NI 43-101 Technical Report Preliminary Economic Assessment

Tonopah Project

Nye County, Nevada

Prepared for:

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1 Executive Summary

1.1 Property Description & Location

The Tonopah property encompasses 8,762 acres in the Ralston Valley, on the northeast side of the San Antonio Mountains in central Nevada, located approximately 20 miles northeast of the town of Tonopah in Nye County (Figure 1-1).

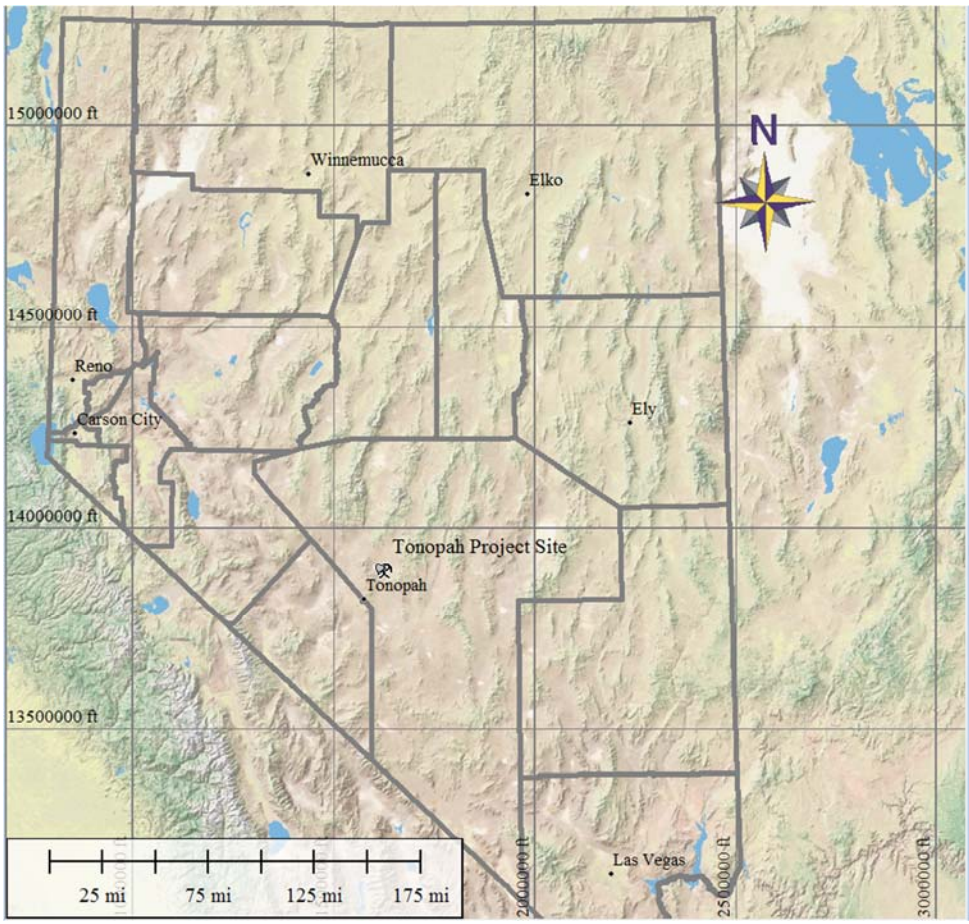

Figure 1-1: Tonopah Location Map

1.2 Ownership

The Project consists of 444 unpatented lode claims covering an area of 8,762 acres. All unpatented claims are 100% controlled by Viva Gold Corporation (Viva); copies of the individual claim notices and location maps are on file with the central BLM office in Reno, Nevada, and with the Nye County Recorder's office in Tonopah, Nevada. The list of claims is included as Appendix C – List of Claims.

A 2% Net Smelter Return Royalty exists on 185 of the unpatented lode mining claims. The surface rights of the unpatented claims located in all claim Sections other than Section 32 are managed by the BLM. Those surface rights located in Section 32 are on lands under private ownership through the Stock Raising Homestead Act (SRHA) of 1916.

1.3 Geology & Mineralization

Surface geology at the Tonopah Property is dominated by valley fill deposits including alluvium, colluvium, sand dunes and playa deposits. With the exception of a single outcrop, the gold-bearing altered and mineralized zones of the Tonopah deposit are masked by these Quaternary deposits. Drilling indicates that the surface sediments are underlain by several rhyolitic to mafic Tertiary volcanics units, which non-conformably overlie Ordovician argillites of the Palmetto formation.

The Tonopah property contains a low-sulfidation epithermal gold system associated with near vertical quartz-adularia-gold veins hosted by Ordovician black argillite of the Palmetto Formation (Opa) and Tertiary rhyolitic volcanics (Tv) and also in association with a discontinuity at the contact with the top of the Palmetto Formation and lower sequence of the Tertiary volcanics. Significant alteration and mineralization is localized within a low-angle zone which includes and often parallels the erosion surface of the Palmetto, as well as several facies in the tertiary volcanics, particularly where veins and mineralized structures intersect this contact zone. It is interpreted that ascending fluids entering the contact zone deposit precious metals in a favorable chemical and textural horizon in the base of the tertiary volcanics.

Structural geology significantly influences the distribution of mineralization and alteration at Tonopah. The Rye Patch fault system is a complex, oblique-slip fault system with numerous northwest trending splays, believed to be associated with north-south trending compressional stress common in the Walker Lane structural trend. Subordinate steeply dipping, north-south striking extension fractures developed between the two bounding strike slip faults. Gold bearing veins occur in a series of en-echelon clusters along a 1.5-mile northwest-trending band of mineralization. Contact related gold mineralization is also seen along this entire band.

Two overlapping mineralized trends have been identified in drilling. The primary trend runs parallel to the west-northwest Rye Patch Fault System, bearing 290-300 degrees over at least 3000 meters, and 500 meters width, and open along strike. Mineralization within this trend is generally within the lower portion of the tertiary volcanics, and sometime in the uppermost argillites, parallel to the Opa/Tv contact and is generally low to moderate grade, from 0.1 ppm to 5 ppm Au.

Secondary extensional fractures range from 345 to 360 degrees strike, are near-vertical in dip, and host veins and hydrothermal breccia's with higher grade mineralization, ranging from 1.0 to over 30 ppm Au. These extensional fracture zones are best represented in drilling in the Discovery and Dauntless zones.

Alteration and mineralization at the Tonopah property are typical of low-sulfidation, volcanic-hosted epithermal gold deposits found elsewhere in Nevada and around the world. The deposit type is characterized by overall low original sulfide content, and quartz-adularia and clay-sericite alteration assemblages, among others. Vein textures are indicative of high level, near surface emplacement and include void fills, crustiform coatings, colloform banding, and comb structures. Similar deposits in Nevada have proven to be economic, including the Midas and Bullfrog deposits. The proximity and similarities of the Tonopah property to other gold deposits does not, on its own, indicate that the Tonopah property should be similarly mineralized.

Vein structures and orientation are best defined in the Discovery Zone, at the center of the project site.

1.4 Exploration Status

Early exploration work was focused on establishing the limits of a large, low-grade gold mineralized system located in the upper portion of the Palmetto formation and in the altered lower units of the tertiary volcanics. Previously issued technical reports (Ristorcelli and Muerhoff, 2002; Ristorcelli, 2003; Gustin and Ristorcelli, 2005) are focused on this interpretation for the deposit.

Midway Gold Corp (MGC) reviewed and compiled subsurface data and targeted exploration on evaluation of higher grade gold mineralization localized around structural zones, quartz veins, and feeders. MGC used this data to evaluate a model focused on the potential for underground mining vein and feeder zones. The previously issued Gustavson 2011 technical report focused on this interpretation of the mineralized system and attempted to model only high grade veins and feeder zones which might be amenable to underground mining.

Viva is focused on understanding both the higher grade and moderate grade portions of the deposit, into a combined model. This consolidated interpretation is more viable because of Viva's reduced royalty structures, which allows for potentially reduced cut-off grades.

1.5 Drilling

A total of 673 holes totaling 91,646 meters have been drilled at the Tonopah Project. 637 of these were completed prior to the acquisition by Viva Gold. Existing drill holes include 12 reverse circulation and auger holes drilled by Midway Gold for hydrology studies, and 12 diamond core holes drilled for geotechnical studies. Approximately half of these drill holes are outside the current resource area, including 100+ holes drilled in the Thunder Mountain area, which is no longer part of the Tonopah Project, and approximately 200 holes drilled west of the current resource area. Drill hole data for the Project is summarized in Table 10-1, and drill hole locations are shown on Figure 10-1. A complete list of drill holes, including year drilled, coordinates, drilling campaign, azimuth and dip, is included as Appendix D.

Viva initiated a drilling program in 2018 designed first to confirm the historical database and secondarily to extend mineralization by targeting areas of inferred which could be upgraded to measured and indicated categories, as well as to provide fresh material for metallurgical test work. Viva initiated an additional drill program in mid-2019 for exploration drilling. Viva drilled a total of 4 Core and 32 RC holes totaling 4,939 meters during the 2018-2019 drilling campaigns.

1.6 Sample Preparation, Analysis & Security

Viva Gold maintains industry standards for drilling, sampling, and assays in its current operations. Historical data were reviewed in detail by independent qualified persons in previous 43-101 reports and are believed to meet industry standards.

1.7 Data Verification

Gustavson considers that the drill data are generally adequate for resource estimation. However, the lack of downhole survey control for many of the historical drill holes may introduce location uncertainty for early sampling at the project.

1.8 Mineral Processing & Metallurgical Testing

Several scoping-level metallurgical studies were undertaken by mining companies from 1990's to 2009 for the Tonopah property. The test work included gravity separation, flotation and cyanidation leaching.

Viva has conducted additional test work to update and expand upon earlier results and can be found in Section 13 of this report. Based on the 2019 test work, this PEA assumes 83% gold recovery for argillite and 58% recovery for volcanics in agglomerated heap leach following 3-stage crushing.

1.9 Mineral Resource Estimate

Gustavson used Leapfrog software to aggregate Palmetto formation, Volcanics, and Gravels within the drill holes for the project, and used the contact points to generate surfaces which represent the contacts between each of the lithologic domains. The contact surface for the top of the Palmetto shows a steep incline to the north which runs generally parallel to the regional Walker Lane Trend, along with two conjugate offsets which correspond to the Discovery and Dauntless zones identified in drilling.

Block grade estimation was completed using Datamine RM software. Grade estimates use ordinary kriging, with an indicator model used to determine impact of the conjugate zone on each block area.

Estimation of block grades uses a two-pass indicator estimation in order to accommodate the two overlapping grade populations and orientations. An indicator model is estimated to determine the level of influence of the higher-grade conjugate zone mineralization, to restrict the influence of this higher-grade mineralization to the shorter variogram range displayed for the higher grade zone, and to honor the geometry of the conjugate zone by orienting search and kriging parameters for the conjugate estimate according to the variography of the zone.

Each grade estimate uses a single pass, with a minimum of 6 and a maximum of 20 2- meter composites used to estimate grades (and IND%). A maximum of 4 composites are used per drill hole, thus requiring at least two drill holes to contribute to each block estimate. Classification is flagged in three passes, with the same data requirements for each pass, but variable ranges, thus SVOL 1 (measured) classification requires a minimum of 6 composites from 2 drill holes within a 17 x 20 x 8 meter search ellipse. Estimates use dynamic anisotropy to orient the PZD and Classification search ellipses parallel to the Palmetto / Volcanic contact, Indicator and CZD searches are oriented with a strike of 170 and a dip of 90 degrees.

Thomas C. Matthews, MMSA-QP, of Gustavson Associates is the Qualified Person with responsibility for the mineral resource estimation in Table 1-1. Resources do not have modifying factors or dilution applied. Gustavson is of the opinion that the resources presented reasonably represent the in-situ resources modeled for the deposit using all available data as of the April 29, 2020 effective date of this report. Resources are presented at a 0.250 g/t (ppm) cutoff grade in argillite and 0.200 g/t (ppm) in volcanics, and inside a $1600/oz Au pit optimization shell. The reporting cutoff grade and constraining the resource inside an optimization shell and two tests for the 'reasonable prospects for economic extraction'.

Table 1-1: Mineral Resource Estimate, Tonopah Project, 2020

Classification	Tonnes (x1000)	Gold Grade grams/tonne	Contained Gold (ozt)
Measured	3,930	1.14	141,000
Indicated	8,900	0.65	185,000
Measured and Indicated	**12,830**	**0.79**	**326,000**
Inferred	8,400	0.67	181,000

Mineral Resources are not Mineral Reserves and have not been demonstrated to have economic viability. There is no certainty that the Mineral Resource will be converted to Mineral Reserves. The quantity and grade or quality is an estimate and is rounded to reflect the fact that it is an approximation. Quantities may not sum due to rounding.

1.10 Mining and Project Development

The Tonopah Property as planned in this report has a project life of 8 years, consisting of 2 years of pre-production, 5 years of mining and heap stacking, and one year of final gold recovery, pad rinsing and reclamation work. Mining will be done using traditional, open pit surface mining techniques.

Lerchs-Grossmann Pit optimization analysis was performed on the resource model. Using initial estimates of the project and financial parameters an optimization was run and a pit shell was selected to guide the design of open pit. The $1,230/oz Au shell was selected as the basis for the mine design. This particular shell captures most of the value of the project while minimizing the mining of marginally economic material.

A three phased pit was designed, Figure 1-2, and material movements scheduled. The mine schedule is then used to develop costs and the cost model used in the financial analysis. The mine schedule targets approximately 2.5 million tonnes of mineralization per year. Total material movement is constrained by the productivity mining, based on a productivity and haulage study of the project, and the fleet size. The mine schedule is shown in Table 1-2.

Table 1-2: Mine Schedule

Parameter	Unit	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Total
Mineral Movement	kt	2,460	2,620	1,990	2,540	2,870		**12,500**
Grade	g/t	0.976	0.732	0.909	0.640	0.708		**0.784**
Contained Au	kg	2,400	1,920	1,810	1,620	2,030		**9,790**
Waste Movement	kt	14,600	12,800	11,500	10,500	8,400		**57,800**
Total Movement	kt	17,000	15,400	13,500	13,100	11,300		**70,300**
Strip Ratio		5.9	4.9	5.8	4.2	2.9		**4.6**
Recovered Au	kg	1,160	1,510	1,500	1,230	1,290	335	**7,030**

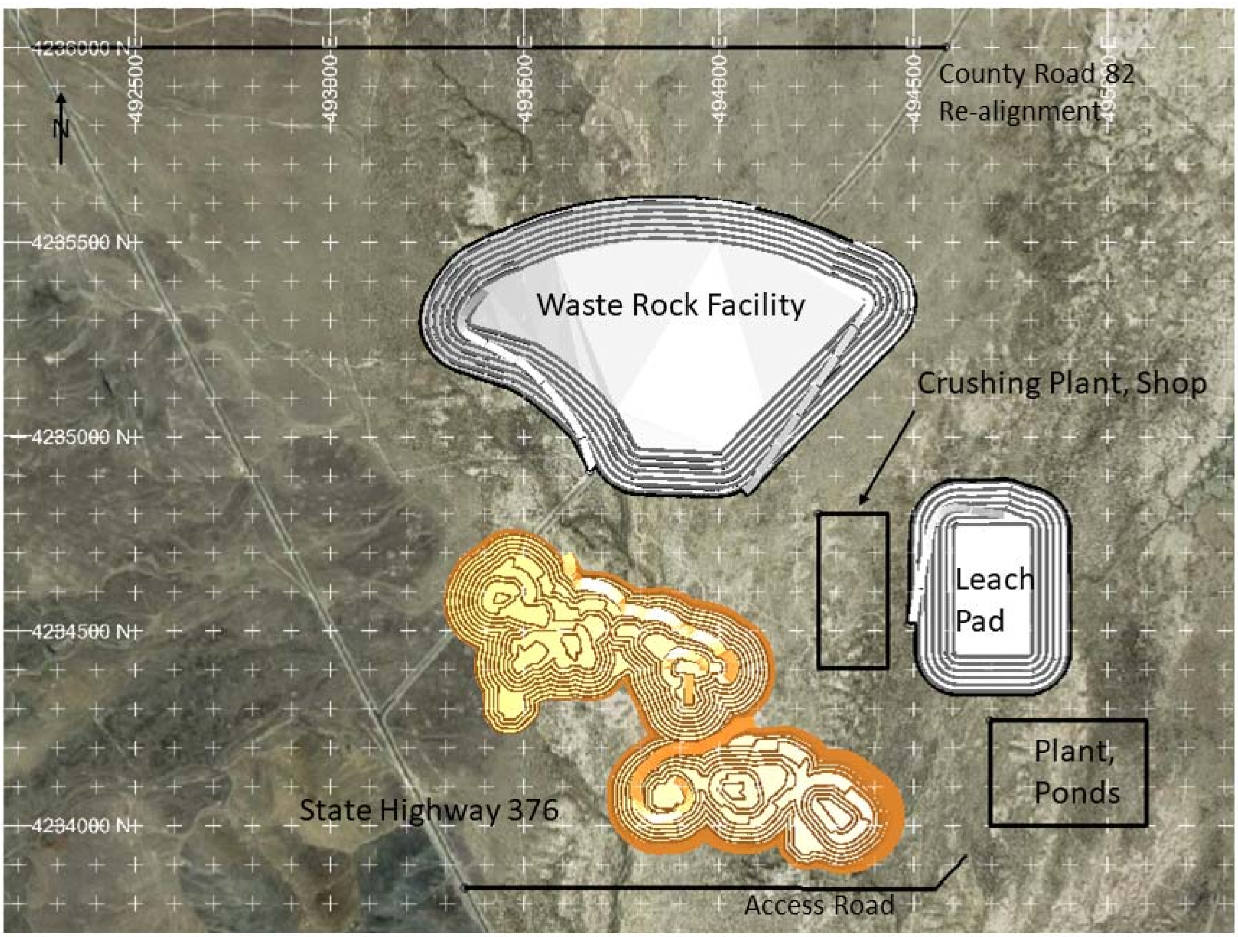

Figure 1-2: Project Layout - Ultimate Pit

The pit design is used to delineate an in-pit resource for the Tonopah project shown in Table 1-3. The in-pit resources shown are included in the economic analysis. Pit constrained mineral resources are not mineral reserves and include inferred resources which are too speculative geologically to have modifying factors applied. There are no mineral reserve estimates for the project at this time. The in-pit mineral resources shown are in-situ, at a gold cutoff grade of 0.20 g/t for argillite mineralization and 0.25 g/tonne for mineralization in volcanics.

Table 1-3: Tonopah Pit Constrained Resource

Classification	Tonnes (x1000)	Gold Grade gram/tonne	Contained Au Ounces (troy)
Measured Resources	1,890	1.261	76,700
Indicated Resources	6,850	0.690	152,000
Measured + Indicated	8,740	0.813	229,000
Inferred Resources	3,740	0.716	86,100

1.11 Recovery Methods

A process for treating the mineralized material was conceptualized based on the scoping level metallurgical test work. The process will consist of three-stage crushing to a nominal 80% passing 10 mesh product, which will be agglomerated and transported to the leach pad using a conveyor system prior to stacking by radial stacker.

The pregnant solution will be processed in a carbon-in-column (CIC) circuit to recover gold and silver.

The planned processing capacity for crushing and heap leach is 7,000 tonnes per day (2.5Mt per year). The planned CIC circuit has a solution processing capacity of 400 cubic meters per hour and is equipped with a 2 tonne carbon strip circuit.

Based on the column leach testing carried out by McClelland Laboratories, Gold recovery from argillite averages 83%, while recovery from volcanics averages 58%.

1.12 Project Infrastructure

The site has excellent logistics and access for exploration, being a short drive from the town of Tonopah, Nevada, with good road access, communications, and access to contractors and labor.

1.13 Environmental Studies

Preliminary data related to environmental and cultural studies have been collected, as detailed in section 20. Permits for exploration activities are discussed in 20.1. No mining permits have yet been sought or secured.

1.14 Capital and Operating Costs

Capital and operating costs for both the mine and processing facilities were developed based on factored and quantity built up estimating techniques and benchmarking similar projects. These costs and equipment requirements were determined from a variety of sources including vendor estimates, the authors' professional experience, production and financial actuals from similar projects in the western United States and third-party mining cost databases. Capital and operating costs are summarized in Table 1-4 and Table 1-5.

Table 1-4: Project Capital Costs

Category	Initial Capital	Sustaining Capital	Total
	($ Millions)		
Mine Development	$7.20	-	$7.20
Mine Mobile Fleet	$4.98	$13.6	$18.6
Process Plant and Heap	$30.5	-$1.05	$29.5
Environmental & Other	$15.2	$2.13	$17.3
Total	**$57.9**	**$14.7**	**$72.6**

Table 1-5: Project Operating Costs

Area	LoM Cost	Average Unit Cost
	($ Millions)	($/tonne processed)
Mining	$90.2	$7.22
Processing	$56.5	$4.52
Site G&A	$8.23	$0.66
Contingency (10%)	$15.5	$1.24
Total	**$170**	**$13.6**

1.15 Economic Analysis

An after tax, discounted cash flow model was developed to assess the economic performance of the Tonopah Project on a PEA basis. This analysis relies on the mining schedule, capital and operating costs, and recovery parameters discussed in the previous sections of the report. The model assumes 100% equity funding, a 5% discount rate, and a gold price of $1,400/oz. The results of the analysis are shown in Table 1-6 and Table 1-7. The analysis presented is a preliminary economic assessment which includes inferred resource and the positive economic outcome presented does not delineate a mineral reserve.

Table 1-6: PEA Economic Results

(USD million) Gold Price ($/oz)	Base Case $1,400
Pre-Tax Economics	
IRR	25%
Cash Flow (Undiscounted)	$69.7
NPV 5% Discount Rate	$43.6
NPV 10% Discount Rate	$25.9
Payback (Years)	2.9
After-Tax Results	
IRR	22%
Cash Flow (Undiscounted)	$60.1
NPV 5% Discount Rate	$36.3
NPV 10% Discount Rate	$20.3

Table 1-7: Project Details

Gold Price ($/oz)	Base Case $1,400
Gold Ounces Sold	226,000
Initial Capital(1)	$58
Sustaining Capital	$16
Avg Cash Cost of Production	$754
All in Sustaining Cost (AISC)	$1,075
Project Life (Years)	6
Total Processed Tonnes (M)	12.5
Average Au Grade (g/t)	0.78
Total Waste Tonnes (M)	57.8
Strip Ratio	4.6
Personnel Employed	135
Average Operating Costs	
Mining Cost ($/t mined)	$1.28
Process Cost ($/t crushed)	$4.52
Gen & Admin Cost	$0.66
Offsite marketing and Refining ($/oz)	$1.50

(1) $1.0 million is included in capital cost to exercise Viva's Option to acquire 1% of the 2% NSR on the project

1.16 Other Relevant Information

Gustavson does not believe there is additional relevant information not disclosed elsewhere in this document.

1.17 Interpretation & Conclusions

1.17.1 Interpretation & Conclusions

Viva has acquired title to the Tonopah project, along with a significant database of technical information, drill data, geologic interpretation, and preliminary metallurgical data. The data are of industry standard quality and can be used for resource estimation for the project.

Viva's work in updating and clarifying the drill database, particularly collar survey locations, improves the overall quality of the database available for resource estimation.

The 2018-2019 drill programs provided confirmation of the historical database, generally intersecting grades and thicknesses of mineralization consistent with the 2018 block model, and providing additional data for geostatistical support of improved resource classification from inferred to measured and indicated.

Additional infill drilling within the main mineralized areas may convert additional inferred resource to measured and indicated, but is unlikely to significantly increase the overall resource quantity. Further drilling should be targeted at extending mineralization along strike on the Opa / Tv contact, particularly where cross faulting or conjugate faulting may provide an environment favorable to higher grade mineralization, or in areas where the Opa/ Tv contact is inferred to be shallow in depth and thus more likely amenable to open pit mining. Additional drilling should target inferred resource within the pit limits, with the objective of upgrading these resources to measured and indicated categories to support feasibility studies.

The Tonopah project contains a significant gold resource with good continuity at relatively low cutoff grades, and with significant contribution from higher-grade zones. The resource as reported is contained within a pit shell and appears amenable to open pit mining methods.

Initial Metallurgical test work shows that the deposit is amenable to cyanide leaching. Initial Column leach test work provides preliminary recovery projections for the Tonopah project that are used in this PEA.

The PEA indicates that at the gold prices considered, the project shows potential to be developed as a mining operation. A sensitivity executed at near-spot prices indicates additional potential for the deposit at higher metal prices. Gustavson recommends that the project proceed to a feasibility study, including appropriate test work to refine the engineering assumptions, with the overall objective of delineating mineral reserves and forming the basis of a production decision.

1.17.2 Risks and Uncertainties

The Tonopah project is subject to risks and uncertainties typical of gold projects, particularly risk in commodity prices and the precious metals equity markets. Lower metals prices or lack of precious metals equity market interest or activity could render the project uneconomic or reduce access to project financing.

Specific risks to the project exploration and subsequent mine development center primarily around water use and non-degradation of waters, cultural resources mitigation, and public road relocation, as discussed in Section 4, Section 24, and Section 20. Each of these risks appears to be manageable; however, there is potential to increase the operating or capital cost for the project, or delay or stop development activities.

The existing exploration data appear to be of high quality, but errors or omissions in the database could potentially reduce the reliability of resource estimates prepared using this information, which could negatively impact the project.

Metallurgical data appears to be of reasonable quality, but it is still preliminary. Incomplete classification of material types or misunderstanding of the representativeness of metallurgical samples could lead to a change in recovery assumptions. Metallurgical recoveries appear to be sensitive to particle size. Further test work in needed to confirm crush sizes for optimal extraction and to refine cost parameters.

This is a preliminary economic assessment, which is based on engineering assumptions related to operating cost, capital cost, recovery, and other engineering inputs. Further test work or analysis may modify these assumptions in ways which negatively impact project economics.

1.18 Recommendations

General Recommendations

Gustavson recommends that ongoing digital database additions/upgrades continue so that the complete database includes all assay data, including gold, silver, and trace element geochemistry, all available primary observational data on lithology and alteration, and appropriate and available metadata about drilling, sampling, and survey. This will improve the reliability and verifiability of the assay database, as well as making alteration and trace element geochemistry available for geological and geometallurgical modeling efforts.

Gustavson recommends that additional specific gravity determinations be made, both for mineral and waste lithologies, to aid in creation of a more robust bulk density model for the deposit.

Gustavson recommends that ongoing drilling campaigns conducted by Viva continue to use the best practices established by previous workers on the project, particularly with regard to sampling and laboratory assay protocols, minimizing potential assay variability caused by coarse gold in the system. Gustavson further recommends that a comprehensive QA/QC program be maintained, including insertion of blanks, standards, and lab duplicates in the sample stream to monitor laboratory performance.

Gustavson recommends that Feasibility level metallurgical testing be completed to support cost and recovery assumptions for further studies. Test work should include PQ core to provide coarse material for comminution studies and to inform more detailed test work at Feasibility level as the project moves forward.

There is an existing inventory of oriented core from various areas on the project, some from peripheral (near-pit-wall) areas, and some from the center of the deposit. Gustavson recommends that the available core be reviewed to determine its utility for geotechnical evaluation of pit slope angles. This review should determine overall slope angles and make recommendations for any additional test work required.

Gustavson recommends that Viva continue advancing the Tonopah Project by completing a Feasibility Study to establish reserves and to clarify the economic potential of the project.

Gustavson recommends that historical data with regard to cyanide shake assay be reviewed as part of the to aid in understanding of possible recovery differences by lithology and alteration type.

Gustavson recommends that exploration be focused on two areas: First, there are areas of inferred mineralization within the PEA pits which should be targeted to confirm grades and to potentially improve classification to measured and indicated. Second, the western section of the resource area has potential for expansion to the west, which would have the potential to expand the resource and to reduce stripping in the western portion of the pit. Exploration drilling should be targeted to step out from the western extents of the estimated resource to determine whether additional resource can be estimated in these areas.

1.18.1 Specific Work Plan:

A proposed drilling program is recommended in two segments. Approximately 1,000 meters of PQ drilling in 4-6 holes are recommended to provide material for metallurgical and comminution studies. Approximately 4,000 meters of RC drilling in 10 -15 holes are recommended to target in-pit inferred material and extensions of the resource zone to the west. Samples from these RC holes could also provide additional fresh material for metallurgical samples.

Metallurgical test work should be completed with the objective of providing information for cost and recovery assumptions to be incorporated into the Feasibility Study, as well as to refine process design criteria.

Long-lead baseline work should be initiated for environmental, water quality monitoring, and archaeological studies to support development efforts.

Existing core and oriented core should be reviewed, and a geotechnical assessment made to propose pit slope angles and to make recommendations for any further geotechnical drilling requirements.

Complete a Feasibility Study to clarify the economic potential of the project, and to declare reserves.

The proposed work plan, including completion of a PFS, metallurgical and exploration drilling, metallurgical test program and ongoing environmental test work, is expected to cost $1.85 million.

1.18.2 2020-2021 Project Budget

Table 1-8: Project Budget

Budget Item	Anticipated Cost	Dependencies
Exploration Drilling and Metallurgical Drill Program	$1,000,000	Drill Targeting
Metallurgical Test Work Program	$300,000	Met Data Review & Sample Availability
Environmental and Archaeological Studies	$200,000	Well-points already installed
Geotechnical Data Review & Recommendations	$50,000	Review of existing core inventory.
43-101 Report – Pre-Feasibility Study	$300,000	Existing Resource Estimate, Detailed Metallurgical Review, Detailed Cost studies
2020-21 Project Budget	**$1,850,000**	

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2 Introduction

2.1 Purpose of Report & Terms of Reference

Viva Gold Corp ("Viva") retained Gustavson Associates, LLC. ("Gustavson") to prepare Preliminary Economic Assessment for the Tonopah Project (formerly the Midway Project) located 20 miles northeast of Tonopah, Nevada. Gustavson previously prepared a May 2019 "Technical Report on Resources for the project for Viva Gold. Gustavson also prepared a March 2018 "Technical Report on Resources" for the project for Viva Gold. Gustavson also prepared a July 2017 "Technical Report on Mineral Exploration Results for the Tonopah Project" for Aintree Resources Inc. ("Aintree"). Aintree changed its name to Viva Gold Corp on January 8, 2018. For consistency, Viva Gold Corp and its predecessor company, Aintree Resources, will be referred to as Viva throughout this document.

The purpose of this report is to update the resource estimate for the project and to prepare preliminary mine plans and an economic assessment to determine whether further investment in the property is warranted.

2.2 Qualifications of Consultants (Gustavson)

The qualified persons responsible for this report is:

- Thomas C. Matthews, Q.P (MMSA), Principal Resource Geologist, Gustavson Associates is a Qualified Person as defined by NI 43-101 and is responsible for sections 1-12, 14, 15, 18, and 22-26 and for the overall content of this report. Mr. Matthews is independent of Viva.
- Christopher Emanuel, SME Member (SME-RM), Senior Mining Engineer, Gustavson Associates is a Qualified Person as defined by NI43-202 and is responsible for sections 19-21, and portions of sections 16 & 17. Mr. Emanuel is independent of Viva.
- Sarah Milne, PE, SME Registered Member (SME-RM), Senior Mining Engineer, Gustavson Associates is responsible for portions of section 16. Ms. Milne is independent of Viva.
- Deepak Malhotra, PhD., SME-RM, President, Resource Development Inc., is a qualified person as defined by NI 43-101 and is responsible for section 13 and portions of section 17. Dr. Malhotra is independent of Viva.

Additional Contributing Authors are:

- Donald E. Hulse, PE, SME Registered Member (SME-RM), Principal Mining Engineer, Gustavson Associates
- Amanda Irons, Geologist, Gustavson Associates
- Todd W. Lewis, Technical Consultant, Lewis Environmental Consulting LLC.
- James Hesketh, CEO of Viva Gold Corp

Mr. Thomas Matthews acted as project manager during preparation of this NI 43-101 Technical Report and visited the site on two occasions. From April 10 to 11, 2017, Mr. Matthews visited the site offices and reviewed drill core and RC chip trays, as well as visiting the claims, where he observed surface geology, including limited outcrops, and observed locations of capped monitoring wells as well as site access and infrastructure. On September 24, 2019, Mr. Matthews visited the property to observe RC drilling conditions during the 2019 drilling campaign. During that visit, Mr. Matthews observed the drill rig, pad conditions, and RC sampling in the main resource area. He also observed surface geology, drill pad conditions and drill hole collars in the Midway Hills area. Mr. Matthews is responsible for the entire content of the technical report. Additional information as to observations from the site visits are detailed in Section 12.

Mr. Lewis contributed text and guidance to Section 20.

Mr. Hesketh contributed text and guidance to Sections 4.2 and 4.3, as well as edits as to form and accuracy and general commentary.

Mr. Hulse and Ms. Irons provided peer review and edits to form and clarity.

Ms. Irons assisted with preparation of databases, resource reporting, statistics, geostatistics, and various tables and graphics.

2.3 Effective Date

The effective date of this report is April 29th, 2020.

2.4 Units of Measurement

Primary data for the project were historically collected in US Commercial Imperial units. A few historical maps are presented in the original units, and are labeled as such. Part of the effort for this report was updating the project to metric units, and maps and cross sections are generally reported in metric units, including grades, which are reported in parts per million (equivalent to grams per metric tonne). For clarity and consistency for TSX reporting, this report uses metric units for grades and tonnages for resource reporting. Gold quantities are generally reported in troy ounces, as this is the unit of trading for gold. Currencies are expressed in constant 2020 US dollars.

3 Reliance on Other Experts

Viva staff provided documentation related to environmental status, land and legal maps, deeds, mineral claims and royalty agreements, which were relied upon to support section 4 of the report.

Mr. Todd W. Lewis of Lewis Environmental Consulting was instrumental in reviewing the current status of environmental and cultural resources in the Tonopah Project area, and drafted sections 4.5 and 4.6 of this report. Gustavson has reviewed the statements and conclusions therein and believes them to be complete and correct according to the records available.

Mr. James Hesketh of Viva Gold Corp was instrumental in reviewing Mineral Titles, Royalty Agreements, and surface rights agreements, and has provided guidance to the authorship of sections 4.2, 4.3 and 4.4, as well as supporting documentation in the form of legal filings for the statements therein. Gustavson has reviewed the statements and documentation and believes them to be complete and accurate according to the records available.

4 Property Description and Location

4.1 Property Description & Location

The Tonopah property encompasses 8,762 acres in the Ralston Valley, on the northeast side of the San Antonio Mountains in central Nevada, located approximately 20 miles northeast of the town of Tonopah in Nye County (Figure 4-1).

The Project site can be found on the U.S.G.S. Henry's Well and Thunder Mountain 1:24,000 scale, 7.5-minute series, topographic quadrangle maps. The geographic center of the property is located at 38°16'N latitude and 117°04'W longitude. Access to the site is provided by State Highway 376, which intersects Nye County Road 82 (Belmont Highway) near the center of the property.

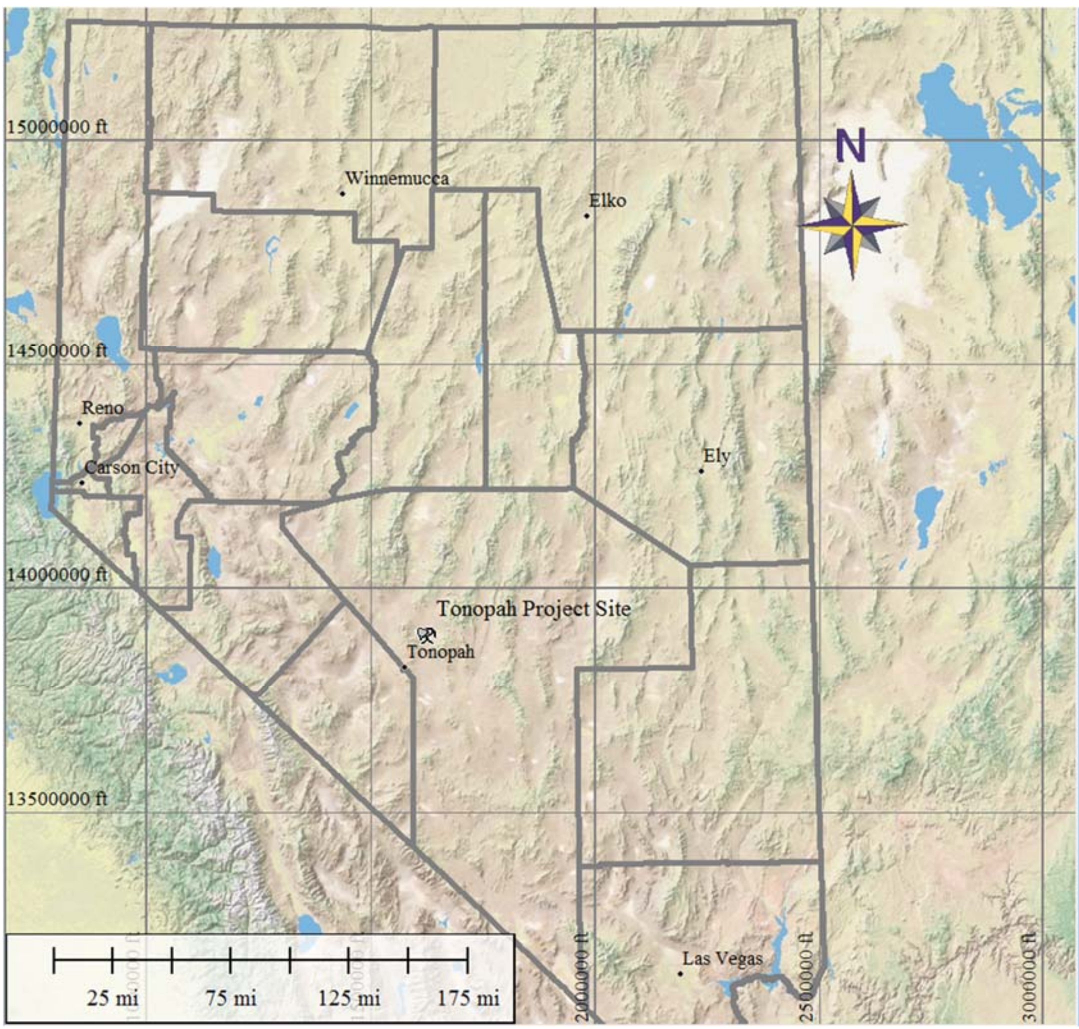


Figure 4-1: Property Location Map

4.2 Mineral Titles

The Tonopah Project mining claims are located in Sections 16 to 21 and 28 to 30 of Township 5 North, Range 44 East (T5NR44E), Mount Diablo Meridian (Figure 4-2). Some claims are also found in Township 5 North, Range 43 East Sections 24 and 25 (T5NR43E).

The Project consists of 444 unpatented lode claims (including 185 royalty claims) covering an area of 8,762 acres. All unpatented claims are 100% controlled by Viva; copies of the individual claim notices and location maps are on file with the central Bureau of Land Management (BLM) office in Reno, Nevada, and with the Nye County Recorder's office in Tonopah, Nevada. The list of claims is included as Appendix E – List of Claims.

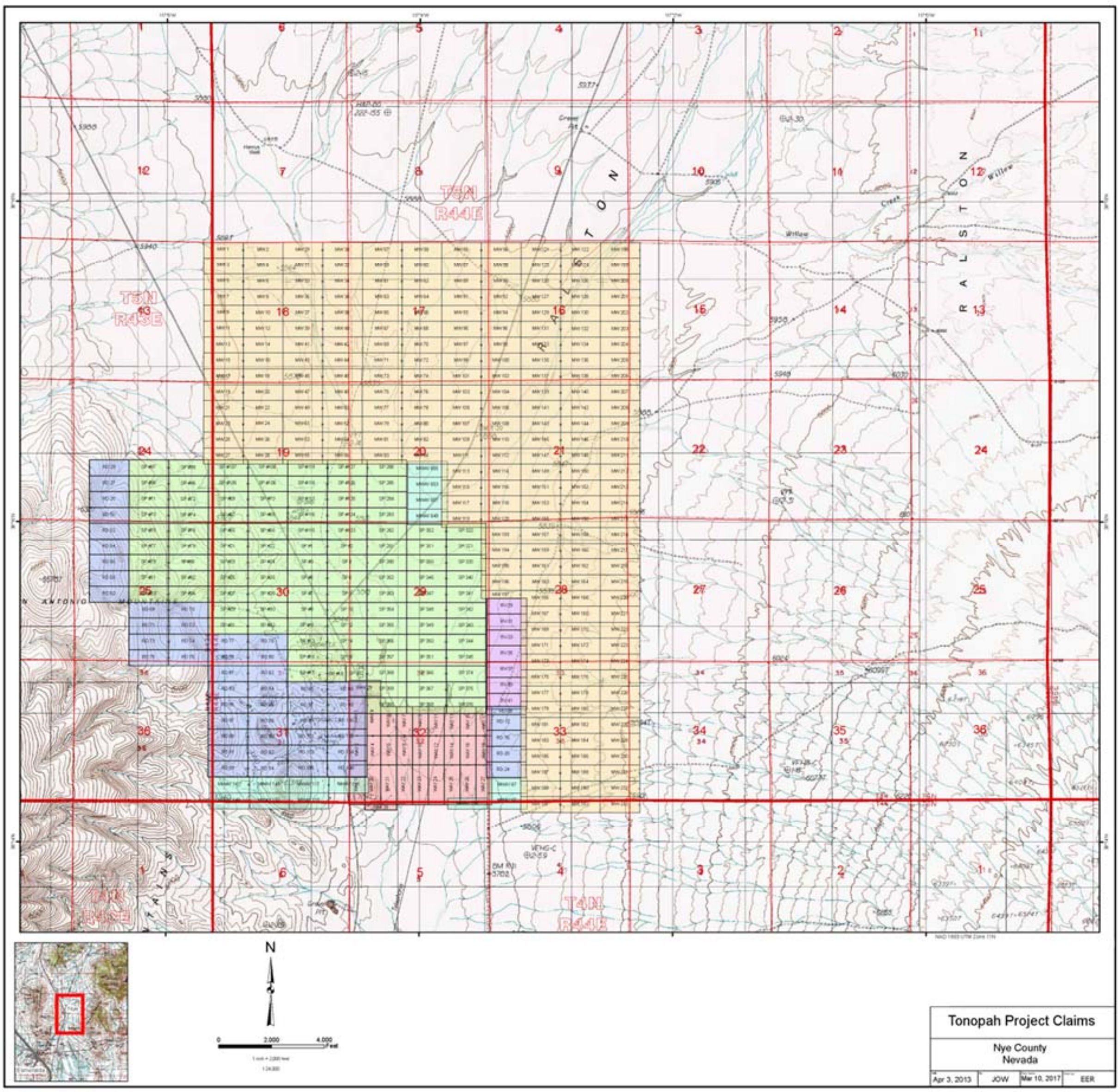


Figure 4-2: Mineral Claim Map

The United States federal law governing locatable minerals is the Mining Law of 1872. This law established a process by which a claimant may locate and extract mineral resources. Location notices for each claim are filed with the BLM and at the courthouse in the county in which the claims are located.

An annual maintenance fee on unpatented claims of US$165 per claim must be paid to the BLM by September 1 at 12 noon each year. A County proof of labor fee of $14.50 for the first claim and $10.50 for each subsequent claim is also assessable on filing of the Federal annual maintenance fees. As of the effective date of this report, Viva is current on all assessment fees.

4.3 Royalties, Agreements, & Encumbrances

The original Midway (now Tonopah) property consisted of 245 claims owned by Paul and Mary Ann Schmidt and Thomas and Linda Patton (Schmidts and Pattons) with each group having a 50% interest. InFaith Community Foundation, a Minnesota nonprofit corporation, now acts as trustee to the Paul and Mary Ann Schmidt 2012 Charitable Trust. InFaith Community Foundation and Thomas and Linda Patton are collectively referred to as the Optionors.

Rex Exploration Corp. (Rex) held an option on the 245 claims under an agreement with the Schmidts and Pattons dated July 2, 2001 and exercised August 5, 2005. Midway Gold Corp (Midway Gold), at the time known as Red Emerald Resource Corp (Red Emerald), held an option on the claims under an agreement with Rex dated August 8, 2001 and exercised October 15, 2002. The original option agreement granted Rex the right to acquire an undivided 100% interest in the Tonopah property by paying the sum of US$3,000,000 to the owners on or before August 15, 2005. US$425,000 was paid between August 2001 and August 2004.

In an amendment dated November 2, 2004, the Schmidts and Pattons granted Rex and Midway Gold the option to purchase the property on payment to the Optionors for an additional US$200,000 (reducing the total purchase price from US$3,000,000 to US$625,000) on or before August 15, 2005. At that time, the property would be transferred to Rex free of all encumbrances except for annual advance royalty payments initiating on August 15, 2006. In addition to these payments, Rex fulfilled the requirement to expend not less than US$1,000,000 on exploration by August 15, 2004. On December 31, 2004, Midway Gold acquired all of the issued and outstanding shares of Rex and assigned the original option agreement to its wholly owned subsidiary MGC on January 1, 2005.

MGC was required to pay to the Optionors an annual advance on royalties that would be payable from commercial production of US$300,000 on or before August 15th of every year until the Project achieved commercial production. These advances were to be credited against future royalties should the Project start commercial production. Once commercial production started, the production royalty would have been based on a sliding Net Smelter Return (NSR) increasing from a 2%NSR at $300 per ounce gold to a maximum 7% NSR at $700 per ounce in increments of 1% for every $100 of price increase.

In 2002, Newmont Mining Corporation entered into a joint venture (JV) agreement with Midway Gold. The JV was terminated in 2004 and Newmont transferred all claims within the agreement's area of interest to Midway Gold, which subsequently assigned them to MGC.

On June 22, 2015 MGC, together with Midway Gold and its affiliated debtors filed petitions under the US Bankruptcy Code (Chapter 11 of Title 11) in US Bankruptcy Court in the District of Colorado. Viva submitted a bid in Bankruptcy Auction to purchase the original property from the debtors. Viva entered into a Royalty Deed Modification and Waiver of Claims Agreement on March 24, 2017 with the Optionors.

The Optionors agreed to support Viva's bid to purchase the property free and clear of the Optionors original royalty and unpaid advanced royalty payment claims against the debtors by terminating the existing royalty agreement with Midway and replacing it with a new royalty agreement negotiated (termed the "Royalty Modification Agreement").

The details of the modified royalty deed and waiver of claims is as follows:

- Upon commercial production the Royalty Modification Agreement granted to the Optionors a 2% NSR over a total of 185 unpatented lode mining claims in the RD08 to RD106 claim group, the RV31 to 41 group, the SP#1 to SP#127 group, the SP4 to SP382 group, the MW26 to 119 group, and the MWAY 649 to 655 group. The claim groups are discontinuous in numerical order.
- Upon commercial production, the Optionors will receive a 2% royalty based on the Net Smelter Return
- Viva paid $25,000 to each of the two royalty holders
- Viva issued 750,000 common shares to each of the two royalty holders
- Viva has the option to buy down 1% of the royalty at any time by paying the Optionors $1.0 million in cash or immediately available funds.

4.4 Surface Rights

The surface rights of the unpatented claims located in Sections 29, 30 and 31 are managed by the BLM. Those surface rights located in Section 32 are on lands under private ownership through the Stock Raising Homestead Act (SRHA) of 1916. This land was transferred to private ownership under SRHA to allow ranchers to privatize lands deemed to be of no value except for livestock grazing and the growing of forage. The federal government retained the subsurface mineral rights, where the right to surface access is granted subject to various conditions under the1872 Mining Law. Viva controls the mineral rights underlying Section 32 as unpatented mining claims. The BLM expects good faith negotiations with the landowners for activities conducted on their surface rights. The Town of Tonopah and two individuals are the owners of the surface rights in Section 32, who allowed the earlier staking of the unpatented claims by agreement.

All surface infrastructure and disturbance in the conceptual designs for this PEA are located on BLM surface.

4.5 Other Significant Factors & Risks

Risk factors to exploration and subsequent mine development center primarily around water use and non-degradation of waters, cultural resources mitigation, and public road relocation(s).

Sub-surface aquifers in the Ralston valley are the primary water source for the Town of Tonopah. Tonopah is located on a heavily mineralized regional trend (Walker Lane Trend) that has been well exploited, and where ground waters are naturally impacted by arsenic content inherent in the geology. Elevated arsenic concentration in groundwater creates issues relative to United States Environmental Protection Agency (EPA) and NDEP Bureau of Safe Drinking Water (BSDW) public drinking water supply standards. TPU's wellfield water supply and distribution system had been located entirely downgradient from the project, below the confluence of the Walker Lane and Sweetwater subsurface aquifers. TPU had difficulty meeting both BSDW and EPA arsenic standards with this wellfield. To rectify this issue, TPU in August 2012, drilled two additional water collection wells upgradient and to the north and east of the Project, located entirely in the Sweetwater aquifer. This allowed TPU to cease primary reliance on its prior downgradient wellfield. Pipelines and power lines were extended to support this new water production field, which is meeting all Town water needs, while also meeting EPA and BSDW drinking water standards. TPU, by taking water out of the aquifer ahead of the project location, may help to reduce future dewatering rates for the Project.

With respect to cultural resources matters, Viva's exploration activities are required to meet all Federal and State cultural resources regulations and stipulations.

A third risk factor includes the potential for local relocation of either or both of Nevada State Route (SR) 376 and Nye County Road 82 (Belmont Road) depending on the scope of a future mining project. This will not be an issue during exploration. SR 376 runs proximal to the Project and may not require relocation. Belmont Road crosses the principal area of mineralization in the Project and may be impacted. This risk is viewed more as a cost and time factor than as a threat to the project as both roads are generally very lightly travelled by local traffic, especially Belmont Road. If any road relocation is necessary due to potential mining operations, Viva would work with the Nye County Road Department, and the Nevada Department of Transportation.

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

5.1 Topography, Elevation, Vegetation and Climate

Local terrain at the Tonopah site is gentle to moderate with seasonal streams and broad washes separating the surrounding pediment slopes near the Ralston Valley bottom. In places, seasonal streams have cut deeply incised channels. Elevation at the property ranges from 1750 to 2100 meters above sea level. Vegetation is typical of high-altitude desert in central Nevada, dominated by desert scrub plant species including shadscale, spiny horsebrush, budsage, winterfat, and prickly pear cacti. Sandy hummocks within defined drainage areas are dominated by greasewood, rubber rabbitbrush, quailbush, and bush seepweed. No noxious weeds were observed during the vegetation survey, though a few weedy species (cheatgrass, halogeton, Russian thistle, poverty weed, and mustards) reportedly do exist within the project area (Gustin, et al., 2005; U.S. Bureau of Land Management, 2003).

The local climate is typical for the high desert of central Nevada and the Basin and Range province. Data from the Western Regional Climate Center (WRCC) shows an average of 12.6 cm of total precipitation per year 36.6 cm of average total snowfall. Average temperatures range from 4.5°C in the winter to 17°C in the summer at Tonopah, Nevada, and daytime temperatures commonly exceed 33°C during the months of July and August (WRCC, 2009). Work can be conducted year-round at the property.

Figure 5-1: Aerial Photo of Tonopah Project Area during 2007 Drill Campaign, Looking SW

5.2 Accessibility & Transportation to the Property

Access to the Tonopah Project site is provided by State Highway 376, a paved road that intersects Nye County Road 82 (Belmont Highway) near the center of the project area. It is approximately 20 miles, via paved road, from Tonopah, Nevada to the Tonopah property. The property is accessible year-round.

5.3 Infrastructure and Local Resources

The Tonopah Project is wholly located on Viva land holdings approximately 20 miles northeast of the Town of Tonopah, Nevada, in the Midway (also known as Rye Patch) Mining District.

The town nearest to the project site, Tonopah, Nevada, hosts a population of 2,478 according to 2010 US Census data. Nye County hosts an area population of 43,946 (US Census Bureau, 2010).

Electrical power is available from the Tonopah well field line, approximately 3 miles east of the project area. Previous exploration campaigns used water from a well located on site for exploration. Water rights for exploration, mine production and process efforts will need to be secured. Potential sources for water include purchase of water from TPU's wellfield, and acquisition of excess water rights available near the towns of Manhattan and Round Mountain, up-drainage from the project area.

Logistical support is available in Tonopah, which currently supports the Round Mountain Mine just 30 miles north of the Tonopah Project. The surrounding region has a long history of mining activity, and mining personnel and resources for operations at Tonopah should be available from the local and regional communities.

5.4 Sufficiency of Surface Rights

Surface rights are described in section 4.4. This PEA outlines a mining area and operating footprint which will assist in understanding what surface rights might be needed for eventual development of the project. However, surface rights for eventual development have not yet been secured.

5.5 Infrastructure

The site has excellent logistics and access for exploration, being a short drive from the town of Tonopah, Nevada, with good road access, communications, and access to contractors and labor. Las Vegas, a city of 2 million people with significant construction and manufacturing infrastructure, is located 210 miles southeast of the project via US Highway 95. There are major Komatsu and Caterpillar dealers and supply depots located in Las Vegas, as well as Cat and Komatsu parts depots and mining-specific machine shops in Round Mountain, approximately 30 miles north of the project. Power and water are available, although water rights will need to be acquired.

There are two water wells already on site. Previous hydrological work has been done on the site due to its proximity to municipal water sources as discussed in Section 4.6 of this report. The report by Water Management Consultants Inc. titled "Hydrologic Assessment and projection of Dewatering Requirements" completed in 2008 confirms that pit or underground mine dewatering activities will be required for the Project and that sufficient sub-surface water supply exists in the drainage to meet the needs of both potential production operations and TPU water supply requirements.

A second dewatering study was completed in 2011 by Schlumberger Water Services which established a plan for dewatering prior to shaft or adit development. This report also includes a hydrogeologic model for the mine area.

Viva does not currently envision an underground mining operation for the project, but the data collected for the hydrological studies will be useful in assessing water management needs for open pit mining.

6 History

6.1 Ownership

The original property consisted of 245 privately held claims which were first optioned in the 1970's. Ownership and operation of the property has changed hands a number of times over the years, and a variety of exploration work has been conducted. Midway Gold gained an option on the claims through an agreement with Rex in 2001 and became the sole owner of the property as of December 31, 2004. MGC, a wholly-owned subsidiary of Midway Gold, conducted exploration drilling, sampling, mapping, and geophysics from assignment of the project on January 1, 2005 through suspension of exploration activities in 2015.

On June 22nd, 2015, Midway Gold filed a voluntary petition for relief under Chapter 11 of Title 11of the United States Code in the United States Bankruptcy Court for the District of Colorado. On March 22nd, 2017 the Court issued an order authorizing the sale of the Tonopah Project by Midway Gold to Viva free and clear of liens, claims and interest pursuant to applicable sections of the Bankruptcy Code.

Viva assumed certain royalty and environmental obligations and provided other valuable considerations, including cash payment. Viva also entered into a Royalty Deed Modification and Waiver of Claims Agreement with underlying royalty holders on the Tonopah Project to waive certain claims by the royalty holders against Midway, eliminate advance royalty payments, and to restructure an onerous sliding scale Net Smelter Royalty (NSR) into a flat 2% NSR structure in exchange for cash consideration and shares of the company.

6.2 Exploration History

Mining and exploration have occurred in the vicinity of the Tonopah Project since the early 1900's. The Tonopah property is located in the Tonopah (or Rye Patch) Mining District. While there is no record of historic gold or silver production at the Tonopah Project site, past production has occurred in the Tonopah Mining District to the south and the Manhattan District immediately to the north of the project area.

At least one shaft and several prospect pits exist as remnants of early mining activity at the Tonopah property, but no data or descriptive information associated with that activity is available. The property was held and explored by Houston Oil and Minerals (Houston) from the 1970s through 1984. Three RC holes were drilled at the property prior to 1981, but it is unclear whether these holes were drilled by Houston or some other company.

In 1981, Felmont drilled 96 RC holes in the Thunder Mountain area, southeast of the Tonopah Project area. No further exploration activity was completed until 1986, when Messrs. Patton and Schmidt staked claims covering the Tonopah property and areas to the north and east. In 1988, Messrs. Patton and Schmidt optioned the property to the Coeur d'Alene Mines Corporation (CDA). CDA conducted preliminary geological, geochemical, and geophysical surveys and drilled three RC holes into targets identified from this exploration. The results of the exploration program were inconclusive and CDA dropped their option on the property.

Rio Algom Ltd., in conjunction with Cour d'Alene optioned the property in 1989 and completed a similar exploration program, including 42 RC holes. This program was completed in an area to the north-

northwest of the Tonopah Mine (now called the Midway Hills Area) and yielded a best intersection of 15 ft. of gold mineralization of 0.6 oz/ton.

Kennecott Exploration Company leased the property from Messrs. Patton and Schmidt in 1992. Kennecott drilled 10 holes in the Midway Hills area in 1992 with limited success. Between 1992 and 1996, Kennecott completed four geophysical programs including airborne magnetic, airborne electromagnetic (EM), gravity, and controlled source audio-frequency magnetotelluric (CSAMT) surveys. Based on the geophysics work, Kennecott switched focus to covered targets east of the Midway hills. Kennecott ultimately drilled 132 RC holes and four core holes, identifying the Discovery Zone.

In August 1996, Mr. Jay W. Hammitt developed a polygonal resource estimate associated with the Discovery Zone. Golconda Resources Ltd. drilled nine RC holes in the Thunder Mountain area, also in 1996. Tombstone Exploration and Kennecott formed a JV in 1997, and Tombstone drilled 14 RC holes in several different areas at the Tonopah property. Late in 1997, rights to the Tonopah property were returned to Messrs. Patton and Schmidt.

In 2001, Rex Exploration Corporation negotiated to acquire a 100% interest in Tonopah from Messrs. Patton and Schmidt. At that time, Rex also entered into an option agreement with Red Emerald Resources Corporation, the predecessor to Midway Gold. In 2002, Red Emerald became Midway Gold and Rex became a wholly owned subsidiary. Between May 2002 and September 2002, Midway Gold drilled 19 RC and 50 core holes at the Tonopah Project (Gustin et al., 2005; MGC Resources, 2008).

In September of 2002, Midway Gold entered into a JV agreement with Newmont Mining Corporation under which Newmont was the operator. Between 2002 and 2004, Newmont completed a regional exploration program that included additional geophysical surveys in the form of ground and airborne radiometric, magnetic and EM/Time-delay electromagnetic (TEM), gravity, CSAMT, Induced Polarization (IP)/resistivity and a small-scale self-potential test over the Discovery Zone. During this period, 75 RC and 46 core holes were drilled at the Tonopah Project and Thunder Mountain areas. Metallurgical testing was also conducted during 2002, and the Northwest and Thunder Mountain areas were mapped, and regional rock and stream sediment geochemical surveys were completed. The Midway – Newmont JV was dissolved in 2004.

Between 2004 and 2012, Midway Gold drilled 90 RC holes and 73 core holes at the Tonopah Project. Midway also collected geotechnical data from and conducted hydrological studies on many of these holes. During this period, Midway also dropped the Thunder Mountain claim area and shrank the claim position to the current holdings.

Historic drilling at the Tonopah Project is summarized in Table 6-1, including drilling conducted by Midway in 2002 and from 2005 through 2012.

Viva initiated a drilling program in January 2018 for confirmation drilling for the property, and potentially to gather fresh material for metallurgical studies. This program is discussed in Section 10, below.

Table 6-1: Tonopah Project Historic Drilling (Pre-Viva Gold)

Company	Year	RC		Core		Total Drill Holes	Total (m)
		No.	m	No.	m		
Felmont	1980-1981	92	9,214			92	9,214
Coeur d'Alene	1988	3	328			3	328
Rio Algom	1990-1991	41	6,026			41	6,026
Kennecott	1992-1996	133	20,486	4	553	137	21,039
Bob Warren	1994	3	361			3	361
Golconda	1996 - 1997	9	515			9	515
Tombstone	1997	14	1,980			14	1,980
Midway Gold	2002	20	3,304	49	4,832	69	8,136
Newmont	2002 - 2004	84	12,692	38	8,022	122	20,714
Midway Gold	2005	22	2,739	1	43	23	2,782
	2006	44	6,899	19	1,289	63	8,188
	2007	11	1,436	8	967	19	2,403
	2008			16	1,051	16	1,051
	2011			26	3,970	26	3,970
Total		476	65,980	161	20,727	637	86,707

6.3 Historical Mineral Resource Estimates

A polygonal resource estimate was completed by Jay Hammitt in August 1996. This resource estimate was described as incomplete because it did not include structural, stratigraphic or geologic controls of gold distribution and did not classify the resources (Gustin et al., 2005). A "preliminary resource" of 270,000oz gold dated 1997 was reported in The Nevada Mineral Industry Special Publication MI-1997 and Special Publication 1998 (Tingley et al., 1997; Tingley et al., 1998). This estimate may be the result of Mr. Hammitt's work. This pre-2000 resource was estimated prior to NI 43-101 reporting standards, was not intended for public reporting, and is not considered representative or current. The estimated tons and grade are not available.

In 2005 Gustin and Ristorcelli completed a Mineral Resource Estimate as part of the, "*Updated Summary Report Midway Gold Project, Nye County, Nevada (2005)*". The estimate was based on 195 drill holes which supported a data base of 8,860 composites. The Inferred Mineral Resource totaled 5.526 million short tons at 0.039 opt Au at a 0.01 opt Au cutoff. This estimate was superseded by the Gustavson 2011 estimate based on an underground mining concept.

Gustavson (March 2011) estimated a mineral resource for the Midway (now Tonopah) project of 114,000 short tons at 0.10 opt Au cutoff in the inferred category, with an average grade of 0.3017 opt Au, containing 34,400 gold. This resource was disclosed in a 43-101 report on resources by Midway Gold Corp, the previous owner of the property. The 2011 study focused on estimating resources for a small underground mining target as reflected by the elevated cutoff grade. This estimate is superseded by the current resource estimate disclosed in this report.

Thomas Matthews, an author of this report, prepared a resource estimate for Viva Gold based on the historical drilling databases available as of March 2018. The results of this estimate were reported in the March 2018 "Technical Report on Mineral Resources for the Tonopah Project". This estimate is superseded by the current resource estimate disclosed in this report.

Thomas Matthews, an author of this report, prepared a resource estimate for Viva Gold based on historical drilling and recent drilling conducted by Viva Gold with an effective date of May 15, 2019. The results of this estimate were reported in the May 2019 "Technical Report on Mineral Resources for the Tonopah Project". This estimate is superseded by the current resource estimate disclosed in this report.

Thomas Matthews, the qualified person for resource estimation for this report, has not done sufficient work to classify any of the historical estimates as current mineral resources or mineral reserves. Viva is not treating any of the historical estimates, nor the superseded estimates as current mineral resources or mineral reserves. The current mineral resource estimate for the Tonopah property is disclosed in section 14 of this report.

6.4 Past Production & Mining

There is no record of historic gold or silver production at the Tonopah Project site; however, past production has occurred in the Tonopah Mining District to the south and the Manhattan District immediately to the north of the project area.

Gustavson does not know of any reserves, compliant to US SEC or Canadian National Instrument standards, which have ever been estimated or reported for the property.

7 Geological Setting & Mineralization

7.1 Regional Geology

The Tonopah property is located on the northeast edge of the Walker Lane structural zone, a zone of sub-parallel, right lateral strike-slip faults that separate the Sierra Nevada batholith from the Basin and Range province (Bonham and Garside, 1979). The project area is situated in the Midway Hills and Rye Patch valley in the eastern San Antonio Mountains, and includes a portion of the Ralston Valley. The San Antonio Mountains are regionally capped by Miocene Red Mountain trachyandesite flows which can reach thicknesses of up to nearly 1,000 ft.

Argillite, chert, limestone, and other fine-grained clastic rocks of the Ordovician Palmetto Formation are exposed in outcrop at the eastern foot of the Tonopah Hills, near the western edge of the Tonopah property. Rocks of the Palmetto Formation strike northwest to east-west, and dip moderately to the north and northeast. Valley fill alluvial, colluvial, aeolian, and playa deposits extend east from the eastern foot of the Tonopah hills into Ralston Valley, masking bedrock geology over most of the Tonopah property. A regional geologic map of the area is presented as Figure 7-1.

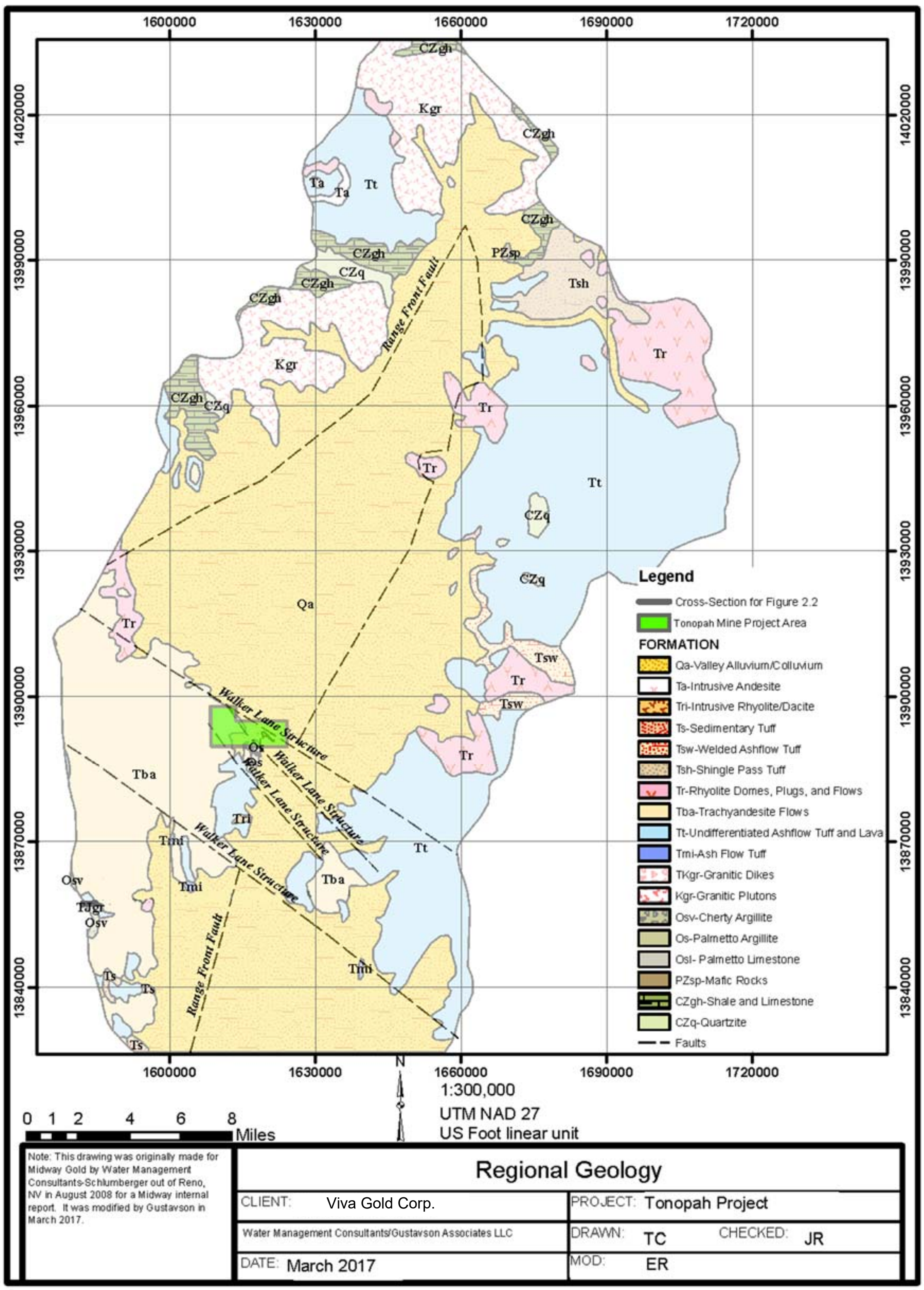


Figure 7-1: Regional Geology

7.2 Local Geology

Local geology in the vicinity of the Tonopah Property is dominated by valley fill deposits including alluvium, colluvium, sand dunes and playa deposits. With the exception of a single outcrop, the gold-bearing altered and mineralized zones of the Tonopah deposit are masked by these Quaternary deposits. Argillite, sandstone, and limestone of the Ordovician Palmetto Formation outcrop in the nearby foothills of the Midway Hills, to the west of the property. These rocks are unconformably overlain by felsic volcanic rocks of the Rye Patch member of the Tonopah Formation (MDA 2003, 2005) or Tombstone Formation (MDA 2002, Panterra 2003).

Intermediate to mafic volcanic flows, presumably of the Red Mountain trachyandesite unit, cap most of the hills to the west of the Tonopah property. These rock types are exposed in a series of north-trending ridges that represent stacked, easterly-directed thrust sheets and low amplitude, open to tight folds. Structure is dominated by the northwest trending Rye Patch fault system, a feature typical of the Walker Lane structural belt.

Rhyolite dikes ranging in width from 1 to 20 meters occur in northwest trending dike swarms in the Palmetto Formation. The dikes are typically clay altered with drusy to chalcedonic quartz veinlets and may host anomalous gold mineralization. Similar felsic dikes have been encountered during drilling.

The current understanding of bedrock geology and the distribution of mineralization and alteration in the Tonopah Project area is based on the results of drilling exploration. A map of the local bedrock geology is presented in Figure 7-2.

7.3 Property Geology

The Tonopah property contains a low-sulfidation epithermal gold system associated with near vertical quartz-adularia-gold veins hosted by Ordovician black argillite of the Palmetto Formation and Tertiary rhyolitic volcanics and also in association with a discontinuity at the contact with the top of the Palmetto Formation and lower sequence of the Tertiary volcanics. Gold bearing veins occur in a series of en-echelon clusters along a 1.5-mile northwest-trending band of mineralization. Contact related gold mineralization is also seen along this entire band. The main altered and mineralized zones are overlain by alluvial gravels, sand dunes, and playa deposits. An idealized stratigraphic column based on drill core logs is presented in Figure 7-3.

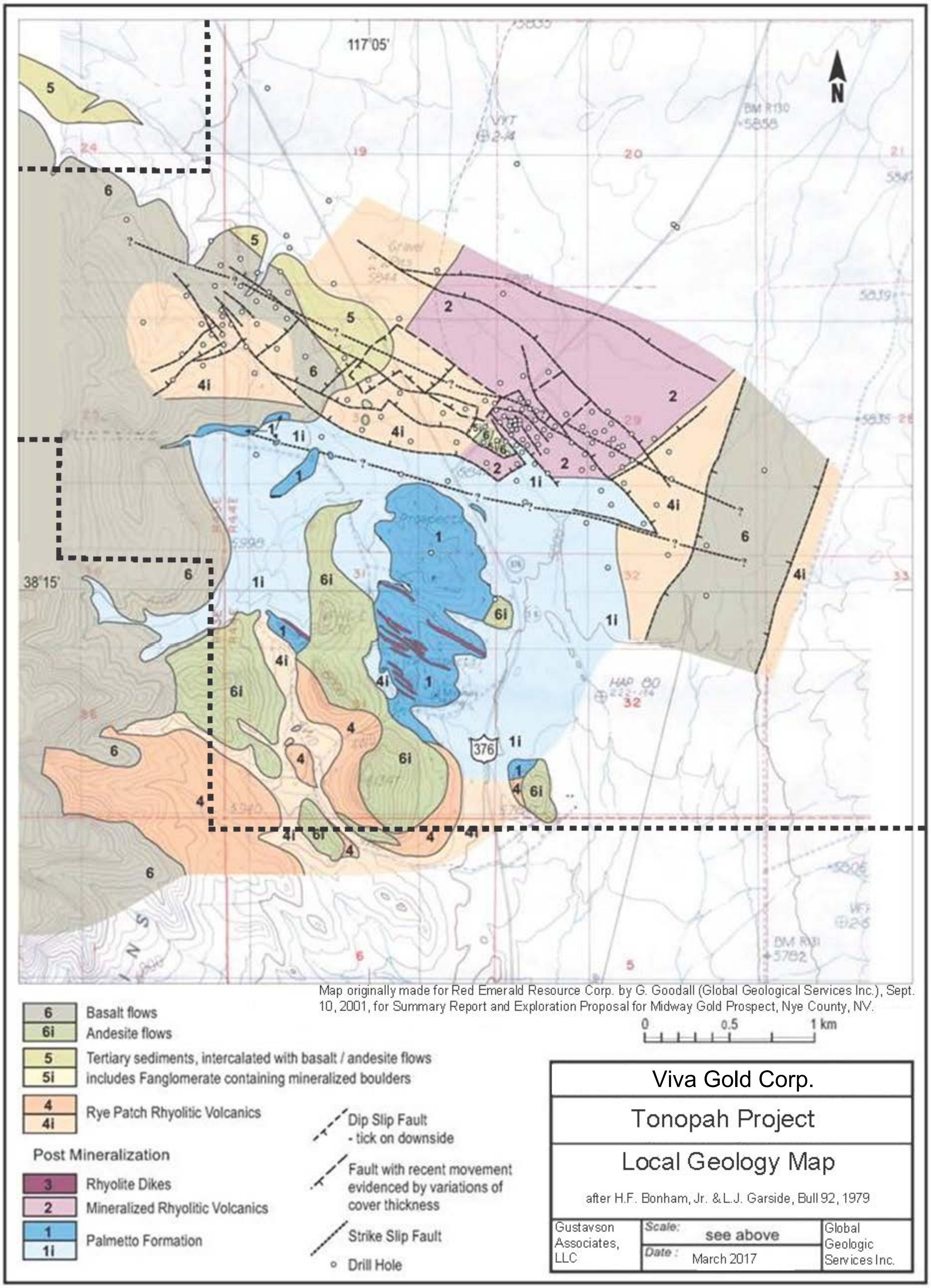
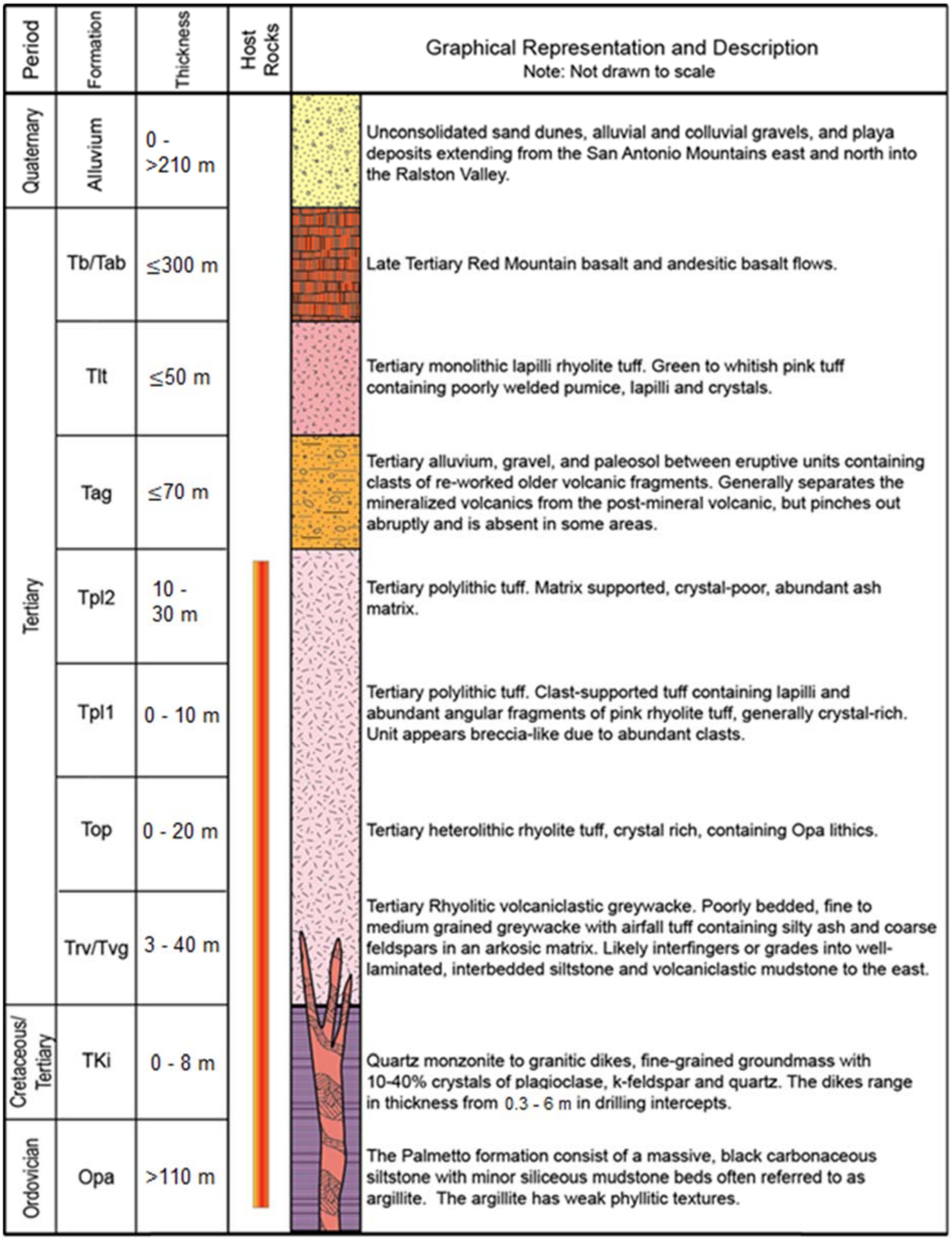


Figure 7-2: Local Bedrock Geology



Period	Formation	Thickness	Host Rocks	Graphical Representation and Description (Note: Not drawn to scale)
Quaternary	Alluvium	0 - >210 m		Unconsolidated sand dunes, alluvial and colluvial gravels, and playa deposits extending from the San Antonio Mountains east and north into the Ralston Valley.
Tertiary	Tb/Tab	≤300 m		Late Tertiary Red Mountain basalt and andesitic basalt flows.
Tertiary	Tlt	≤50 m		Tertiary monolithic lapilli rhyolite tuff. Green to whitish pink tuff containing poorly welded pumice, lapilli and crystals.
Tertiary	Tag	≤70 m		Tertiary alluvium, gravel, and paleosol between eruptive units containing clasts of re-worked older volcanic fragments. Generally separates the mineralized volcanics from the post-mineral volcanic, but pinches out abruptly and is absent in some areas.
Tertiary	Tpl2	10 - 30 m		Tertiary polylithic tuff. Matrix supported, crystal-poor, abundant ash matrix.
Tertiary	Tpl1	0 - 10 m		Tertiary polylithic tuff. Clast-supported tuff containing lapilli and abundant angular fragments of pink rhyolite tuff, generally crystal-rich. Unit appears breccia-like due to abundant clasts.
Tertiary	Top	0 - 20 m		Tertiary heterolithic rhyolite tuff, crystal rich, containing Opa lithics.
Tertiary	Trv/Tvg	3 - 40 m		Tertiary Rhyolitic volcaniclastic greywacke. Poorly bedded, fine to medium grained greywacke with airfall tuff containing silty ash and coarse feldspars in an arkosic matrix. Likely interfingers or grades into well-laminated, interbedded siltstone and volcaniclastic mudstone to the east.
Cretaceous/ Tertiary	TKi	0 - 8 m		Quartz monzonite to granitic dikes, fine-grained groundmass with 10-40% crystals of plagioclase, k-feldspar and quartz. The dikes range in thickness from 0.3 - 6 m in drilling intercepts.
Ordovician	Opa	>110 m		The Palmetto formation consist of a massive, black carbonaceous siltstone with minor siliceous mudstone beds often referred to as argillite. The argillite has weak phyllitic textures.

Figure 7-3: Stratigraphic Column at Tonopah Project *(After Podratz & LeLacheur, 2014)*

Individual lithologic units identified at the project site are described below, from oldest to youngest.

7.3.1 Ordovician Palmetto Formation

The Ordovician Palmetto Formation is the oldest and deepest unit encountered in drill holes at Tonopah. The Palmetto Formation is comprised of siltstone, argillite and chert in the drilled area. Bedding dips moderately, ranging in direction from northeast to northwest in oriented drill core measurements (Rhys, 2003). Pre-Tertiary deformation produced tight to isoclinal folds and a crenulation cleavage in Palmetto rocks; overlying Tertiary volcanic rocks are unaffected

7.3.2 Tertiary Tombstone Formation

Felsic tuffs and volcanoclastic sediments of the Tertiary Tombstone Formation nonconformably overlie the Palmetto Formation. Subsurface mapping and correlation of horizons in drill core or cuttings is difficult due to textural destruction by hydrothermal alteration and rapid lateral facies changes (Rhys, 2003).

7.3.3 Tertiary Intrusive Rocks

Fine to medium grained, and aphanitic felsic dikes and sills intrude the Palmetto and Tombstone Formations, commonly filling faults. These intrusive rocks are altered and mineralized similar to those observed in surface outcrop in the Midway Hills and are likely coeval. Relative age and timing relationships indicate the intrusives are younger or partially coeval with the Tombstone Formation and are syn- to pre-mineral relative to the mineralization/alteration events within the Tombstone.

7.3.4 Tertiary Volcanics (Post-mineral)

A variety of rhyolitic to mafic volcanics unconformably overlie the Tombstone Formation, resting on an interpreted post-mineral paleo-surface. These units have not been studied in any detail.

7.3.5 Quaternary Deposits

Quaternary deposits consisting of a heterogeneous mix of locally derived silt, sand and gravel cover the majority of the Tonopah Property. Mixed dune-playa deposits occur in the central and eastern portion of the property in the lowest areas of the valley floor. Sand dunes are generally small, under 30 meters long and 3 to 4 meters high and are mostly stabilized by vegetation. The mineralized area is buried by 10 to 30 meters of Quaternary cover.

7.4 Structural Geology

Structural geology significantly influences the distribution of mineralization and alteration at Tonopah. The Rye Patch fault system is a complex, oblique-slip fault system with numerous northwest trending splays, believed to be associated with north-south trending compressional stress common in the Walker Lane structural trend. Subordinate steeply dipping, north-south striking extension fractures developed between the two bounding strike slip faults.

Detailed structural studies of bedrock exposures and oriented core from 22 drill holes indicate that alteration and mineralization developed between two moderately northeast dipping faults with right-lateral strike slip movement. Veins and hydrothermal breccias developed along sub-parallel, north-south extension fractures that terminate at the northwest faults.

7.5 Significant Mineralized Zones

Two overlapping mineralized trends have been identified in drilling. The primary trend runs parallel to the west-northwest Rye Patch Fault System, bearing 290-300 degrees over at least 3000 meters, and 500 meters width, and open along strike. Mineralization within this trend is generally within the lower portion of the tertiary volcanics, and sometime in the uppermost argillites, parallel to the Opa/ Tv contact and is generally low to moderate grade, from 0.1 ppm to 5 ppm Au.

Secondary extensional fractures range from 345 to 360 degrees strike, are near-vertical in dip, and host veins and hydrothermal breccias with higher grade mineralization, ranging from 1.0 to over 30 ppm Au. These extensional fracture zones are best represented in drilling in the Discovery and Dauntless zones.

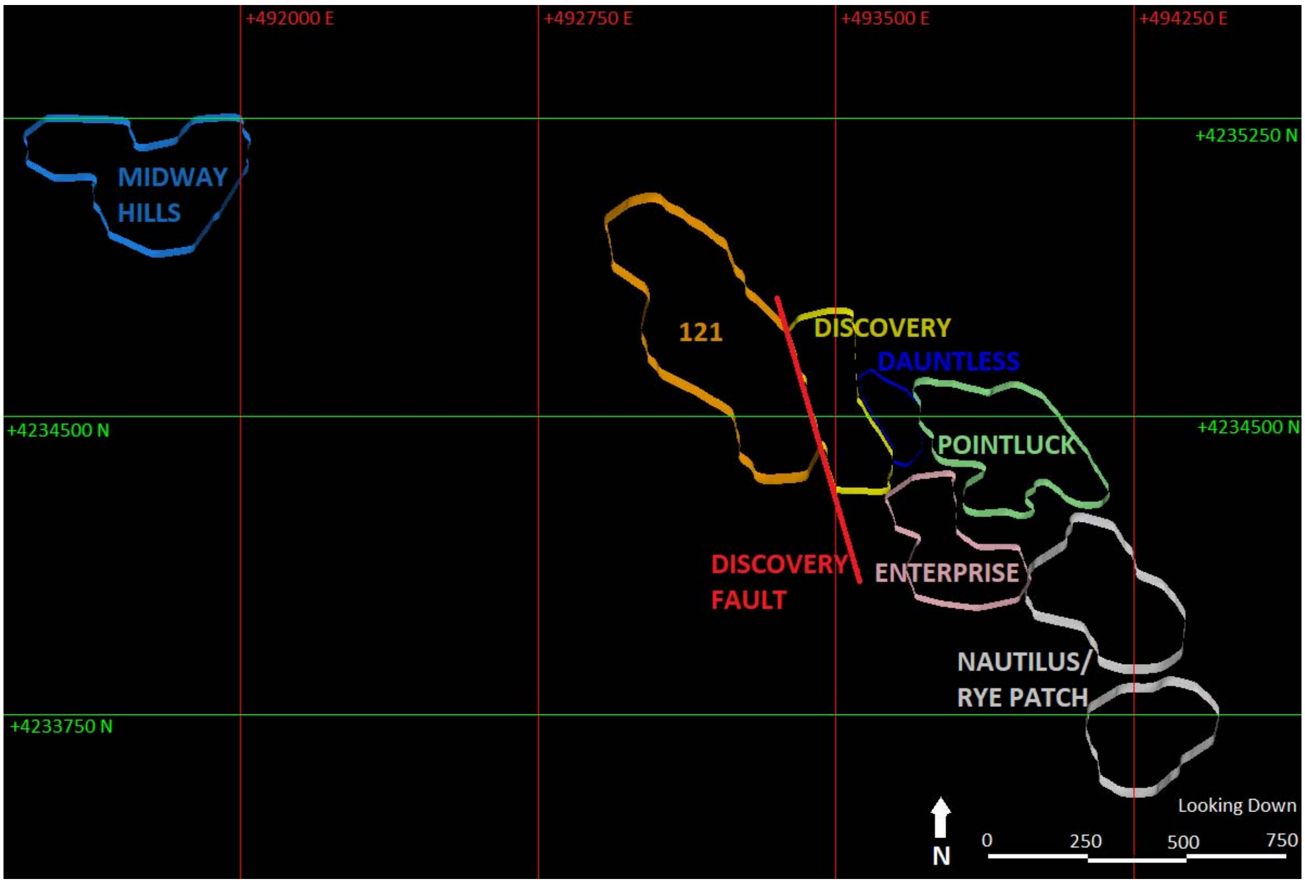


Figure 7-4: Mineralized Zones at the Tonopah project

7.6 Mineralization and Alteration

A discontinuity has been identified in drilling at the top of the Palmetto formation, where tertiary volcaniclastics and ashfall tuffs disconformably overlay the argillite. Significant alteration and mineralization is localized within a low-angle zone which includes and often parallels the erosion surface of the Palmetto, as well as several facies in the tertiary volcanics, particularly where veins and mineralized structures intersect this contact zone.

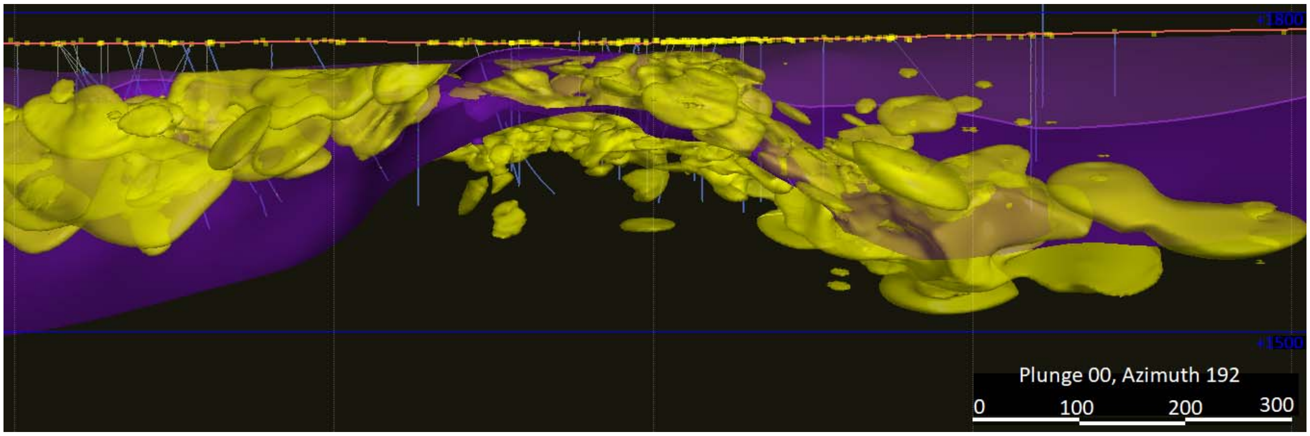


Figure 7-5: Long section looking South (Az 192), showing gold mineralization (gold shell) adjacent to Palmetto argillite contact (purple)

It is interpreted that ascending fluids entering the contact zone deposit precious metals in a favorable chemical and textural horizon in the base of the tertiary volcanics. Figure 7-5 shows the geometry of mineralization along this contact zone. Mineralization within this zone trends parallel to the Rye Patch right-lateral oblique-slip fault system, with a general azimuth of 330 degrees. Higher grade gold mineralization and associated alteration have been identified in a series of north-striking extensional structural zones within the overall mineralized trend, including the Dauntless and Discovery Zones (Figure 7-4). Gold mineralization in the Dauntless and Discovery occurs in zones of massive quartz-adularia alteration in volcanic and volcaniclastic rocks of the Tombstone Formation and in veins, breccias, and silicified faults in both the Tombstone Formation and the underlying Palmetto Formation. Quartz-adularia alteration in the Discovery Zone tends to extend laterally in the Tombstone Formation immediately above and parallel to nonconformable contact with the Palmetto Formation associated low grade disseminated gold mineralization. In the Dauntless Zone, the quartz-adularia forms a funnel-shaped zone that expands upward into the Tombstone Formation above the moderately dipping nonconformity.

Alteration outside of the quartz-adularia zones in the Tombstone Formation is characterized as strong argillic alteration, which persists to the limits of drilling to date. Oxidation is extensive, and only local relict patches of incompletely oxidized pyrite remain in the many of the altered areas.

Significant gold mineralization occurs within the quartz-adularia altered zones, with higher gold grades associated with a variety of siliceous veins, and veinlets including chalcedonic, bladed or drusy quartz, and quartz +/- iron oxide cemented breccias. In the Discovery Zone, to the southwest of the mineralized zones in the Midway Hills and northwest of the Dauntless Zone, there is a strong predominance of steeply dipping north-south trends in mineralized veins and structures of the Tombstone Formation (Rhys, 2003). These structures are interpreted as extension fractures consistent with the structural interpretation described in Section 7.4, Structural Geology.

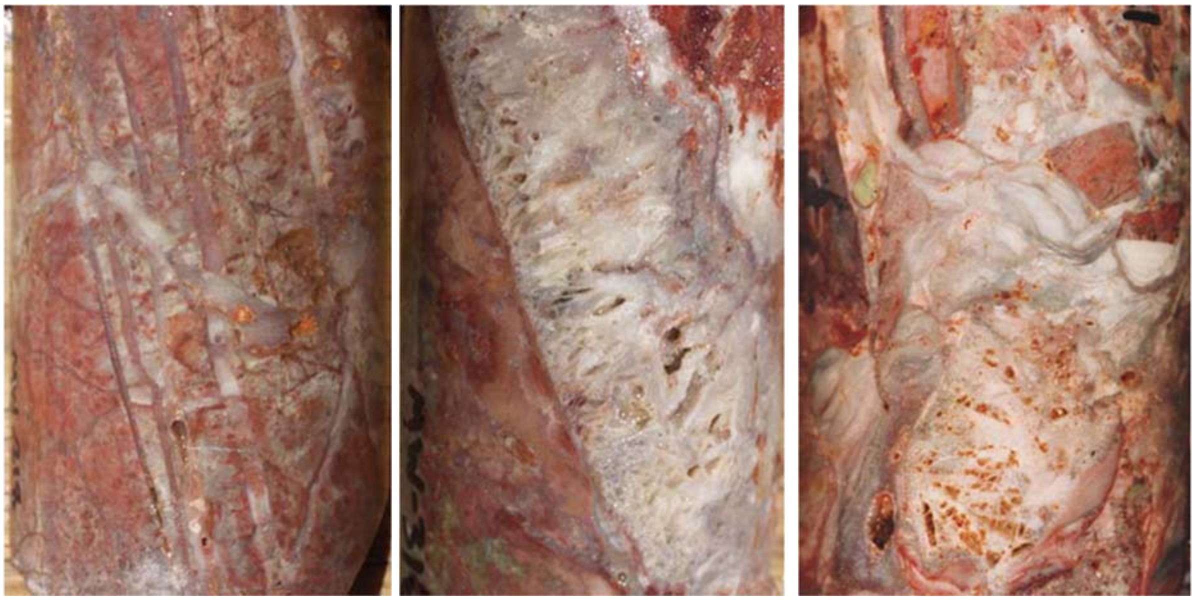


Figure 7-6: Core Photos of Tonopah Vein Structures

The Discovery Zone is the most densely drilled zone at the Tonopah property. Drill holes have intercepted a large number of veins, breccia-veins, and mineralized structures occurring in sub-parallel clusters 3 to 6 meters apart. According to MGC, vein and mineralized structure thicknesses vary from a few centimeters to over 6 meters, averaging 2 meters; Gustavson did not sufficiently review drill core and drill hole data to confirm that estimate. Continuity of veins, vein zones and structures is projected, but not certain, over approximate north-south strike lengths of 60 to 150 meters, and with vertical dimensions that may locally exceed 100 meters. Continuity of gold mineralization and gold grades coincides, approximately, with projections of the veins and structures, but becomes far less certain at progressively higher gold grade cut-offs. At lower cutoff grades, good continuity develops between zones, veins and structures, due largely to lower grade mineralization associated with the discontinuity contact between the Palmetto Formation and the overlying Tombstone volcanics. There is a tendency for well-defined veins in the Palmetto Formation to branch and splay upward into a broader network of veins, vein zones, veinlets in the overlying Tombstone Formation volcanics. Gold mineralization is associated with the veins, breccias and structures, and lower-grade mineralization also spreads laterally in a more disseminated fashion associated with quartz-adularia alteration in the Tombstone volcanics. The system remains open at depth in the Palmetto Formation for lack of sufficient deep drilling.

Visible gold is commonly observed in and along the edges of veins, is frequently associated with hematite, and occurs locally in coarse form. Dendritic gold has been observed in core. Examples of visible gold from the Tonopah property are shown in Figure 7-7.

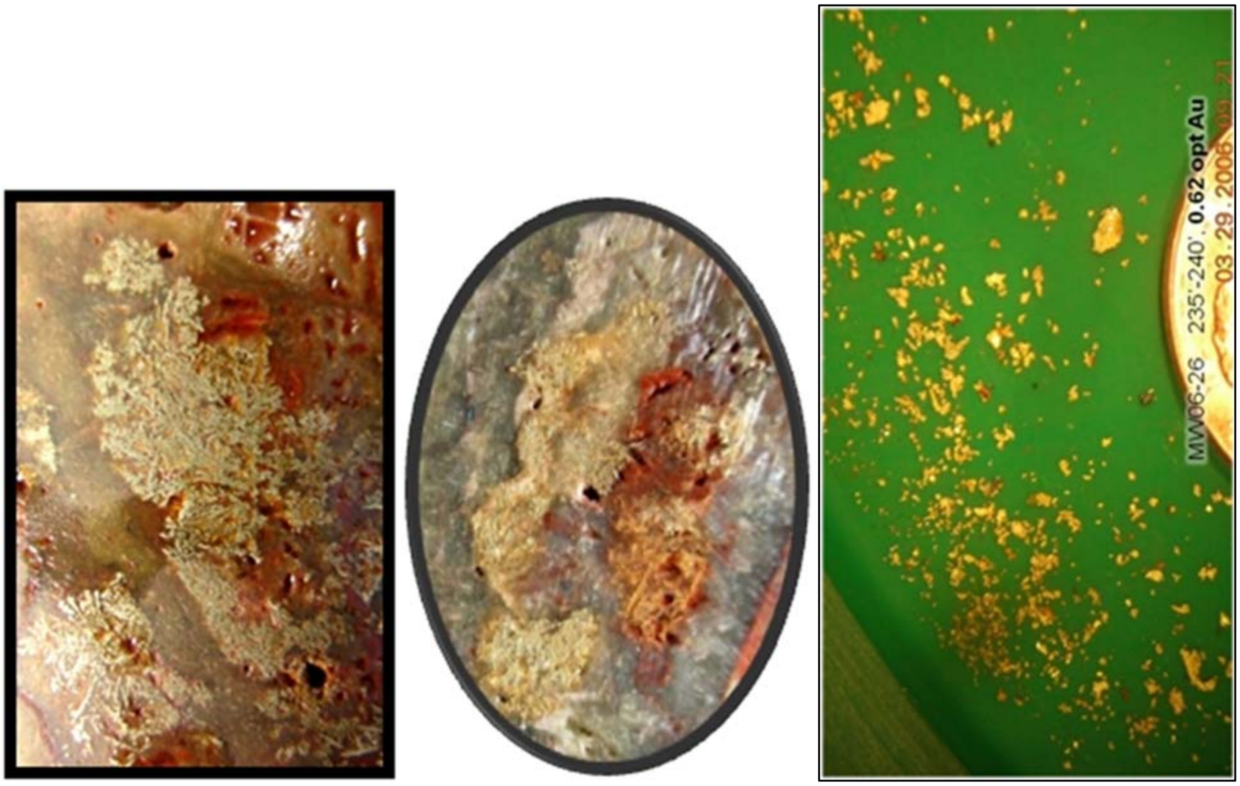


Figure 7-7: Visible Gold from Tonopah Project

Rhys (2003) documents the vertical sequence of veining in the Tombstone Formation:

"Within well mineralized portions of the Discovery Zone, a vertical sequence of veining is frequently apparent in the otherwise massive, intense K-feldspar-quartz alteration zone developed in the Tertiary sequence. High in the zones of K-feldspar-quartz alteration, veinlets are generally rare, but when present, are composed of opaline to chalcedonic quartz, locally with fine-grained drusy quartz lined cavities. Beneath this, significant Au values generally occur within and above a zone containing bladed quartz veins and veinlets that contain lattice-like replacement textures of quartz after calcite. These textures can be traced as a discrete, tabular, 5 to 20 foot thick, shallow northeast dipping textural zone from hole to hole that sits in the upper or central portions of the K-feldspar-quartz altered zone, and which probably records a boiling level in the hydrothermal system. Below this bladed quartz zone, chalcedonic quartz veinlets are common, and progressively increase in abundance downward toward the Palmetto conformity"

Siliceous structures oriented similarly to those in the Tombstone Formation occur in the underlying Palmetto Formation. Veins hosted in the Palmetto Formation form well-defined discrete veins and hydrothermal breccias up to 2 meters wide according to MGC. Alteration in the Palmetto Formation is characterized by argillic alteration extending up to a few hundreds of meters below the nonconformity with the Tombstone Formation. Intense argillic alteration is typically limited to a zone within one to eight meters of the nonconformity, with gradual weakening of bleaching and clay alteration to greater depth. Locally, the zone of intense quartz-adularia alteration in the overlying Tombstone Formation may extend into the Palmetto Formation for a few feet (Rhys, 2003).

8 Deposit Types

Alteration and mineralization at the Tonopah property are typical of low-sulfidation, volcanic-hosted epithermal gold deposits found elsewhere in Nevada and around the world. The deposit type is characterized by overall low original sulfide content, and quartz-adularia and clay-sericite alteration assemblages, among others. Vein textures are indicative of high level, near surface emplacement and include void fills, crustiform coatings, colloform banding, and comb structures. Similar deposits in Nevada have proven to be economic, including the Midas and Bullfrog deposits.

The proximity and similarities of the Tonopah property to other gold deposits does not, on its own, indicate that the Tonopah property should be similarly mineralized.

9 Exploration

9.1 Previous Owner's Exploration Work

A total of 637 drill holes totaling 86,707 meters have been completed in the greater Tonopah Project area by a number of companies beginning in the 1970's. The majority of the work focused specifically on the concealed gold system at the Tonopah Project was conducted beginning in the late 1980's and continued through Midway's ownership.

Early exploration work was focused on establishing the limits of a large, low-grade gold mineralized system located in the upper portion of the Palmetto formation and in the altered lower units of the tertiary volcanics. Previously issued technical reports (Ristorcelli and Muerhoff, 2002; Ristorcelli, 2003; Gustin and Ristorcelli, 2005) are focused on this interpretation for the deposit.

MGC reviewed and compiled subsurface data and targeted exploration on evaluation of higher grade gold mineralization localized around structural zones, quartz veins, and feeders. MGC used this data to evaluate a model focused on the potential for underground mining vein and feeder zones. The superseded Gustavson 2011 technical report focused on this interpretation of the mineralized system and attempted to model only high grade veins and feeder zones which might be amenable to underground mining.

Viva is focused on understanding both the higher grade and moderate grade portions of the deposit, as a combined model. This consolidated interpretation is more viable because of Viva's reduced royalty structures and the near surface aspect of the deposit, which allows for potentially reduced cut-off grades.

The record of exploration conducted prior to 2005 was documented in technical reports previously released by Mine Development Associates (MDA) (Ristorcelli and Muerhoff, 2002; Ristorcelli, 2003; Gustin and Ristorcelli, 2005). Exploration results from 2005 through 2008 are documented in Gustavson 2011, and include annual exploration work conducted by MGC in 2005 through 2008. During the period, MGC completed a large volume of drilling, a reconnaissance soil gas survey, and a limited amount of rock chip sampling in areas peripheral to the mineral system. Physical exploration activity did not occur at the Tonopah property during 2009 and 2010. 26 drill holes were completed during 2011, principally targeting extensions of subvertical vein zone structures. Drill holes completed in 2011 were documented in Gustavson 2017 and Gustavson 2018 and have been included in the resources estimate shown in this report.

Viva's drilling is discussed in section 10 of this report and is included in the resource estimate in Section 14.

The exploration work carried out under previous operators of the Tonopah Project is described in detail in Section 6.2 (Exploration History) of this report.

9.2 Geologic Studies

A number of Geologic studies have been completed, as are referenced in Section 6.2, Exploration History. No geologic studies had been completed by Viva as of the effective date of this report.

9.3 Geologic Mapping

Available geologic mapping to date is summarized in Section 6.2, Exploration History. No geologic mapping had been completed by Viva as of the effective date of this report.

9.4 Geophysical Modelling

A significant amount of geophysical data was collected during historical exploration, as described in Section 6.2, Exploration History. The geophysical database is extensive with uniformly high quality data. It contains data for eight (8) geophysical techniques applied to gold exploration. All the data were generated by either Kennecott Corporation in the early 1990's and/or Newmont Mining Corporation in the early 2000's. Viva has contracted James Wright of Wright Geophysics to review the historical geophysical data in conjunction with the updated geological modelling and drilling information to generate additional insights and drilling targets.

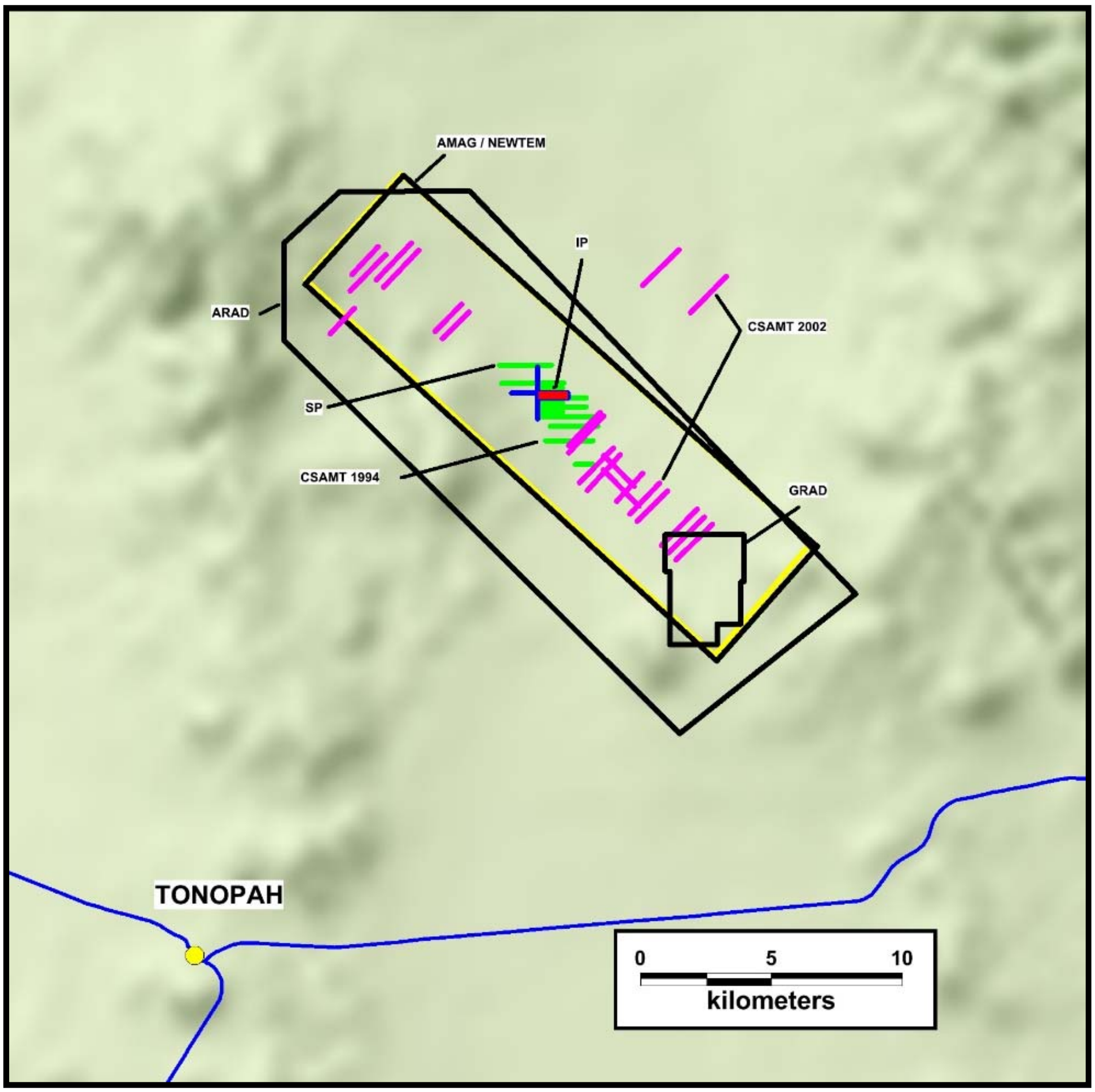


Figure 9-1: Historic Geophysical Survey Index Map (Wright, 2019)

9.5 Surface Sampling

Limited surface sampling has been carried out at the Tonopah property, principally because the main mineralized targets, with the exception of the Discovery outcrop, are covered by post-mineral alluvium, colluvium, and dune sands deposits. Evidence exists to indicate that surface sampling at the western portion of the claim block, where there is outcrop of the tertiary volcanics and Palmetto formation, was completed. Drilling for the last several campaigns has focused on covered areas east of where surface sampling occurred, and the surface samples are not considered material to resource estimation for the Tonopah project.

9.6 Samples in Mine Workings

There are no mine workings of significant extent in the main project area. There are some small prospect pits in the hills to the western portion of the claim block, which have been sampled at surface. Again, this is outside the main mineralized area and not considered material to resource estimation for the project.

10 Drilling

A total of 673 holes totaling 91,646 meters have been drilled at the Tonopah Project. 637 of these were completed prior to the acquisition by Viva Gold. Existing drill holes include 12 reverse circulation and auger holes drilled by Midway Gold for hydrology studies, and 12 diamond core holes drilled for geotechnical studies. Approximately half of these drill holes are outside the current resource area, including 100+ holes drilled in the Thunder Mountain area, which is no longer part of the Tonopah Project, and approximately 200 holes drilled west of the current resource area. Drill hole data for the Project is summarized in Table 10-1, and drill hole locations are shown on Figure 10-1. A complete list of drill holes, including year drilled, coordinates, drilling campaign, azimuth and dip, is included as Appendix D.

Viva initiated a drilling program in 2018 designed first to confirm the historical database and secondarily to extend mineralization by targeting areas of inferred which could be upgraded to measured and indicated categories, as well as to provide fresh material for metallurgical test work. Viva initiated an additional drill program in mid-2019 for exploration drilling. Viva drilled a total of 4 Core and 32 RC holes totaling 4,939 meters during the 2018-2019 drilling campaigns.

Table 10-1: Drill Hole Data Summary at Tonopah Project

Company	Year	RC		Core		Total Drill Holes	Total (m)
		No.	**m**	**No.**	**m**		
Felmont	1980-1981	92	9,214			92	9,214
Coeur d'Alene	1988	3	328			3	328
Rio Algom	1990-1991	41	6,026			41	6,026
Kennecott	1992-1996	133	20,486	4	553	137	21,039
Bob Warren	1994	3	361			3	361
Golconda	1996 - 1997	9	515			9	515
Tombstone	1997	14	1,980			14	1,980
Midway Gold	2002	20	3,304	49	4,832	69	8,136
Newmont	2002 - 2004	84	12,692	38	8,022	122	20,714
Midway Gold	2005	22	2,739	1	43	23	2,782
	2006	44	6,899	19	1,289	63	8,188
	2007	11	1,436	8	967	19	2,403
	2008			16	1,051	16	1,051
	2011			26	3,970	26	3,970
Viva Gold	2018	16	2,195	4	576	20	2,771
	2019	16	2,168			16	2,168
Total		508	70,343	165	21,303	673	91,646

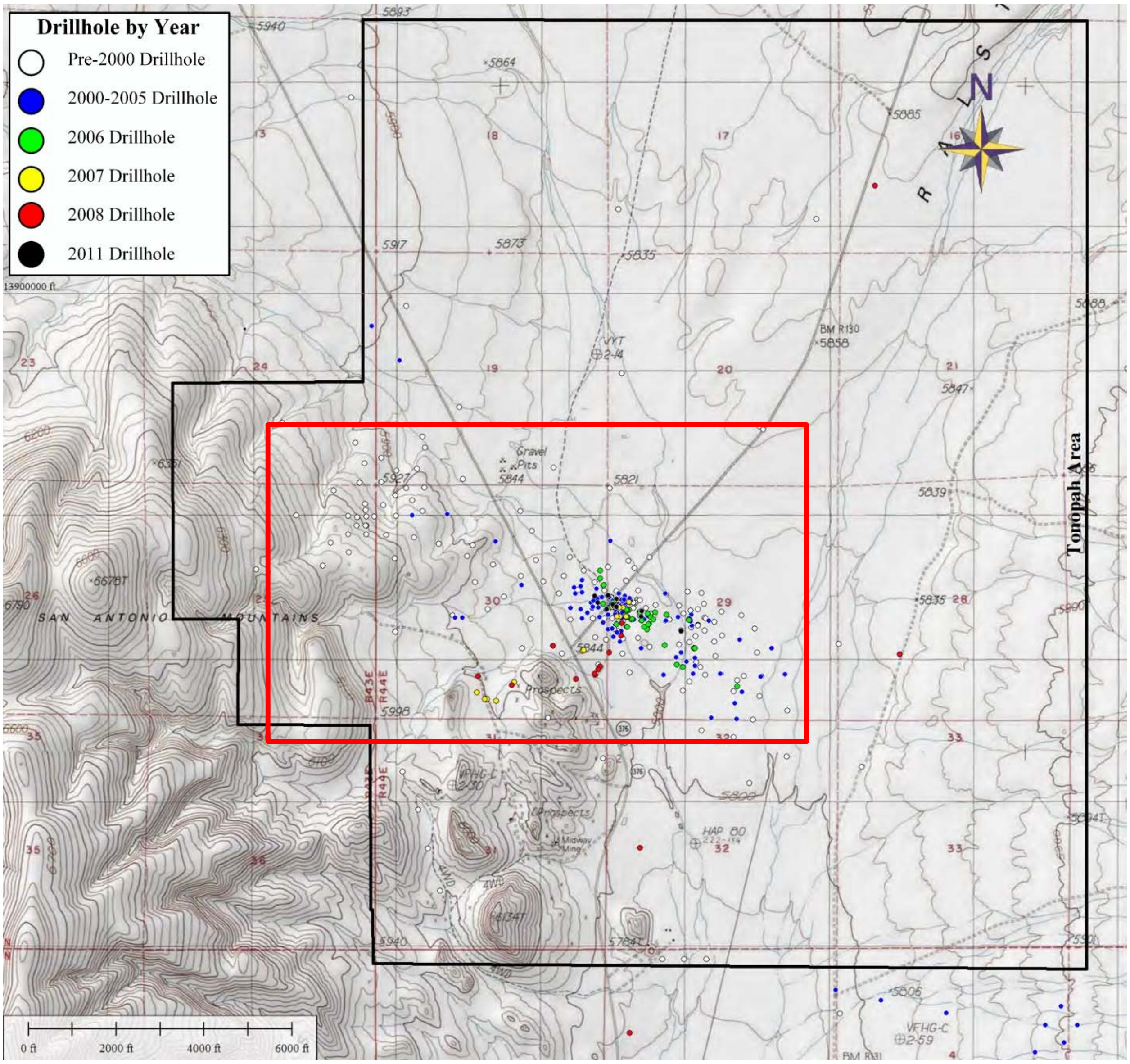


Figure 10-1: Historical (Pre-Viva) Exploration Drilling at the Tonopah Project. Red border shows area of inset in Figure 10-2.

10.1 Drill collar Survey Update

Following the 2019 drilling campaign, Viva conducted a review of all collar survey location data for the Tonopah project. This effort had been previously recommended by Gustavson geologists to resolve concerns about location data from the various drilling campaigns because historical databases are found to have location data in NAD 83 and NAD 27 projection systems, as well as two separate local grids, and because there had been conversions back and forth between US survey feet and metric units.

This review was intended to accomplish two objectives. First, any remaining concerns about collar locations for the projects may be resolved, allowing for higher confidence in the resource models. Secondly, as permitting maps at Tonopah are filed in UTM meters, it is useful to update all project databases, resource estimates, maps, and planning documents to UTM meters as part of the PEA.

To conduct this review, Viva reverted to historical drill collar location data from original log forms and survey sheets, with careful note of the original projection system and coordinate type. Second, all locations were converted to UTM using the National Geodetic Survey Coordinate Conversion and Transformation Tool (NCAT) which accounts both for both translation and rotation adjustments between survey systems.

Most drill hole locations were found to be within 2 meters of the previously recorded location data. However, certain batches of data were found which show offsets of up to 10 meters. It appears that these shifts are related to conversion errors between metric and imperial coordinates. (too few decimal places used in the conversion, which has impact on UTM scale numbers.)

Viva has cross-checked drill hole locations where available by comparison with handheld GPS, and is confident that the updated location data are consistent with field observed locations.

Gustavson has reviewed Viva's work on recompiling survey data and believes the updated information, subject to the degree of accuracy of the original surveys, is the best available location information for the drill holes.

10.2 Drilling Procedures and Conditions

Core logging and drilling conditions prior to 2005 have been described by previous independent reviewers. Drilling procedures described by Gustin et al. (2005) indicate that industry standards were practiced from 1981 to at least 1997. Industry standards were also practiced with regard to drilling, logging and chain of custody from 2002 through 2004. Given the presence of coarse and visible gold at Tonopah, care must be taken with regard to sample collection during both core and RC drilling. Water used during RC drilling may contribute to sample bias, and core samples need to be large in order to provide a representative analytical sample.

Detailed information regarding drilling campaigns prior to 2005 is included in technical reports produced by MDA (Ristorcelli and Muerhoff, MDA, 2002; Ristorcelli, MDA, 2003; and Gustin and Ristorcelli, MDA, 2005). That information is summarized in earlier sections of this report and is not repeated here in detail.

MGC contracted Diversified Drilling of Missoula, Montana to perform reverse circulation drilling in 2005, and Layne Christensen, Las Vegas, Nevada, was contracted for all reverse circulation drilling during 2006-2008. Kirkness Diamond Drilling Co., Inc. and M2 Core Drilling and Cutting, Inc. provided core drilling services in 2007 and 2008, respectively.

Information for the 2011 drilling campaign were well summarized by Podratz and LeLacheur, 2014:

The 2011 Core Drilling campaign was completed by KB drilling of Mound House, Nevada, using a track mounted Versa KMB 1.4 Drill Rig equipped with HQ3 tools for use of split tube. Oriented core was collected using a Reflex ActII down hole tool. A Midway geologist was on site for core drilling. Geotechnical and structural data were logged prior to core being boxed. Boxed core was transported to secure logging facility in Tonopah, NV, by Midway personnel. The drill core was logged for rock type, geologic unit, alteration, mineralization, structural details, and specific gravity.

Drill hole collars were initially located with handheld global positioning system (GPS) units and surveyed afterward by Trimble GPS using UTM NAD 83, Zone 11 projection. Down-hole surveys for each hole were completed by International Directional Services of Elko, Nevada, using a Surface Recording Gyroscope, model DG-69. Upon completion of drilling and down-hole surveying, the holes were abandoned according to Nevada State regulations, including a cement plug at the surface that secures an eye-bolt with a metal tag for identification. The eye-bolt enables post-reclamation location of the drill hole through the use of a metal detector.

10.3 Drill Hole Logging

Available core and RC chips from drilling prior to 2002 were re-logged and entered into the Tonopah Project drill hole database by Newmont geologists. Between 2002 and 2004 all core was photographed, logged, and entered into an electronic drill hole database. Data captured during core logging included geology and RQD measurements. The drill hole database is stored electronically and in hardcopy at the Tonopah, Nevada project office. The drill hole database includes all existing drill logs, analyses, photographs, drill collar locations and down-hole survey information for the Tonopah Project. MGC adheres to procedures established by Newmont for all drilling, core logging and sampling activities (Mosch, 2009) (Podratz & LeLacheur, 2014).

10.4 Sample Procedure

Gustavson personnel were not on-site during any of the drilling programs conducted by MGC in 2005 through 2011, and are reliant on information provided by MGC regarding sample handling and security.

MGC reports that sampling of diamond drill core and reverse circulation cuttings was conducted in accordance with standard industry practices and routine procedures established by Newmont (MDA 2005). The sampling methods and approach employed since 2004 are consistent with those reported by MGC, as follows, in the 2005 technical report prepared by MDA:

"Core sampling procedures for Midway Gold's drill program were being done in accordance with standard industry practices. These practices reportedly remained the same through the transition from Midway Gold to Newmont as operators. Core was stored at the drill site until taken by Midway Gold's geological contractors [since 2004, Midway employees, consultants or M2 Technical Services as contractor] to the logging and core storage warehouse in Tonopah. After photographing and logging, the HQ core was generally sampled in five-foot intervals, but sample intervals do not extend across distinct

geologic breaks. Generally, the maximum length of a sample was five feet, but could be as small as one half of a foot if warranted. Core samples were split by mechanical or hydraulic splitters, or sawed into two halves, with half samples placed in cloth bags that have been pre-numbered with a unique sample identification number. The sample identification did not contain the drill hole name, drill hole number, sample depth or sample length. A sample tag was also placed in each bag. Core samples remained at the logging facility in Tonopah, or were taken to the project site, where they were picked up by Chemex [ALS Chemex] for analysis. One half of the core was retained in the Tonopah warehouse facilities and the other half submitted for analysis.

Sampling of the RC cuttings was done by the drilling contractor under the supervision of the geologist. RC samples were collected on 1.52 m (5 ft) increments over the entire hole. All of Midway Gold's RC drilling used water as a drilling fluid, partly because water was injected down the hole in order to minimize dust (in accordance with BLM requests) but also because all holes drilled by Midway Gold intersected ground water at some point. The slurry of water and drill cuttings was forced up the drill pipe into a cyclone, where it was passed through a rotating wet-sample splitter. The sample was reduced to approximately 7 to 9 kg and collected in a five-gallon bucket. The majority of the liquid was poured off and the sampled cuttings were placed in cloth bags, which were pre-labeled with sample ID numbers. Labeling of RC samples was guided and managed by Midway Gold geological contractors [or employees or consultants], but not necessarily done by them. Samples were given a unique label, which did not relate to either drill hole or depth and only Midway Gold and their geological contractors knew the relationship between sample and location. A sample tag was put in each bag. Representative samples of drill cuttings were collected and stored at the drill site. Chip samples were collected for each five-foot interval.

The RC cuttings remained under the supervision of Midway Gold geological contractors each day until the end of shift, at which time the Midway Gold geological contractors took them to a secure sample storage area. The samples remained stored until picked up by Chemex [ALS Chemex ("Chemex")]. The laboratory generally picked up the core and RC samples about two times per week.

For both RC and drill core, duplicate samples were collected approximately every 31 m (20 samples) [see the QA/QC comments in the Sample Preparation, Analyses and Security section for Gustavson comments on the practices of Midway Gold]. *For RC drilling, this was conducted by placing a Y-splitter on the discharge of the cyclone. MDA feels that due to the presence of free gold, this was not an appropriate way to split wet samples. For core, a duplicate sample was initially collected by splitting the half core to produce a quarter split. Due to the coarse nature of gold encountered at Midway, this practice was changed to sampling the core in two halves.*

Drilling campaigns at the Tonopah Project range from early exploration with widely spaced drill sites to focused investigation with closely spaced, vertical holes. This exploration drilling approach provides data suitable to modeling a low grade, near surface, potential open pit gold prospect. MGC exploration targeted steeply dipping, high gold grade veins and structures with potential for development as an underground mine. This exploration approach used diamond drill holes oriented to crosscut veins and structures.

10.5 Tabulation of 2011 Drilling Results

The results of the 2011 program are presented in Appendix B. Grades intersected are reasonably representative of these zones, but may not be fully representative of average project grades. Because of the complex relationship between subvertical high-grade mineralization and low-angle, lower grade mineralization, it is difficult to estimate true thicknesses for the various drill hole intersections. In general, Gustavson expects that true thicknesses for these intersections are 70-80% of the lengths indicated

10.6 Viva Drilling Campaigns

Viva initiated a drilling campaign in January 2018 with four diamond core holes and followed up with 22 Reverse Circulation drill holes during 2018 and 2019. Drilling was targeted to verify the historical database, to upgrade inferred mineralization to measured and indicated by targeting inferred mineralization in the 2018 resource model, and to provide material for ongoing metallurgical test work. A third drilling campaign was carried out in the summer of 2019 to test mineralization to the west of the main resource area. Figure 10-2 shows drill hole locations from the resource area, extending west to the midway hills area. Collars from the 2018-2019 drill campaigns are labeled in yellow while past drill collars are shown in orange.

Mr. Matthews observed RC drilling during his September 2019 site visit, and found drilling and sampling conditions to be consistent with industry standards.

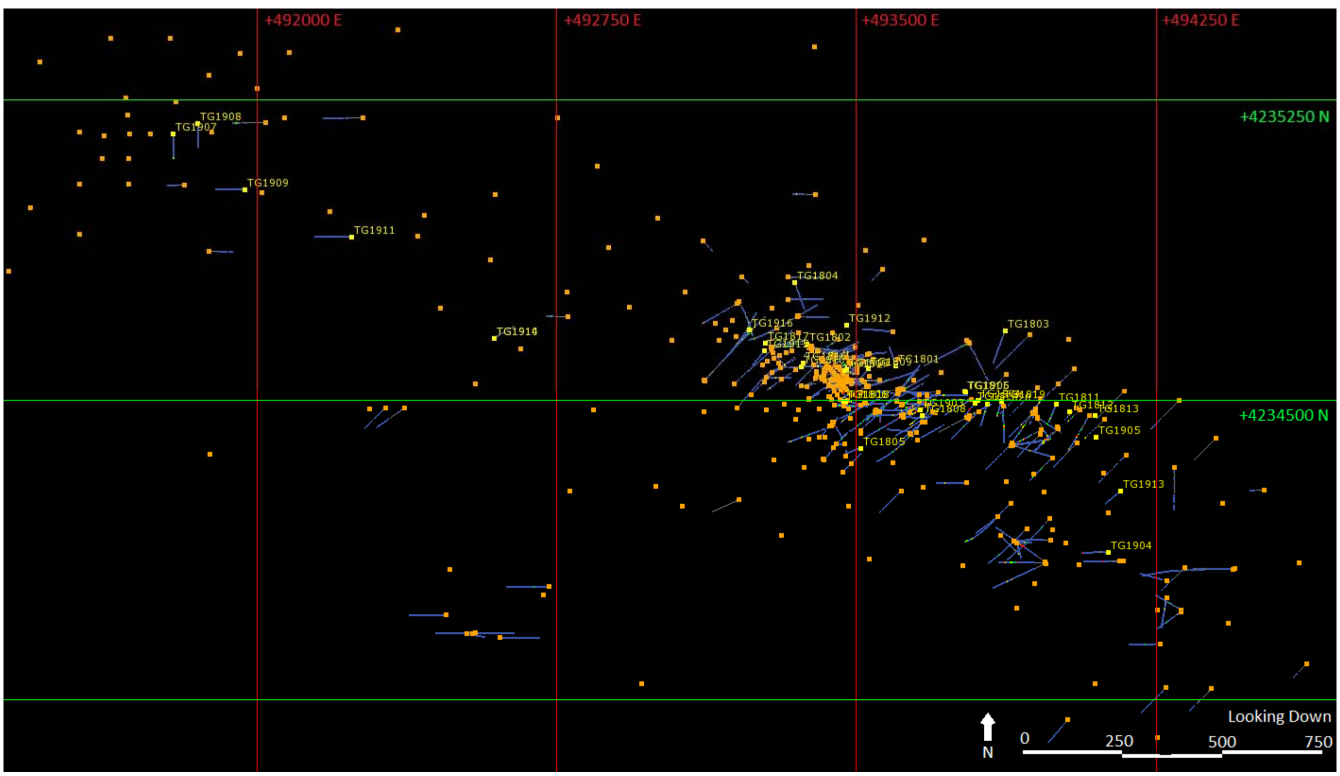


Figure 10-2: Plan view of drilling showing holes from Viva 2018-2019 drill campaigns labeled in yellow (previous drilling in orange).

10.7 Tabulation of Viva Drilling Results

The results of Viva's 2018-2019 drilling campaigns are presented in Appendix C. Grades intersected are consistent with surrounding drill holes and with grade estimates from the 2018 resource model, which constitutes positive validation of the historical database. Because of the complex relationship between subvertical high-grade mineralization and low-angle, lower grade mineralization, it is difficult to estimate true thicknesses for the various drill hole intersections. In general, Gustavson expects that true thicknesses for these intersections are 70-80% of the lengths indicated.

10.8 Cross Sections of Drilling Results

Because of the large number of historical drill holes completed at the Tonopah project, Gustavson believes that showing typical cross sections through the mineralized zones is more representative of drilling results than a tabulation of drilling results, particularly with regard to interpretation of true thickness and continuity of mineralization.

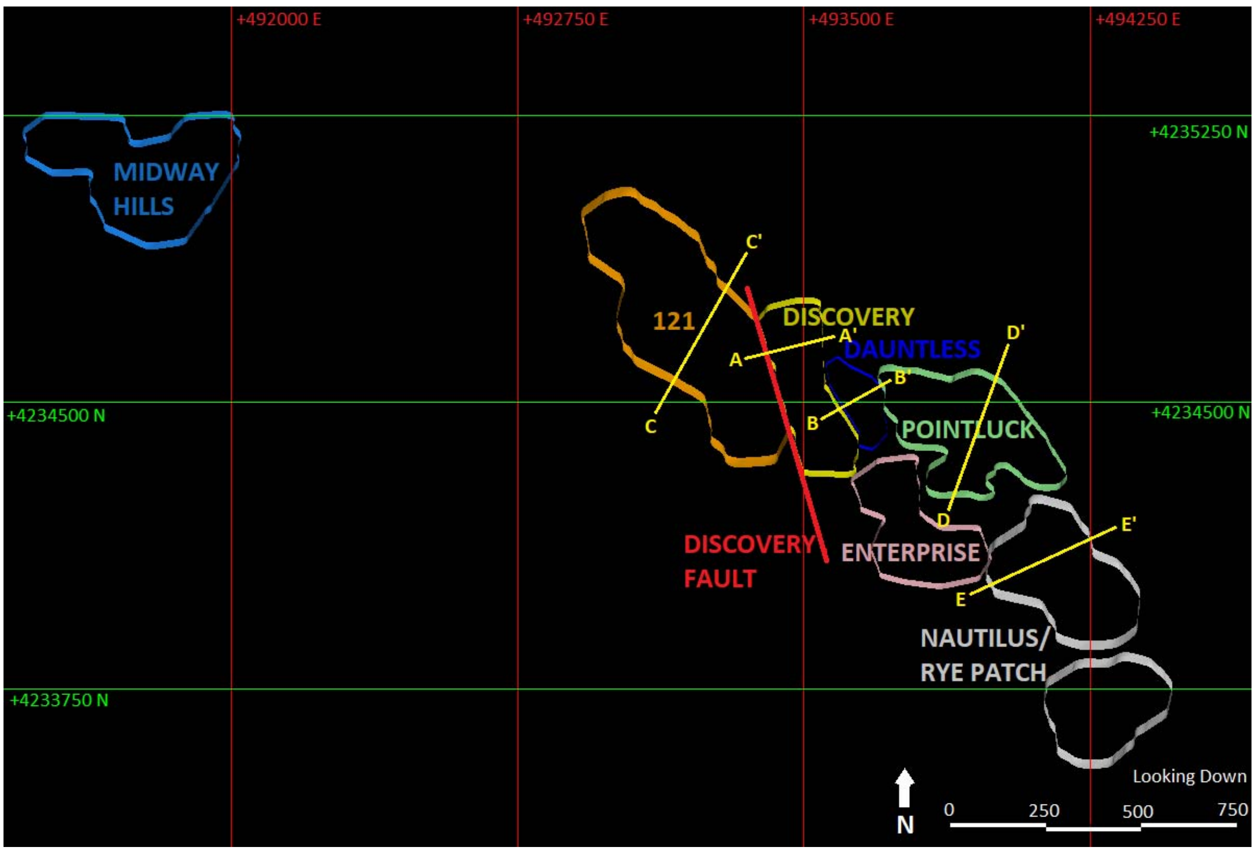


Figure 10-3: Mineralized Zones at the Tonopah project and cross section traces for Section Lines. Section A-A' corresponds to Figure 10-4, Section B-B' corresponds to Figure 10-5, Section C-C' corresponds to Figure 10-6, Section D-D' corresponds to Figure 10-7, and Section E-E' corresponds to Figure 10-8.

A plan view of drilling with section traces is shown in Figure 10-3. Figure 10-4 through Figure 10-8 show 100-foot thick sections through the deposit. Lower-grade mineralization is typically associated with the contact zone between the Tertiary volcanics and the underlying Ordovician argillite. Higher-grade mineralization sometimes parallels these zones, but also may be associated with subvertical structural zones.

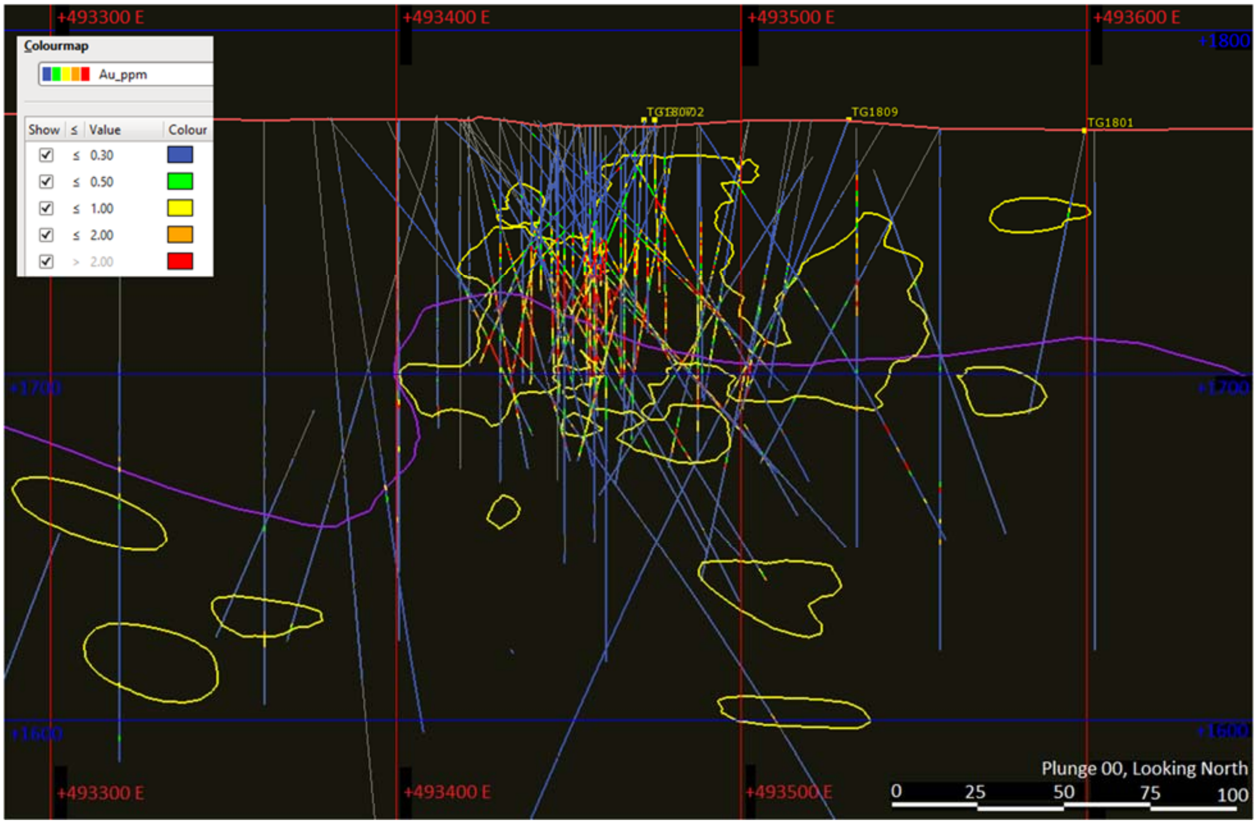


Figure 10-4: Cross Section through Discovery Zone Mineralization (Section A-A' from Figure 10-3)

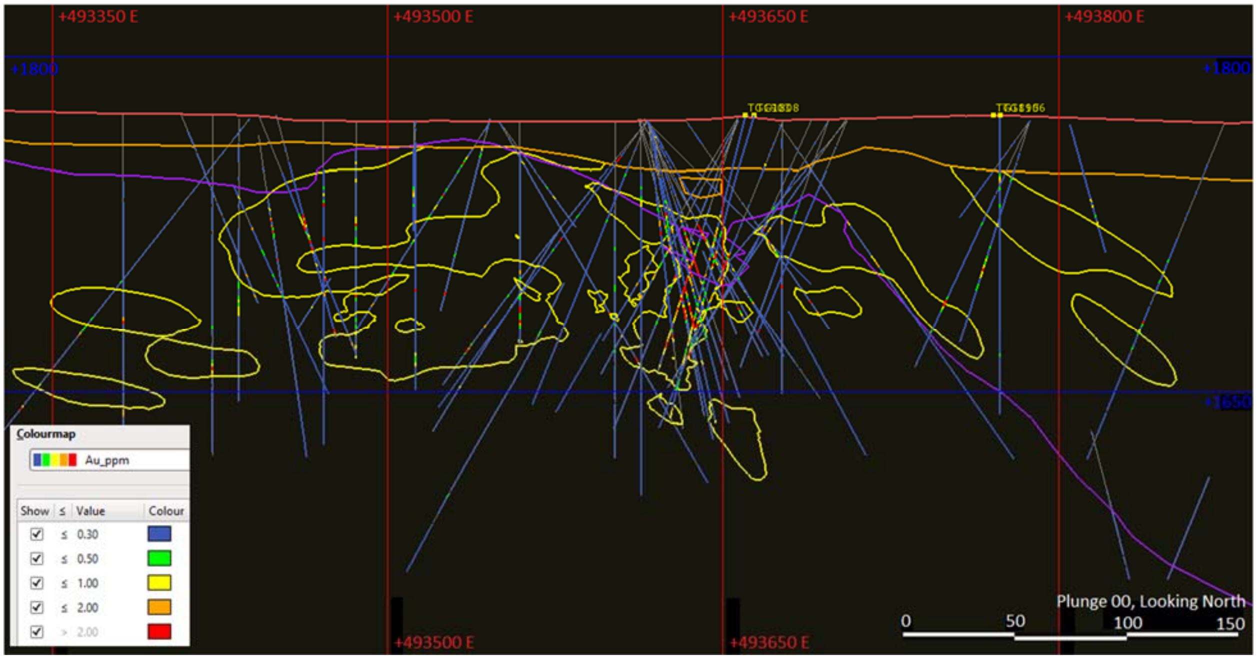


Figure 10-5: Cross Section through Discovery (Left) and Dauntless Zone (Center) Mineralization with 2018-2019 drill holes identified in gold text
(Section B-B' from Figure 10-3)

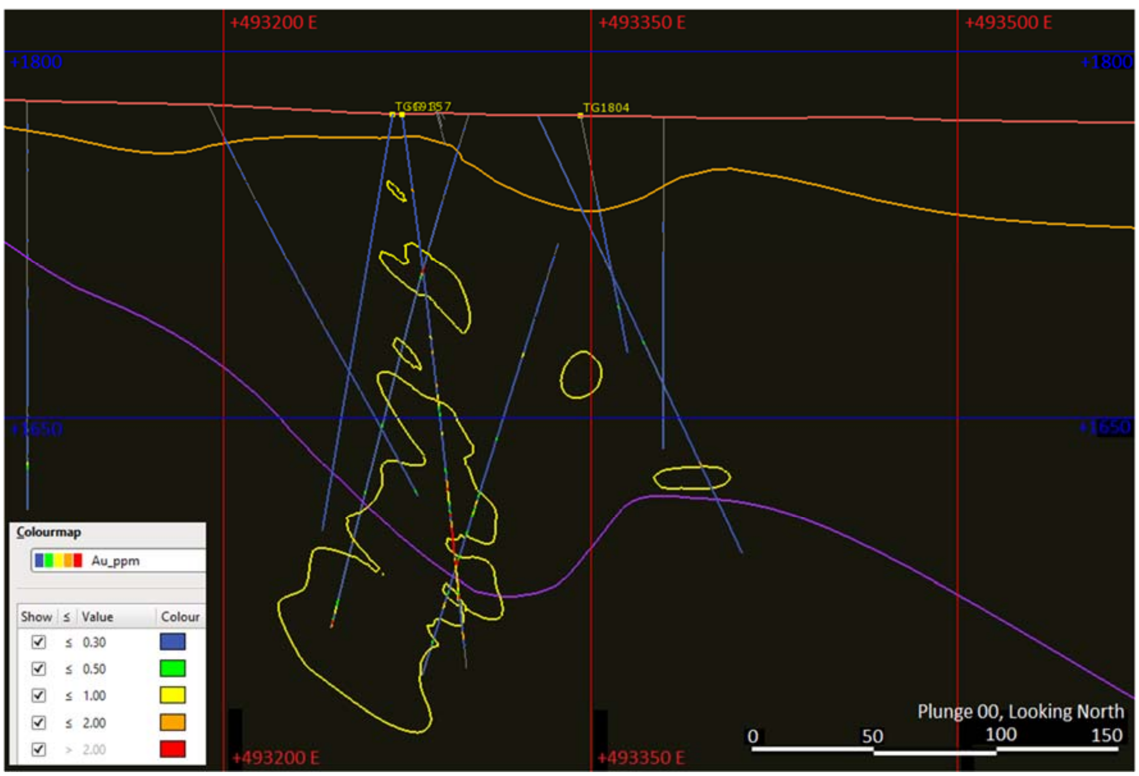


Figure 10-6: Cross Section showing 121 Zone Mineralization with 2018-2019 drill holes identified in gold text.
(Section C-C' from Figure 10-3)

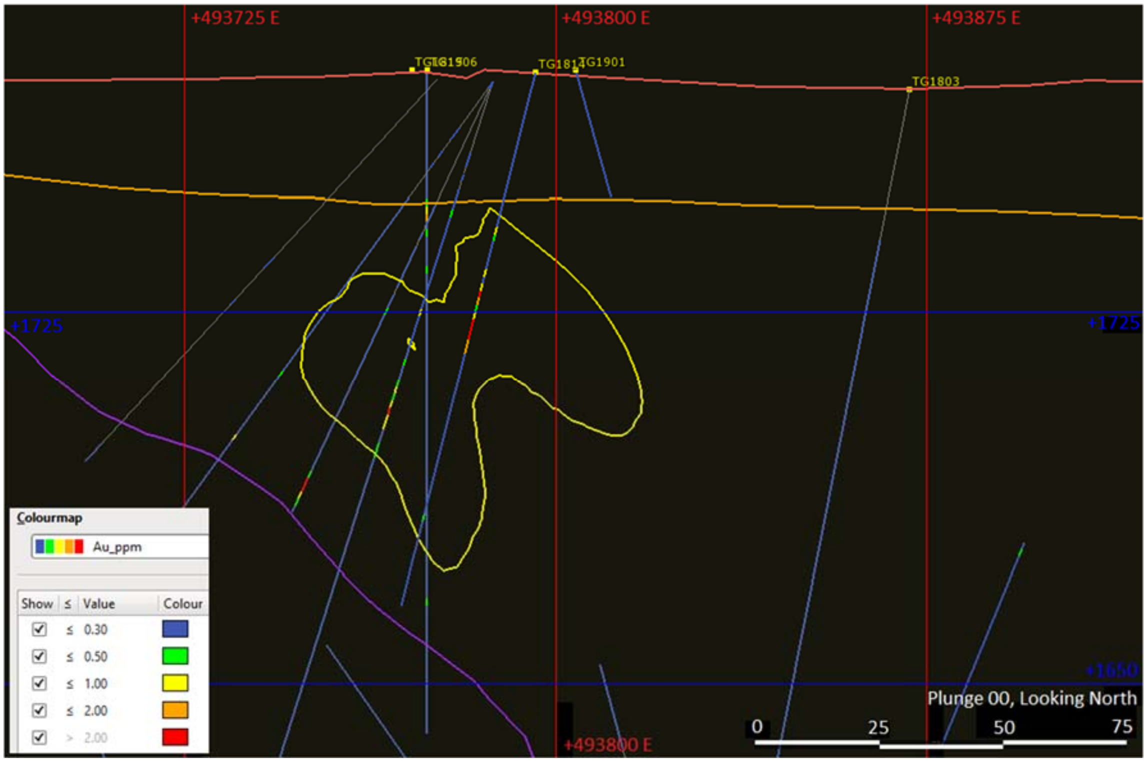


Figure 10-7: Cross Section showing 63-77/Pointluck Zone Mineralization with 2018-2019 drill hole collars identified in gold text. (Section D-D' from Figure 10-3)

Figure 10-8:Cross Section showing Nautilus/Rye Patch Zone Mineralization with 2018-2019 drilling (Section E-E' from Figure 10-3)

10.9 Interpretation and Relevant Results

Viva inherited a database of historical drill hole data which is useful for geological modelling and resource estimation. Historical drilling processes and procedures have been well designed, well documented, and subject to periodic independent review and reporting. Viva has completed a total of 36 drill holes on the property and has generally adopted the well-developed and documented drilling and sampling procedures which have previously been used at the project, with attention given to ensuring a consistent record of QA/QC to monitor laboratory performance.

The Viva drilling campaigns have been effective in verifying grades and locations of mineralization as identified in the historical database, and has provided infill data in certain areas of the deposit.

11 Sample Preparation, Analysis, & Security

11.1 Historic Campaigns

A review of sample preparation, analyses, and security with regard to Tonopah exploration prior 2004 was discussed in previously released technical reports prepared by MDA on Midway Gold Drilling Campaigns. Section 6.2 (Exploration History) summarizes past exploration campaigns at Tonopah.

11.2 Sample Preparation and Assaying Methods

Diamond drill core samples were placed in core boxes at the drill site and were transported daily to the sample warehouse in Tonopah. The core was photographed and marked for splitting, and all pertinent geologic and geotechnical information recorded. The core was cut with a diamond rock saw or split using a manual or hydraulic splitter, if necessary. The half-core for each sample interval was placed in pre-labeled bags, sealed, and stored until the sample was transported to Chemex.

Reverse circulation samples were placed in sample bags at the drill site and were transported daily to the project warehouse in Tonopah. Chip trays were made from each sample. The samples and chip trays were logged and stored securely at the warehouse until they were transported to Chemex.

Overall assay results from the Tonopah Project do not vary substantially between drilling campaigns or operators. Gustavson has no reason to suspect that sample integrity was compromised in any of the historic campaigns or under MGC in recent years.

Sample preparation at Chemex was and is conducted according to the guidelines set out in ISO/IEC Guide 25 – "General requirements for the competence of calibration and testing laboratories" and was certified to the ISO 9002 standard.

At Chemex, Tonopah samples were generally prepared by crushing the entire sample and pulverizing the sample split to 75 microns. Samples were analyzed for gold by fire assay with an atomic absorption finish (AA). Other elements were analyzed by induced coupled plasma (ICP) techniques.

11.2.1 2011 Assay Procedures; Coarse Gold Sampling Review

Sampling for the 2011 Drilling program was designed to test nugget effect and sampling procedures for high-grade intervals. (Podratz & LeLacheur, 2014). Samples were collected in two stages: vein sampling and all other material sampling. For vein sampling, all quartz vein material, quartz vein selvage and breccia, and vein-related silica alteration material was sampled and sent to Florin Labs for Metallic Screen Assay.

Florin Labs Analytical procedure:

1. Crush to 95% passing 32 Mesh
2. Rotary split sample to 1000 g
3. Pulverize to 85% <75 micron
4. A complete fire screen assay process, (assay 100% of coarse metallic fraction + 100 mesh, two 30-gram fire assays on minus 100 mesh fraction) on entire 1000g pulp.

For all other material, samples were sent to ALS Chemex Labs.

ALS Chemex Lab procedure:

1. Crush sample to 70% -20mm
2. Riffle sample split 30g
3. Pulverize split to 85% <75 micron
4. Fire assay, AA finish
5. Then for all intercepts > 0.002 opt
    a. Re-split fine crush reject to 1000g
    b. AU screen fire assay -100-micron, Ore grade FA AA finish

A total of 867 samples were sent to Florin for Metallic Screen Assay from the 2011 drilling campaign.

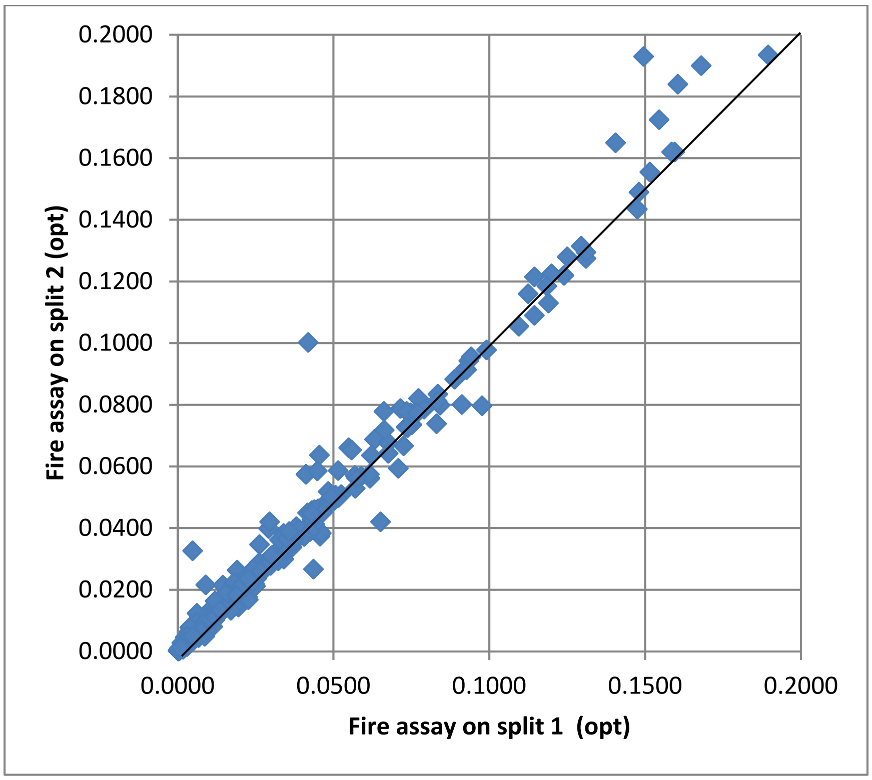


Figure 11-1: Comparison of Fire Assay Split 1 to Split 2

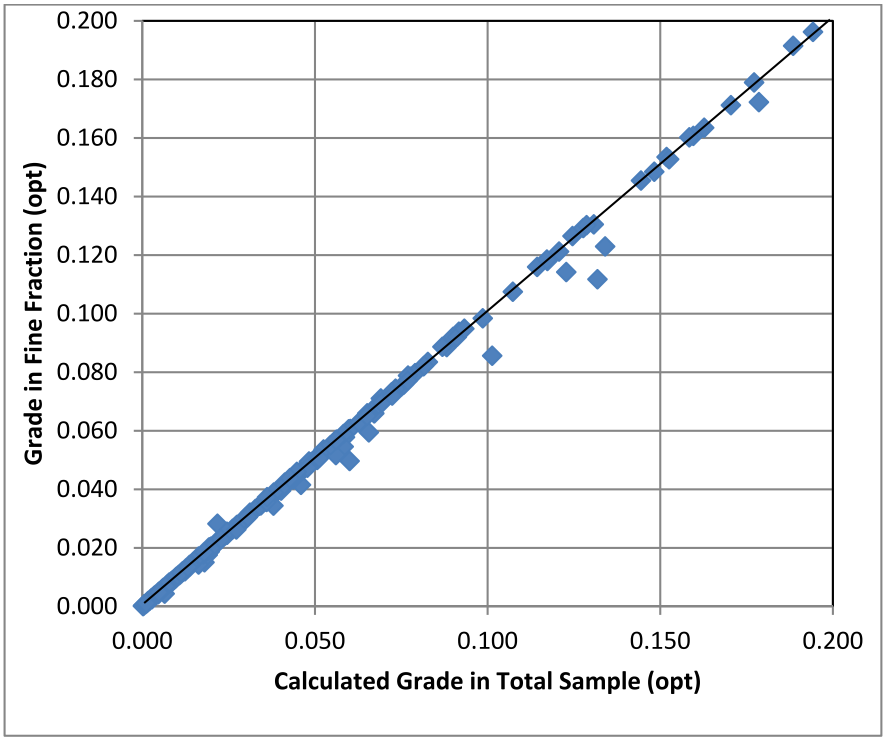


Figure 11-2: Comparison of Gold in Fine Fraction to Gold in total Sample

Comparison of two fire assay splits (Figure 11-1) and fine fraction gold grade to total sample gold grade (Figure 11-2) indicates high reproducibility of gold assays and that gold is reasonably evenly distributed through the samples. With careful sample preparation and splitting, standard 30g fire assays should provide a reasonable gold assay for the Tonopah project.

11.3 Quality Controls and Quality Assurance

QA/QC programs for historic drilling and sampling campaigns at the Tonopah project were addressed in several contemporaneous prior technical reports. Not all the QA/QC data for the programs have been captured in the project database. Gustavson recommends that this information be captured to allow for the full record of data verification to be preserved with the remainder of the data.

MGC submitted a combination of standard samples and duplicate samples at a rate of about 4%, or one of either type of sample with every 24 unknown samples.

While the rate of submission of combined standards and duplicates has remained fairly constant, the mix of standards and duplicates samples appears to vary from drill hole to drill hole and duplicate samples are not included in the sample mix from some drill holes. Gustavson believes the 4% rate for combined standards and duplicates is adequate, but recommends that a more rigorous split between standards and duplicates be maintained.

Midway did not submit blank samples during any of their drilling campaigns from 2004 through 2008. Gustavson recommends that for future drilling programs, Viva submit a blank sample with each drill hole or other sample batch at the rate of 2% (or 1 sample out of each 50) in order to increase the likelihood of contamination detection by MGC, independent of the internal QA/QC program of the lab.

A significant quantity of testing and reporting on gold sampling heterogeneity has been carried out for the Tonopah project. Staley (2003) outlines the results of testing and describes a recommended sampling protocol of crushing of 100% of half-split core to 95% passing 32 mesh, followed by pulverization of a 1 kg split to 100% passing 100 mesh. The recommendation for RC sampling is total collection of the RC sample for size reduction at by crushing 100% of the sample at the assay lab, with pulverization of a 1kg split to 100% passing 100 mesh.

MGC submitted splits of high grade samples for further analysis by metallic screen assays to determine the potential impact of coarse gold on the sampling process. Total gold according to the metallic screen assay was averaged with the original assay for the associated sample interval and entered into the project database. Gustavson recommended that MGC compare the metallic screen assay data to original assay data to determine if bias is generated during the sampling/sample preparation processes. The results of this work (11.2.1, above) indicate that standard fire assays should be usable for resource estimation.

11.4 Viva 2018-2019 Drilling Campaigns

Viva Gold maintains industry standards for drilling, sampling, and assays in its current operations.

For core holes, HQ core was boxed on site by the drill crews and transferred by Viva's geological consultant to the exploration office in Tonopah Nevada for sampling. Core boxes were labeled with drill hole number, start and end depths, and core box number. Core was photographed, logged digitally to excel files, and marked for splitting by Viva's geology consultants. Samples were split by cutting by core saw. ½ sample was placed in numbered sample bags, bundled into rice sacks, and set aside for pickup at site by ALS Chemex Laboratories of Reno Nevada. ½ core was retained in the core storage facility at the Viva exploration office. 100% of HQ core was split and sampled for the Viva drill holes. There are no unsampled or unassayed intervals.

For dry RC Drilling, samples were collected at 5-foot intervals in pre-labeled 12x18" sample bags. Samples are not split. Wet samples are collected into 5-gallon buckets. Samples are allowed to settle for 10-15 minutes. Excess water is decanted, and the damp sample is poured into a pre-labeled sample bag. 100% of sample is bagged and shipped to ALS Chemex for sample preparation, to ensure a large sample. Coarse rejects are returned by ALS for sample inventory.

For all RC samples, a small quantity of material from the sample is captured to a labeled chip tray for logging.

Samples are aggregated into rice bags and were either collected at site by ALS Chemex laboratories of Reno, Nevada, or were transported by Viva's geology consultants to the Tonopah office for pickup.

Samples for the Viva drilling campaign are prepped at ALS Chemex in Reno Nevada by drying in ovens (DRY-21) and crushing according to 31-BY (Crush to 70% less than 2mm, rotary split off 1kg, pulverize split to better than 85% passing 75 microns). A 1kg split is taken by rotary splitter and the entire 1kg is prepared as pulp (minimum 85% passing 75 microns). Pulps are transferred to ALS Vancouver for final analysis. Assays are done by AA23: 30-gram fire assay, AA finish, with GRA21, FA/ Gravimetric finish for overlimit gold assays.

Viva also typically requests AA13, 30-gram cyanide leach assays for Au, Ag with AA finish, in mineralized intervals. For the core holes, AA61, four acid digestion, AAS for Ag, Co, Mo, Pb, Zs, Cu, Ni and Zn, was also used. Multi-element geochem and cyanide leach assay values are captured to the database but are not currently used in the resource estimation.

Pulps and Coarse rejects are currently being stored by Chemex.

Viva submits standards and blanks with the sample stream to the laboratory and monitors laboratory performance for these materials.

ALS Chemex is an internationally recognized commercial assay laboratory, with significant experience and expertise with gold assays. Gustavson notes that Viva is not using metallic screen analysis, but based on the 2011 Florin / Chemex comparisons (11.2.1, above), this is not expected to have a material impact on sample performance.

12 Data Verification

12.1 Site Visits

Gustavson (Thomas C. Matthews) visited the Tonopah Project in April 2017 after Viva's acquisition of the project. Mr. Matthews visited the project site, as well as the mine office in Tonopah, and verified that the core boxes, RC chips, and project records from the MGC drilling programs project are stored at the mine office and warehouse. At the project site, drill sites were noted as having been reclaimed and remediated concurrent with drilling operations, and neither historic nor recent drill hole locations could be verified in the field, with the exception of several vertical holes which are available for groundwater monitoring. The surface geology was reviewed. The only outcrop within the area of immediate interest was visited (the so-called "Discovery Outcrop"), and rock type, alteration and silica veining were confirmed as reported.

During the office portion of the site visit, Mr. Matthews verified that diamond drill core and reverse circulation chip trays from MGC drilling programs are stored in the Tonopah warehouse. Mr. Matthews reviewed core from 20 random drill holes to confirm the consistency of the geologic database and to review the geology and alteration, particularly in mineralized intervals. The geology and alteration observed was found entirely consistent with the database record. Geologic logs of the drill holes were reviewed, and assay sample numbers were compared with assay certificates from Chemex for each drill hole. Assay values reported by Chemex were spot checked against the manual entry on the drill logs, and as entered in the project database. No significant errors were identified. Verification samples were not collected. Gustavson verified that sample rejects and pulps are well organized, numbered and are stored in shipping containers at the Tonopah warehouse site in numbered boxes from Chemex. Geologic logs for 2005 and 2006 drill holes are manual, hand-written logs; whereas logs completed by M2 Technical Services in 2007 and 2008 are electronic. Drill logs from 2011 are electronic.

There are a few drill logs and assay certificates missing from the hard copy files, but geologic information and assays have been captured in the database.

Table 10-1 is a summary of the available drilling including historic drilling. Not all of the historic drilling is on the current claims, particularly the 1980-1981 Felmont drilling, which is focused on a separate claim block to the southeast that is not currently part of the Tonopah Project claims. However, for sake of completeness, all drill holes which were once associated with the project database are maintained as part of the record.

Mr. Matthews conducted a second site visit the Tonopah Project in September 2019, during the 2019 drilling campaign. Mr. Matthews visited the project site and observed RC drilling conditions and sampling methods within the resource area, as well as surface geology, prepared pads, and drill collars in the Midway Hills area. Drilling and sampling conditions were observed to be consistent with industry standards.

12.2 Collar Survey

The updated collar survey database is in UTM, NAD83 meters, with a reported precision of hundredths of meters.

Different generations of drilling used at least UTM NAD 83 and UTM NAD 27, and surveys have been stored in both meters and feet and converted between meters and feet during different drilling programs. Earlier data may have been collected in State Plane Feet and converted. Following the 2019 drilling campaign, Viva reconstructed the drill collar database by reverting to original logs and survey records, determining the survey system and datum employed, and converting the database to UTM NAD 83 Meters. While most drill holes remain in the same location as previously estimated, some sets of holes translate by up to 10 meters. This is believed to be a result of low precision conversion between metric and imperial units. It is expected that this resolves any remaining collar survey issues with the database.

Collar surveys from 2018 and 2019 drilling have been collected by handheld GPS and reported in UTM NAD83 meters. Collar surveys from this drill program are assumed to be accurate to with 6 meters. Gustavson recommends that a high precision collar survey be conducted which covers the 2018 and 2019 drill collars so that precise locations can be collected for the database.

12.3 Downhole Survey

Early drilling was designed to test the relatively low-angle contact between the Palmetto Argillite and the overlying volcanics, and as such typically consists of vertical holes. Since 2006, angled drilling was used to test the importance of higher grade mineralization along sub-vertical structures.

Downhole surveys have been conducted for most drill holes from the 2006 and 2007 campaigns, and for the entire 2011 campaign. Most of the earlier drill holes were vertical, were relatively short in length, and are not expected to deviate significantly. 18.8 percent of holes have downhole surveys, typically with one survey each 50 feet (15 meters) downhole. Because of the lack of downhole survey data for the majority of drill holes, it is expected that individual sample locations may be shifted from their actual location by up to a few meters, particularly with increasing depth. This is not uncommon in historical drill hole databases, but it does decrease the reliability of location information for the drill hole database, which may affect resource classification for future reporting.

Downhole surveys were captured for 16 of the 36 holes of the Viva 2018-2019 drilling programs by a downhole survey contractor. These downhole surveys are stored in the project database.

12.4 Lithology & Alteration

Lithology is recorded in the database as alpha codes. The Lithology data as recorded is consistent with industry standards, and cross-checking the lithology data by review of available drill core yielded no anomalies. Lithology data as recorded corresponds with the overall geological interpretations of the deposit.

Lithology coding in the database allows for geological models to be constructed which segregate Palmetto Argillite, at least 2 generations of tertiary volcanics, and overlying gravels. More detailed distinctions between the volcanic units are preserved in the database, but are not always correlated between drill holes.

Primary alteration type is recorded in the database as alpha codes. Intensity of alteration is not recorded. Gustavson generally prefers to see alteration data recorded as numerical codes or quantitative values by alteration field, as recording information in this way allows for more complete alteration models that recognize both relative intensity as well as the possible influence of secondary or tertiary alteration types. For some of the drill holes, alteration data was recorded in this manner, but has not been captured to the digital database. Gustavson recommends that as the project advances, quantitative alteration data be captured to the main digital database so that it is available for geometallurgical models.

12.5 Assay

Gold assays are recorded in the database, typically reported as selected values in parts per million, as well as in troy ounces per short ton. Final gold values are recorded. It is clear that the final value recorded is selected based on earlier reported assay values, including a number of different laboratories, assay techniques, detection limits, and over-limit assay techniques. While checks of the database against assay certificates have not shown errors in the data, the database does not have total clarity as to the process of selecting assay results, nor would it support redefinition of the selection criteria as currently constructed.

Because there is coarse gold at the Tonopah Project, significant effort and study has been spent on the project to study and recommend optimal sampling methods at site as well as optimal methods for crushing and splitting samples at the laboratory. The overall recommendations are:

- For Core: ½ split of core sent to lab, nominally on 5-foot sampling intervals, except where changes in lithology or alteration dictate a shorter interval. At laboratory, 100% of sample dried, primary and secondary crush to -32 mesh. 1 kg split of crushed material taken and pulverized to pulp. Minimum 1 assay ton split from pulp for assay.
- For RC: 100% collection of RC cuttings from cyclone, wet or dry. 5-foot sampling, intervals. 100% of sample sent to laboratory. At laboratory, 100% of sample dried, primary and secondary crush to -32 mesh. 1 kg split of crushed material taken and pulverized to pulp. Minimum 1 assay ton split from pulp for assay.

Gustavson added the data for the 2018-2019 drill hole program to the database according to its own data entry and review processes. A few inconsistencies in data entry were identified during this process, checked with Viva geologists, and resolved. Gustavson is of the opinion that the resulting assay database is appropriate for use in resource estimation.

12.6 Data Adequacy

The lack of downhole survey control for many of the historical drill holes will introduce location uncertainty for early sampling at the project. The current resource model utilizes 6 x 6 x 6 meter blocks. At this resolution, location uncertainty due to downhole survey control is not expected to have a material impact on resource estimation. However, it is noted that delineation and estimation of discrete high-grade zones could be impacted by location uncertainty.

It is expected that presence of coarse gold will increase the local variability of gold samples in the database, but no bias has been demonstrated in several reviews of the data.

Consistent with Gustavson's previous recommendations, project coordinate systems have been converted to NAD83 UTM meters, starting from original data where possible to resolve any outstanding survey control issues, to re-align the project, survey systems, and permit filings in a single coordinate system. This should allow for reduced survey error rates going forward, and aligns the project survey systems with the filing standards for the State of Nevada. Care should continue to be taken to confirm survey locations where errors may have crept in during the conversion process. Gustavson recommends that a site survey be conducted to locate new drill roads and drill pads at a high precision, and to locate significant landmarks and historical drill holes where possible to confirm the coordinate transformation.

Gustavson considers that the drill data are generally adequate for resource estimation.

13 Mineral Processing & Metallurgical Testing

Several scoping-level metallurgical studies were undertaken by mining companies from 1990's to 2009 for the Tonopah property. The test work included gravity separation, flotation and cyanidation leaching.

Viva has conducted additional test work to update and expand upon earlier results.

13.1 Metallurgical Results from Pre-Viva Testing

13.1.1 Kennecott Cyanide Shake Tests

There is reference in the project records of 350 cyanide shake tests having been carried out for some 350 assays pulp. According to the MDA 2002 report, the data included all rock types, alteration types, and grades, and the mean extraction was 67% of gold on an average sample grade of 0.045 oz. Au/t. This data is considered to indicate that cyanide extraction is likely to be a suitable recovery methodology, but cannot be used to provide exact recoveries for plant operations. The reason is that cyanide shake tests are performed on finely ground pulps (P_{80} = 150 mesh) and the plant operating conditions are going to be different.

13.1.2 McClelland 1995 Cyanidation Test Work

McClelland Laboratories conducted two cyanidation leach tests in October 1995 on a composite of drill core material from MW-87, 128-176.4 feet. One was on 80% passing 200 mesh material that yielded 90.8% gold recovery and 35.5% silver recovery and one on 80% passing 10 mesh yielded 49.6% gold and 14.8% silver recovery. The results indicated that the gold is recoverable by cyanide, and that recovery is likely influenced by grind size. The tests consumed 0.60 lbs/t and 0.15 lbs/t cyanide and 8.9 lbs/t and 5.0 lbs/t lime respectively.

The bulk densities were determined for 15 samples. The average bulk density was reported to be 139.2 lb/ft^3. The specific gravity of the samples was 2.23.

13.1.3 1995 Dawson Metallurgical Laboratories Test Work

Dawson Metallurgical Laboratories, Inc. conducted additional cyanide leach tests to determine the optimal leach time versus reagent consumption. Dawson determined that 82% of the gold was recovered from the sample in the first 24 hours, with a maximum 88% gold extracted after 96 hours of leeching. These results confirmed that cyanide leach process will recover gold.

13.1.4 1996 Rocky Mountain Geochemical Cyanide Shake Test Work

Test work was carried in February 1996 out for Kennecott by Rocky Mountain Geochemical Corp. on a single composite, designated MW-121, with a head grade of 0.105 oz, Au/t, and calculated head grade of 0.114 oz. Au/t. This composite was selected to determine why "sulfide mineralization intervals in this hole indicated low gold recoveries [while] bottle roll test work on the composite yielded 87.8% gold extraction in 96 hours." Cyanide shake tests were run on the composite sample for 1 and 24 hours. The test showed that 50% of the gold extracted after 1 hour and 100% after 24 hours. Hence, it was concluded that "the sulfide mineralization in MW-121 is not refractory [but] requires a longer leach period to extract all the cyanide soluble gold via the standard shake tests."

13.1.5 Newmont 2003 Test Work

Newmont's test work concentrated on proper sampling of core or RC drill samples. Significant differences were noted between fire assay and metallic assay values due to nugget effect. The study concluded that for the RC drill samples, the entire sample needs to be crushed in the laboratory and no splitting at site. Also, recommendations were made that metallic assay procedure should be used for gold values greater than 5 ppm Au.

13.1.6 2006 McClelland Laboratories Gravity Test Work

The test work, performed on composite samples assaying 0.6 g/t to 34 g/t Au, evaluated the response of gold recovery vs. feed grade using gravity concentration. This test work demonstrates a direct correlation between head grades and recovery from gravity concentration. Higher head grade achieved a higher gold recovery as shown in Table 13-1.

Table 13-1: McClelland Laboratories Gravity Test Work Results

Sample	Au Recovery % of Contained Values Nominal Grind Size			Au Recovery Total (%)	Calculated Head Grade, g Au/st	Direct Head g Au/st
	-420mm	150mm	75mm			
Comp. 12	72.5	7.6	8.1	88.2	36.17	34.40
Comp. 13	35.5	12.6	7.7	55.8	8.73	10.36
Comp. 14	23.8	8.2	7.7	39.7	3.44	3.44
Comp. 15	6.1	8.4	32.8	47.3	1.03	0.59

13.1.7 2006 SGS Lakefield Test Work

SGS undertook test work on a single composite involving pre-concentration by floatation and gravity concentration, and cyanidation of pulverized material as well as cyanidation of concentrates plus tails in 2006.

The composite sample assayed 6.45 ppm Au, 82.0% SiO_2, and 0.05% S. Initial gravity concentration testing resulted in the recovery of 15.4% of the total contained Au. Cyanidation of the gravity tails resulted in combined gravity and cyanidation recoveries ranging between 89.5% and 94.1%.

A series of gravity concentration-floatation tests were performed using the composite sample. Combined gravity/floatation test gold recoveries given in Table 13-2 ranged from 43.6% to 52.5%.

Table 13-2: 2006 SGS Gravity and Rougher Con Flotation Test Work Results

Test #	Product	Product Wt. %	Assay Au (g/t)	Au Distribution (%)
F1	Ro Conc. 1-4	12.1	19.7	46.9
	Head (calc.)		5.07	
	Grav + Ro Conc.	12.2	27.7	55.2
F2	Ro Conc. 1-4	7.47	29.5	45.9
	Head (calc.)		4.8	
	Grav + Ro Conc.	7.38	42.5	54.2
F3	Ro Conc. 1-4	11.5	18.9	43.6
	Head (calc.)		4.99	
	Grav + Ro Conc.	11.5	27.3	52.2
F4	Ro Conc. 1-4	10.2	26.5	52.5
	Head (calc.)		5.17	
	Grav + Ro Conc.	8.9	37.3	59.8

Base line cyanidation tests were also performed on the gravity tailings. Depending on grind size, gold recoveries in cyanidation were higher than the gravity-flotation series, ranging from 87.5% to 93.1%. When the gravity concentration was included, overall recovery increased to 94.1%. These results are summarized in Table 13-3.

Table 13-3: SGS Gravity + Cyanide and Cyanidation Test Work Results

Leach Test #	Feed Size K_{80} (µm)	Reagents (g/L, kg/t)			Leach Time (hrs.)	Au Recovery %			Au Tails Grade (g/t)	Au Head Grades (g/t)			
		NaCN Added	Consumption (kg/t)							Calculated		Direct	
			NaCN	CaO		CN	Grav	Grav+CN		CN	Grav	CN	Grav
CN1	169	0.5	0.03	0.38	48	93.1		94.1	0.38	5.49		5.32	
CN2	60	0.5	0.02	0.44	48	87.5	15.4	89.5	0.72	5.74	6.28	5.32	6.45
CN3	60	0.5	0.03	0.42	72	91.4		92.7	0.45	5.23		5.32	
CN4	169	1	0.01	0.39	48	92.5		93.6	0.43	5.64		5.32	

13.1.8 2007 Barrick Goldstrike Metallurgical Services Test Work

In June 2007, a single sample was submitted to Barrick Goldstrike Metallurgical Services for a Bond work index and to determine direct Carbon in Leach gold recovery. Sample head grades, recovery and consumptions are reported in Table 13-4.

The gold extraction of 91% was achieved in the CIL test. The sample had a Bond's ball mill work index of 19.06 thereby indicating the composite material was hard.

Table 13-4: Barrick Goldstrike CIL Gold Recovery Results

Standard ACIL Results	Head/Tail Assays			Head Reconciliation (%)	Au Recovery		Sulfide Sulfur			Carbonate		TCM	
	Head Assay (oz Au/t)	Calc. Head (oz Au/t)	Tail Assay (oz Au/t)		Calc. Recovery (%)	Head-Tail Recovery (%)	Head Sulfide Sulfur (%)	Tail Sulfide Sulfur (%)	Sulfide Oxidation Mass Bal (%)	Head CO^3 (%)	Tail CO^3 (%)	Head TCM (%)	Tail TCM (%)
Sample Name													
050807/1	0.242	0.200	0.018	17.3%	91.01	92.56	0.02	0.02	0.84	0.05	0.15	0.04	0.05
050807/2	0.242	0.210	0.019	13.2%	90.96	92.15	0.02	0.02	0.53	0.05	0.05	0.04	0.06
Average	0.242	0.205	0.019	15.2%	90.98	92.36	0.02	0.02	0.69	0.05	0.10	0.04	0.06

Standard ACIL Results	NaCN			Ore/Slurry				Carbon	
	Addition (lb NaCN/ t ore)	Residual (lb NaCN/ t sol'n)	Consumed (lb NaCN/ t ore)	Mass of Head/Feed Sample (g)	Mass of Dry Tails (g)	Total Slurry Mass (g)	Mass of Solution (g)	Carbon Au Assay (oz Au/t)	Mass of Carbon (g)
Sample Name									
050807/1	5.00	2.70	-0.04	200.00	198.3	571.4	373.1	2.989	12.09
050807/2	5.00	2.61	0.14	200.00	198.9	571.4	372.5	3.145	12.09
Average	5.00	2.66	0.05						

13.1.9 Gekko Systems Test Work

In 2008 and 2009, Midway commissioned Gekko Systems to carry out work on gravity concentration and floatation test work on a single composite. The grade of the sample tested was 22.7 ppm Au and 16.3 ppm Ag. The response of the material to the gravity separation, on a shaking Wilfley table, improved at finer crush sizes. The test 100% passing 450µm showed that 35.5% Au recovery could be achieved into 1.2% of the feed mass. Finer grinding before gravity was shown to improve the gravity recovery at the expense of higher dilution in the concentrate. A gravity circuit inside the circulation load of a mill was recommended.

The flotation response was considered acceptable, however, 2.6 ppm Au was still present in the tails from the best flotation result. Diagnostic test of the flotation tail indicates that cyanide soluble leach recoveries of up to 88% are also achievable for final tail of 0.30 ppm at a grind of 40 µm.

13.2 Viva Gold 2018-2019 Metallurgical Test Work

13.2.1 2018 RDI Cyanide Leach Testing

In late 2018, Viva collected samples from previous drilling campaigns which had been retained by previous operators for bucket leach testing by RDi. The samples tested were assembled from a number of core intercepts from the Company's core inventory, averaging between 0.5 and 1.0 gram per tonne ("g/t") grade. Assayed head grade for the TV sample was 0.88 g/t gold and 0.72 g/t for the OPA sample. The samples were subject to static bucket leach tests for a nominal 1-inch crush material size and bottle roll tests for material ground to nominal 6-mesh and 200-mesh sizes. Results from this work are shown in Table 13-5:.

Table 13-5: RDi Cyanide Leach Test Summary

Composite	Particle Size	Leach Time	Gold Extraction (%)	Silver Extraction (%)
TV	1"	20 days	18.4	9.0
	6 mesh	120 hours	29.7	24.9
	200 mesh	48 hours	91.9	41.6
OPA	1"	20 days	55.5	7.7
	6 mesh	120 hours	51.0	17.9
	200 mesh	48 hours	93.5	41.9

Review of the bucket leach testing shows that the Tv material in holes available as core, and thus selected to provide material for the composites were generally comprised of quartz rich tertiary volcanics from the Discovery zone. As such, it is possible that this material is not fully representative of the project metallurgy. Ongoing test work should properly identify and characterize the material selected for testing so that it can be more fully representative. These preliminary results indicate a potentially strong crush/grind size versus gold recovery relationship, at least in the limited number of samples tested.

13.2.2 2019 McClelland Laboratories Metallurgical Testing

Viva submitted reverse circulation drill chip samples from the 2018 drilling program to McClelland Laboratories, Inc. for metallurgical testing. The test program consisted of head analyses of twenty interval composites, bottle roll tests, preparation of four master composites and bottle roll and column testing of these composites. Gravity concentration tests were also performed on the master composite samples.

The original twenty composites were segregated by drill hole, rock type and depth and represented Palmetto Argillite (OPA) and the three different tertiary volcanic lithologies (TRT, TRV and TVS). The argillitic mineralization comprises approximately 50% of the deposit.

The test results are presented in Table 13-6: and Table 13-7:. The details of the test work are given in the report titled “Report on Bottle Roll and Column Leach Testing – Midway Drill Core Composites, McClelland Laboratories, Inc. December 20, 2019”.

Table 13-6: McClelland Bottle Roll Tests

Composite	Rock Type	Au Recovery, %	gAu/mt feed Extracted	Tail	Calculated Head	Head Assay
TG1806	OPA	55.6	0.20	0.16	0.36	0.35
TG1807C	OPA	61.3	0.73	0.46	1.19	0.98
TG1808B	OPA	64.5	0.20	0.11	0.31	0.32
TG1808C	OPA	79.9	1.31	0.33	1.64	5.92
TG1808D	OPA	81.7	0.85	0.19	1.04	0.90
TG1809A	OPA	51.2	0.44	0.42	0.86	0.91
TG1809B	OPA	85.0	0.68	0.12	0.80	0.64
TG1810B	OPA	48.3	0.14	0.15	0.29	0.33
TG1807A	TRT	47.6	0.20	0.22	0.42	0.36
TG1812B	TRT	72.2	0.26	0.10	0.36	0.30
TG1814A	TRT	52.8	0.28	0.25	0.53	0.42
TG1810A	TRV	42.6	0.52	0.70	1.22	1.36
TG1811A	TRV	71.4	0.20	0.08	0.28	0.30
TG1811B	TRV	68.3	0.28	0.13	0.41	0.38
TG1812A	TRV	68.4	2.21	1.02	3.23	3.07
TG1815	TRV	45.5	0.65	0.78	1.43	1.38
TG1807B	TVS	56.9	1.11	0.84	1.95	1.73
TG1813	TVS	59.2	0.29	0.20	0.49	0.44
TG1814B	TVS	65.0	1.78	0.96	2.74	2.47

Table 13-7: McClelland Composite Column Leach Results

Master Composite	Rock Type	Au Recovery, %	gAu/mt feed Extracted	Tail	Calculated Head	Head Assay
4394-OPA	OPA	83.3	1.80	0.36	2.16	1.98
4394-TRT	TRT	47.9	0.23	0.25	0.48	0.38
4394-TRV	TRV	60.8	0.79	0.51	1.30	1.35
4394-TVS	TVS	63.8	1.11	0.63	1.74	1.68

The test results indicate the following:

- The twenty composite samples assayed 0.32 g/t Au to 11.77 g/t Au and 1.0 g/t Ag to 13.4 g/t Ag.
- The master composite gold grades ranged from 0.38 g/t to 1.98 g/t Au and silver grades ranged from 2.8 g/t to 5.5 g/t Ag.
- The bottle roll tests were performed on as received feed size (nominal 1.7 mm). The actual feed sizes of the master composite were determined to be 66% to 76% passing 1.7 mm.
- The gold extraction in the bottle roll tests for 96-hour leach time (4 days) ranged from 48.3% to 89.1% (average 68.5%) for OPA samples, 47.6% to 72.2% (average 57.5%) for TRT samples, 42.6% to 71.4% (average 59.2%) for TRV samples and 56.9% to 65% (average 60.4%) for TVS samples.

- The extraction results indicated that the gold was still leaching when the tests were terminated. This was confirmed when comparing column results with bottle roll results for the identical sample. Column results were generally higher than bottle roll extractions.
- The argillite material had significantly higher gold extraction than the other material types.
- The silver recovery was low for all material types and ranged from 4.8% to 41.8%.
- Agglomeration strength and stability tests on the four master composites indicated cement addition between 3.6 kg/t and 5.2 kg/t would be sufficient for producing stable agglomerates and minimizing percolation issues during column leach tests.
- Column leach tests on agglomerated "as received" master composites were run for 57 to 66 days at an application rate of 0.005 gpm/ft2 (12 lph/m2) of 1.0 g/L of NaCN. The test results indicated that OPA, TRV and TVS master composites were amenable to simulated heap leach cyanidation treatment. Gold recoveries ranged from 60.8% to 83.3% in 57 to 66 days of leaching.
- Gold recovery was lower at 47.9% for TRT material.
- The majority of the gold leached in 10 to 20 days. However, leaching continued at a slower rate for all master composites except TRT. Extending the leach cycles would have resulted in slightly higher gold recoveries.
- Silver recovery was low for all composites and ranged from 6.9% to 21.4%
- Cyanide consumption was moderate at 0.58 to 0.99 kg/t of feed.
- Gravity concentration tests indicated varied response for the master composites at a particle size of P80 of 212 micrometers. The cleaner concentrate recovered 14.6% to 54.4% of the gold at a concentrate grade of 29.3 to 605 g/t Au. However, silver recovery ranged from 1.2% to 13.6%..

13.2.3 Cyanide Leach Assays

Beginning in 2018, Viva gold added cyanide shake leach assay (whereby a split of the prepared sample pulp is agitated in cyanide solution and the resulting pregnant solution is assayed by AA methods) to the assay suite for exploration samples. The objective of this program is to provide data to determine variability of cyanide recovery through the deposit. In some operations, cyanide shake assays have been found to be useful in predicting cyanide leach recoveries, particularly where there is high variability of recovery for different material types. These data are being maintained as part of the project database, but full evaluation has been delayed pending acquisition of a sufficient number of data points. Gustavson recommends that this data be compared with and registered against the historical Kennecott database of similar sampling technique and evaluated to determine whether it demonstrates measurable variability in recovery by lithology or alteration type.

13.3 Metallurgical Test Work Summary

The historical metallurgical data indicate that gold and silver mineralization from the Tonopah project are amenable to recovery by cyanide leaching. Test work was completed on both fully oxidized and sulfide samples, with little difference noted in recoveries. It is noted in some of the test work that coarse gold present in samples maybe contributing to delayed recovery in cyanide solution.

Gravity pre-concentration was recommended in some of the early studies to segregate coarse gold from the material prior to cyanide leaching. Gravity testing indicates that gravity methods might be useful for pre-concentration, particularly in higher grade materials.

Flotation test work completed on high grade gold samples, indicates that a high percentage of gold can be recovered to concentrate by froth flotation. However, the test work appears to indicate that gravity/cyanidation showed better performance than flotation/cyanidation.

Gustavson believes that sufficient preliminary metallurgical data exists to support determination of cutoff grade for resource estimation, as well as recovery models for scoping-level studies. The cyanidation shake data (Kennecott, Section 13.1.1) appears to be taken from a variety of grades, lithologies and mineralization types. However, the remaining pre-Viva metallurgical data were focused on a limited number of samples of higher grade vein material, and may not be representative of the deposit as a whole.

The Viva Gold metallurgical test work data confirms that gold mineralization from the project is amenable to recovery by cyanide leaching. Argillite material appears to have a somewhat better response than volcanic material, and initial results indicate that there is likely a relationship between particle size and recovery (with smaller size particles showing higher amenability to cyanidation.) It is also noted that the bottle roll test work showed increasing recoveries at the 96-hour timeframe, which generally indicates that leaching was not yet complete. Future bottle roll tests might use a longer time window to allow for additional recovery.

For Scoping level studies, including this PEA, the McClelland work indicates an average recovery of 83% for Argillite material and 58% for Volcanic material. These values are not discounted from the column data because recovery was ongoing at the end of the column leach testing.

Gustavson recommends that further test work be focused on column leach testing for various identified material types, and preferably at different crush sizes, with the objective of predicting heap leach recoveries and testing relationships between comminution and recovery.

14 Mineral Resource Estimates

14.1 Geologic Model

The primary lithologic model for the Tonopah Project consists of Ordovician Palmetto formation Argillites disconformably overlain by Tertiary volcanic rocks. Much of the resource area is further covered by Quaternary gravels. Gustavson used Leapfrog software to aggregate Palmetto formation, Volcanics, and Gravels within the drill holes for the project, and used the contact points to generate surfaces which represent the contacts between each of the lithologic domains. The contact surface for the top of the Palmetto shows a steep incline to the north which runs generally parallel to the regional Walker Lane Trend, along with two conjugate offsets which correspond to the Discovery and Dauntless zones identified in drilling.

Review of long sections and cross sections through the deposit show that mineralization generally follows the contact between the Palmetto and the overlying volcanics, with most of the mineralization occurring within the lower portion of the volcanics and the top of the argillite, except in areas of structural complexity, where feeder structures may exist within the argillite.

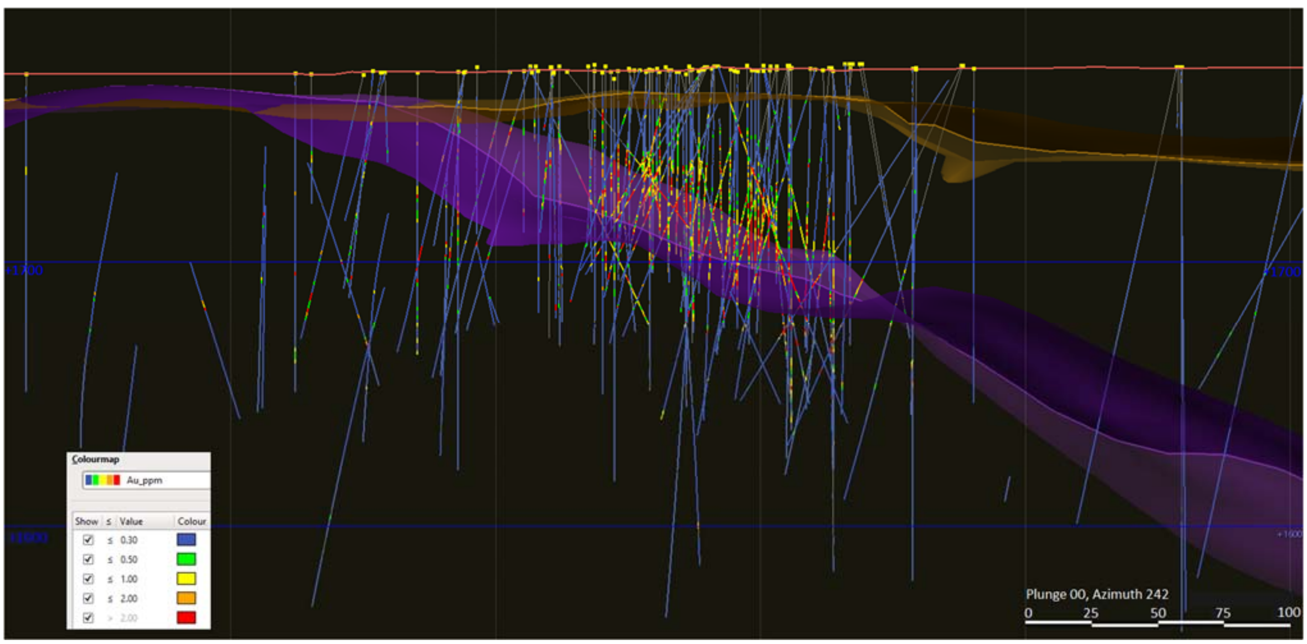


Figure 14-1: Cross section slice showing Opa upper surface (purple) and Tv Upper Surface (orange). Az 242, Scale in meters.

Mineralization is interpreted to primarily occupy zones of favorable structural preparation and lithogeochemical host rock within the lower Tertiary volcanics and underlying Palmetto formation, in an overall trend parallel to the Walker Lane Trend.

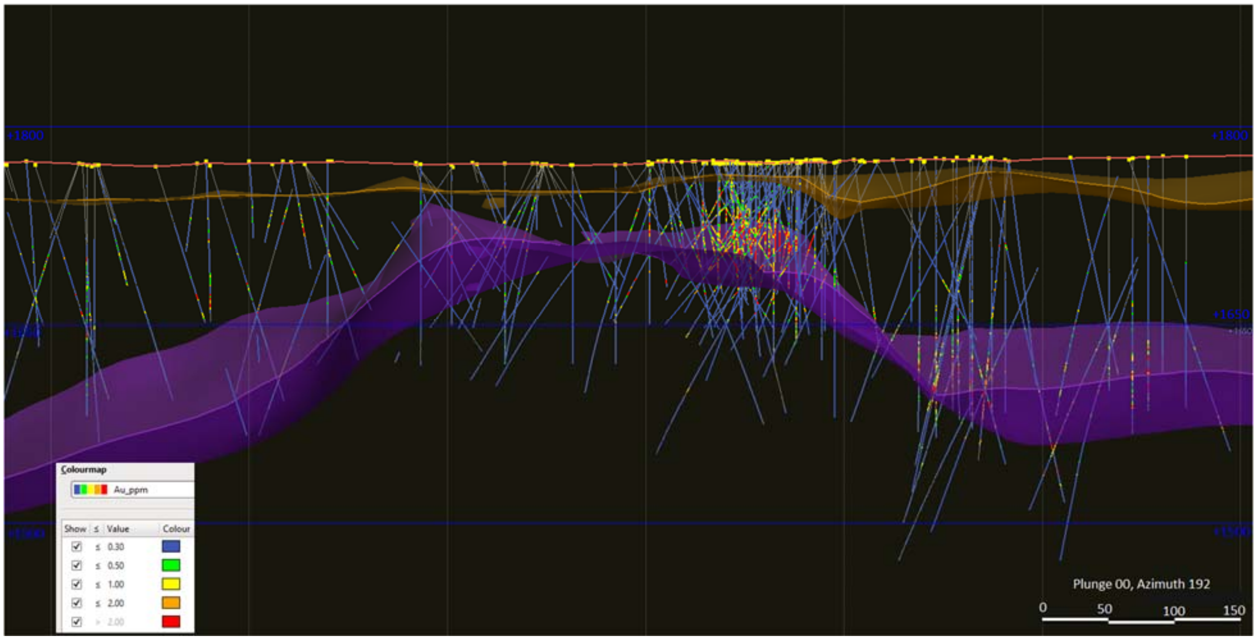


Figure 14-2: Long section view showing mineralization relative to Opa /Tv boundary surface. Az 192. Scale in meters.

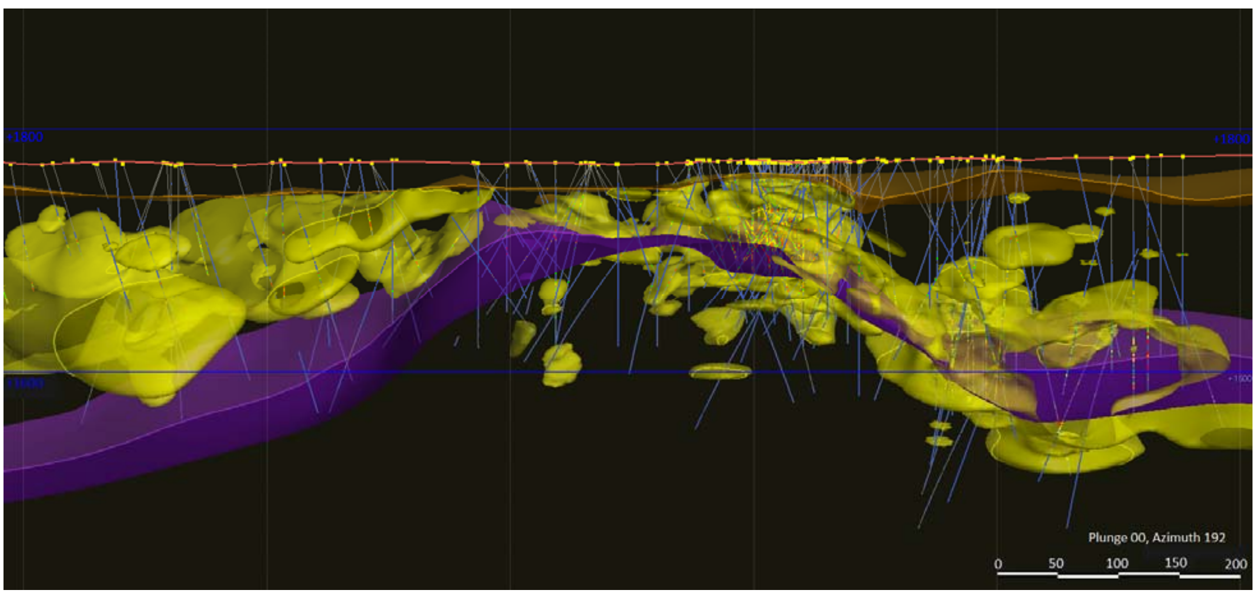


Figure 14-3: Long Section view showing interpreted Primary Zone Domain (PZD, Yellow) log grade shell at 0.150 ppm Au. Azimuth 192, Scale in meters.

14.2 Domains for Resource Estimation

Based on observation that the primary mineralization trend appears to follow the Opa /Tv contact, Gustavson created a series of grade shells in Leapfrog Mining software based on gold grade, using the Opa /Tv contact as a trend surface to guide the interpolation of the solids models. A high anisotropy (8:8:1) was used to counterbalance the complex drilling pattern and orientation, and a log grade shell was estimated to limit projection of the grade shell into areas of sparse drill density. A number of grade shells were considered, at several cutoffs, and using a series of different parameters. The final shell selected is a log shell at 0.150 ppm, which has a good balance between continuity of mineralization without over-projecting grades. This shell is referred to as the primary zone domain (PZD).

A secondary shell was created to allow for separate treatment and understanding of higher-grade mineralization which trends parallel to conjugate extensional structures, particularly the Discovery and Dauntless structures. This shell is a log shell at a 0.8 ppm threshold, with interpreted surfaces for the Discovery and Dauntless structures as trend surfaces for control. The shell is clipped to an area of influence surrounding the Discovery and Dauntless structures. This shell is referred to as the conjugate zone domain (CZD).

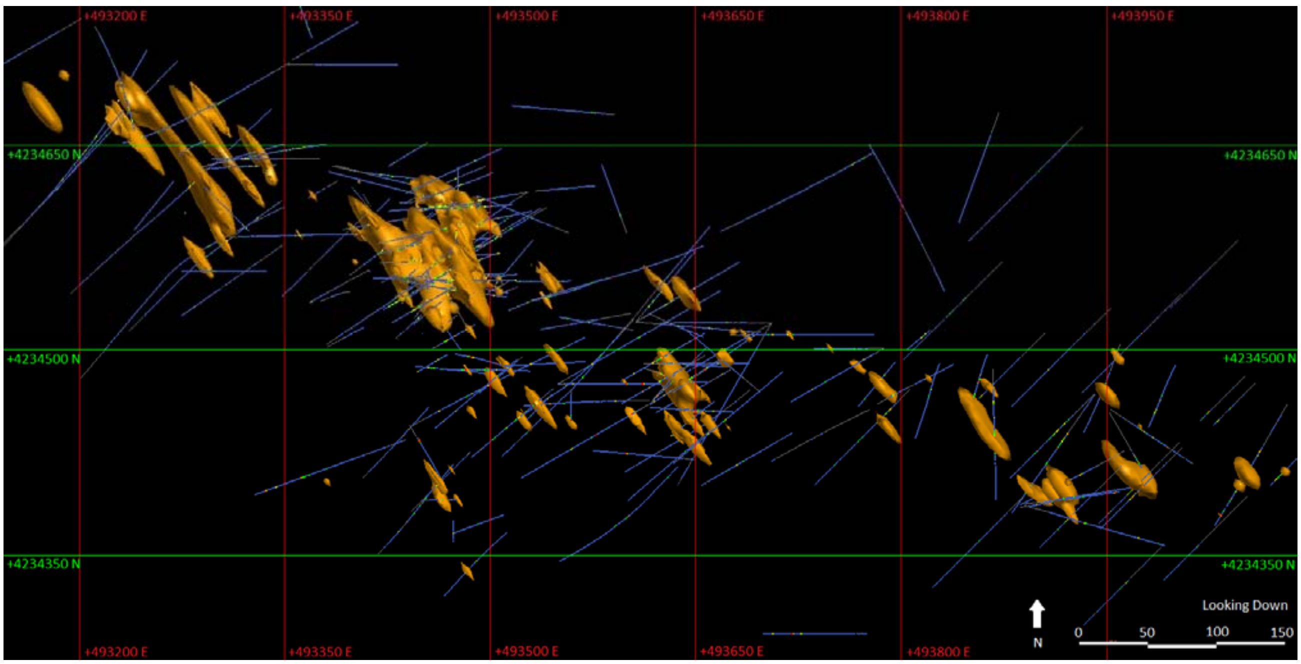


Figure 14-4: Plan view (North is top of page) showing CZD solids (orange). Scale in meters.

The CZD is a subset of the PZD. The subset of the PZD which excludes the CZD is referred to as PZX.

Both domains are clipped at the Tv / Gravel surface, as no mineralization is projected to occur in the overlying gravels.

14.3 Domain Statistics

Drill holes were composited at 2 meter intervals downhole for statistical analysis. and for resource estimation.

Statistical analysis for deposit shows two distinct populations within the PZD, a higher-grade and lower-grade population. Figure 14-5 shows a clear multimodal distribution of Au composite data in the clustered data (left), which resolves into a bimodal distribution in the declustered data (right).

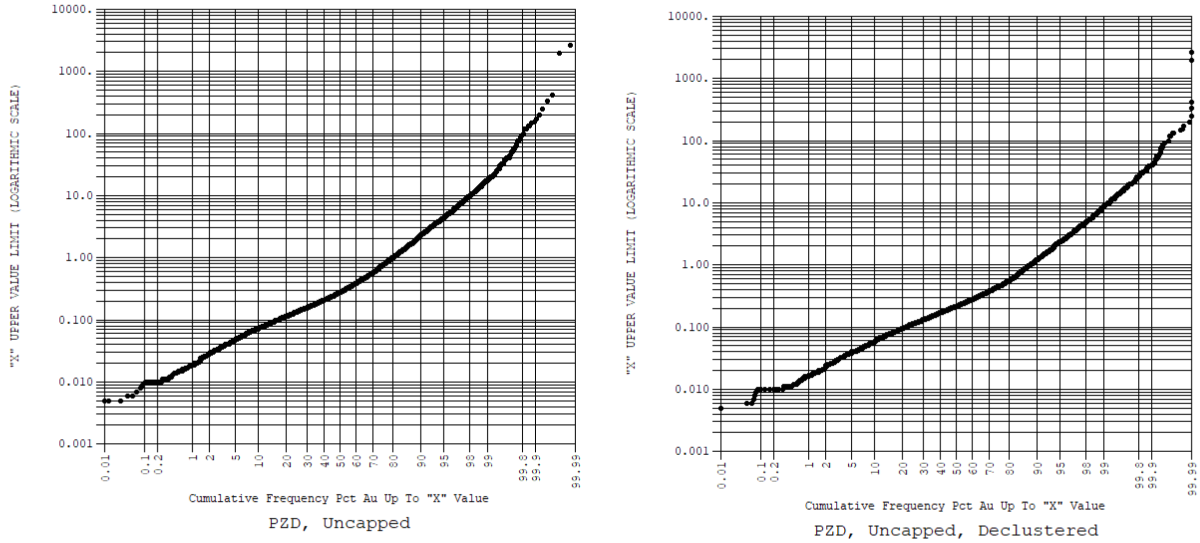
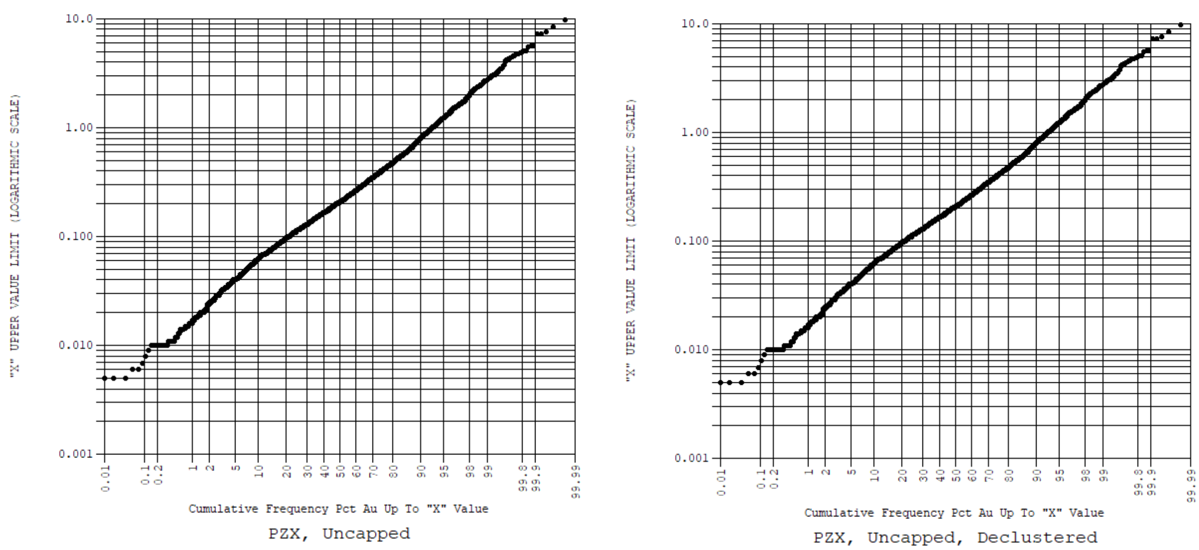


Figure 14-5: Cumulative Frequency Plots, PZD (both domains)

Separation of the deposit into PZX and CZD domains separates the populations, thus demonstrating that it may be reasonable to treat the two data distributions separately for estimation. Figure 14-6 shows composite data for the zones separately. The declustered data (top right) for PZX still shows a relic of the upper data population related to additional conjugate structures which are not captured in the Discover / Dauntless solids area. The CZD declustered data (lower right) shows a clear lognormal distribution of the data population isolated in the CZD. The non-declustered data set, particularly for CZD, is impacted by the large number of closely spaced drill holes in the higher-grade portions of the Discovery & Dauntless zones which comprise the CZD.



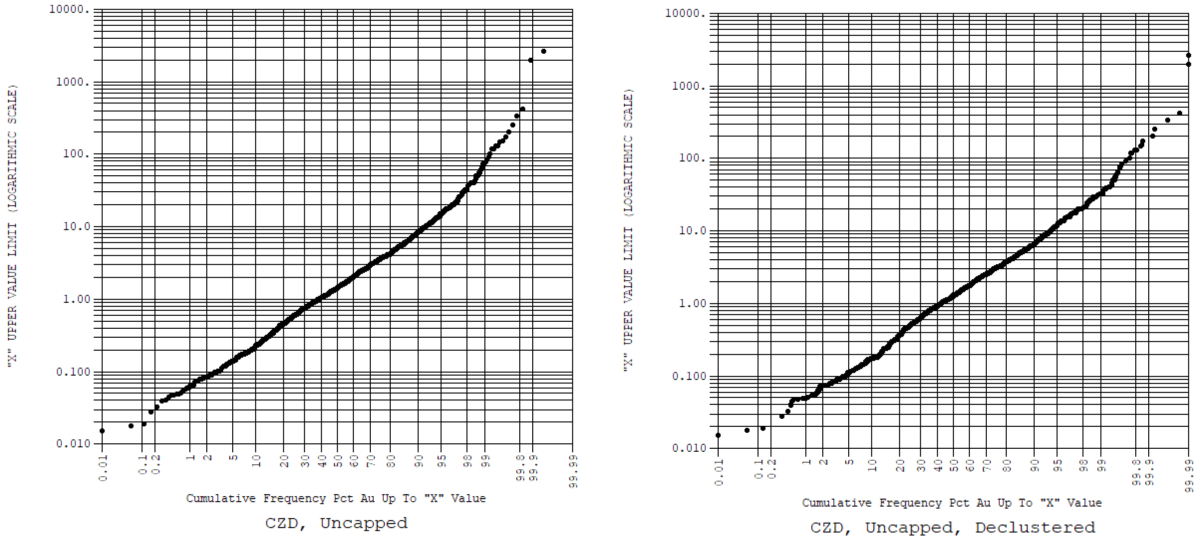


Figure 14-6: Cumulative Frequency Plots:, PZX and CZD domains

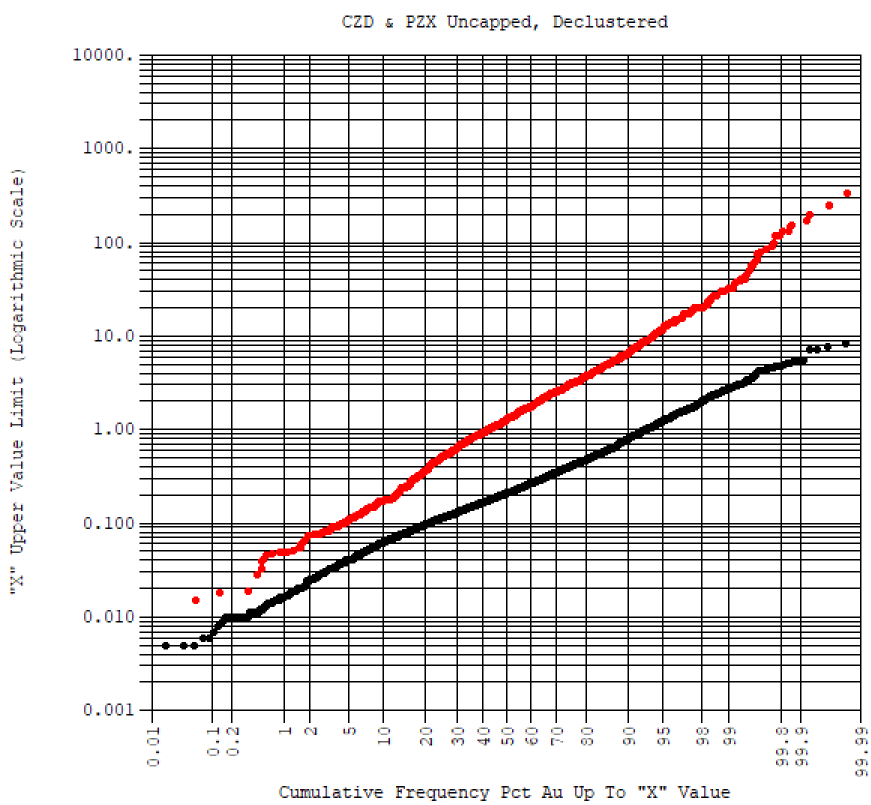


Figure 14-7: CZD and PZX domains Cumulative Frequency Plots

Figure 14-7 shows the cumulative frequency distribution for composites within the CZD and PZX domains, clearly showing the two separate populations. Table 14-1 shows capped, uncapped, declustered, and non-declustered composite statistics by zone for CZD, PZX, and combined PZD domains for reference.

Table 14-1: Composite statistics by zone

Domain	Capping	Number of Samples	Min Value	Max Value	Mean Value	Median Value	Standard Deviation	Coefficient of Variation
CZD	Capped	1220	0	20.000	3.189	1.418	4.54	1.424
	Uncapped	1220	0	3400.103	7.535	1.322	99.161	13.16
PZX	Capped	4779	0	5.000	0.473	0.235	0.735	1.553
	Uncapped	4779	0	191.054	0.558	0.235	3.027	5.423
PZD	Capped	6025	0	5.000	0.779	0.291	1.19	1.527
	Uncapped	6025	0	3400.103	1.969	0.291	44.79	22.743

14.4 Capping & Compositing

Capping is applied separately for CZD and PZX domains prior to estimation. Capping parameters for CZD are based on the CFP's shown in Figure 14-6 (lower left and lower right). There is a shift in the grade population at 20 ppm Au. Accordingly, CZD grades are capped at 20 ppm Au. The CFP for PZX domain indicates a cap of around 15 ppm Au may be optimal. However, because PZX domain appears still to include a separate data population, a more conservative capping parameter of 5ppm Au was chosen.

5% of composites within the CZD domain and 2% of composites within the PZX domain are capped. Gustavson acknowledges that this capping regime may be conservative but considers it prudent at this time because of the risks imposed by more closely clustered drilling in the higher-grade portions of the deposit.

14.5 Geostatistics

Preliminary variography in PZD (Figure 14-8) shows a general trend which parallels the long axis of the mineralization, parallel to the Walker Lane, bearing 120, 0 plunge. However, there is an anomaly in the variogram showing high variability at short range. This is interpreted as the influence of the conjugate 340-degree to 360-degree structural trend, which appears to host very high grade mineralization.

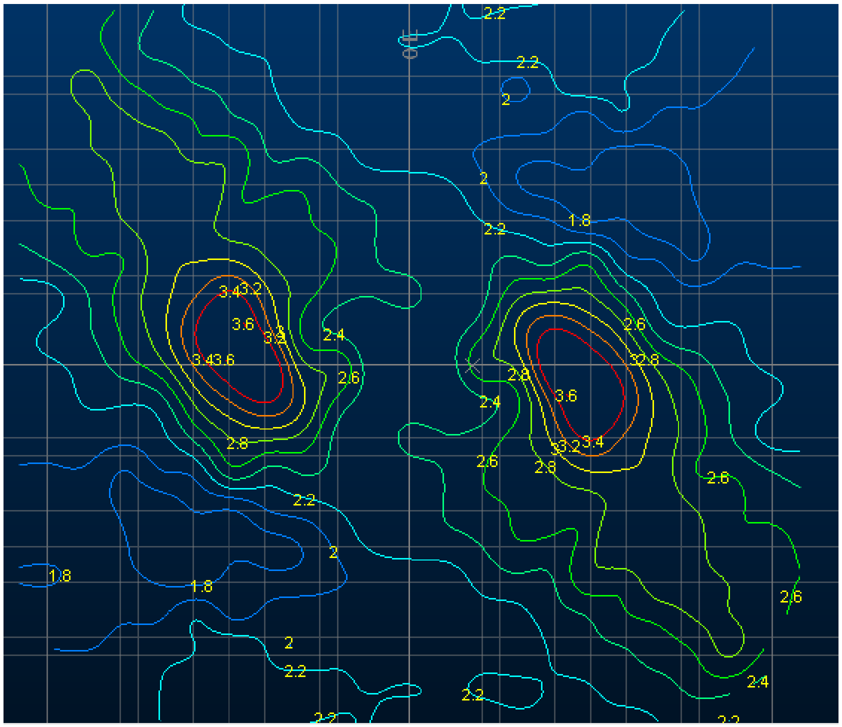


Figure 14-8: PZD Variogram Map, Plan View

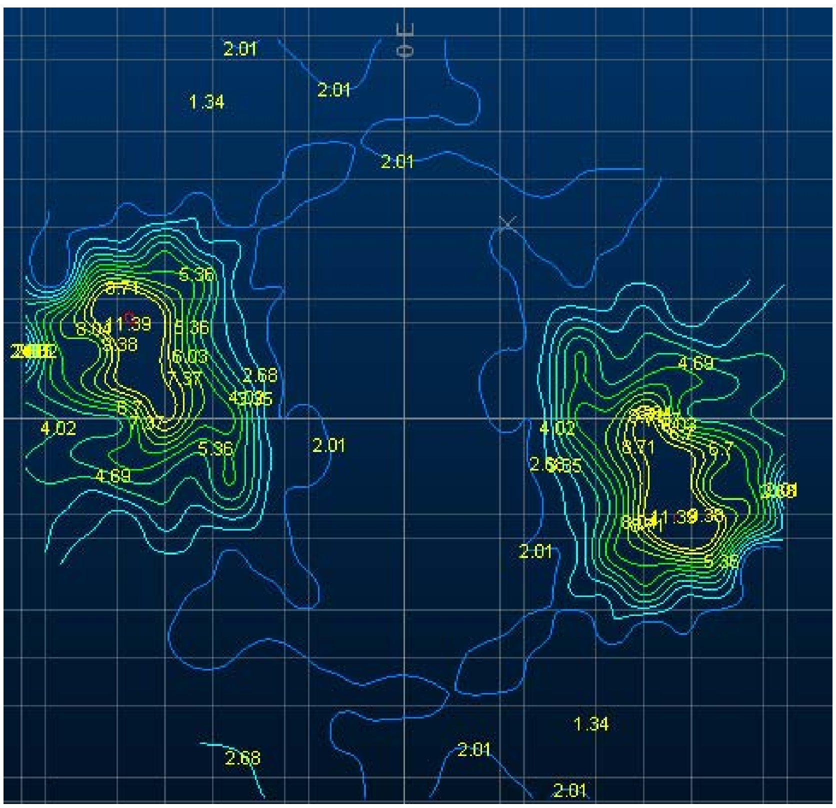


Figure 14-9: CZD variogram map, plan view

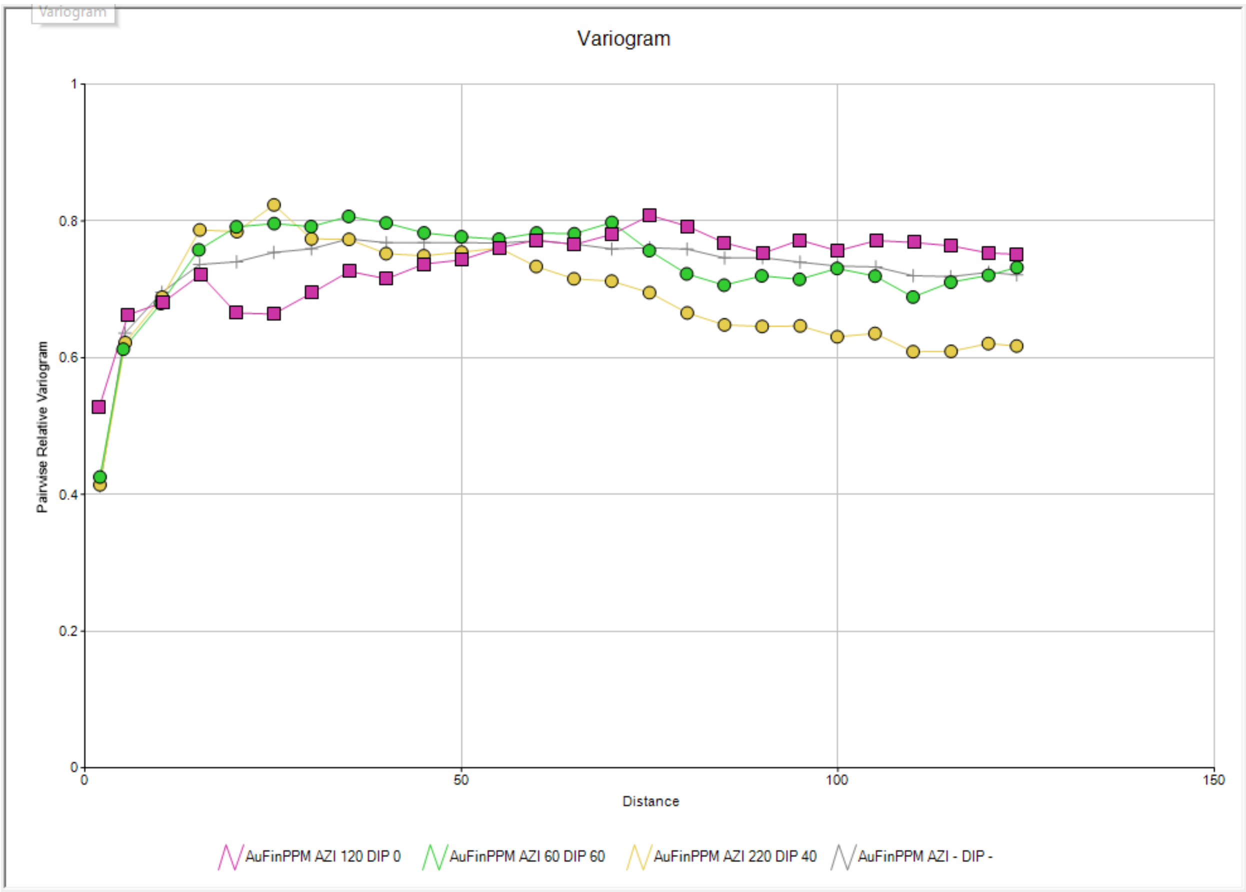


Figure 14-10: Experimental Variograms, PZX zone

Table 14-2: Experimental Variogram Ranges

Composites Set		Bearing	Plunge	Nugget	Type	Sill	Range
PZX	Omni	-	-	.5	SPH	.5	40
	Dip Direction	60	60	.5	SPH	.5	20
	Along Strike	120	0	.5	SPH	.5	40
	Across Strike	220	440	.5	SPH	.5	25
CZD	Omni	-	-	.6	SPH	.4	15
	Dip Direction	90	80	.6	SPH	.4	17
	Along Strike	0	10	.6	SPH	.4	14
	Across Strike	90	-10	.6	SPH	.4	7

14.6 Block Model Parameters

The block model used for resource estimation is a 6 x 6 x 6 meter, orthogonal, non-rotated block model, which is selected as consistent with the likely open pit mining method for the property. Blocks are flagged from the PZD solids, with only blocks inside the domain being estimated. CZD contribution to the block grade estimate is estimated by conjugate zone indicator model, so the CZD subset model is a soft boundaries model.

14.7 Block Grade Estimation Methodology

Block grade estimation was completed using Datamine RM software. Grade estimates use ordinary kriging, with an indicator model used to determine impact of the conjugate zone on each block area.

Estimation of block grades uses a two-pass indicator estimation in order to accommodate the two overlapping grade populations and orientations. An indicator model is estimated to determine the level of influence of the higher-grade conjugate zone mineralization, to restrict the influence of this higher-grade mineralization to the shorter variogram range displayed for the higher grade zone, and to honor the geometry of the conjugate zone by orienting search and kriging parameters for the conjugate estimate according to the variography of the zone.

Step 1: Estimate PZD grades using capped PZD composites, using variography from (PZD), and dynamic anisotropy derived from the Opa / Tv trend surface.

Step 2: Estimate conjugate zone indicator (CZ%) value using all PZD composites set to indicator values (>1.0 ppm =1, < 1.0 ppm = 0) to arrive at an indicator percentage for each block. This estimate uses PZD Indicator variography.

Step 3: Estimate all PZD blocks with CZD composites, using CZD variogram and search ranges.

Step 4: Flag all PZD blocks with SVOL code corresponding to classification methodology using all PZD composites and PZD variography.

Step 5: Aggregate block grades such that the CZD grade is applied to the CZ% of the block, with the remaining percentage having PZD grades applied. Where CZD grades are not estimated due to distance from CZD composites, CZ% is set to zero and the PZD grade is applied.

Parameters for block grade estimation are shown in Table 14-3.

Table 14-3: Block Grade Estimation Parameters

		Search Ranges (m)			Kriging Inputs	
Estimate		Measured	Indicated	Inferred	Nugget %	Sill %
PZX	Dynamic					
	Dip Direction	-	53	-	50	50
	Along Strike	-	80	-	50	50
	Across Strike	-	53	-	50	50
CZD	Strike 170, Dip -90					
	Dip Direction		25		60	40
	Along Strike		25		60	40
	Across Strike		10		60	40
Indicator	Strike 170, Dip -90					
	Dip Direction	-	25	-	50	50
	Along Strike	-	25	-	50	50
	Across Strike	-	10	-	50	50
Classification	Dynamic					
	Dip Direction	17	34	68		
	Along Strike	20	40	60		
	Across Strike	8	16	32		

Each grade estimate uses a single pass, with a minimum of 6 and a maximum of 20 2- meter composites used to estimate grades (and IND%). A maximum of 4 composites are used per drill hole, thus requiring at least two drill holes to contribute to each block estimate. Classification is flagged in three passes, with the same data requirements for each pass, thus SVOL 1 (measured) classification requires a minimum of 6 composites from 2 drill holes within a 17 x 20 x 8 meter search ellipse. Estimates use dynamic anisotropy to orient the PZD and Classification search ellipses parallel to the Palmetto / Volcanic contact, Indicator and CZD searches are oriented with a strike of 170 and a dip of 90 degrees.

14.8 Resource Classification

Resource Classification is based distance from data as compared to the variogram ranges determined for the PZD. Blocks within 50% of the variogram range of two drill holes are classified as measured. Blocks within 100% of the variogram range of two drill holes are classified as indicated. Blocks within 200% of the variogram range of two drill holes are classified as inferred. There are additional blocks within the domains which have no grades estimated, but which constitute targets for additional drilling.

14.9 Cutoff Grade

Cutoff grade used to meet the test of 'reasonable prospects for economic extraction.' Accordingly, the cutoff grade is estimated based on price and recovery assumptions. A separate cutoff grade is applied to the 2 rock types hosting mineralization as they have different recovery.

$$\text{Cutoff Grade} = \text{Cost} / (\text{Metal Price} * \text{Recovery})$$

Using $1400/ oz Au, 58% recovery in Volcanics and 83% recovery in Argillite, and a process cost of $6.20 per tonne yields a cutoff grade of 0.237 ppm (rounded to 0.250 ppm) and 0.166 ppm for Argillite (rounded to 0.200 ppm).

The parameters for the processing cost is based on the life of mine average cost of processing ($4.52/tonne processed), site G&A ($0.66/tonne processed), leach pad construction ($0.50/tonne processed) and contingency of ($0.52/tonne processed) . The above cutoff grades thus constitute reasonable prospects for economic extraction by this test.

14.10 Pit Shell for Resource Reporting

As a second test for 'reasonable prospects for economic extraction', Gustavson constrained the resource estimate within a Lerchs-Grossman pit shell to exclude discontinuous and peripheral areas of mineralization which are less likely to form part of a future mine plan. The parameters for the pit shell are in

Table 14-4.

Table 14-4: Resource Pit Shell Parameters

Parameter	Value
Gold Price (/oz troy)	$1600
Gold Recovery Argillite	83%
Gold Recovery Volcanics	58%
Mining Cost (per ton mined)	$1.55
Process Cost (Includes G&A and Leach Pad Allowance)	$6.20
Overall Highwall Angle	45 degrees
Highwall angle in Gravels	35 degrees

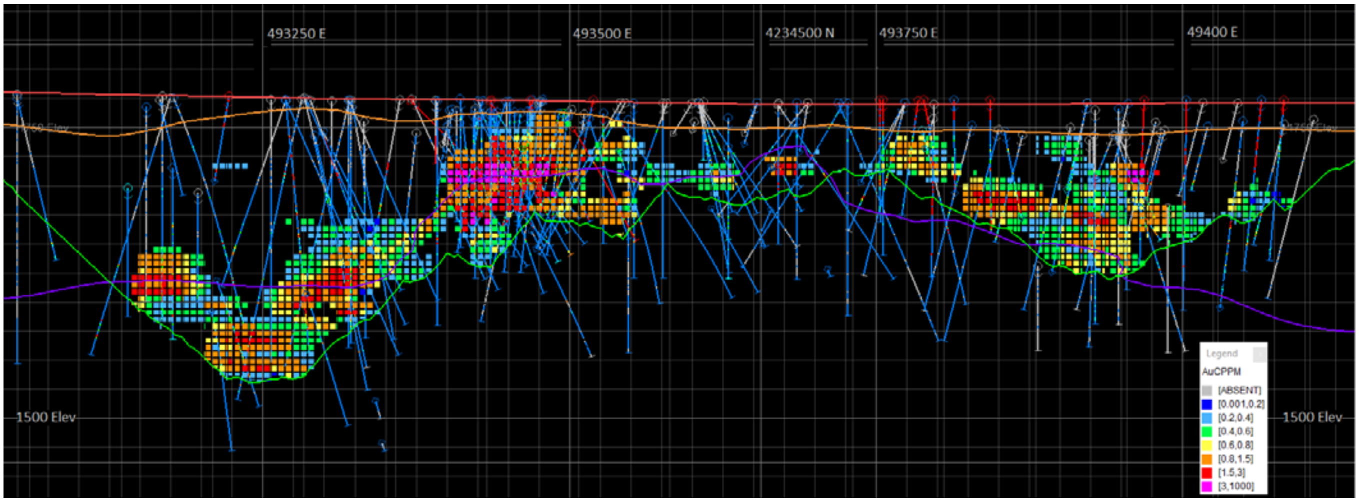


Figure 14-11: Long Section (Looking N20E) view showing drill holes, estimated blocks, and pit shell used to constrain resource estimate.

14.11 Specific Gravity / Density

There is limited bulk density test work available for the project. An SG of 2.46 t/m^3 was used to approximate the density for the mineralized material.

In 2019, McClelland laboratories completed a number of specific gravity tests on column composites comprised of drill cuttings. Specific Gravity data for the cuttings material reported as follows:

Table 14-5 Specific Gravity of Column Composites (triplicate average)

Composite	Specific Gravity
4394-TVS	2.55
4394-TRT	2.57
4394-TRV	2.53
4394-OPA	2.64

Average SG for the volcanic composites is 2.55 t/m^3. Agrillite is higher at 2.64 t/m^3. These are specific gravity estimates rather than bulk density, and ignore potential void space, so they will be slightly higher than bulk density determinations. However, based on this work, it is possible that the 2.46 t/m^3 factor being used is slightly conservative.

Gustavson recommends that specific gravity test work be completed to determine average densities for the different mineral and waste lithologies for the project.

14.12 Validation of Resource Estimate

The resource estimate has been validated by visual review of the block model, by global statistical review, and by swath plots.

14.12.1 Visual Review of Block Model

Visual review of the block model shows good agreement between block and composite grades. Mineralization appears to be well constrained to areas of drilling. Grades generally propagate parallel to the Opa / TV surface. Cross sections are shown with Gravels outlined in yellow, Opa surface as a green trace, and pit shell as an orange trace. Blocks below the pit shell are not reported as resource.

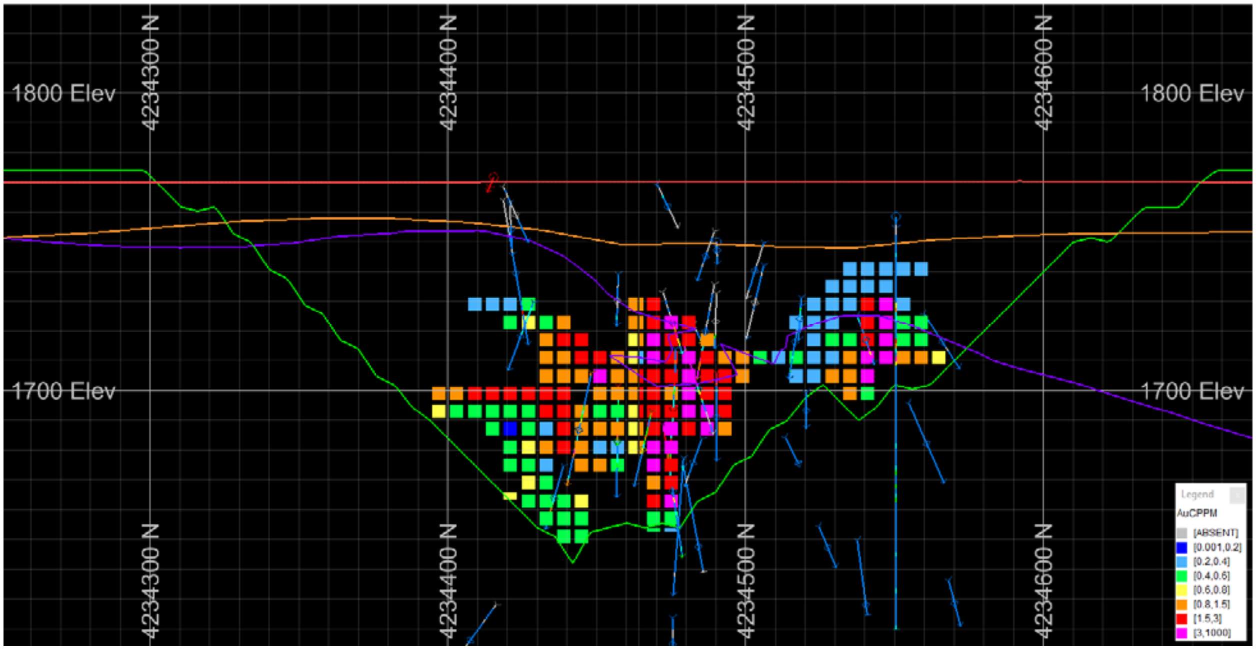


Figure 14-12: Example Cross Section (Looking W) through Tonopah block model (Dauntless Zone). Showing MII blocks and pit trace for reference.

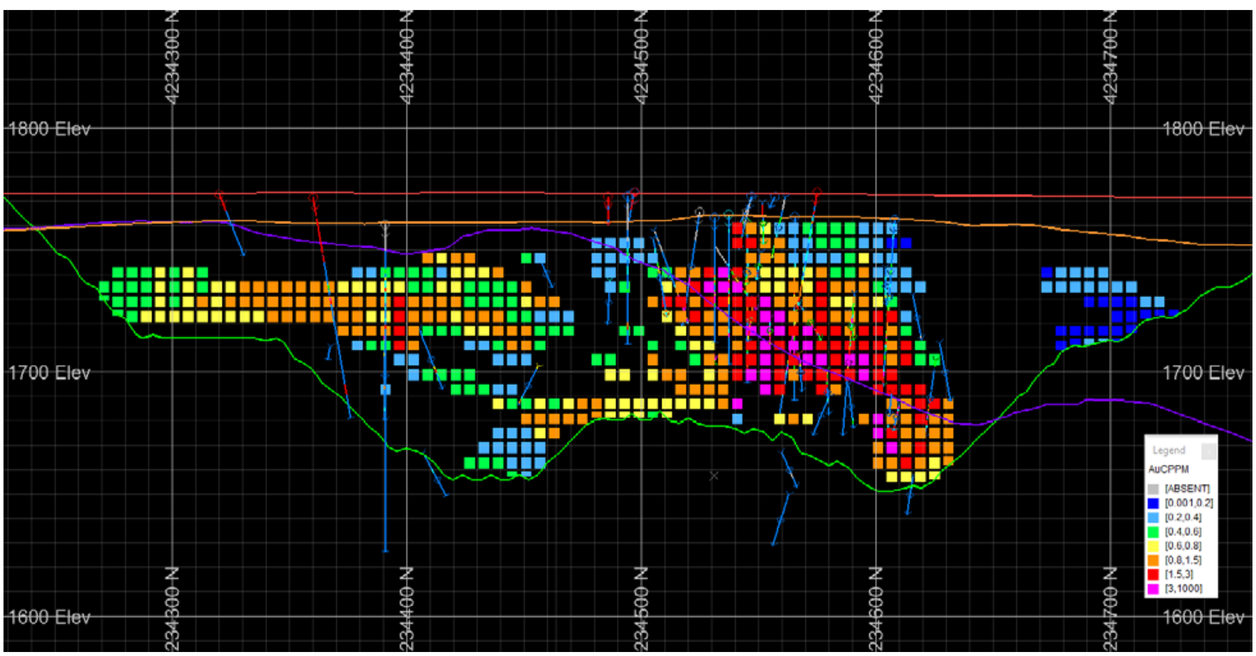


Figure 14-13: Example Cross Section (Looking W) through Tonopah block model (Discovery Zone). Showing MII blocks and pit trace for reference.

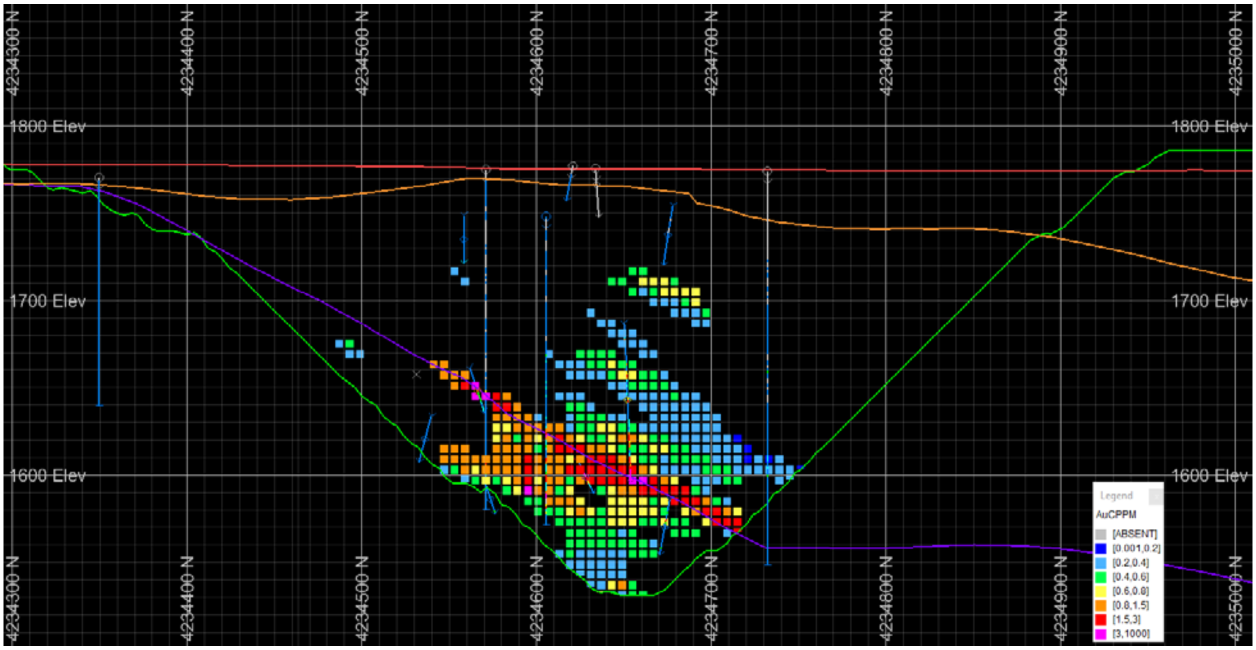


Figure 14-14: Example Cross Section (Looking W) through Tonopah block model (121 Zone). Showing MII blocks and pit trace for reference.

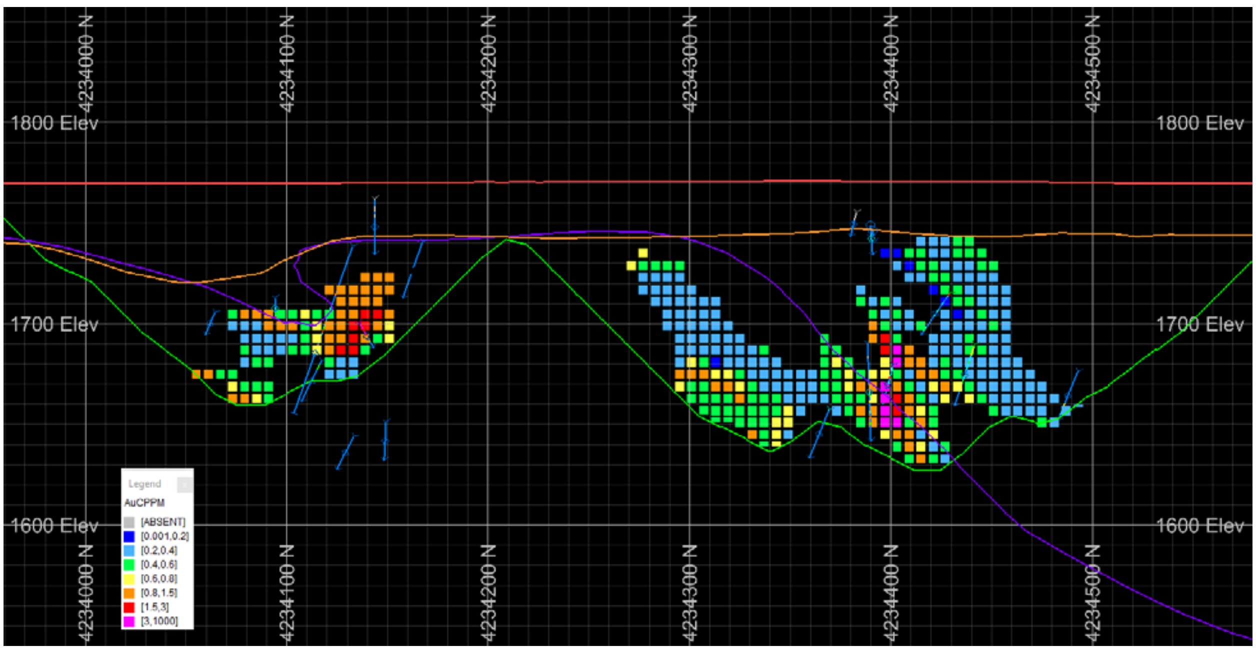


Figure 14-15: Example Cross Section (Looking W) through Enterprise (Left) and Pointluck Zones (right). Showing MII blocks and Pit trace for reference.

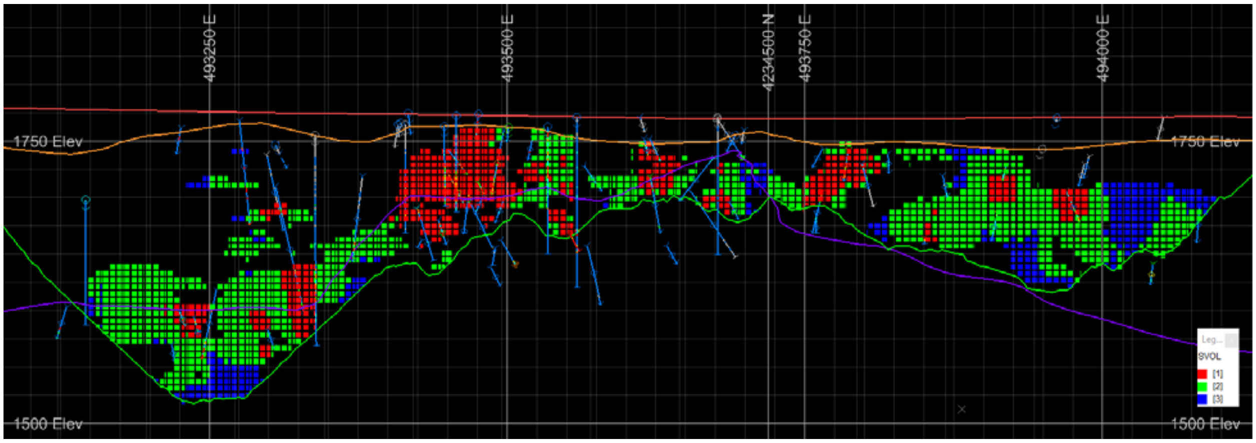


Figure 14-16: Long Section (Looking N20E) through Tonopah Block Model. Showing SVOL (classification) and pit trace for reference.

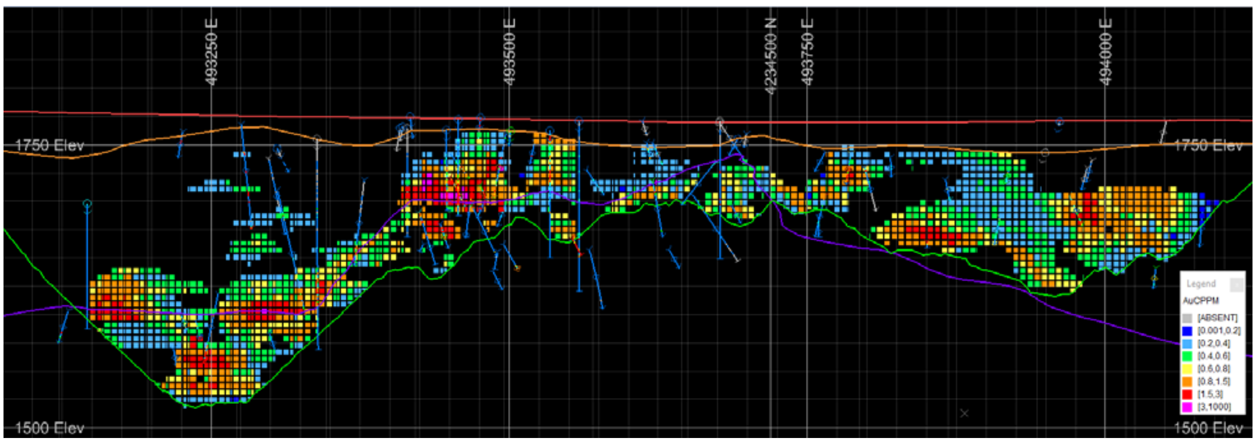
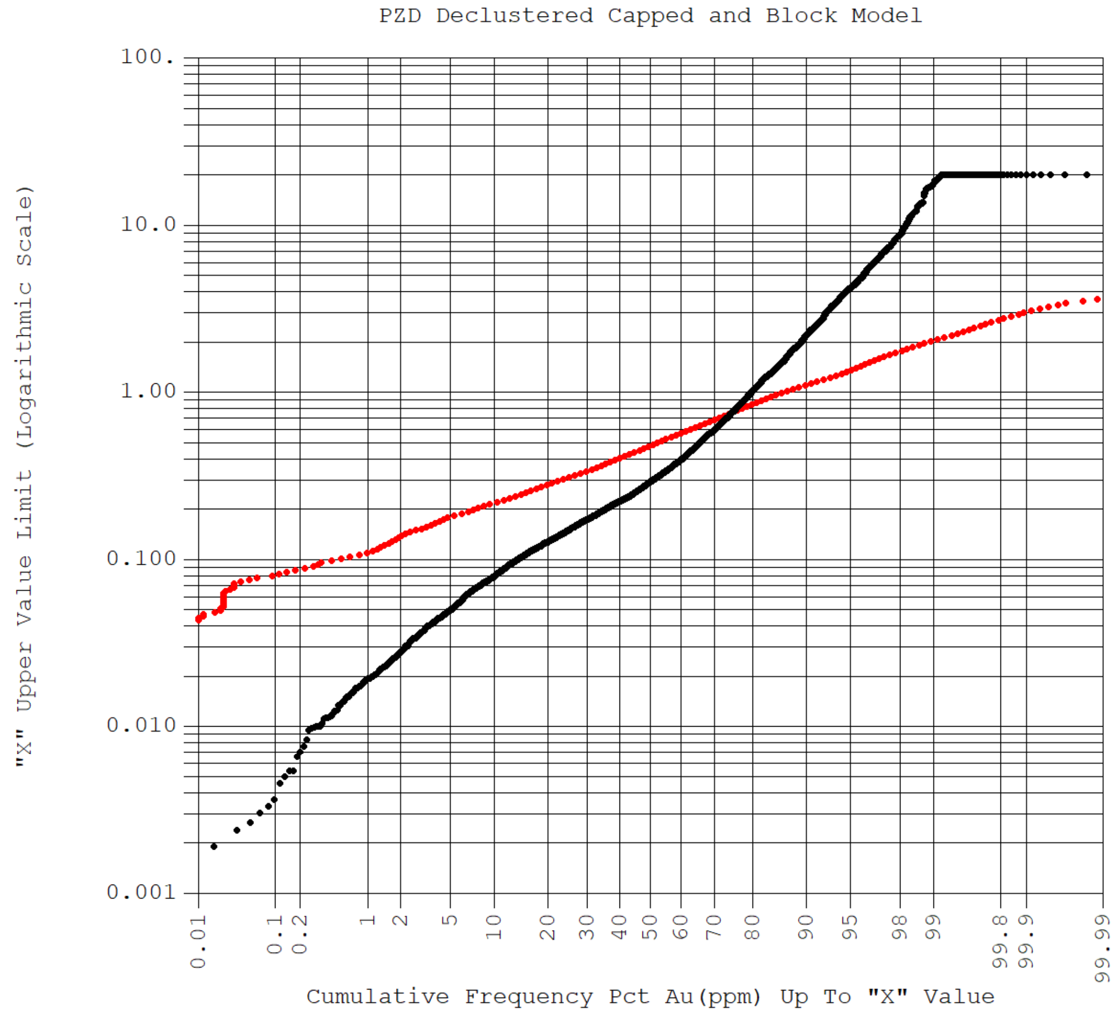


Figure 14-17 Long Section (Looking N20E) showing block grade estimates and pit trace for reference.

14.12.2 Global Statistical Review



PZD Gold Model

	Model	Comp
Number of Samples:	90299	6025
Number Missing:	0	0
Number Below Limits:	31125	20
Number Above Limits:	0	0
Number in Range:	59174	6005
Minimum Value:	0.022	0.002
Maximum Value:	4.315	20.000
Mean Value:	0.592	1.039
Median Value:	0.481	0.293
Variance:	0.162	6.460
Standard Deviation:	0.402	2.542

Figure 14-18: PZD composites and MII Resource Block grades, CFP Comparison.

The statistical review presented in this section is global, or encompassing the entire resource model. A cumulative frequency plot of PZD composite grades compared to MII block grades shows the degree of smoothing of grades within the model. There is significant variance reduction in the block model as compared to composites, partially because of the very high drill density within high grade portions of the deposit. Approximately 90% of the blocks within the block model are above cutoff grade of 0.250 ppm Au. This model may be unreliable for estimation of block grades below this cutoff threshold. Note that the constant grade portion of the composite CFP are the result of capping.

14.12.3 Swath Plots Review

Swath plots were generated comparing capped composite grades, block model grades, and block model tonnages for M & II material by elevation, easting, and northing within the resource area. The swath plots are reasonably well behaved in areas of good drill density and significant resource tonnage, other than some local data effects caused by highly variable drill hole data distributions. (Observed around 5600 elevation in Figure 14-11). The southernmost resource areas in Figure 14-13 have limited drilling and low resource tonnages and behave poorly in the swath plots. This area needs additional drilling to improve confidence in the resource estimate.

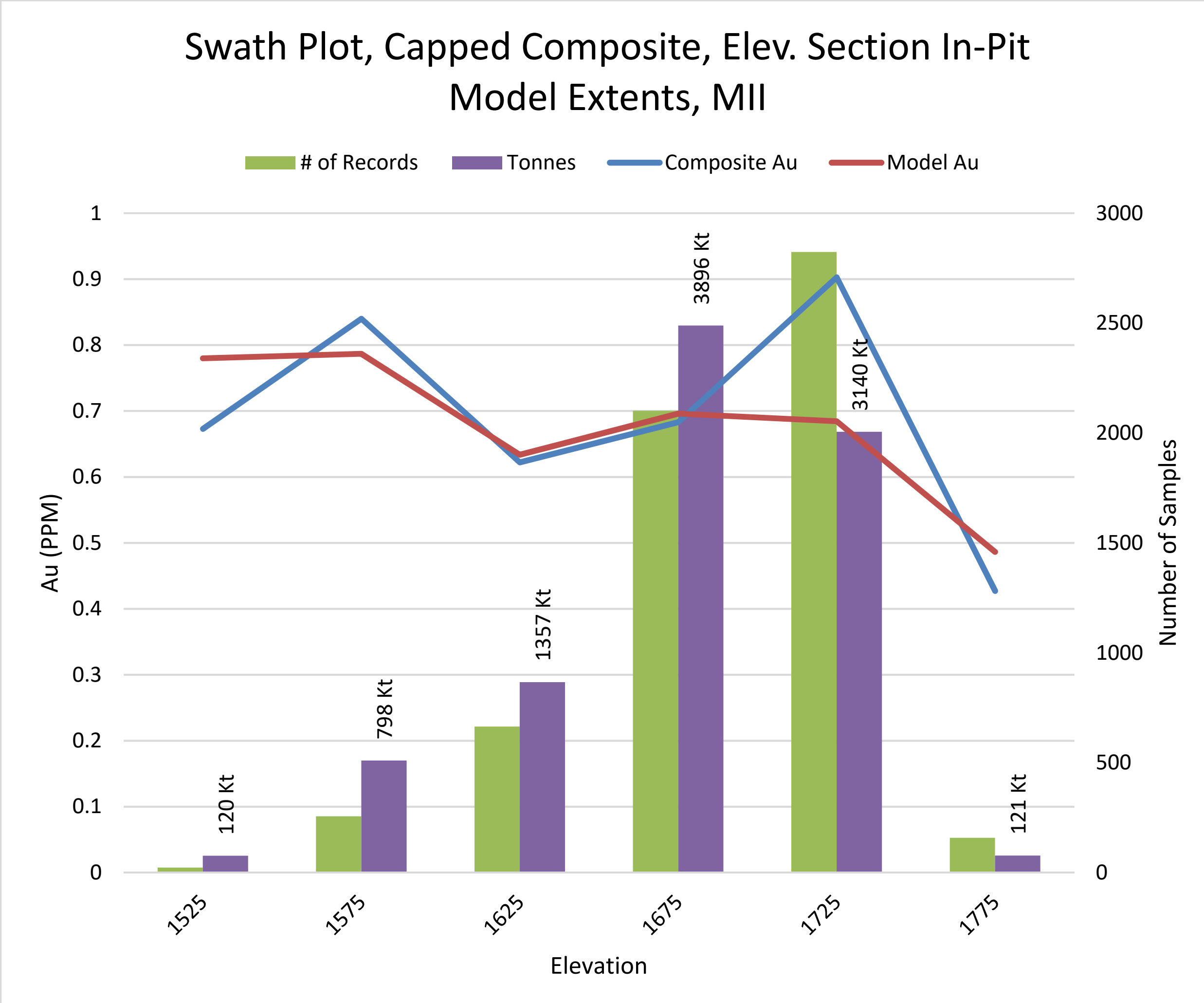


Figure 14-19: Swath Plot by Elevation

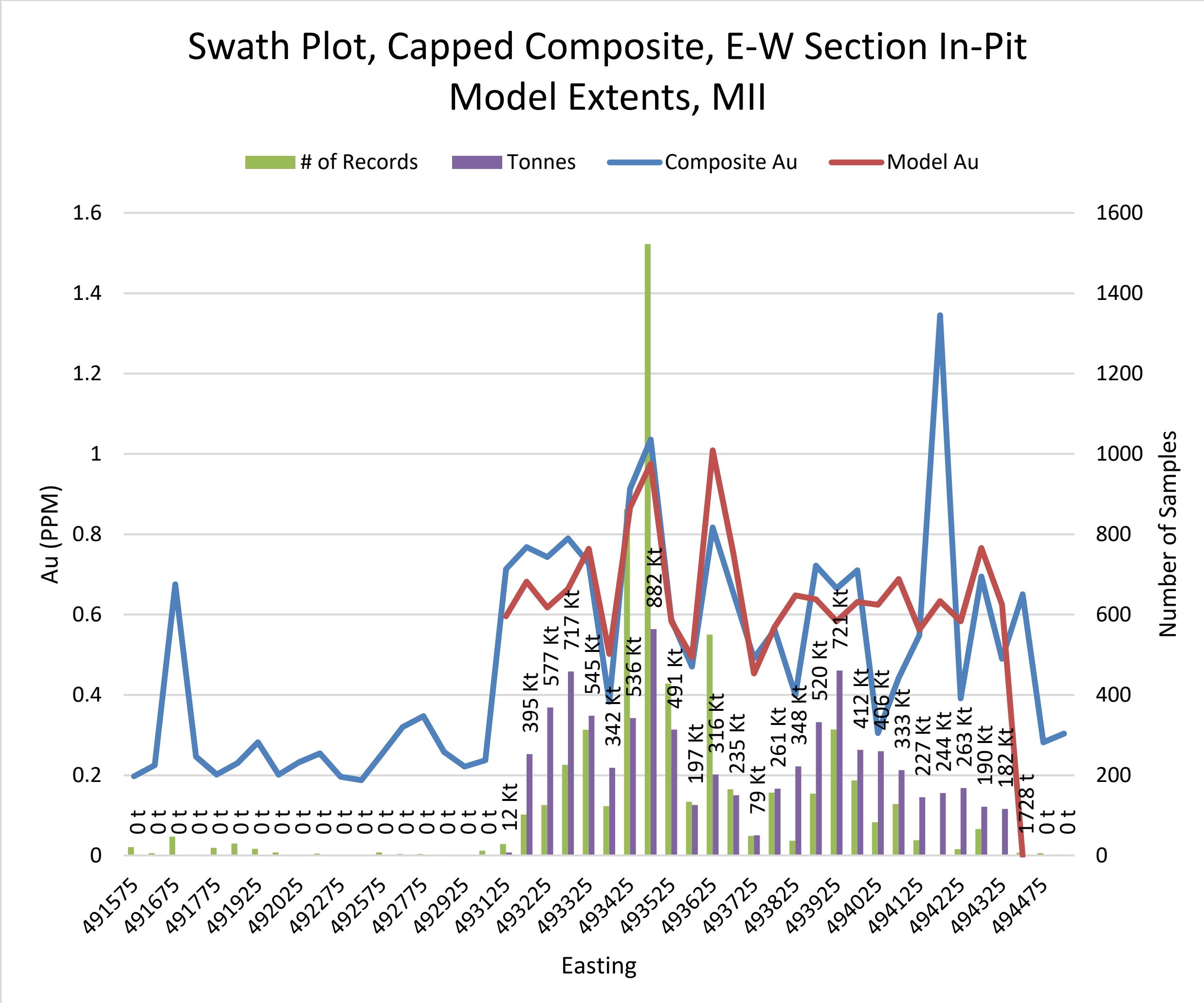


Figure 14-20: Swath Plot by Easting

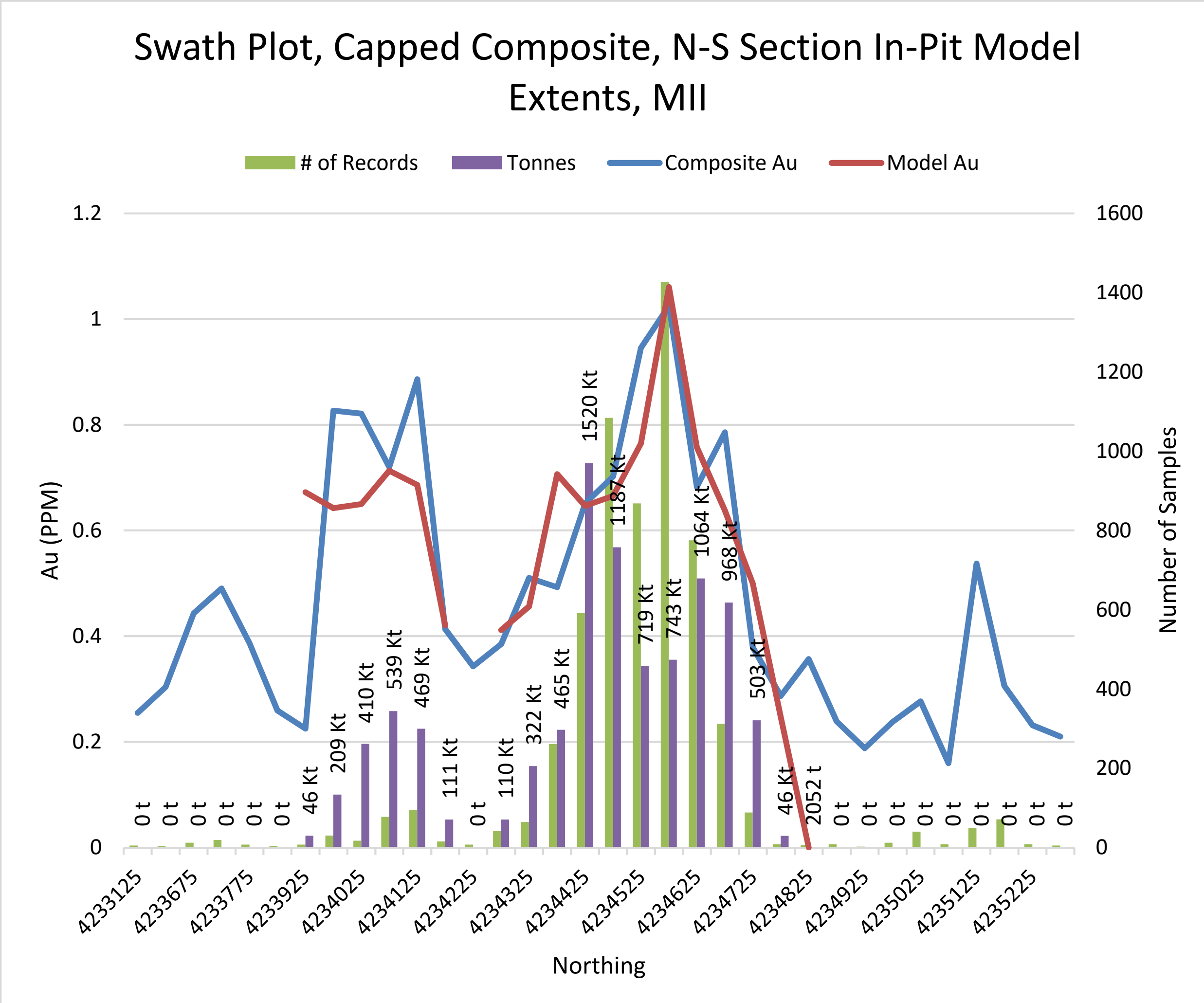


Figure 14-21: Swath Plot by Northing

14.13 Mineral Resource Tabulation

Thomas C. Matthews, MMSA-QP, of Gustavson Associates is the Qualified Person with responsibility for the mineral resource estimation in Table 14-6. Resources do not have modifying factors or dilution applied. Gustavson is of the opinion that the resources presented reasonably represent the in-situ resources modeled for the deposit using all available data as of the April 29, 2020 effective date of this report. Resources are presented at a 0.250 g/t (ppm) cutoff grade in argillite and 0.200 g/t (ppm) in volcanics, and inside a $1600/oz Au pit optimization shell. Parameters used to design this pit are discussed further in Section 16. The reporting cutoff grade and constraining the resource inside an optimization shell constitute two tests for 'reasonable prospects for economic extraction'.

Table 14-6: Mineral Resource Estimate for Tonopah

Classification	Tonnes (x1000)	Gold Grade grams/tonne	Contained Gold (ozt)
Measured	3,930	1.14	141,000
Indicated	8,900	0.65	185,000
Measured plus Indicated	**12,830**	**0.79**	**326,000**
Inferred	8,400	0.67	181,000

Mineral Resources are not Mineral Reserves and have not been demonstrated to have economic viability. There is no certainty that the Mineral Resource will be converted to Mineral Reserves. The quantity and grade or quality is an estimate and is rounded to reflect the fact that it is an approximation. Quantities may not sum due to rounding.

An additional block model tabulation within the identical pit shell is presented as Table 14-7 to show the sensitivity of the block grade estimation at different cutoff grades.

Table 14-7: Block Model Tabulation for grade sensitivities

Classification	Cutoff Grade	Tonnes (x 1,000)	Au Grade Grams/Tonne	Contained Au Ounces
Measured	0.15	3,930	1.12	141,000
	0.20/0.25	**3,380**	**1.14**	**141,000**
	1.00	1,530	2.01	99,000
Indicated	0.15	9,340	0.63	188,000
	0.20/0.25	**8,900**	**0.65**	**185,000**
	1.00	1,210	1.39	54,000
Inferred	0.15	8,990	0.64	185,000
	0.20/0.25	**8,400**	**0.67**	**181,000**
	1.00	1,440	1.33	62,000

The resource (bolded lines) is presented at a cutoff grade of 0.20/0.25 grams per tonne depending on rock type. No differentiation in cutoff by rock type is used for the sensitivities.

15 Mineral Reserve Estimates

There are no Mineral Reserves declared for the Tonopah Property.

16 Mining Methods

16.1 Introduction

Currently, Gustavson is not able to declare a Mineral Reserve at the Tonopah project. Insufficient engineering has been performed to date to declare of Mineral Reserve. Additional work is required to a level of a pre-feasibility study to satisfy the requirements of a reserve declaration. In-pit resources discussed within this section does NOT constitute a Mineral Reserve.

This PEA, including the mine plan within this PEA, includes inferred mineral resource. Inferred mineral resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. No mineral resources in this PEA have been converted to reserves. Mineral resources that are not mineral reserves have no demonstrated economic viability. There is no certainty that the results of this PEA will be realized.

The Tonopah Property as planned in this report has a project life of 8 years, consisting of 2 years of pre-production, 5 years of mining and heap stacking, and one year of final gold recovery, pad rinsing and reclamation work. Mining will be done using traditional, open pit surface mining techniques. Mine operating and capital costs assume self-mining.

16.2 Pit Optimization

The resource model on which the mine designs are based was built using Datamine software from the exploration database provided by Viva Gold Corp. Gustavson performed an economic pit limit analysis using Datamine's NPV Scheduler software which uses the Lerchs-Grossmann algorithm to determine an economic excavation limit based on input parameters. Table 16-1 contains the optimization parameters used. The surface layer is composed of gravels, so this layer was limited to a shallower slope angle than the remainder of the material beneath it.

Table 16-1: Lerchs-Grossmann Optimization Parameters

Parameter	Value
Gold Price (per troy oz)	$1,350
Mining Dilution	0%
Mining Recovery	100%
Mining Cost (per ton mined)	$1.55
Process Cost (per tonne mineralized)	$6.00
Gravels Highwall Angle	35°
Regular Highwall Angle	45°
Gold Recovery % in Volcanics	58%
Gold Recovery % in Argillite	83%

The mining cost of $1.55 per tonne applies to both mineralized material and waste. The cost of $6.00 per tonne processed includes processing and administrative costs and an allowance for leach pad construction. The metallurgical recovery differs in the two material types, volcanics and argillite. The block model delineates these material types and thus a different recovery factor is applied.

The optimization process defines an excavation limit at a specific metal price. The metal price is increased incrementally, and excavation limits or “shells” are created at each price point up to a nominal price of $1350/troy ounce Au. Figure 16-1 shows the results of the optimization process. This graph shows the net present value (NPV), resource tonnes, waste tonnes and total tonnes of the shells plotted as a percentage of the ultimate pit (Y-axis), alongside the gold price.

Figure 16-1 illustrates the reasoning behind selecting a particular optimization shell for the pit design. The $1,230/oz Au shell was selected as the basis for the mine design. This shell contains 66% of the resource but only 51% of the waste yielding a NPV that is 93% of the ultimate pit shell. This shell also does not expand into the west far enough to require the relocation of the state highway, a major investment of capital not accounted for in the NPV shown in this analysis graph.

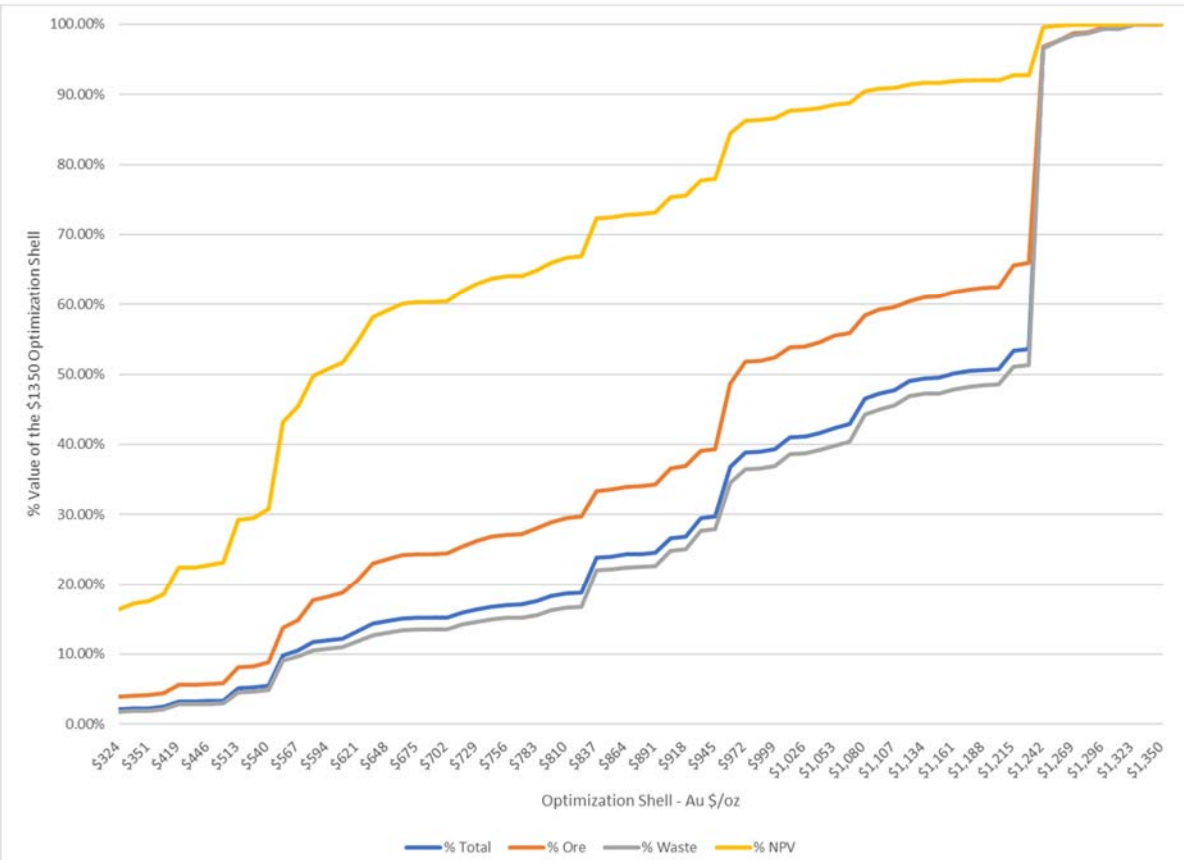


Figure 16-1: Optimization Shell Analysis by Metal Price

16.3 Pit Design

Datamine’s Studio OP software was used to design the ultimate pit limits of the project. Table 16-2: Mine Design Parameters contains the mine design parameters that were used. Note that the road with will accommodate 90 tonne haul trucks. Cut-off grades have been applied to the model depending on lithologies and their respective gold recovery values.

Table 16-2: Mine Design Parameters

Parameter	Value
Bench Height	12m
Regular Catch Bench Width	6.4m
Regular Face Angle	65°
Regular Inter-Ramp Angle	45°
Gravels Catch Bench Width	0m
Gravels Face Angle	35°
Gravels Inter-Ramp Angle	35°
Road Width	30m
Road Gradient	10%
Road Width in Last 2 Benches	20m
Road Gradient in Last 2 Benches	12%
Cutoff Grade in Volcanics (g/tonne)	0.250
Cutoff Grade in Argillite (g/tonne)	0.200

Three successive phases for the pit were designed based on the Lerchs-Grossman shape analysis and minimum push-back widths. These designs account for limitations based on minimum mining widths and wall angles indicative of operational feasibility. Pit exits have been designed to the north and east pit edges where feasible to minimize haul distances to the crusher and waste dump locations. The ultimate pit shape was based on the boundaries in the 91% price Lerchs-Grossman shape discussed in the previous sub-section. The target for road gradient through the main portion of the pits was 10%, with a 12% target in the final two benches. However, some stretches of road exceed the 10% target in small localized areas. The preliminary ultimate pit design is shown in Figure 16-4. A more detailed pit design will be done in future studies.

The ultimate pit shown is the combination of three phases composed of combinations of smaller pit bottoms within the greater geometry. There are a total of seven smaller pit bottoms in the ultimate shape. The first phase takes the first and highest grade pit threshold, Figure 16-2. The second phase includes four additional pit bottoms, Figure 16-3. The third and final phase mines the last two pit bottoms, Figure 16-4. Some waste stripping is common to the pit bottoms, thus the phases ensure that a minimum mining width of 30 meters is for any area that is expanded or pushed back. In scheduling the phases can be intermingled in some years. Given the geometry of the orebody, most of the realization of resource tons is in the intermediate to lowest benches of a given phase or pit bottom. Waste stripping a subsequent phase may often be done while still mining the final and most lucrative benches of the current phase in order to balance material movement flows and the cost/revenue streams.

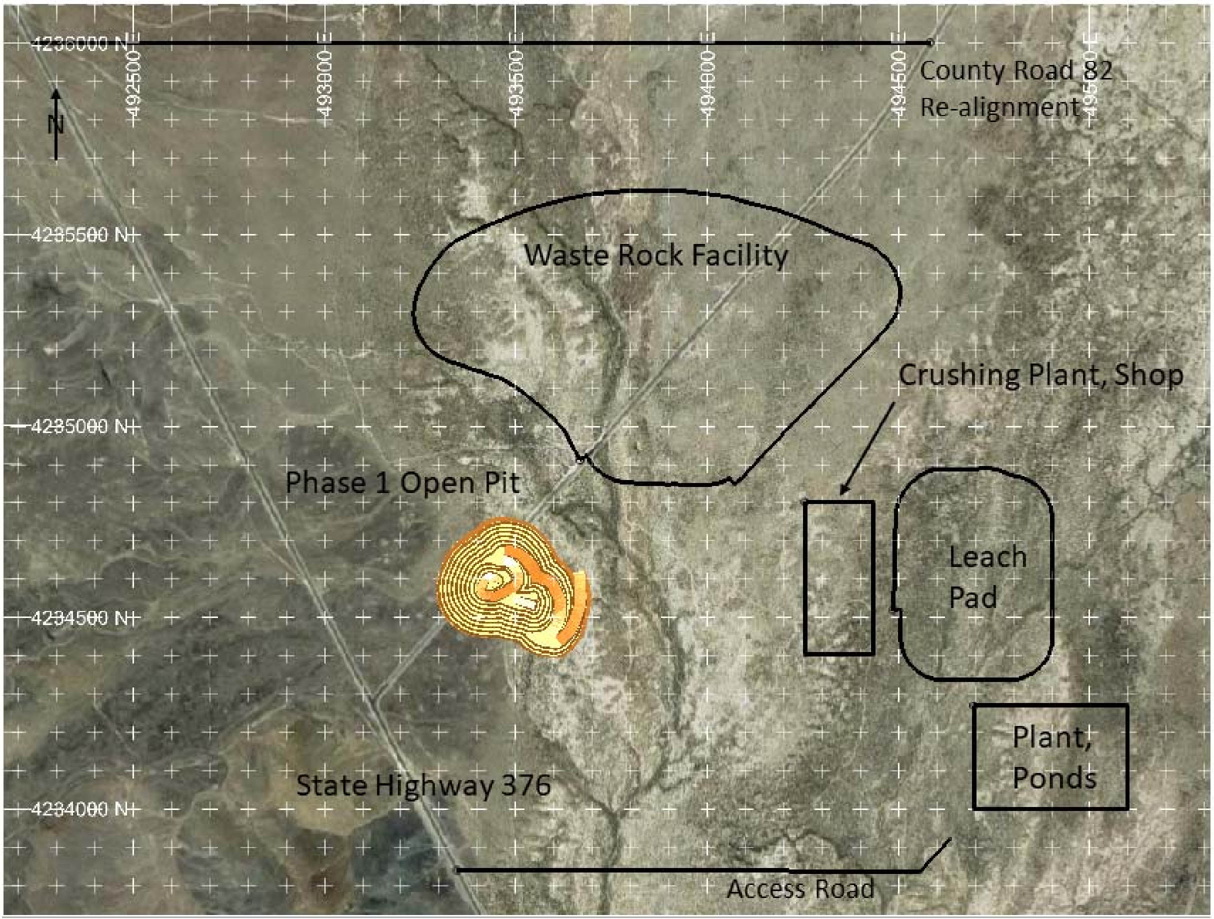


Figure 16-2: Phase 1 Mine Design

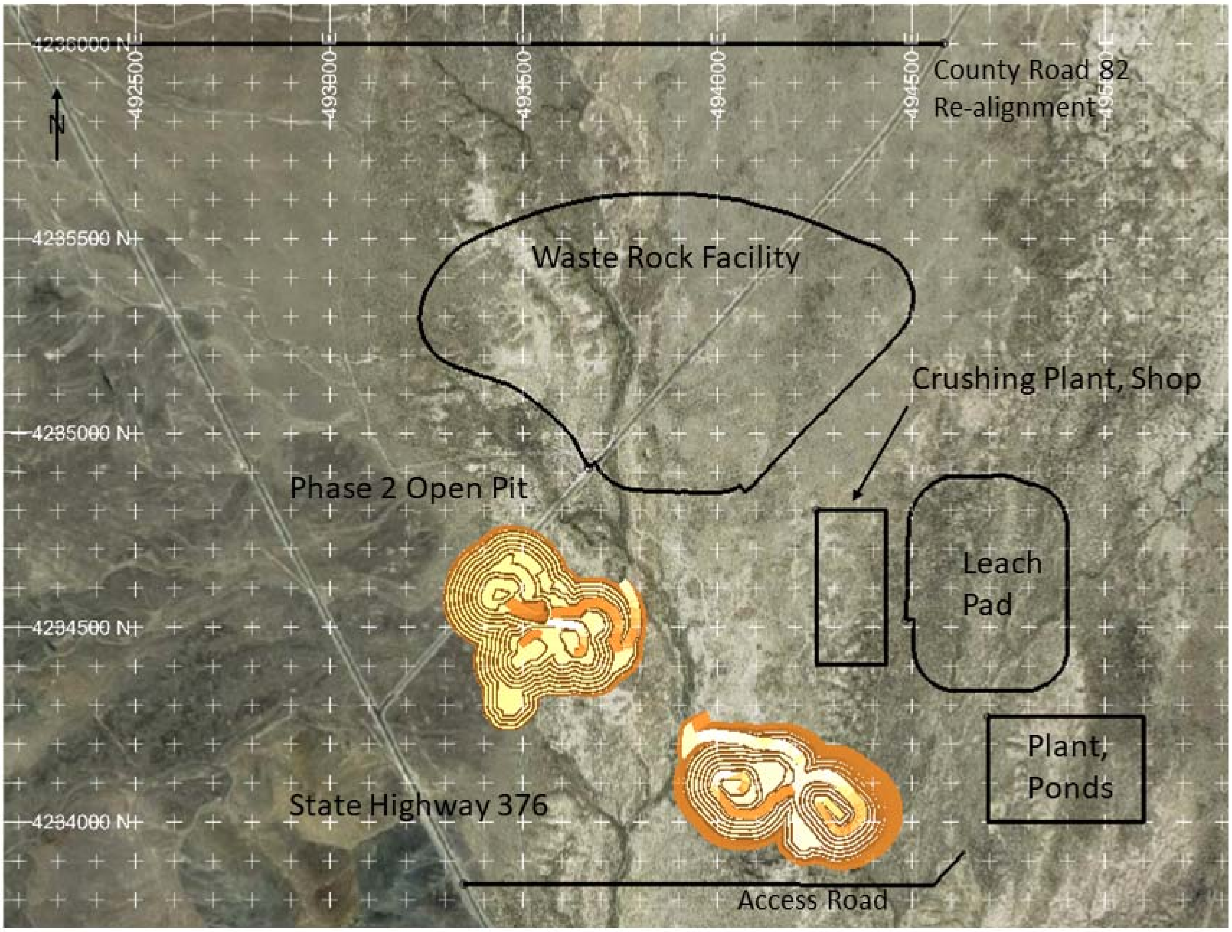


Figure 16-3: Phase 2 Mine Design

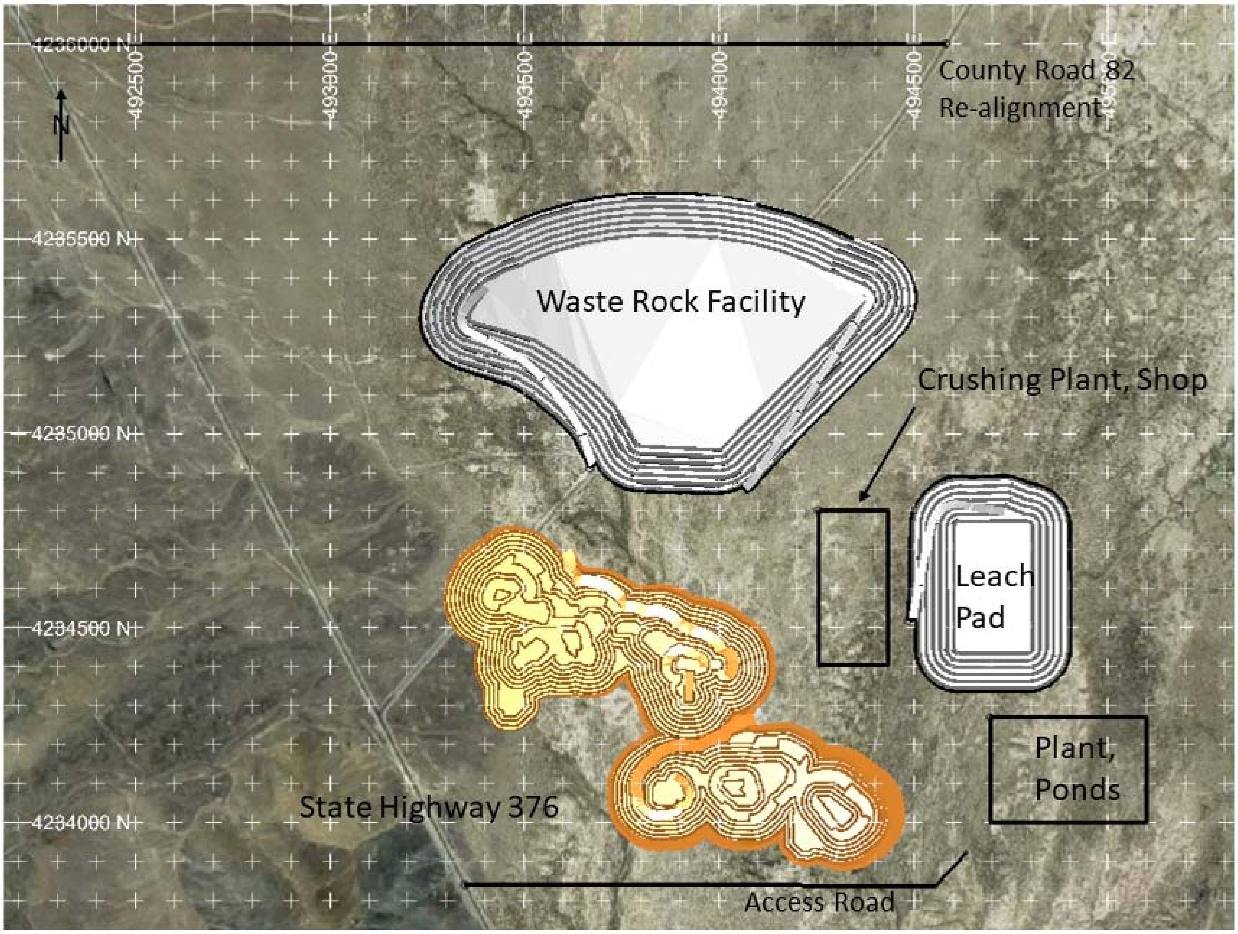


Figure 16-4: Ultimate Mine Design

The in-pit mineral resource estimate shown in Table 16-3 and the mine schedule is based on the above pit design. In-pit mineral resources are not mineral reserves and include inferred resources which are too speculative geologically to have modifying factors applied. There are no mineral reserve estimates for the project at this time. The in-pit mineral resources shown are in-situ, at a gold cutoff grade of 0.20 g/t for argillite mineralization and 0.25 g/tonne for mineralization in volcanics.

Table 16-3: PEA In-Pit Mineral Resource Estimate

Classification	Tonnes (x1000)	Gold Grade gram/tonne	Contained Au Ounces (troy)
Measured Resources	1,890	1.261	76,700
Indicated Resources	6,850	0.690	152,000
Measured + Indicated	8,740	0.813	229,000
Inferred Resources	3,740	0.716	86,100

16.4 Mine Planning and Schedule

Mineralization scheduled for mining is trucked to a 3 stage crushing plan located near the north eastern edge of the final pit. After crushing the material will be conveyed and stacked on the leach pad for gold recovery. Mine planning was carried out on the assumption of a 5-year mine life, attempting to balance the amount of mineralization per period in order to keep consistent feed to the crushing plant. This results in a planning target of approximately 2.5 million tonnes of mineralization per year. Waste materials are trucked to the waste dump to the north of the final pit edge.

After the completion of mining from the two sub-pits in the southeast, waste material will be used to backfill these areas. Waste material movement targets were set by balancing the truck operating hours after the required hours for mineralization movement were accounted for. These hours were calculated through a spreadsheet haulage model based on the length and road gradient traveled from each pit bench to the material destination for Caterpillar 777 haul truck. As pits deepen, leach pads and dumps ascend, haul times escalate, and overage tonnage capacities tend to be reduced. A year-by-year material movement schedule is shown in Table 16-4. Note, recovered gold figures account for leach recovery times and assume a 3-month delay before gold is recovered.

Table 16-4: Scheduled Material Movement

Parameter	Unit	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Total
Mineralized Material	kt	2,460	2,620	1,990	2,540	2,870		**12,500**
Grade	g/t	0.976	0.732	0.909	0.640	0.708		**0.784**
Contained Au	kg	2,400	1,920	1,810	1,620	2,030		**9,790**
Waste Movement	kt	14,600	12,800	11,500	10,500	8,400		**57,800**
Total Movement	kt	17,000	15,400	13,500	13,100	11,300		**70,300**
Strip Ratio		5.9	4.9	5.8	4.2	2.9		**4.6**
Recovered Au	kg	1,160	1,510	1,500	1,230	1,290	335	**7,030**

16.5 General Site Layout

The site is largely flat, slightly dipping to the south. The layout of the pit, waste rock storage, and leach pad are shown in Figure 16-4. To the north of the pit is the waste dump and to the east of the pit is the leach pad. The crushing plant is to be located centrally, between the pit and leach pad.

16.5.1 Waste Rock Storage

Preliminary designs for a waste Dump were made based on the parameters shown in Table 16-5. The waste dump facility was placed to the north or the final pit designs, including a 200m buffer around the $1600 pit edge. The location relative to the ultimate pit and the preliminary leach pad design is shown in Table 16-5.

Table 16-5: Waste Dump Design Parameters

Parameter	Value
Dump Lift Height	9m
Dump Catch Bench Width	6.4m
Dump Face Angle	35°
Dump Inter-Ramp Angle	2H:1V
Dump Road Width	30m
Dump Road Gradient	10%

The dump is seven lifts in height. The first lift is considered partial as it contains about half of the tonnage of the subsequent lifts and brings the overall footprint of the facility to level. Capacity of this facility is 28.8M m^3 for an approximate tonnage of 47.2M tonnes of waste material. The remaining need for waste storage, approximately 10.6M tonnes, will be contained by the two southeastern pit phases after they have been mined out.

16.5.2 Leach Pad Facility

The leach pad facility was designed based on parameters shown in Table 16-6. The stacking was limited to 6 lifts. Road access on the leach pad was made 20 meters wide as this should be sufficient for the conveyors and access for light trucks and small equipment to do maintenance. The facility is to be located 150 meters east of the crushing plant. The total capacity of the current design is 12.9 million tonnes, about 3% more than needed in the tonnage in the schedule.

Table 16-6: Leach Pad Facility Design Parameters

Parameter	Value
Leach Lift Height	9m
Leach Catch Bench Width	6.4m
Leach Face Angle	35°
Leach Inter-Ramp Angle	2H:1V
Leach Road Width	20m
Leach Road Gradient	10%

16.6 Mine Fleet

The mobile equipment fleet is shown in Table 16-7. The quantities of each equipment remain constant throughout the life of mine.

Table 16-7: Mobile Equipment Fleet

Equipment	Quantity
Loader – CAT 992	2
Haul Truck – CAT 777	9
Blast Hole Drills	2
ANFO Loader	1
Dozer – CAT D9	2
Water Truck - CAT 775	1
Grader – CAT 14M	1
Crane Truck	1
Backhoe	1
Crusher Loader – CAT 966	1
Dozer Leach Pad – CAT D6	1

16.7 Labor Requirements

The personnel needed to operate the mine, including administration, technical services, equipment operators, and maintenance are shown in the table below. Staffing levels are constant through the mine life except for year 6 when mining ceases but leaching operations continue

Table 16-8: Staffing Level

Category	Quantity
Loading and Hauling	46
Drilling and Blasting	10
Mine Support	6
Mine Maintenance	18
Crushing Plant	23
Leach and Process Plant	11
Laboratory	4
Administration, Engineering and Supervision	17
Total	**135**

17 Recovery Methods

17.1 Introduction

Process flowsheet and recoveries are based primarily on the 2019 McClelland Laboratories test work, as described in a report titled "Report on Bottle Roll and Column Leach Testing – Midway Drill Core Composites, McClelland Laboratories, Inc. December 20, 2019", and discussed in section 13.2.2.

17.2 Conceptual Process Flowsheet

A process for treating the mineralized material was conceptualized based on the scoping level metallurgical test work. The process will consist of three-stage crushing to a nominal 80% passing 10 mesh product, which will be agglomerated and transported to the leach pad using a conveyor system prior to stacking by radial stacker.

The pregnant solution will be processed in an adsorption-desorption, refining (ADR) plant utilizing a carbon-in-column (CIC) circuit to recover gold and silver.

The planned processing capacity for crushing and heap leach is 7,000 tonnes per day (2.5Mt per year). The planned CIC circuit has a solution processing capacity of 400 cubic meters per hour and is equipped with a 2 - tonne carbon strip circuit.

17.3 PEA Recovery Projections

Based on the column leach testing carried out by McClelland Laboratories, Gold recovery from argillite averages 83%, while recovery from volcanics averages 58%.

Assuming argillite material constitutes 50% of the deposit, the blended recovery of all material types would be ± 71%. However, for the purposes of the PEA, argillite and volcanic mineralized material are tracked separately with the anticipated recovery applied to each.

The gold recovery has not been discounted from the column results because the gold leaching was still on-going when the leaching was terminated in the laboratory column test work.

17.4 PEA Reagent Consumptions

Based on the test work available, the average cement consumption is projected to be 4.4 kg/t. The NaCN consumption is projected at 0.42 kg/t, which is based on 60% of the consumption in the column tests, weighted by material type.

17.5 Recommendations

Gustavson recommends that a detailed metallurgical program be undertaken for the PFS. This should include obtaining PQ core for Column testing, crushing tests be undertaken to produce feed for column tests and run column tests on each material type and blended material. gold recovery is size dependent, HPGR testing for third-stage crushing should be also evaluated.

18 Project Infrastructure

18.1 Water Supply and Dewatering

Water consumption for the project will consist of replacement of leach solution losses, agglomeration of crushed material and dust suppression. Water will be sourced from underground wells accessing a near surface aquifer. Water supply should not be an issue based on hydrological studies conducted by previous operators. Water rights will need to be obtained for this consumptive use.

Because the planned pit intersects this regional aquifer, depression of the local water table is planned using de-watering wells around the perimeter of the open pit. Groundwater will be removed and pumped to infiltration fields where the water will re-enter the aquifer.

Surface water management will be required to limit impacts of storm water runoff. Regulations require the development of a Storm Water Management Program Plan for the entire site to control storm water impacts to the environment. The program plan will outline the measures required to accomplish the above goals.

18.2 Power Supply

The project does not currently have access to the regional electric grid. However, there is a branch of the grid approximately 3 miles east of the project area. This line will be extended to the mine site and a sub-station will be constructed to receive and deliver power to site. Based on the equipment specified the nominal power demand is anticipated to be about 3 MW.

18.3 Labor

Since mining is prolific in this region there is a reasonably large labor pool of experienced miners, staffing the mine is not anticipated to be an issue. The nearby town of Tonopah will provide housing and essential services. Camp accommodations and services will not be required.

18.4 Maintenance/Warehouse/Office

Buildings for mobile equipment maintenance, warehouse, and offices will be required. Planning of these buildings has not been completed. However, funds are allocated in the financial model to construct and equip these facilities. Estimates of the size of these buildings are based on similar sized projects. Fuel storage and dispensing facilities will be located near the shop.

18.5 Processing Facility

The processing facility will consist of a heap leach pad, solution ponds and a building to house the Adsorption, desorption and recovery equipment. The refining area will also be contained within this building. Laboratory facilities will be housed in several shipping containers modified for this purpose.

18.6 Roads

Construction of a short access road is required to access the site from the adjacent state highway 376. The road will be located just south of the open pit. Provisions to restrict public access to the site will be planned for the access road. Additionally, county road 82 will need to be relocated to the north to allow room for mine facilities. Capital costs have been allocated for both road construction projects.

19 Market Studies and Contracts

19.1 Gold Market

Gold is the principal commodity at the Tonopah property and is freely traded in transparent markets worldwide. It is generally assumed that there will be a ready market for gold at market prices. There are no current contracts for the sale of gold produced at the project. There are no issues anticipated in the ability to obtain contracts to sell the bullion produced to a refiner.

19.2 Gold Pricing Assumption

The base case gold price used for this report is $1,400/oz, which is rounded down from the 18 month average of $1,419. Figure 19-1 shows the historic gold price over the last 3 years. Gold prices and averages are based on the London Metals Exchange daily average of the AM and PM fix. Table 19-1 shows the gold price trailing average over several intervals.

Figure 19-1: Historic Gold Price

Table 19-1: Average Gold Price

Interval	Average Gold Price
1 Year	$1,491
18 Months	$1,419
2 Year	$1,375
3 Year	$1,347

20 Environmental Studies, Permitting and Social or Community Impact

20.1 Required Permits and Status

Viva has assumed the permits and authorizations necessary to conduct mineral exploration activities on both public and private land. Authorizations assumed include:

- Decision Record (DR) and Findings of No Significant Impact (FONSI) issued by the United States Department of the Interior Bureau of Land Management (BLM) Casefile NVN-076629, and
- Reclamation Permit 0210 issued by the Nevada Division of Environmental Protection (NDEP) Bureau of Mining Regulation and Reclamation (BMRR).

The BLM DR and FONSI authorize surface disturbance for up to 75 acres for mineral exploration and support activities. Viva has deposited with the BLM Nevada State Office a reclamation bond of $104,786. To-date only 8.4 acres of public land and 0 acres of private land of the total 75 acres of public and private land have been disturbed and remain under reclamation bond.

Temporary groundwater appropriations were issued by the Nevada Division of Water Resources (NDWR) to supply exploration drilling water from an existing well in the Project area. On November 28, 2018 the NDWR approve a waiver to utilize the Red Emerald water well for exploration purposes through November 19, 2019. The use is restricted to dust control and drilling purposes and must not exceed five acre-feet per annum.

Viva also assumed two exploration Notices of Intents (NOI’s), N-95436 and N-95437, the East and North Basins. Bonds posted in regards to these NOI’s amount to $10,454 and $8,235 respectively. The NOI’s involve existing groundwater monitoring and injection wells constructed and used by Midway for groundwater re-infiltration and injection testing. These two NOI’s have been terminated and reclamation activities are ongoing. Bonds will be release on completion of reclamation activity.

Viva’s proposed exploration activities will be located in proximity to two National Register of Historic Places (NRHP) eligible cultural resources sites, CrNV-6-1106 (Ralston Quarry) and CrNV-61-7421 (Midway Archeological Site). As required by Section 106 of the Archaeological Resources Protection Act of 1979 these sites must be protected from disturbance. Viva’s exploration activities are not anticipated to occur near the Ralston Quarry site. The Midway Archaeological Site is located in a very large and extensive dune field complex in which the Project resource area is located. As required by the BLM, Viva will avoid identified cultural features while drilling by establishing 20-meter radius buffer zones at drill sites.

A third cultural resource site, CrNV-61-7482 (Manhattan-Tonopah stage coach route) traverses the Project area from north to south. This site has not been evaluated for NRHP eligibility.

20.2 Environmental Liabilities

Viva is not aware of any current environmental liabilities not identified in this Report resulting from prior Operators’ mineral exploration and testing operations. Field inspections by Agency staff and Viva support staff confirm the existence of water supply and groundwater monitoring and injection wells that require plug and abandon following completion of exploration or potential subsequent mining operations. BLM and Bureau of Mining Regulation and Reclamation (BMRR) regulations require sufficient reclamation bonding to ensure ultimate completion of all reclamation obligations. Review of Midway and Agency records do not report the current presence of residual hydrocarbon (diesel, lubricants, etc.) products resulting from exploration drilling operations in the Project area. Field inspection of the site by the BMRR is conducted annually. No citations or warnings have been issued and no fines or penalties were levied for any environmental or regulatory issues pertaining to the Project under Viva's ownership.

Technical issues related to non-degradation of ground waters of the State, and cultural resources preservation requirements and practices, are not dissimilar to those encountered and managed at mineral exploration projects located elsewhere in the Great Basin of Nevada.

20.3 Previous Environmental Technical and Cultural Resources Studies and Permitting

20.3.1 Previous Environmental Technical Studies

The operator previous to Viva, Midway Gold, undertook several studies to support potential future surface and/or underground mining operations. The studies identified and evaluated baseline hydrogeologic conditions, groundwater quality, storm water controls, mine dewatering requirements, resource and waste rock geochemistry, surplus dewatering water management options including re-infiltration, underground injection and supplemental contribution to the Tonopah Public Utilities (TPU) town water system.

Of note, studies conducted in 2010 predicted an average of 1,000 to 2,000 gallons per minute dewatering rate requirement for a potential underground mining operation. Geochemical testing of waste rock that would be encountered in potential underground decline development reported a low potential for acid rock drainage despite a low net neutralizing potential.

Preliminary data related to environmental and cultural studies have been collected, as detailed and discussed below. Permits for exploration activities are discussed in 20.1. No mining permits have yet been sought or secured.

Nothing has been discovered during these preliminary studies which is expected to have material adverse effects on the eventual permitting and operation of the Tonopah project, although some form of mitigation effort will be required to settle each of the issues discussed.

20.3.1.1 Acid-Base Accounting

Project records indicate that initial acid-base accounting samples were submitted for study to Geomega of Boulder CO. These results were focused on certain types of volcanic tuff and Palmetto Formation argillite, which were considered likely to be the primary waste rock types in an underground mining scenario. While not all possible types of waste rock for all potential mining scenarios have been tested, the results so far demonstrate minimal risk of potential ARD.

The July 1, 2007 Geomega memorandum states:

> *"There are now results for 32 samples of potential waste rock material. On an aggregate basis both the Tertiary volcanic tuff and the Palmetto Formation argillite (with the exception of Outcrop 373) has both low sulfide and low carbonate content. However, while the average*

> *carbonate content of the 25 samples (i.e., excluding Outcrop 373) is low (2.3 ppt) the sulfide content is non-existent to minimal resulting in an average NP:AP of 8.2 which exceeds the EPA criteria of 3."*

Gustavson recommends that additional waste rock characterization work be carried out for additional material types, particularly within alluvial and colluvial deposits, as well as unmineralized Tertiary volcanics overlying the main mineralized zones.

20.3.1.2 Previous Water Studies

A total of 23 separate water monitoring well points have been established in the Project area for water monitoring, which to-date have only been sporadically sampled. No consistent baseline water quality studies have been conducted for the Project. A systematic water sampling program will be required to establish baseline water quality in the project area. Initial injection well and rapid infiltration basin (RIB) testing has been completed to establish costs for processing water pumped during potential underground mine dewatering and returning it the basin.

20.3.1.3 Previous Cultural Resources Studies

Three cultural resources surveys were conducted in the Project area: 1993, 1994 and late 2002-early 2003. These surveys supported mineral exploration activities at that time in the Project area. The Ralston Quarry where Viva proposes no disturbance was originally noted by BLM archaeologist Roberta McGonagle in 1978 with a 1995 follow-up and NRHP eligibility determination in 1978. It is anticipated that updated cultural resource studies are likely required due to the age of past studies, before any major development programs are conducted at the Project.

The Midway Archaeological Site is determined to be potentially eligible to the NRHP with many cultural resource features such as fire-cracked rock, lithic scatters, etc. recorded in the 2002-2003 survey. BLM, Nevada State Historic Preservation Officer and Midway Gold were parties to a Programmatic Agreement (PA) governing development of Midway's exploration activities within the Area of Potential Effect, and administration of the PA to ensure that historic properties are treated to avoid or mitigate effects to the extent practicable and to satisfy BLM Section 106 responsibilities for all aspects of the Project. Midway submitted to the BLM individual work plans (33 to-date) identifying specific locations of proposed disturbance for review and authorization to proceed subject to PA stipulations. The PA facilitated timely authorizations and in-field exploration activities. Viva has assumed Midway Gold's position under the existing PA and filed work plan, #34 to support its recent drill program.

Native American consultations were conducted involving letters, phone calls and two site visits. Concerns expressed by Tribal representatives included potential impacts to the cultural site and impacts to the spiritual value of the Ralston Quarry and Midway Archaeological Site; however, there was no evidence at that time of any recent or current use of the Midway Archaeological Site by Native Americans even though they are aware of the existence of the Site.

Viva's exploration activities should adhere to all Federal and State cultural resources regulations and would inform employees and contractors of the repercussions of collecting cultural artifacts, if any, or damaging cultural resources sites.

20.3.1.4 Previous Permitting Activities

Initial exploration drilling operations involving surface disturbance of less than 5 acres on public land were authorized by the BLM under NOI's. An Exploration Plan of Operations (ExPoO) and Nevada Reclamation Permit application to disturb up to 75 acres for mineral exploration was filed with the BLM and NDEP BMRR in January 2003. The BLM determined it was necessary to prepare an Environmental Assessment (EA) assessing the potential environmental consequences of the proposed exploration activities. The final EA (NV065-2003-037) was published, and a DR and FONSI, issued approving the ExPoO December 12, 2003. NDEP BMRR approved Reclamation Permit 0210 in January 2004. Subsequent ExPoO and Reclamation Permit modifications and amendments followed in 2004-2007, with a Major Modification/Amendment submitted in January 2008 to include construction and operation of an underground mine. Agency processing of the Modification/Amendment was suspended in 2009 as exploration operations at the Project were idled.

21 Capital and Operating Costs

Capital and operating costs for both the mine and processing facilities were developed based on factored and quantity built up estimating techniques and benchmarking similar projects. These costs and equipment requirements were determined from a variety of sources including vendor estimates, the authors' professional experience, review of production and financial actuals from similar projects in the western United States and third-party mining cost databases.

The capital and operating costs detailed in this report have been reviewed by the qualified persons and are reasonable for inclusion in this report. The cost detail meet or exceed the requirements of a Preliminary Economic Assessment accuracy. A 20% contingency is applied to capital costs, except for the mobile equipment fleet, as it reflects current vendor quotations. A 10% contingency is applied to operating costs.

21.1 Capital Cost Estimate

Capital cost were estimated over the entire life of the project of 8 years, including 2 pre-production years, 5 years of mining and processing operations, and a final year of processing operations and reclamation. Initial capital consists of capital spent in the pre-production period and is estimated at $57.9 million. Sustaining capital consists of capital spent in the remaining 6 years of the project and is estimated at $14.7 million, for a project total capital cost of $72.6 million.

Initial and Sustaining capital costs are detailed by area in Table 21-1 Sustaining capital includes credits for the salvage of equipment, return of environmental bonding, and working capital. Costs for each category shown include contingency.

Table 21-1: Project Capital Costs

Category	Initial Capital	Sustaining Capital	Total
	($ Millions)		
Mine Development	$7.20	-	$7.20
Mine Mobile Fleet	$4.98	$13.6	$18.6
Process Plant and Heap	$30.5	-$1.05	$29.5
Environmental & Other	$15.2	$2.11	$17.3
Total	**$57.9**	**$14.7**	**$72.6**

21.1.1 Mine Development Costs

Mine development costs include capital cost for fixed assets, site infrastructure, initial site grading and construction. Items included these groupings include, additional exploration for pit delineation, connection of utilities, are dewatering system, county road re-location and shop/warehouse/office facilities. These costs are typical for a project this size for items such as buildings, utilities, and other items required to support mining operations. The basis for all mine development costs are factored estimates from actual costs or factored estimates from "InfoMine Cost Database." No salvage value is assigned to these capital costs. Sustaining cost for these installations are covered in site wide sustaining capital line item under environmental and other capital. Detail of the mine development costs are available in Table 21-2.

Table 21-2: Mine Development Capital

Item	Cost ($ Millions)
Exploration	$0.50
Dewatering & Infiltration System	$1.50
1300m2 Fully Equipped Shop	$1.50
Office / Warehouse	$0.50
Road Relocation	$0.50
Power Supply	$0.80
Water Supply	$0.20
Access Roads, Site Civil	$0.50
Contingency (20%)	$1.20
Total	**$7.20**

21.1.2 Mine Mobile Equipment Costs

Capital costs for the mobile equipment fleet assume new equipment purchases on the primary mining equipment (haul trucks, wheel loaders, drill rigs and dozers) and used equipment for the mining support equipment (water truck, grader, crane truck, ANFO loader, fuel and lube trucks). Purchases assume using lease to own financing, with financing terms assumes a 5-year term at a 6% interest rate with annual payments beginning one year before delivery of the equipment. All mobile equipment costs assume an additional 10% cost for transportation and 7% for state and local sales taxes over the base equipment price. A salvage value of 30% of the base price, excluding transport and taxes, for mobile equipment is credited to the project in the year after mine operations cease. No contingency costs are applied to the mobile equipment fleet. Table 21-3: contains an itemized table of the mining mobile equipment capital costs.

Table 21-3: Mining Mobile Equipment Capital

Equipment	Quantity	Unit Cost	Principle + Interest	Salvage	Total Capital Cost
	($ Millions)				
Haul Truck - CAT 777	9	$1.29	$13.8	-$3.48	$10.3
Wheel Loader - CAT 992	2	$1.52	$3.61	-$0.91	$2.70
Drill Rig (15-25 cm diameter)	2	$0.98	$2.32	-$0.59	$1.74
Dozer - D9	2	$1.33	$3.16	-$0.80	$2.36
Water Truck – CAT 775	1	$0.62	$0.74	-$0.19	$0.55
Road Grader – CAT 14M	1	$0.31	$0.36	-$0.09	$0.27
Crane Truck – 30 Tonne	1	$0.14	$0.17	-$0.04	$0.12
ANFO Loader	1	$0.09	$0.10	-$0.03	$0.08
Backhoe - 1 m3	1	$0.14	$0.17	-$0.04	$0.12
Fuel /Lube Truck Class 8	2	$0.09	$0.21	-$0.05	$0.16
Light Trucks	5	$0.05	$0.28	-$0.07	$0.21
Total		**$6.55**	**$24.9**	**-$6.29**	**$18.6**

The basis for the estimation of mobile equipment capital costs are, unit costs are based on “InfoMine Equipment Cost Database” with validation of the capital cost for key pieces from bid prices from recent, similar projects. Used prices are estimated at 60% of new prices. Equipment quantities are based on estimated equipment productivity and the material movement schedule.

21.1.3 Process Plant and Heap Costs

Capital costs for the processing plant, including crusher, conveyor and ADR plant are based on vendor quotes for equipment suitable for the scheduled production rate. All equipment is priced as new equipment, except for the leach pad conveyor stacking system which is priced as used equipment. Process equipment costs include an additional cost of 10% for transportation and 7% for state and local sales taxes. The process plant capital cost includes costs for the building, foundations, and construction costs. A 20% contingency is applied to all process plant capital. A salvage value of 35% of the base equipment value, excluding installation and taxes, is used for the process equipment. Table 21-4 contains the process costs by item.

Table 21-4: Process Capital Costs

Item	Capital Cost	Salvage	Total Capital Cost
	($ Millions)		
Crushing Plant	$8.10	-$2.38	$5.72
Leach Pad Construction	$6.21	-	$6.21
Process Plant	$9.80	-$2.15	$7.65
Conveyor and Stacking System	$3.70	-$0.98	$2.72
Laboratory	$0.60	-$0.21	$0.39
Mobile Equipment	$1.51	-$0.38	$1.13
Contingency (20%)	$5.68	-	$5.68
Total	**$35.6**	**-$6.1**	**$29.5**

21.1.4 Other Capital Costs

Other capital costs include items such as environmental bonding, studies, sustaining capital, working capital and other costs. Environmental bonding is modeled as a surety bond where 50% of the bond amount is deposited and 2% of the deposited amount is charged annually as a fee. The initial deposit amount is returned during reclamation. Working capital is 25% of the operating cost in the first production year. General sustaining capital is $1 million per year during the 5 years of mine production.

Table 21-5: Other Project Costs

Item	Initial Capital	Sustaining Capital	Salvage/Credit	Total Capital
	($ millions)			
Environmental Bonding	$2.50	$0.25	-$2.50	$0.25
Feasibility, Permitting	$1.50	-	-	$1.50
Royalty Purchase Option	$1.00	-	-	$1.00
First Fills	$0.25	-	-	$0.25
Sustaining Capital	-	$5.00	-	$5.00
Working Capital	$8.89	-	-$8.89	$0.00
Mine Closure and Reclamation	-	$6.00	-	$6.00
Contingency (20%)	$1.05	$1.05	$1.20	$3.30
Total	**$15.2**	**$12.3**	**-$10.2**	**$17.3**

21.2 Operating Cost Estimate

Operating costs for the project are estimated over the life of the project using a first principles buildup from mine schedule quantities, unit costs, equipment operating hours, labor, and estimated consumables. Fixed costs (labor) and variable costs (equipment operation and consumables) are tabulated separately which leads to variations in the unit operating costs per year due to a varying schedule. Operating costs for major cost centers are shown in Table 21-6.

Table 21-6: Project Operating Costs

Area	**LoM Cost**	**Average Unit Cost**
	($ Millions)	**($/tonne processed)**
Mining	$90.2	$7.22
Processing	$56.5	$4.52
Site G&A	$8.23	$0.66
Contingency (10%)	$15.5	$1.24
Total	**$170**	**$13.6**

The basis of labor costs for all project areas is the number of employees and an annual, burdened wages based on the InfoMine mine cost data base. Wages were verified based on actual cost data from a similar operation. Staffing levels are based on the equipment fleet size or scaled from similar operations.

21.2.1 Mine Operating Costs

A breakdown of the mine operating costs over the life of mine (LoM) is shown in Table 21-7. Load and haul costs are costs associated with the loading of blasted material and transport to the crusher or waste dump. The drill and blast area tracks costs associated with drilling blast holes and explosives consumed. Mine support contains costs associated with dozing at the waste dump and active face, dust suppression, grading of roads and utility work associated with mining. Mine maintenance includes maintenance labor for the mobile equipment fleet and operation of fuel/service trucks. Mine general and administrative (G&A) costs include salaried positions supporting mine operations.

Table 21-7: Mine Operating Costs

Area	LoM Cost	Average Unit Cost	Average Unit Cost
	($ Millions)	($/tonne processed)	($/tonne mined)
Load and Haul	$40.0	$3.21	$0.57
Drill and Blast	$30.6	$2.45	$0.44
Mine Support	$6.51	$0.52	$0.09
Mine Maintenance	$7.58	$0.61	$0.11
Mine G&A	$5.45	$0.44	$0.08
Contingency (10%)	$9.02	$0.72	$0.13
Total	$99.2	$7.94	$1.41

The basis for the mine operating costs are as follows. For all cost areas, machine hours required to meet the mine schedule requirements are calculated and multiplied by an hourly unit cost. For loaders and haul trucks, the machine hours are calculated using an equipment productivity model and a haulage model, on an annual basis. The hourly unit costs are based on a database of equipment and includes fuel, maintenance parts, lubricants, tires and ground engaging wear parts. Table 21-8 through Table 21-12 shows a detail of each mine operating cost area.

Table 21-8: Loading and Hauling OPEX

Area	LoM Cost - ($ Millions)	LoM Quantity	Units	Unit Costs	Units
Variable Costs	$24.3				
Loader	$5.3	50,208	Hours	$98.28	$/hr
Haul Truck	$19.0	238,583		$79.60	
Fixed Costs	$15.7				
Loader Operator	$3.3	8	Employees	$77,566	$/yr
Haul Truck Operator	$12.5	36		$69,245	

Table 21-9: Drilling and Blasting OPEX

Area	LoM Cost - ($ Millions)	LoM Quantity	Units	Unit Costs	Units
Variable Costs	$26.1				
Drill Rig	$3.5	51,433	Hours	$68.75	$/hr
Drill Consumables	$1.0	1,285,819	Meters	$0.75	$/m
Explosive Consumables	$21.4	70,291,431	Tonnes	$0.31	$/t
Shot Truck	$0.2	5,000	Hours	$31.12	$/hr
Fixed Costs	$4.5				
Lead Blaster	$0.7	1	Employees	$130,150	$/yr
Blasting Laborer	$0.3	1		$59,044	
Drill Operator	$3.6	8		$89,050	

Table 21-10: Mine Support OPEX

Area	LoM Cost - ($ Millions)	LoM Quantity	Units	Unit Costs	Units
Variable Costs	$4.37				
Dozer	$2.61	40,000	Hours	$65.15	$/hr
Water Truck	$1.12	20,000		$56.08	
Grader	$0.56	20,000		$27.83	
Crane Truck	$0.05	2,500		$18.36	
Backhoe	$0.04	2,500		$15.22	
Fixed Costs	$2.14				
Operators	$1.55	4	Employees	$77,566	$/yr
Laborers	$0.59	2		$59,044	

Table 21-11: Mine Maintenance OPEX

Area	LoM Cost - ($ Millions)	LoM Quantity	Units	Unit Costs	Units
Variable Costs	$0.25				
Fuel Truck	$0.12	1,000	Hours	$24.63	$/hr
Service Truck	$0.12	1,000		$24.63	
Fixed Costs	$7.33				
Mechanics	$6.15	14	Employees	$87,910	$/yr
Laborers	$1.18	4		$59,044	

Table 21-12: Mine G&A OPEX

Area	LoM Cost - ($ Millions)	LoM Quantity	Units	Unit Costs	Units
Variable Costs	-				
Fixed Cost	$5.45				
Mine Manager	$0.79	1	Employees	$157,550	$/yr
Mine Foreman	$2.60	4		$130,150	
Engineer	$0.65	1		$130,150	
Geologist	$0.65	1		$130,150	
Survey / Technician	$0.75	2		$150,700	

21.2.2 Process Operating Cost

Table 21-13 contains a breakdown of the LoM operating costs for the process area. Crushing plant costs consists of costs associated with operation of the crusher, conveying, and stacking costs and a wheel loader to feed the crushing plant. Leach and process plant costs are costs associated with leach pad operations and operation of the ADR plant.

Table 21-13: Process Operating Costs

Area	LoM Cost	Average Unit Cost
	($ Millions)	($/tonne processed)
Crushing Plant	$28.7	$2.30
Leach and Process Plant	$24.7	$1.98
Laboratory	$3.06	$0.25
Contingency (10%)	$5.65	$0.07
Total	**$62.2**	**$4.59**

The basis for the process operating costs is detailed in Table 21-14 through Table 21-16 and is similar in method to the mine operating costs.

Table 21-14: Crushing Plant OPEX

Area	LoM Cost ($ Millions)	LoM Quantity	Units	Unit Costs	Units
Variable Costs	$18.1				
Wheel Loader	$0.68	22,500	Hours	$30.25	$/hr
Crushing Plant	$14.9	12,486,662	Tonnes Processed	$1.20	$/t
Conveyor & Stacking	$2.46	12,486,662		$0.20	
Fixed Costs	$10.6				
Foreman	$1.78	4	Employees	$89,050	$/yr
Operator	$5.00	12		$83,408	
Mechanic/Electrician	$3.52	8		$87,910	
Laborer	$0.32	1		$64,971	

Table 21-15: Leach and Process OPEX

Area	LoM Cost ($ Millions)	LoM Quantity	Units	Unit Costs	Units
Variable Costs	$19.8				
Dozer	$0.40	15,000	Hours	$26.7	$/hr
ADR Plant	$1.35	12,486,662	Tonnes Processed	$0.10	$/t
Lime/Cyanide	$18.0	12,486,662		$1.44	
Fixed Costs	$4.97				
Process Manager	$0.95	1	Employees	$157,550	$/yr
Plant Operator	$2.08	6		$64,971	
Plant Mechanic	$0.10	2		$87,910	
Laborer	$0.32	1		$64,971	

Table 21-16: Laboratory OPEX

Area	LoM Cost ($ Millions)	LoM Quantity	Units	Unit Costs	Units
Variable Costs					
Supply - Total Cost	$1.07	12,486,662	Tonnes Processed	$0.08	$/t
Fixed Costs	$1.99				
Lab Manager	$0.78	1	Employees	$130,150	$/yr
Technician	$1.21	3		$75,350	

21.2.3 Site G&A Costs

Site G&A costs include salaried employees that perform general and administrative tasks that involve both the mine and process areas such as a general manager, accountant, safety and environmental engineers, clerks and technicians, as well as annual costs such as computers, supplies, insurance & security. Table 21-17 details the buildup of the site G&A costs.

Table 21-17: Site G&A OPEX

Area	LoM Cost ($ Millions)	LoM Quantity	Units	Unit Costs	Units
Variable Costs	$4.24				
General	$3.95	12,486,662	Tonnes Processed	$0.30	$/t
Light Trucks	$0.30	27,000	hours	$10.95	$/hr
Fixed Costs	$3.99				
General Manager	$0.95	1	Employees	$157,550	$/yr
Accountant	$0.78	1		$130,150	
Safety / Environmental Engineer	$1.43	2		$130,150	
Warehouse/AP Clerk	$0.83	2		$75,350	

22 Economic Analysis

The economic analysis of the Tonopah project relies on the mining schedule, capital and operating cost, and recovery parameters discussed in the previous sections of this report. This is a PEA level economic analysis, which includes inferred resource in the model. The positive economic outcome presented here does not delineate a mineral reserve. The economic parameters used are believed to be reasonable but additional information may change these assumptions and impact the analysis. All figures are in constant 2020 US dollars.

22.1 Model Parameters

The economic model is prepared on an after-tax basis. Model parameters are summarized in Table 22-1:

Table 22-1: Economic Model Parameters

Parameter	Value
Project Funding	100% Equity
Working Capital	25% of operating costs
Discount Rate	5%
Contingency Operating Costs	10%
Contingency Capital Costs	20% (except mobile equipment)
Gold Price	$1,400/oz

The model spans 2 years of pre-production, 5 years of mine production and 1-year post-mine production. One of the primary inputs to the model is the mine schedule presented in Table 16-4, which provides the grade and tonnage of the mineralized material mined. Income is based on the quantity of recovered metal, the metal price above and refining terms of 99.9% payable metal and a shipping and refining charge of $1.50/Au oz. The refining charges are based on a refining contract for a similar property located in the western United States.

22.2 Taxes, Royalties, Depreciation and Depletion

The study assumes a royalty of 1% on the net smelter return and that the operator exercises the option to buy out an additional 1% royalty with an upfront payment of $1 million dollars.

A Federal tax rate of 21% is assessed to net income, and a Nevada net proceeds tax ranging between 0% and 5% is based on a sliding scale that considers the net proceeds as a percentage of gross revenue, consistent with current Nevada Tax law. Nevada Sales and Use tax is included in capital and operating cost estimations.

Depreciation for mobile equipment is based on the 7 year Modified Accelerated Cost Recovery System (MACRS) as allowed by the Internal Revenue Service. All other capital costs such as mine development capital, process plant and heap leach capital are depreciated based on a units of production depletion model.

Depletion for federal tax purposes is calculated by the percentage depletion method. For this property the depletion percentage is 15% of the gross revenue less royalties, not to exceed 50% of the taxable income.

22.3 Project Economics – Base Case

The results of the economic analysis are provided in Table 22-2. The complete DCF model is included as Appendix F.

Table 22-2: PEA Economic Results

(USD million) **Gold Price ($/oz)**	**Base Case** **$1,400**
Pre-Tax Economics	
IRR	25%
Cash Flow (Undiscounted)	$69.7
NPV 5% Discount Rate	$43.6
NPV 10% Discount Rate	$25.9
Payback (Years)	2.9
After-Tax Results	
IRR	22%
Cash Flow (Undiscounted)	$60.1
NPV 5% Discount Rate	$36.3
NPV 10% Discount Rate	$20.3

Table 22-3: Project Details

Gold Price ($/oz)	**Base Case $1,400**
Gold Ounces Sold	226,000
Initial Capital(1)	$58 M
Sustaining Capital	$16 M
Avg Cash Cost of Production	$754
All in Sustaining Cost (AISC)	$1,075
Project Life (Years)	6
Total Processed Tonnes (M)	12.5
Average Au Grade (g/t)	0.78
Total Waste Tonnes (M)	57.8
Strip Ratio	4.6
Personnel Employed	135
Average Operating Costs	
Mining Cost ($/t mined)	$1.28
Process Cost ($/t crushed)	$4.52
Gen & Admin Cost	$0.66
Offsite marketing and Refining ($/oz)	$1.50

(1) $1.0 million is included in capital cost to exercise Viva's Option to acquire 1% of the 2% NSR on the project

22.4 Sensitivity Analysis

Sensitivity analysis was performed on the parameters, capital cost, operating cost, and metal price on a before-tax basis. Figure 22-1 and Figure 22-2 show the sensitivity of NPV and IRR. The figures below indicate that the project is most sensitive to changes in metal prices.

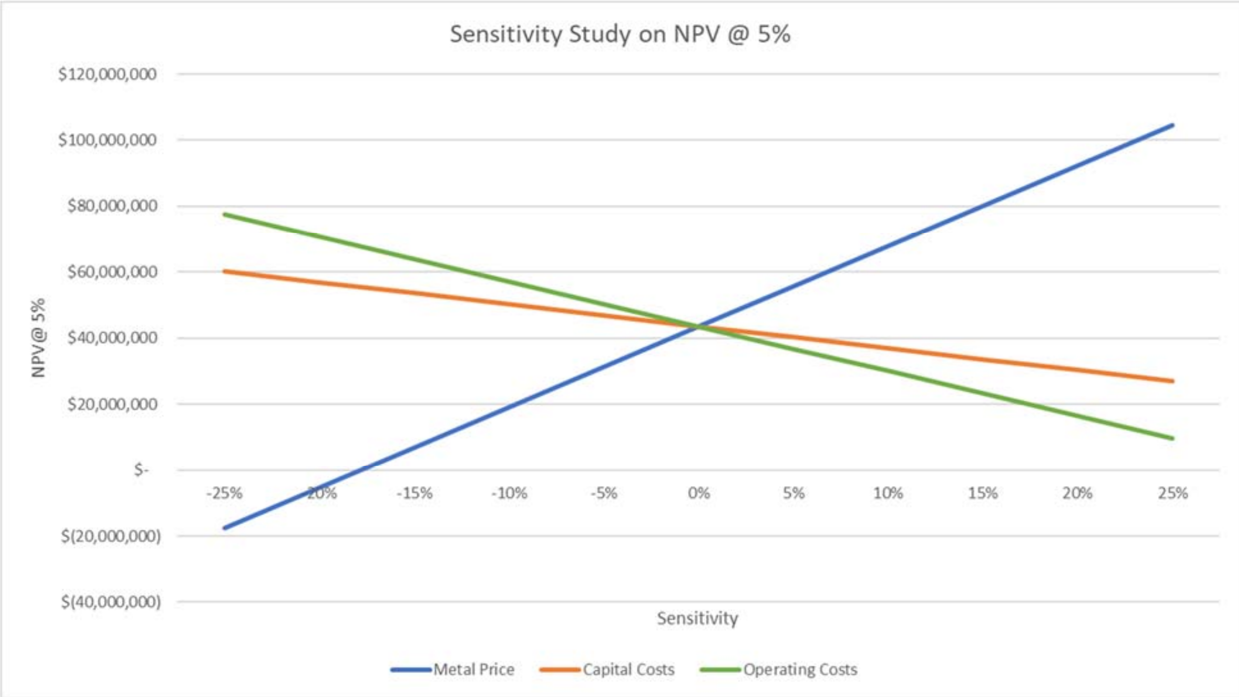


Figure 22-1: Sensitivity Study on NPV

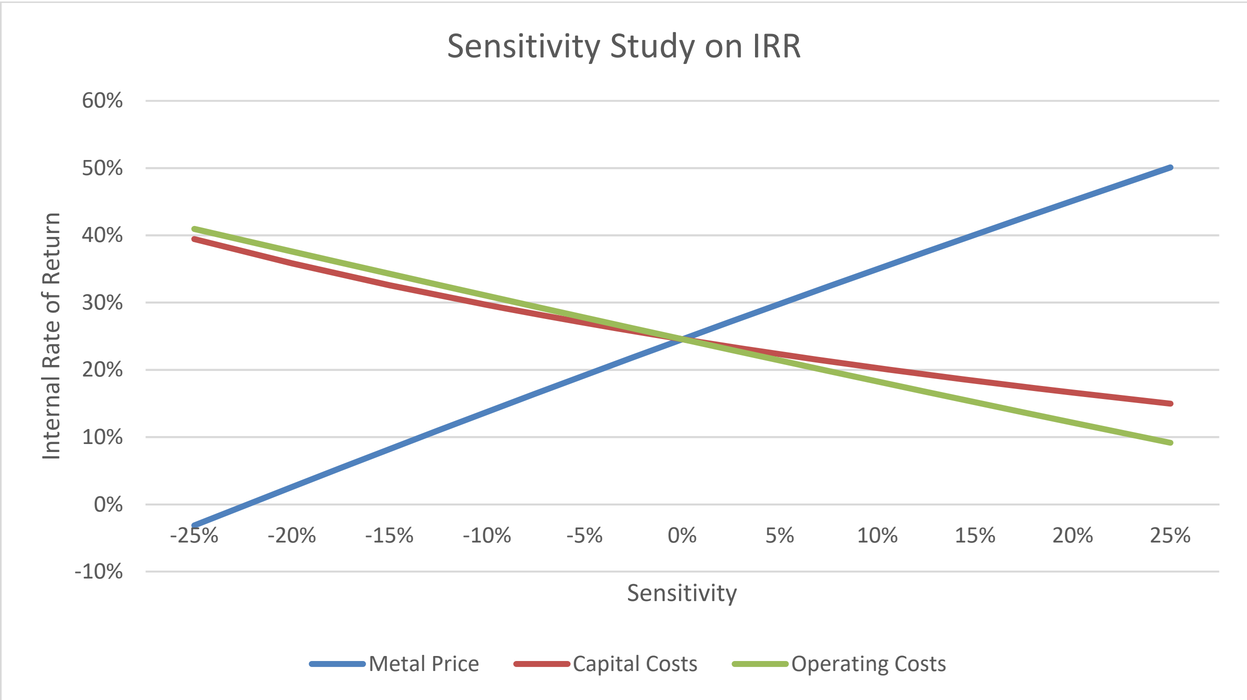


Figure 22-2: IRR Sensitivity

A chart of gold price sensitivities for the PEA is presented in Table 22-4:

Table 22-4: Gold Price Sensitivity Table

Base Case - Pre-Tax (US$MM)					
Gold Price	IRR%	Undiscounted Cash Flow	NPV 5%	NPV 10%	Payback
$1,100	1%	$2.6	($8.7)	($15.6)	n/a
$1,200	9%	$25.0	$8.7	($1.7)	5.1
$1,300	17%	$47.3	$26.1	$12.1	4.1
$1,400	25%	$69.7	$43.6	$25.9	2.9
$1,500	32%	$92.1	$61.1	$39.8	2.5
$1,600	39%	$114.4	$78.5	$53.6	2.2
$1,700	47%	$136.8	$96.0	$67.4	2.0

22.5 Alternate Case - $1600 Gold Price

The Tonopah project pit shells are highly sensitive to gold price. There is a significant step function in the pit shell designs such that the entire western portion of the resource area is excluded from the pit at prices below $1300, but may be included at higher metals prices. At PEA price assumptions of $1400, there is only a marginal value impact from the final pushback. However, Figure 22-3 shows that at a $1600 price, the additional push back captures an additional 15-18% NPV.

Given that spot gold prices as of the effective date are over $1700 per troy ounce, it would be misleading not to consider the impact on the mining plan in a higher gold price environment. Accordingly, Gustavson created a sensitivity design case to explore the impact of $1600 gold price on the preliminary economic assessment.

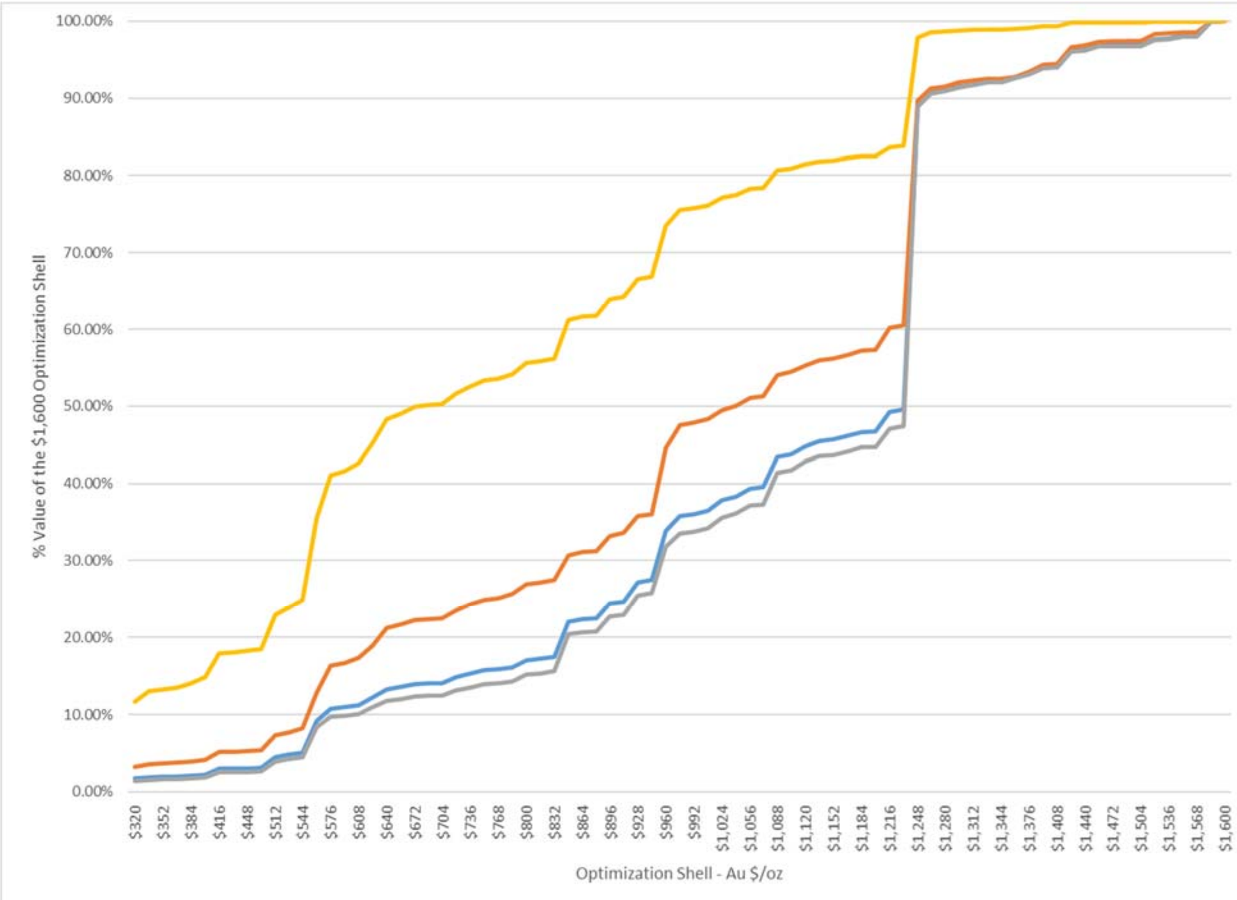


Figure 22-3 Alternative Case Optimization

An analysis was performed to review the impact of $1,600/oz gold price on the project. In this case, a $1600/oz gold price pit optimization shell was used to design a new pit for this case, shown in Figure 22-4. A separate mining schedule was built using similar processing rates and increased mining rates to account for additional waste stripping with the larger pit. Mining operations increase from 5 years in the base case to 8 years. The economic model was scaled for the alternative case and the results are shown in Table 22-5. Capital and operating costs were adjusted accordingly based on the larger project.

Table 22-5: Alternate Case Financial Results

(USD million) **Gold Price ($/oz)**	**Alternate Case** **$1,600**
Pre-Tax Economics	
IRR	26%
Cash Flow (Undiscounted)	$129
NPV 5% Discount Rate	$76.5
NPV 10% Discount Rate	$43.7
Payback (Years)	3.2
After-Tax Results	
IRR	23%
Cash Flow (Undiscounted)	$109
NPV 5% Discount Rate	$62.5
NPV 10% Discount Rate	$34.0

Table 22-6: Alternate Case Details

Gold Price ($/oz)	Alternate Case $1,600
Gold Ounces Sold	347,000
Initial Capital(1)	$66
Sustaining Capital	$42
Avg Cash Cost of Production	$898
All in Sustaining Cost (AISC)	$1,209
Project Life (Years)	9
Total Processed Tonnes (M)	20.9
Average Au Grade (g/t)	0.72
Total Waste Tonnes (M)	123
Strip Ratio	5.9
Personnel Employed	159
Average Operating Costs	
Mining Cost ($/t mined)	$1.24
Process Cost ($/t crushed)	$4.36
Gen & Admin Cost	$0.62
Offsite marketing and Refining ($/oz)	$1.50

(1) $1.0 million is included in capital cost to exercise Viva's Option to acquire 1% of the 2% NSR on the project

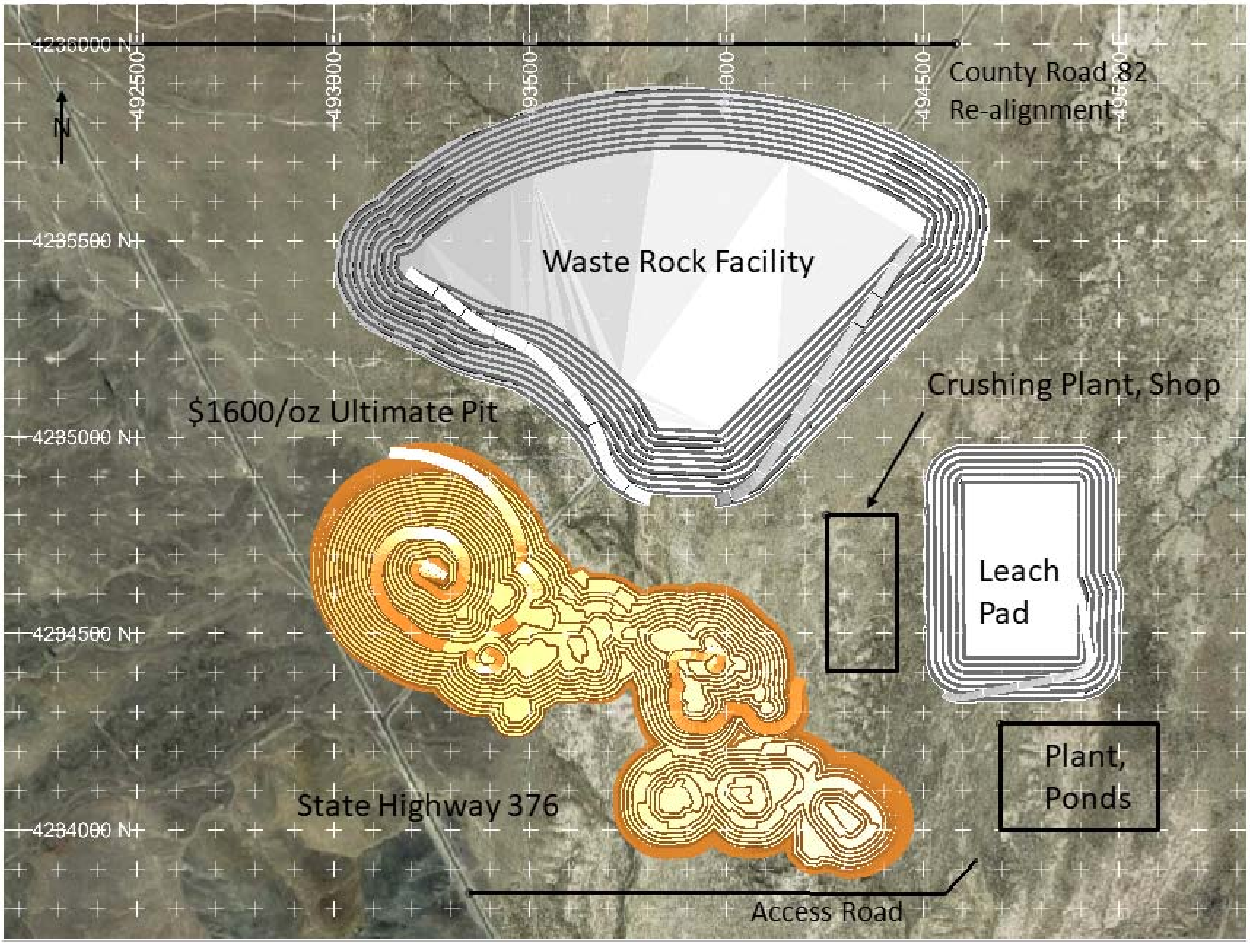


Figure 22-4: Alternate Case Ultimate Mine Design

The DCF model for the $1600 alternate case is included as Appendix G.

23 Adjacent Properties

There are no discovered deposits immediately adjacent to the Tonopah property, although there are a number along the Walker Lane trend.

The Round Mountain Mine is located 30 miles north of the Tonopah property. Round Mountain has been in production, from both historic underground and current open pit operations, since 1906. The Round Mountain deposit is of the low sulfidation, volcanic hosted epithermal gold deposit type. The Round Mountain mine has produced over 10 million ounces since 1977 (Kinross Gold Corporation website, 2017).

The historic mining district of Tonopah lies 20 miles southwest of the Tonopah property. The Manhattan gold deposit, which hosts gold mineralization within a sedimentary sequence of rocks similar to those at the Tonopah property, is located 20 miles to the north. Underground mining was conducted at Manhattan from 1905 to 1947. Large scale, open pit mining operations were active at Manhattan from 1979 to 1988. Manhattan reportedly has proven and probable reserves of 1.7 million tons grading 0.13 oz. Au/ton (4.457 g Au/t) (Goodall, 2001).

The proximity and similarities of the property to these well-documented gold deposits does not, on its own, indicate that the Tonopah property should be similarly mineralized.

24 Other Relevant Data & Information

Gustavson does not believe there is additional relevant information not disclosed elsewhere in this document.

25 Interpretation & Conclusions

25.1 Interpretation & Conclusions

Viva has acquired title to the Tonopah project, along with a significant database of technical information, drill data, geologic interpretation, and preliminary metallurgical data. The data are of industry standard quality and can be used for resource estimation for the project.

Viva's work in updating and clarifying the drill database, particularly collar survey locations, improves the overall quality of the database available for resource estimation.

The 2018-2019 drill programs provided confirmation of the historical database, generally intersecting grades and thicknesses of mineralization consistent with the 2018 block model, and providing additional data for geostatistical support of improved resource classification from inferred to measured and indicated.

Additional infill drilling within the main mineralized areas may convert additional inferred resource to measured and indicated, but is unlikely to significantly increase the overall resource quantity. Further drilling should be targeted at extending mineralization along strike on the Opa / Tv contact, particularly where cross faulting or conjugate faulting may provide an environment favorable to higher grade mineralization, or in areas where the Opa/ Tv contact is inferred to be shallow in depth and thus more likely amenable to open pit mining. Additional drilling should target inferred resource within the pit limits, with the objective of upgrading these resources to measured and indicated categories to support feasibility studies.

The Tonopah project contains a significant gold resource with good continuity at relatively low cutoff grades, and with significant contribution from higher-grade zones. The resource as reported is contained within a pit shell and appears amenable to open pit mining methods.

Initial Metallurgical test work shows that the deposit is amenable to cyanide leaching. Initial Column leach test work provides preliminary recovery projections for the Tonopah project that are used in this PEA.

The PEA indicates that at the gold prices considered, the project shows potential to be developed as a mining operation. A sensitivity executed at near-spot prices indicates additional potential for the deposit at higher metal prices. Gustavson recommends that the project proceed to a feasibility study, including appropriate test work to refine the engineering assumptions, with the overall objective of delineating mineral reserves and forming the basis of a production decision.

25.2 Risks and Uncertainties

The Tonopah project is subject to risks and uncertainties typical of gold projects, particularly risk in commodity prices and the precious metals equity markets. Lower metals prices or lack of precious metals equity market interest or activity could render the project uneconomic or reduce access to project financing.

Specific risks to the project exploration and subsequent mine development center primarily around water use and non-degradation of waters, cultural resources mitigation, and public road relocation, as discussed in Section 4, Section 24, and Section 20. Each of these risks appears to be manageable; however, there is potential to increase the operating or capital cost for the project, or delay or stop development activities.

The existing exploration data appear to be of high quality, but errors or omissions in the database could potentially reduce the reliability of resource estimates prepared using this information, which could negatively impact the project.

Metallurgical data appears to be of reasonable quality, but it is still preliminary. Incomplete classification of material types or misunderstanding of the representativeness of metallurgical samples could lead to a change in recovery assumptions. Metallurgical recoveries appear to be sensitive to particle size. Further test work in needed to confirm crush sizes for optimal extraction and to refine cost parameters.

This is a preliminary economic assessment, which is based on engineering assumptions related to operating cost, capital cost, recovery, and other engineering inputs. Further test work or analysis may modify these assumptions in ways which negatively impact project economics.

26 Recommendations

26.1 General recommendations:

Gustavson recommends that ongoing digital database additions/upgrades continue so that the complete database includes all assay data, including gold, silver, and trace element geochemistry, all available primary observational data on lithology and alteration, and appropriate and available metadata about drilling, sampling, and survey. This will improve the reliability and verifiability of the assay database, as well as making alteration and trace element geochemistry available for geological and geometallurgical modeling efforts.

Gustavson recommends that additional specific gravity determinations be made, both for mineral and waste lithologies, to aid in creation of a more robust bulk density model for the deposit.

Gustavson recommends that ongoing drilling campaigns conducted by Viva continue to use the best practices established by previous workers on the project, particularly with regard to sampling and laboratory assay protocols, minimizing potential assay variability caused by coarse gold in the system. Gustavson further recommends that a comprehensive QA/QC program be maintained, including insertion of blanks, standards, and lab duplicates in the sample stream to monitor laboratory performance.

Gustavson recommends that Feasibility level metallurgical testing be completed to support cost and recovery assumptions for further studies. Test work should include PQ core to provide coarse material for comminution studies and to inform more detailed test work at Feasibility level as the project moves forward.

There is an existing inventory of oriented core from various areas on the project, some from peripheral (near-pit-wall) areas, and some from the center of the deposit. Gustavson recommends that the available core be reviewed to determine its utility for geotechnical evaluation of pit slope angles. This review should determine overall slope angles and make recommendations for any additional test work required.

Gustavson recommends that Viva continue advancing the Tonopah Project by completing a Feasibility Study to establish reserves and to clarify the economic potential of the project.

Gustavson recommends that historical data with regard to cyanide shake assay be reviewed as to aid in understanding of possible recovery differences by lithology and alteration type.

Gustavson recommends that exploration be focused on two areas: First, there are areas of inferred mineralization within the PEA pits which should be targeted to confirm grades and to potentially improve classification to measured and indicated. Second, the western section of the resource area has potential for expansion to the west, which would have the potential to expand the resource and to reduce stripping in the western portion of the pit. Exploration drilling should be targeted to step out from the western extents of the estimated resource to determine whether additional resource can be estimated in these areas.

26.2 Specific Work Plan:

A proposed drilling program is recommended in two segments. Approximately 1,000 meters of PQ drilling in 4-6 holes are recommended to provide material for metallurgical and comminution studies. Approximately 4,000 meters of RC drilling in 10 -15 holes are recommended to target in-pit inferred material and extensions of the resource zone to the west. Samples from these RC holes could also provide additional fresh material for metallurgical samples.

Metallurgical test work should be completed with the objective of providing information for cost and recovery assumptions to be incorporated into the Feasibility Study, as well as to refine process design criteria.

Long-lead baseline work should be initiated for environmental, water quality monitoring, and archaeological studies to support development efforts.

Existing core and oriented core should be reviewed, and a geotechnical assessment made to propose pit slope angles and to make recommendations for any further geotechnical drilling requirements.

Complete a Feasibility Study to clarify the economic potential of the project, and to declare reserves.

The proposed work plan, including completion of a PFS, metallurgical and exploration drilling, metallurgical test program and ongoing environmental test work, is expected to cost $1.85 million.

26.2.1 2020-2021 Project Budget

Table 26-1: Project Budget

Budget Item	Anticipated Cost	Dependencies
Exploration Drilling and Metallurgical Drill Program	$1,000,000	Drill Targeting
Metallurgical Test Work Program	$300,000	Met Data Review & Sample Availability
Environmental and Archaeological Studies	$200,000	Well-points already installed
Geotechnical Data Review & Recommendations	$50,000	Review of existing core inventory.
43-101 Report – Pre-Feasibility Study	$300,000	Existing Resource Estimate, Detailed Metallurgical Review, Detailed Cost studies
2020-21 Project Budget	**$1,850,000**	

27 References (Item 27)

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28 Glossary

28.1 Mineral Resources

The mineral resources and mineral reserves have been classified according to the "CIM Definition Standards for Mineral Resources and Mineral Reserves" (May 10, 2014). Accordingly, the Resources have been classified as Measured, Indicated or Inferred, any Reserves have been classified as Proven, and Probable based on the Measured and Indicated Resources as defined below.

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

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

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

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

28.2 Mineral Reserves

Modifying Factors are considerations used to convert Mineral Resources to Mineral Reserves. These include, but are not restricted to, mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors.

A **Mineral Reserve** is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at Pre-Feasibility or Feasibility level as appropriate that include application of Modifying Factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified.

The reference point at which Mineral Reserves are defined, usually the point where the ore is delivered to the processing plant, must be stated. It is important that, in all situations where the reference point is different, such as for a saleable product, a clarifying statement is included to ensure that the reader is fully informed as to what is being reported.

The public disclosure of a Mineral Reserve must be demonstrated by a Pre-Feasibility Study or Feasibility Study.

A **Probable Mineral Reserve** is the economically mineable part of an Indicated, and in some circumstances, a Measured Mineral Resource. The confidence in the Modifying Factors applying to a Probable Mineral Reserve is lower than that applying to a Proven Mineral Reserve. The Qualified Person(s) may elect, to convert Measured Mineral Resources to Probable Mineral Reserves if the confidence in the Modifying Factors is lower than that applied to a Proven Mineral Reserve.

Probable Mineral Reserve estimates must be demonstrated to be economic, at the time of reporting, by at least a Pre-Feasibility Study.

A **Proven Mineral Reserve** is the economically mineable part of a Measured Mineral Resource. A Proven Mineral Reserve implies a high degree of confidence in the Modifying Factors. Application of the Proven Mineral Reserve category implies that the Qualified Person has the highest degree of confidence in the estimate with the consequent expectation in the minds of the readers of the report. The term should be restricted to that part of the deposit where production planning is taking place and for which any variation in the estimate would not significantly affect the potential economic viability of the deposit.

Proven Mineral Reserve estimates must be demonstrated to be economic, at the time of reporting, by at least a Pre-Feasibility Study. Within the CIM Definition standards the term Proved Mineral Reserve is an equivalent term to a Proven Mineral Reserve.

28.3 Glossary

The following general mining terms may be used in this report.

Table 28-1: Glossary

Term	Definition
Assay:	The chemical analysis of mineral samples to determine the metal content.
Capital Expenditure:	All other expenditures not classified as operating costs.
Crushing:	Initial process of reducing ore particle size to render it more amenable for further processing.
Cut-off Grade (CoG):	The grade of mineralized rock, which determines whether it is economic to recover its mineral content by further concentration.
Dilution:	Waste, which is unavoidably mined with ore.
Dip:	Angle of inclination of a geological feature/rock from the horizontal.
Fault:	The surface of a fracture along which movement has occurred.
Footwall:	The underlying side of an orebody or stope.
Gangue:	Non-valuable components of the ore.
Grade:	The measure of concentration of gold within mineralized rock.
Hangingwall:	The overlying side of an orebody or slope.
Haulage:	A horizontal underground excavation which is used to transport mined ore.
Igneous:	Primary crystalline rock formed by the solidification of magma.
Level:	Horizontal tunnel the primary purpose is the transportation of personnel and materials.
Lithological:	Geological description pertaining to different rock types.
LoM Plans:	Life-of-Mine plans.
LRP:	Long Range Plan.
Material Properties:	Mine properties.
Milling:	A general term used to describe the process in which the ore is crushed and ground and subjected to physical or chemical treatment to extract the valuable metals to a concentrate or finished product.
Mineral/Mining Lease:	A lease area for which mineral rights are held.
Mining Assets:	The Material Properties and Significant Exploration Properties.
Ongoing Capital:	Capital estimates of a routine nature, which is necessary for sustaining operations.
Ore Reserve:	See Mineral Reserve.
Pillar:	Rock left behind to help support the excavations in an underground mine.
RoM:	Run-of-Mine.
Sedimentary:	Pertaining to rocks formed by the accumulation of sediments, formed by the erosion of other rocks.
Shaft:	An opening cut downwards from the surface for transporting personnel, equipment, supplies, ore and waste.
Sill:	A thin, tabular, horizontal to sub-horizontal body of igneous rock formed by the injection of magma into planar zones of weakness.
Stope:	Underground void created by mining.
Stratigraphy:	The study of stratified rocks in terms of time and space.

Term	Definition
Strike:	Direction of line formed by the intersection of strata surfaces with the horizontal plane, always perpendicular to the dip direction.
Sulfide:	A sulfur bearing mineral.
Tailings:	Finely ground waste rock from which valuable minerals or metals have been extracted.
Total Expenditure:	All expenditures including those of an operating and capital nature.

28.4 Definition of Terms

The following abbreviations may be used in this report.

Table 28-2: Abbreviations

Abbreviation	Unit or Term
A	ampere
AA	atomic absorption
A/m^2	amperes per square meter
Ag	silver
Au	gold
°C	degrees Centigrade
CCD	counter-current decantation
CIL	carbon-in-leach
CoG	cut-off grade
cm	centimeter
cm^2	square centimeter
cm^3	cubic centimeter
cfm	cubic feet per minute
ConfC	confidence code
CRec	core recovery
CSS	closed-side setting
CTW	calculated true width
Cu	copper
°	degree (degrees)
dia.	Diameter
EDX	energy dispersive x-ray
EIS	Environmental Impact Statement
EMP	Environmental Management Plan
FA	fire assay
ft	foot (feet)
ft^2	square foot (feet)
ft^3	cubic foot (feet)
g	gram
gal	gallon
g/L	gram per liter
g-mol	gram-mole
gpm	gallons per minute
g/t	grams per tonne
ha	hectares
Hp	Horsepower
HQ	drill core diameter of ~63.5 mm
HTW	horizontal true width
ICP-MS	inductively coupled plasma mass spectrometry
ID2	inverse-distance squared
ID3	inverse-distance cubed
kA	kiloamperes
kg	kilograms

Abbreviation	Unit or Term
km	kilometer
km^2	square kilometer
koz	thousand troy ounce
kt	thousand tonnes
kt/d	thousand tonnes per day
kt/y	thousand tonnes per year
kV	kilovolt
kW	kilowatt
kWh	kilowatt-hour
kWh/t	kilowatt-hour per metric tonne
L	liter
L/sec	liters per second
L/sec/m	liters per second per meter
lb	pound
LHD	Long-Haul Dump truck
LOI	Loss On Ignition
LoM	Life-of-Mine
m	meter
m^2	square meter
m^3	cubic meter
masl	meters above sea level
Ma	millions of years before present
mg/L	milligrams/liter
MLA	mineral liberation analysis
mm	millimeter
mm^2	square millimeter
mm^3	cubic millimeter
MME	Mine & Mill Engineering
Moz	million troy ounces
Mt	million tonnes
MTW	measured true width
MW	million watts
m.y.	million years
NGO	non-governmental organization
NI 43-101	Canadian National Instrument 43-101
NQ	drill core diameter of ~47.5 mm
opt	troy ounce per ton
OSC	Ontario Securities Commission
oz	troy ounce
%	Percent
Pb	lead
PGM	Pilot Gold Mill
PLC	Programmable Logic Controller
PLS	Pregnant Leach Solution
PMF	probable maximum flood
ppb	parts per billion
ppm	parts per million
QA/QC	Quality Assurance/Quality Control

Abbreviation	Unit or Term
RC	rotary circulation drilling
RoM	Run-of-Mine
RQD	Rock Quality Description
Sb	antimony
sec	second
SEM	Scanning Electron Microscope
SG	specific gravity
SPT	standard penetration testing
st	short ton (2,000 pounds)
t	tonne (metric ton) (2,204.6 pounds)
t/h	tonnes per hour
t/d	tonnes per day
t/y	tonnes per year
TSF	tailings storage facility
TSP	total suspended particulates
μm	micron or microns
V	volts
VFD	variable frequency drive
W	Tungsten or watts
XRD	x-ray diffraction
XRF	x-ray fluorescence
Y	Year
Zn	zinc

Appendix A- Certificate of Author Forms

THOMAS C. MATTHEWS

Principal Resource Geologist

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CERTIFICATE of AUTHOR

I, Thomas C. Matthews do hereby certify that:

1. I am currently employed as Principal Resource Geologist by:

   Gustavson Associates, LLC
   270 Union Boulevard, Suite 440
   Lakewood, CO, USA, 80228

2. I graduated with a Bachelor's of Science degree in Geology from University of Rochester in 1994.

3. I am a Qualified Professional (QP) Member of the Mining and Metallurgical Society of America (01455QP) with special expertise in Geology and Ore Reserves.

4. I have worked as a geologist for a total of 24 years since my graduation from university, as an employee of an exploration company, a mining company, and as a consultant. My relevant experience includes exploration, geologic modeling, and resource estimation, reserves definition in feasibility studies, ore control systems, and mine-model reconciliation, particularly for epithermal gold systems.

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 sections 1-12, 14, 15, 18 of the technical report entitled "NI 43-101 Technical Report Preliminary Economic Assessment, Tonopah Project, Nye County NV," dated June 12, 2020, (the "Technical Report"), and am also responsible for the overall organization and content of the document. My last visit to the project site was on September 24, 2019 for one day.

7. I previously worked on the property that is the subject of the Technical Report, as the author of a technical report entitled "NI 43-101 Technical Report on Exploration Results, Tonopah Project, Nye County NV," with an effective date of July 15, 2019, also May 1, 2017, and as the author of a technical report entitled "NI 43-101 Technical Report on Resources, Tonopah Project, Nye County NV," dated March 27, 2018.

8. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.

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

10. 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 on their websites accessible by the public, of the Technical Report.

11. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 12th day of June 2020

/s/ *Thomas C. Matthews*

Signature of Qualified Person

Thomas C. Matthews

Printed Name of Qualified Person

CHRISTOPHER EMANUEL

Senior Mining Engineer

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CERTIFICATE of AUTHOR

I, Christopher Emanuel do hereby certify that:

1. I am currently employed as Senior Mining Engineer by:

   Gustavson Associates, LLC
   270 Union Boulevard, Suite 440
   Lakewood, CO, USA, 80228

2. I graduated with a Bachelor's of Science degree in Mining Engineering from the Colorado School of Mines, Golden, CO.

3. I am a Registered Member of the Society of Mining Metallurgy & Exploration (4151007RM).

4. I have worked as a mining engineer for a total of 15 years since my graduation from university, as an employee of a mining company, underground mine contractor, safety trainer and as a consultant. My relevant experience includes mine design, capital and operating cost estimates, and financial modeling.

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 sections 19 – 21 and portions of 16 & 17 of the technical report entitled "NI 43-101 Technical Report on Preliminary Economic Assessment, Tonopah Project, Nye County NV," dated June 12, 2020, (the "Technical Report"),

7. I previously worked on the property that is the subject of the Technical Report, as a contributing author of a technical report entitled "NI 43-101 Technical Report on Exploration Results, Tonopah Project, Nye County NV," with an effective date of July 15, 2019.

8. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.

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

10. 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 on their websites accessible by the public, of the Technical Report.

11. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 12th day of June 2020

/s/ *Christopher Emanuel*

Signature of Qualified Person

Christopher Emanuel

Printed Name of Qualified Person

DEEPAK MALHOTRA, PH.D.

PRESIDENT

RESOURCE DEVELOPMENT, INC. (RDI)

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CERTIFICATE of AUTHOR

I, Deepak Malhotra, PhD do hereby certify that:

1. I am President of:

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   Wheat Ridge, CO, USA, 80033

2. I graduated with a degree in Master of Science from Colorado School of Mines in 1973. In addition, I have obtained a PhD in Mineral Economics from Colorado School of Mines in 1977.

3. I am a registered member of the Society of Mining, Metallurgy and Exploration, Inc. (SME), member No. 2006420RM.

4. I have worked as a mineral processing engineer and mineral economist for a total of 40 years since my graduation from university. I have experience in similar project types inclusive of those in the Western United States.

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 sections 19 – 21 and portions of 16 & 17 of the technical report entitled "NI 43-101 Technical Report on Preliminary Economic Assessment, Tonopah Project, Nye County NV," dated June 12, 2020, (the "Technical Report").

7. I previously worked on the property that is the subject of the Technical Report, as a contributing author of a technical report entitled "NI 43-101 Technical Report on Exploration Results, Tonopah Project, Nye County NV," with an effective date of July 15, 2019.

8. I am independent of the issuers applying all of the tests in section 1.5 of National Instrument 43-101.

9. I have read NI 43-101 and Form 43-101F1, and the PEA has been prepared in compliance with that instrument and form.

10. As of the effective date of this PEA, to the best of my knowledge, information and belief, the PEA contains all scientific and technical information that is required to be disclosed to make the PEA not misleading.

Dated this 12 day June 2020.

/s/ *Deepak Malhotra*

Signature of Qualified Person

Deepak Malhotra .

Print name of Qualified Person

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CERTIFICATE of AUTHOR

I, Sarah Milne do hereby certify that:

12. I am currently employed as Senior Mining Engineer by:

    Gustavson Associates, LLC
    270 Union Boulevard, Suite 440
    Lakewood, CO, USA, 80228

13. I graduated with a Bachelor's of Science degree in Mining Engineering from the University of Alaska Fairbanks, 2011.

14. I am a Registered Member of the Society of Mining Metallurgy & Exploration and a Colorado Professional Engineer (0056435).

15. I have worked as a mining engineer for a total of 9 years since my graduation from university, as an employee of a mining company and as a consultant. My relevant experience includes mine design, mine scheduling, surface and underground sites, capital and operating cost estimates.

16. 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.

17. I am responsible for portions of section 16 of the technical report entitled "NI 43-101 Technical Report on Preliminary Economic Assessment, Tonopah Project, Nye County NV," dated June 12, 2020, (the "Technical Report").

18. I have not previously worked on the property.

19. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.

20. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

21. 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 on their websites accessible by the public, of the Technical Report.

22. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 12th day of June 2020

/s/ *Sarah Milne*

Signature of Qualified Person

Sarah Milne

Printed Name of Qualified Person

Appendix B- Significant Drill Intersections from 2011 Drilling Program

Hole	Area	Type	From (m)	To (m)	Length (m)	Grade (opt Au)	Grade (ppm Au)
MW11-01C	Discovery	Core	85	89	5	0.011	0.377
			104	109	5	0.015	0.514
			130	134	3	0.011	0.377
MW11-02C	Discovery	Core	96	110	13	0.03	1.029
MW11-03C	Discovery	Core	76	84	8	0.013	0.446
MW11-04C	Discovery	Core	34	87	52	0.058	1.989
	includes:		62	76	14	0.11	3.771
	includes:		81	85	4	0.216	7.406
			108	119	11	0.127	4.354
	includes:		111	117	6	0.215	7.371
			156	160	4	0.023	0.789
MW11-05C	Discovery	Core	45	93	48	0.059	2.023
	includes:		89	90	0	2.569	88.081
MW11-06C	Discovery	Core	59	65	6	0.051	1.749
			69	114	45	0.033	1.131
			129	132	3	0.025	0.857
MW11-07C	Discovery	Core	62	84	22	0.141	4.834
	includes:		76	84	7	0.254	8.709
MW11-08C	Discovery	Core	53	100	47	0.078	2.674
	includes:		60	76	15	0.185	6.343
MW11-09C	Discovery	Core	47	93	46	0.224	7.68
	includes:		64	78	14	0.668	22.903
	includes:		67	69	2	2.287	78.412
	includes:		69	69	0	9.768	334.906
MW11-10C	Discovery	Core	41	57	16	0.02	0.686
			63	80	17	0.103	3.531
	includes:		64	79	15	0.117	4.011
			94	103	9	0.05	1.714
MW11-11C	Discovery	Core	39	70	30	0.061	2.091
	includes:		50	53	3	0.212	7.269
	includes:		62	66	4	0.147	5.04
			76	80	5	0.063	2.16

Hole	Area	Type	From (m)	To (m)	Length (m)	Grade (opt Au)	Grade (ppm Au)
MW11-12C	121 zone	Core	91	94	3	0.014	0.48
			165	190	25	0.045	1.543
MW11-13C	121 zone	Core	134	158	24	0.013	0.446
MW11-14C	121 zone	Core	89	95	6	0.021	0.72
			172	181	10	0.044	1.509
			191	197	6	0.113	3.874
	includes:		193	193	0	1.185	40.629
MW11-15C	121 zone	Core	145	156	10	0.023	0.789
MW11-16C	Dauntless	Core	No significant intercepts				
MW11-17C	Dauntless	Core	No significant intercepts				
MW11-18C	Dauntless	Core	55	75	19	0.026	0.891
MW11-19C	Dauntless	Core	49	62	13	0.023	0.789
			85	91	6	0.027	0.926
MW11-20C	Dauntless	Core	No significant intercepts				
MW11-21C	Dauntless	Core	No significant intercepts				
MW11-22C	63-77 zone	Core	127	153	26	0.031	1.063
MW11-23C	63-77 zone	Core	76	81	5	0.011	0.377
			125	130	5	0.045	1.543
MW11-24C	63-77 zone	Core	106	111	5	0.063	2.16
			120	125	5	0.017	0.583
MW11-25C	63-77 zone	Core	No significant intercepts				

Appendix C- List of significant results from 2018-2019 Drilling Program

Tonopah Project
Drill Results for 2018-2019 Winter RC Drill Program

Hole	Azimuth	Dip	From	To	Length	Gold Grade
			Meter	*Meter*	*Meter*	*Gram/Tonne*
TG1906	**200**	**-90**	**0**	**134.0**		
			25.91	44.20	18.3	0.4
TG1905	**210**	**-69**	**0**	**146.3**		
			32.0	53.3	21.3	0.6
TG1904	**270**	**-60**	**0**	**134.0**		
			36.6	41.1	4.6	0.7
			126.5	131.1	4.6	2.4
	including		*128.0*	*129.5*	*1.5*	*6.7*
TG1903	**275**	**-75**	**0**	**140.2**		
			45.7	48.8	3.0	8.0
	including		*47.2*	*48.8*	*1.5*	*15.4*
			70.1	74.7	4.6	26.9
	including		*70.1*	*71.6*	*1.5*	*50.3*
			82.3	115.8	33.5	2.6
	including		*82.3*	*83.8*	*1.5*	*14.1*
	including		*94.5*	*96.0*	*1.5*	*22.7*
			118.9	128.0	9.1	0.6
			132.6	140.2	7.6	0.8
TG1902	**0**	**-70**	**0**	**146.3**		
			10.7	*111.3*	*100.6*	*1.3*

Hole	Azimuth	Dip	From	To	Length	Gold Grade
	including		*41.15*	*47.24*	*6.10*	*3.3*
	including		*60.96*	*64.01*	*3.05*	*4.8*
	including		*83.82*	*91.44*	*7.62*	*4.1*
TG1901	**50**	**-70**	**0**	**65.5**		
			38.1	*53.3*	*15.2*	*0.4*
TG1820	**200**	**-60**	**0**	**119**		
			35.1	*41.1*	*6.1*	*0.3*
TG 1819	**200**	**-69**	**0**	**201**		
			53.3	57.9	4.6	0.4
			62.5	*65.5*	*3.0*	*1.9*
			89.9	*100.6*	*10.7*	*0.6*
TG1818	**100**	**-65**	**0**	**110**		
			21.3	33.5	12.2	0.9
			71.63	74.68	3.0	46.1
	including		*73.15*	*74.68*	*1.5*	*84.9*
TG1817	**58**	**-80**	0	122		
			112.8	118.9	6.1	1.0
			143.3	189.0	45.7	2.2
	Including		*163.1*	*176.8*	*13.7*	*5.1*
	Including		*172.21*	*173.74*	*1.5*	*13.4*
TG1816	**105**	**-60**	**0**	**164**		
			88.4	96.0	7.6	0.3
			108.2	112.8	4.6	6.1
	Including		*108.2*	*109.7*	*1.5*	*16.4*

0.25 gram/tonne used throughout

Tonopah Project
Drill Results for 2018 RC Drill Program

Hole		Depth From	Depth To	Length	Uncapped Gold Grade	Capped* Gold Grade
		Meter	*Meter*	*Meter*	*Gram/Tonne*	*Gram/Tonne*
TG 1814		32	61	29	1.32	1.32
	including	*47.2*	*48.8*	*1.5*	*4.76*	*4.76*
	Including	*53.3*	*57.9*	*4.6*	*4.05*	*4.05*
TG 1813		129.5	140.2	10.7	0.45	0.45
TG 1811		77.7	83.8	6.1	0.49	0.49
and		103.6	118.9	15.2	0.41	0.41
TG 1809		51.8	56.4	4.6	0.28	0.28
and		68.6	76.2	7.6	0.87	0.87
and		86.9	97.5	10.7	2.57	2.57
	Including	*89.9*	*91.4*	*1.5*	*12.90*	*12.90*
TG1815		68.6	82.3	13.7	1.64	1.64
	including	*77.7*	*79.2*	*1.52*	*8.79*	*8.79*
TG1812		89.9	100.6	10.7	3.07	3.07
	including	*89.9*	*91.4*	*1.5*	*19.2*	*19.20*
and		112.8	120.4	7.6	0.37	*0.37*
TG 1810		91.4	106.7	15.3	1.21	1.21
	including	*96*	*97.5*	1.5	*5.68*	*5.68*
and		121.9	125	3.1	0.35	*0.35*
and		129.5	132.6	3.1	0.57	*0.57*
TG 1808		54.9	57.9	*3.0*	0.5	0.5
and		64.0	73.2	9.1	25.4	5.8
	including	*65.5*	*67.1*	*1.5*	*138.0*	*20.0*

Hole		Depth From	To	Length	Uncapped Gold Grade	Capped* Gold Grade
	including	*70.1*	*71.6*	*1.5*	*8.9*	*8.9*
and		83.8	89.9	6.1	0.4	0.4
and		97.5	102.1	4.6	5.5	5.5
	Including	*99.1*	*100.6*	*1.5*	*14.9*	*14.9*
and		108.2	120.4	12.2	1.2	1.2
and						
and	TD	123.4	125.0	1.5	0.6	0.6
All zones		*54.9*	*125.0*	70.1	*3.9*	*1.4*
TG 1807		10.7	19.8	9.1	0.3	0.3
and		35.1	74.7	39.6	2.0	2.0
	Including	*59.4*	*68.6*	*9.1*	*4.5*	*4.5*
and		80.8	83.8	3.0	0.4	0.4
and	TD	93.0	94.5	1.5	0.4	0.4
All zones		10.7	94.5	83.8	1.0	1.0
TG 1806		21.3	29.0	7.6	0.3	0.3
and	TD	71.6	74.7	3.0	0.7	0.7
TG 1805		38.1	39.6	1.5	0.5	0.5

* Capped at 20 grams/tonne
0.25 gram/tonne used throughout

Appendix D- List of Drill holes

Drill hole Name	Easting	Northing	Elevation	Total Depth	Azimuth	Dip	Drill hole Type	Year	Campaign
TH-01	500927	4222833	1920	91.4	0	-90	RC	1980	Felmont
TH-10	501548	4224661	1920	103.6	0	-90	RC	1980	Felmont
TH-11	500842	4224881	1920	91.4	0	-90	RC	1980	Felmont
TH-12	500123	4225114	1920	91.4	0	-90	RC	1980	Felmont
TH-13	500010	4224390	1920	91.4	0	-90	RC	1980	Felmont
TH-14	499755	4225627	1920	109.7	0	-90	RC	1980	Felmont
TH-15	500954	4225437	1920	91.4	0	-90	RC	1980	Felmont
TH-16	501127	4225821	1920	85.3	0	-90	RC	1980	Felmont
TH-17	500387	4225980	1920	91.4	0	-90	RC	1980	Felmont
TH-18	500763	4226117	1920	91.4	0	-90	RC	1980	Felmont
TH-19	500122	4226462	1920	91.4	0	-90	RC	1980	Felmont
TH-20	499995	4227255	1920	91.4	0	-90	RC	1980	Felmont
TH-21	500447	4225586	1920	73.2	0	-90	RC	1980	Felmont
TH-22	500018	4226155	1920	91.4	0	-90	RC	1980	Felmont
TH-23	499559	4227955	1920	91.4	0	-90	RC	1980	Felmont
TH-24	498694	4228529	1920	91.4	0	-90	RC	1980	Felmont
TH-25	498609	4228146	1920	91.4	0	-90	RC	1980	Felmont
TH-26	499121	4227746	1920	91.4	0	-90	RC	1980	Felmont
TH-27	499573	4227373	1920	91.4	0	-90	RC	1980	Felmont
TH-03	501362	4222449	1920	91.4	0	-90	RC	1981	Felmont
TH-04	500943	4222622	1920	91.4	0	-90	RC	1981	Felmont
TH-05	500657	4222815	1920	91.4	0	-90	RC	1981	Felmont
TH-06	501441	4223114	1920	85.3	0	-90	RC	1981	Felmont
TH-07	501437	4223484	1920	97.5	0	-90	RC	1981	Felmont
TH-08	501206	4223939	1920	91.4	0	-90	RC	1981	Felmont
TH-09	502012	4224066	1920	91.4	0	-90	RC	1981	Felmont
TH-29	499260	4226418	1920	91.4	0	-90	RC	1981	Felmont
TH-30	499560	4227402	1920	99.1	0	-90	RC	1981	Felmont
TH-31	499583	4227329	1920	93.0	0	-90	RC	1981	Felmont
TH-32	499458	4228205	1920	99.1	0	-90	RC	1981	Felmont
TH-33	499607	4227389	1920	93.0	0	-90	RC	1981	Felmont
TH-34	499405	4229577	1920	111.3	0	-90	RC	1981	Felmont
TH-35	499446	4228044	1920	93.0	0	-90	RC	1981	Felmont
TH-36	499554	4227317	1920	111.3	0	-90	RC	1981	Felmont
TH-37	499550	4227426	1920	93.0	0	-90	RC	1981	Felmont
TH-38	499592	4227411	1920	93.0	0	-90	RC	1981	Felmont
TH-39	499609	4227349	1920	93.0	0	-90	RC	1981	Felmont
TH-40	499629	4227397	1920	99.1	0	-90	RC	1981	Felmont
TH-41	499623	4227325	1920	93.0	0	-90	RC	1981	Felmont

Drill hole Name	Easting	Northing	Elevation	Total Depth	Azimuth	Dip	Drill hole Type	Year	Campaign
TH-42	500184	4226897	1920	61.0	0	-90	RC	1981	Felmont
TH-43	500168	4226616	1920	109.7	0	-90	RC	1981	Felmont
TH-44	500342	4226429	1920	91.4	0	-90	RC	1981	Felmont
TH-45	500221	4226967	1920	97.5	0	-90	RC	1981	Felmont
TH-46	500153	4226903	1920	109.7	0	-90	RC	1981	Felmont
TH-47	500127	4226840	1920	109.7	0	-90	RC	1981	Felmont
TH-48	500067	4226821	1920	109.7	0	-90	RC	1981	Felmont
TH-49	500025	4226784	1920	109.7	0	-90	RC	1981	Felmont
TH-50	499738	4227056	1920	109.7	0	-90	RC	1981	Felmont
TH-51	499871	4227118	1920	109.7	0	-90	RC	1981	Felmont
TH-52	499666	4227219	1920	109.7	0	-90	RC	1981	Felmont
TH-53	499764	4227264	1920	109.7	0	-90	RC	1981	Felmont
TH-54	499683	4227427	1920	99.1	0	-90	RC	1981	Felmont
TH-55	499740	4227458	1920	111.3	0	-90	RC	1981	Felmont
TH-56	499522	4227483	1920	93.0	0	-90	RC	1981	Felmont
TH-57	500554	4225904	1920	91.4	0	-90	RC	1981	Felmont
TH-58	500918	4225677	1920	79.2	0	-90	RC	1981	Felmont
TH-59	501267	4225582	1920	109.7	0	-90	RC	1981	Felmont
TH-60	501322	4225647	1920	74.7	0	-90	RC	1981	Felmont
TH-61	500541	4225754	1920	64.0	0	-90	RC	1981	Felmont
TH-62	499993	4225732	1920	115.8	0	-90	RC	1981	Felmont
TH-64	501089	4222988	1920	166.1	0	-90	RC	1981	Felmont
TH-65	501050	4222863	1920	121.9	0	-90	RC	1981	Felmont
TH-66	500929	4222894	1920	121.9	0	-90	RC	1981	Felmont
TH-67	500874	4222910	1920	121.9	0	-90	RC	1981	Felmont
TH-68	500805	4222924	1920	121.9	0	-90	RC	1981	Felmont
TH-69	500796	4222864	1920	109.7	0	-90	RC	1981	Felmont
TH-70	500779	4222802	1920	91.4	0	-90	RC	1981	Felmont
TH-71	500864	4222690	1920	91.4	0	-90	RC	1981	Felmont
TH-72	500840	4222614	1920	91.4	0	-90	RC	1981	Felmont
TH-73	500758	4222640	1920	91.4	0	-90	RC	1981	Felmont
TH-74	500400	4224769	1920	105.2	0	-90	RC	1981	Felmont
TH-75	502095	4222243	1920	86.9	0	-90	RC	1981	Felmont
TH-76	500746	4223066	1920	150.9	0	-90	RC	1981	Felmont
TH-77	500853	4223041	1920	170.7	0	-90	RC	1981	Felmont
TH-78	500971	4223010	1920	152.4	0	-90	RC	1981	Felmont
TH-79	500698	4222952	1920	152.4	0	-90	RC	1981	Felmont
TH-80	500673	4222875	1920	91.4	0	-90	RC	1981	Felmont
TH-82	500415	4222886	1920	97.5	0	-90	RC	1981	Felmont
TH-83	501019	4222748	1920	134.1	0	-90	RC	1981	Felmont
TH-84	500509	4223123	1920	128.0	0	-90	RC	1981	Felmont

Drill hole Name	Easting	Northing	Elevation	Total Depth	Azimuth	Dip	Drill hole Type	Year	Campaign
TH-85	502029	4222301	1920	88.4	0	-90	RC	1981	Felmont
TH-86	501931	4222259	1920	91.4	0	-90	RC	1981	Felmont
TH-87	502047	4222145	1920	91.4	0	-90	RC	1981	Felmont
TH-88	500687	4226068	1920	91.4	0	-90	RC	1981	Felmont
TH-89	500183	4225137	1920	91.4	0	-90	RC	1981	Felmont
TH-90	500069	4225081	1920	91.4	0	-90	RC	1981	Felmont
TH-91	500640	4226064	1920	91.4	0	-90	RC	1981	Felmont
TH-92	500201	4225744	1920	91.4	0	-90	RC	1981	Felmont
TH-93	500981	4225629	1920	91.4	0	-90	RC	1981	Felmont
TH-94	500977	4225760	1920	91.4	0	-90	RC	1981	Felmont
TH-95	500651	4226031	1920	91.4	0	-90	RC	1981	Felmont
TH-96	500946	4225894	1920	91.4	0	-90	RC	1981	Felmont
SP-88-01	491606	4235678	1785	109.7	360	-90	RC	1988	Coeur d'Alene
SP-88-02	492032	4235499	1792	96.0	360	-90	RC	1988	Coeur d'Alene
SP-88-03	492072	4235720	1789	121.9	360	-90	RC	1988	Coeur d'Alene
MW-M-01	492012	4235020	1817	117.3	360	-90	RC	1990	Rio Algom
MW-M-02	492184	4234973	1804	106.7	360	-90	RC	1990	Rio Algom
MW-M-03	491880	4234873	1835	121.9	90	-60	RC	1990	Rio Algom
MW-M-04	491820	4235038	1823	91.4	270	-60	RC	1990	Rio Algom
MW-M-05	491797	4235247	1804	114.3	360	-90	RC	1990	Rio Algom
MW-M-06	492070	4235207	1811	106.7	360	-90	RC	1990	Rio Algom
MW-M-07	492282	4234478	1798	61.0	360	-90	RC	1990	Rio Algom
MW-M-08	491884	4234365	1811	121.9	360	-90	RC	1990	Rio Algom
MW-M-09	491784	4235405	1801	135.6	360	-90	RC	1990	Rio Algom
MW-M-10	491634	4235403	1811	163.1	360	-90	RC	1990	Rio Algom
MW-M-11	491787	4235572	1798	144.8	360	-90	RC	1990	Rio Algom
MW-M-12	491881	4235313	1801	123.4	360	-90	RC	1990	Rio Algom
MW-M-13	491892	4235487	1795	182.9	360	-90	RC	1990	Rio Algom
MW-M-14	491669	4235535	1801	169.2	360	-90	RC	1990	Rio Algom
MW-M-15	491960	4235366	1798	140.2	360	-90	RC	1990	Rio Algom
MW-M-16	491672	4235256	1817	166.1	360	-90	RC	1990	Rio Algom
MW-M-17	491680	4235104	1829	152.4	360	-90	RC	1990	Rio Algom
MW-M-18	491887	4235172	1811	150.9	360	-90	RC	1990	Rio Algom
MW-M-19	492002	4235279	1798	152.4	360	-90	RC	1990	Rio Algom
MW-M-20	492082	4235369	1798	152.4	360	-90	RC	1990	Rio Algom
MW-M-21	491681	4235041	1832	182.9	360	-90	RC	1990	Rio Algom
MW-M-22	491614	4235105	1829	182.9	360	-90	RC	1990	Rio Algom
MW-M-23	491679	4235104	1829	213.4	360	-90	RC	1990	Rio Algom
MW-M-24	491682	4235166	1823	178.3	360	-90	RC	1990	Rio Algom

Drill hole Name	Easting	Northing	Elevation	Total Depth	Azimuth	Dip	Drill hole Type	Year	Campaign
MW-M-25	491734	4235167	1817	182.9	360	-90	RC	1990	Rio Algom
MW-M-26	491557	4235170	1835	199.6	360	-90	RC	1990	Rio Algom
MW-M-27	491557	4235041	1841	182.9	360	-90	RC	1990	Rio Algom
MW-M-28	491558	4234915	1850	152.4	360	-90	RC	1990	Rio Algom
MW-M-29	491433	4234981	1850	182.9	360	-90	RC	1990	Rio Algom
MW-M-30	491678	4235213	1817	163.1	360	-90	RC	1990	Rio Algom
MW-M-31	491618	4235161	1829	181.4	360	-90	RC	1990	Rio Algom
MW-M-32	492420	4234964	1798	121.9	360	-90	RC	1990	Rio Algom
MW-M-33	491457	4235346	1823	152.4	360	-90	RC	1991	Rio Algom
MW-M-34	491379	4234823	1856	152.4	360	-90	RC	1991	Rio Algom
MW-M-35	490918	4234797	1893	152.4	360	-90	RC	1991	Rio Algom
MW-M-36	492353	4235427	1783	123.4	360	-90	RC	1991	Rio Algom
MW-M-37	492752	4235207	1783	105.2	360	-90	RC	1991	Rio Algom
MW-M-38	490353	4237280	1832	152.4	360	-90	RC	1991	Rio Algom
MW-M-39	488555	4238621	1861	152.4	360	-90	RC	1991	Rio Algom
MW-M-40	487976	4238804	1872	91.4	360	-90	RC	1991	Rio Algom
MW-M-41	489287	4238895	1849	146.3	360	-90	RC	1991	Rio Algom
MW-001	490860	4236487	1815	170.7	360	-90	RC	1992	Kennecott
MW-002	491120	4235836	1823	152.4	360	-90	RC	1992	Kennecott
MW-003	491218	4235197	1853	231.6	360	-70	RC	1992	Kennecott
MW-004	489550	4236962	1847	167.6	360	-90	RC	1992	Kennecott
MW-005	489795	4237756	1841	152.4	360	-90	RC	1992	Kennecott
MW-006	492964	4233792	1817	106.7	360	-90	RC	1992	Kennecott
MW-007	493349	4233509	1783	88.4	360	-90	RC	1992	Kennecott
MW-008	493452	4237316	1780	125.0	360	-90	RC	1992	Kennecott
MW-009	493004	4235527	1779	170.7	360	-90	RC	1992	Kennecott
MW-010	489230	4238183	1860	152.4	360	-90	RC	1992	Kennecott
MW-011	493354	4234671	1774	121.9	360	-90	RC	1993	Kennecott
MW-012	493461	4234526	1769	152.4	360	-90	RC	1993	Kennecott
MW-013	493487	4234341	1771	128.0	360	-90	RC	1993	Kennecott
MW-014	493275	4234476	1776	152.4	360	-90	RC	1993	Kennecott
MW-015	493670	4234901	1769	213.4	360	-90	RC	1993	Kennecott
MW-016	493533	4234560	1772	121.9	360	-90	RC	1993	Kennecott
MW-017	493783	4234645	1771	182.9	360	-90	RC	1993	Kennecott
MW-018	493643	4234551	1772	152.4	360	-90	RC	1993	Kennecott
MW-019	493447	4234552	1773	103.6	360	-90	RC	1993	Kennecott
MW-020	493473	4234494	1772	61.0	360	-90	RC	1993	Kennecott
MW-021	493437	4234502	1772	45.7	360	-90	RC	1993	Kennecott
MW-022	493487	4234593	1772	97.5	360	-90	RC	1993	Kennecott
MW-023D	493458	4234540	1772	121.0	360	-90	DDH	1993	Kennecott
MW-024	493448	4234613	1772	150.9	360	-90	RC	1993	Kennecott

Drill hole Name	Easting	Northing	Elevation	Total Depth	Azimuth	Dip	Drill hole Type	Year	Campaign
MW-025	493440	4234669	1773	126.5	360	-90	RC	1993	Kennecott
MW-026	493486	4234456	1771	106.7	360	-90	RC	1993	Kennecott
MW-027	493424	4234583	1773	137.2	360	-90	RC	1993	Kennecott
MW-028	493404	4234624	1773	193.5	360	-90	RC	1993	Kennecott
MW-029	493558	4234551	1771	150.9	360	-90	RC	1993	Kennecott
MW-030	493592	4234345	1771	120.4	360	-90	RC	1993	Kennecott
MW-031	493602	4234496	1771	150.9	360	-90	RC	1993	Kennecott
MW-032	493602	4234605	1771	150.9	360	-90	RC	1993	Kennecott
MW-033	493558	4234586	1771	144.8	292	-60	RC	1993	Kennecott
MW-034	493512	4234487	1772	150.9	360	-90	RC	1993	Kennecott
MW-035	493512	4234417	1771	120.4	360	-90	RC	1993	Kennecott
MW-036	493401	4234565	1774	150.9	360	-90	RC	1993	Kennecott
MW-037	493380	4234837	1773	135.6	360	-90	RC	1993	Kennecott
MW-038	493591	4234673	1771	150.9	275	-60	RC	1993	Kennecott
MW-039	493650	4234420	1771	150.9	275	-60	RC	1993	Kennecott
MW-040	493475	4234548	1772	76.2	240	-45	RC	1994	Kennecott
MW-041	493474	4234547	1772	61.0	240	-64	RC	1994	Kennecott
MW-042	493469	4234557	1772	76.2	240	-45	RC	1994	Kennecott
MW-043	493445	4234524	1773	61.0	35	-65	RC	1994	Kennecott
MW-044	493478	4234501	1772	106.7	325	-65	RC	1994	Kennecott
MW-045	493453	4234584	1772	76.2	242	-65	RC	1994	Kennecott
MW-046	493456	4234584	1772	85.3	360	-90	RC	1994	Kennecott
MW-047	493475	4234562	1772	76.2	240	-65	RC	1994	Kennecott
MW-048	493453	4234561	1773	76.2	158	-65	RC	1994	Kennecott
MW-049	493453	4234560	1773	54.9	175	-65	RC	1994	Kennecott
MW-050	493442	4234535	1773	48.8	360	-90	RC	1994	Kennecott
MW-051	493505	4234737	1772	106.7	260	-90	RC	1994	Kennecott
MW-052	493676	4234513	1771	121.9	360	-90	RC	1994	Kennecott
MW-053	493712	4234571	1771	152.4	360	-90	RC	1994	Kennecott
MW-054	493523	4234875	1772	152.4	360	-90	RC	1994	Kennecott
MW-055	493395	4235384	1774	173.7	360	-90	RC	1994	Kennecott
MW-056	493289	4234732	1774	225.6	360	-90	RC	1994	Kennecott
MW-057	493533	4234103	1772	135.6	360	-90	RC	1994	Kennecott
MW-058	493613	4234461	1771	167.6	360	-90	RC	1994	Kennecott
MW-059	493668	4234281	1771	152.4	360	-90	RC	1994	Kennecott
MW-060	492109	4235665	1788	243.8	360	-90	RC	1994	Kennecott
MW-061	493874	4234297	1771	137.2	360	-90	RC	1994	Kennecott
MW-062	494057	4234089	1771	128.0	360	-90	RC	1994	Kennecott
MW-063	493949	4234469	1771	190.5	360	-90	RC	1994	Kennecott
MW-064	492348	4235946	1789	304.8	360	-90	RC	1994	Kennecott
MW-065	494032	4234653	1772	192.0	360	-90	RC	1994	Kennecott

Drill hole Name	Easting	Northing	Elevation	Total Depth	Azimuth	Dip	Drill hole Type	Year	Campaign
MW-066	494984	4234303	1773	243.8	360	-90	RC	1994	Kennecott
MW-067	495141	4233451	1770	181.4	360	-90	RC	1994	Kennecott
MW-068	493944	4234315	1771	121.9	360	-90	RC	1994	Kennecott
MW-069	494433	4235813	1782	54.9	360	-90	RC	1994	Kennecott
MW-070	494829	4237248	1788	230.1	360	-90	RC	1994	Kennecott
MW-071	491597	4238095	1821	207.3	360	-90	RC	1994	Kennecott
MW-072	494458	4235792	1782	225.6	360	-90	RC	1994	Kennecott
MW-073	494990	4231739	1762	103.6	360	-90	RC	1994	Kennecott
MW-074	493479	4236180	1777	243.8	360	-90	RC	1994	Kennecott
MW-075	491980	4236645	1798	225.6	360	-90	RC	1994	Kennecott
MW-076	489856	4237541	1847	219.5	360	-90	RC	1994	Kennecott
MW-077	494003	4234433	1771	167.6	225	-60	RC	1994	Kennecott
MW-078	493905	4234513	1770	137.2	225	-60	RC	1994	Kennecott
MW-079	494174	4234362	1772	198.1	225	-60	RC	1994	Kennecott
MW-080	494400	4234406	1773	161.5	225	-60	RC	1994	Kennecott
MW-081	494322	4234082	1770	167.6	225	-60	RC	1994	Kennecott
MW-082	494029	4233701	1769	149.4	220	-60	RC	1994	Kennecott
MW-083	494389	4233778	1770	166.1	225	-60	RC	1994	Kennecott
MW-084	494626	4233840	1770	97.5	225	-60	RC	1994	Kennecott
MW-085	494620	4233523	1768	182.9	225	-60	RC	1994	Kennecott
MW-086	493933	4234664	1772	237.7	225	-60	RC	1994	Kennecott
MWRC-1	493889	4232098	1759	117.3	360	-90	RC	1994	Bob Warren
MWRC-2	493737	4232098	1759	97.5	360	-90	RC	1994	Bob Warren
MWRC-3	494041	4232098	1759	146.3	360	-90	RC	1994	Bob Warren
MW-087D	493449	4234551	1771	125.8	360	-90	DDH	1995	Kennecott
MW-088	493402	4234624	1773	54.9	360	-90	RC	1995	Kennecott
MW-089	493942	4234465	1771	139.4	360	-90	RC	1995	Kennecott
MW-089D	493942	4234465	1771	139.4	360	-90	DDH	1995	Kennecott
MW-090D	494005	4234429	1772	167.2	225	-60	DDH	1995	Kennecott
MW-091	494053	4234394	1771	173.7	225	-60	RC	1995	Kennecott
MW-092	493990	4234356	1770	160.0	225	-60	RC	1995	Kennecott
MW-093	493961	4234415	1770	167.6	225	-60	RC	1995	Kennecott
MW-094	494051	4234481	1772	93.0	225	-60	RC	1995	Kennecott
MW-095	493953	4234532	1771	205.7	225	-60	RC	1995	Kennecott
MW-096	493874	4234566	1770	205.7	225	-60	RC	1995	Kennecott
MW-097	494005	4234542	1771	205.7	225	-60	RC	1995	Kennecott
MW-098	494120	4234452	1771	141.7	225	-60	RC	1995	Kennecott
MW-099	494059	4234476	1771	86.9	225	-60	RC	1995	Kennecott
MW-100	494307	4234499	1773	202.7	225	-60	RC	1995	Kennecott

Drill hole Name	Easting	Northing	Elevation	Total Depth	Azimuth	Dip	Drill hole Type	Year	Campaign
MW-101	494170	4234524	1773	196.6	225	-60	RC	1995	Kennecott
MW-102	494046	4234581	1772	141.7	225	-60	RC	1995	Kennecott
MW-103	493806	4234471	1772	196.6	225	-60	RC	1995	Kennecott
MW-104	493889	4234393	1771	187.5	225	-60	RC	1995	Kennecott
MW-105	492403	4234912	1805	213.4	360	-90	RC	1995	Kennecott
MW-106	492459	4234731	1795	201.2	360	-90	RC	1995	Kennecott
MW-107	492586	4234851	1788	201.2	360	-90	RC	1995	Kennecott
MW-108	492776	4234772	1785	195.1	360	-90	RC	1995	Kennecott
MW-109	492661	4234629	1788	152.4	360	-90	RC	1995	Kennecott
MW-110	492548	4234543	1790	121.9	360	-90	RC	1995	Kennecott
MW-111	492842	4234476	1786	152.4	360	-90	RC	1995	Kennecott
MW-112	492784	4234273	1789	115.8	360	-90	RC	1995	Kennecott
MW-113	493064	4234235	1782	121.9	360	-90	RC	1995	Kennecott
MW-114	493408	4234479	1773	198.1	360	-90	RC	1995	Kennecott
MW-115	493355	4234477	1775	182.9	360	-90	RC	1995	Kennecott
MW-116	493421	4234407	1774	152.4	360	-90	RC	1995	Kennecott
MW-117	493481	4234235	1771	121.9	360	-90	RC	1995	Kennecott
MW-118	493293	4234350	1777	137.2	360	-90	RC	1995	Kennecott
MW-119	493120	4234472	1780	167.6	360	-90	RC	1995	Kennecott
MW-120	493288	4234571	1775	195.1	360	-90	RC	1995	Kennecott
MW-121	493172	4234677	1777	192.0	360	-90	RC	1995	Kennecott
MW-122	493039	4234650	1779	192.0	360	-90	RC	1995	Kennecott
MW-123	494252	4233976	1770	97.5	360	-90	RC	1995	Kennecott
MW-124	494118	4234318	1770	198.1	360	-90	RC	1995	Kennecott
MW-125	494129	4234218	1770	152.4	360	-90	RC	1995	Kennecott
MW-126	493991	4234176	1771	106.7	360	-90	RC	1995	Kennecott
MW-127	493901	4233979	1771	106.7	360	-90	RC	1995	Kennecott
MW-128	494417	4234243	1771	189.0	360	-90	RC	1996	Kennecott
MW-129	493072	4234772	1779	231.6	360	-90	RC	1996	Kennecott
MW-130	492933	4234734	1782	189.0	360	-90	RC	1996	Kennecott
MW-131	492880	4234882	1782	201.2	360	-90	RC	1996	Kennecott
MW-132	492851	4235086	1779	243.8	360	-90	RC	1996	Kennecott
MW-133	493002	4234955	1779	237.7	360	-90	RC	1996	Kennecott
MW-134	494253	4233655	1769	170.7	360	-90	RC	1996	Kennecott
MW-135	494363	4233338	1768	167.6	360	-90	RC	1996	Kennecott
MW-136	494431	4233943	1769	170.7	360	-90	RC	1996	Kennecott
TMW-001	491953	4233417	1826	121.9	110	-75	RC	1997	Tombstone
TMW-002	492066	4233152	1815	152.4	110	-60	RC	1997	Tombstone
TMW-003	492133	4232885	1807	182.9	110	-60	RC	1997	Tombstone
TMW-004	492214	4232591	1800	182.9	110	-60	RC	1997	Tombstone
TMW-005	493214	4234809	1774	85.3	135	-75	RC	1997	Tombstone

Drill hole Name	Easting	Northing	Elevation	Total Depth	Azimuth	Dip	Drill hole Type	Year	Campaign
TMW-006	493291	4234621	1777	182.9	135	-75	RC	1997	Tombstone
TMW-007	493566	4234827	1772	152.4	225	-75	RC	1997	Tombstone
TMW-008	493450	4234391	1774	103.6	90	-75	RC	1997	Tombstone
TMW-009	493369	4234333	1774	152.4	360	-90	RC	1997	Tombstone
TMW-010	493313	4234161	1774	152.4	360	-90	RC	1997	Tombstone
TMW-011	494023	4234144	1771	134.1	360	-90	RC	1997	Tombstone
TMW-012	493945	4234042	1769	91.4	360	-90	RC	1997	Tombstone
TMW-013	493115	4234900	1775	144.8	135	-75	RC	1997	Tombstone
TMW-014	494276	4234048	1770	140.2	285	-60	RC	1997	Tombstone
TMT-01	499477	4227397	1908	61.0	360	-90	RC	1998	Golconda
TMT-02	499489	4227379	1908	67.1	360	-90	RC	1998	Golconda
TMT-03	499458	4227386	1908	79.2	100	-60	RC	1998	Golconda
TMT-04	499484	4227359	1907	48.8	10	-60	RC	1998	Golconda
TMT-05	499669	4227236	1908	61.0	325	-60	RC	1998	Golconda
TMT-06	499663	4226954	1923	45.7	285	-60	RC	1998	Golconda
TMT-07	500189	4226877	1939	45.7	32	-60	RC	1998	Golconda
TMT-08	499127	4227959	1902	61.0	215	-60	RC	1998	Golconda
TMT-09	499101	4227901	1896	45.7	65	-60	RC	1998	Golconda
MW-201D	493465	4234545	1773	76.8	360	-90	RC	2002	Midway
MW-202	493189	4234700	1776	208.8	360	-90	RC	2002	Midway
MW-203D	493469	4234537	1772	69.2	360	-90	DDH	2002	Midway
MW-204D	493467	4234539	1772	75.3	230	-65	DDH	2002	Midway
MW-205	493156	4234652	1778	164.6	360	-90	RC	2002	Midway
MW-206D	493473	4234531	1772	60.0	360	-90	DDH	2002	Midway
MW-207	493147	4234693	1777	190.5	360	-90	RC	2002	Midway
MW-208D	493476	4234526	1772	68.6	235	-75	DDH	2002	Midway
MW-209D	493451	4234537	1773	76.8	360	-90	DDH	2002	Midway
MW-210D	493459	4234550	1772	46.9	360	-90	DDH	2002	Midway
MW-211D	493455	4234556	1772	75.3	360	-90	DDH	2002	Midway
MW-212	493199	4234660	1776	176.8	360	-90	RC	2002	Midway
MW-213D	493445	4234568	1773	75.3	360	-90	DDH	2002	Midway
MW-214	493306	4234594	1774	189.0	360	-90	RC	2002	Midway
MW-215D	493448	4234561	1773	75.3	230	-75	DDH	2002	Midway
MW-216	493271	4234547	1776	173.7	360	-90	RC	2002	Midway
MW-217	493264	4234589	1775	192.0	360	-90	RC	2002	Midway
MW-218D	493439	4234574	1773	75.3	360	-90	DDH	2002	Midway
MW-219	493312	4234553	1775	195.1	360	-90	RC	2002	Midway
MW-220D	493436	4234580	1773	75.3	360	-90	DDH	2002	Midway
MW-221	493458	4234548	1772	76.2	360	-90	RC	2002	Midway
MW-222D	493431	4234586	1773	97.2	360	-90	DDH	2002	Midway
MW-224D	493426	4234592	1773	105.5	360	-90	DDH	2002	Midway

Drill hole Name	Easting	Northing	Elevation	Total Depth	Azimuth	Dip	Drill hole Type	Year	Campaign
MW-225D	493446	4234588	1773	81.1	360	-90	DDH	2002	Midway
MW-226D	493458	4234576	1772	84.4	360	-90	DDH	2002	Midway
MW-227	493400	4234574	1773	128.0	360	-90	RC	2002	Midway
MW-228D	493466	4234566	1772	84.1	360	-90	DDH	2002	Midway
MW-229	493471	4234391	1771	144.8	360	-90	RC	2002	Midway
MW-230	493418	4234353	1774	132.6	360	-90	RC	2002	Midway
MW-231D	493478	4234555	1772	63.2	360	-90	DDH	2002	Midway
MW-232	493381	4234403	1775	163.1	360	-90	RC	2002	Midway
MW-233	493433	4234436	1772	126.5	360	-90	RC	2002	Midway
MW-234D	493488	4234543	1772	78.9	360	-90	DDH	2002	Midway
MW-235	493347	4234451	1775	191.1	360	-90	RC	2002	Midway
MW-236D	493488	4234514	1774	102.7	360	-90	DDH	2002	Midway
MW-237	493362	4234536	1774	169.2	360	-90	RC	2002	Midway
MW-238	493320	4234489	1775	187.5	360	-90	RC	2002	Midway
MW-239D	493479	4234477	1774	63.1	360	-90	DDH	2002	Midway
MW-240D	493459	4234502	1774	105.8	360	-90	DDH	2002	Midway
MW-241	493324	4234619	1776	211.8	360	-90	RC	2002	Midway
MW-242D	493459	4234502	1774	102.7	60	-70	DDH	2002	Midway
MW-243	493293	4234605	1777	205.7	360	-90	RC	2002	Midway
MW-244D	493451	4234513	1775	105.8	360	-90	DDH	2002	Midway
MW-245D	493431	4234509	1774	34.7	360	-90	DDH	2002	Midway
MW-246D	493430	4234537	1774	105.8	360	-90	DDH	2002	Midway
MW-247D	493431	4234509	1774	106.4	50	-70	DDH	2002	Midway
MW-248D	493430	4234539	1774	106.1	60	-70	DDH	2002	Midway
MW-249D	493433	4234508	1774	97.8	360	-90	DDH	2002	Midway
MW-250D	493431	4234539	1775	97.8	50	-45	DDH	2002	Midway
MW-251D	493415	4234602	1775	106.4	360	-90	DDH	2002	Midway
MW-252D	493421	4234550	1775	72.5	360	-90	DDH	2002	Midway
MW-253D	493421	4234550	1775	106.1	60	-70	DDH	2002	Midway
MW-254D	493413	4234603	1775	176.5	230	-60	DDH	2002	Midway
MW-255D	493412	4234559	1775	90.8	360	-90	DDH	2002	Midway
MW-256D	493412	4234605	1775	61.6	330	-80	DDH	2002	Midway
MW-257D	493412	4234559	1775	160.6	240	-80	DDH	2002	Midway
MW-258D	493411	4234606	1775	105.5	320	-80	DDH	2002	Midway
MW-259D	493307	4234613	1777	198.7	360	-90	DDH	2002	Midway
MW-260D	493502	4234580	1774	104.2	250	-75	DDH	2002	Midway
MW-261D	493277	4234600	1777	192.9	360	-90	DDH	2002	Midway
MW-262D	493338	4234625	1776	207.3	360	-90	DDH	2002	Midway
MW-263D	493389	4234624	1775	143.6	240	-80	DDH	2002	Midway
MW-264D	493502	4234570	1774	82.9	250	-80	DDH	2002	Midway
MW-265D	493514	4234551	1774	85.6	240	-80	DDH	2002	Midway

Drill hole Name	Easting	Northing	Elevation	Total Depth	Azimuth	Dip	Drill hole Type	Year	Campaign
MW-266D	493520	4234547	1774	78.9	250	-80	DDH	2002	Midway
MW-267D	493529	4234536	1774	96.3	250	-80	DDH	2002	Midway
MW-268D	493517	4234599	1774	138.1	240	-75	DDH	2002	Midway
MW-269D	493482	4234594	1774	107.6	245	-80	DDH	2002	Midway
MW-270D	493476	4234609	1774	150.3	245	-80	DDH	2002	Midway
MW-271D	493781	4234455	1771	132.3	360	-90	DDH	2002	Newmont
MW-272D	493777	4234457	1773	112.5	245	-45	DDH	2002	Newmont
MW-273D	493783	4234461	1772	185.6	60	-70	DDH	2002	Newmont
MW-274D	493952	4234455	1773	115.5	150	-60	DDH	2002	Newmont
MW-275D	493962	4234427	1772	139.0	360	-90	DDH	2002	Newmont
MW-276D	493962	4234427	1772	168.6	240	-60	DDH	2002	Newmont
MW-277D	493962	4234427	1772	161.5	360	-90	DDH	2002	Newmont
MW-278D	493960	4234505	1772	222.2	200	-70	DDH	2002	Newmont
MW-279D	493953	4234457	1772	226.8	123	-70	DDH	2002	Newmont
MW-280D	493541	4234605	1772	246.3	240	-55	DDH	2002	Newmont
MW-281D	493497	4234635	1773	200.3	250	-70	DDH	2002	Newmont
MW-282D	493454	4234652	1773	135.2	240	-75	DDH	2002	Newmont
MW-283D	493380	4234504	1775	290.5	55	-45	DDH	2002	Newmont
MW-284D	493381	4234503	1775	180.9	60	-80	DDH	2002	Newmont
MW-285D	493442	4234445	1772	208.2	60	-45	DDH	2002	Newmont
MW-286D	493441	4234446	1772	158.2	150	-75	DDH	2002	Newmont
MW-287D	493780	4234645	1772	189.9	240	-50	DDH	2003	Newmont
MW-288D	493776	4234651	1772	238.7	150	-60	DDH	2003	Newmont
MW-289D	493867	4234486	1772	191.4	180	-60	DDH	2003	Newmont
MW-290	493767	4234087	1770	97.5	360	-90	RC	2003	Newmont
MW-291	493612	4234273	1772	152.4	225	-60	RC	2003	Newmont
MW-292	493972	4234094	1771	166.1	270	-45	RC	2003	Newmont
MW-293	494168	4234098	1770	61.0	270	-60	RC	2003	Newmont
MW-294	494159	4234097	1771	182.9	270	-60	RC	2003	Newmont
MW-295	494295	4234331	1772	213.4	180	-60	RC	2003	Newmont
MW-296	494098	4233790	1771	128.0	360	-90	RC	2003	Newmont
MW-297	494274	4233781	1769	176.8	225	-60	RC	2003	Newmont
MW-298	494260	4233889	1770	157.0	270	-60	RC	2003	Newmont
MW-299D	493616	4234463	1771	150.0	270	-50	DDH	2003	Newmont
MW-300	494313	4233975	1770	152.4	300	-60	RC	2003	Newmont
MW-301	494312	4233970	1770	152.4	235	-60	RC	2003	Newmont
MW-302	494608	4234093	1771	182.9	360	-90	RC	2003	Newmont
MW-303D	494082	4234461	1771	210.9	360	-90	DDH	2003	Newmont
MW-304	494448	4234078	1770	213.4	270	-60	RC	2003	Newmont
MW-305	494519	4234274	1771	207.3	270	-80	RC	2003	Newmont
MW-306	493559	4234471	1771	103.6	180	-75	RC	2003	Newmont

Drill hole Name	Easting	Northing	Elevation	Total Depth	Azimuth	Dip	Drill hole Type	Year	Campaign
MW-307D	493300	4234682	1774	269.4	240	-70	DDH	2003	Newmont
MW-308	493353	4234710	1774	243.8	240	-70	RC	2003	Newmont
MW-309	493353	4234712	1774	213.4	360	-90	RC	2003	Newmont
MW-310	493356	4234713	1774	213.4	60	-75	RC	2003	Newmont
MW-311D	493206	4234747	1775	216.3	150	-60	DDH	2003	Newmont
MW-312	493201	4234742	1775	243.8	240	-65	RC	2003	Newmont
MW-313	493398	4235015	1773	170.7	270	-70	RC	2003	Newmont
MW-314	492779	4234710	1785	164.6	270	-70	RC	2003	Newmont
MW-315	492597	4235014	1787	219.5	360	-90	RC	2003	Newmont
MW-316D	493483	4234518	1772	78.0	330	-52	DDH	2003	Newmont
MW-317	491935	4236270	1799	152.4	360	-90	RC	2003	Newmont
MW-318	491741	4236510	1803	158.5	360	-90	RC	2003	Newmont
MW-319	487323	4238316	1896	152.4	360	-90	RC	2003	Newmont
MW-320	487558	4238605	1877	182.9	360	-90	RC	2003	Newmont
MW-321	486988	4238519	1912	152.4	360	-90	RC	2003	Newmont
MW-322	486986	4238518	1911	152.4	190	-60	RC	2003	Newmont
MW-323	487013	4236108	1901	152.4	360	-90	RC	2003	Newmont
MW-324	486317	4237107	1927	152.4	360	-90	RC	2003	Newmont
MW-325	486325	4237631	1932	152.4	360	-90	RC	2003	Newmont
MW-326	497703	4239365	1799	152.4	360	-90	RC	2003	Newmont
MW-327	498355	4240078	1806	170.7	360	-90	RC	2003	Newmont
MW-328	494962	4231903	1766	152.4	360	-90	RC	2003	Newmont
MW-329	495731	4231743	1778	105.2	360	-90	RC	2003	Newmont
MW-330	496016	4231224	1785	182.9	360	-90	RC	2003	Newmont
MW-331	496345	4231472	1792	152.4	340	-60	RC	2003	Newmont
MW-332	496415	4231660	1795	91.4	270	-70	RC	2003	Newmont
MW-333	496526	4231790	1799	105.2	270	-70	RC	2003	Newmont
MW-334	496637	4231661	1801	91.4	270	-70	RC	2003	Newmont
MW-335	496545	4231538	1799	91.4	270	-70	RC	2003	Newmont
MW-336	495278	4231831	1768	91.4	360	-90	RC	2003	Newmont
MW-337	495269	4230011	1768	152.4	270	-70	RC	2003	Newmont
MW-338	495924	4229855	1786	152.4	270	-60	RC	2003	Newmont
MW-339	496621	4229271	1810	91.4	90	-60	RC	2003	Newmont
MW-340	497201	4228902	1826	91.4	270	-60	RC	2003	Newmont
MW-341	497445	4229333	1837	74.7	360	-90	RC	2003	Newmont
MW-342	497045	4229913	1819	91.4	270	-70	RC	2003	Newmont
MW-343	497999	4230872	1847	152.4	270	-70	RC	2003	Newmont
MW-344	496217	4227454	1804	152.4	270	-45	RC	2003	Newmont
MW-345	496017	4227452	1800	121.9	270	-45	RC	2003	Newmont
MW-346	496416	4227452	1811	152.4	270	-45	RC	2003	Newmont
MW-347	496613	4227452	1816	61.0	270	-45	RC	2003	Newmont

Drill hole Name	Easting	Northing	Elevation	Total Depth	Azimuth	Dip	Drill hole Type	Year	Campaign
MW-348	496816	4227453	1822	61.0	270	-45	RC	2003	Newmont
MW-349	497215	4227300	1833	121.9	90	-60	RC	2003	Newmont
MW-350	497684	4226994	1846	182.9	45	-45	RC	2003	Newmont
MW-351	498756	4228625	1888	172.2	60	-45	RC	2003	Newmont
MW-352	498679	4228727	1894	152.4	60	-60	RC	2003	Newmont
MW-353	498588	4229057	1880	152.4	225	-50	RC	2003	Newmont
MW-354	498265	4229278	1867	196.6	220	-60	RC	2003	Newmont
MW-355	498441	4228913	1881	121.9	45	-45	RC	2003	Newmont
MW-356	500069	4227804	1947	152.4	270	-60	RC	2003	Newmont
MW-357	499509	4227162	1929	152.4	90	-45	RC	2003	Newmont
MW-358	499348	4227161	1917	152.4	270	-60	RC	2003	Newmont
MW-359	500168	4226781	1954	202.7	360	-50	RC	2003	Newmont
MW-360	499768	4225871	1956	152.4	90	-60	RC	2003	Newmont
MW-361	499561	4225741	1947	182.9	90	-60	RC	2003	Newmont
MW-362	499044	4225811	1910	152.4	90	-60	RC	2003	Newmont
MW-363	498787	4226364	1884	152.4	90	-60	RC	2003	Newmont
MW-364	498631	4227025	1875	152.4	230	-60	RC	2003	Newmont
MW-365	498360	4226874	1866	152.4	45	-60	RC	2003	Newmont
MW-366	497112	4226930	1840	121.9	270	-60	RC	2003	Newmont
MW-367	495728	4231744	1779	182.9	270	-45	RC	2003	Newmont
MW-368	493973	4234092	1770	207.3	245	-45	RC	2003	Newmont
MW-369	493971	4234096	1770	213.4	305	-45	RC	2003	Newmont
MW-370D	493605	4234524	1772	180.1	221.55	-52.47	DDH	2004	Newmont
MW-371D	493664	4234444	1772	169.2	230	-60	DDH	2004	Newmont
MW-372D	493610	4234530	1772	251.2	220.17	-59.27	DDH	2004	Newmont
MW-373D	493986	4234150	1771	193.5	270	-60	DDH	2004	Newmont
MW-374D	493895	4234148	1770	194.5	225.43	-60.58	DDH	2004	Newmont
MW-375D	493690	4234471	1772	223.4	230	-60	DDH	2004	Newmont
MW-376D	493926	4234179	1771	172.0	225	-60	DDH	2004	Newmont
MW-377D	493982	4234204	1771	245.7	219.12	-55.76	DDH	2004	Newmont
MW-378	493546	4234468	1772	91.4	225	-70	RC	2004	Newmont
MW-378A	493549	4234474	1772	22.9	225	-65	RC	2004	Newmont
MW-379	493637	4234415	1772	182.9	224.46	-59.18	RC	2004	Newmont
MW-380D	493237	4234515	1778	314.9	44.43	-58.27	DDH	2004	Newmont
MW-381D	493201	4234480	1778	365.2	41.15	-55.97	DDH	2004	Newmont
MW-382D	493194	4234542	1778	348.1	46.81	-56.18	DDH	2004	Newmont
MW-383D	493118	4234551	1780	349.9	44.74	-50.96	DDH	2004	Newmont
MW-384D	493121	4234550	1780	375.2	39.44	-55.84	DDH	2004	Newmont
MW-385	493442	4234383	1774	121.9	45	-60	RC	2004	Newmont
MW-386D	493407	4234407	1774	137.8	45	-60	DDH	2004	Newmont
MW-387D	493418	4234351	1774	152.0	45	-60	DDH	2004	Newmont

Drill hole Name	Easting	Northing	Elevation	Total Depth	Azimuth	Dip	Drill hole Type	Year	Campaign
MW-388	493466	4234320	1773	121.9	41.66	-60.37	RC	2004	Newmont
MW-389	493886	4234242	1771	182.9	225.29	-55.01	RC	2004	Newmont
MW-390	493854	4234208	1771	182.9	226.49	-56.5	RC	2004	Newmont
MW-391	494442	4234078	1771	310.9	268.81	-45.01	RC	2004	Newmont
MW-392	492022	4235194	1814	169.2	270	-60	RC	2005	Midway
MW-393	492266	4235205	1800	201.2	270	-60	RC	2005	Midway
MW-394	493275	4234558	1776	182.9	90	-70	RC	2005	Midway
MW-395	493440	4234608	1773	24.4	270	-75	RC	2005	Midway
MW-396	493494	4234608	1772	10.7	270	-70	RC	2005	Midway
MW-397	493417	4234602	1775	54.9	45	-75	RC	2005	Midway
MW-398	493494	4234608	1774	137.2	265	-60	RC	2005	Midway
MW-399	493657	4234496	1773	121.9	240	-70	RC	2005	Midway
MW-400	493657	4234496	1773	121.9	240	-55	RC	2005	Midway
MW-401	493706	4234520	1773	152.4	240	-50	RC	2005	Midway
MW-402	493527	4234616	1774	152.4	260	-57	RC	2005	Midway
MW-403	493527	4234616	1774	137.2	236.7	-53.9	RC	2005	Midway
MW-404	493533	4234617	1774	128.0	85	-70	RC	2005	Midway
MW-405	493948	4234479	1770	24.4	225	-50	RC	2005	Midway
MW-406	492321	4234481	1798	121.9	225	-50	RC	2005	Midway
MW-407	492369	4234480	1799	171.3	235	-60	RC	2005	Midway
MW-408	493438	4234615	1773	182.9	65	-65	RC	2005	Midway
MW-409	493616	4234460	1771	140.2	57	-75	RC	2005	Midway
MW-410	493616	4234460	1771	121.9	57	-60	RC	2005	Midway
MW-411	493633	4234469	1772	121.9	55	-50	RC	2005	Midway
MW-412	493616	4234458	1771	140.2	93	-75	RC	2005	Midway
MW-413	493616	4234458	1771	121.9	93	-60	RC	2005	Midway
REW	493350	4234436	1774	42.7	360	-90	HD	2005	Midway
MW06-01	493677	4234487	1772	152.4	240	-65	RC	2006	Midway
MW06-02	493657	4234513	1772	152.4	240	-65	RC	2006	Midway
MW06-03	493657	4234513	1772	134.1	240	-58	RC	2006	Midway
MW06-04	493941	4234469	1771	207.3	225	-65	RC	2006	Midway
MW06-05	493941	4234468	1771	164.6	215	-50	RC	2006	Midway
MW06-06	493546	4234472	1772	121.9	230	-50	RC	2006	Midway
MW06-07	493438	4234435	1772	182.9	250	-50	RC	2006	Midway
MW06-08	493349	4234577	1775	213.4	235	-65	RC	2006	Midway
MW06-09	493606	4234521	1772	152.4	95	-55	RC	2006	Midway
MW06-10	493654	4234401	1771	182.9	60	-65	RC	2006	Midway
MW06-11	493606	4234521	1772	152.4	60	-75	RC	2006	Midway
MW06-12	493606	4234521	1772	152.4	60	-55	RC	2006	Midway
MW06-13	493616	4234461	1772	140.2	240	-65	RC	2006	Midway
MW06-14	493616	4234461	1772	167.6	240	-50	RC	2006	Midway

Drill hole Name	Easting	Northing	Elevation	Total Depth	Azimuth	Dip	Drill hole Type	Year	Campaign
MW06-15	493634	4234417	1771	163.1	50	-75	RC	2006	Midway
MW06-16	493634	4234417	1771	182.9	50	-55	RC	2006	Midway
MW06-17	493787	4234502	1772	158.5	240	-70	RC	2006	Midway
MW06-18	493787	4234502	1772	121.9	240	-50	RC	2006	Midway
MW06-19	493549	4234476	1772	187.5	90	-60	RC	2006	Midway
MW06-20	493549	4234476	1772	213.4	60	-50	RC	2006	Midway
MW06-21	493675	4234511	1770	182.9	90	-55	RC	2006	Midway
MW06-22	493526	4234535	1772	182.9	71.5	-70.8	RC	2006	Midway
MW06-23	493530	4234537	1771	213.4	73.9	-51.5	RC	2006	Midway
MW06-24	493697	4234470	1772	121.9	310	-65	RC	2006	Midway
MW06-25	493697	4234470	1772	121.9	310	-52	RC	2006	Midway
MW06-26	493524	4234495	1772	121.9	230	-70	RC	2006	Midway
MW06-27	493520	4234493	1772	121.9	230	-50	RC	2006	Midway
MW06-28D	493616	4234461	1772	140.2	40	-80	DDH	2006	Midway
MW06-29D	493616	4234461	1772	137.2	45	-65	DDH	2006	Midway
MW06-30D	493616	4234461	1772	64.0	40	-60	DDH	2006	Midway
MW06-31	494276	4234007	1770	137.2	190	-70	RC	2006	Midway
MW06-32	494276	4234007	1770	121.9	190	-50	RC	2006	Midway
MW06-33	493901	4234144	1770	115.8	90	-70	RC	2006	Midway
MW06-34	493901	4234144	1770	121.9	165	-70	RC	2006	Midway
MW06-35	493705	4234520	1772	213.4	210	-50	RC	2006	Midway
MW06-36	493705	4234520	1772	126.5	270	-50	RC	2006	Midway
MW06-37	493860	4234161	1771	152.4	135	-70	RC	2006	Midway
MW06-38	493860	4234161	1771	134.1	135	-55	RC	2006	Midway
MW06-39D	493616	4234461	1772	137.2	45	-75	DDH	2006	Midway
MW06-40	493787	4234502	1772	121.9	270	-65	RC	2006	Midway
MW06-41	493606	4234521	1772	137.2	40	-60	RC	2006	Midway
MW06-42	493610	4234531	1771	121.9	40	-50	RC	2006	Midway
MW06-43D	493398	4234555	1773	152.1	53	-60	DDH	2006	Midway
MW06-45HD	493419	4234562	1773	100.6	360	-90	HD	2006	Midway
MW06-45HM	493420	4234584	1773	47.2	360	-90	HD	2006	Midway
MW06-45HS	493413	4234597	1773	10.4	360	-90	HD	2006	Midway
MW06-46HD	493512	4234521	1772	91.4	360	-90	HD	2006	Midway
MW06-46HM	493516	4234536	1772	54.3	360	-90	HD	2006	Midway
MW06-46HS	493512	4234548	1772	13.4	360	-90	HD	2006	Midway

Drill hole Name	Easting	Northing	Elevation	Total Depth	Azimuth	Dip	Drill hole Type	Year	Campaign
MW06-47HD	493672	4234446	1772	67.1	360	-90	HD	2006	Midway
MW06-47HM	493667	4234442	1772	45.1	360	-90	HD	2006	Midway
MW06-47HS	493657	4234432	1771	19.2	360	-90	HD	2006	Midway
MW06-48HD	493970	4234271	1771	73.2	360	-90	HD	2006	Midway
MW06-48HM	493977	4234272	1771	57.9	360	-90	HD	2006	Midway
MW06-48HS	493991	4234274	1770	19.2	360	-90	HD	2006	Midway
MW06-49HD	493512	4234424	1771	48.8	360	-90	HD	2006	Midway
MW06-49HS	493515	4234414	1772	10.4	360	-90	HD	2006	Midway
MW06-50	493352	4234709	1774	190.5	90	-65	RC	2006	Midway
MW06-51	493602	4234527	1771	115.8	260	-55	RC	2006	Midway
MW06-52	493604	4234515	1772	109.7	280	-60	RC	2006	Midway
MW06-53	493328	4234808	1774	242.3	90	-65	RC	2006	Midway
MW06-54	493775	4234293	1774	152.4	270	-60	RC	2006	Midway
MW06-55	493328	4234753	1774	213.4	90	-65	RC	2006	Midway
MW07-57HD	493581	4234589	1772	140.2	360	-90	HD	2007	Midway
MW07-58	492540	4233917	1799	134.1	270	-45	RC	2007	Midway
MW07-59	492474	4233962	1801	134.1	270	-45	RC	2007	Midway
MW07-60	492546	4233918	1799	138.7	90	-45	RC	2007	Midway
MW07-61	493488	4234552	1773	140.2	92.1	-56.7	RC	2007	Midway
MW07-62	493468	4234486	1772	131.1	89.9	-60.97	RC	2007	Midway
MW07-63	493445	4234488	1772	111.3	95.1	-59.28	RC	2007	Midway
MW07-64	493510	4234489	1772	121.9	87.6	-59.78	RC	2007	Midway
MW07-65	493445	4234604	1773	134.1	90	-58.99	RC	2007	Midway
MW07-66	492526	4233915	1799	91.4	100	-60	RC	2007	Midway
MW07-67	492607	4233906	1797	146.3	90	-45	RC	2007	Midway
MW07-68	492731	4234033	1794	152.4	270	-45	RC	2007	Midway
MW07-69D	493414	4234553	1773	140.2	88.8	-44.01	DDH	2007	Midway
MW07-70D	493436	4234551	1773	108.5	102.4	-43.12	DDH	2007	Midway
MW07-71D	493463	4234552	1772	118.3	88.4	-59.44	DDH	2007	Midway
MW07-72D	493382	4234631	1773	175.0	104.3	-44.27	DDH	2007	Midway
MW07-73D	493207	4234251	1778	106.7	245.69	-44.28	DDH	2007	Midway
MW07-74D	493208	4234251	1778	86.3	189.8	-55.51	DDH	2007	Midway
MW07-75D	493214	4234257	1778	91.4	90	-55	DDH	2007	Midway

Drill hole Name	Easting	Northing	Elevation	Total Depth	Azimuth	Dip	Drill hole Type	Year	Campaign
MW08-76A	495409	4234228	1777	18.3	360	-90	HD	2008	Midway
MW08-77A	495235	4237477	1792	30.5	360	-90	HD	2008	Midway
MW08-78A	493605	4232886	1765	15.2	360	-90	HD	2008	Midway
MW08-79A	493534	4231602	1759	25.9	360	-90	HD	2008	Midway
MW08-80H	492717	4234012	1793	61.9	360	-90	HD	2008	Midway
MW08-81H	492483	4234077	1797	121.9	360	-90	HD	2008	Midway
MW08-82H	492999	4234285	1783	61.9	360	-90	HD	2008	Midway
MW08-83D	493481	4234441	1771	29.0	180	-70	DDH	2008	Midway
MW08-84D	493473	4234360	1771	92.1	360	-80	DDH	2008	Midway
MW08-85D	493158	4234058	1785	106.7	60	-45	DDH	2008	Midway
MW08-86D	493160	4234056	1785	47.2	20	-50	DDH	2008	Midway
MW08-87D	493298	4234083	1774	106.7	230	-45	DDH	2008	Midway
MW08-88D	493330	4234145	1777	74.7	215	-45	DDH	2008	Midway
MW08-89D	493315	4234122	1777	64.0	230	-45	DDH	2008	Midway
MW08-90D	493288	4234097	1774	106.7	270	-45	DDH	2008	Midway
MW08-91D	493616	4234426	1766	88.4	215	-45	DDH	2008	Midway
MW11-01C	493385	4234635	1774	180.4	103.9	-65.48	DDH	2011	Midway
MW11-02C	493387	4234636	1774	173.7	104.5	-55.22	DDH	2011	Midway
MW11-03C	493441	4234608	1773	129.5	70.1	-58.92	DDH	2011	Midway
MW11-04C	493419	4234564	1774	160.3	87.3	-55.41	DDH	2011	Midway
MW11-05C	493422	4234575	1774	160.0	85.4	-59.71	DDH	2011	Midway
MW11-06C	493419	4234579	1774	168.6	77.8	-45.92	DDH	2011	Midway
MW11-07C	493418	4234578	1774	146.0	77.2	-59.21	DDH	2011	Midway
MW11-08C	493392	4234589	1774	138.4	80.8	-69.4	DDH	2011	Midway
MW11-09C	493419	4234571	1774	103.0	77.9	-79.8	DDH	2011	Midway
MW11-10C	493441	4234555	1774	107.6	100.6	-45.81	DDH	2011	Midway
MW11-11C	493440	4234555	1774	94.2	105.5	-59.62	DDH	2011	Midway
MW11-12C	493287	4234634	1776	260.3	75.8	-74.81	DDH	2011	Midway

Drill hole Name	Easting	Northing	Elevation	Total Depth	Azimuth	Dip	Drill hole Type	Year	Campaign
MW11-13C	493289	4234634	1776	190.5	76.5	-59.5	DDH	2011	Midway
MW11-14C	493287	4234633	1776	239.3	89.4	-75.92	DDH	2011	Midway
MW11-15C	493310	4234583	1775	211.8	91.4	-69.06	DDH	2011	Midway
MW11-16C	493615	4234490	1772	104.9	89.8	-44.67	DDH	2011	Midway
MW11-17C	493613	4234490	1772	120.4	87.9	-61.13	DDH	2011	Midway
MW11-18C	493612	4234490	1772	123.4	89.7	-70.94	DDH	2011	Midway
MW11-19C	493612	4234487	1772	147.8	108.4	-74.9	DDH	2011	Midway
MW11-20C	493613	4234528	1772	50.3	0	-90	DDH	2011	Midway
MW11-21C	493617	4234527	1772	105.2	114.4	-64.83	DDH	2011	Midway
MW11-22C	493891	4234388	1772	153.0	82.5	-49.37	DDH	2011	Midway
MW11-23C	493890	4234388	1772	134.1	82.5	-56.71	DDH	2011	Midway
MW11-24C	493887	4234386	1772	157.0	82.3	-77.67	DDH	2011	Midway
MW11-25C	493890	4234387	1772	164.6	104.4	-48.99	DDH	2011	Midway
MW11-26C	493376	4234536	1774	245.4	65.2	-84.5	DDH	2011	Midway
TG18001	493600	4234586	1771	106.7	340	-60	Core	2018	Viva
TG18002	493376	4234640	1774	157.3		-60	Core	2018	Viva
TG18003	493871	4234674	1770	171.8	200	-60	Core	2018	Viva
TG18004	493346	4234794	1774	140.5	160	-60	Core	2018	Viva
TG1805	493510	4234380	1774	80.8	250	-60	RC	2018	Viva
TG1806	493469	4234497	1774	76.2	240	-70	RC	2018	Viva
TG1807	493472	4234575	1774	94.5	200.5	-80.23	RC	2018	Viva
TG1808	493664	4234463	1774	125.0	233.5	-70.68	RC	2018	Viva
TG1809	493531	4234581	1774	100.6	220	-60	RC	2018	Viva
TG1810	493362	4234585	1774	170.7	90	-65	RC	2018	Viva
TG1811	494000	4234489	1774	149.4	206.71	-70.13	RC	2018	Viva
TG1812	494032	4234471	1774	185.9	199.3	-76.27	RC	2018	Viva
TG1813	494097	4234461	1774	164.6	208.4	-60.88	RC	2018	Viva
TG1814	493796	4234492	1773	115.8	219.61	-68.45	RC	2018	Viva
TG1815	493771	4234522	1774	112.8	198.4	-70.91	RC	2018	Viva
TG1816	493366	4234593	1774	170.7	105.5	-59.17	RC	2018	Viva
TG1817	493273	4234643	1774	228.6	54	-80.71	RC	2018	Viva
TG1818	493474	4234497	1774	109.7	99.8	-64.61	RC	2018	Viva
TG1819	493863	4234497	1774	129.5	200.1	-70.57	RC	2018	Viva
TG1820	493828	4234490	1774	118.9	200.7	-61.04	RC	2018	Viva

Drill hole Name	Easting	Northing	Elevation	Total Depth	Azimuth	Dip	Drill hole Type	Year	Campaign
TG1901	493804	4234500	1774	65.5	50	-70	RC	2019	Viva
TG1902	493475	4234578	1774	146.3	0	-69.85	RC	2019	Viva
TG1903	493660	4234476	1774	140.2	275.8	-76.25	RC	2019	Viva
TG1904	494131	4234118	1773	134.1	274	-60.96	RC	2019	Viva
TG1905	494099	4234406	1774	146.3	211.2	-70.92	RC	2019	Viva
TG1906	493774	4234521	1774	134.1	0	-90	RC	2019	Viva
TG1907	491791	4235166	1811	123.444	180	-60	RC	2019	Viva
TG1908	491852	4235193	1806	121.92	180	-60	RC	2019	Viva
TG1909	491970	4235027	1807	147.828	270	-60	RC	2019	Viva
TG1910	492593	4234657	1793	106.68	0	-90	RC	2019	Viva
TG1911	492237	4234908	1812	146.304	270	-50	RC	2019	Viva
TG1912	493474	4234688	1774	121.92	0	-90	RC	2019	Viva
TG1913	494162	4234274	1774	152.4	230	-70	RC	2019	Viva
TG1914	492593	4234657	1793	121.92	60	-60	RC	2019	Viva
TG1915	493269	4234624	1774	176.784	219	-75	RC	2019	Viva
TG1916	493232	4234676	1777	182.88	220	-65	RC	2019	Viva

Appendix E – List of Claims

BLM SERIAL #	CLAIM NAME	DOC	In Faith Royalty
NMC1059873	MW 1	776757	N
NMC1059874	MW 2	776758	N
NMC1059875	MW 3	776759	N
NMC1059876	MW 4	776760	N
NMC1059877	MW 5	776761	N
NMC1059878	MW 6	776762	N
NMC1059879	MW 7	776763	N
NMC1059880	MW 8	776764	N
NMC1059881	MW 9	776765	N
NMC1059882	MW 10	776766	N
NMC1059883	MW 11	776767	N
NMC1059884	MW 12	776768	N
NMC1059885	MW 13	776769	N
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NMC1059887	MW 15	776771	N
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NMC1059890	MW 18	776774	N
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NMC1059893	MW 21	776777	N
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NMC1059896	MW 24	776780	N
NMC1059897	MW 25	776781	N
NMC1059898	MW 26	776782	N
NMC1059899	MW 27	776783	N
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NMC1059906	MW 34	776790	N
NMC1059907	MW 35	776791	N
NMC1059908	MW 36	776792	N
NMC1059909	MW 37	776793	N
NMC1059910	MW 38	776794	N
NMC1059911	MW 39	776795	N
NMC1059912	MW 40	776796	N

BLM SERIAL #	CLAIM NAME	DOC	In Faith Royalty
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NMC1059914	MW 42	776798	N
NMC1059915	MW 43	776799	N
NMC1059916	MW 44	776800	N
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NMC1059949	MW 77	776833	N
NMC1059950	MW 78	776834	N
NMC1059951	MW 79	776835	N
NMC1059952	MW 80	776836	N
NMC1059953	MW 81	776837	N

BLM SERIAL #	CLAIM NAME	DOC	In Faith Royalty
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NMC1059955	MW 83	776839	N
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NMC1059990	MW 118	776874	N
NMC1059991	MW 119	776875	N
NMC1059992	MW 120	776876	N
NMC1059993	MW 121	776877	N
NMC1059994	MW 122	776878	N

BLM SERIAL #	CLAIM NAME	DOC	In Faith Royalty
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NMC1059996	MW 124	776880	N
NMC1059997	MW 125	776881	N
NMC1059998	MW 126	776882	N
NMC1059999	MW 127	776883	N
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NMC1060003	MW 131	776887	N
NMC1060004	MW 132	776888	N
NMC1060005	MW 133	776889	N
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NMC1060031	MW 159	776915	N
NMC1060032	MW 160	776916	N
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BLM SERIAL #	CLAIM NAME	DOC	In Faith Royalty
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NMC1060037	MW 165	776921	N
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NMC1060039	MW 167	776923	N
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NMC1060072	MW 200	776956	N
NMC1060073	MW 201	776957	N
NMC1060074	MW 202	776958	N
NMC1060075	MW 203	776959	N
NMC1060076	MW 204	776960	N

BLM SERIAL #	CLAIM NAME	DOC	In Faith Royalty
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NMC1060078	MW 206	776962	N
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NMC835226	MWAY 118	546957	N
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NMC835255	MWAY 147	546986	N
NMC835256	MWAY 148	546987	N
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NMC835258	MWAY 150	546989	N
NMC835504	MWAY 396	547235	N
NMC845408	MWAY 649	559669	N
NMC845410	MWAY 651	559671	N
NMC845412	MWAY 653	559673	N
NMC845414	MWAY 655	559675	N

BLM SERIAL #	CLAIM NAME	DOC	In Faith Royalty
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NMC835176	MWAY 68	546907	N
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NMC830761	RD 20	539690	Y
NMC830764	RD 24	539693	Y
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NMC831841	RD 27	543107	Y
NMC831843	RD 29	543109	Y
NMC831864	RD 50	543130	Y
NMC831866	RD 52	543132	Y
NMC831868	RD 54	543134	Y
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NMC831895	RD 81	543161	Y
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NMC830753	RD 12	639682	Y
NMC984614	RD 86	706179	Y
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NMC984618	RD 90	706183	Y
NMC984619	RD 91	706184	Y
NMC984620	RD 92	706185	Y
NMC984621	RD 93	706186	Y
NMC984622	RD 94	706187	Y

BLM SERIAL #	CLAIM NAME	DOC	In Faith Royalty
NMC984623	RD 95	706188	Y
NMC984624	RD 96	706189	Y
NMC984625	RD 97	706190	Y
NMC984626	RD 98	706191	Y
NMC984627	RD 99	706192	Y
NMC984628	RD 100	706193	Y
NMC984631	RD 101	706194	Y
NMC984632	RD 102	706195	Y
NMC984629	RD 103	706196	Y
NMC984630	RD 104	706197	Y
NMC984633	RD 105	706198	Y
NMC984634	RD 106	706199	Y
NMC984613	RD 85	706200	Y
NMC688327	RV 29	343968	Y
NMC688329	RV 31	343970	Y
NMC688331	RV 33	343972	Y
NMC688333	RV 35	343974	Y
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NMC688339	RV 41	343980	Y
NMC387816	SP #1	172347	Y
NMC390503	SP #66	143145	Y
NMC387817	SP #2	172348	Y
NMC387818	SP #3	172349	Y
NMC387820	SP #5	172351	Y
NMC387822	SP #7	172353	Y
NMC387824	SP #9	172355	Y
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NMC387828	SP #13	172359	Y
NMC387830	SP #15	172361	Y
NMC387832	SP #17	172363	Y
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NMC387836	SP #21	172367	Y
NMC387837	SP #22	172368	Y
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NMC387840	SP #25	172371	Y
NMC387841	SP #26	172372	Y
NMC387842	SP #27	172373	Y
NMC387843	SP #28	172374	Y
NMC387844	SP #29	172375	Y

BLM SERIAL #	CLAIM NAME	DOC	In Faith Royalty
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NMC387846	SP #31	172377	Y
NMC387847	SP #32	172378	Y
NMC390502	SP #65	173144	Y
NMC390504	SP #67	173146	Y
NMC390505	SP #68	173147	Y
NMC390506	SP #69	173148	Y
NMC390507	SP #70	173149	Y
NMC470114	SP #71	206406	Y
NMC470115	SP #72	206407	Y
NMC470116	SP #73	206408	Y
NMC470117	SP #74	206409	Y
NMC470118	SP #75	206410	Y
NMC470119	SP #76	206411	Y
NMC470120	SP #77	206412	Y
NMC470121	SP #78	206413	Y
NMC470122	SP #79	206414	Y
NMC470123	SP #80	206415	Y
NMC470124	SP #81	206416	Y
NMC470125	SP #82	206417	Y
NMC470126	SP #83	206418	Y
NMC470127	SP #84	206419	Y
NMC470138	SP #95	206430	Y
NMC470139	SP #96	206431	Y
NMC470140	SP #97	206432	Y
NMC470141	SP #98	206433	Y
NMC470148	SP #105	206440	Y
NMC470149	SP #106	206441	Y
NMC470150	SP #107	206442	Y
NMC470151	SP #108	206443	Y
NMC470158	SP #115	206450	Y
NMC470159	SP #116	206451	Y
NMC470160	SP #117	206452	Y
NMC470161	SP #118	206453	Y
NMC470162	SP #119	206454	Y
NMC470166	SP #123	206458	Y
NMC470167	SP #124	206459	Y
NMC470168	SP #125	206460	Y
NMC470169	SP #126	206461	Y
NMC470170	SP #127	206462	Y
NMC502125	SP 281	212803	Y

BLM SERIAL #	CLAIM NAME	DOC	In Faith Royalty
NMC502126	SP 282	212804	Y
NMC502127	SP 283	212805	Y
NMC502128	SP 284	212806	Y
NMC502129	SP 285	212807	Y
NMC502130	SP 286	212808	Y
NMC502144	SP 300	212822	Y
NMC502145	SP 301	212823	Y
NMC502146	SP 302	212824	Y
NMC502164	SP 320	212842	Y
NMC502165	SP 321	212843	Y
NMC502166	SP 322	212844	Y
NMC513285	SP 340	217054	Y
NMC513286	SP 341	217055	Y
NMC513287	SP 342	217056	Y
NMC513288	SP 343	217057	Y
NMC513289	SP 344	217058	Y
NMC513290	SP 345	217059	Y
NMC513291	SP 346	217060	Y
NMC513292	SP 347	217061	Y
NMC513293	SP 348	217062	Y
NMC513294	SP 349	217063	Y
NMC513295	SP 350	217064	Y
NMC513296	SP 351	217065	Y
NMC513303	SP 358	217072	Y
NMC513304	SP 359	217073	Y
NMC513305	SP 360	217074	Y
NMC513309	SP 366	217078	Y
NMC513310	SP 367	217079	Y
NMC513311	SP 368	217080	Y
NMC513317	SP 374	217086	Y
NMC513318	SP 375	217087	Y
NMC513319	SP 376	217088	Y
NMC513325	SP 382	217094	Y
NMC679116	SP 4	333643	Y
NMC679117	SP 6	333644	Y
NMC679118	SP 8	333645	Y
NMC679119	SP 10	333646	Y
NMC679120	SP 12	333647	Y
NMC679121	SP 14	333648	Y
NMC679122	SP 16	333649	Y
NMC679123	SP 280	333650	Y

BLM SERIAL #	CLAIM NAME	DOC	In Faith Royalty
NMC679124	SP 352	333651	Y
NMC679125	SP 353	333652	Y
NMC679126	SP 354	333653	Y
NMC679127	SP 355	333654	Y
NMC679128	SP 356	333655	Y
NMC679129	SP 357	333656	Y
NMC838228	WAY 1	549276	N
NMC838229	WAY 2	249277	N
NMC838242	WAY 15	259290	N
NMC838230	WAY 3	549278	N
NMC838231	WAY 4	549279	N
NMC838232	WAY 5	549280	N
NMC838233	WAY 6	549281	N
NMC838234	WAY 7	549282	N
NMC838235	WAY 8	549283	N
NMC838236	WAY 9	549284	N
NMC838237	WAY 10	549285	N
NMC838238	WAY 11	549286	N
NMC838239	WAY 12	549287	N
NMC838240	WAY 13	549288	N
NMC838241	WAY 14	549289	N
NMC838243	WAY 16	549291	N
NMC838244	WAY 17	549292	N
NMC838245	WAY 18	549293	N
NMC838246	WAY 19	549294	N
NMC838247	WAY 20	549295	N
NMC838248	WAY 21	549296	N
NMC838249	WAY 22	549297	N
NMC838250	WAY 23	549298	N
NMC838251	WAY 24	549299	N
NMC838252	WAY 25	549300	N
NMC838253	WAY 26	549301	N
NMC838254	WAY 27	549302	N
NMC838255	WAY 28	549303	N
NMC838256	WAY 29	549304	N

Appendix F – Base Case Cash Flow Model

	Units	-2	-1	1	2	3	4	5	6	TOTAL
						Years				
Mine Production										
Total Ore	Tonnes			2,464,006	2,623,797	1,993,654	2,537,706	2,867,499		**12,486,662**
grade										
Total Waste	Tonnes			14,583,901	12,786,011	11,498,832	10,539,812	8,396,212		**57,804,768**
Total	Tonnes			17,047,907	15,409,808	13,492,486	13,077,518	11,263,712		**70,291,431**
Total Mined Recoverable Gold	g			1,545,714	1,502,629	1,504,695	1,138,609	1,340,082		**7,031,728**
Leach Pad Inventory	g			386,429	375,657	376,174	284,652	335,021		
Sold Metal	g			1,159,286	1,513,400	1,504,178	1,230,130	1,289,714	335,021	**7,031,728**
Total Project Income				**51,551,856**	**67,298,847**	**66,888,771**	**54,702,223**	**57,351,820**	**14,897,908**	**312,691,424**
Market Price Au	**oz**			1,400	1,400	1,400	1,400	1,400	1,400	
Payable Gold	**%**			99.9%	99.9%	99.9%	99.9%	99.9%	99.9%	
Shipping & Refining Charges	**$/oz**			1.5	1.5	1.5	1.5	1.5	1.5	
Realized Price	**$/oz**			1,399	1,399	1,399	1,399	1,399	1,399	
Realized Price	$/g			45.0	45.0	45.0	45.0	45.0	45.0	
Refiner Receipts	$			52,072,581	67,978,634	67,564,415	55,254,771	57,931,131	15,048,392	**315,849,924**
Royalty	1%			520,726	679,786	675,644	552,548	579,311	150,484	**3,158,499**
Net Income	$			51,551,856	67,298,847	66,888,771	54,702,223	57,351,820	14,897,908	**312,691,424**
Total Project Operating Costs	**$**			**35,520,511**	**34,774,835**	**31,683,470**	**33,289,819**	**33,480,596**	**1,618,093**	**170,367,323**
Contingency	10%			3,229,137	3,161,349	2,880,315	3,026,347	3,043,691	147,099	15,487,938
Mining Cost - Total	**$**			**19,745,373**	**18,530,413**	**17,829,533**	**17,487,623**	**16,568,584**		**90,161,526**
Load Haul - Total Cost	$			8,614,270	8,003,809	8,010,467	7,821,691	7,571,990		40,022,227
Drill / Blasting - Total Cost	$			7,223,804	6,619,305	5,911,766	5,758,633	5,089,294		30,602,803
Mine Support - Total Cost	$			1,301,970	1,301,970	1,301,970	1,301,970	1,301,970		6,509,848
Mine Maintenance - Total Cost	$			1,516,180	1,516,180	1,516,180	1,516,180	1,516,180		7,580,898
Mine G & A - Total Cost	$			1,089,150	1,089,150	1,089,150	1,089,150	1,089,150		5,445,750
Process Cost - Total Cost	**$**			**11,053,368**	**11,542,504**	**9,622,095**	**11,261,106**	**12,254,641**	**755,902**	**56,489,616**
Crushing Plant - Total Cost	$			5,693,836	5,916,425	5,038,635	5,796,500	6,255,902	-	28,701,297
Leach/Process Plant - Total Cost	$			4,806,212	5,059,975	4,067,767	4,905,390	5,413,139	475,402	24,727,886
Laboratory - Total Cost	$			553,320	566,104	515,692	559,216	585,600	280,500	3,060,433
Site G&A - Total				**1,492,632**	**1,540,570**	**1,351,527**	**1,514,742**	**1,613,680**	**715,092**	**8,228,243**
EBITDA				**16,031,345**	**32,524,013**	**35,205,301**	**21,412,404**	**23,871,224**	**13,279,814**	**142,324,101**

	Units	Years -2	-1	1	2	3	4	5	6	TOTAL
Total Project Capital Costs		**2,400,000**	**55,508,638**	**6,542,510**	**10,388,510**	**6,542,510**	**6,542,510**	**1,260,000**	**(16,575,078)**	**72,609,602**
Mining - Subtotal		**600,000**	**11,580,398**	**4,980,398**	**4,980,398**	**4,980,398**	**4,980,398**	**-**	**(6,293,774)**	**25,808,214**
Mining - Expenditures		**600,000**	**6,600,000**							**7,200,000**
Contingency	**20%**	**100,000**	**1,100,000**							**1,200,000**
Exploration			500,000							500,000
Dewatering & Infiltration System			1,500,000							1,500,000
1300m2 Fully Equipped Shop			1,500,000							1,500,000
Office / Warehouse			500,000							500,000
Road Relocation		500,000								500,000
Power Supply			800,000							800,000
Water Supply			200,000							200,000
Access Roads, Site Civil			500,000							500,000
Mining - Capitalized Leasing			**4,980,398**	**4,980,398**	**4,980,398**	**4,980,398**	**4,980,398**	**-**	**(6,293,774)**	**18,608,214**
777 Truck			2,756,262	2,756,262	2,756,262	2,756,262	2,756,262		(3,483,113)	10,298,196
992 Loader			722,160	722,160	722,160	722,160	722,160		(912,600)	2,698,199
Drill Rig (6" - 9")			464,904	464,904	464,904	464,904	464,904		(587,504)	1,737,018
D9			631,334	631,334	631,334	631,334	631,334		(797,823)	2,358,849
775 Water Truck			148,154	148,154	148,154	148,154	148,154		(187,223)	553,546
14M			72,877	72,877	72,877	72,877	72,877		(92,095)	272,290
Crane Truck			33,330	33,330	33,330	33,330	33,330		(42,120)	124,532
ANFO Loader			20,832	20,832	20,832	20,832	20,832		(26,325)	77,833
Backhoe - 1 m3			33,330	33,330	33,330	33,330	33,330		(42,120)	124,532
Fuel /Lube Truck Class 8			41,663	41,663	41,663	41,663	41,663		(52,650)	155,665
Light Trucks			55,551	55,551	55,551	55,551	55,551		(70,200)	207,554
Process - Subtotal			**30,548,113**	**302,113**	**4,148,113**	**302,113**	**302,113**		**(6,101,176)**	**29,501,388**
Process - Expenditures			**30,246,000**		**3,846,000**				**(5,719,393)**	**28,372,607**
Contingency	**20%**		**5,041,000**		**641,000**					**5,682,000**
										-
Crushing Plant			8,100,000						(2,383,609)	**5,716,391**
Leach Pad Construction			3,005,000		3,205,000					**6,210,000**
Process Plant			9,800,000						(2,148,563)	**7,651,437**
Conveyor and Stacking System			3,700,000						(980,569)	**2,719,431**
Laboratory			600,000						(206,652.2)	**393,348**
Process - Leased Equipment			**302,113**	**302,113**	**302,113**	**302,113**	**302,113**		**(381,783)**	**1,128,781**
966 Loader			141,627	141,627	141,627	141,627	141,627		(178,975)	
D6			160,486	160,486	160,486	160,486	160,486		(202,808)	

	Units	-2	-1	1	2	3	4	5	6	TOTAL
					Years					
Other - Expenditures		**1,800,000**	**13,380,128**	**1,260,000**	**1,260,000**	**1,260,000**	**1,260,000**	**1,260,000**	**(4,180,128)**	**17,300,000**
Contingency	20%	300,000	750,000	210,000	210,000	210,000	210,000	210,000	1,200,000	**3,300,000**
Environmental Bonding			2,500,000	50,000	50,000	50,000	50,000	50,000	(2,500,000)	**250,000**
Feasibility, Permitting		1,500,000								**1,500,000**
Royalty Purchase Option			1,000,000							**1,000,000**
First Fills			250,000							**250,000**
Sustaining Capital				1,000,000	1,000,000	1,000,000	1,000,000	1,000,000		**5,000,000**
Working Capital			8,880,128						(8,880,128)	
Mine Closure and Reclamation									6,000,000	**6,000,000**
Before Tax Revenue and Cashflow										
EBITDA				$ 16,031,345	$ 32,524,013	$ 35,205,301	$ 21,412,404	$ 23,871,224	$ 13,279,814	**$ 142,324,101**
EBITDA-Capital		$ (2,400,000)	$ (55,508,638)	$ 9,488,834	$ 22,135,502	$ 28,662,791	$ 14,869,894	$ 22,611,224	$ 29,854,892	**$ 69,714,499**
Cumulative cashflow		$ (2,400,000)	$ (57,908,638)	$ (48,419,804)	$ (26,284,302)	$ 2,378,489	$ 17,248,383	$ 39,859,607	$ 69,714,499	
Pre-tax NPV @ 5% per annum		**$ 43,604,590**	(End of year discounting)							
Pre-tax NPV @ 10% per annum		$ 25,912,888								
IRR		24.5%								
Federal and State Taxes				$ (302,276)	$ (1,723,012)	$ (2,959,857)	$ (1,319,192)	$ (1,840,646)	$ (1,519,161)	$ (9,664,143)
After Tax Cash Flow										
After Tax Cash Flow (ATCF)		$ (2,400,000)	$ (55,508,638)	$ 9,186,559	$ 20,412,490	$ 25,702,933	$ 13,550,702	$ 20,770,578	$ 28,335,732	**$ 60,050,356**
Cumulative ATCF		$ (2,400,000)	$ (57,908,638)	$ (48,722,080)	$ (28,309,589)	$ (2,606,656)	$ 10,944,046	$ 31,714,624	$ 60,050,356	
Post Tax NPV @ 5% per annum		**$ 36,286,079**	(End of year discounting)							
Post Tax NPV @ 10% per annum		$ 20,273,213								
IRR		21.6%								

Appendix G – Alternate Case Cash Flow Model

		Years											
	Units	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**TOTAL**
Mine Production													
Total Ore	Tonnes			2,626,256	2,664,998	2,547,521	2,620,913	2,274,986	2,638,806	2,714,261	2,800,180		**20,887,920**
Total Waste	Tonnes			22,605,983	19,671,103	17,578,194	15,358,017	15,320,368	17,602,514	12,483,596	2,033,587		**122,653,363**
Total	Tonnes			25,232,239	22,336,102	20,125,714	17,978,930	17,595,354	20,241,320	15,197,857	4,833,767		**143,541,283**
													-
Total Mined Recoverable Gold	g			1,626,716	1,691,825	1,241,085	1,509,396	1,013,417	862,095	1,101,618	1,745,453		**10,791,606**
Leach Pad Inventory	g			406,679	422,956	310,271	377,349	253,354	215,524	275,405	436,363		
Sold Metal	g			1,220,037	1,675,548	1,353,770	1,442,318	1,137,412	899,926	1,041,738	1,584,494	436,363	**10,791,606**
Total Project Income		**-**	**-**	**62,012,205**	**85,164,967**	**68,809,580**	**73,310,309**	**57,812,514**	**45,741,527**	**52,949,562**	**80,536,869**	**22,179,524**	**548,517,058**
Market Price Au	**oz**			1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600	
Payable Gold	**%**			99.9%	99.9%	99.9%	99.9%	99.9%	99.9%	99.9%	99.9%	99.9%	
Shipping & Refining Charges	**$/oz**			1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	
Realized Price	**$/oz**			1,599	1,599	1,599	1,599	1,599	1,599	1,599	1,599	1,599	
Realized Price	$/g			51.4	51.4	51.4	51.4	51.4	51.4	51.4	51.4	51.4	
Refiner Receipts	$	-	-	62,638,591	86,025,219	69,504,626	74,050,818	58,396,479	46,203,563	53,484,406	81,350,373	22,403,559	**554,057,634**
Royalty	1%	-	-	626,386	860,252	695,046	740,508	583,965	462,036	534,844	813,504	224,036	**5,540,576**
Net Income	$	-	-	62,012,205	85,164,967	68,809,580	73,310,309	57,812,514	45,741,527	52,949,562	80,536,869	22,179,524	**548,517,058**
Total Project Operating Costs	**$**	**-**	**-**	**43,692,446**	**42,429,662**	**41,035,806**	**40,339,279**	**38,780,219**	**42,086,975**	**37,442,555**	**24,374,224**	**785,824**	**311,622,892**
Contingency	10%	-	-	3,972,041	3,857,242	3,730,528	3,667,207	3,525,474	3,826,089	3,403,869	2,215,839	71,439	28,269,726
Mining Cost - Total	**$**	**-**	**-**	**26,674,198**	**25,395,665**	**24,524,376**	**23,643,867**	**23,392,176**	**25,172,390**	**20,695,935**	**9,053,584**	**-**	**178,552,190**
Load Haul - Total Cost	$	-	-	12,400,829	12,187,593	12,129,358	12,038,508	11,927,909	12,364,580	9,557,529	4,059,350	-	86,665,655
Drill / Blasting - Total Cost	$	-	-	10,212,821	9,147,525	8,334,470	7,544,811	7,403,719	8,747,262	7,077,858	2,618,647	-	61,087,113
Mine Support - Total Cost	$	-	-	1,280,992	1,280,992	1,280,992	1,280,992	1,280,992	1,280,992	1,280,992	753,050	-	9,719,996
Mine Maintenance - Total Cost	$	-	-	1,690,406	1,690,406	1,690,406	1,690,406	1,690,406	1,690,406	1,690,406	869,037	-	12,701,875
Mine G & A - Total Cost	$	-	-	1,089,150	1,089,150	1,089,150	1,089,150	1,089,150	1,089,150	1,089,150	753,500	-	8,377,550
Process Cost - Total Cost	**$**	**-**	**-**	**11,506,666**	**11,625,591**	**11,264,981**	**11,490,267**	**10,428,408**	**11,545,190**	**11,776,808**	**11,513,083**	**-**	**91,806,897**
Crushing Plant - Total Cost	$	-	-	5,839,423	5,893,740	5,729,036	5,831,932	5,346,942	5,857,018	5,962,806	5,555,803	-	46,016,701
Leach/Process Plant - Total Cost	$	-	-	5,048,418	5,109,151	4,924,993	5,040,043	4,497,767	5,068,092	5,186,376	5,321,062	375,402	40,571,304
Laboratory - Total Cost	$	-	-	618,826	622,700	610,952	618,291	583,699	620,081	627,626	636,218	280,500	5,218,892
Site G&A - Total		**-**	**-**	**1,539,541**	**1,551,164**	**1,515,921**	**1,537,939**	**1,434,160**	**1,543,306**	**1,565,943**	**1,591,719**	**714,386**	**12,994,079**
EBDITA		**-**	**-**	**18,319,759**	**42,735,304**	**27,773,774**	**32,971,030**	**19,032,296**	**3,654,552**	**15,507,007**	**56,162,645**	**21,393,699**	**236,894,166**

	Units	-2	-1	1	2	3	4	5	6	7	8	9	TOTAL
							Years						
Total Project Capital Costs		**2,400,000**	**64,014,074**	**8,732,962**	**8,732,962**	**8,732,962**	**15,194,962**	**1,284,000**	**1,284,000**	**1,284,000**	**1,284,000**	**(4,823,112)**	**108,120,811**
Mining - Subtotal		**600,000**	**14,586,849**	**7,146,849**	**7,146,849**	**7,146,849**	**7,146,849**	**-**	**-**	**-**	**-**		**43,774,247**
Mining - Expenditures		**600,000**	**7,440,000**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**8,040,000**
Contingency	**20%**	**100,000**	**1,240,000**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**1,340,000**
Exploration			500,000										**500,000**
Dewatering & Infiltration System			1,500,000										**1,500,000**
Fully Equipped Shop			1,950,000										**1,950,000**
Office / Warehouse			750,000										**750,000**
Road Relocation		500,000											**500,000**
Power Supply			800,000										**800,000**
Water Supply			200,000										**200,000**
Access Roads, Site Civil			500,000										**500,000**
Mining - Capitalized Leasing			**7,146,849**	**7,146,849**	**7,146,849**	**7,146,849**	**7,146,849**	**-**	**-**	**-**	**-**	**(9,031,539)**	**26,702,708**
777 Truck		-	4,287,519	4,287,519	4,287,519	4,287,519	4,287,519					(5,418,176)	**16,019,417**
992 Loader		-	1,083,240	1,083,240	1,083,240	1,083,240	1,083,240					(1,368,900)	**4,047,299**
Drill Rig (6" - 9")		-	697,356	697,356	697,356	697,356	697,356					(881,256)	**2,605,526**
D9		-	631,334	631,334	631,334	631,334	631,334					(797,823)	**2,358,849**
775 Water Truck		-	148,154	148,154	148,154	148,154	148,154					(187,223)	**553,546**
14M		-	72,877	72,877	72,877	72,877	72,877					(92,095)	**272,290**
Crane Truck		-	33,330	33,330	33,330	33,330	33,330					(42,120)	**124,532**
ANFO Loader		-	34,719	34,719	34,719	34,719	34,719					(43,875)	**129,721**
Backhoe - 1 m3		-	33,330	33,330	33,330	33,330	33,330					(42,120)	**124,532**
Fuel /Lube Truck Class 8		-	69,438	69,438	69,438	69,438	69,438					(87,750)	**259,442**
Light Trucks		-	55,551	55,551	55,551	55,551	55,551					(70,200)	**207,554**
Process - Subtotal		**-**	**32,804,113**	**302,113**	**302,113**	**302,113**	**6,764,113**	**-**	**-**	**-**	**-**		**40,474,564**
Process - Expenditures		**-**	**32,502,000**	**-**	**-**	**-**	**6,462,000**	**-**	**-**	**-**	**-**	**(4,766,161)**	**34,197,839**
Contingency	**20%**	**-**	**5,417,000**	**-**	**-**	**-**	**1,077,000**	**-**	**-**	**-**	**-**		
Crushing Plant			8,100,000									(2,430,000)	**5,670,000**
Leach Pad Construction			4,885,000				5,385,000						**10,270,000**
Process Plant			9,800,000									(1,871,460)	**7,928,540**
Conveyor and Stacking System			3,700,000									(284,701)	**3,415,299**
Laboratory			600,000									(180,000.0)	**420,000**
Process - Leased Equipment		**-**	**302,113**	**302,113**	**302,113**	**302,113**	**302,113**	**-**	**-**	**-**	**-**	**(381,783)**	**1,128,781**
966 Loader		-	141,627	141,627	141,627	141,627	141,627					(178,975)	**529,158**
D6		-	160,486	160,486	160,486	160,486	160,486					(202,808)	**599,623**

	Units	-2	-1	1	2	3	4	5	6	7	8	9	TOTAL
							Years						
Other - Expenditures		**1,800,000**	**16,623,112**	**1,284,000**	**1,284,000**	**1,284,000**	**1,284,000**	**1,284,000**	**1,284,000**	**1,284,000**	**1,284,000**	**(4,823,112)**	**23,872,000**
Contingency	20%	300,000	950,000	214,000	214,000	214,000	214,000	214,000	214,000	214,000	214,000	1,600,000	
Environmental Bonding			3,500,000	70,000	70,000	70,000	70,000	70,000	70,000	70,000	70,000	(3,500,000)	**560,000**
Feasibility, Permitting		1,500,000											**1,500,000**
Royalty Purchase Option			1,000,000										**1,000,000**
First Fills			250,000										**250,000**
Sustaining Capital				1,000,000	1,000,000	1,000,000	1,000,000	1,000,000	1,000,000	1,000,000	1,000,000		**8,000,000**
Working Capital			10,923,112									(10,923,112)	**-**
Mine Closure and Reclamation												8,000,000	**8,000,000**
Before Tax Revenue and Cashflow													
EBDTIA		$ -	$ -	$ 18,319,759	$ 42,735,304	$ 27,773,774	$ 32,971,030	$ 19,032,296	$ 3,654,552	$ 15,507,007	$ 56,162,645	$ 21,393,699	**$ 237,550,068**
Revenue-Capital		$ (2,400,000)	$ (64,014,074)	$ 9,586,797	$ 34,002,342	$ 19,040,812	$ 17,776,068	$ 17,748,296	$ 2,370,552	$ 14,223,007	$ 54,878,645	$ 26,216,811	**$ 129,429,257**
Cumulative cashflow		$ (2,400,000)	$ (66,414,074)	$ (56,827,277)	$ (22,824,935)	$ (3,784,122)	$ 13,991,946	$ 31,740,241	$ 34,110,793	$ 48,333,801	$ 103,212,446	$ 129,429,257	**$ 129,429,257**
Pre-tax NPV @ 5% per annum		**$ 76,495,927**	(End of year discounting)										
Pre-tax NPV @ 10% per annum		$ 43,790,153											
IRR		25.7%											
Federal and State Taxes		$ -	$ -	$ (339,117)	$ (2,602,317)	$ (1,598,253)	$ (2,766,110)	$ (1,190,195)	$ -	$ (212,278)	$ (8,552,091)	$ (3,494,401)	$ (20,754,762)
After Tax Cash Flow													
After Tax Cash Flow (ATCF)	$ -	$ (2,400,000)	$ (64,014,074)	$ 9,247,680	$ 31,400,025	$ 17,442,559	$ 15,009,958	$ 16,558,101	$ 2,370,552	$ 14,010,729	$ 46,326,554	$ 22,722,409	**$ 108,674,494**
Cumulative ATCF	$ -	$ (2,400,000)	$ (66,414,074)	$ (57,166,393)	$ (25,766,368)	$ (8,323,809)	$ 6,686,149	$ 23,244,250	$ 25,614,802	$ 39,625,531	$ 85,952,085	$ 108,674,494	
Post Tax NPV @ 5% per annum		**$ 62,469,634**	(End of year discounting)										
Post Tax NPV @ 10% per annum		$ 33,981,413											
IRR		22.8%											