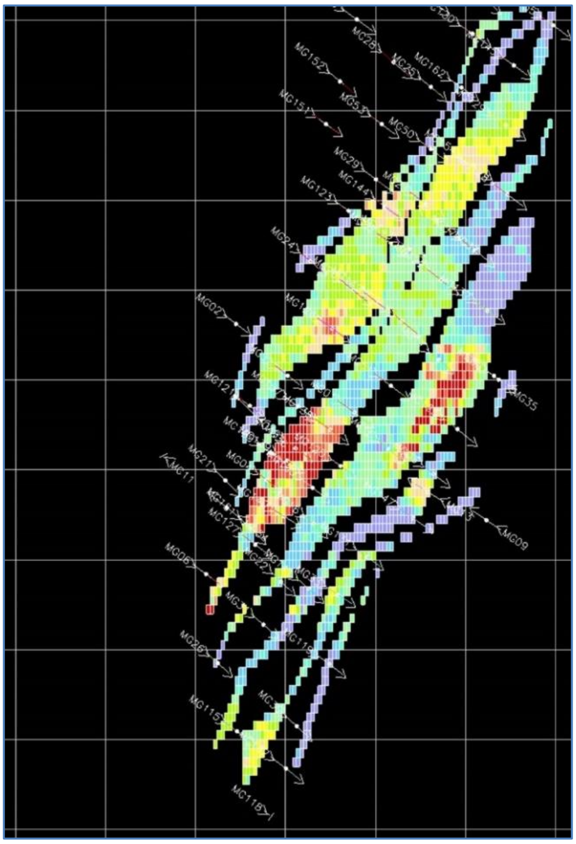
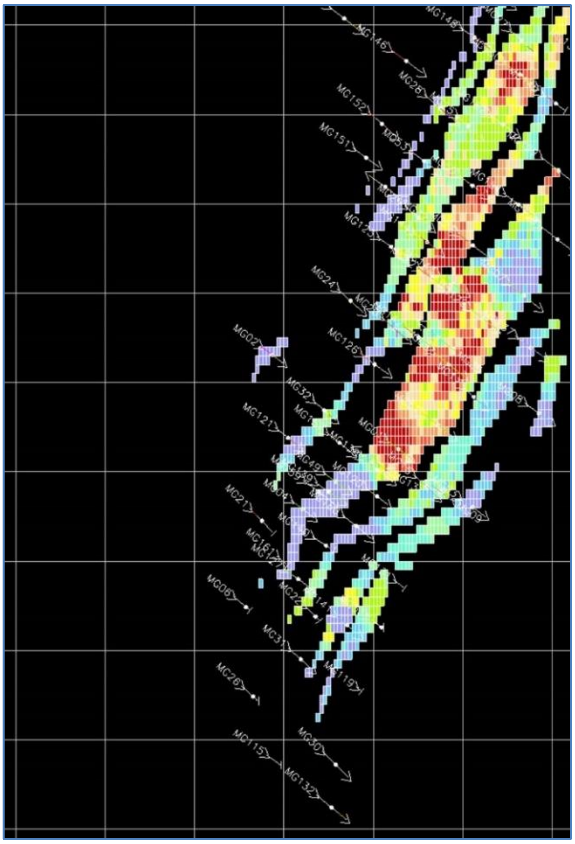


Figure 14.36 – Mungu gold block cross-section 5,367,802N



14.15 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 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**

This gold equivalent calculation was effectively the same as the 2017 Resource estimation – 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).

14.16 Bulk density

Bulk density was studied for the 2017 Resource estimation (11.4) 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/m3
- Transitional 2.59 t/m3
- Fresh 2.64 t/m3

14.17 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:

- 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.

14.17.1 Classification level and thinking

In the previous 2017 Resources a high 93% of estimated blocks by tonnage in the Pipe 1, 2 and 4 deposits were classified as Measured and Indicated, with Measured representing 64% and Indicated representing 36%. The 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 QP's view that the geological model (based as it was on grade shells) took comparatively little account of geology.

Notwithstanding the QP's slightly negative view of the past classification his considered opinion here is that the bulk of material should still be partly classified as Measured and partly as Indicated. Resources but with a slightly higher proportion (than in the past) of Indicated (Table 14.12) given the rigour required to meet the Measured status. Peripheral material (surrounding the other classes where drilling information declines) should be classified as Inferred Resources.

With regard to classification decisions he considers overall that:

1. The deposits are well, closely (~30 m), and fairly uniformly drilled.
2. In-fill drilling since the 2017 estimate has very largely confirmed the previous results, thus raising confidence.
3. Mineralised zones are very clearly continuous over multiple adjacent drill holes, thus giving confidence in the drill hole spacing and geological deposit interpretation.
4. 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.
5. 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.
6. 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.

14.17.2 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. The passes differed in estimation parameter, essentially increasing scan distances relating to different components of the geostatistical variograms.

For JORC classification here the 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.

14.17.3 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.

Table 14.9 - JORC Classification criteria

Resource class	**Average distance AU_D (m)**	**Number points AU_P (#)**	**Class AU_CAT**
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.

14.17.4 Block classification cross-sections

Pipes 1 and 4: Figure 14.37 to Figure 14.39 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 Figure 14.37 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 14.37 – Pipe 1 and 4 distance (D) block cross-section 5,367,000N

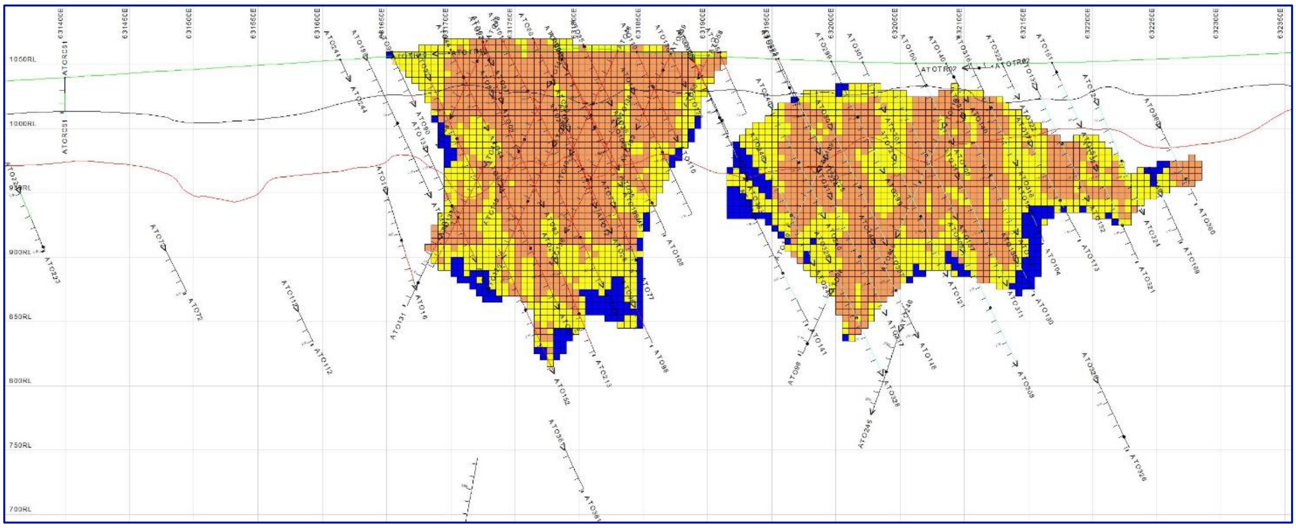


Figure 14.38 – 96 Pipe 1 and 4 points (P) block cross-section 5,367,000N

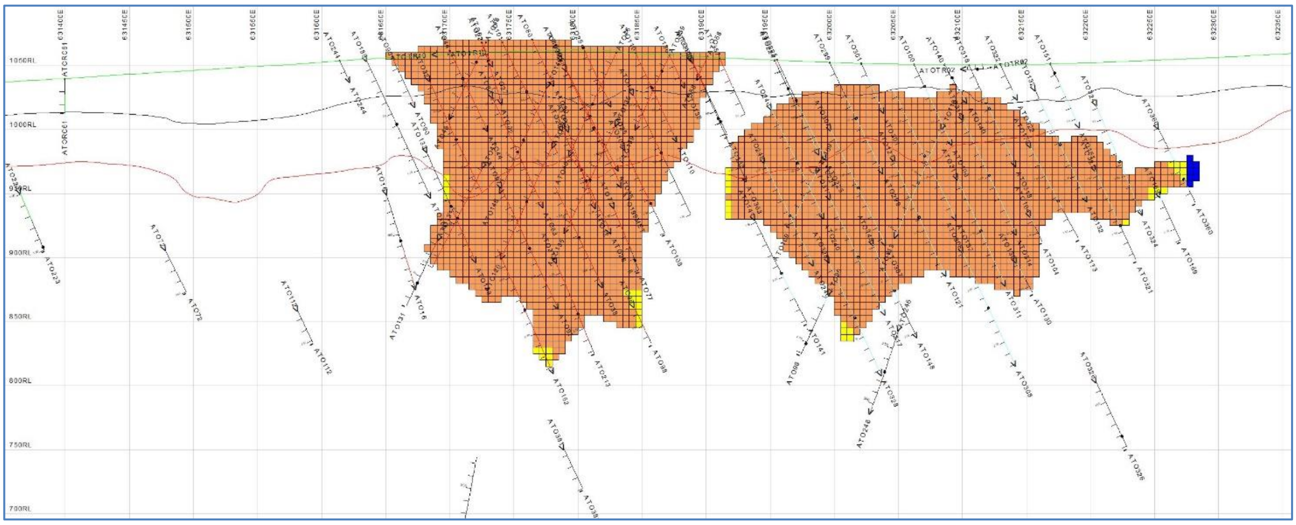
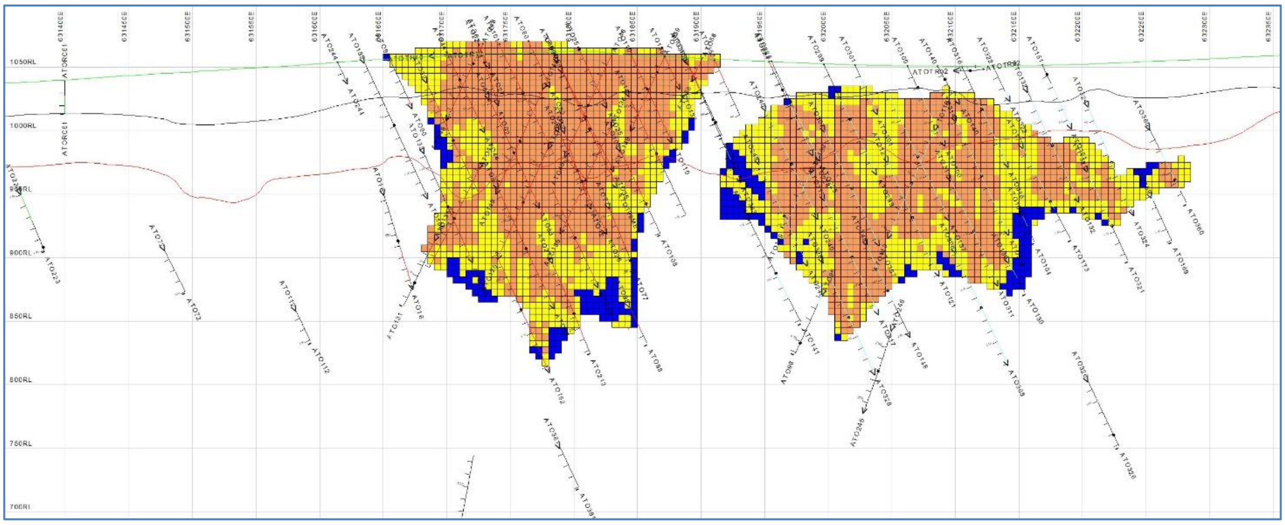


Figure 14.39 – Pipe 1 and 4 classification (CAT) block cross-section 5,367,000N



Mungu: Figure 14.40 to Figure 14.42 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 14.40 – Mungu distance (D) block cross-section 5,367,600N

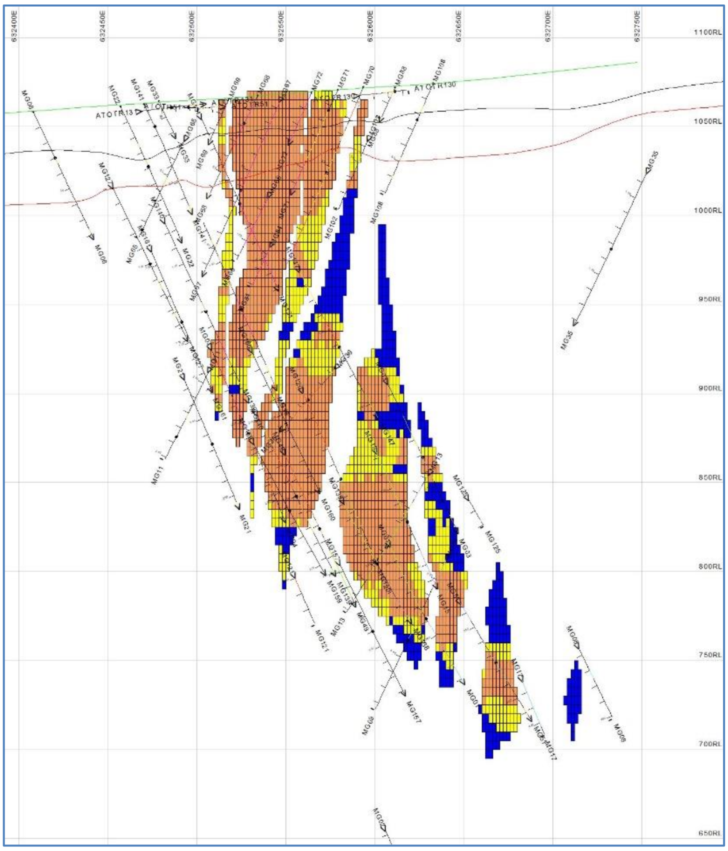


Figure 14.41 – Mungu points (P) block cross-section 5,367,600N

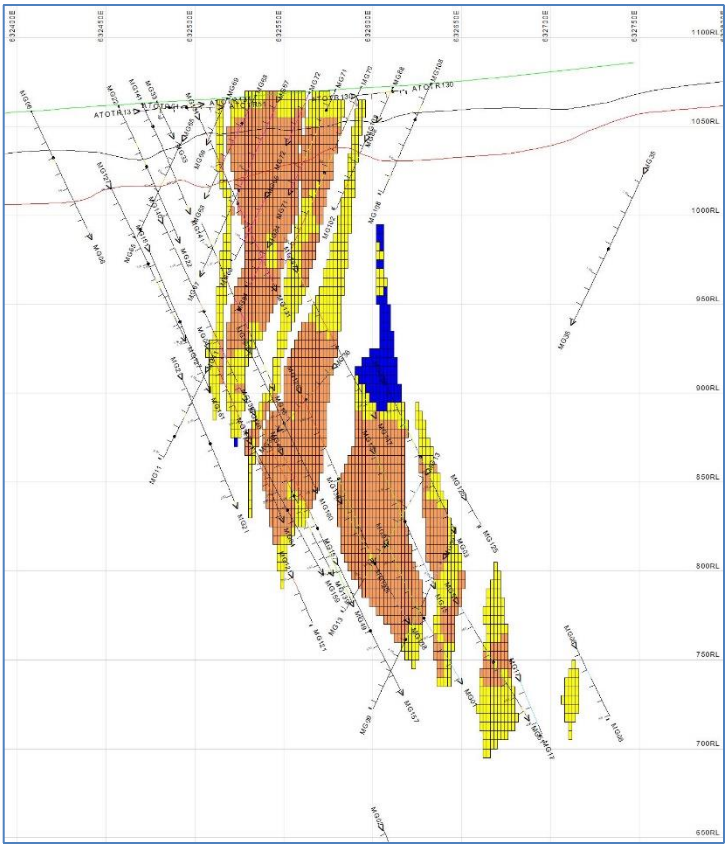
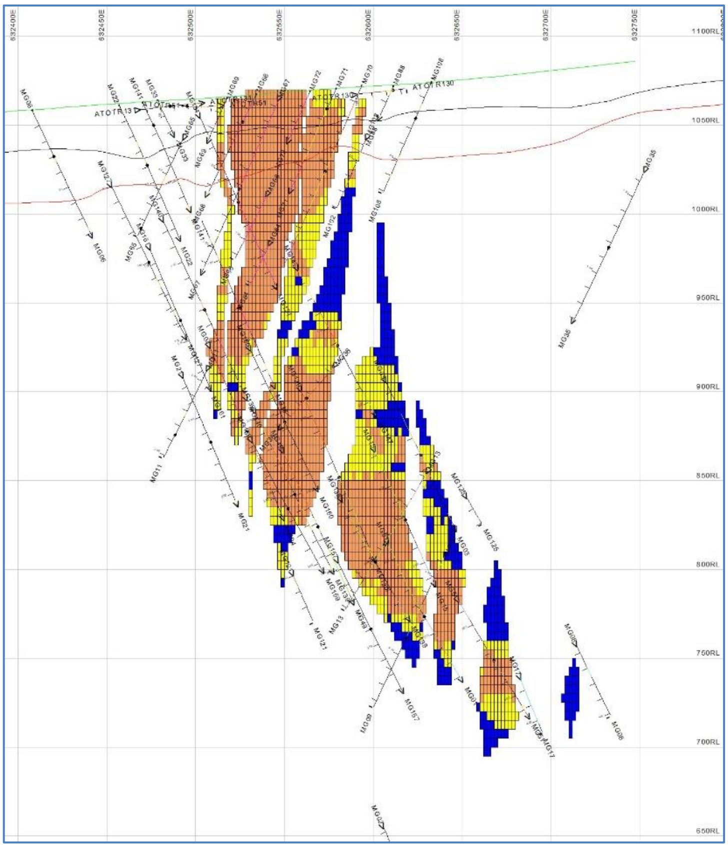


Figure 14.42 – Mungu classification (CAT) block cross-section 5,367,600N



14.18 ATO 2022 JORC Mineral Resources

The QP reports here as at 27 July 2022 the Minerals Resources estimated in late 2021 for the ATO Project. These 2022 Resources update those reported in February 2021 by using July 2022 up-dated oxidation surfaces (dividing oxidised, transitional and fresh rock) and a July 2022 topography surface accounting for open pit mining (in Pipes 1 and 40 to that data.

Requisite statements, certifications and declarations by the QP are made in Section 14.1 above to satisfy the reporting codes governing these Resources. This JORC classification is directly equivalent to CIM categorisation, as stated in the Terms of Reference Section.

The basis for the JORC classification here is given in Section 14.17. 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.

Cut-off grades: Lower cut-off grades to use in reporting were applied by oxidation level. Values used in the 2017 reporting[30] 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 in Section 11.4, 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 at 27th July 2022, in Table 14.. These Resources supersede any previously reported (in particular those reported immediately before at February 2021). Here the deposit names ATO1, ATO2 and ATO4 are interchangeable with names Pipe 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.

[30] 2017 NI 43-101. Section 14.9.4, pp145.

Table 14.10 - ATO July 2022 Measured & Indicated in-situ Mineral Resources - by oxidation level

ATO - Resources remaining at 27/7/2022													
RESOURCES BY OXIDATION LEVEL	Deposit	Cut-off AuEq (g/t)	Bulk density (t/m^3)	Tonnes (Mt)		Grades Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	AuEq (g/t)	Metal Au (k oz)	Ag (k oz)	AuEq (k oz)
MEASURED													
Oxide	ATO1	0.15	2.46	0.5		0.64	6.03	0.33	0.33	0.99	10	96	16
(20220727 down	ATO2	0.15	2.46	1.1		0.47	3.63	0.40	0.31	0.80	17	132	29
(to 2022_OX)	ATO4	0.15	2.46	0.1		0.72	25.15	0.10	0.20	1.20	1	50	2
	Mungu	0.15	2.46	0.2		0.81	30.08	0.00	0.00	1.23	6	230	9
	TOTAL	**0.15**	**2.46**	**1.9**	**9%**	**0.56**	**8.20**	**0.32**	**0.27**	**0.91**	**35**	**508**	**57**
Transition	ATO1	0.40	2.59	0.7		1.83	7.34	0.78	0.98	2.64	42	167	60
(between	ATO2	0.40	2.59	0.1		0.33	3.13	0.44	0.80	0.89	1	9	3
2022_OX	ATO4	0.40	2.59	0.1		0.95	24.28	0.15	0.27	1.47	0	49	3
and 2022_TR)	Mungu	0.40	2.59	0.0		0.72	33.40	0.00	0.01	1.19	1	46	2
	TOTAL	**0.40**	**2.59**	**0.9**	**4%**	**1.56**	**9.32**	**0.66**	**0.87**	**2.31**	**46**	**272**	**67**
Fresh	ATO1	0.40	2.64	6.7		1.02	5.69	0.72	1.38	1.97	221	1,230	426
(below 2022_TR)	ATO2	0.40	2.64	0.5		0.35	4.50	0.75	1.83	1.51	5	70	23
	ATO4	0.40	2.64	6.9		1.36	11.88	0.30	0.54	1.87	303	2,644	416
	Mungu	0.40	2.64	4.6		1.36	44.73	0.01	0.03	2.00	202	6,645	297
	TOTAL	**0.40**	**2.64**	**18.8**	**87%**	**1.21**	**17.56**	**0.39**	**0.75**	**1.93**	**731**	**10,589**	**1,163**
MEASURED	**TOTAL**			**21.6**	**57%**	**1.17**	**16.38**	**0.40**	**0.71**	**1.85**	**811**	**11,370**	**1,287**
INDICATED													
Oxide	ATO1	0.15	2.46	0.3		0.41	5.71	0.30	0.27	0.72	4	56	7
(20220727 down	ATO2	0.15	2.46	1.0		0.44	3.62	0.34	0.26	0.72	14	118	24
(to 2022_OX)	ATO4	0.15	2.46	0.1		0.46	20.01	0.07	0.19	0.85	2	93	4
	Mungu	0.15	2.46	0.1		0.65	22.16	0.00	0.00	0.97	2	77	3
	TOTAL	**0.15**	**2.46**	**1.6**	**10%**	**0.45**	**6.80**	**0.28**	**0.24**	**0.75**	**23**	**344**	**38**
Transition	ATO1	0.40	2.59	0.2		0.96	6.49	0.61	0.71	1.58	6	41	10
(between	ATO2	0.40	2.59	0.1		0.37	2.96	0.34	0.57	0.79	1	6	2
2022_OX	ATO4	0.40	2.59	0.1		0.61	19.94	0.08	0.33	1.07	2	63	3
and 2022_TR)	Mungu	0.40	2.59	0.0		0.57	20.60	0.00	0.01	0.86	0	9	0
	TOTAL	**0.40**	**2.59**	**0.4**	**2%**	**0.75**	**10.00**	**0.40**	**0.56**	**1.28**	**9**	**119**	**15**
Fresh	ATO1	0.40	2.64	4.2		0.76	4.91	0.67	1.34	1.67	103	666	226
(below 2022_TR)	ATO2	0.40	2.64	0.4		0.49	5.28	0.85	2.03	1.78	7	72	24
	ATO4	0.40	2.64	7.4		0.97	14.94	0.23	0.44	1.46	231	3,564	348
	Mungu	0.40	2.64	2.4		0.91	37.26	0.01	0.03	1.45	71	2,907	113
	TOTAL	**0.40**	**2.64**	**14.5**	**88%**	**0.88**	**15.48**	**0.34**	**0.68**	**1.53**	**412**	**7,209**	**712**
INDICATED	**TOTAL**			**16.4**	**43%**	**0.84**	**14.52**	**0.34**	**0.63**	**1.45**	**444**	**7,672**	**765**
MEASURED + INDICATED				**0.0**							**0**	**0**	**0**
Oxide	TOTAL	0.15	2.46	3.5	9%	0.51	7.57	0.31	0.26	0.84	58	852	95
Transition	TOTAL	0.40	2.59	1.3	3%	1.33	9.52	0.59	0.78	2.01	55	391	83
Fresh	TOTAL	0.40	2.64	33.2	87%	1.07	16.65	0.37	0.72	1.75	1,143	17,798	1,875
MEAS + IND	**TOTAL**			**38.0**		**1.03**	**15.58**	**0.37**	**0.68**	**1.68**	**1,255**	**19,042**	**2,052**

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 at February 2021, in Table 14..

Table 14.11 - ATO July 2022 Inferred in-situ Mineral Resources- by oxidation level

ATO - Resources remaining at 27/7/2022													
RESOURCES BY OXIDATION LEVEL	Deposit	Cut-off AuEq (g/t)	Bulk density (t/m^3)	Tonnes (Mt)		Grades Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	AuEq (g/t)	Metal Au (k oz)	Ag (k oz)	AuEq (k oz)
INFERRED													
Oxide	ATO1	0.15	2.46	0.1		0.30	5.36	0.23	0.21	0.55	1	13	1
(20220727 down	ATO2	0.15	2.46	0.3		0.28	3.30	0.21	0.23	0.51	2	27	4
(to 2022_OX)	ATO4	0.15	2.46	0.0		0.31	6.23	0.07	0.13	0.48	0	10	1
	Mungu	0.15	2.46	0.0		0.54	12.48	0.00	0.00	0.72	1	12	1
	TOTAL	**0.15**	**2.46**	**0.4**	**8%**	**0.31**	**4.73**	**0.18**	**0.20**	**0.53**	**4**	**61**	**7**
Transition	ATO1	0.40	2.59	0.0		0.58	6.28	0.33	0.51	1.02	0	3	0
(between	ATO2	0.40	2.59	0.0		0.45	3.63	0.38	0.56	0.88	0	2	0
2022_OX	ATO4	0.40	2.59	0.0		0.40	11.41	0.04	0.25	0.69	0	3	0
and 2022_TR)	Mungu	0.40	2.59	0.0		0.58	8.04	0.00	0.01	0.69	0	1	0
	TOTAL	**0.40**	**2.59**	**0.0**	**1%**	**0.50**	**6.73**	**0.24**	**0.41**	**0.86**	**1**	**9**	**1**
Fresh	ATO1	0.40	2.64	1.0		0.52	4.17	0.59	1.36	1.40	16	132	44
(below 2022_TR)	ATO2	0.40	2.64	0.2		0.27	9.08	1.37	2.87	2.19	2	57	14
	ATO4	0.40	2.64	2.1		0.60	15.34	0.19	0.36	1.04	40	1,030	70
	Mungu	0.40	2.64	1.7		0.84	25.70	0.01	0.02	1.21	45	1,365	64
	TOTAL	**0.40**	**2.64**	**4.9**	**92%**	**0.65**	**16.34**	**0.26**	**0.55**	**1.21**	**103**	**2,584**	**192**
INFERRED	**TOTAL**			**5.4**	**14%**	**0.62**	**15.39**	**0.25**	**0.52**	**1.16**	**108**	**2,655**	**200**

Reporting by class Table 14.12 reports similar Measured and Indicated Resources to those in Table 14. – except summarised by class.

Table 14.12 - ATO July 2022 Measured & Indicated in-situ Mineral Resources - by class

ATO - Resources remaining at 27/7/2022											
RESOURCES BY DEPOSIT	Deposit	Tonnes (Mt)		Grades Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	AuEq (g/t)	Metal Au (k oz)	Ag (k oz)	AuEq (k oz)
MEASURED	ATO1	7.9		1.07	5.86	0.70	1.28	1.97	272	1,493	502
	ATO2	1.7		0.43	3.85	0.50	0.77	1.01	24	212	55
	ATO4	7.0		1.35	12.11	0.30	0.53	1.86	304	2,743	422
	Mungu	4.9		1.33	43.92	0.01	0.03	1.96	209	6,922	308
	TOTAL	**21.6**	**57%**	**1.17**	**16.38**	**0.40**	**0.71**	**1.85**	**811**	**11,370**	**1,287**
INDICATED	ATO1	4.7		0.75	5.03	0.64	1.24	1.60	113	762	243
	ATO2	1.5		0.45	4.06	0.48	0.77	1.02	22	196	49
	ATO4	7.7		0.96	15.10	0.23	0.43	1.44	235	3,721	356
	Mungu	2.5		0.90	36.53	0.01	0.03	1.43	74	2,993	117
	TOTAL	**16.4**	**43%**	**0.84**	**14.52**	**0.34**	**0.63**	**1.45**	**444**	**7,672**	**765**
MEAS + IND	ATO1	12.6	33%	0.95	5.55	0.68	1.27	1.83	385	2,256	745
	ATO2	3.2	8%	0.44	3.95	0.49	0.77	1.01	45	408	105
	ATO4	14.7	39%	1.14	13.67	0.26	0.48	1.64	540	6,463	777
	Mungu	7.5	20%	1.18	41.39	0.01	0.03	1.77	283	9,915	425
MEAS+IND	**TOTAL**	**38.0**		**1.03**	**15.58**	**0.37**	**0.68**	**1.68**	**1,255**	**19,042**	**2,052**

Table 14. reports similar Inferred Resources to those in Table 14. – except summarised by class.

Table 14.13 - ATO July 2022 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**

ATO - Resources remaining at 27/7/2022											
RESOURCES BY DEPOSIT	Deposit	Tonnes		Grades					Metal		
				Au	Ag	Pb	Zn	AuEq	Au	Ag	AuEq
		(Mt)		(g/t)	(g/t)	(%)	(%)	(g/t)	(k oz)	(k oz)	(k oz)
INFERRED	ATO1	1.1	20%	0.51	4.28	0.56	1.27	1.34	17	147	46
	ATO2	0.5	9%	0.28	5.76	0.71	1.36	1.23	4	86	18
	ATO4	2.1	40%	0.59	15.12	0.19	0.35	1.03	41	1,043	71
	Mungu	1.7	31%	0.83	25.40	0.01	0.02	1.20	45	1,379	65
INFERRED	**TOTAL**	**5.4**		**0.62**	**15.39**	**0.25**	**0.52**	**1.16**	**108**	**2,655**	**200**

14.19 Reconciliation – of Resources with other estimates

A reconciliation was done between the immediately previous 2/2021 Resources estimated by the QP and those from 2017 estimated by DRA. Reconciliation could only be done for the three deposits 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[31]). 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. for 2017 reporting (light blue) and 2021 reporting (light green).

[31] 2017 NI 43-101. Section 14.9.3, Tables 14.13 and 14.14, pp143-144.

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%**

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; the 2021 estimate used detailed section-by-section multi-element mineralisation interpretation. The 2021 estimate better integrated 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.

14.20 Potential impact on Resources by other factors

The 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 QP's experience the cut-offs used here are comparatively low.
- Raising cut-offs would reduce the Resources. The 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 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 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 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.
- TheQP 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 QP does not consider that the current model is over optimistic in size.

JORC classification:

- The 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 QP's classification has the Measured and Indicated Resources as being a lower (albeit slightly) 88% of all blocks (compared to the 93%).
- And here the Measured is a lower 58% of the total Measured and Indicated Resources (compared to 64%).
- The QP regards the latest class proportions to be more realistic than the 2017 proportions.
- However the 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 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 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 QP does not know at what depth this consideration should apply from.

15 MINERAL RESERVE ESTIMATES

15.1 Introduction

This section of the report summarises the main considerations in relation to the preparation of the ATO Mineral Reserves update and provides references to the sections of the study where more detailed discussions of particular aspects are covered. Detailed technical information provided in this section relates specifically to this Mineral Reserve update and is based on the Mineral Resource models and estimates as reported in Item 14.

The Mineral Reserves update was compiled with reference to NI 43-101 by the Qualified Person responsible for the reporting of open pit Mineral Reserves, Mr. Grant Walker, who is a Charter Profession and Member of The Australasian Institute of Mining and Metallurgy (AusIMM), and an employee of Xenith. Mr. Walker has sufficient experience, which is relevant to the style of mineralisation and type of deposit under consideration and to the activity which he has undertaken to be a Qualified Person, as defined by NI 43-101. Mr Walker work has been reviewed by Mr Iain Ross who is a Member of The Australasian Institute of Mining and Metallurgy (AusIMM), and an employee of Xenith. Mr Ross has over 30 years of relevant experience in the style of mineralisation and type of deposit under consideration and to the activity which he has undertaken to be a Qualified Person, as defined by NI 43-101.

15.2 Mineral Reserve Pit Determination and Modifying Factors

The conversion of the Mineral Resource estimate to a Mineral Reserve estimate followed a conventional approach commencing with determination of the economic pit limits using the Deswik Pseudo Flow optimisation software ("Deswik Optimiser"). The word 'optimisation' is used as the pit shell generated defines economic mining limits for each mining area using the given input parameters. Deswik Optimiser was not used to "optimise" the overall ATO project development as practical elements such as mine sequence, blending strategy were excluded.

The approach used to identify the final economic pit limits for ATO was:

- Identify any physical constraints to mining, for example, tenement boundaries, infrastructure, protected zones (flora, rivers, roads and road easements);
- Define mining and processing costs as well as selling prices and costs;
- Define the mining loss and dilution;
- Define processing recoveries for Oxide and Fresh ore (includes beneficiation recoveries as well as leach recoveries);
- Define the pit slope design parameters for each mining area and material type;
- Import all above parameters, including geological model into the pit optimisation software;
- Run pit optimisation software to produce a series of nested pit shells at increasing product selling prices;
- Analyse results and select a preferred pit shell for each mining area for guidance in the pit designing process.

The selected ultimate pit outlines (shells) were used to create practical and detailed open pit designs accounting for the siting of in-pit ramps, berms, pumps, and haul roads. These pit designs then provided the bench by bench ore and waste mining inventories for the detailed production schedule that demonstrates viable open pit mining and provides the physical basis for cash flow modelling (refer Item 16).

15.3 Geological Block Models and Topography

As discussed in Item 14 of this report, the resource block models for ATO and Mungu were prepared by GeoRes.

The two models were imported into Deswik. Surveyed mining surfaces as at 27th August 2022 were then applied to the models to deplete mined material. The two block models were subsequently manipulated to add attributes required for optimisation.

15.4 Mineral Reserve Cut-off Grade

A ROM marginal break-even cut-off grade calculation based on the latest revenue assumptions, cost estimation and metallurgical recoveries determined the economic marginal cut-off of to be 0.40 g/t AuEq for Fresh material and 0.43 g/t AuEq for Oxide material. These cut-offs were applied for the estimation of Mineral Reserves.

15.5 Mineral Reserve Dilution and Ore Loss

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.

15.5.1 Geotechnical Parameters

The pit designs followed the recommended geotechnical slopes for the ATO and Mungu Pits, as described in Section 16.2.

15.6 Mining Operating Costs

The unit operating costs used in the pit optimisation process were derived based on the 2022 Budget. Variable mining costs comprising drill, blast, load and haul costs, on a bench by bench basis, were based on the current mining contract.

15.7 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 operating cost, product sales prices (for gold, silver, lead and zinc), and pit and plant operating parameters. The optimiser operates on a net value calculation for the blocks (i.e. revenue from product sales minus the operating 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 Inputs

Description	Unit	Oxide	Transition/Fresh
Ore Mining Cost	$/t ore	2.10	2.10
Waste Mining Cost	$/t waste	1.80	1.80
Milling Cost	$/t milled	6.59	14.20
G&A	$/t milled	7.11	2.10
Gold Recovery	%	70	80
Silver Recovery	%	40	85
Lead Recovery	%	0	88
Zinc Recovery	%	0	88
Gold sales price	$/oz	1700	1700
Silver sales Price	$/oz	22	22
Lead sales price	$/t metal	0	2,200
Zinc sales price	$/t metal	0	2,950
Gold refining charge	$/oz	2.00	2.00
Silver refining charge	$/oz	1.00	1.00
Lead refining charge	$/t metal	0	520
Zinc refining charge	$/t metal	0	525
Royalty	% sales	5	5

15.7.1 Optimisation Results and Shell Selection

Key criteria used in selecting the final pit shell included:

- Maximise cash flow;
- Maximise resource recovery;
- Balance between maximising ore and minimising incremental strip ratio; and
- Practical pit size and shape.

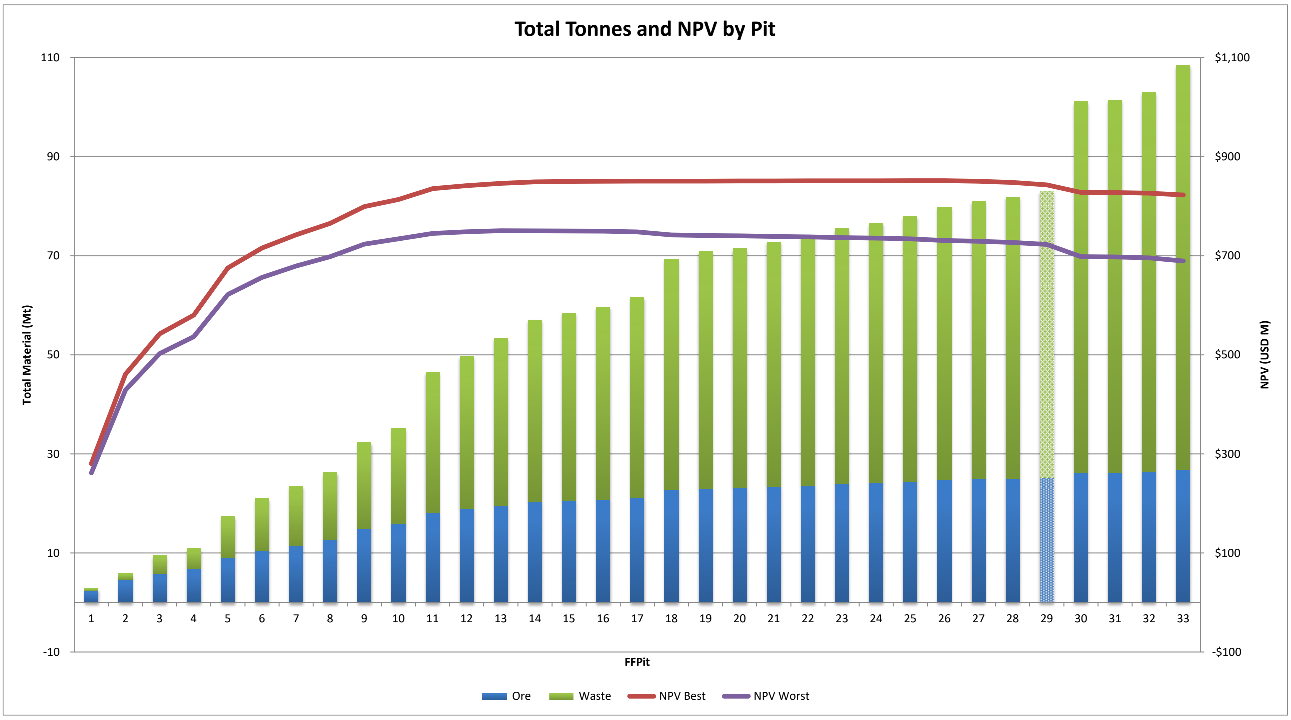
Table 15.2 and Figure 15.1 show the graphical results of pit optimisation at the ATO pits. When varying prices, the best and worst case NPVs increase gradually until it reaches Pits 13 and 14. From these points, the NPV decreases very slightly due to the costs associated with waste mining exceeding revenues. At pit 30 we see a sharp drop in NPV associated with an increase in waste tonnes with small increase in ore tonnes. On this basis, Pit 29 was chosen as the basis for the economic limit. This equates to the 85% revenue factor pit shell

Table 15.2 ATO Optimiser Results

Stage	NPV Best (UDS M)	NPV Worst (UDS M)	Waste (Mt)	Ore (Mt)	Grade AuEq (g/t)	Strip Ratio (t:t)
1	$280	$262	0	2	3.6	0.20
2	$461	$429	1	5	3.1	0.28
3	$542	$503	4	6	3.0	0.61
4	$580	$537	4	7	2.8	0.62
5	$675	$622	8	9	2.6	0.92
6	$715	$656	11	10	2.5	1.02
7	$742	$679	12	11	2.4	1.04
8	$766	$698	14	13	2.3	1.06
9	$799	$723	17	15	2.2	1.17
10	$814	$734	19	16	2.1	1.21
11	$835	$745	28	18	2.1	1.57
12	$842	$748	31	19	2.0	1.63
13	$846	$750	34	20	2.0	1.72
14	$849	$750	37	20	2.0	1.81
15	$850	$750	38	21	2.0	1.84
16	$850	$750	39	21	1.9	1.86
17	$850	$748	40	21	1.9	1.91
18	$850	$742	47	23	1.9	2.05
19	$850	$741	48	23	1.9	2.08
20	$850	$740	48	23	1.9	2.08
21	$850	$739	49	23	1.9	2.11
22	$850	$738	51	24	1.9	2.14
23	$850	$736	52	24	1.8	2.16
24	$850	$735	52	24	1.8	2.18
25	$850	$734	54	24	1.8	2.21
26	$850	$730	55	25	1.8	2.22
27	$850	$729	56	25	1.8	2.25
28	$850	$728	57	25	1.8	2.27
29	$850	$727	58	25	1.8	2.28
30	$828	$698	75	26	1.8	2.85
31	$828	$698	75	26	1.8	2.86
32	$826	$696	77	26	1.8	2.90

Stage	NPV Best (UDS M)	NPV Worst (UDS M)	Waste (Mt)	Ore (Mt)	Grade AuEq (g/t)	Strip Ratio (t:t)
33	$823	$689	82	27	1.8	3.04

Figure 15.1 – ATO Optimiser Results



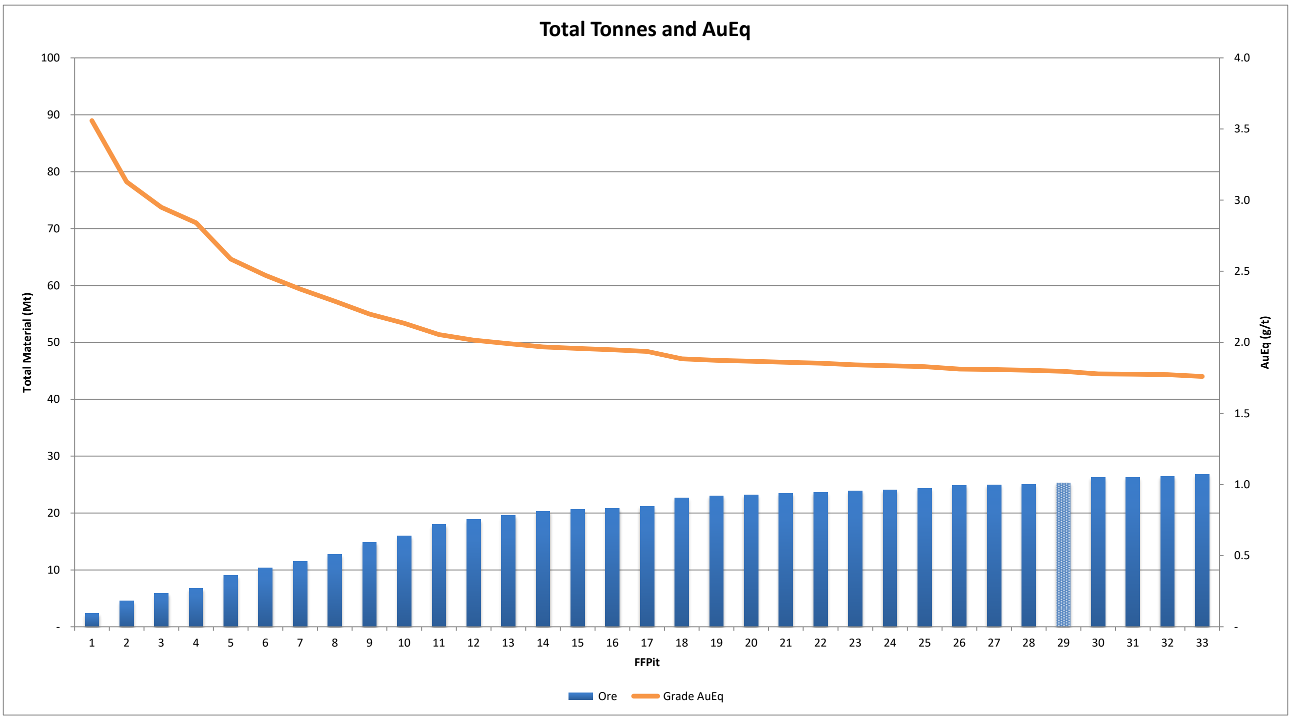


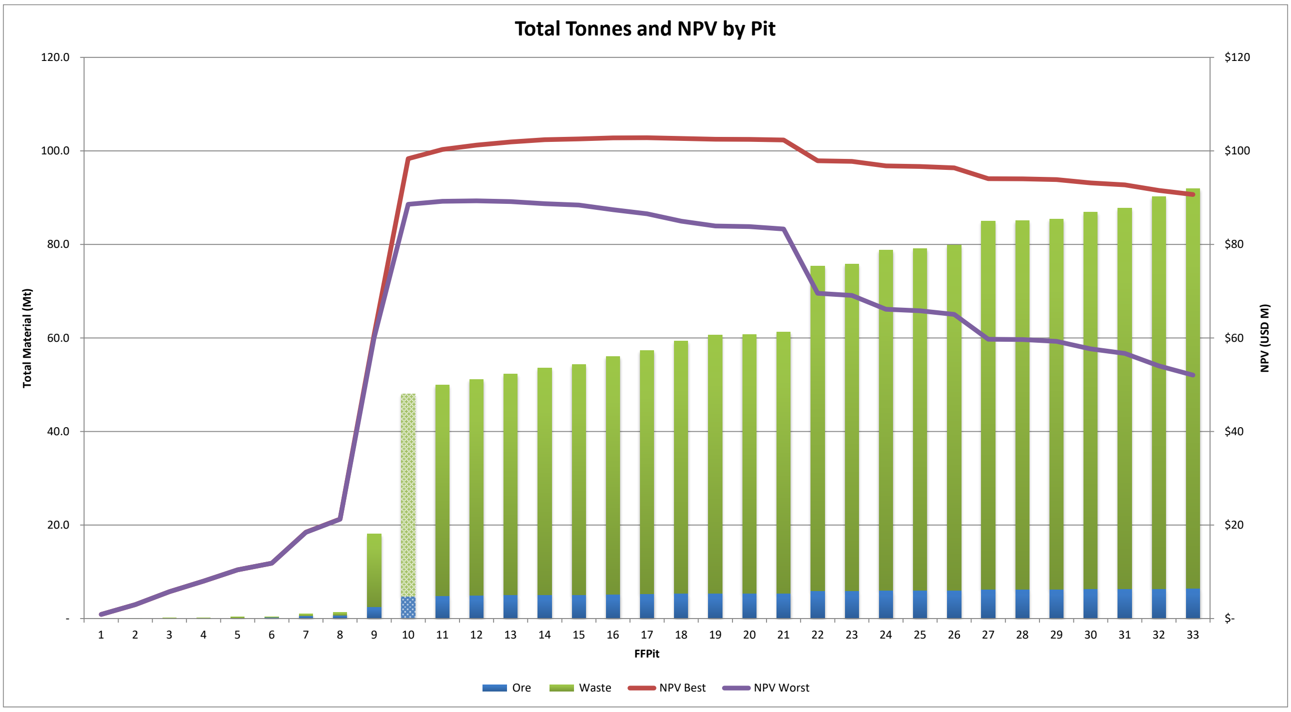
Table 15.3and Figure 15.2present the results of the optimisation at the Mungu pit. When varying gold price, the best case NPV increases gradually while the stripping ratio also steadily increases. Around pit 8 , and 10 the stripping ratio increases significantly, indicating a large quantity of waste is required to obtain relatively little ore. Thus, Pit 10 was chosen despite the jump there is not a significant quantity of ore in the lower pits. The quantity of economic open pit ore at Mungu is constrain by the structure of the ore body. Steppe Gold are carrying out additional drilling in the Mungu area followed by underground mining studies.

Table 15.3 Mungu Optimiser Results

Stage	NPV Best (UDS M)	NPV Worst (UDS M)	Waste (Mt)	Ore (Mt)	Grade AuEq (g/t)	Strip Ratio (t:t)
1	$1	$1	0.0	0.0	2.8	0.04
2	$3	$3	0.0	0.0	2.4	0.36
3	$6	$6	0.0	0.1	2.1	0.47
4	$8	$8	0.1	0.1	1.9	0.49
5	$10	$10	0.1	0.2	1.8	0.61
6	$12	$12	0.2	0.3	1.6	0.63
7	$18	$18	0.4	0.6	1.4	0.76
8	$21	$21	0.6	0.7	1.3	0.90
9	$61	$60	15.6	2.1	1.2	6.15
10	$98	$89	43.3	3.4	1.4	9.08
11	$100	$89	45.0	3.5	1.5	9.21
12	$101	$89	46.2	3.6	1.5	9.30
13	$102	$89	47.3	3.7	1.5	9.40
14	$102	$89	48.5	3.7	1.5	9.52
15	$103	$88	49.2	3.8	1.5	9.58
16	$103	$87	50.8	3.9	1.5	9.73
17	$103	$87	52.0	3.9	1.5	9.86
18	$103	$85	54.0	4.0	1.5	10.07
19	$103	$84	55.2	4.1	1.5	10.18
20	$102	$84	55.3	4.1	1.5	10.19
21	$102	$83	55.9	4.1	1.5	10.24
22	$98	$70	69.5	4.5	1.6	11.79
23	$98	$69	69.9	4.5	1.6	11.83
24	$97	$66	72.8	4.6	1.6	12.12
25	$97	$66	73.1	4.7	1.6	12.14
26	$96	$65	73.8	4.7	1.6	12.20
27	$94	$60	78.7	4.9	1.6	12.56

Stage	NPV Best (UDS M)	NPV Worst (UDS M)	Waste (Mt)	Ore (Mt)	Grade AuEq (g/t)	Strip Ratio (t:t)
28	$94	$60	78.8	4.9	1.6	12.56
29	$94	$59	79.1	4.9	1.6	12.58
30	$93	$58	80.6	5.0	1.6	12.73
31	$93	$57	81.4	5.0	1.6	12.81
32	$92	$54	83.8	5.1	1.6	13.04
33	$91	$52	85.5	5	1.6	13.22

Figure 15.2 – Mungu Optimiser Results



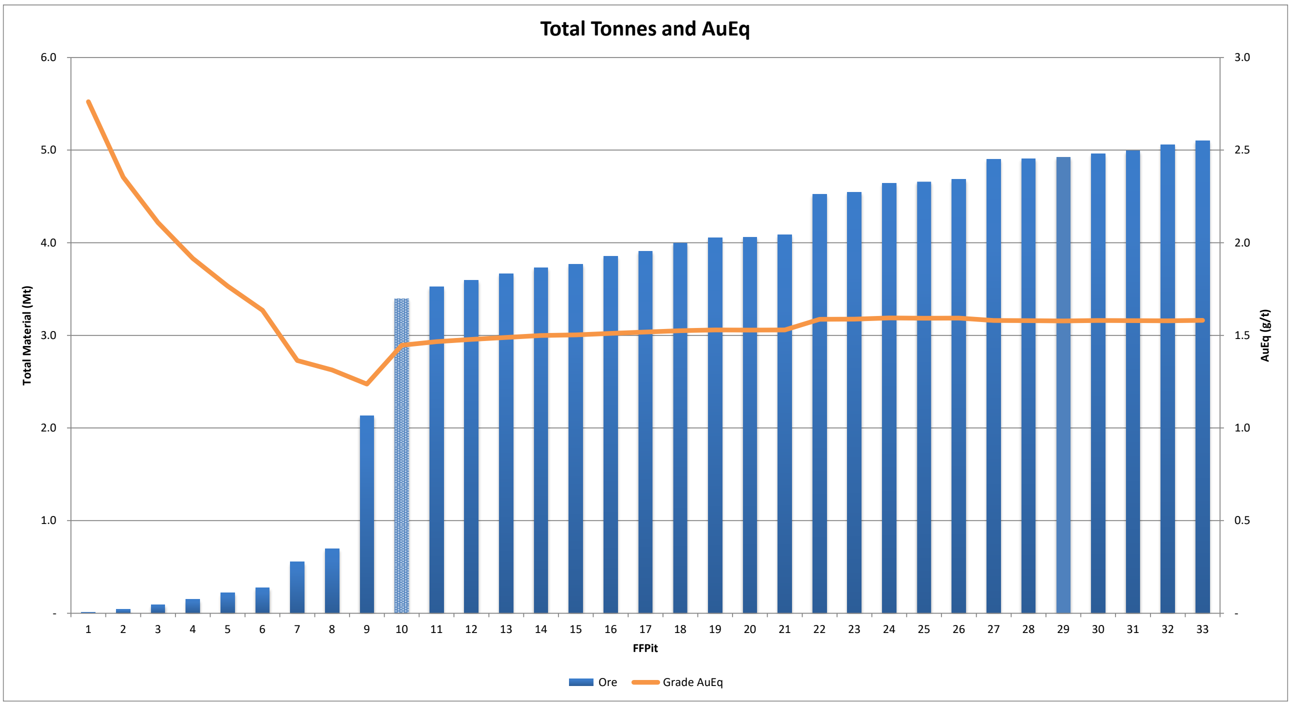


Table 15.4 Selected Optimal Shell Summary

Pit	Waste (Mt)	Ore (Mt)	Strip Ratio (t:t)	AuEq (g/t)
ATO	56.8	25.4	2.24	1.54
Mungu	43.3	3.4	9.1	1.30
Total	100	28.8	3.5	1.50

15.8 Design and Planning Parameters

The mine design work is completed using Deswik.CAD. The Mineral Reserve Life of Mine (LoM) scheduling is completed through Precision Mining Spry Software.

The Mineral Reserve is estimated from mine designs applied to 2022 Mineral Resource block models, which have been depleted for the production through to 27 August 2022.

The mine design methodology considers the Mineral Reserve cut-off grade, mining feasibility and economic assessment of mining blocks, and comprises the following general methodology:

- Determination of the mining method applied to individual areas, based on access options, geological grade distribution, geometry of the lode, historic mining shapes and geotechnical constraints,
- Determination of the mining dilution and recovery factors to apply to design,
- Interrogation of the practical pit against 3D geological block models in Deswik to calculate and assign ore tonnes and grade,
- Mining quantities of Measured and Indicated material above the cut-off grade are identified for further design and assessment,

- Assessment and design of the waste development required to uncover the ore,
- Economic assessment of pit development, based on variable mining costs applicable to the mining method and is inclusive of waste mining, haulage, processing, selling, royalty, and administrative costs,
- Economically viable areas are included in the Mineral Reserve LoM schedule.
- Uneconomic areas are removed or may be re-designed and included in the plan if reassessment proves to be profitable,
- Dependency rules, mining rates and schedule constraints are applied to the to link the mining activities in a logical manner within the Spry scheduling software.
- The resulting Reserve LoM schedule is exported for further economic validation through the financial model.

15.9 Pit Designs

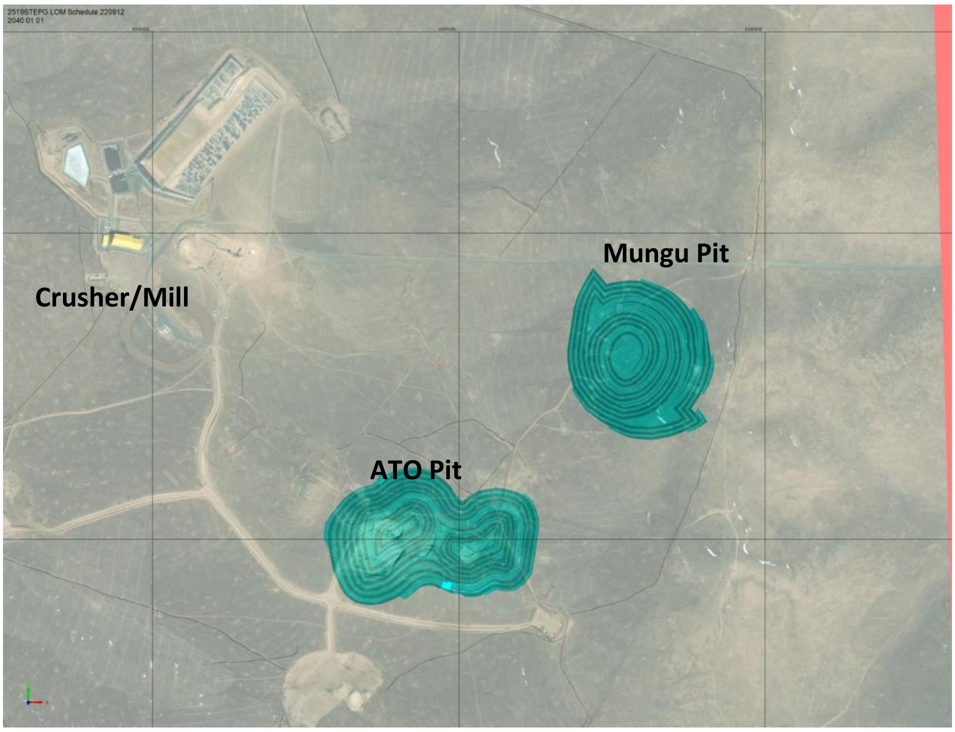
As noted in item 15.2 above, the ultimate pit designs for each mining area are based on the selected ultimate pit shells. These designs were developed in accordance with the design and planning parameters listed in item 15.7.

The following design parameters relate to the design of the two ultimate pits:-

- Benches (interval between berms) are mined to a height of 5 m in 2 x 2.5 m flitches in ore and waste
- Truck ramp and road width including bund = 30 m
- Maximum haul ramp gradient = 1 : 10 (approximately 6°)

Figure 15.3 show the ultimate pit designs produced from respective pit shell outlines.

Figure 15.3 – Pit Shells



15.10 Design Efficiency

Table 15.5provides a validation comparison between the ultimate pit designs and the shells upon which those designs were based.

Table 15.5 Validation between optimiser and pit design

Pit	Waste (Mt)	Ore (Mt)	Strip Ratio (t:t)	AuEq (g/t)
ATO Optimiser	56.8	25.4	2.24	1.54
ATO Design	60.0	25.9	2.25	1.51
Variance	-1.4%	-0.52%	-0.01	2%
Variance %	2%	2%	0.5%	2%
Mungu Optimiser	43.3	3.4	9.1	1.30
Mungu Design	43.5	3.2	13.7	1.25
Variance	0.2	6.0%	4.0	0.05
Variance %	-0.5%	6.0%	14%	4%

ATO optimised pit verse design pit correlate very well. Mungu has a moderate variation as a percentage. This was due to the tightness of the pit in regard to access.

15.11 Mineral Reserve Statement

Mineral Resources are reported inclusive of Mineral Reserves, (that is, Mineral Reserves are not additional to Mineral Resources). Mineral Reserves are subdivided into Proven Mineral Reserves and Probable Mineral Reserves categories to reflect the confidence in the underlying Mineral Resource data and modifying factors applied during mine planning. A Proven Mineral Reserve can only be derived from a Measured Mineral Resource while a Probable Mineral Reserve is typically derived from an Indicated Mineral Resource but can also be made up of a Measured Mineral Resource should the Qualified Person have reason to downgrade the confidence of the estimation.

The mineral reserves estimate with an effective date of August 30, 2022 for the Project is based on the parameters and steps outlined within this report as well as the resource estimate. The mineral reserves for the ATO gold deposit contains combined proven and probable mineral reserves totaling 29.1 million tonnes ("Mt") at 1.13 g/t gold and 12.43 g/t silver, containing 1.1 million ounces of gold and 11.7 million ounces of silver. The reserves have been classified as approximately 59% proven and 41% probable on a tonnage basis. The mineral reserve within the 2022 reserve pit shell was based on a AuEq cut-off grade of 0.43 g/t AuEq for Fresh material and 0.40 g/t AuEq for Oxide material and revenue of $1,700 per ounce gold, $20 per ounce of silver, zinc price of $2,500/t and lead price of $1,970/t. as the price assumptions. To access the ore, a total of 104 Mt of waste rock will need to be extracted at an average stripping ratio of 3.6.

The Mineral Reserves for ATO has been updated as at 27August 2022 and summarised in Table 15.6

Table 15.6 - Mineral Reserves as at August 27 2022

		Ore	Grade					Attributable Metal		
		kt	Au (g/t)	Ag (g/t)	Pb (%)	Zn (%)	AuEq (g/t)	Au (k oz)	Ag (k oz)	AuEq (k oz)
ATO										
Proven										
	Oxide	934	1.14	0.70	5.57	0.46	0.34	21	168	34
	Transition	361	1.57	0.72	10.32	0.41	0.70	8	120	18
	Fresh	13,535	2.10	1.37	8.59	0.49	0.88	597	3,749	917
	Total	**14,830**	**2.03**	**1.31**	**8.44**	**0.48**	**0.84**	**627**	**4,036**	**970**
Probable										
	Oxide	850	0.92	0.55	5.91	0.35	0.25	15	162	25
	Transition	372	1.47	0.70	11.35	0.27	0.48	8	136	18
	Fresh	9,922	1.69	1.09	11.26	0.36	0.68	350	3,603	541
	Total	**11,145**	**1.62**	**1.04**	**10.86**	**0.36**	**0.64**	**373**	**3,901**	**584**
Proven & Probable										
	Oxide	1,785	1.04	0.63	5.73	0.41	0.30	36	330	60
	Transition	733	1.52	0.71	10.84	0.34	0.59	17	256	36
	Fresh	23,457	1.93	1.25	9.72	0.43	0.79	947	7,352	1,458
	Total	**25,975**	**1.85**	**1.19**	**9.48**	**0.43**	**0.75**	**1,000**	**7,938**	**1,554**
Mungu										
Proved										
	Oxide	224	1.13	0.71	25.92	0.38	0.42	5	187	8
	Transition	-	-	-	-	-	-	-	-	-
	Fresh	2,193	1.27	0.64	39.67	0.08	0.09	45	2,805	90
	Total	2,417	1.26	0.65	38.39	0.11	0.12	51	2,993	98
Probable										
	Oxide	54	0.92	0.61	19.23	1.59	1.75	1	34	2
	Transition	-	-	-	-	-	-	-	-	-
	Fresh	684	1.02	0.51	32.33	0.25	0.28	11	713	22
	Total	**738**	**1.01**	**0.52**	**31.37**	**0.35**	**0.39**	**12**	**747**	**24**
Proven & Probable										
	Oxide	278	1.09	0.69	24.62	0.62	0.68	6	221	10
	Transition	-	-	-	-	-	-	-	-	-

		Ore	Grade					Attributable Metal		
	Fresh	2,877	1.21	0.61	37.93	0.12	0.13	57	3,518	113
	Total	**3,156**	**1.20**	**0.62**	**36.75**	**0.16**	**0.18**	**63**	**3,739**	**122**
Combine ATO & Mungu										
Proven										
	Oxide	1,159	1.14	0.70	9.50	0.44	0.36	26	355	43
	Transition	361	1.57	0.72	10.32	0.41	0.70	8	120	18
	Fresh	15,728	1.99	1.27	12.92	0.43	0.77	643	6,554	1,007
	Total	**17,247**	**1.92**	**1.22**	**12.64**	**0.43**	**0.74**	**677**	**7,029**	**1,068**
Probable										
	Oxide	905	0.92	0.56	6.71	0.43	0.34	16	196	27
	Transition	372	1.47	0.70	11.35	0.27	0.48	8	136	18
	Fresh	10,606	1.65	1.06	12.62	0.35	0.65	361	4,316	563
	Total	**11,883**	**1.59**	**1.01**	**12.13**	**0.36**	**0.62**	**385**	**4,648**	**608**
Proven & Probable										
	Oxide	2,063	1.04	0.64	8.28	0.44	0.35	42	551	69
	Transition	733	1.52	0.71	10.84	0.34	0.59	17	256	36
	Fresh	26,334	1.85	1.18	12.80	0.40	0.72	1,004	10,870	1,571
	Total	29,130	1.78	1.13	12.43	0.40	0.69	1,063	11,677	1,676

Notes

1. Mineral Reserves estimate was based on Measured and Indicated Resource Estimate by R. Rankin, QP and effective August 27 2022.
2. ATO and Mungu Mineral Reserves are effective as of August 27, 2022.
3. Mineral Reserves are included in Mineral Resources.
4. Mineral Reserves are reported in accordance with JORC and CIM and NI 43-101 guidelines.
5. Ore dilution is estimated at 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 0.40 g/t AuEq for oxide material and 0.43 g/t AuEq for transition and fresh material.
8. Mineral Reserves are contained within an optimised pit shell based on a gold price of $1,700 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: $1,700/oz gold, $20/oz silver, $1,970/t lead, $2,500/t zinc.
11. Totals may not match due to rounding.
12. The Mineral Reserves are stated as dry tonnes processed at the crusher.

15.12 Mineral Reserve Comparison

Reflecting the updates to the Mineral Resource model and the comparisons made between the depleted June 2021 and August 2022 Mineral Resource estimates (Item 14.14 and Table 14-19), there are commensurate differences between the respective Mineral Reserve estimates for each pit. The 2021 Technical Report contained a Mineral Reserve estimate of 26.4 million tonnes at a grade of 1.86 g/t AuEq. These Mineral Reserves were estimated using a 0.45 g/t AuEq cutoff grade based on a US$1,610 /Oz gold price; US$21 /Oz Silver price; US$1,970 / t metal lead price and US$2,51521 /t metal Zinc price. Since June

2021 the reported Mineral Reserve has been depleted approximately 1.4 Mt by mining and processing at ATO.

Overall, there has been a increase in tonnes of 9% and a decrease in grade of -1%. A summary of the reasons behind the differences for each mining area is as follows:

- Changes in the geological model. The Fresh layer has been remodelled leading to an increase in tonnes of 250 kt. The increase is related to the density difference between Transition and Fresh material.
- Changes in mine design. With the updated revenue assumptions the quantity of economic ore has increased by 1,807 kt in ATO and 2,069 kt at Mungu.
- Depletion from mining and processing.

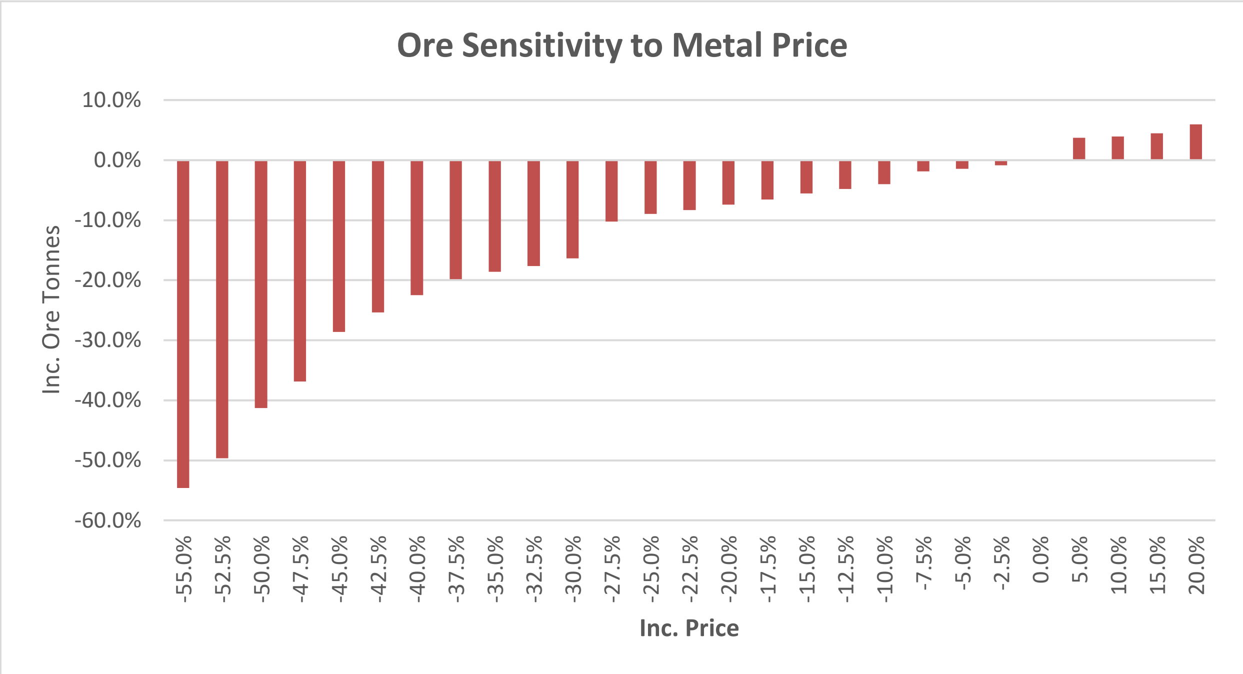
15.13 Mineral Reserve Sensitivity

The sensitivity of the ATO Mineral Reserve has been reviewed at a high level using the Deswik software. As in any mining operation, the results of the sensitivity analysis represent forward looking information that is subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. Areas of uncertainty that may materially impact the ATO Mineral Reserve estimation include:

- Commodity price,
- Capital and operating cost estimates, and
- Ore recoveries.

A high level sensitivity analysis of the deposit to a range of metal prices is presented graphically in Figure 15.2.

Figure 15.4 – Price Sensitivity



16 MINING METHODS

16.1 Mining Operation

The mining method at ATO 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 the current mine plan; however, the Mungu deposit has additional Resources at depth which are out side the current mine plan. Therefore, there is an opportunity for the Mungu Pit to be expanded to an underground mine once the open pit reserves are depleted.

All mining activities to date have occurred in the initial three ATO pits. Mining has been focused on extracting the oxide ore in these areas. No mining has commenced at Mungu.

Steppe Gold has contracted 2, mining contracting companies who are based in Mongolia, to undertake ore and waste mining at ATO under a three year, bcm rate based contract which includes drill and blast activities.

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.

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).

Figure 16.1 – Site Layout

16.1.1 Ore Control

Conventional open pit grade control practices have been put into place, incorporating RC drilling and sampling on a suitably designed drilling pattern to cover multiple bench horizons. Multi element sample assaying is being carried out on each sample. A grade control process has been standardised and implemented as the basis for designing dig blocks. The procedure ensures up to date estimate results are used together with the defined standards for material types as well as the bench and flitch specifications as aligned to the Mineral Reserve process

16.1.2 Drill and Blast

Some near-surface overburden material can be mined as free-dig however most bench development requires blasting in order to achieve excavator productivity targets. Production drilling and blasting is carried out on 6 metre benches using a small range of drilling/charging patterns and relatively low powder factors. Controlled blasting is undertaken on interim and final walls to prevent blast damage and to maintain wall control.

16.2 Geotechnical

The pit slope parameters for ATO and Mungo are based 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 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, Steppe Gold 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 Steppe 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 16.1 Pit Slope Parameters

Domain	Bench Angle (°)	Bench Height (m)	Berm Width (m)	Overall Slope (°)
NE Slope	65	10	6.8	50
SE Slope	65	10	4.7	52
SW Slope	65	10	10.5	42
NW Slope	65	10	4.7	52
Mungu	65	10	7.5	45

The current pit slope parameters do not vary by material type. Steppe Gold have recently carried out geotechnical drilling and are in the process or assessing pit slope parameters.

16.3 Pit and Dump Design

The pit design parameters including berm widths, batter angles, berm spacing and haul road gradients and widths are detailed in Section 15. For ease of rehabilitation, the external waste dumps were designed to have an overall outer slope gradient of 1:4.5. The geotechnical parameters are discussed in Items 15.4.3 and 16.5. The mine design parameters are shown in Table 16.1.

16.3.1 Pit Quantities

The ATO and Mungu final pit designs were previously presented in Figures 15.5 and 15.6, respectively.

16.4 Dump Design

Waste material mined from each of the pits will be placed in two expit waste dumps. The ATO waste dump is located East of the ATO Pit and has a capacity of 31.8 Mlcm.and a maximum height of 50 m. The Mungu waste dump is located West of the Mungu Pit and has a capacity of 2.6 Mlcm.and a maximum height of 33 m.

The Dumps were designed using the following parameters:

- 5 m high lifts;
- 70° lift face angles;
- 10 m berms at every lift;
- 45° overall slope;
- 30% swell factor from bank in-situ lcm

16.5 Production Schedule

With the completion of the detailed ultimate pit designs, detailed life-of-mine (LOM) production scheduling was completed using Spry software. Scheduling assumptions included:

- minimum mining block size (x, y, z) for all pits = 50 m x 50 m x 3 m ;
- mining flitch height = 3 m for monthly and quarterly periods;
- 6 m for annual periods; •
- maximum vertical advance rate = 54 m/year; •
- terrace mining with horizontal lag distance of 100 m to 200 m.

Features of the LOM mining and production schedule as displayed in Table 16-2 are as follows:

- Reported periods are calendar years
- Year 2022 includes Sept, Oct, Nov, Dec only.
- Mining is currently targeting the oxide material in the ATO 1, 2 and 4 pits.
- Production increases in 2024 with the targeting of Fresh waste and the commencement of ATO cutback
- Mining at Mungu is scheduled to commence in 2026,
- The total material to be mined from all pits from January 2022 onwards, amounts to 173 Mt, of which 29Mt is ore, and 144 Mt is Waste.
- The overall life of mine strip ratio (tonnes) is 5.0 to 1.
- The Project life is 14 years to 2036

Figure 16-2 to 16-9 depict the LOM schedule graphical results and end of period surfaces.

Figure 16.2 – Material Mined by Type

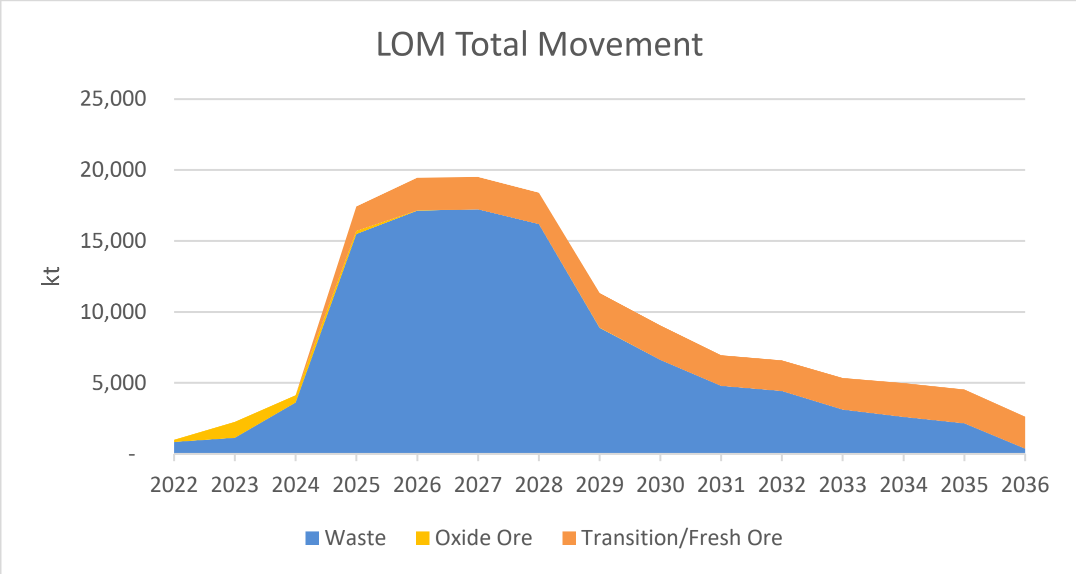


Figure 16.3 – Ore Mined By Location

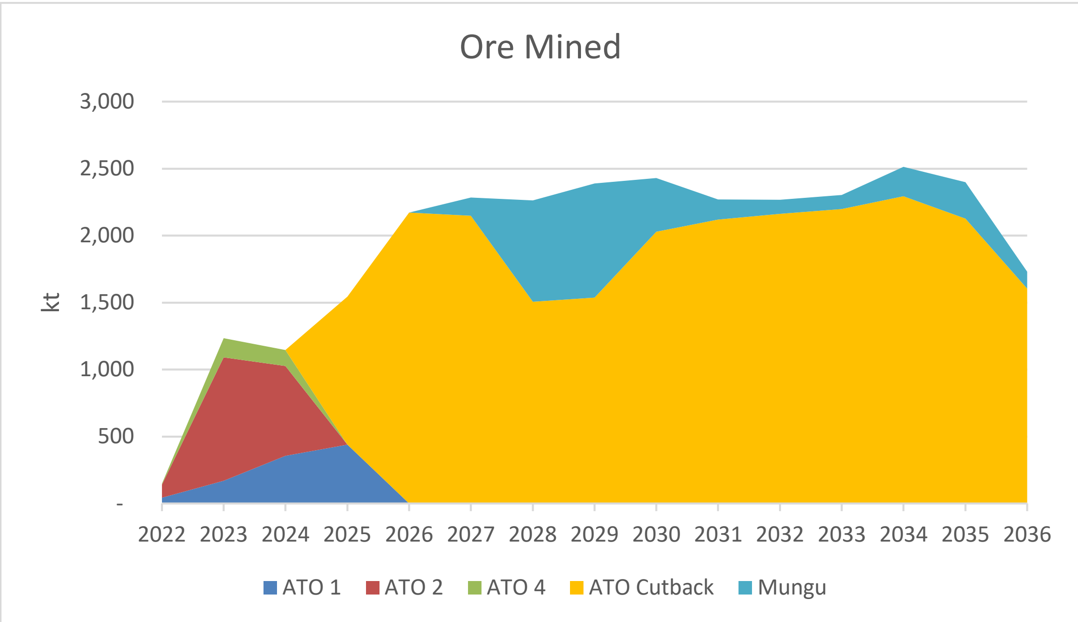


Table 16.2 Mine Production Schedule

			2022*	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	Total/Av
Waste																		
		kt	831	1,127	3,609	15,486	17,133	17,217	16,189	8,862	6,607	4,769	4,416	3,111	2,583	2,126	341	**104,407**
Ore																		
	Oxide	kt	148	1,126	518	205	20	-	-	17	-	-	-	-	-	-	-	**2,033**
	Fresh	kt	-	-	-	1,729	2,305	2,286	2,205	2,459	2,442	2,180	2,176	2,231	2,405	2,414	2,266	**27,098**
	Total	kt	**148**	**1,126**	**518**	**1,934**	**2,325**	**2,286**	**2,205**	**2,476**	**2,442**	**2,180**	**2,176**	**2,231**	**2,405**	**2,414**	**2,266**	**29,130**
Total Material		t: t	979	2,253	4,126	17,420	19,458	19,503	18,394	11,337	9,049	6,950	6,592	5,342	4,988	4,539	2,607	**133,537**
Strip Ratio		kt	5.6	1.0	7.0	8.0	7.4	7.5	7.3	3.6	2.7	2.2	2.0	1.4	1.1	0.9	0.2	**3.6**
Grade																		
	AuEq	g/t	0.53	0.51	0.49	2.17	2.74	2.71	2.08	1.71	1.54	1.47	1.68	1.53	2.08	1.34	1.36	**1.78**
	Au	g/t	0.83	0.62	0.62	1.00	1.41	1.55	1.40	1.01	1.13	0.86	1.11	1.03	1.12	1.14	1.17	**1.13**
	Ag	g/t	12.84	10.24	6.18	17.32	16.90	15.19	13.44	11.05	12.70	14.94	8.85	5.18	14.24	13.33	8.82	**12.43**
	Zn	%	-	-	-	0.45%	0.46%	0.32%	0.27%	0.40%	0.31%	0.40%	0.44%	0.39%	0.53%	0.54%	0.65%	**0.40%**
	Pb	%	-	-	-	0.22%	0.52%	0.61%	0.61%	0.84%	0.65%	0.71%	0.77%	0.77%	0.94%	0.95%	1.08%	**0.69%**

* Includes Sep, Oct, Nov, Dec

16.5.1 Yearly Sequence

Figure 16.4 – Mining Position end of 2022

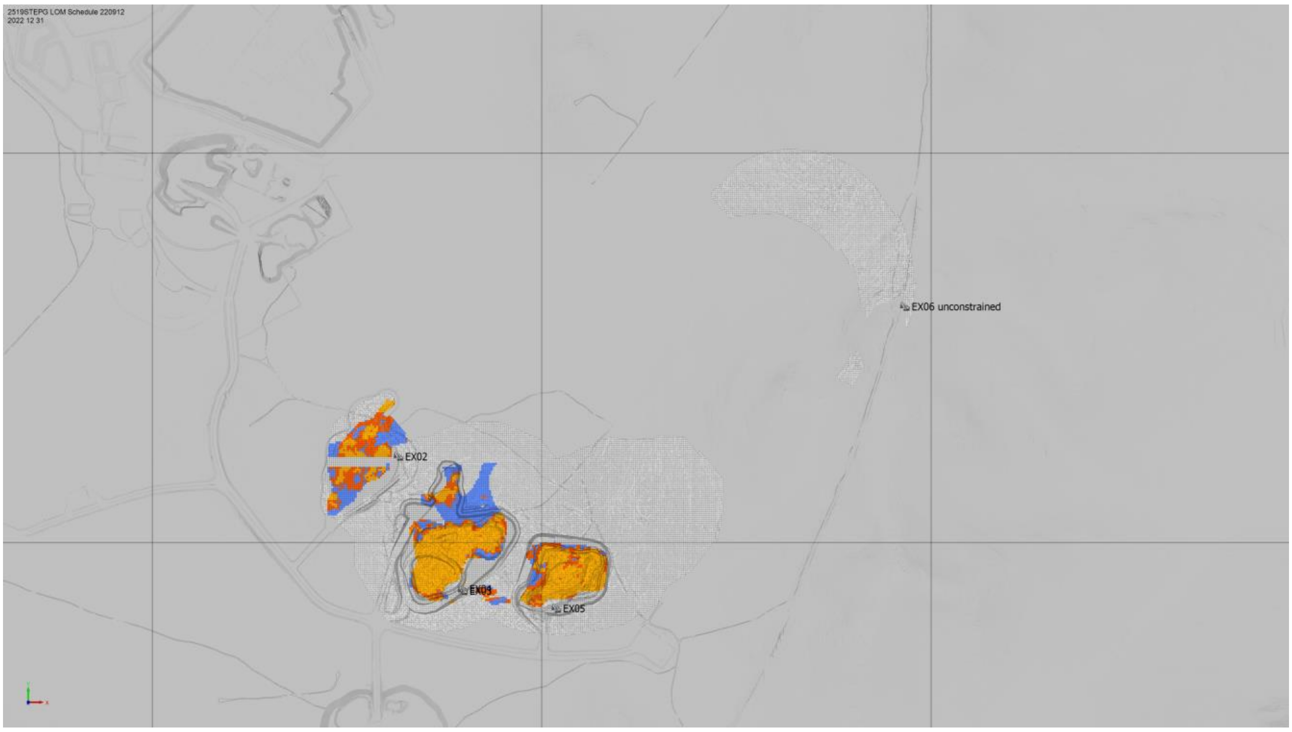


Figure 16.5 – Mining Position end of 2024

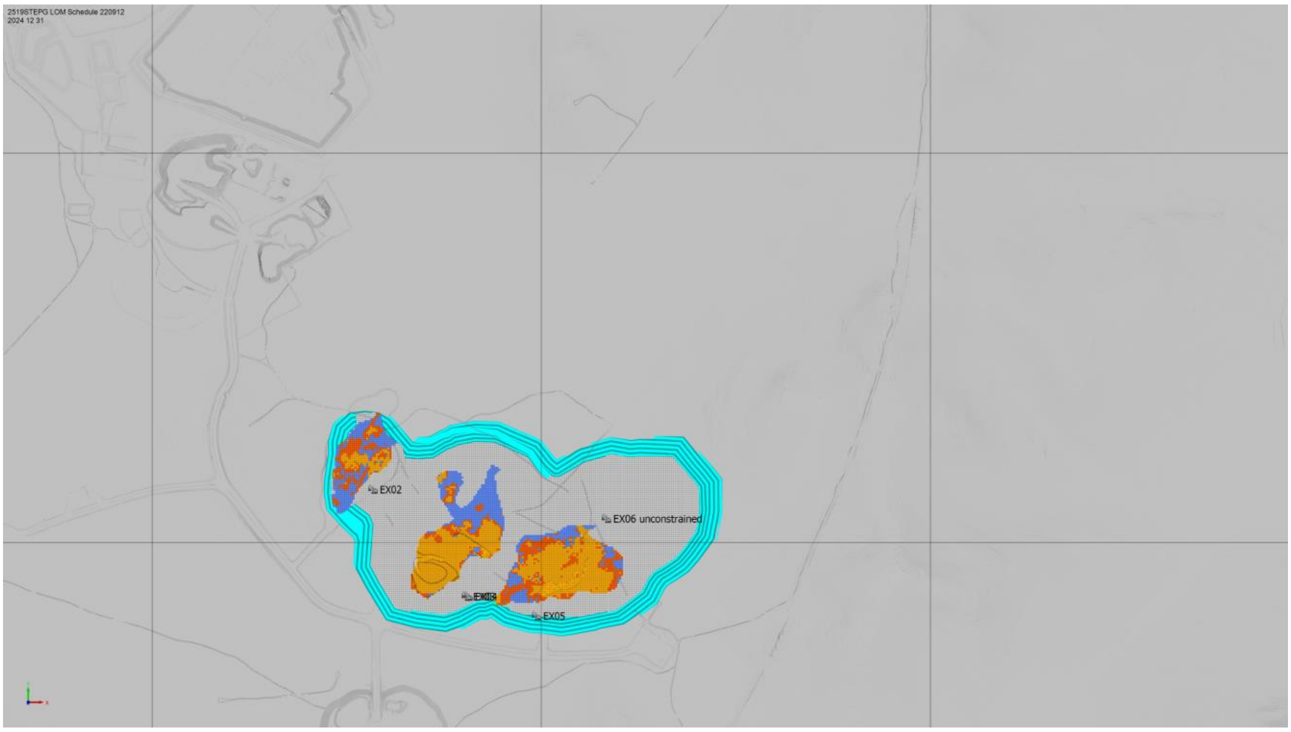


Figure 16.6 – Mining Position end of 2026

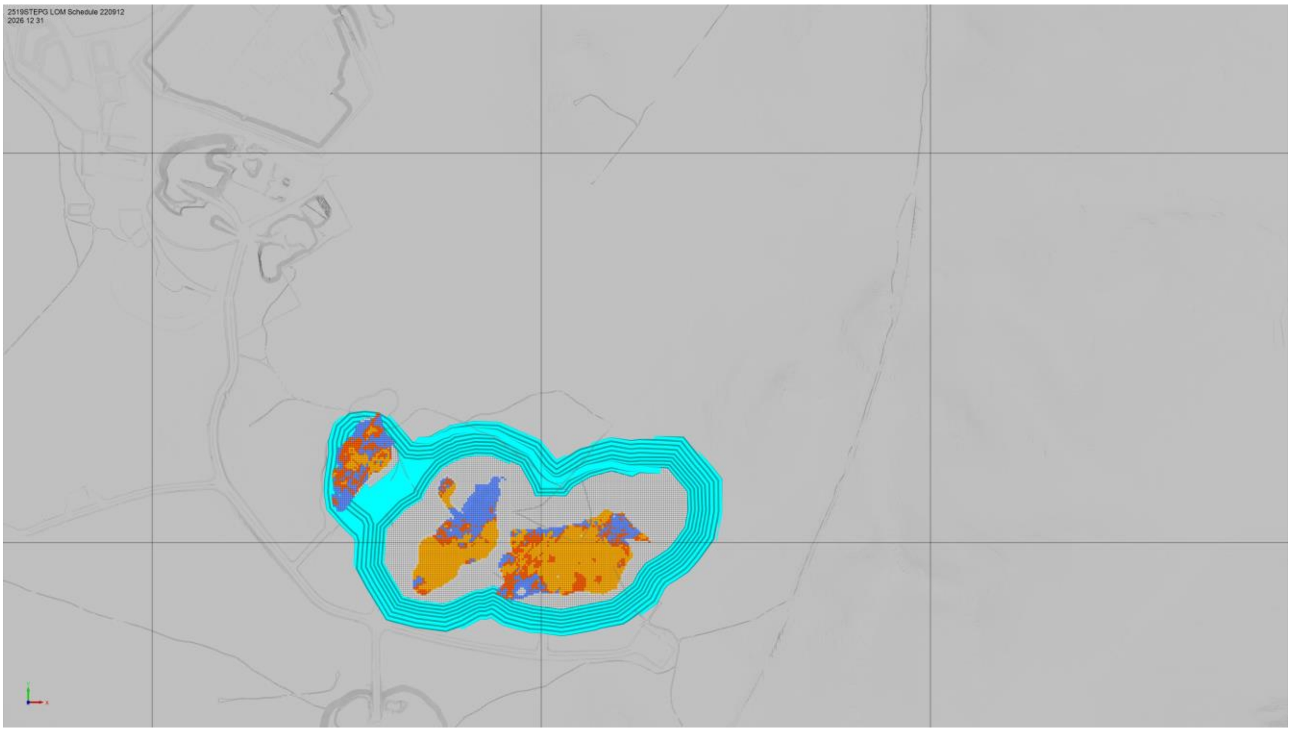


Figure 16.7 – Mining Position end of 2028

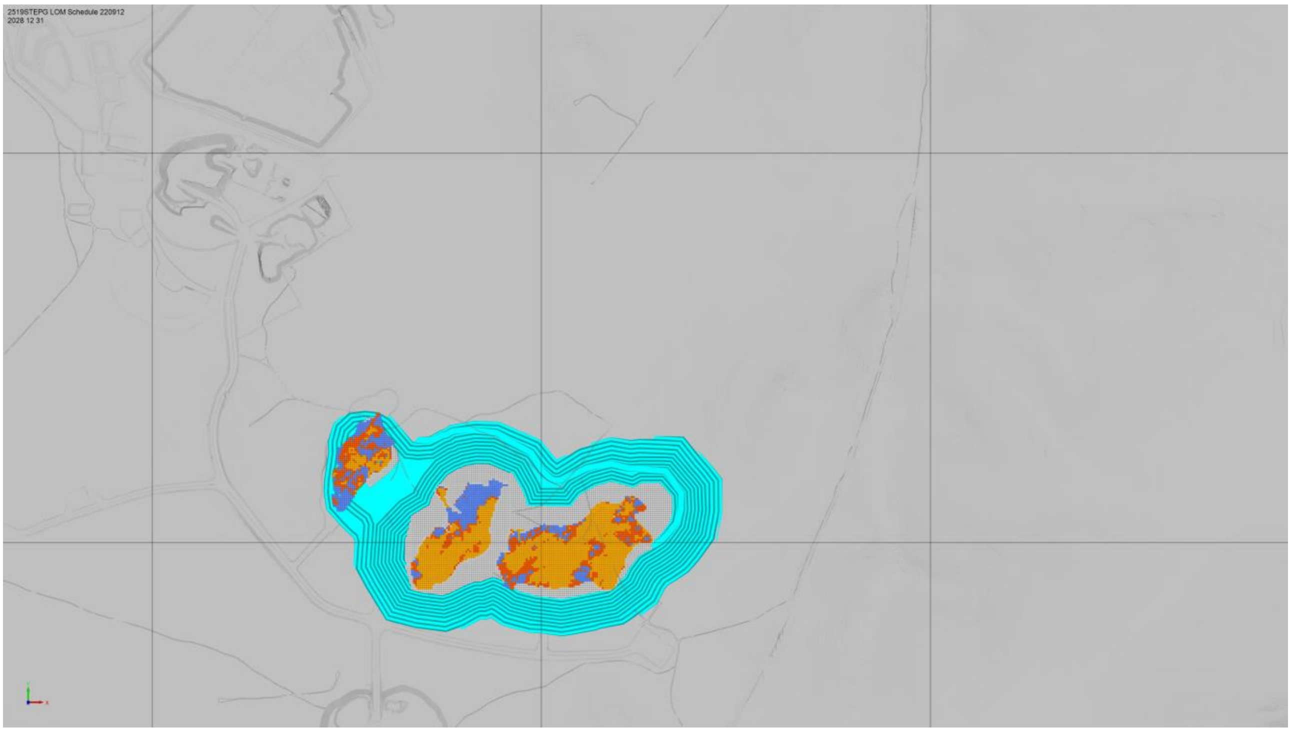


Figure 16.8 – Mining Position end of 2030

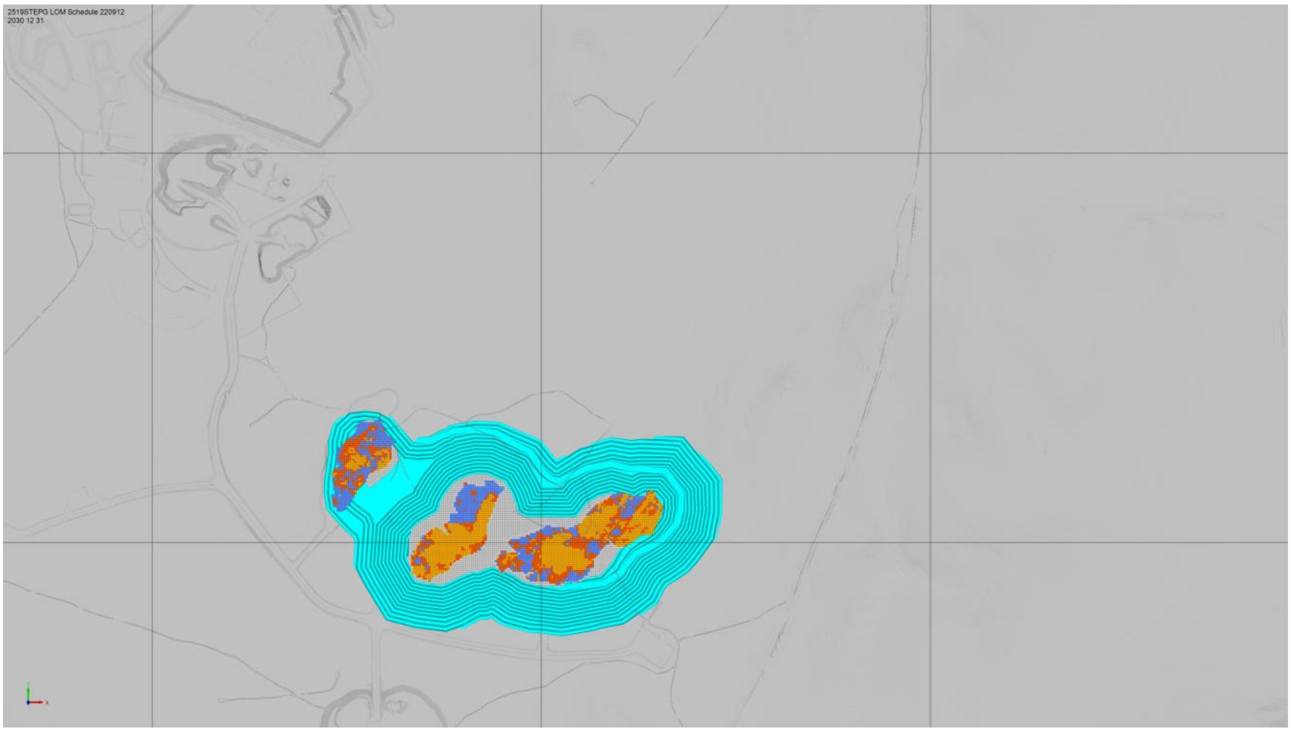


Figure 16.9 – Mining Position end of 2032

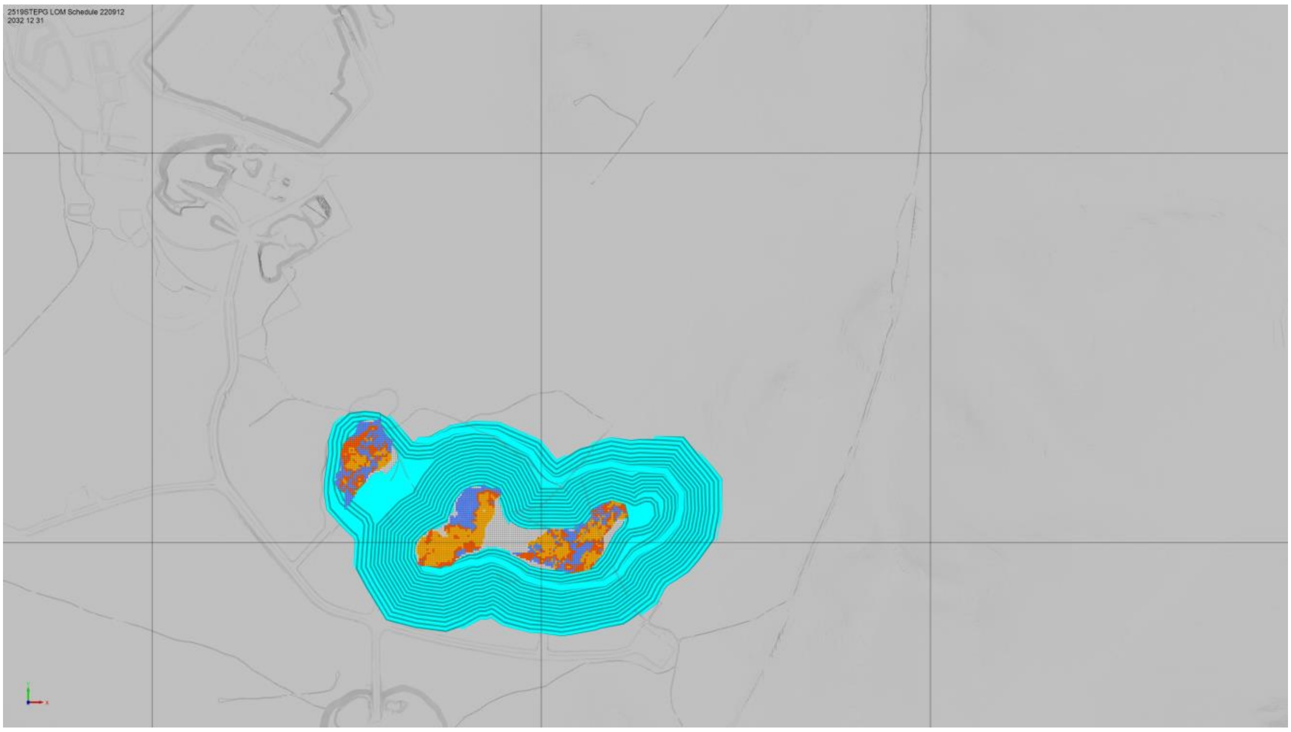
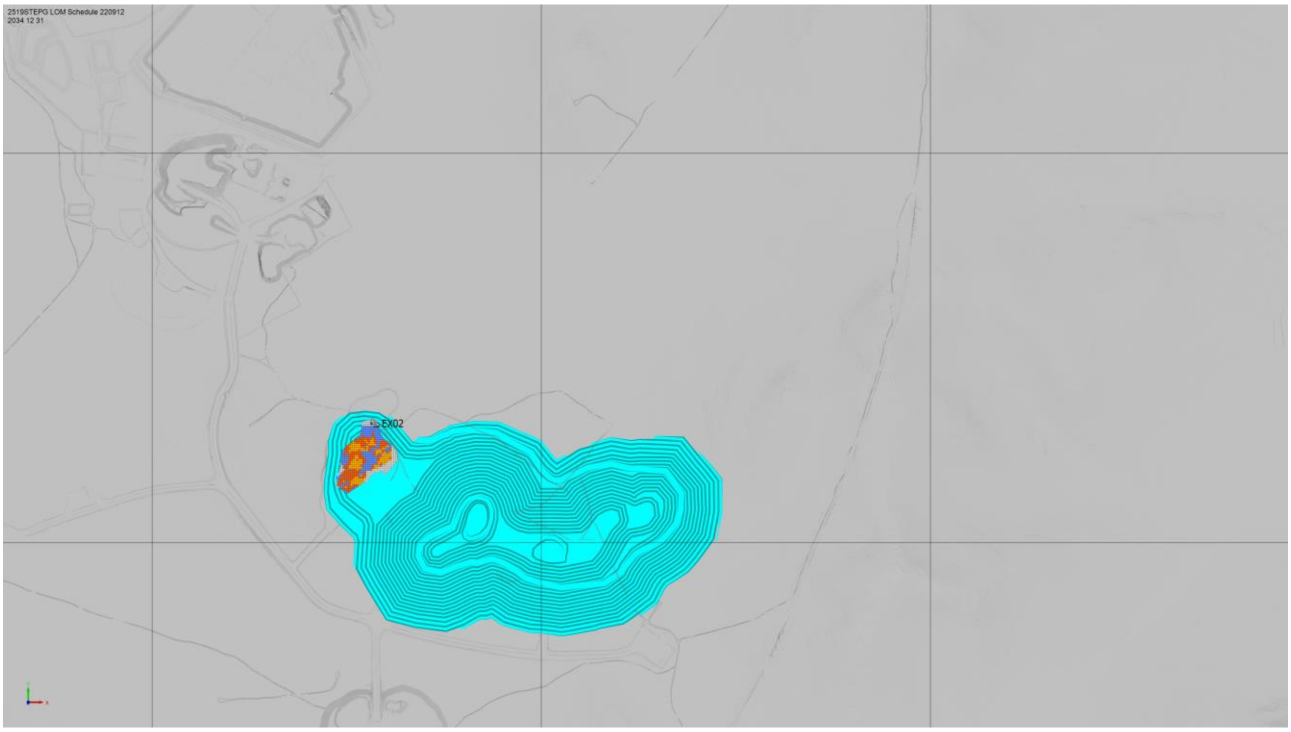


Figure 16.10 – Mining Position end of 2034



16.6 Mining Equipment

The mine is currently operated by a contractor with their own fleet. Table 16.3 presents the current and proposed mining fleet and support equipment at ATO.

The maximum capacity of the mining fleet is 3 Mt per month which is sufficient to meet LOM schedule movement requirements. The capacity of the mining fleet will be adjusted to match the LOM schedule requirements by Steppe Gold controlling the number of shifts worked per day.

Table 16.3 Contractor Mining Fleet

		2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
Hydraulic Excavator	**2.3m^3**	3	3	4	6	6	6	6	6	6	5	5	5	5	4	3
Rear Dump Truck	**32 t**	5	5	9	15	23	23	25	26	26	24	24	24	24	24	20
Dozer	**265 kw**	1	1	1	2	2	2	2	2	2	2	2	1	1	1	1
FEL	**1.5m^3**	1	1	1	2	2	2	2	2	2	2	1	1	1	1	1
Drill	**172mm**	2	2	3	3	3	3	3	3	3	3	3	3	3	2	2
Grader		1	1	1	2	2	2	2	2	2	2	2	2	1	1	1
Water Cart		1	1	1	2	2	2	2	2	2	2	2	2	1	1	1

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 dumps. 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: 60 min/shift;
- Rolling resistance: 3%;
- Maximum speed: 50 km/h, 30km/h across bench; 20 km/h down ramp and around excavator or dump points.

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. Spry software was used to determine truck cycle times.

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 m3 hydraulic excavators (in backhoe configuration) and 1.5 m3 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: 80 min/shift.

16.6.3 Ancillary fleet

Ancillary fleet includes water carts, explosives trucks, lighting plants, as well as fuel, lube and service trucks.

16.6.4 Drill & Blast

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.

16.7 Manning

Manpower requirements for the Project include both Owner and Contractor needs. Owner manpower requirements are presented in Table 16.4 and the proposed Contractor requirements are presented in Table 16.5.

Table 16.4 Owners Labour Numbers

Position	No.
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 16.5 Contractor Labour Numbers

		2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
Hydraulic Excavator		12	12	16	24	24	24	24	24	24	20	20	20	20	16	12
Rear Dump Truck		20	20	36	60	92	92	100	104	104	96	96	96	96	96	80
Dozer		4	4	4	8	8	8	8	8	8	8	8	4	4	4	4
Other operator		10	10	12	18	18	18	18	18	18	18	16	16	12	10	10
Maintenance		5	5	8	9	15	15	15	15	15	15	15	15	10	10	10
Supervisors/admin		2	4	4	8	8	8	8	8	8	8	8	8	8	8	8
Total		53	55	80	127	165	165	173	177	177	165	163	159	150	144	124

17 PROCESSING RECOVERY METHODS

The process described below covers two (2) phases of the overall process design. Phase 1 is deemed to be existing and in place.

- 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 Crushers: 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 Concentrator (Fresh and Transitional 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 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.

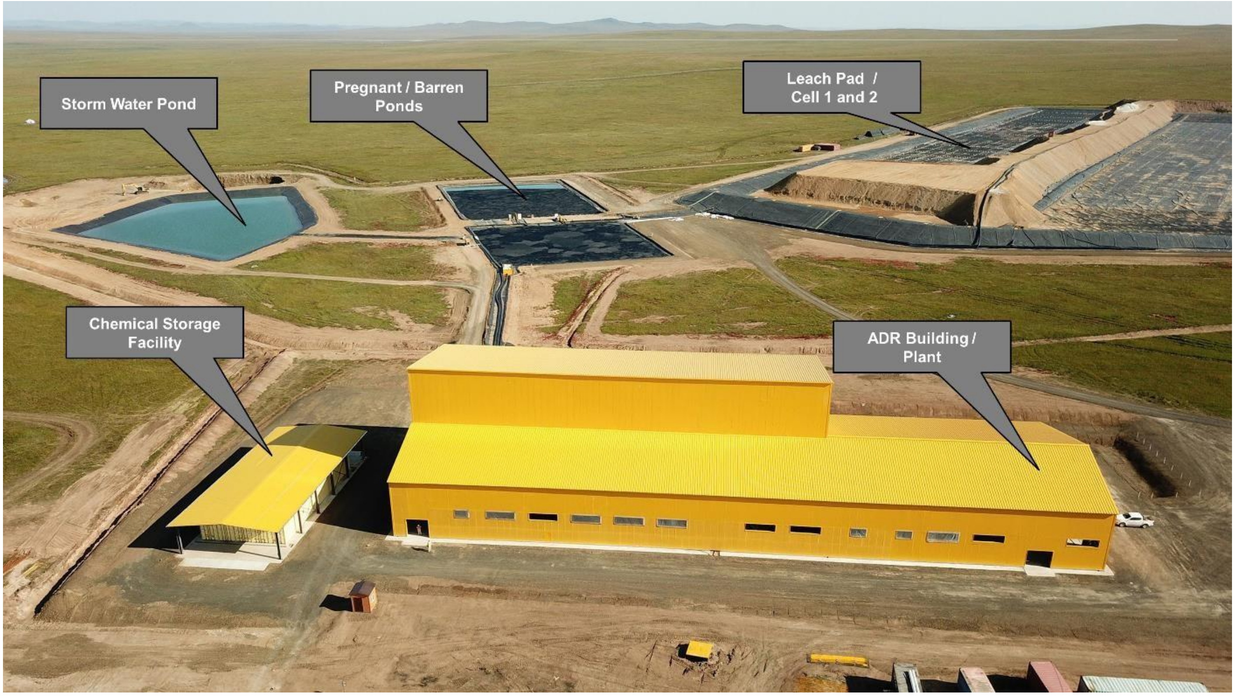
The existing crushing plant was designed to operate at a nominal throughput of 5,860 tpd for 275 days per year (75% utilisation).

A new three-stage crushing plant (currently being commissioned) 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 (P100) 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.

Figure 17.1 - Existing Phase I Site Processing Facilities



Source: Steppe Gold Photo

17.2.1 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 below describes the soon to be commissioned crushing plant

The new crushing circuit is designed for a capacity of 2.2 Mtpa

A three-stage crushing circuit will reduce run-of-mine (ROM) material from an F100 of 800 mm to a P80 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

17.2.2 Heap Leach Facility (HLF)

The existing HLF is designed to allow crushed ore stacking to a maximum height of approximately24 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.
- Permanent and interim perimeter diversion channels and berms to manage surface water flows.
- Perimeter access and ore haulage roads.
- Leach pad liner system

17.2.3 ADR Gold Recovery Plant

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.

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.

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.

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.

Refining

Pregnant solution flows by gravity to a secure gold room. The solution flows through one of two3.54 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.

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.

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;

An overall flow diagram summarising the concentrator plant and process flows is provided in Figure 17.2

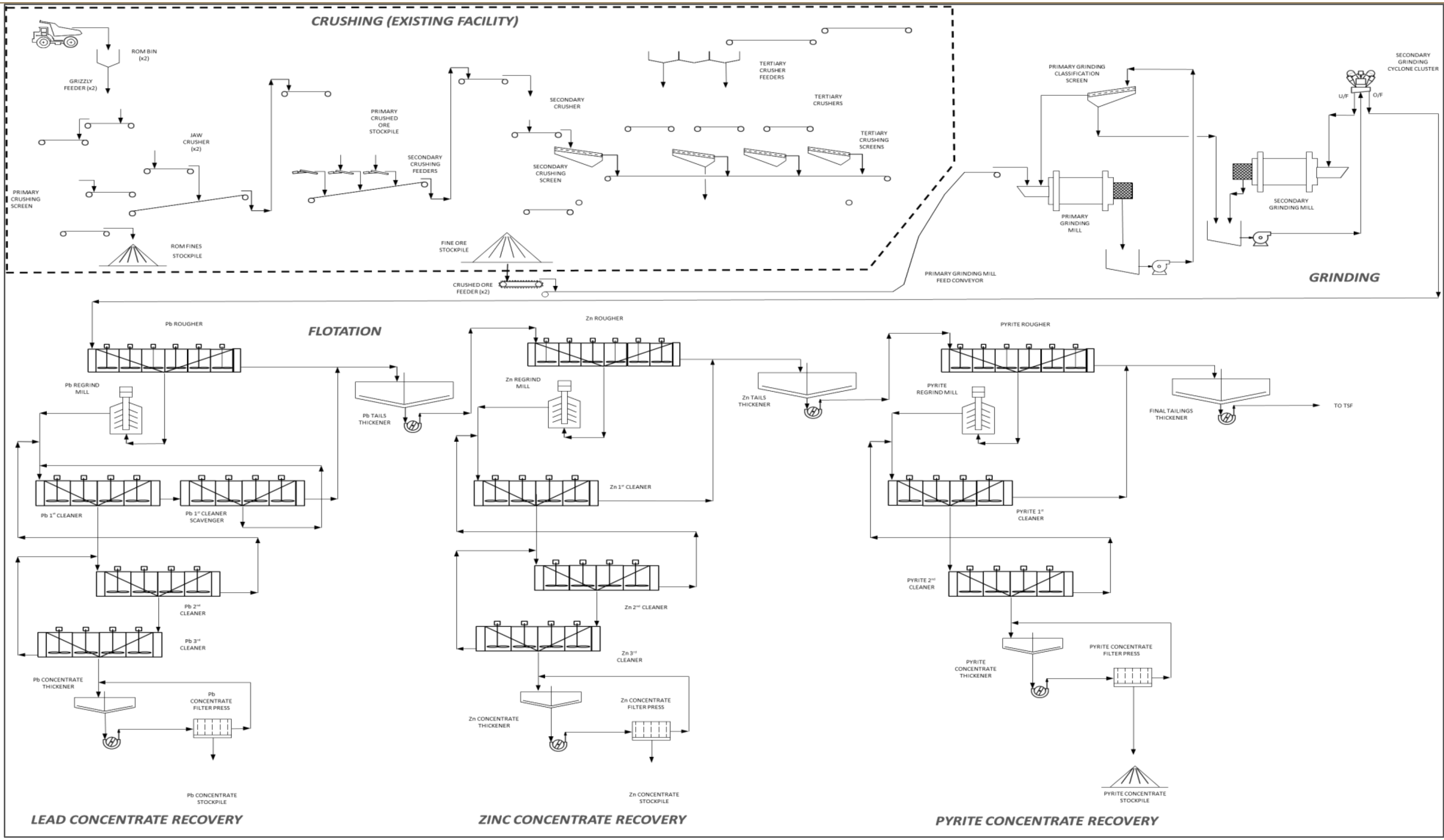
17.3.1 Grinding

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 (T80) 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.

Figure 17.2 – ATO Phase 2 Overall Flowsheet



17.3.2 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 P80 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 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 P80 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.

17.3.4 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 P80 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.

17.3.5 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.

17.3.6 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.

17.3.7 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.

17.3.8 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.

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.

18.1.1 Tank Farm

Currently, diesel is 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.

18.1.2 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 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.

18.1.3 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.

18.1.4 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

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

18.1.5 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

18.1.6 Tailings Storage Facility (TSF)

The TSF for Phase 2 will be located in a south-east facing valley approximately 2 km south-east of the pit

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.

19 MARKET STUDIES & CONTRACTS

The ATO Project is an operating site producing a readily saleable commodity in the form of gold bars. The bars are 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:

3. contains approximately 70% Au by weight and the remaining 30% is a mixture of Ag, base metals and Fe.
4. 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.

20 ENVIRONMENTAL STUDIES, PERMITTING & SOCIAL IMPACT

20.1 Environmental Setting

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.

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 Status of Environmental Social Permitss

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.

20.3 Environmental Management

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.

Management of the Project's significant environmental and social aspects and impacts is achieved through a suite of Management Plans. The following Management Plans have been developed and outline the expected Project performance.

- Air Quality (including noise and vibration) Management Plan
- Water Resources Management Plan
- Land and Soil Management Plan
- Biodiversity Management Plan
- Waste Management Plan
- Hazardous Materials Management Plan
- Occupational and Community Health, Safety and Security Management Plan
- Stakeholder and Communications Management Plan
- Local Cooperation and Development Management Plan

20.4 Waste and Tailings Disposal

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.

20.5 Community 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.

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.

20.6 Mine Closure

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.

21 CAPITAL & OPERATING COSTS

Costs are presented in United States dollars (USD)

21.1 Capital Cost Estimates

The LOM capital cost estimate for ATO is based on the x Year LOM Plan . As noted in other Items of this report, ATO has been operating and expanding the mine and process plant since 2020 and all Phase 1 infrastructure is in place.

The total estimated capital cost is $128 million (M), which primarily consists of capital for the phase 2 Process Plant and tailings dam. The capital estimate is based on the plant design presented in the 2021 NS 43-101 Report.

Table 21.1 ATO Capital Cost Summary

	Total Capital (SD M)
Mining	1.8
Process Plant	75.2
Tailings/ reclaim water and water treatment	13.5
Power	1.7
Indirect	23.3
Owner's Cost	1.5
Contingency	11.5
	128.5

The USD 1.8M capital estimate for mining relates to the development of a new haul road from the pit to the phase 2 Process Plant.

The indirect costs include design, procurement and construction management activities, vendor representatives, spare parts and first fills, and contractor's indirect costs

21.1.1 Sustaining Capital

The sustaining capital requirements for the process plant includes for the purchase of spare parts for equipment, and for replacement of equipment when required. An allowance of USD 17 M has been included in the financial model.

21.1.2 Mine Closure Provision

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 M include for the dismantling and removal of all facilities and services and re-vegetation of the area plus an allowance for contingency.

21.2 Operating Cost Estimates

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 bars during 2022 and 2023 and Phase 2 which includes the new processing plant that will produce lead, zinc and pyrite concentrates from Q1 2024 to Q1 2036.

Life-Of-Mine (LOM) operating costs were calculated based on actual site costs. The summary of average LOM operating costs is listed in Table 21-3. These costs are shown in $/tonne processed.

Table 21.2 Average Operating Costs

	Av. Annual Cost (USD M)	Cost / t ore processed (USD/t)	Total Cost LOM (M USD)
Mining	17.7	7.87	249
Processing	30.2	13.44	425
General Admin	12.2	5.50	174
Total	**60.1**	**26.81**	**848**

21.2.1 Mining Cost

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.

Table 21.3 Mining Operating Costs

	$/t material mined (USD/t)
Contract Mining Waste	1.80
Contract Mining Ore	2.10

21.2.2 Processing and G&A costs

Based on oxides that is leached and the plant design nameplate feed throughput capacities of 2.2 Mtpa for Phase 2, the estimated process operating costs are divided into five (5) main components.

The breakdown of these costs is summarised in vv and is detailed in the following sections

Table 21.4 Processing Costs

	Av. Annual Cost (USD M)	Cost / t ore processed (USD/t)
Consumables	8.7	4.12
Labour	3.0	1.44
G & A (excl. camp)	1.9	0.92
Maintenance	3.9	1.85
Power	10.8	5.12
Total	**28.4**	**13.44**

The General and Administration (G&A) costs include the following categories:

- Corporate;
- Insurance
- Marketing
- Site services.

21.2.3 Metal costs

Mineral production in Mongolia is subject to fixed and sliding scale government royalties.

In addition to royalties, metal costs for AU, AG, Pb and Zn in concentrate comprise:

- concentrate transport charges ;
- concentrate refining charges;
- payable rates for each metal recovered into concentrate.

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%.

Table 22.1 summarise the economic/financial results of the Project for the base case (Phase 1 and Phase 2). All figures are in USD currency.

22.1 Revenue

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:

- \$1,700/oz Au;
- \$20/oz Ag;
- \$1,970/t Pb;
- \$2,500/t Zn.

For the sensitivity analysis prices of metals were evaluated at 80%, 90%, 110% and 120%, prices per metal are shown as follows:

22.2 Escalation and Inflation

There is no adjustment for inflation and escalation in the financial model; all cash flows are in US dollars.

22.3 Depreciation

Depreciation using a straight-line balance method at 10% 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.

22.4 Tax

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.

22.5 Royalty

Mineral production in Mongolia is subject to fixed and sliding scale government royalties. The financial model assumes a flat rate of 5% royalty.

22.6 Cash Flow

The financial analysis results are summarised in Table 22.1.

Table 22.1 Financial Summary

Description	Unit	Value	
LOM Tonnage Ore Processed	kt	29,103	
LOM Feed Grade Processed - Au	g/t	1.01	
LOM Feed Grade Processed - Ag	g/t	8.59	
LOM Feed Grade Processed - Pb	%	0.42	
LOM Feed Grade Processed - Zn	%	0.71	
LOM Recovery - Au	%	79.2	
LOM Recovery - Ag	%	72.6	
LOM Recovery - Pb	%	82.5	
LOM Recovery - Zn	%	85.9	
Total Net Revenue (after payable & streaming)	USD Million	2,003	
Refining/Transport Costs	USD Million	229	
LOM Operating Costs	USD Million	848	
		Pre tax	Post Tax
NPV @ 5%	USD Million	364	242
NPV @ 8%	USD Million	273	176
NPV @ 10%	USD Million	226	142

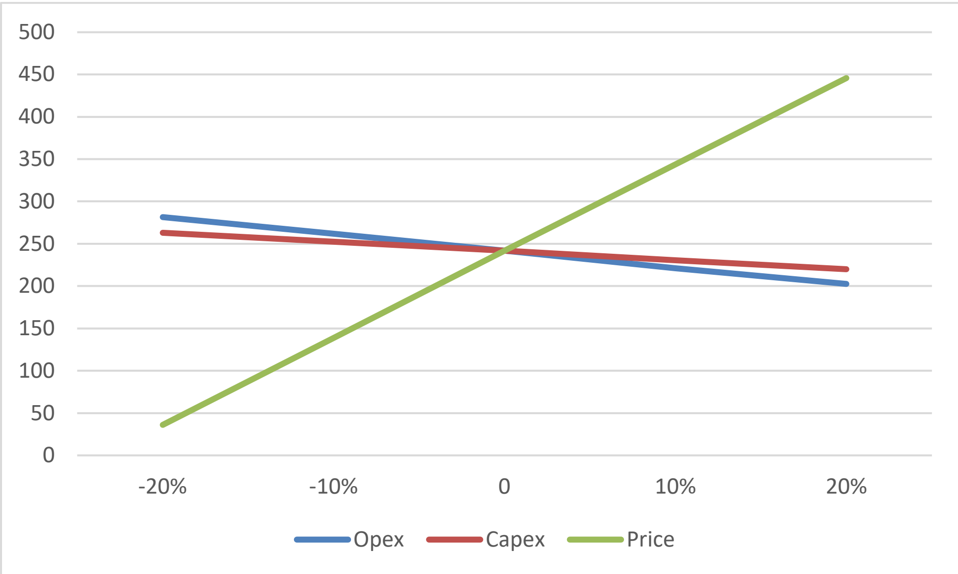
22.7 Sensitivity

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 (initial) capital expenditure ("CAPEX"), operating costs ("OPEX") and product prices ("PRICE") on the Project NPV @ 5 %. 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, indicate that 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 22.1 – Economic Sensitivity



23 ADJACENT PROPERTIES

There are no adjacent properties or relevant information pertaining to adjacent properties that are material to this Technical Report.

24 OTHER RELEVANT DATA & INFORMATION

There is no other relevant information or explanation required to make this Technical Report understandable and not misleading.

25 INTERPRETATION & CONCULSIONS

25.1 Mineral Resource modelling and estimation

The overall interpretation of the estimation and resulting Resources was that it proceeded as expected, confirmed the 2017/2021 modelling and results of Pipes 1, 2 and 4, and produced a more accurate second generation result.

Interpretation of mineralisation at the new Mungu deposit showed it to be more complex and lode-like than the other deposits, with greater potential for increasing the Resource with targeted drilling. Its different shape gives encouragement for further regional exploration to find other deposits of its style.

25.1.1 Conclusions

Although not part of the original Project scope the Author QP nevertheless would share his positive views on the Resource resulting from the Consulting. Conclusions drawn about this 2021 Resource estimate were:

Measured and Indicated Resources:

- Pipes 1, 2 and 4 deposit:
  - The 2021 estimate confirmed the quantum of the 2017 estimate of Pipes 1, 2 and 4.
  - It increased the Resource tonnage significantly (by 25%), decreased the gold grade slightly (by 3%), increased the silver grade reasonable (by 16%), leading to an overall significant increases in metal contents (gold by 22%, silver by 45%). This comparison uses equivalent reporting cut-offs between the two estimates.
  - The increase in deposit volume was not only because of additional drill holes but also because of more practical and geologically controlled deposit shape interpretation.
- Mungu:
  - The cross-sectional interpretations hung together and created a significant deposit.
  - This maiden Measured and Indicated Resource was significant at 7.6 Mt at 1.16 g/t gold (282 k oz gold metal) and 40.75 g/t silver (9,916 k oz silver metal).
  - Mungu now represents 18% by tonnage of the Resources, 20% of the gold metal and 48% of the silver metal (as the silver grade is 320% higher than for Pipes 1, 2 and 4).
- All deposits:
  - The total Resource for all deposits stands at 41.6 Mt at 1.04 g/t gold (1.4 Moz gold metal) and 15.31 g/t silver (20.5 Moz silver metal).
  - The absolute comparison (Table 52) of the 2017 Resource (reported at much higher cut-off grades) with the 2021 Resource shows increases of tonnage by 136%, decreases of average gold grade by 27%, and increases of average silver grade by 53%.
  - The absolute comparison of contained metal shows a 73% increase in gold metal (to 1.4 Moz) and a 262% increase in silver metal (to 20.5 Moz).

Table 52 Absolute Resource differences 2017 / 2021

Estimate		Cut-off AuEq (g/t)	Bulk density (t/m³)	Tonnes (M t)		Au (g/t)		Ag (g/t)		Metal Au (k oz)		Metal Ag (k oz)	
GSTATS 2017	Oxide	0.30	2.46	5.3	30%	1.23		9.96		208		1,680	
	Fresh	1.10	2.64	12.4	70%	1.50		10.03		594		3,980	
	Total			**17.6**		**1.42**		**10.01**		**802**		**5,660**	
GeoRes 2021	Oxide	0.15	2.46	8.0	19%	0.86		10.72		221		2,763	
	Transition	0.40	2.59	10.3	25%	1.24		12.97		412		4,307	
	Fresh	0.40	2.64	23.3	56%	1.01		17.93		757		13,409	
	Total			**41.6**		**1.04**		**15.31**		**1,390**		**20,479**	
2021 difference	Oxide			2.8	53%	-0.37	-30%	0.76	8%	13	6%	1,083	64%
	Fresh			21.2	172%	-0.42	-28%	6.37	64%	575	97%	13,736	345%
	Total			**24.0**	**136%**	**-0.38**	**-27%**	**5.30**	**53%**	**588**	**73%**	**14,819**	**262%**

Adequacy of data & sources: The Author QP believes that all data supplied and used in the estimation was suitable for the purpose of Mineral Resource estimation and JORC classification.

Drilling data: 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.

Compliance with JORC and Canadian NI 43-101 standards:

- To the best of his knowledge the Author QP believes that the Mineral Resource estimation Project, and the reporting of it, comply with the JORC (2012) and NI 43-101 (June 2011) standards.
- He notes that in order to comply with the CIM Definitions requirement of "reasonable prospects for eventual economic extraction" the 2017 Report[32] included details on pit optimisation studies (which incorporated metal recoveries, dilution etc) and pit design leading to the reporting of Mineral Reserves, and made the assertion that the Resources met the requirement.
- The Author QP considers the results of that previous work to have been successful and therefore continue to demonstrate the "reasonable prospects for economic extraction".
- The fact that mining has commenced at the ATO Mine further reinforces this compliance with the Code.

25.1.2 Potential risks to the estimate

The Author QP does not consider that input data, geological interpretation or grade estimation parameters pose any significant risk to the quantum of the reported Resources.

Aspects which could alter the Resources slightly (not significantly) were discussed fully in Section 14.20. Of those mentioned the issue of mining method to apply to Mungu (given its greater depth than the other deposits), which could alter the economic cut-off grade, would seem to be the greatest risk to these

[32] 2017 NI 43-101. Section 25.2, 4th paragraph, pp206.

Resources. However that could only alter Resources for the lowest parts of Mungu and, as Mungu only represents 20% by tonnage of the overall Resources, that risk appears minimal.

25.2 Mine planning and Mineral Reserve estimation

ATO is an established conventional open cut gold/silver mine that has been in operation for several years.

Mine planning and evaluations undertaken using the latest resource models confirm that ATO is both viable and economic. The Mineral Reserve estimate has a relatively high sensitivity to revenue which is controlled by metal prices and payability. It is noted however that at current long term forecast metal prices, the mineral reserve is relatively insenstive to changes in revenue and costs.

Given the mine has been operational for a number of years, technical risk in relation to the Mineral Reserves estimate is deemed to be low.

26 RECOMMENDATIONS

Recommendations were outside the Scope of Work for the Author QP's Resource estimation. And with the ATO Mine now in operation and mine planning assumed to be at a fairly advanced stage the capacity for recommendations is limited.

However the Author QP sees the potential for further drilling to enlarge the defined deposits (particularly at Mungu) and for further exploration to find new ones.

So his simple recommendations would be to:

- Continue drilling around and nearby the deposits in a targeted way (particularly by using cross-sectional data provided in this Report). The drill spacing could be widened to ~45 m.
- Explore locally for other deposits of the Mungu type (as this style of mineralisation may have previously been overlooked).

27 REFERENCE

Canadian National Instrument 43-101 (NI 43-101):

- *Form F1*: British Columbia Securities Administrators (BCSC), 24th June 2011. *Form 43-101F1 Technical Report – Contents of the Technical Report*. Document setting out the specific structure (sections and content) of an NI 43-101 Technical Report.
- *Companion Policy CP*: British Columbia Securities Administrators (BCSC), 25th February 2016. *Companion Policy 43-101CP to National Instrument 43-101 – Standards of Disclosure for Mineral Projects*. Document sets out the views of the Canadian Securities authorities as to how they interpret and apply the NI 43-101 and the Form 43-101F1.

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).

Oyungeral Bayanjargal, 4 October 2017. *NI 43-101 Technical Report – Altan Tsagaan Ovoo Gold Project*. Report for Steppe Gold LLC by GSTATS Consulting LLC, Mongolia. Report reviewed by Qualified Persons Leonid Tokar (GSTATS), Tim Fletcher (DRA Americas Inc), David Frost (DRA) and Dr Martin Stapinsky (DRA). Referenced as the '2017 NI 43-101' or '2017 Report'

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.

Altan Tsagaan Ovoo Project (ATO) 2021 Mineral Resources Technical Report (Amended NI 43-101), Effective 30th March 2021 by Robin A Rankin (GeoRes)

Matthew Wood (Executive Charmain), 27 August 2020. *Geological Consultant engagement letter*. Emailed letter to GeoRes from Steppe Gold Ltd..

NI 43-101 Technical Report Feasibility Study for the Altan Tsagaan Ovoo (ATO) Phase 2 Expansion Project Mongolia. Effective Date of Report: October 27, 2021. Report for Steppe Gold by DRA. Referenced as the '2021 NI 43-101' or '2021 FS Report'

APPENDIX A. DRILL HOLE & COLLAR SURVEYS

Drill hole	Easting (m)	Northing (m)	Elevation (m)	Depth (m)	Azimuth (°)	Dip (°)	Type
AT diamond holes							
ATO01	631,779.9	5,367,067.8	1,063.7	221.4	38	-60	DDH
ATO02	631,725.2	5,366,995.8	1,067.6	242.2	35	-60	DDH
ATO03	631,576.1	5,367,109.1	1,056.7	200.1	35	-60	DDH
ATO04	631,399.1	5,367,217.6	1,049.0	200.3	35	-60	DDH
ATO05	631,241.8	5,367,323.6	1,044.8	200.2	35	-60	DDH
ATO06	631,462.3	5,367,117.2	1,053.1	209.0	35	-60	DDH
ATO07	631,669.5	5,366,908.0	1,050.8	208.2	35	-60	DDH
ATO08	631,458.2	5,366,462.6	1,028.5	176.2	35	-60	DDH
ATO09	631,667.8	5,366,908.3	1,050.9	120.5	215	-60	DDH
ATO10	631,680.8	5,367,024.3	1,060.1	201.7	305	-60	DDH
ATO100	632,062.3	5,367,008.0	1,050.9	225.9	125	-60	DDH
ATO101	631,741.2	5,367,015.1	1,068.0	197.3	125	-60	DDH
ATO102	631,994.8	5,366,909.1	1,051.3	116.0	125	-60	DDH
ATO103	632,327.6	5,367,497.1	1,056.8	200.3	125	-60	DDH
ATO104	632,096.4	5,367,054.8	1,049.4	158.3	125	-60	DDH
ATO105	631,809.4	5,366,894.7	1,052.6	115.7	125	-60	DDH
ATO106	631,791.2	5,366,979.7	1,063.4	110.3	125	-60	DDH
ATO107	632,053.5	5,366,941.5	1,051.0	160.9	125	-60	DDH
ATO108	631,798.2	5,367,046.0	1,065.7	170.3	125	-60	DDH
ATO109	632,026.9	5,366,959.2	1,051.5	147.7	305	-60	DDH
ATO11	631,717.3	5,366,873.3	1,049.8	220.8	35	-60	DDH
ATO110	631,845.0	5,367,011.5	1,059.8	200.3	125	-60	DDH
ATO111	631,980.2	5,366,992.3	1,053.1	500.8	125	-60	DDH
ATO112	631,490.9	5,367,078.8	1,052.6	251.3	125	-60	DDH
ATO113	631,525.6	5,367,128.5	1,061.5	284.5	305	-80	DDH
ATO114	631,571.7	5,367,246.6	1,054.1	144.9	305	-60	DDH
ATO115	632,101.3	5,366,907.4	1,051.1	104.7	125	-60	DDH
ATO116	632,108.5	5,366,972.6	1,051.0	139.2	125	-60	DDH
ATO117	631,489.9	5,367,077.1	1,052.9	108.0	305	-60	DDH
ATO118	631,707.0	5,367,428.5	1,039.7	99.2	125	-60	DDH
ATO119	632,106.7	5,366,970.3	1,051.2	222.0	305	-60	DDH
ATO12	631,771.2	5,366,956.8	1,062.5	201.4	35	-60	DDH
ATO120	631,813.5	5,366,962.8	1,059.0	281.3	305	-60	DDH
ATO121	632,011.5	5,367,042.4	1,052.1	285.9	125	-60	DDH
ATO122	631,841.7	5,367,056.4	1,063.2	224.3	305	-60	DDH
ATO123	632,162.2	5,366,936.2	1,052.4	129.4	125	-60	DDH
ATO124	631,880.2	5,367,061.7	1,058.2	224.3	305	-60	DDH
ATO125	631,720.4	5,366,881.2	1,050.2	224.3	305	-60	DDH
ATO126	632,145.8	5,367,021.4	1,050.6	145.6	305	-60	DDH
ATO127	631,760.8	5,366,888.4	1,051.4	200.3	305	-60	DDH

ATO128	632,015.6	5,366,891.8	1,050.5	140.6	305	-60	DDH
ATO129	631,778.1	5,366,913.9	1,054.0	68.3	125	-60	DDH
ATO13	631,774.9	5,366,958.5	1,062.5	199.9	125	-60	DDH
ATO130	632,070.5	5,367,071.5	1,049.2	205.7	125	-60	DDH
ATO131	631,755.4	5,366,931.4	1,055.8	227.3	305	-60	DDH
ATO132	632,147.6	5,367,023.5	1,050.3	151.8	125	-60	DDH
ATO133	631,647.8	5,367,044.3	1,058.0	323.3	125	-60	DDH
ATO134	632,016.5	5,366,894.5	1,050.7	101.8	125	-60	DDH
ATO135	632,086.4	5,366,989.9	1,050.6	194.4	125	-60	DDH
ATO136	631,854.6	5,366,969.2	1,057.3	269.3	305	-60	DDH
ATO137	632,036.6	5,367,024.9	1,051.5	257.0	125	-60	DDH
ATO138	631,871.3	5,366,995.7	1,058.2	152.3	125	-60	DDH
ATO139	631,871.6	5,366,998.1	1,058.4	236.3	305	-60	DDH
ATO14	631,826.1	5,367,029.2	1,063.8	199.8	125	-60	DDH
ATO140	632,086.0	5,366,994.0	1,050.7	186.5	35	-45	DDH
ATO141	631,879.3	5,367,059.7	1,058.2	242.3	125	-60	DDH
ATO142	631,855.5	5,366,863.4	1,050.4	88.3	125	-60	DDH
ATO143	632,051.3	5,366,942.2	1,051.1	174.7	215	-45	DDH
ATO144	631,769.9	5,366,847.6	1,048.7	77.6	125	-60	DDH
ATO145	632,149.8	5,366,871.5	1,052.3	87.7	125	-60	DDH
ATO146	632,041.1	5,366,875.9	1,050.2	72.4	125	-60	DDH
ATO147	631,894.6	5,366,976.8	1,056.1	199.3	125	-60	DDH
ATO148	631,965.0	5,367,074.7	1,053.7	276.9	125	-60	DDH
ATO149	631,790.1	5,366,977.1	1,063.5	230.3	305	-60	DDH
ATO15	631,827.2	5,367,031.7	1,063.8	200.2	35	-60	DDH
ATO150	631,969.3	5,366,924.8	1,052.2	166.6	125	-60	DDH
ATO151	632,168.5	5,367,004.8	1,051.4	151.9	125	-60	DDH
ATO152	631,666.7	5,367,065.5	1,059.1	305.3	125	-60	DDH
ATO153	632,182.8	5,366,918.9	1,053.1	99.9	125	-60	DDH
ATO154	632,128.8	5,367,102.3	1,048.3	170.1	124	-60	DDH
ATO155	631,838.5	5,366,946.7	1,056.1	87.2	125	-60	DDH
ATO156	632,177.9	5,367,071.0	1,050.6	161.0	125	-60	DDH
ATO157	631,878.2	5,366,953.4	1,055.5	89.3	125	-60	DDH
ATO158	631,812.7	5,366,961.3	1,059.3	92.3	125	-60	DDH
ATO159	632,077.0	5,367,140.3	1,047.3	258.2	125	-60	DDH
ATO16	631,618.7	5,366,944.1	1,050.8	199.9	35	-60	DDH
ATO160	631,722.3	5,366,883.1	1,050.2	75.7	125	-60	DDH
ATO161	632,174.1	5,367,140.8	1,048.8	250.6	125	-60	DDH
ATO162	631,759.9	5,366,885.2	1,051.1	80.3	125	-60	DDH
ATO163	631,887.4	5,367,019.1	1,057.6	71.3	125	-60	DDH
ATO164	631,627.6	5,367,276.6	1,048.3	131.3	305	-60	DDH
ATO165	631,612.5	5,367,361.2	1,042.9	100.3	125	-60	DDH
ATO166	631,957.6	5,366,859.8	1,049.7	121.1	125	-60	DDH
ATO167	632,194.4	5,366,987.3	1,052.2	121.2	125	-60	DDH
ATO168	632,065.7	5,366,855.8	1,050.2	69.2	125	-60	DDH

ATO169	632,204.5	5,367,052.9	1,051.7	160.9	125	-60	DDH
ATO17	631,689.5	5,367,019.0	1,060.4	207.3	35	-60	DDH
ATO170	632,125.9	5,366,888.9	1,051.4	98.3	125	-60	DDH
ATO171	632,133.7	5,366,953.9	1,051.4	182.3	125	-60	DDH
ATO172	632,153.1	5,367,089.8	1,049.1	170.1	125	-60	DDH
ATO173	632,121.9	5,367,038.8	1,049.5	158.3	125	-60	DDH
ATO174	632,105.1	5,367,121.5	1,047.7	230.9	125	-60	DDH
ATO175	632,045.0	5,367,091.5	1,049.9	242.3	125	-60	DDH
ATO176	631,655.8	5,366,890.5	1,049.6	99.7	125	-60	DDH
ATO177	631,829.1	5,366,878.7	1,051.5	108.7	125	-60	DDH
ATO178	631,901.6	5,366,933.3	1,053.8	112.0	125	-60	DDH
ATO179	631,758.5	5,367,146.1	1,056.9	287.4	125	-60	DDH
ATO18	631,523.4	5,367,202.3	1,066.2	199.8	305	-60	DDH
ATO180	631,677.6	5,366,873.4	1,049.0	95.3	125	-60	DDH
ATO181	632,218.4	5,366,969.6	1,053.3	99.9	125	-60	DDH
ATO182	632,137.9	5,367,172.5	1,046.9	269.0	125	-60	DDH
ATO183	631,460.3	5,367,247.2	1,052.1	259.0	125	-60	DDH
ATO184	631,496.3	5,367,296.7	1,047.9	221.9	125	-60	DDH
ATO185	631,634.3	5,367,017.9	1,055.8	270.7	125	-60	DDH
ATO186	631,430.3	5,367,199.4	1,051.8	264.2	125	-60	DDH
ATO187	632,326.7	5,366,966.9	1,058.2	152.0	125	-60	DDH
ATO188	631,843.6	5,367,849.2	1,035.3	155.3	215	-60	DDH
ATO189	631,305.8	5,366,895.3	1,028.9	63.2	125	-60	DDH
ATO19	631,524.7	5,367,204.0	1,066.2	160.7	125	-60	DDH
ATO190	630,989.5	5,367,143.7	1,022.6	118.5	125	-60	DDH
ATO191	630,792.2	5,367,518.0	1,014.6	149.3	125	-60	DDH
ATO192	632,745.9	5,368,929.0	1,079.2	200.0	125	-60	DDH
ATO193	631,197.2	5,367,034.5	1,028.7	149.3	125	-60	DDH
ATO194	632,645.9	5,368,796.7	1,069.8	93.4	125	-60	DDH
ATO195	631,866.5	5,367,192.1	1,053.2	152.3	125	-60	DDH
ATO196	632,686.0	5,366,958.6	1,090.9	201.5	125	-60	DDH
ATO197	631,962.4	5,367,232.9	1,047.8	710.3	215	-60	DDH
ATO198MET	631,679.5	5,366,980.1	1,056.7	145.0	125	-60	DDH
ATO199MET	631,780.1	5,367,024.9	1,069.3	150.0	125	-60	DDH
ATO20	631,525.9	5,367,202.3	1,066.3	208.9	35	-60	DDH
ATO200MET	632,077.7	5,366,928.9	1,051.1	145.0	315	-60	DDH
ATO201MET	631,526.0	5,367,206.2	1,066.0	125.0	125	-60	DDH
ATO202	632,421.7	5,366,912.7	1,064.0	200.1	125	-60	DDH
ATO203	630,511.8	5,367,909.6	1,028.6	299.1	305	-60	DDH
ATO204	633,023.3	5,368,668.6	1,085.4	200.3	125	-60	DDH
ATO205	629,791.4	5,367,498.3	1,007.3	152.3	125	-60	DDH
ATO206	631,156.7	5,367,166.4	1,035.1	200.3	305	-60	DDH
ATO207	631,389.7	5,366,841.1	1,033.8	152.3	305	-60	DDH
ATO208	631,733.6	5,367,166.9	1,055.5	351.6	125	-60	DDH
ATO209	631,775.0	5,367,174.0	1,055.5	285.6	125	-60	DDH

ATO21	631,855.4	5,367,078.5	1,060.1	184.7	35	-60	DDH
ATO210	631,418.8	5,367,132.7	1,049.5	269.2	125	-60	DDH
ATO211	631,741.5	5,367,123.3	1,059.8	286.6	125	-60	DDH
ATO212	631,725.0	5,367,101.4	1,062.8	301.6	125	-60	DDH
ATO213	631,707.9	5,367,075.6	1,061.5	272.2	125	-60	DDH
ATO214	631,709.2	5,367,181.2	1,055.8	371.2	125	-60	DDH
ATO215	632,029.9	5,366,391.2	1,058.7	149.6	360	-90	DDH
ATO216	631,749.4	5,367,187.5	1,053.9	339.4	125	-60	DDH
ATO217	631,790.7	5,367,197.4	1,053.8	298.9	125	-60	DDH
ATO218	631,944.8	5,366,944.0	1,053.3	160.8	125	-60	DDH
ATO219	631,919.8	5,366,957.8	1,054.5	181.7	125	-60	DDH
ATO22	631,736.4	5,366,905.7	1,052.6	118.7	215	-60	DDH
ATO220	631,717.0	5,367,142.6	1,057.8	358.5	125	-60	DDH
ATO221	631,619.0	5,367,211.2	1,053.5	80.1	125	-60	DDH
ATO222	631,694.3	5,367,157.0	1,056.5	164.1	125	-60	DDH
ATO223	631,326.1	5,367,046.8	1,036.8	150.0	125	-60	DDH
ATO224	631,953.5	5,367,009.8	1,054.2	93.4	125	-60	DDH
ATO225	631,955.3	5,367,013.1	1,054.5	358.4	125	-60	DDH
ATO226	631,956.0	5,367,695.6	1,036.5	151.6	270	-60	DDH
ATO227	631,310.2	5,367,205.1	1,043.8	149.2	125	-60	DDH
ATO228	631,614.2	5,367,104.5	1,057.7	149.2	125	-60	DDH
ATO229	631,963.9	5,367,184.5	1,049.8	149.3	125	-60	DDH
ATO23	631,841.0	5,367,054.1	1,062.9	123.4	125	-60	DDH
ATO230	631,453.5	5,367,659.0	1,027.9	152.1	125	-60	DDH
ATO231	631,893.4	5,367,697.6	1,034.6	200.1	270	-60	DDH
ATO232	628,459.7	5,364,649.0	999.8	144.1	70	-60	DDH
ATO233	629,987.0	5,363,690.1	1,016.4	152.1	0	-60	DDH
ATO234	631,935.1	5,366,877.1	1,050.7	100.0	125	-60	DDH
ATO235	631,983.5	5,366,842.0	1,049.1	101.1	125	-60	DDH
ATO236	631,929.2	5,366,916.7	1,052.7	101.1	125	-60	DDH
ATO237	631,875.9	5,366,845.5	1,049.3	90.6	125	-60	DDH
ATO238	631,789.4	5,366,875.7	1,051.2	100.0	125	-60	DDH
ATO239	632,233.8	5,366,885.9	1,055.0	90.6	125	-60	DDH
ATO24	631,793.3	5,367,088.3	1,062.1	239.2	125	-60	DDH
ATO240	631,932.1	5,367,030.2	1,055.4	381.6	125	-60	DDH
ATO241	631,876.6	5,367,460.4	1,039.0	716.0	215	-60	DDH
ATO242	631,912.8	5,366,895.1	1,051.9	116.1	125	-60	DDH
ATO243	631,520.2	5,367,064.5	1,052.2	110.1	125	-60	DDH
ATO244	631,614.0	5,366,992.9	1,053.8	221.1	125	-60	DDH
ATO245	631,453.2	5,367,181.7	1,056.6	215.1	125	-60	DDH
ATO246	632,114.6	5,367,110.3	1,048.0	440.2	215	-60	DDH
ATO247	631,893.9	5,366,872.7	1,050.8	110.1	125	-60	DDH
ATO248	631,907.6	5,367,046.7	1,056.7	422.1	125	-60	DDH
ATO249	631,622.2	5,366,951.7	1,051.0	200.1	125	-60	DDH
ATO25	631,803.4	5,367,003.0	1,065.3	118.2	125	-60	DDH

ATO250	631,628.1	5,366,910.1	1,049.6	142.9	125	-60	DDH
ATO251	631,705.9	5,366,859.4	1,048.6	107.1	125	-60	DDH
ATO252	631,747.9	5,366,867.6	1,049.9	80.1	125	-60	DDH
ATO253	631,837.9	5,366,839.3	1,049.1	101.1	125	-60	DDH
ATO254	631,643.5	5,367,086.7	1,058.0	230.1	125	-60	DDH
ATO255	631,889.0	5,366,914.0	1,053.1	131.1	125	-60	DDH
ATO256	632,011.3	5,366,827.0	1,048.8	81.6	125	-60	DDH
ATO257	631,872.7	5,366,888.4	1,051.8	125.1	125	-60	DDH
ATO258	631,921.9	5,366,853.8	1,049.7	92.1	125	-60	DDH
ATO259	631,952.6	5,366,903.8	1,051.8	116.1	125	-60	DDH
ATO26	631,756.3	5,367,039.9	1,066.3	214.8	125	-60	DDH
ATO260	632,212.0	5,367,120.6	1,051.2	200.2	125	-60	DDH
ATO261	632,261.1	5,367,086.7	1,054.2	200.1	125	-60	DDH
ATO262	631,906.7	5,367,153.8	1,054.1	200.1	125	-60	DDH
ATO263	631,571.6	5,366,878.2	1,044.2	200.1	125	-60	DDH
ATO264	631,620.4	5,366,840.0	1,044.9	200.1	125	-60	DDH
ATO265	631,142.0	5,367,365.0	1,034.3	200.1	125	-60	DDH
ATO266	632,170.8	5,367,221.8	1,047.0	338.1	125	-60	DDH
ATO267	630,029.3	5,368,865.2	1,004.6	76.2	0	-90	DDH
ATO268	629,511.9	5,368,020.3	1,000.1	88.2	0	-90	DDH
ATO269	629,510.7	5,367,130.9	1,000.0	46.2	0	-90	DDH
ATO27	631,725.0	5,366,993.4	1,067.7	198.0	125	-60	DDH
ATO270	629,494.0	5,366,124.7	994.1	34.2	0	-90	DDH
ATO271	629,538.2	5,365,306.1	992.6	52.2	0	-90	DDH
ATO272	629,116.5	5,365,005.1	991.4	61.2	0	-90	DDH
ATO273	629,100.1	5,365,747.8	993.8	50.2	0	-90	DDH
ATO274	629,085.4	5,366,594.1	995.6	95.2	0	-90	DDH
ATO275	629,072.4	5,367,570.9	998.0	55.2	0	-90	DDH
ATO276	629,174.3	5,368,848.1	1,000.6	55.2	0	-90	DDH
ATO277	631,183.1	5,364,996.9	1,026.0	50.2	0	-90	DDH
ATO278	631,289.7	5,364,995.4	1,029.9	62.2	0	-90	DDH
ATO279	630,806.3	5,366,890.4	1,013.8	50.0	0	-90	DDH
ATO28	631,689.7	5,366,938.9	1,053.7	193.8	125	-60	DDH
ATO280	631,006.9	5,366,893.8	1,018.3	49.2	0	-90	DDH
ATO281	630,997.8	5,366,799.7	1,017.9	49.2	0	-90	DDH
ATO282	630,896.6	5,366,799.1	1,015.3	49.6	0	-90	DDH
ATO283	630,795.6	5,366,798.6	1,014.1	31.2	0	-90	DDH
ATO284	630,810.7	5,366,992.6	1,014.0	49.2	0	-90	DDH
ATO285	630,902.0	5,366,996.5	1,016.1	52.2	0	-90	DDH
ATO286	630,999.4	5,366,995.1	1,019.6	40.2	0	-90	DDH
ATO287	630,107.0	5,366,998.1	1,006.3	46.2	0	-90	DDH
ATO288	630,400.9	5,367,200.4	1,008.2	46.2	0	-90	DDH
ATO289	630,198.9	5,367,202.4	1,007.6	41.4	0	-90	DDH
ATO29	631,738.5	5,366,908.1	1,052.6	112.8	125	-60	DDH
ATO290	630,000.9	5,367,200.1	1,005.6	34.2	0	-90	DDH

ATO291	632,459.2	5,366,470.3	1,063.5	271.6	305	-60	DDH
ATO292	632,180.2	5,366,659.6	1,057.8	200.1	305	-60	DDH
ATO293	632,225.0	5,367,190.0	1,049.0	277.7	125	-60	DDH
ATO294	631,700.2	5,366,670.4	1,052.0	293.0	125	-60	DDH
ATO295	631,987.5	5,366,950.5	1,052.3	224.4	125	-60	DDH
ATO296	631,967.4	5,366,967.6	1,053.2	299.0	125	-60	DDH
ATO297	632,010.2	5,366,934.1	1,051.3	254.0	125	-60	DDH
ATO298	631,975.0	5,366,885.5	1,050.3	167.0	125	-60	DDH
ATO299	631,996.5	5,367,017.3	1,052.3	392.0	125	-60	DDH
ATO30	631,569.2	5,367,245.9	1,054.1	172.8	35	-60	DDH
ATO300	632,034.7	5,366,916.7	1,050.8	199.0	125	-60	DDH
ATO301	632,021.0	5,367,000.0	1,051.6	302.0	125	-60	DDH
ATO302	631,999.5	5,366,868.1	1,049.7	161.0	125	-60	DDH
ATO303	632,059.2	5,366,899.4	1,050.4	169.0	125	-60	DDH
ATO304	632,045.5	5,366,982.6	1,051.1	271.0	125	-60	DDH
ATO305	632,083.7	5,366,882.1	1,050.5	158.0	125	-60	DDH
ATO306	631,936.8	5,366,986.1	1,053.9	443.0	125	-60	DDH
ATO307	631,972.0	5,367,034.7	1,053.1	452.0	125	-60	DDH
ATO308	632,031.7	5,367,065.9	1,050.6	407.0	125	-60	DDH
ATO309	632,069.9	5,366,965.3	1,050.9	278.0	125	-60	DDH
ATO31	631,817.2	5,367,070.5	1,062.0	142.7	125	-60	DDH
ATO310	632,094.4	5,366,948.0	1,050.8	209.0	125	-60	DDH
ATO311	632,056.2	5,367,048.6	1,050.3	272.0	126	-60	DDH
ATO312	632,118.9	5,366,930.6	1,051.1	182.0	126	-60	DDH
ATO313	632,115.9	5,367,079.8	1,048.8	200.0	125	-60	DDH
ATO314	632,080.7	5,367,031.2	1,050.2	269.0	125	-60	DDH
ATO315	632,143.4	5,366,913.3	1,051.6	137.0	125	-60	DDH
ATO316	632,167.8	5,366,896.0	1,052.4	110.0	125	-60	DDH
ATO317	631,947.4	5,367,052.2	1,054.4	506.0	125	-60	DDH
ATO318	632,105.2	5,367,013.9	1,050.3	188.0	125	-60	DDH
ATO319	632,140.4	5,367,062.5	1,049.4	179.0	125	-60	DDH
ATO32	631,674.2	5,367,024.2	1,058.9	298.7	125	-60	DDH
ATO320	631,249.9	5,367,363.8	1,040.3	242.0	125	-60	DDH
ATO321	632,164.9	5,367,045.1	1,050.3	176.0	125	-60	DDH
ATO322	632,129.7	5,366,996.6	1,050.8	170.0	125	-60	DDH
ATO323	632,486.5	5,366,450.0	1,064.9	206.0	125	-60	DDH
ATO324	632,189.4	5,367,027.8	1,051.4	173.0	125	-60	DDH
ATO325	632,154.1	5,366,979.2	1,051.4	149.0	125	-60	DDH
ATO326	632,091.4	5,367,097.2	1,048.5	341.0	125	-60	DDH
ATO327	631,502.9	5,367,256.5	1,052.9	251.0	125	-60	DDH
ATO328	631,923.0	5,367,069.3	1,055.4	479.0	125	-60	DDH
ATO329	632,066.9	5,367,114.5	1,048.3	308.0	125	-60	DDH
ATO33	631,865.0	5,367,037.0	1,060.3	84.3	125	-60	DDH
ATO330	631,527.4	5,367,239.1	1,056.7	197.0	125	-60	DDH
ATO331	631,538.1	5,367,305.1	1,046.5	200.0	125	-60	DDH

ATO332	631,562.6	5,367,287.7	1,048.5	209.0	125	-60	DDH
ATO333	631,513.8	5,367,322.3	1,044.4	47.0	125	-60	DDH
ATO334	631,587.1	5,367,270.4	1,049.5	50.0	125	-60	DDH
ATO335	631,611.6	5,367,253.0	1,050.3	50.0	125	-60	DDH
ATO336	631,478.4	5,367,273.7	1,049.5	50.0	125	-61	DDH
ATO337	631,551.9	5,367,221.8	1,059.2	56.0	125	-60	DDH
ATO338	631,576.4	5,367,204.4	1,059.9	62.0	125	-60	DDH
ATO339	631,505.9	5,367,107.3	1,058.3	62.0	125	-60	DDH
ATO34	631,767.1	5,367,106.6	1,062.6	220.8	125	-60	DDH
ATO340	631,481.4	5,367,124.7	1,058.2	56.0	125	-60	DDH
ATO341	631,457.0	5,367,141.9	1,056.6	50.0	125	-60	DDH
ATO342	631,432.4	5,367,159.2	1,053.9	40.0	125	-60	DDH
ATO343	631,467.8	5,367,207.9	1,057.2	53.0	125	-60	DDH
ATO344	631,541.1	5,367,155.8	1,068.2	53.3	125	-60	DDH
ATO345	631,492.1	5,367,190.6	1,062.4	64.0	125	-60	DDH
ATO346	631,516.6	5,367,173.2	1,065.0	59.0	125	-60	DDH
ATO347	632,205.0	5,367,270.0	1,048.3	357.0	125	-60	DDH
ATO348	632,251.0	5,367,174.0	1,051.0	269.0	125	-60	DDH
ATO349	632,275.6	5,367,156.9	1,051.0	239.0	125	-60	DDH
ATO35	631,780.5	5,367,022.3	1,069.2	151.5	125	-60	DDH
ATO350	631,410.1	5,366,842.0	1,033.0	407.0	70	-55	DDH
ATO351	632,254.0	5,367,234.0	1,051.0	299.0	125	-60	DDH
ATO352	632,302.0	5,367,200.0	1,067.0	329.0	125	-60	DDH
ATO353	631,600.0	5,367,186.0	1,057.0	53.0	125	-60	DDH
ATO354	632,284.0	5,367,284.0	1,055.0	302.0	125	-60	DDH
ATO355	631,869.0	5,366,737.0	1,043.0	221.0	125	-60	DDH
ATO356	631,699.0	5,367,116.5	1,058.6	332.0	125	-62	DDH
ATO357	631,910.6	5,367,001.6	1,055.8	311.0	125	-61	DDH
ATO358	632,177.5	5,366,960.9	1,052.2	119.0	125	-60	DDH
ATO359	632,146.5	5,367,240.0	1,045.7	326.0	125	-61	DDH
ATO36	631,829.7	5,366,986.7	1,059.9	90.8	125	-60	DDH
ATO360	632,227.9	5,367,035.4	1,052.7	128.0	125	-60	DDH
ATO361	632,006.2	5,367,082.4	1,051.3	221.5	125	-60	DDH
ATO362	632,102.0	5,367,198.2	1,046.3	321.9	125	-60	DDH
ATO363	631,886.9	5,367,018.3	1,057.7	275.0	125	-60	DDH
ATO364	631,681.7	5,367,163.7	1,056.0	386.0	125	-60	DDH
ATO365	632,076.1	5,367,215.9	1,045.6	350.0	125	-60	DDH
ATO366	632,201.9	5,366,943.0	1,053.0	92.0	125	-60	DDH
ATO367	632,121.4	5,367,148.1	1,047.0	293.0	125	-61	DDH
ATO368	632,194.3	5,367,096.4	1,051.0	197.0	125	-61	DDH
ATO369	631,656.9	5,367,181.3	1,054.0	440.0	125	-62	DDH
ATO37	631,742.2	5,366,977.2	1,068.4	129.4	125	-60	DDH
ATO371	632,154.0	5,367,199.7	1,047.0	305.0	125	-60	DDH
ATO372	632,178.5	5,367,182.2	1,048.0	281.0	125	-60	DDH
ATO373	632,203.1	5,367,165.1	1,049.0	263.0	125	-60	DDH

ATO374	632,227.6	5,367,146.8	1,051.0	221.0	125	-60	DDH
ATO375	632,096.9	5,367,165.5	1,046.7	296.0	125	-60	DDH
ATO376	632,215.6	5,367,227.6	1,049.0	305.0	125	-60	DDH
ATO377	632,169.9	5,367,113.6	1,048.8	212.0	125	-60	DDH
ATO378	632,241.0	5,367,211.0	1,050.9	278.0	125	-60	DDH
ATO379	631,674.2	5,367,134.1	1,057.0	401.0	125	-60	DDH2
ATO38	631,731.9	5,367,055.9	1,063.7	229.7	125	-60	DDH
ATO380	631,685.5	5,367,200.0	1,053.5	461.0	125	-60	DDH2
ATO381	631,644.0	5,367,119.0	1,057.5	390.0	125	-60	DDH2
ATO382	631,649.7	5,367,151.5	1,056.2	494.0	125	-60	DDH2
ATO39	631,642.6	5,366,973.0	1,053.2	202.8	125	-60	DDH
ATO397	632,910.0	5,365,936.0	1,054.0	400.0	125	-60	DDH2
ATO398	630,720.0	5,367,546.0	1,015.0	400.0	125	-60	DDH2
ATO399	631,405.2	5,367,215.9	1,049.5	296.0	125	-60	DDH2
ATO40	631,715.4	5,366,923.2	1,053.5	103.8	125	-60	DDH
ATO400	631,443.2	5,367,224.6	1,051.9	245.0	125	-60	DDH2
ATO41	631,770.4	5,366,954.0	1,062.5	229.6	305	-60	DDH
ATO42	631,871.5	5,367,106.3	1,057.0	94.4	125	-60	DDH
ATO43	631,756.4	5,367,042.2	1,066.2	85.9	305	-60	DDH
ATO44	631,699.9	5,367,009.8	1,061.6	234.3	125	-60	DDH
ATO45	631,856.1	5,366,970.8	1,056.9	208.6	125	-60	DDH
ATO46	631,830.1	5,366,988.5	1,060.4	299.8	305	-60	DDH
ATO47	631,795.7	5,366,939.4	1,057.0	51.1	125	-60	DDH
ATO48	631,728.2	5,366,997.6	1,068.1	94.8	305	-60	DDH
ATO49	631,669.1	5,366,956.2	1,053.6	144.7	125	-60	DDH
ATO50	631,739.7	5,366,910.4	1,052.8	199.8	305	-60	DDH
ATO51	631,485.0	5,367,232.3	1,058.0	220.0	125	-60	DDH
ATO52	631,825.5	5,367,137.6	1,056.8	205.8	125	-60	DDH
ATO53	631,560.7	5,367,182.0	1,063.4	140.5	125	-60	DDH
ATO54	631,850.8	5,367,120.8	1,057.2	150.2	125	-60	DDH
ATO55	631,520.9	5,367,281.0	1,051.4	214.5	125	-60	DDH
ATO56	631,799.7	5,367,154.0	1,056.2	219.3	125	-60	DDH
ATO57	631,571.5	5,367,243.4	1,054.2	130.0	125	-60	DDH
ATO58	631,901.3	5,367,012.1	1,057.1	232.8	305	-60	DDH
ATO59	631,580.0	5,367,314.6	1,046.5	255.0	125	-60	DDH
ATO60	631,478.1	5,367,163.7	1,060.5	180.0	125	-60	DDH
ATO61	631,898.3	5,367,083.8	1,056.3	204.0	305	-60	DDH
ATO62	631,774.2	5,366,951.5	1,062.6	502.0	305	-80	DDH
ATO63	631,670.0	5,366,920.9	1,051.8	134.3	125	-60	DDH
ATO64	631,810.3	5,367,115.8	1,059.1	170.3	125	-60	DDH
ATO65	631,627.3	5,367,273.5	1,048.8	96.2	125	-60	DDH
ATO66	631,857.7	5,367,082.1	1,059.9	98.3	125	-60	DDH
ATO67	631,526.5	5,367,125.0	1,060.3	89.3	125	-60	DDH
ATO68	631,444.1	5,367,114.0	1,049.9	246.2	125	-60	DDH
ATO69	631,729.7	5,366,946.6	1,057.9	116.3	125	-60	DDH

ATO70	631,545.2	5,367,262.9	1,053.7	215.3	125	-60	DDH
ATO71	632,028.7	5,366,955.8	1,051.5	200.3	125	-60	DDH
ATO72	631,410.5	5,367,066.4	1,044.3	200.3	125	-60	DDH
ATO73	632,106.2	5,366,989.1	1,051.1	177.7	215	-60	DDH
ATO74	631,508.9	5,367,214.6	1,065.6	200.3	125	-60	DDH
ATO75	631,680.7	5,366,981.7	1,056.9	215.3	125	-60	DDH
ATO76	631,502.8	5,367,146.6	1,064.7	141.3	125	-60	DDH
ATO77	631,773.5	5,367,063.7	1,064.4	200.4	125	-60	DDH
ATO78	631,467.9	5,367,097.0	1,053.0	149.3	125	-60	DDH
ATO79	631,592.8	5,367,228.9	1,053.7	116.3	125	-60	DDH
ATO80	631,764.2	5,366,998.7	1,070.5	140.2	125	-60	DDH
ATO81	631,602.8	5,367,295.3	1,047.3	146.3	125	-60	DDH
ATO82	631,553.7	5,367,330.0	1,045.4	200.3	125	-60	DDH
ATO83	631,714.9	5,367,030.5	1,064.9	257.3	125	-60	DDH
ATO84	631,586.4	5,367,160.9	1,058.4	89.3	125	-60	DDH
ATO85	631,652.2	5,367,258.8	1,049.9	50.3	125	-60	DDH
ATO86	631,549.3	5,367,109.0	1,056.7	50.3	125	-60	DDH
ATO87	631,781.8	5,367,130.6	1,058.6	276.8	125	-60	DDH
ATO88	631,646.1	5,366,932.1	1,051.4	146.3	125	-60	DDH
ATO89	631,695.8	5,366,897.4	1,051.2	95.3	125	-60	DDH
ATO90	631,655.1	5,367,000.3	1,056.3	251.3	125	-60	DDH
ATO91	631,829.9	5,367,098.8	1,060.0	146.3	125	-60	DDH
ATO92	632,004.5	5,366,975.4	1,052.2	301.8	125	-60	DDH
ATO93	631,705.4	5,366,964.5	1,059.2	131.3	125	-60	DDH
ATO94	632,076.2	5,366,924.8	1,051.0	101.3	125	-60	DDH
ATO95	631,753.8	5,366,929.6	1,055.6	82.0	125	-60	DDH
ATO96	632,076.9	5,366,926.6	1,051.1	261.5	305	-60	DDH
ATO97	631,690.2	5,367,050.6	1,061.2	296.3	125	-60	DDH
ATO98	631,749.3	5,367,081.5	1,064.8	266.1	125	-60	DDH
ATO99	631,090.8	5,367,416.1	1,028.1	197.3	125	-60	DDH
At diamond holes	**385.0**		**Total**	**73,133.8**	**m**		
			Av	**190.0**	**m**		
AT RC holes							
ATORC01	630,692.2	5,368,197.0	1,014.3	20.0	0	-90	RC
ATORC02	630,890.6	5,368,198.1	1,015.8	14.0	0	-90	RC
ATORC03	631,095.5	5,368,199.3	1,020.9	26.0	0	-90	RC
ATORC04	631,295.2	5,368,200.1	1,022.5	20.0	0	-90	RC
ATORC05	631,496.0	5,368,201.2	1,026.4	28.0	0	-90	RC
ATORC06	631,695.7	5,368,201.2	1,033.3	30.0	0	-90	RC
ATORC07	631,400.3	5,367,999.6	1,022.9	24.0	0	-90	RC
ATORC08	631,197.9	5,367,999.6	1,019.9	26.0	0	-90	RC
ATORC09	630,999.3	5,367,998.8	1,017.3	20.0	0	-90	RC
ATORC10	630,798.9	5,367,999.5	1,016.9	18.0	0	-90	RC
ATORC100	630,395.4	5,365,799.4	1,011.3	40.0	0	-90	RC

ATORC101	630,146.3	5,366,203.0	1,019.2	32.0	0	-90	RC
ATORC102	631,032.8	5,364,598.9	1,025.3	54.0	0	-90	RC
ATORC103	630,604.0	5,364,592.1	1,012.1	85.0	0	-90	RC
ATORC104	630,369.8	5,364,200.6	1,010.1	45.0	0	-90	RC
ATORC105	629,947.4	5,364,199.7	999.3	48.0	0	-90	RC
ATORC106	629,744.8	5,363,805.4	998.4	16.0	0	-90	RC
ATORC107	628,821.6	5,360,597.4	988.3	35.0	0	-90	RC
ATORC108	629,044.3	5,361,011.2	987.8	64.0	0	-90	RC
ATORC109	628,662.6	5,360,198.4	985.7	52.0	0	-90	RC
ATORC11	630,498.2	5,367,785.1	1,025.5	16.0	0	-90	RC
ATORC110	628,861.9	5,359,798.9	1,002.9	35.0	0	-90	RC
ATORC111	629,378.0	5,360,996.9	999.3	15.0	0	-90	RC
ATORC112	629,225.7	5,360,599.5	995.6	56.0	0	-90	RC
ATORC113	629,619.4	5,360,601.6	1,006.0	37.0	0	-90	RC
ATORC114	630,066.1	5,360,602.7	1,020.4	29.0	0	-90	RC
ATORC115	630,344.7	5,360,194.3	1,030.2	18.0	0	-90	RC
ATORC116	630,132.0	5,359,804.1	1,028.4	19.0	0	-90	RC
ATORC117	629,882.6	5,360,196.9	1,010.5	34.0	0	-90	RC
ATORC118	629,424.2	5,360,197.6	1,019.6	14.0	0	-90	RC
ATORC119	628,458.6	5,359,801.2	981.1	45.0	0	-90	RC
ATORC12	630,697.8	5,367,792.1	1,019.7	20.0	0	-90	RC
ATORC120	629,043.7	5,359,400.1	1,018.6	20.0	0	-90	RC
ATORC121	628,655.3	5,359,399.5	991.9	34.0	0	-90	RC
ATORC122	628,856.1	5,358,998.4	1,004.7	15.0	0	-90	RC
ATORC123	628,467.1	5,358,998.2	987.6	17.0	0	-90	RC
ATORC124	629,046.4	5,358,601.3	1,015.0	20.0	0	-90	RC
ATORC125	628,681.0	5,358,598.3	999.3	43.0	0	-90	RC
ATORC126	628,864.0	5,358,201.8	1,010.0	51.0	0	-90	RC
ATORC127	628,492.6	5,358,198.2	997.4	46.0	0	-90	RC
ATORC128	627,557.0	5,358,404.8	977.9	14.0	0	-90	RC
ATORC129	627,295.0	5,358,027.8	971.7	16.0	0	-90	RC
ATORC13	630,897.4	5,367,793.8	1,017.2	22.0	0	-90	RC
ATORC130	627,418.1	5,357,803.1	974.8	23.0	0	-90	RC
ATORC131	627,989.1	5,357,800.8	994.6	19.0	0	-90	RC
ATORC132	628,338.1	5,357,804.8	994.1	39.0	0	-90	RC
ATORC133	628,515.6	5,357,398.8	1,004.6	29.0	0	-90	RC
ATORC134	628,072.0	5,357,397.0	989.1	57.0	0	-90	RC
ATORC135	628,302.3	5,357,001.2	996.6	53.0	0	-90	RC
ATORC136	628,104.1	5,356,601.7	988.8	55.0	0	-90	RC
ATORC137	627,916.6	5,356,201.4	985.1	62.0	0	-90	RC
ATORC138	627,723.6	5,355,797.9	980.1	74.0	0	-90	RC
ATORC139	627,548.8	5,355,397.8	969.8	48.0	0	-90	RC
ATORC14	631,094.9	5,367,794.6	1,020.3	24.0	0	-90	RC
ATORC140	626,234.9	5,355,401.3	967.2	40.0	0	-90	RC
ATORC141	625,844.1	5,355,406.1	971.3	37.0	0	-90	RC

ATORC142	625,709.4	5,356,004.6	970.2	25.0	0	-90	RC
ATORC143	625,504.3	5,356,396.3	972.5	26.0	0	-90	RC
ATORC144	627,437.9	5,359,399.7	972.9	20.0	0	-90	RC
ATORC145	627,236.1	5,359,803.9	975.3	27.0	0	-90	RC
ATORC146	627,647.6	5,359,796.8	994.4	15.0	0	-90	RC
ATORC147	627,829.5	5,360,199.3	990.6	17.0	0	-90	RC
ATORC148	628,198.5	5,363,798.0	989.8	32.0	0	-90	RC
ATORC149	627,970.7	5,364,194.1	994.9	26.0	0	-90	RC
ATORC15	631,296.9	5,367,793.1	1,023.3	24.0	0	-90	RC
ATORC150	628,236.5	5,364,596.3	998.6	41.0	0	-90	RC
ATORC151	630,456.7	5,368,597.8	1,011.3	29.0	0	-90	RC
ATORC152	630,887.2	5,368,600.5	1,017.9	27.0	0	-90	RC
ATORC153	631,288.6	5,368,598.9	1,023.9	20.0	0	-90	RC
ATORC154	631,697.0	5,368,597.1	1,035.0	13.0	0	-90	RC
ATORC155	631,510.4	5,369,000.7	1,034.8	16.0	0	-90	RC
ATORC156	631,687.4	5,369,203.0	1,038.6	13.0	0	-90	RC
ATORC157	631,349.7	5,369,192.5	1,033.3	27.0	0	-90	RC
ATORC158	631,784.8	5,367,718.2	1,031.6	180.0	129	-64	RC
ATORC159	632,175.1	5,367,331.5	1,046.2	204.0	138	-63	RC
ATORC16	631,497.8	5,367,793.8	1,025.3	24.0	0	-90	RC
ATORC160	632,223.3	5,367,296.8	1,049.6	199.0	129	-60	RC
ATORC161	632,419.3	5,367,157.4	1,063.7	200.0	121	-60	RC
ATORC162	632,370.1	5,367,192.2	1,060.4	200.0	114	-61	RC
ATORC163	632,321.8	5,367,224.9	1,057.2	199.0	119	-60	RC
ATORC164	632,271.8	5,367,261.8	1,053.4	200.0	124	-61	RC
ATORC165	632,127.9	5,367,474.2	1,043.2	199.0	120	-60	RC
ATORC166	632,030.0	5,367,544.1	1,038.5	137.0	119	-61	RC
ATORC167	631,932.0	5,367,613.1	1,034.9	165.0	127	-60	RC
ATORC168	631,882.5	5,367,648.1	1,034.0	200.0	126	-60	RC
ATORC169	631,834.2	5,367,682.8	1,032.9	155.0	123	-60	RC
ATORC17	631,697.7	5,367,795.0	1,029.5	32.0	0	-90	RC
ATORC170	632,124.9	5,367,363.1	1,043.1	200.0	125	-61	RC
ATORC171	631,342.9	5,366,964.7	1,034.4	100.0	123	-61	RC
ATORC172	631,391.8	5,366,929.5	1,036.1	100.0	122	-60	RC
ATORC173	631,001.0	5,366,995.8	1,019.6	22.0	0	-90	RC
ATORC174	630,907.8	5,366,894.0	1,015.9	70.0	0	-90	RC
ATORC175	630,811.6	5,366,994.0	1,014.0	37.0	0	-90	RC
ATORC176	630,615.5	5,366,993.6	1,010.9	32.0	0	-90	RC
ATORC177	630,500.5	5,367,110.3	1,009.4	56.0	0	-90	RC
ATORC178	630,305.7	5,367,108.7	1,007.2	65.0	0	-90	RC
ATORC179	630,296.7	5,367,286.3	1,009.0	56.0	0	-90	RC
ATORC18	632,091.8	5,367,793.9	1,042.6	20.0	0	-90	RC
ATORC180	630,496.3	5,367,296.9	1,009.3	47.0	0	-90	RC
ATORC181	630,385.8	5,367,784.1	1,030.3	17.0	0	-90	RC
ATORC182	630,495.0	5,367,699.4	1,022.9	35.0	0	-90	RC

ATORC183	631,398.3	5,367,310.9	1,043.6	16.0	0	-90	RC
ATORC184	631,308.7	5,367,398.5	1,037.5	16.0	0	-90	RC
ATORC185	631,500.0	5,367,403.4	1,032.3	49.0	0	-90	RC
ATORC186	631,398.1	5,367,500.0	1,039.3	36.0	0	-90	RC
ATORC187	631,999.1	5,367,498.5	1,037.2	42.0	0	-90	RC
ATORC188	632,104.4	5,367,600.1	1,040.7	64.0	0	-90	RC
ATORC189	631,702.4	5,367,699.9	1,029.3	18.0	0	-90	RC
ATORC19	632,291.5	5,367,790.0	1,048.3	16.0	0	-90	RC
ATORC190	631,788.2	5,367,792.3	1,032.5	14.0	0	-90	RC
ATORC191	631,688.2	5,367,901.6	1,030.5	10.0	0	-90	RC
ATORC192	631,602.0	5,367,788.8	1,027.0	18.0	0	-90	RC
ATORC193	631,496.8	5,367,902.2	1,024.7	19.0	0	-90	RC
ATORC194	631,305.2	5,367,901.5	1,022.3	30.0	0	-90	RC
ATORC195	631,300.9	5,368,099.7	1,021.5	26.0	0	-90	RC
ATORC196	631,498.1	5,368,102.2	1,025.8	28.0	0	-90	RC
ATORC197	631,599.7	5,367,997.1	1,028.1	19.0	0	-90	RC
ATORC198	632,290.4	5,367,099.0	1,054.7	66.0	0	-90	RC
ATORC199	632,500.8	5,367,096.9	1,068.7	36.0	0	-90	RC
ATORC20	632,201.2	5,367,600.9	1,045.9	36.0	0	-90	RC
ATORC200	632,595.4	5,366,999.3	1,077.9	46.0	0	-90	RC
ATORC201	632,398.9	5,366,997.7	1,061.7	38.0	0	-90	RC
ATORC202	632,304.6	5,366,900.7	1,057.5	32.0	0	-90	RC
ATORC203	632,519.5	5,366,902.3	1,072.8	28.0	0	-90	RC
ATORC204	632,315.1	5,366,499.8	1,062.9	40.0	0	-90	RC
ATORC205	632,519.9	5,366,501.9	1,068.3	42.0	0	-90	RC
ATORC206	632,410.3	5,366,397.3	1,063.2	44.0	0	-90	RC
ATORC207	632,195.6	5,366,396.8	1,072.2	56.0	0	-90	RC
ATORC208	632,318.8	5,366,301.4	1,073.5	42.0	0	-90	RC
ATORC209	632,531.1	5,366,298.5	1,062.8	40.0	0	-90	RC
ATORC21	632,002.0	5,367,599.6	1,036.8	28.0	0	-90	RC
ATORC210	632,420.2	5,366,200.3	1,069.8	72.0	0	-90	RC
ATORC211	632,542.1	5,366,095.2	1,062.7	48.0	0	-90	RC
ATORC212	632,305.4	5,367,600.7	1,051.8	50.0	0	-90	RC
ATORC213	632,201.8	5,367,500.1	1,047.7	46.0	0	-90	RC
ATORC214	632,070.1	5,367,699.8	1,040.0	46.0	0	-90	RC
ATORC215	631,964.7	5,367,802.0	1,038.8	40.0	0	-90	RC
ATORC216	631,874.9	5,367,898.3	1,037.8	56.0	0	-90	RC
ATORC217	632,055.8	5,367,900.0	1,046.7	48.0	0	-90	RC
ATORC218	631,956.1	5,367,998.8	1,046.4	36.0	0	-90	RC
ATORC219	631,840.6	5,368,096.3	1,040.9	54.0	0	-90	RC
ATORC22	631,799.9	5,367,599.0	1,031.9	22.0	0	-90	RC
ATORC220	631,926.0	5,368,200.9	1,049.6	40.0	0	-90	RC
ATORC221	632,308.2	5,366,098.8	1,071.4	46.0	0	-90	RC
ATORC222	632,434.4	5,366,002.3	1,069.9	44.0	0	-90	RC
ATORC223	629,514.2	5,360,602.0	1,001.8	29.0	0	-90	RC

ATORC224	629,719.3	5,360,596.5	1,009.5	28.0	0	-90	RC
ATORC225	629,821.6	5,360,598.6	1,012.6	29.0	0	-90	RC
ATORC226	628,963.7	5,359,797.8	1,011.7	28.0	0	-90	RC
ATORC227	628,763.6	5,358,198.2	1,007.5	46.0	0	-90	RC
ATORC23	631,601.3	5,367,600.7	1,030.9	36.0	0	-90	RC
ATORC24	631,400.9	5,367,599.7	1,028.1	26.0	0	-90	RC
ATORC25	631,200.8	5,367,599.6	1,025.1	20.0	0	-90	RC
ATORC26	631,001.7	5,367,601.4	1,019.7	22.0	0	-90	RC
ATORC27	630,800.1	5,367,598.6	1,015.8	20.0	0	-90	RC
ATORC28	630,599.7	5,367,600.0	1,016.6	24.0	0	-90	RC
ATORC29	630,401.0	5,367,599.7	1,018.9	24.0	0	-90	RC
ATORC30	630,201.4	5,367,599.4	1,019.6	26.0	0	-90	RC
ATORC31	630,091.2	5,367,386.3	1,013.9	24.0	0	-90	RC
ATORC32	630,289.7	5,367,388.0	1,011.4	22.0	0	-90	RC
ATORC33	630,490.6	5,367,387.2	1,010.2	16.0	0	-90	RC
ATORC34	630,903.0	5,367,386.6	1,018.3	22.0	0	-90	RC
ATORC35	631,396.3	5,367,398.3	1,037.5	28.0	0	-90	RC
ATORC36	631,799.3	5,367,400.6	1,041.5	30.0	0	-90	RC
ATORC37	631,694.0	5,367,344.1	1,044.4	26.0	0	-90	RC
ATORC38	631,797.0	5,367,287.5	1,048.0	32.0	0	-90	RC
ATORC39	631,895.5	5,367,233.1	1,050.1	30.0	0	-90	RC
ATORC40	631,894.0	5,367,353.2	1,043.9	26.0	0	-90	RC
ATORC41	632,003.7	5,367,291.5	1,044.1	30.0	0	-90	RC
ATORC42	632,301.2	5,367,234.5	1,055.4	18.0	0	-90	RC
ATORC43	632,296.7	5,367,351.7	1,055.9	34.0	0	-90	RC
ATORC44	632,199.8	5,367,400.7	1,047.7	53.0	0	-90	RC
ATORC45	632,003.5	5,367,398.8	1,040.1	32.0	0	-90	RC
ATORC46	632,096.3	5,367,353.9	1,042.0	44.0	0	-90	RC
ATORC47	632,202.8	5,367,292.2	1,048.1	40.0	0	-90	RC
ATORC48	632,202.2	5,367,174.7	1,049.2	54.0	0	-90	RC
ATORC49	632,102.0	5,367,234.5	1,044.8	46.0	0	-90	RC
ATORC50	632,003.2	5,367,176.7	1,048.1	24.0	0	-90	RC
ATORC51	632,594.1	5,366,797.6	1,082.0	18.0	0	-90	RC
ATORC52	632,599.9	5,366,599.3	1,077.8	18.0	0	-90	RC
ATORC53	632,509.9	5,366,702.0	1,069.8	32.0	0	-90	RC
ATORC54	632,399.3	5,366,802.2	1,062.6	32.0	0	-90	RC
ATORC55	632,400.7	5,366,599.6	1,062.1	42.0	0	-90	RC
ATORC56	632,081.7	5,366,951.2	1,051.0	30.0	0	-90	RC
ATORC57	632,156.1	5,367,058.2	1,049.7	32.0	0	-90	RC
ATORC58	631,710.1	5,367,595.2	1,031.5	24.0	0	-90	RC
ATORC59	631,655.1	5,367,499.8	1,035.4	30.0	0	-90	RC
ATORC60	631,556.9	5,367,498.5	1,035.1	28.0	0	-90	RC
ATORC61	631,399.5	5,366,999.6	1,039.5	26.0	0	-90	RC
ATORC62	631,598.4	5,366,800.4	1,042.1	30.0	0	-90	RC
ATORC63	631,797.0	5,366,801.4	1,046.2	24.0	0	-90	RC

ATORC64	631,997.9	5,366,802.2	1,047.8	20.0	0	-90	RC
ATORC65	632,198.1	5,366,803.0	1,052.9	46.0	0	-90	RC
ATORC66	632,311.9	5,366,702.6	1,056.5	30.0	0	-90	RC
ATORC67	632,200.3	5,366,601.0	1,063.9	42.0	0	-90	RC
ATORC68	631,904.5	5,366,593.0	1,043.5	18.0	0	-90	RC
ATORC69	631,702.2	5,366,593.8	1,036.0	22.0	0	-90	RC
ATORC70	631,503.8	5,366,595.2	1,030.8	18.0	0	-90	RC
ATORC71	631,398.6	5,366,401.3	1,026.8	20.0	0	-90	RC
ATORC72	631,198.8	5,366,401.3	1,027.8	22.0	0	-90	RC
ATORC73	631,303.8	5,366,590.8	1,024.2	18.0	0	-90	RC
ATORC74	631,101.2	5,366,594.5	1,019.5	20.0	0	-90	RC
ATORC75	631,199.1	5,366,802.1	1,024.1	22.0	0	-90	RC
ATORC76	630,999.8	5,366,800.4	1,018.0	22.0	0	-90	RC
ATORC77	630,797.1	5,366,799.9	1,014.1	18.0	0	-90	RC
ATORC78	630,598.5	5,366,801.9	1,015.4	16.0	0	-90	RC
ATORC79	630,903.9	5,366,997.8	1,016.1	22.0	0	-90	RC
ATORC80	630,705.0	5,366,999.5	988.9	22.0	0	-90	RC
ATORC81	630,504.1	5,366,998.2	1,009.8	10.0	0	-90	RC
ATORC82	630,305.0	5,366,997.2	1,008.5	12.0	0	-90	RC
ATORC83	630,797.4	5,367,199.9	1,013.6	22.0	0	-90	RC
ATORC84	630,599.3	5,367,201.5	1,009.6	22.0	0	-90	RC
ATORC85	630,398.9	5,367,200.0	1,008.2	14.0	0	-90	RC
ATORC86	630,196.6	5,367,202.2	1,007.7	16.0	0	-90	RC
ATORC87	629,998.7	5,367,200.0	1,005.6	12.0	0	-90	RC
ATORC88	630,691.6	5,367,388.0	1,011.8	20.0	0	-90	RC
ATORC89	630,104.7	5,366,997.7	1,006.5	16.0	0	-90	RC
ATORC90	629,904.0	5,366,996.9	1,004.5	12.0	0	-90	RC
ATORC91	631,676.8	5,364,999.9	1,036.3	38.0	0	-90	RC
ATORC92	631,236.5	5,364,996.5	1,028.1	24.0	0	-90	RC
ATORC93	630,828.2	5,365,000.4	1,015.2	45.0	0	-90	RC
ATORC94	630,395.4	5,365,002.3	1,006.1	68.0	0	-90	RC
ATORC95	629,966.6	5,365,005.0	999.4	82.0	0	-90	RC
ATORC96	628,110.9	5,365,004.9	993.9	74.0	0	-90	RC
ATORC97	631,023.6	5,365,391.5	1,029.7	22.0	0	-90	RC
ATORC98	630,617.4	5,365,399.4	1,012.5	43.0	0	-90	RC
ATORC99	630,199.1	5,365,395.5	1,002.1	51.0	0	-90	RC
AT RC holes	**227.0**		**Total**	**9,441.0**	**m**		
			Av	**41.6**	**m**		
Mungu diamond holes							
MG01	632,483.0	5,367,706.0	1,060.3	399.8	125	-60	DDH
MG02	632,430.3	5,367,743.6	1,057.1	501.4	125	-60	DDH
MG03	632,532.8	5,367,670.2	1,064.3	395.1	125	-60	DDH
MG04	632,446.0	5,367,659.1	1,059.3	370.2	125	-60	DDH
MG05	632,517.6	5,367,754.7	1,059.4	401.3	125	-60	DDH

MG06	632,408.0	5,367,610.5	1,058.4	300.5	125	-60	DDH
MG07	632,548.0	5,367,804.3	1,058.4	401.1	125	-60	DDH
MG08	632,567.9	5,367,719.9	1,063.8	400.8	125	-60	DDH
MG09	632,779.7	5,367,493.1	1,098.6	434.1	305	-60	DDH
MG10	632,544.4	5,367,476.3	1,079.4	170.2	305	-60	DDH
MG100	632,409.7	5,367,389.2	1,067.4	53.0	307	-62	DDH
MG101	632,427.9	5,367,373.0	1,070.3	47.0	305	-61	DDH
MG102	632,609.3	5,367,576.7	1,074.2	80.0	308	-62	DDH
MG103	632,596.8	5,367,548.7	1,075.8	113.0	304	-60	DDH
MG104	632,567.7	5,367,531.7	1,075.7	122.0	307	-62	DDH
MG105	632,395.2	5,367,360.3	1,066.9	31.0	306	-60	DDH
MG106	632,422.0	5,367,345.4	1,071.1	40.0	305	-61	DDH
MG107	632,438.0	5,367,334.8	1,074.5	52.0	303	-59	DDH
MG108	632,632.7	5,367,595.0	1,074.3	71.0	308	-61	DDH
MG109	632,455.3	5,367,323.2	1,076.4	68.0	304	-60	DDH
MG11	632,584.3	5,367,525.7	1,077.5	251.2	305	-60	DDH
MG110	632,543.5	5,367,514.2	1,075.4	107.0	307	-63	DDH
MG111	632,522.3	5,367,492.5	1,075.0	95.0	307	-62	DDH
MG112	632,500.6	5,367,472.1	1,073.8	80.0	308	-61	DDH
MG113	632,484.7	5,367,446.3	1,073.7	77.0	308	-61	DDH
MG114	632,461.3	5,367,421.6	1,072.5	100.0	308	-60	DDH
MG115	632,423.0	5,367,525.0	1,063.0	302.0	125	-60	DDH
MG116	632,546.0	5,367,955.0	1,062.6	368.0	125	-60	DDH
MG117	632,497.6	5,367,991.2	1,062.0	346.0	125	-60	DDH
MG118	632,449.0	5,367,470.0	1,062.0	230.0	125	-60	DDH
MG119	632,475.0	5,367,563.0	1,065.0	291.0	125	-60	DDH
MG12	632,509.4	5,367,425.8	1,079.5	182.0	305	-60	DDH
MG120	632,486.4	5,367,668.3	1,061.6	356.0	125	-61	DDH
MG121	632,436.0	5,367,702.7	1,057.9	329.0	125	-60	DDH
MG122	632,538.0	5,367,777.0	1,058.7	401.0	125	-60	DDH
MG123	632,487.4	5,367,814.0	1,056.5	380.0	125	-60	DDH
MG124	632,486.3	5,367,519.3	1,070.7	146.0	125	-60	DDH
MG125	632,526.7	5,367,712.0	1,062.3	290.0	125	-55	DDH
MG126	632,477.9	5,367,746.6	1,058.2	404.0	125	-60	DDH
MG127	632,432.0	5,367,631.0	1,059.0	386.0	125	-60	DDH
MG128	632,566.1	5,367,829.9	1,061.8	408.0	125	-60	DDH
MG129	632,561.8	5,367,870.0	1,061.0	335.0	125	-60	DDH
MG13	632,752.6	5,367,513.1	1,091.7	371.2	305	-60	DDH
MG130	632,553.3	5,367,912.6	1,060.9	348.0	125	-60	DDH
MG131	632,498.0	5,367,620.0	1,063.9	296.0	125	-60	DDH
MG132	632,448.2	5,367,509.0	1,066.5	335.0	125	-60	DDH
MG133	632,488.3	5,368,204.1	1,062.7	287.0	125	-50	DDH
MG134	632,352.2	5,367,369.1	1,062.7	300.0	125	-60	DDH
MG135	632,595.0	5,367,920.1	1,063.0	305.0	125	-60	DDH
MG136	632,566.0	5,367,975.4	1,066.0	332.0	125	-60	DDH

MG137	632,562.8	5,367,759.3	1,061.3	410.0	125	-60	DDH2
MG138	632,472.0	5,367,694.8	1,060.0	453.0	125	-60	DDH2
MG139	632,452.8	5,367,670.6	1,059.5	452.0	125	-60	DDH2
MG14	632,464.7	5,367,352.4	1,076.7	120.0	270	-45	DDH
MG140	632,451.9	5,367,636.6	1,060.4	305.0	125	-60	DDH2
MG141	632,470.3	5,367,606.3	1,062.8	375.6	125	-60	DDH2
MG142	632,504.8	5,367,762.4	1,058.4	392.0	125	-60	DDH2
MG143	632,524.4	5,367,787.7	1,058.0	431.0	125	-60	DDH2
MG144	632,508.0	5,367,816.5	1,057.0	443.0	125	-60	DDH2
MG145	632,576.6	5,367,895.4	1,062.6	314.0	125	-60	DDH2
MG146	632,485.6	5,367,923.3	1,058.7	470.0	125	-60	DDH2
MG147	632,509.7	5,367,649.6	1,063.6	287.0	125	-60	DDH2
MG148	632,528.5	5,367,929.9	1,060.7	452.0	125	-60	DDH2
MG149	632,522.2	5,367,973.1	1,062.1	389.0	125	-60	DDH2
MG15	632,509.4	5,367,685.8	1,062.6	420.0	125	-60	DDH
MG150	632,542.0	5,367,993.9	1,063.7	381.0	125	-60	DDH2
MG151	632,478.3	5,367,855.4	1,056.1	446.0	125	-60	DDH2
MG152	632,485.0	5,367,888.2	1,057.4	473.0	125	-60	DDH2
MG153	632,473.1	5,368,008.9	1,060.9	560.0	125	-60	DDH2
MG154	632,549.4	5,367,841.9	1,060.1	470.0	125	-60	DDH2
MG155	632,402.9	5,367,359.3	1,067.8	207.0	125	-60	DDH2
MG156	632,461.5	5,367,941.3	1,058.1	491.0	125	-60	DDH2
MG157	632,460.0	5,367,703.9	1,059.5	419.0	125	-60	DDH2
MG158	632,427.2	5,367,341.9	1,071.4	161.0	125	-60	DDH2
MG159	632,441.2	5,367,680.5	1,058.7	329.0	125	-60	DDH2
MG16	632,474.1	5,367,638.1	1,061.7	266.2	125	-60	DDH
MG160	632,465.8	5,367,663.0	1,060.5	332.5	125	-60	DDH2
MG161	632,439.7	5,367,645.3	1,059.3	293.0	125	-60	DDH2
MG162	632,549.4	5,367,878.9	1,060.3	478.0	125	-60	DDH2
MG163	632,390.0	5,367,290.0	1,065.7	257.0	125	-60	DDH2
MG164	632,555.0	5,368,020.0	1,066.9	500.0	125	-60	DDH2
MG165	632,376.7	5,367,374.2	1,064.5	201.0	125	-60	DDH2
MG17	632,541.4	5,367,737.2	1,061.4	407.2	125	-60	DDH
MG18	632,570.2	5,367,460.5	1,082.0	161.2	305	-60	DDH
MG19	632,601.3	5,367,511.6	1,079.8	179.2	125	-60	DDH
MG20	632,492.5	5,367,772.0	1,057.3	401.1	125	-60	DDH
MG21	632,424.7	5,367,673.0	1,057.8	314.1	125	-60	DDH
MG22	632,457.0	5,367,613.4	1,061.4	302.1	125	-60	DDH
MG23	632,527.0	5,367,820.7	1,057.4	401.1	125	-60	DDH
MG24	632,467.6	5,367,789.1	1,055.8	371.1	125	-60	DDH
MG25	632,535.1	5,367,884.7	1,059.5	371.1	125	-60	DDH
MG26	632,423.5	5,367,562.5	1,061.4	302.1	125	-60	DDH
MG27	632,570.0	5,367,936.6	1,062.3	374.1	125	-60	DDH
MG28	632,511.1	5,367,904.7	1,058.3	389.3	125	-60	DDH
MG29	632,500.6	5,367,834.9	1,056.5	448.0	125	-60	DDH

MG30	632,461.6	5,367,536.6	1,066.8	419.7	125	-60	DDH
MG31	632,451.0	5,367,580.8	1,063.4	371.5	125	-60	DDH
MG32	632,458.7	5,367,721.1	1,059.3	328.2	125	-60	DDH
MG33	632,478.6	5,367,593.8	1,064.1	302.7	125	-60	DDH
MG34	632,627.1	5,367,492.1	1,083.7	290.5	305	-60	DDH
MG35	632,786.5	5,367,563.1	1,096.8	398.1	305	-60	DDH
MG36	632,715.4	5,367,462.9	1,089.9	355.6	305	-45	DDH
MG37	632,744.6	5,367,659.1	1,080.8	371.0	305	-60	DDH
MG38	633,000.0	5,367,900.0	1,080.0	419.1	305	-60	DDH
MG49	632,459.6	5,367,682.1	1,059.9	380.4	125	-60	DDH
MG50	632,531.2	5,367,849.8	1,057.8	434.4	125	-60	DDH
MG51	632,500.1	5,367,726.0	1,060.3	458.0	125	-60	DDH
MG52	632,511.7	5,367,795.6	1,057.0	485.0	125	-60	DDH
MG53	632,504.7	5,367,863.4	1,057.2	476.0	125	-60	DDH
MG54	632,505.7	5,367,505.0	1,071.8	68.0	305	-58	DDH
MG55	632,492.1	5,367,514.4	1,069.9	47.0	305	-61	DDH
MG56	632,477.5	5,367,525.6	1,068.1	41.0	305	-60	DDH
MG57	632,459.5	5,367,537.4	1,066.5	53.0	304	-60	DDH
MG58	632,528.5	5,367,525.3	1,072.8	56.0	303	-59	DDH
MG59	632,514.5	5,367,533.7	1,070.5	83.0	305	-60	DDH
MG60	632,499.4	5,367,545.3	1,068.5	38.0	305	-60	DDH
MG61	632,481.6	5,367,558.6	1,066.8	104.0	307	-59	DDH
MG62	632,549.7	5,367,546.2	1,072.6	104.0	304	-60	DDH
MG63	632,536.3	5,367,556.6	1,070.5	80.0	308	-62	DDH
MG64	632,520.6	5,367,568.8	1,068.4	32.0	307	-61	DDH
MG65	632,503.9	5,367,580.4	1,066.5	89.8	302	-62	DDH
MG66	632,570.3	5,367,568.1	1,072.3	113.0	302	-60	DDH
MG67	632,549.0	5,367,585.6	1,069.3	137.0	304	-62	DDH
MG68	632,534.1	5,367,596.6	1,067.5	110.0	306	-59	DDH
MG69	632,518.0	5,367,609.1	1,065.7	56.0	305	-61	DDH
MG70	632,593.5	5,367,588.7	1,072.3	116.0	303	-63	DDH
MG71	632,579.1	5,367,598.1	1,070.6	95.0	306	-61	DDH
MG72	632,564.4	5,367,608.3	1,069.0	57.0	304	-62	DDH
MG73	632,542.9	5,367,623.1	1,066.8	65.0	302	-59	DDH
MG74	632,597.1	5,367,622.8	1,070.5	104.0	308	-60	DDH
MG75	632,582.0	5,367,633.9	1,068.9	90.0	304	-61	DDH
MG76	632,564.3	5,367,645.7	1,067.1	48.0	306	-60	DDH
MG77	632,619.7	5,367,645.5	1,070.4	44.0	308	-62	DDH
MG78	632,484.1	5,367,485.2	1,070.7	65.0	305	-63	DDH
MG79	632,470.6	5,367,493.2	1,068.9	44.0	302	-60	DDH
MG80	632,453.5	5,367,503.9	1,066.7	40.0	305	-59	DDH
MG81	632,437.9	5,367,514.3	1,064.8	39.1	307	-61	DDH
MG82	632,597.8	5,367,661.2	1,068.1	62.0	302	-60	DDH
MG83	632,638.5	5,367,666.5	1,071.0	35.0	306	-62	DDH
MG84	632,583.9	5,367,558.0	1,074.4	131.0	304	-61	DDH

MG85	632,621.4	5,367,681.2	1,069.1	28.0		303	-62	DDH
MG86	632,624.5	5,367,712.5	1,067.5	35.0		307	-60	DDH
MG87	632,642.2	5,367,702.6	1,069.3	17.0		304	-59	DDH
MG88	632,611.3	5,367,610.7	1,072.0	83.0		306	-61	DDH
MG89	632,420.0	5,367,492.1	1,064.2	36.5		302	-62	DDH
MG90	632,436.5	5,367,478.8	1,066.4	62.0		305	-61	DDH
MG91	632,451.6	5,367,468.5	1,068.2	44.0		308	-59	DDH
MG92	632,466.5	5,367,458.3	1,070.4	47.0		304	-62	DDH
MG93	632,412.9	5,367,459.2	1,064.7	50.0		303	-61	DDH
MG94	632,428.8	5,367,445.4	1,066.8	49.0		307	-59	DDH
MG95	632,445.2	5,367,433.1	1,069.9	88.0		306	-62	DDH
MG96	632,397.0	5,367,432.7	1,064.3	44.0		301	-59	DDH
MG97	632,414.1	5,367,420.8	1,066.5	53.0		304	-61	DDH
MG98	632,431.3	5,367,408.9	1,069.0	41.0		305	-63	DDH
MG99	632,395.9	5,367,401.2	1,065.7	41.0		303	-60	DDH
Mungu diamond	**155.0**		**Total**	**37,745.5**	**m**			
holes			**Av**	**243.5**	**m**			
Trenches								
ATOTR01	631,915.2	5,367,187.2	1,049.0	114.0		221	-2	TR
ATOTR02	632,122.2	5,367,015.8	1,048.3	80.0		218	-6	TR
ATOTR03	631,943.5	5,367,026.7	1,052.3	89.0		269	-2	TR
ATOTR04	631,034.8	5,367,324.8	1,023.3	80.0		264	-4	TR
ATOTR05	631,481.3	5,367,319.5	1,041.9	51.0		214	-3	TR
ATOTR06	631,114.7	5,367,284.4	1,030.4	48.0		220	-2	TR
ATOTR07	631,208.2	5,367,398.2	1,036.8	50.0		221	-1	TR
ATOTR08	631,530.3	5,366,569.8	1,027.7	58.6		219	-3	TR
ATOTR09	631,886.2	5,367,208.2	1,050.9	45.0		269	-1	TR
ATOTR10	631,698.8	5,367,011.0	1,058.8	93.0		300	-3	TR
ATOTR11	631,693.6	5,366,927.1	1,049.5	52.0		211	-5	TR
ATOTR110	632,390.5	5,364,409.0	1,078.4	54.2		347	-6	TR
ATOTR110add	632,398.1	5,364,374.8	1,083.1	34.5		348	-9	TR
ATOTR111	632,211.2	5,364,341.0	1,091.3	76.0		347	-14	TR
ATOTR112	632,452.3	5,364,301.7	1,086.7	42.0		147	-12	TR
ATOTR116	632,063.2	5,366,069.0	1,050.0	19.0		83	-4	TR
ATOTR117	632,442.3	5,365,050.0	1,099.7	105.0		4	-11	TR
ATOTR117add	632,443.5	5,365,049.5	1,098.6	56.0		181	-7	TR
ATOTR118	628,456.1	5,364,638.8	1,001.0	30.0		82	-2	TR
ATOTR119	628,509.3	5,364,468.7	1,006.3	31.0		75	-2	TR
ATOTR12	631,664.7	5,366,879.5	1,051.7	11.0		211	-8	TR
ATOTR120	631,189.8	5,367,114.0	1,031.8	101.5		167	-5	TR
ATOTR121	631,238.6	5,367,266.5	1,043.1	127.0		65	0	TR
ATOTR122	632,152.5	5,364,382.6	1,082.1	50.5		170	-7	TR
ATOTR123	632,296.1	5,364,368.7	1,083.0	27.0		183	-11	TR
ATOTR123add	632,294.9	5,364,368.8	1,083.0	21.0		4	-9	TR

ATOTR124	632,425.5	5,364,235.3	1,080.2	76.0	333	-7	TR
ATOTR125	632,165.6	5,364,190.8	1,096.0	76.0	161	-4	TR
ATOTR126	631,925.3	5,364,013.9	1,113.3	25.4	162	-11	TR
ATOTR127	632,496.7	5,365,003.6	1,109.7	101.5	7	-16	TR
ATOTR128	633,788.7	5,368,993.4	1,040.2	133.0	144	-1	TR
ATOTR129	633,119.4	5,369,173.2	1,073.7	63.0	332	-3	TR
ATOTR13	630,560.6	5,366,424.6	1,042.3	95.0	214	-2	TR
ATOTR130	632,510.0	5,367,674.0	1,061.3	133.0	125	-4	TR
ATOTR131	632,442.0	5,367,638.7	1,058.1	167.0	130	-5	TR
ATOTR132	632,566.4	5,367,713.5	1,061.7	84.0	126	-4	TR
ATOTR133	632,441.0	5,367,545.6	1,062.7	115.0	126	-2	TR
ATOTR134	632,343.2	5,367,359.6	1,058.9	135.0	91	-6	TR
ATOTR135	632,613.0	5,367,774.8	1,060.2	89.0	124	-13	TR
ATOTR136	632,393.7	5,367,271.6	1,063.6	88.0	85	-3	TR
ATOTR137	631,991.8	5,367,301.5	1,041.2	2.0	65	-32	TR
ATOTR138	631,879.8	5,367,383.3	1,040.5	2.0	65	-3	TR
ATOTR139	632,483.3	5,367,207.7	1,070.4	99.0	129	-2	TR
ATOTR14	632,351.6	5,367,478.2	1,056.2	35.0	208	-2	TR
ATOTR140	632,592.1	5,367,320.4	1,096.1	89.0	134	-1	TR
ATOTR141	632,700.1	5,367,416.5	1,091.3	80.0	130	-3	TR
ATOTR142	633,131.2	5,369,514.9	1,083.9	88.0	130	-6	TR
ATOTR143	633,204.2	5,369,615.1	1,090.3	97.0	131	-5	TR
ATOTR144	633,296.1	5,369,695.6	1,084.3	82.0	130	-7	TR
ATOTR145	632,658.5	5,367,960.1	1,067.6	74.0	90	0	TR
ATOTR146	632,686.3	5,366,739.7	1,093.1	38.0	270	-5	TR
ATOTR147	633,088.4	5,369,404.3	1,080.7	98.0	130	0	TR
ATOTR148	633,721.2	5,369,458.9	1,065.9	80.0	136	-7	TR
ATOTR149	633,764.8	5,369,580.7	1,058.6	133.0	133	-2	TR
ATOTR15	632,655.1	5,366,614.6	1,081.6	35.0	234	-3	TR
ATOTR150	633,125.8	5,369,127.8	1,072.2	69.0	131	-5	TR
ATOTR151	632,418.5	5,367,186.9	1,062.6	80.0	61	-3	TR
ATOTR152	633,526.0	5,369,850.5	1,066.2	39.0	311	-3	TR
ATOTR153	633,752.5	5,370,577.6	1,053.5	67.0	131	-2	TR
ATOTR154	633,096.4	5,366,509.0	1,067.7	35.0	116	-2	TR
ATOTR155	633,830.4	5,369,627.5	1,050.8	152.0	127	-1	TR
ATOTR156	634,043.4	5,369,462.0	1,034.6	33.0	122	-5	TR
ATOTR157	633,908.7	5,368,632.9	1,018.8	53.0	88	-1	TR
ATOTR158	633,158.4	5,371,936.4	1,058.1	92.0	87	-3	TR
ATOTR159	633,714.1	5,368,935.7	1,034.8	52.0	51	-3	TR
ATOTR16	632,407.3	5,366,819.0	1,060.4	52.0	221	-4	TR
ATOTR160	633,885.6	5,368,623.6	1,019.0	144.0	39	-2	TR
ATOTR161	633,966.3	5,368,703.7	1,024.9	56.0	131	-6	TR
ATOTR162	633,876.5	5,368,820.6	1,034.6	35.0	35	-1	TR
ATOTR163	631,348.3	5,367,069.7	1,036.5	70.0	138	-3	TR
ATOTR164	631,844.4	5,367,809.7	1,032.2	34.0	120	-1	TR

ATOTR165	631,791.1	5,367,726.0	1,029.9	86.0	117	-1	TR
ATOTR166	632,130.8	5,368,248.9	1,066.2	34.0	119	-6	TR
ATOTR167	631,912.0	5,367,772.5	1,033.2	16.0	120	-1	TR
ATOTR168	631,840.6	5,367,747.3	1,030.8	14.0	122	-1	TR
ATOTR169	631,844.0	5,367,706.7	1,031.4	17.0	34	-3	TR
ATOTR17	631,361.4	5,367,180.5	1,043.1	15.0	216	-2	TR
ATOTR170	631,844.1	5,367,654.0	1,031.4	16.0	26	-8	TR
ATOTR174	632,631.8	5,366,310.3	1,063.8	44.0	90	-2	TR
ATOTR18	631,504.5	5,367,148.2	1,064.9	92.0	213	-15	TR
ATOTR180	634,040.4	5,368,793.4	1,025.0	73.0	148	-4	TR
ATOTR181	634,057.7	5,368,710.9	1,022.4	38.0	92	-6	TR
ATOTR182	633,819.8	5,368,720.5	1,025.4	86.0	185	-5	TR
ATOTR183	632,765.6	5,366,616.2	1,097.6	48.0	21	-8	TR
ATOTR184	632,458.3	5,365,077.2	1,095.6	55.0	93	0	TR
ATOTR185	633,119.8	5,371,556.9	1,077.5	160.0	88	0	TR
ATOTR186	632,534.8	5,364,963.8	1,109.9	28.0	51	0	TR
ATOTR187	632,357.8	5,365,090.1	1,082.3	74.0	42	0	TR
ATOTR189	632,119.0	5,366,618.8	1,055.9	40.0	93	0	TR
ATOTR19	631,564.0	5,367,169.0	1,060.2	154.0	307	-11	TR
ATOTR190	632,287.2	5,367,250.2	1,052.9	96.0	122	0	TR
ATOTR191	632,237.2	5,367,171.8	1,049.3	99.0	129	0	TR
ATOTR192	633,865.3	5,369,684.3	1,041.5	102.0	140	0	TR
ATOTR193	633,954.1	5,370,337.7	1,034.6	107.0	125	0	TR
ATOTR194	632,242.5	5,366,067.5	1,064.0	28.0	243	0	TR
ATOTR20	631,530.7	5,367,128.8	1,058.8	38.0	166	-10	TR
ATOTR202	631,039.0	5,367,221.8	1,024.8	54.0	140	0	TR
ATOTR203	630,896.1	5,367,129.5	1,014.6	39.0	133	0	TR
ATOTR204	632,414.1	5,366,999.9	1,059.8	40.0	92	0	TR
ATOTR205	632,422.7	5,366,728.1	1,060.7	41.0	87	0	TR
ATOTR206	632,409.7	5,366,369.1	1,062.5	41.0	87	0	TR
ATOTR207	632,422.7	5,365,944.1	1,067.0	44.0	96	0	TR
ATOTR208	632,410.3	5,365,603.8	1,063.7	47.0	90	0	TR
ATOTR209	632,514.1	5,366,711.9	1,068.3	24.0	89	0	TR
ATOTR21	631,701.8	5,367,178.7	1,053.1	32.0	205	-1	TR
ATOTR210	632,606.1	5,366,965.3	1,078.2	19.0	99	0	TR
ATOTR211	632,188.8	5,366,305.5	1,067.9	32.0	43	0	TR
ATOTR212	632,428.0	5,366,099.3	1,070.9	31.5	200	0	TR
ATOTR213	631,849.2	5,365,724.6	1,056.5	22.0	221	0	TR
ATOTR214	632,052.0	5,365,776.8	1,055.2	20.0	205	0	TR
ATOTR215	632,553.1	5,366,377.4	1,065.8	7.0	84	0	TR
ATOTR216	632,500.2	5,367,177.9	1,070.6	29.0	48	0	TR
ATOTR218	628,558.2	5,364,271.9	1,029.6	28.0	44	0	TR
ATOTR22	631,344.5	5,366,795.5	1,028.4	17.0	175	-3	TR
ATOTR23	631,698.7	5,367,130.6	1,055.3	60.0	152	-4	TR
ATOTR231	632,753.6	5,367,785.1	1,067.2	62.0	126	-3	TR

ATOTR232	632,508.3	5,368,160.7	1,065.9	90.0	123	-1	TR
ATOTR233	632,760.1	5,368,052.2	1,075.0	33.0	116	-5	TR
ATOTR234	633,043.7	5,368,750.0	1,083.2	90.0	118	-5	TR
ATOTR235	633,035.1	5,369,317.4	1,073.9	75.0	124	-5	TR
ATOTR235add	633,097.1	5,369,276.7	1,076.5	86.0	124	-2	TR
ATOTR236	633,421.2	5,369,946.0	1,060.9	44.0	133	-2	TR
ATOTR237	633,851.8	5,370,488.4	1,051.3	82.0	129	-8	TR
ATOTR238	632,793.8	5,364,449.8	1,082.3	89.0	157	-3	TR
ATOTR239	631,471.1	5,364,156.5	1,095.6	38.0	177	-15	TR
ATOTR24	631,761.2	5,366,952.0	1,059.5	21.0	151	-14	TR
ATOTR240	631,841.3	5,364,281.3	1,080.7	46.0	158	-6	TR
ATOTR241	631,540.2	5,364,183.7	1,095.8	42.0	156	-10	TR
ATOTR241add	631,540.2	5,364,183.7	1,095.8	10.0	339	-12	TR
ATOTR242	631,730.2	5,364,214.1	1,080.2	4.0	175	-25	TR
ATOTR243	630,152.1	5,367,857.7	1,025.9	43.0	57	0	TR
ATOTR244	633,515.5	5,368,482.0	1,037.8	66.0	120	0	TR
ATOTR38	633,391.6	5,369,799.5	1,088.5	65.0	111	-4	TR
ATOTR39	632,703.5	5,368,965.0	1,076.8	182.0	119	-1	TR
ATOTR40	629,511.1	5,360,681.5	1,004.7	58.0	310	0	TR
ATOTR48	632,644.9	5,367,343.7	1,099.7	97.5	305	-1	TR
ATOTR49	632,759.5	5,366,891.6	1,106.1	95.0	336	-10	TR
ATOTR50	632,463.3	5,366,918.2	1,063.4	90.0	176	-1	TR
ATOTR51	632,492.6	5,367,599.9	1,060.8	90.2	53	-2	TR
ATOTR52	631,563.5	5,366,958.6	1,046.5	28.0	107	-2	TR
ATOTR53	631,396.6	5,366,870.4	1,031.4	60.0	285	-3	TR
ATOTR54	632,377.6	5,368,118.8	1,057.1	101.0	267	-2	TR
ATOTR55	631,874.5	5,367,876.8	1,035.3	100.0	209	-2	TR
ATOTR56	631,002.6	5,367,828.2	1,018.5	15.0	127	0	TR
ATOTR57	630,711.5	5,367,511.4	1,014.1	19.0	122	-4	TR
ATOTR58	631,973.4	5,366,789.5	1,045.7	16.0	122	0	TR
ATOTR59	633,035.7	5,368,517.5	1,090.2	82.9	270	-5	TR
ATOTR60	632,211.5	5,366,720.4	1,051.3	27.0	229	-2	TR
ATOTR61	632,366.6	5,366,615.7	1,057.5	25.0	250	-4	TR
ATOTR62	632,179.8	5,367,090.4	1,046.5	26.0	37	-5	TR
ATOTR63	632,023.7	5,366,670.0	1,046.2	18.5	57	-3	TR
ATOTR64	632,303.3	5,367,524.6	1,051.7	48.2	210	0	TR
ATOTR65	631,059.1	5,367,110.4	1,024.8	17.0	225	-4	TR
ATOTR66	632,310.7	5,367,512.9	1,052.8	46.8	122	-4	TR
ATOTR67	632,655.1	5,368,756.3	1,066.6	28.0	218	-2	TR
ATOTR68	632,725.7	5,368,854.0	1,075.7	28.0	53	-2	TR
ATOTR69	632,174.6	5,367,917.6	1,049.2	32.0	123	-1	TR
ATOTR70	631,969.1	5,368,007.4	1,046.5	30.0	219	-5	TR
ATOTR71	632,019.6	5,368,059.3	1,056.2	22.2	225	-13	TR
ATOTR72	632,191.7	5,367,066.8	1,048.2	22.0	122	-2	TR
ATOTR73	632,653.3	5,368,809.1	1,069.2	64.0	128	-1	TR

ATOTR74	630,529.8	5,367,897.4	1,024.0	56.0	100	-1	TR
ATOTR75	630,518.1	5,368,405.6	1,008.2	61.2	209	0	TR
ATOTR85	632,445.3	5,367,904.2	1,052.5	136.0	60	-3	TR
ATOTR86	631,300.9	5,367,234.6	1,041.2	90.0	109	-1	TR
ATOTR87	632,351.7	5,367,485.7	1,054.6	152.0	122	6	
ATOTR90	632,388.2	5,366,624.4	1,057.9	24.1	260	0	TR
Trenches	**167.0**		**Total**	**10,184.3**	**m**		
			Av	**61.0**	**m**		

APPENDIX B. ABBREVIATIONS

Abbreviation	Meaning or Units
'	Feet
"	Inch
$	Dollar Sign
$\$/m^2$	Dollar per Square Metre
$\$/m^3$	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/m^3	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
BFA	Bench Face Angle
Bi	Bismuth
BML	Base Metal Laboratories
BoQ	Bill of Quantities
BSG	Bulk Specify Gravity
BSTP	Biological Sewerage Treatment Plant

Abbreviation	Meaning or Units
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
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

Abbreviation	Meaning or Units
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
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

Abbreviation	Meaning or Units
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
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

Abbreviation	Meaning or Units
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
$mg/m^2/day$	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
Mm^3	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

Abbreviation	Meaning or Units
NE	Northeast
NFPA	National Fire Protection Association
NGO	Non-Governmental Organisation
NGR	Neutral Grounding Resistor
Ni	Nickel
NI 43-101	National Instrument 43-101
Nm^3/h	Normal Cubic Metre per Hour
NNE	North - Northeast
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

Abbreviation	Meaning or Units
RF	Revenue Factor
RFQ	Request for Quotation
ROM	Run of Mine
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/m^2	Tonne per Hour per Square Metre
t/m or tpm	Tonne per Month
t/m^2	Tonne per Square Metre
t/m^3	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
TSX	Toronto Stock Exchange
U	Uranium

Abbreviation	Meaning or Units
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

APPENDIX C. Certificate Of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

I, Robin A Rankin, do hereby certify that:

- I am a Geologist.
- I am Principal Consulting Geologist and operator of GeoRes, with business address at 4 Warenda Street, Bowral, New South Wales, Australia, 2576.
- I am an author of the technical report entitled “Altan Tsagaan Ovoo Project (ATO) 2022 Mineral Resources and Reserves Technical Report (NI 43-101)” (the “**Technical Report**”) with an effective date of 6th November 2022 prepared for Steppe Gold Limited (the “**Issuer**”). The Technical Report relates to Mining License MV-017111 constituting the property of the ATO Project (the “Project”) in eastern Mongolia.
- 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. I have practiced my profession (geology) virtually continuously since 1981. 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.
    - 2021: I authored the previous 30th March 2021 NI 43-101 2021 Mineral Resources Technical Report on the Project.
- 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.
- I have read the definition of “qualified person” set out in National Instrument 43-101 (“**NI 43-101**”) and certify that by reason of my education, affiliation and membership designation with an

accepted foreign professional association (as defined in NI 43-101) and relevant experience, I am a "qualified person" for the purposes of NI 43-101.

- I visited the Project in April/May 2022 for 2 days. During that visit I inspected the geology and mining operations on two of the open pits and briefly inspected surrounding surface geological outcrops. I also observed diamond exploration drilling in progress.
- I assisted with the preparation of the Technical Report prepared for the Issuer and, in particular, am the QP responsible for the following aspects of the Technical Report: relevant geological and Resource parts of the introductory Summary (Section 1); the Geology (Section 7 and 8); Resource estimation in general (Section 14); reported Mineral Resources (Section 14.18); relevant aspects of the Interpretation and Conclusions regarding the Resources (Section 25.1); and further exploration aspects of the Recommendations (Section 26).
- I am independent of the Issuer in terms understood from Part 1.5 of NI 43-101 and the tests detailed in Part 1.5 of the Companion Policy 43-101CP.
- I had no prior involvement with the Project that is the subject of the Technical Report, or with the Project operator, or with the Issuer, before undertaking the work leading to the previous 2021 Resource Report.
- I have read NI 43-101 and Form 43-101F1, and the sections of the Technical Report for review of which I was responsible have been prepared in compliance with that instrument and form.
- As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report for review of which I was responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 13th day of **March 2023**.

"Signed and Sealed"

Robin A Rankin

MSc DIC MAusIMM (CPGeo)[33]

Principal Consulting Geologist – **Geo**Res

[33] Accredited by The Australasian Institute of Mining & Metallurgy (The AusIMM) since 2000 as a Chartered Professional (CP) in the Geology discipline.

CERTIFICATE OF QUALIFIED PERSON

I, Grant Walker, do hereby certify that:

1. I am a Mining Engineer.

2. My business address is Level 6 40 Creek Street Brisbane QLD 4000, Australia.

3. My current occupation Principal Advisor   Corporate Solutions at Xenith Consulting Pty Ltd.

4. I am an author of the technical report entitled Altan Tsagaan Ovoo Project (ATO) 2022 Mineral Resources & Reserves Report (NI 43-101) with an effective date of 6th November 2022 (the "**Technical Report**") prepared for Steppe Gold Ltd. (the "**Issuer**").

5. I graduated with a Bachelor of Engineering (Mining) from the University of New South Wales, Sydney, Australia in 1996. In addition, I have obtained a Master of Applied Finance from Kaplan. I have practiced my profession (Mining Engineering) virtually continuously since 1996. A summary of my relevant experience follows:

   1996 – 1998: Technical Services Manager at Petangis Mine for (Darma Henwa/HWE)

   1998 – 2011: Senior/Principal Engineer at Minarco MineConsult. Carried out over 250 various projects in several countries. The projects range from LOM planning, Reserve reporting, Due Diligence and Public Reporting to various stock exchanges.

   2011 – 2022: Manager & Principal Mining Engineer employed by Xenith Consulting. As well as the establishment and continued growth of the Xenith business I have undertaken project manager roles for various studies to identify the key value drivers, acted as a technical adviser in due-diligence and M&A for both buyer and sellers on projects in Australia and overseas, completing numerous public reports in accordance with both the ValMin and JORC codes. In addition, providing mentoring and support to technical services teams across Australia, Indonesia and Mozambique, including implementing planning systems and processes.

   2022 – Present: Principal Advisor   Corporate Solutions employed by Xenith Consulting.

   I am a Chartered Professional member (#230557) of the Australasian Institute of Mining and Metallurgy (MAusIMM CP).

6. I have read the definition of "qualified person" set out in National Instrument 43-101 ("**NI 43-101**") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and experience, I am a "qualified person" for the purposes of NI 43-101.

7. I visited the Altan Tsagaan Ovoo Project in October 2022 for 3 days. During that visit I inspected the mining operations in three of the open pits and the Phase 1 processing plant.

8. I assisted with the preparation of the Technical Report prepared for the Issuer and, in particular, I'm the qualified person responsible for the Mine Plan and Reserve aspects of this Report. I prepared Section 1- Summary, Section 15 – Mineral Reserve Estimates, Section 16 – Mining Methods, Section 21 – Capital & Operating Costs and Section 22 – Economic Analysis.

9. I am independent of the Issuer applying all of the tests in Section 1.5 of NI 43-**101** and had no prior involvement with the property that is the subject of the Technical Report, or with the property operator, the Issuer, before undertaking the work leading to this Technical Report.

10. I have read NI 43-101 and Form 43-101F1, and the sections of the Technical Report for review of which I was responsible have been prepared in compliance with that instrument and form.

11. As at the effective date of the Technical Report, and to the best of my knowledge, information and belief, the sections of the Technical Report for review of which I was responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 13th day of March 2023.

"Signed and Sealed"

Grant Walker (Qualified Person)
Be (Mining) MAusIMM CP(Mining)
Principal Mining Engineer – **Xenith Consulting**

APPENDIX D. Computerised Geological Modelling Methods

Geological modelling is an activity undertaken largely for the purpose of estimating Mineral Resources. It may be done manually on plans and cross-sections or more usually now by using computers. Several methods are commonly used – and their purpose is primarily to interpret the shape of a mineral deposit, interpolate mineral grades throughout it (from drill hole samples), and then estimate the Mineral Resources. Some methods are used in combination.

GeoRes uses **Minex** geological and mining software.

Common geological modelling methods are:

- Polygonal
- Surface
- Wire frame
- Block

POLYGONAL METHOD

A polygonal method typically involves interpreting closed polygons around "ore" zones from the drill holes intercepting them. The polygons are interpreted on cross-sections traversing the deposit and regularly spaced the length of the deposit. Areas of polygons are estimated and average grades assigned to each by length weighting all the drill hole sample assays found within a polygon. Polygon influences are assumed to extend half way to the adjacent polygons, giving a width. Volumes are then calculated from polygon areas multiplied by their influence widths; and tonnages found by applying a density. The polygonal method is manual or computerised, and is generally a stand-alone method.

SURFACE METHOD

Surface modelling methods involve creating computerised surfaces of geological features. The features may be stratigraphic horizons, rock type interfaces, veins, weathering changes, faults or other structures easily characterised by an open surface. Surfaces are particularly suited to modelling thin bodies. Surfaces are created from drill hole intercepts and mapping strings.

Sub-horizontal surfaces are modelled relative to a horizontal datum. Steeply dipping surfaces are modelled relative to an inclined reference plane the orientation of which is chosen closest to closest to sub-parallel to the surface. This latter steep surface modelling is often applicable to "**vein**" deposits.

A collection of stratigraphically stacked surfaces readily models sedimentary or layered geology. If a regular sequence is present it should have to be honoured in every drill hole, consequently an interpretation stage is undertaken to ensure this (principally inserting missing intercepts in appropriate positions). If units are not in a sequence they are simply modelled individually, with cross cutting situations accounted for in various ways.

Surfaces themselves are either collections of **triangles** (where straight lines join the data points) or are **grids** (where the positions of regularly spaced points in space (forming a regular grid pattern or mesh) are interpolated from the surrounding data points. With grids an interpolation algorithm relevant to the type of surface is chosen, typically a growth or trending method, inverse distance weighting or kriged weighting. An advantage of grid surface modelling is the fully 3-dimensional interpolation used to estimate new values between data points. The rolling surfaces produced, particularly with trending algorithms, usually realistically simulate natural surfaces. Gridded surfaces are also amenable to manipulation and mathematical operations, which can enforce stratigraphic rules.

Surface modelling is computerised, may be combined with polygonal or wire frame methods, and the surfaces are usually combined into a 3D block model.

Wire Frame Method

Wire frame methods use computerised closed surfaces to represent geological volumes. The surface is the outside edge of the volume. Wire frame surfaces are extensions of triangle surfaces where the triangles are closed back on themselves. They are usually created from a series of closed cross-sectional polygon "outline" strings (the same starting point to the polygonal method) which are stitched together by "wires". Polygonal outline string digitising is usually based on coloured geological drill hole intercepts projected onto cross-sections, and on mapping strings. Wire frame models are computerised and are usually combined into a 3D block model.

Block Method

Block models are minutely sub-divided representations of geological models in which individual mineral grades may be interpolated into each subdivision (a block) for the ultimate purpose of accumulating (for all blocks) a Mineral Resource. Various other variables may be estimated when interpolating an individual block grade, such as the numbers or distances of samples used, allowing determination of confidence and such things as JORC Resource classifications. Block models are computerised and are usually the source of data for computerised pit optimisation.

Block modelling methods typically involves constructing a framework of adjacent 3D blocks which cover the full volume of the deposit. The framework may simply be defined somewhere in space, or is often created within surface or wire frame models. Blocks are often all the same size, but the X, Y and Z dimensions may be different. They are made small enough to honour shapes and allow for spatial grade variations. Blocks may be reduced in size locally (termed "sub-blocking") to even better reflect geological boundaries. Block grades are calculated independently for each block from nearby drill hole sample assays, usually using an algorithm which proportions individual sample influences based on their distance and direction from the block. The ID algorithm proportions sample weighting in inverse relation to the distance and the distance is often further weighted (such as where it is squared and termed the ID2 algorithm). The Kriging algorithm introduces geostatistical information on sample variances into the distance weighting.

Where directional sample anisotropy exists (samples in some directions away from a block have different weightings to samples in other directions, say in a cross dip direction) then dynamic direction controlling block models may be used. These allow data searching to follow along (i.e. bend around) geological layers, and "**un-folding**" approach. These unfolding block models may also be used to control drill hole sample data searching during geostatistical analysis. Their very powerful control frequently allows good geostatistical analysis where otherwise it is very poor or impossible, and it produces very well controlled stratigraphic grade continuity.

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