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NI 43-101 TECHNICAL REPORT AND MINERAL RESOURCE UPDATE, CONVERSE PROPERTY, HUMBOLDT COUNTY, NEVADA, USA

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1 Summary

1.1 General

The Technical Report provides an updated Mineral Resource Estimate ("MRE") and metallurgy of the Mineral Resources identified within the Converse Property (the "Property"), Pershing County, Nevada, USA. This report is authored by Michael B. Dufresne, P.Geol., P.Geo., APEX Geoscience Ltd. ("APEX"), Philo Schoeman, P.Geo., Pr.Sc.Nat., APEX, R. Mohan Srivastava, P.Geo., RedDot3D Inc. ("RedDot3D") and Ray Walton, P. Eng., Ray Walton Consulting Inc. ("RWC"). The authors are fully independent of Converse Resources LLC ("CRL") and are Qualified Persons ("QPs") as defined by National Instrument (NI) 43-101. Mr. Walton has prepared and is taking responsibility for Section 13 of this report, Mr. Srivastava has prepared and is taking responsibility for Section 14 of this report and Mr. Dufresne and Mr. Schoeman have prepared and are taking responsibility for the remainder of this report. Converse Resources LLC is a Reno, Nevada (NV), based private corporation, that is under a Letter of Intent ("LOI") to be purchased by Axcap Ventures Inc. ("Axcap" or "AVI").

Axcap engaged APEX, RedDot3D and RWC to complete an independent Technical Report (the "Report") using the format specified in Form NI 43-101F1 to provide the results of an updated MRE for the Converse Property. The Technical Report has been prepared in accordance with the Canadian Securities Administration's (CSA's) NI 43-101 Standards of Disclosure for Mineral Projects and the Form NI 43-101F1 guidelines for technical reporting. The updated MRE has been prepared in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) "Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines" dated November 29, 2019 and reported using the CIM "Definition Standards for Mineral Resources and Mineral Reserves" dated May 14, 2014.

The most recent exploration from 2017 to 2019 conducted by CRL on the Property includes a seven hole core drilling program, metallurgical studies, rebuilding of the geological model, environmental studies and permitting.

1.2 Property

The Property is owned by CRL and consists of 286 unpatented mining claims (the "Claims") located on federal land administered by the United States Bureau of Land Management (BLM) and five privately-owned tracts of land (the "Fee Land"). The total land area covered by the Property is approximately 7,784 acres (ac) with 4,588 ac of Claims land and 3,196 ac of Fee land. The Claims are a combination of both owned and leased. The Claims are located within Sections 1 and 12, T32N, R41E, MDM, Pershing County, Sections 6 and 8, T32N, R42E, MDM, Humboldt County, Section 36, T33N, R41E, MDM, Humboldt County, and Sections 2, 4, 8, 10, 16, 18, 20, 22, 28, 30, 32 and 34, T33N, R42E, MDM, Humboldt County, NV. One of the five Fee Land parcels (Section 17, Township 33 North, Range 42 East, MDM) is presently leased to a third party.

The current land holding costs of the Property total approximately US$151,823 annually (including the annual advance royalty payment due under the Converse Lease, as well as BLM maintenance fees and county recording fees for the Claims and property taxes on the Fee Land).

The Property is subject to a number of net smelter returns (NSR) production royalties payable to Royalty Consolidation Company, LLC (RCC) on the sale of any minerals from the Property (the "RCC Royalty"). The RCC Royalty is effectively a blanket 6% NSR royalty on the production of all minerals.

1.3 Access, Permitting and History

The Property is located in Buffalo Valley, in the southeast corner of Humboldt County, NV, with a very minor part of the Property extending into Pershing County at the southwest corner. It lies approximately 10.5 miles south of the Valmy exit on Interstate 80; approximately 15.5 miles west-northwest of Battle Mountain and approximately 30 miles southeast of Winnemucca. Reno is located approximately 205 miles (330 km) to the southwest by highway. The Property is located at approximately 40º43' N latitude and 117º16' W longitude, within Township 33 North, Range 42 East. The Converse Property is located 21 miles northwest of Battle Mountain by road and 45 miles southeast of Winnemucca by road. A well-maintained county gravel road leads 10.3 miles from Exit 216 (Valmy) on the Interstate 80 Highway (I-80; Figure 5.1), past the Marigold Mine and the turnoff to the Trenton Canyon mine, both of which are near the Property. Further access to the Property is by dirt roads and tracks.

Access to electricity and natural gas supplies is within 5 miles of the Property from SSR Mining Inc.'s (SSR) Marigold Mine and within 9 miles of Highway 80. Water required for exploration drilling is supplied by the nearby Marigold Mine. In 2019, CRL purchased 2,560 acre-feet of irrigation water rights from New Nevada Lands, LLC (Permit 71715 and 71716). Once converted to mining and milling use, the acquired water rights would support the construction and operation of a future mine at the Property. An application requesting a change in the water rights' point of diversion, place of use and manner of use was submitted to the Nevada State Engineer on October 29, 2020.

Exploration of the Converse Property is carried out under an Exploration Plan of Operations, NVN065461, approved by the BLM pursuant to Environmental Assessment N20-98-001P and Reclamation Permit #0122 approved by the Nevada Division of Environmental Protection. There is a US$56,330 reclamation bond currently associated with the existing permits.

There are no significant factors or risks that the QPs are aware of that would affect access, title, infrastructure or preclude year-round exploration and potential future mining operations at the Property. Exploration at the Property began in 1988 with initial claims staked by Nevada North Resources and the first holes drilled in 1988 by Kennecott Minerals Co. Numerous exploration and drill campaigns have been completed by several owners. The property has no history of mining or production.

1.4 Geology and Mineralization

The Converse Property is located in the Battle Mountain district, within the Battle Mountain-Eureka Trend, one of the main gold deposit trends in Nevada comprising a northwest trending belt of precious metal deposits with current reserves and past production exceeding 50 million oz Au (Holley et al., 2015).

The regional geology of the Battle Mountain Mining District comprises three Paleozoic rock assemblages, summarized from Yennamani (2010), as follows: Ordovician, Silurian and Devonian aged siliceous sequence of the Roberts Mountains allochthon; Mississippian, Pennsylvanian and Permian Havallah sequence of the Golconda allochthon; and Pennsylvanian-Permian Antler sequence of the Antler orogeny.

The Property region is underlain by the structurally-complex Havallah sequence that consists of basinal sedimentary rocks, including calcareous sandstone and siltstone, quartzite, pebbly limestone, siliceous siltstone, chert, argillite and variable amounts of basalt and greenstone. This unit measures greater than 1,800 ft in thickness. The calcareous sandstone-rich unit is the main host to the Redline deposits. The Havallah sequence is intruded by a porphyry stock and related dykes and/or sills. The intrusive is known as the Redline porphyry stock and varies from dacite to granodiorite and grades into tonalite at its southern contact. The stock diameter measures from 1,475 to 1,970 ft and its estimated age is Tertiary, based on regional correlations and a 41 million year (Ma) isotopic age date (Cleveland, 2000).

Alluvial material, predominantly Quaternary and late Tertiary in age, covers much of the deposit area and ranges in thickness from 20 ft to over 900 ft. The alluvial material comprises sand, sandy gravel, pebble-gravel and discontinuous deposits of sand, silt and mud.

Fault zones identified within the Property are high-angle, interpreted to strike predominantly to the north and include pre-, syn- and post-mineralization faulting.

Two main styles of metamorphic/alteration assemblages are observed at the Property and include early prograde hornfels-skarn and late retrograde skarn assemblages. Skarn assemblages are developed within both the intrusive unit (endoskarn) and Havallah units (exoskarn). Prograde metamorphism includes an early biotite-rich (potassic) hornfels that is overprinted by biotite-amphibole-pyroxene and garnet-pyroxene (calc-silicate) skarn assemblages. The prograde envelope extends ~1,500 ft (or ~450 m) laterally to the north and south from the central granodiorite porphyry. The distribution of strongly developed prograde assemblages can be defined by a radial zonation of up to 800 ft laterally around the central stock. Further outboard at a distance of 800 to 1,500 ft from the intrusive contact is a second zonation of biotite-rich hornfels and minor intervals of pyroxene-garnet skarn minerals associated with a small lobe of the intrusive unit. Less spatially defined are two retrograde assemblages that overprint the prograde assemblages. From oldest to youngest these include partial sericite replacement of the plagioclase along with epidote-chlorite-actinolite-quartz-calcite veins followed by quartz-calcite veins.

The Property hosts two gold-rich skarn deposits known as North Redline and South Redline. Gold mineralization is observed over an approximate 5,000 by 2,500 ft area and extends from a vertical depth of 18 ft below surface to >2,000 ft. Mineralization is spatially associated with all observed alteration/metamorphic assemblages indicating gold deposition occurred throughout the skarn development. Gold predominantly occurs as liberated grains. Silver and copper mineralization are also spatially associated with gold. The sulfide minerals including pyrrhotite, chalcopyrite, pyrite, sphalerite and molybdenite precipitated during development of the prograde and retrograde assemblages. Galena, arsenopyrite and bismuth-tellurium minerals are also present but in minor abundances.

Three redox zones are observed and include an oxide, transition and sulfide zone. The oxide zone has a variable vertical depth profile ranging from 35 to >500 ft below the base of alluvium. Goethite is the dominant iron oxide mineral. The transition zone, consisting of both iron oxide and sulfide minerals, underlies the oxide zone. The vertical depth profile of this zone ranges from <5 to >1,400 ft in thickness. The sulfide zone comprises sulfide minerals and predominantly underlies the transition zone. In the North Redline deposit, localized areas of sulfide material are observed in the oxide zone.

1.5 Drilling, Sampling and Assaying

The Property drillhole database as of December 31, 2020 contained 326 drillholes totalling 254,833.6 ft. The database used for the current MRE consists of 215,123 ft drilled in 249 holes that has provided 31,908 gold assays from intervals totalling 172,325 ft of core or reverse circulation (RC) chips. Core drilling represents approximately 33% of the total footage and 23% of the total drillholes for the Property, with the remaining holes drilled as RC and to a lesser extent as rotary/mud rotary (MR). A nominal drillhole spacing across the deposit is approximately 400 ft and reduced to 100 or 200 ft spacing where infill drilling was completed. No additional drilling has been conducted since December 31, 2020.

The spatial distribution of the drillhole collars used for the MRE cover an area of approximately 6,000 ft long by 3,900 ft wide. RC samples were collected on 5 ft intervals and core hole samples were also predominantly sampled on 5 ft intervals with locally adjusted intervals based on lithological, alteration and mineralization changes. Samples were prepared and analysed by accredited laboratories that included Activation Laboratories (Actlabs), ALS Global (ALS), American Assay Laboratories (AAL), Bondar Clegg and Cone Geochemical Incorporated (Cone). Quality Assurance/Quality Control (QA/QC) samples including blanks, standard reference material (SRMs) and duplicates, which were routinely inserted for the majority of the sampled footage and for most of the drill campaigns.

The QP authors conducted a review of the available analytical data, including QA/QC data, and it is the opinion of the QPs that the sample preparation, security, and analytical procedures adopted meet accepted industry standards and are adequate to ensure overall data quality ensuring that the data is suitable for Mineral Resource Estimation.

1.6 Data Verification and Database

The drillhole database includes data from drillholes that span more than three decades since the Redline deposits were first drilled. Even though the Property has passed through several owners, the documentation and paper trail remain intact and in good condition. There are very few pieces of information (well below 1%) for which the only supporting documentation is hand-written tabulations of laboratory analyses. The digital database was well designed and well maintained, with information that facilitates tracing assay information back to original documents, the majority of which are assay certificates issued by accredited laboratories. Previous verification campaigns have been carried out by MRDI (1997) and SRK (2009, 2012 and 2014).

The APEX QPs conducted a verification program by reviewing 10% of the archived analytical geochemical certificates for the historical drillholes completed between 1996 and 2017 on the Property. The historical holes were randomly selected and reviewed from top to bottom versus the values contained in the drillhole database. There were no significant differences with respect to the company's database and the archived analytical certificates. In the opinion of the QPs, industry standard procedures have been used that are acceptable for ensuring the accuracy of all analytical data pertaining to exploration and drilling work conducted by CRL and its predecessors, and the database is suitable for use in Mineral Resource Estimation.

1.7 Metallurgical Studies

A significant amount of metallurgical test work has been completed to date on both composite and variability samples. The composites were mainly drill core, and some assay rejects, collected from a number of areas around the deposit. The sample gold values utilized for the test work were similar to the MRE values and cover a wide range of oxidation states and other variables. The test work consisted of bottle roll and column leach cyanidation tests, as well as comminution, gravity and flotation testing. From 2004, and up to December 31, 2020, four test work programs were carried out and are summarized in a total of eleven reports and memorandums.

For the column leach tests completed on conventionally crushed material, gold extractions ranged from 43 to 84% based on calculated heads which ranged from 0.487 to 1.343 grams per tonne (g/t) gold (Au). The sodium cyanide consumptions ranged from 1.74 to 2.90 kilograms per tonne (kg/t). The material used in leaching was blended with 0.51 to 1.52 kg/t hydrated lime. For the column leach tests completed on crushed material generated using high pressure grinding roll (HPGR), gold extractions ranged from 50 to 86% based on calculated heads which ranged from 0.460 to 1.355 g/t. The sodium cyanide consumptions ranged from 1.55 to 2.90 kg/t. The material used in leaching was blended with 0.77 to 1.03 kg/t hydrated lime or agglomerated with 3.95 to 4.28 kg/t cement. It should be noted that the average extraction of the conventionally-crushed material was 65%, while the average extraction of the HPGR-crushed material was 68% (an average increase of 3%).

Although gravity concentration, agitation leaching and flotation were considered during the test programs, the preliminary process selection to support the MRE is fine crush and heap leach. The emphasis in the following sections is therefore on summarizing process conclusions and proposing preliminary process design criteria using data from the column testing. Sampling location and testing in the recent 2018 test program is the most thorough and therefore results from that program have been prioritized.

The column test data clearly shows that a fine crush of the order of ¼ inch (in) is required to maximize recoveries. Recoveries using HPGR were 3% higher than conventional crush. While this is significant, it represents only a small incremental increase in revenue at the expense of increased process complexity and cost. The results demonstrate Redline South consistently yields better recoveries than Redline North. It is believed that this is explained by the overall lower copper value in Redline South. Regarding the effect of the oxidation state, samples designated as “oxide” and “transition” by the geologists usually yield similar recoveries, while recoveries from “sulfide” tend to be lower. This is explained by the overall higher copper value of the sulfide samples and the fact that weathering positively affects gold liberation.

Using only the P80 <6.4 millimetre (mm) conventional crush data from the 2018 program on a redox basis, the average gold recoveries achieved were 77% in oxide, 62% in transition and 50% in sulfide material. Using only the 9.5 mm conventional crush data from the 2018 program, the average gold recoveries achieved were 58% from Redline North and 73% from Redline South. This difference is likely explained by the average copper values of the Redline North and South zones, which are 961 and 475 parts per million (ppm), respectively. Recoveries are therefore predicted for Redline North and South deposits, using a single formula, considering the two areas as a single deposit.

When the 2018 test data from the two deposits are analyzed together, there is no clear relationship between the gold recovery and sulfide content. This is due to all samples, except for one, have low sulfur values. Test work results show a relatively narrow range of copper solubility values across the wide range of copper values tested, with no correlation identified. However, a negative correlation does exist between copper content and gold recovery.

Based on existing data, the copper value is used to predict gold recoveries, regardless of which deposit, gold value or oxidation state. The relationship is established using only the 2018 data. However, similar trendlines exist using data from all four metallurgical programs taken together, or individually. It should be noted that the “R2” value is fairly low indicating that some “scatter” can be expected. It should be noted that samples designated as “sulfide” throughout the four programs tend to also have a high copper grade. Samples with very low copper values can be expected to yield gold recoveries over 70%, those with mid-range copper values are predicted to have gold recoveries of approximately 60% and high copper values will result in gold recoveries of 45 to 50%. No deduction from the laboratory testing results was made as recoveries were still increasing, albeit at a slow rate.

1.8 Mineral Resource Estimate

An updated resource block model utilizing updated geological logging and assay data from drillholes completed post 2012 was completed and is discussed herein. In addition, silver (Ag), copper (Cu) and cyanide-soluble gold grades were also estimated to support the evaluation of the Property’s potential technical and economic viability, but have not been reported.

The digital database was well designed and well maintained, with information that facilitates tracing assay information back to original documents, the majority of which are assay certificates issued by accredited laboratories. The QP responsible for the MRE completed a review of the Converse drillhole database and is of the opinion that the database is suitable to support resource estimation.

The geological logging database included information on the lithology, the oxidation state and on the minerals observed. The lithology logging was used to identify the drillhole depths that served as control points for modeling the undulating surface that marked the base of the Quaternary alluvium. It was also used in the development of the three-dimensional (3D) model of the geometry of the main central stock of the intrusion, along with its small offshoots and satellites. The oxidation state was used to develop the model for the three bedrock oxidation zones: oxide, transition and sulfide. DXF files containing triangulated surfaces and solids were provided by CRL for the following features:

- The base of the Quaternary alluvium (Qal) gravels, which lie between the top of bedrock and topography;
- The oxidized bedrock below the Qal;
- The transitional oxide-sulfide zone beneath the oxidized bedrock;
- The sulfide (unoxidized) zone beneath the transitional zone;
- The central intrusive stock.

Each of the geological surfaces and solids was developed in Leapfrog Geo 6.0 using the information from drillhole logs to identify control points for modeling a smooth surface in 3D. The QP verified that the various geological wireframes were updated to include the most recent drillholes, that they honored logging information from drillholes, and that they struck a reasonable balance between complexity that captured important local details and smoothness that removed noise. The results of statistical analysis confirmed that they were well constructed and were suitable for use in resource estimation.

The majority (>90%) of the sample intervals in the assay database are exactly 5 ft in length. Statistical analysis indicates there is no significant change in the variability of gold grades as a function of sample length; there is also no significant difference in the average grade of shorter or longer samples. Since there is no relationship between sample length and variability of gold grades, or between sample length and grade, composites were not created for resource estimation. Instead, grade interpolation was done directly with the drillhole assays.

For each population, the capping level was chosen so that the coefficient of variation (CV) of the capped assays was below 2., The capping value was above the 99th percentile of the gold assay grade distribution for each population. A 30 g/t Au cap was utilized for the gold assays in the strongly-metamorphosed population, which have a CV below 2 even without capping.

The block size used for the MRE is 50 x 50 x 20 ft; no sub-blocking was used. Estimation of the gold grade in each block was done in two steps:

1. The volume proportion of each metamorphic population is estimated using indicator kriging (IK); this gives p1, p2 and p3;
2. The gold grade of each population is estimated using ordinary kriging (OK) of the nearby samples that fall within that population; this gives Au1, Au2 and Au3.

The final block grade was the proportion-weighted average of the grades for each population:

$$Aublock = p1 \times Au1 + p2 \times Au2 + p3 \times Au3$$

The search strategy and variogram model for all the kriged estimates were the same. This avoided the problem of having proportions that did not sum to one, and the complication of the grade estimation for a particular population not being able to find nearby data when that population had a non-zero proportion estimate.

The variogram ranges were aligned with the search ellipsoid, both being 1,000 ft in the direction parallel to the stock and in the vertical direction, and 300 ft in the perpendicular direction, horizontal and radially outward from the stock. Blocks that could not be estimated in the first pass were picked up in a second pass, with the search ellipsoid doubled in size and the constraints on the number of samples relaxed.

Collocated co-kriging (Goovaerts, 1997) was used to avoid the problem of over-estimation of silver, copper and cyanide-soluble gold. Collocated co-kriging is an interpolation method that uses nearby data for the variable being estimated (the "primary" variable) and also incorporates into the estimation the value of another correlated variable (the "secondary" variable). The primary data occur at various locations nearby; the secondary data occurs exactly at the location being estimated, giving rise to the name "collocated".

Dry bulk density was measured on 312 whole-core samples from drillholes using Archimedes' Principle. Approximately ⅓ of the samples were coated in wax to preserve internal porosity and moisture. The uncoated and coated samples give identical averages, to the second decimal place, so the two populations of data were utilized together with no modifying factors.

The block model was classified into Measured, Indicated and Inferred Mineral Resources using a two-step procedure separate from grade estimation. In the first step, integer codes were assigned on a block-by-block basis, using the following criteria from the kriging of the population proportions:

- Blocks were assigned a 1 if they had data in at least four octants and if the average weighted variogram distance to the nearby samples was 33% of the variogram range, or less;
- Blocks were assigned a 2 if they had data in at least four octants and if the average weighted variogram distance to the nearby samples was 66% of the variogram range, or less;
- Blocks were assigned a 3 if they couldn't be assigned a 1 or 2.

The requirement of having data in at least four octants ensured that assays from at least two different drillholes were used to estimate grade. The criteria related to the average weighted variogram distance ensured that samples assigned a 1 were in areas with 200 ft drill spacing, and that samples assigned a 2 were in the areas with 400 ft drill spacing.

In the second step, the integer codes were smoothed by calculating a moving average within a 1,000 x 1,000 x 200 ft rectangular block, a volume which corresponded approximately to three months of mineralization and waste mining. The purpose of this smoothing step was to remove the small-scale chatter in the block-by-block codes and, in so doing, ensure that the classification provides meaningful information about the confidence in grade estimates at the scale of quarterly production.

The current mineral resource estimate for the Converse Property was constructed using gold price of US$1,750 per troy ounce, a lower grade of cutoff of 0.008 oz/ton (0.27 g/t) Au and is presented in Table 1.1.

Table 1.1. Converse Mineral Resource Statement based upon Open Pit Heap Leach.

	US UNITS			METRIC UNITS	
	Tonnage	Au Grade	Contained Metal	Tonnage	Au Grade
Classification	(Mtons)	(oz/ton)	(Moz Au)	(Mtonnes)	(g/t)
Measured	209.0	0.018	3.79	189.6	0.62
Indicated	80.6	0.017	1.38	73.1	0.59
Meas.+Ind.	289.6	0.018	5.17	262.7	0.61
Inferred	29.1	0.019	0.55	26.4	0.65

Notes:

1. *Mineral Resources have an effective date of 31 December 2020. Mr. Mohan Srivastava, of RedDot3D Inc., is the Qualified Person responsible for the Mineral Resource estimate.*
2. *Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.*
3. *Mineral Resources are the portion of the Redline North and Redline South deposits that have reasonable prospects for eventual economic extraction by open pit mining method and processed by gold heap leaching.*
4. *Mineral Resources are constrained oxide and sulfide mineralization inside a conceptual open pit shell. The main parameters for pit shell construction are a gold price of $1,750/oz gold, variable gold recovery for oxide, mixed and sulfide mineralization, open pit mining costs of $1.30/ton, heap leach processing costs of $4.80/ton, general and administrative costs of $0.29/ton processed, pit slope angles of 36 ° for alluvium and 41 ° below base of alluvium, and a 6.0% royalty.*
5. *Mineral Resources are reported above a 0.008 oz/ton(0.27 g/t) gold cut-off grade. This is a marginal cut-off grade that generates sufficient revenue to cover conceptual processing, general and off-site costs given metallurgical recovery and long-range metal prices for gold and silver*
6. *Units are imperial tons.*
7. *Numbers have been rounded as required by reporting guidelines and may result in apparent summation differences.*
8. *The QP is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, or political factors that might materially affect the development of these mineral resource estimates.*

1.9 Interpretations and Conclusions

Gold mineralization at the Property is hosted in two main deposits: Redline North and Redline South. The Mineral Resource Estimate described in this report was prepared by QP Mohan Srivastava, P.Geo. using Micromine 2020.5 software.

Using a 0.008 oz/ton (0.27 g/t) gold cut-off grade, Measured and Indicated Resources are estimated at 289.6 million tons grading 0.018 oz/ton (0.61 g/t) gold; and Inferred Resources are estimated at 29.1 million tons grading 0.019 oz/ton (0.65 g/t) gold. The MRE is constrained within an optimized pit shell wireframe that was generated using a gold price of US$1,750/oz, variable gold recovery for oxide, mixed and sulfide mineralization, open pit mining costs of US$1.30/ton, heap leach processing costs of US$4.80/ton, general and administrative costs of US$0.29/ton processed, pit slope angles of 36° for alluvium and 41° below base of alluvium, and a 6.0% royalty.

Several drillhole data verification campaigns have been undertaken with the most recent materially improving the auditability and quality. Drill programs completed at the Property between 1988 and 2017 have included QA/QC monitoring programs that have incorporated the insertion of CRMs, blanks and duplicates. Some concerns are observed in the QA/QC data but the APEX QPs are of the opinion these issues do not materially impact the global, long term MRE.

1.9.1 Risks

- Data used to inform the block model is historical in nature and incomplete records of original data result in some limitations during verification campaigns. Ongoing improvements should be made to verify the data as additional work is completed.
- The number of bulk density determinations used in the block model are moderate (312). Additional determinations may result in minor changes and impact the tonnage.
- Gold recoveries are affected by increased copper values hence the gold recovery model not only relies on a good oxidation model but also on a good estimation of copper values.
- Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is a degree of uncertainty attributed to the estimation of Mineral Resources. Until resources are actually mined and processed, the quantity of mineralization and grades must be considered as estimates only.

1.9.2 Opportunities

- Gold mineralization has been intersected towards the base of the alluvium and if estimated may result in a minor increase in tons and contained gold for the mineral resource.
- Improvements in the estimation of copper values may provide more gold ounces at better recovery values in the model.
- Resource blocks below the reporting pit shell had estimated gold grades above the reporting cut-off and were classified as Measured or Indicated after the smoothing step of the classification procedure, but do not fall within the reporting pit shell. These represent additional upside potential for resource growth if, in the future, new technical or economic parameters allow the conceptual pit to go deeper.

1.10 Recommendations

1.10.1 Mineral Resource Estimate

- Review and relog any available drilling materials or photos to replace the back-coded metamorphic intervals used in the estimate.
- Complete an analytical program to further investigate the observed bias in the silver and copper datasets due to different ICP assay methods.

1.10.2 Mineral Processing

- Investigate optimal mining scenarios and their impact on LOM capex and opex including at higher cut-offs/lower tonnages.
- Evaluate metallurgical sample coverage and grade within the optimized pit.
- Investigate HPGR tertiary crushing flowsheet with the objective of assessing potential throughput rates.

- Develop additional tests for HPGR tertiary crushing and evaluate impact on recovery.
- Complete metallurgical test work programs to support finer crush size processing options, including development of additional tests for HPGR tertiary crushing and evaluate impact on recovery.
- Evaluate opportunities to implement SART copper recovery methods and assess capital implications.
- Firm up recovery and consumables vs copper value correlations based on the additional tests results.

It is expected that the work programs would have a duration of 12 to 18 months with an estimated budget of $1.5 million.

2 Introduction

APEX Geoscience Ltd. ("APEX"), Ray Walton Consulting ("RWC") and RedDot3D Inc. ("RedDot3D") were engaged by Axcap Ventures Inc. ("Axcap" or "AVI") to prepare an independent Technical Report (the "Report") for Converse Resources LLC (CRL) on an updated Mineral Resource Estimate ("MRE") for the Converse Property located approximately 15.5 miles west of Battle Mountain, Nevada (Figure 2.1). Converse Resources LLC is a Reno, Nevada (NV), based private corporation, that is under a Letter of Intent ("LOI") to be purchased by Axcap.

The Technical Report has been prepared using the format specified in Form NI 43-101F1 to provide the results of an updated MRE for the Converse Property. The Technical Report has been prepared in accordance with the Canadian Securities Administration's (CSA's) NI 43-101 Standards of Disclosure for Mineral Projects and the Form NI 43-101F1 guidelines for technical reporting.

2.1 Terms of Reference

Axcap engaged APEX, RedDot3D and RWC to complete an independent Technical Report to assess the MRE at the Project and identify work required to advance the Project. The MRE was performed in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (November 2019) (the 2019 CIM Best Practice Guidelines) and reported in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves (10 May 2014) (the 2014 CIM Definition Standards).

Units used for the Project are US Customary units and a combination of imperial units (ounces per short ton) and metric units (parts per million or grams per tonne) for grade units. Precious metal values are typically specified in ounces per short ton (oz/ton) or grams per metric tonne (g/t) and the concentrations of other elements are stated in parts per million (ppm) or percent (%). The MRE was developed in Imperial Units for volume and tonnage using metric units for grade. The estimate was converted to metric units for tonnage for the Mineral Resource statement. Monetary units are in United States dollars.

The Converse Property is located in the Battle Mountain district, within the Battle Mountain-Eureka Trend, one of the main gold deposit trends in Nevada comprising a northwest-trending belt of precious metal deposits with current reserves and past production exceeding 50 million ounces ("oz") gold ("Au") (Holley et al., 2015).

The Property is located in the Buffalo Valley in the southeast corner of Humboldt County, NV, with a very minor part of the Property extending into Pershing County at the southwest corner (Figure 2.1). The Property lies approximately 10.5 miles (16.5 kilometres (km)) south of the Valmy exit on Interstate 80; approximately 15.5 miles (25 km) west-northwest of Battle Mountain and approximately 30 miles (48 km) southeast of Winnemucca. Reno is located approximately 205 miles (330 km) to the southwest by highway. The Property is located at approximately 40º43' N latitude and 117º16' W longitude. The Property location is shown in Figure 2.1. The area is relatively flat, with elevations ranging from 4,900 feet (ft) (1,500 metres (m)) to 5,900 ft (1,600 m) above sea level.

2.2 Authors and Site Inspection

The authors of this Technical Report (the "Authors") are as follows:

- Mr. Michael Dufresne, M.Sc., P.Geol., P.Geo, President and a Principal, APEX Geoscience Ltd. (APEX)

- Mr. Philo Schoeman, M.Sc., P.Geo., Pr.Sci.Nat., Senior Project Geologist (APEX)
- Mr. Mohan Srivastava, M.Sc., P.Geo. Senior Geologist (RedDot3D), and
- Mr. Ray Walton, P.Eng., President and Principal Metallurgist (RWC)

The authors are independent of the Issuer and are QPs as defined in NI 43-101. NI 43-101 and CIM define a QP as “an individual who is an engineer or geoscientist with at least five years of experience in mineral exploration, mine development or operation, or mineral project assessment, or any combination of these; has experience relevant to the subject matter of the mineral project and the technical report; and is a member or licensee in good standing of a professional association.”

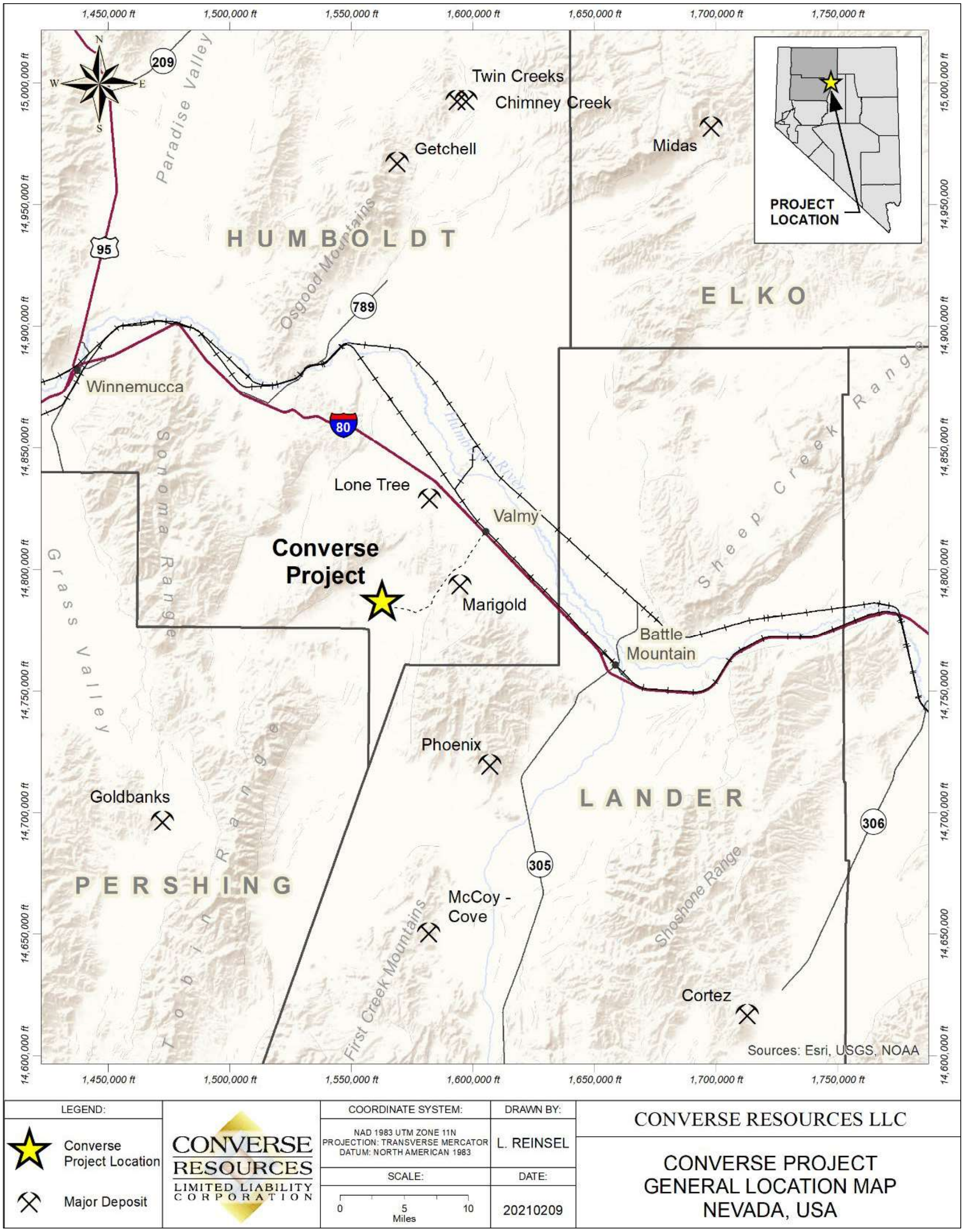
Mr. Dufresne is a Professional Geologist with the Association of Professional Engineers and Geoscientists of Alberta (“APEGA”; Member #: 48439), a Professional Geoscientist with the Engineers and Geoscientists of British Columbia (“EGBC”; Member #: 37074), the Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (“NAPEG”; Member #: L3378), the Association of Professional Engineers & Geoscientists of New Brunswick (“APEGNB”; Member #: F6534) and the Professional Geoscientists of Ontario (“PGO”; Member #: 3903), and has worked as a mineral exploration geologist for more than 40 years since his graduation from university. Mr. Dufresne has been involved in all aspects of mineral exploration and mineral resource estimations for precious and base metal mineral projects and deposits in Canada and globally and specifically for a number of intrusion and skarn related deposit types. Mr. Dufresne takes responsibility for Sections 1 to 11 and 15 to 28 of the Report. Mr. Dufresne has not visited the Property.

Mr. Schoeman is a Professional Geologist with the Association of Professional Engineers and Geoscientists of Alberta (“APEGA”; Member #161717) and a Professional Natural Scientist in the Geological Sciences with the South African Council for Natural Scientific Professions (registration # 400121/03). Mr. Schoeman has worked as a geologist for more than 34 years since his graduation from University and has been involved in all aspects of mineral exploration and evaluation for gold deposits in South Africa, Argentina, Ghana, Niger, Yemen, Canada and specifically on a number of projects with intrusion and skarn related deposit types. Mr. Schoeman takes responsibility for Section 12 of the Report. Mr. Schoeman also made contributions to and is jointly responsible for Sections 1, 11, 25, 26 and 28. Mr. Schoeman visited the Converse Property and the associated core facility from December 20 to 22, 2020. During the site visit Mr. Schoeman inspected collar locations, site access, drill core, drill logs, and assay certificates. Mr. Schoeman collected 15 check core samples during the site visit with the results presented in Section 12.

Mr. Srivastava is a Professional Geologist with the Association of Professional Geoscientists of Ontario (“PGO” Member #0547). He has worked as a geologist, geostatistician, and resource estimation specialist in mineral exploration for more than 40 years since he graduated from university. Mr. Srivastava has been involved in all aspects of mineral exploration and mineral resource estimations for precious and base metal mineral projects and deposits in Canada and globally and specifically for a number of intrusion and skarn related deposit types. Mr. Srivastava takes responsibility for Section 14, of the Report. Mr. Srivastava also made contributions to and is jointly responsible for Sections 1, 25, 26, and 28. Mr. Srivastava last visited the Property in 2005. Mr. Srivastava has not completed a site visit since then due to a combination of COVID restrictions and personal health reasons.

Mr. Walton is a Professional Engineer with the Professional Engineers of Ontario (“PEO” Member #90294521). He has worked as a process metallurgical engineer in the mining sector for more than 47 years since his graduation from university. Mr. Walton has been involved in process design and engineering, project management and the commissioning of metallurgical plants for gold and copper projects around the world. Mr. Walton takes responsibility for Section 13, of the Report. Mr. Walton also made contributions to and is jointly responsible for Sections 1, 25, 26, and 28. Mr. Walton has not visited the Property.

Figure 2.1. Converse location map.



Source: CRL (2021)

2.3 Sources of Information

Key sources of information include the drillhole database and metallurgical test work reports. A list of all information sources used in completing this Report are included in Section 27. This Report summarizes an updated MRE for CRL and the Converse Property. The Company and its wholly-owned subsidiary Chaparral Gold Corp. ("CGC") have been actively exploring the Converse Property, primarily for precious metals, since CGC was spun out from Hochschild Mining PLC in 2013 ("Hochschild"). The most recent exploration conducted on the Property by CRL was from 2017 to 2019 and included a seven hole core drilling program, metallurgical studies, rebuilding of the geological model, environmental studies and permitting.

This Report is a compilation of Company and publicly-available information. Data required for the execution of this report were obtained from CRL in paper and digital format and were the subject of a data validation process conducted by the QPs. These and other important sources of information are documented in Sections 6, 7, 9 to 14 and in Section 27 of the Report. A large portion of the data presented in the Report, notably in Section 6, is historical in nature and was collected prior to CRL's Property ownership. Previous NI 43-101 Technical Reports authored by Srivastava et al. (2012), Sullivan et al. (2004) and Muerhoff et al. (2002) provided the majority of the background information used to prepare the Report. Other sources of information were provided to the authors by CRL. All sources are summarised in Section 27. This Report includes all known and available technical data and information known to CRL and reviewed by the QPs as of the effective date of the Report. The QPs are unaware of any material technical data other than that provided by CRL and reviewed and presented by the authors herein.

In support of the technical sections of this Report, the authors have independently reviewed reports, data, and information derived from work completed by CRL and their consultants. Journal publications listed in Section 27 "References" were used to verify background geological information regarding the regional and local geological setting and mineral deposits of the Property. The authors have deemed these reports, data, and information as valid contributions to the best of their knowledge. Based on the Property visit and review of the available literature and data, the authors take responsibility for the information herein.

Information pertaining to Property ownership and mineral tenure was derived from Converse, and, where possible, was checked against the United States Bureau of Land Management (BLM) mining claim register (MLRS) as recently as October 31, 2024.

2.4 Units of Measure

With respect to units of measure, unless otherwise stated, this Report uses:

- Geographic coordinates are projected in an established local grid known as "CNVS16" developed by previous operators and surveyor Loyal D. Olsen. The projection uses on-site controls and is based the Universal Transverse Mercator ("UTM") system relative Nevada State Plane Coordinates (NAD83) Central Zone; (Olsen, 2008). Local projection "CON3" was renamed to "CNVS16" when a review was completed by CRL. No changes were made.
- Currency in United States dollars (USD$), unless otherwise specified.

Abbreviations used in the report are presented in Table 2.1.

Table 2.1. Abbreviations used in the report

acres	ac	milliliter	mL
cubic feet	ft^3	millimeter	mm
degrees	°	Million ounces	Moz
degrees Fahrenheit	°F	Million tonnes	Mt
feet	ft	Million tons	Mton
gram	g	million years ago	Ma
gram per liter	g/L	meter	m
gram per tonne	g/t	parts per million	ppm
inch	in	percent	%
kilograms per tonne	kg/t	Troy ounce (12 oz to 1 pound)	oz
micron	µm	Troy ounce per ton	oz/ton
mile	mi	weight percent	wt %

3 Reliance on Other Experts

The QPs have relied upon the following other expert reports, which provided information regarding property claim tenure, property contracts and agreements, and environmental liabilities and permits.

3.1 Property Claim Tenure and Agreements

The authors relied on CRL to provide all pertinent information concerning the legal status of the Company, as well as current legal title, material terms of all agreements, and tax matters that relate to the Property. Copies of documents and information related to legal status, property agreements, and mineral tenure were reviewed, and relevant information was included elsewhere in the Report; however, the Report does not represent a legal, or any other, opinion as to the validity of the agreements or mineral titles. The following documents and information, provided by CRL, were relied upon to summarize the legal status and mineral tenure status of the Property. The land position was provided by CRL and was compared with and confirmed using title opinion letters provided by attorneys:

- Parr, Brown, Gee & Loveless (June 26, 2020). Title Report Update [letter to Converse Resources LLC], 8 pp.
- Parr, Brown, Gee & Loveless (September 9, 2020). Title Report [letter to Converse Resources LLC], 12 pp.

Mr. Dufresne and Mr. Schoeman checked the status of all of the BLM mining claims using the BLM's MLRS database and service during October, 2024. The mineral claims were all active with the BLM indicating that maintenance payments were up to date and not due until September 2, 2025.

3.2 Environmental Liabilities and Permits

CRL has assisted with and has provided much of the background information for Section 4.4 "Environmental Liabilities and Permits". This information has been reviewed and confirmed with information available.

4 Property Description and Location

4.1 Description and Location

The Property is located in Buffalo Valley, in the southeast corner of Humboldt County, NV, with a very minor part of the Property extending into Pershing County at the southwest corner of the Property (Figure 2.1). The Property lies ~10.5 miles south of the Valmy exit on I-80; ~15.5 miles west-northwest of Battle Mountain and ~30 miles southeast of Winnemucca. Reno is located approximately 205 miles to the southwest by highway. The Property is located at roughly 40°43' N latitude and 117°16' W longitude, within Township 33 North, Range 42 East.

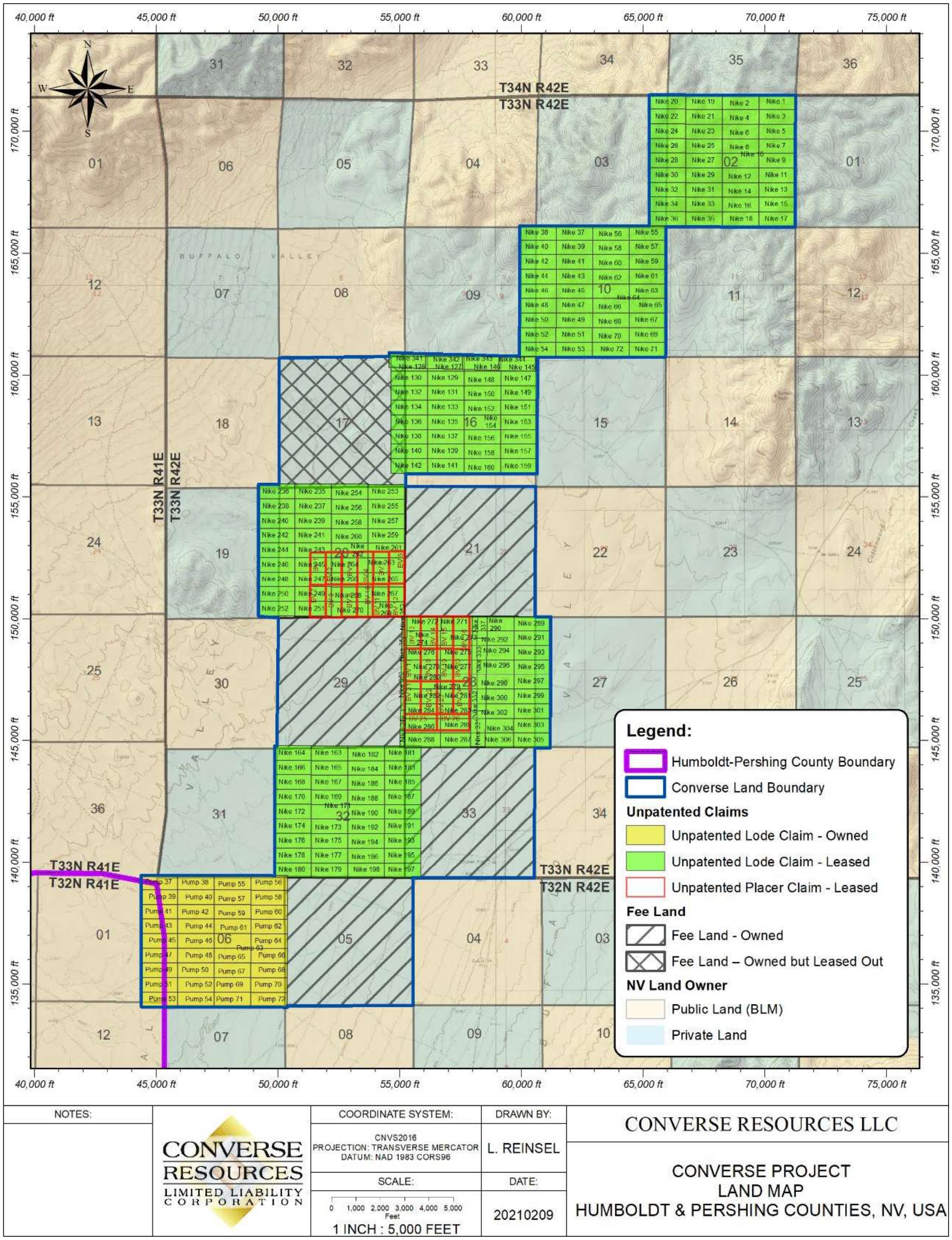
4.2 Mineral Rights and Tenure

The Property consists of 286 unpatented mining claims (the "Claims") located on federal land administered by the BLM and five privately-owned tracts of land (the "Fee Land"). The total land area covered by the Property is approximately 7,784 acres. The Claims are located within Sections 1 and 12, T32N, R41E, MDM, Pershing County, Sections 6 and 8, T32N, R42E, MDM, Humboldt County, Section 36, T33N, R41E, MDM, Humboldt County, and Sections 2, 4, 8, 10, 16, 18, 20, 22, 28, 30, 32 and 34, T33N, R42E, MDM, Humboldt County, Nevada. A Property map showing the land ownership, the Claims and the Fee Land in the area controlled by CRL is presented as Figure 4.1. A detailed list of the Claims is provided in Appendix 1.

Most (260) of the claims are unpatented mining lode claims. The remainder (26) are unpatented placer claims. Unpatented mining claims (both lode and placer) are created and maintained in accordance with the U.S. General Mining Law of 1872 as amended. While the United States retains title to the land within an unpatented mining claim, the claimant, through compliance with federal and state laws, has the right to explore for and exploit certain minerals (including precious metals) within that land. Therefore, no specific mining or exploitation license or permit is required to hold the claims, as is the case in most jurisdictions outside the United States. However, exploration and mining operations must have a variety of other permits, as discussed below in Section 4.3. An individual unpatented lode claim is limited to a maximum of 20.66 acres. The Property's lode claims are generally of that maximum size. An unpatented placer claim can range from 20 to 160 acres in size. Each of the Property's placer claims is 20 acres in size. An annual fee, payable to the BLM, is required to maintain the Claims from year to year. At present, the fee is $200 per unpatented mining claim. Currently there are no federal royalties imposed on mineral production from unpatented mining claims.

Converse owns 36 of the Claims; Pump Claims 37 to 72 (Appendix 1). The other 250 Claims (the "Leased Claims") are leased by Converse from their owner, Nevada North Resources (U.S.A.), Inc., pursuant to an Amended and Restated Mining Lease dated March 29, 2013, but effective August 31, 2012 (the "Converse Lease"). The Converse Lease gives Converse the right to explore, develop and mine on the Leased Claims. The Converse Lease has an initial term of 10 years (until August 31, 2022), after which the term may be extended by Converse until August 31, 2032, and continue so long thereafter as there is development, mining, processing, reclamation or closure activities occurring on the Leased Claims. The term has been extended.

Figure 4.1. Property map for the Converse Project.



Source: CRL (2021)

The Converse Lease requires annual advance royalty payments of US$50,000, US$75,000 or US$100,000 depending on the average gold price for the 12-month period ending July 31 of each year. At current gold prices, the annual advance royalty payments due under the Converse Lease are US$100,000. In addition, the Converse Lease subjects the Leased Claims to a sliding scale 3% to 5% net smelter returns ("NSR") royalty on all minerals produced from the Leased Claims, indexed to the price of gold (the "Leased Claims Royalty"). At current gold prices (at or above US$375 per ounce), the Leased Claims Royalty rate is 5%. Converse has paid approximately US$2,060,000 in annual advance royalties, all of which can be credited in full toward future Leased Claims Royalty payments with production.

The Fee Land comprises a total of 3,195.64 acres, with each of the five tracts of Fee Land covering approximately one square mile or 640 acres. The Fee Land is private property in which the United States government holds no ownership interest. Converse owns 100% of the surface and mineral rights to all of the Fee Land. However, one of the five Fee Land parcels (Section 17, Township 33 North, Range 42 East, MDM) is presently leased to Nevada Gold Mines LLC and as a result is not currently available for exploration or mining by Converse. A description of the Fee Land is presented in Table 4.1. The annual property tax cost for the Fee Land is about US$3,579 per year.

Table 4.1. Fee Land.

Section	Township	Range	Description	Acres	APN #	% SURF	% MIN
5	32	42	All	635.64	07-0481-02	100	100
17	33	42	All	640	07-0451-14	100	100
21	33	42	All	640	07-0451-21	100	100
29	33	42	All	640	07-0451-26	100	100
33	33	42	All	640	07-0451-33	100	100

The current land holding costs of the Property total approximately US$169,964 annually (including the annual advance royalty payment due under the Converse Lease, as well as BLM maintenance fees and county recording fees for the Claims and property taxes on the Fee Land).

4.3 Royalties and Agreements

As discussed above, the Leased Claims are subject to the Leased Claims Royalty on all minerals produced from those claims. The current Leased Claims Royalty rate is 5% (of NSR), and that rate will not change unless the price of gold drops to less than US$375 per ounce.

Newmont USA Limited is entitled to a gold price-related sliding scale NSR royalty of 3% to 5% on the production of gold, and 3% on the production of other minerals, from four of the five Fee Land parcels (those being Section 5, Township 32 North, Range 42 East, MDM and Sections 21, 29 and 33, Township 33 North, Range 42 East, MDM). At gold prices of US$400 per ounce and above, the royalty rate is 5%, such that the current NSR royalty rate on gold is 5%.

Nevada Land and Resource Company, LLC is entitled to a 1% NSR royalty on the production of minerals from one of the Fee Land parcels (that being Section 17, Township 33 North, Range 42 East, MDM). However, as explained above, Section 17 is currently leased to Nevada Gold Mines LLC and this royalty is therefore not applicable to Converse's present development state.

The entire Property is subject to a NSR production royalty payable to Royalty Consolidation Company, LLC on the sale of any minerals from the Property (the "RCC Royalty"). The RCC Royalty rate is 6%, except as to those portions of the Property that were subject, as of the date of the RCC Royalty grant, to existing royalty obligations, in which case the RCC Royalty rate is the difference between 6% and the rate of the existing royalty obligations. Effectively, the RCC Royalty means that the Property is subject to a blanket 6% NSR royalty on the production of all minerals.

4.4 Environmental Liabilities, Permitting and Significant Factors

The Converse Deposit is a greenfield site. All exploration, development and production activities are subject to regulation under one or more of the various state and federal environmental laws and regulations. Many of the regulations require CRL to obtain permits for its activities. CRL must update and review its permits from to time to time and may be subject to environmental impact analyses and public review processes prior to approval of any additional activities. CRL expects to make in the future significant expenditures to expand the scope of its current permits.

Exploration of the Converse Property is carried out under an Exploration Plan of Operations NVN065461, approved by the BLM pursuant to Environmental Assessment N20-98-001P and Reclamation Permit #0122 approved by the Nevada Division of Environmental Protection. There is a US$56,330 reclamation bond currently associated with the existing permits.

In 2019, CRL purchased 2,560 acre-ft of irrigation water rights from New Nevada Lands, LLC (Permit 71715 and 71716). Once converted to mining and milling use, the acquired water rights will support the construction and operation of a future mine at the Property. An application requesting a change in the water rights' point of diversion, place of use and manner of use was submitted to the Nevada State Engineer on October 29, 2020. The change has since been granted.

There are no other significant factors or risks that the QPs are aware of that would affect access, title or the ability to perform work on the Property.

5 Accessibility, Climate, Local Resources, Infrastructure, and Physiography

5.1 Accessibility

The Converse Property is located 21 miles northwest of Battle Mountain and 45 miles southeast of Winnemucca by road in northwestern Nevada. A well-maintained county gravel road leads 10.3 miles from Exit 216 (Valmy) on the I-80 Highway (Figure 5.1), past the Marigold mine and the turnoff to the Trenton Canyon mine, both of which are near the Property. Further access to the Property is by unimproved dirt roads and tracks.

5.2 Climate

The climate of the Property area is semi-arid, characterized by warm, dry summers, and moderately cold, dry winters. Overnight freezing conditions are common during winter. The mean annual high temperature is approximately 67°F and the low is 37°F. Precipitation averages 9.0 in per annum, the majority comes in the form of winter snow or from infrequent summer thunderstorms. The average annual snowfall is 19 in. Prevailing winds are from the south-southwest in the region. The Property has no significant climatic issues and thus work can be completed throughout the year.

5.3 Physiography

The Property is located in a relatively flat valley with Buffalo Mountain to the west-northwest, the Havallah Hills to the north and the Battle Mountain Range to the southeast. The Property covers a gently southwest-sloping alluvial plain with elevations ranging from 4,900 to 5,200 ft above sea level (topographic relief is approximately 300 ft). Vegetation consists of sagebrush and desert grasses, with areas of desert hardpan without vegetation.

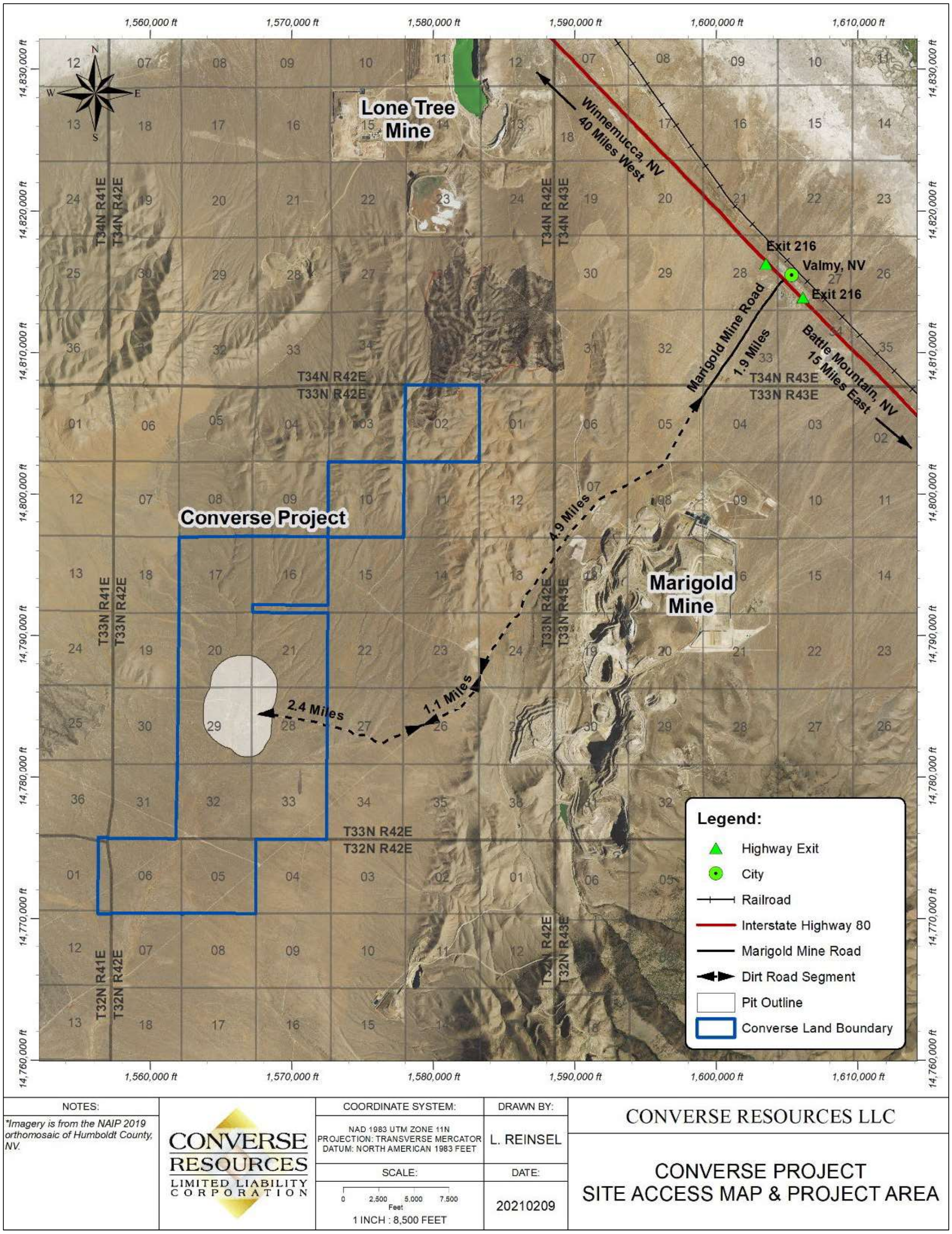
5.4 Local Resources and Infrastructure

The Property lies near to the regional population centers of Battle Mountain (18 miles to the east) and Winnemucca (30 miles to the west) where commercial services, suppliers, accommodation and medical facilities are available. In addition, both of these communities are in close proximity to and support numerous active mining operations, and thus are a potential source of experienced labour as well as mining industry-specific support services.

Electricity and natural gas infrastructure do not currently exist on the Property. NV Energy operates the main powerline that runs along the I-80 corridor, from where power could be obtained. Water required for exploration drilling is supplied by the nearby Marigold Mine owned by SSR.

In the opinion of the QPs, the Property is of sufficient size to accommodate potential exploration and mining facilities, including waste rock disposal and processing infrastructure. There are no other significant factors or risks that the QPs are aware of that would affect access or the ability to perform work on the Property.

Figure 5.1. Converse site access and Property area.



Source: CRL (2021)

6 History

6.1 Historical Work at the Converse Property (1988 to 2014)

Exploration at Converse is summarized in Table 6.1. Descriptions of recent exploration activity and drilling results are provided in Sections 9 and 10 of this Report, respectively. Historical exploration up to 2011 is summarized from a previous Technical Report on the Property (Srivastava et al., 2012). There has been no production from the Property.

Table 6.1. Converse Property area ownership and exploration summary (1988 to 2014).

Operator	Period	Comments
Nevada North Resources	1988	Staked 315 unpatented lode mining claims, known as the "Nike Property".
Kennecott Minerals Co. (Kennecott)	1988	Completed two RC holes totaling 585 ft.
Chevron Resources (Chevron)	1989-1991	Optioned the Nike Property and carried out reconnaissance geological mapping, geochemical sampling, gravity and gradient array induced polarization (IP) surveys Completed 11 RC holes totaling 4,810 ft, of which three holes all failed to reach bedrock.
Cyprus Mines Corp. (Cyprus)	1991-1992	Acquired the option from Chevron and carried out limited geological mapping and acquired ground magnetic data. Completed 15 RC holes, totaling 4,070 ft, with anomalous gold values intercepted in bedrock material.
Independence Mining Co.	1993-1994	Leased the Property from Nevada North Resources and carried out a bulk leach extractable gold (BLEG) survey. Completed ten mud-rotary holes, totaling 4,950 ft, and intersected significant mineralization in two holes.
Uranerz U.S.A. Inc. (UUI)	1994	Executed a lease agreement for the Property with Nevada North Resources.
Romarco Nevada Inc. (Romarco)	1995	Executed a joint venture (JV) agreement, known at the Nike Venture, with UUI, who remained the operator, on a 50%-50% basis. Completed further gravity surveys as well as an enzyme leach geochemical survey. Completed ten mud-rotary holes, totaling 4,760 ft.
	1996	The Nike JV staked 36 unpatented lode claims. The Nike JV entered into an exploration agreement, known as the Converse Agreement, with Santa Fe Pacific Gold (SFPG, now Newmont), whom leased adjacent fee land. Sixteen RC holes totaling 10,465 ft, six core holes with RC pre-collars totaling 6,440 ft were completed.
	1997	Newmont Gold Corp acquired SFPG. Thirty RC holes totaling 21,148 ft and three core holes with RC pre-collars totaling 3,611 ft were completed. Initial metallurgical test work was completed. A resource estimate was completed by the Nike JV.

Operator	Period	Comments
	1998	Cameco acquired UUI and changed names to UUS Inc. (UUS). Fifty-two RC holes (totaling 42,012 ft) were completed and further metallurgical test work carried out.
	1999	Fifteen RC holes 9,335 ft were completed. An updated historical mineral estimate was generated by the Nike JV.
Metallic Ventures Gold Inc. (MVG)	2001	Romarco NV was acquired by Metallic Ventures Gold Inc. (MVG).
	2002	Romarco NV acquired USS' interest in the Nike JV and Converse Agreement, as well as acquired Newmont's interest in the Converse Agreement. Mine Development Associates (MDA) generated an updated historical mineral estimate and NI 43-101 technical report.
	2003	Zonge Geoscience completed three-line miles of Controlled Source Audio-frequency Magnetotellurics (CSAMT) on the Property. Eighteen RC holes (totaling 14,988 ft) and eight core holes with mud-rotary pre-collars (totaling 5,307.2 ft) were completed.
	2004	Twenty-eight RC holes (totaling 24,622.5 ft) were completed. Metallurgical test work at Kappes Cassiday and Associates (KCA) was initiated. Watts Griffis & McOuat (WGM) completed an updated historical mineral estimate and NI 43-101 technical report.
	2007	Fifty-three RC holes (totaling 37,480 ft) and eight core holes (totaling 7,332.2 ft) were completed.
	2008-2009	Metallurgical test work at McClelland Laboratory Inc. (MLI), Reno, and geotechnical evaluations were completed. FSS Canada generated an updated historical mineral estimate.
International Minerals Corp. (IMC)	2010	MVG was acquired by International Minerals Corp. (IMC).
	2011	Eight core holes (totaling 13,945.5 ft) and six core holes with RC pre-collars holes (totaling 7,700.4 ft) were completed.
	2012	Four core holes (totaling 5,028.6 ft) and 10 core holes with RC pre-collars holes (totaling 16,064.2 ft) were completed. Micon completed an updated historical mineral estimate and Preliminary Economic Assessment on behalf of IMC.
Chaparral Gold Corp. (Chaparral)	2013	IMC was acquired by Hochschild Mining plc and the Converse Property along with the other Nevada assets were spun out into Chaparral Gold Corp.
	2014	R. Mohan Srivastava completed an updated historical mineral resource estimate.
Converse Resources LLC (CRL)	2014	Chaparral was acquired by CRL (through Waterton Global Resource Management).
	2017	Completed seven core drillholes on the Property totalling 5,944 ft for metallurgical purposes.
	2018	Completed metallurgical test work that included bottle rolls, agglomeration and compaction tests and column leach tests.
	2019	CRL purchased 2,560 ac-ft of irrigation water rights from New Nevada Lands, LLC (Permit 71715 and 71716).

7 Geological Setting and Mineralization

7.1 Regional Geology

The Converse Property is located in the Battle Mountain district, within the Battle Mountain-Eureka Trend, one of the main gold deposit trends in Nevada comprising a northwest-trending belt of precious metal deposits with current reserves and past production exceeding 50 million oz Au (Holley et al., 2015). The regional geological setting and history of north-central Nevada, including the Battle Mountain-Eureka Trend, has been well documented by several authors. The following section on the geological and tectonic history of north-central Nevada has been summarized from reports by Breit et al. (2015), Cleveland (2000), Cline et al. (2005), Fithian (2015), Leonardson (2015), Price (2010) and Wallace et al. (2004).

7.2 Geological and Tectonic History of North-Central Nevada

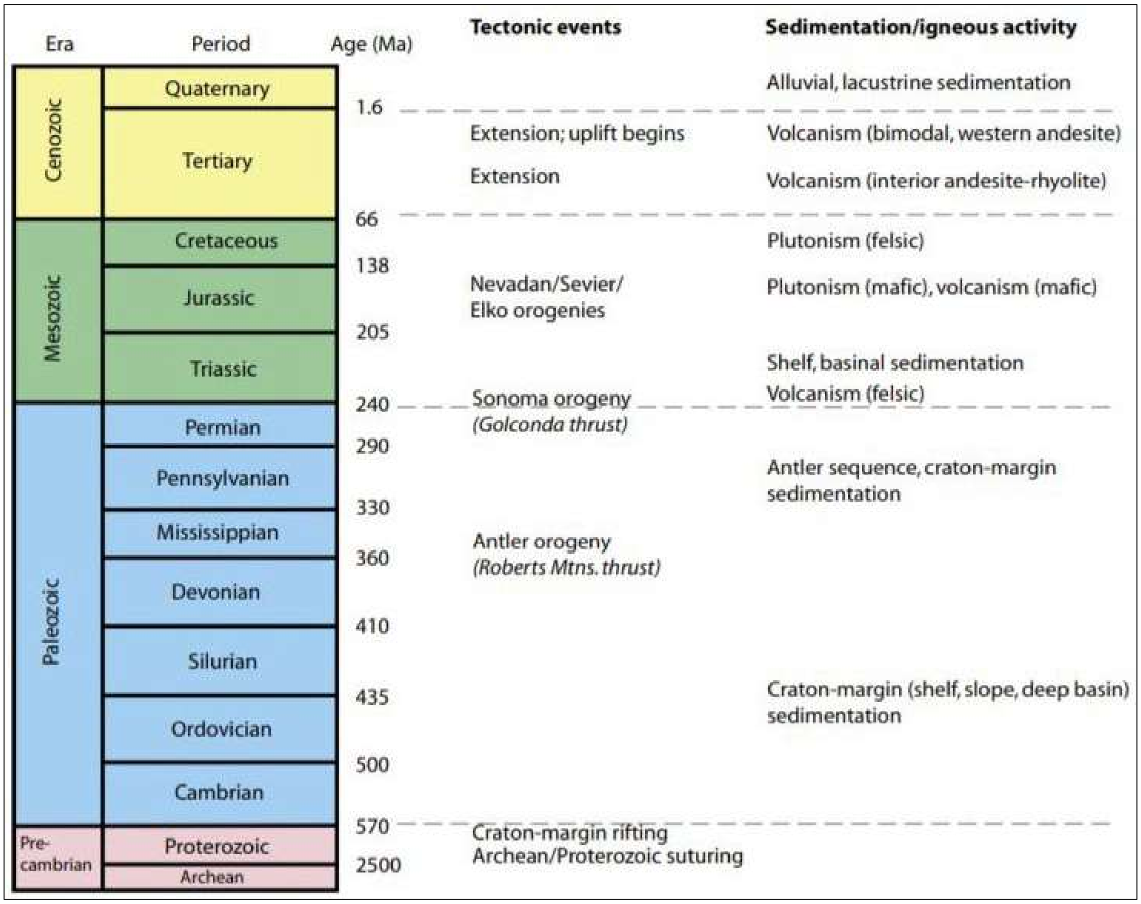
Paleoproterozoic terranes were accreted to the Wyoming craton during the assembly of Laurentia (Cline et al., 2005) forming several northwest- and north-striking faults. The Wyoming craton became the future Cheyenne Lineament, the most significant structural suture zone and mobile belt in Nevada (Leonardson, 2015) and host to the most significant known Carlin-type deposits. Rifting in the Meso- and Neoproterozoic resulted in a westward-thinning margin of continental crust as Laurentia separated from an adjoining crustal block (Cline et al., 2005). A westward-thickening sedimentary sequence was deposited in the early Paleozoic along the edge of the North American craton as indicated by Stewart, 1972 and Poole et al. 1992 (cited in Cline et al., 2005; Wallace et al., 2004).

The Roberts Mountain Thrust Formed during the Devonian to early Mississippian Antler orogeny with marine rocks thrust over the miogeoclinal shelf sequence (as indicated by Roberts et al. and cited in Leonardson, 2015; Cline et al., 2005; Wallace et al., 2004). The Antler orogeny continued up until the Permian. The Golconda allochthon was emplaced during the Sonoma orogeny in the late Permian to early Triassic; deep Paleozoic sediments were thrust eastward over rocks of the Roberts Mountains thrust (Siberling and Roberts, 1962). During the Antler and Sonoma orogeny's, deformation regressed to the west as major thrust plates were emplaced in the region of prior thrusting (Price, 2010). An east-dipping subduction zone formed along the western margin of North America by the Middle Triassic (Cline et al., 2005).

Regarding magmatism, north-central Nevada magmatism commenced in the Middle Jurassic with back-arc volcanic-plutonic complexes and lamprophyre dykes. Lipman et al. and Hickey et al. (2003a and b) indicate that the magmatism shifted into Colorado at approximately 65 Ma and did not resume in Nevada until approximately 42 Ma (Cline et al., 2005).

A timeline of the major geological and stratigraphic events in northern Nevada is shown in Figure 7.1 and Figure 7.2.

Figure 7.1. Timeline of geological events of northern Nevada.



Source: Modified from Wallace et al., (2004)

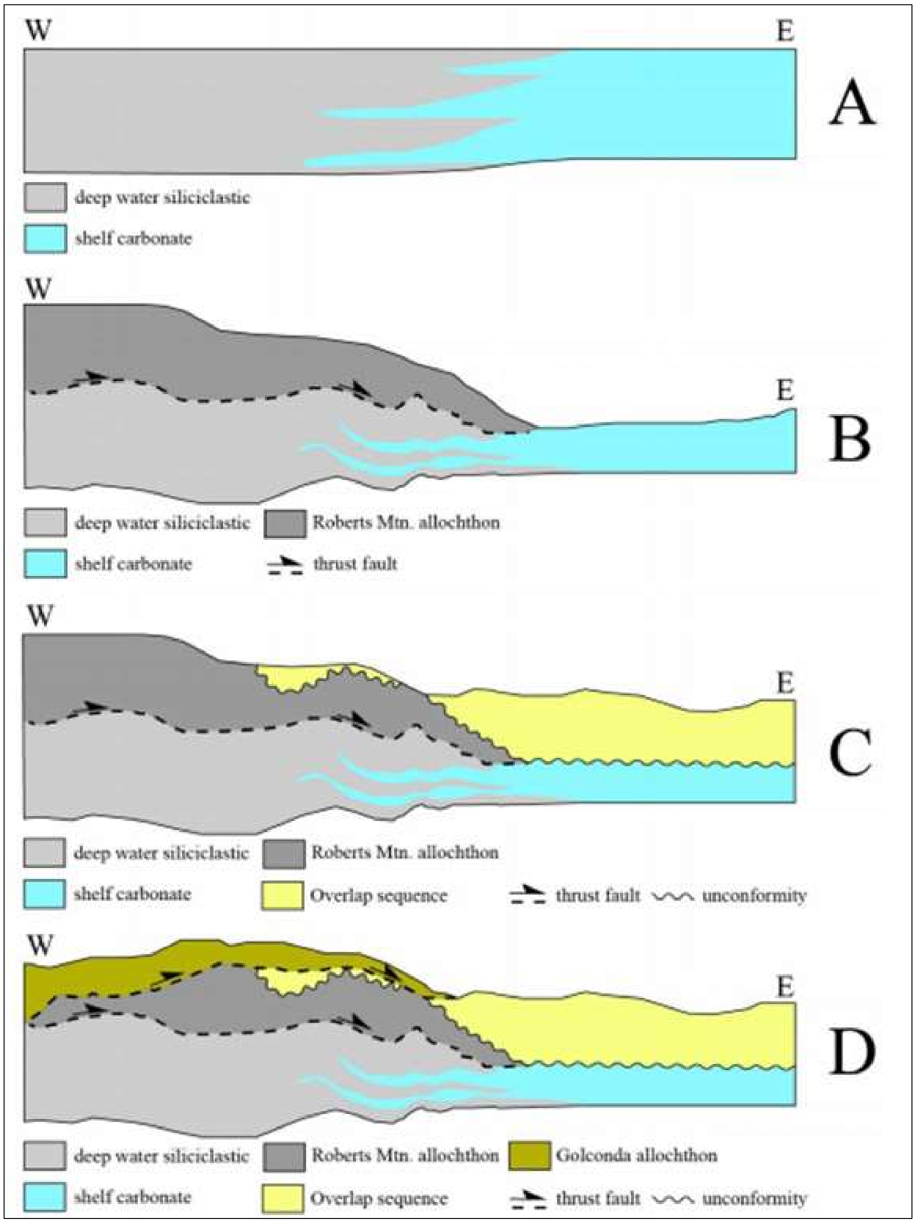
7.3 Regional Geology of Battle Mountain Mining District

The regional geology of the Battle Mountain Mining District comprises three Paleozoic rock assemblages (Figure 7.3), summarized from Yennamani (2010), as follows:

- Ordovician, Silurian and Devonian aged siliceous sequence of the Roberts Mountains allochthon.
- Mississippian, Pennsylvanian and Permian Havallah sequence of the Golconda allochthon.
- Pennsylvanian-Permian Antler sequence of the Antler orogeny.

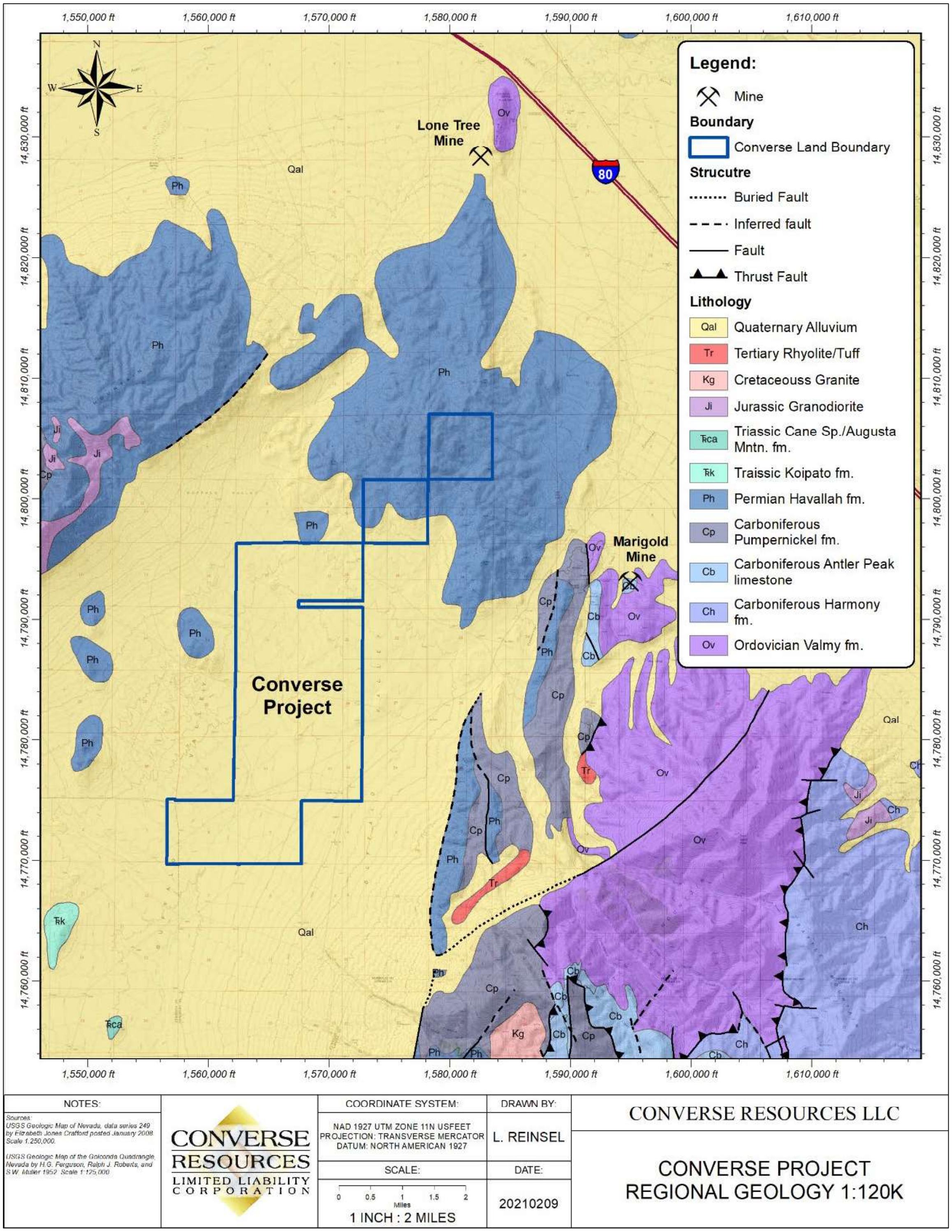
The Property region is underlain by the structurally complex Havallah sequence that comprises the upper plate of the Golconda allochthon (Cleveland, 2000). The Havallah sequence consists of basinal sedimentary rocks, including calcareous sandstone and siltstone, quartzite, pebbly limestone, siliceous siltstone, chert, argillite and variable amounts of basalt and greenstone. The Havallah sequence overlies the Antler sequence, that includes the Middle Pennsylvanian Battle Formation, the Pennsylvanian and Permian Antler Peak Limestone and the Permian Edna Mountain Formation. The Antler sequence overlies the siliciclastic sedimentary rocks of the Roberts Mountains allochthon, including the Cambrian(?) Harmony Formation, Ordovician Valmy Formation and Devonian Scott Canyon Formation. A stratigraphic column of the Havallah sequence is shown in Figure 7.4.

Figure 7.2. Tectonostratigraphic events of northern Nevada during the: A) Devonian, B) Devonian to Mississippian, C) Mississippian to Permian, and D) Permian to Triassic.



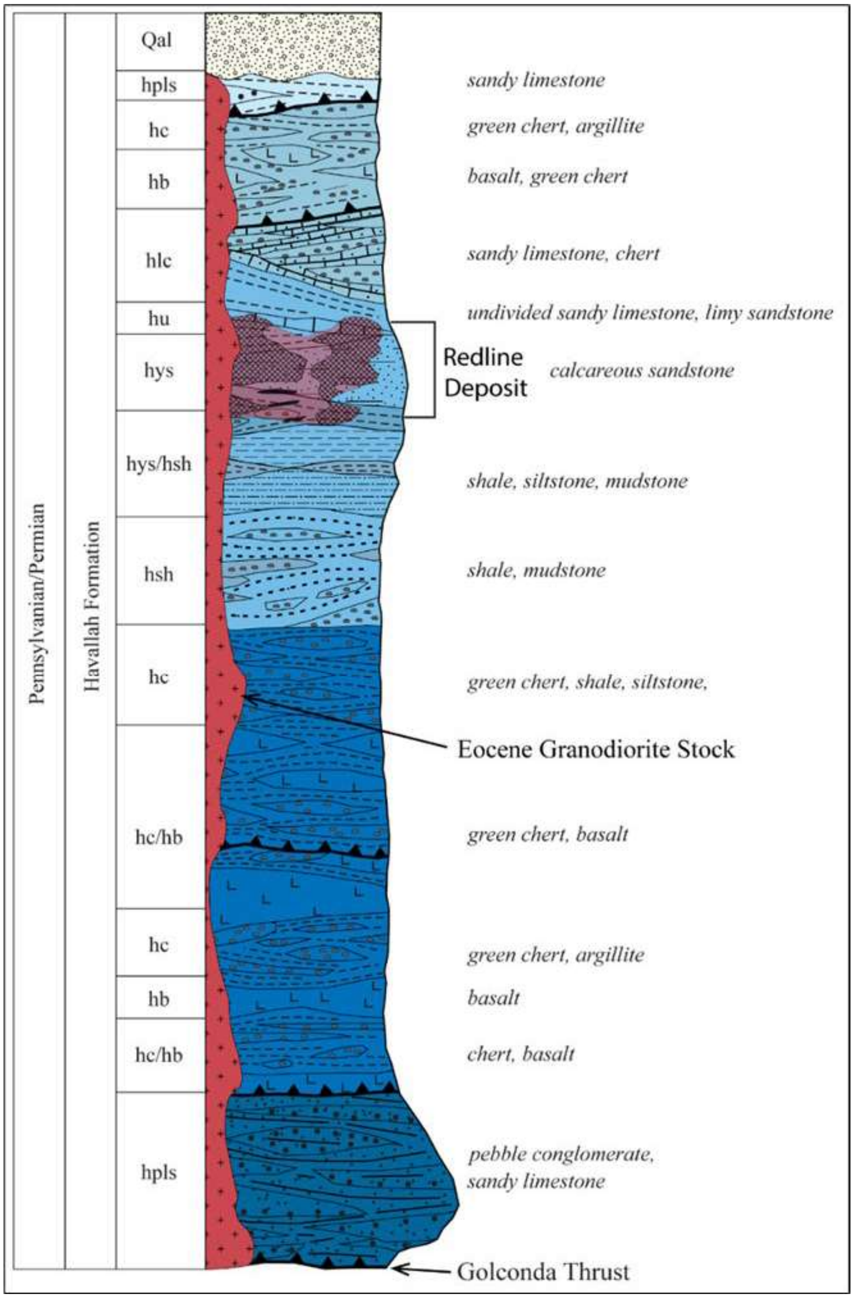
Source: Fithian (2015)

Figure 7.3. Converse Property regional geology.



Source: CRL (2021)

Figure 7.4. Stratigraphic column of the Havallah sequence.



Source: Modified from Bloomstein et al., (1993); Roberts and Arnold, (1965)

7.4 Property Geology

The following description of geology and mineralization at the Converse Property has been adapted or taken directly from previous studies or Technical Reports written on the Property by Cleveland (2000) and Srivastava et al. (2012).

A calcareous sandstone-rich unit of the Havallah sequence predominantly underlies the alluvium (Figure 7.5). This unit measures greater than 1,800 ft in thickness and includes interbedded sandy to pebbly limestone, calcareous to siliceous siltstone, chert and argillite of turbiditic origin. The calcareous sandstone-rich unit dips to the west at 20° to 35°, hosts the Redline deposits and correlates with subunit “hys” shown on Figure 7.4. A unit of siltstone, argillite and chert with interbedded calcareous sandstone turbidite layers underlies the calcareous sandstone-rich unit. This unit is at least 400 ft thick and, based on drilling, lies at the eastern boundary of the Redline deposits.

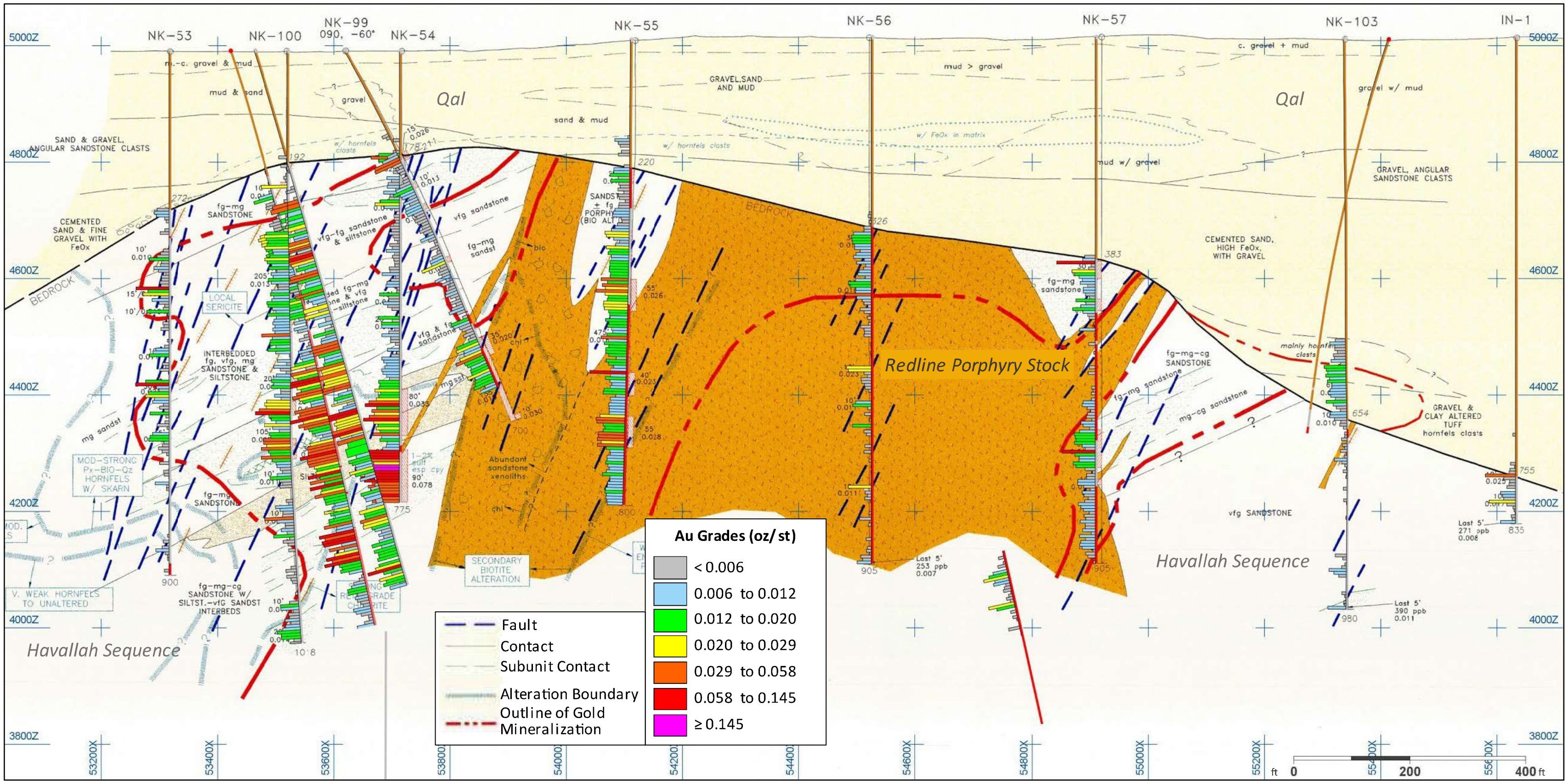
The Havallah sequence is intruded by a porphyry stock and related dykes and/or sills (Figure 7.5). The intrusive is known as the Redline porphyry stock and comprises phenocrysts of plagioclase, hornblende, and minor quartz and biotite in a fine-grained quartzofeldspathic groundmass. The composition of the stock varies from dacite to granodiorite, and grades into tonalite at its southern contact. The stock diameter measures from 1,475 to 1,970 ft and its estimated age is Tertiary, based on regional correlations and a 41 Ma rhenium-osmium (Re-Os) age date of molybdenite (Cleveland, 2000).

Alluvial material, predominantly Quaternary and late Tertiary in age, covers much of the deposit area and ranges in thickness from 20 ft to over 900 ft (Figure 7.3 and Figure 7.5). The alluvial material comprises sand, sandy gravel, pebble-gravel and discontinuous deposits of sand, silt and mud. Clast composition of the alluvial material includes Havallah sequence rocks and minor amounts of intrusive and volcanic clasts.

7.5 Structure

Fault zones identified within the Property are high-angle (Figure 7.5), interpreted to strike predominantly to the north and include pre-, syn- and post-mineralization faulting. Second-order west- to northwest- and northeast-striking faults are also inferred from drill logging and interpretations. Slickensides observed in core indicate strike-slip and dip-slip movements on individual faults. Cleveland (2000) suggests that the Redline stock was emplaced at the intersection of a pre-existing northwest-trending structural zone with one or perhaps two north- to north-northeast striking faults. The stock is situated along a northwest-trending aeromagnetic high and may represent the apical part of an intrusive complex that underlies this magnetic anomaly.

Figure 7.5. Cross-section with lithologic units, interpreted high-angle faults and outline of gold mineralization. Section 148200N looking to the north and width of window is 100 ft.



Source: CRL (2021)

7.6 Metamorphic/Alteration

Two main styles of metamorphic/alteration assemblages are observed at the Property and include early prograde hornfels-skarn and late retrograde skarn assemblages. Skarn assemblages are developed within both the intrusive unit (endoskarn) and Havallah units (exoskarn).

Prograde metamorphism includes an early biotite-rich hornfels that is overprinted by calc-silicate skarn assemblage. The prograde envelope extends ~1,500 ft laterally to the north and south from the central granodiorite porphyry. Mineral assemblage and intensity are strongly controlled by protolith interaction and relative distance to the central intrusive body. Replacement of the original host minerals varies from the presence of minor disseminated and veinlet biotite to complete replacement by garnet and pyroxene skarn. The more calcareous sandstones commonly contain a higher percentage of calc-silicate skarn minerals, such as diopside, garnet and Ca-plagioclase, whereas clay-rich siltstones are altered to a biotite-potassium (K)-feldspar-rich hornfels. Endoskarn formed in the intrusive unit comprises replacement of mafic minerals, mainly hornblende, by pyroxene (diopside) and amphibole (actinolite), but the groundmass and porphyritic texture of the stock, dikes and sills are preserved.

The distribution of prograde metamorphic alteration can be defined by a radial zonation around the central intrusive. Proximal to the granodiorite stock, the endoskarn and potassic hornfels are overprinted by biotite-amphibole-pyroxene hornfels and garnet-pyroxene skarn and assemblages. These assemblages form a strongly metamorphosed band up to 800 ft wide laterally from the intrusive contact. Further outboard at a distance of 800 to 1,500 ft from the contact of the central intrusive body is a second zonation of biotite-rich hornfels and minor intervals of pyroxene-garnet skarn minerals associated with a small lobe of the intrusive unit.

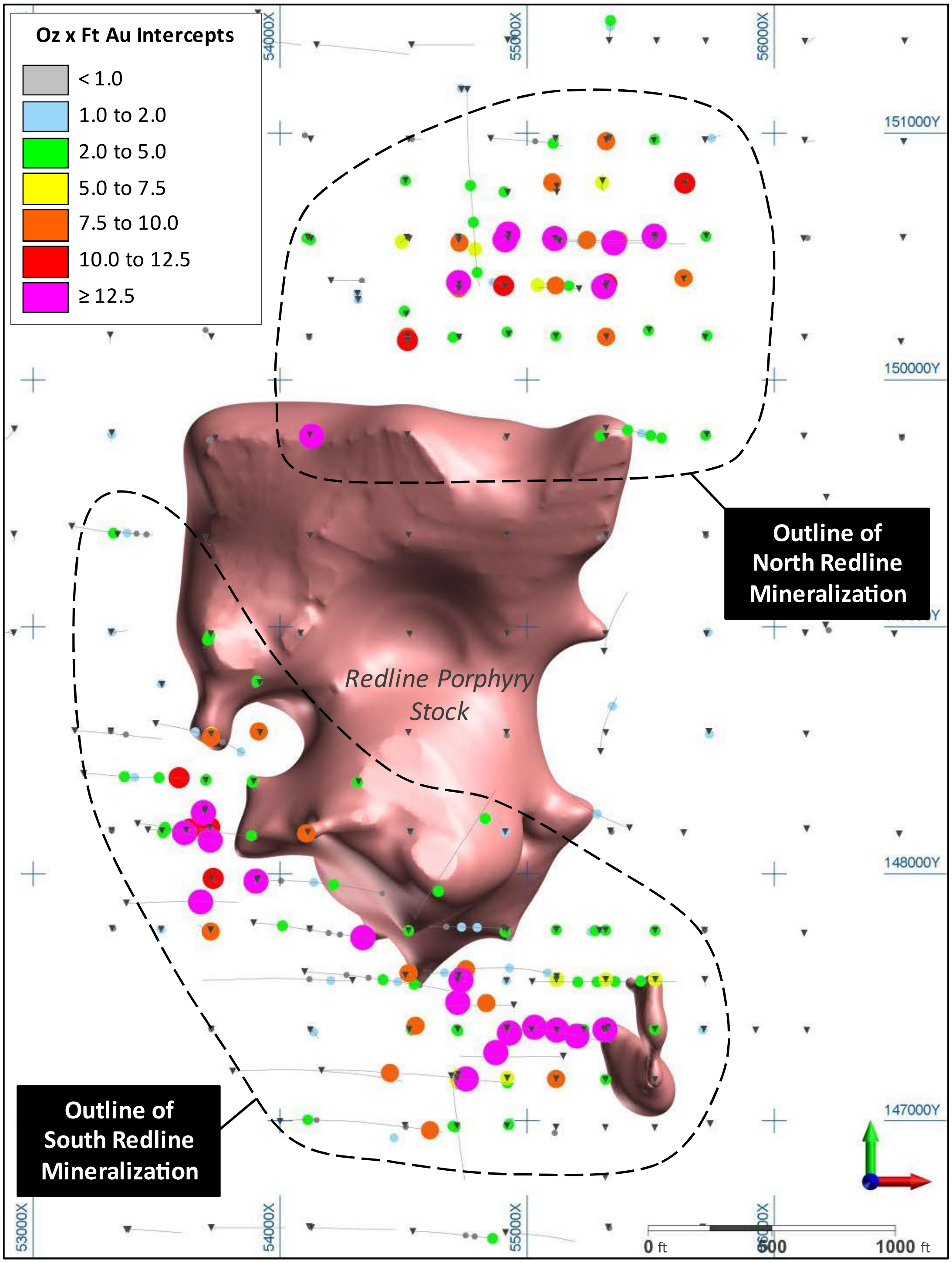
Less spatially defined are two retrograde assemblages that overprint the prograde assemblages. From oldest to youngest, these include epidote-chlorite-actinolite-quartz-calcite-veins followed by quartz-calcite veins. Replacement by this hydrous retrograde assemblage varies from minor rims to complete. Partial sericite replacement of the plagioclase phenocrysts is common. Overall, the retrograde assemblages are predominantly found in areas of complex structural activity and within dykes ands sills distal from the main intrusive body.

7.7 Mineralization

The Property hosts two gold-rich skarn deposits known as North Redline and South Redline (Figure 7.6). Gold mineralization is observed over an approximate 5,000 ft by 2,500 ft area and extends from a vertical depth 18 ft below surface to >2,000 ft.

Mineralization is spatially associated with all observed alteration and metamorphic assemblages indicating gold deposition occurred throughout the skarn development. Gold occurs as liberated grains. Silver and copper mineralization are also spatially associated with gold. The sulfide minerals including pyrrhotite, chalcopyrite, pyrite, sphalerite and molybdenite, which were precipitated during development of the prograde and retrograde alteration assemblages. Galena, arsenopyrite and bismuth-tellurium minerals are also present but in minor abundances.

Figure 7.6. Outline of North and South Redline deposits and significant Au intercepts from drillholes.



Source: CRL (2021)

Gold mineralization was intersected in alluvium material predominantly immediately above the contact with the bedrock and is assumed to be contained in clast-supported horizons.

7.8 Redox

Three redox zones are observed at the Converse Property and include an oxide, transition, and sulfide zone.

The oxide zone has a variable vertical depth profile ranging from 35 to >500 ft below the base of alluvium. Goethite is the dominant iron-oxide mineral (Muerhoff et al., 2002). Sulfide sulfur LECO values in the oxide zone average 0.05 wt % and the average gold solubility value using a 0.003 oz/ton Au cut-off is 0.83 (median is 0.85) based on cyanide to fire assay gold ratios.

The transition zone comprised of both iron oxide and sulfide minerals underlies the oxide zone. The vertical depth profile of this zone ranges from <5 ft to >1,400 ft) in thickness. Sulfide sulfur LECO values in the transition zone average 0.08 wt % and the average gold solubility value using a 0.003 oz/ton Au cut-off is 0.75 (median is 0.80) based on cyanide to fire assay gold ratios.

The sulfide zone is comprised of sulfide minerals and predominantly underlies the transition zone. In the North Redline deposit, localized areas of sulfide material are observed in the oxide zone. Sulfide sulfur LECO values in the sulfide zone average 0.18 wt % and the average gold solubility value using a 0.003 oz/ton Au cut-off is 0.72 (median is 0.75) based on cyanide to fire assay gold ratios.

8 Deposit Types

The Converse Property is being explored primarily for skarn mineralization and, specifically, gold-skarn mineralization, which includes the currently identified Redline deposits. The following is a brief summary of a skarn deposits, and precious metal skarns in particular, after Meinert (1993).

Skarns are a category of intrusion-related mineral deposits that occur world-wide and have been mined for a wide variety of commodities including Fe, W, Cu, Pb, Zn, Mo, Au, Ag, U, REE, F, B and Sn. Skarns can develop in shallow and deep crustal levels in a variety of geological settings. The common characteristic of skarn deposits is the predominance of calc-silicate mineralogy, which normally includes garnet and pyroxene. Skarn formation is a dynamic process affected by many variables, including temperature, pressure, and host-rock chemistry as well as the chemistry of the intrusion(s) and the mineralizing fluids they generate. Large skarn systems are typically characterized by several phases of 'skarn' development from early, normally isochemical, hornfels phase; followed by structurally- and/or stratigraphically-controlled reaction skarn development; then a main phase of proximal, metasomatic, coarse-grained skarn development at peak temperatures; followed by retrograde skarn development as temperatures cool.

Precious metal skarns are often related to ilmenite-bearing granodioritic plutons or intrusive complexes. The skarn mineralogy is generally dominated by iron-rich mineralogy including hedenbergitic pyroxene and intermediate (grossular to andraditic) garnets. Other common minerals include potassium feldspar, scapolite, vesuvianite, apatite and high-chlorine aluminous amphibole. Distal, or early-stage alteration, can often include significant potassic (k-feldspar ± biotite) hornfels development. Arsenopyrite and pyrrhotite are the most common sulfide minerals associated with precious metal mineralization.

The Redline deposits at Converse are interpreted as gold-rich skarn deposits. Gold mineralization is associated with the precipitation of sulfides (pyrite-pyrrhotite-chalcopyrite-sphalerite-molybdenite) during one prograde (garnet-pyroxene-K-feldspar) and two retrograde (chlorite-epidote-actinolite-quartz-calcite and quartz-calcite) assemblage events. The skarn assemblages developed subsequent to the emplacement of a dioritic intrusive stock. Alteration minerals occur mainly as replacements of carbonate minerals in the matrix of calcareous sandstones and also as cross-cutting veinlets. As observed in drill core, much of the prograde skarn replaces bedding planes (dipping shallowly to the west).

9 Exploration

In 2017, CRL completed seven core drillholes on the Property totalling 5,944 ft. The details of the CRL drill program are discussed in Section 10. In 2018, CRL completed metallurgical test work, the details of the CRL metallurgical work are discussed in Section 13.

10 Drilling

The drillhole database as of December 31, 2020 contained 326 drillholes totalling 254,833.6 ft. All drilling within the database is presented in Table 10.1 and includes drillholes outside the Converse resource area, as illustrated in Figure 10.1. The number of holes and footage differs from Section 6 reports of drilling due to subsequent changes in the mineral claim boundaries.

Holes drilled within the resource area and used for the 2020 MRE consist of 215,123 ft drilled in 249 holes (Table 10.2) that have provided 31,908 gold assays from intervals totalling 172,325 ft of core or RC chips. Core drilling represents approximately 33% of the total footage and 23% of the total holes for the Project.

Table 10.1. Drilling statistics for drillholes in the Converse database.

Year	Company	Core		MR-/RC-Core*		RC		Rotary		Total	
		No.	Ft.	No.	Ft.	No.	Ft.	No	Ft.	No.	Ft.
1989	Kennecott Minerals Co.					2	585.0			2	585.0
1989	Chevron Resources					8	3,695.0			8	3,695.0
1991	Chevron Resources					3	1,115.0			3	1,115.0
1992	Cyprus Mines Corp.					15	4,070.0			15	4,070.0
1994	Independence Mining Co.							10	4,950.0	10	4,950.0
1995	Uranerz U.S.A. Inc./Romarco							10	4,760.0	10	4,760.0
1996	Uranerz U.S.A. Inc./Romarco			6	6,440.0	16	10,645.0			22	17,085.0
1997	Uranerz U.S.A. Inc./Romarco			3	3,611.0	30	21,148.0			33	24,759.0
1998	Uranerz U.S.A. Inc./Romarco					52	42,012.0			52	42,012.0
1999	Uranerz U.S.A. Inc./Romarco					15	9,335.0			15	9,335.0
2003	Metallic Ventures Group			8	5,307.2	18	14,988.0			26	20,295.2
2004	Metallic Ventures Group					28	24,622.5			28	24,622.5
2007	Metallic Ventures Group	8	7,332.2			53	37,480.0			61	44,812.2
2011	International Minerals Corp.	8	13,945.5	6	7,700.4					14	21,645.9
2012	International Minerals Corp.	4	5,028.6	10	16,064.2	6	4,055.0			20	25,147.8
2017	Converse Resources LLC	7	5,944.0							7	5944.0

Year	Company	Core		MR-/RC-Core*		RC		Rotary		Total	
		No.	Ft.	No.	Ft.	No.	Ft.	No	Ft.	No.	Ft.
Total		27	32,250.3	33	39,122.8	246	173,750.5	20	9,710.0	326	254,833.6

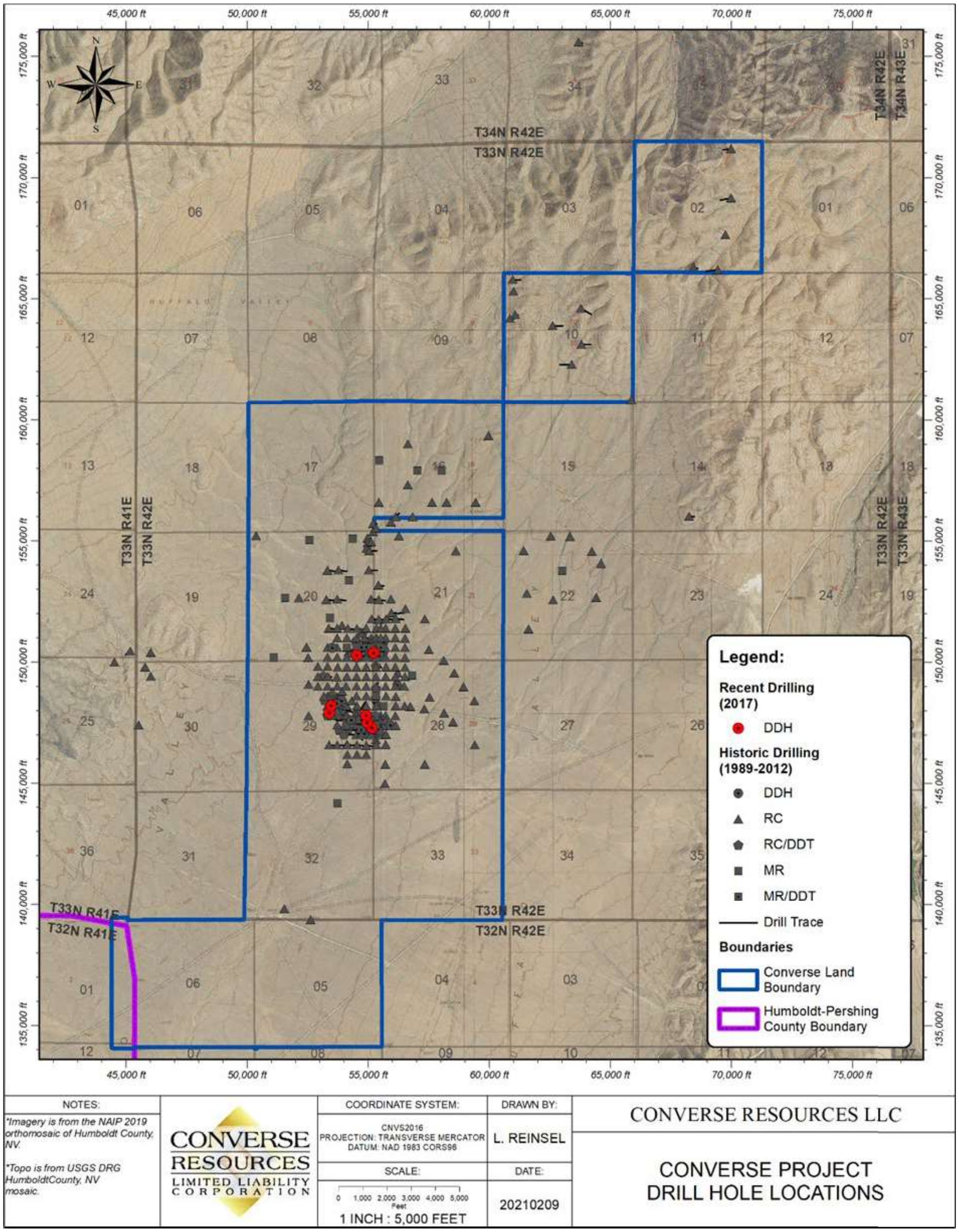
* A number of RC holes were initiated with RC and then completed with a diamond core tail.

Table 10.2. Converse resource area drillhole summary.

Year	Company	Core		MR-/RC-Core		RC		Rotary		Total	
		No.	Ft.	No.	Ft.	No.	Ft.	No.	Ft.	No.	Ft.
1989	Kennecott Minerals Co.					2	585.0			2	585.0
1989-1991	Chevron Resources					11	4,810.0			11	4,810.0
1992	Cyprus Mines Corp.					15	4,070.0			15	4,070.0
1994	Independence Mining Co.							10	4,950.0	10	4,950.0
1995-1999	Uranez U.S.A. Inc./Romarco			9	10,051	113	83,140.0	10	4,760.0	132	97,651.0
2003-2007	Metallic Ventures Group	8	7,332.2	8	5,307.2	99	77,090.5			115	89,729.9
2011-2012	International Minerals Corp.	12	18,974.1	16	23,764.6	6	4,055.0			34	46,793.7
2017	Converse Resources LLC	7	5,944.0							7	5944.0
Total		27	32,250.3	33	39,122.8	246	173,750.5	20	9,710.0	326	254,833.6

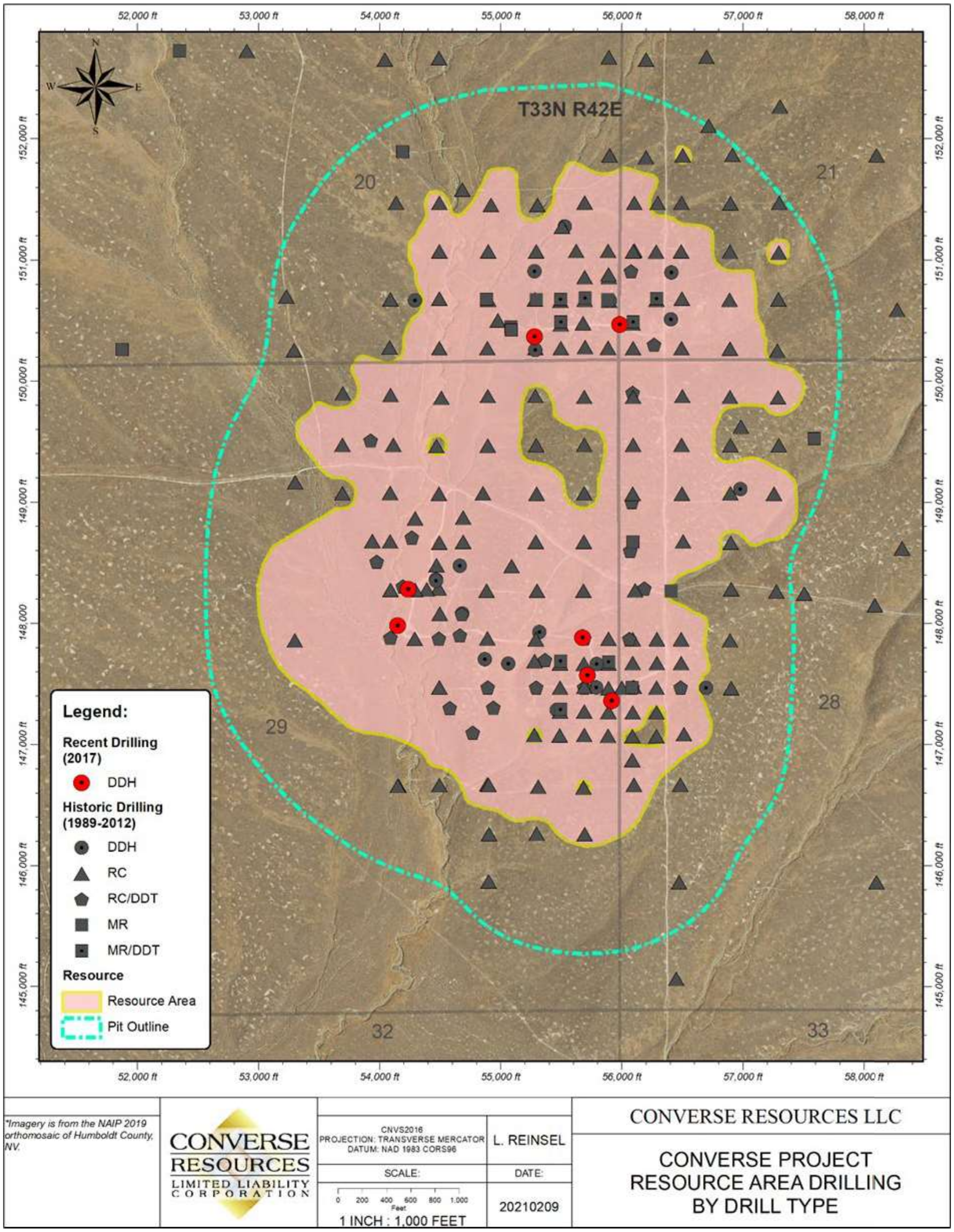
The spatial distribution of the drillhole collars used for the MRE are shown in Figure 10.2. Holes drilled for resource estimation were drilled covering an area of approximately 6,000 ft long by 3,900 ft wide. A nominal drillhole spacing across the deposit is approximately 400 ft and reduced to 100 or 200 ft spacing where infill drilling was completed.

Figure 10.1. Drillhole location map by type, Converse Property.



Source: CRL (2021)

Figure 10.2. Drillholes within the Project area with outline of gold mineralization.



Source: CRL (2021)

10.1 Drill Methods

The drilling contractor used by Independence in 1994 was B & B Drilling, of Grand Junction, Colorado. Kennecott's, Chevron's and Cyprus' drilling contractor(s) could not be determined.

10.1.1 RC Drilling

Between 1995 and 1999, UUI/Romarco RC drilling was performed by Eklund Drilling Company of Reno, NV. RC hole diameters were between 6 and 5¼ in, the alluvial portion of the holes were drilled using rotary tricone bits. A booster and auxiliary compressors were used in the deeper portions of most holes, typically below ~900 ft. RC drilling was completed using TH-100A and Explorer 1500 rigs and was completed wet using a cyclone and rotary wet splitter. Mud rotary drilling was completed by B & B Drilling, of Grand Junction, Colorado, with hole diameters of 5½ to 5¼ in.

Between 2003 and 2007, MVG's RC drilling was performed by Eklund Drilling Company and Rimrock Drilling of Reno, NV, generally using MPD 1,500 rubber-tire Explorer and truck-mounted deep-hole IR-75C rigs. RC drilling was carried out using a 5¾ in diameter hammer bit.

In 2011 and 2012, IMC's RC drilling was performed by Rimrock Drilling generally using a TH75E rig. RC drilling was carried out using a 5¾ in diameter hammer bit. The hole conditions, any drilling problems and the water depth and flow were reported by the driller on the driller's log.

Srivastava (2012) reported that historical and recent RC drilling procedures were similar. Generally, the first 20 m of each drillhole was drilled dry. Most of the mineralized intervals were drilled wet and split by a rotary wet splitter. A sample technician was assigned to each rig to ensure that the sample collection did not overflow the collection bucket. A representative portion of the rock chip sample was collected from the reject material for each sample and placed in a covered plastic tray for later logging. A geologist logged each interval and the driller reported drillhole conditions, drilling conditions, water depth and water quantity.

10.1.2 Core Drilling

Between 1996 and 1997, UUI/Romarco's core drilling was performed by Connors Drilling using a Longyear 44 core rig. Core sizes were HQ (2.5 in) and reduced to NQ (1.87 in) where necessary. Most of the core holes were pre-collared using an RC rig.

Between 2003 and 2007, MVG's core drilling was performed by Boart Longyear Drilling Company (Boart Longyear) using an LS 244 truck-mounted core rig. Core sizes were NQ and HQ Most of the core holes were pre-collared using a RC rig.

In 2011 and 2012, IMC's core drilling was performed by American Drilling and Boart Longyear. The type of rigs used is unknown. Core sizes were HQ. Most of the core holes were pre-collared using an RC rig.

In 2017, CRL's core drilling was performed by Major Drilling of Salt Lake City, Utah, using a truck mounted LF-230 rig. The purpose of the 2017 core drilling program was to collect samples for metallurgical studies. A total of seven PQ size (3.35 in) core holes were drilled for a total of 5,944.0 ft.

Srivastava (2012) reported that core handling procedures were similar across historical and recent campaigns. Whole core was first washed and photographed and then placed on benches for rock quality determination (RQD), core recovery measurements and geologic review. The photographs of the core were initially taken using a single lens reflex camera and conventional colour film. Recent campaigns utilized digital photography stored on a server and external hard drives for back up. After the core has been split for sampling with a diamond saw core was usually photographed again. Geological and engineering log data are hand-written on pro-forma log sheets and a technician enters them into the drillhole database. More recent campaigns had the data entered directly into excel and then into the drillhole database.

10.2 Geologial Logging

The historical RC logging campaigns collected a variety of information that predominantly included mineralogy, lithological unit, color, alteration, metamorphic assemblages, quartz vein intensity, and oxide state with intensity of iron oxides and sulfides.

The historical core logging campaigns collected a variety of information that predominantly included recovered core length, mineralogy, lithological unit, color, alteration, metamorphic assemblages, oxide state with intensity of iron oxides and sulfides, and vein type and abundance.

The 2017 CRL core was transported to a secure logging facility in Lovelock, NV, where the CRL geologists and technicians completed the following:

- Core boxes were arranged sequentially on the logging tables and drill mud was washed from the core;
- Geotechnical measurements, including recovery, rock quality designation (RQD) and rock mass rating (RMR) were captured;
- Geological data, including mineralogy, lithological unit and texture, color, structural type and style, redox, alteration type and intensity, mineralization type and percentage were captured by CRL geologists directly into a Microsoft Excel logging template;
- Sample boundaries were marked with wax pen, and sample tags were stapled to the inside of the core box at the beginning of the interval. Digital core photographs were taken of wet core with the sample tags visible and the box number and footage (from–to) labeled;
- Following logging, sample markup and photography, the core boxes were placed on pallets, wrapped in plastic, and stored within the secured laydown yard at the Lovelock facility. An independent transportation company transported pallets of core with a signed inventory list to the ALS Global facility for sample cutting, bagging, and analysis, as described in Section 11.

10.3 Recovery

The core recoveries for the companies not specifically stated in this section are unknown as no information was available. MVG core drilling recoveries averaged approximately 90%. Approximately 92% of the core was from the bedrock where the overall recovery was 94%. IMC core recovery was 94% in bedrock and 70% in alluvium. The combined average recovery was 92% for the entire drill program.

The 2017 CRL core recovery values averaged 98% in the bedrock and 82% in the overlying alluvium units. The combined average recovery was 95% for the entire drill program.

10.4 Collar Surveys

MVG drill collars were surveyed by a registered contract land surveyor; however, it is unknown who completed the work and with what instrument. IMC drill collars were surveyed but it is also unknown by whom and with what instrument. The 2017 CRL drill collars were surveyed by Daniel Park of Elko Mining Group using a high accuracy real time kinematic (RTK) global positioning system (GPS) equipment with centimeter accuracy.

10.5 Downhole Surveys

Downhole surveys for the Romarco/UUI drilling were completed by Silver State Surveys Inc. of Tucson, Arizona (AZ), and by Wellbore Navigation Inc. (Wellbore) of Elko, NV. Both companies used gyroscopic instruments with measurements recorded on 50 ft intervals. Downhole surveys for the MVG drillholes were collected using a gyroscopic instrument operated by Wellbore. Measurements were recorded on 50 ft intervals. Downhole surveys for the IMC drillholes were collected using a gyroscopic instrument operated by International Directional Services, with measurements recorded on 50 ft intervals.

The 2017 CRL drillholes were downhole surveyed using a Reflex EZ-Trac gyroscopic tool operated by Major, with measurements recorded on 50 ft intervals. Downhole survey deviations were recorded on a tablet and sent via electronic mail to GRL personnel. The REFLEX GYRO is not affected by magnetic interference and can be used within steel drill rods. Surveys were checked for erroneous records such as large deviations between readings. Questionable survey data was flagged in the database and excluded from the Company's database exports.

10.6 Metallurgical Drilling

In 2017, CRL collected drill core from seven PQ-size core holes for metallurgical test work. Drillholes names, coordinates, depths and hole diameters are summarized in Table 10.3.

Table 10.3. Metallurgical drillholes sampled by CRL.

Hole ID	East (ft)	North (ft)	Elevation (ft)	Depth (ft)	Hole Diameter
CNR-MET17-001	55,146.1	147,262.5	5,018.7	1,076.5	PQ
CNR-MET17-002	54,945.6	147,469.4	5,016.4	1,070.0	PQ
CNR-MET17-004	53,377.7	147,881.9	4,989.9	813.5	PQ
CNR-MET17-005	53,464.9	148,180.0	4,991.8	955.0	PQ
CNR-MET17-006	55,210.1	150,368.2	5,031.9	647.0	PQ
CNR-MET17-007	54,509.7	150,266.0	5,019.5	582.0	PQ

10.7 Sample Length/True Thickness

Calculation of the true thickness of each core or RC interval is dependent on the orientation and dip of the drillhole and the mineralized zone. Mineralized intercepts of core holes drilled in 2017 have a calculated true thickness of between approximately 70% and 100% of the drilled thickness.

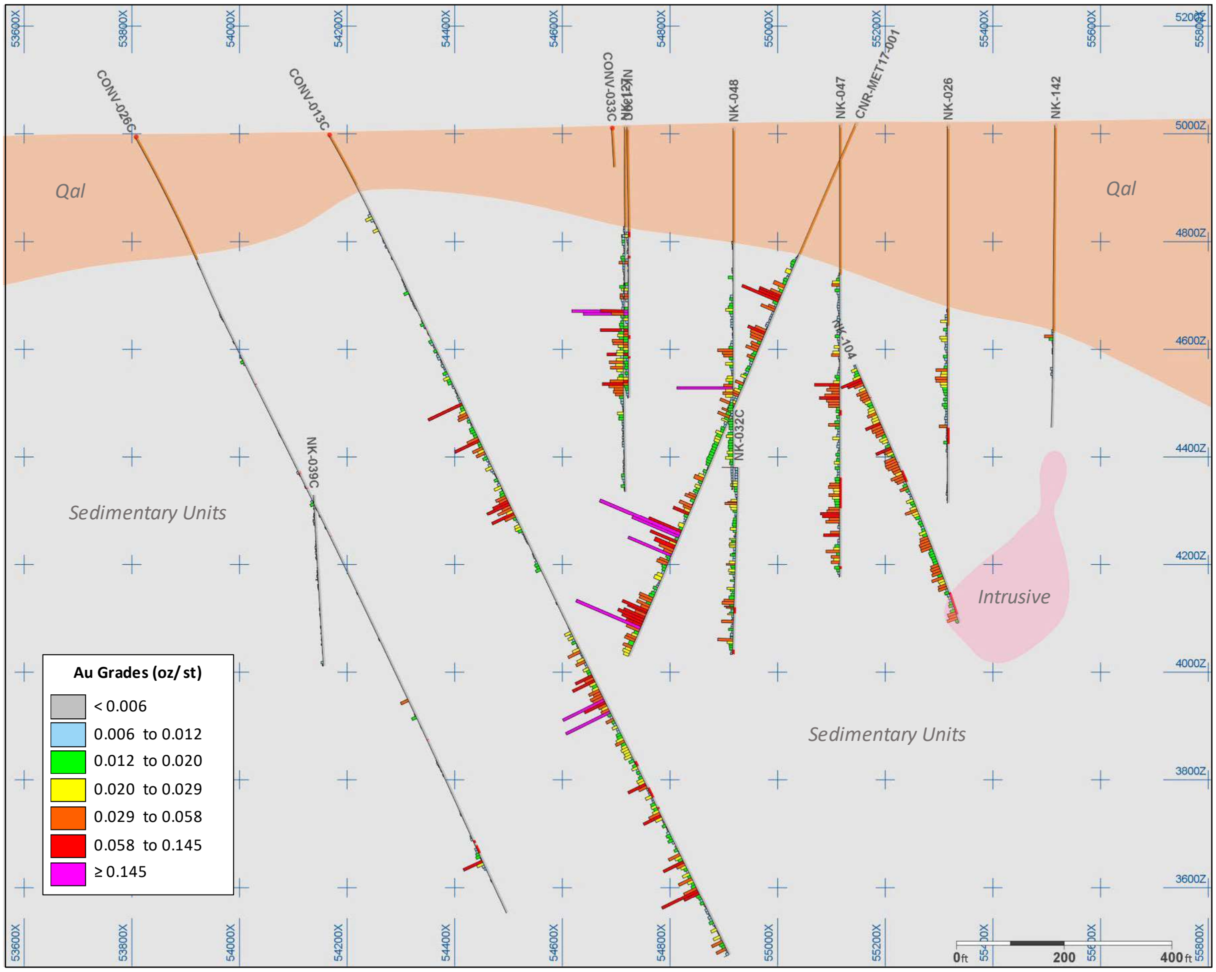
10.8 Summary of Drill Intercepts

Table 10.4 list the significant intercepts for the CRL 2017 drill program. Figure 10.3 and Figure 10.4 show downhole assays from the CNR-MET17-001 and CNR-MET17-006.

Table 10.4. 2017 program significant intercepts (≥0.006 oz/ton Au).

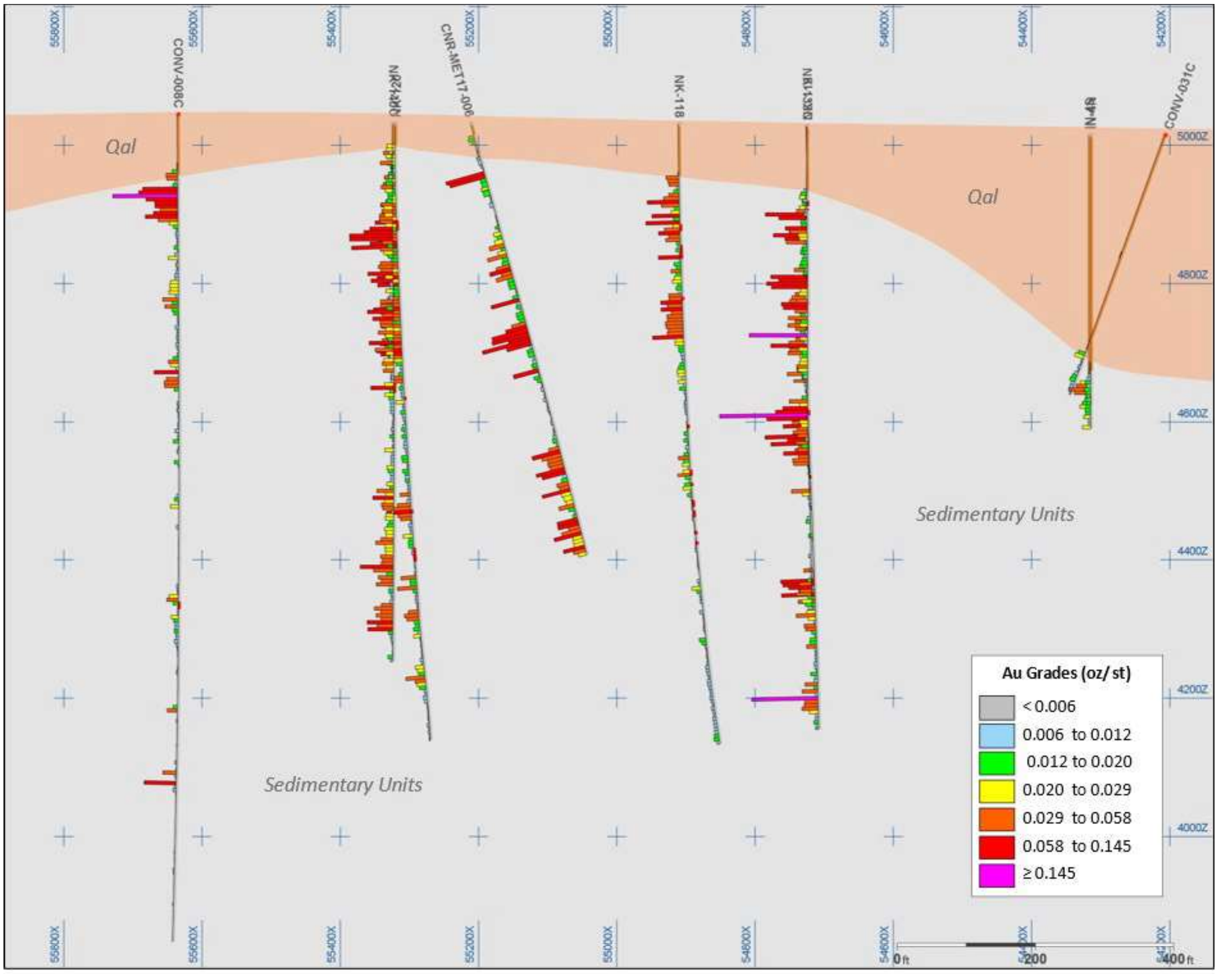
Hole ID	From (ft)	To (ft)	Interval (ft)	Au (oz/ton)	Ag (oz/ton)
CNR-MET17-001	272.0	1,076.5	804.5	0.033	0.028
CNR-MET17-002	211.5	561.0	349.5	0.025	0.065
CNR-MET17-002	586.5	1,057.0	470.5	0.028	0.077
CNR-MET17-003	252.0	346.5	94.5	0.018	0.538
CNR-MET17-003	392.0	547.0	155.0	0.010	0.101
CNR-MET17-003	612.0	666.5	54.5	0.011	0.048
CNR-MET17-004	355.0	801.0	446.0	0.028	0.098
CNR-MET17-005	266.0	474.0	208.0	0.021	0.094
CNR-MET17-005	498.5	955.0	456.5	0.023	0.115
CNR-MET17-006	17.5	125.5	108.0	0.019	0.096
CNR-MET17-006	153.5	387.5	234.0	0.032	0.093
CNR-MET17-006	462.0	647.0	185.0	0.035	0.176
CNR-MET17-007	202.0	317.0	115.0	0.022	0.068
CNR-MET17-007	362.0	582.0	220.0	0.019	0.095

Figure 10.3. Drillhole CNR-MET17-001 with view to the north. Window slice of 200 ft.



Source: CRL (2020)

Figure 10.4. Drillhole CNR-MET17-006 with view to the north. Window slice of 200 ft.



Source: CRL (2020)

11 Sample Preparation, Analyses and Security

11.1 Sampling Methods

Drilling by Kennecott, Chevron, Cyprus and Independence Mining Co. was completed prior to 1995. There are no available supporting documents for these programs which represent <6% of the total footage at the Converse Property.

11.1.1 RC Sampling

11.1.1.1 UUI/Romarco RC Sampling

The historical RC drilling was completed wet using a cyclone and rotary wet splitter. One split per 5 ft interval was collected in alluvium, and two splits per 5 ft interval (A and B) were collected in bedrock. The A split was used for assaying. The B split was saved on the ground for potential bottle roll tests or backup assays. Sample size for A splits generally varied from 8 to 10 lb in 1996 and from 1997 onwards sample sizes were increased to between 10 to 15 lb. B splits were generally in the 8 to 10 lb range. All bedrock and generally the lower 20 to 25 ft of alluvium were assayed.

The mud rotary holes drilled by UUI in 1995 were sampled by shoveling the drill cuttings, which were allowed to settle in a mud trough, into sample bags. All historical mud rotary sample intervals were 5 ft in length.

11.1.1.2 MVG RC Sampling

MVG RC samples were collected by a sampling technician provided by the drilling contractor at 5 ft intervals after passing through a cyclone and rotating wet splitter attached to the drill rig. The wet splitter was set to acquire the desired sample volume (usually 11 to 22 lbs). The samples were all placed in pre-numbered bags. Excess water was allowed to filter out of the sample bag on site prior to shipment to the assay laboratory. A small representative portion of the cuttings was collected from the wet splitter for each 5 ft drilling interval and placed in a covered plastic chip tray and taken to Sparks, NV for geological logging.

11.1.2 Core Sampling

11.1.2.1 UUI/Romarco Core Sampling

The core was transported by UUI/Cameco personnel to their warehouse in Battle Mountain, NV for logging and sampling. After logging was completed, the geologist would determine sample intervals based on geological, mineralogical, or structural features. Core sample intervals ranged from 0.5 to 7.5 ft in length, with an average of 3.9 ft. The core was split longitudinally into two halves using a hydraulic core splitter, with one half submitted for assay and the other half archived at the Battle Mountain facility. Core samples were picked up at the warehouse by assay laboratory personnel; sample rejects and assay pulps were returned after analyses were completed.

11.1.2.2 MVG Core Sampling

MVG drill core was transported to the company's warehouse facility in Sparks, NV, for processing. After washing and photographing with a digital camera, the drill core was sampled in 5 ft intervals unless significant zones were encountered. Mineralized zones were sampled utilizing intervals adjusted appropriately based upon geology and mineralization. Core sampling was carried out depending on the nature of the recovered material. Competent core was sampled using a diamond saw and collecting a ½ split (HQ or smaller sizes) or a ¼ split (PQ size) of the original whole core to be sent for assay. The remaining core was returned to the core box. Broken core was sampled by hand selecting a representative half portion of the larger pieces and then combining this with half of the finer material obtained using a modified drywaller's corner trowel.

11.1.2.3 CRL Core Sampling

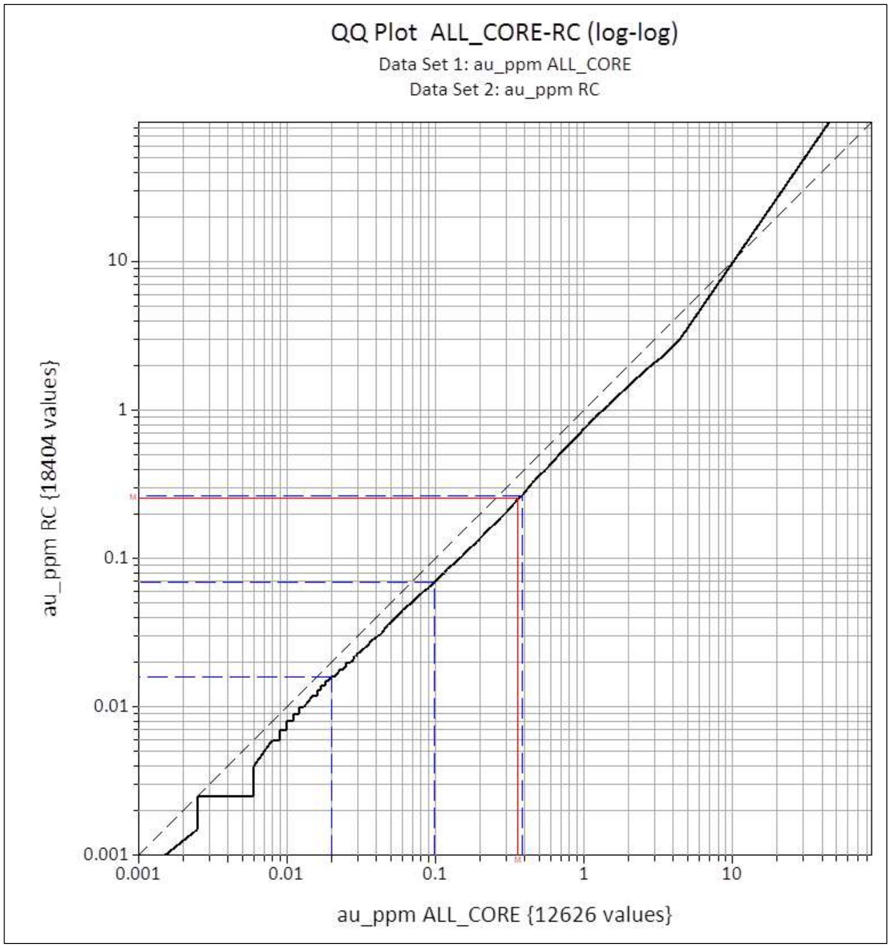
During the 2017 core drill program, the drilling company transported core from the drill rig to a nearby secure logging facility (Lovelock, NV), at the end of every shift. Upon completion of geotechnical and geological data collection, sample intervals were delineated and cut lines marked. Samples were marked at approximately 5 ft intervals and were adjusted based on geological boundaries. Sample boundaries were marked with a wax pen and sample tags were stapled to the inside of the core box at the beginning of the interval. Following core photography, the core boxes were placed on pallets, wrapped in plastic, and stored within the secured laydown yard at the Lovelock facility. An independent transportation company transported pallets of core with a signed inventory list to the ALS Global facility.

ALS Global personnel completed sample cutting, bagging, and analyses of all core material. Bulk density samples were collected prior to core cutting and set aside for later density determination.

11.1.3 RC vs Core Sampling Analysis

Gold fire assay ("FA") data were analyzed for RC vs drill core sample collection bias by CRL with statistical and visual comparisons completed. Quantile-quantile plots indicate a minor systematic high bias in the core samples compared to the RC samples (Figure 11.1). This bias is related to a greater number of RC samples along the periphery of the deposit, which includes lower-grade samples, than the core data set. Visual comparison of neighboring RC and core holes show similar grade ranges and lengths of mineralized intersection. Histograms and cumulative distribution plots suggest some local variance between RC and core grade distributions but with no significant effect on global grade trends within the mineralized zones.

Figure 11.1. RC vs core sample assay comparison.



Source: CRL (2020)

11.2 Metallurgical Sampling

Refer to Section 13.

11.3 Density Determinations

UUI/ Romarco bulk density determinations (6 samples) were performed by KCA of Reno, NV using the wax method.

MVG bulk density determinations on bedrock samples were performed by KCA (35 samples) using the wax method, as well as McClelland Laboratories (109 samples) using the acrylic coating density analysis method.

Determinations on rock and alluvium samples were performed at ALS Global using the water immersion method (ALS code OA-GRA08a) for 74 core samples.

CRL bulk density determinations were performed by ALS Global and calculated using the paraffin wax-coated and water immersion methods (ALS code OA-GRA09a) for 78 whole core samples collected approximately every 30 ft.

11.4 Analysis and Test Laboratories

Numerous independent laboratories were contracted for analytical test work over the different years, as listed in Table 11.1. These laboratories were all independent of the company conducting the Converse exploration at that time and were and are independent of the authors and QPs of this Technical Report.

Table 11.1. Laboratories used for analytical test work.

Name and Location	Accreditation	Year	Test Work Performed
Acme Laboratories (Acme)	Unknown	1995-1997	Multi-element geochemistry
Activation Laboratories (Actlabs)	Unknown	1994-1999	Multi-element geochemistry
ALS Global (ALS; previously ALS Chemex), Reno, NV	ISO 9001:2000 ISO 17025:2000	1991-1992, 2007, 2011-2012, 2017	Au assays, multi-element geochemistry, and density determinations
American Assay Laboratories (AAL), Sparks, NV	ISO 17025	2003-2004	Assays and density determinations
Bondar Clegg, Sparks (acquired by ALS Chemex, 2001)	Unknown	1988/1989	Assays
Cone Geochemical (Cone)	Unknown	1989, 1994-1999	Assays, multi-element geochemistry

11.5 Sample Preparation and Analysis

Sample preparation and analysis for Kennecott was completed at Bondar Clegg but methods are unknown.

11.5.1 Chevron

Drilling samples from the Chevron campaigns were submitted to Cone for sample preparation and analyses. Sample preparation methods are unknown. The analytical method utilized a one assay-ton aliquot (1AT; ~29.2 g) for FA digest and a finish with atomic absorption spectroscopy (AAS).

11.5.2 Cyprus

Samples from the Cyprus drilling campaign were submitted to ALS Global (ALS Chemex at the time). Sample preparation methods are unknown. The analytical method was FA with AAS finish. Aliquot size for the analytical method is unknown.

11.5.3 Independence and UUI/Romarco

Samples of drillhole cuttings and core in 1994 to early 1997 were assayed using a conventional sample preparation and FA procedure by Cone. Sample preparation included drying and crushing the entire sample to >50% minus 10 mesh. A 300 g split was pulverized to >90% minus 200 mesh. In 1994 to 1996, i.e., holes IN-1 to IN-9 and NKM-10 to NK-41, a 20 g aliquot was used for final gold assay by FA digest and AAS finish.

From early 1997 to 1999, UUI/Romarco sample preparation consisted of drying and crushing the entire sample to >50% minus 10 mesh. A 2 kg split was collected and pulverized to 70% minus 100 mesh using a Bico plate pulverizer. A 300 g split was collected from the 2 kg pulp and reduced to >90% minus 200 mesh in a ring-and-puck pulverizer. Analytical method for holes NK-042C to NK-109 included a 30 g aliquot used for FA digest and AAS gold finish. In 1997 and 1998, select intervals were submitted to Bondar-Clegg for gold check assays using a 30 g aliquot with FA digest and AAS finish.

All UUI/Romarco mineralized intervals in drillholes were subsequently analyzed by a hot cyanide shake assay method. Hot cyanide assays were performed on the same pulp as the original fire assay analysis. The cyanide assay is performed on a one assay-ton (1AT) aliquot using 60 mL of solution and analyzed by AAS finish. No cyanide assays were performed in 1998.

UUI multi-element analyses were performed on 20 ft composites of cuttings and core that were prepared by Cone. Multi-element analyses were normally performed by Actlabs for a 48-element package with an iron-oxide titration and neutron activation (INAA) finish used to determine 35 elements and an inductively-coupled plasma (ICP) mass spectrometry (MS) (ICP-MS) finish used to determine a further 19 elements. Six elements are in both packages, i.e. Ag, Ca, Mo, Ni, Sr and Zn. Cone also completed Cu, Mo, Pb, Zn and Ag analyses on select 5 ft intervals using a 4-acid digest with AAS finish. In 1995, 1996, and early 1997, Actlabs ICP analyses for holes NKM-10 to NK-41, NKC-42 to 387 ft (RC portion), NKC-43 to 317 ft (RC portion), and NK-44 were subcontracted to Acme Laboratories (Acme) who used a four-acid digestion followed by ICP-MS finish. During most of 1997, the ICP-MS analyses for holes NKC-43 and NK-45 to NK-66 were completed by Actlabs, also using a four-acid digestion.

LECO analyses for carbon and sulfur were performed on composites from seventeen 1996 and 1997 holes by Newmont's Lone Tree mine laboratory. Details on the analytical method are unknown.

11.5.4 MVG

MVG drill samples from 2003 and 2004 were analysed by AAL with preparation as follows: sample was weighed, dried and crushed to <70% passing 10 mesh (2 mm). A 300 g split was pulverized +80% passing 150 mesh (or ~100 μm). A 30 g charge was split for FA digest and atomic absorption (AA) finish.

MVG drill samples from 2007 were prepared and analysed by ALS. The samples were weighed, and dried and crushed to <70% passing 2 mm. A 250 g split was pulverized +85% passing 75 µm. A 30 g charge was split for FA digest and AA finish.

Selective samples from 2003, 2004 and 2007 were also submitted to ALS' Vancouver facility for a 33-element package with a four-acid digest and ICP atomic emission spectroscopy (ICP-AES) finish.

MVG routinely inserted standards after every 10th sample interval as part of its quality assurance and quality control (QA/QC) program. ALS also routinely introduced blanks and standards into the sample stream as part of its own internal QA/QC program.

11.5.5 IMC

IMC sample preparation and analysis procedures remained unchanged from those used by MVG in 2007 (refer to Section 11.5.4), with the exception of multi-element analysis that was changed to a 51-element package with an aqua regia digest and ICP-MS finish.

11.5.6 CRL

For 2017 PQ core sampling, a fillet representing approximately ¼ of the core was cut parallel to the long axis and along the side of the core using a manual-feed electric core saw and placed into sample bags. The ALS 2017 core sample preparation and analyses flow chart is shown in Figure 11.2. The unsampled core, approximately ¾, was retained for metallurgical test work at KCA in Reno. This core was picked up from the ALS Global facility by KCA personnel.

11.6 Quality Assurance and Quality Control (QA/QC)

11.6.1 Historical

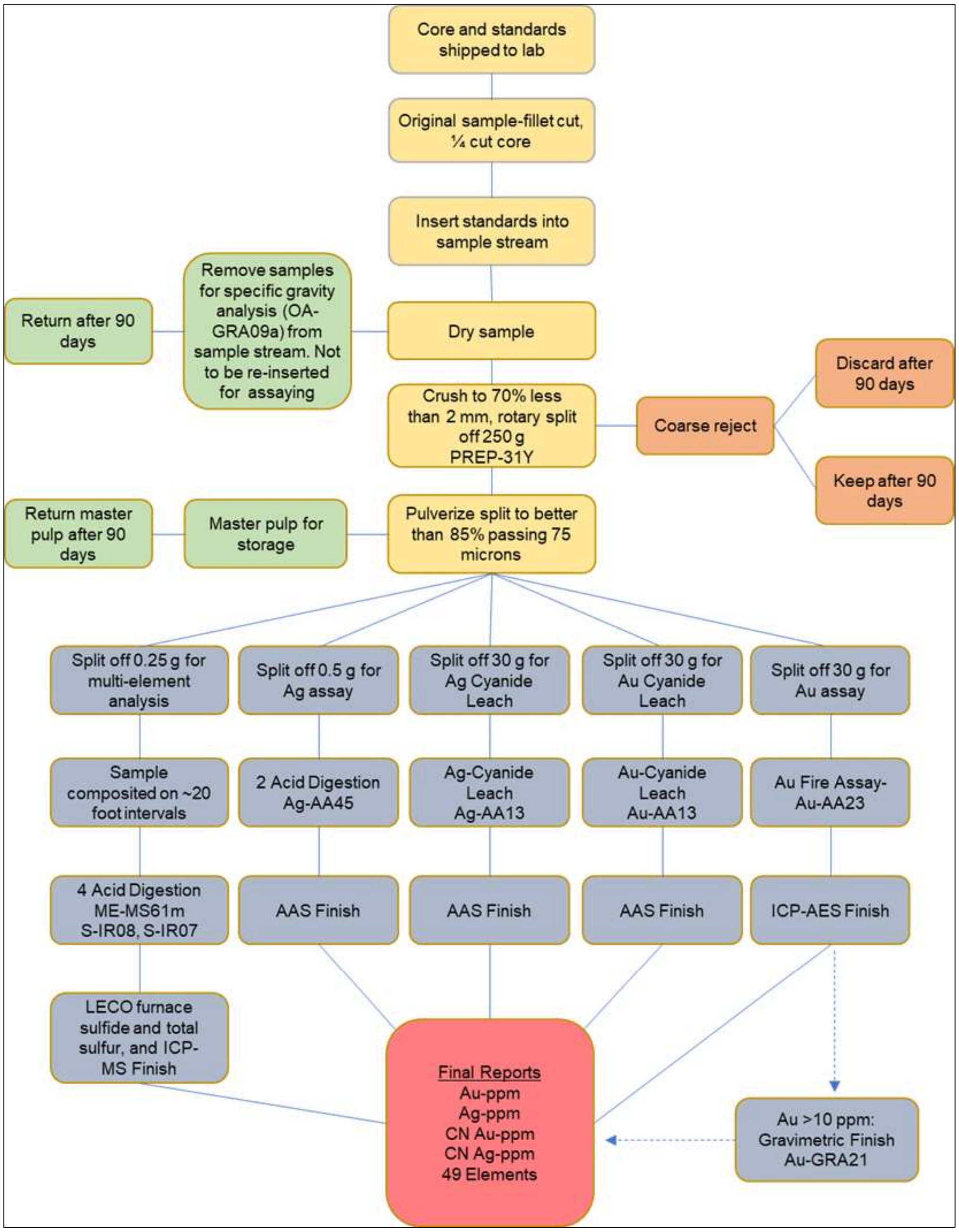
The Nike JV recognized discrepancies between original and check assay data sets. In particular, individual assay reproducibility was considered problematic. The matter was studied, and it was concluded that an intermediate pulverizing step was required. Following crushing of the entire sample to 50% passing 10 mesh, a 2 kg split was pulverized to 70% passing 200 mesh. A 300 g sample was then further reduced to 90% passing 200 mesh at which point a 30 g sample was fire assayed with an AAS finish.

Substantial differences were still present between assays derived from the same original whole sample; however, the average difference between the mean grades of all samples was 1% and correlations between datasets were at or above 0.95.

Muerhoff et al. (2002) concluded there was no systematic bias in the check assay data, and that the variances were related to the in-situ heterogeneity of the deposit and its effect on sampling and sample preparation. Muerhoff et al. (2002) were satisfied that the data were of sufficient quality to support resource estimation.

The APEX QPs concur with the results of this work and view this variance as reasonable for hydrothermal gold systems.

Figure 11.2. CRL sample flow chart for 2017 program.



Source: CRL (2017)

11.6.2 MVG

Between 2003 and 2004 MVG did not routinely insert standards and blanks into the sample stream. Coarse reject material was collected for check assays at ALS (Reno) or Bondar Clegg (Sparks). AAL inserted standard reference materials (SRMs) and blanks as part of the laboratory's internal QA/QC program.

For MVG's 2007 program, blank and SRMs were inserted into the sample stream for submittals to ALS. No field duplicates were included in the sample stream. MVG submitted 621 QA/QC samples (~9%; see Table 11.2) to ALS, together with 6,458 original RC and core samples.

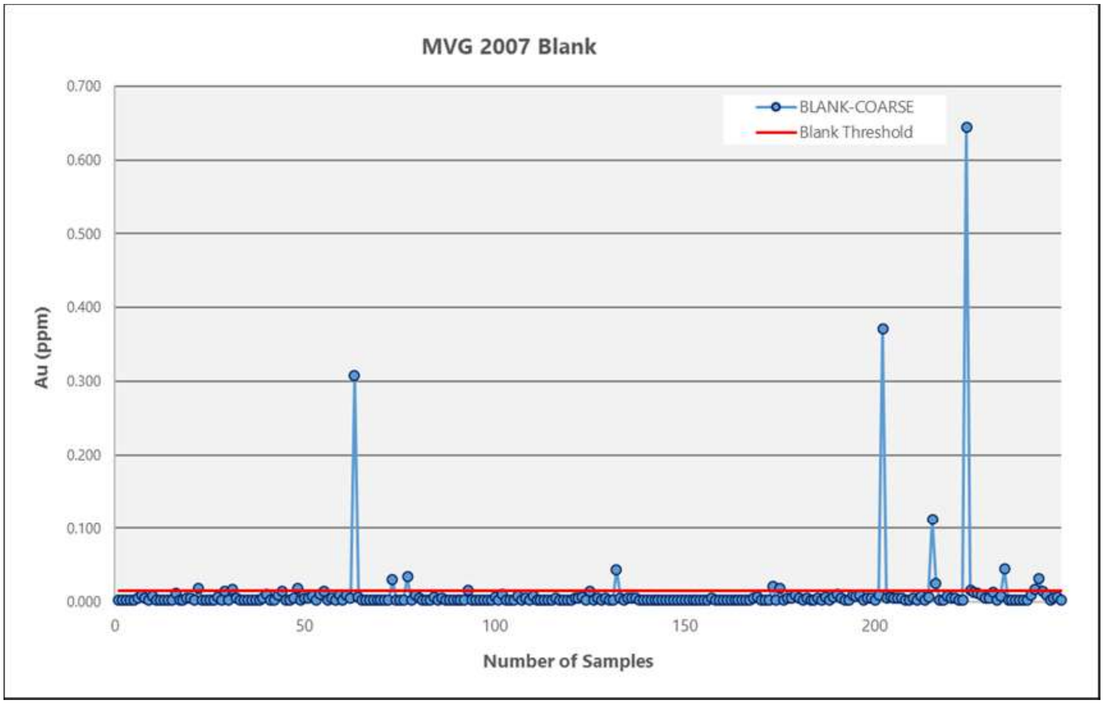
Table 11.2. Summary of QA/QC samples inserted into sample stream by MVG.

No. of Blanks	No. of CRMs	No. of Field Duplicates
248	373	0

11.6.2.1 MVG Blanks

MVG inserted 248 blanks into the sample stream. The source of the blank material used is unknown. Ninety-three per cent (or 230 samples) of samples passed with 18 samples (~7%) exceeding the threshold of x3 the detection limit (or 0.015 ppm). Overall, the performance of the blanks is considered acceptable with only three samples yielded values above the lower cutoff for the potential MRE (Figure 11.3).

Figure 11.3. Blank results from MVG gold fire assays.



11.6.2.2 MVG SRM

MVG used 10 SRMs (Table 11.3) from Mineral Exploration & Environmental Geochemistry of Reno, NV. All standard types have counts of <50 and relative standard deviation (RSD) values range from 50.5 to 4.4% and bias values range from -10.0 to 3.1%. Standard S104004X returned the highest RSD and largest bias; however, the low recommended value is extremely low at 0.032 ppm Au relative to the overall anticipated lower cutoff value for the MRE. Overall, the performance of the SRMs for FA analyses are considered acceptable.

Table 11.3. Summary of MVG's SRM results for FA analyses (note: 1SD = first standard deviation).

SRM	Recommended Au Value (ppm)	1SD (ppm)	Count (#)	RSD (%)	Bias (%)	Percentage Within 2SD (%)	Percentage Within 3SD (%)
S104004X	0.032	0.006	38	50.5	-10.0	82	92
S105002X	0.440	0.020	38	4.9	-0.1	92	95
S105003X	0.525	0.026	35	10.5	-2.2	69	86
S104008X	0.662	0.017	39	5.3	1.5	82	82
S104007X	0.750	0.016	39	9.8	1.5	67	82
C404002X	1.315	0.050	38	4.4	-0.5	92	97
S105001X	1.843	0.085	40	6.1	3.1	88	93
S105005X	2.416	0.083	37	5.2	2.0	81	92
S105004X	3.752	0.200	34	7.5	2.7	82	97
S105006X	4.516	0.099	36	4.4	-0.2	72	81

11.6.3 IMC

IMC inserted QA/QC samples in the sample stream at an interval of 1 in 10 and alternated between a blank and certified reference material (CRM). No field duplicates were included in the sample stream. IMC submitted 773 QA/QC samples (~9%; see Table 11.4) to ALS, together with 8,266 original RC and core samples.

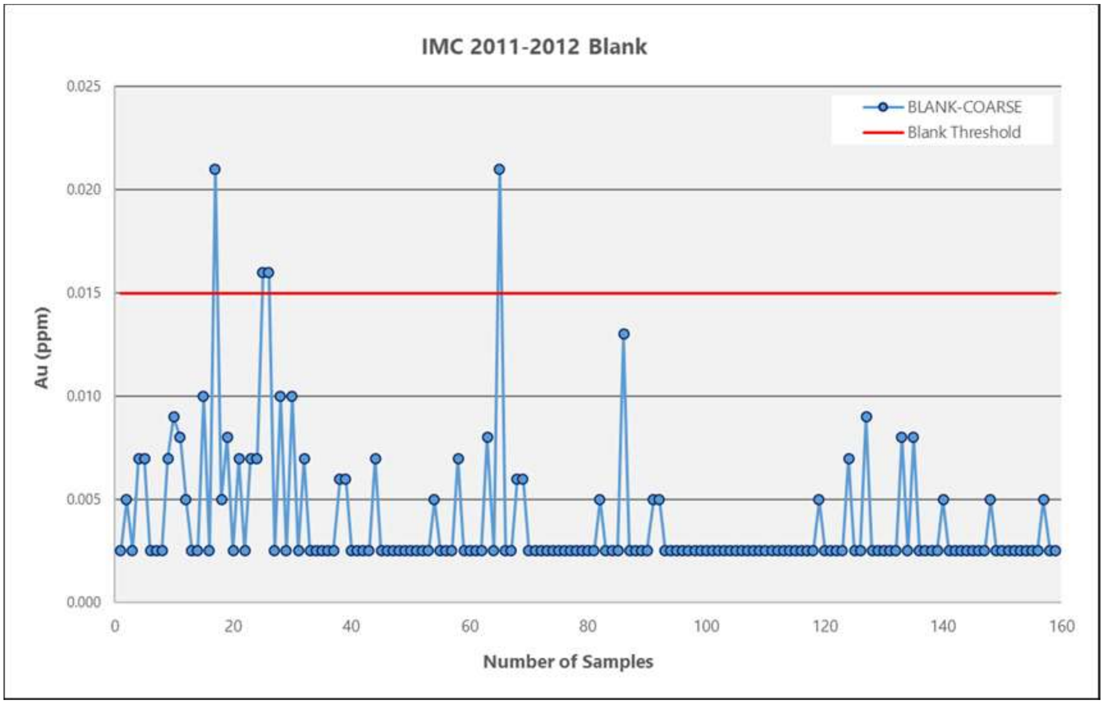
Table 11.4. Summary of QA/QC samples inserted into sample stream by IMC.

No. of Blanks	No. of CRMs	No. of Field Duplicates
159	614	0

11.6.3.1 IMC Blanks

IMC inserted 159 blanks into the sample stream. The source of the blank material used is unknown. Ninety-seven per cent (or 155) of samples passed with only four samples (~3%) slightly exceeding the threshold of x3 the detection limit (or 0.015 ppm; Figure 11.4). Overall, the performance of the blanks is considered acceptable.

Figure 11.4. Blank results from IMC gold fire assays.



11.6.3.2 IMC CRM

IMC used eight CRMs (Table 11.5) two from Rocklabs Ltd. (Rocklabs) of Auckland, New Zealand (SE44 and SE58), and six from CDN Resource Laboratories Ltd. (CDN) of Langley, BC, Canada (CDN-GS-1F, -1H, -2F, -2G, -2J, -7B). CDN and Rocklabs are commercial providers of certified reference materials.

Table 11.5. Summary of IMC's CRM results for FA analyses.

CRM/SRM	Certificate Au Value (ppm)	1SD (ppm)	Count (#)	RSD (%)	Bias (%)	Percentage Within 2SD (%)	Percentage Within 3SD (%)
SE44	0.606	0.017	35	2.3	0.2	100	100
SE58	0.607	0.019	123	4.1	-2.4	93	97
CDN-GS-1H	0.972	0.054	123	5.8	2.1	89	98
CDN-GS-1F	1.16	0.065	40	9.2	6.6	75	88
CDN-GS-2F	2.16	0.12	12	7.5	3.6	75	92
CDN-GS-2G	2.26	0.095	30	4.7	5.3	77	90
CDN-GS-2J	2.36	0.1	123	4.2	4.7	79	97
CDN-GS-7B	6.42	0.23	127	4.7	1.2	87	98

CRMs with counts of >50 (Table 11.5) returned acceptable RSD (<6%) and bias (-2.4 to 4.7%) values, compared to standards with fewer counts that had slightly higher RSD (<10%) and bias (0.2 to 6.6%). Overall, the performance of the CRMs for FA is considered acceptable.

11.6.4 CRL

For the 2017 drilling program, CRL inserted QA/QC samples in the sample stream at an interval of 1 in 10 and alternated between a blank and standard. No field duplicates were inserted as the remaining core was used for metallurgical test work. QA/QC sample tags were stapled in the core box and followed the original sample tag for reference purposes. CRL submitted 97 QA/QC samples (~10%; see Table 11.6) to ALS, along with 919 original core samples.

Table 11.6. Summary of QA/QC samples inserted into sample stream by CRL

No. of Blanks	No. of CRMs/SRMs	No. of Field Duplicates
35	62	0

All QA/QC data from the CRL drill program were exported from DataShed and imported into MS Excel to generate summary statistics and charts. Charts were created to identify anomalies. Guidelines for results that triggered further investigation included:

- A CRM outside ±3 SD;
- Two consecutive CRMs above +2 or below -2 SD;
- Blank control samples reporting a value >0.015 ppm Au

Sample investigation protocols included:

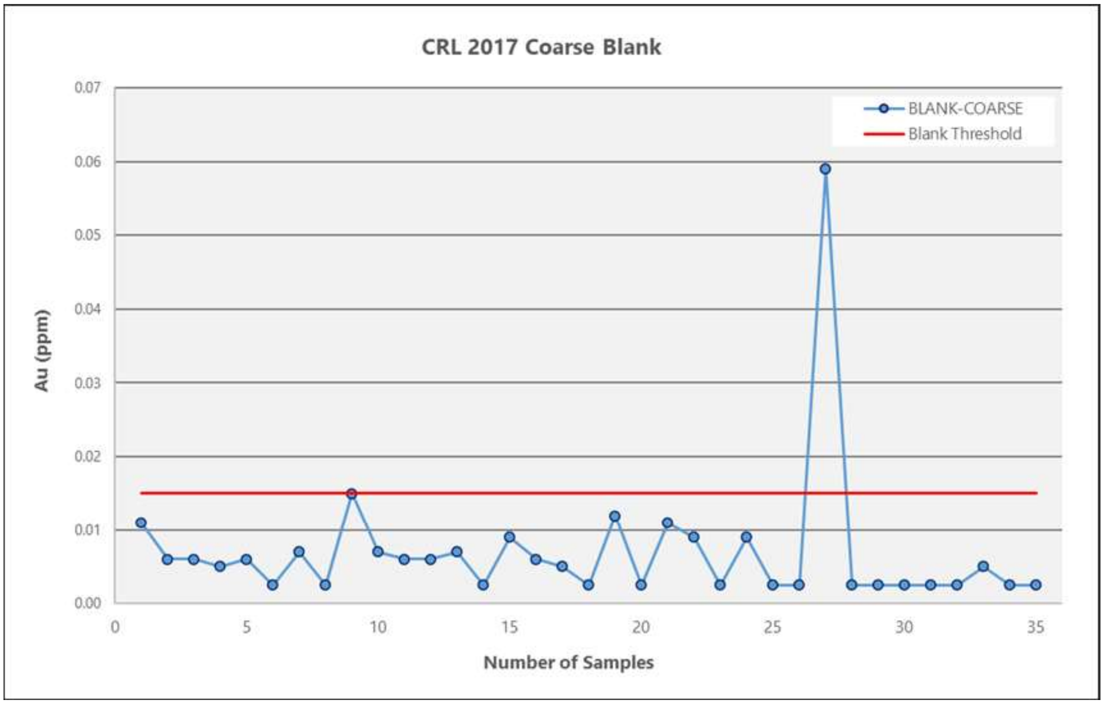
- Check for possible mis-labels or switched samples;
- Check reported sample weights;
- Check if the failure was within a mineralized interval;
- If the failed QA/QC samples were considered immaterial (e.g., a CRM above 3SD in an interval with below-detection-limit gold values), accept the results;
- If failed QA/QC results were considered material, notify the laboratory, and request that intervals containing QA/QC failures be re-assayed if justified;
- Review results from the re-assayed intervals;
- If results are acceptable, import the re-assays into the database, attaching “Corrected” to the batch identification;
- For the final database, export only those assay results that were accepted by the qualified geologist.

11.6.4.1 CRL Blanks

CRL inserted 35 blank samples in the sample stream for the 2017 drill program. The blank material was landscape marble acquired from a Home Depot in Reno, NV.

Gold analyses were carried out using the Au-AA23 method, which has a lower detection limit of 0.005 pm Au. One sample (or 3%) returned a value of 0.059 ppm Au but the immediately above samples were low grade ranging from 0.044 to 0.131 ppm Au, and the silver values were below detection limit. The remaining 34 blank samples (or 97%) were below the threshold limit (3x the lower detection limit), and the results were therefore considered acceptable (Figure 11.5).

Figure 11.5. Blank results from CRL gold fire assays.



11.6.4.2 CRL CRM

CRL used three gold CRM types in 2017 which were acquired from Rocklabs (OXG99 and SE68), and CDN (CDN-GS-2L). Overall, the results for CRMs used during the CRL drilling program were acceptable (Table 11.7; Figures 11.6 to 11.8) given the low number of results per CRM. The RSD ranged from 3.0 to 6.3%, and the bias ranged from −1.5 to 1.5%.

Table 11.7. Summary of CRL's CRM results.

CRM/SRM	Certificate Au Value (ppm)	1SD (ppm)	Count (#)	RSD (%)	Bias (%)	Percentage Within 2SD (%)	Percentage Within 3SD (%)
SE68	0.599	0.013	22	3.0	-0.6	82	100
OxG99	0.932	0.02	22	3.6	-1.5	82	91
CDN-GS-2L	2.34	0.12	18	6.3	1.5	83	100

Figure 11.6. CRM SE68 used by CRL in 2017.

Figure 11.7. CRM OXG99 used by CRL in 2017.

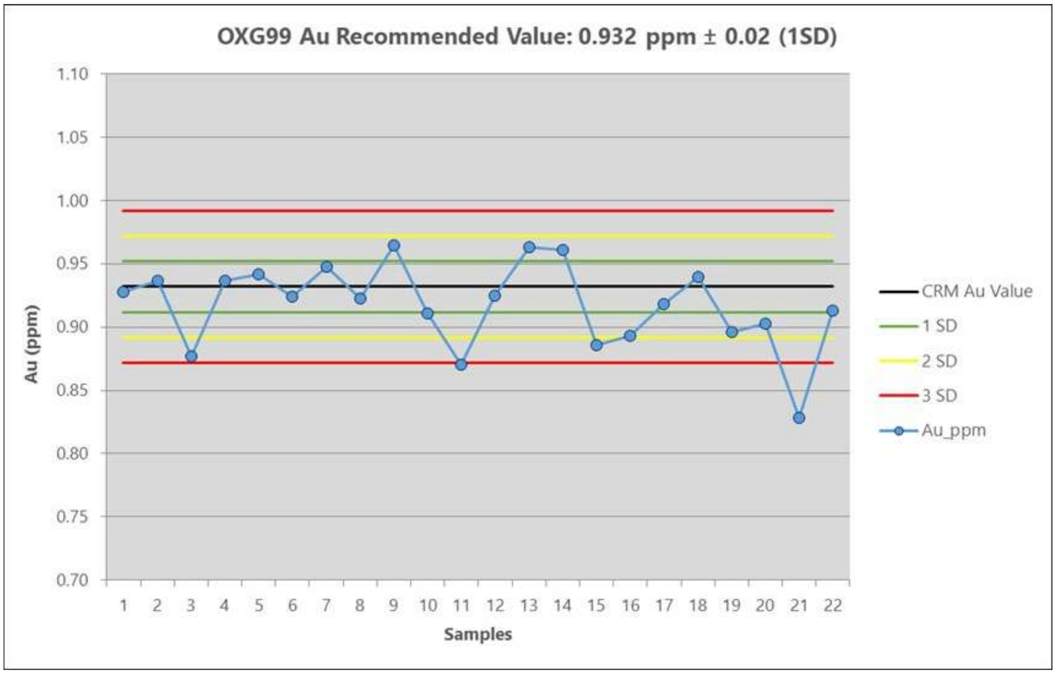
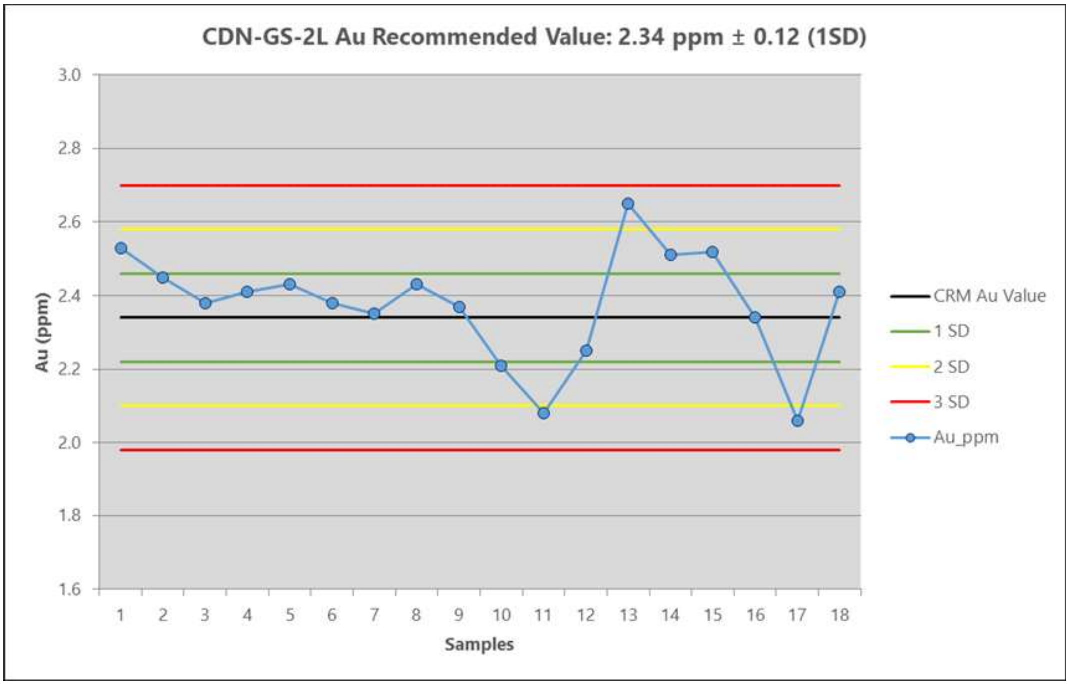


Figure 11.8. CRM CDN-GS-2L used by CRL in 2017



11.7 Databases

CRL imported historical drillhole data into DataShed software. All data generated during CRL's 2017 program was collected using MS Excel and imported directly into DataShed.

Final assay certificates were posted on ALS's internet-based Webtrieve portal, which facilitated downloading in .csv and .pdf formats. CRL directly uploaded these .csv files into DataShed. Imports were verified to ensure that DataShed values matched the final assay certificates. QC samples were also reviewed to confirm expected results.

The CRL database was maintained on the company's server in Salt Lake City, Utah, and nightly back-ups were made to a secure off-site location.

11.8 Sample Security

For the MVG drill program, samples were stored at the drill site until picked up by AAL. Sample pickups were scheduled to coincide with the drilling company's work schedule. For example, samples from the last hole drilled were picked at the end of their shift and were picked up the same day as the end of their rotation. For the IMC drill program, samples were held at the drill site, in possession of the drillers, until transported (either by the drilling contractor or by IMC personnel) to IMC's Reno storage facility.

For the CRL drill program, samples were held at the drill site, in possession of the drillers, until transported (either by the drilling contractor or by CRL personnel) to company's Lovelock logging and storage facility.

11.9 Comments

QA/QC procedures form a key component in supporting sample precision and accuracy, and therefore the validity of the data on which MREs are based. Through evaluating the QA/QC results for a combination of blanks, SRMs, CRMs, and different types of sample duplicates (field, crushed and pulverized), it is possible to assess potential sources of grade variability within the samples.

The APEX QPs reviewed the supplied blank, SRM, CRM, and duplicate sample submissions, and the laboratory and assay methods used. Based on the QA/QC results, the QPs are of the opinion that the sample preparation and assay methods are free from significant contamination. Assay methods are also considered to be reasonably accurate and, in the case of the SRM/CRM samples, to have a good level of precision.

It is the opinion of the APEX QPs of this report that the sample preparation, security, and analytical procedures adopted meet accepted industry standards, are adequate to ensure overall data quality and considered acceptable for the use herein as part of the MRE process.

12 Data Verification

12.1 Analytical Data Verification Procedures

The APEX QPs, Mr. Dufresne and Mr. Schoeman, reviewed 10% of the archived analytical geochemical certificates for historical drillholes completed between 1996 and 2017 on the Property. The historical holes were randomly selected and reviewed from top to bottom versus the values contained in the drillhole database. There were no significant differences with respect to the company's databases and the archived analytical certificates. In the opinion of the APEX QPs, industry standard procedures have been used that are acceptable for ensuring the accuracy of all analytical data pertaining to exploration and drilling work conducted by CRL and its predecessors.

All of the results for the 2017 QA/QC samples inserted by CRL personnel and by ALS at the laboratory were reviewed. In general, the Company-inserted SRMs and blanks yielded reasonable results with no significant analytical issues identified.

12.2 QA/QC Data Review 2017 Drillhole Program

All of the results for the 2017 QA/QC samples inserted by CRL personnel and by ALS at the laboratory were reviewed. In general, the Company-inserted SRMs and blanks yielded reasonable results with no significant analytical issues identified. Specific graphs and tables of the results are provided in Section 11.

The following QA/QC protocol was applied by the Company for the 2017 core sampling program at Converse:

- Blanks: one blank was inserted by Company or ALS personnel every 30 core samples.
- CCRMs: 1 CRM was inserted by Company or ALS personnel for every 10 core samples.

Table 12.1 provides a summary of all the Company-inserted standards, blanks and coarse blanks used by Converse. Most of the failures are considered marginal and not material.

Table 12.1. Summary of 2017 QA/QC samples used at the Converse Property

QA/QC samples used	Number used	Certified Value	Failed
CDN-GS-2L	18	2.34±0.12 ppm Au	3
OXG99	22	0.932±0.02 ppm Au	1
SE68	22	0.599±0.013 ppm Au	4

The initial QA/QC data review for the Converse Property indicated there were a number of significant failures for the ALS laboratory-inserted SRMs and blanks. However, most of these failures were related to use of a cyanide leach methodology on CRMs that were not designed to be tested using cyanide leach techniques.

12.3 Drillhole Collar and Downhole Survey Verification

The APEX QPs reviewed 10% of the cover sheets of the geological logs for historical drillholes completed between 1996 and 2012 on the Property in order to compare collar locations on the logs versus the database. The historical holes were randomly selected and reviewed versus the values contained in the drillhole database. There were two significant collar location differences with respect to the Company's databases and the archived geological logs. The database was confirmed as correct based upon a review of the survey files and the current locations of ground disturbances in the form of visible historical drill pads. A number of drill pads were verified on the ground and in Google Earth.

In the opinion of the APEX QPs, industry-standard procedures were used that are acceptable for ensuring the accuracy of all analytical data pertaining to exploration and drilling work conducted by CRL and its predecessors.

12.4 Qualified Person Site Inspection

Mr. Philo Schoeman, M.Sc., P.Geo., Pr.Sci.Nat. visited the Property on December 20, 2020, and verified the collar stake positions for the 2017 drilling program by handheld GPS. Collar stakes with numbered tags were located within 3.3 to 6.6 ft from the published collar positions for holes CNR-MET17-001 and 002, collar stakes only (no tags) for CNR-MET17-004 and 005 and depressions only on pads for CNR-MET17003, 006 and 007. Signs of extensive rehabilitation with a dozer on the drill pads were observed and likely resulted in the removal of the missing collar stakes and tags by accident.

On December 21, 2020, Mr. Schoeman collected 15 core samples from 1997, 2013 and 2017 historical drill core at the CRL core storage facility in Lovelock, NV (Table 12.2). These samples were delivered to the ALS Elko Laboratory on December 23, 2020 for verification analyses. ALS is an internationally accredited independent analytical company with ISO9001 and ISO/IEC 17025 certification. ALS is independent of the Company and the QPs of this Report. The assay results and a comparison with assay values obtained in 1997, 2012 and 2017 are shown in Table 12.2 and Figures 12.1 and 12.2.

There is some variance up and down between the 2020 sample results and the historical original assay results, however, the APEX QPs deem the variance to be within acceptable limits for field duplicates in hydrothermal gold systems based upon their experience. No discerable bias is detected and in general, the 2020 results are similar to the original results. The APEX QPs deem the data acceptable .

Table 12.2. Results of verification core sampling of 2017 and historical core from the Converse Property.

Hole ID	Batch No.	Original Sample ID	Depth From (ft)	Depth To (ft)	Historical				2020		
					Au (ppm) FA30 AAS	Au (ppm) CL30 AAS	Ag (ppm) AR-AAS or AR-ICPMS	APEX Sample ID	Au (ppm) FA30 AAS	Au (ppm) CL30 AAS	Ag (ppm) MS61 ICPMS
CNR-MET17-001	RE17144163	R857421	817	822	0.481	0.43	1.5	20PSC-001	0.747	0.56	1.65
CNR-MET17-001	RE17144163	R857422	822	826.5	4.4	4.27	0.9	20PSC-002	4.28	3.42	2.07
CONV-033C	RE12221800	CONV-033C 1210-1215	1210	1215	0.062	N/A	0.77	20PSC-003	0.07	0.05	1.11
CONV-033C	RE12221800	CONV-033C 1215-1220	1215	1220	1.71	N/A	31.5	20PSC-004	0.676	0.29	22
CONV-033C	RE12221800	CONV-033C 1220-1225	1220	1225	0.019	N/A	0.31	20PSC-005	0.015	<0.03	0.29
CONV-028C	WN12166026	CONV-028C 835-840	835	840	0.075	N/A	4.13	20PSC-006	0.099	0.05	4.03
CONV-028C	WN12166026	CONV-028C 840-845	840	845	3.53	N/A	4.73	20PSC-007	2.91	0.74	3.75
CONV-028C	WN12166026	CONV-028C 845-850	845	850	0.009	N/A	0.26	20PSC-008	0.015	<0.03	0.29
NK-032C	97-0702	NKC-32 890-893	890	893	0.605	N/A	N/A	20PSC-009	0.76	0.76	1.32
NK-032C	97-0702	NKC-32 893-894.5	893	894.5	1.24	N/A	N/A	20PSC-010	0.812	0.77	1.56
CNR-MET17-005	RE17173740_Rev2	V663384	912	917	0.944	0.18	3.1	20PSC-011	0.695	0.28	3.22
CNR-MET17-005	RE17173740_Rev2	V663385	917	922	2.57	0.75	1.8	20PSC-012	1.87	0.5	3.61
CNR-MET17-005	RE17173740_Rev2	V663387	922	927	0.447	0.27	1.1	20PSC-013	0.409	0.22	0.83
CNR-MET17-007	RE17178165_Rev	V663530	192	197	0.165	0.1	1.5	20PSC-014	0.251	0.13	1.37
CNR-MET17-007	RE17178165_Rev	V663531	197	202	0.175	0.14	1.6	20PSC-015	0.206	0.11	1.07

Figure 12.1. Comparison of 2017 and historical core fire assay values vs 2020 site visit fire assay values.

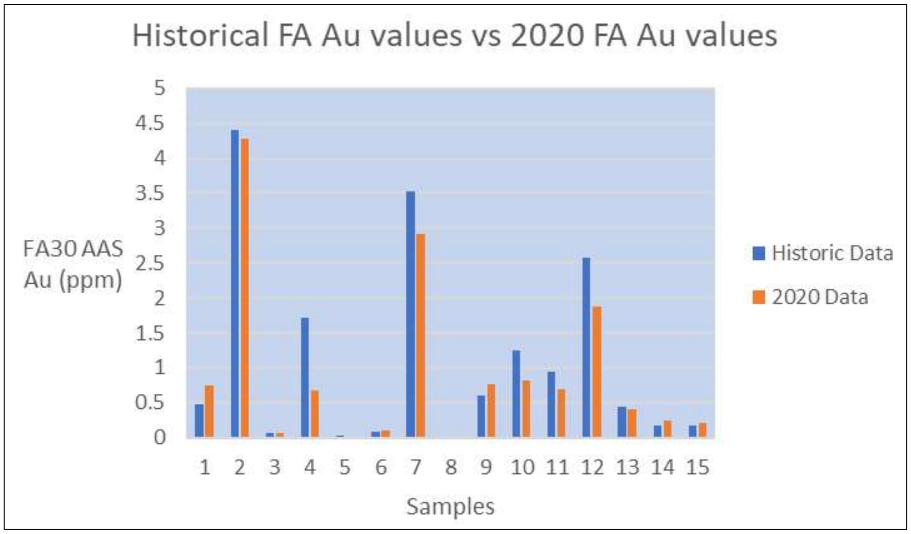
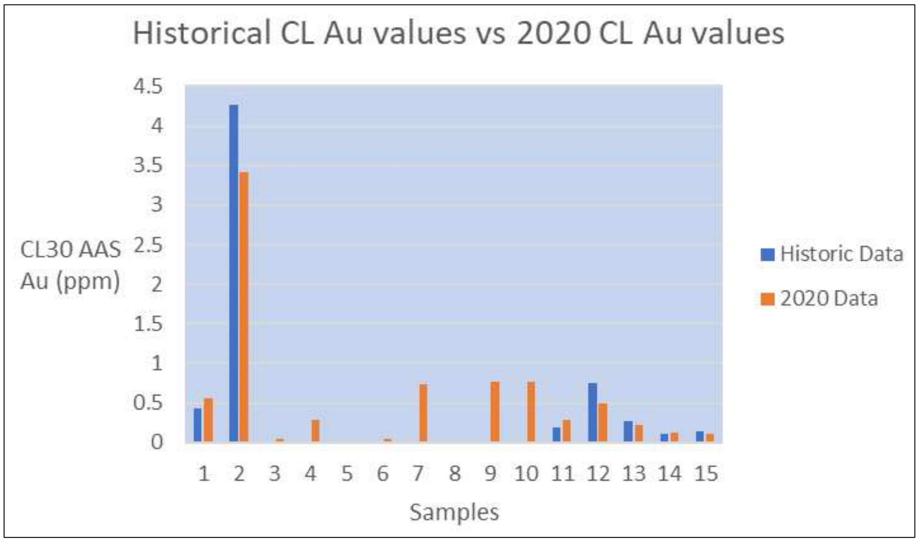


Figure 12.2. Comparison of 2017 and historical core cyanide leach assay values vs 2020 site visit cyanide leach assay values



12.5 Adequacy of the Data

The APEX QPs have reviewed the adequacy of the Converse Property's drillhole database. It is the opinion of the QPs that the data in the drillhole database is of sufficient quality for the purposes used in this Technical Report, including Mineral Resource Estimation.

13 Mineral Processing and Metallurgical Testing

13.1 Summary

A significant amount of metallurgical test work has been completed to date on both composite and variability samples. The samples were mainly drill core, and some assay rejects, collected from around the deposit. The sample grades are similar to the MRE grades and cover a wide range of oxidation states and other variables. The test work consists of bottle roll and column leach cyanidation tests, as well as comminution, gravity, and flotation testing.

The emphasis in the following summary sections is on gold recovery from the column testing. Bottle roll results are also included, where relevant. Other testing, including silver recovery, is mentioned briefly.

In conclusion, it is suggested to estimate gold recoveries from both the North and South Redline deposits using a simple formula based on the copper grade. Sulfide samples tend to have lower gold recoveries than transition and oxide samples, which are similar. This is explained by the fact that sulfide samples tend to have higher copper grades.

13.2 Metallurgical Test Work

From 2004, and up to December 31, 2020, four test work programs were carried out by KCA and MLI, with procedures and results summarized in a total of eleven reports and memoranda.

13.2.1 KCA, 2004/2005

This program is summarized in three reports:

- The first KCA program of test work (KCA, 2005a) included gravity concentration, fine and coarse grind bottle rolls. The samples used were nine composites selected by grade and degree of oxidation from eight drillholes. The grade of these composite samples varied from 0.021 to 0.044 oz/ton;

- The second KCA test program (KCA, 2005b) consisted of 750 bottle roll tests on assay reject (core and RC) samples from the exploration drilling program;

- The third KCA test program (KCA, 2005c) used the same samples as the first program (KCA, 2005a) and included column tests and Bond comminution tests.

The results from coarse bottle roll testing on samples ground to 80% passing 10 mesh (KCA, 2005b) indicate that gold extraction from samples in the North Redline deposit is slightly lower and the cyanide consumption is slightly higher than the South Redline deposit. Furthermore, the results suggest that at those relatively fine particle sizes, gold recovery is not particularly sensitive to mineralized zone, depth, grade, or oxidation state.

The KCA minus 200 mesh bottle roll leach tests using the nine composite samples (KCA, 2005a) gave gold recoveries of between 95 and 98%. All samples returned a gold tail grade of 0.001 oz/ton.

The column testing methodology used in this program is unusual in that the material used in the minus 1½ in test was removed from the column, crushed further to minus ¼ in and then re-leached. The recoveries obtained should be considered indicative only.

The tails from each one of these tests was collected and analyzed to determine the effect of crush size vs. recovery. Because the minus 1½ in material was essentially used for two separate leach tests, the amounts extracted from each phase of leaching were added to develop an overall expected recovery for the minus ¼ in material.

The overall average recoveries were then calculated for each size distribution. A trendline was fitted to the graph and a theoretical recovery prediction was generated out to a particle size of 6 in. This was used to obtain the theoretical recovery prediction. Table 13.1 shows the results of this analysis.

Table 13.1. 2004/2005 column test summary.

KCA Comp No	Zone	Description	Avg Head Au Grade (oz/ton)	% Au Rec at <1½ in	% Au Rec at 3/8 in	Expected %Au Rec at <1/4 in
32101	North	Mixed, Low-grade	0.027	28	59	63
32102	North	Mixed, Medium grade	0.048	28	60	63
32103	North	Oxide, Medium grade	0.040	32	60	61
32104	North	Sulfide, Medium grade	0.034	26	53	56
32105	North	Sulfide, Medium grade	0.036	21	54	58
32106	South	Mixed, Low-grade	0.025	35	67	66
32107	South	Mixed, Medium grade	0.027	56	74	79
32108	South	Oxide, Low-grade	0.027	44	74	74
32109	South	Sulfide, Low-grade	0.023	22	54	58

Despite the unusual methodology, it can be seen that the recovery increases at finer crush sizes as would be expected.

Three composite samples were selected for grindability testing (Table 13.2). The testing consisted of Bond rod mill (Rwi), ball mill (Bwi) and abrasion index (Ai) testing. As would be expected, the sulfide sample showed the highest overall hardness index while the oxide showed the lowest. All samples were moderately hard and very abrasive.

Table 13.2. Summary of Bond rod mill, ball mill and abrasion index testing.

KCA Comp No.	Oxidation State	Rwi (kWhr/ton)	Bwi (kWhr/ton)	Ai (g)
33103	Oxide	15.7	14.6	0.490
33104	Mixed	18.0	15.4	0.662
33105	Sulfide	18.0	15.5	0.701

13.2.2 MLI, 2009

A detailed metallurgical testing program was undertaken by MLI in 2009 on a total of 50 drill core composites. The program evaluated metallurgical response variability by conducting bottle roll tests on each of 39 sample variability drill core composites. These composites were prepared based on mineralization zone, oxidation, depth, grade (gold and copper) and rock type. Results from those tests were then used to prepare 11 larger metallurgical composites for further testing. Column testing on four lower value composites and one overall master (mineralization grade) composite included evaluation of crush size, reagent addition and consumption. The higher value composites were evaluated for gravity concentration, gravity/cyanidation, and bulk sulfide flotation treatment methods.

Summary results from the bottle roll tests, conducted on 39 of the drill core composites at an 80% passing 1.7 mm feed size, are shown in Table 13.3.

Table 13.3. Summary bottle roll tests.

Mineralization Type	No. of Composites	Au Recovery (%)	Calc. Head Au Grade (g/t)	NaCN Consumed (kg/t)
North Redline	17	65.7	2.00	1.10
South Redline	22	64.6	0.89	0.55
Oxide	14	67.7	0.64	0.29
Mixed	22	63.0	1.83	1.02
Sulfide	3	68.6	1.44	1.49
Siltstone	7	64.3	1.29	0.94
Sandstone	8	66.7	1.05	0.74
Chert	5	60.9	1.33	1.16
Porphyry	3	54.1	0.57	0.62

The testing confirmed that a reasonable recovery can be achieved at the 1.7 mm particle size. It showed that the various mineralization types evaluated were consistent in their response to cyanidation. Overall, the recovery was not particularly sensitive to mineralization zone, depth, grade or oxidation. Cyanide consumption tended to increase with increasing copper content of the ore.

Column leach tests were conducted on four low-grade core composites, designated North and South low-grade oxide (NLGOx & SLGOx), low-grade high copper (LGHiCu Master) and low-grade high sulfur (LGSulf Master), at an 80% passing 9.5 mm feed size, to determine heap leach amenability and evaluate reagent addition and consumption. After preliminary results from those tests were reviewed, column leach tests were conducted on a master core composite, at feed sizes ranging from 54% passing 25 mm to 80% passing 1.7 mm, to select the optimum heap leach feed size. Summary results from the column tests are presented in Table 13.4.

Table 13.4. Summary column and bottle roll tests

Composite	Test	Feed Size (mm)	NaCN (g/L)	Leach Time (days)	Au Rec (%)	Head Au Grade (g/t)	NaCN Cons. (kg/t)	Lime Added (kg/t)
NLGOx	Column	9.5	1.00	141	51.0	0.49	1.69	1.1
NLGOx	Bottle Roll	1.7	1.00	7	72.0	0.93	0.66	1.9
SLGOx	Column	9.5	1.00	142	65.3	0.49	1.91	1.7
SLGOx	Bottle Roll	1.7	1.00	7	68.9	0.61	0.51	2.9
LGHiCu Master	Column	9.5	1.00	188	54.8	1.04	3.37	1.7
LGHiCu Master	Bottle Roll	1.7	1.00	7	64.9	1.14	1.89	2.8
LGSulf Master	Column	9.5	1.00	118	34.5	0.55	1.52	1.7
LGSulf Master	Bottle Roll	1.7	1.00	7	62.7	0.59	2.48	2.8
Master Comp	Column	25	0.50	120	35.1	0.74	1.14	1.0

Composite	Test	Feed Size (mm)	NaCN (g/L)	Leach Time (days)	Au Rec (%)	Head Au Grade (g/t)	NaCN Cons. (kg/t)	Lime Added (kg/t)
Master Comp	Column	13	0.50	122	45.5	0.88	1.29	1.0
Master Comp	Column	9.5	0.50	117	50.6	0.89	1.37	1.0
Master Comp	Column	6.3	0.50	122	59.6	0.89	1.38	1.0
Master Comp	Column	1.7	0.50	124	78.2	0.87	1.63	1.0
Master Comp	Bottle Roll	1.7	1.00	7	67.9	1.06	1.06	1.4

Note: only the tests carried out at a cyanide addition of 1.0 g/L are shown. In most cases, these tests gave the highest recoveries.

Gold recoveries obtained from the NLGOx, SLGOx and LGHiCu Master composites, at an 80% passing 9.5 mm feed size were 46.3%, 49.1% and 52.0%, respectively, after 140 to 188 days of leaching and rinsing. Gold recovery obtained from the low-grade "high sulfur" (0.81% total S) composite (LGSulf Master) was significantly lower (33.3%).

Results from these tests confirmed the sensitivity to crush size and indicated that conventional crushing to approximately 6 mm would be required to obtain acceptable heap leach gold recoveries. Gold extraction was progressing at a very slow rate when leaching was terminated. Longer leaching cycles would improve gold recoveries slightly.

Cyanide consumptions were fairly high and increased somewhat with decreasing feed size. The 1.0 kg/t lime added before leaching was sufficient for maintaining protective alkalinity during leaching.

Gravity recoverable gold (GRG) tests were conducted on six high value mineralization composites to determine response to gravity concentration treatment. The composites responded reasonably well to gravity concentration treatment. Total GRG content ranged from 55.6 to 84.3%. Gravity recoverable gold tended to be liberated at relatively coarse (850 to 212 µm) grind sizes. These results indicate good potential for producing high-grade gravity concentrates from the high-grade mineralization.

Scoping-level gravity concentration tests with agitated cyanidation of the gravity tailings were conducted on the same six high-grade mineralization composites to evaluate potential for a grinding circuit, or a combined grinding/heap leaching ("pulp agglomeration" type) circuit for processing the high-grade material. Results showed that all six composites were readily amenable to gravity/cyanidation treatment at 80% passing 212 µm and 75 µm grind sizes. Combined (gravity/cyanidation) gold recoveries ranged from 84.9 to 96.6% at the 212 µm feed size and from 88.3 to 98.0% at the 75 µm feed size.

Scoping-level flotation tests were conducted on four high-grade, high sulfur and high copper composites, at an 80% passing 75 µm feed size, to determine response to conventional bulk sulfide flotation treatment. Results showed that all four composites responded reasonably well to conventional flotation treatment. Gold recoveries to the flotation rougher concentrates ranged from 79 to 89%. Copper recoveries from the high copper composites were high.

13.2.3 MLI, 2013

13.2.3.1 MLI, 2013a

A total of 20 drill core composites were prepared for the testing program. These consisted of seven DHC composites, and thirteen UC composites. Of these, four of the DHC composites were combined to produce two master composites: 004C DHC1 and 013 DHC1.

Direct head assays show that the composites generally ranged in grade from 0.37 to 1.46 g/t Au. Silver grades were generally 5 g/t Ag or lower. Silver grades of the 007 UC3 and 010C UC2 composites were significantly higher (13 and 10 g/t Ag, respectively).

A bottle roll test was conducted on each composite at an 100% passing 9.5 mm feed size, to determine gold and silver recoveries, leach times and reagent consumptions. A parallel bottle roll test was conducted on each composite, at an 80% passing 75 µm feed size, to evaluate feed size sensitivity. Column leach tests were conducted on each composite at a 100% passing 9.5 mm feed size, to determine heap leach amenability. Summary results are presented in Tables 13.5 and 13.6.

Table 13.5. Summary cyanidation tests (DHC composites).

Composite	Test	Feed Size	Au Rec (%)	Avg Head Au Grade (g/t)	NaCN Cons (kg/t)	Lime Added (kg/t)
004C DHC1	CLT	100%<9.5 mm	65.2	0.66	2.41	0.5
004C DHC1	BRT	100%<9.5 mm	46.9	0.66	0.31	0.5
004C DHC1	BRT	80%<75 µm	95.8	0.66	0.56	0.9
005C DHC1	CLT	100%<9.5 mm	61.0	0.62	2.54	0.5
005C DHC1	BRT	100%<9.5 mm	33.3	0.62	0.47	0.5
005C DHC1	BRT	80%<75 µm	95.2	0.62	0.94	0.9
007C DHC1	CLT	100%<9.5 mm	50.0	0.57	2.41	0.5
007C DHC1	BRT	100%<9.5 mm	37.3	0.57	0.68	0.5
007C DHC1	BRT	80%<75 µm	94.7	0.57	1.12	0.9
010C DHC1	CLT	100%<9.5 mm	54.3	0.38	2.76	0.5
010C DHC1	BRT	100%<9.5 mm	48.6	0.38	0.60	0.5
004C DHC1	CLT	100%<9.5 mm	65.2	0.66	2.41	0.5
004C DHC1	BRT	100%<9.5 mm	46.9	0.66	0.31	0.5
004C DHC1	BRT	80%<75 µm	95.8	0.66	0.56	0.9
005C DHC1	CLT	100%<9.5 mm	61.0	0.62	2.54	0.5

Note: 1) BRT = bottle roll test (1.0 g/L NaCN, 48 hours), CLT = column leach test (1.0 g/L NaCN, 57-85 days).

Table 13.6. Summary cyanidation tests (UC composites).

Composite	Test	Feed Size	Au Rec (%)	Avg Head Au Grade (g/t)	NaCN Cons (kg/t)	Lime Added (kg/t)
004C UC1	CLT	100%<9.5 mm	69.0	0.77	2.82	0.5
004C UC1	BRT	100%<9.5 mm	42.0	0.77	0.16	0.5
004C UC1	BRT	80%<75 µm	97.2	0.77	0.10	0.6
004C UC2	CLT	100%<9.5 mm	65.6	0.93	2.66	0.8
004C UC2	BRT	100%<9.5 mm	56.8	0.93	0.15	0.8
004C UC2	BRT	80%<75 µm	98.9	0.93	0.50	0.7
004C UC3	CLT	100%<9.5 mm	52.2	0.88	2.38	0.6
004C UC3	BRT	100%<9.5 mm	53.5	0.88	0.23	0.6
004C UC3	BRT	80%<75 µm	96.6	0.88	0.85	0.5
005C UC1	CLT	100%<9.5 mm	58.4	1.00	3.41	0.9
005C UC1	BRT	100%<9.5 mm	44.6	1.00	0.30	0.9
005C UC1	BRT	80%<75 µm	95.6	1.00	0.93	0.7
005C UC2	CLT	100%<9.5 mm	46.3	0.88	3.11	0.9
005C UC2	BRT	100%<9.5 mm	36.9	0.88	0.46	0.9
005C UC2	BRT	80%<75 µm	95.3	0.88	1.32	0.7
005C UC3	CLT	100%<9.5 mm	54.5	0.41	2.17	0.8
005C UC3	BRT	100%<9.5 mm	46.7	0.41	0.30	0.8
005C UC3	BRT	80%<75 µm	97.6	0.41	0.70	0.7
007C UC1	CLT	100%<9.5 mm	54.5	0.42	2.66	0.9
007C UC1	BRT	100%<9.5 mm	55.6	0.42	0.92	0.9
007C UC1	BRT	80%<75 µm	91.7	0.42	1.49	0.7
007C UC2	CLT	100%<9.5 mm	41.4	0.91	2.70	0.5
007C UC2	BRT	100%<9.5 mm	31.6	0.91	0.91	0.5
007C UC2	BRT	80%<75 µm	90.1	0.91	1.57	0.7
007C UC3	CLT	100%<9.5 mm	51.4	1.23	2.82	1.1
007C UC3	BRT	100%<9.5 mm	41.5	1.23	1.58	1.1
007C UC3	BRT	80%<75 µm	91.8	1.23	2.04	0.9
010C UC1	CLT	100%<9.5 mm	78.7	0.96	4.08	0.5
010C UC1	BRT	100%<9.5 mm	62.1	0.96	0.67	0.5
010C UC1	BRT	80%<75 µm	96.9	0.96	0.91	0.7
010C UC2	CLT	100%<9.5 mm	55.3	0.47	3.56	0.5
010C UC2	BRT	100%<9.5 mm	54.4	0.47	1.58	0.5
010C UC2	BRT	80%<75 µm	95.1	0.47	1.66	0.7
013C UC1	CLT	100%<9.5 mm	59.0	0.59	2.92	0.5
013C UC1	BRT	100%<9.5 mm	46.8	0.59	0.48	0.5
013C UC1	BRT	80%<75 µm	91.1	0.59	1.04	0.5
013C UC2	CLT	100%<9.5 mm	50.0	0.64	2.81	0.5

Composite	Test	Feed Size	Au Rec (%)	Avg Head Au Grade (g/t)	NaCN Cons (kg/t)	Lime Added (kg/t)
013C UC2	BRT	100%<9.5 mm	40.4	0.64	0.39	0.5

Note: 1) BRT = bottle roll test (1.0 g/L NaCN, 48 hours), CLT = column leach test (1.0 g/L NaCN, 57-85 days).

Overall, the results show that recoveries were very sensitive to feed size. Column leach test gold recoveries, at a 100% passing 9.5 mm feed size, ranged from 35.5 to 59.0% for most (13) of the composites. Silver recoveries ranged from 7.5 to 67.7%.

Column test gold recovery rates were rapid and gold extractions were substantially complete in 20 days of leaching. Gold extraction was still progressing at the end of the leaching cycle, albeit at a slow rate. Cyanide consumptions were high and lime requirements were low.

Bottle roll test gold recoveries, at a 100% passing 9.5 mm feed size, ranged from 31.6 to 62.1%. Grinding the composites finer to an 80% passing 75 µm feed size improved the range of gold recoveries to 90.1 to 98.1%.

Bottle roll test silver recoveries from the composites that contained greater than 5 g/t Ag ranged from 19.0 to 56.4%, at an 100% passing 9.5 mm feed size, and from 25.6 to 81.0%, at an 80% passing 75 µm feed size. Silver recoveries from the remaining composites ranged from 5.4 to 44.1%, at an 100% passing 9.5 mm feed size, and from 8.8 to 85.7%, at an 80% passing 75 µm feed size.

Grinding/cyanidation gold recovery rates were rapid. Cyanide consumptions ranged from moderate to high and lime requirements were low.

13.2.3.2 MLI, 2013b

As part of the program and reported in MLI (2013b), further tests were conducted on five drill core composites to select the optimum grind size for agitated cyanidation treatment. A bottle roll test was conducted on each of the five "DHC1" composites, at feed sizes of 80% passing 420 µm, 212 µm, 150 µm, 106 µm and 75 µm. Head assays showed that the five composites ranged in grade from 0.51 to 0.76 g/t Au. None of the composites contained greater than 5.0 g/t Ag. Sulfide sulfur content ranged from 0.03 to 0.52%. Summary results are shown inTable 13.7.

Table 13.7. Summary bottle roll tests, drill core composites.

Composite	Feed Size (µm)	Au Rec (%)	Avg Head Au Grade (g/t)	NaCN Cons (kg/t)	Lime Added (kg/t)
004C	420	86.6	0.70	0.54	0.6
	212	89.6	0.70	0.44	0.8
	150	92.4	0.70	0.57	0.7
	106	97.2	0.70	0.60	0.8
	75	>98.4	0.70	0.56	0.9
005C	420	81.8	0.68	0.81	0.8
	212	88.2	0.68	0.98	0.6
	150	94.3	0.68	1.06	0.7
	106	95.5	0.68	1.16	0.7
	75	95.4	0.68	1.17	0.8

Composite	Feed Size (µm)	Au Rec (%)	Avg Head Au Grade (g/t)	NaCN Cons (kg/t)	Lime Added (kg/t)
007C	420	79.7	0.62	1.01	0.8
	212	86.8	0.62	1.05	0.6
	150	90.6	0.62	1.27	0.7
	106	93.9	0.62	1.34	0.6
	75	95.2	0.62	1.24	0.9
010C	420	80.6	0.43	1.09	0.7
	212	88.1	0.43	1.05	0.6
	150	>97.2	0.43	1.04	0.8
	106	>97.4	0.43	1.06	0.9
	75	>97.5	0.43	1.17	1.0
013C	420	80.8	0.74	0.64	0.8
	212	89.5	0.74	0.68	0.8
	150	94.6	0.74	0.75	0.7
	106	97.0	0.74	0.70	0.9
	75	98.7	0.74	0.66	1.0

Bottle roll results showed that all five of the composites were readily amenable to grinding/cyanidation treatment, at the feed sizes evaluated, and recoveries were sensitive to feed size. The indicated optimum feed size, with respect to gold recovery, was 80% passing 75 µm. Gold recoveries achieved at that feed size ranged from 95.2 to 98.7%. Gold recovery decreased with coarsening grind size and was an average of 15% lower at the coarsest size evaluated (80% passing 420 µm).

Gold recovery rates were rapid, and gold extraction was substantially complete in 16 to 24 hours of leaching. Cyanide consumptions were moderate to high and tended to increase incrementally with decreasing feed size. Cyanide consumptions for the 75 µm feeds ranged from 0.56 to 1.24 kg/t NaCN and averaged 0.96 kg/t NaCN. Lime requirements were low and did not exceed 1 kg/t.

13.2.3.3 MLI, 2013c

As part of the program and reported in MLI (2013c), five drill core composites from an earlier testing program (Ref. MLI Project No. 3729) were combined to produce two drill core composites designated North and South Redline deposits.

Predicted gold head grades for the North and South Redline composites (0.63 and 0.72 g/t Au, respectively) agreed well with the head grades calculated from the bottle roll tests. Predicted silver grades were 3.1 and 1.7 g/t Ag, respectively.

Whole-ore grinding/cyanidation tests were conducted on each composite, using a solids density of 40% and cyanide concentrations of 0.5 and 1.0 g/L NaCN, to optimize solution cyanide concentration during leaching. Comparative grinding/cyanidation tests were conducted on the South composite at a higher solids density (50%) and cyanide concentrations of 0.5 and 1.0 g/L NaCN, to optimize solution cyanide concentration and solids density during leaching. All tests were conducted at an 80% passing 150 µm feed size. Summary results from cyanidation tests are presented in Table 13.8.

Table 13.8. Summary bottle roll tests.

Composite	Solids (%)	NaCN Conc (g/L)	Addition (hours)	Rec Au (%)	Calc Avg Head Au (g/t)	Rec Ag (%)	NaCN Cons (kg/t)	Lime Added (kg/t)
North	40	0.50	36	89.1	0.55	0.59	53.3	0.45
North	40	0.50	24	89.7	0.58	0.59	55.2	0.50
North	40	1.00	36	90.2	0.61	0.59	56.7	0.54
North	40	1.00	24	90.0	0.60	0.59	53.3	0.64
South	40	0.50	36	92.6	0.68	0.67	47.1	0.15
South	40	0.50	24	92.6	0.68	0.67	47.1	0.15
South	40	1.00	36	92.5	0.67	0.67	47.1	0.30
South	40	1.00	24	92.6	0.68	0.67	47.1	0.32
South	50	0.50	36	92.4	0.66	0.67	47.1	0.26
South	50	1.00	36	91.9	0.62	0.67	50.0	0.29

Bottle roll test results show that the North and South composites were readily amenable to cyanidation, under the conditions evaluated. Optimization testing showed that no significant difference in gold recovery occurred by varying solution cyanide concentration from 0.5 to 1.0 g/L NaCN. Gold recoveries obtained from these composites at 0.5 and 1.0 g/L NaCN ranged from 89.1 to 92.6%. Silver recoveries ranged from 47.1 to 56.7%. Cyanide consumptions were moderate, and not sensitive to cyanide concentration, addition schedule or solids density. Lime requirements were low.

13.2.3.4 MLI, 2013d

As part of the program and reported in MLI (2013d), a total of 50 drill core composites were received. Of these, 25 were selected for testing. Direct head assays showed that the composites ranged in grade from 0.15 to 1.76 g/t Au. Silver grades were generally 5.4 g/t Ag or lower. Silver grades of the 002C (260-270 ft), 008C (355-365 ft), and 013C (320-330 ft) composites were significantly higher (11.0, 12.5, and 8.0 g/t Ag, respectively). Copper grades ranged from 41 to 1,189 ppm for 22 of the composites, and from 1,822 to 4,910 ppm for the remaining composites. Sulfide sulfur grades from all the composites tested ranged from <0.01 to 1.36%.

Whole-ore grinding/cyanidation tests were conducted on each composite at feed sizes ranging from 80% passing 212 µm to 80% passing 75 µm to optimize grind size. Overall metallurgical results show that most (22) of the composites were readily amenable to whole-ore grinding/cyanidation treatment, at the feed sizes evaluated. Gold recoveries obtained from these composites at an 80% passing 212 µm feed size ranged from 79.1 to 93.7%. The remaining three composites were not particularly amenable to whole-ore grinding/cyanidation treatment, at the feed sizes evaluated. Gold recoveries obtained from these composites at an 80% passing 212 µm feed size ranged from 38.0 to 54.3%. Two of these three composites came from drillhole 002C, and two contained relatively high sulfide sulfur levels (1.34 to 1.36%). Grinding from 212 to 150 µm improved gold recoveries (0.4 to 9.5%) for 17 of the 25 composites. Grinding from 150 to 106 µm improved gold recoveries (0.3 to 3.9%) for 14 of the 25 composites. Grinding from 106 to 75 µm improved gold recoveries (0.2 to 7.5%) for 17 of the 25 composites. The observed improvements in gold recovery generally fell within experimental and analytical precision limits, considering the low-grade nature of many of the samples. Only three composites contained greater than 6 g/t Ag. Silver recoveries obtained from those composites were as high as 32.0% (002C (260-270 ft), 35.2% 008C (355-365 ft), and 61.0% 013C (320-330 ft). Cyanide consumptions ranged from low to very high (0.07 to 4.45 kg/t NaCN). Lime requirements were low (0.4 to 1.3 kg/t).

13.2.4 KCA, 2018

This test work program is the most detailed of the four programs completed to date.

Seven metallurgical holes were drilled, within the proposed North and South pit limits, as shown in Figure 13.1. The holes covered the mineralized zones and aimed to cut across different redox zones and depths. Test data on the drill core confirmed the influence of copper on gold solubility (CN/FA%). Based on this, five distinctive metallurgical zones were identified:

- North – low & high copper;
- South – low & high copper;
- South – high sulfide.

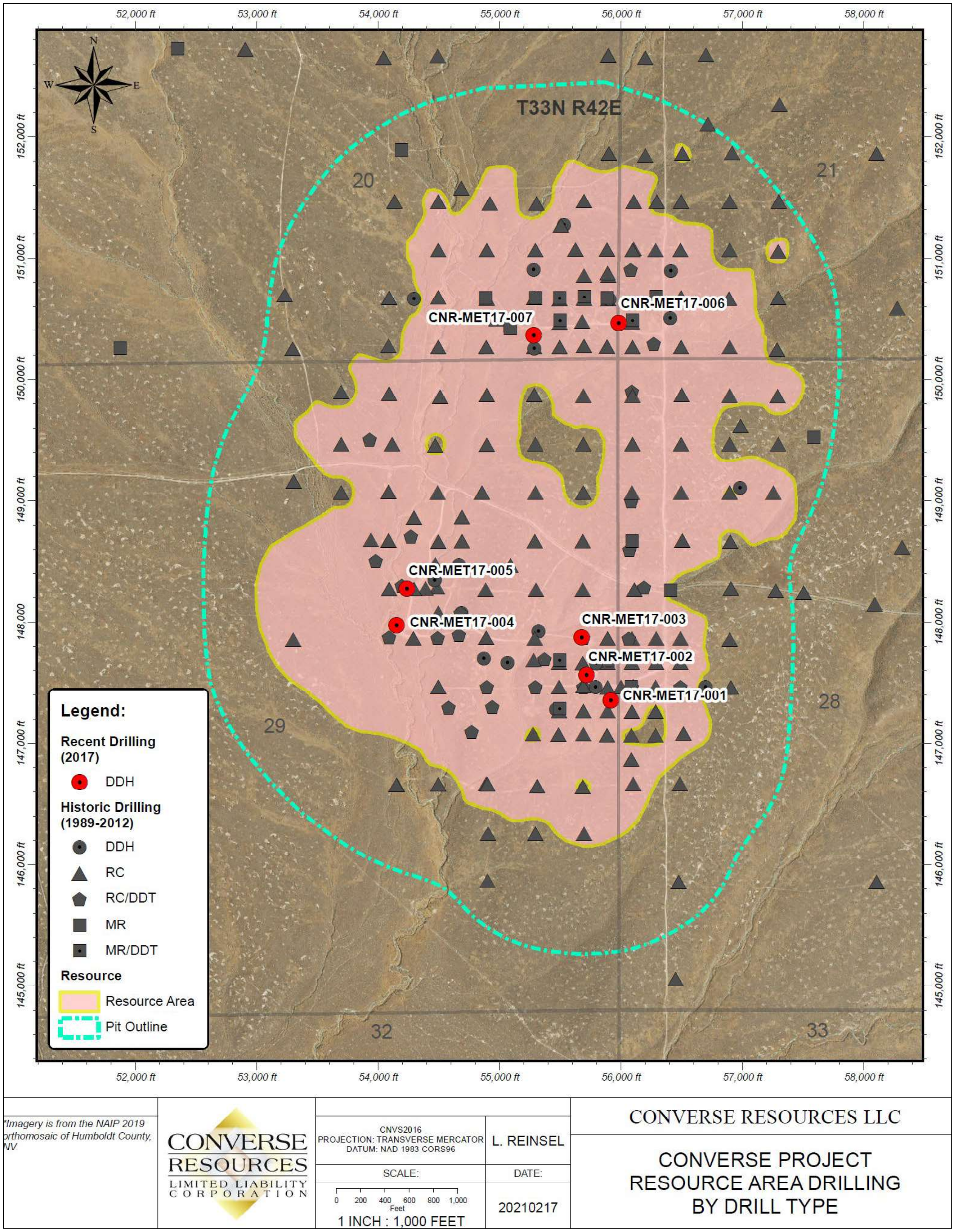
KCA received thirty-one pallets of core from the seven drillholes. From this, five south zone composites (PS 1-5; Table 13.9) and five north zone composites (PN 1-5; Table 13.10) were prepared. These composites were conventionally crushed and used for head analyses, head screen analyses with assays by size fraction, bottle roll leach, agglomeration, and column leach test work. Analytical and other details of these composites are shown in the tables, together with the column test gold recovery and cyanide consumption.

Additional samples were then cut from each composite for high pressure grinding roll (HPGR) crushing test work. The HPGR crushed material was used for agglomeration and column leach test work. In addition to the ten composites listed above, 11 variability composites were prepared (VS 1-6 and VN 1-4). These composites were used for bottle roll leach and agglomeration test work. Similarly, four composites were prepared for comminution test work, and five composites were generated for optical sorting tests.

Table 13.9. PS composites.

Composite Description	PS1 Base	PS2 High Cu	PS3 Low Au	PS4 High Au	PS5 Sulfide
Redox Class	1-21	1-21	21	1-21	3
Au, g/t	0.732	0.890	0.504	1.157	0.667
Cu, ppm	91	933	254	76	1,020
Sulfide S, %	0.01	0.08	0.01	0.01	2.39
9.5 mm Column test Au recovery %	81	84	74	69	56
9.5 mm Column test NaCN consumption kg/t	1.74	2.86	1.96	2.04	2.26

Figure 13.1. Metallurgical hole locations.



Source: CRL (2021)

Table 13.10. PN composites.

Composite Description	PN1 Base	PN2 High Cu	PN3 Low Au	PN4 High Au	PN5 Sulfide
Redox Class	21-22-23	21-23	21-22-23	21-22-23	24-3
Au, g/t	0.855	1.353	0.535	0.993	0.989
Cu, ppm	811	1,470	500	766	1,260
Sulfide S, %	0.12	0.11	0.02	0.01	0.13
9.5 mm Column test Au recovery %	62	68	56	60	43
9.5 mm Column test NaCN consumption kg/t	2.26	2.90	2.0	2.4	2.36

13.2.4.1 Agglomeration and Compaction Tests

Preliminary agglomeration test work was conducted on the crushed material. The agglomerated material was placed in a column with no compressive load and then tested for permeability.

Agglomeration tests were conducted on samples of conventionally-crushed material and HPGR-crushed material. This material was agglomerated with 0, 2, 4, and 8 kg/t cement. All conventionally crushed samples passed the KCA criteria. Of the HPGR crushed samples, five failed the criteria (PS1, PS2, PS5, PN3 and PN4) when no cement or 2 kg/t of cement were used. It was determined from these tests that a cement level of 4 kg/t would be used in the column leach tests. All conventionally crushed, variability composites passed the KCA criteria.

After column leaching, compacted permeability test work was conducted on column tailings. Separate test samples were loaded into a column and subjected to loads equivalent to 20, 40, 60, 80, 100, 120 and 140 m of overall heap height.

All tests passed with regard to flow. However, the following tailings material failed due to excessive slump:

- Conventionally crushed PS1 at 140 m;
- Conventionally crushed PS2 at 120 and 140 m;
- Conventionally crushed PN2 at 140 m;
- Conventionally crushed PN5 at 140 m;
- HPGR crushed PS1 at 120 and 140 m;
- HPGR crushed PS2 at 100, 120 and 140 m;
- HPGR crushed PN3 at 120 and 140 m;
- HPGR crushed PN5 at 140 m.

13.2.4.2 Column Tests

Column leach tests were conducted using conventionally crushed, as well as HPGR-crushed material. During testing, the material was leached for between 132 and 143 days. After the leaching process, the tailings material from 12 of the column leach tests (PS1, PS2, PS5, PN2, PN3 and PN5 material crushed conventionally and by HPGR) were used for compacted permeability test work.

The results of the column leach test work for the PS and PN composites are summarized in Table 13.11.

Table 13.11. Summary of column leach tests.

KCA Sample	KCA Test No	Sample	Crush Size (mm)	Calc Head Au (g/t)	Extraction Au (%)	Calc Tail (p80 mm)	Days of Leach	Calc Cu Extraction (%)	Cons NaCN, (kg/t)	Lime, (kg/t)	Cement, (kg/t)
80501 B	80543	PS1	9.5	0.614	81	6.2	133	30	1.74	1.01	0.00
80511 C	80576	PS1	HPGR	0.609	81	7.6	132	27	2.09	0.00	4.03
80502 B	80546	PS2	9.5	0.798	84	5.9	133	33	2.86	1.52	0.00
80512 C	80579	PS2	HPGR	0.780	86	8.4	132	27	2.38	0.00	4.11
80503 B	80549	PS3	9.5	0.487	74	6.2	133	11	1.96	0.75	0.00
80513 C	80582	PS3	HPGR	0.489	74	8.4	132	11	1.86	1.01	0.00
80504 B	80552	PS4	9.5	1.097	69	6.1	133	37	2.04	0.76	0.00
80514 C	80585	PS4	HPGR	1.058	75	8.5	132	30	1.55	1.02	0.00
80505 B	80555	PS5	9.5	0.685	56	6.5	142	16	2.26	1.00	0.00
80515 C	80588	PS5	HPGR	0.584	57	9.0	142	22	2.47	0.00	4.28
80506 B	80558	PN1	9.5	1.181	62	6.4	142	18	2.26	0.75	0.00
80516 C	81201	PN1	HPGR	1.172	63	8.6	142	25	2.30	0.95	0.00
80507 B	80562	PN2	9.5	1.343	68	6.2	143	31	2.90	0.76	0.00
80517 C	81204	PN2	HPGR	1.355	68	8.3	142	23	2.90	1.03	0.00
80508 B	80565	PN3	9.5	0.573	56	6.2	143	14	2.00	0.51	0.00
80518 C	81207	PN3	HPGR	0.460	55	9.2	142	23	2.11	0.00	3.95
80509 B	80568	PN4	9.5	0.968	60	6.4	132	12	2.40	0.76	0.00
80519 C	81210	PN4	HPGR	0.901	67	8.2	133	9	2.08	0.00	4.08
80510 B	80571	PN5	9.5	1.148	43	6.2	132	25	2.36	0.51	0.00
80520 C	81213	PN5	HPGR	1.150	50	9.2	133	19	1.88	0.77	0.00

KCA Sample	KCA Test No	Sample	Crush Size (mm)	Calc Head Au (g/t)	Extraction Au (%)	Calc Tail (p80 mm)	Days of Leach	Calc Cu Extraction (%)	Cons NaCN, (kg/t)	Lime, (kg/t)	Cement, (kg/t)
Avg PS		PS	9.5	0.736	73	6.2	135	25	2.17	1.01	0.00
Avg PN		PN	9.5	1.043	58	6.3	138	20	2.38	0.66	0.00
Avg o'all		O'all	9.5	0.889	65	6.2	137	23	2.28	0.83	0.00
Avg PS		PS	HPGR	0.704	75	8.4	134	23	2.07	0.41	2.48
Avg PN		PN	HPGR	1.008	61	8.7	138	20	2.25	0.55	1.61
Avg o'all		O'all	HPGR	0.856	68	8.5	136	22	2.16	0.48	2.05

For the column leach tests completed on conventionally-crushed material, gold extractions ranged from 43 to 84% based on calculated heads which ranged from 0.487 to 1.343 g/t. The sodium cyanide consumptions ranged from 1.74 to 2.90 kg/t. The material used in leaching was blended with 0.51 to 1.52 kg/t hydrated lime.

For the column leach tests completed on HPGR-crushed material, gold extractions ranged from 50 to 86% based on calculated heads which ranged from 0.460 to 1.355 g/t. The sodium cyanide consumptions ranged from 1.55 to 2.90 kg/t. The material used in leaching was blended with 0.77 to 1.03 kg/t hydrated lime or agglomerated with 3.95 to 4.28 kg/t cement.

It should be noted that the average extraction of the conventionally-crushed material was 65%, while the average extraction of the HPGR-crushed material was 68% (an average increase of 3%).

The test work results show a relatively narrow range of copper solubility values across the wide range of copper grades tested, with no correlation identified. However, a negative correlation does exist between copper content and gold recovery.

13.2.4.3 Bottle Roll Testing

Bottle roll leach testing was conducted on portions of material from each of the PS, PN and variability samples. A portion of head material was reduced to sizes of 80% passing 1.70 mm and 80% passing 0.150 mm for leach testing. The results of the bottle roll leach test work are summarized in Table 13.12.

Table 13.12. Summary of bottle roll leach tests.

Sample	Target p80 p100 (mm)	Calc Head (g/t Au)	Au Extraction (%)	Leach Time (hours)	Cons NaCN (kg/t)	Addition Lime (kg/t)
PS1	1.70	0.584	75	144	0.24	1.00
PS1	0.15	0.613	95	96	0.21	1.25
PS2	1.70	0.749	75	144	0.56	1.50
PS2	0.15	0.799	94	96	0.79	2.00
PS3	1.70	0.453	64	144	0.17	0.75
PS3	0.15	0.476	92	96	0.21	1.00
PS4	1.70	1.101	54	144	0.21	0.75

Sample	Target p80 p100 (mm)	Calc Head (g/t Au)	Au Extraction (%)	Leach Time (hours)	Cons NaCN (kg/t)	Addition Lime (kg/t)
PS4	0.15	0.954	95	96	0.24	1.00
PS5	1.70	0.643	60	144	0.65	1.00
PS5	0.15	0.590	88	96	3.86	1.75
PN1	1.70	0.988	65	144	0.50	0.75
PN1	0.15	1.113	95	96	1.21	0.75
PN2	1.70	1.246	69	144	1.22	0.75
PN2	0.15	1.525	95	96	2.06	0.75
PN3	1.70	0.517	71	144	0.51	0.50
PN3	0.15	0.432	89	96	0.92	0.75
PN4	1.70	0.848	64	144	0.37	0.75
PN4	0.15	0.795	93	96	0.56	1.00
PN5	1.70	1.083	52	144	0.63	0.50
PN5	0.15	0.996	95	96	2.34	0.50
Avg PS	1.70	0.706	66	144	0.37	1.00
Avg PN	1.70	0.936	64	144	0.65	0.65
Avg o'all	1.70	0.821	65	144	0.51	0.83
Avg PS	0.15	0.687	93	96	1.06	1.40
Avg PN	0.15	0.972	93	96	1.42	0.75
Avg o'all	0.15	0.829	93	96	1.24	1.08

For the bottle roll tests completed on nominal 1.70 mm material, gold extractions ranged from 52 to 75% based on calculated heads which ranged from 0.453 to 1.246 g/t. The sodium cyanide consumptions ranged from 0.17 to 1.22 kg/t. The material used in leaching was blended with 0.50 to 1.50 kg/t hydrated lime.

For the bottle roll tests completed on pulverized material, gold extractions ranged from 88 to 95% based on calculated heads which ranged from 0.432 to 1.525 g/t. The sodium cyanide consumptions ranged from 0.21 to 3.86 kg/t. The material used in leaching was blended with 0.50 to 2.00 kg/t hydrated lime. Results are summarized in Table 13.13.

Table 13.13. Summary of bottle roll leach tests.

Sample	Target p80 p100 (mm)	Calc Head Au (g/t)	Au Extraction (%)	Leach Time (hours)	Cons NaCN (kg/t)	Addition Lime (kg/t)
VS1	1.70	0.984	49	144	0.20	0.50
VS1	0.15	1.062	93	96	0.23	0.50
VS2	1.70	1.808	62	144	0.25	1.25
VS2	0.15	2.195	92	96	0.31	1.25
VS3	1.70	0.344	62	144	0.23	1.25
VS3	0.15	0.349	92	96	0.35	1.50
VS4	1.70	0.398	63	144	0.26	0.75
VS4	0.15	0.445	95	96	0.42	0.75
VS5	1.70	1.248	45	144	2.05	3.50

Sample	Target p80 p100 (mm)	Calc Head Au (g/t)	Au Extraction (%)	Leach Time (hours)	Cons NaCN (kg/t)	Addition Lime (kg/t)
VS5	0.15	1.187	90	96	6.70	4.00
VS6	1.70	0.775	72	144	0.33	1.50
VS6	0.15	0.852	96	96	0.41	1.75
VS7	1.70	0.657	65	144	0.58	1.00
VS7	0.15	0.518	94	96	0.61	1.25
VN1	1.70	1.637	71	144	0.49	1.00
VN1	0.15	1.745	99	96	0.85	1.25
VN2	1.70	1.349	38	144	0.38	0.50
VN2	0.15	1.043	91	96	0.83	0.50
VN3	1.70	0.396	58	144	0.30	0.50
VN3	0.15	0.408	86	96	0.39	0.50
VN4	1.70	0.854	58	144	0.31	0.50
VN4	0.15	0.747	84	96	0.68	0.50
Avg VS	1.70	0.888	60	144	0.56	1.39
Avg VN	1.70	1.059	56	144	0.37	0.63
Avg O'all	1.70	0.950	58	144	0.49	1.11
Avg VS	0.15	0.944	93	96	1.29	1.57
Avg VN	0.15	0.986	90	96	0.69	0.69
Avg O'all	0.15	0.959	92	96	1.07	1.25

For the bottle roll tests completed on nominal 1.70 mm material, gold extractions ranged from 38 to 72% based on calculated heads which ranged from 0.344 to 1.808 g/t. The sodium cyanide consumptions ranged from 0.20 to 2.05 kg/t. The material used in leaching was blended with 0.50 to 3.50 kg/t hydrated lime.

For the bottle roll tests completed on pulverized material, gold extractions ranged from 84 to 99% based on calculated heads which ranged from 0.349 to 2.195 g/t (refer to Table 13.13). The sodium cyanide consumptions ranged from 0.23 to 6.70 kg/t. The material used in leaching was blended with 0.50 to 4.00 kg/t hydrated lime.

13.3 Metallurgical Variability

A significant amount of metallurgical test work has been completed to date on both composite and variability samples, taken from around the deposit. The samples were mainly drill core, and some assay rejects. The sample grades are similar to the MRE grades and cover a wide range of oxidation states and other variables.

13.4 Deleterious Elements

A wide range of analyses were carried out on the composites. Results for the PS and PN composites are given in the following sub-sections. The main deleterious elements are copper and mercury.

13.4.1 Copper

As referenced in the KCA (2018) summary, copper analyses were performed on the tailings material from each column carried out on the PS and PN composites. The tail values were compared with the copper head analyses in order to calculate an approximate copper extraction. Copper extractions ranged from 9 to 37%.

Copper in solution generally forms a strong complex with cyanide, increasing the cyanide consumption and potentially interfering with gold extraction. Copper cyanide can be removed from solution using the SART process (sulfidization, acidification, recycling and thickening). It produces a saleable copper concentrate product and recycles the cyanide.

13.4.2 Mercury

Mercury contents in the composites are low at <0.03 g/t. Although mercury loadings on carbon were carried out, additional work would need to be done to generate a reliable mercury balance. In general, the low levels of mercury do not represent a risk to processing.

13.4.3 Other

In addition to total carbon and sulfur analyses, speciation for organic and inorganic carbon and speciation for sulfide and sulfate sulfur were conducted. The analyses were conducted to identify potential impacts on gold recovery and reagent consumption from preg-robbing material and sulfur minerals. The results of the carbon and sulfur analyses are presented in Table 13.14. Table 13.15 shows mercury and copper.

Semi-quantitative analyses were conducted by means of an iCAP optical emission spectroscopy (OES) for a series of individual elements and whole rock constituents (lithium metaborate fusion/iCAP). The results of the multi-element analyses are presented in Table 13.16.

Table 13.14. Head analyses – carbon and sulfur.

Description	Total Carbon (%)	Organic Carbon (%)	Inorganic Carbon (%)	Total Sulfur (%)	Sulfide Sulfur (%)	Sulfate Sulfur (%)
PS1	0.13	0.07	0.06	0.02	0.01	0.01
PS2	0.19	0.07	0.12	0.24	0.08	0.16
PS3	0.09	0.03	0.06	0.01	0.01	0.00
PS4	0.25	0.05	0.20	0.02	0.01	0.01
PS5	0.18	0.13	0.05	2.74	2.39	0.35
PN1	0.57	0.07	0.51	0.27	0.12	0.16
PN2	0.86	0.08	0.78	0.26	0.11	0.16
PN3	0.44	0.07	0.37	0.13	0.02	0.11
PN4	0.41	0.04	0.36	0.04	0.01	0.04
PN5	0.36	0.07	0.29	0.25	0.13	0.13

Table 13.15. Head analyses – mercury and copper.

Description	Total Hg (%)	Total Cu (ppm)	Cyanide Soluble Cu (ppm)	Cyanide Soluble Cu (%)
PS1	0.02	91	29	32
PS2	0.03	933	203	22
PS3	0.03	254	26	10
PS4	0.03	76	15	20
PS5	0.03	1,020	253	25
PN1	0.02	811	304	37
PN2	0.03	1,470	652	4
PN3	0.03	500	246	49
PN4	0.03	766	181	24
PN5	0.02	1,260	659	52

Table 13.16. Head analyses – Multi-element analyses.

Constituent	Units	PS1	PS2	PS3	PS4	PS5	PN1	PN2	PN3	PN4	PN5
Al	%	3.36	2.86	3.25	2.76	2.83	2.20	1.95	1.87	3.61	1.66
As	ppm	11	45	3	4	14	64	35	24	30	27
Ba	ppm	1770	1420	2110	2080	436	1340	718	652	1270	683
Bi	ppm	<2	<2	<2	<2	<2	3	<2	<2	<2	4
C(total)	%	0.13	0.19	0.09	0.25	0.18	0.57	0.86	0.44	0.41	0.36
C(organic)	%	0.07	0.07	0.03	0.05	0.13	0.07	0.08	0.07	0.04	0.07
C(inorganic)	%	0.06	0.12	0.06	0.2	0.05	0.51	0.78	0.37	0.36	0.29
Ca	%	4.88	2.47	2.93	5.37	3.96	7.25	8.16	5.44	5.56	6.18
Cd	ppm	5	11	3	5	10	12	9	7	7	7
Co	ppm	8	19	7	8	19	12	12	8	8	9
Cr	ppm	209	222	152	173	190	170	202	202	140	193
Cu(total)	ppm	91	933	254	76	1020	811	1470	500	766	1260
Cu(CN sol)	ppm	29	203	26	15	253	304	652	246	181	659
Fe	%	2.87	4.05	1.53	2.8	7.14	4.52	4.93	3.65	2.66	4.23
Hg	ppm	0.02	0.03	0.03	0.03	0.03	0.02	0.03	0.03	0.03	0.02
K	%	1.59	0.92	1.8	1.52	0.54	0.81	0.38	0.41	0.69	0.67
Mg	%	1.51	0.75	1.47	1.63	1.21	1.90	1.19	0.95	0.86	1.47
Mn	ppm	696	617	417	896	543	1610	1420	1080	814	1310
Mo	ppm	<1	10	2	3	40	20	20	7	9	28
Na	%	0.61	0.34	0.46	0.35	0.45	0.25	0.13	0.08	0.33	0.07
Ni	ppm	45	59	45	43	56	42	49	42	39	30
Pb	ppm	19	122	11	19	<10	219	60	62	39	83
S(total)	%	0.02	0.24	0.01	0.02	2.74	0.27	0.26	0.13	0.04	0.25
S(sulfide)	%	0.01	0.06	0.01	0.01	2.39	0.12	0.11	0.02	0.01	0.13

Constituent	Units	PS1	PS2	PS3	PS4	PS5	PN1	PN2	PN3	PN4	PN5
S(sulfate)	%	0.01	0.16	0	0.01	0.35	0.16	0.16	0.11	0.04	0.13
Sb	ppm	<2	3	<2	<2	<2	2	<2	2	<2	<2
Se	ppm	<5	7.4	<5	<5	24.3	5.5	5.2	<5	<5	5.7
Sr	ppm	213	201	206	177	127	151	147	105	290	85
Te	ppm	6	8	<5	<5	10	<5	6	<5	<5	5
Ti	%	0.24	0.19	0.23	0.2	0.18	0.15	0.16	0.14	0.21	0.15
V	ppm	56	101	57	56	64	46	50	64	78	29
W	ppm	<10	17	<10	12	23	16	12	8	5	8
Zn	ppm	82	307	86	117	75	442	202	145	154	119

13.5 Conclusions

Although gravity concentration, agitation leaching and flotation have been considered during the test programs, the preliminary process selection is fine crush and heap leach. The discussion summarizes process conclusions and proposes preliminary process design criteria using data from the column testing. Sampling location and testing in the 2018 KCA test program is the most thorough and therefore results from that program have been prioritized.

Crush Size: Column test data clearly shows that a fine crush of the order of ¼ in is required to maximize recoveries. A preliminary design of the crushing plant is for a conventional plant, that is three-stage, open-circuit to produce a product with a particle size of 100% passing 9.5 mm, or 80% passing 6 mm. Recoveries using HPGR were 3% higher than conventional crush. While this is significant, it represents only a small incremental increase in revenue at the expense of increased process complexity and cost.

Recoveries: Table 13.17 contains averages from the three test programs that includes copper analyses and pit location. Only data from column tests on finely-crushed material are included in the averages. The results demonstrate the South zone consistently yields better recoveries than the North zone. This is interpreted to be related to the lower copper grade in the South zone. Regarding the effect of the oxidation state, samples designated as “oxide” and “transition” by the geologists usually yield similar recoveries, while recoveries from “sulfide” tend to be lower. This is partially explained by the higher copper grade of the sulfide samples and due to weathering that positively affects gold liberation. Using only the P80 <6.4 mm conventional crush data from the KCA 2018 program on a redox basis, the average recoveries achieved were 77% oxide, 62% transition and 50% sulfide.

Table 13.17. Program averages – gold recovery and copper grade.

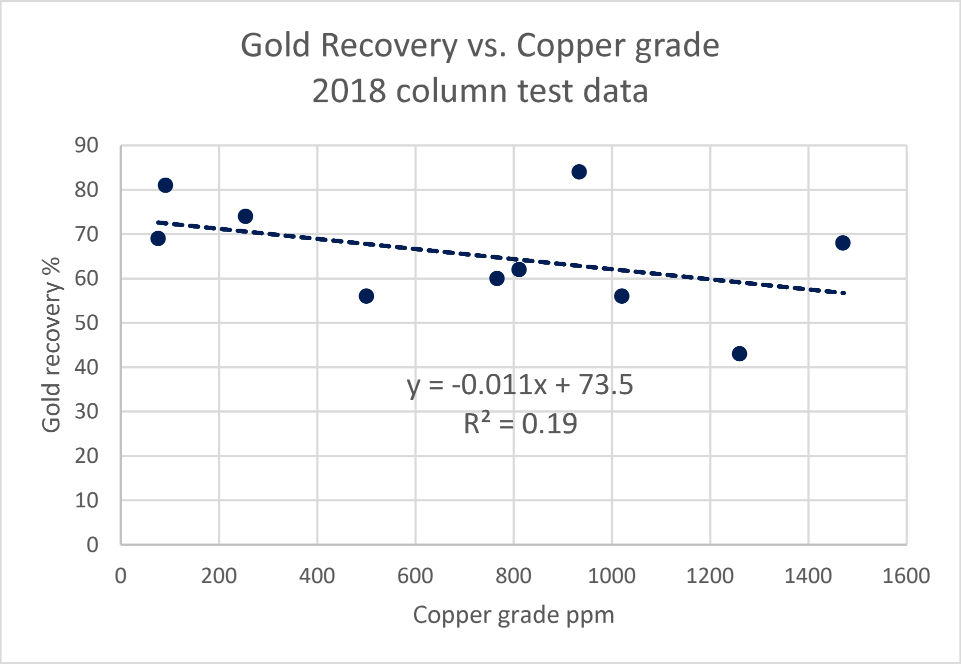
Test Program	Redox or Location	Column Test (#)	Crush Size (mm)	Au Rec (%)	Cu (ppm)
KCA 2005	North	5	P100<6.4	60	-
	South	4	P100<6.4	69	-
	Oxide	2	P100<6.4	68	-
	Transition	4	P100<6.4	68	-
	Sulfide	3	P100<6.4	57	-
MLI 2009	North	6	P80<9.5	47	617
	South	6	P80<9.5	53	552
	Oxide	12	P80<9.5	50	584
	Transition	11	P80<9.5	52	1542
	Sulfide	3	P80<9.5	34	1190
KCA 2018 (conv. crush)	North	5	P80<6.4	58	961
	South	5	P80<6.4	73	475
	Oxide	4	P80<6.4	77	339
	Transition	4	P80<6.4	62	887
	Sulfide	2	P80<6.4	50	1140
Overall (average of above)	North	-	-	55	789
	South	-	-	65	514
	Oxide	-	-	65	462
	Transition	-	-	61	1215
	Sulfide	-	-	47	1165

Using only the 9.5 mm conventional crush data from the KCA 2018 program, the average recoveries achieved were 58% from the North zone and 73% from the South zone. This difference is likely explained by the average copper grades of the North and South zones, which are 961 and 475 ppm respectively.

When the 2018 test data from the two zones are analyzed together, there is no clear relationship between the gold recovery and sulphide content. However, this is because all samples, except for one, are fairly low-grade sulphur. Similarly, there is no clear relationship between gold recovery and gold grade. There is, however, a relationship between gold recovery and copper grade which holds for all four test programs.

It is therefore suggested that the copper grade be used to predict gold recoveries, regardless of which zone they came from, gold grade or oxidation state. The relationship is shown in Figure 13.2, which uses only the 2018 data. Similar trendlines exists using data from all four programs taken together, or individually. It should be noted that the "R^2" value is fairly low indicating that some "scatter" can be expected. It should also be noted that samples designated as "sulfide" throughout the four programs tend to also have a high copper grade.

Figure 13.2. Gold recovery vs. copper grade (2018 data).



Samples with very low copper can be expected to yield gold recoveries over 70%, those with mid-range copper approximately 60% and high copper samples 45 to 50%. No deduction from the laboratory testing results has been made as recoveries were still increasing, albeit at a slow rate. Gold recoveries from individual samples or zones can be predicted using the simple formula (Figure 13.2):

Gold recovery % = 73.5 - 0.011 x copper content (ppm)

Heap Construction: As is typically the case for fine material, the heap should be constructed using grasshopper conveyors and a stacker, rather than trucks.

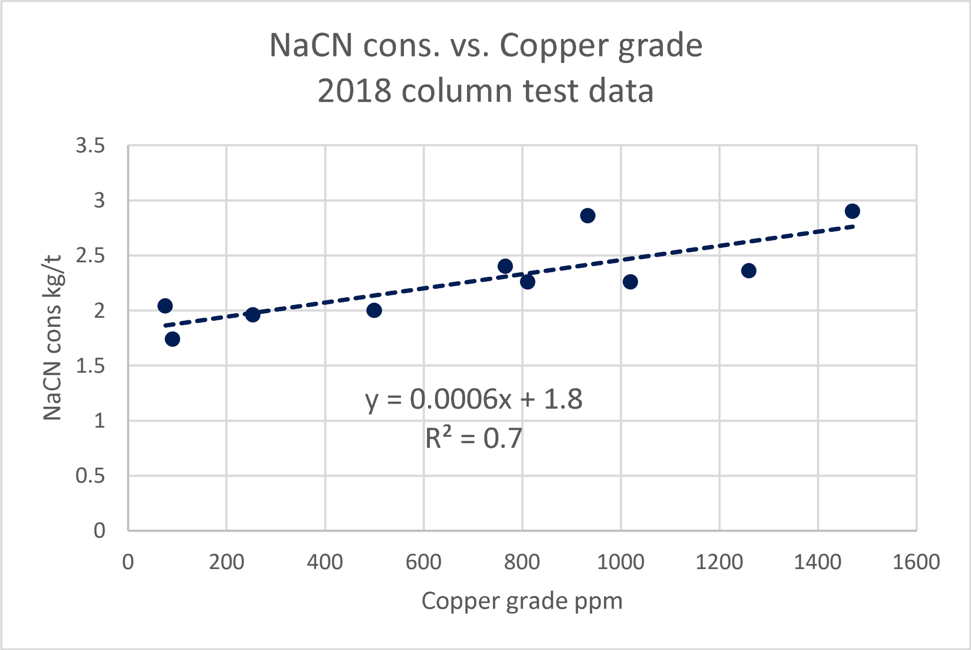
Agglomeration: It is concluded from the preliminary KCA agglomeration tests that 4 kg/t cement is required, and that a maximum heap height of 100 m can be used for preliminary design. It also appears that a heap constructed using HPGR material may be less stable than a heap constructed using conventionally-crushed material.

SART Process: It is recommended to use the SART process (sulfidization, acidification, recycling and thickening), to minimize the negative effects on cyanide consumption and gold recovery by dissolution of copper by cyanide. It produces a saleable copper concentrate product and recycles the cyanide.

Cycle Times: Laboratory cycle times are of the order of 130 days. There are a number of methods used to infer “field days” from “column days”. In this case the preferred method is that the first 30 column days are increased to 90 field days, the second 30 column days are increased to 60 days and no adjustment is made to the remaining days. Therefore, the “column days” of 130 becomes 220 “field days”. As typically occurs in actual heap leach operations, the final leach times will be much higher as the solution from higher lifts percolates through lower lifts.

Reagent Consumption: Column test cyanide consumptions are fairly high. As would be expected there is a relationship between copper content and cyanide consumption. This is shown in Figure 13.3, which uses the 2018 data only. Similar trendlines exist using data from all four programs taken together, or individually. It should be noted that the "R2" value is fairly high, indicating that it is an accurate indicator.

Figure 13.3. Cyanide consumption vs. Copper grade (2018 data).



Laboratory column tests typically consume approximately three times the cyanide than is experienced in the actual operation. It is suggested that the following simple formulae be used to estimate cyanide consumption.

Lab: NaCN consumption (kg/t) = 1.8 + 0.0006 x copper content (ppm)

Actual: NaCN consumption (kg/t) = 0.6 + 0.0006 x copper content (ppm)

The 1.0 kg/t lime added before leaching was sufficient for maintaining protective alkalinity during leaching. When 4 kg/t cement is added to assist in agglomeration, no lime is required.

13.6 Preliminary Process Design Criteria

To direct the design of a future processing plant, the following preliminary design criteria are recommended (Table 13.18).

Table 13.18. Preliminary design criteria.

Item	Units	Value
Resource tons and grade	tons and g/t	
Throughput	tons/day	46,575
Availability	%	75
Throughput	tons/hour	2,588
Crushing circuit		3-stage conventional
Crush size	100%< mm	9
	80%< mm	6
Cycle time	d	220
Cyanide consumption	kg/t NaCN	See formula
Lime addition	kg/t CaO	1 (if no cement added)
Cement addition	kg/t	4

14 Mineral Resource Estimates

The Mineral Resource Estimate (MRE) was based upon the historical drilling from 1989 to 2012 and drilling conducted by Converse in 2017.

A new resource block model was developed for the Converse Project to take into account geological logging and assay data from drillholes not available at the time of the previous 2012 estimate, to incorporate the latest improvements in the understanding of the geological controls on mineralization, and to add estimates of silver, copper and cyanide-soluble gold grades that may play a role in the evaluation of the Project's technical and economic viability.

Statistical analysis, three-dimensional (3D) modelling and resource estimation was completed by Mr. Srivastava the QP responsible for this section. The kriging workflow for estimation of gold grades in the Converse MRE was completed using the commercial mine planning software Micromine 2020.5. The co-kriging workflow for estimation of silver, copper and for the cyanide-soluble gold recovery was completed using RedDot3D's in-house software developed by the QP.

Converse provided APEX and RedDot3D with a Project drillhole database that consisted of analytical, geological, density, collar survey information and downhole survey information. The provided data was reviewed in detail in 2011 and 2012 (Srivastava *et al.,* 2012) and again in 2019–2020 by the QPs. A review of the 2017 drilling was conducted prior to the start of the current MRE. In the opinion of the QPs, the current Converse drillhole database is considered to be in good condition and suitable to use in ongoing resource estimation studies.

The MRE was estimated using a block model size of 50 (X) by 50 (Y) by 20 ft (Z). RedDot3D estimated the gold grade for each block using indicator kriging (IK) and ordinary kriging (OK) with locally varying anisotropy (LVA) to ensure that local changes in the direction of maximum grade continuity were reproduced in the block model.

Modelling was conducted in the CON3 coordinate system developed by Loyal Olsen (2008). The database consisted of 326 drillholes containing useable downhole data completed between 1989 to 2017, of which 249 drillholes were used in the current resource modelling.

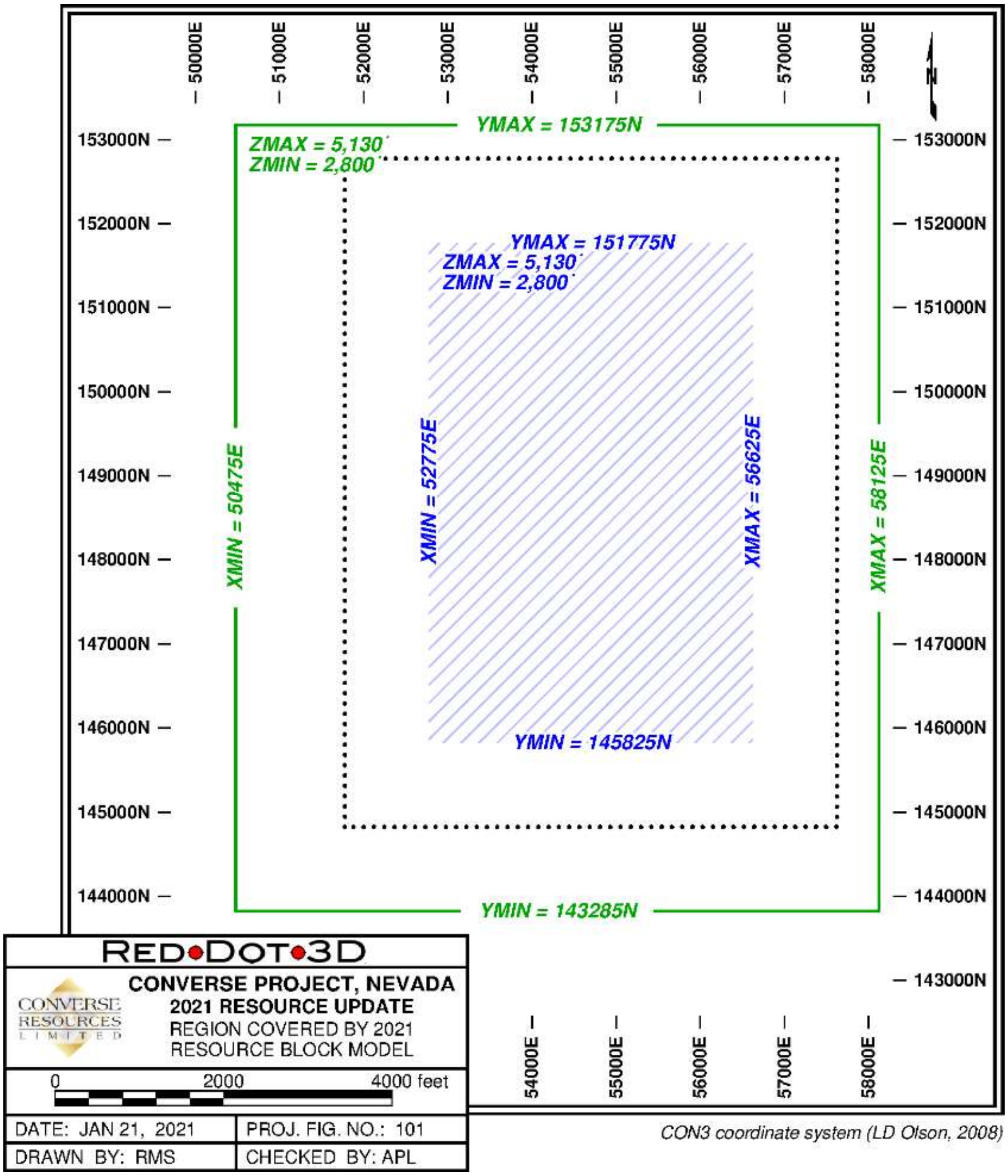
14.1 Region Covered by Current Block Model

Figure 14.1 shows the region covered by the current resource block model. Gold, silver and copper grades were estimated for blocks within the central blue hatched region; outside this region, all block grade estimates were set to zero but the tonnages were reported to facilitate the optimization of pit shells whose crests will extend beyond the central hatched area. The outer green rectangle of the block model area extended far enough to accommodate waste stripping at gold prices much higher than used in the estimate, i.e. up to US$3,000/oz gold.

The geological surfaces that influence the assignment of densities extended past the outer edge of the block model region.

The region within which drillhole data affect grade estimates is shown by the dotted line in Figure 14.1, which extends 1,000 ft beyond the central hatched region. Although the Project database includes holes drilled outside this region, the search parameters used for grade interpolation cause holes beyond the dotted line to have no influence on grade estimates.

Figure 14.1. Region covered by the current resource block model.



In the vertical direction, the block model extended upwards to an elevation of 5,140 ft, one full bench above the highest point on the topography in the areal footprint of the block model. It extended down to an elevation of 2,800 ft; even though deep drillholes do show scattered mineralization below this elevation, the block model extended 200 ft below the bottom of a preliminary pit shell optimized at a gold price intentionally much higher than the peak spot price in 2020 to ensure that the resource block model extended deep enough for the purposes of future mine planning and economic assessment.

14.2 Data

14.2.1 Drillhole Database

14.2.1.1 Drillhole Collars

A total of 249 drillholes in the Converse database were used for resource estimation (Table 14.1); these are the holes located within the dotted line in Figure 14.1. The locations of the drillholes are shown in Figure 14.2.

There have been four drilling technologies used at Converse: reverse circulation drillholes (RC); core drillholes (DD); mud-rotary holes (MR); and holes drilled as reverse circulation holes through the alluvium and completed as core drillholes in the bedrock (RC-DD). Of the samples that have been analysed for gold, roughly half of those samples have been also analysed for silver and copper.

In the North Redline and South Redline areas of the project, the drillhole spacing was approximately 200 ft; elsewhere, the drillholes were approximately 400 ft apart. On the southwestern side of the stock there was also dense drilling in an unnamed region that was also regarded as being generally higher in grade.

The 2017 holes drilled since the 2012 MRE are the seven holes shown in magenta in Figure 14.2; these were core drillholes that were completed primarily for metallurgical studies.

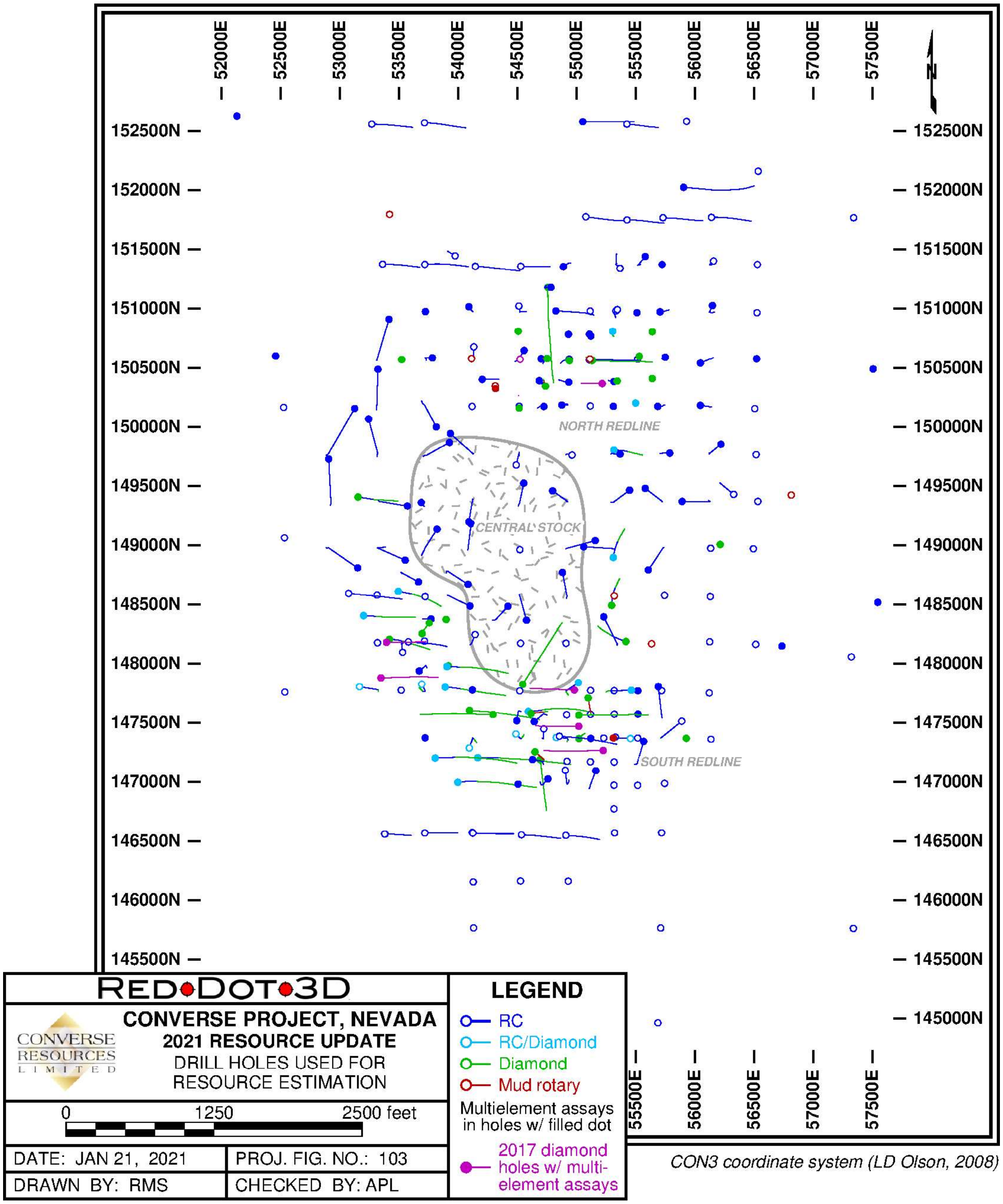
Table 14.1. Summary of drillholes used for the current resource block model.

Type	Number	Length (ft)	Number of Au Assays	Length of Au assays (ft)	Number of Ag Assays	Length of Ag Assays (ft)	Number of Cu Assays	Length of Cu Assays (ft)
RC	180	139,699	18,888	105,309	5,456	27,910	5,362	27,435
DD	39	43,042	7,681	40,242	7,359	38,369	6,440	33,670
MR	10	5.080	249	1,245	22	110	10	50
RC-DD	20	27,302	5,090	25,529	3,514	32,280	3,515	18,808
Total	249	215,123	31,908	172,325	16,351	85,197	15,326	79,963

14.2.1.2 Downhole Surveys

Of the 249 drillholes used for resource estimation, 183 have downhole surveys. A total of 66 have only a collar survey and no downhole survey, and were presumed to exactly follow their collar orientation for their entire length; 62 of these are vertical. Checks of the deviations in the vertical and inclined holes that had downhole surveys confirmed that the uncertainty of the exact sample locations caused by lack of downhole surveys was very minor in vertical holes and less than ±50 ft in the four inclined holes that were not surveyed.

Figure 14.2. Drillholes used for resource estimation.



The potential deviations in the four holes with no downhole surveys that were inclined at -55° to -70° can be quantified using surveyed holes with similar collar orientations. At the depths where these four unsurveyed holes have bedrock samples, the surveyed holes with similar orientations show that if the only available information on the hole's orientation is the collar survey, then there is a 20% chance that the location of a bedrock sample in these four holes might be off by more than 50 ft, i.e. in an adjacent block in the resource block model rather than its proper block. The chance of one of the bedrock samples in these four holes being mislocated by 100 ft (two blocks in the resource block model) is below 0.5%.

With only four holes having location uncertainty due to the lack of downhole surveys, and with a very low chance that any of the 150 bedrock samples from these holes has been mislocated to an adjacent block, the lack of downhole surveys in these few holes has no significant impact on resource estimates. The initial azimuth and dip for those holes as been used for the entire length of the hole.

14.2.1.3 Assays

Gold

The assay database includes gold assays from several analytical methods outlined in Section 11. For any sample interval where more than one type of gold assay was available, the assay database identifies in a separate field the "Au_final" value to be used for resource estimation, following a systematic hierarchy that gives priority to fire assays, to larger aliquots and to gravimetric finishes, when available. Of the 31,908 Au final values in the area where grades were estimated, all but one sample had its final value derived from fire assays. There was a single sample interval for which the aqua regia ICP assay was used.

Greater than 60% of the gold assays are from samples derived from RC holes. Studies of the possibility of downhole contamination in RC holes showed no evidence of the gradual accumulation of gold with depth, or of anomalous spikes in gold grade at rod changes.

Cyanide-Soluble Gold

Cyanide leach gold assays were used to add to the resource block model an estimate of the percentage of gold that is cyanide-soluble. None of the cyanide leach gold assays were used as the Au final value for resource estimation purposes. Approximately 14% of the sample intervals in the assay data base have cyanide-soluble gold assays; none of these were used to estimate gold grades for the purpose of block model gold estimation.

Silver and Copper

Almost all of the Ag and Cu assays came from multielement ICP analytical methods; very few (<0.4%) of the Project's early Ag assays were completed by fire assay. The ICP analyses used either a four-acid or aqua regia digestion. ICP assays done with an aqua regia digestion are slightly more common (approximately 55% of the Ag ICP assays, and 52% of the Cu ICP assays) than those done with a four-acid digestion. A four-acid digestion is preferable for most trace elements because it more completely dissolves most metals into the acidic liquid analyzed by the ICP instrument; the notable exception is gold, for which aqua regia is more effective.

With an aqua regia digestion not getting all of the Ag or Cu into solution, the Ag and Cu ICP assays completed with an aqua regia digestion had a negative bias: they systematically under-represent the true total concentrations of the metal in the original sample. Ag and Cu ICP assays completed with an aqua regia digestion were adjusted upwards based on analysis of the silver-gold and copper-gold trends for the two types of ICP assays.

The majority of the silver and copper assays were from core drillholes or from the core tail of holes drilled as RC through the alluvium. As seen in Figure 14.2, where open circles are used to mark collars of holes with no multielement assays, there are areas where there were no multielement assays for several thousand feet, particularly toward the south.

Assay Values Below Detection Limit

For Au, Ag and Cu, assays below the detection limit were set to half the detection limit. For cyanide-soluble gold, if the assay was reported as below detection limit it was set to half the detection limit if the total gold value was at or above the detection limit; otherwise (i.e. when the total gold value was also below detection limit), the CN-Au assay was treated as missing.

Missing Assay Values

Of the drillhole sample intervals that affect resource modeling in the bedrock, 69 are missing a gold assay, which is less than 0.2%. For these intervals that were missing assays, no grade values were assigned; they were left as unknown or effectively not sampled for grade interpolation purposes.

14.2.1.4 Geological Logging

The geological logging database includes information on the lithology, the oxidation state and on the minerals observed. The lithology logging was used to identify the drillhole depths that serve as control points for modeling the undulating surface that marks the base of the Quaternary alluvium. The logging was used for the development of the 3D model of the geometry of the main central stock, together with its small offshoots and satellites, and for the assignment of prograde metamorphic indicators that were used in resource estimation.

The oxidation state was used to develop the model of the three bedrock oxidation zones: oxide, transition and sulphide. Based upon the logged oxidation state, wireframe surfaces or polygons were built in order to flag the block model with the oxidation state.

Observations on mineralogy were used to supplement the logging database fields in order to assess and document the degree of contact metamorphism, an important control on gold mineralization. The logging of the metamorphic mineral assemblage was completed in order to characterise prograde and retrograde metamorphism. Although retrograde metamorphism does influence gold mineralization, most of the samples have no information on retrograde metamorphism, leaving the logging database with only sparse information on the degree of retrograde metamorphism. Since 64% of the samples have information on the degree of prograde metamorphism, the prograde metamorphic logging was usable as information that improves the accuracy of resource estimation.

For the 36% of the bedrock sample intervals that did not have information on prograde metamorphism, those intervals were assigned to one of the prograde populations based on their location relative to the central stock.

14.2.1.5 Adequacy of Drillhole Database

The drillhole database for the Converse Project includes data from holes that span more than three decades since the Redline deposits were first drilled in 1989. Even though the Project has passed through several owners, the documentation and paper trail remain intact and are considered in excellent shape. There are very few pieces of information (well below 1%) for which the only supporting documentation was hand-written tabulations of laboratory analyses. Geological logs and assay certificates were available for most of the drillholes.

The digital database has been well designed and well maintained, with information that facilitates tracing assay information back to original documents, the majority of which were assay certificates issued by accredited laboratories. The QPs are of the opinion that the Converse drillhole database is suitable for the purposes of resource estimation.

14.2.2 Topography

The Converse Project topography came from a high-resolution digital elevation model with two-foot contour intervals that was developed in 2008 by Aerographics from aerial photography and a set of 350 ground control points surveyed by Loyal Olson in the CON3 local coordinate system established by him in 2008. Figure 14.3 shows the topography over the resource block model area, averaged into 25 x 25 ft grid cells.

The high-precision CON3 local coordinate system (Olson, 2008) continues to be used for the Project, with new hole collars being surveyed in Universal Transverse Mercator (UTM) coordinates that are converted to CON3 coordinates using formulas provided by Olson.

The QP is of the opinion that the topography, survey and hole-orientation data is highly reliable, and suitable for the purposes of resource estimation.

14.2.3 Geological Wireframes

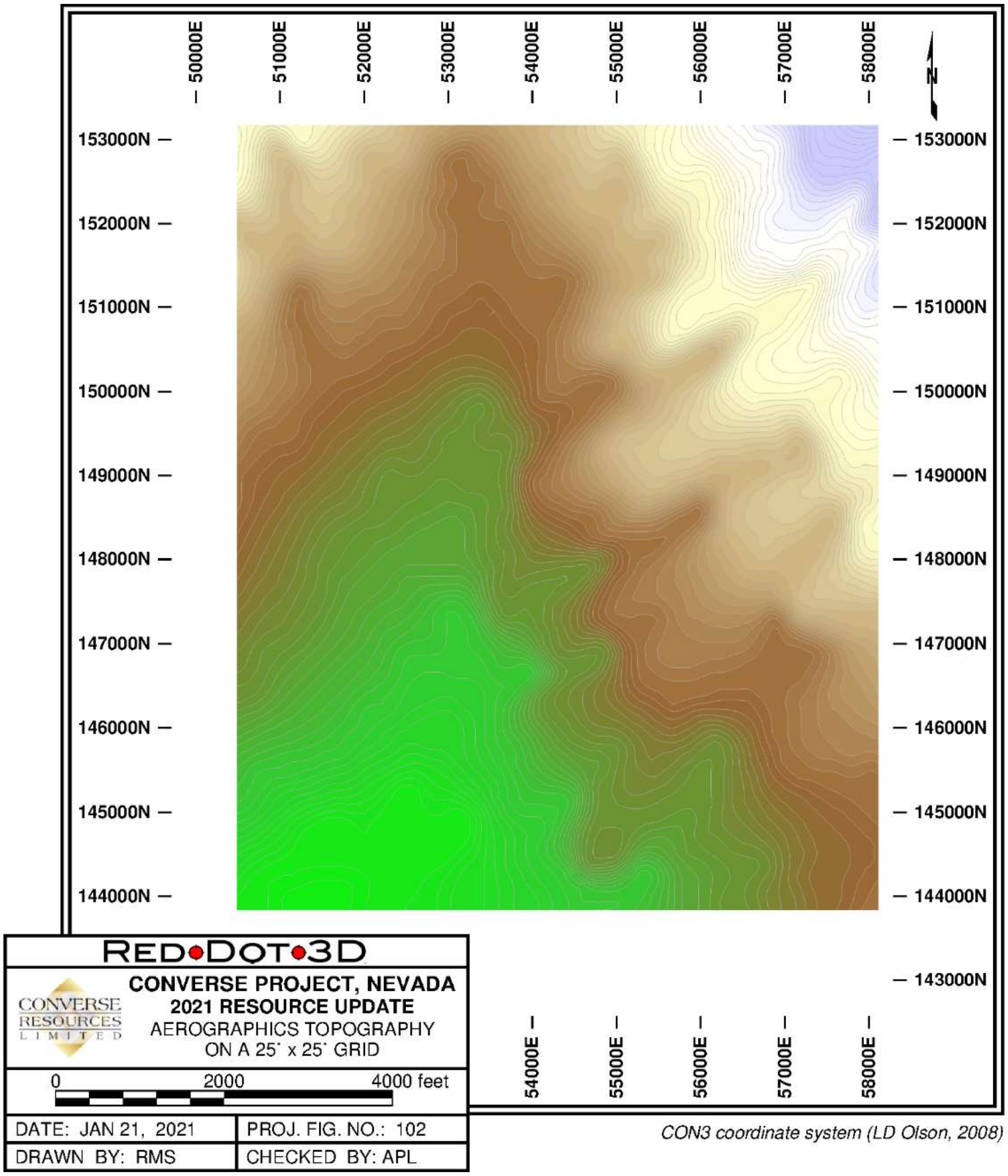
DXF files containing triangulated surfaces and solids were provided by CRL for the following features:

- The base of the Quaternary alluvium (Qal) gravels, which lie between the top of bedrock and topography;
- The oxidized bedrock below the Qal;
- The transitional oxide-sulfide zone beneath the oxidized bedrock;
- The sulfide (unoxidized) zone beneath the transitional zone;
- The central intrusive stock.

Each of the geological surfaces and solids was developed in Leapfrog Geo 6.0 using the information from drillhole logs to identify control points for modeling a smooth surface in 3D.

The QP verified that the various geological wireframes were updated to include the most recent drillholes, that they honored logging information from drillholes, and that they struck a reasonable balance between complexity that captures important local details and smoothness that removes noise. The results of statistical analysis that makes use of these geological wireframes confirmed that they were well constructed and were suitable for the purposes of resource estimation.

Figure 14.3. Aerographics topography on a 25 x 25 ft grid.



14.3 Geological Controls

14.3.1 The Role of the Base of Quaternary Alluvium

The first of the geological controls, the base of Quaternary alluvium (Qal), influences grade estimation, classification and density. It serves as a hard boundary between the bedrock domain and the alluvium domain. Although there is gold mineralization in the alluvial gravels, the grades are much lower than in the bedrock, and the direction of maximum continuity is different. Above the Qal boundary, where the gold mineralization is patchy and thin, blocks were classified as Inferred. The density of the poorly-consolidated Quaternary gravels is lower than that of bedrock, so the base of Qal controls the assignment of block densities above it. The base of Qal also provides information on the local strike and dip of the gently undulating bedrock surface on which the alluvial gravels accumulated; this orientation information was used to guide the directions of maximum continuity for gold grade estimates in the alluvium.

14.3.2 The Role of the Oxidation Zones

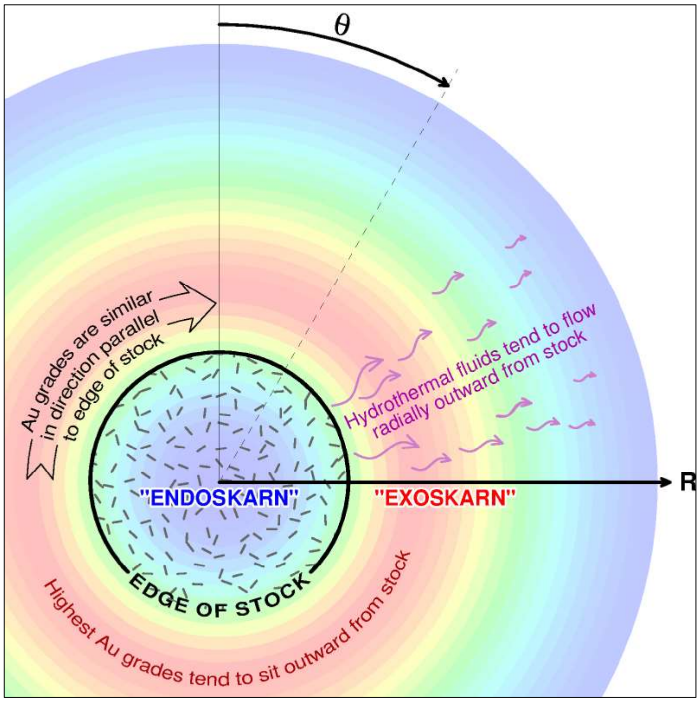
The oxide, transition and sulfide surfaces influence only the density, which is higher, on average, in the transitional and fresh bedrock at depth, where it’s the same, and lower in the oxidized bedrock just below the alluvium.

14.3.3 The Role of the Central Stock

The central stock influenced the grade estimation. This intrusion served as the heat source for the metamorphism of the surrounding sedimentary rocks. Mineralized hydrothermal fluids would generally have moved away from the stock, creating a deposit in which there is a clear trend in gold grade in the radial direction, outward from the stock. Gold grades are highest slightly outward from the central stock, where gold would have precipitated quickly from fluids as their temperature cooled. As the fluids move further away from the central stock, past the peak of gold mineralization, the grades generally drop since it is difficult to keep gold in solution as the pressure and temperature of the fluids drop.

The wireframe of the stock was used to define a radial coordinate system in which the radius I was the distance away from the stock, with negative values for locations inside the stock (the “endoskarn”) and positive values for locations outside the stock (the “exoskarn”). The angle (θ) was the direction from the center of the stock. As shown schematically in Figure 14.4, the gold grades rise and fall in the radial direction, and are continuous in the circumferential direction.

Figure 14.4. Schematic map view showing how the central stock affects gold mineralization.



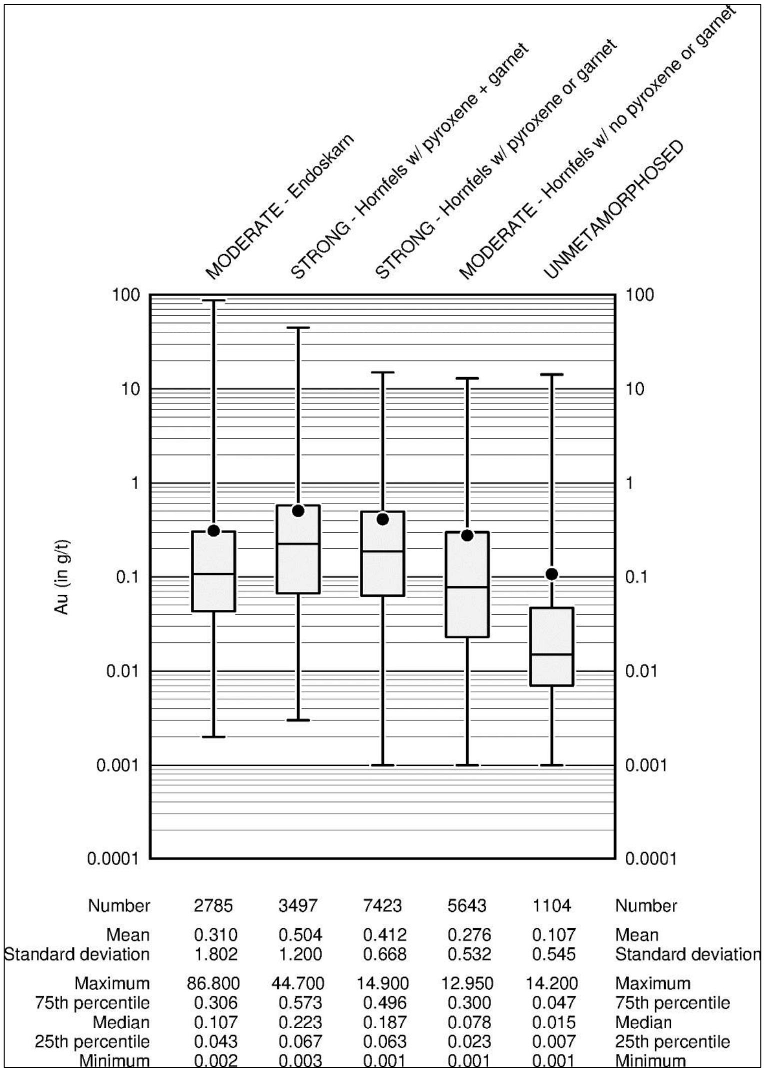
14.4 Exploratory Data Analysis

14.4.1 Effect of Prograde Metamorphism on Gold Grade

Figure 14.5 shows side-by-side boxplots of the gold grades for bedrock samples, separated into three major inter-mixed groupings of prograde metamorphic codes:

- Unmetamorphosed: samples whose mineral assemblages did not indicate any prograde metamorphism. These generally lay in the outer blue ring in the schematic in Figure 14.4, far away from the central stock, but could occur anywhere in the deposit. Hydrothermal fluids did not percolate uniformly; even close to the stock, rocks may show no evidence of metamorphism;
- Moderate metamorphism: samples whose mineral assemblages have been logged as indicative of endoskarn or weak hornfels metamorphism, with no specific observations of garnet or pyroxene. These generally lay in the light-blue to yellow bands in the schematic in Figure 14.4:
  - Inside the central stock (the endoskarn);
  - Outward and close to its edge;
  - Further away, past the peak of gold mineralization, about 500 to 1,200 ft away from the stock (the weak and undifferentiated hornfels of the exoskarn);
- Strong metamorphism: samples whose mineral assemblages have been logged as indicative of strong hornfels metamorphism, with garnet and/or pyroxene having been specifically observed. These generally lay in the orange-red bands in the schematic in Figure 14.4, about 400 to 800 ft from the central stock, but can occur elsewhere.

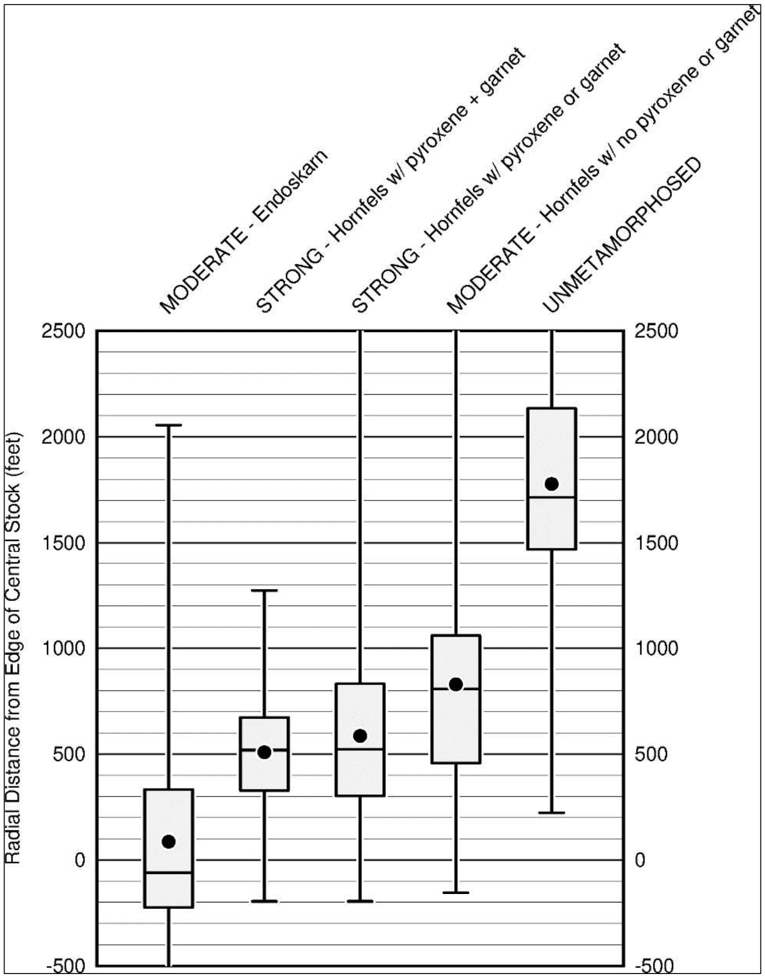
Figure 14.5. Side-by-side boxplots of gold grades separated into major prograde metamorphic groups



The locations of the samples in each of these groups are shown by the side-by-side boxplots in Figure 14.6. The left-to-right ordering of the boxplots in Figure 14.5 and Figure 14.6 reflects the general position relative to the central stock, with the endoskarn boxplot generally lying inside or closest to the stock, followed in an outward direction by the exoskarn groups: strong hornfels, weak hornfels and unmetamorphosed.

The rise and fall of gold grades from left to right across Figure 14.5 is consistent with the geological model shown schematically in Figure 14.4, and demonstrates the reliability of the prograde metamorphic coding in the logging data base. There is variability within every group, but the unmetamorphosed samples have an average grade that is ⅓ that of the moderately metamorphosed samples, and 1/5 that of the strongly metamorphosed samples.

Figure 14.6. Side-by-side boxplots of distance from the central stock for the major metamorphic groups, with zero distance on the y-axis marking the edge of the stock, negative values corresponding to samples in the endoskarn inside and near the stock, and positive.

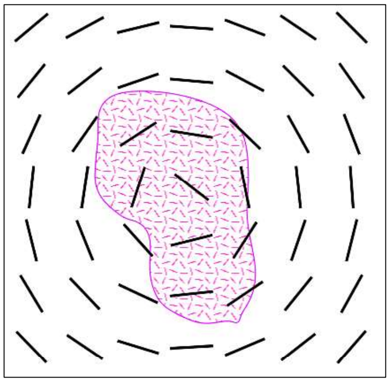


Grade interpolation for the resource block model took into account the effect of prograde metamorphism by treating the three major inter-mixed groups; strong, moderate and unmetamorphosed, as separate populations. Because the samples in the populations do not fall into discrete regions that can be separated into an interlocking set of domains, the resource estimation methodology used a "mixture model" that accommodated the possibility that any resource block could have contributions from any population. Within each block, the proportion contributed by each population was estimated by IK, and the gold grade of each population was estimated by OK using only the nearby samples from that population. A single estimate of block grade was then calculated using a proportion-weighted average of the gold grade estimates for each of the three populations. In order to ensure that a gold grade can be estimated in any block that has a non-zero estimate of a particular population, the same search parameters were used for both the IK and for the gold grade estimates. In order to avoid proportion estimates that sum to one, the same variogram model (Table 14.3) was used for all three populations, for both the gold grade and for the metamorphic indicators.

14.4.2 Influence of the Central Stock on Direction of Maximum Continuity

Figure 14.5 and Figure 14.6 are consistent with the geological model presented schematically in Figure 14.4, showing grades that rise and fall in the radially-outward direction. The same geological model predicts that there will be no single compass direction that serves as the dominant direction of maximum continuity. Locally, there is a clear direction of maximum continuity: gold grades will be most similar in the direction parallel to the stock at equal distances from the stock contact; but this direction swings all the way around the compass, as shown in Figure 14.7.

Figure 14.7. Local directions of maximum continuity of gold grade calculated from the shape of the edge of the central intrusive stock. The black lines are parallel to the edge of the stock and run in the direction referred to as "azimuth" in the text.



As predicted by the geological model, experimental variograms of gold grades calculated in different compass directions show no clear anisotropy; but when calculated in a radius-and-azimuth coordinate system, the anisotropy becomes clear, with longer ranges of correlation in the directions parallel to the stock, i.e. that wrap around the stock in a cylindrical manner. In map view, these directions run parallel to the stock and, in cross-sectional view, they run parallel to the approximately vertical edge of the stock. The shortest ranges of correlation run perpendicular to the stock in map view.

Figure 14.8 shows the experimental variogram for the raw sample gold grades calculated in the "radius" direction that runs perpendicular to the black lines in Figure 14.7, radially outward from the stock. Figure 14.9 shows the experimental variogram calculated in the "azimuth" direction that runs parallel to the black lines, wrapping around the stock. Gold grades have a range of correlation of approximately 1,000 ft in the direction locally parallel to the nearest edge of the stock, and of only 300 ft in the perpendicular direction, radially outward from the stock.

Figure 14.8. Experimental pairwise relative variogram of gold grades, calculated in the "radius" direction, radially outward.

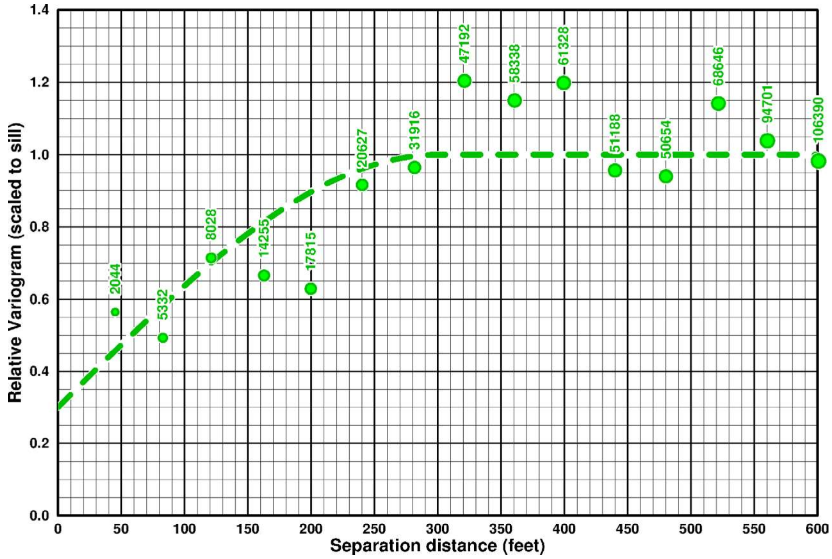


Figure 14.9. Experimental pairwise relative variogram of gold grades calculated in the "azimuth" direction, locally parallel to the edge of the stock.

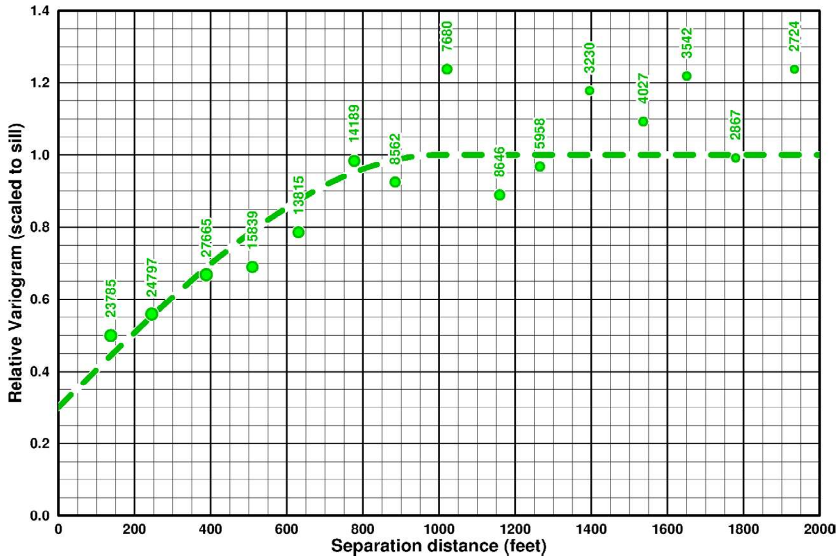
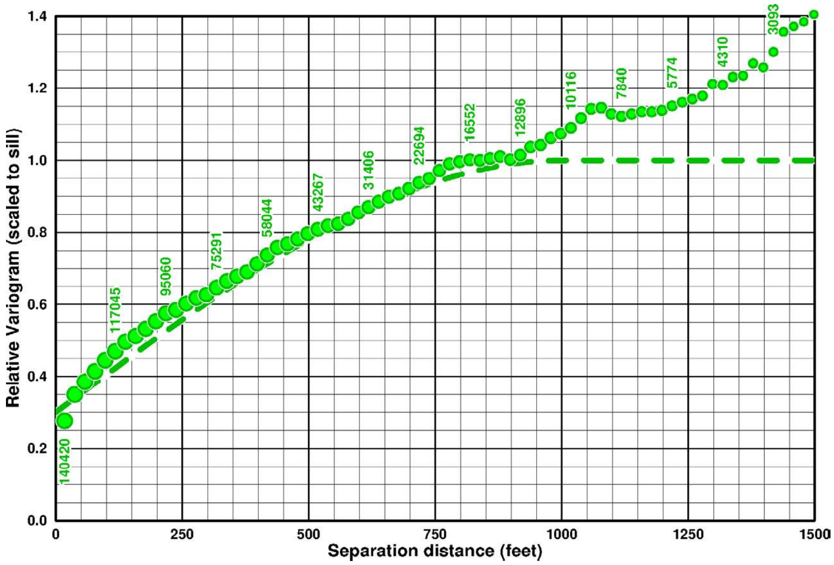


Figure 14.10 shows the experimental downhole variogram calculated in the vertical direction. With many of the holes being vertical, or within 30° of vertical, this is very close to a vertical variogram and shows much more short-scale detail than can be seen on horizontal variograms, for which the closest sample spacing is the closest drillhole spacing: about 200 ft.

Figure 14.10. Experimental pairwise downhole variogram of gold grades.



The range in the downhole direction is slightly smaller than 1,000 ft; but the downhole direction is slightly oblique to the true vertical direction in many holes, which will cause its range to appear slightly shorter than the true vertical range. Consistent with the fact that the intrusion is close to vertical, the range of continuity in the vertical direction was modeled as 1,000 ft. The zones of similar gold grades shown schematically in Figure 14.4 will have the shape of vertical cylindrical bands. Gold grades have their greatest continuity along these bands, either horizontally or vertically, and will have their minimum continuity in the direction perpendicular to these bands, running horizontally and radially outward from the stock.

14.5 Assignment of Missing Metamorphic Population Codes

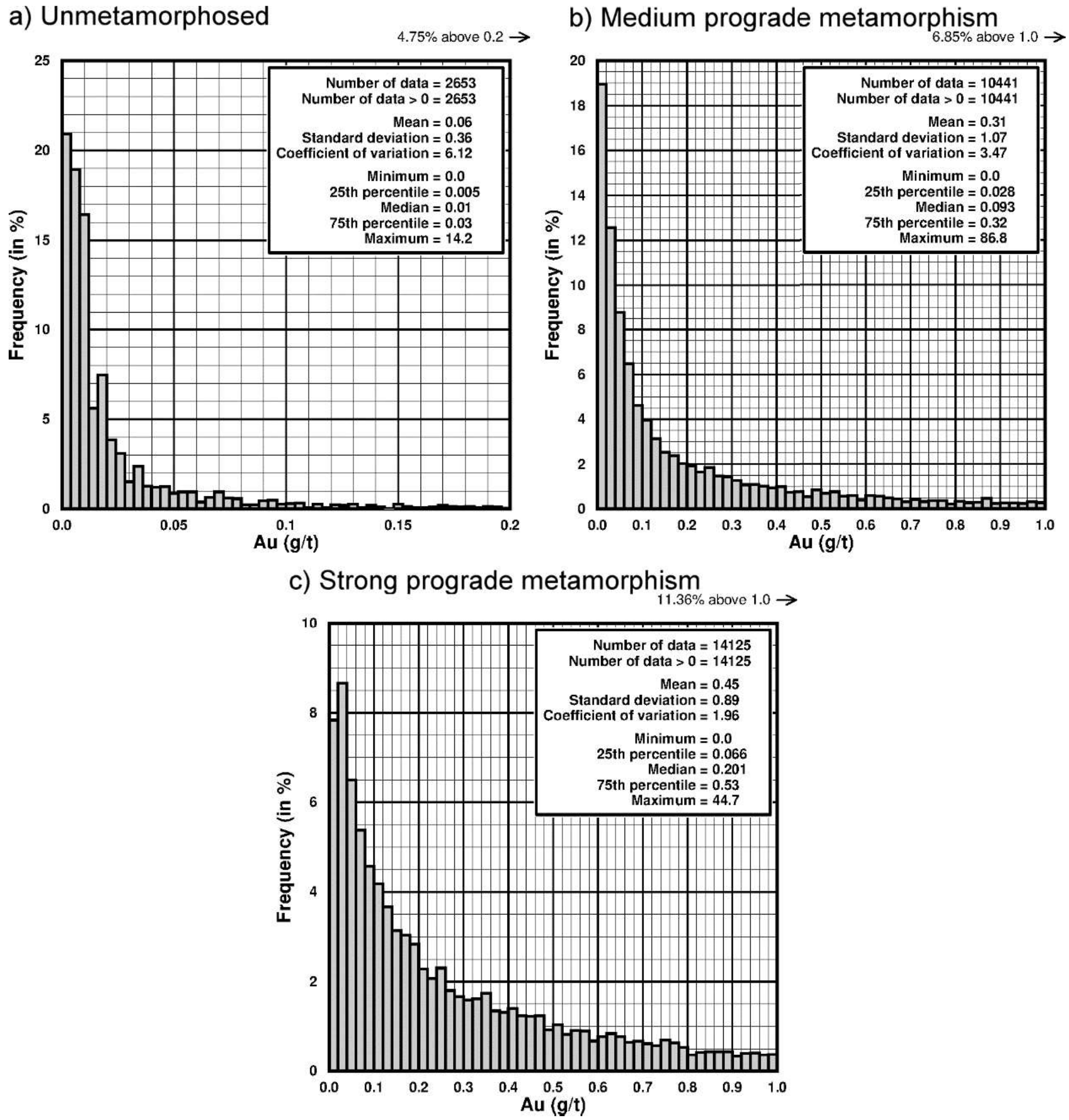
The majority of bedrock sample intervals (64%) were logged for prograde metamorphism. The remaining 36% needed to be assigned to one of the three metamorphic populations because their grade information was important to grade estimation. This assignment was done using the distance to the central stock, with samples located near the stock, on its outside edge, being coded as having "strong" prograde metamorphism, samples more than 1,500 ft from the stock as being unmetamorphosed, and the rest of the unlogged sample intervals being coded as having "medium" prograde metamorphism.

14.6 Grade Distributions and Summary Statistics

14.6.1 Gold

Figure 14.11 shows the distributions of bedrock gold assays in each of the metamorphic populations, with their summary statistics.

Figure 14.11. Gold grade histograms and summary statistics in the three metamorphic populations.



The average gold grade is 4-5x higher in the population with medium metamorphism than in the unmetamorphosed population, and 6-7x higher in the population with strong prograde metamorphism. Assay gold grades above 1 g/t can occur in any population, but are rare in the unmetamorphosed population. One gram/tonne is the 93rd percentile of the distribution in the medium population, and the 89th percentile in the strongly metamorphosed population. Of the gold metal contained in the drillhole assay database, 66% of it lies in strongly metamorphosed samples, 33% in samples with medium metamorphism, and only 1% in the unmetamorphosed population. The coefficient of variation (CV), a barometer of the influence of erratic high values on grade estimates, is slightly below 2 in the strongly-metamorphosed population that accounts for most of the metal content. Treating the deposit as three domains, separate but inter-mixed, had the beneficial effect of making the influence of erratic high values manageable in the population that carried most of the metal content.

14.6.1.1 Capping of Gold Grade by Metamorphic Population

In order to limit the influence of erratic high values, gold assays were capped using an upper limit that brings the CV on each of the metamorphic populations down to 2. Table 14.2 summarizes the capping levels for each population and the effect that capping has on the average grade.

Table 14.2. Gold capping levels and the effect of capping in each population.

	Capping Value (g/t Au)	Decrease in Average Grade After Capping (%)
Unmetamorphosed	2	14
Medium prograde metamorphism	15	3
Strong prograde metamorphism	30	<1

14.6.2 Secondary Metals: Silver, Copper and Cyanide-Soluble Gold

The current resource block model contains estimates of grades for silver and copper. Although gold accounts for well over 90% of the economic value of the deposit, grade estimates for these other metals will be useful for future technical and economic analysis. Similarly, block-by-block estimates of the cyanide-soluble gold may be needed for further engineering studies.

14.6.2.1 Silver

Figure 14.12 shows the histograms of silver grades from four-acid ICP analysis and from aqua-regia ICP analysis; the aqua-regia silver assays were clearly significantly lower, on average, by almost 50%. This is not surprising since four-acid digestion gets almost all of the silver into solution while aqua-regia results in only a partial digestion of silver.

Checks were done of the possibility that the 50% difference was due to sample location (i.e. aqua-regia ICP assays were done in regions with lower grades), but this was not the case. With the low-bias in aqua-regia ICP assays being well understood and well documented, silver assays completed by ICP with an aqua-regia digest were increased by 50% to remove the known bias.

Figure 14.12. Histograms and summary statistics of Ag assays by: a) 4-acid ICP; and, b) Aqua-regia ICP.

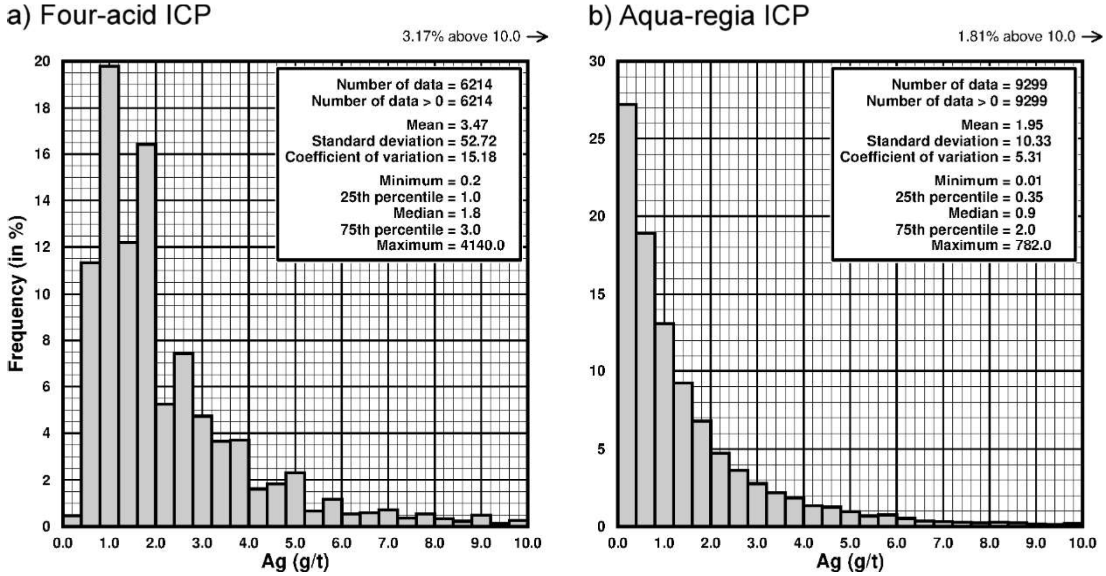
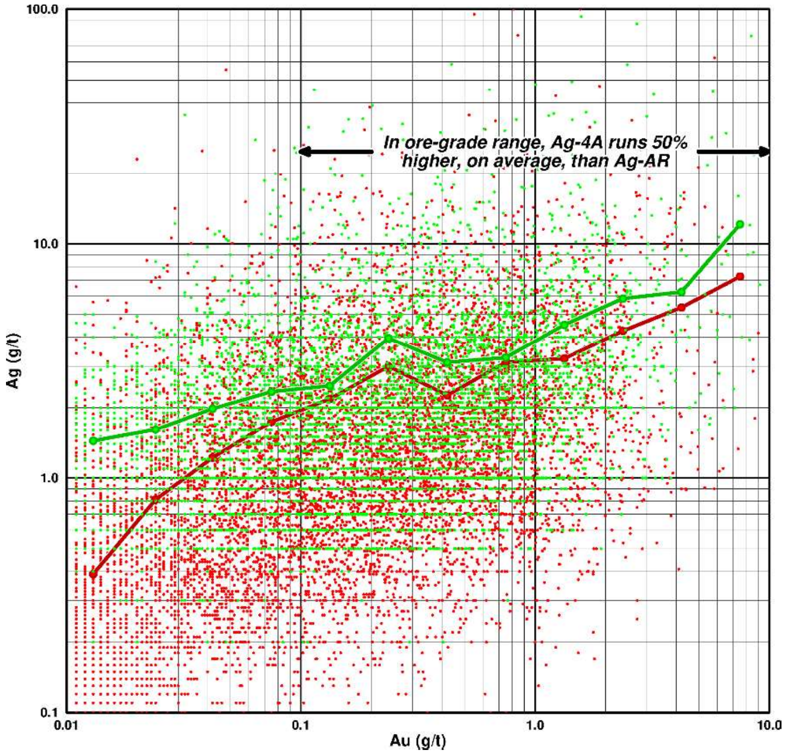


Figure 14.13 shows that the large difference in the average silver grades is not a product of the locations where each ICP method was used. With silver and gold being positively correlated, gold grades can serve as a yardstick by which to judge whether the aqua-regia ICP assays are lower because they happened to be taken from low-grade regions of the deposit. The moving-average lines in Figure 14.13 show that aqua-regia silver assays are low, on average, regardless of whether the region was low-grade (for both gold and silver) or high-grade. Above 0.1 g/t, the difference is approximately 50%. For the purposes of adding silver grades to the Converse block model, aqua-regia ICP analyses of silver were increased by 50% to take into account their systematic low bias. This is a slightly conservative adjustment, lower than the 78% seen in the comparison of the means in Figure 14.14, and consistent with the 50% seen in the comparison of the moving-averages above 0.1 g/t Au in Figure 14.13.

Following the 50% adjustment to the aqua-regia silver assays, all the silver assays were combined into a single variable for grade estimation, and capped at 100 g/t. The capping level was chosen in the same way it was for the 2012 resource estimate, by visual examination of the high-grade tail on the cumulative probability plot of silver assays. The top 10 silver assays were affected by the 100 g/t cap.

Figure 14.13. Silver versus gold for four-acid Ag assays (green) and aqua-regia Ag assays (red).



14.6.2.2 Copper

Copper assays analyzed by ICP were affected by the same issue discussed in the previous sub-section: ie aqua-regia does not get all of the copper into the solution analyzed by the ICP instrument. Figure 14.14 shows the histograms of copper for the two types of copper assays; on average, the aqua-regia copper assays were about 25% lower than the four-acid copper assays. Copper assays analyzed by ICP with an aqua-regia digest were increased by 25% to remove the known bias.

For the purposes of adding copper grades to the Converse block model, aqua-regia ICP analyses of copper were increased by 25% to take into account their systematic low bias. This is consistent with 25% seen in the comparison of the moving-averages above 0.1 g/t Au in Figure 14.15.

Figure 14.14. Histograms and summary statistics of Cu assays by: a) 4-acid ICP; and, b) Aqua-regia ICP.

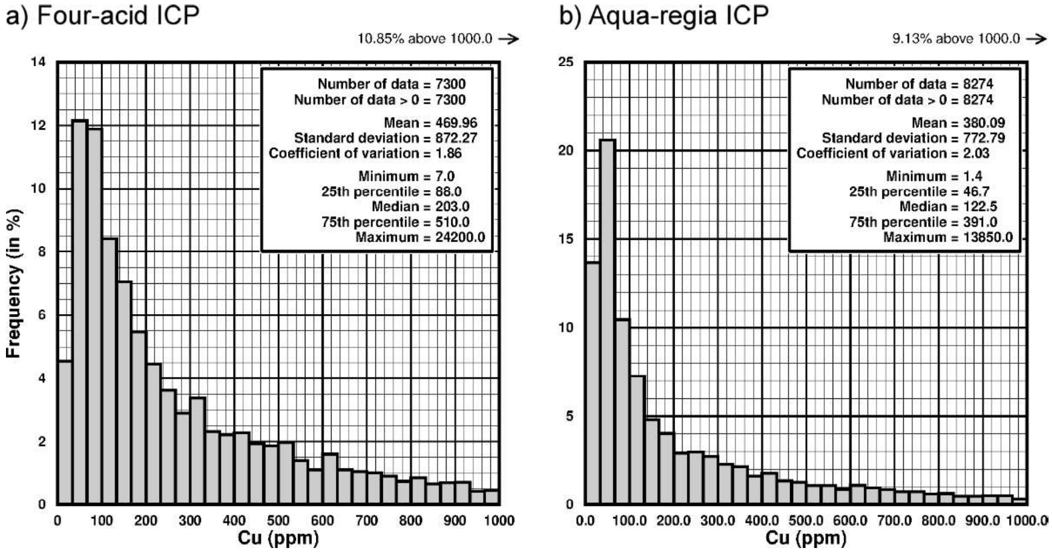
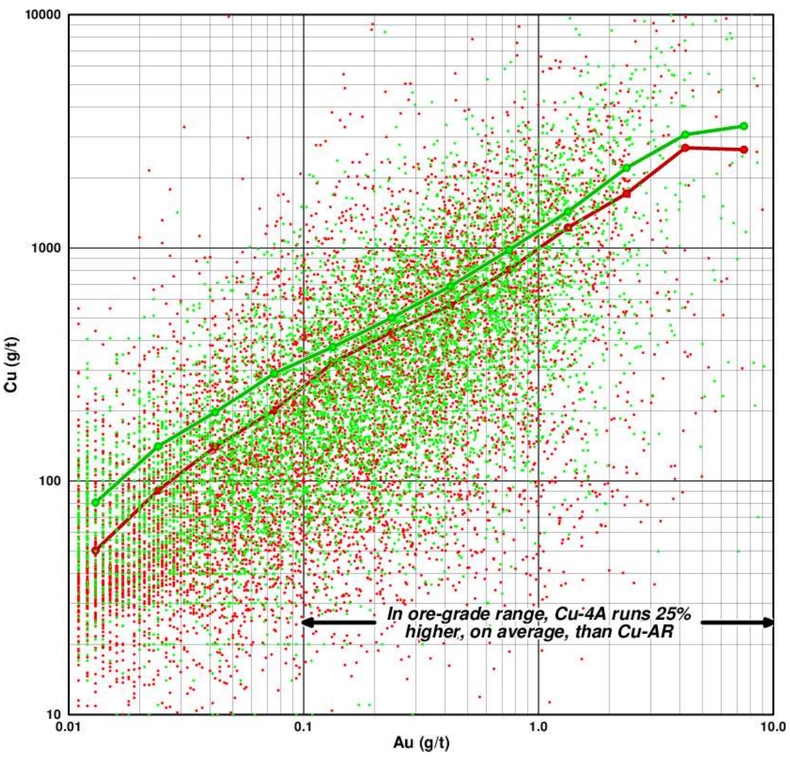


Figure 14.15. Copper versus gold for four-acid Cu assays (green) and aqua-regia Cu assays (red).



Following the 25% adjustment to the aqua-regia copper assays, all the copper assays were combined into a single variable for grade estimation and capped at 10,000 ppm, based on visual inspection of the upper tail of the cumulative distribution; this affected the top 11 copper assays.

14.6.2.3 Cyanide-Soluble Gold

Cyanide-soluble gold grades were capped at the total-gold grade, so that cyanide-soluble gold recovery would not exceed 100%.

14.7 Use of Assays

The majority (>90%) of the sample intervals in the assay database are exactly 5 ft in length. Since there is no relationship between sample length and variability of gold grades, or between sample length and grade, composites were not created for resource estimation. Instead, grade interpolation was done directly with the drillhole assays.

14.8 Grade Estimation

14.8.1 Block Model Configuration

The block size is 50 x 50 x 20 ft; no sub-blocking was used. Within the central area where grades were estimated, the block model had 77 columns in the east-west direction, 119 rows in the north-south direction, and 117 benches in the vertical direction.

For the larger area (the green outline in Figure 14.1), the block model had 153 columns in the east-west direction, 187 rows in the north-south direction, and 117 benches in the vertical direction.

14.8.2 Gold Estimation

14.8.2.1 Bedrock

Estimation of the gold grade in each block was done in two steps:

1. The volume proportion of each metamorphic population is estimated using IK; this gives p_1, p_2 and p_3. The search-neighbourhood parameters and the variogram model parameters for each of these IK runs were identical to those used for the grade estimation (Table 14.3).

2. The gold grade of each population was then estimated using OK of the nearby samples that fell within that population; this gives Au_1, Au_2 and Au_3.

The final block grade is the proportion-weighted average of the grades for each population:

$$Au_{block} = p_1 \times Au_1 + p_2 \times Au_2 + p_3 \times Au_3$$

The same search strategy and variogram models were used for all of the kriging runs. The search and variogram parameters are given in Table 14.3. The variogram ranges were aligned with the search ellipsoid, both being 1,000 ft in the direction parallel to the stock and in the vertical direction, and 300 ft in the perpendicular direction, horizontal and radially outward from the stock.

Blocks that could not be estimated in the first pass were picked up in a second pass, with the search ellipsoid doubled in size and the constraints on the number of samples relaxed.

Table 14.3. Estimation parameters used for ordinary kriging of gold and indicator kriging of volume proportions.

Variogram Model	Relative nugget effect	30% of sill	
	Long range	1,000 ft	
	Intermediate range	1,000 ft	
	Short range	300 ft	
	Long direction	Horizontal, parallel to edge of stock	
	Intermediate direction	Vertical	
	Short direction	Horizontal, perpendicular to stock	
	Variogram model shape	Spherical	
Search Strategy		First Pass	Second Pass
	Size of search ellipsoid	1x variogram ranges	2x variogram ranges
	Minimum # of samples	2	2
	Maximum # of samples per octant	4	4
Block Discretization	Number of points in X, Y and Z	50 points spaced 10 ft apart in all directions on a regular 5 x 5 x 2 grid	

14.8.2.2 Quaternary Alluvium

Alluvium samples do occasionally show mineralization above the resource reporting cut-off, which suggests that there may be opportunities during stripping to treat some of the alluvium as mineralized material. For this reason, gold grades were estimated for blocks in alluvium; these blocks were always classified as Inferred.

Assay gold grades in alluvium were capped at 1 g/t Au, based on visual inspection of the upper tail on the plot of the cumulative probability distribution. This capping affects ten of the 4,804 sample intervals with gold grades in the Quaternary alluvium.

Block grades were estimated using OK with the same search and variogram parameters as were used for bedrock, with the two long axes of the search ellipse and variogram model lying parallel to the alluvium/bedrock contact. This flat-lying orientation was chosen because the gold in the gravels is a product of erosion of the underlying bedrock. Lenses with moderate gold mineralization likely formed in alluvial fans whose bedding lay within a few degrees of the slope on which they formed.

14.8.3 Silver, Copper and Cyanide-Soluble Gold

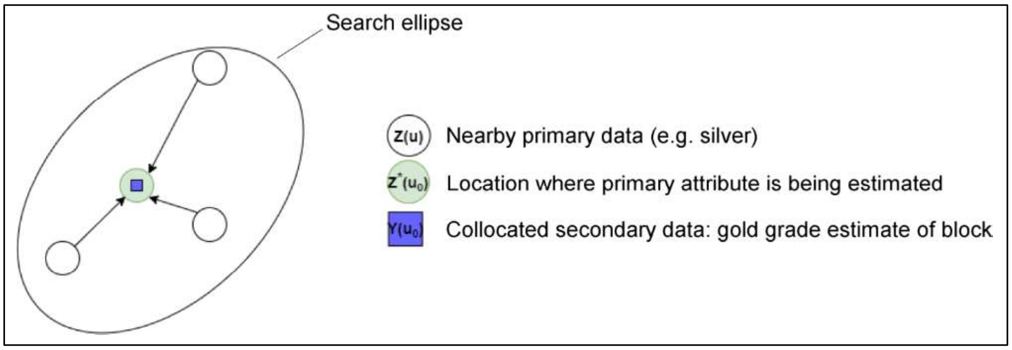
Further engineering studies for the property will benefit from having estimates of silver, copper and cyanide-soluble gold in every block in the block model. As seen in Figure 14.2, many of the drillholes did not have multi-element ICP data and were, therefore, missing silver and copper assays. This creates large gaps in the silver and copper assays, especially in areas with lower silver and copper grades. Estimates of silver, copper and cyanide-soluble gold grades could be created by using the available data with search neighbourhoods that are sufficiently large to span the gaps in the assay data. But with the gaps tending to have lower grades, the approach of using very large search neighborhoods would result in overestimation of the silver and copper grades in the areas that lack assays, i.e. the higher grades in far away drillholes will simply be smeared into the gaps. The approach used for the current resource model was to use cokriging, a method that brings in secondary data to improve the accuracy of estimates. With gold grades being positively correlated with silver, copper and cyanide-soluble gold grades, and with gold assays being present everywhere, even in the drillholes in regions with lower grade, collocated co-kriging (Goovaerts, 1997) with gold as the secondary variable provided a technically sound method for estimating silver, copper and cyanide-soluble copper grades throughout the study area.

14.8.3.1 Co-Kriging

Collocated co-kriging (Goovaerts, 1997) was used to avoid the problem of over-estimation of silver, copper and cyanide-soluble gold. As shown in Figure 14.16, collocated co-kriging is an interpolation that uses nearby data for the variable being estimated (the "primary" variable) and also incorporates into the estimation the value of another correlated variable (the "secondary" variable). The primary data occur at various locations nearby; the secondary data occurs exactly at the location being estimated, giving rise to the name "collocated".

In the conventional implementation of collocated co-kriging, the variogram models of the primary and secondary variables are assumed to have the same shape, and the cross-variogram between primary and secondary is assumed to also have the same shape, but with a relative nugget effect equal to one minus the correlation coefficient between the two variables. This assumption is justified when the secondary value at the location being estimated completely screens out the relevance of any other secondary data (Goovaerts, 1997). In the case of the Converse Project, this condition says that if one is estimating silver grades, and using gold grade to assist that estimation because of sparse silver data, then the only gold grade one needs to take into account is the gold grade at exactly the same location where silver is being estimated, i.e. gold grades at other locations aren't relevant.

Figure 14.16. Schematic showing how collocated co-kriging combines nearby primary data with the one piece of secondary data at the location being estimated.



Source: Samson and Deutsch, (2020)

14.9 Validation of Grade Block Models

14.9.1 Visual Checks

The consistency of the resource block model with drillhole assay data and geology was checked visually on plan. Figure 14.17 shows an example from the North Redline area; Figure 14.18 shows an example from the

South Redline area. The block model and drillholes were also plotted on cross-sections. Figure 14.19 shows an example through North Redline and Figure 14.20 shows an example through South Redline. With the ranges of correlation extending up to 1,000 ft with the drillholes having different orientations and with the direction of maximum continuity changing continuously around the stock, it is sometimes difficult to see the drillhole samples responsible for certain blocks on a simple 2D map or cross-section. The 3D visualizer of Micromine, the commercial software used for interpolation of metamorphic population proportions and for gold grades, was also used to investigate individual blocks and their surrounding drillhole data.

Resource classification regions were overlaid on the maps and sections to check that blocks were being correctly classified. The reporting pit shell was also overlaid to check that blocks outside the pit were not included in the resource tabulation.

There are locations in the deposit where the resource block model shows a halo of well-mineralized blocks below drillholes whose last sample was well-mineralized. These, however, are few; and the vertical extrapolation of grade is consistent with the geological understanding. Furthermore, drillholes that ran deeper in the same area (but beyond the range of correlation) also often showed similar mineralization, so the prediction of occasional pods of deep mineralization is not inconsistent with data or geology. In all cases, the classification of blocks passed from Measured or Indicated near the bottom of drillholes to Inferred below the hole if there are no other nearby drillholes to support grade estimation. The few halos of high-grade estimates at the bottom of holes that bottomed in high-grade mineralization were not adjusted or removed from the block model but were classified as Indicated since they meet the definition of an Indicated resource: the continuity from assays can be reasonably assumed, but not confirmed. Further drilling in these areas will be needed to confirm the details of the tonnage and the geometry of the deep high-grade regions in the block model.

14.9.2 Statistical Checks

14.9.2.1 Average Assay Grade versus Block Estimate for Blocks Penetrated by Drilling

For those blocks that are penetrated by drilling, Figure 14.21 shows a plot of the average gold assay grade versus the block grade estimate for the Measured and Indicated blocks.

14.9.2.2 Correlation Between Gold Grade Estimates and Estimates of Other Attributes

Figure 14.22 shows a plot of the block estimates of silver versus gold. This confirms that the collocated co-kriging achieved its goal of imparting plausible correlation even in areas where there were few silver assays. The correlation parameter used for the collocated co-kriging of silver was 0.64, which is very close to the correlation actually achieved: 0.59.

The same check was carried out for copper and cyanide-soluble gold estimates, both of which also correlate with gold assays in drillholes and should, therefore, also be correlated at the block level. For copper, the collocated co-kriging correlation parameter was 0.49, which is very close to the correlation actually achieved: 0.42. For cyanide-soluble recovery, the collocated co-kriging correlation parameter was 0.44, which is very close the correlation actually achieved: 0.37.

Figure 14.17. Map showing gold grade estimates in the North Redline area on the 3960-3980 ft bench and drillhole assays within ±200 ft above and below the bench.

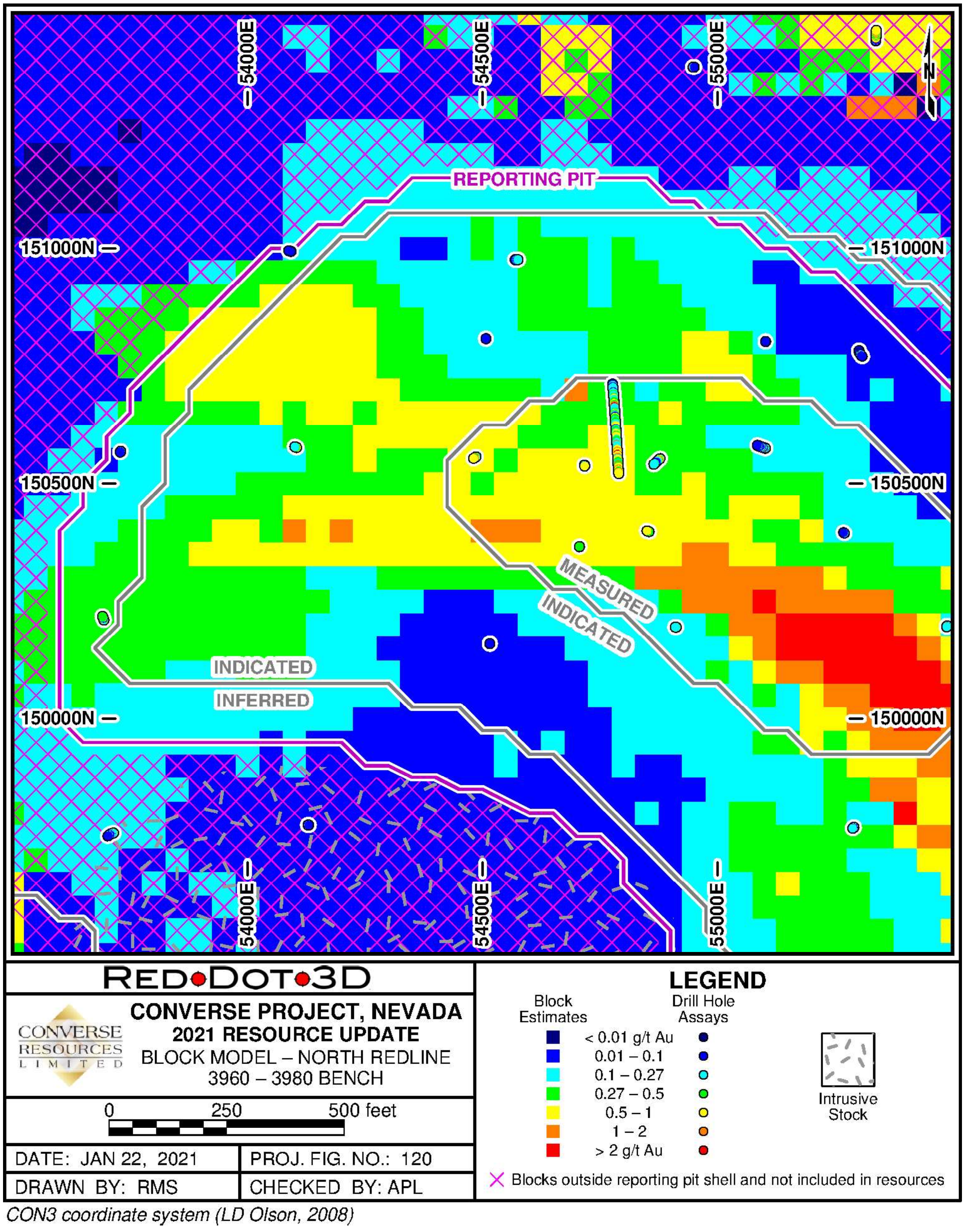


Figure 14.18. Map showing gold grade estimates in the South Redline area on the 3960-3980 ft bench and drillhole assays within ±200 ft above and below the bench.

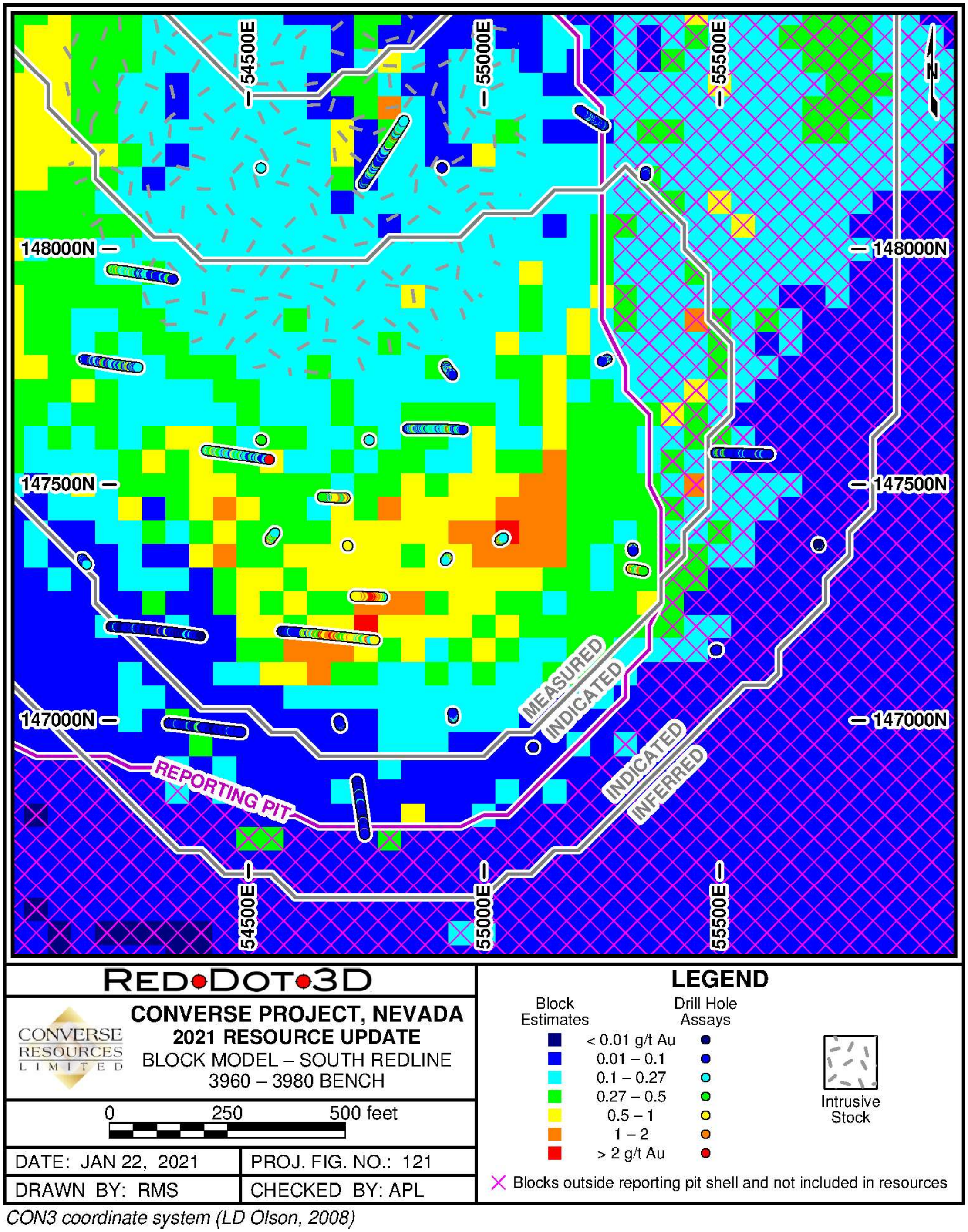


Figure 14.19. North-facing cross-section through the North Redline area showing gold grade estimates at 150500N and drillhole assays within ±200 ft on either side.

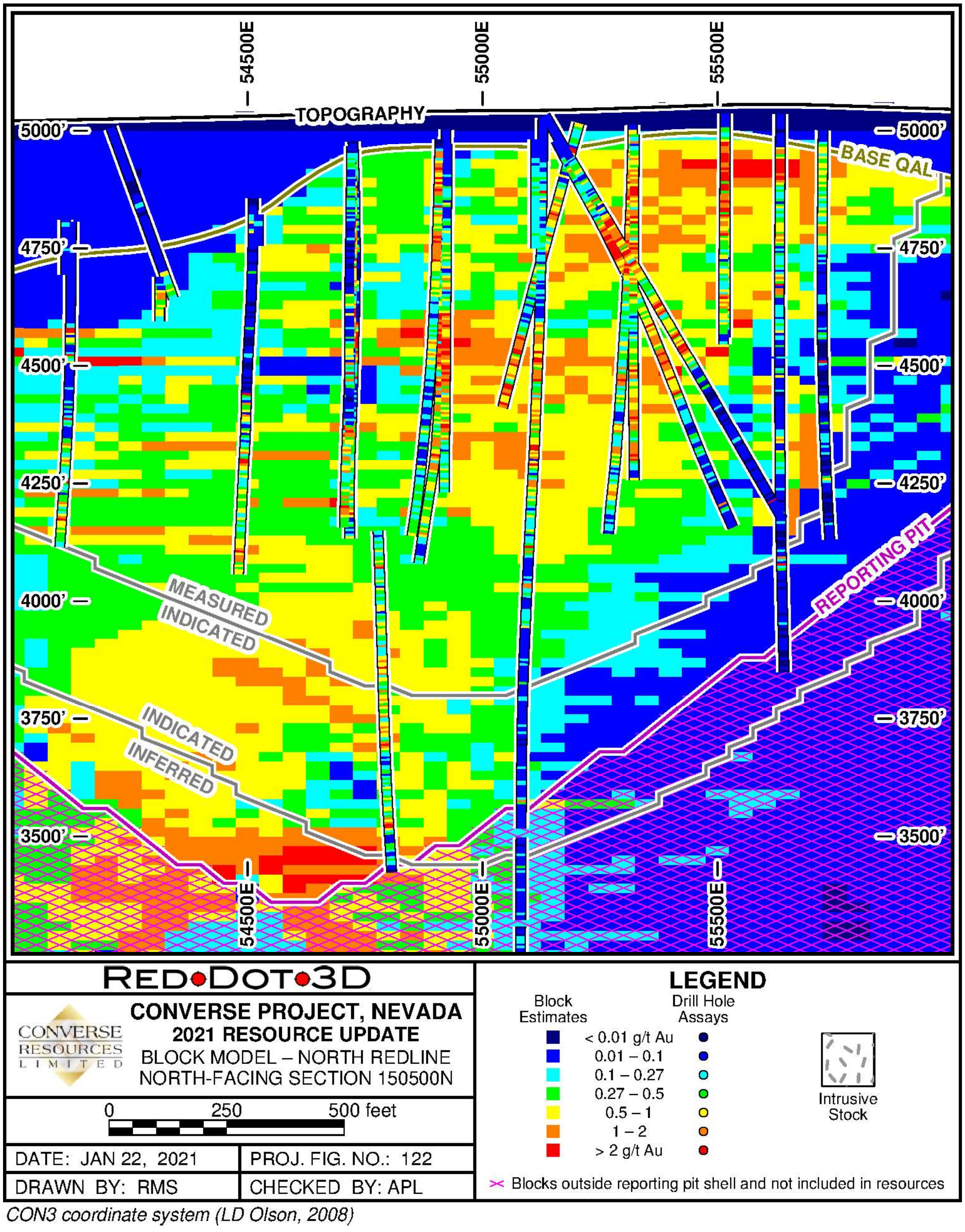
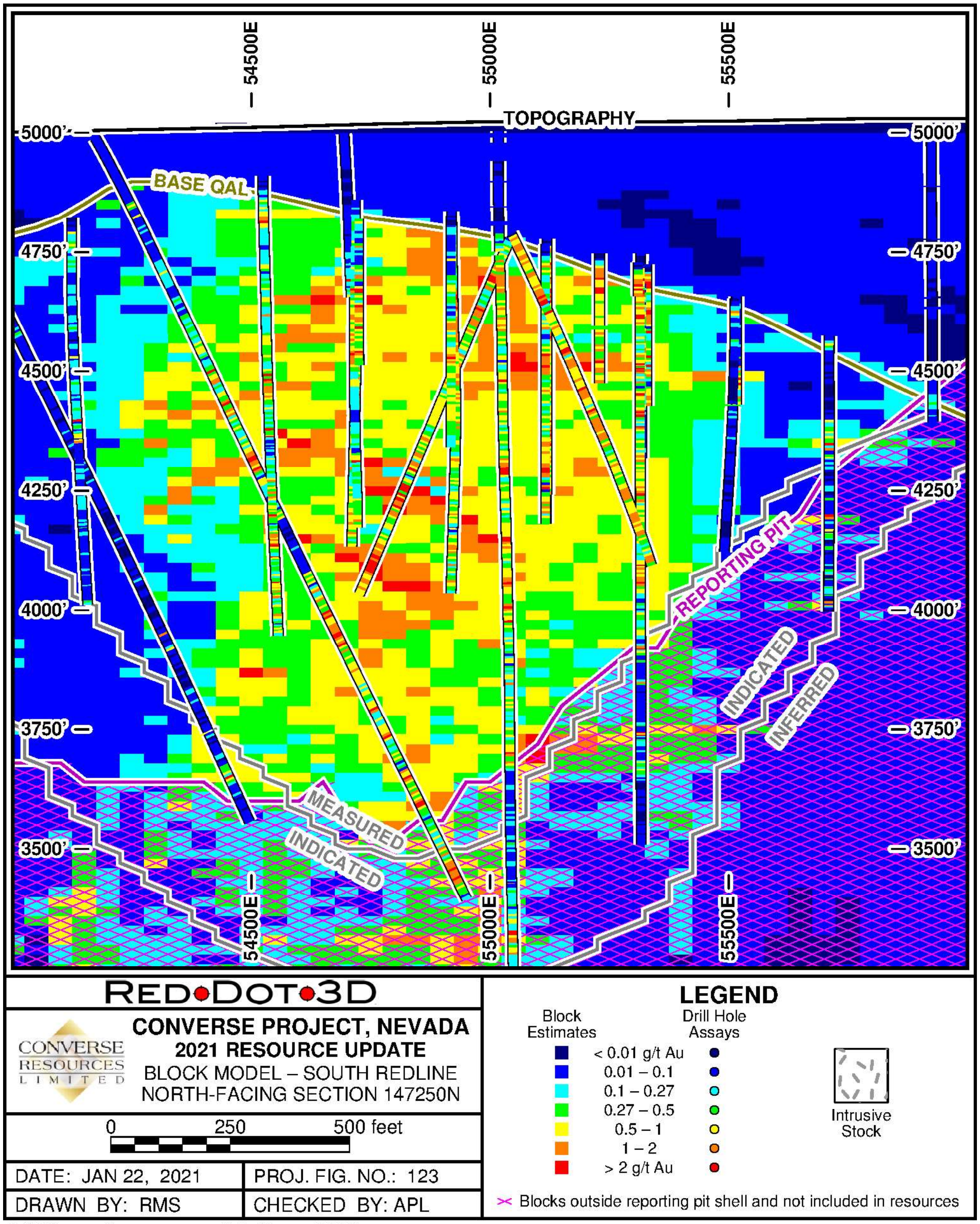


Figure 14.20. North-facing cross-section through the South Redline area showing gold grade estimates at 147250N and drillhole assays within ±200 ft on either side.



CON3 coordinate system (LD Olson, 2008)

Figure 14.21. Gold block estimates versus average of gold assays within the same block.

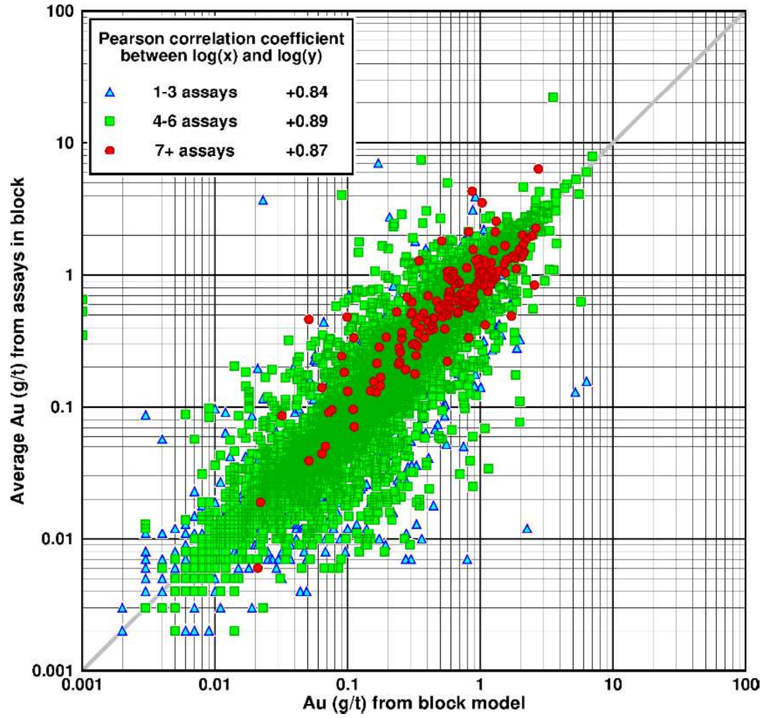
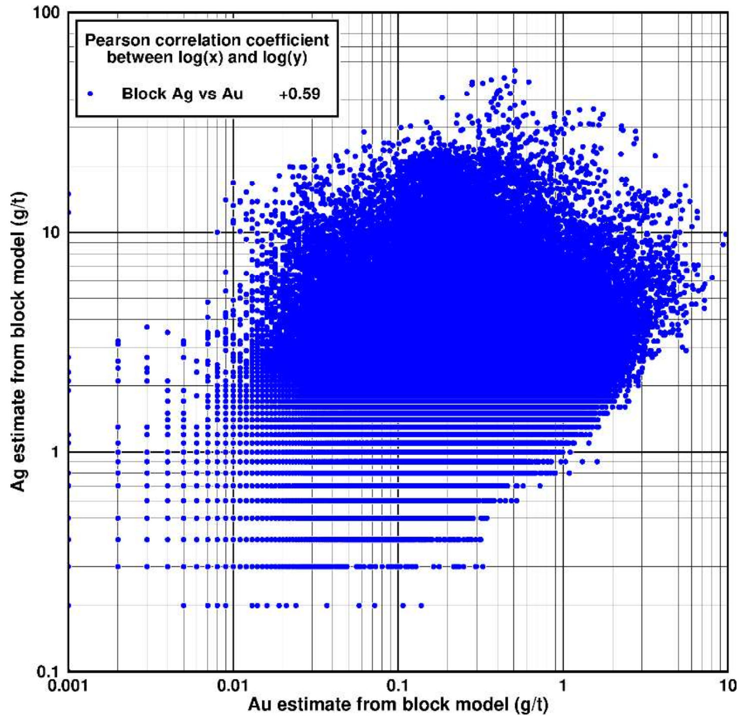


Figure 14.22. Silver versus gold grade estimates for Measured and Indicated blocks.

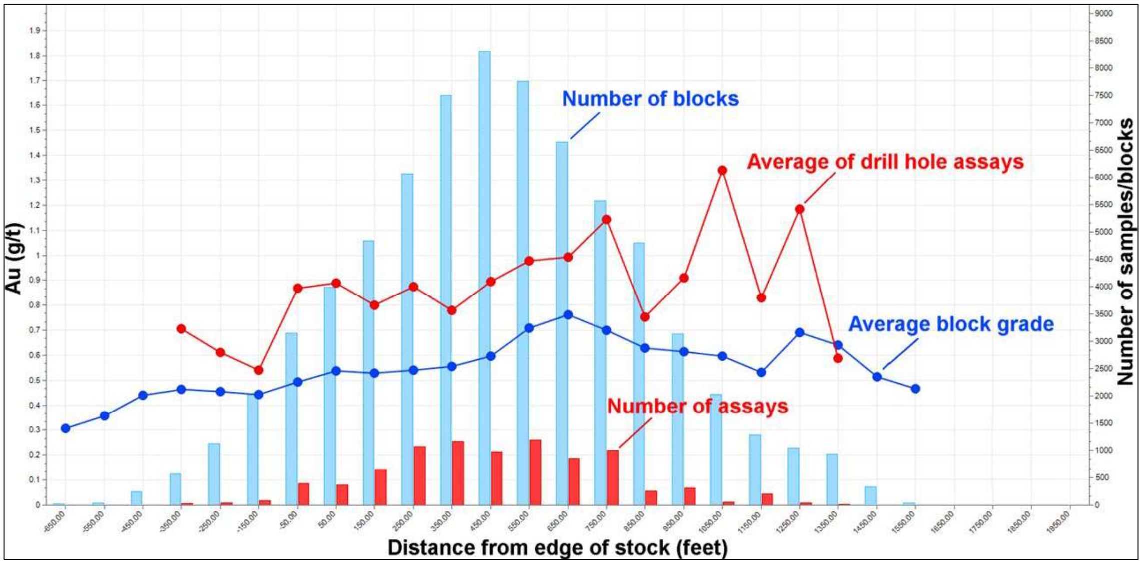


14.9.2.3 Swath Plots

Figure 14.23 shows a swath plot comparing gold estimates to drillhole data as a function of radial distance from the edge of the stock, the direction in which there is a clear trend.

The swath plots in the X, Y and Z directions confirm that the block model does capture the broad spatial sense of the trend in gold grades, rising as one moves away from the central stock and the falling as one moves further out into the unmetamorphosed rocks. All of these plots showed an acceptable agreement between block estimates and drillhole data, there is no strong trend in any of those directions, so the swath plots simply show gold block estimates and gold assays from drillholes fluctuating around the overall average for the deposit.

Figure 14.23. Swath plot of gold block estimates and drillhole data in the radial distance direction, for all Measured and Indicated regions inside the reporting pit shell.



14.9.3 Grade-Tonnage Curves and Global Volume-Variance Correction of Gold Distribution

Figure 14.24 and Figure 14.25 compare the global grade-tonnage curves from the Measured and Indicated blocks inside the pit shell to the grade-tonnage curves calculated using a global change of support (Parker, 1980). The variance adjustment factor used for the global change of support was 0.21, which was calculated from the variogram model, assuming that the selective mining unit is the same size as the resource blocks: 50 x 50 x 20 ft.

With the tonnage-above-cut-off and grade-above-cut-off curves both matching well, this check confirms that there is an appropriate degree of smoothing in the block estimates.

Figure 14.24. Tonnage above cut-off calculated from the Measured and Indicated blocks in the resource block model, and from a global change-of-support adjustment to the distribution of gold assay grades.

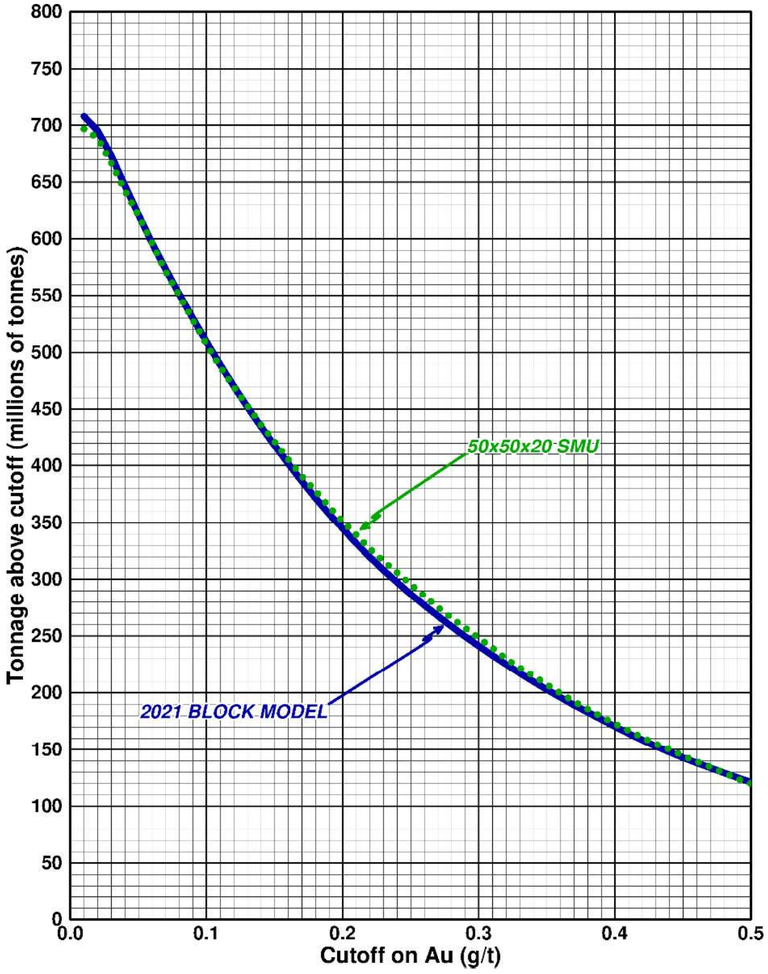
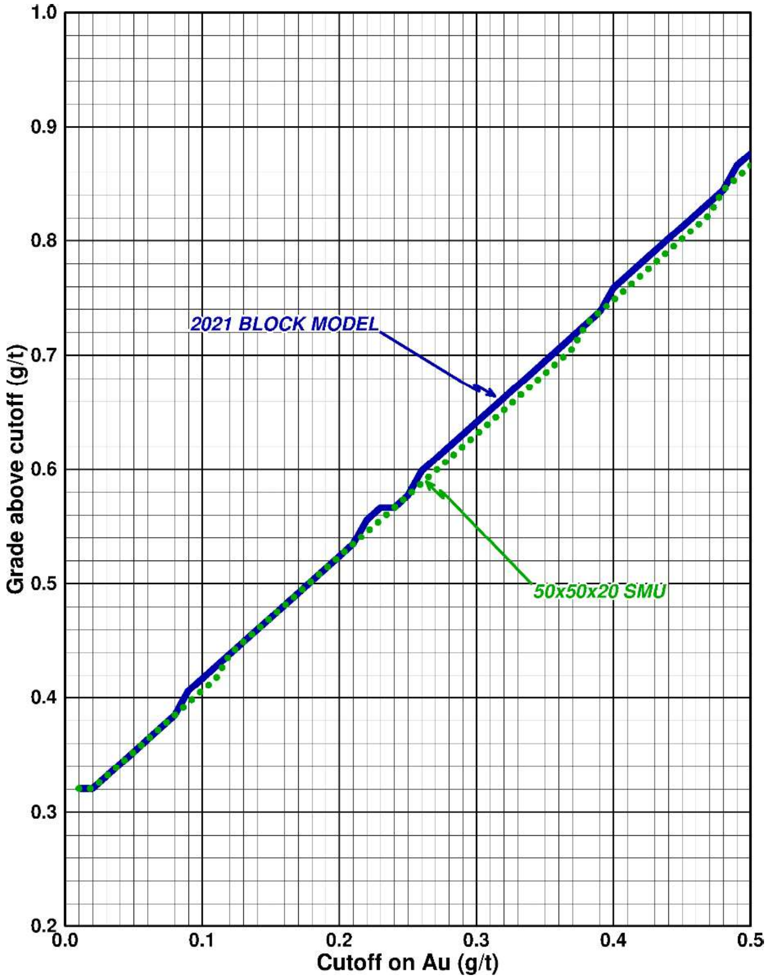


Figure 14.25. Grade above cut-off calculated from the Measured and Indicated blocks in the resource block model, and from a global change-of-support adjustment to the distribution of gold assay grades.



14.10 Assignment of Density and Tonnage

Bulk density determination methods were described in Section 11.3. Tabe 14.4 shows that average dry bulk density of the samples in the alluvium and three bedrock redox layers.

Table 14.4. Average dry bulk density for alluvium and three redox layers

	Count	Average Density (g/cm3)
Quaternary alluvium	74	1.93 ± 0.06
Oxide	57	2.56 ± 0.08
Mixed	109	2.67 ± 0.02
Sulfide	72	2.67 ± 0.02
Total	312	2.47 ± 0.04

14.11 Classification of Mineral Resources

The resource block model was classified into Measured, Indicated and Inferred Mineral Resources using a two-step procedure. In the first step, integer codes were assigned on a block-by-block basis, using the following criteria from the kriging of the population proportions:

- Blocks were assigned a 1 if they had data in at least four octants and if the average weighted variogram distance to the nearby samples was 33% of the variogram range, or less;

- Blocks were assigned a 2 if they had data in at least four octants and if the average weighted variogram distance to the nearby samples was 66% of the variogram range, or less;

- Blocks were assigned a 3 if they couldn't be assigned a 1 or 2.

In the second step, the integer codes were smoothed to remove small islands of one classification that were completely surrounded by a different classification. The requirement of having data in at least four octants ensured that assays from at least two different drillholes were used to estimate grade. The criteria related to the average weighted variogram distance ensured that samples assigned a 1 were in areas with 200 ft drill spacing, and that samples assigned a 2 were in the areas with 400 ft drill spacing.

In the second step, the integer codes were smoothed by calculating a moving average within a 1,000 x 1,000 x 200 ft (305 x 305 x 61 m) rectangular block, a volume which corresponded approximately to three months of mineralization and waste mining. The purpose of this smoothing step was to remove the small-scale local variation in the block-by-block codes and, in so doing, ensure that the classification provided meaningful information about the confidence in grade estimates and classification at the scale of quarterly production.

Figure 14.26 shows the classification on one bench. On the left, the colours show the original integer codes with significant local variation. As is usually the case with block-by-block classification using criteria linked to the data in the search neighbourhood, these preliminary codes showed short-scale variability: single blocks coded as 3 (blue), for example, but entirely surrounded by blocks coded as 2 (yellow) or vice-versa.

The left side of Figure 14.26 shows the classification following the smoothing step, where the Measured region now follows the well-drilled areas of North Redline, South Redline and the high-grade pod to the west of the intrusion. All blocks above the bedrock, in the alluvium, were classified as Inferred.

Figure 14.26. Preliminary integer codes, on the left, that identify different levels of confidence at the block-by-block level (red = 1, yellow = 2, blue = 3); and, on the right, classification codes obtained by smoothing the preliminary integers (red = Measured, yellow = Indicated, blue = Inferred).

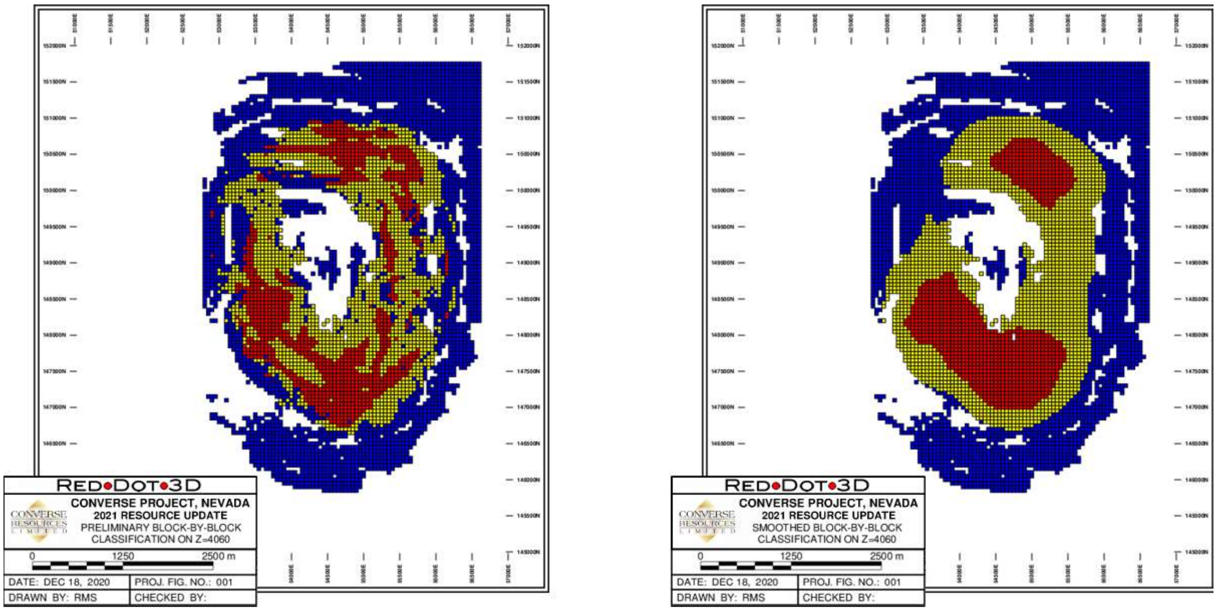
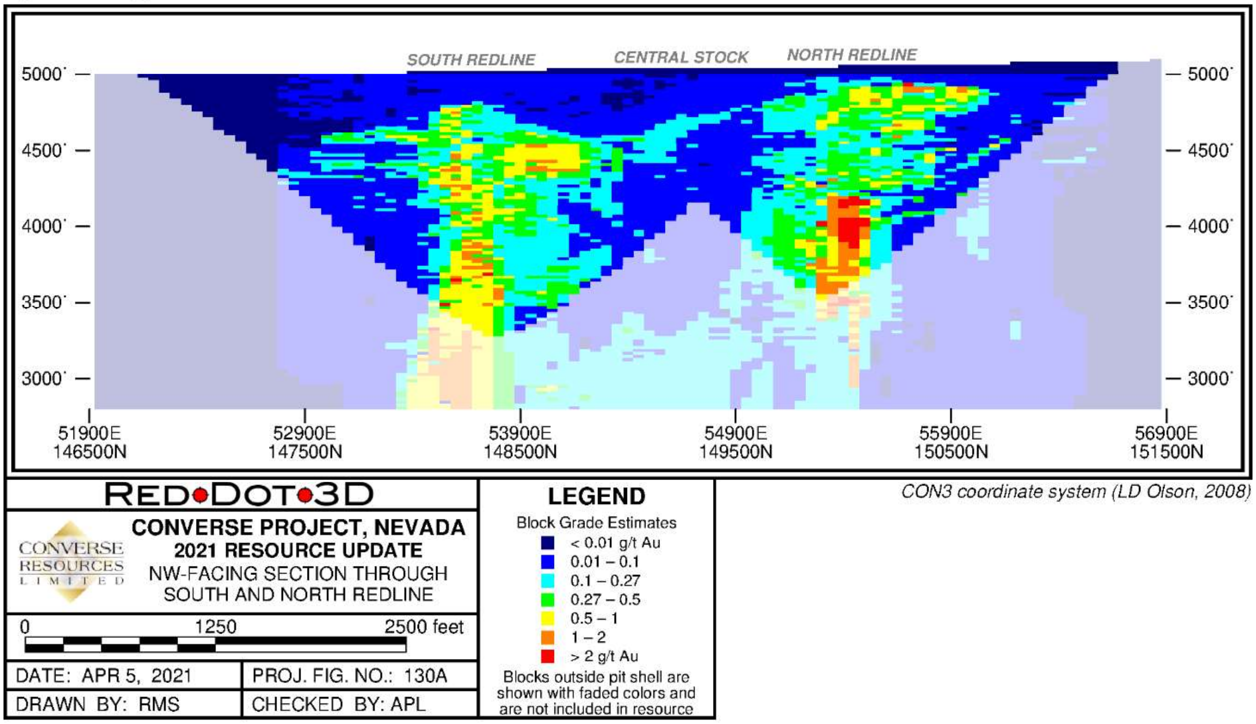


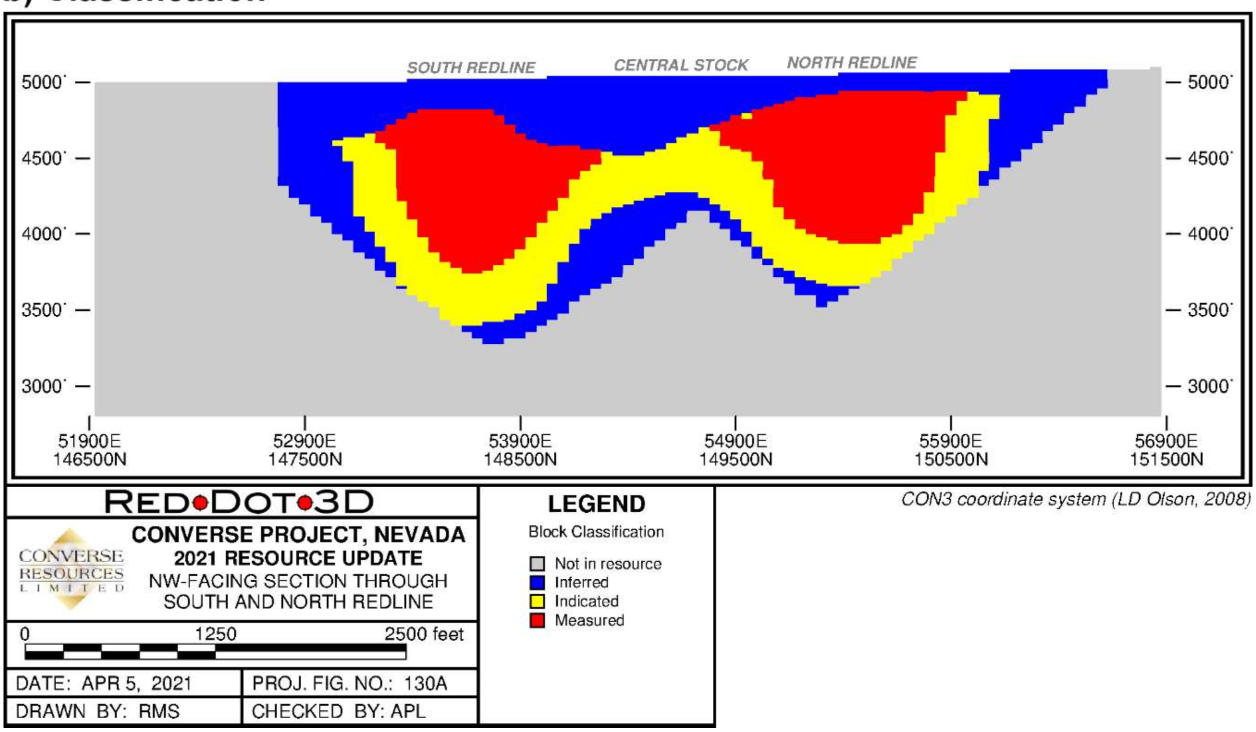
Figure 14.27 shows a southwest-northeast cross-section of gold grades and classification, running from South Redline on the left through the central stock and across to North Redline on the right.

Figure 14.27. Northwest-facing cross-section through the block model, with grade estimates shown at the top and classification shown at the bottom.

a) Gold grades



b) Classification



14.12 Reasonable Prospects of Eventual Economic Extraction

A Mineral Resource pit shell was constructed to define the portion of the Converse MRE having reasonable prospects for eventual economic extraction (RPEEE) amenable to open pit mining and processing by run of mine heap leaching using the 50 x 50 x 20 ft block model and the Whittle Lerchs-Grossman (LG) open pit optimization algorithm. Conceptual mining, processing and economic assumptions for the open pit resource shell are presented in Table 14.5.

Table 14.5. Technical and economic parameters for pit shell construction.

Parameter	Rate / Unit
Gold Price	$1,750/oz
Mining Cost	$1.30/ton mined
Process Costs	$4.80/ton processed
G&A	$0.29/ton processed
Classification	Meas., Ind. & Inf.
Recoveries	
Crushed Oxide Au	77%
Crushed Mixed Au	62%
Crushed Sulfide Au	50%
Treatment and Refining Charges	
Payable % (Au)	99.9%
Refining Cost (Au)	$2.50/oz
Royalty	6% NSR
Maximum Slope of Pit Wall	
Alluvium	36°
Bedrock	41°

14.13 Mineral Resource Statement

Table 14.6 provides the MRE for the Converse Project, tabulated at a reporting cut-off of 0.008 oz/ton (0.27 g/t) Au, which reflects a blended breakeven lower gold cutoff for the deposit. The MRE is that portion of the estimated blocks that are within the pit shell and overall is reported as undiluted and total ounces of gold.

Table 14.6. Mineral Resource Estimate Statement above a cut-off of 0.008 oz/ton Au (as of December 31st, 2020).

	US UNITS				METRIC UNITS	
	Tonnage	Au Grade	Contained Metal		Tonnage	Au Grade
Classification	(Mtons)	(oz/ton)	(Moz Au)		(Mtonnes)	(g/t)
Measured	209.0	0.018	3.79		189.6	0.62
Indicated	80.6	0.017	1.38		73.1	0.59
Meas.+Ind.	289.6	0.018	5.17		262.7	0.61
Inferred	29.1	0.019	0.55		26.4	0.65

Notes:

1. *Mineral Resources have an effective date of 31 December 2020. Mr. Mohan Srivastava, of RedDot3D Inc., is the Qualified Person responsible for the Mineral Resource estimate.*
2. *Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.*
3. *Mineral Resources are the portion of the Redline North and Redline South deposits that have reasonable prospects for eventual economic extraction by open pit mining method and processed by gold heap leaching.*
4. *Mineral Resources are constrained oxide and sulfide mineralization inside a conceptual open pit shell. The main parameters for pit shell construction are a gold price of $1,750/oz gold, variable gold recovery for oxide, mixed and sulfide mineralization, open pit mining costs of $1.30/ton, heap leach processing costs of $4.80/ton, general and administrative costs of $0.29/ton processed, pit slope angles of 36° for alluvium and 41° below base of alluvium, and a 6.0% royalty.*
5. *Mineral Resources are reported above a 0.008 oz/ton (0.27 g/t) gold cut-off grade. This is a marginal cut-off grade that generates sufficient revenue to cover conceptual processing, general and off-site costs given metallurgical recovery and long-range metal prices for gold and silver*
6. *Units are imperial tons.*
7. *Numbers have been rounded as required by reporting guidelines and may result in apparent summation differences.*
8. *The QP is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, or political factors that might materially affect the development of these mineral resource estimates.*

14.13.1 Sensitivity to Cut-off

Table 14.7 shows changes in tonnage, grade and contained metal at different cut-off grades for Measured and Indicated resources. The base case is bolded at 0.008 oz/ton (0.27 g/t) gold.

Table 14.7. Sensitivity of Measured + Indicated Mineral Resources to reporting cut-off.

		Imperial Units			Metric Units	
Cut-off (oz/ton)	Classification	Tonnage (Mtons)	Au Grade (oz/ton)	Contained Metal (Moz Au)	Tonnage (Mtonnes)	Au Grade (g/t)
0.006	Meas.+Ind.	371.3	0.015	5.74	336.8	0.53
0.008	**Meas.+Ind.**	**289.6**	**0.018**	**5.17**	**262.7**	**0.61**
0.010	Meas.+Ind.	228.8	0.020	4.62	207.6	0.69

14.13.2 Comparison to Previous Resource Estimate

The new current MRE has a similar metal content to the historical MRE developed in 2014, but with lower tonnage and a higher grade. The additional holes that have been drilled since the historical 2014 MRE account for some local changes, but most of the decrease in tonnage and increase in grade in the current MRE is attributable to the use of an approach that integrates more of the geological information. The prograde metamorphic coding allows the gold assays to be separated into distinct populations, which avoids some of the smoothing of previous models.

15 Mineral Reserve Estimates

This section is not relevant to this report.

16 Mining Methods

This section is not relevant to this report.

17 Recovery Methods

This section is not relevant to this report.

18 Project Infrastructure

This section is not relevant to this report.

19 Market Studies and Contracts

This section is not relevant to this report.

20 Environmental Studies, Permitting and Social or Community Impact

This section is not relevant to this report.

21 Capital and Operating Costs

This section is not relevant to this report.

22 Economic Analysis

This section is not relevant to this report.

23 Adjacent Properties

The Converse Property is located on the western margin of the Battle Mountain Gold Belt, a northwest linear trend that extends from the Twin Creeks gold deposits in the north to Cove-McCoy gold deposits in the south. This belt hosts several prospects and mines that account for over 50 million ounces of cumulative gold production and mineral resources over the past 25 years (Holley et al., 2015). It is the second most prolific gold belt in Nevada after the Carlin trend, and it includes variants of Carlin-Type Gold Deposits, distal type sediment hosted deposits as well as skarn and copper–gold porphyry systems.

Three major gold deposits lie adjacent to the Converse Property: the Fortitude-Phoenix Mine located approximately 20 km south-southeast of the Converse Property; the Lone Tree gold mine (now closed) located 14 km north-northeast; and the Marigold deposit located 8 km to the east (Figure 23.1). Past production and Mineral Reserves and Resources for the adjacent properties are summarized in Table 23.1.

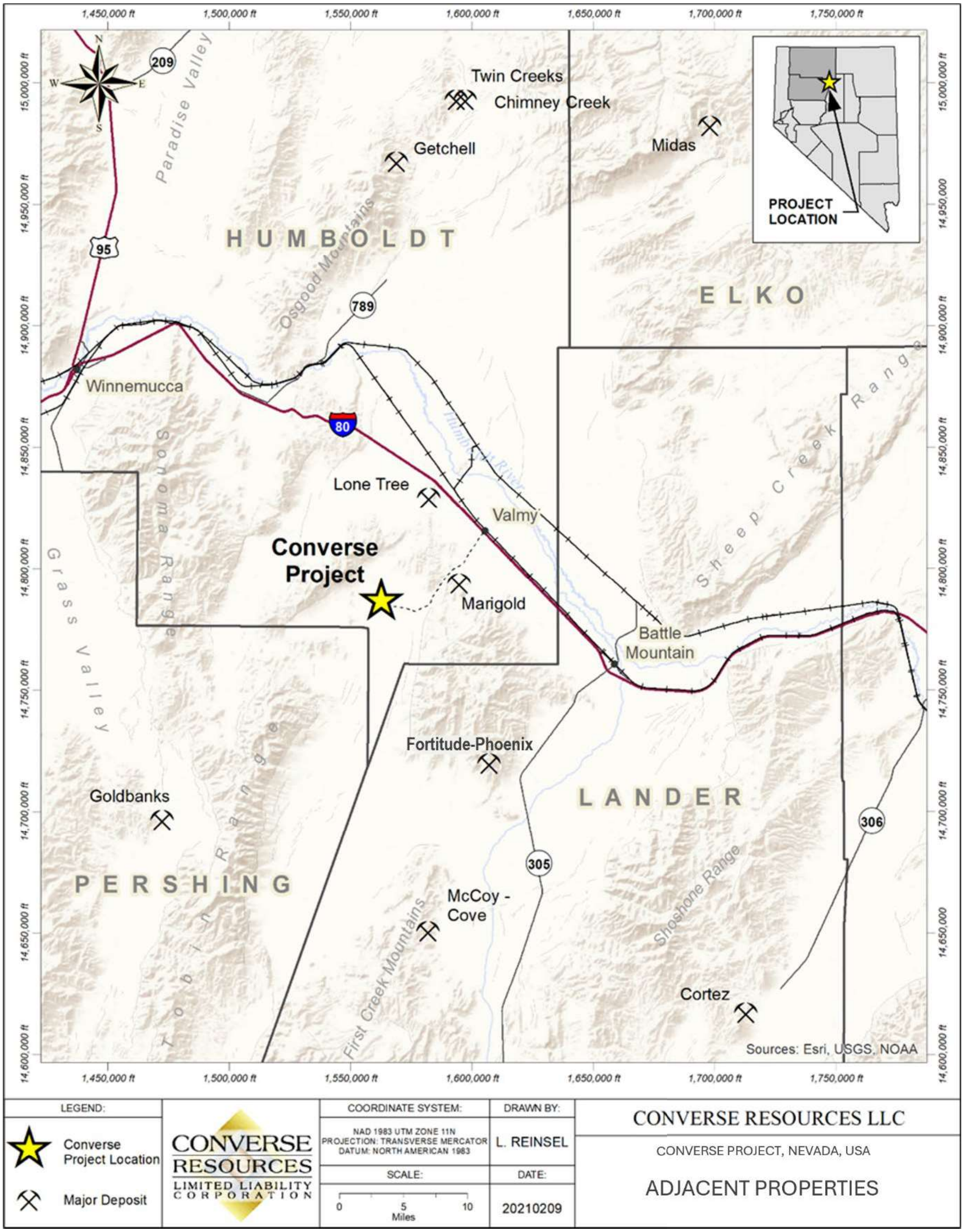
This section discusses mineral properties that occur outside of the Converse Property. The QPs have not visited any of these projects and are unable to verify the information pertaining to mineralization on the adjacent properties, and therefore, the information in the following sections is not necessarily indicative of the mineralization on the Property that is the subject of this Report. The information provided in this section is simply intended to describe examples of the types and tenor of mineralization that exists in the region and is being explored for at the Converse Property.

Table 23.1. Summary of past production, Mineral Reserves and Mineral Resources.

Property	Owner	Years of Production	Gold Produced (Moz)	Proven and Probable Mineral Reserves	Measured and Indicated Mineral Resources	Inferred Mineral Resources
Fortitude-Phoenix[1]	Nevada Gold Mines LLC (NGM)	2006 - Present	Unknown	3.27 Moz at 0.60 g/t Au 830 Mlbs at 0.17% Cu	3.27 Moz at 0.45 g/t Au 910 Mlbs at 0.14% Cu	580 Koz at 0.4 g/t Au 150 Mlbs at 0.1% Cu
Lone Tree[2]	i-80 Gold Corp.	1991 - 2006	4.6	n/a	410 Koz at 1.77 g/t Au	2.76 Moz at 1.69 g/t Au
Marigold[3]	SSR Mining Inc.	1989 - 2023	4.75	2.98 Moz at 0.47 g/t Au	1.47 Moz at 0.44 g/t Au	0.22 Moz at 0.36 g/t Au

[1] Nevada Gold Mines (2022)
[2] Samal (2021)
[3] SLR (2024)
Moz=Million ounces, Koz= Thousand ounces; Mlbs=Million pounds.
Note: The authors of this Technical Report have not verified the mineral reserves and resources reported. However, the resources were prepared by qualified persons in accordance with NI43-101 or S-K 1300 guidelines and the authors have no reason to question their validity. The mineral reserves and resources presented above are not necessarily indicative of the mineralization at the Converse Property.

Figure 23.1. Properties situated adjacent to the Converse Property.



Source: Modified from CRL (2021)

23.1 Fortitude-Phoenix Mine

The Fortitude-Phoenix Mine is located in the Battle Mountain Mining District, approximately 26 km south of Battle Mountain and 20 km south-southeast of the Converse Property (Figure 23.1). In 2019, Newmont and Barrick established the Nevada Gold Mines joint venture encompassing both company's assets across Nevada, including the Fortitude-Phoenix Mine. The joint venture company, Nevada Gold Mines LLC ("NGM"), ownership is split 61.5% Barrick and 38.5% Newmont, with Barrick remaining as the operator.

The Fortitude-Phoenix deposit was originally a prospect located on the Copper Canyon properties which were initially mined by underground methods for copper in the early 1900's after copper and silver were discovered in Copper Canyon in 1864. The Copper Canyon properties were acquired by Duval Corp. ("Duval") in the early 1960s from American Smelting and Refining Co. Duval mined the deposits as an open pit Cu-Au-Ag mine from 1967 to 1984. During the late 1970s with depressed copper prices and rising precious metal prices the mine was gradually converted to a gold producer. Subsequently, gold-silver skarn mineralization was discovered at the Tomboy and Minnie deposits. In 1984, Battle Mountain Gold Co. ("BMGC") was formed to assume the gold mining operations of Duval. BMGC adopted the name Phoenix project for the gold prospects in the area surrounding the old copper mines, particularly in the Copper Canyon area. The Phoenix Mine was acquired by Newmont in January 2001 and commenced production in late 2006 (Doebrich, 1995; Doebrich and Theodore, 1996).

The mineral deposit type(s) discovered to date at the Fortitude-Phoenix Property are high-grade, structurally controlled fault/veins and low-grade, disseminated precious metals skarns and replacements associated with north-trending structures and Tertiary intrusives (Kennedy, 2000). Newmont characterizes Fortitude-Phoenix as a skarn-hosted polymetallic massive sulphide replacement deposit. The Fortitude skarn deposit was formed in calcareous siltstone and conglomerate of the Battle Formation, with the lower zone of mineralization formed in the Antler Peak Limestone located west of and in the hanging wall of the Virgin Fault (Doebrich, 1995). Sulphide mineralization is vertically and concentrically zoned around intrusions and along northward trending structural corridors. The mineral zones roughly correspond to the silicate mineral alteration zones with an inner copper-gold, middle gold-silver, outer lead-zinc-silver-gold and possible distal arsenic-antimony zonation (Guo, 2020).

The Fortitude-Phoenix Mine produces approximately 241,000 oz. gold and 32 million lbs of copper annually. As of December 31, 2021, the Proven and Probable Reserves at the Phoenix Complex were estimated at 169.3 million tonnes at 0.60 g/t Au for 3.27 million ounces gold; 226.1 million tonnes at 0.17% Cu for 830 million lbs Cu and 169.3 million tonnes at 6.43 g/t Ag for 35.01 million ounces Ag (Table 23.1; Nevada Gold Mines, 2022).

The authors of this Technical Report have not visited the Fortitude-Phoenix Mine and have not verified the mineral reserves and resources reported above. However, the resources were prepared by qualified persons in accordance with NI 43-101 guidelines and the authors have no reason to question their validity. The information presented above is not necessarily indicative of the mineralization at the Converse Property that is the subject of this Technical Report.

23.2 Lone Tree Mine

The Lone Tree Mine is located in the Battle Mountain Mining District, approximately 32 km northwest of Battle Mountain and 14 km north-northeast of the Converse Property (Figure 23.1). The Lone Tree Property was acquired in 2021 by i-80 Gold Corporation (i-80) from NGM.

The Lone Tree Mine began operations in 1991 with Newmont as owner and operator. In 2006, Newmont discontinued operations due to increased production costs. Approximately 4.6 million ounces of gold were produced from the Lone Tree Mine during that time (Samal, 2021). The Lone Tree Mine is now closed, with i-80 planning to resume production.

The Lone Tree deposit is characterized as a pluton-related or distal-disseminated Ag-Au deposit (Samal, 2021). The deposit appears to be related genetically to porphyry systems although the gold-silver mineralization may be over one km away from the causative intrusions. For this reason, the deposit has also been historically characterized as a distal disseminated system. Due to complex tectonic and extension in the region, the mineralization substantially different geometric relations to the intrusive centers and is hosted in different stratigraphic horizons (Samal, 2021). Gold mineralization is primarily controlled by structure, with three principal mineralized structural zones and at least one lesser zone currently recognized at Lone Tree. The mineralization is largely structurally controlled along the north-south Powerline Fault. In addition, gold mineralization occurs in three intensely fractured stratigraphic horizons (the Valmy Formation, the Antler Sequence of the Battle Mountain and Edna Mountain Formations, and the Havallah Sequence), similar to other deposits in the region (e.g., Marigold Mine; Samal, 2021).

Hydrothermal breccias, with as much as 25% matrix expansion, host a significant portion of the gold mineralization at Lone Tree. High grade mineralization occurs at fault or fracture intersections, or at jogs in the faults, which form dilatant zones. Gold mineralization is associated with sericitic and argillic alteration of the siliciclastic rocks and dikes and with decarbonatization and iron carbonate alteration of the carbonate-bearing units, as well as in iron-arsenic sulfide and fine grained quartz alteration of all rock types. Gold is hosted in arsenopyrite indicating higher temperatures of mineralization formation in comparison to typical Carlin-type deposits where gold is hosted in arsenian pyrite (Samal, 2021).

Assuming a gold price of $1,650/oz Au and an open-pit cut-off grade of 0.65 g/t Au, the Lone Tree Mine Mineral Resources are estimated at 410,000 ounces of gold Indicated Mineral Resources within 7.2M tonnes grading 1.77 g/t Au, and 2,764,000 ounces of gold Inferred Mineral Resources within 50.7M tonnes grading 1.69 g/t Au (Table 23.1; Samal, 2021).

The authors of this Technical Report have not visited the Lone Tree Mine and have not verified the mineral reserves and resources reported above. However, the resources were prepared by qualified persons in accordance with NI 43-101 guidelines and the authors have no reason to question their validity. The information presented above is not necessarily indicative of the mineralization at the Converse Property that is the subject of this Technical Report.

23.3 Marigold Mine

The Marigold Mine is a open pit gold mine located near the northern limits of the Battle Mountain-Eureka trend of mineral deposits, approximately 23 km west-northwest of Battle Mountain and 8 km to the east of the Converse Property (Figure 23.1). It is owned and operated by SSR Mining Inc. (formerly Silver Standard Resources Inc.).

The deposits at Marigold are distal disseminated silver-gold deposits described as disemminated equivalents of polymetallic vein deposits, with a geochemical signature that includes silver, gold, lead, manganese, copper, zinc, antimony, mercury, arsenic, and tellurium.

The gold deposits at Marigold cumulatively define a northtrending alignment of gold mineralized rock more than eight kilometres long. Gold mineralization at the Marigold Mine occurs in/near fault zones and is finely disseminated within sedimentary and metasedimentary rock units, including quartzite, limestone, basalt, siltstone, and sandstone. Gold deposits at Marigold are described as Carlin-type gold deposits, a type of disseminated, sedimentary rock-hosted gold deposit. Fault structure and lithology, as well as a tertiary influence by fold geometry, controlled the gold mineralizing fluids at Marigold. Gold deposition at Marigold is restricted to fault zones and quartzite dominant horizons within the Valmy Formation, and high permeability units within the Antler sequence (SLR, 2024).

As of September 30, 2023, the Marigold deposit consisted of 103.72 million tonnes Indicated Mineral Resources at an average gold grade of 0.44 g/t containing 1.47 million ounces Au, and an additional 19.09 million tonnes at an average grade of 0.36 g/t Au containing 0.22 million ounces of Inferred Mineral Resources. Total Probable Mineral Reserves at the Marigold Mine are estimated to be 174.8 million tonnes grading 0.47 g/t Au containing 2.98 million ounces Au, including the 0.346 million ounces Au contained within the leach pad inventory (SLR, 2024).

The authors of this Technical Report have not visited the Marigold Mine and have not verified the mineral reserves and resources reported above. However, the resources were prepared by qualified persons in accordance with S-K 1300 guidelines and the authors have no reason to question their validity. The information presented above is not necessarily indicative of the mineralization at the Converse Property that is the subject of this Technical Report.

24 Other Relevant Data and Information

This section is not relevant to this report.

25 Interpretation and Conclusions

The Converse Property is located in the Battle Mountain district, within the Battle Mountain-Eureka Trend, one of the main gold deposit trends in Nevada comprising a northwest-trending belt of precious metal deposits with current reserves and past production exceeding 50 million ounces (oz) gold (Au) (Holley *et al*., 2015).

The updated and current MRE herein has been prepared in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) “Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines” dated November 29, 2019 and reported using the CIM “Definition Standards for Mineral Resources and Mineral Reserves” dated May 14, 2014.

The QPs note the following interpretations and conclusions in their respective areas of expertise, based on the review of data available for this Report.

25.1 Mineral Tenure, Mineral Rights, Access and Permitting

The Property is owned by CRL and comprises 286 unpatented mining claims located on federal land administered by the BLM and five privately-owned tracts of fee land. The total land area covered by the Property is approximately 7,784 acres. One of the five Fee Land parcels is presently leased to a third party.

The Converse Lease requires annual advance royalty payments of US$100,000 at current gold prices. The current land holding costs of the Property total approximately US$169,964 annually (including the annual advance royalty payment due under the Converse Lease, as well as BLM maintenance fees and county recording fees for the Claims and property taxes on the Fee Land).

The Property is subject to a number of net smelter returns (NSR) production royalties payable to RCC on the sale of any minerals from the Property. The RCC Royalty is effectively a blanket 6% NSR royalty on the production of all minerals.

Access to electricity and natural gas supplies is within 5 miles of the Property from SSR Mining Inc.’s (SSR) Marigold Mine and within 9 miles of Highway 80. Water required for exploration drilling is supplied by the nearby Marigold Mine. In 2019, CRL purchased 2,560 acre-feet of irrigation water rights from New Nevada Lands, LLC (Permit 71715 and 71716). Once converted to mining and milling use, the acquired water rights would support the construction and operation of a future mine at the Property. An application requesting a change in the water rights’ point of diversion, place of use and manner of use was submitted to the Nevada State Engineer on October 29, 2020. The change has been granted.

Exploration of the Converse Property is carried out under an Exploration Plan of Operations, NVN065461, approved by the BLM pursuant to Environmental Assessment N20-98-001P and Reclamation Permit #0122 approved by the Nevada Division of Environmental Protection. There is a US$56,330 reclamation bond currently associated with the existing permits.

25.2 Geology and Mineralization

The Property hosts two gold-rich skarn deposits known as the North Redline and South Redline deposits. Skarn assemblages developed subsequent to the emplacement of an intrusive stock. Gold mineralization is observed over an approximate 5,000 by 2,500 ft area and extends from a vertical depth of 18 ft below surface to >2,000 ft.

Mineralization at Converse is spatially associated with all observed alteration and metamorphic assemblages indicating gold deposition occurred throughout skarn development. Gold occurs as liberated grains. Silver and copper mineralization are spatially associated with gold. Sulfide minerals including pyrrhotite, chalcopyrite, pyrite, sphalerite and molybdenite precipitated during development of the prograde and retrograde contact metamorphic assemblages. Galena, arsenopyrite and bismuth-tellurium minerals are also present, but in minor abundances.

Alteration minerals occur mainly as replacements of carbonate minerals in the matrix of calcareous sandstones and also as cross-cutting veinlets. As observed in drill core, much of the prograde skarn replaces bedding planes (dipping shallowly to the west). Three redox zones are observed and include an oxide, transition and sulfide zone. The oxide zone has a variable vertical depth profile ranging from 35 to >500 ft below the base of alluvium.

25.3 Drilling, Sampling and Assaying

The drillhole database as of December 31, 2020 contained 326 drillholes totalling 254,833.6 ft. of drilling Drill types include RC, mud rotary, mixed RC-core and core holes. The drillhole database used for the current MRE consists of 215,123 ft drilled in 249 holes that have provided 31,908 gold assays from intervals totalling 172,325 ft of core or RC chips. Core drilling represents approximately 33% of the total footage and 23% of the total drillholes for the Project.

25.4 QA/QC and Data Verification/Database

Numerous data validation campaigns were conducted on the drillhole data for the Converse Project. The data are considered to be well validated and suitable for Mineral Resource Estimation. Reasonable to good QA/QC data including SRMs, CRMs, blanks and duplicate samples were used for most of the 2003 to 2017 drilling, which represents close to half of the drillholes supporting the MRE.

Mr. Philo Schoeman, M.Sc., P.Geo., Pr.Sci.Nat. visited the Property on December 20, 2020, and verified the collar stake positions for the 2017 drilling program by handheld GPS. On December 21, 2020, Mr. Schoeman collected 15 core samples from 1997, 2013 and 2017 historical drill core at the CRL core storage facility in Lovelock, NV. The Collar locations and the assay results were all within expected ranges.

The APEX QPs have reviewed the adequacy of the Converse Property's drillhole database. It is the opinion of the QPs that the data in the drillhole database is of sufficient quality for the purposes used in this Technical Report, including Mineral Resource Estimation.

25.5 Metallurgical Test Work

Several metallurgical studies including column leach tests have been conducted on Redline deposit material from both the north and south deposits. The most recent work was conducted by KCA in 2018-2019 using the 2017 drill core. Gold recovery forecasts for the P80 <6.4 millimetre (mm) conventional crush heap leach material used in establishing the cut-off grades and pit optimization shell include 77% for oxide, 62% for transition and 50% for sulfide material.

25.6 Mineral Resource Estimate

Gold mineralization at the Property is hosted in two main deposits: Redline North and Redline South. The current MRE presented was completed by Mohan Srivastava, M.Sc., P.Geo. using Micromine 2020.5 software. The current MRE was completed in 2021 using ordinary kriging and was estimated into a regularized block model with blocks 50 × 50 x 20 ft and no sub-blocking. Gold estimation was performed using raw assays with capping based upon which alteration domain each block fell within. Locally varying anisotropy and variography were used to determine search ellipse distances and orientation during the estimation process.

For the purpose of establishing reasonable prospects for eventual economic extraction, the MRE was reported in an optimized pit shell based on a $1,750/oz gold price, with gold recoveries and densities dependent upon the oxidation zone. The conceptual pit shell included assumed mining and processing costs, G&A costs and an overall pit wall angle of 41° in bedrock. The MRE was classified as Measured, Indicated and Inferred Mineral Resources and is reported using a lower gold cut-off of 0.008 oz/ton (0.27 g/t) gold and is provided in Table 25.1 below.

Table 25.1. Converse Mineral Resource Statement based upon Open Pit Heap Leach.

	US UNITS			METRIC UNITS	
	Tonnage	**Au Grade**	**Contained Metal**	**Tonnage**	**Au Grade**
Classification	**(Mtons)**	**(oz/ton)**	**(Moz Au)**	**(Mtonnes)**	**(g/t)**
Measured	209.0	0.018	3.79	189.6	0.62
Indicated	80.6	0.017	1.38	73.1	0.59
Meas.+Ind.	289.6	0.018	5.17	262.7	0.61
Inferred	29.1	0.019	0.55	26.4	0.65

Notes:

1. *Mineral Resources have an effective date of 31 December 2020. Mr. Mohan Srivastava, of RedDot3D Inc., is the Qualified Person responsible for the Mineral Resource estimate.*
2. *Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.*
3. *Mineral Resources are the portion of the Redline North and Redline South deposits that have reasonable prospects for eventual economic extraction by open pit mining method and processed by gold heap leaching.*
4. *Mineral Resources are constrained oxide and sulfide mineralization inside a conceptual open pit shell. The main parameters for pit shell construction are a gold price of $1,750/oz gold, variable gold recovery for oxide, mixed and sulfide mineralization, open pit mining costs of $1.30/ton, heap leach processing costs of $4.80/ton, general and administrative costs of $0.29/ton processed, pit slope angles of 36 ° for alluvium and 41 ° below base of alluvium, and a 6.0% royalty.*
5. *Mineral Resources are reported above a 0.008 oz/ton(0.27 g/t) gold cut-off grade. This is a marginal cut-off grade that generates sufficient revenue to cover conceptual processing, general and off-site costs given metallurgical recovery and long-range metal prices for gold and silver*
6. *Units are imperial tons.*
7. *Numbers have been rounded as required by reporting guidelines and may result in apparent summation differences.*
8. *The QP is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, or political factors that might materially affect the development of these mineral resource estimates.*

Measured and Indicated Resources are estimated at 289.6 million tons grading 0.018 oz/ton (0.61 g/t) gold; and Inferred Resources are estimated at 29.1 million tons grading 0.019 oz/ton (0.65 g/t) gold. The MRE is constrained within an optimized pit shell wireframe that was generated using a gold price of US$1,750/oz, variable gold recovery for oxide, mixed and sulfide mineralization, open pit mining costs of US$1.30/ton, heap leach processing costs of US$4.80/ton, general and administrative costs of US$0.29/ton processed, pit slope angles of 36° for alluvium and 41° below base of alluvium, and a 6.0% royalty.

25.7 Risks

- Data used to inform the block model is historical in nature and incomplete records of original data result in some limitations during verification campaigns. Ongoing improvements should be made to verify the data as additional work is completed.

- The number of bulk density determinations used in the block model are moderate (312). Additional density determinations should be completed and may result in minor changes and impact the tonnage.

- Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is a degree of uncertainty attributed to the estimation of Mineral Resources. Until resources are actually mined and processed, the quantity of mineralization and grades must be considered as estimates only.

- Areas of uncertainty that could materially affect the Mineral Resource Estimate include the following: commodity pricing; interpretations of fault geometries; lithological interpretations on a local scale, including the thickness and amenability of the sedimentary units to host mineralization; geotechnical assumptions related to the open pit and underground mine designs, rock quality and material behavior; additional dilution considerations that may be refinements to open pit and underground mining methods in operation, metal recovery assumptions; product quality assumptions; assumptions as to operating costs used when assessing reasonable prospects of eventual economic extraction; and changes to drill spacing assumptions used to support confidence classification categories.

25.8 Opportunities

- Gold mineralization has been intersected towards the base of the alluvium and if estimated may result in a minor increase in tons and contained gold to the MRE.

- Resource blocks below the reporting pit shell had estimated gold grades above the reporting cut-off and were classified as Measured or Indicated after the smoothing step of the classification procedure, but do not fall within the reporting pit shell. These represent additional upside potential for resource growth if, in the future, new technical or economic parameters allow the conceptual pit to go deeper.

26 Recommendations

26.1 Mineral Resource Estimate

- Review and relog any available drilling materials or photos to replace the back-coded metamorphic intervals used in the estimate;
- Complete an analytical program to further investigate the observed bias in the silver and copper datasets due to different ICP assay methods centered on the use of aqua regia versus four acid digestion.

26.2 Mineral Processing

- Investigate optimal mining scenarios and their impact on life-of-mine capital and operating cost estimates including at higher cut-offs/lower tonnages;
- Evaluate metallurgical sample coverage and grade within the optimized pit;
- Investigate HPGR tertiary crushing flowsheet with the objective of increasing throughput;
- Develop additional tests for HPGR tertiary crushing and evaluate impact on recovery;
- Complete metallurgical test work programs to support finer crush size processing options, including development of additional tests for HPGR tertiary crushing and evaluate impact on recovery;
- Evaluate opportunities to implement SART copper recovery methods and the impact on LOM capital;
- Firm up recovery and consumables vs copper grade correlations based on the additional tests results.

The anticipated cost of the recommended activities will total approximately $1.5M (see Table 26.1) and be carried out over a 12 to 18 month period.

Table 26.1. Proposed budget for Converse activities.

Item	Budget ($M)
Geological Relog and Assays	0.3
Drilling, Sampling and Assaying	1.2
Total	1.5

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28 Certificate of Authors

28.1 Michael Dufresne Certificate of Author

I, Michael Dufresne, M.Sc., P.Geol., P.Geo., do hereby certify that:

1) I am President and a Principal of APEX Geoscience Ltd., Suite 100, 11450 – 160th Street NW, Edmonton, AB T5M 3Y7 Canada.
2) I graduated with a B.Sc. Degree in Geology from the University of North Carolina at Wilmington in 1983 and a M.Sc. Degree in Economic Geology from the University of Alberta in 1987.
3) I am and have been registered as a Professional Geologist with the Association of Professional Engineers and Geoscientists of Alberta ("APEGA") since 1989 and a Professional Geoscientist with the Association of Professional Engineers and Geoscientists of British Columbia ("EGBC") since 2012, Northwest Territories and Nunavut ("NAPEG") since 2017, New Brunswick ("APEGNB") since 2022, and the Association of Professional Geoscientists of Ontario ("PGO") since 2023.
4) I have worked as a geologist for more than 40 years since my graduation from University and have extensive experience with exploration for, and the evaluation of, gold deposits of various types, including epithermal, intrusion related and skarn mineralization in the Western US and Canada.
5) I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
6) I am responsible for, or directly supervised, Sections 1 – 11 and 15 - 28 of the "Technical Report and Mineral Resource Update, Converse Property, Humboldt County, Nevada, USA", with an effective date of November 15, 2024 (the "Technical Report"). I have not visited the Property.
7) To the best of my knowledge, information and belief, the Technical Report contains all relevant scientific and technical information that is required to be disclosed, to make the Technical Report not misleading.
8) I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
9) I am independent of the issuer, the vendor and the Property applying all of the tests in Section 1.5 of both NI 43-101 and 43-101CP.
10) I participated in an independent resource update for the Property from 2020 to 2022 that is the subject of the Technical Report.
11) I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files or their websites.

Effective date: November 15, 2024
Edmonton, Alberta, Canada

"Signed & Sealed"

Michael B. Dufresne, M.Sc., P.Geol., P.Geo.

28.2 Philo Schoeman Certificate of Author

I, Philo Schoeman, M.Sc., P.Geo., Pr.Sci.Nat., do hereby certify that:

1) I am a Senior Project Geologist with APEX Geoscience Ltd., Suite 100, 11450 – 160th Street NW, Edmonton, AB T5M 3Y7 Canada.
2) I graduated with a B.Sc. Degree in Geology from the University of Port Elizabeth in South Africa in 1985, a B.Sc. Honours in Geology from the University of Cape Town in South Africa in 1989 and with a M.Sc. in Geology from Rhodes University in Grahamstown in South Africa in 1996.
3) I am and have been registered as a Professional Natural Scientist, registration number 400121/03 in the Geological Sciences with the South African Council for Natural Scientific Professions since 2003. I am and have been registered as a Professional Geologist with the Association of Professional Engineers and Geoscientists of Alberta since 2013.
4) I have worked as a geologist for more than 33 years since my graduation from University and have been involved in all aspects of mineral exploration and evaluation for gold deposits in South Africa, Argentina, Ghana, Niger, Yemen and Canada.
5) I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
6) I am responsible for Section 12, the Verification and Qualified Person Visit Inspection section of Technical Report titled "Technical Report and Mineral Resource Estimate, Converse Property, Humboldt County, Nevada, USA", with an effective date of November 15, 2024 (the "Technical Report"). I also made contributions to and am jointly responsible for Sections 1, 11, 25, 26 and 28. I personally conducted a site visit to the Converse Property on December 20, 2020.
7) At the effective date of the Technical Report, to the best of my knowledge, information and belief, the report contains all the relevant scientific and technical information that is required to be disclosed, to make the Technical Report not misleading.
8) I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
9) I am independent of the issuer and the Property applying all of the tests in Section 1.5 of both NI 43-101 and 43-101CP.
10) I participated in an independent resource update for the Property from 2020 to 2022 that is the subject of the Technical Report.
11) I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files or their websites.

Effective date: November 15, 2024
Edmonton, Alberta, Canada

"Signed & Sealed"

Philo Schoeman, M.Sc., P.Geo., Pr.Sci.Nat.

28.3 R. Mohan Srivastava Certificate of Author

I, R. Mohan Srivastava, M.Sc., P.Geo., do hereby certify that:

1) I am a geological and geostatistical consultant, an employee of RedDot3D Inc., with my office at #1100 – 120 Eglinton Avenue East, Toronto, Ontario, Canada M4P 1E2.
2) This certificate applies to the Technical Report titled "Technical Report and Mineral Resource Update, Converse Property, Humboldt County, Nevada, USA", with an effective date of November 15, 2024 (the "Technical Report").
3) I hold the following academic qualifications:
   - B.Sc. in Earth Sciences from the Massachusetts Institute of Technology
   - M.Sc. in Geostatistics from Stanford University.
4) I have worked as a geologist, geostatistician and resource estimation specialist since graduation from university in 1979 and have more than five years of experience in mineral exploration, mine development or operation or mineral project assessment that is relevant to my professional degree and area of practice. My relevant experience for the purpose of this Technical Report includes:
   - 1979 to Present - Consultant specializing in mineral resource estimation, reviews and audits for mining projects in their exploration and development phases, including precious and base metals projects in Canada.
   - 2016 to 2021 - Vice President of TriStar Gold Inc., responsible for field programs and technical studies including: mineral tenure issues, historical data compilation and verification, geological modeling and interpretation, prospect identification, drilling, petrophysics, QA/QC of analytical laboratories, public disclosure, and calculation and reporting of significant intervals from exploration drillholes.
5) I have been a Practicing Member (#0547) of the Professional Geoscientists of Ontario continuously since 2003.
6) I meet all the education, work experience and professional registration requirements of a "Qualified Person" as defined in Section 1.1 of National Instrument 43-101.
7) I last visited the Converse project site in 2005.
8) I am solely responsible for Section 14 and contributed to Sections 1, 25 and 26 of this Technical Report.
9) I am independent of the issuer and owner of the property, AxCap Resources.
10) I have worked as a consulting geologist on the Converse Project since 2005.
11) I have read National Instrument 43-101; this Technical Report has been prepared in compliance with this Instrument.
12) At the Effective Date of the Technical Report, to the best of my knowledge, information, and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and sealed in Toronto, Ontario, Canada, on November 15, 2024.

/s/ "R. Mohan Srivastava"

R. Mohan Srivastava (B.Sc., M.Sc., P.Geo.)

28.4 Ray Walton Certificate of Author

I, R. Walton, B.Tech., P.Eng., do hereby certify that:

1) I am President and Principal of Ray Walton Consulting Inc. with my office at Unit 424, Leader Lane, Toronto, ON Canada M5E 0B2.
2) This certificate applies to the Technical Report titled "Technical Report and Mineral Resource Update, Converse Property, Humboldt County, Nevada, USA", with an effective date of November 15, 2024 (the "Technical Report").
3) I graduated with a B.Tech. (Hons) degree in Materials Science and Technology from Brunel University, Uxbridge, Middlesex, in1977.
4) I am and have been registered as a Professional Engineer of Ontario (License No. 90294521) since 1991. I am also a member of the National Canadian Institute of Mining and Metallurgy.
5) I have worked as a metallurgist continuously for more than 47 years since my graduation from Brunel University and have extensive experience in gold, silver and copper metallurgy and extraction processes.
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 past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
7) I am responsible for, or directly supervised and take responsibility for Section 13 and and contributed to Sections 1, 25 and 26 of this Technical Report. I have not visited the Property.
8) At the effective date of the Technical Report, to the best of my knowledge, information and belief, the report contains all the relevant scientific and technical information that is required to be disclosed, to make the Technical Report not misleading.
9) I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
10) I am independent of the issuer and the Property applying all of the tests in section 1.5 of both NI 43-101 and 43-101CP. I participated in an independent resource update for the Property from 2020 to 2022 that is the subject of the Technical Report.
11) I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files or their websites.

Effective date: November 15, 2024
Edmonton, Alberta, Canada

"Signed & Sealed"

Raymond Walton, B.Tech., P.Eng..

Appendix 1 – Unpatented Lode and Placer Mining Claims

Claim Name	BLM Serial No.	Location Date	County Doc. No.	Amended County No.	Claim Type
NIKE 1	NMC476586	2/10/1988	289722	1996-2896	Lode
NIKE 2	NMC476587	2/10/1988	289723	1996-2896	Lode
NIKE 3	NMC476588	2/10/1988	289724	1996-2896	Lode
NIKE 4	NMC476589	2/10/1988	289725	1996-2896	Lode
NIKE 5	NMC476590	2/10/1988	289726	1996-2896	Lode
NIKE 6	NMC476591	2/10/1988	289727	1996-2896	Lode
NIKE 7	NMC476592	2/10/1988	289728	1996-2896	Lode
NIKE 8	NMC476593	2/10/1988	289729	1996-2896	Lode
NIKE 9	NMC476594	2/10/1988	289730	1996-2896	Lode
NIKE 10	NMC476595	2/10/1988	289731	1996-2896	Lode
NIKE 11	NMC476596	2/10/1988	289732	1996-2896	Lode
NIKE 12	NMC476597	2/10/1988	289733	1996-2896	Lode
NIKE 13	NMC476598	2/10/1988	289734	1996-2896	Lode
NIKE 14	NMC476599	2/10/1988	289735	1996-2896	Lode
NIKE 15	NMC476600	2/10/1988	289736	1996-2896	Lode
NIKE 16	NMC476601	2/10/1988	289737	1996-2896	Lode
NIKE 17	NMC476602	2/10/1988	289738	1996-2896	Lode
NIKE 18	NMC476603	2/10/1988	289739	1996-2896	Lode
NIKE 19	NMC476604	2/11/1988	289740	1996-2896	Lode
NIKE 20	NMC476605	2/11/1988	289741	1996-2896	Lode
NIKE 21	NMC476606	2/11/1988	289742	1996-2896	Lode
NIKE 22	NMC476607	2/11/1988	289743	1996-2896	Lode
NIKE 23	NMC476608	2/11/1988	289744	1996-2896	Lode
NIKE 24	NMC476609	2/11/1988	289745	1996-2896	Lode
NIKE 25	NMC476610	2/11/1988	289746	1996-2896	Lode
NIKE 26	NMC476611	2/11/1988	289747	1996-2896	Lode
NIKE 27	NMC476612	2/11/1988	289748	1996-2896	Lode
NIKE 28	NMC476613	2/11/1988	289749	1996-2896	Lode
NIKE 29	NMC476614	2/11/1988	289750	1996-2896	Lode
NIKE 30	NMC476615	2/11/1988	289751	1996-2896	Lode
NIKE 31	NMC476616	2/11/1988	289752	1996-2896	Lode
NIKE 32	NMC476617	2/11/1988	289753	1996-2896	Lode

Claim Name	BLM Serial No.	Location Date	County Doc. No.	Amended County No.	Claim Type
NIKE 33	NMC476618	2/11/1988	289754	1996-2896	Lode
NIKE 34	NMC476619	2/11/1988	289755	1996-2896	Lode
NIKE 35	NMC476620	2/11/1988	289756	1996-2896	Lode
NIKE 36	NMC476621	2/11/1988	289757	1996-2896	Lode
NIKE 37	NMC476622	2/12/1988	289758	1996-2896	Lode
NIKE 38	NMC476623	2/12/1988	289759	1996-2896	Lode
NIKE 39	NMC476624	2/12/1988	289760	1996-2896	Lode
NIKE 40	NMC476625	2/12/1988	289761	1996-2896	Lode
NIKE 41	NMC476626	2/12/1988	289762	1996-2896	Lode
NIKE 42	NMC476627	2/12/1988	289763	1996-2896	Lode
NIKE 43	NMC476628	2/12/1988	289764	1996-2896	Lode
NIKE 44	NMC476629	2/12/1988	289765	1996-2896	Lode
NIKE 45	NMC476630	2/12/1988	289766	1996-2896	Lode
NIKE 46	NMC476631	2/12/1988	289767	1996-2896	Lode
NIKE 47	NMC476632	2/12/1988	289768	1996-2896	Lode
NIKE 48	NMC476633	2/12/1988	289769	1996-2896	Lode
NIKE 49	NMC476634	2/12/1988	289770	1996-2896	Lode
NIKE 50	NMC476635	2/12/1988	289771	1996-2896	Lode
NIKE 51	NMC476636	2/12/1988	289772	1996-2896	Lode
NIKE 52	NMC476637	2/12/1988	289773	1996-2896	Lode
NIKE 53	NMC476638	2/12/1988	289774	1996-2896	Lode
NIKE 54	NMC476639	2/12/1988	289775	1996-2896	Lode
NIKE 55	NMC476640	2/12/1988	289776	1996-2896	Lode
NIKE 56	NMC476641	2/12/1988	289777	1996-2896	Lode
NIKE 57	NMC476642	2/12/1988	289778	1996-2896	Lode
NIKE 58	NMC476643	2/12/1988	289779	1996-2896	Lode
NIKE 59	NMC476644	2/12/1988	289780	1996-2896	Lode
NIKE 60	NMC476645	2/12/1988	289781	1996-2896	Lode
NIKE 61	NMC476646	2/12/1988	289782	1996-2896	Lode
NIKE 62	NMC476647	2/12/1988	289783	1996-2896	Lode
NIKE 63	NMC476648	2/12/1988	289784	1996-2896	Lode
NIKE 64	NMC476649	2/12/1988	289785	1996-2896	Lode
NIKE 65	NMC476650	2/12/1988	289786	1996-2896	Lode
NIKE 66	NMC476651	2/12/1988	289787	1996-2896	Lode

Claim Name	BLM Serial No.	Location Date	County Doc. No.	Amended County No.	Claim Type
NIKE 67	NMC476652	2/12/1988	289788	1996-2896	Lode
NIKE 68	NMC476653	2/12/1988	289789	1996-2896	Lode
NIKE 69	NMC476654	2/12/1988	289790	1996-2896	Lode
NIKE 70	NMC476655	2/12/1988	289791	1996-2896	Lode
NIKE 71	NMC476656	2/12/1988	289792	1996-2896	Lode
NIKE 72	NMC476657	2/12/1988	289793	1996-2896	Lode
NIKE 127	NMC476712	2/15/1988	289848	1996-2896	Lode
NIKE 128	NMC476713	2/15/1988	289849	1996-2896	Lode
NIKE 129	NMC476714	2/15/1988	289850	1996-2896	Lode
NIKE 130	NMC476715	2/15/1988	289851	1996-2896	Lode
NIKE 131	NMC476716	2/15/1988	289852	1996-2896	Lode
NIKE 132	NMC476717	2/15/1988	289853	1996-2896	Lode
NIKE 133	NMC476718	2/15/1988	289854	1996-2896	Lode
NIKE 134	NMC476719	2/15/1988	289855	1996-2896	Lode
NIKE 135	NMC476720	2/15/1988	289856	1996-2896	Lode
NIKE 136	NMC476721	2/15/1988	289857	1996-2896	Lode
NIKE 137	NMC476722	2/15/1988	289858	1996-2896	Lode
NIKE 138	NMC476723	2/15/1988	289859	1996-2896	Lode
NIKE 139	NMC476724	2/15/1988	289860	1996-2896	Lode
NIKE 140	NMC476725	2/15/1988	289861	1996-2896	Lode
NIKE 141	NMC476726	2/15/1988	289862	1996-2896	Lode
NIKE 142	NMC476727	2/15/1988	289863	1996-2896	Lode
NIKE 145	NMC476730	2/15/1988	289866	1996-2896	Lode
NIKE 146	NMC476731	2/15/1988	289867	1996-2896	Lode
NIKE 147	NMC476732	2/15/1988	289868	1996-2896	Lode
NIKE 148	NMC476733	2/15/1988	289869	1996-2896	Lode
NIKE 149	NMC476734	2/15/1988	289870	1996-2896	Lode
NIKE 150	NMC476735	2/15/1988	289871	1996-2896	Lode
NIKE 151	NMC476736	2/15/1988	289872	1996-2896	Lode
NIKE 152	NMC476737	2/15/1988	289873	1996-2896	Lode
NIKE 153	NMC476738	2/15/1988	289874	1996-2896	Lode
NIKE 154	NMC476739	2/15/1988	289875	1996-2896	Lode
NIKE 155	NMC476740	2/15/1988	289876	1996-2896	Lode
NIKE 156	NMC476741	2/15/1988	289877	1996-2896	Lode

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NIKE 157	NMC476742	2/15/1988	289878	1996-2896	Lode
NIKE 158	NMC476743	2/15/1988	289879	1996-2896	Lode
NIKE 159	NMC476744	2/15/1988	289880	1996-2896	Lode
NIKE 160	NMC476745	2/15/1988	289881	1996-2896	Lode
NIKE 163	NMC476748	5/3/1988	289884	1996-2896	Lode
NIKE 164	NMC476749	5/3/1988	289885	1996-2896	Lode
NIKE 165	NMC476750	5/3/1988	289886	1996-2896	Lode
NIKE 166	NMC476751	5/3/1988	289887	1996-2896	Lode
NIKE 167	NMC476752	5/3/1988	289888	1996-2896	Lode
NIKE 168	NMC476753	5/3/1988	289889	1996-2896	Lode
NIKE 169	NMC476754	5/3/1988	289890	1996-2896	Lode
NIKE 170	NMC476755	5/3/1988	289891	1996-2896	Lode
NIKE 171	NMC476756	5/3/1988	289892	1996-2896	Lode
NIKE 172	NMC476757	5/3/1988	289893	1996-2896	Lode
NIKE 173	NMC476758	5/3/1988	289894	1996-2896	Lode
NIKE 174	NMC476759	5/3/1988	289895	1996-2896	Lode
NIKE 175	NMC476760	5/3/1988	289896	1996-2896	Lode
NIKE 176	NMC476761	5/3/1988	289897	1996-2896	Lode
NIKE 177	NMC476762	5/3/1988	289898	1996-2896	Lode
NIKE 178	NMC476763	5/3/1988	289899	1996-2896	Lode
NIKE 179	NMC476764	5/3/1988	289900	1996-2896	Lode
NIKE 180	NMC476765	5/3/1988	289901	1996-2896	Lode
NIKE 181	NMC476766	5/3/1988	289902	1996-2896	Lode
NIKE 182	NMC476767	5/3/1988	289903	1996-2896	Lode
NIKE 183	NMC476768	5/3/1988	289904	1996-2896	Lode
NIKE 184	NMC476769	5/3/1988	289905	1996-2896	Lode
NIKE 185	NMC476770	5/3/1988	289906	1996-2896	Lode
NIKE 186	NMC476771	5/3/1988	289907	1996-2896	Lode
NIKE 187	NMC476772	5/3/1988	289908	1996-2896	Lode
NIKE 188	NMC476773	5/3/1988	289909	1996-2896	Lode
NIKE 189	NMC476774	5/3/1988	289910	1996-2896	Lode
NIKE 190	NMC476775	5/3/1988	289911	1996-2896	Lode
NIKE 191	NMC476776	5/3/1988	289912	1996-2896	Lode
NIKE 192	NMC476777	5/3/1988	289913	1996-2896	Lode

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NIKE 193	NMC476778	5/3/1988	289914	1996-2896	Lode
NIKE 194	NMC476779	5/3/1988	289915	1996-2896	Lode
NIKE 195	NMC476780	5/3/1988	289916	1996-2896	Lode
NIKE 196	NMC476781	5/3/1988	289917	1996-2896	Lode
NIKE 197	NMC476782	5/3/1988	289918	1996-2896	Lode
NIKE 198	NMC476783	5/3/1988	289919	1996-2896	Lode
NIKE 235	NMC476820	2/17/1988	289956	1996-2896	Lode
NIKE 236	NMC476821	2/17/1988	289957	1996-2896	Lode
NIKE 237	NMC476822	2/17/1988	289958	1996-2896	Lode
NIKE 238	NMC476823	2/17/1988	289959	1996-2896	Lode
NIKE 239	NMC476824	2/17/1988	289960	1996-2896	Lode
NIKE 240	NMC476825	2/17/1988	289961	1996-2896	Lode
NIKE 241	NMC476826	2/17/1988	289962	1996-2896	Lode
NIKE 242	NMC476827	2/17/1988	289963	1996-2896	Lode
NIKE 243	NMC476828	2/17/1988	289964	1996-2896	Lode
NIKE 244	NMC476829	2/17/1988	289965	1996-2896	Lode
NIKE 245	NMC476830	2/17/1988	289966	1996-2896	Lode
NIKE 246	NMC476831	2/17/1988	289967	1996-2896	Lode
NIKE 247	NMC476832	2/17/1988	289968	1996-2896	Lode
NIKE 248	NMC476833	2/17/1988	289969	1996-2896	Lode
NIKE 249	NMC476834	2/17/1988	289970	1996-2896	Lode
NIKE 250	NMC476835	2/17/1988	289971	1996-2896	Lode
NIKE 251	NMC476836	2/17/1988	289972	1996-2896	Lode
NIKE 252	NMC476837	4/17/1988	289973	1996-2896	Lode
NIKE 253	NMC476838	2/17/1988	289974	358551 1996-2896	Lode
NIKE 254	NMC476839	2/17/1988	289975	1996-2896	Lode
NIKE 255	NMC476840	2/17/1988	289976	358552 1996-2896	Lode
NIKE 256	NMC476841	2/17/1988	289977	1996-2896	Lode
NIKE 257	NMC476842	2/17/1988	289978	1358553 996-2896	Lode
NIKE 258	NMC476843	2/17/1988	289979	1996-2896	Lode
NIKE 259	NMC476844	2/17/1988	289980	358554 1996-2896	Lode
NIKE 260	NMC476845	2/17/1988	289981	1996-2896	Lode

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NIKE 261	NMC476846	2/17/1988	289982	358555 1996-2896	Lode
NIKE 262	NMC476847	2/17/1988	289983	1996-2896	Lode
NIKE 263	NMC476848	2/17/1988	289984	358556 1996-2896	Lode
NIKE 264	NMC476849	2/17/1988	289985	1996-2896	Lode
NIKE 265	NMC476850	2/17/1988	289986	358557 1996-2896	Lode
NIKE 266	NMC476851	2/17/1988	289987	1996-2896	Lode
NIKE 267	NMC476852	2/17/1988	289988	358558 1996-2896	Lode
NIKE 268	NMC476853	2/17/1988	289989	1996-2896	Lode
NIKE 269	NMC476854	2/17/1988	289990	358559 1996-2896	Lode
NIKE 270	NMC476855	2/17/1988	289991	1996-2896	Lode
NIKE 271	NMC476856	2/18/1988	289992	1996-2896	Lode
NIKE 272	NMC476857	2/18/1988	289993	358560 1996-2896	Lode
NIKE 273	NMC476858	2/18/1988	289994	1996-2896	Lode
NIKE 274	NMC476859	2/18/1988	289995	358561 1996-2896	Lode
NIKE 275	NMC476860	2/18/1988	289996	1996-2896	Lode
NIKE 276	NMC476861	2/18/1988	289997	358562 1996-2896	Lode
NIKE 277	NMC476862	2/18/1988	289998	1996-2896	Lode
NIKE 278	NMC476863	2/18/1988	289999	358563 1996-2896	Lode
NIKE 279	NMC476864	2/18/1988	290000	1996-2896	Lode
NIKE 280	NMC476865	2/18/1988	290001	358564 1996-2896	Lode
NIKE 281	NMC476866	2/18/1988	290002	1996-2896	Lode
NIKE 282	NMC476867	2/18/1988	290003	358565 1996-2896	Lode
NIKE 283	NMC476868	2/18/1988	290004	1996-2896	Lode
NIKE 284	NMC476869	2/18/1988	290005	358566 1996-2896	Lode
NIKE 285	NMC476870	2/18/1988	290006	1996-2896	Lode
NIKE 286	NMC476871	2/18/1988	290007	358567 1996-2896	Lode
NIKE 287	NMC476872	2/18/1988	290008	1996-2896	Lode

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NIKE 288	NMC476873	2/18/1988	290009	358568 1996-2896	Lode
NIKE 289	NMC476874	2/18/1988	290010	1996-2896	Lode
NIKE 290	NMC476875	2/18/1988	290011	1996-2896	Lode
NIKE 291	NMC476876	2/18/1988	290012	1996-2896	Lode
NIKE 292	NMC476877	2/18/1988	290013	1996-2896	Lode
NIKE 293	NMC476878	2/18/1988	290014	1996-2896	Lode
NIKE 294	NMC476879	2/18/1988	290015	1996-2896	Lode
NIKE 295	NMC476880	2/18/1988	290016	1996-2896	Lode
NIKE 296	NMC476881	2/18/1988	290017	1996-2896	Lode
NIKE 297	NMC476882	2/18/1988	290018	1996-2896	Lode
NIKE 298	NMC476883	2/18/1988	290019	1996-2896	Lode
NIKE 299	NMC476884	2/18/1988	290020	1996-2896	Lode
NIKE 300	NMC476885	2/18/1988	290021	1996-2896	Lode
NIKE 301	NMC476886	2/18/1988	290022	1996-2896	Lode
NIKE 302	NMC476887	2/18/1988	290023	1996-2896	Lode
NIKE 303	NMC476888	2/18/1988	290024	1996-2896	Lode
NIKE 304	NMC476889	2/18/1988	290025	1996-2896	Lode
NIKE 305	NMC476890	2/18/1988	290026	1996-2896	Lode
NIKE 306	NMC476891	2/18/1988	290027	1996-2896	Lode
Nike 331	NMC742663	5/13/1996	1996-6607		Lode
Nike 332	NMC742664	5/13/1996	1996-6608	2020-2337	Lode
Nike 333	NMC742665	5/13/1996	1996-6609	2020-2337	Lode
Nike 337	NMC742669	5/13/1996	1996-6613		Lode
Nike 341	NMC742673	5/16/1996	1996-6617		Lode
Nike 342	NMC742674	5/16/1996	1996-6618		Lode
Nike 343	NMC742675	5/16/1996	1996-6619		Lode
Nike 344	NMC742676	5/16/1996	1996-6620		Lode
Nike 349	NMC742681	5/16/1996	1996-6625		Lode
Nike 350	NMC742682	5/16/1996	1996-6626		Lode
Nike 351	NMC742683	5/16/1996	1996-6627		Lode
Nike 352	NMC742684	5/16/1996	1996-6628		Lode
PUMP #37	NMC740333	3/23/1996	1996-4653		Lode
PUMP #38	NMC740334	3/23/1996	1996-4654		Lode

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PUMP #39	NMC740335	3/23/1996	1996-4655		Lode
PUMP #40	NMC740336	3/23/1996	1996-4656		Lode
PUMP #41	NMC740337	3/23/1996	1996-4657		Lode
PUMP #42	NMC740338	3/23/1996	1996-4658		Lode
PUMP #43	NMC740339	3/23/1996	1996-4659		Lode
PUMP #44	NMC740340	3/23/1996	1996-4660		Lode
PUMP #45	NMC740341	3/23/1996	1996-4661		Lode
PUMP #46	NMC740342	3/23/1996	1996-4662		Lode
PUMP #47	NMC740343	3/23/1996	1996-4663		Lode
PUMP #48	NMC740344	3/23/1996	1996-4664		Lode
PUMP #49	NMC740345	3/23/1996	1996-4665		Lode
PUMP #50	NMC740346	3/23/1996	1996-4666		Lode
PUMP #51	NMC740347	3/23/1996	1996-4667		Lode
PUMP #52	NMC740348	3/23/1996	1996-4668		Lode
PUMP #53	NMC740349	3/23/1996	1996-4669		Lode
PUMP #54	NMC740350	3/23/1996	1996-4670		Lode
PUMP #55	NMC740351	3/23/1996	1996-4671		Lode
PUMP #56	NMC740352	3/23/1996	1996-4672		Lode
PUMP #57	NMC740353	3/23/1996	1996-4673		Lode
PUMP #58	NMC740354	3/23/1996	1996-4674		Lode
PUMP #59	NMC740355	3/23/1996	1996-4675		Lode
PUMP #60	NMC740356	3/23/1996	1996-4676		Lode
PUMP #61	NMC740357	3/23/1996	1996-4677		Lode
PUMP #62	NMC740358	3/23/1996	1996-4678		Lode
PUMP #63	NMC740359	3/23/1996	1996-4679		Lode
PUMP #64	NMC740360	3/23/1996	1996-4680		Lode
PUMP #65	NMC740361	3/23/1996	1996-4681		Lode
PUMP #66	NMC740362	3/23/1996	1996-4682		Lode
PUMP #67	NMC740363	3/23/1996	1996-4683		Lode
PUMP #68	NMC740364	3/23/1996	1996-4684		Lode
PUMP #69	NMC740365	3/23/1996	1996-4685		Lode
PUMP #70	NMC740366	3/23/1996	1996-4686		Lode
PUMP #71	NMC740367	3/23/1996	1996-4687		Lode
PUMP #72	NMC740368	3/23/1996	1996-4688		Lode

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BV 1	NMC937492	10/13/2006	2006-7382		Placer
BV 2	NMC937493	10/13/2006	2006-7383		Placer
BV 3	NMC937494	10/13/2006	2006-7384		Placer
BV 4	NMC937495	10/13/2006	2006-7385		Placer
BV 5	NMC937496	10/13/2006	2006-7386		Placer
BV 6	NMC937497	10/13/2006	2006-7387		Placer
BV 7	NMC937498	10/13/2006	2006-7388		Placer
BV 8	NMC937499	10/13/2006	2006-7389		Placer
BV 9	NMC937500	10/13/2006	2006-7390		Placer
BV 10	NMC937501	10/13/2006	2006-7391		Placer
BV 11	NMC937502	10/13/2006	2006-7392		Placer
BV 12	NMC937503	10/13/2006	2006-7393		Placer
BV 13	NMC937504	10/13/2006	2006-7394		Placer
BV 14	NMC937505	10/13/2006	2006-7395		Placer
BV 15	NMC937506	10/13/2006	2006-7396		Placer
BV 16	NMC937507	10/13/2006	2006-7397		Placer
BV 17	NMC937508	10/13/2006	2006-7398		Placer
BV 18	NMC937509	10/13/2006	2006-7399		Placer
BV 19	NMC937510	10/13/2006	2006-7400		Placer
BV 20	NMC937511	10/13/2006	2006-7401		Placer
BV 21	NMC937512	10/13/2006	2006-7402		Placer
BV 22	NMC937513	10/13/2006	2006-7403		Placer
BV 23	NMC937514	10/13/2006	2006-7404		Placer
BV 24	NMC937515	10/13/2006	2006-7405		Placer
BV 25	NMC937516	10/13/2006	2006-7406		Placer
BV 26	NMC937517	10/13/2006	2006-7407		Placer