Viva Gold Corp.

Preliminary Economic Assessment NI 43-101 Technical Report

Tonopah Gold Project, Nevada, USA

CA0004670.0343-001-TR-Rev0

August 20, 2025

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Notice to Readers

This National Instrument 43-101 Technical Report for Tonopah Gold Project (the Project) was prepared and executed by the Qualified Persons named herein as Authors. This report contains the expressions of professional opinions of the Authors based on (i) information available at the time of preparation, (ii) data supplied by Viva Gold Corp. (Viva), and (iii) the assumptions, conditions, and qualifications set forth in this report. The quality of information, conclusions, and estimates contained herein are consistent with the stated levels of accuracy as well as the circumstances and constraints under which the mandate was performed. This Report was prepared in accordance with a contract between WSP Canada Inc. and Viva which permits Viva to file this report as a Technical Report with Canadian securities regulators pursuant to *National Instrument 43-101 - Standards of Disclosure for Mineral Projects*. Except for the purposes legislated under Canadian securities law, any use of this Report by any third party is at that party’s sole risk.

Date and Signature Page

This Technical Report on the Tonopah Gold Project is submitted to Viva Gold Corp. and is effective as of August 20, 2025.

Qualified Person	Responsible for Parts
Signed by Brian Thomas Brian Thomas, P.Geo. (WSP Canada Inc.) Date Signed: August 20, 2025	Responsible for Items: 1.8, 1.11.1.1, 1.11.2.1, 1.11.2.6, 14.0, 25.1, 26.1, 26.6
Signed by Róisín Kerr Róisín Kerr, P.Geo. (WSP Canada Inc.) Date Signed: August 20, 2025	Responsible for Items: 1.1, 1.2, 1.3, 1.4, 1.5, 1.6, 1.11.1.1, 1.11.2.1, 1.11.2.6, 2.0, 3.0, 4.0, 5.0, 6.0, 7.0, 8.0, 9.0, 10.0, 11.0, 12.0, 23.0, 24.0, 25.1, 26.1, 26.6, 27.0
Signed by Jason Baker Jason Baker, P.Eng. (WSP Canada Inc.) Date Signed: August 20, 2025	Responsible for Items: 1.7.1, 1.11.1.2, 1.11.2.2, 1.11.2.6, 15.0, 16.0, 18.1.3, 25.2, 26.2, 26.6
Signed by William Richard McBride William Richard McBride, P.Eng. (WSP Canada Inc.) Date Signed: August 20, 2025	Responsible for Items: 1.9, 1.10, 1.11.1.5, 1.11.1.6, 1.11.2.5, 19, 21.1, 21.2.1, 21.2.2, 21.2.3, 21.2.4, 21.2.5, 21.2.6, 21.2.7, 21.2.9, 21.2.10, 21.2,11, 21.3.1, 21.3.4, 22, 25.5, 26.5
Signed by Randal Huffsmith Randal Huffsmith, P.E. (WSP USA Inc.) Date Signed: August 20, 2025	Responsible for Items: 1.11.1.4, 1.11.2.4, 1.11.2.6, 20.0, 25.4, 26.4, 26.6
Signed by Caleb D. Cook Caleb D. Cook, P.E. (Kappes, Cassiday & Associates) Date Signed: August 20, 2025	Responsible for Items: 1.7.2, 1.7.3, 1.7.4, 1.11.1.3, 1.11.2.3, 1.11.2.6, 13.0, 17.0, 18.1.1, 18.1.2, 18.2, 18.3, 18.4, 18.5, 18.6, 21.2.8, 21.3.2, 21.3.3, 25.3, 26.3, 26.6

CERTIFICATE OF QUALIFIED PERSON BRIAN THOMAS

I, Brian Thomas, state that:

(a) I am a Principal Geologist at:
WSP Canada Inc.
33 Mackenzie Street, Suite 100
Sudbury, Ontario, P3C 4Y1

(b) This certificate applies to the technical report titled Preliminary Economic Assessment NI 43-101 Technical Report on the Tonopah Gold Project, Nevada, USA; with an effective date of: August 20, 2025 (the "Technical Report").

(c) I am a "qualified person" for the purposes of National Instrument 43-101 ("NI 43-101"). My qualifications as a qualified person are as follows. I am a graduate of Laurentian University with a B.Sc. in Geology from 1994, I am a member in good standing of the Association of Professional Geoscientists of Ontario (#1366). My relevant experience after graduation, for the purpose of the Technical Report, includes over 30 years of experience in mine geology and mineral resource evaluation of mineral projects nationally and internationally in a variety of commodities including 8 years of direct working experience in gold mining operations located in northern Ontario, 9 years of experience in base metals operations in Sudbury, Ontario, and 14 years of consulting experience with a strong focus on gold and base metals related projects.

(d) I have not completed a personal inspection of the property described in the Technical Report.

(e) I am responsible for Item(s) 1.8, 1.11.1.1, 1.11.2.1, 1.11.2.6, 14.0, 25.1, 26.1, and 26.6 of the Technical Report.

(f) I am independent of the issuer as described in section 1.5 of NI 43-101.

(g) My prior involvement with the property that is the subject of the Technical Report consists of internal grade estimation, and supervision of grade estimation of the Tonopah Project for Viva Gold.

(h) I have read NI 43-101 and the parts of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101; and

(i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible, contain(s) all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Sudbury, Ontario this 20th of August 2025.

Thomas, Brian (gld_bthomas)
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Brian Thomas; P.Geo.

CERTIFICATE OF QUALIFIED PERSON RÓISÍN KERR

I, Róisín Kerr, state that:

(a) I am a Senior Geologist at:
WSP Canada Inc.
3300, 237 – 4th Avenue SW
Calgary, Alberta, T2P 4K3

(b) This certificate applies to the technical report titled Preliminary Economic Assessment NI 43-101 Technical Report on the Tonopah Gold Project, Nevada, USA with an effective date of: August 20, 2025 (the "Technical Report").

(c) I am a "qualified person" for the purposes of National Instrument 43-101 ("NI 43-101"). My qualifications as a qualified person are as follows. I am a graduate of the University of Calgary with a B.Sc. in Geology and a B.Sc. in Earth Science from 2011, I am a member in good standing of the Association of Professional Engineers and Geoscientists of Alberta (#102171). My relevant experience after graduation and over 13 years of experience in mining and resource geology, for the purpose of the Technical Report includes 7 years of mine geology experience, 6 years of consulting experience with a strong focus on gold and base metals related projects, and 3 years of experience working on the Tonopah Gold Project.

(d) My most recent personal inspection of each property described in the Technical Report occurred on June 6, 2023 and was for a duration of 2 days.

(e) I am responsible for Item(s) 1.1, 1.2, 1.3, 1.4, 1.5, 1.6, 1.11.1.1, 1.11.2.1, 1.11.2.6, 2.0, 3.0, 4.0, 5.0, 6.0, 7.0, 8.0, 9.0, 10.0, 11.0, 12.0, 23.0, 24.0, 25.1, 26.1, 26.6, and 27.0 of the Technical Report.

(f) I am independent of the issuer as described in section 1.5 of NI 43-101.

(g) My prior involvement with the property that is the subject of the Technical Report is as follows. I have provided internal drilling data reviews and updates, and internal geological modelling and grade estimation for the Tonopah Gold Project for Viva Gold Corp.

(h) I have read NI 43-101 and the parts of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101; and

(i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible, contain(s) all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Calgary, Alberta this 20th of August, 2025.

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Róisín Kerr, P.Geo.

CERTIFICATE OF QUALIFIED PERSON JASON BAKER

I, Jason Baker, state that:

(a) I am a Senior Principal Mining Engineer at:
WSP Canada Inc.
1 Spectacle Lake Drive
Dartmouth, Nova Scotia, B3B 1X7

(b) This certificate applies to the technical report titled Preliminary Economic Assessment NI 43-101 Technical Report on the Tonopah Gold Project, Nevada, USA with an effective date of: August 20, 2025 (the "Technical Report").

(c) I am a "qualified person" for the purposes of National Instrument 43-101 ("NI 43-101"). My qualifications as a qualified person are as follows. I am a graduate of Dalhousie University with a Bachelor of Mining Engineering in 2000. My relevant experience after graduation and over 28 years for the purpose of the Technical Report includes over 5 years in open pit gold operations, over 15 years in open pit mine operations, and over 10 years in gold mineral reserve estimation experience.

(d) The requirement for a site visit is not applicable to me.

(e) I am responsible for Item(s) 1.7.1, 1.11.1.2, 1.11.2.2, 1.11.2.6, 15.0, 16.0, 18.1.3, 25.2, 26.2, and 26.6 of the Technical Report.

(f) I am independent of the issuer as described in section 1.5 of NI 43-101.

(g) I have not had prior involvement with the property that is the subject of the Technical Report.

(h) I have read NI 43-101 and the parts of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101; and

(i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible, contain(s) all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Dartmouth, Nova Scotia this 20th of August, 2025.

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Jason Baker; P.Eng.

CERTIFICATE OF QUALIFIED PERSON WILLIAM RICHARD MCBRIDE

I, William Richard McBride, state that:

(a) I am a Senior Principal Mining Engineer at:
WSP Canada Inc.
33 Mackenzie Street, Suite 100
Sudbury, Ontario, P3C 4Y1

(b) This certificate applies to the technical report titled Preliminary Economic Assessment NI 43-101 Technical Report on the Tonopah Gold Project, Nevada, USA with an effective date of: August 20, 2025 (the "Technical Report").

(c) I am a "qualified person" for the purposes of National Instrument 43-101 ("NI 43-101"). My qualifications as a qualified person are as follows. I am a graduate of Queen's University (Kingston) with a Bachelor of Science degree in Mining Engineering granted in 1973. I am a Registered Member of the Professional Engineers of Ontario (PEO), License Number 29888013. My relevant experience after graduation and over 50 years of working as a mining engineer and consultant for the purpose of the Technical Report includes working on projects involving multiple commodities such as copper, gold, and nickel and projects involving public disclosure reporting.

(d) The requirement for a site visit is not applicable to me.

(e) I am responsible for Item(s) 1.9, 1.10, 1.11.1.5, 1.11.1.6, 1.11.2.5, 19, 21.1, 21.2.1, 21.2.2, 21.2.3, 21.2.4, 21.2.5, 21.2.6, 21.2.7, 21.2.9, 21.2.10, 21.2,11, 21.3.1, 21.3.4, 22, 25.5, and 26.5 of the Technical Report.

(f) I am independent of the issuer as described in section 1.5 of NI 43-101.

(g) I have not had prior involvement with the property that is the subject of the Technical Report.

(h) I have read NI 43-101 and the parts of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101; and

(i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible, contain(s) all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Sudbury, Ontario this 20th of August, 2025.

Rick.McBride@wsp.com Digitally signed by Rick.McBride@wsp.com DN: cn=Rick.McBride@wsp.com Date: 2025.08.20 14:00:34 -04'00'

William Richard McBride, P.Eng.

William Richard McBride, P.Eng.

CERTIFICATE OF QUALIFIED PERSON RANDAL HUFFSMITH

I, Randal Huffsmith, state that:

(a) I am a Senior Vice President at:
WSP USA Inc.
815 Great Northern Blvd, Suite 304
Helena, MT 596012

(b) This certificate applies to the technical report titled Preliminary Economic Assessment NI 43-101 Technical Report on the Tonopah Gold Project, Nevada, USA with an effective date of: August 20, 2025 (the “Technical Report”).

(c) I am a “qualified person” for the purposes of National Instrument 43-101 (“NI 43-101”). My qualifications as a qualified person are as follows. I am a graduate of University of Wyoming with a BS in Ag/Civil Engineering and an MS in Ag/Civil Engineering and a Professional Engineer registered in Montana (Civil) and Professional Engineer (Civil) registered in Arizona. My relevant experience after graduation and over the last 40 years for the purpose of the Technical Report includes environmental remediation and environmental compliance for mining and industrial projects including work for many of the largest mining companies in the world, regulatory agencies and state and local government. I am a Board Certified Environmental Engineer for the American Academy of Environmental Engineers and have worked on mine permitting, surface and groundwater sampling, water treatment and waste encapsulation. In addition, I have supported mine planning phases and mine design and waste system design, groundwater supply and evaluation and remediation of groundwater contamination and worked with experts to collaborate on cultural and biological resources and mitigation and protection of the same.

(d) The requirement for a site visit is not applicable to me.

(e) I am responsible for Item(s) 1.11.1.4, 1.11.2.4, 1.11.2.6, 20.0, 25.4, 26.4, and 26.6 of the Technical Report.

(f) I am independent of the issuer as described in section 1.5 of NI 43-101.

(g) I have not had prior involvement with the property that is the subject of the Technical Report.

(h) I have read NI 43-101 and the parts of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101; and

(i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible, contain(s) all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Helena, Montana this 20th of August, 2025.

Huffsmith, Randy (USRH716293)

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Randy L. Huffsmith, P.E. MT PE #10554PE

CERTIFICATE OF QUALIFIED PERSON

I, Caleb Cook, P.E., of Reno, Nevada, USA, Engineering Manager at Kappes, Cassiday & Associates, as an author of this report entitled “Preliminary Economic Assessment NI 43-101 Technical Report on the Tonopah Gold Project, Nevada, USA”, prepared for Viva Gold Corp. (the “**Issuer**”) do hereby certify that:

1. I am employed as the Engineering Manager at Kappes, Cassiday & Associates, an independent metallurgical and engineering consulting firm, whose address is 7950 Security Circle, Reno, Nevada 89506.

2. This certificate applies to the technical report “Preliminary Economic Assessment NI 43-101 Technical Report on the Tonopah Gold Project, Nevada, USA”, effective date August 20, 2025 (the “**Technical Report**”).

3. I am a Professional Engineer in the state of Nevada (No. 025803) and my qualifications include experience applicable to the subject matter of the Technical Report. In particular, I am a graduate of the University of Nevada with a B.S. in Chemical Engineering (2010) and have practiced my profession continuously since graduating. Most of my professional practice has focused on the development of gold-silver leaching projects.

4. I am familiar with National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* (“**NI 43-101**”) and by reason of education, experience and professional registration I fulfill the requirements of a “qualified person” as defined in NI 43-101.

5. I visited the Tonopah Gold Project for a total of one day on 8 May 2025.

6. I am responsible for Items 1.7.2, 1.7.3, 1.7.4, 1.11.1.3, 1.11.2.3, 1.11.2.6, 13.0, 17.0, 18.1.1, 18.1.2, 18.2, 18.3, 18.4, 18.5, 18.6, 21.2.8, 21.3.2, 21.3.3, 25.3, 26.3, and 26.6 of the Technical Report.

7. I am independent of the Issuer as described in section 1.5 of NI 43-101.

8. I have had no prior involvement with the property that is the subject of the Technical Report.

9. I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101.

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

Dated this 20th day of August, 2025

Caleb Cook Digitally signed by Caleb Cook Date: 2025.08.20 11:16:20 -07'00'

Caleb Cook, P.E. Chemical Engineering
Engineering Manager at
Kappes, Cassiday & Associates

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1.0 Executive Summary

This Technical Report was prepared for Viva Gold Corp. (Viva) and presents an updated Mineral Resource estimate and Preliminary Economic Assessment (PEA) for the Tonopah Gold Project located near Tonopah, Nevada, USA (previous Technical Report effective date: January 1, 2022). Viva owns a 100% interest in the Project.

The Mineral Resource estimate and Technical Report were prepared by WSP Canada Inc. (WSP), in conjunction with Kappes, Cassiday & Associates (KCA) for the metallurgy, recovery, and infrastructure components and Lewis Environmental Consulting LLC (LEC) for the environmental components of the study, and reviewed by WSP. The Mineral Resource estimate is disclosed in accordance with the Canadian Securities Administrator's National Instrument (NI) 43-101, and this Technical Report follows the requirements of Form 43-101F.

1.1 Property Description and Location

The Tonopah property spans approximately 4,092 hectares (10,112 acres) and is situated about 30 kilometres (km) northeast of the town of Tonopah, Nevada (Figure 4.1). The Project is in Nye Country and centered on Universal Transverse Mercator (UTM) Zone 11 North, North American 1983 Datum (NAD83) 493,880 metres (m) east and 4,235,070 m north.

Viva has 508 unpatented lode claims (including 184 royalty claims) that are 100% controlled by Viva and filed with both the Bureau of Land Management (BLM) and Nye County (Figure 4.2). A maintenance fee on unpatented claims of United States (US) $200 per claim must be paid to the BLM annually. Surface rights are managed by the BLM except for those within Section 32, which are under private ownership through the Stock Raising Homestead Act (SRHA). Within Section 32, Viva controls a 16.2-hectare (40-acre) parcel of the private surface land with 929 square metre (m^2) (10,000 square feet [ft^2]) allocated to a long-term lease arrangement for telecommunications infrastructure.

The Project is subject to a royalty agreement following various historical transactions, bankruptcy proceedings, and modifications. The current arrangement grants the Optionors a 2% Net Smelter Return (NSR) royalty on 184 claims upon commercial production, with an option for Viva to buy down half of the royalty. Viva has also issued cash payments and company shares to the royalty holders. Viva has the right at any time to acquire 1% of the 2% royalty for US$1.0 million (M).

Environmental management is overseen through a series of permits and reclamation bonds held with the BLM and Nevada agencies, covering surface disturbance, groundwater monitoring, and access to both public and private lands. Surface disturbance up to 30.4 hectares is authorized for mineral exploration and support activities. To date, environmental liabilities are minimal, with only a small area requiring reclamation and no significant regulatory issues identified. Agency field inspections confirm compliance, and ongoing monitoring addresses water quality, wildlife, and cultural resource preservation.

Viva holds the permits and authorizations required to carry out mineral exploration, environmental monitoring, and testing activities on both public and private land, as well as to access privately-owned property via public land.

Access, title, and operational risks are primarily associated with water resource management, compliance with environmental standards, and potential infrastructure adjustments, such as road relocations.

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

The Project is located in central Nevada, characterized by gentle to moderate terrain, seasonal streams, and elevations ranging from 1,750 to 2,100 m above mean sea level (amsl). The area supports typical high-altitude desert vegetation, including various scrub plants and cacti, with some weedy species also present. The high desert climate of central Nevada features approximately 21 centimetres (cm; 8-inches) of annual precipitation and 50 cm (20-inches) of snowfall (Climate-Data.org, 2021). Average temperatures range from 3°C (37°F) in winter to 18°C (65°F) in summer, with July and August highs often above 30°C (87°F) (WeatherSpark, 2023). Year-round work is possible at the site.

The Tonopah site is accessible year-round via State Route (SR) 376, which connects with Nye County Road (CR) 82 near the Project's center. The property is about 30 km (20-miles) from Tonopah, Nevada, by paved road. Infrastructure in the region is robust, supported by local utilities, including electrical power sourced from a line connecting to the Nevada grid. The Project is located 4 km from a 15-kilovolt (kV) power line, with the potential for the power line to be upgraded to 25 kV under existing permits. Water for exploration and operational needs can be obtained from on-site wells, third-party suppliers, or directly from public utility sources, as the Ralston Valley basin has substantial, under-allocated groundwater resources.

Logistical support for the Project is provided from the town of Tonopah, which also serves the Round Mountain Mine located approximately 50 km (31-miles) to the north, as well as other exploration and mineral resource projects in Nye and Esmeralda counties. The area has a history of mining activity, with personnel and resources for operations at Tonopah expected to be sourced from local and regional communities. Reno and Las Vegas are both situated within a 3.5-hour (h) drive from the site.

As discussed in Item 1.1, the majority of surface rights are held by the BLM. This PEA defines the proposed mining area and operational footprint, providing insight into the surface rights that may be required for future project development.

1.3 History

The Tonopah property has a complex history of ownership and exploration stretching back to the early 1900s. Originally consisting of 245 private claims, the property has changed ownership numerous times, with significant activities beginning in the 1970s. Notably, Midway Gold Corp. (Midway Gold) acquired full ownership in 2004 and conducted extensive exploration until 2015, when bankruptcy proceedings led to the property's sale to Viva. This transition included the assumption and restructuring of royalty and environmental obligations by Viva.

Exploration at Tonopah and the surrounding mining districts has been ongoing for over a century, though there is no record of significant gold (Au) or silver (Ag) production directly from the Tonopah Project site. Early mining activity left behind a shaft and several prospect pits, but detailed records are lacking. The area has seen a succession of companies conduct various drilling and geophysical surveys, including Houston Oil and Minerals, Coeur d'Alene Mines Corporation, Rio Algom Ltd., Kennecott Exploration Company (Kennecott), Newmont Mining Corporation (Newmont), Midway Gold, and others. These efforts focused on both the property itself and adjacent areas, resulting in the identification of mineralized zones such as the Discovery Zone.

From the late 1990s onward, a series of joint ventures and agreements furthered exploration, culminating in detailed drilling campaigns and geochemical studies by Midway Gold and its partners. Between 2002 and 2012, hundreds of drill holes and comprehensive surveys provided a substantial dataset for resource estimation and geological understanding. Over time, the claim area was reduced to its current size, and focus narrowed to the most promising zones.

1.4 Geology and Mineralization

The Tonopah property is located on the northeastern margin of Nevada's Walker Lane structural zone, an area defined by right-lateral strike-slip faults separating the Sierra Nevada from the Basin and Range Province (Bonham and Garside, 1979). The geology is complex, comprising ancient Ordovician Palmetto Formation (Op) rocks overlain by a sequence of younger Miocene volcanic and volcaniclastic units, as well as extensive Quaternary valley fill deposits that obscure much of the bedrock (Figure 7.1). The structural framework is dominated by the Rye Patch fault system and associated relay zones, which are critical controls on mineralization.

Au mineralization at Tonopah is hosted in a low-sulphidation epithermal system, characterized by nearly vertical quartz-adularia veins developed predominantly within the Op argillite. These veins extend into overlying Tertiary volcanic and volcaniclastic rocks, where broader zones of lower-grade, disseminated Au can be found. The nonconformity between the Op and Tertiary volcanics marks a significant mineralized horizon, with high permeability and porosity that allows for the widespread distribution of mineralized fluids.

The local geology reveals that mineralization is structurally controlled, with Au concentrated along north-to-northwest trending extension fractures and veins, primarily within the Discovery and Dauntless zones (Figure 7.5). Visible Au is frequently observed in drill core, particularly in quartz veins and hydrothermal breccias. These mineralized trends are often associated with alteration zones marked by argillic and quartz-adularia assemblages, and the highest grades are linked to distinctive siliceous vein textures suggestive of boiling within the hydrothermal system.

Overall, the complex interplay of structure, lithology, and alteration at Tonopah has produced multiple, overlapping mineralized zones that remain open along strike. Drill data indicates continuity of veins and Au mineralization over significant strike lengths and vertical extents, although the distribution of higher grades becomes less predictable at stricter cut-off levels. The area's geology and mineralization style are typical of productive epithermal gold systems found elsewhere in Nevada.

1.5 Exploration

The Project has undergone extensive exploration activities since 1986, involving multiple operators and a range of geophysical and drilling programs. Initial exploration started with geological surveys and claim staking, followed by major geophysical surveys in the 1990s and early 2000s, conducted by Kennecott and Newmont. These efforts established a robust geophysical database, used for refining drilling targets and developing the geological model for the property. Viva commissioned an aerial flyover of the property in 2022, producing an orthophoto and digital surface model (DSM) at a resolution of 25 cm.

1.5.1 Drilling

Drilling at Tonopah has been a central component of the exploration strategy, with a total of 626 drill holes and 90,716 m drilled to date, including both reverse circulation (RC) and diamond drill (DD) core methods (Table 10.1). The drilling campaigns have evolved from early RC techniques with continuous sampling to more sophisticated programs that include geotechnical and hydrological studies.

Viva has conducted annual drilling programs since 2018, focusing on confirming historic data, extending mineralization, and collecting fresh samples for metallurgical testing. Between 2018 and 2024, 106 RC and 15 DD holes were drilled, totaling 17,984 m. Drilling activities were supervised by Mr. Ed Bryant, a qualified geology sub-contractor to Viva, with suitable documentation of drilling methods and drill hole details.

Drill collar locations were initially marked using handheld Global Positioning System (GPS), with higher-precision surveys conducted for some collars in 2020 and additional coordinate adjustments made following the 2022 aerial flyover.

DD core and RC chip handling included careful labelling, boxing, and transportation to the logging and storage facility, where samples were checked, photographed, and logged. Both core and RC chips were sampled at consistent intervals, labelled with unique identifications (IDs), and securely stored prior to analysis. Core samples were typically split or sawed, with half retained for reference and half sent for laboratory analysis, while RC chip samples followed similar collection and security protocols.

1.5.2 Sample Preparation, Analyses, and Security

Assaying and analytical work has been carried out by independent, ISO-certified laboratories such as ALS Limited (ALS) and American Assay Laboratories (AAL), both of which adhere to strict chain-of-custody protocols. Sample preparation involves crushing, splitting, and pulverizing to prescribed specifications, and analytical methods include fire assay for Au and multi-acid digestion for Ag. Over 53,000 samples have been collected and analyzed since 1988, with the majority of assay results and certificates securely archived by program and drilling campaign.

Quality assurance and quality control (QA/QC) programs have been implemented to industry standards, especially by Viva since 2018. This includes the insertion of certified reference materials (CRMs), blanks, and laboratory duplicates in the sample stream at regular intervals. The performance of these QA/QC measures is monitored using statistical criteria, with the majority of control samples falling within acceptable tolerance ranges. Historical QA/QC protocols varied among prior operators, but Viva’s contemporary protocols are consistently applied.

Bulk density measurements have been performed using both wax-sealed and unsealed rock samples, following standard industry methods based on water displacement and mass calculations. The compiled geological, geotechnical, and analytical data supports the reliability of the Project’s resource models and ensures compliance with regulatory standards for technical reporting.

1.6 Data Verification

The WSP Qualified Person (QP) completed a 2-day site visit that included observation and review of drilling, logging, and sampling procedures, and observation of the core storage facility. The QP was able to verify the location of a selection of drill hole collars and send a representative suite of samples for replicate analytical comparison. All reviewed drill logs and collar data were found to be consistent with the Viva database and no material differences were identified.

Drill hole database verification included extensive validation of drill hole data for accuracy, completeness, and consistency. This encompassed checks for duplicate records, data gaps, unit conversion issues, and alignment of drill hole collar coordinates with the control points from the 2022 aerial survey. Historical differences in lithology coding were reconciled, and assay data tables were rebuilt and validated against original laboratory certificates where available. Additional data verification included reviewing and adjusting collar elevations to match the 2022 aerial survey DSM.

Verification on data was limited as the QP could not directly observe earlier drilling and sampling programs. Nevertheless, forensic review and industry-standard database checks were conducted to ensure the reliability of the geological database for modelling and resource estimation. The QP concluded that the data collection and management procedures met industry best practices and were suitable for Mineral Resource estimation.

The KCA QP visited the site in May 2025 to observe and evaluate the proposed heap leach and other processing facilities locations and reviewed local utilities and infrastructure.

1.7 Mining and Metallurgy

1.7.1 Mining Method

Various mining scenarios were analyzed for Tonopah, considering different processing strategies and economic inputs. The base case used an Au price of US$2,150 per troy ounce (oz) and involved a combined processing approach: a 2,000 tonnes per day (tpd) mill and an 8,000 tpd heap leach facility, with final pit selection maximizing resources and economic margin. A mining recovery of 100% and 0% dilution were assumed for pit optimization and scheduling. Au recovery was assumed at 92% for the mill and 75% for the heap leach. Based on the chosen Au price, the cut-off grades (COGs) are 0.18 grams per tonne (g/t) Au for heap leach, 0.28 g/t Au for mill feed, and 0.76 g/t Au as the break-even grade between the two processing routes.

The mine design features three pit phases, each tailored to prioritize higher-grade material early and manage pre-stripping efficiently. Access and haul roads, ramp geometry, and bench configurations were optimized for 100-ton class equipment, maintaining safety and operational efficiency. Waste rock storage was strategically planned to accommodate both current and future needs. Approximately 16% of waste rock was planned to dump within the pit, a strategic move that significantly reduced costs as the mine approached closure.

Production planning is focused on maximizing project net present value (NPV) while maintaining practical equipment requirements and minimizing fluctuations in mining rates. The schedule fills both mill and heap leach circuits to capacity when possible, prioritizing high-grade material for the mill and routing lower-grade material to the heap leach. The COGs and material routing ensure that mineral processing is both efficient and economically advantageous, with only minimal rehandling of material required.

Supporting the mine plan is an owner-operated equipment fleet, including 3 Caterpillar (CAT) 992 front-end loaders, 8 to 11 CAT 777 haul trucks, 3 CAT MD6250 drill rigs, and ancillary support vehicles, all sized to match production targets. This approach leverages local skilled labour, ensuring strong mineralized material control and operational flexibility. The overall strategy is designed to maintain a steady production profile, efficient resource utilization, and adaptability for future expansion or modification based on ongoing operational results and economic factors.

1.7.2 Mineral Processing and Metallurgical Testing

Scoping-level metallurgical studies were undertaken by various operators, including Kennecott (1994-1996), Newmont (2003) and Midway Gold (2006 to 2009) for the Tonopah property which was primarily focused on testing higher-grade vein and breccia type material. Additional test work was commissioned by Viva between 2018 and 2023 and completed by Resource Development, Inc (RDi), McClelland Laboratories (MLI), and KCA; which form the basis for the conclusions derived in this study. These results indicate that Au and Ag mineralization from the Tonopah project are amenable to recovery by gravity concentration and cyanide leaching with the following recommended parameters:

- Heap Leach Parameters:
  - Crush size of 100% passing 12.5 millimetre (mm)
  - Variable recoveries for Au and Ag based on head grade and material type with an overall Au recovery of 76% for grades of 0.5 g/t or less for both volcanic and argillic material types and Ag recoveries of 15% and 19% for Ag grades of 1.0 g/t or less for argillic and volcanic material, respectively
  - Design leach cycle of 120 days
  - Agglomeration will be required with an estimated cement addition of 4.0 kilograms per tonne (kg/t) material
  - Cyanide consumption of 0.26 kg/t material
- Mill parameters:
  - Grind size of 80% minus 106 µm (micron) (150 mesh)
  - Au recovery of 95% for argillic material and 90% for volcanic material
  - Ag recovery of 36% for argillic material and 38% for volcanic material
  - 48 h leach residence time
  - Lime addition of 0.60 kg/t material
  - Cyanide consumption of 0.58 kg/t material

In general, the Tonopah deposit shows some variability in recoveries by the two primary material types with heap recoveries for Au being more sensitive to head grade than product crush size. Coarse Au is present in some of the material and may be contributing to the lower recoveries for high-grade heap leach material. Au recoveries for milled material were consistently high at both 106 and 75 µm grind sizes (150 and 200 mesh) and gravity pre-concentration is recommended to recover coarse Au particles.

1.7.3 Recovery Methods

Mineralized material from the Project will be processed by a combination of heap leaching and milling with Carbon-in-Leach (CIL) recovery at a combined rate of 10,000 tpd, including 8,000 tpd of low-grade heap leach material and 2,000 tpd of high-grade mill material.

Mineralized material will be crushed to 100% passing 12.5 mm using a three-stage closed crushing circuit. High-grade and low-grade material will be campaigned through the crushing circuit and stockpiled separately using a radial stacking conveyor. Low-grade material will be reclaimed from the low-grade stockpile and agglomerated with cement, which will also act as a pH buffer, before being conveyor stacked in 10 m lifts onto a permanent geomembrane-lined heap leach pad and leached with a dilute cyanide solution. The resulting pregnant leach solution will flow by gravity to a pregnant solution tank before being pumped to a carbon adsorption circuit. Au values will be loaded onto activated carbon and then periodically transported off site to be toll-processed, where the loaded carbon will be stripped and regenerated before being returned to the project for re-use. Barren solution leaving the adsorption circuit will flow by gravity to a barren solution tank before being pumped back to the heap.

High-grade material will be reclaimed and ground to 80% passing 150 Mesh (106 µm) in a single stage ball mill which will operate in a closed circuit. Lime will be added to the high-grade material for pH control before being fed to the ball mill along with process solution. Ball mill discharge will be pumped to a hydrocyclone cluster for classification with a portion of the cyclone underflow being diverted to a gravity concentrator for the recovery of coarse metal with the remaining cyclone underflow being returned to the ball mill feed. Cyclone overflow material will be thickened before reporting to a six-stage CIL circuit where the thickened slurry will be mixed counter-flow with activated carbon and will flow from one stage to the next through carbon interstage screens. Sodium cyanide (NaCN) will be added to the first three CIL tanks as needed. Leached slurry leaving the last tank will be discharged as tailings to a tailings thickener before being pumped to a filter feed tank, filtered using a filter press, and dry-stacked using trucks onto a dedicated portion of the heap leach pad. Loaded carbon from the first tank of the CIL will be toll-processed along with carbon from the heap leach circuit.

1.7.4 Project Infrastructure

The Project is accessed by paved SR 376 and Nye CR 82. Nye CR 82 will need to be relocated during the project development. Internal site roads will be primarily dirt roads which will connect the site and process facilities for access.

The Project will use a combination of pre-fabricated office trailers and pre-engineered steel buildings for operations. Buildings considered include administrative and process office trailers, a site laboratory, mine truck shop and warehouse facility and roof over the milling facility.

Power for the operation will be provided to the project site via an existing 15 kV power line which runs along the eastern side of the property and is only lightly utilized. The line connects to the Nevada Energy power grid and may be upgraded under existing permits to a 25 kV service and is assumed to have sufficient capacity, to meet the project needs. Power from the existing line will be stepped down and distributed using overhead power lines at 4.16 kV, 3 phase, 60-hertz (Hz) and will be further stepped down to 480-volts (V), 220 V and 110 V as required. Power will be supplied to Motor Control Centers (MCCs) or distribution panels at 4160 V for motors above 373-kilowatts (kW) (500 horsepower [hp]) and 480 V for 220/110 V for motors below 373 kW and low voltage systems. All overhead distribution lines will be connected to a main switchgear which will include synchronization, control panels, disconnects, circuit breakers, instrumentation and data logging.

Raw water will be provided by production wells and will be pumped to a head tank for distribution to other areas. A portion of the head tank will be used to provide fire water storage. Potable water is planned to be delivered to the site from Tonopah via truck and distributed using a potable water storage and transfer pumping system.

1.8 Mineral Resource

The Mineral Resource estimate for the Tonopah Gold Project has been prepared in accordance with NI 43-101 following the requirements of Form 43-101F. The Mineral Resource estimate follows the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Estimation of Mineral Resources and Reserves Best Practice Guidelines (MRMR Best Practice Guidelines), issued November 29, 2019, and was classified following CIM Definition Standards (CIMDS) for Mineral Resources and Mineral Reserves adopted May 10, 2014.

This Mineral Resource estimate was completed by Róisín Kerr, P.Geo. an employee of WSP Canada Inc., under the supervision of Brian Thomas, P.Geo., an independent QP, as defined under NI 43-101 and an employee of WSP Canada Inc. The Effective Date of the Mineral Resource estimate is June 13, 2025.

The Mineral Resource estimate outlined in this Technical Report was determined from drill hole data provided by Viva and a geological model developed by WSP, using a Three-Dimensional (3D) block modelling approach in Maptek Vulcan™ (Vulcan) software and reported within a constrained volume for open-pit (OP) mining.

This sub-item contains forward-looking information related to Mineral Resource estimates for the Project. The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts, or projections in the forward-looking information include any significant differences from one or more of the material factors or assumptions that were set forth in this sub-item, including geological and grade interpretations and controls, assumptions and forecasts associated with establishing the prospects for eventual economic extraction.

Table 1.1 summarizes the Mineral Resource estimate for the Tonopah Gold Project. Mineral Resources were evaluated for reasonable prospects of eventual economic extraction (RPEEE) by reporting OP resources within a constrained pit shell at an Au COG of 0.15 g/t.

Table 1.1: Mineral Resource Estimate

Resource Classification	Tonnes (kt)	Grade		Contained Metal	
		Au (g/t)	Ag (g/t)	Au (oz)	Ag (oz)
Measured	1,690	1.41	3.11	77,000	169,000
Indicated	25,000	0.53	1.98	427,000	1,593,000
Measured + Indicated	**26,690**	**0.59**	**2.05**	**504,000**	**1,762,000**
Inferred	6,905	0.37	1.81	83,000	402,000

Notes:

1. The Mineral Resource estimate for the potentially surface mineable resource were constrained by a conceptual pit shell for the purpose of establishing reasonable prospects of eventual economic extraction based on potential mining, metallurgical and processing grade parameters identified by studies performed to date on the Project.
2. Key constraint inputs included reasonable assumptions for operating costs, geotechnical slope parameters, forecast Au prices, and a minimum Cut-off Grade of 0.15 g/t Au.
3. The Cut-off Grade assumes an Au price of US$2,200 and a revenue factor of 1.2 (equivalent to US$2,640 Au price), and includes all material that can be economically processed
4. Heap leach recovery of 75% was assumed.
5. Tonnage and contained metal estimates are rounded to the nearest 1,000.
6. kt = kilotonnes; g/t = grams per tonne; oz/t = troy ounces per tonne.
7. Mineral Resource categorization of Measured, Indicated and Inferred Mineral Resources presented in the summary table is in accordance with the CIM definition standards (CIMDS, 2014).
8. No mining recovery, dilution or other similar mining parameters have been applied.
9. Although the Mineral Resources presented in this Technical Report are believed to have a reasonable expectation of being extracted economically, they are not Mineral Reserves. Estimation of Mineral Reserves requires the application of modifying factors and a minimum of a Pre-Feasibility Study (PFS).
10. The reported Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves.
11. There is no certainty that all or any part of this Mineral Resource will be converted into Mineral Reserve.
12. Mineral Resource estimates are not precise calculations, being dependent on the interpretation of limited information on the location, shape and continuity of the occurrence and on the available sampling results. All figures are rounded to reflect the relative accuracy of the estimates.

1.9 Capital and Operating Costs

The capital cost estimate (CAPEX) and operating cost estimate (OPEX) were compiled by WSP and are based on the scope of work presented in the Items of this Technical Report.

WSP developed the CAPEX and OPEX for the mining and mine infrastructure for the Project described in this Report. All contributors to the CAPEX and OPEX are as follows:

- **WSP**: Overall responsibility for CAPEX/OPEX, Open Pit Mining, Minesite Infrastructure
- **KCA**: Provided process plant initial and sustaining CAPEX and OPEX. Provided General and Administrative (G&A) cost estimate.
- **Piteau Associates (Piteau)**: Provided Dewatering CAPEX and OPEX; initial and sustaining costs
- **LEC**: Provided Bond Premium and Closure Costs

1.9.1 Capital Cost Estimate

The CAPEX was developed to deliver an Association for the Advancement of Cost Estimating (AACE) class 5 estimate in support of a PEA. AACE guidelines identify class 5 estimates as having an accuracy range with a low of -20% to -50% and a high of +30% to +100%.

As each portion was estimated individually by the responsible parties, the accuracy ranges and contingency allowances are identified within each party's sub-item. KCA states for the processing, "Capital cost estimates were based on the second quarter (Q2) of 2025 United States dollars (US$) and are considered to have an accuracy of +/ - 35%.", thereby falling within the range described in the AACE guidelines for a class 5 estimate.

Table 1.2 presents separate summaries of the Initial CAPEX, and Sustaining CAPEX distributed over the life-of-mine (LoM) with the two indicated categories of capital then totaled. Owner's costs, contingencies and risk amounts are included in this CAPEX.

Table 1.2: Capital Cost Summary

Area	Description	Initial CAPEX (US$M)	Sustaining CAPEX (US$M)	Total (US$M)
Mine	Equipment lease payments	13.4	55.3	**68.7**
Mine	Shops and Other Surface Infrastructure	8	0	**8**
Mill	Processing Capital	120.7	9.6	**130.2**
G&A	Dewatering Capital	9.9	5.6	**15.5**
G&A	Royalty Purchase Option	1	0	**1**
Indirects	Mine Indirect, First Fills, & Owners Costs	5.2	0	**5.2**
Indirects	Process Plant Indirect, First Fills, & Owners Costs	11.1	0	**11.1**
Indirects	Engineering, Procurement & Construction Management (EPCM)	17.2	0	**17.2**
Indirects	Mine Contingency (12% of total* directs and indirects)	10	0	**10**
Indirects	Process Plant Contingency (15% of total** directs and indirects)	23.4	0	**23.4**
	Total Major Area CAPEX	219.9	70.5	**290.4**
Other	Working Capital	22.2	-22.2	**0**
Other	Surety Bond Premium	0.4	OPEX G&A	**0.4**
Other	Reclamation Bond Restricted Cash Collateral	4.6	-5.5	**-0.9**
Other	Project Closure and Rehabilitation	0	12	**12**
Other	Equipment Salvage Value	0	-16.2	**-16.2**
	Total CAPEX	**247.1**	**38.6**	**285.7**

Notes:

1. *Mine Contingency reflective of 10% applied to budgetary priced CAT equipment and 30% applied to non-quoted equipment and infrastructure.
2. **Mill contingency applied to total capital including initial and Sustaining Capital
3. Numbers may not sum precisely due to rounding.
4. M = Million

The capital is representative of a mine mobile equipment lease for the loaders, haul trucks, production drills, and auxiliary equipment.

The CAPEX is based on the following key assumptions:

- All relevant permits in a timely manner to meet the Project schedule.
- Quotes from Vendors for leasing equipment to be valid for budget purposes.
- Suitable backfill material is available locally. Soil conditions are adequate for foundation bearing pressures.
- Engineering and Construction activities will be carried out in a continuous program with full funding available, including contingency.
- Bulk materials such as cement, rebar, structural steel and plate, cable, cable tray, and piping are all readily available in the scheduled timeframe.

- Capital equipment is available in the timeframe shown.
- The construction and commissioning period begins upon receipt of construction permit, including Open Pit stripping, process plant, and infrastructure development.
- The production plan assumes the mill is immediately constructed and available at full capacity 2000 tpd for 365 days per year throughout the Life-of-Mine Plan (LoMP).

The following items were not included in the CAPEX:

- Provision for inflation, escalation, currency fluctuations and interest incurred during construction.
- Schedule delays and associated costs.
- Scope changes.
- Unidentified ground conditions.
- Extraordinary climatic events.
- Force majeure.
- Labour disputes.
- Insurance, bonding, permits and legal costs beyond that related to the closure bond.
- Schedule recovery or acceleration.
- Cost of financing, Property taxes, corporate and mining taxes, duties; and salvage values.
  - However, they are considered in the Economic Analysis.
- There will be no construction camp and catering in this CAPEX.
  - It is assumed that non-local contractors will source local accommodation.
  - It is further assumed that accommodation is available in the surrounding areas for the Engineering, Procurement, and Construction Management (EPCM) personnel and vendor supervisors.

1.9.2 Operating Cost Estimate

A summary of the overall LoM project OPEX is presented in Table 1.3 The OPEX costs presented in the table exclude pre-production OPEX allowances for mining, process and G&A. The unit costs shown are based on 23.6 million tonnes (Mt) of mineralized material mined and processed (Table 1.4).

Table 1.3: OPEX by Major Area

Area	LoM Total OPEX (US$M)	OPEX per Total Tonne Mined	OPEX per Tonne Mill Feed	OPEX per Tonne Heap Leach Feed	Average OPEX (US$/t processed)
Mining	$227.90	$1.95			$9.67
Mill	$74.20		$16.43		$3.15
Heap Leach	$130.30			$6.62	$5.53
Dewatering	$6.50				$0.28
G&A	$31.60				$1.34
Total OPEX	**$470.50**				**19.97**

Notes: Numbers may not sum precisely due to rounding.

1.10 Economic Analysis

The results of the economic analysis contain forward-looking information under Canadian securities law. The results rely on inputs that are subject to known and unknown risks, uncertainties, and other factors, which may cause actual results to differ materially from those presented here.

At assumed long-term prices for Au of US$2,400/oz and for Ag of US$27.70/oz, the financial results indicate a positive post-tax NPV at 5% of US$111.6M, an after-tax Internal Rate of Return (IRR) of 17.6% and a payback period of 3.6 years

The economic analysis is based on the discounted cash flow (DCF) method on a pre-tax and after-tax basis. The key metrics determined in the analysis are the NPV at a discount rate of 5%, the IRR, and the Payback Period. For the purposes of the evaluation, it is assumed that the operations are established within a single corporate entity. The Project has been evaluated on an unlevered, all-equity basis.

The cash flow model uses inputs from all elements of the Project to provide a comprehensive financial projection for the entire Project, on an annual basis, supporting an 8-year life of the Project with 7-year mining and milling period. Heap leach extends into year 8 to recover the residual leachate material. Years 8 to 10 use on-hand mining equipment to facilitate closure. All prices and costs are in US dollars and accurate as of Q2 2025. No provisions have been made for the effects of inflation.

Table 1.4 provides a summary of the key technical assumptions and inputs.

Table 1.4: Economic Analysis Parameters

Description	Unit	Value
Macroeconomic Parameters		
Au Price	US$/oz	2,400.00
Ag Price	US$/oz	27.70
Discount Rate	%	5
Project Parameters		
Mine Life	years	7
Mineable Mineral Resource (LoM)	Mt	23.6
Au Grade Mined (LoM Average)	g/t Au	0.63
Ag Grade Mined (LoM Average)	g/t Ag	2.42
Peak Mill Throughput	tpa	730,000

Description	Unit	Value
Mill Feed Grade (LoM Average)	g/t Au	1.75
Peak Heap Leach Throughput	tpa	2,920,000
Heap Leach Feed Grade (LoM Average)	g/t Au	0.37
Recoveries	%	
Au Mill Argillite	%	95
Au Mill Volcanics	%	90
Au Heap Leach Argillite	%	75
Au Heap Leach Volcanics	%	75
Ag Mill Argillite	%	36
Ag Mill Volcanics	%	38
Ag Heap Leach Argillite	%	12
Ag Heap Leach Volcanics	%	16.5
Heap Leach Delay to Next Year	%	13
Au Payability (LoM average)	%	99.9
Ag Payability (LoM Average)	%	98
Total Au Produced (LoM)	koz	404.1
Total Ag Produced (LoM)	koz	354.5
Average Annual Au Production (LoM)	oz	57,201
Average Annual Au Production (first five years)	oz	63,893
Treatment Charge - Au	$/oz	2
Treatment Charge - Ag	$/oz	0
Royalty – Au and Ag (NSR)	%	1
Capital Cost Estimate		
Initial Capital (direct + indirect + contingencies)	US$ M	247
Sustaining Capital (LoM)	US$ M	27
Closure Costs	US$ M	12
LoM Operating Unit Costs		
Mining	US$ per tonne processed	9.67
Processing Mill	US$ per tonne processed	3.15
Processing Heap Leach	US$ per tonne processed	5.53
Dewatering	US$ per tonne processed	0.28
General & Administrative	US$ per tonne processed	1.34
Total OPEX	**US$ per tonne processed**	**19.97**
LoM Average All-in Sustaining Cost[1]	**US$/oz Au**	**1,269**

Notes:

1. All-in sustaining costs are a non-GAAP (Generally Accepted Accounting Principles) financial measure or ratio that have no standardized meaning under International Financial Reporting Standards Accounting Standards (IFRS) and may not be comparable to similar measures used by other issuers. As the Project is not in production, Viva does not have historical non-GAAP financial measures nor historical comparable measures under IFRS, and therefore the foregoing prospective non-GAAP financial measures or ratios may not be reconciled to the nearest comparable measures under IFRS.
2. oz = troy ounce, koz = kilotroy ounce (1000 ounces), M = million, Mt = million tonnes, tpa = tonnes per annum
3. Numbers may not sum precisely due to rounding

The cashflow analysis results in the following pre-tax and after-tax metrics in Table 1.5.

Table 1.5: Project Cash Flow Analysis Summary

Description	Unit	Pre-Tax	Post-Tax
NPV @ 5%	US$ M	138.6	111.6
IRR	%	20.6	17.6
Payback Period	Years	3.3	3.6

Notes: M = million

Sensitivity analyses were conducted, using the cashflow analysis in Item 22.4 as the base case, to assess the impact of changes in metal prices, CAPEX, and OPEX on the Project's NPV (5% discount rate) and IRR. The impact of each variable is examined individually with an interval of 40% and increments of 10% applied. It is to be noted that margin of error for cost estimates at the PEA study is typically -50% +50% while KCA states their estimate is +/-35% level of accuracy.

The post-tax results of the sensitivity analysis are shown in Figure 1.1, and Figure 1.2. The NPV of the project is most sensitive to changes in the metal prices and OPEX, while the IRR is most sensitive to changes in the metal prices and CAPEX.

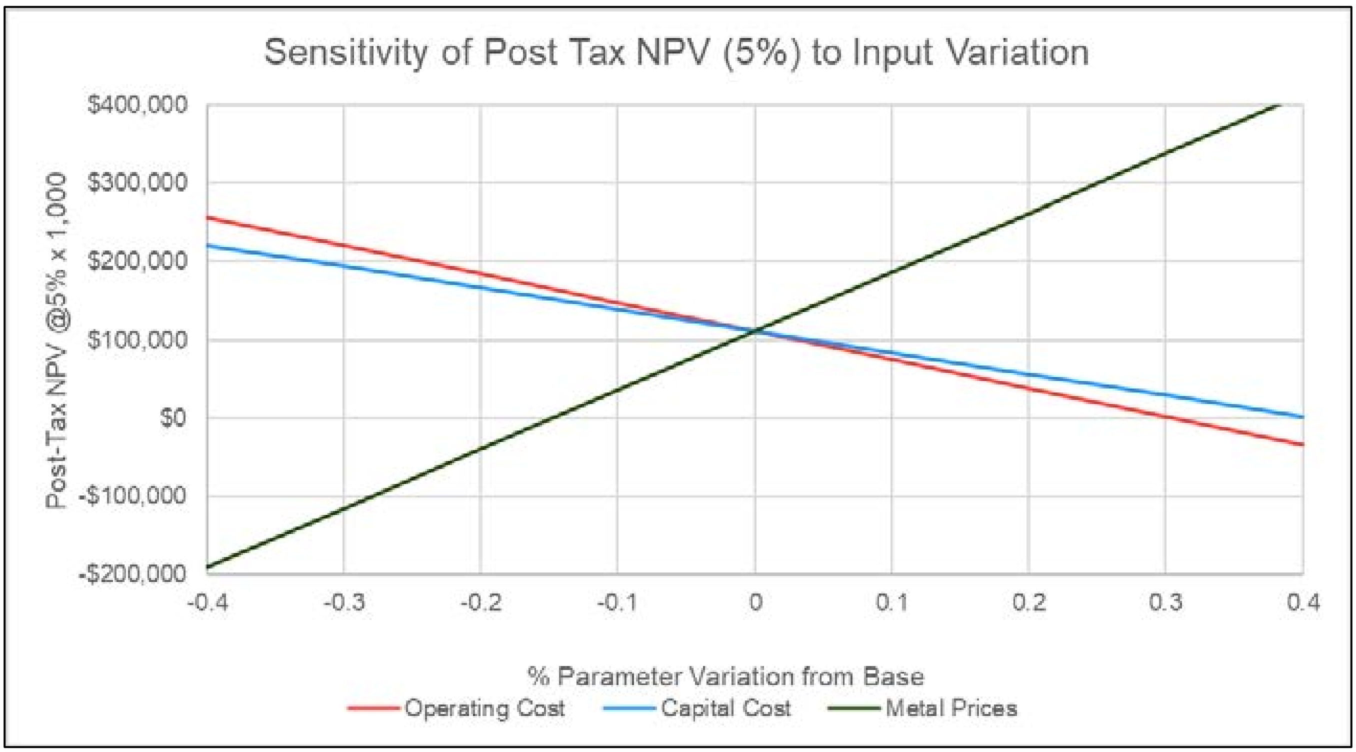


Figure 1.1: Post-Tax NPV (5%): Sensitivity to OPEX, CAPEX and Metal Prices

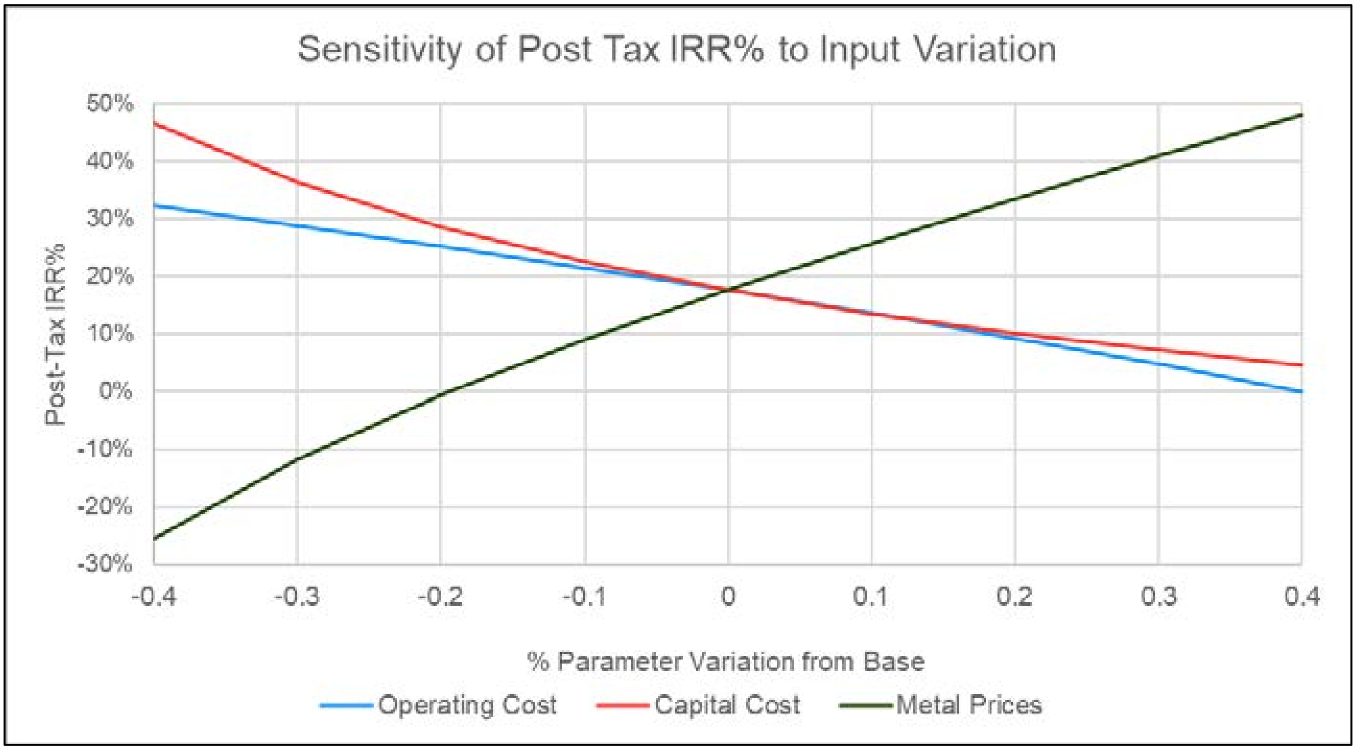


Figure 1.2: Post-Tax IRR (%): Sensitivity to OPEX, CAPEX, and Metal Prices

Table 1.6 shows the effect of a higher Au price on the project's Post-Tax NPV @ 5% and IRR results.

Table 1.6: Economic Metrics Sensitivity to Variations in Au Price

Au Price (US$)	Post-Tax NPV 5% (US$M)	Post-Tax IRR (%)
1,920	(38.4)	-0.4
2,160	36.7	9.0
2,400	111.6	17.6
2,640	186.5	25.7
2,880	261.3	33.3
3,120	336.1	40.7
3,360	411.0	47.9

1.11 Qualified Person’s Conclusions and Recommendations

1.11.1 Conclusions

1.11.1.1 Geology and Mineral Resources

The QP completed extensive data verification checks for the 2025 Tonopah Gold Mineral Resource estimate. The verification process included a 2-day personal inspection of the Project site to review geological procedures, chain of custody of DD core and RC chip samples and assay certificates, drill collar inspections and the collection of 60 sample pulp rejects for independent sample analysis.

Data verification also included a full independent build of the Viva assay database using original assay lab certificates where available, and a review of QA/QC performance for drilling completed by Viva.

It is the QP’s opinion that the analytical methods, and the QA/QC procedures used by Viva are consistent with industry standards and that the geological database and the assay data is of suitable quality to support the 2025 Mineral Resource estimate, as reported in Item 1.8.

The Tonopah Gold Project is defined by its complex geological setting, characterized as a low-sulphidation epithermal system that is further complicated by numerous interacting faults within an oblique-slip fault framework. Au and Ag mineralization is controlled by both lithological and structural features, resulting in two overlapping orientations. High-grade Au is typically concentrated in veins, breccia-veins, and mineralized structures, with visible Au often occurring along vein margins and associated with hematite. Occasionally, Au appears in a coarse form. The highest Au grades are found in the Discovery Zone—a well-explored area bounded by multiple faults—where significant mineralization is encountered just 30 m below the surface in both the Tertiary volcanic lower (TVL) and Ordovician Palmetto (OP) geological units. By contrast, the Rye Patch zone is primarily mineralized within the TVL unit, which exhibits Au at both shallow depths and beyond 50 m.

Recent advancements include an updated geological model and Mineral Resource estimate (MRE) that incorporate data from 59 new drill holes completed since 2022. This increased drill coverage has improved spatial resolution, enabling delineation of both high-grade and non-mineralized zones. The development of a 3D CSAMT (Controlled Source Audio-frequency Magnetotelluric) model has strengthened the Project's structural interpretation, providing more precise definitions of mineralization boundaries and estimation domains. These ongoing efforts suggest significant potential for discovering new mineralized zones, as evidenced by recent exploration successes during the 2022, 2023, and 2024 drilling campaigns.

The latest MRE has upgraded parts of the resource from Inferred to Indicated, reflecting greater confidence in the geological interpretation due to improved data density. Nevertheless, the MRE remains highly sensitive to the modelling methodology and interpretative domain boundaries; as a result, future revisions in these domains could lead to substantial changes in overall resource estimates. Further refinement of the structural interpretation, as well as the possible integration of an alteration model, are recommended to enhance confidence in these boundaries. Additionally, the management of statistical outliers—especially in areas of low drill density within the Inferred category—remains a significant factor. Increasing the density of drilling would further boost reliability and may allow for additional resources to be reclassified into higher-confidence categories.

The Tonopah Gold Project continues to demonstrate strong exploration upside. Ongoing infill and step-out drilling are expected to further increase resource confidence and may lead to the identification of new zones of Au and Ag mineralization in the future.

1.11.1.2 Mining

The Mineral Resource block model was adapted for use in GEOVIA Whittle™ (Whittle) software by adding an additional categorical field, WCODE, that grouped blocks according to lithology and resource classification. Both mineralized and non-mineralized blocks were coded, enabling clear differentiation between potential feed and waste material. Geotechnical domains were used to define an Inter-ramp Slope Angle (ISA) attribute that were used for more precise slope design for the pit.

The pit optimization for the Tonopah Project used Whittle to design a three-phase mining plan that maximizes economic value. The ultimate pit reaches approximately 228 m in depth, with a mine life of seven years. The selected final pit shell, corresponding to revenue factor (RF) 0.98, is used for the ultimate pit design, and calculations and production scheduling for both heap leach and mill options. Pit slope parameters for the open pit design were derived from ISA's coded in the block model. The ISA's applied during pit optimization and detailed pit design ranged primarily from 35° to 46°, based on updated geotechnical domain models and supported by further details in Items 14.0 and 16.0.

The final pit contains 23.6 Mt of material, with an average Au grade of 0.63 g/t and a strip ratio of 3.92. Mineralized material will be processed through both a mill (2,000 tpd) and a heap leach facility (8,000 tpd), using COGs of 0.18 g/t for heap leach and 0.28 g/t for mill feed. Waste storage is planned both on the surface and, for about 16% of the total, within the pit. The mine plan is sequenced to prioritize high-grade material and minimize waste movement, with processing capacities and costs based on 2025 estimates. Au recovery rates are 92% for the mill and 75% for heap leach, supporting a flexible and efficient operation. The Tonopah Project's open pit mine plan is technically viable and economically reasonable under current market conditions. It is designed with flexibility, operational safety, and resource optimization.

1.11.1.3 Metallurgy and Processing

The Tonopah Gold Project proposes a dual recovery process for Au and Ag, combining heap leaching and milling with a CIL plant to treat an average of 10,000 tpd. Low-grade mineralized material will be processed on a permanent heap leach pad using NaCN, with Au adsorbed onto carbon and recirculated, while high-grade mineralized material will be milled, gravity-concentrated, and leached in a CIL circuit. Both circuits will send loaded carbon off-site for toll processing, ensuring efficient recovery of precious metals.

Test results indicate that the project's mineralized material is suitable for cyanide leaching and gravity concentration. Heap leach recoveries for Au correlate strongly with head grade, with higher-grade material showing lower recoveries due to coarse Au and reduced leach kinetics, while milling recoveries are less grade-dependent. Estimated recoveries stand at 75% for Au and 24% for Ag from heap leaching, and between 90–95% for Au and 36–38% for Ag in the mill circuit, depending on mineralized material type.

1.11.1.4 Environmental

Environmental and permitting matters related to the Project are very similar to those effectively managed by other operators at multiple surface mining projects on public and private lands in northern Nevada and authorized by Federal, State and local regulatory authorities.

Reasonably foreseeable risks may include, but not be limited to:

- Lengthy permitting schedules resulting from lessened availability of consulting services and regulatory agency staffing and technical expertise.

- Challenges by Non-Governmental Organizations (NGOs) and interested parties to technical data and agency decisions regarding management of hydrologic, biologic and cultural resources at the Project.

No operational or post-closure chemical treatment of dewatering water or post-mining pit lake water quality is currently anticipated; however, this is based on a limited data set and must be confirmed in later studies. Water management strategies will be designed to comply with applicable water quality standards and to protect groundwater resources. A waste rock management plan approved by the agencies will present methods to isolate and protect groundwaters of the State when Potentially Acid Generating (PAG) material is encountered.

1.11.1.5 Capital and Operating Cost Estimates

The estimated capital and operating costs are reasonable for a PEA level analysis.

1.11.1.6 Economic Analysis

Based on the available information, the project has an after-tax NPV of US$111.6M at a discount rate of 5%, an IRR of 17.6%, and a payback of 3.6 years. The sensitivity analysis indicates that the Project economics are most sensitive to the Au price with the breakeven Au price 20% below the base case of US$2,400/oz.

1.11.2 Recommendations

1.11.2.1 Geology and Mineral Resource Estimate

The QP has the following recommendations for Viva to consider regarding data collection:

- Drill hole collars should be surveyed with a differential GPS to improve accuracy and marked with a permanent ground marker that will survive environmental conditions. An example would be a concrete cap or monument to mark the collar, with the drill hole ID written either in the concrete or written on a metal tag attached to the monument. The use of a handheld GPS is acceptable for exploration stage projects but requires increased level of accuracy for PEA and above level studies.

- An increase in the number of DD holes vs. RC holes is recommended for improved data quality and understanding of the deposit. DD holes allow the collection of bulk density measurements, rock quality information, fault/geotechnical parameters locations and angles, and alteration zones, that are not easily (if at all) available from the collection of RC drilling samples.

- Additional bulk density measurements should be taken on future DD drilling, including intervals of known waste rock. Suitable QA/QC procedures should be included as part of the bulk density measurements, and the use of a waterproof coating should be used, especially in fractured rock. If possible, bulk density measurements should be taken on historical DD core, spaced evenly throughout the main zone of the current resource pit shell extents.

- Field duplicate samples should be collected for all future DD drilling to evaluate the accuracy of the sampling at core splitting stage. Viva should also collect several duplicate RC samples to test the variability between samples of the same interval using this method of drilling.

- Down-hole deviation measurements for all drill holes, as well as core recovery, and rock quality designation (RQD) for DD holes should continue to be collected on all future drilling programs.

- Regular check sampling by a third-party (umpire) analytical laboratory should be conducted.

- As recommended by the CIM MRMR Best Practice Guidelines, drill logs and sample results should be stored in a relational database that provides proper control and security. The database should contain all relevant data for each drill hole, including, but not limited tom drill hole ID, collar location and orientation, total depth, down-hole deviation measurements, hole diameter, geological data (lithology, alteration, core recovery, RQD), and analytical data (unique sample ID, analytical laboratory name, analytical certificate number, assay results, including any trace or deleterious elements, geometallurgical results, bulk density, QA/QC data). A Microsoft Access database would be suitable for this purpose.

For all future DD core programs, WSP recommends obtaining additional data such as bulk density measurements, core recovery, RQD, and the location and angles of major faults. Additional bulk density measurements, including those from intervals of known waste rock, will improve the current specific gravity (SG) and tonnage estimate for the deposit. Further geotechnical data will refine the existing structural interpretation and its effect on Au mineralization. Developing an alteration model could improve understanding of its impact on Au mineralization and potentially identify new drill targets.

All these recommendations would fall under the next phase of the Project as part of a Pre-Feasibility Study (PFS). The estimated costs are approximately US$300,000.

1.11.2.2 Mining

A detailed trade-off study comparing the leasing versus purchasing of production equipment, including the potential for a hybrid approach, should be undertaken. This study would highlight opportunities to reduce initial capital requirements and assess the impact on overall operating costs.

A more detailed mining phase plan for open pit mining at the PFS level should be developed. Given the nature of spatial grade distribution and strip ratio of the deposit, detailed phasing would allow for bringing more high-grade material earlier in the mine life, allowing for enhanced capital cost recovery.

The development of geotechnical slope design domains should be advanced through continued DD drilling and geological modelling. This approach would help maintain consistent slope stability throughout the mine’s operational life and minimize the risk of deviations.

A comprehensive hydrological assessment should be undertaken to identify potential water inflows and drainage concerns. The study is essential to mitigate operational disruptions and ensure long-term pit stability.

The expected cost for next stage of trade-off study and PFS is approximately US$3.0M, which is exclusive of the expenses associated with any field programs described in other sub-items. By undertaking detailed and multidisciplinary assessment, stakeholders will be able to identify risks, optimize design parameters, and establish a solid foundation for the next stages of project development.

1.11.2.3 Metallurgy and Processing

Results from metallurgical test work suggests that there is a possibility of improved recoveries with longer leaching due to potential coarse Au. The results also suggest that similar heap leach recoveries may be achievable at coarser crush sizes. As part of future test programs, variability testing at coarser product sizes and longer leach cycles should be evaluated to determine whether further process optimization is possible to improve project economics.

Toll processing of carbon is considered for the project without any formal contract terms in place. Due diligence efforts were made to ensure reasonable numbers; however, should there be any significant shifts in the market this could result in increased prices or delays in Au and Ag production. As part of future work, Viva should engage with multiple toll stripping groups to ensure production capacity and costs.

KCA recommends additional metallurgical studies including:

- High-grade mill and gravity variability testing
- Variability column testing at various crush sizes (9.5 mm, 12.5 mm, 25 mm and 38 mm) for a 120 to 180-day period.
- Additional characterization work.

Samples for KCA’s metallurgical program may be captured in a future DD core program. The cost of a 1,000 m PQ drill program including assay, televiewer/oriented core study is approximately US$500,000 not including additional cost for SG testing. The estimated cost of the metallurgical test work is US$475,000. Quotations are pending for an updated geotechnical study.

1.11.2.4 Environmental

As part of the specific work plan, ongoing long-lead baseline studies will be continued to support environmental permitting, mine development, and eventual closure activities. These studies will include:

- Environmental monitoring, including updated hydrogeologic investigations;
- Cultural resources surveys, with a focus on areas not surveyed or resurveyed within the past ten years;[1]
- Biological studies, including updated habitat and species evaluations;
- Hydrogeologic studies, to inform the development of a site-specific numerical groundwater model.

The work plan will also include the installation of three groundwater monitoring wells: one upgradient replacement well and two downgradient wells. In support of the groundwater model and permitting requirements, two 30-day aquifer tests will be conducted – one from the existing bedrock production/monitoring well and one from the existing alluvial production/monitoring well.

A soil infiltration testing program should be implemented to evaluate the capacity of alluvial soils to accept excess mine dewatering water without adversely impacting groundwater quality, in accordance with state water protection standards.

As the proposed pit design extends below the regional water table, it is recommended that the PFS and Feasibility Study (FS) phases include further hydrogeologic investigations. These studies should evaluate potential impacts to groundwater quantity and quality, assess the suitability of pit water for wildlife, and determine whether pit lake formation could affect other ecological receptors. Additionally, the potential for hydraulic connectivity to nearby water users, streams, or springs should be assessed to ensure that mining activities do not adversely impact existing water rights or natural hydrologic features.

To support accurate design and permitting of the dewatering system, it is recommended that a longer-duration and more robust pumping test be conducted in the vicinity of the proposed dewatering zone. This test should be

[1] Assumes that survey standards at the time of the original work are still current and no significant changes (e.g., new discoveries, land use changes, or updated regulations) have occurred in the area since the survey was completed.

designed to improve predictions of groundwater inflow rates and to characterize the quality of the dewatering water. If water quality results indicate that discharge would not meet applicable groundwater discharge standards, treatment will be required prior to discharge into Rapid Infiltration Basins (RIBs).

In parallel, a pilot-scale infiltration basin test is recommended, coupled with a calibrated groundwater flow model, to evaluate the infiltration capacity and long-term sustainability of the proposed discharge approach. While pumping and discharge to RIBs is generally considered a sustainable and effective water management strategy, its success depends on accurate site-specific hydrogeologic data, proper operation of infiltration basins, and discharge water quality. These investigations are typically conducted during the PFS or FS stages to inform engineering design, permitting, and environmental impact assessments.

Additional predictive modelling may be required by regulatory agencies to demonstrate that discharge of dewatering water to the RIBs will not adversely affect groundwater resources. As outlined in Item 20.0, a regulatory precedent exists for similar operations; however, for water quality that does not meet standards or if agencies determine that site-specific factors such as local hydrogeology, background water quality, and surrounding land use warrant further analysis, agencies may require additional analysis to confirm compliance and protect water quality.

For a PEA many studies utilize existing data and information. Updated studies, such as a PFS or FS, will need to meet current expectations for data quality, modelling transparency, or regulatory rigor. New guidance (e.g., Nevada Division of Environmental Protection's (NDEP) 2021 geochemical modelling guidance) emphasizes integration with groundwater flow models, which may not have been standard practice in earlier studies. The Nevada Modified Sobek Procedure has also evolved since 2012. Studies should include comprehensive baseline water quality or hydrologic data, which are now required to support predictive modelling of pit lakes and waste rock impacts. Future detailed investigations should include geochemical models that rely on site-specific mineralogy, hydrology, or climate conditions.

A Class III cultural resources survey will be completed for all unsurveyed or outdated areas within the projected Project boundary, consistent with federal and state requirements.

To ensure compliance with the Bald and Golden Eagle Protection Act, a two-year aerial survey program targeting Golden Eagles and other raptors will be conducted. The results will inform mitigation planning and permitting, if necessary.

The Nevada-approved Standardized Reclamation Cost Estimator (SRCE) Version 1.4.1 Build 017b (Revised May 16, 2019), along with the SRCE_Cost_Data_File_1_12_Std_2024.xlsm, was used to estimate the preliminary reclamation bond for the Project based on operational and closure assumptions outlined in this PEA. For future studies and more refined planning, it is recommended that the most current version of the SRCE model and associated cost data files be utilized. This will allow alignment with evolving regulatory expectations, cost structures, and best practices in reclamation planning.

Table 1.7 presents the estimated environmental and permitting costs to move the project into the next phases of a PFS. These estimates costs are provided at an order-of-magnitude level of accuracy and are intended for preliminary planning purposes.

Table 1.7: Estimated PFS Environmental and Permitting Costs

Category	Estimated Cost (US$)	Notes
Hydrogeology Studies	$500,000	Excludes capital cost of wells construction
Cultural Resources Survey	$200,000	Areas not surveyed, or surveyed >10 years ago
Raptor Surveys	$100,000	2 years Golden Eagle & Raptor surveys
General Consulting	$100,000	Environmental permitting support
Heap Leach Pad	$250,000	Studies to support the material characterization and stability analysis for the heap leach pad
Environmental Total	**$1,150,000**	

1.11.2.5 Costs and Financials

For the advancement of the Project, specific cost workups based on multi sourced quotations for equipment, construction materials, consumables, pre-stripping contracts, are a prerequisite. With fast-track project execution aspirations to advantage the metal markets then the multi sourced quotes would benefit from Front-End Engineering Design (FEED) leading to firm tender bids for these recommended quotes thereby allowing equipment procurement immediately upon approval of funding.

Contracts for market level agreements, power supply, site preparation, fuel and lubricants, maintenance agreements need consideration of advancement.

1.11.2.6 Budget for Recommended Work

Table 1.8 provides the estimated budget for recommended work.

Table 1.8: Budget for Recommended Work

Item	Category	Estimated Cost (US$)
1.0	Geology and Mineral Resources	$300,000
2.0	Metallurgy and Processing	$1,000,000
3.0	Environmental Studies, Permitting, Social or Community Impact and Government Relations	$1,150,000
4.0	Engineering and Field Work to Complete PFS and Reporting	$3,000,000
	Total	**$5,450,000**

2.0 Introduction

The Tonopah Gold Project is a near surface low-sulphidation epithermal gold system located northeast of the town of Tonopah, Nevada, USA. Viva Gold Corp. (Viva) owns a 100% interest in the Project and is publicly traded on the TSX Venture Exchange (TSX-V) “VAU”.

WSP Canada Inc. (WSP), Kappes, Cassiday & Associates (KCA), and Lewis Environmental Consulting LLC (LEC) were retained by Viva to prepare an updated Mineral Resource estimate (MRE) and Preliminary Economic Assessment (PEA) Technical Report prepared in accordance with Canadian Securities Administrator’s National Instrument (NI) 43-101. The MRE and Technical Report were prepared by WSP, in conjunction with KCA for the metallurgy, recovery, and infrastructure components and LEC for the environmental components of the study. This Technical Report follows the requirements of Form 43-101F.

The purpose of this Technical Report update is to support the disclosure of a material change to the MRE based on new drilling completed since the 2022 PEA (effective date: January 1, 2022), as well as updated mine design based on additional metallurgical and geotechnical studies that have been completed. Previous MREs for the project have only reported gold (Au) resources, the updated 2025 MRE reports silver (Ag) resources for the first time.

The effective date of the MRE presented in this Technical Report is June 13, 2025. The MRE is stated per the definitions and guidance provided in the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards on Mineral Resources and Reserves (CIMDS), adopted May 10, 2014.

2.1 Sources of Information

The primary sources of information for this Technical Report and the MRE documented within are the data and observations collected by Viva (and its predecessors) personnel during various exploration campaigns on the Project property between 1988 and 2023. In addition to the drilling performed by Viva and its predecessors, historical drilling data performed by third party entities was also used for the geology and resource models.

All Project-specific data, observations, and reports, including third-party consultant technical reports for the Project area, were provided by Viva. Specific references are cited through the Technical Report and listed in Item 27.0.

2.2 Language, Currency, and Measurement Status

This Technical Report uses Canadian English spelling and metric units, unless otherwise stated. Currencies outlined in the report are in United States Dollars (US$), unless otherwise stated.

Unless otherwise indicated, coordinates in this Technical Report are presented in metric units using the Universal Transverse Mercator (UTM) Zone 11 North projection, and the North American Datum of 1983 (NAD83). In instances where data has been provided in other units or coordinate projections, the data has been converted to metric units and UTM Zone 11N, NAD83 coordinates using in-built transformations in commercially available Geographic Information System (GIS) software. Elevations are reported as metres above mean sea level (amsl).

Au and Ag grades are presented in parts per million (PPM) or its equivalent, grams per tonne (g/t). Metal content is presented in troy ounces (oz).

Table 2.1: Key Acronyms and Definitions

Abbreviation	Description
.csv	Comma-separated value (file format)
.pdf	Portable Document Format
°C	Degrees Celsius
°F	Degrees Fahrenheit
µm	Micron (micrometre)
2D	Two-Dimensional
3D	Three-Dimensional
AACE	Association for the Advancement of Cost Estimating
AAL	American Assay Laboratories Inc.
AAS	Atomic Absorption Spectroscopy
ABA	Acid-base accounting
Ag	Silver
AGP	Acid Generating Potential
ALS	ALS Chemex
amsl	Above mean sea level
ANP	Acid Neutralizing Potential
APE	Area of Potential Effect
ARD	Acid Rock Drainage
Au	Gold
BAPC	Bureau of Air Pollution Control
BFA	Bench Face Angle
bgs	Below ground surface
BLM	Bureau of Land Management
BMRR	Bureau of Mining Regulation and Reclamation
BNAF	Baseline Data Needs Assessment Form
BSDW	Bureau of Safe Drinking Water
BWPC	Bureau of Water Pollution Control
CAD	Canadian Dollar
CaO	Calcium oxide (quicklime)
CAPEX	Capital Expenditure
CAT	Caterpillar Inc.
CDA	Coeur d'Alene Mines Corporation
CIC	Carbon-in-Column
CIL	Carbon-in-Leach
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIMDS	Canadian Institute of Mining, Metallurgy and Petroleum Definition Standards
cm	Centimetre
CNI	Call & Nicholas, Inc.
COG	Cut-Off Grade
CR	County Road
CRM	Certified Reference Material
CSAMT	Controlled Source Audio-Frequency Magnetotelluric
CV	Coefficient of Variation
DCF	Discounted cash flow
DD	Diamond Drill
DL	Detection Limit
DR	Decision Record
DSM	Digital Surface Model
EA	Environmental Assessment
EDA	Exploratory Data Analysis
EM	Electromagnetic
EPA	Environmental Protection Agency
EPCM	Engineering, Procurement, and Construction Management
ESG	Environmental, Social, and Governance
ExPoO	Exploration Plan of Operations
FA	Fire Assay
FEED	Front-End Engineering Design
FONSI	Finding of No Significant Impact
FS	Feasibility Study
ft	Foot/feet
g	Gram
G&A	General and Administrative
g/t	Grams per metric tonne
GAAP	Generally Accepted Accounting Principles
GCL	Geosynthetic Clay Liner
GeoTIFF	Geographic Tagged Image File Format
GHG	Greenhouse Gas
GIS	Geographic Information System
gpm	Gallons per minute
GPS	Global Positioning System
h	Hour
HAP	Hazardous Air Pollutant
HARD	Half Absolute Relative Difference
HCT	Humidity Cell Test
HDPE	High-Density Polyethylene
hp	Horsepower
HPGR	High Pressure Grinding Roll
HQ	HQ-size core (standardized drill core diameter, 63.5 mm)
HRD	Half Relative Difference
Hz	Hertz
IBA	ICE Benchmark Administration
ICP	Inductively Coupled Plasma
ICP-AES	Inductively Coupled Plasma Atomic Emission Spectroscopy
ICP-OES	Inductively Coupled Plasma Optical Emission Spectroscopy
ID	Identification
ID2	Inverse Distance Squared
ID3	Inverse Distance Cubed
IFRS	International Financial Reporting Standards
IP	Induced Polarization
IRA	Inter-Ramp Angle
IRR	Internal Rate of Return
ISA	Inter-ramp Slope Angle
JV	Joint Venture
K	Thousand
KCA	Kappes, Cassiday & Associates
kg	Kilogram
kg/t	Kilograms per tonne
km	Kilometre
koz	Kilotroy ounces (1000 troy ounces)
kt	Kilotonne (1000 metric tonnes)
kV	Kilovolt
kW	Kilowatts
kWh	Kilowatt-hour
L	Litre
lb	Pound
LEC	Lewis Environmental Consulting LLC
LLDPE	Linear Low-Density Polyethylene
LMBA	London Bullion Market Association
LoM	Life-of-Mine

Abbreviation	Description
LoMP	Life-of-Mine Plan
LPG	Liquefied Petroleum Gas
M	Million
m	Metre
m^2	Square metre
m^3	Cubic metre
Ma	Megaannum (Million years ago)
MCC	Motor Control Center
MDA	Mine Development Associates
MDB&M	Mount Diablo Base and Meridian
MEG	Moment Exploration Geochemistry
mg	Milligram
MGC	MGC Resources Inc.
MLI	McClelland Laboratories Inc.
mm	Millimetre
MPoO	Mine Plan of Operations
MRMR	Mineral Resources and Mineral Reserves
MSHA	Mine Safety and Health Administration
MW	Megawatt
MWMP	Meteoric Water Mobility Procedure
NAAQS	National Ambient Air Quality Standards
NAC	Nevada Administrative Code
NaCN	Sodium cyanide
NAD27	North American Datum 1927
NAD83	North American Datum 1983
NAG	Net Acid Generation
NDEP	Nevada Division of Environmental Protection
NDOT	Nevada Department of Transportation
NDOW	Nevada Department of Wildlife
NDWR	Nevada Division of Water Resources
NEPA	National Environmental Policy Act
NGO	Non-Governmental Organization
NI 43-101	National Instrument 43-101 (Canadian disclosure standards for mineral projects)
NN	Nearest Neighbour
NOAA	National Oceanic and Atmospheric Administration
NOI	Notice of Intent
NP:AP	Neutralization Potential to Acid Potential ratio
NPV	Net Present Value
NRHP	National Register of Historic Places
NRS	Nevada Revised Statutes
NRV	Nevada Reference Value
NSR	Net Smelter Return
NTP	Notice to Proceed
NV	Nevada
O&M	Operations and Maintenance
OK	Ordinary Kriging
OP	Open pit
Op	Ordovician Palmetto Formation
Opa	Ordovician Palmetto Formation argillite
OPEX	Operating Expenditure
opt	Troy ounces per short ton
OREAS	Ore Research & Exploration Assay Standards
oz	Ounce
PA	Programmatic Agreement
PAG	Potentially Acid Generating
PCPE	Perforated Corrugated Polyethylene

Abbreviation	Description
PEA	Preliminary Economic Assessment
PFS	Pre-Feasibility Study
pH	Potential of Hydrogen (acidity/alkalinity)
PLC	Programmable Logic Controller
PPM	Parts per million
PQ	PQ-size core (standardized drill core diameter, 85.0 mm)
PVC	Polyvinyl Chloride
QA/QC	Quality Assurance/Quality Control
QAL	Quaternary Alluvium
QP	Qualified Person
RA	Regulatory Authorities
RBF	Radial Basis Function
RC	Reverse circulation
RDi	Resource Development Inc.
Rex	Rex Exploration Corp.
RF	Revenue Factor
RIB	Rapid Infiltration Basin
ROM	Run-of-Mine
ROW	Right-of-Way
RQD	Rock Quality Designation
SAD	Surface Area Disturbance Permit
SDR	Standard Dimension Ratio
SEM	Scanning Electron Microscope
SG	Specific Gravity
SHPO	State Historic Preservation Office
SLS	Solid to Liquid System
SR	State Route
SRCE	Standardized Reclamation Cost Estimator
SRHA	Stock Raising Homestead Act
SRM	Standard Reference Material
StDev	Standard Deviation
SU	Standard Units
SWS	Schlumberger Water Services
t	Metric tonne
TD	Total depth
TEM	Time-domain Electromagnetics
tpa	Tonnes per annum (year)
tpd	Tonnes per day
tph	Tonnes per hour
TPU	Tonopah Public Utilities
TSF	Tailing Storage Facility
TVL	Tertiary Volcanics Lower
TVU	Tertiary Volcanics Upper
UAV	Unmanned Aerial Vehicle
US	United States
US$	United States Dollar
USGS	United States Geological Survey
UTM	Universal Transverse Mercator
V	Volt
Viva	Viva Gold Corp.
WMCI	Water Management Consultants, Inc.
WPCP	Water Pollution Control Permit
WRMP	Waste Rock Management Plan
WRSF	Waste Rock Storage Facility
WSP	WSP Canada, Inc.
XRD	X-ray Diffraction

2.3 Personal Inspection Summary

As part of the data and methodology verification process, Ms. Róisín Kerr, P.Geo., WSP QP for data verification and Mr. Jordi Bascompte-Vaquero, mining engineer, conducted a site visit to the Tonopah property and offices on June 6 and 7, 2023. The purpose of the site visit was to allow the QP to observe key aspects of the Project site, including deposit geology, current and previous exploration programs, and site infrastructure. Mr. James Hesketh, President and CEO of Viva, and Mr. Edward Bryant, Viva's contract geologist, were available for discussion and verification of current and historical methods and results and to discuss any concerns and recommendations.

Activities performed during the site visit included the following:

- Verify a selection of drill hole collar coordinates.
- Observe and review the drilling, logging, and sampling procedure.
- Observe the core storage facility.
- Select a representative suite of samples for replicate analytical comparison.

As part of the mineral processing and infrastructure verification process, Mr. Caleb Cook, P.E., KCA QP for mineral processing and metallurgical testing, recovery methods, and project infrastructure, visited the Tonopah Project site and offices on May 8, 2025. During the visit, Mr. Cook visited the proposed heap leach and other processing facilities locations to evaluate the general site conditions, reviewed utilities and infrastructure near the project site and looked at drill core. Mr. Cook was accompanied by Mr. James Hesketh, President and CEO of Viva, to discuss any questions or concerns.

3.0 Reliance on Other Experts

In this Technical Report and as described in this Item, the QPs relied on: a) a report, opinion, statement, of another expert who is not a QP, or on information provided by the issuer concerning legal, political, environmental, or tax matters relevant to the technical report; or b) a report, opinion, or statement of another expert who is not a QP concerning the pricing of commodities for which pricing is not publicly available.

This Technical Report has been prepared by WSP and KCA for Viva. The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to WSP and KCA at the time of report preparation;
- Assumptions, conditions, and qualifications as set forth in this report; and
- Data, reports, and other information supplied by Viva and other third-party sources.

In Items 4.2 Property Ownership, 4.3 Permitting, 4.4 Royalties, Encumbrances, and Other Obligations, 4.5 Environmental Liabilities, and 4.6 Significant Risk Factors of this Technical Report, the QPs have relied upon, and believe there is a reasonable basis for this reliance on, information provided by Viva regarding mineral tenure, surface rights, ownership details, and agreements relating to the Tonopah Gold Project, royalties, environmental obligations, permitting requirements and applicable legislation relevant to the Tonopah Gold Project. The QPs have not independently verified the information in these items and have fully relied upon, and disclaim responsibility for, information provided by Viva in these Items.

Information related to the environmental studies, permitting, and social or community impact in Item 20 of this Technical Report were provided by Viva, LEC and Piteau Associates (Piteau). The QPs have not independently verified the permitting, environmental, or reclamation and closure cost analyses. The QPs have fully relied upon, and disclaim responsibility for, information provided by Viva and LEC.

Information related to capital cost estimate (CAPEX) and operating cost estimate (OPEX) in this Technical Report were provided by Viva and KCA, and third-party experts, Piteau and LEC, for inclusion in the cost estimation for the Project by WSP. This information included Owner's Costs and OPEX data (Viva), dewatering CAPEX and OPEX; initial and sustaining costs (Piteau) and bond premium and closure costs (LEC). WSP also relied upon Viva for information related to Item 19.0 Market Studies and Contracts. The QPs have not independently verified the information for these costs and have fully relied upon, and disclaim responsibility for, information provided by Viva, KCA, Piteau, and LEC.

Information related to Economic Analysis in this Technical Report was provided by KCA, for inclusion in the cashflow established for the Project by WSP. This information included metal recoveries of the various metallurgies at the site from mill and heap leach processing of the WSP defined mined materials that was used to establish the revenues realized from the production plan as presented.

4.0 Property Description and Location

4.1 Location and Area

The Tonopah property encompasses 4,092 hectares (10,112 acres) in the Ralston Valley, on the northeast side of the San Antonio Mountains in central Nevada, located approximately 30 kilometres (km) northeast of the town of Tonopah in Nye County (Figure 4.1).

The Project site can be found on the United States Geological Survey (USGS) Henry's Well and Thunder Mountain 1:24,000 scale, 7.5-minute series, topographic quadrangle maps. The geographic center of the property is centered on UTM Zone 11N, NAD83 493,880 metres (m) east and 4,235,070 m north (38°16'N latitude and 117°04'W longitude). Access to the site is provided by State Route (SR) 376, which intersects Nye CR 82 (Belmont Road) near the center of the property.

4.2 Mineral Tenure, Surface, and Other Rights

The Tonopah Project mining claims are in Sections 16 to 21, 28 to 33 of Township 5 North, Range 44 East (T5NR44E); and Sections 4 and 5 of Township 4 North, Range 44 East (T4NR44E), Mount Diablo Base and Meridian (MDB&M) as illustrated on Figure 4.2. Some claims are also found in Township 5 North, Range 43 East Sections 13, 24 and 25 (T5NR43E).

The Project consists of 508 unpatented lode claims (including 184 royalty claims) covering an area of approximately 4,092 hectares (10,112 acres). All claims are 100% controlled by Viva; copies of the individual claim notices and location maps are on file with the Bureau of Land Management (BLM) Nevada State Office in Reno, Nevada, and with the Nye County Recorder's office in Tonopah, Nevada. The list of claims is shown in Table 4.1.

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

An annual maintenance fee on unpatented claims of US$200 per claim must be paid to the BLM by September 1 at 12 noon each year. A County proof of labour fee of US$12.00 per claim is also assessable on filing of the Federal annual maintenance fees. As of the effective date of this report, Viva is current on all assessment fees.

The surface rights of the unpatented claims located in all Sections, with the exception of Section 32, T5N R44E, are managed by the BLM. Those surface rights located in Section 32 are on lands under private ownership through the Stock Raising Homestead Act (SRHA) of 1916. This land was transferred to private ownership under SRHA to allow ranchers to privatize lands deemed to be of no value except for livestock grazing and the growing of forage. The federal government retained the subsurface mineral rights, where the right to surface access is granted subject to various conditions under the1872 Mining Law. Viva controls the mineral rights underlying Section 32 as unpatented mining claims. The BLM expects good faith negotiations with the landowners for activities conducted on their surface rights. Twenty-three unpatented claims were re-located on this section in 2025 after filing and serving required notice to surface land holders.

LEGEND

- PROJECT CLAIM BOUNDARY
- STATE BOUNDARIES
- COUNTY BOUNDARIES
- TOWN OF TONOPAH
- MAJOR CITY
- TOWN
- FEDERAL INTERSTATE
- U.S. HIGHWAY
- STATE HIGHWAY
- SECONDARY ROADS

0 50 100

1:1,500,000 KILOMETRES

NOTE(S)

REFERENCE(S)
NEVADA DEPARTMENT OF TRANSPORTATION
NEVADA BUREAU OF MINES AND GEOLOGY
PROJECTED COORDINATE SYSTEM: NAD 1983 UTM ZONE 11N

CLIENT
VIVA GOLD CORP.

PROJECT
TONOPAH GOLD PROJECT

TITLE
PROJECT LOCATION

CONSULTANT

YYYY-MM-DD	2025-08-07
DESIGNED	RK
PREPARED	RK
REVIEWED	JS
APPROVED	JS

PROJECT NO.	CONTROL	REV.	FIGURE
CA0004670.0343		0	4.1

Table 4.1: Mineral Tenure Rights

Claim Name	Serial Number	Acres	Hectares	Date Of Location	Next Payment Due Date	Meridian Township Range Section	Royalty
MW 1	NV106322800	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 013	N
MW 2	NV106322801	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 018	N
MW 3	NV106322802	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 018	N
MW 4	NV106322803	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 018	N
MW 5	NV106322804	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 018	N
MW 6	NV106322805	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 018	N
MW 7	NV106322806	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 018	N
MW 8	NV106322807	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 018	N
MW 9	NV106322808	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 018	N
MW 10	NV106322809	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 018	N
MW 11	NV106322810	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 018	N
MW 12	NV106322811	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 018	N
MW 13	NV106322812	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 013	N
MW 14	NV106322813	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 018	N
MW 15	NV106322814	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 013	N
MW 16	NV106322815	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 018	N
MW 17	NV106322816	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 013	N
MW 18	NV106322817	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 019	N
MW 19	NV106322818	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 024	N
MW 20	NV106322819	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 019	N
MW 21	NV106322820	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 024	N
MW 22	NV106322821	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 019	N
MW 23	NV106322822	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 024	N
MW 24	NV106322823	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 019	N
MW 25	NV106322824	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0430E 024	**Y**
MW 26	NV106322825	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 019	**Y**
MW 27	NV106322826	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 019	**Y**
MW 28	NV106322827	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 019	**Y**
MW 29	NV106322828	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 018	N
MW 30	NV106322829	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 018	N
MW 31	NV106322830	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 018	N
MW 32	NV106322831	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 018	N
MW 33	NV106322832	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 018	N
MW 34	NV106322833	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 018	N
MW 35	NV106322834	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 018	N
MW 36	NV106322835	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 018	N
MW 37	NV106322836	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 018	N
MW 38	NV106322837	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 017	N
MW 39	NV106322838	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 018	N
MW 40	NV106322839	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 017	N
MW 41	NV106322840	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 018	N
MW 42	NV106322841	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 017	N
MW 43	NV106322842	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 018	N
MW 44	NV106322843	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 017	N
MW 45	NV106322844	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 018	N
MW 46	NV106322845	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 017	N
MW 47	NV106322846	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 019	N
MW 48	NV106322847	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 019	N
MW 49	NV106322848	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 019	N
MW 50	NV106322849	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 019	N
MW 51	NV106322850	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 019	N
MW 52	NV106322851	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 019	N
MW 53	NV106322852	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 019	**Y**
MW 54	NV106322853	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 019	**Y**
MW 55	NV106322854	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 019	**Y**
MW 56	NV106322855	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 019	**Y**
MW 57	NV106322856	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 017	N
MW 58	NV106322857	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 017	N
MW 59	NV106322858	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 017	N
MW 60	NV106322859	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 017	N
MW 61	NV106322860	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 017	N

Claim Name	Serial Number	Acres	Hectares	Date Of Location	Next Payment Due Date	Meridian Township Range Section	Royalty
MW 62	NV106322861	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 017	N
MW 63	NV106322862	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 017	N
MW 64	NV106322863	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 017	N
MW 65	NV106322864	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 017	N
MW 66	NV106322865	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 017	N
MW 67	NV106322866	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 017	N
MW 68	NV106322867	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 017	N
MW 69	NV106322868	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 017	N
MW 70	NV106322869	20.66	8.36	2023-09-17	2025-09-02	21 0050N 0440E 017	N
MW 71	NV106322870	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 017	N
MW 72	NV106322871	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 017	N
MW 73	NV106322872	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 020	N
MW 74	NV106322873	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 020	N
MW 75	NV106322874	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 020	N
MW 76	NV106322875	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 020	N
MW 77	NV106322876	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 020	N
MW 78	NV106322877	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 020	N
MW 79	NV106322878	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 020	N
MW 80	NV106322879	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 020	N
MW 81	NV106322880	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 020	**Y**
MW 82	NV106322881	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 020	**Y**
MW 83	NV106322882	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 020	**Y**
MW 84	NV106322883	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 020	**Y**
MW 85	NV106322884	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 017	N
MW 86	NV106322885	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 016	N
MW 87	NV106322886	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 017	N
MW 88	NV106322887	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 017	N
MW 89	NV106322888	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 017	N
MW 90	NV106322889	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 016	N
MW 91	NV106322890	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 017	N
MW 92	NV106322891	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 016	N
MW 93	NV106322892	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 017	N
MW 94	NV106322893	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 016	N
MW 95	NV106322894	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 017	N
MW 96	NV106322895	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 017	N
MW 97	NV106322896	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 017	N
MW 98	NV106322897	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 017	N
MW 99	NV106322898	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 017	N
MW 100	NV106322899	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 016	N
MW 101	NV106322900	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 017	N
MW 102	NV106322901	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 016	N
MW 103	NV106322902	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 020	N
MW 104	NV106322903	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 020	N
MW 105	NV106322904	20.66	8.36	2023-09-17	2025-09-02	21 0050N 0440E 020	N
MW 106	NV106322905	20.66	8.36	2023-09-17	2025-09-02	21 0050N 0440E 020	N
MW 107	NV106322906	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 020	N
MW 108	NV106322907	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 020	N
MW 109	NV106322908	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 020	N
MW 110	NV106322909	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 020	N
MW 111	NV106322910	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 020	N
MW 112	NV106322911	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 020	N
MW 113	NV106322912	20.66	8.36	2023-09-17	2025-09-02	21 0050N 0440E 020	N
MW 114	NV106322913	20.66	8.36	2023-09-17	2025-09-02	21 0050N 0440E 020	N
MW 115	NV106322914	20.66	8.36	2023-09-17	2025-09-02	21 0050N 0440E 020	N
MW 116	NV106322915	20.66	8.36	2023-09-17	2025-09-02	21 0050N 0440E 021	N
MW 117	NV106322916	20.66	8.36	2023-09-17	2025-09-02	21 0050N 0440E 020	N
MW 118	NV106322917	20.66	8.36	2023-09-17	2025-09-02	21 0050N 0440E 021	N
MW 119	NV106322918	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 020	**Y**
MW 120	NV106322919	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 020	N
MW 121	NV106322920	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 016	N
MW 122	NV106322921	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 016	N
MW 123	NV106322922	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 016	N

Claim Name	Serial Number	Acres	Hectares	Date Of Location	Next Payment Due Date	Meridian Township Range Section	Royalty
MW 124	NV106322923	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 016	N
MW 125	NV106322924	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 016	N
MW 126	NV106322925	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 016	N
MW 127	NV106322926	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 016	N
MW 128	NV106322927	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 016	N
MW 129	NV106322928	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 016	N
MW 130	NV106322929	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 016	N
MW 131	NV106322930	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 016	N
MW 132	NV106322931	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 016	N
MW 133	NV106322932	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 016	N
MW 134	NV106322933	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 016	N
MW 135	NV106322934	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 016	N
MW 136	NV106322935	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0440E 016	N
MW 137	NV106322936	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 021	N
MW 138	NV106322937	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 021	N
MW 139	NV106322938	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 021	N
MW 140	NV106322939	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 021	N
MW 141	NV106322940	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 021	N
MW 142	NV106322941	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 021	N
MW 143	NV106322942	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 021	N
MW 144	NV106322943	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 021	N
MW 145	NV106322944	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 021	N
MW 146	NV106322945	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 021	N
MW 147	NV106322946	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 021	N
MW 148	NV106322947	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 021	N
MW 149	NV106322948	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 021	N
MW 150	NV106322949	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 021	N
MW 151	NV106322950	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 021	N
MW 152	NV106322951	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 021	N
MW 153	NV106322952	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 021	N
MW 154	NV106322953	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 021	N
MW 155	NV106322954	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 021	N
MW 156	NV106322955	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 028	N
MW 157	NV106322956	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 028	N
MW 158	NV106322957	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 028	N
MW 159	NV106322958	20.66	8.36	2023-09-14	2025-09-02	21 0050N 0440E 028	N
MW 160	NV106322959	20.66	8.36	2023-09-14	2025-09-02	21 0050N 0440E 028	N
MW 161	NV106322960	20.66	8.36	2023-09-14	2025-09-02	21 0050N 0440E 028	N
MW 162	NV106322961	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 028	N
MW 163	NV106322962	20.66	8.36	2023-09-14	2025-09-02	21 0050N 0440E 028	N
MW 164	NV106322963	20.66	8.36	2023-09-14	2025-09-02	21 0050N 0440E 028	N
MW 165	NV106322964	20.66	8.36	2023-09-14	2025-09-02	21 0050N 0440E 028	N
MW 166	NV106322965	20.66	8.36	2023-09-14	2025-09-02	21 0050N 0440E 028	N
MW 167	NV106322966	20.66	8.36	2023-09-14	2025-09-02	21 0050N 0440E 028	N
MW 168	NV106322967	20.66	8.36	2023-09-14	2025-09-02	21 0050N 0440E 028	N
MW 169	NV106322968	20.66	8.36	2023-09-14	2025-09-02	21 0050N 0440E 028	N
MW 170	NV106322969	20.66	8.36	2023-09-14	2025-09-02	21 0050N 0440E 028	N
MW 171	NV106322970	20.66	8.36	2023-09-14	2025-09-02	21 0050N 0440E 028	N
MW 172	NV106322971	20.66	8.36	2023-09-14	2025-09-02	21 0050N 0440E 028	N
MW 173	NV106322972	20.66	8.36	2023-09-14	2025-09-02	21 0050N 0440E 028	N
MW 174	NV106322973	20.66	8.36	2023-09-14	2025-09-02	21 0050N 0440E 033	N
MW 175	NV106322974	20.66	8.36	2023-09-14	2025-09-02	21 0050N 0440E 033	N
MW 176	NV106322975	20.66	8.36	2023-09-14	2025-09-02	21 0050N 0440E 033	N
MW 177	NV106322976	20.66	8.36	2023-09-14	2025-09-02	21 0050N 0440E 033	N
MW 178	NV106322977	20.66	8.36	2023-09-14	2025-09-02	21 0050N 0440E 033	N
MW 179	NV106322978	20.66	8.36	2023-09-14	2025-09-02	21 0050N 0440E 033	N
MW 180	NV106322979	20.66	8.36	2023-09-14	2025-09-02	21 0050N 0440E 033	N
MW 181	NV106322980	20.66	8.36	2023-09-14	2025-09-02	21 0050N 0440E 033	N
MW 182	NV106322981	20.66	8.36	2023-09-14	2025-09-02	21 0050N 0440E 033	N
MW 183	NV106322982	20.66	8.36	2023-09-14	2025-09-02	21 0050N 0440E 033	N
MW 184	NV106322983	20.66	8.36	2023-09-14	2025-09-02	21 0050N 0440E 033	N
MW 185	NV106322984	20.66	8.36	2023-09-14	2025-09-02	21 0050N 0440E 033	N

Claim Name	Serial Number	Acres	Hectares	Date Of Location	Next Payment Due Date	Meridian Township Range Section	Royalty
MW 186	NV106322985	20.66	8.36	2023-09-14	2025-09-02	21 0050N 0440E 033	N
MW 187	NV106322986	20.66	8.36	2023-09-14	2025-09-02	21 0050N 0440E 033	N
MW 188	NV106322987	20.66	8.36	2023-09-14	2025-09-02	21 0050N 0440E 033	N
MW 189	NV106322988	20.66	8.36	2023-09-14	2025-09-02	21 0050N 0440E 033	N
MW 190	NV106322989	20.66	8.36	2023-09-14	2025-09-02	21 0050N 0440E 033	N
MW 191	NV106322990	20.66	8.36	2023-09-14	2025-09-02	21 0040N 0440E 004	N
MW 192	NV106322991	20.66	8.36	2023-09-14	2025-09-02	21 0050N 0440E 033	N
MW 193	NV106322992	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 029	N
MW 194	NV106322993	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 029	N
MW 195	NV106322994	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 029	N
MW 196	NV106322995	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 029	N
MW 197	NV106322996	10.33	4.18	2023-09-16	2025-09-02	21 0050N 0440E 028	N
MW 198	NV106322997	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 015	N
MW 199	NV106322998	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 015	N
MW 200	NV106322999	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 015	N
MW 201	NV106323000	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 015	N
MW 202	NV106323001	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 015	N
MW 203	NV106323002	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 016	N
MW 204	NV106323003	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 015	N
MW 205	NV106323004	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 015	N
MW 206	NV106323005	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 015	N
MW 207	NV106323006	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 022	N
MW 208	NV106323007	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 022	N
MW 209	NV106323008	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 022	N
MW 210	NV106323009	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 022	N
MW 211	NV106323010	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 022	N
MW 212	NV106323011	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 022	N
MW 213	NV106323012	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 022	N
MW 214	NV106323013	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 022	N
MW 215	NV106323014	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 022	N
MW 216	NV106323015	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 028	N
MW 217	NV106323016	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 028	N
MW 218	NV106323017	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 028	N
MW 219	NV106323018	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 028	N
MW 220	NV106323019	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 028	N
MW 221	NV106323020	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 027	N
MW 222	NV106323021	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 027	N
MW 223	NV106323022	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 027	N
MW 224	NV106323023	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 027	N
MW 225	NV106323024	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 034	N
MW 226	NV106323025	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 034	N
MW 227	NV106323026	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0440E 034	N
MW 228	NV106323027	20.66	8.36	2023-09-17	2025-09-02	21 0050N 0440E 034	N
MW 229	NV106323028	20.66	8.36	2023-09-17	2025-09-02	21 0050N 0440E 034	N
MW 230	NV106323029	20.66	8.36	2023-09-17	2025-09-02	21 0050N 0440E 034	N
MW 231	NV106323030	20.66	8.36	2023-09-17	2025-09-02	21 0050N 0440E 033	N
MW 232	NV106323031	20.66	8.36	2023-09-17	2025-09-02	21 0050N 0440E 034	N
MW 233	NV106323032	20.66	8.36	2023-09-17	2025-09-02	21 0050N 0440E 034	N
MWAY 67	NV106322786	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 032	N
MWAY 68	NV106322787	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 033	N
MWAY 117	NV106322788	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 031	N
MWAY 118	NV106322789	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 031	N
MWAY 119	NV106322790	20.66	8.36	2023-09-11	2025-09-02	21 0040N 0440E 006	N
MWAY 147	NV106322791	20.66	8.36	2023-09-13	2025-09-02	21 0050N 0430E 036	N
MWAY 148	NV106322792	20.66	8.36	2023-09-13	2025-09-02	21 0050N 0440E 031	N
MWAY 149	NV106322793	20.66	8.36	2023-09-13	2025-09-02	21 0040N 0430E 001	N
MWAY 150	NV106322794	20.66	8.36	2023-09-13	2025-09-02	21 0040N 0440E 006	N
MWAY 396	NV106322795	13.77	5.57	2023-09-13	2025-09-02	21 0040N 0440E 005	N
MWAY 649	NV106322796	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 029	**Y**
MWAY 651	NV106322797	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 020	**Y**
MWAY 653	NV106322798	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 020	**Y**
MWAY 655	NV106322799	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 020	**Y**

Claim Name	Serial Number	Acres	Hectares	Date Of Location	Next Payment Due Date	Meridian Township Range Section	Royalty
RD 8	NV106322734	10.33	4.18	2023-09-12	2025-09-02	21 0050N 0440E 032	Y
RD 12	NV106322735	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 033	Y
RD 16	NV106322736	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 033	Y
RD 20	NV106322737	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 032	Y
RD 24	NV106322738	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 032	Y
RD 25	NV106322739	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 024	Y
RD 27	NV106322740	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 024	Y
RD 29	NV106322741	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 024	Y
RD 50	NV106322742	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 025	Y
RD 52	NV106322743	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 025	Y
RD 54	NV106322744	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 025	Y
RD 56	NV106322745	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 025	Y
RD 58	NV106322746	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 025	Y
RD 60	NV106322747	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 025	Y
RD 69	NV106322748	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0430E 025	Y
RD 70	NV106322749	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0430E 025	Y
RD 71	NV106322750	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0430E 014	Y
RD 72	NV106322751	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0430E 025	Y
RD 73	NV106322752	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0430E 025	Y
RD 74	NV106322753	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0430E 025	Y
RD 75	NV106322754	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0430E 025	Y
RD 76	NV106322755	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0430E 036	Y
RD 77	NV106322756	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 030	Y
RD 78	NV106322757	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 030	Y
RD 79	NV106322758	20.66	8.36	2023-09-13	2025-09-02	21 0050N 0430E 025	Y
RD 80	NV106322759	20.66	8.36	2023-09-13	2025-09-02	21 0050N 0440E 030	Y
RD 81	NV106322760	20.66	8.36	2023-09-13	2025-09-02	21 0050N 0430E 036	Y
RD 82	NV106322761	20.66	8.36	2023-09-13	2025-09-02	21 0050N 0440E 031	Y
RD 83	NV106322762	20.66	8.36	2023-09-13	2025-09-02	21 0050N 0430E 036	Y
RD 84	NV106322763	20.66	8.36	2023-09-13	2025-09-02	21 0050N 0440E 031	Y
RD 85	NV106322764	20.66	8.36	2023-09-13	2025-09-02	21 0050N 0430E 036	Y
RD 86	NV106322765	20.66	8.36	2023-09-13	2025-09-02	21 0050N 0440E 031	Y
RD 87	NV106322766	20.66	8.36	2023-09-13	2025-09-02	21 0050N 0430E 036	Y
RD 88	NV106322767	20.66	8.36	2023-09-13	2025-09-02	21 0050N 0440E 031	Y
RD 89	NV106322768	20.66	8.36	2023-09-13	2025-09-02	21 0050N 0430E 036	Y
RD 90	NV106322769	20.66	8.36	2023-09-13	2025-09-02	21 0050N 0440E 031	Y
RD 91	NV106322770	20.66	8.36	2023-09-13	2025-09-02	21 0050N 0430E 036	Y
RD 92	NV106322771	20.66	8.36	2023-09-13	2025-09-02	21 0050N 0440E 031	Y
RD 93	NV106322772	20.66	8.36	2023-09-13	2025-09-02	21 0050N 0440E 031	Y
RD 94	NV106322773	20.66	8.36	2023-09-13	2025-09-02	21 0050N 0440E 031	Y
RD 95	NV106322774	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 031	Y
RD 96	NV106322775	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 031	Y
RD 97	NV106322776	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 031	Y
RD 98	NV106322777	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 031	Y
RD 99	NV106322778	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 031	Y
RD 100	NV106322779	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 031	Y
RD 101	NV106322780	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 031	Y
RD 102	NV106322781	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 031	Y
RD 103	NV106322782	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 031	Y
RD 104	NV106322783	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 031	Y
RD 105	NV106322784	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 031	Y
RD 106	NV106322785	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 031	Y
RV 29	NV106322727	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 028	Y
RV 31	NV106322728	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 028	Y
RV 33	NV106322729	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 028	Y
RV 35	NV106322730	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 028	Y
RV 37	NV106322731	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 028	Y
RV 39	NV106322732	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 032	Y
RV 41	NV106322733	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 032	Y
SP 1	NV106322525	20.66	8.36	2023-09-10	2025-09-02	21 0050N 0440E 030	Y
SP 2	NV106322526	20.66	8.36	2023-09-10	2025-09-02	21 0050N 0440E 029	Y
SP 3	NV106322527	20.66	8.36	2023-09-10	2025-09-02	21 0050N 0440E 030	Y

Claim Name	Serial Number	Acres	Hectares	Date Of Location	Next Payment Due Date	Meridian Township Range Section	Royalty
SP 4	NV106322528	20.66	8.36	2023-09-10	2025-09-02	21 0050N 0440E 030	Y
SP 5	NV106322529	20.66	8.36	2023-09-10	2025-09-02	21 0050N 0440E 030	Y
SP 6	NV106322530	20.66	8.36	2023-09-10	2025-09-02	21 0050N 0440E 030	Y
SP 7	NV106322531	20.66	8.36	2023-09-10	2025-09-02	21 0050N 0440E 030	Y
SP 8	NV106322532	20.66	8.36	2023-09-10	2025-09-02	21 0050N 0440E 029	Y
SP 9	NV106322533	20.66	8.36	2023-09-10	2025-09-02	21 0050N 0440E 030	Y
SP 10	NV106322534	20.66	8.36	2023-09-10	2025-09-02	21 0050N 0440E 030	Y
SP 11	NV106322535	20.66	8.36	2023-09-10	2025-09-02	21 0050N 0440E 030	Y
SP 12	NV106322536	20.66	8.36	2023-09-10	2025-09-02	21 0050N 0440E 030	Y
SP 13	NV106322537	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 030	Y
SP 14	NV106322538	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 030	Y
SP 15	NV106322539	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 031	Y
SP 16	NV106322540	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 032	Y
SP 17	NV106322541	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 031	Y
SP 18	NV106322542	20.66	8.36	2023-09-10	2025-09-02	21 0050N 0440E 031	Y
SP 21	NV106322543	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0430E 025	Y
SP 22	NV106322544	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 030	Y
SP 23	NV106322545	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0430E 025	Y
SP 24	NV106322546	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 030	Y
SP 25	NV106322547	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0430E 025	Y
SP 26	NV106322548	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 030	Y
SP 27	NV106322549	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0430E 025	Y
SP 28	NV106322550	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 030	Y
SP 29	NV106322551	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0430E 025	Y
SP 30	NV106322552	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 030	Y
SP 31	NV106322553	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0430E 025	Y
SP 32	NV106322554	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 030	Y
SP 65	NV106322555	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0430E 025	Y
SP 66	NV106322556	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 030	Y
SP 67	NV106322557	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0430E 024	Y
SP 68	NV106322558	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 030	Y
SP 69	NV106322559	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0430E 024	Y
SP 70	NV106322560	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 019	Y
SP 71	NV106322561	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0430E 024	Y
SP 72	NV106322562	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0430E 024	Y
SP 73	NV106322563	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0430E 025	Y
SP 74	NV106322564	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0430E 024	Y
SP 75	NV106322565	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0430E 025	Y
SP 76	NV106322566	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0430E 025	Y
SP 77	NV106322567	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0430E 025	Y
SP 78	NV106322568	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0430E 025	Y
SP 79	NV106322569	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0430E 025	Y
SP 80	NV106322570	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0430E 025	Y
SP 81	NV106322571	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0430E 025	Y
SP 82	NV106322572	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0430E 025	Y
SP 83	NV106322573	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0430E 025	Y
SP 84	NV106322574	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0430E 025	Y
SP 95	NV106322575	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0430E 024	Y
SP 96	NV106322576	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0430E 024	Y
SP 97	NV106322577	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0430E 024	Y
SP 98	NV106322578	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0430E 024	Y
SP 105	NV106322579	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 019	Y
SP 106	NV106322580	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 019	Y
SP 107	NV106322581	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 019	Y
SP 108	NV106322582	20.66	8.36	2023-09-12	2025-09-02	21 0050N 0440E 019	Y
SP 115	NV106322583	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 030	Y
SP 116	NV106322584	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 019	Y
SP 117	NV106322585	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 019	Y
SP 118	NV106322586	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 019	Y
SP 119	NV106322587	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 019	Y
SP 123	NV106322588	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 029	Y
SP 124	NV106322589	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 019	Y

Claim Name	Serial Number	Acres	Hectares	Date Of Location	Next Payment Due Date	Meridian Township Range Section	Royalty
SP 125	NV106322590	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 019	Y
SP 126	NV106322591	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 019	Y
SP 127	NV106322592	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 020	Y
SP 280	NV106322593	20.66	8.36	2023-09-10	2025-09-02	21 0050N 0440E 029	Y
SP 281	NV106322594	20.66	8.36	2023-09-10	2025-09-02	21 0050N 0440E 029	Y
SP 282	NV106322595	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 029	Y
SP 283	NV106322596	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 020	Y
SP 284	NV106322597	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 020	Y
SP 285	NV106322598	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 020	Y
SP 286	NV106322599	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 020	Y
SP 300	NV106322600	20.66	8.36	2023-09-10	2025-09-02	21 0050N 0440E 029	Y
SP 301	NV106322601	20.66	8.36	2023-09-10	2025-09-02	21 0050N 0440E 029	Y
SP 302	NV106322602	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 029	Y
SP 320	NV106322603	20.66	8.36	2023-09-10	2025-09-02	21 0050N 0440E 029	Y
SP 321	NV106322604	20.66	8.36	2023-09-10	2025-09-02	21 0050N 0440E 029	Y
SP 322	NV106322605	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 029	Y
SP 340	NV106322606	20.66	8.36	2023-09-10	2025-09-02	21 0050N 0440E 029	Y
SP 341	NV106322607	20.66	8.36	2023-09-10	2025-09-02	21 0050N 0440E 029	Y
SP 342	NV106322608	20.66	8.36	2023-09-10	2025-09-02	21 0050N 0440E 029	Y
SP 343	NV106322609	20.66	8.36	2023-09-10	2025-09-02	21 0050N 0440E 029	Y
SP 344	NV106322610	20.66	8.36	2023-09-10	2025-09-02	21 0050N 0440E 029	Y
SP 345	NV106322611	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 029	Y
SP 346	NV106322612	20.66	8.36	2023-09-10	2025-09-02	21 0050N 0440E 029	Y
SP 347	NV106322613	20.66	8.36	2023-09-10	2025-09-02	21 0050N 0440E 029	Y
SP 348	NV106322614	20.66	8.36	2023-09-10	2025-09-02	21 0050N 0440E 029	Y
SP 349	NV106322615	20.66	8.36	2023-09-10	2025-09-02	21 0050N 0440E 029	Y
SP 350	NV106322616	20.66	8.36	2023-09-10	2025-09-02	21 0050N 0440E 029	Y
SP 351	NV106322617	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 029	Y
SP 352	NV106322618	20.66	8.36	2023-09-10	2025-09-02	21 0050N 0440E 029	Y
SP 353	NV106322619	20.66	8.36	2023-09-10	2025-09-02	21 0050N 0440E 029	Y
SP 354	NV106322620	20.66	8.36	2023-09-10	2025-09-02	21 0050N 0440E 029	Y
SP 355	NV106322621	20.66	8.36	2023-09-10	2025-09-02	21 0050N 0440E 029	Y
SP 356	NV106322622	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 029	Y
SP 357	NV106322623	20.66	8.36	2023-09-11	2025-09-02	21 0050N 0440E 029	Y
SP 358	NV106715124	20.66	8.36	2025-01-13	2025-09-02	21 0050N 0440E 032	Y
SP 359	NV106715125	20.66	8.36	2025-01-13	2025-09-02	21 0050N 0440E 032	Y
SP 360	NV106715126	20.66	8.36	2025-01-13	2025-09-02	21 0050N 0440E 032	Y
SP 366	NV106715127	20.66	8.36	2025-01-13	2025-09-02	21 0050N 0440E 032	Y
SP 367	NV106715128	20.66	8.36	2025-01-13	2025-09-02	21 0050N 0440E 032	Y
SP 368	NV106715129	20.66	8.36	2025-01-13	2025-09-02	21 0050N 0440E 032	Y
SP 374	NV106715130	20.66	8.36	2025-01-13	2025-09-02	21 0050N 0440E 032	Y
SP 375	NV106715131	20.66	8.36	2025-01-13	2025-09-02	21 0050N 0440E 032	Y
SP 376	NV106715132	20.66	8.36	2025-01-13	2025-09-02	21 0050N 0440E 032	Y
TG 1	NV106322658	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 013	N
TG 2	NV106322659	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 024	N
TG 3	NV106322660	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 024	N
TG 4	NV106322661	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 024	N
TG 5	NV106322662	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 024	N
TG 6	NV106322663	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 024	N
TG 7	NV106322664	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 024	N
TG 8	NV106322665	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 013	N
TG 9	NV106322666	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 024	N
TG 10	NV106322667	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 013	N
TG 11	NV106322668	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 024	N
TG 12	NV106322669	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 024	N
TG 13	NV106322670	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 024	N
TG 14	NV106322671	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 024	N
TG 15	NV106322672	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 024	N
TG 16	NV106322673	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 024	N
TG 17	NV106322674	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 024	N
TG 18	NV106322675	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 024	N
TG 19	NV106322676	12.39	5.01	2023-09-15	2025-09-02	21 0050N 0430E 013	N

Claim Name	Serial Number	Acres	Hectares	Date Of Location	Next Payment Due Date	Meridian Township Range Section	Royalty
TG 20	NV106322677	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 024	N
TG 21	NV106322678	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 024	N
TG 22	NV106322679	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 024	N
TG 23	NV106322680	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 026	N
TG 24	NV106322681	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 025	N
TG 25	NV106322682	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 025	N
TG 26	NV106322683	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 036	N
TG 27	NV106322684	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 025	N
TG 28	NV106322685	20.66	8.36	2023-09-16	2025-09-02	21 0050N 0430E 025	N
TG 29	NV106322686	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 025	N
TG 30	NV106322687	20.66	8.36	2023-09-15	2025-09-02	21 0050N 0430E 025	N
TG 31	NV106322688	20.66	8.36	2023-09-11	2025-09-02	21 0040N 0440E 005	N
TG 32	NV106322689	20.66	8.36	2023-09-14	2025-09-02	21 0040N 0440E 004	N
TG 33	NV106322690	20.66	8.36	2023-09-14	2025-09-02	21 0040N 0440E 004	N
TG 34	NV106322691	17.21	6.96	2023-09-14	2025-09-02	21 0040N 0440E 004	N
TG 35	NV106322692	17.21	6.96	2023-09-14	2025-09-02	21 0040N 0440E 004	N
TG 36	NV106322693	17.21	6.96	2023-09-14	2025-09-02	21 0040N 0440E 004	N
TG 37	NV106322694	20.66	8.36	2023-09-14	2025-09-02	21 0040N 0440E 005	N
TG 38	NV106322695	20.66	8.36	2023-09-14	2025-09-02	21 0040N 0440E 005	N
TG 39	NV106322696	20.66	8.36	2023-09-14	2025-09-02	21 0040N 0440E 004	N
TG 40	NV106322697	20.66	8.36	2023-09-14	2025-09-02	21 0040N 0440E 004	N
TG 41	NV106322698	20.66	8.36	2023-09-14	2025-09-02	21 0040N 0440E 004	N
TG 42	NV106322699	20.66	8.36	2023-09-14	2025-09-02	21 0040N 0440E 004	N
TG 43	NV106322700	20.66	8.36	2023-09-14	2025-09-02	21 0040N 0440E 004	N
TG 44	NV106322701	20.66	8.36	2023-09-14	2025-09-02	21 0040N 0440E 004	N
TG 45	NV106322702	20.66	8.36	2023-09-14	2025-09-02	21 0040N 0440E 004	N
TG 46	NV106322703	20.66	8.36	2023-09-14	2025-09-02	21 0040N 0440E 004	N
TG 47	NV106322704	20.66	8.36	2023-09-14	2025-09-02	21 0040N 0440E 003	N
TG 48	NV106322705	20.66	8.36	2023-09-14	2025-09-02	21 0040N 0440E 005	N
TG 49	NV106322706	20.66	8.36	2023-09-14	2025-09-02	21 0040N 0440E 005	N
TG 50	NV106322707	20.66	8.36	2023-09-14	2025-09-02	21 0040N 0440E 005	N
TG 51	NV106322708	20.66	8.36	2023-09-14	2025-09-02	21 0040N 0440E 005	N
TG 52	NV106322709	20.66	8.36	2023-09-14	2025-09-02	21 0040N 0440E 004	N
TG 53	NV106322710	20.66	8.36	2023-09-14	2025-09-02	21 0040N 0440E 009	N
TG 54	NV106322711	20.66	8.36	2023-09-14	2025-09-02	21 0040N 0440E 004	N
TG 55	NV106322712	20.66	8.36	2023-09-14	2025-09-02	21 0040N 0440E 004	N
TG 56	NV106322713	20.66	8.36	2023-09-14	2025-09-02	21 0040N 0440E 004	N
TG 57	NV106322714	20.66	8.36	2023-09-14	2025-09-02	21 0040N 0440E 004	N
TG 58	NV106322715	20.66	8.36	2023-09-14	2025-09-02	21 0040N 0440E 004	N
TG 59	NV106322716	20.66	8.36	2023-09-14	2025-09-02	21 0040N 0440E 004	N
TG 60	NV106322717	20.66	8.36	2023-09-14	2025-09-02	21 0040N 0440E 004	N
TG 61	NV106322718	20.66	8.36	2023-09-14	2025-09-02	21 0040N 0440E 004	N
TG 62	NV106322719	20.66	8.36	2023-09-14	2025-09-02	21 0040N 0440E 004	N
TG 63	NV106322720	20.66	8.36	2023-09-14	2025-09-02	21 0040N 0440E 004	N
TG 64	NV106322721	20.66	8.36	2023-09-14	2025-09-02	21 0040N 0440E 004	N
TG 65	NV106322722	20.66	8.36	2023-09-14	2025-09-02	21 0040N 0440E 004	N
TG 66	NV106322723	20.66	8.36	2023-09-14	2025-09-02	21 0040N 0440E 004	N
TG 67	NV106322724	20.66	8.36	2023-09-14	2025-09-02	21 0040N 0440E 004	N
TG 68	NV106322725	20.66	8.36	2023-09-14	2025-09-02	21 0040N 0440E 003	N
TG 69	NV106322726	20.66	8.36	2023-09-14	2025-09-02	21 0040N 0440E 003	N
WAY 3	NV106715110	20.66	8.36	2025-01-13	2025-09-02	21 0050N 0440E 032	N
WAY 4	NV106715111	15.15	6.13	2025-01-13	2025-09-02	21 0050N 0440E 032	N
WAY 5	NV106715112	15.15	6.13	2025-01-13	2025-09-02	21 0050N 0440E 032	N
WAY 6	NV106715113	15.15	6.13	2025-01-13	2025-09-02	21 0050N 0440E 032	N
WAY 7	NV106715114	15.15	6.13	2025-01-13	2025-09-02	21 0050N 0440E 032	N
WAY 8	NV106715115	15.15	6.13	2025-01-13	2025-09-02	21 0050N 0440E 032	N
WAY 9	NV106715116	15.15	6.13	2025-01-13	2025-09-02	21 0050N 0440E 032	N
WAY 10	NV106715117	15.15	6.13	2025-01-13	2025-09-02	21 0050N 0440E 032	N
WAY 11	NV106715118	15.15	6.13	2025-01-13	2025-09-02	21 0050N 0440E 032	N
WAY 12	NV106715119	15.15	6.13	2025-01-13	2025-09-02	21 0050N 0440E 032	N
WAY 13	NV106715120	20.66	8.36	2025-01-13	2025-09-02	21 0050N 0440E 032	N
WAY 14	NV106715121	15.15	6.13	2025-01-13	2025-09-02	21 0050N 0440E 032	N

Claim Name	Serial Number	Acres	Hectares	Date Of Location	Next Payment Due Date	Meridian Township Range Section	Royalty
WAY 15	NV106715122	13.13	5.31	2025-01-13	2025-09-02	21 0050N 0440E 032	N
WAY 16	NV106715123	15.15	6.13	2025-01-13	2025-09-02	21 0050N 0440E 032	N
WAY 17	NV106322647	15.15	6.13	2023-09-13	2025-09-02	21 0050N 0440E 032	N
WAY 18	NV106322648	15.15	6.13	2023-09-13	2025-09-02	21 0050N 0440E 032	N
WAY 20	NV106322649	20.66	8.36	2023-09-13	2025-09-02	21 0040N 0440E 005	N
WAY 21	NV106322650	20.66	8.36	2023-09-13	2025-09-02	21 0040N 0440E 005	N
WAY 22	NV106322651	20.66	8.36	2023-09-13	2025-09-02	21 0040N 0440E 005	N
WAY 23	NV106322652	20.66	8.36	2023-09-13	2025-09-02	21 0040N 0440E 005	N
WAY 24	NV106322653	20.66	8.36	2023-09-13	2025-09-02	21 0040N 0440E 005	N
WAY 25	NV106322654	20.66	8.36	2023-09-13	2025-09-02	21 0040N 0440E 005	N
WAY 26	NV106322655	20.66	8.36	2023-09-13	2025-09-02	21 0050N 0440E 032	N
WAY 27	NV106322656	20.66	8.36	2023-09-13	2025-09-02	21 0050N 0440E 033	N
WAY 28	NV106322657	5.5	2.23	2023-09-13	2025-09-02	21 0040N 0440E 005	N

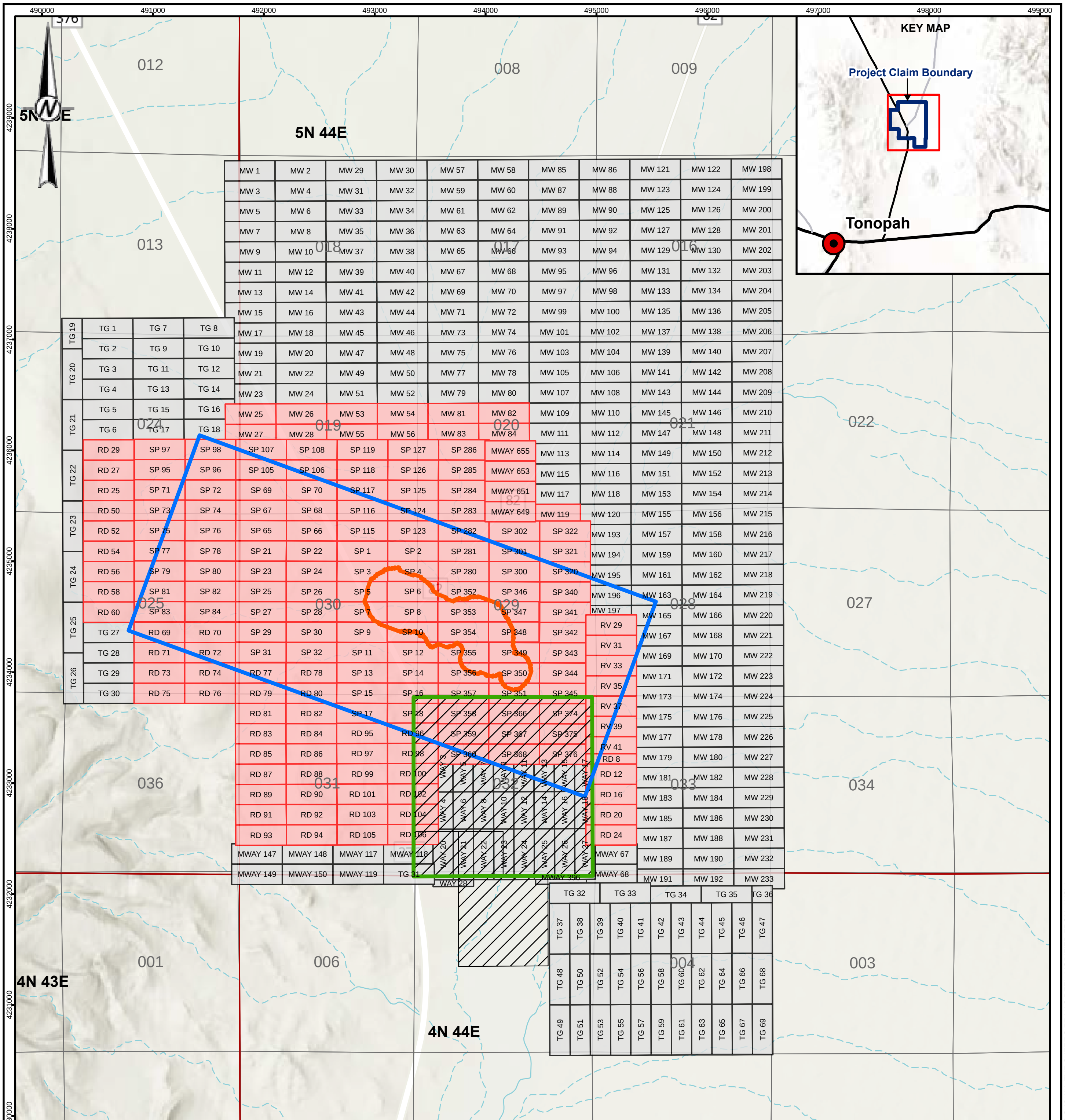
In September 2022, Viva acquired, in the name of its Nevada subsidiary, 0862130 Corp, a 16.2-hectare (40-acre) parcel of the private SRHA surface land located in the SW ¼ SW ¼ Sec. 32 T5N R44E. On July 7, 2023, the Company entered into a lease agreement with TOWERCO 2013 LLC (Towerco) to lease approximately 929 square metres (m^2) (10,000 square feet [ft^2]) of this parcel. As per the agreement, the initial term of the lease will be five years with 19 additional options of five-year terms (for a total of 100 years). Towerco completed construction of a telecommunication cellular tower at the leased premises and has commenced paying to 0862130 Corp US$1,000 per month for the first year of the lease term. On April 22, 2025, Viva was notified by Towerco of their intent to buy-out their monthly lease obligation for a lump sum of US$150,000 as per the terms of the lease agreement.

All surface infrastructure and disturbance in the conceptual designs for the PEA are located on BLM surface and Viva-owned private surface land.

4.3 Permits

Viva has received permits and authorizations necessary to conduct mineral exploration and environmental monitoring and testing activities on both public and private land, as well as access across public land to privately-owned property. Authorizations received with Viva's Nevada-registered subsidiary 0862130 Corp. as the Operator/Permittee include:

- Decision Record (DR) and Finding of No Significant Impact (FONSI) issued by the BLM as BLM Serial NVNV106037004 (legacy Casefile NVN-076629),
- Nevada Reclamation Permit 0210 issued by the Bureau of Mining Regulation and Reclamation (BMRR),
- Land Use Permit Serial NVNV1058663574 (legacy Casefile NVN-101549) issued by the BLM,
- Right-of-Way (ROW)/Roads Serial NVNV105959651 (legacy Casefile NVN-062845) issued by the BLM,
- Waiver MM-232 to use Test Well PW07-01 for exploration drill water issued by the Nevada Division of Water Resources (NDWR),
- Waiver M/O to construct 2 upgradient groundwater monitoring wells issued by the NDWR,
- Waiver M/O to use Test Well PW22-02 as long-term groundwater monitoring well MW22-02 issued by the NDWR



LEGEND

- UNPATENTED LODE CLAIM
- UNPATENTED LODE CLAIM (ROYALTY)
- MINERAL RESOURCE MODEL BOUNDARY
- RESOURCE PIT SHELL OUTLINE
- FEDERALLY OWNED SUBSURFACE MINERAL RIGHTS
- PRIVATELY OWNED SURFACE RIGHTS
- PUBLIC LAND SURVEY, SECTION GRID
- PUBLIC LAND SURVEY, TOWNSHIP RANGE

0 1 2

1:50,000 KILOMETRES

REFERENCE(S)

BUREAU OF LAND MANAGEMENT
CLAIM BOUNDARIES PROVIDED BY VIVA GOLD CORP.
PROJECTED COORDINATE SYSTEM: NAD 1983 UTM ZONE 11N

CLIENT
VIVA GOLD CORP.

PROJECT
TONOPAH GOLD PROJECT

TITLE
MINERAL CLAIMS

CONSULTANT

YYYY-MM-DD	2025-08-05
DESIGNED	RK
PREPARED	RK
REVIEWED	JS
APPROVED	BT

PROJECT NO.	CONTROL	REV.	FIGURE
CA0004670.0343		0	4.2

IF THIS MEASUREMENT DOES NOT MATCH WHAT IS SHOWN, THE SHEET SIZE HAS BEEN MODIFIED FROM: ANSI A

25mm

0

Temporary groundwater appropriations were issued by the NDWR to supply exploration drilling water from an existing well in the Project area. On November 16, 2022, the NDWR granted an extension to use the project's existing water supply well for exploration and fugitive dust suppression purposes through November 17, 2027. The use is restricted to dust control and drilling purposes and must not exceed 6,136 cubic metres (m^3) per annum.

The current reclamation bond to ensure reclamation authorized under BLM Serial NVNV106037004 (legacy Casefile NVN-076629) and Reclamation Permit 0210 is US$103,557. Agency Decisions concurring with the updated estimate were received September 11 and November 9, 2023.

Viva also received in 2022 Waivers from NDWR to construct groundwater monitoring wells and one aquifer test well. Extension of the Waivers for the groundwater monitoring wells, and conversion of the aquifer test well to a long-term groundwater monitoring well were received October 24, 2022, and August 16, 2023, respectively.

4.4 Royalties, Encumbrances, Other Obligations

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

Rex Exploration Corp. (Rex) held an option on the 245 claims under an agreement with the Schmidts and Pattons dated July 2, 2001, and exercised August 5, 2005. Midway Gold Corp (Midway Gold), at the time known as Red Emerald Resource Corp (Red Emerald), held an option on the claims under an agreement with Rex dated August 8, 2001, and exercised October 15, 2002.

On December 31, 2004, Midway Gold acquired all of the issued and outstanding shares of Rex and assigned the original option agreement to its wholly owned subsidiary MGC Resources Inc. (MGC) on January 1, 2005.

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

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

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

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

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

- Upon commercial production the Royalty Modification Agreement granted to the Optionors a 2% NSR over a total of 184 unpatented lode mining claims in the RD08 to RD106 claim group, the RV31 to 41 group, the SP#1 to SP#127 group, the SP4 to SP382 group, the MW26 to 119 group, and the MWAY 649 to 655 group. The claim groups are discontinuous in numerical order.
- Upon commercial production, the Optionors will receive a 2% royalty based on the NSR
- Viva paid US$25,000 to each of the two royalty holders
- Viva issued 750,000 common shares to each of the two royalty holders
- Viva has the option to buy down 1% (half) of the 2% royalty at any time by paying the Optionors US$1.0M in cash or immediately available funds.

In 2023, Viva re-staked and filed all claims. In this process seven overlapping claims were eliminated to control the same area of land and royalty deeds with the Optionors were modified accordingly.

4.5 Environmental Liabilities

The BLM DR and FONSI, and Nevada Reclamation Permit, authorize surface disturbance for up to 30.4 hectares for mineral exploration and support activities. Viva’s current reclamation bond liability deposited with the BLM Nevada State Office is US$103,557 for reclamation of disturbance authorized under BLM Serial NVNV106037004 (legacy Casefile NVN-076629) and Nevada Reclamation Permit 0210. To-date, only 7.2 hectares of public land and 0.4 hectares of private land of the total 30.4 hectares are reported as disturbed and remain under reclamation bond. The 7.6 hectares will require reclamation prior to release of bonds.

The BLM has also authorized 0.3 hectares for construction of 2 upgradient groundwater monitoring wells and 2 Rapid Infiltration Basin (RIB) test pits under 43 CFR 2920 Land Use Permit Serial NVNV1058663574 (legacy Casefile NVN-101549). The Permit expires November 10, 2025. Viva will apply to renew the Permit in September 2025.

The BLM has also granted ROW access across public land to Viva’s privately-owned 16.2-hectare property in SW ¼ SW ¼ Sec. 32 T5NR44E under ROW/Roads Serial NVNV105959651 (legacy Casefile NVN-062845). The ROW expires December 31, 2049. Viva’s current reclamation bond liability deposited with the BLM is US$2,922.

Viva is not aware of any current environmental liabilities not identified in this Report resulting from prior Operators’ mineral exploration and testing operations. Field inspections by Agency staff and Viva support staff confirm the existence of water supply and groundwater monitoring wells that require plug and abandon following completion of exploration or potential subsequent mining operations. BLM and Nevada Division of Environmental Protection (NDEP) BMRR regulations require sufficient reclamation bonding to ensure ultimate completion of all reclamation obligations. Review of Company and Agency records do not report the current presence of residual hydrocarbon (diesel, lubricants, etc.) products resulting from exploration drilling operations in the Project area.

Field inspection of the site by the BLM and NDEP BMRR is conducted periodically. No citations or warnings have been issued, and no fines or penalties have been levied for any environmental or regulatory issues pertaining to the Project under Viva's ownership.

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

4.6 Significant Factors or Risks Affecting Access

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

Sub-surface aquifers in the Ralston Valley serve as the primary water source for the Town of Tonopah. Located along the heavily mineralized Walker Lane Trend, the region's groundwater is naturally impacted by high arsenic content due to its geological composition. This elevated arsenic concentration poses challenges concerning compliance with US Environmental Protection Agency (EPA) and NDEP Bureau of Safe Drinking Water (BSDW) public drinking water supply standards. The initial Tonopah Public Utilities (TPU) wellfield was situated downgradient from the project, below the confluence of the Walker Lane and Sweetwater subsurface aquifers, making it difficult to meet both BSDW and EPA arsenic standards.

To resolve this issue, TPU drilled two additional water production wells in August 2012, positioning them upgradient to the north and east of the project within the Sweetwater aquifer. This strategic move allowed TPU to reduce reliance on the previously downgradient wellfield for the Town of Tonopah's potable water supply. Supporting infrastructure, including a water pipeline and a 15-kilovolt (kV) power line, was extended across the eastern side of the Tonopah project claims to facilitate this new water production field. The new system successfully meets the town's water supply needs while adhering to EPA and BSDW drinking water standards, based on data collected from mandated periodic monitoring samples.

The power line, located within 4 km of the project site, connects to the Nevada Energy power grid at the Tonopah substation. Currently lightly utilized, the power line can potentially be upgraded to a 25 kV service under existing permits. Subject to further study, this power line may have sufficient capacity to support a production operation at the project. Additionally, potable water may be commercially purchased from the pipeline through TPU to fulfill the project's potable water requirements. By extracting water from the aquifer upgradient of the project location, TPU may also contribute to reducing future dewatering rates for the project.

An alternative connection to the Nevada power grid could involve a 256 kV power transmission line that crosses SR 376, approximately 16 km north of the site toward Round Mountain. Existing powerline easements associated with SR 376 may be exploitable for power-line construction.

Viva's exploration activities must comply with all Federal and State cultural resources regulations. Additional surveys are needed on Viva-controlled lands to support future Mine Plan of Operations (MPoO) and Reclamation Permit surface disturbance beyond current permitted areas. Wildlife baseline surveys show certain rodents and raptors in or near the project area, and Federal and State regulations may require mitigation to minimize exploration impacts.

Another risk factor is the potential relocation of SR 376 and Nye CR 82 (Belmont Road) depending on future mining scope. Exploration does not affect these roads; however, SR 376 likely won't need relocation, but Belmont Road may require relocation due to its proximity to mineralization and site facilities layout; ex. waste rock dump placement. If relocation is required, Viva will collaborate with the Nye County Road Department and the Nevada Department of Transportation (NDOT)

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

5.1 Topography, Elevation, Vegetation, and Climate

Local terrain at the Tonopah site is gentle to moderate with seasonal streams and broad washes separating the surrounding pediment slopes near the Ralston Valley bottom. In places, seasonal streams have cut deeply incised channels. Elevation at the property ranges from 1,750-to-2,100 m above mean sea level (m amsl) (5,741-to-6,890 feet [ft] amsl). Vegetation is typical of high-altitude desert in central Nevada, dominated by desert scrub plant species, including shadscale, spiny horsebrush, budsage, winterfat, and prickly pear cacti. Sandy hummocks within defined drainage areas are dominated by greasewood, rubber rabbitbrush, quailbush, and bush seepweed. A few weedy species (cheatgrass, halogeton, Russian thistle, poverty weed, and mustards) reportedly do exist within the project area (Gustin and Ristorcelli, 2005; US Bureau of Land Management, 2003). An example of the terrain and vegetation at the Tonopah Project is shown in Figure 5.1.

The local climate is typical for the high desert of central Nevada and the Basin and Range province. Data from the National Oceanic and Atmospheric Administration (NOAA) shows an average of 21 centimetres (cm; 8-inches) of annual precipitation and 50 cm (20-inches) of snowfall (Climate-Data.org, 2021). Average temperatures range from 3°C (37°F) in winter to 18°C (65°F) in the summer at Tonopah, Nevada, and daytime temperatures commonly exceed 30°C (87°F) during the months of July and August (WeatherSpark, 2023). Work can be conducted year-round at the property.

Source: WSP during June 2023 Site Visit

Figure 5.1: Example of the Terrain and Vegetation of the Tonopah Project

5.2 Accessibility

Access to the Tonopah Project site is provided by SR 376, a paved road that intersects Nye CR 82 (Belmont Road) near the center of the project area. It is approximately 30 km (~20 miles), via paved road, from Tonopah, Nevada to the Tonopah property. The property is accessible year-round.

5.3 Infrastructure and Local Resources

The Project is wholly located on Viva land holdings approximately 30 km (~20 miles) northeast of the Town of Tonopah, Nevada, in the Midway (also known as Rye Patch) Mining District.

The town nearest to the project site, Tonopah, Nevada, hosts a population of approximately 1,938 residents. Nye County hosts an area population of 55,990 per the US Census Bureau 2024 data.

Electrical power to the Project is available from the Tonopah well field line, approximately 5 km (3 miles) east of the project area. This line connects to the Nevada grid at a regional substation located in the Town of Tonopah.

Exploration campaigns use water for drilling water and fugitive dust suppression, which has either been sourced from a permitted water supply well located on site, or the Project purchases water from TPU. Ralston Valley Hydrographic Basin 141, where the Project resides, is a designated groundwater basin per the Nevada State Water Engineer. Basin 141 contains substantial unutilized water resources and remains one of the few under-allocated hydrographic basins remaining in the State of Nevada. The water table in the basin is typically found at between 20 m to 30 m (66 ft to 98 ft) depth. TPU is the principal water rights holder and consumer of water resources in the basin. A public water utility, TPU is a logical provider of water rights and resources for the project, a concept that has been discussed with Town management. A 36.6 cm (14-inch) pipeline from the Belmont Road wellfield crosses Project mineral claims within 920 m (3,000 ft) of the proposed plant site. Alternative water sources include third party lease/purchase and direct application for water rights with the State.

Logistical support is available in Tonopah, which currently supports the Round Mountain Mine just 50 km (31 miles) north of the Tonopah Project, as well as other exploration and mineral resources projects nearby in Nye and Esmeralda counties. The surrounding region has a long history of mining activity, and mining personnel and resources for operations at Tonopah should be available from the local and regional communities. The major cities of Reno and Las Vegas are within 3.5-hour (h) drive of the site.

5.4 Sufficiency of Surface Rights, Sites, and Local Resources

Surface rights are described in Item 4.2. This PEA outlines a mining area and operating footprint which will assist in understanding what surface rights might be needed for eventual development of the project. However, surface rights for eventual mining development have not yet been secured.

The site has excellent logistics and access for exploration, being a short drive from the town of Tonopah, Nevada, with good road access, communications, and access to contractors and labour. The Las Vegas metropolitan area has a population of approximately 2.4 million people with significant construction and manufacturing infrastructure and is located 340 km (211 miles) southeast of the project via US Highway 95. There are major Komatsu and Caterpillar dealers and supply depots located in Las Vegas, as well as Caterpillar and Komatsu parts depots and mining-specific machine shops in Round Mountain, approximately 50 km (31 miles) north of the project. Power and water are available, although water rights will need to be acquired or leased.

There is one water production well already on site. Previous hydrological work has been done on the site due to its proximity to municipal water sources as discussed in Item 4.6 of this report. The report by Water Management

Consultants, Inc. (WMCI) titled “Hydrologic Assessment and projection of Dewatering Requirements” completed in 2008 confirms that pit or underground mine dewatering activities will be required for the Project and that sufficient sub-surface water supply exists in the drainage to meet the needs of both potential production operations and TPU water supply requirements.

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

Viva does not currently envision an underground mining operation for the project, but the data collected for the hydrological studies will be useful in assessing water management needs for open pit mining. Hydrogeologic monitoring at the project resumed in 2020 and continues with a quarterly or semi-annual frequency. Additional seep and springs studies have also been performed in a 15 km (9 mile) radius around the project. Aquifer testing in November-December 2022 provided additional data to support a required updated hydrogeologic groundwater model for open pit mining.

6.0 History

6.1 Prior Ownership and Ownership Changes

The original property comprised 245 privately held claims, first optioned in the 1970s. Over the years, ownership and management of the property have changed multiple times, accompanied by various exploration activities. Midway Gold acquired an option on the claims through an agreement with Rex in 2001, eventually becoming the sole owner of the property as of December 31, 2004. MGC, a wholly-owned subsidiary of Midway Gold, conducted exploration drilling, sampling, mapping, and geophysics from the assignment of the project on January 1, 2005, until the suspension of exploration activities in 2015.

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

Viva assumed certain royalty and environmental bonding obligations, outstanding drill road and pad reclamation liabilities (as discussed in Item 4.6), and provided other valuable considerations, including cash payment. Additionally, Viva entered into a Royalty Deed Modification and Waiver of Claims Agreement with underlying royalty holders on the Tonopah Project. This agreement waived certain claims by the royalty holders against Midway, eliminated advance royalty payments, and restructured a burdensome sliding scale NSR into a flat 2% NSR structure in exchange for cash consideration and shares of the company.

6.2 Exploration and Development History

Mining and exploration have occurred in the vicinity of the Tonopah Project since the early 1900's. The Tonopah property is located in the Tonopah (or Rye Patch) Mining District. While there is no record of historic Au or Ag production at the Tonopah Project site, past production has occurred in the Tonopah Mining District to the south and the Manhattan District immediately to the north of the project area. At least one shaft and several prospect pits exist as remnants of early mining activity at the Tonopah property, but no data or descriptive information associated with that activity is available. The property was held and explored by Houston Oil and Minerals (Houston) from the 1970s through 1984. Three reverse circulation (RC) holes were drilled at the property prior to 1981, but it is unclear whether these holes were drilled by Houston or some other company.

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

Rio Algom Ltd., in conjunction with CDA, optioned the property in 1989 and completed a similar exploration program, including 42 RC holes. This program was completed in an area to the north-northwest of the Tonopah Project (now called the Midway Hills Area) and yielded a best intersection of 16.9 g/t of Au over 4.6 m.

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

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

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

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

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

6.3 Historical Mineral Resource Estimates

In 2005 Gustin and Ristorcelli completed a Mineral Resource estimate as part of the “Updated Summary Report Midway Gold Project, Nye County, Nevada (2005)”. The estimate was based on 195 drill holes which was supported by a database of 8,860 sample composites. The Inferred Mineral Resource totaled 5.526 million short tons at 0.039 oz per short ton (opt) Au at a 0.01 opt Au cutoff.

In 2011, Gustavson Associates LLC (Gustavson) estimated a Mineral Resource for the Midway (now Tonopah) project of 114,000 short tons at 0.10 opt Au cutoff in the Inferred category, with a mean grade of 0.3017 opt Au, containing 34,400 oz Au. This Mineral Resource was disclosed in an NI 43-101 Technical Report by Midway Gold Corp, the previous owner of the property. The 2011 study focused on estimating Mineral Resources for a small underground mining target as reflected by the elevated COG.

In 2019, Gustavson QP Thomas Matthews prepared a Mineral Resource estimate for Viva based on historical drilling and the 2018 and 2019 Viva drilling with an effective date of May 15, 2019. The results of this estimate reported 2,500 kilotonnes (kt) at 1.38 g/t Au in the Measured category, 6,300 kt at 0.69 g/t Au in the Indicated category, and 6,000 kt at 0.64 g/t Au in the Inferred category, disclosed in the May 2019 NI 43-101 for Viva.

In 2020, Gustavson QP Thomas Matthews updated the Mineral Resource estimate for Viva with additional 2019 drilling with an effective date of April 29, 2020. The results of this estimate reported 3,930 kt at 1.14 g/t Au in the Measured category, 8,900 kt at 0.65 g/t Au in the Indicated category, and 8,400 kt at 0.67 g/t Au in the Inferred category, disclosed in the June 2020 NI 43-101 for Viva.

The most recent Mineral Resource estimate was reported in 2022, Gustavson QP Don Hulse prepared an updated Mineral Resource estimate for Viva with additional drilling from 2020 and 2021, with an effective date of January 1, 2022. The results of this estimate reported 4,764 kt at 0.83 g/t Au in the Measured category, 11,440 kt at 0.73 g/t Au in the Indicated category, and 7,352 kt at 0.87 g/t Au in the Inferred category, disclosed in the February 2022 (amended April 2022) NI 43-101 for Viva.

The QP has not done sufficient work to classify any of the historical estimates as current Mineral Resources. Viva is not treating any of the historical estimates, nor the superseded estimates as current Mineral Resources, and they should no longer be relied upon. The current Mineral Resource estimate for the Tonopah property is disclosed in Item 14.0 of this report.

6.4 Production from the Property

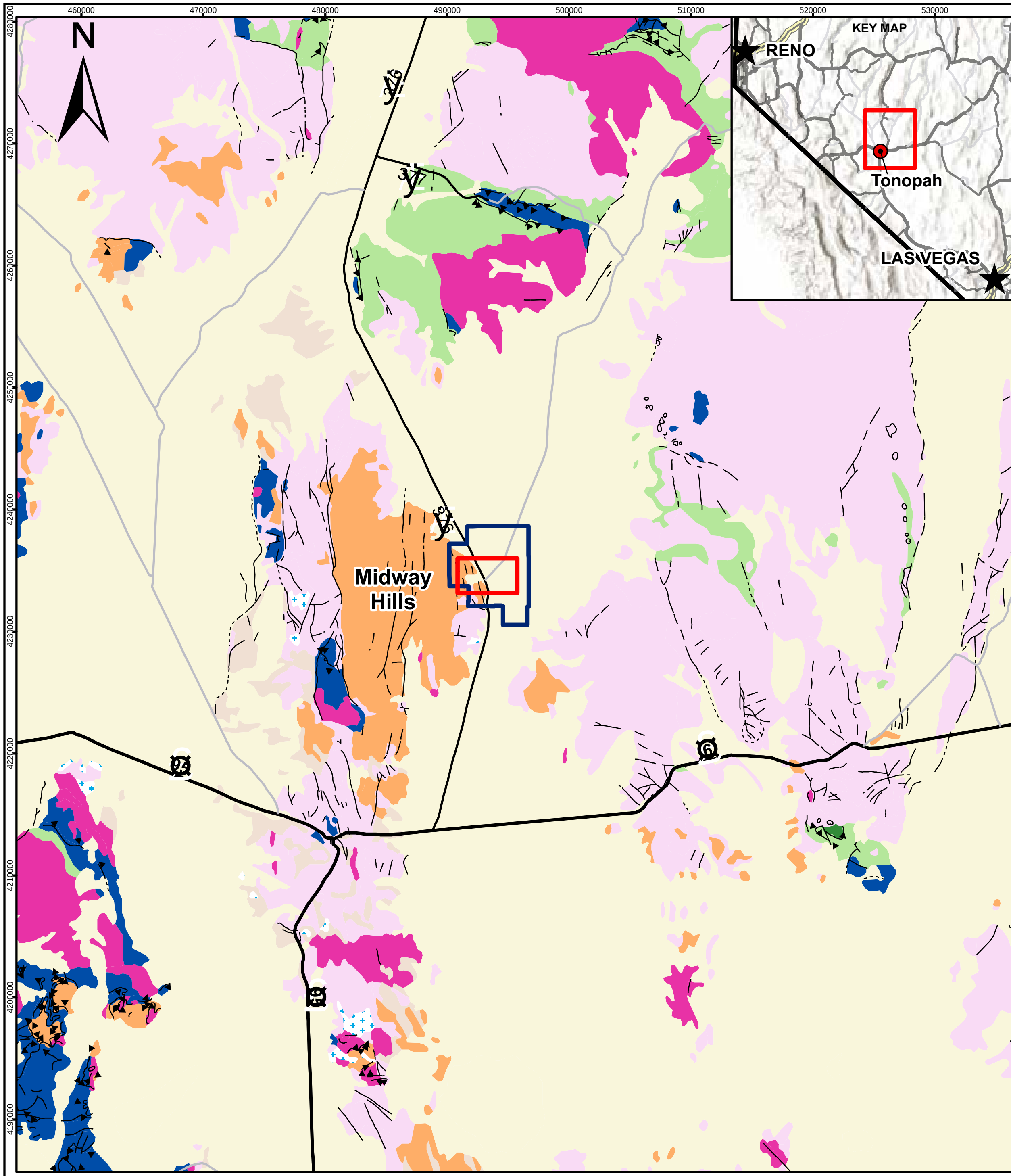
There is no record of historic Au or Ag production at the Tonopah Project site.

7.0 Geological Setting and Mineralization

7.1 Regional Geology

The Tonopah property (the Project) is situated on the northeast edge of the Walker Lane structural zone, which consists of sub-parallel, right lateral strike-slip faults that separate the Sierra Nevada batholith from the Basin and Range province (Bonham and Garside, 1979). The Project area is located in the Midway Hills and Rye Patch Valley within the eastern San Antonio Mountains, including part of the Ralston Valley. The San Antonio Mountains are covered by Miocene Red Mountain trachyandesite flows with thicknesses reaching up to approximately 305 m.

Argillite, quartzite, chert, limestone, and other fine-grained clastic rocks of the Ordovician Palmetto Formation (Op) are exposed at the eastern foot of the Tonopah Hills, near the western edge of the Tonopah property. The rocks of the Op are predominantly folded, striking northwest to east-west and dipping moderately to the north and northeast. Unconformably deposited on the Op are Miocene volcanic and volcaniclastic rocks. Valley-fill alluvial, colluvial, aeolian, and playa deposits extend east from the eastern foot of the Tonopah Hills into Ralston Valley, concealing bedrock geology over most of the Tonopah property. A regional geologic map of the area is provided as Figure 7.1.



LEGEND

- Lithology Model Boundary
- Project Claim Boundary
- U.S. Highway
- State Highway
- Secondary Roads

STATEWIDE FAULTS

- Known fault
- Queried fault
- Inferred fault
- Inferred queried fault
- Concealed fault
- Concealed queried fault
- Known thrust fault
- Inferred thrust fault

GEOLOGIC UNITS

- Igneous Hypabyssal
- Igneous Plutonic
- Igneous Tuff
- Igneous Volcanic Felsic
- Igneous Volcanic Mafic
- Metamorphic
- Sedimentary Carbonate
- Sedimentary Clastic
- Unconsolidated

0 10 20

1:500,000 KILOMETRES

NOTE(S)

REFERENCE(S)

NEVADA BUREAU OF MINES AND GEOLOGY, UNIVERSITY OF NEVADA, RENO
CRAFFORD, A.E.J., 2007, GEOLOGIC MAP OF NEVADA: U.S. GEOLOGICAL SURVEY DATA SERIES 249, 1 CD-ROM, 46 P., 1 PLATE.
PROJECTED COORDINATE SYSTEM: NAD 1983 UTM ZONE 11N

CLIENT
VIVA GOLD CORP.

PROJECT
TONOPAH GOLD PROJECT

TITLE
REGIONAL SURFACE GEOLOGY

CONSULTANT

YYYY-MM-DD	2025-08-06
DESIGNED	RK
PREPARED	RK
REVIEWED	JS
APPROVED	BT

PROJECT NO.	CONTROL	REV.	FIGURE
CA0004670.0343		0	7.1

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25mm

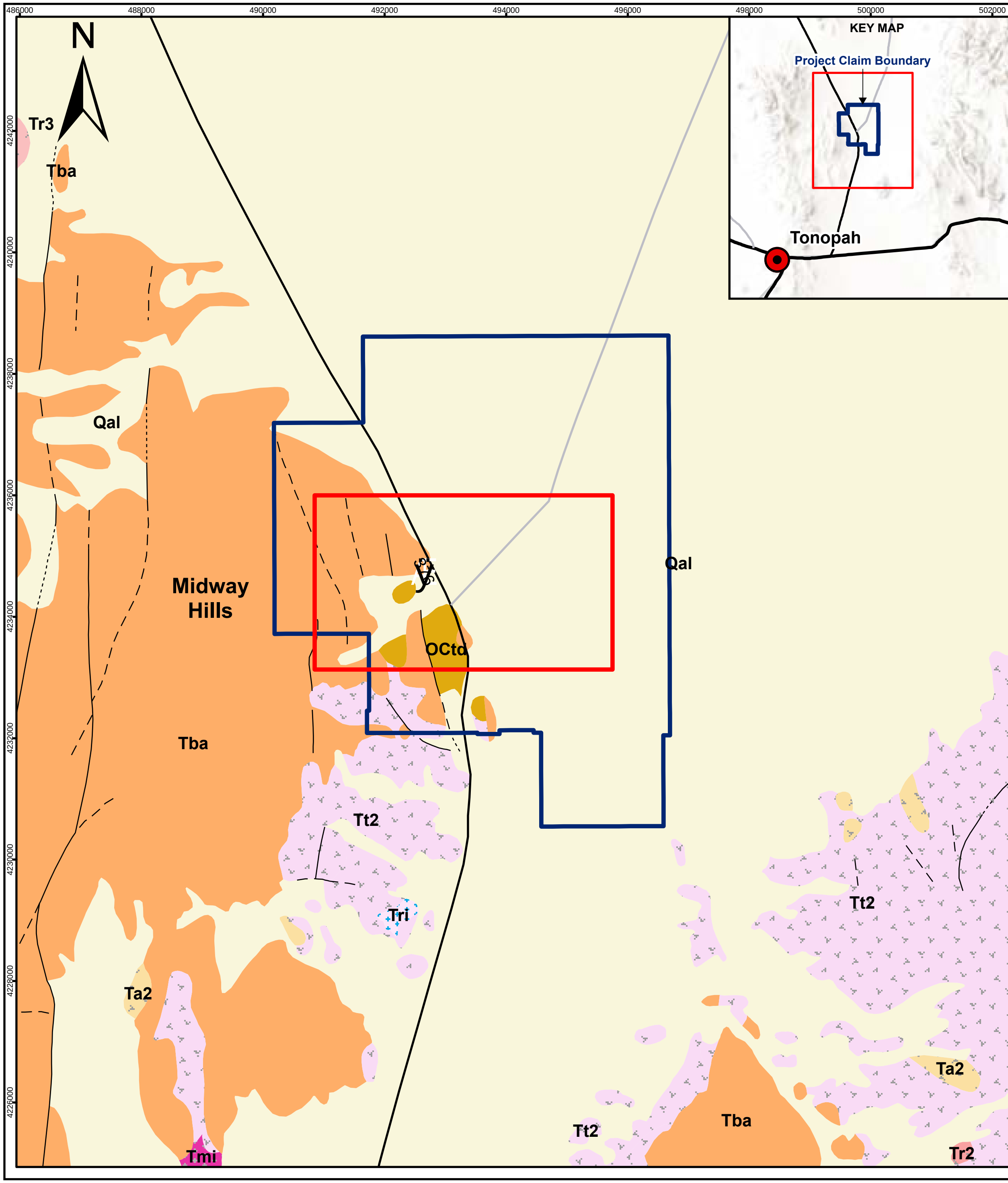
7.2 Local Geology

Local geology near the Project consists predominantly of valley fill deposits, including alluvium, colluvium, sand dunes, and playa deposits. The Au-bearing altered and mineralized zones of the Tonopah deposit are concealed by these Quaternary deposits except for a single outcrop.

Argillite, quartzite, and limestone from the Op are found in the nearby foothills of the Midway Hills, west of the property. These rocks are overlain unconformably by felsic volcanic rocks belonging to either the Rye Patch member of the Tonopah Formation (Gustin and Ristorcelli, 2005) or the Tombstone Formation (Ristorcelli and Muerhoff, 2002; Rhys, 2003). These are mainly tuffs and volcaniclastic sediments derived from tuffs, which seem to have been deposited into basins several hundred metres deep. Intermediate to mafic volcanic flows, likely the Red Mountain trachyandesite unit, cover most of the hills west of the Tonopah property. The structure is mainly defined by the northwest-trending Rye Patch fault system, characteristic of the Walker Lane structural belt.

Rhyolite dykes, ranging from 1 to 20 m in width, occur in northwest-trending dyke swarms within the Op. These dykes are generally fine to very fine-grained and often show clay mineral alteration with drusy to chalcedonic quartz veinlets, especially along the margins. These dykes may contain anomalous Au mineralization. Similar felsic dykes have been found during drilling. Some dykes, particularly those in felsic volcanic rocks near the Op nonconformity, appear to have a more mafic composition and strong clay mineral alteration.

The current understanding of bedrock geology and the distribution of mineralization and alteration in the Tonopah Project area is derived from drilling exploration results. A map of the local bedrock geology is presented in Figure 7.2.



LEGEND

- Lithology Model Boundary
- Project Claim Boundary
- State Highway
- Secondary Roads

NEVADA SURFACE GEOLOGY

- Qal - Alluvium, undifferentiated
- Tba - Andesite and basalt flows (Miocene and Oligocene)
- Tr3 - Younger rhyolitic flows and shallow intrusive rocks (Miocene)
- Ta2 - Intermediate andesite and intermediate flows and breccias (lower Miocene and Oligocene)
- Tt2 - Intermediate silicic ash flow tuff (lower Miocene and Oligocene)
- Tr2 - Intermediate rhyolitic flows and shallow intrusive rocks (lower Miocene and Oligocene)
- Tmi - Mafic phaneritic intrusive rocks (Miocene to middle Eocene)
- Tri - Rhyolitic intrusive rocks with aphanitic groundmass (Miocene to middle Eocene)
- OCtd - Shale, chert, phyllite, quartzite, and limestone (Ordovician to Cambrian)

STATEWIDE FAULTS

- Known fault
- Inferred fault
- Concealed fault

0 2 4

1:100,000 KILOMETRES

NOTE(S)

REFERENCE(S)

NEVADA BUREAU OF MINES AND GEOLOGY, UNIVERSITY OF NEVADA, RENO
CRAFFORD, A.E.J., 2007, GEOLOGIC MAP OF NEVADA: U.S. GEOLOGICAL SURVEY DATA SERIES 249, 1 CD-ROM, 46 P., 1 PLATE.
PROJECTED COORDINATE SYSTEM: NAD 1983 UTM ZONE 11N

CLIENT
VIVA GOLD CORP

PROJECT
TONOPAH GOLD PROJECT

TITLE
LOCAL BEDROCK GEOLOGY

CONSULTANT

YYYY-MM-DD	2025-08-13
DESIGNED	RK
PREPARED	RK
REVIEWED	JS
APPROVED	BT

PROJECT NO.	CONTROL	REV.	FIGURE
CA0004670.0343		0	7.2

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7.3 Property Geology

The Tonopah property contains a low-sulphidation epithermal gold system characterised by nearly vertical quartz-adularia-Au veins within argillite of the Op. Folding within the argillite can locally extend and enhance vein formation. These quartz-adularia-Au veins continue into the overlying Tertiary volcanic and volcaniclastic rocks, but a broad zone of lower grade, disseminated mineralization extends well beyond the actual veins. The rhyolitic composition of the tuffs and tuffaceous clastic rocks, along with the increased permeability of the volcaniclastic rocks, allows hydrothermal fluids to penetrate tens to hundreds of metres beyond the large veins.

The nonconformity between the Op argillite and the overlying Tertiary volcanics varies; in some areas it appears as a clear fault, while in others it resembles a paleosol. Where the paleosol is preserved, it appears as a deeply weathered rubble zone dominated by boulders of Op argillite and derived soil, grading upwards to a mixed zone of argillite and volcanic debris, potentially indicating the onset of Tertiary volcanic activity. When mineralized, this zone forms a shallowly dipping, manto-like area of mineralization similar to those found in the overlying Tertiary volcanic sequences.

High-grade Au-bearing quartz-adularia veins occur in a series of north-to-north northwest-striking en-echelon clusters along a 2.4 km northwest-trending band of mineralization. In the Op argillite, there is minimal alteration and disseminated mineralization within the wall rock along the margins of the major quartz-adularia veins, but in the Tertiary volcanic and clastic rocks, the wall rock has extensive alteration and disseminated mineralization. The primary altered and mineralized zones are overlain by alluvial gravels, sand dunes, and playa deposits. An idealised stratigraphic column based on drill core logs is presented in Figure 7.3. Descriptions of individual lithologic units identified at the Project site follow, from oldest to youngest.

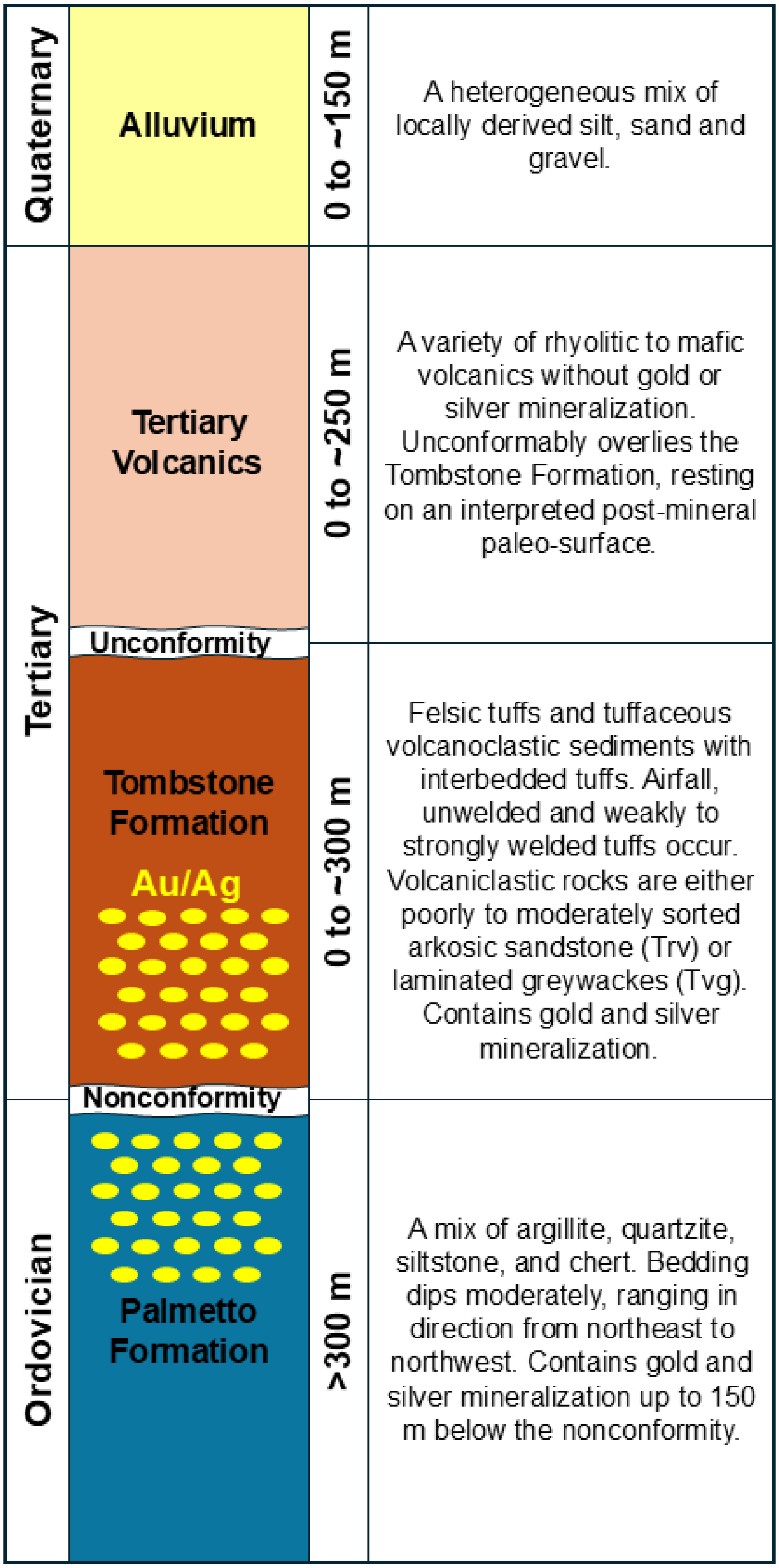


Figure 7.3: Stratigraphic Column of Tonopah Project

7.3.1 Ordovician Palmetto Formation (Op)

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

7.3.2 Nonconformity Zone

The top of the Op Formation is marked by a nonconformity at its contact with overlying volcanic rocks, representing a gap of approximately 400 million years (Ma). The upper argillite layer is heavily weathered where it meets Miocene and younger volcanic deposits, suggesting prolonged exposure as an erosional surface. No post-Paleozoic sedimentary rocks were deposited until mid-Cenozoic volcanism began. This nonconformity likely persisted through the Paleocene and Eocene. Evidence of erosion is present in both core samples and RC chips, showing a pronounced zone of bleaching and clayey rubble with iron oxides before transitioning to unweathered argillite. Some cores exhibit classic paleosol features, including soil, gravel, and cobbles, while others show minor or absent paleosol, indicating a fault between Paleozoic and Tertiary layers. When present, the paleosol indicates high permeability and porosity and is often mineralized.

7.3.3 Tertiary Intrusive Rocks

Fine to medium-grained aphanitic felsic dykes and sills intrude the Op and Tombstone Formations, often filling faults. These altered, mineralized intrusions resemble those in the Midway Hills and are likely of the same age. They are younger or partially coeval with the Tombstone Formation and occurred during or before the mineralization/alteration events within the Tombstone.

7.3.4 Tertiary Tombstone Formation

Felsic tuffs and volcanoclastic sediments of the Tertiary Tombstone Formation unconformably overlie the Op. Subsurface mapping and correlation of horizons in drill core or cuttings is challenging due to textural destruction by hydrothermal alteration and rapid lateral facies changes (Rhys, 2003). Generally, the majority of Tertiary rocks are tuffaceous volcaniclastic rocks with interbedded tuffs. Airfall, unwelded and weakly to strongly welded tuffs occur. Away from strongly altered zones where sedimentary textures are preserved, the volcaniclastic rocks are either poorly to moderately sorted arkosic sandstone (Trv) or laminated greywackes (Tvg). While generally destroyed in more altered zones, the greywackes of the Tvg appear to be varved and generally include calcareous siltstones and occasional limestone beds. These deposits suggest formation within a caldera-like basin. Drilling in the northwest portion of the project also indicates the presence of a caldera margin. The group of mineralized Tertiary volcanics are referred to as Tertiary volcanic lower (TVL).

7.3.5 Tertiary Volcanics (Post-mineral)

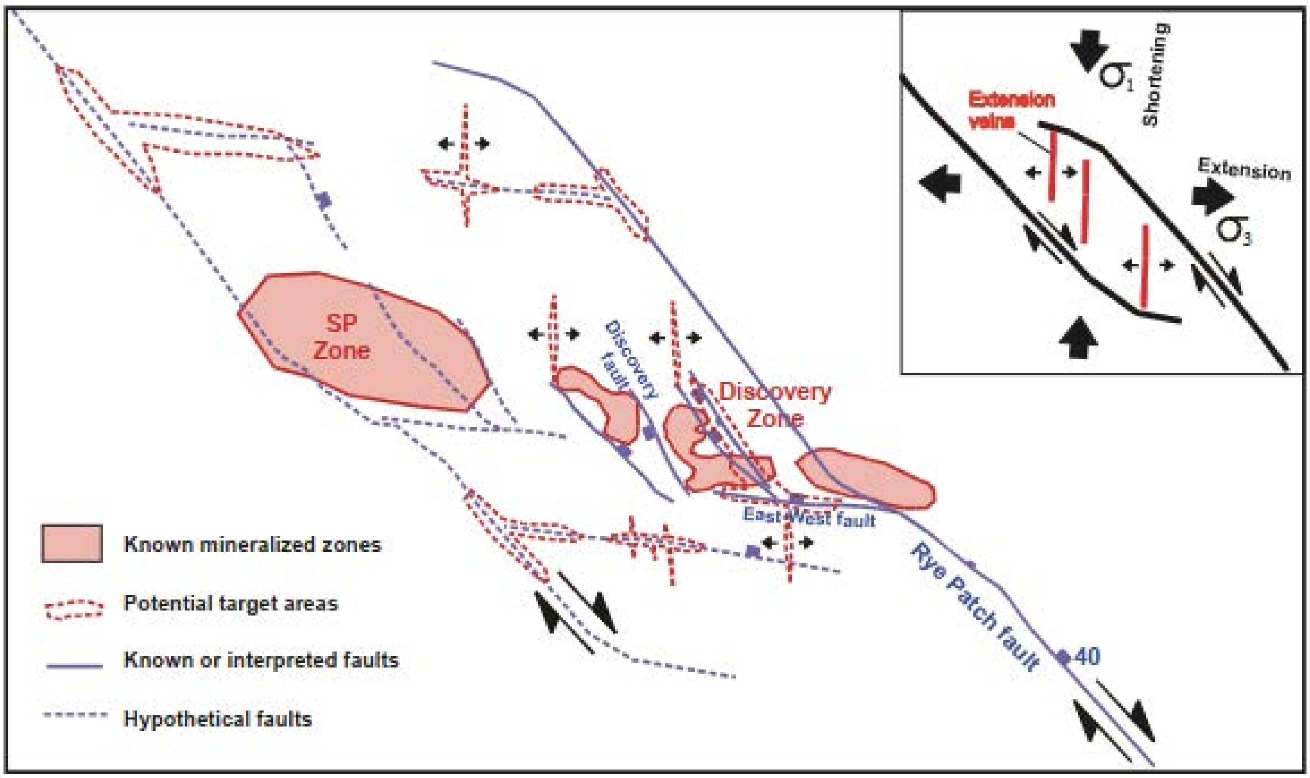
A variety of rhyolitic to mafic volcanics unconformably overlie the Tombstone Formation, resting on an interpreted post-mineralization paleo-surface. These units have not been studied in any detail. This upper, unmineralized unit is referred to as Tertiary volcanic upper (TVU).

7.3.6 Quaternary Deposits

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

7.3.7 Structural Geology

The Tonopah Project is located within the tectonically complex Basin and Range Province of the western United States. Structural geology significantly influences the distribution of mineralization and alteration at Tonopah. The Rye Patch fault system is a complex, oblique-slip fault system believed to be associated with north-south trending compressional stress common in the Walker Lane structural trend. The system is bounded by north-west trending pre-mineral faults to the east and west. Where these two bounding faults overlap, a complex relay zone has formed between them, linking them and accommodating the offset of the whole structural system. This relay zone, recognized by Mr. David Rhys of Panterra Geoservices, Inc., and later by WSP during Three-Dimensional (3D) structural modelling, includes steeply dipping north-south striking extension fractures. The inferred structural patterns in the relay zone are shown in Figure 7.4 (Rhys, 2003).



Source: (Rhys, 2003)

Figure 7.4: Hypothetical Structural Model for the Central Midway Property During Mineralization

Detailed structural studies (including Rhys, 2003) of bedrock exposures and oriented core from 22 drill holes indicate that these bounding faults dip moderately to the northeast, accommodating right lateral strike-slip movement. These studies also indicate that development of alteration and mineralization occurred between these two bounding faults, with veins and hydrothermal breccias developing along sub-parallel, north-south extensional faults that form the complex network of linking second order structures in the relay zone. Extensional relay zones, such as at Tonopah, are typically areas of lower pressure with extensive faulting and fracturing, making them favorable for mineralization. One significant fault, the Discovery fault, identified by Mr. Rhys, accommodates extension within the relay zone and is described as a steep, southwesterly dipping clay gouge filled fault with approximately 200 ft (~60 m) of apparent normal displacement (Rhys, 2003). This fault displaces Au mineralization and shows maximum offset at the north end, decreasing to zero towards the south, characteristic of a scissor fault.

Walker Lane faulting occurred around 12 Ma, or middle to middle late Miocene. Given that the age dates for the deposit, limited though they are, suggest early late to middle early Miocene age, the main mineralized structures may be purely basin-and-range extensional features. A pull apart extensional basin would present the same structural relationships, with the subsequent rotation from E-W to NE-SW resulting from ductile décollement associated with later Walker Lane faulting.

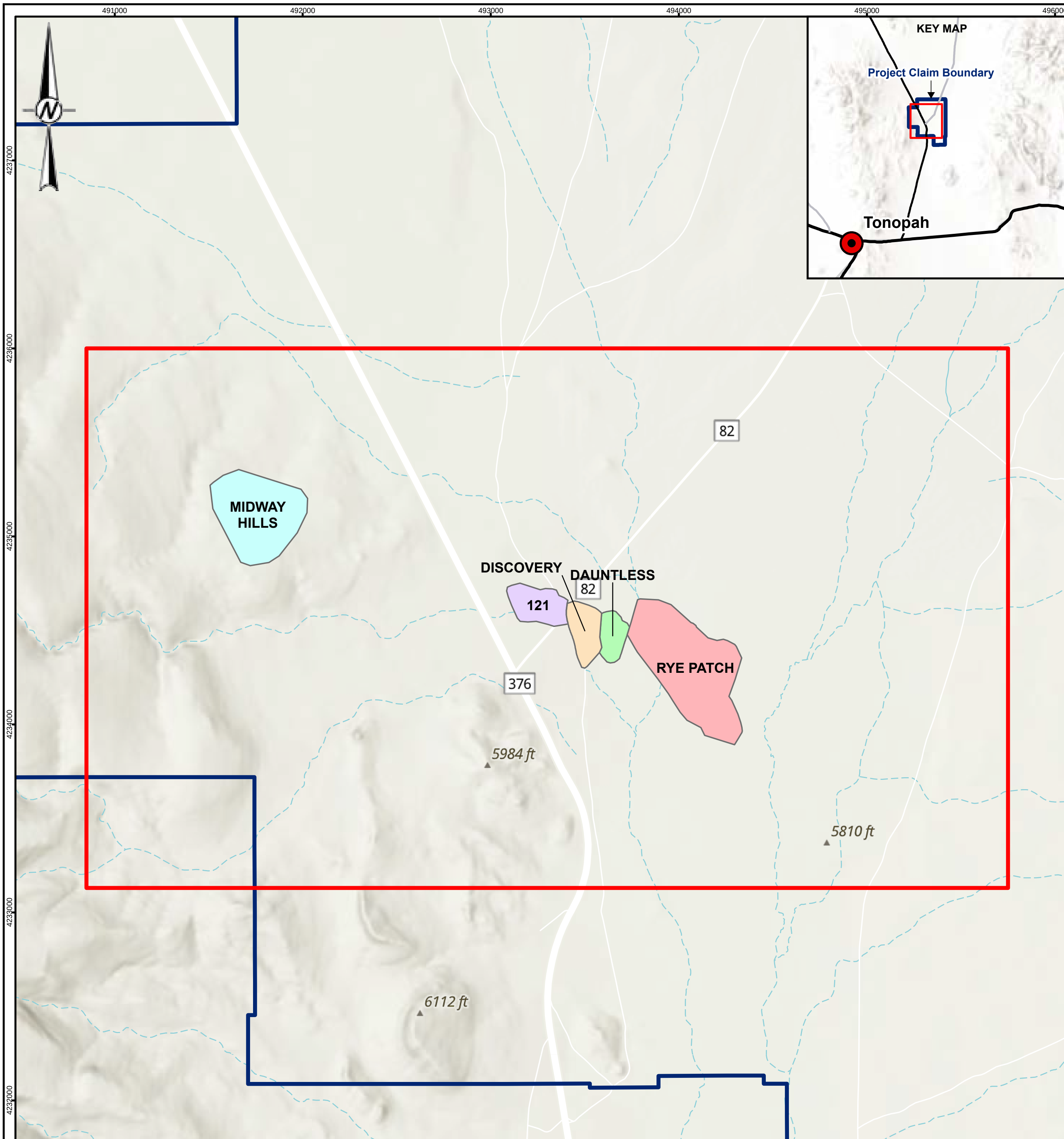
7.4 Mineralization and Alteration

Two overlapping mineralized trends have been identified in drilling. The primary trend runs parallel to the west-northwest Rye Patch Fault System, bearing 290-300 degrees over at least 3,000 m, and 500 m width, and open along strike.

Significant alteration and mineralization are localized within a low-angle zone near the TVL/Op nonconformity, which includes and often parallels the erosion surface of the Op, as well as several facies in the Tertiary volcanics, particularly where veins and mineralized structures intersect this contact zone. It is interpreted that ascending fluids entering the contact zone, depositing precious metals in a favourable horizon in the base of the Tertiary volcanics.

Five main mineralization zones are shown on Figure 7.5. The highest-grade mineralization occurs in the Discovery and Dauntless zones. In the western part of the deposit in the Discovery and Dauntless zones, and the down-drop side of the Discovery Fault (121 zone), mineralization occurs above and below the TVL/Op nonconformity. In the Rye Patch area in the eastern portion of the deposit, mineralization primarily occurs in the TVL unit. Further to the west is a minor mineralization zone with limited drilling, named Midway Hills. As discussed in Item 7.3.7, mineralization appears to be confined to the structural relay zone.

Au mineralization is concentrated in secondary extensional fractures that range from 345 to 360 degrees strike, are near-vertical in dip, and host veins and hydrothermal breccias with higher grade mineralization, ranging from 1.0 to over 30 PPM Au. These extensional fracture zones are best represented in drilling in the Discovery and Dauntless zones, primarily in zones of massive quartz-adularia alteration in volcanic and volcaniclastic rocks and in veins, breccias, and silicified faults. Higher Au grades are associated with a variety of siliceous veins, and veinlets, including chalcedonic, bladed or drusy quartz, and quartz +/- iron oxide cemented breccias; these textures all suggest boiling was occurring. In the Dauntless Zone, the quartz-adularia forms a funnel-shaped zone that expands upward into the Tombstone Formation above the moderately dipping nonconformity.



LEGEND

MINERALIZATION ZONES

- 121
- DAUNTLESS
- DISCOVERY
- MIDWAY HILLS
- RYE PATCH

- LITHOLOGY MODEL BOUNDARY
- PROJECT CLAIM BOUNDARY

0 500 1,000

1:30,000 METRES

NOTE(S)

REFERENCE(S)

PROJECTED COORDINATE SYSTEM: NAD 1983 UTM ZONE 11N

CLIENT

VIVA GOLD CORP.

VIVA GOLD CORP

PROJECT

TONOPAH GOLD PROJECT

TITLE

MINERALIZATION ZONES

CONSULTANT

wsp

YYYY-MM-DD	2025-08-06
DESIGNED	RK
PREPARED	RK
REVIEWED	JS
APPROVED	BT

PROJECT NO.	CONTROL	REV.	FIGURE
CA0004670.0343		0	7.5

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25mm

0

The Discovery Zone is the most densely drilled zone at the Tonopah property. Drill holes have intercepted a large number of veins, breccia-veins, and mineralized structures occurring in sub-parallel clusters three to six m apart. Vein and mineralized structure thicknesses vary from a few cm to over six m, averaging two m (MGC Resources, 2008). Continuity of veins, vein zones and structures are projected, but not certain, over approximate north-south strike lengths of 30 to 100 m, and with vertical dimensions that may locally exceed 100 m. Continuity of Au mineralization and Au grades coincide, approximately, with projections of the veins and structures, but becomes far less certain at progressively higher Au grade cutoffs. At lower COGs, good continuity develops between zones, veins and structures, due largely to lower grade mineralization associated with the nonconformity between the Op and the overlying TVL. There is a tendency for well-defined veins in the Op to branch and splay upward into a broader network of veins, vein zones, veinlets in the overlying TVL. Visible Au is commonly observed in and along the edges of veins, is frequently associated with hematite, and occurs locally in coarse form. Dendritic Au has been observed in core.

In the Discovery Zone, a vertical sequence of veining is observed in the extensive K-feldspar-quartz alteration zone in the Tertiary sequence. Upper parts have rare veinlets of opaline to chalcedonic quartz, sometimes with fine-grained drusy quartz-lined cavities. Below this, significant Au values are present within and above a zone containing bladed quartz veins and veinlets with lattice-like textures of quartz after calcite. This two to seven m thick, shallow northeast-dipping textural zone indicates a boiling level in the hydrothermal system. Chalcedonic quartz veinlets become more common downward toward the Op nonconformity. (Rhys, 2003)

Alteration outside of the quartz-adularia zones in the Tombstone Formation is characterized as strong argillic alteration. Oxidation is extensive, with local relict patches of incompletely oxidized pyrite in altered areas.

Siliceous structures oriented similarly to those in the Tombstone Formation occur in the underlying Op. Veins hosted in the Op form well-defined discrete veins and hydrothermal breccias up to two m wide (MGC Resources, 2008). Alteration in the Op is characterized by argillic alteration extending up to approximately 100 m below the nonconformity with the Tombstone Formation. Intense argillic alteration is typically limited to a zone within one to eight m of the nonconformity, with gradual weakening of bleaching and clay alteration to greater depth, this coincides with the deeply weathered surface developed at the top of the Op. Locally, the zone of intense quartz-adularia alteration in the overlying Tombstone Formation may extend into the Op for a few metres (Rhys, 2003).

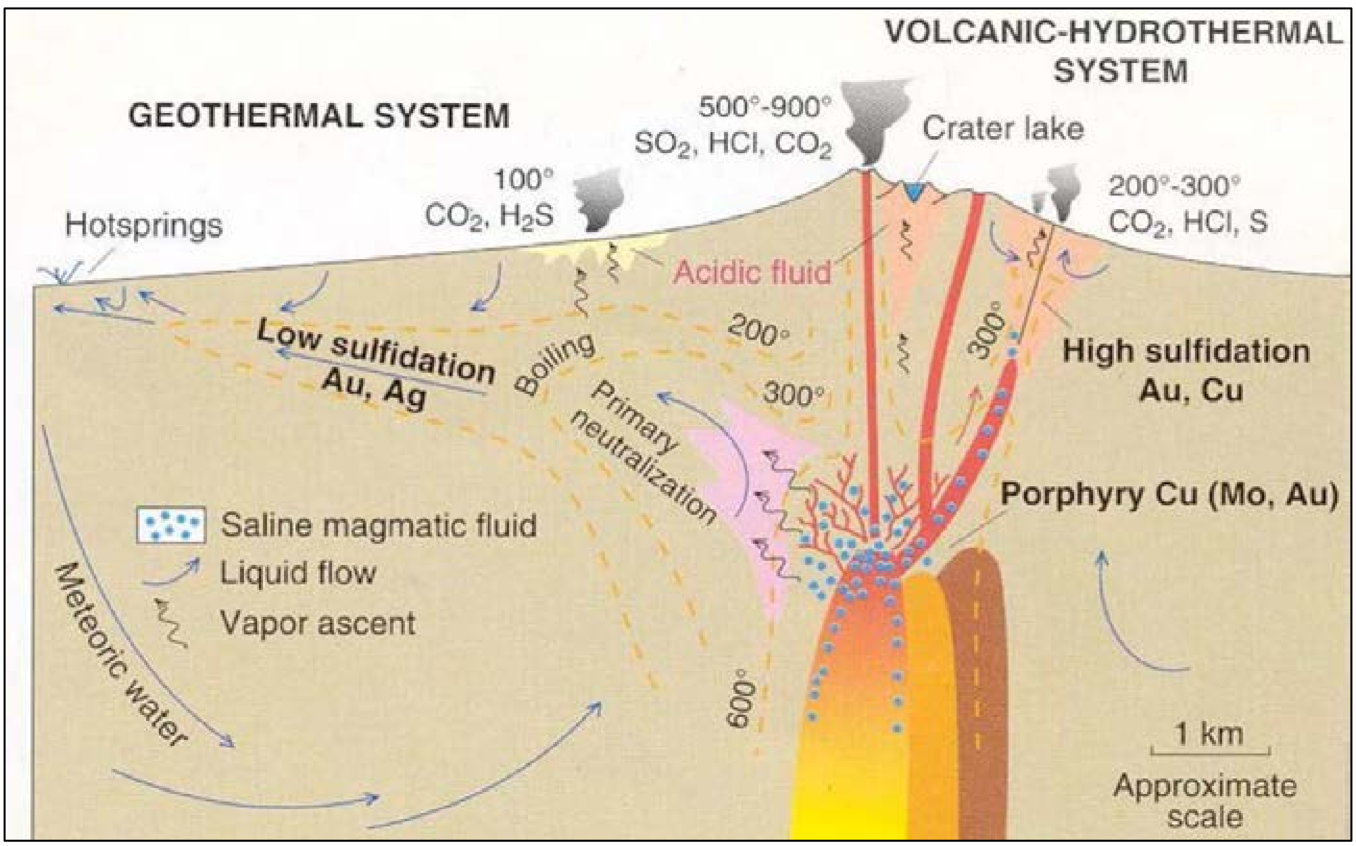
8.0 Deposit Types

Alteration and mineralization at the Tonopah property are typical of low-sulphidation, volcanic-hosted epithermal gold deposits found elsewhere in Nevada and around the world. They are typically associated with rhyolitic to andesitic volcanic centers which have formed in the shallow crustal settings. Mineralization is often structurally controlled.

The Tonopah deposit type is characterized by overall low original sulphide content, and quartz-adularia and clay-sericite alteration assemblages, among others. Vein textures are indicative of high level, near surface emplacement and include void fills, crustiform coatings, colloform banding, and comb structures. Similar deposits in Nevada have proven to be economic, including the Round Mountain, Midas and Bullfrog deposits.

The proximity and similarities of the Tonopah property to other Au deposits does not, on its own, indicate that the Tonopah property should be similarly mineralized. Figure 8.1 illustrates a general example of how low-sulphidation epithermal gold deposits are formed (Taksavasu, 2017, citing Hedenquist and Lowenstern, 1994).

The exploration programs and results presented in this report were designed and implemented with this deposit type in mind.



Note: Figure reproduced from Taksavasu (2017), citing Hedenquist and Lowenstern (1994).

Figure 8.1: Schematic of a Volcanic-Hydrothermal System

9.0 Exploration

This Item focuses on non-drilling components of the 2018 to 2024 exploration programs. Detailed information on exploration drilling programs from 2018 to 2024 is included in Item 10.0 (Drilling). Information on exploration programs prior to 2018 is covered in Item 6.0 (History).

9.1 Historical Exploration Activities

Exploration activities on the Tonopah property have spanned several decades, beginning in 1986 when Messrs., Patton and Schmidt staked claims. CDA conducted preliminary geological, geochemical, and geophysical surveys in 1988, followed by similar programs by Rio Algom Ltd. in 1989. From 1992 to 1996, Kennecott carried out multiple geophysical programs, including airborne magnetic, airborne EM, gravity, and CSAMT surveys.

Between 2002 and 2004, Newmont, alongside Midway Gold, conducted extensive geophysical surveys, including ground and airborne radiometric, magnetic, EM/TEM, gravity, CSAMT, and IP/resistivity. Additionally, a small-scale self-potential test was conducted over the Discovery Zone, and metallurgical testing was carried out. From 2004 to 2012, Midway Gold collected geotechnical data and conducted hydrological studies on drilled holes, further enhancing the understanding of the Tonopah property.

The historical exploration work carried out on the Tonopah Project is described in detail in Item 6.2.

9.2 Viva Exploration Activities

Viva's exploration activities have been primarily limited to drilling and sampling, however, there has been some exploration activities since 2018.

A significant amount of geophysical data was collected during historical exploration, as described in Item 6.2. The geophysical database is extensive with uniformly high-quality data. It contains data for 8 geophysical techniques applied to Au exploration. All the data were generated by either Kennecott in the early 1990's and/or Newmont in the early 2000's. In 2019, Viva commissioned Wright Geophysics to review the historical geophysical data in conjunction with the updated geological modelling and drilling information to generate additional insights and drilling targets. WSP conducted a further review of the geophysical data during the recent model updates and incorporated the CSAMT data into the geological model to help refine the structural interpretation.

In 2022, an aerial flyover of the Tonopah property was performed by MWH Geo-Surveys International Inc. using a Professional Mapping unmanned aerial vehicle (UAV), producing a Geographic Tagged Image File Format (GeoTIFF) orthophoto and digital surface model (DSM) at a resolution of 25 cm. WSP converted the DSM to a topography triangulation surface with a 10 m triangle resolution using Global Mapper GIS software for use in the geological model and for collar validation checks.

10.0 Drilling

10.1 Summary

A total of 626 drill holes totaling 90,716 m has been drilled at the Tonopah Project to date, including 458 (73%) RC and 168 (27%) DD core holes. A total of 477 (81%) of these drill holes were completed prior to the acquisition by Viva. Existing drill holes include 14 holes drilled by Midway Gold for hydrology studies, and 11 DD holes drilled for geotechnical studies. WSP has not included 148 drill holes which are outside the current claim boundaries in the Thunder Mountain area for which Viva owns the historical drilling data. Drill hole data for the Project is summarized in Table 10.1, and drill hole locations are shown on Figure 10.1. Figure 10.2 through Figure 10.5 illustrate representative cross-sections through the 121, Discovery, Dauntless and Rye Patch zones.

Viva conducted an initial drilling program in 2018 designed first to confirm the historical database and secondarily to extend mineralization by targeting historical areas of Inferred Mineral Resources which had the potential to be upgraded, as well as to provide fresh material for metallurgical test work. Viva has conducted drill programs annually from 2019 through 2024 for exploration and infill drilling. Viva has drilled a total of 15 DD holes and 106 RC holes totaling 17,984 m from 2018 to 2024.

Given the presence of coarse and visible Au at Tonopah, care must be taken with regard to sample collection during both DD and RC drilling. Water used during RC drilling may contribute to sample bias, and core samples need to be large in order to provide a representative analytical sample.

Due to the complex relationship between subvertical high-grade mineralization and low-angle, lower-grade mineralization, it is difficult to estimate true thicknesses for the various drill hole intersections. In general, it is expected that true thicknesses for these intersections are 70-80% of the lengths indicated.

Table 10.1: Tonopah Project Drill Hole Summary

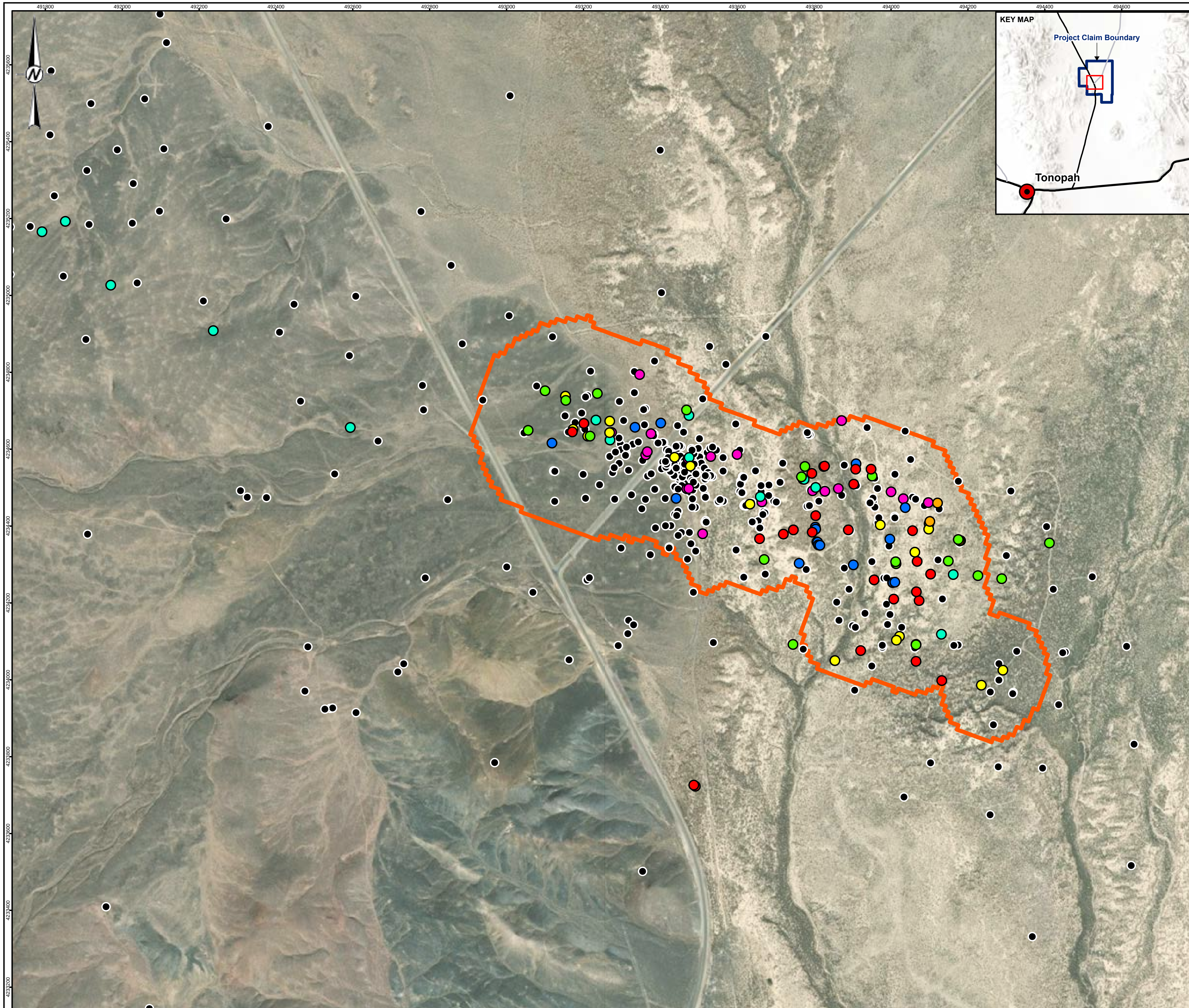
Company	Year	RC		Core		Total No. of Drill Holes	Total Depth (m)
		Count	Depth (m)	Count	Depth (m)		
Coeur d'Alene	1988	3	328			3	328
Rio Algom	1990-1991	41	6,027			41	6,027
Kennecott	1992-1996	132	20,347	4	553	136	20,900
Bob Warren	1994	3	361			3	361
Tombstone	1997	14	1,980			14	1,980
Newmont	2002-2004	74	11,890	88	12,932	162	24,822
Midway Gold Corp.	2005-2008, 2011	85	11,545	61	6,769	146	18,314
Viva Gold Corp.	2018	16	2,086	4	575	20	2,661
	2019	16	2,169			16	2,169
	2020	11	1,928	5	602	16	2,530
	2021	4	637			4	637
	2022	16	2,499	6	1,301	22	3,799
	2023	18	2,574			18	2,574
	2024	25	3,613			25	3,613
Total		**458**	**67,984**	**168**	**22,732**	**626**	**90,716**

10.2 Historical Drilling

The DD and RC logging and drilling conditions prior to 2018 have been described previously in several Technical Reports for the Project. This Item presents a summary of this information. Further details on these programs can be found in Item 6.2.

10.2.1 Early Drilling (1987 to 2002)

Drilling prior to 2002 was mostly completed using RC drilling techniques and the drill holes were 95 millimetre (mm) (3¾-inch) in diameter. Button tricone drill bits were used near the surface, with a down-hole hammer used in compacted and consolidated material. The geological sample was recovered through the center of a double wall drill pipe and discharged at the surface via a cyclone exiting directly onto a three-tiered Jones splitter. Typically, drill cuttings were sampled continuously on 1.5-to-3 m (5-to-10 ft) intervals, producing, on average, a 16-kilogram (kg) sample. Two geological samples, one washed and one unwashed, were collected from the split fraction for each sample interval. Samples for analysis were shipped by bonded carrier to recognized commercial laboratory facilities in Nevada and northern California. Additional details on commercial laboratories used in past drilling campaigns are presented in Item 11.2 of this report.



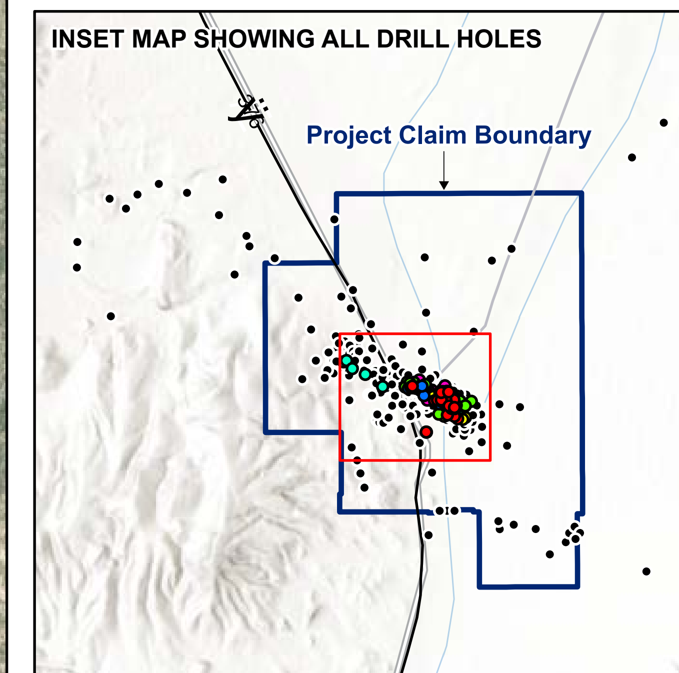
LEGEND

RESOURCE PIT SHELL OUTLINE

PRE-VIVA DRILL HOLES

VIVA DRILL HOLES

- 2018
- 2019
- 2020
- 2021
- 2022
- 2023
- 2024



0 200 400

1:10,000 METRES

NOTE(S)

REFERENCE(S)
DRILL HOLE DATA PROVIDED BY VIVA GOLD CORP.
PROJECTED COORDINATE SYSTEM: NAD 1983 UTM ZONE 11N

CLIENT
VIVA GOLD CORP.

PROJECT
TONOPAH GOLD PROJECT

TITLE
DRILL HOLE LOCATIONS

CONSULTANT

YYYY-MM-DD	2025-08-08
DESIGNED	RK
PREPARED	RK
REVIEWED	JS
APPROVED	BT

PROJECT NO.	CONTROL	REV.	FIGURE
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25mm

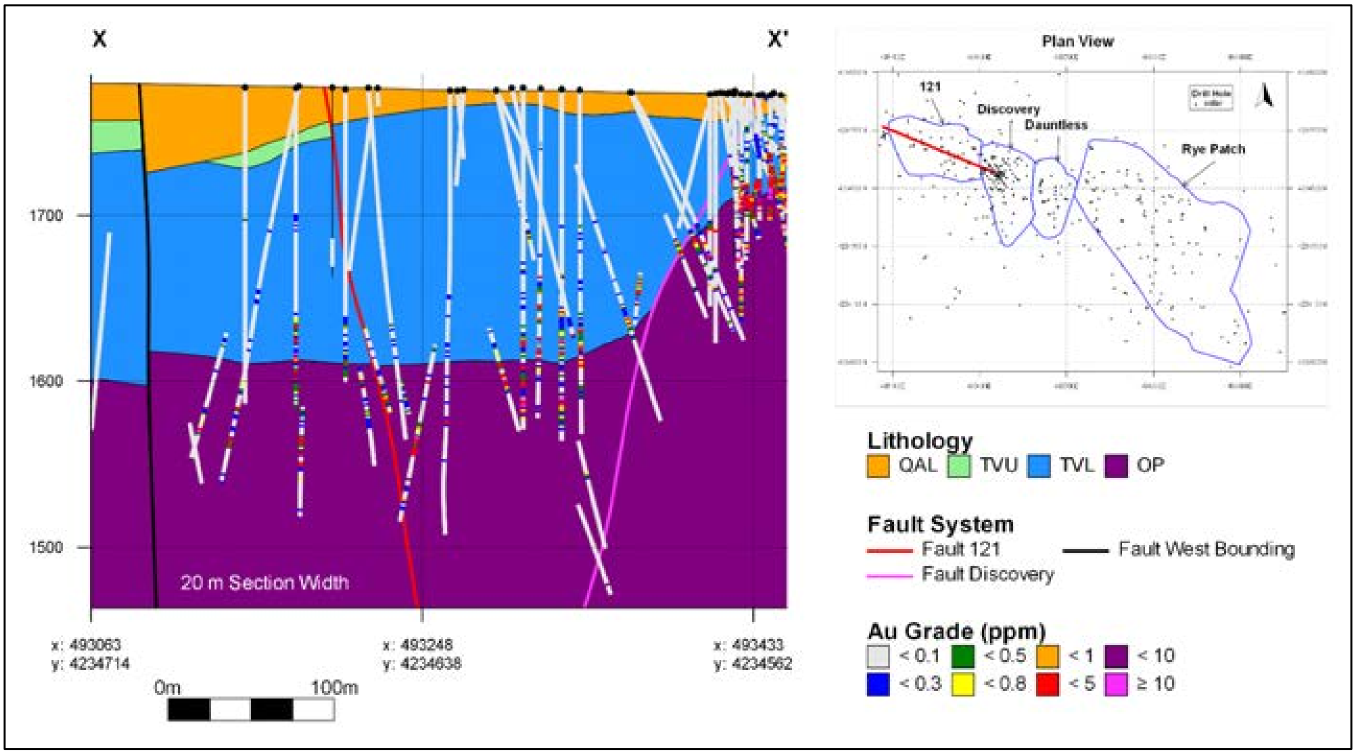


Figure 10.2: Representative Long-Section 121 Zone

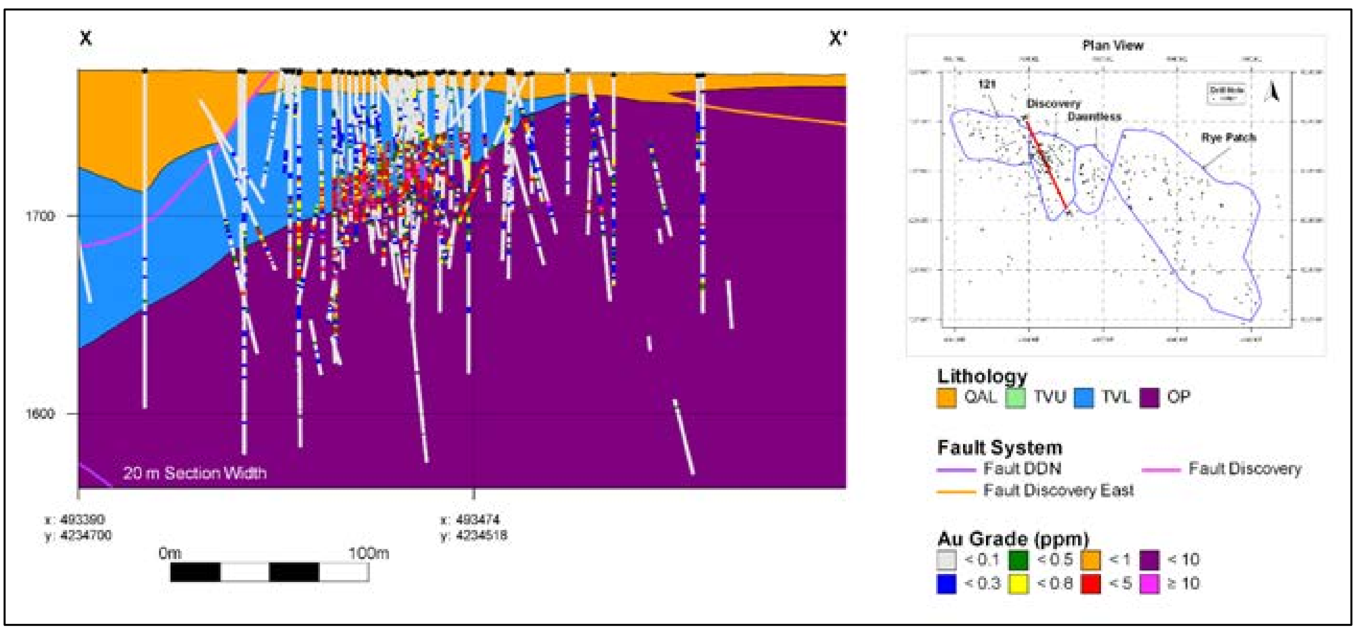


Figure 10.3: Representative Cross-Section Discovery Zone

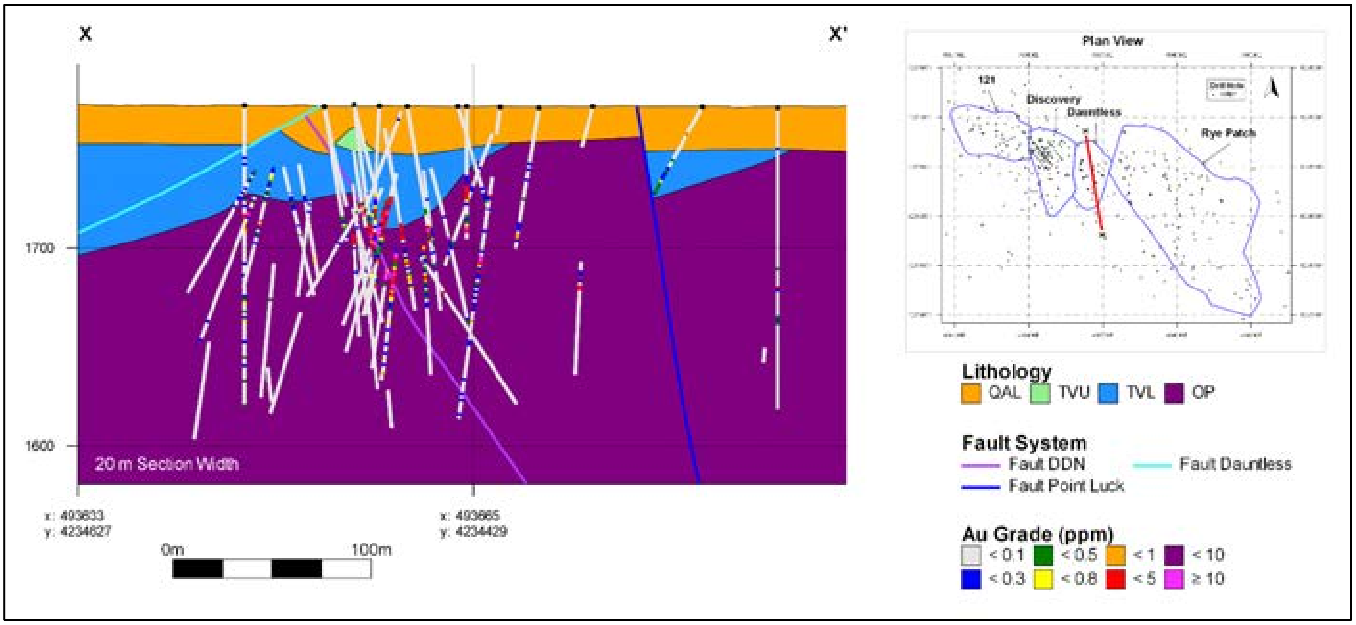


Figure 10.4: Representative Cross-Section Dauntless Zone

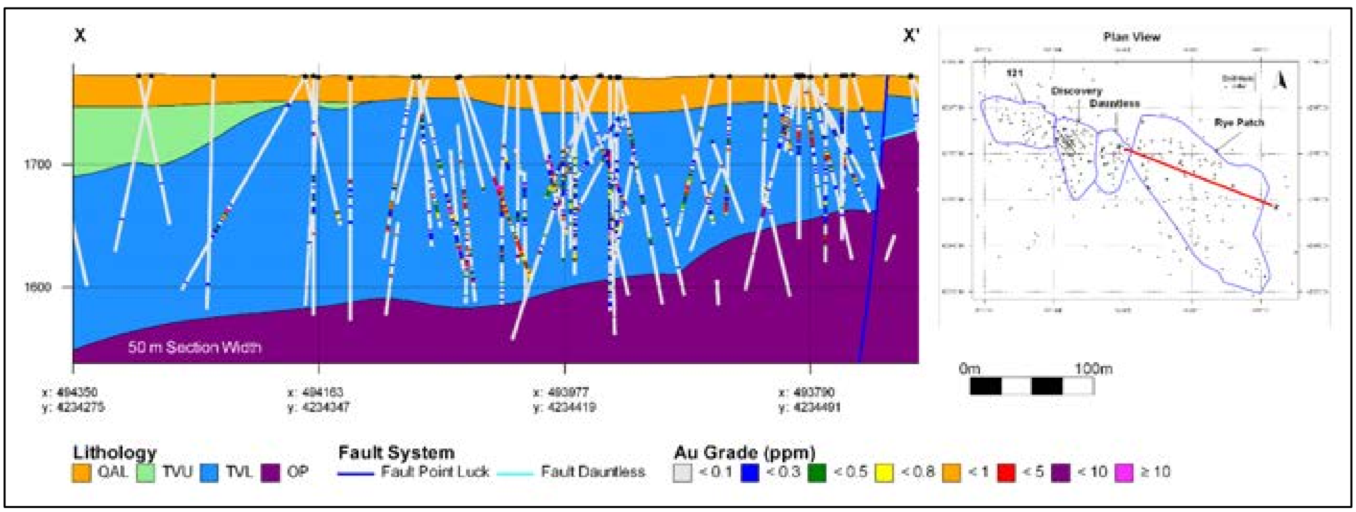


Figure 10.5: Representative Long-Section Rye Patch Zone

10.2.2 Midway Gold and Newmont Drilling (2002 to 2004)

From 2002 to 2004, the JV between Midway Gold and Newmont drilled a total of 74 RC holes totaling 11,890 m and 88 DD holes totalling 12,932 m. The 2004 drilling included 23 angle holes, with seven drilled by RC methods and the remainder as HQ-size (63.5 mm core diameter) core DD holes with RC pre-collars. The drilling aimed to test high-grade zones in the Op and explore untested areas in the greater Discovery area. The diamond drilling was conducted by Action Drilling and RC drilling by Layne Christensen and Eklund Drilling, using HQ-size core and RC drilling with a 13.3 cm (5¼-inch) hammer bit or tricone bit. Core recovery averaged 93%, with an average rock quality designation (RQD) of 51%.

Core was generally sampled in 1.5 m (5 ft) intervals, with samples split by mechanical or hydraulic splitters or sawed into halves. Samples were placed in cloth bags with unique identification numbers and stored securely until delivery to ALS Chemex (now known as ALS) for analysis. RC samples were also collected on 1.5 m intervals, with wet samples split using a rotating wet-sample splitter. Samples were labelled, stored securely, and delivered to ALS Chemex for analysis.

Photographs were taken of all core samples. The geology of all DD and RC drill holes was recorded on drill log paper forms and entered electronically into a database. RQD measurements were collected during the core logging procedure on 85 DD holes. All drill collar locations were surveyed by differential Global Positioning System (GPS). This information was recorded in both UTM Zone 11N, North American Datum 1927 (NAD27) format and local grid coordinates.

10.2.3 Midway Gold Drilling (2005 to 2011)

Midway Gold drilled 85 RC holes totaling 11,545 m and 61 DD holes totaling 6,769 m from 2005 to 2008 and in 2011. Diversified Drilling of Missoula, Montana conducted the RC drilling in 2005, and Layne Christensen, Las Vegas, Nevada, was contracted for all RC drilling from 2006 to 2008. Kirkness Diamond Drilling Co., Inc. and M2 Core Drilling and Cutting, Inc. provided core drilling services in 2007 and 2008, respectively. The 2011 core drilling campaign was completed by KB drilling of Mound House, Nevada, using a track mounted Versa KMB 1.4 Drill Rig equipped with HQ3 tools for use of split tube.

Photographs were taken of all core samples. The geology of all DD and RC drill holes was recorded on drill log paper forms and entered electronically into a database. RQD measurements were collected during the core logging procedure on seven DD holes from 2007 and 2008.

Drill hole collars were initially located with handheld GPS units, and surveyed afterward by Trimble GPS using UTM Zone 11N, NAD83 projection. Down-hole surveys for each hole were completed by International Directional Services of Elko, Nevada, using a Surface Recording Gyroscope, model DG-69. Upon completion of drilling and down-hole surveying, the holes were abandoned according to Nevada State regulations. Drill hole collars were generally marked with a wooden stake and labelled with a metal tag.

MCG drilled 14 monitoring wells in 2006 and 4 water wells in 2008. The deepest of the monitoring wells were capped with a cement monument and steel casing, as shown in Figure 10.6.

Source: WSP June 2023 Site Visit

Figure 10.6: Midway Gold Monitoring Well MW06-49HD

10.3 Viva Drilling

Viva has completed several annual drilling programs between 2018 and 2024 using both RC and DD techniques, completing 106 RC and 15 DD holes totaling 17,984 m. All drilling was supervised by Mr. Ed Bryant, a geology sub-contractor to Viva, who was responsible for sample collection and security. Table 10.2 summarizes the drilling contractors, drill hole size and total metreage by drilling program. Table 10.3 summarizes the Viva drilling since 2018.

In 2022 three rotary drill holes totalling 200 m and of differing diameters, were drilled under the supervision of Piteau for monitoring well purposes, these are not included in the drill hole database, as they were not logged or assayed.

Table 10.2: Summary of Viva Drilling Contractors by Program

Year	Type	Contractor	Count	Total Depth Drilled (m)	Hole Diameter (mm)	Start Date
2018	RC	DeLong Drilling	16	2,086	140	2018-03-17
2019		Drillrite	16	2,169	165	2019-09-03
2020		Drillrite	11	1,928	165	2020-07-21
2021		Drillrite	4	637	165	2021-07-12
2022		Major Drilling	16	2,499	140	2022-03-10
2023		DeLong Drilling	18	2,574	140	2023-04-06
2024		Major Drilling	10	1,448	165	2024-01-25
		DeLong Drilling	15	2,165	140	2024-10-01
		Total RC	**106**	**15,506**		
2018	DD	Major Drilling	4	575	HQ/89	2018-02-03
2020		Drillrite	5	602	PQ/114	2020-11-17
2022		National EWP	6	1,301	HQ/89	2022-05-10
		Total DD	**15**	**2,478**		
		Total All Viva Drilling	**121**	**17,984**		

Notes: *In 2022 Major Drilling completed one production and two monitoring wells of varying wellbore size using rotary drilling.

Table 10.3: Viva Drilling 2018 to 2024

Drill Hole ID	Easting (m)	Northing (m)	Elevation (m)	Total Depth (m)	Azimuth	Dip	Hole Type	Year Drilled
TG1801	493,600	4,234,586	1,772	105.16	340	-60	DD	2018
TG1802	493,376	4,234,640	1,774	157.28	159	-60	DD	2018
TG1803	493,871	4,234,674	1,772	171.79	200	-60	DD	2018
TG1804	493,346	4,234,794	1,774	140.51	160	-60	DD	2018
TG1805	493,510	4,234,380	1,771	77.72	250	-60	RC	2018
TG1806	493,469	4,234,497	1,773	76.20	240	-70	RC	2018
TG1807	493,472	4,234,575	1,773	94.49	195	-80	RC	2018
TG1808	493,664	4,234,463	1,772	124.97	232	-71	RC	2018
TG1809	493,531	4,234,581	1,772	100.58	220	-60	RC	2018
TG1810	493,362	4,234,585	1,774	170.69	90	-65	RC	2018
TG1811	494,000	4,234,489	1,772	149.35	203	-71	RC	2018
TG1812	494,032	4,234,471	1,772	185.93	203	-77	RC	2018
TG1813	494,097	4,234,461	1,772	164.59	211	-64	RC	2018
TG1814	493,796	4,234,492	1,772	115.82	220	-68	RC	2018
TG1815	493,771	4,234,522	1,773	112.78	198	-70	RC	2018
TG1816	493,366	4,234,593	1,774	164.59	109	-66	RC	2018
TG1817	493,273	4,234,643	1,776	201.17	64	-82	RC	2018
TG1818	493,474	4,234,497	1,773	109.73	106	-68	RC	2018
TG1819	493,863	4,234,497	1,772	118.87	197	-71	RC	2018
TG1820	493,828	4,234,490	1,772	118.87	198	-61	RC	2018
TG1901	493,804	4,234,500	1,772	65.53	50	-70	RC	2019
TG1902	493,475	4,234,578	1,773	146.30	132	-67	RC	2019
TG1903	493,660	4,234,476	1,772	140.21	261	-76	RC	2019
TG1904	494,131	4,234,118	1,772	134.11	269	-60	RC	2019

Drill Hole ID	Easting (m)	Northing (m)	Elevation (m)	Total Depth (m)	Azimuth	Dip	Hole Type	Year Drilled
TG1905	494,099	4,234,406	1,772	146.30	214	-71	RC	2019
TG1906	493,774	4,234,521	1,772	134.11	0	-90	RC	2019
TG1907	491,791	4,235,166	1,820	123.44	180	-60	RC	2019
TG1908	491,852	4,235,193	1,812	121.92	180	-60	RC	2019
TG1909	491,970	4,235,027	1,827	147.83	270	-60	RC	2019
TG1910	492,593	4,234,657	1,791	106.68	0	-90	RC	2019
TG1911	492,237	4,234,908	1,808	146.30	270	-50	RC	2019
TG1912	493,474	4,234,688	1,773	121.92	0	-90	RC	2019
TG1913	494,162	4,234,274	1,772	152.40	230	-70	RC	2019
TG1914	492,593	4,234,657	1,791	121.92	60	-60	RC	2019
TG1915	493,269	4,234,624	1,776	176.78	219	-75	RC	2019
TG1916	493,232	4,234,676	1,776	182.88	220	-65	RC	2019
TG2001	493,153	4,234,738	1,777	231.65	228	-83	RC	2020
TG2002	493,173	4,234,652	1,778	243.84	293	-79	RC	2020
TG2003	493,211	4,234,634	1,777	213.36	92	-83	RC	2020
TG2004	494,098	4,234,392	1,772	158.50	175	-69	RC	2020
TG2005	494,062	4,234,332	1,771	106.68	90	-75	RC	2020
TG2006	494,291	4,234,025	1,771	137.16	66	-81	RC	2020
TG2007	494,235	4,233,986	1,772	67.06	250	-80	RC	2020
TG2008	494,022	4,234,113	1,772	160.02	46	-69	RC	2020
TG2009	493,854	4,234,050	1,771	170.69	50	-75	RC	2020
TG2010	494,014	4,234,103	1,772	170.69	107	-88	RC	2020
TG2011	493,268	4,234,643	1,776	268.22	191	-87	RC	2020
TGM2001	493,478	4,234,556	1,773	99.97	200	-75	DD	2020
TGM2002	493,633	4,234,457	1,772	104.24	30	-75	DD	2020
TGM2003	493,268	4,234,673	1,775	153.47	270	-85	DD	2020
TGM2004	493,972	4,234,403	1,772	151.03	0	-90	DD	2020
TGM2005	493,437	4,234,579	1,773	93.57	90	-80	DD	2020
TG2101	494,181	4,234,362	1,772	201.17	110	-60	RC	2021
TG2102	494,121	4,234,460	1,772	207.26	180	-70	RC	2021
TG2103	494,101	4,234,412	1,772	182.88	225	-60	RC	2021
TG2104	494,178	4,234,362	1,772	45.72	n/a	n/a	RC	2021
TG2201	493,217	4,234,634	1,777	192.02	194	-71	DD	2022
TG2202	493,100	4,234,752	1,778	218.69	219	-72	DD	2022
TG2203	493,056	4,234,649	1,780	251.76	66	-74	DD	2022
TG2204	493,236	4,234,745	1,776	237.90	17	-80	DD	2022
TG2205	494,226	4,234,271	1,772	203.00	25	-70	DD	2022
TG2206	494,149	4,234,309	1,771	197.21	31	-71	DD	2022
TG2207	494,175	4,234,365	1,772	195.07	0	-90	RC	2022
TG2208	494,412	4,234,356	1,772	198.12	225	-60	RC	2022
TG2209	494,013	4,234,302	1,771	195.99	25	-70	RC	2022
TG2210	493,776	4,234,555	1,773	115.82	335	-70	RC	2022
TG2211	493,803	4,234,397	1,772	152.40	340	-75	RC	2022
TG2212	493,825	4,234,555	1,773	152.40	177	-81	RC	2022
TG2213	493,468	4,234,702	1,773	152.40	180	-65	RC	2022
TG2214	493,670	4,234,313	1,771	152.40	335	-60	RC	2022
TG2215	493,154	4,234,727	1,778	213.36	28	-69	RC	2022
TG2216	494,288	4,234,263	1,772	140.21	360	-60	RC	2022
TG2217	494,065	4,234,092	1,772	121.92	0	-90	RC	2022
TG2218	494,065	4,234,091	1,772	117.35	200	-65	RC	2022
TG2219	494,012	4,234,307	1,771	121.92	170	-80	RC	2022

Drill Hole ID	Easting (m)	Northing (m)	Elevation (m)	Total Depth (m)	Azimuth	Dip	Hole Type	Year Drilled
TG2220	493,951	4,234,530	1,772	164.59	360	-70	RC	2022
TG2221	493,745	4,234,092	1,771	182.88	20	-60	RC	2022
TG2222	493,767	4,234,528	1,773	121.92	335	-70	RC	2022
TG2301	493,803	4,234,397	1,772	164.59	60	-60	RC	2023
TG2302	493,807	4,234,359	1,772	124.97	337	-76	RC	2023
TG2303	493,809	4,234,358	1,772	53.34	120	-80	RC	2023
TG2304	493,902	4,234,299	1,772	140.21	345	-71	RC	2023
TG2305	493,761	4,234,303	1,773	121.92	0	-90	RC	2023
TG2306	494,004	4,234,253	1,771	117.35	0	-90	RC	2023
TG2307	494,005	4,234,253	1,771	152.40	55	-60	RC	2023
TG2308	493,997	4,234,366	1,772	152.40	0	-90	RC	2023
TG2309	494,037	4,234,448	1,772	198.12	196	-70	RC	2023
TG2310	493,909	4,234,562	1,771	198.12	210	-70	RC	2023
TG2311	493,441	4,234,472	1,773	121.92	90	-75	RC	2023
TG2312	493,334	4,234,657	1,775	249.94	239	-83	RC	2023
TG2313	493,401	4,234,668	1,774	170.69	0	-90	RC	2023
TG2314	493,118	4,234,616	1,778	201.17	0	-90	RC	2023
TG2315	494,010	4,234,254	1,771	91.44	140	-75	RC	2023
TG2316	493,811	4,234,351	1,772	114.30	210	-75	RC	2023
TG2317	493,815	4,234,350	1,772	91.44	170	-75	RC	2023
TG2318	493,804	4,234,392	1,772	109.73	250	-75	RC	2023
TG2401	493,827	4,234,556	1,773	105.16	270	-70	RC	2024
TG2402	493,903	4,234,509	1,771	128.02	268	-68	RC	2024
TG2403	493,908	4,234,548	1,771	128.02	273	-70	RC	2024
TG2404	494,068	4,234,308	1,771	158.50	273	-72	RC	2024
TG2405	494,103	4,234,275	1,771	158.50	270	-71	RC	2024
TG2406	494,132	4,233,998	1,772	140.21	320	-70	RC	2024
TG2407	494,056	4,234,388	1,772	176.78	268	-69	RC	2024
TG2408	493,804	4,234,427	1,772	158.50	270	-70	RC	2024
TG2409	493,746	4,234,390	1,773	123.44	55	-60	RC	2024
TG2410	493,889	4,234,390	1,772	170.69	269	-62	RC	2024
TG2411	493,658	4,234,367	1,772	147.83	311	-78	RC	2024
TG2412	493,720	4,234,379	1,772	143.26	270	-72	RC	2024
TG2413	493,795	4,234,384	1,772	164.59	270	-70	RC	2024
TG2414	493,794	4,234,537	1,772	152.40	64	-89	RC	2024
TG2415	493,171	4,234,645	1,778	260.60	105	-87	RC	2024
TG2416	493,201	4,234,667	1,777	114.30	0	-90	RC	2024
TG2417	493,491	4,233,723	1,771	152.40	95	-61	RC	2024
TG2417A	493,487	4,233,726	1,771	60.96	90	-50	RC	2024
TG2418	494,065	4,234,048	1,773	131.06	40	-80	RC	2024
TG2419	493,921	4,234,076	1,771	140.21	60	-60	RC	2024
TG2420	494,007	4,234,210	1,771	121.92	0	-90	RC	2024
TG2421	493,956	4,234,260	1,771	152.40	0	-90	RC	2024
TG2422	494,073	4,234,206	1,772	121.92	0	-90	RC	2024
TG2423	494,066	4,234,229	1,772	140.21	261	-73	RC	2024
TG2424	493,948	4,234,548	1,772	161.54	271	-74	RC	2024

Note: TG2104 terminated in alluvium

Table 10.4 and Table 10.5 present mineralization highlights from the 2023 and 2024 drilling programs. Further details on the mineralized intercepts and analytical results will be discussed in detail in Item 11.3 and 14.1 of this Technical Report.

Table 10.4: Mineralization Highlights from the 2023 Drilling Programs

Drill Hole	Azimuth	Dip	From (m)	To (m)	Length (m)	Au (g/t)	Ag (g/t)	Rock Type
TG2301	**60**	**-60**			**165.0**			
			110.0	116.0	6.1	0.8	2.2	TVL/Opa
TG2302	**340**	**-75**			**125.0**			
			44.0	59.0	15.2	0.5	3.1	TVL/Opa
			72.0	105.0	33.5	1.4	4.0	Opa
		including	*85.0*	*94.0*	*9.1*	*3.9*	*5.9*	
TG2303	**120**	**-80**			**53.0**			
						NSS	NSS	
TG2304	**340**	**-75**			**140.0**			
			52.0	58.0	6.1	0.8	3.8	Opa
			122.0	123.0	1.5	0.8	3.2	
			139.0	140.0	1.5	0.3	4.0	
TG2305	**0**	**-90**			**122.0**			
			35.0	37.0	1.5	0.4	1.5	Opa
			47.0	52.0	4.6	0.3	2.5	
			88.0	90.0	1.5	0.4	1.4	
			107.0	108.0	1.5	0.8	1.5	
TG2306	**0**	**-90**			**117.0**			
			35.0	38.0	3.0	0.5	16.4	Opa
			59.0	70.0	10.7	0.5	6.8	
TG2307	**55**	**-60**			**152.0**			
			79.0	90.0	10.7	0.5	2.7	TVL
			93.0	143.0	50.3	0.9	3.6	TVL/Opa
		including	*116.0*	*119.0*	*3.0*	*2.8*	*5.1*	Opa
		including	*125.0*	*130.0*	*4.6*	*3.2*	*3.8*	Opa
		including	*140.0*	*143.0*	*3.0*	*3.1*	*3.6*	Opa
			149.0	151.0	1.5	1.1	2.0	Opa
TG2308	**0**	**-90**			**152.0**			
			114.0	117.0	3.0	0.4	1.1	TVL
			133.0	136.0	3.0	0.3	2.5	Opa
			146.0	148.0	1.5	4.2	3.3	
TG2309	**190**	**-70**			**198.0**			
			58.0	62.0	4.6	0.7	8.1	TVL
			94.0	122.0	27.4	1.6	3.1	
		including	*94.0*	*98.0*	*3.0*	*10.7*	*9.5*	
			146.0	160.0	13.7	0.7	1.5	

Table 10.4: Mineralization Highlights from the 2023 Drilling Programs, continued

Drill Hole	Azimuth	Dip	From (m)	To (m)	Length (m)	Au (g/t)	Ag (g/t)	Rock Type
TG2310	**210**	**-70**			**198.0**			
			72.0	94.0	22.9	3.5	3.6	TVL
		including	*73.0*	*75.0*	*1.5*	*6.9*	*3.9*	
		including	*78.0*	*84.0*	*6.1*	*6.8*	*6.4*	
			98.0	113.0	15.2	0.5	0.6	
			117.0	126.0	9.1	0.7	1.7	
			133.0	134.0	1.5	1.0	2.1	
TG2311	**90**	**-75**			**122.0**			
			24.0	34.0	9.1	2.4	7.2	TVL/Opa
		including	*26.0*	*27.0*	*1.5*	*9.5*	*1.5*	
			49.0	58.0	9.1	3.0	4.2	Opa
		including	*49.0*	*50.0*	*1.5*	*15.7*	*12.4*	
			70.0	111.0	41.1	1.5	3.1	
		including	*107.0*	*110.0*	*3.0*	*10.4*	*4.0*	
TG2312	**350**	**-80**			**250.0**			
			104.0	107.0	3.0	0.7	3.5	TVL
			146.0	155.0	9.1	0.3	2.4	
			169.0	172.0	3.0	0.3	3.0	
			183.0	186.0	3.0	0.4	3.1	TVL/Opa
TG2313	**0**	**-90**			**171.0**			
			116.0	117.0	1.5	0.4	1.4	
TG2314	**0**	**-90**			**201.0**			
			149.0	152.0	3.0	0.3	3.5	TVL/Opa
TG2315	**140**	**-75**			**91.0**			
			47.0	49.0	1.5	0.3	6.5	Opa
			85.0	90.0	4.6	2.6	3.9	
TG2316	**210**	**-75**			**114.0**			
			87.0	90.0	3.0	0.3	5.3	Opa
TG2317	**170**	**-75**			**91.0**			
			62.0	67.0	5.0	0.3	2.4	Opa
TG2318	**250**	**-75**			**110.0**			
			38.0	93.0	54.9	1.0	6.1	TVL/Opa
		including	*44.0*	*58.0*	*13.7*	*1.6*	*12.4*	TVL/Opa
		including	*58.0*	*62.0*	*4.6*	*4.2*	*8.1*	Opa

Notes:

TVL = Tertiary volcanic lower

Opa = Ordovician Palmetto argillite.

COG = 0.2 Au g/t

Table 10.5: Mineralization Highlights from the 2024 Drilling Programs

Drill Hole	Azimuth	Dip	From (m)	To (m)	Length (m)	Au (g/t)	Ag (g/t)	Rock Type
TG2401	**270**	**-70**			**105.0**			
			40.0	90.0	50.3	0.7	3.0	TVL
		including	*40.0*	*50.0*	*10.7*	*1.5*	*6.3*	TVL
		including	*69.0*	*73.0*	*4.6*	*1.5*	*2.9*	TVL
TG2402	**270**	**-70**			**128.0**			
			52.0	56.0	4.6	0.5	1.5	TVL
TG2403	**270**	**-70**			**128.0**			
			64.0	82.0	18.3	0.6	5.8	TVL
TG2404	**270**	**-70**			**159.0**			
			90.0	126.0	36.6	3.2	6.9	TVL/Opa
		including	*94.0*	*102.0*	*7.6*	*8.9*	*17.4*	TVL
		including	*108.0*	*110.0*	*1.5*	*15.4*	*11.2*	TVL
			134.0	142.0	7.6	0.4	2.7	Opa
			146.0	158.0	12.2	0.5	3.4	Opa
TG2405	**270**	**-70**			**159.0**			
			99.0	116.0	16.8	0.9	4.5	TVL/Opa
TG2406	**270**	**-70**			**159.0**			
			61.0	64.0	3.0	0.5	2.2	Opa
TG2407	**270**	**-70**			**177.0**			
			87.0	177.0	89.9	2.0	4.4	TVL/Opa
		including	*90.0*	*98.0*	*7.6*	*11.8*	*23.6*	TVL
		including	*94.0*	*98.0*	*3.0*	*19.5*	*29.4*	TVL
TG2408	**270**	**-70**			**159.0**			
			40.0	58.0	18.3	4.4	7.5	TVL
		including	*41.0*	*50.0*	*9.1*	*7.6*	*12.2*	TVL
			75.0	78.0	3.0	0.5	0.4	TVL
			90.0	98.0	7.6	0.3	2.7	Opa
TG2409	**55**	**-60**			**171.0**			
			55.0	76.0	21.3	5.1	7.2	TVL
		including	*56.0*	*64.0*	*7.6*	*11.7*	*10.6*	TVL
			81.0	119.0	38.1	1.1	4.2	Opa
TG2410	**270**	**-60**			**171.0**			
			73.0	78.0	4.6	0.3	2.4	TVL
TG2412	**270**	**-70**			**143.0**			
			88.0	99.0	10.7	1.0	3.2	Opa
		including	*96.0*	*98.0*	*1.5*	*3.6*	*2.4*	Opa
TG2414	**n/a**	**-90**			**152.0**			
			49.0	52.0	3.0	0.3	1.3	TVL
			61.0	70.0	9.1	1.0	2.6	TVL
		including	*64.0*	*67.0*	*3.0*	*2.4*	*3.0*	TVL
			98.0	110.0	12.2	0.3	5.0	TVL
			114.0	122.0	7.6	0.3	2.9	TVL
			130.0	139.0	9.1	1.0	1.9	TVL

Table 10.5: Mineralization Highlights from the 2024 Drilling Programs, cont.

Drill Hole	Azimuth	Dip	From (m)	To (m)	Length (m)	Au (g/t)	Ag (g/t)	Rock Type
TG2415	**n/a**	**-90**			**261.0**			
			155.0	160.0	4.6	0.4	1.6	Opa
			174.0	181.0	7.6	3.3	18.1	
		including	*178.0*	*180.0*	*1.5*	*12.7*	*15.7*	
			189.0	200.0	11.0	0.5	10.2	
			210.0	232.0	21.0	1.6	4.8	
		including	*212.0*	*216.0*	*4.6*	*2.6*	*5.6*	
		including	*221.0*	*224.0*	*3.0*	*4.2*	*9.1*	
TG2420	**n/a**	**-90**			**122.0**			
			24.0	29.0	4.6	0.6	2.0	Opa/TVL
			55.0	58.0	3.0	0.7	4.1	Opa
			66.0	67.0	1.5	0.7	6.5	
			107.0	108.0	1.5	0.9	2.9	
TG2421	**n/a**	**-90**			**152.0**			
			41.0	44.0	3.0	2.4	14.8	Opa
			107.0	111.0	4.6	4.6	7.7	
TG2422	**n/a**	**-90**			**122.0**			
			23.0	53.0	30.5	2.3	7.4	TVL
		including	*24.0*	*38.0*	*13.7*	*4.5*	*8.6*	
			62.0	78.0	15.2	0.4	4.8	Opa
			99.0	105.0	6.1	4.4	4.0	
		including	*99.0*	*101.0*	*1.5*	*16.3*	*3.9*	
TG2423	**270**	**-70**			**140.0**			
			34.0	37.0	3.0	0.8	1.0	TVL
			49.0	61.0	12.2	0.6	3.0	Opa/TVL
			67.0	72.0	4.6	0.7	2.3	Opa
TG2424	**270**	**-70**			**170.0**			
			128.0	158.0	30.5	0.9	2.1	TVL
		including	*131.0*	*136.0*	*4.6*	*2.2*	*4.7*	
		including	*154.0*	*158.0*	*4.6*	*2.6*	*4.5*	

Notes:

TVL = Tertiary volcanic lower

Opa = Ordovician Palmetto argillite.

COG = 0.2 Au g/t

10.3.1 Collar Surveys

Drill hole locations were initially spotted, and once complete, surveyed, using a handheld GPS. Approximately 35 of the 2018 and 2019 drill hole collars were formally surveyed in 2020 using a high-precision Trimble GPS device by Solarus, LLC of Ely, Nevada. DD and RC holes drilled since 2020 have not been formally surveyed with a high-precision GPS as of the effective date of this Technical Report.

Most of the pre-2018 drilling locations are no longer visible; however, those that remain are marked with a wooden stake and a metal tag showing the drill hole number. Viva drilling locations are marked with a wooden stake placed in the cement plug of the completed drill hole and labelled with permanent marker. Figure 10.7 illustrates the collar locations of a Viva and Newmont drill hole taken during the 2023 site visit.

Source: WSP June 2023

Figure 10.7: Example of Viva Drill Hole TG2312 (A) and Newmont Drill Hole MW-217 (B)

An aerial flyover of the Tonopah property was performed in 2022, producing an orthophoto and DSM at a resolution of 25 cm. The project data has changed coordinate systems and been converted from imperial to metric units over time. To ensure the collar data has been properly converted to UTM Zone 11N, NAD 83 m, in September 2022 DeMaris Integrated GeoSpatial Solutions LLC, was contracted by Viva directly to verify and adjust the drill hole collar coordinates based on surveyed ground control points and the 2022 aerial flyover. The aerial flyover does not cover the full area of all available drilling. Figure 10.8 shows the coverage of the aerial flyover relative to all available drilling. Drill holes located outside of the aerial flyover are also outside the boundary of the current geologic model.

10.3.2 Downhole Surveys

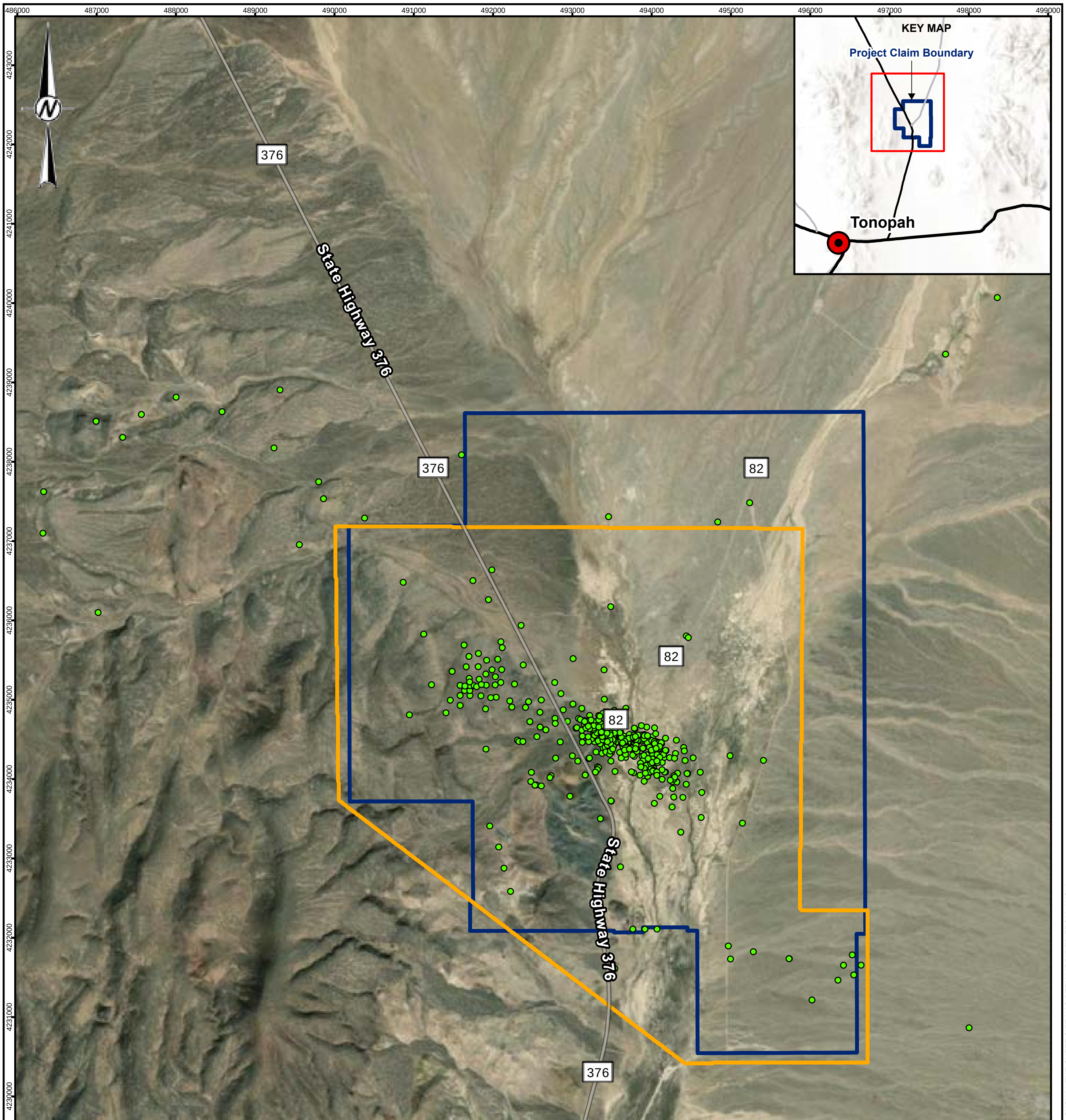
Downhole surveys for all Viva campaigns were performed by International Directional Services of Elko, Nevada, using a Surface Recording Gyroscope, model DG-69. A total of 48 of 121 drill holes have been surveyed and the data stored electronically in the drill hole file archive.

10.3.3 Core Recovery

Core recovery and RQD was recorded for the DD holes drilled by Newmont, Midway Gold and Viva. In 2020, Call & Nicholas, Inc. (CNI) conducted a review of all the historic core recovery and RQD measurements, as well as the results of the 2020 Viva diamond drilling, as part of a geotechnical study to determine preliminary slope angles for the proposed open pit (CNI 2020). In the study, CNI defined eight geomechanical domains which were based off the geology and the location relative to the major faults within the conceptual pit. The mean core recovery within the Op domains was 98% and the mean RQD was 51%. For the volcanics CNI separated the results into hanging wall and footwall domains, with the hanging wall volcanics having a mean core recovery of 84% and RQD of 24%, while the footwall volcanics had a mean core recovery of 86% and RQD of 64%. A second program of geotechnical drilling was completed by CNI in 2022, with a mean core recovery of 83% and mean RQD of 28%.

10.3.4 Core and Chip Handling Procedures

The core was boxed in cardboard boxes at the drill rig and labelled with the drill hole identification (ID) and box number (Figure 10.9). Footage blocks were inserted at the end of each drill run. A lid was placed on the box, secured with rubber bands, and transported by truck to the logging and storage facility in Tonopah by the Viva geologist. After arrival at the logging and storage facility, the boxes were opened, and the core checked for block errors and for mistakes in core placement, and the core was photographed. Logging and sampling of the core followed as per the procedures in Item 10.3.5 and 10.3.6. RC chips were collected by the driller every 1.52 m (5 ft) increment and arranged in increasing depth at the drill. A reference sample of each interval was retained in plastic trays, as shown in Figure 10.10.



LEGEND

- 2022 AERIAL FLYOVER BOUNDARY
- PROJECT CLAIM BOUNDARY
- DRILL HOLE COLLAR

0 2 4

1:70,000 KILOMETRES

NOTE(S)

REFERENCE(S)

2022 AERIAL DATA PROVIDED BY VIVA GOLD CORP.
DATA PROVIDED BY VIVA GOLD CORP.
PROJECTED COORDINATE SYSTEM: NAD 1983 UTM ZONE 11N

CLIENT
VIVA GOLD CORP.

PROJECT
TONOPAH GOLD PROJECT

TITLE
2022 AERIAL FLYOVER BOUNDARY

CONSULTANT

YYYY-MM-DD	2025-08-08
DESIGNED	RK
PREPARED	RK
REVIEWED	JS
APPROVED	BT

PROJECT NO.	CONTROL	REV.	FIGURE
CA0004670.0343		0	10.8

Source: Viva 2022

Figure 10.9: Example of Core from TG2202

Source: WSP June 2023

Figure 10.10: Example of RC Chips at Drill, with Labelled Chip Tray for TG2317

10.3.5 Logging Procedures

The DD core and RC chip logging procedures used by Viva were similar to those that Newmont implemented during their programs. Core logging was done at the logging and storage facility in Tonopah on paper sheets then transcribed into excel files. Scans of all paper logs are stored in the drill hole file for reference. All core is wet photographed prior to sawing or splitting for sampling. RC chips were logged at the drill on paper sheets and then further logged under a binocular microscope at the logging and storage facility. Logging data was then transcribed into excel, and a scan of the original paper log stored with the drill hole file. RC chips are not regularly photographed; however, all chips are stored at the logging and storage facility and are referenced frequently. All core and chip logging was completed by Mr. Edward Bryant, a qualified geologist very familiar with the Tonopah deposit, and a Viva sub-contractor.

10.3.6 Core and Chip Sampling

Core sampling procedures have remained essentially the same through the transition from Newmont to Midway Gold to Viva as operators. Both core and RC chips were stored at the drill site until taken to the logging and storage facility in Tonopah, where drill core was photographed and logged.

Core sampling was guided by the lithology and was generally sampled in 1.52 m (5 ft) intervals. During the 2018 program, core samples were split by mechanical or hydraulic splitters, or sawed into two halves, with half samples placed in cloth bags that have been pre-numbered with a unique sample identification number. One half of the core was retained in the logging and storage facility in Tonopah and half submitted for analysis. Core samples were picked then up by ALS Limited (ALS) for analysis. The core collected in 2020 and 2022 was shipped whole to American Assay Laboratories, Inc. (AAL), and cut there. Viva coordinated the cutting procedure.

Sampling of the RC chips was done by the drilling contractor under the supervision of the Viva geologist. RC samples were collected on 1.52 m (5 ft) increments over the entire hole, an example of which is shown in Figure 10.10. The sampled cuttings were placed in cloth bags, which were pre-labelled with sample ID numbers (Figure 10.11). Labelling of RC samples was guided and managed by the Viva geologist, but not necessarily done by them. Samples were given a unique label. Representative samples of drill cuttings were collected and placed in numbered trays and stored at the drill site until transferred to the logging and storage facility in Tonopah.

Source: WSP June 2023

Figure 10.11: Example of RC Chip Samples for Assaying

The RC chips were first loaded into a bin at the drill site and allowed to drain. Prior to pickup, the bins were brought to the laydown yard and then loaded on either the ALS or AAL trucks for transport to the laboratory. The site was fenced and gated for security. The laboratory generally picked up the core and RC samples approximately two times per week.

11.0 Sample Preparation, Analyses, and Security

11.1 Summary

A total of 53,564 samples have been collected on the Tonopah Project since 1988, totaling 79,587 m of sampled intervals, including 39,849 (74%) RC chip samples and 13,715 (26%) DD core samples. A total of 42,302 (79%) of these samples were obtained prior to the acquisition by Viva. A summary of historical and Viva sampling is presented in Table 11.1.

Table 11.1: Tonopah Project Sampling Summary

Company	Year	Laboratory	RC Drilling			DD Drilling		
			Total No. of Samples	Total Length of Assays (m)	Mean Sample Length (m)	Total No. of Samples	Total Length of Assays (m)	Mean Sample Length (m)
Coeur d'Alene	1988	Bondar-Clegg Inc.	108	328	3.03			
Rio Algom	1990-1991	*No record*	2,878	4,386	1.52			
Kennecott	1992-1994	Barringer Laboratories Inc.	6,237	9,830	1.58	77	113	1.47
	1995-1996	Shasta Geochemistry Laboratory	4,928	7,510	1.52	226	370	1.64
Bob Warren	1994	*No record*	180	274	1.52			
Tombstone	1997	Chemex	1,294	1,974	1.53			
Newmont	2002-2003	ALS Chemex	6,452	9,830	1.52	6,020	7,798	1.30
	2004	American Assay Laboratories Inc.	700	1,067	1.52	2,307	3,326	1.44
Midway Gold Corp.	2005-2008, 2011	ALS Chemex	6,944	10,582	1.52	2,789	3,528	1.27
	2007-2008	SGS Canada Inc.	195	297	1.52	160	244	1.52
	2011	Florin				789	939	1.19
	2008*	American Assay Laboratories Inc.				18	27	1.52
Viva Gold Corp.	2018	ALS	1,368	2,085	1.52	276	420	1.52
	2019		1,423	2,169	1.52			
	2020		1,260	1,920	1.52			
	2020	American Assay Laboratories Inc.				328	502	1.53
	2021		356	543	1.52			
	2022		1,630	2,484	1.52	725	1,105	1.52
	2023		1,689	2,574	1.52			
	2024		2,207	3,363	1.52			
Total			**39,849**	**61,215**	**1.49**	**13,715**	**18,373**	**1.49**

Note: *18 DD core samples from the 2008 Midway Gold program were analyzed by Viva in 2022

11.2 Historical Drilling

The DD and RC sample preparation, analyses, and security prior to 2018, have been described previously in several Technical Reports for the Project. This Item presents a summary of this information. Further details on these programs can be found in Item 6.2.

The various companies who have drilled on the Tonopah Project prior to 2018 have used recognized independent laboratories, including:

- Bondar Clegg Inc., Sparks Nevada (acquired in 2001 by ALS Chemex);
- Barringer Laboratories Inc., Reno, Nevada (acquired by Inspectorate in 2000 then Bureau Veritas in 2010);
- Shasta Analytical Geochemistry laboratory, Redding, California (acquired by Barringer in 1998);
- Chemex Labs (acquired by ALS in 1999 as ALS Chemex, ALS Limited since 2012) in Reno, Nevada and Vancouver, British Columbia;
- American Assay Laboratories Inc., in Sparks, Nevada;
- SGS Canada Inc., Toronto, Ontario; and
- Florin Analytical Services in Reno, Nevada (analytical division of Kappes, Cassiday & Associates).

Many of the laboratories used since 2000 have held accreditation from ISO during the period the samples were analyzed. WSP could not verify the accreditation of the laboratories used prior to 2000.

Drilling programs prior to 2002 were primarily completed using RC drilling techniques, which utilized a double wall drill pipe, and discharged rock cuttings at the surface via a cyclone exiting directly onto a three-tiered Jones splitter, as reported by MDA (2005) and summarized in Item 10.2.1. Samples were typically collected continuously on 1.5-to-3 m (5-to-10 ft) intervals, producing on average, a 16 kg (35 pound [lb]) sample. Samples were placed into cloth sacks, sealed, and labelled with a unique sample number for shipping by bonded carrier to a registered laboratory.

Plastic sample trays were filled with a representative sample for each sampled interval and retained at the logging and storage facility in Tonopah for reference (Figure 11.1).

Source: WSP 2023

Figure 11.1: Example of RC Chip Tray Storage

MDA (2005) reported that samples were crushed to <2 mm, with a 250 gram (g) split taken and then pulverized to 75 microns (µm). Au was analyzed by fire assay (FA) with atomic absorption spectroscopy (AAS) for quantification, and trace elements by Inductively Coupled Plasma - Atomic Emission Spectroscopy (ICP-AES).

The 2002 to 2004 JV between Midway Gold and Newmont, and the following Midway Gold 2005 to 2011 campaigns, collected and assayed both core and RC samples. The sample preparation, analyses and security protocols implemented by Newmont were followed during the JV and continued to be followed during the proceeding Midway Gold campaigns.

Core was generally sampled in 1.5 m (5 ft) intervals, with samples cut by mechanical or hydraulic splitters or sawed into halves. One half of the core was retained in the core box and stored at the Tonopah logging and storage facility (Figure 11.2), and the other half sent for analysis. Samples were placed in cloth bags with unique identification numbers and stored securely until delivery to ALS Chemex (now known as ALS) in Winnemucca, Nevada or AAL, in Sparks, Nevada for analysis. RC samples were also collected on 1.5 m intervals, with wet samples split using a rotating wet-sample splitter. Samples were placed in cloth bags, labelled, stored securely, and delivered to ALS Chemex or AAL for analysis. A reference sample of each interval was retained in plastic trays (Figure 11.1).

Both ALS Chemex and AAL conducted sample preparation and analyses, following ISO/IEC 17025:1999 and ISO 9001:2000 standards. Samples were received at the laboratory, weighed, and assigned a unique barcode which was the entered into the internal laboratory information management system. Samples were then dried and weighed again, prior to crushing. Samples were then crushed to 70% passing <2 mm, split using a riffle splitter, then a 1.0-to-1.5 kg sample was further pulverized to 85% passing <75 µm, and a 30 g sample was analyzed by fire assay with AAS finish. Samples that returned >5 PPM on the fire assay AAS analysis, were further analyzed by fire assay with a gravimetric finish.

Newmont also conducted metallic screen analysis on approximately 1,400 assay samples in 2004. Metallic screen fire assays employ a larger sample volume that is screened to effectively segregate coarse Au particles from finer material. This technique ensures accurate quantification of both coarse and fine Au, thereby providing a comprehensive evaluation of the Au content.

Samples remained under the control of the geologist from collection at the drill rig through delivery to the courier. ALS Chemex and AAL provided a complete chain of custody for each sample through the analytical process. Following analysis, sample pulps were returned to Newmont and Midway Gold and securely stored at the core storage facility in Tonopah (Figure 11.4).

11.3 Viva 2018 to 2024 Core and Chip Sampling

11.3.1 Core and Chip Handling, Sampling and Security

Since 2018, Viva has collected a total of 11,262 samples, totalling 17,164 m, of which 1,329 (12%) were DD core samples and 9,933 (88%) were RC chip samples. All samples were analyzed for Au and Ag.

Viva has drilled 15 DD holes to date and continues to follow the core handling procedures implemented by Newmont. HQ and PQ (85 mm core diameter) sized core was boxed on site by the drill crews and transferred by Viva’s geological consultant, Ed Bryant, to the secure logging and storage facility in Tonopah, Nevada for sampling. Core boxes were labelled with drill hole number, start and end depths, and core box number. Core was photographed, logged on standardized paper logging sheets, transcribed digitally to excel files, and then marked for splitting by Viva’s geology consultant.

HQ core samples were split in half lengthwise via rotary or mechanical cutting methods. One half of the core sample was placed in a numbered plastic sample bag, then placed inside rice sacks or bins as a sample batch and set aside for pickup at the warehouse. The remaining half core was retained in the core storage facility at the Viva secure logging and storage facility. All mineralized HQ core was split and sampled for the Viva drill holes (Figure 11.2). The PQ core collected in 2020 for metallurgical testing was cut by American Assay at their laboratory.

Assays were performed by either ALS in Reno, Nevada or AAL in Sparks, Nevada, who collected the samples at the storage facility for transport to the laboratory. Both ALS and AAL maintain a robust chain of custody program through internal protocols and adhere to ISO/IEC 17025:2017 and ISO 9001:2015 standards. Viva prepared specific assay submittal forms for ALS or AAL, which accompanied each shipment, and which were verified upon arrival at the laboratories.

Source: WSP 2023

Figure 11.2: Example of Retained DD Core at the Logging and Storage Facility in Tonopah

For dry RC drilling, chip samples were collected at 1.5 m (5 ft) intervals in pre-labelled 12-x-18-inch sample bags. The samples were not divided further after they were discharged from the Jones splitter. Wet samples were collected into sample bags placed in 19 litre (L) (5-gallon) buckets. Samples were allowed to de-water prior to being placed into a bin for transport to the laboratory (Figure 11.3). To ensure a sufficient sample mass, the entire collected chip sample for each 1.5 m interval, minus the reference sample discussed below, was shipped to one of the labs for sample preparation. For all RC samples, a small quantity of material from the sample was retained in a labelled plastic chip tray for logging and future reference (Figure 11.1).

Source: WSP 2023

Figure 11.3: RC Samples Collected at Drill for Transport to Laboratory

Samples were aggregated into bins and were collected at site by either ALS or AAL laboratories. Following analysis, the sample pulps were returned to Viva for sample inventory and securely stored at the logging and storage facility in Tonopah (Figure 11.4).

Source: WSP 2023

Figure 11.4: Returned Assay Pulps from ALS

11.3.2 Analytical Procedures

Two independent certified laboratories were used by Viva for Au analyses for the Project. A total of 4,327 samples were prepared and analyzed at ALS in Reno, Nevada in 2018, 2019, and 2020, and a total of 6,910 samples were prepared and analyzed at AAL in Sparks, Nevada for the campaigns between 2020 and 2024. Both ALS and AAL are independent from Viva and are certified under ISO/IEC 17025:2017 and ISO 9001:2015. Two routine Au analytical packages were selected by Viva for the analysis completed at ALS and AAL, including:

1) Fire assay with AAS or an Inductively Coupled Plasma Optical Emission Spectroscopy (ICP-OES) finish (ALS method Au-AA23, AAL method FA-PB30-ICP);
2) Fire assay with a gravimetric finish (ALS method Au-GRA21, AAL method GRAVAu30), for overlimit samples.

For the fire assay procedure, the sample was first dried in high temperature ovens, then the entire sample was crushed to 70% passing <2 mm, mechanically split into a 1 kg sample using a rotary splitter, then pulverized to at least 85% passing <75 µm. A 30 g sample of the pulverized sample was then used for the fire assay procedure with either an AAS or ICP-OES finish. If the sample returned an Au value of >5 PPM, then a second fire assay analysis with gravimetric finish was completed. The gravimetric assay was carried forward as the final interval value in drill hole database, however, both results are retained for reference.

Viva also typically requested 30 g cyanide leach assays for Au and Ag with AAS finish (ALS method Au-AA13, AAL method AuCN30), in mineralized intervals. Ag was analyzed at ALS by cyanide leach (method Ag-AA13). Ag was analyzed at AAL either by five-acid digestion (method D5A-0.5) using nitric acid (HNO_3), hydrochloric Acid (HCl), sulphuric acid (H_2SO_4), hydrofluoric acid (HF) and perchloric acid ($HClO_4$) with a 0.5 g sample, or by cyanide leach (method +AgCN) using 30 g sample.

11.3.3 Data Management

All existing exploration data for the Tonopah Project, including historical data as well as data collected between 2018 and 2024 by Viva, is stored in electronic folders by drilling program or operator. Viva does not use a commercially available database, such as MX Deposit or acQuire to store exploration data, rather all geologist collected drilling data, including collar, lithology and sampling, was first recorded on paper then transcribed into Excel. A scanned copy of the paper drill log, stored as a Portable Document Format (.pdf) was included in each drill hole folder by program for reference and to validate the transcription. Assay certificates from ALS were provided to Viva as both a secured .pdf and an unsecured comma-separated value (.csv) file. These were stored electronically by program. Assay certificates from AAL were provided to Viva as an unsecured .pdf and a .csv. These were stored electronically by program.

11.4 Bulk Density

In 1995, Kennecott collected 15 samples from 3 DD holes, which were analyzed for bulk density by McClelland Laboratories, Inc. (MLI). The core samples were weighed, then oven dried, prior to having the bulk density measured. The oven dried pieces were coated with acrylic enamel and bulk densities were measured using the standard volume displacement method with a weight differential check (McClelland, 1995).

Midway Gold systematically collected 336 bulk density measurements from 20 DD holes during the 2011 drilling program. The methodology for measuring the bulk density followed Archimedes' Principle, where the mass of the dry sample is compared to the mass of the sample submerged in water. Midway Gold followed the standard practice for this methodology which was to seal the sample with a waterproof coating (wax) to prevent water from entering the void space of the rock where necessary. A total of 275 measurements were taken from samples sealed with wax, and 61 were taken from unsealed samples. Where a sample was sealed with wax, the loggers recorded the dry weight, the sealed dry weight, and the sealed wet weight.

The following formula was used to calculate the final bulk density where a wax seal was used:

$$Bulk\ Density = \frac{Unsealed\ dry\ weight}{Unsealed\ dry\ weight - \left(Sealed\ wet\ weight + \frac{(Sealed\ dry\ weight - unsealed\ dry\ weight)}{0.9} - (Sealed\ dry\ weight - unsealed\ dry\ weight)\right)} * Balance\ Calculation\ Factor$$

Where a wax seal was not used, the loggers recorded the dry weight, the initial unsealed wet weight, and the final unsealed wet weight after 1 minute submerged in water. The bulk density was calculated for both the initial unsealed sample after being submerged in water, and the bulk density of the sample after being submerged in water for 1 min using the formulas listed below. These values were then averaged to determine the final bulk density used for the sample.

$$Initial\ Bulk\ Density = \frac{Unsealed\ dry\ weight}{(Unsealed\ Dry\ Weight - Initial\ Unsealed\ Wet\ Weight))} * Balance\ Calculation\ Factor$$

$$1\ min\ Bulk\ Density = \frac{Unsealed\ dry\ weight}{(Unsealed\ Dry\ Weight - 1\ min\ Unsealed\ Wet\ Weight))} * Balance\ Calculation\ Factor$$

$$Final\ Bulk\ Density = \ Average\ of\ Initial\ Bulk\ Density\ and\ 1\ Min\ Bulk\ Density$$

CNI collected 19 bulk density measurements from 7 DD holes as part of the 2022 and 2023 geotechnical strength testing study (CNI, 2023a). Methodology for the bulk density measurements were not provided by CNI.

The majority of bulk density measurements were collected within the Discovery, Dauntless, and 121 zones (Figure 11.5). Of the 370 samples collected, three are in the Quaternary alluvium (QAL) and one is in the TVU, both of which are considered waste lithologies. Within the mineralized lithologies, 208 samples are in the TVL, and 158 are in the Op. The waste lithologies and the Rye Patch mineralized zone are currently underrepresented, and the QP recommends additional bulk density measurements be taken on all future DD drilling programs as well as historical DD core, spaced evenly throughout the main zone of the current resource pit shell extents. Additional bulk density measurements, including those from intervals of known waste rock, will improve the current specific gravity (SG) and tonnage estimate for the deposit.

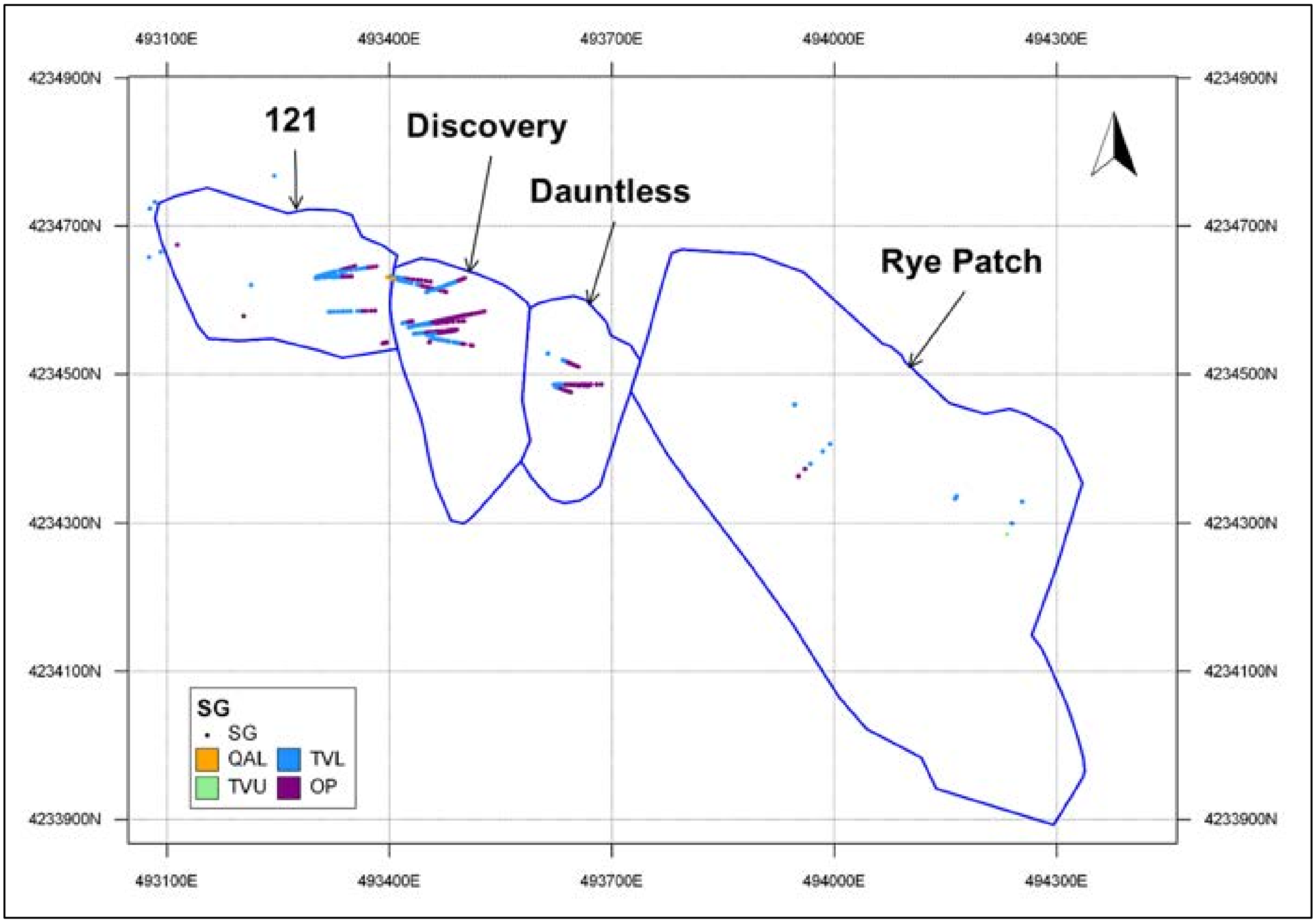


Figure 11.5: Plan View of Bulk Density Measurements within Mineralized Zones

11.5 Quality Assurance and Quality Control Sampling Procedures and Results

11.5.1 Summary

Quality assurance and quality control (QA/QC) procedures are typically established to ensure the precision, accuracy and overall reliability of the exploration data and are essential industry standard practice. In general, this will include inserting certified reference materials (CRM) or standard reference materials (SRM) samples to measure the accuracy of the laboratory, duplicate samples to measure the reliability of the laboratory, and blanks to detect contamination during sample preparation and analysis.

Prior to 2002 there were limited QA/QC programs implemented by the various operators. SRMs, duplicates and blanks were inserted at different rates, though there wasn't a consistent approach across the operators. SRK (2009) reported the following:

- CDA (1988) – no SRM's, one duplicate per drill hole
- Kennecott (1992-1996) – limited number of SRM's and blanks (amounts not specified)
- Tombstone (1997) – no QA/QC records

- Newmont (2002-2004) – one SRM every 60 m (200 ft), one duplicate every 30 to 60 m (150 to 200 ft)
- Midway Gold (2002) – one duplicate every 30 to 60 m (100 to 200 ft), and (2005 to 2008) – one SRM every 30 m (100 ft)

Previous Technical Reports have summarized the QA/QC procedures implemented by the various operators. Results for these programs were found to be within an acceptable range. For the purposes of this Technical Report, WSP is focusing on the QA/QC protocols implemented by Viva from 2018 to 2024.

11.5.2 Viva QA/QC Protocols

Viva regularly inserts CRMs and blanks into the sample stream at a rate of one of each type per every 20 samples. Viva does not collect field duplicate samples; however, they do monitor the performance of the laboratory preparation and pulp duplicates. Table 11.2 summarizes the type, expected values and source of the CRMs and Blanks used by Viva from 2018 to 2024. The following presents a summary of the results of the Viva QA/QC program. A summary of the number and type of QA/QC samples by drill campaign is presented in Table 11.3.

Table 11.2: Summary of CRM and Blanks Certified Values and Source Material

QA/QC Sample	Type of Control	CRM	Certified Value	1SD	95% Confidence Limits		Method Name	Mineralization Style/Matrix
					Low	High		
Standards	Low Grade	MEG-Au.13.04	0.013	0.002	0.009	0.019	FA	Pediment gravels from Washoe Valley, Nevada, USA
		OREAS 15F	0.334	0.016	0.326	0.341	FA	Blend of barren alkali olivine basalt from Epping, Victoria, Australia and gold-bearing Magdala ore from the Stawell Gold Mine, west central Victoria, Australia.
		MEG-Au.22.03	0.683	0.018	0.648	0.718	FA	Ore grade silty limestone material from the Santa Gertrudis deposit, Mexico.
		MEG-Au.19.09	0.711	0.032	0.646	0.776	FA	Low sulfidation ore from mixed Nevada volcanics
		MEG-Au.12.25	0.720	0.032	0.655	0.782	FA	*not listed*
		MEG-Au.17.09	0.767	0.046	0.675	0.859	FA	Mineralized rock from the Hycroft (Brimstone oxide) pit, Humboldt County, NV
	Mid Grade	MEG-Au.19.10	0.813	0.036	0.741	0.884	FA	Low sulfidation ore from mixed Nevada volcanics
		OREAS 2PD	0.885	0.030	0.871	0.898	FA	Blend of oxidised ore and barren material taken from the flanks of a mineralised shear zone within Ordovician flysch sediments in the Blackwood area of central Victoria
		MEG-Au.22.04	0.953	0.042	0.868	1.037	FA	Ore grade igneous and quartz vein material from Tonopah, Nevada
		MEG-Au.17.21	1.107	0.067	0.973	1.241	FA	Mineralized rock from the Hycroft (Brimstone oxide) pit, Humboldt County, NV
		OxH29	1.298	0.033	1.283	1.313	FA	Feldspars with minor quantities of finely divided gold-containing minerals that have been screened to ensure there is no gold nugget effect
		MEG-Au.21.05	1.723	0.092	1.540	1.907	FA	Ore grade calcareous siltstone within an Au-skarn at Santa Gertrudis, Mexico
	High Grade	S106007X	2.264	0.123	2.017	2.509	FA	*not listed*
		MEG-Au.11.15	3.457	0.232	2.992	3.921	FA	Weakly gold and silver mineralized volcanic rock from Rosebud, Nevada
		OxK35	3.489	0.111	3.442	3.536	FA	Feldspars with minor quantities of finely divided gold-containing minerals that have been screened to ensure there is no gold nugget effect
		OREAS 62C	8.790	0.210	8.690	8.880	FA	High grade epithermal vein-style gold mineralization hosted by andesitic volcanics, Cracow, Queensland, Australia
Blanks	Fine (pulp)	MEG-Au.13.04	0.013	0.002	0.009	0.019	FA	Pediment gravels from Washoe Valley, Nevada, USA
		MEG-Au.13.05	0.010	0.001	0.009	0.013	FA	
		MEG-BLANK.17.10	0.003	0.001	0.000	0.006	FA	Created from barren silica sand from Lane Mountain, WA
		MEG-BLANK.17.11	0.003	0.001	0.000	0.006	FA	
		SiBLANK.21.01	0.005	0.001	0.003	0.007	FA	
		SiBLANK.21.02	0.005	0.001	0.003	0.007	FA	
	Coarse[1]	Landscape Marble	n/a	n/a	n/a	n/a	n/a	n/a

Notes:

1. Only used in 2018.

Table 11.3: Summary of QA/QC Samples by Drilling Campaign

QA/QC Control Samples		Drilling Campaign						
		2018	2019	2020	2021	2022	2023	2024
Total Samples per Campaign		**1,844**	**1,553**	**1,773**	**404**	**2,626**	**1,845**	**2,557**
QA/QC Sample Type								
CRMs	MEG-Au.13.04					16	3	
	OREAS 15F	3						
	MEG-Au.22.03						27	23
	MEG-Au.19.09					13	5	14
	MEG-Au.12.25	18	6					
	MEG-Au.17.09		7	19	6			
	MEG-Au.19.10							22
	OREAS 2PD	2						
	MEG-Au.22.04							6
	MEG-Au.17.21	19	10	20	5	16	6	
	OxH29		11					
	MEG-Au.21.05						34	31
	S106007X	23	2					
	MEG-Au.11.15		5	19	7	15	3	6
	OxK35		15					
	OREAS 62C	8				18		
Blanks	Fine	62	74	60	18	78	78	102
	Coarse	13						
Total QA/QC Samples		**148**	**130**	**118**	**36**	**156**	**156**	**204**
Percent QA/QC Samples		8.0%	8.4%	6.7%	8.9%	5.9%	8.5%	8.0%
Duplicates[1]	Pulp[2]	74	168	4				
	Preparation	16	26	85	46	291	140	246
Total Lab Duplicate Samples		**90**	**194**	**89**	**46**	**291**	**140**	**246**

Notes:

1. Lab duplicates.
2. ALS completed both internal preparation and pulp duplicates, AAL only completed internal preparation duplicates.

11.5.2.1 CRMs

Several types of commercially prepared Au CRMs were obtained by Viva directly from Ore Research & Exploration Assay Standards (OREAS), Moment Exploration Geochemistry (MEG; formerly Shea Clarke Smith) and Rocklabs, and were inserted in alternating order. CRMs are used to evaluate the analytical laboratories accuracy against a certified value. Assay results for a CRM should be within ±3 standard deviation (SD) tolerance range of the certified value, otherwise they are considered to have failed. WSP used the Half Absolute Relative Difference (HARD) method to assess CRM accuracy. HARD is determined by the following formula and expressed as a percentage:

$$HARD = \frac{(x1 - x2)}{(x1 + x2)} \times 100$$

Where: x1 = certified value, x2 = assay value.

The HARD value indicates the percentage of the difference between the expected value of the CRM and analyzed value of the CRM. The lower the HARD value, the higher the precision between the expected and analyzed value. The Half Relative Difference (HRD) determines the relative differences between the certified and analyzed value and can be positive or negative. The HRD indicates potential biases in the measurements, while the HARD measures the overall precision.

Among the 187 CRMs analyzed by ALS, 5 were outside of the ±3SD tolerance range. Similarly, out of the 276 samples analyzed by AAL, 6 values fell outside the ±3SD tolerance range. Viva followed up with each laboratory regarding each of the anomalous samples, however, samples were not re-submitted for analysis. Overall, there was approximately 97% of the CRMs that passed the QC analysis, with all but MEG-Au.13.04 returning HARD less than 5%, indicating consistently high precision of analysis at both ALS and AAL. Table 11.4 summarizes the CRMs used by Viva from 2018 through 2024, by laboratory, including the type and certified value.

Table 11.4: QA/QC CRM Sample Counts and Statistics by Laboratory

CRM	Years	Counts	Certified Value (Au ppm)	Mean (Au ppm)	HRD (%)	HARD (%)	Outliers	Percent Passing QC
ALS								
OREAS 15F	2018	3	0.334	0.320	-2.24	2.24	0	100.0%
MEG-Au.12.25	2019-2019	24	0.720	0.731	0.75	1.33	0	100.0%
MEG-Au.17.09	2019-2020	26	0.767	0.810	2.68	3.38	0	100.0%
OREAS 2PD	2018	2	0.885	0.872	-0.75	0.75	0	100.0%
MEG-Au.17.21	2018-2020	49	1.107	1.134	1.17	1.73	0	100.0%
OxH29	2019	11	1.298	1.259	-1.66	1.95	1	90.9%
S106007X	2018-2019	25	2.264	2.405	2.83	2.99	2	91.7%
MEG-Au.11.15	2019-2020	24	3.457	3.645	2.47	3.67	2	91.7%
OxK35	2019	15	3.489	3.511	0.31	0.62	0	100.0%
OREAS 62C	2018	8	8.790	8.616	-1.01	1.01	0	100.0%
Total / *Mean*		**187**				***1.97***	**5**	***97.4%***
AAL								
MEG-Au.22.03	2023-2024	50	0.683	0.704	1.49	1.62	3	94.0%
MEG-Au.19.09	2022-2024	32	0.711	0.723	0.75	2.21	0	100.0%
MEG-Au.17.09	2021	6	0.767	0.806	2.47	2.47	0	100.0%
MEG-Au.19.10	2024	22	0.813	0.802	-0.73	1.89	0	100.0%
MEG-Au.22.04	2024	6	0.953	1.038	4.24	4.24	1	83.3%
MEG-Au.17.21	2021-2023	27	1.107	1.070	-1.77	2.67	1	96.3%
MEG-Au.21.05	2023-2024	65	1.723	1.815	2.58	2.75	1	98.5%
MEG-Au.11.15	2021-2024	31	3.457	3.684	3.15	3.24	0	100.0%
OREAS 62C	2022	18	8.790	8.863	0.4	1	0	100.0%
MEG-Au.13.04	2022-2023	19	0.013	0.015	4.85	9.43	0	100.0%
Total / *Mean*		**276**				***3.15***	**6**	***97.2%***

11.5.2.2 Blanks

Blank samples were used to assess contamination at both the preparation (coarse blank) and analytical (pulp blank stage) stage. Viva primarily submitted commercially prepared pulp blank CRMs (Table 11.2) between 2018 and 2024, with the coarse blanks (landscape marble) only being submitted in 2018. Samples were considered to have passed if they were within 5 times the detection limit (5x DL) of the analysis method. For ALS this was 0.025 PPM (DL = 0.005), for AAL it was 0.015 PPM (DL = 0.003). Viva used two very low-grade Au CRMs (MEG-Au.13.04 and MEG-Au.13.05) as blanks in 2021, 2023 and 2024. These samples did not perform well, as shown in Table 11.5 and Figure 11.6, likely due to the background Au values of the CRM rather than potential contamination. Viva recognized this and switched to a true blank CRM part way through the 2024 program. WSP agrees with this action and recommends that these types of CRMs not be used in future QA/QC programs as it is difficult to assess true sample contamination. The true blank samples that both ALS and AAL analyzed all fell within the 5x DL tolerance, with no outliers, as shown in Table 11.5.

Table 11.5: QA/QC Blank Sample Counts and Statistics by Laboratory

CRM	Years	Counts	Certified Value (Au ppm)	Mean (Au ppm)	Upper Limit (Au ppm)	Outliers	Percent Passing QC
ALS							
MEG-BLANK.17.10	2018-2019	111	0.003	0.004	0.025	0	100.0%
MEG-BLANK.17.11	2019-2020	85	0.003	0.004	0.025	0	100.0%
Landscape Marble	2018	13	0.003	0.003	0.025	0	100.0%
Total / *Mean*		**209**				**0**	***100.0%***
AAL							
MEG-Au.13.04*	2021, 2023-2024	106	0.013	0.017	0.015	67	36.8%
MEG-Au.13.05*	2024	50	0.010	0.014	0.015	25	50.0%
MEG-BLANK.17.11	2021	14	0.003	0.006	0.015	0	100.0%
SiBlank.21.01	2022-2023	100	0.005	0.004	0.015	0	100.0%
SiBlank.21.02	2024	6	0.005	0.003	0.015	0	100.0%
Total / *Mean*		**276**				**92**	***77.4%***

Notes: * MEG-Au.13.04 and MEG-Au.13.05 are very-low grade Au CRMs

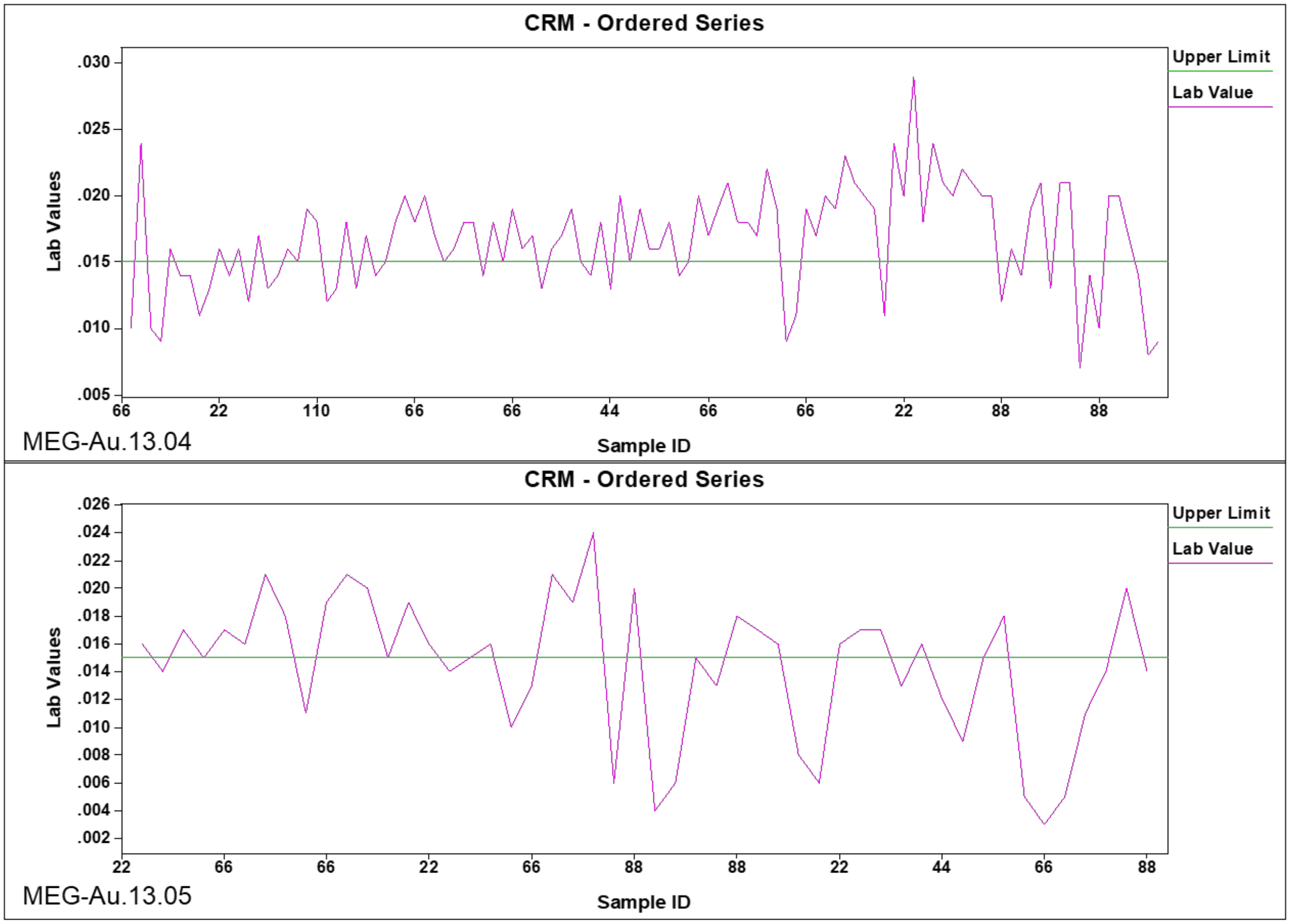


Figure 11.6: QA/QC Blank Control Charts for MEG-Au.13.04 and MEG-Au.13.05

11.5.2.3 Duplicates

Duplicates assess the accuracy of the sampling at different stages in the process. Pulp duplicates assess accuracy at the laboratory analytical stage, preparation at the sample crushing and preparation stage, and field (twin) at the sample cutting stage (half or quarter core). Viva did not take field duplicate samples, rather they have relied on the ALS and AAL internal QA/QC processes for preparation and pulp duplicates.

The HARD value indicates the percentage of the difference between the value of the original sample and that of the duplicate. If the HARD value is below 10% it is considered acceptable. According to Marcotte and Dutaut (2020), the threshold range varies for each type of duplicate, depending on their precision, as follows:

- Pulp duplicates should have 90% of the samples having less than 10% difference
- Preparation duplicates should have 90% of the samples having less than 15% difference
- Field duplicates should have 90% of the samples having less than 20% difference

Table 11.6 summarizes the duplicate samples from each laboratory.

Table 11.6: QA/QC Duplicate Sample Counts and Statistics by Laboratory

Duplicate	Lab	Pair Counts	Mean Au (Original) (ppm)	Mean Au (Duplicate) (ppm)	HRD (%)	HARD (%)	Samples Passing (%)
Pulp[1]	ALS	93	0.407	0.539	-1.09	7.14	78
Pulp[2]		150	0.363	0.351	0.23	4.99	87
Preparation		43	0.146	0.156	-1.46	12.58	67
Pulp	AAL	*Not analyzed*					
Preparation		806	0.261	0.258	0.33	6.38	83

Notes:

1. ALS duplicate for method Au-AA13 (DL = 0.03)
2. ALS duplicate for method Au-AA23 (DL = 0.005)

ALS conducted duplicate analysis for samples using both the Au-AA13 method, and the Au-AA23 method. The Au-AA13 Pulp duplicates showed slightly higher variability (78% of samples <10% HARD) than the Au-AA23 method (87% of samples <10% HARD), both were below the 90% threshold for Pulp duplicates. The ALS Preparation duplicates were also below the threshold (67% of samples <10% HARD), which indicates some challenges with repeatability of sample preparation at ALS, and may also be due to the nature of the Tonopah mineralization.

The AAL Preparation duplicates performed well, with 83% of the samples returning a HARD value below 10%. This indicates a good degree of repeatability in the sample preparation. AAL did not complete any Pulp duplicates of the Viva assays, and WSP would recommend requesting this for future assaying to assess the analytical accuracy.

11.6 Qualified Person Statement on the Adequacy of Sample Preparation, Security, and Analytical Procedures

It is the QP’s opinion that the analytical methods, and the QA/QC procedures used by Viva are consistent with industry standards and that the geological database and the assay data is of suitable quality to support the 2025 Mineral Resource estimate, as reported in Item 14.0.

12.0 Data Verification

12.1 Site Visit

As part of the data and methodology verification process, Ms. Róisín Kerr, P.Geo., WSP QP for data verification and Mr. Jordi Bascompte-Vaquero, mining engineer, conducted a site visit to the Tonopah property and offices on June 6 and 7, 2023. The purpose of the site visit was to allow the QP to observe key aspects of the project site, including deposit geology, current and previous exploration programs, and site infrastructure. Mr. James Hesketh, President and CEO of Viva, and Mr. Edward Bryant, Viva's contract geologist, were available for discussion and verification of current and historical methods and results and to discuss any concerns and recommendations.

Activities performed during the site visit included the following:

- Verify a selection of drill hole collar coordinates.
- Observe and review the drilling, logging, and sampling procedures.
- Observe the core storage facility.
- Select a representative suite of samples for replicate analytical comparison.

An additional site visit was carried out by the mineral processing and infrastructure KCA QP, Mr. Caleb Cook, P.E., who visited the Tonopah Project site and offices on May 8, 2025. During the visit, Mr. Cook visited the proposed heap leach and other processing facilities locations to evaluate the general site conditions, reviewed utilities and infrastructure near the project site. Mr. Cook was accompanied by Mr. James Hesketh, President and CEO of Viva, to discuss any questions or concerns.

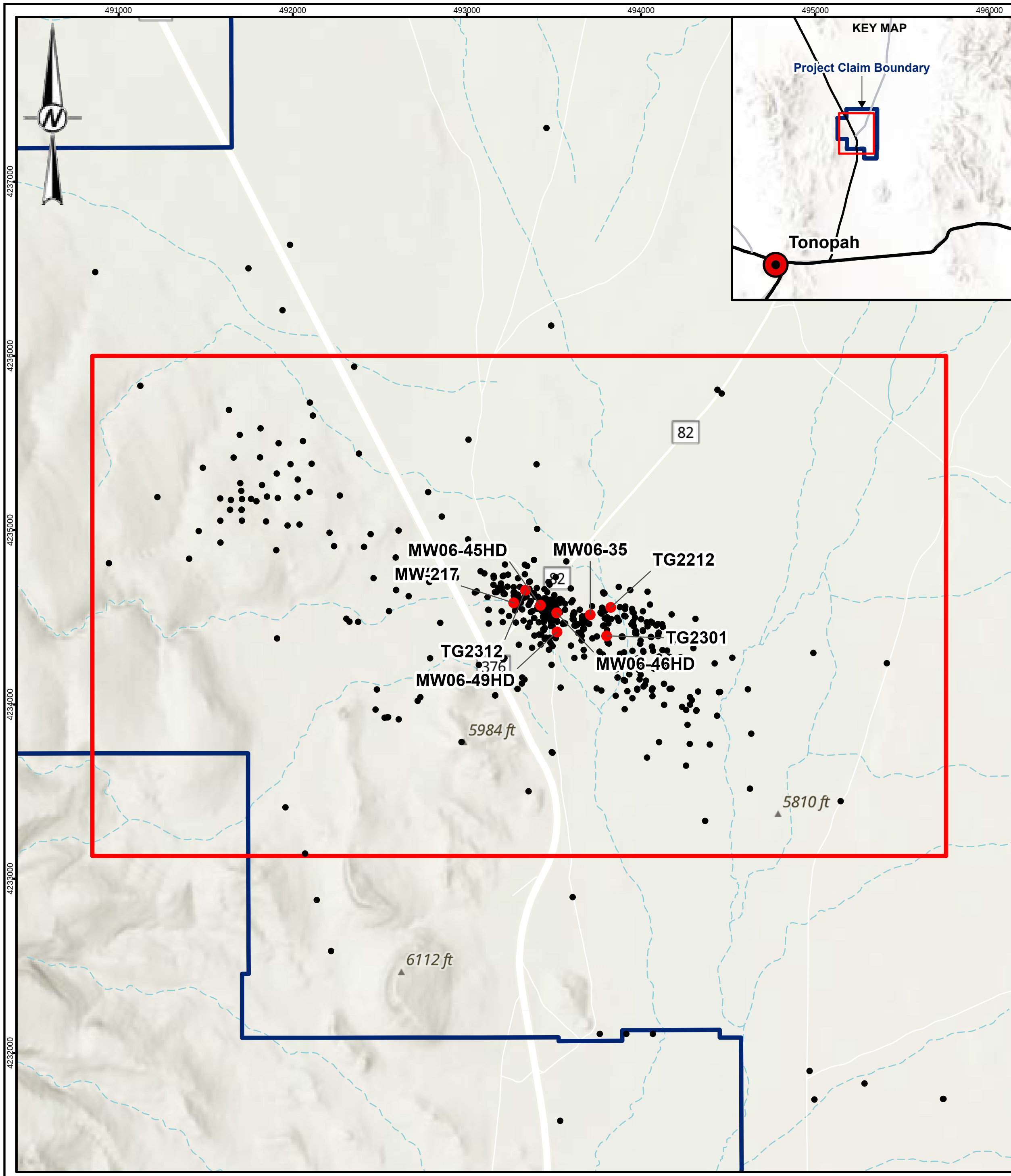
12.1.1 Drill Hole Collar Verification

Eight drill collar locations were visited and surveyed using a handheld non-differential GPS to confirm the collar survey data provided by Viva. All collar locations were found to be within the accuracy of the GPS as summarized in Table 12.1.

Figure 12.1 shows the locations of the drill holes verified during the site visit.

Table 12.1: Comparison of Drill Hole Collar Coordinates

DHID	GPS			Database			Difference		
	Easting (m)	Northing (m)	Elevation (m)	Easting (m)	Northing (m)	Elevation (m)	Easting (m)	Northing (m)	Elevation (m)
MW06-35	493,708	4,234,515	1,771	493,711	4,234,514	1,772	-4	1	-1
MW06-45HD	493,422	4,234,569	1,775	493,422	4,234,567	1,773	-1	2	2
MW06-46HD	493,515	4,234,525	1,775	493,516	4,234,522	1,772	-1	3	3
MW06-49HD	493,517	4,234,416	1,773	493,518	4,234,414	1,771	-1	2	2
MW-217	493,268	4,234,583	1,775	493,268	4,234,581	1,775	0	2	0
TG2212	493,825	4,234,557	1,773	493,825	4,234,555	1,773	0	2	1
TG2301	493,802	4,234,393	1,773	493,803	4,234,397	1,772	-1	-4	1
TG2312	493,336	4,234,655	1,774	493,334	4,234,657	1,775	2	-2	-1
Average							**-1**	**1**	**1**



LEGEND

- LITHOLOGY MODEL BOUNDARY
- PROJECT CLAIM BOUNDARY
- DRILL HOLE COLLAR
- VERIFIED DRILL HOLE COLLAR

0 1 2

1:35,000 KILOMETRES

NOTE(S)

REFERENCE(S)

DATA PROVIDED BY VIVA GOLD CORP.
PROJECTED COORDINATE SYSTEM: NAD 1983 UTM ZONE 11N

CLIENT
VIVA GOLD CORP.

PROJECT
TONOPAH GOLD PROJECT

TITLE
DRILL HOLE COLLAR VERIFICATION

CONSULTANT

YYYY-MM-DD	2025-08-06
DESIGNED	RK
PREPARED	RK
REVIEWED	JS
APPROVED	BT

PROJECT NO.	CONTROL	REV.	FIGURE
CA0004670.0343		0	12.1

Most historical drill hole collars, including early Viva collars, were no longer visible. Newer holes drilled by Viva were marked with a wooden stake with the drill hole name written on the stake with permanent marker (Figure 12.2). Some historical holes were marked with a wooden stake with a metal label stapled to it (Figure 12.3). Figure 12.4 shows an example of one of the permanent water wells.

Figure 12.2: Drill Hole Collar Location of TG2312

Figure 12.3: Drill Hole Collar Location of Holes MW-217 and MW06-35

Figure 12.4: Example of Drill Hole Collar Location of a Water Well

The project data has changed coordinate systems and has been converted from imperial to metric units over time. To ensure the collar data had been properly converted to UTM Zone 11N, NAD83 m, DeMaris Integrated GeoSpatial Solutions, LLC, was contracted in September 2022 by Viva to verify and adjust the drill hole collar coordinates based on surveyed ground control points and the 2022 aerial flyover.

12.1.2 Exploration Drill Rig and Core Shed

WSP observed one active drill rig completing exploration RC drilling as part of the 2023 Viva drilling campaign (Figure 12.5 and Figure 12.6). The drill site review included a review of the drill hole location, drilling methods, chip recovery, and sampling methodology and chain of custody.

Figure 12.5: Exploration RC Drill Rig

Figure 12.6: Active Drill Hole Sample Inspection and Collection

WSP visited the Viva office and core shed located in the town of Tonopah to review DD core and RC chip storage (Figure 12.7 and Figure 12.8). This review included a discussion on core/chip handling and security, drill core/chip logging, sample identification and selection, analytical QA/QC sample insertion, drill core/chip storage, and sample reject (coarse and pulp) storage.

Figure 12.7: Viva DD Core and Sample Reject Storage

Figure 12.8: Viva RC Chip Storage

12.1.3 Independent Sample Verification

WSP reviewed drill core from high-grade intercepts within the mineralized zone of the resource. Figure 12.9 provides examples of high-grade Au mineralization hosted in the Tertiary volcanic unit. The Viva drill logs were found to match the observed core reasonably well and no material issues were identified.

Figure 12.9: Drill Core Inspection of High-grade Au Intercept in MW-220D

Sixty (60) samples from 14 RC holes were chosen for independent sample analysis. Samples ranging from 0.1 PPM to 78 PPM Au were chosen from historical drilling campaigns, with focus placed on the Kennecott campaign where no original assay certificates were available for review. Duplicates of the mineralized DD core were not taken due to the limited availability of the core (most core of interest had already been sent for secondary analysis, either verification or metallurgical testing). All samples sent for analysis were sample pulp rejects. The samples were sent to ALS laboratory in Carson City, Nevada for preparation and were analyzed at the ALS laboratories in Reno, Nevada, and North Vancouver, British Columbia. For Au, WSP chose fire assay analysis (30 g pulp) with AAS finish (Au-AA25) and Au by cyanide leach and AAS finish (30 g pulp) (Au-AA13). For Ag, WSP chose Ag by four acid digestion with AAS finish (Ag-OG62) and Ag by cyanide leach and AAS finish (30 g pulp) (Ag-AA13) was used. Table 12.2 summarizes the intervals sampled and compares the verification results to the Viva assay values.

Table 12.2: Independent Sample Verification Results

DHID	From (m)	To (m)	Length (m)	Year	Campaign	Type	Viva Au (ppm)	WSP Au (ppm)	Viva Ag (ppm)	WSP Ag (ppm)
MW-027	56.39	57.91	1.52	1993	Kennecott	RC	3.43	3.51	11.4	24.0
MW-027	57.91	59.44	1.53	1993	Kennecott	RC	0.33	0.43	3.6	8.0
MW-027	59.44	60.96	1.52	1993	Kennecott	RC	0.77	0.78	2.3	5.0
MW-027	60.96	62.48	1.52	1993	Kennecott	RC	10.17	10.30	5.0	13.0
MW-027	62.48	64.01	1.53	1993	Kennecott	RC	4.40	4.32	4.6	11.0
MW-027	64.01	65.53	1.52	1993	Kennecott	RC	11.45	9.72	2.9	10.0
MW-027	65.53	67.06	1.53	1993	Kennecott	RC	5.20	4.80	1.6	4.0
MW-027	67.06	68.58	1.52	1993	Kennecott	RC	26.82	33.10	4.5	11.0
MW-048	38.1	39.62	1.52	1994	Kennecott	RC	8.09	7.89	-	11.0
MW-048	39.62	41.15	1.53	1994	Kennecott	RC	1.44	1.68	-	8.0
MW-048	41.15	42.67	1.52	1994	Kennecott	RC	1.16	1.38	-	7.0
MW-048	42.67	44.2	1.53	1994	Kennecott	RC	5.66	6.46	-	21.0
MW-048	45.72	47.24	1.52	1994	Kennecott	RC	5.34	5.20	-	25.0
MW-048	47.24	48.77	1.53	1994	Kennecott	RC	6.72	6.34	-	16.0
MW-048	53.34	54.86	1.52	1994	Kennecott	RC	3.41	3.29	-	7.0
MW-048	54.86	56.39	1.53	1994	Kennecott	RC	3.40	4.20	-	8.0
MW-048	56.39	57.91	1.52	1994	Kennecott	RC	1.01	1.57	-	4.0
MW-048	57.91	59.44	1.53	1994	Kennecott	RC	4.53	5.13	-	7.0
MW-048	59.44	60.96	1.52	1994	Kennecott	RC	1.60	1.86	-	9.0
MW-048	60.96	62.48	1.52	1994	Kennecott	RC	2.49	2.91	-	7.0
MW-120	128.02	129.54	1.52	1995	Kennecott	RC	0.37	0.38	0.9	4.0
MW-120	129.54	131.06	1.52	1995	Kennecott	RC	31.27	30.40	13.2	23.0
MW-120	131.06	132.59	1.53	1995	Kennecott	RC	0.19	0.24	0.6	2.0
MW-121	156.97	158.5	1.53	1995	Kennecott	RC	0.19	0.22	2.6	4.0
MW-121	158.5	160.02	1.52	1995	Kennecott	RC	0.28	0.30	2.1	3.0
MW-121	161.54	163.07	1.53	1995	Kennecott	RC	9.02	5.86	20.6	22.0
MW-121	163.07	164.59	1.52	1995	Kennecott	RC	0.99	1.09	4.9	9.0
MW-121	170.69	172.21	1.52	1995	Kennecott	RC	3.84	3.21	6.7	14.0
MW-121	172.21	173.74	1.53	1995	Kennecott	RC	8.19	6.00	13.1	15.0
MW-121	173.74	175.26	1.52	1995	Kennecott	RC	6.41	-	21.3	33.0
MW-121	175.26	176.78	1.52	1995	Kennecott	RC	2.57	3.02	15.4	25.0
MW-123	38.1	39.62	1.52	1995	Kennecott	RC	2.47	2.25	2.2	6.0
MW-123	39.62	41.15	1.53	1995	Kennecott	RC	14.64	16.10	16.8	41.0
MW-123	41.15	42.67	1.52	1995	Kennecott	RC	2.16	1.79	2.9	6.0
MW-369	97.54	99.06	1.52	2003	Newmont	RC	7.72	3.67	5.3	4.0
MW-369	99.06	100.58	1.52	2003	Newmont	RC	7.65	5.16	2.2	5.0
MW-369	100.58	102.11	1.53	2003	Newmont	RC	5.33	3.79	1.5	4.0
MW-369	102.11	103.63	1.52	2003	Newmont	RC	2.42	2.64	1.5	2.0
MW-369	103.63	105.16	1.53	2003	Newmont	RC	9.08	9.59	1.9	4.0
MW11-05C	92.51	94.18	1.67	2011	Midway Gold Corp.	DD	0.16	0.20	0.8	3.0
MW11-07C	69.8	71.32	1.52	2011	Midway Gold Corp.	DD	1.15	1.39	3.2	5.0
MW11-07C	71.32	72.85	1.53	2011	Midway Gold Corp.	DD	1.05	1.08	2.4	4.0
MW11-07C	72.85	73.76	0.91	2011	Midway Gold Corp.	DD	1.94	2.06	3.0	7.0
MW11-07C	89.92	91.44	1.52	2011	Midway Gold Corp.	DD	0.36	0.33	0.5	2.0
MW11-07C	108.2	109.73	1.53	2011	Midway Gold Corp.	DD	0.33	0.32	0.7	2.0
MW11-07C	109.73	110.89	1.16	2011	Midway Gold Corp.	DD	0.47	0.41	0.7	2.0
MW11-07C	127.41	128.32	0.91	2011	Midway Gold Corp.	DD	0.20	0.20	0.7	2.0
MW11-09C	48.43	49.07	0.64	2011	Midway Gold Corp.	DD	0.21	0.22	0.2	1.0
MW11-09C	49.07	50.29	1.22	2011	Midway Gold Corp.	DD	0.17	0.18	0.1	1.0
MW11-09C	64.4	65.53	1.13	2011	Midway Gold Corp.	DD	2.98	2.53	1.8	5.0

DHID	From (m)	To (m)	Length (m)	Year	Campaign	Type	Viva Au (ppm)	WSP Au (ppm)	Viva Ag (ppm)	WSP Ag (ppm)
MW11-09C	65.53	67.06	1.53	2011	Midway Gold Corp.	DD	0.53	0.55	1.3	5.0
MW11-09C	67.06	68.58	1.52	2011	Midway Gold Corp.	DD	78.40	83.00	28.8	31.0
MW11-12C	225.25	226.77	1.52	2011	Midway Gold Corp.	DD	0.87	0.86	1.7	2.0
MW11-12C	248.11	249.63	1.52	2011	Midway Gold Corp.	DD	0.17	0.16	1.6	1.0
MW11-13C	167.64	169.16	1.52	2011	Midway Gold Corp.	DD	0.17	0.16	1.5	2.0
MW11-18C	88.39	89.92	1.53	2011	Midway Gold Corp.	DD	0.19	0.04	0.7	2.0
MW11-22C	125.58	127.1	1.52	2011	Midway Gold Corp.	DD	0.23	0.24	0.1	0.5
MW11-23C	79.25	80.77	1.52	2011	Midway Gold Corp.	DD	0.45	0.51	0.7	1.0
MW11-23C	80.77	82.3	1.53	2011	Midway Gold Corp.	DD	0.31	0.29	0.2	1.0
MW11-09C	50.29	51.82	1.53	2011	Midway Gold Corp.	DD	1.10	0.93	-	4.0

Figure 12.10 and Figure 12.11 provide a graphical comparison of the WSP verification and Viva assays for Au and Ag, respectively. The Au samples compare well, while there is more variation in the Ag samples; however, this is likely due to the differences in analysis method. The QP did not identify any material bias in the Viva sample data, and the comparison results were found to be reasonable given the nature of mineralization in the deposit.

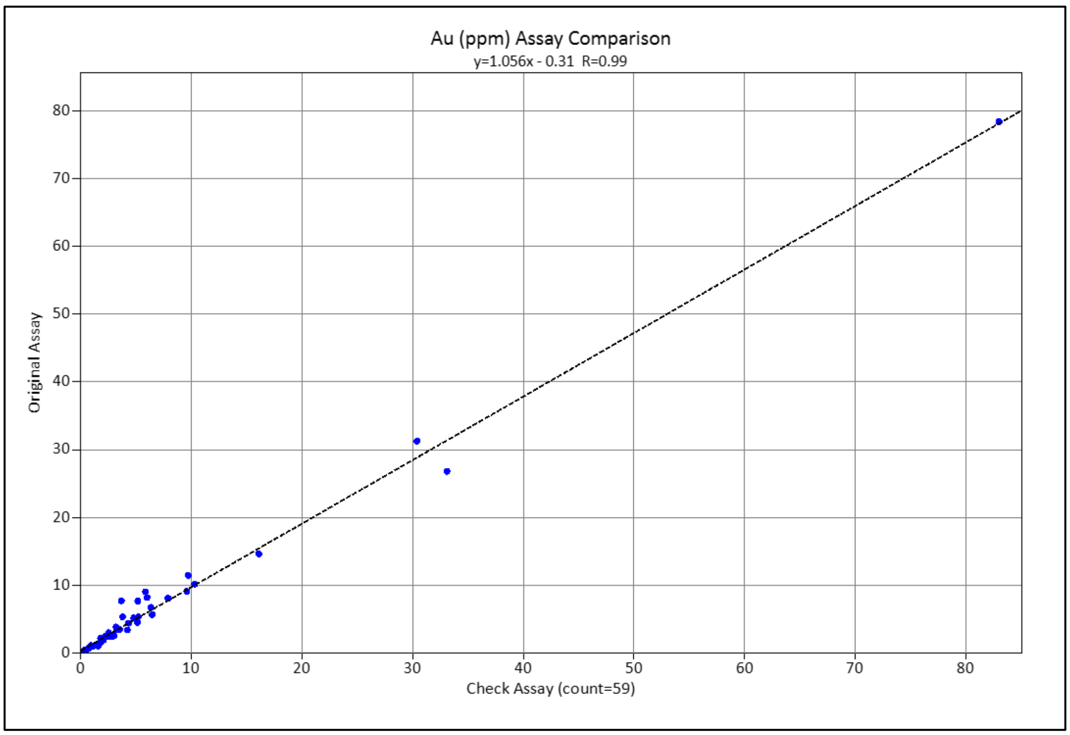


Figure 12.10: Scatterplot Comparison of Verification Sample Results for Au

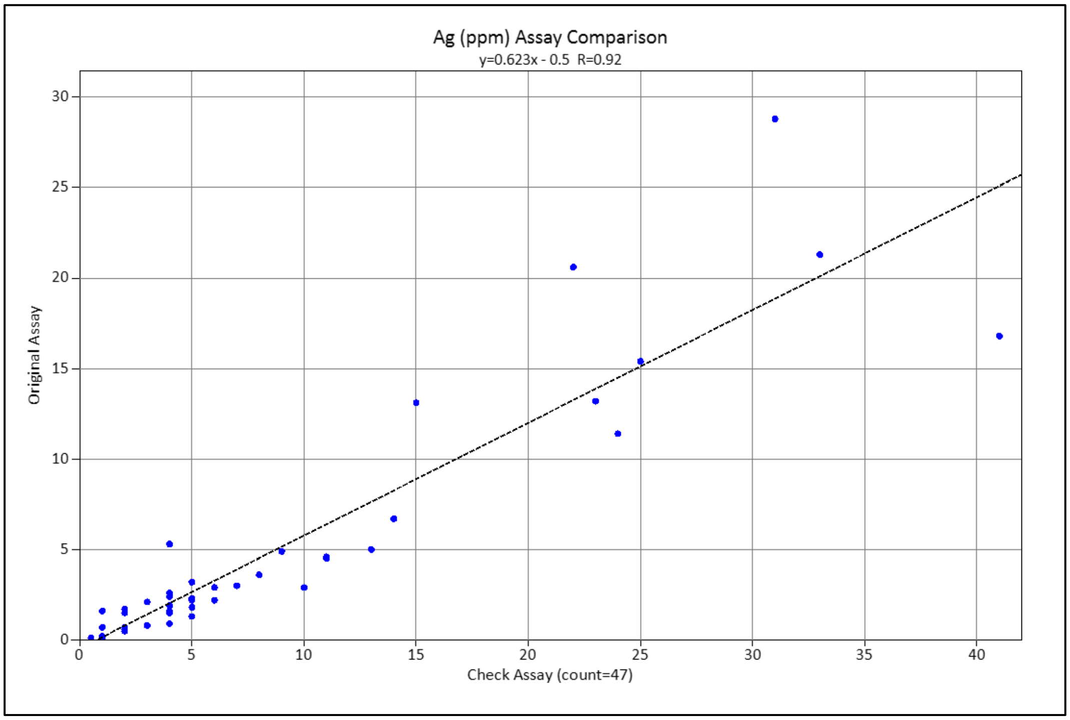


Figure 12.11: Scatterplot Comparison of Verification Sample Results for Ag

12.2 Database Verification

WSP was previously contracted by Viva in August 2022 to update the Tonopah geological model that had been built in Leapfrog modelling software by a third-party consultant and had been last updated in January 2022. During the previous scope of work, WSP reviewed the geological model and drill hole data, including the collar, survey, lithology logging, and assay data included inside the Leapfrog project. Additional drilling and verification were completed following WSP's validation of the resource database and update to the geology model in May 2023 and February 2025. The following is a summary of the steps taken to verify the overall resource database for the Project.

12.2.1 Drill Hole Data Verification

WSP was provided the 2022, 2023 and 2024 drilling data in the form of individual Microsoft (MS) Excel spreadsheets for collar, downhole survey, lithology logging, and assay results, as well as .pdf format documents. The drill hole tables within the Leapfrog model were exported as .csv files. Both sets of drilling information were collated into a Microsoft Access™ drill hole database for review.

The QP performed a review of the data included in the Leapfrog project, as well as the 2022, 2023, and 2024 drilling. As discussed in Item 10.0, as of January 2025, there are a total of 626 drill holes in the database, including 168 (27%) cored DD holes and 458 (73%) RC drill holes.

Drill hole data validation checks were performed in MS Access and MS Excel using a series of in-house data procedures to evaluate for common drill hole data errors, including, but not limited to, data gaps and omissions, overlapping lithology or sample intervals, miscorrelated units, unit conversion checks, and other indicators of data corruption, including transcription and keying errors.

- Identify duplicate or twinned drill holes with identical collar positions:
  - Two drill holes were identified as sharing identical collar and downhole survey, the newer hole was determined to be a re-drill to a deeper depth and was chosen to supersede the original hole.
- Check drill hole collar elevation against topography elevation:
  - An assessment of the drill holes against the topographic surface was completed, as discussed further in Item 12.2.3.
- Check that total hole depths on the collar table match the total depth of the lithological table:
  - If any did not match, WSP reviewed downhole geological data as well as drilling records to reconcile the difference. Once the error was identified, the erroneous data field was corrected.
- Check that from and to depths from surface on the lithology and assay tables increase downhole:
  - If any did not match, WSP reviewed downhole geological data to correct the errors.
- Check for from and to depth overlaps in lithology and assay tables:
  - Where overlaps were identified, WSP reviewed the original lithology and sample logs to correct the errors.
- Identify drill holes that had no lithological, assay, survey:
  - Any drill holes missing all geological data were excluded from the geological model.
- Check that original interval footage has been converted to metres correctly:
  - WSP added the original logged footage to the lithology and assay tables and re-converted the intervals to metres.
- Check for data gaps and omissions:
  - Where a data gap was identified, WSP reviewed downhole geological data to ensure the gap was real and inserted the missing interval with a “not logged” or “no sample” code.

After the initial drill hole database validation, the collar, downhole survey, and geological unit intervals were imported into Leapfrog and reviewed in cross-section. The following additional checks were performed:

- Check lithology followed the geological chronological order downhole:
  - Drill holes that had logging that was out of chronological order were validated against original logs where available, reviewed against surrounding holes, and removed from the geological model if determined to have conflicting logging.
- Check lithology agrees with surrounding drill holes:
  - Holes that were determined to have conflicting logging were removed from the geological model.

Review of the lithology table indicated all holes prior to 2011 had been using lithology codes applied by former property owner Midway Gold that were different from the original logged lithology codes. The original logging codes were added back to the lithology table and used for the geological model update.

12.2.2 Grade Data Verification

The resource database used for the Mineral Resource Estimate effective January 2022 did not contain Ag analytical data that had been collected by Viva and its predecessors. During the internal modelling scope of work completed May 2023, WSP compiled both Au and Ag assay values from original certificates and laboratory spreadsheets where available, and historical compiled spreadsheets where original certificates were unavailable. As a result, the entire resource assay table was rebuilt and validated against original source information. During the review, errors noted included but were not limited to: missing below detection values, lack of substitution of high-grade Au values with gravimetric or metallic screened results where fire assay AAS or ICP results were beyond method range, intervals with zero length, errors due to conversion of Au PPM results to opt and back to PPM, errors due to incorrect conversion from interval feet to metres, truncation of decimal places in original footages, and inclusion of QA/QC samples.

A significant issue identified in the assay table was the lack of below detection values. Where there should have been a below detection value (i.e., a value of <0.005 PPM), there was no value. Below detection values are denoted by either a < or - symbol preceding the value in the digital assay results provided by the assay lab. Of the 11,201 values added to the assay table, 9,651 were below detection values. The validation process also resulted in the correction of 5,755 assay values that were either an incorrect value or assigned to the wrong interval, including 8 values recorded in the final Au PPM column (AuFinPPM) as > 10 PPM (indicating the grade is higher than the range of the analytical method).

All original logging and sampling were recorded in feet, so for ease of reference, the original intervals in feet were added to the table beside the conversion to metres. Comments were added to the assay table where results were updated, stating source of the values. Original certificate and sample identification numbers were also added to the assay table for reference.

The following rules were applied to the rebuilt assay table:

- All assays reported as below detection by the reporting analytical laboratory were set to half the detection limit of the assay method as reported on the certificate of analysis (i.e., <0.05 PPM was changed to 0.025 PPM).
- Duplicate samples were averaged.
- Where gravimetric analysis grades were available, they were used as the final grade.
- Where metallic screening analysis grades were available, they were used as the final grade, replacing gravimetric analysis where both were available.

As part of WSP’s standard analytical data reviews, tabular grade data is compared against signed assay certificates from the laboratories that performed the analytical test work to ensure the tabular data is free from transcription errors or omissions.

12.2.3 Other Data Verification

During the previous scope of work, WSP performed high level reviews of the topographic data and topographic surface models for the Project using the drill hole collar elevations as spot checks against the topographic model elevations.

In 2022, an aerial flyover of the Tonopah property was performed by MWH Geo-Surveys International Inc. using a Professional Mapping UAV, producing a GeoTIFF orthophoto and DSM at a resolution of 25 cm. WSP converted the DSM to a topography triangulation surface with a 10 m triangle resolution using Global Mapper GIS software for use in the geological model and for collar validation checks.

The collar locations for the drill holes used in the model were validated against the topography surface in Maptek Vulcan™ (Vulcan) modelling software. The Z coordinates of the collars were compared to the Z coordinates of the surface at the same X-Y coordinates. 563 of 626 drill holes are within the aerial flyover area. The drill hole collars were both above and below the surface with a minimum absolute difference of 0.1 m, a maximum difference of 22.39 m, and a mean difference of 1.5 m. WSP assigned a flag between 1 to 5 to each drill hole, ranking the severity of the difference. Table 12.3 summarizes the number of holes in each flag category.

Table 12.3: Number of Holes in each Elevation Difference Category

Rank	Difference Range	No. of Holes
1	0 to 1 m	325
2	1 to 2 m	137
3	2 to 3 m	39
4	3 to 6 m	38
5	>6 m	24

After discussion with Viva, the decision was made to adjust the collar elevation of all holes with a difference greater than 2 m to match the 2022 topography surface. All holes drilled by Viva were also adjusted to match the topography surface, regardless of the difference.

12.2.4 Limitations on Data Verification

The WSP QP was not directly involved in the exploration drilling and sampling programs that formed the basis for collecting the data used in the geological modelling and Mineral Resource estimates for this Project. During a site visit in June 2023 (see Item 12.1), the WSP QP was able to observe the drilling and sampling that was in progress for the 2023 drill program. The WSP QP was not able to observe the drilling, sampling, or sample preparation while in progress for the pre-2023 programs and therefore WSP has had to rely upon forensic review of the exploration program data, documentation, and standard database validation checks to ensure the resultant geological database is representative and reliable for use in geological modelling and Mineral Resource estimation.

12.3 Qualified Persons Opinion of the Adequacy of Data

The QP has validated and verified the data used in the preparation of the Mineral Resource. In addition, the QP observed recent Viva drilling, geological logging, and sampling procedures at Tonopah and considers the data and procedures to be generally consistent with industry best practices as described in CIM Estimation of Mineral

Resources and Reserves Best Practice Guidelines (MRMR Best Practice Guidelines), issued November 29, 2019. The QP considers the data suitable for supporting the Mineral Resource stated in Item 14.0.

The QP has the following recommendations for Viva to consider regarding data collection:

- Drill hole collars should be surveyed with a differential GPS to improve accuracy and marked with a permanent ground marker that will survive environmental conditions. An example would be a concrete cap or monument to mark the collar, with the drill hole ID written either in the concrete or written on a metal tag attached to the monument. The use of a handheld GPS is acceptable for exploration stage projects but requires increased level of accuracy for PEA and above level studies.
- An increase in the number of DD core holes vs. RC holes is recommended for improved data quality and understanding of the deposit. Core holes allow the collection of bulk density measurements, rock quality information, fault/geotechnical parameters locations and angles, and alteration zones, that are not easily (if at all) available from the collection of RC drilling samples.
- Additional bulk density measurements should be taken on future core drilling, including intervals of known waste rock. Suitable QA/QC procedures should be included as part of the bulk density measurements, and the use of a waterproof coating should be used, especially in fractured rock. If possible, bulk density measurements should be taken on historical drill core, spaced evenly throughout the main zone of the current resource pit shell extents.
- Field duplicate samples should be collected for all future diamond drilling to evaluate the accuracy of the sampling at core splitting stage. Viva should also collect several duplicate RC samples to test the variability between samples of the same interval using this method of drilling.
- Down-hole deviation measurements should be taken on all drill holes.
- Regular check sampling by a third-party (umpire) analytical laboratory should be conducted.
- As recommended by the CIM MRMR Best Practice Guidelines, drill logs and sample results should be stored in a relational database that provides proper control and security. The database should contain all relevant data for each drill hole, including, but not limited to drill hole ID, collar location and orientation, total depth, down-hole deviation measurements, hole diameter, geological data (lithology, alteration, core recovery, RQD), analytical data (unique sample ID, analytical laboratory name, analytical certificate number, assay results, including any trace or deleterious elements, geometallurgical results, bulk density, QA/QC data). An MS Access database would be suitable for this purpose.

13.0 Mineral Processing and Metallurgical Testing

The assumptions or predictions regarding recovery estimates are forward-looking information. The material factors that could cause actual results to differ materially from these assumptions and predictions include actual plant feed characteristics that are different from the historical operations or from samples tested to date, and actual plant flowsheet, equipment and operational performance that yield different results from the historical operations and historical and current test work results.

Several scoping-level metallurgical studies were undertaken by various operators, including Kennecott (1994-1996), Newmont (2003) and Midway Gold (2006 to 2009) for the Tonopah property and are summarized in Table 13.1.

Table 13.1: Pre-Viva Metallurgical Test Work

Operator	Year	Laboratory	Description
Kennecott Exploration/ Kennecott Minerals Company	1994	Barringer Laboratories, Inc.	Head analysis and cyanide shake tests on 350 drill pulp samples of all rock types, alteration types, and grades to evaluate cyanide extraction as a suitable methodology for processing
	1995	McClelland Laboratories, Inc.	Bottle roll cyanidation leach tests on a composite of drill core material from 1 hole at 2 grind sizes to determine if recovery is influenced by grind size, and bulk density measurements of 15 core samples from 3 drill holes
	1996	Dawson Metallurgical Laboratories, Inc.	Bottle roll cyanidation leach test on a composite of drill pulp from 1 hole at a coarse grind size to determine Au extraction from coarse grinding
	1996	Rocky Mountain Geochemical Corp.	Cyanide shake tests at longer leach times on a single composite from 1 drill hole to determine why 1 h shake tests on sulfide mineralization intervals in this hole indicated low Au recoveries, while bottle roll test work at 96 h yielded high recovery
Newmont Mining Corporation	2003	American Assay Labs, ALS Chemex, Newmont	Head analysis to optimize the sample collection and preparation procedures for Au assaying due to significant differences in assays from the nugget effect
Midway Gold Corp	2006	McClelland Laboratories, Inc.	Gravity concentration testing on 4 composites to evaluate the response of Au recovery vs. feed grade
	2006	SGS Lakefield	Pre-concentration testing on 1 composite sample by flotation and gravity concentration, as well as cyanidation of the gravity tails
	2007	Barrick Goldstrike Metallurgical Services	Bond Work Index and Direct Carbon-in-Leach (CIL) testing on 1 sample to determine hardness and CIL Au recovery for potential processing at Round Mountain
	2008 - 2009	Gekko Systems	Gravity concentration at two grind sizes and flotation test work followed by cyanide leaching on a single composite to determine best processing methods

Most of the pre-Viva metallurgical test work focused on a limited number of samples of higher-grade vein and breccia type material and may not be representative of the deposit as a whole. Additional metallurgical test work programs were commissioned by Viva between 2018 and 2023 and completed by Resource Development Inc (RDi), MLI, and KCA; which form the basis for the conclusions derived in this study. Selected test work and results from these programs are summarized chronologically below and are referenced in this report. Although condensed, for the sake of completeness as much relevant data as practical are presented herein.

13.1 Viva 2018-2023 Metallurgical Test Work Programs

13.1.1 2018 RDi Cyanide Leach Testing

In late 2018, Viva collected samples from previous drilling campaigns which had been retained by previous operators for bucket leach testing by RDi. The samples tested were assembled from a number of core intercepts from the Company's core inventory, averaging between 0.5 and 1.0 g/t grade. Assayed head grade for the Tertiary volcanic sample was 0.88 g/t Au and 0.72 g/t for the Op argillite sample. The samples were subject to static bucket leach tests for a nominal 1-inch (2.54 cm) crush size and bottle roll tests for material ground to nominal 3.35 mm and 75 µm (6-mesh and 200-mesh) sizes. Results from this work are shown in Table 13.2.

Table 13.2: RDi Cyanide Leach Test Summary

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

Notes: TV = Tertiary volcanic, OP = Ordovician Palmetto argillite

The bottle roll results showed improved recoveries for both material types at finer product sizes for both Au and Ag values. It was noted that the Tertiary volcanic material selected for the composite may not be fully representative of the project metallurgy, since it was generally comprised of quartz rich Tertiary volcanics from the discovery zone.

13.1.2 2019 McClelland Laboratories Metallurgical Testing

Results for the 2019 MLI test program summarized herein are extracted from the MLI report titled "Report on Bottle Roll and Column Leach Testing – Midway Drill Core Composites MLI Job No. 4394" dated 20 December 2019.

The 2019 MLI program was completed on coarse RC drilled assay rejects submitted by Viva. In total, 156 bags or material at a nominal size of 1.7 mm was received which was sorted to produce 20 composite samples as directed by Viva. Each of the composites was subjected to head analyses and bottle roll leach testing. Four master composites were generated from the 20 composite samples which were used for agglomerate strength and stability tests. column tests, and scoping gravity concentration tests.

The original twenty composites were segregated by drill hole, rock type and depth and represented Op argillite and the three different Tertiary volcanic lithologies (TRT, TRV and TVS). A summary of the head assay results for Au and Ag are presented in Table 13.3.

Table 13.3: MLI Head Assay Summary

Composite	Au			Ag		
	Average Au (g/t)	Std. Deviation	Precision (%)	Average Ag (g/t)	Std. Deviation	Precision (%)
TG1806	0.35	0.02	94.3	2.1	0.0	100.0
TG1807A	0.36	0.02	94.4	3.5	0.3	91.4
TG1807B	1.73	0.13	92.5	6.7	0.2	97.0
TG1807C	0.98	0.07	92.9	5.5	0.1	98.2
TG1808A	11.77	4.96	57.9	13.0	1.0	92.3
TG1808B	0.32	0.02	93.8	3.7	0.2	94.6
TG1808C	5.92	5.87	0.8	4.5	0.7	84.4
TG1808D	0.90	0.33	63.3	4.5	0.3	93.3
TG1809A	0.91	0.15	83.5	2.5	0.1	96.0
TG1809B	0.64	0.25	60.9	3.6	0.2	94.4
TG1810A	1.36	0.18	86.8	3.9	0.2	94.9
TG1810B	0.33	0.05	84.8	5.7	0.1	98.2
TG1811A	0.30	0.02	93.3	1.0	0.1	90.0
TG1811B	0.38	0.13	65.8	1.2	0.0	100.0
TG1812A	3.07	0.31	89.9	5.3	0.3	94.3
TG1812B	0.30	0.15	50.0	1.0	0.1	90.0
TG1813	0.44	0.07	84.1	1.8	0.1	94.4
TG1814A	0.42	0.02	95.2	2.6	0.2	92.3
TG1814B	2.47	0.10	96.0	6.8	0.1	98.5
TG1815	1.38	0.29	79.0	12.0	0.0	100.0

13.1.2.1 2019 MLI Bottle Roll Leach Tests

Bottle roll leach tests were performed on each of the 20 composite samples and four master composites at the as-received material size (approximately 1.7 mm). The bottle roll tests were performed on 1 kg splits and leached for 96 h with a sodium cyanide (NaCN) concentration of 1.0 g NaCN/L. Results for the bottle roll tests are presented in Table 13.4.

Table 13.4: MLI Bottle Roll Tests Results

Composite	Rock Type	Au Recovery (%)	Au Calculated Head (g/t)	Ag Recovery (%)	Ag Calculated Head (g/t)	Reagent Requirements (kg/t)	
						NaCN Cons.	Lime Added
TG1806	Opa	55.6	0.36	10	2	0.07	1.4
TG1807C	Opa	61.3	1.19	9.1	5.5	0.11	1.5
TG1808A	Opa	89.1	11.49	41.8	13.4	0.14	1.3
TG1808B	Opa	64.5	0.31	10.8	3.7	<0.07	0.9
TG1808C	Opa	79.9	1.64	22.2	3.6	0.13	0.9
TG1808D	Opa	81.7	1.04	15.6	4.5	0.2	1
TG1809A	Opa	51.2	0.86	12.5	2.4	<0.07	0.9
TG1809B	Opa	85	0.8	14.3	3.5	0.13	0.9
TG1810B	Opa	48.3	0.29	7.4	5.4	0.15	1.1
TG1807A	TRT	47.6	0.42	5.7	3.5	0.17	1.1
TG1812B	TRT	72.2	0.36	16.7	1.2	<0.07	1.1
TG814A	TRT	52.8	0.53	11.5	2.6	0.08	0.7
TG1810A	TRV	42.6	1.22	15.8	3.8	0.12	1.1
TG1811A	TRV	71.4	0.28	9.1	1.1	<0.07	0.7
TG1811B	TRV	68.3	0.41	14.3	1.4	0.08	0.6
TG1812A	TRV	68.4	3.23	24.5	5.3	0.2	2.1
TG1815	TRV	45.5	1.43	4.8	10.4	0.06	1.5
TG1807B	TVS	56.9	1.95	16.4	6.1	<0.07	1
TG1813	TVS	59.2	0.49	16.7	1.8	0.09	0.8
TG1814B	TVS	65	2.74	15.6	6.4	<0.07	1.5
4394-OPA	Opa	89.2	2.95	30.4	5.6	0.08	1.2
4394-TRT	TRT	55.3	0.38	6.9	2.9	<0.07	1.1
4394-TRV	TRV	52.1	1.21	11.6	4.3	0.08	1.2
4394-TSV	TSV	53	1.64	14.5	5.5	0.07	1.2

Results for the bottle roll tests show variable recoveries for Au and Ag ranging between 42.6% and 89.2% for Au and 5.7% and 41.8% for Ag. Au recoveries for the argillic material averaged 70.6% compared to 57.9% for the volcanic material types with average Ag recoveries of 17.4% and 13.2%, respectively.

13.1.2.2 2019 MLI Column Leach and Agglomerate Strength Tests

Agglomerate strength tests were performed on 10 kg splits of each of the master composites to determine how much cement would be required to generate stable agglomerates and ensure no ponding or percolation issues occur during the column test program. The tests were performed by agglomerating the samples with cement, curing for 72 h, soaking for a period of 24 h, and observing the degree of agglomerate breakdown. Based on this method, a recommended cement addition ranging between 3.6 kilogram per tonne (kg/t) and 5.2 kg/t for the different material types were determined.

Column leach tests were performed on each of the master composites samples which were agglomerated with the cement requirements determined from the agglomerate strength tests. Results from the column leach tests are presented in Table 13.5.

Table 13.5: MLI Composite Column Leach Tests

Sample	Test No.	Leach/Rinse Time (Days)	Au Recovery (%)	Au Calculated Head (g/t)	Ag Recovery (%)	Ag Calculated Head (g/t)	Reagent Requirements (kg/t)	
							NaCN Cons.	Cement Added
4394-OPA	P-1	57	83.3	2.16	20.8	4.8	0.68	5.2
4394-TRT	P-2	57	47.9	0.48	6.9	2.9	0.58	3.6
4394-TRV	P-3	64	60.8	1.30	11.1	4.5	0.83	5.2
4394-TVS	P-4	66	63.8	1.74	16.7	5.4	0.99	4.4

Column recoveries at the as-received material size ranged from 47.9% and 83.3% for Au and 6.9% and 20.8% for Ag with low to moderate cyanide consumptions. Similar to the bottle roll tests, recoveries for the argillic material were higher compared to the volcanic material composites.

13.1.2.3 2019 MLI Gravity Concentration Tests

Gravity concentration tests were completed on splits from each of the four master composites to evaluate the amenability of the material to gravity concentration. Each sample was ground in a laboratory steel ball mill to a size of 80% -212 µm (65 mesh). The ground product was processed once through a Knelson concentrator to produce a rougher concentrate and rougher tailings. The rougher concentrate was cleaned once by hand panning to produce a cleaner concentrate and cleaner tailings. The gravity test results are presented in Table 13.6.

Table 13.6: MLI Gravity Concentration Tests

Composite	Weight, %				Assay, Au (g/t)						Au Distribution, %			
	Cl. Conc.	CL. Tail	Ro. Conc.	Ro. Tail	Cl. Conc.	CL. Tail	Ro. Conc.	Ro. Tail	Calc'd. Head	Avg. Head	Cl. Conc.	CL. Tail	Ro. Conc.	Ro. Tail
Opa	0.19	0.56	0.75	99.25	605.00	34.10	179.00	0.78	2.11	1.98	54.40	9.00	63.40	36.60
TRT	0.19	0.49	0.68	99.32	29.30	3.57	10.80	0.31	0.38	0.38	14.60	4.60	19.20	80.80
TRV	0.19	0.53	0.72	99.28	198.00	21.80	68.30	0.76	1.25	1.35	30.20	9.30	39.50	60.50
TVS	0.19	0.54	0.73	99.27	234.00	18.30	74.40	1.10	1.64	1.68	27.20	6.00	33.20	66.80

The gravity concentration results, although varied, showed that the material generally responded well to gravity concentration, although no free Au was observed.

13.1.3 2021-2023 Kappes, Cassiday & Associates Metallurgical Test Work

Two test work programs were completed by KCA between 2021 and 2023 and are titled “Tonopah Gold Project Argillite and Volcanic Composites Report of Metallurgical Test Work” dated February 2022 and “Tonopah Pulp Agglomeration Report of Metallurgical Test Work” dated March 2023. The first program evaluated High Pressure Grinding Roll (HPGR) fine crushing to potentially improve heap leach recoveries with the second program evaluating the potential of pulp agglomeration for treating a combination of high- and low-grade material. Results from these programs are discussed in the following sub-items.

13.1.3.1 2022 KCA Test Program

KCA received 9 pallets containing core boxes representing material from five (5) drill holes (TGM2001 through TGM2005) from the Tonopah Gold Project. The material was described as argillite or volcanic material and was received as half split, quarter split and broken PQ core.

The received material was utilized to generate ten composite samples, including three argillite composites, six volcanic composites, and one Master Composite. The composite samples were crushed using a HPGR crushing method and the crushed products were then utilized for metallurgical test work. The Master Composite MC1 was a 1:1 blend of argillite material and volcanic material split out from select composites. The composite samples were utilized for column leach testing on HPGR product and bottle roll leach testing on HPGR product, fine crushed, milled and pulverized material. For the coarse leach test work, a portion of the final HPGR products (edge + center material) for each composite was utilized for leach testing. Head assays for Au and Ag are presented in Table 13.7.

Table 13.7: 2022 KCA Head Assay Results

KCA Sample No.	Description	Au Average Head Assay (g/t)	Ag Average Head Assay (g/t)	Calc. P_{80} Size (mm)	Wt. Passing 1.70 mm (%)	Wt. Passing 0.212 mm (%)	Au Weighted Avg. Head Assay (g/t)	Ag Weighted Avg. Head Assay (g/t)
92339 A	Argillite Composite 1	1.430	3.80	6.1	57.2	23.2	1.041	2.07
92340 A	Argillite Composite 2	4.397	13.30	6.9	44.5	12.7	4.501	10.73
92341 A	Argillite Composite 3	1.545	4.80	5.2	52.6	20.5	1.539	2.76
	Average:	**2.457**	**7.30**	**6.1**	**51.4**	**18.8**	**2.360**	**5.19**
92342 A	Volcanic Composite 1	1.498	6.60	7.0	44.8	13.7	1.065	5.30
92343 A	Volcanic Composite 2	0.297	5.61	6.7	47.7	13.1	0.268	3.64
92344 A	Volcanic Composite 3	0.327	2.40	7.5	39.5	11.4	0.312	0.83
92345 A	Volcanic Composite 4	0.765	2.61	8.7	33.9	9.2	0.744	2.65
92346 A	Volcanic Composite 5	2.469	5.90	8.0	38.6	9.5	2.622	6.45
92347 A	Volcanic Composite 6	0.379	3.31	7.2	44.0	13.2	0.312	2.53
	Average:	**0.956**	**4.40**	**7.5**	**41.4**	**11.7**	**0.887**	**3.57**
92348 A	Master Composite MC1	2.331	7.10	6.3	48.3	15.9	2.367	7.17

The direct assays and calculated head assays for Au varied significantly during the test program, likely due to nugget effects. The most significant variations were noted in the argillite composites, where the relative standard deviations ranged from 10% to 28% across all of the head assays and calculated heads. In the volcanic composites, the relative standard deviations ranged from 6% to 15% across all of the head assays and calculated heads. The Master Composite MC1 had a relative standard deviation of 8% across the head assays and calculated heads.

13.1.3.1.1 2022 KCA Bottle Roll Leach Tests

Bottle roll leach tests were conducted on HPGR and conventional fine crushed material, milled material and pulverized material. Results of the bottle roll tests are summarized in Table 13.8.

Table 13.8: 2022 KCA Bottle Roll Leach Test Results

KCA Sample No.	Description	Crush Type	Target P_{80} (mm)	P_{80} Size (mm)	Au Calculated Head (g/t)	Au Extracted (%)	Ag Calculated Head (g/t)	Ag Extracted (%)	Leach Time (hours)	Consumption NaCN (kg/t)	Addition $Ca(OH)_2$ (kg/t)
92339 A	Argillite Composite 1	HPGR	--	4.8	0.953	45%	2.50	12%	144	0.01	0.50
92339 A	Argillite Composite 1	Conv.	1.7	--	0.939	56%	2.89	12%	96	0.00	0.75
92339 A	Argillite Composite 1	Pulv.	0.106	--	1.069	97%	3.07	30%	48	0.24	0.75
92339 A	Argillite Composite 1	Milled	0.075	--	1.215	96%	2.53	39%	48	0.71	0.50
				Average:	**1.044**		**2.75**				
92340 A	Argillite Composite 2	HPGR	--	6.2	4.320	40%	13.11	12%	144	0.04	0.75
92340 A	Argillite Composite 2	Conv.	1.7	--	3.913	53%	12.72	16%	96	0.07	1.00
92340 A	Argillite Composite 2	Pulv.	0.106	--	3.622	91%	12.96	38%	48	0.20	1.00
92340 A	Argillite Composite 2	Milled	0.075	--	3.817	93%	13.27	42%	48	0.86	0.75
				Average:	**3.918**		**13.01**				
92341 A	Argillite Composite 3	HPGR	--	7.8	1.156	58%	3.23	13%	144	0.04	0.50
92341 A	Argillite Composite 3	Conv.	1.7	--	1.582	62%	4.00	20%	96	0.03	0.75
92341 A	Argillite Composite 3	Pulv.	0.106	--	1.296	94%	3.94	29%	48	0.19	0.75
92341 A	Argillite Composite 3	Milled	0.075	--	1.993	96%	4.08	37%	48	0.18	0.75
				Average:	**1.507**		**3.81**				
92342 A	Volcanic Composite 1	HPGR	--	7.2	1.025	38%	5.73	17%	144	0.06	0.50
92342 A	Volcanic Composite 1	Conv.	1.7	--	1.274	52%	5.87	21%	96	0.03	0.75
92342 A	Volcanic Composite 1	Pulv.	0.106	--	1.114	90%	5.98	35%	48	0.20	0.50
92342 A	Volcanic Composite 1	Milled	0.075	--	1.162	92%	8.46	59%	48	1.79	0.50
				Average:	**1.144**		**6.51**				
92343 A	Volcanic Composite 2	HPGR	--	6.9	0.276	83%	3.86	7%	144	0.03	0.50
92343 A	Volcanic Composite 2	Conv.	1.7	--	0.250	75%	4.15	7%	96	0.03	0.50
92343 A	Volcanic Composite 2	Pulv.	0.106	--	0.283	96%	4.61	22%	48	0.08	0.75
92343 A	Volcanic Composite 2	Milled	0.075	--	0.317	96%	4.14	22%	48	1.43	0.50
				Average:	**0.282**		**4.19**				

KCA Sample No.	Description	Crush Type	Target P_{80} (mm)	P_{80} Size (mm)	Au Calculated Head (g/t)	Au Extracted (%)	Ag Calculated Head (g/t)	Ag Extracted (%)	Leach Time (hours)	Consumption NaCN (kg/t)	Addition $Ca(OH)_2$ (kg/t)
92344 A	Volcanic Composite 3	HPGR	--	7.7	0.304	83%	1.19	25%	144	0.03	0.50
92344 A	Volcanic Composite 3	Conv.	1.7	--	0.284	80%	1.77	20%	96	0.07	0.50
92344 A	Volcanic Composite 3	Pulv.	0.106	--	0.309	97%	1.96	60%	48	0.17	0.50
92344 A	Volcanic Composite 3	Milled	0.075	--	0.374	94%	1.16	56%	48	0.87	0.50
				Average:	**0.318**		**1.52**				
92345 A	Volcanic Composite 4	HPGR	--	7.5	0.846	66%	1.04	42%	144	0.01	0.63
92345 A	Volcanic Composite 4	Conv.	1.7	--	0.748	89%	2.17	25%	96	0.07	0.75
92345 A	Volcanic Composite 4	Pulv.	0.106	--	0.745	99%	1.83	51%	48	0.14	1.00
92345 A	Volcanic Composite 4	Milled	0.075	--	0.745	95%	2.07	49%	48	1.35	0.50
				Average:	**0.771**		**1.78**				
92346 A	Volcanic Composite 5	HPGR	--	6.8	2.657	34%	5.81	11%	144	0.06	0.50
92346 A	Volcanic Composite 5	Conv.	1.7	--	2.695	42%	5.86	15%	96	0.10	0.50
92346 A	Volcanic Composite 5	Pulv.	0.106	--	2.293	87%	6.36	39%	48	0.17	0.75
92346 A	Volcanic Composite 5	Milled	0.075	--	2.457	93%	6.00	43%	48	1.08	0.50
				Average:	**2.526**		**6.01**				
92347 A	Volcanic Composite 6	HPGR	--	6.3	0.359	48%	1.47	32%	144	0.06	0.88
92347 A	Volcanic Composite 6	Conv.	1.7	--	0.379	47%	2.37	36%	96	0.07	0.75
92347 A	Volcanic Composite 6	Pulv.	0.106	--	0.386	84%	1.99	45%	48	0.17	0.75
92347 A	Volcanic Composite 6	Milled	0.075	--	0.391	86%	2.26	48%	48	0.29	0.75
				Average:	**0.379**		**2.02**				
92348 A	Master Composite MC1	HPGR	--	7.1	2.744	36%	7.88	13%	144	0.04	0.63
92348 A	Master Composite MC1	Conv.	1.7	--	2.305	54%	7.49	20%	96	0.07	0.75
92348 A	Master Composite MC1	Pulv.	0.106	--	2.174	91%	7.73	39%	48	0.16	0.75
92348 A	Master Composite MC1	Milled	0.075	--	2.187	92%	7.52	38%	48	1.24	0.50
				Average:	**2.352**		**7.65**				

In the bottle roll leach tests, the Au extractions improved with finer crush sizes. In all of the composite samples, a notable increase in Au extraction was observed when the crush size was reduced from 1.70 mm to 0.106 mm. The Au extractions increased by 32% to 41% in the argillite composites, 10% to 46% in the volcanic composites and 37% in the Master Composite. Further size reduction did not have as significant an effect. The Au extractions in the 0.106 mm tests and the 0.075 mm tests were similar for most of composite samples, with the largest difference observed in the Volcanic Composite 5 (5% increase).

13.1.3.2 2022 KCA Preliminary Agglomeration and Column Leach Tests

Preliminary agglomeration test work was conducted on portions of the final HPGR product for each composite sample. Additionally, compacted permeability test work was conducted on portions of the final HPGR product for the Master Composite MC1. Each separate composite was tested with 0 (no cement addition), 2, 4 and 8 kg/t of cement agglomeration levels. Half of the composite samples tested passed the KCA criteria for flow rate with no cement addition. The effluent flow rate was too low without cement for Argillite Composite 1, Argillite Composite 3, Volcanic Composite 2, Volcanic Composite 5, and the Master Composite MC1. Compacted permeability test results for the Master Composite are presented in Table 13.9. The samples passed at both 2 and 4 kg/t cement additions at simulated heap heights up to 60 m. Column leach tests were conducted on each of the composites and are summarized in Table 13.10.

Table 13.9: 2022 KCA Compacted Permeability Test Work Results

KCA Sample No.	KCA Test No.	Sample Desc.	Material Size	Calc. P_{80} Size (mm)	Test Phase	Cement Added (kg/t)	Effective Height (m)	Flow Rate (L/H/m^2)	Flow Result (Pass/Fail)	Incremental Slump (%)	Cum. Slump (% Slump)	Slump Result (Pass/Fail)	Overall (Pass/Fail)
92348 A	92385 A	Master Composite MC1	HPGR Crushed	6.3	Primary	4	10	2,185	Pass	1%	1%	Pass	Pass
					Stage Load		20	1,979	Pass	1%	3%	Pass	Pass
					Stage Load		40	1,865	Pass	1%	4%	Pass	Pass
					Stage Load		60	1,844	Pass	0%	4%	Pass	Pass
92348 A	92385 B	Master Composite MC1	HPGR Crushed	6.3	Primary	2	10	2,536	Pass	2%	2%	Pass	Pass
					Stage Load		20	1,766	Pass	2%	4%	Pass	Pass
					Stage Load		40	749	Pass	2%	6%	Pass	Pass
					Stage Load		60	688	Pass	0%	6%	Pass	Pass

Table 13.10: KCA Column Leach Test Results

KCA Sample No.	Description	Au Calculated Head (g/t)	Extracted Au (g/t)	Extracted (% Au)	Ag Calculated Head (g/t)	Extracted (% Ag)	Calculated Tail P_{80} Size (mm)	Days of Leach	Consumption NaCN (kg/t)	Addition Hydrated Lime (kg/t)	Addition Cement (kg/t)
92339 A	Argillite Composite 1	0.91	0.64	70%	3.10	14%	5.6	84	0.85	0.00	4.00
92340 A	Argillite Composite 2	4.27	1.92	45%	12.93	13%	6.5	84	0.92	0.00	4.02
92341 A	Argillite Composite 3	1.34	0.86	64%	4.23	12%	4.6	84	0.85	0.00	4.00
92342 A	Volcanic Composite 1	1.34	0.87	65%	6.30	14%	7.1	84	0.76	0.00	4.00
92343 A	Volcanic Composite 2	0.32	0.26	80%	4.81	6%	6.8	84	0.52	0.00	4.02
92344 A	Volcanic Composite 3	0.34	0.30	86%	1.69	23%	7.3	81	0.68	1.00	0.00
92345 A	Volcanic Composite 4	0.83	0.73	88%	2.19	21%	7.3	81	0.65	1.01	0.00
92346 A	Volcanic Composite 5	2.62	1.01	39%	7.38	7%	7.7	81	0.74	0.00	4.03
92347 A	Volcanic Composite 6	0.37	0.22	61%	2.68	21%	6.9	81	0.94	1.01	0.00
92348 A	Master Composite MC1	2.39	1.26	53%	8.37	13%	5.9	81	0.74	0.00	4.01

Results from the column leach tests showed variable recovery ranging from 39% to 88% for Au and 6% to 23% for Ag. In general, higher recoveries were associated with lower grade material. Assays by size fraction on the column leach test tailings also indicate a benefit to size reduction. For most of the tests, the assays of the column tails were notably lower for the size fractions less than 0.212 mm than those above 0.212 mm.

Compared to bottle roll leach tests on HPGR crushed material, the longer leach times in the column tests resulted in significantly higher Au extractions for most of the composites. The Au extractions increased by 4% to 26% in the argillite composites and 17% in the Master Composite. One volcanic composite (Volcanic Composite 2) showed a slightly lower (3%) Au extraction in the column test versus the bottle roll test, but the rest of the volcanic composites showed increased Au extractions, ranging from 3% to 27%.

13.1.3.3 2022 KCA Physical Characterization Work

Two samples representing the volcanic material and the argillite material were submitted to Hazen Research, Inc. in Golden, Colorado for comminution testing. Test work was completed to provide Bond Crusher Work indices and abrasion indices for the samples. The results are summarized in Table 13.11.

Table 13.11: KCA Bond Crusher Work Indices and Abrasion Indices

KCA Sample No.	Drill Hole ID	Rock Type	Cw_i (kWh/t)	A_i (g)
92317 A	TGM2001	Volcanic	10.1	0.3542
92318 A	TGM2001	Argillite	4.6	0.0101

13.1.3.4 2023 KCA Test Program

Stored reject material from the Tonopah Gold Project was utilized to generate a high-grade composite sample and a low-grade composite sample for pulp agglomeration test work, which considers agglomerating a milled high-grade material with a coarse crushed low-grade material for recovery by heap leaching. Head analyses for the composites are presented in Table 13.12.

Table 13.12: KCA 2023 Pulp Agglomeration Composite Head Analyses

KCA Sample No.	Description	Au Assay 1 (g/t)	Au Assay 2 (g/t)	Au Average Assay (g/t)
93919	Low-Grade Composite	0.183	0.185	0.184
93920	High-Grade Composite	3.434	3.333	3.383

KCA Sample No.	Description	Ag Assay 1 (g/t)	Ag Assay 2 (g/t)	Ag Average Assay (g/t)
93919	Low-Grade Composite	2.40	2.09	2.25
93920	High-Grade Composite	11.30	11.90	11.60

Notes: The detection limit for silver with FAAS finish is 0.21 g/t. For the purpose of calculation, a value of 1/2 the detection limit is utilized for assays less than the detection limit.

13.1.3.5 2023 KCA Pulp Agglomeration Testing

Direct and Carbon-in-Leach (CIL) bottle roll leach testing was conducted on milled portions of the high-grade composite material with portions of the tail material from selected CIL bottle roll leach tests being hand blended with the low-grade composite material in a 1:4 part ratio to generate pulp agglomerated material utilized for column leach testing.

The weighted average Au extraction for the CIL bottle roll leach tests utilized for the column leach test composite was 65% after an eight (8) h leach period based on a weighted average head grade of 2.419 g/t Au. NaCN consumption was 0.56 kg/t. Hydrated lime addition was 0.75 kg/t.

Preliminary agglomeration test work as well as compacted permeability test work was conducted on portions of the pulp agglomeration test material. The pulp agglomerated material tested with 6 and 12 kg/t cement additions both passed the KCA test criteria for slump and flow rate at 10 m of overall heap height. However, the material agglomerated with both 6 and 12 kg/t cement failed at overall heap heights of 20, 40 and 60 m.

A column leach test was conducted utilizing the pulp agglomeration test material. The test material was blended with 17.64 kg/t of cement prior to loading the column. During testing, the material was leached for 86 days with a NaCN solution.

The overall Au extraction for the pulp agglomerated material was 77% based on a calculated head grade of 1.153 g/t Au. Twenty-seven percent (27%) of the total Au extracted was extracted in the CIL bottle roll leach tests. The remaining fifty percent (50%) of the total Au extracted was extracted in the column leach test. The overall Au extraction balance is presented in Table 13.13.

Table 13.13: KCA Pulp Agglomeration Au Extraction Balance

Description	Au Calculated Head (g/t)	Au Equiv. BRT Extracted (g/t)	Au Column Extracted (g/t)	Au Overall Extracted (g/t)	Au Column Tail (g/t)
Total Feed (HG BRT + HG/LG Column)	1.153	0.314	0.579	0.893	0.26
Wt. Fract Au	100%	27%	50%	77%	23%

Notes: BRT = Bottle roll test, HG = High-grade, LG = Low-grade

The pulp agglomeration testing showed reasonable recoveries; however, the cement agglomeration requirements were very high. Although promising, it was noted that pulp agglomeration was unlikely to be economically beneficial with the high cement requirements.

A second round of pulp agglomeration testing was performed on half core from the Tonopah Project which was used to generate a low-grade composite sample and a high-grade composite sample. A column leach test was conducted utilizing the low-grade test material. The test material was blended with 1 kg/t of hydrated lime prior to loading the column. During testing, the material was leached for 87 days with a NaCN solution and realized an Au extraction of 58% based on a calculated head grade of 0.228 g/t Au.

CIL bottle roll leach testing was conducted on milled portions of the high-grade composite material. Portions of the tail material from selected CIL bottle roll leach tests were hand blended with the low-grade composite material in a 1:4 part ratio to generate pulp agglomerated material utilized for column leach testing.

The weighted average Au extraction for the CIL bottle roll leach tests utilized for the column leach test composite was 66% after a twenty-four (24) h leach period based on a weighted average head grade of 3.299 g/t Au. NaCN consumption was 1.07 kg/t. Hydrated lime addition was 0.75 kg/t.

Preliminary agglomeration test work as well as compacted permeability test work was conducted on portions of the pulp agglomeration test material. The pulp agglomerated material tested with 18 kg/t cement addition both passed the KCA test criteria for slump and flow rate at 10, 20, 40 and 60 m of overall heap height.

A column leach test was conducted utilizing the pulp agglomeration test material. The test material was blended with 18 kg/t of cement prior to loading the column. During testing, the material was leached for 87 days with a NaCN solution.

The column leach test Au extraction for the pulp agglomerated material was 69% based on a calculated head grade of 0.406 g/t Au. Fifty-two percent (52%) of the total Au extracted was extracted in the CIL bottle roll leach tests. The remaining thirty-three (33%) of the total Au extracted was extracted in the column leach test. The overall Au extraction balance is presented in Table 13.14.

Table 13.14: KCA Pulp Agglomeration Au Extraction Balance Round 2

Description	Au Calculated Head (g/t)	Au Equiv. BRT Extracted (g/t)	Au Column Extracted (g/t)	Au Overall Extracted (g/t)	Au Column Tail (g/t)
Total Feed (HG BRT + HG/LG Column)	0.8422	0.4362	0.281	0.7172	0.125
Wt. Fract Au	100%	52%	33%	85%	15%

Notes: BRT = Bottle roll test, HG = High-grade, LG = Low-grade

The overall pulp agglomerated column leach test was an Au extraction of 85%. Results from the second round of testing provided confirmation of the results from the first round which suggest that cement addition requirements are likely to limit the potential benefits of pulp agglomerating.

13.1.3.6 2023 KCA Solid-Liquid-Separation Test Results

Solid Liquid Separation test work was conducted by Pocock Industrial (Pocock) on one CIL Pulp Tails sample on behalf of KCA as part of the 2023 test program. The overall objective of the test was to develop a general set of data for design of thickening and filtration equipment intended to dewater the material prior to further processing.

Prior to conducting formal solid / liquid separation studies, flocculant screening tests were performed on small pulp samples to determine the relative effectiveness of flocculant in areas such as floccule particle formation, the capture of fines, liquor release and the approximate dosage level required. Although many types of flocculants provided some degree of response, SNF AN910 SH (a medium to high molecular weight, 10-15 charge density, anionic polyacrylamide) was shown to produce a slightly more robust floccule structure than the other types tested.

Table 13.15 provides high-rate thickener design criteria and operating parameters for the material tested. The design conditions shown represent a moderately aggressive design philosophy and correspond to the feed and overflow solids concentrations and flocculant dosages indicated.

Table 13.15: Recommended High-Rate Thickener Operating Parameter Ranges

Material Tested	Tested Feed Solids (1) (%)	Flocculant			Design Basis Net Feed Loading (m^3/m^2h) (5)	Predicted Overflow TSS Conc. Range (mg/L) (6)	Predicted Underflow Density (7)
		Type (2)	Dose (3) (g/t)	Conc. (4) (g/L)			
CIL Pulp Tails	25	SNF AN 910 SH	50 - 55	0.1	2.57	150 - 250	59%

Notes:

1) Feed solids concentration range required for thickener operation (wt. %) at maximum design Net Feed Loading Rate. Note: Maintaining feed solids concentration in the ranges shown is critical to thickener performance and operation at design rates shown.
2) Recommended SNF 910 SH flocculant type. Flocculants from other manufacturers with similar specifications would also serve.
3) Recommended flocculant dose in grams per metric tonne (g/t).
4) Recommended flocculant concentration prior to contact with the pulp.
5) Recommended design basis (net feed loading rate) in cubic metres of feed slurry per hour per square metre of thickener area (m^3/m^2 h). This basis can be used to calculate the required thickener area based on the volumetric feed rate at the design solids concentration. The feed loading rates shown correspond to the feed solids concentrations shown in the table. Since hydraulic design bases are specified independent of solids tonnage, an operable feed solids concentration range is required to properly specify a thickener designed using hydraulic feed loading rate. Recommended design net feed loading rates are provided without scale-up or safety factors.
6) Overflow suspended solids conc. in milligrams per litre as measured using a 0.45 m septum.
7) Maximum underflow solids concentration recommended based on viscosity considerations and experience with similar materials.

13.2 Metallurgical Test Summary and Conclusions

Results from the metallurgical test data indicate that Au and Ag mineralization from the Tonopah project are amenable to recovery by gravity concentration and cyanide leaching. Test work was completed on both fully oxidized and sulphide samples, with little difference noted in recoveries.

Recommended key design parameters for the project are summarized as follows and discussed in greater detail in the following sub-item:

- Heap Leach Parameters:
  - Crush size of 100% passing 12.5 mm.
  - Variable recoveries for Au and Ag based on head grade and material type with an overall Au recovery of 76% for grades of 0.5 g/t or less for both volcanic and argillic material types and Ag recoveries of 15% and 19% for Ag grades of 1.0 g/t or less for argillic and volcanic material, respectively.
  - Design leach cycle of 120 days.
  - Agglomeration will be required with an estimated cement addition of 4.0 kg/t material.
  - Cyanide consumption of 0.26 kg/t material.
- Mill parameters:
  - Grind size of 80% minus 106 µm (150 mesh).
  - Au recovery of 95% for argillic material and 90% for volcanic material.
  - Ag recovery of 36% for argillic material and 38% for volcanic material.
  - 48 h leach residence time.

- Lime addition of 0.60 kg/t material.
- Cyanide consumption of 0.58 kg/t material.

In general, the Tonopah deposit shows some variability in recoveries by the two primary material types with heap recoveries for Au being more sensitive to head grade than product crush size. Coarse Au is present in some of the material and may be contribution to the lower recoveries for high-grade heap leach material. Au recoveries for milled material were consistently high at both 106 and 75 µm grind sizes (150 and 200 mesh) and gravity pre-concentration is recommended to recover coarse Au particles.

Sufficient test work has been completed to date to support the project evaluation at a preliminary economic assessment level. Future test programs should include additional variability testing for both material product size and grades and should include long duration column leach tests. Full cycle testing for milled material with gravity pre-concentration should also be performed.

13.2.1 Heap Leach Recovery and Reagents

Results from the column leach tests were evaluated based on crushed product size (Figure 13.1) and head grade (Figure 13.2 for Au and Figure 13.3 for Ag) to determine the effect on Au and Ag recoveries. The results show that for the crush sizes tested, there is not a significant correlation between the P_{80} crush size and recoveries for either Au or Ag; however, recoveries for Au do appear to be affected by the head grade with higher grades achieving lower recoveries. Ag recoveries for volcanic material also appear to decrease with higher Ag head grades while argillic material for Ag does not appear to be affected. Reduced recoveries for the higher-grade material are assumed to be a result of lower leach kinetics due to coarse Au; however, additional long duration column tests on high-grade material are needed to confirm if this is the case.

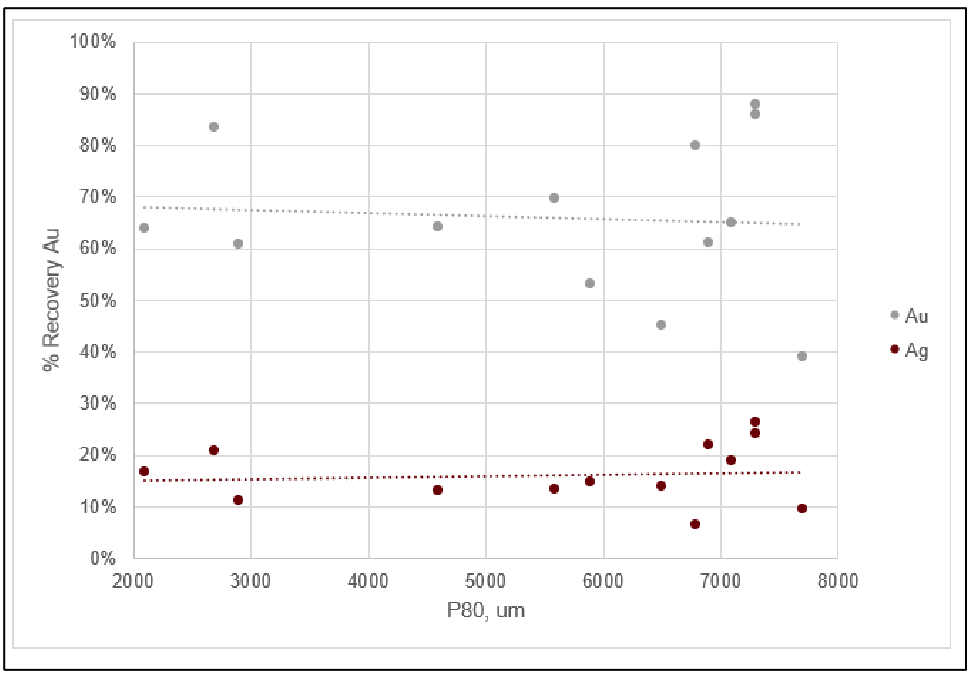


Figure 13.1: Column Leach Tests P_{80} Crush Size vs. Recovery

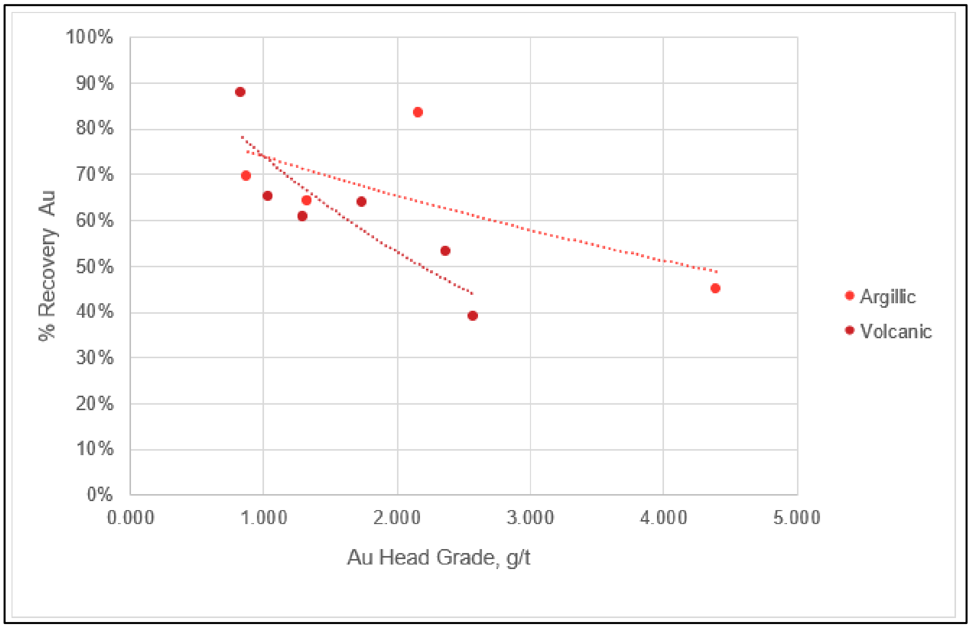


Figure 13.2: Column Leach Tests Head Grade vs. Recovery, Au

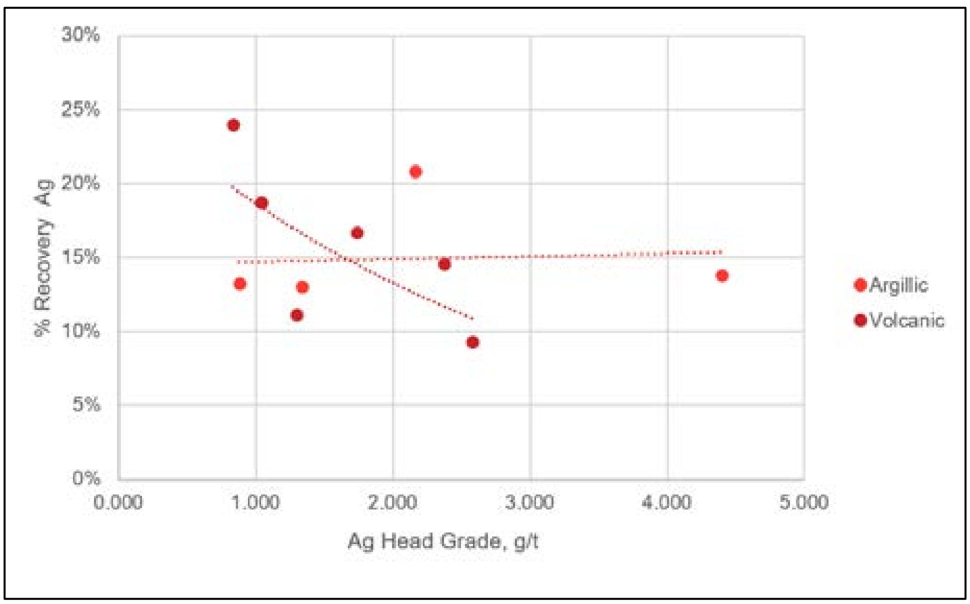


Figure 13.3: Column Leach Tests Head Grade vs. Recovery, Ag

Trend lines were plotted against the data which were used calculate Au and Ag recoveries for different head grade ranges. Estimated recoveries by head grade for Au are presented in Table 13.16 and Ag in Table 13.17. Because the tests did not show a large variation in recoveries based on crush size, a target crush size of 12.5 mm is recommended and additional test work at coarser crush sizes should be performed as part of future test work. The recoveries presented do not include a lab to field deduction, which has been excluded because all of the columns appeared to still be leaching when they were stopped.

Table 13.16: KCA Estimated Heap Leach Au Recoveries

Au Grade (g/t)	Argillic (% Recovery Au)	Volcanic (% Recovery Au)
<0.5	76.0%	76.0%
0.5 to 1.0	75.0%	75.0%
1.0 to 1.5	71.8%	68.4%
1.5 to 2.0	67.6%	57.9%
2.0 to 2.5	63.6%	49.1%
2.5 to 3.0	59.8%	41.6%
3.0 to 3.5	56.3%	35.2%
3.5 to 4.0	53.0%	29.8%
4.0 to 4.5	49.8%	25.3%
4.5 to 5.0	46.9%	21.4%
5.0 to 5.5	44.1%	18.1%
>6.0	35.0%	15.0%

Table 13.17: KCA Estimated Heap Leach Ag Recoveries

Ag Grade (g/t)	Argillic (% Recovery Ag)	Volcanic (% Recovery Ag)
<1	15.0%	19.0%
1.0 to 2.0	15.0%	16.4%
2.0 to 3.0	15.0%	11.5%
3.0 to 4.0	15.0%	8.4%
4.0 to 5.0	15.0%	6.1%
5.0 to 6.0	15.0%	4.3%
6.0 to 7.0	15.0%	2.8%

Cyanide consumption for the column leach tests averaged 0.78 kg/t with an estimated field consumption of 0.26 kg/t with the field consumption being estimated at 33% of the lab cyanide consumption.

Cement will be used for pH buffering and will also help ensure the heap will remain permeable during operations. A cement addition of 4.0 kg/t has been selected based on results from the pulp agglomeration test work; however, additional compacted permeability tests are recommended to confirm the cement requirements.

13.2.2 Heap Leach Cycle

The Tonopah Gold leach cycle has been estimated based on the column test work completed by evaluating the leach curves for Au. The leach cycle considers tonnes of solution per tonne of material as well as the total time required to reach the ultimate recovery in the column leach tests. The selected leach cycle for the Tonopah material is 120 days. It was noted that all of the columns appeared to still be leaching at a very slow rate when they were stopped. Additional longer duration column leach tests should be completed to determine what the final heap recoveries might be.

13.2.3 Mill Recoveries and Reagents

Recoveries for the milled material have been estimated based on the high-grade bottle roll leach test results with recoveries versus P_{80} grind size presented in Figure 13.4 for Au and Figure 13.5 for Ag. The results show good recoveries with fine grinding with very little recovery improvement between 75 and 106 µm. Based on an average of the test results completed at 75 and 106 µm, the mill has estimated recoveries of 95% and 90% for Au for argillic and volcanic material types, respectively and Ag recoveries of 36% and 38%. No discount for lab to field recoveries were applied.

Limited gravity test results provided good results with 30-40% of Au being recovered to a gravity concentrate. Additional gravity testing is required but these tests suggest that improved recoveries and shorter leach times may be possible with a combination of gravity concentration and leaching.

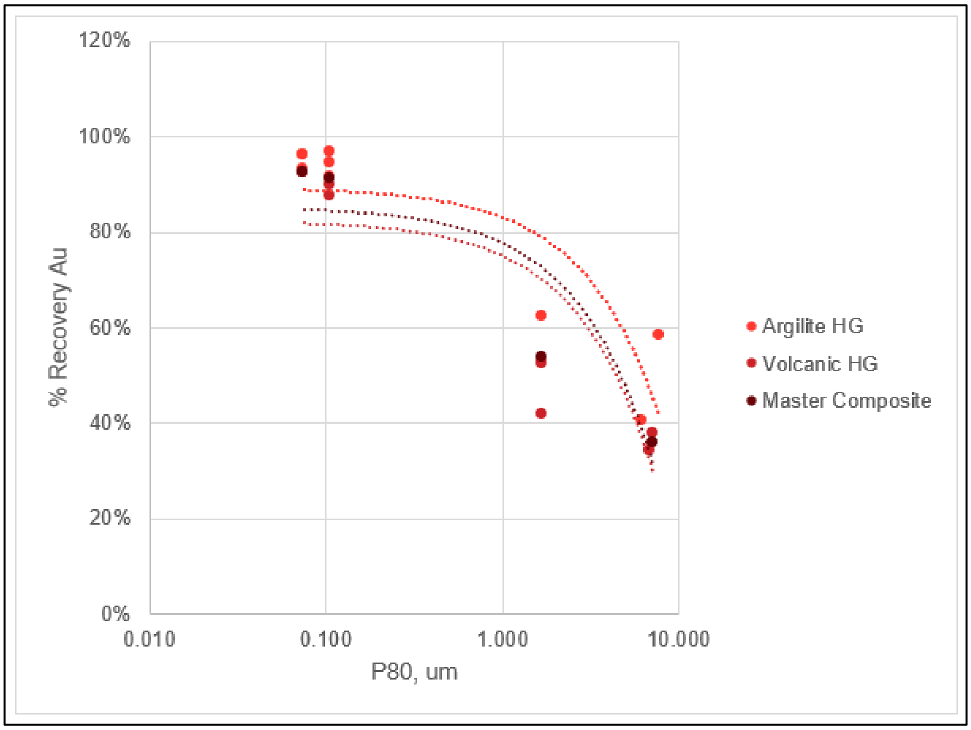


Figure 13.4: Bottle Roll Leach Test P_{80} Grind Size vs. Recovery, Au

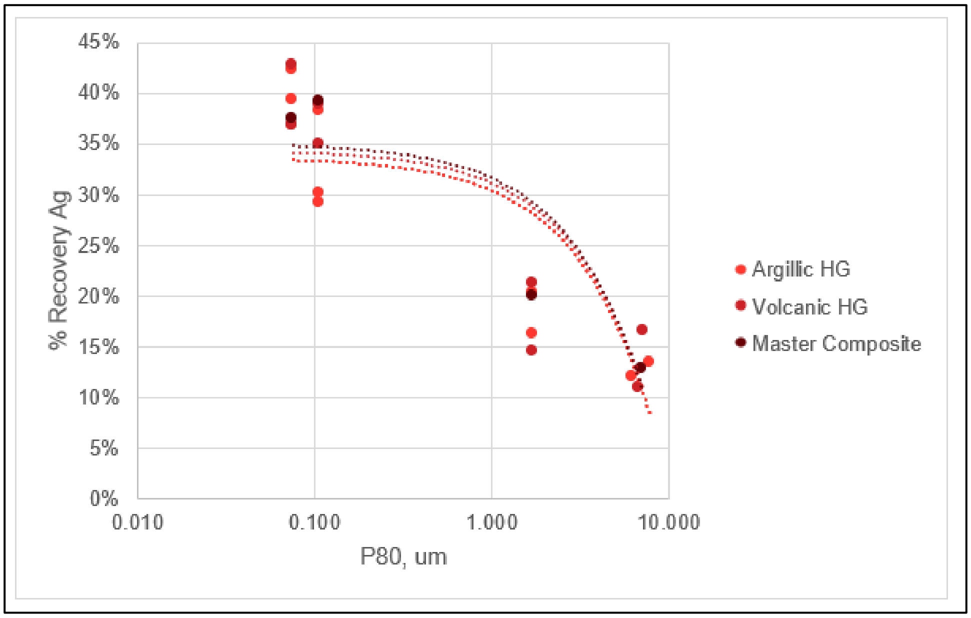


Figure 13.5: Bottle Roll Leach Test P_{80} Grind Size vs. Recovery, Ag

Cyanide consumption and lime addition was calculated based on the average of the bottle test results at 0.58 kg/t and 0.60 kg/t, respectively.

13.3 Recommendations for Future Test Work

The following additional test work is recommended as part of future test programs for the Tonopah Gold Project:

- Variability column leach tests evaluating coarser crushed product sizes and head grades.
- Long term column leach tests (>90 days)
- Additional compacted permeability tests at different cement additions for crushed material only (no pulp agglomeration).
- Full cycle gravity pre-concentration tests following by cyanide leaching at variable P_{80} grind sizes.

14.0 Mineral Resource Estimates

This sub-item contains forward-looking information related to Mineral Resource estimates for the Project. The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts, or projections in the forward-looking information include any significant differences from one or more of the material factors or assumptions that were set forth in this sub-item, including geological and grade interpretations and controls, assumptions and forecasts associated with establishing the prospects for eventual economic extraction.

14.1 Key Assumptions, Parameters, and Methods Used to Estimate the Mineral Resources

14.1.1 Introduction

The Tonopah Mineral Resource estimate included the estimation of Au and Ag. The estimate follows the CIM Estimation of Mineral Resources and Reserves Best Practice Guidelines (MRMR Best Practice Guidelines), issued November 29, 2019.

14.1.2 Available Data

The drill hole database consists of a total of 626 holes consisting of 168 DD holes and 458 RC holes. The database contains exploration holes that are beyond the limits of the current lease boundaries or are a considerable distance away from the main mineralization; therefore, not all available holes were used for the Mineral Resource estimate. Table 14.1 summarizes the available analytical data for Au and Ag and the data used for the resource estimate. The drill hole data was collected between 1988 and 2024, as shown in Table 10.1.

Table 14.1: Summary of Available Data vs. Data Used for Resource Model

Variable	Available Data			Used for Resource Model		
	No. of Drill Holes	No. of Samples	Total Assayed Length (m)	No. of Drill Holes	No. of Samples	Total Assayed Length (m)
Au	596	53,564	79,587	552	49,727	73,689
Ag	388	31,543	47,492	348	28,109	42,208

The data was reviewed for interval errors and out of range assays values prior to import into Maptek Vulcan™ (Vulcan) software. Material issues were identified and corrected prior to modelling as summarized in Item 12.0.

The assay data was modified to account for below detection values and unsampled intervals. All assays reported as below detection by the reporting analytical laboratory were set to half the detection limit of the assay method as reported on the certificate of analysis (i.e., <0.05 PPM was changed to 0.025 PPM). All unsampled intervals were reviewed individually and changed to either 0.0001 PPM or a null value (-99). Where an unsampled interval was within the QAL or TVU lithologies, or within a sequence of assays reported as below detection, the interval was assumed to be waste rock, and the grade value was set to 0.0001 PPM. All other unsampled intervals were set to a null value.

Four sample intervals were excluded from the modelling process due to overlap with a newer hole that had been re-drilled at the same collar and azimuth to a deeper depth.

The geological model and Mineral Resource estimate were constrained by the August 2022 topography surface discussed in Item 9.0.

14.1.3 Geological Modelling

The geological model for the Project was developed as a lithologically constrained grade block model using a combination of Leapfrog Geo™ (Leapfrog) and Vulcan modelling software. All modelling software selected are industry recognized computer-assisted geological, grade modelling, and estimation software applications.

14.1.3.1 Lithology Model

The lithology model for the Tonopah project was previously developed in Leapfrog and was updated by the QP in Leapfrog Geo version 2024.1.2. The lithology model forms the basis of the estimation domain model.

The lithology model extents were confined within the limits of the 2022 aerial flyover boundary and exclude 68 drill holes that either did not have lithology information, were beyond the limits of the current lease boundaries, or were a considerable distance away from the main mineralization. Figure 14.1 shows the lithology model boundary relative to the aerial flyover boundary and the collars of all available drill holes.

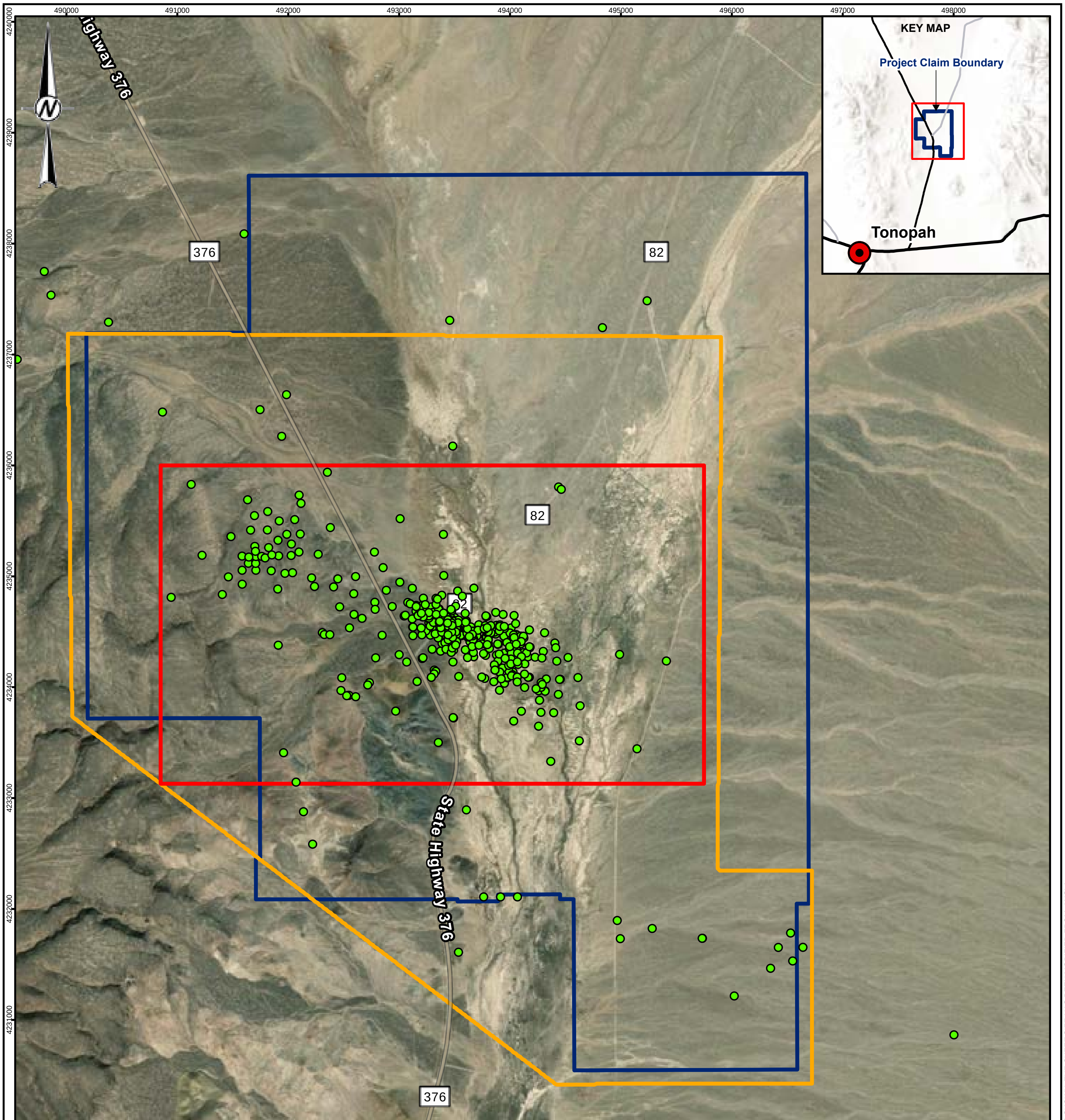
As described in Item 12.0 of this Technical Report, the QP performed data validation on the drill hole database records using available underlying data and documentation, including, but not limited to, original drill hole descriptive logs and core photos.

The Tonopah lithology logging codes have not been consistent throughout Project history. There are 77 different lithology codes recorded in the lithology table. For the purposes of modelling, these codes have been grouped into four major units (Table 14.2). Of the four major units, only the TVL and Op units are significantly mineralized, with the majority of mineralization occurring proximal to a nonconformity separating the two units. Undifferentiated units are excluded from the lithology model.

Table 14.2: Tonopah Lithology Model Major Units

Major Unit	Description
QAL	Quaternary Alluvium
TVU	Tertiary Volcanics - Upper
TVL	Tertiary Volcanics - Lower
Op	Ordovician Palmetto Formation
Undiff	Undifferentiated (faults, intrusions, etc.)

The lithology codes had been grouped into the major units in previous updates to the model, however previous iterations of the lithology model did not utilize all available historic drill holes. During a previous scope of work, WSP added all available drill holes within the model extents to the lithology model, which included lithology codes that had previously not been assigned to a major unit. WSP reviewed both historic and recent drill holes in cross-section within Leapfrog to determine appropriate grouping into a major unit. Table 14.3 shows the final unit assignments for each lithology code.



LEGEND

- LITHOLOGY MODEL BOUNDARY
- 2022 AERIAL FLYOVER BOUNDARY
- PROJECT CLAIM BOUNDARY
- DRILL HOLE COLLAR

0 1 2

1:50,000 KILOMETRES

NOTE(S)

REFERENCE(S)

2022 AERIAL DATA PROVIDED BY VIVA GOLD CORP.
DATA PROVIDED BY VIVA GOLD CORP.
PROJECTED COORDINATE SYSTEM: NAD 1983 UTM ZONE 11N

CLIENT
VIVA GOLD CORP.

PROJECT
TONOPAH GOLD PROJECT

TITLE
LITHOLOGY MODEL BOUNDARY

CONSULTANT

YYYY-MM-DD	2025-08-06
DESIGNED	RK
PREPARED	RK
REVIEWED	JS
APPROVED	BT

PROJECT NO.	CONTROL	REV.	FIGURE
CA0004670.0343		0	14.1

IF THIS MEASUREMENT DOES NOT MATCH WHAT IS SHOWN, THE SHEET SIZE HAS BEEN MODIFIED FROM: ANSI A

25mm

Table 14.3: Lithology Logging Codes

Logged Lithology	Unit	Description
G	QAL	Gravel
OB	QAL	Overburden
QA	QAL	Quaternary Alluvium
QAL	QAL	Quaternary Alluvium
QOA	QAL	Quaternary Alluvium (Older alluvial deposits)
RHY	TVU	Rhyolite Flow
SLST	TVU	Volcaniclastic Sediments (Mostly Finer Grained)
TAB	TVU	Tertiary Red Mountain Andesite - Basalt Flows
TAF	TVU	Tertiary Alluvium
TAG	TVU	Tertiary hornblende-biotite andesite
TAL	TVU	Tertiary Alluvial Horizon (Older)
TB	TVU	Tertiary Scoriaceous Basalt Flows
TBA	TVU	Tertiary Basalt
TFB	TVU	Tertiary Flow-Banded Rhyolite
TFT	TVU	Tertiary Volcanic ash-flow tuff
TG	TVU	Tertiary Alluvial Fan Deposit (gravel)
TGR	TVU	Tertiary Crystal-Rich, Welded Rhyolite Tuff; Distinctive Green Color
TPB	TVU	Tertiary Porphyritic Basalt Flows
TRH	TVU	Tertiary Volcanic Undivided
TRT	TVU	Tertiary Rhyolite Tuff (Undivided)
TVS	TVU	Tertiary Volcaniclastic Siltstone; Distinctive Thin-Bedded Character
TVU	TVU	Tertiary Volcanic Upper
TAR	TVL	Tuffaceous Arkose
TLAP	TVL	Lapilli Tuff
TLT	TVL	Tertiary Monolithic Lapilli Tuff
TMONO	TVL	Monolithic Tuff
TOP	TVL	Tertiary Heterolithic Lapilli Tuff Characterized by black Palmetto fragments
TOP1	TVL	Tertiary Heterolithic Crystal Tuff (Younger); Contains Distinctive Palmetto Formation Clasts
TOP2	TVL	Tertiary Heterolithic Ash to Lapilli Tuff
TPL	TVL	Tertiary Polylithic Tuff
TPL1	TVL	Tertiary Polylithic Lapilli Tuff; Abundant Rhyolite Tuff Clasts
TPL2	TVL	Tertiary Polylithic Andesitic Tuff; Clast to Matrix-Supported
TPL3	TVL	Tertiary Polylithic Tuff
TPL4	TVL	Tertiary Polylithic Tuff
TPL5	TVL	Tertiary Polylithic Tuff
TPOLY	TVL	Polylithic Tuff
TRFB	TVL	Tertiary Volcanic Undivided
TROP	TVL	Tertiary Volcanic Undivided
TRP	TVL	Tertiary Volcanic Undivided
TRU	TVL	Tertiary Rye Patch Rhyolite Tuff (Lithic-Rich)
TRV	TVL	Tertiary Volcaniclastic Arkose/Greywacke / Tertiary Siltstone (Tuff of Ralston Valley)
TRV/TSL	TVL	Tertiary Volcaniclastic Arkose/Greywacke / Tertiary Siltstone
TRV/TVG	TVL	Tertiary Volcaniclastic Arkose/Greywacke / Tertiary Volcaniclastic Siltstone; Distinctive Thin-Bedded Character
TRVG	TVL	Tertiary Crystal-Rich, Welded Rhyolite Tuff; Distinctive Green Color
TRVT	TVL	Tertiary Volcaniclastic Arkose/Greywacke
TVG	TVL	Tertiary Volcaniclastic Siltstone, Mudstone and Arkosic Greywacke; Distinctive Thin-Bedded Character
TWR	TVL	Tertiary Welded Rhyolite Tuff
TXL	TVL	Crystal Tuff
ARG	Op	Argillite
FNC	Op	Unknown (occurs in historic Midway Gold holes only)
OP	Op	Ordovician Palmetto Formation: chert and argillite
OPA	Op	Ordovician Palmetto Formation: mostly argillite

Logged Lithology	Unit	Description
OPA/TRV MELANGE	Op	Ordovician Palmetto Formation: Mixed with Tertiary Volcaniclastic unit
OPL	Op	Ordovician Palmetto Formation: mostly limestone
OPQ	Op	Ordovician Palmetto Formation
TI	Op	Tertiary Intrusive
TKI	Op	Tertiary Intrusive
BX	Undiff	Breccia
BXF	Undiff	Bx Fault
BXV	Undiff	Pyroclastic
CL	Undiff	Clay Seam
CY	Undiff	Clay
FAULT	Undiff	Fault
FG	Undiff	Fault Gouge
IN	Undiff	Intrusive
INT	Undiff	Intrusive
NC	Undiff	No Core
NL	Undiff	Not Logged
NR	Undiff	No Recovery
NS	Undiff	No Recovery
QL	Undiff	Quartz Latite
QV	Undiff	Quartz Veins
SEDS	Undiff	Sediments
TKP	Undiff	Tertiary Rhyolite Quartz Monzonite Intrusive; Mostly Dikes in Palmetto Fm
TLD	Undiff	Tertiary Quartz Latite Intrusive; Mostly Thin Dikes Within Palmetto Fm and Lower Tertiary Volcanics
TMD	Undiff	Tertiary Ultramafic to Mafic Intrusive; Emplaced as Sill/Dike Along Rye Patch Fault
TRI	Undiff	Tertiary Rhyolite dikes and intrusive breccias
TRPU	Undiff	Tuffs of Rye Patch: upper ash-flow tuff

Where historical lithology codes did not align with the lithology of more recent drilling, the intervals were ignored within Leapfrog for the purposes of modelling.

14.1.3.2 CSAMT Model

Previous studies on the geophysical data collected for the Project (Item 9.0) demonstrated the CSAMT data could be used to identify structural features and recommended their incorporation into the geological interpretation (Wright, 2019). Within the main mineralized area of the deposit, the CSAMT data was collected along east-west section lines ranging from 90 m to 700 m apart, with most sections approximately 180 m apart. The QP imported the CSAMT data into Leapfrog and modelled the inverted resistivity as a numeric model using a radial basis function (RBF) interpolant, generating a 3D interpretation (Figure 14.2). The CSAMT dataset also included sectional two-dimensional (2D) inversions on coloured sections with contours, an example for Line 12700 is shown in Figure 14.3. Both the 3D and 2D interpretations were used as part of the structural interpretation for the Project.

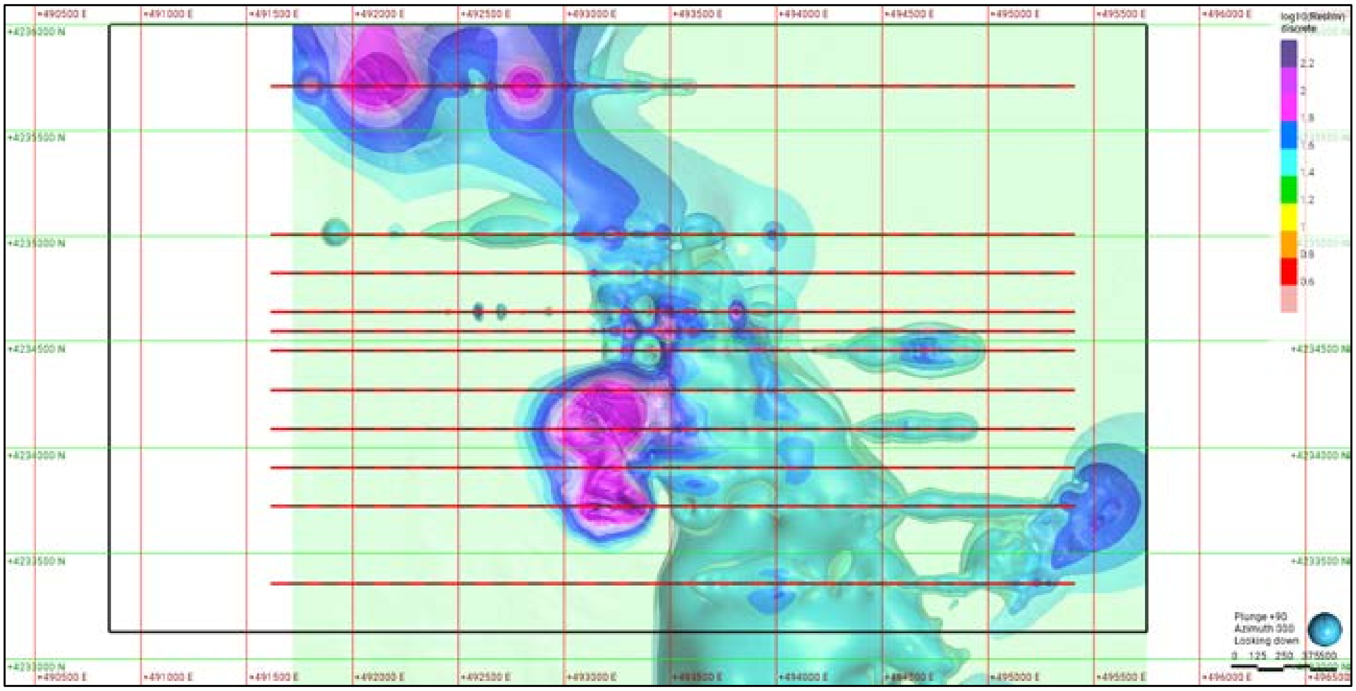


Figure 14.2: Plan View of 3D CSAMT Numeric Model within Lithology Model Boundary (Black Outline), Survey Section Lines shown in Red

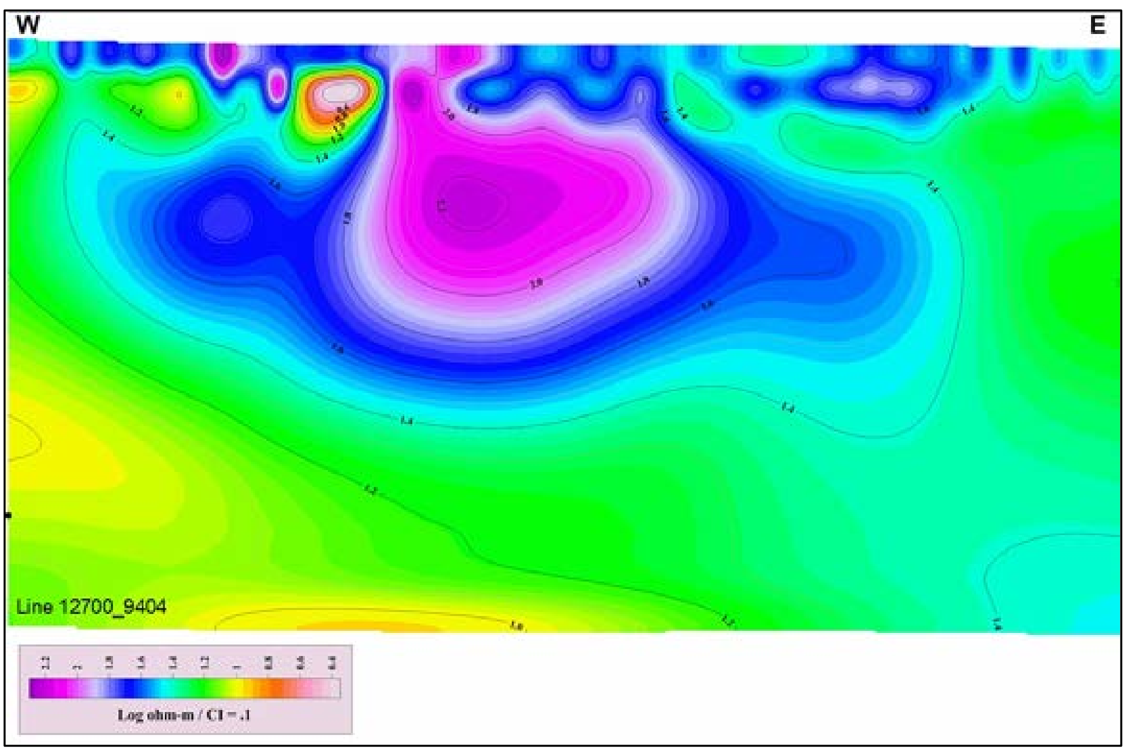


Figure 14.3: Line 12700 Inverted Resistivity Section

14.1.3.3 Structural Controls

As discussed in Item 7.3.7, the Tonopah Project is located within the Rye Patch fault system, a complex, oblique-slip fault system (Rhys, 2003). The faults constrain mineralization and offset both mineralization and lithology. Fourteen (14) faults were modelled as part of the structural interpretation, using a combination of lithology logging data, geotechnical logging data, historic mapping data, and CSAMT data:

- **Nautilus Fault**: forms the northern boundary of the relay system. The interpreted location is speculative, primarily based on the regional geology map from Rhys, 2003, with some input from the CSAMT model.
- **Dauntless Fault**: a northwest-southeast striking, northeast dipping normal fault in the northern part of the system, within the main mineralized area. It appears to terminate Au mineralization to the north of it and form part of the contact between the TVL and Op. It is based on drill hole data and the CSAMT model. The interpreted location has moderate confidence. It potentially continues to the southeast, forming the contact between the TVL and Op units, as what has been termed the Rye Patch fault in previous studies. Additional interpretation is recommended.
- **DDN (Discovery-Dauntless North) Fault**: a northwest-southeast striking, steeply southwest dipping normal fault, directly south and parallel to the Dauntless fault. It offsets both Au mineralization and lithology and likely has a northwest dipping counterpart directly south of it that runs parallel to it (not modelled). It is based on drill hole data, the CSAMT model, and was interpreted by Rhys (Rhys, 2003). The interpreted location has moderate to high confidence.
- **Discovery Fault**: a north-south striking, west dipping scissor fault that offsets the Au mineralization and lithology by approximately 60 m. This displacement is readily apparent when viewing the Au samples in cross-section (see Figure 14.4). The Discovery Fault had been previously modelled by Midway Gold circa 2006 but was not being used in the lithology model. It is based on over 30 drill hole intercepts, with some input from the CSAMT model, and was interpreted by Rhys (Rhys, 2003). The interpreted location has high confidence.
- **Discovery East Fault**: a north-south striking, east dipping fault, directly east and parallel to the Discovery fault. It does not appear to offset Au mineralization or lithology, and its interpretation is based entirely on the CSAMT model. Based on its proximity to the high-grade mineralization of the Discovery and Dauntless zones, it’s possible it is a mineralizing fault. The interpreted location has low confidence.
- **Point Luck**: a northeast-southwest striking, steeply southeast dipping normal fault in the northern part of the system, within the main mineralized area. It appears to terminate Au mineralization to the west of it and offsets lithology. It is based on drill hole data and the CSAMT model. The interpreted location has moderate confidence north of its junction with the Enterprise East fault but is low south of it.
- **Enterprise East Fault**: a northwest-southeast striking, steeply southwest dipping normal fault in the eastern part of the system, within the main mineralized area. It appears to terminate Au mineralization to the south of it where it terminates against the Point Luck fault and accounts for lithology offset seen in the drill hole logging. It is based on drill hole data and the CSAMT model. The interpreted location has moderate confidence.
- **Enterprise West Fault**: a northwest-southeast striking vertical fault, directly southwest and roughly parallel to the Enterprise East fault. It appears to terminate Au mineralization to the west of it, though drilling in this area is limited. It is based on the CSAMT model with some drill hole data input. The interpreted location has low to moderate confidence.

- **121 Fault**: a north-south striking, steeply east dipping fault located west and parallel to the Discovery fault, in the western part of the system. It appears to minorly offset the Au mineralization but does not appear to offset lithology. It is based on drill hole data and the CSAMT model. The interpreted location has moderate confidence.
- **West Bounding Fault**: a north-south striking, steeply east dipping fault located west and parallel to the 121 fault. It appears to terminate Au mineralization to the west of it, though drilling in this area is limited. It is based on drill hole data and the CSAMT model. The interpreted location has low to moderate confidence.
- **Midway Hills Fault**: a northwest-southeast striking, northeast dipping normal fault that spans the western part of the system, terminating against the Discovery fault in the east. It appears to align with the start of the TVL overlaying the OP in this part of the deposit but does not form the contact between the two units. The interpreted location is speculative, primarily based on the regional geology map from Rhys, 2003, with some input from the CSAMT model.
- **SouthSS (Strike-Slip) Fault**: an east-west striking strike-slip fault in the southern part of the system, outside the main mineralized area. The interpreted location is speculative and primarily based on the regional geology map from Rhys, 2003. It is interpreted to be the southern limit of the Discovery, Point Luck, Enterprise West, and Enterprise East faults.
- **South17 Fault**: a north-south striking fault in the southern part of the system, outside the main mineralized area. The interpreted location is speculative and primarily based on the CSAMT model and one drill hole intercept.
- **South Bounding Fault**: forms the southern boundary of the relay system. The interpreted location is speculative and primarily based on the CSAMT model.

Figure 14.5 shows a plan view section of the modelled faults at 1,700 m amsl elevation with the lithology model, and Figure 14.6 shows the same plan view section of the modelled faults with the CSAMT numeric model. A representative long-section of the updated lithology model is shown in Figure 14.7. The structural interpretation does not represent all the faults in the system and additional interpretation may be beneficial to the overall geologic interpretation.

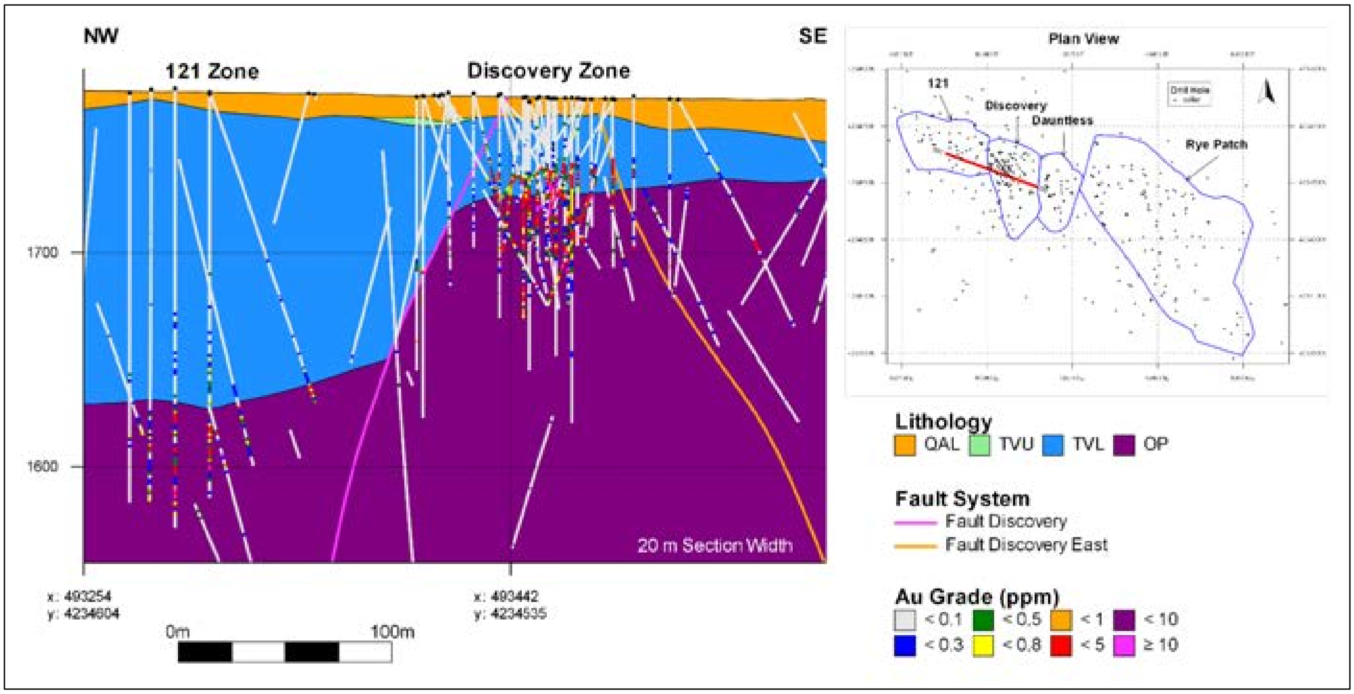


Figure 14.4: Cross-Section of Au Mineralization Offset Due to Discovery Fault

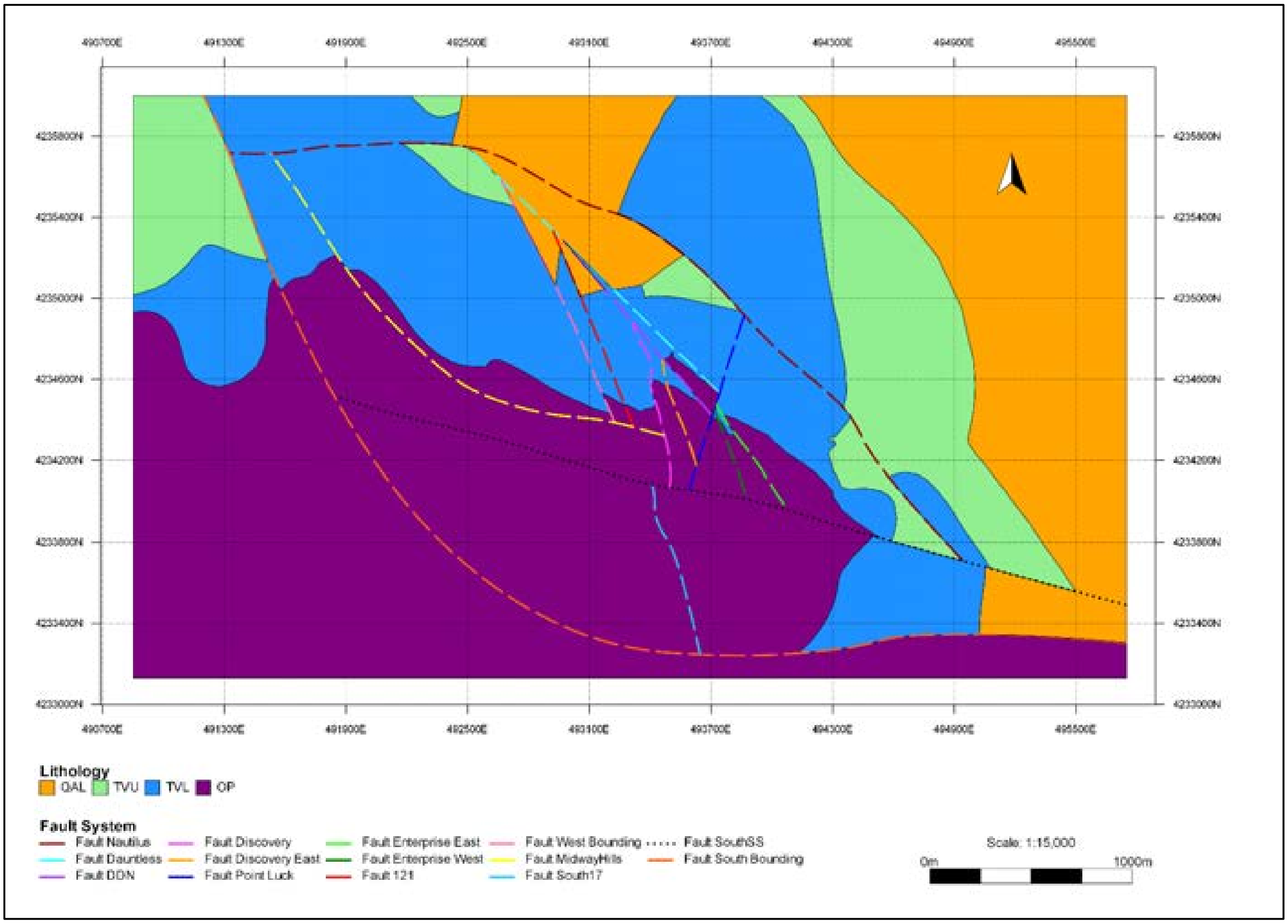


Figure 14.5: Plan Section of Modelled Faults with Lithology at 1,700 m Elevation

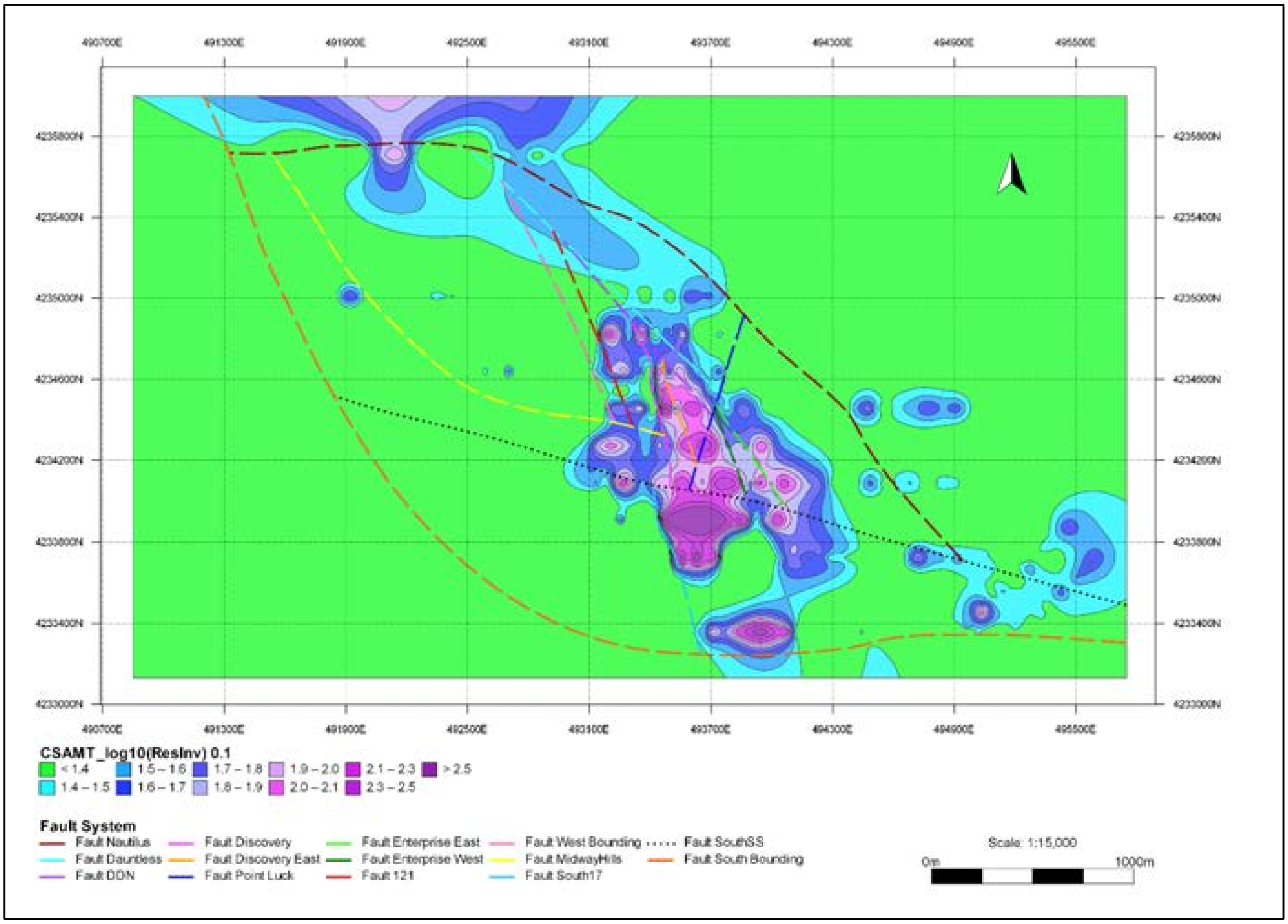


Figure 14.6: Plan Section of Modelled Faults with CSAMT Model at 1,700 m Elevation

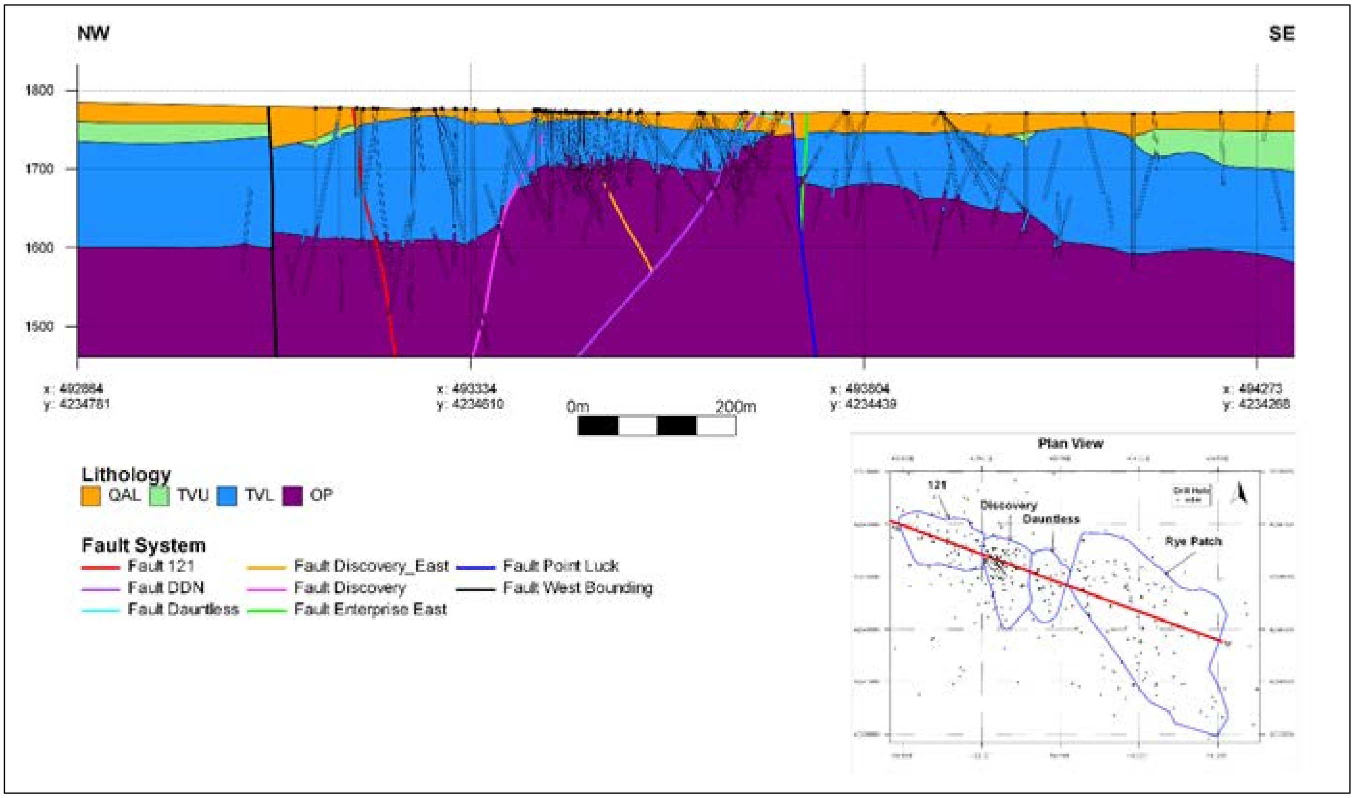


Figure 14.7: Tonopah Lithology Model Representative Long-Section

14.1.3.4 Estimation Domain Model

The estimation domain model consists of 10 zones separated by lithology contacts, faults, a change in the characteristics of the Au mineralization and geology, and drilling density. The QAL and TVU lithology units are not considered mineralized and any mineralized samples that occur within them are considered anomalous, therefore these units have been excluded from the estimation domains. Due to the significant differences in sample density across the deposit, a limiting boundary was defined around the area of high drill density (Figure 14.8). The purpose of the limiting boundary was to restrict areas of very low drilling density that would not meet the definition of a mineral resource. The estimation domains are described as follows:

1) **Domain 1 - 121**: Characterized by Au mineralization concentrated proximal to TVL-Op nonconformity, located west of the Discovery Fault with downward displacement and bounded by the West Boundary fault on the west side and the Midway Hills fault on the south side. The TVL and Op units have been combined for this domain. Located within the higher drilling density limiting boundary.

2) **Domain 2 - Discovery/Dauntless**: Characterized by Au mineralization concentrated proximal to the TVL-Op nonconformity, bounded by the Discovery Fault on the west side and by the Point Luck fault on the east side. Contains the highest-grade mineralization of the deposit. The TVL and Op units have been combined for this domain. Located within the higher drilling density limiting boundary.

3) **Domain 3 - Enterprise West**: Characterized by Au mineralization occurring only in the Op unit, between the Point Luck and Enterprise West faults. This domain has a limited number of drill holes and samples within it.

The TVL and Op units have been combined for this domain. Located within the higher drilling density limiting boundary.

4) **Domain 4 - Enterprise East**: Characterized by Au mineralization occurring only in the Op unit, between Enterprise East and Enterprise West faults. The TVL and Op units have been combined for this domain. Located within the higher drilling density limiting boundary.

5) **Domain 5 - Rye Patch TVL**: Characterized by Au mineralization occurring within the TVL unit, bounded by the Point Luck fault on the west side. Au mineralization is concentrated in the TVL unit, above the TVL-Op nonconformity, with less mineralization in the Op unit compared to previous domains. This domain consists only of the TVL unit. Located within the higher drilling density limiting boundary.

6) **Domain 6 - Rye Patch OP**: Characterized by Au mineralization occurring within the Op unit. Au mineralization is significantly less than the TVL unit above. This domain consists only of the Op unit. Located within the higher drilling density limiting boundary.

7) **Domain 7 - Midway Hills**: Characterized by lower drill hole density and Au mineralization concentrated near the TVL-Op nonconformity, located approximately 1,500 m northwest of the mineralization in Domain 1. The TVL and Op units have been combined for this domain. Located outside the drilling density limiting boundary.

8) **Domain 8 - Enterprise Outer**: Characterized by very low drilling density, located south of Domain 3 and 4 and outside the drilling density limiting boundary. The TVL and Op units have been combined for this domain.

9) **Domain 9 - Rye Patch TVL Outer**: Characterized by very low drilling density, located north of Domain 5 and outside the drilling density limiting boundary. This domain consists only of the TVL unit.

10) **Domain 10 - Discovery/Dauntless Outer**: Characterized by very low drilling density, located north of Domain 2 and outside the drilling density limiting boundary. The TVL and Op units have been combined for this domain.

A plan view section of the estimation domains is shown in Figure 14.9.

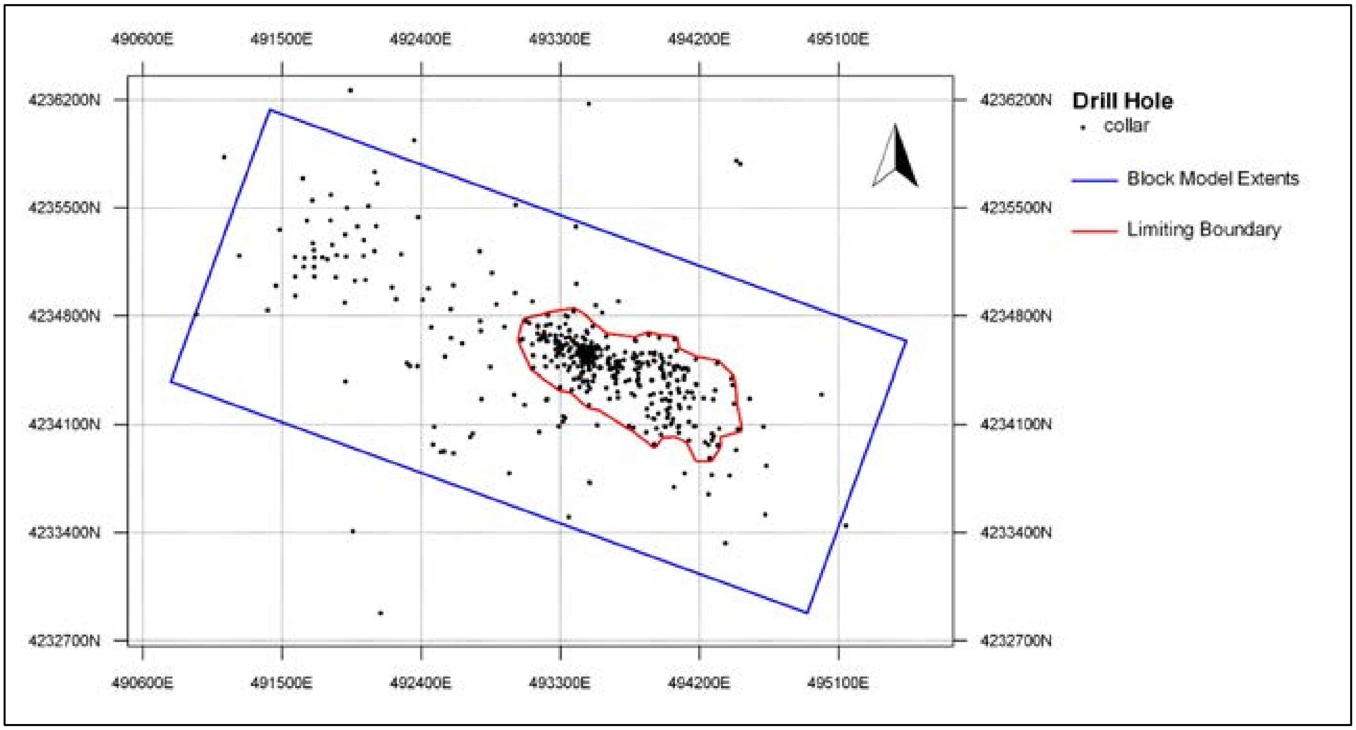


Figure 14.8: Estimation Limiting Boundary Relative to Drill Hole Collars and Block Model Extents

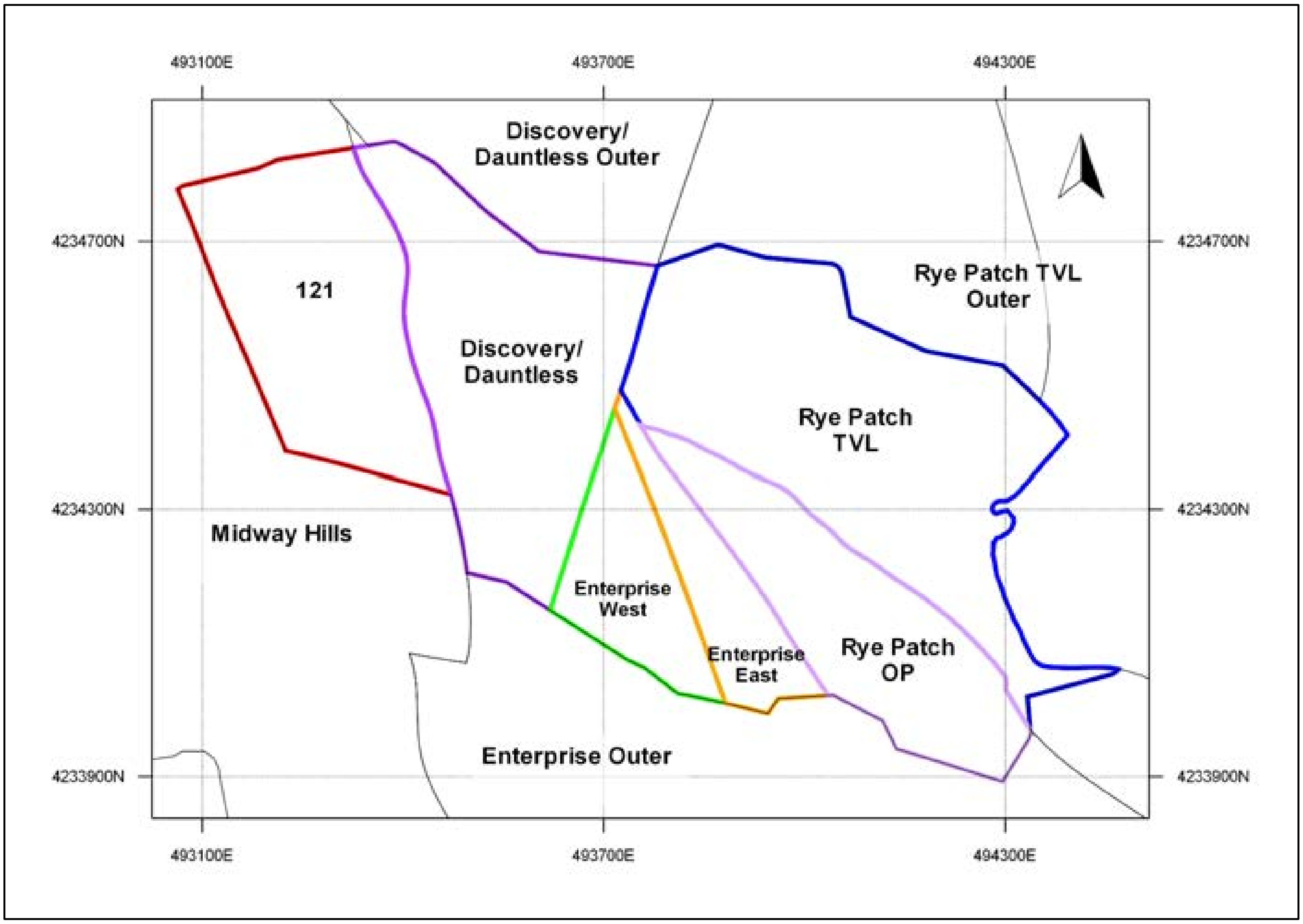


Figure 14.9: Plan Section of Estimation Domains at 1,700 m Elevation

14.1.4 Exploratory Data Analysis

The QP performed Exploratory Data Analysis (EDA) comprised of statistical and geostatistical analysis of the verified data to allow for evaluation of the statistical and spatial variability of the geological data. The EDA aided in the evaluation of the geological domains used in modelling by evaluating statistical and spatial trends in the data for the identified geological domains.

14.1.4.1 Descriptive Statistics

The initial phase of the EDA was completed on the global model dataset to evaluate for trends and outliers in the data prior to establishing mineralization domains. A second pass EDA was performed following the domaining process (see discussion in Item 14.1.4.5 below). The QP used Phinar Software X10-Geo™ (X10) software to develop descriptive univariate statistics, box and whisker graphs, histograms, probability statistics, and scatter plots for all the available assay data within the model limits. Table 14.4 summarizes the length-weighted statistics for each grade variable used for block estimation. The Au and Ag populations were found to have a highly positively skewed distribution with the presence of several very high outlier grade values. Figure 14.10 and Figure

14.11 show the length-weighted log histogram and log probability plot for Au. Figure 14.12 and Figure 14.13 show the length-weighted log histogram and log probability plot for Ag.

Table 14.4: Summary Statistics for Estimated Variables

Grade Variable	Raw Sample Count	Minimum	Maximum	Mean	Variance	StDev	CV	Skewness	Kurtosis	Median
Au (ppm)	49,727	0.00	5,474.39	0.36	354.20	18.82	51.67	262.90	75,149.00	0.03
Ag (ppm)	28,109	0.00	446.00	1.00	12.15	3.49	3.49	45.24	4,346.00	0.40

Notes: StDev = Standard Deviation, CV = Coefficient of Variation

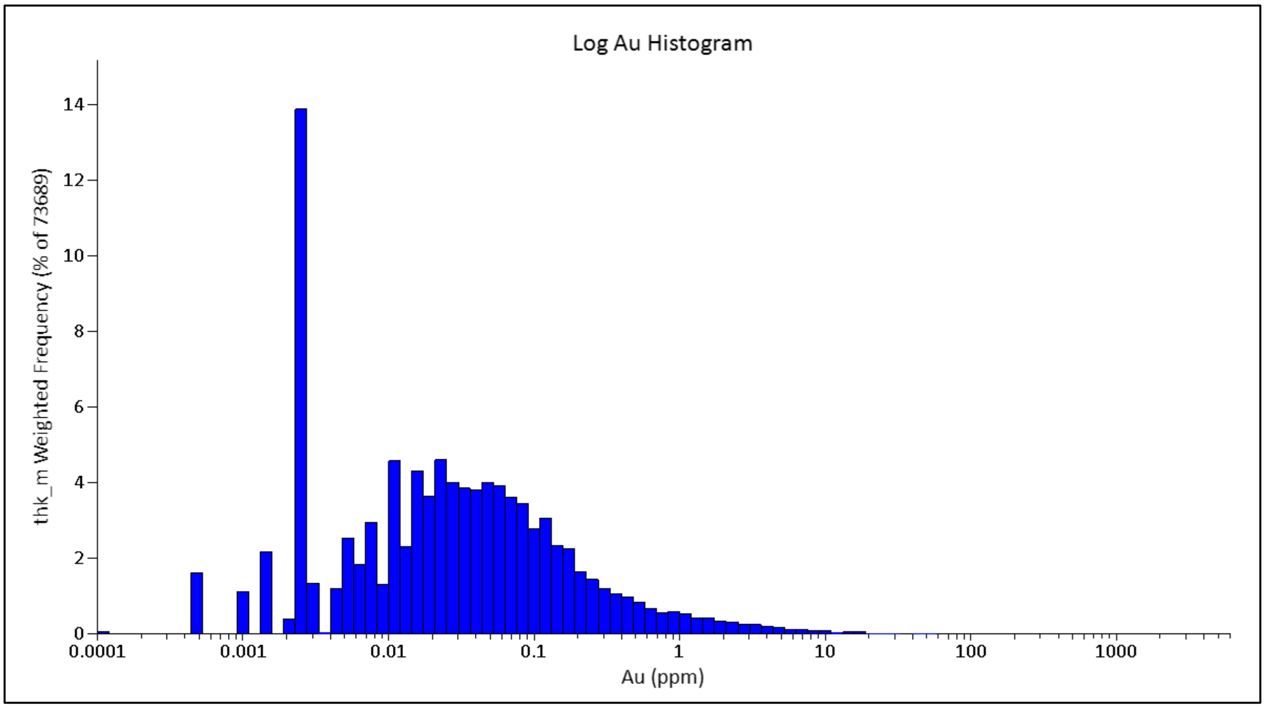


Figure 14.10: Log Histogram of Raw Au Samples

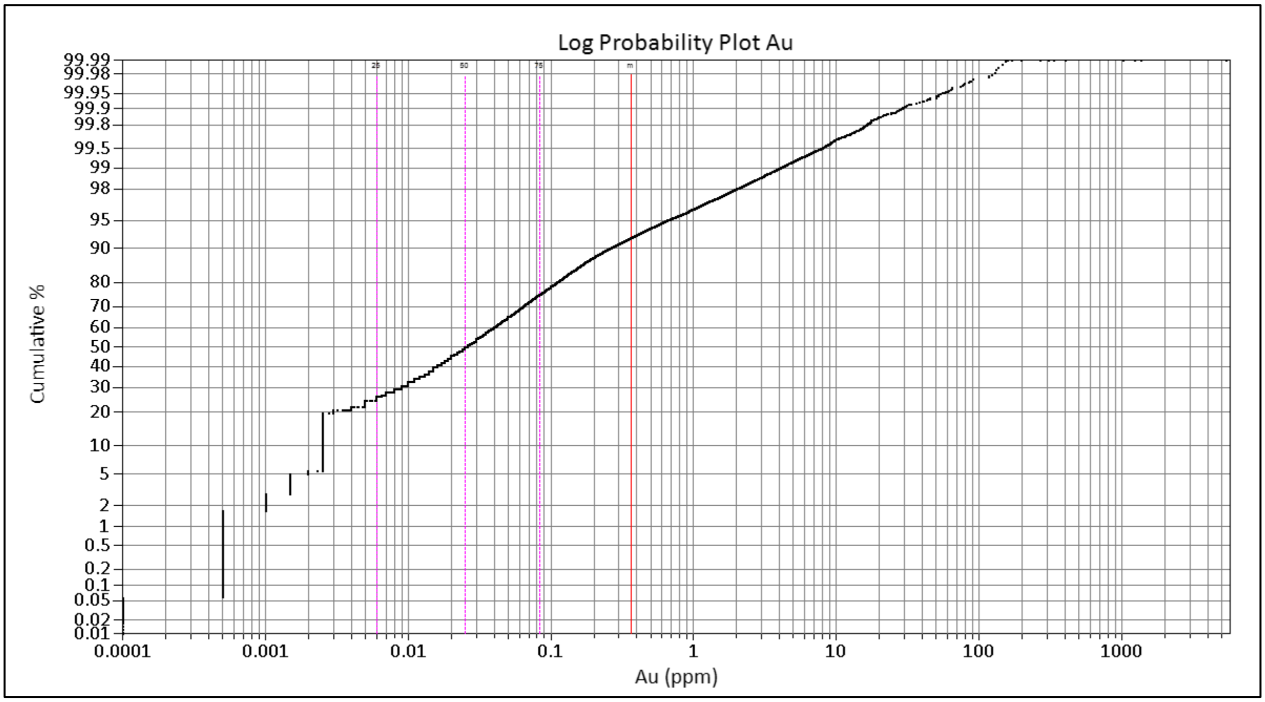


Figure 14.11: Log Probability Plot of Raw Au Samples

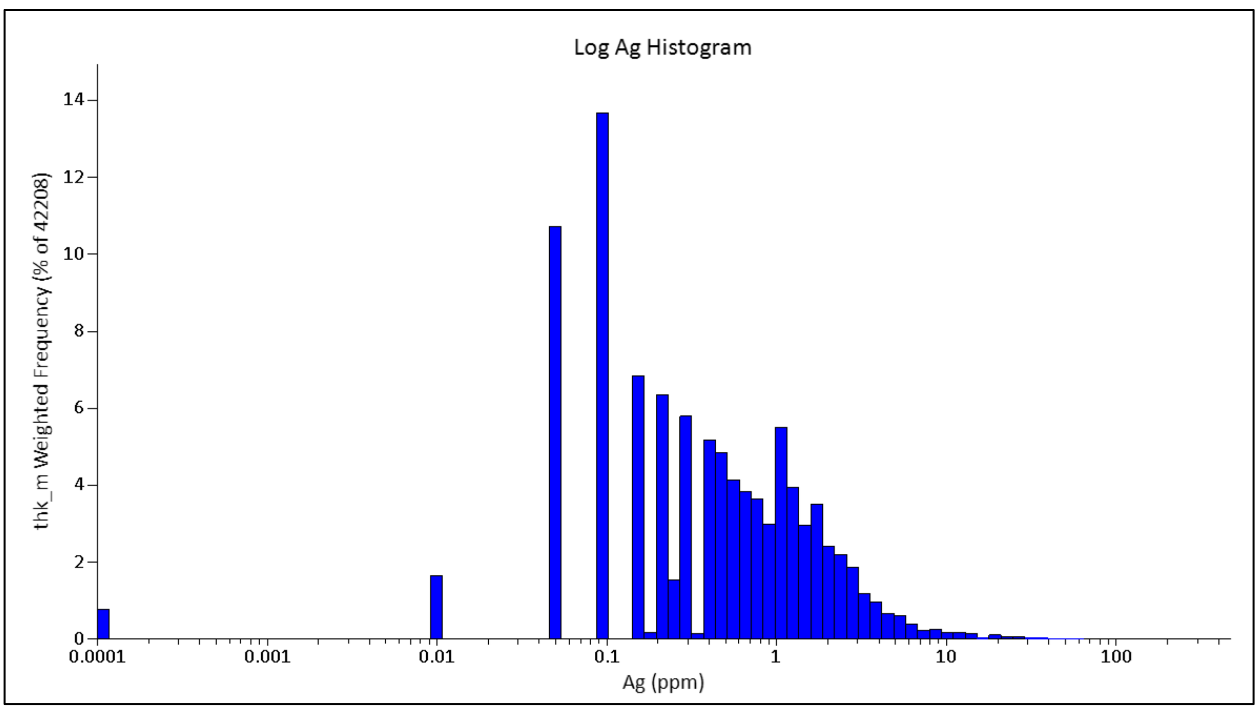


Figure 14.12: Log Histogram of Raw Ag

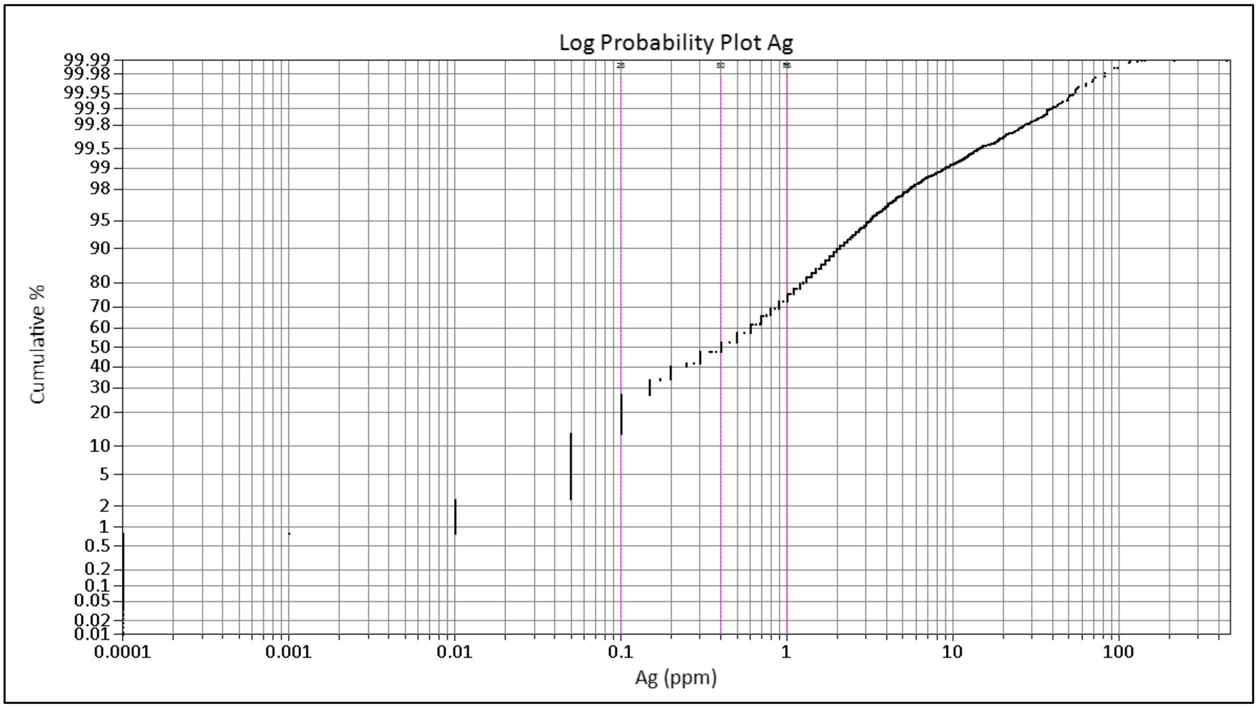


Figure 14.13: Log Probability Plot of Raw Ag

14.1.4.2 Correlation

The statistical correlation between Au and Ag was compared using a scatterplot (Figure 14.14). The Ag grade shows a fairly strong positive correlation to the Au grade with a correlation R value of 0.78. The maximum Ag grade of 446 PPM corresponds to an Au grade of 1,031 PPM. This correlation was confirmed with visual analysis of the Au and Ag samples in cross-section.

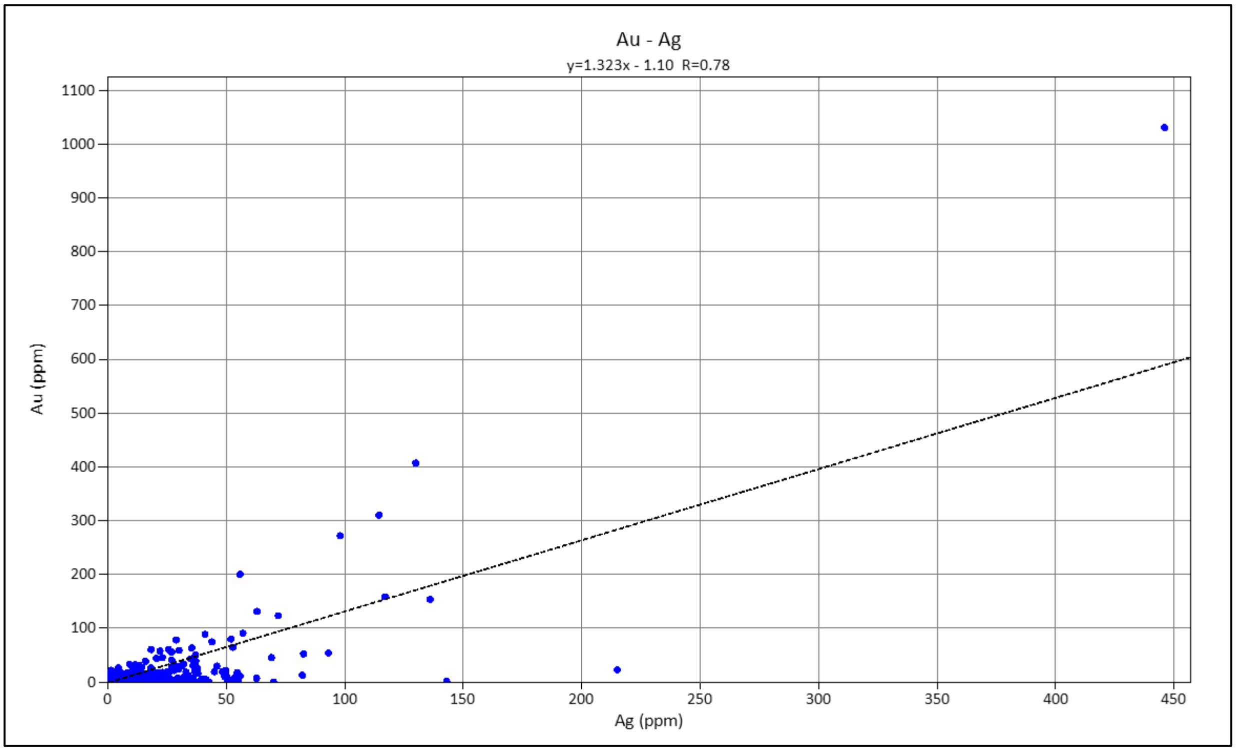


Figure 14.14: Au-Ag Correlation Scatterplot

14.1.4.3 Sample Compositing

Compositing samples is a technique used to give each sample relatively equal length weighting to reduce the potential for bias due to uneven sample lengths. A histogram of the raw sample length was generated to determine the most common (modal) sample length (Figure 14.15).

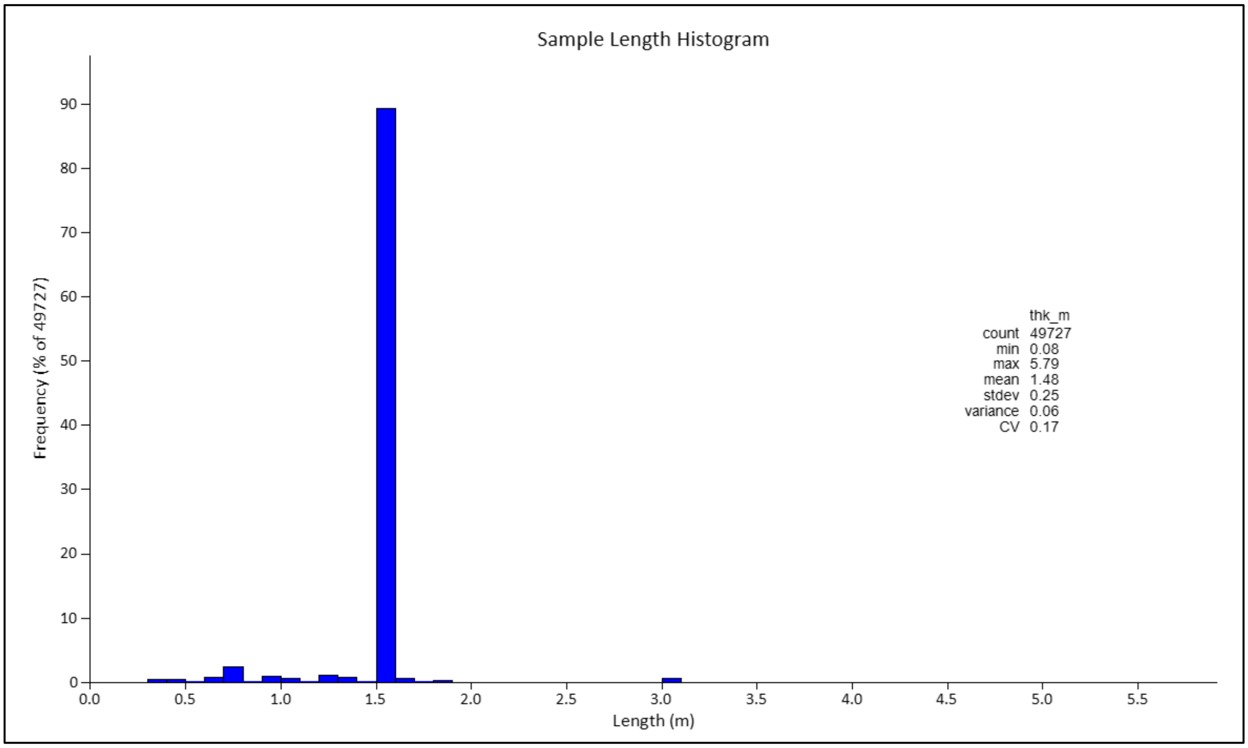


Figure 14.15: Raw Sample Length Histogram

The modal length of 1.5 m combined with the block size in the Z direction of 6 m were both considered when determining the final compositing length of 3 m. Compositing was completed in Vulcan. All composites were constrained by the estimation domain (i.e., composites were not allowed to span boundaries of units) with no overlaps. Small composites that occurred at domain boundaries or at the end of drill holes with a length of less than 0.5 m were added to the previous interval. The raw sample length and composite length statistics, for all domains combined, are summarized in Table 14.5.

Table 14.5: Raw and Composite Sample Length Statistics

Raw Sample Length					Composite Sample Length				
Count	Mean	Mode	Minimum	Maximum	Count	Mean	Mode	Minimum	Maximum
49,727	1.5	1.5	0.1	5.8	27,725	2.9	3.0	0	3.5

14.1.4.4 Outlier Evaluation

High-grade outlier data has the potential to bias local block model grades if they are not handled by grade capping (also known as top cutting) or otherwise restricting their influence through other estimation criteria. The scatterplot of the Au grades vs. the sample length (Figure 14.16) indicates that outliers and extreme values are related to short sample lengths relative to the modal length, therefore the outlier evaluation was performed on the 3 m composited samples. Log probability plots of the composited data were used to determine the capping values for Au (Figure 14.17) and Ag (Figure 14.18). Outliers were also examined in 3D space to determine spatial correlation.

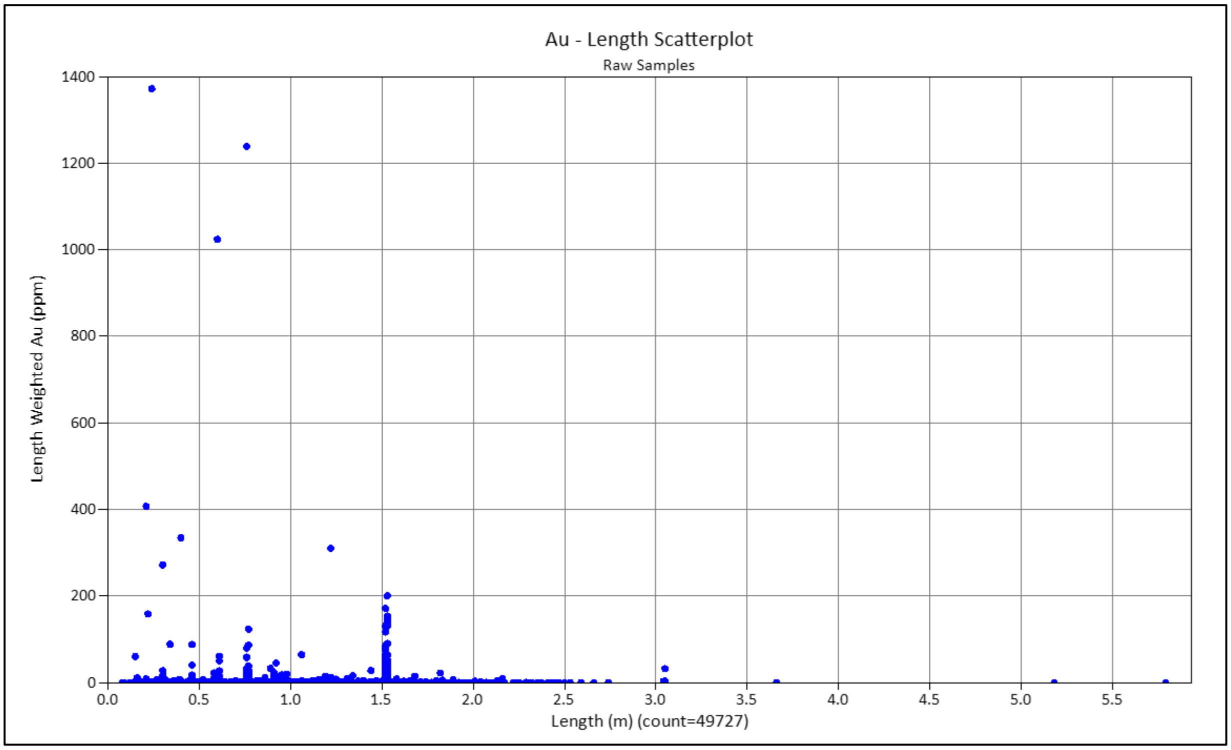


Figure 14.16: Scatterplot of Au Grades vs. Sample Length

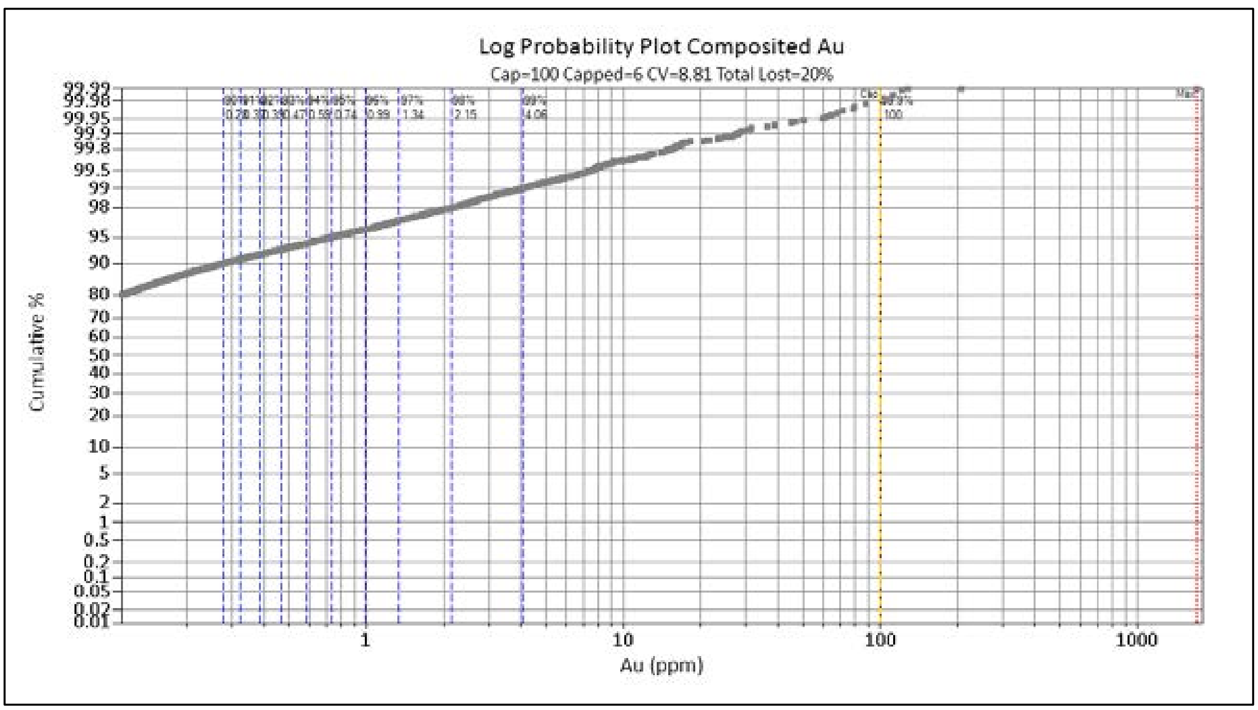


Figure 14.17: Log Probability Plot of Composited Au Samples

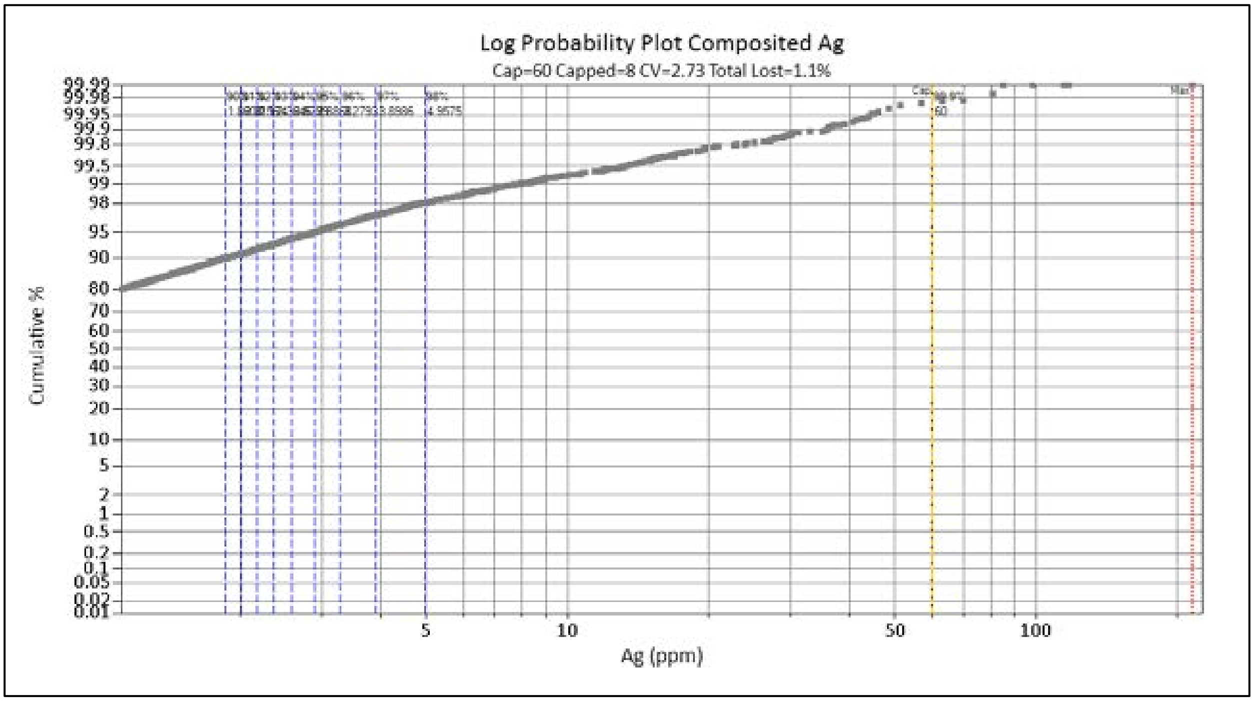


Figure 14.18: Log Probability Plot of Composited Ag Samples

A significant number of high-grade samples occurred spatially clustered together in areas of high drilling density. Examination of the available drill core for these samples indicates the high-grade mineralization occurs primarily in rock with quartz veins, silicification, and brecciation. A combination of grade capping and high-grade search restriction was used to allow the high grades to be estimated locally while still controlling their influence further away from the block. Grade capping was applied after compositing for the purposes of grade estimation. The capping values for each variable, the number of values for each variable affected by the capping process, as well as the comparison of the uncapped and capped mean and Coefficient of Variation (CV) are presented in Table 14.6.

Table 14.6: Grade Cap Per Variable and Capping Statistics

Variable	Grade Cap (PPM)	Number of Samples Capped	Uncapped Mean (PPM)	Capped Mean (PPM)	Uncapped CV	Capped CV
Au	100	6	0.33	0.26	32.45	8.81
Ag	60	8	0.85	0.84	3.11	2.73

A high-grade search restriction limits samples above a certain grade from being estimated beyond a specified ellipsoid search distance, in this case 24 m x 12 m x 6 m in the major, semi-major, and minor directions, respectively. The restricted grade for the Au and Ag estimation was determined by evaluating natural breaks in the sample distribution of the log probability plots for the composited samples within each estimation domain. Table 14.7 outlines the grade restriction used for each domain as well as the number of samples restricted.

Table 14.7: Restricted Grade Applied to Composited Samples

Domain	High-Grade Search Restriction			
	Restricted Grade Au (PPM)	No. of Samples Restricted	Restricted Grade Ag (PPM)	No. of Samples Restricted
1 - 121	5.5	20	17	9
2 - Discovery/Dauntless	22	33	32	15
3 - Enterprise West	-	-	3	5
4 - Enterprise East	7	2	3.1	5
5 - Rye Patch TVL	10	14	20	10
6 - Rye Patch Op	5	8	10.5	4
7 - Midway Hills	1	6	10.5	3
8 - Enterprise Outer	-	-	-	-
9 - Rye Patch TVL Outer	-	-	-	-
10 - Discovery/Dauntless Outer	-	-	-	-

14.1.4.5 Domain Statistics

A second phase of EDA was undertaken in X10 once the drill hole samples were composited within the estimation domains. To allow for the evaluation of trends and patterns in the domained data, the QP developed descriptive univariate statistics as well as a series of statistical plots, including histograms, box and whisker plots, and probability plots for each variable in each domain. Table 14.8 provides a summary of the descriptive statistics for the capped composited sample populations within each estimation domain. Figure 14.19, Figure 14.20, and Figure 14.21 provide the frequency distribution for the Au sample populations within each estimation domain.

Table 14.8: Descriptive Statistics for Capped – Composited Samples by Domain

Domain	Variable	Count	Min	Max	Mean	Variance	StDev	CV	Skewness	Kurtosis	Median
1 - 121	Au (ppm)	4,325	0.00	46.75	0.18	1.11	1.06	5.94	25.10	947.30	0.03
	Ag (ppm)	2,385	0.00	57.00	0.98	6.82	2.61	2.66	11.30	180.10	0.38
2 - Discovery/ Dauntless	Au (ppm)	7,528	0.00	100.00	0.65	17.97	4.24	6.55	17.29	351.10	0.07
	Ag (ppm)	3,065	0.00	60.00	1.56	12.10	3.48	2.23	9.67	124.40	0.88
3 - Enterprise West	Au (ppm)	322	0.00	1.18	0.08	0.02	0.15	1.92	4.96	29.18	0.04
	Ag (ppm)	228	0.02	60.00	1.20	11.92	3.45	2.88	12.36	170.50	0.78
4 - Enterprise East	Au (ppm)	730	0.00	21.70	0.18	1.23	1.11	6.27	14.35	243.90	0.03
	Ag (ppm)	490	0.01	14.67	0.87	0.87	0.93	1.08	8.06	105.60	0.72
5 - Rye Patch TVL	Au (ppm)	3,659	0.00	50.40	0.29	2.20	1.48	5.11	18.13	488.70	0.04
	Ag (ppm)	2,527	0.00	60.00	1.04	7.94	2.82	2.72	10.39	155.30	0.29
6 - Rye Patch OP	Au (ppm)	1,738	0.00	12.81	0.18	0.45	0.67	3.66	10.16	141.10	0.04
	Ag (ppm)	1,233	0.00	36.11	1.58	3.68	1.92	1.21	7.73	109.90	1.12
7 - Midway Hills	Au (ppm)	3,291	0.00	11.84	0.04	0.06	0.24	6.27	40.02	1,851.00	0.01
	Ag (ppm)	2,178	0.00	22.66	0.58	1.25	1.12	1.92	8.32	114.60	0.25
8 - Enterprise Outer	Au (ppm)	464	0.00	1.32	0.05	0.02	0.13	2.56	6.91	54.68	0.02
	Ag (ppm)	362	0.00	3.55	0.60	0.42	0.65	1.08	2.08	5.28	0.44
9 - Rye Patch TVL Outer	Au (ppm)	147	0.00	0.62	0.02	0.00	0.06	3.54	8.18	80.62	0.00
	Ag (ppm)	147	0.00	3.07	0.17	0.13	0.36	2.06	5.27	35.34	0.06
10 - Discovery/ Dauntless Outer	Au (ppm)	201	0.00	0.84	0.04	0.01	0.09	1.91	5.57	42.35	0.02
	Ag (ppm)	201	0.05	1.43	0.24	0.07	0.27	1.11	2.23	5.73	0.15

Notes: StDev = Standard Deviation, CV = Coefficient of Variation

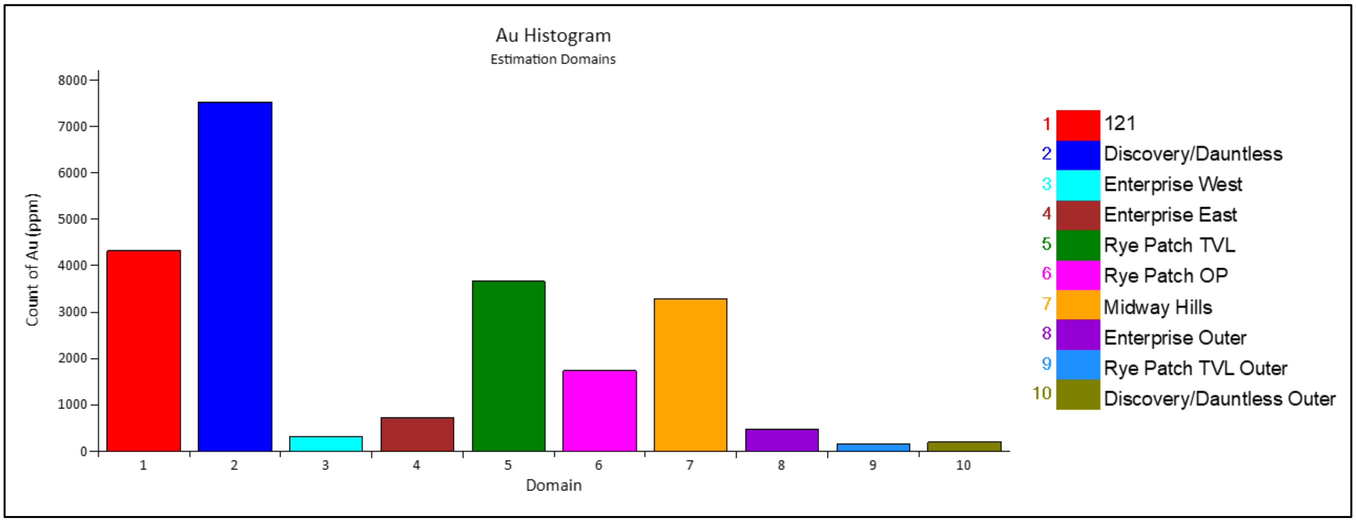


Figure 14.19: Frequency of Au Composited Samples by Estimation Domain

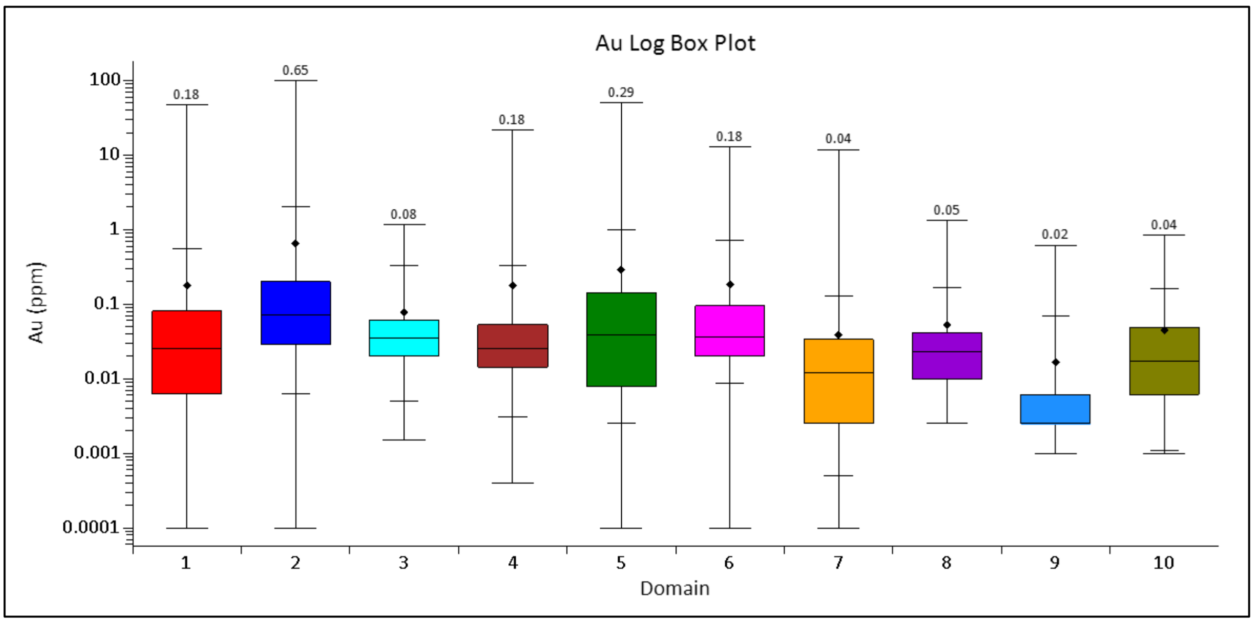


Figure 14.20: Log Box Plot of Au Composited Samples by Estimation Domain

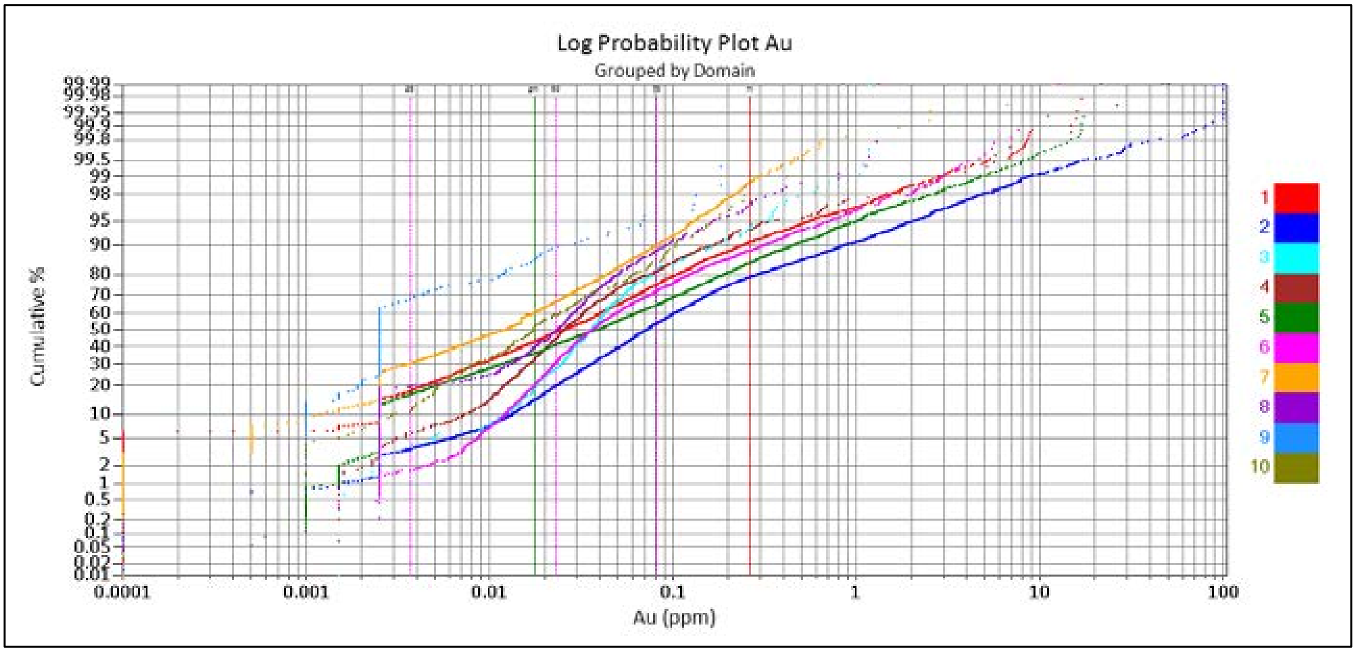


Figure 14.21: Log Probability Plot of Au Composited Samples by Estimation Domain

14.1.5 Spatial Continuity

The QP evaluated the spatial continuity of mineralization using variography. Experimental semi-variograms were generated for Au within each estimation domain using Vulcan Data Analyzer software. The variogram models were used to assign Kriging weights to the composite samples for Ordinary Kriging (OK) estimation, and the ranges of the models were used to determine search ellipsoid distances and to assist with the definition of resource categorization parameters.

Due to the highly skewed distribution, extremely high values, and uneven distribution of sample density, it was necessary to use a normal score transformation on the data prior to evaluating the variography. A normal score transformation transforms a skewed data distribution into a “normal” or Gaussian distribution, which produces a more stable experimental variogram. The experimental variograms are then back-transformed before being modelled and used for grade estimation. Both the transformation and back-transformation are performed by Vulcan Data Analyzer.

The directional orientation, or anisotropy, of mineralization was evaluated using fan variograms (also known as variogram maps), which plot the variance of the sample data along an azimuth and distance. The direction of lowest variance (or gamma) was chosen for three principal directions. Figure 14.22 shows an example of the fan variograms for Domain 5.

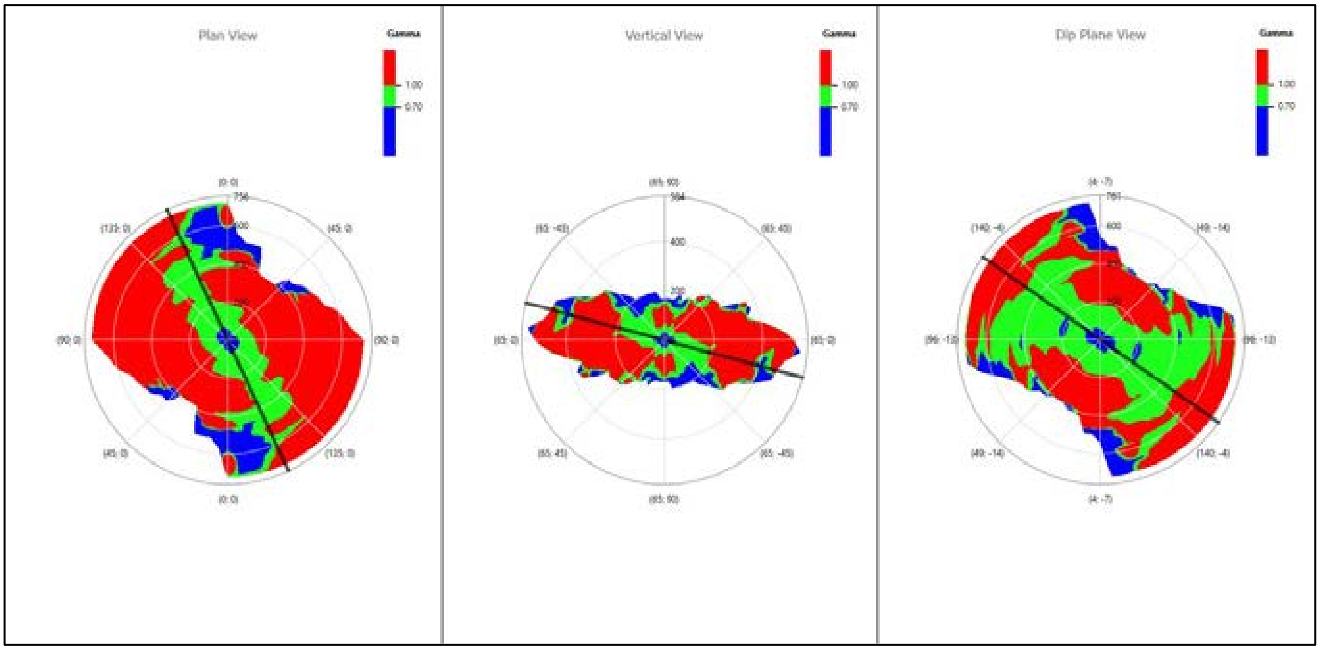


Figure 14.22: Example Fan Variograms for 3 Principal Directions for Domain 5

The chosen directions were then visualized against the sample data in 3D and cross-section to ensure it reflected the data spatially.

The experimental variograms were calculated and modelled with a range of lag distances and tolerances to identify the most robust experimental variogram structure. Directional variography requires search tolerances to be used for calculation of variograms, to address the fact that the drill hole samples are not perfectly aligned in each direction in 3D space and are not equally spaced along that direction. This requires the use of angular and distance tolerances. The parameters used to generate experimental variograms are outlined in Table 14.9.

Table 14.9: Experimental Variogram Parameters

Variogram Parameter	Value	Unit
Search Radius	300	m
Lag size	20	m
Lag tolerance	10	m
Horizontal Angle Tolerance	22.5	°
Vertical Angle Tolerance	7.5	°
Horizontal Distance Tolerance	500	m
Vertical Distance Tolerance	10	m

Two-structure spherical variograms were modelled for Au in each estimation domain. Example variogram models for Au in Domains 2 and 5 are shown in Figure 14.23 and Figure 14.24. A summary of the Au variogram model parameters for all domains are shown in Table 14.10. Due to the low number of samples, Domains 3 and 10 used the same variogram model as Domain 2, Domain 8 used the same variogram model as Domain 4, and Domain 9 used the same variogram as Domain 5. The downhole variogram was used to obtain the nugget for each domain.

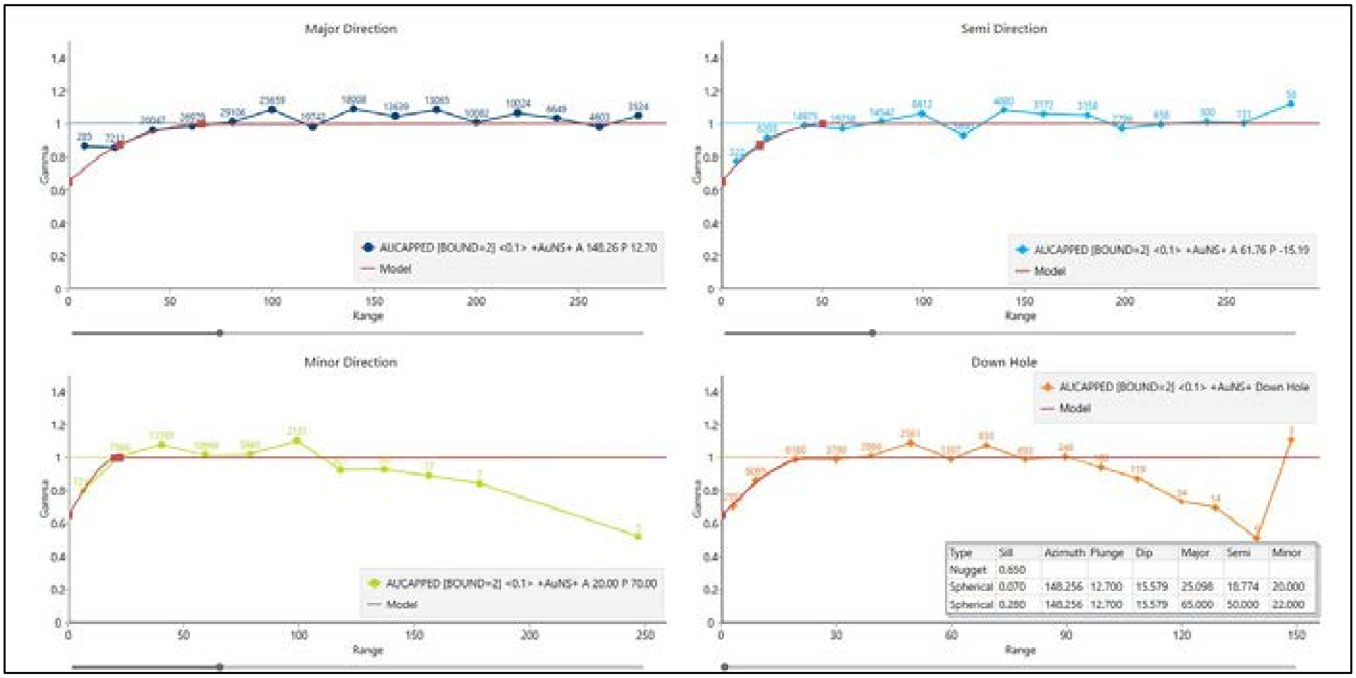


Figure 14.23: Variogram Model for Au in Domain 2

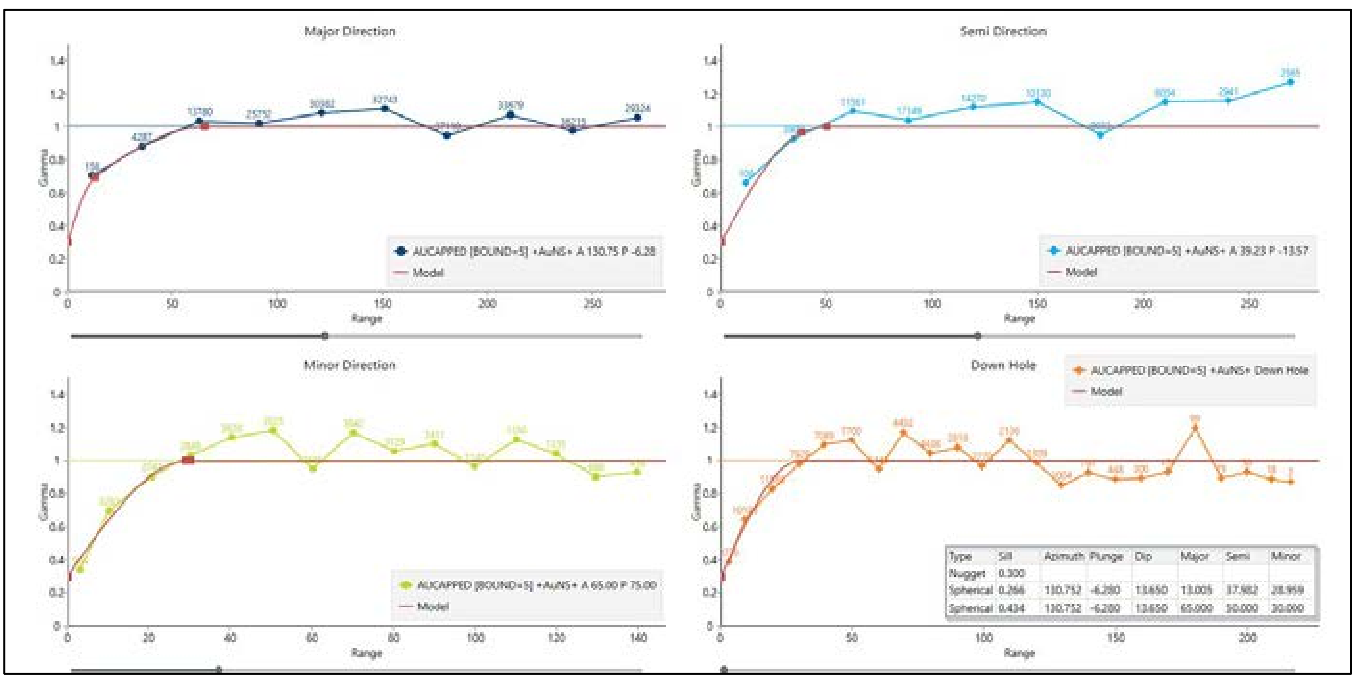


Figure 14.24: Variogram Model for Au in Domain 5

Table 14.10: Summary of Au Variogram Model Parameters

Variable	Domain	Rotation			Variogram Model					Range (m)		
		Bearing	Plunge	Dip	Variogram Type	Model Type	Nugget	Structure No.	Sill	Major	Semi	Minor
Au	1 - 121	125	1	-5	Anisotropic Semivariogram	Spherical	0.20	1	0.46	26	40	39
								2	0.34	140	90	40
Au	2 - Discovery/ Dauntless	148	13	16	Anisotropic Semivariogram	Spherical	0.65	1	0.07	25	19	20
								2	0.28	65	50	22
Au	4 - Enterprise East	51	25	-2	Anisotropic Semivariogram	Spherical	0.30	1	0.57	18	19	16
								2	0.13	75	55	17
Au	5 - Rye Patch TVL	131	-6	14	Anisotropic Semivariogram	Spherical	0.30	1	0.27	13	38	29
								2	0.43	65	50	30
Au	6 - Rye Patch OP	165	-1	-5	Anisotropic Semivariogram	Spherical	0.25	1	0.60	45	35	18
								2	0.16	140	60	45
Au	7 - Midway Hills	115	10	2	Anisotropic Semivariogram	Spherical	0.20	1	0.02	66	66	34
								2	0.78	115	95	70

The variogram directions for each domain generally follow the orientations of nearby faults, in line with the interpretation that mineralization is structurally controlled.

14.1.6 Grade Model

14.1.6.1 Model Extents

A 3D block model was generated using Vulcan conventional block modelling tools. The structure of the block model was rotated to align with the strike orientation of the most continuous trend of mineralization. The block model origin and rotation are outlined in Table 14.11. The block model extents and block size parameters are summarized in Table 14.12. Sub-blocking (or block splitting) was used along lithology boundaries, and within the area of the model with increased drill hole density (see Figure 14.25). The model block size parameters were driven by the drill hole spacing and using guidance from WSP mining engineers based on early high-level evaluations of mining methods.

Table 14.11: Block Model Origin and Rotation

Origin (m)		Rotation (°)	
X Coordinate	490,780	Bearing[1]	110
Y Coordinate	4,234,377	Plunge[2]	0
Z Coordinate	1,460	Dip[3]	0

Notes:
1. absolute bearing of X axis around Z axis
2. relative rotation of X axis around Y axis
3. relative rotation of Y axis around X axis

Table 14.12: Block Model Extents and Block Size Parameters

Direction	Offset from Origin (m)	Extent (m)	Parent		Sub-block	
			Block Size (m)	No. of Blocks	Block Size (m)	No. of Blocks
Easting (X)	0	4,380	12	365	6	730
Northing (Y)	0	1,872	12	156	6	312
RL (Z)	0	540	12	45	6	90

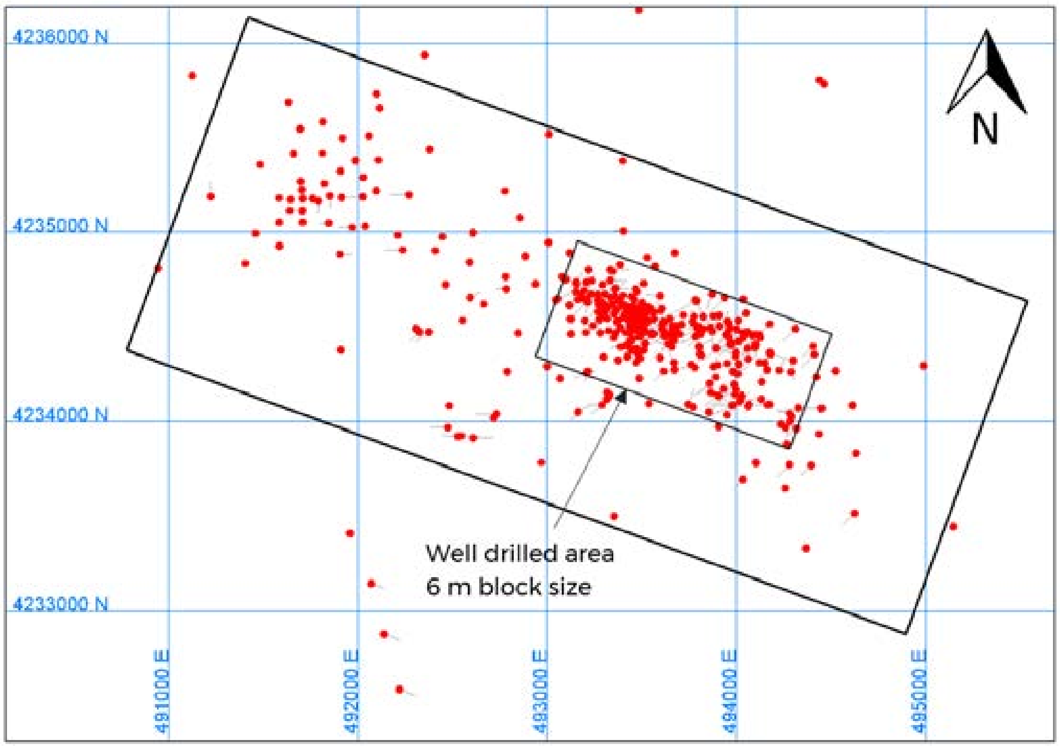


Figure 14.25: Plan View of Block Model Extents Showing Area of 6 m Block Size Relative to Drill Hole Collars (red dots)

14.1.6.2 Block Model Definition

Geological and grade parameter fields for the block model are summarized in Table 14.13. A default (null) value of -99 has been assigned to numerical block parameters.

Table 14.13: Block Model Variables

Variable	Default Value	Default Value	Description
au_ok	-99	Double	Au (PPM) Ordinary Kriging Estimation
ag_id2	-99	Double	Ag (PPM) Inverse Distance Squared Estimation
sg	-99	Double	Specific gravity by lithology type
tonnes	-99	Double	Volume*SG
au_metal	-99	Double	Au Contained Metal (au_ok*tonnes/31.1034768)
lith_txt	null	Name	Lithology Name
lith_int	-99	Integer	Lithology Number
estdom_int	-99	Integer	Estimation Domain Integer
aoi	0	Integer	Area of Interest for 6 m sub-blocking
est_limit	0	Integer	Estimation limit - 20 m boundary around well drilled area
class_txt	null	Name	Resource Classification (Measured, Indicated or Inferred)
class_int	-99	Integer	Resource Classification (Mea = 1, Ind = 2, Inf = 3 and Exp = 0)
gtk_domain_int	-99	Integer	Geotech Domain from CNI
isa	-99	Integer	Inter-ramp Slope Angle based on Geotech Domain
wcode	0	Integer	Whittle code

14.1.6.3 Search Parameters

The Au composite data was estimated into the block model using OK interpolation and Ag using inverse distance squared (ID^2) interpolation.

The composited sample database and the blocks in the model were flagged by the estimation domain wireframes. Grades were interpolated within the domains using only samples within those units (i.e. hard boundaries). The sample search strategy consisted of three search passes, using the modelled variogram ranges to determine the distances. As discussed in Item 14.1.3.4, a limiting boundary was defined around the area of high drill density (Figure 14.8). The purpose of the limiting boundary was to restrict the estimation from interpolating into areas of very low drilling density. The areas outside the boundary were estimated separately from the area within the boundary. The search parameters for all domains are defined in Table 14.14. Domains outside the limiting boundary are denoted with “outer” in the name. All domain boundaries, including those separated by the limiting boundary, were treated as hard boundaries, meaning samples were not permitted to contribute to the estimation outside of their flagged domain. As discussed in Item 14.1.4.4, a high-grade search restriction was applied to the estimation. For all domains, the search distance for samples above the grades defined in Table 14.7 were limited to 24 m x 12 m x 6 m in the major, semi-major, and minor directions, respectively. The Ag estimation used the same search parameters as the Au estimation.

Table 14.14: Grade Estimation Search Parameters

Au and Ag	Orientation				Search Distance (m)			Discretization			Sample Counts			
Domain	Bearing	Plunge	Dip	Search Pass	Major Axis	Semi-Major Axis	Minor Axis	X	Y	Z	Minimum	Maximum	Maximum per drill hole	Minimum Drill Holes
1 - 121 Area	125	1	-5	1	70	45	20	4	4	2	10	16	4	3
				2	140	90	40				10	16		3
				3	280	180	80				8	16		2
2 - Discovery/ Dauntless 3 - Enterprise West 10 - Discovery/ Dauntless Outer	148	13	16	1	32.5	25	11	4	4	2	10	16	4	3
				2	65	50	22				10	16		3
				3	130	100	44				8	16		2
4 - Enterprise East 8 - Enterprise Outer	51	25	-2	1	37.5	27.5	8.5	4	4	2	10	16	4	3
				2	75	55	17				10	16		3
				3	150	110	34				8	16		2
5 - Rye Patch TVL 9 - Rye Patch TVL Outer	131	-6	14	1	32.5	25	15	4	4	2	10	16	4	3
				2	65	50	30				10	16		3
				3	130	100	60				8	16		2
6 - Rye Patch OP	165	-1	-5	1	70	30	22.5	4	4	2	10	16	4	3
				2	140	60	45				10	16		3
				3	280	120	90				8	16		2
7 - Midway Hills	115	10	2	1	57.5	47.5	35	4	4	2	10	16	4	3
				2	115	95	70				10	16		3
				3	230	190	140				8	16		2

14.1.6.4 Density Determination

SG data for the block model was compiled from three sources, as discussed in Item 11.4:

- Kennecott/MLI (1995): 15 measurements from 3 DD holes
- Midway Gold (2011): 336 measurements from 20 DD holes
- Viva/CNI (2022-2023): 19 measurements from 7 DD holes

The compiled bulk density results were imported into Leapfrog and reviewed against the lithology model. Upon reviewing the values in cross-section within Leapfrog, it was observed that the SG within the TVL unit was significantly higher near the TVL-Op nonconformity compared to the upper boundary of the TVL. After discussion with Mr. Bryant, the decision was made to model a unit referred to as the "OP-Melange," representing what is interpreted to be a deeply eroded paleosurface above the nonconformity. The unit is highly fractured and contains a mixture of rock from both the Op and TVL lithologies and is generally highly mineralized, particularly in the Discovery Zone. The OP-Melange lithology has been observed by Mr. Bryant during more recent logging but has not been historically recorded in lithology logs and is therefore not easily modelled as a separate lithological unit. Under guidance from Mr. Bryant, the unit was modelled as an offset of the TVL-Op nonconformity, projected 5 m downward into the Op unit, and 20 m upward into the TVL unit. Figure 14.26 shows the OP-Melange unit relative to the lithology model and SG measurements.

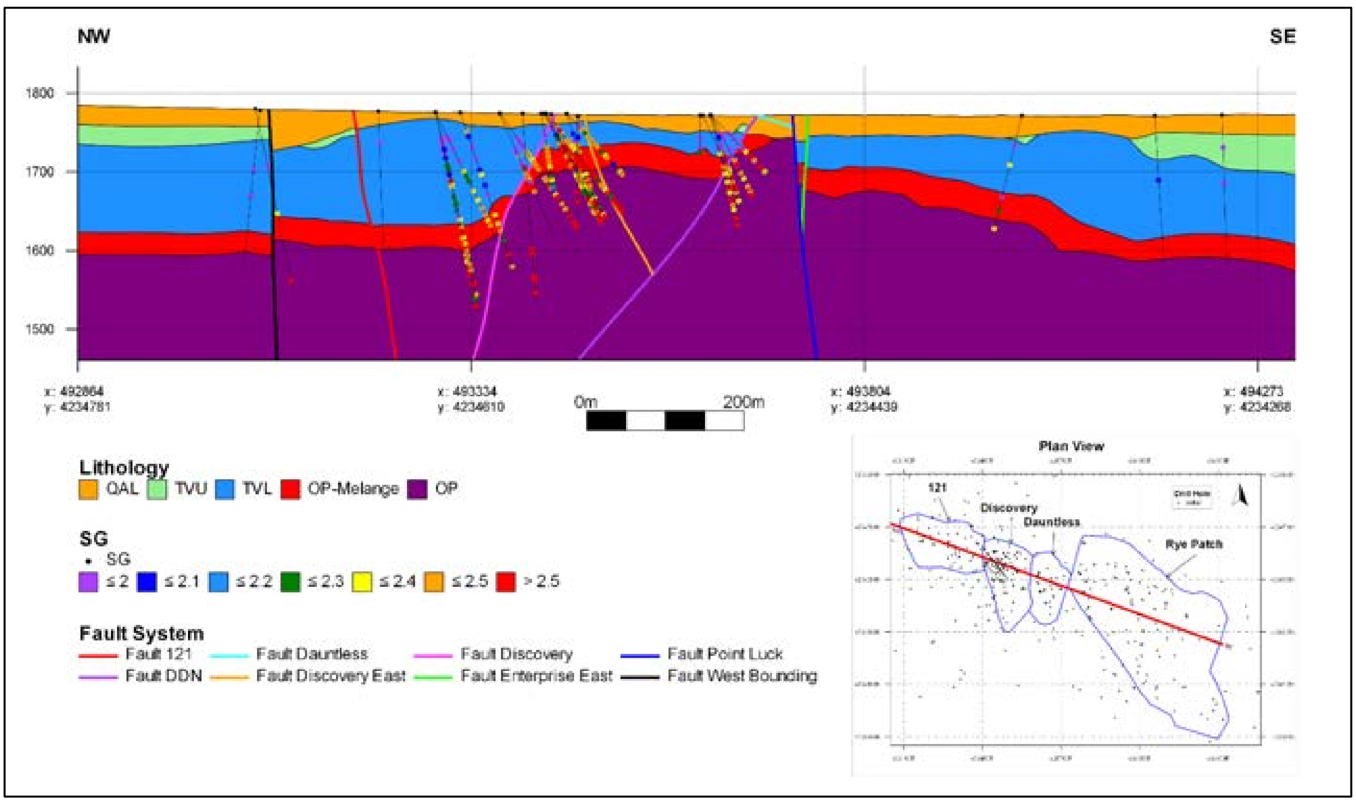


Figure 14.26: Cross-Section of OP-Melange Unit with SG Measurements

The compiled SG values were flagged by the modified lithology model with the OP-Melange unit and exported for statistical analysis within X10. Table 14.15 summarizes the descriptive statistics for the SG within each lithology unit. The unmineralized QAL and TVU units did not have enough samples to determine an average SG for those units. After discussion with Mr. Bryant, an SG of 2.0 was decided for the QAL based on the unconsolidated nature of the unit, and the TVL SG was used for the TVU based on their shared lithological characteristics. The SG for the TVL, OP-Melange, and Op units were determined from the mean SG for each of those units. Table 14.16 summarizes the final SG determinations for each unit. The block model was flagged by the wireframes for each unit, including the OP-Melange, and a script was used to assign a static SG to the blocks based on the flagged units.

Table 14.15: Descriptive Statistics of SG by Lithology Unit

Lithology	Variable	Count	Min	Max	Mean	Variance	StDev	CV	Median
Total	Density	370	1.36	2.73	**2.33**	0.08	0.28	0.12	2.42
QAL	Density	3	1.92	2.12	**2.00**	0.01	0.11	0.05	1.94
TVU	Density	1	1.83	1.83	**1.83**	0.00	0.00	0.00	1.83
TVL	Density	129	1.36	2.55	**2.13**	0.10	0.32	0.15	2.20
OP-Melange	Density	102	1.71	2.60	**2.40**	0.04	0.20	0.08	2.48
Op	Density	135	2.04	2.73	**2.47**	0.02	0.13	0.05	2.48

Notes: StDev = Standard Deviation, CV = Coefficient of Variation.

Table 14.16: Final Block Model SG by Unit

Unit	SG
QAL	**2.00**
TVU	**2.13**
TVL	**2.13**
OP-Melange	**2.41**
Op	**2.47**

14.1.7 Model Review and Validation

The QP performed internal reviews and validations of the block model using a combination of visual inspection and statistical analysis checks between drill hole data and modelled surfaces, thicknesses, and grades.

Visual inspection included review of regularly spaced sections through the block model. Figure 14.27 shows a cross-section of the block model grades with the composited samples. No material issues were identified in the visual inspection.

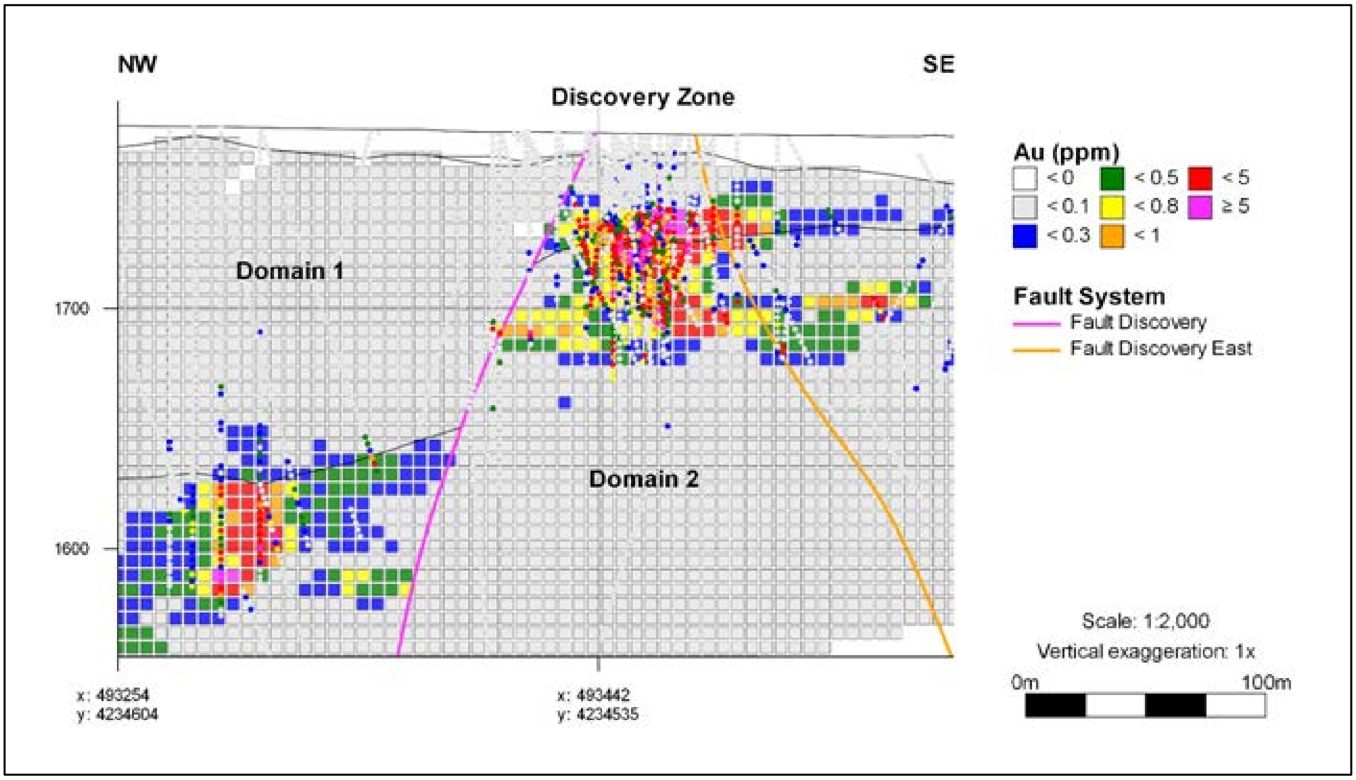


Figure 14.27: Comparison of Block Grades and Composite Sample for Au (PPM) in the Discovery Zone area, Looking Northeast

In addition to the OK interpolation, the QP also estimated grade variables by ID^2, Inverse Distance Cubed (ID^3) and Nearest Neighbour (NN) interpolation for the purposes of assessing global bias and grade smoothing. Global statistical comparisons were made between the ID^2, ID^3, NN, and final OK estimation in a series of statistical tables and swath plots. Clustering of the drill hole data can result in differences between the global mean of the composites and the block estimates, the NN estimation represents the declustered composited data. The Au swath plot analysis and corresponding table are illustrated in Figure 14.28 and Table 14.17. No significant global grade bias was found in the block estimated grade.

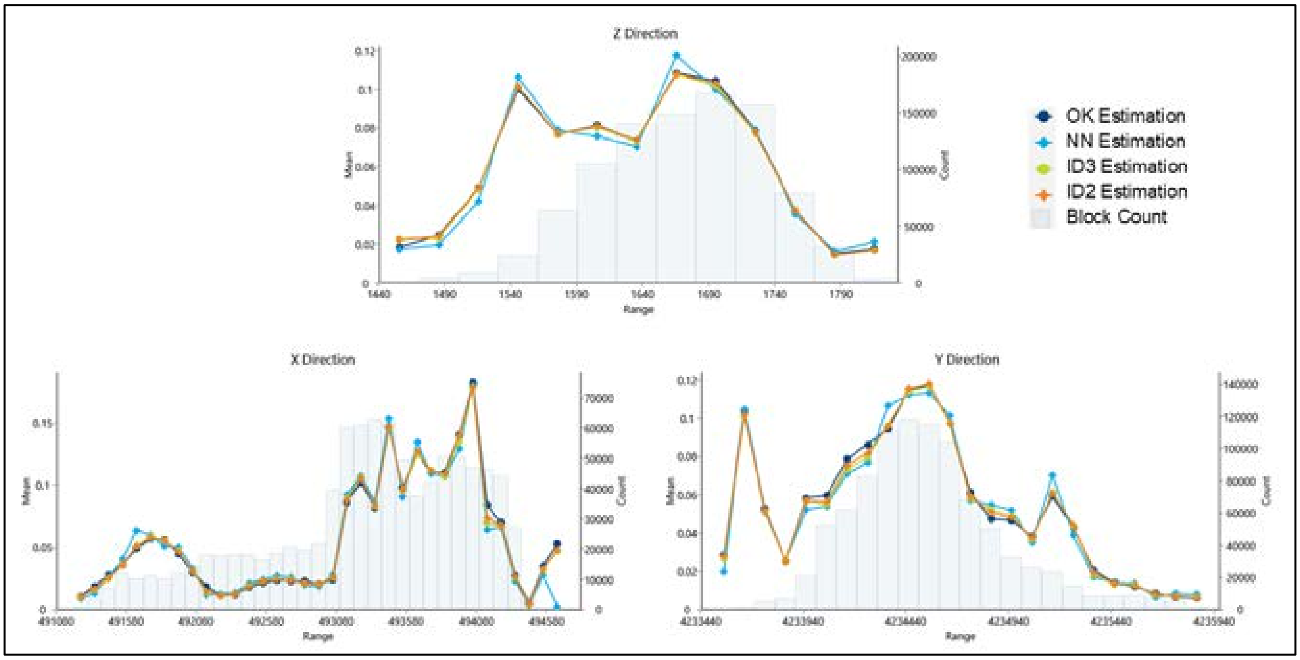


Figure 14.28: Au Swath Plots

Table 14.17: Validation Comparison of Global Mean Grades

Variable name	Count	Min	Max	Mean	Variance	StDev	CV	Median	Variance % with NN
Au ID^2	935,726	0.00	25.00	0.081	0.07	0.27	3.32	0.03	0.0
Au ID^3	935,726	0.00	31.02	0.081	0.09	0.30	3.76	0.03	0.0
Au NN	935,726	0.00	100.00	0.081	0.35	0.59	7.29	0.02	-
Au OK	935,726	0.00	22.48	0.082	0.06	0.25	3.06	0.03	1.2

Notes: StDev = Standard Deviation, CV = Coefficient of Variation.

14.2 Mineral Resource Estimate

This sub-item contains forward-looking information related to Mineral Resource estimates for the Project. The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts or projections in the forward-looking information include any significant differences from one or more of the material factors or assumptions that were set forth in this sub-item, including geological and grade interpretations and controls and assumptions and forecasts associated with establishing the prospects for eventual economic extraction.

14.2.1 Basis for Mineral Resource Estimate

The basis of the Mineral Resource estimates for the Tonopah Gold deposit and the methods in which they were prepared are summarized in this sub-item. For estimating the Mineral Resources for the Tonopah deposit, the QP has applied the definitions of “Mineral Resource” as set forth in the CIMDS adopted May 10, 2014.

Under CIMDS, a Mineral Resource is defined as:

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

A Mineral Resource can be estimated for material where the geological characteristics and the continuity are known or reasonably assumed and where there is the potential for production at a profit.

Mineral Resources are subdivided into categories of Measured, Indicated, and Inferred, with the level of confidence reducing with each category respectively. Mineral Resources are always reported as in situ tonnage and are not adjusted for mining losses or mining recovery.

The Mineral Resource estimates for Tonopah presented in this Technical Report were based on geological and grade block models generated from exploration drilling and sampling data collected by Viva (and its predecessors); the source data are described in detail in Items 6.0, 10.0, and 11.0 of this Technical Report while the data aggregation and modelling methodology is described in detail in Item 14.1.3 of this Technical Report.

14.2.2 Mineral Resource Classification and Categorization

Mineral Resource classification and categorization assigned to the Mineral Resource estimates as presented in this Technical Report were in accordance with NI 43-101, which provides for the classification of a mineral deposit into Mineral Resources and/or Mineral Reserves. Under the NI 43-101 definitions, Mineral Resources can only be categorized under Measured, Indicated and/or Inferred categories, as applicable given the confidence of the estimator in the basis of the estimates. NI 43-101 requires the disclosure of these categories of Mineral Resources in technical reports.

At present, only Mineral Resources have been estimated and there are no Mineral Reserves for the Project.

The Mineral Resource categorization applied by the QP has included the consideration of data reliability, spatial distribution and abundance of data and continuity of geology and grade parameters. The QP performed a statistical and geostatistical analysis for evaluating the confidence of continuity of the geological units and grade parameters. The results of this analysis were applied to developing the Mineral Resource categorization criteria.

Following CIM MRMR Best Practice Guidelines, the QP used the following numeric-based parameters to define the Mineral Resource categories:

- The number of drill holes used
- The estimation pass used to estimate a given block
- The drill hole spacing

The blocks were initially categorized using numerical criteria that considered the number of holes used, the search pass, and the average distance to the three closest drill holes (Table 14.18).

Table 14.18: Initial Classification Criteria

Resource Category	Average Distance to 3 Closest Drill Hole Samples	Number of Holes	Search Pass	Additional Restriction
Measured	≤ 10 m	3	1	Within high drill density limiting boundary
Indicated	≤ 35 m	3	1 or 2	Within high drill density limiting boundary
Inferred	≤ 80 m	2	1 to 3	

The distance to the three closest drill holes was estimated into the model to represent the average drill hole spacing. A script was used to categorize the blocks based on the criteria in Table 14.18. The Measured and Indicated categories were restricted to within the limiting boundary discussed in Item 14.1.6.3. The classification was then smoothed to reduce the “spotted dog” effect, whereby isolated blocks within a category were adjusted to the match the category of the blocks around them.

Figure 14.29 shows a plan section view at 1,700 m amsl of the final Mineral Resource classification and Figure 14.30 shows a long section view.

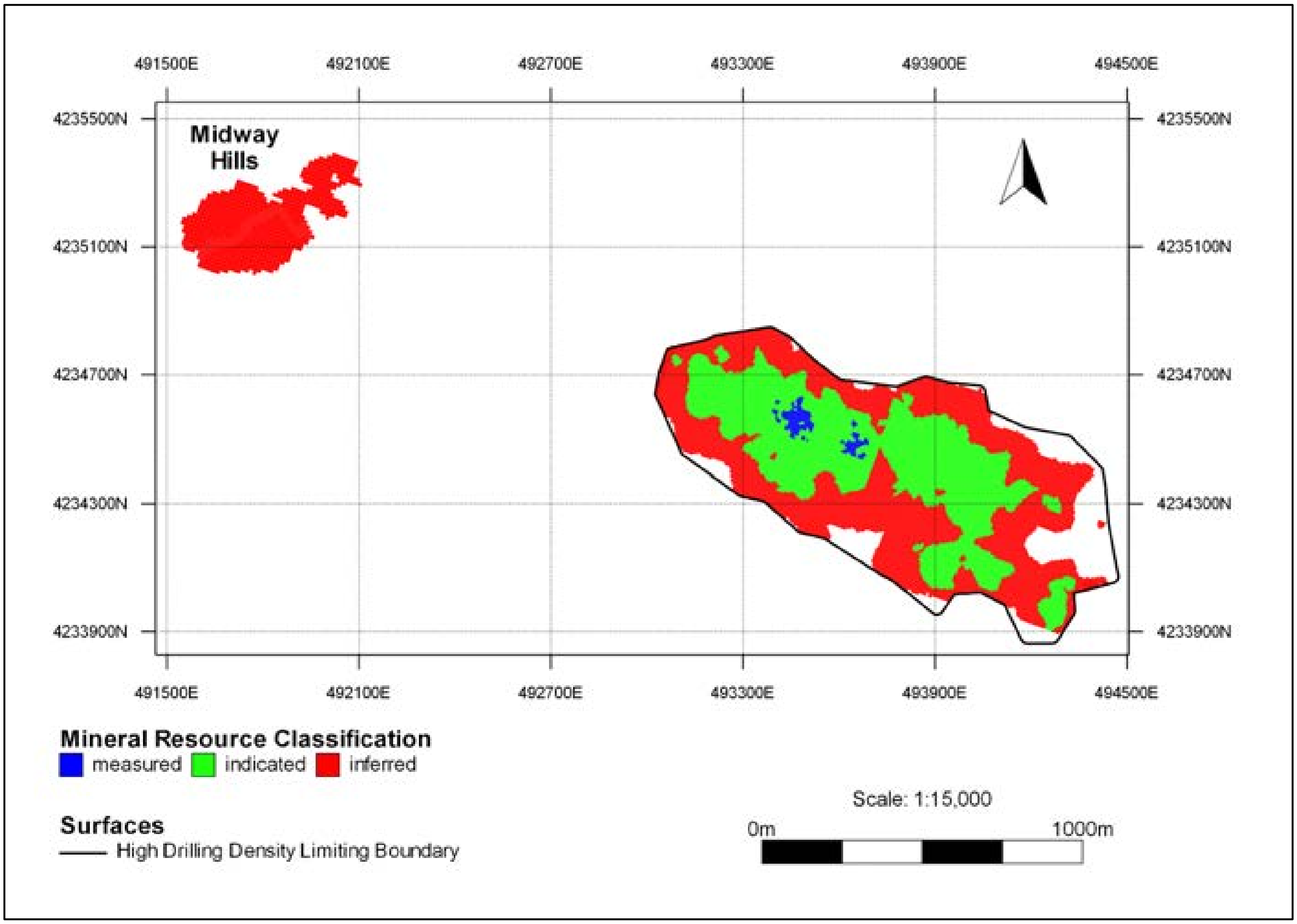


Figure 14.29: Plan Section of Mineral Resource Classification at 1,700 m Elevation

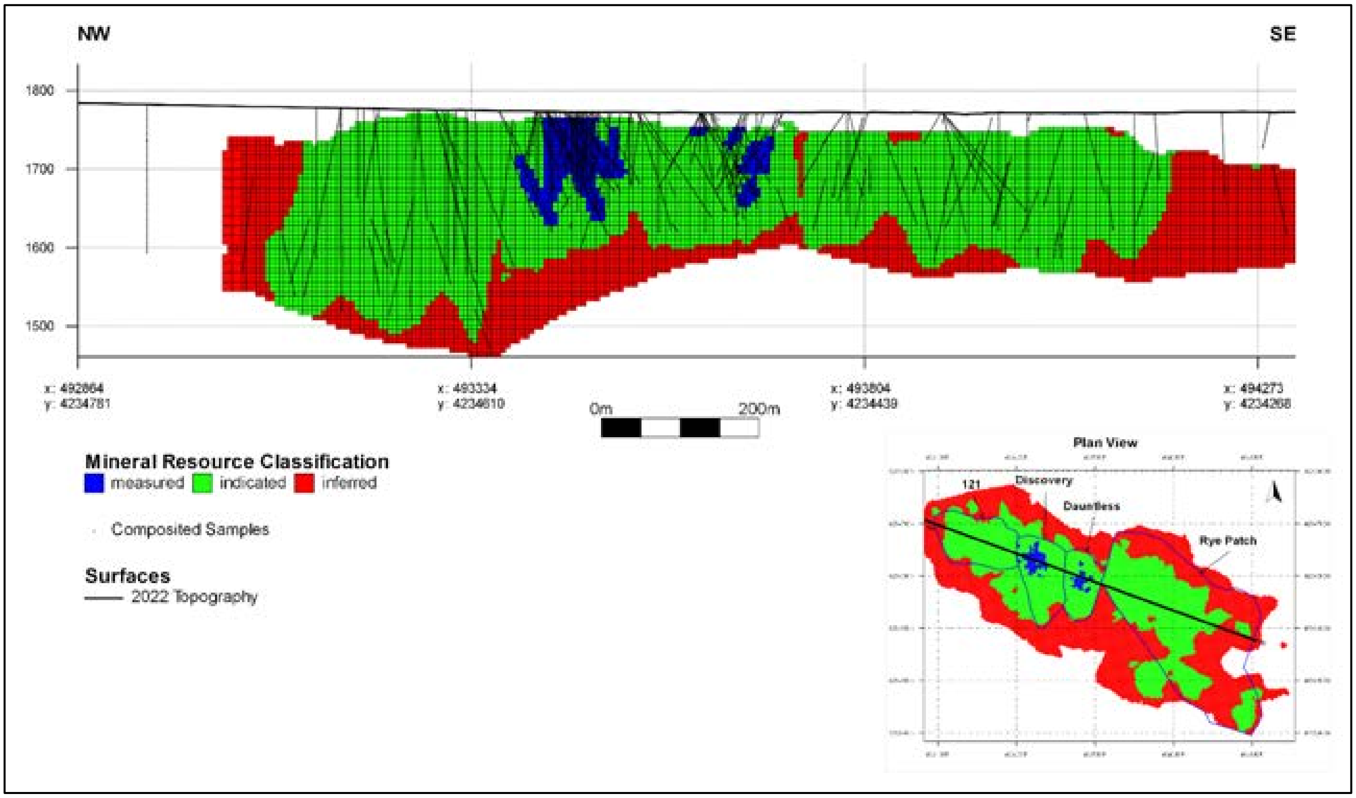


Figure 14.30: Long-Section of Mineral Resource Classification

14.2.3 Reasonable Prospects for Eventual Economic Extraction

The Mineral Resource estimates for the potentially surface mineable resources at Tonopah were constrained by conceptual resource pit shells for the purpose of establishing reasonable prospects of eventual economic extraction (RPEEE) based on potential mining, metallurgical recovery and processing parameters identified by mining, metallurgical, and processing studies performed to date on the Project.

Key constraint inputs included reasonable assumptions for operating costs, geotechnical slope parameters, Au forecast prices, as summarized in Table 14.19, resulting in a minimum Cut-off Grade (COG) of 0.15 g/t Au.

14.2.3.1 Cut-off Grade (COG)

The COG assumes an Au price of US$2,200 and a revenue factor (RF) of 1.2 (equivalent to US$2,640 Au price), and includes all material that can be potentially economically processed. Heap leach processing is the lowest cost processing method used in the operation; therefore, to determine the break-even COG to report the Mineral Resources, the heap leach processing costs and recoveries have been used. The break-even reporting COG determined by WSP for the Mineral Resource was 0.15 g/t Au, as outlined in Table 14.19. Refer to Item 16.0 for discussion of the parameters.

Table 14.19: Break-Even Cut-off Grade for Mineral Resources

Parameter	Unit	Value
Processing Costs (incl. Sustaining CAPEX) + G&A	$/t	7.12
Processing Recovery	%	75.0%
Refining Recovery/Payable	%	99.9%
Royalty	% NSR	1.0%
Refining Cost/Selling Cost	$/oz Au	2
Resource Au Price at RF 1.2	$/oz Au	2,640
Cut-off grade	**g/t Au**	**0.15**

Notes:

- G&A = General and Administration
- t = tonnes
- NSR = Net smelter return
- Oz = ounces

14.2.3.2 Resource Pit Shell

WSP utilized GEOVIA Whittle™ (Whittle) Pit Optimizer software to develop the resource pit shell. Whittle Pit Optimizer uses the Pseudoflow algorithm, along with the user defined input parameters and constraints, to assign a value to each block within a block model, to produce pit shells for selected commodity prices. Refer to Item 16.0 for a detailed discussion of the inputs and constraints. The Whittle Pit Optimization program was used with the input parameters presented in Table 14.20 to provide guidance for establishing RPEEE. Figure 14.31 shows the Au blocks above 0.15 g/t constrained to the resource pit shell.

Table 14.20: Resource Pit Shell Input Parameters

Mining Parameter	Unit	Value
Waste Mining Cost[1]	$/t	1.90
Mineral Mining Cost[1]	$/t	1.90
Overburden Mining Cost[1]	$/t	1.60
Mining Sustaining Capital Cost[2]	$/t	0.24
Mining Recovery[3]	%	100
Mining Dilution[3]	%	0
Processing Parameter	**Unit**	**Value**
Mill Recovery	%	92.5
Heap Leach Recovery	%	75.0
Mill COG	g/t	1.0
Heap Leach COG	-	breakeven
Mill Processing Cost + G&A	$/t	17.50
Mill Processing Sustaining Capital Cost[4]	$/t	0.11
Heap Leach Processing Cost + G&A	$/t	8.70
Heap Leach Processing Sustaining Capital Cost[5]	$/t	0.62
Selling Parameter	**Unit**	**Value**
Au Price at RF 1.2	$/oz	2,640
Au Royalty	%	1.0
Selling Cost	$/oz	2.00
Au Payable	%	99.9

Notes:

1. The mineral and waste mining costs were based on escalated mining costs from similar projects in Nevada and nearby states escalated to Q2 2025 US$ value. The overburden mining cost is the cost of free digging the overburden, without drilling and blasting.
2. Mining sustaining capital cost of 0.24 US$/t was calculated based on the escalated April 2020 PEA cost estimate to Q2 2025 US$ value and was included in the pit optimization to the mining cost.
3. The block model inherently includes a level of dilution. Viva recommended to use 100% mining recovery and 0% dilution, and it is the QP’s opinion that this logic is reasonable for a PEA-level study.
4. Mill processing sustaining capital cost of 0.11 US$/t was obtained from the April 2020 PEA cost estimate and escalated to Q2 2025 US$ value.
5. Heap leach processing sustaining capital cost of 0.62 US$/t was obtained from industry benchmarking, and both were included in the pit optimization to the processing cost for all scenarios.

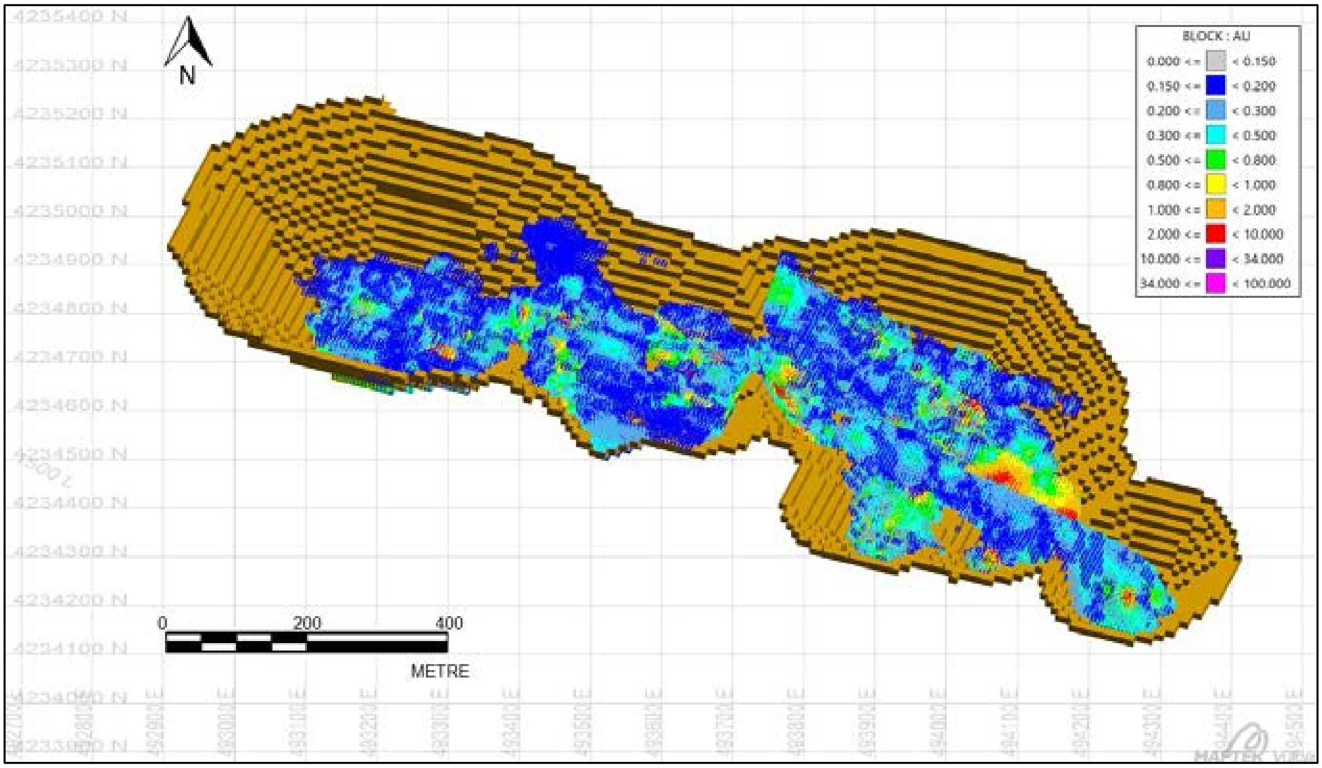


Figure 14.31: Oblique View of Resource Pit Shell with Au Block Grades Above 0.15 g/t

14.2.4 Mineral Resource Statement

Note to readers: The Mineral Resources presented in this Item are not Mineral Reserves and do not reflect demonstrated economic viability. The reported Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that all or any part of this Mineral Resource will be converted into Mineral Reserve. All figures are rounded to reflect the relative accuracy of the estimates and totals may not add correctly.

The categorized estimated Mineral Resources for Tonopah are presented in Table 14.21. Mineral Resource categorization of Measured, Indicated, and Inferred Mineral Resources presented in Table 14.21 is in accordance with CIMDS.

This Mineral Resource Estimate was completed by Róisín Kerr, P.Geo., under the supervision of Brian Thomas, P.Geo., WSP QP for the Mineral Resource. The Effective Date of the Mineral Resource Estimate is June 13, 2025.

Table 14.21: Summary of Estimated Mineral Resources – Effective Date: June 13, 2025

Resource Classification	Tonnes (kt)	Grade		Contained Metal	
		Au (g/t)	Ag (g/t)	Au (oz)	Ag (oz)
Measured	1,690	1.41	3.11	77,000	169,000
Indicated	25,000	0.53	1.98	427,000	1,593,000
Measured + Indicated	**26,690**	**0.59**	**2.05**	**504,000**	**1,762,000**
Inferred	6,905	0.37	1.81	83,000	402,000

Notes:

1. The MRE for the potentially surface mineable resource were constrained by a conceptual pit shell for the purpose of establishing reasonable prospects of eventual economic extraction based on potential mining, metallurgical and processing grade parameters identified by studies performed to date on the Project.
2. Key constraint inputs included reasonable assumptions for operating costs, geotechnical slope parameters, forecast Au prices, and a minimum Cut-off Grade of 0.15 g/t Au.
3. The Cut-off Grade assumes an Au price of US$2,200 and a revenue factor of 1.2 (equivalent to US$2,640 Au price), and includes all material that can be economically processed
4. Heap leach recovery of 75% was assumed.
5. Tonnage and contained metal estimates are rounded to the nearest 1,000.
6. kt = kilotonnes; g/t = grams per tonne; oz/t = troy ounces per tonne.
7. Mineral Resource categorization of Measured, Indicated and Inferred Mineral Resources presented in the summary table is in accordance with the CIM definition standards (CIMDS, 2014).
8. No mining recovery, dilution or other similar mining parameters have been applied.
9. Although the Mineral Resources presented in this Technical Report are believed to have a reasonable expectation of being extracted economically, they are not Mineral Reserves. Estimation of Mineral Reserves requires the application of modifying factors and a minimum of a Pre-Feasibility Study (PFS).
10. The reported Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves.
11. There is no certainty that all or any part of this Mineral Resource will be converted into Mineral Reserve.
12. Mineral Resource estimates are not precise calculations, being dependent on the interpretation of limited information on the location, shape and continuity of the occurrence and on the available sampling results. All figures are rounded to reflect the relative accuracy of the estimates.

The Mineral Resource estimates presented in this Technical Report are based on the factors related to the geological and grade models, and the criteria for RPEEE presented in Item 14.2.3 of this Technical Report. The Mineral Resource estimates may be affected positively or negatively by additional exploration that expands the geological database and models of mineralized zones for the individual deposit areas. The Mineral Resource estimates could also be materially affected by any significant changes in the assumptions regarding forecast prices, costs, or other economic factors that were used in the resource pit shell development process. If the price assumptions are decreased or the assumed costs increased significantly, then the minimum Au grade must be increased and, if so, the potential impacts on the Mineral Resource estimates would likely be material and need to be re-evaluated. Table 14.22 shows the sensitivity of the Mineral Resource estimate to changes in COG.

Table 14.22: Cut-off Grade Sensitivity

Au Cut-off	Measured Category					Indicated Category					Inferred Category				
	Tonnes (kt)	Grade		Contained Metal		Tonnes (kt)	Grade		Contained Metal		Tonnes (kt)	Grade		Contained Metal	
		Au (g/t)	Ag (g/t)	Au (oz)	Ag (oz)		Au (g/t)	Ag (g/t)	Au (oz)	Ag (oz)		Au (g/t)	Ag (g/t)	Au (oz)	Ag (oz)
0.10	1,920	1.26	2.89	78,000	179,000	32,215	0.44	1.81	455,000	1,871,000	10,050	0.30	1.59	95,000	515,000
0.15	1,690	1.41	3.11	77,000	169,000	25,000	0.53	1.98	427,000	1,593,000	6,905	0.37	1.81	83,000	402,000
0.20	1,500	1.57	3.32	76,000	160,000	20,245	0.62	2.13	400,000	1,383,000	5,090	0.45	1.96	73,000	320,000
0.25	1,360	1.71	3.51	75,000	153,000	16,875	0.69	2.25	376,000	1,222,000	4,055	0.50	2.07	65,000	269,000
0.30	1,245	1.84	3.66	74,000	147,000	14,220	0.77	2.37	353,000	1,081,000	3,245	0.56	2.16	58,000	225,000

Notes:

1. Numbers shown are to demonstrate sensitivity of the MRE to changes in COG only; COG below 0.15 g/t do not meet the requirements of RPEEE
2. The official Mineral Resource COG of 0.15 g/t is highlighted.
3. kt = kilotonnes; g/t = grams per tonne; oz/t = troy ounces per tonne.

The Mineral Resource estimates for Tonopah are also based on assumptions that a mining project may be developed, permitted, constructed, and operated at each of these individual advanced exploration properties. Any material changes in these assumptions would materially and adversely affect the Mineral Resource estimates for these deposits; potentially reducing to zero. Examples of such material changes include extraordinary time required to complete or perform any required activities, or unexpected and excessive taxation or regulation of mining activities that become applicable to any proposed mining projects. Except as described in this report, the QP does not know of environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the Mineral Resource estimates.

14.2.5 Comparison with Previous Mineral Resource Estimates

The previous Mineral Resource Estimate was reported in “NI 43-101 Technical Report, Preliminary Economic Assessment of the Tonopah Project” effective date January 1, 2022. Table 14.23 summarizes the changes from the 2022 reported Mineral Resource Estimate.

Table 14.23: Comparison of 2022 and 2025 Mineral Resource Estimates

Resource Classification	2022			2025			Difference		
	Tonnes (kt)	Grade	Contained Metal	Tonnes (kt)	Grade	Contained Metal	Tonnes (kt)	Grade	Contained Metal
		Au (g/t)	Au (ozt)		Au (g/t)	Au (ozt)		Au (g/t)	Au (ozt)
Measured	4,764	0.83	127,000	1,690	1.41	77,000	-3,074	0.58	-50,000
Indicated	11,440	0.73	267,000	25,000	0.53	427,000	13,560	-0.20	160,000
Measured + Indicated	**16,204**	**0.76**	**395,000**	**26,690**	**0.59**	**504,000**	10,486	-0.17	109,000
Inferred	7,352	0.87	206,000	6,905	0.37	83,000	-447	-0.50	-123,000

The changes between the two estimates are due to several factors, outlined below:

- The 2022 geology model was based on 66 drill holes, the 2025 WSP geology model was based on 558 drill holes.
- The 2022 geology model had no faults modelled, the 2025 WSP geology model used the re-interpreted Discovery Fault as well as 13 additional faults to offset and constrain the lithology and resource block model.
- A 10 m upward offset had been applied to the TVL-Op nonconformity in the 2022 geology model; this was removed in the 2025 WSP geology model.
- For the 2025 geology model, WSP adjusted the drill hole collar elevations to match the 2022 DSM.

- The assay database was extensively validated and rebuilt from original source data where available, and intervals were re-converted to metres to ensure correct conversion. The 2022 resource database was missing a significant number of below detection values that were re-added and included in the 2025 model update.
- From the 2022-2024 drill hole programs, 59 new drill holes were used in the 2025 WSP geology model.
- The 2022 block model used a larger block size (20 m x 20 m x 6 m), the WSP block model uses 6 m x 6 m x 6 m block size in the main part of the model, and 12 m x 12 m x 12 m in the outer part of the model, where drill hole spacing is increased.
- The 2022 block model capped Au grade at 10 PPM, the 2023 WSP model uses a grade cap of 100 PPM with the addition of a high-grade search restriction.
- The 2022 block model estimation domains were based on a combination Leapfrog derived grade shells and lithology, the 2025 WSP estimation domains are based on a combination of lithology, faults, and a change in the characteristics of the Au mineralization and geology, as discussed in Item 14.1.3.4.
- The 2022 block model resource classification was based on average spacing of drill holes around the block based on declustering weights, the 2025 WSP block model uses a combination of numerical categories, as outlined in Item 14.2.2.
- The 2022 block model uses a static SG for all blocks, the 2025 WSP block model uses a different SG for each lithology unit, as discussed in Item 14.1.6.4.
- Additional geotechnical and metallurgical testing has been completed since the 2022 estimate, which has modified the input parameters for the pit optimization resource shell, as discussed in Item 16.0.

14.2.6 Mineral Resource Risks and Opportunities

It is the QP’s opinion that the information presented in this Technical Report is representative of the Project, and based on the data verification completed, concludes that the sample database is of a suitable quality to provide the basis for the conclusions and recommendations reached in this Technical Report.

The QP has taken reasonable steps to ensure the block model and MRE are representative of the Tonopah data, but notes that there are risks related to the accuracy of the estimate related to the following:

- The accuracy and quality of historical data.
- The assumptions used by the QP to prepare the data for resource estimation.
- The accuracy of the geological interpretation, including the structural interpretation and estimation domains. Any revisions to domain boundaries may result in significant changes to the overall MRE.
- The variable and structurally complex nature of the deposit geology.
- The impact of outlier grade data, particularly within the Inferred category where drill density is limited.
- Estimation parameters used by the QP.
- Parameters used to support RPEEE.

For these and other reasons, actual results may differ materially from the estimate.

Additional refinement of the structural model and/or the development of an alteration model could potentially improve the confidence in the estimation domain boundaries. Additional drilling to reduce the drill hole spacing will improve the confidence of the estimate and potentially upgrade resource categories. There is also potential to identify new resources through additional exploration or infill drilling, as demonstrated by the new zones identified by the 2022, 2023, and 2024 drilling programs.

15.0 Mineral Reserve Estimates

This Item is not applicable as no Mineral Reserve Estimates have been completed to date.

16.0 Mining Methods

This Item contains forward-looking information related to mining methods, mine design, equipment selection and production plans for the Project. The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts or projections in the forward-looking information include any significant differences from one or more of the following material factors or assumptions that were applied in drawing the conclusions or making the estimates, designs, forecasts or projections set forth in this Item: Mineral resource model, geotechnical, hydrogeological and other surface and underground characteristics described and design criteria; labour and equipment availability and productivity.

16.1 Mining Block Model

The Open Pit Mineral Resource block model described in Item 14.0 of this Technical Report was adapted by WSP for use in PEA level pit optimization, mine design, and scheduling presented in this Item. For this purpose, WSP created a new numeric categorical field, WCODE, to distinguish between blocks based on lithology and resource classification in a format compatible with Whittle.

The WCODE field combines the values of the lithology (LITH) and classification (CLASS) fields.

The logic is as follows:

- Blocks with LITH = 0 represent air and were assigned WCODE = 00.
- Blocks with LITH = 1 (QAL) were considered overburden and non-mineralized; they were assigned WCODE = 10.
- Blocks with LITH = 2 (TVU) were also considered non-mineralized and assigned WCODE = 20.
- Blocks with LITH = 3 (TVL, mineralized):
  - CLASS = 1, 2, or 3 (Measured, Indicated, or Inferred): assigned WCODE = 31, 32, or 33, respectively.
  - All other classification codes (including unclassified or exploration material): assigned WCODE = 30.
- Blocks with LITH = 4 (Op, interpreted as argillic-altered mineralized rock):
  - CLASS = 1, 2, or 3: assigned WCODE = 41, 42, or 43, respectively.
  - All other classification codes: assigned WCODE = 40.

This categorization was designed to support Whittle optimization by grouping all potential mineralized blocks (codes 31–33 and 41–43) as potential mill and heap leach feed, while blocks coded as 10, 20, 30, and 40 were treated as waste.

A geotechnical slope category field, Inter-ramp Slope Angle (ISA), was assigned to each block to support the application of inter-ramp slope angles during pit optimization and design. The initial ISA domain definitions were determined by Call & Nicholas in 2023 (CNI, 2023b). These domains were subsequently refined to reflect the updated geological understanding gained from the 2023 and 2024 drilling campaigns and the revised geological model described in Item 14.0 of this Technical Report. Four ISA domains were defined in total, each corresponding to distinct litho-structural conditions interpreted from the updated geological framework. The ISA coding was applied throughout the block model and used as the basis for assigning slope parameters in Whittle and pit design. Additional details on the geotechnical characterization and slope design criteria are presented in Item 16.2 and 16.4.1 respectively.

The finalized block model, incorporating both WCODE and ISA, was exported for mine planning purposes. Topographic control was established using the 2022 topography surface file discussed in Item 9.0. This surface was used to clip and validate the model extents, serve as a base for pit and Waste Rock Storage Facility (WRSF) designs, and guide infrastructure placement across the site.

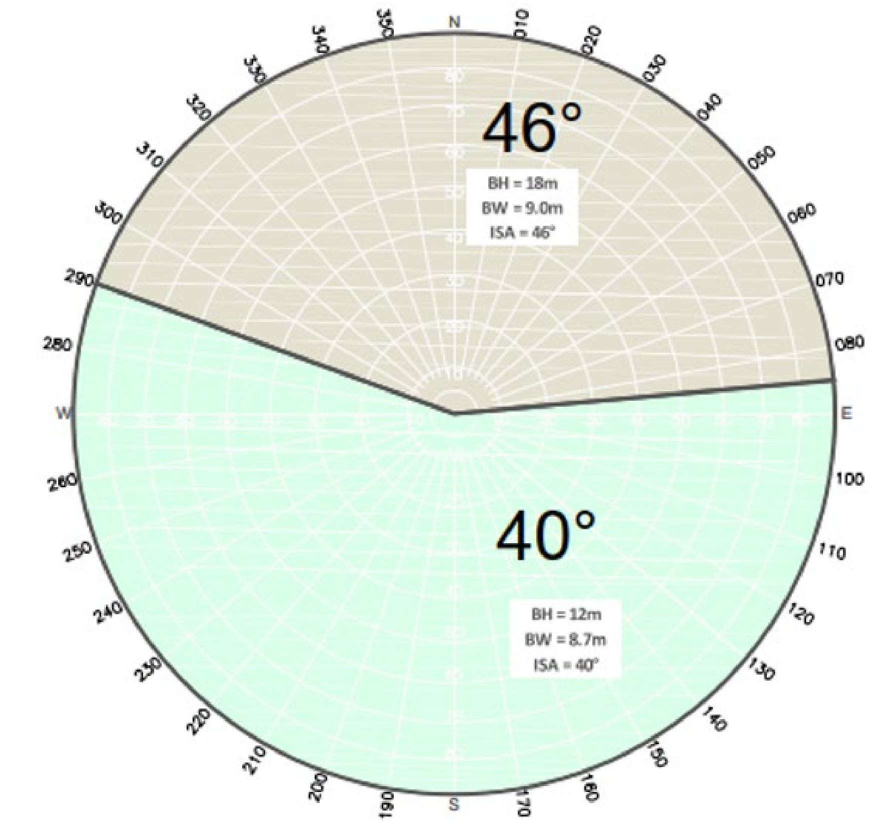
16.2 Geotechnical

The pit slope parameters used in the preparation of the open pit mine design are based on the ISA domains coded in the block model, as described in Item 16.1. The updated ISA framework reflects four slope design domains, each representing distinct combinations of lithology, alteration, and structural fabric that influence rock mass behavior.

The inter-ramp slope angles applied in the pit optimization and detailed pit design were assigned based on this updated domain model. The slope parameters from CNI 2023b are summarized in Table 16.1 and shown visually in Figure 16.1 and Figure 16.2.

Table 16.1: ISA Design Domains (CNI, 2023b)

Domain	ISA (°)	Bench Height (m)	Bench Width (m)	Bench Face Angle (°)	Geomechanical Domain
1	46	18	9	65	West Opa, West FW Volcanics
2	40	12	8.7	65	West HW Volcanics, Disc HW Volcanics, Disc Opa, East Opa
3	See rosette (Figure 16.1)			65	East HW Volcanics
4	35	-	-	65	Qal



Domain	Wall Sector	Wall Dip Direction (degrees)	ISA (degrees)	Bench Height (m)	Bench Width (m)
Domain 3	North	290 - 085	46	18	9.0
	South	085 - 290	40	12	8.7

LEGEND
BH = Bench Height
BW = Bench Width
ISA = Inter-ramp Slope Angle

CALL & NICHOLAS, INC.
TUCSON, ARIZONA USA

ISA DESIGN DOMAIN 3 ROSETTE

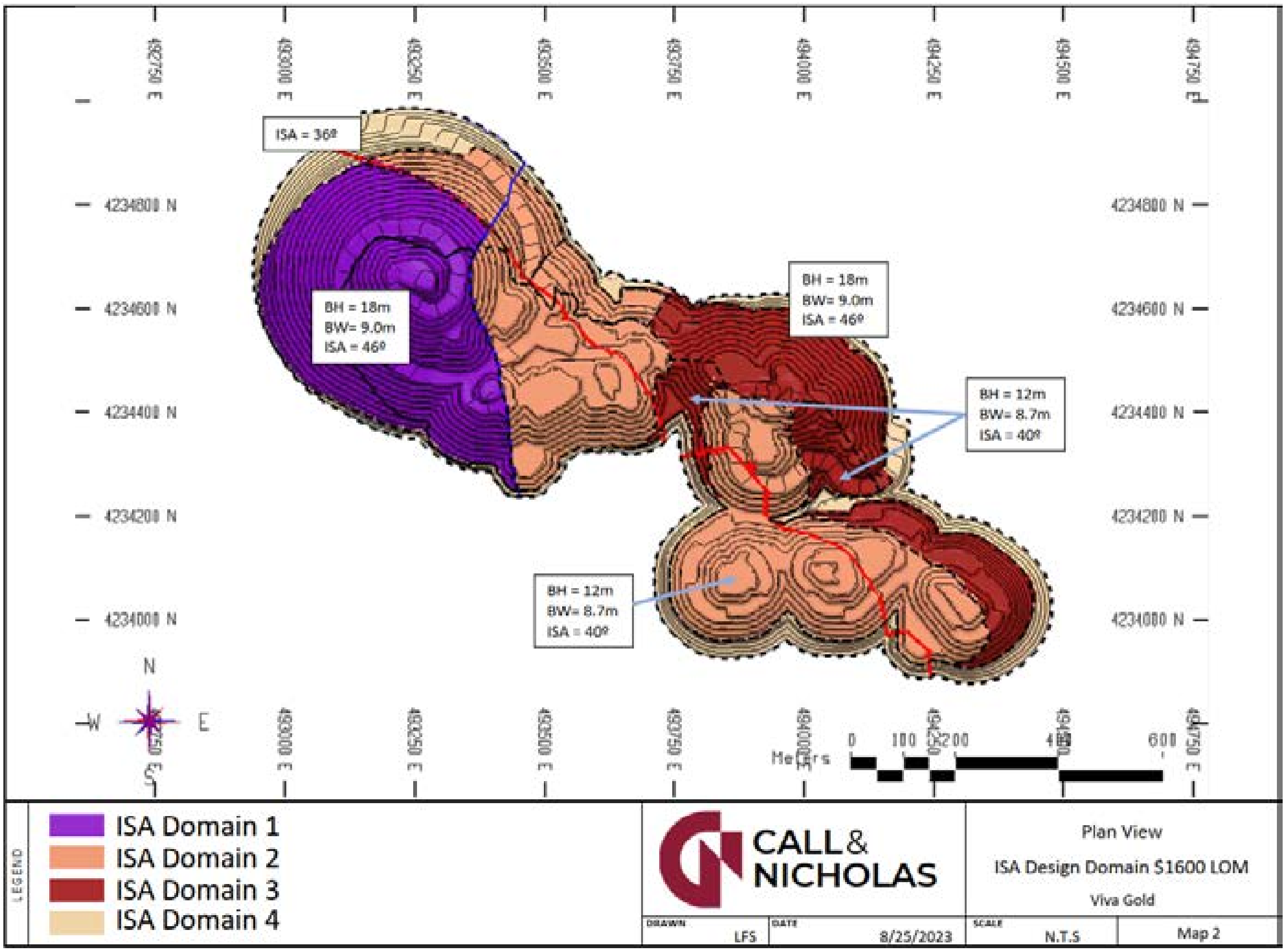
DRAWING | DATE 08/23 | REVISED 8/25/2023 3:51 PM

VIVA GOLD

SCALE NA | Figure 1

Source: Call & Nicholas, 2023

Figure 16.1: ISA Geotechnical Design Domains 3 Rosette



Source: Call & Nicholas, 2023b

Figure 16.2: ISA Geotechnical Design Domains Plan View

The original CNI 2023b recommendations considered variable bench heights of 12 m and 18 m depending on domain-specific geotechnical conditions. For the purpose of the 2025 PEA, all geotechnical domains were standardized to a constant bench height of 12 m to simplify the pit design, facilitate mining operations, and ensure consistency across phases. The Bench Face Angle (BFA), berm width, and Inter-Ramp Angle (IRA) were adjusted accordingly to preserve or exceed the conservatism of the original overall slope angles. The adapted design adheres to a minimum berm width of 7.6 m, consistent with Mine Safety and Health Administration (MSHA) guidelines. The updated geotechnical parameters used in the mine design are summarized in Table 16.2.

Table 16.2: ISA Design Domains for 2025 PEA

Domains	Bench Face Angle (°)	Bench Height (m)	Berm Width (m)	Inter Ramp Angle (°)
Original Geotech Parameters with Varying Bench Height:				
Domain 1	65.0	18.0	9.0	46.0
Domain 2	65.0	12.0	8.7	40.0
Domain 3 North	65.0	18.0	9.0	46.0
Domain 3 South	65.0	12.0	8.7	40.0
Domain 4	65.0	12.0	11.0	36.0
Adapted Geotech Parameters with 12 m Bench Height:				
Domain 1	71.6	12.0	7.6	46.0
Domain 2	65.0	12.0	8.7	40.0
Domain 3 North	71.6	12.0	7.6	46.0
Domain 3 South	65.0	12.0	8.7	40.0
Domain 4	65.0	12.0	11.0	36.0

16.3 Hydrological

No hydrological constraints or considerations were taken into account in the PEA Life-of-Mine Plan (LoMP). Capital and operating costs for a pit dewatering system has been included in project economics as discussed in Item 22.0. The system is designed to lower the ground water table below the bottom of the designed pit to support suggested pit slope angles. Additional review of this point will be required in further studies.

16.4 Pit Optimization

Pit optimization was carried out using Whittle software to establish a set of nested pit shells and evaluate the economic potential of the Tonopah Project deposit under multiple development scenarios. The optimization incorporated the mining block model described in Item 16.1, which includes both lithology-classification-based WCODE values and the geotechnical ISA domain codes used for slope application, as well as the Au and Ag grades and the SG from the block model described in Item 14.0.

The objective of the optimization is to define a series of pit shells that maximize project value under varying revenue conditions, and to support selection of a suitable shell for detailed pit design. Whittle runs were performed using updated economic inputs summarized in Item 16.4.2 and geotechnical parameters detailed in Item 16.4.1. The processing configurations and associated costs and recoveries reflect the latest metallurgical results at the time and scenarios developed with KCA and are detailed in Item 17.0.

Nested shells were generated using a range of RFs from 0.20 to 1.20, in 0.02 increments, relative to a base-case Au price of US$2,150/oz.

A variety of trade-off scenarios were analyzed during the pit optimization stage. These included a mill-only configuration, a heap leach-only configuration, and a combined processing scenario. Additional scenarios considered the impact of keeping or relocating the state road SR 376, which potentially intersects the pit area and acts as a constraint in the base case. Various COG thresholds were also evaluated to delineate high-grade mill feed from low-grade heap leach material, along with different mill throughput configurations. This Technical

Report only details the Base Case scenario, which assumes a combined processing strategy with a 2,000 tonnes per day (tpd) mill and an 8,000 tpd heap leach facility. The state road is assumed to remain in place.

16.4.1 Geotechnical Slope Parameters

Geotechnical slope parameters used in the pit optimization process were derived from the ISA domains coded into the block model (refer to Item 16.2). The ISA coding reflects geotechnical domains defined by CNI 2023b and refined through updated structural and geological interpretation from the 2023 and 2024 drilling programs.

Table 16.3 summarizes the final bench configurations used for each domain, including BFA, berm widths, and IRA. These adapted geometries were used as input in Whittle's slope configuration file and applied according to each block's ISA domain code. Overall slope angles were not explicitly reduced to account for haul ramps due to the shallow depth of the pit and limited sensitivity observed in prior slope reduction scenarios.

Table 16.3: Whittle Simplified Geotechnical Design Domains

Domain	ISA (°)	Bench Height (m)	Bench Width (m)	Bench Face Angle (°)	Geomechanical Domain
1	46	18	9	65	West Opa, West FW Volcanics
2	40	12	8.7	65	West HW Volcanics, Disc HW Volcanics, Disc Opa, East Opa
3	43	18	11	65	East HW Volcanics
4	35	-	-	65	Qal

Overall slope angles were not reduced from the inter-ramp slope angles coded in the ISA block model field to account for proposed haul roads. The relative shallowness of final pit causes the pit economics to be less sensitive to the pit slopes and a couple of degrees change in a sector would not have a large impact on the overall strip ratio, at a PEA-level study such approach is believed to be within the confidence margin.

WSP then ran a sensitivity scenario by reducing the overall slope angles to account for the proposed haul roads. Based on the number of ramp intercepts within a region, the inter-ramp angles were reduced. The differences were in the range of 1-3% for the main parameters: rock tonnage, mineral tonnage, while metal contained and grade were largely the same, therefore confirming the low sensitivity of the pit shell to the impact of the haul ramp.

16.4.2 Economic Input Parameters

Economic input parameters used in the pit optimization were based on updated 2025 estimates reflecting benchmark data from other projects in the area, escalated data from previous studies, current metallurgical estimates, and site-specific cost assumptions. The parameters include mining, processing, and selling assumptions and are summarized in Table 16.4, Table 16.5, and Table 16.6, respectively.

No incremental mining cost due to depth was included due to the relatively shallow pit geometry. Mining costs assume an owner-operated fleet. The selling cost applied in Whittle incorporates transport, refining, royalties, and other deductions. All costs are stated in Q1 2025 US$.

Table 16.4: Mining Parameters

Parameter	Unit	Value
Waste Mining Cost[1]	US$/t	1.90
Mineral Mining Cost[1]	US$/t	1.90
Overburden Mining Cost[1]	US$/t	1.60
Rehandling Cost	-	direct to crusher
Mining Sustaining Capital Cost[2]	US$/t	0.24
Mining Recovery[3]	%	100%
Mining Dilution[3]	%	0%

Notes:

1. The mineral and waste mining cost were based on escalated mining cost from similar projects in Nevada and nearby states escalated to Q2 2025 US$ value. The overburden mining cost is the cost of free digging the overburden, without drilling and blasting.
2. Mining sustaining capital cost of 0.24 US$/t was calculated based on the escalated April 2020 PEA cost estimate to Q2 2025 US$ value and was included in the pit optimization to the mining cost.
3. The block model inherently includes a level of dilution. Viva recommended to use 100% mining recovery and 0% dilution, and it is the QP's opinion that this logic is reasonable for a PEA-level study.

Processing costs include mill processing costs, heap leach processing costs, tailings handling, environmental expenses, and general & administrative costs. Processing costs and metal recoveries were provided by KCA.

Table 16.5: Processing Parameters

Parameter	Unit	Value
Heap Leach Processing Cost + G&A	US$/t	8.70
Mill Processing Cost + G&A	US$/t	17.50
Heap Leach Sustaining Capital Cost[1]	US$/t	0.62
Mill Sustaining Capital Cost[2]	US$/t	0.11
Heap Leach Au Recovery (Variable)	%	75-76
Mill Au Recovery	%	92
High-Grade Cut-off (COG)	g/t Au	1.00

Notes:

1. Heap leach processing sustaining capital cost of 0.62 US$/t was obtained from industry benchmarking, and both were included in the pit optimization to the processing cost for all scenarios.
2. Mill processing sustaining capital cost of 0.11 US$/t was obtained from the April 2020 PEA cost estimate and escalated to Q2 2025 US$ value.

Table 16.6: Selling Parameters

Parameter	Unit	Value
Au Price	US$/oz	2,150
Au Royalty	% NSR	1.0
Au Selling Cost	US$/oz	2.00
Au Payability	%	99.9

Ag is not included as saleable product in the pit optimization, it does not affect the pit optimization, the mine design or the mine LoMP. There may be opportunities to increase Mineral Resources if Ag is included in the pit optimization. Economic value from Ag credits will be added in the economic analysis in Item 22.0.

16.4.3 Pit Optimization Results

The results of the pit optimization are presented in Table 16.7 and Figure 16.3. Nested shells were generated using RFs ranging from 0.20 to 1.20, in increments of 0.02, using the economic, geotechnical, mining and processing inputs described in previous sub-items.

The selected final pit shell corresponds to RF 0.98 (Shell 40). This shell was selected to maximize pit-constrained resources while maintaining a reasonable economic margin. Shell 40 contains a total of 106.9 million tonnes (Mt) of rock, composed of 82.7 Mt of waste, 9.0 Mt of heap leach feed, and 15.2 Mt of mill feed, for a total mineral tonnage of 24.2 Mt. The average diluted Au grade is 0.641 g/t, containing approximately 498 kilotroy ounces (koz) of Au, and yielding a strip ratio of 3.42.

Shells beyond RF 0.98 showed minimal incremental economic improvement, with value curves flattening across Shells 31 through 51. Viva selected Shell 40 to maximize pit-constrained resources, and the QP considers this selection reasonable for a PEA-level study based on current Au market conditions.

Two interim phases were selected to facilitate operational sequencing, and reduce pre-strip:

- Phase 0: RF 0.52 (Shell 17)
- Phase 1: RF 0.78 (Shell 30)

Table 16.7: Summary of Nested Pit Shell Results by Revenue Factor

Pit	Revenue Factor	Rock Tonnes (t)	Waste Tonnes (t)	Mineral Tonnes (t)	Strip Ratio (w:o)	Au Grade (g/t)	Contained Au (oz)
1	0.2	3,042,859	2,685,664	357,195	7.52	3.806	43,709
2	0.22	3,735,510	3,200,754	534,756	5.99	3.115	53,568
3	0.24	7,672,467	6,778,104	894,363	7.58	2.878	82,768
4	0.26	7,676,657	6,731,949	944,708	7.13	2.765	83,983
5	0.28	7,927,464	6,903,341	1,024,123	6.74	2.639	86,889
6	0.3	8,383,167	7,224,063	1,159,104	6.23	2.463	91,779
7	0.32	8,703,234	7,451,665	1,251,569	5.95	2.358	94,882
8	0.34	9,957,937	8,490,100	1,467,837	5.78	2.192	103,452
9	0.36	10,295,400	8,692,974	1,602,426	5.42	2.077	107,032
10	0.38	11,025,277	9,267,484	1,757,793	5.27	1.982	112,001
11	0.4	19,705,034	16,982,419	2,722,615	6.24	1.761	154,167
12	0.42	20,273,257	17,353,604	2,919,653	5.94	1.692	158,841
13	0.44	23,497,536	19,991,221	3,506,315	5.70	1.565	176,444
14	0.46	26,720,317	22,508,738	4,211,579	5.34	1.442	195,239
15	0.48	28,044,356	23,518,240	4,526,116	5.20	1.393	202,772
16	0.5	29,187,339	24,313,928	4,873,411	4.99	1.340	210,023
17	0.52	32,681,457	27,275,895	5,405,562	5.05	1.289	223,979
18	0.54	42,650,783	35,309,811	7,340,972	4.81	1.140	269,017
19	0.56	43,036,821	35,364,460	7,672,361	4.61	1.108	273,408
20	0.58	44,906,990	36,553,579	8,353,411	4.38	1.060	284,647
21	0.6	46,206,230	37,375,053	8,831,177	4.23	1.029	292,205
22	0.62	47,265,351	37,974,439	9,290,912	4.09	1.000	298,851
23	0.64	48,700,131	38,802,000	9,898,131	3.92	0.966	307,426
24	0.66	51,282,979	40,659,358	10,623,621	3.83	0.934	318,910
25	0.68	51,958,210	40,960,339	10,997,871	3.72	0.915	323,425
26	0.7	52,627,908	41,234,565	11,393,343	3.62	0.895	327,982
27	0.72	54,628,381	42,533,005	12,095,376	3.52	0.867	337,351
28	0.74	56,808,186	44,260,154	12,548,032	3.53	0.853	344,315
29	0.76	57,568,631	44,535,437	13,033,194	3.42	0.834	349,382
30	0.78	58,595,186	45,061,872	13,533,314	3.33	0.816	354,872
31	0.8	78,733,590	62,869,978	15,863,612	3.96	0.789	402,487
32	0.82	84,031,558	66,986,920	17,044,638	3.93	0.765	419,084
33	0.84	85,889,393	67,888,158	18,001,235	3.77	0.741	428,643
34	0.86	86,318,665	67,878,093	18,440,572	3.68	0.729	432,264
35	0.88	100,615,200	79,869,437	20,745,763	3.85	0.702	468,370
36	0.9	101,598,620	80,228,152	21,370,468	3.75	0.690	473,839
37	0.92	102,935,772	80,824,498	22,111,274	3.66	0.676	480,474
38	0.94	103,995,611	81,291,579	22,704,032	3.58	0.665	485,637
39	0.96	105,071,062	81,662,805	23,408,257	3.49	0.653	491,389
40	0.98	106,866,847	82,688,526	24,178,321	3.42	0.641	498,358
41	1	107,756,443	82,964,060	24,792,383	3.35	0.631	503,098
42	1.02	108,373,296	83,047,648	25,325,648	3.28	0.623	506,941
43	1.04	108,453,518	82,714,807	25,738,711	3.21	0.616	509,358
44	1.06	110,107,426	83,643,931	26,463,495	3.16	0.606	515,352
45	1.08	110,583,883	83,633,598	26,950,285	3.10	0.598	518,547
46	1.1	111,428,324	83,928,573	27,499,751	3.05	0.591	522,401
47	1.12	113,470,204	85,174,051	28,296,153	3.01	0.581	528,912
48	1.14	115,799,613	86,655,528	29,144,085	2.97	0.572	535,910
49	1.16	119,215,498	89,210,003	30,005,495	2.97	0.564	543,928
50	1.18	120,015,471	89,451,458	30,564,013	2.93	0.557	547,571
51	1.2	121,186,655	90,026,456	31,160,199	2.89	0.551	551,724

Notes: t = tonnes, w:o = waste to mineralized material ratio, g/t = grams per tonne, oz = troy ounces

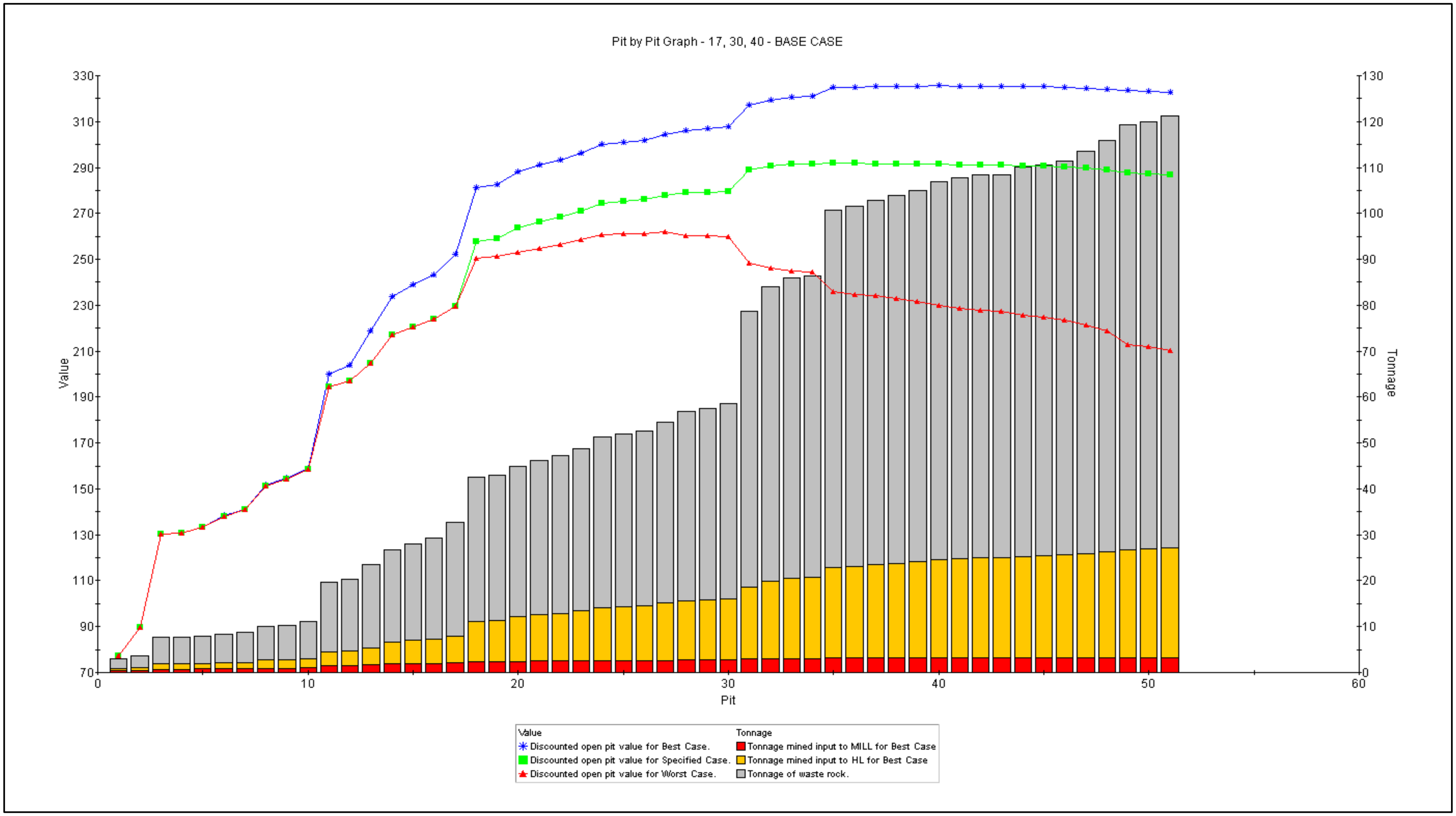


Figure 16.3: Pit-by-Pit Graph

16.5 Mine Design

16.5.1 Phases and Ultimate Pit Design

The open pit was designed using the geotechnical parameters summarized in Table 16.2 and the pit and ramp design parameters listed in Table 16.8. The largest truck considered in the design is the Caterpillar (CAT) 777G (100-ton class), with a maximum vehicle operating width of 6.7 m. Based on this, single-lane and double-lane road widths were set at 13.4 m and 23.4 m (drivable surface), respectively. Ramp widths (all-in, including safety berms and drainage ditches) were set at 17.8 m for single-lane and 27.8 m for double-lane segments, with a maximum gradient of 10%, for the bottom 3-5 benches a maximum gradient of 11% was allowed if necessary. Minimum catch bench width was set at 7.6 m in accordance with MSHA guidelines.

Three pit phases were defined in the mine plan:

- Phase 0: designed based on nested shell RF 0.52, designed to prioritize high-grade material early in the schedule and reduce initial stripping. See Figure 16.4.
- Phase 1: designed based on nested shell RF 0.78. See Figure 16.5.
- Phase 2: the ultimate pit was designed based on nested shell RF 0.98. See Figure 16.6.

Where possible, the final wall was established within the interim phase designs, and a minimum spacing of 35 m between phases was maintained to ensure safe and efficient operation with 100-ton class equipment. The main exit ramp is located on the northeast side of the pit to connect with the primary crusher, mill, heap leach pad, and WRSF. The pit design is constrained on the west-southwest side by existing road SR 376, which remains in place under the Base Case; however, the road could be relocated in future scenarios should resource growth justify expansion in that direction.

Tonnages and grades by phase are summarized in Table 16.9.

Table 16.8: Pit Ramp and Road Design Parameters

Description	Units	Value	Additional Information
Largest truck	short ton	100	CAT 777G
Maximum vehicle operating width	m	6.7	CAT 777G
Road width - double lane	m	23.4	3.5x maximum vehicle operating width, drivable surface only
Road width - single lane	m	13.4	2x maximum vehicle operating width, drivable surface only
Maximum ramp gradient	%	10.0	
Safety berm height	m	1.4	Equal to 1/2 largest vehicle wheel radius
Safety berm width	m	3.6	Angle of repose 37° where a drop >3 m exists
Ditch depth highwall side	m	0.5	Ditch on highwall to capture runoff from pit wall
Ditch width highwall side	m	0.8	
Ramp width (all-in) - double lane	m	27.8	Road width plus safety berm and ditch
Ramp width (all-in) - single lane	m	17.8	Road width plus safety berm and ditch. Single lane ramp in the bottom 3-5 benches.
Ramp turning radius - double lane	m	34.8	1.25x ramp width. 14.2 m OEM turning circle clearance radius
Ramp turning radius - single lane	m	22.2	1.25x ramp width. 14.2 m OEM turning circle clearance radius
Minimum catch bench width	m	7.6	Program Information Bulletin No. MSHA-P10-09

Notes: OEM = Original Equipment Manufacturer

Table 16.9: Tonnes and Grades by Pit Phase

Phase/ Grade Bin	Tonnage (t)	Contained Metal		Grade		Strip Ratio
		Au (oz)	Ag (oz)	Au (g/t)	Ag (g/t)	
Phase 0	**18,670,597**					**2.59**
Mineral	5,207,188	150,398	393,912	0.03	0.08	
HG	1,035,012	94,573	160,627	2.84	4.83	
LG	3,731,120	43,422	196,293	0.36	1.64	
MG	441,055	12,402	36,992	0.87	2.61	
Waste	13,463,409					
Phase 1	**41,651,787**					**3.46**
Mineral	9,341,702	178,391	582,300	0.02	0.06	
HG	1,296,415	72,902	128,195	1.75	3.08	
LG	7,258,622	83,432	401,026	0.36	1.72	
MG	786,665	22,057	53,079	0.87	2.10	
Waste	32,310,085					
Phase 2	**55,664,342**					**5.17**
Mineral	9,015,793	152,130	542,285	0.02	0.06	
HG	835,815	48,497	98,079	1.80	3.65	
LG	7,628,595	88,184	402,223	0.36	1.64	
MG	551,383	15,449	41,983	0.87	2.37	
Waste	46,648,549					
Total	**115,986,725**					**3.92**
Mineral	23,564,683	480,918	1,518,497	0.63	2.00	
HG	3,167,243	215,972	386,902	2.12	3.80	
LG	18,618,337	215,038	999,542	0.36	1.67	
MG	1,779,102	49,908	132,053	0.87	2.31	
Waste	92,422,042					

Notes:
- Low-grade (LG): 0.18-0.76 g/t Au
- Mid-grade (MG): 0.76-1.00 g/t Au
- High-grade (HG): >1.00 g/t Au

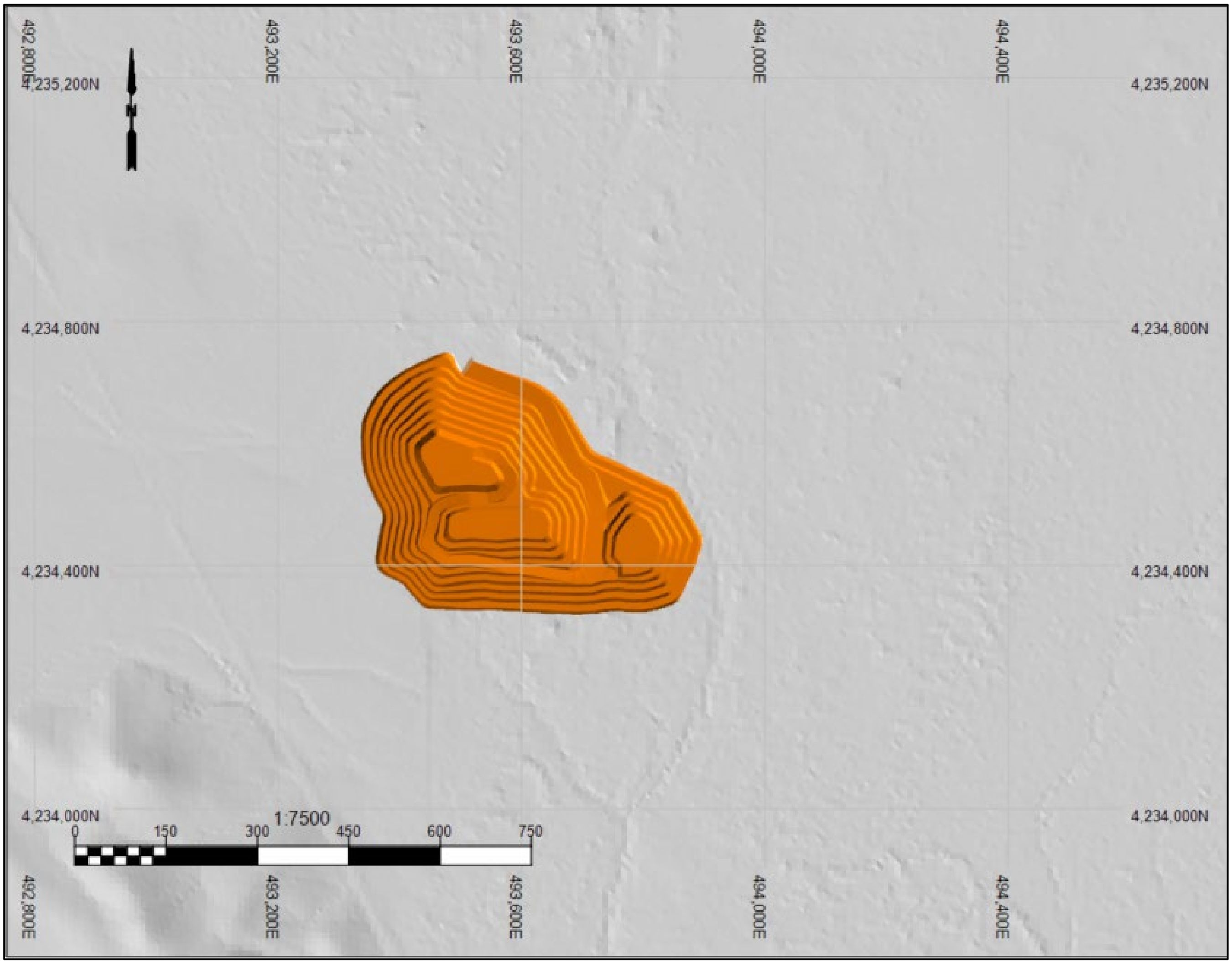


Figure 16.4: Phase 0 Pit Design

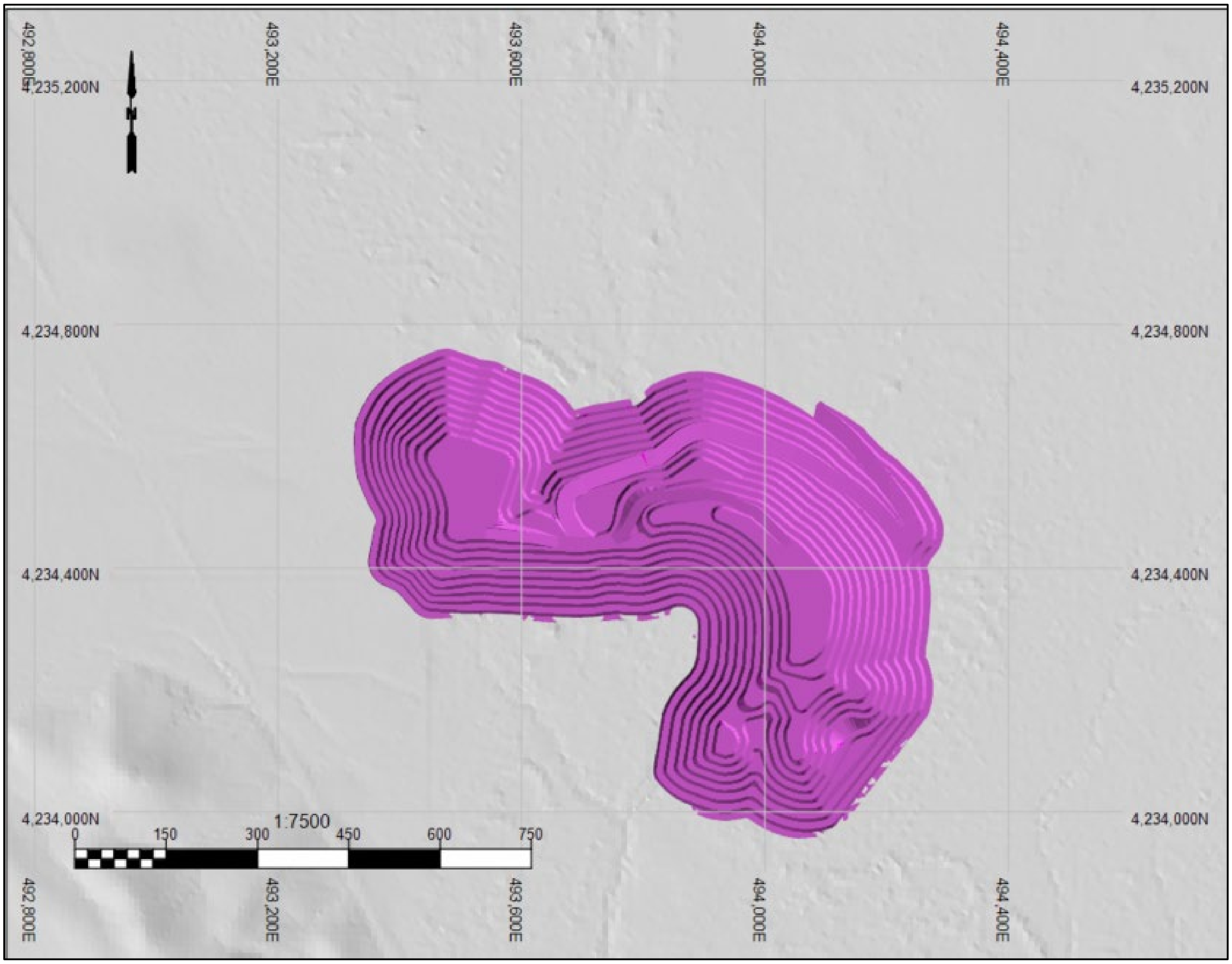


Figure 16.5: Phase 1 Pit Design

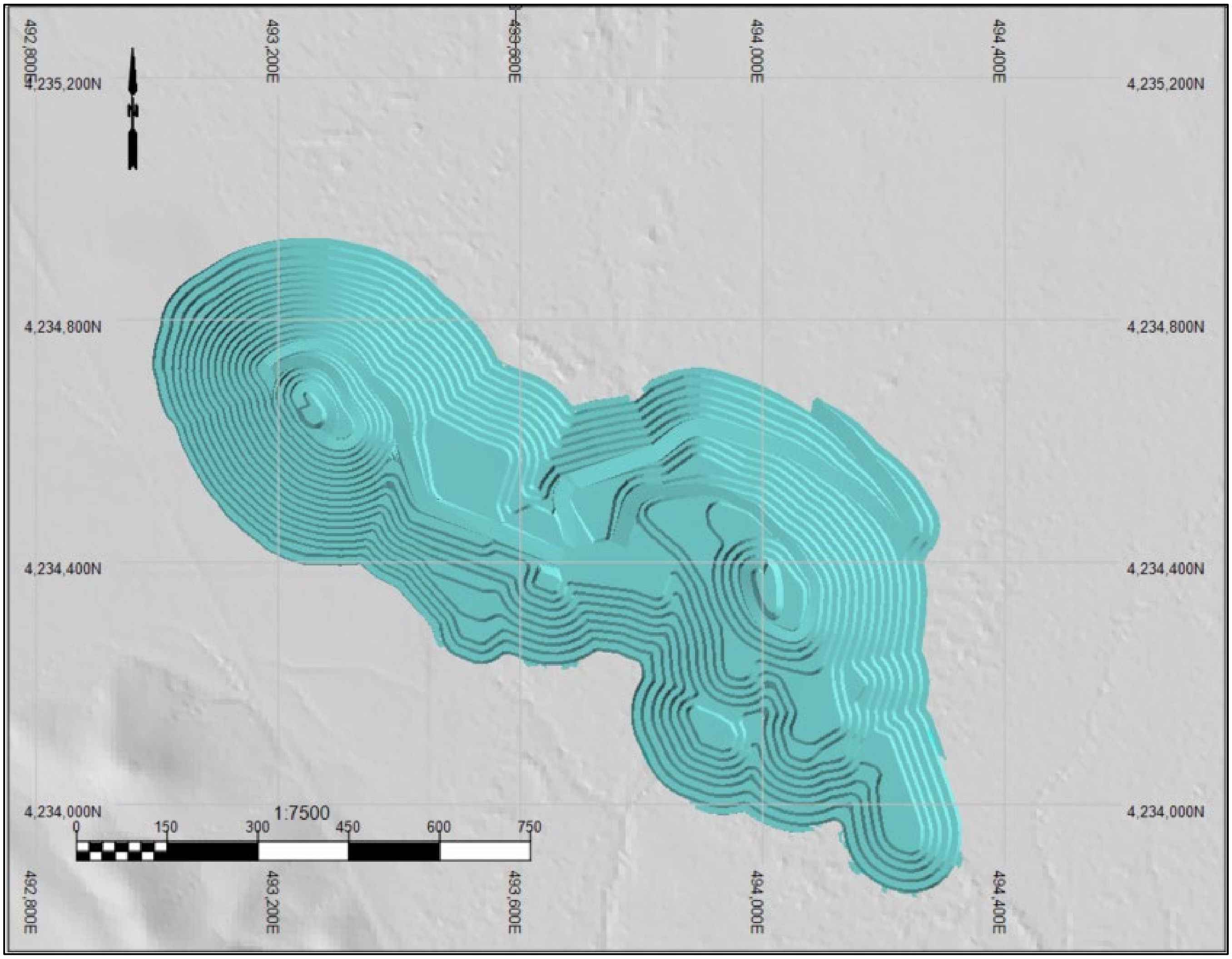


Figure 16.6: Phase 2 (Ultimate Pit) Design

16.5.2 Waste Storage

WRSF design was based on truck-dumped, benched lift construction with inter-ramp angles, catch benches, and ramp dimensions consistent with industry practices and MSHA guidelines. WRSF design parameters are summarized in Table 16.10. A lift height of 9 m was adopted with a minimum 7.6 m wide catch bench and a 27° inter-ramp angle. Ramps were designed for 100-ton trucks with a 27.8 m double-lane width and a 10% gradient.

The WRSF is located northeast of the open pit in proximity to the mill, primary crusher, and heap leach pad. The design capacity is approximately 4.5 Mt, which exceeds current PEA waste volume requirements and provides room for future expansion should additional resources be incorporated into an updated mine plan (Figure 16.7).

In addition, approximately 15 Mt of waste from the northwest portion of the pit is planned to be in-pit dumped into the southeast pit toward the end of the mine life. This in-pit dumping strategy will reduce haulage distances and operating costs in the final years of production.

Table 16.10: WRSF Design Parameters

Description	Units	Value
WRSF Lift Height	m	9.0
WRSF Catch Bench Width	m	7.6
WRSF Face Angle	°	35.0
WRSF Inter-Ramp Angle	°	27.0
WRSF Ramp width (all-in) - double lane	m	27.8
WRSF Road Gradient	%	10.0

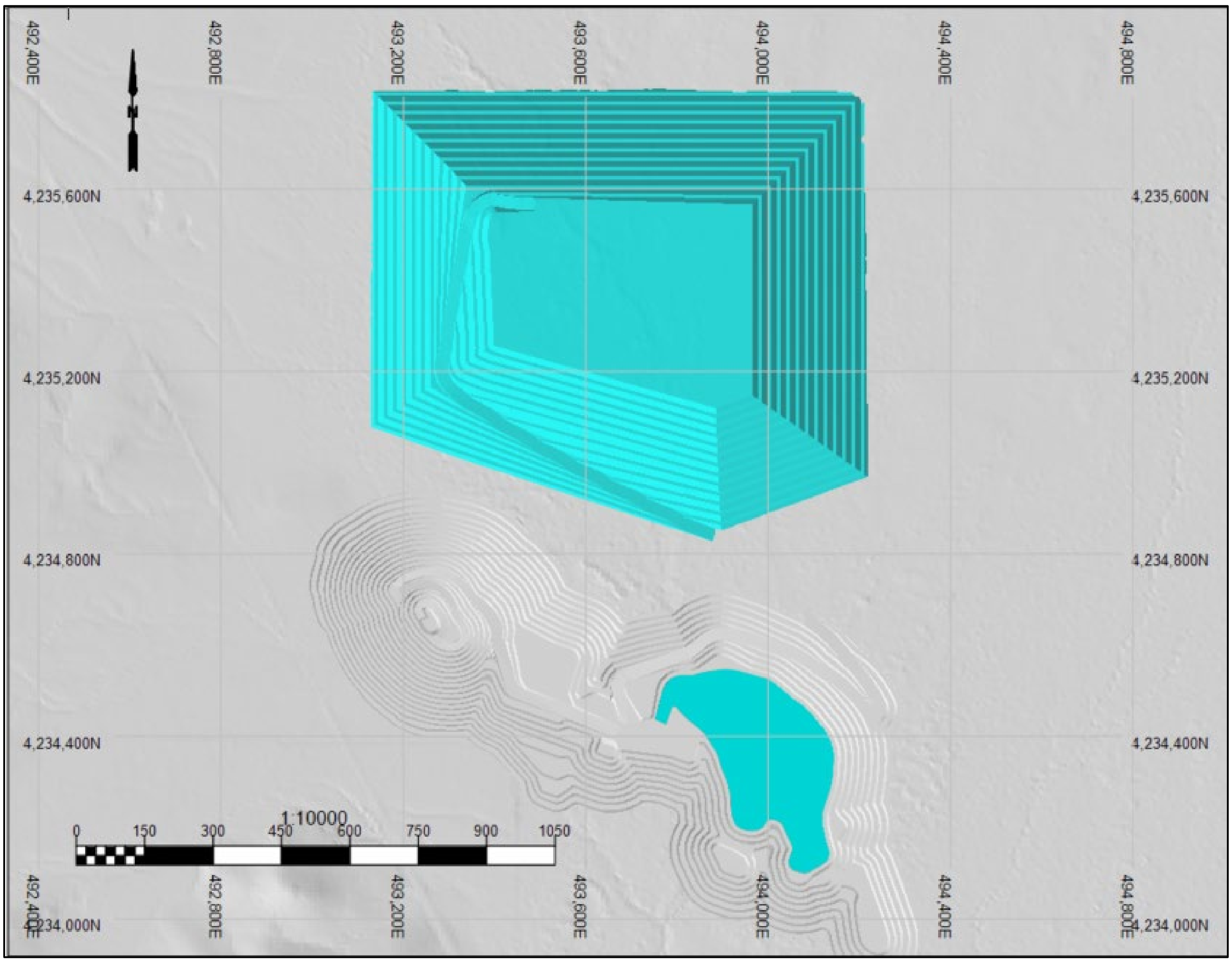


Figure 16.7: Waste Rock Storage Facility Design

16.6 Life-of-Mine Plan

16.6.1 Cut-Off Grade

COGs for the PEA LoMP were calculated based on Au price and cost estimates available at the time of pit optimization and scheduling. An Au price of US$2,150/oz was used for the base case. The resulting COGs reflect the economic thresholds for material classification as mineralized material and the preferred processing route.

- Heap Leach COG: 0.18 g/t Au
- Mill COG: 0.28 g/t Au
- Break-even Mill-to-Heap Leach COG: 0.76 g/t Au

Any material above 0.18 g/t Au is considered mineralized. Material between 0.18–0.76 g/t Au is better suited for the heap leach facility due to higher economic margin under current cost and recovery assumptions. Material between 0.76–1.00 g/t Au (mid-grade) yields slightly higher margin if processed via the mill and may be redirected accordingly depending on mill capacity availability. High-grade material above 1.00 g/t Au is prioritized and only sent for mill feed.

The COGs used in the LoMP were derived from the cost structure summarized in Table 16.11. More refined processing cost estimates were later developed by KCA for the financial model, which included slightly lower processing costs. As a result, the LoMP uses conservative COG values, and any downstream economic evaluation benefits from increased processing margins on all classified mineralized material.

Table 16.11: COG Summary Inputs and Results

Parameter	Unit	Mill	Heap Leach
Processing Cost + G&A	US$/t processed	17.61	9.32
Processing Recovery	%	92.0	76.0
Refining Recovery & Payable	%	99.9	99.9
Royalty	% NSR	1.0	1.0
Refining & Selling Cost	US$/oz	2.0	2.0
Au Price	US$/oz	2,150	2,150
Calculated COG	g/t Au	0.28	0.18
Break-Even COG (Mill vs. Heap Leach)	g/t Au	-	0.76

16.6.2 Production Schedule

The Life-of-Mine (LoM) production schedule was developed to maximize project net present value (NPV) while maintaining practical constraints on mining rates, processing capacities, and equipment requirements. The schedule fills both the heap leach and mill circuits to nameplate capacity where possible, while minimizing the number of truck units required and avoiding sudden fluctuations in equipment demand. Pre-strip capital in the early years is reduced, and mining rates are smoothed to ensure a steady equipment profile over the mine life.

Material was categorized into three economic grade bins for scheduling purposes:

- Low-grade: 0.18-0.76 g/t Au, preferentially routed to the heap leach pad.
- Mid-grade: 0.76-1.00 g/t Au, directed to the mill only when mill capacity is not filled with high-grade.
- High-grade: >1.00 g/t Au, always prioritized for mill feed.

The mill has a nameplate capacity of 730 kt/year. High-grade material is scheduled first into the mill, and mid-grade is used to fill any available capacity. Mid-grade not processed through the mill is instead routed to the heap leach pad. The heap leach facility, with a nameplate capacity of 2,920 kt/year, receives primarily low-grade material. If heap leach capacity is fully utilized, surplus low-grade is stockpiled and rehandled in later years when capacity becomes available. Rehandling cost has been accounted for in the margin calculation. Only 40 kt of heap leach material are rehandled over the LoM. High-grade and mid-grade are not stockpiled and are processed in the year they are mined to preserve their economic contribution.

The scheduling logic also seeks to delay waste and mineralized material mining where possible without compromising feed continuity. No pre-strip year is required, as mineralized material is located within 10 m of surface and accessible from the start of Phase 0. Early mineral access allows for immediate feeding of both the heap leach and mill circuits without requiring a dedicated pre-production stripping campaign.

Phase 0 is mined during Years 1 and 2. Phase 1 is pre-stripped in the latter part of Year 1 and mined from Years 2 to 5. Phase 2 is pre-stripped in the latter part of Year 4 and mined through Years 5 to 7.

Although the LoMP was developed assuming a 2,000 tpd mill, a smaller mill configuration could be studies if that allowed for a reduced initial capital spend in the mill. A 1,400 tpd mill would be sufficient to process all high-grade material over the 7-year mill operating window, while a 1,500 to 1,800 tpd facility would offer additional flexibility to accommodate mid-grade if there are periods of limited high-grade availability.

A summary of yearly mined tonnage by grade bin, mill and heap leach feed, and strip ratio is provided in Table 16.12.

Table 16.12: Life-of-Mine Production Schedule

Name	Units	LOM Total	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
MINED TONNAGE			-	-	-	-	-	-	-
Waste Tonnage	tonnes	91,564,056	18,350,004	18,650,017	12,000,020	12,000,004	11,350,102	13,500,014	5,713,887
Low-Grade Tonnage	tonnes	18,618,328	2,776,648	2,623,158	2,920,002	2,867,768	2,920,003	1,846,390	2,664,359
Mid-Grade Tonnage	tonnes	1,779,102	265,000	326,842	313,201	227,995	255,274	138,828	251,964
High-Grade Tonnage	tonnes	3,167,241	602,008	635,610	464,650	512,800	356,911	167,769	427,493
Mineralized Tonnage	tonnes	23,564,672	3,643,655	3,585,611	3,697,853	3,608,564	3,532,187	2,152,987	3,343,816
Total Tonnage	tonnes	115,128,727	21,993,659	22,235,628	15,697,872	15,608,567	14,882,289	15,653,001	9,057,702
GRADE AU									
Low-Grade Grade	g/t Au	0.36	0.35	0.35	0.36	0.35	0.38	0.36	0.37
Mid-Grade Grade	g/t Au	0.87	0.87	0.88	0.88	0.87	0.87	0.86	0.88
High-Grade Grade	g/t Au	2.12	2.92	2.38	1.71	1.92	1.76	1.52	1.84
GRADE AG									
Low-Grade Grade	g/t Ag	1.67	1.62	1.97	1.66	1.42	1.63	1.61	1.79
Mid-Grade Grade	g/t Ag	2.31	2.95	2.11	1.97	2.27	2.08	1.95	2.78
High-Grade Grade	g/t Ag	3.80	5.90	3.48	2.48	3.42	3.20	2.75	4.12
ADJUSTED MILL FEED									
Mill Tonnes	tonnes	4,518,238	730,000	730,000	730,000	730,000	612,184	306,597	679,457
Mill Grade Au	g/t Au	1.75	2.56	2.18	1.41	1.60	1.39	1.22	1.48
Mill Grade Ag	g/t Ag	3.35	5.39	3.30	2.29	3.08	2.73	2.39	3.62
ADJUSTED HEAP LEACH FEED									
Heap Leach Tonnes	tonnes	19,046,434	2,913,655	2,855,611	2,920,000	2,920,000	2,920,000	1,852,809	2,664,359
Heap Leach Grade Au	g/t Au	0.37	0.37	0.40	0.36	0.35	0.38	0.36	0.37
Heap Leach Grade Ag	g/t Ag	1.69	1.68	1.98	1.67	1.43	1.63	1.61	1.79
Processed Au	oz Au	480,918	95,087	87,646	67,256	70,275	62,664	33,488	64,501
Processed Ag	oz Ag	1,518,496	284,117	259,446	210,454	206,309	206,515	119,395	232,259
Strip Ratio	waste: mineral	3.9	5.0	5.2	3.2	3.3	3.2	6.3	1.7

16.7 Modifying Factors

The following modifying factors were applied or considered during the development of the pit optimization, mine design, and LoMP:

- A mining recovery of 100% and 0% dilution were assumed in the pit optimization and scheduling. This is considered appropriate for a PEA-level study (see Item 16.1 and Table 16.4).
- Au recovery was assumed at 92% for the mill and 75% for the heap leach, based on metallurgical test work available at the time. These values were used in the COG and routing logic. Ag is treated as a credit in the financial model but was not included in pit optimization or scheduling. A more refined heap leach recovery model, differentiated by lithology and grade bin, was developed by KCA and applied in the financial model (Item 22.0). While more accurate, it does not affect the mine plan.
- COGs were calculated at an Au price of US$2,150/oz using the cost and recovery assumptions in Item 16.6.1. The resulting values were 0.18 g/t Au for heap leach, 0.28 g/t Au for mill feed, and 0.76 g/t Au as the break-even grade between the two processing routes.
- Rehandling was limited to approximately 40 kt of low-grade material, representing less than 1% of total heap leach feed. This material was stockpiled and later re-fed to the heap leach. Rehandling cost was included in the heap leach margin calculation.
- A minimum spacing of 35 m was maintained between pit phases to accommodate CAT 777G-class equipment. Ramp geometry and road design followed the criteria in Table 16.2 and Table 16.8. The main exit ramp is located on the northeast side of the pit, providing direct access to the plant and WRSF.
- Processing capacities of 730 kt/year for the mill and 2,920 kt/year for the heap leach were used in the schedule and treated as nameplate throughout the LoM.
- The state road on the southwest side of the deposit remains in place in the Base Case and was treated as a fixed pit limit. It has minimal impact on the current resource envelope but could be relocated if future resource expansion justifies it.
- No factors related to permitting, environmental offsets, or surface disturbance constraints were applied at this stage. These will be addressed in future study phases.

16.8 Equipment Fleet

The Tonopah Project has been planned assuming an owner-operated mining fleet sized to meet production requirements while maintaining operational flexibility, controlling costs, and ensuring effective mineralized material control throughout the LoMP. Owner operation is considered the preferred approach in this region due to the high availability of skilled equipment operators in southwestern Nevada and the broader US Southwest. Contract mining is not cost competitive in this context, and more critically, contractors typically do not maintain the level of mineralized material control required for a deposit of this type. Mineralized material control is expected to be a key factor in the successful execution of the mine plan, and maintaining this function under direct site supervision is important.

Primary loading operations will be carried out using three CAT 992 front-end loaders. Material hauling will be performed with CAT 777 haul trucks, with a maximum of 11 units required in Year 2 and between 8 and 10 units operating in subsequent years, depending on annual mining volume and haul distances. Production and pre-split drilling will be performed with three CAT MD6250 drill rigs.

The ancillary fleet includes the following:

- Two CAT D9 dozers: one assigned to the active loading areas as a floating unit and one dedicated to the waste dump and general mine support.
- One CAT 16M grader for haul road maintenance.
- One CAT 773 water truck for pit and haul road dust suppression, supported by one smaller 1,000-gallon water truck for localized control.
- Two utility mechanical trucks and one fuel/lube service truck for field support.
- One CAT 352 excavator assigned to ditch clearing, wall cleaning, and other general pit maintenance tasks.
- Light-duty vehicles, including pick-up trucks and other utility equipment for operations, supervision, and support functions.

The fleet is sized to match the production profile described in Item 16.6 and has been selected based on equipment class commonly used in similar open-pit Au operations in the region.

17.0 Recovery Methods

This Item contains forward-looking information related to Handling and processing methods, plant design and equipment selection, and processing rates and recoveries for the Project. The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts or projections in the forward-looking information include any significant differences from one or more of the following material factors or assumptions that were applied in drawing the conclusions or making the estimates, designs, forecasts or projections set forth in this Item: Plant feed characteristics and rate, mineral processing flowsheet, equipment selection and plant design, metals recovery factors.

17.1 Process Design Basis

Test work developed by KCA and Viva and carried out by KCA and MLI in Reno, Nevada has indicated that the potentially minable resource for the Tonopah Gold Project is amenable to gravity concentration and cyanide leaching for the recovery of Au and Ag values.

The process considers processing 10,000 tpd of mineralized Run-of-Mine (ROM) material, including 8,000 tpd of low-grade and 2,000 tpd of high-grade material. Mineralized material will be crushed to 100% passing 12.5 mm using a three-stage closed crushing circuit. High-grade and low-grade material will be campaigned through the crushing circuit and stockpiled separately using a radial stacking conveyor. Low-grade material will be reclaimed from the low-grade stockpile and agglomerated with cement, which will also act as a pH buffer, before being conveyor stacked in 10 m lifts onto a permanent geomembrane-lined heap leach pad and leached with a dilute cyanide solution. The resulting pregnant leach solution will flow by gravity to a pregnant solution tank before being pumped to a carbon adsorption circuit. Au values will be loaded onto activated carbon and then periodically transported off site to be toll-processed where the loaded carbon will be stripped and regenerated before being returned to the project for re-use. Barren solution leaving the adsorption circuit will flow by gravity to a barren solution tank before being pumped back to the heap.

High-grade material will be reclaimed and ground to 80% passing 150 Mesh (106 µm) in a single stage ball mill which will operate in a closed circuit. Lime will be added to the high-grade material for pH control before being fed to the ball mill along with process solution. Ball mill discharge will be pumped to a hydrocyclone cluster for classification with a portion of the cyclone underflow being diverted to a gravity concentrator for the recovery of coarse metal with the remaining cyclone underflow being returned to the ball mill feed. Cyclone overflow material will be thickened before reporting to a six-stage CIL circuit where the thickened slurry will be mixed counter-flow with activated carbon and will flow from one stage to the next through carbon interstage screens. NaCN will be added to the first three CIL tanks as needed. Leached slurry leaving the last tank will be discharged as tailings to a tailings thickener before being pumped to a filter feed tank, filtered using a filter press, and dry-stacked using trucks onto a dedicated portion of the heap leach pad. Loaded carbon from the first tank of the CIL will be toll-processed along with carbon from the heap leach circuit.

A summary of the processing design criteria is presented in Table 17.1.

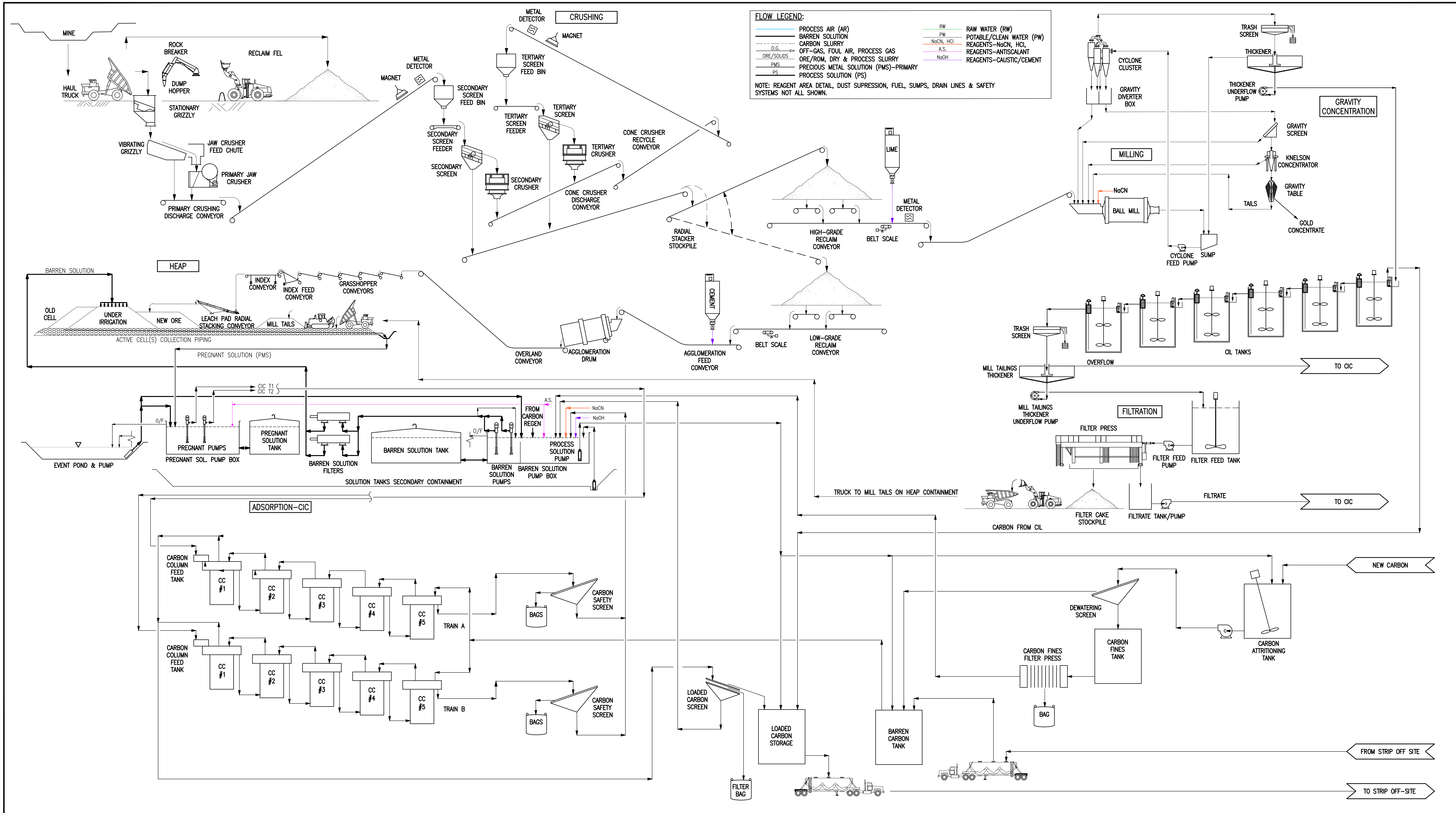
Table 17.1: Processing Design Criteria Summary

Item	Design Criteria
Annual Tonnage Processed	3,650,000 tonnes
Grade	
High-grade Average Au	2.1 g/t (0.074 opt)
Low-grade Average Au	0.44 g/t (0.016 opt)
Production Rate	
Crushing Rate	10,000 tonnes/day (365 days/year)
CIL Milling Rate	2,000 tonnes/day (365 days/year)
Leach Pad Stacking Rate	8,000 tonnes/day (365 days/year)
Recovery	
High-grade Au Recovery, Average	93%
Low-grade Au Recovery, Average	75%
Operation	12 hours/shift, 2 shifts/day, 7days/week, 365 days/year
Leach Cycle	120 days
Reagents	
High-grade NaCN Consumption, kg/t	0.58
Low-grade NaCN Consumption, kg/t	0.26
High-grade CaO Addition, kg/t	0.6
Low-grade Cement Addition, kg/t	4

The barren and pregnant solution tanks will be located within the same pond containment which will be maintained at an empty level during operation. An event pond will be included to contain seasonal accumulations of leach solutions and/or upset conditions that cannot be managed during normal operations. Event solution will be returned to the barren tank as makeup solution as soon as practical.

The simplified process flowsheet is presented in Figure 17.1. The overall site layout is presented in Figure 17.2.

All of the selected processes and equipment are established technologies used in Au processing plants.



REFERENCE DRAWINGS	
DWG. NO.	TITLE

REVISIONS						
NO.	BY	CK'D	APPROVED		DATE	DESCRIPTION

REVISIONS						
NO.	BY	CK'D	APPROVED		DATE	DESCRIPTION
A	DSG	AJH	TDS		21SEP23	ISSUED FOR REVIEW
B	DSG	AJH	TDS		12OCT23	ISSUED FOR REVIEW
C	SDR	SDR	CC		19JUL24	UPDATED, ISSUED FOR REVIEW
D	WCC	JDW	CC		14MAY25	UPDATED TO INCLUDE MILL, ISSUED FOR REVIEW

DESIGNED BY	AJH/TDS
DRAWN BY	DSG
CHECKED BY	AJH
APPROVED 1	TDS
APPROVED 2	
CLIENT APP'L	

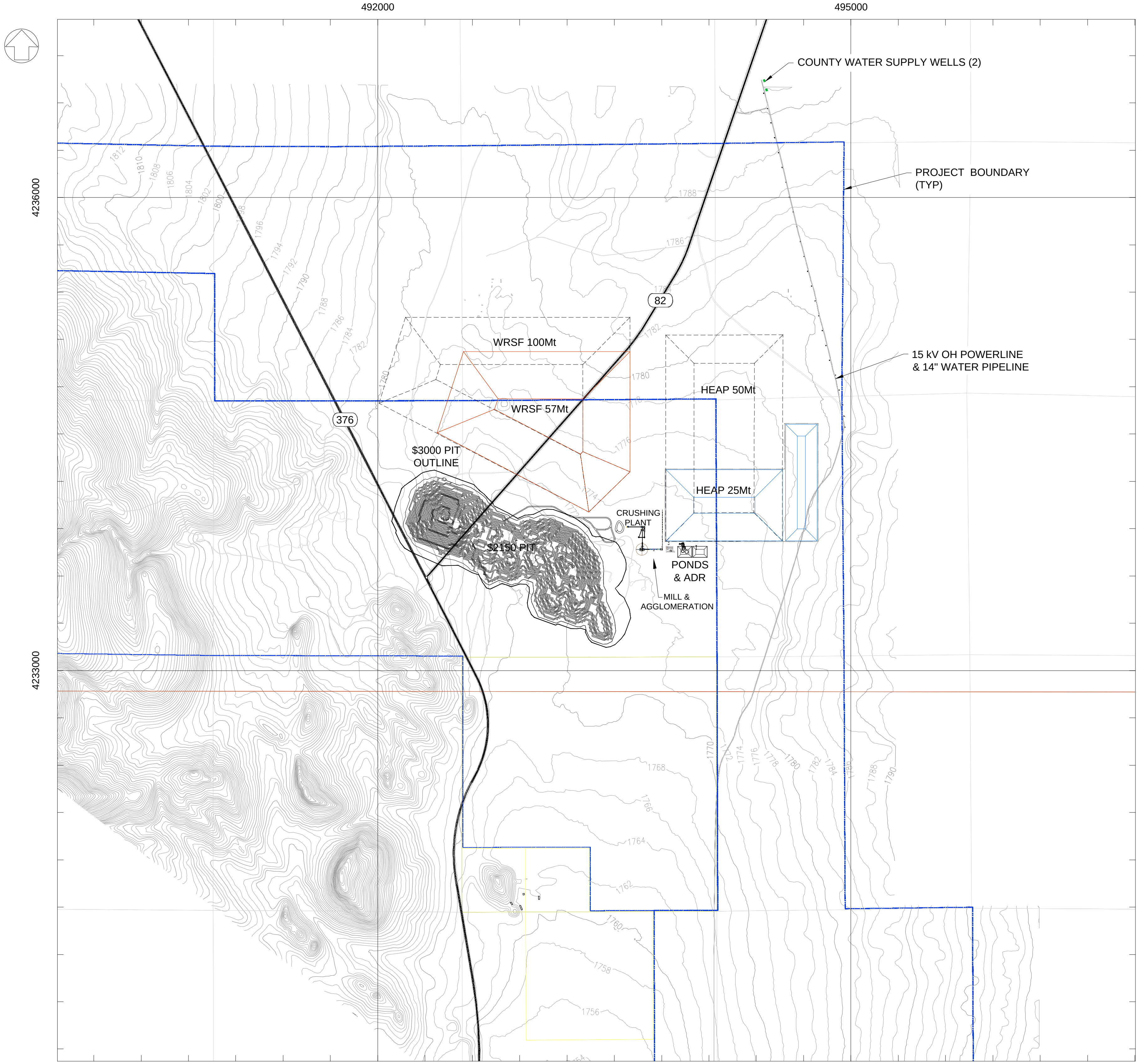
PREPARED FOR

VIVA GOLD CORP

PREPARED BY

Kappes, Cassiday & Associates
Reno, Nevada
KCA

Project:	TONOPAH HEAP LEACH PROJECT	
Title:	FLOWSHEET SIMPLIFIED OVERALL	
Job No. 9431H	Scale N.T.S.	ACAD Filename: 9431-0110-11-100.DWG
9431	0110-11-100	REV D



PLAN VIEW: OVERALL SITE LAYOUT (PRELIMINARY)

SCALE 1:15,000

PRELIMINARY
NOT FOR CONSTRUCTION

0 500 1000 Meters

CONTOUR INTERVALS:
PROPOSED: 5m, PIT = 100m
EXISTING: 2m MINOR,10m MAJOR

REFERENCE DRAWINGS

DWG. NO.	TITLE

REVISIONS

NO.	BY	CK'D	APPROVED		DATE	DESCRIPTION

REVISIONS

NO.	BY	CK'D	APPROVED		DATE	DESCRIPTION
A	DSG	AJH	TDS		03OCT23	ISSUED FOR REVIEW
B	DSG	AJH	TDS		09OCT23	ISSUED FOR REVIEW
C	DSG	AJH	TDS		27MAR25	ISSUED FOR REVIEW
D	DSG	AJH	TDS		28APR25	ISSUED FOR REVIEW
E	DSG	AJH	TDS		15MAY25	ISSUED FOR REVIEW

DESIGNED BY AJH
DRAWN BY DSG
CHECKED BY AJH
APPROVED 1 TDS
APPROVED 2
CLIENT APP'L

PREPARED FOR
VIVA GOLD CORP

PREPARED BY
Kappes, Cassiday & Associates
Reno, Nevada
KCA

Project: **TONOPAH HEAP LEACH PROJECT**

Title: GENERAL ARRANGEMENT
OVERALL SITE PLAN
PRELIMINARY
1:15,000

Job No.	Scale	ACAD Filename:
9431H	AS SHOWN	9431-0000-13-100.DWG

9431 **0000-13-100** REV E

NAME: J:\DRAWINGS\Active Projects\Tonopah Viva\DWG-WORKING\13-GENERAL ARRANGEMENTS\9431-0000-13-100.dwg LAST UPDATE: 15-May-25, 4:46 PM PLOT DATE:15-May-25 BY: PROSALES PLOT SCALE: 1:3

17.2 Crushing

The crushing circuit for high-grade and low-grade material has been designed as a three-stage crushing system with the third (tertiary) stage operating in closed circuit, and an overall availability of 75%. The high- and low-grade feed material will be campaigned as required, utilizing a shared radial stacker and separate product stockpiles each equipped with reclaim tunnels and conveyors.

ROM material will be transported from the mine in surface haul trucks to the primary crusher area. The material will be stockpiled and fed to the primary crushing dump hopper either by direct dump from 100-ton trucks, or front-end loader.

While campaigning one or the other materials, excess deliveries will be stockpiled ahead of the primary crusher in the respective high and low-grade stockpiles. The combined ROM stockpiles are each sized for 5 days of mining, or 50,000 tonnes. The crushing plant will process an average of 417 tonnes of material per h.

The primary crushing module will include a stationary grizzle, primary crusher feed bin, vibrating grizzly feeder, primary jaw crusher, and under-crusher conveyor. Material from the primary dump hopper will be fed to the jaw crusher via the vibrating grizzly feeder. Minus 90 mm undersize from the vibrating grizzly feeder will bypass the jaw crusher while plus 90 mm oversize material will feed into the jaw crusher. Undersize from the vibrating grizzly feeder and primary crusher discharge will fall onto the primary crusher discharge conveyor, which in turn will discharge onto the secondary screen feed bin feed conveyor. The primary crushed product size is expected to be approximately 80% passing 144 mm.

The secondary crushing module will include a secondary screen feed bin, double-deck inclined vibrating screen, 373 kilowatts (kW) (500 horsepower [hp]) standard cone crusher with a 33 mm closed side setting, which will discharge to the common cone crusher discharge conveyor (shared with the tertiary crusher). The secondary screen oversize (+13 mm) will feed the secondary crusher, and undersize will report to the product conveyor. The secondary crushed product size is expected to be approximately 80% passing 38 mm.

The closed-circuit tertiary crushing unit will include a tertiary screen feed bin, double deck vibrating screen, two 373 kW (500 hp) short head cone crushes, and cone crusher recycle conveyors. The tertiary crushers discharge together with the secondary crusher discharge will be collected on the common cone crusher discharge conveyor and will be recycled back to the tertiary screen feed bin via the cone crusher recycle conveyors. The tertiary screen oversize (+13 mm) will drop into the tertiary cone crushers with a 13 mm closed side setting, which will discharge to the cone crusher discharge conveyor. Tertiary screen undersize material will discharge onto the product conveyor, which will be fed to a radial stacker along with the secondary screen undersize material.

The final crushed product size will be 100% passing 12.5 mm. The crushed material will be stockpiled using a radial stacker in either the high-grade crushed product stockpile or the low-grade crushed product stockpile. A modular control center will be located proximal to the primary crusher dump hopper. All of the equipment and conveyors will be interlocked so that if one trips out, all upstream equipment will also trip. This interlocking will prevent large spills and equipment damage and is considered necessary to meet the design utilization for the system.

Water sprays will be located at all material transfer points to reduce dust generation by the crushing circuit.

17.3 Crushed Material Reclaim

The high-grade and low-grade crushed product stockpiles have been sized for 12 h of live capacity (1,000 tonnes for high-grade and 4,000 tonnes for low-grade). A dozer will be used if required to maintain feed to the reclaim system.

Both stockpiles will have separate under-pile reclaim systems with a reclaim tunnel, belt feeders, and a reclaim conveyor to reclaim the crushed material from the stockpile. Both stockpiles will be equipped with two belt feeders (one operating, one standby) which will feed either high-grade material onto the ball mill feed conveyor or low-grade material onto the agglomeration feed conveyor.

The high-grade reclaim conveyor will be equipped with a belt scale and metal detector. Pebble quicklime for pH control will be metered onto the reclaim conveyor from a 100-ton storage silo at a nominal rate of 0.6 kg/t before discharging to the ball mill feed conveyor.

The low-grade reclaim conveyor will be equipped with a belt scale and will discharge to the agglomeration feed conveyor. Cement for pH control and heap permeability/stability will be metered onto the agglomeration feed conveyor from a 200-ton storage silo at a nominal rate of 4 kg/t before discharging into an agglomeration drum.

17.4 High-Grade Material Processing

17.4.1 Grinding & Gravity Concentration

The grinding circuit has been designed as a single-stage closed circuit system with hydrocyclones to grind an average of 2,000 tpd of crushed high-grade material at an average rate of 83 tonnes per h (tph) from 100% passing 12.5 mm to 80% passing 106 µm (150 mesh), with an overall availability of 90% and a 300% recirculating load. A gravity concentration circuit will be coupled with the grinding circuit to recover the coarse Au and Ag present in the high-grade material.

Crushed high-grade material will be fed to the ball mill by the ball mill feed conveyor where it will be combined with process solution to make a slurry of 70% solids by weight. The slurry will discharge from the ball mill through a trommel to the cyclone feed pump box. Process solution will be added to the cyclone feed pump box to adjust the slurry density. The slurry in the cyclone feed pump box will be pumped to the hydrocyclone cluster by the cyclone feed pumps (one operating, one standby).

A fixed amount of the recirculating load (slurry from the cyclone underflow) will be diverted to the gravity concentration circuit by the gravity diverter box. Slurry fed to the gravity concentration circuit will first pass through a vibrating screen to separate the oversize and undersize material. Oversize material (+1.68 mm,12 mesh) will be returned to the ball mill feed and undersize material (-1.68 mm) will discharge to a Knelson Concentrator. The screen will be rinsed with process solution to prevent plugging.

The Knelson Concentrator will concentrate coarse Au and Ag from the slurry. The concentrate will be periodically discharged in batches to a concentrate storage cone. The concentrate will be fed in daily batches to an automated tabling system to produce a clean Au concentrate suitable for quick drying and direct smelting. Gravity concentration tailings will be pumped back to the ball mill feed.

Slurry from the cyclone feed pump box will be pumped by the cyclone feed pumps to the hydrocyclone cluster for classification. Oversize particles will report to the cyclone underflow and will be recycled to the ball mill feed for further size reduction. The cyclone overflow grinding product (80% passing 106 µm) will pass through a vibrating trash screen to remove any tramp material and then will discharge to the grinding thickener feed mix box.

17.4.2 Grinding Thickener & Process Solution

Flocculant will be added to the cyclone overflow slurry at a rate of 50 g flocculant per tonne of mineralized material in the thickener feed mix box before discharging to the center feed well of a high rate grinding thickener. The overflow slurry will be thickened to 50% solids by weight with the underflow pumped to the CIL circuit via the thickener underflow pumps (1 operating, 1 standby), and the overflow discharging to the cyclone feed sump.

17.4.3 CIL

Grinding thickener underflow at 50% solids by weight will be pumped to the first of six identically sized CIL tanks which will operate in series. Each tank will provide eight hours residence time for a total of 48 h at the nominal rate of 92.2 tph. The leach tanks will be fitted with double axial flow impeller agitators. Compressed air will be injected into each tank using a “down shaft” aeration system. NaCN solution will be added to any of the first three tanks in series.

The CIL tanks will be maintained at a concentration of 15 g/L of activated carbon. As soon as the Au and Ag are extracted, they will be adsorbed onto the carbon. The carbon will be advanced counter-current to the slurry. Each of the tanks will be equipped with an interstage pump screen to prevent the carbon from flowing from tank to tank with the slurry. Carbon advance will take place approximately 3 h per day from each tank. Carbon advanced from the first CIL tank, loaded with precious metals up to approximately 5,000 g Au and Ag per tonne carbon, will be directed to the recovery screen where the carbon will be separated from slurry and rinsed with process solution. The rinsed carbon will be recovered as oversize into the loaded carbon tank, and the undersize slurry will be returned to the first CIL tank. A tanker truck will transport the loaded carbon off site for toll-processing.

Slurry overflowing from the last CIL tank will discharge via a carbon safety screen into the tailings thickener feed mix tank.

17.4.4 Tailings Thickener

Flocculant will be added to the CIL discharge slurry at a rate of 50 g flocculant per tonne of mineralized material in the thickener feed mix tank before being discharged to the center feed well of a high-rate tailings thickener. The CIL discharge slurry will be thickened to 55% solids by weight with the underflow pumped via the thickener underflow pumps (1 operating, 1 standby) to the agitated filter feed tank. The thickener overflow will discharge to the tailings thickener overflow tank where it will be pumped to the heap leach Carbon-in-Column (CIC) circuit using a thickener overflow pump. Tailings thickener overflow will also be used as make-up solution to the grinding circuit.

17.4.5 Tailings Filtration

Tailings thickener underflow slurry will be pumped to the agitated filter feed tank, which has been sized for 6 h of retention at the nominal flow. The slurry will be fed to the tailings filters by the filter feed pumps to remove most of the remaining moisture from the thickened tailings slurry. Each filter will have a dedicated filter feed pump. The filtration circuit consists of two recessed plate filter presses which have been sized to produce a final filter cake that will be at least 85% solids by weight.

After completion of a filter cycle, the plates will be opened and the cake will be discharged to a collection area followed by a cloth rinse cycle using raw water via the rinse pumps (1 operating, 1 standby). The filter cloths will be washed with raw water using a high-pressure cloth wash system once per day or as needed.

Filter cake in the collection area will be reclaimed using a front-end loader and loaded into trucks to be transported to a designated area of the heap leach pad for final disposal. A dedicated dozer will be used to spread the tailings material on the heap followed by compaction.

The filtrate and cloth rinse water will flow to the filtrate tank where it will be pumped via the filtrate pump to the heap CIC circuit.

17.5 Low-Grade Material Processing

17.5.1 Agglomeration and Conveyor Stacking

Agglomeration of the low-grade material with cement will be accomplished using one agglomeration drum. The agglomeration drum will mix the crushed material and cement with barren process solution to produce agglomerates. The cement will bind fine particles to coarser particles which will increase the permeability and stability of material stacked onto the heap. The agglomeration drum will be installed over a lined concrete containment area to contain any process solution or material contacted by process solution in the event of a spill or other disruption.

Agglomerated material from the agglomeration drum will discharge to a series of overland conveyors which will transfer feed to the conveyor stacking system. The final overland conveyor will be equipped with a tripper conveyor which will feed the material to the mobile conveyor stacking system at the active stacking site of the heap. The mobile conveyor stacking system will consist of 13 each grasshopper ramp conveyors, an index feed conveyor, a horizontal index conveyor, and a radial stacker. A tripper on the overland conveyor feeds material to the grasshopper conveyors at the active stacking site of the heap. The horizontal index conveyor and radial stacker are able to retreat and stack material onto the heap. The number of grasshopper conveyors required will vary depending on the area of the heap being stacked with a maximum of 13 grasshopper conveyors being required.

Each of the grasshopper and stacking conveyors will include an onboard transformer and interlocked Programmable Logic Controller (PLC) to allow for the removal or addition of conveyors. The master PLC will be installed at the radial stacker for initiating the conveyor start sequence. Each of the stacking system conveyors will include a strobe and horn alarm which will sound before the equipment starts up. Movement for the radial stacker and horizontal index conveyor will be controlled manually at the equipment. Each conveyor will be equipped with pull-cords and emergency stops. If one conveyor in the stacking line is tripped, all upstream conveyors will also stop.

Once a lift of cells has finished leaching and is sufficiently drained, a new lift can be stacked over the top of the old lift. The old lift will be cross-ripped prior to stacking new material on top to break up any compacted or cemented sections. Stacked lifts will progress in a stair-step manner with a planned heap height of approximately 70 m.

17.6 Heap Leach Pad Design

The heap leach pad will be a single-use, multi-lift type leach pad which will also accommodate the filtered mill tailings material with a combined total capacity of 25 Mt and has been designed to allow for the development of future resources. The heap has been designed with a maximum height of 70 m with an overall slope of 2.5:1 (height: vertical).

The heap lining system has been designed in accordance with International Cyanide Code requirements and meets or exceeds the NDEP requirements.

The leach pad will be constructed by clearing the pad area by stripping vegetation and stockpiling growth media. Only minor grading of the leach pad area will e required as the natural topography is within the required range for solution drainage and stability.

The leach pad liner will be composed of the following lining system from top to bottom:

- 0.6 m (minimum) gravel overliner (mine waste or sub-grade mineralized material).
- 80 mm Linear Low-Density Polyethylene (LLDPE) geomembrane liner.
- 300 mm of compacted low-permeability soil or clay liner with a permeability of 1x10-6 cm/sec
- Leak detection system under the primary solution collection pipes which will route solution to a collection area
- Prepared subgrade.

The overliner cover will act as a protective layer that resides above the LLDPE geomembrane. The main purpose of this material will be to protect the geomembrane layer and collection piping from damage during material placement.

Gravity solution collection pipes will be installed on top of the geomembrane liner and covered with overliner material. The pipes are sized to operate at 50% full to contain the design production flows from the upgradient tributary area, allowing additional capacity to accommodate excess solution from storm events.

The gravity solution collection pipes will consist of 100 mm diameter perforated corrugated polyethylene (PCPE) tertiary pipes spaced on 8 m centres flowing into larger double walled PCPE secondary and primary pipes of up to 450 mm in diameter. The primary solution collection pipes will exit the heap through a concrete weir to the solution collection channel. The pipes will be solid walled as they enter the solution collection channel that flows to the pregnant solution tank.

Should solution flows exceed the capacity of the heap outlet pipes, solution head will build at the leach pad discharge area, causing excess solution to overflow the concrete weir into the solution collection channel.

The leak detection system will consist of 50 mm perforated Polyvinyl Chloride (PVC) pipe which will be installed under the main solution collection pipes. The leak detection pipes will discharge outside of the heap perimeter berm within the solution collection channel area. At the perimeter berm the perforated PVC pipe will transition to solid pipe and will pass through a 1000 mm bentonite plug to ensure solutions are contained. The leak detection pipes will be checked daily to ensure no leaks are present.

17.7 Heap Leaching Solution Handling & Storage

The heap leach solution storage and management system for the Tonopah Gold project includes a barren solution tank, pregnant solution tank and event pond. The pregnant solution and barren solution tanks will be constructed within a common pond for containment, which will be maintained at empty levels.

Crushed and agglomerated material will be irrigated using drip tubes spaced at 0.91 m (3 ft) intervals. Reusable Yelomine or High-Density Polyethylene (HDPE) pipes will be used to distribute the solution to the drip tubes on top of the heap. Antiscalant will be added at the barren and pregnant solution tanks to reduce the potential for scaling problems within the system.

The total leach cycle for the material is estimated to be 120 days. Leach solutions will be applied to the material at a nominal application rate of 10L/h/m^2 (0.004 gallons per minute [gpm]/ft^2) with an approximate maximum cyanide concentration of 250-300 PPM to the heap. Vertical turbine pumps mounted on a pump box connected to the barren tank will be used for solution application to the heap leach. High-strength cyanide and antiscalant will be added to the pump box by metering pumps. The nominal flow for barren solution to the stacked material is estimated at 640 m^3/h.

Au bearing solutions draining from the leach pad will be collected at the bottom of the heap by a network of perforated drainage pipes and directed to the pregnant solution tank where it will be pumped to the CIC circuit using vertical turbine pumps mounted on a pump box connected to the pregnant solution tank.

The event pond and pregnant and barren tank containment pond will each have a designed capacity of about 7.4 M gallons (28 M L) and have been sized to accommodate the 24-h heap drain down plus the 100-year, 24 h storm even over the full lined area and 24-h working volume and 24-h heap drain down, respectively.

The ponds will have a composite lining system with a leak detection system and will be composed of the following from top to bottom:

- 80-mm textured HDPE geomembrane liner.
- HDPE geonet leak detection layer.
- 60-mm textured HDPE geomembrane liner.
- Geosynthetic Clay Liner (GCL).
- Prepared subgrade.

Leak detection will be utilized in the ponds. Leak detection riser pipes will be provided beneath the primary and secondary liners of the ponds and will allow for monitoring and pumping of solutions from within the leak detection sumps if the liner is found to be leaking.

The leak detection risers will include the use of a 12-inch (30.5 cm) diameter Standard Dimension Ratio (SDR)-32 HDPE pipe that will run down the slope of the pond into a sump between the primary and secondary liners. The sump will be filled with drain gravel that will be free-draining and porous in nature. The 12-inch (30.5 cm) diameter pipe will be perforated within the sump to accumulate fluids, if they exist in the sump. The non-perforated portion of the pipe will boot through the anchor trench and terminate at the crest of the pond.

Submersible pumps on pump slides in the ponds will be used to return collected solution to the system.

Solution management for the system will be generally simple. The pregnant and barren solution tanks will be operated approximately half full to allow for some surge capacity within the tanks. The event pond is intended for emergency solution storage only and will be kept empty or at low levels whenever possible. When solution is diverted to the event pond, it should be pumped back to the leach system as soon as practical. Every effort will be made to avoid storing solution in the event pond over an extended period of time.

The solution storage tanks and event pond have been sized to ensure that solution discharge will not be required and that all the leach solutions can be managed in a controlled manner. In the event of a significant upset condition resulting in the pregnant and barren tank containment pond filling with solution, valves on the tank will automatically open allowing the volume of solution in the tank and ponds to equilibrate so the tanks will not float.

17.8 Process Water Balance

A monthly water balance model was developed for the heap leach operation. The model was used to evaluate the annual heap leach facility performance, evaluate the event pond complex utilization, and estimate average make-up water flowrates. The water balance simulation model is incremented on a monthly basis to reflect seasonal variations in precipitation and evaporation. The model has been developed using an Excel spreadsheet, with versions for low, average, and high precipitation years.

The model considers the discrete increments of lined area as the leach pad phases are constructed throughout the life of the project, the increment of areas covered with fresh material, and the areas under irrigation.

The water balance model is based upon the leach facility operating for 365 days a year. Inflow to the system consists of precipitation on all lined areas, including all leach pad areas and collection ponds, applied leach solution, and moisture in the material as stacked on the pad. Outflow from the system consists of evaporation from the lined areas and heaps, and solution flow to the collection ponds. Moisture absorbed by material stacked onto the heap from the as-stacked condition to its field capacity is included as a loss.

17.8.1 Climate Data

Water balance computations included different precipitation datasets, including considerations for above and below average precipitation to estimate the system accumulation of process solution and the makeup water required. The wet year reflects the most water likely to affect the system, and the dry year will indicate the most makeup water needed.

The monthly rainfall data for the site was estimated from two sets of precipitation data obtained from two nearby stations: Tonopah (268160, period of record 5/1/1902 - 6/9/2016), and Tonopah Airport (268170, period of record 6/11/1954 - 06/09/2016). The Tonopah station is located approximately 24 km (15 miles) southwest from the mine site. The Tonopah Airport station is located approximately 21 km (13 miles) south from the mine site. The site rainfall data was obtained utilizing the arithmetic average method, determining the precipitation profile for the average, the driest and wettest years, as shown in Table 17.2.

Table 17.2: Rainfall Data Estimated for the Site

Month	Mean (mm)	Wettest Year 1978 (mm)	Driest Year 1927 (mm)
January	9.0	18.0	3.8
February	10.5	39.4	2.0
March	13.1	60.5	10.9
April	12.6	44.2	0.0
May	10.9	0.0	0.0
June	6.5	0.0	0.0
July	11.0	3.8	1.3
August	13.1	7.9	0.0
September	9.0	52.1	3.3
October	11.9	19.3	22.9
November	9.3	13.5	0.3
December	8.6	11.7	4.3
Annual	**125.6**	**270.3**	**48.8**

Monthly average pan evaporation data was obtained from Silver Peak station, which is located approximately 74 km (46 miles) southwest from the Project site, shown in Table 17.3. For the purposes of this model, monthly pan evaporation data was not varied for wet and dry years in the above cases. Instead, the evaporation from the heap is limited to the lesser of the amount of precipitation that has fallen in that period, or the factored pan evaporation. Pan evaporation assumes water is available for evaporation, but in dry months only the amount of precipitation is available.

Table 17.3: Pan Evaporation Data

Month	Pan Evaporation (mm)
January	0.0
February	97.5
March	184.4
April	257.3
May	345.4
June	414.3
July	456.7
August	404.4
September	287.5
October	174.8
November	74.7
December	0.0
Annual	**2,697.0**

The water balance model does not account for other climatological factors such as wind speed, humidity and cloud cover.

17.8.2 Water Balance

The model shows that the normal operation of the facility is characterized by a regular deficit on the water balance mainly due to the climate characteristics such as high evaporation and low rainfall throughout the years. During dry to average years, the event pond is expected to be utilized at a very minimum with low chance of accumulating water. During wet years some accumulation of water is expected in the event pond during the wettest months.

Water deficit will be balanced with the addition of make-up raw water supplied from nearby water wells. The flowrate requirement for the heap leach will vary month to month depending mainly on the climate patterns, with average flowrates ranging from 39 m^3/h to 102 m^3/h (170 to 450 gpm) in dry years. Table 17.4 shows the summary of estimated average and maximum flowrates required for make-up water for each of the three scenarios modelled. Note these make-up requirements are for the heap leach only, and do not include other requirements such as road dust suppression, and miscellaneous building and infrastructure use.

Table 17.4: Estimated Flow Rates for Make-up Water

	Average Year (m^3/h)	Driest Year (m^3/h)	Wettest Year (m^3/h)
Average	39	102	32
Maximum	59	132	56

17.9 Recovery Plant

The recovery plant at the Tonopah Gold Project has been designed to recover Au and Ag values onto carbon using a CIC and a CIL process. Metal values will be recovered from carbon via toll-stripping off site.

Loaded carbon from the CIC and CIL circuits will be collected in a storage tank and transferred to trucks which will deliver the loaded carbon to the toll-strip facility. Barren carbon will be returned from the toll-strip facility and stored in a storage tank before it is added back to the CIC or CIL circuits with new carbon being added as required.

17.9.1 CIC Adsorption

Adsorption of Au and Ag from pregnant solution of the heap leach onto carbon will occur in the CICs.

The adsorption circuit will consist of two trains of five, cascade type open-top up-flow mild-steel CICs. Each of the carbon columns will be sized to hold 3 tonnes of carbon.

The two column trains will be operated in parallel. Pregnant solution from the pregnant solution pump box will be pumped to the adsorption circuit for each column train at a rate of 320 m^3/h, for a total of approximately 640 m^3/h. Each train will be supplied by a separate pump from the pregnant solution pump box. Each column train will be equipped with a static safety screen to separate any floating carbon from barren solution leaving the columns, then the solution will flow by gravity back to the barren solution tank or pump box.

Antiscalant will be added at the pregnant solution pump box to prevent scaling of carbon and reduction of the carbon loading capability. Magnetic flowmeters equipped with totalizers will measure solution flow to the adsorption circuit. Pregnant solution will flow by gravity through each set of five columns in series, exiting the lowest column as barren solution. Pregnant and barren solution continuous samplers will be installed at the feed and discharge end of each carbon column train, respectively. Solution samples will be used to measure pregnant and barren solution Au concentrations.

Adsorption of Au and Ag from pregnant leach solutions from the heap circuit will be a continuous process. Once the carbon in the lead column achieves the desired precious metal load it will be advanced to loaded carbon tank using screw type centrifugal pumps. Carbon in the remaining columns will be advanced counter current to the solution flow to the next column in series. New or acid washed/regenerated carbon will be added to the last column in the train.

Generally, the transfer of carbon will occur once each day.

17.9.2 Carbon Handling & Storage

The carbon handling includes all equipment required to store, prepare and transfer carbon. All loaded carbon will be transported by trucks off site for recovery of metal values and regeneration of the carbon before being returned and added back to the system.

New and returned carbon from off-site processing will be stored in the barren carbon storage tank and transferred to the adsorption or CIL circuits as makeup carbon. Carbon being transferred to the barren carbon storage tank will first pass through a carbon sizing screen to remove any carbon fines from the system. Carbon fines will be stored in a carbon fines storage tank, which will be periodically pumped through the carbon fines filter press; carbon fines from the filter press will be stored in bulk bags for removal from the system.

New carbon being added to the system will first be attritioned in the carbon attritioning tank before being pumped to the carbon sizing screen to remove carbon fines and then transferred to the barren carbon storage tank.

Loaded carbon from the lead CIC column from each column train will be pumped to the loaded carbon screen which will dewater and separate carbon into +0.85 mm or -0.85 mm (20 mesh) size fractions. Carbon that is +0.85 mm will be transferred to the loaded carbon storage tank; carbon finer than 0.85 mm will be collected in bulk bags and transported off site for processing.

Loaded carbon advanced from the first CIL tank will be directed to the recovery screen where it will be rinsed with process solution. The rinsed carbon will be recovered as oversize into the loaded carbon tank, and the undersize slurry will be returned to the first CIL tank.

Periodically loaded carbon from the loaded carbon storage tank will be pumped to trucks to be transported off site for processing and recovery of Au values from carbon with stripped and regenerated carbon being trucked back to the project site and transferred to the barren carbon storage tank. Approximately 11.2 tonnes of loaded carbon is expected to be transferred off site for processing each week.

17.10 Process Reagents and Consumables

The reagent handling system will include equipment used to mix and/or store all reagents required for the process. Reagent mixing and storage will be at ambient temperature and pressure.

Average estimated annual reagent and consumable consumption quantities for the processing area are shown in Table 17.5.

Table 17.5: Projected Annual Reagents and Consumables

Item	Form	Annual Usage
Sodium Cyanide	Liquid Delivery, 30% NaCN by Weight, or Briquettes in Isotanks	1,175 t NaCN solid
Lime (Pebble CaO)	Bulk Delivery Truck	438 t
Cement	Bulk Delivery Truck	11,680 t
Activated Carbon	500 kg Supersacks	17.5 t
Flocculant	Dry Solid Sacks	73 t
Antiscalant	Liquid Tote 1 m^3 Bins	43.2 t
Grinding Balls	Bulk Delivery Truck	756 t
Ball Mill Liners		81 t

Notes: t = tonne, kg = kilogram, m^3 = cubic metres

17.10.1 Sodium Cyanide

NaCN will be used in the leaching process and will be delivered in tanker trucks as a liquid at 30% concentration by weight (1.15 SG) or in briquettes in Isotanks for the Solid to Liquid System (SLS). NaCN will be stored in a 98 m^3 steel tank at the Recovery Plant area and a 24.6 m^3 tank at the mill area within concrete containment and will be distributed by metering pumps to points of use.

All cyanide distribution lines will be double-containment, either by “pipe-within-pipe” or “pipe-over-liner” containment systems. Cyanide will be consumed at an estimated 0.26 kg/t solid NaCN for the low-grade heap leach material and 0.58 kg/t solid NaCN for the high-grade mill material.

17.10.2 Lime

Pebble quicklime (CaO) will be used to treat the high-grade material prior to milling and cyanide leaching to maintain the alkaline pH. Lime will be delivered in bulk by 20-ton trucks, which will be off-loaded pneumatically into the storage silo. Lime will be metered onto the ball mill feed conveyor with a rotary feeder from the 100-ton storage silo.

Lime will be consumed at an estimated 0.6 kg/t for mill (high-grade) material.

17.10.3 Cement

Cement will be used to treat the low-grade material prior to cyanide leaching to maintain the alkaline pH, heap stability, and proper percolation. Cement will be delivered in bulk by 20-ton trucks, which will be off-loaded pneumatically into the storage silo. Cement will be metered onto the Agglomeration Drum feed conveyor with a rotary feeder from the 200-ton storage silo.

Cement will be consumed at an estimated 4 kg/t for heap (low-grade) material.

17.10.4 Activated Carbon

Activated carbon will be used to adsorb precious metals from the leach solution in the adsorption columns and CIL circuit. Make-up carbon will be 6-x-12 mesh and will be delivered in 500 kg supersacks. It is estimated that approximately 3% of the carbon delivered will have to be replaced due to carbon fines losses. Consumption has been estimated at 17.5 tonnes per annum (tpa).

17.10.5 Flocculant

Flocculant will be used to aid settling in the thickeners. Flocculant will be received in 50 lb (22.7 kg) bags on pallets. Estimated consumption will be at 0.1 kg/t solids for the thickeners.

17.10.6 Antiscalant

Antiscalant will be used to prevent the build-up of scale in the process solutions and heap irrigation lines. Antiscalant will be added directly into pipelines or tanks, and consumption will vary depending on the concentration of scale-forming species in the process stream. Delivery will be in liquid form in 1 m^3 tote bins.

Antiscalant will be added directly from the supplier tote bins into the pregnant, barren, and desorption pumping systems using variable speed, chemical-metering pumps. On average, antiscalant consumption is expected to be about 6 PPM of for leach solutions.

18.0 Project Infrastructure

This Item contains forward-looking information related to locations and designs of facilities comprising infrastructure for the Project. The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts or projections in the forward-looking information include any significant differences from one or more of the material factors or assumptions that were set forth in this sub-Item including Project development plan and schedule, available routes and facilities sites with the characteristics described, facilities design criteria, access and approvals timing.

18.1 Roads

18.1.1 Access Roads

The Tonopah Gold Project is located approximately 30 km northeast of the town of Tonopah and is accessed by paved SR 376 and Nye CR 82. As part of the project development, Nye CR 82 will need to be relocated.

18.1.2 Site Roads

Internal site roads will be primarily dirt roads which will connect site and process facilities for access.

18.1.3 Haul Roads

Table 18.1 shows the haulage distances supporting the mining operation. It details various in-pit and ex-pit routes, with notable entries such as the dump ramp at 1,032 m, the extensive Phase 0 Inpit at 1,585 m, and multiple distinct phases above and below certain elevations in the pit. The list also includes process-related ex-pit and in-pit ramps.

Table 18.1: Haulage Distances

Haulage Area	Length (m)
Dump Ramp	1,032
Mine Ex-pit	116
Phase 0 In-pit	1,585
Phase 01 In-pit	1,415
Phase 02 above1664 m In-pit	1,905
Phase 02 1676 m to 1736 m In-pit	1,657
Phase 02 S2 Above 1736 m In-pit	2,010
Phase 02 S2 below 1676 m In-pit	1,844
Phase 02 S3 above 1672 m In-pit	1,778
Phase 02 S3 below 1672 m In-pit	1,197
Phase 02 S3 In-pit	1,051
Phase 02 S3 In-pit dump	478
Process Ex-pit 1	229
Process Ex-pit 2	101
Process Heap Leach Ex-pit	54
Process mill Ex-pit	53
Process stockpile Ex-pit	35

18.2 Project Buildings and Facilities

Buildings for the Tonopah Gold project will primarily be mobile trailer type buildings or pre engineered steel buildings. Planned buildings for the project include:

- Administrative office trailer (7.3 m x 18.3 m double wide pre-fabricated trailer)
- Process office trailer (3.65 m x 18.3 m double wide pre-fabricated trailer)
- Milling Facilities (roofed mill area)
- Site laboratory (pre-engineered steel building)
- Truck shop and warehouse (pre-engineered steel building)

18.3 Fuel Storage

The fuel storage system will consist of above ground tanks; including a diesel tank and a gasoline tank which will be equipped with all necessary fuel dispensing equipment for operation. Fuel will be delivered to the mine site via tanker trucks.

The diesel and gasoline tanks will be insulated and heated to prevent fuel gelling and will be contained within in a lined containment berm to ensure fuel cannot leak into the environment.

18.4 Power Supply, Communications, and IT

18.4.1 Power Supply and Distribution

Power for the operation will be provided to the project site via power line. An existing 15 kV power line on runs along the eastern side of the property and is only lightly utilized. The line connects to the Nevada Energy power grid and may be upgraded under existing permits to a 25 kV service. This power line may have sufficient capacity, subject to additional study, to meet the needs of a production operation at the Tonopah Gold Project.

The Project will have an estimated attached power of 8.7 Megawatt (MW) with an average demand of 5.1 MW. Power will be distributed using overhead power lines at 4.16 kV, 3 phase, 60 Hz and stepped down to 480 V, 220 V and 110 V as required. Power will be supplied to Motor Control Centers (MCCs) or distribution panels and 4160 V for motors above 373 kW (500 hp) and 480 V for 220/110 V for motors below 373 kW and low voltage systems. All overhead distribution lines will be connected to a main switchgear which will include synchronization, control panels, disconnects, circuit breakers, instrumentation and data logging.

18.4.2 Estimated Electrical Power Consumption

The estimated electrical power demand for the project by area is presented in Table 18.2.

Table 18.2: Tonopah Gold Project Power Summary

Area	Attached Power (kW)	Operating Demand (kW)
Area 113 - Crushing	1,865	1,049
Area 115 - Agglomeration & Stacking	1,235	660
Area 118 - Grinding, Gravity, & Thickening	1,766	1,126
Area 119 - CIL	417	266
Area 120 - Tailings Thickener	52	33
Area 121 - Filtration	713	455
Area 122 - Heap Leach Pad & Ponds	1,259	708
Area 128 - Carbon Adsorption & Handling	174	98
Area 134 - Reagents	49	27
Area 338 - Laboratory	225	127
Area 362 - Water Supply, Storage & Distribution	316	177
Area 368 - Compressed Air & Fuel	357	201
Area 366 - Facilities	240	135
Total	**8,669**	**5,063**

18.4.3 Emergency Power

In the event of a power failure or power interruption, a diesel-fired backup generator will be used to supply emergency power for project safety and security. The emergency generator will be located next to the recovery plant. The emergency generator is sized at 1,000 kW and is sufficiently sized to run all of the major process pumping circuits, including the pregnant and barren solution pumps. A fuel tank will be provided for the generator to maintain a 24 h fuel supply. The fuel storage system will also include a concrete containment area sized for 110% of the capacity of the tank.

18.4.4 Communications

The site will be connected to the local phone and internet data network using a microwave or other through the air method. Cellular service is currently available throughout the Project site via a newly constructed cellular phone tower located within the Project property boundary.

18.5 Water

18.5.1 Raw Water Supply & Distribution

The project will require raw water supply for the following uses:

- Mining operations for dust control, drilling, etc.;
- Crushing for dust control;
- Makeup water for the heap leach and milling circuit;
- Process plant and laboratory;

Raw water will be provided by production wells and will be pumped to a head tank for distribution to other areas. A portion of the head tank will be used to provide fire water storage.

Water from the head tank will be distributed to the required project areas via a combination of buried and insulated / heat traced pipes to prevent freezing.

18.5.2 Potable & Domestic Water

Potable water will be delivered to the offices as well as various facilities around the plant. Potable water is planned to be delivered to the site from Tonopah via truck and distributed using a potable water storage and transfer pumping system.

18.5.3 Fire Water

Fire water storage will be accommodated using the raw water head tank and will supply fire water to automatic sprinklers, standpipe systems, and hydrants as applicable. An electric fire water pump with diesel backup pump will supply fire water from the tank at the required pressure and flow rates.

18.6 Waste Disposal

18.6.1 Sewage

Sewage for the planned operations will be treated in a sewage disposal system consisting of septic tank system with leach field.

18.6.2 Solid Waste

Solid waste removal will be contracted to a local or regional waste management service. Waste will be temporarily stored on site in a fenced area with regular pickups and disposal by the waste management service. Where appropriate, recyclable materials such as scrap metal, tires, glass, recyclable plastics and drink containers will be separated, containerized as appropriate, and temporarily stored until sufficient volumes are available for shipment to a recycling point.

Special or hazardous wastes such as waste oil, coolant, solvent fluids, used oil filters, used batteries, and contaminated fuel, will be handled, stored, transported, and disposed of in accordance with appropriate Hazardous Waste Regulations. A certified transport and disposal company will collect all waste to transport off site for final disposal.

19.0 Market Studies and Contracts

This Item contains forward-looking information related to Commodity demand and prices for the Project. The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts or projections in the forward-looking information include any significant differences from one or more of the following material factors or assumptions that were applied in drawing the conclusions or making the estimates, designs, forecasts or projections set forth in this Item: including prevailing economic conditions, commodities demand and prices are as forecast over the Study period.

19.1 Gold and Silver Market

Au is the principal commodity while Ag is a byproduct at the Tonopah property. Both are freely traded in transparent markets worldwide. It is a reasonable assumption that there will be a ready market for Au and Ag at market prices. There are no current contracts for the sale of Au or Ag produced at the project. There are no issues anticipated in the ability to obtain contracts to sell the Doré produced to a refiner.

19.2 Gold and Silver Pricing Assumption

The base case Au price used for this report is US$2,400/oz and for Ag is US$27.70, which, as shown in Table 19.1, is rounded down from the 2-year trailing average of US$2,424 for Au and US$27.71 for Ag.

Au and Ag prices have shown a steady increase over the last three years. Figure 19.1 shows the historic Au price and Figure 19.2 shows the historic Ag price. Au and Ag prices and averages shown in the tables are based on the London Bullion Market Association (LMBA) daily AM fix, which is independently administrated by the ICE Benchmark Administration (IBA).

Table 19.1 shows the Au and Ag price trailing average for three different intervals, as of June 28, 2025. The three-year trailing average price of Au has been increasing rapidly on a monthly basis due to sustained Au prices above the US$3,200/oz level, leading to the use of the 2-year trailing average Au price range for the economic analysis.

Table 19.1: Average Au and Ag Prices

Interval	Average Au Price (US$/oz)	Average Ag Price (US$/oz)
1 Year	$2,784	$31.35
2 Year	$2,424	$27.71
3 Year	$2,226	$25.77

Source: LBMA Posted Price Series

Figure 19.1: Historic Au Price

Source: LBMA Posted Price Series

Figure 19.2: Historic Ag Price

20.0 Environmental Studies, Permitting, and Social or Community Impact

This Item contains forward-looking information related to applications, permits, approvals and consents required and time to approvals for the Project. The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts or projections in the forward-looking information include any significant differences from one or more of the following material factors or assumptions that were applied in drawing the conclusions or making the estimates, designs, forecasts or projections set forth in this Item.

20.1 Environmental Technical Studies and Known Environmental Issues

20.1.1 BLM Baseline Needs Assessment Form

The BLM issued a Baseline Data Needs Assessment Form (BNAF) July 12, 2021, describing the updated and/or additional environmental resources surveys required for the BLM to evaluate a Mining Plan of Operations for the Project under National Environmental Policy Act (NEPA). Status of the BNAF is shown in Table 20.1.

20.1.2 Previous Environmental Technical Studies

Midway Gold, the operator and permittee prior to Viva, undertook several studies to support potential future surface and/or underground mining operations. The studies identified and evaluated baseline hydrogeologic conditions, groundwater quality, storm water controls, mine dewatering requirements, resource and waste rock geochemistry, surplus dewatering water management options, including re-infiltration, underground injection and supplemental contribution to the TPU town water system. The studies were primarily focused toward a potential future underground mining operation.

Of note, baseline hydrogeologic studies were conducted from 2006 to 2012. Those studies predicted an average of 107 to 151 liters per second dewatering rate requirement for a potential underground mining operation. They also evaluated the potential for local alluvial soils to accept reinfiltration to the regional aquifer of dewatering pumpage in excess of that necessary for process water make-up. Geochemical testing of waste rock that would be encountered in potential underground decline development reported a low potential for acid rock drainage despite a low net neutralizing potential.

Midway Gold completed baseline environmental and cultural resources surveys in the early 2000s through 2012 that supported initial regulatory permitting applications, authorizations, exploration operations and plans for an underground mining operation. The surveys are summarized below. Permits for exploration activities are discussed in Item 20.4. No surface mining permits have yet been sought or secured; however, Viva has conducted updated and expanded baseline environmental and cultural resources surveys and studies as required by BLM guidance as discussed in Item 20.1.2.4

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

20.1.2.1 Mineralized Material and Waste Rock Geochemical Characterization Studies

Project records indicate that initial acid-base accounting samples were submitted for study by Geomega of Boulder, CO. These results were focused on certain types of Tertiary volcanic tuff and Op argillite, which were considered likely to be the primary waste rock types in an underground mining scenario. While not all possible types of waste rock for all potential mining scenarios were tested, the results demonstrated minimal risk of potential Acid Rock Drainage (ARD).

The July 1, 2007, Geomega memorandum states:

"There are now results for 32 samples of potential waste rock material. On an aggregate basis both the Tertiary volcanic tuff and the Palmetto Formation argillite (with the exception of Outcrop 373) has both low sulfide and low carbonate content. However, while the average carbonate content of the 25 samples (i.e., excluding Outcrop 373) is low (2.3 ppt) the sulfide content is non- existent to minimal resulting in an average NP:AP (Neutralization Potential to Acid Potential) of 8.2 which exceeds the Environmental Protection Agency (EPA) criteria of 3."

Beginning in Q4 2020, additional acid-base accounting (ABA) and net acid generation (NAG) analyses were conducted on samples collected during recent exploration drilling campaigns. The samples represented Project mineralized material, anticipated waste rock, and rock to be potentially exposed in a final post-mining pit configuration. The analytical data was evaluated to determine the potential for the materials to degrade Waters of the State of Nevada during an open pit mining operation and closure/final reclamation/post-closure periods. In addition to the ABA and NAG analyses, select samples were analyzed for specific constituents via the Meteoric Water Mobility Procedure (MWMP) to determine potential concentrations in meteoric waters that contact the mineralized material and waste rock.

Following Regulatory Authorities' (RAs) concurrence with a supplemental geochemical characterization program, Viva initiated testing of 10 Humidity Cell Test (HCT) cells composed of materials that, under final permanent closure conditions, may impact the quality of long-term heap leach pad drain down, water contacting reclaimed waste rock disposal stockpiles, and the water quality of a persistent post-mining pit lake resulting from permitting and mining an open pit mine at the Project. Two cells with alluvium/basalt material, 3 cells with Tertiary volcanic material and 3 cells with Op material were classified as waste rock. One cell with Tertiary volcanic material and 1 cell with Op Formation material were classified as mineralized material. All cells have been terminated following BLM and NDEP BMRR's concurrence with 4 separate Requests for Termination.

Six cells were terminated after collection of 37 total weeks of data reporting no acid generation potential for the 6 cells. Five of the 6 initial terminated cells represented alluvium/basalt, Tertiary volcanic, and Op Formation waste rock and 1 initial terminated cell represented Op Formation mineralized material. Two more cells were terminated after collection of 54 total weeks of data reporting stabilized pH, low sulphate production, and minor amounts of iron. The extended length of time for consumption of remaining Acid Neutralizing Potential (ANP) supported termination of the Tertiary volcanic waste rock and Op mineralized material cells. The ninth cell was terminated after collection of 72 total weeks of data reporting low reaction rates and minor release of sulphide oxidation by-products suggesting the Op waste rock material in the cell is generally inert. The tenth cell was terminated after collection of 88 total weeks of data. The tenth cell representing Tertiary volcanic material classified as mineralized material continued to produce circum-neutral leachate with very low dissolved load pH throughout the 88 weeks of testing. Available ANP was consumed by Week 74 with stable leachate chemistry remaining for the subsequent 14 weeks.

A summary of the results from HCTs can be reported as follows:

- Generally, slightly acidic leachates (pH ranging from 7.5 Standard Units (SU) to 5.8 SU) and low in constituent concentrations.

- Material of one Tertiary volcanic waste rock material cell, 1 Op waste rock cell and 1 Tertiary volcanic mineralized material cell may be considered Potentially Acid Generating (PAG) under NDEP BMRR criteria of an ANP/Acid Generating Potential (AGP) ratio <1.2. An additional Tertiary volcanic waste rock cell and additional Op waste rock cell may be considered PAG under NDEP BMRR criteria of an ANP/AGP ratio <1.2 and BLM criteria for PAG material of an ANP/AGP ration of <3.

Additional ABA analyses were performed on all pre- and post-HCT material, including NAG analyses. Scan search X-ray Diffraction (XRD), Scanning Electron Microscope (SEM) and petrography mineralogy analyses were performed on 3 waste rock samples and 1 mineralized material sample.

It is important to note that volumes and spatial distribution of all rock types identified in plans reported in this PEA, a subsequent Pre-Feasibility Study (PFS) and Feasibility Study (FS), and mine permitting applications will evaluate the nature and extent of potential impacts, if any, of waste and mineralized materials mined at the Project. Data gathered and analyzed to-date do not indicate issues of permitting or operational concern to the success of the Project relative to geochemistry and protection of post-mining groundwater and surface waters.

A Waste Rock and Pit Wall Geochemical Characterization baseline resources report will be prepared evaluating all data generated to-date. A Waste Rock Management plan (WRMP) will utilize data and recommended practices for material handling based on recommendations developed from the geochemical characterization report. The Waste Rock and Pit Wall Geochemical Characterization report and WRMP will be submitted to the BLM and NDEP BMRR as required in conjunction with the overall mine permitting process.

20.1.2.2 Baseline Water Resources Characterization Studies

A total of 23 separate groundwater level and groundwater quality monitoring well data points were established in the Project area by Midway Gold in 2007 through 2011. The wells were only sporadically sampled during this period and no consistent baseline water level and water quality studies were conducted for the Project. Viva resumed baseline groundwater level and groundwater quality monitoring in select wells in December 2020 and continued bi-monthly, then quarterly, and now at least semi-annual monitoring data collection during the spring and fall hydrologic seasons. The initial seeps and springs survey was conducted in April 2021 with follow-up seasonal surveys conducted in July and October 2021, and February 2022.

Two new groundwater level and groundwater quality monitoring wells were approved and constructed in Q4 2022 upgradient from the Project to augment the existing baseline water resources monitoring program. Quarterly monitoring of these wells commenced in Q1 2023 through Q2 2024 with semi-annual monitoring beginning Q4 2024.

Construction and monitoring of additional groundwater monitoring wells, including two groundwater level and groundwater quality monitoring wells downgradient from the Project, will be necessary to determine baseline water quality for the broader area that may be impacted by mine dewatering and reinfiltration operations. Initial RIB testing was completed in 2008 to establish costs for processing water pumped during potential underground mine dewatering and returning it to the hydrographic basin. Additional RIB testing is planned to support the updated water impacts analysis (numerical groundwater model) required for future mine dewatering and reinfiltration permitting of an open pit mining operation.

Baseline groundwater quality data for most site monitoring wells report arsenic concentrations consistently below the Nevada Reference Value (NRV) for arsenic of 0.01 milligrams (mg)/L. Two (2) wells located in the heart of the mineralized zone report concentrations between 0.01 mg/ and <0.02 mg/L. One monitoring well located west of

SR 376 and upgradient of the Project area also reports baseline arsenic concentrations of <0.02 mg/L. This groundwater flows eastwards towards the heart of the Project area. One upgradient monitoring well north of CR 82 reports pH values exceeding the 6.5 SU to 8.5 SU NRV and may be the result of cement contamination during well construction.

The NDEP BMRR has approved for another permittee's Project located in Humboldt County a Water Pollution Control Permit (WPCP) with a dewatering water discharge-to-RIBs action-level limit of 0.019 mg/L based on background groundwater quality and the most recent predictive modelling results. WPCPs require predictive model updates with every 5-year permit renewal application, and certain step-by-step actions if the action-level limit is exceeded.

A seven-day aquifer test was conducted at the Project in November through December 2022. Variable rate testing consisted of four 100-minute testing periods at approximately 3 litres per second (L/s), 6 L/s, 9 L/s and 11 L/s, respectively. Measured drawdown during the variable rate test was 7.6 m. A constant rate aquifer pump test lasting the seven-day test period reported total pumpage discharged of 6,643 m^3 at an average pumping rate of 11 liters per second. Measured drawdown during the seven-day constant rate pump test was 11.6 m. A draft preliminary peak dewatering rate of 20 m^3 per minute during mining operations was estimated based on the aquifer testing.

Further review of Midway Gold and the 2022 test data, and water level monitoring data collected 2020-present, has provided an updated draft LoM estimated average dewatering rate corrected for lateral flow constraints of 14 m^3 per minute. The estimate is based on current groundwater elevations in the PEA pit area, a dewatering target elevation of approximately 252 m below ground surface (bgs), a strategy of constructing 5 extra-pit alluvial water interceptor wells and 2 in-pit bedrock aquifer dewatering wells, and a hydraulic conductivity range of 0.3-0.6 m/day assuming a saturated thickness of 252 m.

Data gathered by the baseline water resources characterization program will be utilized to prepare an updated water resources impact analysis (groundwater model) required by the BLM and NDEP BMRR in conjunction with the overall Project permitting process. A predictive post-mining pit lake water quality study will also be prepared following completion of the Waste Rock and Pit Wall Geochemical Characterization and Baseline Water Resources Characterization reports.

20.1.2.3 Cultural Resources Survey Studies

Three initial cultural resources surveys were conducted in the Project area: 1993, 1994 and late 2002-early 2003. These surveys supported mineral exploration activities at that time in the Project area. The Ralston Quarry where Viva proposes no disturbance was originally noted by BLM archaeologist Roberta McGonagle in 1978 with a 1995 follow-up, and National Register of Historic Places (NRHP) eligibility determination in 1978.

The Midway Archaeological Site is determined to be potentially eligible for listing on the NRHP with many cultural resource features such as fire-cracked rock, lithic scatters, etc. recorded in the 2002-2003 survey. BLM, State Historic Preservation Office (SHPO) and Midway Gold were parties to a Programmatic Agreement (PA) governing development of Midway's exploration activities within the Area of Potential Effect (APE), and administration of the PA to ensure that historic properties are treated to avoid or mitigate effects to the extent practicable and to satisfy BLM Section 106 responsibilities for all aspects of the Project. Midway Gold submitted to the BLM thirty-three individual work plans identifying specific locations of proposed disturbance for review and authorization to proceed subject to PA stipulations. The PA facilitated timely authorizations and in-field exploration activities.

Viva assumed Midway Gold’s position under the existing PA and filed eight additional work plans to support its recent drill programs with subsequent Notice to Proceed (NTPs) issued by the BLM.

Native American consultations were conducted by Midway Gold involving letters, phone calls, and site visits on May 28 and July 8, 2003. Concerns expressed by Tribal representatives included potential impacts to the cultural site and impacts to the spiritual value of the Ralston Quarry and Midway Archaeological Site; however, there was no evidence at that time of any recent or current use of the Midway Archaeological Site by Native Americans even though they are aware of the existence of the Site.

Viva’s exploration activities adhere to all Federal and State cultural resources regulations and employees and contractors are informed of the repercussions of collecting cultural artifacts, if any, or damaging cultural resources sites.

A cultural resources re-survey of approximately 708 hectares located in Sections 29 and 30, and portions of Section 19, 20 and 32 T5N R44E was performed in November 2020 with the intent to identify artifacts exposed by moving sand dunes since conduct of prior surveys in the early- to mid-2000s. The re-survey results have been reviewed and accepted by the BLM and SHPO. Updated Class III surveys and Class III surveys of unsurveyed areas will be required to support future open pit mining operations permitting once a full potential disturbance footprint has been determined.

20.1.2.4 Environmental Resources Studies

Surveys of site and area biological resources were conducted in 2003 to support BLM Environmental Assessment (EA) level analysis of the Exploration Plan of Operations (ExPoO) under NEPA. Surveys included identification of geologic; air quality; soils; vegetation; range; invasive non- native plant species; wildlife; threatened, endangered and sensitive plant and animal species; and wild horses’ resources in addition to the geochemical and water resources investigations described in Sections 3.3.1.1 and 3.3.1.2 of the EA. No significant impacts to the surveyed resources were identified by the BLM in the 2003 EA and the DR authorizing Project exploration activities was issued December 12, 2003. To build on data developed by previous Project operators, Viva resumed in December 2020 periodic hydrogeologic resources monitoring and characterization studies discontinued by Midway Gold in 2011.

Viva and its contracted technical support team met with BLM and NDEP BMRR staff in March 2021 to initiate Agency contact, present Viva’s project and team, and coordinate and receive guidance on necessary baseline environmental resources information required for Agency review of future mining operation permitting. The Agencies reviewed Viva’s plans, including supplemental geochemical characterization plans. The Agencies concurred with the proposed supplemental geochemical characterization plans for additional MWMP and ABA analysis and HCTs. The BLM issued a BNAF July 12, 2021, describing the updated and/or additional environmental resources surveys required for the BLM to evaluate an MPoO under NEPA. Viva continued/initiated data collection and surveys for resources identified in the BNAF and Table 20.1 reports the status of each.

In addition to environmental resources described in Sections 3.3.1.1 and 3.3.1.2 of the EA and supplemented by Viva’s programs 2018 to present, Viva contracted in response to the BNAF a golden eagle and raptor nest survey, and a survey of those wildlife resources that could reasonably be conducted in accordance with season-specific protocol for general wildlife, bat habitat, pygmy rabbit and kangaroo mouse. Survey reports were submitted to the BLM in December 2021. Additional surveys were contracted in 2022 to complete gathering the data necessary for the BNAF’s required wildlife and botanical baseline survey reports, including:

- Botanical survey,
- Vegetation community mapping,
- Special Status Plant Species survey,
- Noxious weed survey,
- Wildlife surveys,
- General wildlife survey, and
- Special Status Wildlife Species surveys.

The Baseline Biological Survey Report was submitted to the BLM September 19, 2022, and following Agency review with no comments, was accepted by the BLM November 29, 2022, and represents a completed requirement of the BNAF.

Additional baseline resources reports required by the BNAF will be completed and submitted for RAs review as Project planning and engineering guide preparation of the MPoO.

Table 20.1: 2021 BNAF Surveys and Status

Resource	Resource Status	Baseline Required	Status/Comments
Noxious Weeds, Invasive Non-Native Species	Present/ May Be Affected	Yes	Weed survey completed; reviewed & accepted by BLM. Noxious weed control plan will be appended to MPoO when submitted to BLM.
Noise	Present/ May Be Affected	Yes	Baseline noise study to be prepared once site facilities are determined for MPoO.
Soils	Present/ May Be Affected	Yes	Surveys completed, baseline submitted to BLM Q3 2022, reviewed, no comments, accepted.
Vegetation	Present/ May Be Affected	Yes	Surveys completed, baseline submitted to BLM Q3 2022, reviewed, no comments, accepted.
Forestry	Present/ May Be Affected	Yes	Surveys completed, baseline submitted to BLM Q3 2022, reviewed, no comments, accepted.
Geology/Minerals	Present/ May Be Affected	Yes	Geochemical characterization complete. Baseline report submittal to RAs Q3 2025.
Water Quality/Quantity	Present/ May Be Affected	Yes	Baseline characterization ongoing. Updated numerical groundwater model, pit lake predictive water quality model, ERA, WRMP baseline reports scheduled for submittal RAs 1H 2026.
Wetlands/Riparian Zones	Present/ May Be Affected	Yes	S&S baseline to be incorporated into water quality/quantity baseline. WOUS to be addressed in baseline.
Floodplains	Present/ May Be Affected	Yes	Baseline scheduled for submittal to BLM Q4 2025.
Land Use Authorization	Present/ May Be Affected	Yes	Viva will address in MPoO; BLM will address, augment in NEPA review.

Resource	Resource Status	Baseline Required	Status/Comments
Air Quality	Present/ May Be Affected	Yes	Air Quality Impact Assessment (AQIA), Greenhouse Gas (GHG) accounting, National Ambient Air Quality Standards (NAAQS), Hazardous Air Pollutant (HAPS) emissions inventory, Mercury (Hg) emissions baseline preparation to commence following finalization of mine planning/process engineering. Baseline to be submitted to RAs during permitting process.
Visual Resources	Present/ May Be Affected	No	Viva will contract for visual resources analyses once facilities locations are determined in Feasibility Study. Analyses will be reported to BLM for address in NEPA document.
Wilderness	Present/ May Be Affected	No	BLM discusses 2 LWCs in SERs & NEPA document.
Paleontological Resources	Not Present	No	
Native American C&C	Present/ May Be Affected	No	BLM will address C&C in conjunction with NEPA review.
Environmental Justice	Present/ May Be Affected	No	BLM will address in NEPA review.
Social & Economic Values	Present/ May Be Affected	No	Viva will address in MPoO; BLM will address, augment in NEPA review.
Recreational Resources	Present/ May Be Affected	No	BLM will address in NEPA review.
Mining Law Administration	Present/ May Be Affected	No	BLM will address in NEPA review.
Wastes, Hazardous or Solid	Present/ May Be Affected	No	Viva will address in MPoO; BLM will address, augment in NEPA review.
Wild Horses & Burros	Not Present	No	
Wild & Scenic Rivers	Not Present	No	
Grazing Management	Present/ May Be Affected	No	Viva will address in MPoO; BLM will address, augment in NEPA review.
Human Health & Safety	Not Present	No	
Farmlands (Prime or Unique)	Not Present	No	
Areas of Critical Environmental Concern	Not Present	No	

20.2 Requirements and Plans for Waste and Tailings Disposal, Site Monitoring and Water Management During Operations and Post-Closure

The BLM administers 43 CFR 3809 regulations pertaining to mining operations on public lands. Stipulations identified in a DR issued by the BLM authorizing the mining operation surface disturbance will state the BLM requirements for adherence to plans published in the MPoO and EIS, including site monitoring and water management during operations and post-closure.

The NDEP BMRR, in coordination with other state, federal, and local agencies, regulates mining activities under Nevada Administrative Code (NAC) Chapters 445A.350-NAC 445A.447 and 519A.010-NAC 519A.415, which implement the requirements of the Nevada Revised Statutes (NRS) Chapters 445A.300-445A.730 and 519A.010-519A.290. WPCPs for mining operations and the discharge of excess mine dewatering water are administered by the NDEP BMRR Regulation Branch. These permits establish requirements for the construction, operation, and closure of mine facilities, including waste rock dumps, tailings storage, site monitoring, and water management during active operations and post-closure. Specific permit conditions are determined by agency review of WPCP permit applications, which must include baseline, site-specific hydrologic, hydrogeologic, geochemical data as well as Viva's proposed operational and monitoring plans.

20.3 Requirements for Pit Dewatering

The Tonopah open pit will extend below the existing water table. As a result, a pit dewatering system ahead of mining advance is required. A conceptual dewatering system design for the Project was developed by Piteau Associates of Reno, Nevada, Viva's long-term hydrologic consultant for the project, dated June 20, 2025, using the following assumptions:

- Dewatering target of approximately 252 m bgs.
- Design dewatering rate of 14 m^3 per minute.
- Individual dewatering wells estimated to produce between 1.9 and 5.7 m^3 per minute.
- 5 alluvial interceptor dewatering wells and 2 bedrock dewatering wells with a range of productivity based on local hydrogeology and compartmentalization (3 to 5.7 m^3 per minute for interceptor wells, 1.9 to 3.8 m^3 per minute for bedrock wells). Interceptor wells total depth (TD) 152 m and bedrock wells TD 213 m.
- 3 RIBs with a 75% duty cycle to infiltrate excess dewatering water not required for process make-up.
- Each basin volume 49,000 m^3 with design infiltration capacity of 0.004 m^3 per minute at dewatering rate of 13.6 m^3 per minute

Conclusions of the conceptual dewatering system design are as follows:

- The dewatering rate is estimated to peak at 14 m^3 per minute with the final rate adjusted with a radial flow correction factor.
- Five 152 m interceptor wells producing between 3 to 5.7 m^3 per minute.
- Two 213 m in-pit wells producing 1.9 to 3.8 m^3 per minute.
- Three RIBs operating at a 75% duty cycle.

- 6,700 m of conveyance pipeline, 35.56 cm to 50.8 cm diameter, and 20.32 cm connection lines.
- The dewatering rate is based on highly variable hydraulic parameters that will be confirmed by two longer duration (30-day), higher volume (0.95-3.79 m^3 per minute) pumping tests to fully stress the groundwater system.
- Capital Cost to construct the wellfield is estimated at US$15.1M.
- Dewatering Operations and Maintenance (O&M) cost range is estimated between US$335K and US$1,251K annually.
- Total O&M through the 8 years of dewatering (1 year prior to mining plus 7-year mine life) is estimated at US$6.9M.
- The system design is based on a preliminary estimate of dewatering rates and cost.
- Dewatering rates will change depending on mining rate driving the required rate of drawdown.

20.4 Environmental Permitting and Bonding Requirements

20.4.1 Current Permits

Initial exploration drilling operations involving surface disturbance of less than 2 hectares on public land were authorized by the BLM under Notice of Intents (NOIs). The ExPoO and Nevada Reclamation Permit application to disturb up to 30.4 hectares for mineral exploration was filed with the BLM and NDEP BMRR in January 2003. The BLM determined it was necessary to prepare an EA assessing the potential environmental consequences of the proposed exploration activities. The final EA (NV065-2003-037) was published, and a DR and FONSI, issued approving the ExPoO December 12, 2003. NDEP BMRR approved Reclamation Permit 0210 in January 2004. Subsequent ExPoO and Reclamation Permit modifications and amendments followed in 2004-2007, with a Major Modification/Amendment submitted in January 2008 to include construction and operation of an underground mine. Agency processing of the Modification/Amendment was suspended in 2009 as exploration operations at the Project were idled.

The ExPoO and Reclamation Permit were transferred to Viva's Nevada-registered subsidiary 0862130 Corp. as Owner and Operator in 2017 following Viva's posting of the required reclamation bond. BLM Serial NVNV106037004 (legacy Casefile NVN-076629) and Nevada Reclamation Permit 0210 remain in full force and effect. Only 7.6 hectares have been disturbed under the ExPoO and Permit to-date. Viva amended the ExPoO and Reclamation Permit in 2022 to include a temporary aquifer testing well and submitted an updated targeted reclamation bond cost estimate addressing the plug and abandon costs of the test well. Viva also permitted through the BLM in 2022 surface disturbance of less than 2 hectares public land outside of the ExPoO boundary under 2920 Land Use Authorization Permit Serial NVNV1058663574 (legacy Casefile NVN-101549). The 2920 Land Use Permit authorized construction and long-term monitoring of 2 upgradient groundwater level and groundwater quality monitoring wells, and construction and testing at 2 locations of 2 test pits to support future RIBs permitting. An application to extend the 2920 Land Use Permit beyond its November 10, 2025, expiration was submitted to the BLM June 16, 2025.

Waiver MM-232 issued by the Nevada Division of Water Resources for temporary use of less than 6,167 m^3 per annum groundwater to support exploration drilling operations and fugitive dust suppression was extended to

November 17, 2027. Waivers M/O-2418 and M/O-2454 authorizing long-term groundwater level and groundwater quality monitoring of 3 wells expire October 24, 2027, and extension of the Waivers will be requested in early Q3 2027.

Viva intends to continue collecting baseline environmental resources information and submitting reports to the Agencies during 2025-2026 to support future Agency review of applications for mining operation permits at the Project.

20.4.2 Permits Required for Mining Operations

Permits required for the proposed surface mining operation will include but not be limited to those identified in Table 20.2.

Table 20.2: Permits Required for Mining Operations

Permit Name	Agency	Description
Plan of Operations	BLM	Plan of Operations is required for all mining and processing activities and exploration exceeding 5 acres of surface disturbance on public lands managed by the BLM. The BLM approves the plan and determines the required environmental studies, usually an Environmental Assessment (EA) or an Environmental Impact Statement (EIS) based on the requirements outlined in the National Environmental Policy Act (NEPA).
National Environmental Policy Act - Environmental Impact Statement (EIS), DR	BLM	It is assumed an EIS and ROD will be required for full-production mining.
Water Pollution Control Permit (Facilities)	NDEP, BMRR - Regulation Branch	Mines operating in the State of Nevada are required to have a Water Pollution Control Permit (WPCP) to protect waters of the State during mining activities.
Water Pollution Control Permit (Rapid Infiltration Basins)	NDEP, BMRR - Regulation Branch	Water Pollution Control Permit (WPCP) for infiltration of water from the underground mine operations into Rapid Infiltration Basins (RIBs).
Water Rights	Nevada Division of Water Resources (NDWR)	Water rights are issued by the Nevada Division of Water Resources and State Engineer based on Nevada water law which allocates rights based on appropriation and beneficial use within the water basin. Prior appropriation (also known as "first in time, first in right") allows for the orderly use of the state's water resources by granting priority to parties with senior water rights. This concept warrants the senior uses are protected, even as new uses for water are allocated. Mining water rights are considered temporary in nature.

Permit Name	Agency	Description
Nevada Reclamation Permit	NDEP, BMRR - Reclamation Branch	The BMRR Reclamation Branch works in coordination with the BLM for projects on public land to establish reclamation guidelines and a reclamation cost estimate to support project bonding. This permit and associated bond identifies land disturbed by mining activities requiring reclamation to safe and stable conditions to promote safe and stable post-mining land use. A permit is required for any disturbance of 5 acres or more. The reclamation cost estimate (RCE) is financially secured with a posted surety. The posted surety amount provides assurance that reclamation will be pursuant to the approved reclamation plan in the event that the State has to perform reclamation or is held until reclamation has been successfully conducted.
Air Quality Operation Permit	NDEP, Bureau of Air Pollution Control (BAPC)	An owner or operator of any proposed stationary source must submit an application for and obtain an appropriate operating permit before commencing construction or operation. Class II Air Permit - Typically for facilities that emit less than 100 tons per year for any one regulated pollutant and emit less than 25 tons per year of total Hazardous Air Pollutants (HAP's) and emit less than 10 tons per year of any one HAP.
Air Quality Surface Area Disturbance Permit	NDEP, BAPC	A Surface Area Disturbance Permit (SAD) is required for any project that disturbs 5 or more acres of ground. Annual updates show what areas have been disturbed.
Industrial Artificial Pond Permit	Nevada Department of Wildlife (NDOW)	NDOW oversees wildlife management of industrial artificial ponds at mine sites. The ponds are required to have wildlife protection design standards and quarterly mortality reports are submitted to document any deceased wildlife discovered in the ponds.
Storm Water Control Permit	NDEP, Bureau of Water Pollution Control (BWPC)	Required for storm water runoff from waste rock piles, haul roads, facilities and other mine areas that have not mixed with process solutions or other contaminant sources. Typical pollutants include suspended and dissolved solids and minerals eroded from exposed surfaces.
LPG License	NV LP-Gas Board	An LPG License is required for any business engaged in activities involving liquefied petroleum gas (LP-Gas), including installation, delivery, and resale. Mining operations that store, dispense, or use LP-Gas for equipment or heating purposes require one of these licenses depending on the volume and type of use.

20.4.3 Bonding Requirements

The BLM and NDEP BMRR require a reclamation bond to ensure completion of reclamation and closure of the Project in the event the permittee is no longer able to complete the regulatory requirements due to bankruptcy or other factors.

The Nevada-approved Standardized Reclamation Cost Estimator (SRCE) Version 1.4.1 Build 017b (Revised 16 May 2019) with Cost Data File SRCE_Cost_Data_File_1_12_Std_2024.xlsm was used to estimate a potential reclamation bond amount for the Project based on operations and closure assumptions used in this PEA. Operational and Maintenance Costs were estimated to be US$17.5M. An estimated total reclamation bond of

US$23.7M includes US$6.2M of Indirect Costs required by the Federal and State Agencies to complete reclamation and closure operations. This bond estimate is considered reasonable when measured against analog projects involving heap leach component closure, operational mine dewatering, and post-mining pit lake(s). Triennial reclamation bond cost estimate updates are required by Reclamation Permit 0210; therefore, the bond amount required by the Federal and State Agencies will change over time. By pursuing concurrent reclamation as operations progress in later years of operation the bond amount may be reduced by Agency approval of earthworks and vegetation of individual components as they meet reclamation standards.

Viva would intend to post a 20% cash collateral on the US$23.7M bond of US$4.7M requiring a surety of US$19.0M to be obtained from commercial surety sources. The premium of a commercial surety of US$19.0M is estimated to be US$20 per US$1,000 of surety face value for an annual premium cost of US$380,000 per year.

Actual reclamation and closure costs are estimated to be approximately US$12M when reclamation and closure is achieved by concurrent reclamation, mining operation labour at non-Federally mandated labour rates, and Viva and contractor staff during operations, closure and post-closure periods. Those costs are expected to be incurred beginning in Year 1 following completion of mining. Reclamation earthworks on the waste rock dump, dry stack tails and heap leach pad will occur in Year 1 as residual process fluid in the heap drains down. Other closure projects in Years 1 through 3 will involve plug and abandon of mine dewatering wells and removal of pipelines, reclamation of the RIBs sites, removal of the mill and process infrastructure, rapid-fill of the post-mining pit lake, and construction of the long-term heap draindown evaporation cells. Salvage value of the mill and process infrastructure will offset potential costs associated with removal from the site of these components.

20.5 Social and Community-Related Requirements and Plans

At present there are no social and community-related requirements by Nye County, the Town of Tonopah, or Federal or State Agencies. Viva’s Environmental, Social, and Governance (ESG) commitment guides the company’s outreach efforts that include local expenditures for lodging, meals, office rental, fuel and supplies. Viva shares groundwater monitoring data with the Town of Tonopah and its hydrology consultant reporting groundwater level and groundwater quality data relevant to TPU’s proximal public water supply wells. Viva hires local contractors and labour to support current exploration operations.

Viva plans to hire local labour whenever feasible for the planned surface mining and processing, and long-term reclamation and closure operations. The local labour force currently employed at other mining operations in Nye County will provide a labour source locally resident which will lessen the burden of providing housing and services for the Project. As mine permitting proceeds Viva will engage with Non-Governmental Organizations (NGOs) and interested parties as required by the Federal and State permits review process.

20.6 Mine Closure Requirements and Costs

The Project is required by Federal and State regulations to leave the Project with stable landforms vegetated to regulatory standards and ensure that unnecessary and undue degradation of the environment and, in particular, groundwaters of the State has not and will not occur.

Site facilities, including buildings and structures, will be removed or remain for a future post-mining land use if approved by the Federal and State agencies. Production and mine dewatering wells will be plugged and abandoned in accordance with NDWR requirements. Groundwater monitoring wells will remain to support long-term groundwater quality monitoring until such time they are no longer required. They then will also be plugged and abandoned in accordance with NDWR requirements. The waste rock dump, heap leach pad and dry-stack tailings pad will be recontoured and revegetated to present a stable landform. Long-term draindown of heap leach fluid will be directed to evaporation cells constructed within the existing process fluid ponds in accordance with agency-approved mine closure plans. Once discharge of excess mine dewatering water ceases, pipelines will be removed from the site and the RIBs will be filled in with the stockpiled material originally excavated to construct the basins. RIB monitoring piezometers and wells will be plugged and abandoned as required. The final mine pit will be rapid-filled with pumped groundwater to reduce the time involved in reaching a stable final post-mining pit lake elevation towards initiation of the post-closure monitoring period. The post-mining pit lake water quality will be monitored as required to confirm the lake water does not affect adversely the health of human, terrestrial or avian life in accordance with NAC 445A.429.3.(b). Pit lake water quality predictions and the size and characteristics of the potential pit lake will be generated during the next stages of the project based upon geochemical modelling and rates of inflow estimated by long term pump tests along with predictions of the rates of evaporation and amount groundwater interaction.

Actual reclamation and closure costs are estimated to be approximately US$12M when reclamation and closure is achieved by concurrent reclamation, mining operation labour at non-Federally mandated labour rates, and Viva and contractor staff during operations, closure and post-closure periods.

21.0 Capital and Operating Costs

This Item contains forward-looking information related to both the Capital and Operating cost estimates for the Tonopah Gold Project (the Project). The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts or projections in the forward-looking information include any significant differences from one or more of the following material factors or assumptions that were applied in drawing the conclusions or making the estimates, designs, forecasts or projections set forth in this Item: unit costs as estimated are not escalated and thereby are in constant (or real) dollar terms, projected labour and equipment productivity levels may not be realized, scaling factors as used might differ when enter into contracts to build the facilities as described, firm price tenders for equipment and consumable supply can vary from budgetary and database factors, contingency might not be sufficient to account for changes in material factors or assumptions.

Estimates may fluctuate due to changes in material prices, labour rates, market conditions, and unforeseen events. Viva should periodically review and confirm the estimates.

The CAPEX and OPEX were provided by multiple parties as outlined below in Table 21.1 for compilation by WSP; and are based on the scope of work presented in prior sections of this Technical Report.

WSP under the supervision of QP, William Richard McBride, P. Eng., developed the CAPEX and OPEX for the open pit mining, and mine site infrastructure for the Project scope described in this Technical Report.

The cost estimates herein follow methodologies and procedures with due care and professional judgement, aiming for accuracy within specified margins for the Project.

Table 21.1: CAPEX/OPEX Responsibilities

Contributor	Scope
WSP	Overall responsibility for CAPEX/OPEX, Open Pit Mining, Minesite Infrastructure
KCA	Provided process plant initial and sustaining CAPEX and OPEX.
	Provided General and Administrative (G&A) cost estimate.
Piteau	Provided Dewatering CAPEX and OPEX; initial and sustaining costs
LEC	Provided Bond Premium and Closure Costs

21.1 Capital Costs

The CAPEX was developed to deliver an Association for the Advancement of Cost Estimating (AACE) class 5 estimate in support of a PEA. As shown in Table 21.2, AACE guidelines identify class 5 estimates as having an accuracy range with a low of -20% to -50% and a high of +30% to +100%. Ranges can vary significantly, as shown, dependant on the occurrence of unusual risks. For this reason, class 5 estimates are expected to have a contingency allowance aligned to the accuracy range. As each portion was estimated individually by the responsible parties, the accuracy ranges and contingency allowances are identified within each party's sub-item.

Table 21.2: AACE Cost Estimate Classification Table

Estimate Class	Primary Characteristic	Secondary Characteristic		
	Maturity Level of Project Definition Deliverables	End Usage	Methodology	Expected Accuracy Range
	Expressed as % of Complete Definition	*Typical Purpose of Estimate*	*Typical Estimating Method*	*Typical variation in low (L) and high (H) ranges*
Class 5	0% to 2%	Conceptual Planning	Capacity factored, parametric models, judgement, or analogy	L: -20% to -50% H: +30% to +100%
Class 4	1% to 15%	Screening options	Equipment factored or parametric models	L: -15% to -30% H: +20% to +50%
Class 3	10% to 40%	Funding authorization	Semi-detailed unit costs with assembly level line items	L: -10% to -20% H: +10% to +30%
Class 2	30% to 75%	Project control	Detailed unit cost with forced detailed take-off	L: -5% to -15% H: +5% to +20%
Class 1	65% to 100%	Fixed price bid, check estimate	Detailed unit cost with detailed take-off	L: -3% to -10% H: +3% to +15%

In the case of the 2025 Viva Tonopah PEA, the CAPEX reflects an EPCM-type execution model. Although some individual elements of the CAPEX may not achieve the target level of accuracy, the overall CAPEX should fall within the parameters of the intended accuracy.

All CAPEX and OPEX costs are expressed in United States Dollars (US$) and are based on Q2 2025 pricing.

21.2 Overall Capital Cost Estimate

21.2.1 Capital Cost Summary

The CAPEX consists of direct and indirect capital costs as well as contingency. Provisions for sustaining capital are also included, for mining, tailings storage facility expansion, dewatering wells additions, and heap leach pad growth. Amounts for the mine closure and rehabilitation of the site have been estimated as well. The capital is representative of a mine mobile equipment lease for the loaders, haul trucks, production drills, and auxiliary equipment.

Table 21.3 presents separate summaries of the Initial CAPEX, and Sustaining CAPEX distributed over the LoM with the two indicated categories of capital then totaled. Owner's costs, contingencies and risk amounts are included in this CAPEX.

Table 21.3: Capital Cost Summary

Area	Description	Initial CAPEX (US$M)	Sustaining CAPEX (US$M)	Total (US$M)
Mine	Equipment lease payments	13.4	55.3	**68.7**
Mine	Shops and Other Surface Infrastructure	8.0	0.0	**8.0**
Mill	Processing Capital	120.7	9.6	**130.2**
G&A	Dewatering Capital	9.9	5.6	**15.5**
G&A	Royalty Purchase Option	1.0	0.0	**1.0**
Indirects	Mine Indirect, First Fills, & Owners Costs	5.2	0.0	**5.2**
Indirects	Process Plant Indirect, First Fills, & Owners Costs	11.1	0.0	**11.1**
Indirects	Engineering, Procurement & Construction Management (EPCM)	17.2	0.0	**17.2**
Indirects	Mine Contingency	10.0	0.0	**10.0**
Indirects	Process Plant Contingency	23.4	0.0	**23.4**
	Total Major Area CAPEX	219.9	70.5	**290.4**
Other	Working Capital	22.2	-22.2	**0**
Other	Surety Bond Premium	0.4	OPEX G&A	**0.4**
Other	Reclamation Bond Restricted Cash Collateral	4.6	-5.5	**-0.9**
Other	Project Closure and Rehabilitation	0.0	12.0	**12.0**
Other	Equipment Salvage Value	0.0	-16.2	**-16.2**
	Total CAPEX	**247.1**	**38.6**	**285.7**

Note: Numbers may not sum precisely due to rounding. M = Million

21.2.2 Sustaining Capital

The Sustaining CAPEX commences upon production of concentrate and continues throughout the mine life. The Sustaining costs are included in the overall Project CAPEX.

These sustaining capital costs cover several areas, including mining, Tailing Storage Facility (TSF), dewatering, and heap leach pad expansion.

21.2.3 Closure and Rehabilitation Costs

At the end of the Project life, it is required that all disturbed areas are rehabilitated, and equipment and buildings are disposed of. Closure costs have been included in the sustaining capital. Surety bond cash collateral is a component of the initial capital and the bond interest generated is then an offsetting factor as part of the sustaining capital. Major equipment is shown to be salvaged at the end of mine life.

21.2.4 Major Assumptions

The CAPEX is based on the following key assumptions:

- All relevant permits in a timely manner to meet the Project schedule.
- Quotes from Vendors for leasing equipment to be valid for budget purposes.

- Suitable backfill material is available locally. Soil conditions are adequate for foundation bearing pressures.
- Engineering and Construction activities will be carried out in a continuous program with full funding available, including contingency.
- Bulk materials such as cement, rebar, structural steel and plate, cable, cable tray, and piping are all readily available in the scheduled timeframe.
- Capital equipment is available in the timeframe shown.
- The construction and commissioning period begins upon receipt of construction permit, including Open Pit stripping, process plant, and infrastructure development.
- The production plan assumes the mill is immediately constructed and available at full capacity 2000 tpd for 365 days per year throughout the LoMP.

21.2.5 Major Exclusions

The following items were not included in the CAPEX:

- Provision for inflation, escalation, currency fluctuations and interest incurred during construction.
- Schedule delays and associated costs.
- Scope changes.
- Unidentified ground conditions.
- Extraordinary climatic events.
- Force majeure.
- Labour disputes.
- Insurance, bonding, permits and legal costs beyond that related to the closure bond.
- Schedule recovery or acceleration.
- Cost of financing, Property taxes, corporate and mining taxes, duties; and salvage values.
  - However, they are considered in the Economic Analysis.
- There will be no construction camp and catering in this CAPEX.
  - It is assumed that non-local contractors will source local accommodation.
  - It is further assumed that accommodation is available in the surrounding areas for the EPCM personnel and vendor supervisors.

21.2.6 Currencies

The base currency for the PEA is United States Dollars (US$).

Where currency exchange rates were required, they were determined by WSP based on the 2022-2024 Annual Exchange Rates - Bank of Canada and included in the CAPEX and OPEX, as shown in Table 21.4.

Table 21.4: Currency Conversion Rates

Currency	Code Name	US Dollar Equivalent to 1.00 Currency	Currency Equivalent to 1.00 US$
US$	United States Dollar	1.00	1.00
CAD	Canadian Dollar	0.75	1.34

21.2.7 Open Pit Mine CAPEX

21.2.7.1 Summary of Estimated Open Pit Mine Capital Costs

Table 21.5 presents WSP's CAPEX for the open pit mine. The estimate's accuracy level aligns with the PEA's required accuracy as stated in Item 21.1. The mine equipment capital is representative of an equipment lease.

Table 21.5: Open Pit Mine CAPEX

CAPEX Category	Initial Capital (US$K)	Sustaining Capital (US$K)	Total Capital (US$K)
Mine Equipment	13,473	54,877	68,350
Mine Infrastructure	7,962	0	7,962
Mine Equipment Indirects	3,959	381	4,340
Mine Infrastructure Indirects	1,194	0	1,194
Contingency	10,036	0	10,036
Total Mine CAPEX	**36,624**	**55,257**	**91,881**

Note: Numbers may not sum precisely due to rounding

21.2.7.2 Basis for the Open Pit Mine CAPEX

The database of cost inputs for the estimate was derived from the following sources:

- Quotations from suppliers of new equipment
- Cost data from other mines and projects
- Internal benchmarks based on past project experience

CAPEX for the open pit mine are categorized into Mine Equipment, Mine Infrastructure, Equipment Indirects, Infrastructure Indirects, and Contingency.

21.2.7.3 Assumptions

The following assumptions have been made in the preparation of the cost estimate for the open pit mine:

- Quotations obtained from vendors for leasing equipment and procuring materials are provisional and are intended solely for budgetary purposes. WSP has reviewed and deemed them reasonable for the PEA.
- Viva will be able to lease all equipment required for the open pit mine.

21.2.7.4 Exclusions and Exceptions

The mine capital estimate only includes costs for work within the defined battery limit for the open pit mine. Items associated with processing, non-mine surface infrastructure, or pre-production operating costs are excluded from the cost estimate.

21.2.8 Process & Process Infrastructure Capital Cost

21.2.8.1 Process and Process Infrastructure Capital Cost Basis

All equipment and material requirements are based on the design information described in this report. Budgetary capital cost estimates were developed based on budgetary project specific quotes or recent quotes from similar projects in KCA's files for all major and most minor equipment. Where recent quotes were not available, reasonable cost estimates or allowances were made based on cost guide data. All capital cost estimates were based on the purchase of equipment quoted new from the manufacturer or to be fabricated new. Capital cost estimates were based on the second quarter of 2025 US dollars and are considered to have an accuracy of +/ - 35%.

The process and process infrastructure cost estimate has been separated into the following disciplines, as applicable:

- Major earthworks & liner;
- Civil (concrete);
- Structural steel;
- Platework;
- Mechanical equipment;
- Piping;
- Electrical;
- Instrumentation; and
- Infrastructure & Buildings.

Pre-production process and infrastructure costs by area and by discipline are presented in Table 21.6 and Table 21.7, respectively.

Table 21.6: Process & Process Infrastructure Pre-production Costs by Area

Item	Total Supply Cost (US$M)	Install (US$M)	Grand Total (US$M)
Area 113 - Crushing	$12.0	$4.9	$16.8
Area 115 - Agglomeration & Stacking	$17.7	$5.1	$22.8
Area 118 - Grinding, Gravity, & Thickening	$10.0	$5.4	$15.4
Area 119 - CIL	$4.9	$2.3	$7.2
Area 120 - Tailings Thickener	$1.3	$1.0	$2.4
Area 121 - Filtration	$8.8	$4.8	$13.6
Area 122 - Heap Leach Pad & Ponds	$4.6	$13.6	$18.2
Area 128 - Carbon Adsorption & Handling	$6.8	$3.7	$10.4
Area 131 - Refinery	$0.0	$0.0	$0.0
Area 134 – Reagents	$0.6	$0.6	$1.3
Area 338 - Laboratory	$1.2	$0.2	$1.4
Area 360 - Power	$0.5	$0.2	$0.7
Area 362 - Water Supply, Storage & Distribution	$1.4	$0.5	$1.8
Area 368 - Compressed Air & Fuel	$1.3	$0.6	$1.9
Area 366 - Facilities	$0.7	$0.2	$0.9
Area 008 - Plant Mobile Equipment	$3.3	$0.0	$3.3
Plant & Infrastructure Total Direct Costs	**$75.1**	**$43.1**	**$118.3**
Spare Parts	$1.9		$1.9
Sub Total with Spare Parts			**$120.2**
Contingency			$23.4
Plant & Infrastructure Total Direct Costs with Contingency			**$143.6**
Indirect Field Costs			$3.6
Other Owner's Costs			$7.2
EPCM			$17.2
Initial Fills			$0.4
Total Process Plant and Leach Pad Pre-Production Capital Cost			**$171.9**
D6 Dozer – Heap Leach Pad Maintenance			$0.5
Total Processing Pre-Production Capital Cost			**$172.4**

Table 21.7: Process & Process Infrastructure Pre-production Costs by Discipline

Item	Total Supply Cost (US$M)	Install (US$M)	Grand Total (US$M)
Major Earthworks & Liner		$10.5	$10.5
Civils (Supply & Install)		$9.7	$9.7
Structural Steelwork (Supply & Install)	$3.7	$1.1	$4.8
Platework (Supply & Install)	$5.2	$1.6	$6.8
Mechanical Equipment	$47.3	$13.6	$61.0
Piping	$4.8	$1.4	$6.2
Electrical	$9.6	$3.5	$13.1
Instrumentation	$1.8	$1.0	$2.8
Infrastructure	$2.8	$0.5	$3.3
Spare Parts			$1.9
Contingency			$23.4
Plant Total Direct Costs	**$75.1**	**$43.1**	**$143.6**

Freight, sales tax, and installation costs were also considered for each discipline. Freight costs were based on loads as bulk freight and were estimated at 10% of the equipment cost.

Sales tax for Nye County in Nevada is 7.6% and was applied to the supply cost of all equipment and materials.

Installation estimates were based on the equipment type and included all installation labour, tools and equipment usage at an average hourly installation rate of US$114 based on KCA's experience from recent projects.

21.2.8.2 Process Plant Major Earthworks & Liner

Earthworks and liner quantities for the Project were estimated by KCA for all Project areas. Earthworks and liner supply and installation were assumed to be performed by contractors with costs for the heap leach estimated based on an overall cost per m^2 based on KCA's experience with recent similar projects. Costs for miscellaneous earthworks have been estimated as allowances based on recent similar projects.

21.2.8.3 Process Plant Civils

Civils include detailed earthworks and concrete. Concrete quantities were estimated by KCA based on layouts, similar equipment installations, vibrating equipment, major equipment weights and on slab areas. Unit costs for concrete supply, which include production (supply of aggregates, water, and cement, batching and mixing), and delivery of concrete and concrete installation which include all excavations, formwork, rebar, placement, and curing were based on recent contractor quotes in KCA's files.

21.2.8.4 Process Plant Structural Steel

Costs for structural steel, including steel grating, and handrails was estimated based on a percentage of the mechanical equipment cost benchmarked against other recent similar projects.

21.2.8.5 Process Plant Platework

The platework discipline includes costs for the supply and installation of steel tanks, bins, and chutes. Platework costs were estimated based on preliminary weights and recent supplier cost information for similar tanks or included as part of complete equipment supply packages.

21.2.8.6 Process Plant Mechanical Equipment

Costs for mechanical equipment were based on a detailed equipment list developed of all major equipment for the process. Costs for all major and most minor equipment items were based on budgetary quotes from suppliers. Where Project specific supplier quotes were not available, reasonable allowances were made based on recent quotes from KCA's files or cost guide data. All costs assume equipment purchased new from the manufacturer or to be fabricated new.

Installation costs for mechanical equipment were based on estimated installation hours and hourly contractor rates from KCA's experience on recent similar projects.

21.2.8.7 Process Plant Piping

Major piping, including heap irrigation and gravity solution collection pipes, were based on recent estimates from similar sized projects in the United States. An allowance of US$600,000 was included for water delivery and distribution. Additional ancillary piping, fittings, and valve costs were estimated on a percentage basis of the mechanical equipment supply costs by area ranging from 0% to 20%.

Installation costs for piping were based on estimated installation hours and hourly contractor rates from KCA's experience on recent similar projects.

21.2.8.8 Process Plant Electrical

Electrical costs were estimated as percentages of the mechanical equipment supply cost for each process area and range between 0% and 30% based on benchmarked costs from recent similar projects.

Installation of electrical equipment and ancillary electrical items were estimated based on estimated installation hours and hourly contractor rates from KCA's experience on recent projects.

21.2.8.9 Process Plant Instrumentation

Instrumentation costs were primarily estimated as percentages of the mechanical equipment supply cost for each process area and range between 0% and 8%.

21.2.8.10 Process Plant Infrastructure & Buildings

New buildings for the Project will include an administration office trailer, process office trailer, mill facility and laboratory. Costs for the buildings were based on recent budgetary quotes from suppliers and reasonable allowances based on KCA experience.

21.2.8.11 Process Mobile Equipment

Process mobile equipment includes a 2-ton forklift, 5-ton boom truck, 10-ton telehandler, mechanic service truck, flatbed truck, two each skid steer, heap leach pad dozer (CAT D6 or equivalent), crusher area loader (CAT 988 or equivalent), personnel van and 6 each ¾-ton pickup trucks. Costs for mobile equipment are included in the mechanical equipment discipline.

21.2.8.12 Process Plant Spare Parts

Spare parts costs were estimated at 4% of the mechanical equipment supply costs.

21.2.8.13 Process Plant Construction Indirect Costs & Other Owner Construction Costs

Indirect construction field costs include temporary construction facilities, construction services, quality control, survey support, warehouse and fenced yards, support equipment, etc. These costs were estimated based on the preliminary construction schedule, recent contractor quotes, and reasonable allowances based on KCA's recent experience. Most of the construction indirect costs will be the responsibility of the construction contractors and these costs were built into the contractor quotes.

Other Owner's construction costs are intended to cover the owner's team costs for labour, offices, home office support, travel during construction, software, taxes and permit fees, legal fees and workplace health safety during construction. Other Owner's costs were estimated based on the preliminary construction schedule using reasonable allowances based on KCA's recent experience on other similar projects.

21.2.8.14 Process Plant Engineering, Procurement & Construction Management

The estimated EPCM costs for the development, construction, and commissioning are based on a percentage of the direct capital cost. The total EPCM cost was estimated at 12% of the process and infrastructure direct costs.

The EPCM costs cover services and expenses for the following areas:

- Project Management.
- Detailed Engineering.
- Engineering Support.
- Procurement.
- Construction Management.
- Commissioning.
- Vendors Reps.

21.2.8.15 Process Plant Contingency

Contingency for the Process Pant portion of the Project was applied to the total direct costs by discipline and category. Contingency was applied ranging from 15% to 25% for process and infrastructure. The overall contingency for process and infrastructure was estimated at 19.5% of the direct costs.

Contingency for process and infrastructure sustaining capital was estimated at 20% of the sustaining costs. Sustaining capital contingency is included in the sustaining cost estimate.

21.2.8.16 Process Plant Initial Fills

The initial fills consist of consumable items stored on site at the outset of operations, which includes NaCN, lime, cement, activated carbon, antiscalants, flocculant and grinding media. An allowance of US$350,000 is considered for the project initial fills.

21.2.8.17 Process Plant Sustaining Capital

Process sustaining capital includes the Phase 2 heap leach pad expansion, which is assumed to be constructed during Year 2 of operation. Sustaining capital for the process, including contingency, is estimated at US$9.6M.

21.2.8.18 Process Plant Exclusions

The following capital cost considerations have been excluded from the scope of supply and estimate:

- Finance charges and interest during construction.
- Escalation costs.
- Currency exchange fluctuations.

21.2.9 Dewatering Capital Costs

Capital Cost to construct the wellfield is estimated at US$15.5M

21.2.10 Closure and Rehabilitation Costs

At the end of the Project life, it is required that all disturbed areas are rehabilitated, and equipment and buildings are disposed of. Closure costs have been included in the sustaining capital at US$12.0M.

21.2.11 Pre-Production Operating Costs

Pre-production operating costs are estimated for Year 0 of the project. These costs consist entirely of a surety bond premium that must be paid annually in the amount of US$0.38M per year.

21.3 Operating Costs

21.3.1 Mine Operating Costs

Mine Operating costs were estimated by WSP and consist of costs associated with mine equipment operations, drill and blast activities, auxiliary equipment operations, and staffing of the mine. Table 21.8 presents the mining costs on a LoM total basis and per-tonne mined basis.

Table 21.8: LoM and Unit Mine OPEX

OPEX Category	LoM Total OPEX (US$M)	Average OPEX (US$/t mined)
Loading	14.1	0.12
Hauling	57.8	0.50
Production Drilling	16.6	0.14
Pre-Split Drilling	3.4	0.03
Blasting	33.6	0.29
Auxiliary Equipment	23.6	0.20
Mine Staffing	78.8	0.67
Total Mine OPEX	**227.9**	**1.95**

Table 21.9 presents the mining costs in millions of US$ on an annualized basis over the LoM.

Table 21.9: Annual Mine OPEX

OPEX	Total US$M	Year						
		1	2	3	4	5	6	7
Loading	**14.1**	2.6	2.8	1.9	1.9	1.8	1.9	1.1
Hauling	**57.8**	9.8	9.9	7.8	7.8	7.7	9	5.7
Production Drilling	**16.6**	2	3.1	2.2	2.1	2.6	2.8	1.8
Pre-Split Drilling	**3.5**	0.2	0.5	0.7	0.6	0.5	0.5	0.5
Blasting	**33.6**	4.1	6.4	4,.4	4.3	5.2	5.5	3.6
Auxiliary Equipment	**23.6**	3.8	4	3.3	3,4	3.2	3.5	2.4
Mine Staffing	**78.8**	12.1	13.1	11.1	11.1	11.1	12.1	8.1
Total OPEX	**227.9**	34.7	39.9	31.5	31.1	32	35.3	23.3
Total Tonnes Mined	**116.7**	22	23.8	15.7	15.6	14.9	15.7	9.1
OPEX Cost per Total Tonne Mined	**1.95**	1.58	1.67	2.01	1.99	2.15	2.26	2.57

Figure 21.1 shows the annual mining OPEX by cost category.

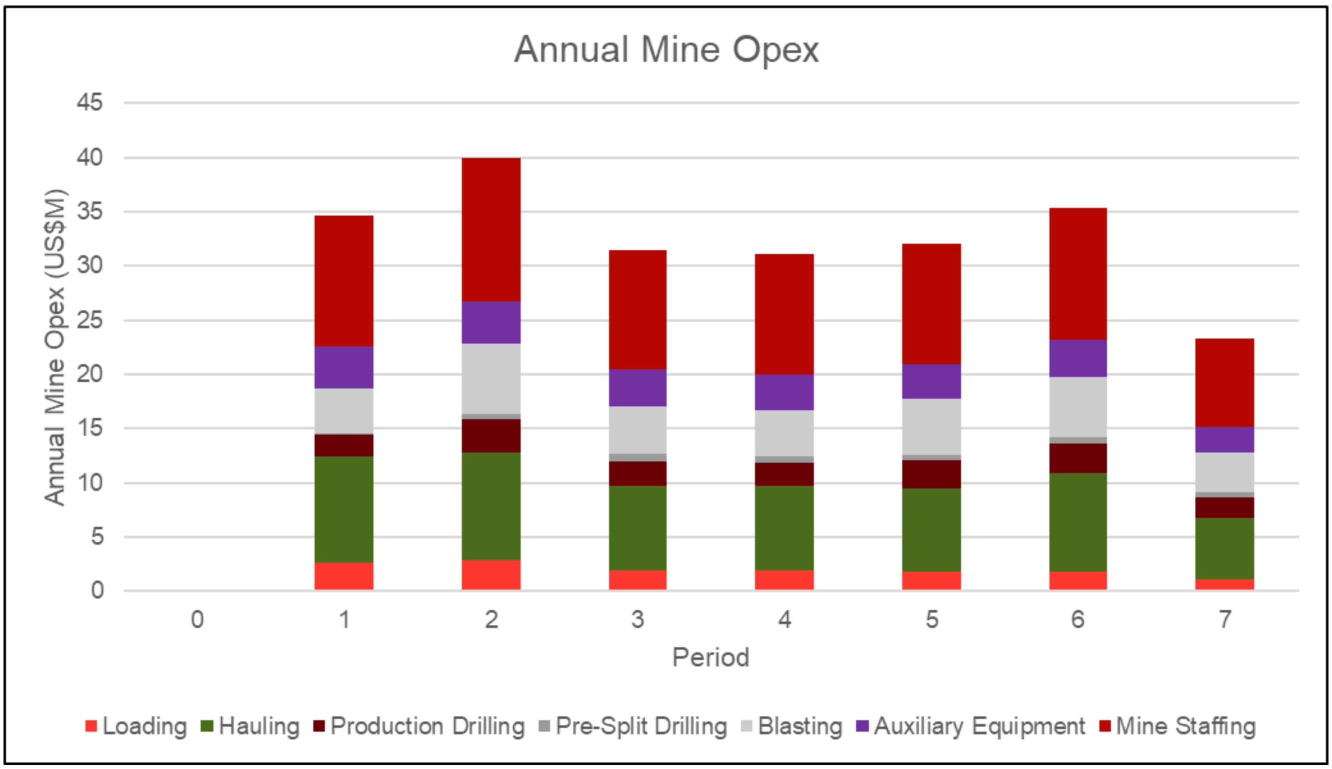


Figure 21.1: Annual Mine OPEX

Figure 21.2 shows the annual mining unit cost in comparison to the annual tonnes mined.

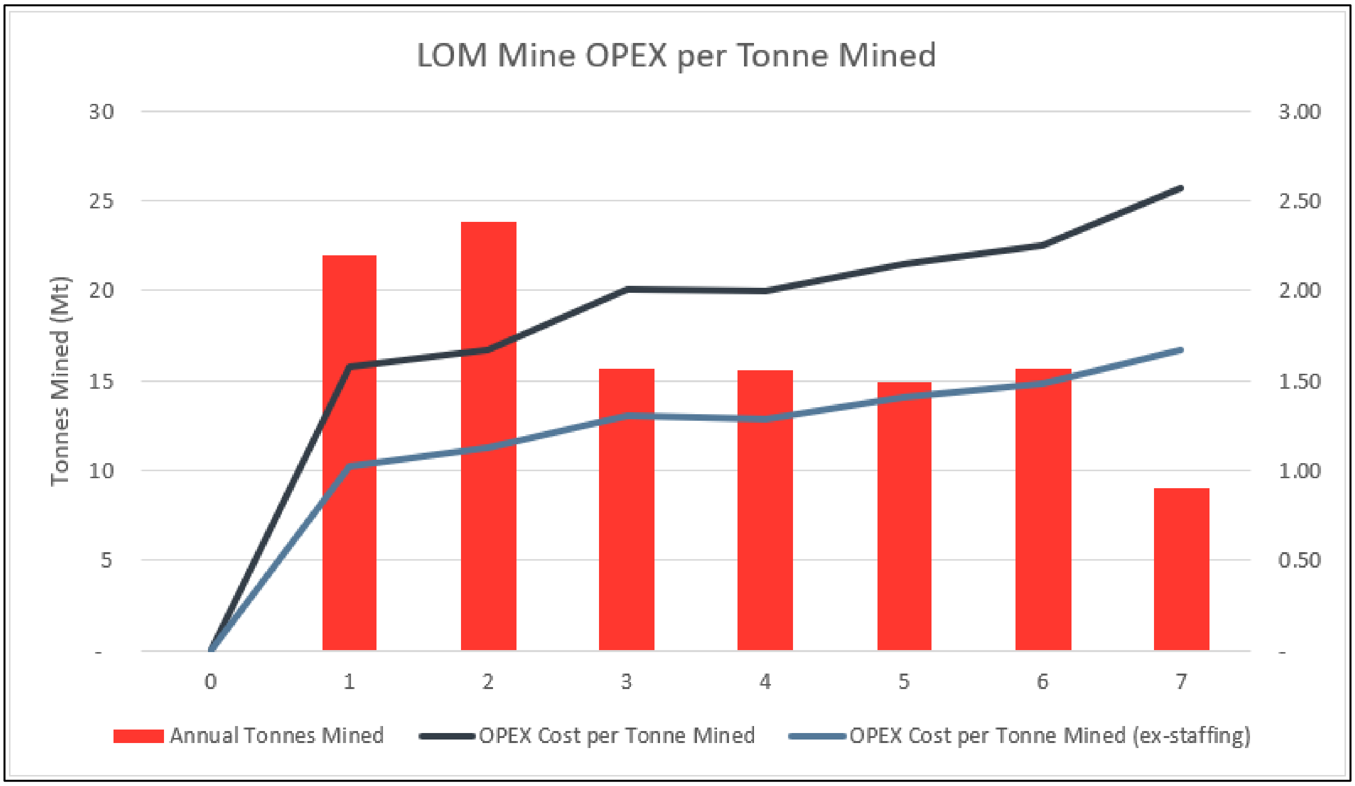


Figure 21.2: LoM Mine OPEX per Tonne Mined

The mining OPEX estimate generally excludes:

- Costs for processing, dewatering, and general and administrative costs.
- Pre-production operating costs.
- Any factors for contingency or escalation.

21.3.2 Process Operating Costs

Process Operating costs were estimated by KCA from first principles. Labour costs were estimated using project specific staffing, salary and wage and benefit requirements. Unit consumptions of materials, supplies, power, water and delivered supply costs were also estimated. Average processing costs are presented in Table 21.10 and Table 21.11 for the heap leach and mill material, respectively. Operating Costs for shared services, including crushing, reagents, support services, and select labour, have been distributed based on tonnage.

Table 21.10: Average Heap Leach Process and Support Services Operating Costs

	Units	Cost Type	Qty	Unit Cost (US$)	Total Annual Costs (US$K)	US$/Tonne
Labor - All Process Areas						
Process Labor	Persons	Fixed	42		$5,192	$1.78
Laboratory Labor	Persons	Fixed	6		$778	$0.27
Labor - All Process Areas Subtotal		Total			$5,970	$2.04
Area 113 - Crushing						
Power	kWh/t	Variable	4.76	$0.10	$1,433	$0.49
988 Loader	h/mo	Fixed	292	$102.15	$30	$0.01
Wear	$/t	Variable			$730	$0.25
Overhaul / Maintenance	$/t	Variable			$1,022	$0.35
Area 113 - Crushing Subtotal		Total			$3,215	$1.10
Area 115 - Agglomeration & Stacking						
Power	kWh/t	Variable	1.98	$0.10	$596	$0.20
Maintenance Supplies	$/t	Variable			$292	$0.10
Area 115 - Agglomeration & Stacking Subtotal		Total			$888	$0.30
Area 122 - Heap Leach Pad & Ponds						
Power	kWh/t	Variable	1.18	$0.10	$355	$0.12
Piping/Drip tubing	$/t	Variable			$88	$0.03
Maintenance Supplies	$/t	Variable			$29	$0.01
Area 122 - Heap Leach Pad & Ponds Subtotal		Total			$472	$0.16
Area 128 - Carbon Adsorption & Handling						
Power	kWh/t	Variable	0.24	$0.10	$71	$0.02
Carbon	t/year	Variable	17.49	$2,680	$47	$0.02
Misc. Operating Supplies	$/t	Variable			$73	$0.03
Maintenance Supplies	$/t	Variable			$58	$0.02
Carbon Treatment (Toll Strip + Regen)	$ (dry tonne)	Variable	325.48	$4,460	$1,452	$0.50
Carbon Treatment (Just Refiners burn fee, wet tons)	$ (dry tonne)	Variable		-	$0	$0.00
Area 128 - Carbon Adsorption & Handling Subtotal		Total			$1,701	$0.58

Table 21.10: Average Heap Leach Process and Support Services Operating Costs, continued

	Units	Cost Type	Qty	Unit Cost (US$)	Total Annual Costs (US$K)	US$/Tonne
Area 134 - Reagents						
Power	kWh/t	Variable	0.06	$0.10	$19	$0.01
Cyanide Low-Grade Ore Heap	kg/t	Variable	0.26	$3.10	$2,333	$0.80
Cyanide High-Grade Ore Mill	kg/t	Variable		$3.10	$0	$0.00
Lime	kg/t	Variable		$0.38	$0	$0.00
Cement	kg/t	Variable	4	$0.25	$2,920	$1.00
Flocculant	kg/t	Variable		$6.50	$0	$0.00
Caustic	kg/year	Variable	25,000	$0.81	$20	$0.01
Antiscalant	kg/year	Variable	43,225	$3.78	$163	$0.06
Maintenance Supplies	$/t	Variable			$29	$0.01
Area 134 - Reagents Subtotal		Total			$5,484	$1.88
Area 338 - Laboratory						
Power	kWh/t	Variable	0.3	$0.10	$91	$0.03
Assays, Solids	No./day	Fixed	150	$6.00	$329	$0.11
Assays, Solutions	No./day	Fixed	100	$3.00	$110	$0.04
Miscellaneous Supplies	$/t	Variable			$73	$0.03
Area 338 - Laboratory Subtotal		Total			$602	$0.21
Area 360 - Power						
Power	kWh/t	Variable	-	$0.10	$0	$0.00
Overhaul / Maintenance	$/t	Variable			$15	$0.01
Area 360 - Power Subtotal		Total			$15	$0.01
Area 362 - Water Supply, Storage & Distribution						
Power	kWh/t	Variable	0.31	$0.10	$93	$0.03
Maintenance Supplies	$/t	Variable			$29	$0.01
Area 362 - Water Supply, Storage & Distribution Subtotal		Total			$123	$0.04

Table 21.10: Average Heap Leach Process and Support Services Operating Costs, continued

	Units	Cost Type	Qty	Unit Cost (US$)	Total Annual Costs (US$K)	US$/Tonne
Area 368 - Compressed Air & Fuel						
Power	kWh/t	Variable	0.48	$0.10	$145	$0.05
Maintenance Supplies	$/t	Variable			$29	$0.01
Area 368 - Compressed Air & Fuel Subtotal		Total			$174	$0.06
Area 366 - Facilities						
Facilities / Infrastructure					$0	
Power - Buildings/Misc.	kWh/t	Fixed	0.32	$0.10	$95	$0.03
Heating - WH/Admin	LPG gallon/yr	Fixed	20,000.00	$1.76	$35	$0.01
Heating - Truckshop	LPG gallon/yr	Fixed	27,200.00	$1.76	$48	$0.02
Area 366 - Facilities Subtotal		Total			$178	$0.06
Area 008 - Plant Mobile Equipment						
Fork Lift	h/mo	Fixed	96	$7.45	$9	$0.00
Boom Truck	h/mo	Fixed	64	$47.31	$36	$0.01
Mechanic Service Truck	h/mo	Fixed	144	$45.83	$79	$0.03
Backhoe/Loader	h/mo	Fixed	64	$24.42	$19	$0.01
Pickup Truck	h/mo	Fixed	960	$24.86	$286	$0.10
Telehandler	h/mo	Fixed	64	$30.37	$23	$0.01
Flatbed Truck	h/mo	Fixed	96	$22.21	$26	$0.01
Skid Steer / Bobcat Loader	h/mo	Fixed	144	$9.00	$16	$0.01
Area 008 - Plant Mobile Equipment Subtotal		Total			$494	$0.17
Total					$19,316	**$6.62**
Fixed Costs					$7,110	$2.44
Variable Costs					$12,206	$4.18

Notes: kWh = kilowatt hour, t = tonne, h = hour, mo = month, yr = year, kg = kilogram, No. = number, LPG = Liquified Petroleum Gas

Table 21.11: Average Mill Process and Support Services Operating Costs

	Units	Cost Type	Qty	Unit Cost (US$)	Total Annual Costs (US$K)	US$/Tonne
Labor - All Process Areas						
Process Labor	Persons	Fixed	26		$3,117	$4.27
Laboratory Labor	Persons	Fixed	2		$194	$0.27
Labor - All Process Areas Subtotal		Total			$3,311	$4.54
Area 113 - Crushing						
Power	kWh/t	Variable	4.76	$0.10	$358	$0.49
988 Loader	h/mo	Fixed	73	$102.15	$7	$0.01
Wear	$/t	Variable			$183	$0.25
Overhaul / Maintenance	$/t	Variable			$256	$0.35
Area 113 - Crushing Subtotal		Total			$804	$1.10
Area 118 - Grinding, Gravity, & Thickening						
Power	kWh/t	Variable	13.51	$0.10	$1,016	$1.39
Grinding Media (Ball Mill)	kg/kWh	Variable	0.09	$1.42	$1,075	$1.47
Mill Liners (Ball Mill)	kg/kWh	Variable	0.01	$4.03	$238	$0.33
Wear	$/t	Variable			$73	$0.10
Overhaul / Maintenance	$/t	Variable			$146	$0.20
Area 118 - Grinding, Gravity, & Thickening Subtotal		Total			$2,548	$3.49
Area 119 - CIL						
Power	kWh/t	Variable	3.19	$0.10	$240	$0.33
Wear	$/t	Variable			$110	$0.15
Area 119 - CIL Subtotal		Total			$349	$0.48
Area 120 - Tailings Thickener						
Power	kWh/t	Variable	0.4	$0.10	$30	$0.04
Wear	$/t	Variable			$37	$0.05
Area 120 - Tailings Thickener Subtotal		Total			$67	$0.09

Table 21.11: Average Mill Process and Support Services Operating Costs, continued

	Units	Cost Type	Qty	Unit Cost (US$)	Total Annual Costs (US$K)	US$/Tonne
Area 121 - Filtration						
Power	kWh/t	Variable	5.46	$0.10	$410	$0.56
Filter Cloths	$/t	Variable			$219	$0.30
966 Loader	h/mo	Fixed	620.5	$29.51	$220	$0.30
Haul Trucks	h/mo	Fixed	-	$29.19	$0	$0.00
Dozer	h/mo	Fixed	310.25	$50.52	$188	$0.26
Maintenance Supplies	$/t	Variable			$110	$0.15
Operating Supplies	$/t	Variable			$7	$0.01
Area 121 - Filtration Subtotal		Total			$1,154	$1.58
Area 128 - Carbon Adsorption & Handling						
Power	kWh/t	Variable	0.24	$0.10	$18	$0.02
Carbon	t/year	Variable	13.83	$2,680	$37	$0.05
Misc. Operating Supplies	$/t	Variable			$18	$0.03
Maintenance Supplies	$/t	Variable			$15	$0.02
Carbon Treatment (Toll Strip & Regen)	$ (dry tonne)	Variable	257.47	$4,460	$1,148	$1.57
Carbon Treatment (burn fee, wet tons)	$ (dry tonne)	Variable	-		0	$0.00
Area 128 - Carbon Adsorption & Handling Subtotal		Total			$1,236	$1.69
Area 134 - Reagents						
Power	kWh/t	Variable	0.06	$0.10	$5	$0.01
Cyanide LG Ore Heap	kg/t	Variable		$3.10	$0	$0.00
Cyanide HG Ore Mill	kg/t	Variable	0.58	$3.10	$1,313	$1.80
Lime	kg/t	Variable	0.6	$0.38	$166	$0.23
Cement	kg/t	Variable		$0.25	$0	$0.00
Flocculant	kg/t	Variable	0.1	$6.50	$475	$0.65
Caustic	kg/year	Variable		$0.81	$0	$0.00
Antiscalant	kg/year	Variable		$3.78	$0	$0.00
Maintenance Supplies	$/t	Variable			$7	$0.01
Area 134 - Reagents Subtotal		Total			$1,966	$2.69

Table 21.11: Average Mill Process and Support Services Operating Costs, continued

	Units	Cost Type	Qty	Unit Cost (US$)	Total Annual Costs (US$K)	US$/Tonne
Area 338 - Laboratory						
Power	kWh/t	Variable	0.3	$0.10	$23	$0.03
Assays, Solids	No./day	Fixed	100	$6.00	$219	$0.30
Assays, Solutions	No./day	Fixed	50	$3.00	$55	$0.08
Miscellaneous Supplies	$/t	Variable			$18	$0.03
Area 338 - Laboratory Subtotal		Total			$315	$0.43
Area 360 - Power						
Power	kWh/t	Variable	-	$0.10	$0	$0.00
Overhaul / Maintenance	$/t	Variable			$4	$0.01
Area 360 - Power Subtotal		Total			$4	$0.01
Area 362 - Water Supply, Storage & Distribution						
Power	kWh/t	Variable	0.31	$0.10	$23	$0.03
Maintenance Supplies	$/t	Variable			$7	$0.01
Area 362 - Water Supply, Storage & Distribution Subtotal		Total			$31	$0.04
Area 368 - Compressed Air & Fuel						
Power	kWh/t	Variable	0.48	$0.10	$36	$0.05
Maintenance Supplies	$/t	Variable			$7	$0.01
Area 368 - Compressed Air & Fuel Subtotal		Total			$44	$0.06
Area 366 - Facilities						
Facilities / Infrastructure						
Power - Buildings/Misc.	kWh/t	Fixed	0.32	$0.10	$24	$0.03
Heating - WH/Admin	LPG gallon/yr	Fixed	5,000.00	$1.76	$9	$0.01
Heating - Truckshop	LPG gallon/yr	Fixed	6,800.00	$1.76	$12	$0.02
Area 366 - Facilities Subtotal		Total			$45	$0.06

Table 21.11: Average Mill Process and Support Services Operating Costs, continued

	Units	Cost Type	Qty	Unit Cost (US$)	Total Annual Costs (US$K)	US$/Tonne
Area 008 - Plant Mobile Equipment						
Fork Lift	h/mo	Fixed	24	$7.45	$2	$0.00
Boom Truck	h/mo	Fixed	16	$47.31	$9	$0.01
Mechanic Service Truck	h/mo	Fixed	36	$45.83	$20	$0.03
Backhoe/Loader	h/mo	Fixed	16	$24.42	$5	$0.01
Pickup Truck	h/mo	Fixed	240	$24.86	$72	$0.10
Telehandler	h/mo	Fixed	16	$30.37	$6	$0.01
Flatbed Truck	h/mo	Fixed	24	$22.21	$6	$0.01
Skid Steer / Bobcat Loader	h/mo	Fixed	36	$9.00	$4	$0.01
Area 008 - Plant Mobile Equipment Subtotal		Total			$123	$0.17
Total					**$11,995**	**$16.43**
Fixed Costs					$4,168	$5.71
Variable Costs					$7,827	$10.72

Notes: kWh = kilowatt hour, t = tonne, h = hour, mo = month, yr = year, kg = kilogram, No. = number, LPG = Liquified Petroleum Gas

Operating costs were estimated based on Q2 2025 US dollars and are presented with no added contingency based upon the design and operating criteria present in this report and are considered to have an accuracy of +/- 35%. Sales tax was not included in the operating cost estimate.

The operating costs presented are based upon the ownership of all process production equipment and site facilities, including the on-site laboratory. The owner will employ and direct all operating maintenance and support personnel for all site activities.

Where specific data do not exist, cost allowances were based upon consumption and operating requirements from other similar properties for which reliable data exist. Freight costs were estimated where delivered prices were not available.

21.3.2.1 Process Plant Personnel & Staffing

Staffing requirements for process were estimated by KCA based on experience with similar sized operations and input from Viva. Total process personnel were estimated at 76 persons, including 8 laboratory workers. The labour estimate considers 18 dedicated workers for the heap, 20 dedicated workers for the mill and 38 shared workers.

21.3.2.2 Process Plant Power

Power usage for the process and process-related infrastructure was derived from estimated connected loads assigned to powered equipment from the mechanical equipment list. Equipment power demands under normal operation were assigned and coupled with estimated on-stream times to determine the average energy usage and cost. Power requirements for the Project are presented in Item 18.0.

Power is assumed to be supplied by tying into an existing transmission line near the project site. The power cost is estimated at US$0.103 /kilowatt hour (kWh) and is based on recent published rates.

21.3.2.3 Process Plant Consumable Items

Operating supplies were estimated based upon unit costs and consumption rates predicted by metallurgical tests and have been broken down by area. Freight costs were included in all operating supply and reagent estimates. Reagent consumptions were estimated from test work and from design criteria considerations. Other consumable items were estimated by KCA based on experience with other similar operations.

Operating costs for consumable items were distributed based on tonnage and carbon batches, as appropriate.

21.3.2.4 Process Laboratory

Fire assaying and solution assaying of samples will be conducted in the on-site laboratory. It was estimated that approximately 150 solids assays and 100 solutions assays will need to be performed each day for the heap and 100 solids assays and 50 solution assays for the mill.

21.3.2.5 Process Plant Miscellaneous Operating & Maintenance Supplies

Overhaul and maintenance of equipment along with miscellaneous operating supplies for each area were estimated as allowances based on tons processed. The allowances for each area were developed based on published data as well as KCA's experience with similar operations.

21.3.2.6 Carbon Toll Processing

The project considers toll-processing or toll-stripping of carbon loaded on site. The costs for toll-processing have been estimated at US$4,460 per tonne of carbon treated and includes transportation to and from the toll processing facility, carbon stripping and recovery of metals to Doré, carbon acid washing and thermal regeneration.

21.3.2.7 Process Mobile Support Equipment

Mobile and support equipment will be required for the process and include a 2-ton forklift, 5-ton boom truck, mechanic service truck, backhoe loader (CAT 430E or equivalent), heap leach pad dozer (CAT D6 or equivalent), dry-stacked tailings dozer (CAT D6 or equivalent), crusher area loader (CAT 988 or equivalent), personnel van, six each ¾ ton pickup trucks, 10-ton telehandler, flatbed truck and skid steer. The costs to operate and maintain each piece of equipment were estimated primarily using published information and project specific fuel costs. Where published information was not available, allowances were made based on KCA's experience from similar operations.

21.3.3 General and Administrative Costs

General and Administrative (G&A) costs include administration labour costs and expenses associated with the project. G&A labour requirements were estimated by KCA with input from Viva and include 10 persons. G&A expenses, including surety premium are expected to average US$4.4M per year and include costs for off site offices, insurance, office supplies, communications, environmental and social management, health and safety supplies, security, travel, and legal expenses. For the cost estimate, G&A expenses were represented primarily as fixed costs.

21.3.4 Dewatering Operating Costs

- Dewatering O&M cost range is estimated between US$671K and US$1,251K annually.
- The system design is based on a preliminary estimate of dewatering rates and cost.
- Dewatering rates will change depending on mining rate driving the required rate of drawdown.

22.0 Economic Analysis

The economic analysis of the Viva Tonopah Project is preliminary in nature and per allowances for PEA level of study, includes Inferred Mineral Resources, which are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. As a result, there is no certainty that this 2025 PEA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The economic analysis presented in this Item contains forward-looking information under Canadian securities law. The results of the analysis rely on inputs that are subject to known and unknown risks, uncertainties, and other factors, which may cause actual results to differ materially from those presented here.

22.1 General

The economic analysis is based on the discounted cash flow (DCF) method that measures the pre-tax and after-tax metrics of future cash flow streams. Current United States and Nevada tax rules were used to assess corporate tax liabilities. The key metrics determined in the analysis are the NPV at a discount rate of 5%, the Internal Rate of Return (IRR), and the Payback Period. A sensitivity analysis was carried out to assess the impact of variations in Au price, CAPEX, and OPEX on the financial metrics.

For the purposes of the evaluation, it is assumed that the operations are established within a single corporate entity. The Project has been evaluated on an unlevered, all-equity basis.

The production schedule used in this analysis is based on the LoM Mine and Process Plan detailed in Items 16 and 17. The economic analysis is developed in terms of financial years with appropriate adjustments made to the production schedules to convert the data from mine plan years. The capital and operating costs are taken from the estimates detailed in Item 21.0.

All costs and pricing are in Q2 2025 US Dollars. No provision is made for the effects of inflation in this analysis.

22.2 Forward Looking Information

The results of the economic analyses discussed in this Item represent forward-looking information as defined under Canadian securities law. The results of the analysis depend on inputs that are subject to known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented here. Forward-looking information includes assumptions and estimations of:

- Price of Au.
- Amount of mineralized material and material grade.
- Proposed mine production plan.
- Mining dilution and mining recovery.
- Geotechnical or hydrogeological considerations during mining.
- Process plant production plan.
- Recovery rates of Au in the processing plant.

- Ability of plant, equipment, processes to operate as anticipated.
- Capital costs of equipment, installations and heap leach pad.
- Sustaining and operating costs.
- Closure costs.
- Unforeseen reclamation expenses.
- Environmental, social, and licensing risks.
- Taxation policy and tax rate.
- Royalty agreements.
- Cost inflation.
- Ability to maintain social license to operate.
- Unrecognized environmental risks.

22.3 Economic Criteria

The data that forms the basis for the LoM economic analysis are outlined in Table 22.1.

Table 22.1: LoM Economic Analysis Parameters

Parameter	Units	Values
Recoveries		
Au Mill Argillite	%	95.0
Au Mill Volcanics	%	90.0
Au Heap Leach Argillite	%	75.0
Au Heap Leach Volcanics	%	75.0
Ag Mill Argillite	%	36.0
Ag Mill Volcanics	%	38.0
Ag Heap Leach Argillite	%	12.0
Ag Heap Leach Volcanics	%	16.5
Heap Leach Delay to Next Year	%	13.0
Payables		
Au Payable	%	99.9
Ag Payable	%	98.0
Metal Prices		
Au Price	US$/oz	2,400.00
Ag Price	US$/oz	27.70
Treatment Charges		
Au Refining Cost	$/oz	2.00
Ag Refining Cost	$/oz	0.00

Parameter	Units	Values
Royalties		
Au Royalty	%	1.0
Ag Royalty	%	1.0
Operating Costs		
Mine OPEX	US$/tonne mined	1.95
Mine OPEX	US$/tonne processed	9.67
Mill Processing OPEX	US$/tonne milled	16.43
Heap Leach Processing OPEX	US$/tonne leached	6.62
Residual Heap Leach OPEX LoM-End	US$M	4.3
Dewatering OPEX	US$M/year	0.7 to 1.3
G&A OPEX	US$M/year	4.0 + Surety Bond Premium
Capital Costs		
Mine Capital Direct Costs	US$M	21.4
Mine Capital Indirect Costs	US$M	5.2
Processing Capital Direct Costs	US$M	120.6
Processing Capital Indirect Costs	US$M	11.1
Dewatering Capital Initial	US$M	9.9
NSR Royalty Buyout Option Exercised	US$M	1.0
Engineering, Procurement, Construction Management (EPCM)	US$M	17.2
Contingency	US$M	33.4
Total Initial Capital	US$M	219.9
Mine Capital Sustaining	US$M	55.3
Processing CAPEX Sustaining	US$M	9.6
Dewatering Capital Sustaining	US$M	5.6
Total Sustaining Capital	US$M	70.4
Mining Equipment Salvageable	US$M	13.1
Processing Equipment Salvageable	US$M	47.3
Processing Equipment Salvage Value	%	10%
Other Costs		
Environmental Bonding	US$M	23.7
Bond Cash Collateral	%	20%
Surety Bond Premium	US$M	0.4
Interest on Bond Cash Collateral	%	2%
Closure and Reclamation Cost	US$M	12.0
Working Capital in Year 1	day	45
Working Capital Annual Release	%	20%
Project Parameters		
Mine Life	years	7

Parameter	Units	Values
Total Tonnes Mined	Mt	115.1
Tonnes Mineral Resources Mined	Mt	23.6
Au Grade Mined (LoM Average)	g/t Au	0.63
Ag Grade Mined (LoM Average)	g/t Ag	2.42
Strip Ratio (LoM Average)	Waste to Mined Mineral	3.89
Annual Mill Throughput	tpa	730,000
Mill Feed Grade (LoM Average)	g/t Au	1.75
Heap Leach Throughput	tpa peak	2,920,000
Heap Leach Feed Grade (LoM Average)	g/t Au	0.37

Notes: Mt = million tonnes, g/t = grams per tonne, tpa = tonnes per annum

22.4 Cash Flow Analysis

For an Au price of US$2,400/oz and an Ag price of US$27.70/oz, the cashflow analysis results in the following pre-tax and after-tax metrics in Table 22.2:

Table 22.2: Project Cash Flow Analysis Summary

Description	Unit	Pre-Tax	Post-Tax
NPV @ 5%	US$M	138.6	111.6
IRR	%	20.6	17.6
Payback Period	Years	3.3	3.6
Cash Cost	US$/oz Au payable	1,164	
AISC[1]	US$/oz Au payable	1,269	

[1]All-in sustaining costs are a non-GAAP (Generally Accepted Accounting Principles) financial measure or ratio that have no standardized meaning under International Financial Reporting Standards Accounting Standards (IFRS) and may not be comparable to similar measures used by other issuers. As the Project is not in production, Viva does not have historical non-GAAP financial measures nor historical comparable measures under IFRS, and therefore the foregoing prospective non-GAAP financial measures or ratios may not be reconciled to the nearest comparable measures under IFRS.

Figure 22.1 presents the annual and cumulative cash flow of the project on a post-tax basis and Table 22.3 summarizes the project's annual mine plan and cash flows:

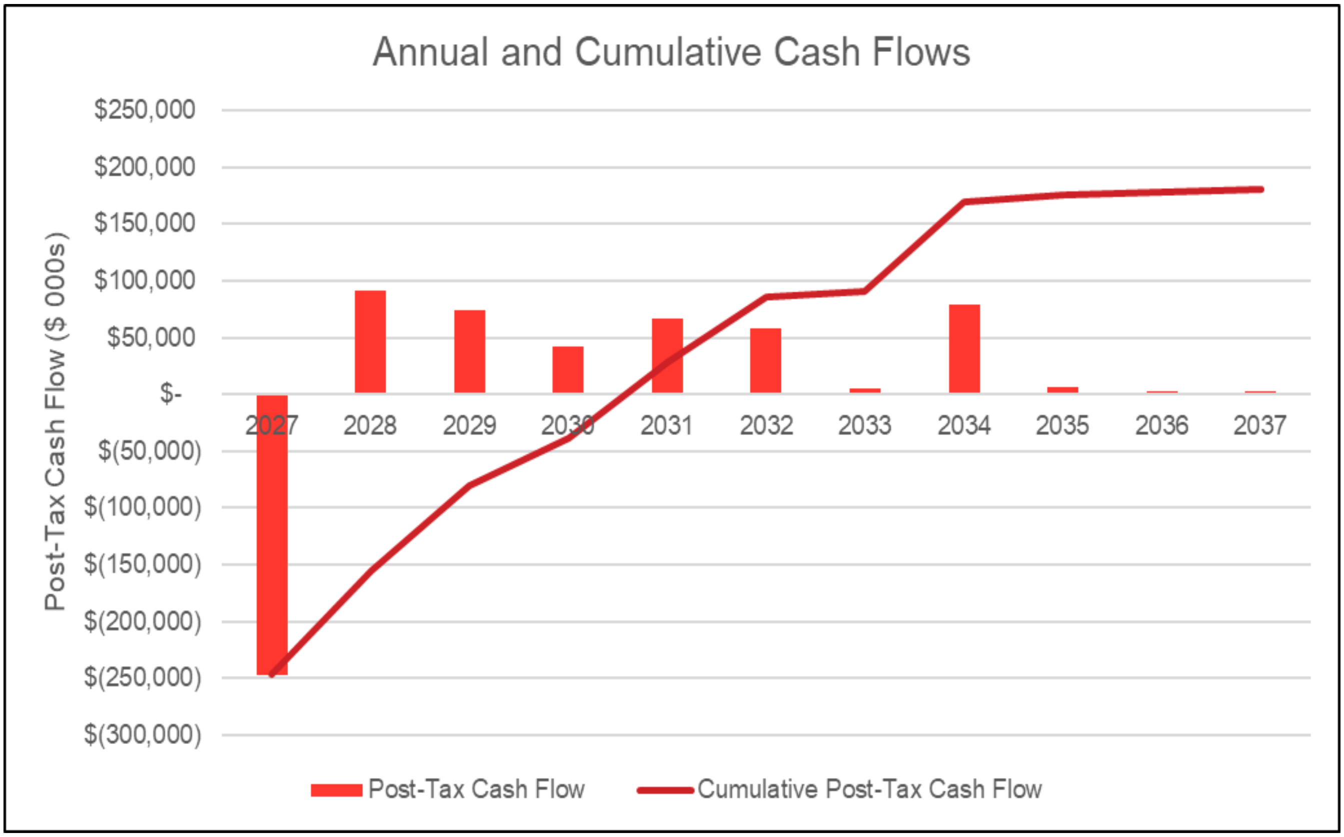


Figure 22.1: Annual and Cumulative Post-Tax Cashflows

Table 22.3: Annual Mine Plan and Cash Flows

Field	Units	LoM Total	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Mine Plan													
Waste Tonnage	Mt	91.6		18.4	18.7	12.0	12.0	11.4	13.5	5.7			
Mineral Tonnage	Mt	23.6		3.6	3.6	3.7	3.6	3.5	2.2	3.3			
Total Tonnage	Mt	115.1		22.0	22.2	15.7	15.6	14.9	15.7	9.1			
Au Grade	g/t Au	0.63		0.81	0.76	0.57	0.6	0.55	0.48	0.6			
Ag Grade	g/t Ag	2.42		3.13	2.75	2.08	2.15	2.11	1.92	2.62			
Strip Ratio	w:mineral	3.89		5.04	5.2	3.25	3.33	3.21	6.27	1.71			
Processing Feed													
Mill Feed													
Mill Tonnage	Mt	4.5		0.7	0.7	0.7	0.7	0.6	0.3	0.7			
Mill Au Grade	g/t Au	1.75		2.56	2.18	1.41	1.6	1.39	1.22	1.48			
Mill Ag Grade	g/t Ag	3.35		5.39	3.3	2.29	3.08	2.73	2.39	3.62			
Heap Leach Feed													
Heap Leach Tonnage	Mt	2.9		2.9	2.9	2.9	2.9	2.9	1.9	2.7			
Heap Leach Au Grade	g/t Au	0.37		0.37	0.4	0.36	0.35	0.38	0.36	0.37			
Heap Leach Ag Grade	g/t Ag	1.69		1.68	1.98	1.67	1.43	1.63	1.61	1.79			
Metal Production													
Mill													
Au from Mill	koz Au	234.2		55.3	47.5	30.4	34.5	25.3	11.2	30.0	0.0		
Ag from Mill	koz Ag	180.5		47.0	28.6	20.1	27.0	19.9	8.7	29.3	0.0		
Leach Pad													
Au from Heap Leach with Delay	koz Au	170.3		22.8	27.2	25.9	24.6	26.3	17.4	23.0	3.1		
Ag from Heap Leach with Delay	koz Ag	181.2		23.1	29.6	30.2	23.5	24.7	17.3	28.7	4.0		
Total													
Au Produced	koz Au	404.5		78.2	74.7	56.3	59.1	51.6	28.6	53.0	3.1		
Ag Produced	koz Ag	361.7		70.1	58.2	50.3	50.5	44.6	26.1	58.0	4.0		
Au Payable	koz Au	404.1		78.1	74.6	56.2	59.0	51.5	28.6	52.9	3.1		
Ag Payable	koz Ag	354.5		68.7	57.0	49.3	49.5	43.7	25.5	56.9	3.9		
Revenue													
Au & Ag Revenue	US$M	979.6		189.3	180.7	136.3	143.0	124.8	69.2	128.6	7.6	0.0	0.0
Refining & Selling Cost	US$M	-0.8		-0.2	-0.1	-0.1	-0.1	-0.1	-0.1	-0.1	0.0	0.0	0.0
Au & Ag Royalty	US$M	-9.7		-1.9	-1.8	-1.3	-1.4	-1.2	-0.7	-1.3	-0.1	0.0	0.0
Net Revenue	US$M	**969.1**		**187.3**	**178.8**	**134.8**	**141.5**	**123.5**	**68.5**	**127.3**	**7.5**	**0.0**	**0.0**
Operating Expenses													
Mine OPEX	US$M	-227.9		-34.7	-39.9	-31.5	-31.1	-32.0	-35.3	-23.3	0.0	0.0	0.0
Processing Mill OPEX	US$M	-74.2		-12.0	-12.0	-12.0	-12.0	-10.1	-5.0	-11.2	0.0	0.0	0.0
Processing Heap Leach OPEX	US$M	-130.3		-19.3	-18.9	-19.3	-19.3	-19.3	-12.3	-17.6	-4.3	0.0	0.0
Dewatering OPEX	US$M	-6.5		-0.7	-0.7	-0.8	-1.1	-1.0	-1.3	-1.0	0.0	0.0	0.0
G&A OPEX	US$M	-31.2		-4.3	-4.3	-4.3	-4.3	-4.3	-4.3	-4.3	-0.7	-0.1	0.0
Total OPEX	US$M	**-470.2**		**-71.0**	**-75.8**	**-68.0**	**-67.9**	**-66.8**	**-58.2**	**-57.4**	**-5.0**	**-0.1**	**0.0**
Gross Income	US$M	**499**		**116.3**	**102.9**	**66.8**	**73.6**	**56.7**	**10.3**	**69.9**	**2.6**	**-0.1**	**0.0**

Table 22.3: Annual Mine Plan and Cash Flows, continued

Field	Units	LoM Total	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Capital Expenses													
Mine CAPEX	US$M	-91.9	-36.6	-17.9	-17.7	-17.7	-2.0	0.0	0.0	0.0	0.0	0.0	0.0
Processing CAPEX	US$M	-182.0	-172.4	0.0	-9.6	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Dewatering CAPEX	US$M	-15.5	-9.9	-0.2	-0.1	-3.2	-1.9	-0.1	-0.1	-0.1	0.0	0.0	0.0
Royalty Purchase Option	US$M	-1.0	-1.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Equipment Salvage Value	US$M	16.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	5.4	5.4	5.4
Total CAPEX	US$M	**-274.2**	**-219.9**	**-18.1**	**-27.4**	**-20.9**	**-3.9**	**-0.1**	**-0.1**	**-0.1**	**5.4**	**5.4**	**5.4**
Other Expenses													
Environmental Bonding	US$M	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Environmental Bond Cash Collateral	US$M	0.0	-4.7	0.0	0.0	0.0	0.0	0.0	0.0	0.0	1.6	1.6	1.6
Environmental Bond Interest Generation	US$M	0.9	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.0	0.0
Working Capital	US$M	0.0	-22.2	0.3	4.4	-0.5	1.6	5.4	-4.9	14.8	1.1	0.0	0.0
Closure and Reclamation	US$M	-12.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	-4.0	-4.0	-4.0
Total Other Expenses	US$M	**-11.1**	**-26.8**	**0.4**	**4.5**	**-0.5**	**1.7**	**5.5**	**-4.8**	**14.9**	**-1.3**	**-2.4**	**-2.4**
Before-tax Cash Flow	US$M	**213.3**	**-247.1**	**98.6**	**80.0**	**45.5**	**71.4**	**62.1**	**5.5**	**84.7**	**6.7**	**2.9**	**3.0**
After-tax Cash Flow	US$M	**181.1**	**-247.1**	**91.7**	**74.4**	**42.3**	**66.4**	**57.8**	**5.1**	**78.8**	**6.2**	**2.7**	**2.8**
Post-Tax NPV 5%	**US$M**	**$111.60**											
Post-Tax NPV 8%	**US$M**	**$78.50**											
Post-Tax NPV 10%	**US$M**	**$59.20**											
Post-Tax IRR		**17.60%**											
Payback in Years		**3.6**											
Cash Cost	**US$/oz**	**$1,164**											
All-In-Sustaining-Cost	**US$/oz**	**$1,269**											

Notes:

1. Mt = Million tonnes
2. g/t = grams per tonne
3. oz = ounces; koz = kilotroy (1000) ounces
4. $ M = Million dollars

22.5 Sensitivity Analysis

Sensitivity analyses were conducted, using the cashflow analysis in Item 22.4 as the base case, to assess the impact of changes in metal prices, CAPEX, and OPEX on the Project's NPV (5% discount rate) and IRR. The impact of each variable is examined individually with an interval of 40% and increments of 10% applied. It is to be noted that margin of error for cost estimates at the PEA study is typically -50% +50%.

The post-tax results of the sensitivity analysis are shown in Table 22.4, Table 22.5, Table 22.6, Figure 22.2, and Figure 22.3. The NPV of the project is most sensitive to changes in the metal prices and OPEX, while the IRR is most sensitive to changes in the metal prices and CAPEX.

Table 22.4: Economic Metrics Sensitivity to Variations in Operating Cost

Operating Cost Variation	Units	-40%	-30%	-20%	-10%	0%	10%	20%	30%	40%
Post-Tax NPV (5%)	US$M	257.2	220.8	184.4	148	111.6	75.2	38.5	1.8	-34.9
Post-Tax IRR	%	32.4%	28.8%	25.2%	21.5%	17.6%	13.6%	9.3%	4.8%	0.0%

Table 22.5: Economic Metrics Sensitivity to Variations in Capital Cost

Capital Cost Variation	Units	-40%	-30%	-20%	-10%	0%	10%	20%	30%	40%
Post-Tax NPV (5%)	US$M	221.9	194.3	166.7	139.2	111.6	84.1	56.5	28.9	1.4
Post-Tax IRR	%	46.6%	36.4%	28.7%	22.5%	17.6%	13.5%	10.1%	7.2%	4.6%

Table 22.6: Economic Metrics Sensitivity to Variations in Metal Prices

Metal Price Variation	Units	-40%	-30%	-20%	-10%	0%	10%	20%	30%	40%
Post-Tax NPV (5%)	US$M	-192.3	-115.9	-40	36	111.6	187.2	262.8	338.4	414
Post-Tax IRR	%	0.0%	0.0%	0.0%	9.0%	17.6%	25.7%	33.5%	40.9%	48.2%

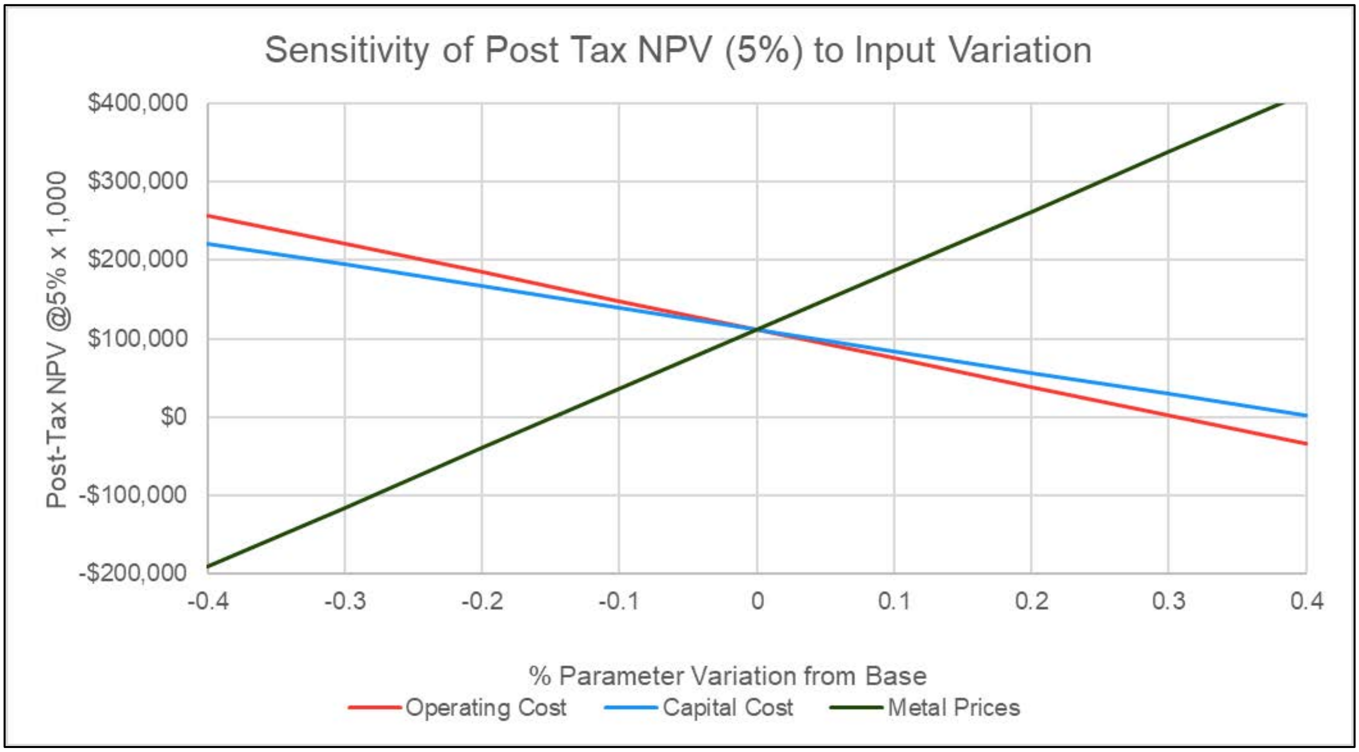


Figure 22.2: Post-Tax NPV (5%): Sensitivity to OPEX, CAPEX and Metal Prices

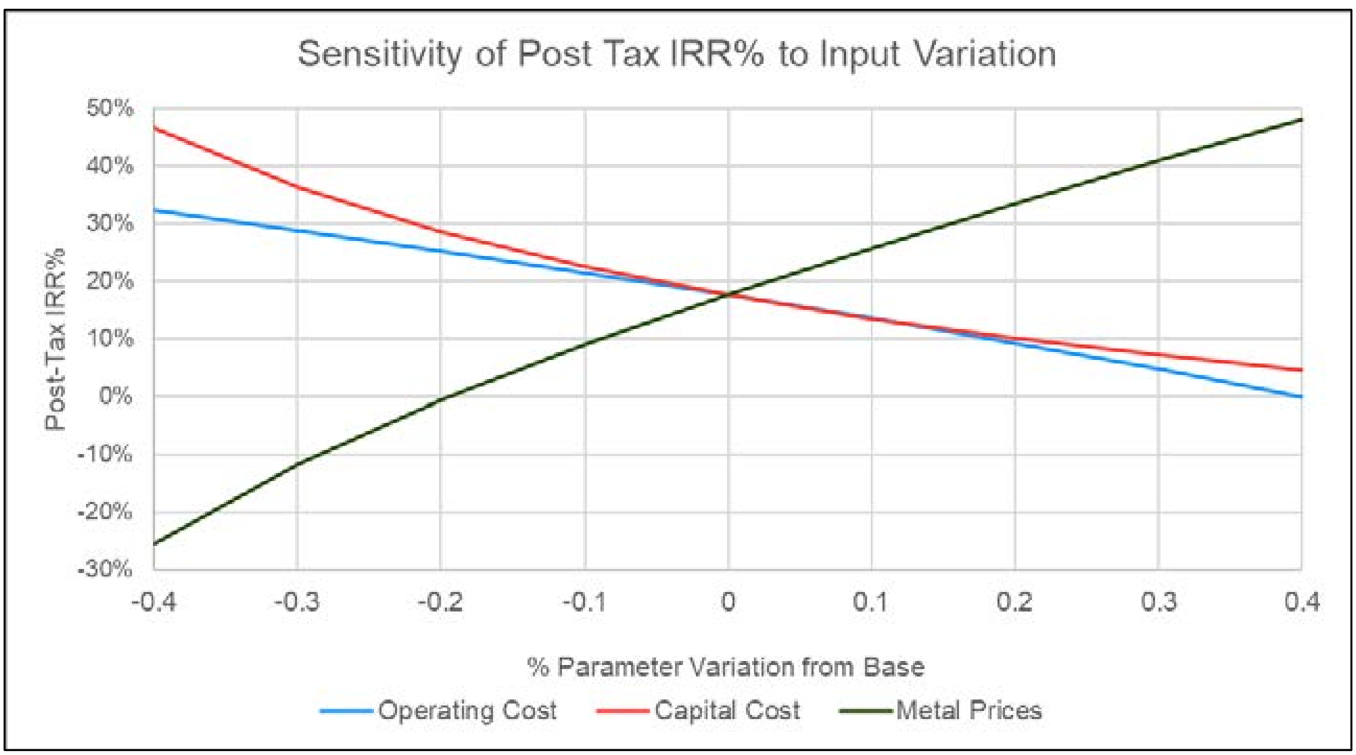


Figure 22.3: Post-Tax IRR (%): Sensitivity to OPEX, CAPEX, and Metal Prices

Table 22.7shows the effect of a higher Au price on the project's Post-Tax NPV @ 5% and IRR results.

Table 22.7: Economic Metrics Sensitivity to Variations in Au Price

Au Price (US$)	Post-Tax NPV 5% (US$M)	Post-Tax IRR (%)
1,920	(38.4)	-0.4
2,160	36.7	9.0
2,400	111.6	17.6
2,640	186.5	25.7
2,880	261.3	33.3
3,120	336.1	40.7
3,360	411.0	47.9

23.0 Adjacent Properties

There are no active deposits directly adjacent to the Tonopah Project; however, several have been identified along the Walker Lane trend.

The historic Midway Mine is approximately 2 km southwest of the Tonopah project, and was in operation in the early 1900’s. The Thunder Mountain Au-Ag prospect is located approximately 10 km to the southeast of the Tonopah Project.

The Round Mountain Mine is situated approximately 50 km north of the Tonopah Project. Since 1906, it has been in production through both historic underground operations and current open-pit methods. The Round Mountain deposit is classified as a low-sulphidation, volcanic-hosted epithermal gold deposit. Since Kinross Gold acquired the property in 2003, the Round Mountain mine has produced over 15 million oz of Au, with production planned until approximately 2027.

The historic mining district of Tonopah lies 35 km southwest of the Tonopah property. The Tonopah Mining District has produced approximately 1.8 million oz of Au and 174 million oz of Ag from several different historical mines over its history.

The Manhattan Mining District is approximately 40 km north of the Tonopah Project and has had a long history of Au production since its discovery in 1905, with continuous underground mining until 1947 at the Manhattan deposit. Exploration and drilling from 1972 to 1975 defined significant Au reserves, leading to heap-leaching and milling operations at the site. Houston Oil & Minerals Company increased reserves and resumed mining in 1984, operating open pits and a mill until 1990. Round Mountain Gold Corporation acquired the site, continuing heap leaching until 1993, followed by reclamation efforts from 1994 to 2001.

The QP has been unable to verify the information for the properties discussed, and the information is not necessarily indicative of the mineralization on the Tonopah Project that is the subject of this Technical Report. No data or other information from adjacent properties was used in the modelling and estimation procedures for the Tonopah Project.

24.0 Other Relevant Data and Information

The QPs are not aware of any other relevant data concerning the Tonopah Project.

25.0 Interpretation and Conclusions

This Item contains forward-looking information related to various elements for the Project. The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts or projections in the forward-looking information include any significant differences from one or more of the following material factors or assumptions that were applied in drawing the conclusions or making the estimates, designs, forecasts or projections set forth in this Item: geological and grade interpretations and controls and assumptions and forecasts associated with establishing the prospects for economic extraction; grade continuity analysis and assumptions; Mineral Resource model tonnes and grade and mine design parameters; mining strategy and production rates; expected mine life and mining unit dimensions; prevailing economic conditions, commodity markets and prices over the LoM period; permitting schedules; potential challenges from NGOs or other interested parties to applications; estimated capital and operating costs; and project schedule and approvals timing with availability of funding.

25.1 Geology and Mineral Resource Estimate

The Tonopah Gold Project exhibits the defining features of a low-sulphidation epithermal system, further complicated by numerous interacting faults within an oblique-slip fault framework. Au and Ag mineralization is governed by both lithological and structural controls, revealing two overlapping orientations. High-grade Au is predominantly concentrated in veins, breccia-veins, and mineralized structures, while broader disseminated zones host lower-grade material. Visible Au is frequently present along vein margins, commonly associated with hematite, and occasionally appears in coarse form. The highest grades are contained within the Discovery Zone, an area extensively drilled and sampled throughout the Project's history. This zone is bounded by multiple faults, with high-grade mineralization encountered just 30 m below surface in both the TVL and Op units. The Rye Patch zone is also significantly mineralized, characterized primarily by mineralization within the TVL unit, which occurs at both shallow depths and beyond 50 m depth.

The updated geological model and MRE for the Project now incorporates data from 59 new drill holes since 2022, improving drill spacing and identifying new high-grade and non-mineralized zones. Previous iterations of the lithology model were not utilizing data from all available drill holes; the incorporation of this historic data has resulted in an improved interpretation and geologic understanding of the deposit. The creation of a 3D CSAMT model has contributed to the development of a robust structural model, helping define mineralization boundaries and geologically defined estimation domains. These results indicate potential for discovering additional mineralized zones.

Data verification included a full independent rebuild of the Tonopah assay database using original assay lab certificates where available, including the addition of missing below detection assay values. This has resulted in higher confidence in the underlying data used for the MRE, and better control on block estimation, reducing overestimation of higher grades in areas of lower sample density. The analysis and inclusion of historic bulk density data has resulted in a more realistic tonnage estimate for the resource. On completion of the data verification process for the Tonopah Gold Project, it is the QP’s opinion that the geologic data collection, analytical methods, and QA/QC procedures used by Viva are generally consistent with industry standards, WSP has recommendations to improve the data collection procedures, as well as additional recommendations listed in Item 26.1.

It is the QP's opinion that the geological database and assay data is of suitable quality to support the 2025 MRE, as reported in Item 14.0. The updated MRE has converted Inferred mineral resources to Indicated, reflecting increased confidence in the data and geologic interpretation.

The Mineral Resource estimate for the Tonopah Gold Project has been prepared in accordance with NI 43-101 following the requirements of Form 43-101F. The Mineral Resource estimate follows the CIM MRMR Best Practice Guidelines, issued November 29, 2019, and was classified following CIMDS for Mineral Resources and Mineral Reserves adopted May 10, 2014. It is the QP's opinion that the information presented in this Technical Report is representative of the Project, and based on the data verification completed, concludes that the sample database is of a suitable quality to provide the basis for the conclusions and recommendations reached in this Technical Report.

The QP has taken reasonable steps to ensure the block model and MRE are representative of the Tonopah data, but notes that there are risks related to the accuracy of the estimate related to the following:

- The accuracy and quality of historical data.
- The assumptions used by the QP to prepare the data for resource estimation.
- The accuracy of the geological interpretation, including the structural interpretation and estimation domains. Any revisions to domain boundaries may result in significant changes to the overall MRE.
- The variable and structurally complex nature of the deposit geology.
- The impact of outlier grade data, particularly within the Inferred category where drill density is limited.
- Estimation parameters used by the QP.
- Parameters used to support RPEEE.

For these and other reasons, actual results may differ materially from the estimate.

25.2 Mining

Based on the available information, brief analysis, and technical assessments, the Tonopah Project demonstrates the potential for an open pit mine. Key data supporting this conclusion include block modelling, geotechnical domain refinement based on drilling data, and systematic application of slope and ramp design parameters. Mine design, optimization, and LoM planning have all been carefully considered.

The use of Whittle software to generate nested pit shells and three mining phases, as well as to optimize economic outcomes, ensures that mineral resources are maximized under reasonable, varied scenarios. Economic input parameters, such as cost estimates and Au price assumptions, are carefully integrated. No incremental mining cost due to depth was included because of the relatively shallow pit geometry. The selected final pit shell corresponds to RF 0.98 and is used for COG calculations and production scheduling for both heap leach and mill options.

Waste rock and material production rates, mining fleet and equipment choices, and operational sequencing are all based on industry benchmarks and site-specific conditions. Owner-operation is preferred, as it allows for effective mineralized material control and cost efficiency. The ability to access mineralized material from the surface partly eliminates the need for pre-production stripping. About 16% of waste from the northwest portion of the pit is planned to be in-pit dumped into the southeast pit toward the end of the mine life, which will reduce haulage distances and operating costs in the final years of production. Material was classified as low-, mid-, or high-grade and directed accordingly to heap leach or mill, with high-grade material always prioritized for milling.

Geotechnical and hydrological considerations were limited, however were substantiated by recommendations and compliance with industry standards and regulatory guidelines. Waste storage capacity has been designed with future expansion in mind, and operational safety measures meet industrial standards. For mine design and scheduling, factors such as mining recovery and dilution are incorporated to maintain moderate yet reasonable projections for a PEA-level study.

The Tonopah Project’s open pit mine plan is technically viable and economically reasonable under current market conditions. It is designed with flexibility, operational safety, and resource optimization at its core. The methodology and data inputs meet industry practices and regulatory requirements, positioning the project for successful progression to advanced study phases and potential development.

25.3 Metallurgy and Process

The Tonopah Gold Project considers recovering Au and Ag values by a combination of heap leaching and milling with a CIL plant for recovery at an average combined rate of 10,000 tpd (8,000 tpd heap and 2,000 tpd mill). Material will be crushed in a shared crushing plant where low-grade and high-grade material will be campaigned and stockpiled separately using a radial stacker conveyor. Low-grade heap leach material will be reclaimed; drum agglomerated with cement and conveyor stacked onto a permanent heap leach pad where it will be leached using a low concentration NaCN solution. Pregnant solution from the heap will be pump to a carbon adsorption circuit where Au is adsorbed onto carbon and the resulting barren solution will be pumped back to the heap as barren leach solution.

High-grade material will be reclaimed and milled in a closed, single stage ball mill circuit to a target grind size of 80% passing 106 µm. A gravity concentration circuit is also considered to recover coarse Au from the circuit. After grinding, the mill product will be thickened and leached in a CIL circuit with the mill tailings being filtered and struck stacked onto a dedicated section of the leach pad. Loaded carbon from the heap leach and mill circuits will be transported off site for toll processing where Au and Ag values will be recovered and the carbon regenerated and returned to be added back to the circuits.

Test work developed for the project indicates that the material is amenable to cyanide leaching and gravity concentration for recovery of Au and Ag values. The test work completed to date shows a strong dependence on head grade for heap leach Au recoveries with high-grade material having lower overall recoveries which is assumed to be due to coarse Au and reduced leach kinetics. Head grade does not appear to be a significant factor for Au recoveries in the milling circuit.

Overall heap recoveries are estimated at 75% for Au and 24% for Ag based on a recovery by grade correlation applied to the block model. Overall mill recoveries are estimated at 92% for Au and 37% for Ag which are based on volcanic material recoveries of 90% for Au and 38% for Ag and argillic material recoveries of 95% for Au and 36% for Ag.

25.4 Environmental

Environmental and permitting matters related to the Project are very similar to those effectively managed by other operators at multiple surface mining projects on public and private lands in northern Nevada and authorized by Federal, State and local regulatory authorities.

Reasonably foreseeable risks may include, but not be limited to:

- Lengthy permitting schedules resulting from lessened availability of consulting services and regulatory agency staffing and technical expertise.
- Challenges by NGOs and interested parties to technical data and agency decisions regarding management of hydrologic, biologic and cultural resources at the Project.

No operational or post-closure chemical treatment of dewatering water or post-mining pit lake water quality is currently anticipated; however, this is based on a limited data set and must be confirmed in later studies. Water management strategies will be designed to comply with applicable water quality standards and to protect groundwater resources. A waste rock management plan approved by the agencies will present methods to isolate and protect groundwaters of the State when PAG material is encountered.

25.5 Costs and Financials

The capital and operating costs were estimated using methods described in this Report, which are typical of a PEA-level cost estimation exercise.

Based on the available information, the project has an after-tax NPV of US$111.6M at a discount rate of 5%, an IRR of 17.6%, and a payback of 3.6 years. The sensitivity analysis indicates that the Project economics are most sensitive to the Au price with the breakeven Au price 20% below the base case of US$2,400/oz. Table 25.1 is a reproduction of Table 22.7 and shows the effect of a higher Au price on the project's Post-Tax NPV @ 5% and IRR results.

Table 25.1: Economic Metrics Sensitivity to Variations in Au Price

Au Price (US$)	Post-Tax NPV 5% (US$M)	Post-Tax IRR (%)
1,920	(38.4)	-0.4
2,160	36.7	9.0
2,400	111.6	17.6
2,640	186.5	25.7
2,880	261.3	33.3
3,120	336.1	40.7
3,360	411.0	47.9

26.0 Recommendations

26.1 Geology and Mineral Resource Estimate

The QP has the following recommendations for Viva to consider regarding data collection:

- Drill hole collars should be surveyed with a differential GPS to improve accuracy and marked with a permanent ground marker that will survive environmental conditions. An example would be a concrete cap or monument to mark the collar, with the drill hole ID written either in the concrete or written on a metal tag attached to the monument. The use of a handheld GPS is acceptable for exploration stage projects but requires increased level of accuracy for PEA and above level studies.
- An increase in the number of DD holes vs. RC holes is recommended for improved data quality and understanding of the deposit. DD holes allow the collection of bulk density measurements, rock quality information, fault/geotechnical parameters locations and angles, and alteration zones, that are not easily (if at all) available from the collection of RC drilling samples.
- Additional bulk density measurements should be taken on future DD drilling, including intervals of known waste rock. Suitable QA/QC procedures should be included as part of the bulk density measurements, and the use of a waterproof coating should be used, especially in fractured rock. If possible, bulk density measurements should be taken on historical DD core, spaced evenly throughout the main zone of the current resource pit shell extents.
- Field duplicate samples should be collected for all future DD drilling to evaluate the accuracy of the sampling at core splitting stage. Viva should also collect several duplicate RC samples to test the variability between samples of the same interval using this method of drilling.
- Down-hole deviation measurements for all drill holes, as well as core recovery, and RQD for DD holes should continue to be collected on all future drilling programs.
- Regular check sampling by a third-party (umpire) analytical laboratory should be conducted.
- As recommended by the CIM MRMR Best Practice Guidelines, drill logs and sample results should be stored in a relational database that provides proper control and security. The database should contain all relevant data for each drill hole, including, but not limited tom drill hole ID, collar location and orientation, total depth, down-hole deviation measurements, hole diameter, geological data (lithology, alteration, core recovery, RQD), and analytical data (unique sample ID, analytical laboratory name, analytical certificate number, assay results, including any trace or deleterious elements, geometallurgical results, bulk density, QA/QC data). An MS Access database would be suitable for this purpose.

For all future DD core programs, WSP recommends obtaining additional data such as bulk density measurements, core recovery, RQD, and the location and angles of major faults. Additional bulk density measurements, including those from intervals of known waste rock, will improve the current SG and tonnage estimate for the deposit. Further geotechnical data will refine the existing structural interpretation and its effect on Au mineralization. Developing an alteration model could improve understanding of its impact on Au mineralization and potentially identify new drill targets.

All these recommendations would fall under the next phase of the Project as part of a PFS. The estimated costs are approximately US$300,000.

26.2 Mining

A detailed trade-off study comparing the leasing versus purchasing of production equipment, including the potential for a hybrid approach, should be undertaken. This study would highlight opportunities to reduce initial capital requirements and assess the impact on overall operating costs.

A more detailed mining phase plan for open pit mining at the PFS level should be developed. Given the nature of spatial grade distribution and strip ratio of the deposit, detailed phasing could allow for bringing more high-grade material earlier in the mine life, allowing for enhanced capital cost recovery.

The development of geotechnical slope design domains should be advanced through continued DD drilling and geological modelling. This approach would help maintain consistent slope stability throughout the mine's operational life and minimize the risk of deviations.

A comprehensive hydrological assessment should be undertaken to identify potential water inflows and drainage concerns. The study is essential to mitigate operational disruptions and ensure long-term pit stability.

The expected cost for next stage of trade-off studies and PFS is approximately US$3.0M, which is exclusive of the expenses associated with any field programs described in other sections. By undertaking detailed and multidisciplinary assessment, stakeholders will be able to identify risks, optimize design parameters, and establish a solid foundation for the next stages of project development.

26.3 Metallurgy and Process

Results from metallurgical test work suggests that there is a possibility of improved recoveries with longer leaching due to potential coarse Au. The results also suggest that similar heap leach recoveries may be achievable at coarser crush sizes. As part of future test programs, variability testing at coarser product sizes and longer leach cycles should be evaluated to determine whether further process optimization is possible to improve project economics.

Toll processing of carbon is considered for the project without any formal contract terms in place. Due diligence efforts were made to ensure reasonable numbers; however, should there be any significant shifts in the market this could result in increased prices or delays in Au and Ag production. As part of future work, Viva should engage with multiple toll stripping groups to ensure production capacity and costs.

KCA recommends additional metallurgical studies including:

- High-grade mill and gravity variability testing
- Variability column testing at various crush sizes (9.5 mm, 12.5 mm, 25 mm and 38 mm) for a 120 to 180-day period.
- Additional characterization work.

Samples for KCA's metallurgical program may be captured in a future DD core program. The cost of a 1,000 m PQ drill program including assay, televiewer/oriented core study is approximately US$500,000 not including additional cost for SG testing. The estimated cost of the metallurgical test work is US$475,000. Quotations are pending for an updated geotechnical study.

26.4 Environmental

26.4.1 Specific Work Plan

As part of the specific work plan, long-lead baseline studies will be initiated to support environmental permitting, mine development, and eventual closure activities. These studies will include:

- Environmental monitoring, including updated hydrogeologic investigations;
- Cultural resources surveys, with a focus on areas not surveyed or resurveyed within the past ten years;[2]
- Biological studies, including updated habitat and species evaluations;
- Hydrogeologic studies, to inform the development of a site-specific numerical groundwater model.

The work plan will also include the installation of three groundwater monitoring wells: one upgradient replacement well and two downgradient wells. In support of the groundwater model and permitting requirements, two 30-day aquifer tests will be conducted – one from the existing bedrock production/monitoring well and one from the existing alluvial production/monitoring well.

A soil infiltration testing program should be implemented to evaluate the capacity of alluvial soils to accept excess mine dewatering water without adversely impacting groundwater quality, in accordance with state water protection standards.

As the proposed pit design extends below the regional water table, it is recommended that the PFS and FS phases include further hydrogeologic investigations. These studies should evaluate potential impacts to groundwater quantity and quality, assess the suitability of pit water for wildlife, and determine whether pit lake formation could affect other ecological receptors. Additionally, the potential for hydraulic connectivity to nearby water users, streams, or springs should be assessed to ensure that mining activities do not adversely impact existing water rights or natural hydrologic features.

To support accurate design and permitting of the dewatering system, it is recommended that a longer-duration and more robust pumping test be conducted in the vicinity of the proposed dewatering zone. This test should be designed to improve predictions of groundwater inflow rates and to characterize the quality of the dewatering water. If water quality results indicate that discharge would not meet applicable groundwater discharge standards, treatment will be required prior to discharge into RIBs.

In parallel, a pilot-scale infiltration basin test is recommended, coupled with a calibrated groundwater flow model, to evaluate the infiltration capacity and long-term sustainability of the proposed discharge approach. While pumping and discharge to RIBs is generally considered a sustainable and effective water management strategy, its success depends on accurate site-specific hydrogeologic data, proper operation of infiltration basins, and discharge water quality. These investigations are typically conducted during the PFS or FS stages to inform engineering design, permitting, and environmental impact assessments.

Additional predictive modelling may be required by regulatory agencies to demonstrate that discharge of dewatering water to the RIBs will not adversely affect groundwater resources. As outlined in Item 20.0, a regulatory precedent exists for similar operations; however, for water quality that does not meet standards or if

[2] Assumes that survey standards at the time of the original work are still current and no significant changes (e.g., new discoveries, land use changes, or updated regulations) have occurred in the area since the survey was completed.

agencies determine that site-specific factors such as local hydrogeology, background water quality, and surrounding land use warrant further analysis, agencies may require additional analysis to confirm compliance and protect water quality.

For a PEA many studies utilize existing data and information. Updated studies, such as a PFS or FS, will need to meet current expectations for data quality, modelling transparency, or regulatory rigor. New guidance (e.g., NDEP's 2021 geochemical modelling guidance) emphasizes integration with groundwater flow models, which may not have been standard practice in earlier studies. The Nevada Modified Sobek Procedure has also evolved since 2012. Studies should include comprehensive baseline water quality or hydrologic data, which are now required to support predictive modelling of pit lakes and waste rock impacts. Future detailed investigations should include geochemical models that rely on site-specific mineralogy, hydrology, or climate conditions.

A Class III cultural resources survey will be completed for all unsurveyed or outdated areas within the projected Project boundary, consistent with federal and state requirements.

To ensure compliance with the Bald and Golden Eagle Protection Act, a two-year aerial survey program targeting Golden Eagles and other raptors will be conducted. The results will inform mitigation planning and permitting, if necessary.

The Nevada-approved SRCE Version 1.4.1 Build 017b (Revised May 16, 2019), along with the SRCE_Cost_Data_File_1_12_Std_2024.xlsm, was used to estimate the preliminary reclamation bond for the Project based on operational and closure assumptions outlined in this PEA. For future studies and more refined planning, it is recommended that the most current version of the SRCE model and associated cost data files be utilized. This will allow alignment with evolving regulatory expectations, cost structures, and best practices in reclamation planning.

Table 26.1 presents the estimated environmental and permitting costs to move the project into the next phases of a PFS. These estimates costs are provided at an order-of-magnitude level of accuracy and are intended for preliminary planning purposes.

Table 26.1: Estimated PFS Environmental and Permitting Costs

Category	Estimated Cost (US$)	Notes
Hydrogeology Studies	$500,000	Excludes capital cost of wells construction
Cultural Resources Survey	$200,000	Areas not surveyed, or surveyed >10 years ago
Raptor Surveys	$100,000	2 years Golden Eagle & Raptor surveys
General Consulting	$100,000	Environmental permitting support
Heap Leach Pad	$250,000	Studies to support the material characterization and stability analysis for the heap leach pad
Environmental Total	$1,150,000	

26.5 Costs and Financials

For the advancement of the Project, specific cost workups based on multi sourced quotations for equipment, construction materials, consumables, pre-stripping contracts, are a prerequisite. With fast-track project execution aspirations to advantage the metal markets then the multi sourced quotes would benefit from FEED leading to firm tender bids for these recommended quotes thereby allowing equipment procurement immediately upon approval of funding.

Contracts for market level agreements, power supply, site preparation, fuel and lubricants, maintenance agreements need consideration of advancement.

26.6 Budget for Recommended Work

Table 26.2 is a reproduction of Table 1.8 and provides and estimated budget for recommended work. These estimated costs would be finalized once work packages are tendered.

Table 26.2: Budget for Recommended Work

Item	Category	Estimated Cost (US$)
1.0	Geology and Mineral Resources	$300,000
2.0	Metallurgy and Processing	$1,000,000
3.0	Environmental Studies, Permitting, Social or Community Impact and Government Relations	$1,150,000
4.0	Engineering and Field Work to Complete PFS and Reporting	$3,000,000
	Total	**$5,450,000**

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