NI 43-101 Technical Report

Preliminary Economic Assessment of the Getchell Gold Corp. Fondaway Canyon Project Nevada, USA

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1. SUMMARY

This Technical Report (the "Technical Report") has been prepared by Forte Dynamics, Inc. (Forte) and APEX Geoscience Ltd. (APEX), for the Issuer, Getchell Gold Corp. (Getchell Gold or the Company). The Fondaway Canyon Project (Fondaway Canyon, the Project, or the Property) is located on the western flank of the Stillwater Range in Churchill County, northwestern Nevada, 140 kilometers (km) northeast of Reno, Nevada, and 58 km northeast of Fallon, Nevada.

1.1 Mineral Tenure

The Project consists of 253 contiguous, unpatented mining lode claims, covering approximately 1,806 hectares (4,463 acres), on land administered by the U.S. Bureau of Land (BLM). The claims are currently held by Richard E. Fisk under an agreement with the Company, and by Getchell Gold Nevada Inc., a wholly owned subsidiary of Getchell Gold. The claims are listed in Table 4-1 and shown in Figure 4-2 grouped by the registered claimant.

1.2 Environmental Liabilities and Permitting

At the execution of a definitive option agreement to acquire 100% of the Fondaway Canyon Project from Canarc Resource Corp., now known as Canagold Resources Ltd., on January 3, 2020 (Canagold Option), the Fondaway Canyon Property was encroached on three sides by the Stillwater Wilderness Study Area. As of December 23rd, 2022, following the passage of the National Defense Authorization Act (NDAA), the Stillwater Wilderness Study Area was released. The Numunaa Nobe National Conservation Area (NCA) (Figure 4-2) was established with a reduced footprint. The newly established boundaries of the NCA formalized in the NDAA opened additional area for exploration and mining around the existing claim group. Current exploration, including drilling, is being carried out under an existing 5-acre Surface Management Notice disturbance permit (NVN95628). The reclamation bond is currently set at US $22,619.

1.3 Drilling

Total drilling on the Fondaway Canyon Property includes 765 drill holes for over 64,419 meters (m) completed between 1981 and 2022 by various operators including Getchell Gold. A brief summary of historical drilling is provided in Section 10.1 with additional details included in Section 6. Drilling conducted by Getchell Gold is summarized in Section 10.2.

1.4 Geology

The gold mineralization at Fondaway Canyon appears to conform to an orogenic intrusion-related mesothermal gold system. Although this is the most likely model for mineralization, structurally controlled, low-sulfidation epithermal mineralization cannot be entirely ruled out. A schematic showing the types of mineralization typically associated with this deposit type is provided in Figure 8-1.

Gold deposition occurs adjacent to first-order, deep-crustal fault zones with interpreted long-lived structural controls. These first-order faults, which can be hundreds of kilometers long and kilometers wide, show complex structural histories. Economic mineralization typically formed as vein fill of second- and third-order shears and faults, particularly at jogs or changes in strike along the crustal fault zones. Mineralization styles vary from stockworks and breccias in shallow, brittle regimes, through laminated crack-seal veins and sigmoidal vein arrays in brittle-ductile crustal regions, to replacement- and disseminated-type orebodies in deeper, ductile environments. The specific style of gold mineralization at Fondaway can be classified as both structurally controlled, vein associated and locally disseminated in zones of silicification and/or brecciation.

1.5 Metallurgical Testing

Several scoping level mineralogical and metallurgical scoping level studies have been undertaken on the Fondaway Canyon Property from 1984 to 2017.

Getchell Gold contracted Forte Analytical in 2024 to complete a scoping level metallurgical study for the Fondaway Canyon Project with the primary objective to develop a conceptual process flowsheet for the oxide and sulfide samples that minimize both capital expenditures (CAPEX) and operating expenditures (OPEX). The test program was expanded to include processing of oxide ore which occurs on the surface of the sulfide deposit.

The scoping level metallurgical study evaluated several processing options following the test work on deportment of gold which indicated that a majority of the gold was refractory and associated with pyrite.

Both oxide and sulfide ore can be readily floated to produce a concentrate containing 80% to 90% of gold. The concentrate can be upgraded to reduce concentrate weight and increase the gold grade of the concentrate.

Preliminary scoping studies indicate that deleterious elements are not in sufficient quantity to negatively impact the sale of concentrates and should be readily marketable. Additional test work is needed to refine the preliminary conclusions.

A review of the CAPEX and OPEX for various processing options indicated that the most promising approach at this stage of the study is to produce a gold-rich concentrate, ± 20 grams/tonne (g/t Au) and ship/sell it to a processing facility in Nevada.

1.5.1 Recommendations

The metallurgical study should be continued to optimize the flotation process in order to produce a high-grade concentrate with high gold recovery.

1.6 Mineral Resource Estimate

Getchell Gold engaged APEX to prepare a Mineral Resource Estimate (MRE) for the Fondaway Canyon Project. The 2024 Fondaway MRE has an effective date of October 31, 2024. The MRE was completed by Kevin Hon, B.Sc., P.Geo., Senior Geologist with APEX under the direct supervision of Michael B. Dufresne, M.Sc., P.Geol., P.Geo., President of APEX and QP for the Mineral Resource Estimate.

Fondaway Canyon gold mineralization is localized along a trend of over 3.5 km (2 miles) of en echelon, east-northeast trending and steeply south dipping structures developed within fine grained Triassic carbonaceous siliciclastic sedimentary rocks and Jurassic limestone, cut by Tertiary dikes (Norred and Henderson, 2017).

The area is interpreted as an east-west district left lateral shear zone with a dilation zone (releasing bend) with north-northeast mineralized structural strands hosting the Main Zone resource and linking a throughgoing ~east-west district-scale mineralized fault zone. Dilation zone and brittle zone quartz veins and stockworks along with sulfide mineralization likely developed late in the history of the shear zone.

1.6.1 Oxidation Modeling and Interpretation

Oxidation logging was reviewed from the historical and modern drilling data and evaluated for a potential zone of alteration. In total, 386 drill holes contain oxidation logging information. Oxidation logging only exists at the Main and Silica Ridge – Hamburger Hill zones.

1.6.2 Bulk Density

A total of 1,377 modern density samples were collected from nine drill holes completed between 2020 – 2022. These drill holes intersect both mineralized and non-mineralized material from various lithology types. All the samples came from the Main Zone estimation domain. The density values ranged from 2.4 g/cm^3 to 2.99 g/cm^3.

1.6.3 Compositing Methodology

The drill hole sample interval lengths within the estimation domains at Fondaway Canyon vary from 0.09 to 9.15 m, as illustrated in Figure 14-8. A composite length of 5 feet (1.53 m) was chosen because 97% of the sample intervals are equal to or shorter than this length.

Composites were capped to a specified maximum value to ensure metal grades are not overestimated by including outlier values during estimation. Probability plots illustrating each composite's values are used to identify outlier values that appear greater than expected relative to each estimation domain's commodity distribution. If outliers are identified as part of a high-grade trend that still requires grade capping, the capping level applied may be less stringent than the level used for controlling isolated high-grade outliers.

1.6.4 Grade Estimation of Mineralized Material

Ordinary Kriging (OK) was used to estimate metal grades for the 2024 Fondaway Canyon MRE block model. Only blocks that intersect the resource estimation domains were estimated.

Estimation used locally varying anisotropy (LVA), which employs different rotation angles to set the variogram model's principal directions and search ellipsoid for each block. Trend surface wireframes assign these angles to blocks within the estimation domain, enabling local structural complexities to be captured in the estimated block model.

Contact analysis of the boundaries between adjacent estimation domains showed that the metal profile at the boundary is hard or semi-hard, where the profiles trend toward each other over a very short distance. Consequently, only data from within each domain was used for grade estimation within that specific domain.

A multiple-pass estimation method was used to control Kriging's smoothing effect and limit the influence of high-grade samples, ensuring accurate grade and tonnage estimates at the block scale. Each pass considered up to 30 composites, with a minimum of one required for estimation. Table 14-8 details the restricted search parameters and limits the number of composites from each drill hole. While these rules may introduce local bias, they improve the global accuracy of grade and tonnage estimates above the reporting cutoff.

Measured Mineral Resources are currently not defined. For future resource assessments, ranking historical drill holes based on confidence in their collar and downhole surveys is recommended. Only drill holes with high confidence should be considered for measured resources in conjunction with modern drilling data. Additionally, a more robust geological model would provide more confidence to the Project to more accurately construct the estimation domains.

1.7 Mineral Resource Estimate Statement

The resource block model underwent several pit optimization scenarios using Deswik's Pseudoflow pit optimization. Table 14-10 outlines the economic assumptions used for pit optimization and to establish the reporting cutoff of 0.3 g/t Au.

After evaluating the continuity of the manually flagged blocks below the open-pit shell, a cutoff of 1.75 g/t Au was chosen for reporting of the potential underground mineral resource.

Table 1-1: Summary of 2024 Indicated and Inferred Mineral Resources on the Fondaway Canyon Project

Mineral Resource Area	Cutoff Au (g/t)	Classification	Tonnes (kt)	Au (g/t)	Au (toz/st)	Au (koz)
Pit-Constrained Mineral Resource Estimate						
Main	0.3	Indicated	13,518	1.49	0.043	648
		Inferred	37,983	1.09	0.032	1,335
Mid Realm - South Mouth	0.3	Inferred	2,516	0.95	0.028	77
Silica Ridge - Hamburger Hill (HH)	0.3	Inferred	2,977	1.45	0.042	139
Underground Mineral Resource Estimate						
Main/ Silica Ridge - HH	1.75	Inferred	1,353	2.74	0.080	119
Total Mineral Resource Estimate						
All	0.3/1.75	Indicated	13,518	1.49	0.043	648
		Inferred	44,829	1.16	0.034	1,670

Notes:

1) Michael B. Dufresne, M.Sc., P.Geol., Senior Consultant, Mineral Resources of APEX Geoscience Ltd., who is deemed a qualified person as defined by NI 43-101 is responsible for the completion of the mineral resource estimation, with an effective date of October 31, 2024.
2) The mineral resources presented are not mineral reserves, and they do not have demonstrated economic viability. There is no guarantee that any part of the resources defined by the MRE will be converted to a mineral reserve in the future.
3) The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
4) The Inferred Mineral Resource in this estimate has a lower level of confidence than that applied to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of the Inferred Mineral Resource could potentially be upgraded to an Indicated Mineral Resource or a higher classification with continued exploration.
5) The Mineral Resources were estimated in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources & Reserves, Definitions (2014) and Best Practices Guidelines (2019) prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council.
6) Economic assumptions used include US$1,950/oz Au, process recoveries of 92% for Au, a US$15/t processing cost, G&A cost of US$2/t, and a 1% royalty.
7) The constraining pit optimization parameters were US$2.7/t mineralized and waste material mining cost and 45° pit slopes. Pit-constrained Mineral Resources are reported at an Au cutoff of 0.3 g/t.
8) The Underground Mineral Resources include blocks outside the constraining pit shell that form continuous and potentially mineable shapes. A mining cost of US$83/t and the economic assumptions above result in the Underground Au cutoff of 1.75 g/t. Mining shapes encapsulate material within domains with a minimum horizontal width of 1.5 meters, perpendicular to the strike, and target vertical and horizontal dimensions of approximately 15 meters. Blocks narrower than the required mining thickness are only included if their diluted grade exceeds the cutoff when adjusted to the minimum mining width.

1.8 Risk and Uncertainty in the Mineral Resource Estimate

The 2024 Fondaway Canyon MRE database is dominated by historical drilling. The drilling of 15 core holes by Nevada Contact Gold and Canagold in 2002 to 2017 and a further 28 holes by Getchell Gold from 2020 to 2022 in the Main Zone resource area has greatly improved the understanding of the geological model that was used in the construction of the 2024 MRE for the Main Zone. The geological and mineralization domains were improved and adjusted based upon this drilling. However, the geological model has changed from a discrete quartz vein model with higher grades, to a lower grade vein and stockwork mineralization zone model that is more suited to a bulk tonnage open pit extraction scenario for the resource. Uncertainty in the geological model still exists in areas of Inferred Mineral Resources with little to no modern drilling.

Historical metallurgical testing has focused on previous geological interpretations of quartz vein stockworks and sulfide halos in carbonaceous to calcareous sedimentary host rocks. Additionally, modern data analysis has identified a significant portion of near surface oxidized mineralization. Modern metallurgical test work would increase the confidence of the recovery methodology of the new geological interpretations, and the recovery value of the identified near surface oxide mineralization.

1.9 Mining Methods

The Fondaway Canyon Project will consist of an open pit mining operation using conventional equipment. The Project is a conventional hard rock open pit mine that will use a contractor for mining. Mining is planned on 6-meter benches using haul trucks, and conventional drill and blast activities. Processed material is planned to be mined at a rate of 8,000 tonnes per day.

Pits were developed using the economic parameters shown in Table 1-2.

Table 1-2: Pit Optimization Parameters

Modifying Factor	Units	Value
Gold Price	US$/toz	$2,200
Gold Price	US$/gr	$70.7
Gold Refining Charges	%	0
Royalties	%	1.0
Cost		
Mining	US$/tonne	$2.60
Processing	US$/tonne	$30.0
G&A	US$/tonne	$2.0
Plant Recovery	%	88
Slopes	degrees	45

A pit shell smaller than the maximum possible was selected due to the high waste stripping requirements and long lead times of the largest economic pits. Pit shell 40 was selected as the optimum pit shell which correspond to a 40% Revenue Factor. This shell has a total tonnage of 199.2 million tonnes (Mt) including 35.4 Mt of processed material at an average grade of 1.51 g/t Au for 1.7 million ounces (Moz) of contained gold, with average stripping ratio of 4.73. Incremental stripping ratios for the larger pit shells exceed 10:1. This can be seen in Table 16-3 and Figure 16-2. The final designed pit and estimated block grades are shown in Figure 1-1.

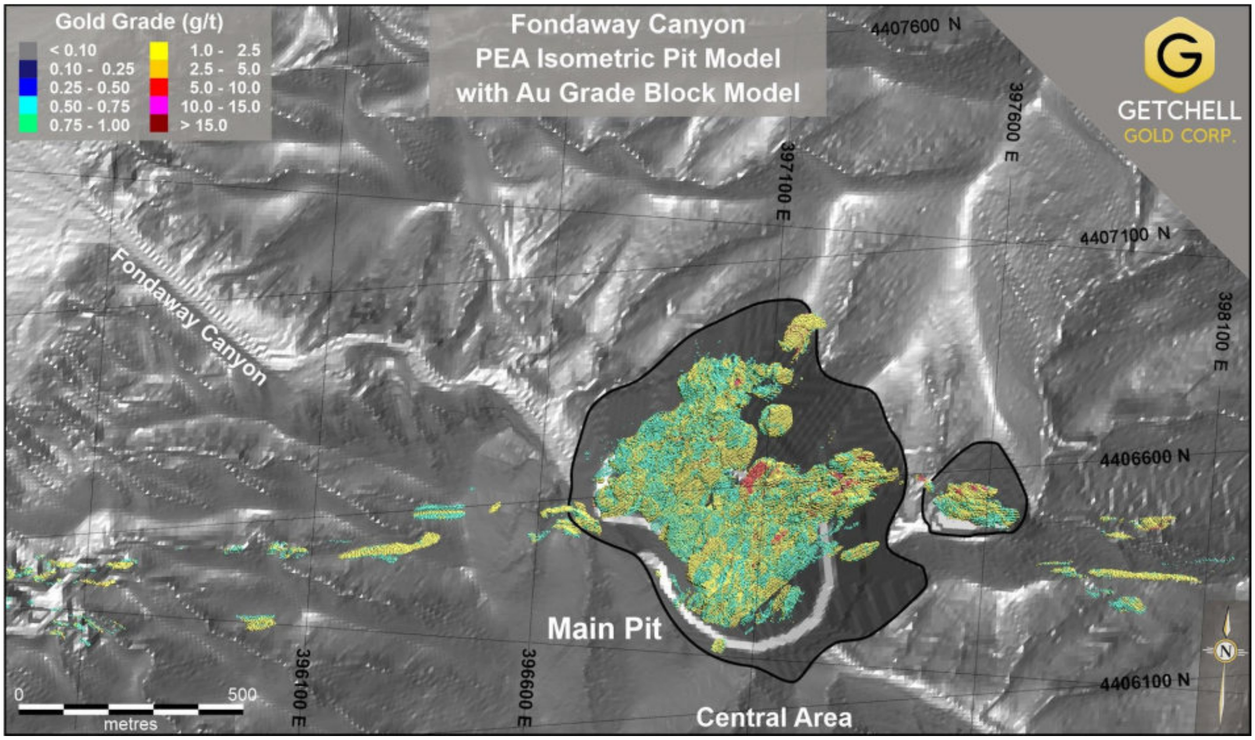

Figure 1-1: Getchell Fondaway Canyon Project Final Pit

(Source: Forte Dynamics, Inc. 2024)

The summary of in-pit mineral resources is shown in Table 1-3.

Table 1-3: In-Pit Mineral Resources by Pit Phase

		Processed Resource				Waste	Total	
Cut	Material	ktonnes	Au g/t	Au ktoz Contained	Au ktoz Recovered	ktonnes	ktonnes	Stripping Ratio
PIT-1A	2-Indicated	533	2.12	36	32	174	707	0.33
	3-Inferred	173	1.19	7	6	114	287	0.66
						2,850	2,850	0.00
PIT-1B	2-Indicated	5,011	1.76	283	249	2,598	7,609	0.52
	3-Inferred	2,359	1.34	102	90	3,574	5,933	1.51
						28,388	28,388	0.00
PIT-2	2-Indicated	3,885	1.63	204	180	1,704	5,590	0.44
	3-Inferred	7,293	1.41	329	290	8,897	16,190	1.22
						38,315	38,315	0.00
PIT-34	2-Indicated	2,326	1.76	131	116	975	3,301	0.42
	3-Inferred	8,877	1.33	378	333	9,461	18,338	1.07
						46,228	46,228	0.00
Total	2-Indicated	11,756	1.73	655	576			
	3-Inferred	18,702	1.36	816	718			
						143,277	173,734	4.7

1.10 Mineral Reserve Estimate

There are currently no mineral reserves estimated for Fondaway Canyon.

1.11 Mining Method

The Fondaway Canyon Project will consist of an open pit mining operation using conventional equipment. The Project is a conventional hard rock open pit mine that will use a contractor for mining. Mining is planned on 6-meter benches using haul trucks, and conventional drill and blast activities. Processed material is planned to be mined at a rate of 8,000 tonnes per day (t/d or tpd).

The open pit optimization assessed the sensitivity of the pit optimizations to the fluctuation in the revenue generated, as defined by the Revenue Factor, a parameterization of the metal price and cost numbers, as well as the impact of pit size and stripping ratio on the Projects' NPV. This procedure yields a series of nested pit shells that prioritize the extraction of the most economically viable and most economically robust material based on factoring the costs shown in Table 1-4.

Table 1-4: Design Metal Prices, Cost, and Recoveries

Description	Units	Fondaway
Mining Cost	US$/tonne	$3.54
Processing Cost Processed Material	US$/tonne	$15.00
G&A Cost Processed Material	US$/tonne	$2.00
Gold Price	US$/toz	$2,250
Gold Price	US$/gr	$72.3
Transport and Refining Cost Processed Material	US$/tonne	$10.00
Recovery	%	84

The shell selection is presented in Table 16-3, and a sectional view of the shells is in Figure 16-3. The shells selected for pushback designs and the eventual mine scheduling were the 20%, 30%, 40% 50% and 60%, although the 50% and 60% pits were eliminated as they required stripping ratios greater than 12:1 and were marginally profitable.

Pit shell 40 was selected as the optimal pit shell which corresponds to a 40% Revenue Factor. Revenue Factors are used to calculate pit shells by varying the commodity prices but keeping the costs the same. This shell has a total tonnage of 199.2 Mt including 35.4 Mt of processed material at an average grade of 1.51 g/t Au for 1.6 Moz of contained gold. The average stripping ratio is 4.73. The mineral resources by phase are shown in Table 1-3.

The final pit limit contains a total tonnage of 173.7 Mt including 11.7 Mt of Indicated Mineral Resource at 1.73 g/t Au, and 18.7 Mt of Inferred Mineral Resource at 1.36 g/t Au to be processed for 1.47 Moz of contained gold. Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability. There has been insufficient exploration to define the Inferred Resources tabulated above as an Indicated or Measured Mineral Resource, however, it is reasonably expected that many of the Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration. There is no guarantee that any part of the Mineral Resources discussed herein will be converted into a Mineral Reserve in the future.

The phase designs were used to create a life of mine (LOM) schedule for the site. This schedule considers open-pit operations. The yearly mine schedule is presented in Table 16-6, and the production profile is shown in Figure 1-2. The schedule was completed quarterly for the first year of mining and yearly for the rest of the mine life. The production schedule is driven by the nominal rate of 8,000 t/d processed material (2.9 Mt/y) and the average LOM stripping ratio is 4.73:1 waste-to-ore, using a 0.5 g/t Au cutoff grade.

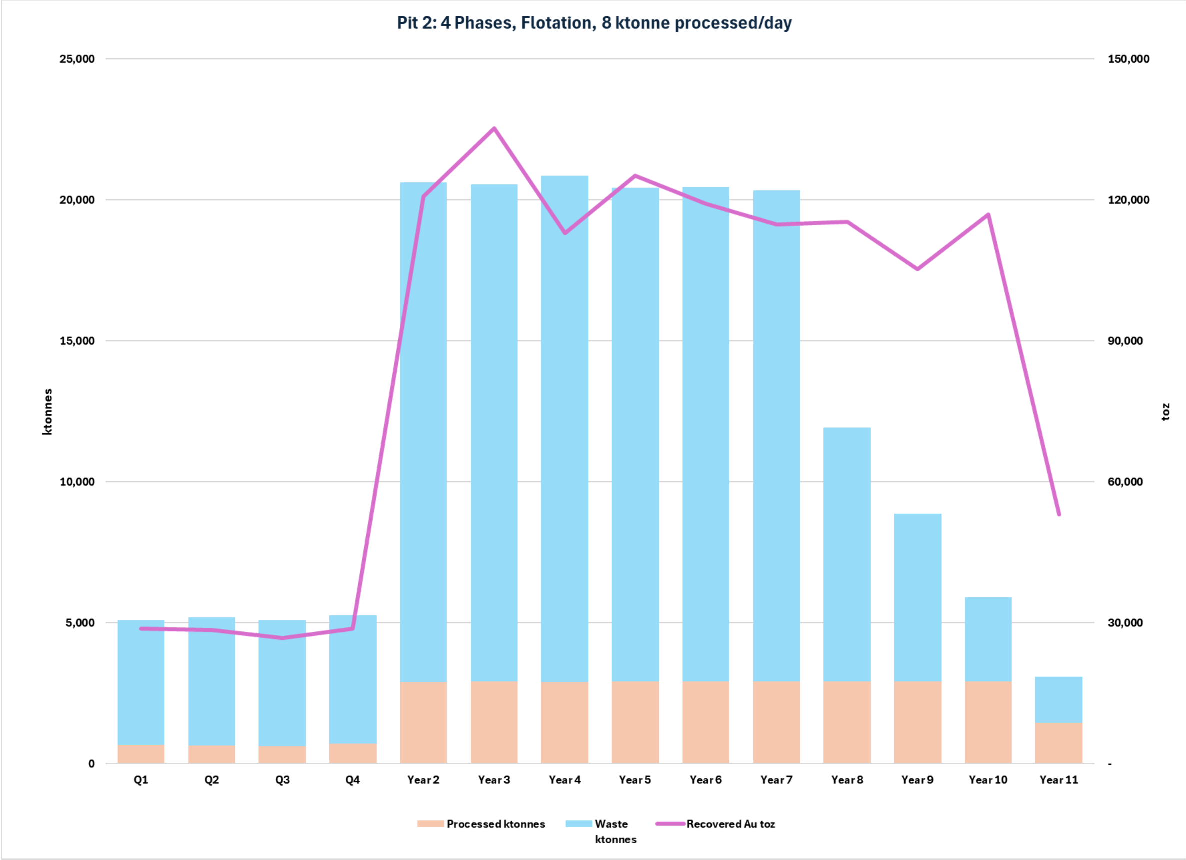

Figure 1-2: LOM Production Schedule

(Source: Forte Dynamics, Inc. 2024)

1.12 Recovery Methods

A conceptual process flowsheet was developed based on the historical and scoping level study undertaken by Forte Analytical in 2024. A techno-economic evaluation of the various processing options was undertaken. The options included (1) whole ore pressure oxidation (POX) followed by cyanidation, (2) crush-grind-flotation of gold and sulfides followed by POX and cyanidation, and (3) crush-grind-flotation and sell flotation concentrate. The last processing option is the most viable for Getchell Gold at this time.

The process flowsheet used for planning and cost estimation is based on the following assumptions:

- The optimum comminution circuit for 8,000 tpd will be three-stage crushing followed by a closed circuit ball mill-cyclone system.
- Rougher flotation will recover 88% to 90% of the gold in 20% of the weight.
- Two-stage cleaner flotation will reject 50% of the weight while recovering 95% of the gold in the rougher concentrate.
- The second-cleaner concentrate will have 10% of the weight of the original feed (i.e. 800 tpd) assaying about 20 g/t Au.

- The gold recovery in the pressure oxidation circuit is estimated at 95% of the contained gold in the flotation concentrate.

The process flowsheet will consist of three stages of crushing followed by ball-mill-cyclone configuration. The cyclone overflow will be sent to the rougher flotation. The rougher-concentrate will be sent to the first-cleaner flotation. The first-cleaner tailing will be combined with the rougher tailing and sent to the tailings pond. The first-cleaner concentrate will be further upgraded in second-cleaner flotation. The conceptual process flowsheet is shown in Figure 17-1.

1.13 Project Infrastructure

The Property is accessed from Fallon east on Highway 50 and then north on Highway 116 to the settlement of Stillwater and then north on the gravel East County Road 30 miles (50 km) along the front range of the Stillwater Range to the mouth of Fondaway Canyon. Existing mine roads provide access into the canyon and there remains a dense network of drill roads developed over decades of exploration and mining in various states of use.

There are no structures at Fondaway Canyon. There are the remains of old mine workings, both underground and open pit, as well as two water wells and the existing exploration roads. There is ample space for a plant on the overburden deposits in the basin at the mount of the canyon which is within the current permit limit.

1.14 Market Study

The estimated gold price used was based on historical market conditions giving credence to current price trends. The price estimate was based on 2/3 of a three year trailing average, and 1/3 of a bank consensus future forecast compiled on January 3, 2025 by Scottsdale Bullion & Coin (SBC), which is shown in Table 19-2. This analysis resulted in a forecast of US $2,287/toz Au, rounded to US $2,250/toz Au for this study.

1.15 Environmental Impacts

The Project status changed as of December 23rd, 2022, following the passage of the National Defense Authorization Act ("NDAA"), the Stillwater Wilderness Study Area was released. The Numunaa Nobe National Conservation Area ("NCA") (Figure 4-2) was established with a reduced footprint.

Exploration work, including drilling, is being carried out under an existing 5-acre Surface Management Notice disturbance permit (NVN95628). The reclamation bond is currently set at US $22,619.

No other permits currently exist, although as the Project has been mined previously, the QP believes that the Project can be permitted. The recommendations will include a budget to bring the permitting to the next step.

1.16 Capital and Operating Costs

The capital and operating costs used in this report were based on costs from similar project work performed recently by Forte, and by interpolation from CostMine™ models. The QP's believe that the estimates are appropriate for inclusion in this report. The QP believes that these costs comply with the precision requirements for a Preliminary Economic Assessment (PEA).

Capital costs for the mine and the plant were estimated by interpolating published data from CostMine™. Mine capital cost does not include the mobile fleet as it is included in the contract mining cost. Capital cost includes both the construction and startup phases. The initial capital cost, which includes process, preproduction, and facilities, is estimated at US $226.47 million with a 20% contingency. There is no sustaining capital at this stage as mining contractors are planned to be used for all major mining work. The CAPEX summary is shown in Table 1-5.

Table 1-5: Project Capital Cost Summary

Category	US $M
Process Capital CostMine™ Model	$131.74
Preproduction and Facilities	$56.98
Summary CAPEX	$188.72
Contingency (20%)	$37.75
Total CAPEX	**$226.47**

Operating costs for the mine and the plant were estimated by interpolating published data from CostMine™. Table 1-6 provides a detailed breakdown of operating costs for the Project.

Table 1-6: Project Operating Cost Summary

Operating Costs	$/tonne Mined	LOM (US $M)	$/oz Au Produced
Mining to Process	$ 3.54	$ 107.4	$ 87.2
Mining Waste	$ 3.54	$ 507.4	$ 412.1
Processing	$ 13.25	$ 402.0	$ 326.5
Mine Site G&A	$ 2.00	$ 60.7	$ 49.3
Total Operating Costs:		$ 1,077.5	$ 875.0
Transportation and Refining	$ 10.00	$ 303.4	$ 246.4
Royalties	3%	$ 83.0	$ 67.5
Total Cash Costs:		$ 1,464.0	$ 1,188.9

1.17 Economic Analysis

This report presents a PEA level analysis, which incorporates Inferred Resources in the economic model. The favorable economic results presented do not define a mineral reserve. While the economic parameters used in this technical report are considered reasonable, additional information could alter these assumptions and affect the analysis. All figures are expressed in constant 2024 US dollars.

The costs used in the economic model are summarized in Table 1-6.

Table 1-7: Cost Summary

Prices	
Gold Price ($/toz)	$2,250
Initial Capital	$226.47M
Sustaining Capital	$0M
Project Life (Years)	10.5
Production	
Total Mined Processed Material (ktonnes)	30,343
Total Mined Waste (ktonnes)	143,392
Total Mined Gold (k toz)	1,466
Au Grade (opst)	0.044
Au Grade (g/t)	1.50
Average Operating Costs	
Open Pit Mining Cost ($/t)	$3.54
Process Cost ($/t)	$13.25
Transportation and Refining Cost ($/t)	$10.00
Gen. & Admin. Cost ($/t processed material)	$2.00
NSR Royalty (%)	3.0

The operating and capital costs have been used with the mine production schedule to produce a discounted cash flow model. The model is presented in Appendix B and the summary results are shown in Table 1-8 and Table 1-9.

Table 1-8: Pre-Tax NPV Summary

Pre-Tax NPV	US $M
NPV @ 0%	$1,080.13
NPV @ 5%	$761.12
NPV @ 8%	$622.38
NPV @ 10%	$545.73
NPV @ 12%	$479.29
IRR	51.2%
Payback Period	3.1 years

Table 1-9: After-Tax NPV Summary

After-Tax NPV	US $M
NPV @ 0%	$953.37
NPV @ 5%	$667.51
NPV @ 8%	$542.93
NPV @ 10%	$474.01
NPV @ 12%	$414.23
IRR	46.7%
Payback Period	3.2 years

1.18 Conclusions

Based on the estimated quantity of mineral resources with economic potential for an open pit operation, the Fondaway Canyon Project is economically robust with an 8,000 t/d operation and a 10.5 year mine life. The discounted cashflow economic analysis returns a pre-tax NPV of US $545.7 million, and an after-tax NPV of US $474.0 million at a discount rate of 10% with an initial capital investment of US $226.5 million.

Getchell Gold has a clear title to the Fondaway Project, including a significant database of technical information, drill data, geologic interpretation, and preliminary metallurgical data. The data are of industry standard quality and are suitable to be used for resource estimation and future work for the Project.

Their interpretations of the Project as a surface mineable producer of flotation concentrate have overcome issues of refractory gold and attempting to pursue high grade underground targets within the system. This is of course enhanced by the shift in gold prices since 2020.

The 2020 drill program provided confirmation for the geological model. The 2021-2022 drill programs continued to delineate the mineralization. All drilling programs from 2020-2022, completed by Getchell Gold, assisted in providing confirmation of the historical drilling database along with yielding greater confidence in the Mineral Resource Estimate, as well as enhancing the understanding of the mineralized and non-mineralized contacts.

The Fondaway Canyon Project contains a significant gold resource with good continuity at relatively low cutoff grades, and significant contribution from higher-grade zones. The resource as reported is contained within an economic pit shell and appears to be amenable to open pit mining methods. Due to the complex geometry of the canyon, the pushbacks, designed to provide robust economic returns, were explicitly designed to provide economic confidence in the early production years, and to assure the potential of successfully pre-stripping successive pushbacks.

Initial metallurgical test work confirms that the deposit is refractory for cyanidation; however, as much of the gold is associated with pyrite and other sulfides, froth flotation shows the potential to create a high-grade gold bearing sulfide concentrate which could be processed via pressure oxidation to achieve economic recoveries.

The PEA indicates that at the gold prices considered, the Project shows potential to be developed as an open pit surface mining operation. A sensitivity study executed at near-spot prices indicates additional potential for the deposit at higher metal prices. The QP's believe that before proceeding to a potential next phase Pre-Feasibility Study (PFS), it would be beneficial to complete additional step out drilling which may increase the mineable mineral resource, infill drilling to increase the confidence in the resource, and appropriate test work to refine the metallurgical assumptions and process flow sheet.

1.19 Risks and Uncertainties

The Fondaway Canyon Project is subject to the risks and uncertainties typical of gold projects, particularly risk in commodity prices and the precious metals equity markets. Lower metal 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 confirmation of transitional grades around structural zones, that material used for metallurgical testing is not representative of the overall deposit, and with the handling of water inflows to the pit, and/or of adequate availability of water for the mill operation.

Each of these risks appear to be manageable; however, there is potential to increase the operating or capital cost for the Project, or delay development activities.

The life of mine (LOM) plan includes a majority percentage of Inferred Mineral Resources, compared to the amount of Indicated Mineral Resources (there are no Measured Mineral Resources). The current mineable resource demonstrates economic viability but will need to be upgraded to become a mineral reserve.

Metallurgical data appears to be of reasonable quality but is considered preliminary. Incomplete classification of material types or misunderstanding of the representativeness of metallurgical samples could lead to a change in recovery or process cost assumptions. Further test work is 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 the Project economics.

1.20 Opportunities

There is potential to increase the Project mineral resource inventory. The mineralized areas of Fondaway Canyon are open along strike and down dip, and there are zones within the pit design developed in this study that did not have sufficient data to be classified as a mineral resource. This offers a path to increasing potentially economic mineral resources along with lowering the stripping ratio. Upgrading the classification of Inferred Mineral Resources to Indicated and/or Measured Mineral Resources would improve confidence in the mineral resource inventory and may have potential to increase the mineable resources. There are also zones of higher-grade material which may be amenable to underground exploitation if they can be connected and/or expanded.

Optimization of the operation of the flotation plant will offer opportunities to produce a more marketable concentrate, improving downstream revenues and reducing downstream costs.

1.21 Recommendations

The PEA has highlighted the potential of an open pit surface operation with a flotation concentrator to produce a gold concentrate for further treatment. The Fondaway Canyon Project is robust and demonstrates positive returns over a range of prices and costs. The discounted cashflow economic analysis returns a pre-tax NPV of US $545.7 million, and an after-tax NPV of US $474.0 million at a discount rate of 10% with an initial capital investment of US $226.5 million.

Based on the positive economic results from this PEA, there are several steps that the QP's feel should be taken that could progress the Project and/or prior to proceeding to a potential Pre-Feasibility Study (PFS), that can better define the overall potential of the Project.

1.21.1 Mineral Resource Expansion and Exploration

There is potential to increase the Project mineral resource inventory. The mineralized areas of Fondaway Canyon are open along strike and down dip beyond the designed pit limits, and there are zones within the pit design developed in this study that did not have sufficient data to be classified as a mineral resource.

There is also potential to upgrade mineral resources from Inferred to Indicated and Indicated to Measured, which will improve resource confidence and increase the potential mineable resource inventory and the potential for an economic mineral reserve estimate.

There are several areas within the Fondaway Canyon Project that the Company believes warrant further exploration. In addition to resource definition within the designed pit limits, there is potential to expand the current modeled mineralized zones to the west for the Mid Realm and South Mouth areas, and to the east for the Silicon Ridge and Hamburger Hill areas.

1.21.2 Geological Model and Resource Domains

Review input data of geological, structural, and overall mineralization controls to refine the domain definitions for the Mineral Resource Estimate. The addition of structural data through drilling (see Table 26-1 below) could improve the understanding of structural controls on mineralization (and geology) and enhance the confidence in grade estimation and continuity, which could improve future mineral resource estimates.

1.21.3 Geotechnical Drilling

Specific geotechnical drilling and analysis of the pit highwalls is recommended to better understand the fracture behavior and rock strength characteristics and de-risk in-pit safety concerns.

1.21.4 Metallurgical Test Work

Additional metallurgical test work is recommended to provide greater confidence for input cost parameters, recovery, crush sizes for optimal extraction, and subsequent processing details. Flotation work on grind sizes and reagent consumption may improve recovery and increase concentrate grade with potential benefits to the Project economics.

1.21.5 Market Potential of Concentrates

The QP's recommend initial discussions with potential buyers of concentrates to gain a better understanding of the current and future market conditions.

1.21.6 Recommended Work Programs

A single-phase work program is recommended. The focus of the work program will be to enhance the confidence and potentially expand the current Mineral Resource Estimate. This could further outline the overall shape and orientation of the resource, and based on the results of this phase, additional drilling may be warranted. Additional metallurgical test work and other studies may be needed to further de-risk the Project.

Table 1-10: Recommended Work Programs

Budget Item	Estimated Cost
Resource Definition & Expansion Drilling	$2,000,000
Metallurgical Test Work & ARD	$125,000
Geotechnical Drilling	$100,000
Total	**$2,225,000**

2. INTRODUCTION

This Technical Report (the "Technical Report") has been prepared by Forte Dynamics, Inc. (Forte) and APEX Geoscience Ltd. (APEX), for the Issuer, Getchell Gold Corp. (Getchell Gold or the Company), a British Columbia (BC), Canada, based exploration company that is focused on gold and copper in Nevada (NV), USA. Getchell Gold entered into definitive option agreement with Canagold Resources Ltd. (Canagold) on January 3rd, 2020 to acquire 100% of the Fondaway Canyon Project.

The Fondaway Canyon Project (Fondaway Canyon, the Project, or the Property) is located on the western flank of the Stillwater Range in Churchill County, northwestern Nevada, 140 km northeast of Reno, Nevada, and 58 km northeast of Fallon, Nevada. The Property comprises 253 unpatented lode mineral claims.

2.1 Terms of Reference

This Technical Report is presented as a Preliminary Economic Assessment (PEA) of the Fondaway Canyon Gold Project in Nevada to support the economic potential of the Project. Forte and APEX have prepared the PEA to meet National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101), and in accordance with Form 43-101F1 Technical Report.

Mineral Resources are reported in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources & Mineral Reserves (2014) and the CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (2019).

2.2 Qualified Persons

The following persons serve as Qualified Person (QPs) as defined in National Instrument 43-101:

- Mr. Donald Hulse, SME-R.M., Director of Mining Resources with Forte Dynamics.
- Mr. Jonathan R. Heiner, SME-R.M., Director of Mining with Forte Dynamics.
- Dr. Deepak Malhotra, SME-R.M., Director of Metallurgy with Forte Dynamics.
- Mr. Michael B. Dufresne, P. Geol., P.Geo., President of APEX Geoscience.

2.3 Effective Dates

The Report has multiple effective dates:

- Date of database cutoff for Mineral Resource Estimation: April 31, 2024.
- Date of Mineral Resource Estimate: October 31, 2024.
- The overall Report effective date is January 15, 2025.
- The Report signing data is February 6, 2025.

2.4 Sources of Information and Data

Reports and documents listed in Section 3 (Reliance on Other Experts) and Section 27 (References) were used to support the preparation of the Report. Getchell Gold provided project data and information from their files in preparation of the Report. Additional regional and property geological information was available publicly in online libraries.

2.5 Site Visit Details and QP Inspections

The Property was visited by Mr. Donald Hulse on September 16-18, 2024.

The Property was visited by Mr. Jonathan Heiner on September 16-18, 2024.

On this visit the team visited various outcrops and historical workings, as well as potential infrastructure and access routes. They reviewed geology and structure, and they examined core to investigate mineralization styles and ore-waste contact areas.

The Property was not visited by Dr. Deepak Malhotra, although he has personally supervised the metallurgical test work in the Forte Analytical laboratory in Colorado.

The Property was visited by Mr. Mike Dufresne on May 7-8, 2022.

2.6 Units of Measure

All references to dollars in this report are to US dollars (US$) unless otherwise noted. Distances, areas, volumes, and masses are expressed in the metric system unless indicated otherwise.

For this report, common measurements are given in metric units. All tonnages shown are in tonnes of 1,000 kilograms, and precious metal grade values are given in grams per tonne (g/t), as well as troy ounces per short ton (opt).

Table 2-1: Units of Measure and Abbreviations

	Imperial Customary		Metric	
	Units	**Description**	**Units**	**Description**
Time	y	year	y	year
	d	day	d	day
	h	hour	h	hour
	min	minute	min	minute
	s	seconds	s	seconds
Length	ft	feet	m	meter
	in	inch	cm	centimeter
	mil	one thousandth of an inch	mm	millimeter
	mi	miles	µm	microns
Area	ft^2	square feet	m^2	square meters
			ha	hectare
Mass	st	short ton	mt or t	metric tonne
	kst	kilo ton	kt	kilo tonne
	dst	dry short tons	dmt	dry metric tonnes
	kst	thousand dry short tons	kmt	thousand dry metric tonnes
	lb	pound	kg	kilogram
	toz, troz, or troy oz	troy ounce	g	gram
Grade	opt or opst	troy ounces per short ton	g/t or gpt	grams per tonne

	Imperial Customary		Metric	
	opmt	troy ounces per metric tonne		
Volume	ft^3	cubic feet	m^3	cubic meter
	gal	gallons	L	liter
Density	lb/ft^3	pounds per cubic foot	t/m^3	tonnes per cubic meter
	sg	specific gravity	sg	specific gravity
Percent Solids	wt%	percent solids by weight		
Work Index (Hardness)	kWh/st	kilowatt-hours per short ton	kWh/t	kilowatt-hours per tonne
Elevation – Above Mean Sea Level (AMSL)	FASL	feet above sea level	MASL	meters above sea level
Volumetric Flow rate	gpm	gallons per minute	Lpm	liters per minute
	scfm	standard cubic feet per minute	m^3/hr	cubic meters per hour
Throughput	st/h or stph	short tons per hour	t/h or tph	metric tonnes per hour
	st/d or stpd	short tons per day	t/d, tpd, or mtpd	metric tonnes per day
	st/y or stpy	short tons per year	t/y or tpy	metric tonnes per year
Temperature	˚F	degrees Fahrenheit	˚C	degrees Celsius
Concentration	ppm	parts per million	mg/L	milligrams per liter
			g/L	grams per liter
Work Index	kWh/st	kilowatt hour per short ton	kWh/t	kilowatt hour per metric tonne
Power	hp	horsepower	kW	kilowatt
Mill Speed	rpm	revolutions per minute	rpm	revolutions per minute
Pressure	psi	pounds per square inch	kPa	kilopascal
Voltage			kV	kilvolt
			kVA	kilovolt-amperes

3. RELIANCE ON OTHER EXPERTS

The QPs have relied upon statements and information provided by Getchell Gold and its representatives concerning legal, political, environmental and tax matters relevant to the Technical Report.

3.1 Royalites, Mineral Claims & Agreements

Details regarding the nature of royalties, mineral claims and agreements were provided to the QP by Getchell Gold in the following documents:

- E-mail dated November 17th, 2022 from Michael Sieb, President of Getchell Gold.
- E-mail dated January 9th, 2023 from Michael Sieb, President of Getchell Gold.
- E-mail dated November 11th, 2024 from Michael Sieb, President of Getchell Gold.

3.2 Environmental Compliance

Details regarding the Stillwater Wilderness Study Area were provided by the following:

- Bureau of Land Management (BLM) Headquarters e-mails dated November 28th, 2022, November 29th, 2022, January 11th, 2023, January 12th, 2023, January 13th, 2023.
- Getchell Gold e-mails dated December 6th, 2023, January 12th, 2023.
- The James M. Inhofe National Defense Authorization Act for Fiscal Year 2023 (NDAA).

The QP is not qualified to provide a title opinion and have relied upon information provided by Getchell Gold, however, it should be noted that the QP reviewed the BLM register of lode claims (MLRS) on January 15th, 2025 and can confirm that the mineral claims listed in Section 4 (Table 4-1) are listed as Active.

This information is used in Section 4 of the Report and in support of the Mineral Resource Estimate in Section 14.

3.3 Tax Guidance

Forte reviewed the tax treatment of the economic model cash flow with Dennis Workman of MNP who provides tax services to Getchell Gold.

4. PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Fondaway Canyon Property is located on the western flank of the Stillwater Range in Churchill County, northwestern Nevada, 140 km northeast of Reno, NV, and 58 km northeast of Fallon, Nevada in Churchill County (Figure 4-1).

The Project centroid is 396213 E, 4406565 N, WGS 84 / UTM 11N (EPSG: 32611).

Figure 4-1: Project Location Map

(Source: Getchell Gold, 2022)

4.2 Mineral Tenure

The Project consists of 253 contiguous, unpatented mining lode claims, covering approximately 1,806 hectares (4,463 acres), on land administered by the U.S. Bureau of Land (BLM). The claims are currently held by Richard E. Fisk under an agreement with the Company, and by Getchell Gold Nevada Inc., a wholly owned subsidiary of Getchell Gold. The claims are listed in Table 4-1 and shown in Figure 4-2 grouped by the registered claimant.

Table 4-1: Fondaway Canyon Property Claims

Claim Name	Serial Number	Lead File	Location Date	Claimant	Status
Extension	NV101601202	NV101601202	1979-02-04	Richard E. Fisk	Active
Extension #4	NV102520786	NV102520786	1979-05-26	Richard E. Fisk	Active
Extension #5	NV101494812	NV101494812	1979-05-26	Richard E. Fisk	Active
Extension #6	NV101502168	NV101502168	1979-05-26	Richard E. Fisk	Active
Extension #7	NV101529453	NV101529453	1979-05-26	Richard E. Fisk	Active
Gold Hill # 1	NV101496539	NV101496539	1975-10-25	Richard E. Fisk	Active
Gold Hill # 2	NV101522630	NV101522630	1975-10-25	Richard E. Fisk	Active
Gold Hill # 3	NV101407103	NV101407103	1980-11-13	Richard E. Fisk	Active
Gold Hill # 4	NV101341961	NV101341961	1980-11-13	Richard E. Fisk	Active
Gold Hill # 5	NV101491354	NV101491354	1980-11-13	Richard E. Fisk	Active
Gold Hill # 6	NV101459086	NV101459086	1980-11-13	Richard E. Fisk	Active
White Caps	NV101604923	NV101604923	1961-01-12	Richard E. Fisk	Active
White Caps # 1	NV101604107	NV101604107	1961-01-14	Richard E. Fisk	Active
White Cap # 2	NV101478002	NV101478002	1968-10-14	Richard E. Fisk	Active
White Cap # 3	NV101406145	NV101406145	1968-10-14	Richard E. Fisk	Active
White Cap # 4	NV101600672	NV101600672	1968-10-14	Richard E. Fisk	Active
I Told You	NV101458019	NV101458019	1968-02-29	Richard E. Fisk	Active
"Little John, a/k/a Littel John"	NV101405714	NV101405714	1957-08-10	Richard E. Fisk	Active
Quicktung	NV101756687	NV101756687	1956-03-16	Richard E. Fisk	Active
Quicktung # 1	NV101300209	NV101300209	1956-07-03	Richard E. Fisk	Active
Quicktung # 2	NV101600847	NV101600847	1956-07-05	Richard E. Fisk	Active
Quicktung # 3	NV101303782	NV101303782	1956-07-08	Richard E. Fisk	Active
Quicktung # 4	NV101480349	NV101480349	1956-07-20	Richard E. Fisk	Active
Quicktung # 5	NV101600889	NV101600889	1956-09-18	Richard E. Fisk	Active
Quicktung # 6	NV101480015	NV101480015	1956-09-18	Richard E. Fisk	Active
Quicktung # 7	NV101350397	NV101350397	1957-03-04	Richard E. Fisk	Active
Sunrise Pike	NV101550156	NV101550156	1957-04-20	George Fisk & Richard E. Fisk and Wayne Fisk	Active
Sunrise Pike # 1	NV101521093	NV101521093	1957-05-04	Richard E. Fisk	Active
Chucker	NV101543201	NV101543201	1957-08-10	Richard E. Fisk	Active
FC # 14	NV101347052	NV101347052	1988-02-02	Richard E. Fisk	Active
FC # 16	NV101500005	NV101500005	1988-02-02	Richard E. Fisk	Active
FC # 18	NV101525337	NV101525337	1988-02-02	Richard E. Fisk	Active

Claim Name	Serial Number	Lead File	Location Date	Claimant	Status
FC # 20	NV101456272	NV101456272	1981-03-24	Richard E. Fisk	Active
FC # 22	NV101456387	NV101456387	1981-03-24	Richard E. Fisk	Active
FC # 24	NV101491171	NV101491171	1981-03-24	Richard E. Fisk	Active
FC # 26	NV101452864	NV101452864	1981-03-25	Richard E. Fisk	Active
FC # 28	NV101457457	NV101457457	1981-03-25	Richard E. Fisk	Active
FC # 30	NV101610375	NV101610375	1981-03-25	Richard E. Fisk	Active
FC # 55	NV101524435	NV101524435	1981-03-31	Richard E. Fisk	Active
FC # 56	NV101456102	NV101456102	1981-03-31	Richard E. Fisk	Active
FC # 57	NV101524453	NV101524453	1981-03-31	Richard E. Fisk	Active
FC # 58	NV101457732	NV101457732	1981-03-31	Richard E. Fisk	Active
FC # 59	NV101401897	NV101401897	1981-03-31	Richard E. Fisk	Active
FC # 60	NV101754339	NV101754339	1981-03-31	Richard E. Fisk	Active
FC # 61	NV101496184	NV101496184	1981-03-31	Richard E. Fisk	Active
FC # 62	NV101548955	NV101548955	1981-03-31	Richard E. Fisk	Active
FC # 63	NV101524413	NV101524413	1981-03-31	Richard E. Fisk	Active
FC # 64	NV101458607	NV101458607	1981-03-31	Richard E. Fisk	Active
FC # 65	NV101605235	NV101605235	1988-02-02	Richard E. Fisk	Active
FC # 66	NV101456641	NV101456641	1981-03-31	Richard E. Fisk	Active
FC # 67	NV101602992	NV101602992	1988-02-03	Richard E. Fisk	Active
FC # 68	NV101455273	NV101455273	1981-03-31	Richard E. Fisk	Active
FC # 69	NV101453631	NV101453631	1988-02-03	Richard E. Fisk	Active
FC # 70	NV101540408	NV101540408	1981-03-31	Richard E. Fisk	Active
FC # 71	NV101609486	NV101609486	1988-02-03	Richard E. Fisk	Active
FC # 72	NV101548617	NV101548617	1981-03-31	Richard E. Fisk	Active
FC # 73	NV101451938	NV101451938	1988-01-29	Richard E. Fisk	Active
FC # 74	NV101477880	NV101477880	1988-02-16	Richard E. Fisk	Active
FC # 75	NV101731283	NV101731283	1988-01-29	Richard E. Fisk	Active
FC # 76	NV101751264	NV101751264	1988-01-24	Richard E. Fisk	Active
FC # 77	NV101302045	NV101302045	1981-03-27	Richard E. Fisk	Active
FC # 78	NV101608683	NV101608683	1988-01-24	Richard E. Fisk	Active
FC # 79	NV101601178	NV101601178	1981-03-27	Richard E. Fisk	Active
FC # 80	NV101731136	NV101731136	1988-01-24	Richard E. Fisk	Active
FC # 81	NV101495815	NV101495815	1988-02-13	Richard E. Fisk	Active
FC # 82	NV101499852	NV101499852	1988-02-13	Richard E. Fisk	Active
FC # 83	NV101459865	NV101459865	1988-02-12	Richard E. Fisk	Active
FC # 84	NV101603419	NV101603419	1988-01-24	Richard E. Fisk	Active
FC # 85	NV101459348	NV101459348	1988-02-12	Richard E. Fisk	Active
FC # 86	NV102521592	NV102521592	1988-02-14	Richard E. Fisk	Active
FC # 87	NV101609018	NV101609018	1988-02-14	Richard E. Fisk	Active
FC # 88	NV101754074	NV101754074	1981-03-28	Richard E. Fisk	Active
FC # 89	NV101402610	NV101402610	1988-02-14	Richard E. Fisk	Active

Claim Name	Serial Number	Lead File	Location Date	Claimant	Status
FC # 90	NV101607077	NV101607077	1988-03-27	Richard E. Fisk	Active
FC # 91	NV101407043	NV101407043	1988-02-14	Richard E. Fisk	Active
FC # 92	NV101508216	NV101508216	1988-03-27	Richard E. Fisk	Active
FC # 93	NV101405524	NV101405524	1988-02-15	Richard E. Fisk	Active
FC # 94	NV101528413	NV101528413	1988-03-27	Richard E. Fisk	Active
FC # 95	NV101408545	NV101408545	1988-03-27	Richard E. Fisk	Active
FC # 96	NV101527227	NV101527227	1988-03-27	Richard E. Fisk	Active
FC # 98	NV101496093	NV101496093	1981-03-28	Richard E. Fisk	Active
FC # 100	NV101758117	NV101758117	1981-03-28	Richard E. Fisk	Active
FC # 107	NV101348253	NV101348253	1981-03-30	Richard E. Fisk	Active
FC # 109	NV101347710	NV101347710	1981-03-30	Richard E. Fisk	Active
FC # 111	NV101348081	NV101348081	1981-03-30	Richard E. Fisk	Active
FC # 113	NV102520704	NV102520704	1981-03-30	Richard E. Fisk	Active
FC # 115	NV101303514	NV101303514	1981-03-29	Richard E. Fisk	Active
FC # 117	NV101349586	NV101349586	1981-03-29	Richard E. Fisk	Active
FC # 119	NV101303093	NV101303093	1981-03-28	Richard E. Fisk	Active
FC # 121	NV101521003	NV101521003	1981-03-28	Richard E. Fisk	Active
FC # 123	NV101522039	NV101522039	1981-03-28	Richard E. Fisk	Active
FC # 125	NV101605890	NV101605890	1981-03-26	Richard E. Fisk	Active
FC # 127	NV101609296	NV101609296	1981-03-26	Richard E. Fisk	Active
FC # 129	NV101490754	NV101490754	1981-03-26	Richard E. Fisk	Active
FC # 131	NV101491593	NV101491593	1981-04-01	Richard E. Fisk	Active
FC # 133	NV101491150	NV101491150	1981-04-01	Richard E. Fisk	Active
FC # 135	NV101405966	NV101405966	1981-04-01	Richard E. Fisk	Active
FC # 137	NV101404794	NV101404794	1981-04-01	Richard E. Fisk	Active
FC # 139	NV101365003	NV101365003	1981-04-01	Richard E. Fisk	Active
Fond Fraction #9	NV101455612	NV101455612	1988-12-12	Richard E. Fisk	Active
Fond Fraction #10	NV101457655	NV101457655	1988-12-12	Richard E. Fisk	Active
Fond Fraction #11	NV101525274	NV101525274	1988-12-12	Richard E. Fisk	Active
Fond Fraction #12	NV101731759	NV101731759	1988-12-12	Richard E. Fisk	Active
Fond Fraction #14	NV101478582	NV101478582	1988-12-12	Richard E. Fisk	Active
Fond Fraction #15	NV101607935	NV101607935	1988-12-12	Richard E. Fisk	Active
FCW 1	NV101382914	NV101382914	2001-12-28	Richard E. Fisk	Active
FCW 2	NV101382915	NV101382915	2001-12-28	Richard E. Fisk	Active
FCW 3	NV101382916	NV101382916	2001-12-28	Richard E. Fisk	Active
FCW 4	NV101382917	NV101382917	2001-12-28	Richard E. Fisk	Active
FCW 5	NV101382918	NV101382918	2001-12-28	Richard E. Fisk	Active
FCW 6	NV101382919	NV101382919	2001-12-28	Richard E. Fisk	Active
FCW 7	NV101382920	NV101382920	2001-12-28	Richard E. Fisk	Active
FCW 8	NV101382921	NV101382921	2001-12-28	Richard E. Fisk	Active
FCW 9	NV101384048	NV101384048	2001-12-28	Richard E. Fisk	Active

Claim Name	Serial Number	Lead File	Location Date	Claimant	Status
FCW 10	NV101384049	NV101384049	2001-12-28	Richard E. Fisk	Active
FCW 11	NV101384050	NV101384050	2001-12-28	Richard E. Fisk	Active
FCW 12	NV101384051	NV101384051	2001-12-28	Richard E. Fisk	Active
FCW 13	NV101384052	NV101384052	2001-12-28	Richard E. Fisk	Active
FCW 14	NV101384053	NV101384053	2001-12-28	Richard E. Fisk	Active
FCW 15	NV101384054	NV101384054	2001-12-28	Richard E. Fisk	Active
FCW 16	NV101384055	NV101384055	2001-12-28	Richard E. Fisk	Active
FCW 17	NV101384056	NV101384056	2001-12-28	Richard E. Fisk	Active
FCW 18	NV101384057	NV101384057	2001-12-28	Richard E. Fisk	Active
FON 3	NV101868691	NV101868691	2013-10-16	Getchell Gold Nevada Inc	Active
FON 4	NV101868692	NV101868692	2013-10-16	Getchell Gold Nevada Inc	Active
FON 5	NV101868693	NV101868693	2013-10-16	Getchell Gold Nevada Inc	Active
FON 6	NV101868694	NV101868694	2013-10-16	Getchell Gold Nevada Inc	Active
FON 9	NV101868695	NV101868695	2013-10-16	Getchell Gold Nevada Inc	Active
FON 12	NV101868696	NV101868696	2013-10-16	Getchell Gold Nevada Inc	Active
FON 15	NV101868697	NV101868697	2013-10-16	Getchell Gold Nevada Inc	Active
FON 17	NV101868698	NV101868698	2013-10-16	Getchell Gold Nevada Inc	Active
FON 18	NV101868699	NV101868699	2013-10-16	Getchell Gold Nevada Inc	Active
FON 19	NV101868700	NV101868700	2013-10-16	Getchell Gold Nevada Inc	Active
FON 20	NV101868701	NV101868701	2013-10-16	Getchell Gold Nevada Inc	Active
FON 21	NV101868702	NV101868702	2013-10-18	Getchell Gold Nevada Inc	Active
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NFC#9	NV101616656	NV101616656	2020-02-08	Getchell Gold Nevada Inc	Active
NFC#10	NV101616657	NV101616657	2020-02-08	Getchell Gold Nevada Inc	Active
NFC#11	NV101616658	NV101616658	2020-02-08	Getchell Gold Nevada Inc	Active
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NFC#14	NV101616661	NV101616661	2020-02-08	Getchell Gold Nevada Inc	Active
NFC#15	NV101617405	NV101617405	2020-02-08	Getchell Gold Nevada Inc	Active
NFC#16	NV101617406	NV101617406	2020-02-08	Getchell Gold Nevada Inc	Active
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NFC#19	NV101617408	NV101617408	2020-02-08	Getchell Gold Nevada Inc	Active
NFC#20	NV101617409	NV101617409	2020-02-08	Getchell Gold Nevada Inc	Active

Claim Name	Serial Number	Lead File	Location Date	Claimant	Status
NFC#21	NV101617410	NV101617410	2020-02-08	Getchell Gold Nevada Inc	Active
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NFC#23	NV101617412	NV101617412	2020-02-08	Getchell Gold Nevada Inc	Active
NFC#24	NV101617413	NV101617413	2020-02-08	Getchell Gold Nevada Inc	Active
NFC#25	NV101617414	NV101617414	2020-02-08	Getchell Gold Nevada Inc	Active
NFC#26	NV101617415	NV101617415	2020-02-08	Getchell Gold Nevada Inc	Active
NFC#27	NV101617416	NV101617416	2020-02-08	Getchell Gold Nevada Inc	Active
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NFC#30	NV101617419	NV101617419	2020-02-08	Getchell Gold Nevada Inc	Active
NFC#31	NV101617420	NV101617420	2020-02-08	Getchell Gold Nevada Inc	Active
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NFC#36	NV101617425	NV101617425	2020-02-08	Getchell Gold Nevada Inc	Active
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FCG 3	NV105830859	NV105830857	2023-03-17	Getchell Gold Nevada Inc	Active
FCG 4	NV105830860	NV105830857	2023-03-17	Getchell Gold Nevada Inc	Active
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FCG 6	NV105830862	NV105830857	2023-03-17	Getchell Gold Nevada Inc	Active
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FCG 12	NV105830868	NV105830857	2023-03-17	Getchell Gold Nevada Inc	Active
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FCG 18	NV105830874	NV105830857	2023-04-15	Getchell Gold Nevada Inc	Active
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FCG 20	NV105830876	NV105830857	2023-04-15	Getchell Gold Nevada Inc	Active
FCG 21	NV105830877	NV105830857	2023-03-16	Getchell Gold Nevada Inc	Active
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FCG 25	NV105830881	NV105830857	2023-03-16	Getchell Gold Nevada Inc	Active

Claim Name	Serial Number	Lead File	Location Date	Claimant	Status
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FCG 27	NV105830883	NV105830857	2023-03-16	Getchell Gold Nevada Inc	Active
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FCG 29	NV105830885	NV105830857	2023-03-16	Getchell Gold Nevada Inc	Active
FCG 30	NV105830886	NV105830857	2023-03-16	Getchell Gold Nevada Inc	Active
FCG 31	NV105830887	NV105830857	2023-02-18	Getchell Gold Nevada Inc	Active
FCG 32	NV105830888	NV105830857	2023-02-18	Getchell Gold Nevada Inc	Active
FCG 33	NV105830889	NV105830857	2023-02-18	Getchell Gold Nevada Inc	Active
FCG 34	NV105830890	NV105830857	2023-02-18	Getchell Gold Nevada Inc	Active
FCG 35	NV105830891	NV105830857	2023-02-18	Getchell Gold Nevada Inc	Active
FCG 36	NV105830892	NV105830857	2023-02-10	Getchell Gold Nevada Inc	Active
FCG 37	NV105830893	NV105830857	2023-02-10	Getchell Gold Nevada Inc	Active
FCG 38	NV105830894	NV105830857	2023-02-10	Getchell Gold Nevada Inc	Active
FCG 39	NV105830895	NV105830857	2023-02-10	Getchell Gold Nevada Inc	Active
FCG 40	NV105830896	NV105830857	2023-02-10	Getchell Gold Nevada Inc	Active
FCG 41	NV105830897	NV105830857	2023-02-10	Getchell Gold Nevada Inc	Active
FCG 42	NV105830898	NV105830857	2023-02-10	Getchell Gold Nevada Inc	Active
FCG 43	NV105830899	NV105830857	2023-02-10	Getchell Gold Nevada Inc	Active
FCG 44	NV105830900	NV105830857	2023-02-10	Getchell Gold Nevada Inc	Active
FCG 45	NV105830901	NV105830857	2023-02-10	Getchell Gold Nevada Inc	Active
FCG 46	NV105830902	NV105830857	2023-02-10	Getchell Gold Nevada Inc	Active
FCG 47	NV105830903	NV105830857	2023-02-18	Getchell Gold Nevada Inc	Active
FCG 48	NV105830904	NV105830857	2023-02-18	Getchell Gold Nevada Inc	Active
FCG 49	NV105830905	NV105830857	2023-02-18	Getchell Gold Nevada Inc	Active
FCG 50	NV105830906	NV105830857	2023-02-18	Getchell Gold Nevada Inc	Active
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FCG 60	NV105830910	NV105830857	2023-02-10	Getchell Gold Nevada Inc	Active
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FCG 63	NV105830913	NV105830857	2023-02-10	Getchell Gold Nevada Inc	Active
FCG 64	NV105830914	NV105830857	2023-02-10	Getchell Gold Nevada Inc	Active
FCG 65	NV105830915	NV105830857	2023-02-10	Getchell Gold Nevada Inc	Active
FCG 66	NV105830916	NV105830857	2023-02-10	Getchell Gold Nevada Inc	Active

Claim Name	Serial Number	Lead File	Location Date	Claimant	Status
FCG 67	NV105830917	NV105830857	2023-02-10	Getchell Gold Nevada Inc	Active
FCG 68	NV105830918	NV105830857	2023-02-10	Getchell Gold Nevada Inc	Active
FCG 69	NV105830919	NV105830857	2023-02-10	Getchell Gold Nevada Inc	Active
FCG 70	NV105830920	NV105830857	2023-02-10	Getchell Gold Nevada Inc	Active
FCG 71	NV105830921	NV105830857	2023-02-10	Getchell Gold Nevada Inc	Active
FCG 76	NV105830922	NV105830857	2023-04-17	Getchell Gold Nevada Inc	Active
FCG 77	NV105830923	NV105830857	2023-04-17	Getchell Gold Nevada Inc	Active
FCG 78	NV105830924	NV105830857	2023-04-17	Getchell Gold Nevada Inc	Active
FCG 79	NV105830925	NV105830857	2023-04-17	Getchell Gold Nevada Inc	Active
FRAC 1	NV106372364	NV106372364	2024-04-16	Getchell Gold Nevada Inc.	Filed
FRAC 2	NV106372365	NV106372364	2024-04-16	Getchell Gold Nevada Inc.	Filed
FRAC 3	NV106372366	NV106372364	2024-04-16	Getchell Gold Nevada Inc.	Filed
FRAC 4	NV106372367	NV106372364	2024-04-16	Getchell Gold Nevada Inc.	Filed
Extension 1	NV106372360	NV106372360	2024-04-16	Getchell Gold Nevada Inc	Filed
Extension 2	NV106372361	NV106372360	2024-04-16	Getchell Gold Nevada Inc	Filed
Extension 3	NV106372362	NV106372360	2024-04-16	Getchell Gold Nevada Inc	Filed

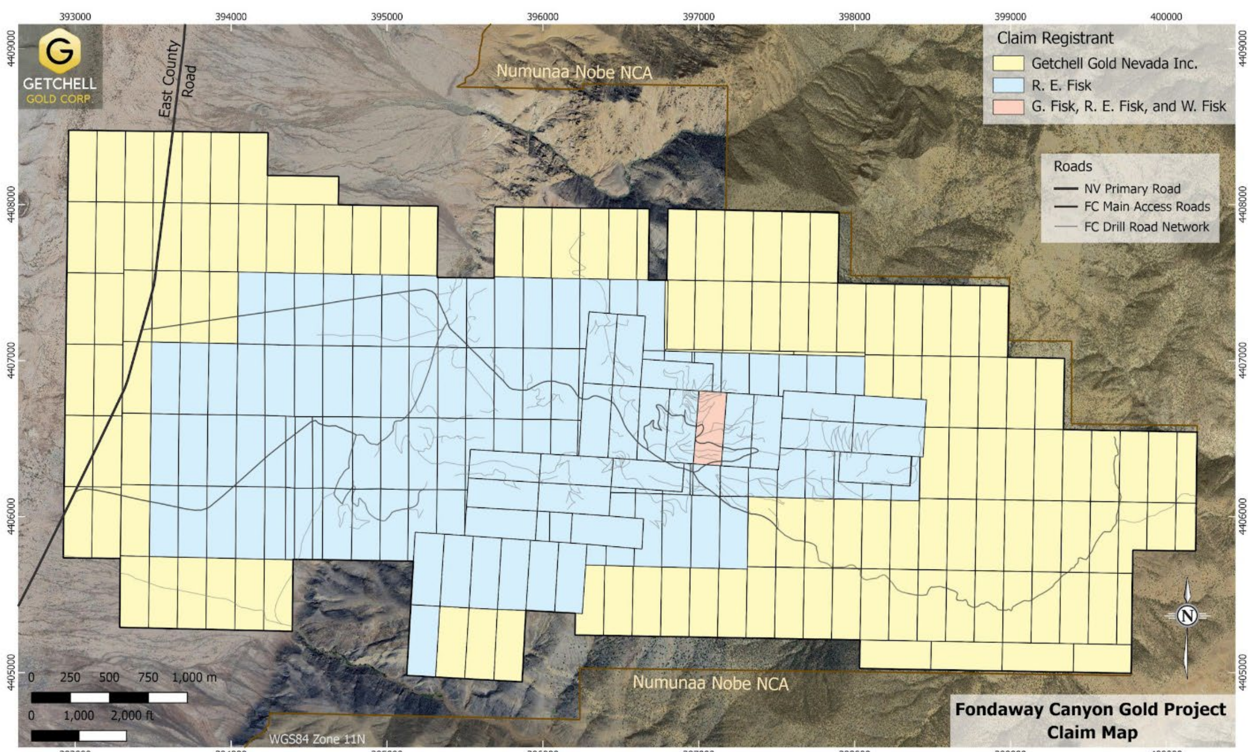


Figure 4-2: Fondaway Canyon Property Claims

(Source: Getchell Gold, 2024)

4.3 Royalties and Agreements

The claims are currently held by Getchell Gold under a Mining Lease/Purchase Agreement with the owner, Richard E. Fisk. The Property is subject to a Net Smelter Royalty (NSR) of 3% to Richard Fisk that can be purchased for US $600,000. An advance payment of US $35,000 is made by the project operator every year and counted towards the royalty purchase. To date, a total of US $420,000 has been paid towards the Fisk royalty purchase. Upon fulfillment of the royalty to Fisk, a 2% NSR held by Canagold Resources Ltd. will be triggered. Getchell Gold can purchase half of this royalty for US $1.0 million. An additional 2% NSR is held by Hale Capital, this royalty can be purchased for US $2.0 million. A fee of $200 per claim is payable to the BLM before September 1 each year, and $12 per claim and $18 per filing is payable to Churchill County by November 1st each year.

4.4 Environmental Liabilities and Permitting

At the execution of a definitive option agreement to acquire 100% of the Fondaway Canyon Project from Canarc Resource Corp., now known as Canagold Resources Ltd., on January 3, 2020 (Canagold Option), the Fondaway Canyon Property was encroached on three sides by the Stillwater Wilderness Study Area. As of December 23rd, 2022, following the passage of the National Defense Authorization Act (NDAA), the Stillwater Wilderness Study Area was released. The Numunaa Nobe National Conservation Area (NCA) (Figure 4-2) was established with a reduced footprint. The newly established boundaries of the NCA formalized in the NDAA opened additional area for exploration and mining around the existing claim group. Subsequent to the establishment of the NCA, Getchell Gold expanded the claim package through the staking of additional mining claims, the FCG group of claims, to the North, East, and South up to the boundary of the NCA. Of note, the NCA does not infringe on the mining claims, does not limit expansion of the mineral resource, and allows ample area in support of potential future mining activities such as those associated with an open pit operation envisioned herein.

Exploration, including drilling, is being carried out under an existing 5-acre Surface Management Notice disturbance permit (NVN95628). The reclamation bond is currently set at US$22,619. Reclamation of the drill pads from the 2020-2022 exploration programs are still pending at the time of this report. A number of small historical open pit excavations exist on the Property along with some minor dumps and equipment.

4.5 Other Significant Factors and Risks

The QP is not aware of any environmental liabilities affecting the Property. There are no other significant factors and risks that may affect the access, title, or the right or ability to perform work on the Project that are not discussed in this Report.

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

5.1 Accessibility

The Fondaway Canyon Property is located on the western flank of the Stillwater Range in northwestern Nevada, 140 km northeast of Reno, NV, and 58 km northeast of Fallon in Churchill County.

The Property is accessed from Fallon, Nevada east on Highway 50 and then north on Highway 116 to the settlement of Stillwater and then north on the gravel East County Road 30 miles (50 km) along the front range of the Stillwater Range to the mouth of Fondaway Canyon.

Existing mine roads provide access into the canyon and there remains a dense network of drill roads developed over decades of exploration and mining in various states of use.

5.2 Topography, Elevation and Vegetation

The Property sits at an elevation ranging from 5,000 to 6,000 ft (1,500 m to 1,830 m) above sea level (“asl”). Access east into Fondaway Canyon is at a gentle grade with the north and south slopes variably steep but adequate with existing mine or drill roads.

The terrain in the immediate vicinity consists of variably steep rounded hills and overall rugged mountainous ridges with no discernible timber line. Water is scarce and regional elevation ranges from 3,458 ft (1,053 m) to 7,414 ft (2,260 m) asl. The Stillwater Range was subject to a detailed ecological and wilderness review as part of the regional (Stillwater) Wilderness Study Area (WSA) inventory for which there is detailed information available. Recent legislation has seen the Stillwater WSA released and a conservation area created with mining rights preserved.

Vegetation types range from pinyon-juniper and juniper, sage types in the higher elevations, sagebrush and grass types at moderate elevations, and scrub and greasewood types in the valley bottoms. Poisonous plants that are known to occur in limited quantities in the North Stillwater Range HMA are deathcamas, larkspur, loco weed, alogeton, halogeton, and horsebrush. These species appear in limited quantities throughout the range.

5.3 Climate

Winters are cold and summers are hot with little rainfall. The area is considered a cold desert because winter temperatures fall below freezing. Operations can be completed year-round. Any future mining activity would also be year-round. The average monthly weather in Fallon, Nevada is shown in Table 5-1.

Table 5-1: Average Monthly Weather – Fallon, NV

	January	February	March	April	May	June	July	August	September	October	November	December
Avg. Temperature °C / °F	1.2 °C	3.9 °C	8.4 °C	11.9 °C	17.3 °C	23.4 °C	28.2 °C	26.6 °C	21.6 °C	13.6 °C	6.2 °C	0.9 °C
	34.2 °F	39 °F	47.2 °F	53.4 °F	63.2 °F	74.1 °F	82.8 °F	79.9 °F	70.9 °F	56.5 °F	43.2 °F	33.6 °F
Min. Temperature °C / °F	-3.1 °C	-1.2 °C	2.2 °C	5.4 °C	10.4 °C	15.7 °C	20 °C	18.5 °C	14.1 °C	7.1 °C	1.1 °C	-3.1 °C
	26.4 °F	29.8 °F	36 °F	41.7 °F	50.7 °F	60.3 °F	68 °F	65.3 °F	57.4 °F	44.8 °F	34 °F	26.4 °F
Max. Temperature °C / °F	7.1 °C	10.4 °C	15.5 °C	19.1 °C	24.4 °C	30.7 °C	35 °C	33.5 °C	28.7 °C	20.7 °C	12.5 °C	6.4 °C
	44.9 °F	50.7 °F	59.9 °F	66.4 °F	75.9 °F	87.2 °F	95 °F	92.2 °F	83.6 °F	69.2 °F	54.5 °F	43.5 °F
Precipitation mm / in	25 mm	21 mm	22 mm	18 mm	20 mm	9 mm	2 mm	2 mm	6 mm	12 mm	15 mm	21 mm
	1.0 in	0.8 in	0.9 in	0.7 in	0.8 in	0.4 in	0.1 in	0.1 in	0.2 in	0.5 in	0.6 in	0.8 in

(Source: climate-data.org, Data 1991 – 2021)

5.4 Local Resources and Infrastructure

There are no public utilities, including electrical power on the Property. Two permitted water wells are present on the Property, with water available for mining use under the lease agreement, permitted for 24 million gallons per year, renewed annually.

The closest significant communities are Reno, Nevada 140 km to the west-southwest, Lovelock, Nevada 78 km to the northwest, and Fallon 58 km to the southwest. Fallon is the county seat with above normal resources for the area (e.g. supplies and accommodations) primarily due to the contribution of the Fallon Naval Air Station and the generous agricultural setting as a draw and support for the region.

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

6. HISTORY

6.1 History of Ownership and Operators

The initial lode mining claims of the Fondaway Canyon Property were staked in 1956 by George and Richard Fisk. Occidental Minerals (Occidental) optioned the claims from the Fisks in 1980 and staked surrounding claims that covered much of the identified mineralization. Occidental conducted exploration between 1980 and 1982 while the Fisks continued small volume mining.

Tundra Gold Mines (Tundra) acquired the Occidental option in 1983 and joint-ventured the property in 1984 with New Beginnings Resource Corp. (New Beginnings). Homestake Mining Company sub-leased the property from 1984 to 1985. In 1985, Mill Creek Mining took over, followed by Tenneco Minerals whom leased the property from 1986 to1996 and increased the property size to 647 unpatented mining claims. Consolidated Granby leased the property from 1996 to 1997, with no significant exploration activity and Stillwater Gold leased the property in 1999.

Nevada Contact Inc (NCI), a subsidiary of Agnico Eagle, leased the property from 2001 to 2002, then Royal Standard Minerals leased the property from 2003 to 2013. In 2013, the lease was acquired by American Innovative Minerals (AIM) from Royal Standard. Aorere Resources Limited obtained an option to purchase the AIM properties in February 2016, which expired at the end of January 2017. Canarc Resource Corp. acquired the Fondaway Canyon Property along with substantially all of the mineral properties held by AIM in March 2017.

Getchell Gold entered into a definitive option agreement to acquire 100% of the Fondaway Canyon Project from Canarc Resource Corp., now known as Canagold, on January 3, 2020 (Canagold Option). Under the terms of the Canagold Option, Getchell Gold acquired 100 percent of the Fondaway Canyon Project on December 29, 2023, by satisfying the Canagold Option earn-in requirements comprising work commitments totaling US$1.45 million, paying Canagold a total of US $2 million in cash, and issuing US$2 million in Getchell Gold shares.

Getchell Gold's ownership remains subject to the Mining Lease/Purchase Agreement with the underlying owner, Richard Fisk, and fulfillment of the outstanding NSR royalty payments (refer to Section 4.2). The Fisk family has continuously owned the core mining claims to the present day.

6.2 Exploration and Development Work Conducted by Previous Owners

The initial lode mining claims of the Fondaway Canyon Property were staked in 1956 by George and Richard Fisk. Approximately 10,000 tons of tungsten mineralization were mined by the Fisks, recovering 200,000 lbs of tungsten trioxide (WO_3). The Fisks also produced 47 flasks of mercury and three tons of antimony during this period. Later, the Fisks discovered gold at Fondaway Canyon in the mid 1970's and produced approximately 2,500 ounces of gold from shallow, oxide material from about 1977 to 1983 (Norred and Henderson, 2017).

Figure 6-1: Fisk Crusher and Vat Leach Operation

(Source: Getchell Gold, 2021, Photos from Tundra Gold Mines, Akright, 1983)

Occidental Minerals optioned the property from 1980 to 1982 and explored the property while the Fisks continued small volume mining. Occidental conducted extensive geological, geochemical, and geophysical surveys over the area which identified disseminated gold mineralization hosted within select argillite horizons and tungsten mineralization in scheelite veins (Oliver 1982; Akright, 1983). Occidental Minerals drilled 15 reverse circulation (RC) holes in 1981 and 3 core holes in 1982.

Between 1983 and 1984 Tundra Gold Mines Ltd. conducted several miles of VLF-EM and magnetometer surveys, and completed mapping, surface grab sampling and channel sampling largely focused over the Central area of the Property. Tundra identified least 27 anomalies, labeled “A” through “V” (Scott, 1983). Tundra drilled 29 core holes in 1983. The New Beginnings/Tundra joint-venture drilled 18 RC holes and 6 core holes in 1984.

Homestake Mining Company sub-leased the property between 1984 and 1985. Homestake sampled the underground workings on the property, and commissioned mineralogy and petrographic studies, as well as metallurgical testing. They drilled 4 core holes. Mill Creek Mining (Mill Creek) took over in 1985. Mill Creek drilled 69 RC holes, totaling 6,805 feet, and drilled numerous, shallow percussion holes. They mined near-surface oxide ore, and attempted vat leach processing, with no significant recoveries.

Tenneco Minerals leased the property from 1986 to1996. They increased the property size to 647 unpatented claims, and took thousands of rock, soil, and stream sediment samples. Tenneco drilled over 500 RC holes, totaling 130,000 ft (~40,000 m) of drilling. They drove an adit with 540 ft of workings to take bulk samples of the mineralized Half Moon zone. They commissioned extensive metallurgical testing at Hazen Labs, showing over 85% recovery for oxide material.

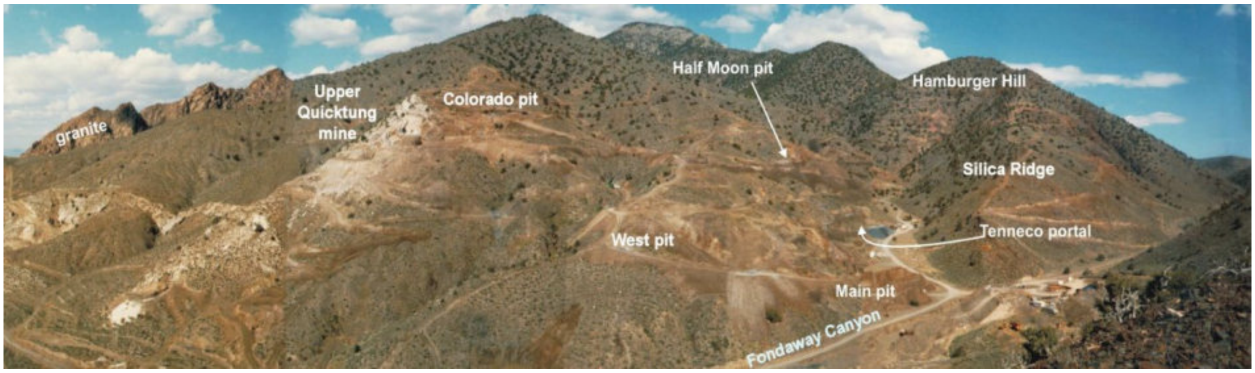


Figure 6-2: Panorama North Slope Fondaway Canyon Circa 1989

(Source: Getchell Gold, 2020, modified from Strachan, 2003)

Tenneco built a 1,500 tons per day (tpd) heap leach with a 230 gallons per minute (gpm) Merrill-Crowe processing plant. Tenneco mined the South Mouth, Reed Pit, Paperweight and Halfmoon. From August 1989 through August 1990, they mined and processed 186,000 tons of material, and recovered 5,402 ounces of gold, with a reported 87% average recovery (Cohan, 1997). Tenneco completed final reclamation of their mining and processing area areas in 2004.

Figure 6-3: Tenneco Porta and Heap Leach Pad

(Source: Getchell Gold, 2020, Photos from Williams, 2005, and Tenneco Minerals Company, 1990)

Consolidated Granby leased the property from 1996 to 1997, with no significant exploration activity. Stillwater Gold leased the property in 1999 and conducted extensive field mapping and sampling. The detailed mapping and geological interpretation by Michael Brady for Stillwater (Brady, 1997) are the basis for much of the work by later companies, including the Resource modeling reported in Norred and Henderson (2017).

Nevada Contact Inc (NCI), a subsidiary of Agnico Eagle, leased the property from 2001-2002. They organized the previously-collected data into a GIS and geological database. The compiled database contained 2,451 rock chip samples, 457 soil samples, and 146 stream sediment samples. Nevada Contact drilled 3 RC holes and 8 RC/Core holes, totaling 5,335 ft of RC and 6,317 ft of core.

Royal Standard Minerals leased the property from 2003 to 2013, with little reported exploration activity. The technical report commissioned by Royal Standard mentioned the 2002 Nevada Contact drilling, but did not incorporate the drilling results into their historical resource model (Strachan, 2003). The lease was acquired by American Innovative Minerals (AIM) from Royal Standard in 2013. AIM compiled previous drill holes and samples into a GIS database. They collected and assayed more than 250 rock chip samples, as well as grab samples from stockpiles, dumps, and the leach pad. AIM conducted metallurgical tests on the stockpile material and on the tungsten mineralization, in order to evaluate the economics of selling the material.

Aorere Resources Limited (Aorere) obtained an option to purchase the AIM properties in February 2016, which expired at the end of January 2017. Aorere commissioned a Scoping Report (Norred, 2016). They sampled the 2002 core and sent six representative samples to Applied Petrologic Services & Research (APSAR) for detailed petrologic studies (Coote, 2016). Additional core samples were selected and submitted to McClelland Laboratories for metallurgical testing (McPartland, 2017). Aorere contracted Techbase International to compile and validate the drilling and other data from the property, and to produce a resource estimate. The Mineral Resource Estimate that is the subject of the Norred and Henderson (2017), report was originally produced for Aorere. New drilling has been completed since the 2017 Mineral Resource Estimate was completed and therefore it is considered historical in nature and is superseded by the resource estimate presented as part of this Technical Report.

Canarc Resource Corp. (now Canagold) acquired the Fondaway Canyon Property in March 2017. Work included geological mapping, rock-chip sampling, a ground magnetics survey (Figure 6-4), a topographic survey, drilling seven deep core holes and radiometric dating. Interpretation of Canagold's ground magnetics survey data was integrated with the geological information to refine the property geology. Norred and Henderson (2017) reported a Mineral Resource Estimate for the Property that is now considered historical in nature as discussed in Section 6.5.

A total of 2,943 rock chip samples have been collected by the historical property operators to date. The results from the analyzed chip samples are provided in Figure 6-5.

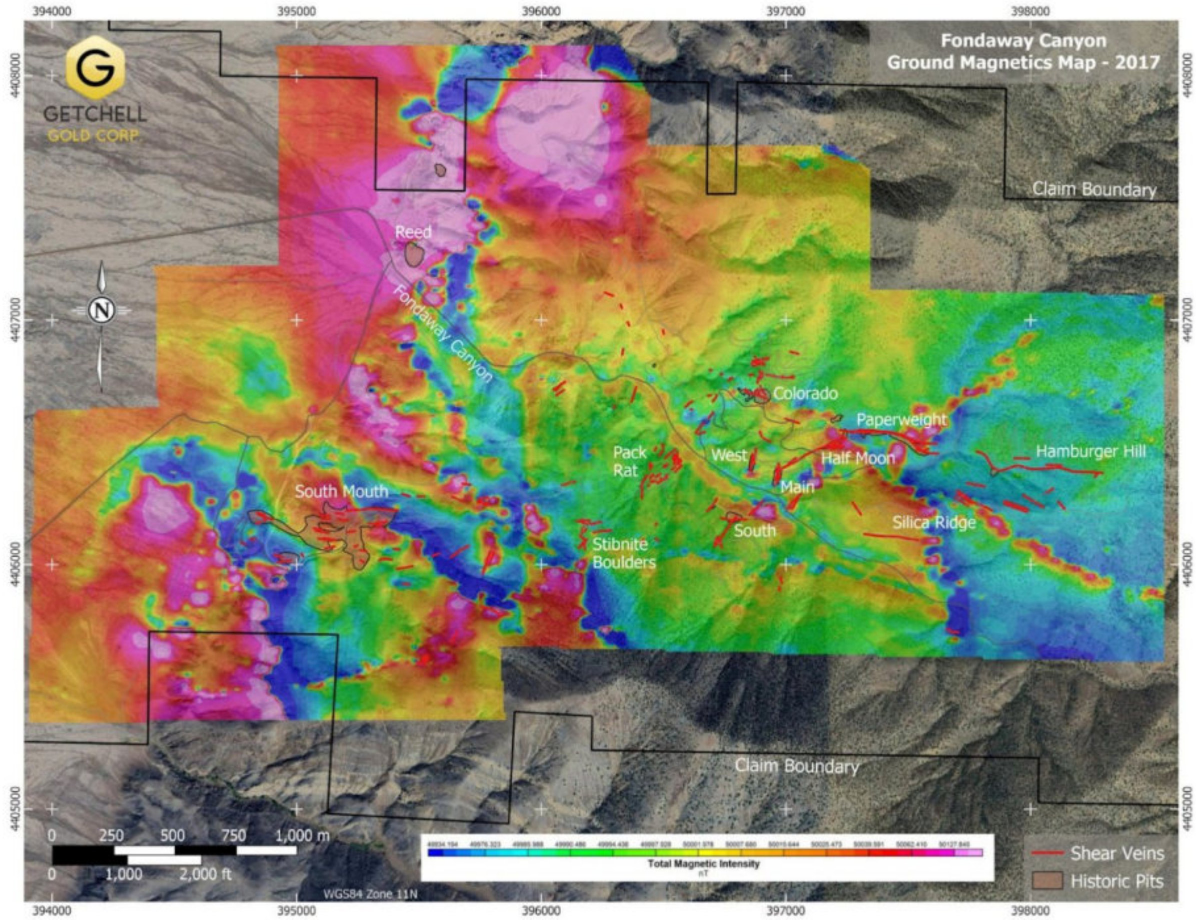


Figure 6-4: Total Field Magnetics, Ground Survey in 2017

(Source: Getchell Gold, 2024)

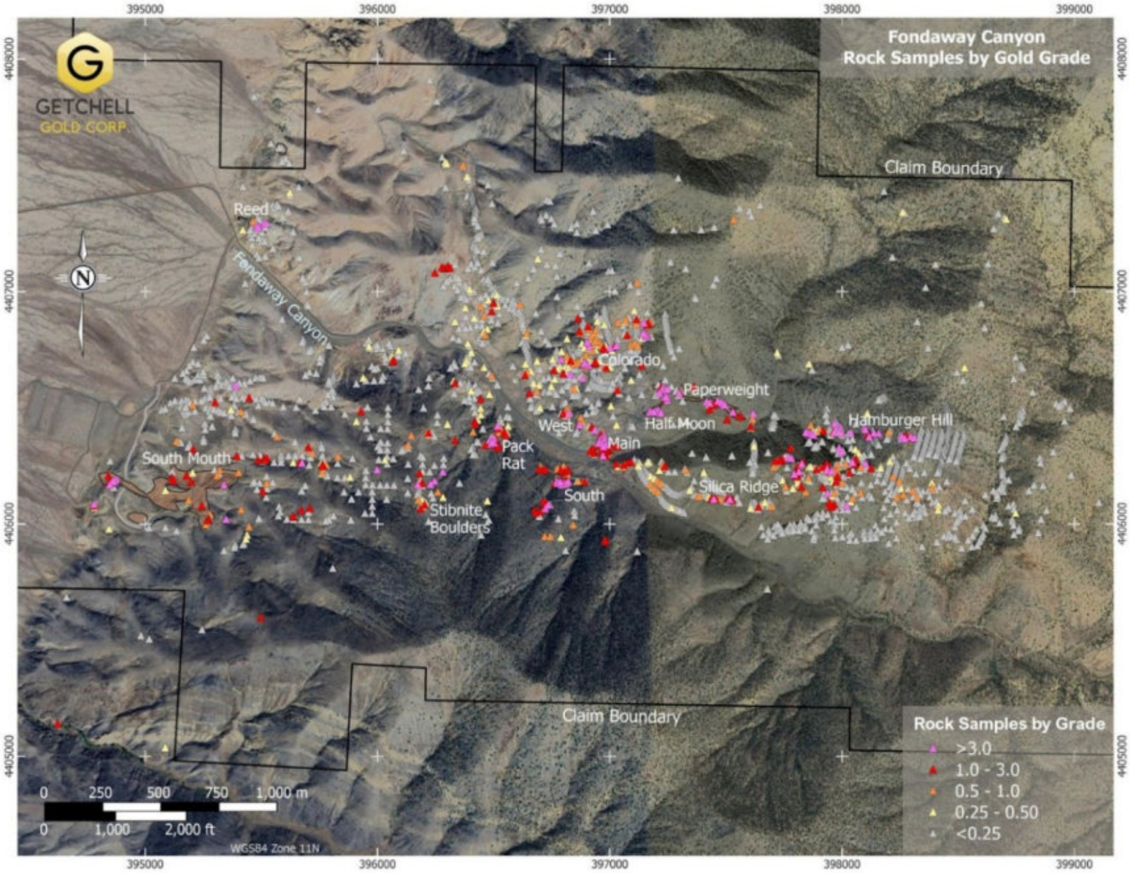


Figure 6-5: Compilation of Historical Rock Chip Sampling Results

(Source: Getchell Gold, 2024)

6.3 Historical Drilling

A total of 735 drill holes totaling over 63,800 m have been completed on the Fondaway Canyon Property between 1981 and 2017 by various operators (Table 6-1 and Figure 6-6).

Table 6-1: Drilling Programs at Fondaway Canyon

		RC Drilling		Core Drilling	
Year(s)	**Company**	**Holes**	**Meters**	**Holes**	**Meters**
1981-1982	Occidental	15	>1,409.4*	3	>121.9
1983	Tundra			29	4,644.0
1984	New Beginnings/Tundra	18	616.3	6	938.9
1984-1985	Homestake			4	780.6
1985	Mill Creek	69	2,074.2		
1987-1996	Tenneco	573	>37,149.0*		
2002	Nevada Contact	3	783.3	8	2769.4
2017	Canagold (Canarc)			7	2533.7
	Total	**678**	**42,032.2**	**57**	**11,788.5**

**Total depth was not available for all drill holes, meterage represents a minimum total of meters drilled.*

The majority of the drilling completed on the Property has been reverse circulation (RC) totaling 678 RC drill holes for over 42,000 m with the balance comprising 57 core drill holes totaling over 11,790 m. The historical drilling extended along 3.5 kms of the east-west gold trend and predominantly targeted two prospective areas: the South Mouth Area on the western extent and the Central Area encompassing the Colorado, Paperweight, Half Moon, Main Pit, South Pit, and Pack Rat mineralized zones.

6.3.1 Historical Drilling 1981-1996

Occidental Minerals drilled 15 reverse circulation (RC) holes in 1981 and 3 core holes in 1982, totaling 1,784.9 m (5,856 feet) of drilling. Drilling was completed by Eklund Drilling Co. (Ekland). Drilling targeted mineralized veins and disseminated mineralization. Drill holes targeting the veins intersected 0.234 opt Au over 9 m (30 feet) of 0.234 opt Au. Drill holes targeting disseminated mineralization intersected 0.018 opt Au over 54 m (180 feet) (Oliver 1982).

Tundra drilled 29 core holes in 1983 totaling 4,644 m (15,236.2 feet). Drilling was completed by the Boyles Brothers Drilling Company and Coates Drilling using HQ sized rigs. In 1984 New Beginnings/Tundra drilled 18 RC holes totaling 616.3 m (2,020 feet) and 6 core holes totaling 938.9 m using Boyles Brothers Drilling Company. Core holes were completed using a HQ sized rig. The drill programs resulted in the partial delineation of seven gold-bearing zones on the Property. The zones were delineated over a strike length of 1.6 km (Descarreaux, 1984).

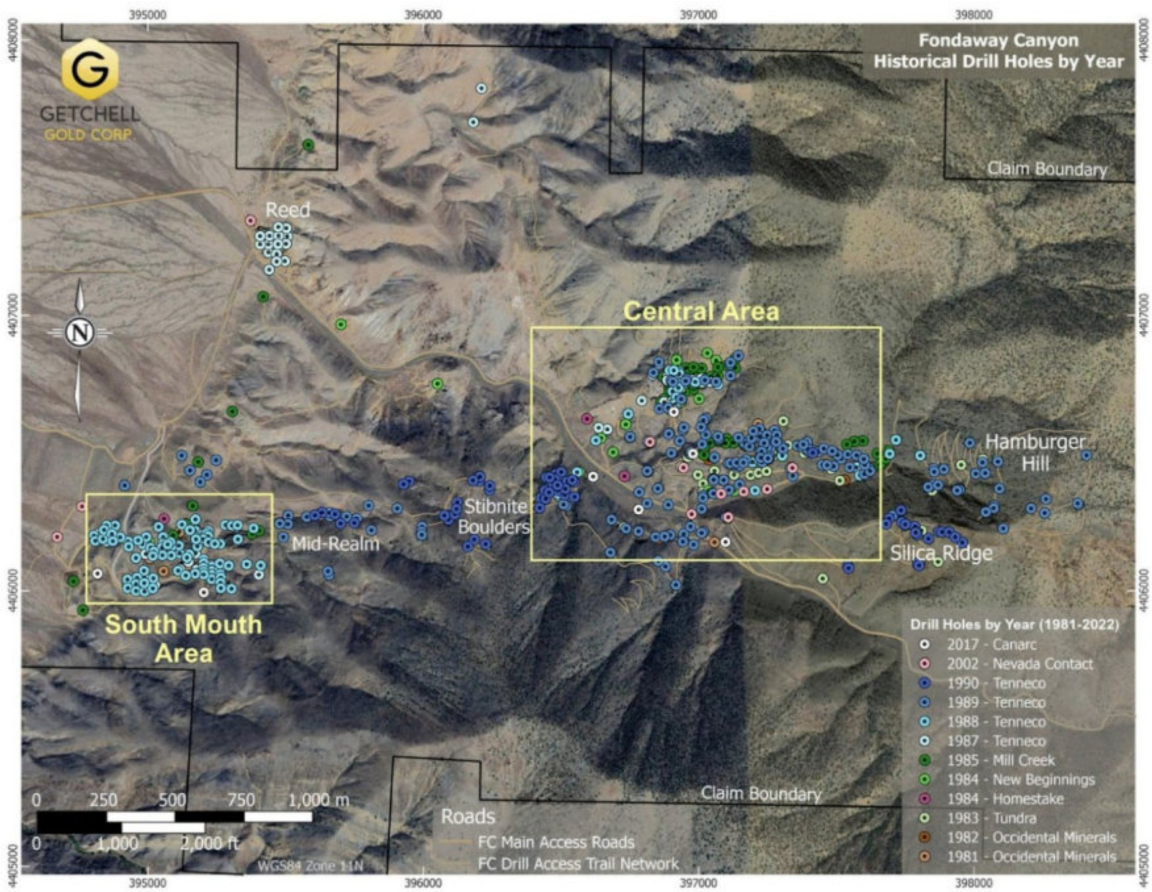


Figure 6-6: Historical Drill Holes Over the Fondaway Canyon Property

(Source: Getchell Gold, 2024)

In 1984-1985 Homestake drilled 4 HQ-sized core holes totaling 780.6 m (2,561 feet). Three holes targeted the westward extension of the gold mineralization in the Central target area, all holes intersected gold mineralization. A single hole followed up on gold mineralization intersected by Occidental at the Range Front target / South Mouth area, no gold mineralization was intersected by this hole (Homestake, 1984).

In 1985 New Beginnings / Mill Creek drilled 69 vertical, percussion drill holes totaling 2,074.2 m (6,805 ft).

Between 1987 and 1996 Tenneco completed an extensive drilling program targeting shallow disseminated mineralization as well as deeper mineralization that was vein hosted. Tenneco completed over 570 RC drill holes on the Property. No issues were reported by Tenneco with respect to drilling in the mineralized zones. Variable information is available for the Tenneco drill holes. Tenneco used a number of different companies for their programs. The majority of the drilling was completed by Ponderosa Drilling (67 holes), other drilling companies used by Tenneco include C&L Drilling Co., Rough Country Contracting, Drift and Dateline. Total depth records are available for 573 holes which indicate total drilling of at least 37,149 m (121,880 ft). Based on the favorable results from their drill programs Tenneco constructed a plant for processing near surface mineralization (Cohan, 1997; Norred and Henderson, 2017).

6.3.2 Historical Drilling 2002-2017

Nevada Contact drilled 8 core holes totaling 2,769.3 m (9,085.6 feet) to test the down dip extension of known mineralization in the Half Moon, Paperweight and Deep Dive areas. Three RC holes were also completed totaling 783.3 m (2,570 feet) to test blind exploration targets along the Range Front fault and potential extensions of known mineralization in the South Mouth and Reed Pit areas. All the core holes were “pre-collared” with the RC rig to expedite the program. Nevada Contact used Ekland to complete the drill program (Nevada Contact, 2002). During the program Ekland was acquired by Boart Longyear.

In 2017, Canarc drilled 7 HQ core holes targeting the Pack Rat zone at depth, the Colorado area, the Half Moon Zone, the South Pit and the South Mouth area. Nevada Contact used IDEA Drilling to complete the program.

6.3.3 West Area Drill Summary

The West Area contains several prospective targets (Figure 6-7). The Pediment Target is the westernmost known gold occurrence along the 3.5 km long E-W trending Fondaway Canyon gold mineralization corridor. Two of Nevada Contact’s RC holes, 02FC-10 and 02FC-11, targeted the Pediment area, west of the South Mouth area. The Pediment target area is on trend with the South Mouth gold bearing shear zone and is located west of the Range Front fault that is situated at the western margin of the Stillwater range. Both of these vertical holes, 185 m apart and 100-150 m onto the Pediment, intersected zones of low-grade mineralization within limestone host rocks. Drill hole 02FC-10 intersected 27.4 m returning an average assay of 0.82 g/t Au between 256.0 m to 283.5 m. Drill hole 02FC-11 intersected 36.6 m returning an average assay of 0.52 g/t Au between 179.8 m and 216.4 m (Strachan, 2003). Drill hole 02FC-6 targeted the Reed Pit mineralization located 1.2 km to the north. The drill hole was terminated at 175 m due to slow penetration in the silicified carbonate rocks and failed to intersect anomalous gold values.

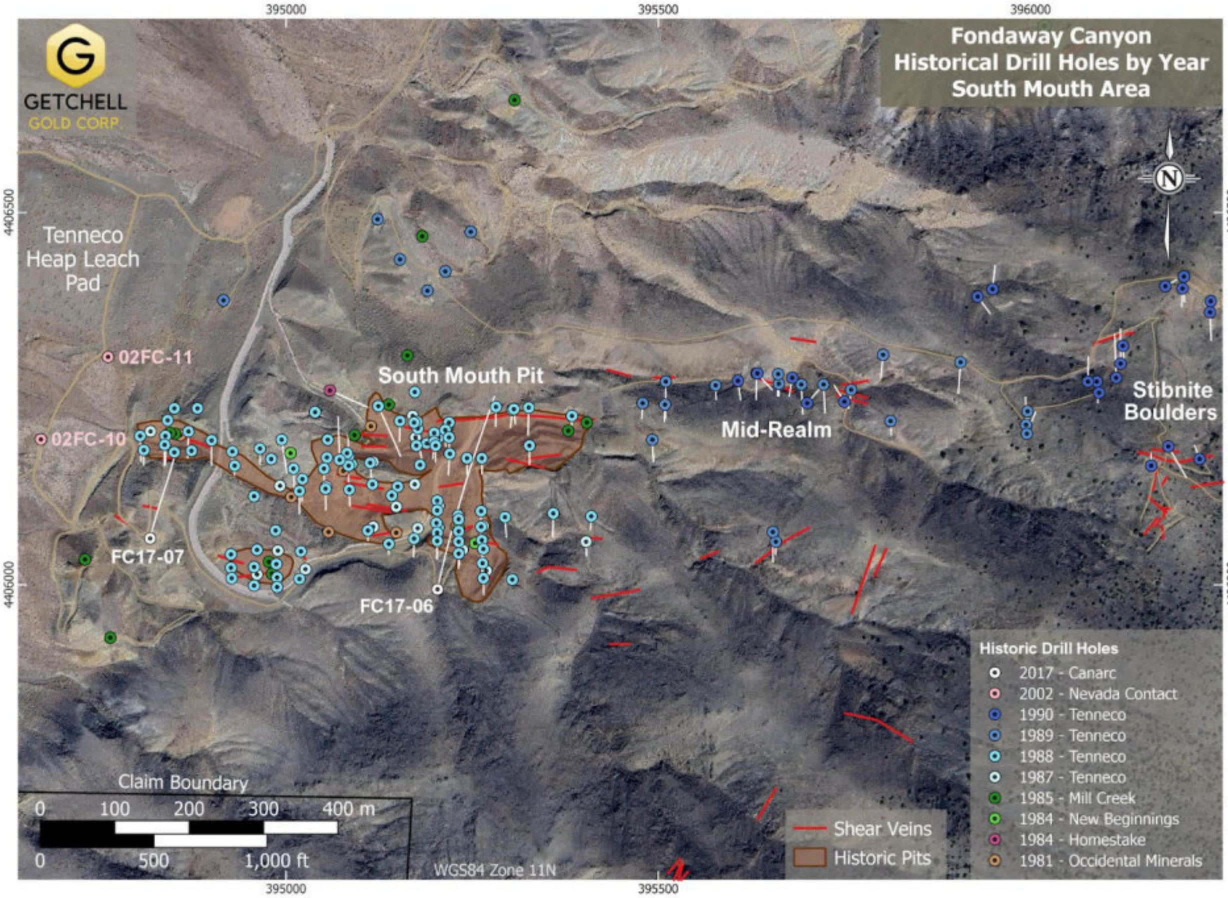


Figure 6-7: Historical Drill Hole Locations West Area

(Source: Getchell Gold, 2024)

The South Mouth area was the site of small-scale open-pit mining in the late 1980's. The gold mineralization at South Mouth occurs within a 300 m wide, east striking, steeply dipping shear zone, hosting shear-type veins within a broader disseminated lower grade halo. The historical drilling was quite shallow and primarily tested the near surface mineralization in support of the open pit operation.

The eastern part of the South Mouth open pit area was tested by Canarc's drill hole FC17-06. Four zones of low-grade gold mineralization returning assays between 0.4 to 0.7 g/t Au, over intersections of 4 to 10 m in length were intersected in the upper parts of the hole. Consistent gold mineralization averaging 1.29 g/t Au over the last 6.1 m, from 364.5 m to 370.6 m was intersected at the bottom of the hole. The mineralization intersected by drill hole FC17-06 is located 200 m west of, and on trend with, the Mid-Realm zone. Mineralization in the area remains open in all directions.

The western part of the South Mouth area was tested by Canarc's core drill hole FC17-07. The hole was collared 400 m west of drill hole FC17-06 targeting mineralization below the vein-stockwork zone evident in the pit. The hole was abandoned before it reached the targeted mineralized zone due to drilling difficulties caused by broken ground within a shear zone. An interval of stockwork quartz veins, intersected near the bottom of the hole between a depth of 161.8 m and 167.0 m, returned an average assay of 2.06 g/t Au over 5.2 m including 6.0 g/t Au over 1.2 m.

Assay highlights from the West Area for drill results from the 2002 and 2017 drill programs are provided in Table 6-2.

Table 6-2: Highlights of West Area Drill Results – 2002 to 2017

Zone	Drill Hole	Au (g/t)	Interval (m)	Depth From (m)	Depth To (m)	Depth From (ft)	Depth To (ft)
South Mouth	**FC17-06**	1.29	6.1	364.5	370.6	1,195.9	1,215.9
	FC17-07	2.06	5.2	161.8	167.0	530.8	547.9
Pediment	**02FC-10**	0.82	27.4	256.0	283.5	839.9	930.1
	including	1.07	18.3	256.0	274.3	839.9	899.9
	02FC-11	0.52	36.6	179.8	216.4	589.9	710.0
	including	0.62	21.3	195.1	216.4	640.1	710.0

**Note: True width can vary from 50% up to 100% of core length depending upon drill hole intersection angles*

6.3.3.1 Central Area Drill Results – 2002 to 2017

In the Central Area, Nevada Contact completed 8 core holes totaling 2,769 m (9,085 ft) to test the down-dip extensions of known mineralization (Figure 6-8). Six of the holes intersected mineralization considered to be associated with the Half Moon and Paperweight veins at depth, with drill holes 02FC-04 and 05 returning the higher gold intercepts. Assay highlights are presented in Table 6-3. Canarc drilled 7 holes in the area totaling 2,533.7 m.

Table 6-3: Highlights of Central Area Drill Results – 2002 to 2017

Zone	Drill Hole	Gold g/t	Interval	Depth From (m)	Depth To (m)	Depth From (ft)	Depth To (ft)
Paperweight	**02FC-04**	4.20	16.70	265.2	281.9	870.0	925.0
Pack Rat	**FC17-01**	1.29	4.63	319.1	365.8	1,047.0	1,200.0
	including	2.10	7.01	319.1	326.6	1,047.0	1,071.5
	including	1.56	26.97	332.6	359.6	1,091.3	1,179.8
Colorado	**02FC-01**	1.46	49.07	172.2	221.3	565.0	726.0
	FC17-02	2.08	21.64	189.3	210.9	621.0	692.0
	FC17-02	1.77	62.94	253.1	316.0	830.5	1,036.8
	FC17-03	2.83	65.83	122.7	188.1	402.5	617.0
	including	7.69	9.75	154.5	164.3	507.0	539.0
	including	5.28	7.92	180.1	188.1	591.0	617.0
Halfmoon	**02FC-05**	1.88	59.44	174.7	234.1	573.0	768.0
	including	4.70	16.80	217.3	234.1	712.9	768.0
Halfmoon	**FC17-04**	1.01	66.14	226.2	292.3	742.0	959.0
	including	1.36	10.67	226.2	236.8	742.0	777.0
	including	1.98	21.03	267.9	289.0	879.0	948.0
	FC17-04	5.91	3.72	333.8	337.5	1,095.0	1,107.2
South Pit	**FC17-05**	6.55	2.44	320.7	323.1	1,052.0	1,060.0
	FC17-05	3.37	3.96	334.4	338.3	1,097.0	1,110.0
	FC17-05	3.48	12.80	345.3	358.1	1,133.0	1,175.0
	including	5.97	6.10	345.4	353.6	1,133.0	1,160.0

**Note: True width can vary from 50% up to 100% of core length depending upon drill hole intersection angles*

Drill hole 02FC-01 was drilled to test the down-dip extension of the Colorado vein system. The hole intersected 49.1 m core length averaging 1.5 m g/t Au from 172.2 m to 221.3 m.

Drill hole 02FC-04 was drilled to test the down-dip extension of the Paperweight vein system. It encountered an anomalous intercept of 4.2 g/t Au over 16.7 m between 265.2 m and 281.9 m. This core length intercept is significantly deeper than intersected by previous drilling in the area.

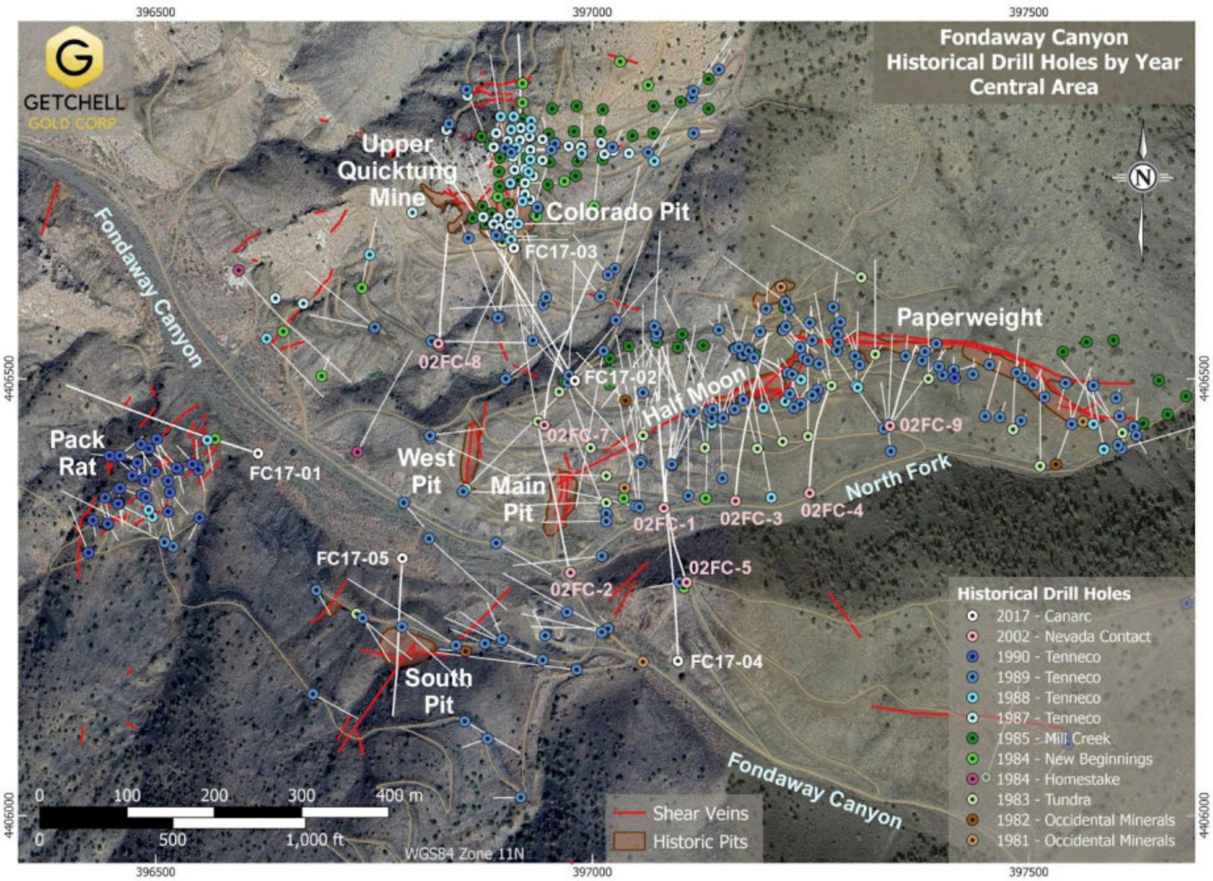


Figure 6-8: Historical Drill Hole Locations in the Central Area

(Source: Getchell Gold, 2024)

Drill hole 02FC-05 targeted the intersection of the NE-SW trending Half Moon vein system with the N-S trending, east dipping fault exposed in the Main Pit. The hole intersected 59.4 m core length averaging 1.9 m g/t Au from 174.7 m to 234.1 m including 16.8 m averaging 4.7 m g/t Au from 217.3 m to 234.1 m.

Drill hole FC17-01 targeted the Pack Rat zone at depth. The Pack Rat zone is located approximately 400 m to the southwest of the Colorado area along an extensional fault zone, the Pack Rat fault. The Pack Rat fault is considered to be one of the mineralized structures at Fondaway Canyon. Drill hole FC17-01 intersected 46.6 m core length with an average grade of 1.29 g/t Au between 319.1 m to 365.8 m ending in mineralization at the bottom of the hole.

Drill hole FC17-02 was drilled in the Colorado area to twin the historical Tenneco RC drill hole TF-11. Drill hole TF-11 intersected 7.4 g/t Au over 48.8 m from 176.8 m to 225.6 m depth. Drill hole FC17-02 intersected 2.1 g/t Au over 21.6 m from 189.3 to 210.9 m and a second zone that returned 1.8 g/t Au over 62.9 m between 253.1 m and 316.1 m.

Drill hole FC17-03 assessed the continuity and down dip extent of the Colorado zone. The hole intersected gold mineralization over a 65.4 m interval returning an average assay of 2.83 g/t Au, including 1.77 g/t Au over 62.9 m, 7.69 g/t Au over 9.8 m, and 5.28 g/t Au over 7.9 m. These results support the continuity and extent of the mineralization 250 meters down dip of the surface expression of the Colorado zone.

Drill hole FC17-04 tested the northeast striking quartz-vein stock-work hosted shear zone down dip from the Half Moon gold zone. Drill hole FC17-04 reported a 66.1 m intersection with an assay of 1.01 g/t Au from 226.2 m to 292.3 m and extending mineralization about 70 m down-dip from previous drilling.

Drill hole FC17-05 tested the South Pit area that is situated at the southwestern extent, 500 m to the southwest of the start of the Half Moon zone, of an extensional fault zone parallel to the Pack Rat fault. The hole intersected two intervals, 2.4 to 4.0 m in width with grades 6.6 and 3.4 g/t respectively, before being completed in mineralization that returned 3.48 g/t Au over 12.8 m from 345.3 to 358.1 m. The mineralization encountered at the bottom of the hole is a previously unknown gold zone that lies outside of the known extents of mineralization at Fondaway Canyon.

6.4 Historical Metallurgical Analysis

6.4.1 Historical Tenneco Results

Over a short period between 1989 and 1990, Tenneco operated an open pit mine on the Fondaway Canyon Property. Tenneco mined and processed approximately 186,000 tons of oxide ore with an average grade of 0.034 opt (1.06 g/t) with a reported recovery of approximately 87% (Cohan, 1997).

The oxide ore was crushed in a primary jaw crusher and a secondary cone crusher in an open circuit to minus two inches, then agglomerated. The crushed ore was stacked on the leach pads in 20-foot lifts, then cyanide leached. Gold was recovered from the pregnant solution using a Merrill-Crowe precipitation process (Tenneco, 1990).

6.4.2 Historical Metallurgical Testing

The mineralized oxide material at Fondaway Canyon was found to be readily leachable. However, the mineralized sulfide material was found to contain organic carbon which has the ability to re-absorb gold from solution (preg-robbing). In 1988, Tenneco commissioned a Hazen Research testing program to determine the most economical means of recovering gold from the high grade, mineralized sulfide material. Results from the Hazen 1988 testing are shown in Table 6-4.

Table 6-4: Hazen 1988 Test Results

Extraction Method	Recovery
Standard Cyanide leaching	< 0.1%
Carbon-in-leach (CIL) leaching	22.4 to 72%
Acidic High-Pressure pre-treatment with CIL	55.1 to 85.4%
Alkaline High Pressure pre-treatment with CIL	62.3 to 69.8%
Chlorine pre-treatment with CIL	50.9 to 59.5%
Nitrate pre-treatment with CIL	36.3 to 75.2%
Air/Caustic pre-treatment with CIL	51.1 to 74.2%
Roasting pre-treatment with CIL (high grade from Colorado area)	79.1 to +88%
Phase III Roasting with CIL (high grade from various veins)	86 to 95%

Hazen concluded that Carbon-in-Leach (CIL) was the best leaching process, due to the preg-robbing characteristics of the sulfide material. Additionally, Hazen found that an oxidizing pre-treatment would be required prior to CIL leaching with roasting found to be most effective, over a range of vein composites and samples (1990).

Tenneco also did some preliminary testing on biological oxidation of the sulphides, followed by CIL. They reported recovery rates from 72.3 to 92.8% (Cohan, 1997).

In late 1990, Tenneco commissioned American Barrick to conduct a series of flotation tests on samples collected from the Half Moon vein in the Tenneco adit. The testing was designed to collect the sulphides and organic carbon in two separate concentrates by selectively floating the carbon first, and the carbon second, leaving "clean" tailings for treatment by direct cyanidation. The results were reported to be very encouraging, with 83% of the total gold reporting to the concentrates, and CIL leaching of the flotation tails recovering an additional 12% of the total gold, for an overall recovery ranging from 93 to 95% (Cohan, 1997).

6.4.3 2016 Aorere Metallurgical Testing

A total of 9 core samples were described, photographed and sent to McClelland Labs for flotation testing. Samples were included from drill holes 02FC-02, 02FC-04, and 02FC-05. The goal was to make a composite grading 0.20 opt (6.25 g/t) or better from the carbonaceous, sulfidic mineralization. The samples totaled 88.5 lbs (40 kg). The results of the testing were reported to Canarc in McPartland (2017).

Initially, each of the individual samples was assayed, with grades ranging from 0.42 to 12.31 g/t Au, and the remaining material from the samples was combined to produce a metallurgical composite. The composite head grade for testing was 5.92 g/t Au, 1.30 g/t silver (Ag). The composite also contained 0.12% antimony, 0.84% arsenic, 1.77% sulfide sulfur, and 0.43% organic carbon.

Initial flotation testing included a single test (F-2) to determine response of the composite to bulk sulphide flotation treatment, and another test (F-1) to attempt to differentially float organic carbon, gold bearing minerals, and antimony bearing minerals. Based on results from those tests, a series of tests was conducted to optimize grind size (F-4 through F-7).

After results from those tests were reviewed, a single kinetic rougher flotation test was conducted (F-3), and a series of tests was conducted to evaluate cleaner flotation of a bulk sulfide rougher concentrate (F-8 through F-10). Summary results from those tests are shown in Table 6-5.

Results from the initial bulk sulfide flotation test (F-2) showed that the composite responded reasonably well at an 80%-75µm feed size. The rougher concentrate was 24.2% of the feed weight and recovered 85.4% of the gold, and the cleaner concentrate was 9.7% of the feed weight, assayed 46.7 g/t Au, and represented gold and sulfide sulfur recoveries of 78.6% and 74.4%, respectively.

Table 6-5: McClelland Summary Flotation Test Results, Fondaway Canyon Drill Core Composite 4136-001

Test	Feed Size P_{80}	Weight, %				Assay, g Au/mt				Au Distribution, %			
		Cl Conc	Cl Tail	Ro. Conc	Ro. Tail	Cl Conc	Cl Tail	Ro. Conc	Ro. Tail	Cl Conc	Cl Tail	Ro. Conc	Ro. Tail
F-1	75 µm	-	-	31.7	68.3	-	-	14.66	1.04	-	-	86.7	13.3
F-2	75 µm	9.7	14.5	24.2		46.7	2.7	20.34	1.11	78.6	6.8	85.4	14.6
F-3	75 µm	-	-	19.3	80.7	-	-	6.28	1.40	-	-	82.0	18.0
F-4	150 µm	-	-	19.5	80.5	-	-	24.5	1.96	-	-	75.2	24.8
F-5	75 µm	-	-	26.5	73.5	-	-	20.4	1.45	-	-	83.5	16.5
F-6	53 µm	-	-	22.6	77.4	-	-	23.8	1.40	-	-	83.2	16.8
F-7	45 µm	-	-	24.2	75.8	-	-	22.0	1.36	-	-	83.8	16.2
F-8	75 µm	10.5	9.4	19.9	80.1	45.0	3.74	25.51	1.58	74.5	5.5	80.0	20.0
F-9	75 µm	9.4	13.4	22.8	77.2	48.5	3.16	21.85	1.74	72.1	6.7	78.8	21.2
F-10	75 µm	7.8	10.7	18.5	81.5	57.4	3.16	26.03	1.81	71.2	5.4	76.6	23.4

Source: McPartland, 2017

An attempt (Test F-l) was made to sequentially float organic carbon, followed by a gold rich pyrite concentrate and finally an antimony rich concentrate. Overall recovery was similar to bulk flotation. Although it was possible to selectively upgrade the targeted minerals in the respective concentrates, the selectivity achieved was not sufficient for a viable process. Extensive further testing would be required to properly evaluate the selective flotation of these targeted components.

A series of tests (F-4 through F-7) were run to optimize feed size for bulk sulfide flotation. Grinding from 80%-150µm to 80%-75µm improved gold recovery from 75.2% to 83.5%. Further grinding did not improve recovery.

A kinetic flotation test (F-3) was conducted at an 80%-75pm feed size, to better establish the relationship between flotation time, mass pull, concentrate grade and recoveries. That test employed an initial carbon pre-flotation stage, followed by bulk sulfide flotation. Analysis of the carbon concentrate (4.2% mass pull) confirmed that gold (34.2% of total) and antimony (30.3% of total) tended to report with the naturally floatable organic carbon (35.1% of total). Overall, results from the kinetic flotation test were consistent with those from the initial bulk sulfide flotation test, and showed relatively slow gold and sulfide flotation kinetics.

Cleaner flotation testing (F-8 through F-10) attempted to improve cleaner flotation recoveries. The best results, F-10, were produced by regrinding the rougher concentrate, and adding additional reagents, resulting in a cleaner concentrate with 71.2% of the gold in 7.8% of the feed weight.

Separate testing was conducted for gravity concentration. The feed was ground to 80%-75µm, then passing the milled sample, as a slurry, one time through a Knelson concentrator to produce a rougher concentrate. The rougher concentrate was 2.31% of the feed weight and represented a gold recovery of 20.1%.

The 2016 metallurgical testing provided confidence that the mineralized material tested to date can be treated appropriately to concentrate 79-85% of the gold in less than 10% weight percent via flotation processes. Further testing was recommended of a combined gravity – flotation circuit to determine if any of the gold values recovered by gravity concentration are not otherwise recovered by flotation. Further testing is also needed to determine whether additional gold could be recovered from the flotation tails using cyanide leaching as demonstrated in the American Barrick metallurgical tests.

6.5 Historical Mineral Resource Estimates

Tenneco (1990), Cohan (1997), Brady (1997), and Strachan (2003) each produced a technical report which provided a Mineral Resource Estimate (MRE) for the Fondaway Canyon Project. The historical MRE's were calculated prior to the implementation of the standards set forth in NI 43-101 and current CIM standards for Mineral Resource estimation. Resource definitions, terminology, and reporting standards have changed significantly since these series of reports. The estimates in these reports are all considered historical in nature and a QP has not done sufficient work to evaluate these resources as current resources. Therefore, the Company and the QPs of this report are treating these estimates as historical in nature.

In 2017 Canarc released a Mineral Resource Estimate for the Fondaway Canyon Deposit prepared by Techbase International Ltd. of Reno, NV (Norred and Henderson, 2017). The Mineral Resource Estimate was prepared based on a potential underground mining scenario. The Mineral Resource Estimate was prepared in accordance with NI 43-101 and CIM standards at that time and used acceptable classes of mineral resources. The Mineral Resource Estimate used a cutoff grade of 3.43 g/t Au and is presented in Table 6-6. The Mineral Resource Estimate included drilling results up to 2016.

The Mineral Resource Estimate was compiled from 591 drill holes (49,086 m) with Techbase software that used a polygonal method for each interpreted vein. Cutoff parameters of 0.10 opt (3.43 g/t) Au and 1.8 m horizontal vein width were used. A total of twelve veins were deemed to have sufficient composited intercepts and continuity with sulfide mineralization to be included in the Mineral Resource Estimate. No capping or cutting of grades was applied. Mineral resources based upon a polygonal method of estimation along with no proper statistical evaluation, including capping of high-grade outlier values, is not considered appropriate based upon current CIM guidelines and standards. The 2017 Mineral Resource Estimate is superseded by the updated Mineral Resource Estimate presented herein.

Table 6-6: Canarc Mineral Resource Estimate

Resource Category	Tonnes[1] (t)	Grade (g/t) Au	Ounces[2] (oz) Au	Type
Indicated	2,050,000	6.18	409,000	UG/Sulfide
Inferred	3,200,000	6.40	660,000	UG/Sulfide

[1] Resource based on cutoff of 1.8 m horizontal width >= 3.43 g/t Au
[2] Rounding differences may occur

6.6 Historical Production

Tungsten mining occurred at the Upper and Lower Quick Tung mines during the 1950's with production recorded as 10,000 tons with a recovered 200,000 lbs of WO_3. Small scale production of antimony and mercury took place at the historical Quick Tung mine through 1976 (Lawrence, 1977).

Gold was discovered in the mid-1970s with the first commercial deposit identified in 1977. The Fisk family conducted open pit mining from various pits (e.g. South, Main, West, Fisk, Upper and Lower Stibnite, and Oxy pits) from 1978 through to 1983. A reported 25,000 tons of ore were mined over this period producing 2,500 troy ounces of gold recovered by a vat leach extraction operation at site (Figure 6-1).

During 1989 and 1990, Tenneco operated an open pit mine with heap leach processing. Tenneco mined approximately 171,000 tons of oxide mineralization from the South Mouth pits at an average grade of 1.1 g/t Au. They supplemented this production with 12,000 tons of oxide material from the Reed Pit and 4,000 tons of oxide material from the Half Moon Stibnite Pits. The total gold produced from the Tenneco mining was 6,324 ounces.

Table 6-7: Fondaway Canyon Historic Gold Mining Production

Year	Company	Zone	Processed (tons)	Waste (tons)	Processed (tonnes)	Grade (opt Au)	Grade (g/t)	Contained Au oz	Recovered Au oz	Strip Ratio
1978-1983	Fisk Mining	South, Main, West, Fisk, Upper/Lower Stibnite and Oxy Pits	**25,000**	25,000	**22,680**	0.200	6.86	5,000	2,500	1.0 : 1
1989-1990	Tenneco	South Mouth Pit	171,000	1,048,000	155,128	0.032	1.10	5,527		6.1 : 1
		Reed Pit	12,000	43,000	10,886	0.030	1.03	361		3.6 : 1
		Stibnite	4,000	13,000	3,629	0.109	3.74	436		3.3 : 1
		Tenneco Total:	**186,000**	1,138,000	**168,736**	0.034	1.17	6,324	5,402	5.9 : 1
		Grand Total:	**211,000**	1,163,000	**191,416**	0.054	1.85	11,324	7,902	5.5 : 1

Source: Cohan, 1997

High-grade sulfide gold was mined from the Tenneco Drift (Figure 6-9 and Figure 6-10) but was not put on the heap leach pads. No record exists of gold being recovered from the mined adit. It is estimated that 1,500 tons at an average grade of 1.2 g/t Au was mined and stockpiled on surface (Figure 6-11) for metallurgical testing.

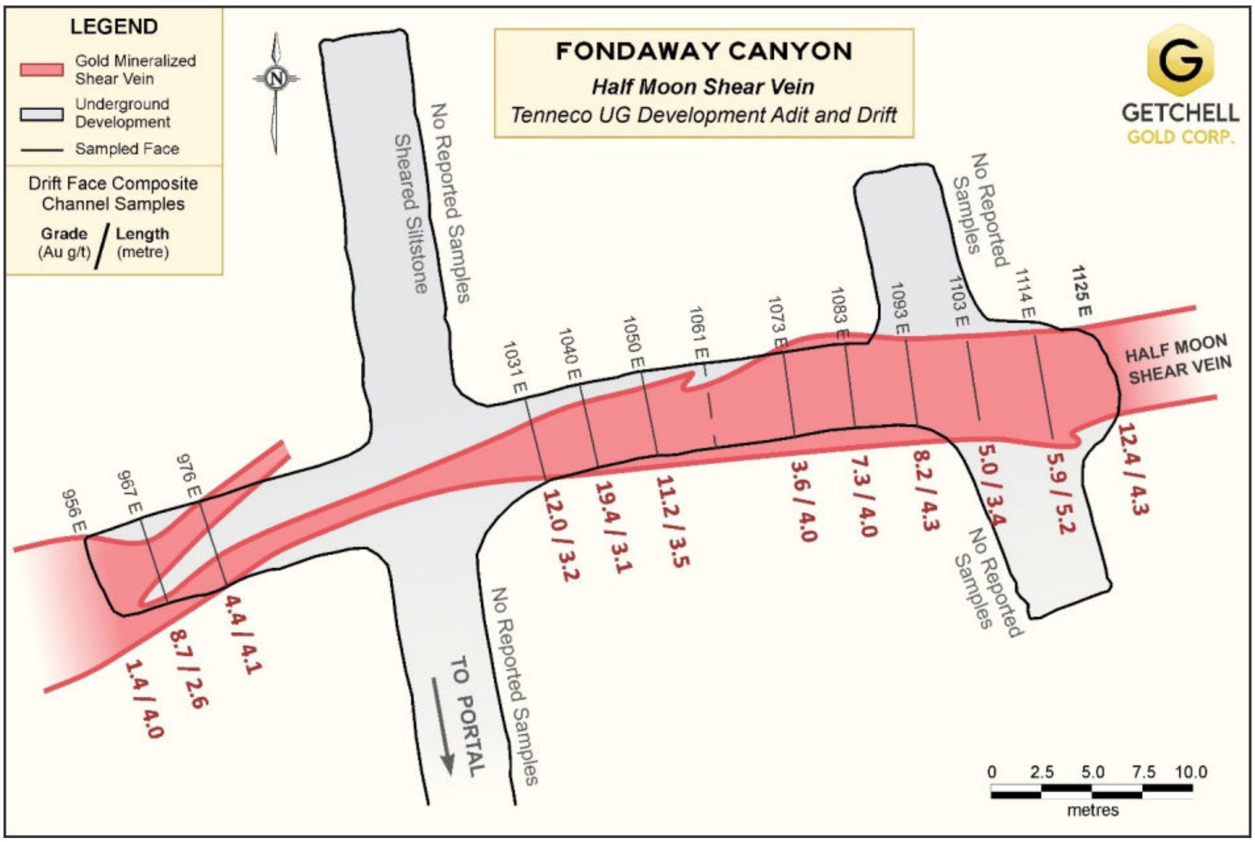


Figure 6-9: Tenneco Half Moon Shear Vein UG Drift Development and Gold Grades

(Source: Getchell Gold, 2020, modified from Tenneco Minerals Company, 1989)

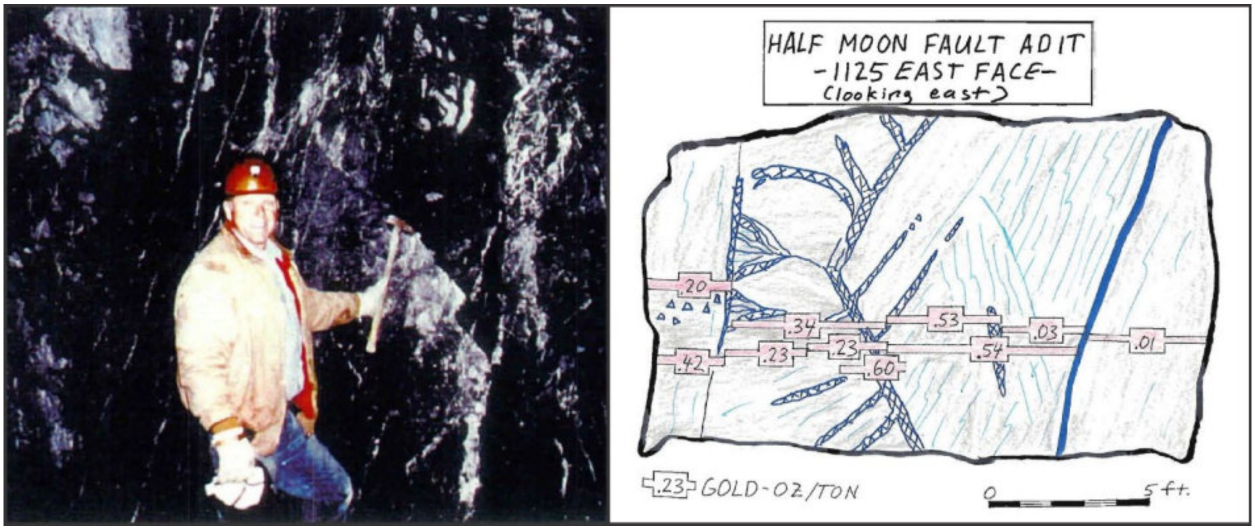


Figure 6-10: Tenneco Drift Development Face 1125E and UG Photo of Dick Fisk

(Source: Getchell Gold, 2020, Photo from Tenneco Minerals Company, 1990, and drift face modified from Tenneco Minerals Company, 1989)

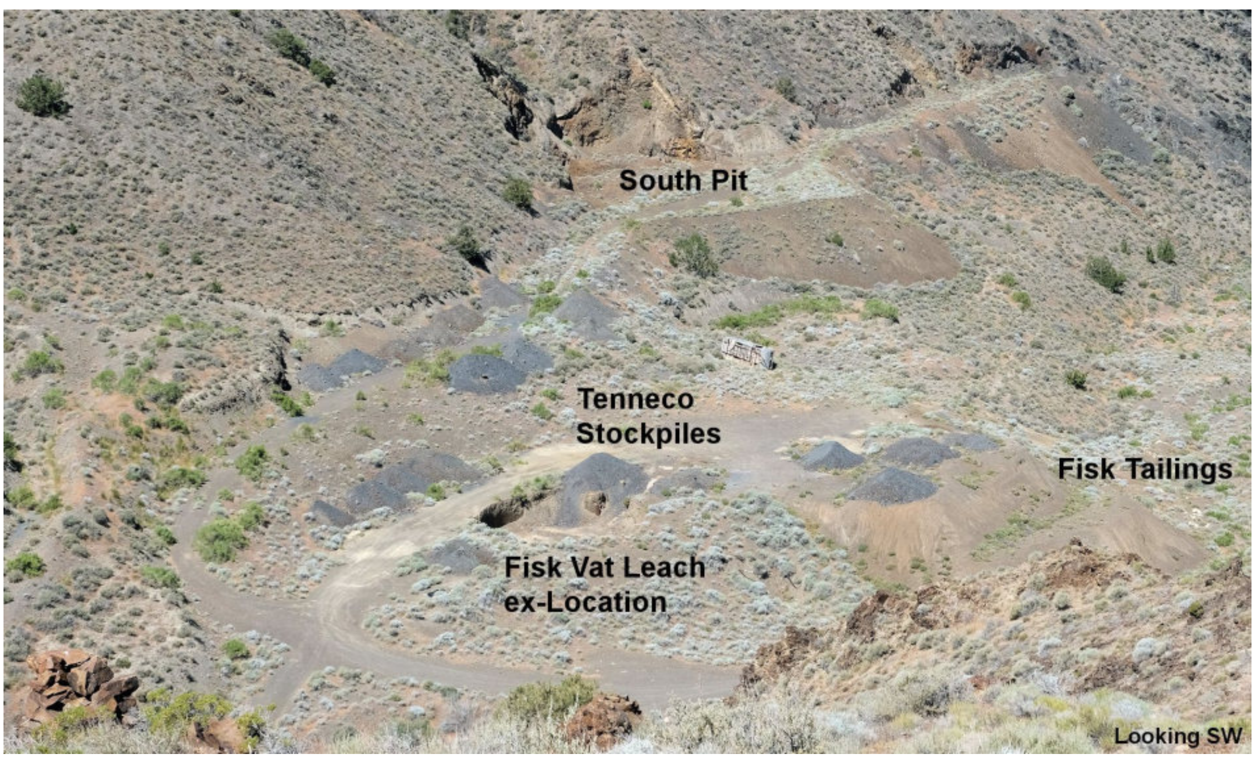


Figure 6-11: Tenneco Stockpiles, Fisk Tailings, and South Pit

(Source: Getchell Gold, 2024)

7. GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Stillwater Range lies within a region underlain by Triassic-Jurassic sedimentary and volcanic rocks, Mesozoic to Miocene intrusive rocks and, locally, Oligo-Miocene volcanic rocks (Figure 7-1; Crawford, 2007). Rocks exposed along the west flank of the range in the area of Fondaway Canyon are mostly Triassic black shales that are weakly metamorphosed to phyllite with bedding-parallel foliation and comprise a sequence that may be as much as 3 km thick (Page, 1965). Minor quartzite and limestone are also present, and fossils indicate an Upper Triassic age (Page, 1965). Quartzite and limestone (marble) of possible Jurassic age are exposed above a thrust fault around the mouth of Fondaway Canyon ("Boyer Thrust"; Page, 1965) that is likely part of the regional, Jurassic, Luning-Fencemaker fold and thrust belt (Wyld, 2002; Figure 7-1). Volcanic rocks dip gently east along the crest and east flank of the range; similar volcanic rocks 20 km to the south have been dated as Oligocene (~25-30 Ma; Colgan et al., 2018).

Several styles of gold-silver deposits occur in the region, largely in and adjacent to the Humboldt range 35-100 km north of Fondaway Canyon. These include the Middle Miocene and younger epithermal deposits at Florida Canyon (Fifarek et al., 2011), Willard-Colado (Conelea and Howald, 2011), Dixie Comstock (Vikre, 1994) and at least a part of the Relief Canyon district (Fifarek et al., 2015); Oligocene (23-27 Ma), locally intrusion-related, volcanic- and sediment-hosted Au-Ag-Cu deposits at Trinity and Majuba Hill (John and Muntean, 2006) and at least part of Relief Canyon; and Mesozoic, intrusion-related systems typified by the world-class Rochester deposit (Ag rich; Vikre, 1981; Hohbach and Johnson, 2015) and possibly Spring Valley (Crosby and Thompson, 2015).

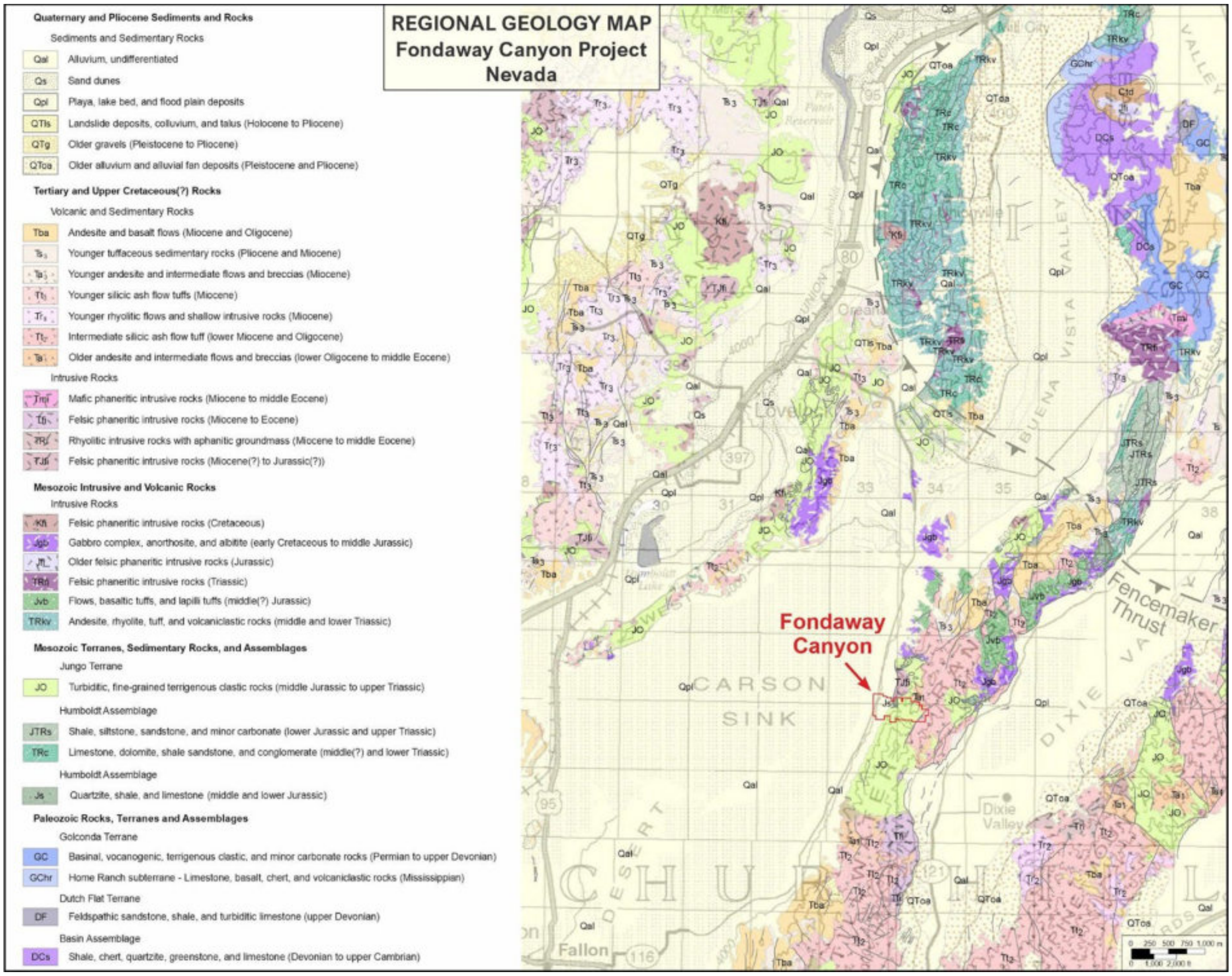


Figure 7-1: Regional Geology of Fondaway Canyon Project

(Source: Modified from Crafford, 2007)

7.2 Local and Property Geology

A detailed description of the Fondaway Canyon local geology is contained within a paper published by Jakob Margolis, formerly of Canagold, for the 2020 Geological Society of Nevada Symposium (Margolis, 2020).

The Fondaway Canyon area is mostly underlain by a Triassic black shale sequence consisting of thinly-bedded, black, carbonaceous (phyllitic) shale and siltstone which contain a well-developed penetrative cleavage. The sequence largely strikes west-northwest and dips moderately to steeply southwest; vertical and locally overturned bedding is common. However, in areas of mineralization, mafic dikes and high-angle faults, bedding is more chaotic and commonly broadly parallel to dikes and faults.

Host rocks for the majority of the mineralization at Fondaway Canyon (Half Moon, Paperweight, Hamburger Hill and South Pit Zones) are primarily shale and mudstone of the Triassic Age Grass Valley Formation (Figure 7-2). The Grass Valley Formation has been regionally metamorphosed to phyllite and folded into east-west trending folds with approximately 180 m amplitude across the folds and vertical to slightly overturned limbs. Jurassic Age Boyer Ranch limestone and quartzite is mapped at the Colorado-Deep Dive areas and appears to be overthrust by Grass Valley phyllite.

East-west faulting crosscuts the metamorphosed sedimentary units and forms a 3.5 km long structural corridor that hosts the majority of the gold mineralization at Fondaway Canyon. Sets of north trending mineralized and post-mineral faults displace east-west trending mineralized faults. The north trending post mineral faults are probably related to basin and range development (Young, 1989). A low-angle fault, termed the "Boyer Thrust" by Page (1965) crosses the western part of Fondaway Canyon (Figure 7-2). The fault is expressed as a thick (> 10 m in some areas) zone of brecciation, shearing and strong iron-oxide development.

An Oligocene age granitic stock, called the White Cloud Canyon stock, is exposed north of Fondaway Canyon and covers an area of about 5 by 2 km (Figure 7-2). Outside of this stock, the only other exposed granitoid in the area is a Cretaceous age granite occurring about 700 m north-northwest of the Colorado zone, which is possibly underlying the tungsten skarn deposits in the central mined area.

Tertiary age west-striking dacite dykes and west-northwest striking basaltic-andesite mafic dykes, typically of 1-4 m width, occur at Fondaway Canyon and are broadly parallel to mineralized trends and structures. These dykes are altered but not strongly mineralized.

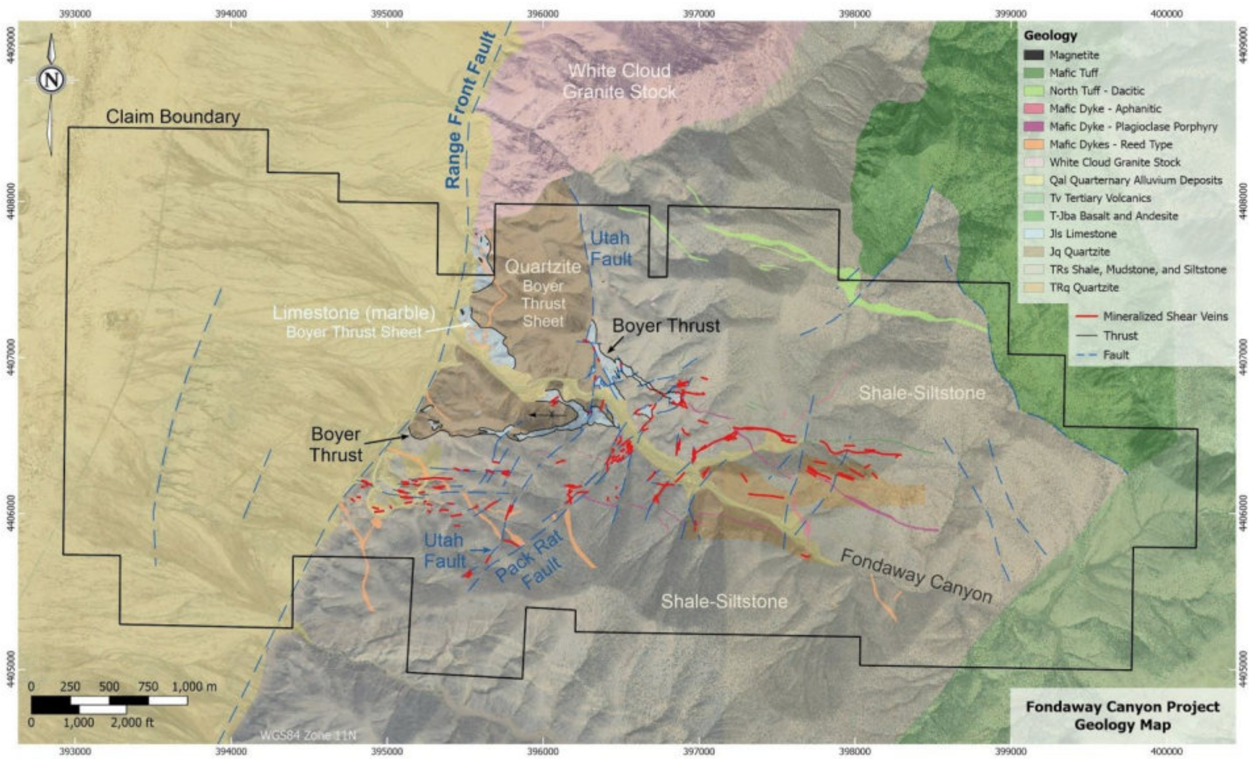


Figure 7-2: Local Surface Geology of Fondaway Canyon Project

(Source: Getchell Gold, 2024, modified from Margolis, 2020, Brady, 1997, Proffett, 1989, and Howell, 1984)

Note: Faults (green lines, except black for the Boyer thrust) are shown as solid (definitive) for clarity, although their traces in many areas are inferred; faults and some dikes are shown projected through alluvium, again for clarity, as they do not cut alluvium.

7.3 Mineralization

Gold mineralization at Fondaway Canyon occurs within sharp-walled or more diffuse shear veins within carbonaceous shale-siltstone that are characterized by strong brittle-ductile fabrics (crushing, lenticular and lens-like textures); a high sulfide content; diffuse, broken quartz—Fe-carbonate—sulfide veins and fillings; and silicification (Margolis, 2020).

The precious metal mineralization at Fondaway Canyon characteristically has a low Au:Ag ratio of less than 1:1, is interpreted to be structurally controlled mesothermal and is associated with the sulfide minerals pyrite, stibnite, arsenopyrite and lesser amounts of tetrahedrite, chalcopyrite, galena and pyrrhotite. Thin sections identify the gold to be 5 to 20 microns in size and found to occur in quartz veins and zones of silicification and sulfides with pyrite, arsenopyrite, quartz and brecciated carbonaceous siltstone (Hazen Research Petrographic Report, 1989). The carbonaceous host may account for 10 to 20% of the mineralization and is likely to exhibit preg-robbing characteristics (defined as a phenomenon in which a metal of interest is adsorbed or retained by minerals, especially due to the presence of species like carbonaceous matter and silicates, therefore reducing its recovery potential).

The major gold mineralization occurs spatially related to faults in silicified vertical to steeply-dipping (70 – 85 degrees south) east-trending shear zones; but low-angle veins also occur, likely occupying Mesozoic thrust planes. Individual shears are typically 1–5 m wide but combine to form broader shear-zone stockwork corridors locally 100–150 m thick that dip more moderately than the contained individual mineralized zones. Gold mineralization is restricted to the shear zone and does not disseminate into the wallrock shale and siltstone of the Upper Grass Valley Formation unless there are stockworks of fracture quartz veins and silica replacement that permitted the migration of mineralization into the wallrock. The vertical extent of the gold mineralization is greater than 300 m based on the recent drilling by Nevada Contact and Getchell Gold. The most persistent vein zone strike length is 900 m on the Paperweight – Hamburger Hill Zone. Vein width is commonly 1.5 to 6.0 m. However, the QP observed numerous stockwork, breccia zones and silicified zones with gold mineralization that are likely spatially related to the mineralized faults with high carbon, pyrite, barite, arsenic, antimony, mercury.

8. DEPOSIT TYPE

8.1 Overview

The gold mineralization at Fondaway Canyon appears to conform to an orogenic intrusion-related mesothermal gold system. Although this is the most likely model for mineralization, structurally controlled, low-sulfidation epithermal mineralization cannot be entirely ruled out. A schematic showing the types of mineralization typically associated with this deposit type is provided in Figure 8-1.

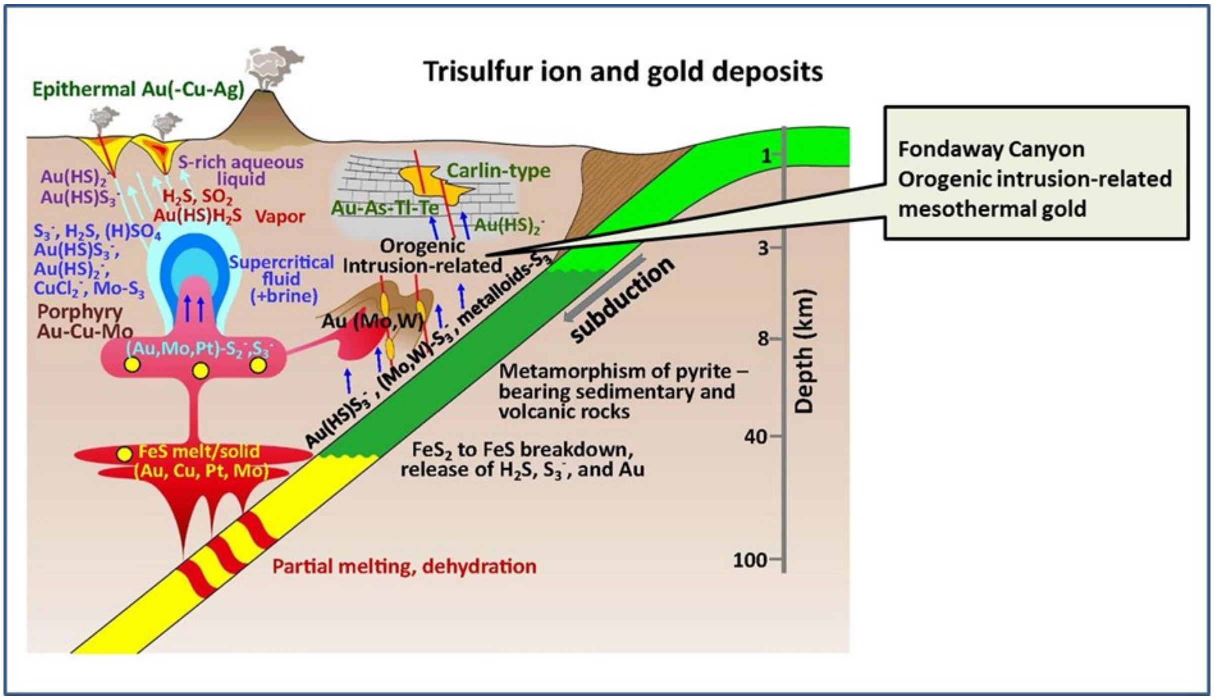


Figure 8-1: Gold Mineralization Systems

(Source: Pokrovski, 2015)

8.2 Geological Setting

The structural setting, alteration mineralogy and mineralization characteristics at the Fondaway Canyon Property are consistent with orogenic gold deposits as defined in Moritz (2000), Goldfarb et al. (2005), Groves et al. (1998; 2003), and Johnston et al. (2015).

Orogenic gold deposits occur in variably deformed metamorphic terranes formed during Middle Archean to younger Precambrian, and continuously throughout the Phanerozoic. The host geological environments are typically volcano–plutonic or clastic sedimentary terranes, but gold deposits can be hosted by any rock type. There is a consistent spatial and temporal association with granitoids of a variety of compositions. Host rocks are metamorphosed to greenschist facies, but locally can achieve amphibolite or granulite facies conditions.

8.3 Mineralization

Gold deposition occurs adjacent to first-order, deep-crustal fault zones with interpreted long-lived structural controls. These first-order faults, which can be hundreds of kilometers long and kilometers wide, show complex structural histories. Economic mineralization typically formed as vein fill of second- and third-order shears and faults, particularly at jogs or changes in strike along the crustal fault zones. Mineralization styles vary from stockworks and breccias in shallow, brittle regimes, through laminated crack-seal veins and sigmoidal vein arrays in brittle-ductile crustal regions, to replacement- and disseminated-type orebodies in deeper, ductile environments. The specific style of gold mineralization at Fondaway can be classified as both structurally controlled, vein associated and locally disseminated in zones of silicification and/or brecciation.

Orogenic gold deposits in Nevada are situated along the Argentoro belt (Luning-Fencemaker Fold-and Thrust Belt of Wyld et al., 2000, 2001; DeCelles, 2004), a 700-km long, north-south trending belt extending from south-eastern California to the Nevada-Oregon border. The belt formed between ~100 Ma and 70 Ma synchronous with low-grade metamorphism and brittle-ductile deformation. District-scale controls consist of high-angle, N-striking strike-slip faults, while deposit-scale controls consist of NW-, EW-, and NE-striking dip-slip fracture arrays.

Johnston et at. (2015) outline that Nevada orogenic gold deposits are defined by: 1) widespread low to moderate-grade metamorphism in Mesozoic rocks, 2) low-sulfide bearing, mesothermal “bull-quartz” veins emplaced in shear zones, 3) ubiquitous quartz-sericite-pyrite alteration of wall rocks, 4) diluted CO_2-rich ore fluids, 5) coarse gold in veins, 6) elevated concentrations of Ag, Sb, As, and Hg, and 7) abundant placer gold deposits.

A tungsten rich garnetiferous skarn deposit is developed in a contact metamorphism envelope in a limestone along the West Side of the Central gold resource area. The skarn contains gold mineralization where silicification of possibly a later hydrothermal event has overprinted the skarn alteration. The tungsten mineralization is coarse crystalline scheelite in marble and garnetiferous exoskarn. An intrusion of igneous rock has not been observed or reported in association with the skarn to date, however, the Company has conducted little to no work on the skarn and the associated historical mines developed on it.

9. EXPLORATION

During 2020 Getchell Gold compiled a Microsoft Access database, reviewed historical drill results, produced a new geological model for the deposit and designed a drill program to test the model and the extents of the known mineralized zones. In addition, approximately 2,800 core photos were indexed, and the majority of the drill logs were converted from static paper copies to digital format with the significant geological attributes coded into a standardized digital database. The new interpretation of the geological model was aided by using the Seequent Ltd. software products Target and Leapfrog 3D (Frostad, 2021; 2022). The historical data compilation and geological model were then used to delineate drill targets at the Property, with the Getchell Gold drilling programs detailed in Section 10 of this Technical Report.

The Fondaway Canyon Property is an advanced stage gold project that warrants continued exploration work. However, the QP recommends that future exploration activities are mainly centered on additional metallurgical test work, and on exploration drilling to test areas of the Property that have not been drill-tested in the past, as well as infill and expansion drilling.

10. DRILLING

Total drilling on the Fondaway Canyon Property includes 765 drill holes for over 64,419 m completed between 1981 and 2022 by various operators including Getchell Gold. A brief summary of historical drilling is provided in Section 10.1 with additional details included in Section 6. Drilling conducted by Getchell Gold is summarized in Section 10.2.

10.1 Historical Drilling Summary

Data available for historical drill programs at the Fondaway Canyon Property is variable dependent on the operator and age of the drill program. Historical drilling is described in detail in Section 6.3 and summarized in the following text.

Based on Getchell Gold's current database, a total of 735 drill holes totaling over 53,800 m have been completed historically on the Fondaway Canyon Property between 1981 and 2017 by various operators. The majority of the drilling has been reverse circulation (RC) with 678 RC drill holes completed on the Property totaling over 42,000 m. Additionally, 57 core drill holes have been completed totaling over 11,788 m. Companies that carried out drilling historically over the Fondaway Canyon Property include Occidental Minerals (1981-1982), Tundra Gold (1983), Homestake Mining (1984-1985), New Beginnings (1984), Mill Creek Mining (1985), Tenneco Minerals (1987-1996), Nevada Contact (2002) and Canarc Resources (2017) (Table 6-1; Figure 6-6).

The historical drilling programs resulted in the delineation of several gold-bearing zones on the Property. The historical drilling programs primarily targeted two areas: the West Area and the Central (Main) Area, each of which contain numerous prospective mineralized zones. See Section 6 for additional information on drill results including best results returned from each historical drill campaign.

Samples from historical drilling were analyzed at various laboratories that include Cone Geochemical, (Denver CO), Geochemical Services Inc., (Reno, NZ), Shasta Analytical Geochemistry Laboratory, (Redding CA) and G.D. Resources Inc., (Sparks, NV). All the listed analytical laboratories are independent of the authors and the issuer of this Technical Report. Although some of the laboratories are no longer in business, all laboratories were certified and known in the industry for professional procedures and quality results.

Gold was measured by fire assay with an Atomic Absorption finish and copies of the original assay sheets were made available to the QP. The laboratories employed a quality assurance/quality control (QA/QC) protocol that included periodic duplicate analyses of core pulps at least for Tenneco Minerals (1988-1990), Nevada Contact (2002), and Canarc Resources (2017) drilling programs. Additional QA/QC data available to the authors include certified reference materials (standards) and blanks inserted by the laboratory for Canarc Resources' 2017 drilling program. No other QA/QC data is available from the historical drilling campaigns from either the operator with inserted QA/QC samples or from the laboratory.

The compiled drill hole database used for the Mineral Resource Estimate (MRE) calculation contains a total of 647 historical drill holes (collars and assays) totaling 53,676 m for drill holes completed between 1981 and 2017 by previous operators (Table 6-1). Drill holes with incomplete data (i.e. missing collar locations, missing collar ID, missing assays) were not included in the final MRE database.

10.2 Getchell Gold Drilling Programs

Getchell Gold carried out three diamond drill programs for 30 drill holes of HQ sized core totaling 10,619 m between 2020 and 2022. They were primarily carried out in the Central Area of the Fondaway Canyon Project (Figure 10-1). The combined programs consisted of 28 completed (10,454 m) and 2 abandoned diamond drill holes (165 m) totaling 10,619 m (34,839 ft). The drilling contractor for the 2020-2022 drill programs was First Drilling of Montrose, Colorado and the assay laboratory used was Bureau Veritas Laboratories' ("BVL") of Sparks, Nevada. BVL is accredited to ISO/IEC 17025 and ISO 9001, and is independent of the issuer and the authors of this report.

The initial drill program was conducted in 2020 totaling 1,996 m in six diamond drill holes, FCG20-01 to FCG20-06. This program resulted in three major discoveries: the Colorado SW and the Juniper zones to the SW and down dip of the Colorado Pit zone, and the North Fork zone to the SSW of the Half Moon Shear Vein.

The 2021 exploration program consisted of ten completed diamond drill holes (3,875 m) and one abandoned drill hole (95m), FCG21-07 to FCG21-16 and FCG21-10A, totaling 3,970 m. This program expanded upon the zones discovered during the 2020 drill program and identified high-grade structures.

The 2022 exploration program included twelve completed diamond drill holes (4,583 m) and one abandoned hole (70 m), FCG22-17 to FCG22-28 and FCG22-17A, totaling 4,653 m. Only 3 of the 12 drill holes, totaling 1,107 m, were completed by the data cut off date of the previous technical report (Dufresne et al., 2023). This Technical Report includes data for the 9 additional drill holes completed.

Table 10-1 shows a breakdown of the 2020, 2021 and 2022 Getchell Gold drilling programs. Figure 10-1 shows the location of the 2020 to 2022 drill holes.

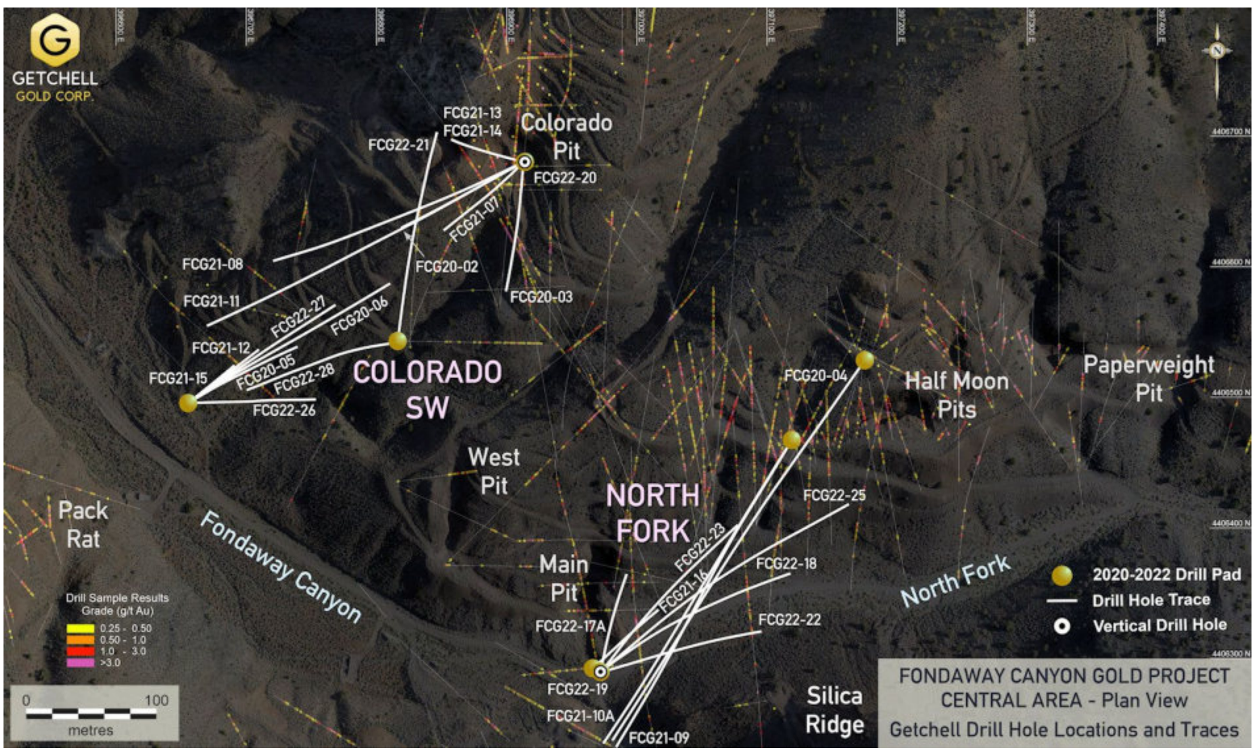


Figure 10-1: Fondaway Canyon Project Central Area Drill Programs

(Source: Getchell Gold, 2022)

Table 10-1: Fondaway Canyon Project Drill Programs Summary

Company	Year	# of Holes	Azimuth (◦)	Dip (◦)	Length (m)	Length (ft)
Getchell Gold Corp.	2020	6*	13 to 240	-54 to -73	1,996	6,548
	2021	11 (10)*	41 to 284	-48 to -87	3,970 (3,875)*	13,025 (12,713)*
	2022	13 (12)*	7 to 360	-47 to -90	4,653 (4,583)*	15,266 (15,037)*
TOTAL		30 (28)*			10,619 (10,454)*	34,839 (34,298)*

**Completed holes.*

10.2.1 2020 Getchell Gold Drilling Summary and Results

The 2020 drill program discovered three new zones within the Central Area of the Fondaway Canyon Project. These three new zones are referred to as Colorado SW, Juniper, and North Fork. The initial drill program conducted in 2020 totaled 1,996 m in six holes (FGC20-01 to 06; Table 10-2, Figure 10-2).

Table 10-2: Getchell Gold 2020 Collar Information

Hole ID	Year	UTM Northing* (m)	UTM Easting* (m)	Elevation (m)	Elevation (ft)	Azimuth (◦)	Dip (◦)	Depth (m)	Depth (ft)
FCG20-01	2020	4406172	394667	1,322	4,337	13	-67	253.5	831.7
FCG20-02	2020	4406680	396913	1,585	5,200	240	-66	353.9	1,161.1
FCG20-03	2020	4406680	396913	1,585	5,200	185	-68	295.0	967.9
FCG20-04	2020	4406528	397175	1,603	5,259	215	-54	499.0	1,637.2
FCG20-05	2020	4406495	396655	1,482	4,862	56	-73	289.0	948.2
FCG20-06	2020	4406495	396655	1,482	4,862	56	-57	305.4	1,002.0
							TOTAL	**1,996**	**6,548**

** Coordinate system: NAD 1983 / UTM Zone 11N*

The majority of the high-grade gold mineralization intersected during the 2020 drill program was associated with quartz carbon breccia and hosted by carbonaceous mudstone/siltstone. Re-mobilized carbon, finely disseminated pyrite and arsenopyrite, silicification and multiple episodes of brecciation and quartz veining were key indicators associated with these high-grade zones.

Results from the 2020 drill program suggested that a broad zone of mineralization was present below the Colorado pit and that it dipped shallowly to the southwest. Drill hole FCG20-05 returned the most notable intercept of the Colorado SW zone with 2.7 g/t Au over 51.8 m core length. Above the Colorado SW zone, high-grade gold mineralization was intersected by drill holes FCG20-02 and FCG20-03 and named the Juniper zone returning 6.2 g/t Au over 21.9 m core length and 4.3 g/t Au over 21.1 m core length respectively. Another gold discovery, 350 m to the SE, named the North Fork Gold Zone, was intersected by FCG20-04 returning 2.5 g/t Au over 58.0 m core length.

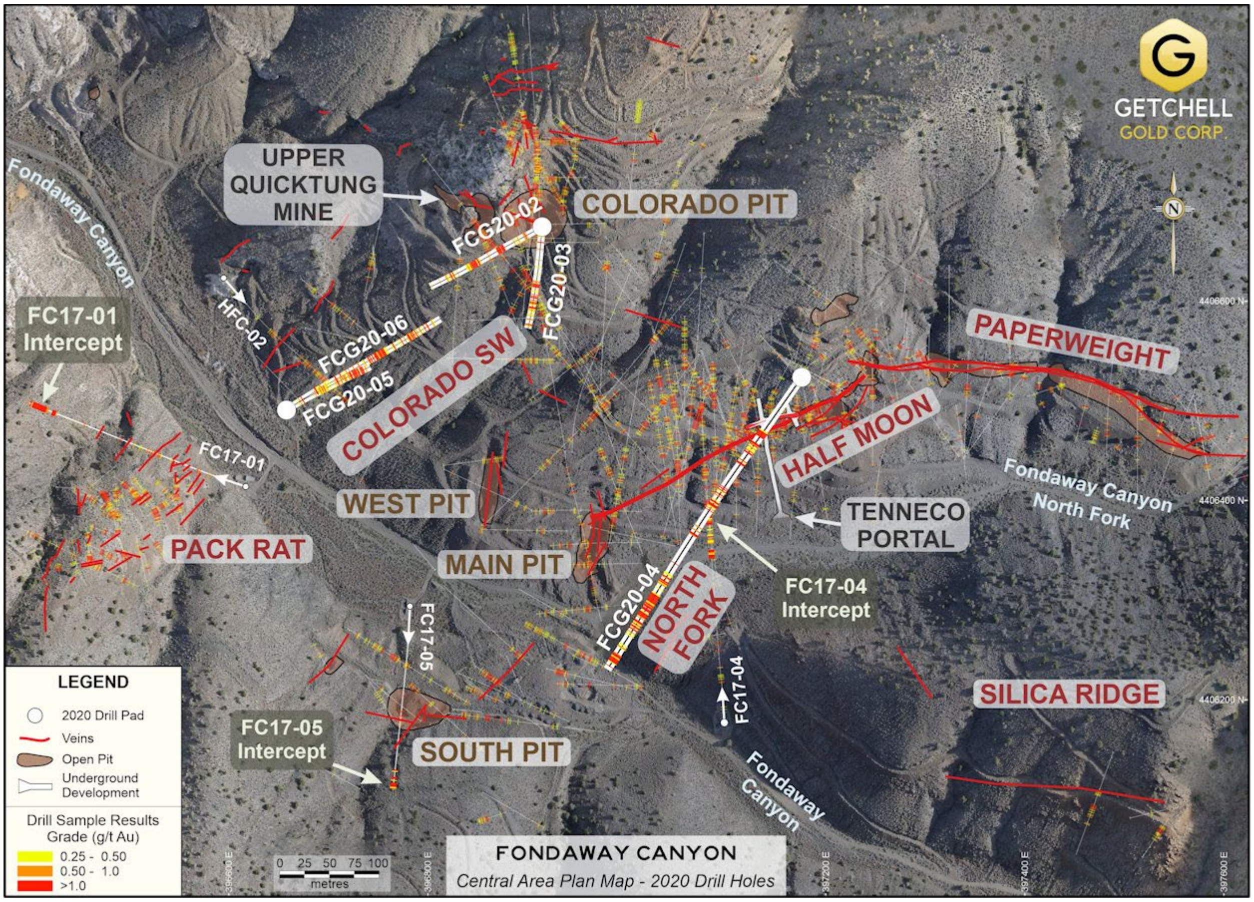


Figure 10-2: Getchell Gold 2020 Drill Hole Locations

(Source: Getchell Gold, 2020)

10.2.1.1 Results and Highlights

Table 10-3 provides highlights of the gold assay results from the 2020 drill program. Summary intervals provided are average gold grade over core length for all intervals and holes (not true thickness).

The Pediment Target is the westernmost known gold mineralized occurrence along the 3.5 km long east-west trending Fondaway Canyon gold mineralization corridor. The area is completely blanketed by a broad alluvium cover which is typical of the range and basin geomorphology for the area.

Drill hole FCG20-01 was drilled at Pediment targeting the midway point between the two gold bearing intervals intersected by historical RC drill holes 02FC-10 and 02FC-11 to characterize and model the mineralization geometry (Figure 10-3). The wide intersection of andesite dyke that was encountered at the top of the drill hole coincides with a northwest-southeast trending dyke mapped on surface within the South Mouth pit area. The interpreted dip of the dyke, based on oriented core measurements, also aligns the lower contact with the upper dyke intersected by 02FC-11. No limestone was seen within FCG-01 although wide limestone intercepts were logged within both of the proximal 2002 RC drill holes. The hole was lost within a fault zone prior to reaching the target depth. The last series of samples at the bottom of the hole showed an increase in gold values, 0.25 g/t over 3.2 m core length, and is interpreted as the top of the targeted gold zone.

Table 10-3: 2020 Getchell Gold Drilling Program Highlights

Zone	Drill Hole	Au g/t	Interval* (m)	Depth From (m)	Depth To (m)	Depth From (ft)	Depth To (ft)
Colorado	FCG20-02	2.5	8.5	41.1	49.5	134.8	162.4
	FCG20-02	6.2	21.9	106.1	128.0	348.1	420.0
	including	9.6	12.0	116.0	128.0	380.6	420.0
	including	20.4	3.2	120.5	123.7	395.3	405.9
	FCG20-02	1.9	43.5	181.0	224.5	593.8	736.6
	including	4.2	14.9	192.1	207.0	630.3	679.1
	FCG20-02	1.1	12.3	265.6	277.9	871.4	911.8
Colorado	FCG20-03	1.5	17.1	2.7	19.8	8.9	65.0
	FCG20-03	5.4	3.0	39.0	42.0	128.0	137.8
	FCG20-03	4.3	21.1	148.7	169.8	487.9	557.1
	including	8.7	9.4	159.6	169.0	523.6	554.5
	including	14.6	3.4	163.4	166.8	536.1	547.2
	FCG20-03	2.0	49.0	188.3	237.3	617. 8	778.5
	including	3.6	12.9	205.1	218.0	672.9	715.2
	including	3.4	7.0	224.9	231.9	737.9	760.8
	FCG20-03	4.4	2.2	262.30	264.5	860. 6	867. 8
	FCG20-03	1.2	4.9	277.1	282.0	909.1	925.2
Colorado	FCG20-05	2.1	4.0	62.5	66.5	205.1	218.2
	FCG20-05	0.6	28.0	119.0	147.0	390.4	482.3
	FCG20-05	6.3	3.3	165.7	169.0	543.6	554.5
	FCG20-05	1.8	90.0	177.5	267.5	582.4	877.6
	including	2.7	51.8	215.7	267.5	707.7	877.6
	including	3.0	45.3	222.2	267.5	729.0	877.6
	including	4.4	11.1	241.4	252.5	792.0	828.4
Colorado	FCG20-06	0.7	13.2	63.2	76.4	207.4	250.7
	FCG20-06	1.5	3.7	168.0	205.7	551.2	674.9
	including	2.1	192.0	181.0	200.2	593.8	656.8
	FCG20-06	1.1	38.3	243.5	281.8	798.9	924.5
	including	2.5	10.6	245.0	255.6	803.8	838.6
North Fork	FCG20-04	8.6	9.8	108.1	117.9	354.7	386.8
	FCG20-04	2.7	20.5	128.5	149.0	421.6	488.9
	FCG20-04	6.3	3.3	165.7	169.0	543.6	554.5
	FCG20-04	0.70	15.8	209.0	224.8	685.7	737.5
	FCG20-04	3.20	15.6	233.0	248.6	764.4	815.6
	including	5.50	8.5	23.0	241.5	75.5	792.3
	FCG20-04	1.30	3.9	286.0	289.9	938.3	951.1
	FCG20-04	1.30	13.5	356.0	369.5	1,168.0	1,212.3
	FCG20-04	2.50	58.0	383.0	441.0	1,256.7	1,446.9
	including	3.50	36.1	384.8	420.9	1,262.5	1,380.9

Zone	Drill Hole	Au g/t	Interval* (m)	Depth From (m)	Depth To (m)	Depth From (ft)	Depth To (ft)
	including	10.30	5.2	414.6	149.8	1,360.2	491.5
	FCG20-04	2.60	14.5	478.5	493.0	1,569.9	1,617.5

**Note: Intervals represent core length. True width can vary from 50% up to 100% of core length depending upon drill hole intersection angles.*

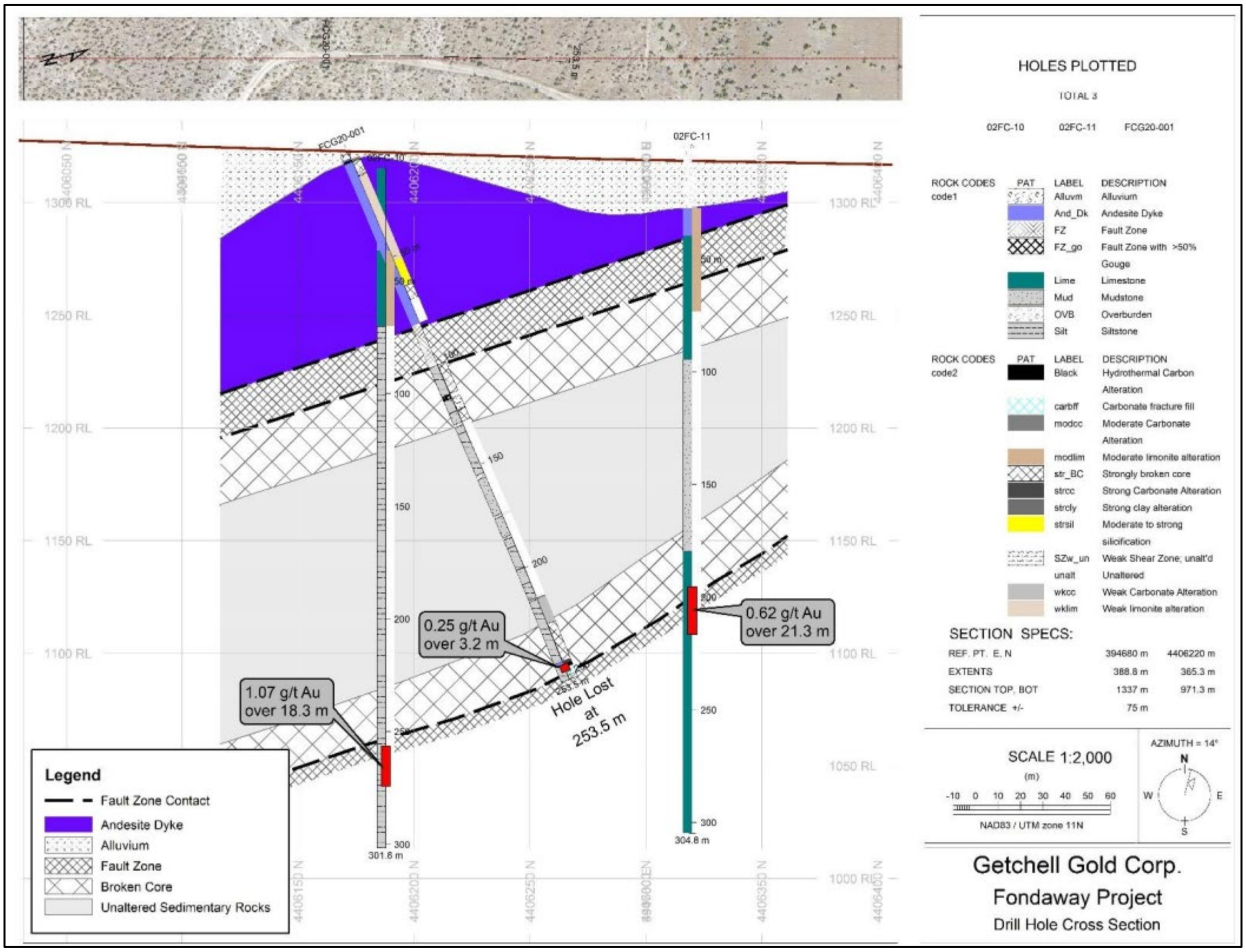


Figure 10-3: Drill Hole Section for FCG20-01

(Source: Getchell Gold, Frostad, 2021)

Note: Intercepts represent core length. True width can vary from 50% up to 100% of core length depending upon drill hole intersection angles.

Drill holes FCG20-02 and 03 were both collared from the historical Colorado Pit (Figure 10-4) and successfully extended the known gold mineralization towards the southwest. Since these drill holes were drilled using different azimuths, 240° for FCG20-02 and 185° for FCG20-03, the northeast-looking aspect of the interpreted section (Figure 10-4) provides the best separation of the holes for visualization purposes. It is important to note that the distance between the holes increases at depth and that interpreted structures in the lower portion of these holes is considered to dip towards the southwest.

Drill hole FCG20-02 (Figure 10-4) was drilled to the southwest along a plane connecting the Colorado Pit to Pack Rat zone and intersected a significant structural zone of high-grade gold mineralization higher up in the hole, the Juniper zone, than originally expected. Of 17 consecutive samples extending 21.9 m downhole, only one sample assayed less than 1.0 g/t Au with the highest sample grading 25.5 g/t Au (1.7 m sample). The mineralized interval graded 6.2 g/t Au over 21.9 m core length including 9.6 g/t Au over 12.0 m and included an intercept of 20.4 g/t Au over 3.2 m core length. As shown in Figure 10-4, this high-grade zone may be related to the upper FCG20-03 intercept. Further evaluation is required to properly determine the strike and dip of the mineralized structure.

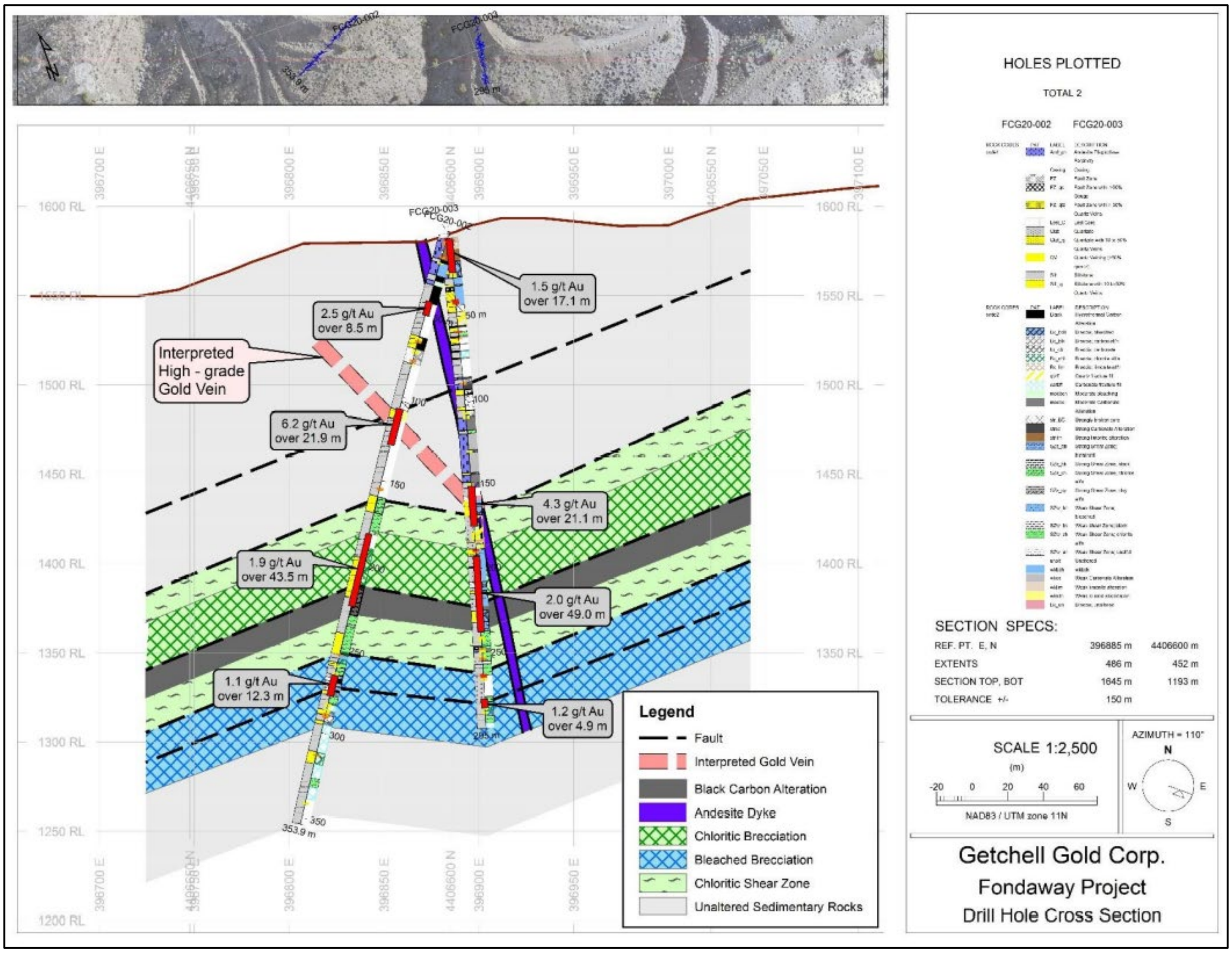


Figure 10-4: Drill Hole Section for FCG20-02 and FCG20-03

(Source: Getchell Gold, Frostad, 2021)

Note: Intercepts represent core length. True width can vary from 50% up to 100% of core length depending upon drill hole intersection angles.

Drill hole FCG20-02 encountered a wide mineralized structural zone between a drill depth of 150 and 300 meters. The mineralization was intersected where predicted by the geological model and down-dip from the Colorado Zone and named the Colorado SW Zone. The hole intersected 1.9 g/t Au over 43.5 m core length from 181.0 m to 224.5 m including 4.2 g/t Au over 14.9 m; and 1.1 g/t Au over 12.3 m core length from 265.6 to 277.9 m.

The broad Colorado SW structural zone that hosts the gold mineralization encountered in FCG20-02 is thought to have a true thickness of approximately 100 m and to dip shallowly to the southwest. The structural zone is comprised of strongly brecciated and sheared sedimentary rocks that are chloritized within the upper portion and bleached within the lower portion.

Drill hole FCG20-03 (Figure 10-4) was drilled towards the south and collared at the same drill pad as FCG20-02. As previously noted, a significant structural zone of high-grade gold mineralization that is considered to have been also intersected by FCG20-02, was drilled between 148.7 and 169.8 m and returned 4.3 g/t Au over 21.1 m core length including 8.7 g/t Au over 9.4 m and 14.6 g/t Au over 3.4 m. The hole then encountered a second major mineralized interval returning 2.0 g/t Au over 49.0 m core length from 188.3 to 237.3 m on trend with the Colorado SW zone. The location of the mineralized structure in this hole is approximately 120 m east-southeast of the FCG20-02 main structural zone intercepts.

Drill hole FCG20-04 (Figure 10-5) was collared north of where the Half Moon Vein is exposed on surface and drilled to the southwest. The hole was designed to pierce the Half Moon vein to characterize the mineralization and to extend the gold mineralization intersected in historical drill hole FC17-04 down-dip to the southwest (Figure 10-5). The hole encountered the high-grade Half Moon Shear Vein 108.1 m downhole and 54 m vertically below surface. In addition, a second notable gold intercept was encountered further down the hole that is interpreted to be a splay of the main Half Moon Gold Shear Vein. The Half Moon Shear Vein related gold intercepts returned 8.6 g/t Au over 9.8 m core length between 108.1 and 117.9 m and 2.7 g/t Au over 20.5 m core length from 128.5 and 149.0 m.

Further down the hole, FCG20-04 encountered a broad 144 m intercept of gold mineralization, newly identified as the North Fork Gold Zone, extending to the bottom of the hole with the final samples of hole FCG20-04 returning 2.6 g/t Au over 14.5 m core length between 478.5 m to 493.0 m suggesting the lower extent of the North Fork Gold Zone may not have been reached. The broad North Fork mineralization returned 2.5 g/t Au over 58.0 m core length between 383.0 m and 441.0 m, including 3.5 g/t Au over 36.1 m and 2.8 g/t Au over 13.4 m, and an additional 2.6 g/t Au over 14.5 m core length between 478.5 m and 493.0 m.

The North Fork Gold Zone is geologically modelled as a 40 to 50 m thick, shallowly dipping to the southwest, zone of gold mineralization and the results observed in FCG20-04 supported this model. In addition, the North Fork Gold Zone represented a 200 m step out to the southwest from drill hole FC17-04 and was open laterally and down-dip. There were no proximal drill holes that had targeted the North Fork Gold Zone's depth horizon. Of note is the location of historical drill hole FC17-05 (Figure 10-6) that ended within a significantly mineralized structure (3.48 g/t Au over 12.8 m core length). FC17-05 is 300 m distant from the end of drill hole FCG20-04, and was interpreted as the potential untested down-dip extension of the North Fork Gold Zone.

Drill holes FCG20-05 and 06, were stationed on the same pad near the canyon floor and drilled to the northeast along a plane connecting the Colorado Pit to Pack Rat zone and on plane with drill hole FCG20-02 (Figure 10-6). These two holes were designed to test the down-dip extension of the mineralization observed at surface at the historical Colorado Pit and the mineralization encountered in drill holes FGC20-02 and 03. Both drill holes, FCG20-05 and 06, encountered broad 100-metermeter-thick zones of gold mineralization within what is now referred to as the Colorado SW Zone.

Drill hole FCG20-05 (Figure 10-6) encountered the Colorado SW Zone between a downhole depth of 177.5 and 267.5 m. The hole intersected two mineralized intervals within the structural zone; 0.7 g/t Au over 31.8 m core length between 177.5 and 209.3 m and an additional 2.7 g/t Au over 51.8 m core length between 215.7 and 267.5 m. The lower intercept included 11.1 m core length of 4.4 g/t Au between 241.4 and 252.5 m. These strongly mineralized intervals are considered to represent a 150-200 m step out to the southwest from the mineralization intersected in drill hole FC20-02 and was open laterally and down-dip.

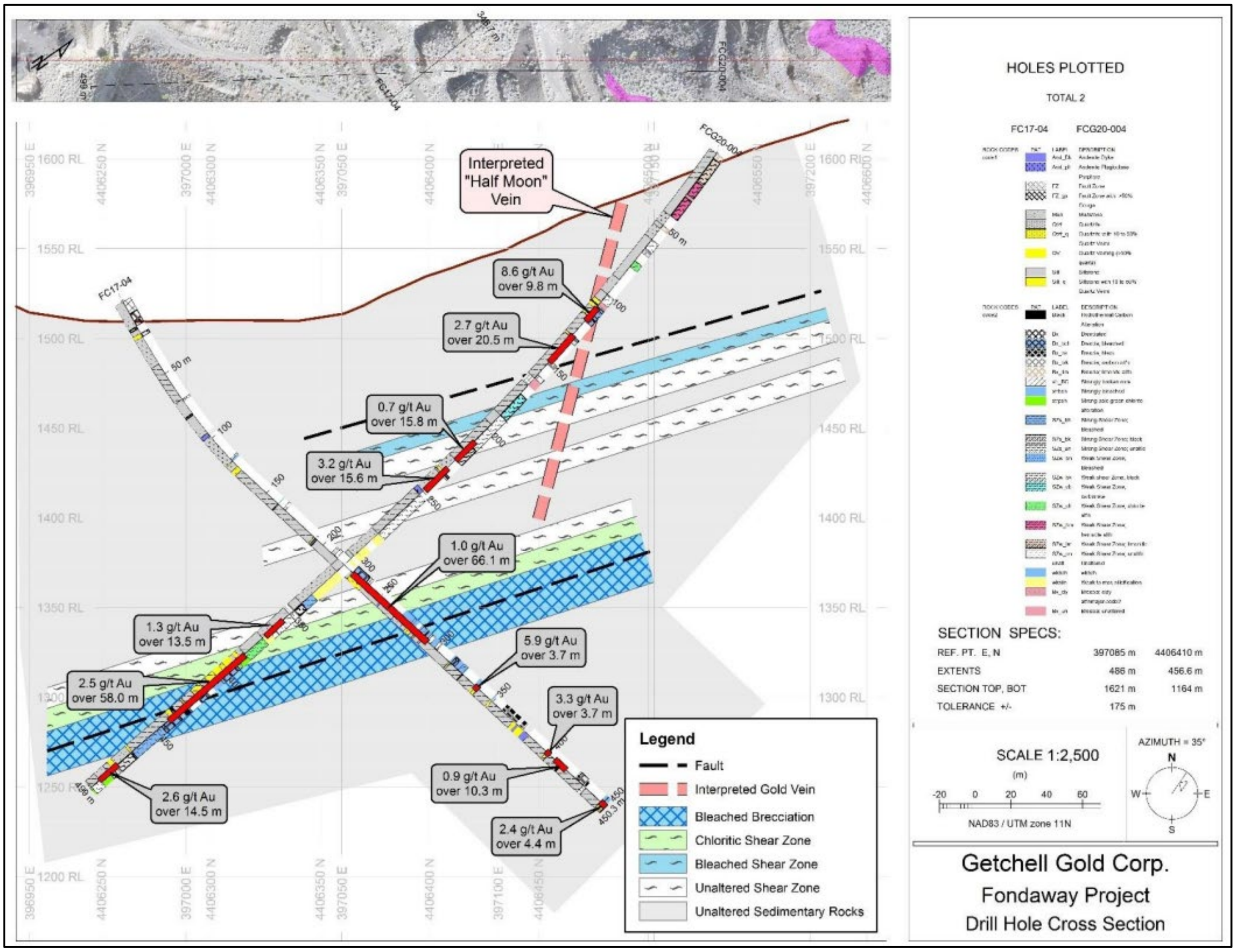


Figure 10-5: Drill Hole Section for FCG20-04

(Source: Getchell Gold, Frostad, 2021)

Note: Intercepts represent core length. True width can vary from 50% up to 100% of core length depending upon drill hole intersection angles.

Of note is historical drill hole FC17-01 (Figure 10-2) that encountered an intercept of gold mineralization at the bottom of the hole (46.6 m core length of 1.29 g/t Au). The FC17-01 intercept is located 250 m from drill hole FCG20-05, and is within and on plunge with the down-dip projection of the Colorado SW Zone suggesting the potential for a significant continuation of the mineralized structural zone.

Drill hole FCG20-06 (Figure 10-6) encountered the Colorado SW Zone between a depth of 165 and 285 m downhole. The hole intersected two mineralized intervals; 1.5 g/t Au over 37.7 m core length between 168.0 and 205.7m including 2.1 g/t Au over 19.2 m; and an additional 1.1 g/t Au over 38.3 m core length from 243.5 and 281.8m that included 2.5 g/t Au over 10.6 m.

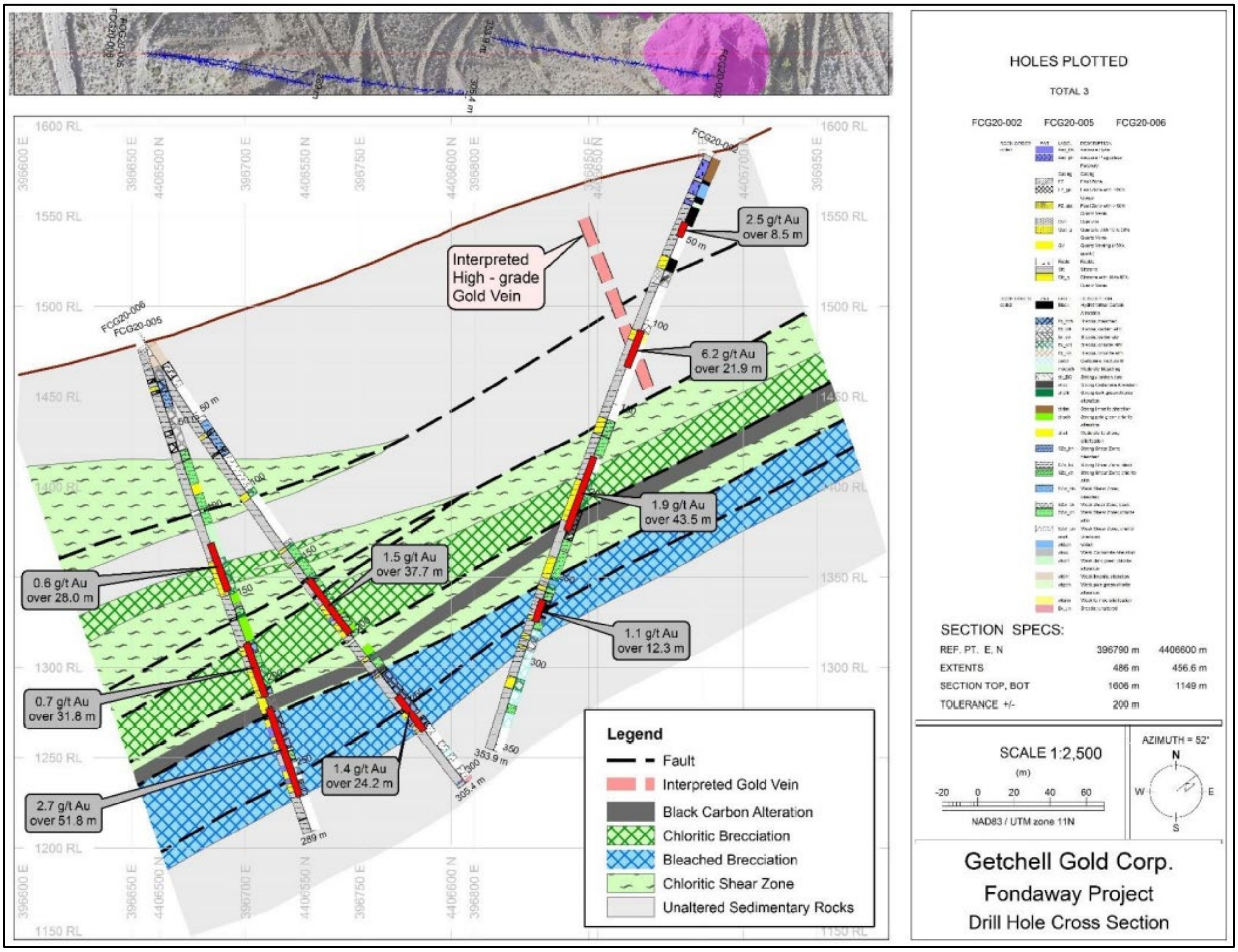


Figure 10-6: Drill Hole Section for FCG20-03, FCG20-05, and FCG20-06

(Source: Getchell Gold, Frostad, 2021)

Note: Intercepts represent core length. True width can vary from 50% up to 100% of core length depending upon drill hole intersection angles.

10.2.2 2021 Getchell Gold Drilling Summary and Results

The 2021 drill program was designed as follow-up on the 2020 discoveries referred to as Colorado SW, Juniper, and North Fork, located within the Central Area Gold Zone of the Fondaway Canyon Project. The program served to further define and extend the new zones.

The 2021 exploration program consisted of a diamond drill program with ten diamond drill holes completed (3,875 m) and one drill hole abandoned (FCG22-010; 95m) for a total of 3,970 m (Table 10-4; Figure 10-7).

Table 10-4: Getchell Gold 2021 Collar Information

Hole ID	Year	UTM Northing* (m)	UTM Easting* (m)	Elevation (m)	Elevation (ft)	Azimuth (◦)	Dip (◦)	Depth (m)	Depth (ft)
FCG21-07	2021	4406680	396913	1,585	5,200	264.6	-72	264.6	868.2
FCG21-08	2021	4406680	396913	1,585	5,200	459.2	-62	459.2	1,506.6
FCG21-09	2021	4406467	397119	1,567	5,141	506.6	-48	506.6	1,662.2
FCG21-10	2021	4406467	397119	1,567	5,141	94.6	-57	94.6	310.4
FCG21-10A	2021	4406467	397119	1,567	5,141	522.1	-57	522.1	1,713.0
FCG21-11	2021	4406680	396913	1,585	5,200	493.2	-58	493.2	1,618.2
FCG21-12	2021	4406495	396655	1,482	4,862	356.0	-80	356.0	1,168.0
FCG21-13	2021	4406680	396913	1,585	5,200	335.0	-80	335.0	1,099.1
FCG21-14	2021	4406680	396913	1,585	5,200	127.7	-66	127.7	419.0
FCG21-15	2021	4406495	396655	1,482	4,862	437.1	-87	437.1	1,434.1
FCG21-16	2021	4406292	396966	1,509	4,951	373.8	-80	373.8	1,226.0
							TOTAL	**3,970**	**13,025**

** Coordinate system: NAD 1983 / UTM Zone 11N*

The 2021 drill program was designed with four objectives in mind: 1) to test the high-grade Juniper zone, 2) determine the continuity of the Colorado SW Zone between drill holes FCG20-02 and FCG20-06, 3) extend the Colorado SE Zone further to the southeast, and 4) to follow-up the discovery by FCG20-04 of the North Fork Zone.

The Colorado SW Zone was successfully intersected and extended during the 2021 drilling by six of the seven drill holes that targeted the mineralized structure. Drill hole FCG21-08, intersected the Colorado SW Zone for over 200 m with mineralized intervals that included: 4.2 g/t Au over 27.5 m core length, 2.8 g/t Au over 24.5 m core length, 1.4 g/t over 30.7 m core length, and 1.3 g/t Au over 16.8 m core length. The hole also intersected the Juniper zone returning 4.7 g/t Au over 25.9 m core length.

The North Fork Zone was targeted by three drill holes during the 2021 program with all holes intersecting the mineralized structure. The final hole of the program, FCG21-16, returned high-grade intercepts (core length) that included 6.3 g/t Au over 50.7 m, 3.1 g/t Au over 33.4 m and 2.1 g/t Au over 14.1 m.

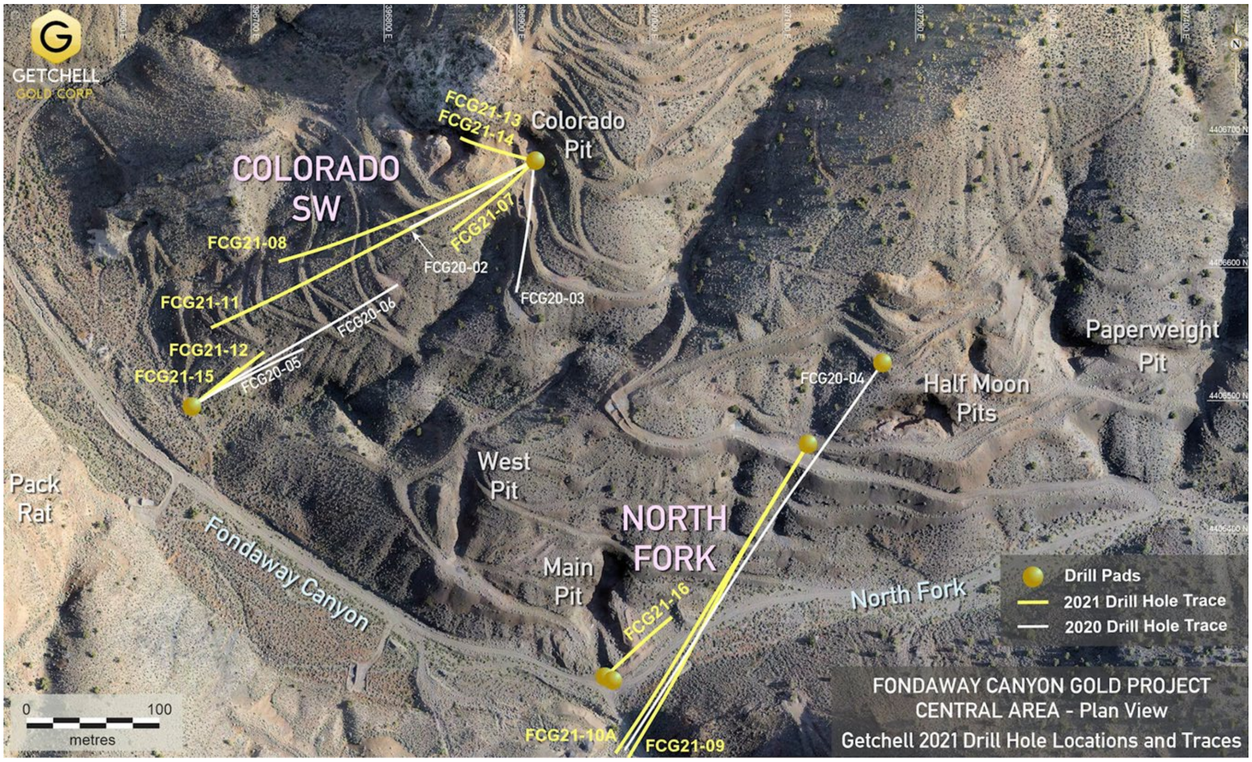


Figure 10-7: Getchell Gold 2021 Drill Hole Location Map

(Source: Getchell Gold, 2021)

10.2.2.1 Results and Highlights

Table 10-5 provides highlights of the gold assay results from the 2021 drill program. Summary intervals provided are average gold grade over core length for all intervals and holes.

Table 10-5: 2021 Getchell Gold Drilling Program Highlights

Zone	Drill Hole	Au (g/t)	Interval* (m)	Depth From (m)	Depth To (m)	Depth From (ft)	Depth To (ft)
Colorado	FCG21-07	2.9	3.2	143.3	146.5	470.1	480.6
	FCG21-07	2.2	5.1	155.6	160.7	510.5	527.2
	FCG21-07	3.8	3.2	167.2	170.4	548.6	559.1
	FCG21-07	3.0	33.0	209.1	242.1	686.0	794.3
	including	7.8	4.6	214.2	218.8	702.8	717.9
Colorado	FCG21-08	1.9	6.1	83.2	89.3	273.0	293.0
	FCG21-08	4.7	25.9	104.0	129.9	341.2	426.2
	including	11.4	5.5	124.4	129.9	408.1	426.2
	FCG21-08	0.6	30.0	190.1	220.1	623.7	722.1
	FCG21-08	4.2	27.5	223.4	250.9	732.9	823.2
	including	13.0	4.5	243.9	248.2	800.2	814.3
	FCG21-08	2.8	24.5	261.5	286.0	857.9	938.3
	FCG21-08	0.5	20.3	299.0	319.3	981.0	1,047.6
	FCG21-08	1.4	30.7	323.5	354.2	1,061.4	1,162.1
	including	5.1	5.6	345.8	351.4	1,134.5	1,152.9
	FCG21-08	1.3	16.8	274.0	390.8	899.0	1,282.2
Colorado	FCG21-11	1.5	5.40	86.5	91.9	283.8	301.5
	FCG21-11	8.8	8.2	107.8	116.0	353.7	380.6
	FCG21-11	1.4	14.9	250.3	265.2	821.2	870.1
	FCG21-11	1.0	52.2	274.4	326.9	900.3	1,072.5
	FCG21-11	2.2	9.1	333.1	342.2	1,092.9	1,122.7
	FCG21-11	0.8	10.5	347.7	358.2	1,140.8	1,175.2
	FCG21-11	0.5	8.8	362.8	371.6	1,190.3	1,219.2
	FCG21-11	1.4	9.1	382.3	391.4	1,254.3	1,284.1
	FCG21-11	0.7	5.0	424.6	429.6	1,393.0	1,409.5
	FCG21-11	0.6	6.6	459.9	466.5	1,508.9	1,530.5
	FCG21-11	2.0	9.2	484.0	493.2	1,587.9	1,618.1
Colorado	FCG21-12	0.9	11.6	139.5	151.1	457.7	495.7
	FCG21-12	0.9	5.0	198.3	203.3	650.6	667.0
	FCG21-12	6.3	3.6	224.2	228.0	735.6	748.0
	FCG21-12	2.5	24.5	235.5	260.0	772.6	853.0
	FCG21-12	1.7	3.5	263.5	267.0	864.5	876.0
	FCG21-12	1.6	25.5	271.9	297.4	892.1	975.7
	FCG21-12	0.8	14.6	301.9	316.5	990.5	1,038.4
Colorado	FCG21-13	1.7	6.4	1.0	7.4	3.3	24.38
	FCG21-13	2.4	5.8	16.7	22.5	54.8	73.8
	FCG21-13	0.9	20.1	30.0	50.1	98.4	164.4
	FCG21-13	9.3	1.9	72.5	74.4	237.9	244.1
	FCG21-13	5.7	11.6	85.0	96.6	278.9	316.9

Zone	Drill Hole	Au (g/t)	Interval* (m)	Depth From (m)	Depth To (m)	Depth From (ft)	Depth To (ft)
	FCG21-13	1.0	19.7	170.2	189.9	558.4	623.0
	including	7.8	1.6	178.6	180.2	586.0	591.2
	FCG21-13	1.9	11.8	197.9	209.7	649.3	688.0
	FCG21-13	1.2	29.1	224.2	253.3	735.6	831.0
	including	2.8	8.7	244.6	253.3	802.5	831.0
Colorado	FCG21-14	2.6	18.5	2.9	21.4	9.5	70.2
	including	6.8	5.4	12.6	18.0	41.3	59.1
Colorado	FCG21-15	3.3	10.6	134.4	145.0	440.9	475.7
	including	17.6	1.6	135.2	13.8	443.6	45.3
	FCG21-15	2.3	3.9	215.5	219.4	707.0	719.8
	FCG21-15	1.2	33.6	249.6	283.2	818.9	929.1
	FCG21-15	1.9	26.1	288.6	315.0	946.9	1,033.5
	including	7.4	2.6	305.1	307.7	1,001.0	1,009.5
	FCG21-15	1.6	7.7	328.9	336.6	1,079.1	1,104.3
	FCG21-15	1.5	12.6	372.1	384.7	1,220.8	1,262.1
North Fork	FCG21-09	2.4	7.4	227.2	234.6	745.4	769.7
	FCG21-09	1.2	32.6	272.5	305.1	894.0	1,001.0
	including	2.0	14.1	279.8	293.9	918.0	964.2
	FCG21-09	1.3	13.3	341.0	354.1	1,118.8	1,161.8
	FCG21-09	1.1	4.2	401.1	405.3	1,315.9	1,329.7
	FCG21-09	4.1	5.4	422.2	427.6	1,385.2	1,402. 9
	FCG21-09	1.4	5.1	477.9	483.0	1,567.9	1,584.7
North Fork	FCG21-10A	4.2	3.6	52.9	56.5	173.6	185.4
	FCG21-10A	2.1	7.7	244.0	251.7	800.5	825.8
	FCG21-10A	3.0	41.8	275.5	317.3	903.9	1,041.0
	including	47.0	1.5	293.3	294.8	962.3	967.2
	FCG21-10A	4.6	9.8	326.4	336.2	1,070.9	1,103.0
	FCG21-10A	1.0	14.0	343.4	357.7	1,126.6	1,173.6
	FCG21-10A	2.1	12.1	401.0	413.1	1,315.6	1,355.3
North Fork	FCG21-16	2.1	14.1	75.6	89.7	248.0	294.3
	FCG21-16	6.3	50.7	117.5	168.2	385.5	551.8
	including	10.4	25	139.9	164.9	459.0	541.0
	FCG21-16	5.0	6.7	191.9	198.6	629.6	651.6
	FCG21-16	1.7	4.3	206.5	210.8	677.5	691.6
	FCG21-16	3.1	33.4	265	298.4	869.4	979.0
	FCG21-16	1.6	4.1	329.4	333	1,080.7	1,092.5
	FCG21-16	4.5	2.7	354.9	357.60	1,164.4	1,173.2

**Note: Intervals represent core length. True width can vary from 50% up to 100% of core length depending upon drill hole intersection angles.*

Five 2021 drill holes, FCG21-07, 08, 11, 12 and 15, were completed on the same section as 2020 drill holes FCG20-02, 05 and 06 (Figure 10-8). The significant 2021 assays returned from drilling along Section 1 are provided in Table 10-5. Downhole sample interval lengths within this report are not representative of true width and true width will be less than the reported core length intervals by a certain factor.

FCG21-07, the first drill hole of the 2021 program, was drilled southwest from the Colorado Pit with two holes from the 2020 drill program, FCG20-02 and 03, being drilled from the same pad. The gold intercepts encountered in drill holes FCG20-02 and FCG20-03, 1.9 g/t Au over 43.5 m and 2.0 g/t Au over 49.0 m core length respectively, are 75 m apart from each other and FCG21-07 was drilled between these two 2020 gold intercepts to establish the lateral continuity of the Colorado SW Zone across this broad distance. The hole intersected a higher-grade gold interval than the neighboring drill holes, grading 3.0 g/t Au over 33.0 m core length of uninterrupted mineralization including an interval grading 7.8 g/t Au over 4.6 m core length.

Drill hole FC21-08 was drilled from the same drill pad as FCG21-07 and was designed to test the Colorado SW Zone down-dip, and to the west, of FCG20-02. The hole intersected the Colorado SW Zone over a distance greater than 200 m downhole. Four significant core length intercepts include: 4.2 g/t Au over 27.5 m core length from 223.4 to 250.9 m that included 13.0 g/t Au over 4.3 m from 243.9-248.2 m, 2.8 g/t Au over 24.5 m from 261.5 to 286.0 m, 1.4 g/t Au over 30.7 m from 323.5 to 354.2 m, and 1.3 g/t Au over 16.8 m from 374.0 to 390.8 m.

Drill hole FCG21-08 also tested the Juniper Zone, located within 100 m of surface, with a 10 m vertical step out from FCG20-02. The hole intersected the Juniper Zone between 104.0-129.9 m returning 4.7 g/t Au over 25.9 m that included 11.4 g/t Au over 5.5 m core length. The Juniper Zone was discovered in 2020 by FCG20-02 that intersected 6.2 g/t Au over 21.9 m that included 20.4 g/t Au over 3.2 m core length.

FCG21-11 was designed to extend the Colorado SW gold zone approximately 30 to 50 m to the southeast down-dip of drill hole FCG21-08 and 40 m to the northwest on-strike from drill holes FCG20-05 and FCG20-06. The hole was collared at the Colorado Pit on the same drill pad as FCG21-08 and drilled towards the southwest. Multiple significant gold intercepts were intersected within the Colorado SW Zone over a downhole depth greater than 240 meters. Three significant FCG21-11 core length intercepts include: 1.4 g/t Au over 14.9 m from 250.3 to 265.2 m, 1.0 g/t Au over 52.5 m from 274.4 to 326.9 m, and 2.2 g/t Au over 9.1 m from 333.1 to 342.2 m.

FCG21-11 was also designed to test the near surface high grade Juniper gold zone down dip from FCG21-08 that reported 4.7 g/t Au over 25.9 m. The hole intersected a substantially higher-grade core length interval reporting 8.8 g/t Au over 8.2 m from 107.8 to 116.0 m including one sample that graded 22.9 g/t Au over 1.7 m.

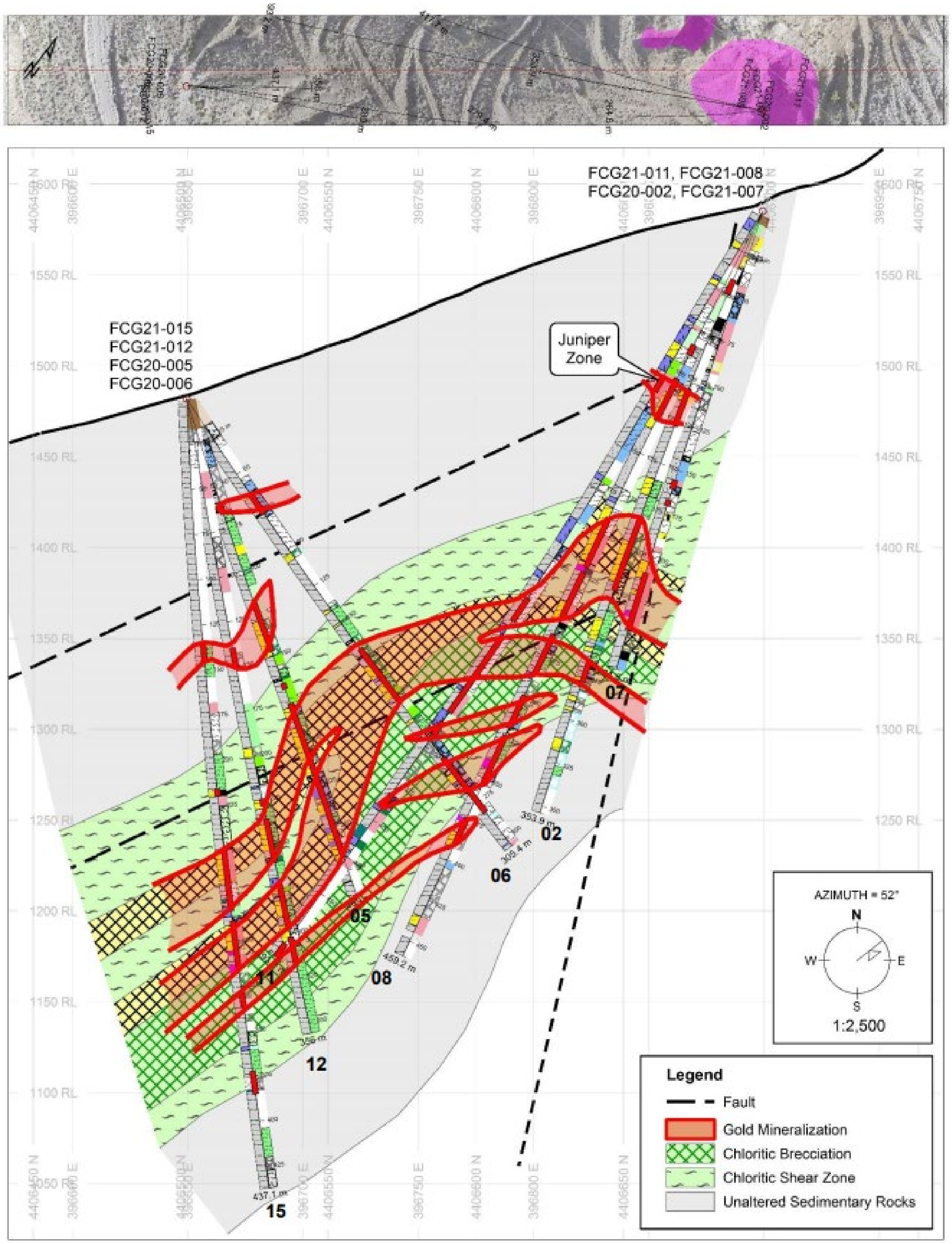


Figure 10-8: Colorado SW Zone – Section 1

(Source: Getchell Gold, Frostad, 2022)

FCG21-12 was collared near the canyon floor, drilled steeply to the northeast, and was designed to test the down-dip extent of the Colorado SW gold mineralization encountered in FCG20-05 with a 40-metermeter step out. The hole intersected the Colorado SW Zone of gold mineralization over 92 m with core length intercepts that included: 6.3 g/t Au over 3.6 m from 224.4 to 228.0 m, 2.5 g/t Au over 24.5 m from 235.5 to 260.0 m, and 1.6 g/t Au over 25.5 m from 271.9 to 297.4 m.

FCG21-15 was collared at the same location as FCG21-12 and was also drilled steeply to the northeast. The drill hole was designed to test the down-dip extent of the Colorado SW gold mineralization encountered in FCG21-12 with a 30-meter step out. FCG21-15 intersected the Colorado SW zone of gold mineralization over an 87 m downhole distance (Table 10-5; Figure 10-8) with three notable core length drill intercepts including: 1.2 g/t Au over 33.6 m from 249.6 to 283.2 m, 1.9 g/t Au over 26.4 m from 288.6 to 315.0 m, and 1.6 g/t Au over 7.7 m from 328.9 to 336.6 m.

A significant intercept was encountered by FCG21-15 higher up the hole returning 3.3 g/t Au over 10.6 m core length including 17.6 g/t Au over 1.6 m. The extent and orientation of this lens of mineralization will need to be determined by additional drilling. The drill hole was extended well below the modelled envelope of the Colorado SW Zone and encountered a notable intercept grading 1.5 g/t Au over 12.6 m at a downhole depth of 370 m. The intercept represents the deepest gold interval encountered to date and reinforces the untested potential of the mineralizing system at Fondaway Canyon.

Two 2021 drill holes, FCG21-13 and 14, were collared at the Colorado pit using similar azimuths of 284 degrees. Drill hole FCG21-13 was drilled with a dip of -80 degrees while FCG21-14 was drilled with a dip of -66 degrees (Figure 10-9). The significant 2021 assays returned from drilling along Section 2 are provided in Table 10-5.

FCG21-13 was designed to test the gold mineralization directly under the Colorado Pit exposed at surface (the Colorado Zone), the Juniper shear zone and the Colorado SW gold zone. The Colorado Zone mineralization was encountered at the top of the hole returning 1.7 g/t Au over 6.4 m core length from 1.0 to 7.4 m, 2.4 g/t Au over 5.8 m from 16.7 to 22.5 m, and 0.9 g/t Au over 20.1 m from 30.0 to 50.1 m. The high-grade Juniper zone was intersected with two core length intervals: 9.3 g/t Au over 1.9 m from 72.5 to 74.4 m, and 5.7 g/t Au over 11.6 m from 85.0 to 96.6 m. The Colorado SW Zone was intersected with multiple intervals over a downhole depth of approximately 100 meters including 1.0 g/t Au over 19.7 m from 170.2 to 189.9 m, 1.9 g/t Au over 11.8 m from 197.9 to 209.7 m, and 1.2 g/t Au over 29.1 m from 224.2 to 253.3 m.

FCG21-14 was designed to test the gold mineralization below the Colorado Pit and determine the boundary location of a known limestone fault block to assist with resource modeling. Immediately situated to the west of the Colorado Pit, the historic Upper Quick-Tung Tungsten Mine is hosted within an isolated fault block composed of marbleized limestone. The marble unit is an isolated and relatively thin thrust sheet in a fault relationship with the surrounding siltstone/argillite unit host to the Colorado, Juniper, and Colorado SW gold zones. Gold mineralization is present in the adjoining siltstone/argillite both at surface to the north and east of the marble block and exists at depth below the lower contact as demonstrated by numerous historic drill holes.

FCG21-14 intersected the Colorado Zone at surface returning 2.6 g/t Au over 18.5 m core length including 6.8 g/t Au over 5.4 m from 12.6 to 18.0 m drill depth. Shortly downhole from the above gold intersection, the drill crossed into the fault contact boundary zone and then penetrated the marble block (Figure 10-9). The hole was terminated before reaching the targeted depth due to the extreme hardness of the intensely silicified marble unit. The depth extent and geometry of the marble block has yet to be determined.

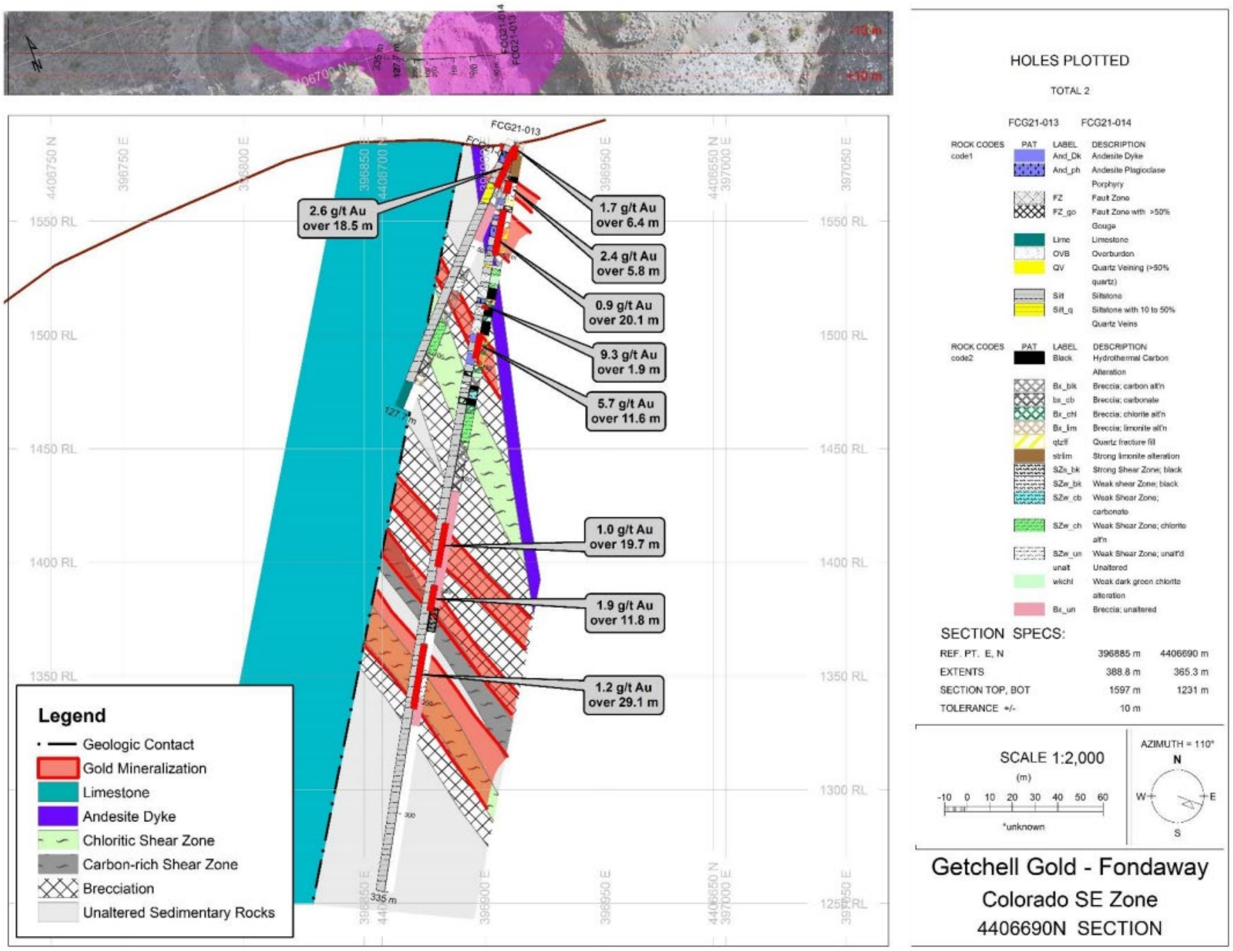


Figure 10-9: Colorado SW Zone – Section 2

(Source: Getchell Gold, Frostad, 2022)

Note: Intercepts represent core length. True width can vary from 50% up to 100% of core length depending upon drill hole intersection angles.

Three 2021 drill holes, FCG21-09, 10A and 16 followed-up on the North Fork gold mineralization discovered in 2020 by drill hole FCG20-04 (Figure 10-10). FCG21-10 was abandoned after drilling 94.6 m due to drilling difficulties and recollared as FCG21-10A. The significant 2021 assays returned from the North Fork drilling are provided in Table 10-5.

FCG21-09 was designed to parallel drill hole FCG21-04, spaced 50 m above, and to test the down dip extent of the North Fork Zone. Drill hole FCG21-09 intersected a broad zone of gold mineralization grading 1.2 g/t Au over 32.6 m core length at a higher elevation than initially projected for the North Fork Zone. The hole then intersected additional mineralization including 1.3 g/t Au over 13.1 m and 4.1 g/t Au over 5.4 m that is considered to represent the North Fork Zone.

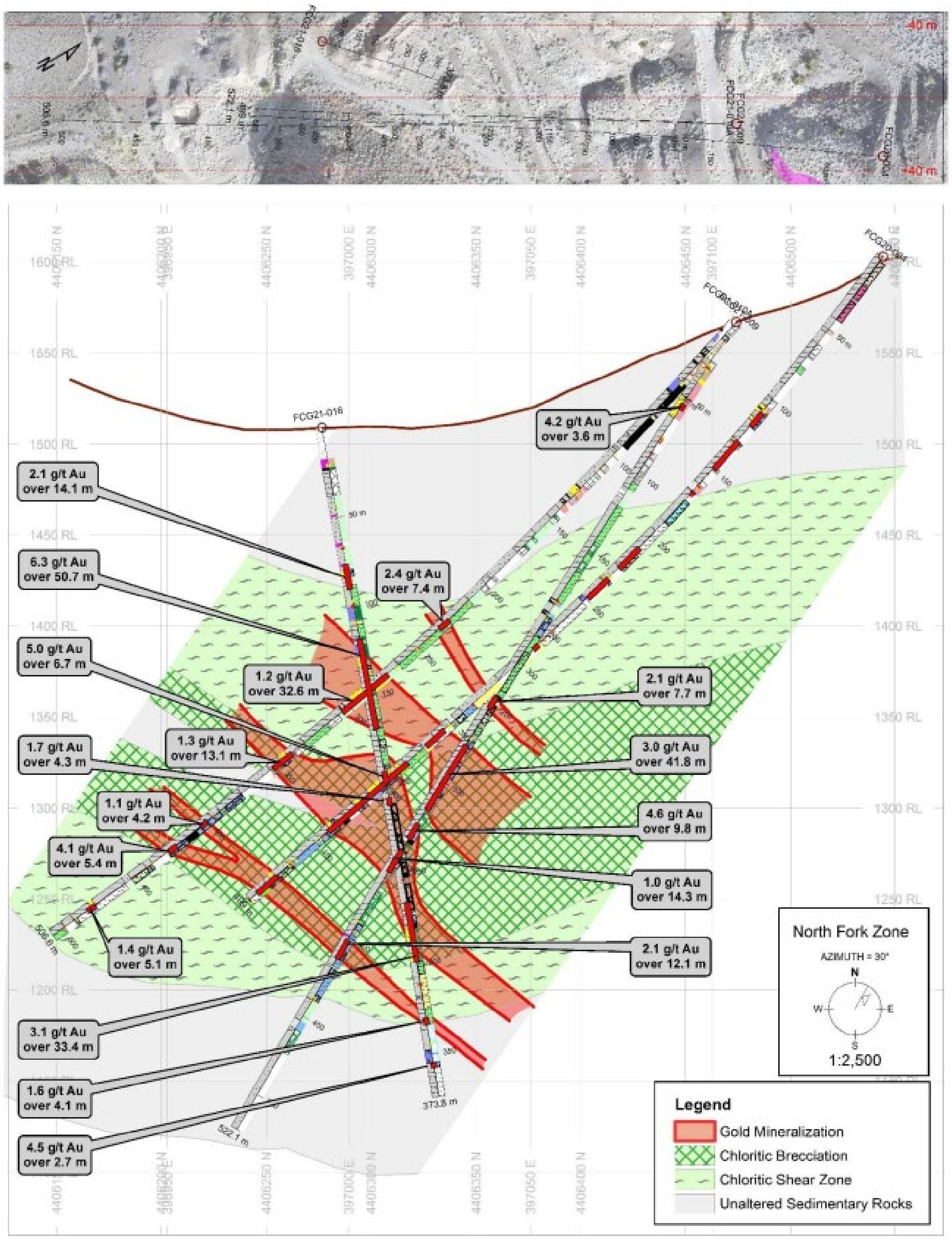


Figure 10-10: North Fork Zone Section

(Source: Getchell Gold, Frostad, 2022)

Note: Intercepts represent core length. True width can vary from 50% up to 100% of core length depending upon drill hole intersection angles.

FCG21-10A intersected the North Fork Zone mineralization over approximately 80 m core length (Table 10-5). One interval graded 3.0 g/t Au over 41.6 m core length that included 47.0 g/t Au over 1.5 m while a second interval, 9.1 m lower in the drill hole, returned 4.6 g/t Au over 9.8 m core length.

FCG21-16, the last drill hole of the 2022 drill program, stationed on the canyon floor at the junction of Fondaway Canyon and the North Fork branch, was drilled steeply to the northeast and designed to further delineate the North Fork mineralized zone. The hole intersected core length intervals of 2.1 g/t Au over 14.1 m from 75.6 to 89.7 m, 6.3 g/t Au over 50.7 m from 117.5 to 168.2 m that includes 10.4 g/t Au over 25.0 m, and 3.1 g/t Au over 33.4 m from 265.0 to 298.4 m that included two internal zones grading 9.6 g/t Au over 3.0 m and 6.1 g/t Au over 6.1 m.

Notably, drill hole FCG21-16 returned the greatest 'gold grade x thickness' interval (10.4 g/t Au over 25.0 m) in the history of gold exploration at the Fondaway Canyon Project.

10.2.3 2022 Getchell Gold Drilling Summary and Results

The 2022 drill program was designed to follow-up on high-grade gold discoveries from the previous year, and to continue to bracket and expand upon the Colorado and North Fork mineralization, and consisted of a diamond drill program with 12 drill holes completed (4,583 m) and one abandoned hole (FCG22-017; 70m) for a total of 4,653 m (Table 10-6 and Figure 10-11).

All completed 2022 drill holes, FCG22-017A through FCG22-028, are included in the Mineral Resource Estimate presented in Section 14 of this Technical Report.

Table 10-6: Getchell Gold 2022 Drill Hole Locations

Hole ID	Year	UTM Northing* (m)	UTM Easting* (m)	Elevation (m)	Elevation (ft)	Azimuth (◦)	Dip (◦)	Depth (m)	Depth (ft)
FCG22-017	2022	4406289	396972	1508	4948	13	-77	70.1	230.0
FCG22-017A	2022	4406289	396972	1508	4948	13	-77	348.7	1144.1
FCG22-018	2022	4406289	396972	1508	4948	50	-70	437.1	1434.1
FCG22-019	2022	4406289	396972	1508	4948	360	-90	321.9	1056.2
FCG22-020	2022	4406680	396913	1585	5200	360	-90	386.2	1267.1
FCG22-021	2022	4406541	396815	1529	5016	7	-58	308.8	1013.2
FCG22-022	2022	4406289	396972	1508	4948	78	-75	461.2	1503.2
FCG22-023	2022	4406289	396972	1508	4948	43	-70	484	1588.0
FCG22-024	2022	4406320	394674	1319	4327	163	-76	290.5	953.1
FCG22-025	2022	4406289	396972	1508	4948	50	-65	491.6	1612.9
FCG22-026	2022	4406495	396655	1482	4862	88	-72	362	1187.7
FCG22-027	2022	4406495	396655	1482	4862	56	-47	237.3	778.6
FCG22-028	2022	4406537	396821	1529	5016	264	-70	453.2	1486.9
							TOTAL	**4,653**	**15,266**

** Coordinate system: NAD 1983 / UTM Zone 11N*

10.2.3.1 Results and Highlights

Table 10-7 provides highlights of the gold assay results from the 2022 drill program. Summary intervals provided are average gold grade over core length for all intervals and drill holes.

Table 10-7: 2022 Getchell Gold Drilling Program Highlights

Zone	Drill Hole	Au (g/t)	Interval* (m)	Depth From (m)	Depth To (m)	Depth From (ft)	Depth To (ft)
North Fork	FCG22-17A	5.4	51.9	66.1	118.0	216.9	387.1
	including	12.2	5.3	72.4	77.7	237.5	254.9
	including	17.7	9.9	94.7	104.6	310.7	343.2
	FCG22-17A	2.0	22.9	129.1	152.0	423.6	498.7
	FCG22-17A	1.9	15.9	169.9	185.8	557.4	609.6
North Fork	FCG22-18	4.1	6.0	108.5	114.5	356.0	375.7
	FCG22-18	2.5	43.4	180.6	224.0	592.5	734.9
	including	5.8	7.1	188.7	195.8	619.1	642.4
	FCG22-18	4.8	5.9	246.5	252.4	808.7	828.1
	FCG22-18	2.0	29.6	256.9	286.5	842.9	940.00
	FCG22-18	3.4	3.2	290.2	293.4	952.1	962.6
	FCG22-18	4.8	12.1	327.4	336.5	1,074.2	1,104.0
	including	10.5	4.9	333.0	337.9	1,092.5	1,108.6
	FCG22-18	1.4	27.7	344.4	372.1	1,129.9	1,220.8
	FCG22-18	2.0	22.1	377.9	400.0	1,239.8	1,312.3
North Fork	FCG22-19	0.6	8.3	19.2	27.5	63.0	90.2
	FCG22-19	0.7	5.6	105.8	111.4	347.1	365.5
	FCG22-19	1.8	107.5	120.0	227.5	393.7	746.4
	including	1.4	9.3	120.0	129.3	393.7	424.2
	including	2.9	32.9	139.9	172.8	459.0	566.9
	including	2.0	4.9	176.8	181.7	580.1	596.1
	including	2.3	10.6	185.8	196.4	609.6	644.4
	including	2.0	24.8	202.7	227.5	665.0	746.4
	FCG22-19	2.1	10.8	240.1	250.9	787.7	823.2
	FCG22-19	2.5	4.1	265.5	269.6	871.1	884.5
Colorado	FCG22-20	0.9	15.3	1.8	17.1	5.9	56.1
	FCG22-20	1.4	10.2	104.9	115.1	344.2	377.6
	FCG22-20	0.8	7.4	119.8	127.2	393.1	417.3
	FCG22-20	1.7	56.6	160.4	217.0	526.3	712.0
Colorado	FCG22-21	1.2	4.8	139.4	144.2	457.4	473.1
	FCG22-21	0.9	74.3	191.7	266.0	629.0	872.8
	including	2.7	10.8	213.8	224.6	701.5	736.9
North Fork	FCG22-22	3.0	59.3	159.0	218.3	521.7	716.2
	including	8.8	8.1	172.0	180.1	564.3	590.9
	FCG22-22	1.5	5.4	224.0	229.4	734.9	752.7
	FCG22-22	2.4	21.7	238.2	259.9	781.5	852.7
	including	7.1	5.3	239.4	244.7	785.5	802.9
	FCG22-22	0.8	41.6	290.7	332.3	953.8	1090.3

Zone	Drill Hole	Au (g/t)	Interval* (m)	Depth From (m)	Depth To (m)	Depth From (ft)	Depth To (ft)
	FCG22-22	1.1	25.8	370.8	396.6	1216.6	1301.2
North Fork	FCG22-23	1.8	12.8	65.1	77.9	213.6	255.6
	FCG22-23	3.4	44.6	164.1	211.5	538.4	693.9
	FCG22-23	1.5	7.1	245.5	252.6	805.5	828.8
	FCG22-23	2.2	7.1	308.5	316.8	1012.2	1039.4
Pediment	FCG22-24	0.7	3.0	140.0	143.0	459.3	469.2
	FCG22-24	0.6	1.6	239.3	240.9	785.1	790.4
North Fork	FCG22-25	3.4	31.4	254.4	285.8	834.7	937.7
	including	14.1	2.2	254.7	256.9	835.7	842. 9
	FCG22-25	1.3	17.4	406.7	424.1	1334.4	1391.5
Colorado	FCG22-26	1.8	29.4	108.3	137.7	355.3	451.8
	FCG22-26	0.8	18.7	175.1	193.8	574.5	635.8
	FCG22-26	1.1	83.8	229.8	313.6	754.0	1028.9
	including	5.4	4.8	247.9	252.7	813.4	829.1
Colorado	FCG22-27	1.2	29.9	143.1	173.0	469.5	567.6
	FCG22-27	0.9	6.1	227.3	233.4	745.8	765.7
Colorado	FCG22-28	0.9	17.9	139.0	156.9	456.1	514.8
	FCG22-28	0.8	98.0	182.5	280.5	598.8	920.3
	FCG22-28	1.3	58.0	293.9	351.9	964.3	1154.5

**Note: Intervals represent core length. True width can vary from 50% up to 100% of core length depending upon drill hole intersection angles.*

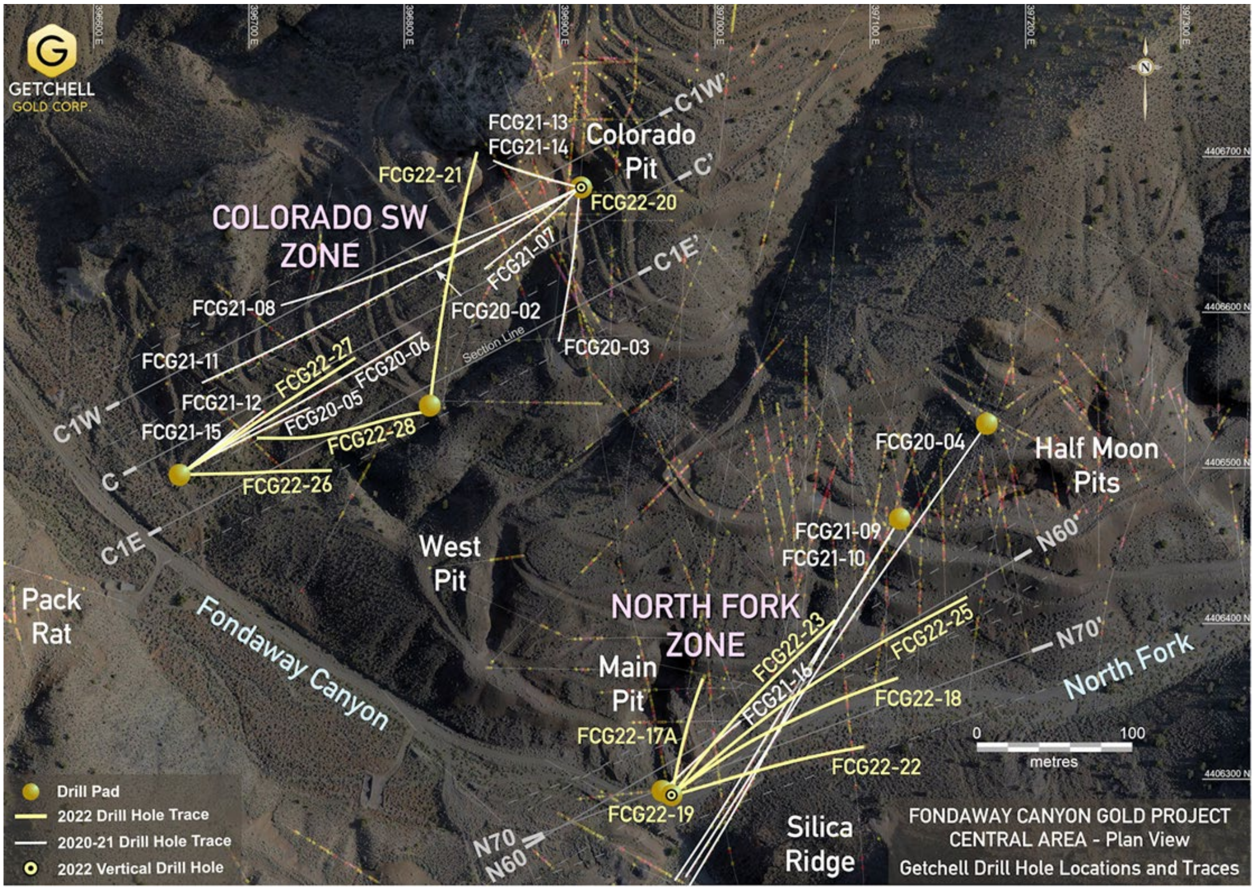


Figure 10-11: Fondaway Canyon Central Area Drill Location Plan Map Highlighting Getchell Gold 2022 Drill Holes

(Source: Getchell Gold, 2022)

Drill hole FCG22-17 is the first in a series of holes tasked with delineating the high-grade gold discovered by FCG21-16. FCG21-16 encountered a high-grade gold interval grading 6.3 g/t Au over 50.7 m core length (117.5-168.2 m drill depth) that includes 10.4 g/t Au over 25.0 m core length (139.9-164.9 m). This latter interval contained 12 samples reporting >10 g/t Au revealing strong internal high-grade gold consistency.

FCG22-17 was collared on the canyon floor, at the junction of Fondaway Canyon and the North Fork branch, on the same drill pad as drill hole FCG21-16 (Figure 10-2 and Figure 10-11), however this hole was abandoned after drilling 70.1 m due to deviation beyond acceptable parameters and recollared as FCG21-17A. FCG22-17A was designed to target the North Fork mineralized zone as a 25 m step out to the northwest from the high-grade intercept encountered in FCG21-16. FCG22-17A intersected significant gold mineralization grading 5.4 g/t Au over 51.9 m core length at a shallow downhole depth of 66.1 m including an exceptionally high-grade gold zone grading 17.7 g/t Au over 9.9 m core length (94.7m - 104.6 m; Figure 10-12 and Figure 10-13). This latter interval contains ten consecutive samples reporting >9 g/t Au revealing strong internal high-grade gold consistency. The 51.9 m interval was closely followed by two intervals grading 2.0 g/t Au over 22.9 m (129.1 m - 152.0 m) and 1.9 g/t Au over 15.9 m (169.9 m - 185.8 m) that combined for an overall gold mineralized zone spanning 120 m downhole.

Drill hole FCG22-18 was designed as the second hole to follow up on the high-grade gold discovered by FCG21-16.

FCG22-18 was collared on the canyon floor, at the junction of Fondaway Canyon and the North Fork branch, on the same drill pad as drill hole FCG21-16 (Figure 10-2 and Figure 10-11). FCG22-18 targeted the North Fork mineralized zone as a 30 m step out to the northeast from the high-grade intercept encountered in FCG21-16 (Figure 10-14). FCG22-18 intersected multiple significant intervals of gold mineralization, encountered from 180.6 to 400 m downhole (Figure 10-14). The broader core length intervals graded 2.5 g/t Au over 43.4 m, 2.0 g/t Au over 29.6 m, 4.8 g/t Au over 12.1 m,1.4 g/t Au over 27.7 m, and 2.0 g/t Au over 22.1 m (detailed in Table 10-6). The latter gold intervals, extending over a 72.6 m downhole distance, were encountered in an area outside and to the east of previous drilling, and 75 m distant from the nearest drill hole.

Drill hole FCG22-19 was designed as the third hole bracketing the high-grade gold discovered by FCG21-16. FCG22-19, drilled vertically from the same drill pad as drill hole FCG21-16 (Figure 10-2, Figure 10-11, and Figure 10-15), targeted the North Fork mineralized zone as a 30 m step out to the southwest.

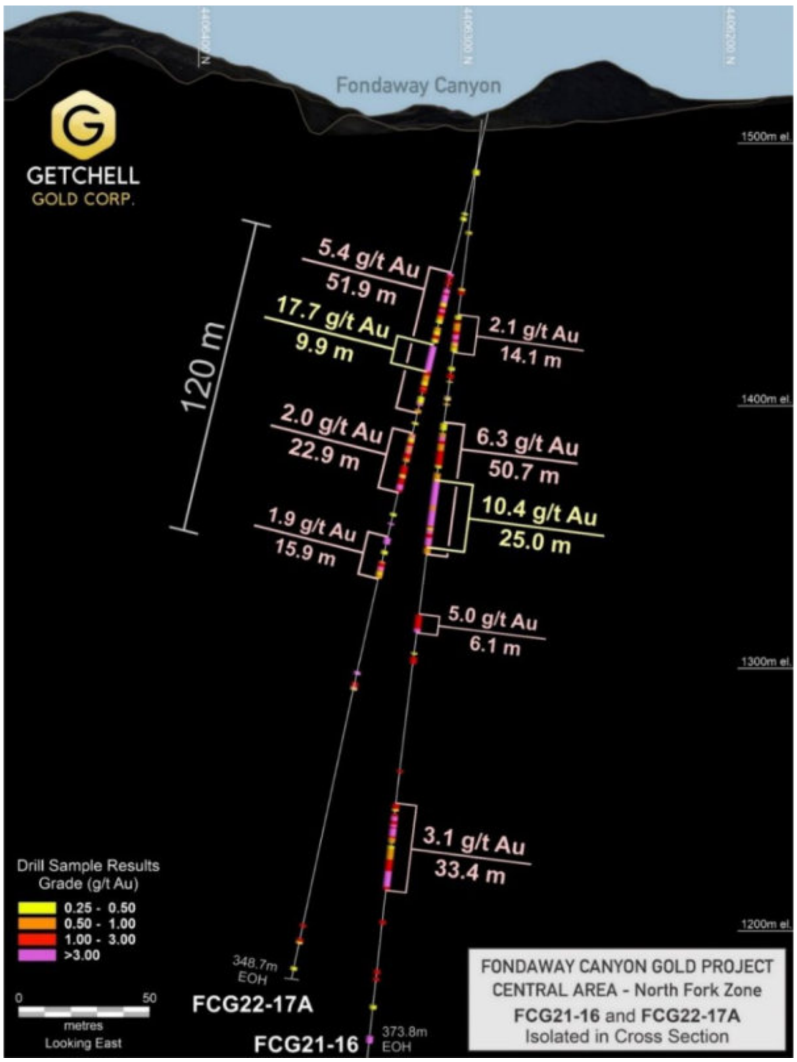


Figure 10-12: Cross-Section Highlighting Gold Intervals in Holes FGC21-16 and FGC22-17A

(Source: Getchell Gold, 2022)

Note: Intercepts represent core length. True width can vary from 50% up to 100% of core length depending upon drill hole intersection angles.

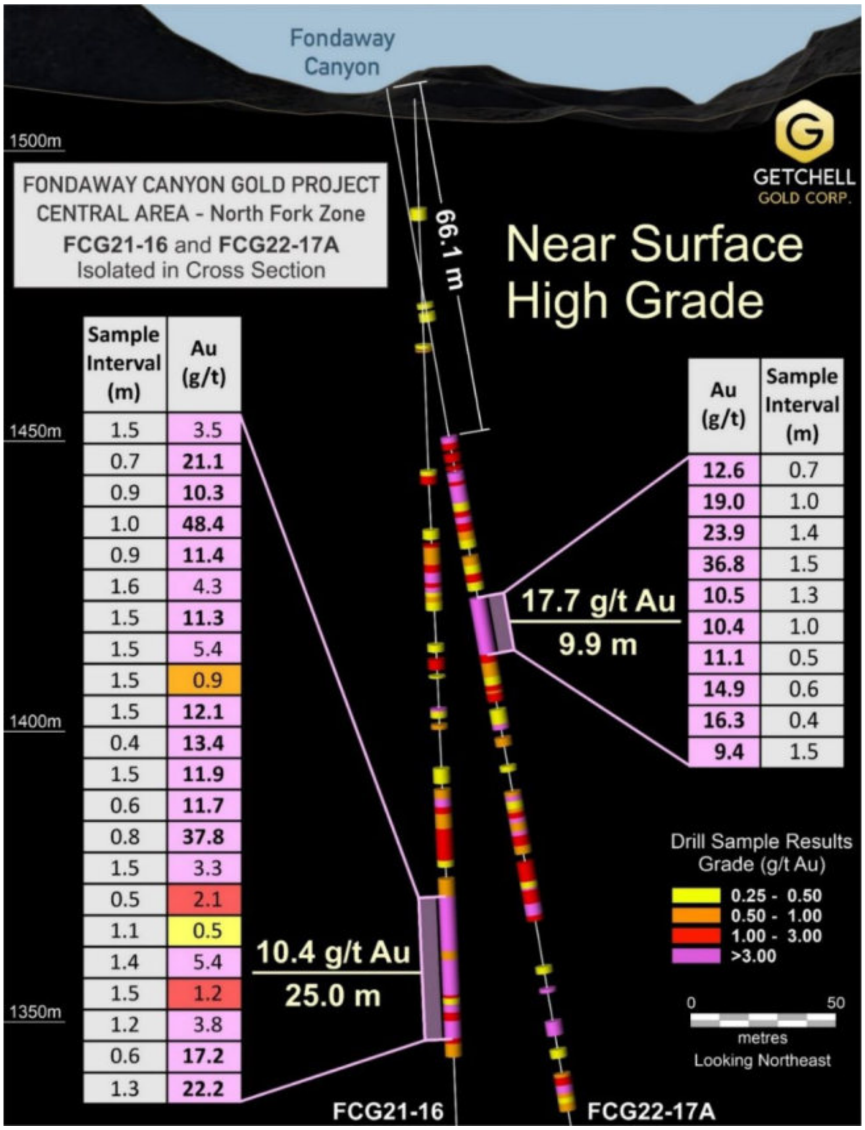


Figure 10-13: Cross-Section Highlighting Gold Assays in Holes FGC21-16 and FGC22-17A

(Source: Getchell Gold, 2022)

Note: Intercepts represent core length. True width can vary from 50% up to 100% of core length depending upon drill hole intersection angles.

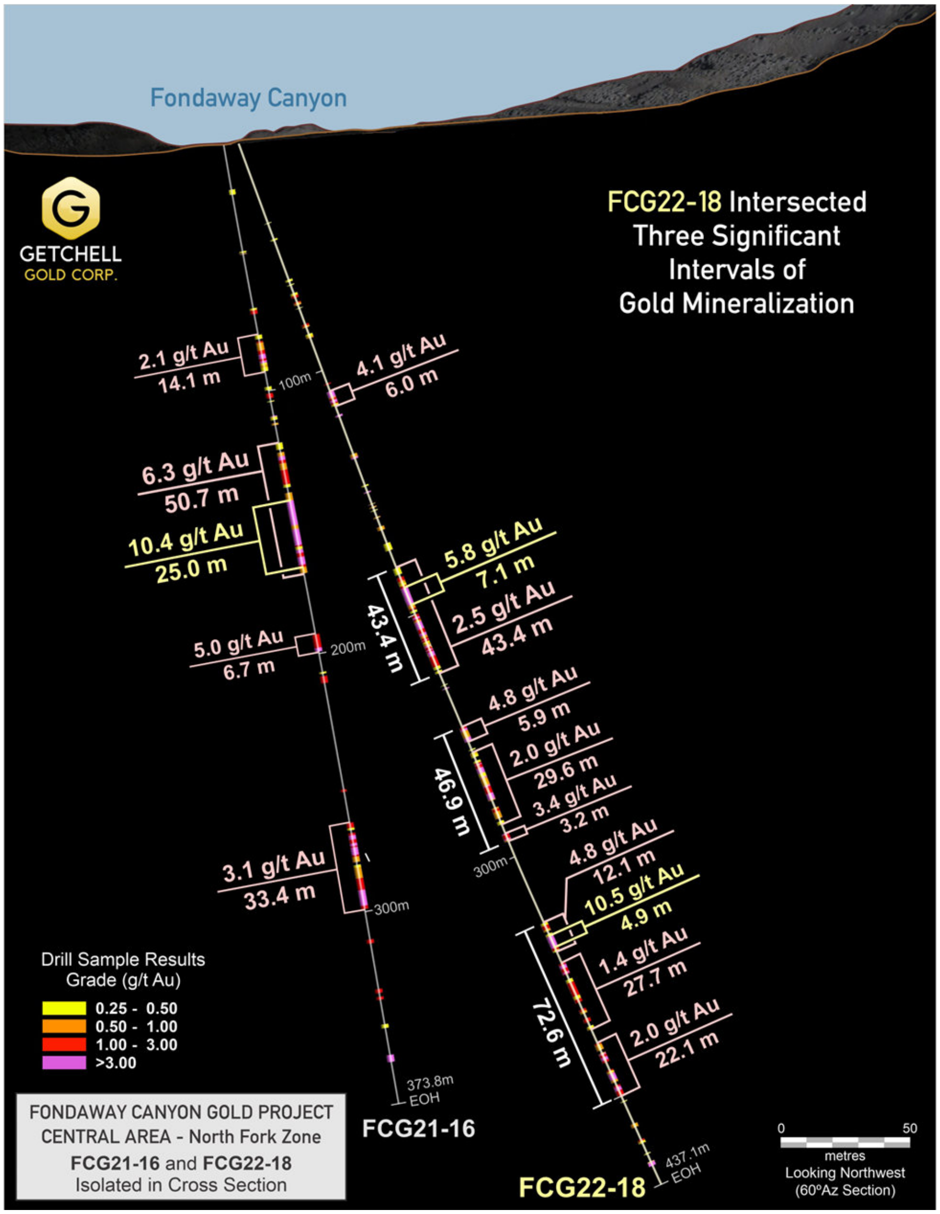


Figure 10-14: Cross-Section Highlighting Gold Assays in Holes FGC21-16 and FGC22-18

(Source: Getchell Gold, 2022)

Note: Intercepts represent core length. True width can vary from 50% up to 100% of core length depending upon drill hole intersection angles.

FCG22-19 intersected multiple significant intervals of gold mineralization along a 145.1 m drill length, from 105.8 to 250.9 m downhole with a core length mineralized zone grading 1.8 g/t Au over 107.5 m from 120.0 to 227.5 m downhole (Table 10-6; Figure 10-15).

Drill hole FCG22-17A intersected 3.8 g/t Au over 85.9 m core length and FCG22-18 intersected 2.5 g/t Au over 43.4 m and 2.1 g/t Au over 46.5 m core length, with respective step-outs of 15 m to the northwest and 50 m to the east, FCG22-19 (Figure 10-15).

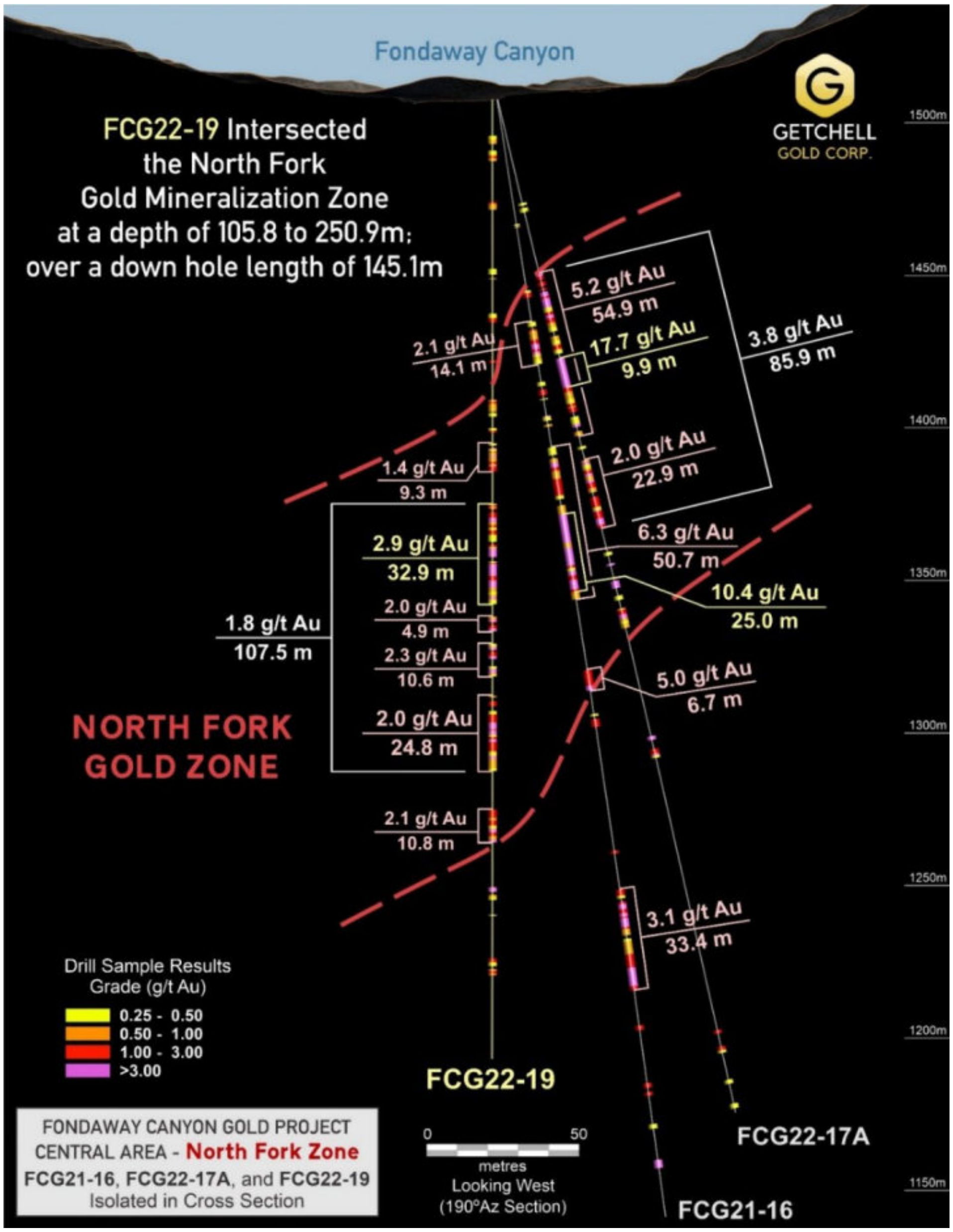


Figure 10-15: Cross-Section Highlighting Gold Assays in Holes FGC21-16, FGC22-17A and FGC22-19

(Source: Getchell Gold, 2022)

Note: Intercepts represent core length. True width can vary from 50% up to 100% of core length depending upon drill hole intersection angles.

Drill hole FCG22-20 was a vertical hole drilled from the Colorado Pit (Figure 10-11 and Figure 10-16), a site of small-scale mining during the 1980s, and was designed to test the up-dip extension of the Colorado SW mineralization. FCG22-20 intersected four significant gold mineralized intervals starting from surface including a major interval grading 1.7 g/t Au over 56.6 m from 160.4 to 217.0 m downhole (Figure 10-16; Table 10-7). This interval represents a 35-meter step out to the east with the Colorado SW zone remaining open and untested to the east and northeast.

Drill hole FCG22-21, stationed midway up the north slope of Fondaway Canyon (Figure 10-11), was designed to crosscut the Colorado SW zone of mineralization to assist with modeling and to test the extents of the mineralization to the northwest, as encountered by drill hole FCG21-08 and FCG20-02 (Figure 10-16). Drill hole FCG22-21 intersected an interval of gold mineralization grading 0.9 g/t Au over 74.3 m from 191.7 to 266.0 m downhole, representing a 50 meter step out to the north-northwest from previous drilling. The Colorado SW zone remains open and untested to the north and west from this drill hole.

Drill holes FCG22-22, 23, and 25 were designed as step outs to test the extent of the North Fork mineralization encountered in drill hole FCG21-16. Drill holes FCG22-22, 23, and 25 were additionally designed as step outs to test the extent of the lower North Fork gold zone discovered in FCG22-18 that graded 1.9 g/t Au over 72.6 m core length.

Drill hole FCG22-22 was drilled eastward from the southern margin of the Main Pit (Figure 10-11), a site of small-scale mining during the 1970’s and 1980s. The hole was designed to test for a continuation of the North Fork mineralization towards the southeast. Drill hole FCG22-22 intersected four significant gold mineralized core length intervals (Table 10-7) consisting of: 3.0 g/t Au over 59.3 m downhole including 8.8 g/t Au over 8.1 m (Upper North Fork); 2.4 g/t Au over 21.7 m including 7.1 g/t Au over 5.3 m; 0.8 g/t Au over 41.6 m; and 1.1 g/t Au over 25.8 m (Figure 10-17). The upper high-grade interval correlates well with the high-grade gold mineralization encountered in drill holes FCG21-16 and FCG22-17.

Drill hole FCG22-23 was drilled as a 45 m up-dip step out to the northeast of the high-grade gold mineralization encountered in drill hole FCG21-16. Drill hole FCG22-23 extended the North Fork gold mineralization intersecting 3.4 g/t Au over 44.6 m (Table 10-7; Figure 10-18). In addition, FCG22-23 encountered a shallow interval, 60 meters below surface, which graded 1.8 g/t Au over 12.8 m (Table 10-7; Figure 10-18).

Drill hole FCG22-24 was designed to test the Pediment target located 2 km to the west of the Central Area. Drill hole FCG22-24 encountered two mineralized core length intervals grading 0.72 g/t Au over 3.0 m and 0.64 g/t Au over 1.6 m at respective downhole depths of 140.0 and 239.3 m. Both mineralized intervals are hosted within a shear structure and exhibit characteristics indicative of mineralization peripheral to a main zone.

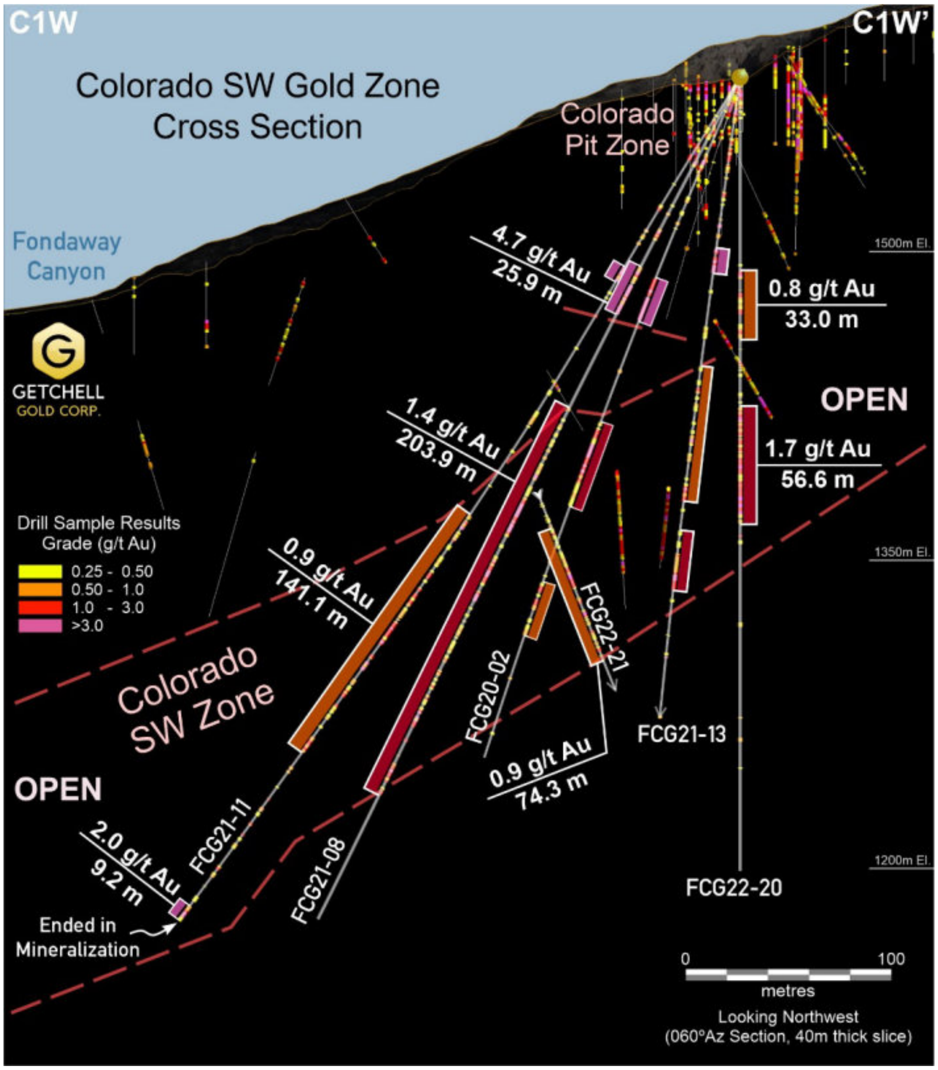


Figure 10-16: Cross-Section Highlighting Gold Intervals in Drill Holes FCG21-20 and FCG22-21

(Source: Getchell Gold, 2022)

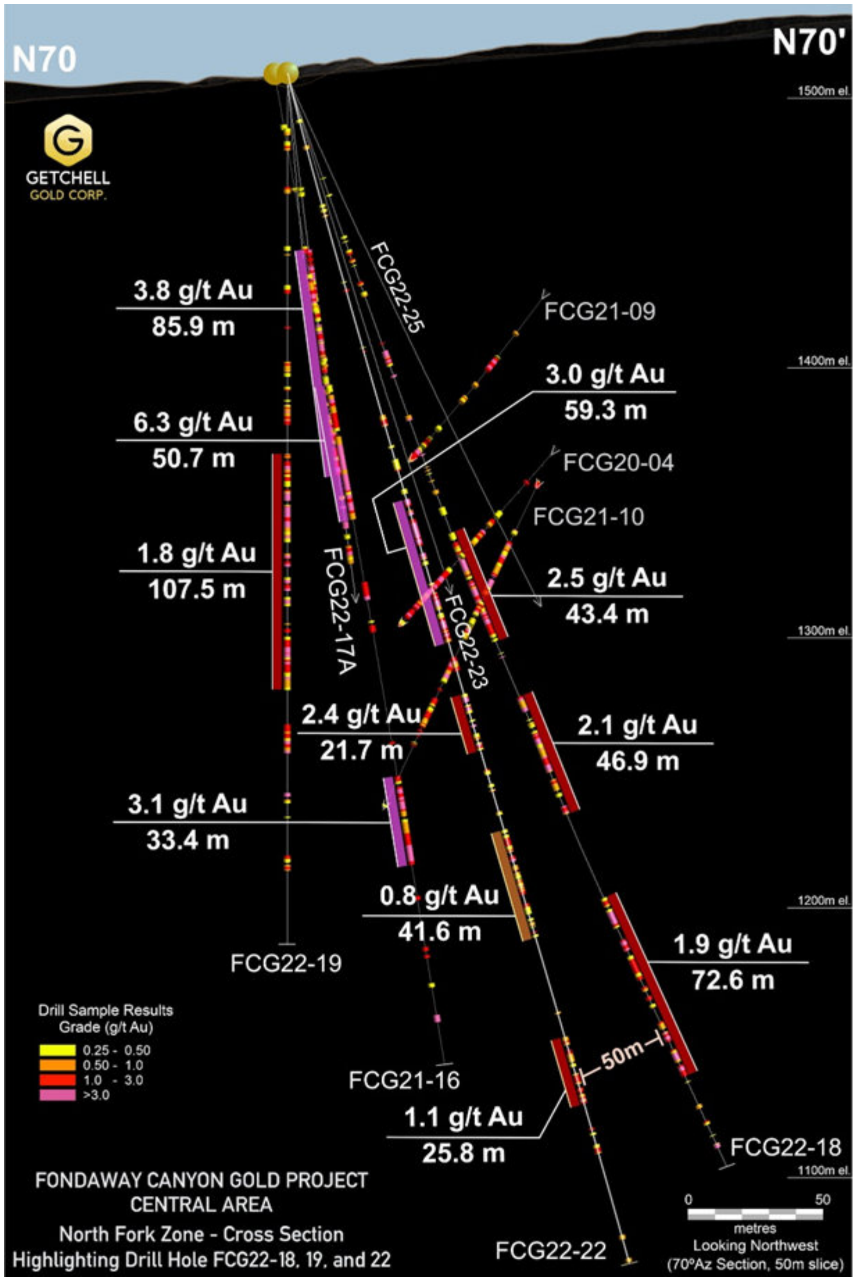


Figure 10-17: North Fork Cross-Section Highlighting Gold Intervals in Drill Holes FCG22-18, 19 and 22

(Source: Getchell Gold, 2022)

Note: Intercepts represent core length. True width can vary from 50% up to 100% of core length depending upon drill hole intersection angles.

Drill hole FCG22-25 was primarily designed to test the up-dip continuation of the lower series of gold intervals encountered at North Fork by FCG22-18 that includes 2.1 g/t Au over 46.9 m. FCG22-25 encountered a gold interval grading 3.4 g/t Au over 31.5 m core length that correlates well and represents a 30 m extension to the lower zone of mineralization at North Fork zone (Table 10-7; Figure 10-18). In addition, FCG22-25 encountered multiple gold intervals above and below that interval (Table 10-7; Figure 10-18).

Drill holes FCG22-26, 27 and 28, stationed near the canyon floor on the Colorado SW section, were designed to respectively target the eastern strike, test the up-dip continuation, and target the down dip extent of the Colorado SW zone. Both holes revealed good gold grade consistency and mineralized thickness in respect to the adjoining drill section, intersecting the Colorado SW zone over a 200m downhole length (Figure 10-19).

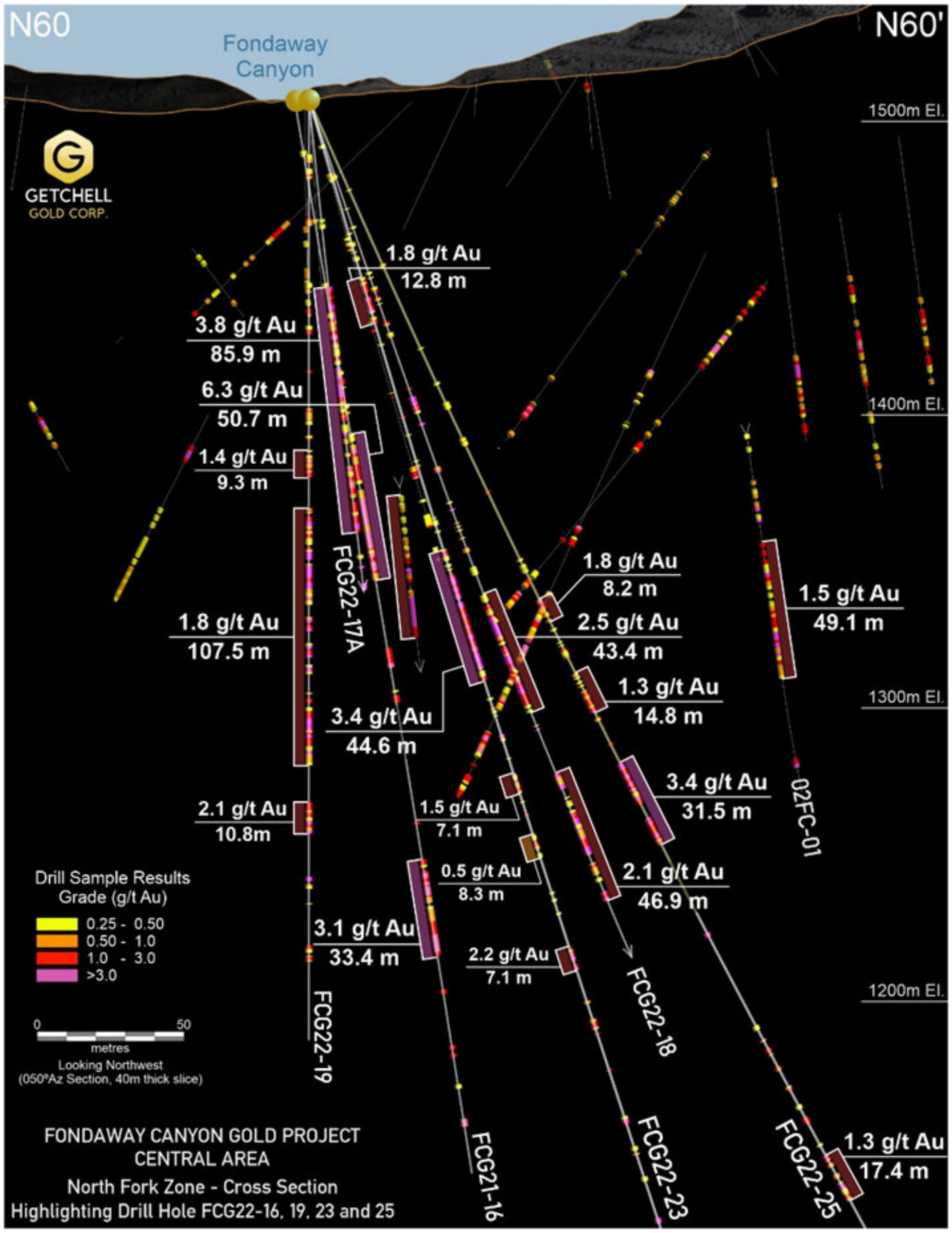


Figure 10-18: North Fork Cross-Section Highlighting Gold Intervals in Drill Holes FCG22-16, 19, 23 and 25

(Source: Getchell Gold, 2022)

Note: Intercepts represent core length. True width can vary from 50% up to 100% of core length depending upon drill hole intersection angles.

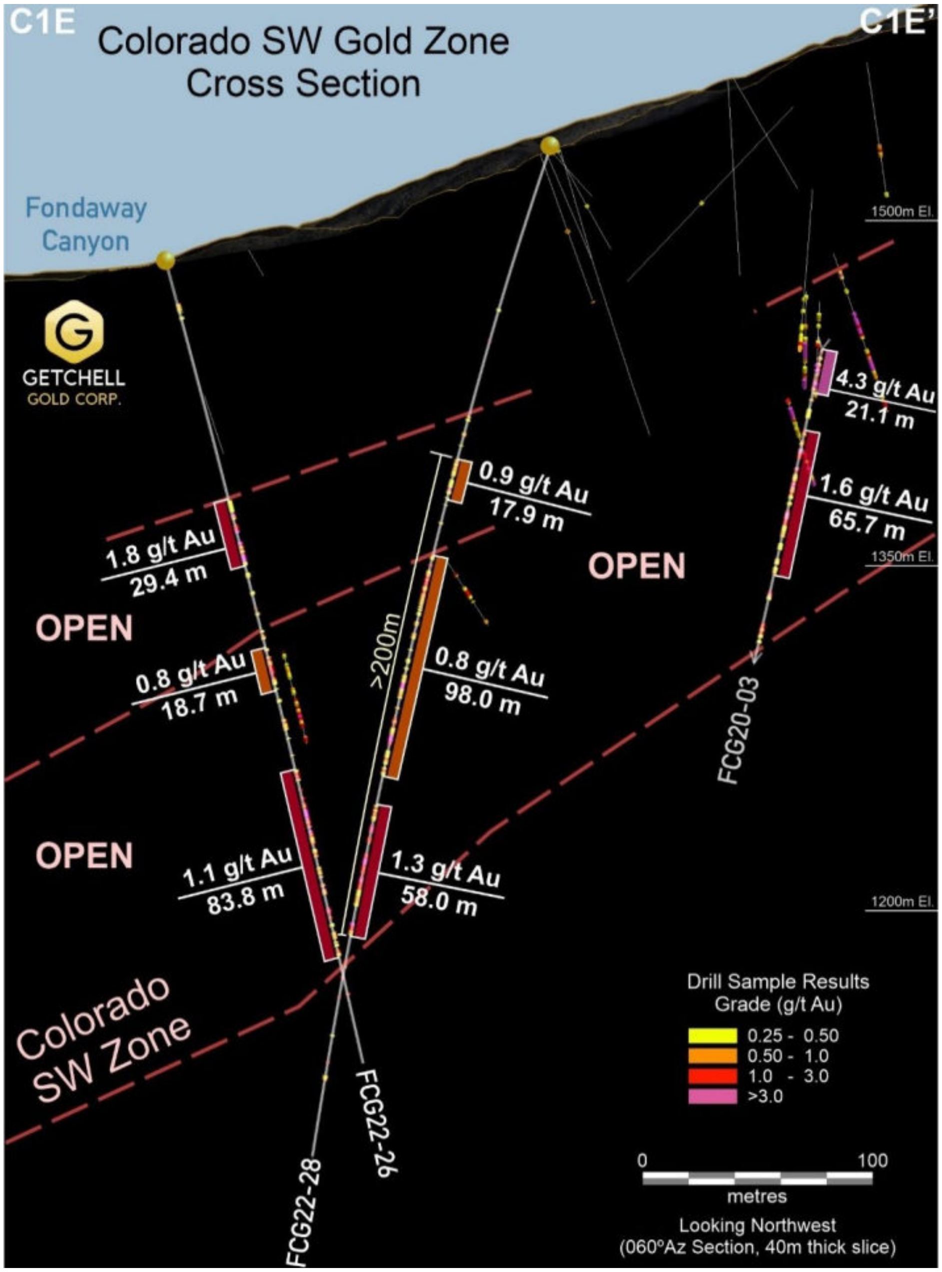


Figure 10-19: Cross-Section Highlighting Gold Intervals in Drill Holes FCG22-26 and FCG22-28

(Source: Getchell Gold, 2022)

Note: Intercepts represent core length. True width can vary from 50% up to 100% of core length depending upon drill hole intersection angles.

Drill hole FCG22-27 was designed to test the up-dip continuation of the Colorado SW zone on section with drill hole FCG20-06. Drill hole FCG22-27 intersected an upper interval returning 1.2 g/t Au over 29.9 m, but the drill rods became stuck and the hole was lost partway through the targeted zone at a depth of 240m (Figure 10-20).

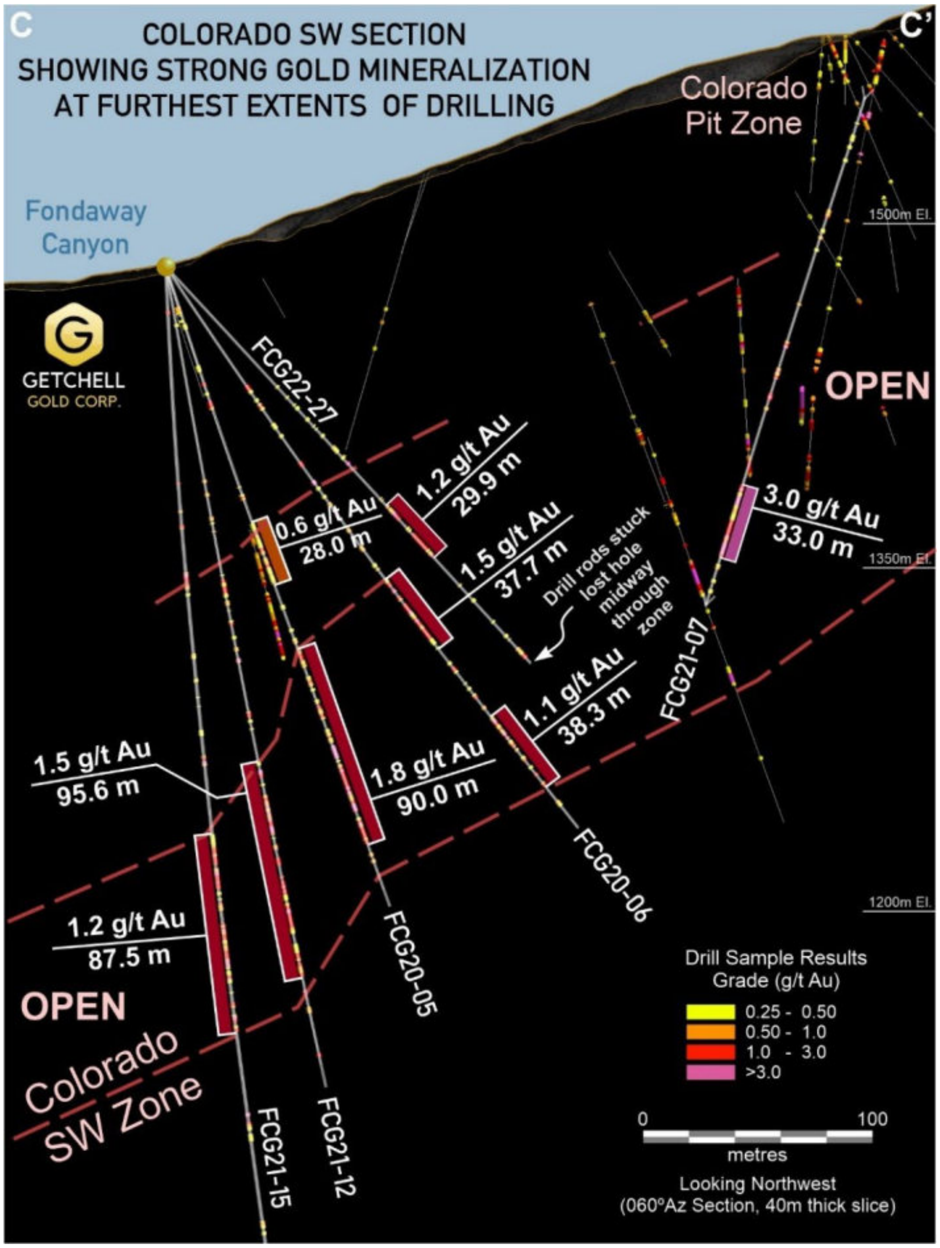


Figure 10-20: Cross-Section Highlighting Gold Intervals in Drill Hole FCG22-27

(Source: Getchell Gold, 2022)

Note: Intercepts represent core length. True width can vary from 50% up to 100% of core length depending upon drill hole intersection angles.

Drill hole FCG22-28 marked the last hole drilled during the 2022 drill program with all gold zones drill-tested remaining open along strike and up and down dip. The gold mineralized intervals encountered in the 2022 drill holes FCG2217A through FCG22-28 have been incorporated into the database and represents the cut off point for data inclusion into the Mineral Resource Estimate provided in Section 14.

Drill results on the overall Colorado SW Zone (Figure 10-19) show good gold grade consistency and mineralized thickness. This section is highlighted by drill holes FCG21-08 and FCG21-11 that intersected the Colorado SW zone grading 1.4 g/t Au over 203.9 m core length and 0.9 g/t Au over 141.1 m core length, respectively. This section is the most northwestern drilled section on the Property and is bounded to the northwest by a region absent of any drilling.

10.2.4 Drilling Procedures

A brief overview of drilling procedures used by Getchell Gold during their drill programs is included below.

10.2.4.1 Collar Surveys

On the drill site, each drill set-up is surveyed for azimuth, inclination or dip, and collar coordinates. Drill holes are surveyed using a Reflex Gyrocompass. The survey data obtained from the drill holes are then transferred to Getchell Gold's databases.

10.2.4.2 Downhole Surveys

Downhole procedures for the 2020 to 2022 Getchell Gold drilling included hole deviation readings and oriented core readings. Downhole orientation readings were taken every 30 m with a Reflex EZ shot survey tool. Oriented drill core markings were made on the drill core for each drill run using a Reflex ACT III Core orientation tool.

A Reflex ACT III Tool was used by the drillers to mark the core orientation reference point, the lowermost point on the top face of a run of core. The geologists then pieced the run of core back together (if possible) and extended a line along the run of core from the reference point. An Ezy-Logger™ Goniometer was then used to measure the alpha and beta angles of bedding, foliations, fractures, veins, lithologic contacts and gouges.

Downhole deviations, as measured by the drillers using a Reflex EZ-Gyro, were entered into the GeoCalculator software by R. Holcombe along with the goniometer alpha and beta measurements to determine true dips and strikes of planar structures. The measurements were then entered into the Stereonet 10.0 software by Richard W. Allmendinger to create Schmidt Stereonet Plots and Rose Diagrams of foliations and Kamb Contour Diagrams of foliation poles. The mean azimuth and dip of the foliation was also calculated for each exploration area using the results from the oriented core.

10.2.4.3 Logging and Sampling

Data collected from the drill core included geological descriptions, core recovery, rock quality determination (RQD), and fracture count. Oriented drill core measurements, recorded using a goniometer, included shears, foliation, slip surfaces, fault gouge, fractures and veins.

The 2020, 2021, and 2022 drill core was cut at Bureau Veritas Laboratories' ("BVL") facilities in Sparks, Nevada, with the samples analyzed for gold and multi-element analysis in BVL's Sparks, Nevada and Vancouver, BC laboratories respectively. BVL is accredited to ISO/IEC 17025 and ISO 9001 and is independent of the issuer and the authors of this report.

Gold analysis was completed by fire assay with an Atomic Absorption finish on a 30-gram sample (BV code FA430) with over limits re-analyzed using method FA530 (30g Fire Assay with gravimetric finish). The multi-element analysis was performed by ICP-MS following aqua regia digestion on a 30 g sample (BV code AQ250). Quality control measures in the field included the systematic insertion of standards and blanks.

10.2.4.4 Sample Length and True Thickness

Unless otherwise stated, intercepts presented throughout Section 10 represent core length. The true thickness is currently not well constrained but can vary from 50% up to 100% of core length depending upon drill hole intersection angles.

10.3 Specific Gravity Collection Program

In the Spring of 2024, Getchell Gold carried out a specific gravity (SG) collection program on core from select drill holes from the 2020-2022 drilling programs. The SG measurements were collected in order to be used in the revised Mineral Resource Estimate presented in Section 14.

The SG measurements were first taken in the field by Getchell Gold's personnel. A portion of those core samples were sent to Bureau Veritas of Sparks, Nevada for SG analysis to be compared to the measurements taken in the field.

A total of 1,382 core samples from 9 drill holes (FCG20-06, FCG21-09, FCG21-11, FCG21-15, FCG22-17A, FCG22-19, FCG22-20, FCG22-21 and FCG22-23) were subject to SG measurements in the field, sampled approximately every 2 meters downhole. Core samples were selected from both mineralized and non-mineralized intervals, and well representative of the lithologies and alteration contained within the Mineral Resource.

The methodology for SG measurement was as follows: a piece of core is selected (ideally 10 cm-long minimum), weighed dry in air, then weighed fully immersed in water. The SG value is then obtained by dividing the weight of the sample in air by its weight in air minus its weight in water.

Getchell Gold subsequently submitted 121 of the 1,382 core pieces to Bureau Veritas to complete both paraffin wax-coated (lab code SPG03) and regular non-wax-coated (lab code SPG02) SG analysis, in order to constrain if there was a porosity/void filling effect on the field SG measurements. Samples were selected every 4 to 6 meters downhole, down to about 200-330 meters, from 4 holes (FCG22-19 and FCG22-17A from the North Fork Zone; FCG22-20 and FCG21-11 from the Colorado Zone).

Results show that the field, lab non-wax-coated and lab wax-coated SG measurements have an average value of 2.75, 2.75, and 2.73 respectively. The lab non-wax-coated and lab wax-coated results are both within the margin of error of the field measurements results; however, lab wax-coated results display a slightly lower SG average (about 0.02 SG or 0.7% difference). Out of the 121 samples submitted for analysis, 8 samples (6.6%) were or exceeded a -0.05 SG differential from the field measurements. A density of 2.74 g/cm3, representing a 7% increase, was derived from the specific gravity study and assigned to the rock hosting the mineralized zones and waste.

11. SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Historical Drilling

11.1.1 Sample Collection, Preparation and Security

For each of the historical RC drilling programs, the RC samples were collected at the drill rigs, using industry-standard practices, under the supervision of the company geologists. RC samples were split with a Jones splitter when dry and with a rotary splitter when wet. Duplicate RC samples were taken from the rotary splitter at the drill rig.

For the historical core drilling programs, the core was logged and sampled under the supervision of the company geologists. The core was split at important geological contacts, and into equal, typically five-foot lengths within geological units. Competent core was sawed in half for analysis, and the core that was broken into rubble had approximately half selected by the geologist. In either case, the remainder of the core was stored in labeled core boxes.

The samples were prepared and assayed by reputable, certified laboratories. The labs included Cone Geochemical (Denver, CO), Geochemical Services (Reno, NV), Shasta Analytical (Redding, CA), G.D. Resources (Sparks, NV), and American Assay Labs (Reno, NV). All of these labs are independent of Canarc. Although some of the labs are no longer in business, all of the labs were certified and known in the industry for professional procedures and quality results.

The samples were dried, then crushed (typically >85% 6-mesh), then Jones riffle-split to obtain ½ to 1 pound splits, with the remainder of the crushed reject. The splits were then ring and puck pulverized to 120 to 150 mesh and stored in a labeled packet.

11.1.2 Analytical Procedures

Samples from historical drilling were analyzed at various laboratories that include Cone Geochemical, (Denver CO), Geochemical Services Inc., (Reno, NZ) and Shasta Analytical Geochemistry Laboratory, (Redding CA) and G.D. Resources Inc., (Sparks, NV). All of these laboratories are independent of the QP and the issuer. Although some of the labs are no longer in business, all of the labs were certified and known in the industry for professional procedures and quality results.

Gold was measured by fire assay with an Atomic Absorption finish and copies of the original assay sheets were made available to the QP.

11.1.3 Quality Assurance – Quality Control

The laboratories employed a QA/QC protocol that included periodic duplicate analyses of core pulps at least for Tenneco Minerals (1988-1990), Nevada Contact (2002), and Canarc Resources (2017) drilling programs. Additional QA/QC data available to the authors include certified reference materials (standards) and blanks inserted by the laboratory for Canarc Resources' 2017 drilling program. No other QA/QC data is available from the historical drilling campaigns from either the operator with inserted QA/QC samples or from the laboratories.

11.2 Getchell Gold Drilling

11.2.1 Sample Collection, Preparation and Security

The same procedure was used for the 2020, 2021 and 2022 drill programs.

Diamond drill core is placed in core boxes by the drill company and transported to the Getchell Gold core logging building in Fallon, NV by the drilling company. The Project geologists log the core for lithologic characteristics and the geological technicians log the core for core recovery, rock quality determination (RQD), fracture count, magnetic susceptibility and conductivity.

Samples of drill core were chosen for analysis by a qualified geologist based on the lithology, structure, percentage of quartz veining and alteration. Core to be sampled by splitting was marked, sample intervals were recorded in a sample ticket book, then sample number tags from the sample ticket book were stapled to the core box at the beginning of each sample interval. After the core was marked for sampling, it was photographed both wet and dry in good lighting conditions.

The 2020, 2021, and 2022 drill core was cut at Bureau Veritas Laboratories' ("BVL") facilities in Sparks, NV. Core designated for cutting was stacked on pallets, wrapped in stretch wrap and loaded onto a BVL flatdeck truck for transport to the Sparks laboratory. Sample submittal sheets were sent to the lab electronically for an official record of samples submitted along with instructions for prep and analysis.

All pulps, coarse rejects and split core are then transported and stored in a central warehouse in Fallon, NV for the Getchell Gold drilling, under their management and control.

11.2.2 Analytical Procedures

The BVL facilities in Sparks, NV, analyzed the samples for gold while the multielement analysis was conducted at their Vancouver, BC laboratory. Gold values were produced by fire assay with an Atomic Adsorption finish on a 30-gram sample (BV code FA430) with over limits re-analyzed using method FA530 (30 g Fire Assay with gravimetric finish). The multi-element analysis was performed by ICP-MS following aqua regia digestion on a 30 g sample (BV code AQ250). Results from the analyses are transmitted by email directly to Getchell Gold's senior management and the signed paper assay certificates are mailed. BVL is accredited to ISO/IEC 17025 and ISO 9001 and is independent of the issuer and the authors of this report.

11.2.3 Quality Assurance – Quality Control

Getchell Gold inserts control samples at a frequency of one standard, a Certified Reference Material (CRM), every 20 samples and one blank every 30 samples, with a goal of achieving approximately a 10% QAQC sample insertion rate. Duplicate samples were inserted into the batch sample stream and analyzed by BVL.

During 2020-2022 drill programs, Getchell Gold used thirteen different CRMs with gold values ranging from 0.039 g/t to 11.229 g/t. For blanks Getchell Gold used commercially acquired silica blanks. A total of 549 CRMs, 206 blanks and 771 lab duplicates were analyzed during the 2020-2022 drill program (Table 11-1).

Table 11-1: QA/QC Samples Used in 2020-2022 Drill Programs

Drill Program	CRMs	Blanks	Duplicates
2020	83	32	142
2021	204	75	293
2022	262	99	336
Total	**549**	**206**	**771**

The BV laboratory QA/QC protocol incorporates a granite or quartz sample-prep blank(s) carried through all stages of preparation and analysis as the first sample(s) in the job. Typically, an analytical batch will be comprised of 34-36 samples, a pulp duplicate to monitor analytical precision, a -10 mesh reject duplicate to monitor subsampling variation, a reagent blank to measure background and an aliquot of Certified Reference Material. Using these inserted control samples each analytical batch and complete job is reviewed and validated prior to release. No issues were reported by the lab with respect to their internal QA/QC sample results. Results of Laboratory duplicates are shown in section 11.2.3.3.

Getchell Gold's QA/QC protocols follow industry best practices.

11.2.3.1 Certified Reference Material (CRM)

Getchell Gold purchased Certified Reference Material (CRM or standard) for insertion into the sample stream. The gold standard reference material was purchased from Moment Exploration Geochemistry LLC (MEG), Lamoille NV. A total of 13 certified gold CRMs were used over the three years of core drilling: STD906, STD1113, STD1115, STD1134, STD1213, STD1227, STD1303, STD1708, STD1706, STD1723, STD1903, STD1907, STD1910.

Results are presented using statistical process control charts (control charts, for short). In the chart the "accepted" or average value appears as a black horizontal line. Control limits at 2 Standard Deviation (2SD) of the accepted value appear as dashed red lines above and below the line showing the accepted value and for 3SD as solid red lines. The assay result values for the standard appear on the chart as green circles. Assays results falling outside of the 3SD limits are considered failures. Certified assay values and 90% confidence intervals for each of the CRMs are presented in Table 11-2.

Table 11-2: Certified Au Values and Statistics for the CRMs

Standard ID	Expected Value	STDEV	%RSD	3SD (90% confidence interval)	
	Au (ppm)			Min Au (ppm)	Max Au (ppm)
STD 906	11.229	0.459	4.1	9.852	12.606
STD 1113	1.806	0.081	4.5	1.563	2.049
STD 1115	3.457	0.2323	6.7	2.7599	4.1537
STD 1134	2.113	0.172	8.2	1.597	2.629
STD 1213	0.879	0.059	6.7	0.702	1.056
STD 1227	2.931	0.258	8.8	2.157	3.705
STD 1303	1.823	0.107	5.9	1.502	2.144
STD 1706	0.099	0.004	4.0	0.087	0.111
STD 1708	0.410	0.014	3.5	0.368	0.452
STD 1723	0.126	0.006	4.7	0.108	0.144
STD 1903	0.039	0.003	7.7	0.03	0.048
STD 1907	0.331	0.016	4.8	0.283	0.379
STD 1910	0.811	0.036	4.4	0.703	0.919

For the 2020-2022 drilling, CRM results for all standards are shown in Figure 11-1 through Figure 11-13. The overall failure rate is 18.2%, which is considered somewhat high by the QP, but many of the failures are considered marginal failures i.e. close to the 3SD limits. The analytical results for standard STD1903 had the greatest number of analytical failures for gold and should be investigated further; however, in general, the results of the standard analyses completed by Getchell Gold from 2020 to 2022 show no significant issues. In the opinion of the QP, the data is considered acceptable for use in this Technical Report, with recommendations for future protocols provided below in Section 11.2.4.

No failures were recorded.

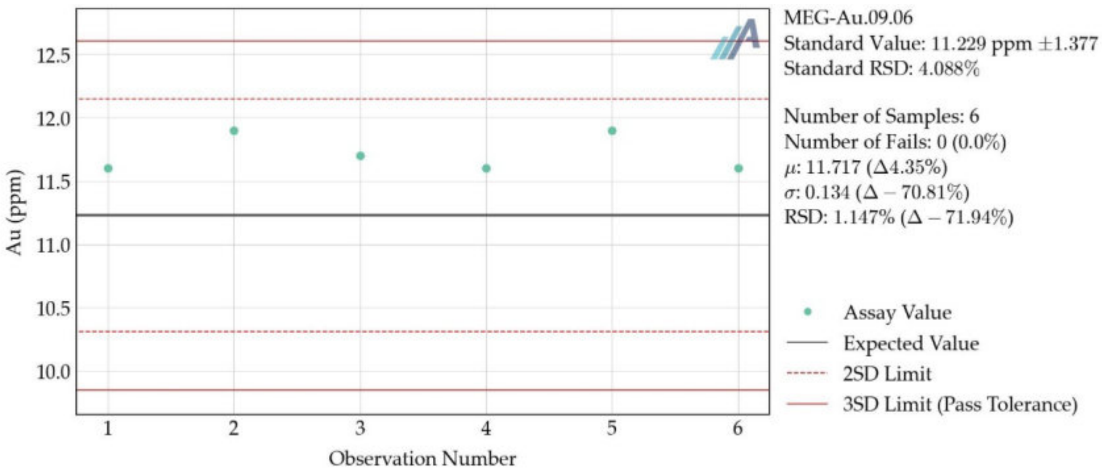


Figure 11-1: Standard STD906 – Gold Results

(Source: APEX Geoscience Ltd.)

STD1113 reported 2 failures out of 34 standard samples outside of 3SD.

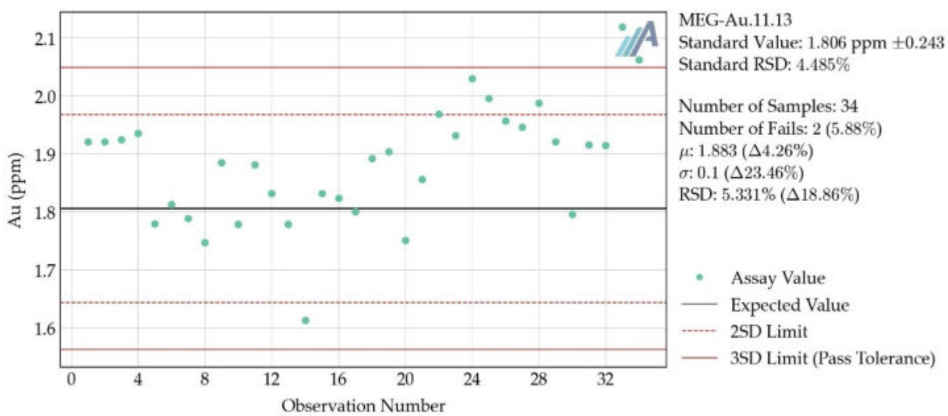


Figure 11-2: Standard STD 1113 – Gold Results

(Source: APEX Geoscience Ltd.)

STD1115 reported 6 failures out of 60 standard samples outside of 3SD.

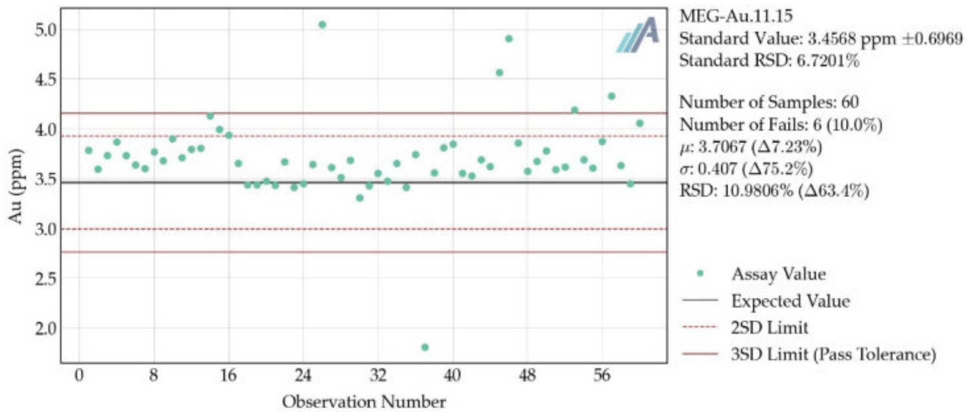


Figure 11-3: Standard STD 1115 – Gold Results

(Source: APEX Geoscience Ltd.)

STD1134 reported 5 failures out of 21 standard samples outside of 3SD.

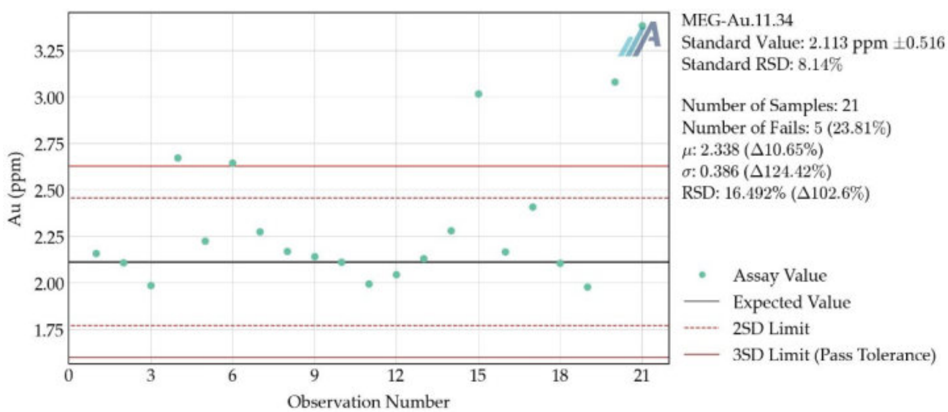


Figure 11-4: Standard STD 1134 – Gold Results

(Source: APEX Geoscience Ltd.)

No failures were recorded for STD1213.

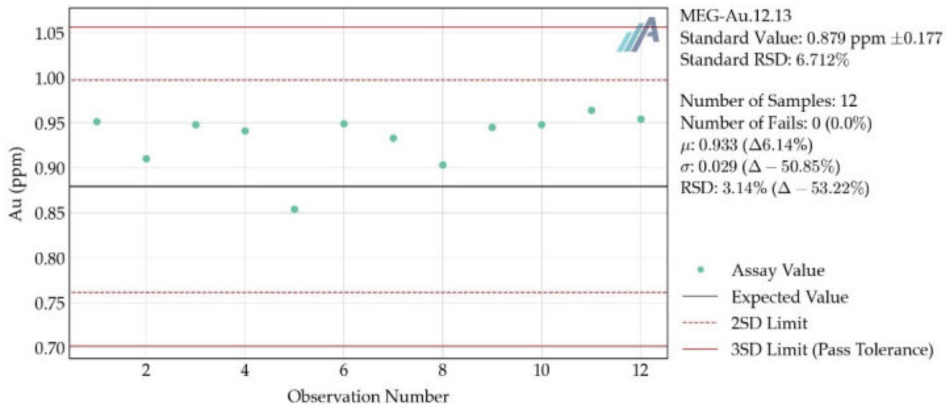


Figure 11-5: Standard STD 1213 – Gold Results

(Source: APEX Geoscience Ltd.)

STD1227 reported 9 failures out of 68 standard samples outside of 3SD.

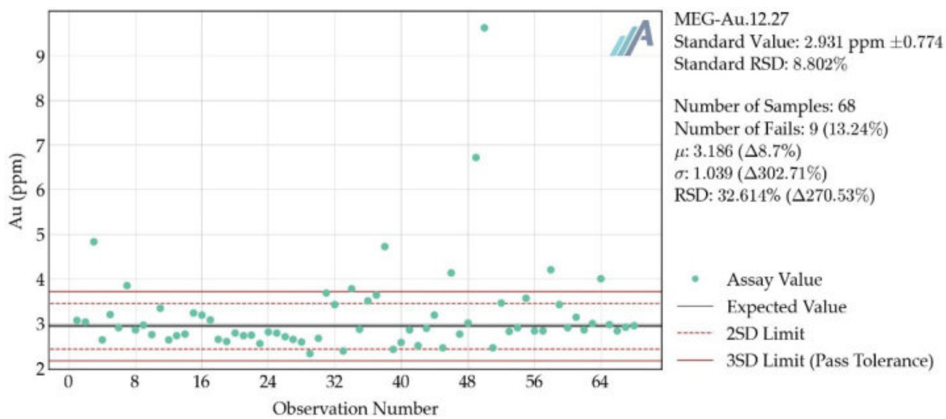


Figure 11-6: Standard STD 1227 – Gold Results

(Source: APEX Geoscience Ltd.)

STD1303 reported 1 failure out of 13 standard samples outside of 3SD.

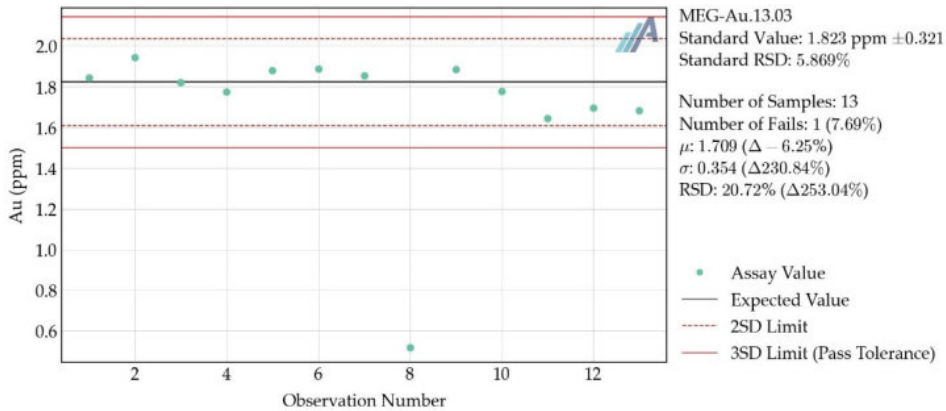


Figure 11-7: Standard STD 1303 – Gold Results

(Source: APEX Geoscience Ltd.)

STD11706 reported 9 failures out of 58 standard samples outside of 3SD.

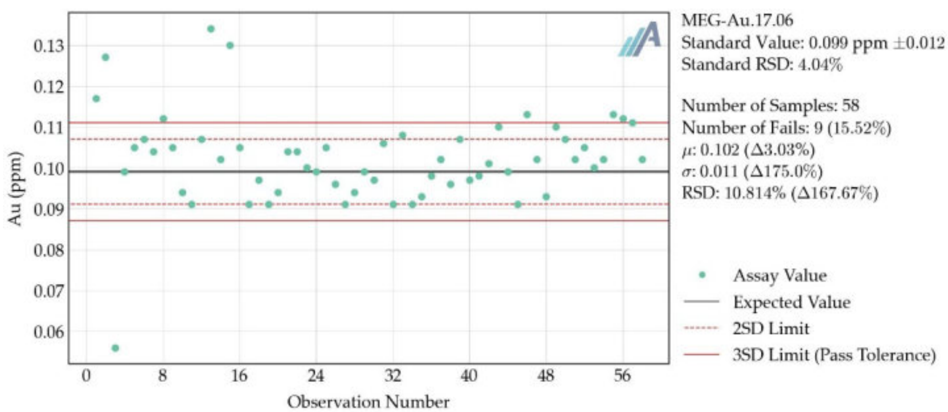


Figure 11-8: Standard STD 1706 – Gold Results

(Source: APEX Geoscience Ltd.)

No failures were recorded for STD1708.

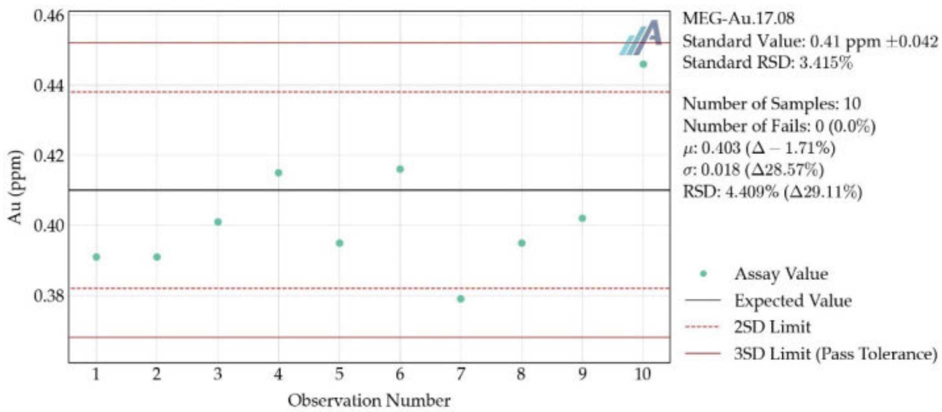


Figure 11-9: Standard STD 1708 – Gold Results

(Source: APEX Geoscience Ltd.)

STD1723 reported 10 failures out of 69 standard samples outside of 3SD.

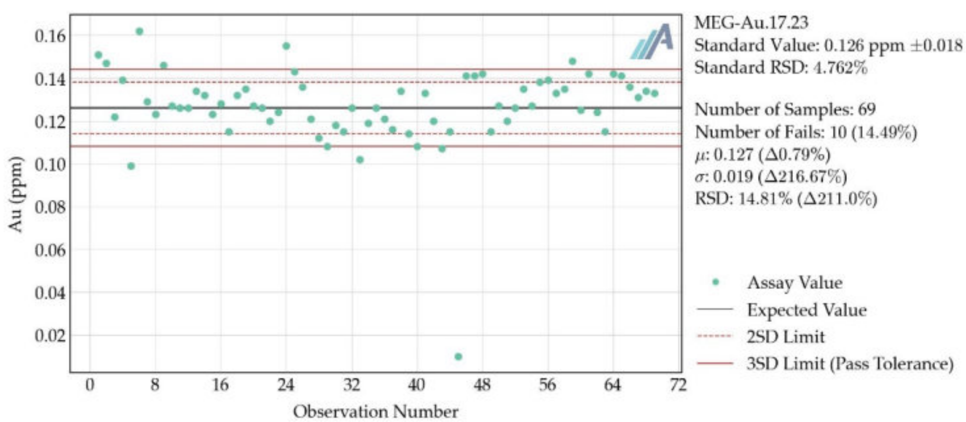


Figure 11-10: Standard STD 1723 – Gold Results

(Source: APEX Geoscience Ltd.)

STD1903 reported 16 failures out of 69 standard samples outside of 3SD.

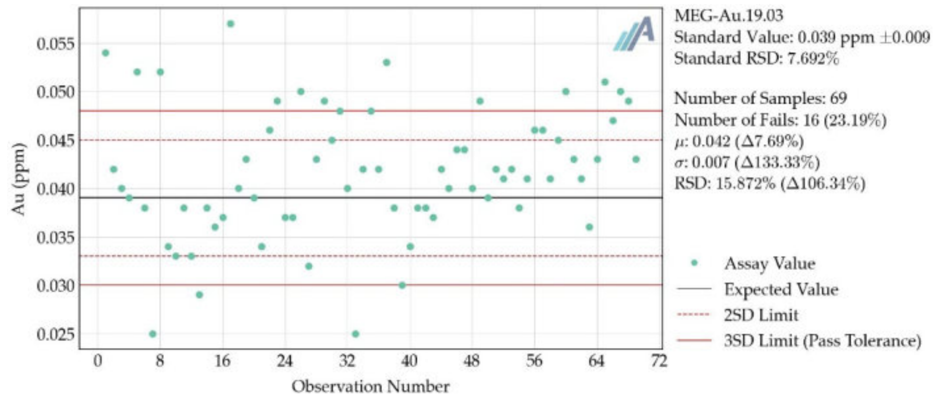


Figure 11-11: Standard STD 1903 – Gold Results

(Source: APEX Geoscience Ltd.)

STD1907 reported 7 failures out of 71 standard samples outside of 3SD.

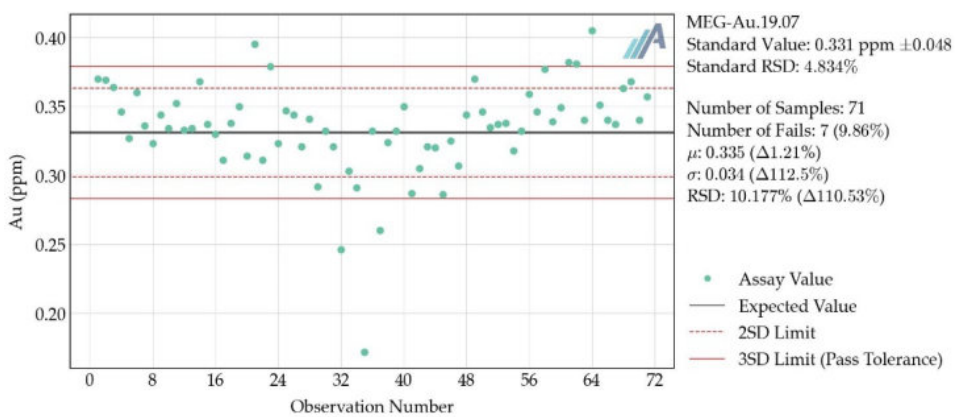


Figure 11-12: Standard STD 1907 – Gold Results

(Source: APEX Geoscience Ltd.)

STD1910 reported 2 failures out of 58 standard samples outside of 3SD.

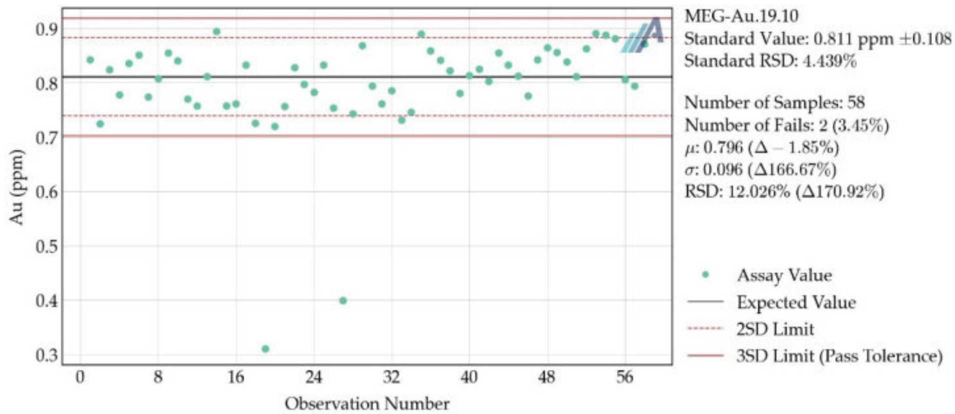


Figure 11-13: Standard STD 1910 – Gold Results

(Source: APEX Geoscience Ltd.)

11.2.3.2 Blank Samples

For the 2020-2022 drilling programs a commercially acquired silica blank was utilized for insertion into the sample stream. Analyses of the material by BVL returned no significant Au results. The majority of blanks returned assays below detection, with only 3 samples (1.46%) returning assays above the maximum allowable value which is equal to 3 times the detection limit (Figure 11-14). In the opinion of the QP, the results from the blank sample analyses for the Getchell Gold sampling completed during 2020 and 2022 display no significant issues and are considered acceptable for use in this Technical Report.

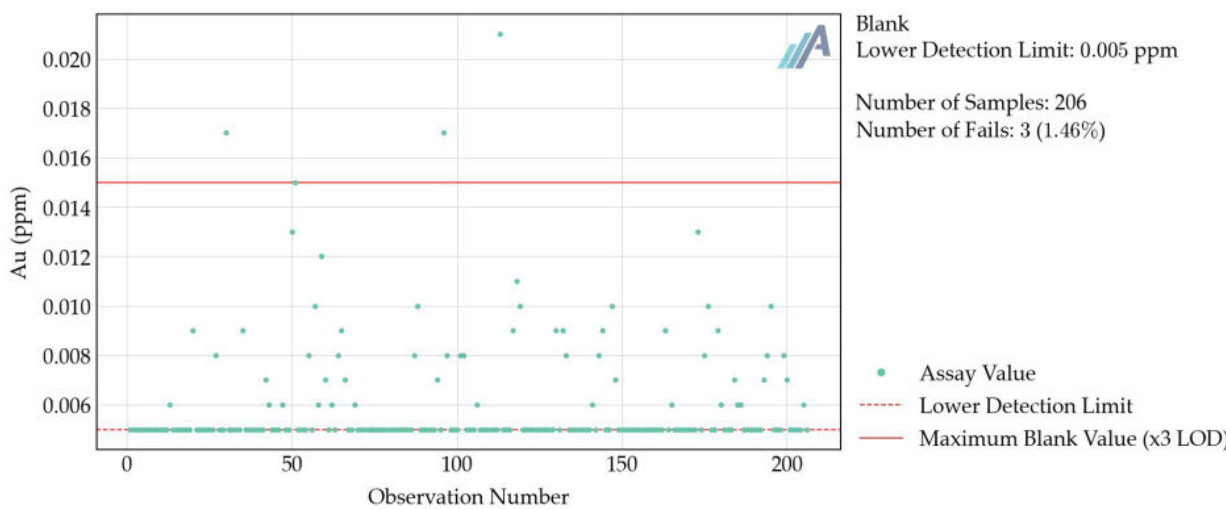


Figure 11-14: Au Assays for Blank Samples

(Source: APEX Geoscience Ltd.)

11.2.3.3 Laboratory Duplicate Samples

BVL analyzed 332 pulp duplicates to monitor analytical precision and 286 -10 mesh reject duplicates to monitor subsampling variation. Results of the comparison assays are presented below in Figure 11-15 and Figure 11-16. Failures rates of 1.8% and 2.5% respectively were reported, which are considered acceptable. In the opinion of the QP, the results of the laboratory duplicate sample analyses for the Getchell Gold sampling completed during 2020 to 2022 display no significant issues and are acceptable for use in this Technical Report.

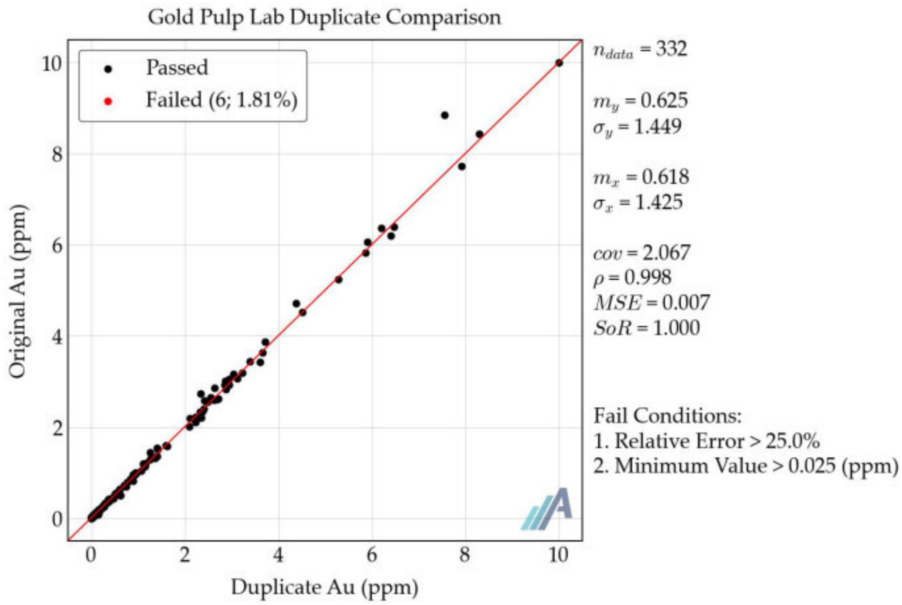


Figure 11-15: Pulp Duplicate Au Assay Comparison

(Source: APEX Geoscience Ltd.)

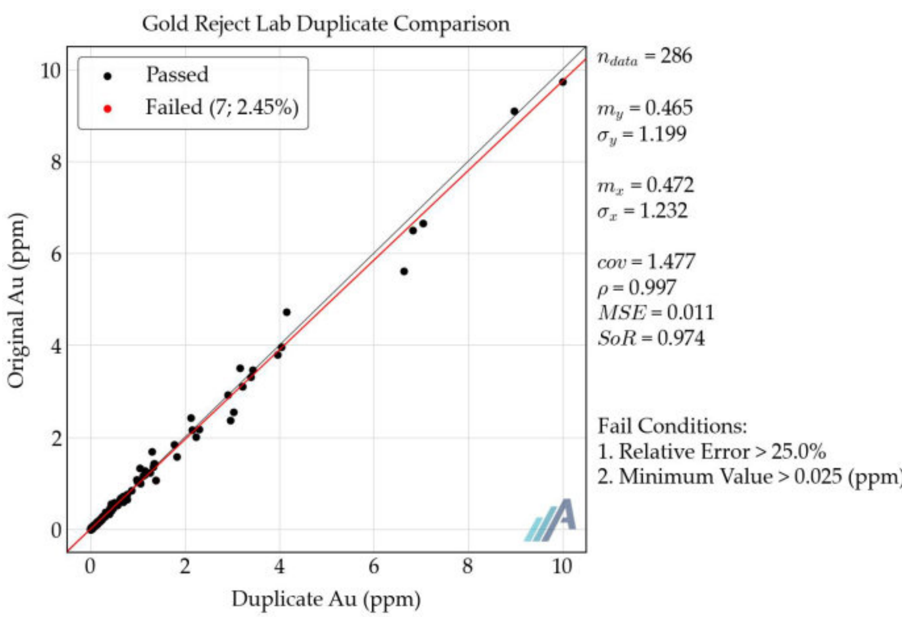


Figure 11-16: Reject Duplicate Au Assay Comparison

(Source: APEX Geoscience Ltd.)

11.2.4 QAQC Recommendations

For future exploration programs, it is recommended that the QA/QC program should include the re-analysis of failures outside of the accepted ranges (>3SD) for standards that are within mineralized zones. The re-runs should include 10 samples above the failed standard, the standard, and 10 samples below the failed standard.

The Company should reconsider using certain CRMs from Moment Exploration Geochemistry LLC (MEG) as some of these materials display inconsistencies and biases in using CRMs with elevated failure rates resulting from higher-than-expected assay variability. A different CRM provider may offer improved accuracy, more precise values, or greater consistency across batches, which can improve the reliability of results. The QP recommends CRMs from suppliers such as Rock Labs, OREAS, and CDN laboratories.

In addition, standardizing the CRM selection and utilizing fewer high-quality CRMs to improve continuity and increase sample populations would ensure a more accurate trend analysis. In general, a low grade, medium grade and high-grade CRM of representative mineralogy types along with a blank pulp CRM should be sufficient for QA/QC evaluation and are recommended by the QP.

In addition, the QP believes that future exploration can be enhanced by:

- A thorough preparation program for the standard samples with adequate mixing and splitting of the samples entered into the analytical stream.
- Use of certified blank samples or round robin analysis of the blank material, rather than assuming the metal concentration.
- Prompt review of the QAQC results to include the laboratory in an appropriate re-analysis if warranted.
- Check assay program at an umpire lab for future drilling and sampling programs.

11.3 Getchell Gold Specific Gravity Measurements

11.3.1 Sample Collection, Preparation and Security

The 2024 Getchell Gold specific gravity (SG) collection program was carried out on core from select drill holes from the 2020-2022 drilling programs. The SG measurements were first taken in the field by Getchell Gold's personnel. A portion of those core samples were sent to Bureau Veritas Laboratories' ("BVL") facilities in Sparks, NV for additional SG analysis . A total of 1,382 core were subject to SG measurements in the field, and 121 of them were subsequently sent to BLV. BVL is accredited to ISO/IEC 17025 and ISO 9001 and is independent of the issuer and the authors of this report.

For field measurements, core samples were taken approximately every 2 meters downhole. For lab measurements, samples were taken every 4 to 6 meters downhole, down to about 200-330 meters. Core samples were ideally at least 10-cm long and were selected from both mineralized and non-mineralized intervals, and well representative of the lithologies and alteration contained within the Mineral Resource.

At BLV, samples underwent SG measurements through two series of measurements: 1) at the state they were submitted (without any coating), and 2) coated with paraffin wax.

11.3.2 Analytical Procedures

The methodology for SG measurement was as follows:

1. A specific gravity balance is placed on a level stable surface and calibrated.

2. The sample is weighed dry in air on the balance, and its weight is recorded.

3. The sample is fully immersed in water via a basket hung on a hook under the balance, and its weight is recorded.

4. The SG value is then obtained by dividing the weight of the sample in air by its weight in air minus its weight in water.

11.4 Adequacy of Sample Collection, Preparation, Security and Analytical Procedures

Based upon a review of Getchell Gold and other companies' 1981 to 2022 sample collection, sample preparation, security, analytical procedures, and QA/QC procedures used at the Fondaway Canyon Project, it is the opinion of the QP that they are appropriate for the type of mineralization that is being evaluated and the stage of the Project. Assay results from modern drilling including Getchell Gold's drilling largely confirm results from the historical drill holes. The QA/QC measures, including the insertion rates and performance of blanks, standards, and duplicates for the 2020, 2021, and 2022 drilling by Getchell Gold indicate the observed failure rates are within reasonable expected ranges and no significant assay biases were apparent.

Based upon the evaluation of the drilling, sampling and QA/QC programs completed by Getchell Gold and reviewed by APEX personnel, it is the opinion of the QP that the Fondaway Canyon Project's drill and assay data are appropriate for use in the resource modeling and subsequent estimation work discussed in Section 14.

12. DATA VERIFICATION

12.1 Data Verification Procedures

Getchell Gold provided APEX two separate Microsoft Access databases containing relevant drill hole data including drill hole collar locations, downhole surveys, assays, QA/QC data, downhole geological and geotechnical information. The databases were found to be well organized. Verification of the drill hole database by the QP included a review of the various digital drill hole data tables provided by Getchell Gold which were compared against scans of hard copy logs, surveys and collar files for historical and Getchell Gold drill programs.

Assay certificates were available for 75% of the assay results. Over the course of the exploration programs various laboratories have been used for analysis. Assay certificates from Shasta, Barringer, Cone, GSI, GDR, American Assays, and BVL clearly state the analysis method used (ex. FA30, FA430) and provide comprehensive assay data. Assay certificates from GDR for 7,250 samples do not list the analytical method that was used. It is assumed that these analyses were completed using fire assay because the other drill holes with TF- prefix were analyzed by fire assay. A total of 9,353 samples have handwritten assays recorded on drill logs with 6,981 out of 9,353 samples are noted to be analyzed by fire assay. A total of 2,282 assays with unknown analysis methods are sourced from handwritten log sheets.

A total of 386 sample assays do not have any assay certificates or corresponding values on drill logs. These samples correlated with drill holes TF-036 – 042, 044, 046, 048, 051, 052, and TF-064.

APEX personnel, under the supervision of Mr. Dufresne, have randomly checked around 13.8% of assays by comparing the database recorded assay value to the original assay certificate value (Table 12-1). Only 19 errors have been identified in 3,975 checked assay records. However, the identified errors are negligible with most being in the third decimal digit. Additionally, APEX personnel compared original assay certificates to assay results in the Getchell Gold database for drill holes FCG22-020 to FCG22-028 completed by Getchell Gold in 2022 and used in the 2024 Fondaway MRE detailed in Section 14 of this Technical Report. The certified PDF assay values were checked against the drill hole database. Seven minor typos were found and rectified.

Table 12-1: Assay Data Verification Outcome

Drill Holes by Drilling Program	# of Assays	# of Assays Checked	Assay Verification Percentage	# of Errors Found	Assay Error Percentage	Comments
OXYR-01 - OXYR-18	472	27	5.7%	3	11.1%	Missed decimal digit
HFC-1 - HFC-4	559	30	5.4%	0	0.0%	
NBRC-01 - NBRC-18	405	32	7.9%	0	0.0%	
RC-19 - RC-87	1361	190	14.0%	0	0.0%	
SM-002 - SM-122	2910	197	6.8%	1	0.5%	It is 0.005 not 0.001 opt
T-01 - T-35	3958	196	5.0%	2	1.0%	Incorrect average of two repeats
TF-001 - TF-340	19229	891	4.6%	5	0.6%	Rounding issue
M-01 - M-19	533	58	10.9%	0	0.0%	
P-01 - P-30	677	36	5.3%	0	0.0%	
CR-09 - CR-14	180	40	22.2%	0	0.0%	Long decimal issue needs to be rounded
02FC-01 - 02FC-11	2075	258	12.4%	1	0.4%	02FC-11 990-1000 sample result was omitted
FC17-01 - FC17-07	1892	153	8.1%	0	0.0%	Long decimal issue needs to be rounded
FCG20-001 - FCG21-016	5121	214	4.2%	0	0%	
FCG22-017 – FCG22-019	804	0	0	0	0	Missing
FCG21-020 – FCG21-028	3761	3761	100%	7	0.2%	Typos
Total	**44226**	**6083**	**13.8%**	**19**	**0.3%**	

In the opinion of the QP, the Fondaway Canyon drill hole database is reasonably free of any material or systematic errors and is suitable for use in this Report, and to support geological interpretations and the 2024 Fondaway Canyon MRE, detailed in Section 14 of this Technical Report.

12.2 Qualified Person Site Inspection

Michael Dufresne, M.Sc., P. Geol, P. Geo., author and QP, conducted a site inspection of the Fondaway Canyon Property for data verification purposes on May 7th and 8th, 2022, while the 2022 drill program was in progress. The site visit included a property tour facilitated by Mr. Mike Sieb, a geologist and President of Getchell Gold. The general geology, mineralization style and alteration were observed and compared with published interpretations and select drill collar locations and orientations were verified and cross-checked against the exploration database. Additionally, time was spent at the core facility reviewing the recent and historical core stored at that facility and collecting verification samples. Core handling, sampling and QA/QC procedures were discussed with Mr. Mike Sieb. Access to the site was via secondary highways and gravel roads.

The objectives of the site visit included:

- Verification of selected drill hole collar locations.
- Observation and sampling of historical showings in outcrop.
- Examination of drill core and observation of mineralized intercepts.
- Collection of verification samples.

The Property visit included stops at the South Mouth, Mid-Realm, Colorado, Upper and Lower Stibnite (Half Moon) Pits. Historical drill collars are rarely present and are mostly marked with stacked rocks covering the collar and a wooden stake. On occasion they are marked with a cement plug. Drill collars encountered during the site visit were located using a hand-held GPS (Table 12-2; Figure 12-1). Getchell Gold drill hole collars and drill pads were also visited for 12 holes drilled from 4 drill pads. The locations of the Getchell Gold drill holes recorded by the QP agree within error of those recorded in the database (Table 12-2; Figure 12-1). Getchell Gold is currently drilling on the Property and re-using multiple drill pads through various years so the 2021 – 2022 drill pads have not yet been reclaimed.

Table 12-2: Drill Hole Collar Location Verification

	Site Visit		Database		Difference (m)	
Drill Hole	**X N83 Z11**	**Y N83 Z11**	**X**	**Y**	**X**	**Y**
FGC21-09 & 10	397118	4406474	397119	4406467	-1	7
FGC20-02 & 03; FGC21-07, 08, 13, 14	396912	4406681	396913	4406680	-1	1
FGC20-05 & 06; FGC21-15	396654	4406493	396655	4406495	-1	-2
FGC20-01	394668	4406171	394667	4406172	1	-1

A total of six surface composite rock grab verification samples were collected from selected outcrops at the Mid Realm – South Mouth area and at the Main Central Zone. Rock grab samples were collected from quartz vein stockworks hosted in gossanous metasediments and breccias. The samples yielded anomalous gold values consistent with the style and tenor of mineralization previously described on the Property. Verification rock grab sample descriptions and assays are presented in Table 12-3 and shown on Figure 12-2.

During the site visit, selected intervals of mineralized core from the drilling program were examined at the Fallon facility. A total of 8 core holes were reviewed from the 2021 and 2022 drill programs. The observed geology was consistent with the drill database descriptions. Additionally, the intervals examined contained sulfide assemblages and/or gossan consistent with the reported mineralization. Two verification samples were collected for assay from drill hole FGC21-08. In general, there is reasonable agreement between the original assay results and verification sample results (Table 12-4), despite difference in sample size (half-core vs. quarter randomly selected core). The results for the QP verification samples both returned higher assay values than the original samples, but within reason for a gold rich system.

All verification samples were submitted for analysis to ALS Global's (ALS) facility in Vancouver, BC. ALS is an International Standard ISO/IEC 17025:2005 certified laboratory and is independent of the Company and the authors of this Technical Report. Samples were analyzed using ALS's ME-MS61 48 element, four-acid ICP-MS package.

In the opinion of the QP, visual inspection and verification sampling confirm the presence and style of historically and recently reported mineralization.

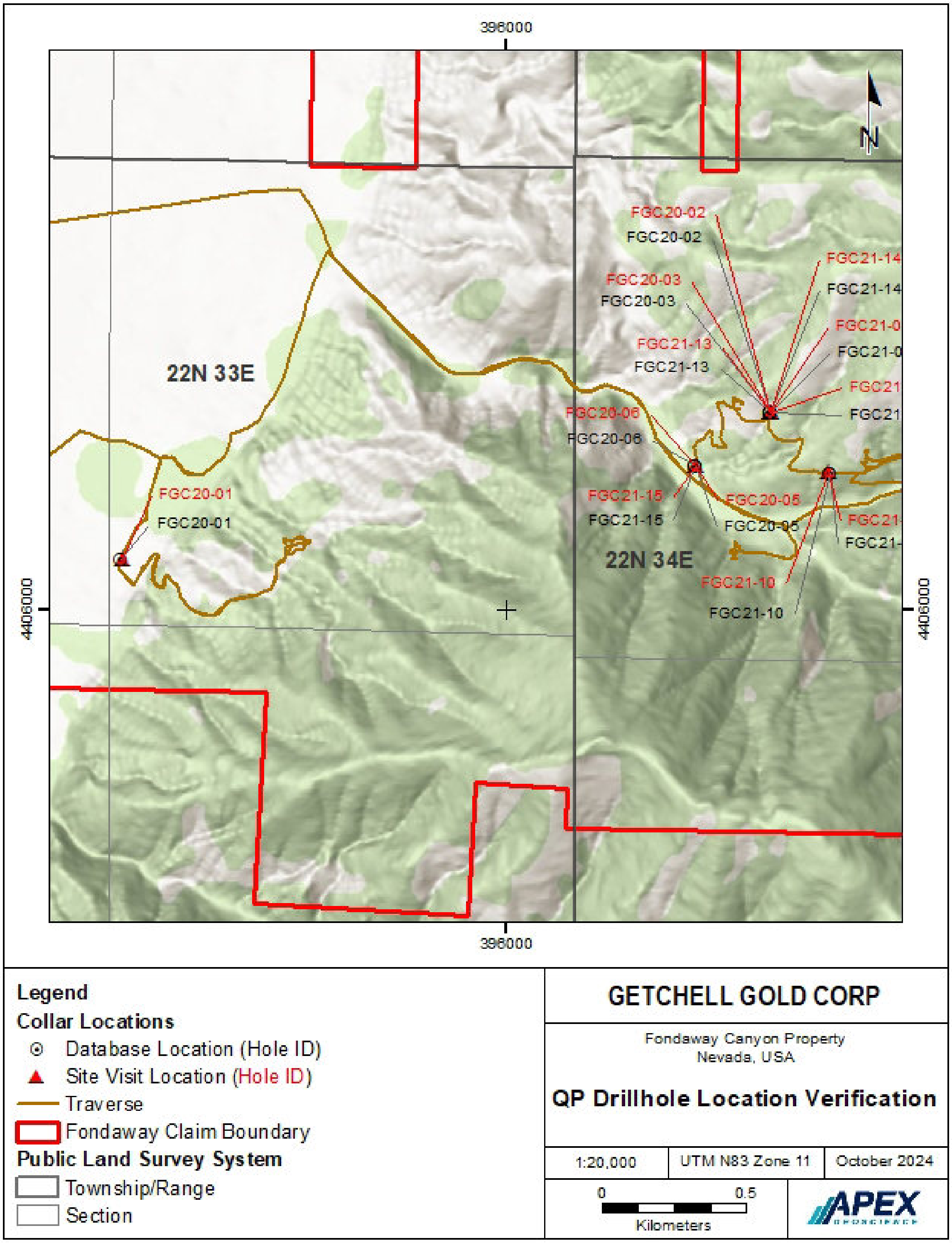


Figure 12-1: QP Drill Hole Collar Location Verification

(Source: APEX Geoscience Ltd.)

Table 12-3: Verification Grab Sample Results from the Fondaway Canyon Property

Sample	Easting N83Z11	Northing N83Z11	Au ppm	Comments
22MDP401	395299	4406207	0.148	Quartz veined gossanous metasediment - south wall of E-W Pit at South Mouth/Mid Realm area
22MDP402	395300	4406221	0.062	Composite of quartz veined (epithermal textures) rubble from outcrop north side of E-W Pit at South Mouth/Mid Realm area
22MDP403	396792	4406177	4.43	Black Mudstone/Shale - Quartzite band; Gossanous & lots of carbon - brecciated - some quartz vein material; Little Pit South of the Main Canyon Road
22MDP404	396798	4406173	3.90	Gossanous altered Phyllite - 0.5 - 1 cm flat quartz vein and vein stockwork; comp grab across phyllite and stockwork zone
22MDP405	397246	4406547	20.0	East Side of Upper Stibnite Pit - Sample of hydrothermal breccia in sediments - vertical structures - E-W Half Moon Trend coming thru N-S Pit - Composite over 1+m
22MDP406	397242	4406547	22.6	West Side of Upper Stibnite Pit - blasted hydrothermal breccia and mélange of qtz vein-stockwork material in argillaceous sediments - intersection of NE-SW structure and E-W Half Moon - comp over 1+m
22MDP407	396913	4406680	5.20	Dup of core sample 593149 (4.51 ppm Au) in drill hole FGC21-008 and at 283.7 m to 285.5 m - qtz vein stockwork in chippy mudstone with fine qtz veinlets and pyrite
22MDP408	396913	4406680	8.67	Dup of core sample 593150 (6.916 ppm Au) in drill hole FGC21-008 and at 285.5 m to 286.0 m - qtz vein stockwork in chippy mudstone with fine qtz veinlets and pyrite up against grey andesite dyke

Table 12-4: Comparison of QP Verification Core Sample Results vs Original Results

Hole FGC21-08	X N83Z11	Y N83Z11	From (m)	To (m)	QP Sample (ppm)	Original (ppm)	Difference (ppm)	Difference %
22MDP407	396913	4406680	283.7	285.5	5.20	4.51	0.69	15%
22MDP408	396913	4406680	285.5	286.0	8.67	6.92	1.75	25%

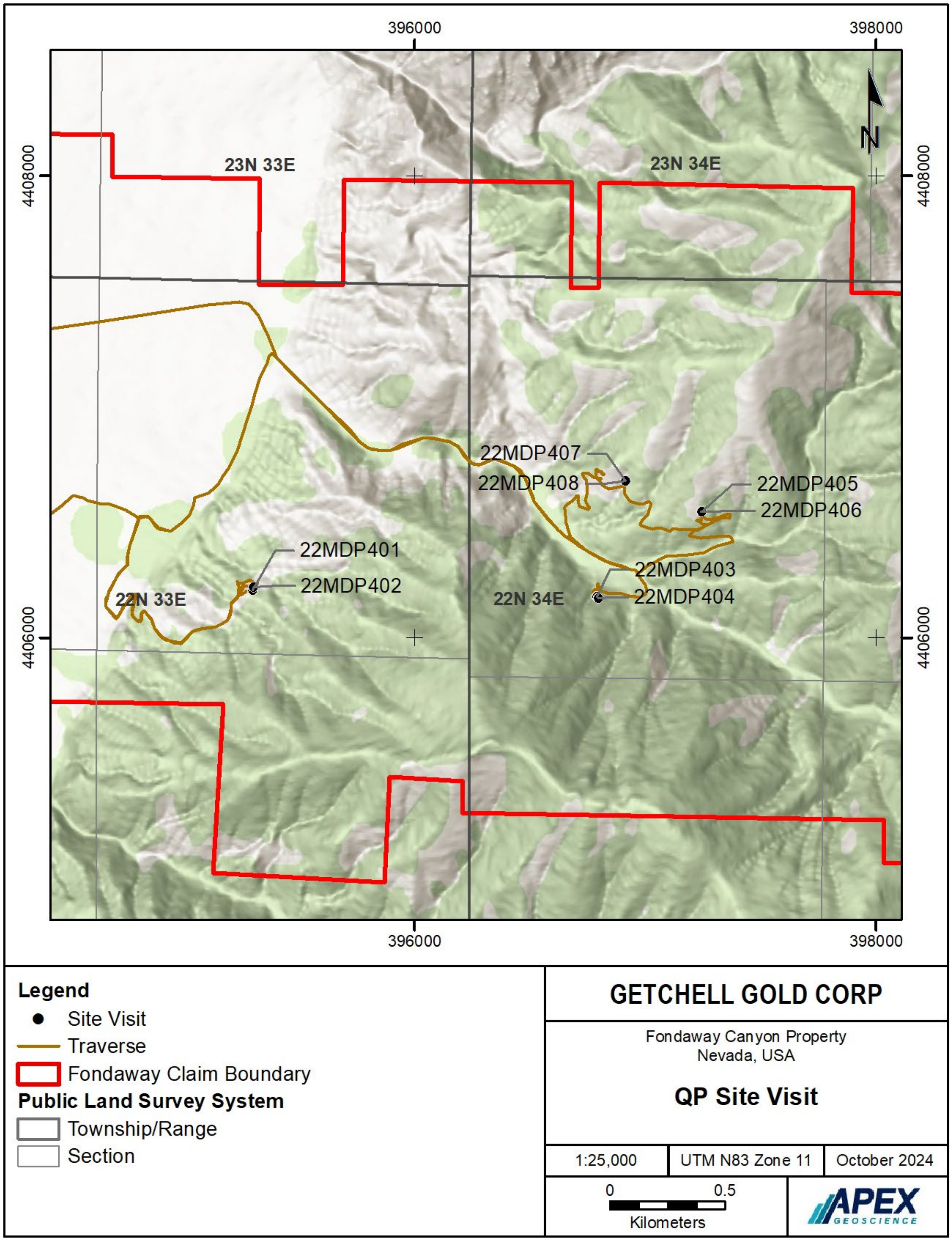


Figure 12-2: 2022 QP Verification Sample Locations

(Source: APEX Geoscience Ltd.)

12.3 Validation Limitations

Assay certificates for some of the older historical drill holes were not available and assays have been verified against values recorded on drill logs.

Based on the site inspection, verification sampling, and data review, the QP has no reason to doubt the reported geology and exploration results.

12.4 Adequacy of the Data

The QP has reviewed the adequacy of the exploration information and the visual, physical, and geological characteristics of the Property and found no significant issues or inconsistencies that would cause one to question the validity of the data.

Based upon the evaluation of the drilling, sampling and QA/QC programs completed by historical operators and Getchell Gold, and reviewed by APEX personnel, it is Mr. Dufresne's opinion that the Fondaway Canyon drill and assay data are appropriate for use in the resource modeling and estimation work discussed in this Technical Report and Section 14.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

Several scoping level mineralogical and metallurgical studies have been undertaken on the Fondaway Canyon Property from 1984 to 2017. Getchell Gold undertook a scoping level metallurgical study in 2024 to advance the Project by developing a conceptual process flowsheet for the oxide and sulfide ores that required minimum CAPEX and OPEX.

13.1 Historical Metallurgical Test Work

The following documents were reviewed to summarize the historical test work undertaken on the Project:

- Hazen Research Reports dated July 7, 1988, February 28, 1989, June 28, 1989, and September 11, 1989 for Tenneco Minerals.
- Flotation Testing on Fondaway Canyon Samples, Barrick memo dated December 6, 1990.
- Pertrologic Studies of Drill Core from Fondaway Canyon Gold for Aorere Resources Ltd, September 2016.
- Laboratory Scale Flotation and Gravity Concentration Testing – Fondaway Canyon Drill Core, McClelland Laboratories, Inc. for Canarc Resources Corp, March 27, 2017.
- Canarc Resources Corp NI 43-101 Technical Report, Norred and Henderson, 2017.
- Desk-Top Due Diligence Review of Metallurgical Test Programs and Documentation for the Fondaway and Dixie Comstock Projects, Nevada, Samuel Engineering, March 9, 2020 (Kuestermeyer, A., 2020).
- Getchell Gold Corp NI 43-101 Technical Report, 2022.

The mineralogy and metallurgical results have been reviewed in the two Technical Reports. The highlights of the several studies on the samples from the prospect indicate the following:

- The deposit is characteristic of carbonaceous pyritic refractory gold ore.
- Conflicting mineralogical information was noted in the reports:
    - o “Gold seen in these samples is extremely fine grained (5-20 microns), and is either enclosed by quartz or associated in space with the relatively abundant organic carbon” (Russ Honea, May 22, 1984)
    - o “The complex sulfosalt mineral assemblage… is shown to be present in both drill holes. Carbon is common in both drill holes. Extremely fine grained sulfides – including pyrite and arsenopyrite – often accompany the carbon.” (Russ Honea, July 27, 1984).
    - o “Native gold is present as a very small grain (4 microns) enclosed by pyrite in sample” (Russ Honea, August 7, 1984).
- The majority of the metallurgical studies have been performed on material assaying ± 6 g/t Au.
- Cyanidation with/without carbon indicate the preg robbing characteristics of the ore. However, surface oxide ore can be leached by cyanide.
- The best gold extraction was obtained by roasting the ore at 675°C – 750°C followed by acid Pox as second best.

- Flotation tests performed in 2017 on samples assaying 3 to 6 g/t Au indicated decent recovery of gold (± 80%). These samples contained 0.25% to 2% As and ± 1.64% C_T.

- The ore is not amenable to gravity concentration.

13.2 2024 Forte Analytical Metallurgical Test Work

Getchell Gold contracted Forte Analytical in 2024 to complete a scoping level metallurgical study for the Fondaway Canyon Project with the primary objective to develop a conceptual process flowsheet for the oxide and sulfide samples that minimizes both CAPEX and OPEX. The test program was expanded to include processing of oxide ore which occurs on the surface of the sulfide deposit.

The preliminary metallurgical report is given in Appendix A.

13.2.1 Sample Preparation and Head Analyses

Forte Analytical received approximately 180 kgs of average-grade sulfide and ± 50 kgs each of high-grade and low-grade sulfide analytical rejects for the study.

Two shipments of oxide analytical rejects and core consisting of ± 20 kgs and ± 50 kgs were received for the testing of oxide samples.

There are a total of five composites, which are listed in Table 13-1.

Table 13-1: Description of Five Composites for Metallurgical Test Work

Composite	Description	Material
1	Average Grade Sulfide	Analytical Rejects
2	Low Grade Sulfide	Analytical Rejects
3	High Grade Sulfide	Analytical Rejects
4	Oxide Composite 1	Analytical Rejects
5	Oxide Composite 2	Core

The samples were stage crushed to 100% passing 6 mesh, blended and split into 1 kg and 10 kg charges. One 1 kg charge was split and a portion was pulverized for head analyses. A portion of the average grade composite was sent for mineralogical study.

The test results are presented in Table 13-2 through Table 13-4. The test results indicate the following:

- The average-grade composite assayed 1.49 g/t Au, 1.60 g/t Ag, 2.58% S_{Sulfur}, and 0.33% $C_{Organic}$.

- The average-grade sample contained 1,775 ppm As and 3.58% Fe, whereas As assays of low- and high-grade composites were 2,548 ppm and 1,666 ppm, respectively.

- The high-grade composite assayed 4.93 g/t Au and the low-grade composite assayed 0.53 g/t Au. The low-grade sample contained 0.33% organic carbon.

- The oxide composites assayed 1.5 g/t Au and 1.86 g/t Au.

- None of the samples contained an economic quantity of silver.

Table 13-2: Head Analyses of Sulfide Samples

Assay	Composite		
	Average Grade	Low Grade	High Grade
Au, g/t	1.49	0.53	4.93
Ag, g/t	1.60	1.0	1.10
S_{Total}, %	2.66	0.13	0.04
$S_{Sulfide}$, %	2.58	0.05	0.01
$S_{Sulfate}$, %	0.08	0.08	0.03
C_{Total}, %	1.74	0.59	0.11
$C_{Organic}$, %	0.33	0.33	0.05
$C_{Inorganic}$, %	1.40	0.26	0.06

Table 13-3: ICP Analyses of Sulfide and Oxide Composites

Element	Average Grade	Low Grade	High Grade	Oxide 1 (Comp #4)	Oxide 2 (Comp #5)
As	1755	2548	1630	BD	577
Ba	2444	1666	1497	47	786
Ca	30266	21379	10853	BD	25026
Cr	89.3	72.7	102	BD	91
Fe	35761	35614	19968	BD	25523
K	16258	25133	16129	90	17228
Mg	11548	9033	6238	BD	5700
Mn	379	345	226	BD	602
Na	482	BD	BD	BD	1039
Ni	24.2	27.6	15.2	BD	23
P	497	567	311	BD	52.2
Pb	28.9	27.5	20.8	BD	29
S	24580	28556	20057	BD	4850
Sb	40.7	BD	33.1	24	BD
Sr	214	213	106	BD	180
Ti	3175	2452	2084	49	2397
U	142	142	106	BD	125
V	75.3	74.6	52	BD	102
Zn	70.5	66.6	36.2	BD	81

Note: BD – Below Detection

Table 13-4: Head Analyses of Oxide Composites

	Composite	
	Composite 4	Composite 5 (Core)
Au, g/t	1.38	1.86
Ag, g/t	1.1	1.8
STotal, %	0.02	0.05
SSulfide, %	0.02	0.0
SSulfate, %	0.01	0.04
CTotal, %	0.85	1.0
COrg, %	0.21	0.20
Cinorg,%	0.64	0.80

13.2.2 Mineralogical Evaluation of Sulfide Composites

The mineralogy study was undertaken to determine the major minerals in the ore and liberation characteristics of gold particles. The highlights of the study indicated the following:

- The major minerals in the ore are quartz and orthoclase with minor amounts of pyrite, muscovite, ankerite, dolomite, and calcite.
- Gold particles observed in average-and low-grade composites were approximately 5 microns and were associated with pyrite.
- Two free particles at approximately 5 microns in size were identified in the high-grade composite.

13.2.3 Comminution

Bond's ball mill work indices were determined at a P_{80} of 100 mesh for the composite samples except for Oxide Composite 1. The comminution data, summarized in Table 13-5, indicates that the oxide ore has an average hardness whereas the sulfide ores can be designated as slightly hard ores.

Table 13-5: Bond's Ball Mill Work Indices for Composite Samples

Composite	BWi (Kwh/st)
Average Grade Sulfide	15.54
Low Grade Sulfide	15.82
High Grade Sulfide	15.46
Oxide Composite 2	13.62

13.2.4 Diagnostic Leach (Gold Deportment) of Average-Grade Sulfide Ore

A series of sequential leach tests were performed with intermediate roasting steps to determine the association of gold with various minerals (i.e., free milling, associated with pyrite, arsenopyrite, etc.).

The test flowsheet is given in Figure 13-1.The test results, summarized in Table 13-6, indicate the following:

- The ore is refractory with 1% of the gold leaching in the direct cyanidation process.
- Only 5.8% of the gold is associated with arsenopyrite.
- A majority of the gold is associated with pyrite (77.5%).
- Approximately 15.7% of the gold is encapsulated in silica.

These results correlate with the mineralogy which has indicated gold association with pyrite and being extremely fine (± 5 microns) which may require fine grind to expose it to cyanide for leaching.

Table 13-6: Deportment of Gold in Average Grade Sulfide Composite

Composite	Feed g/t Au	% Extraction Au			
		Free Milling	Arsenopyrite Association	Pyrite Association	Silica Encapsulation
Average Grade Sulfide	1.55	1.0	5.8	77.5	15.7

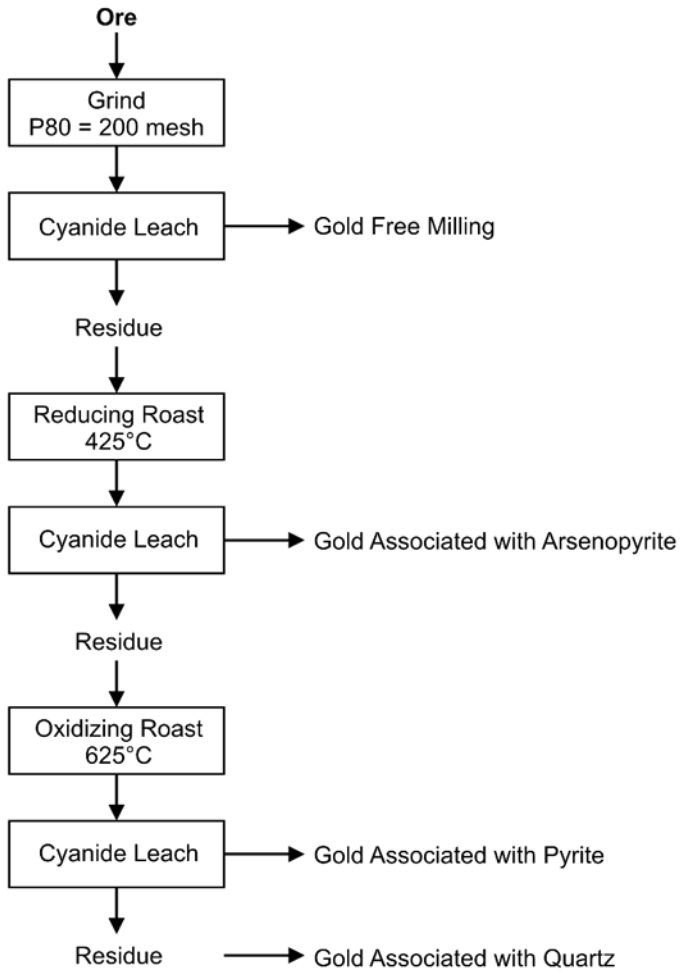


Figure 13-1: Diagnostic Test Procedure for Deportment of Gold

(Source: Forte Dynamics, Inc. 2024)

13.2.5 Flotation Tests

The test work was initiated on average grade sulfide ore with the objective of determining the techno-economically viable process flowsheet.

The sulfide composites and Oxide 2 composite were ground in a laboratory rod mill which simulates a ball mill-cyclone circuit in an actual operation. Several grinding tests for varying grind times were performed to determine the relationship between grind time and grind size. The grind times were determined for achieving P_{80} of 100, 150, 200, and 270 mesh.

13.2.5.1 Average-Grade Composite

A series of flotation tests were performed using the average-grade composite to determine the optimum grind size, flotation time, and reagent dosages to maximize gold recovery in the concentrate. The reagent suite consisted of potassium amyl xanthate, Aeropromotor 404, and frothers MIBC and AF65. These reagent combinations float both sulfides and gold.

The test results, summarized in Table 13-7, indicated the following:

- The finer the grind, the higher the gold recovery. Approximately 81.6% of gold was recovered in 19.4% of the weight. The concentrate assayed 7.40 g/t Au.

- The tailing assay for P_{80} of 270 mesh was lower than that for 200 mesh (0.36 g/t vs. 0.41 g/t Au), though the recovery was only 80%. This was due to lower calculated feed grade.

Table 13-7: Flotation Test Results for Average Grade Composites

Test No.	Grind Size, P_{80} Mesh	Rougher Recovery %		Grade, g/t Au		
		Wt	Au	Concentrate	Tailing	Feed
1	100	14.7	73.8	7.85	0.48	1.57
2	150	17.5	76.9	6.69	0.43	1.53
3	200	19.4	81.6	7.40	0.41	1.79
4	270	17.9	80.0	6.58	0.36	1.48
6	200	21.3	84.9	6.40	0.31	1.60
7	270	26.8	87.3	4.92	0.26	1.51

Note: Flotation Time = 12 min.

The flotation tailing from Test 1 (P_{80} of 100 mesh) was subjected to gravity concentration using a Knelson concentrator. The test results, summarized in Table 13-8, indicate that one could get 33.6% of the gold lost to the flotation tailing by gravity. This would increase the flotation plus gravity recovery from 80% to ± 88%.

Table 13-8: Gravity Concentration of Flotation Tailing

Product	Recovery %		Grade, g/t Au
	Wt	Au	
Gemeni Concentrate	0.5	6.5	3.57
Gemeni Tail	5.7	27.1	1.22
Cal. Knelson Concentrate	6.2	33.6	1.41
Knelson Tails	93.8	66.4	0.18
Cal. Feed	100	100	0.25

13.2.5.2 10 kg Rougher Flotation Test

A 10 kg rougher flotation test was performed at a primary grind of P_{80} of 270 mesh to generate a concentrate for cyanide leaching to recover gold. The flotation time and reagent additions were scaled up for larger flotation test.

The test results, summarized in Table 13-9, indicated that **89.8% of the gold was recovered in a concentrate assaying 6.07 g/t Au**.

Table 13-9: Flotation Test Results for One-Cubic Foot Flotation Cell (10 kg Charge, Test 5)

Product	Recovery %		Grade, g/t Au
	Wt	Au	
Rougher Concentrate	20.7	89.8	6.07
Rougher Tail	79.3	10.2	0.18
Cal. Feed	100	100	1.40

The tailing from the test assayed 0.18 g/t Au. One kilogram of the tailing was taken and floated for additional time to evaluate if one could recover additional gold. The test results, summarized in Table 13-10, indicate that ore could recover approximately 32.4% of gold in an additional 7.8% of weight. The concentrate assayed 0.76 g/t Au.

Table 13-10: Effect of Additional Flotation Time on Gold Recovery (Test 5, Scavenger Float)

Product	Flotation Time, min	Recovery %		Grade, g/t Au
		Wt	Au	
Scavenger Conc. 1	3	7.8	32.4	0.76
Scavenger Conc. 2	3	1.1	3.3	0.55
Scavenger Conc. 3	3	1.0	0.7	0.12
Scavenger Tail		90.1	63.6	0.18
Cal. Scavenger Feed		100	100	0.18

A portion of the bulk concentrate generated in the flotation test was analyzed by XRD to determine the major minerals. The data, summarized in Table 13-11, indicated that the major minerals in the concentrate were quartz, mica/illite, pyrite, and dolomite.

Table 13-11: XRD Analyses of Bulk Concentrate

Mineral	Approx. Weight %
Quartz	37
Mica / Illite	35
Kaolinite	<3
Dolomite	7
Calcite	<2
Rutile	<1
Pyrite	13
Arsenopyrite	<2
Unidentified	<5

Two additional flotation tests were performed at P_{80} of 200 and 270 mesh. Higher weight pull resulted in higher gold recovery. Approximately 87.3% of gold was recovered when weight recovery was 26.8%. The larger scale flotation test recovered 89.8% of gold in 20.7% of the weight.

These results indicate that one needs to recover ± 20% of weight to get 87% - 88% of gold recovery.

13.2.5.3 Low-Grade Composite

Rougher flotation tests were performed with low-grade composite using the optimum process parameters developed for average-grade composite.

The test results, summarized in Table 13-12, indicate the following:

- Finer grind did not improve gold recovery. One needed to float ± 20% of weight in order to get gold recovery in the high 70s.

Table 13-12: Flotation Test Results for Low-Grade Composite

Test No.	Grind Size, P80 mesh	Rougher Recovery %		Grade, g/t Au		
		Wt	Au	Concentrate	Tailing	Feed
8	200	23.4	79.1	1.74	0.14	0.51
9	270	20.6	77.0	2.01	0.16	0.54

13.2.5.4 High-Grade Composite

Rougher flotation tests were also performed on high-grade composite at P_{80} of 200 and 270 mesh.

The test results, summarized in Table 13-13, indicate the following:

- Gold recovery was independent of particle size. A grind of 200 mesh appears to be optimum.
- Weight recovery of ± 20% was required to achieve ± 85% of gold.

Table 13-13: Flotation Test Results for High-Grade Composite

Test No.	Grind Size, P80 mesh	Rougher Recovery %		Grade, g/t Au		
		Wt	Au	Concentrate	Tailing	Feed
10	200	20.7	85.3	22.4	1.01	5.45
11	270	22.2	85.8	21.2	1.00	5.47

13.2.5.5 Production of Rougher Concentrate for Leaching Tests

Rougher flotation tests were performed for the three sulfide composites to generate rougher concentrate for cyanidation leach tests.

Average grade concentrate was produced in a one-cubic flotation machine using 10 kg ore, whereas three 1 kg tests each were run for the other two composites.

The test results, summarized in Table 13-14, were similar to those obtained for the composites in earlier tests except for the average-grade composite. The recovery was ± 8% lower due to higher tailing assay (i.e. 0.38 vs. 0.18 g/t Au).

The concentrates were submitted for multi-element analyses. The results indicated that the rougher concentrates assayed ± 1% As, ± 200ppm Sb, and ± 4.5 ppm Hg. This concentrate will be upgraded in two cleaner flotation stages to produce a product assaying ± 20 g/t Au and should be readily marketable to smelters or POX facilities.

Table 13-14: Flotation Tests to Generate Concentrate for Leaching (P_{80} = 270 mesh)

Test No.	Test Type	Rougher Recovery %		Grade, g/t Au		
		Wt	Au	Concentrate	Tailing	Feed
12	Avg. Grade, 10kg	20.7	81.1	6.23	0.38	1.59
13	Low Grade 3 1kg Tests	26.1	78.2	1.62	0.16	0.54
14	High Grade 3 1kg Tests	26.4	85.4	17.2	1.05	5.31

13.2.6 Whole Ore Leaching (WOL)

The whole ore leaching tests were performed for the oxide and sulfide composites. The ore was ground to P_{80} of 100 and 200 mesh and leached for 48 hours at 40% solids with varying cyanide concentration (1 to 2 g/L NaCN).

The test results, summarized in Table 13-15 and Table 13-16, indicate the following:

- Direct cyanidation of oxide ore extracted 50% to 62% of gold in 2 hours. The gold recovery dropped to 37% to 49% in 48 hours of leaching, thereby indicating the ore exhibited preg-robbing properties.
- The carbon-in-leach for oxide ore recovered 62.1% to 71.6% of gold in 24 hours at a grind of P_{80} of 270 mesh. The NaCN consumption was reasonable at ± 1 kg/t and the lime consumption was ± 2.5 kg/t.

- The sulfide ores also exhibited preg-robbing properties besides being refractory ore. The gold extraction for average-grade sulfide ore was 9.5% in 2 hours and dropped to 4.2% in 48 hours. The high-grade sulfide composite had 41.8% of gold extraction in 4 hours but dropped to 29.4% in 48 hours.

Table 13-15: Whole Ore Leach of Oxide Composites

Parameter	Test No					
	4	5	6	7	12	13
Sample	Comp #4	Comp #4	Comp #5	Comp #5	Comp #4	Comp #5
Grind, P_{80} mesh	100	200	100	200	270	270
Extraction, % Au						
2 hr	50.3	51.4	57.9	62.2		
4 hr	50.3	50.9	55.9	61.6		
8 hr	48.8	48.6	55.2	60.0		
24 hr	44.3	42.7	50.6	52.9	62.1	71.6
48 hr	41.2	37.6	48.3	49.1		
Residue, g/t Au	0.79	0.82	1.03	1.00		
Cal. Feed, g/t Au	1.34	1.31	1.99	1.96		
Consumption, kg/t						
Lime	2.903	2.699	3.116	3.007	2.583	2.562
NaCN	0.363	0.422	0.419	0.842	1.025	0.955

Note: Tests 12 & 13 are CIL

Table 13-16: Whole Ore Leach of Sulfide Composites

Parameter	Test No		
	9	10	11
Sample	Average Grade	Low Grade	High Grade
Grind, P_{80} mesh	200	200	200
Extraction, % Au			
2 hr	9.5	22.5	27.1
4 hr	8.2	23.4	41.8
8 hr	5.5	22.3	39.9
24 hr	4.2	20.8	33.2
48 hr	4.2	20.3	29.4
Residue, g/t Au	1.41	0.46	3.67
Cal. Feed, g/t Au	1.47	0.58	5.20
Consumption, kg/t			
Lime	1.495	1.597	0.897
NaCN	1.678	1.679	1.798

13.2.7 Leaching of Sulfide Flotation Concentrate

The leach tests were performed on flotation concentrates generated from the sulfide ores. The concentrates were also reground to determine if one could liberate gold from sulfides (i.e. pyrite/arsenopyrite) and leach it. The test results, summarized in Table 13-17, indicate the following:

- The flotation concentrate exhibited preg-robbing properties. The gold extraction tended to decrease with leach time.

- The flotation concentrate from average-grade composite recovered ± 20% of gold.

- Regrind of concentrate did improve gold extraction to 37.5% but decreased as the leaching process continued. Hence, regrinding concentrate enhanced preg-robbing properties.

- Carbon-in-leach (CIL) did improve gold extraction to 50% to 55% for the sulfide composites.

- Leaching of flotation tailing assaying 0.18 g/t Au resulted in gold extraction of 64% in two hours but dropped to 50.9% in 48 hours of leach time. Hence, even flotation tailing exhibited preg-robbing properties.

The leaching of flotation concentrate recovered a maximum of ± 60% of gold extraction, indicating that the ore is both refractory and preg-robbing. Consequently, pre-treatment methods for improving project economics were evaluated.

Table 13-17: Leaching of Flotation Concentrate

Parameter	Test No						
	2	3	8	14	15	16	17
Sample	Avg Grade Concentrate	Avg Grade 4hr Regrind Concentrate	Tailing	Avg Grade Concentrate Reground	Avg Grade Reground	Low Grade Concentrate	High Grade Concentrate
Flotation Test No	5	5	5	12	12	13	14
NaCN, g/L	2	2	1	5	5	5	5
Extraction, % Au							
2 hr	22.7	4.9	63.9				
4 hr	23.0	17.2	56.6				
8 hr	23.0	37.5	56.7				
24 hr	22.9	16.7	51.9	53.9			
36 hr	-	-	-	-	56.2	42.4	54.6
48 hr	20.3	10.8	50.9				
Residue, g/t Au	4.92	5.54	0.09	3.17	3.02	0.94	7.96
Cal. Feed, g/t Au	6.17	6.21	0.18	6.91	6.96	1.64	17.55
Consumption, kg/t							
Lime	2.092	2.627	3.596	1.087	0.598	6.092	4.547
NaCN	2.435	7.449	0.605	13.593	16.286	2.409	2.717

13.2.8 Roast Plus Leach Process

The test results had indicated two reasons for poor gold recovery, namely, refractoriness of ore and preg-robbing properties of the ore. A series of roasting tests at 325°C (normally designated calcining test) and 650°C under oxidizing conditions were performed for average- and high-grade sulfide composites and oxide composite 2 followed by cyanidation.

The test results, summarized in Table 13-18 and Table 13-19, indicate the following:

- Oxidizing roast at 325°C did not eliminate the preg-robbing characteristics of the ore.

- CIL tests did eliminate the preg-robbing characteristics of the ore. The gold extraction for high-grade sulfide and oxide ore improved to 53% - 57%.

Oxidizing roast at 650°C did improve the gold extraction to 89.6% for average grade sulfide, 93.1% for high-grade sulfide and 85.4% for Oxide 2 composites.

Table 13-18: Leaching of Composites Following 325°C Oxidizing Roast

Parameter	Test No.					
	18	19	22	23	26	27
Sample	**Average Grade Sulfide**	**Average Grade Sulfide**	**High Grade Sulfide**	**High Grade Sulfide**	**Oxide Comp 2**	**Oxide Comp 2**
NaCN, g/L	2	2	2		2	2
Extraction, % Au						
2hr	10.1		31.4		57.3	
4hr	8.6		29.6		56.9	
8hr	7.6		27.9		55.4	
24hr	5.7		22.4		50.6	
48hr	4.7	8.6	20.0	53.5	48.1	57.4
Residue, g/t Au	1.43	0.89	3.97	2.48	0.99	0.46
Cal. Feed, g/t Au	1.50	0.97	4.96	5.34	1.91	1.08
Consumption, kg/t						
Lime	17.856	15.476	18.363	21.628	9.163	9.057
NaCN	0.676	0.833	0.617	0.862	0.728	1.151

Note: Tests 19, 23, and 27 are CIL

Table 13-19: Leaching of Composites Following 650°C Oxidizing Roast

Parameter	Test No.					
	20	21	24	25	28	29
Sample	**Average Grade Sulfide**	**Average Grade Sulfide**	**High Grade Sulfide**	**High Grade Sulfide**	**Oxide Comp 2**	**Oxide Comp 2**
NaCN, g/L	2	2	2	2	2	2
Extraction, % Au						
2hr	81.1		88.5		82.7	
4hr	86.0		90.2		83.9	
8hr	87.5		90.0		85.1	
24hr	86.9		91.5		84.6	
48hr	86.0	89.6	92.2	93.1	85.4	82.5
Residue, g/t Au	0.22	0.20	0.44	0.44	0.32	0.43
Cal. Feed, g/t Au	1.58	1.92	5.62	6.38	2.20	2.46
Consumption, kg/t						
Lime	52.475	4.8222	8.393	7.822	-	-
NaCN	0.597	1.722	0.82	2.06	0.636	0.887

Note: Tests 21, 25, and 29 are CIL

13.2.9 Conceptual Process Flowsheet

The scoping level metallurgical study evaluated several processing options following the test work on deportment of gold which indicated that a majority of the gold was refractory and associated with pyrite.

The processing option evaluated included whole ore leach, production of flotation concentrate and fine grind concentrate and leach, roasting of concentrate at two different temperatures and leaching. Pressure oxidation of ore and flotation concentrate is on-going.

Both oxide and sulfide ore can be readily floated to produce a concentrate containing 80% to 90% of gold. The concentrate can be upgraded to reduce concentrate weight and increase the gold grade of the concentrate.

A review of the CAPEX and OPEX for various processing options indicated that the most promising approach at this stage of the study is to produce a gold-rich concentrate (± 20 g/t Au) and ship/sell it to a processing facility in Nevada.

The simplified conceptual process flowsheet is given in Figure 13-2. The flowsheet would process both oxide and sulfide ores and consist of three stage crushing, closed circuit grinding, and rougher and two stages of cleaner flotation.

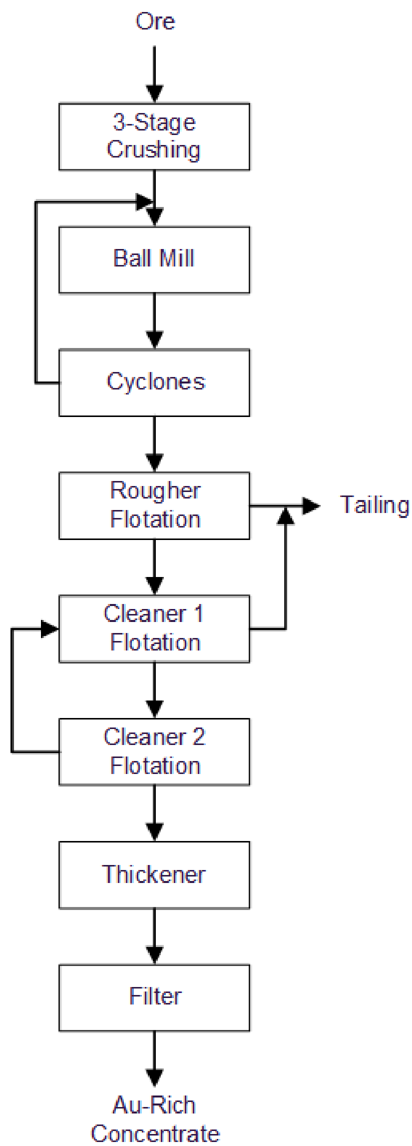


Figure 13-2: Simplified Conceptual Flowsheet

(Source: Forte Dynamics, Inc. 2024)

13.2.10 Conclusions

The following conclusions can be drawn based on the scoping level study:

- The average grade of the prospect is 1.5 g/t Au, 1.6 g/t Ag, 2.56% S_{Sulfur} and 0.33% $C_{organic}$.
- The oxide ore also averaged 1.88 g/t Au and 0.21% $C_{organic}$.
- The mineralogical study indicated that gold particles in the sulfide ore were approximately 5 microns and were associated with pyrite.
- The diagnostic leach study also confirmed that gold was associated with pyrite and the ores were preg-robbing.
- The Bond's ball mill work index for sulfide ore was ± 15.5 kwh/st and for oxide ore was 13.6 kwh/st. The sulfide ore can be designated as slightly hard ore.
- A simple reagent suite for the flotation process can recover 80% to 88% of gold in the rougher concentrate for the average-grade ore.
- Whole ore leach confirmed that the ore was not only refractory but exhibited preg-robbing properties.
- Gold can be readily extracted following the roasting or pressure oxidation of the flotation concentrate.
- Preliminary scoping studies indicate that deleterious elements are not in sufficient quantity to negatively impact the sale of concentrates. Additional test work is needed to refine the preliminary conclusions.

13.2.11 Recommendations

The metallurgical study should be continued to optimize the flotation process in order to produce a high-grade concentrate with high gold recovery.

14. MINERAL RESOURCE ESTIMATES

14.1 Introduction

Getchell Gold engaged APEX to prepare a Mineral Resource Estimate (MRE for the Fondaway Canyon Project. The 2024 Fondaway MRE has an effective date of October 31, 2024. The MRE was completed by Kevin Hon, B.Sc., P.Geo., Senior Geologist with APEX under the direct supervision of Michael B. Dufresne, M.Sc., P.Geol., P.Geo., President of APEX. Mr. Dufresne is an independent Qualified Person as defined in NI 43-101 and takes responsibility for the 2024 Fondaway MRE and Section 14 herein. Tyler Acorn, M.Sc., Geostatistician with APEX assisted with the workflow and completed a peer review.

The workflow implemented for the calculation of the 2024 Fondaway MRE was completed using Micromine commercial resource modeling and mine planning software (v2024.0), Resource Modeling Solutions Platform (RMSP; v1.14.0), and Deswik CAD pit optimization (v2024.1). Supplementary data analysis was completed using the Anaconda Python distribution and a custom Python package developed by APEX.

Mineral Resource modeling was conducted in the UTM coordinate system relative to the North American Datum (NAD) 1983 Zone 11N (EPSG: 26911). The MRE utilized a block model with a size of 3.0 meters (X) by 3.0 meters (Y) by 3.0 meters (Z) to honor the mineralization wireframes for estimation; no sub-blocking was used. Gold (Au) grades were estimated for each block using Ordinary Kriging (OK) with locally varying anisotropy (LVA) to ensure grade continuity in various directions are reproduced in the block model. The MRE is reported as undiluted.

The 2024 Fondaway MRE is reported in accordance with the Canadian Securities Administrators' NI 43-101 rules for disclosure and has been estimated using the CIM “Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines” dated November 29, 2019, and CIM “Definition Standards for Mineral Resources and Mineral Reserves” dated May 10, 2014. The effective date of the Mineral Resource Estimate is October 31, 2024.

14.2 Drill Hole Description

The 2024 Fondaway Canyon MRE drill hole database consists of a total of 525 drill holes that intersect the mineralization domains. The drilling inside the mineralization domains is summarized in Table 14-1. There is 19,148.0 meters (m) of drilling within the estimation domains. Any sample intervals with explicit documentation that drilling did not return enough material to allow for analysis are classified as insufficient recovery (IR) and left blank. Portions of the drill holes not sampled for unknown reason were assumed unmineralized, summarized in Table 14-1. These intervals were assigned a nominal waste value, set at half the detection limit of modern assay methods (Table 14-2).

Table 14-1: Summary of Drilling Inside the Mineralized Estimation Domains for Fondaway Canyon Project Drill Hole Database

Resource Area	Variable	Number of Drill holes	Total Meters	Total Samples	Number of Non-Null Assays
Fondaway	Au	525	19,148.0	14,260	14,234

Table 14-2: Nominal Waste Values Assigned to Unsampled Intervals in the Fondaway Canyon Project Drill Hole Database and Inside the Estimation Domains

Resource Area	Variable	Unit	Nominal Waste	Meters Not Sampled and Assumed Unmineralized	% Not Sampled	Number of Zero Assays
Fondaway	Au	ppm	0.0025	61.70	0.18	26

14.2.1 Data Verification

APEX validated the Mineral Resource database by checking for inconsistencies in analytical units, duplicate entries, interval, length, or distance values less than or equal to zero, blank or zero-value assay results, out-of-sequence intervals, intervals or distances greater than the reported drill hole length, inappropriate collar locations, survey and missing interval and coordinate fields. A small number of errors were identified and corrected in the database. Mr. Dufresne considers the Fondaway Canyon Project drill hole database suitable for Mineral Resource estimation.

14.3 Grade Estimation Domain Interpretation

14.3.1 Geological Interpretation of Mineralization Domains

At Fondaway Canyon, gold mineralization is localized along a trend of over 3.5 km (2 miles) of en echelon, east-northeast trending and steeply south dipping structures developed within fine grained Triassic carbonaceous siliciclastic sedimentary rocks and Jurassic limestone, cut by Tertiary dikes (Norred and Henderson, 2017).

The structural model for the Fondaway Canyon area shows that there are several schematic veins and vein (stockwork-like) zones (Figure 14-1). The zones show a reasonable degree of consistency in location, thickness, and grade. This consistency has allowed for the interpretation of mineralized zones which are used as distinct domains during the development of the resource model.

The Fondaway Canyon area is interpreted as an east-west district left lateral shear zone with a dilation zone (releasing bend) with north-northeast mineralized structural strands hosting the Main Zone resource and linking a throughgoing ~east-west district-scale mineralized fault zone. Dilation zone and brittle zone quartz veins and stockworks along with sulfide mineralization likely developed late in the history of the shear zone.

The current resource model is primarily based on the structural model shown in Figure 14-1. However, the adjacent mineralized areas with a similar structure are combined into three main zones to model.

- Main Zone (Colorado, Main Pit, Half Moon, Paperweight, South Pit, West Pits, and Pack Rat)
- Mid Realm and South Mouth
- Silica Ridge and Hamburger Hill

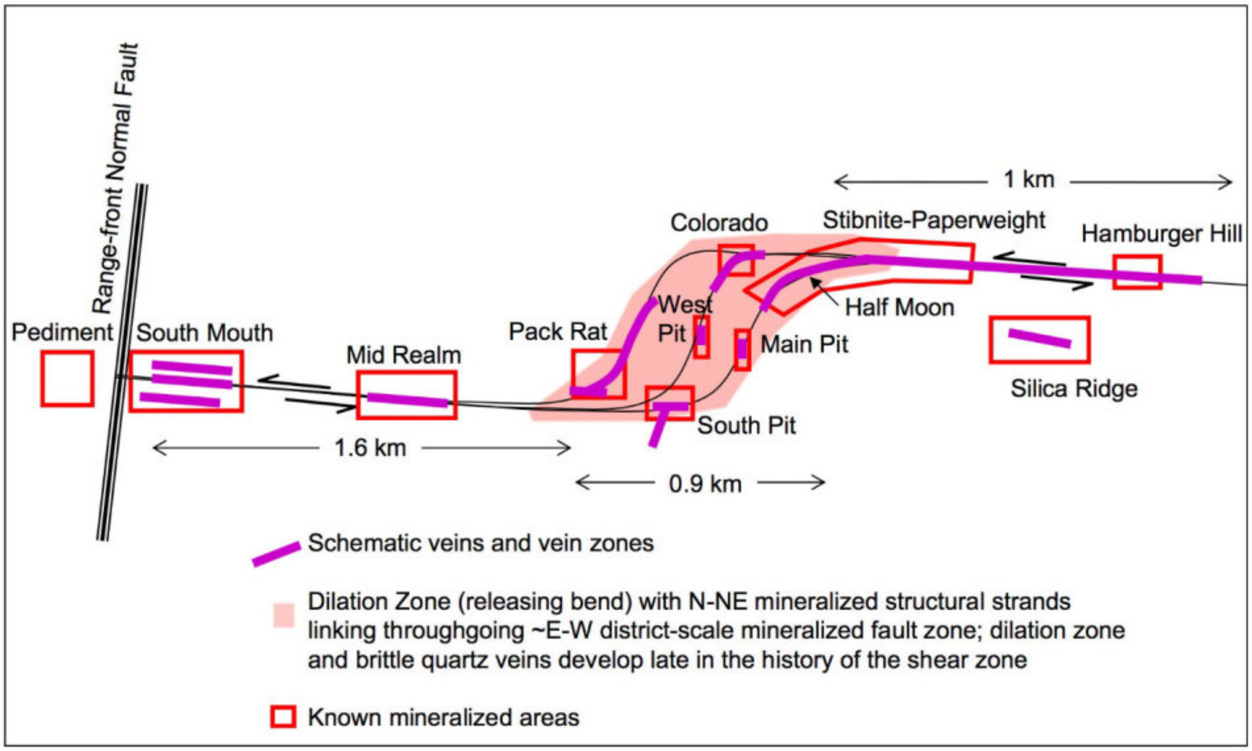


Figure 14-1: Fondaway Canyon Structural Model

(Source: Margolis, 2020)

14.3.2 Oxidation Modeling and Interpretation

Oxidation logging was reviewed from the historical and modern drilling data and evaluated for a potential zone of alteration. In total, 386 drill holes contain oxidation logging information. Oxidation logging only exists at the Main and Silica Ridge – Hamburger Hill zones. The historic data was compared against the modern data to provide a consistent oxidation dataset. Oxidation was logged based on visual strength from 1-3 with blank intervals assumed to be non-oxidized. Based on the comparison of modern and historic data, an oxidation model was created from the intervals logged as 2 to 3 on the visual strength. This created a consistent, near surface zone of oxidation that is mappable across numerous drill holes, shown in Figure 14-2.

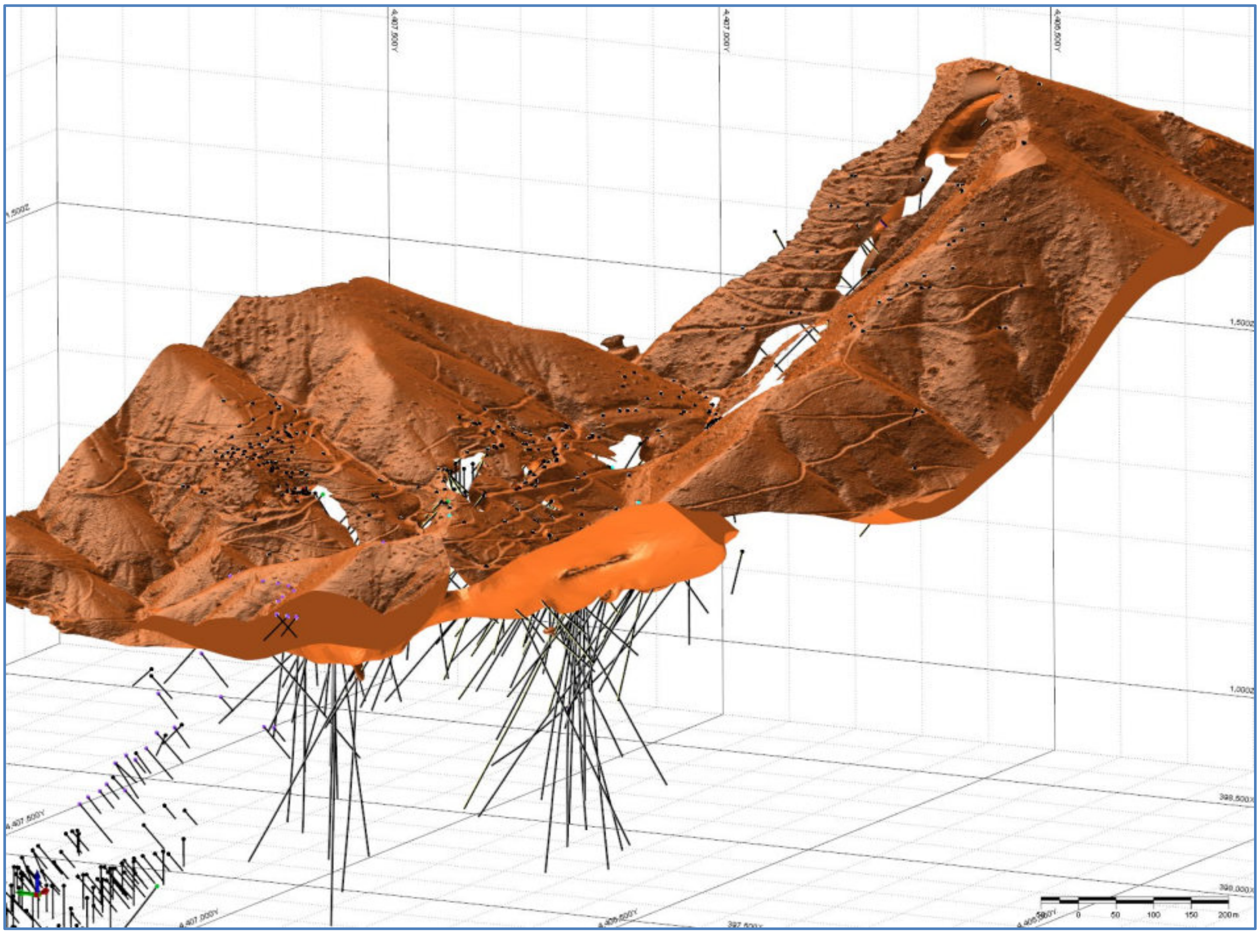


Figure 14-2: Fondaway Canyon Oxidation Model

(Source: APEX Geoscience Ltd. 2024)

14.3.3 Estimation Domains Interpretation Methodology

APEX personnel used an implicit modeling approach for constraining three estimation domains to a gold grade shell while still honoring interpretations of local geological controls on mineralization. The raw drill hole analytical data was composited and classified as either mineralized or waste. Those composites were then used as input for implicit modeling to generate the 3-D estimation domain wireframes that honor the observed geological controls on mineralization.

The mineralization domain construction utilized an approximate lower cutoff of 0.1 ppm Au for the interpretation and joining of mineralization shapes. The estimation domains were evaluated in 3-D and on a section-by-section basis. Control points were inserted to constrain spurious features in the generated wireframes and ensure that the underlying geology was honored. The control points were used in a second pass of the implicit model to construct the final estimation domains.

Plan view of the extents of the estimation domains projected to surface with the drill hole collar locations is shown in Figure 14-3, and Figure 14-4 to Figure 14-6 show cross-sections of each estimation domain in relation to drill hole gold assays.

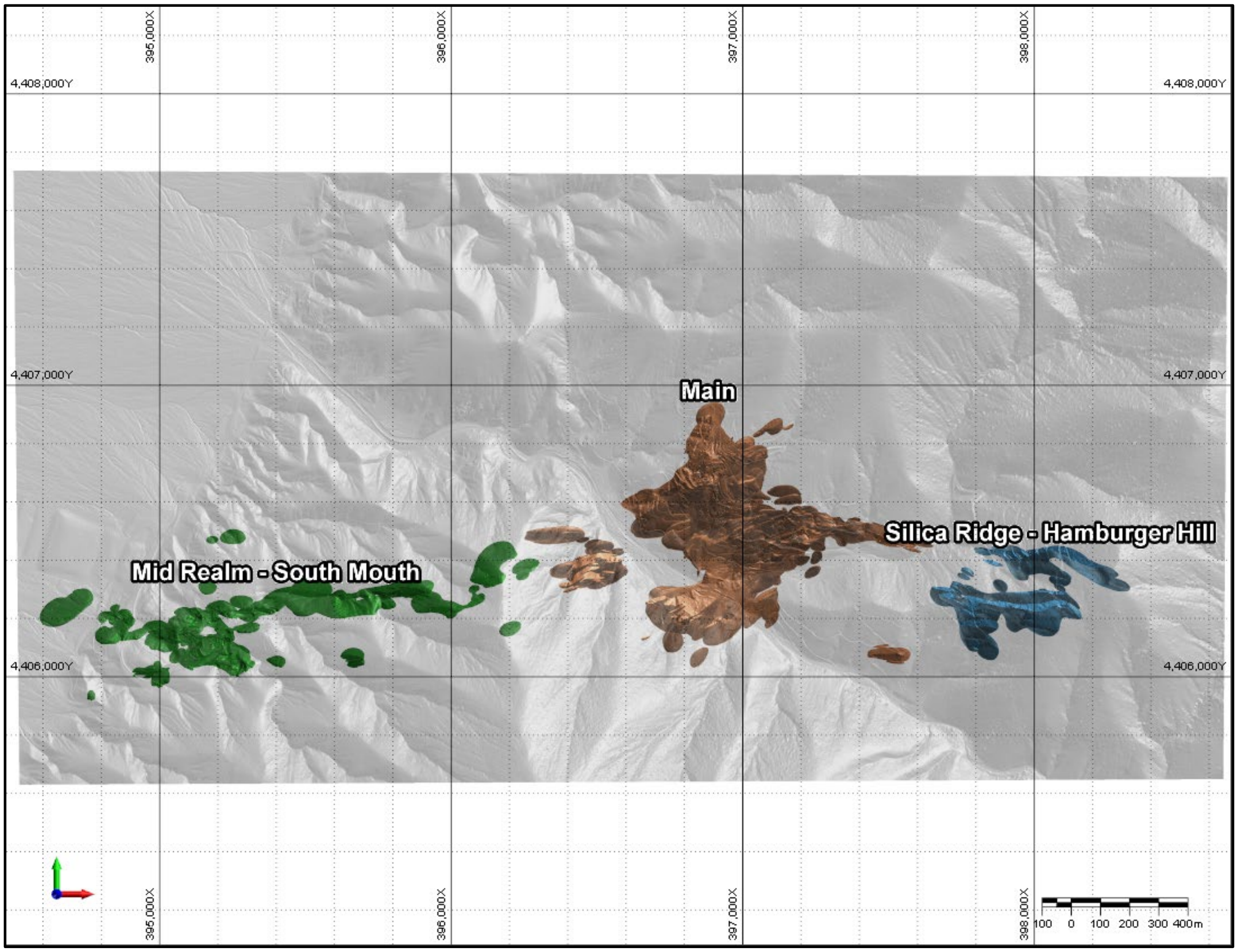
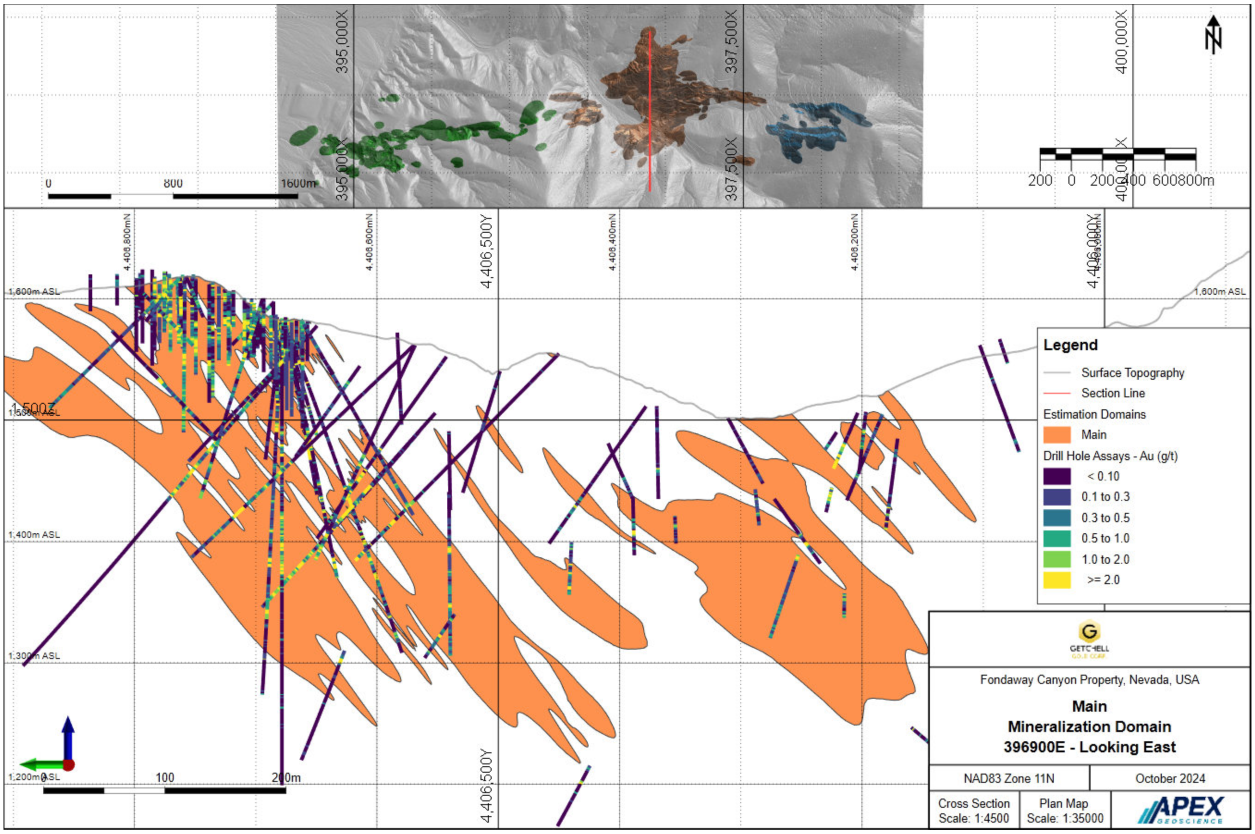


Figure 14-3: Plan View of the Fondaway Canyon Project Grade Estimation Domains

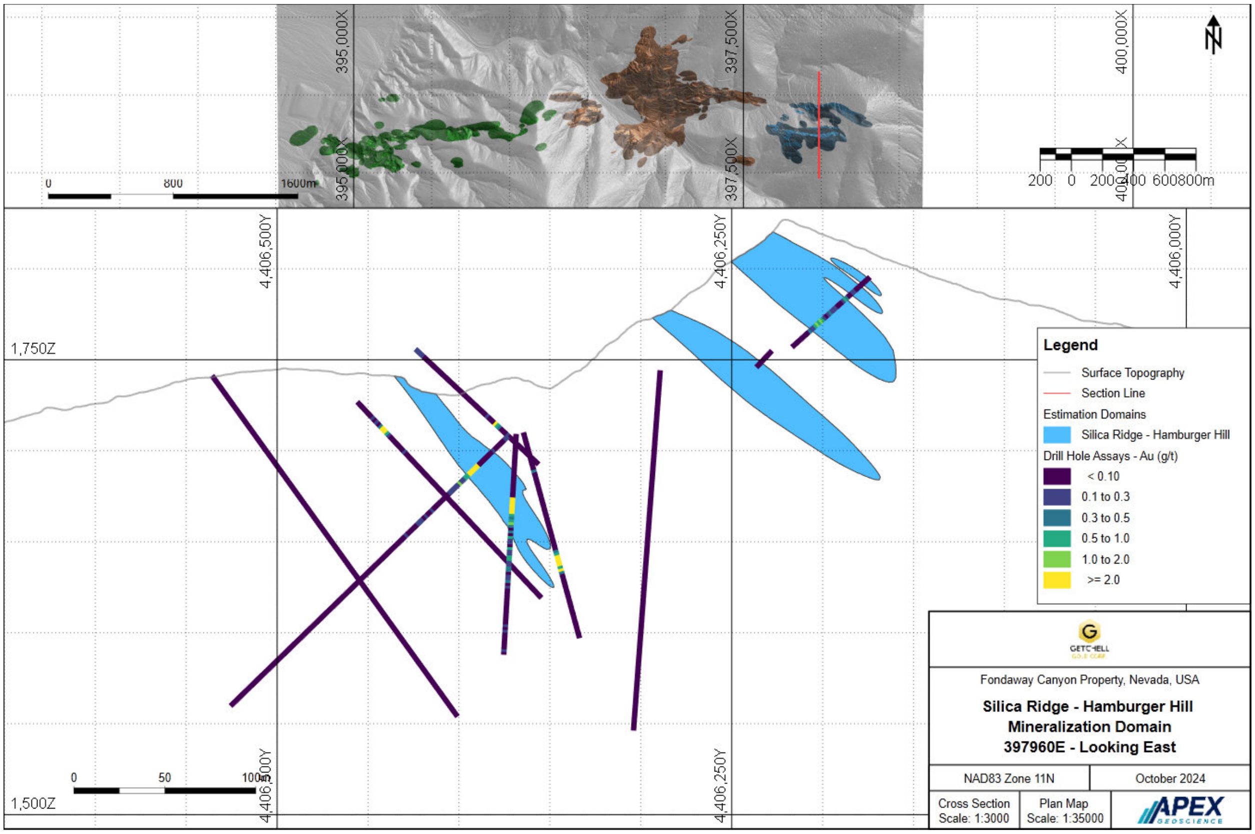
(Source: APEX Geoscience Ltd. 2024)



Note: Section window extents +/- 40 m.

Figure 14-4: Cross-Section of the Main Zone Estimation Domains

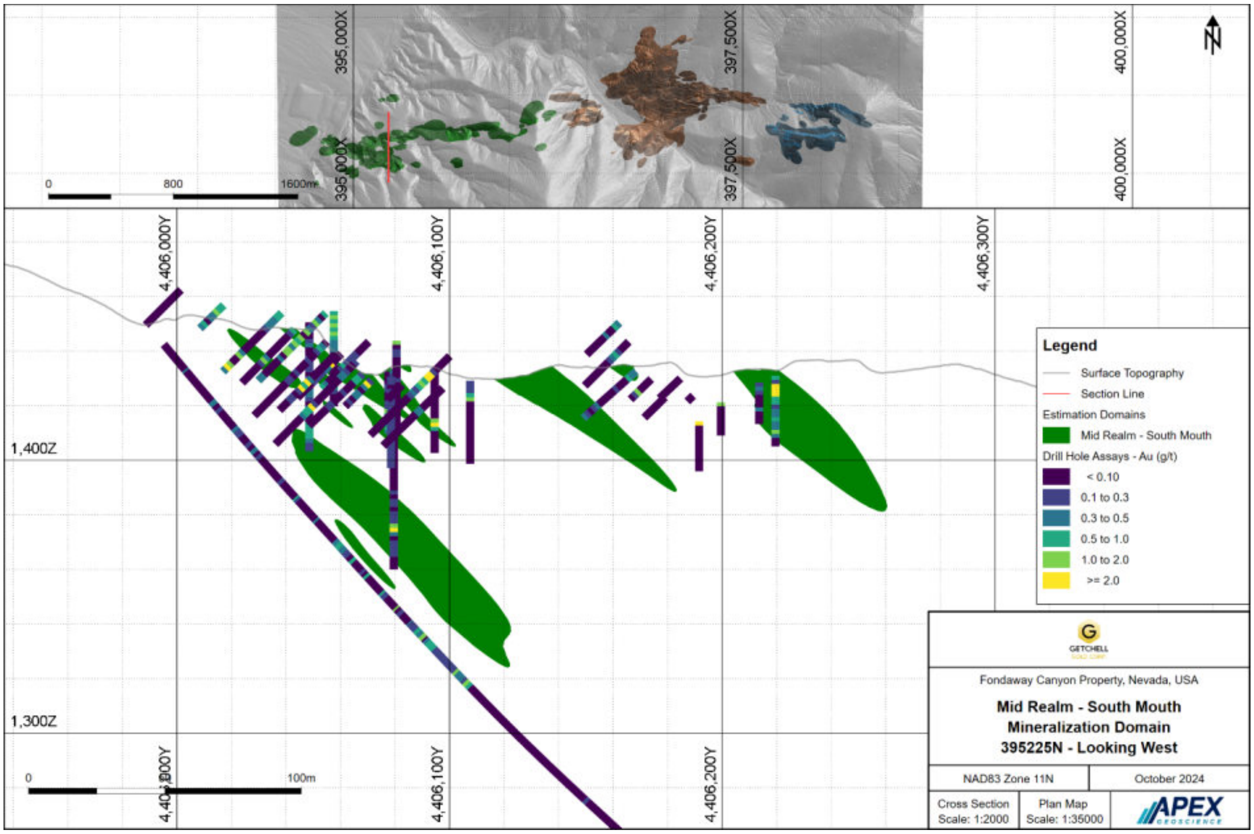
(Source: APEX Geoscience Ltd. 2024)



Note: Section window extents +/- 40 m.

Figure 14-5: Cross-Section of the Silica Ridge – Hamburger Hill Estimation Domain

(Source: APEX Geoscience Ltd. 2024)



Note: Section window extents +/- 40 m.

Figure 14-6: Cross-Section of the Mid Realm – South Mouth Estimation Domain

(Source: APEX Geoscience Ltd. 2024)

14.3.4 Bulk Density

A total of 1,377 modern density samples were collected from nine drill holes completed between 2020 – 2022. These holes intersect both mineralized and non-mineralized material from various lithology types. All the samples came from the Main Zone estimation domain. The density values ranged from 2.4 g/cm^3 to 2.99 g/cm^3. The samples were investigated to determine if any unique density populations exist based on geological, lithological, or mineralization characteristics; however, no unique density populations were found. As an example, Figure 14-7 shows the lack of unique density populations separated by structural groupings as compared to the overall density population. The same analysis was done for other geological characteristics such as domains, lithology, and grade, resulting in a similar outcome. The overall density population mean value of 2.74 g/cm^3 was used for all material.

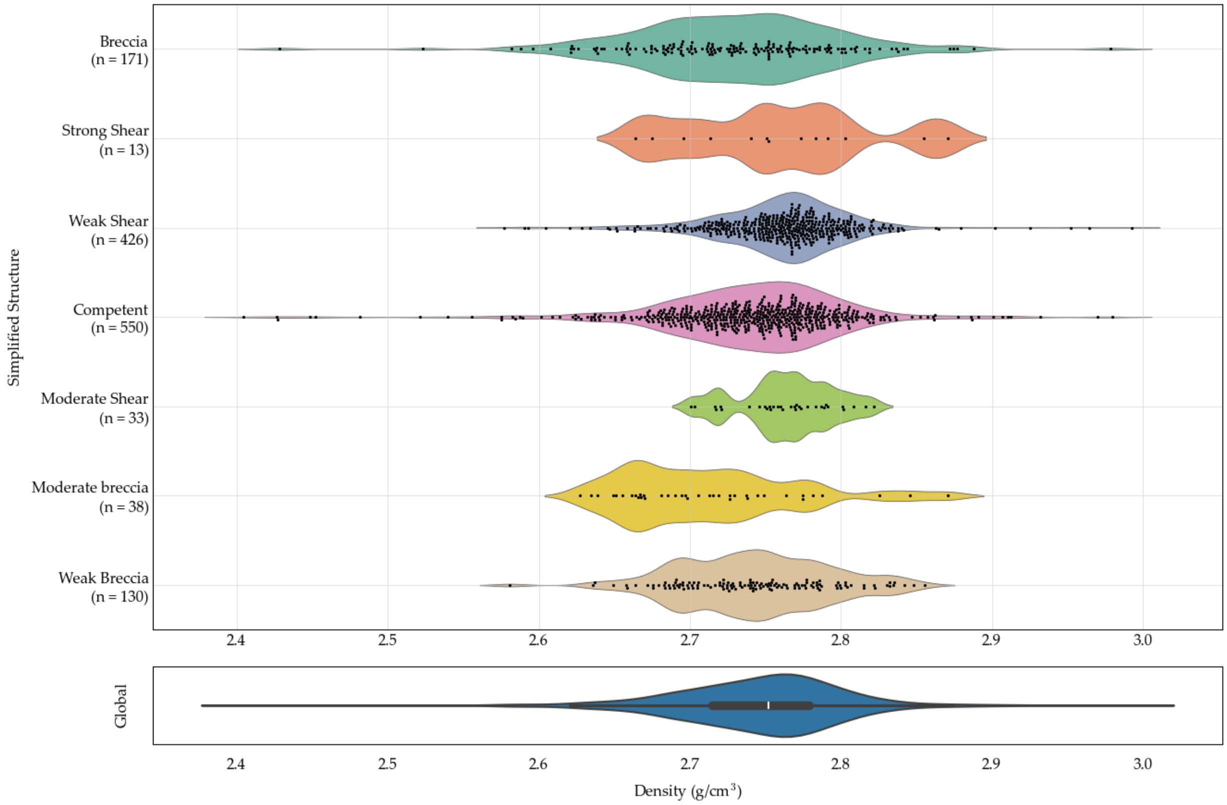


Figure 14-7: Violin Plot of Density Measurements by Structure Type

(Source: APEX Geoscience Ltd. 2024)

14.3.5 Raw Analytical Data

Table 14-3 presents the summary statistics for the raw (not composited) assays from sample intervals within the estimation domains. The assays within each estimation domain exhibit a single coherent statistical population.

Table 14-3: Raw Assay Statistics for the 2024 Fondaway MRE

	Au (ppm)
Count	14260
Mean	1.195
Median	0.34
Standard Deviation	2.595
Variance	6.732
Coefficient of Variation	2.171
Minimum	0.002
25 Percentile	0.17
50 Percentile (Median)	0.34
75 Percentile	1.03
Maximum	59.1

14.3.6 Compositing Methodology

The drill hole sample interval lengths within the estimation domains at Fondaway Canyon vary from 0.09 to 9.15 meters, as illustrated in Figure 14-8. A composite length of 5 feet (1.53 m) was chosen because 97% of the sample intervals are equal to or shorter than this length.

An explicit compositing method was selected, which uses a set composite length for all sample intervals in each contiguous unit, defined as the drill hole segment between domain boundary contacts. The length-weighted compositing process starts from the drill hole collar and ends at the bottom of the hole. However, the final composite intervals along the drill hole cannot cross contacts between estimation domains that demonstrate a hard boundary. Therefore, composites extending downhole are truncated when one of these contacts are intersected. A new composite begins at these contacts and continues to extend downhole until the maximum composite interval length is reached, or another truncating contact is intersected.

The composite length was chosen to ensure that most of the sample intervals were included in the composites, while maintaining a balance between the number of composites and the number of orphans. Of the 12,566 composites, 13 (0.1%) of them fell outside the 50% tolerance of the selected composite length, were considered orphans, and were excluded from the estimation process.

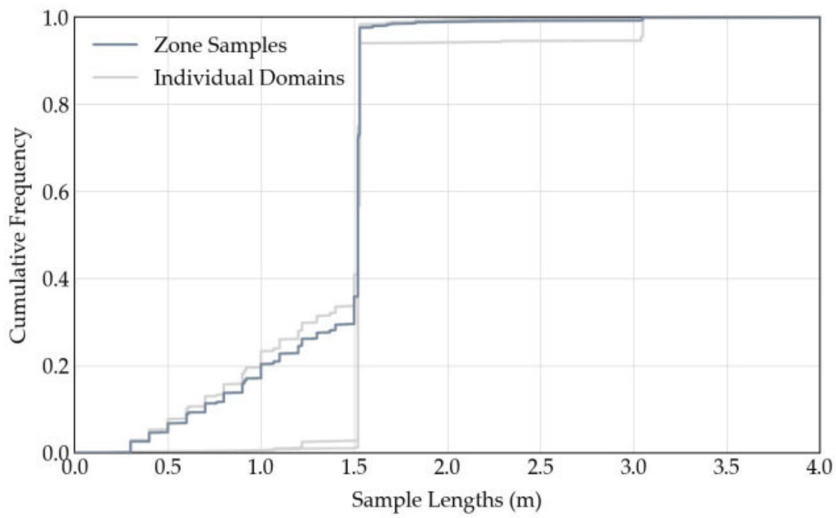


Figure 14-8: Distribution of Raw Interval Lengths Within the Estimation Domains

(Source: APEX Geoscience Ltd. 2024)

14.3.7 Grade Capping

Composites were capped to a specified maximum value to ensure metal grades are not overestimated by including outlier values during estimation. Probability plots illustrating each composite's values are used to identify outlier values that appear greater than expected relative to each estimation domain's commodity distribution. Composites identified as potential outliers on the log-probability plots were evaluated in 3-D to determine whether they are part of a high-grade trend. If outliers are identified as part of a high-grade trend that still requires grade capping, the capping level applied may be less stringent than the level used for controlling isolated high-grade outliers.

Grade capping is completed by assessing the composites within individual domains. Table 14-4 indicates the grade capping levels determined using the log-probability plots. Visual inspection of the potential outliers revealed they have no spatial continuity with each other. Therefore, the grade capping levels detailed in Table 14-4 was applied to all composites used to calculate the 2024 Fondaway Canyon Project MRE.

Table 14-4: Grade Capping Levels

Variable	Capping Level Unit	Domain	Capping Level	No. of Capped Composites	No. of Composites
Au	ppm	Main	32	5	10,632
Au	ppm	Mid Realm – South Mouth	NA	0	1,267
Au	ppm	Silica Ridge – Hamburger Hill	32	8	654

14.3.8 Declustering

Data collection often focuses on high-value areas, leaving sparse areas underrepresented in the raw composite statistics and distributions. Spatially representative (declustered) statistics and distributions are necessary to achieve accurate validation. Declustering techniques assign a weight to each composite within an estimation domain, giving more weight to sparsely sampled areas and less to densely sampled regions. Declustering cell sizes of 23 m, 20 m, and 32.0 m were used for the Main, Mid Realm – South Mouth, and Silica Ridge – Hamburger Hill areas respectively.

14.3.9 Final Composite Statistics

Summary statistics for the declustered and capped composites contained within the interpreted grade estimation domains are presented in Table 14-5. The composites within each grade estimation domain generally exhibit coherent individual statistical populations.

Table 14-5: Final Composite Statistics for the 2024 Fondaway Canyon Project MRE

	Au (ppm)
Count	12,553
Mean	1.11
Standard Deviation	2.25
Coefficient of Variation	2.02
Minimum	0.00
25 Percentile	0.17
50 Percentile (Median)	0.342
75 Percentile	1.00
Maximum	32.00

Note: Statistics consider declustering weights and capping.

14.4 Variography

Experimental semi-variograms are calculated along the major, minor, and vertical principal directions of continuity, defined by three Euler angles. These angles describe the orientation of anisotropy through a series of left-hand rule rotations that are:

- Angle 1: A rotation about the Z-axis (azimuth), where positive angles represent clockwise rotation, and negative angles represent counterclockwise rotation.
- Angle 2: A rotation about the X-axis (dip), where positive angles represent counterclockwise and negative angles represent clockwise rotation.
- Angle 3: A rotation about the Y-axis (tilt), where positive angles represent clockwise rotation, and negative angles represent counterclockwise rotation.

APEX calculated standardized correlograms for each estimation domain using composite data. In domains with enough composites for experimental variogram calculations, the primary geological factors influencing mineralization guided the main continuity directions, which formed the basis for the variogram calculations.

Figure 14-9 to Figure 14-11 illustrate the modeled gold variograms for each estimation domain. Table 14-6 outlines the variogram parameters used for kriging.

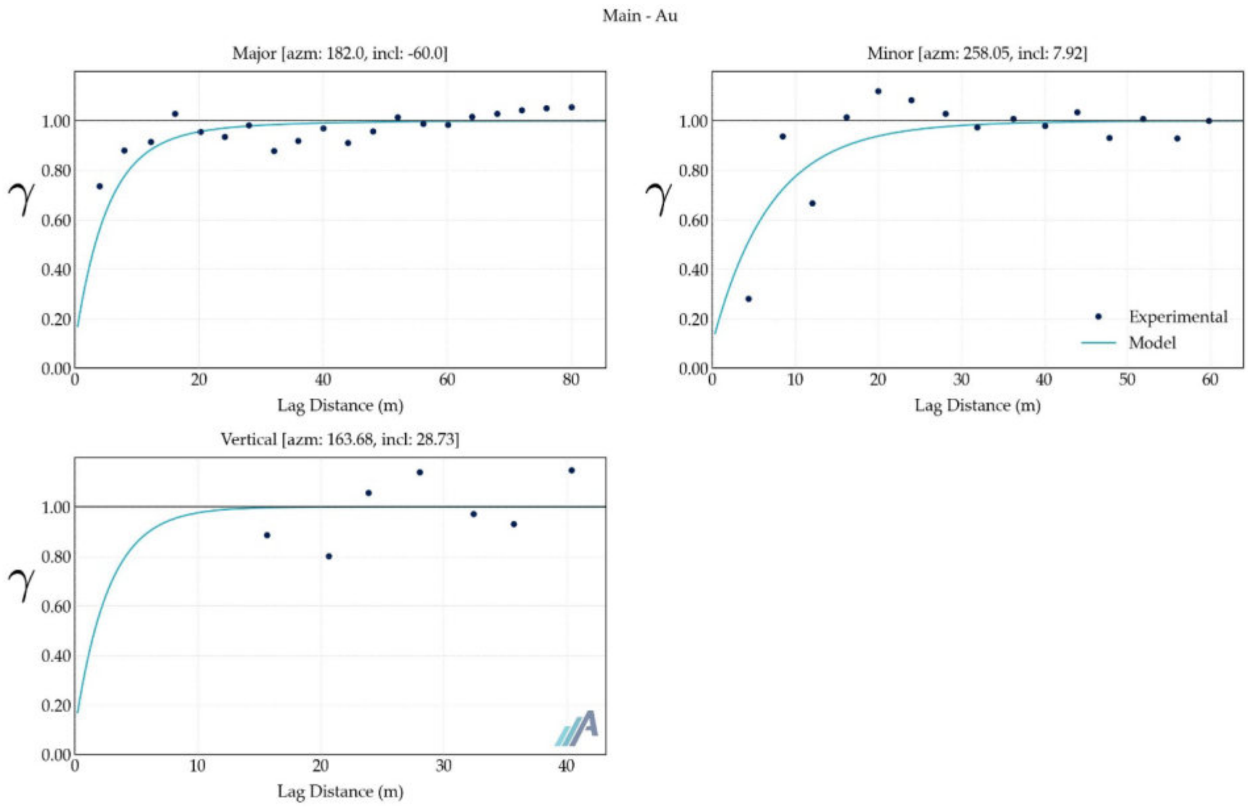


Figure 14-9: Modeled Gold Variogram for the Main Zone Domain

(Source: APEX Geoscience Ltd. 2024)

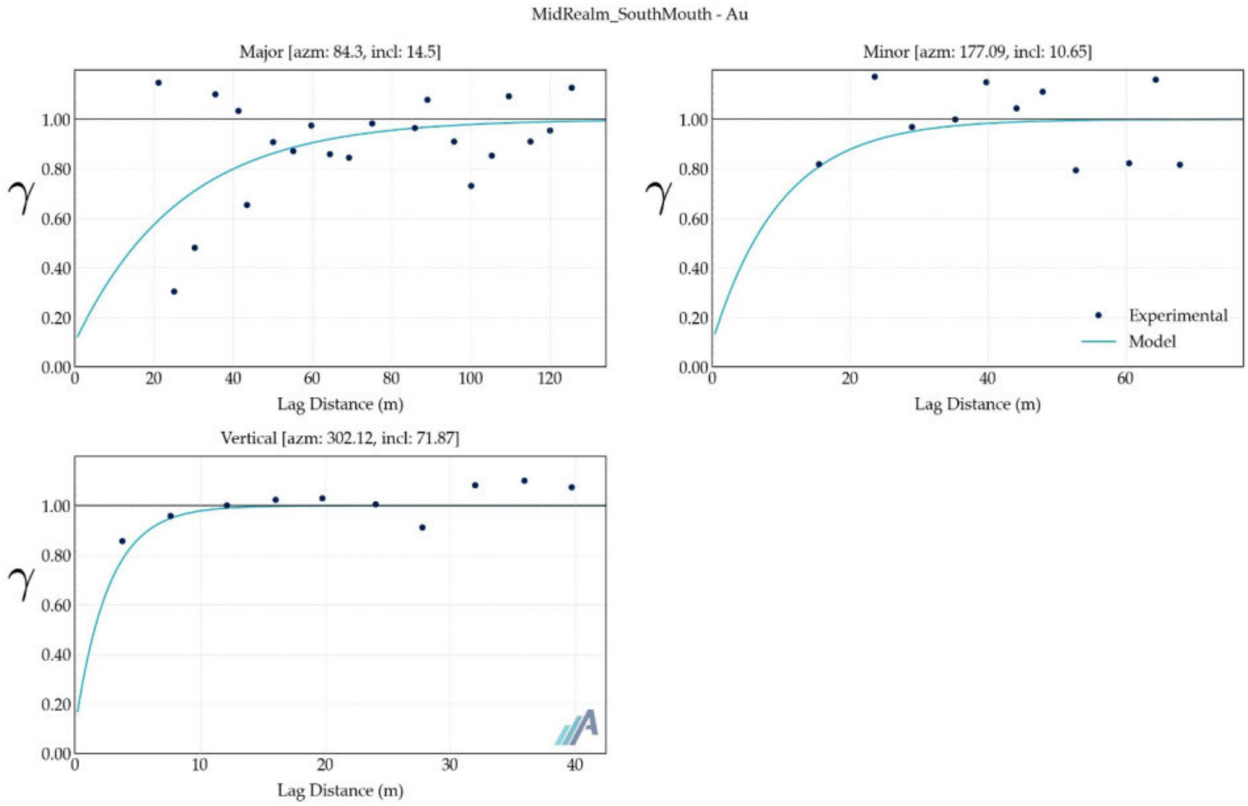


Figure 14-10: Modeled Gold Variogram for the Mid Realm – South Mouth Domain

(Source: APEX Geoscience Ltd. 2024)

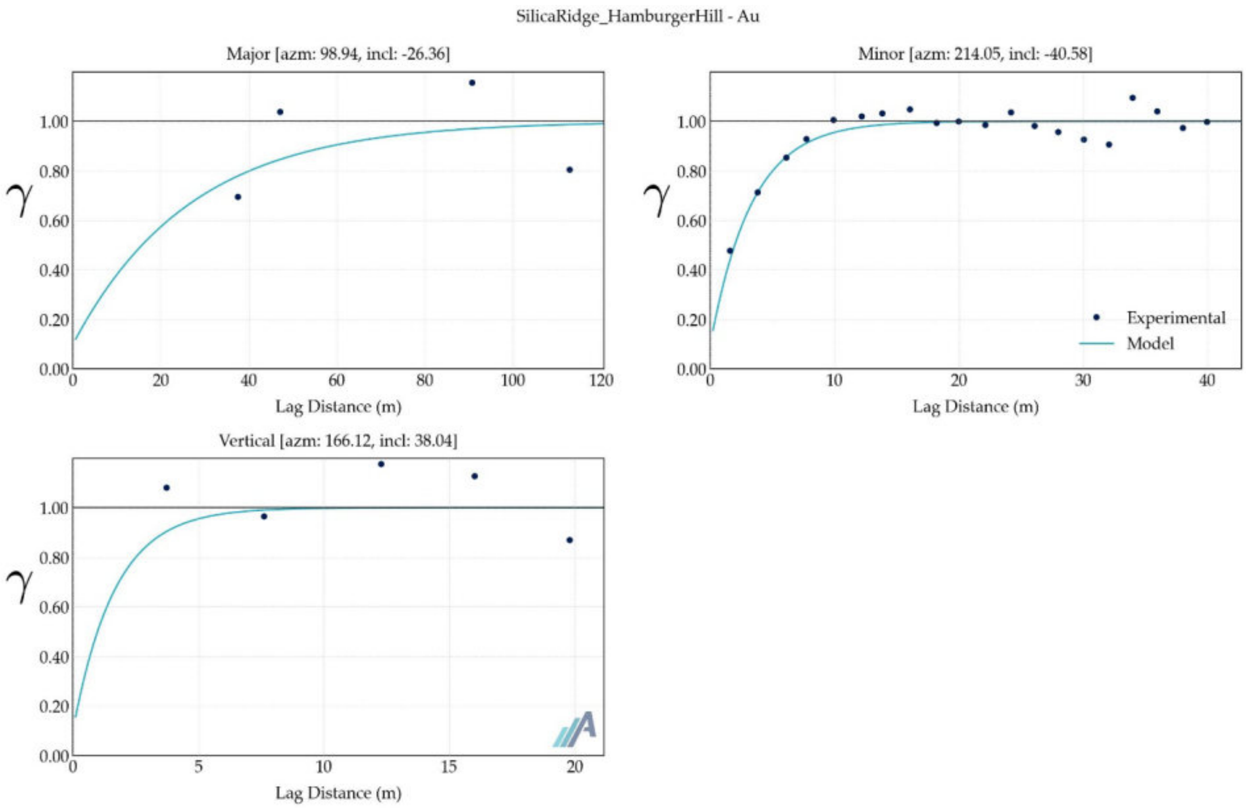


Figure 14-11: Modeled Gold Variogram for the Silica Ridge – Hamburger Hill Domain

(Source: APEX Geoscience Ltd. 2024)

Table 14-6: Standardized Variogram Parameters

Domain	Rotation Angles			C0	Variogram Structures					
	1	2	3		Structure	Type	CC	Ranges (ft)		
								Major	Minor	Vertical
Main	182	-60	-16	0.1	1	Exponential	0.8	15	20	8
					2	Spherical	0.1	50	40	10
Mid Realm - South Mouth	84.3	14.5	-11	0.1	1	Exponential	0.9	80	30	8
Silica Ridge - Hamburger Hill	99	-26	47	0.1	1	Exponential	0.9	80	10	5

Abbreviations: C0 – nugget effect, CC – covariance contributions.
Note: the sill and covariance contributions are standardized to 1.

14.5 Block Model

14.5.1 Block Model Parameters

The block model used to calculate the 2024 Fondaway Canyon Project MRE fully encapsulates the Main, Mid Realm – South Mouth, and Silica Ridge – Hamburger Hill domains described in Section 14.3. No blocks are estimated outside of the estimation domains. The block model extents are described in Table 14-7.

A block factor is calculated to represent the percentage of each block's volume within each estimation domain. This factor is used to:

- Identify the primary domain by volume for each block.
- Determine the percentage of mineralized material and waste material within each block.

Table 14-7: 2024 Fondaway MRE Block Model Definition

Axes	Origin*	No. of Blocks	Block Size (m)	Rotation**
X	394,325	1,525	3	0
Y	4,405,650	620	3	0
Z	1,000	310	3	0

** In RMSP, a block model's origin represents the block's centroid coordinates with the minimum U, V, and Z. After rotation, the U and V axes correspond to the X and Y axes, respectively.*

*** Rotations are applied sequentially about the Z, Y, and X axes, following the convention outlined in Section 14-5.*

14.5.2 Volumetric Checks

Wireframe and block model volumes are compared to ensure tonnages are not significantly over- or underestimated. Each block's volume is scaled using its calculated block factor to determine the total block model volume. The maximum percent difference calculated is 0.92 %. This is considered reasonable and within tolerance.

14.6 Grade Estimation Methodology

14.6.1 Grade Estimation of Mineralized Material

Ordinary Kriging (OK) is used to estimate metal grades for the 2024 Fondaway Canyon MRE block model. Only blocks that intersect the resource estimation domains are estimated.

Estimation uses locally varying anisotropy (LVA), which employs different rotation angles to set the variogram model's principal directions and search ellipsoid for each block. Trend surface wireframes assign these angles to blocks within the estimation domain, enabling structural complexities to be captured in the estimated block model.

During grade estimation for each domain, the nugget effect and covariance contributions of the standardized variogram model are scaled to match the variance of the composites within that domain. The ranges used for each mineralized zone are unchanged from the standardized variogram model.

Contact analysis of the boundaries between adjacent estimation domains shows that the metal profile at the boundary is hard or semi-hard, where the profiles trend toward each other over a very short distance. Consequently only data from within each domain can be used for grade estimation within that specific domain.

A multiple-pass estimation method is used to control Kriging's smoothing effect and limit the influence of high-grade samples, ensuring accurate grade and tonnage estimates at the block scale. Each pass considers up to 30 composites, with a minimum of one required for estimation. Table 14-8 details the restricted search parameters and limits the number of composites from each drill hole. While these rules may introduce local bias, they improve the global accuracy of grade and tonnage estimates above the reporting cutoff.

Table 14-8: 2024 Fondaway MRE Interpolation Parameters

Domain	Variable	Pass	Max No. Comps	Max No. Comps per Drill hole	Min. No. Comps	Search Ranges		
						Major	Minor	Vertical
Main	Au	1	30	6	1	30	30	5
		2	30	4	1	50	40	5
		3	30	4	1	100	80	20
Mid Realm - South Mouth	Au	1	30	4	3	20	20	5
		2	30	4	1	80	30	8
		3	30	4	1	160	60	20
Silica Ridge - Hamburger Hill	Au	1	30	8	2	80	20	5
		2	30	4	1	160	40	15

14.6.2 Grade Estimation of Waste Material

The open pit optimization for evaluating reasonable prospects for eventual economic extraction relies on a whole block grade. Therefore, blocks that contain more than or equal to 1% waste by volume are diluted by estimating a waste grade that is then volume-weight averaged with the estimated grades.

It is desired that the behavior at the estimation domain to waste boundary is accurately reproduced. The behavior of mineralization at the mineralized/waste contact was evaluated and used to determine a window to flag composites used to condition a waste estimate for blocks containing waste material. The profile along the mineralized/waste contact behaves statistically hard, where the grades of the composite centroids flagged within an estimation domain transitions from mineralized to waste with no transition. Only composites outside the estimation domains were used to estimate a waste grade for the 2024 Fondaway Canyon MRE.

14.7 Model Validation

14.7.1 Statistical Validation

Statistical checks were completed to validate that the block model accurately reflects the informing drill hole data and ensuring grade trends were maintained. Swath plots confirmed local directional trends, while volume-variance analysis verified global metal quantity and grades estimated at the reporting cutoff.

14.7.1.1 Direction Trend Analysis Validation

Swath plots verify that the estimated block model grades honor local directional trends and identifies potential areas of over- or under-estimating grade. The swath plots are generated by calculating the average metal grades of composites and the OK estimated blocks over different swaths. Examples of the swath plots used to validate the Mineral Resource Estimate are illustrated in Figure 14-12 to Figure 14-14.

Overall, the block model compares well with the composites. Some local over- and under-estimation has been observed. Due to the limited amount of conditioning data available for grade estimation in those areas, this result is expected.

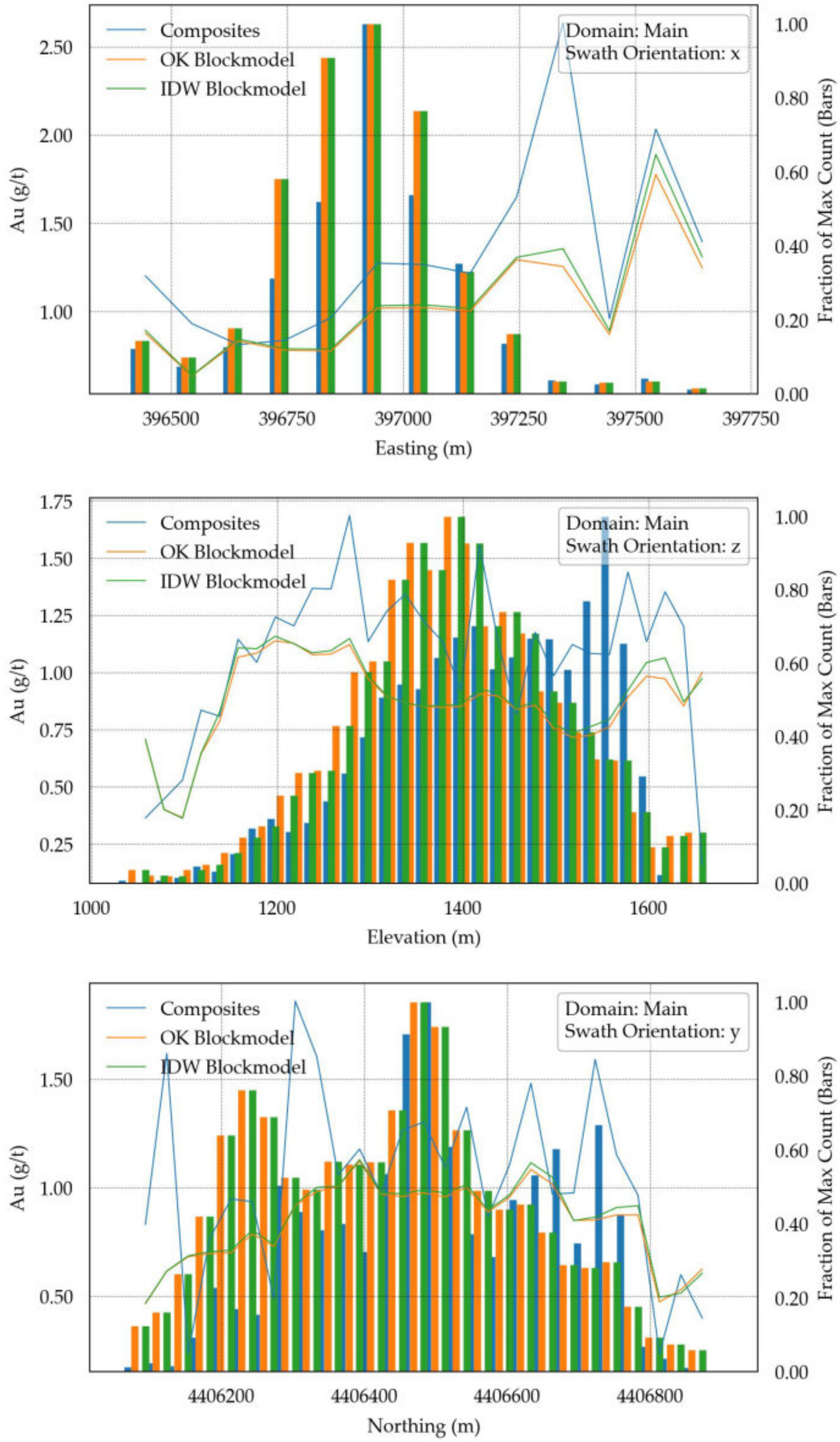


Figure 14-12: Swath Plots of the Estimated Gold Grades for the Main Domain

(Source: APEX Geoscience Ltd. 2024)

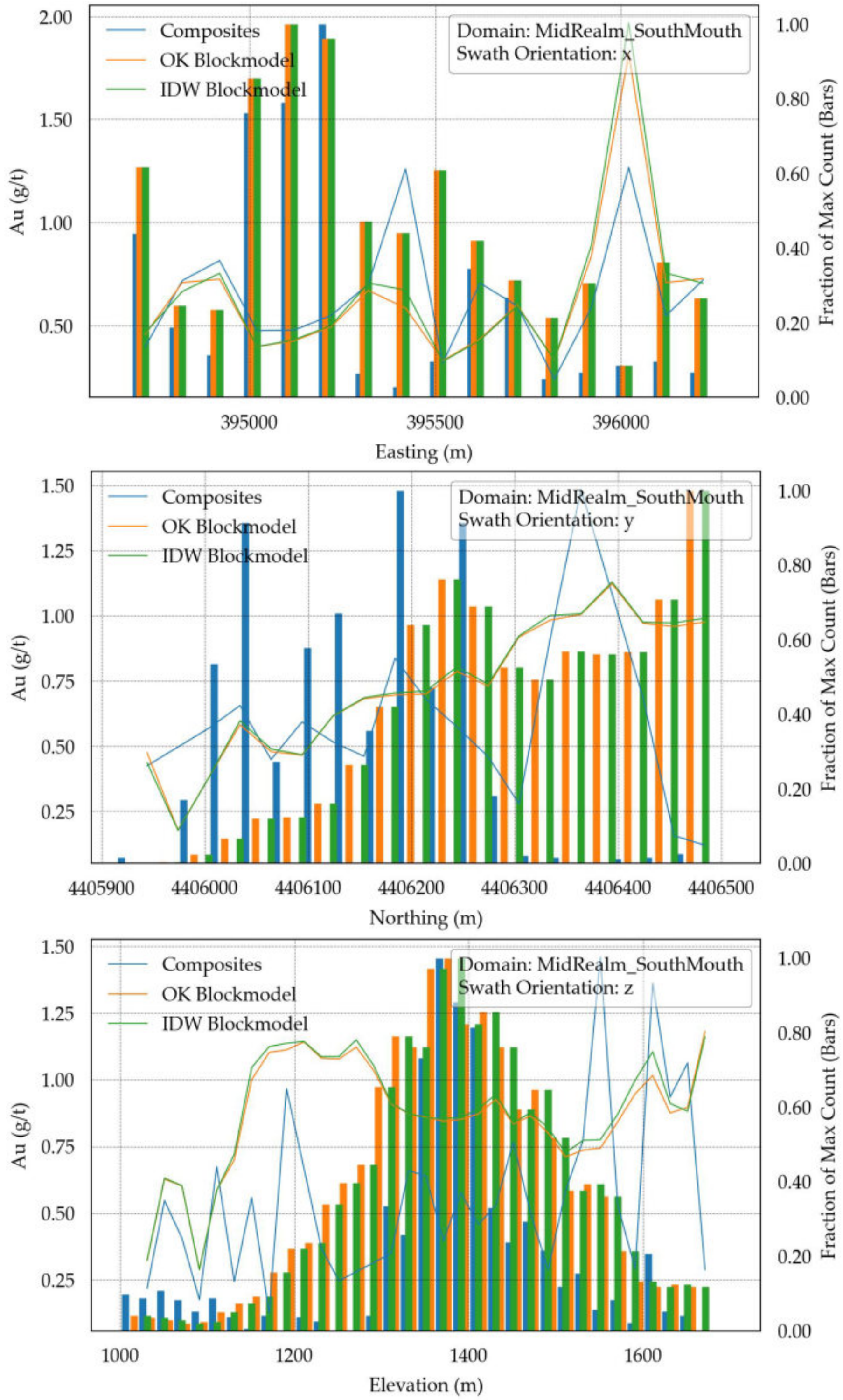


Figure 14-13: Swath Plots of Estimated Gold Grades for the Mid Realm – South Mouth Domain

(Source: APEX Geoscience Ltd. 2024)

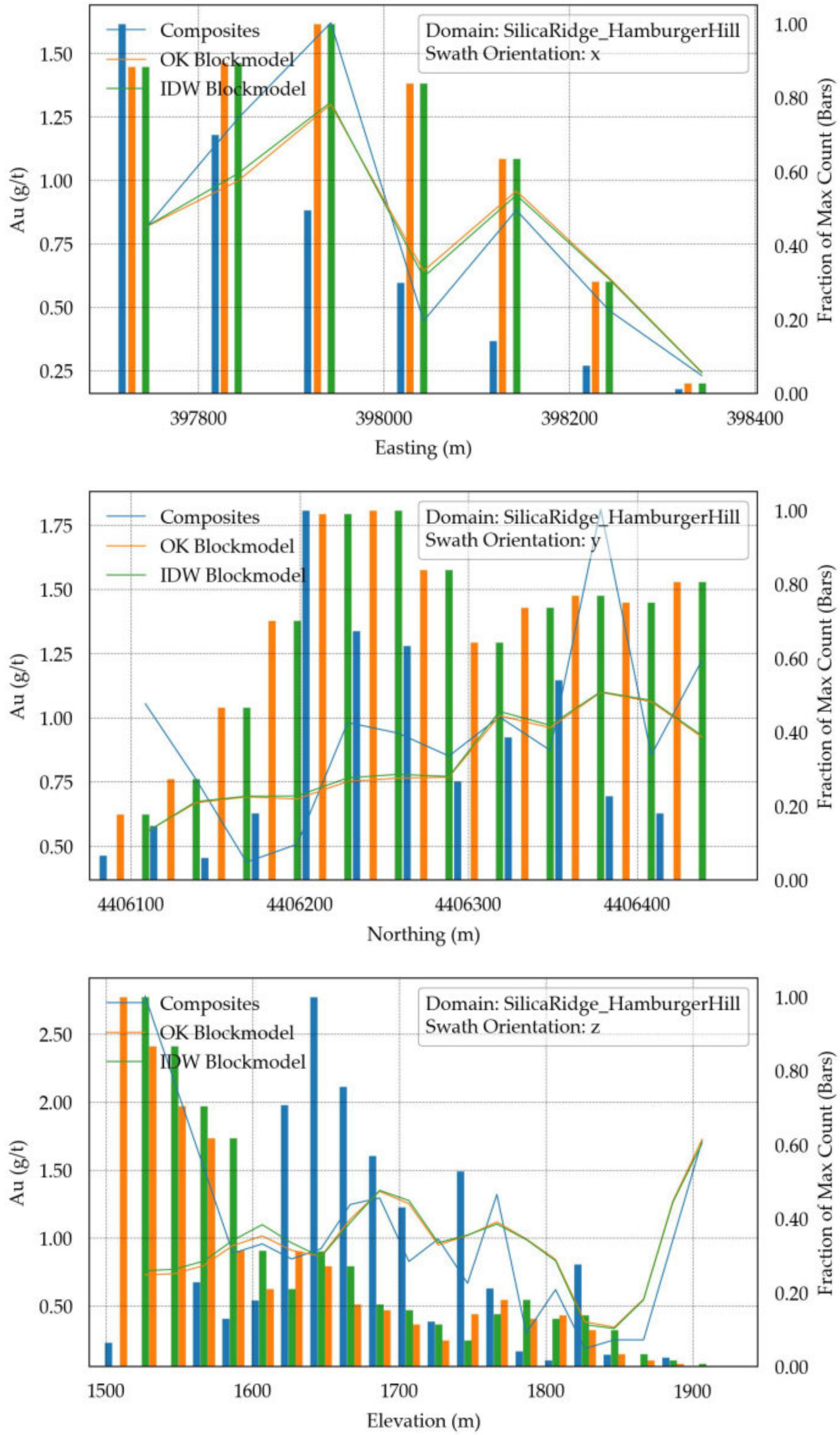


Figure 14-14: Swath Plots of Estimated Gold Grades for the Silica Ridge – Hamburger Hill Domain

(Source: APEX Geoscience Ltd. 2024)

14.7.1.2 Volume-Variance Analysis Validation

Smoothing is an intrinsic property of Kriging, and it is critical to validate that the estimated model, when restricted to a specific cutoff, predicts appropriate estimates of both grade and tonnes. Considering the selective mining unit (SMU) and the information effect, target distributions are calculated using a discrete Gaussian model, with composites and variograms as parameters. The distribution of the scaled composites illustrates the anticipated tonnes and average grades above various cutoff grades at the SMU scale. As described in Section 14-6, the searches used during OK are restricted to mitigate Kriging's smoothing effects and ensure the estimated model matches the target distribution. A comparison between the expected SMU distribution of grade and tonnes and the estimated model (Figure 14-15) confirms that the appropriate level of smoothing is achieved at the reporting cutoff. The goal of the volume-variance analysis is to achieve a global rather than local validation. There appears to be some smoothing within the domains, but it is likely due to the size of the individual domains. Improved estimation domains that more accurately align with the geological features could improve the volume-variance analysis. Further modifications to the search strategy to achieve a closer match would introduce excessive bias.

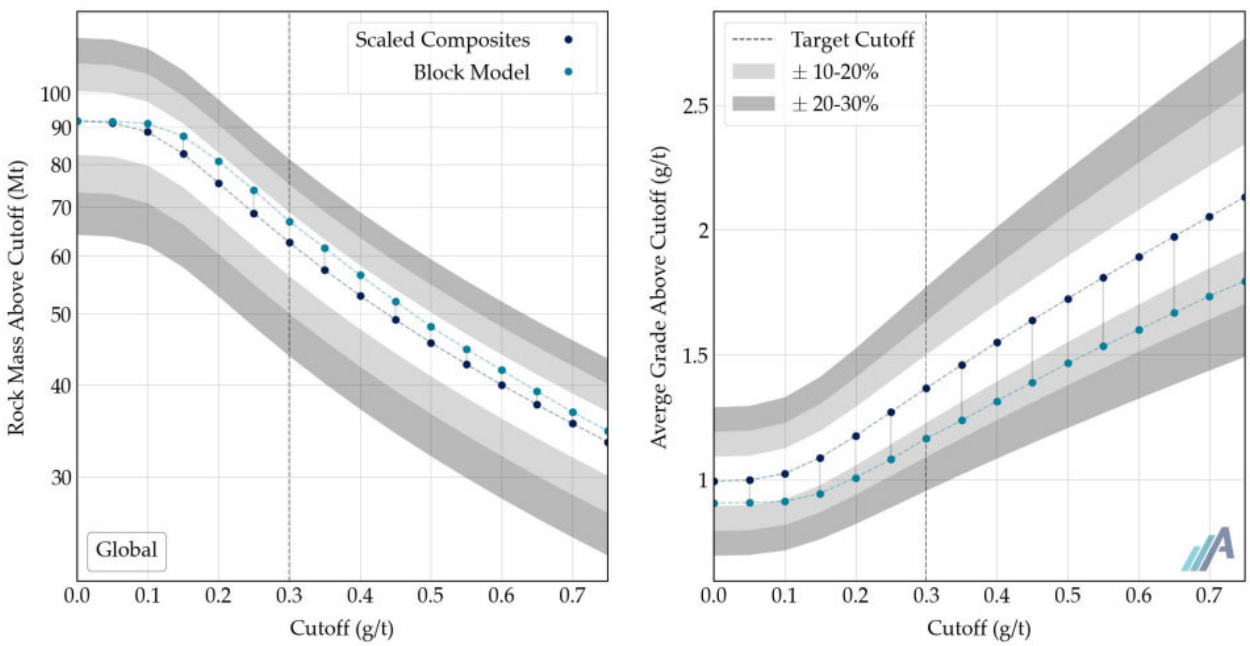


Figure 14-15: Comparison of Target Gold Distribution and Estimated Distribution

(Source: APEX Geoscience Ltd. 2024)

14.7.2 Visual Validation

APEX personnel visually reviewed the estimated block model grades in cross-sectional views, comparing the estimated block model grades to the input composited drill hole assays and the modelled mineralization trends. The block model compares very well to the informing composite data. Local high- and low-grade zones within the Mineral Resource areas are reproduced as desired, and the locally varying anisotropy adequately maintains variable mineralization orientations. Figure 14-16 to Figure 14-18 illustrates the grade estimation blocks used for the MRE.

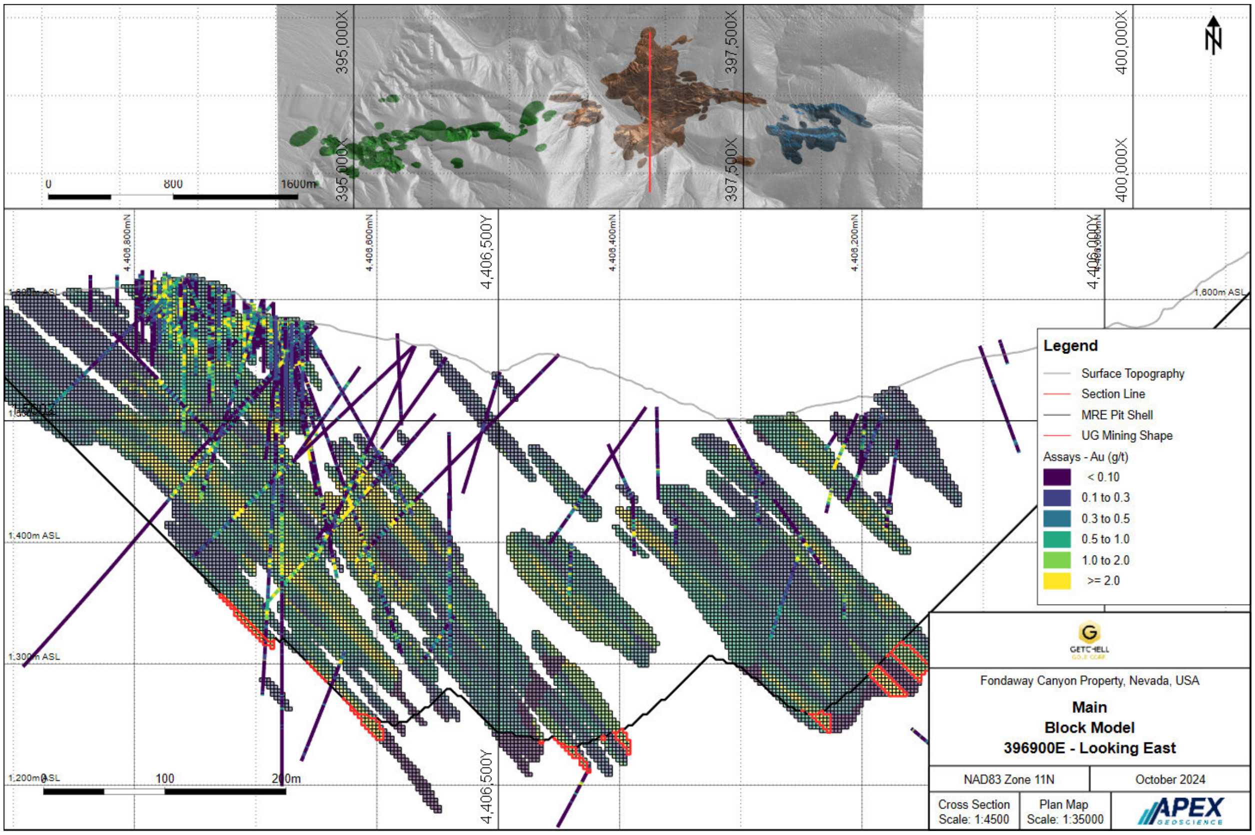


Figure 14-16: Cross-Section of the 2024 Fondaway MRE Block Model of the Main Domain

(Source: APEX Geoscience Ltd. 2024)

Note: Section window extents +/- 40 m and looking east along 396950E illustrating estimated grades.

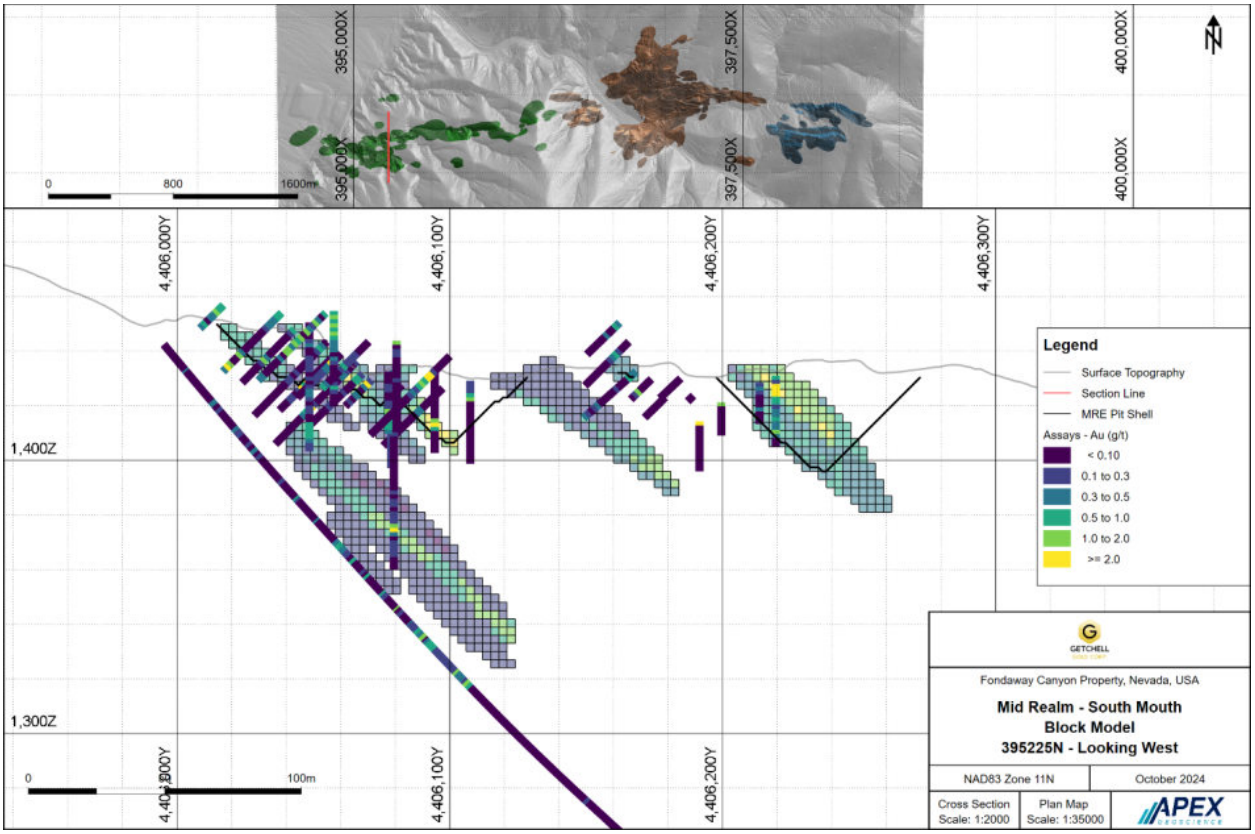


Figure 14-17: Oblique Section of the 2024 Fondaway MRE Block Model of the Mid Realm – South Mouth Domain

(Source: APEX Geoscience Ltd. 2024)

Note: Section window extents +/- 40 m and looking west along 395225N illustrating estimated grades.

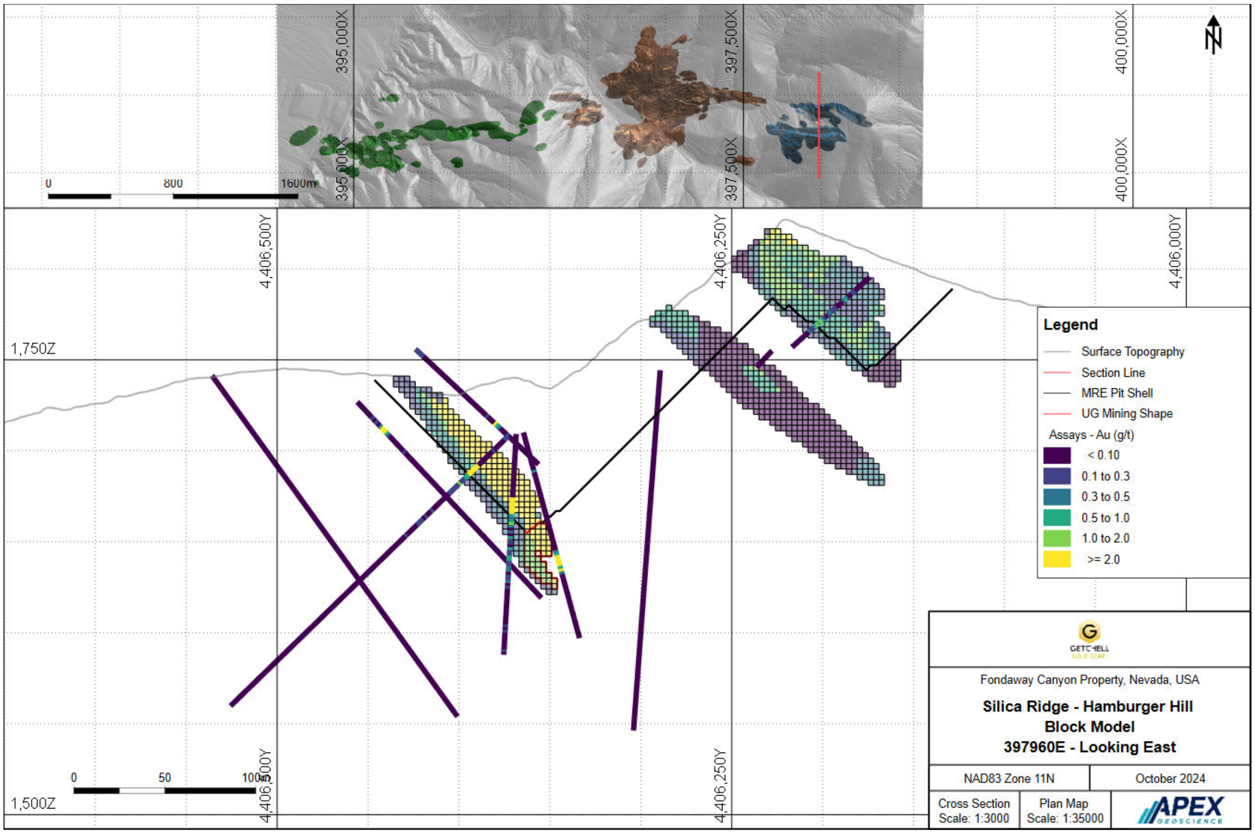


Figure 14-18: Cross-Section of the 2024 Fondaway MRE Block Model of the Silica Ridge – Hamburger Hill Domain

(Source: APEX Geoscience Ltd. 2024)

Note: Section window extents +/- 40 m and looking north along 398050E illustrating estimated grades.

14.8 Mineral Resource Classification

14.8.1 Classification Definitions

The 2024 Fondaway MRE discussed in this Technical Report is classified following guidelines established by the CIM "Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines" dated November 29, 2019, and CIM "Definition Standards for Mineral Resources & Mineral Reserves" dated May 19, 2014.

A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity.

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.

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.

14.8.2 Classification Methodology

According to the CIM definition standards, the 2024 Fondaway Canyon MRE is classified as Indicated and Inferred. The classification of the Indicated and Inferred Mineral Resources is based on geological confidence, data quality and grade continuity of the data. The most relevant factors used in the classification process are the following:

- Density of conditioning data.
- Level of confidence in drilling results and collar locations.
- Level of confidence in the geological interpretation.
- Continuity of mineralization.
- Level of confidence in the assigned densities.

Mineral Resource classification is determined using a multiple-pass strategy that consists of a sequence of runs that flag each block with the run number of the block that first meets a set of search restrictions. With each subsequent pass, the search restrictions decrease, representing a decrease in confidence and classification from the previous run. For each run, a search ellipsoid is centered on each block and oriented in the same way described in Section 14-7. This process is completed separately from grade estimation.

Table 14-10 details the range of the search ellipsoids and the number of composites that must be found within the ellipse for a block to be flagged with that run number. The runs are executed in sequence from run 1 to run 2. Classification is determined by relating the run number to each block that is flagged as Indicated (run 1) or Inferred (run 2). Classification is capped at Inferred for the Mid Realm – South Mouth and Silica Ridge – Hamburger Hill areas due to a limited understanding of the mineralization controls and orientation. Figure 14-19 to Figure 14-21 illustrate the classification model used for the MRE.

Measured Mineral Resources are currently not defined. For future resource assessments, ranking historical drill holes based on confidence in their collar and downhole surveys is recommended. Only drill holes with high confidence should be considered for measured resources in conjunction with modern drilling data. Additionally, a more robust geological model would provide more confidence to the Project to more accurately construct the estimation domains.

Table 14-9: Parameters for Search Restrictions in the Multiple-Pass Classification Strategy

Classification	Run	Minimum No. of Drill holes	Ranges (m)		
			Major	Minor	Vertical
Indicated	1	3	55	40	10
Inferred	2	2	120	120	20

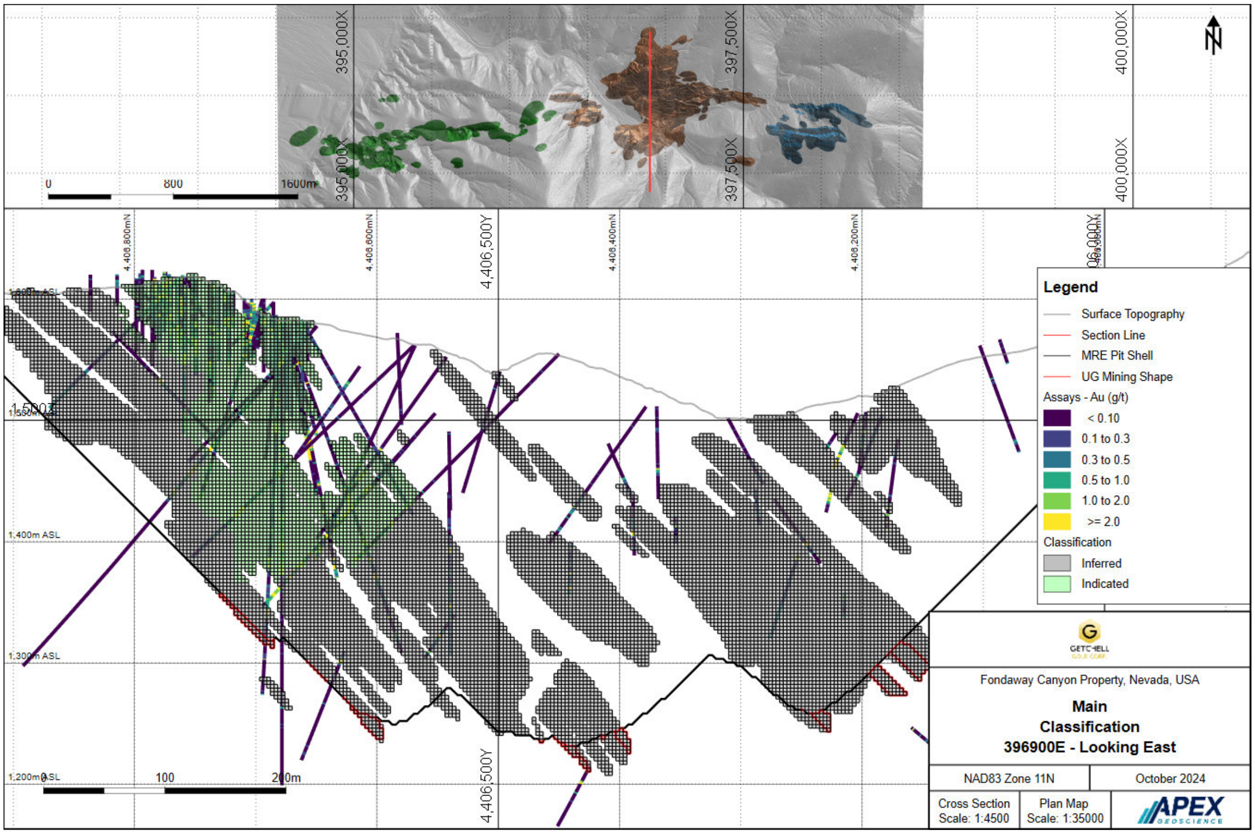


Figure 14-19: Cross-Section of the Classification Block Model for the Main Zone Estimation Domain

(Source: APEX Geoscience Ltd. 2024)

Note: Section window extents +/- 40 m and is looking north along 396925E.

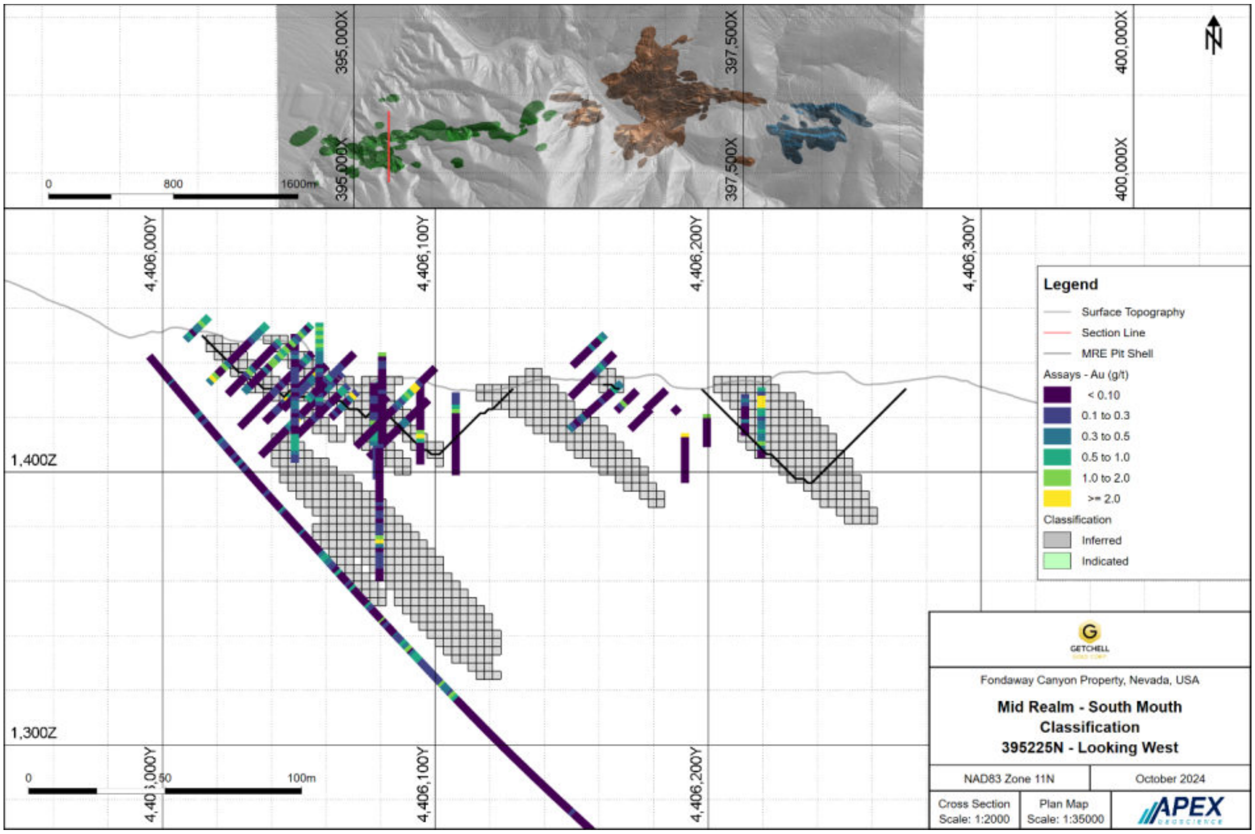


Figure 14-20: Cross-Section of the Classification Block Model for the Mid Realm – South Mouth Estimation Domain

(Source: APEX Geoscience Ltd. 2024)

Note: Section window extents +/- 40 m and is looking west along 395225N.

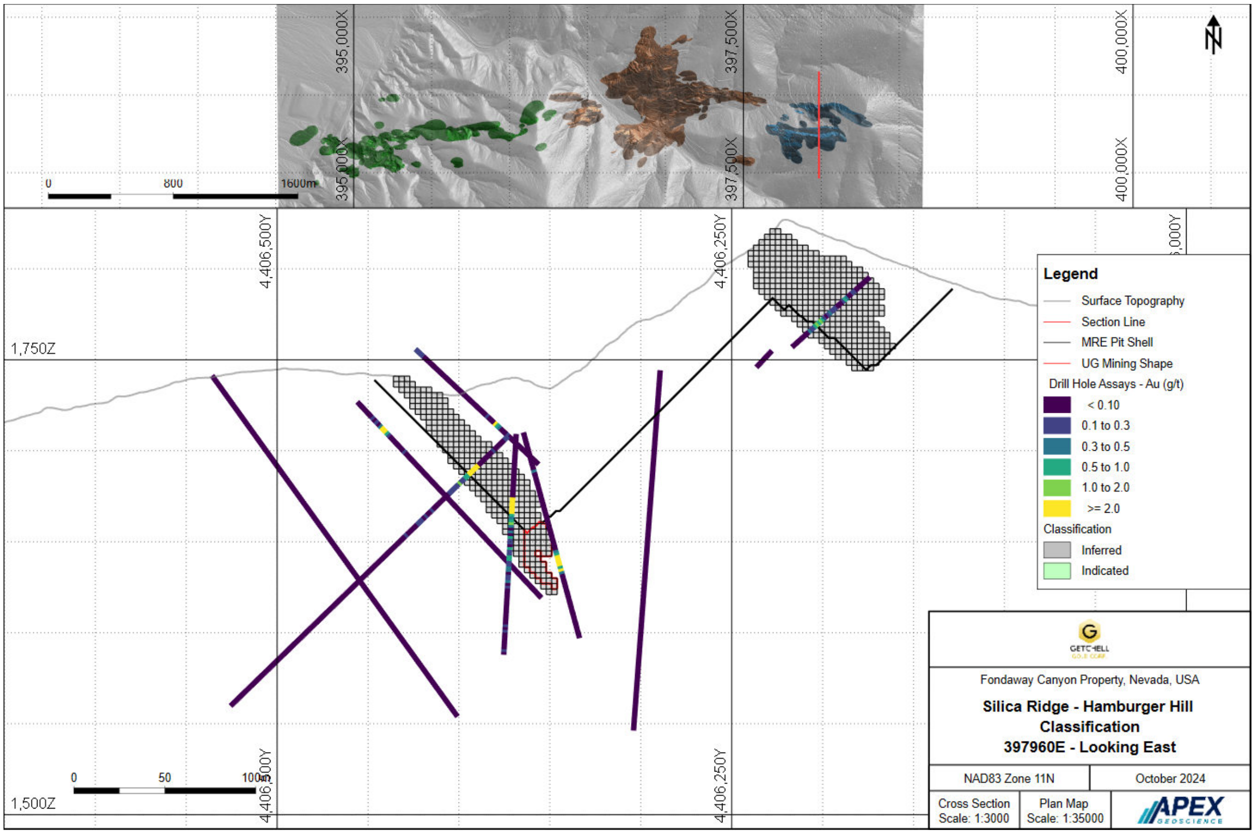


Figure 14-21: Cross-Section of the Classification Block Model for Silica Ridge – Hamburger Hill Estimation Domain

(Source: APEX Geoscience Ltd. 2024)

Note: Section window extents +/- 40 m and is looking north along 397935E.

14.9 Reasonable Prospects for Eventual Economic Extraction

According to CIM guidelines, reported mineral resources must demonstrate reasonable prospects for eventual economic extraction (RPEEE). The following section describes the parameter assumptions and methodologies used to constrain the 2024 Fondaway MRE statement.

14.9.1 Open Pit Mineral Resource Parameters

The resource block model underwent several pit optimization scenarios using Deswik's Pseudoflow pit optimization. Table 14-10 outlines the economic assumptions used for pit optimization and to establish the reporting cutoff of 0.3 g/t Au.

Table 14-10: Parameter Assumptions for Pit Optimization

Parameter	Unit	Value
Exchange Rate	C$/US$	0.75
Mining Cost – Waste	US$/tonne	2.7
Mining Cost – Mineralized	US$/tonne	2.7
G&A Cost	US$/tonne	2.0
Processing Cost	US$/tonne	15.0
Pit Slope	Degrees	45
Gold Recovery	%	92.0
Reporting Cutoff	Au g/t	0.3
Gold Price	US$/toz	1950
Royalty	%	1

14.9.2 Underground Mineral Resource Parameters

To demonstrate that the Fondaway Canyon Project MRE has RPEEE in a potential underground mining scenario, APEX personnel manually flagged blocks above a 1.0 g/t Au underground cutoff where the domains were 1.5 m thick or greater.

After evaluating the continuity of the manually flagged blocks below the open-pit shell, a cutoff of 1.75 g/t Au was chosen for reporting of the potential underground mineral resource. Table 14-11 outlines the economic assumptions used for reported underground cutoff for material within the potentially mineable shapes.

Blocks within domains narrower than the required underground mining thickness are only considered for inclusion in potential mining shapes if their diluted grade exceeds the cutoff when adjusted to meet the required minimum mining width. The dilution is calculated by adjusting the original grade based on the ratio of the minimum required thickness to the domain's actual thickness, effectively bulking the grade for a larger, standardized volume.

Table 14-11: Parameter Assumptions Used to Produce the Underground MRE Cutoff

Parameters	Unit	Value
Gold Price	US$/toz	1,950
Gold Recovery	%	92
Mining Cost – Open Stope	US$/t mined	83.0
Processing Cost	US$/t milled	15.0
G&A Cost	US$/t milled	2.0

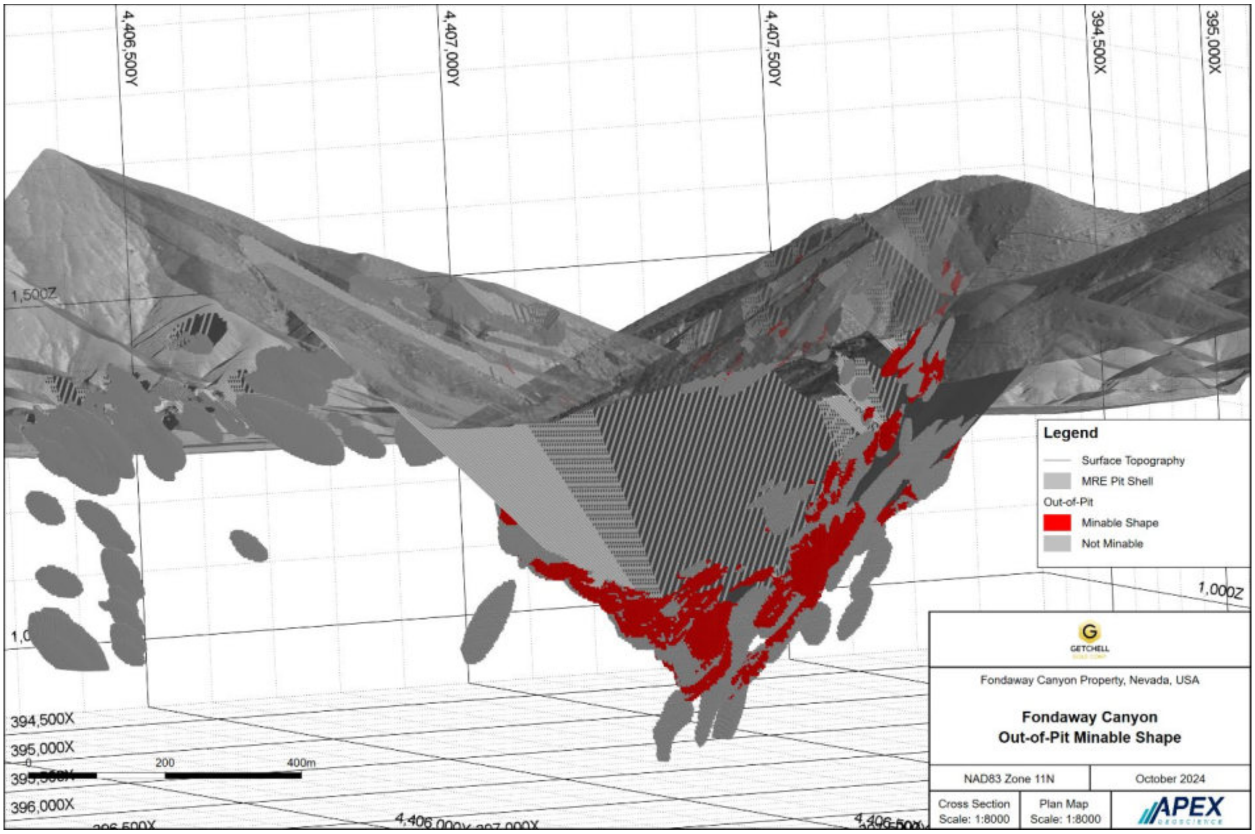


Figure 14-22: Orthogonal Illustrating the Potential Mineable Shapes Used to Constrain the Underground MRE

(Source: APEX Geoscience Ltd. 2024)

Note: Only blocks below the optimized Open-Pit shell are illustrated.

14.10 Mineral Resource Estimate Statement

The 2024 Fondaway Canyon MRE is reported in accordance with the Canadian Securities Administrators' NI 43-101 rules for disclosure and has been estimated using the CIM “Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines” dated November 29, 2019, and CIM “Definition Standards for Mineral Resources & Mineral Reserves” dated May 19, 2014. The effective date of the Mineral Resource is October 31, 2024.

Mineral Resource modeling was conducted in the UTM coordinate system relative to the North American Datum (NAD) 1983 Zone 11N (EPSG: 26911). The MRE utilized a block model with a size of 3.0 meters (X) by 3.0 meters (Y) by 3.0 meters (Z) to honor the mineralization wireframes for estimation. Gold (Au) grades were estimated for each block using Ordinary Kriging (OK) with locally varying anisotropy (LVA) to ensure grade continuity in various directions are reproduced in the block model. The MRE is reported as undiluted.

The reported open-pit resources utilize a cutoff of 0.3 g/t Au. The resource block model underwent several pit optimization scenarios using Deswik's Pseudoflow pit optimization. The resulting pit shell is used to constrain the reported open-pit resources. The reported underground MRE is constrained within mining shapes, assuming a long-hole open stope mining method and a grade cutoff of 1.75 g/t Au. The mining shapes were manually constructed, constraining contiguous material above the gold cutoff that met the minimum thickness of 1.5 m and volume requirements.

The 2024 Fondaway Canyon MRE comprises Indicated Mineral Resources of 648,000 troy ounces (648 koz) gold at a grade of 1.49 g/t Au, within 13,518,000 tonnes (13,518 kt) and an Inferred Mineral Resource of 1,670 koz at 1.16 g/t Au within 44,829 kt. Table 14-11 presents the complete 2024 Fondaway Canyon MRE statement. Table 14-12 illustrates the reported Mineral Resource broken down by zone, classification and mining category.

Table 14-12: Summary of 2024 Indicated and Inferred Mineral Resources on the Fondaway Canyon Project[1-9]

Mineral Resource Area	Cutoff Au (g/t)	Classification	Tonnes (kt)	Au (g/t)	Au (toz/st)	Au (koz)
Pit-Constrained Mineral Resource Estimate						
Main	0.3	Indicated	13,518	1.49	0.043	648
		Inferred	37,983	1.09	0.032	1,335
Mid Realm - South Mouth	0.3	Inferred	2,516	0.95	0.028	77
Silica Ridge - Hamburger Hill (HH)	0.3	Inferred	2,977	1.45	0.042	139
Underground Mineral Resource Estimate						
Main/ Silica Ridge - HH	1.75	Inferred	1,353	2.74	0.080	119
Total Mineral Resource Estimate						
All	0.3/1.75	Indicated	13,518	1.49	0.043	648
		Inferred	44,829	1.16	0.034	1,670

Notes:

1) Michael B. Dufresne, M.Sc., P.Geol., Senior Consultant, Mineral Resources of APEX Geoscience Ltd., who is deemed a qualified person as defined by NI 43-101 is responsible for the completion of the mineral resource estimation, with an effective date of October 31, 2024.
2) The mineral resources presented are not mineral reserves, and they do not have demonstrated economic viability. There is no guarantee that any part of the resources defined by the MRE will be converted to a mineral reserve in the future.
3) The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
4) The Inferred Mineral Resource in this estimate has a lower level of confidence than that applied to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of the Inferred Mineral Resource could potentially be upgraded to an Indicated Mineral Resource or a higher classification with continued exploration.
5) The Mineral Resources were estimated in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources & Reserves, Definitions (2014) and Best Practices Guidelines (2019) prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council.
6) Economic assumptions used include US$1,950/oz Au, process recoveries of 92% for Au, a US$15/t processing cost, G&A cost of US$2/t, and a 1% royalty.
7) The constraining pit optimization parameters were US$2.7/t mineralized and waste material mining cost and 45° pit slopes. Pit-constrained Mineral Resources are reported at an Au cutoff of 0.3 g/t.
8) The Underground Mineral Resources include blocks outside the constraining pit shell that form continuous and potentially mineable shapes. A mining cost of US$83/t and the economic assumptions above result in the Underground Au cutoff of 1.75 g/t. Mining shapes encapsulate material within domains with a minimum horizontal width of 1.5 meters, perpendicular to the strike, and target vertical and horizontal dimensions of approximately 15 meters. Blocks narrower than the required mining thickness are only included if their diluted grade exceeds the cutoff when adjusted to the minimum mining width.

14.11 Mineral Resource Estimate Sensitivity

Mineral Resources can be sensitive to the selection of the reporting cutoff grade. For sensitivity analyses, other cutoff grades are presented for review. Mineral Resources at various cutoff grades are presented for the Pit-Constrained Mineral Resources in Table 14-13 and the Underground-Constrained Mineral Resources Table 14-14.

Table 14-13: Sensitivities of the Pit-Constrained 2024 Fondaway Canyon MRE

Cutoff Au (g/t)	Indicated				Inferred			
	Tonnes (k)	Au (g/t)	Au (toz/st)	Au (koz)	Tonnes (k)	Au (g/t)	Au (toz/st)	Au (koz)
0.1	15,697	1.31	0.038	663	58,888	0.87	0.025	1,655
0.2	14,847	1.38	0.04	659	52,145	0.97	0.028	1,620
0.3	**13,518**	**1.49**	**0.043**	**648**	**43,476**	**1.11**	**0.032**	**1,551**
0.4	12,349	1.60	0.047	635	36,734	1.25	0.036	1,475
0.5	11,249	1.71	0.050	619	30,778	1.40	0.041	1,389
0.6	10,256	1.82	0.053	601	26,530	1.54	0.045	1,315
0.8	8,502	2.06	0.060	562	20,264	1.80	0.053	1,175
1.0	7,047	2.30	0.067	520	15,947	2.05	0.060	1,051
1.5	4,569	2.87	0.084	422	9,503	2.61	0.076	797
2.0	3,040	3.45	0.084	337	5,615	3.22	0.094	581

Table 14-14: Sensitivities of the Underground-Constrained 2024 Fondaway Canyon MRE

Cutoff Au (g/t)	Tonnes (k)	Au (koz)	Au (g/t)	Au (toz/st)
1.0	2,759	179	2.01	0.059
1.25	2,137	156	2.27	0.066
1.5	1,654	135	2.53	0.074
1.75	**1,353**	**119**	**2.74**	**0.080**
2.0	1,014	99	3.03	0.088

Combined Main and Silica Ridge – Hamburger Hill

14.12 Risk and Uncertainty in the Mineral Resource Estimate

The 2024 Fondaway Canyon MRE database is dominated by historical drilling. The drilling of 15 core holes by Nevada Contact Gold and Canagold in 2002 to 2017 and a further 28 holes by Getchell Gold from 2020 to 2022 in the Main Zone resource area has greatly improved the understanding of the geological model that was used in the construction of the 2024 MRE for the Main Zone. The geological and mineralization domains were improved and adjusted based upon this drilling. However, the geological model has changed from a discreet quartz vein model with higher grades, to a lower grade vein and stockwork mineralization zone model that is more suited to a bulk tonnage open pit extraction scenario for the resource. Uncertainty in the geological model still exists in areas of Inferred Mineral Resources with little to no modern drilling.

The MRE, and in particular the Inferred Mineral Resources, depend largely on a significant amount of pre-2000 drilling. The complete drill hole and assay database comprises assays from a number of drilling programs from 1981 to 2022, utilizing numerous analytical labs. The uniformity of analytical data across these generations of data collection is difficult to characterize because of the large number of drilling programs, the different laboratories used, and the lack of appropriate QAQC data, which provides a source of risk. To date, data verification of historical data has been completed to industry standards as described in Section 12.

Historical metallurgical testing has focused on previous geological interpretations of quartz vein stockworks and sulfide halos in carbonaceous to calcareous sedimentary host rocks. Additionally, modern data analysis has identified a significant portion of near surface oxidized mineralization. Modern metallurgical test work would increase the confidence of the recovery methodology of the new geological interpretations, and the recovery value of the identified near surface oxide mineralization.

The estimation domains are subject to several risks and uncertainties due to limitations in the geological model and the absence of a detailed structural model. The resource model is informed by drill hole data and an early-stage geological model; however, critical elements—such as detailed structural information—are lacking. This can affect the accuracy of domain interpretation and the continuity of mineralization across the deposit.

The variograms are very limited due to the lack of variable spatial orientation and variability in data spacing, which restricts the ability to model spatial relationships accurately. Additional drilling will improve the variability of the spatial distribution of data within each domain, improving the ability to model variograms accurately.

14.13 QP Comments on Section 14

Mineral Resources for the Project have been estimated using core and RC drill hole data, have been performed using industry best practices, and adhere to the requirements of the CIM Definition Standards for Mineral Resources & Mineral Reserves (2014).

The QP has checked the data used for the Mineral Resource and finds the resource model to be suitable to support future exploration and mining work, including Preliminary Economic Assessment level studies.

15. MINERAL RESERVE ESTIMATES

There is not currently a Mineral Reserve Estimate for the Fondaway Canyon Project.

16. MINING METHODS

Getchell Gold's Fondaway Canyon Project will consist of an open pit mining operation using conventional equipment. The Project is a conventional hard rock open pit mine that will use a contractor for mining. Mining is planned on 6-meter benches using haul trucks, and conventional drill and blast activities. Processed material is planned to be mined at a rate of 8,000 tonnes per day.

16.1 Initial Pit Limit Evaluations

The open pit optimization was performed using the network flow algorithm in Micromodel. By incorporating mining cost, processing cost, selling cost, gold recovery values, and an overall pit slope, the pit optimizer delineates an economic pit shell that maximizes the value of the extractable material. Creating a series of pit shells across a range of revenue factors, reducing the positive revenue by a percentage factor, to generate a series of pit shells which can be evaluated to determine which of the pits are relatively insensitive to economic factors.

This process assessed the sensitivity of the pit optimizations to the fluctuation in the revenue generated, as well as the impact of pit size and stripping ratio on the Projects' NPV. This procedure yields a series of nested pit shells that prioritize the extraction of the most economically viable and most economically robust material. Less profitable material, characterized by lower gold grade, higher stripping ratios, or higher ratios of the tonnage per ounce of gold may be mined later in the mine life, or not at all. These "robust" shells are used to develop the pushback designs.

The pit optimizations use reasonable and relevant economic, cost, recovery, and pit slope assumptions. Only resource blocks classified as Indicated and Inferred were included in the pit optimizer. The model lacks any blocks classified as Measured.

16.2 Open Pit Economic Parameters

During the pit limit analysis phase, the Project was envisioned as a 10 to 12 thousand tonnes per day operation with a mill and a roaster. This resulted in a marginal cutoff grade of 0.53 g/t Au. The pit analysis was performed with an average 45° pit slope in all directions. Due to the location of the pit within the valley, the pit was run with alternative slopes to effectively place the haul road in a lower area. The pit slope by azimuth is summarized in Table 16-1. The key optimization parameters used to define the economic pit shells for the deposits are summarized in Table 16-2.

Table 16-1: Pit Slope by Azimuth

Azimuth (degrees)	Slope (degrees)
0	50
90	50
180	43
250	35
300	35
345	50
360	50

Table 16-2: Pit Optimization Parameters

Modifying Factor	Units	Value
Gold Price	US$/toz	$2,200
Gold Price	US$/gr	$70.7
Gold Refining Charges	%	0
Royalties	%	1.0
Costs		
Mining	US$/tonne	$2.60
Processing	US$/tonne	$30.0
G&A	US$/tonne	$2.0
Plant Recovery	%	85
Slopes	deg	45

Figure 16-1 shows the results for each revenue factor shell, for processed and waste tonnes, along with profit. The shells selected for pushback designs and the eventual mine scheduling were 20%, 30%, 40%, 50%, and 60%, although the 50% and 60% pits were eliminated as they required stripping ratios greater than 12:1 and were marginally profitable.

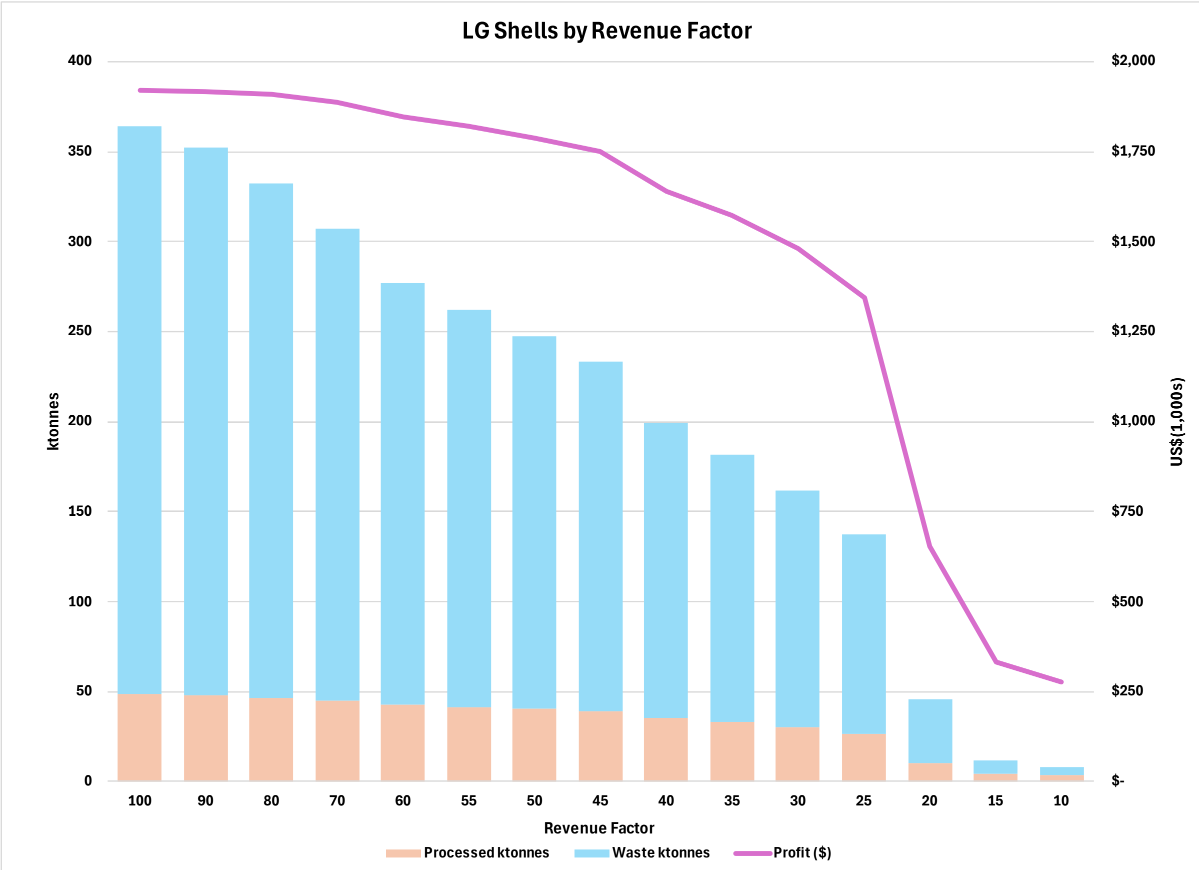


Figure 16-1: LG Shells by Revenue Factor

(Source: Forte Dynamics, Inc. 2024)

Table 16-3: Profit Factor for Optimization Results

Percent	Processed ktonnes	Au g/t	Waste ktonnes	Total ktonnes	Stripping Ratio	Au ktoz	Revenue 000s $	Mining Cost 000s $	Processing Cost 000s $	Total Op Ex 000s $	Net 000s $	Total tonne/toz Au	Profit/ tonne
100	48,508	1.48	315,371	363,879	6.5	2,311	$ 4,235	$ 946	$ 1,455	$ 2,401	$ 1,834	157.46	$ 5.04
90	47,905	1.48	304,067	351,972	6.35	2,284	$ 4,186	$ 915	$ 1,437	$ 2,352	$ 1,834	154.10	$ 5.21
80	46,700	1.49	285,633	332,334	6.12	2,233	$ 4,092	$ 864	$ 1,401	$ 2,265	$ 1,827	148.83	$ 5.50
70	45,059	1.49	261,808	306,868	5.81	2,159	$ 3,957	$ 798	$ 1,352	$ 2,150	$ 1,807	142.12	$ 5.89
60	42,734	1.50	233,946	276,679	5.47	2,058	$ 3,772	$ 719	$ 1,282	$ 2,001	$ 1,770	134.43	$ 6.40
55	41,556	1.50	220,862	262,418	5.31	2,006	$ 3,676	$ 682	$ 1,247	$ 1,929	$ 1,747	130.84	$ 6.66
50	40,312	1.50	206,836	247,148	5.13	1,947	$ 3,567	$ 643	$ 1,209	$ 1,852	$ 1,716	126.96	$ 6.94
45	39,045	1.50	194,270	233,314	4.98	1,887	$ 3,459	$ 607	$ 1,171	$ 1,778	$ 1,681	123.62	$ 7.20
40	**35,439**	**1.51**	**163,785**	**199,224**	**4.62**	**1,724**	**$ 3,159**	**$ 518**	**$ 1,063**	**$ 1,581**	**$ 1,577**	**115.59**	**$ 7.92**
35	33,254	1.52	148,101	181,355	4.45	1,626	$ 2,981	$ 472	$ 998	$ 1,469	$ 1,512	111.50	$ 8.33
30	30,065	1.55	131,240	161,305	4.37	1,499	$ 2,747	$ 419	$ 902	$ 1,321	$ 1,425	107.63	$ 8.84
25	26,267	1.58	111,331	137,598	4.24	1,332	$ 2,441	$ 358	$ 788	$ 1,146	$ 1,295	103.31	$ 9.41
20	9,902	1.80	36,092	45,994	3.64	573	$ 1,049	$ 120	$ 297	$ 417	$ 633	80.33	$ 13.76
15	4,445	1.86	7,277	11,722	1.64	266	$ 487	$ 30	$ 133	$ 164	$ 323	44.12	$ 27.56
10	3,547	1.90	4,161	7,709	1.17	216	$ 397	$ 20	$ 106	$ 126	$ 270	35.61	$ 35.06

The shell selection is presented in Table 16-3. Pit shell 40 was selected as the optimal pit shell, which corresponds to a 40% Revenue Factor. Revenue Factors are used to calculate pit shells by varying the commodity prices but keeping the costs the same. This shell has a total tonnage of 199.2 Mt including 35.4 Mt of processed material at an average grade of 1.51 g/t Au for 1.7 Moz of contained gold. The average stripping ratio is 4.73. Figure 16-2 shows the percentage of profit, processed material, and rock by revenue factor shell. Figure 16-3 and Figure 16-4 are cross sections showing the LG pit shells and the estimated block grades. Figure 16-5 shows the location of these cross sections.

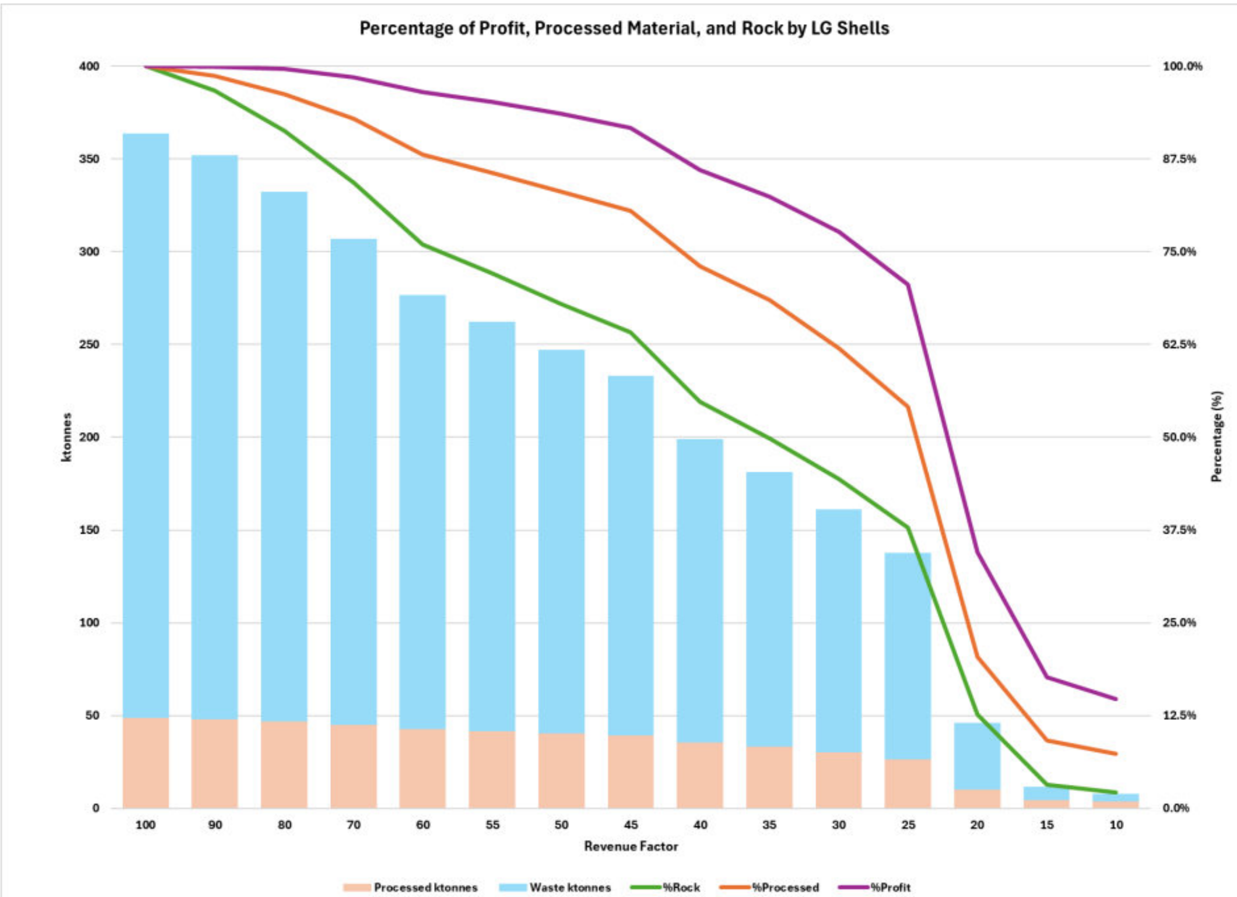


Figure 16-2: Percentage of Profit, Processed Material, and Rock by LG Shells

(Source: Forte Dynamics, Inc. 2024)

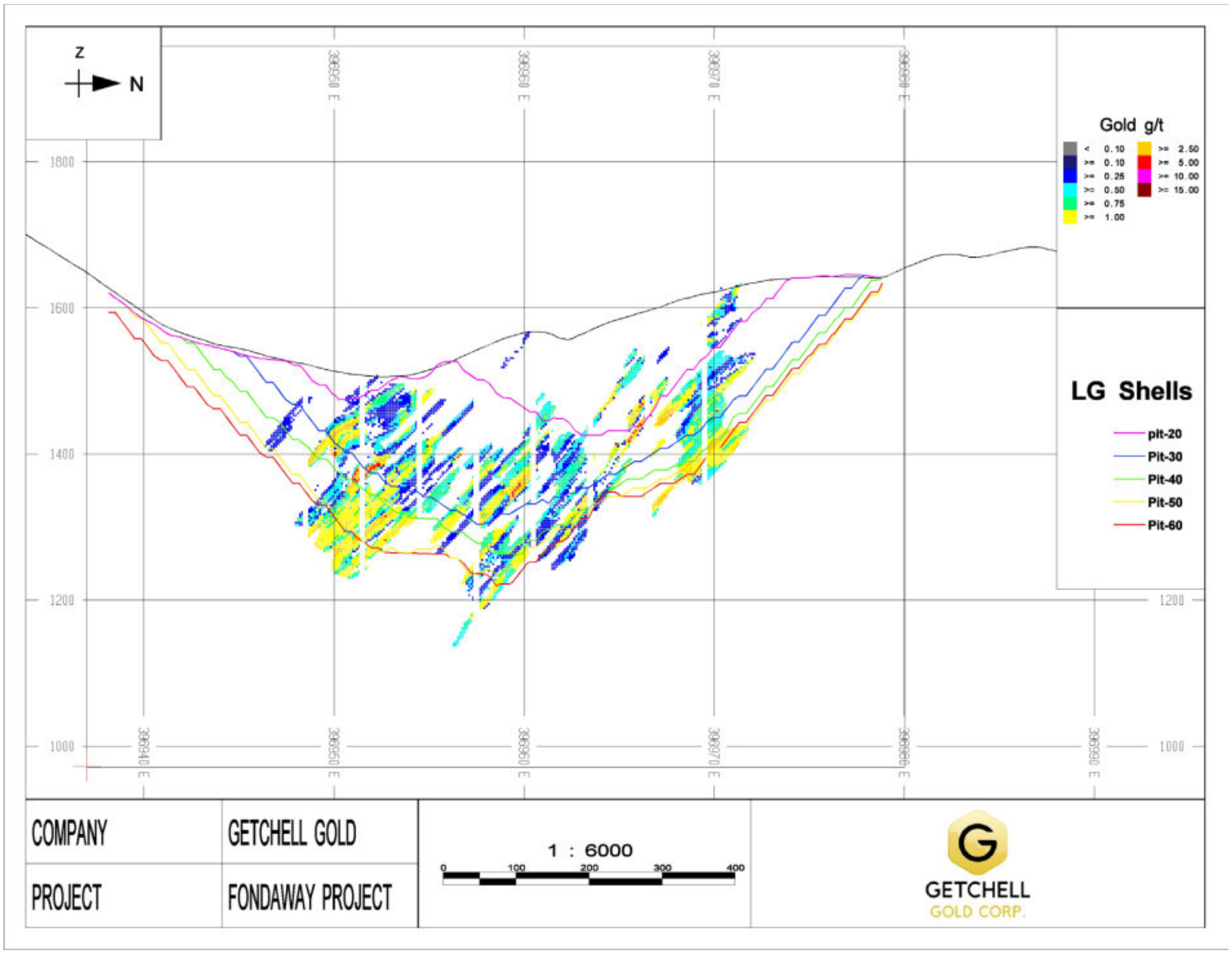


Figure 16-3: Pit Optimization Looking West (Section B – B')

(Source: Forte Dynamics, Inc. 2024)

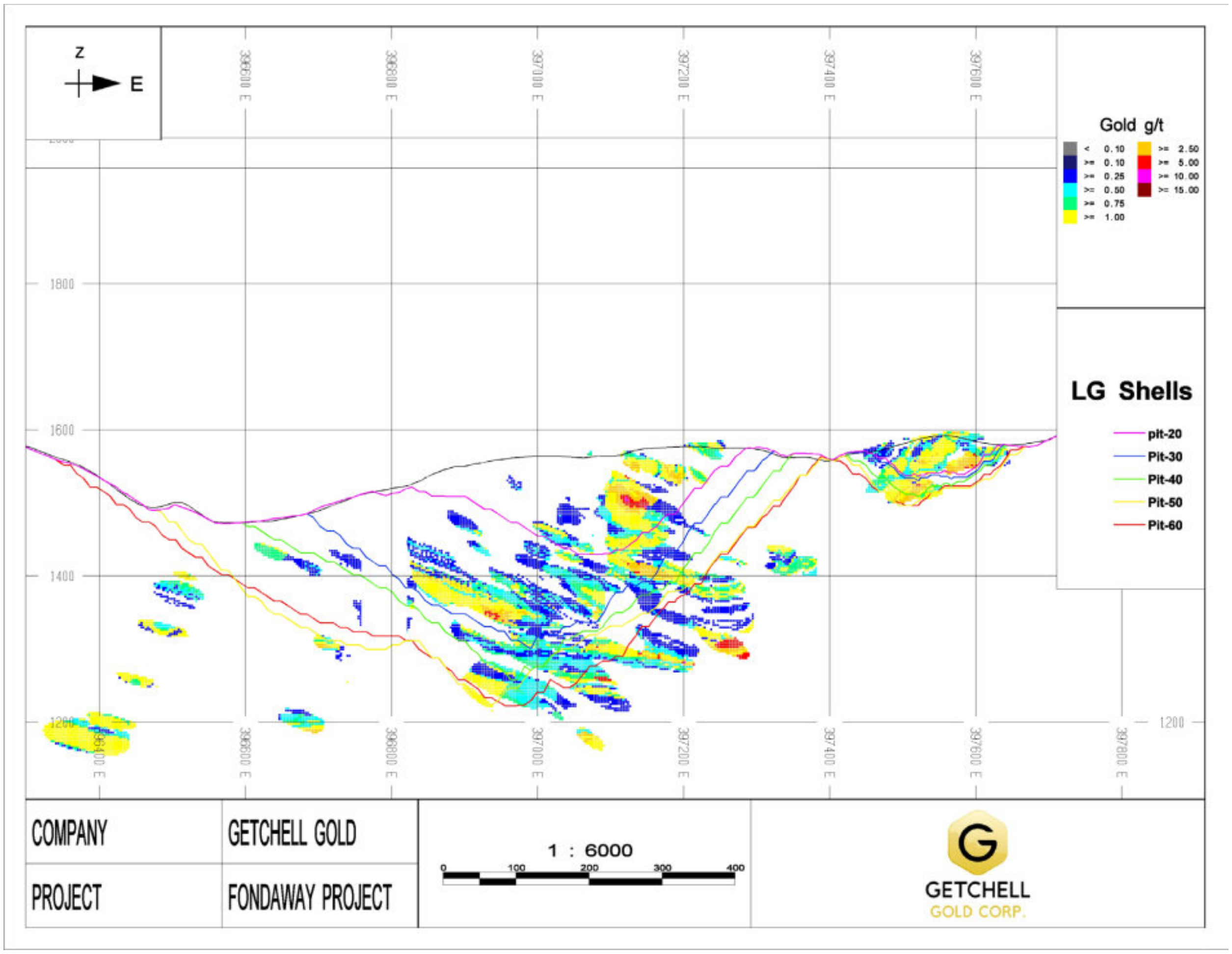


Figure 16-4: Pit Optimization Looking North (Section A – A')

(Source: Forte Dynamics, Inc. 2024)

16.3 Pit Designs

The pit shells and the block model were used as a basis for preliminary life of mine (LOM) open pit mine designs. Ramps were limited at 10% grade for in-pit haulage, ensuring safe operations. Table 16-4 shows the pit design parameters used. Figure 16-5 shows all phases of the pit designs and cross section locations, Figure 16-6 - Figure 16-9 show the individual pit phase designs, Figure 16-10 shows the final pit design and estimated block model, and Figure 16-11 shows a cross section view of all phases of the pit designs.

Table 16-4: Pit Design Parameters

Parameter	Units	Value
Face Angle	degrees	50
Bench Height	m	6
Berm Width	m	3
Inter-Ramp Angle (IRA)	degrees	70
Ramp Width	m	30
Ramp Width One Lane	m	20
Road Gradient	%	10
Minimum Mining Width	m	30

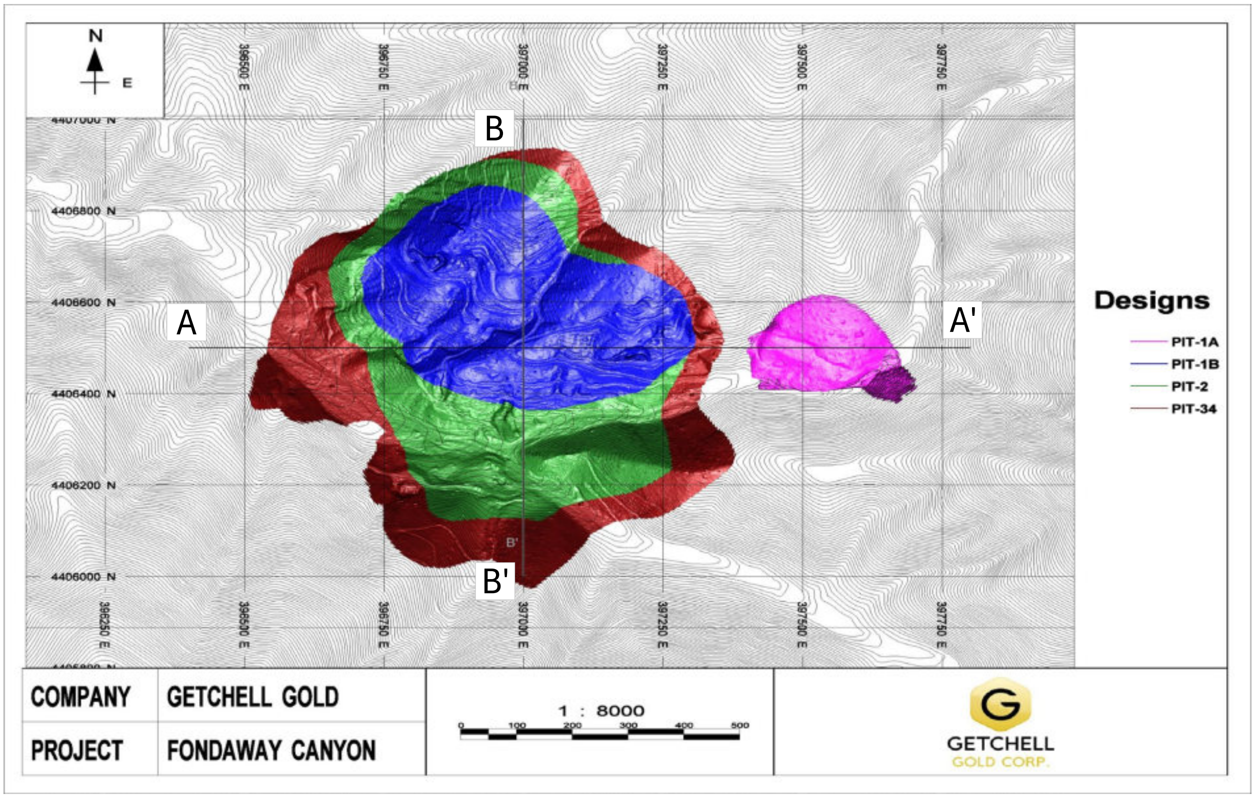


Figure 16-5: Pit Design All Phases (Includes A-A' and B-B' Section Lines)

(Source: Forte Dynamics, Inc. 2024)

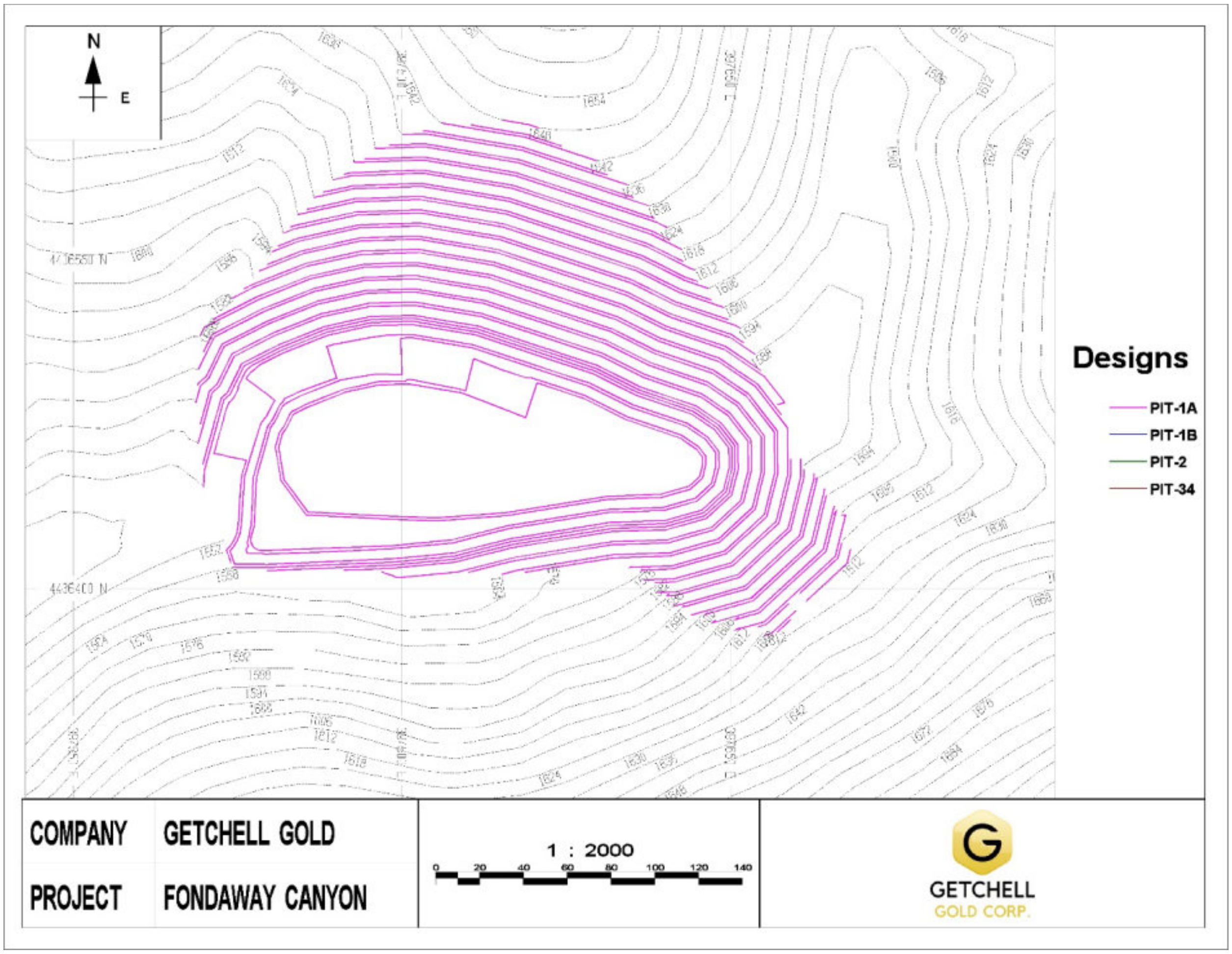


Figure 16-6: Pit Design on Phase 1A

(Source: Forte Dynamics, Inc. 2024)

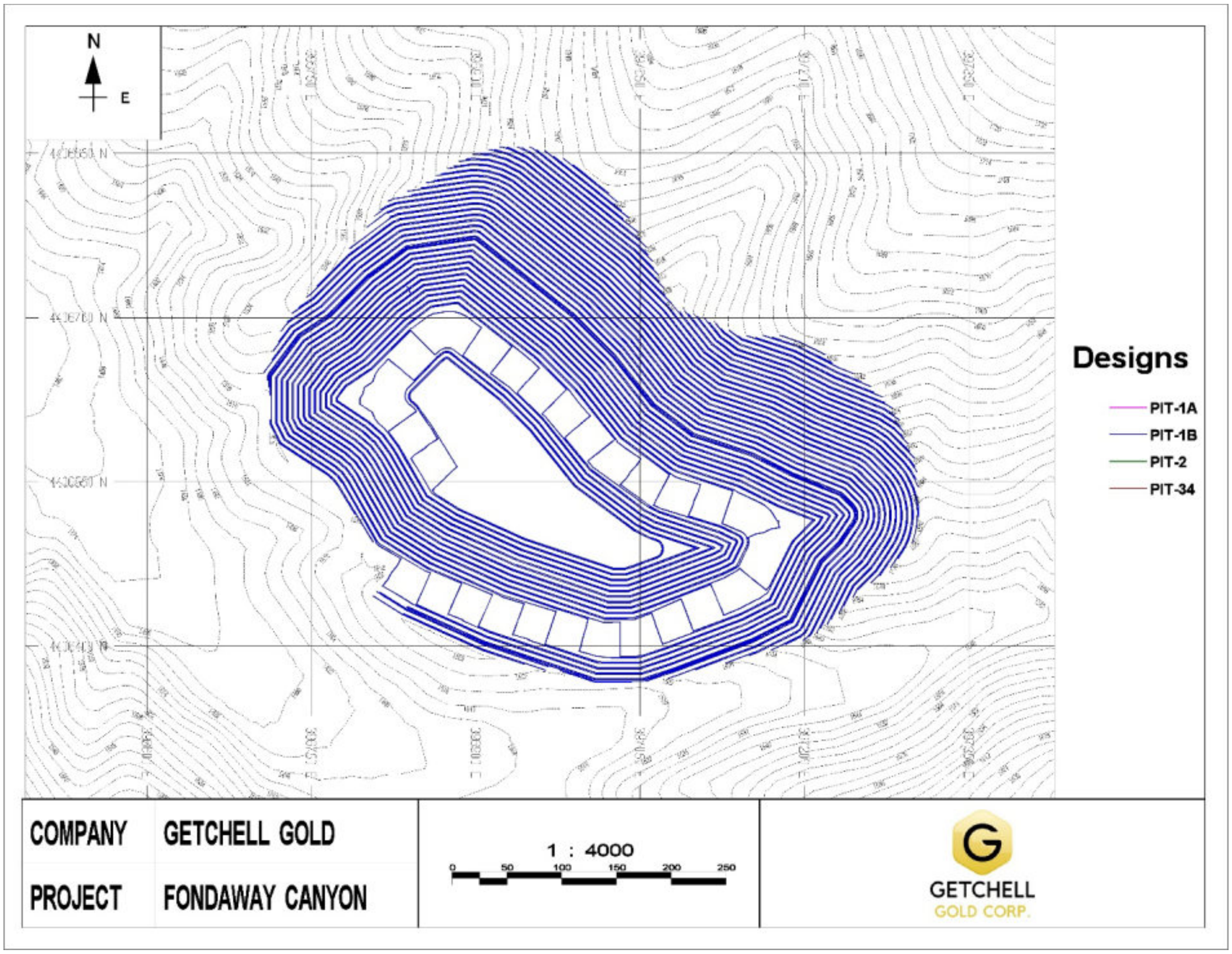


Figure 16-7: Pit Design on Phase 1B

(Source: Forte Dynamics, Inc. 2024)

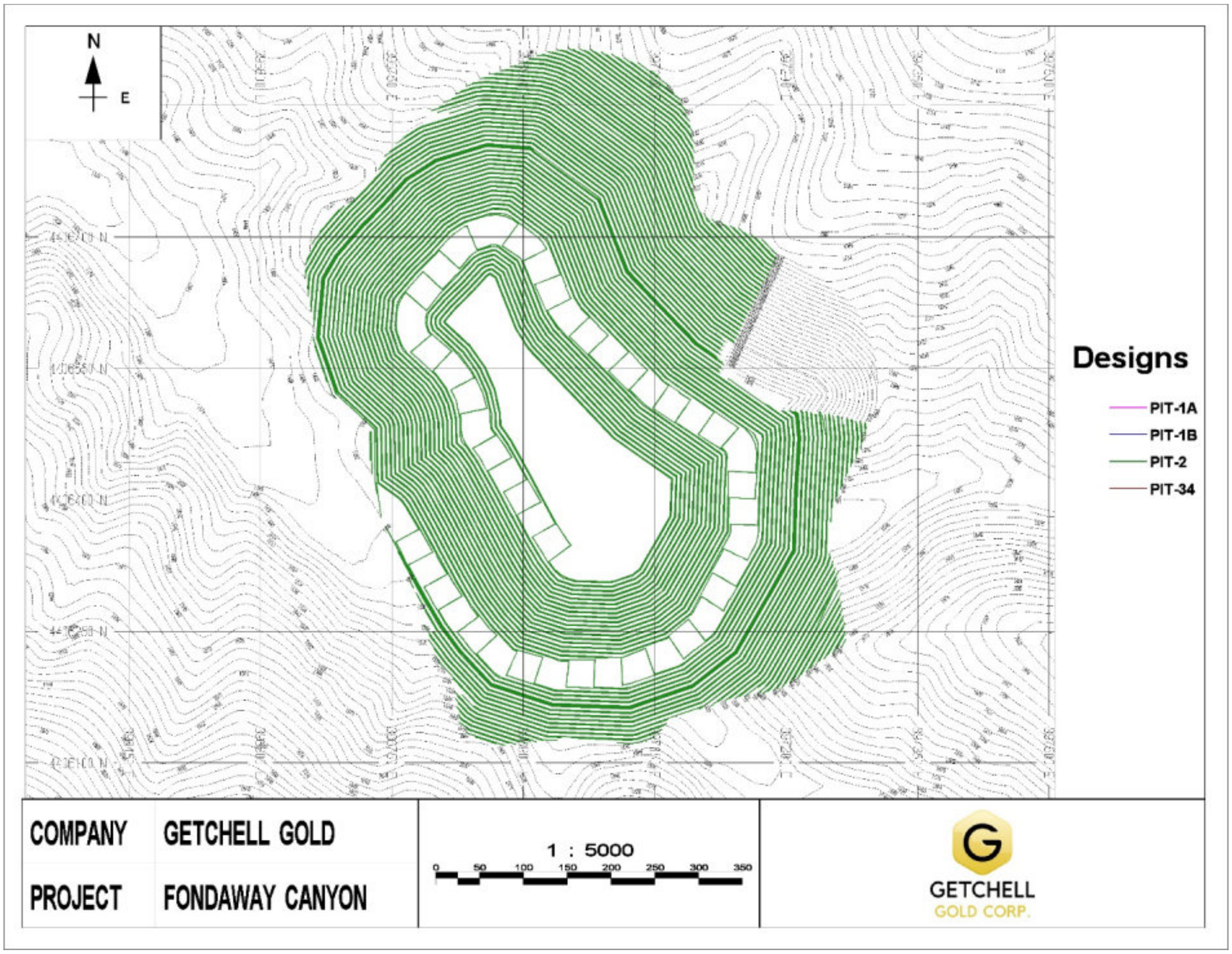


Figure 16-8: Pit Design on Phase 2

(Source: Forte Dynamics, Inc. 2024)

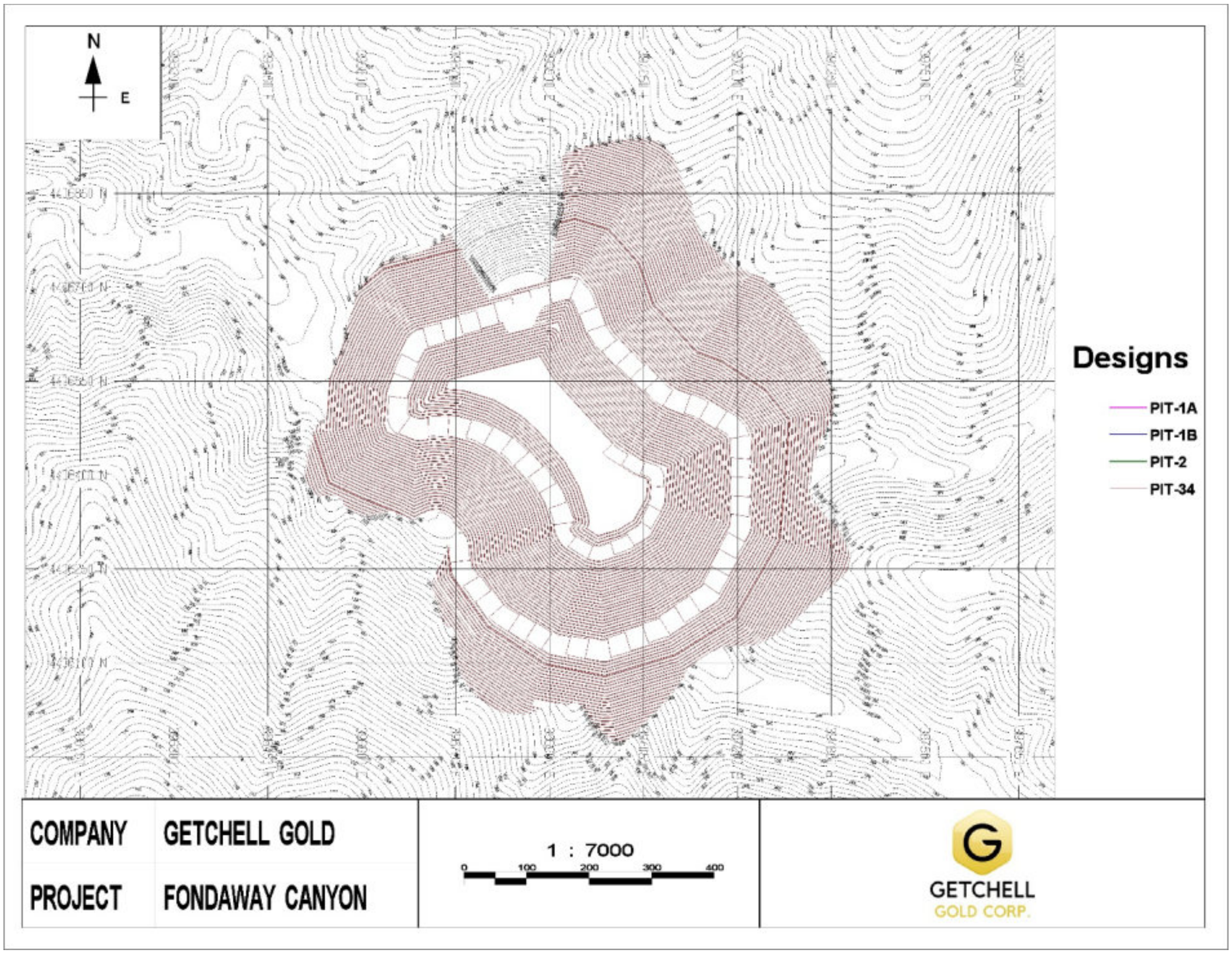


Figure 16-9: Pit Design on Phase 34

(Source: Forte Dynamics, Inc. 2024)

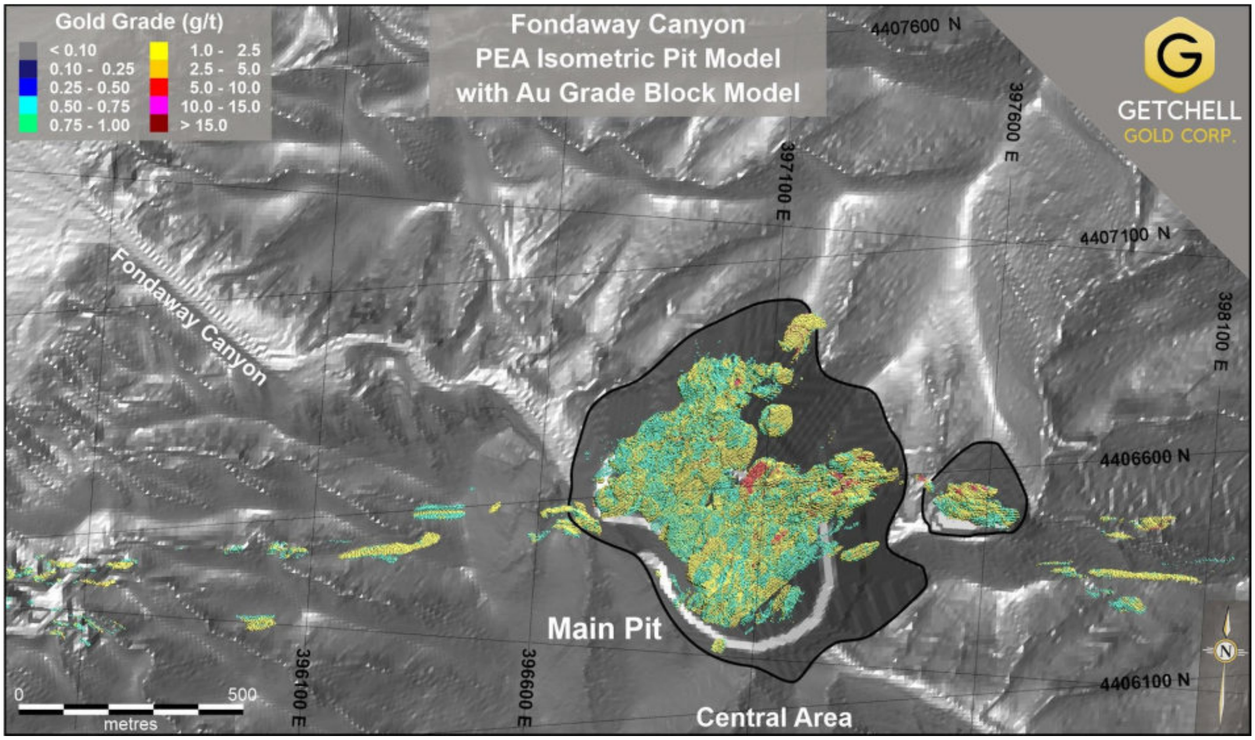


Figure 16-10: Getchell Fondaway Canyon Final Pit and Estimated Block Model

(Source: Forte Dynamics, Inc. 2024)

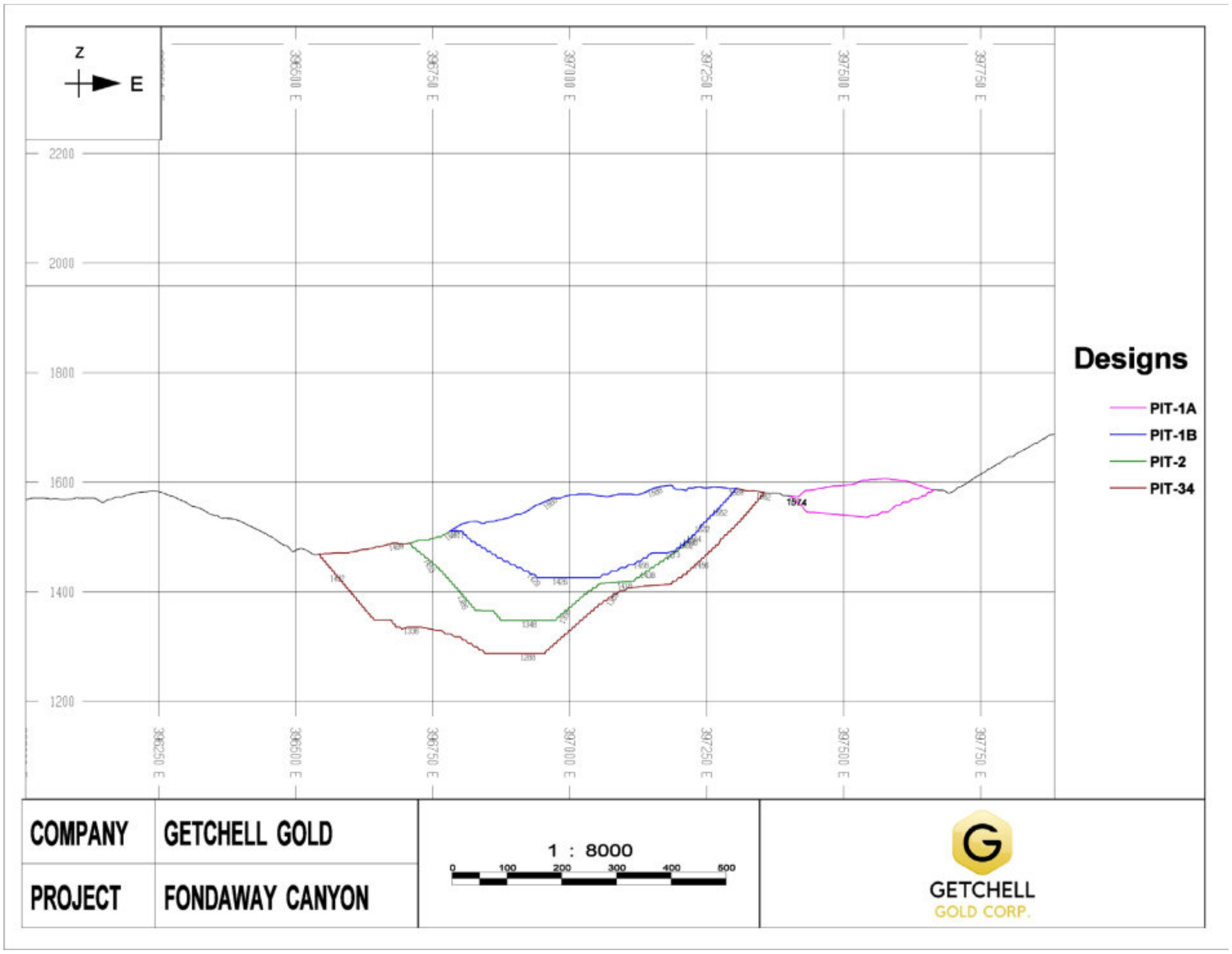


Figure 16-11: Getchell Fondaway Canyon Project EW Section A-A' All Phases

(Source: Forte Dynamics, Inc. 2024)

16.4 Haul Road Design

Existing roads are planned to be utilized where possible. New haul roads will have to be built to the top of each phase for waste mining. This will require the removal of vegetation and any topsoil for the construction of these haul roads.

Haul roads were designed to be wide enough for two-lane traffic, except for the bottom four benches, which were designed for single-lane travel to minimize waste stripping requirements.

16.5 Economic Evaluation

The economic evaluation parameters are different than the pit limit runs. Test work during this study determined that a flotation mill to produce a concentrate for sale to a pressure oxidation plant for refining. Refining and transportation costs have been estimated in the economic analysis.

16.6 Cutoff Grade

The processed/waste cutoff grades for mineable resource reporting were based on the economic parameters and the individual metal grades within each block. These parameters have been modified from the numbers in Table 16-2 due to changes discovered during metallurgical testing, and due to an updated analysis of the potential gold markets. The prices and cutoffs for the economic evaluation is shown in Table 16-5.

Table 16-5: Design Metal Prices, Cost, and Recoveries

Description	Units	Value
Mining Cost	US$/tonne mined	$3.54
Processing Cost	US$/tonne processed material	$15.00
G&A Cost	US$/tonne processed material	$2.00
Gold Price	US$/toz	$2,250
Gold Price	US$/gr	$72.3
Transport and Refining Cost	US$/tonne ore	$10.00
Recovery	%	84

$$COG\left(\frac{\text{g}}{\text{ton}}\right) = \frac{(\text{Mining OP Cost+ ProcessCost+G\&A cost+Transport and Refining Cost})}{(\text{Gold Price}-\text{Selling Cost})\text{x Recovery}}$$

Where:

Process is the total on site processing cost

Recovery is the metallurgical recovery to concentrate and refining (%)

Selling cost includes transport and refining at 10% of gold contained in the concentrate

Using the inputs from Table 16-5 and the above cutoff grade equation, the cutoff grade is rounded to 0.50 g/t Au.

16.7 Pit Design Inventories

Indicated and Inferred Mineral Resource inventories of the preliminary open pit designs are tabulated in Table 16-6. In summary, the final pit limit contains a total tonnage of 173.7 Mt including 11.7 Mt of Indicated Mineral Resource at 1.73 g/t Au, and 18.7 Mt of Inferred Mineral Resource at 1.36 g/t Au to be processed for 1.47 Moz of contained gold. Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability. There has been insufficient exploration to define the Inferred Resources tabulated above as an Indicated or Measured Mineral Resource, however, it is reasonably expected that the majority of the Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration. There is no guarantee that any part of the Mineral Resources discussed herein will be converted into a Mineral Reserve in the future.

Table 16-6: In-Pit Mineral Resources by Pit Phase

Cut	Material	Processed Resource ktonnes	Au g/t	Au ktoz Contained	Au ktoz Recovered	Waste ktonnes	Total ktonnes	Stripping Ratio
PIT-1A	2-Indicated	533	2.12	36	32	174	707	0.33
	3-Inferred	173	1.19	7	6	114	287	0.66
						2,850	2,850	0.00
PIT-1B	2-Indicated	5,011	1.76	283	249	2,598	7,609	0.52
	3-Inferred	2,359	1.34	102	90	3,574	5,933	1.51
						28,388	28,388	0.00
PIT-2	2-Indicated	3,885	1.63	204	180	1,704	5,590	0.44
	3-Inferred	7,293	1.41	329	290	8,897	16,190	1.22
						38,315	38,315	0.00
PIT-34	2-Indicated	2,326	1.76	131	116	975	3,301	0.42
	3-Inferred	8,877	1.33	378	333	9,461	18,338	1.07
						46,228	46,228	0.00
Total	**2-Indicated**	**11,756**	**1.73**	**655**	**576**			
	3-Inferred	**18,702**	**1.36**	**816**	**718**			
						143,277	**173,734**	**4.7**

16.8 Drilling and Blasting

Primary fragmentation for mining will be carried out using traditional drill and blast techniques that are standard in open pit mining. This study used a powder factor of 0.8 kg/m3 for mineralized material and waste rock.

Production drilling and blasting will be included in the mining contract. Benches are blasted and mined in 6-meter benches. Buffer and pre-trimmed rows are planned to be allowed for controlled blasting and to minimize back breaking damages to the highwalls.

16.9 Production Schedules

The mine designs were used to create a LOM schedule for the site. This schedule considers open-pit operations. The yearly mine schedule is presented in Table 16-6. The schedule below was completed quarterly for the first year of mining and yearly for the rest of the mine life. The production schedule is driven by the nominal rate of 8,000 t/d processed material (2.9 Mt/y) and the average LOM stripping ratio is 4.73:1 waste-to-ore, using a 0.5 g/t Au cutoff grade.

Table 16-7 details the LOM production by year. Figure 16-12 shows the LOM production schedule for processed material, waste and recovered Au toz.

Table 16-7: LOM Production Schedule

Period	Days	Processed ktonnes	Au g/t	Waste ktonnes	Total Mined ktonnes	Stripping Ratio	Tonnes / Day			Total Metal	
							Processed ktonnes	Waste ktonnes	Total ktonnes	Contained Au toz	Recovered Au toz
Q1	91	662	1.61	4,444	5,106	6.7	7.3	48.8	56.1	34,267	28,784
Q2	91	644	1.64	4,545	5,189	7.1	7.1	49.9	57.0	33,964	28,530
Q3	91	627	1.58	4,478	5,105	7.1	6.9	49.2	56.1	31,806	26,717
Q4	92	712	1.49	4,564	5,276	6.4	7.7	49.6	57.3	34,195	28,724
Year 2	365	2,898	1.54	17,726	20,624	6.1	7.9	48.6	56.5	143,690	120,700
Year 3	365	2,910	1.72	17,637	20,547	6.1	8.0	48.3	56.3	161,056	135,287
Year 4	365	2,907	1.44	17,959	20,866	6.2	8.0	49.2	57.2	134,514	112,991
Year 5	365	2,920	1.59	17,520	20,440	6.0	8.0	48.0	56.0	149,002	125,161
Year 6	365	2,920	1.51	17,527	20,447	6.0	8.0	48.0	56.0	141,915	119,209
Year 7	365	2,920	1.46	17,406	20,327	6.0	8.0	47.7	55.7	136,632	114,771
Year 8	366	2,928	1.46	9,002	11,930	3.1	8.0	24.6	32.6	137,262	115,300
Year 9	365	2,920	1.34	5,952	8,872	2.0	8.0	16.3	24.3	125,372	105,312
Year 10	365	2,920	1.48	2,992	5,912	1.0	8.0	8.2	16.2	139,172	116,904
Year 11	365	1,454	1.35	1,639	3,093	1.1	4.0	4.5	8.5	63,115	53,017
Total		**30,343**	**1.503**	**143,392**	**173,734**	**4.73**				**1,465,962**	**1,231,408**

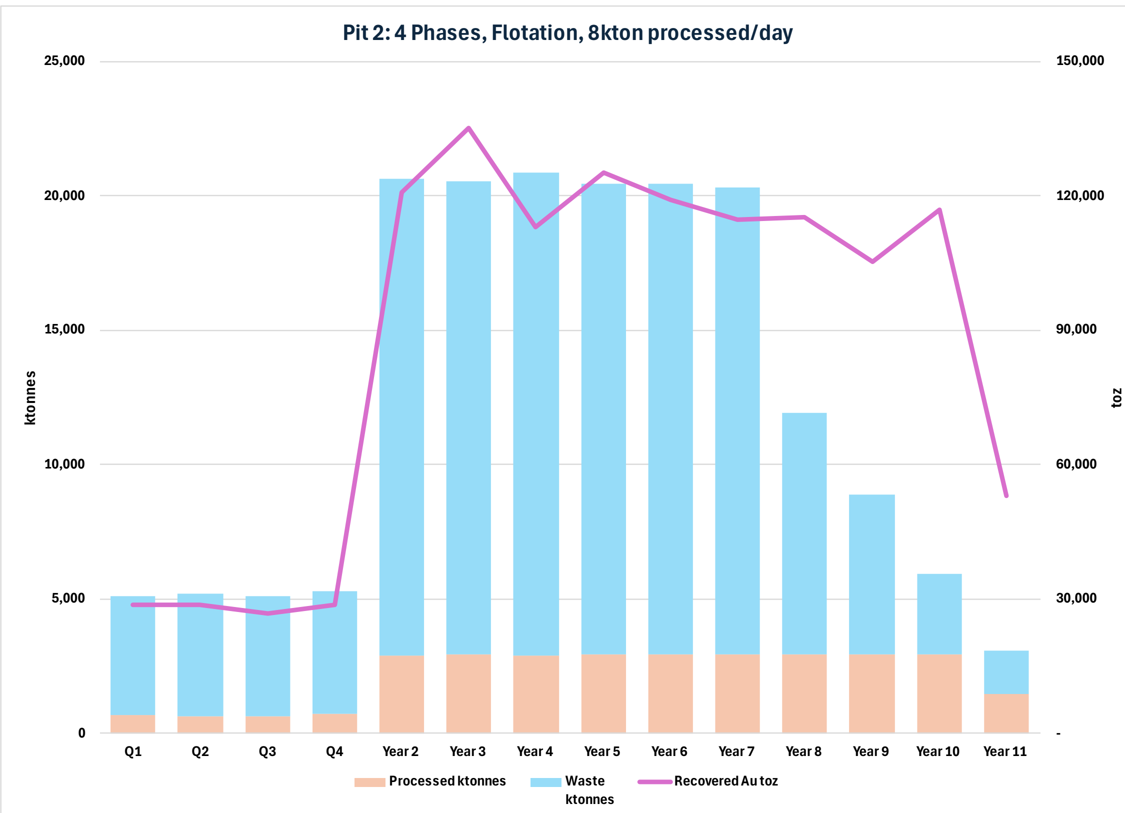


Figure 16-12: LOM Production Schedule

(Source: Forte Dynamics, Inc. 2024)

16.10 Mine Fleet

Mining equipment will be supplied by the mining contractor which will be also responsible for management of mining crews.

16.11 Dewatering

Dewatering may be necessary later in the pit life but has not been quantified at this time. Dewatering is addressed in more detail in Section 18.3.

17. RECOVERY METHODS

17.1 Introduction

A conceptual process flowsheet was developed based on the historical and scoping level study undertaken by Forte Analytical in 2024.

A techno-economic evaluation of the various processing options was undertaken. The options included (1) whole ore pressure oxidation (POX) followed by cyanidation, (2) crush-grind-flotation of gold and sulfides followed by POX and cyanidation, and (3) crush-grind-flotation and sell flotation concentrate. The last processing option is the most viable for Getchell Gold at this time.

17.2 Process Flowsheet

The following assumptions were made to develop the conceptual process flowsheet given in Figure 17-1:

- The optimum comminution circuit for 8,000 t/d will be three-stage crushing followed by a closed circuit ball mill-cyclone system.
- Rougher flotation will recover 88% to 90% of the gold in 20% of the weight.
- Two-stage cleaner flotation will reject 50% of the weight while recovering 95% of the gold in the rougher concentrate.
- The second-cleaner concentrate will have 10% of the weight of the original feed (i.e. 800 t/d) assaying about 20 g/t Au.
- The gold recovery in the pressure oxidation circuit is estimate at 95% of the contained gold in the flotation concentrate.

Additional test work is needed to confirm this assumption.

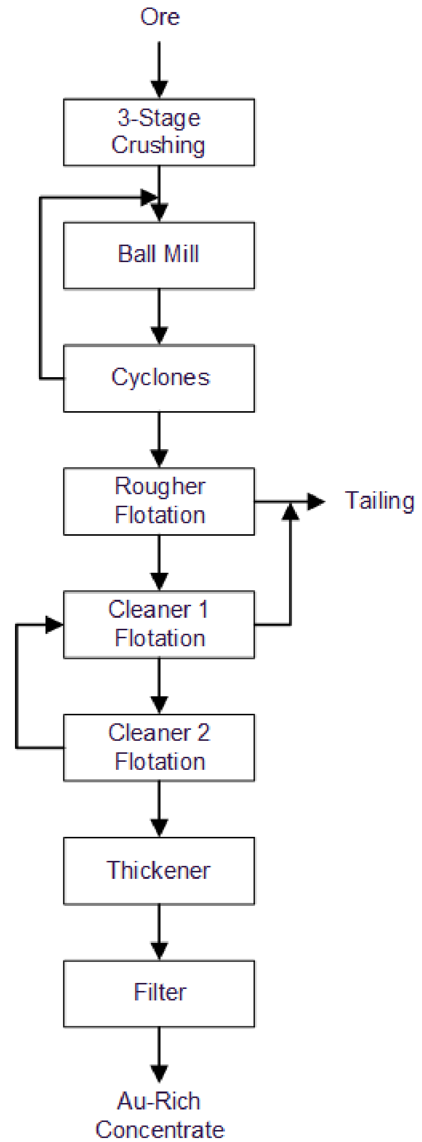


Figure 17-1: Conceptual Process Flowsheet

(Source: Forte Dynamics, Inc. 2024)

The process flowsheet will consist of three stages of crushing followed by ball-mill-cyclone configuration. The cyclone overflow will be sent to the rougher flotation. The reagents, namely xanthate, AP 404 and AF 65 will be added to the mill.

The rougher-concentrate will be sent to the first-cleaner flotation. The first-cleaner tailing will be combined with the rougher tailing and sent to the tailings pond. The first-cleaner concentrate will be further upgraded in second-cleaner flotation to produce a saleable concentrate assaying ± 20 g/t Au. The second-cleaner tailing will be sent to first-cleaner flotation.

18. PROJECT INFRASTRUCTURE

The Property is accessed from Fallon east on Highway 50 and then north on Highway 116 to the settlement of Stillwater and then north on the gravel East County Road 30 miles (50 km) along the front range of the Stillwater Range to the mouth of Fondaway Canyon. Existing mine roads provide access into the canyon and there remains a dense network of drill roads developed over decades of exploration and mining in various states of use.

There are no public utilities, including electrical power, on the Property. Two permitted water wells are present on the Property, with water available for mining use under the lease agreement.

The closest significant communities are Reno, 140 km to the west-southwest, Lovelock, 78 km to the northwest, and Fallon, 58 km to the southwest. Fallon is the county seat with above normal resources for the area (e.g. supplies and accommodations) primarily due to the contribution of the Fallon Naval Air Station and the generous agricultural setting as a draw and support for the region.

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

18.1 Infrastructure

18.1.1 Historical Installations

During 1989 and 1990, Tenneco operated an open pit mine with heap leach processing. Tenneco mined approximately 171,000 tons of oxide mineralization from the South Mouth pits at an average grade of 1.1 g/t Au. They supplemented this production with 12,000 tons of oxide material from the Reed Pit and 4,000 tons of oxide material from the Half Moon Stibnite Pits. The total gold produced from the Tenneco mining was 6,324 ounces.

The leach pads and plant site have been reclaimed. The access road to the South Mouth pit has been armored with cobbles to prevent erosion.

18.1.2 Current Infrastructure

There are no structures at Fondaway Canyon. There are the remains of old mine workings, both underground and open pit, as well as two water wells and the existing exploration roads. There is ample space for a plant and overburden deposits in the basin at the mount of the canyon which is within the current permit limit. Figure 18-1 shows the general infrastructure layout; the map uses contour line spacing at 40 meters.

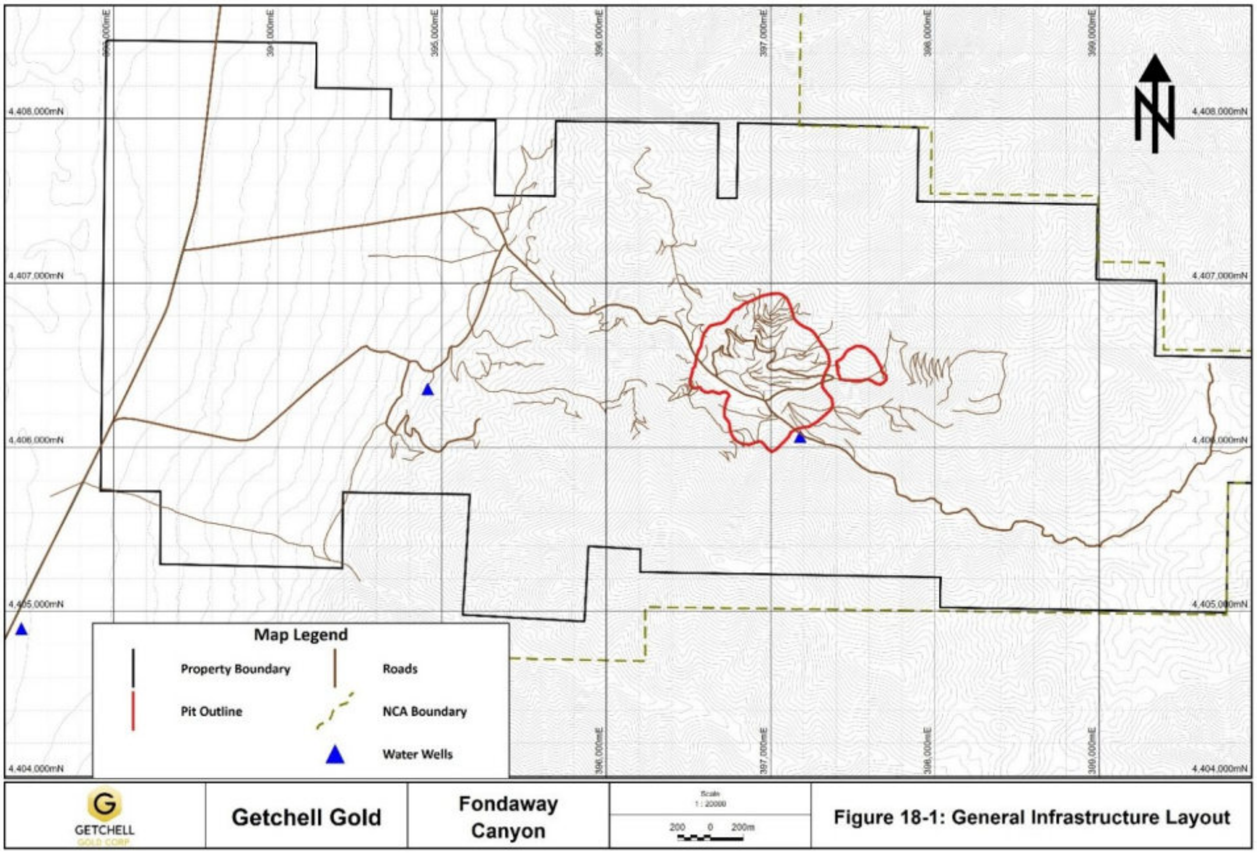


Figure 18-1: General Infrastructure Layout

(Source: Forte Dynamics, Inc. 2024)

18.1.3 Access & Site Roads

Fondaway Canyon is accessible from either Fallon or Lovelock Nevada. From Fallon, proceed east on Highway 50 and then north on Highway 116 to the settlement of Stillwater and then north on the gravel East County Road 30 miles (50 km) along the front range of the Stillwater Range to the mouth of Fondaway Canyon. From Lovelock, proceed east on I-80 approximately 6 miles to Hwy 396. Follow the Coal Canyon-Stillwater road, which becomes the Iron Mine Road, and eventually the Old Emigrant Trail for about 23 miles south turning onto East County Road 30 for about 13 miles to the mouth of Fondaway Canyon.

An existing network of dirt roads connect the various areas of the Project site including previously mined pits and exploration areas. The existing roads have been maintained and may be improved for use during startup and development.

18.2 Project Buildings

Physical infrastructure from prior operations has been removed and reclaimed.

18.3 Water Supply and Dewatering

Water consumption for the Project during mining, processing, and reclamation and closure will consist of dust control, process water, and potable water supply. 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. In-pit wells, and in-pit horizontal and vertical borehole drains may also be used. 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 Plan for the entire site to control storm water impacts to the environment. The plan will outline the measures required to accomplish the above goals.

18.4 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.

18.5 Labor

Northern Nevada has several larger mining resource centers along I-80, including Elko, Winnemucca, and Reno. Lovelock and, to a lesser extent, Fallon are regional mining centers. There is a labor pool of experienced miners, and staffing the mine is not anticipated to be an issue. Both towns can provide housing, supplies, and industrial services.

18.6 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.

19. MARKET STUDIES AND CONTRACTS

A Fondaway Canyon operation is planned to produce a flotation concentrate with high grade gold values for further processing via pressure oxidation (POX) and leaching. There are other producers in Nevada with these facilities, and Getchell Gold will investigate terms for downstream processing. For this study, the recovery, transportation, and purchase terms are assumed to be equal to 10% of the gold contained in the concentrates.

19.1 Gold Market

Gold is the principal commodity at Fondaway Canyon and is freely traded in transparent markets worldwide. It is assumed that there will be a ready market for gold at market prices.

Due to the refractory nature of the Fondaway mineralized material, current plans are to produce a flotation concentrate for sale to one of several autoclave facilities (pressure oxidation) in Nevada. There are no current contracts for the transportation, processing, and refining of the gold produced, although the autoclaves are known to accept third party feed from other miners. The current transportation and processing of these concentrates is currently assumed at 10% of the contained gold. There are no issues anticipated in the ability to obtain contracts to sell the concentrate.

19.2 Price Assumptions

The base case gold price used for this report is US $2,250/toz, which is approximately the three-year trailing average. The spot market has held above $2,100 since early March 2024, and is currently trading in a range between $2,600 and $2,700 through the end of November 2024. Gold prices and averages are based on Kitco Metals daily close. Table 19-1 shows the trailing average gold price over several intervals as of January 1, 2025 (gold price in US $/toz).

Table 19-1: Trailing Average Gold Price

3-year	US $2,056
2-year	US $2,182

To give credence to current price trends, the price was based on 2/3 of a three year trailing average, and 1/3 of a bank consensus future forecast compiled on January 3, 2025 by Scottsdale Bullion & Coin (SBC), which is shown in Table 19-2. This analysis resulted in a forecast of US $2,287/toz Au, rounded to US $2,250/toz Au for this study.

Table 19-2: SBC Gold Price Consensus Forecast[1]

Bank	2024	2025	Time
Citibank	$2,100	**$3,000**	2025
Bank of America	$2,400	**$3,000**	By 2025
Commonwealth Bank	$2,800	**$3,000**	Q4 2025
Goldman Sachs	$2,700	**$3,000**	Early 2025
World Gold Council	–	**$3,000**	2025
ANZ	$2,394	**$2,900**	End of 2025
Société Générale (SocGen)	$2,460	**$2,800**	2025 (avg.)
ING	$2,150	**$2,700**	2025 (avg.)
TD Securities	$2,350	**$2,700**	2024
UBS	$2,500	**$2,700**	Mid-2025
BMI		**$2,700**	2024
J.P. Morgan	$2,500	**$2,600**	2024
Commerzbank	$2,200	**$2,600**	Mid-2025
World Bank Group	$1,900	**$2,350**	2024 (avg.)

[1] [1] https://www.sbcgold.com/blog/experts-boost-gold-price-forecasts-for-2024-2025-again/

20. ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

The Project status changed as of December 23rd, 2022, following the passage of the National Defense Authorization Act ("NDAA"), the Stillwater Wilderness Study Area was released. The Numunaa Nobe National Conservation Area ("NCA") (Figure 4-2) was established with a reduced footprint. The newly established boundaries of the NCA formalized in the NDAA opened additional areas for exploration and mining around the existing claim group.

Subsequent to the establishment of the NCA, Getchell Gold expanded the claim package through the staking of additional mining claims, the FCG group of claims, to the North, East, and South up to the boundary of the NCA. The NCA does not impact the mining claims, does not limit expansion to the mineral resource, and allows ample area in support of potential future mining activities such as those associated with an open pit operation envisioned herein.

Reclamation of the drill pads from the 2020-2022 exploration programs are still pending at the time of this report.

20.1 Permits

Exploration work, including drilling, is being carried out under an existing 5-acre Surface Management Notice disturbance permit (NVN95628). The reclamation bond is currently set at US $22,619.

No other permits currently exist, although as the Project has been mined previously, the QP believes that the Project can be permitted. The recommendations will include a budget to bring the permitting to the next step.

20.2 Environmental Liabilities

Several small historical open pit excavations exist on the Property along with some minor dumps and equipment. The Tenneco portal was closed during reclamation that took place in 1999-2000. All areas of the site successfully passed inspection and achieved bond release in 2002, with the exception of the heap leach pad area, which was released in the fall 2004.

21. CAPITAL AND OPERATING COSTS

The capital and operating costs used in this report were based on costs from similar project work performed recently by Forte, and by interpolation from CostMine™ models. The QP's believe that the estimates are appropriate for inclusion in this report. The QP believes that these costs comply with the precision requirements for a PEA.

21.1 Capital Cost Estimate

Capital costs for the mine and the plant were estimated by interpolating published data from CostMine™. Mine capital cost does not include the mobile fleet, as it is included in the contract mining cost. Capital cost includes both the construction and startup phases. The initial capital cost, which includes process, preproduction, and facilities, is estimated at US $226.47 million with a 20% contingency. There is no sustaining capital at this stage, as mining contractors are planned to be used for all major mining work.

Table 21-1 provides a detailed breakdown of initial capital costs for the Project.

Table 21-1: Project Capital Cost Summary

Category	US $M
Process Capital CostMine™ Model	$131.74
Preproduction and Facilities	$56.98
Summary CAPEX	$188.72
Contingency (20%)	$37.75
Total CAPEX	**$226.47**

21.1.1 Mine Equipment Costs

The ownership equipment costs for the open pit operation are included in the mining cost per tonne. Mining will be performed by a contractor, and no equipment purchases are necessary.

In-pit development and overburden removal is built into the production schedule. The project will need to build or improve access roads, maintenance and office area, and a tailings storage facility.

21.1.2 Mine Dewatering

There has been minimal water in the exploration drilling.

21.1.3 Tailings and Waste Rock Storage Facilities

Tailings and waste rock storage capital and operating costs are assumed to be within the initial capital estimate of the Project.

21.2 Operating Cost Estimate

Operating costs for the mine and the plant were estimated by interpolating published data from CostMine™.

Table 21-2 provides a detailed breakdown of operating costs for the Project.

Table 21-2: Project Operating Cost Summary

Operating Costs	$/tonne Mined	LOM (US $M)	$/oz Au Produced
Mining to Process	$ 3.54	$ 107.4	$ 87.2
Mining Waste	$ 3.54	$ 507.4	$ 412.1
Processing	$ 13.25	$ 402.0	$ 326.5
Mine Site G&A	$ 2.00	$ 60.7	$ 49.3
Total Operating Costs:		$ 1,077.5	$ 875.0
Transportation and Refining	$ 10.00	$ 303.4	$ 246.4
Royalties	3%	$ 83.0	$ 67.5
Total Cash Costs:		$ 1,464.0	$ 1,188.9

Labor costs for all project areas are determined by the number of employees and their annual burdened wages, sourced from CostMine™ models. Validation was conducted with actual cost data from a comparable operation and the previous experience of qualified professionals in the region. Staffing levels are aligned with the size of the equipment fleet or scaled from similar operations.

21.2.1 Mine Operating Costs

Open pit operating costs were developed based on production models from the CostMine™ references of Mining Cost Service. These costs were benchmarked against Forte's experience in Nevada.

21.2.2 Mineral Processing Costs

Based on Forte experience with similar froth flotation plants, the mineral processing cost is estimated at US $15.00/t of processed material. For economic analysis, an additional allowance of $0.50/t has been allocated for duties and pumping requirements for dust control and process water. Power cost is estimated at $0.09 per kilowatt-hour (kWh).

22. ECONOMIC ANALYSIS

The economic analysis of the Fondaway Canyon Project is based on the mining schedule, capital and operating costs, recovery parameters, and royalties outlined in earlier sections of this report. This is a Preliminary Economic Assessment (PEA) level analysis, which incorporates Inferred Resources in the economic model. The favorable economic results presented do not define a mineral reserve. While the economic parameters used in this technical report are considered reasonable, additional information could alter these assumptions and affect the analysis. All figures are expressed in constant 2024 US dollars.

22.1 Principal Assumptions

The mine will utilize surface production only as of the time of this report.

Mineral processing is planned at 8,000 t/d. The mine will be operated by contractors, and the plant by Getchell Gold personnel.

Table 22-1: Economic Model Parameters

Parameter	Value
Project Funding	100% Equity
Working Capital	$0
Discount Rate	10%
Contingency Capital Cost	20%
Gold Price	$2,250/toz

The Project contingency of 20% is considered reasonable for a PEA.

The model encompasses 1.0 year of production ramp-up with year 1 averaging 7.3k t/d, followed by 9 years at 8k t/d of mine production, ending with year 11 averaging 4k t/d. It is assumed that closure costs will be included in the initial bond estimate. A key input to the model is the mine schedule, detailed in Table 16-7: LOM Production Schedule, which outlines the grade and tonnage of the mined mineralized material. Revenue is derived from the amount of recovered metal, the specified metal price, and royalties incurred.

22.1.1 Working Capital

Working capital is assumed to be within the initial contingency at this stage of the Project.

22.2 Operating Cost

Mine and process operating costs were interpolated from published numbers contained in CostMine™, a publication updated annually with average pricing from mine operators in the USA and Canada. The QP believes that these are appropriate for this level of preliminary study.

22.2.1 Capital Costs

Mine capital costs were based on CostMine™ tables, but did not include the mobile fleet, as it will be provided by the contractor. Mine capital will include offices, warehouses, fuel and lubricant storage, powder magazines and maintenance facilities.

Mill capital was interpolated from CostMine™ for a single product flotation mill at 8,000 t/d production rate. The QP believes that these are appropriate for this level of preliminary study.

22.2.2 Tailings

The operation plans to produce dry stack tailings. Filtration costs are included in the mineral processing budget. An allowance of US $250,000 has been made for initial impoundment construction

22.2.3 General and Administrative

General and Administrative or overhead costs are the costs not directly incurred during production.

At the Project, no camp facility is required, and most overhead will be carried by the corporation, allowing a distribution of the costs between projects. It was estimated by Getchell Gold at $2.00/t of processed material.

22.2.4 Refining Costs

Treatment and refining costs are assumed from experience with similar projects. Transportation of concentrates to a pressure oxidation/leach facility and the costs of refining are assumed to total 10% of contained gold in the concentrate. Based on national averages for truck transportation, the estimated cost for the 250 mile transport to an autoclave in Carlin, NV is about $0.16/t-mi, or about $38.00/ton. The 10% payment would produce a total cost for transportation and refining of $100/ton of concentrate or $10/tonne of processed material, which the QP believes is a reasonable estimate to determine if the Project holds future economic potential.

22.3 Cost Summary

The costs used in the economic model are summarized in Table 22-2.

Table 22-2: Cost Summary

Prices	
Gold Price ($/toz)	$2,250
Initial Capital	$226.47M
Sustaining Capital	$0M
Project Life (Years)	10.5
Production	
Total Mined Processed Material (ktonnes)	30,343
Total Mined Waste (ktonnes)	143,392
Total Mined Gold (ktoz)	1,466
Au Grade (opst)	0.044
Au Grade (g/t)	1.50
Average Operating Costs	
Open Pit Mining Cost ($/t)	$3.54
Process Cost ($/t)	$13.25
Transportation and Refining Cost ($/t)	$10.00
Gen. & Admin. Cost ($/t processed material)	$2.00
NSR Royalty (%)	3.0

22.4 Discounted Cash Flow Model

A summary of the discounted cash flow (DCF) model is provided in Appendix B. Additionally, a high-level summary of the Pre-Tax Net Present Value (NPV) is provided in Table 22-3, while the summary After-Tax is included in Table 22-4. The Project is economically robust and has a positive return and short payback period of less than three years.

Table 22-3: Pre-Tax NPV Summary

Pre-Tax NPV	US $M
NPV @ 0%	$1,080.13
NPV @ 5%	$761.12
NPV @ 8%	$622.38
NPV @ 10%	$545.73
NPV @ 12%	$479.29
IRR	51.2%
Payback Period	3.1 years

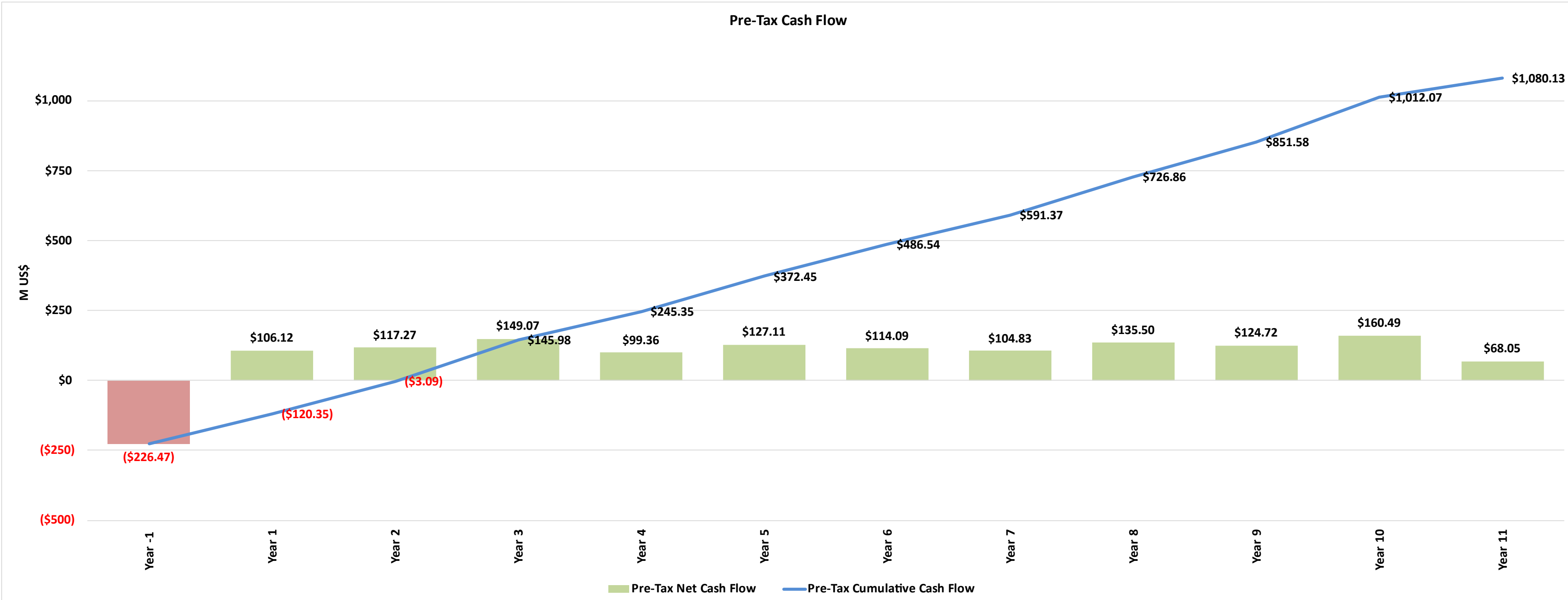


Figure 22-1: Pre-Tax Cash Flow

(Source: Forte Dynamics, Inc. 2024)

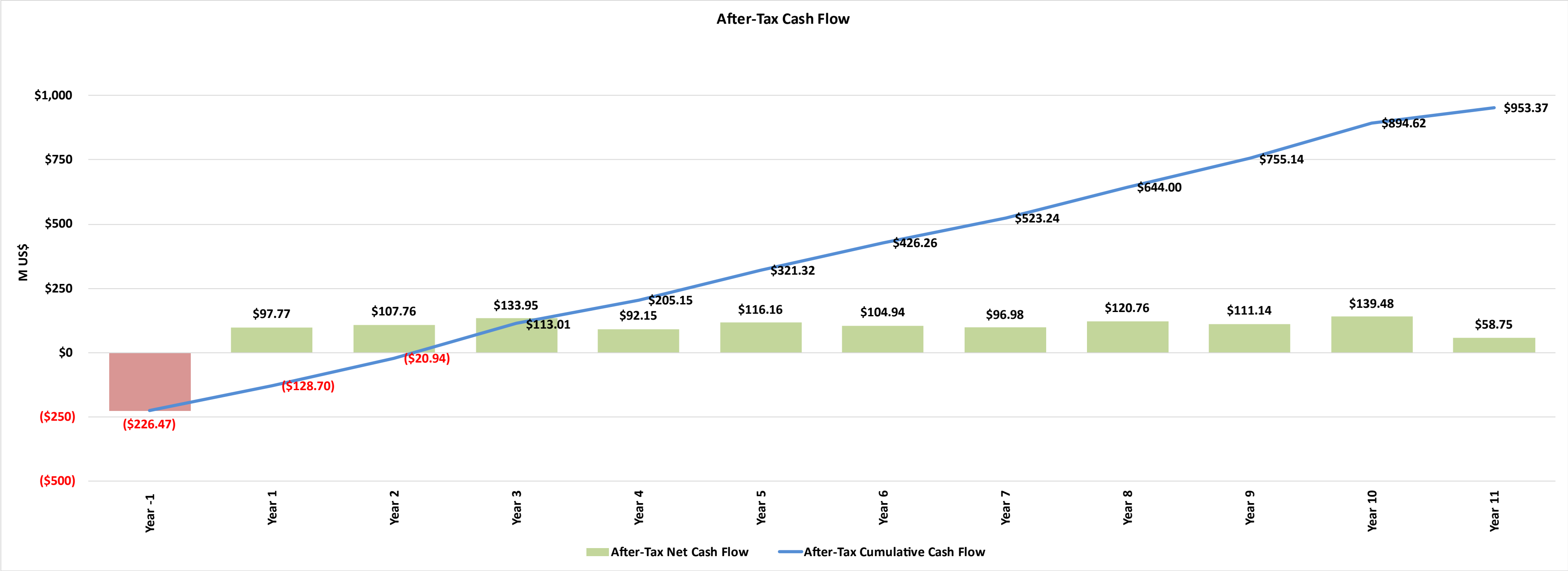


Figure 22-2: After-Tax Cash Flow

(Source: Forte Dynamics, Inc. 2024)

22.4.1 Taxes and Royalties

Royalties are discussed in detail in Section 4.3. Getchell Gold plans to buy out the remainder of the Fisk Royalty and, following that, buy out 1% of the Canagold 2% royalty. This leaves a 1% royalty to Canagold associated with purchase of the asset, and 2% Hale Capital. The buyout of these royalties is assumed included in the initial capital.

Taxes are calculated as required for a project in Nevada. A summary of the After-Tax NPV is included in Table 22-4. The Project will pay a total of US $92.5 million dollars in federal taxes and US $64.7 million dollars in state taxes during the life of mine.

Table 22-4: After-Tax NPV Summary

After-Tax NPV	US $M
NPV @ 0%	$953.37
NPV @ 5%	$667.51
NPV @ 8%	$542.93
NPV @ 10%	$474.01
NPV @ 12%	$414.23
IRR	46.7%
Payback Period	3.2 years

22.4.2 Sensitivity Analysis

A sensitivity analysis was conducted on the parameters of capital cost, operating cost, and metal price, all assessed on a pre-tax and after-tax basis. The sensitivity to the gold price for a variety of scenarios is shown in Table 22-5. Figure 22-3 and Figure 22-4 show the sensitivity of NPV and IRR pre-tax. Figure 22-5 and Figure 22-6 show the sensitivity of NPV and IRR after-tax.

Based on the economic sensitivity study, the Project is very robust regarding both capital and operating costs. It is most sensitive to metal price and by direct correlation, to metal recovery. Metal prices include gold only.

Table 22-5 Economic Sensitivity to Gold Price

Gold Price (US$/oz)	$2,000 (Low Case)	$2,250 (Base Case)	$2,500 (High Case)	$2,750 (Spot Price)
Pre-Tax $NPV_{10\%}$	$ 365 M	$ 546 M	$727 M	$ 908 M
Pre-Tax IRR	38.2 %	51.2 %	63.9 %	76.4 %
Pre-Tax Payback	3.5 years	3.1 years	2.6 years	2.4 years
After-Tax $NPV_{10\%}$	$ 322 M	$474 M	$ 618 M	$ 760 M
After-Tax IRR	35.5 %	46.7 %	57.0 %	66.9 %
After-Tax Payback	3.6 years	3.2 years	2.8 years	2.6 years

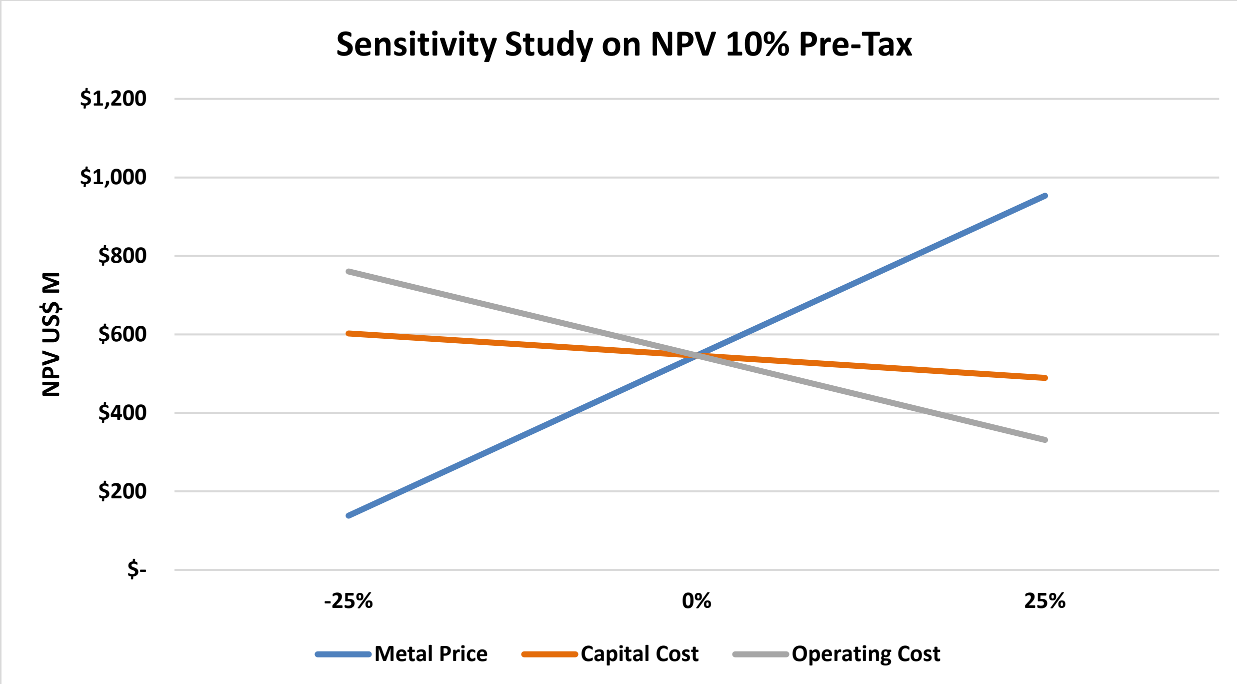


Figure 22-3: Sensitivity Study on NPV at 10% Pre-Tax

(Source: Forte Dynamics, Inc. 2024)

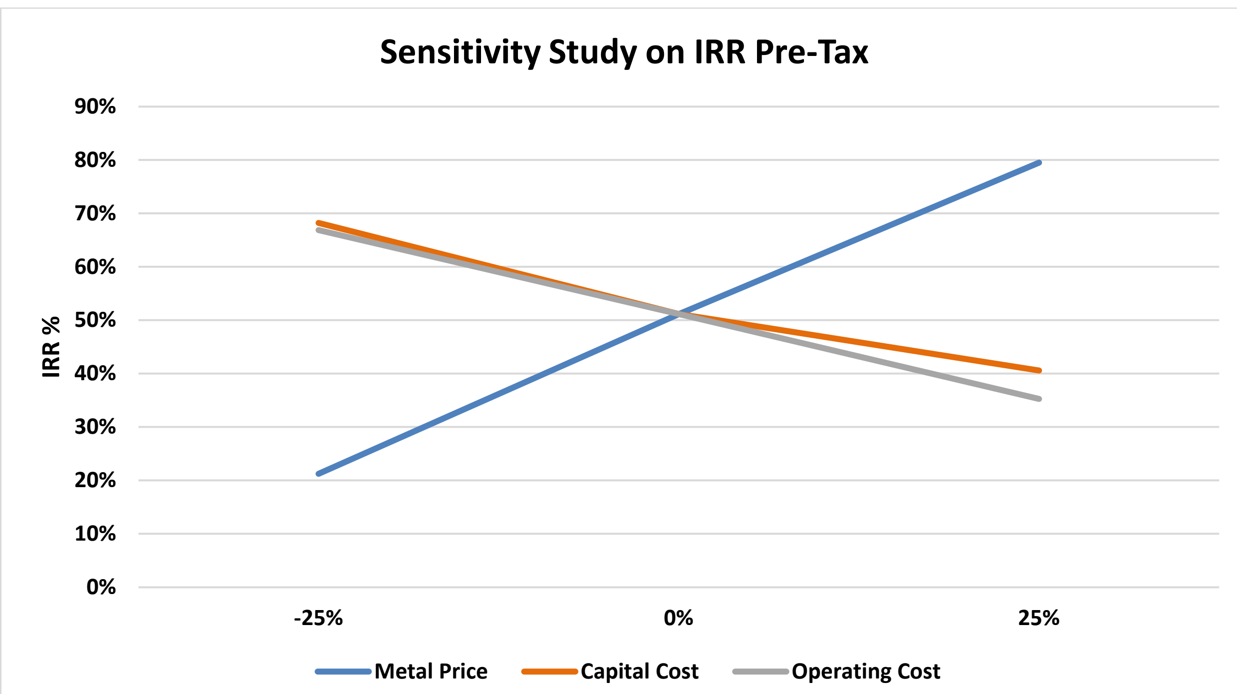


Figure 22-4: Sensitivity Study on IRR Pre-Tax

(Source: Forte Dynamics, Inc. 2024)

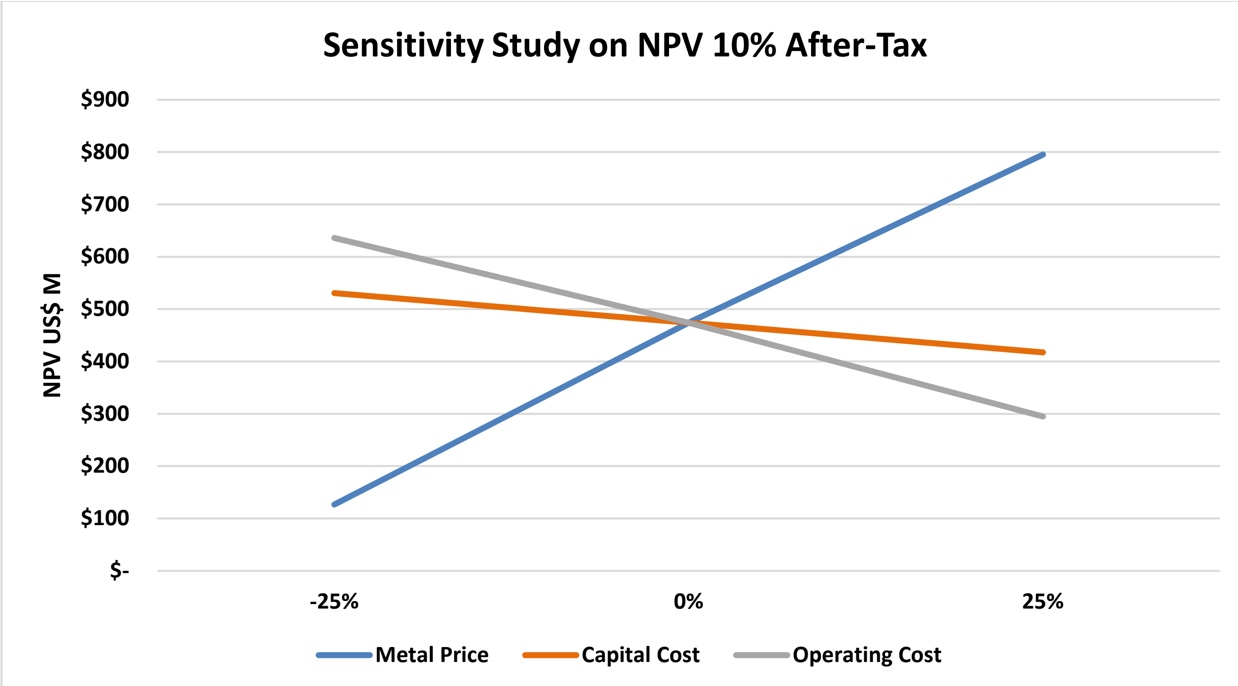


Figure 22-5: Sensitivity Study on NPV at 10% After-Tax

(Source: Forte Dynamics, Inc. 2024)

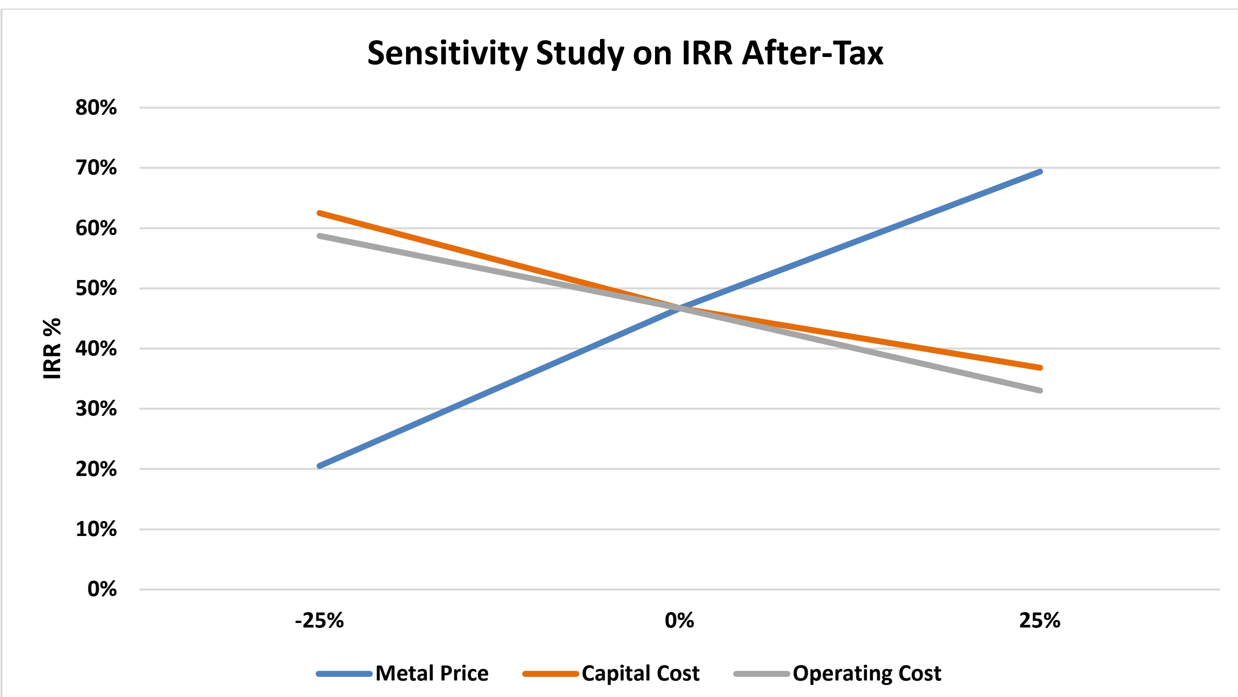


Figure 22-6: Sensitivity Study on IRR After-Tax

(Source: Forte Dynamics, Inc. 2024)

23. ADJACENT PROPERTIES

There are two privately held claims, the Skarn 1 and Skarn 2 claims, contiguous to the north of the claim block in a two-claim size indentation in the boundary right at/along the range front. There are a few small pits within the two claims from an iron ore operation which most recently mined out a small amount of magnetite / iron ore in 2024, utilizing the Fondaway Canyon access road that branches off the East County Road. These claims cover iron ore deposits which are unrelated to the Fondaway Canyon mineral resource and are not considered to have a material impact on the project.

24. OTHER RELEVANT DATA AND INFORMATION

The QP is not aware of any additional relevant data or information.

25. INTERPRETATION AND CONCLUSIONS

25.1 Conclusions

Based on the estimated quantity of mineral resources with economic potential for an open pit operation, the Fondaway Canyon Project is economically robust with an 8,000 t/d operation and a 10.5 year mine life. The discounted cashflow economic analysis returns a pre-tax NPV of US $545.7 million, and an after-tax NPV of US $474.0 million at a discount rate of 10% with an initial capital investment of US $226.5 million.

Getchell Gold has a clear title to the Fondaway Project, including a significant database of technical information, drill data, geologic interpretation, and preliminary metallurgical data. The data are of industry standard quality and are suitable to be used for resource estimation and future work for the Project.

Their interpretations of the Project as a surface mineable producer of flotation concentrate have overcome issues of refractory gold and attempting to pursue high grade underground targets within the system. This is of course enhanced by the shift in gold prices since 2020.

The 2020 drill program provided confirmation for the geological model. The 2021-2022 drill programs continued to delineate the mineralization. All drilling programs from 2020-2022, completed by Getchell Gold, assisted in providing confirmation of the historical drilling database along with yielding greater confidence in the Mineral Resource Estimate, as well as enhancing the understanding of the mineralized and non-mineralized contacts.

The Fondaway Canyon Project contains a significant gold resource with good continuity at relatively low cutoff grades, and significant contribution from higher-grade zones. The resource as reported is contained within an economic pit shell and appears to be amenable to open pit mining methods. Due to the complex geometry of the canyon, the pushbacks, designed to provide robust economic returns, were explicitly designed to provide economic confidence in the early production years, and to assure the potential of successfully pre-stripping successive pushbacks.

Initial metallurgical test work confirms that the deposit is refractory for cyanidation; however, as much of the gold is associated with pyrite and other sulfides, froth flotation shows the potential to create a high-grade gold bearing sulfide concentrate which could be processed via pressure oxidation to achieve economic recoveries. Preliminary scoping studies indicate that deleterious elements are not in sufficient quantity to negatively impact the sale of concentrates and should be readily marketable. Additional test work is needed to refine the preliminary conclusions.

The PEA indicates that at the gold prices considered, the Project shows potential to be developed as an open pit surface mining operation. A sensitivity study executed at near-spot prices indicates additional potential for the deposit at higher metal prices. The QP's believe that before proceeding to a potential next phase Pre-Feasibility Study (PFS), it would be beneficial to complete additional step out drilling which may increase the mineable mineral resource, infill drilling to increase the confidence in the resource, and appropriate test work to refine the metallurgical assumptions and process flow sheet.

25.2 Risks and Uncertainties

The Fondaway Canyon Project is subject to the risks and uncertainties typical of gold projects, particularly risk in commodity prices and the precious metals equity markets. Lower metal 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 confirmation of transitional grades around structural zones, that material used for metallurgical testing is not representative of the overall deposit, and with the handling of water inflows to the pit, and/or of adequate availability of water for the mill operation.

Each of these risks appear to be manageable; however, there is potential to increase the operating or capital cost for the Project, or delay development activities.

The life of mine (LOM) plan includes a majority percentage of Inferred Mineral resources, compared to the amount of Indicated Mineral resources (there are no Measured Mineral resources). The current mineable resource demonstrates economic viability but will need to be upgraded to become a mineral reserve.

Metallurgical data appears to be of reasonable quality but is considered preliminary. Incomplete classification of material types or misunderstanding of the representativeness of metallurgical samples could lead to a change in recovery or process cost assumptions. Further test work is 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 the Project economics.

25.3 Opportunities

There is potential to increase the Project mineral resource inventory. The mineralized areas of Fondaway Canyon are open along strike and down dip, and there are zones within the pit design developed in this study that did not have sufficient data to be classified as a mineral resource. This offers a path to increasing potentially economic mineral resources along with lowering the stripping ratio. Upgrading the classification of Inferred Mineral Resources to Indicated and/or Measured Mineral Resources would improve confidence in the mineral resource inventory and may have potential to increase the mineable resources. There are also zones of higher-grade material which may be amenable to underground exploitation if they can be connected and/or expanded.

Optimization of the operation of the flotation plant will offer opportunities to produce a more marketable concentrate, improving downstream revenues and reducing downstream costs.

26. RECOMMENDATIONS

The PEA has highlighted the potential of an open pit surface operation with a flotation concentrator to produce a gold concentrate for further treatment. The Fondaway Canyon Project is robust and demonstrates positive returns over a range of prices and costs. The discounted cashflow economic analysis returns a pre-tax NPV of US $545.7 million, and an after-tax NPV of US $474.0 million at a discount rate of 10% with an initial capital investment of US $226.5 million.

Based on the positive economic results from this PEA, there are several steps that the QP's feel should be taken that could progress the Project and/or prior to proceeding to a potential Pre-Feasibility Study (PFS), that can better define the overall potential of the Project.

26.1 Mineral Resource Expansion and Exploration

There is potential to increase the Project mineral resource inventory. The mineralized areas of Fondaway Canyon are open along strike and down dip beyond the designed pit limits, and there are zones within the pit design developed in this study that did not have sufficient data to be classified as a mineral resource.

There is also potential to upgrade mineral resources from Inferred to Indicated and Indicated to Measured, which will improve resource confidence and increase the potential mineable resource inventory and the potential for an economic mineral reserve estimate.

There are several areas within the Fondaway Canyon Project that the Company believes warrant further exploration. In addition to resource definition within the designed pit limits, there is potential to expand the current modeled mineralized zones to the west for the Mid Realm and South Mouth areas, and to the east for the Silicon Ridge and Hamburger Hill areas.

26.2 Geological Model and Resource Domains

Review input data of geological, structural, and overall mineralization controls to refine the domain definitions for the Mineral Resource Estimate. The addition of structural data through drilling (see Table 26-1 below) could improve the understanding of structural controls on mineralization (and geology) and enhance the confidence in grade estimation and continuity, which could improve future mineral resource estimates.

26.3 Geotechnical Drilling

Specific geotechnical drilling and analysis of the pit highwalls is recommended to better understand the fracture behavior and rock strength characteristics and de-risk in-pit safety concerns.

26.4 Metallurgical Test Work

Additional metallurgical test work is recommended to provide greater confidence for input cost parameters, recovery, crush sizes for optimal extraction, and subsequent processing details. Flotation work on grind sizes and reagent consumption may improve recovery and increase concentrate grade with potential benefits to the Project economics.

26.5 Market Potential of Concentrates

The QP's recommend initial discussions with potential buyers of concentrates to gain a better understanding of the current and future market conditions.

26.6 Recommended Work Programs

A single-phase work program is recommended. The focus of the work program will be to enhance the confidence and potentially expand the current Mineral Resource Estimate. This could further outline the overall shape and orientation of the resource, and based on the results of this phase, additional drilling may be warranted. Additional metallurgical test work and other studies may be needed to further de-risk the Project.

Table 26-1: Recommended Work Programs

Budget Item	Estimated Cost
Resource Definition & Expansion Drilling	$2,000,000
Metallurgical Test Work & ARD	$125,000
Geotechnical Drilling	$100,000
Total	**$2,225,000**

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28. CERTIFICATES OF QUALIFIED PERSONS

CERTIFICATE OF QUALIFIED PERSON

Donald E. Hulse P.E. SME-RM
Director of Mining Resources - Forte Dynamics, Inc.
120 Commerce Drive, Units 3 & 4
Fort Collins, CO 80524
Email: dhulse@fortedynamics.com

This certificate applies to the report entitled: "Preliminary Economic Assessment of the Getchell Gold Corp. Fondaway Canyon Project Nevada, USA", effective date January 15, 2025.

I, **Donald E. Hulse**, do hereby certify that:

1) I am the Director of Mining Resources for Forte Dynamics, Inc., with a business address of 120 Commerce Drive, Units 3-4, Fort Collins, CO 80524.
2) I graduated with a degree in Mining Engineering, Bachelor of Science in 1982 from the Colorado School of Mines in Golden, Colorado. I have worked as a mining engineer for 42 years with specific expertise in mine design, mine strategic planning, mineral resource estimation in a variety of deposits. I am a Registered Member of the Society of Mining Engineers, and a Professional Engineer in the State of Colorado, USA.
3) I have read the definition of "qualified person" set out in National Instrument 43-101- Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" within the meaning of NI 43-101.
4) I have personally inspected the property that is a subject of this Mineral Resource Estimate in September 2024.
5) I am the QP responsible for Sections 1-6, 18-20, 22-27, and a contributor to the overall content of this report.
6) I am independent of the issuer, Getchell Gold Corp., according to Section 1.5 of NI 43-101.
7) I have had no prior involvement with the property that is the subject of the Technical Report.
8) I have read NI 43-101, Form 43-101 F1 -Technical Report, 43-101 CP-Standards of Disclosure for Mineral Projects, and confirm that the Technical Report has been prepared in compliance with such instrument, form, and companion policy.
9) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10) I consent to the filing of the Technical Report with any securities regulatory authority, stock exchange and other regulatory authority and any publications by them, including electronic publication in the public company files on their websites accessible by the public.

Dated this 6th day of February 2025.

//s// Donald E. Hulse

Donald E. Hulse P.E., SME-RM

CERTIFICATE OF QUALIFIED PERSON

Jonathan R. Heiner, SME-RM
Director of Mining - Forte Dynamics, Inc.
120 Commerce Drive, Units 3 & 4
Fort Collins, CO 80524
Email: jheiner@fortedynamics.com

This certificate applies to the report entitled: “Preliminary Economic Assessment of the Getchell Gold Corp. Fondaway Canyon Project Nevada, USA”, effective date January 15, 2025.

I, **Jonathan R. Heiner**, do hereby certify that:

1) I am the Director of Mining for Forte Dynamics, Inc., with a business address of 120 Commerce Drive, Units 3-4, Fort Collins, CO 80524.
2) I graduated with a Bachelor of Science in Mining Engineering in 2009 from the University of Utah in Salt Lake City UT. and with a Professional Masters in Explosive Engineering in 2018 from Missouri Science and Technical Institute in Rolla MO. I have worked as a mining engineer for 15 years with specific expertise in open pit design and scheduling and blasting. I am a Registered Member of the Society of Mining Engineers, and a Professional Engineer licensed in Utah USA.
3) I have read the definition of "qualified person" set out in National Instrument 43-101- Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" within the meaning of NI 43-101.
4) I personally inspected the property that is a subject of this Mineral Resource Estimate in September 2024.
5) I am the QP responsible for Sections 15, 16, and 21, and a contributor to the overall content of this report.
6) I am independent of the issuer, Getchell Gold Corp., according to Section 1.5 of NI 43-101.
7) I have had no prior involvement with the property that is the subject of the Technical Report.
8) I have read NI 43-101, Form 43-101 F1 -Technical Report, 43-101 CP-Standards of Disclosure for Mineral Projects, and confirm that the Technical Report has been prepared in compliance with such instrument, form, and companion policy.
9) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10) I consent to the filing of the Technical Report with any securities regulatory authority, stock exchange and other regulatory authority and any publications by them, including electronic publication in the public company files on their websites accessible by the public.

Dated this 6th day of February 2025.

//s// Jonathan R. Heiner

Jonathan R. Heiner, SME-RM

CERTIFICATE OF QUALIFIED PERSON

Deepak Malhotra, Ph.D., SME-RM
Director of Metallurgy - Forte Dynamics, Inc.
120 Commerce Drive, Units 3 & 4
Fort Collins, CO 80524
Email: dmalhotra@fortedynamics.com

This certificate applies to the report entitled: "Preliminary Economic Assessment of the Getchell Gold Corp. Fondaway Canyon Project Nevada, USA", effective date January 15, 2025.

I, **Deepak Malhotra**, do hereby certify that:

1) I am the Director of Metallurgy for Forte Dynamics, Inc., with a business address of 120 Commerce Drive, Units 3-4, Fort Collins, CO 80524.
2) I graduated with a degree in Metallurgical Engineering, Master of Science in 1973 from the Colorado School of Mines in Golden, Colorado. In addition, I graduated with a degree in Mineral Economics, Ph.D. in 1978 from the Colorado School of Mines in Golden, Colorado. My relevant experience includes working as a metallurgist and mineral economist for 50+ years since my graduation with specific expertise in mineral processing, metallurgical testing, and recovery methods. I am a member of the Society of Mining Engineers.
3) I have read the definition of "qualified person" set out in National Instrument 43-101- Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" within the meaning of NI 43-101.
4) I have not personally inspected the property that is a subject of this Mineral Resource Estimate.
5) I am the QP responsible for Section 13 and 17.
6) I am independent of the issuer, Getchell Gold Corp., according to Section 1.5 of NI 43-101.
7) I have had no prior involvement with the property that is the subject of the Technical Report.
8) I have read NI 43-101, Form 43-101 F1 -Technical Report, 43-101 CP-Standards of Disclosure for Mineral Projects, and confirm that the Technical Report has been prepared in compliance with such instrument, form, and companion policy.
9) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10) I consent to the filing of the Technical Report with any securities regulatory authority, stock exchange and other regulatory authority and any publications by them, including electronic publication in the public company files on their websites accessible by the public.

Dated this 6th day of February 2025.

//s// Deepak Malhotra

____

Deepak Malhotra, Ph.D., SME-RM

CERTIFICATE OF QUALIFIED PERSON

Mike Dufresne, M.Sc., P.Geo., P.Geol.
President - APEX Geoscience Ltd.
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Email: mdufresne@apexgeoscience.com

This certificate applies to the report entitled: "Preliminary Economic Assessment of the Getchell Gold Corp. Fondaway Canyon Project Nevada, USA", effective date January 15, 2025.

I, **Mike Dufresne**, do hereby certify that:

1) I am President and a Principal of APEX Geoscience Ltd. ("APEX"), with a business address of 100, 11450 – 160 St. NW, Edmonton, Alberta, T5M 3Y7, Canada.
2) I am the QP responsible for Sections 7-12, 14, and 25-27 of this Technical Report.
3) I graduated with a B.Sc. Degree in Geology from the University of North Carolina at Wilmington in 1983 and a M.Sc. Degree in Economic Geology from the University of Alberta in 1987. I have worked as a geologist for more than 40 years since my graduation from university and have been involved in all aspects of mineral exploration and mineral resource estimations for precious and base metal mineral projects and deposits in Canada and internationally.
4) I am and have been registered as a Professional Geologist with the Association of Professional Engineers and Geoscientists ("APEGA") of Alberta since 1989 and a Professional Geoscientist with the Association of Professional Engineers and Geoscientists ("EGBC") of British Columbia since 2012. I am a 'Qualified Person' in relation to the subject matter of this Technical Report.
5) I personally inspected the property that is a subject of this Technical Report May 7th and 8th, 2022.
6) I am independent of the property and the issuer, Getchell Gold Corp., according to Section 1.5 of NI 43-101. I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Company. I am not aware of any other information or circumstance that could interfere with my judgment regarding the preparation of the Technical Report.
7) I have had previous involvement with the Property that is the subject of the Technical Report. In 2023, I co-authored an NI 43-101 Technical Report written on behalf of Getchell Gold Corp. for the Fondaway Canyon Project. The published reference related to this work is included in Section 27, References (see Dufresne et al., 2023).
8) I have read and understand National Instrument 43-101 and Form 43-101F1 and the Report has been prepared in compliance with the instrument.
9) 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.
10) I consent to the filing of the Technical Report with any securities regulatory authority, stock exchange and other regulatory authority and any publications by them, including electronic publication in the public company files on their websites accessible by the public.

Dated this 6th day of February 2025.

//s// Mike Dufresne

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APPENDIX A – PRELIMINARY METALLURGICAL REPORT

Getchell Gold Corp.

Scoping Level Metallurgical Testing of Fondaway Canyon Gold Ores

Project No. 225-23041

Prepared by:

FORTE ANALYTICAL, LLC
120 Commerce Drive
Units 3 & 4
Fort Collins, CO 80524

Revision	Date	Status	Prepared By	Checked By	Approved By
REV 0	10/9/2024	Issued for Review	D.Malhotra		

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1. INTRODUCTION

Getchell Gold Corp. contracted Forte Dynamics / Forte Analytical to complete a scoping level metallurgical study for the Fondaway Canyon Project with the primary objective to develop a conceptual process flowsheet for the sulfide samples that minimizes both CAPEX and OPEX. The test program was expanded to include processing of oxide ore which occurs on the surface of the sulfide deposit.

This progress report discusses the test procedures and results obtained to date for the on-going test program.

2. SAMPLE PREPARATION AND HEAD ANALYSES

Forte Analytical received approximately 180 kgs of average grade sulfide and ± 75 kgs each of high grade and low grade sulfide analytical rejects and core for the study.

Two shipments of oxide analytical rejects and core consisting of ± 20 kgs and ± 50 kgs were received for the testing of oxide samples.

There are a total of five composites, which are listed in Table 1. The description of the samples constituting these composites are given in Appendix A.

Table 1: Description of Five Composites for Metallurgical Test Work

Composite	Description
1	Average Grade Sulfide
2	Low Grade Sulfide
3	High Grade Sulfide
4	Oxide Composite 1 (Analytical Rejects)
5	Oxide Composite 2 (Core)

The samples were stage crushed to 100% passing 6 mesh, blended and split into 1 kg and 10 kg charges. One 1 kg charge was split and a portion was pulverized for head analyses. A portion of the average grade composite was sent for mineralogical study.

The test data are given in Appendix B and the results are presented in Table 2 through Table 4. The test results indicate the following:

- The average grade composite assayed 1.49 g/t Au, 1.60 g/t Ag, 2.58% S_{Sulfur}, and 0.33% $C_{Organic}$.
- The average grade sample contained 1775 ppm As and 3.576% Fe, whereas As assays of low and high grade composites were 2548 ppm and 1666 ppm, respectively.
- The high grade composite assayed 4.93 g/t Au and the low grade composite assayed 0.53 g/t Au. The low grade sample contained 0.33% of organic carbon.
- The oxide composites assayed 1.5 g/t Au and 1.86 g/t Au.
- None of the samples contained an economic quantity of silver.

Table 2: Head Analyses of Sulfide Samples

Assay	Composite		
	Average Grade	Low Grade	High Grade
Au, g/t	1.49	0.53	4.93
Ag, g/t	1.60	1.0	1.10
S_{Total}, %	2.66	0.13	0.04
$S_{Sulfide}$, %	2.58	0.05	0.01
$S_{Sulfate}$, %	0.08	0.08	0.03
C_{Total}, %	1.74	0.59	0.11
$C_{Organic}$, %	0.33	0.33	0.05
$C_{Inorganic}$, %	1.40	0.26	0.06

Table 3: ICP Analyses of Sulfide and Oxide Composites

Element	Average Grade	Low Grade	High Grade	Oxide 1 (Comp #4)	Oxide 2 (Comp #5)
As	1755	2548	1630	BD	577
Ba	2444	1666	1497	47	786
Ca	30266	21379	10853	BD	25026
Cr	89.3	72.7	102	BD	91
Fe	35761	35614	19968	BD	25523
K	16258	25133	16129	90	17228
Mg	11548	9033	6238	BD	5700
Mn	379	345	226	BD	602
Na	482	BD	BD	BD	1039
Ni	24.2	27.6	15.2	BD	23
P	497	567	311	BD	52.2
Pb	28.9	27.5	20.8	BD	29
S	24580	28556	20057	BD	4850
Sb	40.66	BD	33.1	24	BD
Sr	214	213	106	BD	180
Ti	3175	2452	2084	49	2397
U	142	142	106	BD	125
V	75.3	74.6	52	BD	102
Zn	70.5	66.6	36.2	BD	81

Note: BD – Below Detection

Table 4: Head Analyses of Oxide Composites

	Composite	
	Composite 4	Composite 5 (Core)
Au, g/t	1.38	1.86
Ag, g/t	1.1	1.8
STotal, %	0.02	0.05
SSulfide, %	0.02	0.0
SSulfate, %	0.01	0.04
CTotal, %	0.85	1.0
COrg, %	0.21	0.20
Cinorg,%	0.64	0.80

3. MINERALOGICAL EVALUATION OF SULFIDE COMPOSITES

The mineralogy study was undertaken to determine the major minerals in the ore and liberation characteristics of gold particles. The mineralogy report is given in Appendix C. The highlights of the study indicated the following:

- The major minerals in the ore are quartz and orthoclase with minor amounts of pyrite, muscovite, ankerite, dolomite, and calcite.
- Gold particles observed in average and low grade composites were approximately 5 microns and were associated with pyrite.
- Two free particles at approximately 5 microns in size were identified in the high grade composite.

4. COMMINUTION

Bond's ball mill work indices were determined at a P_{80} of 100 mesh for the composite samples except for Oxide Composite 1. The data is given in Appendix D and summarized in Table 5.

Table 5: Bond's Ball Mill Work Indices for Composite Samples

Composite	BWi (Kwh/st)
Average Grade Sulfide	15.54
Low Grade Sulfide	15.82
High Grade Sulfide	15.46
Oxide Composite 2	13.62

The comminution data indicates that the oxide ore has an average hardness whereas the sulfide ores can be designated as slightly hard ores.

5. DIAGNOSTIC LEACH (GOLD DEPORTMENT) OF AVERAGE GRADE SULFIDE ORE

A series of sequential leach tests were performed with intermediate roasting steps to determine the association of gold with various minerals (i.e., free milling, associated with pyrite, arsenopyrite, etc.). The test flowsheet is given in Figure 1 and the results are summarized in Table 6. The test data are given in Appendix E.

Table 6: Deportment of Gold in Average Grade Sulfide Composite

Composite	Feed g/t Au	% Extraction Au			
		Free Milling	Arsenopyrite Association	Pyrite Association	Silica Encapsulation
Average Grade Sulfide	1.55	1.0	5.8	77.5	15.7

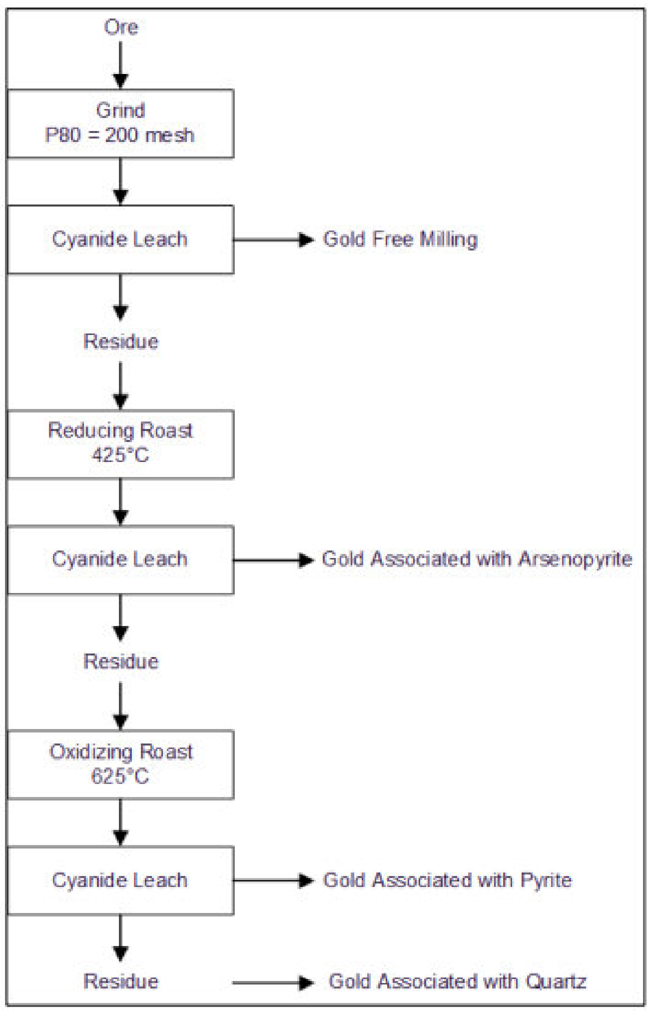

Figure 1: Diagnostic Test Procedure for Deportment of Gold

The test results indicate the following:

- The ore is refractory with 1% of the gold leching in the direct cyanidation process.
- Only 5.8% of the gold is associated with arsenopyrite.
- A majority of the gold is associated with pyrite (77.5%).
- Approximately 15.7% of the gold is encapsulated in silica.

These results correlate with the mineralogy which has indicated gold association with pyrite and being extremely fine (± 5 microns) which will require fine grind to expose it to cyanide for leaching.

6. GRIND STUDY

The test work was initiated on average grade sulfide ore with the objective of determining the techno-economically viable process flowsheet.

The sulfide composites and oxide 2 composite were ground in a laboratory rod mill which simulates a ball mill-cyclone circuit in an actual operation. Several grinding tests for varying grind times were performed to determine the relationship between grind time and grind size. The data are given in Appendix F.

The grind times were determined for achieving P_{80} of 100, 150, 200, and 270 mesh.

7. FLOTATION TESTS

7.1 Average Grade Composite

A series of flotation tests were performed using the average grade composite to determine the optimum grind size, flotation time, and reagent dosages to maximize gold recovery in the concentrate. The reagent suite consisted of potassium amyl xanthate, Aeropromotor 404, and frothers MIBC and AF65. These reagent combination tests to float both sulfides and gold.

The test data is given in Appendix G and the results are summarized in Table 7.

Table 7: Flotation Test Results for Average Grade Composites

Test No.	Grind Size, P_{80} Mesh	Rougher Recovery %		Grade, g/t Au		
		Wt	Au	Concentrate	Tailing	Feed
1	100	14.7	73.8	7.85	0.48	1.57
2	150	17.5	76.9	6.69	0.43	1.53
3	200	19.4	81.6	7.40	0.41	1.79
4	270	17.9	80.0	6.58	0.36	1.48
6	200	21.3	84.9	6.40	0.31	1.60
7	270	26.8	87.3	4.92	0.26	1.51

Note: Flotation Time = 12 min.

The test results indicated the following:

- The finer the grind, the higher the gold recovery. Approximately 81.6% of gold was recovered in 19.4% of the weight. The concentrate assayed 7.40 g/t Au.

- The tailing assay for P_{80} of 270 mesh was lower than that for 200 mesh (0.36 g/t vs. 0.41 g/t Au) though the recovery was only 80%. This was due to lower calculated feed grade.

The flotation tailing from Test 1 (P_{80} of 100 mesh) was subjected to gravity concentration using a Knelson concentrator. The test data is given in Appendix G and the results are summarized in Table 8.

Table 8: Gravity Concentration of Flotation Tailing

Product	Recovery %		Grade, g/t Au
	Wt	Au	
Gemeni Concentrate	0.5	6.5	3.57
Gemeni Tail	5.7	27.1	1.22
Cal. Knelson Concentrate	6.2	33.6	1.41
Knelson Tails	93.8	66.4	0.18
Cal. Feed	100	100	0.25

The test results indicate that one could get 33.6% of the gold lost to the flotation tailing by gravity. This would increase the flotation plus gravity recovery from 80% to ± 88%.

7.2 10 kg Rougher Flotation Test

A 10 kg rougher flotation test was performed at a primary grind of P_{80} of 270 mesh to generate a concentrate for cyanide leaching to recover gold. The flotation time and reagent additions were scaled up for larger flotation test.

The test data is given are Appendix G and the results are summarized in Table 9.

Table 9: Flotation Test Results for One-Cubic Foot Flotation Cell (10 kg Charge, Test 5)

Product	Recovery %		Grade, g/t Au
	Wt	Au	
Rougher Concentrate	20.7	89.8	6.07
Rougher Tail	79.3	10.2	0.18
Cal. Feed	100	100	1.40

The test results indicated that **89.8% of the gold was recovered in a concentrate assaying 6.07 g/t Au**.

The tailing from the test assayed 0.18 g/t Au. One kg of the tailing was taken and floated for additional time to evaluate if one could recover additional gold.

The test data is given in Appendix G and the results are summarized in Table 10.

Table 10: Effect of Additional Flotation Time on Gold Recovery (Test 5, Scavenger Float)

Product	Flotation Time, min	Recovery %		Grade, g/t Au
		Wt	Au	
Scavenger Conc. 1	3	7.8	32.4	0.76
Scavenger Conc. 2	3	1.1	3.3	0.55
Scavenger Conc. 3	3	1.0	0.7	0.12
Scavenger Tail		90.1	63.6	0.18
Cal. Scavenger Feed		100	100	0.18

The test results indicate that ore could recovery approximately 32.4% of gold in additional 7.8% of weight. The concentrate assayed 0.76 g/t Au.

A portion of the bulk concentrate generated in the flotation test was analyzed by XRD to determine the major minerals. The data is given in Appendix G and summarized in Table 11.

Table 11: XRD Analyses of Bulk Concentrate

Mineral	Approx. Weight %
Quartz	37
Mica / Illite	35
Kaolinite	<3
Dolomite	7
Calcite	<2
Rutile	<1
Pyrite	13
Arsenopyrite	<2
Unidentified	<5

The results indicated that the major minerals in the concentrate were quartz, mica/illite, pyrite, and dolomite.

Two additional flotation tests were performed at P_{80} of 200 and 270 mesh. Higher weight pull resulted in higher gold recovery. Approximately 87.3% of gold was recovered when weight recovery was 26.8%. The larger scale flotation test recovered 89.8% of gold in 20.7% of the weight.

These results indicate that one needs to recover ± 20% of weight to get 87% - 88% of gold recovery.

7.3 Low-Grade Composite

Rougher flotation tests were performed with low-grade composite using the optimum process parameters developed for average-grade composite.

The test data are given in Appendix G and the results are summarized in Table 12.

Table 12: Flotation Test Results for Low-Grade Composite

Test No.	Grind Size, P80 mesh	Rougher Recovery %		Grade, g/t Au		
		Wt	Au	Concentrate	Tailing	Feed
8	200	23.4	79.1	1.74	0.14	0.51
9	270	20.6	77.0	2.01	0.16	0.54

The test results indicate the following:

- Finer grind did not improve gold recovery. One needed to float ± 20% of weight in order to get gold recovery in the high 70s.

7.4 High-Grade Composite

Rougher flotation tests were also performed on high-grade composite at P_{80} of 200 and 270 mesh.

The test data are given in Appendix G and the results are summarized in Table 13.

Table 13: Flotation Test Results for High-Grade Composite

Test No.	Grind Size, P80 mesh	Rougher Recovery %		Grade, g/t Au		
		Wt	Au	Concentrate	Tailing	Feed
10	200	20.7	85.3	22.4	1.01	5.45
11	270	22.2	85.8	21.2	1.00	5.47

The test results indicate the following:

- Gold recovery was independent of particle size. A grind of 200 mesh appears to be optimum.
- Weight recovery of ± 20% was required to achieve ± 85% of gold.

7.5 Production of Rougher Concentrate for Leaching Tests

Rougher flotation tests were performed for the three sulfide composites to generate rougher concentrate for cyanidation leach tests.

Average grade concentrate was produced in a one-cubic flotation machine using 10 kg ore, whereas three 1 kg tests each were run for the other two composites.

The test data are given in Appendix G and the results are summarized in Table 14.

Table 14: Flotation Tests to Generate Concentrate for Leaching (P_{80} = 270 mesh)

Test No.	Test Type	Rougher Recovery %		Grade, g/t Au		
		Wt	Au	Concentrate	Tailing	Feed
12	Avg. Grade, 10kg	20.7	81.1	6.23	0.38	1.59
13	Low Grade 3 1kg Tests	26.1	78.2	1.62	0.16	0.54
14	High Grade 3 1kg Tests	26.4	85.4	17.2	1.05	5.31

The results were similar to those obtained for the composites in earlier tests except for the average-grade composite. The recovery was ± 8% lower due to higher tailing assay (i.e. 0.38 vs. 0.18 g/t Au).

8. WHOLE ORE LEACHING (WOL)

The whole ore leaching tests were performed for the oxide and sulfide composites. The ore was ground to P_{80} of 100 and 200 mesh and leached for 48 hours at 40% solids with varying cyanide concentration (1 to 2 g/L NaCN).

The test data are given in Appendix H and the results are summarized in Tables 15 and 16. The assay-by-size data for oxide ore is given in Appendix I.

Table 15: Whole Ore Leach of Oxide Composites

Parameter	Test No					
	4	5	6	7	12	13
Sample	Comp #4	Comp #4	Comp #5	Comp #5	Comp #4	Comp #5
Grind, P_{80} mesh	100	200	100	200	270	270
Extraction, % Au						
2 hr	50.3	51.4	57.9	62.2		
4 hr	50.3	50.9	55.9	61.6		
8 hr	48.8	48.6	55.2	60.0		
24 hr	44.3	42.7	50.6	52.9	62.1	71.6
48 hr	41.2	37.6	48.3	49.1		
Residue, g/t Au	0.79	0.82	1.03	1.00		
Cal. Feed, g/t Au	1.34	1.31	1.99	1.96		
Consumption, kg/t						
Lime	2.903	2.699	3.116	3.007	2.583	2.562
NaCN	0.363	0.422	0.419	0.842	1.025	0.955

Note: Tests 12 & 13 are CIL

Table 16: Whole Ore Leach of Sulfide Composites

Parameter	Test No		
	9	10	11
Sample	Average Grade	Low Grade	High Grade
Grind, P_{80} mesh	200	200	200
Extraction, % Au			
2 hr	9.5	22.5	27.1
4 hr	8.2	23.4	41.8
8 hr	5.5	22.3	39.9
24 hr	4.2	20.8	33.2
48 hr	4.2	20.3	29.4
Residue, g/t Au	1.41	0.46	3.67
Cal. Feed, g/t Au	1.47	0.58	5.20
Consumption, kg/t			
Lime	1.495	1.597	0.897
NaCN	1.678	1.679	1.798

The test results indicate the following:

- Direct cyanidation of oxide ore extracted 50% to 62% of gold in 2 hours. The gold recovery dropped to 37% to 49% in 48 hours of leaching thereby indicating the ore exhibited pre-robbing properties.

- The carbon-in-leach for oxide ore recovered 62.1% to 71.6% of gold in 24 hours at a grind of P_{80} of 270 mesh. The NaCN consumption was reasonable at ± 1 kg/t and the lime consumption was ± 2.5 kg/t.

- The sulfide ores also exhibited preg-robbing properties besides being refractory ore. The gold extraction for average-grade sulfide ore was only 9.5% in 2 hours and dropped to 4.2% in 48 hours. The high-grade sulfide composite had 41.8% of gold extraction in 4 hours but dropped to 29.4% in 48 hours.

9. LEACHING OF SULFIDE FLOTATION CONCENTRATE

The leach tests were performed on flotation concentrates generated from the sulfide ores. The concentrates were also reground to determine if one could liberate gold from sulfides (i.e. pyrite/arsenopyrite) and leach it.

The test data are given in Appendix J and summarized in Table 17.

Table 17: Leaching of Flotation Concentrate

Parameter	Test No						
	2	3	8	14	15	16	17
Sample	Avg Grade Concentrate	Avg Grade 4hr Regrind Concentrate	Tailing	Avg Grade Concentrate Reground	Avg Grade Reground	Low Grade Concentrate	High Grade Concentrate
Flotation Test No	5	5	5	12	12	13	14
NaCN, g/L	2	2	1	5	5	5	5
Extraction, % Au							
2 hr	22.7	4.9	63.9				
4 hr	23.0	17.2	56.6				
8 hr	23.0	37.5	56.7				
24 hr	22.9	16.7	51.9	53.9			
36 hr	-	-	-	-	56.2	42.4	54.6
48 hr	20.3	10.8	50.9				
Residue, g/t Au	4.92	5.54	0.09	3.17	3.02	0.94	7.96
Cal. Feed, g/t Au	6.17	6.21	0.18	6.91	6.96	1.64	17.55
Consumption, kg/t							
Lime	2.092	2.627	3.596	1.087	0.598	6.092	4.547
NaCN	2.435	7.449	0.605	13.593	16.286	2.409	2.717

The test results indicate the following:

- The flotation concentrate also exhibited preg-robbing properties. The gold extraction tended to decrease with leach time.
- The flotation concentrate from average-grade composite recovered ± 20% of gold.
- Regrind of concentrate did improve gold extraction to 37.5% but decreased as the leaching process continued. Hence, regrinding concentrate enhanced preg-robbing properties.
- Carbon-in-leach (CIL) did improve gold extraction to 50% to 55% for the sulfide composites.
- Leaching of flotation tailing assaying 0.18 g/t Au resulted in gold extraction of 64% in two hours but dropped to 50.9% in 48 hours of leach time. Hence, even flotation tailing exhibited preg-robbing properties.

The leaching of ore on flotation concentrate recovered a maximum of ± 60% of gold extraction, indicating that the ore is both refractory and preg-robbing.

Hence, one would need to evaluate pre-treatment methods for improving project economics.

10. ROAST PLUS LEACH PROCESS

The test results had indicated two reasons for poor gold recovery, namely, refractoriness of ore and preg-robbing properties of the ore. A series of roasting tests at 325°C (normally designated calcining test) and 650°C under oxidizing conditions were performed for average- and high-grade sulfide composites and oxide composite 2 followed by cyanidation.

The test data are given in Appendix K and the results are summarized in Tables 18 and 19.

Table 18: Leaching of Composites Following 350°C Oxidizing Roast

Parameter	Test No.					
	18	19	22	23	26	27
Sample	**Average Grade Sulfide**	**Average Grade Sulfide**	**High Grade Sulfide**	**High Grade Sulfide**	**Oxide Comp 2**	**Oxide Comp 2**
NaCN, g/L	2	2	2		2	2
Extraction, % Au						
2hr	10.1		31.4		57.3	
4hr	8.6		29.6		56.9	
8hr	7.6		27.9		55.4	
24hr	5.7		22.4		50.6	
48hr	4.7	8.6	20.0	53.5	48.1	57.4
Residue, g/t Au	1.43	0.89	3.97	2.48	0.99	0.46
Cal. Feed, g/t Au	1.50	0.97	4.96	5.34	1.91	1.08
Consumption, kg/t						
Lime	17.856	15.476	18.363	21.628	9.163	9.057
NaCN	0.676	0.833	0.617	0.862	0.728	1.151

Note: Tests 19, 23, and 27 are CIL

Table 19: Leaching of Composites Following 350°C Oxidizing Roast

Parameter	Test No.					
	20	21	24	25	28	29
Sample	Average Grade Sulfide	Average Grade Sulfide	High Grade Sulfide	High Grade Sulfide	Oxide Comp 2	Oxide Comp 2
NaCN, g/L	2	2	2	2	2	2
Extraction, % Au						
2hr	81.1		88.5		82.7	
4hr	86.0		90.2		83.9	
8hr	87.5		90.0		85.1	
24hr	86.9		91.5		84.6	
48hr	86.0	89.6	92.2	93.1	85.4	82.5
Residue, g/t Au	0.22	0.20	0.44	0.44	0.32	0.43
Cal. Feed, g/t Au	1.58	1.92	5.62	6.38	2.20	2.46
Consumption, kg/t						
Lime	52.475	4.8222	8.393	7.822	-	-
NaCN	0.597	1.722	0.82	2.06	0.636	0.887

Note: Tests 21, 25, and 29 are CIL

The test results indicate the following:

- Oxidizing roast at 35°C did not eliminate the preg-robbing characteristics of the ore.
- CIL tests did eliminate the preg-robbing characteristics of the ore. The gold extraction for high-grade sulfide and oxide ore improved to 53% to 57%.
- Oxidizing roast at 650°C did improve the gold extraction to 89.6% for average grade sulfide, 93.1% for high-grade sulfide and 85.4% for oxide composite.

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APPENDIX A: DESCRIPTION OF SAMPLES

Getchell Au Nevada Project #: Getchell 23041 2/5/2024

ID's	Depth (m)	Sample wt (kg.)	Interval (m)	Zone	Lab wt (kg.)	Au ppm	Grade
FCGM20-A1							
591388	207-208.5	4.872	1.5	Colorado SW	5.61	1.2	Average
591434	265.6-266.7	3.589	1.1	Colorado SW	4.11	1.58	Average
591691	216.5-218	3.326	1.5	Colorado SW	3.74	1.39	Average
591643	166.8-167.9	4.766	1.1	Colorado SW	5.49	1.4	Average
591703	230.9-231.9	2.949	1	Colorado SW	3.4	1.25	Average
FCGM20-A2							
592041	356-357.5	4.577	1.5	North Fork	5.1	1.64	Average
592074	391.2-392.3	3.731	1.1	North Fork	4.13	1.58	Average
592085	401.2-402	3.075	0.8	North Fork	3.53	1.65	Average
592088	402.9-403.9	3.229	1	North Fork	3.7	1.72	Average
592095	409.5-410.5	3.538	1	North Fork	3.98	1.34	Average
592280	130.5-132	4.554	1.5	Colorado SW	5.09	1.3	Average
FCGM20-A3							
592576	190.7-192.2	4.156	1.5	Colorado SW	4.94	1.42	Average
592554	169.2-170.3	3.482	1.1	Colorado SW	3.84	1.29	Average
592378	236.6-238	4.197	1.4	Colorado SW	4.77	1.54	Average
592345	203.7-205	4.058	1.5	Colorado SW	4.95	1.43	Average
FCGM20-B1							
591389	208.5-210	3.785	1.5	Colorado SW	4.34	0.58	Low
592586	201.7-203.3	4.635	1.6	Colorado SW	5.18	0.45	Low
592051	368-369.5	4.671	1.5	North Fork	5.45	0.52	Low
FCGM20-C1							
592571	185.2-186.5	2.974	1.3	Colorado SW	3.46	6.32	High
592102	417.5-418.6	3.696	1.1	North Fork	4.37	6.44	High
592077	394.5-395.5	2.6	1	North Fork	3.04	5.87	High
591637	159.6-161	5.196	1.4	Colorado SW	5.9	5.04	High
FCGM21-A1							
669658	343.4-344	5.383	1.3	North Fork	5.85	1.26	Average
669652	339.3-339.8	2.476	0.5	North Fork	2.85	1.74	Average
593133	269-269.4	0.882	0.4	Colorado SW	1.25	1.72	Average
593131	266.5-267.5	2.969	1	Colorado SW	3.37	1.46	Average
593112	248.2-249.3	4.566	1.1	Colorado SW	4.88	1.42	Average
592862	226.8-227.3	2.047	0.5	Colorado SW	2.46	1.26	Average
FCGM21-A2							
670194	356.2-357.2	4.562	1	Colorado SW	5.3	1.51	Average
670152	315.5-317	5.732	1.5	Colorado SW	6.32	1.37	Average
670137	300.4-301.6	4.752	1.2	Colorado SW	5.28	1.44	Average
669674	355.4-355.7	1.734	0.3	North Fork	2.04	1.42	Average
669669	350.6-352.1	5.897	1.5	North Fork	6.85	1.61	Average
FCGM21-A3							
678093	129.2-130.7	3.83	1.5	North Fork	4.31	1.61	Average
670545	237-238	4.066	1	Colorado SW	4.42	1.29	Average
684576	164.9-165.8	1.443	0.9	North Fork	1.82	1.52	Average
678089	124.2-125.4	3.171	1.2	North Fork	3.79	1.48	Average
670537	226.5-228	4.51	1.5	Colorado SW	4.9	1.71	Average
FCGM21-B1							
678045	81-81.6	3.181	0.6	North Fork	3.66	0.58	Low
670102	261.6-263.1	6.036	1.5	Colorado SW	6.57	0.49	Low
669572	285.8-286.8	3.804	1	North Fork	4.7	0.58	Low
593075	212.2-213.5	4.818	1.3	Colorado SW	5.14	0.55	Low
FCGM21-C1							
684571	158.9-160.3	5.034	1.4	North Fork	5.75	5.37	High
670178	341.6-342.2	1.704	0.6	Colorado SW	2.42	5.97	High
669587	297.5-297.8	1.35	0.3	North Fork	1.72	5.13	High
669600	305.8-306.2	1.309	0.4	North Fork	1.64	5.18	High
593135	269.4-270.6	3.15	1.2	Colorado SW	3.89	5.79	High
FCGM22-A1							
685401	272.9-273.7	1.941	0.8	North Fork	2.3	1.53	Average
692632	170-170.9	1.664	0.9	Colorado SW	2.29	1.39	Average
685341	220.6-221.2	2.071	0.6	North Fork	2.4	1.23	Average
692659	184.8-185.8	4.196	1	Colorado SW	4.74	1.36	Average
692621	163.9-164.3	1.365	0.4	Colorado SW	1.98	1.62	Average
692620	163.5-163.9	1.628	0.4	Colorado SW	2.07	1.2	Average
684886	146.5-147.5	2.53	1	North Fork	3	1.41	Average
684878	138-138.6	1.664	0.6	North Fork	2.07	1.75	Average
FCGM22-A2							
812694	191.3-191.7	0.631	0.4	North Fork	1.12	1.35	Average
812689	188.4-188.8	0.888	0.4	North Fork	1.2	1.37	Average
812186	227.9-229.4	6.676	1.5	North Fork	7.21	1.36	Average
811821	221-221.6	1.088	0.6	Colorado SW	1.46	1.71	Average
692682	199.4-200.1	1.072	0.7	Colorado SW	1.53	1.56	Average
811822	221.6-222.5	3.65	0.9	Colorado SW	4.12	1.33	Average
692660	185.8-186.4	2.015	0.6	Colorado SW	2.69	1.63	Average
FCGM22-A3							
814519	186.2-187.2	4.367	1	Colorado SW	4.83	1.2	Average
814216	165-165.7	2.34	0.7	Colorado SW	2.76	1.52	Average
814214	162-163.5	4.501	1.5	Colorado SW	5.04	1.43	Average
813330	279.5-280.1	1.846	0.6	North Fork	2.21	1.51	Average
812710	201.1-201.6	1.566	0.5	North Fork	2.09	1.6	Average
812698	194.2-194.8	2.148	0.6	North Fork	2.49	1.42	Average
FCGM22-B1							
813266	226.5-227.0	1.063	0.5	North Fork	1.45	0.56	Low
812723	208.1-208.5	1.418	0.4	North Fork	1.89	0.47	Low
811801	206.8-207.9	3.628	1.1	Colorado SW	4.16	0.47	Low
692676	195.5-196.3	1.961	0.8	Colorado SW	2.49	0.48	Low
692221	167.4-168.8	4.001	1.4	North Fork	4.43	0.56	Low
692640	175.6-176.0	1.762	0.4	Colorado SW	2.18	0.56	Low
FCGM22-C1							
812695	191.7-192.3	1.457	0.6	North Fork	1.98	5.35	High
812681	181.1-181.9	2.086	0.8	North Fork	2.55	5.57	High
811819	218.3-219.6	3.697	1.3	Colorado SW	4.09	4.92	High
692691	204.8-205.7	3.169	0.9	Colorado SW	3.95	5.67	High
685320	205.3-205.8	1.524	0.5	North Fork	1.9	4.83	High
685299	192.2-192.8	1.366	0.6	North Fork	1.68	6.4	High

FCGM20 Image FCGM21

FCGM20 Image

FCGM21

FCGM22

Appendix A Description of Samples

Hole_id	Sample_ic	Zone	epth_from	epth_to_	Interval_m	Wgt_kg_Lab	Au_ppm	Au_ppb	Ag_ppb	As_ppm	Hg_ppb	Sb_ppm	W_ppm	S_pct	Ca_pct	Cu_ppm	Pb_ppm	Zn_ppm	Mo_ppm	Ni_ppm	Mg_pct	Al_pct	Co_ppm	Mn_ppm	Fe_pct	Th_ppm	Sr_ppm	Cd_ppm	P_pct	La_ppm	Ba_ppm	Ti_pct	Na_pct	K_pct	Sc_ppm	Ti_ppm
Average Grade																																				
FCG20-002	591388	Colorado SW	207.00	208.50	#	#	1.20	106.70	389	344.60	392	12.29	0.50	3.31	4.99	16.03	12.97	70.80	1.51	22.10	0.21	0.39	7.20	283	3.13	2.70	456.80	0.29	0.04	2.90	93.90	0.00	0.00	0.17	2.60	0.17
FCG20-002	591434	Colorado SW	265.60	266.70	#	#	1.58	251.00	813	625.60	379	16.56	0.50	4.33	2.47	23.98	21.65	38.00	0.98	28.20	0.69	0.55	11.50	309	4.16	5.60	284.40	0.44	0.03	3.40	28.90	0.00	0.01	0.24	3.40	0.22
FCG20-003	591643	Colorado SW	166.80	167.90	#	#	1.40	170.10	561	3869.00	482	20.60	0.70	2.91	1.55	58.97	20.03	88.70	0.67	37.80	0.56	0.73	17.70	383	3.82	6.50	199.60	0.26	0.07	6.80	90.90	0.00	0.01	0.24	3.40	0.30
FCG20-003	591691	Colorado SW	216.50	218.00	#	#	1.39	333.70	264	4477.20	512	13.91	0.50	3.04	1.95	25.16	20.49	79.70	0.74	30.80	0.77	0.67	13.90	352	3.42	6.60	154.60	0.10	0.05	6.20	109.10	0.00	0.01	0.36	4.70	0.41
FCG20-003	591703	Colorado SW	230.90	231.90	#	#	1.25	450.20	316	530.50	196	9.50	0.30	3.85	1.14	31.29	12.92	89.90	0.40	32.30	0.48	0.41	15.20	212	3.65	5.90	167.60	0.20	0.06	6.00	61.80	0.00	0.01	0.26	4.00	0.16
FCG20-004	592041	North Fork	356.00	357.50	#	#	1.64	135.00	302	1927.00	101	14.18	0.20	2.50	2.16	34.70	27.33	86.60	1.16	37.20	1.12	1.16	14.10	663	4.23	4.10	191.20	0.29	0.05	4.80	124.00	0.00	0.01	0.43	3.70	0.15
FCG20-004	592074	North Fork	391.20	392.30	#	#	1.58	1763.10	1146	3245.50	109	118.71	0.40	3.01	0.90	55.52	30.50	74.90	3.19	40.70	0.35	0.55	20.30	204	3.56	11.20	75.10	0.79	0.07	5.60	126.60	0.00	0.01	0.24	2.40	0.21
FCG20-004	592085	North Fork	401.20	402.00	#	#	1.65	525.20	530	2614.80	113	169.83	0.30	2.48	2.01	18.78	10.28	31.30	0.75	22.30	0.79	0.36	11.10	622	2.83	5.30	97.30	0.08	0.04	3.20	89.00	0.00	0.00	0.20	1.60	0.16
FCG20-004	592088	North Fork	402.90	403.90	#	#	1.72	815.40	281	2001.40	69	123.49	0.20	1.66	1.81	16.11	16.05	55.10	0.68	12.00	0.72	0.27	5.90	264	2.29	3.10	87.90	0.13	0.02	3.80	48.70	0.00	0.00	0.14	1.20	0.11
FCG20-004	592095	North Fork	409.50	410.50	#	#	1.34	579.60	242	2675.60	57	67.53	0.30	0.78	1.83	26.61	8.37	66.70	1.20	30.20	0.98	0.75	13.90	465	3.69	7.80	112.80	0.11	0.05	10.20	73.20	0.00	0.00	0.27	2.90	0.13
FCG20-005	592280	Colorado SW	130.50	132.00	#	#	1.30	144.80	786	2360.40	659	23.71	0.30	3.05	1.71	31.41	15.55	106.10	1.18	36.40	0.64	0.61	11.90	496	3.28	5.80	145.90	0.70	0.07	5.00	54.90	0.00	0.00	0.28	2.80	0.35
FCG20-005	592345	Colorado SW	203.70	205.20	#	#	1.43	159.90	507	585.10	325	15.51	0.30	2.36	4.49	18.18	20.51	67.90	1.46	19.30	0.83	0.43	6.50	400	2.71	2.90	170.30	0.71	0.06	3.00	59.80	0.00	0.00	0.19	2.70	0.21
FCG20-005	592378	Colorado SW	236.60	238.00	#	#	1.54	270.40	296	1934.40	476	23.52	0.40	2.32	2.98	26.99	16.45	91.10	0.72	28.40	0.97	0.51	11.90	432	3.08	5.00	124.80	0.50	0.06	5.80	63.90	0.00	0.01	0.24	3.30	0.32
FCG20-006	592554	Colorado SW	169.20	170.30	#	#	1.29	122.20	181	3660.30	348	30.08	0.30	1.67	2.13	28.42	7.28	76.10	0.90	34.30	0.75	0.95	13.20	487	2.98	5.30	111.50	0.34	0.05	7.60	73.70	0.00	0.01	0.26	3.00	0.29
FCG20-006	592576	Colorado SW	190.70	192.20	#	#	1.42	258.10	316	1593.50	544	23.37	0.60	4.02	2.31	28.32	16.32	80.00	0.55	37.80	0.99	0.54	17.20	430	3.96	4.00	238.60	0.23	0.04	3.50	100.90	0.00	0.01	0.25	2.80	0.18
FCG21-007	592862	Colorado SW	226.80	227.30	#	#	1.26	186.10	345	645.80	237	7.33	0.30	4.00	1.17	27.52	17.55	75.30	1.13	31.90	0.29	0.46	14.50	130	3.85	6.50	96.00	0.17	0.05	5.00	24.30	0.00	0.01	0.18	2.50	0.16
FCG21-008	593112	Colorado SW	248.20	249.30	#	#	1.42	87.20	458	633.50	655	12.38	0.50	2.18	2.90	39.34	21.78	73.70	1.86	45.00	0.99	0.51	15.20	444	3.07	6.40	187.90	0.24	0.18	7.40	175.40	0.00	0.01	0.28	3.80	0.37
FCG21-008	593131	Colorado SW	266.50	267.50	#	#	1.46	204.30	178	1131.90	301	5.29	0.30	1.88	3.26	20.48	13.17	68.70	0.71	17.90	1.19	0.69	9.70	585	2.71	5.40	183.30	0.15	0.05	6.90	149.00	0.00	0.01	0.26	4.00	0.18
FCG21-008	593133	Colorado SW	269.00	269.40	#	#	1.72	1915.50	439	348.60	373	8.18	0.20	1.76	3.48	10.04	8.89	27.80	1.01	15.40	0.91	0.55	5.10	563	2.81	3.70	195.10	0.04	0.03	5.20	161.70	0.00	0.00	0.12	1.90	0.18
FCG21-010A	669652	North Fork	339.30	339.80	#	#	1.74	769.40	249	2873.10	20	15.49	0.50	1.58	2.44	21.73	10.78	63.00	0.43	22.90	1.01	0.68	10.30	544	3.37	6.00	117.90	0.19	0.05	9.10	110.20	0.00	0.01	0.31	2.50	0.11
FCG21-010A	669658	North Fork	343.40	344.70	#	#	1.26	200.90	556	2297.30	254	25.43	1.00	3.80	2.51	24.85	29.89	72.40	2.31	40.20	0.97	0.66	16.10	540	4.36	6.20	149.20	0.25	0.06	4.40	147.20	0.00	0.01	0.34	3.00	0.21
FCG21-010A	669669	North Fork	350.60	352.10	#	#	1.61	56.70	856	998.00	515	38.82	0.30	3.79	1.52	27.13	33.02	81.10	1.99	48.80	0.56	0.37	13.60	346	3.85	8.00	107.80	0.88	0.08	2.50	114.50	0.00	0.01	0.23	2.20	0.37
FCG21-010A	669674	North Fork	355.40	355.70	#	#	1.42	38.30	956	1045.60	164	63.28	0.40	3.32	2.34	30.57	114.63	108.60	5.65	55.80	0.93	0.70	14.40	729	4.04	8.00	170.90	1.57	0.12	3.90	181.00	0.00	0.01	0.38	3.20	0.35
FCG21-011	670137	Colorado SW	300.40	301.60	#	#	1.44	214.00	228	2270.20	113	10.25	0.20	2.53	2.51	32.74	18.69	74.00	0.75	31.20	0.85	1.44	13.50	431	4.08	7.80	264.00	0.26	0.06	8.50	83.20	0.00	0.01	0.40	4.20	0.23
FCG21-011	670152	Colorado SW	315.50	317.00	#	#	1.37	143.20	657	902.50	406	22.10	0.30	2.62	3.48	36.45	24.78	74.90	1.54	28.80	1.39	1.44	12.90	717	3.56	6.70	310.10	0.44	0.06	6.20	119.50	0.00	0.01	0.27	3.20	0.35
FCG21-011	670194	Colorado SW	356.20	357.20	#	#	1.51	71.60	523	726.20	659	20.00	0.30	1.61	4.11	31.54	20.13	197.90	4.08	31.40	1.46	0.55	9.10	432	2.75	3.80	162.70	2.97	0.06	5.00	209.40	0.00	0.01	0.26	2.60	0.30
FCG21-012	670537	Colorado SW	226.50	228.00	#	#	1.71	162.60	686	529.60	472	14.24	0.30	3.03	9.11	25.79	21.48	81.50	1.95	32.30	0.71	0.31	6.90	663	3.28	2.10	412.40	0.29	0.06	2.50	84.10	0.00	0.01	0.17	3.20	0.20
FCG21-012	670545	Colorado SW	237.00	238.00	#	#	1.29	221.50	163	1444.90	211	9.94	0.30	2.11	3.34	22.27	23.33	42.90	0.30	31.30	1.30	0.44	12.90	327	3.24	4.70	168.00	0.12	0.05	6.70	172.00	0.00	0.01	0.27	2.80	0.16
FCG21-016	678089	North Fork	124.20	125.40	#	#	1.48	881.80	298	4672.40	93	19.17	0.30	2.20	2.04	22.71	26.72	63.90	0.98	23.90	0.85	0.61	8.60	387	3.46	5.30	194.40	0.29	0.04	7.40	170.30	0.00	0.01	0.29	2.60	0.12
FCG21-016	678093	North Fork	129.20	130.70	#	#	1.61	492.80	189	3618.90	122	27.93	0.30	2.14	4.45	23.11	18.78	73.80	0.92	32.30	1.69	0.62	10.10	773	4.26	5.00	268.70	0.15	0.04	5.50	128.50	0.00	0.01	0.32	3.20	0.19
FCG21-016	684576	North Fork	164.90	165.80	#	#	1.52	217.50	922	3431.30	114	32.97	0.30	2.19	1.86	39.47	55.03	54.70	2.97	22.10	0.56	0.71	8.90	425	3.36	5.70	73.50	1.23	0.02	4.60	110.70	0.00	0.01	0.16	2.20	0.14
FCG22-017A	684878	North Fork	138.00	138.60	#	#	1.75	96.70	474	6570.70	198	10.26	0.40	4.13	1.52	9.35	15.91	26.10	1.89	24.70	0.59	0.40	10.80	607	4.39	3.40	177.60	0.18	0.04	3.10	78.70	0.00	0.01	0.20	1.70	0.15
FCG22-017A	684886	North Fork	146.50	147.50	#	#	1.41	65.40	512	2204.40	148	8.43	0.30	2.98	1.23	20.76	38.12	110.00	2.92	27.90	0.47	0.69	13.20	367	3.41	4.40	89.50	0.65	0.04	6.30	90.00	0.00	0.01	0.28	2.70	0.14
FCG22-018	685341	North Fork	220.60	221.20	#	#	1.23	373.30	195	1156.80	91	14.79	0.10	1.28	0.74	8.45	13.30	37.90	0.36	11.80	0.31	0.23	5.20	182	1.46	2.40	38.20	0.09	0.02	4.70	54.40	0.00	0.00	0.14	0.90	0.09
FCG22-018	685401	North Fork	272.90	273.70	#	#	1.53	653.90	139	249.70	30	9.17	0.30	0.76	1.14	6.38	7.11	10.60	2.47	3.50	0.35	0.18	2.30	278	1.05	1.90	52.40	0.05	0.03	4.10	132.10	0.00	0.00	0.09	0.50	0.05
FCG22-020	692620	Colorado SW	163.50	163.90	#	#	1.20	105.10	282	321.50	368	9.22	0.20	3.22	0.57	18.39	18.46	62.50	0.96	37.90	0.26	0.42	17.60	109	3.00	5.70	62.90	0.42	0.05	8.90	87.90	0.00	0.01	0.26	2.80	0.21
FCG22-020	692621	Colorado SW	163.90	164.30	#	#	1.62	158.80	452	369.80	292	13.60	0.30	6.29	0.22	22.65	21.11	53.70	0.91	42.30	0.09	0.53	15.70	67	5.65	5.20	31.30	0.32	0.04	8.40	67.20	0.00	0.01	0.30	2.00	0.18
FCG22-020	692632	Colorado SW	170.00	170.90	#	#	1.39	92.00	207	244.70	648	7.52	0.30	3.12	0.90	12.18	10.56	42.50	0.73	15.40	0.05	0.45	6.20	81	3.07	4.40	70.40	0.16	0.11	9.20	189.50	0.00	0.01	0.24	1.50	0.16
FCG22-020	692659	Colorado SW	184.80	185.80	#	#	1.36	94.50	691	267.20	800	10.86	0.40	2.47	6.80	25.79	13.22	76.30	4.53	30.40	0.96	0.44	9.40	557	2.76	3.50	349.00	0.72	0.07	4.30	185.90	0.00	0.01	0.25	3.30	0.20
FCG22-020	692660	Colorado SW	185.80	186.40	#	#	1.63	32.50	728	194.80	1523	15.73	1.00	2.83	6.35	32.44	17.19	126.00	28.25	46.20	0.84	0.35	9.40	568	2.93	3.40	339.90	1.27	0.09	3.70	156.00	0.00	0.01	0.20	3.00	0.40
FCG22-020	692682	Colorado SW	199.40	200.10	#	#	1.56	436.40	503	408.70	1549	8.69	0.30	2.88	0.50	28.76	17.33	50.70	3.98	30.70	0.16	0.40	8.90	84	3.09	2.80	88.30	0.34	0.04	5.40	108.50	0.00	0.01	0.20	1.50	0.12
FCG22-021	811821	Colorado SW	221.00	221.60	#	#	1.71	251.00	346	295.40	991	16.71	0.60	3.71	4.87	23.18	15.04	69.40	2.31	42.30	1.45	0.64	15.30	589	4.00	3.30	200.40	0.24	0.06	4.00	120.90	0.00	0.01	0.35	4.10	0.28
FCG22-021	811822	Colorado SW	221.60	222.50	#	#	1.33	81.80	387	111.50	950	16.89	0.80	2.30	10.53	19.39	8.40	97.20	11.99	30.80	1.40	0.45	7.80	774	2.79	1.90	599.00	0.59	0.06	3.20	150.30	0.00	0.01	0.24	3.00	0.25
FCG22-022	812186	North Fork	227.90	229.40	#	#	1.36	763.50	325	1097.40	45	64.48	0.20	1.00	1.04	34.22	90.32	103.40	0.51	34.30	0.78	0.76	12.20	355	3.48	6.60	94.30	0.22	0.05	13.00	106.00	0.00	0.01	0.46	3.00	0.22
FCG22-023	812689	North Fork	188.40	188.80	#	#	1.37	205.90	278	1071.90	179	29.37	0.30	2.49	2.07	12.55	17.31	32.40	7.44	14.90	0.62	0.24	8.50	383	3.12	2.20	74.40	0.26	0.03	2.50	68.40	0.00	0.01	0.12	1.20	0.16
FCG22-023	812694	North Fork	191.30	191.70	#	#	1.35	572.70	342	445.20	104	37.34	0.30	1.99	1.39	18.44	19.21	44.60	0.80	15.30	0.56	0.23	7.20	229	2.45	3.40	74.60	0.27	0.03	3.80	98.10	0.00	0.01	0.15	1.60	0.09
FCG22-023	812698	North Fork	194.20	194.80	#	#	1.42	611.30	425	1855.60	102	28.44	0.20	2.68	1.10	17.60	16.56	30.60	0.63	26.50	0.31	0.33	12.10	266	2.83	4.80	72.10	0.19	0.06	5.10	178.90	0.00	0.01	0.21	1.30	0.11
FCG22-023	812710	North Fork	201.10	201.60	#	#	1.60	814.90	186	5528.60	73	21.68	0.20	1.81	3.64	13.31	12.27	44.80	2.48	15.80	0.93	0.21	7.00	763	2.71	2.30	126.20	0.19	0.04	3.00	86.30	0.00	0.00	0.12	1.30	0.13
FCG22-025	813330	North Fork	279.50	280.10	#	#	1.51	84.60	210	819.20	398	16.66	0.30	1.41	4.73	35.75	22.67	92.50	1.95	37.40	1.74	0.51	12.60	648	3.42	4.50	154.80	0.35	0.09	5.60	133.30	0.00	0.01	0.28	2.20	0.30
FCG22-027	814214	Colorado SW	162.00	163.50	#	#	1.43	468.50	528	1578.80	692	18.00	0.90	3.26	2.13	26.11	15.51	57.20	0.64	26.10	0.96	0.32	11.30	472	3.44	3.70	188.60	0.49	0.05	3.60	21.90	0.00	0.01	0.18	2.00	0.13
FCG22-027	814216	Colorado SW	165.00	165.70	#	#	1.52	411.40	419	976.80	668	12.81	0.60	2.42	1.86	22.03	9.33	218.00	0.58	20.10	0.74	0.29	8.60	489	2.57	3.10	176.50	1.94	0.03	4.50	29.70	0.00	0.01	0.16	2.70	0.09
FCG22-028	814519	Colorado SW	186.20	187.20	#	#	1.20	61.90	339	976.80	99	12.71	0.50	2.08	2.44	30.12	14.28	82.60	0.84	31.40	0.99	0.69	11.30	684	3.58	4.30	172.20	0.25	0.05	5.10	148.60	0.00	0.01	0.21	2.30	0.10

	Samples		########	#########	1.45
Colorado SW	30			#########	
North Fork	37			#########	

Hole_id	Sample_ic	Zone	epth_from	epth_to_	Interval_m	Wgt_kg_Lab	Au_ppm	Au_ppb	Ag_ppb	As_ppm	Hg_ppb	Sb_ppm	W_ppm	S_pct	Ca_pct	Cu_ppm	Pb_ppm	Zn_ppm	Mo_ppm	Ni_ppm	Mg_pct	Al_pct	Co_ppm	Mn_ppm	Fe_pct	Th_ppm	Sr_ppm	Cd_ppm	P_pct	La_ppm	Ba_ppm	Ti_pct	Na_pct	K_pct	Sc_ppm	Ti_ppm
Low Grade																																				
FCG20-002	591389	Colorado SW	208.50	210.00	#	#	0.58	81.60	535	275.40	437	12.93	0.40	3.46	3.43	24.19	10.77	78.30	1.91	30.00	0.13	0.43	11.00	229	3.18	4.30	259.50	0.30	0.06	4.60	114.00	0.00	0.01	0.25	3.20	0.19
FCG20-004	592051	North Fork	368.00	369.50	#	#	0.52	62.60	254	2622.00	68	17.45	0.20	2.49	1.64	25.97	30.17	60.80	6.40	22.20	0.87	1.16	11.40	558	4.09	4.70	168.50	0.51	0.05	5.60	185.10	0.00	0.01	0.39	2.50	0.19
FCG20-006	592586	Colorado SW	201.70	203.30	#	#	0.45	151.30	107	972.20	87	14.07	0.20	0.74	4.11	22.40	14.85	83.20	0.29	24.50	1.34	0.82	10.90	616	3.01	3.80	278.50	0.16	0.05	6.30	396.40	0.00	0.01	0.26	3.00	0.17
FCG21-008	593075	Colorado SW	212.20	213.50	#	#	0.55	51.50	2142	381.20	240	19.05	0.40	2.56	6.20	44.95	22.96	127.20	0.99	27.50	1.56	1.08	12.90	653	3.75	5.80	608.30	0.88	0.26	6.10	45.20	0.00	0.01	0.26	4.90	0.15
FCG21-010A	669572	North Fork	285.80	286.80	#	#	0.58	44.50	241	3478.90	79	48.45	0.40	3.12	2.02	31.40	10.66	108.70	2.20	52.00	0.72	0.78	17.50	483	3.99	6.00	116.90	0.51	0.08	5.30	173.70	0.00	0.01	0.44	3.20	0.18
FCG21-011	670102	Colorado SW	261.60	263.10	#	#	0.49	67.20	497	617.70	269	9.62	0.20	2.19	2.91	22.43	16.97	43.20	1.53	23.20	0.93	0.80	11.90	520	2.95	8.00	276.10	0.04	0.05	6.80	143.20	0.00	0.01	0.31	4.40	0.20
FCG21-016	678045	North Fork	81.00	81.60	#	#	0.58	129.50	149	4725.00	62	10.70	0.30	3.61	0.95	12.32	28.00	55.20	1.01	31.30	0.51	0.79	19.80	383	4.25	5.50	106.60	0.15	0.07	9.60	101.90	0.00	0.01	0.36	2.50	0.12
FCG22-019	692221	North Fork	167.40	168.80	#	#	0.56	36.20	574	4452.40	88	42.81	0.30	2.45	1.69	39.09	63.93	276.30	1.14	38.70	0.70	1.39	17.70	458	4.33	6.20	77.60	1.65	0.06	6.00	96.10	0.00	0.01	0.35	2.20	0.17
FCG22-020	692640	Colorado SW	175.60	176.00	#	#	0.56	63.10	436	233.90	925	15.87	0.30	4.47	1.05	49.81	8.66	95.40	0.59	42.40	0.08	0.53	13.40	83	4.00	6.70	96.60	0.41	0.20	12.20	80.80	0.00	0.01	0.28	2.10	0.21
FCG22-020	692676	Colorado SW	195.50	196.30	#	#	0.48	34.80	789	327.00	574	9.17	0.30	3.44	1.29	37.17	24.70	86.60	0.93	43.00	0.45	0.55	17.40	159	3.45	6.40	128.20	0.37	0.06	9.70	75.50	0.00	0.01	0.28	2.60	0.15
FCG22-021	811801	Colorado SW	206.80	207.90	#	#	0.47	10.40	1067	229.00	355	10.38	0.50	2.98	7.91	22.67	11.01	68.80	9.12	26.00	0.25	0.46	6.70	404	3.08	1.90	543.90	0.49	0.07	4.10	29.80	0.00	0.00	0.19	2.30	0.14
FCG22-023	812723	North Fork	208.10	208.50	#	#	0.47	199.80	49	587.00	44	71.15	0.20	1.15	0.77	28.14	3.30	48.50	0.66	29.90	0.57	0.33	12.90	253	3.26	6.50	65.60	0.14	0.04	7.30	71.50	0.00	0.00	0.20	2.10	0.29
FCG22-025	813266	North Fork	226.50	227.00	#	#	0.56	29.90	553	456.50	182	51.94	0.60	3.13	5.45	13.31	58.78	168.30	4.17	40.10	0.81	0.67	10.00	859	4.12	3.70	241.90	0.70	0.47	5.20	81.20	0.00	0.01	0.32	1.90	0.19
					########	#########	0.52																													

Hole_id	Sample_ic	Zone	epth_from	epth_to_	Interval_m	Wgt_kg_Lab	Au_ppm	Au_ppb	Ag_ppb	As_ppm	Hg_ppb	Sb_ppm	W_ppm	S_pct	Ca_pct	Cu_ppm	Pb_ppm	Zn_ppm	Mo_ppm	Ni_ppm	Mg_pct	Al_pct	Co_ppm	Mn_ppm	Fe_pct	Th_ppm	Sr_ppm	Cd_ppm	P_pct	La_ppm	Ba_ppm	Ti_pct	Na_pct	K_pct	Sc_ppm	Ti_ppm
High Grade																																				
FCG20-003	591637	Colorado SW	159.60	161.00	#	#	5.04	3846.10	430	437.60	222	8.06	0.20	1.22	0.36	7.87	16.11	16.10	0.28	8.10	0.23	0.26	4.20	103	1.52	3.30	73.20	0.11	0.03	7.30	257.20	0.00	0.01	0.11	1.20	0.14
FCG20-004	592077	North Fork	394.50	395.50	#	#	5.87	11213.20	758	6634.00	161	267.34	0.20	0.82	0.85	5.25	36.78	23.60	3.70	7.50	0.33	0.12	3.60	256	1.45	2.30	54.80	1.08	0.02	2.90	54.60	0.00	0.00	0.06	0.80	0.24
FCG20-004	592102	North Fork	417.50	418.60	#	#	6.44	2441.30	631	1257.90	127	74.05	0.30	3.44	1.23	15.98	10.29	39.20	1.37	16.40	0.52	0.28	7.50	271	3.44	4.60	62.80	0.18	0.04	4.00	149.20	0.00	0.00	0.16	1.30	0.17
FCG20-006	592571	Colorado SW	185.20	186.50	#	#	6.32	3039.60	425	1072.00	822	20.31	0.30	2.25	4.11	20.17	14.52	66.50	0.53	24.00	1.71	0.36	10.20	897	2.98	4.40	191.10	0.22	0.04	4.20	153.30	0.00	0.00	0.18	4.00	0.27
FCG21-008	593135	Colorado SW	269.40	270.60	#	#	5.79	2422.10	650	378.80	581	8.40	0.30	2.23	3.49	14.60	11.01	43.20	1.54	16.90	1.10	0.84	6.90	540	3.28	3.30	162.70	0.07	0.05	4.60	138.50	0.00	0.01	0.19	3.30	0.24
FCG21-010A	669587	North Fork	297.50	297.80	#	#	5.13	818.00	237	4643.20	56	46.32	0.40	2.25	2.49	26.77	14.21	86.40	1.16	28.60	0.91	0.60	13.00	464	3.83	5.00	106.10	0.32	0.05	6.40	97.40	0.00	0.01	0.30	3.00	0.20
FCG21-010A	669600	North Fork	305.80	306.20	#	#	5.18	2178.40	346	1617.10	66	32.65	0.20	1.14	1.47	27.51	9.21	55.70	0.65	27.30	0.58	0.45	11.40	394	1.84	6.00	84.50	0.17	0.06	8.90	179.50	0.00	0.01	0.28	2.20	0.12
FCG21-011	670178	Colorado SW	341.60	342.20	#	#	5.97	2399.20	200	914.70	307	9.97	0.30	0.98	4.53	16.51	6.12	34.40	1.10	14.10	1.09	0.80	7.20	456	2.48	4.00	182.40	0.24	0.04	6.10	109.30	0.00	0.01	0.25	2.50	0.30
FCG21-016	684571	North Fork	158.90	160.30	#	#	5.37	483.80	1098	5715.30	210	33.21	0.40	4.23	0.62	49.64	16.08	99.00	0.94	32.40	0.26	0.57	15.00	140	4.31	7.60	44.30	0.61	0.03	5.50	101.20	0.00	0.01	0.27	2.00	0.20
FCG22-018	685320	North Fork	205.30	205.8	########	#########	4.83	2898.05	342	2771.60	77	50.3	0.35	2.7	0.42	13.74	12.75	35.4	0.72	16.3	0.14	0.43	6.75	125	2.93	3.95	35.9	0.16	0.025	7.8	106.6	0.001	0.005	0.27	1.15	0.12
FCG22-020	692691	Colorado SW	204.80	205.7	########	#########	5.67	1548.6	193	680.80	1167	9.09	0.3	3.07	1.43	21.18	14.71	47.9	1.16	23.3	0.57	0.39	7.7	231	3.34	3	272.3	0.08	0.049	4.8	100.5	0.001	0.006	0.21	2.5	0.39
FCG22-021	811819	Colorado SW	218.30	219.6	########	#########	4.92	1542.4	605	326.20	1665	16.78	0.6	3.15	1.27	23.65	10.64	90.8	4.36	29.4	0.33	0.36	8.9	170	3.13	3.1	81	0.4	0.065	5.2	139.3	0.001	0.005	0.2	1.9	0.32
FCG22-023	812681	North Fork	181.10	181.9	########	#########	5.57	4466.2	226	219.40	149	17.48	0.1	0.81	0.29	6.18	5.92	12.2	0.43	7	0.11	0.11	3	81	1.18	1.4	21.9	0.09	0.012	3.4	46.3	0.001	0.003	0.07	0.5	0.06
FCG22-023	812695	North Fork	191.70	192.3	########	#########	5.35	3565.9	349	365.80	172	18.15	0.2	1.29	0.23	8.73	9.71	41.4	0.63	16.7	0.06	0.2	7.9	57	1.53	3.1	25.7	0.17	0.04	5.4	76.7	0.001	0.005	0.12	0.6	0.12
					########	#########	5.56																													

APPENDIX B: HEAD ANALYSES DATA

Appendix B Head Analysis Data

Project: Getchell - 23041
Revised: 4/11/2024
Author: J.Axen

Average Grade Head Comp Assay Data

Fire Assay Au/Ag

Sample Name	Gold mg/kg	Silver mg/kg
Average Grade	1.49	BD
Average Grade Dup	1.47	BD

Cyanide Shake Leach Data

Sample	Description	Leach Parameters NaCN, gpL	Wt., gms	Vol., mL / g	Au, mg/L	Ag, mg/L	Extraction, g Au/MT Ore	Extraction, g Ag/MT Ore
AN24-0229	Avg. Grade Head	2	15.02	30.15	BD	BD	0.00	0.00
AN24-0229 D	Avg. Grade Head	2	15.03	30.13	BD	BD	0.00	0.00
AN24-0229 T	Avg. Grade Head	2	15.01	30.10	BD	BD	0.00	0.00

Leco Forms of Carbon and Sulfur Data

		CTotal		Corganic	Cinorganic	STotal		Sorg/Ssulfide	Sinorg/Ssulfate
Project Id	Sample Name	Total C %	HCl Insol Carbon	Carbonate (by Calculation) Organic %	(HCl Treated C) Inorganic by Difference %	Total S %	HCl Insol Sulfur	Insoluble Sulfur (by Calculation) Sulfide %	(HCl Treated S) Sulfate by Difference %
23041	Average Grade	1.74%	0.33%	1.40%	0.33%	2.66%	2.58%	0.08%	2.58%

AMICS - Eagle Engineering
received 4/24/24

Florin Analytical Silver Data

Sample Name	Silver g/mt
Average Grade	1.10
Average Grade Dup	2.10

Appendix B Head Analysis Data

Project: Getchell - 23041
Revised: 8/23/2024
Author: J.Axen

High and Low Grade Head Comp Assays

Fire Assay Au/Ag (4/11/24)

Sample Name	Gold mg/kg	Silver mg/kg
High Grade	4.88	3
High Grade dup	4.97	2
Low Grade	0.53	<1
Low Grade dup	0.53	2

Sample	Description	Leach Parameters							Results	
		NaCN, gpL	Wt., gms	Vol., mLs	Temp., °C	Time, hours	Au, mg/L	Ag, mg/L	Extraction, g Au/MT Ore	Extraction, g Ag/MT Ore
AN24-0758 1/3	Low Grade Head Comp	2	15.05	30.07	20	24	0	0.02	**0.00**	**0.04**
AN24-0758 2/3	Low Grade Head Comp	2	15.02	30.22	20	24	0	0.02	**0.01**	**0.03**
AN24-0758 3/3	Low Grade Head Comp	2	15.03	30.44	20	24	0	0.01	**0.01**	**0.03**
AN24-0759 1/3	High Grade Head Comp	2	15.05	30.17	20	24	0.5	0.1	**0.99**	**0.19**
AN24-0759 2/3	High Grade Head Comp	2	15.05	30.11	20	24	0.56	0.1	**1.11**	**0.20**
AN24-0759 3/3	High Grade Head Comp	2	15.04	30.01	20	24	0.51	0.1	**1.02**	**0.21**

Leco Forms of Carbon and Sulfur Data

		CTotal	Corganic	Cinorganic	STotal	Sulfide	Sulfate
Project Id	**Sample Name**	**Total C %**	**Carbonate Organic %**	**(HCl Treated C) Inorganic by Difference %**	**Total S %**	**Insoluble Sulfur Sulfide %**	**(HCl Treated S) Sulfate by Difference %**
23041	Getchell Low Grade	0.59	0.27	0.33	0.13	0.05	0.08
23041	Getchell High Grade	0.11	0.06	0.05	0.04	0.01	0.02

AMICS - Eagle Engineering
received 4/24/24

Florin Analytical Silver Data (4-19/24)

Sample Name	Silver g/mt
High Grade	1.40
High Grade dup	0.80
Low Grade	0.80
Low Grade dup	0.80

Appendix B Head Analysis Data

Project: Getchell - 23041
Revised: 8/23/2024
Author: J.Axen

Oxide Composite 1 and 2 Head Assays
Oxide Composite 1: reject material received 6/12. material already crushed, PSD performed 6/18/24
Oxide Composite 2: 49 kg of core received 6/21. crushed, composited and prepped to 6 M charges week of 6/24. Grind Study completed 7/5/24

Fire Assay Au/Ag

Sample Name	Gold mg/kg	Silver mg/kg
Oxide Head Comp 1	1.38	2.2
Oxide Head Comp 1 Dup	1.39	<0.3
Oxide Shipment 2 Head	1.88	1.9
Oxide Shipment 2 Head Dup	1.84	1.7

Cyanide Shake Leach Data Oxide Composite 2

Sample	Description	Leach Parameters					Results			
		NaCN, gpL	Wt., gms	Vol., mLs	Temp., °C	Time, hours	Au, mg/L	Ag, mg/L	Extraction, g Au/MT Ore	Extraction, g Ag/MT Ore
AN24-1821	Getechell Oxide Comp 2	2	15.01	30.18	Ambient	24	0.50	0.52	**1.01**	**1.05**
AN24-1821 D	Getechell Oxide Comp 2	2	15.01	30.28	Ambient	24	0.52	0.60	**1.04**	**1.20**
AN24-1821 T	Getechell Oxide Comp 2	2	15.04	30.07	Ambient	24	0.53	0.48	**1.06**	**0.96**

Cyanide Shake Leach Data Oxide Composite 1

Sample	Description	Leach Parameters					Results			
		NaCN, gpL	Wt., gms	Vol., mLs	Temp., °C	Time, hours	Au, mg/L	Ag, mg/L	Extraction, g Au/MT Ore	Extraction, g Ag/MT Ore
AN24-1858	Getechell Oxide Comp 1	2	15.02	30.01	Ambient	24.00	0.31	0.42	0.62	0.84

Leco Forms of Carbon and Sulfur Data

FORTE ANALYTICAL

		CTotal	Corganic	Cinorganic	STotal	Sulfide	Sulfate
Project Id	**Sample Name**	**Total C %**	**Organic Carbon%**	**(HCl Treated C) Inorganic Carbon by Difference %**	**Total S %**	**Insoluble Sulfur Sulfide %**	**(HCl Treated S) Sulfate by Difference %**
23041	Oxide Comp 1	0.85	0.21	0.64	0.02	0.02	0.00
23041	Oxide Comp 2	1.00	0.20	0.80	0.05	0.00	0.04

Florin Analytical Silver Assays

Sample Name	Silver g/mt
Oxide Head Comp 1	1.20
Oxide Head Comp 1 Dup	1.60
Oxide Shipment 2 Head	2.00
Oxide Shipment 2 Head Dup	1.60

Appendix B Head Analysis Data

FORTE ANALYTICAL

Project: 23041 Getchell
Sample: Oxide Head Composites
Date: 8/5/2024
Reviewed by: J.Axen

ICP-OES Assay post 3-Acid Dissolution

Sample Name	Ag (PPM)	Al (PPM)	As (PPM)	Ba (PPM)	Be (PPM)	Ca (PPM)	Cd (PPM)	Co (PPM)
	22	113	19	27	21	366	20	18
Oxide Comp 1	BD	1609	BD	47	BD	BD	BD	BD
Oxide Comp 2	BD	46893	577	786	BD	25026	BD	BD

Sample Name	Cr (PPM)	Cu (PPM)	Fe (PPM)	K (PPM)	Mg (PPM)	Mn (PPM)	Mo (PPM)	Na (PPM)	Ni (PPM)
	26	118	124	8	23	24	21	435	8
Oxide Comp 1	BD	BD	BD	90	BD	BD	BD	BD	BD
Oxide Comp 2	91	BD	25523	17228	5700	602	BD	1039	23

Sample Name	P (PPM)	Pb (PPM)	S (PPM)	Sb (PPM)	Se (PPM)	Sn (PPM)	Sr (PPM)	Th (PPM)	Ti (PPM)
	156	18	744	22	21	14	26	27	21
Oxide Comp 1	BD	BD	BD	24	BD	BD	BD	BD	49
Oxide Comp 2	552	29	4850	BD	BD	BD	180	BD	2397

Sample Name	Tl (PPM)	U (PPM)	V (PPM)	Zn (PPM)
	8	15	22	20
Oxide Comp 1	20	BD	BD	BD
Oxide Comp 2	BD	125	102	81

Appendix B Head Analysis Data

FORTE ANALYTICAL

Project: 23041 Getchell
Sample: high, Low Head Composites
Date: 4/8/2024
Reviewed by: J.Axen

Sample Name	Ag	Al	As	Ba	Be	Ca	Cd
	(PPM)	(PPM)	(PPM)	(PPM)	(PPM)	(PPM)	(PPM)
	22	113	19	27	21	366	20
Getchell Low Grade	BD	63948	2548	1666	BD	21379	BD
Getchell High Grade	BD	41646	1630	1497	BD	10853	BD

Sample Name	Co	Cr	Cu	Fe	K	Mg	Mn
	(PPM)	(PPM)	(PPM)	(PPM)	(PPM)	(PPM)	(PPM)
	18	26	118	124	8	23	24
Getchell Low Grade	BD	72.7	BD	35614	25133	9033	345
Getchell High Grade	BD	102	BD	19968	16129	6238	226

Sample Name	Mo	Na	Ni	P	Pb	S	Sb
	(PPM)	(PPM)	(PPM)	(PPM)	(PPM)	(PPM)	(PPM)
	21	435	8	156	18	744	22
Getchell Low Grade	BD	BD	27.6	567	27.5	28556	BD
Getchell High Grade	BD	BD	15.2	311	20.8	20057	33.1

Sample Name	Se	Sn	Sr	Th	Ti	Tl	U
	(PPM)	(PPM)	(PPM)	(PPM)	(PPM)	(PPM)	(PPM)
	21	14	26	27	21	8	15
Getchell Low Grade	BD	BD	213	BD	2452	BD	142
Getchell High Grade	BD	BD	106	BD	2084	BD	106

Sample Name	V	Zn
	(PPM)	(PPM)
	22	20
Getchell Low Grade	74.6	66.6
Getchell High Grade	52	36.2

Appendix B Head Analysis Data

FORTE ANALYTICAL

Project: 23041 Getchell
Sample: Average Head Composite
Date: 4/11/2024
Reviewed by: J.Axen

Sample Name	Ag (PPM)	Al (PPM)	As (PPM)	Ba (PPM)	Be (PPM)	Ca (PPM)	Cd (PPM)	Co (PPM)
	22	113	19	27	21	366	20	18
Average Grade Head	BD	63391	1775	2444	BD	30266	BD	BD

Sample Name	Cr (PPM)	Cu (PPM)	Fe (PPM)	K (PPM)	Mg (PPM)	Mn (PPM)	Mo (PPM)	Na (PPM)
	26	118	124	8	23	24	21	435
Average Grade Head	89.3	BD	35761	16258	11548	379	BD	482

Sample Name	Ni (PPM)	P (PPM)	Pb (PPM)	S (PPM)	Sb (PPM)	Se (PPM)	Sn (PPM)	Sr (PPM)
	8	156	18	744	22	21	14	26
Average Grade Head	24.2	497	28.9	24580	40.663815	BD	BD	214

Sample Name	Th (PPM)	Ti (PPM)	Tl (PPM)	U (PPM)	V (PPM)	Zn (PPM)
	27	21	8	15	22	20
Average Grade Head	BD	3175	BD	142	75.3	70.5

Florin Analytical Services

7950 Security Circle - Reno, Nevada 89506 - Phone (775) 677-2177 - FAX (775) 972-4567

Certificate of Analysis

Submitted By: Forte Analytical	Laboratory No.: 241183
11475 West I-70 Frontage Road North	Client Number: F945
Wheat Ridge, CO 80033	Date Received: 19 Mar 2024
Attention: Jessica Axen	Date Completed: 03 Apr 2024

Method: 1/2-AT Fire assay with AA finish for Au & gravimetric finish for Ag. PO No.: 23041

Lab code: 4001	4001	4001
Element:	Silver	Silver
Detection Limit (@ 1 AT):	1.7	1.7
Units:	g/MT	g/MT
Average Grade Head Composite	1.1	2.1

Karen Boldi

Karen Boldi, Quality Control Supervisor

Florin Analytical Services

7950 Security Circle - Reno, Nevada 89506 - Phone (775) 677-2177 - FAX (775) 972-4567

Certificate of Analysis

Submitted By: Forte Analytical	Laboratory No.: 241224
11475 West I-70 Frontage Road North	Client Number: F945
Wheat Ridge, CO 80033	Date Received: 09 Apr 2024
Attention: Jessica Axen	Date Completed: 19 Apr 2024
Method: 4-Acid digestion, AAS analysis.	PO No.: 23041 Getchell

Lab code: 7048

Element:	Silver	Silver
Detection Limit:	0.5	0.5
Units:	g/MT	g/MT
Getchell High Grade Comp	1.4	0.8
Getchell Low Grade Comp	0.8	0.8
Blank	<0.5	
GBM 917-2	10.6	

Certified Reference Material	**Ag**
GBM 917-2	10.3
95% Confidence Interval of Mean	0.2

Mickyle O'Neal, Chemist

Nevada Assembly Bill No. 519.130 requires the following statement: The results of this assay were based solely upon the content of the sample submitted. Any decision to invest should be made only after the potential investment value of the claim or deposit has been determined based on the results of assays of multiple samples of geologic materials collected by the prospective investor or by a qualified person selected by him/her and based on an evaluation of all engineering data which is available concerning any proposed project.

Florin Analytical Services

7950 Security Circle - Reno, Nevada 89506 - Phone (775) 677-2177 - FAX (775) 972-4567

Certificate of Analysis

Submitted By: Forte Analytical	Laboratory No.: 241601
11475 West I-70 Frontage Road North	Client Number: F945
Wheat Ridge, CO 80033	Date Received: 22 Jul 2024
Attention: Jessica Axen	Date Completed: 31 Jul 2024
Method: 4-Acid digestion, AAS analysis.	PO No.: 23041 Getchell

Lab code: 7048

Element:	Silver	Silver
Detection Limit:	0.5	0.5
Units:	g/MT	g/MT
23041 Getchell Oxide Comp 2	2.0	1.6
23041 Getchell Oxide Comp 1Head	1.2	1.6
Blank	<0.5	
GBM 917-2	9.2	
Certified Reference Material	**Ag**	
GBM 917-2	10.3	
95% Confidence Interval of Mean	0.2	

Mickyle O'Neal, Chemist

Batch ID: 23041-1 Getchell Avg Head
Report Date: 2/21/2024
Report to: JA/DM

Analytical Report

Project 23041 Getchell Gold

Lab ID	Sample Name	Gold mg/kg	Silver mg/kg
	Average Grade	1.49	<7
	Average Grade Duplicate run	1.47	<7
	MB	<0.02	<7
CRM 609b	4.97 mg/kg Au, 24.6 mg/kg Ag	5.14	27.6

Analysis Method: litharge fire assay fusion with aqua regia dissolution of prill and final analysis by AAS for gold. Silver by calculation

Reviewed and Approved by: ______________________

Jessica Axen
Director of Exploration Services

Reporting Limit:	1AT	1/2 AT
Gold (mg/kg):	0.02	0.04
Silver (mg/kg):	0.3	7

Batch ID: 23041-3 High and Low Comp
Report Date: 4/11/2012
Report to: JA/DM

Analytical Report

Project 23041 Getchell Gold

Lab ID	Sample Name	Gold mg/kg	Silver mg/kg
	Getchell Low Grade	0.53	BD
	Getchell High Grade	4.88	BD
	Getchell High Grade dup	4.97	BD
	Getchell Low Grade Dup	0.53	BD
	MB	<0.02	BD
CRM 62j	10.54 Au 7.69 Ag	10.30	10

Analysis Method: litharge fire assay fusion with aqua regia dissolution of prill and final analysis by AAS for gold. Silver by calculation

Reviewed and Approved by: ______________________

Jessica Axen
Director of Exploration Services

Reporting Limit:	1AT	1/2 AT
Gold (mg/kg):	0.02	0.04
Silver (mg/kg):	0.3	7

Reporting Sheet
23041

Location: Wheat Ridge
Department: Exploration

Batch ID: 23041-11 Oxide Shipment 2 Head
Report Date: 7/8/2024
Report to: JA/DM

Preliminary Analytical Report

Project 23041 Getchell Gold

Lab ID	Sample Name	Gold mg/kg	Silver mg/kg
	Oxide Shipment 2 Head	1.88	2
	Oxide Shipment 2 Head Dup	1.84	2
MB		<0.02	<0.3
CRM 609b	4.97 Au, 24.6 Ag	4.94	20

Analysis Method: litharge fire assay fusion with aqua regia dissolution of prill and final analysis by AAS for gold. Silver by calculation

Reviewed and Approved by: ______________________

Jessica Axen
Director of Exploration Services

eporting Limit:	1AT	1/2 AT
Gold (mg/kg):	0.02	0.04
Silver (mg/kg):	0.3	7

Batch ID: 23041-14 Oxide Head Comp 1
Report Date: 7/14/2024
Report to: JA/DM

Analytical Report

Project 23041 Getchell Gold

Lab ID	Sample Name	Gold mg/kg	Silver mg/kg
	Getchell Oxide Head Comp 1	1.38	2
	Getchell Oxide Head Comp 1 du	1.39	<0.3
	MB	<0.02	<0.3
CRM 609b	4.97 Au, 24.6 Ag	5.19	30

Analysis Method: litharge fire assay fusion with aqua regia dissolution of prill and final analysis by AAS for gold. Silver by calculation

Reviewed and Approved by: ______________________

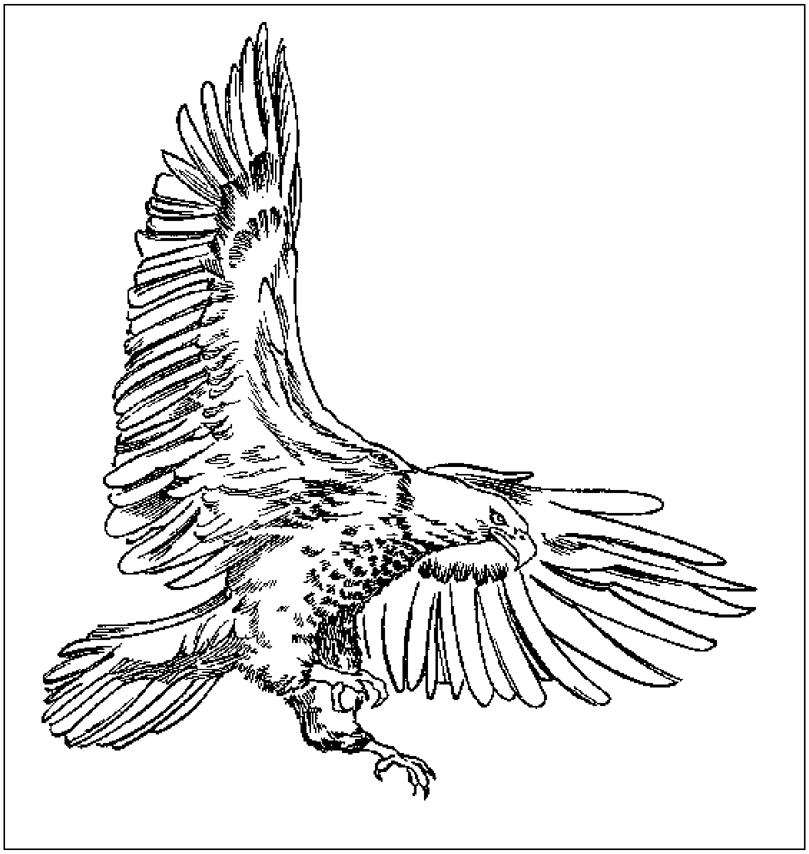
Jessica Axen
Director of Exploration Services

Reporting Limit:	1AT	1/2 AT
Gold (mg/kg):	0.02	0.04
Silver (mg/kg):	0.3	7

APPENDIX C: MINERALOGY REPORT FOR AVERAGE GRADE SULFIDE

AMICS ANALYSIS

FORTE ANALYTICAL



Eagle Engineering

April 24, 2024

AMICS ANALYSIS

Prepared for: Brendan Fetter

Project: Getchel Modal and Gold Analysis

Email: bfetter@forteanalytical.com

By

Paul Miranda, PhD
Chief Metallurgist/Mineralogist
E-mail: eaglemt711@gmail.com

April 24, 2024

EXECUTIVE SUMMARY

Three (3) samples, Getchel Average Grade, Getchel Low Grade, ad Getchel High Grade were received for AMICS analysis. The scope of work was to determine modal mineralogy for all samples. Secondly, a brightness search was conducted to determine gold containing minerals. From AMICS data, backscatter images of gold containing minerals were placed into the report.

Modal mineralogy and phase analysis were determined using energy dispersive x-ray analysis (EDX) along with associated AMICS mineralogy. According to the data, two major minerals, quartz and orthoclase, were identified with minor amounts of ankerite, clinochlore, muscovite, dolomite, and pyrite.

A brightness search was conducted on received samples for determination of gold containing minerals. According to the data, Getchel Average and Getchel Low grade gold particles were observed at approximately 5 microns and associated with pyrite. For Getchel High Grade sample, two free gold particles were identified at approximately 5 microns in size.

Paul Miranda, Ph. D
April 24, 2024

Qualifying Statement

This confidential report was prepared for Forte Analytical and is based on information available at the time of the report preparation. It is believed the information, estimates, conclusions and recommendations contained herein are reliable under the conditions and subject to the qualifications set forth herein. The information, estimates, conclusions and recommendations herein are based on our experience and data supplied by others, but the actual result of the work is dependent, in part, on factors over which we have no control. This report is intended to be used exclusively by Forte Analytical. We disclaim any assumption of responsibility for any reliance on this report by any person other than Forte Analytical, or for any purpose other than that for which it was prepared. We disclaim all liability to any other party for all costs, losses, damages, and liabilities that the other party might suffer or incur arising from or relating to the contents of this report, the provision of this report to the other party, or the reliance on this report by the other party.

Scope of Work

Three (3) samples, Getchel Average Grade, Getchel Low Grade, ad Getchel High Grade were received for AMICS analysis. The scope of work was to determine modal mineralogy for all samples. Secondly, a brightness search was conducted to determine gold containing minerals. From AMICS data, backscatter images of gold containing minerals were placed into the report.

Experimental Work and Results

For received samples, they were mounted, polished, and carbon coated for AMICS analysis. For AMICS analysis, minerals were determined. Next, additional analysis for gold particles were performed to determine gold minerals and associations.

Modal Mineralogy

Modal mineralogy and phase analysis were determined using energy dispersive x-ray analysis (EDX) along with associated AMICS mineralogy. Results are shown table 1. According to the data, two major minerals, quartz and orthoclase, were identified with minor amounts of ankerite, clinochlore, muscovite, dolomite, and pyrite.

Table 1. Modal Mineralogy.

Mineral	Chemistry	Getchel Average Grade	Getchel Low Grade	Getchel High Grade
Albite	$NaAlSi_3O_8$	0.07	0.09	0.07
Almandine	Al_2SiO_5	0.10	0.12	0.04
Andalusite	$Ca,Mn,Fe(CO_3)_2$	0.04	0.04	0.02
Ankerite	$Ca_5(PO_4)_3OH$	2.85	2.00	1.13
Anorthite	FeAsS	0.06	0.07	0.01
Apatite	$K(Mg,Fe)_3AlSi_3O_{10}(OH)_2$	0.22	0.11	0.03
Arsenopyrite	FeAsS	0.09	0.20	0.16
Barite	$BaSO_4$	0.03	0.01	0.00
Calcite	$CaCO_3$	0.95	2.36	0.16
Clinochlore	$(Mg,Fe)_5AlSi_3O_{10}(OH)_8$	1.46	1.54	0.55
Diopside	$CaMgSi_2O_6$	0.08	0.01	0.01
Dolomite	$Ca,Mg(CO_3)_2$	2.29	0.80	1.33
Epidote	$Ca_2(Fe,Al)_3Si_3O_{12}(OH)$	0.07	0.07	0.02
Hedenbergite	$CaFeSi_2O_6$	0.05	0.05	0.01
Hematite	Fe_2O_3	0.02	0.01	0.00
Molybdenite	MoS_2	0.11	0.00	0.00
Muscovite	$KAl_2Si_3O_{10}(OH)_2$	6.59	8.29	3.27
Orthoclase	$KAlSi_3O_8$	34.02	37.33	32.54
Plagioclase	$(Na,Ca)AlSi_3O_8$	0.03	0.02	0.01
Pyrite	FeS_2	4.16	5.22	1.75
Quartz	SiO_2	46.47	41.58	58.81
Rutile	TiO_2	0.09	0.04	0.07
Siderite	$FeCO_3$	0.01	0.01	0.00
Titanite	$CaSiTiO_5$	0.02	0.01	0.01
Wollastonite	$CaSiO_3$	0.03	0.02	0.00
Zircon	$ZrSiO_4$	0.09	0.00	0.00

Brightness Search

A brightness search was conducted on received samples for determination of gold containing minerals. Results are shown in figures 1 through 4. According to the data, Getchel Average and Getchel Low grade gold particles were observed at approximately 5 microns and associated with pyrite. For Getchel High Grade sample, two free gold particles were identified at approximately 5 microns in size.

Figure 1. Gold Particle Associated with Pyrite for Getchel Average Grade Sample.

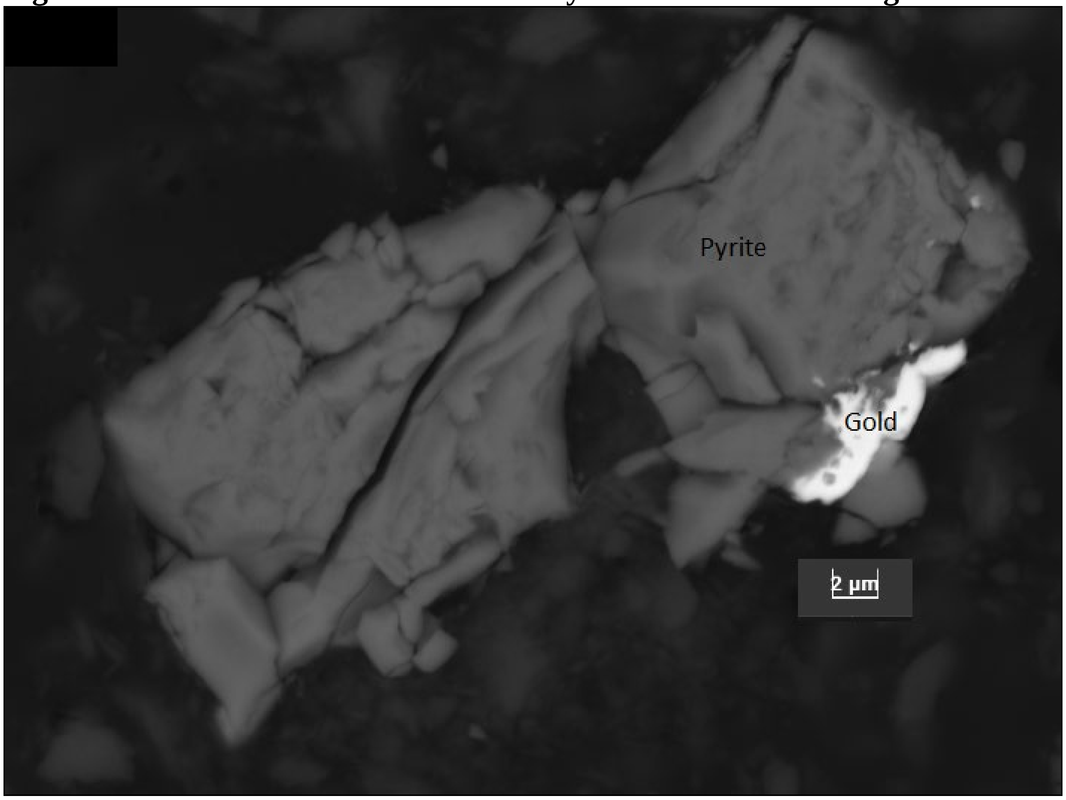


Figure 2. Gold Particle Encapsulated by Pyrite for Getchel Low Grade Sample.

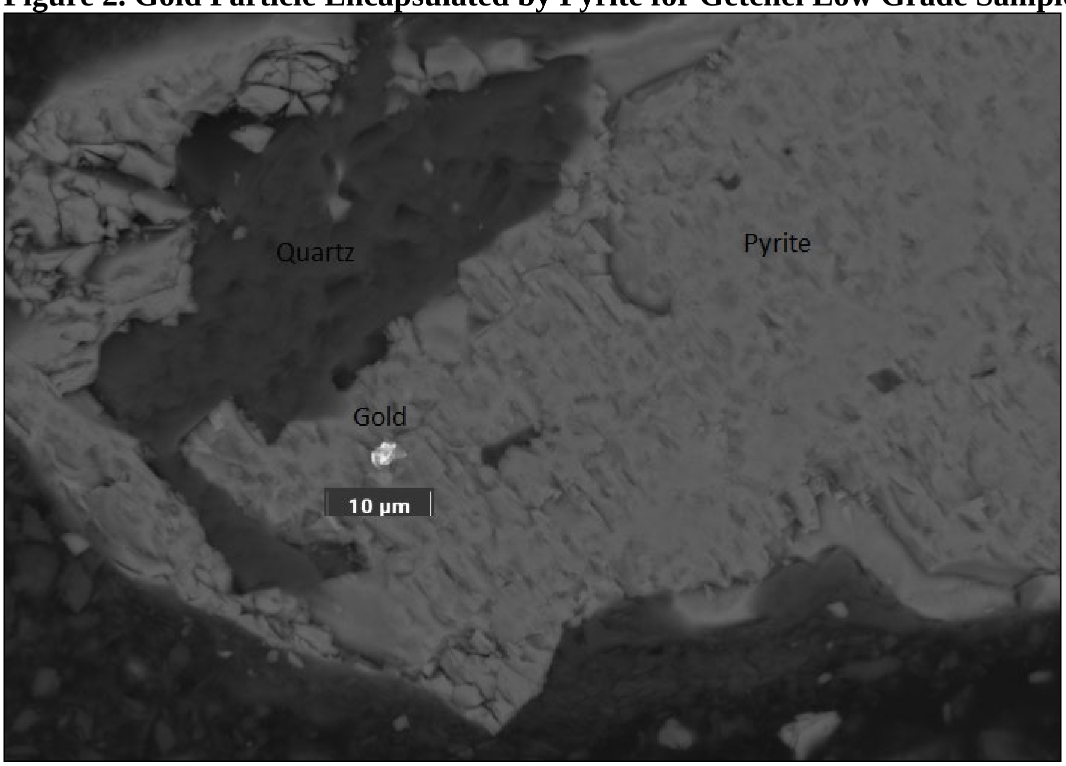


Figure 3. Free Gold Particle for Getchel High Grade Sample.

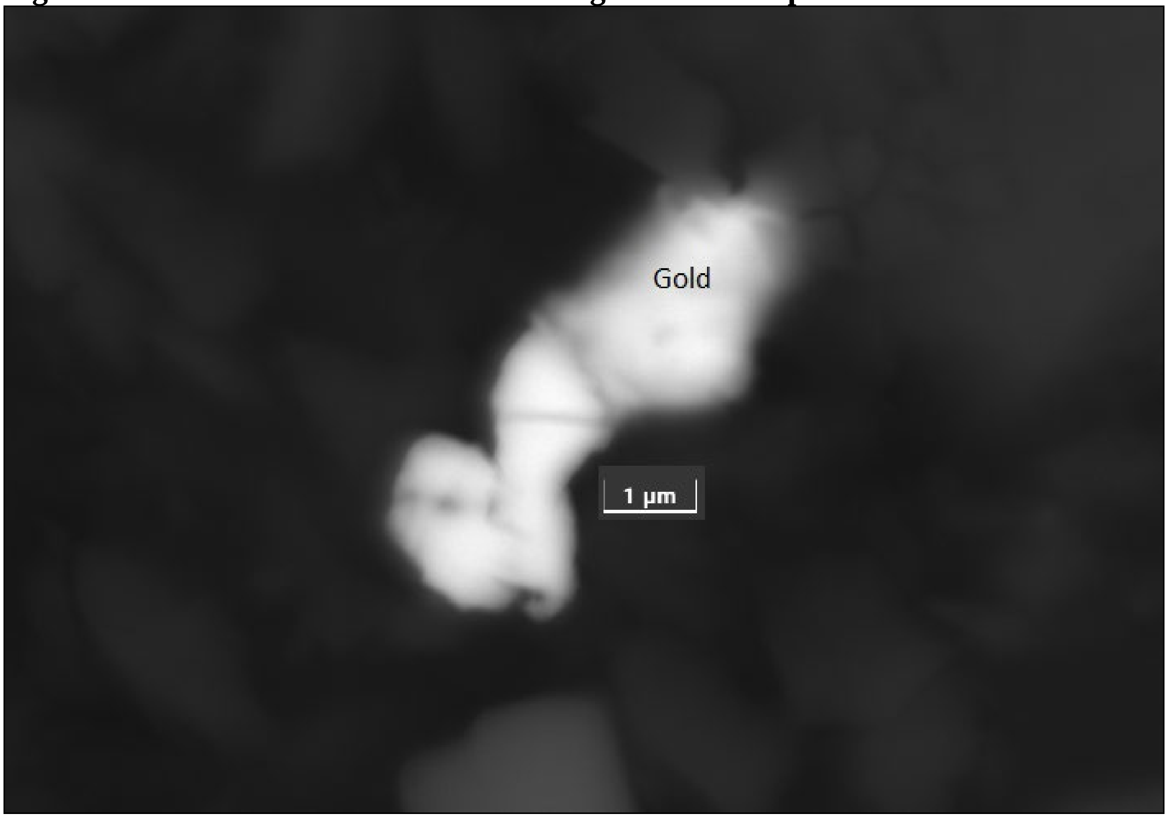
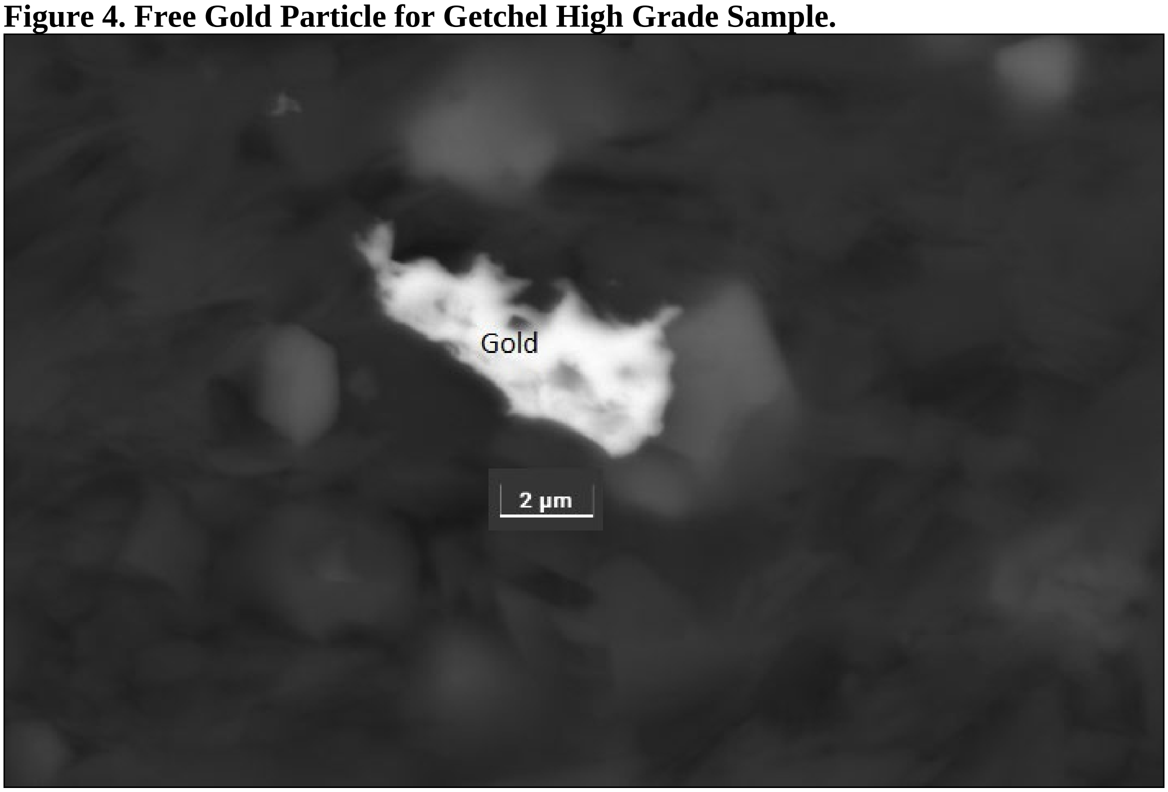


Figure 4. Free Gold Particle for Getchel High Grade Sample.



APPENDIX D: BOND'S BALL MILL WORK INDEX DATA

Appendix D: Bond Work Index

FORTE ANALYTICAL	**Engineer**	JA	**Test ID**	Average Grade
	Technician	TD/MR	**Date**	6/10/2024
	Project Name	Getchell Gold		
	Project No.	23041		

Purpose: To determine the bond work index that can be used to size grind and mills for the project's comminution circuit based on the selected P80.

Procedure: The feed samples were screened and stage-crushed to minus 3,350 microns as per test specification.The Bwi results were run with a closing size of 150 microns. Several quality-control measured, as well as rigorous closing criteria were followed including; minimum of 6 cycles, average grams per mill revolution was less than 3% for last three (3) cycles with inflection, target weight of undersize within 4-10 grams, circulating load ratio of 2.47 or higher, last cycle was wet screened for product P80 size (semi-log interpolated).

Sample: Average Grade

Results:

Bwi - From Graph	15.71	**BWi - Avg, kWh/st**	15.54
Bwi - Interpolated	15.37		

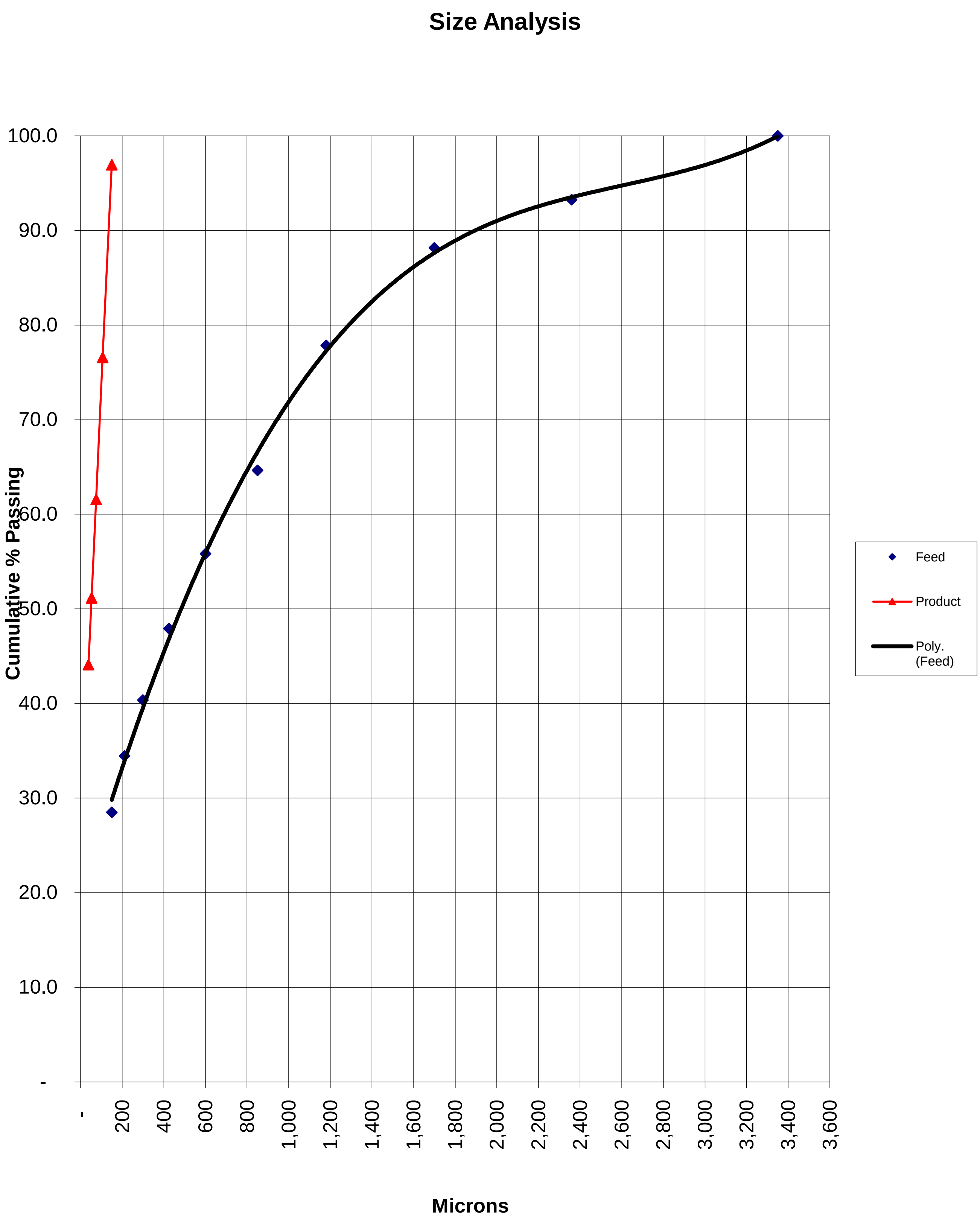
Size		Mill Feed			
Tyler Mesh	Microns	Weight (g)	Weight %	Retained %	Passing %
6	3,350	0.0	-	-	100.0
8	2,360	14.6	6.8	6.8	93.2
10	1,700	11.0	5.1	11.8	88.2
14	1,180	22.2	10.3	22.1	77.9
20	850	28.5	13.2	35.4	64.6
28	600	19.0	8.8	44.2	55.8
35	425	17.1	7.9	52.1	47.9
48	300	16.3	7.6	59.6	40.4
65	212	12.7	5.9	65.5	34.5
100	150	12.9	6.0	71.5	28.5
Pan		61.4	28.5	100.0	
Total		215.5	100.0		

Cycle #	grams / revolution
n-2	1.492
n-1	1.492
n	1.477
Average	1.487

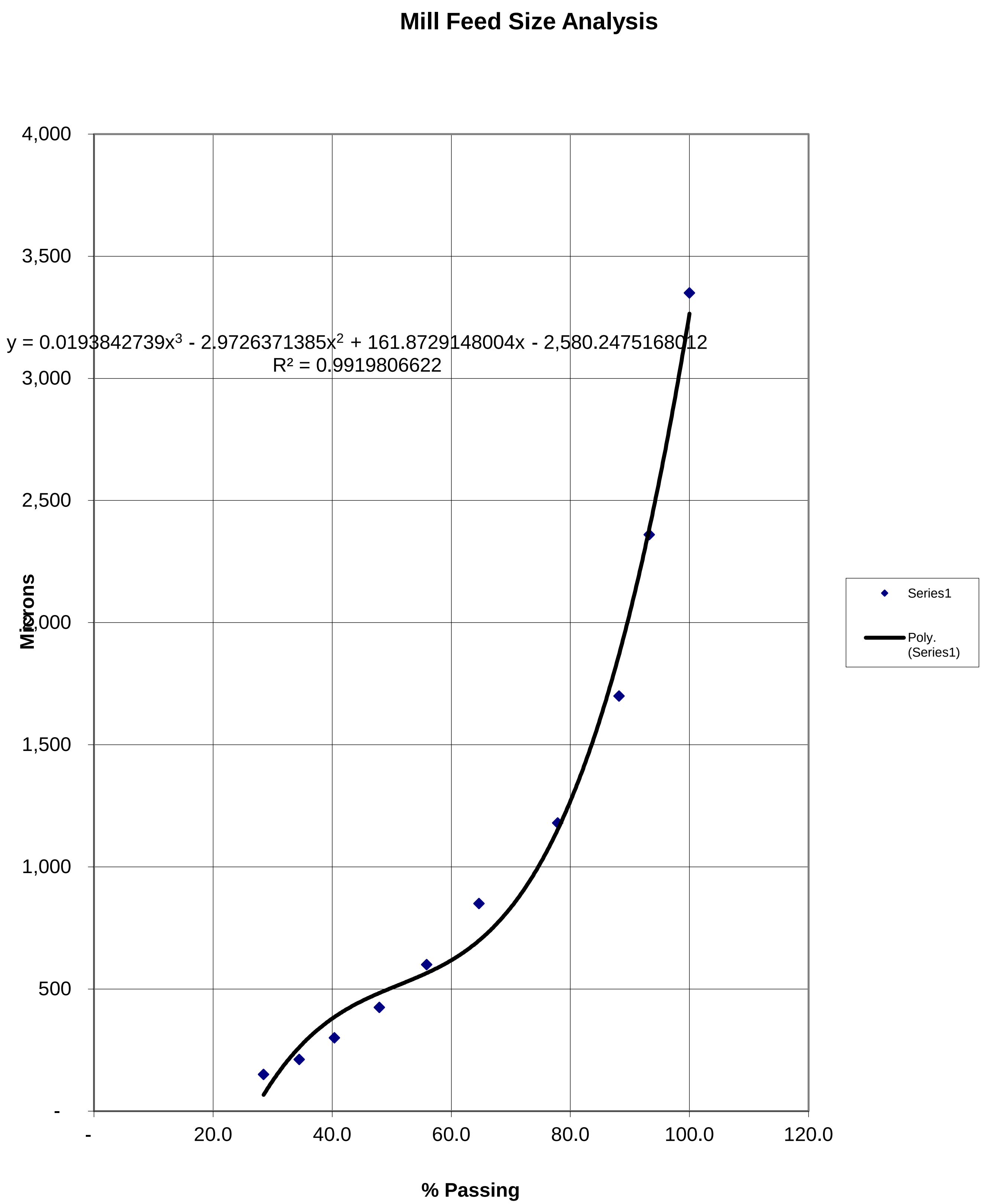
Mill Feed Weight (grams)	1300.1
Desired Mesh of Grind:	100
Desired Micron of Grind:	150
Circulating Load (%)	250
Circulating Load (grams):	371.5

Size		Ground Product			
Tyler Mesh	Microns	Weight (g)	Weight %	Retained %	Passing %
48	300	-			
65	212	-	-	-	100.0
100	150	11.80	3.1	3.1	96.9
150	106	78.6	20.4	23.4	76.6
200	75	58.0	15.0	38.4	61.6
270	53	40.3	10.4	48.9	51.1
400	38	27.2	7.0	55.9	44.1
Pan	0	170.3	44.1	100.0	-
Total		386.2	100.0		

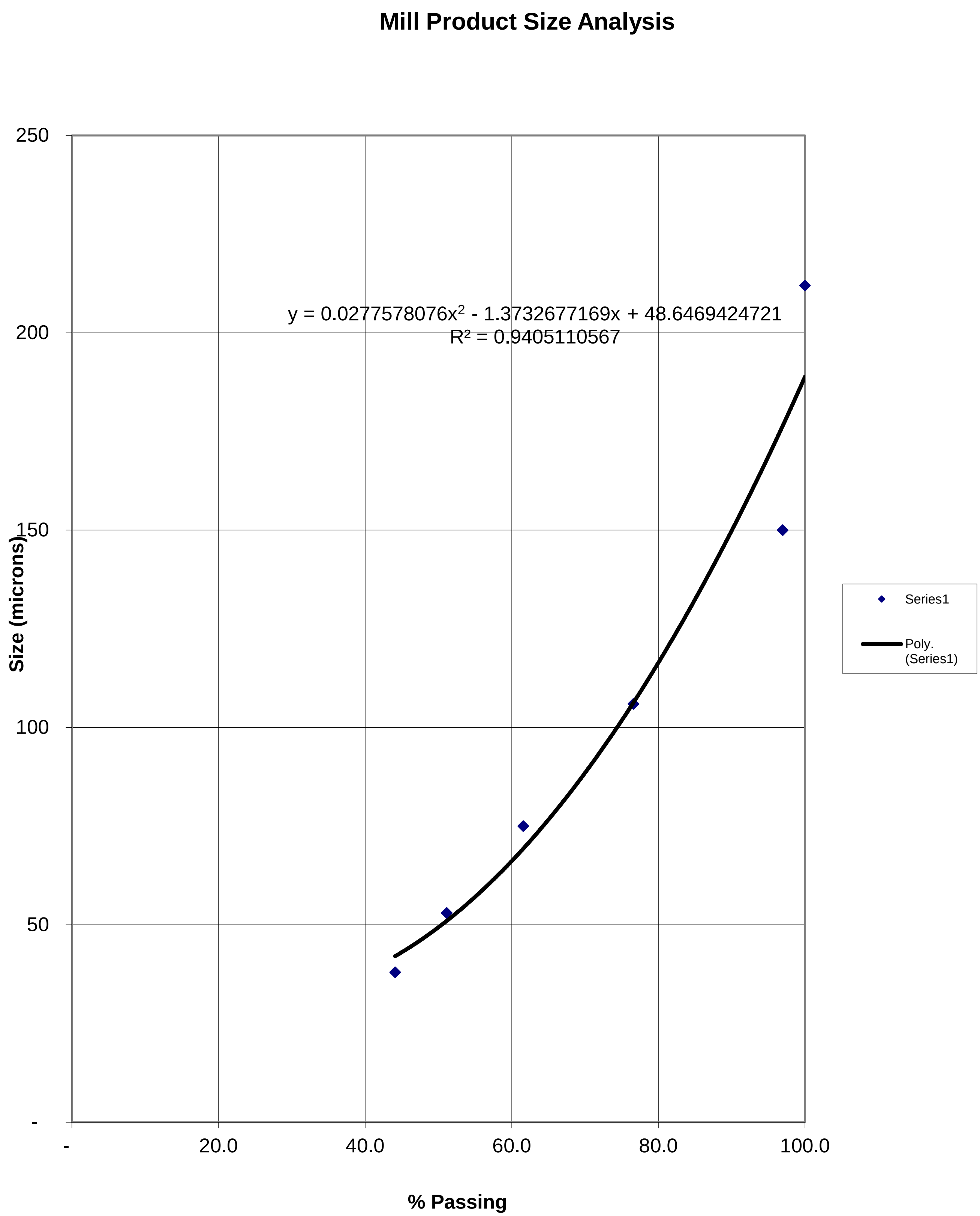
	Interpolated	Graphic
F80	1,288	1,269
P80	113	116



Size Analysis
Cumulative % Passing
100.0
90.0
80.0
70.0
60.0
50.0
40.0
30.0
20.0
10.0
-
-
200
400
600
800
1,000
1,200
1,400
1,600
1,800
2,000
2,200
2,400
2,600
2,800
3,000
3,200
3,400
3,600
Microns
Feed
Product
Poly. (Feed)



Mill Feed Size Analysis
4,000
3,500
3,000
2,500
2,000
1,500
1,000
500
-
Microns
y = 0.0193842739x3 - 2.9726371385x2 + 161.8729148004x - 2,580.2475168012
R² = 0.9919806622
-
20.0
40.0
60.0
80.0
100.0
120.0
% Passing
Series1
Poly. (Series1)


Mill Product Size Analysis
y = 0.0277578076x2 - 1.3732677169x + 48.6469424721
R² = 0.9405110567
Size (microns)
% Passing
250
200
150
100
50
-
-
20.0
40.0
60.0
80.0
100.0
Series1
Poly. (Series1)

APPENDIX D

Bond Ball Mill

Feed

$y = 0.0193842739x^3 - 2.9726371385x^2 + 161.8729148004x - 2580.2475168012$
$R^2 = 0.991981$

Y	X
1269.45622	80

	Microns	% Passing
6	3350	100.00
8	2360	93.24
10	1700	88.15
14	1180	77.85
20	850	64.65
28	600	55.85
35	425	47.93
48	300	40.36
100	150	28.49

Product

$y = 0.0312056748x^2 - 1.8831263070x + 69.9244658479$
$R^2 = 0.9686975$

Y	X
116.435494	80

Microns	% Passing
212	100.0
150	96.9
106	76.6
75	61.6
53	51.1
38	44.1

Average wt of 3 - 700 ml samples	
A1 wt (1)	1418.9
A1 wt (2)	1439.8
A1 wt (3)	1424.4
Average	1427.7

Y= A1 divided by 3.5 for 250% circulating load
X= average wt % of undersize from screening

Y=	407.9
X=	28.5

Variable	A	B	C	D	E	F	G
Calcs	average wt of 3 700 ml samples	A multiplied by X	Y minus B	C1 divided by 1.2	wt of product	E minus B	F divided by D
	A2 = E1, A3 = E2, etc.			C2 divided by G1			

		Undersize (g)				Mill Production (g)	
Cycle #	Feed (g)	In Feed	To be Produced	Mill Revs	Wt of Undersize (g)	Total Net	Per Rev
1	1427.7	406.7	1.2	25	505.0	98.3	3.931
2	505	143.9	264.1	67	288.9	145.0	2.159
3	288.9	82.3	325.6	151	342.9	260.6	1.728
4	342.9	97.7	310.2	180	371.9	274.2	1.528
5	371.9	105.9	302.0	198	444.2	338.3	1.711
6	444.2	126.5	281.4	164	389.3	262.8	1.598

S I Z E			M I L L F E E D				
Tyler Mesh	Microns	(g)	Screen #2 Weight (g)	Screen #3 Weight (g)	Weight %	Retained %	Passing %
6	3,350	0.0	0.0	0.0	-	-	100.0
8	2,360	12.1	16.6	15.0	6.8	6.8	93.2
10	1,700	10.6	11.1	11.2	5.1	11.8	88.2
14	1,180	22.4	22.2	22.0	10.3	22.1	77.9
20	850	28.0	29.9	27.5	13.2	35.4	64.6
28	600	19.1	19.0	18.8	8.8	44.2	55.8
35	425	17.3	16.9	17.0	7.9	52.1	47.9
48	300	16.3	16.1	16.5	7.6	59.6	40.4
65	212	12.6	12.7	12.9	5.9	65.5	34.5
100	150	12.8	12.6	13.2	6.0	71.5	28.5
Pan		61.3	61.3	61.6	28.5	100.0	
Total		212.5	218.4	215.7	100.0		

Appendix D: Bond Work Index

	Engineer	JA	**Test ID**	High Grade
	Technician	TD/MR	**Date**	6/12/2024
	Project Name	Getchell Gold		
	Project No.	23041		

Purpose: To determine the bond work index that can be used to size grind and mills for the project's comminution circuit based on the selected P80.

Procedure: The feed samples were screened and stage-crushed to minus 3,350 microns as per test specification.The Bwi results were run with a closing size of 150 microns. Several quality-control measured, as well as rigorous closing criteria were followed including; minimum of 6 cycles, average grams per mill revolution was less than 3% for last three (3) cycles with inflection, target weight of undersize within 4-10 grams, circulating load ratio of 2.47 or higher, last cycle was wet screened for product P80 size (semi-log interpolated).

Sample: High Grade

Results:

Bwi - From Graph	15.55	**BWi - Avg, kWh/st** 15.46
Bwi - Interpolated	15.36	

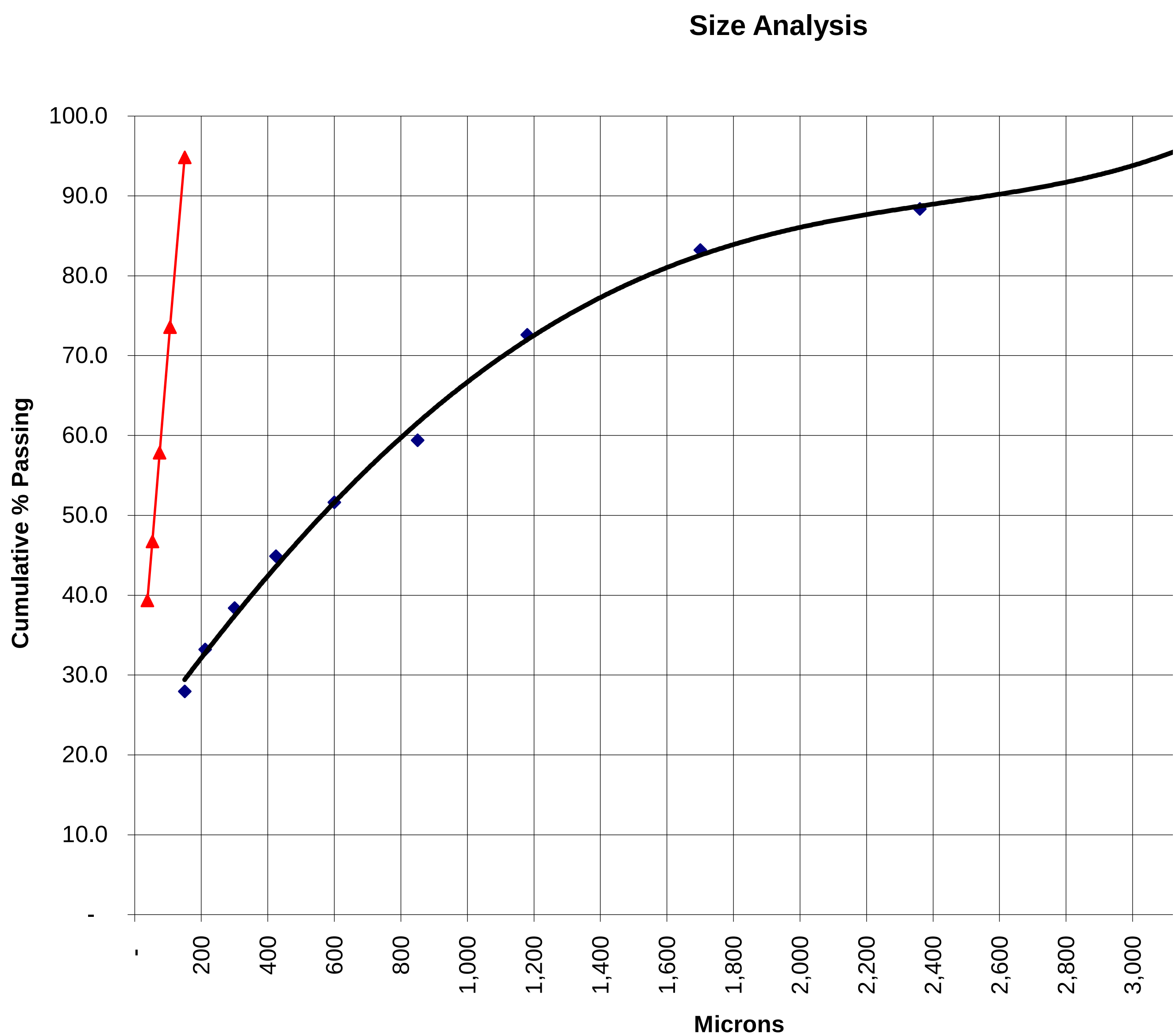
Size		Mill Feed			
Tyler Mesh	Microns	Weight (g)	Weight %	Retained %	Passing %
6	3,350	0.0	-	-	100.0
8	2,360	24.9	11.6	11.6	88.4
10	1,700	11.1	5.2	16.8	83.2
14	1,180	22.8	10.6	27.4	72.6
20	850	28.3	13.2	40.6	59.4
28	600	16.7	7.8	48.4	51.6
35	425	14.5	6.7	55.1	44.9
48	300	13.9	6.5	61.6	38.4
65	212	11.1	5.2	66.8	33.2
100	150	11.3	5.3	72.0	28.0
Pan		59.9	28.0	100.0	
Total		214.3	100.0		

Cycle #	grams / revolution
n-2	1.492
n-1	1.492
n	1.477
Average	1.487

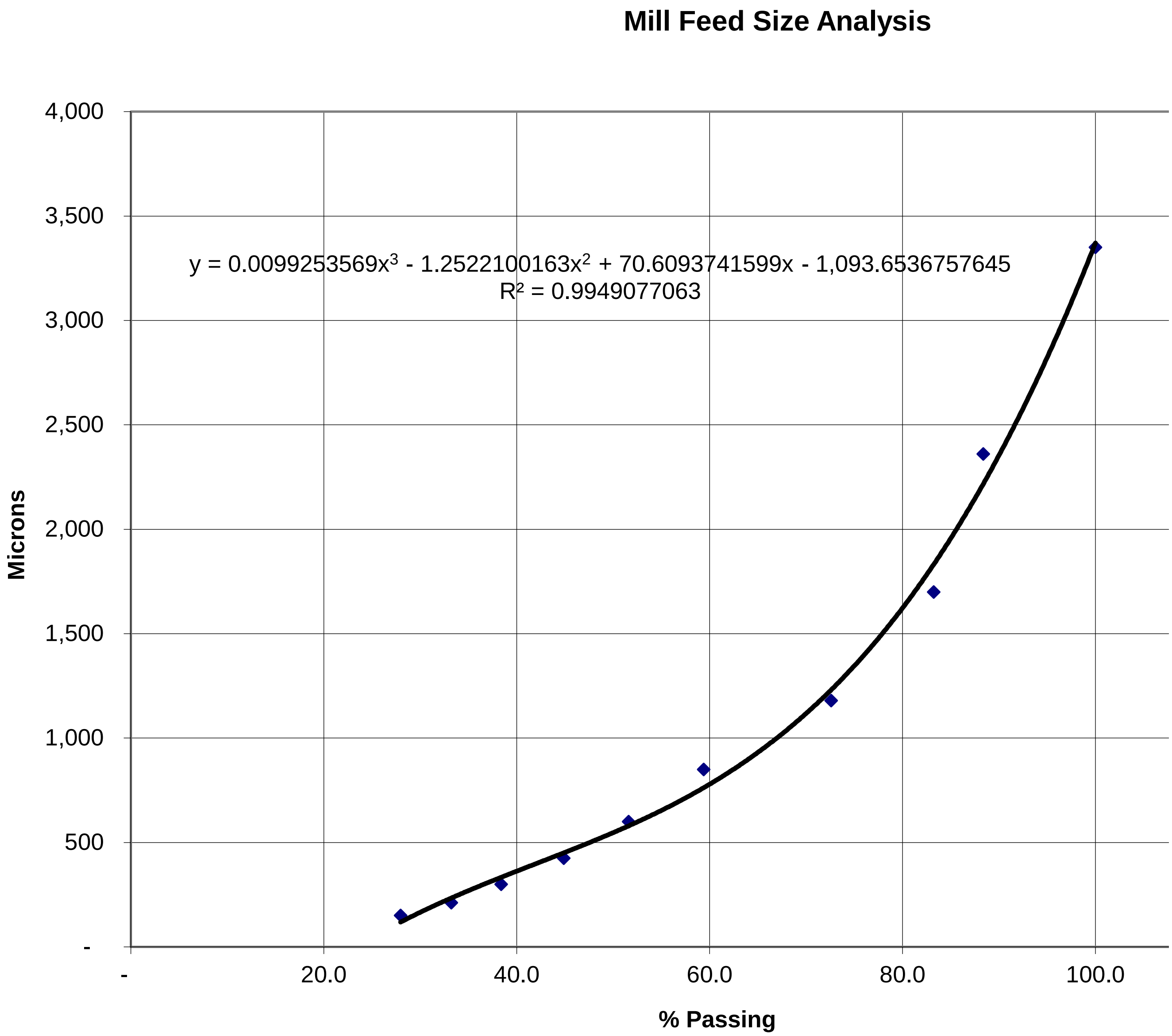
Mill Feed Weight (grams)	1300.1
Desired Mesh of Grind:	100
Desired Micron of Grind:	150
Circulating Load (%)	250
Circulating Load (grams):	371.5

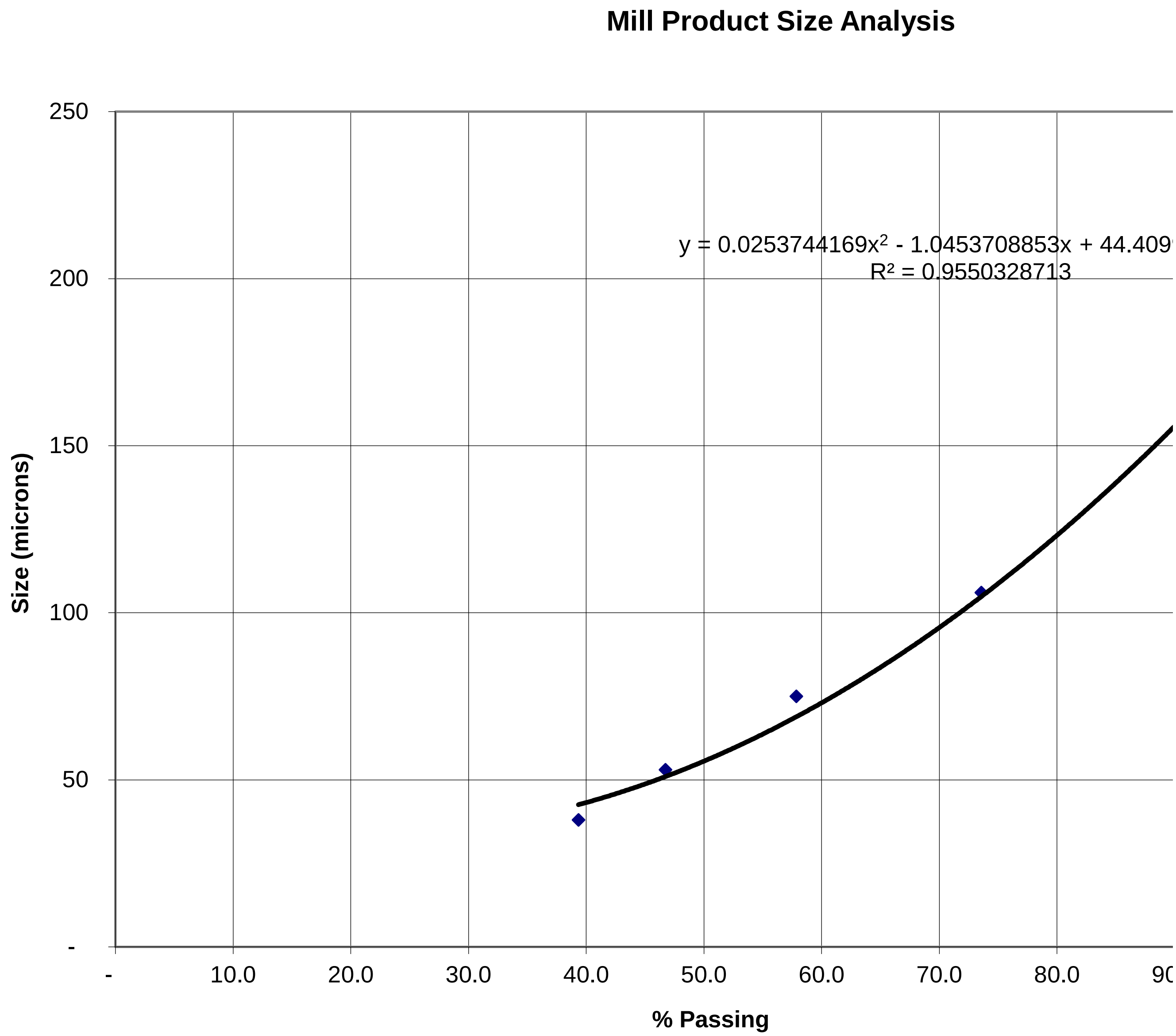
Size		Ground Product			
Tyler Mesh	Microns	Weight (g)	Weight %	Retained %	Passing %
48	300	-			
65	212	0.40	0.1	0.1	99.9
100	150	20.20	5.1	5.2	94.8
150	106	85.0	21.3	26.4	73.6
200	75	62.8	15.7	42.1	57.9
270	53	44.4	11.1	53.3	46.7
400	38	29.6	7.4	60.7	39.3
Pan	0	157.2	39.3	100.0	-
Total		399.6	100.0		

	Interpolated	Graphic
F80	1,542	1,623
P80	119	123



Size Analysis
100.0
90.0
80.0
70.0
60.0
50.0
40.0
30.0
20.0
10.0
-
Cumulative % Passing
-
200
400
600
800
1,000
1,200
1,400
1,600
1,800
2,000
2,200
2,400
2,600
2,800
3,000
Microns


Mill Feed Size Analysis
y = 0.0099253569x3 - 1.2522100163x2 + 70.6093741599x - 1,093.6536757645
R2 = 0.9949077063
4,000
3,500
3,000
2,500
2,000
1,500
1,000
500
-
Microns
-
20.0
40.0
60.0
80.0
100.0
% Passing



Mill Product Size Analysis
y = 0.0253744169x² - 1.0453708853x + 44.409
R² = 0.9550328713
Size (microns)
250
200
150
100
50
-
-
10.0
20.0
30.0
40.0
50.0
60.0
70.0
80.0
% Passing

APPENDIX D

Bond Ball Mill

Feed

$y = 0.0099253569x^3 - 1.2522100163x^2 + 70.60937415992x - 1093.6536757645$
$R^2 = 0.994908$

Y	X
1622.73489	80

	Microns	% Passing
6	3350	100.00
8	2360	88.38
10	1700	83.22
14	1180	72.60
20	850	59.39
28	600	51.62
35	425	44.87
48	300	38.38
100	150	27.96

Product

$y = 0.0253744169x^2 - 1.0453708853x + 44.4099958938$
$R^2 = 0.966033$

Y	X
123.176593	80

Microns	% Passing
212	99.9
150	94.8
106	73.6
75	57.9
53	46.7
38	39.3

Average wt of 3 - 700 ml samples	
A1 wt (1)	1456.9
A1 wt (2)	1432.0
A1 wt (3)	1466.9
Average	1451.9

Y= A1 divided by 3.5 for 250% circulating load
X= average wt % of undersize from screening

Y=	414.8
X=	28.0

Variable	A	B	C	D	E	F	G
Calcs	average wt of 3 700 ml samples	A multiplied by X	Y minus B	C1 divided by 1.2	wt of product	E minus B	F divided by D
	A2 = E1, A3 = E2, etc.			C2 divided by G1			

		Undersize (g)				Mill Production (g)	
Cycle #	Feed (g)	In Feed	To be Produced	Mill Revs	Wt of Undersize (g)	Total Net	Per Rev
1	1451.9	406.0	8.8	25	506.1	100.1	4.004
2	506.1	141.5	273.3	68	252.5	111.0	1.626
3	252.5	70.6	344.2	212	396.5	325.9	1.539
4	396.5	110.9	304.0	197	441.2	330.3	1.673
5	441.2	123.4	291.5	174	425.8	302.4	1.736
6	425.8	119.1	295.8	170	404.0	284.9	1.672
7	404	113.0	301.9	181		-113.0	-0.626

S I Z E			M I L L F E E D				
Tyler Mesh	Microns	(g)	Screen #2 Weight (g)	Screen #3 Weight (g)	Weight %	Retained %	Passing %
6	3,350	0.0	0.0	0.0	-	-	100.0
8	2,360	20.5	28.0	26.2	11.6	11.6	88.4
10	1,700	9.6	12.6	11.0	5.2	16.8	83.2
14	1,180	21.3	24.2	22.8	10.6	27.4	72.6
20	850	28.2	29.4	27.3	13.2	40.6	59.4
28	600	17.4	16.7	15.9	7.8	48.4	51.6
35	425	15.2	14.4	13.8	6.7	55.1	44.9
48	300	15.0	13.2	13.5	6.5	61.6	38.4
65	212	11.7	10.4	11.1	5.2	66.8	33.2
100	150	11.8	10.4	11.6	5.3	72.0	28.0
Pan		62.7	56.0	61.1	28.0	100.0	
Total		213.4	215.3	214.3	100.0		

Appendix D: Bond Work Index

Engineer	JA	**Test ID**	Low Grade
Technician	TK	**Date**	4/24/2024
Project Name	Getchell		
Project No.	23041		

Purpose: To determine the bond work index that can be used to size grind and mills for the project's comminution circuit based on the selected P80.

Procedure: The feed samples were screened and stage-crushed to minus 3,350 microns as per test specification.The Bwi results were run with a closing size of 150 microns. Several quality-control measured, as well as rigorous closing criteria were followed including; minimum of 6 cycles, average grams per mill revolution was less than 3% for last three (3) cycles with inflection, target weight of undersize within 4-10 grams, circulating load ratio of 2.47 or higher, last cycle was wet screened for product P80 size (semi-log interpolated).

Sample: Low Grade

Results:

Bwi - From Graph	15.38	**BWi - Avg, kWh/st**	15.82
Bwi - Interpolated	16.26		

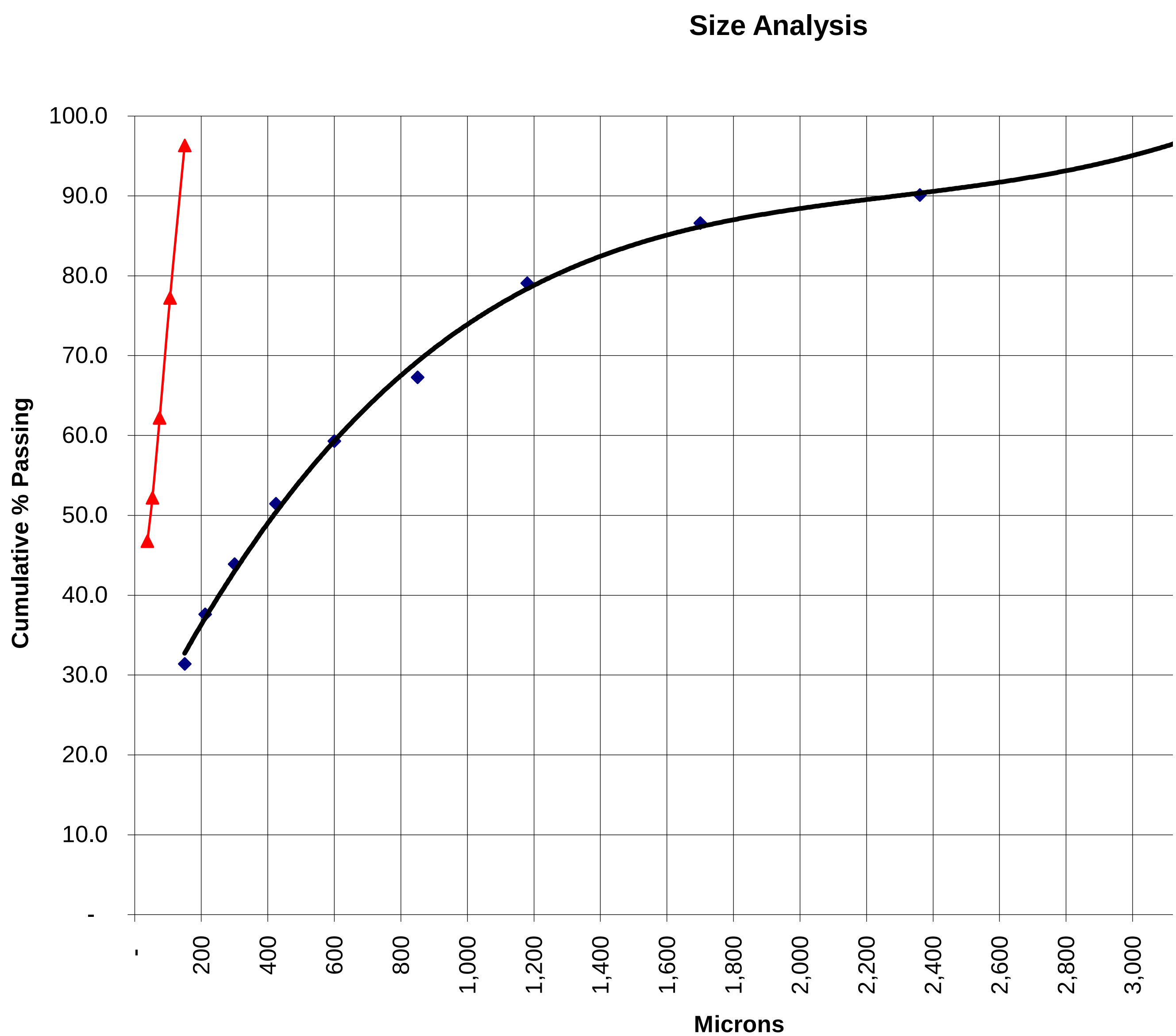
Size		Mill Feed			
Tyler Mesh	Microns	Weight (g)	Weight %	Retained %	Passing %
6	3,350	0.0	-	-	100.0
8	2,360	21.2	9.9	9.9	90.1
10	1,700	7.5	3.5	13.4	86.6
14	1,180	16.1	7.5	20.9	79.1
20	850	25.3	11.8	32.7	67.3
28	600	17.1	8.0	40.7	59.3
35	425	16.8	7.8	48.5	51.5
48	300	16.2	7.6	56.1	43.9
65	212	13.5	6.3	62.4	37.6
100	150	13.3	6.2	68.6	31.4
Pan		67.3	31.4	100.0	
Total		214.4	100.0		

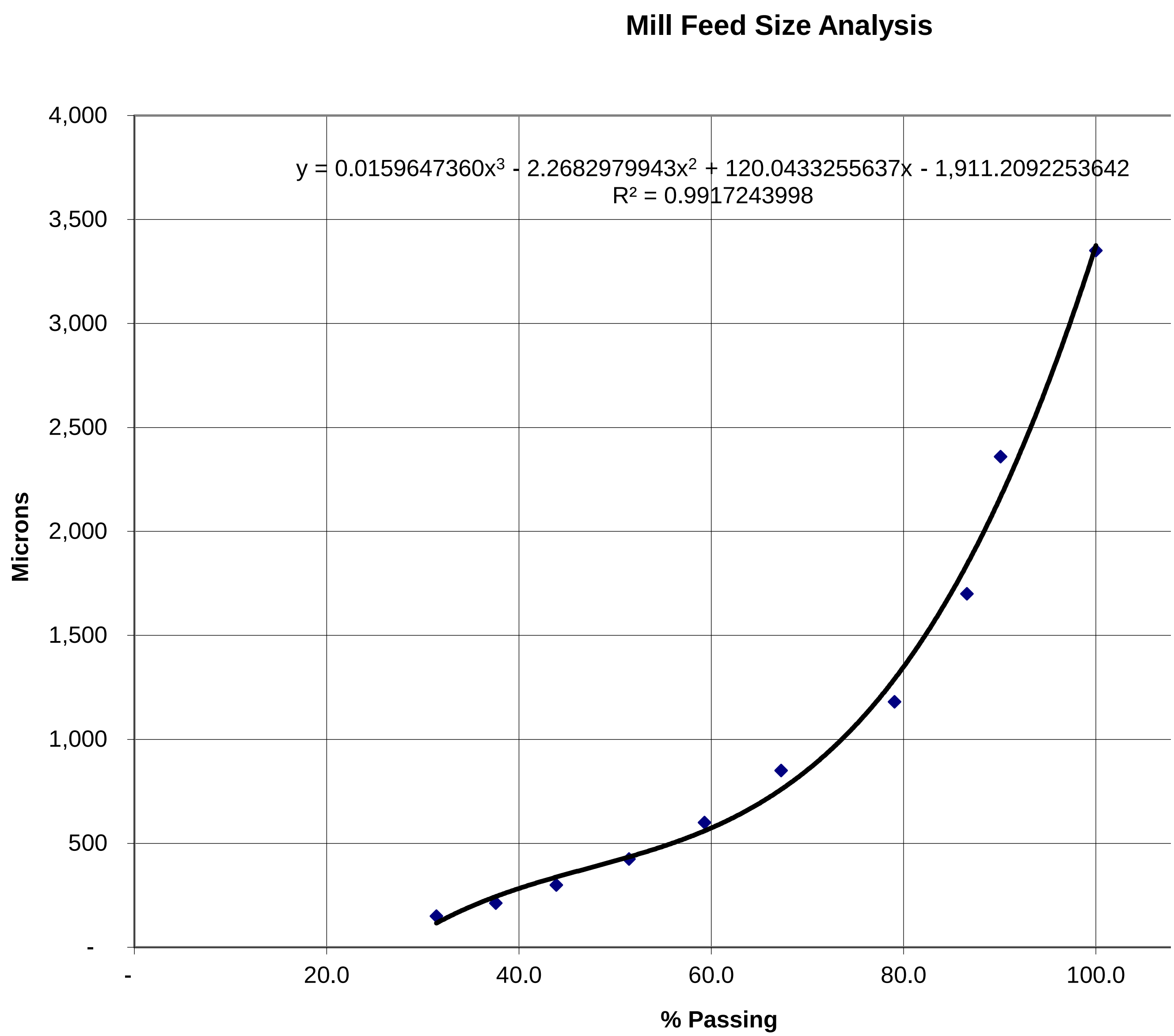
Cycle #	grams / revolution
n-2	1.492
n-1	1.492
n	1.477
Average	1.487

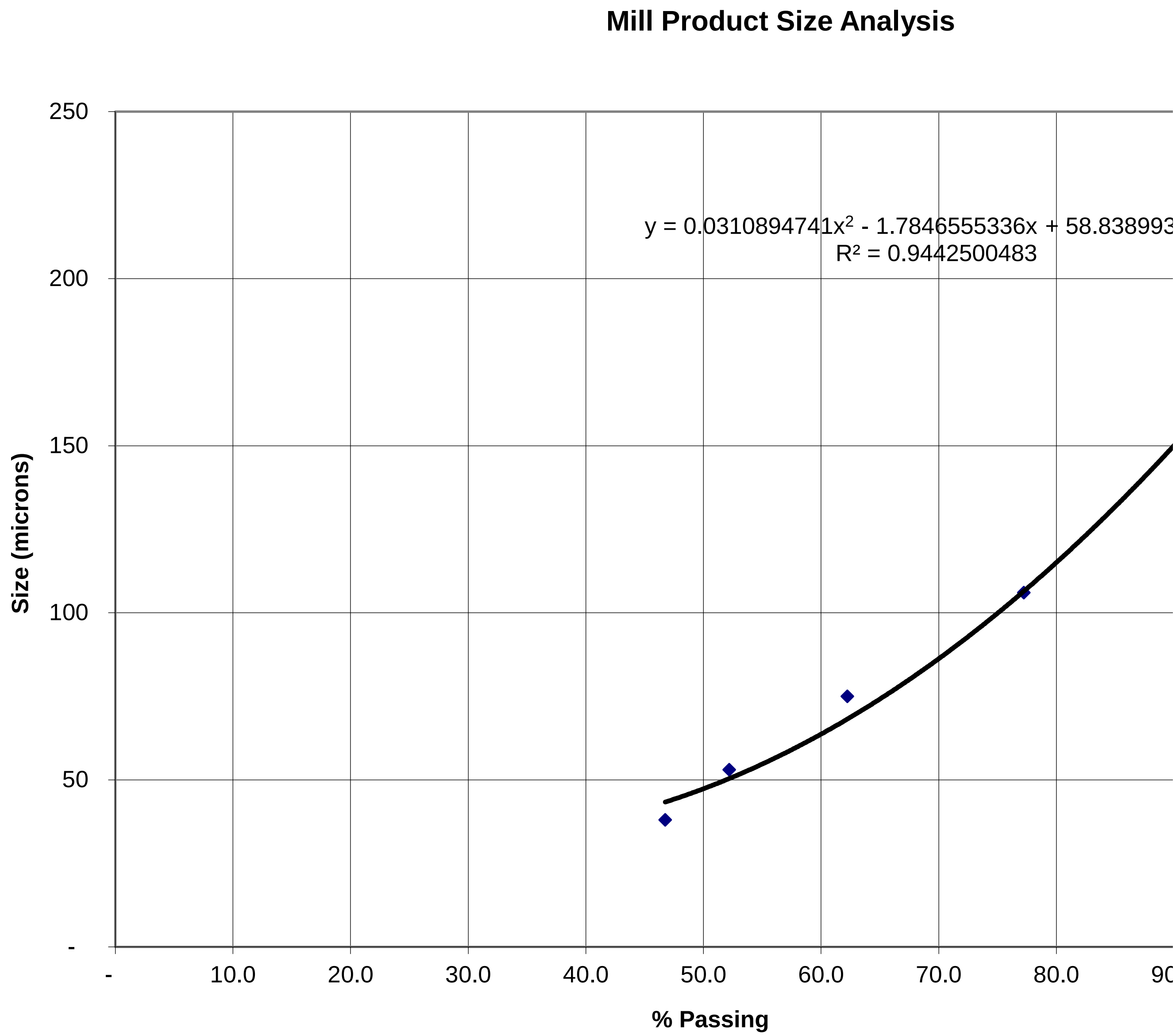
Mill Feed Weight (grams)	1300.1
Desired Mesh of Grind:	100
Desired Micron of Grind:	150
Circulating Load (%)	250
Circulating Load (grams):	371.5

Size		Ground Product			
Tyler Mesh	Microns	Weight (g)	Weight %	Retained %	Passing %
48	300	-			
65	212	0.90	0.2	0.2	99.8
100	150	13.20	3.4	3.7	96.3
150	106	73.3	19.1	22.8	77.2
200	75	57.7	15.0	37.8	62.2
270	53	38.5	10.0	47.8	52.2
400	38	20.9	5.4	53.3	46.7
Pan	0	179.5	46.7	100.0	-
Total		384.0	100.0		

	Interpolated	Graphic
F80	1,244	1,349
P80	121	115


Size Analysis
100.0
90.0
80.0
70.0
60.0
50.0
40.0
30.0
20.0
10.0
-
Cumulative % Passing
-
200
400
600
800
1,000
1,200
1,400
1,600
1,800
2,000
2,200
2,400
2,600
2,800
3,000
Microns


Mill Feed Size Analysis
y = 0.0159647360x3 - 2.2682979943x2 + 120.0433255637x - 1,911.2092253642
R² = 0.9917243998
4,000
3,500
3,000
2,500
2,000
1,500
1,000
500
-
Microns
-
20.0
40.0
60.0
80.0
100.0
% Passing


Mill Product Size Analysis
y = 0.0310894741x2 - 1.7846555336x + 58.838993
R² = 0.9442500483
Size (microns)
250
200
150
100
50
-
-
10.0
20.0
30.0
40.0
50.0
60.0
70.0
80.0
90
% Passing

APPENDIX D

Bond Ball Mill

Feed $y = 0.0159647360x^3 - 2.2682979943x^2 - 120.0433255637x - 1911.2092253642$

$R^2 = 0.991724$

Y	X
1349.09449	80

	Microns	% Passing
6	3350	100.00
8	2360	90.09
10	1700	86.60
14	1180	79.07
20	850	67.27
28	600	59.29
35	425	51.45
48	300	43.88
100	150	31.41

Product $y = 0.0310894741x^2 - 1.7846555336x + 58.8389937002$

$R^2 = 0.94425$

Y	X
115.039185	80

Microns	% Passing
212	99.8
150	96.3
106	77.2
75	62.2
53	52.2
38	46.7

Average wt of 3 - 700 ml samples	
A1 wt (1)	1361.3
A1 wt (2)	1381.8
A1 wt (3)	1367.8
Average	1370.3

Y= A1 divided by 3.5 for 250% circulating load
X= average wt % of undersize from screening

Y=	391.5
X=	31.4

Variable	A	B	C	D	E	F	G
Calcs	average wt of 3 700 ml samples	A multiplied by X	Y minus B	C1 divided by 1.2	wt of product	E minus B	F divided by D
	A2 = E1, A3 = E2, etc.			C2 divided by G1			

		Undersize (g)				Mill Production (g)	
Cycle #	Feed (g)	In Feed	To be Produced	Mill Revs	Wt of Undersize (g)	Total Net	Per Rev
1	1370.3	430.4	-38.9	25	540.4	110.0	4.399
2	540.4	169.7	221.8	50	292.9	123.2	2.443
3	292.9	92.0	299.5	123	303.9	211.9	1.728
4	303.9	95.5	296.1	171	380.4	284.9	1.664
5	380.4	119.5	272.0	164	406.5	287.0	1.755
6	406.5	127.7	263.8	150	398.0	270.3	1.798
7	398	125.0	266.5	148	393.3	268.3	1.810
8	393.3	123.5	268.0	148	392.1	268.6	1.814
9	392.1	123.2	268.4	148	385.6	262.4	1.774
10	385.6	121.1	270.4	152		-121.1	-0.795

S I Z E			M I L L F E E D				
Tyler Mesh	Microns	(g)	Screen #2 Weight (g)	Screen #3 Weight (g)	Weight %	Retained %	Passing %
6	3,350	0.0	0.0	0.0	-	-	100.0
8	2,360	23.4	18.7	21.6	9.9	9.9	90.1
10	1,700	7.7	7.4	7.4	3.5	13.4	86.6
14	1,180	16.6	16.4	15.4	7.5	20.9	79.1
20	850	24.6	25.7	25.6	11.8	32.7	67.3
28	600	16.3	17.8	17.2	8.0	40.7	59.3
35	425	15.9	17.4	17.1	7.8	48.5	51.5
48	300	15.5	16.7	16.5	7.6	56.1	43.9
65	212	13.0	13.8	13.6	6.3	62.4	37.6
100	150	12.9	13.6	13.3	6.2	68.6	31.4
Pan		66.1	68.7	67.2	31.4	100.0	
Total		212.0	216.2	214.9	100.0		

Appendix D: Bond Work Index

	Engineer	JA	**Test ID**	Oxide Comp 2
	Technician	MR	**Date**	7/8/2024
	Project Name	Getchell		
	Project No.	23041		

Purpose: To determine the bond work index that can be used to size grind and mills for the project's comminution circuit based on the selected P80.

Procedure: The feed samples were screened and stage-crushed to minus 3,350 microns as per test specification.The Bwi results were run with a closing size of 150 microns. Several quality-control measured, as well as rigorous closing criteria were followed including; minimum of 6 cycles, average grams per mill revolution was less than 3% for last three (3) cycles with inflection, target weight of undersize within 4-10 grams, circulating load ratio of 2.47 or higher, last cycle was wet screened for product P80 size (semi-log interpolated).

Sample: Oxide Comp 2

Results:

Bwi - From Graph	13.76	**BWi - Avg, kWh/st**	13.62
Bwi - Interpolated	13.48		

Size		Mill Feed			
Tyler Mesh	Microns	Weight (g)	Weight %	Retained %	Passing %
6	3,350	0.0	-	-	100.0
8	2,360	52.6	26.4	26.4	73.6
10	1,700	41.1	20.6	46.9	53.1
14	1,180	34.0	17.0	63.9	36.1
20	850	19.5	9.8	73.7	26.3
28	600	14.1	7.1	80.8	19.2
35	425	8.8	4.4	85.2	14.8
48	300	6.7	3.4	88.5	11.5
65	212	5.0	2.5	91.0	9.0
100	150	3.9	2.0	93.0	7.0
Pan		14.0	7.0	100.0	
Total		199.7	100.0		

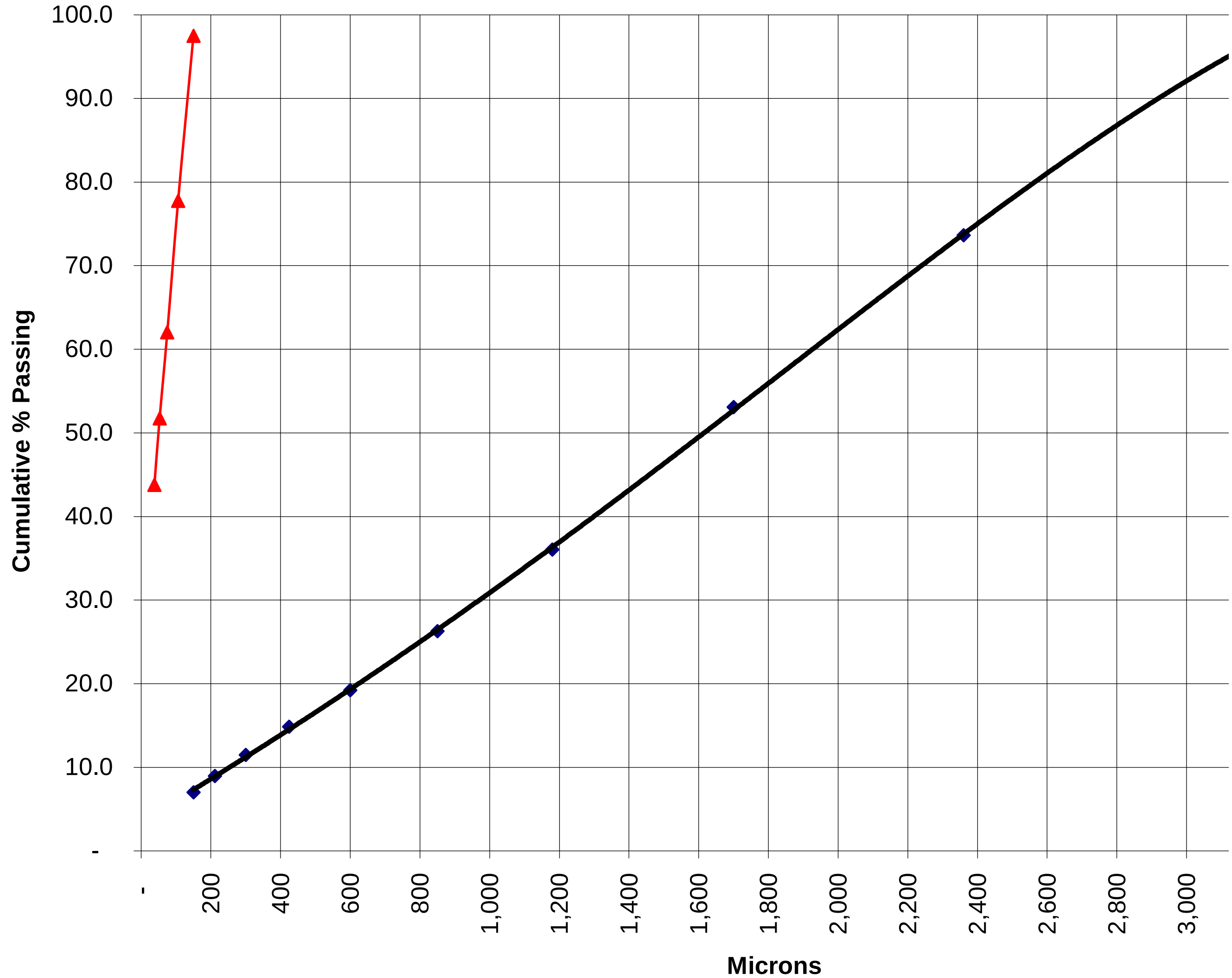
Cycle #	grams / revolution
n-2	1.492
n-1	1.492
n	1.477
Average	1.487

Mill Feed Weight (grams)	1300.1
Desired Mesh of Grind:	100
Desired Micron of Grind:	150
Circulating Load (%)	250
Circulating Load (grams):	371.5

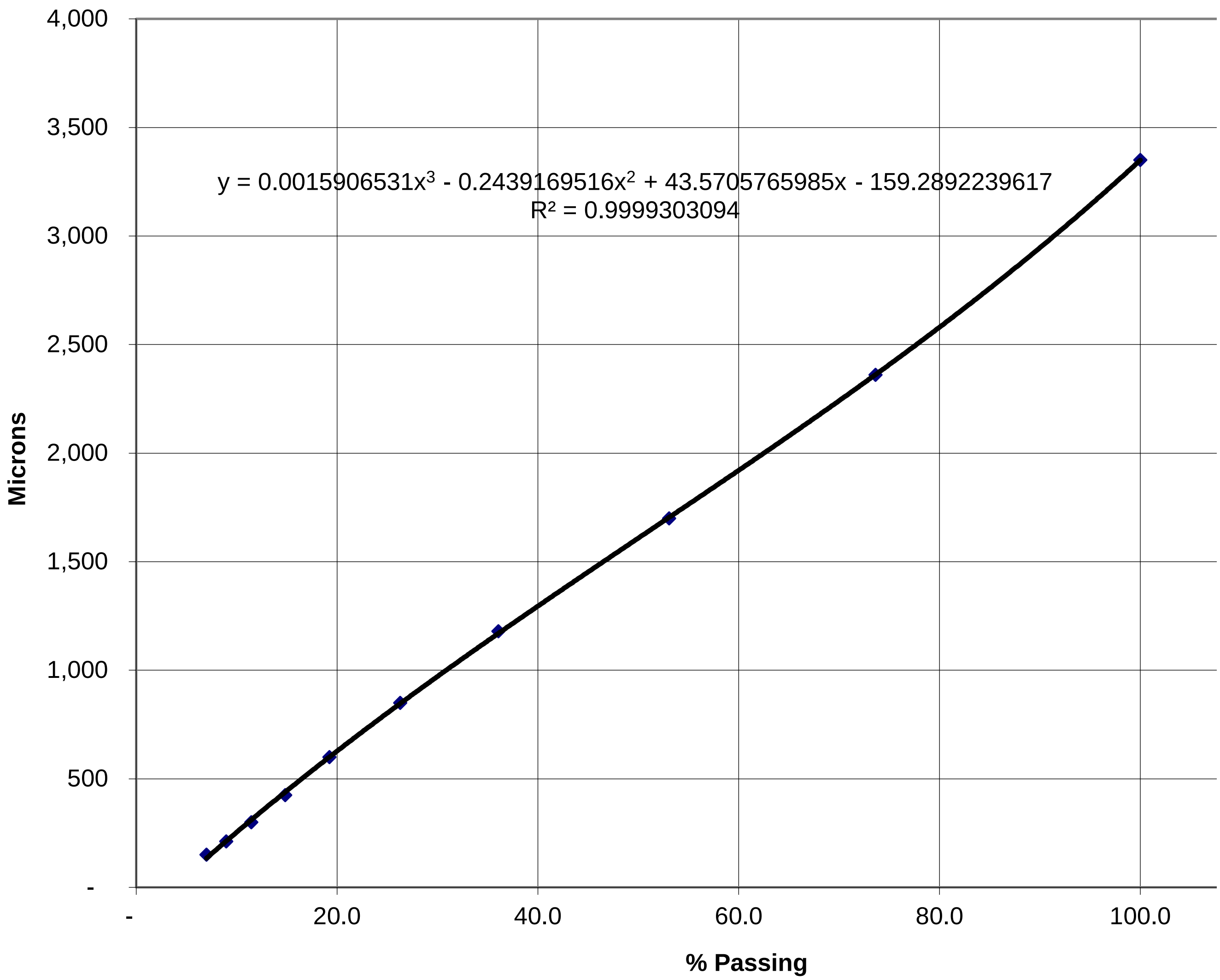
Size		Ground Product			
Tyler Mesh	Microns	Weight (g)	Weight %	Retained %	Passing %
48	300	-			
65	212	0.10	0.0	0.0	100.0
100	150	8.80	2.5	2.5	97.5
150	106	70.5	19.7	22.2	77.8
200	75	56.2	15.7	38.0	62.0
270	53	36.8	10.3	48.3	51.7
400	38	28.4	8.0	56.2	43.8
Pan	0	156.4	43.8	100.0	-
Total		357.2	100.0		

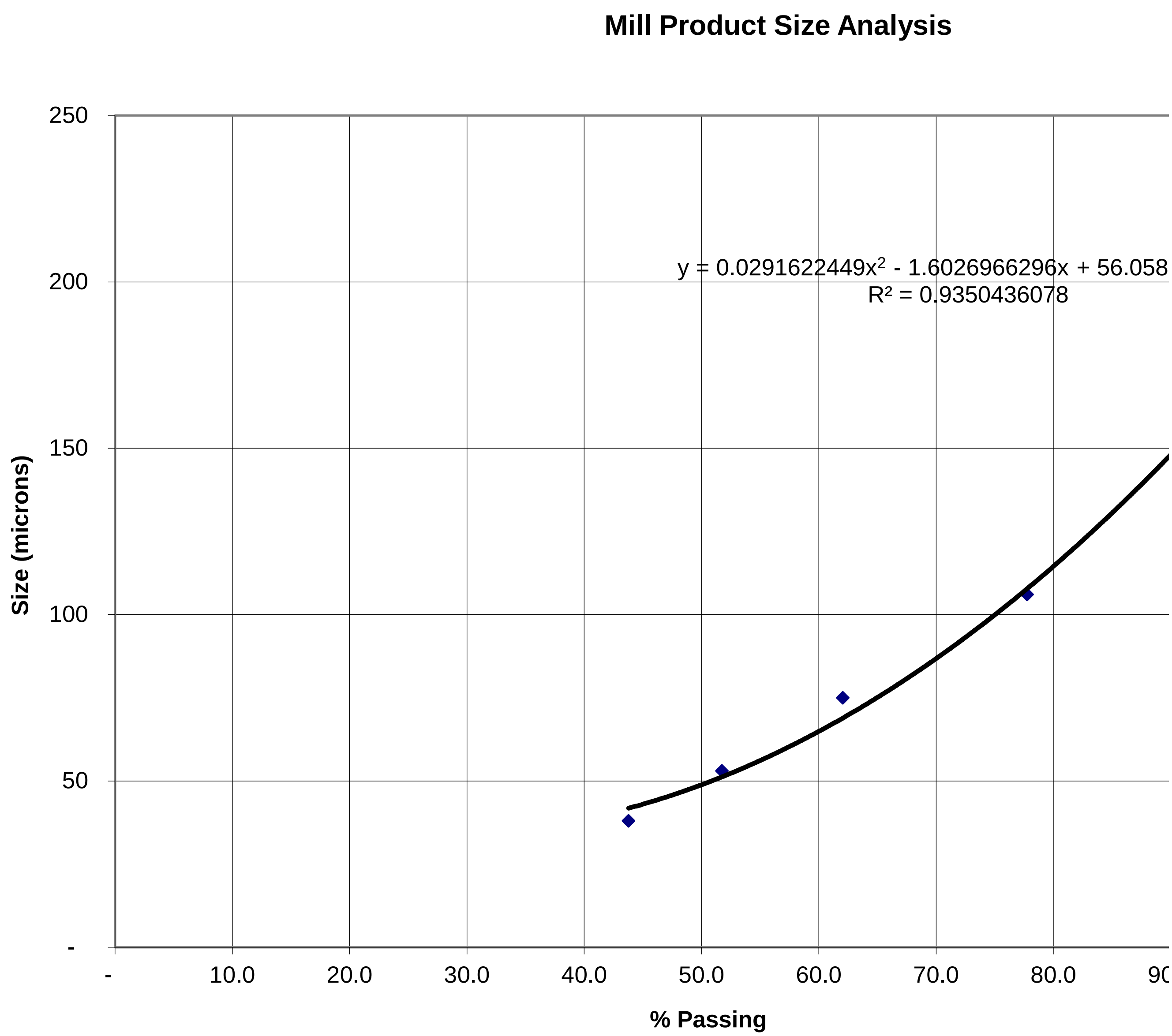
	Interpolated	Graphic
F80	2,599	2,580
P80	111	114

Size Analysis



Mill Feed Size Analysis




Mill Product Size Analysis
y = 0.0291622449x2 - 1.6026966296x + 56.058
R² = 0.9350436078
Size (microns)
250
200
150
100
50
-
-
10.0
20.0
30.0
40.0
50.0
60.0
70.0
80.0
90
% Passing

APPENDIX D

Bond Ball Mill

Feed

$y = 0.0015906531x^3 - 0.2439169516x^2 + 43.5705765985x - 159.2892239617$
$R^2 = 0.99993$

Y	X
2579.7028	80

	Microns	% Passing
6	3350	100.00
8	2360	73.64
10	1700	53.07
14	1180	36.06
20	850	26.29
28	600	19.23
35	425	14.84
48	300	11.47
100	150	6.99

Product

$y = 0.0291622449x^2 - 1.6026966296x + 56.0580200049$
$R^2 = 0.93504$

Y	X
114.480657	80

Microns	% Passing
212	100.0
150	97.5
106	77.8
75	62.0
53	51.7
38	43.8

Average wt of 3 - 700 ml samples

A1 wt (1)	1255.4
A1 wt (2)	1243.7
A1 wt (3)	1222.7
Average	1240.6

Y= A1 divided by 3.5 for 250% circulating load
X= average wt % of undersize from screening

Y=	354.5
X=	7.0

Variable	A	B	C	D	E	F	G
Calcs	average wt of 3 700 ml samples	A multiplied by X	Y minus B	C1 divided by 1.2	wt of product	E minus B	F divided by D
	A2 = E1, A3 = E2, etc.			C2 divided by G1			

		Undersize (g)				Mill Production (g)	
Cycle #	Feed (g)	In Feed	To be Produced	Mill Revs	Wt of Undersize (g)	Total Net	Per Rev
1	1240.6	86.8	267.7	228	488.0	401.2	1.761
2	488	34.1	320.3	182	371.4	337.3	1.854
3	371.4	26.0	328.5	177	364.7	338.7	1.912
4	364.7	25.5	328.9	172	364.9	339.4	1.973
5	364.9	25.5	328.9	167	360.8	335.3	2.011
6	360.8	25.2	329.2	164	353.6	328.4	2.006
7	353.6	24.7	329.7	164	350.9	326.2	1.984
8	350.9	24.5	329.9	166	356.5	332.0	1.996

S I Z E			M I L L F E E D				
Tyler Mesh	Microns	(g)	Screen #2 Weight (g)	Screen #3 Weight (g)	Weight %	Retained %	Passing %
6	3,350	0.0	0.0	0.0	-	-	100.0
8	2,360	45.6	56.8	55.5	26.4	26.4	73.6
10	1,700	35.7	45.5	42.0	20.6	46.9	53.1
14	1,180	33.2	34.4	34.3	17.0	63.9	36.1
20	850	20.7	18.3	19.5	9.8	73.7	26.3
28	600	16.1	12.6	13.6	7.1	80.8	19.2
35	425	10.5	7.5	8.3	4.4	85.2	14.8
48	300	8.4	5.5	6.3	3.4	88.5	11.5
65	212	6.4	4.0	4.6	2.5	91.0	9.0
100	150	5.1	3.2	3.5	2.0	93.0	7.0
Pan		17.5	11.8	12.6	7.0	100.0	
Total		199.2	199.6	200.2	100.0		

APPENDIX E: DIAGNOSTIC LEACH DATA

Getchell Diagnostic Test Results Summary

Revised: 5/16/2024

Author: JA

Diagnostic Leach Results Summary

Test #	Leach Stage	Sample	Grind (P_{80})	% Recovery (Au)	Assayed Head Grade Au (mg/kg)	Calc. Head Grade Au (mg/kg)	Residue Grade Au (mg/kg)	% Recovery (Ag)	Assayed Head Grade Ag (mg/kg)	Calc. Head Grade Ag (mg/kg)	Residue Grade Ag (mg/kg)	As (mg/kg)	Sulfur (total) (mg/kg)	NaCN Consumption (kg/mt)	Lime Consumption (kg/mt)
GBR-1	1st CN leach	Average Grade	200	1.0	1.49	1.55	1.53	NA	BD	BD	BD	1587	2.61	2.399	0.699
425 C Roast															
GBR-1B	2nd CN Leach	Average Grade	200	5.9	1.53	1.51	1.42	NA	BD	BD	BD	1743	1.01	1.531	24.99
625 C Roast															
GBR-1C	3rd CN Leach	Average Grade	200	63.2	1.42	1.57	0.56	NA	BD	BD	BD	TBD	TBD	0.197	0.00
GBR-2 RR*	3rd CN Leach	Average Grade	200	83.2	1.42	1.43	0.22	NA	BD	BD	BD	1722	1.37	0.413	12.22

BD = Below detection limit

RR* = a second 1 kg sample was roasted at 650 C for 6 hours and the residue leached to evaluate gold extraction post roast

425 C Roast for 4 hours

	sample	mass loss in roast (g)	% mass loss in roast
GBR-1	Average Grade	3.3	0.35

Note:

625 C Roast for 4 hours

	sample	mass loss in roast (g)	% mass loss in roast
GBR-1B	Average Grade	29.3	3.2

Notes:

650 C Roast for 6 hours Re-Run

	sample	mass loss in roast (g)	% mass loss in roast
GBR-1B	Average Grade	29.5	2.9

Notes:

Appendix E

Bottle Roll Leaching Test 1

Project: 23041 Getchell Gold
Date: 20-Mar-24

Purpose: To examine the mineral association of gold and silver via diagnostic leach. Initial Bottle Roll

Sample: Approximately 1000 grams of Average Grade Composite, P80 200

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~11 with hydrated lime and sodium cyanide was added to a calculated level of 1.0 g/l. At 6, and 24 hours, the pH and free cyanide were determined. A sample of solution was removed and assayed for gold and silver. Sodium cyanide was added to return the level to 1.0 g/L and the pH was adjusted to 11 with hydrated lime if needed. After 48 hours, the solution was measured to determine pH, free cyanide, and gold and silver contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold and silver contents by fire assay techniques.

Conditions:

Grind	Leach Time	NaCN Concentration	% Solids
80% minus 200 mesh	48 hours	1.00 g/L	40% Solids

Summary of Results:

Parameter	Au	Ag
Extraction, % (1)	1.02	1.3
Assayed Head, g/mt	1.49	BD
Calculated Head, g/mt	1.55	5.0
Final Tail Assay, g/mt	1.53	5.0

Extraction, % (1)			NaCN Consumed
Hr.	Au	Ag	kg/mt
6	2.4	0.8	1.258
24	1.0	0.8	1.798
48	1.0	1.3	2.399

Cyanide Consumption	2.399	kg NaCN/metric ton ore
Lime Added	0.699	kg $Ca(OH)_2$/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g NaCN	$Ca(OH)_2$	Carbon		Residual Reagents NaCN g/L		pH Initial	pH Adjust
0	2500	1499	1.50	0.70					8.6	10.7
6	2500	1499	1.26				0.16		10.9	
24	2500	1499	0.54				0.64		10.5	
48	2497	1496					0.60		10.3	
Total			3.30	0.70						
				(0.5)	CaO Equivalent					

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L			Volume Thief
Feed (analyzed)	1000		1.49		BD				
Feed (computed)			1.55		5.0				
6 hour Preg		1499		0.03		0.03			25
24 hour Preg		1499		0.01		0.03			25
48 hour Preg		1496		0.01		0.04			25
48 hour Dry Residue	1001.4		1.53		5.0				

(1) Based on calculated head assays.

Appendix E
Bottle Roll Leaching Test 1B

Project: 23041 Getchell Gold
Date: 1-Apr-24

Purpose: To examine the mineral association of gold and silver via diagnostic leach post 425 C roast

Sample: Approximately 940 grams of 425 C roasted Average Grade Composite, P80 200

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~11 with hydrated lime and sodium cyanide was added to a calculated level of 1.0 g/l. At 6, and 24 hours, the pH and free cyanide were determined. A sample of solution was removed and assayed for gold and silver. Sodium cyanide was added to return the level to 1.0 g/L and the pH was adjusted to 11 with hydrated lime if needed. After 48 hours, the solution was measured to determine pH, free cyanide, and gold and silver contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold and silver contents by fire assay techniques.

Conditions:

Grind	Leach Time	NaCN Concentration	% Solids
80% minus 200 mesh	48 hours	1.00 g/L	40% Solids

Summary of Results:

Parameter	Au	Ag
Extraction, % (1)	5.9	0.8
Assayed Head, g/mt	1.5	5.0
Calculated Head, g/mt	1.5	3.0
Final Tail Assay, g/mt	1.42	3.0

Extraction, % (1) Hr.	Au	Ag	NaCN Consumed kg/mt
6	4.7	1.0	1.283
24	4.6	1.0	1.469
48	5.9	0.8	1.531

Cyanide Consumption	1.531	kg NaCN/metric ton ore
Lime Added	24.985	kg $Ca(OH)_2$/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g NaCN	$Ca(OH)_2$	Carbon		Residual Reagents NaCN g/L		pH Initial	Adjust
0	2358	1397	1.40	24.00					4.4	10.6
6	2358	1397	1.24				0.12		12.3	
24	2358	1397	0.17				0.88		12.0	
48	2356	1395					0.96		11.7	
Total			2.81	24.00						
				(18.2)	CaO Equivalent					

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L			Volume Thief
Feed (analyzed)	943		1.53		5.0				
Feed (computed)			1.51		3.0				
6 hour Preg		1397		0.05		0.02			25
24 hour Preg		1397		0.05		0.02			25
48 hour Preg		1395		0.06		0.02			25
0 hour Dry Residue	960.6		1.42		3.0				

(1) Based on calculated head assays.

weight gain due to initial lime addition requried to increase pH to > 10.5

Appendix E
Bottle Roll Leaching Test 1C

Project: 23041 Getchell Gold
Date: 10-Apr-24

Purpose: To examine the mineral association of gold and silver via diagnostic leach post 425 C roast

Sample: Approximately 960 grams of 625 C roasted Average Grade Composite, P80 200

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~11 with hydrated lime and sodium cyanide was added to a calculated level of 1.0 g/l. At 6, and 24 hours, the pH and free cyanide were determined. A sample of solution was removed and assayed for gold and silver. Sodium cyanide was added to return the level to 1.0 g/L and the pH was adjusted to 11 with hydrated lime if needed. After 48 hours, the solution was measured to determine pH, free cyanide, and gold and silver contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold and silver contents by fire assay techniques.

Conditions:

Grind	Leach Time	NaCN Concentration	% Solids
80% minus 200 mesh	48 hours	1.00 g/L	40% Solids

Summary of Results:

Parameter	Au	Ag
Extraction, % (1)	63.9	151.3
Assayed Head, g/mt	1.42	3.0
Calculated Head, g/mt	1.55	0.1
Final Tail Assay, g/mt	0.56	0.1

Hr.	Extraction, % (1) Au	Extraction, % (1) Ag	NaCN Consumed kg/mt
6	62.9	35.0	-0.004
24	65.1	673.1	0.058
48	63.9	151.3	0.197

Cyanide Consumption	0.197	kg NaCN/metric ton ore
Lime Added	0.000	kg $Ca(OH)_2$/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g NaCN	Reagents Added, g $Ca(OH)_2$	Reagents Added, g Carbon		Residual Reagents NaCN g/L		pH Initial	pH Adjust
0	2196	1324	1.32						12.3	
6	2196	1324					1.00		12.1	
24	2194	1322	0.46				0.96		11.8	
48	2169	1297					1.24		11.6	
Total			1.78	0.00						
				(0.0)	CaO Equivalent					

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L			Volume Thief
Feed (analyzed)	878		1.42		3.0				
Feed (computed)			1.55		0.1				
6 hour Preg		1324		0.64		0.03			25
24 hour Preg		1322		0.65		0.51			25
48 hour Preg		1297		0.64		0.11			25
0 hour Dry Residue	872.2		0.56		0.1				

(1) Based on calculated head assays.

Appendix E

Bottle Roll Leaching Test 1C RR

Project: 23041 Getchell Gold
Date: 6-May-24

Purpose: To examine the mineral association of gold and silver via diagnostic leach post 650 C 6 hr roast

Sample: Approximately 970 grams of 650 C roasted Average Grade Composite, P80 200

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~11 with hydrated lime and sodium cyanide was added to a calculated level of 1.0 g/l. At 6, and 24 hours, the pH and free cyanide were determined. A sample of solution was removed and assayed for gold and silver. Sodium cyanide was added to return the level to 1.0 g/L and the pH was adjusted to 11 with hydrated lime if needed. After 48 hours, the solution was measured to determine pH, free cyanide, and gold and silver contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold and silver contents by fire assay techniques.

Conditions:

Grind	Leach Time	NaCN Concentration	% Solids
80% minus 200 mesh	48 hours	1.04 g/L	40% Solids

Summary of Results:

Parameter	Au	Ag
Extraction, % (1)	84.4	2.0
Assayed Head, g/mt	1.42	3.0
Calculated Head, g/mt	1.41	5.0
Final Tail Assay, g/mt	0.22	5.0

	Extraction, % (1)		NaCN Consumed
Hr.	Au	Ag	kg/mt
6	82.8	1.2	0.292
24	85.5	1.5	0.408
48	84.4	2.0	0.413

Cyanide Consumption	0.413	kg NaCN/metric ton ore
Lime Added	12.219	kg $Ca(OH)_2$/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g NaCN	$Ca(OH)_2$	Carbon		Residual Reagents NaCN g/L		pH Initial	pH Adjust
0	2432	1441	1.50	11.60					6.2	10.6
6	2432	1441	0.23	0.20			0.84		10.4	10.6
24	2432	1441	0.12	0.30			0.92		10.4	10.7
48	2431	1441					1.00		10.5	
Total			1.85	12.10						
				(9.2)	CaO Equivalent					

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L			Volume Thief
Feed (analyzed)	972.6		1.42		3.0				
Feed (computed)			1.41		5.0				
6 hour Preg		1441		0.80		0.04			25
24 hour Preg		1441		0.82		0.05			25
48 hour Preg		1441		0.79		0.07			25
0 hour Dry Residue	990.2		0.22		5.0				

(1) Based on calculated head assays.

Appendix E

Diagnostic Leach Series 1st Roast

Forte Project: 23041 Getchell
Date: 30-Mar-24

Purpose: To examine the mineral association of gold and silver via diagnostic leach

Sample: approximately 1000 g of Average Grade Composite bottle roll residue from the first cyanide bottle roll leach

Procedure: each sample was roasted in weigh boats at 425 C in a muffle furnace with no added air flow for 4 hour with mixing of the material at approximately 2 hours

	sample	Boat 1 tare (g)	pre-roast (g)	post roast (g)	Boat 2 (g)	pre-roast (g)	post roast (g)	Sample mass pre-roast (g)	sample mass post roast (g)	mass loss in roast (g)	% mass loss in roast
GBR-1	Average Grade	676.3	1117.3	1116	614.8	1120.5	1118.5	946.7	943.4	3.3	0.35

Appendix E

Diagnostic Leach Series 2nd Roast

Forte Project: 23041 Getchell Gold
Date: 8-Apr-24

Purpose: To examine the mineral association of gold and silver via diagnostic leach

Sample: approximately 1000 g of Average Grade Composite bottle roll residue from the second cyanide bottle roll leach

Procedure: each sample was roasted in weigh boats at 625 C in a muffle furnace with added air flow for 4 hour with mixing of the material at approximately 2 hours

	sample	Boat 1 tare (g)	pre-roast (g)	post roast (g)	Boat 2 (g)	pre-roast (g)	post roast (g)	Sample mass pre-roast (g)	sample mass post roast (g)	mass loss in roast (g)	% mass loss in roast
GBR-1B	Average Grade	676	978.2	968.1	1294.5	1899.2	1880	906.9	877.6	29.3	3.2

Notes:

Diagnostic Leach Series 2nd Roast Re-run to confirm sulfur oxidation

Forte Project: 23041 Getchell Gold
Date: 4-May-24

Purpose: To examine the mineral association of gold and silver via diagnostic leach

Sample: approximately 1000 g of Average Grade Composite

Procedure: the sample was roasted in weigh boats at 650 C in a muffle furnace with added air flow for 6 hour with mixing of the material at approximately every 2 hours

	sample	Boat 1 tare (g)	pre-roast (g)	post roast (g)	Boat 2 (g)	pre-roast (g)	post roast (g)	Sample mass pre-roast (g)	sample mass post roast (g)	mass loss in roast (g)	% mass loss in roast
GBR-1B	Average Grade	676	1102.2	1089.7	614.8	1190.6	1173.6	1002	972.5	29.5	2.9

Notes: re-ran this test with a new split of average grade composite to fully oxidize the sulfides for the third leach to maximize precious metal recoveries

APPENDIX F: GRIND STUDY DATA

Appendix F Grind Study Data

FORTE ANALYTICAL

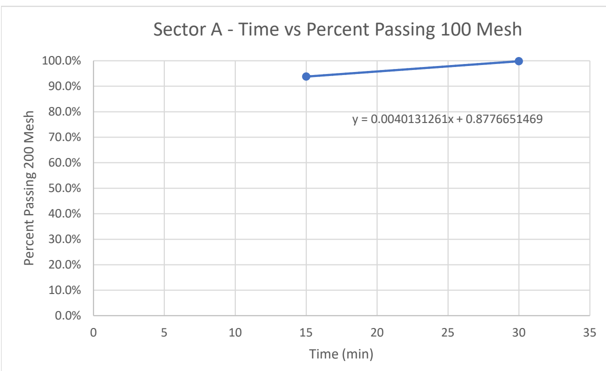
	A	B	C
Tab Name:	15 min	30 min	45 min

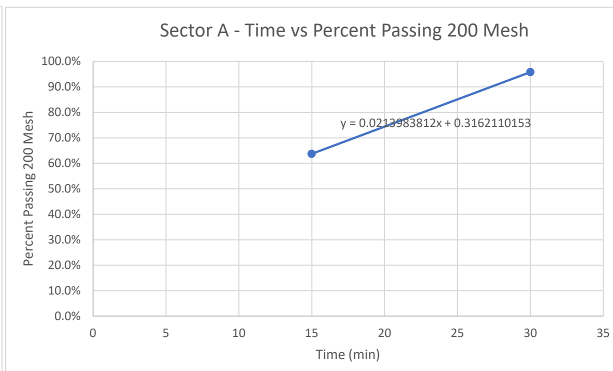
Average Grade Comp

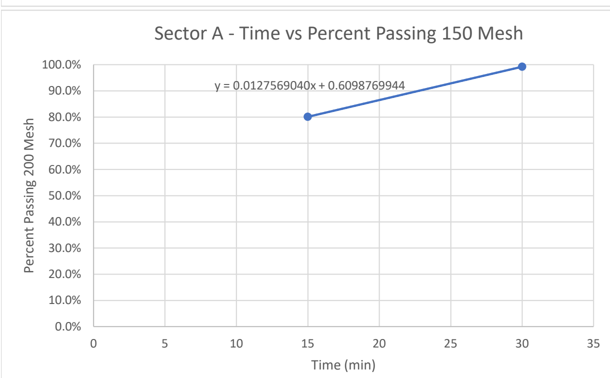
				Average Grade			
			Col of Sizes C	Average Grade 15 min	Average Grade 30 min	Average Grade 45 min	
Mesh Size	Col of Data	Row of Data	Size Fraction, µm	Cumulative Weight Passing, %			
	J	2		Average Grade 15 min	Average Grade 30 min	Average Grade 45 min	
100	J	11	150	93.8%	99.8%	99.9%	
150	J	12	105	80.1%	99.3%	99.7%	
200	J	13	75	63.7%	95.8%	99.3%	
270	J	14	53	52.4%	84.2%	96.0%	
<270	J	15	< 53	0.0%	0.0%	0.0%	

J	20	0	0	0	0	
J	3	Time, minutes	15	30	45	
		Mesh Size	100	150	200	

Sector A (100 Mesh)	Sector A (150 Mesh)	Sector A (200 Mesh)
80%	80%	80%
NA	15	23





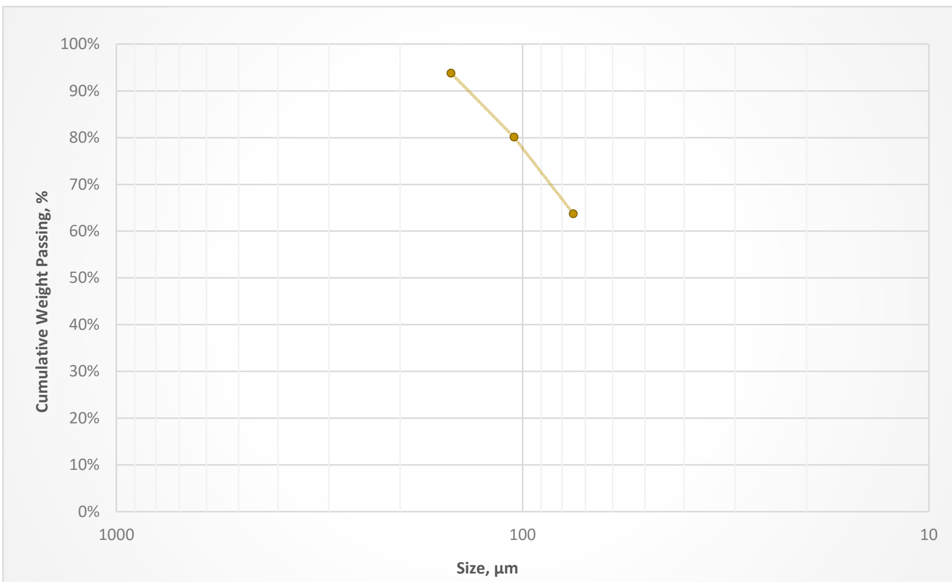


	Engineer	J.Axen	Test ID	Average Grade 15 min
	Technician	TK	Time (min)	15
	Project Name	Getchell Gold	Date	3/17/2024
FA.F-TS01	Project No.	23041		

Original Sample Weight, g	999.55	RPM	53.2

Average Grade Comp

Screen Size		Wet Screen Analysis			
Inches/ US mesh	microns	Sample Dry Weight, g	Weight Distribution, %	Cumulative Weight Retained, %	Cumulative Weight Passing, %
No. 100	150	62.11	6.2%	6.2%	93.8%
No. 150	105	136.57	13.7%	19.9%	80.1%
No. 200	75	163.97	16.4%	36.3%	63.7%
No. 270	53	113.03	11.3%	47.6%	52.4%
<No. 270	< 53	523.87	52.4%	100.0%	0.0%
Total		999.55	100.0%	Calc'd P80, µm	105
				Calc'd P80, in.	0.0041

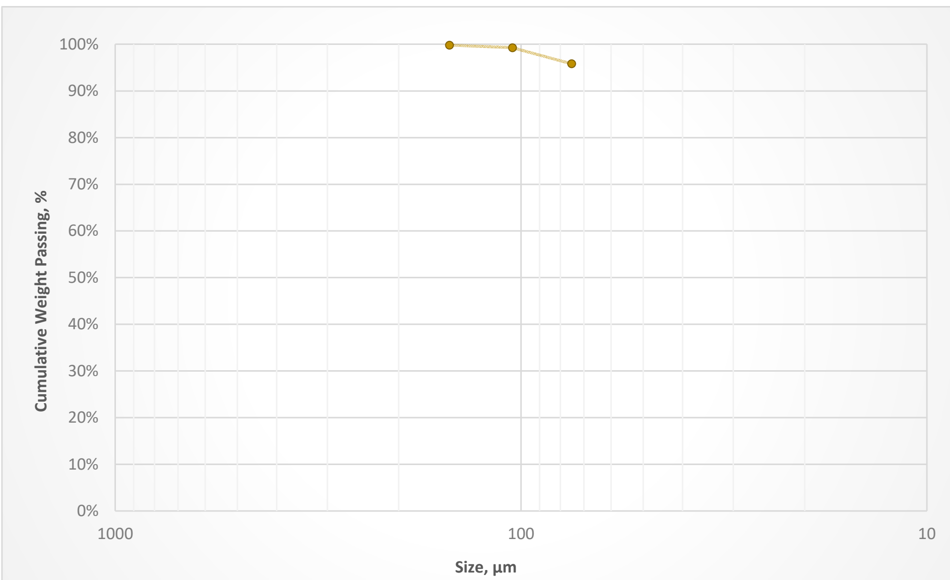


	Engineer	J. Axen	Test ID	Average Grade 30 min
	Technician	TK	Time (min)	30
	Project Name	Getchell Gold	Date	3/11/2024
FA.F-TS01	Project No.	23066		

Original Sample Weight, g	1004.6	RPM	53.3

Average Grade Comp

Screen Size		Wet Screen Analysis			
Inches/ US mesh	microns	Sample Weight, g	Weight Distribution, %	Cumulative Weight Retained, %	Cumulative Weight Passing, %
No. 100	150	1.95	0.2%	0.2%	99.8%
No. 150	105	5.50	0.5%	0.7%	99.3%
No. 200	75	34.58	3.4%	4.2%	95.8%
No. 270	53	116.61	11.6%	15.8%	84.2%
<No. 270	< 53	845.96	84.2%	100.0%	0.0%
Total		1004.60	100.0%	Calc'd P80, µm	63
				Calc'd P80, in.	0.002

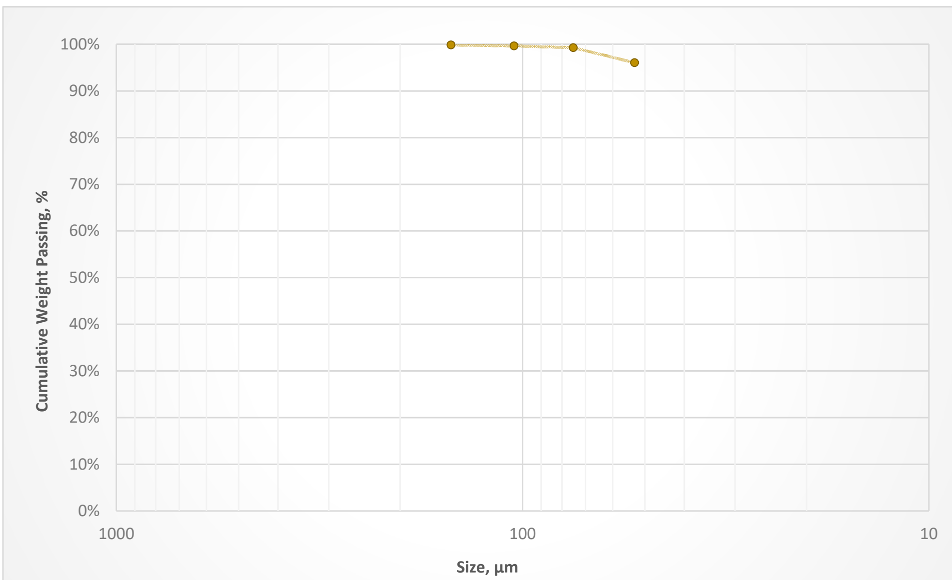


	Engineer	J. Axen	Test ID	Average Grade 45 min
	Technician	TK	Time (min)	45
	Project Name	Getchell Gold	Date	3/18/2024
FA.F-TS01	Project No.	23041		

Original Sample Weight, g	1010.09	RPM	53

Average Grade Comp

Screen Size		Wet Screen Analysis			
Inches/ US mesh	microns	Sample Weight, g	Weight Distribution, %	Cumulative Weight Retained, %	Cumulative Weight Passing, %
No. 100	150	1.44	0.1%	0.1%	99.9%
No. 150	105	2.08	0.2%	0.3%	99.7%
No. 200	75	3.87	0.4%	0.7%	99.3%
No. 270	53	32.52	3.2%	4.0%	96.0%
<No. 270	< 53	970.18	96.0%	100.0%	0.0%
Total		**1010.09**	**100.0%**	**Calc'd P80, µm**	60
				Calc'd P80, in.	0.002



Appendix F Grind Study Data

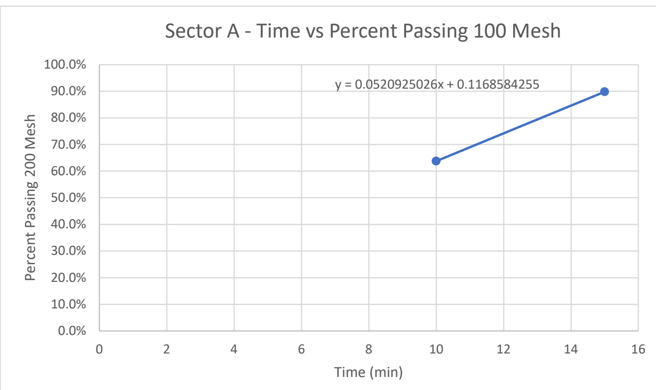
	A	B	C
Tab Name:	10 min	15 min	30 min

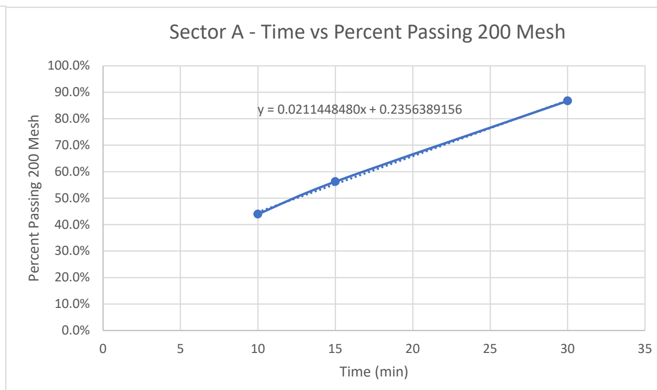
High Grade Comp

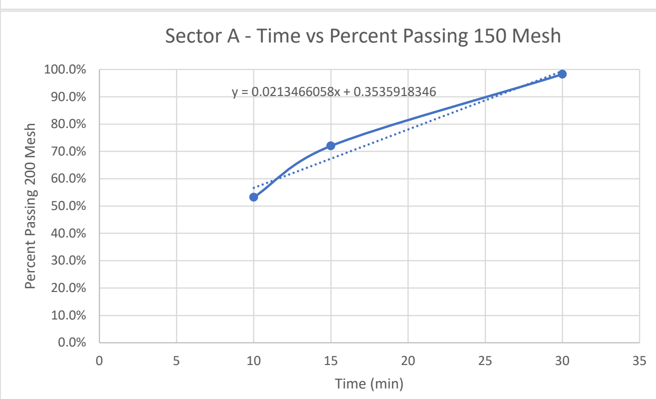
				High Grade			
			Col of Sizes	High Grade 10 min	High Grade 15 min	High Grade 30 min	
			C				
Mesh Size	Col of Data	Row of Data	**Size Fraction, µm**	**Cumulative Weight Passing, %**			
	J	2		**High Grade**	**High Grade**	**High Grade**	
100	J	11	**150**	63.8%	89.8%	99.6%	
150	J	12	**105**	53.2%	72.0%	98.2%	
200	J	13	**75**	44.0%	56.3%	86.8%	
270	J	14	**53**	37.0%	45.9%	68.2%	
<270	J	15	**< 53**	0.0%	0.0%	0.0%	

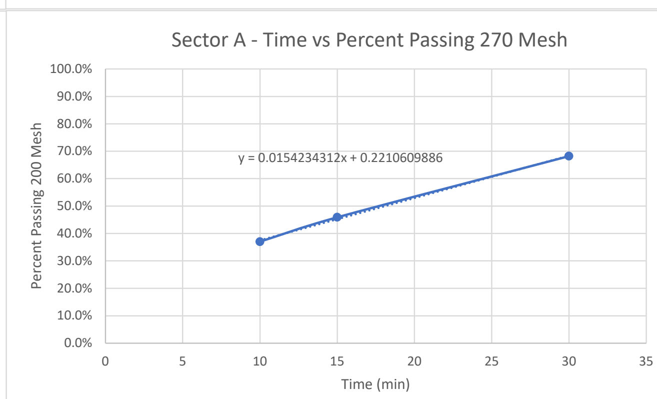
J	20	**0**	0	0	0	
J	3	**Time, minutes**	10	15	30	
		Mesh Size	100	150	200	

Sector A (100 Mesh)	Sector A (150 Mesh)	Sector A (200 Mesh)	Sector A (270 Mesh)
80%	80%	80%	80%
13	21	27	38







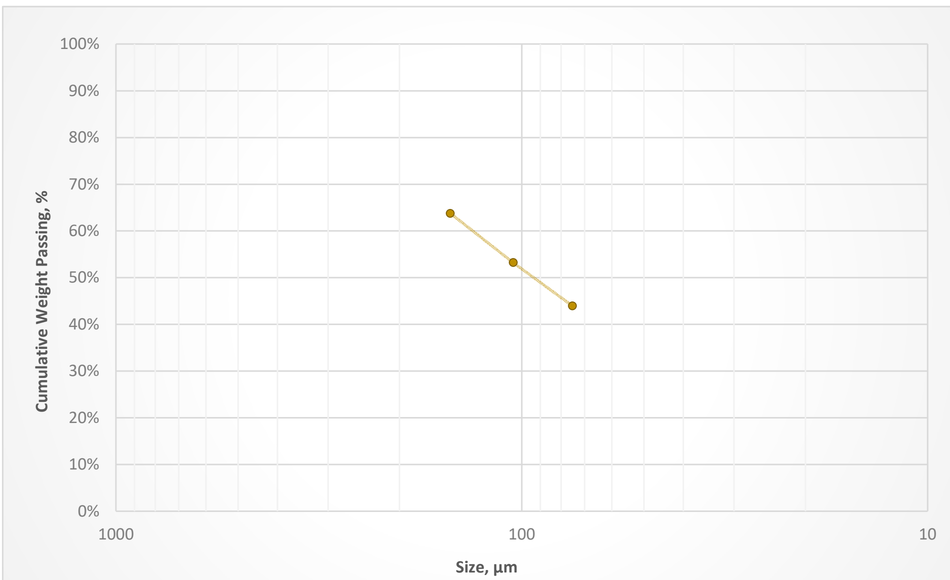


Engineer	J.Axen	Test ID	High Grade
Technician	MR	Time (min)	10
Project Name	Getchell Gold	Date	7/12/2024
Project No.	23041		

FA.F-TS01

Original Sample Weight, g	998.85	RPM	53.2

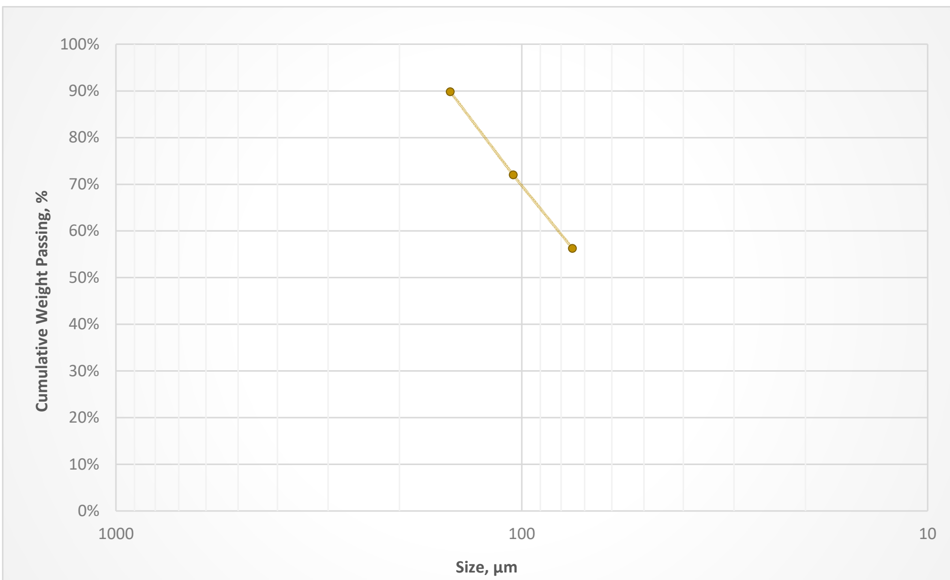
Screen Size		Wet Screen Analysis			
Inches/ US mesh	microns	Sample Dry Weight, g	Weight Distribution, %	Cumulative Weight Retained, %	Cumulative Weight Passing, %
No. 100	150	361.80	36.2%	36.2%	63.8%
No. 150	105	105.30	10.5%	46.8%	53.2%
No. 200	75	92.50	9.3%	56.0%	44.0%
No. 270	53	69.50	7.0%	63.0%	37.0%
<No. 270	< 53	369.75	37.0%	100.0%	0.0%
Total		998.85	100.0%	Calc'd P80, μm	188
				Calc'd P80, in.	0.0074



	Engineer	J. Axen	Test ID	High Grade
	Technician	MR	Time (min)	15
	Project Name	Getchell Gold	Date	7/10/2024
FA.F-TS01	Project No.	23041		

Original Sample Weight, g	1003.4	RPM	53.3

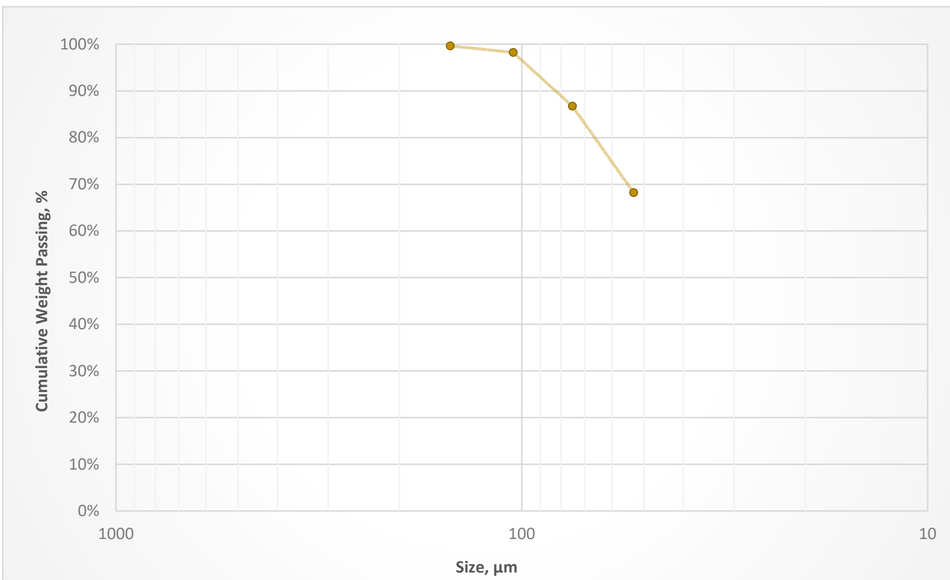
Screen Size		Wet Screen Analysis			
Inches/ US mesh	microns	Sample Weight, g	Weight Distribution, %	Cumulative Weight Retained, %	Cumulative Weight Passing, %
No. 100	150	102.10	10.2%	10.2%	89.8%
No. 150	105	178.80	17.8%	28.0%	72.0%
No. 200	75	158.00	15.7%	43.7%	56.3%
No. 270	53	103.70	10.3%	54.1%	45.9%
<No. 270	< 53	460.80	45.9%	100.0%	0.0%
Total		1003.40	100.0%	Calc'd P80, µm	125
				Calc'd P80, in.	0.005



	Engineer	J. Axen	Test ID	High Grade
	Technician	MR	Time (min)	30
	Project Name	Getchell Gold	Date	7/11/2024
FA.F-TS01	Project No.	23041		

Original Sample Weight, g	1007.1	RPM	53

Screen Size		Wet Screen Analysis			
Inches/ US mesh	microns	Sample Weight, g	Weight Distribution, %	Cumulative Weight Retained, %	Cumulative Weight Passing, %
No. 100	150	3.60	0.4%	0.4%	99.6%
No. 150	105	14.10	1.4%	1.8%	98.2%
No. 200	75	115.70	11.5%	13.2%	86.8%
No. 270	53	186.80	18.5%	31.8%	68.2%
<No. 270	< 53	686.90	68.2%	100.0%	0.0%
Total		1007.10	100.0%	Calc'd P80, µm	69
				Calc'd P80, in.	0.003



Appendix F Grind Study Data

FORTE ANALYTICAL

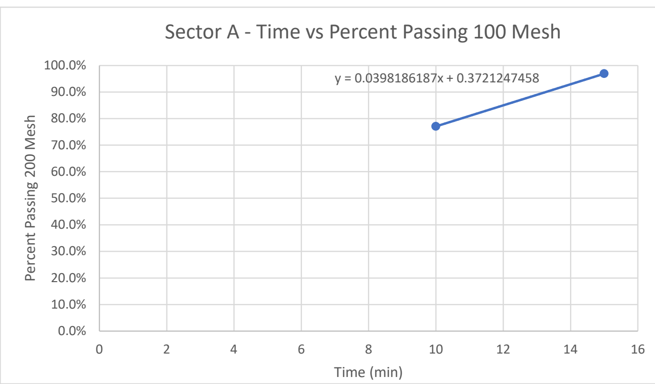
	A	B	C
Tab Name:	10 min	15 min	30 min

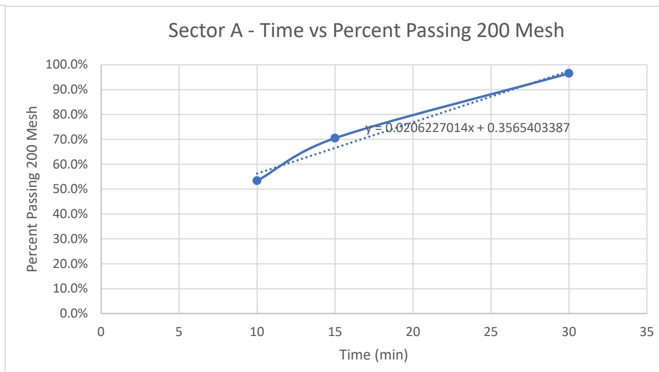
Low Grade Comp

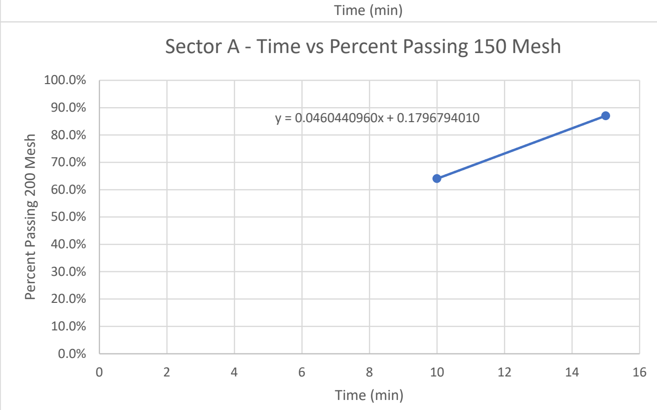
			Col of Sizes	Low Grade			
			C	Low Grade 10 min	Low Grade 15 min	Low Grade 30 min	
Mesh Size	Col of Data	Row of Data	**Size Fraction, µm**	**Cumulative Weight Passing, %**			
	J	2		**Low Grade**	**Low Grade**	**Low Grade**	
100	J	11	**150**	77.0%	96.9%	99.8%	
150	J	12	**105**	64.0%	87.0%	99.5%	
200	J	13	**75**	53.4%	70.5%	96.5%	
270	J	14	**53**	45.8%	58.2%	85.8%	
<270	J	15	**< 53**	0.0%	0.0%	0.0%	

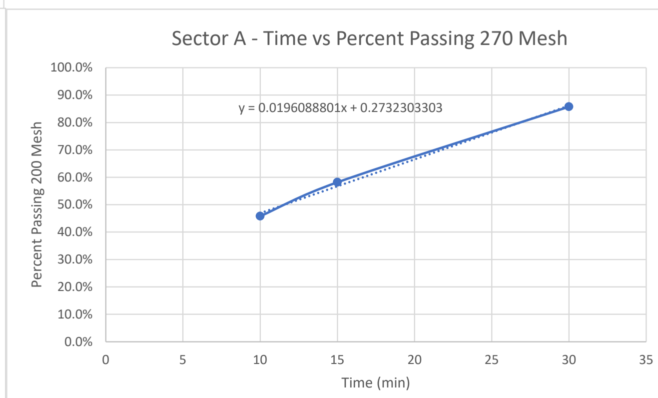
Col of Data	Row of Data					
J	20	**0**	0	0	0	
J	3	**Time, minutes**	10	15	30	
		Mesh Size	100	150	200	

Sector A (100 Mesh)	Sector A (150 Mesh)	Sector A (200 Mesh)	Sector A (270 Mesh)
80%	80%	80%	80%
11	13	22	27





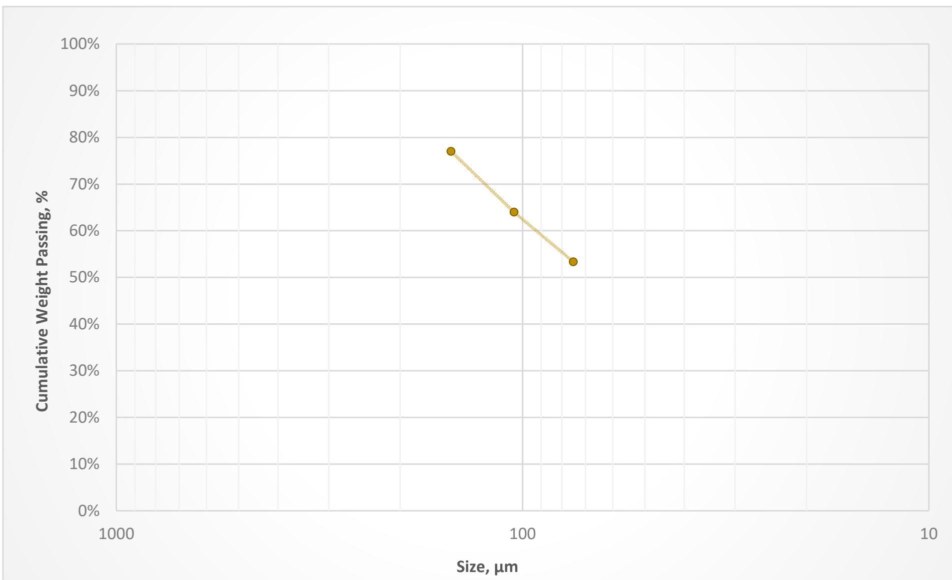




FA.F-TS01	**Engineer**	J.Axen	**Test ID**	Low Grade
	Technician	MR	**Time (min)**	10
	Project Name	Getchell Gold	**Date**	7/12/2024
	Project No.	23041		

Original Sample Weight, g	997	**RPM**	53.2

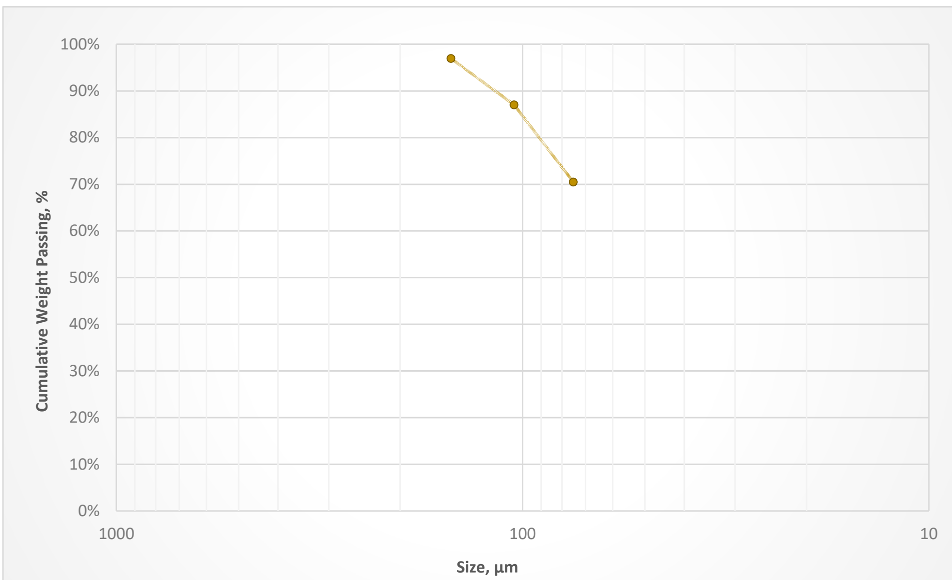
Screen Size		**Wet Screen Analysis**			
Inches/ US mesh	**microns**	**Sample Dry Weight, g**	**Weight Distribution, %**	**Cumulative Weight Retained, %**	**Cumulative Weight Passing, %**
No. 100	150	229.00	23.0%	23.0%	77.0%
No. 150	105	129.80	13.0%	36.0%	64.0%
No. 200	75	106.30	10.7%	46.6%	53.4%
No. 270	53	74.80	7.5%	54.2%	45.8%
<No. 270	< 53	457.10	45.8%	100.0%	0.0%
Total		**997.00**	**100.0%**	**Calc'd P80, µm**	156
				Calc'd P80, in.	0.0061



	Engineer	J. Axen	Test ID	Low Grade
	Technician	MR	Time (min)	15
	Project Name	Getchell Gold	Date	7/10/2024
FA.F-TS01	Project No.	23041		

Original Sample Weight, g	1003.4	RPM	53.3

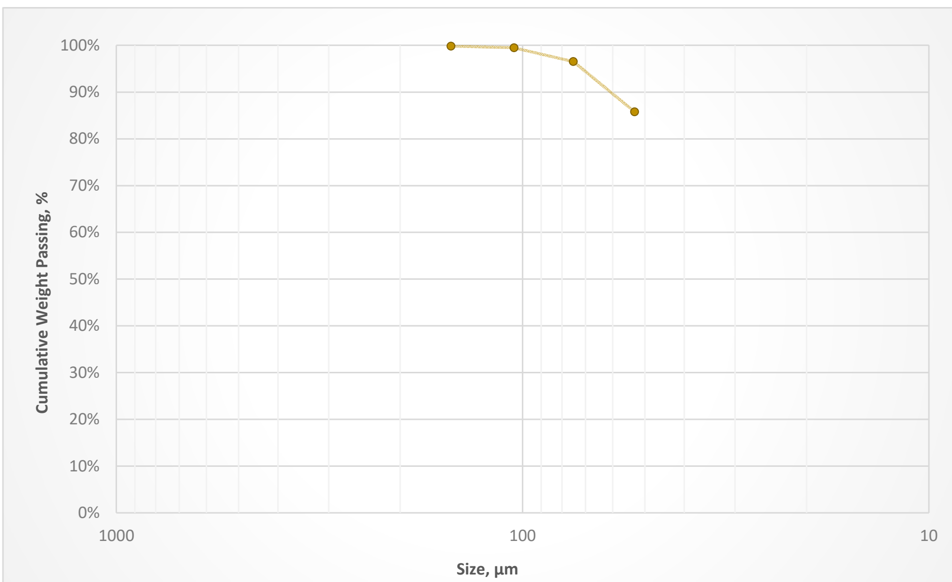
Screen Size		Wet Screen Analysis			
Inches/ US mesh	microns	Sample Weight, g	Weight Distribution, %	Cumulative Weight Retained, %	Cumulative Weight Passing, %
No. 100	150	30.70	3.1%	3.1%	96.9%
No. 150	105	99.40	9.9%	13.0%	87.0%
No. 200	75	166.00	16.5%	29.5%	70.5%
No. 270	53	123.50	12.3%	41.8%	58.2%
<No. 270	< 53	583.80	58.2%	100.0%	0.0%
Total		1003.40	100.0%	Calc'd P80, µm	92
				Calc'd P80, in.	0.004



FA.F-TS01	Engineer	J. Axen	Test ID	Low Grade
	Technician	MR	Time (min)	30
	Project Name	Getchell Gold	Date	7/11/2024
	Project No.	23041		

Original Sample Weight, g	1004.8	RPM	53

Screen Size		Wet Screen Analysis			
Inches/ US mesh	microns	Sample Weight, g	Weight Distribution, %	Cumulative Weight Retained, %	Cumulative Weight Passing, %
No. 100	150	1.60	0.2%	0.2%	99.8%
No. 150	105	3.70	0.4%	0.5%	99.5%
No. 200	75	29.40	2.9%	3.5%	96.5%
No. 270	53	108.10	10.8%	14.2%	85.8%
<No. 270	< 53	862.00	85.8%	100.0%	0.0%
Total		1004.80	100.0%	Calc'd P80, µm	62
				Calc'd P80, in.	0.002



Appendix F Grind Study Data

Oxid Comp 2

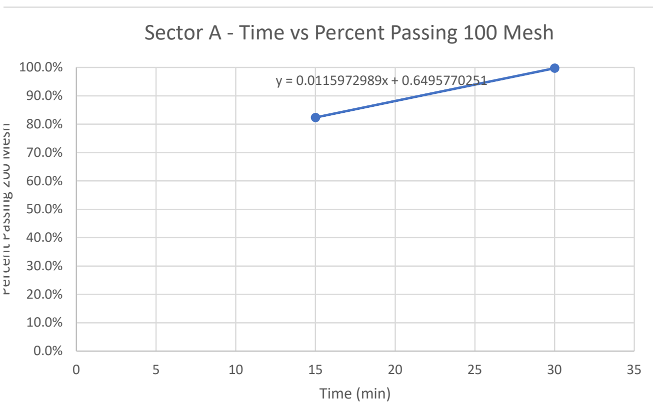
	A	B	C
Tab Name:	15 min	30 min	45 min

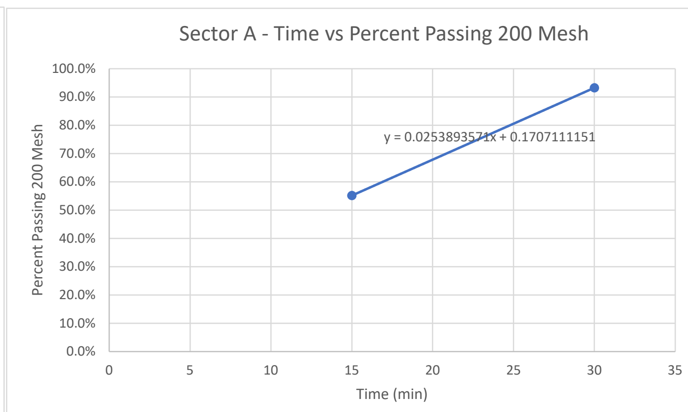
	Oxide Comp 2			
Col of Sizes C	Oxide Comp 2 15 min	Oxide Comp 2 30 min	Oxide Comp 2 45 min	

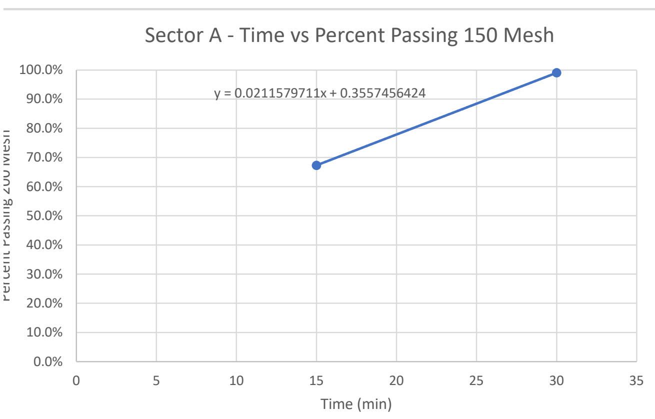
Mesh Size	Col of Data	Row of Data	Size Fraction, µm	Cumulative Weight Passing, %			
	J	2		Oxide Comp 2	Oxide Comp 2	Oxide Comp 2	
100	J	11	150	82.4%	99.7%	99.8%	
150	J	12	105	67.3%	99.0%	99.6%	
200	J	13	75	55.2%	93.2%	99.1%	
270	J	14	53	46.5%	78.4%	94.0%	
<270	J	15	< 53	0.0%	0.0%	0.0%	

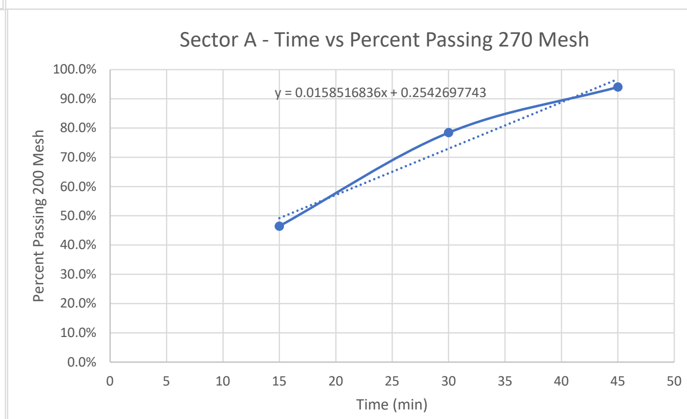
Col of Data	Row of Data					
J	20	0	0	0	0	
J	3	Time, minutes	15	30	45	
		Mesh Size	100	150	200	

Sector A (100 Mesh)	Sector A (150 Mesh)	Sector A (200 Mesh)	Sector A (270 Mesh)
80%	80%	80%	80%
13	21	25	34





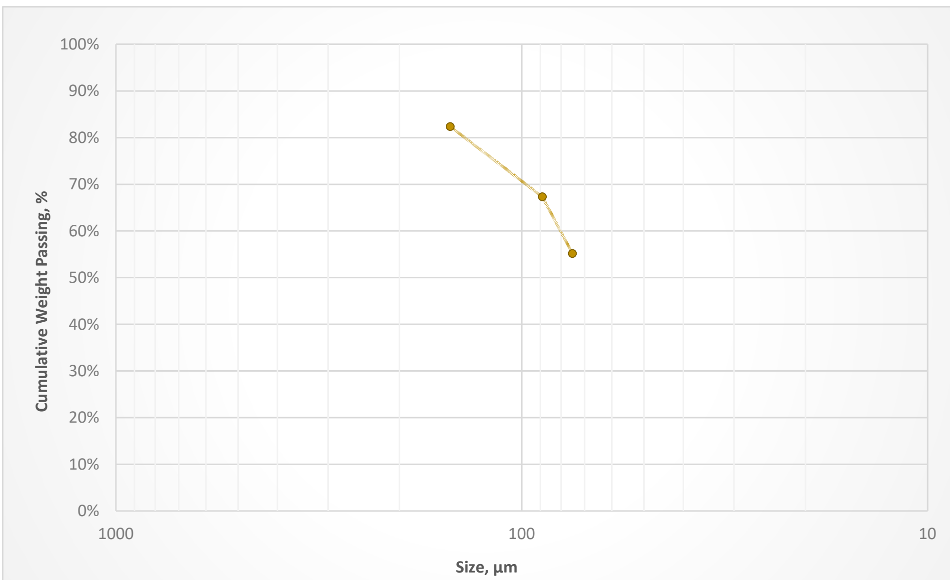




	Engineer	J. Axen	Test ID	Oxide Comp 2
	Technician	TK	Time (min)	15
	Project Name	Getchell Gold	Date	7/2/2024
FA.F-TS01	Project No.	23041		

Original Sample Weight, g	994.54	RPM	53.3

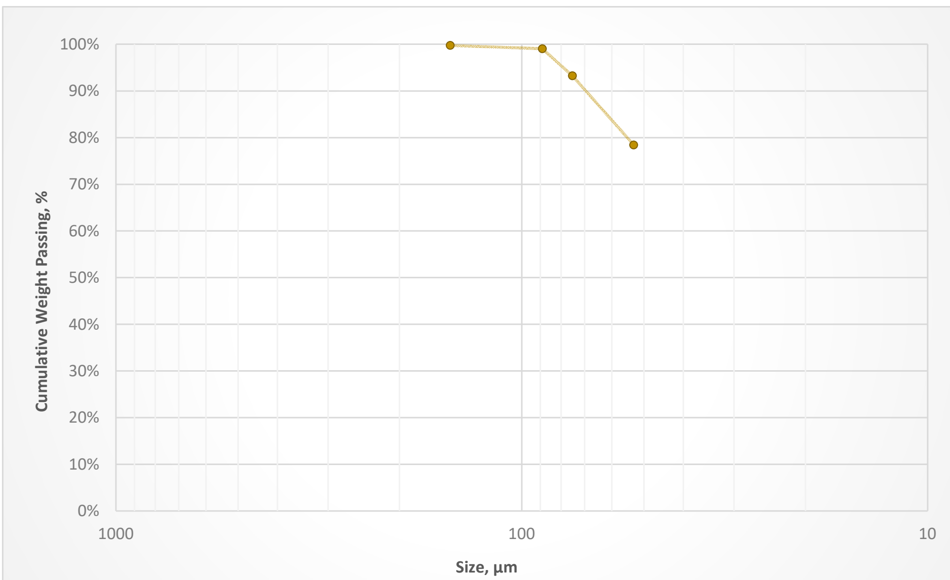
Screen Size		Wet Screen Analysis			
Inches/ US mesh	microns	Sample Weight, g	Weight Distribution, %	Cumulative Weight Retained, %	Cumulative Weight Passing, %
No. 100	150	175.50	17.6%	17.6%	82.4%
No. 150	89	149.60	15.0%	32.7%	67.3%
No. 200	75	120.90	12.2%	44.8%	55.2%
No. 270	53	86.30	8.7%	53.5%	46.5%
<No. 270	< 53	462.24	46.5%	100.0%	0.0%
Total		994.54	100.0%	Calc'd P80, µm	140
				Calc'd P80, in.	0.006



	Engineer	J. Axen	Test ID	Oxide Comp 2
FORTE ANALYTICAL	Technician	TK	Time (min)	30
	Project Name	Getchell Gold	Date	7/3/2024
FA.F-TS01	Project No.	23041		

Original Sample Weight, g	998.4	RPM	53

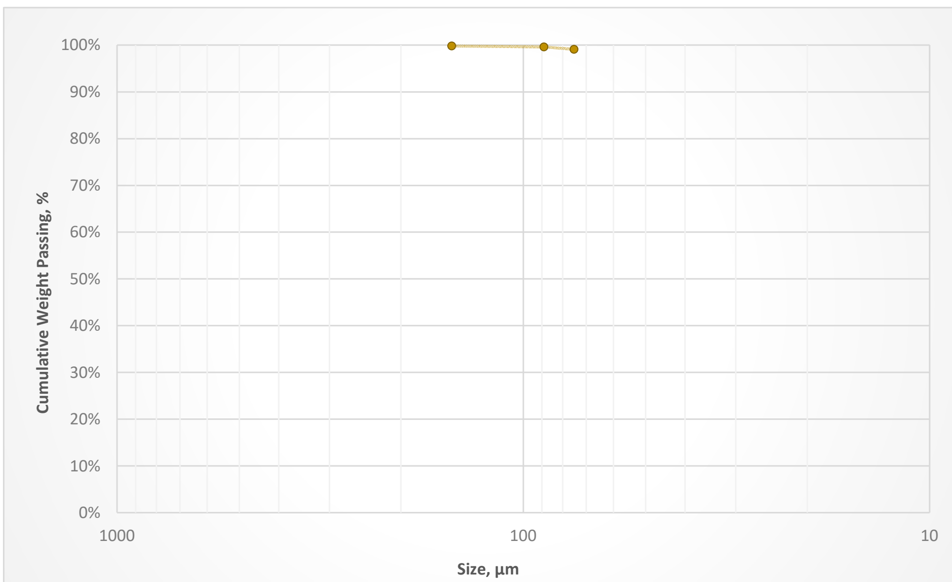
Screen Size		Wet Screen Analysis			
Inches/ US mesh	microns	Sample Weight, g	Weight Distribution, %	Cumulative Weight Retained, %	Cumulative Weight Passing, %
No. 100	150	2.50	0.3%	0.3%	99.7%
No. 150	89	7.00	0.7%	1.0%	99.0%
No. 200	75	58.00	5.8%	6.8%	93.2%
No. 270	53	147.80	14.8%	21.6%	78.4%
<No. 270	< 53	783.10	78.4%	100.0%	0.0%
Total		998.40	100.0%	Calc'd P80, µm	64
				Calc'd P80, in.	0.003



FA.F-TS01	Engineer	J.Axen	Test ID	Oxide Comp 2
	Technician	TK	Time (min)	45
	Project Name	Getchell Gold	Date	7/3/2024
	Project No.	23041		

Original Sample Weight, g	1005.5	RPM	53.2

Screen Size		Wet Screen Analysis			
Inches/ US mesh	microns	Sample Dry Weight, g	Weight Distribution, %	Cumulative Weight Retained, %	Cumulative Weight Passing, %
No. 100	150	1.90	0.2%	0.2%	99.8%
No. 150	89	2.00	0.2%	0.4%	99.6%
No. 200	75	5.20	0.5%	0.9%	99.1%
No. 270	53	50.90	5.1%	6.0%	94.0%
<No. 270	< 53	945.50	94.0%	100.0%	0.0%
Total		1005.50	100.0%	Calc'd P80, µm	61
				Calc'd P80, in.	0.0024



APPENDIX G: FLOTATION TEST DATA

Appendix G Flotation Testing

Getchell Testwork Summary

Updated: 8/8/2024

Author: JA

Flotation Tests on Average Grade Composite at Varying Particle Sizes

Average Grade

Stage	GTFT-1, P80 100 M						
	Cumulative Flotation Time (min)	Product Weight (g)	Mass Distribution (%)	Cumulative % Mass Distribution	Au (mg/kg)	Individual Au Distribution (%)	Cumulative Au Distribution (%)
Feed (Measured)		1000			1.49		
Feed (Calculated)		1000.7	100.0		1.57	100.0	
Rougher Conc-1	3	45.3	4.52	4.52	15.32	44.2	44.2
Rougher Conc-2	6	35.7	3.57	8.09	7.61	17.32	61.5
Rougher Conc-3	9	30.3	3.03	11.1	3.68	7.12	68.7
Rougher Conc-4	12	36.1	3.61	14.7	2.24	5.2	73.8
Rougher Tail		853.3	85.3	100.0	0.48	26.2	100.0

Average Grade

Stage	GTFT-2, P80 150 M						
	Cumulative Flotation Time (min)	Product Weight (g)	Mass Distribution (%)	Cumulative % Mass Distribution	Au (mg/kg)	Individual Au Distribution mg	Cumulative Au Distribution (%)
Feed (Measured)		1000			1.49		
Feed (Calculated)		1001.7	100.0		1.53	100.0	
Rougher Conc-1	3	61.4	6.13	6.13	13.27	53.2	53.2
Rougher Conc-2	6	41.1	4.11	10.2	5.02	13.5	66.7
Rougher Conc-3	9	40.8	4.08	14.3	2.39	6.37	73.1
Rougher Conc-4	12	32.5	3.24	17.6	1.80	3.82	76.9
Rougher Tail		825.8	82.4	100.0	0.43	23.1	100.0

Average Grade

Stage	GTFT-3, P80 200 M						
	Cumulative Flotation Time (min)	Product Weight (g)	Mass Distribution (%)	Cumulative % Mass Distribution	Au (mg/kg)	Individual Au Distribution (%)	Cumulative Au Distribution (%)
Feed (Measured)		1000			1.49		
Feed (Calculated)		1003.3	100.0		1.79	100.0	
Rougher Conc-1	3	65.5	6.53	6.53	13.8	50.2	50.2
Rougher Conc-2	6	57.4	5.72	12.2	7.40	23.6	73.8
Rougher Conc-3	9	37.2	3.71	16.0	2.21	4.57	78.4
Rougher Conc-4	12	34.8	3.47	19.4	1.69	3.3	81.6
Rougher Tail		808.4	80.6	100.0	0.41	18.4	100.0

Average Grade

Stage	GTFT-4, P80 270 M						
	Cumulative Flotation Time (min)	Product Weight (g)	Mass Distribution (%)	Cumulative % Mass Distribution	Au (mg/kg)	Individual Au Distribution (%)	Cumulative Au Distribution (%)
Feed (Measured)		1000			1.49		
Feed (Calculated)		1009.3	100.0		1.48	100.0	
Rougher Conc-1	3	49.8	4.94	4.94	15.5	51.7	51.7
Rougher Conc-2	6	50.6	5.01	9.95	5.23	17.7	69.4
Rougher Conc-3	9	40.6	4.02	14.0	2.42	6.58	76.0
Rougher Conc-4	12	40.6	4.02	18.0	1.48	4.0	80.0
Rougher Tail		827.6	82.0	100.0	0.36	20.0	100.0

Notes:
Ag feed assay (measured) is assay data from Florin Analytical Labs.

Average Grade

Stage	GTGT-1, Gravity Test on GTFT-3 Tail, P80 200 M Product Weight (g)	Mass Distribution (%)	Cumulative % Mass Distribution	Au (mg/kg)	Individual Au Distribution (%)	Cumulative Au Distribution (%)	Ag (mg/kg)
Feed (Measured)	1000			41.00			BD
Feed (Calculated)	732.0	100.0		0.25	100.0		BD
Gemini Conc	3.4	0.46	0.46	3.57	6.5	6.5	BD
Gemini Tail	41.5	5.66	6.13	1.22	27.1	33.6	BD
Knelson Tail	687.1	93.87	100.0	0.18	66.36	100.0	BD

Size by Size Analysis of Getchell Oxide Batch 1

Oxide Batch 1 (as rec'd)	Product Weight (g)	Mass Distribution (%)	Cumulative % Mass Distribution	Au (mg/kg)	Individual Au Distribution (%)	Cumulative % Au Distribution	mg Au
Feed (analyzed)	200			TBD			134.9
Feed (calculated)	200.5	100		1.35	100		
+100 M	138	68.8	68.8	1.24	63.3	63.3	85.35
100x400 M	27.8	13.9	82.7	1.39	14.3	77.5	19.3
-400 M	34.7	17.3	100.0	1.75	22.5	100.0	30.3

Bulk Flotation to produce a concentrate for CN leaching
Average Grade

Stage	GTFT-5, P80 270 M Flotation Time (min)	Product Weight (g)	Mass Distribution (%)	Cumulative % Mass Distribution	Au (mg/kg)	Individual Au Distribution (%)	Cumulative Au Distribution (%)
Feed (Measured)		10000			1.49		
Feed (Calculated)		10000.0	100.0		1.40	100.0	
Rougher Conc-1	18	2070.1	20.70	20.7	6.07	89.8	89.8
Rougher Conc-2		7929.9	79.30	100.0	0.18	10.2	100.0

Scavenger Flotation of Bulk Flotation Tails (GTFT-5)
Average Grade

Stage	GTFT-5, P80 270 M Flotation Time (min)	Product Weight (g)	Mass Distribution (%)	Cumulative % Mass Distribution	Au (mg/kg)	Individual Au Distribution (%)	Cumulative Au Distribution (%)
Feed (Measured)		7930			0.18		
Feed (Calculated)		7929.9	100.0		0.18	100.0	
Scav Conc 1	4.5	621.9	7.84	7.84	0.76	32.4	32.4
Scav Conc 2	4.5	86.6	1.09	8.94	0.55	3.27	35.7
Scav Conc 3	4.5	81.1	1.02	10.0	0.12	0.67	36.3
Scav Tail (calc)		7140.3	90.04	100.0	0.13	63.7	100.0

Kinetic Flotation at P80 200, 270 M
Average Grade

Stage	GTFT-6, P80 200 M Flotation Time (min)	Product Weight (g)	Mass Distribution (%)	Cumulative % Mass Distribution	Au (mg/kg)	Individual Au Distribution (%)	Cumulative Au Distribution (%)	Au (mg)	Cumulative Au Grade (mg/kg)
Feed (Measured)		1000			1.49				
Feed (Calculated)		1000.0	100.0		1.60	100.0			
Rougher Conc-1	3	87.4	8.74	8.74	12.8	69.6	69.6	1.12	12.8
Rougher Conc-2	3	43.5	4.35	13.1	2.94	7.98	77.6	0.13	9.51
Rougher Conc-3	3	42.7	4.27	17.4	1.62	4.31	81.9	0.07	7.57
Rougher Conc-4	3	39.1	3.91	21.3	1.22	2.98	84.9	0.05	6.40
Rougher Tail		787.4	78.7	100.0	0.31	15.1	100.0	0.24	1.60

Average Grade

Stage	GTFT-7, P80 270 M								
	Flotation Time (min)	Product Weight (g)	Mass Distribution (%)	Cumulative % Mass Distribution	Au (mg/kg)	Individual Au Distribution (%)	Cumulative Au Distribution (%)	Au (mg)	Cumulative Au Grade (mg/kg)
Feed (Measured)		1000			1.49				
Feed (Calculated)		1001.1	100.0		1.51	100.0			
Rougher Conc-1	3	105.3	10.52	10.5	9.6	67.1	67.1	1.01	9.62
Rougher Conc-2	3	63.8	6.37	16.9	2.89	12.2	79.3	0.18	7.08
Rougher Conc-3	3	44.7	4.47	21.3	1.48	4.37	83.7	0.07	5.91
Rougher Conc-4	3	54.1	5.40	26.8	1.00	3.59	87.3	0.05	4.92
Rougher Tail		733.3	73.2	100.0	0.26	12.7	100.0	0.19	1.51

Low Grade

Stage	GTFT-8, P80 200 M								
	Flotation Time (min)	Product Weight (g)	Mass Distribution (%)	Cumulative % Mass Distribution	Au (mg/kg)	Individual Au Distribution (%)	Cumulative Au Distribution (%)	Au (mg)	Cumulative Au Grade (mg/kg)
Feed (Measured)		1000			0.53				
Feed (Calculated)		1000.0	100.0		0.51	100.0			
Rougher Conc-1	3	86.2	8.62	8.62	3.19	53.5	53.5	0.27	3.19
Rougher Conc-2	3	57.6	5.76	14.4	1.19	13.3	66.8	0.07	2.39
Rougher Conc-3	3	48.7	4.86	19.2	0.70	6.64	73.5	0.03	1.96
Rougher Conc-4	3	41.1	4.11	23.4	0.70	5.61	79.1	0.03	1.74
Rougher Tail		766.5	76.6	100.0	0.14	20.9	100.0	0.11	0.51

Low Grade

Stage	GTFT-9, P80 270 M								
	Flotation Time (min)	Product Weight (g)	Mass Distribution (%)	Cumulative % Mass Distribution	Au (mg/kg)	Individual Au Distribution (%)	Cumulative Au Distribution (%)	Au (mg)	Cumulative Au Grade (mg/kg)
Feed (Measured)		1000			0.53				
Feed (Calculated)		1001.0	100.0		0.54	100.0			
Rougher Conc-1	3	74.2	7.41	7.41	4.27	58.6	58.6	0.32	4.27
Rougher Conc-2	3	49.2	4.92	12.3	0.96	8.78	67.4	0.05	2.95
Rougher Conc-3	3	42.3	4.22	16.6	0.71	5.57	73.0	0.03	2.38
Rougher Conc-4	3	40.7	4.06	20.6	0.54	4.05	77.0	0.02	2.01
Rougher Tail		794.6	79.4	100.0	0.16	23.0	100.0	0.12	0.54

High Grade

Stage	GTFT-10, P80 200 M								
	Flotation Time (min)	Product Weight (g)	Mass Distribution (%)	Cumulative % Mass Distribution	Au (mg/kg)	Individual Au Distribution (%)	Cumulative Au Distribution (%)	Au (mg)	Cumulative Au Grade (mg/kg)
Feed (Measured)		1000			4.88				
Feed (Calculated)		998.5	100.0		5.45	100.0			
Rougher Conc-1	3	81.0	8.12	8.12	45.1	67.2	67.2	3.66	45.1
Rougher Conc-2	3	45.3	4.54	12.7	11.0	9.20	76.4	0.50	32.9
Rougher Conc-3	3	41.6	4.16	16.8	7.13	5.44	81.8	0.30	26.5
Rougher Conc-4	3	39.0	3.90	20.7	4.85	3.47	85.3	0.19	22.4
Rougher Tail		791.6	79.3	100.0	1.01	14.7	100.0	0.80	5.45

High Grade

Stage	GTFT-11, P80 270 M								
	Flotation Time (min)	Product Weight (g)	Mass Distribution (%)	Cumulative % Mass Distribution	Au (mg/kg)	Individual Au Distribution (%)	Cumulative Au Distribution (%)	Au (mg)	Cumulative Au Grade (mg/kg)
Feed (Measured)		1000			4.88				
Feed (Calculated)		1004.6	100.0		5.47	100.0			
Rougher Conc-1	3	81.4	8.11	8.11	44.8	66.3	66.3	3.65	44.8
Rougher Conc-2	3	49.5	4.92	13.0	11.1	10.0	76.3	0.55	32.1
Rougher Conc-3	3	46.4	4.62	17.7	6.62	5.58	81.9	0.31	25.4
Rougher Conc-4	3	45.2	4.50	22.2	4.75	3.90	85.8	0.21	21.2
Rougher Tail		782.1	77.8	100.0	1.00	14.2	100.0	0.78	5.47

Average Grade 10 kg Bulk Concentrate Production for CIL

	GTFT-12, P80 270 M								
Stage	Flotation Time (min)	Product Weight (g)	Mass Distribution (%)	Cumulative % Mass Distribution	Au (mg/kg)	Individual Au Distribution (%)	Cumulative Au Distribution (%)	Au (mg)	Cumulative Au Grade (mg/kg)
Feed (Measured)		10000			1.49				
Feed (Calculated)		10000.0	100.0		1.59	79.3			
Bulk Concentrate	18	2070.1	20.70	20.7	6.23	67.1	81.1	12.90	6.23
Rougher Tail		7929.9	79.30	100.0	0.38	12.2	100.0	3.01	1.59

Low Grade 3 x 1 kg Bulk Concentrate Production for CIL

	GTFT-13, P80 270 M								
Stage	Flotation Time (min)	Product Weight (g)	Mass Distribution (%)	Cumulative % Mass Distribution	Au (mg/kg)	Individual Au Distribution (%)	Cumulative Au Distribution (%)	Au (mg)	Cumulative Au Grade (mg/kg)
Feed (Measured)		10000			0.53				
Feed (Calculated)		3000.0	100.0		0.54	100.0			
Bulk Concentrate	12*	783.2	26.11	26.1	1.62	78.2	78.2	1.27	1.62
Rougher Tail		2216.8	73.89	100.0	0.16	21.8	100.0	0.35	0.54

High Grade 3 x 1 kg Bulk Concentrate Production for CIL

	GTFT-14, P80 270 M								
Stage	Flotation Time (min)	Product Weight (g)	Mass Distribution (%)	Cumulative % Mass Distribution	Au (mg/kg)	Individual Au Distribution (%)	Cumulative Au Distribution (%)	Au (mg)	Cumulative Au Grade (mg/kg)
Feed (Measured)		10000			4.88				
Feed (Calculated)		3000.0	100.0		5.31	100.0			
Bulk Concentrate	12*	791.0	26.4	26.4	17.2	85.4	85.4	13.6	17.2
Rougher Tail		2209.0	73.6	100.0	1.05	14.6	100.0	2.32	5.31

*Flotation time is for 1 kg float, not the total bulk concentrate collection time over the three floats

Appendix G: Flotation Testing

Engineer	JA	**Test ID**	Flotation Test 1
Technician	SK	**Date**	4/18/2024
Project Name	Getchell Gold	**Sample**	Average Grade
Project No.	23041	**P80**	100 mesh

Purpose: Kinetic Flotation Test to Evaluate Particle Size Optimization

Sample: Approximately 1000 g of Average Grade Composite, P80 100 Mesh

Procedure: A 1 kg charge was floated as a kinetic test generating 4 concentrates and a tail for analysis.

	Conditions				Reagents, g/mt of flotation feed				
Operations	min.	Solids %	pH-Start	pH-End	PAX	404	MIBC	AF-65	RPM
Rod Mill Grinding	13.0	50		8.0	0	0	0	0	1650
Conditioning	1.0	26	8.0	8.1	50	50	20	5	1650
Rougher 1	3.0	26	8.1	8.3	0	0	0	0	1650
Conditioning	1.0	25	8.3	8.3	25	25	5	5	1650
Rougher 2	3.0	25	8.3	8.3	0	0	0	0	1650
Conditioning	1.0	24	8.3	8.3	25	25	5	5	1650
Rougher 3	3.0	24	8.3	8.2	0	0	0	0	1650
Conditioning	1.0	24	8.2	8.3	25	25	5	5	1650
Rougher 4	3.0	24	8.3	8.3	0	0	0	0	1650
Total Reagent Used, g/mt of feed					125	125	35	20	
Solution Concentration (% or g/drop)					1.00%	1.00%	0.0050	0.0050	

Results:

Products	Weight		Chemical Analysis		Percent Distribution	
	gr.	%	Au mg/kg	Ag mg/kg	Au	Ag
Feed (analyzed)	1,000		1.49	1.1		
Feed (calculated)	1,000.7	100.0	1.57	8.6	100.0	100.0
Rougher Conc 1	45.3	4.5	15.32	12.5	44.2	6.6
Rougher Conc 2	35.7	3.6	7.61	14.1	17.3	5.8
Rougher Conc 3	30.3	3.0	3.68	7.6	7.1	2.7
Rougher Conc 4	36.1	3.6	2.24	8.1	5.2	3.4
Rougher Tail	853.3	85.3	0.48	8.2	26.2	81.5

Observations:

1 kg charge in the 750 g cell at ~1650 rpm

Rougher Conc 1+2	80.9	8.1	11.92	13.2	61.5	12.4
Rougher Conc 1+2+3	111.3	11.1	9.68	11.7	68.7	15.1
Combined Rougher Conc	147.4	14.7	7.86	10.8	73.8	18.5

Appendix G: Flotation Testing

Engineer	JA	**Test ID**	Flotation Test 2
Technician	SK	**Date**	4/18/2024
Project Name	Getchell Gold	**Sample**	Average Grade
Project No.	23041	**P80**	150 mesh

Purpose: Kinetic Flotation Test to Evaluate Particle Size Optimization

Sample: Approximately 1000 g of Average Grade Composite, P80 150 Mesh

Procedure: A 1 kg charge was floated as a kinetic test generating 4 concentrates and a tail for analysis.

	Conditions				Reagents, g/mt of flotation feed				
Operations	min.	Solids %	pH-Start	pH-End	PAX	404	MIBC	AF-65	RPM
Rod Mill Grinding	15.0	50		8.0	0	0	0	0	1650
Conditioning	1.0	26	8.0	8.1	50	50	10	10	1650
Rougher 1	3.0	26	8.1	8.3	0	0	0	0	1650
Conditioning	1.0	25	8.3	8.3	25	25	5	5	1650
Rougher 2	3.0	25	8.3	8.3	0	0	0	0	1650
Conditioning	1.0	24	8.3	8.3	25	25	5	5	1650
Rougher 3	3.0	24	8.3	8.3	0	0	0	0	1650
Conditioning	1.0	23	8.3	8.3	25	25	5	5	1650
Rougher 4	3.0	23	8.3	8.3	0	0	0	0	1650
Total Reagent Used, g/mt of feed					125	125	25	25	
Solution Concentration (% or g/drop)					1.00%	1.00%	0.0050	0.0050	

Results:

Products	Weight		Chemical Analysis		Percent Distribution	
	gr.	%	Au mg/kg	Ag mg/kg	Au	Ag
Feed (analyzed)	1,000		1.49	1.1		
Feed (calculated)	1,001.7	100.0	1.53	10.4	100.0	100.0
Rougher Conc 1	61.4	6.1	13.27	8.1	53.2	4.8
Rougher Conc 2	41.1	4.1	5.02	54.5	13.5	21.5
Rougher Conc 3	40.8	4.1	2.39	48.2	6.4	18.9
Rougher Conc 4	32.5	3.2	1.80	0.1	3.8	0.0
Rougher Tail	825.8	82.4	0.43	6.9	23.1	54.8

Observations:

1 kg charge in the 750 g cell at ~1650 rpm

Rougher Conc 1+2	102.5	10.2	9.96	26.7	66.7	26.3
Rougher Conc 1+2+3	143.4	14.3	7.80	32.8	73.1	45.1
Combined Rougher Conc	175.9	17.6	6.69	26.8	76.9	45.2

Appendix G: Flotation Testing

Engineer	JA	**Test ID**	Flotation Test 3
Technician	SK	**Date**	4/18/2024
Project Name	Getchell Gold	**Sample**	Average Grade
Project No.	23041	**P80**	200 mesh

Purpose: Kinetic Flotation Test to Evaluate Particle Size Optimization

Sample: Approximately 1000 g of Average Grade Composite, P80 200 Mesh

Procedure: A 1 kg charge was floated as a kinetic test generating 4 concentrates and a tail for analysis.

	Conditions				Reagents, g/mt of flotation feed				
Operations	min.	Solids %	pH-Start	pH-End	PAX	404	MIBC	AF-65	RPM
Rod Mill Grinding	23.0	50		8.0	0	0	0	0	1650
Conditioning	1.0	26	8.0	8.1	50	50	10	10	1650
Rougher 1	3.0	26	8.1	8.4	0	0	0	0	1650
Conditioning	1.0	25	8.4	8.4	25	25	5	5	1650
Rougher 2	3.0	25	8.4	NA	0	0	0	0	1650
Conditioning	1.0	23	NA	8.4	25	25	5	5	1650
Rougher 3	3.0	23	8.4	8.4	0	0	0	0	1650
Conditioning	1.0	23	8.4	8.4	25	25	5	5	1650
Rougher 4	3.0	23	8.4	NA	0	0	0	0	1650
Total Reagent Used, g/mt of feed					125	125	25	25	
Solution Concentration (% or g/drop)					1.00%	1.00%	0.0050	0.0050	

Results:

Products	Weight		Chemical Analysis		Percent Distribution	
	gr.	%	Au mg/kg	Ag mg/kg	Au	Ag
Feed (analyzed)	1,000		1.49	1.1		
Feed (calculated)	1,003.3	100.0	1.79	10.2	100.0	100.0
Rougher Conc 1	65.5	6.5	13.80	24.0	50.2	15.4
Rougher Conc 2	57.4	5.7	7.40	14.4	23.6	8.1
Rougher Conc 3	37.2	3.7	2.21	21.0	4.6	7.7
Rougher Conc 4	34.8	3.5	1.69	15.2	3.3	5.2
Rougher Tail	808.4	80.6	0.41	8.1	18.4	63.7

Observations:

1 kg charge in the 750 g cell at ~1650 rpm

Rougher Conc 1+2	122.9	12.2	10.81	19.5	73.8	23.5
Rougher Conc 1+2+3	160.1	16.0	8.81	19.9	78.4	31.1
Combined Rougher Conc	194.9	19.4	7.54	19.0	81.6	36.3

Appendix G: Flotation Testing

Engineer	JA	**Test ID**	GTGT-1
Technician	SK	**Date**	5/20/2024
Project Name	Getchell Gold	**Sample**	Average Grade
Project No.	23041	**P80**	200 mesh

Purpose: Gravity test of GTFT-3 Tails to evaluate gold deportment in flotation tails

Sample: Approximately 730 g of GTFT-3 Rougher 1 Tail

Procedure: 730 g was run through Knelson concentration, the Knelson concentrate was run through the Gemini and the products assayed for gold.

Results:

Products	Weight gr.	Weight %	Chemical Analysis Au mg/kg	Chemical Analysis Ag mg/kg	Percent Distribution Au	Percent Distribution Ag
Feed (analyzed)	732		0.41	1		
Feed (calculated)	732.0	100.0	0.25	0	100.0	100.0
Gemini Conc	3.40	0.5	3.57	0	6.5	0.5
Gemini Tail	41.5	5.7	1.22	0	27.1	5.7
Knelson Tail	687.1	93.9	0.18	0	66.4	93.9

Observations:

Knelson Conc	44.9	6.1	1.40	0.4	33.6	6.1

Appendix G: Flotation Testing

Engineer	JA	**Test ID**	Flotation Test 4
Technician	SK	**Date**	5/9/2024
Project Name	Getchell Gold	**Sample**	Average Grade
Project No.	23041	**P80**	270 mesh

Purpose: Kinetic Flotation Test to Evaluate Particle Size Optimization

Sample: Approximately 1000 g of Average Grade Composite, P80 270 Mesh

Procedure: A 1 kg charge was floated as a kinetic test generating 4 concentrates and a tail for analysis.

	Conditions				Reagents, g/mt of flotation feed				
Operations	min.	Solids %	pH-Start	pH-End	PAX	404	MIBC	AF-65	RPM
Rod Mill Grinding	28.0	50		8.0	0	0	0	0	1650
Conditioning	1.0	26	8.0	8.1	50	50	10	0	1650
Rougher 1	3.0	26	8.1	8.4	0	0	0	0	1650
Conditioning	1.0	25	8.4	8.4	25	25	10	0	1650
Rougher 2	3.0	25	8.4	8.4	0	0	0	0	1650
Conditioning	1.0	24	8.4	8.4	25	25	10	0	1650
Rougher 3	3.0	24	8.4	8.4	0	0	0	0	1650
Conditioning	1.0	23	8.4	8.4	25	25	10	0	1650
Rougher 4	3.0	23	8.4	8.4	0	0	0	0	1650
Total Reagent Used, g/mt of feed					125	125	40	0	
Solution Concentration (% or g/drop)					1.00%	1.00%	0.0050	0.0050	

Results:

Products	Weight		Chemical Analysis		Percent Distribution	
	gr.	%	Au mg/kg	Ag mg/kg	Au	Ag
Feed (analyzed)	1,000		1.49	1.1		
Feed (calculated)	1,009.3	100.0	1.48	4.6	100.0	100.0
Rougher Conc 1	49.8	4.9	15.5	27.2	51.7	28.9
Rougher Conc 2	50.6	5.0	5.23	11.2	17.7	12.1
Rougher Conc 3	40.6	4.0	2.42	3.4	6.6	2.9
Rougher Conc 4	40.6	4.0	1.48	14.8	4.0	12.8
Rougher Tail	827.6	82.0	0.36	2.5	20.0	43.3

Observations:

1 kg charge in the 750 g cell at ~1650 rpm

Rougher Conc 1+2	100.4	9.9	10.32	19.1	69.4	41.0
Rougher Conc 1+2+3	141.0	14.0	8.04	14.6	76.0	43.9
Combined Rougher Conc	181.7	18.0	6.58	14.6	80.0	56.7

Appendix G: Flotation Testing

Engineer	JA	**Test ID**	Flotation Test 5
Technician	SK	**Date**	6/20/2024
Project Name	Getchell Gold	**Sample**	Average Grade
Project No.	23041	**P80**	270 mesh

Purpose: A bulk flotation to generate a concentrate for leach tests

Sample: Approximately 10000 g of Average Grade Composite, P80 270 Mesh

Procedure: A 10 kg charge was floated and the concentrates were combined for downstream testing.

	Conditions						Reagents, g/mt of flotation feed				
Operations	min.	Solids %	pH-Start	pH-End			PAX	404	MIBC	AF-65	
Rod Mill Grinding	29.0	50		8.0			0	0	0	0	
Conditioning	1.0	29	8.0	8.1			50	50	4	0	
rougher 1	4.5	29	8.1	8.4			0	0	0	0	
Conditioning	1.0	29	8.4	8.4			25	25	4	0	
Rougher 2	4.5	29	8.4	8.4			0	0	0	0	
Conditioning	1.0	29	8.4	8.4			25	25	4	0	
Rougher 3	4.5	29	8.4	8.4			0	0	0	0	
Conditioning	1.0	29	8.4	8.4			25	25	4	0	
Rougher 4	4.5	29	8.4	8.4			0	0	0	0	
Total Reagent Used, g/mt of feed							125	125	16	0	
Solution Concentration (% or g/drop)							1.00%	1.00%	0.0050	0.0050	

Results:

Products	Weight		Chemical Analysis						Percent Distribution			Cum. % Distribution		
	gr.	%	Au mg/kg	Ag mg/kg					Au	Ag	0	Au	Ag	0
Feed (analyzed)	10,000		1.49	1										
Feed (calculated)	10,000.0	100.0	1.40	1					100.0	100.0				
Bulk Ro Conc	2,070.1	20.7	6.07	4					89.8	91.3		89.8	91.3	
Rougher Tail (calc)	7,929.9	79.3	0.18	0					10.2	8.7		100.0	100.0	

Observations:

wet conc cake weight 2620 g

10 kg charge in the cubic foot cell

Appendix G: Flotation Testing

Engineer	JA	**Test ID**	Flotation Test 5 Scavenger
Technician	SK	**Date**	6/24/2024
Project Name	Getchell Gold	**Sample**	Average Grade
Project No.	23041	**P80**	270 mesh

Purpose: Kinetic Flotation Test to Evaluate Scavenger Flotation

Sample: Approximately 1000 g of Average Grade Composite, P80 270 Mesh

Procedure: a 1 kg charge was floated as a kinetic test generating 4 concentrates and a tail for analysis.

Operations	Conditions				Reagents, g/mt of flotation feed			
	min.	Solids %	pH-Start	pH-End	PAX	404	MIBC	AF-65
Rod Mill Grinding	0.0	50		8.0	0	0	0	0
Conditioning	1.0	24	8.0	8.1	63	63	5	0
Scavenger 1	3.0	24	8.1	8.4	0	0	0	0
Conditioning	1.0	22	8.4	8.4	32	32	5	0
Scavenger 2	3.0	22	8.4	8.4	0	0	0	0
Conditioning	1.0	22	8.4	8.4	32	32	5	0
Scavenger 3	3.0	22	8.4	8.4	0	0	0	0
Conditioning	1.0	22	8.4	8.4	32	32	5	0
Scavenger 4	3.0	22	8.4	8.4	0	0	0	0
Total Reagent Used, g/mt of feed					158	158	20	0
Solution Concentration (% or g/drop)					1.00%	1.00%	0.0050	0.0050

Results:

Products	Weight		Chemical Analysis		Percent Distribution	
	gr.	%	Au mg/kg	Ag mg/kg	Au	Ag
Feed (analyzed)	7,930		0.18	0.1		
Feed (calculated)	7,929.9	100.0	0.18	1.1	100.0	100.0
Scav Conc 1	621.9	7.8	0.76	2	32.4	14.1
Scav Conc 2	86.6	1.1	0.55	5	3.3	4.9
Scav Conc 3	81.1	1.0	0.12	0	0.7	0.1
Scav Tail (calc)	7,140.3	90.0	0.13	1	63.7	80.9

Observations:

8 kg charge in the cubic foot cell

Scav Con 1+2	708.6	8.9	0.73	2.4	35.7	19.0
Combined Scav Con	789.6	10.0	0.67	2.1	36.3	19.1

Appendix G: Flotation Testing

Engineer	JA	**Test ID**	Flotation Test 6
Technician	SK	**Date**	7/16/2024
Project Name	Getchell Gold	**Sample**	Average Grade
Project No.	23041	**P80**	200 mesh

Purpose: Kinetic Flotation Test to Evaluate Particle Size Optimization

Sample: Approximately 1000 g of Average Grade Composite, P80 200 Mesh

Procedure: A 1 kg charge was floated as a kinetic test generating 4 concentrates and a tail for analysis.

	Conditions				Reagents, g/mt of flotation feed				
Operations	min.	Solids %	pH-Start	pH-End	PAX	404	MIBC	AF-65	RPM
Rod Mill Grinding	23.0	50		8.1	0	0	0	0	1650
Conditioning	1.0	26	8.1	8.1	50	50	15	5	1650
Rougher 1	3.0	26	8.1	8.4	0	0	0	0	1650
Conditioning	1.0	24	8.4	8.4	25	25	5	5	1650
Rougher 2	3.0	24	8.4	8.3	0	0	0	0	1650
Conditioning	1.0	23	8.3	8.3	25	25	10	0	1650
Rougher 3	3.0	23	8.3	8.3	0	0	0	0	1650
Conditioning	1.0	22	8.3	8.3	25	25	10	0	1650
Rougher 4	3.0	22	8.3	8.3	0	0	0	0	1650
Total Reagent Used, g/mt of feed					125	125	40	10	
Solution Concentration (% or g/drop)					1.00%	1.00%	0.0050	0.0050	

Results:

Products	Weight		Chemical Analysis		Percent Distribution	
	gr.	%	Au mg/kg	Ag mg/kg	Au	Ag
Feed (analyzed)	1,000		1.49	1.1		
Feed (calculated)	1,000.0	100.0	1.60	3.0	100.0	100.0
Rougher Conc 1	87.4	8.7	12.8	15.4	69.6	45.3
Rougher Conc 2	43.5	4.3	2.94	0.6	8.0	0.9
Rougher Conc 3	42.7	4.3	1.62	3.7	4.3	5.3
Rougher Conc 4	39.1	3.9	1.22	10.6	3.0	14.0
Rougher Tail	787.4	78.7	0.31	1.3	15.1	34.5

Observations:

1 kg charge in the 750 g cell at ~1650 rpm

Rougher Conc 1+2	130.8	13.1	9.51	10.5	77.6	46.2
Rougher Conc 1+2+3	173.6	17.4	7.57	8.8	81.9	51.5
Combined Rougher Conc	212.7	21.3	6.40	9.1	84.9	65.5

Appendix G: Flotation Testing

Engineer	JA	**Test ID**	Flotation Test 7
Technician	SK	**Date**	7/16/2024
Project Name	Getchell Gold	**Sample**	Average Grade
Project No.	23041	**P80**	270 mesh

Purpose: Kinetic Flotation Test to Evaluate Particle Size Optimization

Sample: Approximately 1000 g of Average Grade Composite, P80 270 Mesh

Procedure: A 1 kg charge was floated as a kinetic test generating 4 concentrates and a tail for analysis.

Operations	Conditions: min.	Solids %	pH-Start	pH-End	Reagents, g/mt of flotation feed: PAX	404	MIBC	AF-65	RPM
Rod Mill Grinding	28.0	50		8.1	0	0	0	0	1650
Conditioning	1.0	26	8.1	8.1	50	50	15	5	1650
Rougher 1	3.0	26	8.1	8.4	0	0	0	0	1650
Conditioning	1.0	24	8.4	8.4	25	25	5	5	1650
Rougher 2	3.0	24	8.4	8.3	0	0	0	0	1650
Conditioning	1.0	22	8.3	8.3	25	25	10	0	1650
Rougher 3	3.0	22	8.3	8.3	0	0	0	0	1650
Conditioning	1.0	21	8.3	8.3	25	25	10	0	1650
Rougher 4	3.0	21	8.3	8.3	0	0	0	0	1650
Total Reagent Used, g/mt of feed					125	125	40	10	
Solution Concentration (% or g/drop)					1.00%	1.00%	0.0050	0.0050	

Results:

Products	Weight gr.	Weight %	Chemical Analysis Au mg/kg	Ag mg/kg	Percent Distribution Au	Ag
Feed (analyzed)	1,000		1.49	1.1		
Feed (calculated)	1,001.1	100.0	1.51	3.1	100.0	100.0
Rougher Conc 1	105.3	10.5	9.62	12.9	67.1	44.5
Rougher Conc 2	63.8	6.4	2.89	13.9	12.2	29.1
Rougher Conc 3	44.7	4.5	1.48	12.4	4.4	18.2
Rougher Conc 4	54.1	5.4	1.00	0.6	3.6	1.1
Rougher Tail	733.3	73.2	0.26	0.3	12.7	7.2

Observations:

1 kg charge in the 750 g cell at ~1650 rpm

Rougher Conc 1+2	169.0	16.9	7.08	13.3	79.3	73.6
Rougher Conc 1+2+3	213.7	21.3	5.91	13.1	83.7	91.7
Combined Rougher Conc	267.8	26.8	4.92	10.6	87.3	92.8

Appendix G: Flotation Testing

Engineer	JA	**Test ID**	Flotation Test 8
Technician	SK	**Date**	7/16/2024
Project Name	Getchell Gold	**Sample**	Low Grade
Project No.	23041	**P80**	200 mesh

Purpose: Kinetic Flotation Test to Evaluate Particle Size Optimization

Sample: Approximately 1000 g of Low Grade Composite, P80 200 Mesh

Procedure: A 1 kg charge was floated as a kinetic test generating 4 concentrates and a tail for analysis.

	Conditions				Reagents, g/mt of flotation feed				
Operations	min.	Solids %	pH-Start	pH-End	PAX	404	MIBC	AF-65	RPM
Rod Mill Grinding	22.0	50		8.0	0	0	0	0	1650
Conditioning	1.0	26	8.0	8.1	50	50	15	5	1650
Rougher 1	3.0	26	8.1	8.3	0	0	0	0	1650
Conditioning	1.0	24	8.3	8.3	25	25	5	5	1650
Rougher 2	3.0	24	8.3	8.3	0	0	0	0	1650
Conditioning	1.0	23	8.3	8.3	25	25	10	0	1650
Rougher 3	3.0	23	8.3	8.2	0	0	0	0	1650
Conditioning	1.0	22	8.2	8.2	25	25	10	0	1650
Rougher 4	3.0	22	8.2	8.2	0	0	0	0	1650
Total Reagent Used, g/mt of feed					125	125	40	10	
Solution Concentration (% or g/drop)					1.00%	1.00%	0.0050	0.0050	

Results:

Products	Weight		Chemical Analysis		Percent Distribution	
	gr.	%	Au mg/kg	Ag mg/kg	Au	Ag
Feed (analyzed)	1,000		0.53	1.1		
Feed (calculated)	1,000.0	100.0	0.51	0.8	100.0	100.0
Rougher Conc 1	86.2	8.6	3.19	3.2	53.5	33.1
Rougher Conc 2	57.6	5.8	1.19	0.6	13.3	4.2
Rougher Conc 3	48.7	4.9	0.70	0.6	6.6	3.6
Rougher Conc 4	41.1	4.1	0.70	0.6	5.6	3.0
Rougher Tail	766.5	76.6	0.14	0.6	20.9	56.1

Observations:

1 kg charge in the 750 g cell at ~1650 rpm

Rougher Conc 1+2	143.8	14.4	2.39	2.1	66.8	37.3
Rougher Conc 1+2+3	192.4	19.2	1.96	1.7	73.5	40.9
Combined Rougher Conc	233.5	23.4	1.74	1.5	79.1	43.9

Appendix G: Flotation Testing

Engineer	JA	**Test ID**	Flotation Test 9
Technician	SK	**Date**	7/16/2024
Project Name	Getchell Gold	**Sample**	Low Grade
Project No.	23041	**P80**	270 mesh

Purpose: Kinetic Flotation Test to Evaluate Particle Size Optimization

Sample: Approximately 1000 g of Low Grade Composite, P80 270 Mesh

Procedure: A 1 kg charge was floated as a kinetic test generating 4 concentrates and a tail for analysis.

	Conditions						Reagents, g/mt of flotation feed				
Operations	min.	Solids %	pH-Start	pH-End			PAX	404	MIBC	AF-65	RPM
Rod Mill Grinding	27.0	50		8.0			0	0	0	0	1650
Conditioning	1.0	26	8.0	8.1			50	50	15	5	1650
Rougher 1	3.0	26	8.1	8.4			0	0	0	0	1650
Conditioning	1.0	24	8.4	8.3			25	25	5	5	1650
Rougher 2	3.0	24	8.3	8.3			0	0	0	0	1650
Conditioning	1.0	23	8.3	8.3			25	25	10	0	1650
Rougher 3	3.0	23	8.3	8.2			0	0	0	0	1650
Conditioning	1.0	22	8.2	8.2			25	25	10	0	1650
Rougher 4	3.0	22	8.2	8.1			0	0	0	0	1650
Total Reagent Used, g/mt of feed							125	125	40	10	
Solution Concentration (% or g/drop)							1.00%	1.00%	0.0050	0.0050	

Results:

Products	Weight		Chemical Analysis						Percent Distribution		
	gr.	%	Au mg/kg	Ag mg/kg					Au	Ag	
Feed (analyzed)	1,000		0.53	1.1							
Feed (calculated)	1,001.0	100.0	0.54	1.0					100.0	100.0	
Rougher Conc 1	74.2	7.4	4.27	4.7					58.6	35.0	
Rougher Conc 2	49.2	4.9	0.96	2.5					8.8	12.1	
Rougher Conc 3	42.3	4.2	0.71	0.6					5.6	2.5	
Rougher Conc 4	40.7	4.1	0.54	0.6					4.1	2.4	
Rougher Tail	794.6	79.4	0.16	0.6					23.0	47.8	

Observations:

1 kg charge in the 750 g cell at ~1650 rpm

Rougher Conc 1+2	123.4	12.3	2.95	3.8	67.4	47.2
Rougher Conc 1+2+3	165.7	16.6	2.38	3.0	73.0	49.7
Combined Rougher Conc	206.4	20.6	2.01	2.5	77.0	52.2

Appendix G: Flotation Testing

Engineer	JA	**Test ID**	Flotation Test 10
Technician	SK	**Date**	7/16/2024
Project Name	Getchell Gold	**Sample**	High Grade
Project No.	23041	**P80**	200 mesh

Purpose: Kinetic Flotation Test to Evaluate Particle Size Optimization

Sample: Approximately 1000 g of High Grade Composite, P80 200 Mesh

Procedure: A 1 kg charge was floated as a kinetic test generating 4 concentrates and a tail for analysis.

	Conditions						Reagents, g/mt of flotation feed				
Operations	min.	Solids %	pH-Start	pH-End			PAX	404	MIBC	AF-65	RPM
Rod Mill Grinding	27.0	50		7.8			0	0	0	0	1650
Conditioning	1.0	26	7.8	7.8			50	50	15	5	1650
Rougher 1	3.0	26	7.8	8.3			0	0	0	0	1650
Conditioning	1.0	24	8.3	8.2			25	25	5	5	1650
Rougher 2	3.0	24	8.2	8.2			0	0	0	0	1650
Conditioning	1.0	23	8.2	8.2			25	25	10	0	1650
Rougher 3	3.0	23	8.2	8.1			0	0	0	0	1650
Conditioning	1.0	22	8.1	8.1			25	25	10	0	1650
Rougher 4	3.0	22	8.1	8.1			0	0	0	0	1650
Total Reagent Used, g/mt of feed							125	125	40	10	
Solution Concentration (% or g/drop)							1.00%	1.00%	0.0050	0.0050	

Results:

Products	Weight		Chemical Analysis		Percent Distribution	
	gr.	%	Au mg/kg	Ag mg/kg	Au	Ag
Feed (analyzed)	1,000		4.88	3.0		
Feed (calculated)	998.5	100.0	5.45	7.0	100.0	100.0
Rougher Conc 1	81.0	8.1	45.1	0.6	67.2	0.7
Rougher Conc 2	45.3	4.5	11.0	43.3	9.2	28.0
Rougher Conc 3	41.6	4.2	7.13	0.6	5.4	0.4
Rougher Conc 4	39.0	3.9	4.85	0.6	3.5	0.3
Rougher Tail	791.6	79.3	1.01	6.3	14.7	70.7

Observations:

1 kg charge in the 750 g cell at ~1650 rpm

Rougher Conc 1+2	126.4	12.7	32.90	15.9	76.4	28.7
Rougher Conc 1+2+3	167.9	16.8	26.52	12.1	81.8	29.0
Combined Rougher Conc	206.9	20.7	22.44	10.0	85.3	29.3

Appendix G: Flotation Testing

Engineer	JA	**Test ID**	Flotation Test 11
Technician	SK	**Date**	7/16/2024
Project Name	Getchell Gold	**Sample**	High Grade
Project No.	23041	**P80**	270 mesh

Purpose: Kinetic Flotation Test to Evaluate Particle Size Optimization

Sample: Approximately 1000 g of High Grade Composite, P80 270 Mesh

Procedure: A 1 kg charge was floated as a kinetic test generating 4 concentrates and a tail for analysis.

	Conditions						Reagents, g/mt of flotation feed				
Operations	min.	Solids %	pH-Start	pH-End			PAX	404	MIBC	AF-65	RPM
Rod Mill Grinding	38.0	50		8.0			0	0	0	0	1650
Conditioning	1.0	26	8.0	8.0			50	50	15	5	1650
Rougher 1	3.0	26	8.0	8.4			0	0	0	0	1650
Conditioning	1.0	24	8.4	8.2			25	25	5	5	1650
Rougher 2	3.0	24	8.2	8.3			0	0	0	0	1650
Conditioning	1.0	23	8.3	8.2			25	25	10	0	1650
Rougher 3	3.0	23	8.2	8.2			0	0	0	0	1650
Conditioning	1.0	22	8.2	8.2			25	25	10	0	1650
Rougher 4	3.0	22	8.2	8.1			0	0	0	0	1650
Total Reagent Used, g/mt of feed							125	125	40	10	
Solution Concentration (% or g/drop)							1.00%	1.00%	0.0050	0.0050	

Results:

Products	Weight		Chemical Analysis		Percent Distribution	
	gr.	%	Au mg/kg	Ag mg/kg	Au	Ag
Feed (analyzed)	1,000		4.88	3.0		
Feed (calculated)	1,004.6	100.0	5.47	3.2	100.0	100.0
Rougher Conc 1	81.4	8.1	44.8	8.0	66.3	20.5
Rougher Conc 2	49.5	4.9	11.10	0.6	10.0	0.9
Rougher Conc 3	46.4	4.6	6.62	0.6	5.6	0.9
Rougher Conc 4	45.2	4.5	4.75	0.6	3.9	0.9
Rougher Tail	782.1	77.8	1.00	3.1	14.2	76.8

Observations:

1 kg charge in the 750 g cell at ~1650 rpm

Rougher Conc 1+2	130.9	13.0	32.06	5.2	76.3	21.5
Rougher Conc 1+2+3	177.3	17.7	25.40	4.0	81.9	22.3
Combined Rougher Conc	222.5	22.2	21.20	3.3	85.8	23.2

Appendix G: Flotation Testing

Engineer	JA	**Test ID**	Flotation Test 12
Technician	SK	**Date**	7/26/2024
Project Name	Getchell Gold	**Sample**	Average Grade
Project No.	23041	**P80**	270 mesh

Purpose: bulk flotation to generate a concentrate for CIL Tests

Sample: Approximately 10,000 g of Average Grade Composite, P80 270 Mesh

Procedure: A 10 kg charge was floated and the concentrates were combined for downstream leach tests.

Operations	Conditions				Reagents, g/mt of flotation feed				
	min.	Solids %	pH-Start	pH-End	PAX	404	MIBC	AF-65	RPM
Rod Mill Grinding	28.0	50		7.9	0	0	0	0	1650
Conditioning	1.0	29	7.9	7.9	50	50	18	2	1650
Rougher 1	4.5	29	7.9	8.3	0	0	0	0	1650
Conditioning	1.0	29	8.3	8.2	25	25	8	2	1650
Rougher 2	4.5	29	8.2	8.3	0	0	0	0	1650
Conditioning	1.0	29	8.3	8.3	25	25	10	0	1650
Rougher 3	4.5	29	8.3	8.3	0	0	0	0	1650
Conditioning	1.0	29	8.3	8.3	25	25	10	0	1650
Rougher 4	4.5	29	8.3	8.4	0	0	0	0	1650
Total Reagent Used, g/mt of feed					125	125	46	4	
Solution Concentration (% or g/drop)					1.00%	1.00%	0.01	0.0050	

Results:

Products	Weight		Chemical Analysis		Percent Distribution			Cum. % Distribution		
	gr.	%	Au mg/kg	Ag mg/kg	Au	Ag	0	Au	Ag	0
Feed (analyzed)	10,000		1.49	1						
Feed (calculated)	10,000.0	100.0	1.59	0	100.0	100.0				
Bulk Ro Conc	2,070.1	20.7	6.23	0	81.1	43.9		81.1	43.9	
Rougher Tail (calc)	7,929.9	79.3	0.38	0	18.9	56.1		100.0	100.0	

Observations:

10 kg charge in the cubic foot cell

Appendix G: Flotation Testing

Engineer	JA	**Test ID**	Flotation Test 13
Technician	SK	**Date**	8/1/2024
Project Name	Getchell Gold	**Sample**	Low Grade
Project No.	23041	**P80**	270 mesh

Purpose: 3 x 1 kg test to produce a bulk concentrate for CIL Testing

Sample: Approximately 3 x 1000 g of Low Grade Composite, P80 270 Mesh

Procedure: Three 1 kg charges were floated and the products combined to produce a bulk concentrate for CIL testing

	Conditions				Reagents, g/mt of flotation feed				
Operations	min.	Solids %	pH-Start	pH-End	PAX	404	MIBC	AF-65	RPM
Rod Mill Grinding	27.0	50		8.3	0	0	0	0	1650
Conditioning	1.0	26	8.3	8.3	50	50	15	5	1650
Rougher 1	3.0	26	8.3	8.6	0	0	0	0	1650
Conditioning	1.0	26	8.6	8.6	25	25	10	5	1650
Rougher 2	3.0	26	8.6	8.5	0	0	0	0	1650
Conditioning	1.0	26	8.5	8.5	25	25	10	5	1650
Rougher 3	3.0	26	8.5	8.4	0	0	0	0	1650
Conditioning	1.0	26	8.4	8.4	25	25	10	5	1650
Rougher 4	3.0	26	8.4	8.4	0	0	0	0	1650
Total Reagent Used, g/mt of feed					125	125	45	20	
Solution Concentration (% or g/drop)					1.00%	1.00%	0.0050	0.0050	

Results:

Products	Weight		Chemical Analysis		Percent Distribution			Cum. % Distribution		
	gr.	%	Au	Ag	Au	Ag	0	Au	Ag	0
	1000		mg/kg	mg/kg						
Feed (analyzed)	3,000		0.53	1						
Feed (calculated)	3,000.0	100.0	0.54	0	100.0	100.0				
Rougher Conc	783.2	26.1	1.62	1	78.2	74.6		78.2	74.6	
Rougher Tail	2,216.8	73.9	0.16	0	21.8	25.4		100.0	100.0	

Observations:

1 kg charge in the 750 g cell at ~1650 rpm

Appendix G: Flotation Testing

Engineer	JA	**Test ID**	Flotation Test 14
Technician	SK	**Date**	8/1/2024
Project Name	Getchell Gold	**Sample**	High Grade
Project No.	23041	**P80**	270 mesh

Purpose: 3 x 1 kg test to produce a bulk concentrate for CIL Testing

Sample: Approximately 3 x 1000 g of High Grade Composite, P80 270 Mesh

Procedure: Three 1 kg charges were floated and the products combined to produce a bulk concentrate for CIL testing

	Conditions				Reagents, g/mt of flotation feed				
Operations	min.	Solids %	pH-Start	pH-End	PAX	404	MIBC	AF-65	RPM
Rod Mill Grinding	38.0	50		8.2	0	0	0	0	1650
Conditioning	1.0	26	8.2	8.2	50	50	15	5	1650
Rougher 1	3.0	26	8.2	8.6	0	0	0	0	1650
Conditioning	1.0	26	8.6	8.6	25	25	10	5	1650
Rougher 2	3.0	26	8.6	8.6	0	0	0	0	1650
Conditioning	1.0	26	8.6	8.5	25	25	10	5	1650
Rougher 3	3.0	26	8.5	8.5	0	0	0	0	1650
Conditioning	1.0	26	8.5	8.5	25	25	10	5	1650
Rougher 4	3.0	26	8.5	8.4	0	0	0	0	1650
Total Reagent Used, g/mt of feed					125	125	45	20	
Solution Concentration (% or g/drop)					1.00%	1.00%	0.0050	0.0050	

Results:

Products	Weight		Chemical Analysis		Percent Distribution			Cum. % Distribution		
	gr.	%	Au	Ag	Au	Ag	0	Au	Ag	0
	1000		mg/kg	mg/kg						
Feed (analyzed)	3,000		4.88	3						
Feed (calculated)	3,000.0	100.0	5.31	1	100.0	100.0				
Rougher Conc	791.0	26.4	17.20	4	85.4	96.6		85.4	96.6	
Rougher Tail	2,209.0	73.6	1.05	0	14.6	3.4		100.0	100.0	

Observations:

1 kg charge in the 750 g cell at ~1650 rpm

July 11, 2024
Lab no. 224143

Ms. Jess Axen
Forte Analytical
11475 West I-70 Frontage Road North
Wheat Ridge, Colorado 80033

Dear Ms. Axen:

Enclosed are the x-ray diffraction (XRD) results for sample "23041 GTFT-5 Bulk Conc" received earlier this week. This report will be mailed and emailed to you.

A representative portion the sample was ground to approximately -400 mesh in a steel swing mill, packed into a well-type plastic holder and then scanned with the diffractometer over the range, 3-61° 2θ using Cu-Kα radiation. The results of the scan are summarized as approximate mineral weight percent concentrations on the enclosed table. Estimates of mineral concentrations were made using our XRF-determined elemental composition and the relative peak areas on the XRD scan. The detection limit for an average mineral in this sample is 1-3% and the analytical reproducibility is approximately equal to the square root of the amount. "Unidentified" accounts for that portion of the XRD scan which could not be resolved.

Thank you for the opportunity to be of continuing service Forte Analytical.

Sincerely,

Peggy Dalheim

Research Development - Forte Analytical | **July 11, 2024**

XRD Results for Sample "23041 GTFT-5 Bulk Conc." | **Lab no. 224143**

Mineral Name	Chemical Formula	Approx. Wt %
Quartz	SiO_2	37
Mica/illite	$(K,Na,Ca)(Al,Mg,Fe)_2(Si,Al)_4O_{10}(OH,F)_2$	35
Kaolinite	$Al_2Si_2O_5(OH)_4$	<3
Dolomite	$Ca(Mg,Fe)(CO_3)_2$	7
Calcite	$CaCO_3$	<2
Rutile	TiO_2	<1
Pyrite	FeS_2	13
Arsenopyrite	FeAsS	<2
"Unidentified"	?	<5

Initial __________

Date ____________

Analysis performed by The Mineral Lab, Inc

APPENDIX H: WHOLE ORE LEACH TEST DATA

Appendix H

Bottle Roll Leaching Test 4

Project: 23041-Getchell Gold
Date: 15-Jul-24

Purpose: To examine gold and silver leach amenability at Different Grind Sizes

Sample: Approximately 1000 g of Oxide Comp 1, 100 M

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~11 with hydrated lime and sodium cyanide was added to a calculated level of 1.0 g/l.At 2, 4, 8, and 24 hours, the pH and free cyanide were determined. A sample of solution was removed and assayed for gold. Sodium cyanide was added to return the level to 1.0 g/L and the pH was adjusted to 11 with hydrated lime if needed. After 48 hours, the solution was measured to determine pH, free cyanide, and gold contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold and silver contents by fire assay techniques.

Conditions:

Grind	Leach Time	NaCN Concentration	% Solids	
80% minus 100 mesh	48 hours	1.00 g/L	40% Solids	

Summary of Results:

Parameter	Au	Ag
Extraction, % (1)	41.2	NA
Assayed Head, g/mt	1.38	2.0
Calculated Head, g/mt	1.34	1.0
Final Tail Assay, g/mt	0.79	1.0

Hr.	Extraction, % (1) Au	Ag	NaCN Consumed kg/mt
2	50.3	NA	0.239
4	50.3	NA	0.179
8	48.8	NA	0.239
24	44.3	NA	0.300
48	41.2	NA	0.362

Cyanide Consumption 0.362 kg NaCN/metric ton ore
Lime Added 2.903 kg $Ca(OH)_2$/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g NaCN	$Ca(OH)_2$	Carbon		Residueal Reagents NaCN g/L	Dissolved Oxygen Initial mg/L	Adjust mg/L	pH Initial	Adjust
0	2500	1501	1.50	0.90						8.4	10.7
2	2500	1501	0.24	1.40			0.84			10.0	10.8
4	2500	1501					1.04			10.6	
8	2500	1501		0.40			1.00			10.4	10.7
24	2499	1500	0.06	0.20			0.96			10.5	10.7
48	2498	1498					0.96			10.6	
Total			1.80	2.90		0.00					
				(2.2)	CaO Equivalent						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L			Volume Thief
Feed (analyzed)	1000		1.38		2.0				
Feed (computed)			1.34		1.0				
2 hour Preg		1501		0.45		NA			25
4 hour Preg		1501		0.44		NA			25
8 hour Preg		1501		0.42		NA			25
24 hour Preg		1500		0.38		NA			25
48 hour Preg		1498		0.34		NA			25
48 hour Carbon									
48 hour Dry Residue	999.1		0.79		1.0				

(1) Based on calculated head assays.

Appendix H

Bottle Roll Leaching Test 5

Project: 23041-Getchell Gold
Date: 15-Jul-24

Purpose: To examine gold and silver leach amenability at Different Grind Sizes

Sample: Approximately 1000 g of Oxide Comp 1, 200 M

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~11 with hydrated lime and sodium cyanide was added to a calculated level of 1.0 g/l.At 2, 4, 8, and 24 hours, the pH and free cyanide were determined. A sample of solution was removed and assayed for gold. Sodium cyanide was added to return the level to 1.0 g/L and the pH was adjusted to 11 with hydrated lime if needed. After 48 hours, the solution was measured to determine pH, free cyanide, and gold contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold and silver contents by fire assay techniques.

Conditions:

Grind	Leach Time	NaCN Concentration	% Solids	
80% minus 200 mesh	48 hours	1.00 g/L	40% Solids	

Summary of Results:

Parameter	Au	Ag
Extraction, % (1)	37.6	NA
Assayed Head, g/mt	1.38	2.0
Calculated Head, g/mt	1.31	2.0
Final Tail Assay, g/mt	0.82	2.0

Extraction, % (1) Hr.	Au	Ag	NaCN Consumed kg/mt
2	51.4	NA	0.240
4	50.9	NA	0.120
8	48.6	NA	0.174
24	42.7	NA	0.300
48	37.6	NA	0.422

Cyanide Consumption	0.422	kg NaCN/metric ton ore
Lime Added	2.699	kg $Ca(OH)_2$/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g NaCN	$Ca(OH)_2$	Carbon		Residueal Reagents NaCN g/L	Dissolved Oxygen Initial mg/L	Adjust mg/L	pH Initial	Adjust
0	2500	1500	1.50	1.00						8.5	10.6
2	2500	1500	0.24	1.70			0.84			10.0	11.1
4	2500	1500					1.08			10.8	
8	2506	1506					1.04			10.7	
24	2500	1500	0.06				0.96			10.6	
48	2498	1498					0.92			10.4	
Total			1.80	2.70		0.00					
				(2.0)	CaO Equivalent						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L			Volume Thief
Feed (analyzed)	1000		1.38		2.0				
Feed (computed)			1.31		2.0				
2 hour Preg		1500		0.45		NA			25
4 hour Preg		1500		0.44		NA			25
8 hour Preg		1506		0.41		NA			25
24 hour Preg		1500		0.35		NA			25
48 hour Preg		1498		0.30		NA			25
48 hour Carbon									
48 hour Dry Residue	1000.3		0.82		2.0				

(1) Based on calculated head assays.

Appendix H

Bottle Roll Leaching Test 6

Project: 23041-Getchell Gold
Date: 11-Jul-24

Purpose: To examine gold and silver leach amenability at Different Grind Sizes

Sample: Approximately 1000 g of Oxide Comp 2, 100 M

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~11 with hydrated lime and sodium cyanide was added to a calculated level of 1.0 g/l.At 2, 4, 8, and 24 hours, the pH and free cyanide were determined. A sample of solution was removed and assayed for gold and silver. Sodium cyanide was added to return the level to 1.0 g/L and the pH was adjusted to 11 with hydrated lime if needed. After 48 hours, the solution was measured to determine pH, free cyanide, and gold and silver contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold and silver contents by fire assay techniques.

Conditions:

Grind	Leach Time	NaCN Concentration	% Solids	
80% minus 100 mesh	48 hours	1.00 g/L	40% Solids	

Summary of Results:

Parameter	Au	Ag
Extraction, % (1)	48.3	90.8
Assayed Head, g/mt	1.88	2.0
Calculated Head, g/mt	1.99	1.1
Final Tail Assay, g/mt	1.03	0.1

Hr.	Extraction, % (1) Au	Extraction, % (1) Ag	NaCN Consumed kg/mt
2	57.9	81.6	0.237
4	55.9	86.8	0.176
8	55.2	88.0	0.230
24	50.6	88.8	0.416
48	48.3	90.8	0.419

Cyanide Consumption 0.419 kg NaCN/metric ton ore
Lime Added 3.116 kg $Ca(OH)_2$/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g NaCN	$Ca(OH)_2$	Carbon		Residueal Reagents NaCN g/L	Dissolved Oxygen Initial mg/L	Dissolved Oxygen Adjust mg/L	pH Initial	pH Adjust
0	2500	1505	1.50	1.00						8.8	10.7
2	2500	1505	0.24	1.20			0.84			10.1	10.9
4	2500	1505					1.04			10.6	
8	2506	1511		0.90			1.00			10.4	11.0
24	2502	1507	0.18				0.88			10.7	
48	2499	1504					1.00			10.6	
Total			1.92	3.10		0.00					
				(2.3)	CaO Equivalent						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L			Volume Thief
Feed (analyzed)	1000		1.88		2.0				
Feed (computed)			1.99		1.1				
2 hour Preg		1505		0.76		0.59			25
4 hour Preg		1505		0.72		0.62			25
8 hour Preg		1511		0.70		0.61			25
24 hour Preg		1507		0.63		0.61			25
48 hour Preg		1504		0.59		0.62			25
48 hour Carbon									
48 hour Dry Residue	994.9		1.03		0.1				

(1) Based on calculated head assays.

Appendix H

Bottle Roll Leaching Test 7

Project: 23041-Getchell Gold
Date: 11-Jul-24

Purpose: To examine gold and silver leach amenability at Different Grind Sizes

Sample: Approximately 1000 g of Oxide Comp 2, 200 M

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~11 with hydrated lime and sodium cyanide was added to a calculated level of 1.0 g/l.At 2, 4, 8, and 24 hours, the pH and free cyanide were determined. A sample of solution was removed and assayed for gold and silver. Sodium cyanide was added to return the level to 1.0 g/L and the pH was adjusted to 11 with hydrated lime if needed. After 48 hours, the solution was measured to determine pH, free cyanide, and gold and silver contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold and silver contents by fire assay techniques.

Conditions:

Grind	Leach Time	NaCN Concentration	% Solids	
80% minus 200 mesh	48 hours	1.00 g/L	40% Solids	

Summary of Results:

Parameter	Au	Ag
Extraction, % (1)	49.1	51.2
Assayed Head, g/mt	1.88	2.0
Calculated Head, g/mt	1.96	2.0
Final Tail Assay, g/mt	1.00	1.0

Hr.	Extraction, % (1) Au	Ag	NaCN Consumed kg/mt
2	62.2	48.8	0.299
4	61.6	49.0	0.479
8	60.1	49.8	0.537
24	52.9	51.4	0.541
48	49.1	51.2	0.842

Cyanide Consumption 0.842 kg NaCN/metric ton ore
Lime Added 3.007 kg $Ca(OH)_2$/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g NaCN	$Ca(OH)_2$	Carbon		Residueal Reagents NaCN g/L	Dissolved Oxygen Initial mg/L	Adjust mg/L	pH Initial	Adjust
0	2500	1502	1.50	1.70						8.9	10.9
2	2500	1502	0.30				0.80			10.5	
4	2500	1502	0.18	0.70			0.88			10.4	10.8
8	2502	1504	0.06				0.96			10.7	
24	2498	1501		0.60			1.00			10.5	10.6
48	2498	1500					0.80			10.6	
Total			2.04	3.00		0.00					
				(2.3)	CaO Equivalent						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L			Volume Thief
Feed (analyzed)	1000		1.88		2.0				
Feed (computed)			1.96		2.0				
2 hour Preg		1502		0.81		0.66			25
4 hour Preg		1502		0.79		0.66			25
8 hour Preg		1504		0.76		0.66			25
24 hour Preg		1501		0.65		0.67			25
48 hour Preg		1500		0.59		0.65			25
48 hour Carbon									
48 hour Dry Residue	997.7		1.00		1.0				

(1) Based on calculated head assays.

Appendix H

Bottle Roll Leaching Test 9

Project: 23041-Getchell Gold
Date: 15-Jul-24

Purpose: To examine gold and silver leach amenability of the Average Grade Composite

Sample: Approximately 1000 g of Average Grade Comp, P80 200 M

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~11 with hydrated lime and sodium cyanide was added to a calculated level of 1.0 g/l.At 2, 4, 8, and 24 hours, the pH and free cyanide were determined. A sample of solution was removed and assayed for gold and silver. Sodium cyanide was added to return the level to 1.0 g/L and the pH was adjusted to 11 with hydrated lime if needed. After 48 hours, the solution was measured to determine pH, free cyanide, and gold and silver contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold and silver contents by fire assay techniques.

Conditions:

Grind	Leach Time	NaCN Concentration	% Solids	
80% minus 200 mesh	48 hours	1.00 g/L	40% Solids	

Summary of Results:

Parameter	Au	Ag
Extraction, % (1)	4.2	NA
Assayed Head, g/mt	1.49	1.0
Calculated Head, g/mt	1.47	0.3
Final Tail Assay, g/mt	1.41	0.3

Hr.	Extraction, % (1) Au	Extraction, % (1) Ag	NaCN Consumed kg/mt
2	9.5	NA	1.077
4	8.2	NA	1.319
8	5.5	NA	1.438
24	4.2	NA	1.558
48	4.2	NA	1.678

Cyanide Consumption 1.678 kg NaCN/metric ton ore
Lime Added 1.495 kg $Ca(OH)_2$/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g NaCN	$Ca(OH)_2$	Carbon		Residueal Reagents NaCN g/L	Dissolved Oxygen Initial mg/L	Dissolved Oxygen Adjust mg/L	pH Initial	pH Adjust
0	2500	1497	1.50	1.50						8.1	NA
2	2500	1497	1.08				0.28			11.6	
4	2500	1497	0.24				0.84			11.3	
8	2500	1497	0.12				0.92			1.1	
24	2500	1497	0.12				0.92			10.9	
48	2500	1497					0.92			10.8	
Total			3.06	1.50		0.00					
				(1.1)	CaO Equivalent						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L			Volume Thief
Feed (analyzed)	1000		1.49		1.0				
Feed (computed)			1.47		0.3				
2 hour Preg		1497		0.09		NA			25
4 hour Preg		1497		0.08		NA			25
8 hour Preg		1497		0.05		NA			25
24 hour Preg		1497		0.04		NA			25
48 hour Preg		1497		0.04		NA			25
48 hour Carbon									
48 hour Dry Residue	1003.3		1.41		0.3				

(1) Based on calculated head assays.

Appendix H

Bottle Roll Leaching Test 10

Project: 23041-Getchell Gold
Date: 16-Jul-24

Purpose: To examine gold and silver leach amenability of the Low Grade Composite

Sample: Approximately 1000 g of Low Grade Comp, P80 200 M

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~11 with hydrated lime and sodium cyanide was added to a calculated level of 1.0 g/l.At 2, 4, 8, and 24 hours, the pH and free cyanide were determined. A sample of solution was removed and assayed for gold. Sodium cyanide was added to return the level to 1.0 g/L and the pH was adjusted to 11 with hydrated lime if needed. After 48 hours, the solution was measured to determine pH, free cyanide, and gold contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold and silver contents by fire assay techniques.

Conditions:

Grind	Leach Time	NaCN Concentration	% Solids
80% minus 200 mesh	48 hours	1.00 g/L	40% Solids

Summary of Results:

Parameter	Au	Ag
Extraction, % (1)	20.3	NA
Assayed Head, g/mt	0.53	1.0
Calculated Head, g/mt	0.58	0.3
Final Tail Assay, g/mt	0.46	0.3

Hr.	Extraction, % (1) Au	Ag	NaCN Consumed kg/mt
2	22.5	NA	1.258
4	23.4	NA	1.319
8	22.3	NA	1.379
24	20.8	NA	1.558
48	20.3	NA	1.679

Cyanide Consumption 1.679 kg NaCN/metric ton ore
Lime Added 1.597 kg $Ca(OH)_2$/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g NaCN	$Ca(OH)_2$	Carbon		Residueal Reagents NaCN g/L	Dissolved Oxygen Initial mg/L	Adjust mg/L	pH Initial	Adjust
0	2500	1498	1.50	1.00						8.2	11.1
2	2500	1498	1.26				0.16			11.2	
4	2500	1498	0.06				0.96			11.0	
8	2500	1498	0.06				0.96			10.8	
24	2500	1498	0.18	0.60			0.88			10.5	10.9
48	2499	1497					0.92			11.0	
Total			3.06	1.60		0.00					
				(1.2)	CaO Equivalent						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L			Volume Thief
Feed (analyzed)	1000		0.53		1.0				
Feed (computed)			0.58		0.3				
2 hour Preg		1498		0.09		NA			25
4 hour Preg		1498		0.09		NA			25
8 hour Preg		1498		0.08		NA			25
24 hour Preg		1498		0.08		NA			25
48 hour Preg		1497		0.07		NA			25
48 hour Carbon									
48 hour Dry Residue	1001.9		0.46		0.3				

(1) Based on calculated head assays.

Appendix H
Bottle Roll Leaching Test 11

Project: 23041-Getchell Gold
Date: 16-Jul-24

Purpose: To examine gold and silver leach amenability of the High Grade Composite

Sample: Approximately 1000 g of High Grade Comp, P80 200 M

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~11 with hydrated lime and sodium cyanide was added to a calculated level of 1.0 g/l.At 2, 4, 8, and 24 hours, the pH and free cyanide were determined. A sample of solution was removed and assayed for gold. Sodium cyanide was added to return the level to 1.0 g/L and the pH was adjusted to 11 with hydrated lime if needed. After 48 hours, the solution was measured to determine pH, free cyanide, and gold contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold and silver contents by fire assay techniques.

Conditions:

Grind	Leach Time	NaCN Concentration	% Solids
80% minus 200 mesh	48 hours	1.00 g/L	40% Solids

Summary of Results:

Parameter	Au	Ag
Extraction, % (1)	29.4	NA
Assayed Head, g/mt	4.88	3.0
Calculated Head, g/mt	5.20	0.3
Final Tail Assay, g/mt	3.67	0.3

Hr.	Extraction, % (1) Au	Ag	NaCN Consumed kg/mt
2	27.1	NA	1.316
4	41.8	NA	1.498
8	39.9	NA	1.499
24	33.2	NA	1.618
48	29.4	NA	1.798

Cyanide Consumption 1.798 kg NaCN/metric ton ore
Lime Added 0.897 kg $Ca(OH)_2$/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g NaCN	$Ca(OH)_2$	Carbon		Residueal Reagents NaCN g/L	Dissolved Oxygen Initial mg/L	Adjust mg/L	pH Initial	Adjust
0	2500	1497	1.50	0.80						8.2	10.8
2	2501	1498	1.32				0.12			11.4	
4	2500	1497	0.18				0.88			11.1	
8	2500	1497					1.00			10.9	
24	2500	1497	0.12	0.10			0.92			10.7	10.7
48	2499	1496					0.88			10.7	
Total			3.12	0.90		0.00					
				(0.7)	CaO Equivalent						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L			Volume Thief
Feed (analyzed)	1000		4.88		3.0				
Feed (computed)			5.20		0.3				
2 hour Preg		1498		0.94		NA			25
4 hour Preg		1497		1.44		NA			25
8 hour Preg		1497		1.35		NA			25
24 hour Preg		1497		1.09		NA			25
48 hour Preg		1496		0.94		NA			25
48 hour Carbon									
48 hour Dry Residue	1003.0		3.67		0.3				

(1) Based on calculated head assays.

Appendix H

Bottle Roll Leaching Test 12

Project: 23041-Getchell Gold
Date: 30-Jul-24

Purpose: To examine gold and silver extraction of whole ore using carbon in leach

Sample: Approximately 1000 g of Oxide Composite 1, P80 270 M

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~11 with hydrated lime and sodium cyanide was added to a calculated level of 2.0 g/l. Hydrogen peroxide was added as necessary to increase dissolved oxygen. Activated carbon was added a calculated level of 20 g/L. At 2, 4, and 8 hours, the pH and free cyanide were determined. A sample of solution was removed and assayed for gold and silver. Sodium cyanide was added to return the level to 2.0 g/L and the pH was adjusted to 11 with hydrated lime if needed. Hydrogen peroxide was added as necessary to increase dissolved oxygen. After 24 hours, the solution was measured to determine pH, free cyanide, and gold and silver contents. The slurry was screened, washed, re-pulped, filtered, and dried. After drying, representative samples of the residue and carbon were submitted for determination of gold and silver contents by fire assay techniques.

Conditions:

Grind	Leach Time	NaCN Concentration	% Solids	H2O2 (3%)
80% minus 270 mesh	24 hours	1.98 g/L	40% Solids	4.90 g added

Summary of Results:

Parameter	Au	Ag
Extraction, % (1)	62.1	86.5
Assayed Head, g/mt	1.38	2.0
Calculated Head, g/mt	1.56	0.7
Final Tail Assay, g/mt	0.59	0.1

Hr.	Extraction, % (1) Au	Ag	NaCN Consumed kg/mt
2			0.386
4			0.379
8			0.561
24	62.1	86.5	1.025

Cyanide Consumption	1.025	kg NaCN/metric ton ore
Lime Added	2.583	kg $Ca(OH)_2$/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g NaCN	$Ca(OH)_2$	Carbon	H2O2	Residueal Reagents NaCN g/L	Dissolved Oxygen Initial mg/L	Adjust mg/L	pH Initial	Adjust
0	2525	1518	3.00	1.60	25.0	4.90		0.23	8.34	8.3	10.7
2	2525	1518	0.42	0.70			1.72	3.14		10.3	10.7
4	2526	1519					2.00	3.18		10.7	
8	2526	1519	0.18	0.30			1.88	3.06		10.6	10.8
24	2524	1493					1.72	3.33		10.6	
Total			3.60	2.60	25.0	4.90					
				(2.0)	CaO Equivalent						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L			Volume Thief
Feed (analyzed)	1000		1.38		2.0				
Feed (computed)			1.56		0.7				
2 hour Preg		1518		0.04		0.02			25
4 hour Preg		1519		0.04		0.00			25
8 hour Preg		1519		0.04		0.00			25
24 hour Preg		1493		0.03		0.00			25
24 hour Carbon	23.96		39		27				
24 hour Dry Residue	1006.8		0.59		0.1				

(1) Based on calculated head assays.

Appendix H

Bottle Roll Leaching Test 13

Project: 23041-Getchell Gold
Date: 29-Jul-24

Purpose: To examine gold and silver extraction of whole ore using carbon in leach

Sample: Approximately 1000 g of Oxide Composite 2, P80 270 M

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~11 with hydrated lime and sodium cyanide was added to a calculated level of 2.0 g/l. Hydrogen peroxide was added as necessary to increase dissolved oxygen. Activated carbon was added a calculated level of 20 g/L. At 2, 4, and 8 hours, the pH and free cyanide were determined. A sample of solution was removed and assayed for gold and silver. Sodium cyanide was added to return the level to 2.0 g/L and the pH was adjusted to 11 with hydrated lime if needed. Hydrogen peroxide was added as necessary to increase dissolved oxygen. After 24 hours, the solution was measured to determine pH, free cyanide, and gold and silver contents. The slurry was screened, washed, re-pulped, filtered, and dried. After drying, representative samples of the residue and carbon were submitted for determination of gold and silver contents by fire assay techniques.

Conditions:

Grind	Leach Time	NaCN Concentration	% Solids	H2O2 (3%)
80% minus 270 mesh	24 hours	1.90 g/L	39% Solids	9.70 g added

Summary of Results:

Parameter	Au	Ag
Extraction, % (1)	71.6	45.2
Assayed Head, g/mt	1.88	2.0
Calculated Head, g/mt	2.18	3.7
Final Tail Assay, g/mt	0.62	2.0

Hr.	Extraction, % (1) Au	Extraction, % (1) Ag	NaCN Consumed kg/mt
2			0.344
4			0.430
8			0.601
24	71.6	45.2	0.955

Cyanide Consumption 0.955 kg NaCN/metric ton ore
Lime Added 2.582 kg $Ca(OH)_2$/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g NaCN	$Ca(OH)_2$	Carbon	H2O2	Residueal Reagents NaCN g/L	Dissolved Oxygen Initial mg/L	Dissolved Oxygen Adjust mg/L	pH Initial	pH Adjust
0	2585	1578	3.00	1.70	25.0	8.20		0.27	12.07	8.7	10.7
2	2586	1579	0.48	0.40			1.68	3.43		10.4	10.7
4	2594	1587	0.12	0.10			1.92	3.78		10.6	10.7
8	2600	1593	0.18	0.40		1.50	1.88	2.40	6.36	10.5	10.8
24	2597	1566					1.80	3.33		10.6	
Total			3.78	2.60	25.0	9.70					
				(2.0)	CaO Equivalent						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L			Volume Thief
Feed (analyzed)	1000		1.88		2.0				
Feed (computed)			2.18		3.7				
2 hour Preg		1579		0.04		0.18			25
4 hour Preg		1587		0.04		0.04			25
8 hour Preg		1593		0.03		0.06			25
24 hour Preg		1566		0.03		0.00			25
24 hour Carbon	24.01		63.30		69				
24 hour Dry Residue	1006.9		0.62		2.0				

(1) Based on calculated head assays.
Gross starting weight was over by 60 g of water due to mis-calculation

APPENDIX I: OXIDE ASSAY-BY-SIZE DATA

Appendix I: Oxide Assay-by-Size

Engineer	JA	**Test ID**	Oxide Batch 1 SxS
Technician	MK	**Date**	6/17/2024
Project Name	Getchell Gold	**Sample**	Oxide Batch 1 SxS
Project No.	23041	**P80**	as received

Purpose: size by size assay at +100 M, 100x400 M, -400 M to evaluate gold deportment in as received material

Sample: Approximately 200 g of as received Oxide Composite

Procedure: A grab sample was screened at +100 M, 100 x 400 +, -400 M sizes, and the resulting fractions analyzed for Au by fire assay

Results:

Oxide Batch 1 As Received

Products	Weight gr.	Weight %	Chemical Analysis Au mg/kg	Chemical Analysis Ag mg/kg	Percent Distribution Au	Percent Distribution Ag	Cum. % Distribution Au	Cum. % Distribution Ag
Feed (analyzed)	200		0.08	BD				
Feed (calculated)	200.5	100.0	1.35	BD	100.0	100.0		
+100M	138.0	68.8	1.24	4	63.3	71.9	63.3	71.9
100x400 M	27.8	13.9	1.39	4	14.3	14.5	77.5	86.4
-400 M	34.7	17.3	1.75	3	22.5	13.6	100.0	100.0

APPENDIX J: LEACHING OF FLOTATION ROUGHER CONCENTRATE DATA

Appendix J Leaching of Flotation Rougher Concentrate
23041 Getchell Gold Bottle Roll Summary
Updated: 8/14/2024
author: J.Axen

CN Leach of Flotation Concentrate, with and without Re-Grind

Test #	Feed Material	Composite	Re-Grind	Assayed Float Head Grade Au (mg/kg)	*Assayed Flot. Conc Head to BR Au (mg/kg)	Calc. BR Head Grade Au (mg/kg)	% Recovery BR (Au)	BR Residue Grade Au (mg/kg)	NaCN Consumption (kg/mt)	Lime Consumption (kg/mt)
GBR-2	GTFT-5 conc	Average Grade	No	1.49	6.07	6.19	20.2	4.92	2.435	2.092
GBR-3	GTFT-5 conc	Average Grade	Yes	1.49	6.07	6.22	10.8	5.54	7.449	2.627

CN Leach of Flotation Tail

Test #	Feed Material	Composite	Re-Grind	Assayed Float Head Grade Au (mg/kg)	*Assayed Flot. Conc Head to BR Au (mg/kg)	Calc. BR Head Grade Au (mg/kg)	% Recovery BR (Au)	BR Residue Grade Au (mg/kg)	NaCN Consumption (kg/mt)	Lime Consumption (kg/mt)
GBR-8	GTFT-5 Tail	Average Grade	No	1.49	0.18	0.18	50.5	0.09	0.605	3.596

Whole Ore CN Leach of Oxide Composite 2 at Two Grind Sizes

Test #	Feed Material	Grind Size Mesh	Assayed Au Head Grade (mg/kg)	Calc. BR Head Grade Au (mg/kg)	% Recovery BR (Au)	BR Residue Grade Au (mg/kg)	NaCN Consumption (kg/mt)	Lime Consumption (kg/mt)
GBR-6	Oxide Comp 2	100	1.88	2.01	47.9	1.03	0.419	3.116
GBR-7	Oxide Comp 2	200	1.88	1.98	48.7	1.00	0.842	3.007

Whole Ore CN Leach of Oxide Composite 1 at Two Grind Sizes

Test #	Feed Material	Grind Size Mesh	Assayed Au Head Grade (mg/kg)	Calc. BR Head Grade Au (mg/kg)	% Recovery BR (Au)	BR Residue Grade Au (mg/kg)	NaCN Consumption (kg/mt)	Lime Consumption (kg/mt)
GBR-4	Oxide Comp 1	100	1.38	1.35	40.9	0.79	0.362	2.903
GBR-5	Oxide Comp 1	200	1.38	1.32	37.4	0.82	0.422	2.699

Whole Ore CN Leach of Low, Average and High Composites

Test #	Feed Material	Grind Size Mesh	Assayed Au Head Grade (mg/kg)	Calc. BR Head Grade Au (mg/kg)	% Recovery BR (Au)	BR Residue Grade Au (mg/kg)	NaCN Consumption (kg/mt)	Lime Consumption (kg/mt)
GBR-10	Low Grade	200	0.53	0.58	20.3	1.41	1.679	1.597
GBR-9	Average Grade	200	1.49	1.47	4.20	0.46	1.678	1.495
GBR-11	High Grade	200	4.88	5.22	29.2	3.67	1.798	0.897

Whole Ore CIL of Oxide Composites, P80 270 M, 2 g/L NaCN

Test #	Feed Material	Grind Size Mesh	Assayed Au Head Grade (mg/kg)	Calc. BR Head Grade Au (mg/kg)	% Recovery BR (Au)	BR Residue Grade Au (mg/kg)	NaCN Consumption (kg/mt)	Lime Consumption (kg/mt)
GBR-12	Oxide Comp 1	270	1.38	1.56	62.0	0.59	1.025	2.583
GBR-13	Oxide Comp 2	270	1.88	2.18	71.5	0.62	0.955	2.582

CIL of Average Grade Bulk FT Concentrate (GTFT-12), P80 < 270 M (4 hour re-grind prior to CIL), 5 g/L NaCN

Test #	Feed Material	Leach Time, Hr	Assayed Au Head Grade (mg/kg)	Calc. BR Head Grade Au (mg/kg)	% Recovery BR (Au)	BR Residue Grade Au (mg/kg)	NaCN Consumption (kg/mt)	Lime Consumption (kg/mt)
GBR-14	GTFT-12 Conc	24	6.23	6.92	53.8	3.17	13.593	1.087
GBR-15	GTFT-12 Conc	36	6.23	6.97	56.1	3.02	16.286	0.598

CIL of Low, High Grade Bulk FT Concentrate (GTFT-13,14), P80 270 M, 5 g/L NaCN

Test #	Composite	Feed Material	Leach Time, Hr	Assayed Au Head Grade (mg/kg)	Calc. BR Head Grade Au (mg/kg)	% Recovery BR (Au)	BR Residue Grade Au (mg/kg)	NaCN Consumption (kg/mt)	Lime Consumption (kg/mt)
GBR-16	Low Grade	GTFT-13 Conc	24	1.62	1.65	42.4	0.94	2.409	6.092
GBR-17	High Grade	GTFT-14 Conc	24	17.2	17.55	54.6	7.96	2.717	4.547

Appendix J
Bottle Roll Leaching Test 2

Project: 23041-Getchell Gold
Date: 27-Jun-24

Purpose: To examine gold and silver leach amenability of a flotation concentrate without re-grind

Sample: Approximately 1000 g of Average Grade Flotation Concentrate (GTFT-5), no regrind, P80 270M

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~11 with hydrated lime and sodium cyanide was added to a calculated level of 2.0 g/l. Hydrogen peroxide was added as necessary to increase dissolved oxygen. At 2, 4, 8, and 24 hours, the pH and free cyanide were determined. A sample of solution was removed and assayed for gold and silver. Sodium cyanide was added to return the level to 2.0 g/L and the pH was adjusted to 11 with hydrated lime if needed. Hydrogen peroxide was added as necessary to increase dissolved oxygen. After 48 hours, the solution was measured to determine pH, free cyanide, and gold and silver contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold and silver contents by fire assay techniques.

Conditions:

Grind	Leach Time	NaCN Concentration	% Solids	H2O2 (3%)
80% minus 270 mesh	48 hours	1.94 g/L	38% Solids	4.70 g

Summary of Results:

Parameter	Au	Ag
Extraction, % (1)	20.3	13.7
Assayed Head, g/mt	6.07	4.0
Calculated Head, g/mt	6.17	7.0
Final Tail Assay, g/mt	4.92	6.0

Hr.	Extraction, % (1) Au	Extraction, % (1) Ag	NaCN Consumed kg/mt
2	22.7	7.5	0.359
4	23.0	8.8	0.733
8	23.0	9.2	0.982
24	22.9	12.7	1.682
48	20.3	13.7	2.435

Cyanide Consumption	2.435	kg NaCN/metric ton ore
Lime Added	2.092	kg $Ca(OH)_2$/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g NaCN	$Ca(OH)_2$	Carbon	H2O2 (3%)	Residueal Reagents NaCN g/L	Dissolved Oxygen Initial mg/L	Dissolved Oxygen Adjust mg/L	pH Initial	pH Adjust
0	2620	1616	3.14	2.10				3.90		7.7	10.9
2	2620	1616	0.44				1.72	4.50		10.7	
4	2620	1616	0.38				1.76	4.10		10.6	
8	2620	1616	0.25				1.84	4.40		10.5	
24	2620	1616	0.69			4.70	1.56	1.90	3.9	10.9	
48	2620	1615					1.52	5.80		10.5	
Total			4.90	2.10		4.70					
				(1.6)	CaO Equivalent						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L			Volume Thief
Feed (analyzed)	1048		6.07		4.0				
Feed (computed)			6.17		7.0				
2 hour Preg		1616		0.87		0.32			25
4 hour Preg		1616		0.87		0.37			25
8 hour Preg		1616		0.86		0.39			25
24 hour Preg		1616		0.84		0.53			25
48 hour Preg		1615		0.73		0.57			25
48 hour Carbon									
48 hour Dry Residue	1004.1		4.92		6.0				

(1) Based on calculated head assays.

Appendix J

Bottle Roll Leaching Test 3

Project: 23041-Getchell Gold
Date: 27-Jun-24

Purpose: To examine gold and silver leach amenability of a flotation concentrate with re-grind

Sample: Approximately 1000 g of Average Grade Flotation Concentrate (GTFT-5), 4 hour regrind

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~11 with hydrated lime and sodium cyanide was added to a calculated level of 2.0 g/l. Hydrogen peroxide was added as necessary to increase dissolved oxygen. At 2, 4, 8, and 24 hours, the pH and free cyanide were determined. A sample of solution was removed and assayed for gold and silver. Sodium cyanide was added to return the level to 2.0 g/L and the pH was adjusted to 11 with hydrated lime if needed. Hydrogen peroxide was added as necessary to increase dissolved oxygen. After 48 hours, the solution was measured to determine pH, free cyanide, and gold and silver contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold and silver contents by fire assay techniques.

Conditions:

Grind	Leach Time	NaCN Concentration	% Solids	H2O2 (3%)
80% < 270 mesh	48 hours	2.02 g/L	41% Solids	9.10 g

Summary of Results:

Parameter	Au	Ag
Extraction, % (1)	10.8	41.7
Assayed Head, g/mt	6.07	4.0
Calculated Head, g/mt	6.21	3.4
Final Tail Assay, g/mt	5.54	2.0

Hr.	Extraction, % (1) Au	Ag	NaCN Consumed kg/mt
2	4.9	0.0	1.532
4	17.2	0.0	4.000
8	37.5	41.0	5.045
24	16.7	44.2	6.087
48	10.8	41.7	7.449

Cyanide Consumption 7.449 kg NaCN/metric ton ore
Lime Added 2.627 kg $Ca(OH)_2$/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g NaCN	$Ca(OH)_2$	Carbon	H2O2 (3%)	Residueal Reagents NaCN g/L	Dissolved Oxygen Initial mg/L	Adjust mg/L	pH Initial	Adjust
0	2620	1554	3.14	2.40				3.70		9.5	11.1
2	2636	1570	1.63			1.00	0.96	1.20	4.7	11.6	
4	2646	1580	2.64			2.00	0.32	0.90	NA	11.7	
8	2653	1587	1.13			6.10	1.28	1.60	6.4	12.0	
24	2668	1602	1.13	0.40			1.28	4.70		10.4	10.7
48	2667	1601					1.08	4.40		10.3	
Total			9.67	2.80		9.10					
				(2.1)	CaO Equivalent						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L			Volume Thief
Feed (analyzed)	1048		6.07		4.0				
Feed (computed)			6.21		3.4				
2 hour Preg		1570		0.21		0.00			25
4 hour Preg		1580		0.72		0.00			25
8 hour Preg		1587		1.55		0.95			25
24 hour Preg		1602		0.65		1.00			25
48 hour Preg		1601		0.40		0.92			25
48 hour Carbon									
48 hour Dry Residue	1066.0		5.54		2.0				

(1) Based on calculated head assays.

Appendix J

Bottle Roll Leaching Test 8

Project: 23041-Getchell Gold
Date: 11-Jul-24

Purpose: To examine gold and silver leach amenability of a flotation tail

Sample: Approximately 1000 g of GTFT-5 Tail

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~11 with hydrated lime and sodium cyanide was added to a calculated level of 1.0 g/l.At 2, 4, 8, and 24 hours, the pH and free cyanide were determined. A sample of solution was removed and assayed for gold and silver. Sodium cyanide was added to return the level to 1.0 g/L and the pH was adjusted to 11 with hydrated lime if needed. After 48 hours, the solution was measured to determine pH, free cyanide, and gold and silver contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold and silver contents by fire assay techniques.

Conditions:

Grind	Leach Time	NaCN Concentration	% Solids	
80% minus 270 mesh	48 hours	1.00 g/L	40% Solids	

Summary of Results:

Parameter	Au	Ag
Extraction, % (1)	50.9	29.3
Assayed Head, g/mt	0.18	0.1
Calculated Head, g/mt	0.18	0.1
Final Tail Assay, g/mt	0.09	0.1

Hr.	Extraction, % (1) Au	Ag	NaCN Consumed kg/mt
2	63.9	24.4	0.176
4	56.6	33.2	0.241
8	56.7	28.5	0.241
24	51.9	32.1	0.480
48	50.9	29.3	0.605

Cyanide Consumption 0.605 kg NaCN/metric ton ore
Lime Added 3.596 kg $Ca(OH)_2$/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g NaCN	$Ca(OH)_2$	Carbon		Residueal Reagents NaCN g/L	Dissolved Oxygen Initial mg/L	Adjust mg/L	pH Initial	Adjust
0	2500	1499	1.50	1.70						7.7	10.8
2	2506	1505	0.18	0.50			0.88			10.4	10.9
4	2500	1499	0.06				0.96			10.8	
8	2500	1499					1.00			10.7	
24	2500	1499	0.24	1.40			0.84			10.3	11.2
48	2495	1494					0.92			10.7	
Total			1.98	3.60		0.00					
				(2.7)	CaO Equivalent						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L			Volume Thief
Feed (analyzed)	1000		0.18		0.1				
Feed (computed)			0.18		0.1				
2 hour Preg		1505		0.08		0.02			25
4 hour Preg		1499		0.07		0.03			25
8 hour Preg		1499		0.07		0.03			25
24 hour Preg		1499		0.06		0.03			25
48 hour Preg		1494		0.06		0.03			25
48 hour Carbon									
48 hour Dry Residue	1001.2		0.09		0.1				

(1) Based on calculated head assays.

Appendix J

Bottle Roll Leaching Test 14

Project: 23041-Getchell Gold
Date: 1-Aug-24

Purpose: To examine gold and silver extraction from reground flotation concentrate using carbon in leach at 24 hr

Sample: Approximately 1000 g of Average Grade Flotation Concentrate GTFT-12, P80 , 270 M (4 hr regrind)

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~11 with hydrated lime and sodium cyanide was added to a calculated level of 2.0 g/l. Hydrogen peroxide was added as necessary to increase dissolved oxygen. Activated carbon was added a calculated level of 20 g/L. At 2, 4, and 8 hours, the pH and free cyanide were determined. A sample of solution was removed and assayed for gold and silver. Sodium cyanide was added to return the level to 2.0 g/L and the pH was adjusted to 11 with hydrated lime if needed. Hydrogen peroxide was added as necessary to increase dissolved oxygen. After 24 hours, the solution was measured to determine pH, free cyanide, and gold and silver contents. The slurry was screened, washed, re-pulped, filtered, and dried. After drying, representative samples of the residue and carbon were submitted for determination of gold and silver contents by fire assay techniques.

Conditions:

Grind	Leach Time	NaCN Concentration	% Solids	H2O2 (3%)
80% minus 270 mesh	24 hours	4.94 g/L	40% Solids	62.90 g added

Summary of Results:

Parameter	Au	Ag
Extraction, % (1)	53.9	4.9
Assayed Head, g/mt	6.23	2.0
Calculated Head, g/mt	6.91	13.7
Final Tail Assay, g/mt	3.17	13.0

Extraction, % (1)			NaCN Consumed
Hr.	Au	Ag	kg/mt
2			7.287
4			8.260
8			9.973
24	53.9	4.9	13.593

Cyanide Consumption 13.593 kg NaCN/metric ton ore
Lime Added 1.087 kg $Ca(OH)_2$/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g NaCN	Ca(OH)$_2$	Carbon	H2O2	Residueal Reagents NaCN g/L	Dissolved Oxygen Initial mg/L	Dissolved Oxygen Adjust mg/L	pH Initial	pH Adjust
0	2532	1520	7.50	1.10	30.0	17.60		0.25	12.3	8.6	11.5
2	2580	1568	7.38			34.50	0.08	1.60	5.7	12.5	
4	2595	1583	1.32			7.30	4.12	1.40	11.9	11.6	
8	2603	1591	1.74			3.50	3.84	1.40	3.4	11.5	
24	2602	1561					2.68	2.80		11.4	
Total			17.94	1.10	30.0	62.90					
				(0.8)	CaO Equivalent						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L			Volume Thief
Feed (analyzed)	1000		6.23		2.0				
Feed (computed)			6.91		13.7				
2 hour Preg		1568		0.20					25
4 hour Preg		1583		0.45					25
8 hour Preg		1591		0.31					25
24 hour Preg		1561		0.38					25
24 hour Carbon	29.2		109		23				
24 hour Dry Residue	1012.0		3.17		13.0				

(1) Based on calculated head assays.

Appendix J

Bottle Roll Leaching Test 15

Project: 23041-Getchell Gold
Date: 1-Aug-24

Purpose: To examine gold and silver extraction from flotation concentrate using carbon in leach at 36 hr

Sample: Approximately 1000 g of Average Grade Flotation Concentrate GTFT-12, P80 < 270 M (4 hour re-grind)

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~11 with hydrated lime and sodium cyanide was added to a calculated level of 5.0 g/l. Hydrogen peroxide was added as necessary to increase dissolved oxygen. Activated carbon was added a calculated level of 20 g/L. At 2, 4, and 24 hours, the pH and free cyanide were determined. A sample of solution was removed and assayed for gold and silver. Sodium cyanide was added to return the level to 5.0 g/L and the pH was adjusted to 11 with hydrated lime if needed. Hydrogen peroxide was added as necessary to increase dissolved oxygen. After 36 hours, the solution was measured to determine pH, free cyanide, and gold and silver contents. The slurry was screened, washed, re-pulped, filtered, and dried. After drying, representative samples of the residue and carbon were submitted for determination of gold and silver contents by fire assay techniques.

Conditions:

Grind	Leach Time	NaCN Concentration	% Solids	H2O2 (3%)
80% minus 270 mesh	36 hours	4.91 g/L	40% Solids	60.80 g added

Summary of Results:

Parameter	Au	Ag
Extraction, % (1)	56.2	11.8
Assayed Head, g/mt	6.23	2.0
Calculated Head, g/mt	6.96	5.7
Final Tail Assay, g/mt	3.02	5.0

Hr.	Extraction, % (1) Au	Extraction, % (1) Ag	NaCN Consumed kg/mt
2			7.350
4			8.545
24			12.041
36	56.2	11.8	16.286

Cyanide Consumption 16.286 kg NaCN/metric ton ore
Lime Added 0.598 kg $Ca(OH)_2$/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g NaCN	$Ca(OH)_2$	Carbon	H2O2	Residueal Reagents NaCN g/L	Dissolved Oxygen Initial mg/L	Dissolved Oxygen Adjust mg/L	pH Initial	pH Adjust
0	2531	1527	7.50	0.60	30.0	20.80		2.48	10.7	8.8	11.3
2	2570	1567	7.38			25.90	0.08	1.60	4.2	12.2	
4	2596	1593	1.56			4.40	3.96	1.10	4.2	11.5	
8	2610	1606	1.56			4.80	3.96	1.00	4.7	11.5	
24	2630	1627	3.48			4.90	2.68	1.60	3.7	11.4	
36	2630	1626					3.16	2.80		11.3	
Total			21.48	0.60	30.0	60.80					
				(0.5)	CaO Equivalent						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L			Volume Thief
Feed (analyzed)	1000		6.23		2				
Feed (computed)			6.96		6				
2 hour Preg		1567		0.25					25
4 hour Preg		1593		0.45					25
8 hour Preg		1606		0.34					25
24 hour Preg		1627		0.39					25
36 hour Preg		1626		0.43					25
36 hour Carbon	29.3		109.60		23				
36 hour Dry Residue	1003.3		3.02		5.0				

(1) Based on calculated head assays.

Appendix J

Bottle Roll Leaching Test 16

Project: 23041-Getchell Gold
Date: 5-Aug-24

Purpose: To examine gold and silver extraction from flotation concentrate using carbon in leach at 24 hr

Sample: Approximately 770 g of Low Grade Flotation Concentrate GTFT-13, P80 270 M

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~11 with hydrated lime and sodium cyanide was added to a calculated level of 5.0 g/l. Hydrogen peroxide was added as necessary to increase dissolved oxygen. Activated carbon was added a calculated level of 20 g/L. At 2, 4, and 8 hours, the pH and free cyanide were determined. A sample of solution was removed and assayed for gold and silver. Sodium cyanide was added to return the level to 5.0 g/L and the pH was adjusted to 11 with hydrated lime if needed. Hydrogen peroxide was added as necessary to increase dissolved oxygen. After 24 hours, the solution was measured to determine pH, free cyanide, and gold and silver contents. The slurry was screened, washed, re-pulped, filtered, and dried. After drying, representative samples of the residue and carbon were submitted for determination of gold and silver contents by fire assay techniques.

Conditions:

Grind	Leach Time	NaCN Concentration	% Solids	H2O2 (3%)
80% minus 270 mesh	24 hours	4.89 g/L	40% Solids	11.90 g added

Summary of Results:

Parameter	Au	Ag
Extraction, % (1)	42.4	90.4
Assayed Head, g/mt	1.62	0.6
Calculated Head, g/mt	1.64	1.0
Final Tail Assay, g/mt	0.94	0.1

Hr.	Extraction, % (1) Au	Ag	NaCN Consumed kg/mt
2			0.237
4			0.729
8			1.271
24	42.4	90.4	2.409

Cyanide Consumption	2.409	kg NaCN/metric ton ore
Lime Added	6.092	kg $Ca(OH)_2$/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g NaCN	$Ca(OH)_2$	Carbon	H2O2	Residueal Reagents NaCN g/L	Dissolved Oxygen Initial mg/L	Adjust mg/L	pH Initial	Adjust
0	1952	1181	5.77	4.70	30.0	7.90		0.80	11	7.8	11.2
2	1955	1184	0.32			0.80	4.72	2.70	3.8	11.5	
4	1953	1181	0.37			0.90	4.68	2.30	3.9	11.5	
8	1952	1181	0.42			2.30	4.64	2.40	8.8	11.5	
24	1952	1152					4.36	3.00		11.7	
Total			6.88	4.70	30.0	11.90					
				(3.6)	CaO Equivalent						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L			Volume Thief
Feed (analyzed)	769		1.62		0.6				
Feed (computed)			1.64		1.0				
2 hour Preg		1184		0.10					25
4 hour Preg		1181		0.09					25
8 hour Preg		1181		0.09					25
24 hour Preg		1152		0.09					25
24 hour Carbon	28.91		14.87		25				
24 hour Dry Residue	771.5		0.94		0				

(1) Based on calculated head assays.

Appendix J

Bottle Roll Leaching Test 17

Project: 23041-Getchell Gold
Date: 5-Aug-24

Purpose: To examine gold and silver extraction from flotation concentrate using carbon in leach at 24 hr

Sample: Approximately 792 g of High Grade Flotation Concentrate GTFT-14, P80 270 M

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~11 with hydrated lime and sodium cyanide was added to a calculated level of 5.0 g/l. Hydrogen peroxide was added as necessary to increase dissolved oxygen. Activated carbon was added a calculated level of 20 g/L. At 2, 4, and 8 hours, the pH and free cyanide were determined. A sample of solution was removed and assayed for gold and silver. Sodium cyanide was added to return the level to 5.0 g/L and the pH was adjusted to 11 with hydrated lime if needed. Hydrogen peroxide was added as necessary to increase dissolved oxygen. After 24 hours, the solution was measured to determine pH, free cyanide, and gold and silver contents. The slurry was screened, washed, re-pulped, filtered, and dried. After drying, representative samples of the residue and carbon were submitted for determination of gold and silver contents by fire assay techniques.

Conditions:

Grind	Leach Time	NaCN Concentration	% Solids	H2O2 (3%)
80% minus 270 mesh	24 hours	4.88 g/L	39% Solids	9.70 g added

Summary of Results:

Parameter	Au	Ag
Extraction, % (1)	54.6	19.9
Assayed Head, g/mt	17.20	4.0
Calculated Head, g/mt	17.55	5.0
Final Tail Assay, g/mt	7.96	4.0

	Extraction, % (1)		NaCN Consumed
Hr.	Au	Ag	kg/mt
2			0.153
4			0.787
8			1.400
24	54.6	19.9	2.717

Cyanide Consumption	2.717	kg NaCN/metric ton ore
Lime Added	4.547	kg $Ca(OH)_2$/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time	Net Pulp Weight	Net Soln Volume	Reagents Added, g				Residueal Reagents	Dissolved Oxygen		pH	
							NaCN	Initial	Adjust		
hrs	g	ml	NaCN	$Ca(OH)_2$	Carbon	H2O2	g/L	mg/L	mg/L	Initial	Adjust
0	2010	1218	5.94	3.60	30.0	6.20		0.50	3.1	8.0	11.3
2	2014	1223	0.29			0.80	4.76	2.90	4.3	11.4	
4	2011	1219	0.48			1.20	4.60	1.50	3.8	11.4	
8	2010	1218	0.48			1.50	4.60	2.80	1.5	11.3	
24	2009	1188					4.24	2.90		11.3	
Total			7.19	3.60	30.0	9.70					
				(2.7)	CaO Equivalent						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L			Volume Thief
Feed (analyzed)	792		17.20		4.0				
Feed (computed)			17.55		5.0				
2 hour Preg		1223		0.11					25
4 hour Preg		1219		0.09					25
8 hour Preg		1218		0.09					25
24 hour Preg		1188		0.10					25
24 hour Carbon	29.07		256.94		27				
24 hour Dry Residue	791.7		7.96		4.0				

(1) Based on calculated head assays.

APPENDIX K: ROAST PLUS LEACH TEST DATA

Appendix K

Bottle Roll Leaching Test 18

Project: 23041-Getchell Gold
Date: 9-Sep-24

Purpose: To examine gold and silver leach amenability of whole ore after 350 C, 4 hr, oxidizing roast

Sample: Approximately 500 g of Average Grade Composite, P80 270 M

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~11 with hydrated lime and sodium cyanide was added to a calculated level of 2.0 g/l. Hydrogen peroxide was added as necessary to increase dissolved oxygen. At 2, 4, 8, and 24 hours, the pH and free cyanide were determined. A sample of solution was removed and assayed for gold contents. Sodium cyanide was added to return the level to 2.0 g/L and the pH was adjusted to 11 with hydrated lime if needed. Hydrogen peroxide was added as necessary to increase dissolved oxygen. After 48 hours, the solution was measured to determine pH, free cyanide, and gold contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold and silver contents by fire assay techniques.

Conditions:

Grind	Leach Time	NaCN Concentration	% Solids	H2O2 (3%)
80% minus 270 mesh	48 hours	2.01 g/L	40% Solids	4.00 g

Summary of Results:

Parameter	Au	Ag
Extraction, % (1)	4.7	
Assayed Head, g/mt	1.49	1.00
Calculated Head, g/mt	1.50	NA
Final Tail Assay, g/mt	1.43	1.0

	Extraction, % (1)		NaCN Consumed
Hr.	Au	Ag	kg/mt
2	10.1		0.134
4	8.6		0.194
8	7.6		0.313
24	5.7		0.432
48	4.7		0.676

Cyanide Consumption 0.676 kg NaCN/metric ton ore
Lime Added 17.856 kg $Ca(OH)_2$/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time	Net Pulp Weight	Net Soln Volume	Reagents Added, g				Residueal Reagents	Dissolved Oxygen		pH	
hrs	g	ml	NaCN	$Ca(OH)_2$	Carbon	H2O2 (3%)	NaCN g/L	Initial mg/L	Adjust mg/L	Initial	Adjust
0	1250	746	1.50	9.00		4.00		1.40	15.7	6.6	11.4
2	1250	746	0.06				1.92	4.90		11.8	
4	1250	746	0.03				1.96	5.90		11.7	
8	1250	746	0.06				1.92	5.80		11.6	
24	1250	746	0.06				1.92	5.30		12.0	
48	1248	744					1.84	4.70		11.7	
Total			1.71	9.00		4.00					
				(6.8)	CaO Equivalent						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L			Volume Thief
Feed (analyzed)	500		1.49		1.0				
Feed (computed)			1.50		NA				
2 hour Preg		746		0.10		NA			25
4 hour Preg		746		0.08		NA			25
8 hour Preg		746		0.07		NA			25
24 hour Preg		746		0.05		NA			25
48 hour Preg		744		0.04		NA			25
48 hour Carbon									
48 hour Dry Residue	504.0		1.43		1.0				

(1) Based on calculated head assays.

Appendix K

Bottle Roll Leaching Test 19

Project: 23041-Getchell Gold
Date: 9-Sep-24

Purpose: To examine gold and silver extraction of whole ore, roasted at 350 C for 4 hours, using carbon in leach

Sample: Approximately 500 g of Average Grade Composite, P80 270 M

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~11 with hydrated lime and sodium cyanide was added to a calculated level of 2.0 g/l. Hydrogen peroxide was added as necessary to increase dissolved oxygen. Activated carbon was added a calculated level of 20 g/L. At 2, 4, 8, and 24 hours, the pH and free cyanide were determined. A sample of solution was removed and assayed for gold contents. Sodium cyanide was added to return the level to 2.0 g/L and the pH was adjusted to 11 with hydrated lime if needed. Hydrogen peroxide was added as necessary to increase dissolved oxygen. After 48 hours, the solution was measured to determine pH, free cyanide, and gold contents. The slurry was screened, washed, re-pulped, filtered, and dried. After drying, representative samples of the residue and carbon were submitted for determination of gold and silver contents by fire assay techniques.

Conditions:

Grind	Leach Time	NaCN Concentration	% Solids	H2O2 (3%)
80% minus 270 mesh	48 hours	1.99 g/L	40% Solids	0.70 g

Summary of Results:

Parameter	Au	Ag
Extraction, % (1)	8.6	
Assayed Head, g/mt	1.49	1.0
Calculated Head, g/mt	0.97	
Final Tail Assay, g/mt	0.89	0.6

Hr.	Extraction, % (1) Au	Extraction, % (1) Ag	NaCN Consumed kg/mt
2			0.111
4			0.111
8			0.292
24			0.536
48	8.6		0.833

Cyanide Consumption 0.833 kg NaCN/metric ton ore
Lime Added 15.476 kg $Ca(OH)_2$/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g NaCN	$Ca(OH)_2$	Carbon	H2O2 (3%)	Residueal Reagents NaCN g/L	Dissolved Oxygen Initial mg/L	Dissolved Oxygen Adjust mg/L	pH Initial	pH Adjust
0	1250	752	1.50	7.70	15.0	0.70		2.90	6.4	6.7	11.4
2	1250	752	0.06				1.92	6.50		11.6	
4	1250	752					2.00	4.80		11.6	
8	1250	752	0.09				1.88	4.40		11.5	
24	1249	752	0.12				1.84	4.70		11.8	
48	1249	737					1.84	3.10		11.6	
Total			1.77	7.70		0.70					
				(5.8)	CaO Equivalent						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L			Volume Thief
Feed (analyzed)	500		1.49		1.0				
Feed (computed)			0.97		2.1				
2 hour Preg		752		0.07		NA			25
4 hour Preg		752		0.06		NA			25
8 hour Preg		752		0.05		NA			25
24 hour Preg		752		0.04		NA			25
48 hour Preg		737		0.05		NA			25
48 hour Carbon	14.77		0.10		51				
48 hour Dry Residue	497.5		0.89		0.6				

(1) Based on calculated head assays.

Appendix K

Bottle Roll Leaching Test 20

Project: 23041-Getchell Gold
Date: 11-Sep-24

Purpose: To examine gold and silver leach amenability of whole ore after 650 C, 4 hr, oxidizing roast

Sample: Approximately 500 g of Average Grade Composite, P80 270 M

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~11 with hydrated lime and sodium cyanide was added to a calculated level of 1.0 g/l.At 2, 4, 8, and 24 hours, the pH and free cyanide were determined. A At 2, 4, 8, and 24 hours, the pH and free cyanide were determined. A sample of solution was removed and assayed for gold contents. was adjusted to 11 with hydrated lime if needed. After 48 hours, the solution was measured to determine pH, free cyanide, and gold and silver contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold and silver contents by fire assay techniques.

Conditions:

Grind	Leach Time	NaCN Concentration	% Solids	H2O2 (3%)
80% minus 270 mesh	48 hours	2.04 g/L	41% Solids	0.00 g

Summary of Results:

Parameter	Au	Ag
Extraction, % (1)	86.0	
Assayed Head, g/mt	1.49	1.0
Calculated Head, g/mt	1.58	
Final Tail Assay, g/mt	0.22	0.2

	Extraction, % (1)		NaCN Consumed
Hr.	Au	Ag	kg/mt
2	81.1		0.116
4	86.0		0.233
8	87.5		0.295
24	86.9		0.468
48	86.0		0.597

Cyanide Consumption 0.597 kg NaCN/metric ton ore
Lime Added 52.475 kg $Ca(OH)_2$/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time	Net Pulp Weight	Net Soln Volume	Reagents Added, g				Residueal Reagents	Dissolved Oxygen		pH	
hrs	g	ml	NaCN	$Ca(OH)_2$	Carbon	H2O2 (3%)	NaCN g/L	Initial mg/L	Adjust mg/L	Initial	Adjust
0	1218	715	1.46	26.40				5.10		9.0	10.4
2	1218	715	0.03				1.96	4.80		12.6	
4	1218	715	0.06				1.92	4.30		12.6	
8	1218	715	0.03				1.96	5.80		12.6	
24	1218	715	0.09				1.88	5.80		12.4	
48	1216	713					1.92	6.60		12.4	
Total			1.67	26.40		0.00					
				(20.0)	CaO Equivalent						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L			Volume Thief
Feed (analyzed)	487.1		1.49		1.0				
Feed (computed)			1.58		0.2				
2 hour Preg		715		0.90		NA			25
4 hour Preg		715		0.92		NA			25
8 hour Preg		715		0.91		NA			25
24 hour Preg		715		0.87		NA			25
48 hour Preg		713		0.83		NA			25
48 hour Carbon									
48 hour Dry Residue	503.1		0.22		0.2				

(1) Based on calculated head assays.
added 1.6 g NaOH at start to bring pH from 9.86 to 10.36

Appendix K

Bottle Roll Leaching Test 21

Project: 23041-Getchell Gold
Date: 9-Sep-24

Purpose: To examine gold and silver extraction of whole ore, roasted at 650 C for 4 hours, using carbon in leach

Sample: Approximately 500 g of Average Grade Composite, P80 270 M

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~11 with hydrated lime and sodium cyanide was added to a calculated level of 2.0 g/l. Hydrogen peroxide was added as necessary to increase dissolved oxygen. Activated carbon was added a calculated level of 20 g/L. At 2, 4, 8, and 24 hours, the pH and free cyanide were determined. A sample of solution was removed and assayed for gold contents. Sodium cyanide was added to return the level to 2.0 g/L and the pH was adjusted to 11 with hydrated lime if needed. Hydrogen peroxide was added as necessary to increase dissolved oxygen. After 48 hours, the solution was measured to determine pH, free cyanide, and gold contents. The slurry was screened, washed, re-pulped, filtered, and dried. After drying, representative samples of the residue and carbon were submitted for determination of gold and silver contents by fire assay techniques.

Conditions:

Grind	Leach Time	NaCN Concentration	% Solids	H2O2 (3%)
80% minus 270 mesh	48 hours	1.97 g/L	39% Solids	0.00 g

Summary of Results:

Parameter	Au	Ag
Extraction, % (1)	89.6	
Assayed Head, g/mt	1.49	1.0
Calculated Head, g/mt	1.92	
Final Tail Assay, g/mt	0.20	4.0

Hr.	Extraction, % (1) Au	Extraction, % (1) Ag	NaCN Consumed kg/mt
2			0.699
4			0.686
8			0.811
24			1.248
48	89.6		1.722

Cyanide Consumption 1.722 kg NaCN/metric ton ore
Lime Added 4.822 kg $Ca(OH)_2$/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g NaCN	Reagents Added, g $Ca(OH)_2$	Reagents Added, g Carbon	Reagents Added, g H2O2 (3%)	Residueal Reagents NaCN g/L	Dissolved Oxygen Initial mg/L	Dissolved Oxygen Adjust mg/L	pH Initial	pH Adjust
0	1218	741	1.46	2.30	15.0			4.60		9.2	10.5
2	1218	741	0.35				1.52	4.90		10.1	
4	1218	741					2.00	4.00		10.6	
8	1218	741	0.06				1.92	4.50		10.8	
24	1218	741	0.20				1.72	3.90		10.7	
48	1218	726					1.72	4.60		10.7	
Total			2.07	2.30		0.00					
				(1.7)	CaO Equivalent						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L			Volume Thief
Feed (analyzed)	487.1		1.49		1.0				
Feed (computed)			1.92		4.4				
2 hour Preg		741		0.22		NA			25
4 hour Preg		741		0.20		NA			25
8 hour Preg		741		0.19		NA			25
24 hour Preg		741		0.19		NA			25
48 hour Preg		726		0.18		NA			25
48 hour Carbon	14.97		45		13				
48 hour Dry Residue	477.0		0.20		4.0				

(1) Based on calculated head assays.
added 3.6 g NaOH at start of test to bring pH up to 10.53

Appendix K

Bottle Roll Leaching Test 22

Project: 23041-Getchell Gold
Date: 9-Sep-24

Purpose: To examine gold and silver leach amenability of whole ore after 350 C, 4 hr, oxidizing roast

Sample: Approximately 500 g of High Grade Composite, P80 270 M

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~11 with hydrated lime and sodium cyanide was added to a calculated level of 1.0 g/l.At 2, 4, 8, and 24 hours, the pH and free cyanide were determined. A At 2, 4, 8, and 24 hours, the pH and free cyanide were determined. A sample of solution was removed and assayed for gold contents. was adjusted to 11 with hydrated lime if needed. After 48 hours, the solution was measured to determine pH, free cyanide, and gold and silver contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold and silver contents by fire assay techniques.

Conditions:

Grind	Leach Time	NaCN Concentration	% Solids	H2O2 (3%)
80% minus 270 mesh	48 hours	2.02 g/L	41% Solids	0.00 g

Summary of Results:

Parameter	Au	Ag
Extraction, % (1)	20.0	
Assayed Head, g/mt	4.88	3.0
Calculated Head, g/mt	4.96	
Final Tail Assay, g/mt	3.97	6.0

Hr.	Extraction, % (1) Au	Extraction, % (1) Ag	NaCN Consumed kg/mt
2	31.4		0.083
4	29.6		0.201
8	27.9		0.203
24	22.4		0.438
48	20.0		0.617

Cyanide Consumption 0.617 kg NaCN/metric ton ore
Lime Added 18.363 kg $Ca(OH)_2$/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g NaCN	Reagents Added, g $Ca(OH)_2$	Reagents Added, g Carbon	Reagents Added, g H2O2 (3%)	Residueal Reagents NaCN g/L	Dissolved Oxygen Initial mg/L	Dissolved Oxygen Adjust mg/L	pH Initial	pH Adjust
0	1250	744	1.50	9.30				3.50		6.6	11.3
2	1250	744	0.03				1.96	5.30		11.8	
4	1250	744	0.06				1.92	4.70		11.8	
8	1250	744					2.00	5.20		11.7	
24	1250	744	0.12				1.84	4.60		12.0	
48	1250	743					1.88	3.80		12.0	
Total			1.71	9.30		0.00					
				(7.0)	CaO Equivalent						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L			Volume Thief
Feed (analyzed)	500		4.88		3.0				
Feed (computed)			4.96		6.0				
2 hour Preg		744		1.06		NA			25
4 hour Preg		744		0.97		NA			25
8 hour Preg		744		0.87		NA			25
24 hour Preg		744		0.66		NA			25
48 hour Preg		743		0.56		NA			25
48 hour Carbon									
48 hour Dry Residue	506.4		3.97		6.0				

(1) Based on calculated head assays.

Appendix K

Bottle Roll Leaching Test 23

Project: 23041-Getchell Gold
Date: 9-Sep-24

Purpose: To examine gold and silver extraction of whole ore, roasted at 350 C for 4 hours, using carbon in leach

Sample: Approximately 500 g of High Grade Composite, P80 270 M

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~11 with hydrated lime and sodium cyanide was added to a calculated level of 2.0 g/l. Hydrogen peroxide was added as necessary to increase dissolved oxygen. Activated carbon was added a calculated level of 20 g/L. At 2, 4, 8, and 24 hours, the pH and free cyanide were determined. A sample of solution was removed and assayed for gold contents. Sodium cyanide was added to return the level to 2.0 g/L and the pH was adjusted to 11 with hydrated lime if needed. Hydrogen peroxide was added as necessary to increase dissolved oxygen. After 48 hours, the solution was measured to determine pH, free cyanide, and gold contents. The slurry was screened, washed, re-pulped, filtered, and dried. After drying, representative samples of the residue and carbon were submitted for determination of gold and silver contents by fire assay techniques.

Conditions:

Grind	Leach Time	NaCN Concentration	% Solids	H2O2 (3%)
80% minus 270 mesh	48 hours	2.02 g/L	41% Solids	0.00 g

Summary of Results:

Parameter	Au	Ag
Extraction, % (1)	53.5	
Assayed Head, g/mt	4.88	3.0
Calculated Head, g/mt	5.34	
Final Tail Assay, g/mt	2.48	7.0

Extraction, % (1) Hr.	Au	Ag	NaCN Consumed kg/mt
2			0.092
4			0.151
8			0.210
24			0.560
48	53.5		0.862

Cyanide Consumption 0.862 kg NaCN/metric ton ore
Lime Added 21.628 kg $Ca(OH)_2$/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g NaCN	$Ca(OH)_2$	Carbon	H2O2 (3%)	Residueal Reagents NaCN g/L	Dissolved Oxygen Initial mg/L	Adjust mg/L	pH Initial	Adjust
0	1250	741	1.50	11.00	15.0			3.80		6.6	11.4
2	1250	741	0.03				1.96	4.70		11.8	
4	1250	741	0.03				1.96	4.10		11.8	
8	1250	742	0.03				1.96	4.60		11.7	
24	1250	742	0.18				1.76	5.20		11.9	
48	1247	724					1.84	4.10		11.8	
Total			1.77	11.00		0.00					
				(8.3)	CaO Equivalent						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L			Volume Thief
Feed (analyzed)	500		4.88		3.0				
Feed (computed)			5.34		7.8				
2 hour Preg		741		0.13		NA			25
4 hour Preg		741		0.11		NA			25
8 hour Preg		742		0.09		NA			25
24 hour Preg		742		0.07		NA			25
48 hour Preg		724		0.04		NA			25
48 hour Carbon	14.84		95		29				
48 hour Dry Residue	508.6		2.48		7.0				

(1) Based on calculated head assays.

Appendix K

Bottle Roll Leaching Test 24

Project: 23041-Getchell Gold
Date: 11-Sep-24

Purpose: To examine gold and silver leach amenability of whole ore after 650 C, 4 hr, oxidizing roast

Sample: Approximately 500 g of High Grade Composite, P80 270 M

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~11 with hydrated lime
and sodium cyanide was added to a calculated level of 1.0 g/l.At 2, 4, 8, and 24 hours, the pH and free cyanide were determined. A
At 2, 4, 8, and 24 hours, the pH and free cyanide were determined. A sample of solution was removed and assayed for gold contents.
was adjusted to 11 with hydrated lime if needed. After 48 hours, the solution was measured to determine pH, free cyanide, and gold
and silver contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted
contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for
determination of gold and silver contents by fire assay techniques.

Conditions:

Grind	Leach Time	NaCN Concentration	% Solids	H2O2 (3%)
80% minus 270 mesh	48 hours	1.97 g/L	39% Solids	0.00 g

Summary of Results:

Parameter	Au	Ag
Extraction, % (1)	92.2	
Assayed Head, g/mt	4.88	3.0
Calculated Head, g/mt	5.62	
Final Tail Assay, g/mt	0.44	3.0

Extraction, % (1) Hr.	Au	Ag	NaCN Consumed kg/mt
2	88.5		0.641
4	90.2		0.195
8	92.0		0.505
24	91.5		0.816
48	92.2		0.820

Cyanide Consumption 0.820 kg NaCN/metric ton ore
Lime Added 8.393 kg NaOH/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g NaCN	NaOH	Carbon	H2O2 (3%)	Residueal Reagents NaCN g/L	Dissolved Oxygen Initial mg/L	Adjust mg/L	pH Initial	Adjust
0	1217	740	1.46	2.30				4.00		9.0	10.6
2	1217	740	0.32	1.10			1.56	4.80		10.0	11.6
4	1217	740					2.28	4.50		11.0	
8	1217	740					2.08	4.90		10.8	
24	1217	740	0.09	0.60			1.88	5.00		10.3	11.8
48	1216	740					2.00	4.90		11.2	
Total			1.87	4.00		0.00					
				~~(3.0)~~	~~CaO Equivalent~~						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L			Volume Thief
Feed (analyzed)	486.6		4.88		3.0				
Feed (computed)			5.62		3.0				
2 hour Preg		740		3.20		NA			25
4 hour Preg		740		3.16		NA			25
8 hour Preg		740		3.12		NA			25
24 hour Preg		740		2.99		NA			25
48 hour Preg		740		2.92		NA			25
48 hour Carbon									
48 hour Dry Residue	476.6		0.44		3.0				

(1) Based on calculated head assays.

NaOH pellets used to bring pH to >10.6

Appendix K

Bottle Roll Leaching Test 25

Project: 23041-Getchell Gold
Date: 11-Sep-24

Purpose: To examine gold and silver leach amenability of whole ore after 650 C, 4 hr, oxidizing roast, using carbon in leach

Sample: Approximately 500 g of High Grade Composite, P80 270 M

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~11 with hydrated lime and sodium cyanide was added to a calculated level of 2.0 g/l. Hydrogen peroxide was added as necessary to increase dissolved oxygen. Activated carbon was added a calculated level of 20 g/L. At 2, 4, 8, and 24 hours, the pH and free cyanide were determined. A sample of solution was removed and assayed for gold contents. Sodium cyanide was added to return the level to 2.0 g/L and the pH was adjusted to 11 with hydrated lime if needed. Hydrogen peroxide was added as necessary to increase dissolved oxygen. After 48 hours, the solution was measured to determine pH, free cyanide, and gold contents. The slurry was screened, washed, re-pulped, filtered, and dried. After drying, representative samples of the residue and carbon were submitted for determination of gold and silver contents by fire assay techniques.

Conditions:

Grind	Leach Time	NaCN Concentration	% Solids	H2O2 (3%)
80% minus 270 mesh	48 hours	1.96 g/L	39% Solids	0.00 g

Summary of Results:

Parameter	Au	Ag
Extraction, % (1)	93.1	
Assayed Head, g/mt	4.88	3.0
Calculated Head, g/mt	6.38	
Final Tail Assay, g/mt	0.44	0.2

Hr.	Extraction, % (1) Au	Extraction, % (1) Ag	NaCN Consumed kg/mt
2			0.823
4			0.620
8			0.998
24			1.251
48	93.1		2.060

Cyanide Consumption 2.060 kg NaCN/metric ton ore
Lime Added 7.822 kg NaOH/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g NaCN	NaOH	Carbon	H2O2 (3%)	Residueal Reagents NaCN g/L	Dissolved Oxygen Initial mg/L	Dissolved Oxygen Adjust mg/L	pH Initial	pH Adjust
0	1217	744	1.46	2.30	15.0			3.80		9.0	10.6
2	1217	744	0.41	0.90			1.44	4.50		10.0	11.1
4	1217	744					2.12	4.30		10.5	
8	1217	744	0.09	0.50			1.88	3.90		10.4	11.5
24	1217	744	0.12				1.84	5.40		10.6	
48	1216	727					1.52	5.90		10.4	
Total			2.08	3.70		0.00					
				~~(2.8)~~	~~CaO Equivalent~~						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L			Volume Thief
Feed (analyzed)	486.6		4.88		3.0				
Feed (computed)			6.38		0.5				
2 hour Preg		744		0.23		NA			25
4 hour Preg		744		0.23		NA			25
8 hour Preg		744		0.23		NA			25
24 hour Preg		744		0.23		NA			25
48 hour Preg		727		0.23		NA			25
48 hour Carbon	15.05		174		11				
48 hour Dry Residue	473.0		0.44		0.2				

(1) Based on calculated head assays.

NaOH pellets used to bring pH to >10.6

Appendix K

Bottle Roll Leaching Test 26

Project: 23041-Getchell Gold
Date: 9-Sep-24

Purpose: To examine gold and silver leach amenability of whole ore after 350 C, 4 hr, oxidizing roast

Sample: Approximately 500 g of Oxide Composite 2, P80 270 M

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~11 with hydrated lime and sodium cyanide was added to a calculated level of 1.0 g/l.At 2, 4, 8, and 24 hours, the pH and free cyanide were determined. A At 2, 4, 8, and 24 hours, the pH and free cyanide were determined. A sample of solution was removed and assayed for gold contents. was adjusted to 11 with hydrated lime if needed. After 48 hours, the solution was measured to determine pH, free cyanide, and gold and silver contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for determination of gold and silver contents by fire assay techniques.

Conditions:

Grind	Leach Time	NaCN Concentration	% Solids	H2O2 (3%)
80% minus 270 mesh	48 hours	2.01 g/L	40% Solids	2.80 g

Summary of Results:

Parameter	Au	Ag
Extraction, % (1)	48.1	
Assayed Head, g/mt	1.88	2.0
Calculated Head, g/mt	1.91	
Final Tail Assay, g/mt	0.99	3.0

Hr.	Extraction, % (1) Au	Ag	NaCN Consumed kg/mt
2	57.3		0.247
4	56.9		0.247
8	55.4		0.426
24	50.6		0.547
48	48.1		0.728

Cyanide Consumption 0.728 kg NaCN/metric ton ore
Lime Added 9.163 kg $Ca(OH)_2$/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g NaCN	$Ca(OH)_2$	Carbon	H2O2 (3%)	Residueal Reagents NaCN g/L	Dissolved Oxygen Initial mg/L	Adjust mg/L	pH Initial	Adjust
0	1250	748	1.50	4.60		2.80		2.20	15.7	7.7	10.8
2	1250	748	0.12				1.84	4.50		10.8	
4	1250	748					2.00	4.00		10.8	
8	1250	748	0.09				1.88	4.40		10.8	
24	1250	748	0.06				1.92	4.90		10.9	
48	1249	747					1.88	4.80		10.7	
Total			1.77	4.60		2.80					
				(3.5)	CaO Equivalent						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L			Volume Thief
Feed (analyzed)	500		1.88		2.0				
Feed (computed)			1.91		3.0				
2 hour Preg		748		0.73		NA			25
4 hour Preg		748		0.71		NA			25
8 hour Preg		748		0.66		NA			25
24 hour Preg		748		0.58		NA			25
48 hour Preg		747		0.53		NA			25
48 hour Carbon									
48 hour Dry Residue	502.0		0.99		3.0				

(1) Based on calculated head assays.

Appendix K

Bottle Roll Leaching Test 27

Project: 23041-Getchell Gold
Date: 9-Sep-24

Purpose: To examine gold and silver extraction of whole ore, roasted at 350 C for 4 hours, using carbon in leach

Sample: Approximately 500 g of Oxide Composite 2, P80 270 M

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~11 with hydrated lime and sodium cyanide was added to a calculated level of 2.0 g/l. Hydrogen peroxide was added as necessary to increase dissolved oxygen. Activated carbon was added a calculated level of 20 g/L. At 2, 4, 8, and 24 hours, the pH and free cyanide were determined. A sample of solution was removed and assayed for gold contents. Sodium cyanide was added to return the level to 2.0 g/L and the pH was adjusted to 11 with hydrated lime if needed. Hydrogen peroxide was added as necessary to increase dissolved oxygen. After 48 hours, the solution was measured to determine pH, free cyanide, and gold contents. The slurry was screened, washed, re-pulped, filtered, and dried. After drying, representative samples of the residue and carbon were submitted for determination of gold and silver contents by fire assay techniques.

Conditions:

Grind	Leach Time	NaCN Concentration	% Solids	H2O2 (3%)
80% minus 270 mesh	48 hours	2.01 g/L	40% Solids	2.40 g

Summary of Results:

Parameter		Au	Ag
Extraction, % (1)		57.4	
Assayed Head, g/mt		1.88	2.0
Calculated Head, g/mt		1.08	
Final Tail Assay, g/mt		0.46	0.2
Cyanide Consumption	1.151	kg NaCN/metric ton ore	
Lime Added	9.057	kg $Ca(OH)_2$/metric ton ore	

Hr.	Extraction, % (1) Au	Ag	NaCN Consumed kg/mt
2			0.075
4			0.135
8			0.375
24			0.736
48	57.4		1.151

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g NaCN	$Ca(OH)_2$	Carbon	H2O2 (3%)	Residueal Reagents NaCN g/L	Dissolved Oxygen Initial mg/L	Adjust mg/L	pH Initial	Adjust
0	1238	741	1.49	4.50	15.0	2.40		1.90	13.3	7.7	10.8
2	1238	741	0.03				1.96	4.30		10.8	
4	1238	741	0.06				1.96	3.90		10.8	
8	1238	741	0.09				1.88	5.40		10.8	
24	1238	741	0.18				1.76	4.50		11.0	
48	1238	726					1.76	4.60		11.0	
Total			1.85	4.50		2.40					
				(3.4)	CaO Equivalent						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L			Volume Thief
Feed (analyzed)	495		1.88		2.0				
Feed (computed)			1.08		0.6				
2 hour Preg		741		0.07		NA			25
4 hour Preg		741		0.06		NA			25
8 hour Preg		741		0.06		NA			25
24 hour Preg		741		0.05		NA			25
48 hour Preg		726		0.04		NA			25
48 hour Carbon	14.76		18		15				
48 hour Dry Residue	496.9		0.46		0.2				

(1) Based on calculated head assays.

Appendix K

Bottle Roll Leaching Test 28

Project: 23041-Getchell Gold
Date: 11-Sep-24

Purpose: To examine gold and silver leach amenability of whole ore after 650 C, 4 hr, oxidizing roast

Sample: Approximately 500 g of Oxide Composite 2, P80 270 M

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~11 with hydrated lime
and sodium cyanide was added to a calculated level of 1.0 g/l.At 2, 4, 8, and 24 hours, the pH and free cyanide were determined. A
At 2, 4, 8, and 24 hours, the pH and free cyanide were determined. A sample of solution was removed and assayed for gold contents.
was adjusted to 11 with hydrated lime if needed. After 48 hours, the solution was measured to determine pH, free cyanide, and gold
and silver contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted
contents. The slurry was washed, re-pulped, filtered, and dried. After drying, a representative sample of the solids was submitted for
determination of gold and silver contents by fire assay techniques.

Conditions:

Grind	Leach Time	NaCN Concentration	% Solids	H2O2 (3%)
80% minus 270 mesh	48 hours	2.00 g/L	40% Solids	0.00 g

Summary of Results:

Parameter		Au	Ag
Extraction, % (1)		85.4	
Assayed Head, g/mt		1.88	2.0
Calculated Head, g/mt		2.20	
Final Tail Assay, g/mt		0.32	1.0
Cyanide Consumption	0.636	kg NaCN/metric ton ore	
Lime Added	0.000	kg $Ca(OH)_2$/metric ton ore	

Hr.	Extraction, % (1) Au	Ag	NaCN Consumed kg/mt
2	82.7		0.123
4	83.9		0.188
8	85.1		0.283
24	84.6		0.509
48	85.4		0.636

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g NaCN	$Ca(OH)_2$	Carbon	H2O2 (3%)	Residueal Reagents NaCN g/L	Dissolved Oxygen Initial mg/L	Adjust mg/L	pH Initial	Adjust
0	1209	724	1.45					4.40		12.3	
2	1209	724	0.06				1.92	5.00		12.4	
4	1208	724	0.03				1.96	3.90		12.3	
8	1215	731	0.06				1.92	4.10		12.2	
24	1220	736	0.12				1.84	4.10		11.7	
48	1220	735					1.92	5.30		11.5	
Total			1.72	0.00		0.00					
				(0.0)	CaO Equivalent						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L			Volume Thief
Feed (analyzed)	483.3		1.88		2.0				
Feed (computed)			2.20		1.0				
2 hour Preg		724		1.22		NA			25
4 hour Preg		724		1.19		NA			25
8 hour Preg		731		1.16		NA			25
24 hour Preg		736		1.10		NA			25
48 hour Preg		735		1.08		NA			25
48 hour Carbon									
48 hour Dry Residue	484.5		0.32		1.0				

(1) Based on calculated head assays.

Appendix K

Bottle Roll Leaching Test 29

Project: 23041-Getchell Gold
Date: 11-Sep-24

Purpose: To examine gold and silver leach amenability of whole ore after 650 C, 4 hr, oxidizing roast, using carbon in leach

Sample: Approximately 500 g of Oxide Composite 2, P80 270 M

Procedure: The material was transferred to a bottle and was adjusted to 40% solids. The pH of the slurry was adjusted to ~11 with hydrated lime and sodium cyanide was added to a calculated level of 2.0 g/l. Hydrogen peroxide was added as necessary to increase dissolved oxygen. Activated carbon was added a calculated level of 20 g/L. At 2, 4, 8, and 24 hours, the pH and free cyanide were determined. A sample of solution was removed and assayed for gold contents. Sodium cyanide was added to return the level to 2.0 g/L and the pH was adjusted to 11 with hydrated lime if needed. Hydrogen peroxide was added as necessary to increase dissolved oxygen. After 48 hours, the solution was measured to determine pH, free cyanide, and gold contents. The slurry was screened, washed, re-pulped, filtered, and dried. After drying, representative samples of the residue and carbon were submitted for determination of gold and silver contents by fire assay techniques.

Conditions:

Grind	Leach Time	NaCN Concentration	% Solids	H2O2 (3%)
80% minus 270 mesh	48 hours	2.00 g/L	40% Solids	0.00 g

Summary of Results:

Parameter	Au	Ag
Extraction, % (1)	82.5	
Assayed Head, g/mt	1.88	2.0
Calculated Head, g/mt	2.46	
Final Tail Assay, g/mt	0.43	0.2

Extraction, % (1) Hr.	Au	Ag	NaCN Consumed kg/mt
2			0.055
4			0.199
8			0.372
24			0.579
48	82.5		0.887

Cyanide Consumption 0.887 kg NaCN/metric ton ore
Lime Added 0.000 kg $Ca(OH)_2$/metric ton ore

Detailed Results:

A. Cyanidation Conditions

Time hrs	Net Pulp Weight g	Net Soln Volume ml	Reagents Added, g NaCN	$Ca(OH)_2$	Carbon	H2O2 (3%)	Residueal Reagents NaCN g/L	Dissolved Oxygen Initial mg/L	Adjust mg/L	pH Initial	Adjust
0	1208	726	1.45		15.0			4.60		12.4	
2	1208	726	0.03				1.96	4.00		12.4	
4	1218	736	0.09				1.88	3.70		12.4	
8	1222	740	0.09				1.88	3.60		12.3	
24	1233	750	0.12				1.84	4.50		11.7	
48	1232	735					1.84	5.30		11.6	
Total			1.78	0.00		0.00					
				(0.0)	CaO Equivalent						

B. Products and Analyses

Leach Product	Weight g	Volume ml	Au g/mt	Au mg/L	Ag g/mt	Ag mg/L			Volume Thief
Feed (analyzed)	483.3		1.88		2.0				
Feed (computed)			2.46		0.2				
2 hour Preg		726		0.13		NA			25
4 hour Preg		736		0.11		NA			25
8 hour Preg		740		0.10		NA			25
24 hour Preg		750		0.07		NA			25
48 hour Preg		735		0.05		NA			25
48 hour Carbon	14.95		62		1				
48 hour Dry Residue	482.1		0.43		0.2				

(1) Based on calculated head assays.

APPENDIX B – ECONOMIC MODEL AND DISCOUNTED CASH FLOW

Getchell Gold Corp.
Fondaway Canyon Project
Economic Model

			Period (yr)	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
				0	1	1	1	1	1	1	1	1	1	1	1
				2029	1/1/2030	1/1/2031	1/1/2032	1/1/2033	1/1/2034	1/1/2035	1/1/2036	1/1/2037	1/1/2038	1/1/2039	1/1/2040
Production		**Total/Average**	ore tonne/day		7,249	7,940	7,973	7,964	8,000	8,000	8,000	8,022	8,000	8,000	3,983
	Total Processed Material	**30,342,924**	ore tonne/yr		**2,645,789**	**2,898,109**	**2,910,311**	**2,907,001**	**2,919,939**	**2,919,913**	**2,920,032**	**2,927,986**	**2,920,034**	**2,919,977**	**1,453,833**
	Surface processed material	30,342,924			2,645,789	2,898,109	2,910,311	2,907,001	2,919,939	2,919,913	2,920,032	2,927,986	2,920,034	2,919,977	1,453,833
	Surface waste	143,391,501			18,029,819	17,726,129	17,636,567	17,959,385	17,520,172	17,526,992	17,406,493	9,001,660	5,952,390	2,992,445	1,639,449
	Ore tons per day	7,552			7,249	7,940	7,952	7,964	8,000	8,000	7,978	8,022	8,000	8,000	3,972
	Gold oz contained metal	1,465,962			134,231.57	143,690.45	161,055.89	134,513.67	149,001.76	141,914.92	136,631.96	137,262.43	125,372	139,172	63,115
	Au grade toz/ston	0.044			0.051	0.050	0.055	0.046	0.051	0.049	0.047	0.047	0.043	0.048	0.043
	Recovered silver ounces	-			-	-	-	-	-	-	-	-	-	-	-
	Recovered gold ounces	**1,231,408**			**112,755**	**120,700**	**135,287**	**112,991**	**125,161**	**119,209**	**114,771**	**115,300**	**105,312**	**116,904**	**53,017**
Revenue from Concentrate															
	Au revenue	**$2,770,667,321**			**$253,697,674**	**$271,574,948**	**$304,395,624**	**$254,230,839**	**$281,613,325**	**$268,219,200**	**$258,234,397**	**$259,425,985**	**$236,953,038**	**$263,035,025**	**$119,287,265**
	Less Royalites (3%)	$83,120,020			$7,610,930	$8,147,248	$9,131,869	$7,626,925	$8,448,400	$8,046,576	$7,747,032	$7,782,780	$7,108,591	$7,891,051	$3,578,618
Total															
	Total Revenue	**$2,687,547,301**			**$246,086,743**	**$263,427,700**	**$295,263,756**	**$246,603,914**	**$273,164,925**	**$260,172,624**	**$250,487,365**	**$251,643,206**	**$229,844,447**	**$255,143,974**	**$115,708,647**
Operating Costs			$/tonne												
	Surface Ore	$107,374,168	$ 3.54		$9,362,623	$10,255,507	$10,298,685	$10,286,973	$10,332,755	$10,332,664	$10,333,086	$10,361,231	$10,333,091	$10,332,889	$5,144,663
	Surface Waste	$507,417,911	$ 3.54		$63,801,919	$62,727,255	$62,410,322	$63,552,677	$61,998,438	$62,022,573	$61,596,164	$31,854,075	$21,063,656	$10,589,332	$5,801,500
	Power	$0													
	Processing	$402,043,744	$ 13.25		$35,056,701	$38,399,948	$38,561,619	$38,517,767	$38,689,189	$38,688,849	$38,690,430	$38,795,814	$38,690,448	$38,689,691	$19,263,287
	Transportation and Refining	$303,429,241	$ 10.00		$26,457,888	$28,981,093	$29,103,108	$29,070,013	$29,199,388	$29,199,131	$29,200,325	$29,279,860	$29,200,338	$29,199,767	$14,538,330
	G&A	$60,685,848	$ 2.00		$5,291,578	$5,796,219	$5,820,622	$5,814,003	$5,839,878	$5,839,826	$5,840,065	$5,855,972	$5,840,068	$5,839,953	$2,907,666
Total Operating Costs		**$1,380,950,911**		**$0**	**$139,970,709**	**$146,160,022**	**$146,194,355**	**$147,241,433**	**$146,059,648**	**$146,083,043**	**$145,660,070**	**$116,146,953**	**$105,127,602**	**$94,651,631**	**$47,655,445**
Before Tax Cash Flow		**$1,306,596,390**		**$0**	**$106,116,035**	**$117,267,677**	**$149,069,400**	**$99,362,481**	**$127,105,277**	**$114,089,581**	**$104,827,296**	**$135,496,253**	**$124,716,845**	**$160,492,343**	**$68,053,202**
Capital Costs															
	Process capital ($20,000/ton)	$131,739,400													
	Mine equipment capital														
	Preproduction and Facilities	$56,985,705													
	Capex summary	$188,725,105			$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
	Contingency (20%)	$37,745,021													
Total Capital Cost		**$226,470,126**			**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**
Net Cash Flow (Pre-Tax)		**0%**		**($226,470,126)**	**$106,116,035**	**$117,267,677**	**$149,069,400**	**$99,362,481**	**$127,105,277**	**$114,089,581**	**$104,827,296**	**$135,496,253**	**$124,716,845**	**$160,492,343**	**$68,053,202**
	Discounted Cash Flow	5%		($226,470,126)	$101,062,890	$106,365,240	$128,771,753	$81,745,759	$99,590,310	$85,135,402	$74,498,802	$91,709,198	$80,393,590	$98,528,376	$39,789,298
	Discounted Cash Flow	8%		($226,470,126)	$98,255,588	$100,538,132	$118,336,096	$73,034,390	$86,505,715	$71,895,789	$61,165,720	$73,204,410	$62,389,473	$74,339,008	$29,186,852
	Discounted Cash Flow	**10%**		**($226,470,126)**	**$96,469,122**	**$96,915,436**	**$111,998,047**	**$67,865,911**	**$78,922,376**	**$64,400,594**	**$53,792,978**	**$63,210,002**	**$52,892,117**	**$61,876,746**	**$23,852,232**
	Discounted Cash Flow	12%		($226,470,126)	$94,746,460	$93,485,074	$106,104,655	$63,146,653	$72,122,947	$57,801,333	$47,418,545	$54,724,664	$44,974,145	$51,674,239	$19,563,669
Cumulative Pre-Tax Cash Flow				**($226,470,126)**	**($120,354,091)**	**($3,086,414)**	**$145,982,986**	**$245,345,467**	**$372,450,744**	**$486,540,325**	**$591,367,621**	**$726,863,874**	**$851,580,719**	**$1,012,073,062**	**$1,080,126,264**
Taxes															
	Revenue	$ 2,687,547,301		$ -	$ 246,086,743	$ 263,427,700	$ 295,263,756	$ 246,603,914	$ 273,164,925	$ 260,172,624	$ 250,487,365	$ 251,643,206	$ 229,844,447	$ 255,143,974	$ 115,708,647
	Operating Costs	$ (1,380,950,911)		$ -	$ (139,970,709)	$ (146,160,022)	$ (146,194,355)	$ (147,241,433)	$ (146,059,648)	$ (146,083,043)	$ (145,660,070)	$ (116,146,953)	$ (105,127,602)	$ (94,651,631)	$ (47,655,445)
	Sustaining Capital	$ -		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
	Depreciation	$ (415,195,231)		$ -	$ (38,017,579)	$ (40,696,558)	$ (45,614,863)	$ (38,097,476)	$ (42,200,848)	$ (40,193,686)	$ (38,697,425)	$ (38,875,989)	$ (35,508,331)	$ (39,416,817)	$ (17,875,659)
	Depletion	$ (386,095,516)		$ -	$ (34,049,228)	$ (38,285,560)	$ (44,289,563)	$ (30,632,502)	$ (40,974,739)	$ (36,947,948)	$ (33,064,936)	$ (37,746,481)	$ (34,476,667)	$ (38,271,596)	$ (17,356,297)
	Amortization	$ -		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -		
	State Proceeds Tax	$ (34,227,433)		$ -	$ (2,260,878)	$ (2,651,185)	$ (4,301,995)	$ (1,735,289)	$ (2,995,501)	$ (2,539,952)	$ (1,899,233)	$ (3,759,069)	$ (3,327,893)	$ (5,353,777)	$ (3,402,660)
	Gold and Silver Excise Tax	$ (30,477,341)		$ -	$ (2,790,674)	$ (2,987,324)	$ (3,348,352)	$ (2,796,539)	$ (3,097,747)	$ (2,950,411)	$ (2,840,578)	$ (2,853,686)	$ (2,606,483)	$ (2,893,385)	$ (1,312,160)
	Loss Carry Forward (Corporate)	$ -													
	Interest Expense	$ -		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
	Tax Loss Carry Forward	$ -		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -			
Taxable Income		$ 440,600,869		$ -	$ 28,997,675	$ 32,647,050	$ 51,514,626	$ 26,100,674	$ 37,836,442	$ 31,457,584	$ 28,325,124	$ 52,261,028	$ 48,797,471	$ 74,556,767	$ 28,106,426
	Federal Tax (21%)	$ (92,526,183)		$ -	$ (6,089,512)	$ (6,855,880)	$ (10,818,072)	$ (5,481,142)	$ (7,945,653)	$ (6,606,093)	$ (5,948,276)	$ (10,974,816)	$ (10,247,469)	$ (15,656,921)	$ (5,902,350)
Net Income		$ 348,074,687		$ -	$ 22,908,164	$ 25,791,169	$ 40,696,555	$ 20,619,533	$ 29,890,789	$ 24,851,492	$ 22,376,848	$ 41,286,212	$ 38,550,002	$ 58,899,846	$ 22,204,077
	Depreciation	$ 415,195,231			$ 38,017,579	$ 40,696,558	$ 45,614,863	$ 38,097,476	$ 42,200,848	$ 40,193,686	$ 38,697,425	$ 38,875,989	$ 35,508,331	$ 39,416,817	$ 17,875,659
	Depletion	$ 386,095,516			$ 34,049,228	$ 38,285,560	$ 44,289,563	$ 30,632,502	$ 40,974,739	$ 36,947,948	$ 33,064,936	$ 37,746,481	$ 34,476,667	$ 38,271,596	$ 17,356,297
	Amortization	$ -			$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
	Capital Expenditures (Less Interest)	$ (226,470,126)		$ (226,470,126)	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Net Cash Flow (After-Tax)		**0%**		**($226,470,126)**	**$97,765,645**	**$107,760,612**	**$133,949,334**	**$92,146,050**	**$116,164,122**	**$104,943,537**	**$96,979,787**	**$120,762,368**	**$111,141,483**	**$139,481,645**	**$58,748,192**
	Discounted Cash Flow	5%		($226,470,126)	$93,110,138	$97,742,051	$115,710,471	$75,808,784	$91,017,629	$78,310,483	$68,921,724	$81,736,724	$71,642,791	$85,629,630	$34,348,851
	Discounted Cash Flow	8%		($226,470,126)	$90,523,745	$92,387,356	$106,333,300	$67,730,098	$79,059,350	$66,132,229	$56,586,774	$65,244,150	$55,598,412	$64,606,990	$25,196,093
	Discounted Cash Flow	**10%**		**($226,470,126)**	**$88,877,859**	**$89,058,357**	**$100,638,117**	**$62,936,992**	**$72,128,781**	**$59,237,891**	**$49,765,965**	**$56,336,536**	**$47,134,838**	**$53,776,212**	**$20,590,883**
	Discounted Cash Flow	12%		($226,470,126)	$87,290,754	$85,906,100	$95,342,490	$58,560,481	$65,914,643	$53,167,662	$43,868,730	$48,773,895	$40,078,733	$44,909,357	$16,888,701
Cumulative After-Tax Cash Flow				**($226,470,126)**	**($128,704,481)**	**($20,943,869)**	**$113,005,464**	**$205,151,515**	**$321,315,637**	**$426,259,173**	**$523,238,960**	**$644,001,328**	**$755,142,811**	**$894,624,456**	**$953,372,648**

Pre-Tax US$	
(Cumulative Cash Flow) NPV@0%	$1,080,126,264
NPV@5%	$761,120,492
NPV@8%	$622,381,047
NPV@10%	$545,725,436
NPV @ 12%	$479,292,258
IRR	51.2%
LOM Cash Flow	$1,080,126,264

After-Tax US$	
(Cumulative Cash Flow) NPV@0%	$953,372,648
NPV@5%	$667,509,151
NPV@8%	$542,928,370
NPV@10%	$474,012,304
NPV @ 12%	$414,231,420
IRR	46.7%
LOM Cash Flow	$953,372,648