ARCHITECTURE

ENGINEERING

CONSTRUCTION MANAGEMENT

Mexican Hat Project

NI 43-101 Technical Report

Preliminary Economic Assessment

Cochise County, Arizona

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Prepared For:

GMV MINERALS INC.

DATE AND SIGNATURES PAGE

The effective date of this Technical Report is December 6, 2018. The issue date of this report is December 18, 2018. See Appendix A, Feasibility Study Contributors and Professional Qualifications, for certificates of qualified persons. These certificates are considered the date and signature of this report and the effective date of this report in accordance with Form 43-101F1.

MEXICAN HAT PROJECT
FORM 43-101F1 TECHNICAL REPORT

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- Certificate of Qualified Person ("QP")

1 EXECUTIVE SUMMARY

This preliminary economic assessment (PEA) has been prepared for GMV Minerals Inc. (the "Company" or "GMV") for the Mexican Hat Project gold deposit located in Cochise County, Arizona ("Mexican Hat" or the "project") in accordance with the National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101). GMV Minerals Inc. is a publicly traded exploration company solely focused on developing precious metal assets in Arizona.

1.1 PROPERTY DESCRIPTION AND LOCATION

The property is located in Cochise County, immediately north of the Gleeson Courtland district, 10 km (6 miles) south of Pearce, Arizona, and approximately 140 km (90 miles) by road from Tucson, Arizona, and is centered on approximately N 31° 48' 9.23" / W 109° 48' 26.17" (612,875 mE, 3,519,245 mN NAD83 Zone 12).

1.2 GEOLOGY AND MINERALIZATION

The Mexican Hat Property is considered to host a low sulphidation alkaline epithermal gold deposit in Tertiary volcanic rocks. Similar deposits of similar age within the same basin and range province in Nevada, such as Round Mountain and the Midas Deposit host many millions of ounces of gold.

Initial work completed by GMV in 2014 sought to confirm previous work. The previous work was confirmed, and data verification allows for integration of all datasets except that of Kalahari into a comprehensive model for assay data.

Work completed by GMV since 2014 includes additional surface mapping, collection of 1,123 surface soil geochemistry samples, completion of an aerial DEM and photogrammetric survey, completion of 85.6 line kilometers of ground magnetics, completion of three lines of audiomagnetotellurics geophysics, three gravity profiles, and drilling including 15 reverse circulation (RC) holes totalling 4,776.5 m in 2016, and in 2017 completed eight HQ core holes totalling 1,979.3 m and 15 RC holes totalling 4,032.9 m.

A new structural interpretation was synthesized from existing and recently acquired data. Three prominent faults are mapped on the project which both host and may offset mineralization. Two subparallel north-south trending faults, the West Fault occurring to the west of Mexican Hat and the Victoria Fault occurring to the east of Mexican Hat, are connected by a left lateral north dipping jog referred to as the 120 fault or the SMAG fault.

Gold and silver mineralization is associated with moderate to strong oxidized zones of hematite and limonite, directly related to and fill, in part, dominant NE/SW related fractured zones including secondary NW/SE fault and fractured zones. Mineralization remains open at depth along these faults, and to the south along the Victoria Fault. Mapped gold mineralization is hosted in structures within all observed major rock units.

1.3 METALLURGY

Two column leach test programmes were carried out in addition to bottle roll tests. The initial programme had two columns running on minus 38 mm material and yielded 95 and 77% recovery. The second programme had a single column running 150 mm material and yielded 57% recovery. However, analysis of the test results revealed that the finer fractions (minus 25 mm) gave recoveries of over 90% and indicated a residue grade of approximately 0.05 g/t could be achieved. In this PEA, it is designed for a crush size of 80% passing 21.4 mm (P80) and an assumed residue grade of 0.055 g/t.

The material tested was a hard, competent rock and would present no problems in constructing a 21.4 mm (P80) crushed rock heap leach without the need to agglomerate. Leaching and gold recovery will be by conventional well proven heap leach and carbon absorption technology.

1.4 MINERAL RESOURCE ESTIMATE

Resource modeling has been completed using available assay results from 45 core holes (totalling 2,650 assays), 27 reverse circulation (RC) holes (totalling 2,308 assays), 120 rotary holes (totalling 5,536 assays) and 149 channel samples. The geological database has been reviewed and verified for use in mineral resource estimation.

A grade model was developed based on an approximate 0.2 g/t Au lower cut-off grade resulting in nine (9) mineralized domains to represent the West Fault and 120 Fault, the NE-SW trending fractured zones, and a higher-grade core (>1.0 g.t) which appears to be present which is surrounded by a lower grade halo.

A block model was developed for the Mexican Hat Property which has been constrained to a conceptual open pit model for statement of Mineral Resources. The Mexican Hat Deposit hosts an Inferred Mineral Resource of 32,876,000 tonnes grading 0.62 g/t Au, equivalent to 651,000 troy ounces of gold using a 32 g/t capping grade and a 0.20 g/t cut-off grade.

Table 1-1: Mineral Resource Statement, Mexican Hat Project, Arizona, USA, Tetra Tech, Effective Date June 22, 2018

Category	Cut-off (g/t Au)	Grade (Au, g/t)	Tonnes	Gold Oz	Strip Ratio
Inferred	0.20	0.616	32,876,000	651,000	2.56

- The Mineral Resource Estimate has been constrained to a preliminary optimized pit shell, using the following parameters: SG = 2.57 t/m^3 based on testwork, mining costs = $1.50/tonne, mining recovery =98%, mining dilution = 2%, process cost = $3.25 per tonne, G&A = $0.55 per tonne, gold price = $1,300 per troy ounce Au, throughput at 15,000 tpd, discount rate = 7%. Top cut at 32 g/t Au.
- Mineral Resources constrained to optimized pit shells are not mineral reserves and do not have demonstrated economic viability.
- Conforms to NI 43-101, Companion Policy 43-101CP, and the CIM Definition Standards for Mineral Resources and Mineral Reserves. Inferred Resources have been estimated from geological evidence and limited sampling and must be treated with a lower level of confidence than Measured and Indicated Resources.
- All numbers are rounded. Overall numbers may not be exact due to rounding.
- There are no known legal, political, environmental, or other risks that could materially affect the potential development of the mineral resources.

1.5 MINING

The Mexican Hat Project is planned as a conventional hard rock surface truck-and-loader mining operation. There are two independent pits which are developed with 5 phase or pushback designs.

Mexican Hat is planned to produce 15,000 tpd (5,475 kt/a) of mineralized feed to the heap leach plant with a total mining rate of 58,000 tpd (20,880 kt/a) of material. The mine life is approximately 5-6 years where there will be a year of preproduction and a partial year in year 5. Over the course of the 5 year mine life, 25.1 Mtonnes of mineralization are planned for processing out of a total material movement of 96.4 Mtonnes.

There is no mineral reserve at Mexican Hat at this time. A component of the mineral resource, inclusive of inferred mineralization has been designated as potentially minable material for this PEA.

The total resource that is planned for processing on Table 1-2 is inferred mineralization. The reader is cautioned that this mine plan is based on a mineral resource which is entirely an inferred category mineralization. The reader is cautioned that mineral resources are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the mineral resources will be realized or that they will convert to mineral reserves.

The final pit and dump layout can be seen in Figure 1-1.

Table 1-2: Mexican Hat Mine Production Schedule

	Direct Crusher Feed				Low-grade >=0.25 Au-gpt				
Year	Cut-off Grade au-gpt	Feed Ktonnes	Gold au-gpt	Rec Gold au-gpt	LG Ktonnes	Gold au-gpt	Rec Gold au-gpt	Waste Ktonnes	Total Ktonnes
0	0.30	1,000	0.702	0.623	199	0.273	0.218	3,971	5,170
1	0.30	4,475	0.769	0.683	966	0.273	0.218	15,439	20,880
2	0.22	5,475	0.634	0.559				15,405	20,880
3	0.22	5,475	0.542	0.474				15,405	20,880
4	0.22	5,475	0.671	0.592				15,413	20,888
5	0.22	2,045	0.991	0.882				5,643	7,688
Total		23,945	0.680	0.601	1,165	0.273	0.218	71,276	96,386

	Process Production Schedule			
Year	Cut-off Grade au-gpt	Feed Ktonnes	Gold au-gpt	Rec Gold au-gpt
0		0		
1	0.30	5,475	0.757	0.672
2	0.22	5,475	0.634	0.559
3	0.22	5,475	0.542	0.474
4	0.22	5,475	0.671	0.592
5	0.22-0.25	3,210	0.730	0.641
Total		25,110	0.661	0.583

Figure 1-1: Final Pit and Dump Layout (End of Mine Life, End of Year 05)

1.6 RECOVERY METHODS

The Mexican Hat gold deposit will be mined as an open pit. Mineralization will be crushed and stacked on a heap for cyanidation in order to produce a 'pregnant' solution. The crushing plant is slated to run at a nominal capacity of 833 mtph utilizing an open-circuit design with one primary jaw crusher and one secondary cone crusher.

Pregnant solution from the heap leach will be processed by a 3-tonne carbon ADR plant. In the carbon plant, gold will be recovered from the pregnant solution by adsorbing the dissolved gold onto activated carbon followed by desorption, electrowinning, retorting and smelting to recover the gold as a final doré product.

The recovery of the cyanide heap leach process for the Mexican Hat mine is projected in Table 1-3.

Table 1-3: Projected Metal Recovery

Metal	Head Grade	Heap Leach Recovery
Au	0.661 g/t	88.2%

The latest mine plan was based on the recovery models and operating cost models that were used in the financial analysis in this study.

1.7 PROJECT INFRASTRUCTURE

The Mexican Hat Project is located in the southeastern part of the State of Arizona, approximately 115 km east-southeast of Tucson, and can be accessed from the Old Ghost Town Rd., a gravel road extending south of the Town of Pearce.

Groundwater will be used as the source of water for mining operations. No permitting restrictions or quantity issues are anticipated.

A 69 kV powerline to site will be supplied by Sulphur Springs Valley Electric Cooperative. The power plant is located 30 km north of the site.

1.8 ENVIRONMENTAL AND SOCIO-ECONOMICS

The Mexican Hat Project will require various state and federal authorizations, licenses and permits for project construction, operation, closure and post-closure. Comprehensive environmental and socio-economic baseline studies will be required. No environmental baseline programs have been conducted.

No known factors exist to preclude a successful permitting effort; however, the length and the effort of the permitting process can be difficult to predict due to the multiple agencies that will be involved, including the possibility of the project being subject to the National Environmental Policy Act.

1.9 CAPITAL AND OPERATING COSTS

1.9.1 Capital Costs

The capital cost for initial developed for the Mexican Hat Project totals $130.1 million as summarized in Table 1-4.

Table 1-4: Mexican Hat Project Initial Capital Summary

Cost Item	Total ($M)
Site, Process and Infrastructure	
Project Directs	$63.3
Project Indirects	$12.7
Contingency (25%)	$19.0
Subtotal	**$95.0**
Initial Fleet Lease	$14.9
Pre-Production Waste Removal	$15.2
Owner's Costs	$5.0
Total	$130.1

1.9.2 Operating Costs

The mine operating costs were calculated to average $1.84 per tonne moved.

Table 1-5: Mine Operating Costs

Area	Unit Cost ($/t moved)
Drilling	0.26
Blasting	0.30
Loading	0.17
Hauling	0.43
Auxiliary Equipment	0.25
Mine and Maintenance General	0.18
Mine General and Administration	0.25
Total Cost	1.84

The life of mine operating costs were calculated to average $12.30 per tonne feed.

Table 1-6: LOM Mine Operating Costs

	Cost per Tonne of Mineralized Material Processed
Mining	$6.62
Process	$4.18
G&A	$1.50
Total Site Operating Cost	$12.30

1.10 ECONOMIC ANALYSIS

At a $1,300 per ounce gold price, the base case economic analysis of the Mexican Hat Project indicates an NPV at 5% discount rate (NPV$_5$) of $77.8 million and an after-tax IRR of 29.1%.

Table 1-7: Sensitivity Analysis After-Tax ($M)

	NPV @ 5%	IRR%	Payback (yrs)
Gold Price			
+20%	$165.3	53.8%	1.3
+10%	$122.1	41.7%	1.7
Base Case	$77.8	29.1%	2.3
-10%	$32.9	15.5%	3.4
-20%	$-0.4	-0.1%	7.0
Initial Capital			
+20%	$55.5	19.4%	3.1
+10%	$66.7	23.9%	2.8
Base Case	$77.8	29.1%	2.3
-10%	$88.8	35.4%	1.9
-20%	$99.8	43.3%	1.6
Operating Cost			
+20%	$32.1	15.4%	3.4
+10%	$55.1	22.4%	2.9
Base Case	$77.8	29.1%	2.3
-10%	$100.3	35.4%	1.9
-20%	$122.3	41.4%	1.7

1.11   RECOMMENDATIONS

To move the project forward, a geotechnical slope stability should be conducted to refine the pit slopes assumed for the pit designs. Additional drilling will be required to transfer inferred category material to indicated category material.

Environmental baseline studies should be started to support the permitting process. Consultation with the community and the regulatory agencies should be initiated.

2 INTRODUCTION

The Mexican Hat Property (the "Property" or "Project") is located in Cochise County, immediately north of the Gleeson Courtland district, south of Pearce, Arizona. The property covers 42 mineral claims over 316 hectares, along with an additional 1,831.11 surrounding hectares of exploration permits for a combined total of 2,152.11 secured hectares of interest related to the Mexican Hat Project.

GMV Minerals Inc. (GMV) is a publicly listed and Canadian registered gold exploration company that is actively exploring and developing their Mexican Hat Project in Arizona. GMV's stock is listed on the Toronto Venture Stock Exchange (TSXV) and trades under the symbol GMV. GMV's head office is located at 595 Howe Street, Vancouver BC V6C 2T5.

2.1 SOURCES OF INFORMATION

There have been two Technical Reports previously filed on the Mexican Hat Project:

- Technical Report on a Geological Assessment of the Mexican Hat Property, Cochise County, Arizona, USA, dated February 26, 2016.
- 2018 Technical Report and Mineral Resource Estimate on the Mexican Hat Project, Cochise County, Arizona, USA dated effective June 22, 2018.

Additional information was obtained by M3 or provided by GMV and is contained and referenced herein.

2.2 QUALIFIED PERSONS AND SITE VISITS

The Qualified Persons for this Technical Report are as follows:

- **Daniel Roth**, of M3 Engineering & Technology Corp. –Project Infrastructure, Process Plant and Infrastructure Capital Costs, and Economic Analysis
- **Dave R Webb**, of DRW Geological Consultants Ltd. – Property Description and Location, Accessibility, Climate, Local Resources, Infrastructure, and Physiography, History, Geological Setting and Mineralization, Deposit Types, Exploration, Drilling, Sample Preparation, Analysis, and Security, Market Studies and Contracts, Adjacent Properties
- **Justin Black**, of M3 Engineering & Technology Corp. – Recovery Methods and Process Operating Costs
- **John R.W. Fox**, of Laurion Consulting Inc. – Mineral Processing and Metallurgical Testing
- **John M. Marek**, of Independent Mining Consultants, Inc. – Mining Methods, Mine Capital and Operating Costs
- **James Barr**, of Tetra Tech Inc. – Data Verification and Mineral Resources Estimate
- **Dawn Garcia**, Independent consulting geologist – Environmental Studies, Permitting and Social or Community Impact

Table 2-1: Areas of Responsibility and Site Visit Dates

QP Name	Certification	Site Visit Date	Area of Responsibility
Daniel Roth	PE, P.Eng.	August 28, 2018	Sections 2, 18, 21.1.2, 21.1.3, 22, 27, and corresponding sections of 1, 25 and 26.
Dave R. Webb	PhD., P.Eng., P.Geo.	100 days between 2014 and 2018	Sections 3, 4, 5, 6, 7, 8, 9, 10, 11, 15, 19, 23, 24 and corresponding sections of 1, 14, 25 and 26.
Justin Black	PE	N/A	Section 17, 21.2.2 and corresponding sections of 1, 25 and 26.
John R.W. Fox	P.Eng.	N/A	Section 13, and corresponding sections of 1, 25 and 26.
John Marek	RM-SME	August 28, 2018	Sections 16, 21.1.1, 21.2.1, and corresponding sections of 1, 25 and 26.
James Barr	P.Geo.	July 18-19, 2017	Sections 12, 14, and corresponding sections of 1, 25 and 26.
Dawn Garcia	CPG, P.G.	November 5, 2018	Section 20, and corresponding sections of 1, 18.3, 25 and 26.

2.3   EFFECTIVE DATE

The effective date of December 6, 2018, applied to this report reflects the date the capital and operating costs were completed and the cut-off date by which all scientific and technical information was received and used for the preparation of the Technical Report.

2.4   TOPOGRAPHIC REFERENCE

The coordinate system used for the Mexican Hat Project is Universal Transverse Mercator (UTM) zone 12 and references the North American 1983 (NAD83) datum. The accuracy of the topography and surveyed drillhole collar locations as provided is assumed to be reliable. The Tetra Tech QP has approximately verified drill collar surveys in the field using handheld GPS.

The Mexican Hat Property is centered at approximately N 31° 48' 9.23" / W 109° 48' 26.17" (612,875 mE, 3,519,245 mN) and is located on the Turquoise Mountain Map 1:24,000 Map Sheet.

The topographic digital elevation model currently used for the property was developed by Cooper Aerial Surveys Co., located in Phoenix, Arizona.

2.5   TERMS AND DEFINITIONS

The report considers US Dollars ($) unless otherwise noted. Most units are metric, however, as noted and as standard for projects of this nature, certain statistics and scientific units are reported as avoirdupois or English units such salable precious metals which are described in terms of troy ounces and salable base metals which are described in terms of English pounds.

Table 2-2 shows the abbreviations that may be used in this report.

Table 2-2: Terms and Abbreviations

Abbreviation	Unit or Term
$	United States Dollars
$M	Million Dollars
% (grade)	Percent by weight (grade)
2-D	Two-Dimensional
3-D	Three-Dimensional
4WD	Four-Wheel Drive
AA	Atomic Absorption
AAS	Atomic Absorption Spectrometry
ABA	Acid Base Accounting
AG	Autogenous Grinding
Ag	Silver
As	Arsenic
AuEq	Gold Equivalent
ADR	Adsorption, Desorption, Regeneration
AGP	Acid Generation Potential
ANP	Acid Neutralization Potential
ARD	Acid Rock Drainage
AT	Assay Ton
Au	Gold
Bi	Bismuth
BLM	Bureau of Land Management
CDH	Core drillholes
CIC	Carbon-in-column
CIM	Canadian Institute of Mining Metallurgy and Petroleum
COG	Cut-off grade
CN	Cyanide
Co	Cobalt
Cu	Copper
CuEq	Copper Equivalent
CV	Coefficient of Variation (standard deviation/mean)
DEM	Digital elevation model
dmt	Dry metric tonne
dmt/h	Dry metric tonnes per hour
dmtpd	Dry metric tonnes per day
dmtpy	Dry metric tonnes per year
EIA	Environmental Impact Assessment
FA	Fire Assay
Fe	Iron
ft	feet
g	gram
g/cm^3	grams per cubic centimeter

Abbreviation	Unit or Term
g/t	grams per metric tonne
GMV	GMV Minerals Inc.
GPS	Global Positioning System
ha	hectare
HC	Humidity Cell
Hg	mercury
HP / hp	Horsepower
IRR	Internal Rate of Return
k	thousands
kg	kilograms
kg/t	kilograms per metric tonne
km	kilometer
km^2	square kilometer
kPa	kilopascal
kV	kilovolt
kW	kilowatt
kW-h	Kilowatt-hour
L	Liters
lb	pound
LOM	Life of Mine
m	meter
M	millions
m^3	cubic meter
M m^3	millions of cubic meters
m^3/h	cubic meters per hour
Ma	Million years old
masl	meters above sea level
MHGP	Mexican Hat Gold Property
Mo	Molybdenum
Mt	Megatonnes, or one thousand metric tonnes
MTPD	Metric Tonnes per Day
MW	megawatt
MY	Million years old
MWSF	Mine Waste Storage Facility
NGO	non-governmental organizations
NaCN	Sodium Cyanide
Ni	Nickel
NPV	Net Present Value
NSR	Net Smelter Return
ODI	Oneida Resources Inc.
opt	Troy ounces per English ton
oz/t	troy ounce per short ton

Abbreviation	Unit or Term
PAX	Potassium Amyl Xanthate
PDI	Placer Dome (USA) Inc.
PEA	Preliminary Economic Assessment
Pb	Lead
ppb	part per billion
ppm	Part per million
PROP	Propylitic
PSD	Particle Size Distribution
QA/QC	Quality Assurance/Quality Control
QFP	Quartz Feldspar Porphyry
QP	Qualified Person
RC	Reverse Circulation
RDP	Relative percent difference
RMR	rock mass rating
ROM	Run of Mine
rpm	revolutions per minute
RQD	rock quality designation
S	Sulphur

Abbreviation	Unit or Term
Sb	Antimony
SG	Specific gravity
SRF	stress reduction factor
t/d	metric tonnes per day
t/h	metric tonnes per hour
t/m^3	Tonne per cubic meter
$t/h/m^2$	Tonnes per hour per square meter
tonne	metric tonne
tonnes	dry metric tonnes (where one tonne = 1.1023 short tons)
tpa	Tonnes per annum
tpy	Tonnes per year
UTM	Universal Transverse Mercator coordinate system
VFD	Variable Frequency Drive
wmt	Wet Metric Tonne
µm	micrometer or micron

3   RELIANCE ON OTHER EXPERTS

The authors of this technical report have relied upon independent legal experts to establish and verify the legal status and ownership of the Company's mineral concessions and surface properties.

Reports received from other experts who are not authors of this technical report have been reviewed for factual errors by the authors. Any changes made as a result of these reviews did not involve any alteration to the conclusions made. Hence, the statements and opinions expressed in these documents are given in good faith and in the belief that such statements and opinions are not false or misleading at the date of this report.

This report has been prepared using publicly available documents, and information provided by GMV as part of the Mexican Hat Property data room. The data room included databases prepared previously by others, annual assessment reports, and various consulting and/or engineering reports completed since 1989.

Property tenure was examined to confirm that GMV Minerals Inc. has executed agreements to the property described in "Section 4: Property Description and Location", however this is not a legal opinion and as such we rely upon GMV's due diligence. The location and nature of the lode claims, and exploration permits, and their validity is based on discussions with GMV's legal counsel and documents provided by GMV and the state of Arizona. The QP's rely on these experts.

4   PROPERTY DESCRIPTION AND LOCATION

4.1   LOCATION

The property is located in Cochise County, immediately north of the Gleeson Courtland district, 10 km (6 miles) south of Pearce, Arizona, and approximately 140 km (90 miles) by road from Tucson, Arizona, and is centered on approximately N 31° 48' 9.23" / W 109° 48' 26.17" (612,875 mE, 3,519,245 mN) (Figure 4-1).

Figure 4-1: Location of Mexican Hat Project, Cochise County, Arizona (after McCleod, 2011)

4.2   MINERAL TENURE

The Mexican Hat Project consists of: (i) unpatented lode mining claims (the "Claims"), and (ii) State of Arizona Mineral Exploration Permits (the "Permits") (together, the "Property"). The mineral tenure boundaries are shown on Figure 4-2.

4.2.1   Leased Claims

Unpatented mining claims owned by Manuel R. Hernandez and leased to GMV Minerals (Nevada) LLC are described below (Mining Property Lease Agreement between Norman A. Pearson and Manuel R. Hernandez dated December 14, 2002, as assigned to GMV Minerals Inc. by Mining Property Lease Assignment Agreement dated May 14, 2014, as assigned to GMV Minerals (Nevada) LLC by Mining Property Lease Assignment Agreement dated May 31, 2014).

The claims cover certain areas whereby the surface estate is owned by the United States and managed by the Bureau of Land Management (S/2NE/4 and SE/4 of Section 4; and NE/4NW/4 and NW/4NE/4 of Section 9, T19S, R25E) and certain areas whereby the surface estate was patented pursuant to the Stock Raising Homestead Act and is owned by a private party, Ms. Kay Graham (Lots 1, 2, 3, 4, S/2NW/4 and SW/4 of Section 4, T19S, R25E).

Unpatented mining claims and sites situated in the Turquoise (Courtland, Gleeson) Mining District, in Section 33, Township 18 South, Range 25 East; and Sections 4 and 9, Township 19 South, Range 25 East; G&SRB&M, Cochise County, Arizona, the names of which together with the Fee # of recording of the location notices, and amendments thereto, in the official records of said county, and the serial numbers assigned by the Arizona State Office of the Bureau of Land Management, are as follows:

Table 4-1: Claims Leased to GMV

No.	Name of Claim	Fee #	BLM Serial No.
1	Victor #1	070308320	AMC379641
2	Victor #2	070308321	AMC379642
	Victor #2 / Amended	071137676	
3	Victor #3	070308322	AMC379643
4	Victor #4	070308323	AMC379644
5	Victor #5	070308324	AMC379645
6	Victor #6	070308325	AMC379646
7	Victor #7	070308326	AMC379647
8	Victor #8	070308327	AMC379648
9	Victor #9	070308328	AMC379649
	Victor #9 / Amended	071137677	
10	Victor #10	070308329	AMC379650
	Victor #10 / Amended	071137678	
11	Victor #11	070308330	AMC379651
	Victor #11 / Amended	071137679	
12	Victor #12	070308331	AMC379652
13	Victor #13	070308332	AMC379653
14	Victor #14	070308333	AMC379654
	Victor #14 / Amended	071137680	
15	Victor #15	070308334	AMC379655
16	Victor #16	070308335	AMC379656
17	Victor #17	070308336	AMC379657
	Victor #17 / Amended	071137681	
18	Victor #18	070308337	AMC379658
19	Victor #19	070308338	AMC379659
20	Victor #20	070308339	AMC379660
21	Victor #21	070308340	AMC379661
22	Victor #22	070308341	AMC379662
23	Victor #23	070308342	AMC379663
	Victor #23 / Amended	071137682	
24	Victor #24	070308343	AMC379664
	Victor #24 / Amended	071137683	
25	Victor #25	070308344	AMC379665

No.	Name of Claim	Fee #	BLM Serial No.
	Victor #25 / Amended	071137684	
	Victor #25 / Amended	080205183	
26	Victor #26	070308345	AMC379666
27	Victor #27	070308346	AMC379667
	Victor #27 / Amended	071137685	
	Victor #27 / Amended	080205184	
28	Victor #28	070308347	AMC379668
29	Victor #29	070308348	AMC379669
	Victor #29 / Amended	071137686	
30	Victor #30	070308349	AMC379670
	Victor #30 / Amended	071137687	
31	Victor #31	070308350	AMC379671
	Victor #31 / Amended	071137688	
32	Victor #32	070308351	AMC379672
	Victor #32 / Amended	071137689	
33	Victor #33	070308352	AMC379673
	Victor #33 / Amended	071137690	
34	Victor #34	070308353	AMC379674
	Victor #34 / Amended	071137691	
35	Victor #35	070308354	AMC379675
	Victor #35 / Amended	071137692	
36	Victor #36	070308355	AMC379676
	Victor #36 / Amended	071137693	
37	Victor #37	070308356	AMC379677
	Victor #37 / Amended	071137694	
38	Victor #38	070308357	AMC379678
	Victor #38 / Amended	071137695	
39	Victor #39	070308358	AMC379679
	Victor #39 / Amended	071137696	
40	Victor #40	070308359	AMC379680
	Victor #40 / Amended	071137697	

4.2.2 GMV Claims

Unpatented mining claims owned by GMV Minerals (Nevada) LLC are as described below. These claims cover certain areas whereby the surface estate is owned by the United States and managed by the Bureau of Land Management (NE/4NW/4 and NW/4NE/4 of Section 9, T19S, R25E), certain areas whereby the surface estate was patented pursuant to the Stock Raising Homestead Act and is owned by a private party, Ms. Kay Graham (SE/4NE/4 of Section 5, T19S, R25E), and certain areas whereby the surface estate was patented pursuant to the Taylor Grazing Act and is owned by the State of Arizona (Lot 1 of Section 5, T19S, R25E).

Unpatented mining claims and sites situated in the Turquoise (Courtland, Gleeson) Mining District, in Sections 5 and 9, Township 19 South, Range 25 East, G&SRB&M, Cochise County, Arizona, the names of which together with the document numbers of recording of the location notices, and amendments thereto, in the official records of said county, and the serial numbers assigned by the Arizona State Office of the Bureau of Land Management, are as follows:

Table 4-2: GMV Unpatented Mining Claims

No.	Name of Claim	Document No.	BLM Serial No.
1	Vicfract E*	2014-21306	AMC430047
2	Vicfract W*	2014-21307	AMC430048
3	GMV #1	2017-05878	AMC443113
4	GMV #2	2017-05879	AMC443114
5	GMV #3	2017-05880	AMC443115
6	GMV #4	2017-05881	AMC443116
7	GMV #5	2017-05882	AMC443117
8	GMV #6	2017-05883	AMC443118

*The Vicfract E and Vicfract W are overstaked fractions.
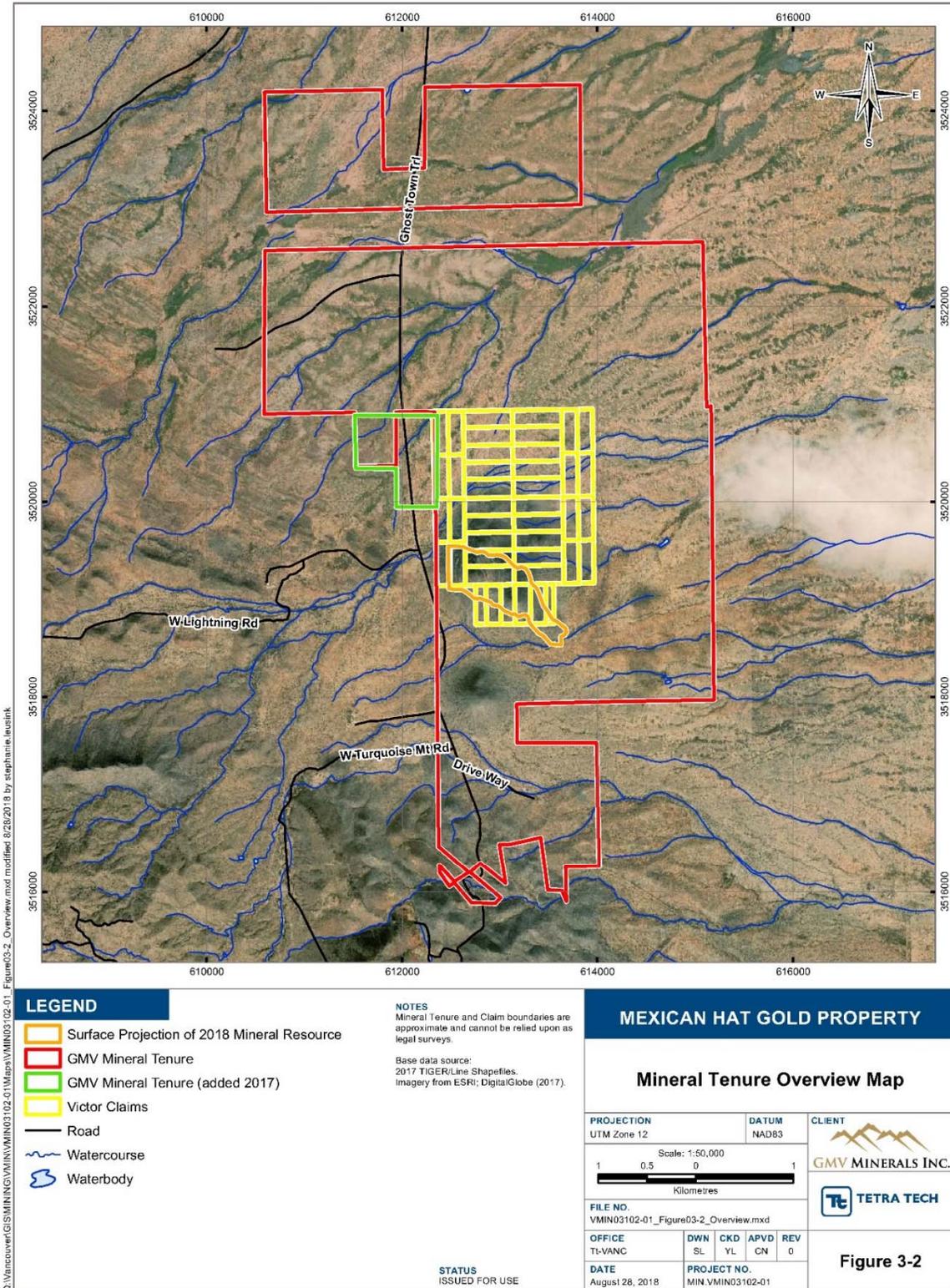
Figure 4-2: GMV Mineral Tenure, Overview Map

The Company also controls ten (10) Arizona Exploration Permits totaling 1,836.11 hectares (4,537.12 acres) as part of the Property.

Table 4-3: Arizona Exploration Permits held by GMV

Permit No.	Legal Description	Acres	Effective Date	Final Term. Date	Last Renewed Through Date
08-117862	Section 16, T19S, R25E; Lots 1, 2, 3, 4, 6, and 7, N2, NESE	482.66	10/23/2014	10/22/2019	10/22/2018
08-117863	Section 9, T19S, R25E; SW, W2NW, SENW, SWNE, N2SE, E2NE	480.00	10/23/2014	10/22/2019	10/22/2018
08-118106	Section 3, T19S, R25E; Lots 2, 3, and 4, S2NW, SWNE, SW, W2SE	521.90	5/7/2015	5/6/2020	5/6/2019
08-118167	Section 10, T19S, R25E; W2NE, NW, N2SW, NWSE	360.00	7/9/2015	7/8/2020	7/8/2019
08-119123	Section 33, T18S, R25E; All	640.00	3/28/2017	3/27/2022	3/27/19
08-119124	Section 34, T18S, R25E; W2, W2E2	480.00	3/28/2017	3/27/2022	3/27/19
08-119128	Section 28, T18S, R25E; N2, N2S2	480.00	3/28/2017	3/27/2022	3/27/19
08-119129	Section 29, T18S, R25E; NW, W2NE, N2S2	400.00	3/28/2017	3/27/2022	3/27/19
08-119130	Section 32, T18S, R25E; All	640.00	3/28/2017	3/27/2022	3/27/19
08-119131	Section 5, T19S, R25E; Lot 2	52.56	3/28/2017	3/27/2022	3/27/19

Permits are granted for a period of five years and give the right to explore for minerals pursuant to the terms and conditions of the Permit. During the five-year period, each Permit must be renewed on an annual basis by paying an annual filing fee of US$500.00, a US$1.00/acre (0.4 ha) rental fee, and by meeting minimum exploration work requirements or paying the cash equivalent ($10.00/acre for years 1-2, and $20.00/acre for years 3-5). If a mineral discovery is made in the Permit area, then a Mineral Lease is required before mining can commence. Mineral Leases have a twenty (20) year term and may be renewed for an additional term. Both rents and royalties for Mineral Leases are determined by appraisal with a statutory minimum royalty rate of 2% gross value.

The Claims must be maintained on an annual basis by paying the BLM a maintenance fee of US$155.00/claim no later than September 1st each year. The right to explore and mine on the Claims is governed by the General Mining Law of 1872 (30 U.S.C.A. § 22, et. seq.), BLM regulations, other federal, state and local laws and regulations, and an appropriate agreement with any private surface estate owners.

4.3   SURFACE OWNERSHIP AND LAND ACCESS AGREEMENTS

GMV Minerals Inc. acquired 100% of the leasehold interest in the Mexican Hat Property by way of an assignment agreement with the previous lessee and the underlying mining claim owner (New Release; GMV Minerals Inc., dated May 30, 2014).

4.4   LIABILITIES

No major environmental liabilities have been noted by the QP (Dave Webb). Exploration permits are applied for and permitted on an as-need basis.

4.5   PERMITS

Currently, GMV does not have any exploration permits in place, however, no issues are foreseen in applying for, and receiving permits for future exploration campaigns. GMV has agreements secured with neighbouring land owners to obtain and complete the necessary exploration and work programs on the Property.

4.6 ROYALTIES AND LIENS

The Mexican Hat Property is subject to a 3% net smelter returns royalty (NSR Royalty) in favour of Hernandez created pursuant to the terms of the Lease. The NSR Royalty is subject to a buy-back right pursuant to which 1.5% of the NSR Royalty can be purchased by the lessee in consideration of US$1,500,000. Pursuant to the terms of the Lease, the Company will be required to make advance royalty payments to Hernandez in the amount of US$4,500, payable quarterly. In addition, the Company will be required to do and record sufficient assessment work, make annual filings, and pay taxes, fees and rents as required to maintain the Mexican Hat Property in good standing. No additional factors or risks are known to the QP (Dave Webb) which would affect access, title, or the right or ability to perform work on the Property.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESSIBILITY

The present property holdings are a group of lode mineral claims and state mineral exploration permits. These lands are accessible under the provisions of the Mining Law of 1872, subject to obtaining approval from the US Forest Service or Bureau of Land Management. Ownership of the claims and mineral exploration permits gives the right, subject to federal, state and local permits and approvals, to explore for and develop mineral resources, but gives no surface rights.

The property is in the southeastern part of the State of Arizona, approximately 115 km east-southeast of Tucson, and can be accessed from the Old Ghost Town Rd., a gravel road extending south of the Town of Pearce or north from Gleeson Rd. Food and accommodations can be found in Pearce. Wilcox and Benson are both located about a 30 minute drive away from Pearce and are larger communities with a greater selection in accommodations.

Access on BLM controlled surface lands has been obtained using Letter of Intent with bonding. Additional or expanded surface access may require Plan of Operation filings. Access to State controlled lands has been obtained by Exploration Plan of Operations. Access to privately controlled surface lands has been obtained by written agreements with the owner.

5.2 CLIMATE

The property has a typical dry desert climate with hot summers and cool winters and is best described as semi-arid. The property area experiences an average of 30 cm of annual precipitation of which about 30% may occur as snow equivalent at higher elevations. Summers are typically hot with temperatures averaging about 25 degrees Celsius and occasionally exceeding 40 degrees Celsius. Temperatures in winter average about 5 to 10 degrees Celsius and occasionally reach lows of -10 degrees Celsius. The climate is generally amenable to year-round exploration work with adequate preparation.

Table 5-1: Average Climatic Data for Pearce, AZ between March 1950 and December 2005
From the Western Regional Climate Center: www.wrcc.dri.edu, 2014

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Annual
Ave. Max. Temp. (F)	61.1	64.5	70.0	77.7	86.1	94.9	93.9	91.1	88.8	80.6	68.9	60.7	78.2
Ave. Min. Temp. (F)	29.6	32.0	36.0	41.7	49.5	58.6	64.4	62.4	56.7	46.3	35.5	29.4	45.2
Ave. Total Precip. (in.)	0.8	0.7	0.5	0.2	0.2	0.5	2.8	3.1	1.2	0.8	0.5	0.8	12.1
Ave. Total Snow Fall (in.)	0.4	0.3	0.1	0.1	0.0	0.0	0.0	0.0	0.0	0.0	0.1	0.0	1.0
Ave. Snow Depth (in.)	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0

5.3 PHYSIOGRAPHY

The physiography of the Sulphur Springs Valley is in part defined by the basin and range province. The valley lies at an approximate elevation of 1,250 m and has an average width of 24 km. It is bounded on the west by the Dragoon Mountains and on the east by the Swisshelm Mountains. Further to the east lie the Chiricahua Mountains where Chiricahua Peak rises to 2,975 m. The project area lies within the southern terminus of the Dragoon Mountains. The dominant physiographic feature on the project area is Mexican Hat Hill which rises about 150 m above the ground level and attains an elevation of approximately 1,585 m. This feature is dominated by Tertiary age volcanic rocks that have undergone fracture controlled silicification and possible mineralization. The general features of project area are repeated on a smaller scale to its south, east and southeast as evidenced by the occurrence of other smaller, rounded, cone-shaped volcanic hills that in part form a north-easterly trending "train" into the valley. These may be a residual feature of underlying, low angle (thrust) or detachment faults.

In several locations about the area are occurrences of gold-bearing unconsolidated material as and/or desert wash, colluvium, alluvium and playa deposits of Tertiary age or younger. These occurrences which have undergone some development but apparently all have proven to be sub-economic. More recent unconsolidated deposits are localized about Mexican Hat Hill.

The physiographic setting of the property can be described as open, semi-arid range in the valley and within the confinement of bordering rugged mountain ranges on the west and east well beyond the project boundaries. The surface has been modified both by fluvial and wind erosion and the depositional (drift cover) effects of infilling. Thickness of drift cover in the valleys may vary considerably from very little to around 100 m. Santa Fe Gold Corp. reverse circulation drilling of 29 holes in 1996 disclosed that 8 holes encountered zero cover while the remaining 21 holes had an average of 10 m of cover with the deepest being 30 m. Drilling by the Company has not encountered more than 30 m of overburden.

5.4   LOCAL RESOURCES

5.4.1   Water

Water may be obtained from privately-owned and operated wells in the vicinity; however expanded operations will likely require purpose-built access to subsurface waters. Water is an important commodity in the southwest. Within the general area and about the project area, successful water wells that are presently shut-in have apparently been drilled (Hernandez, pers. com, 2014). Water has been encountered in every drillhole completed on the property by GMV, often at depths less than 50 m.

5.4.2   Infrastructure and Power

The Courtland-Pearce, Ghost Town Trail Road extends along the western project area along which runs an active power line. Storage facilities such as tailings, waste, potential heap leach areas and processing plant sites may be acquired either through the State, private land holder or BLM.

5.4.3   Community Services

The Towns of Willcox, Benson and Sierra Vista, AZ are the supply centers of Cochise County, as well as distribution hubs for local trade and commerce related to the well-developed agricultural, tourism and mineral industries. These Towns have the necessary resources to support mineral exploration such as, accommodations, communications, equipment and supplies and an available, knowledgeable work force and contractors. Larger or more specialized equipment can likely be acquired in the City of Tucson. Limited facilities to support mineral exploration exist to the south of Willcox, AZ and at Sunsites - Pearce, AZ.

Casual labour is provided from the local ranches and nearby towns.

Contract work for surveying and small heavy-duty equipment operations has been provided from the towns of Sierra Vista, Benson and Pearce.

Geophysical surveys have been contracted from firms with offices in Tucson, and assaying facilities are available in Tucson, or with preparation services in Tucson.

5.5   STORAGE AND WAREHOUSING

The Company currently rents an office and locked warehousing facility located north of the Property approximately 18 kilometers (11 miles) along Southern US Route 191, north of the community of Sunsites.

6   HISTORY

6.1   PRE-1980 HISTORY

There is a general lack of recorded information available on the project in the historical record prior to the 1980's. The following historical summary was provided by Mr. Hernandez, the property vendor as reported by McLeod, 2011.

The area around and immediately south of the project area underwent mining activity during the 16th or 17th century by early Indians seeking turquoise, a hydrous copper aluminum phosphate, semi-precious gemstone used for decorative purposes. Later Spanish explorers apparently worked the area for gold.

Mr. Hernandez reported that during the 1930's there was a gold exploration project carried-out on a portion of the present project area under the property name of the Gold Band prospect. It has been suggested (Ikona 2003) that this work may have been done on what is now referred to as the Victoria Shaft area. An early description of this work is that one vein, was developed "by an inclined shaft about 40 feet deep, and averages about four feet wide, with well-defined walls". It continues that "The vein strikes north - northwest and south - south east, and dips deeply to the East-south East". Further, "Short drifts have been run about north and south of the shaft, samples taken from faces of these drifts running as follows (paraphrased by the authors, C.K. Ikona, P. Eng. and R.G. Friesen, P. Geol., 2003):

- South Drift: 0.53 opt gold, 0.60 opt silver

- North Drift: 0.52 opt gold, 0.70 opt silver

- Vein along footwall: 0.31 opt gold, 0.50 opt silver

- General Shaft Sample, 20 ft. depth: 0.05 opt gold, 0.10 opt silver"

The authors, C.K. Ikona, P. Eng. and R.G. Friesen, P. Geol., state that, "The authors believe the shaft referred to in this report is probably the Victoria Shaft on the southeast flank of Mexican Hat Mountain, which was viewed by the authors".

6.2   1989-1995 HISTORY

In 1989, Oneida Resources Inc. ("ODI") of Vancouver, B.C. optioned what was a smaller part of the present day Mexican Hat Gold Property ("MHGP") from Mr. Manuel Hernandez and conducted surface exploration work completing a 1,524 m percussion drill program comprised of 20 holes.

During early 1989 Oneida geologists collected rock chip samples along 61 m (200') spaced E-W gridlines at 30 m (100') intervals, channel sample bulldozer trenches and had the samples analyzed for gold. When Placer Dome (USA) Inc ("PDI") assumed operation of the project, in the same year, they had the pulp samples of the ODI sampling fire assayed for gold (Au) and silver (Ag) including a 14-element geochemical package. This suite consisted of arsenic (As), antimony (Sb), copper (Cu), lead (Pb), zinc (Zn), molybdenum (Mo), cobalt (Co), nickel (Ni), bismuth (Bi), gallium (Ga), thallium (Tl), tellurium (Te), mercury (Hg) and fluorine (F). PDI expanded their trenching program where all samples were fire assayed for gold and silver and selected samples underwent the multi-element analyses. Only Au, Ag, As and Hg display a coherent and somewhat coincident distribution pattern. As presented by Gray (1990) the results are summarized for these four elements as follows:

Gold

Anomalous Au concentrations (several > 1 g/t) are coincident with the surface trace of the Victoria Fault Zone (VFZ) and they are irregularly present over 900 m of exposed strike length, on the southeast and east flank of MHGP. The east flank anomaly is coincident with an underlying zone mapped as exhibiting propylitic alteration. This irregular

shaped zone is approximately 210 m long by 30 m wide and still open to the east. It should be noted that to the southeast and east is where the SFPMI - PDH data is derived from.

Silver

Anomalous Ag concentrations (> 0.34 g/t) are coincident with the VFZ over 700 m of strike length SE of Mexican Hat Mountain. A 240 m by 610 m Ag anomaly straddles MHGP from W-E and is open on both ends.

Arsenic

Arsenic distribution closely mimics that of Ag. Anomalous As concentrations (> 20 ppm) follow the VFZ on the southeast of MHGP. An irregular shaped As anomaly 300 m wide by 610 m long trends ~ W-E over MHGP. The anomaly is open to the west and partially closed to the east.

Mercury

The Hg distribution reveals several scattered one point anomalies and a 185 m wide by 610 m long one that is centered over MHGP from W-E. The Hg anomaly is coincident with Ag and As and is open on both the W and E.

In 1989, Oneida entered into a joint venture with Placer Dome Inc "PDI" who during the period May 1989 - August 1990 spend $1.9 M (US) and subsequently earned a 60% interest in a portion of the MHGP. The drill totals were 18,939 meters (62,120 feet) that included 137 reverse circulation, rotary percussion drillholes (PDH) and 17 diamond core drillholes (CDH). In 1990 PDI conducted geostatistical analyses and historical estimate calculations utilizing 120 percussion drillholes ("PDH") and 15 core drillholes ("CDH") to stay within the area influenced by the drillholes. Auracle have not undertaken any independent investigation of the resource estimates nor have they independently analyzed the results of the previous exploration work to verify the resources.

During 1990 PDI conducted bottle roll intensive-cyanide leach tests on composite drillhole composites from the MHGP at their Golden Sunlight mine in Montana. They estimated that the gold bearing material was amenable to cyanide leaching. They reported gold recoveries averaging 93%, while NaCN consumption averaged 0.76 lb/ton and lime consumption averaged 5.43 lb/ton.

In 1989 Santa Fe Pacific Mining Inc. (SFPMI) who had a major exploration presence in the immediate area, acquired the portion of Section 9 that was not in the MHGP at that time and in 1990 they conducted a 29 RC program totaling 3,811 m along the west-south-east boundary perimeter of the MHGP. There is some speculation as to the exact location of each of these holes, but QP (Dave Webb) has seen the collar of SFPMI PDH #9, 16, 17, and 22. Their historical estimates are reported for intersections greater than 0.010 ounces of gold per ton (see Webb, 2015).

By far the most abundant data collection performed to date on the MHGP was conducted by PDI during the period 1989-1990 that comprised Phase 1 and Phase 2 exploration programs. When the Oneida - PDI joint venture was formed in 1989 Oneida had already performed some exploration work - rock chip sampling, trenching and a PDH (percussion drillhole) program comprised of 20 holes. This exploration work was carried-out on 40 contiguous, lode mineral claims that Oneida had leased that was centered about the MHGP.

When the joint venture began, PDI became the operator of the project and they incorporated the following sampling and assaying guidelines:

- ~2,155 rock channel samples were collected, where each was sampled over a 5-foot interval and weighed ~ 10 lbs. During the Phase 1 program, these were analyzed geochemically for As, Sb, Hg, Cu, Pb and Zn and fire assayed for Au and Ag. Subsequent samples were analyzed for Au and Ag by fire assay because the

multi-elements analyses did not appear to correlate with the Au values. During the Phase 2 program the channel samples were only fire assayed for Au.

- CDH holes 1- 6 were logged, split with a saw. One split was fire assayed at 5' intervals or less, the other split was saved as a record or for additional geochemical testing. CDH holes 7-17 were logged and then sampled by first taking representative sections of each lithologic or altered interval section that was saved for reference. The remaining material intervals underwent cyanide (CN) digestion and Au concentrations > 0.006 opt were fire assayed.

- PDH were sampled at 5' intervals and were split into two equal sized samples, one for analysis and the other that was kept for reference or metallurgical work. All holes were logged on site by a PDI geologist with the exception of PDH 1-21 that were drilled under the supervision of Oneida. Samples from PDH 1-41 (Phase 1 drilling) were fire assayed for Au and Ag. For PDH hole 42-125 (Phase 2 drilling) they were assayed for Au using a CN digestion and an atomic absorption (AA) detection method and samples with Au concentrations > 0.006 opt were subsequently fire assayed. Check assay samples were collected from 10% of the samples from the PDH. Check samples were analyzed at three independent laboratories of which two did the majority of the work (1 and 2) while a third (3) did some re-assay checks.

Reproducibility between the labs was reported to be acceptable.

The following is a comparison by twinning seven rotary percussion drillholes (PDH) with core drillholes (CDH (HQ size)) and in two instances twinned again by another CDH and again by another PDH. The sample twins were collared within a maximum of 20' of one another and generally within 10' of each other. The twin sets are listed in Table 6-1.

Table 6-1: Twinned Drillholes (after McLeod, 2011)

Percussion Hole	Twin Core Hole	Twin Percussion	Twin Core Hole	Remarks
89-10	1		89-10	drilled down the strike of a mineralized structure?
89-16	2		89-16	drilled down the strike of a mineralized structure?
89-28	5			CDH values < gold than PDH
89-38	6			To test correlation of gold from the West Fault, correlation poor by high variability. CDH intercepted gold where PDH did not.
90-98	7		8	Excellent correlation between mineral intercepts. Poor correlation between PDH & CDH.
89-41	12	89-79		Good correlation of PDH's moderate between PDH & CDH.
89-4	13			To test the correlation in a high-grade section of the Victoria Fault Zone. While the grade was < and the top of section was higher the CDH gave a similar thickness of the high grade to the PDH.

In 1991, Oneida was unable to proceed with its 40% interest in the joint venture which then was reduced to a 20% interest and PDI planned to sell its interest.

In 1992, Oneida announced that it had purchased the PDI 80% interest in the Victor Claims.

During the period, 1992-1995 Oneida works on project financing, but did not perform further work.

6.2.1   1992 Metallurgical Test Work

Historical metallurgical testing reported by Gray (1992) includes bottle roll leaching using 2.0 lb NaCN per ton at 25% solids at a pH of 11 and a 48 hour leach time and a 35% +100 mesh sample size yielding recoveries ranging from 82.8 to 97.4 from both oxide and sulfide ores with feed grades between 1.097 and 33.394 g/t gold. Both gold and lime consumption were reported as low.

The same material was retested at a finer grind (10% +100 mesh) under the same conditions yielded gold recoveries ranging from 97.4 to 98.7%.

6.3   1995-1996 HISTORY

During December 1995, Oneida reported that Kalahari Resources Inc. of Vancouver, B.C. could earn a 60% interest in, what at the time, was a smaller area of the MHGP by spending $2.25 million and producing a feasibility report.

In 1996, an 18 percussion drillhole program totaling 12,375 feet was completed by Kalahari. A follow-up three phase program was planned that was to include a further 10,000 feet of PDH and 2,000 feet of CDH that was not completed.

6.4   2008 TO 2013 HISTORY

In 2002, Mr. N.A. Pearson (Lessee) of Burnaby, B.C. was granted a twenty-year lease by Manuel Hernandez (Lessor) on the original 40 Mexican Hat Gold Property claims. Mr. Pearson carried-out some initial sampling at several unconsolidated material sites, acquiring sample concentrates and having concentrate samples analyzed.

In 2003, Mr. Pearson transferred all of his interest to Capitol Hill Gold Corp. of Vancouver, B.C. Mr. Pearson at that time commissioned Pamicon Developments Ltd. of Vancouver, B.C., (Geological Consultants) to undertake a more rigorous unconsolidated material sampling program. This program was carried-out and results "of possible economic interest" having been obtained from two areas on the eastern flank of MHGP referred to as the Victoria Shaft area and the Hot Linda area.

In 2004, Capitol Hill undertook a limited diamond core drilling program on the 40 claims area of the MHGP. The program included drilling four NG drillholes for a total of 517 feet of a planned total of 1,000'. All four holes suffered overage costs due to bad ground conditions and the program was adjusted. The program was conducted under the direction and supervision of Pamicon.

During the subsequent period, 2004-08 Mr. N.A. Pearson kept the Lease in good standing, fulfilling his obligations to the registered owners while preparing to arrange an exploration campaign.

During May 2009 Mr. Pearson optioned the Lease to MH Holdings (then named Auracle Resources Ltd.).

Auracle Resources Ltd. conducted a diamond drill, surface sampling, and geophysics program between 2009 and 2011. Most of the road cuts over the Mexican Hat hill and environs were chip sampled, 206.4-line km of ground magnetic and VLF surveys and a 19 drillhole (2,579.5 m) program was conducted (Game, 2013).

6.4.1   Geophysics

Historical Ground magnetic and VLF electromagnetic surveys were completed by Auracle Resources and have since been updated by GMV during recent exploration activities.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

The Mexican Hat Property lies within the southeastern portion of the Basin and Range Province, a physiographic domain encompassing much of Nevada, southeast California, southern Arizona and parts of New Mexico (Figure 7-1).
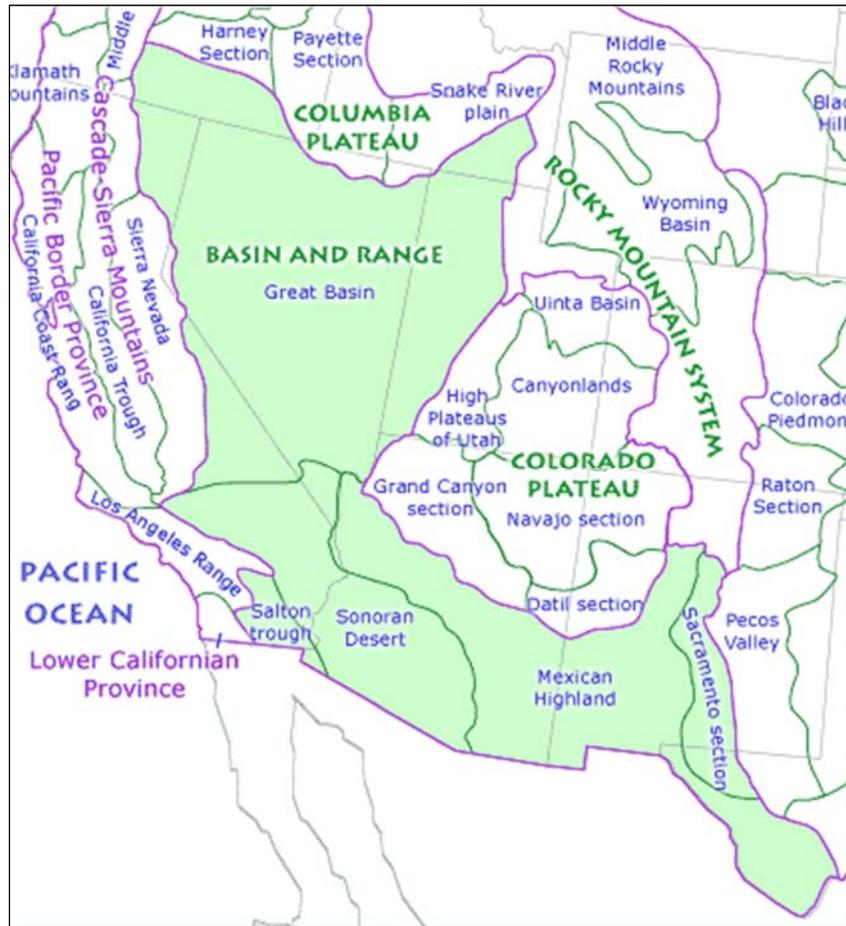
Ref. www.geomaps.wr.usgs.gov/parks/province. 2014.

Figure 7-1: Location of the Basin and Range Physiographic Domain and Five Subdomains.

The general area about the MHGP is underlain by rock units ranging in age from Precambrian through to Tertiary (Figure 7-2). Some basement units in the general area have been dated at 1.7 billion years and are composed of metamorphosed sediments of what is termed the Pinal Schist. These basement units are overlain by a thick sequence 1,700 m. of Paleozoic sedimentary rock units in which there is no record of intrusive igneous activity. A thicker sequence of Mesozoic sedimentary and volcanic rock units has been described to overly the Paleozoic units. The youngest rock units observed in the region are Tertiary age volcanic rocks. The host units at the MHGP are believed to be mid-Tertiary age volcanic rocks. Many parts of the general area are overlain by unconsolidated surface material that may be comprised of and/or desert wash, colluvium, alluvium and playa deposits of Quaternary age. Igneous activity was widespread during the Jurassic, Cretaceous-Tertiary (including the Laramide orogeny) and middle Tertiary periods.
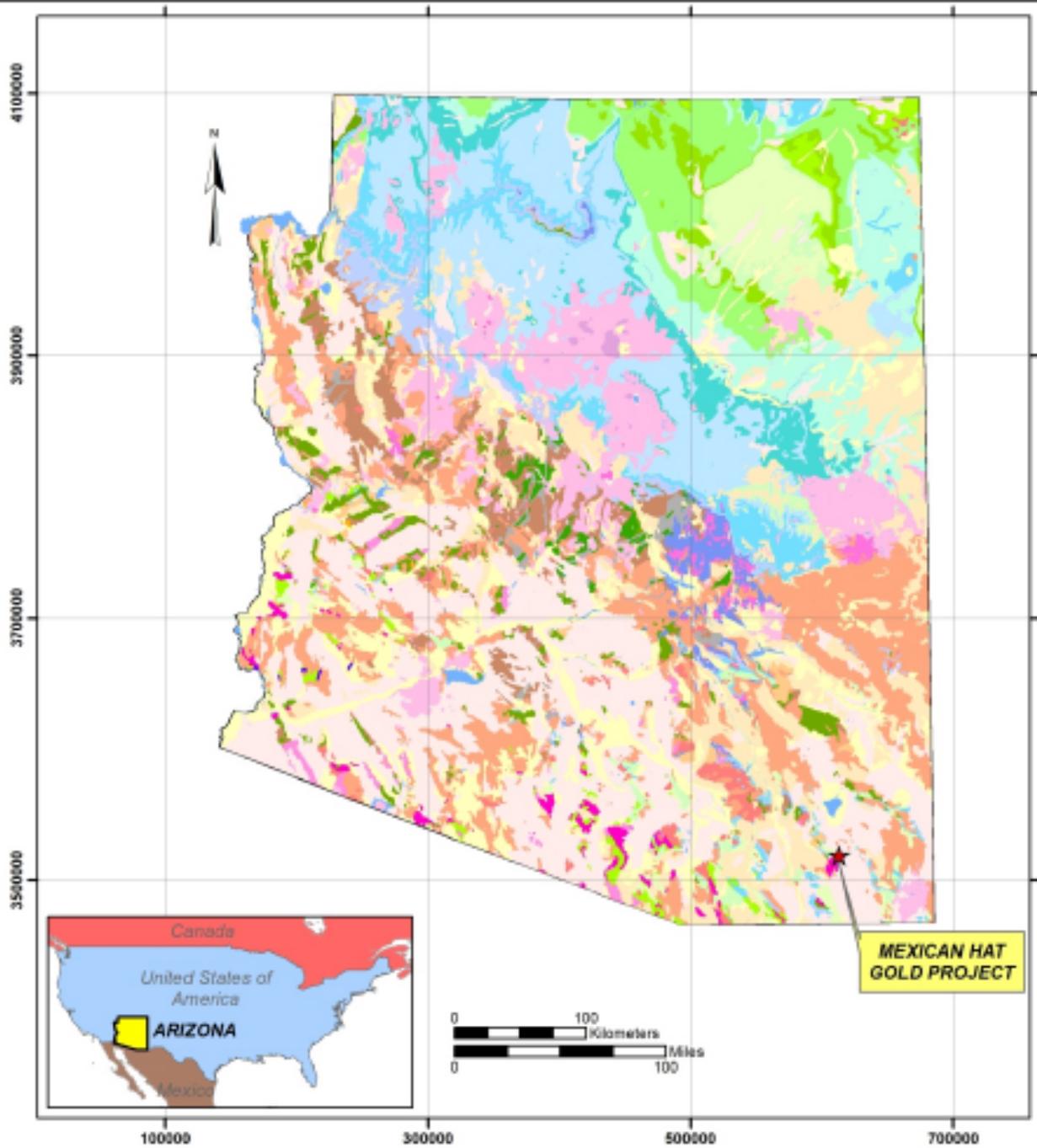
Map Legend on Following Page

Figure 7-2: General Geology of Arizona showing Location of Mexican Hat Project, after Game (2013)

Map Legend

- Q - Quaternary Surficial deposits, undivided (0-2 Ma)
- QTb - Holocene to Middle Pliocene Basaltic Rocks (0-4 Ma)
- QTv - Holocene to Middle Pliocene Volcanic Rocks (0-4 Ma)
- Qr - Holocene River Alluvium (0-10 ka)
- Qy - Holocene Surficial Deposits (0-10 ka)
- Qm - Late And Middle Pleistocene Surficial Deposits (10-750 ka)
- Qo - Early Pleistocene to Latest Pliocene Surficial Deposits (0.75-3 Ma)
- QTs - Early Pleistocene to Late Miocene Basin Deposits (0.75-10 Ma)
- Tvy - Pliocene to Middle Miocene Volcanic Rocks (2-12 Ma)
- Tsy - Pliocene to Middle Miocene Deposits (2-16 Ma)
- Tby - Pliocene to Late Miocene Basaltic Rocks (4-8 Ma)
- Tb - Late to Middle Miocene Basaltic Rocks (8-16 Ma)
- Tsv - Middle Miocene to Oligocene Volcanic And Sedimentary Rocks, Undivided (11-32 Ma)
- Tsm - Middle Miocene to Oligocene Sedimentary Rocks (11-32 Ma)
- Tv - Middle Miocene to Oligocene Volcanic Rocks (11-38 Ma)
- Tg - Middle Miocene to Oligocene Granitic Rocks (14-26 Ma)
- Ti - Middle Miocene to Oligocene Shallow Intrusions (14-35 Ma)
- TXgn - Tertiary to Early Proterozoic Gneissic Rocks (15-1800 Ma)
- Tso - Oligocene to Paleocene[?] Sedimentary Rocks (30-65 Ma)
- TKgm - Early Tertiary to Late Cretaceous Muscovite-Bearing Granitic Rocks (50-80 Ma)
- TKg - Early Tertiary to Late Cretaceous Granitic Rocks (50-82 Ma)
- Kv - Early Tertiary to Late Cretaceous Volcanic Rocks (50-82 Ma)
- KJo - Orocopia Schist (Cretaceous - Jurassic, 65-165 Ma)
- KJs - Cretaceous to Upper Jurassic Sedimentary Rocks with Minor Volcanic Rocks (80-160 Ma)
- Kmv - Sedimentary Rocks of the Upper Cretaceous Mesaverde Group (84-88 Ma)
- Ks - Cretaceous Sedimentary Rocks (about 88-97 Ma)
- Jm - Morrison Formation (Late Jurassic, about 145-160 Ma)
- Jsv - Jurassic Sedimentary and Volcanic Rocks (150-170 Ma)
- Jg - Jurassic Granitic Rocks (150-180 Ma)
- Jv - Jurassic Volcanic Rocks (160-200 Ma)
- J^ - Jurassic And Triassic Sedimentary and Volcanic Rocks (160-240 Ma)
-)[- Jurassic to Cambrian Metamorphosed Sedimentary Rocks (160-540 Ma)
- Js - San Rafael Group (Late to Middle Jurassic, about 160-180 Ma)
- Jgc - Glen Canyon Group (Early Jurassic, about 180-210 Ma)
- ^c - Chinle Formation (Late Triassic, 210-230 Ma)
- ^cs - Shinarump Conglomerate Member
- ^m - Moenkopi Formation (Middle(?) and Early Triassic, 230-245 Ma)
- | - Paleozoic Sedimentary Rocks (248-544 Ma)
- P - Permian Sedimentary Rocks (270-280 Ma)
- P^ - Permian to Pennsylvanian Sedimentary Rocks (280-310 Ma)
- M_ - Mississippian, Devonian, And Cambrian Sedimentary Rocks (330-540 Ma)
- Ys - Middle Proterozoic Sedimentary Rocks (700-1300)
- Yd - Middle Proterozoic Diabase (1050-1150 Ma)
- YXg - Proterozoic Granitic Rocks (1400-1800 Ma)
- Yg - Middle Proterozoic Granitic Rocks (1400-1450 Ma)
- Xg - Early Proterozoic Granitic Rocks (1600-1800 Ma)
- Xms - Early Proterozoic Metasedimentary Rocks (1600-1800 Ma)
- Xq - Early Proterozoic Quartzite (1650? -1700 Ma)
- Xmv - Early Proterozoic Metavolcanic Rocks (1650 to 1800 Ma)
- Xm - Early Proterozoic Metamorphic Rocks (1600-1800 Ma)

Calcareous or limey rocks exposed on the west-side of Pearce Road (Ghost town Trail) appear to be part of the Bisbee Group which underlies much of the nearby Turquois Mountains (Figure 7-3).
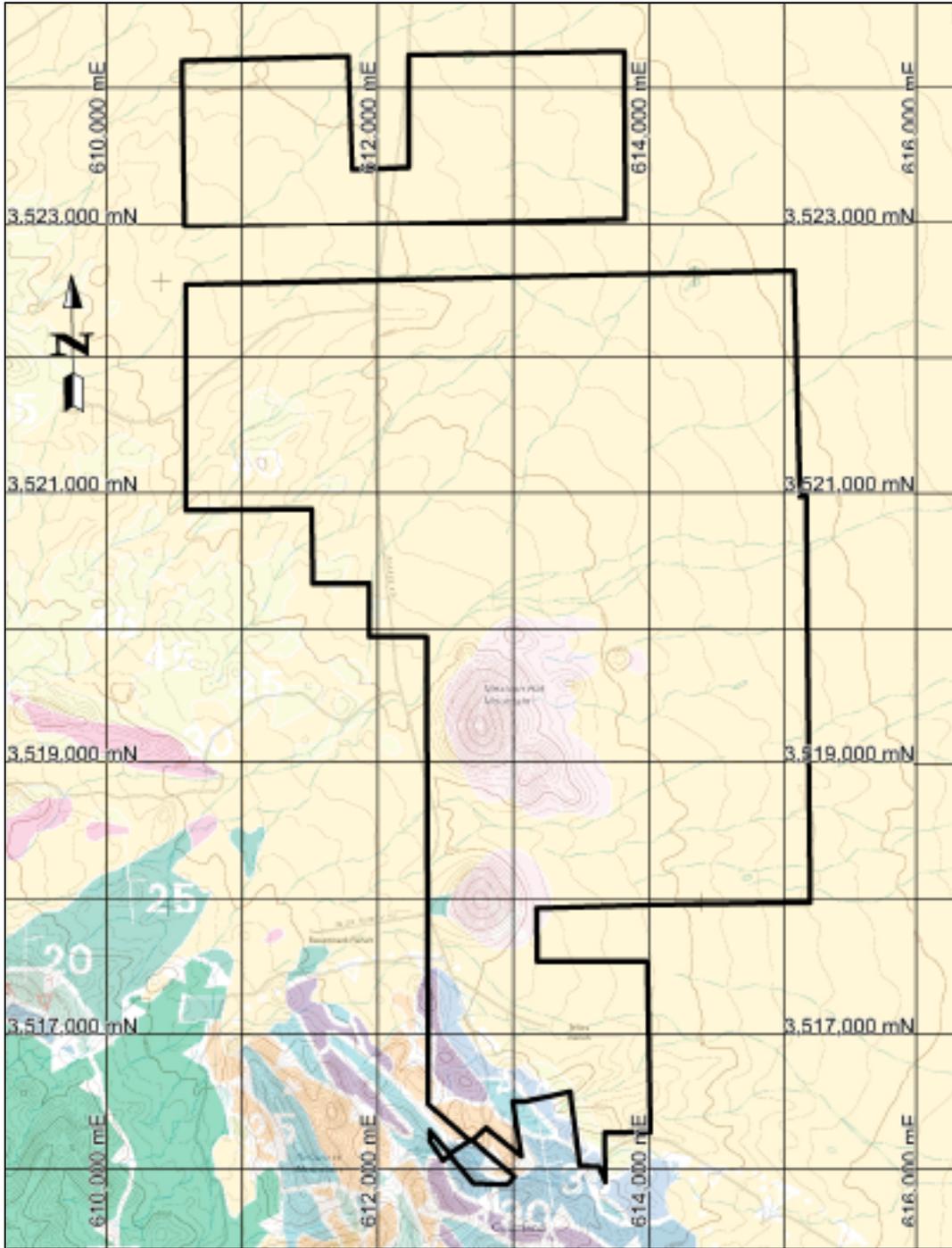
Legend following below

Figure 7-3: General Geology after Map I-1109 east, Drewes, 2002

LIST OF MAP UNITS

SURFICIAL DEPOSITS

Qg - Gravel, sand, and silt (younger surficial deposits)

Qd - Sand and silt (younger surficial deposits)

Qp - Gravel, sand, silt and clay (younger surficial deposits)

QTg - Gravel, sand, and silt (older or undifferentiated surficial deposits)

QTgu - Gravel, sand, and silt (older or undifferentiated surficial deposits)

QTb - Basalt (older or undifferentiated surficial deposits)

Tuc - Upper conglomerate, gravel, and sand (older or undifferentiated surficial deposits)

IGNEOUS AND SEDIMENTARY ROCKS

Tb - Basalt (upper igneous and sedimentary rocks)

Tc - Conglomerate (upper igneous and sedimentary rocks)

Tva - Extrusive andesite and dacite (upper igneous and sedimentary rocks)

Tv - Extrusive rhyolite and rhyodacite (upper igneous and sedimentary rocks)

Tug - Granitoid rocks (upper igneous and sedimentary rocks)

Ti - Intrusive rhyolite and rhyodacite (upper igneous and sedimentary rocks)

Ta - Andesite (upper igneous and sedimentary rocks)

Tlc - Lower conglomerate, gravel, and sand (upper igneous and sedimentary rocks)

CORDILLERAN (LARAMIDE) IGNEOUS AND SEDIMENTARY ROCKS

Tg - Granitoid rocks (uppermost Cordilleran (Laramide) igneous rocks)

Tlp - Quartz latite porphyry (uppermost Cordilleran (Laramide) igneous rocks)

Tlv - Lower volcanic rocks (uppermost Cordilleran (Laramide) igneous rocks)

Tlg - Lower granitoid rocks (uppermost Cordilleran (Laramide) igneous rocks)

TKp - Porphyritic and aplitic intrusive rocks (main Cordilleran (Laramide) igneous rocks)

Kd - Diorite and quartz diorite (main Cordilleran (Laramide) igneous rocks)

Kq - Quartz monzonite (main Cordilleran (Laramide) igneous rocks)

Kg - Granodiorite (main Cordilleran (Laramide) igneous rocks)

Kus - Upper sedimentary rocks (lower Cordilleran (Laramide) igneous and sedimentary rocks)

Kr - Rhyodacite tuff and welded tuff (lower Cordilleran (Laramide) igneous and sedimentary rocks)

Ka - Andesitic to dacitic volcanic breccia (lower Cordilleran (Laramide) igneous and sedimentary rocks)

Ca - Abrigo Formation and Bolsa Quartzite, undifferentiated

Yd - Diabase

Ys - Dripping Spring Formation and Pioneer Formation (Apache Group)

Yg - Granodiorite and quartz monzonite (granitoid rocks)

Yw - Wrong Mountain Quartz Monzonite (granitoid rocks)

Yr - Rincon Valley Granodiorite (granitoid rocks)

Yc - Continental Granodiorite (granitoid rocks)

Yt - Tungsten King Granite (granitoid rocks)

Xj - Johnny Lyon Granodiorite (granitoid rocks)

Xp - Pinal Schist

Xi - Rhyolite porphyry

Contact, dotted where concealed, queried where uncertain

Contact, unspecified local marker horizon

Contact, unspecified local marker horizon, concealed

Contact, base of Mural Limestone where extensively exposed

Contact, base of Mural Limestone, concealed

Fault, unknown offset, dotted where concealed

Normal fault, dotted where concealed

Reverse fault, dotted where concealed

Left-lateral strike-slip fault, dotted where concealed

Left-lateral strike-slip fault with normal motion, dotted where concealed, dashed where approximate

Left-lateral strike-slip fault with reverse motion, dotted where concealed

Right-lateral strike-slip fault, dotted where concealed

Right-lateral strike-slip fault with normal motion, dotted where concealed

Thrust fault, dotted where concealed, teeth on upper plate

Glide fault, dotted where concealed, teeth on upper plate

Reactivated fault, dotted where concealed, teeth on upper plate

Anticline, dotted where concealed

Anticline, overturned

Syncline, dotted where concealed

Syncline, overturned

Kuvs - Volcanic and sedimentary rocks, undifferentiated (lower Cordilleran (Laramide) igneous and sedimentary rocks)

Kq - Lower quartz monzonite and granodiorite (lower Cordilleran (Laramide) igneous and sedimentary rocks)

Ks - Sedimentary rocks (lowest Cordilleran (Laramide) sedimentary rocks)

Kl - Rhyodacite porphyry (lowest Cordilleran (Laramide) sedimentary rocks)

IGNEOUS AND SEDIMENTARY ROCKS

Kb - Upper part of Bisbee Formation or Group, undifferentiated, and related rocks (Bisbee Formation or Group, undifferentiated)

Kbu - Upper part of Bisbee Formation or Group, undifferentiated, and related rocks (Bisbee Formation or Group, undifferentiated)

Kbg - Glance Conglomerate of Bisbee Group or Glance Conglomerate of Bisbee Formation

Klvs - Andesitic to rhyolitic volcanic rocks, conglomerate, and sandstone (lower volcanic and sedimentary rocks)

Jg - Stocks of pinkish-gray coarse-grained rock (granite and quartz monzonite)

JRi - Rhyolitic porphyry plutons, dikes, and sills (Intrusive rocks)

JRvs - Rhyolitic tuff, welded tuff, lava, sandstone, and conglomerate (volcanic and sedimentary rocks)

Tm - Stocks of dark-gray very coarse-grained monzonite and quartz monzonite (monzonitic rocks)

Ts - Red mudstone, sandstone, and conglomerate, and intercalated rhyodacite volcanic rocks

Tvs - Rhyolitic to andesitic lava and pyroclastic rocks and intercalated sandstone, quartzite, and some conglomerate

Ps - Rainvalley Formation to Bolsa Quartzite, undifferentiated

PPn - Rainvalley Formation, Concha Limestone, Scherrer Formation, Epitaph Dolomite, Colina Limestone, Earp Formation and Horquilla Limestone, undifferentiated (Naco Group)

Ps - Sedimentary rocks of the Rainvalley Formation, Concha Limestone, and Scherrer Formation, undifferentiated (Naco Group)

PPs - Sedimentary rocks of the Epitaph Dolomite, Colina Limestone, and Earp Formation, undifferentiated (Naco Group)

Ph - Horquilla Limestone (Naco Group)

MDs - Escabrosa Limestone and Martin Formation, undifferentiated

OЄs - El Paso Limestone, Abrigo Formation and Bolsa Quartzite, undifferentiated

Ti - intrusive rhyolite and rhyodacite - laccoliths, and dikes

Tip - quartz latite porphyry - plugs, b and dikes

Tkp - porphyritic and aplitic intrusive

Kg - granodiorite - stocks of gray, me locally porphyritic rock

Aplite dikes

Local tuff marker beds in upper cong sand and gravel unit (Tuc)

Maar crater

Paleoplaya boundary

Political boundary, state, national, ar

Map boundary (lines of latitude or lor

⊕ Horizontal bedding

Inclined bedding

Vertical bedding

Overturned bedding

Inclined foliation

Vertical foliation

→ Lineation

Dip of fault

Collection site, query mark to left of symbol where precise location uncertain

Cinder cone, queried where uncertain

Plunge of fold axis

7.2   LOCAL GEOLOGY

The property occurs on the east-side of the Dragoon Mountains and is underlain by mixed mafic to felsic peralkaline volcanic rocks. A fault of regional extent bounds the property to the west, and alluvium covers the eastern, northern and southern extents of the property. A prominent hill, the Mexican Hat Hill dominates, and several smaller hills define the physiography of the property. Several structures of more local extent can be observed on the flanks of the hills on the property and defined by geophysics.

The basal unit is a light-coloured thick rhyolite breccia consisting of a polylithic tuff to tuff breccia including fragments of limestone, argillite, andesite and latite up to 20% in places. No primary structures were observed. Fragments up to several meters were noted within the dominantly lapilli-tuff. Rhyolite is exposed near the base of Mexican Hat Hill to the south and southeast as well as to the northwest with limited exposures to the west.

A dark grey-green porphyritic basalt (trachybasalt) flow conformably overlies the rhyolite. Large breccia-sized fragments of this unit occur within the rhyolite immediately below the basalt. Euhedral phenocrysts of pyroxene, amphibole, and to the south, biotite (pseudomorphs after pyroxene) occur within the basalt. A transition to a trachyandesitic flow is demarked by the addition of euhedral feldspar phenocrysts to the flow. The preponderance of phenocrysts to the east of Mexican Hat Hill gives the unit a decidedly trachytic appearance. The basalt – andesite is exposed to the north, east and in a limited fashion to the south of Mexican Hat Hill.

A medium to light-grey latite to quartz latite is exposed at higher elevations on Mexican Hat Hill, and on some smaller hills to the east-northeast and south-southeast of Mexican Hat Hill. This unit occurs dominantly as a crystal to lapilli tuff, to agglomerate and minor tuff breccia. In places, the latite displays lamination or banding interpreted as bedding.

The local geology is shown in Figure 7-4.

The rock names and descriptions are supported by petrographic analysis carried out by Vancouver Petrographics (Leitch, 2014).
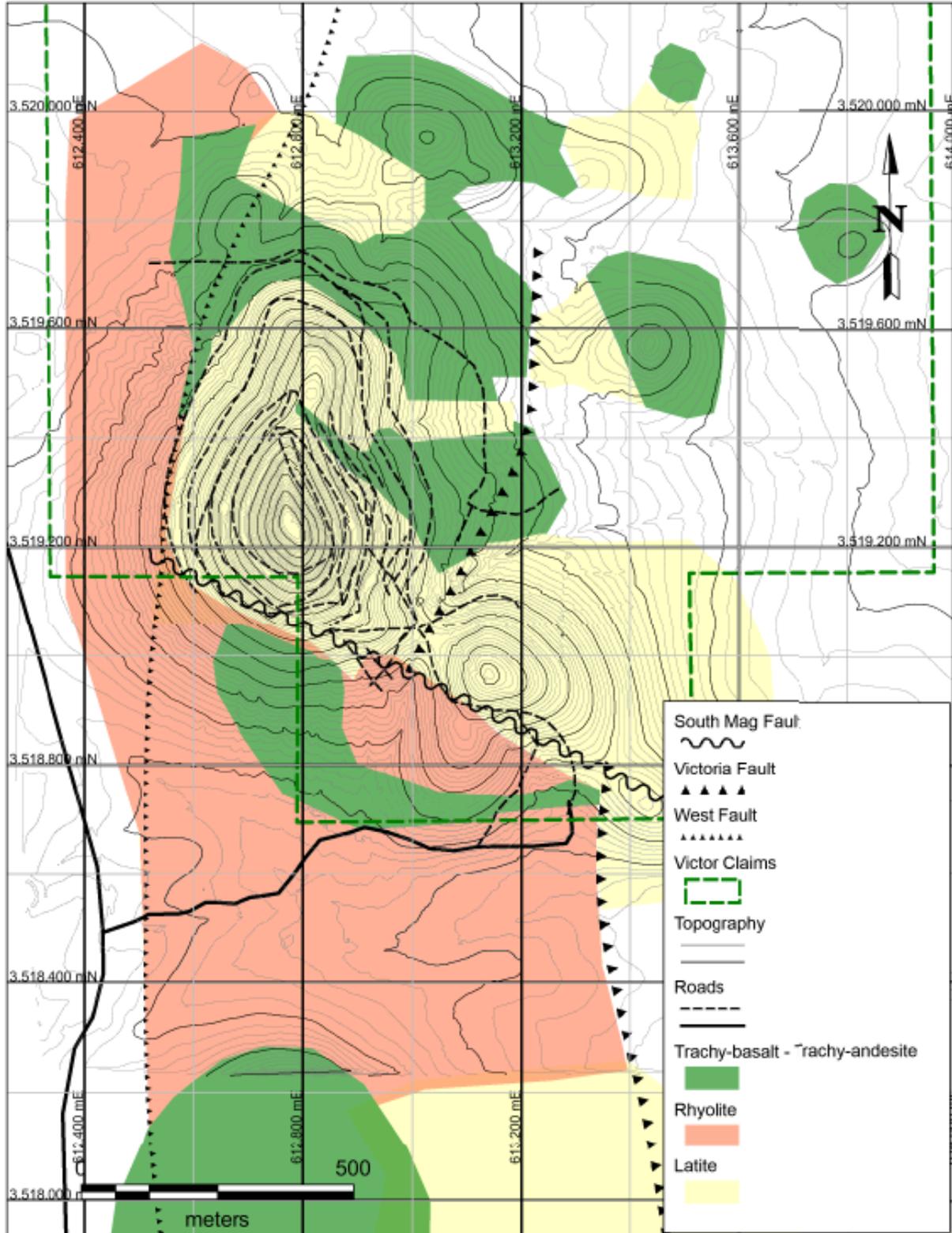
Figure 7-4: Simplified Property Geology, from Webb, 2015

7.2.1 Mineralization

Gold and silver mineralization is associated with moderate to strong oxidized zones of hematite and limonite. Hematite and limonite are directly related to and fill, in part, dominant NE/SW fault and related fractured zones including secondary NW/SE fault and fractured zones. A dark metallic sulphide has been locally observed (Trench TR 14) and may be a silver sulfa-salt. Hematite may be the result of low-temperature alteration of primary or secondary magnetite. Limonite, a secondary mineral after pyrite is common in surface oxidized zones. Malachite and azurite was observed in several locations across the property within trachyte andesite (Webb, 2015). In addition, the main elevated metal assemblage from assay results include Au, Ag, As, Hg and Sb. Limits of mineralized zones have not been determined. No sulphides or visible gold has been identified from surface exploration campaigns.

7.2.2 Alteration

Three main types of alteration are recognized on the Mexican Hat property. First, carbonate alteration is the most common within all Tertiary volcanic rocks mapped on the property. Carbonate alteration is pervasive and ranges from weak to strong. Second, weak propylitic alteration (epidote + weak chlorite in part) was observed and is association with zones of fracturing. Third, weak to moderate silicification was observed within and adjacent to some extensional structures. Sericite and K-Feldspar alteration typical of low sulphidation alkali epithermal deposits may be present but these types of alteration were not observed. A Pima or suitable alteration survey may be useful in identifying all alterations present in the main elevated gold/silver zones.

7.2.3 Structure

Brittle faults were mapped at the Mexican Hat property within Tertiary volcanic rocks. Faulting is more common than fractures in oxidized mineralized (hematite/limonite) zones. Three types of faulting were recognized and are listed below in order of importance:

- Oblique-Slip faults are most common and were observed and mapped (Detailed mapping: TR 1, TR 2 and TR 4) in most continuous chip trenches. Oblique-slip faults display both a strike-slip and dip-slip component and results from a combination of shearing and tension produced by compressional forces.

- Normal Faults occur throughout the mapped trenches on the Mexican Hat property and are caused by tensional forces and results in extension.

- Reverse faults are less common on the Mexican Hat property. This fault motion is caused by compressional forces and results in shortening.

Brittle faults mapped in detail (TR 1, TR 2 and TR 4) typically trend northeast–southwest (dominant fault direction – approximately three times more common than the NW-SE fault direction) with fault planes dipping generally between 58 to 90 degrees to the southeast (most common) and northwest. Gold and silver assays are generally higher in faulted areas that trend NE-SW and contain hematite. Brittle faults were also mapped trending northwest-southeast. These brittle extensional faults identified on the Mexican Hat property are typical in the Sierra Madre Occidental province where, for example, extensional forces caused the Baja California land mass to separate from western Sonora. The extensional forces also created and reactivated northeast and, in part, northwest orientated fault and fracture zones that acted as hosts to mineralizing fluids.

Fractures are common on the Mexican Hat property and include mineralized and non-mineralized fractures. Detailed trench mapping shows that the dominant mineralized fracture direction is northeast-southwest (twice as many mineralized fractures trend NE-SW than NW-SE) and dip steeply to the SE and NW.

Possible volcanic layering or laminations observed in trenches trend NW/SE (175/71 SW and 297/55 NE) and dip moderately to the southwest and northeast at 55 and 71 degrees. Northeast trending laminations or volcanic layering were observed in trench 16 dipping moderately to the northwest (49/73 NW).

Possible sedimentary bedding orientation observed in TR 7 was mapped at approximately 341/46 NE.

A calcite veinlet observed in TR 1 trends northwesterly at 333/55 NE. A barren quartz/chalcedony veinlet approximately 1-4 cm wide was also mapped in TR 14 trending NW/SE at 123/54 SW and hosted in latite volcanic rocks.

These structures are shown graphically below in Figure 7-5 through Figure 7-8.
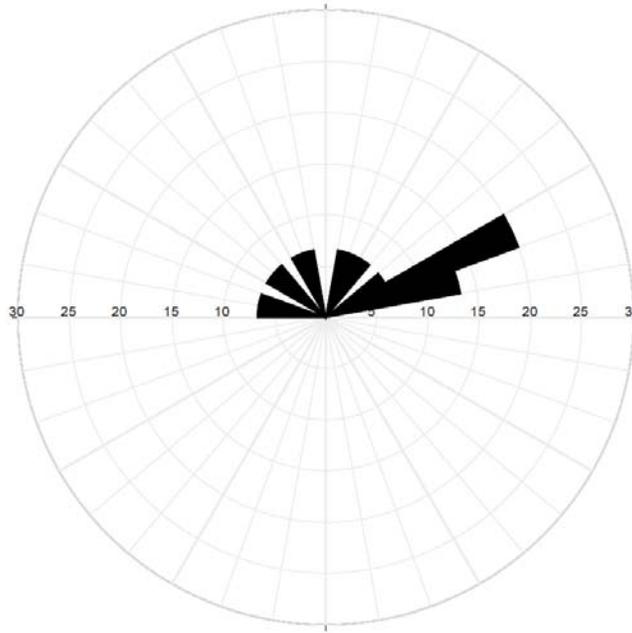
Figure 7-5: Rose Diagram of Strikes of all Faults Measured in Trenching Program
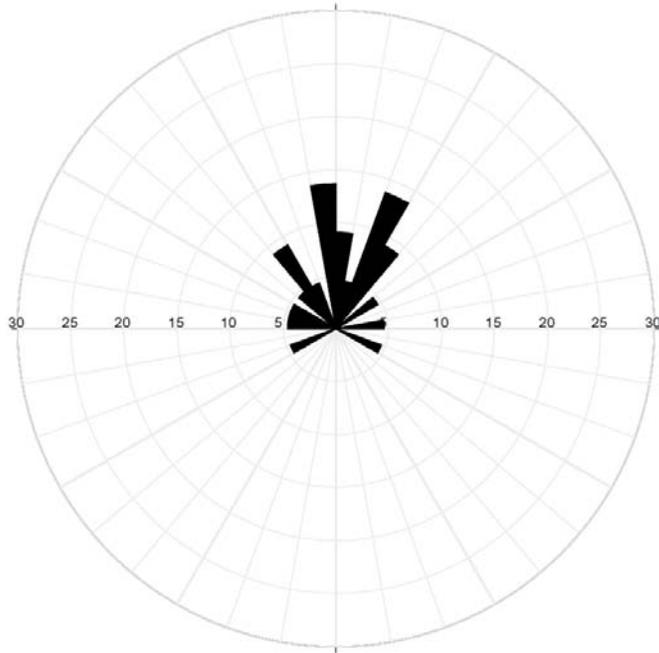
Figure 7-6: Rose Diagram of all Fractures with no Apparent Movement Measured in Trenching Program
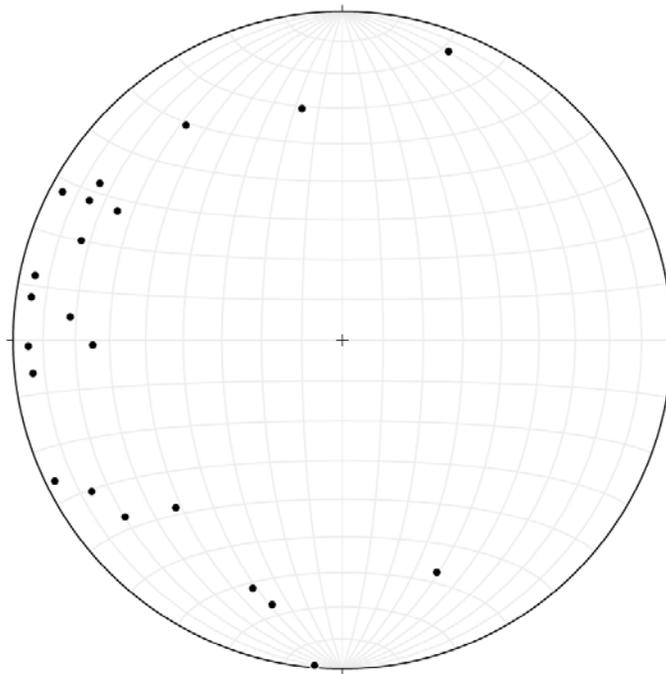
Figure 7-7: Stereonet Plot of Poles to all Fractures Measured in Trenches
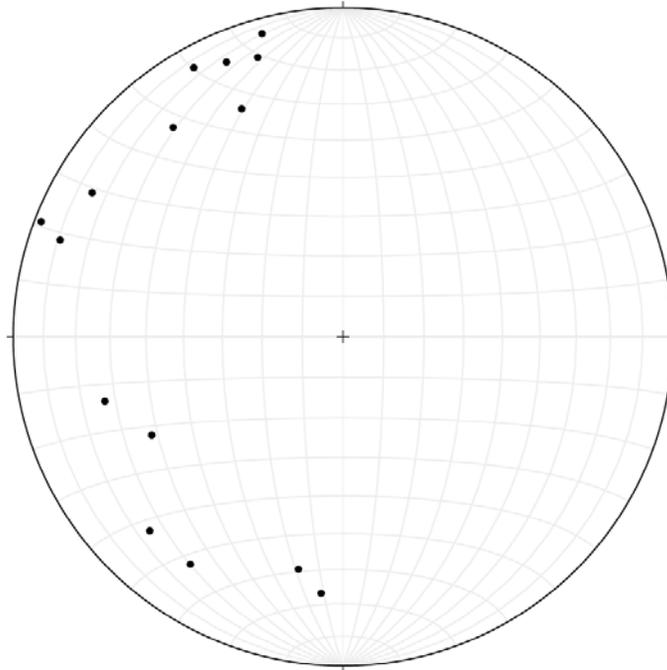
Figure 7-8: Stereonet Plot of Poles to all Faults Measured in Trenches

Previous correlations and names of the faults on the property are not consistent. The following is the Dave Webb's interpretation based on his observations.

The Bisbee Fault is interpreted to occur immediately west of the property, separating bedded Cretaceous sedimentary sequences including limestone, limey argillite from the Tertiary volcanic rocks described above. The Bisbee Fault is interpreted to be a north-striking thrust fault dipping shallowly to the east.

A fault was observed in outcrop on the southern flank of Mexican Hat Hill, corresponding to a magnetic lineament noted by Game (2013). Grey (1990) correlates this structure to a northwest striking portion of the Victoria Fault, whereas Game (2013) shows a different location for the Victoria Fault, and no structure corresponding to this geophysical lineament. This fault, called the south magnetic fault or SMAG Fault strikes 296o, dips steeply to the northeast and truncates geophysical lineaments correlating to the Victoria Fault, with an apparent sinistral offset of 400 m of this fault. The eastern exposure of this fault separates rhyolite from latite in a road cut immediately north of the Victoria Shaft.

The Victoria Fault is exposed on the east flank of Mexican Hat Hill and strikes northerly and dips moderately to shallowly to the east according to Gray (1990) as does Game (2013), however Gray (1990) shows a section with a west-dipping Victoria Fault.

The West Fault is exposed on the western flank of Mexican Hat Hill, striking northerly and dipping moderately to shallowly to the east and it apparently cross-cuts the SMAG Faults

Numerous minor structures were observed during the detailed mapping of the trenches during the 2014 work.

8 DEPOSIT TYPES

Porphyry-style mineralization including skarn-type end members containing economically recoverable copper, gold, and sometimes lead, zinc, silver and molybdenum occur in this part of Arizona. The Courtland Gleeson district extends up to and may include the Mexican Hat Property. This mineralization is primarily hosted within Mesozoic sediments and younger intrusions. Placer gold deposits occur in places in washes near the base of the Turquois Mountains.

Younger mineralization hosted in rocks including Tertiary volcanic rocks occur in the area, and at Mexican Hat. The association of alkaline to subalkaline volcanic rocks and the presence of low sulphide concentrations together with the geochemistry of these rocks indicates that the Mexican Hat Property is a low sulphidation epithermal gold deposit. A selected list of Tertiary low-sulphidation deposits in Nevada are listed Table 8-1.

Table 8-1: Selected List of Tertiary Low-sulphidation gold Deposits from Nevada from Cuffney, 2008

Deposit	Au oz/t	Au k oz	Age Ma	Mineralization Style	Alteration	Ore Mineralogy (primary)	Gangue Mineralogy	Host rocks
Rosebud	0.452	538	14.7	stockwork & disseminated	argillic propylitic, minor silica	electrum, silver sulfides, selenides, sulfosalts	illite, quartz, calcite, barite, adularia	Miocene-Oligocene rhyolites and volcaniclastics
Hog Ranch	0.036	306	15.2	disseminated, veins, breccias	argillic qtz-adularia	native gold	quartz, adularia, pyrite, marcasite, realgar, stibnite	Miocene rhyolites. lacustrine sediments
Hollister	1.38	827	15.1	disseminated banded veins	argillic silicification	electrum, silver sulfides, selenides	quartz, clay, adularia	Miocene volcanics, Ordovician argillites
Hycroft	0.015	2000	3.9	disseminated, breccias	opal, late acid sulfate	native gold	chalcedony, pyrite, marcasite	Pliocene conglomerate, & volcanics
Midas	0.630	2400	15.3	banded vein	quartz-adularia, argillic	Au, electrum, silver selenides	quartz-adularia,	Miocene volcanics
Mule Cnyn	0.112	1433	15.6	stockwork veins, breccia	argillic silicification	electrum, silver sulfides, selenides	quartz, clay, pyrite	Miocene volcanics
Rawhide	0.027	1625	15.7	stockwork & disseminated	potassic, argillic, propylitic	electrum, silver sulfides, selenides, sulfosalts	quartz, adularia, pyrite, illite	Miocene volcanics
Round Mt	0.019	>10000	25.9	stockwork & disseminated	potassic, argillic, propylitic	electrum, silver sulfides, selenides, sulfosalts	quartz, adularia, pyrite, illite	Oligocene volcanics Ordovician argillites
Sleeper	0.030	1680	16.1	stockwork & banded vein	silicification potassic	electrum	quartz-pyrite, adularia, calcite	Miocene volcanics

Note: Ounces of gold reported are Ore Reserves, Measured and Indicated plus production as reported by the Nevada Bureau of Mines.

9   EXPLORATION

9.1   WORK COMPLETED IN 2014

9.1.1   Trench Mapping

During the 2014 exploration program selected (TR 1, TR 2 and TR 4) continuous chip sample trenches were geologically mapped at one to 100 scale (1:100). Results of geological mapping determined that there are two types of structural controls on mineralization observed within Tertiary volcanic rocks on the Mexican Hat property.

1. Brittle Fault controlled mineralization observed consisting mainly of hematite, +/- limonite. Narrow zones of weak breccias believed to be in part hydrothermally induced breccias, and weak to moderate silicification adjacent to some fault envelopes were recognized.

2. Fracture controlled mineralization consisting of hematite and limonite. Mineralized fractures generally trend in a dominant NE/SW direction. Not all fractures are mineralized.

The most common rock unit identified within the continuous rock chip trenches is latite. Latite is described as weakly to strongly faulted and fractured, beige/grey/whitish, medium to coarse grained, porphyritic volcanic. The latite has no visible quartz phenocrysts and contains approximately 80% feldspar phenocrysts with 15-20% mafic minerals (biotite, hornblende and pyroxene). Minor (1-2%) sub-rounded lithic lapilli fragments of argillite and latite composition are common. Some latite outcrops have agglomerate size fragments but are less common on the property. Quartz latite was observed in the eastern end of continuous chip trench TR 15. The quartz latite unit contained up to approximately 7% quartz phenocrysts but this unit is rare on the property and the contact between quartz latite and latite is gradational. Other rock units within the Mexican Hat property trenches include; trachyte basalt, a fine grained light green, grey to maroon color volcanic with mafic phenocryst (pyroxene, hornblende). Second; trachyte andesite, a fine grained, green to maroon color rock with feldspar and mafic phenocrysts (pyroxene, hornblende). Third; arkosic sandstone is described as medium grained, reddish to beige sedimentary rock (possibly a fine-grained volcanic rock) and locally mixed with trachyte basalt in TR 14. Fourth; a rhyolite tuff to tuff breccia that appears similar to a pebble conglomerate found in trench TR 7. Fifth; an argillite unit associated with the rhyolite and is described as fine grained, grey to black sedimentary rock.

A total of 16 separate continuous chip samples were collected in 2015, some including breaks where there was no exposure, shown on the Figure 9-1 below.
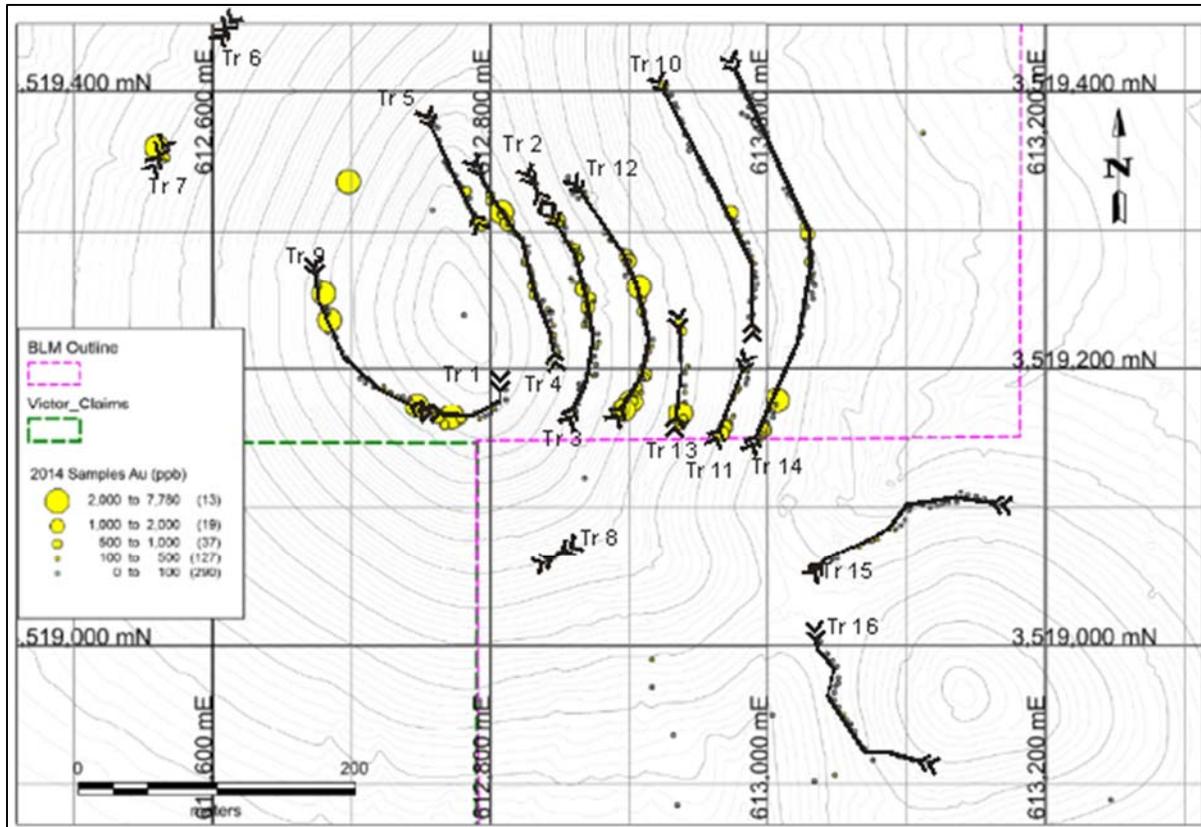
Figure 9-1: Location of Chip Sampling Program with Gold Values from 2014 Trench Program

9.1.2 Sampling

9.1.2.1 Sample Collection

A total of 567 samples have been collected from trenches and diamond drill core re-sampling for geochemical analysis using Bureau Veritas' AQ252 process with all samples reporting greater than 0.1 g/t gold subjected to fire assay using Bureau Veritas' FA430 process.

<u>Historical Core Resampling</u>

The drill core resampling was completed over two weeks in December 2014. Auracle drillholes that are proximal to Placer Dome drillholes were selected to a) verify the Auracle assays and b) by proximity, verify the Placer Dome Inc. drillholes. There was no material from Placer Dome's work to verify directly.

The drillholes were relogged and resampled and compared to the Auracle data.

The samples were bagged, tagged, and securely stored for up to seven days prior to shipping by commercial shippers to Inspectorate Laboratories in Elko Nevada.

The analytical gold assay results of the resampling program are compared with the historical gold assay results in Figure 9-2 through Figure 9-4.

The Placer Dome holes are within 30 meters and parallel to the Auracle drillhole and yields similar but not identical assays.

Table 9-1: Location of BTM-11-09 Showing Proximity to Other Drillholes

Hole_ID	td__m_	Azimuth	Dip	East_m	North_m	Elev_m	Drill_Type
BTM 11-09	155.8	360	-60	613240	3519187	1430	CORE
MH 11-9	152.4	360	-60	613240	3519187	1430	CORE
MH-89-41	138.684	0	-60	613255.9	3519158	1431.676	ROTARY
MH-89-79	103.632	0	-60	613253.5	3519158	1432.191	ROTARY
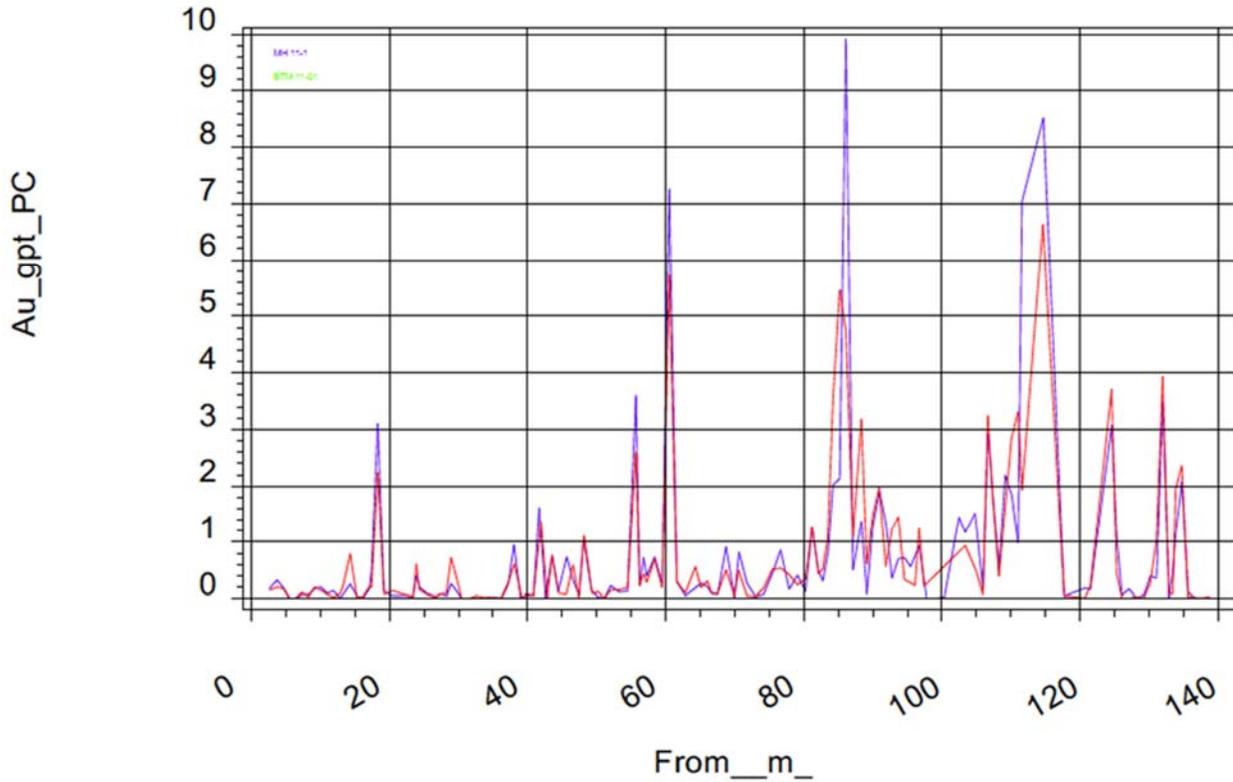
Figure 9-2: Line Plot of Gold Values in MH 11-1 (blue) and Re-assay Data as BTM 11-01 (red)
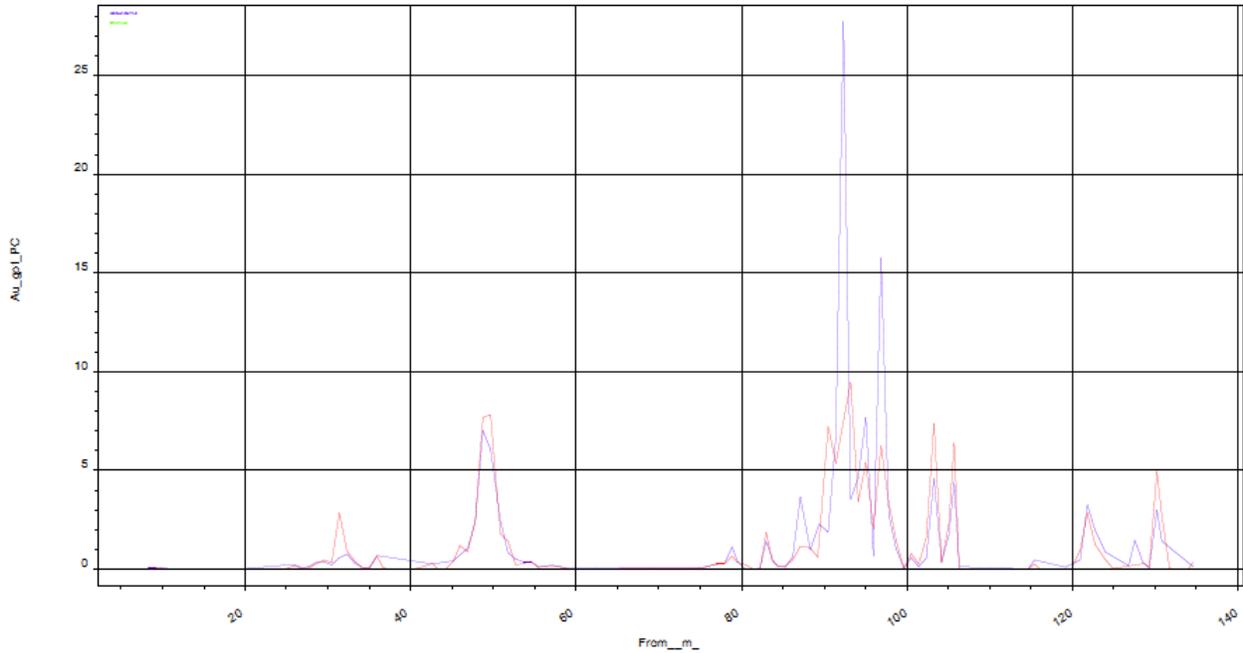
Figure 9-3: Line Plot of Gold Values in MH 11-2 (blue) and Re-assay Data as BTM 11-02 (red)
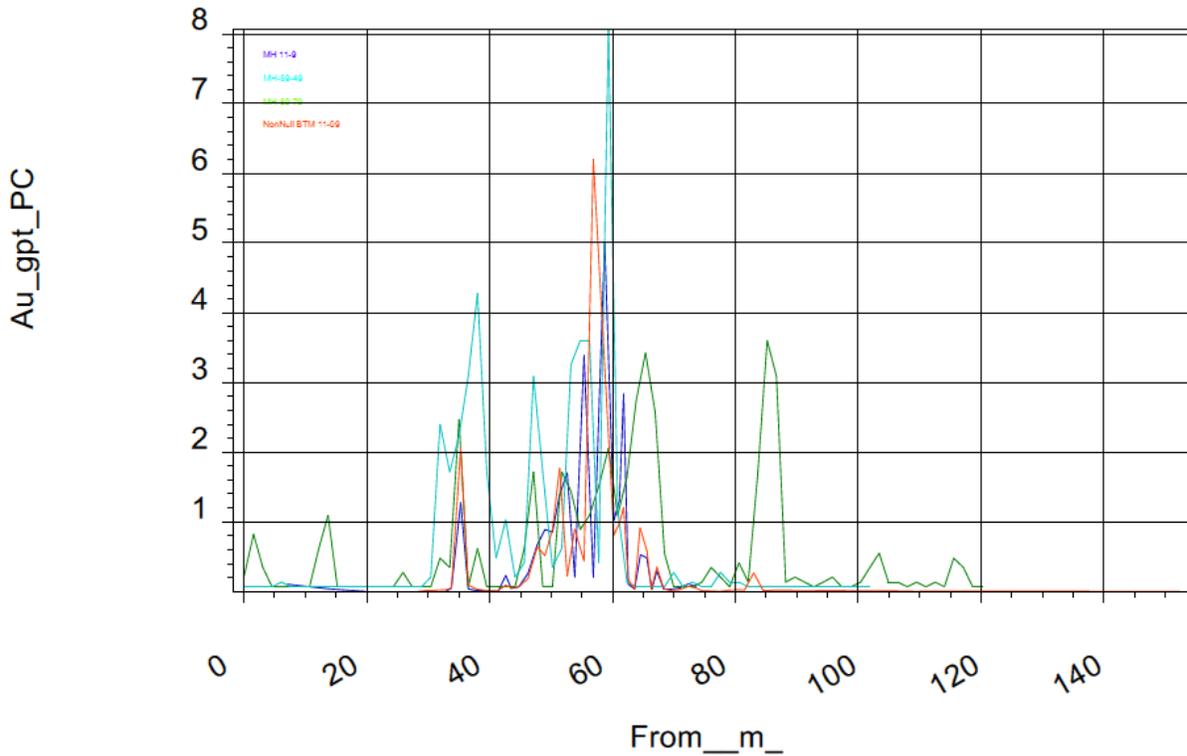
Figure 9-4: Line Plot of Gold Values in MH 11-9 (blue (dark)), a Twin of MH 89-79 (light blue) and Nearby MH 89-41 (green) and Re-assay Data as BTM-09 (red)

Trench Sample Collection

All trench samples were collected using a mechanical hammer (Bosch Bulldog) in near continuous samples along road cuts and/or mechanically excavated road cuts using a Caterpillar 420 backhoe. Additional regional grabs samples were collected and included into the sample stream (Photographs 8-1 and 8-2).

Photograph 9-1: Near continuous chip sample were collected in road cuts (Webb, 2014)

Photograph 9-2: A backhoe was used to open or clean up road cuts prior to sampling (Webb, 2014)

9.1.2.2 Lithogeochemistry

A total of 677 samples exclusive of duplicates and standards were collected and together with repeats and duplicates samples from the laboratory, were analyzed by geochemical methods. Statistics are summarized in Table 9-2.

Table 9-2: Univariate Statistics for Geochemistry of all Surface Samples Collected in 2014, Above Detection Limits (D.L.)

Field	Units	Count >D.L.	Min	Max	Range	Mean	Median	Mode	Variance	Standard Deviation
Au g/t	PPM	321	0.045	9.476	9.431	1.093	0.44	0.119	2.477	1.574
Mo	PPM	677	0.07	73.39	73.32	1.873	0.64	0.19	21.860	4.675
Cu	PPM	676	2.91	358.17	355.26	38.185	27.245	6.27	1801.808	42.448
Pb	PPM	677	0.93	40.34	39.41	8.240	7.94	4.84	13.516	3.676
Zn	PPM	677	3.8	144.7	140.9	69.072	67.6	68.2	149.317	12.220
Ag	PPB	677	5	6888	6883	467.112	276	20	393153	627.020
Ni	PPM	677	1.6	74.1	72.5	22.605	17.2	16.2	166.188	12.891
Co	PPM	677	0.4	30.2	29.8	13.621	12.4	11.8	15.797	3.975
Mn	PPM	677	20	1633	1613	640.285	627	571	32506	180.296
Fe	%	677	0.87	4.77	3.9	2.874	2.83	2.88	0.315	0.561
As	PPM	677	0.4	606.6	606.2	57.347	30.4	2.3	5593.678	74.791
U	PPM	677	0.35	2.67	2.32	0.989	0.91	0.71	0.154	0.392
Au	PPB	674	0.4	11127.9	11127.5	539.161	86.7	5.2	1602215	1265.787
Th	PPM	677	0.9	8.1	7.2	4.133	4.8	1.3	4.036	2.009
Sr	PPM	677	13.7	225.7	212	43.309	35.6	26.5	711.984	26.683
Cd	PPM	655	0.01	0.81	0.8	0.078	0.06	0.06	0.004	0.064
Sb	PPM	676	0.04	154.9	154.86	4.242	1.06	0.19	185.349	13.614
Bi	PPM	511	0.02	2.34	2.32	0.089	0.05	0.03	0.032	0.179
V	PPM	677	8	147	139	63.421	60	56	381.351	19.528
Ca	%	677	0.11	3.9	3.79	1.142	0.88	0.37	0.624	0.790
P	%	677	0.021	0.285	0.264	0.115	0.107	0.099	0.001	0.025
La	PPM	677	6.9	51.3	44.4	27.258	27.6	28.2	15.462	3.932
Cr	PPM	677	17.7	114.4	96.7	47.813	44.4	39.9	305.552	17.480
Mg	%	677	0.03	2.87	2.84	1.315	1.25	1.07	0.188	0.434
Ba	PPM	677	14.7	897.7	883	50.851	47.4	49.2	1592.095	39.901
Ti	%	675	0.002	0.231	0.229	0.079	0.069	0.004	0.003	0.054
B	PPM	660	1	11	10	3.759	3	3	2.438	1.561
Al	%	677	0.47	2.67	2.2	1.509	1.5	1.49	0.117	0.342
Na	%	677	0.004	0.188	0.184	0.049	0.045	0.044	0.000	0.017
K	%	677	0.03	0.43	0.4	0.121	0.11	0.09	0.003	0.054
W	PPM	675	0.06	30.14	30.08	1.954	0.73	0.3	10.784	3.284
Sc	PPM	677	0.7	12.9	12.2	4.844	4.7	4.7	2.192	1.480
Tl	PPM	609	0.02	0.86	0.84	0.097	0.08	0.06	0.004	0.060
S	%	64	0.02	0.59	0.57	0.065	0.04	0.03	0.008	0.089
Hg	PPB	503	5	1100	1095	24.732	15	11	3189.033	56.472
Se	PPM	213	0.1	10.7	10.6	0.277	0.2	0.2	0.535	0.731
Te	PPM	123	0.02	0.23	0.21	0.051	0.04	0.03	0.001	0.032
Ga	PPM	677	1.2	19.3	18.1	9.898	9.9	10	4.328	2.080

The different Numbers of pairs for Pearson correlation coefficient determination requires different values for significance at 99.9% confidence (strong) and 99% confidence (Table 9-3 through Table 9-7). For gold as measured by fire assay show have strong positive correlations with Ag, Fe, As, Au (by AQ252), Sr, V, Cr, B and modest positive

correlations with Mo, Pb, Zn, Ba, W, and S. Gold shows strong negative correlations with Th, Ti, Al, and modest negative correlations with La, Mg and Sc.

Table 9-3: Pearson Correlation Coefficients for all Surface Samples Collected in 2014, Part 1. Significant Correlations are High-lighted in yellow (95% CI) or gold (99% CI) for First Six Elements

Field	n	Au g/t	Mo	Cu	Pb	Zn	Ag	Ni	Co
Au g/t	321	1	0.1751	-0.0082	0.1705	0.1755	0.4846	0.0387	-0.0785
Mo	677	0.1751	1	-0.0021	0.2204	-0.0220	0.5322	-0.1377	-0.1590
Cu	676	-0.0082	-0.0021	1	0.1774	0.1893	0.0935	0.0166	0.0510
Pb	677	0.1705	0.2204	0.1774	1	0.1380	0.3066	-0.1238	-0.1990
Zn	677	0.1755	-0.0220	0.1893	0.1380	1	0.1793	0.2720	0.4883
Ag	677	0.4846	0.5322	0.0935	0.3066	0.1793	1	-0.1753	-0.1552
Ni	677	0.0387	-0.1377	0.0166	-0.1238	0.2720	-0.1753	1	0.7919
Co	677	-0.0785	-0.1590	0.0510	-0.1990	0.4883	-0.1552	0.7919	1
Mn	677	-0.1267	-0.0751	-0.0962	-0.0936	0.2998	-0.1287	0.3017	0.5227
Fe	677	0.3530	0.0287	0.1016	-0.1576	0.4173	0.1467	0.4933	0.6802
As	677	0.3415	0.2288	0.0397	0.2000	0.1305	0.3800	-0.1701	-0.1761
U	677	-0.1417	0.0667	-0.0694	0.3216	-0.2406	0.0122	-0.3096	-0.3739
Au	674	0.9330	0.2627	0.0597	0.2258	0.1862	0.5940	-0.0889	-0.1323
Th	677	-0.2927	0.0799	-0.0778	0.3319	-0.3508	-0.0300	-0.4997	-0.6295
Sr	677	0.1902	-0.0739	0.0190	-0.1074	-0.0239	-0.0823	0.3450	0.3029
Cd	655	-0.0898	0.0230	0.0452	0.3124	0.2364	0.1490	-0.1077	-0.0817
Sb	676	0.1083	-0.0230	0.5087	0.1502	0.2110	0.1136	0.0095	0.0686
Bi	511	0.0048	-0.0064	0.2767	0.3309	0.2801	0.0654	0.0985	0.1721
V	677	0.2763	-0.0062	0.1655	0.0468	0.4176	0.1213	0.6940	0.7000
Ca	677	-0.1301	-0.2056	-0.0118	-0.2098	-0.0835	-0.3545	0.4008	0.3919
P	677	0.0512	-0.1259	-0.0772	-0.2670	0.3222	-0.1134	0.4096	0.5924
La	677	-0.1644	-0.0308	-0.1310	0.1146	0.2175	-0.0346	0.0111	0.0852
Cr	677	0.2253	-0.0460	-0.0681	-0.0579	0.0585	-0.0321	0.7524	0.4226
Mg	677	-0.1436	-0.2312	-0.0725	-0.3395	0.3243	-0.2755	0.8015	0.8291
Ba	677	0.1590	-0.0365	0.0060	0.1643	-0.0178	-0.0258	0.0112	-0.0329
Ti	675	-0.2678	-0.2411	-0.1455	-0.0367	-0.1294	-0.3919	0.4434	0.3079
B	660	0.2356	0.2070	0.1245	0.3040	0.0341	0.2846	-0.1202	-0.1755
Al	677	-0.2061	-0.1700	-0.0019	-0.2602	0.1901	-0.2408	0.5365	0.6637
Na	677	0.1090	-0.0487	-0.0656	-0.0371	0.1083	-0.0383	0.1697	0.2300
K	677	-0.1300	0.0675	0.0363	0.3794	-0.2112	0.1104	-0.5508	-0.6116
W	675	0.1619	0.0943	0.4067	0.2241	0.2085	0.2864	-0.0422	0.0138
Sc	677	-0.1541	-0.2193	-0.1089	-0.0899	0.1086	-0.2991	0.6976	0.6039
Tl	609	0.0666	0.1838	0.1140	0.3271	0.0408	0.1885	-0.1240	-0.0831
S	64	0.3524	0.0647	-0.1199	-0.1586	0.1582	0.2010	0.0046	0.0464
Hg	503	0.0723	-0.0024	0.1773	0.0679	0.1644	0.2154	0.0515	0.1583
Se	213	-0.1660	-0.0003	-0.0684	-0.0192	-0.3754	0.0066	-0.1290	-0.2563
Te	123	0.1579	-0.0686	0.0405	0.0658	-0.0078	0.0392	-0.0759	-0.0946
Ga	677	0.1196	0.0212	0.1882	0.0126	0.5589	0.1686	0.2688	0.4297

Table 9-4: Pearson correlation coefficients for all surface samples collected in 2014, Part 2

Field	n	Mn	Fe	As	U	Au	Th	Sr	Cd
Au g/t	321	-0.1267	0.3530	0.3415	-0.1417	0.9330	-0.2927	0.1902	-0.0898
Mo	677	-0.0751	0.0287	0.2288	0.0667	0.2627	0.0799	-0.0739	0.0230
Cu	676	-0.0962	0.1016	0.0397	-0.0694	0.0597	-0.0778	0.0190	0.0452
Pb	677	-0.0936	-0.1576	0.2000	0.3216	0.2258	0.3319	-0.1074	0.3124
Zn	677	0.2998	0.4173	0.1305	-0.2406	0.1862	-0.3508	-0.0239	0.2364
Ag	677	-0.1287	0.1467	0.3800	0.0122	0.5940	-0.0300	-0.0823	0.1490
Ni	677	0.3017	0.4933	-0.1701	-0.3096	-0.0889	-0.4997	0.3450	-0.1077
Co	677	0.5227	0.6802	-0.1761	-0.3739	-0.1323	-0.6295	0.3029	-0.0817
Mn	677	1	0.3473	-0.1605	-0.1336	-0.1481	-0.2274	0.0641	0.1250
Fe	677	0.3473	1	0.2698	-0.3975	0.2891	-0.7084	0.3379	-0.1405
As	677	-0.1605	0.2698	1	-0.0658	0.4850	-0.0752	0.2589	0.0748
U	677	-0.1336	-0.3975	-0.0658	1	-0.0831	0.5748	-0.1163	0.1496
Au	674	-0.1481	0.2891	0.4850	-0.0831	1	-0.1410	0.0852	-0.0319
Th	677	-0.2274	-0.7084	-0.0752	0.5748	-0.1410	1	-0.4131	0.1159
Sr	677	0.0641	0.3379	0.2589	-0.1163	0.0852	-0.4131	1	-0.0878
Cd	655	0.1250	-0.1405	0.0748	0.1496	-0.0319	0.1159	-0.0878	1
Sb	676	-0.2116	0.0901	0.0339	-0.1293	0.1456	-0.2034	0.0367	-0.0161
Bi	511	0.0189	0.0644	-0.0904	-0.0942	0.0040	-0.1929	-0.0328	0.0918
V	677	0.2108	0.7530	0.0542	-0.2177	0.1747	-0.5418	0.2892	-0.0489
Ca	677	0.3559	0.1382	-0.4188	0.0369	-0.2808	-0.2422	0.3424	0.0116
P	677	0.3599	0.5706	0.0102	-0.3339	-0.0628	-0.5932	0.2836	0.0355
La	677	0.2205	-0.0038	0.0270	-0.1288	-0.0917	0.1273	-0.2146	0.2659
Cr	677	0.0159	0.2723	-0.0370	-0.1995	0.0247	-0.2851	0.2771	-0.1221
Mg	677	0.5199	0.5081	-0.3362	-0.3241	-0.2158	-0.5430	0.2298	-0.1192
Ba	677	-0.0308	-0.0484	0.0805	0.1673	0.1338	0.0794	0.0831	0.0624
Ti	675	0.1808	-0.0628	-0.4408	0.3648	-0.3423	0.0569	0.2156	0.0204
B	660	-0.2442	0.0884	0.3084	-0.0219	0.2886	0.0471	-0.0806	0.1070
Al	677	0.5720	0.5738	-0.2259	-0.1944	-0.1913	-0.3445	0.1833	-0.1586
Na	677	0.0744	0.2223	0.0523	-0.0300	0.0237	-0.1719	0.3841	-0.0113
K	677	-0.3332	-0.5548	0.2056	0.4381	-0.0009	0.6139	-0.1881	0.2728
W	675	-0.2142	0.1245	0.0645	-0.1451	0.2056	-0.1355	-0.1012	0.0689
Sc	677	0.3895	0.3025	-0.3466	0.0712	-0.2377	-0.2560	0.2761	-0.0271
Tl	609	0.0354	-0.0013	0.2618	0.0580	0.1350	0.1293	0.1215	0.1625
S	64	-0.1037	0.3904	0.4293	-0.2116	0.3728	-0.3556	0.6345	-0.1003
Hg	503	0.0377	0.2061	0.1991	-0.0107	0.1223	-0.1591	0.0950	0.4413
Se	213	-0.2495	-0.1490	-0.0022	0.3149	-0.0396	0.0313	0.4091	-0.0743
Te	123	-0.1888	-0.0174	-0.0201	-0.0719	0.2241	-0.0019	-0.0719	-0.0417
Ga	677	0.2641	0.5489	0.1907	-0.2855	0.1701	-0.3219	-0.0204	-0.0216

Table 9-5: Pearson Correlation Coefficients for all Surface Samples Collected in 2014, Part 3

Field	n	Bi	V	Ca	P	La	Cr	Mg	Ba
Au g/t	321	0.0048	0.2763	-0.1301	0.0512	-0.1644	0.2253	-0.1436	0.1590
Mo	677	-0.0064	-0.0062	-0.2056	-0.1259	-0.0308	-0.0460	-0.2312	-0.0365
Cu	676	0.2767	0.1655	-0.0118	-0.0772	-0.1310	-0.0681	-0.0725	0.0060
Pb	677	0.3309	0.0468	-0.2098	-0.2670	0.1146	-0.0579	-0.3395	0.1643
Zn	677	0.2801	0.4176	-0.0835	0.3222	0.2175	0.0585	0.3243	-0.0178
Ag	677	0.0654	0.1213	-0.3545	-0.1134	-0.0346	-0.0321	-0.2755	-0.0258
Ni	677	0.0985	0.6940	0.4008	0.4096	0.0111	0.7524	0.8015	0.0112
Co	677	0.1721	0.7000	0.3919	0.5924	0.0852	0.4226	0.8291	-0.0329
Mn	677	0.0189	0.2108	0.3559	0.3599	0.2205	0.0159	0.5199	-0.0308
Fe	677	0.0644	0.7530	0.1382	0.5706	-0.0038	0.2723	0.5081	-0.0484
As	677	-0.0904	0.0542	-0.4188	0.0102	0.0270	-0.0370	-0.3362	0.0805
U	677	-0.0942	-0.2177	0.0369	-0.3339	-0.1288	-0.1995	-0.3241	0.1673
Au	674	0.0040	0.1747	-0.2808	-0.0628	-0.0917	0.0247	-0.2158	0.1338
Th	677	-0.1929	-0.5418	-0.2422	-0.5932	0.1273	-0.2851	-0.5430	0.0794
Sr	677	-0.0328	0.2892	0.3424	0.2836	-0.2146	0.2771	0.2298	0.0831
Cd	655	0.0918	-0.0489	0.0116	0.0355	0.2659	-0.1221	-0.1192	0.0624
Sb	676	0.1307	0.2162	-0.0269	-0.0296	-0.1491	-0.0654	-0.0667	0.0220
Bi	511	1	0.1119	-0.0842	-0.0467	-0.0644	0.0129	0.0333	0.0157
V	677	0.1119	1	0.2316	0.4030	-0.0314	0.4726	0.5874	0.0521
Ca	677	-0.0842	0.2316	1	0.3385	-0.1482	0.1476	0.4841	0.0089
P	677	-0.0467	0.4030	0.3385	1	0.2576	0.1871	0.5218	-0.0051
La	677	-0.0644	-0.0314	-0.1482	0.2576	1	0.0122	0.0490	0.0275
Cr	677	0.0129	0.4726	0.1476	0.1871	0.0122	1	0.4410	0.0480
Mg	677	0.0333	0.5874	0.4841	0.5218	0.0490	0.4410	1	-0.0812
Ba	677	0.0157	0.0521	0.0089	-0.0051	0.0275	0.0480	-0.0812	1
Ti	675	-0.1931	0.1971	0.5995	0.2529	-0.1067	0.3470	0.4063	0.1379
B	660	0.0593	0.0729	-0.2457	-0.0762	0.0859	-0.0292	-0.3146	0.0587
Al	677	-0.0143	0.4564	0.4098	0.3925	-0.0539	0.2094	0.7722	-0.0701
Na	677	-0.1183	0.1907	0.2008	0.3250	-0.0211	0.1484	0.1631	0.0775
K	677	0.0070	-0.4970	-0.4306	-0.3860	0.2183	-0.2233	-0.6684	0.2197
W	675	0.1187	0.2460	-0.1732	-0.0213	0.0457	-0.0760	-0.1036	-0.0225
Sc	677	-0.0325	0.4802	0.5713	0.4574	0.0291	0.4663	0.6974	0.0608
Tl	609	0.0697	-0.0240	-0.3416	-0.0645	0.2629	0.0844	-0.2585	0.2479
S	64	-0.0792	0.2584	-0.2970	0.2349	-0.2865	0.1164	-0.0918	-0.0423
Hg	503	0.1441	0.2671	-0.0842	0.0128	-0.0500	-0.0076	0.0434	0.1764
Se	213	-0.0261	-0.1755	-0.1051	-0.2650	-0.3461	0.1147	-0.2156	0.0779
Te	123	-0.0523	-0.0065	-0.0899	-0.1847	0.0083	-0.0540	-0.1755	-0.0070
Ga	677	0.2056	0.5003	-0.1333	0.1970	0.1720	0.0037	0.4314	-0.1161

Table 9-6: Pearson Correlation Coefficients for all Surface Samples Collected in 2014, Part 4

Field	n	Ti	B	Al	Na	K	W	Sc	Tl
Au g/t	321	-0.2678	0.2356	-0.2061	0.1090	-0.1300	0.1619	-0.1541	0.0666
Mo	677	-0.2411	0.2070	-0.1700	-0.0487	0.0675	0.0943	-0.2193	0.1838
Cu	676	-0.1455	0.1245	-0.0019	-0.0656	0.0363	0.4067	-0.1089	0.1140
Pb	677	-0.0367	0.3040	-0.2602	-0.0371	0.3794	0.2241	-0.0899	0.3271
Zn	677	-0.1294	0.0341	0.1901	0.1083	-0.2112	0.2085	0.1086	0.0408
Ag	677	-0.3919	0.2846	-0.2408	-0.0383	0.1104	0.2864	-0.2991	0.1885
Ni	677	0.4434	-0.1202	0.5365	0.1697	-0.5508	-0.0422	0.6976	-0.1240
Co	677	0.3079	-0.1755	0.6637	0.2300	-0.6116	0.0138	0.6039	-0.0831
Mn	677	0.1808	-0.2442	0.5720	0.0744	-0.3332	-0.2142	0.3895	0.0354
Fe	677	-0.0628	0.0884	0.5738	0.2223	-0.5548	0.1245	0.3025	-0.0013
As	677	-0.4408	0.3084	-0.2259	0.0523	0.2056	0.0645	-0.3466	0.2618
U	677	0.3648	-0.0219	-0.1944	-0.0300	0.4381	-0.1451	0.0712	0.0580
Au	674	-0.3423	0.2886	-0.1913	0.0237	-0.0009	0.2056	-0.2377	0.1350
Th	677	0.0569	0.0471	-0.3445	-0.1719	0.6139	-0.1355	-0.2560	0.1293
Sr	677	0.2156	-0.0806	0.1833	0.3841	-0.1881	-0.1012	0.2761	0.1215
Cd	655	0.0204	0.1070	-0.1586	-0.0113	0.2728	0.0689	-0.0271	0.1625
Sb	676	-0.2092	0.1946	-0.2000	-0.1062	-0.0227	0.7796	-0.1347	0.0338
Bi	511	-0.1931	0.0593	-0.0143	-0.1183	0.0070	0.1187	-0.0325	0.0697
V	677	0.1971	0.0729	0.4564	0.1907	-0.4970	0.2460	0.4802	-0.0240
Ca	677	0.5995	-0.2457	0.4098	0.2008	-0.4306	-0.1732	0.5713	-0.3416
P	677	0.2529	-0.0762	0.3925	0.3250	-0.3860	-0.0213	0.4574	-0.0645
La	677	-0.1067	0.0859	-0.0539	-0.0211	0.2183	0.0457	0.0291	0.2629
Cr	677	0.3470	-0.0292	0.2094	0.1484	-0.2233	-0.0760	0.4663	0.0844
Mg	677	0.4063	-0.3146	0.7722	0.1631	-0.6684	-0.1036	0.6974	-0.2585
Ba	677	0.1379	0.0587	-0.0701	0.0775	0.2197	-0.0225	0.0608	0.2479
Ti	675	1	-0.3083	0.2921	0.3081	-0.1515	-0.3349	0.7786	-0.2371
B	660	-0.3083	1	-0.3052	-0.0415	0.2220	0.3305	-0.2412	0.2171
Al	677	0.2921	-0.3052	1	0.1805	-0.5056	-0.2171	0.5163	-0.1464
Na	677	0.3081	-0.0415	0.1805	1	-0.0918	-0.1711	0.3245	-0.0981
K	677	-0.1515	0.2220	-0.5056	-0.0918	1	0.0622	-0.4071	0.5494
W	675	-0.3349	0.3305	-0.2171	-0.1711	0.0622	1	-0.2113	0.1691
Sc	677	0.7786	-0.2412	0.5163	0.3245	-0.4071	-0.2113	1	-0.2829
Tl	609	-0.2371	0.2171	-0.1464	-0.0981	0.5494	0.1691	-0.2829	1
S	64	-0.2490	-0.0630	0.0315	0.8295	0.0803	-0.1541	-0.0474	-0.0450
Hg	503	-0.1535	0.0364	0.0570	-0.0226	0.0042	0.1752	-0.0825	0.0547
Se	213	0.0647	-0.0821	-0.2306	-0.2072	0.1314	-0.0630	-0.2244	0.2776
Te	123	-0.1552	0.0436	-0.2286	-0.1467	-0.0006	0.1426	-0.1545	0.0142
Ga	677	-0.3492	0.0526	0.4310	-0.0264	-0.2966	0.3177	0.0709	0.0509

Table 9-7: Pearson Correlation Coefficients for all Surface Samples Collected in 2014, Part 5

Field	n	S	Hg	Se	Te	Ga
Au g/t	321	0.3524	0.0723	-0.1660	0.1579	0.1196
Mo	677	0.0647	-0.0024	-0.0003	-0.0686	0.0212
Cu	676	-0.1199	0.1773	-0.0684	0.0405	0.1882
Pb	677	-0.1586	0.0679	-0.0192	0.0658	0.0126
Zn	677	0.1582	0.1644	-0.3754	-0.0078	0.5589
Ag	677	0.2010	0.2154	0.0066	0.0392	0.1686
Ni	677	0.0046	0.0515	-0.1290	-0.0759	0.2688
Co	677	0.0464	0.1583	-0.2563	-0.0946	0.4297
Mn	677	-0.1037	0.0377	-0.2495	-0.1888	0.2641
Fe	677	0.3904	0.2061	-0.1490	-0.0174	0.5489
As	677	0.4293	0.1991	-0.0022	-0.0201	0.1907
U	677	-0.2116	-0.0107	0.3149	-0.0719	-0.2855
Au	674	0.3728	0.1223	-0.0396	0.2241	0.1701
Th	677	-0.3556	-0.1591	0.0313	-0.0019	-0.3219
Sr	677	0.6345	0.0950	0.4091	-0.0719	-0.0204
Cd	655	-0.1003	0.4413	-0.0743	-0.0417	-0.0216
Sb	676	-0.1031	0.1653	-0.0253	0.1756	0.1970
Bi	511	-0.0792	0.1441	-0.0261	-0.0523	0.2056
V	677	0.2584	0.2671	-0.1755	-0.0065	0.5003
Ca	677	-0.2970	-0.0842	-0.1051	-0.0899	-0.1333
P	677	0.2349	0.0128	-0.2650	-0.1847	0.1970
La	677	-0.2865	-0.0500	-0.3461	0.0083	0.1720
Cr	677	0.1164	-0.0076	0.1147	-0.0540	0.0037
Mg	677	-0.0918	0.0434	-0.2156	-0.1755	0.4314
Ba	677	-0.0423	0.1764	0.0779	-0.0070	-0.1161
Ti	675	-0.2490	-0.1535	0.0647	-0.1552	-0.3492
B	660	-0.0630	0.0364	-0.0821	0.0436	0.0526
Al	677	0.0315	0.0570	-0.2306	-0.2286	0.4310
Na	677	0.8295	-0.0226	-0.2072	-0.1467	-0.0264
K	677	0.0803	0.0042	0.1314	-0.0006	-0.2966
W	675	-0.1541	0.1752	-0.0630	0.1426	0.3177
Sc	677	-0.0474	-0.0825	-0.2244	-0.1545	0.0709
Tl	609	-0.0450	0.0547	0.2776	0.0142	0.0509
S	64	1	0.2845	-0.0638	0.3699	0.2410
Hg	503	0.2845	1	-0.0405	0.3601	0.1431
Se	213	-0.0638	-0.0405	1	0.0374	-0.3223
Te	123	0.3699	0.3601	0.0374	1	-0.0879
Ga	677	0.2410	0.1431	-0.3223	-0.0879	1

Gold values returned 321 samples above detection limits and these had a mean, median and mode of 1.09, 0.44, and 0.119 g/t.

Silver values returned 677 samples above detection limits and these had a mean, median and mode of 0.47, 0.27, and 0.02 g/t.

There were no other significantly elevated base or precious metals detected.

Deleterious elements are present at generally low concentrations, including As, Bi, Sb, Hg, S, Se, and Te. Of these, Hg is perhaps the greatest concern with 503 samples returning values above detection limits with mean, median and mode at 25, 15, and 11 ppb respectively with a high value of 1,100 ppb (Sample 81511) which correlates to a prospecting sample south of the Victoria Shaft in basalt near the underlying rhyolite contact with very elevated copper values.

9.1.2.3   Discussion of Correlations

The positive correlations are to be expected from an alkaline epithermal system whereas the negative correlations all relate to common rock-forming and/or least mobile elements and suggested significant dilution of the host rock by mobile components such as silica and carbonate. This indicates that the hydrothermal system as sampled is likely very large.

9.1.3   Interpretation of 2014 Work

Numerous weakly to moderately siliceous zones are identified adjacent to brittle faults and associated fracture zones. These zones trend dominantly in a northeast-southwest direction and to a lesser extent to the northwest-southeast. The zones are from < 1 cm to approximately 4.0 meters in width, where measured and mapped in detailed (1:100 scale).

Mineralization is in the form of hematite and limonite, commonly with carbonate that fill open spaces in regions where extensional structures were mapped (eastern flanks of Mexican Hat Peak). Malachite and azurite was also identified in trachyte andesites in many locations on the property. Results from initial trenching including 1.074 g/t Au over 25.0 meters (not true width – some sampling along fault fracture planes) and an elevated mineral assemblage (Au, Ag, As, Hg and Sb) from assays confirms the presence of a low sulphidation epithermal system hosted mainly in Tertiary extrusive volcanic (latite) rocks. Higher Au and Ag assays are directly related to zones with moderate to strong hematite filling extensional structures with +/- limonite and weak to moderate silicification. Some faulted and fractured zones were described with weak breccias and weak vuggy quartz which is consistent with epithermal systems.

Auracle twinned a number of Placer Dome drillholes, and DRW re-sampled a number of Auracle drillholes, including two that were proximal to Placer Dome drillholes. MH 11-1, MH 11-2, and MH 11-9 were tested, and assay values confirmed the results reported by Auracle Resources.

The results are of similar magnitude an equivalent location to provide confidence that these drillholes appropriately represent the mineralization that was tested.

9.2   WORK COMPLETED IN 2016 AND 2017

9.2.1   Surficial Geochemistry

Two phases of soil sampling were conducted in 2016 and 2017 covering the area shown on Figure 9-5.
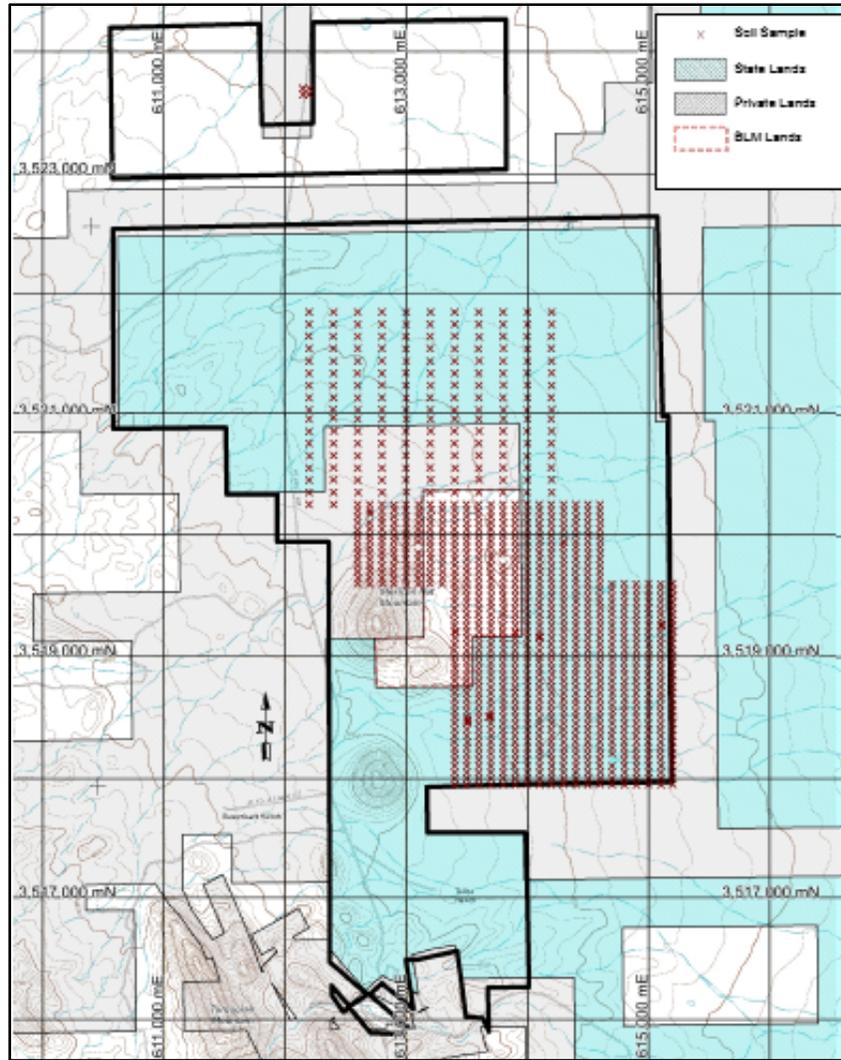
Figure 9-5: Location of 2016 and 2017 Soil Samples

Pearson correlation coefficients on the 1,123 soil samples are shown below on in Table 9-8 to Table 9-11.

Table 9-8: Pearson Correlation Coefficients on N=1123 Soil Samples 1 of 3

Field	Mo	Cu	Pb	Zn	Ag	Ni	Co	Mn	Fe	As	U	Au
Mo	1.0000	0.2904	0.6504	0.3708	0.2951	-0.3435	-0.2021	0.4347	0.1015	-0.3289	0.5769	-0.0371
Cu	0.2904	1.0000	0.5799	0.7231	0.5150	0.3594	0.4211	0.5099	0.5018	0.3623	0.2815	0.0527
Pb	0.6504	0.5799	1.0000	0.6694	0.5398	-0.0427	0.0858	0.6117	0.3647	-0.0817	0.5004	-0.0228
Zn	0.3708	0.7231	0.6694	1.0000	0.6323	0.4204	0.4764	0.7350	0.6000	0.1362	0.4589	0.0437
Ag	0.2951	0.5150	0.5398	0.6323	1.0000	0.2428	0.1874	0.4811	0.3653	0.2571	0.3381	0.2461
Ni	-0.3435	0.3594	-0.0427	0.4204	0.2428	1.0000	0.8621	0.1774	0.6475	0.6051	-0.2877	0.1945
Co	-0.2021	0.4211	0.0858	0.4764	0.1874	0.8621	1.0000	0.4065	0.6838	0.4072	-0.1849	0.1054
Mn	0.4347	0.5099	0.6117	0.7350	0.4811	0.1774	0.4065	1.0000	0.2627	-0.0856	0.4200	-0.0315
Fe	0.1015	0.5018	0.3647	0.6000	0.3653	0.6475	0.6838	0.2627	1.0000	0.2563	0.1445	0.1367
As	-0.3289	0.3623	-0.0817	0.1362	0.2571	0.6051	0.4072	-0.0856	0.2563	1.0000	-0.2766	0.2796
U	0.5769	0.2815	0.5004	0.4589	0.3381	-0.2877	-0.1849	0.4200	0.1445	-0.2766	1.0000	0.0124
Au	-0.0371	0.0527	-0.0228	0.0437	0.2461	0.1945	0.1054	-0.0315	0.1367	0.2796	0.0124	1.0000
Th	0.5719	0.0650	0.3817	0.1906	0.1857	-0.4669	-0.3899	0.1904	0.0002	-0.4050	0.8738	0.0136
Sr	-0.4274	0.1882	-0.1614	0.2372	-0.0199	0.5501	0.4923	-0.0154	0.2819	0.3922	-0.1963	0.0463
Cd	0.4438	0.6736	0.7616	0.7563	0.4580	0.1822	0.3516	0.7298	0.3474	0.0399	0.3734	-0.0187
Sb	-0.3383	0.1739	-0.2603	0.0380	-0.0056	0.5171	0.4461	-0.0481	0.1777	0.7215	-0.2660	0.1559
Bi	0.7274	0.5053	0.8360	0.6191	0.4829	-0.2782	-0.1525	0.5084	0.2522	-0.2768	0.6545	-0.0675
V	-0.2794	0.3268	-0.0702	0.2976	0.0816	0.7892	0.8001	0.0063	0.8189	0.4696	-0.2102	0.1455
Ca	-0.3866	0.0511	-0.2056	0.0746	-0.0738	0.3077	0.1895	-0.1404	0.0099	0.3313	-0.1542	0.0352
P	-0.0044	0.5827	0.2331	0.7189	0.2852	0.6141	0.6431	0.4281	0.4846	0.3666	0.1477	0.0464
La	0.3638	0.4861	0.7091	0.6104	0.5399	0.1180	0.2255	0.6297	0.3985	0.0375	0.4271	0.0231
Cr	-0.3229	0.2674	-0.0947	0.2972	0.2008	0.9337	0.8122	0.0499	0.7007	0.5378	-0.3055	0.2033
Mg	-0.1631	0.4821	0.1115	0.6082	0.3789	0.8284	0.6765	0.2812	0.4941	0.6183	-0.0671	0.1803
Ba	-0.1312	0.2681	0.1708	0.4074	0.2220	0.3301	0.2966	0.3340	0.2243	0.2165	0.0589	-0.0118
Ti	-0.3581	-0.0772	-0.3778	-0.0790	-0.0710	0.4928	0.4397	-0.0773	0.2260	0.3679	-0.2958	0.1303
Al	-0.1709	0.5633	0.3070	0.5798	0.4945	0.7532	0.6087	0.1888	0.6721	0.5953	-0.0938	0.1693
Na	-0.0535	0.1994	-0.0728	0.1032	0.0473	0.2970	0.3344	-0.0331	0.3439	0.1531	-0.1534	0.0265
K	0.2279	0.4982	0.5969	0.6677	0.5507	0.3319	0.2120	0.4062	0.4611	0.1946	0.2298	0.0936
W	0.2821	0.2204	-0.0170	0.1561	0.0229	0.1123	0.1408	0.0836	0.1552	0.1018	0.0734	0.0158
Sc	-0.1842	0.4724	0.1945	0.4596	0.4546	0.8270	0.7554	0.2154	0.7000	0.5945	-0.2006	0.2085
Tl	0.1168	0.2510	0.4682	0.1695	0.4298	0.0446	-0.0423	0.1012	0.2169	0.2305	0.0746	0.0928
S	0.0450	-0.0395	0.0214	0.0040	0.0753	-0.0593	-0.0542	0.0414	-0.0256	-0.0702	0.0359	-0.0133
Hg	0.0178	0.4356	0.3513	0.2343	0.3398	0.2634	0.3410	0.1719	0.2961	0.3467	-0.0341	0.0497
Se	0.0178	0.0299	0.0672	0.0335	0.0731	0.0290	0.0122	0.0323	0.0368	0.0262	-0.0048	-0.0350
Te	0.6492	0.5108	0.7273	0.5954	0.4595	-0.2072	-0.1092	0.4464	0.2241	-0.1684	0.5886	-0.0277
Ga	-0.0621	0.6146	0.3567	0.6967	0.6199	0.7564	0.6131	0.3095	0.6778	0.6211	0.0346	0.2107

Table 9-9: Pearson Correlation Coefficients of N=1,123 Soil Samples 2 of 3

Field	Th	Sr	Cd	Sb	Bi	V	Ca	P	La	Cr	Mg	Ba
Mo	0.5719	-0.4274	0.4438	-0.3383	0.7274	-0.2794	-0.3866	-0.0044	0.3638	-0.3229	-0.1631	-0.1312
Cu	0.0650	0.1882	0.6736	0.1739	0.5053	0.3268	0.0511	0.5827	0.4861	0.2674	0.4821	0.2681
Pb	0.3817	-0.1614	0.7616	-0.2603	0.8360	-0.0702	-0.2056	0.2331	0.7091	-0.0947	0.1115	0.1708
Zn	0.1906	0.2372	0.7563	0.0380	0.6191	0.2976	0.0746	0.7189	0.6104	0.2972	0.6082	0.4074
Ag	0.1857	-0.0199	0.4580	-0.0056	0.4829	0.0816	-0.0738	0.2852	0.5399	0.2008	0.3789	0.2220
Ni	-0.4669	0.5501	0.1822	0.5171	-0.2782	0.7892	0.3077	0.6141	0.1180	0.9337	0.8284	0.3301
Co	-0.3899	0.4923	0.3516	0.4461	-0.1525	0.8001	0.1895	0.6431	0.2255	0.8122	0.6765	0.2966
Mn	0.1904	-0.0154	0.7298	-0.0481	0.5084	0.0063	-0.1404	0.4281	0.6297	0.0499	0.2812	0.3340
Fe	0.0002	0.2819	0.3474	0.1777	0.2522	0.8189	0.0099	0.4846	0.3985	0.7007	0.4941	0.2243
As	-0.4050	0.3922	0.0399	0.7215	-0.2768	0.4696	0.3313	0.3666	0.0375	0.5378	0.6183	0.2165
U	0.8738	-0.1963	0.3734	-0.2660	0.6545	-0.2102	-0.1542	0.1477	0.4271	-0.3055	-0.0671	0.0589
Au	0.0136	0.0463	-0.0187	0.1559	-0.0675	0.1455	0.0352	0.0464	0.0231	0.2033	0.1803	-0.0118
Th	1.0000	-0.3816	0.1629	-0.4012	0.5692	-0.3601	-0.3062	-0.1291	0.2544	-0.4350	-0.2975	-0.1259
Sr	-0.3816	1.0000	0.0811	0.3092	-0.2618	0.5114	0.8437	0.6291	0.0158	0.4429	0.5859	0.6180
Cd	0.1629	0.0811	1.0000	-0.0582	0.6371	0.0874	-0.0330	0.5461	0.5632	0.0652	0.3457	0.2572
Sb	-0.4012	0.3092	-0.0582	1.0000	-0.3928	0.4628	0.2643	0.3275	-0.1476	0.4748	0.4382	0.0765
Bi	0.5692	-0.2618	0.6371	-0.3928	1.0000	-0.2237	-0.2662	0.1511	0.5724	-0.2972	-0.0695	0.0410
V	-0.3601	0.5114	0.0874	0.4628	-0.2237	1.0000	0.2381	0.5085	0.0717	0.8427	0.5816	0.2021
Ca	-0.3062	0.8437	-0.0330	0.2643	-0.2662	0.2381	1.0000	0.4988	-0.1130	0.1975	0.4287	0.5746
P	-0.1291	0.6291	0.5461	0.3275	0.1511	0.5085	0.4988	1.0000	0.2831	0.4482	0.7889	0.4950
La	0.2544	0.0158	0.5632	-0.1476	0.5724	0.0717	-0.1130	0.2831	1.0000	0.0340	0.2241	0.3504
Cr	-0.4350	0.4429	0.0652	0.4748	-0.2972	0.8427	0.1975	0.4482	0.0340	1.0000	0.7033	0.1727
Mg	-0.2975	0.5859	0.3457	0.4382	-0.0695	0.5816	0.4287	0.7889	0.2241	0.7033	1.0000	0.4101
Ba	-0.1259	0.6180	0.2572	0.0765	0.0410	0.2021	0.5746	0.4950	0.3504	0.1727	0.4101	1.0000
Ti	-0.3912	0.1810	-0.1789	0.4955	-0.5210	0.5236	0.0083	0.0863	-0.2086	0.5779	0.3373	-0.1108
Al	-0.2590	0.5074	0.3601	0.2656	0.0809	0.6167	0.3223	0.5969	0.4025	0.6698	0.7481	0.4775
Na	-0.1861	0.3728	-0.0172	0.0617	-0.1217	0.4866	0.1932	0.2470	0.0285	0.3122	0.2658	0.2457
K	0.1116	0.1528	0.4981	-0.0360	0.4274	0.1298	0.1043	0.4111	0.5188	0.1985	0.4243	0.4704
W	-0.0070	0.1082	0.0644	0.1938	0.0095	0.2708	0.0955	0.2576	-0.0755	0.1023	0.2271	0.0730
Sc	-0.3332	0.3303	0.2986	0.3385	-0.0557	0.6922	0.0617	0.4711	0.3132	0.8050	0.6966	0.2585
Tl	0.0826	-0.1007	0.1658	-0.0187	0.3046	-0.0422	-0.0631	-0.1474	0.4684	0.0362	-0.0415	0.1860
S	0.0454	-0.0506	-0.0030	-0.0589	0.0188	-0.0648	-0.0145	-0.0388	0.0310	-0.0580	-0.0423	0.0745
Hg	-0.1024	0.1050	0.3624	0.1496	0.1985	0.2362	-0.0317	0.1800	0.3684	0.2769	0.2280	0.0485
Se	-0.0039	-0.0239	0.0502	-0.0097	0.0163	0.0058	0.0022	0.0437	0.0702	0.0128	0.0582	0.0376
Te	0.4770	-0.1472	0.5898	-0.2664	0.8795	-0.1648	-0.1405	0.2284	0.4739	-0.2341	0.0207	0.0733
Ga	-0.1779	0.4335	0.4308	0.3323	0.1615	0.5918	0.2494	0.6477	0.4547	0.6695	0.8211	0.4223

Table 9-10: Pearson Correlation Coefficients of n=1123 Soil Samples 3 of 3

Field	Ti	Al	Na	K	W	Sc	Tl	S	Hg	Se	Te	Ga
Mo	-0.3581	-0.1709	-0.0535	0.2279	0.2821	-0.1842	0.1168	0.0450	0.0178	0.0178	0.6492	-0.0621
Cu	-0.0772	0.5633	0.1994	0.4982	0.2204	0.4724	0.2510	-0.0395	0.4356	0.0299	0.5108	0.6146
Pb	-0.3778	0.3070	-0.0728	0.5969	-0.0170	0.1945	0.4682	0.0214	0.3513	0.0672	0.7273	0.3567
Zn	-0.0790	0.5798	0.1032	0.6677	0.1561	0.4596	0.1695	0.0040	0.2343	0.0335	0.5954	0.6967
Ag	-0.0710	0.4945	0.0473	0.5507	0.0229	0.4546	0.4298	0.0753	0.3398	0.0731	0.4595	0.6199
Ni	0.4928	0.7532	0.2970	0.3319	0.1123	0.8270	0.0446	-0.0593	0.2634	0.0290	-0.2072	0.7564
Co	0.4397	0.6087	0.3344	0.2120	0.1408	0.7554	-0.0423	-0.0542	0.3410	0.0122	-0.1092	0.6131
Mn	-0.0773	0.1888	-0.0331	0.4062	0.0836	0.2154	0.1012	0.0414	0.1719	0.0323	0.4464	0.3095
Fe	0.2260	0.6721	0.3439	0.4611	0.1552	0.7000	0.2169	-0.0256	0.2961	0.0368	0.2241	0.6778
As	0.3679	0.5953	0.1531	0.1946	0.1018	0.5945	0.2305	-0.0702	0.3467	0.0262	-0.1684	0.6211
U	-0.2958	-0.0938	-0.1534	0.2298	0.0734	-0.2006	0.0746	0.0359	-0.0341	-0.0048	0.5886	0.0346
Au	0.1303	0.1693	0.0265	0.0936	0.0158	0.2085	0.0928	-0.0133	0.0497	-0.0350	-0.0277	0.2107
Th	-0.3912	-0.2590	-0.1861	0.1116	-0.0070	-0.3332	0.0826	0.0454	-0.1024	-0.0039	0.4770	-0.1779
Sr	0.1810	0.5074	0.3728	0.1528	0.1082	0.3303	-0.1007	-0.0506	0.1050	-0.0239	-0.1472	0.4335
Cd	-0.1789	0.3601	-0.0172	0.4981	0.0644	0.2986	0.1658	-0.0030	0.3624	0.0502	0.5898	0.4308
Sb	0.4955	0.2656	0.0617	-0.0360	0.1938	0.3385	-0.0187	-0.0589	0.1496	-0.0097	-0.2664	0.3323
Bi	-0.5210	0.0809	-0.1217	0.4274	0.0095	-0.0557	0.3046	0.0188	0.1985	0.0163	0.8795	0.1615
V	0.5236	0.6167	0.4866	0.1298	0.2708	0.6922	-0.0422	-0.0648	0.2362	0.0058	-0.1648	0.5918
Ca	0.0083	0.3223	0.1932	0.1043	0.0955	0.0617	-0.0631	-0.0145	-0.0317	0.0022	-0.1405	0.2494
P	0.0863	0.5969	0.2470	0.4111	0.2576	0.4711	-0.1474	-0.0388	0.1800	0.0437	0.2284	0.6477
La	-0.2086	0.4025	0.0285	0.5188	-0.0755	0.3132	0.4684	0.0310	0.3684	0.0702	0.4739	0.4547
Cr	0.5779	0.6698	0.3122	0.1985	0.1023	0.8050	0.0362	-0.0580	0.2769	0.0128	-0.2341	0.6695
Mg	0.3373	0.7481	0.2658	0.4243	0.2271	0.6966	-0.0415	-0.0423	0.2280	0.0582	0.0207	0.8211
Ba	-0.1108	0.4775	0.2457	0.4704	0.0730	0.2585	0.1860	0.0745	0.0485	0.0376	0.0733	0.4223
Ti	1.0000	0.0948	0.1966	-0.2288	0.1749	0.3304	-0.2929	-0.0637	-0.0367	-0.0502	-0.4587	0.1803
Al	0.0948	1.0000	0.3363	0.6563	0.0774	0.8525	0.4522	-0.0348	0.4464	0.1037	0.1105	0.9492
Na	0.1966	0.3363	1.0000	0.0789	0.5842	0.3025	-0.0139	0.0331	0.1214	-0.0087	-0.0763	0.2610
K	-0.2288	0.6563	0.0789	1.0000	0.0507	0.4373	0.5847	0.0400	0.2064	0.0971	0.3746	0.6516
W	0.1749	0.0774	0.5842	0.0507	1.0000	0.0228	-0.1464	-0.0114	-0.0469	0.0132	0.0849	0.1220
Sc	0.3304	0.8525	0.3025	0.4373	0.0228	1.0000	0.3052	-0.0401	0.4852	0.0610	-0.0425	0.8423
Tl	-0.2929	0.4522	-0.0139	0.5847	-0.1464	0.3052	1.0000	0.0483	0.4367	0.1073	0.2416	0.3909
S	-0.0637	-0.0348	0.0331	0.0400	-0.0114	-0.0401	0.0483	1.0000	-0.0289	0.0671	-0.0071	-0.0261
Hg	-0.0367	0.4464	0.1214	0.2064	-0.0469	0.4852	0.4367	-0.0289	1.0000	0.0685	0.1731	0.4107
Se	-0.0502	0.1037	-0.0087	0.0971	0.0132	0.0610	0.1073	0.0671	0.0685	1.0000	0.0295	0.0925
Te	-0.4587	0.1105	-0.0763	0.3746	0.0849	-0.0425	0.2416	-0.0071	0.1731	0.0295	1.0000	0.1972
Ga	0.1803	0.9492	0.2610	0.6516	0.1220	0.8423	0.3909	-0.0261	0.4107	0.0925	0.1972	1.0000

All elements for the soil samples are reported in ppm except for Au and Hg which are reported in ppb, and Fe, Ca, P, Mg, Ti, Al, Na, K, and S.

Table 9-11: Basic Statistics for Soil Samples

Field	Count_n	Count Valid	Count Invalid	Minimum	Maximum	Mean	Median	Range	Mode	Std Dev
Mo	1123	1123	0	0.13	2.84	0.781	0.75	2.71	0.71	0.271
Cu	1123	1123	0	9.69	189.36	29.362	26.64	179.67	30.64	13.830
Pb	1123	1123	0	3.56	36.47	18.370	17.38	32.91	14.39	5.091
Zn	1123	1123	0	15.6	91.4	42.728	39.6	75.8	27.4	15.961
Ag	1123	1123	0	13	387	101.634	86	374	64	53.167
Ni	1123	1123	0	3.5	29.3	8.923	7.4	25.8	6.4	4.056
Co	1123	1123	0	2.9	22.4	7.523	6.8	19.5	6.7	2.493
Mn	1123	1123	0	166	1400	511.035	465	1234	403	181.721
Fe	1123	1123	0	1.03	4.4	2.011	1.93	3.37	1.64	0.426
As	1123	1123	0	0.6	32.8	4.928	3.9	32.2	2.8	3.401
U	1123	1123	0	0.4	6.1	1.430	1.2	5.7	1	0.638
Au	1123	1123	0	0.1	2202.9	18.261	3	2202.8	2	95.185
Th	1123	1123	0	0.7	74.8	12.834	11.5	74.1	3.1	8.066
Sr	1123	1123	0	5.4	292.5	29.560	19.5	287.1	14.2	30.722
Cd	1123	1123	0	0.04	0.61	0.191	0.17	0.57	0.15	0.077
Sb	1123	1123	0	0.07	3.04	0.410	0.33	2.97	0.26	0.251
Bi	1123	1123	0	0.06	2.51	0.820	0.68	2.45	0.61	0.414
V	1123	1123	0	15	115	36.427	32	100	31	13.158
Ca	1123	1123	0	0.04	16.1	0.700	0.23	16.06	0.1	1.691
P	1123	1123	0	0.007	0.132	0.036	0.026	0.125	0.014	0.025
La	1123	1123	0	6.6	46.8	28.327	27.8	40.2	28.8	4.461
Cr	1123	1123	0	5	46.7	13.523	11.7	41.7	10.3	5.476
Mg	1123	1123	0	0.09	1.44	0.316	0.25	1.35	0.13	0.210
Ba	1123	1123	0	33.3	847.5	106.315	96.6	814.2	74.6	52.203
Ti	1123	1123	0	0.006	0.122	0.028	0.026	0.116	0.026	0.014
B	1123	1	1122	30	30	30.000	30	0	30	0.000
Al	1123	1123	0	0.56	3.49	1.220	1.1	2.93	0.8	0.483
Na	1123	1123	0	0.0005	0.167	0.007	0.006	0.1665	0.005	0.007
K	1123	1123	0	0.09	0.47	0.203	0.19	0.38	0.14	0.065
W	1123	968	155	0.1	5.4	0.196	0.2	5.3	0.2	0.191
Sc	1123	1123	0	1.5	7.3	3.041	2.8	5.8	2.3	0.970
Tl	1123	1123	0	0.04	0.42	0.166	0.16	0.38	0.16	0.040
S	1123	90	1033	0.02	0.11	0.029	0.03	0.09	0.02	0.016
Hg	1123	1123	0	2.5	128	32.138	30	125.5	28	12.677
Se	1123	244	879	0.1	0.5	0.186	0.2	0.4	0.2	0.087
Te	1123	1080	43	0.02	0.24	0.077	0.07	0.22	0.05	0.043
Ga	1123	1123	0	1.9	12.6	4.259	3.9	10.7	2.7	1.654

Gold is distributed downslope from the Mexican Hat Deposit as shown on Figure 9-6.
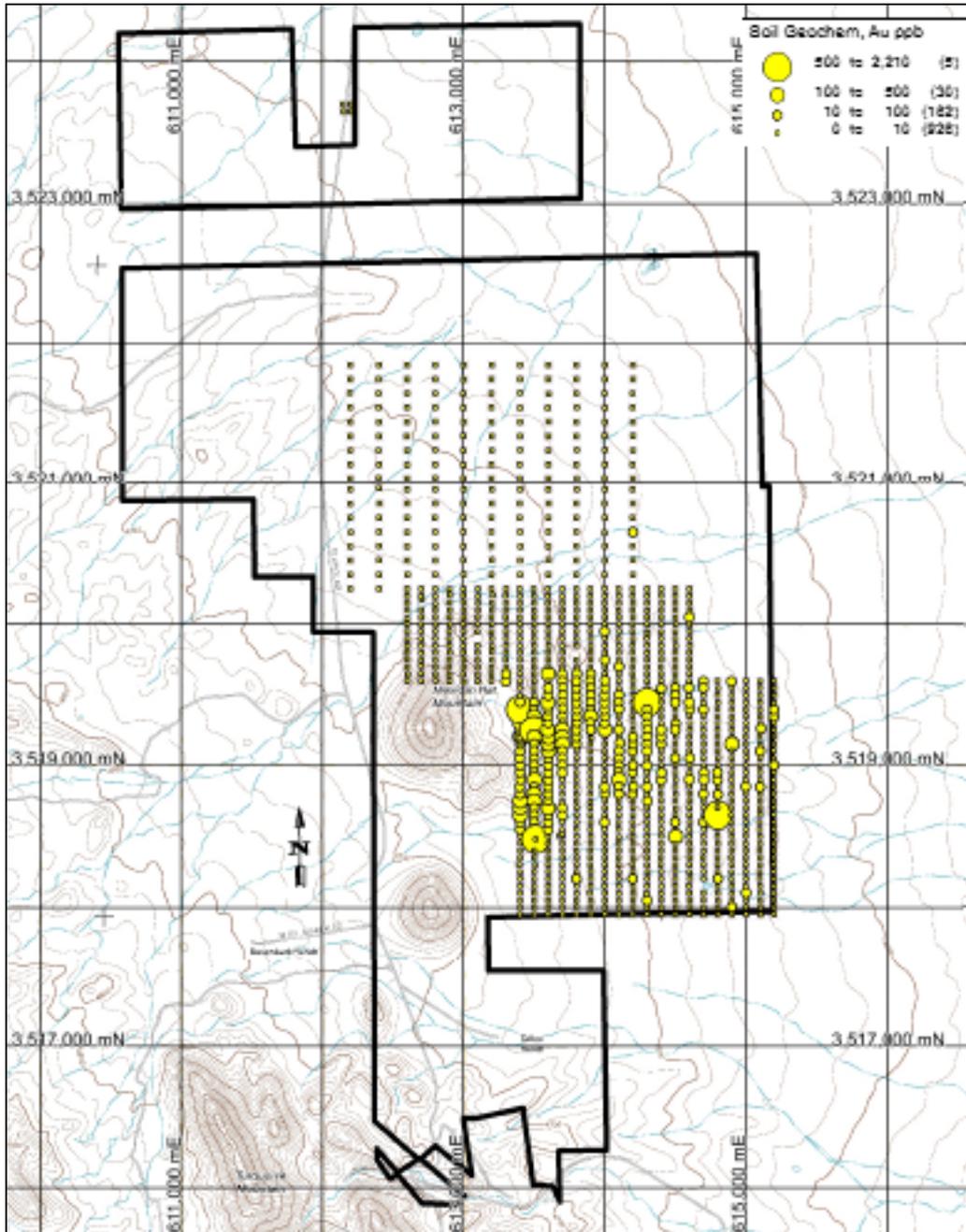
Figure 9-6: Distribution of Gold in Soils

For n>300, significant correlations at the 99th percentile is 0.148. Gold correlates positively with Ag, Ni, Fe, As, Sb, V, Cr, Mg, Al, Sc, Ga and negatively with no elements. The distribution of these elements is shown on Figure 9-7 through Figure 9-11 below.
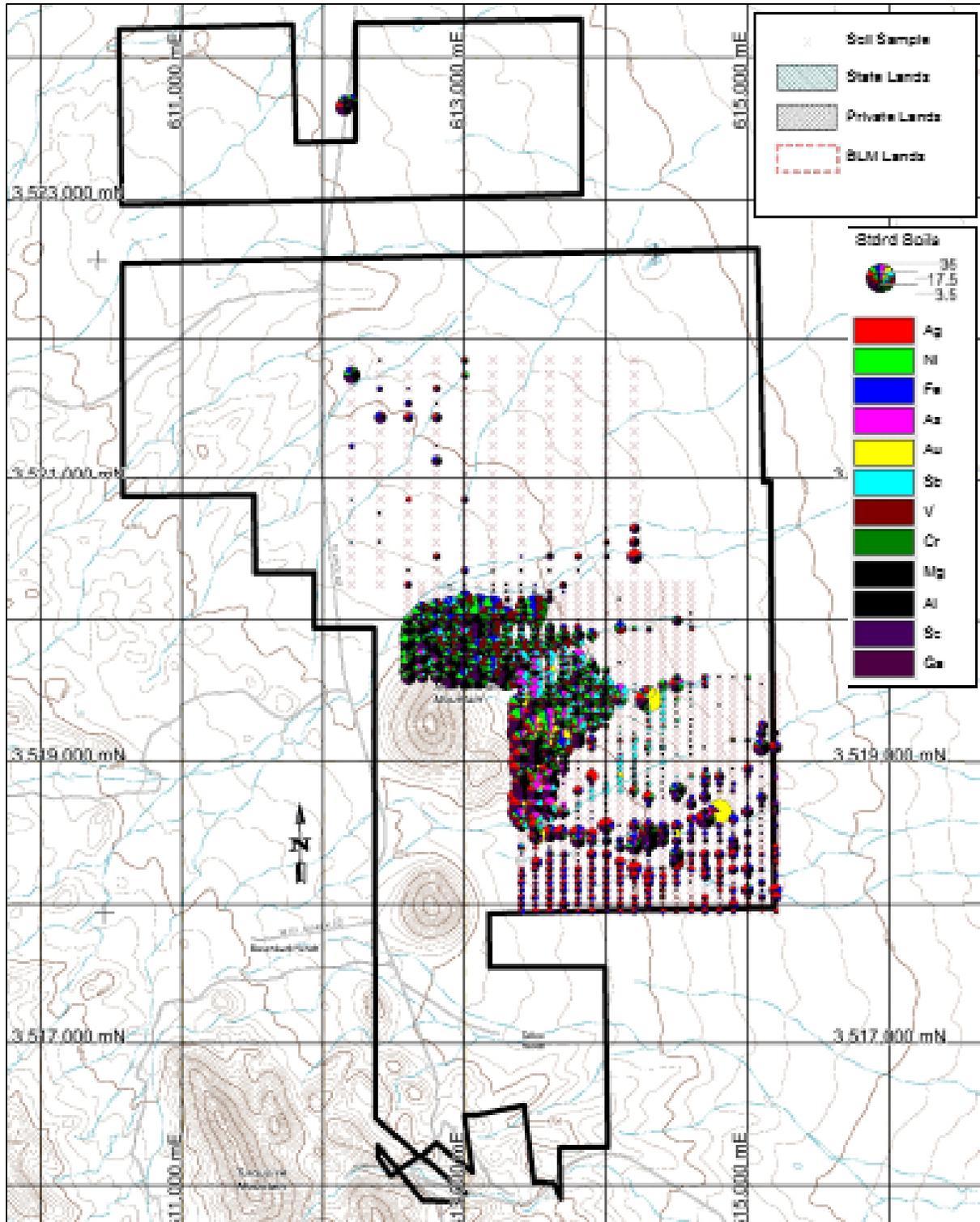
Figure 9-7: Distribution of all Elements Positively Correlating with Gold
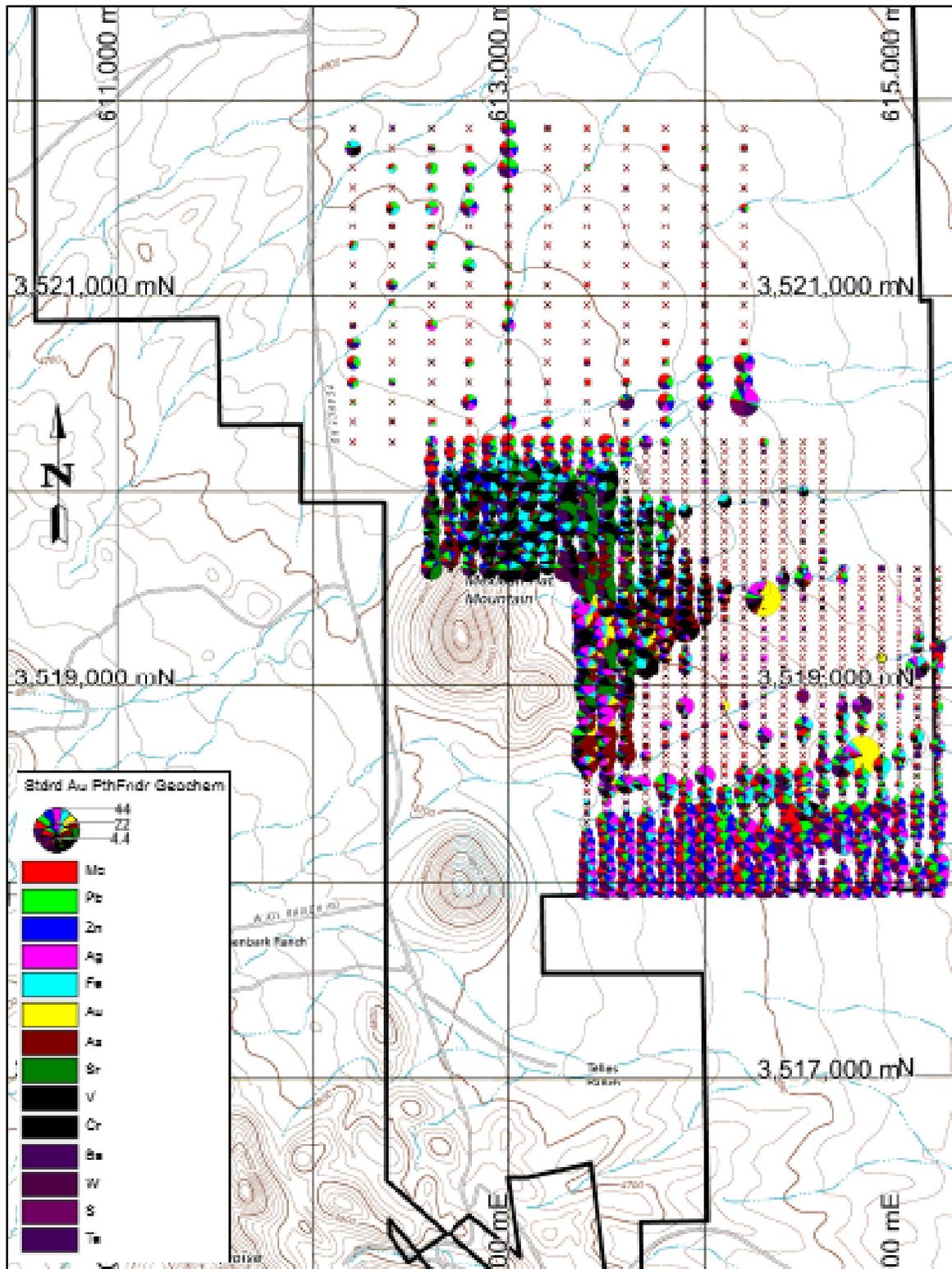
Figure 9-8: All Gold Pathfinders from Rock Geochemistry on Soils
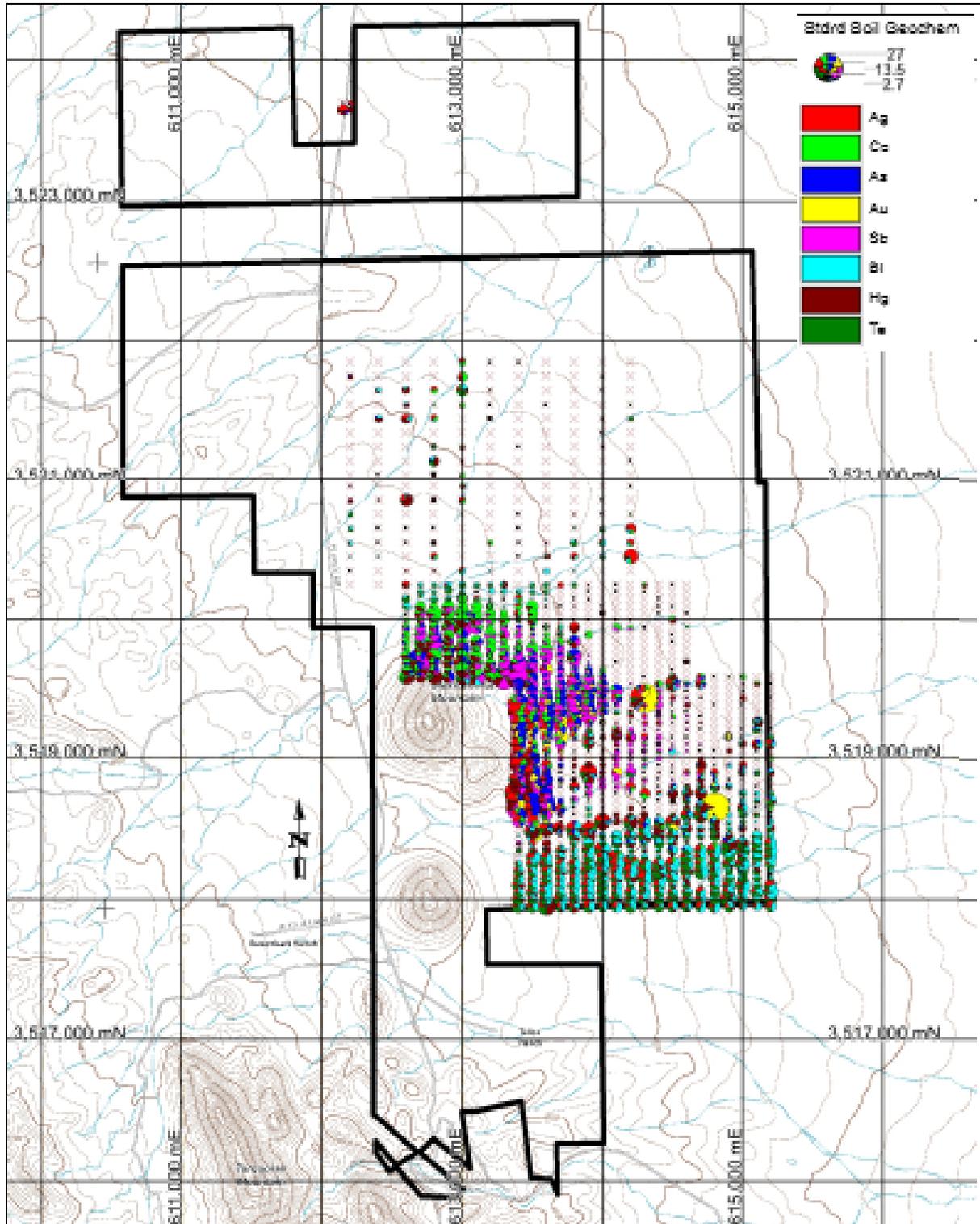
Figure 9-9: Distribution of Typical Epithermal Pathfinder Elements
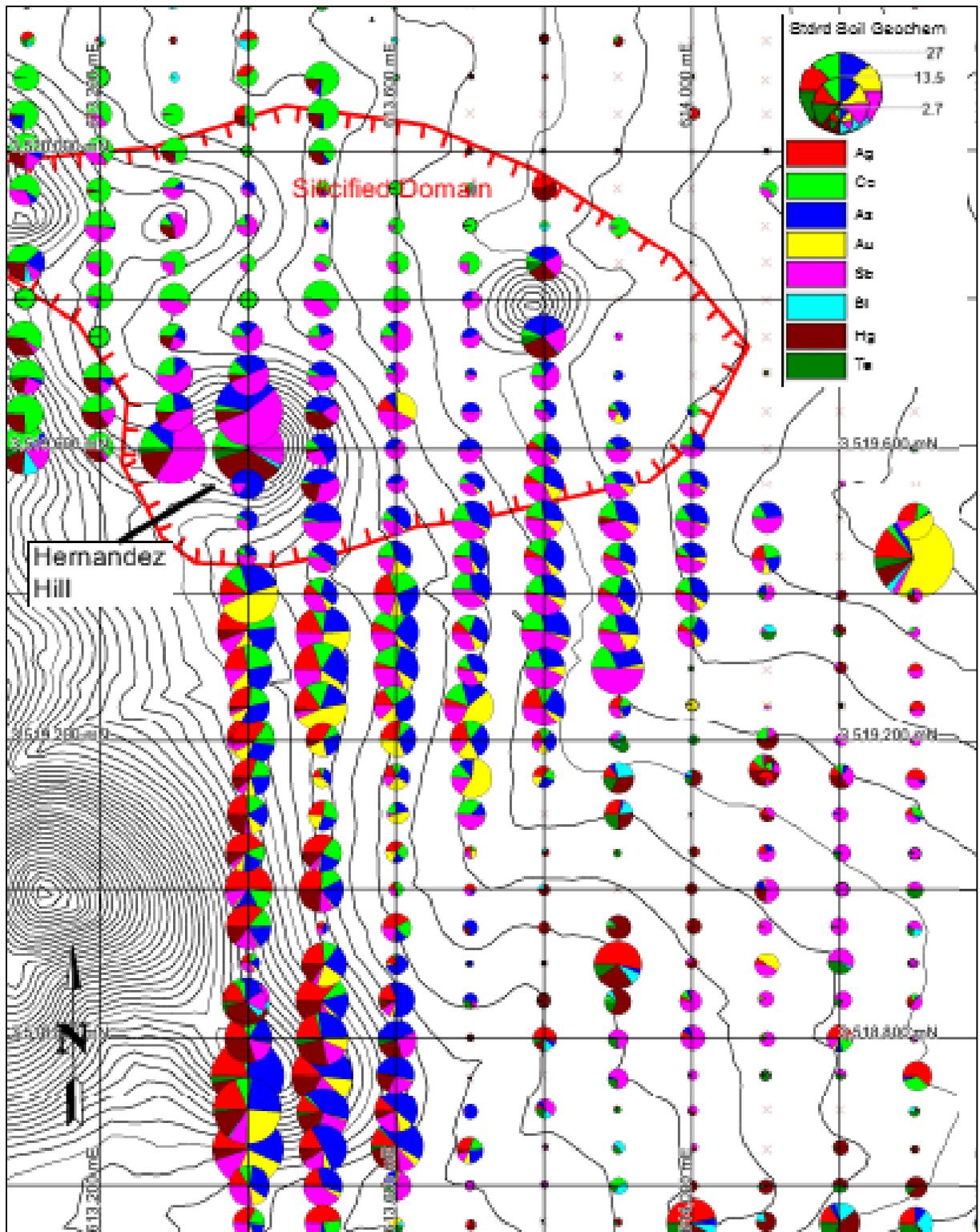
Figure 9-10: Close-up of Hernandez Hill Area
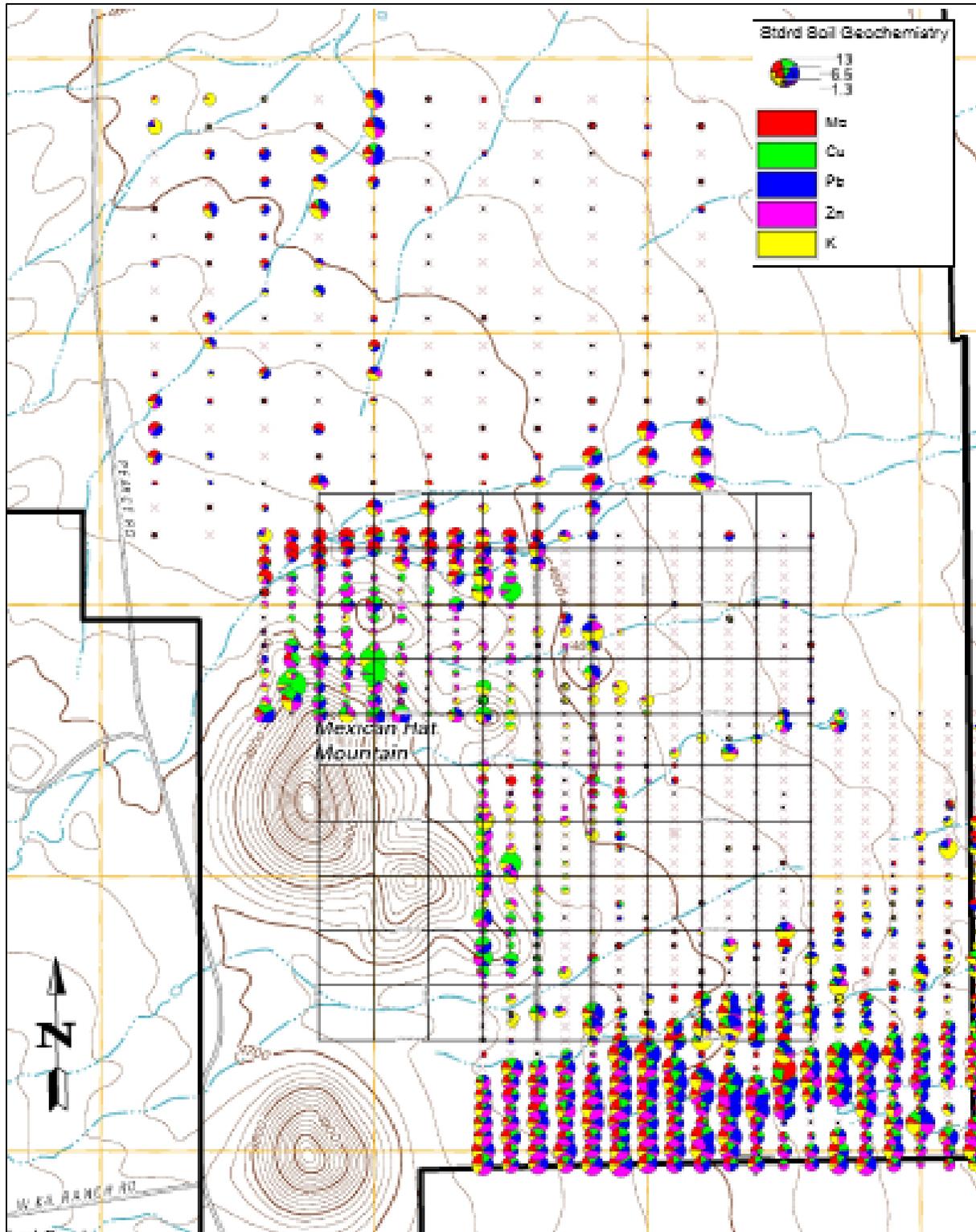
Figure 9-11: Distribution of Common Porphyry Copper Pathfinder Elements

9.2.2   Airborne Photogrammetry

Cooper Aerial Surveys Ltd., from Phoenix was contracted to set some ground GPS stations and to fly the property to establish better topographic controls.

9.2.3   Geophysics

Zonge International Inc. was contracted to complete new ground magnetic surveys to expand the existing ground magnetic surveys completed by Auracle, to test three AMT lines and to collect some gravity data across portions of the property. They were also to provide a synthesis of these data.

9.2.3.1   Magnetics

Ground magnetic surveys were completed by Zonge International of Tucson to expand the existing survey. A total of 85.6 line km of survey was completed using a GEM Systems GSM-19W Overhauser-effect magnetometer. The GSM-19 magnetometer has a resolution of 0.01 nT and an accuracy of 0.2 nT over the operating range. Positioning was made with an integrated Novatel Superstar II DGPS board. The GPS data were differentially corrected in real-time and positions are integrated with the raw magnetometer readings. The system provides sub-meter accuracy under standard operating conditions.

This data was then merged with the 2011 data collected by Geotronix Consulting Inc. for Auracle. The combined RTP and first derivative of the RTP data survey is shown in Figure 9-12 and Figure 9-13.
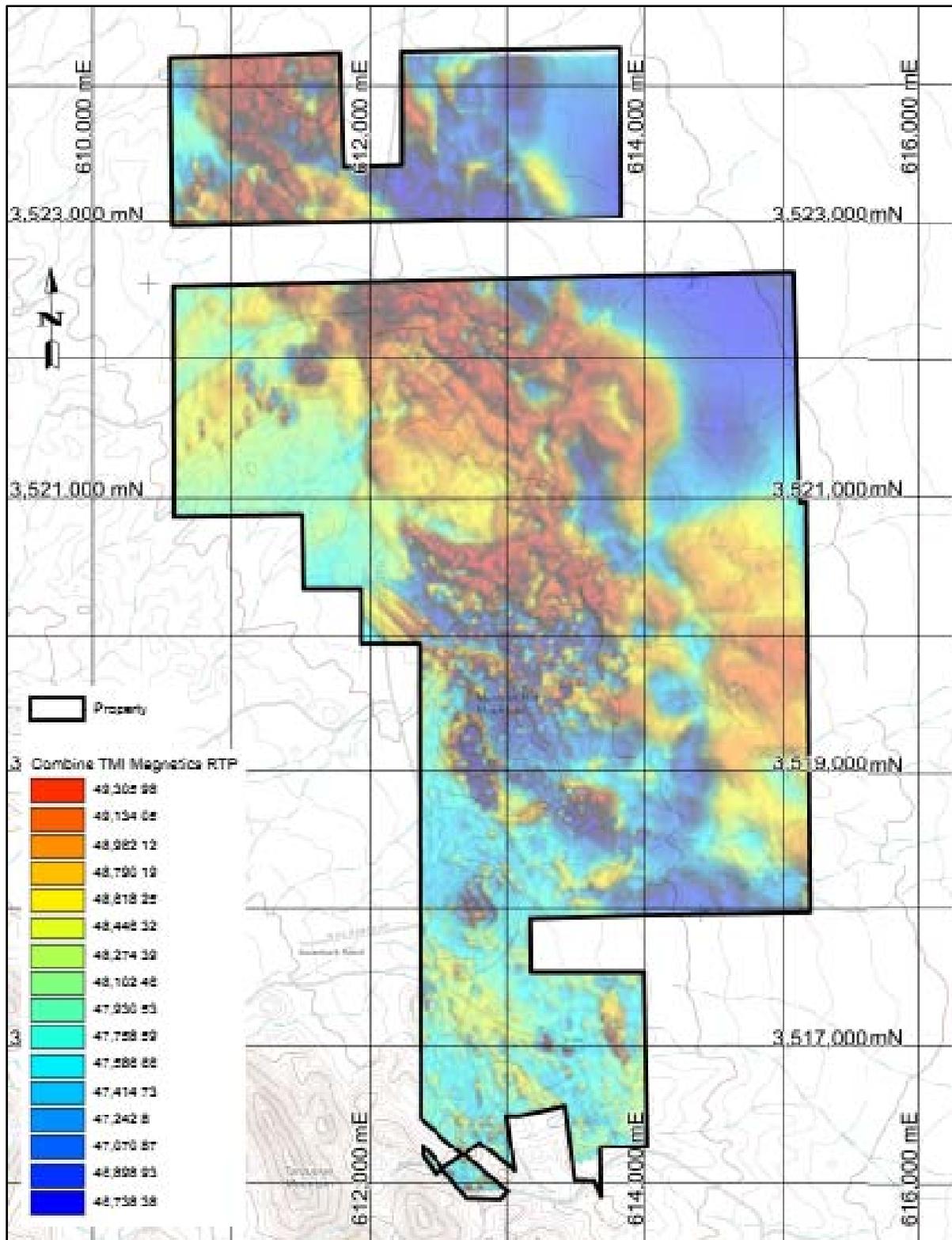
Figure 9-12: Combined TMI Survey RTP after Zonge (2017)
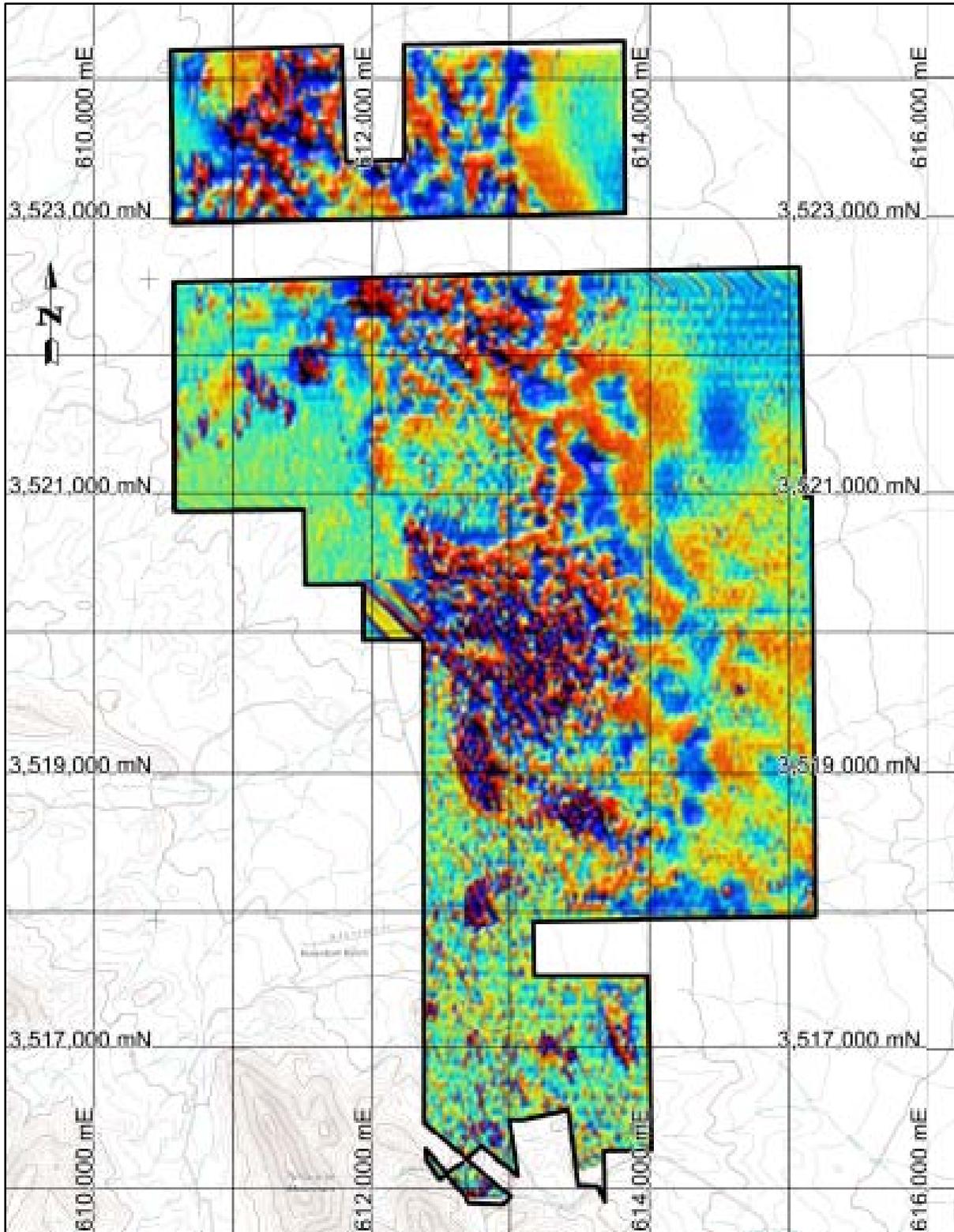
Figure 9-13: First Vertical Derivative of Combined TMI Survey, RTP after Zonge (2017)

9.2.3.2   Audiomagnetotelluric

Three lines of audiomagnetotelluric were tested across the eastern portion of the Mexican Hat Deposit as shown on Figure 9-14.
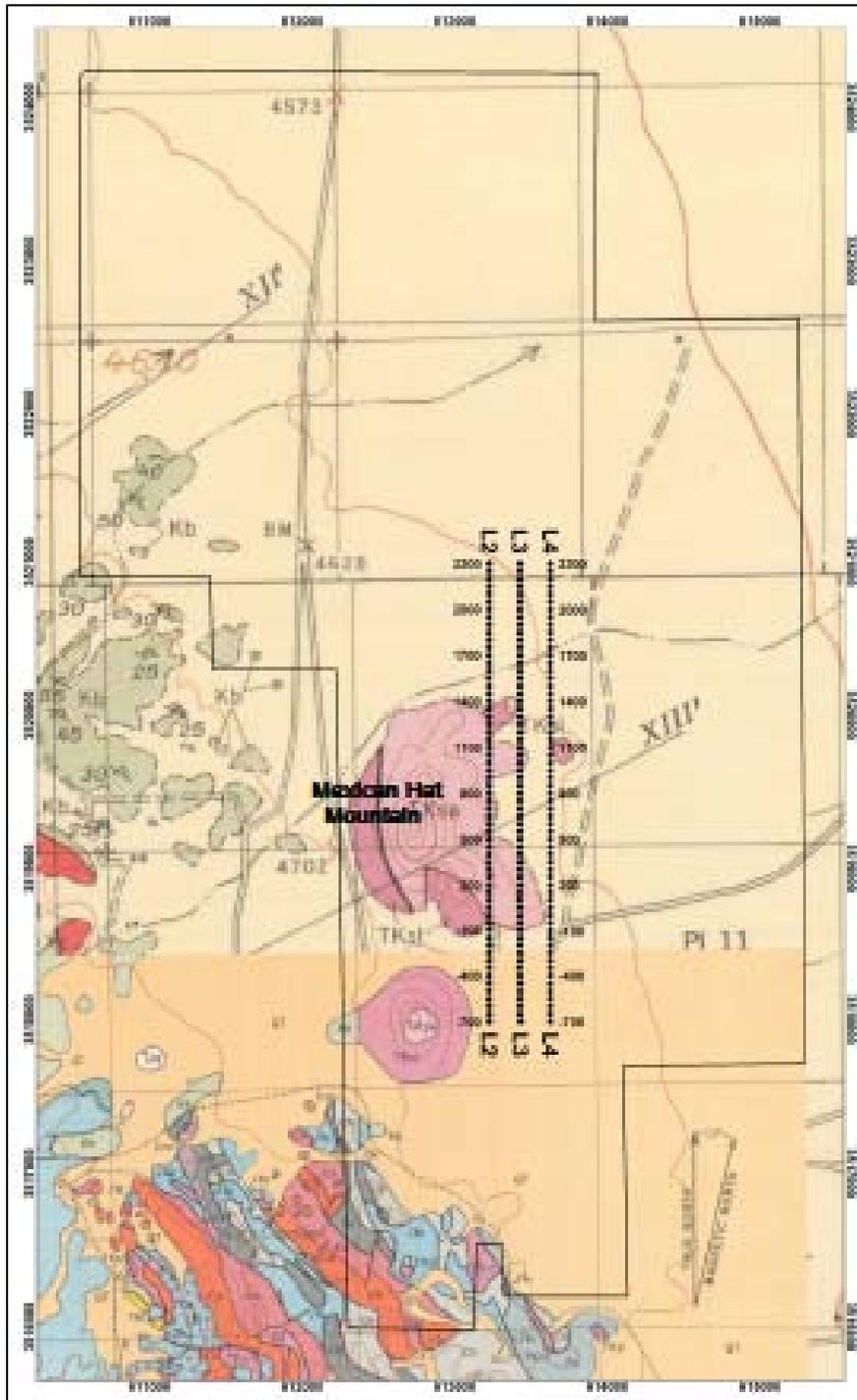
Figure 9-14: Location of AMT Test Lines
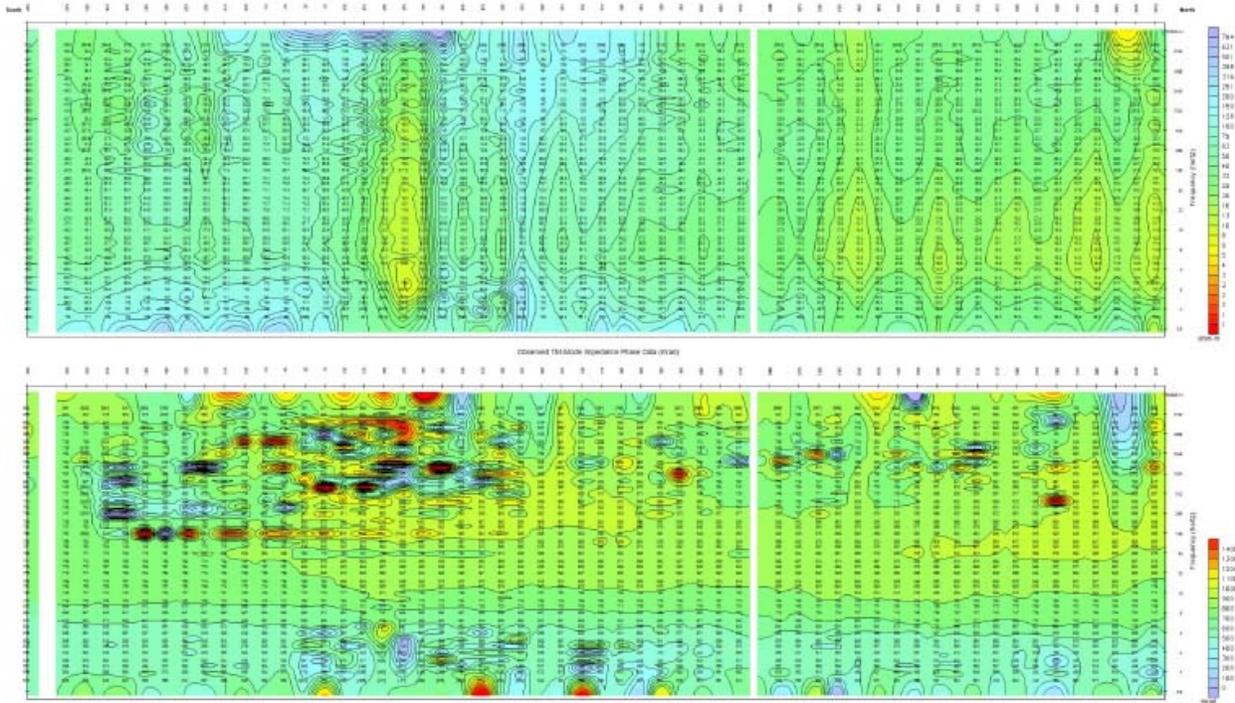
Figure 9-15: West-facing Apparent Resistivity and Phase Pseudosection of Line 2
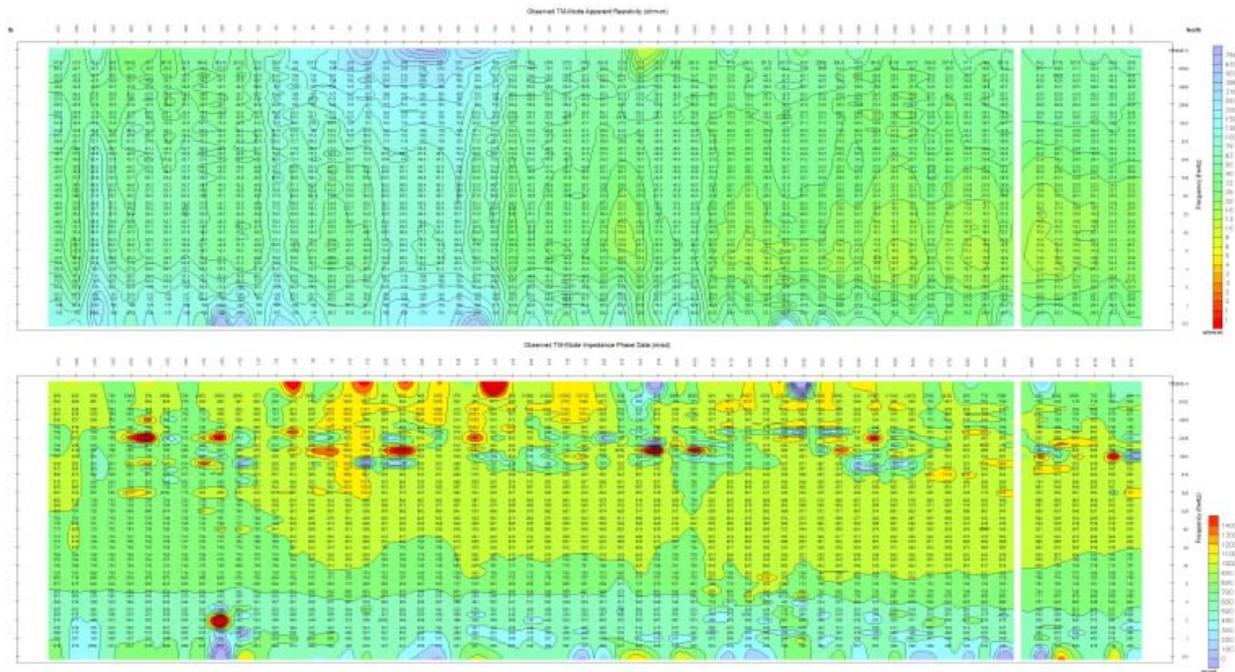
Figure 9-16: West-facing Apparent Resistivity and Phase Pseudosection of Line 3
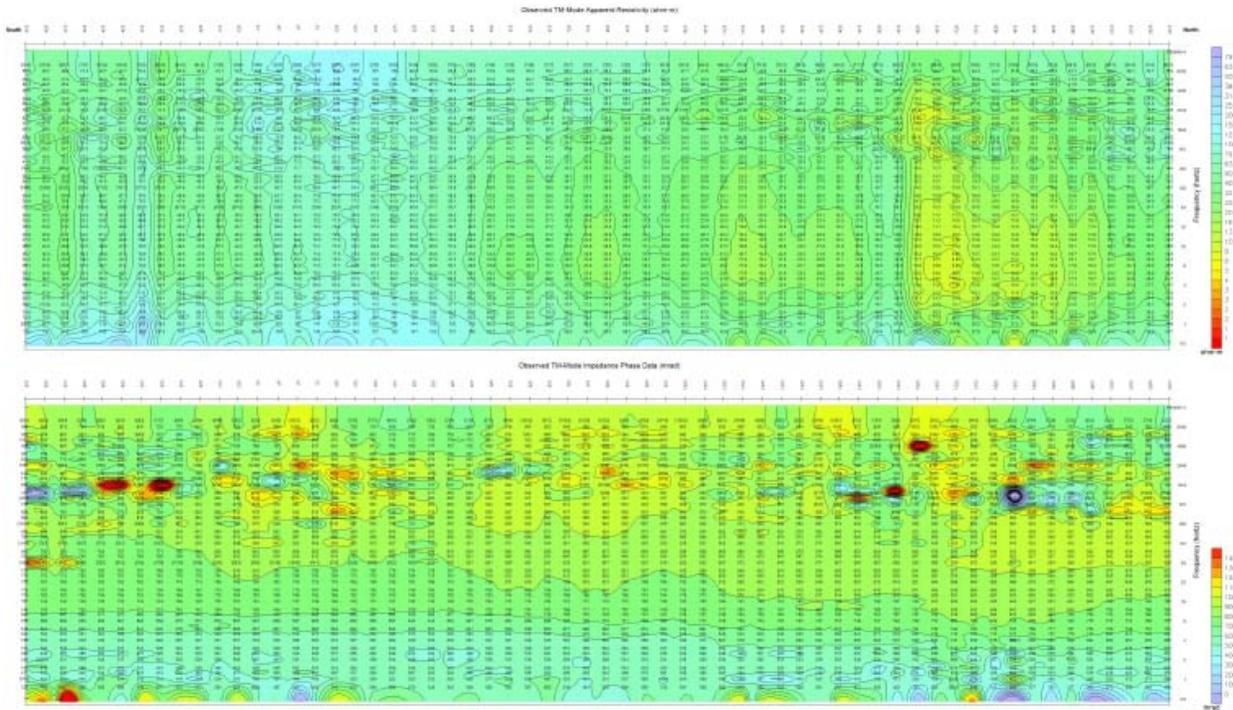
Figure 9-17: West-facing Apparent Resistivity and Phase Pseudosection of Line 4
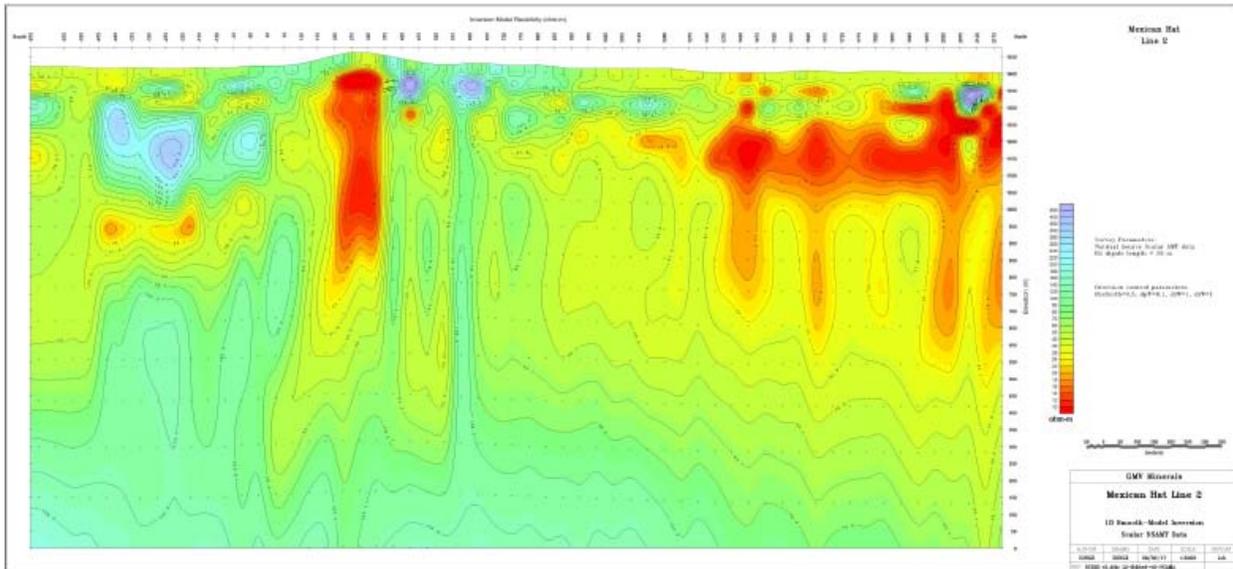
Figure 9-18: Section Showing West-facing 1D Inversion of Line 2 AMT Data
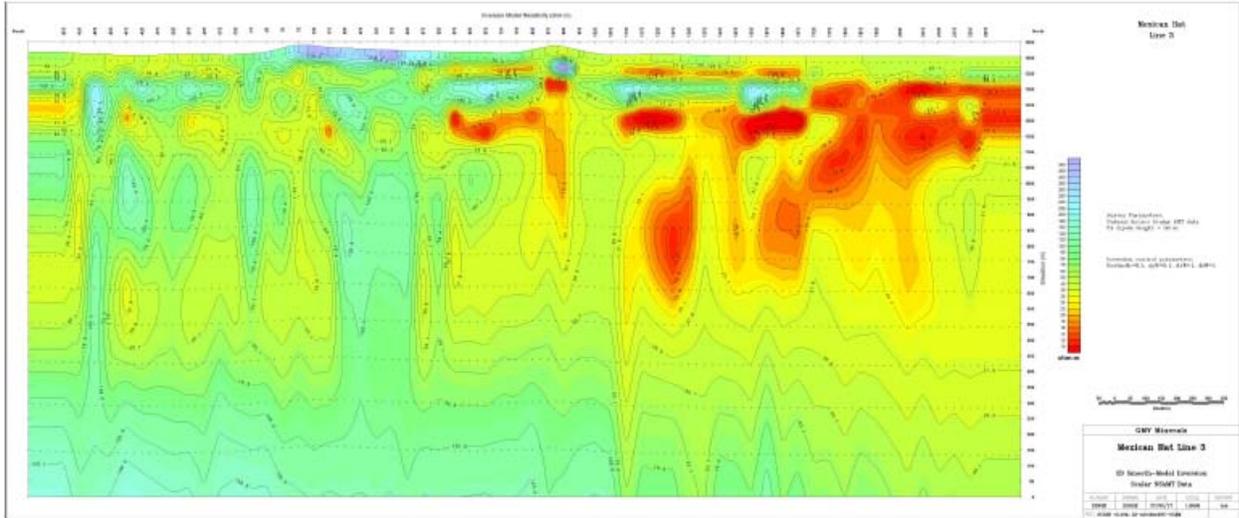
Figure 9-19: Section Showing West-facing 1D Inversion of Line 3 AMT Data
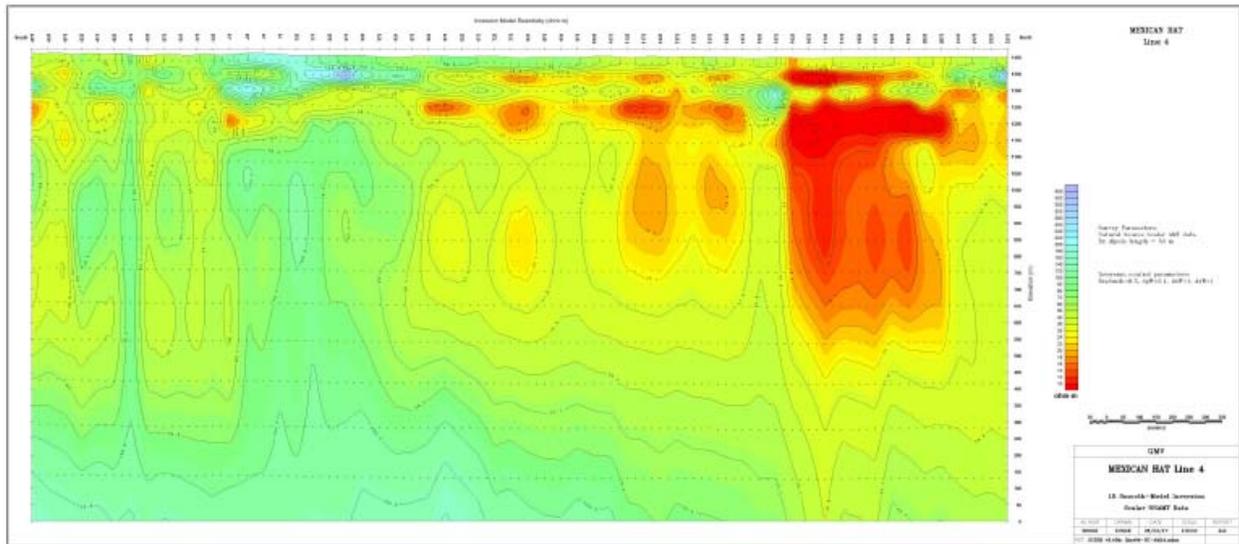
Figure 9-20: Section Showing West-facing 1D Inversion of Line 4 AMT Data

Zonge (2017) reported that, in general, the modeled resistivities are low over the entire cross-sections of the lines. The resistivity range of the majority of the cross sections is between about 30 ohm-m to 150 ohm-m. High resistivity anomalies are only around 300 ohm-m. Low resistivity zones are about 10 ohm-m.

AMT Line 2 is the line collected nearest Mexican Hat Mountain (Figure 9-18). There is a prominent low resistivity anomaly along the line in the vicinity of the mountain between stations 175 and 375. The resistivities in this zone indicate a layered structure consistent with the interpreted geology. There is a high resistivity zone about 50 m thick overlying a low resistivity zone, also about 50 m thick. Below that is a second low resistivity zone that is sharply bounded to the north by what appears to be a near vertical fault at about station 375. The deeper low resistivity zone extends to a depth of about 500 meters. This low resistivity anomaly would be consistent with mineralization extending vertically through a zone of fractures. There is a minor magnetic low over the resistivity low, consistent with magnetite destruction or alteration of magnetite to maghemite hematite. The northern third of Line 2 has a second low resistivity zone which appears as a horizontal layer about 200 m thick. It is overlain by thinner, more resistive layers. The low resistivity zone

begins about station 1300 and continues northward to the end of the line. The vertical variability in resistivity in the north points to a layered earth of different rock types, e.g. sand over volcanic flows over consolidated sediments, rather than thick unconsolidated basin fill.

Line 3 is offset 200 m east of Line 2, and consequently is further from Mexican Hat Mountain, but as can be seen in Figure 9-19, stations pass over two lobes that extend eastward from the main body. Similar to Line 2, near the mountain there is evidence of a layered structure over a deeper low resistivity zone, but it is displaced further north than on Line 2, between stations 850 and 950, an interval near the north eastward extending lobe. Also, as with Line 2, the northern portion of the line has a thick conductive layer at about 150 m. However, at about station 1925, the conductive zone dips to the south at about 40 degrees.

AMT Line 4 is offset from Line 3 by 200 m. The deep vertical conductive zones in the vicinity of Mexican Hat Mountain, which appear in Lines 2 and 3, are absent in Line 4. The high resistivity horst like feature appears between about stations -125 and 375. There is strong indication of horizontal layering in the north half of the line, and at station 1575 there is what appears to be a vertical fault or contact bounding a low resistivity zone that extends to a depth of about 800 m. The resistive pods at about 100 m depth could represent volcanic flows that could have strong positive magnetization.

9.2.3.3   Gravity

Three gravity profiles were collected by Zonge in 2017. One was oriented north-south and was coincident with the southern end of AMT Line 4. One angled to the northeast across Mexican Hat Mountain. A third line trended east-west, just north of the north ends of the AMT lines. The data were reduced to terrain corrected Bouguer anomalies using densities from 2.00 g/cc to 3.00 g/cc. The terrain corrected anomaly that used a 2.40 g/cc reduction was chosen for 2D inverse modeling. The 2D modeling used the GM-SYS program and fit the observed data with a model computed from 2D polygons having varying densities and shapes.
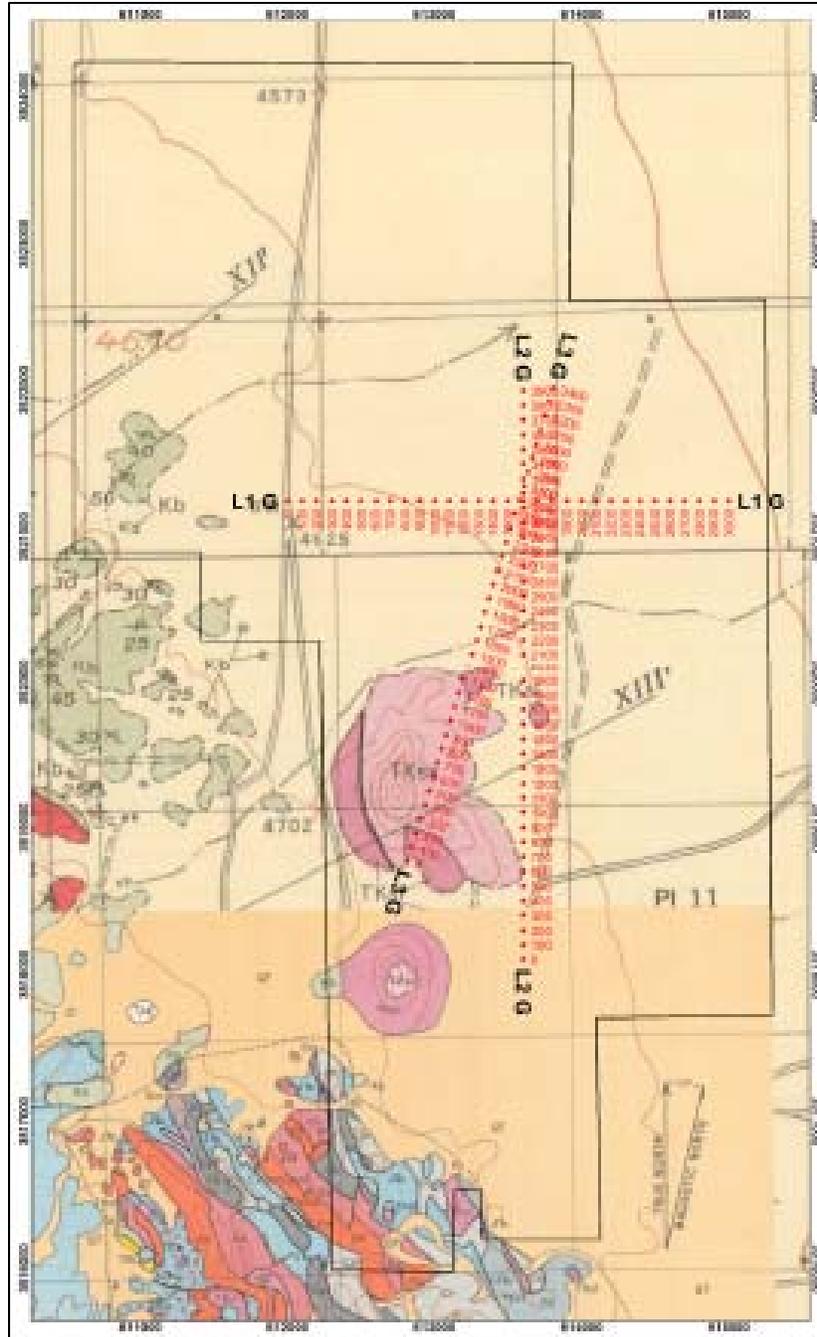
Figure 9-21: Location of Three Gravity Profile Lines

The following is drawn from Zonge (2017). Data from west-to-east gravity Line 1G is shown in Figure 9-22, along with RTP magnetic data sampled along the same line path. The data have been terrain corrected to a reduction density of 2.40 g/cc. The gravity profile monotonically decreases from west to east, with a small inflection at the center of the line. The magnetic data also show a pronounced negative anomaly at the gravity inflection. The inverse model is shown in Figure 9-23. The model has three rock types: rock 1 represents unconsolidated basin fill; rock 2 represents units in the west that may contain more and denser basaltic or other volcanic units; rock 3 represents sedimentary units below the basin fill and volcanics. The inverse model shows basin fill that in largely thin in the west and increases in thickness to

the east. The volcanic-rich unit, rock 2, has variable thickness, but is thicker in the east and thins to the west. In the far west, the high density volcanic-rich units are no longer required for a good data fit. Volcanic units may still be present, but not in great enough volume to affect the gross density in the western part of the line.
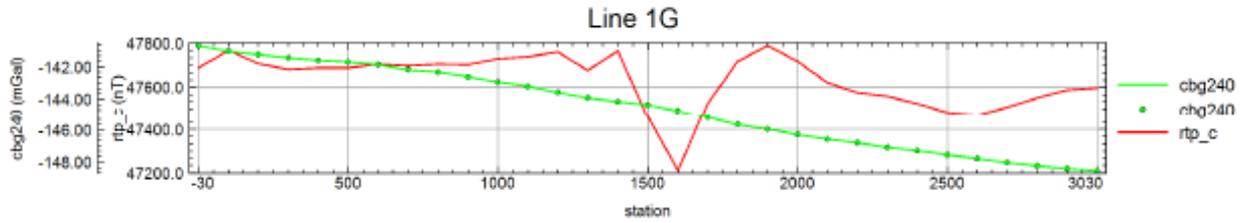
Figure 9-22: Section Along Gravity Line 1 with RTP Magnetic Data
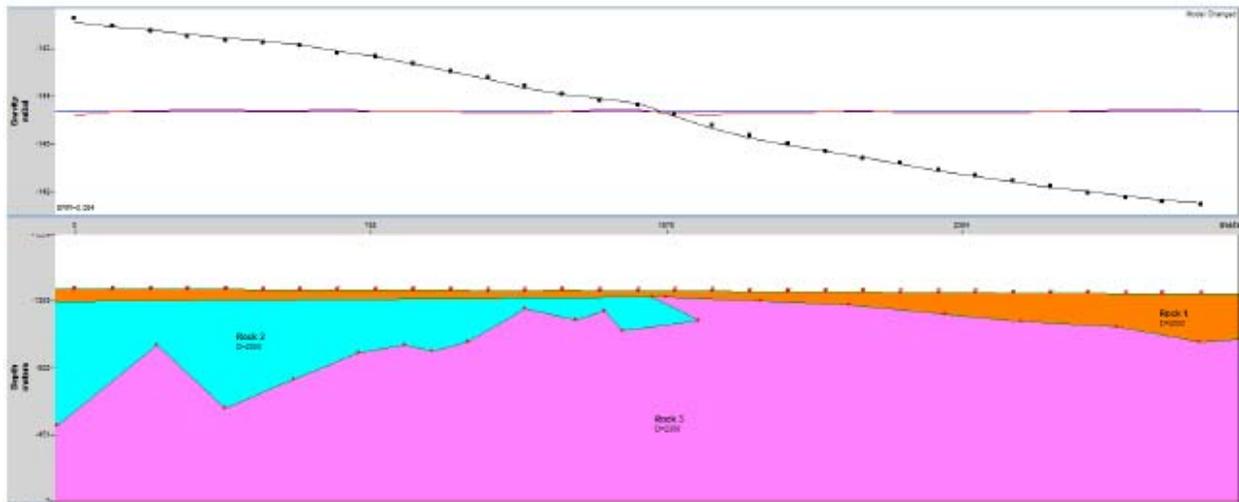
Figure 9-23: 1G Gravity Data Fit and Inverse Model for Line 1G. Upper Panel Shows Gravity Data (black dots). The Model Fit (solid black curve) and Fit Error (red curve)

Data from south-to-north gravity Line 2G is shown in Figure 9-24, along with RTP magnetic data sampled along the same line path. The data have been terrain corrected to a reduction density of 2.40 g/cc. The gravity profile monotonically decreases from south to north, with small positives near gravity stations 700 and 2000. The magnetic data also show a pronounced negative anomaly at station 550. The inverse model is shown in Figure 9-25. The model has four rock types: rock 1 represents unconsolidated basin fill; rock 2 represents units in the west that may contain more and denser basaltic or other volcanic units; rock 3 represents sedimentary units below the basin fill and volcanic rocks. Rock 4 represents dense basement rock, or alternatively, igneous intrusions. The inverse model shows a model in which the basement slopes slightly to the north. The volcanic-rich unit, rock 2, has variable thickness, but is thin through the length of the profile. Near station 700, the high-density intrusion is required for a good data fit. This intrusive unit at 700 appears in the AMT data to extend from the deepest portion of the resistivity model. The data near gravity station 2000 (AMT station 1500) also require a dense intrusive structure for a satisfactory fit. In the AMT inverse model, this is near a narrow vertical resistive structure adjacent to low resistivity zone to the north.
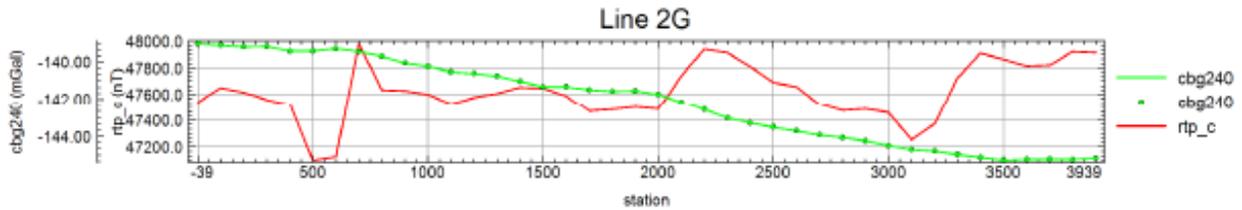
Figure 9-24: Terrain Corrected Gravity and RTP Magnetic Data along Line 2G
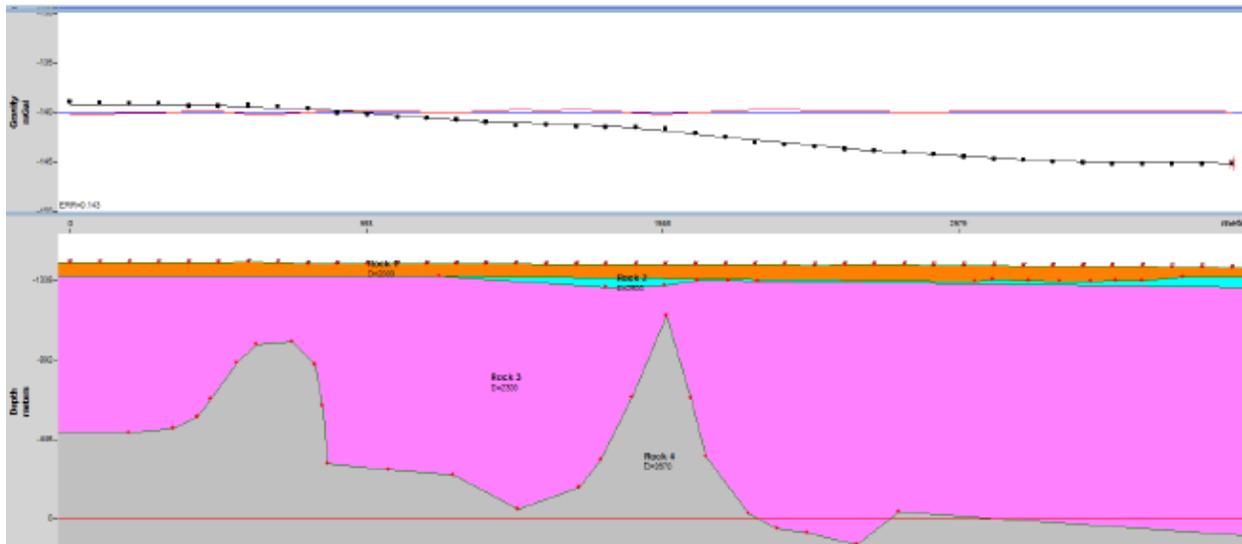
Figure 9-25: Gravity Data Fit and Inverse Model for Line 2G. Upper Panel Shows Gravity Data (black dots), the Model Fit (solid black curve), and Fit Error (red curve)

Data from southwest-to-northeast gravity Line 3G is shown in Figure 9-26, along with RTP magnetic data sampled along the same line path. The data have been terrain corrected to a reduction density of 2.40 g/cc. The gravity profile monotonically decreases from southwest to northeast. The inverse model is shown in Figure 9-27. The model used has four rock types: rock 1 represents unconsolidated basin fill; rocks 2a and 2b represents units may contain denser volcanic units; rock 3 represents sedimentary units below the basin fill and volcanic units. The line starts at Mexican Hat Mountain, which is modeled as a thick high density unit. A second thick dense unit is modelled at station 1600, and may be related to the broad magnetic high zone in the same location, from about 1500 to 1900. In the resistivity model, a south dipping conductor is in this is the same location in AMT Line 3.
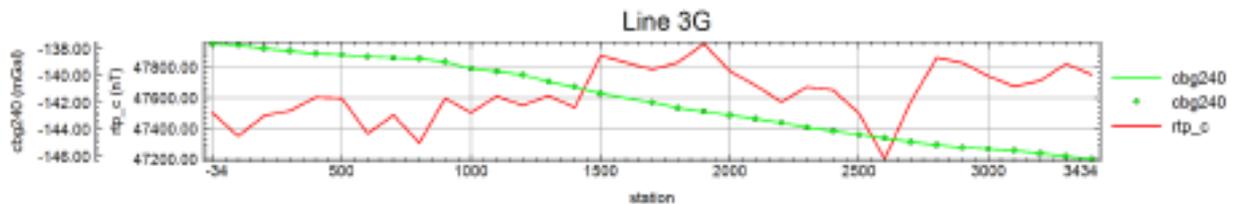
Figure 9-26: Terrain Corrected Gravity and RTP Magnetic Data along Line 3G
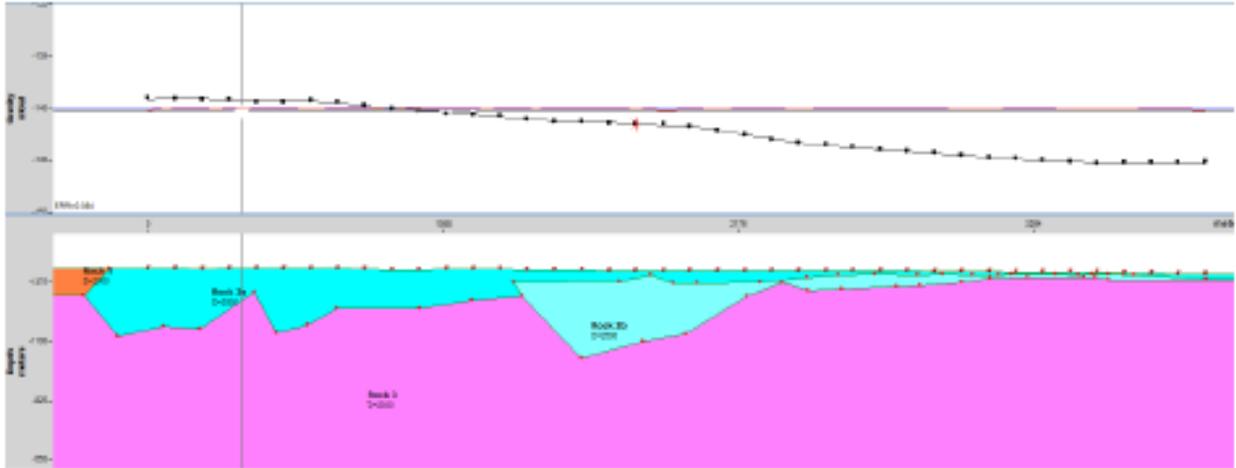
Figure 9-27: Gravity Data Fit and Inverse Model for Line 3G. Upper Panel Shows Gravity Data (black dots), the Model Fit (solid black curve), and Fit Error (red curve)

10   DRILLING

GMV has completed three drill campaigns on the Mexican Hat Property since 2014, completing 15 reverse circulation (RC) holes totalling 4,776.5 m in 2016, and in 2017 completed 8 HQ core holes totalling 1,979.3 m and 15 RC holes totalling 4,032.9 m.

All drilling completed on project by GMV and previous operators to date is summarized and listed in Table 10-1 and is shown on Figure 10-1.

Table 10-1: Summary of Drilling Completed on the Mexican Hat Property

Year	Company	Sample Type	Number of Holes	Total Meters	Comments
1989					
	PDI				
		CORE	17	2,446.8	
		ROTARY	88	12,515.7	
1990					
	PDI				
		ROTARY	32	3,977.3	
1996					
	Kalahari	ROTARY	18	3,771.9	Holes not used in mineral resource estimate
2011					
	Auracle				
		CORE	19	2,586.9	
2016					
	GMV				
		RC	15	4,776.5	
2017					
	GMV				
		CORE	8	1,979.3	
		RC	15	4,032.9	
Grand Total			212	36,087.3	
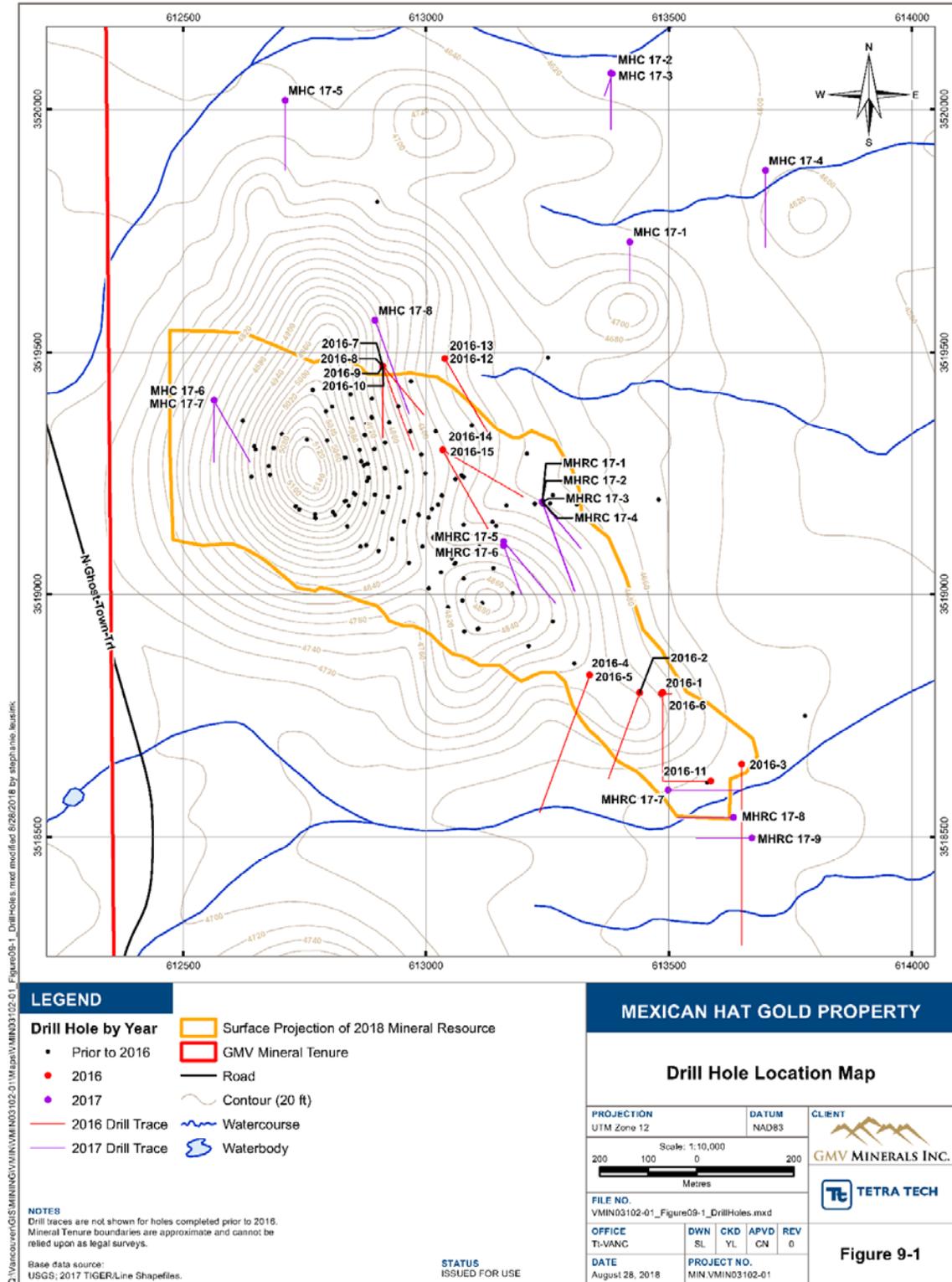
Figure 10-1: Overview of Drilling Completed on the Mexican Hat Property

10.1   2016 DRILLING

Fifteen RC holes were completed on the eastern portion of the Mexican Hat Deposit, specifically targeting the 120o striking zone. Table 10-2 below summarizes the drilling completed in 2016.

O'Keefe Drilling Company of Butte, Montana was contracted to provide drilling services.

Table 10-2: 2016 RC Drillholes

hole_id	azimuth	dip	east_m	north_m	elev_m	td__m_	drill_type
2016 1	180	-50	613487.0	3518797.0	1430.2	283.0	RC
2016 2	200	-50	613440.0	3518797.0	1437.9	293.0	RC
2016 3	180	-45	613650.0	3518650.0	1410.6	530.0	RC
2016 4	200	-85	613337.0	3518833.0	1440.1	300.0	RC
2016 5	200	-60	613337.0	3518833.0	1440.1	600.0	RC
2016 6	90	-85	613485.0	3518795.0	1430.7	238.0	RC
2016 7	180	-60	612911.0	3519472.0	1476.4	299.0	RC
2016 8	160	-50	612911.0	3519472.0	1476.4	287.0	RC
2016 9	140	-60	612911.0	3519472.0	1476.4	262.0	RC
2016 10	140	-80	612911.0	3519472.0	1476.4	299.0	RC
2016 11	270	-60	613586.0	3518615.0	1411.6	192.0	RC
2016 12	150	-50	613039.0	3519487.0	1444.9	299.0	RC
2016 13	150	-60	613039.0	3519487.0	1444.9	305.0	RC
2016 14	150	-50	613034.0	3519297.0	1457.9	290.0	RC
2016 15	120	-50	613034.0	3519297.0	1457.9	299.0	RC

10.1.1   Collar Surveys

All holes were numbered in accordance to a nomenclature scheme which reflected the chosen drilling method for that hole. Diamond drillholes contained the prefix MHC, where as RC holes contained the prefix MHRC. Overall, holes were labeled using the following template "Prefix Identifier" dash "year of drilling" dash "the hole number". For example, MHC-17-02.

All holes were placed and oriented using field GPS and compass and checked upon completion for accuracy.

10.1.2   Downhole Survey

Down hole surveys were taken at the end of each hole, along with a top of the hole survey.

10.1.3   Logging

Logging and sampling of diamond drill core was carried out at GMV Minerals Inc's field office near Pearce, Arizona. Core was logged by the geologist into a hand paper log, and once the hole was complete, the log transposed into excel spreadsheets on a laptop computer. Once the drill core was logged the previous sample intervals were recorded and sample intervals were marked onto the core with a wax pencil and double checked.

Photograph 10-1: GMV Logging Facility

10.1.4 Recovery

Recoveries varied during the 2016 drilling with occasional voids or broken zones encountered where drilling advanced suddenly 0.1 to 0.3 m. It is the QP's (Dave Webb) opinion that this is not material at this time but should be considered in the future drilling.

10.1.5 Significant Drilling Results

Significant drillhole intercepts from the 2016 RC program are shown in Table 10-3. The intercepts are listed as downhole widths and have not been estimated as true widths.

Table 10-3: Significant Results from 2016 RC Drillholes

RC Hole	From	To	Length	Gold
GMV 2016-1	15.2	18.3	3	0.59
GMV 2016-1	54.9	61	6.1	0.29
GMV 2016-2	70.1	79.2	9.1	0.89
GMV 2016-2	112.8	115.8	3	0.23
GMV 2016-3	NSV			
GMV 2016-4	30.5	36.6	6.1	0.40
GMV 2016-5	64	67.1	3	0.34
GMV 2016-6	NSV			
GMV 2016-7	57.9	67.1	9.1	0.65
GMV 2016-7	103.6	106.7	3	0.21
GMV 2016-7	115.8	121.9	6.1	0.39
GMV 2016-7	137.2	140.2	3.0	0.23
GMV 2016-7	152.4	155.4	3	0.45
GMV 2016-7	167.6	179.8	12.2	0.20
GMV 2016-7	185.9	195.1	9.1	0.23
GMV 2016-7	222.5	225.6	3	0.25
GMV 2016-8	152.4	176.8	24.4	0.69
GMV 2016-8	152.4	173.7	21.3	0.74
GMV 2016-8	201.2	204.2	3	0.66
GMV 2016-8	210.3	213.4	3	0.21
GMV 2016-8	222.5	228.6	6.1	0.60
GMV 2016-8	268.2	277.4	9.1	0.34
GMV 2016-9	121.9	131.1	9.1	2.03
GMV 2016-9	146.3	149.4	3	0.37
GMV 2016-9	167.6	170.7	3	0.27
GMV 2016-9	192	195.1	3	0.53
GMV 2016-9	237.7	246.9	9.1	0.48
GMV 2016-10	NSV			
GMV 2016-11	9.1	82.3	73.2	0.60
GMV 2016-11	15.2	48.8	33.5	0.92
GMV 2016-11	164.6	167.6	3	0.20
GMV 2016-12	NSV			
GMV 2016-13	NSV			
GMV 2016-14	9.1	30.5	21.3	0.44
GMV 2016-14	73.2	79.2	6.1	0.21
GMV 2016-14	243.8	283.5	39.6	0.38
GMV 2016-14	243.8	265.2	21.3	0.56
GMV 2016-15	6.1	9.1	3	0.20
GMV 2016-15	12.2	15.2	3	0.20
GMV 2016-15	18.3	21.3	3	0.28
GMV 2016-15	51.8	57.9	6.1	0.23
GMV 2016-15	268.2	271.3	3	0.30

NSV = No significant values

10.2 GMV DRILLING – 2017

The 2017 drilling campaign saw the completion of 20 drillholes on the Mexican Hat Property. The particulars of the drilling campaign are presented below in the following subsections.

10.2.1   Collar Surveys

All holes were numbered in accordance to a nomenclature scheme which reflected the chosen drilling method for that particular hole. Diamond drillholes contained the prefix MHC, where as reverse circulation holes contained the prefix MHRC. Overall, holes were labeled using the following template "Prefix Identifier" dash "year of drilling" dash "the hole number". For example, MHC-17-02.

All holes were placed and oriented using field GPS and compass and checked upon completion for accuracy. Table 10-4 below summarizes the particulars of each hole completed during the 2017 program.

Table 10-4: 2017 Mexican Hat Drilling Collar Information

Hole_ID	Azimuth	Dip	East_m	North_m	Elev_m	td__m_	Drill_Type
MHC 17-1	180	-70	613420	3519727	1408.933	233.5	Core
MHC 17-2	180	-60	613381	3520074	1406.3	231	Core
MHC 17-3	199	-65	613383	3520073	1406.284	115.2	Core
MHC 17-4	180	-55	613699	3519874	1400.62	274.9	Core
MHC 17-5	180	-60	612710	3520018	1422.525	285.8	Core
MHC 17-6	180	-55	612563.2	3519401	1472.196	223.7	Core
MHC 17-7	150	-55	612563.2	3519401	1472.196	258.3	Core
MHC 17-8	160	-55	612895	3519566	1482.838	356.9	Core
MHRC 17-1	140	-54	613239	3519190	1433.05	213.4	RC
MHRC 17-10	0	-90	613218	3521528	1396.863	207.3	RC
MHRC 17-11	0	-90	613369	3521530	1394.367	304.8	RC
MHRC 17-12	0	-90	613520	3521531	1394.054	304.8	RC
MHRC 17-2	140	-70	613239	3519190	1433.05	305	RC
MHRC 17-3	160	-50	613239	3519190	1433.05	304.8	RC
MHRC 17-4	160	-50	613239	3519190	1433.05	304.8	RC
MHRC 17-5	140	-55	613159.8	3519109	1458.001	289.6	RC
MHRC 17-6	160	-70	613159.8	3519101	1461.976	317	RC
MHRC 17-7	90	-55	613498	3518596	1412.98	262.1	RC
MHRC 17-8	270	-55	613633	3518540	1411.789	201.2	RC
MHRC 17-9	270	-55	613671	3518497	1411.234	201.2	RC

10.2.2   Downhole Surveys

No downhole surveys were completed.

10.2.3   Logging

All core holes were delivered to the Company's facilities near the community of Sunsites where they were sorted, cleaned, logged for geology, alteration and structure, and RQD measurements. The core was marked, cut in half and sampled by company personnel, tagged and bagged along with blanks, and standards. Selected core was quartered to provide duplicate samples. RC samples were collected, and chip trays were developed for each interval. In addition, sludge boards were generated for all but four RC holes. These have been preserved in the Company's logging facilities.

10.2.4   Recovery

Recoveries for core drilling were generally good for the 2017 program, where an average of 99% recovery is estimated.

Recovery measurements were not collected for the 2016 / 2017 RC program; however, sufficient volume was collected for analysis for each drill run.

10.2.5   2017 Significant Drilling Results

Core holes MHC 17-1 to 5, did not intersect significant values. An exploration target, Hernandez Hill, was targeted by hole MHC17-1. Other drillholes MHC 17-6 to 8 intersected values as shown on Table 10-5, below. All core sampling obtained excellent recoveries with no significant lost core.

A summary of the significant values intersected during the RC and core drilling program is shown below in Table 10-5. Significant intersections from the 2017 drilling are summarized in Table 10-5. The intercepts are listed as downhole widths and have not been estimated as true widths.

Table 10-5: 2017 Significant Drill Core Intersections (Left), 2017 Significant Reverse Circulation Intersections (Right). All Intersection Lengths are Drill Lengths and Not True Widths

Drillhole	From	To	Length (m)	Gold (g/t)
MHC 17-6	6.20	7.50	1.20	0.25
MHC 17-6	6.20	11.00	4.70	0.12
MHC 17-6	41.10	42.80	1.70	0.14
MHC 17-6	54.90	63.70	6.90	0.48
MHC 17-7	19.80	25.00	5.20	0.16
MHC 17-7	103.60	105.60	2.00	1.30
MHC 17-7	116.30	119.50	3.20	0.25
MHC 17-8	199.50	200.30	0.80	0.12
MHC 17-8	245.10	246.00	0.90	0.13
MHC 17-8	284.80	288.40	3.60	0.24
MHC 17-8	294.30	295.30	1.00	0.26
MHC 17-8	297.90	327.50	5.00	0.78
MHC 17-8	348.10	354.60	6.50	0.27

Drillhole	From	To	Length	Gold (g/t)
MHRC 17-1	18.30	27.40	9.10	0.80
MHRC 17-1	70.10	73.20	3.10	0.19
MHRC 17-2	42.70	48.80	6.10	0.46
MHRC 17-2	54.90	57.90	3.00	0.23
MHRC 17-2	67.10	73.20	6.10	0.24
MHRC 17-2	219.50	225.60	6.10	0.52
MHRC 17-3	24.40	42.70	18.30	0.57
MHRC 17-3	57.90	70.10	12.20	1.37
MHRC 17-3	225.60	234.70	9.10	0.20
MHRC 17-3	283.50	304.80	21.30	0.39
MHRC 17-4	3.00	12.20	9.20	0.23
MHRC 17-4	45.70	48.80	3.10	0.27
MHRC 17-4	143.30	146.30	3.00	0.28
MHRC 17-5	76.20	88.40	12.20	0.52
MHRC 17-5	143.30	146.30	3.00	0.46
MHRC 17-5	204.20	210.30	6.10	0.45
MHRC 17-6	106.70	112.80	6.10	0.25
MHRC 17-6	125.00	143.30	18.30	2.07
MHRC 17-6	182.90	201.20	18.30	0.73
MHRC 17-7	42.70	45.70	3.00	0.23
MHRC 17-7	54.90	61.00	6.10	0.32
MHRC 17-7	82.30	85.30	3.00	0.31
MHRC 17-8	54.90	57.90	3.00	0.58
MHRC 17-9	91.40	94.50	3.10	0.30

11   SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1   2016 AND 2017 SAMPLING METHODS

11.1.1   Sample Collection

The 2016 and 2017 drill programs were completed under the direct supervision of D.R. Webb or B.T. Malahoff, both of DRW Geological. All samples remained in the possession of either D.R. Webb or B.T. Malahoff until they were shipped by bonded courier to Bureau Veritas Laboratories' preparation facilities in Elko, Nevada. Prepped samples were then sent by the laboratory to Bureau Veritas Commodities Canada Ltd. (Formerly Acme Analytical Laboratories Ltd.) in Vancouver, BC for analyses.

11.1.2   Sampling

Core and RC sampling were handled as described in Section 10: Drilling.

Logging and sampling of diamond drill core was carried out at GMV Minerals Inc.'s field office near Pearce, Arizona. Core was logged by hand into spreadsheets, then transposed into excel spreadsheets on a laptop computer. Once the drill core was logged the previous sample intervals were recorded and sample intervals were marked onto the core with a wax pencil and double checked. For quality control, the sample sequence required the insertion of standards, blanks and duplicates nominally at 10 sample intervals for blank/standards and 30 sample intervals for duplicates. The sample sequence, in general, was left to the discretion of the geologist. In this way, standard and/or blanks could be inserted within a sequence of visibly mineralized material to ensure sample prep/analysis quality.

Drill core was first photographed and then halved using a diamond core saw. Half-core was taken from each sample interval and placed into pre-marked plastic sample bags containing the corresponding sample ticket and secured with a plastic cable tie. All samples comprised half-core except for designated duplicate intervals where the core had been quartered and a quarter of core went into each of two bags.

Soil samples were collected and bagged in the field in standard kraft paper bags with GPS control and slope, sample colour and matrix were recorded. Samples were consolidated in GMV's locked facilities until sufficient material was at hand for shipping to Bureau Veritas's laboratory in Nevada by commercial shipper.

Photograph 11-1: Samples Awaiting Transport for Analysis at GMV Sample Processing Facility

11.1.3   Sample Preparation

All drill core samples were labeled, bagged and stored in a locked facility under the control of Webb and Malahoff. Approximately once per week the samples were shipped to Bureau Veritas' sample preparation facilities in Elko, Nevada where they were received and prepared (crushed, split and pulverized) (PRP 70-250). The prepared samples were then sent to the Bureau Veritas Commodities Canada Ltd. in Vancouver, B.C. for analyses.

All reverse circulation drilling was sampled in 10' (3.05 m) intervals. The discharge stream was run through a rotary sampler and placed into pre-marked bags by the drilling company under the supervision of a geologist. The sampler was cleaned and flushed after each sample.

The bagged samples were brought to the logging facility and laid out to dry for several days prior to consolidating with blanks and standards into ~20 kg rice bags and labeled for shipping. Samples were held in a locked storage until sufficient material was at hand for shipping to Bureau Veritas's laboratory in Nevada by commercial shipper.

On receipt at the laboratory, the drill samples were dried, crushed to 1kg ≥ 70% passing 2 mm, and then riffle split in accordance with PRP70-250 (Inspectorate), an aliquot was then separated by riffle where 250 grams were then pulverized to ≥ 85% μm according to PRP70-250 (Inspectorate).

11.1.4   Analytical Methodology

Samples were analyzed for 37 elements on 30 gm samples digested in aqua regia and analyzed using ICP mass spectrometer techniques (Bureau Veritas Commodities Canada Ltd., AQ 252). All samples returning gold values >0.1

g/t were re-analyzed using 30 gm fire assay with atomic adsorption techniques (Bureau Veritas Commodities Canada Ltd., FA330). Bureau Veritas Commodities Canada Ltd. have a "Quality Management System" and accreditation (ISO 9001: 2008 accredited). Bureau Veritas Commodities Canada Ltd. and Inspectorate America Corporation preparation facilities are independent of GMV Minerals Inc. and were contracted to provide sample preparation and geochemical assays for drill core and RC samples.

Soil samples were dried, screened to -80 mesh and analyzed for 37 elements on 30 gm samples digested in aqua regia and analyzed using ICP mass spectrometer techniques (Bureau Veritas Commodities Canada Ltd., AQ 250). Bureau Veritas Commodities Canada Ltd. have a "Quality Management System" and accreditation (ISO 9001: 2008 accredited). Bureau Veritas Commodities Canada Ltd. and Inspectorate America Corporation preparation facilities are independent of GMV Minerals Inc. and were contracted to provide sample preparation and geochemical assays for soil samples.

11.2 QUALITY CONTROL OF LABORATORY ANALYSIS

QA/QC included the insertion and continual monitoring of numerous standards and blanks into the sample stream at a frequency of 1 per 10 samples. Duplicates were taken on approximately every 20th sample and processed as either course reject or pulp duplicates.

Fourteen drill logs from 2016 and 2017 were cross-referenced with assay certificates containing geochemical data from ALS Laboratories on samples, standards, blanks and duplicates.

11.2.1 Certified Reference Materials

Two certified reference materials (CRM) were used to check for analytical accuracy, and one analytical blank was used to check for potential contamination during sample preparation.

Samples of the CDN-GS-1M and CDN-GS-P5C, prepared by CDN Resource Laboratories Ltd. in Langley, BC Canada, were used as the CRM's. The recommended value and 'between laboratory' two standard deviations for the CDN-GS-P5C and CDN-GS-1M reference materials are 0.571 g/t ± 0.048 g/t Au, and 1.07 ± 0.09 g/t Au, respectively.

One analytical blank, CDN-BL-10, also from CDN Laboratories was used as the analytical blank. The prepared samples had certified gold concentration of <0.01 g/t.

The number of analyses completed for each CRM, by method, and blank material is outlined in Table 11-1. Discussion of the QAQC control program is included below.

Table 11-1: Number of Analyses of Gold Completed for each CRM and Blank

Standard or Blank	No. of Analyses
CDN-BL-10	119
CDN-GS-IM (ICP)	52
CDN-GS-IM (Fire Assay)	95
CDN-GS-P5C (ICP)	55
CDN-GS-P5C (Fire Assay)	90
Unknown, unconfirmed standard/blank	25

11.2.1.1   CDN GS-P5C

In total, 55 CRM standards with expected values of 0.571 g/t gold were submitted into the sample stream and assayed via ICP analysis, along with an additional 90 CRM standards which tested the laboratories fire assay accuracy. No ICP analyzed standards failed outside of two standard deviations (Figure 11-1), with only 1 fire assay analyzed standard failing outside of two standard deviations (Figure 11-2). The one standard which failed did not fall outside of three standard deviations. Additionally, no appreciable assay bias for the 0.571 g/t standard was detected for either ICP or fire assay analytical methods.
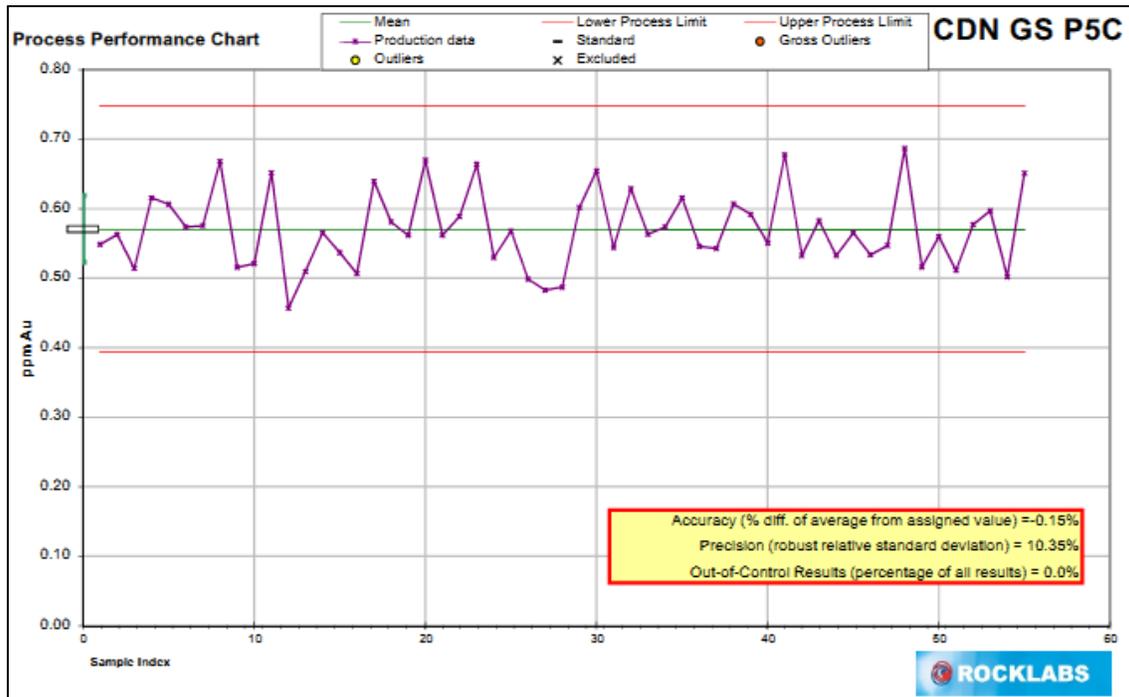
Figure 11-1: Certified Standard CDN GS-P5C, ICP Results
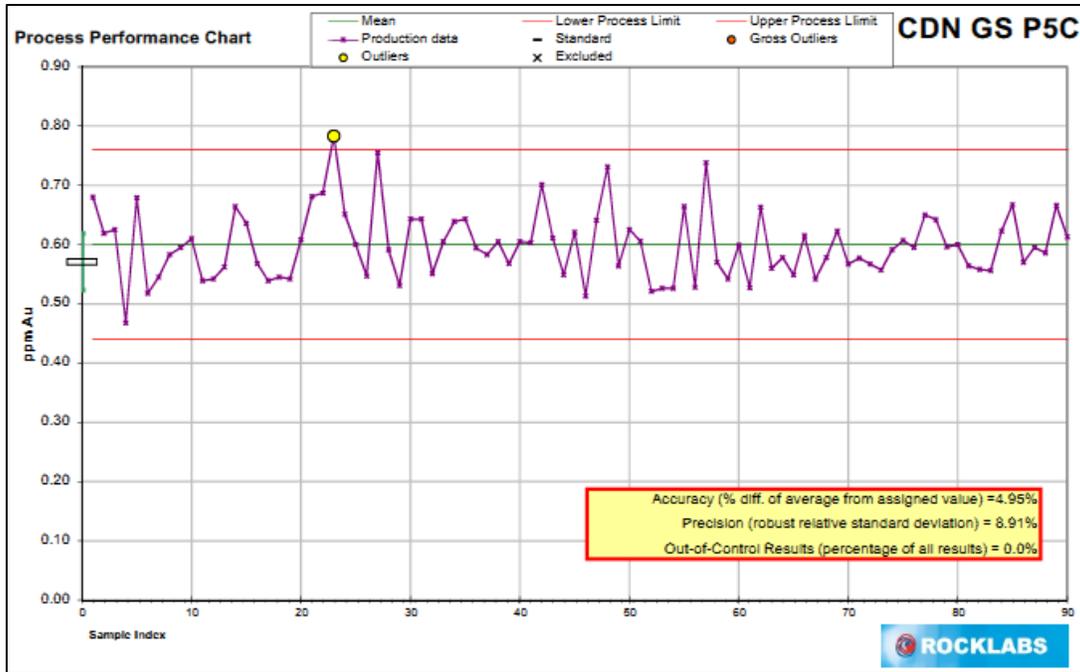
Figure 11-2: Certified Standard CDN GS P5C, fire Assay Results

11.2.1.2   CDN GS-1M

In total, 52 CRM standards with expected values of 1.07 g/t gold were submitted into the sample stream and assayed via ICP analysis, along with an additional 95 CRM standards which tested the laboratories fire assay accuracy. Overall, 1 ICP analyzed standard failed outside of two standard deviation accepted limit (Figure 11-3), along with 1 fire assay analyzed standard failing outside of two standard deviations (Figure 11-4). Neither of the samples which failed fell outside of three standard deviations. No appreciable assay bias was detected for either ICP or fire assay analytical methods.
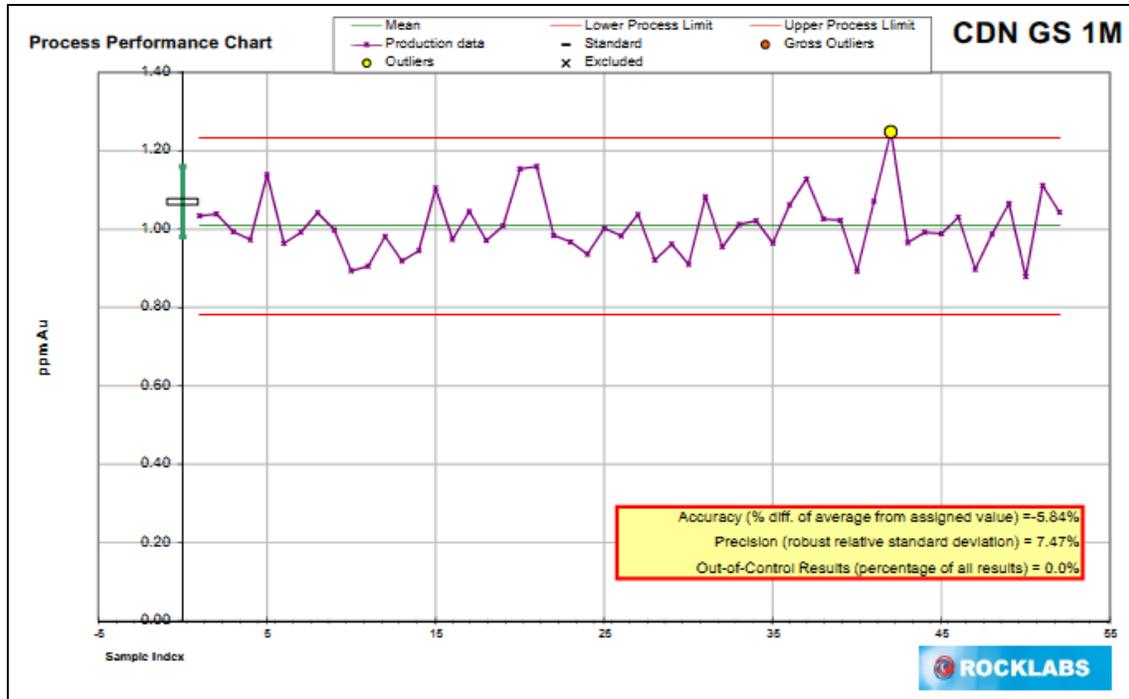
Figure 11-3: Certified Standard CDN GS-1M, ICP Results
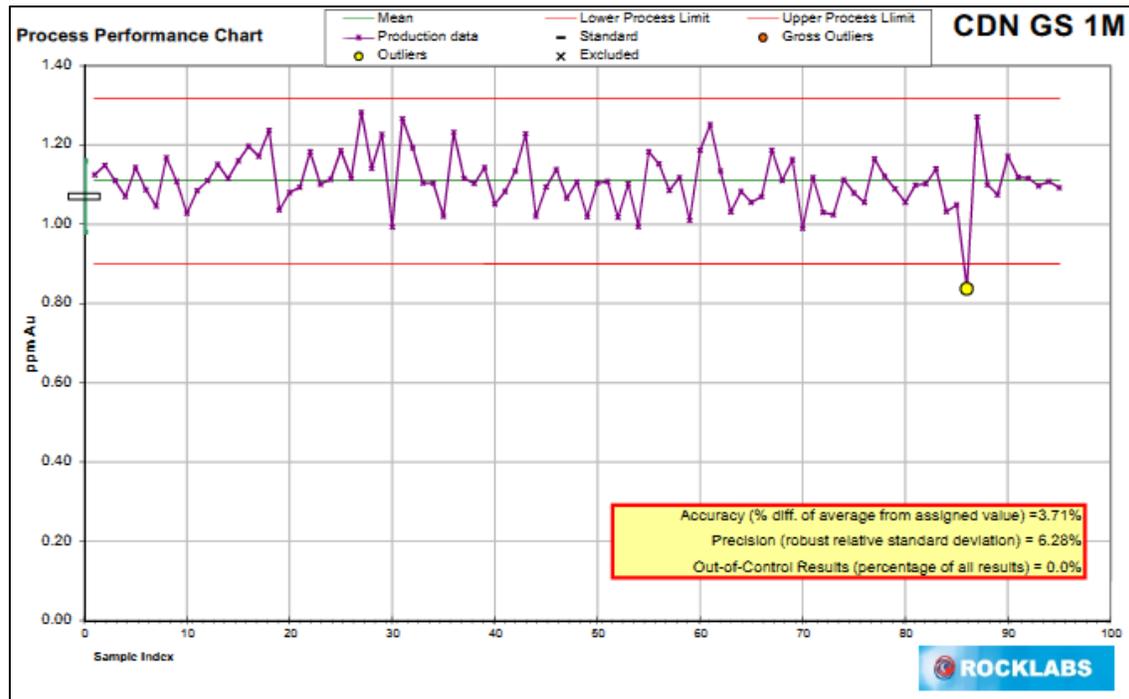
Figure 11-4: Certified Standard CDN GS 1M, Fire Assay Results

11.2.1.3   Analytical Blank

CDN Laboratory provided CDN BL-10, a certified blank material for gold that returned seven (7) samples above the declared value or <0.01 g/t Au (Figure 11-5). With the exception of one sample, all were less than 0.03 g/t Au. This is considered acceptable for this program.
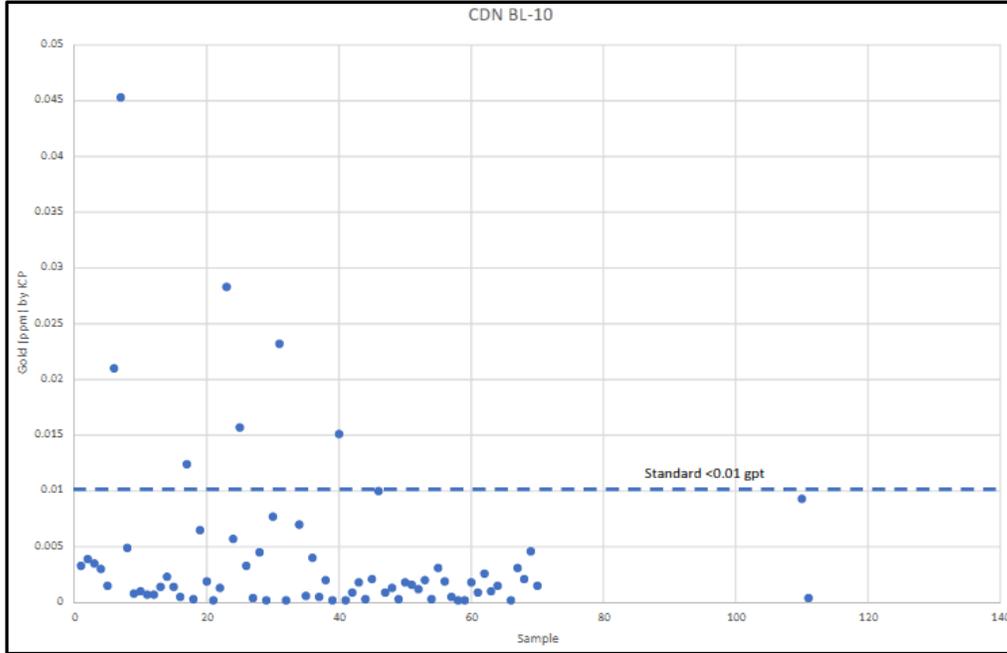
Figure 11-5: Plot of all CDN BL-10 ICP Results

Fire assays gave better results as shown on Figure 11-6 below, with only three sample above the certified value, and these are all below 0.015 g/t Au.
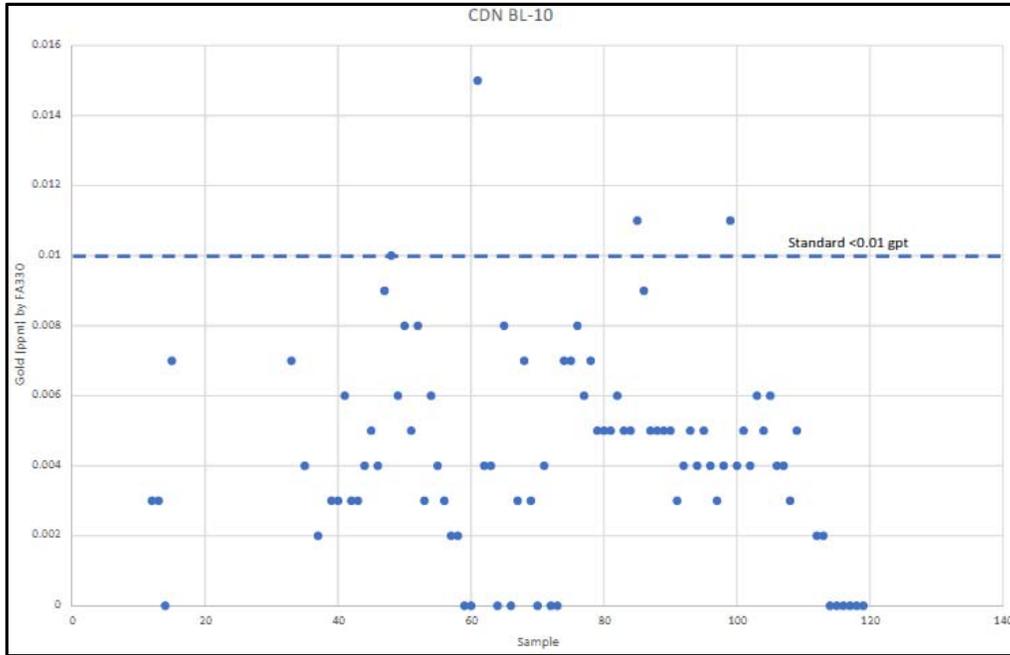
Figure 11-6: Plot of all CDN BL-10 FAA Results

11.2.2 Coarse Reject Duplicates

A total of 31 coarse reject duplicates were collected randomly from drill core. These samples were analyzed and compared to the original assay grade to assess the precision of analytical methods.

Generally, a higher margin of error is expected from preparation (coarse reject) duplicate in comparison to analytical (pulp) duplicates, where it is desired to measure 90% of the sample population above detection limit to have less than 20% relative percent difference (RPD).

On average, the coarse reject duplicate data reported an RPD of 12.94% for gold, with 7 of the 31 samples (22.5% of population) falling outside of the 20% RPD threshold (Table 11-2 below). Of these 7 failures, none of the failures occurred within samples greater than the resource cut-off grade of 0.2 g/t gold. It is noted that only 3 of 31 coarse reject duplicate samples are above the resource cut-off grade of 0.2 g/t gold.

Table 11-2: Raw Coarse Reject Duplicate Data showing Relative Percent Difference (RPD), Au Grades Shown in ppb (1 g/t=1000ppb)

Sample Number	Primary Au (ppb)	Duplicate Au (ppb)	%RPD	Sample Number	Primary Au (ppb)	Duplicate Au (ppb)	%RPD
195725	324.00	367.00	12.45	195979	48.00	46.00	4.26
195793	982.00	922.00	6.30	202519	26.00	21.00	21.28
196731	52.00	49.00	5.94	202565	76.00	64.00	17.14
196775	6.00	6.00	0.00	202604	321.00	308.00	4.13
196860	3.00	2.00	40.00	202775	92.00	77.00	17.75
196894	22.00	16.00	31.58	202673	14.00	15.00	6.90
196604	12.90	12.00	7.23	202726	1.00	1.00	0.00
196664	6.70	6.40	4.58	202879	25.00	21.00	17.39
195504	8.00	5.00	46.15	195808	77.00	83.70	8.34
195550	9.00	9.00	0.00	195858	28.30	21.90	25.50
196926	10.00	11.00	9.52	84641	17.70	31.00	54.62
195579	5.00	5.00	0.00	82936	1.00	1.00	0.00
195695	31.00	35.00	12.12	82970	1.00	1.00	0.00
196975	3.00	3.00	0.00	82454	1.00	1.00	0.00
195940	26.00	35.00	29.51	202621	7.00	6.00	15.38
				195852	33.00	34.00	2.99

Figure 11-7 below shows that none of the coarse reject failures occurred at values greater than 0.2 g/t (200ppb). Additionally, the duplicate values and their associated failures are shown at greater resolution in Figure 11-8.
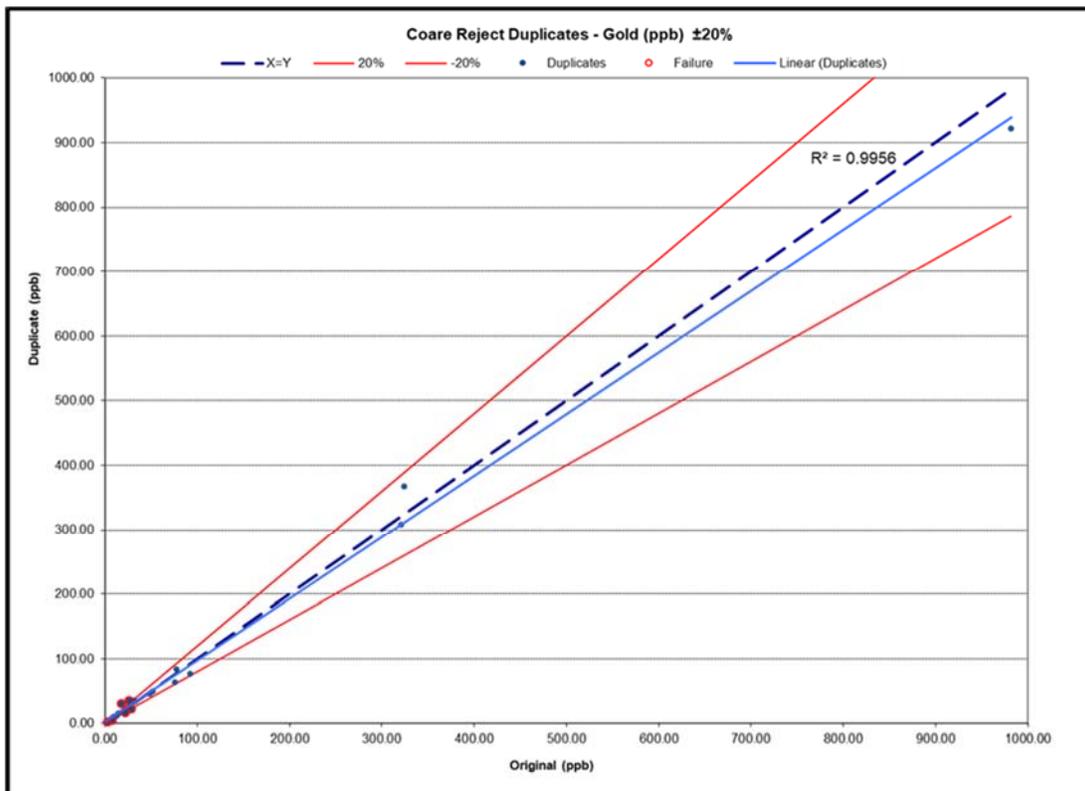
Figure 11-7: Coarse Reject Original Vs Duplicate Assays
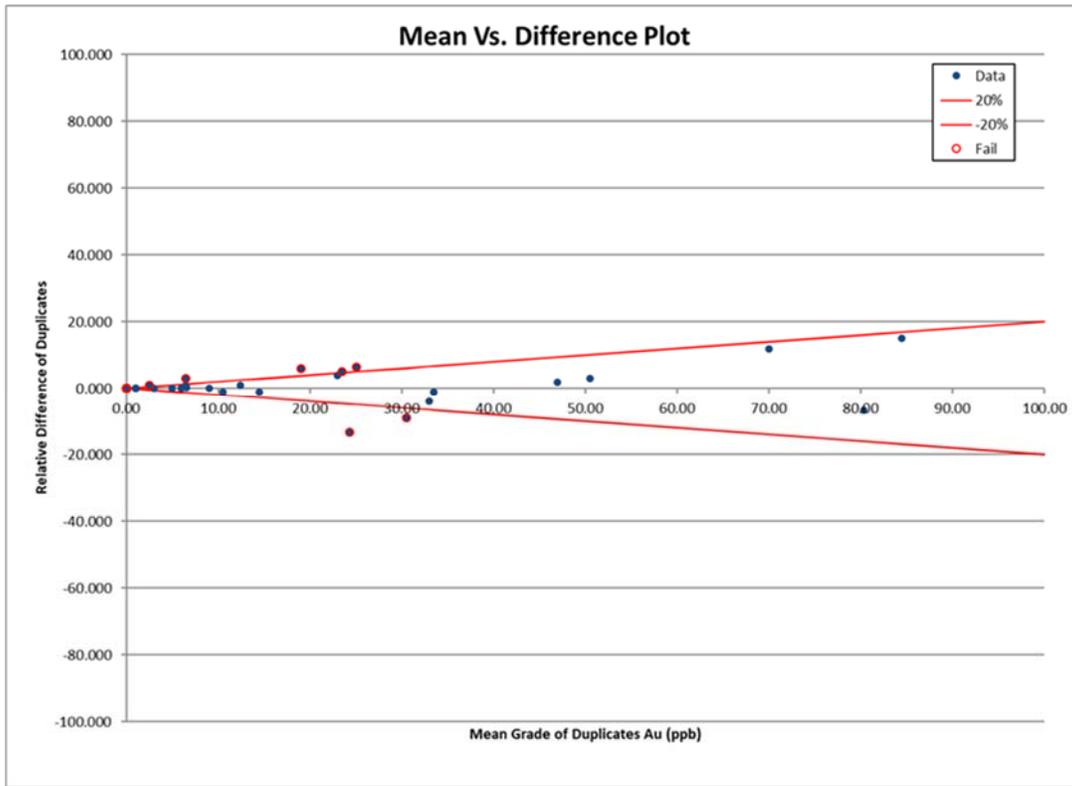
Figure 11-8: Mean Vs Duplicate Data for Au <100ppb

11.2.3   Pulp Duplicates

A total of 43 pulp duplicates were assayed during the 2016, 2017 field program. For pulp duplicates its desired that 90% of the of the duplicated samples have less than a 10% RPD when compared to the original assay value.

Overall, of the 43 pulp duplicates which were submitted, 12 of the samples (27.9% of population) failed outside of the 10% accepted value threshold. When this failure rate is further investigated (Table 11-3), it can be observed that the majority of these samples (10 of the 12 samples) contained original assay values of less than 0.2 g/t (200ppb) which is below the cut-off grade of the resource estimate, and therefore, non-material failures (Figure 11-9). It is noted that only 5 of 43 pulp duplicate samples are above the resource cut-off grade of 0.2 g/t gold.

Table 11-3: Raw Pulp Duplicate Data showing Relative Percent Difference (RPD)

Sample Number	Primary Au (ppb)	Duplicate Au (ppb)	%RPD	Sample Number	Primary Au (ppb)	Duplicate Au (ppb)	%RPD
195715	18	18	0	202504	92	95	3.21
195785	22	21	4.65	202593	254	283	10.8
196772	5	5	0	202505	80	73	9.15
196726	43	42	2.35	202660	28	26	7.41
196894	22	22	0	202784	34	37	8.45
196800	643	549	15.77	202630	4	9	76.92
196857	6	6	0	202719	1	1	0
196801	590	574	2.75	202780	56	51	9.35
196616	4.1	5.2	23.66	202880	199	201	1
196551	9.8	10.5	6.9	202680	13	14	7.41
196676	4.6	2.5	59.15	202837	2850	2992	4.86
196990	6	6	0	202674	8	18	76.92
196915	18	23	24.39	195903	20.5	25.6	22.13
195579	5	5	0	195840	12.4	13.5	8.49
195697	7	7	0	82953	1	1	0
196973	639	687	7.24	82993	1	1	0
195715	18	18	0	202863	43	33	26.32
195785	22	21	4.65	202526	27	26	3.77
196772	5	5	0	195851	28	28	0
196726	43	42	2.35	82986	1	1	0
195957	124	105	16.59	202658	17	16	6.06
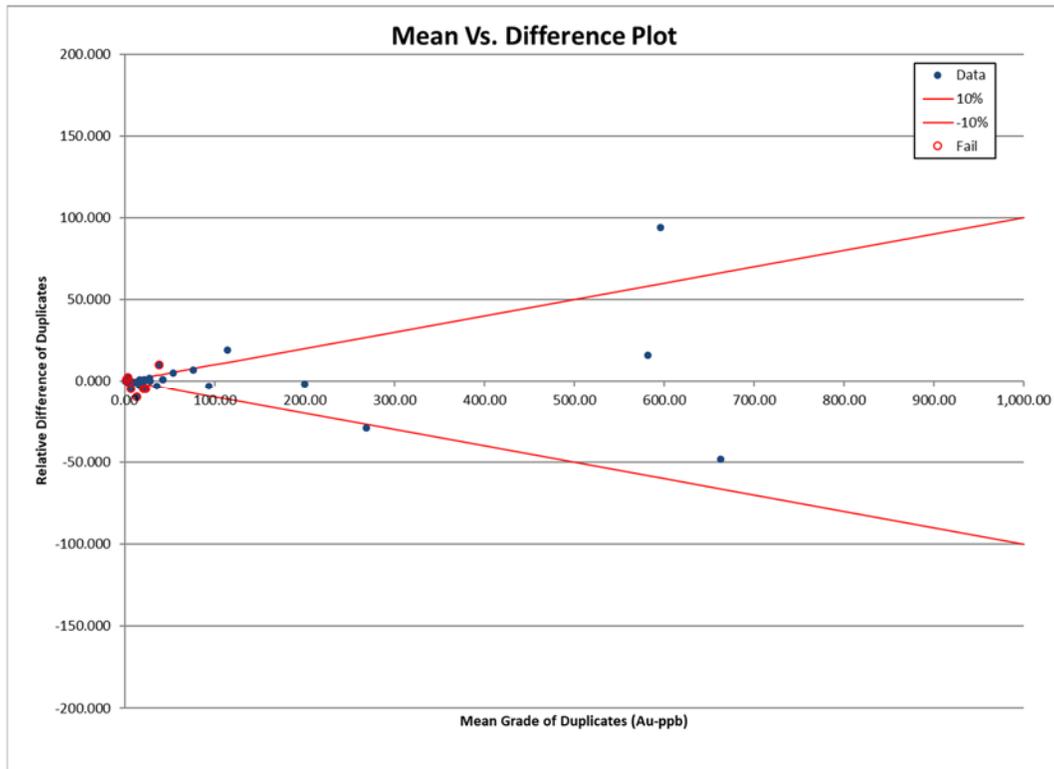
Figure 11-9: Pulp Duplicates Mean vs Difference. Samples with >10% RPD Circled in red (Gold grades presented in ppb)

11.3   QP OPINION ON SAMPLE PREPARATION, ANALYSIS AND SECURITY

It is the QP's (Dave Webb) opinion that all samples (soil, RC and drill core samples) were adequately prepared, adequately security was provided, and adequate analytical procedures were followed.

The sampling was conducted according to industry standard practice which benefitted from insertion of certified blank and reference standards, and collection of duplicate samples at the laboratory preparation and analytical stages which is considered adequate for this phase of work.

The standards have all returned adequate values with fire assays providing for acceptable results. Geochemical methods are also acceptable.

12   DATA VERIFICATION

12.1   TETRA TECH QP INDEPENDENT VERIFICATION

12.1.1   Database Audit

The Tetra Tech QP conducted a review and audit of the Mexican Hat geological database provided by GMV. The outcomes of the audit are listed below.

Data comprising the 2016 field program and previous Property investigations were provided to Tetra Tech in advance of the Tetra Tech QP site visit. The data included reports from historical operators, but no drill logs, assay certificates or sample material was available to be verified by the QP. The data collected during the 2017 field season was provided to Tetra Tech in late 2017 for review.

12.1.1.1   Collar Locations

Drill collar and trench locations were examined during the site visit in July 2017.

Coordinates for the GMV collars were collected using a handheld GPS unit and recorded in UTM (NAD83), the same datum used in the resource database. Numerous marked collars from the GMV drilling and trenching campaigns were identified in the field, with reasonable location error using handheld GPS.

Historical collar locations were unmarked in the field and could not be verified.

12.1.1.2   Property Digital Elevation Model

All collar locations were plotted against a new georeferenced aerial photograph and digital elevation model (DEM) to investigate their plotted locations in comparison to the visual pad locations present on the aerial photographs. This study showed many of the 2016 and 2017 collar locations reliably plotted, however, many of the historical drill collars locations (1989 and 1990) did not visually correspond to a drill pad, with many holes appearing to have undergone a co-ordinate shift. Tetra Tech discussed this discrepancy with GMV, and it was agreed that drill collars coordinates reported in the database appeared to have shifted. GMV reviewed the drill database and in conjunction with Cooper Aerial Surveys were able to correct all the collars in question to more accurately reflect the spatial location on the ground.

The identification of the shift in collar locations in a relative small sample set would suggest that additional verification work in the form of a complete collar survey using a professional differential GPS would increase the confidence of the drillhole locations. Additionally, all future holes should have surveyed in with a differential GPS upon completion of the hole. Tetra Tech QP recommends this to be undertaken prior to updating the resources in the future.

12.1.1.3   Collar Elevations

Collar elevations in the database were applied based on the DEM surface elevation following adjustments made to the collar locations discussed above.

12.1.1.4   Drillhole Orientation Surveys

Drillhole orientation surveys listed in the drillhole database were compared with original drill log records and where discrepancies existed were discussed with GMV. The drillhole database was updated to include the confirmed drillhole orientation surveys.

Channel samples are included as continuous "trench' samples in the database, with a surveyed collar origin and projected 'down trench' orientation based on surface mapping compass bearing. Often, the projected trench does not correspond with elevation and contour of the DEM topography. The samples were kept in the database, but required special treatment for mineral resource estimation, as discussed in Section 13.6.2.

No drillhole orientation surveys are reported in the database for historical drilling.

12.1.1.5   Drillhole Intervals

The downhole interval data which including assay results and lithology quick logs were reviewed for error such as overlapping intervals, interval gaps and logged depths exceeding the total drillhole depth. Discrepancies were flagged and compared with original drill logs. The discrepancies were corrected within the drillhole database.

Historical drillhole data is only available for gold assay. Data for lithology, geochemistry or other parameters are not available for historical drilling.

12.1.1.6   Translation of Ounce per Tonne to Gram per Tonne

Prior to GMV work, samples were assayed for gold and recorded in units of ounces per ton. Consequently, during the conversion of the historical assay database into units of grams per tonne to be consistent with the GMV assays, a larger population of samples showing assay values of 0.07 g/t exist in the data base. This population was introduced from samples which were at the detection limit of 0.002 ounces per ton of the analytical procedure. Given that the grade of gold is less than the cut-off that the resource is reported at, these samples do not have the ability to skew the final estimate.

12.1.1.7   Review of QAQC Data

A review of the QAQC program that was implemented by GMV was undertaken, which included a review of the 2017 drill logs, review of the QAQC database and review of the analytical certificates provided by Bureau Veritas.

Some drill logs referenced CRM standards and blanks clearly, such as in drill log GMV-2016-1, 2016-2, and 2016-13. However, the majority of drill logs required assumptions to be made for when and which CRM standard or blank were submitted to the lab. In this case, by assumption, the standard, GSP5C (Au = 531-755 ppm) or GS1M (Au = 993-1266 ppm), or blank (Au ~ 2-6 ppm) was designated by its range of known Au value.

Drill logs occasionally were missing sample numbers in the numbering sequence, without indication of what the missing sample number represented. In this case, a sample number was noted in a QAQC log, but without a standard or blank name. Many of these samples were not found in any assay certificate (drill logs 2017-9, 2017-10, and 2017-11). Numerous standards and blanks had measured values returned as I.S. (insufficient sample) in assay certificate REN17000021, representing QAQC in drill log GMV-2016-3 and 2016-4. In some instances, sample numbers could not be located in any assay certificates, thus could not be cross-referenced to assay data (GMV-2016-3, 2016-14, 2017-7, 2017-8, 2017-9, 2017-10, and 2017-11).

12.1.2   Site Visit

The Tetra Tech QP, James Barr, P.Geo, visited the Mexican Hat Project between July 18 and 19, 2017 to observe the ground conditions, nature of mineralization, and to collect rock samples for independent verification of the mineralization. During the site visit, Mr. Barr was accompanied by Mr. Dave Webb, Ph.D., P.Eng. and conducted meetings with Mr. Brian Malahoff, P.Geo. The results of the independent verification are presented below in the following subsections.

12.1.2.1   Verification Samples

Three representative rock samples were collected from outcrop on the Property to confirm gold mineralization and test the various lithological hosts of mineralization. The first rock sample collected, TtMH-001 was a Latite / Andesite tuff within the iron-rich alteration halo of a fracture/structure. The second and third samples, TtMH-002 and TtMH-003 were both Latite / Andesite and were collected from a structure which is interpreted to host gold mineralization. TtMH-004 represents a QAQC standard (CDN ME-7, Au = 0.219+/- 0.024) which was inserted into the data validation samples for quality control purposes.

The samples were sent to the ALS Minerals laboratory in North Vancouver. Samples were crushed to 70% passing 2 mm and 1,000 g sub-samples pulverized to 85% passing 75 µm. The samples were analyzed using package ALS labs CCP-PKG03 which provides a complete sample characterization by combining whole rock analysis, aqua regia digestion for the volatile trace elements along with gold. The package includes trace elements analysis using aqua regia digestion and ICP-MS (ME MS42, not Au), four acid digestion and ICP-MS (Au, Ag, Zn, and Pb only, OG62), whole rock lithogeochemistry (ME-MS81 and ME-XRF26), and 30 g Au fire assay and atomic absorption (Au-AA23). Samples TtMH-001 and -002 were also submitted for a 500g bottle roll (bulk leachable) over 12 hours in sodium cyanide and analysis with ICP-MS (Au CN-11: for Au, Ag and Cu) to further test the sample for cyanide solubility of gold.

Mineralized samples from the 2017 drilling program were not available on site at the time of the site visit while being stored at the Bureau Veritas laboratory.

Table 12-1 below summarizes the results of the reconnaissance grab sample validation. The sampling confirms gold mineralization on the property related to structural control. These samples collected from within the weathered surface outcrops show high proportion of gold solubility in cyanide relative to the near total fire assay concentrations. The CRM standard performance was within accepted range and was considered valid.

Table 12-1: Analytical Results of Au, Ag and Cu Grades from Tetra Tech QP Independent Sampling

Sample	Au (Fire Assay, AA23) g/t	Au (CN-11) g/t	Ag (4-Acid, AAS) g/t	Ag (CN-11) g/t	Cu (4-Acid, AAS) ppm	Cu (CN-11) ppm
TtMH-001	0.16	0.16	2.2	0.56	21	0.45
TtMH-002	8.04	7.99	6.7	1.67	39	0.91
TtMH-003	1.42	n/a	1.4	n/a	28	n/a
TtMH-004 (CDN-ME-7)	0.238	n/a	>100	n/a	2430	n/a

12.2   QP OPINION ON DATA VERIFICATION

The Tetra Tech QP has conducted a review of the project database, has compared analytical certificates with reported assay results for drill core and rock samples, has visited the Property and collected mineralized samples from the Property. It is the QP's opinion that the data reported for the Project can be verified and is acceptable for mineral resource estimation. Results of the database audit have been considered in the classification of the Mineral Resource Estimate.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

Two metallurgical test programmes were carried out on mineralization from the Mexican Hat Project. These programmes were supervised by John Fox, P.Eng who reviewed test conditions and results at the respective laboratories. The first stage programme carried out by McClelland Lab of Sparks, NV should be viewed as a scoping study. Following this, a second programme was carried out at the Bureau Veritas of Richmond, BC.

Preliminary geological assessment identified 4 types of mineralization. Latite about 80% of the resource with Andesite and Basalt contributing about 8% each and Dacite 4%. As a result, initial testwork will focus on Latite.

13.1 MCCLELLAND TESTWORK

Two bulk samples were sent to the McClelland lab for testwork. These were taken from Trench 3 and Trench 12 and assayed 1.58 g/t Au and 1.36 g/t Au respectively. Silver was 1.4 and 2.0 g/t respectively as follows.

Table 13-1: McClelland Benchscale Testwork Summary

Sample	Leach time Days	Size P80	Au % Rec.	Au g/t Tail	Au g/t Calc Hd	Ag % Rec.	Ag g/t Calc Hd	NaCN kg/t	Lime Kg/t
Tr-3	98	38mm	95%	0.07	1.39	4.8	1.4	0.92	1.4
Tr-3	4	1.7mm	84.9	0.24	1.59	23.1	1.3	0.10	1.5
Tr-3	3	0.075mm	96.9	0.05	1.60	47.4	<1.9	0.07	1.8
Tr-12	98	38mm	77.1	0.30	1.31	3.2	3.1	0.96	1.6
Tr-12	4	1.705mm	82.2	0.26	1.36	11.5	2.6	<0.07	1.8
Tr-12	3	0.075mm	97.6	0.05.	1.25	38.1	2.1	0.16	2.1

The high recovery after the 75-micron grind indicates that there are no refractory issues related to the samples tested and this is supported by the low cyanide consumption. The 1.7 mm test data suggests at this size a longer leach time would have been beneficial because at the even coarser crush of 38 mm gave 95% recovery on the trench 3 sample, although only 77% on the trench 12 sample. The increase in cyanide consumption in the column tests is almost certainly due to the long leach time at relatively high cyanide strength used in these tests. Looking at the leach residue, Trench 3 material does not show any very marked variation trend related to size, but the Trench 12 material does with a steady rise in recovery from the 37 mm size fraction of 61.5% to approximately 90% at 12 mm, with everything less than 12 mm having over 90% recovery and over 95% for everything finer than 1.7 mm.

Silver, in the samples, was quite low and did not leach very well. As it contributes so little to the value of the mineralization, no time or effort was made to optimize the recovery of silver.

13.2 BUREAU VERITAS TESTWORK

The good results from the McClelland column leach testwork suggested that a Run-of Mine leach may be viable. In practical terms, a true run of mine leach is not practical to run at laboratory scale, but it was considered that a column run on nominal 150 mm material could approximate conditions likely to be encountered in a Run of Mine heap at a manageable scale. A bulk sample was produced at Mexican Hat and shipped to Richmond. The total of approximately 18 tonnes of material was packed into 24 bulk bags and each bag enclosed in a wooden crate. This material was received at the Richmond laboratory in good condition with all boxes and bags intact. In addition, some smaller samples from the drilling programme were procured for bottle roll tests. These samples represented Rhyolite, from 13 m from surface, Trachybasalt from 122 m, and a sub sample of the bulk sample (latite) taken from surface. These samples were submitted to bottle roll tests with the following results:

Table 13-2: BV Lab Benchscale Testwork Summary

Sample	Leach Time hr.	Size P$_{80}$ Micron	Au % Rec.	Au g/t Tail	Au g/t Calc Hd		Au g/t Assay Hd	NaCN kg/t	Lime Kg/t
Bulk	48	74	96.4	0.02	0.56		0.45	0.43	0.99
Bulk	96	1961	82,9	0.10	0.56		0.45	1.03	0.88
Rhyolite	48	69	98.2	0.02	0.85		0.66	0.71	1.04
Rhyolite	96	1671	80.9	0.16	0.84		0.66	1.29	0.59
Basalt	48	97	96.9	0.03	0.97		0.74	1.12	2.08
Basalt	96	2600	95.0	0.05.	0.89		0.74	2.47	1.82

These results indicate that all the samples tested leach very well when ground and are similar to other samples tested, however reagent consumption is highly variable.

The 1 meter diameter by 6m tall column was loaded with the bulk sample material broken to a nominal 150 mm passing size. Lime water was passed through the column until the exit pH was over 10 and then 0.5kg/m^3 sodium cyanide leach solution was applied to the column. The pregnant leach solution (PLS) was passed through a carbon column before being adjusted and returned to the column as leach solution. The progress of the leach was monitored based on the head calculated from a sample of feed material that was screened and assayed by size fraction. After 40 days, the leaching stalled at an estimated 27% leach extraction. The column was drained down and leaching restarted a few days later. Leaching reached about 35% recovery by day 100 and continued until day 137 when recovery was about 57% and further leaching appeared to have stopped. The test was terminated. After draining and rinsing, the column was unloaded, and a sample was taken and screened by size fraction and the individual size fractions assayed (as had previously been done with the sample before loading into the column).

It became apparent that the coarser sizes were difficult to sample and get a representative sample. The 100 kg sub-sample had only one rock in the coarsest size range (150 mm) representing the 8 tonnes loaded into the column.

Analysis of the leach residue showed a general trend to lower residue grade the finer the material with the very finest size being aberrant. The results as first presented had negative recoveries for the coarser size fractions (over 75 mm) owing to the difficulties of taking representative samples at these coarse sizes. A separate study looked at individual rocks randomly taken from the residue (and feed) and in the 150 mm residue size the 5 rocks averaged 0.17 g/t with the range from 0.02-0.52 for individual rocks. When all the data was analysed, leaching on the minus 25 mm material left a residue below 0.1 g/t Au (average 0.045g/t) which would represent about 85% recovery. The aberrant value in the finest size fraction (0,17 g/t Au in the minus 80 mesh (177 microns)) is sometimes seen and may indicate leached gold has reprecipitated under adverse ORP conditions.

A sample of the 80 mesh fine residue from the column was taken and bottle roll leached. It gave over 85% leach recovery, and a residue of 0.03 g/t indicating that under the right conditions it would easily leach and was not inherently refractory.

The detailed results have been reviewed with a target crush close to 25 mm yield optimum leach results. A relatively economical and practical crushing circuit targeting close to 25 mm crush size would have atop size of about 30 mm and would yield a residue of 0.055g/t after a 100 day leach, equivalent to a 91.9% leach extraction on average grade ore.

14 MINERAL RESOURCE ESTIMATES

14.1 BASIS OF CURRENT RESOURCE ESTIMATE

This mineral resource estimate has been completed by Tetra Tech to incorporate new information collected from the Mexican Hat Project since completion of the previous NI 43-101 Technical Report in 2015 (Webb, 2015). This new work has been completed using Datamine Studio RM v 1.2.47.0.

Since the completion of the previous resource estimate, an additional eight diamond drillholes and 30 reverse circulation (RC) drillholes, totaling 1,979.3 m and 8,809.40 m respectively have been completed, along with the collection of 304 drill core and 1,962 RC samples. The program focused on continued drilling of the Mexican Hat Project to increase the tenor of the known resources.

14.2 PREVIOUS ESTIMATES

A Mineral Resource Estimate was previously reported for the Mexican Hat Project in 2015 by DRW Geological. This work included the results from historical drilling and channel sampling, and the results of the GMV channel sampling up to and including the 2015 campaign. Geological modeling was completed using Discover 3D software, and the block model was developed using Discover 3D. The Mineral Resource Estimate was reported at a 0.20 g/t cut-off and is summarized in Table 14-1 below.

The 2015 resource statement is superseded by the current resource statement and is no longer relied upon.

Table 14-1: Previous Mineral Resource Estimate, Effective Date February 2015

Category	Cut-off (g/t Au)	Tonnes	Grams Au	Grade (Au, g/t)	Ounces Au
Inferred	0.20	23,452,000	16,528,000	0.70	531,400

14.3 DATABASE

GMV maintains all geological data in a Microsoft Excel© database containing header, survey, assays, and lithology tables. The database consists of surface trenching (treated as drillholes), rotary and core drilling from Placer Dome, surface samples (treated as drillholes) and core drilling from Auracle Resources, along with the drilling completed by GMV Minerals. No holes completed by Kalahari Resources (1996) were included in the resource estimate, nor was trench MHT-133 as it appeared to unduly influence the resource estimate.

A copy of the header, survey, and assays was provided to Tetra Tech. GMV does not have a detailed lithological database. It is strongly recommended that a robust lithology database be compiled in order to better understand, target, and model the zones present at Mexican Hat.

Table 14-2: GMV Database for Mineral Resource Estimate

Type	Borehole and Trenching Records	Survey Records	Assay Records
Core	45	52	2,650
RC	27	27	2,308
Rotary	120	120	5,536
Channel (trench)	149	183	1,864
Total	341	382	12,358

14.4   INPUT DATA ANALYSIS

14.4.1   Assays

Various analytical techniques were employed by each group, along with duplicate and repeat samples. In all instances, where possible, fire assays were used. If no fire assays exist, geochemical assays were used. An investigation into the sample data and quality collected by each pervious operator showed that Kalahari's dataset was statistically distinct from the other operators, and so it was removed from further consideration within the database.

Overall, the Mexican Hat deposit has been sampled by 12,358 gold assays, of which, 9,775 samples where constrained by wireframes to make up the current mineral resource. The remaining assays were either below the lower grade shell grade boundary or were not able to be interpreted and assigned to a discrete zone. Assay intervals within each zone were captured by clipping the boreholes to the encompassing wireframe zone and then translating them into individual borehole files. These files were reviewed to ensure all the proper assay intervals were captured. Any un-sampled intervals were replaced with zero grade. Table 14-3 summarizes the basic statistics for the assays in the various wireframe domains. Figure 14-1 presents a histogram of the sample lengths before compositing.

Table 14-3: Mexican Hat Mineral Domain Drillhole Statistics

Zone	Field	No. of Samples	Minimum	Maximum	Mean	Variance	Standard Deviation
1	Length (m)	73	1.50	22.90	3.15	12.73	3.57
	Au (g/t)	73	0.00	7.02	0.35	0.99	0.99
2	Length (m)	149	1.50	21.40	2.33	4.70	2.17
	Au (g/t)	149	0.00	7.03	0.37	0.82	0.90
3	Length (m)	303	0.60	40.00	2.11	6.47	2.54
	Au (g/t)	303	0.00	13.20	0.39	1.00	1.00
4	Length (m)	780	0.30	45.20	1.93	6.58	2.56
	Au (g/t)	780	0.00	13.54	0.37	0.97	0.99
5	Length (m)	5302	0.10	126.50	2.55	21.39	4.62
	Au (g/t)	5302	0.00	30.69	0.32	0.75	0.86
6	Length (m)	1751	0.20	80.80	1.85	8.90	2.98
	Au (g/t)	1751	0.00	613.43	1.60	249.07	15.78
7	Length (m)	57	0.92	35.10	4.34	47.91	6.92
	Au (g/t)	57	0.00	1.03	0.10	0.03	0.16
8	Length (m)	1304	0.20	50.30	2.02	7.37	2.71
	Au (g/t)	1304	0.00	80.23	0.57	8.36	2.89
9	Length (m)	83	0.30	12.50	1.96	3.02	1.74
	Au (g/t)	83	0.00	6.72	0.52	0.92	0.96
All Drillholes	Length (m)	9775	0.10	126.50	2.30	15.50	3.94
	Au (g/t)	9775	0.00	613.43	0.59	46.52	6.82
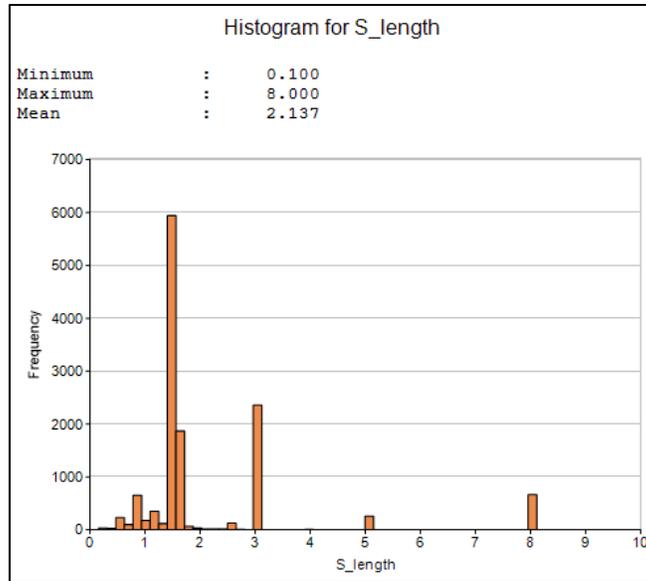
Figure 14-1: Mexican Hat Sample Length Histogram

14.4.2   Composites

Compositing of all assay data within the wireframes was completed at 1.5 m intervals based upon the predominance of 1.5 m samples within the raw assay dataset. The compositing procedure was undertaken such that the composite intervals honoured the geological solids. Flexibility was allowed with the composite lengths by using the backstitching method whereby the composite lengths could be marginally adjusted for each drillhole in order to accommodate the last sample interval.

Table 14-4 summarizes the statistics for the samples after compositing.

Table 14-4: Mexican Hat Composite Statistics

Zone	Field	Composite Length	No. of Samples	Minimum	Maximum	Mean	Variance	Standard Deviation
1	Length (m)	1.5	151	1.50	1.53	1.52	0.00	0.00
	Au (g/t)	1.5	151	0.00	7.02	0.26	0.89	0.94
2	Length (m)	1.5	229	1.48	1.60	1.52	0.00	0.02
	Au (g/t)	1.5	229	0.00	7.03	0.26	0.54	0.73
3	Length (m)	1.5	420	1.48	1.57	1.52	0.00	0.02
	Au (g/t)	1.5	420	0.00	13.20	0.29	0.73	0.85
4	Length (m)	1.5	1001	1.48	1.55	1.51	0.00	0.02
	Au (g/t)	1.5	1001	0.00	12.41	0.28	0.59	0.77
5	Length (m)	1.5	9011	1.40	1.62	1.50	0.00	0.01
	Au (g/t)	1.5	9011	0.00	29.68	0.20	0.40	0.63
6	Length (m)	1.5	2148	1.43	1.68	1.51	0.00	0.02
	Au (g/t)	1.5	2148	0.00	355.99	1.12	80.85	8.99
7	Length (m)	1.5	165	1.47	1.54	1.50	0.00	0.02
	Au (g/t)	1.5	165	0.00	1.02	0.04	0.01	0.10
8	Length (m)	1.5	1746	1.42	1.57	1.51	0.00	0.02
	Au (g/t)	1.5	1746	0.00	78.93	0.42	4.90	2.21
9	Length (m)	1.5	108	1.49	1.53	1.51	0.00	0.01
	Au (g/t)	1.5	108	0.00	2.85	0.41	0.29	0.54

14.4.3   Capping

When frequency distributions are skewed, a very small number or proportion of samples may represent a large amount of the contained metal in the resource. Frequently, these samples may be scattered through the deposit and not restricted to spatially identifiable or continuous zones. Sometimes, small clusters of high-grade mineralization may be present, and it may or may not be possible or practical to restrict their influence. Other times, the very high-grade samples may be the result of laboratory errors; pulps sometimes segregate high specific gravity materials like electrum or pyrite and may produce biased results if the pulps are not re-homogenized prior to aliquot selection for analysis.

Even when the assays are valid, linear interpolation (weighted average) grade estimation methods can be adversely affected. When these methods are used, the inclusion of a high-grade sample will have a greater influence on the estimate than a lower grade sample. This can lead to undue projection (or smearing) of the effect of high-grade material into areas for which there is no evidence on hand that the grade material continues to occur. Under such circumstances, restriction of the influence of the higher-grade material is mandatory.

For the Mexican Hat Project, the mitigation of undue high-grade influence was achieved using the following two methods:

- Generation of a "high grade" mineralization shell within a lower grade envelope;

- Statistical capping of gold grades which were deemed outliers.

14.4.3.1   Grade Shells

Beneath the topographic high which is referred to as Mexican Hat, a higher grade core appears to be present which is surrounded by a lower grade halo. In order to constrain the influence of this higher grade material, a 1.0 g/t gold grade shell was generated, which was surrounded by a lower grade, 0.2 g/t gold grade shell. Figure 14-2 below shows drillhole grade, 1.0 g/t Au wireframed grade shell, and results of the grade constraint after block modeling.
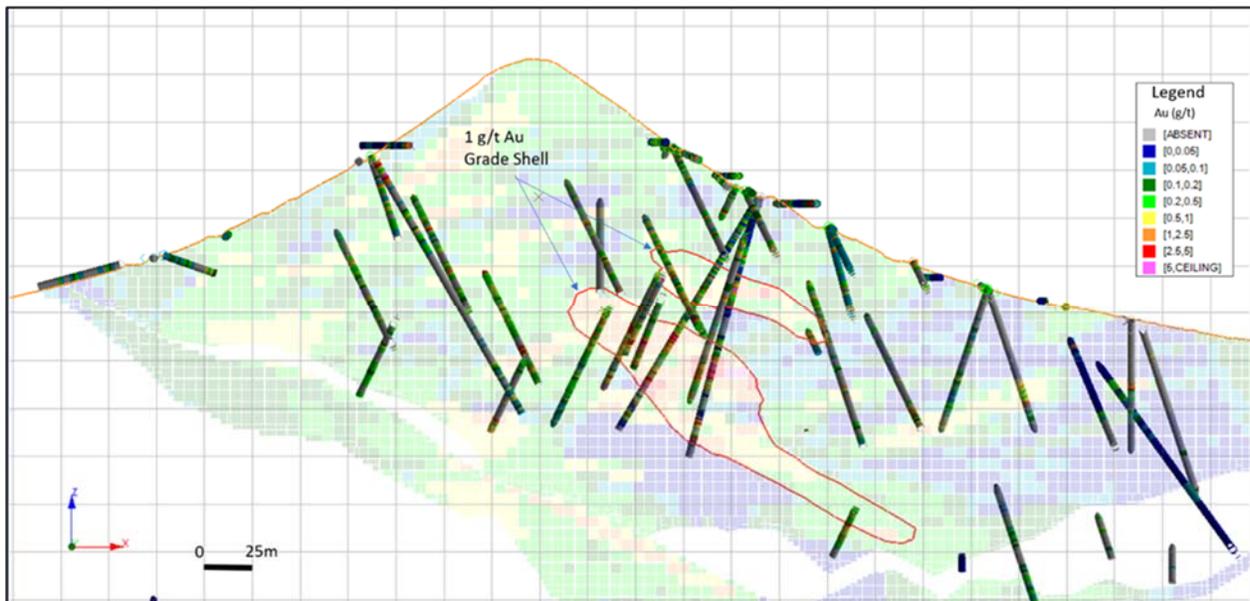
Figure 14-2: Mexican Hat Grade Shells

14.4.3.2   Statistical Capping of Gold

Based on a review of probability (Figure 14-4) and histogram plots (Figure 14-3) of the composited assays, it was concluded that there was reasonable justification for capping of the Au values. Breaks in the slope of the cumulative probability imply that a few higher grade samples are spatially discontinuous from the remainder of the data set. The samples were then further investigated by decile and percentile analysis where the top decile (greatest 10% of the sample population) contained more than 40% of the total metal content (Table 14-5). Based on this analysis, capping of the composited assay data was applied at 32 g/t Au for the Mexican Hat Project.

A total of seven (7) samples were capped within the database.
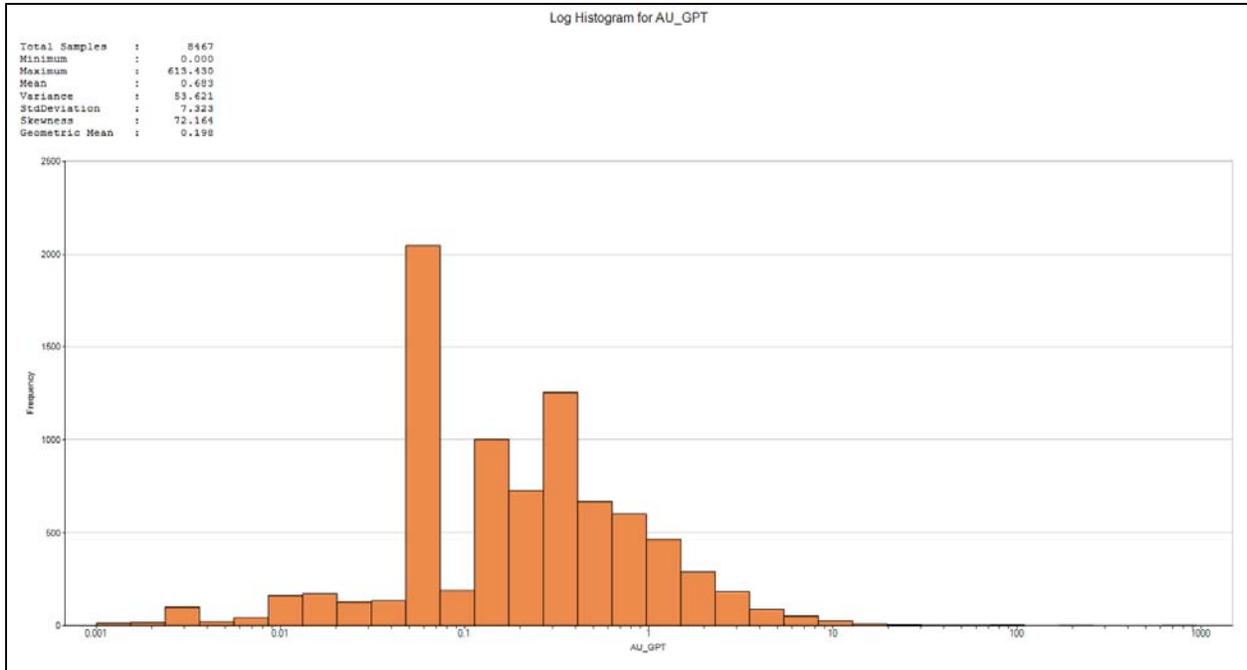
Figure 14-3: Log Histogram of Gold Assays in Wireframes
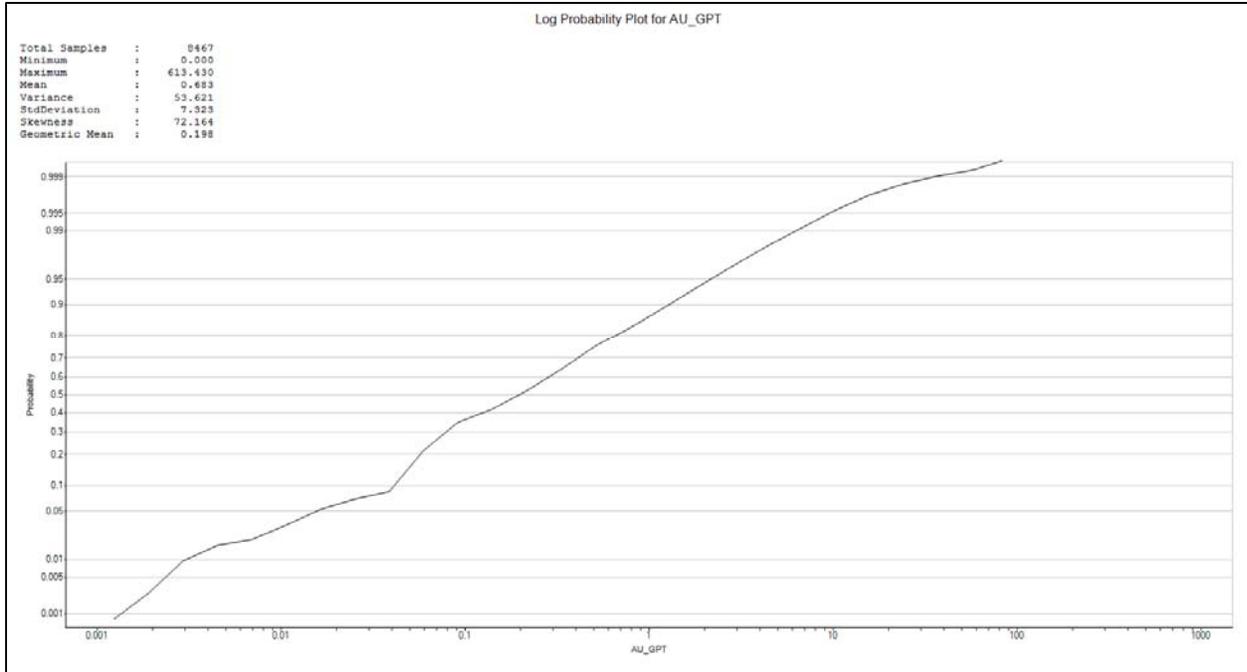
Figure 14-4: Log Probability Plot of Gold Assays

Table 14-5: Parrish Decile Analysis for Capping of Gold Grades

Q%_From	Q%_To	Nsamples	Mean	Minimum	Maximum	Metal	Metal%
0	10	840	0.06	0.05	0.07	81.33	0.98
10	20	838	0.07	0.07	0.07	88.71	1.07
20	30	842	0.09	0.07	0.12	115.85	1.40
30	40	839	0.14	0.12	0.15	173.73	2.10
40	50	840	0.19	0.15	0.22	239.48	2.90
50	60	840	0.26	0.22	0.31	332.81	4.03
60	70	840	0.37	0.31	0.45	470.66	5.70
70	80	840	0.56	0.45	0.70	714.61	8.65
80	90	840	0.91	0.71	1.22	1156.40	14.00
90	100	841	3.86	1.22	355.99	4888.51	59.17
90	91	84	1.27	1.22	1.32	161.27	1.95
91	92	84	1.36	1.32	1.41	172.95	2.09
92	93	84	1.49	1.41	1.58	188.75	2.28
93	94	84	1.68	1.58	1.77	212.16	2.57
94	95	84	1.88	1.77	2.00	238.21	2.88
95	96	84	2.11	2.00	2.23	267.46	3.24
96	97	84	2.51	2.24	2.74	317.33	3.84
97	98	84	3.05	2.75	3.45	385.53	4.67
98	99	84	4.20	3.47	5.23	531.29	6.43
99	100	85	18.87	5.23	355.99	2413.55	29.21
0	100	8400	0.65	0.05	355.99	8262.09	100.00

Table 14-6: Post Capping Stats

Zone	Field	Composite Length	No. Of Samples	Samples Capped	Minimum	Maximum	Mean	Variance	Standard Deviation
1	Au (g/t)	1.5	151	0	0	7.02	0.26	0.89	0.94
2	Au (g/t)	1.5	229	0	0	7.03	0.26	0.54	0.73
3	Au (g/t)	1.5	420	0	0	13.2	0.29	0.73	0.85
4	Au (g/t)	1.5	1001	0	0	12.41	0.28	0.59	0.77
5	Au (g/t)	1.5	9011	0	0	29.68	0.2	0.4	0.63
6	Au (g/t)	1.5	2148	5	0	32	0.85	5.41	2.32
7	Au (g/t)	1.5	165	0	0	1.02	0.04	0.01	0.1
8	Au (g/t)	1.5	1746	2	0	32	0.4	1.93	1.39
9	Au (g/t)	1.5	108	0	0	2.85	0.41	0.29	0.54

14.4.4   Bulk Density

Material density measurements represent an important parameter for a mineral resource estimate, as this parameter is used to calculate the tonnage of a deposit. Specific gravity (SG) represents density in relation to the density of a standard or reference (usually water) and can be used as a proxy for bulk density. Given that the density of pure water is so close to 1 (0.9976 grams per cubic centimeter), specific gravity and density are essentially the same.

SG data for the Mexican Hat property collected from various lithology types within the 2017 GMV drill core and sent to ALS laboratories in North Vancouver for SG testing. SG measurements were collected using the water displacement wax method. The wax method applies a paraffin coating to the rocks and is particularly suitable for friable material as the coating helps to maintain the integrity of the sample. The calculation used for the water displacement wax method is presented below:

$$\text{Density} = \text{Weight in Air} / (\text{Weight in Air}) \times (\text{Weight in Water})$$

Compiled SG values are shown in Table 14-7.

Table 14-7: Average SG Results by Lithologic Unit

Lithology	Average Value	Standard Deviation	Minimum Value	Maximum Value	Number of Samples
Latite	2.47	0.177	2.32	2.83	6
Andesite	2.535	0.182	2.33	2.86	4
Basalt	2.81	0.265	2.44	3.05	3
Average	2.57	0.243	2.32	3.02	13

14.4.5   Variography

Variography is the measure of degree of variation two samples will exhibit based upon varying spatial separations. In general, it can be expected that samples located closer together will contain less grade variation than samples collected at a greater distance. The correlogram statistically measures this correlation between data values as a function of their separation distance and direction. Close spaced samples with similar grades can expect correlation coefficients which approach a value 1.0. As sample separation increases, increased grade variation is typically expected, and the correlogram will decrease towards 0.0. The distance at which the correlogram reaches zero is called the "range of correlation", or simply the range. The range of the correlogram corresponds roughly to the more qualitative notion of the "range of influence" of a sample; it is the distance over which sample values show some persistence or correlation.

Attempts to generate directional sample correlograms were calculated along horizontal azimuths of 0, 30, 60, 90, 120, 150, 180, 210, 240, 270, 300, and 330 degrees in Datamine Studio RM. For each azimuth, sample correlograms were also calculated at dips of 30 and 60 degrees and horizontally. Based upon the results of the variography it was not possible to obtain any meaningful variograms. This can indicate a high degree of grade variability within the mineralization sampled on the Project, or the presence of an isotropic variogram.

14.5   INTERPOLATION PLAN

14.5.1   Search Parameters

The interpolation plan of the Mexican Hat resource estimation was completed using the following methods: nearest neighbor (NN), inverse distance squared (ID2) and inverse distance cubed (ID3).

The estimations were designed as a three pass system which were run independently within each individual wireframe using composite data constrained by the wireframe. Table 14-8 below summarizes search distances and rotations for estimating a block as well as minimum and maximum number of composites required.

Table 14-8: Mexican Hat Search Ellipse Parameters

Pass Number	Au_Cap	Zone	Search Distance			Rotation			Number of Composites		
			X	Y	Z	Z	X	Z	Min	Max	Max per Drillhole
Pass 1	32	1	37.5	37.5	22.5	337	35	0	8	16	4
		2	37.5	37.5	22.5	337	35	0	8	16	4
		3	37.5	37.5	22.5	337	35	0	8	16	4
		4	37.5	37.5	22.5	337	35	0	8	16	4
		5	37.5	37.5	22.5	337	35	0	8	16	4
		6	35	35	15	30	20	0	8	16	4
		7	37.5	37.5	22.5	337	35	0	8	16	4
		8	50	50	25	37	40	0	8	16	4
		9	37.5	37.5	22.5	65	60	0	8	16	4
Pass 2	32	1	56.25	56.25	33.75	337	35	0	5	16	4
		2	56.25	56.25	33.75	337	35	0	5	16	4
		3	56.25	56.25	33.75	337	35	0	5	16	4
		4	56.25	56.25	33.75	337	35	0	5	16	4
		5	56.25	56.25	33.75	337	35	0	5	16	4
		6	52.5	52.5	22.5	30	20	0	5	16	4
		7	56.25	56.25	33.75	337	35	0	5	16	4
		8	75	75	37.5	37	40	0	5	16	4
		9	56.25	56.25	33.75	65	60	0	5	16	4
Pass 3	32	1	75	75	45	337	35	0	4	16	4
		2	75	75	45	337	35	0	4	16	4
		3	75	75	45	337	35	0	4	16	4
		4	75	75	45	337	35	0	4	16	4
		5	75	75	45	337	35	0	4	16	4
		6	70	70	30	30	20	0	4	16	4
		7	75	75	45	337	35	0	4	16	4
		8	100	100	50	37	40	0	4	16	4
		9	75	75	45	65	60	0	4	16	4

14.6 MODEL DEVELOPMENT

14.6.1 Geological Interpretation

A total of nine three-dimensional (3D) wireframe models of mineralization were modeled in Datamine along with topography. The mineralization at Mexican Hat was broken into multiple domains reflecting the difference in either grade shell domain or mineralization type (structural vs disseminated). This modeling follows the interpretation that multiple mineralizing events have occurred, which have results in at least two distinct styles of gold mineralization.

The wireframes were based on at least two composite samples with average values greater than or equal to 0.2 g/t Au to construct a 3D model for each of the zones. Where necessary, 1.0 g/t grade shells where generated within regions with a higher grade domain. Internal dilution of samples below the 0.2 g/t cut-off was maintained were geological continuity of a wider mineralized between sections was observed.

The H Zone (zones 8 and 9, Figure 14-5), oriented at approximately 310 / -40, represent a mineralized fault which is interpreted as the "feeder" fault for the deposit. All other zones were truncated to the H Zone to honor this interpretation and constrain mineralization.

Drilling directly underneath the predominant Mexican hat hill is a broad low grade disseminated gold zone (zone 5, Figure 14-5). Encompassed within this zone is a structurally controlled high-grade gold core (zone 6). This relatively flat laying core was modeled using a minimum grade of 1.0 g/t Au over a minimum of two composited samples.

To the south, mineralization occurs within 4 discreet, structurally hosted corridors, which are up to 20m in width and display an average orientation of 245 / -35 (zones 1-4, Figure 14-5). The angle between the H-zone fault and the southern mineralized structures is approximately 50 degrees.

Interpretations were made in Datamine on a sectional basis, and these interpretations were linked with tag strings and triangulated to build 3D solids. Table 14-9 tabulates the solids and associated volumes. The solids were validated in Datamine software and no errors were found.
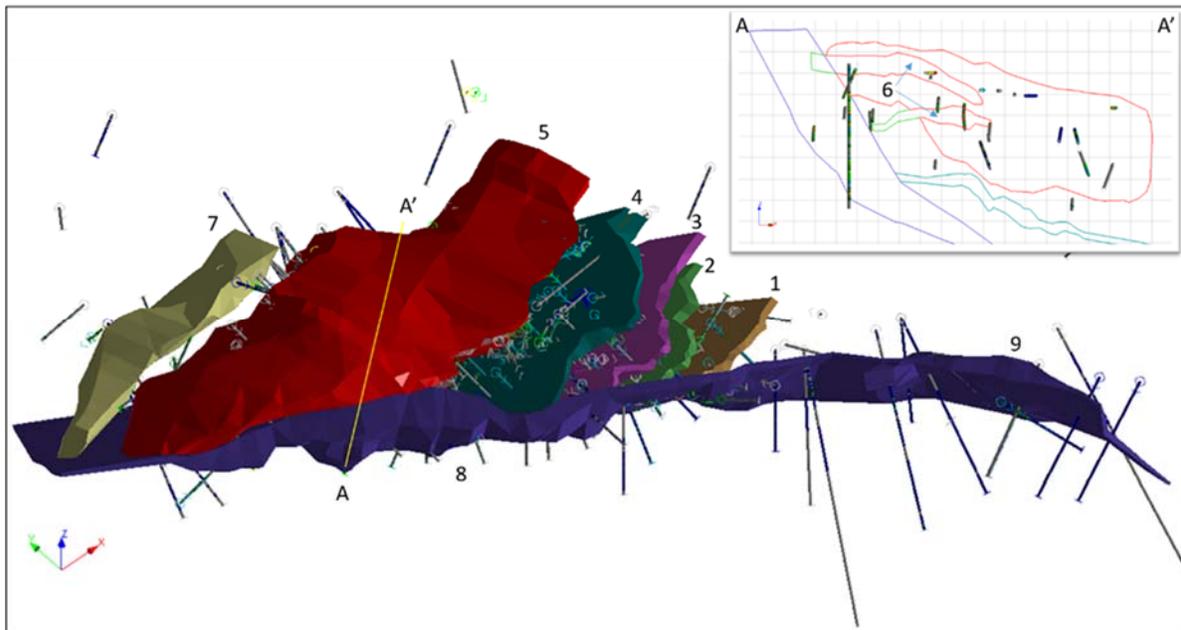
Figure 14-5: Oblique View of Mexican Hat Mineralized Domains, Looking Down to Northeast

Table 14-9: Mexican Hat Wireframe Volumes

Zone	Minimum X	Maximum X	Minimum Y	maximum Y	Minimum Z	Maximum Z	Volume (m³)
1	613063.94	613350.14	3518839.83	3519298.54	1200.75	1497.36	1,337,063.62
2	613035.54	613309.22	3518875.58	3519321.70	1214.39	1506.99	1,158,951.72
3	612962.88	613344.43	3518902.74	3519353.83	1240.88	1515.15	1,842,204.29
4	612889.69	613327.42	3518947.76	3519393.49	1256.45	1523.31	1,895,023.86
5	612468.98	613265.62	3519074.64	3519463.89	1228.18	1599.77	22,345,648.57
6	612773.63	613083.24	3519069.30	3519415.99	1315.14	1523.97	1,877,109.02
7	612440.38	612844.69	3519355.86	3519573.64	1299.40	1554.65	4,266,902.27
8	612345.22	613622.83	3518646.34	3519591.15	1144.05	1541.15	13,549,154.09
9	613398.63	613693.77	3518425.29	3518750.80	1177.34	1537.37	1,726,808.52

14.6.2   Block Model

Individual block models were established in Datamine for the discreet mineralized zones using one parent model as the origin. A non- rotated block model was utilized. The particulars of the parent block model are presented below in Table 14-10.

Drill spacing varies from 25-100 m along the sections, and 25-100 m between the sections across the deposit. A block size of 6 m by 6m by 6m was selected. To accommodate the local wireframe anisotropies, each parent cell within the 070 trending zones was allowed to be spilt into two subcells in in the X-Y direction, along with a variable width in the Z direction. The north-north-west trending H-Zone allowed sub celling in the Y-Z direction, along with a variable width in the X direction. This allowed the blocks to more accurately fill the volume of the wireframes. Estimation on each block was completed on the parent blocks and the grades assigned to the sub-cell blocks.

Occasionally, the trenching data did not plot directly along the provided topography, with trenches occasionally plotting up to 5 meters above the topographic surface. To allow the trench data to be used in the interpolation process, trenches located above topography were captured in the wireframes. However, to avoid any overestimate of material above topography, above surface "air" blocks were generated during the block modeling process. These air blocks were flagged with a unique identifier, and then superimposed overtop of the mineralization model. Once combined, any mineralized blocks which then contained the unique "air block" identifier were deleted, thereby leaving only subsurface mineralization. This process is presented visually in Figure 14-6.

Upon completion of the block modeling, all zones were combined to form the Mexican Hat Model.

Table 14-10: Mexican Hat Parent Model Parameters

Origin			Cell Size			Number of Cells		
X Origin	Y Origin	Z Origin	XINC	YINC	ZINC	NX	NY	NZ
612,345	3518425	1144	6	6	6	225	195	76

Table 14-11: Mexican Hat Wireframe Volumes vs Block Volumes

Zone	Wireframe Volume (m³)	Unfilled Block Volume (m³)	Volume Difference (m³)	Relative Difference (%)
1	1,337,063.62	1,336,456.98	-606.64	-0.01
2	1,158,951.72	1,158,997.44	45.72	0.00
3	1,842,204.29	1,734,883.24	-107,321.05	-1.50
4	1,895,023.86	1,895,202.61	178.75	0.00
5	22,345,648.57	22,343,418.40	-2,230.17	0.00
6	1,877,109.02	1,877,920.34	811.32	0.01
7	4,266,902.27	4,199,973.32	-66,928.95	-0.40
8	13,549,154.09	13,549,361.82	207.73	0.00
9	1,726,808.52	1,727,667.85	859.33	0.01

A: Section showing Trenches Plotted above Topography (orange line).   B: Grade Filled Zones, with Blocks Extending Above Topography.
C: Addition / Superimposing of "Air" blocks (Red).   D: Removal of any Blocks Above Topography, Leaving Final Block Model.
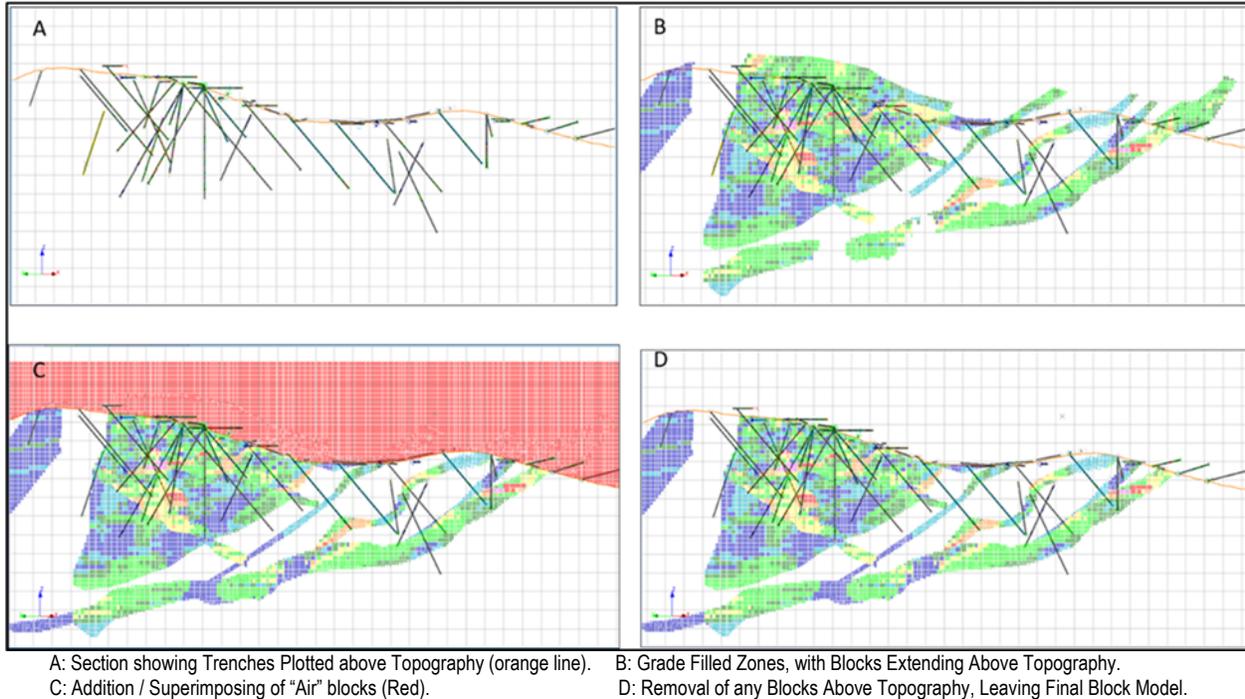
Figure 14-6: Process of Allowing Trenching Data to be Captured for Estimation and Removing any above Surface Mineralization

14.7   MINERAL RESOURCE STATEMENT

14.7.1   Classification

Mineral Resource Classification was performed in accordance with CIM Best Practices. In accordance with CIM Definitions Standards (2014) the Tetra Tech QP is of the opinion that the Mexican Hat Deposit is a reasonable prospect for eventual extraction by open pit and heap leach mining, on the basis of:

- Location of the deposit in reasonable proximity to power and road infrastructure;

- Demonstrated size and grade of the mineral resource estimate in comparison to similar deposit types in Arizona; and

- No known environmental, permitting, legal, title, taxation, socio-economic, marketing or other relevant issues are known to the QP that may affect the estimate of a mineral resource.

All resources calculated for the Mexican Hat deposit are classified as Inferred, based upon the following reasons:

- Large proportion of drilling data is historical and cannot be verified,

- A moderate to high degree has been observed of grade variability between twinned and close spaced drillholes,

- Due to abundance of RC drilling, uncertainty exists regarding the structural controls on mineralization,

- Lack of supporting geochemical analysis to support a more detailed geological model, and

- Lack of high resolution drillhole and trench location surveys.

To determine the quantities of material offering "reasonable prospects for eventual economic extraction by an open pit, Tetra Tech used a Whittle pit optimizer and reasonable mining assumptions to evaluate the proportions of the block models that could be "reasonably expected" to be mined from an open pit. The results are used as a guide to assist in the preparation of a mineral resource statement and select an appropriate resource reporting cut-off grade (Table 14-12).

The reader is cautioned that the results from the pit optimization are used solely for testing the "reasonable prospects for eventual economic extraction" by an open pit, and do not represent mineral reserves which can only be estimated based on an economic evaluation that is used in a preliminary feasibility study of a mineral project. As such, no reserves have been estimated. As per NI 43-101, mineral resources, which are not mineral reserves, do not have to demonstrate economic viability.

Figure 14-7 shows a cross section of a designated Whittle shell and the within which the resources have been reported for the Mexican Hat deposit.

Table 14-13 represents the inferred resources for the Mexican Hat Project within the Whittle shell at 0.2 g/t gold cut-off.
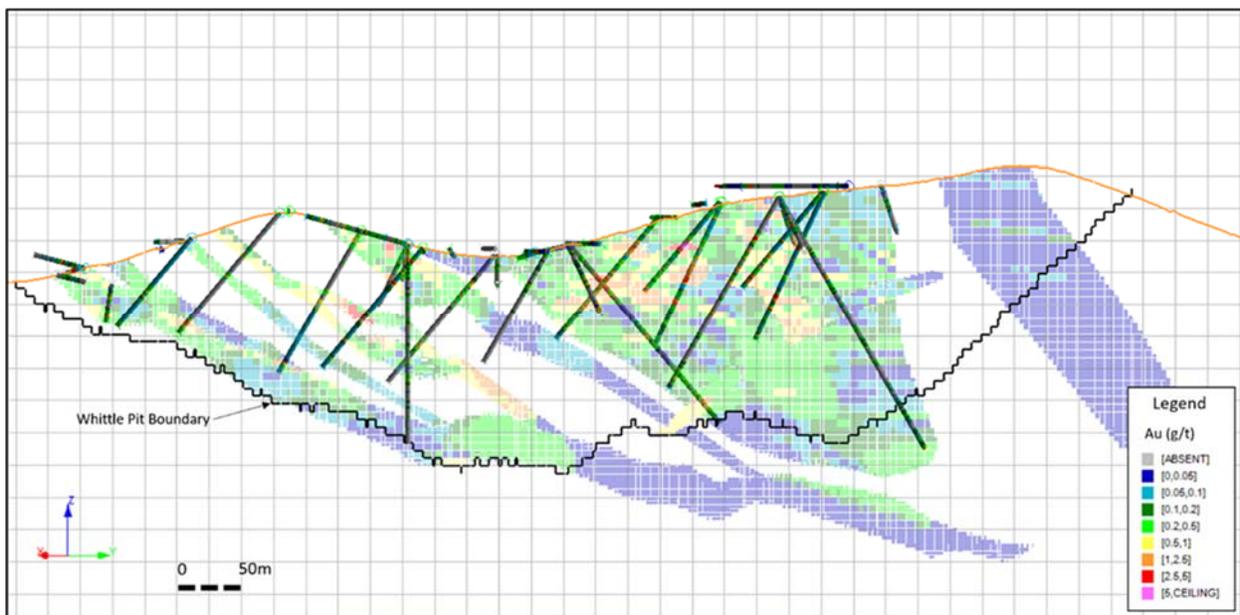
Figure 14-7: Mexican Hat Cross-section of the Whittle Shell and Estimated Gold Block Grades, Looking West

14.7.2 Resource Tabulation

To determine the quantities of material offering "reasonable prospects for eventual economic extraction" by an open pit, Tetra Tech used a Whittle pit optimizer and reasonable mining assumptions to evaluate the proportions of the block models that could be "reasonably expected" to be mined from an open pit. The results are used as a guide to assist in the preparation of a mineral resource statement and to select an appropriate resource reporting cut-off grade (Table 14-12).

Table 14-12: Whittle™ Optimization Parameters for Resource Estimation Constraint

Whittle Input Parameters	
Input Parameter	Value
Mining Cost ($/t)	$1.50
Mining Recovery (%)	98%
Mining Dilution (%)	2%
Process Cost ($/t processed)	$3.25
G&A ($/t processed)	$0.55
Gold Price ($/oz)	$1,300
Process Throughput (t/day)	15,000
Process Throughput (t/year)	5,475,000
Total Mining Limit (t/year)	25,000,000
Discount Rate (%)	7%

Table 14-13: Mineral Resource Statement, Mexican Hat Project, Arizona, USA, Tetra Tech, Effective Date June 22, 2018

Category	Cut-off (g/t Au)	Grade (Au, g/t)	Tonnes	Gold Oz	Strip Ratio
Inferred	0.20	0.616	32,876,000	651,000	2.56

- The Mineral Resource Estimate has been constrained to a preliminary optimized pit shell, using the following parameters: SG = 2.57 gm/cc based on testwork, mining costs = $1.50/tonne, mining recovery =98%, mining dilution = 2%, process cost = $3.25 per tonne, G&A = $0.55 per tonne, gold price = $1,300 per troy ounce, throughput at 15,000 tpd, discount rate = 7%. Top cut at 32 g/t.
- Mineral Resources constrained to optimized pit shells are not mineral reserves and do not have demonstrated economic viability.
- Conforms to NI 43-101, Companion Policy 43-101CP, and the CIM Definition Standards for Mineral Resources and Mineral Reserves. Inferred Resources have been estimated from geological evidence and limited sampling and must be treated with a lower level of confidence than Measured and Indicated Resources.
- All numbers are rounded. Overall numbers may not be exact due to rounding.
- There are no known legal, political, environmental, or other risks that could materially affect the potential development of the mineral resources.

14.7.3   Grade Sensitivity Analysis

The mineral resources at the Mexican Hat Property are sensitive to the selection of the reporting cut-off grade. To Illustrate this sensitivity, the block model quantities and grade estimates are presented at various cut-offs in a grade tonnage curve, presented in Figure 14-8. The reader is cautioned that the values presented on this chart should not be construed with a Mineral Reserve Statement. The values are only presented to show the sensitivity of the block model estimates to the selection of cut-off grade.
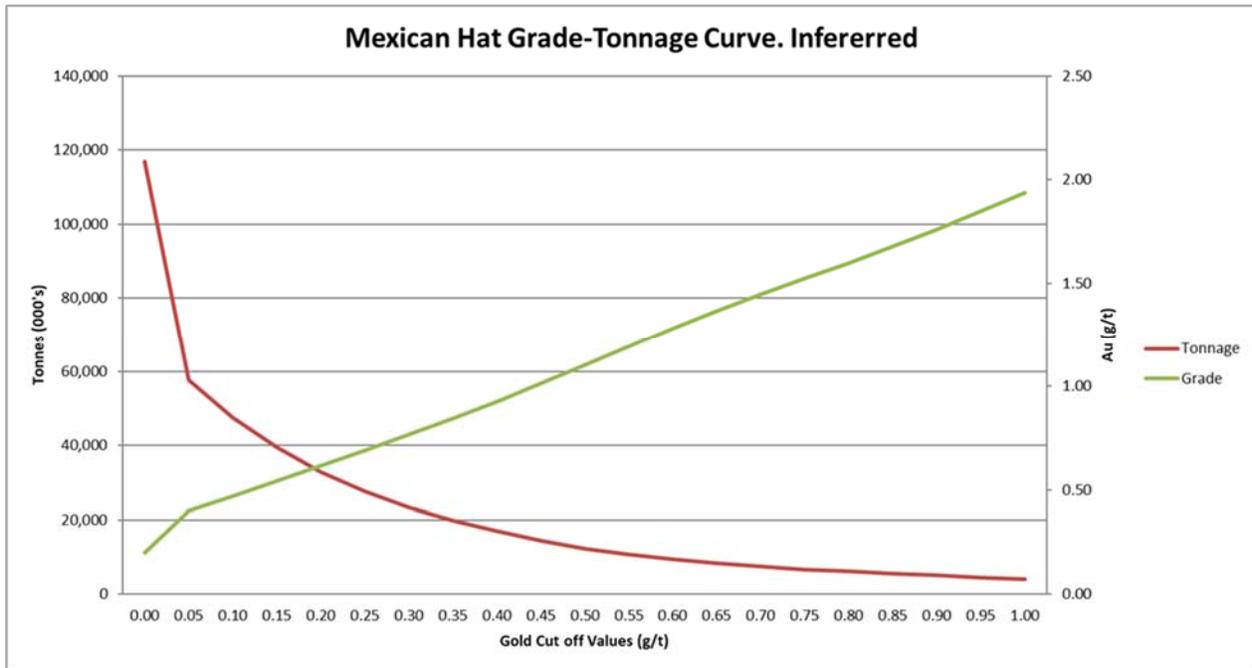
Figure 14-8: Mexican Hat Inferred Category Grade Tonnage Curves

14.7.4 Model Validation

Model validation is undertaken to demonstrate that the input data has been fairly and accurately represented in outputs of the block modelling process. Substantial deviations to the data distribution or mean tendency, or inflations to high grade ranges can lead to misrepresentation or overstatement of the mineral resource estimate.

Methods used to validate the models include visual spatial comparison of input drillhole composite data on cross-section with block model output, comparison of descriptive statistics by means of a histogram analysis, and swath plot analysis. Additionally, the swath plots include a comparison of the Inverse Distance Weighted (ID, to Power of 2 and 3) interpolation results and nearest neighbor interpolation. These comparisons provide qualitative comparison of the results.

The model validation indicates that the input data has been reasonably represented in the model, at a confidence of an Inferred Mineral Resource.

14.7.4.1 Model Statistic Comparison

The global block model estimation of the OK Method was compared to that of the global ID2 and NN model values as well as the composite drillhole data. Table 14-14 shows this comparison of the global estimates for the three estimation method calculations. In general, there is agreement between the OK model, the ID2 model, and the NN model. Larger discrepancies are reflected as a result of lower drill density in some portions of the model. There is a degree of smoothing apparent when compared to the diamond drill statistics. Comparisons were made using all blocks at a 0.00 g/t gold cut-off.

Table 14-14: Mexican Hat Comparison by Estimation Method

Estimation Method	Au g/t Cut-off	Tonnes	Au g/t	Contained Ounces	% Difference Total metal
Nearest Neighbor	0.2	24,152,761	0.88	685,752	5.39
Inverse Distance Squared	0.2	33,740,268	0.61	663,672	2.00
Inverse Distance Cubed	0.2	32,875,980	0.62	650,656	Base Case

14.7.4.2   Visual Comparison

Visual comparison of the input data with the output block model resulted in decent correlation. Grade trends in certain areas can be improved in future modelling by incorporating additional structural and geological controls.
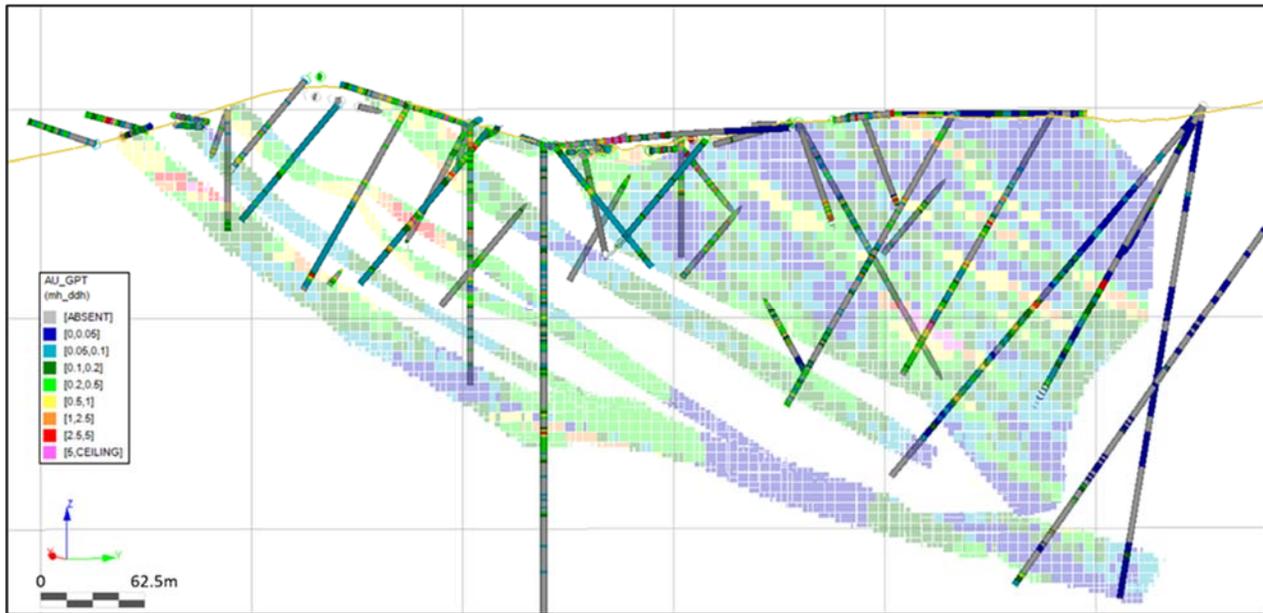
Figure 14-9: Mexican Hat Model, Looking West South West

14.7.4.3   Swath Plots

Swath plots provide a qualitative method to observe preservation of the input composite grade trends on a spatial basis in the block model results. The data is plotted with average values along discrete intervals along the Cartesian X, Y and Z axis (i.e., easting, northing, and elevation). Input sample data used for these swath plots is composited and capped, resulting in a slightly smoother trend than raw data. However, the sample data can be clustered and may misrepresent areas of high grade mineralization that have been oversampled. The block data is based on the composited and capped data and can also appear clustered due to the creation of subblocks. Both datasets have been constrained to the geological and grade shell models.

The block model swaths show good correlation between the ID2, ID3, and NN models, where ID2 and ID3 are more smoothed than the NN model. Overall, all three models are somewhat smoothed in comparison to the average capped and composited grades shown for each section. This is attributed to the reginal averaging of capped / composited grades into each block and is considered to be a reasonable spatial estimation for the composite sample grades.
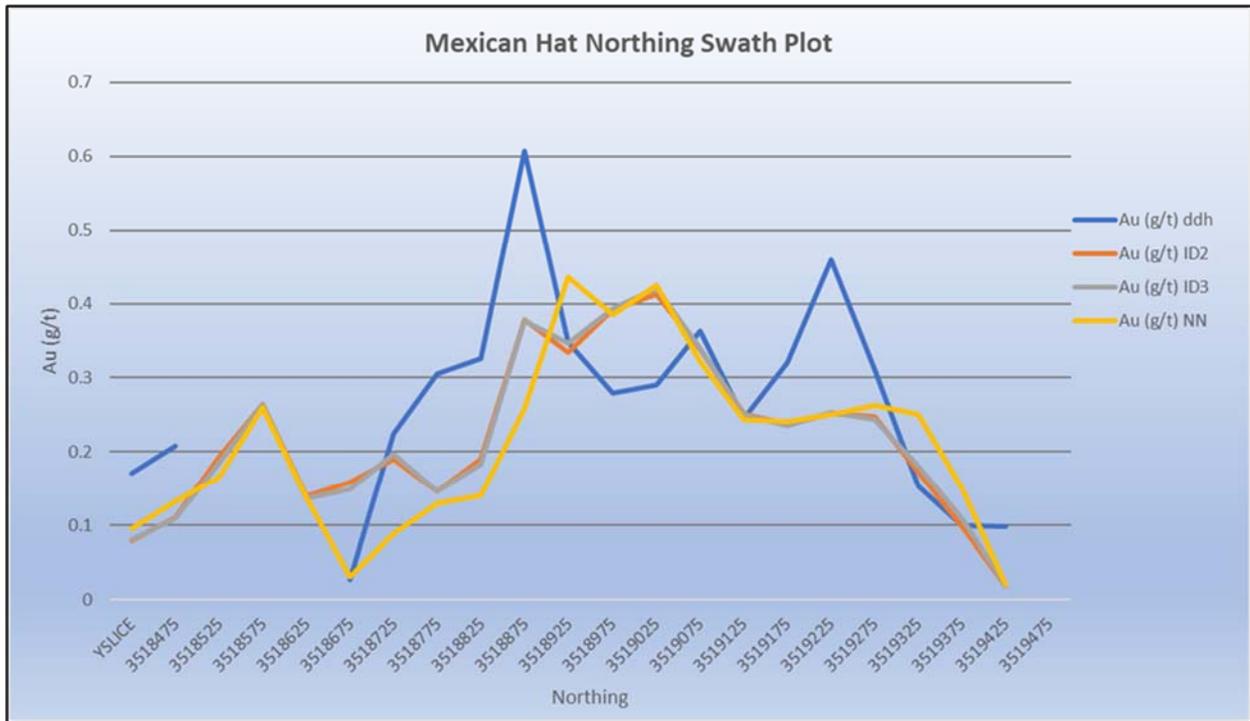
Figure 14-10: Swath Plot along Northings for the Mexican Hat Model
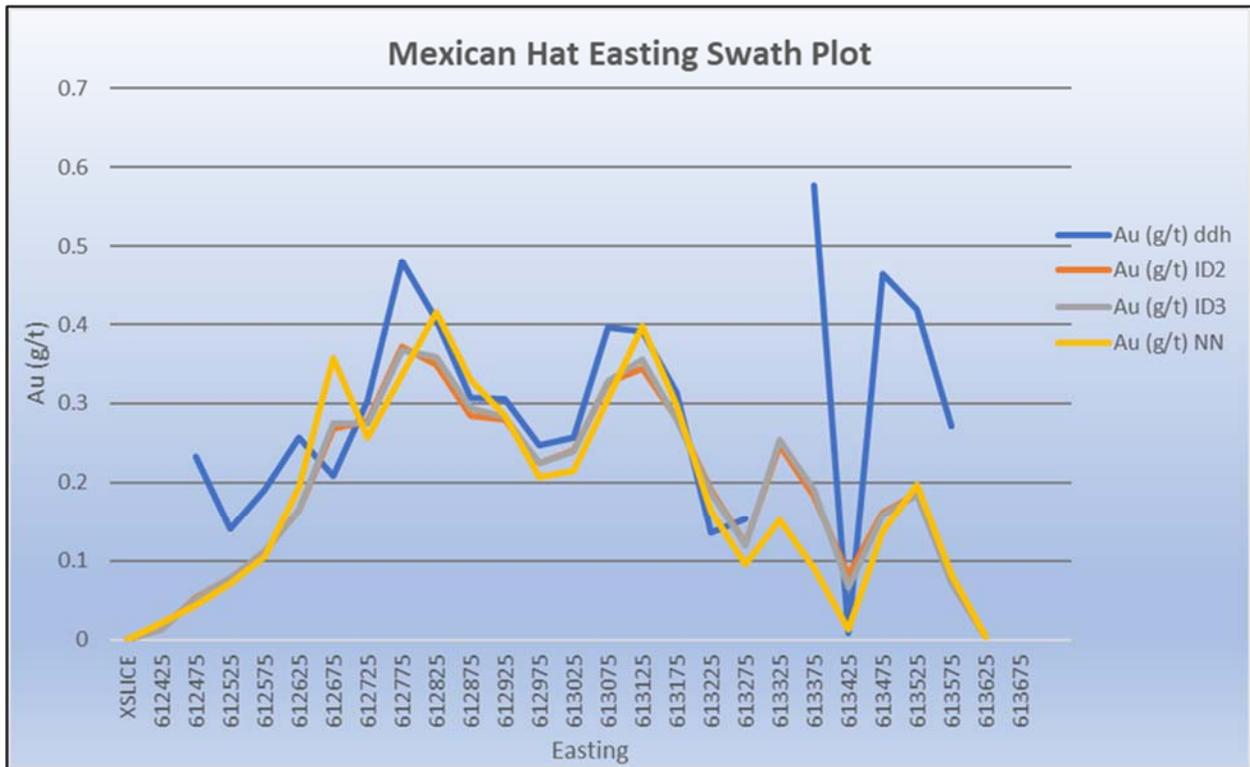
Figure 14-11: Swath Plot along Eastings for the Mexican Hat Model

Figure 14-12: Swath Plot along Elevations for the Mexican Hat Model

15   MINERAL RESERVE ESTIMATES

All material at Mexican Hat is categorized as inferred material. There is no material that can be considered a reserve.

16 MINING METHODS

This section summarizes the results of a mine plan that was developed as part of the Preliminary Economic Assessment (PEA) evaluation of the Mexican Hat deposit. The reader is cautioned that this mine plan is based on a Mineral Resource which is entirely an inferred category mineralization. The reader is cautioned that Mineral Resources are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the Mineral Resources will be realized or that they will convert to Mineral Reserves.

There is no Mineral Reserve at Mexican Hat at this time. A component of the Mineral Resource, inclusive of inferred mineralization has been designated as potentially minable material for this PEA.

The Mexican Hat Project is planned as a conventional hard rock surface truck-and-loader mining operation producing 15,000 tpd (5,475 kt/a) of Inferred Mineral Resource feed for the heap leach pad supported with a total mining rate of 58,000 tpd (20,880 kt/a) of material. The mine life is approximately 5-6 years where there will be a year of preproduction and a partial year in year 5.

The mine production schedule was developed with the goal of filling the plant at required feed rate while maximizing the project return on investment. The mine is scheduled to operate 365 days/yr with two, 12 hour shifts/day with an allowance of 5 days of lost per year due to weather or other delays.

Pit bench heights will be 6 meters high for mining. Drilling will be completed with a fleet of five rotary blast hole rigs with 23,500 kg pull down force. Blast holes will be drilled with 6 3/4 in. (17.15 cm) diameter holes. The blasted rock will be loaded into 100 t (89.4 tn) haul trucks using three 11.5 cu m front end loaders.

The mine plan was developed with a phase approach. The phase designs, mine schedule, and mine equipment requirements are summarized in this section.

Within this section, the following units are utilized. Tonnes mean metric tons. Ktonnes means 1,000 metric tonnes. Gold grades are in grams per metric tonne (gpt or gm/tonne). All tonnages reported in the mine plan are dry tonnes.

Mining Modifying Factors

The model received by IMC was a nonstandard block size, sub-block block model that was developed by Tetra Tech per Chapter 15. In preparing the block model for mine planning usage, IMC applied mining modifying factors in which the sub-blocks were diluted to 6x6x6 m whole blocks. Where the summation coverage of the model estimated nonstandard sub-blocks did not constitute a whole block, the remainder of the whole block was diluted with zero grade waste material. The sub-block model also did not reference a practical bench level designation applicable to a surface open-pit operation and so it was re-blocked to a whole block model that translated the block bench levels down in elevation by 0.06 m. This resulted in bench levels that were whole numbers – for example, the 1414 level as opposed to the original 1414.06 level.

16.1 PHASE DESIGN

A pit optimization algorithm was used as a guide to the design of the phases or pushbacks. Multiple optimized pits were developed using the costs, slope angles and recoveries are outlined in Table 16-1. The optimized pits were run in increments of metal prices in order to establish a series of multiple nested cone geometries. The results of this work laid out a framework for the starting point, final pit and the extraction sequence that maximized the NPV throughout the mine life.

Phases or pushbacks are incremental expansions of the mine. Designed phases include proper mine access to every bench and assure practical working room and geometries for the mine equipment. A phase does not represent a time period. Multiple phases are operated at any time during the mine life.

Mexican Hat has two independent pits with a total of five phases. The main pit, Pit 1, is comprised of four phases and, pit 2 is a single phase satellite pit. Incremental pits were run based on a revenue factor from 25-100 of the base metal price as seen on Table 16-1. Pit 95 was chosen for design of the ultimate pit limit with aspects of each pit selected based on mine operating constraints. Pits 50, 65, and 75 were used to guide the design of the pit phases. Figure 16-2 illustrates on the 1414 bench map the approximate locations of all of the different phases derived from the optimized pit analysis.

Figure 16-1 is a map of the optimized pit at a revenue factor of 0.95. Note the small satellite pit between the main pit and the far southeast satellite (located at 613,400 E, 3,518,800 N) did not prove to be economically positive with the addition of a road design and so was not included in the final mine plan as can be seen in Figure 16-2.

The following mine planning parameters were assumed in the design of the pit phases. These follow the pit optimization parameters as laid out in Table 16-1.

Mine Planning Parameters:

Haul Road Width	25 meters
Haul Road Grade	10% Maximum
Interramp Slope Angles	45 degrees
Operating width between pushbacks	100 meters nominal

The phases were tabulated from the block model and those five tabulations were used as input to the development of the mine production schedule. Figure 16-2 illustrates the relative position of the pits and phases on the 1414 elevation.

Table 16-1: Pit Optimization Parameters

Base Case Metal Prices		
1,250 $/ozt-au Gold Price (Rev. Fact. 1.0)		
Operating Costs		
<u>Units</u>	<u>Type of Cost</u>	<u>Source</u>
1.95 $/tonne	Mining Cost	Based on Similar 2018 NV Gold Project
5.07 $/tn feed material	Processing Cost	M3 Preliminary Process Cost
<u>1.32 $/tn feed material</u>	<u>Processing G&A Cost</u>	M3 Preliminary Process G&A Cost
6.39 $/tn feed material	Process + G&A Cost	
Processing Recovery		
Up to 0.55 gpt	Apply (Gold-0.055)/Gold*100% Recovery	
Above 0.55 gpt	Apply 90% Recovery	
Refining Charges	**Royalty**	
5 $/oz-au, Refining Charge	3.0% Hernandez Royalty	
100.0% Payable Gold	(Victor Claims)	
Cut-off Grades	**Discount Rate**	
Au (gpt)		
Internal 0.22	7.0% Annual Discount Rate	
Breakeven 0.27	0.7% Discount Rate/Bench	
Slope Parameters		
45 Deg, Inter-ramp Slope Angle		
40 Deg, Overall Slope Angle		
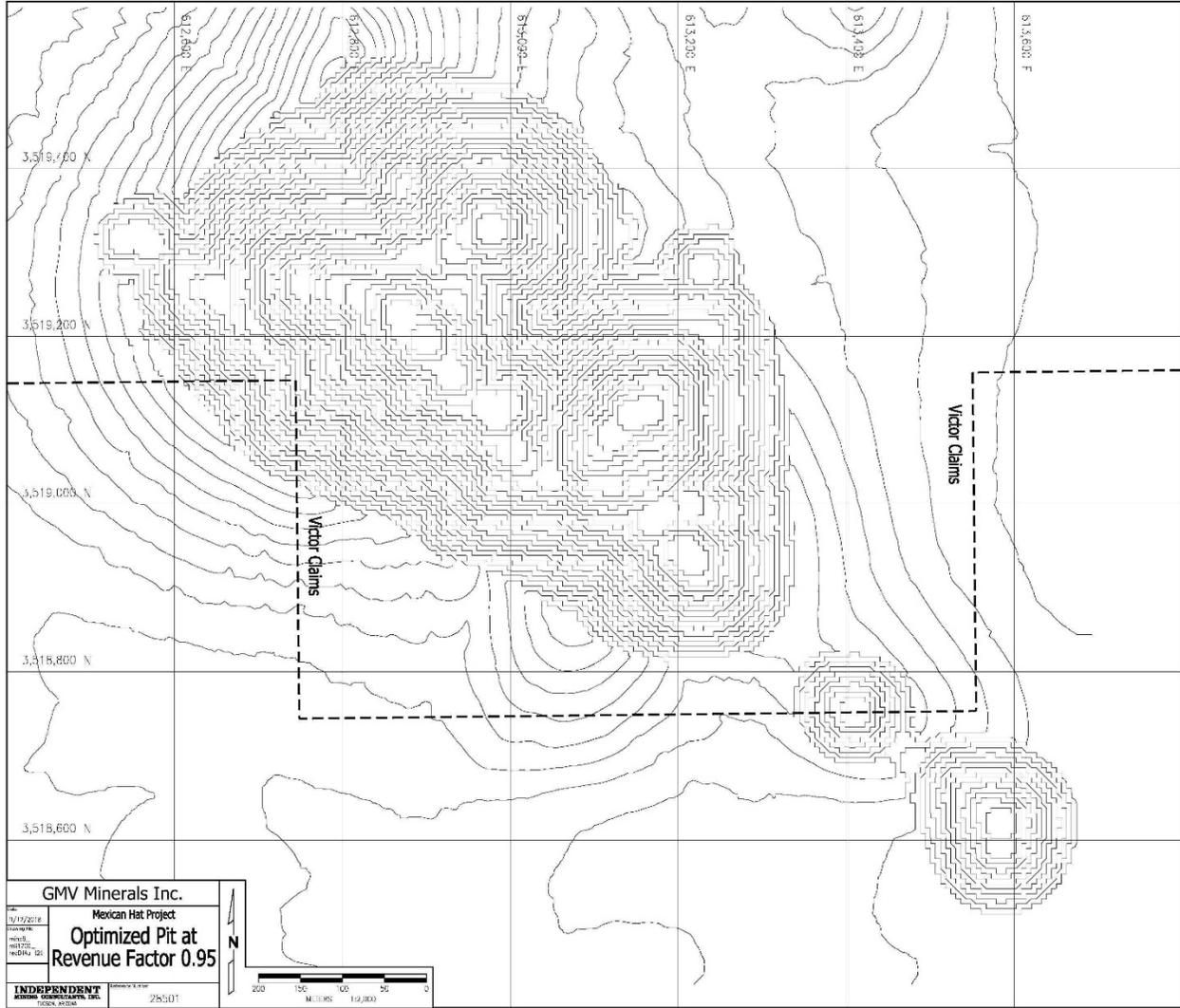
Figure 16-1: Optimized Pit (Revenue Factor of 95)
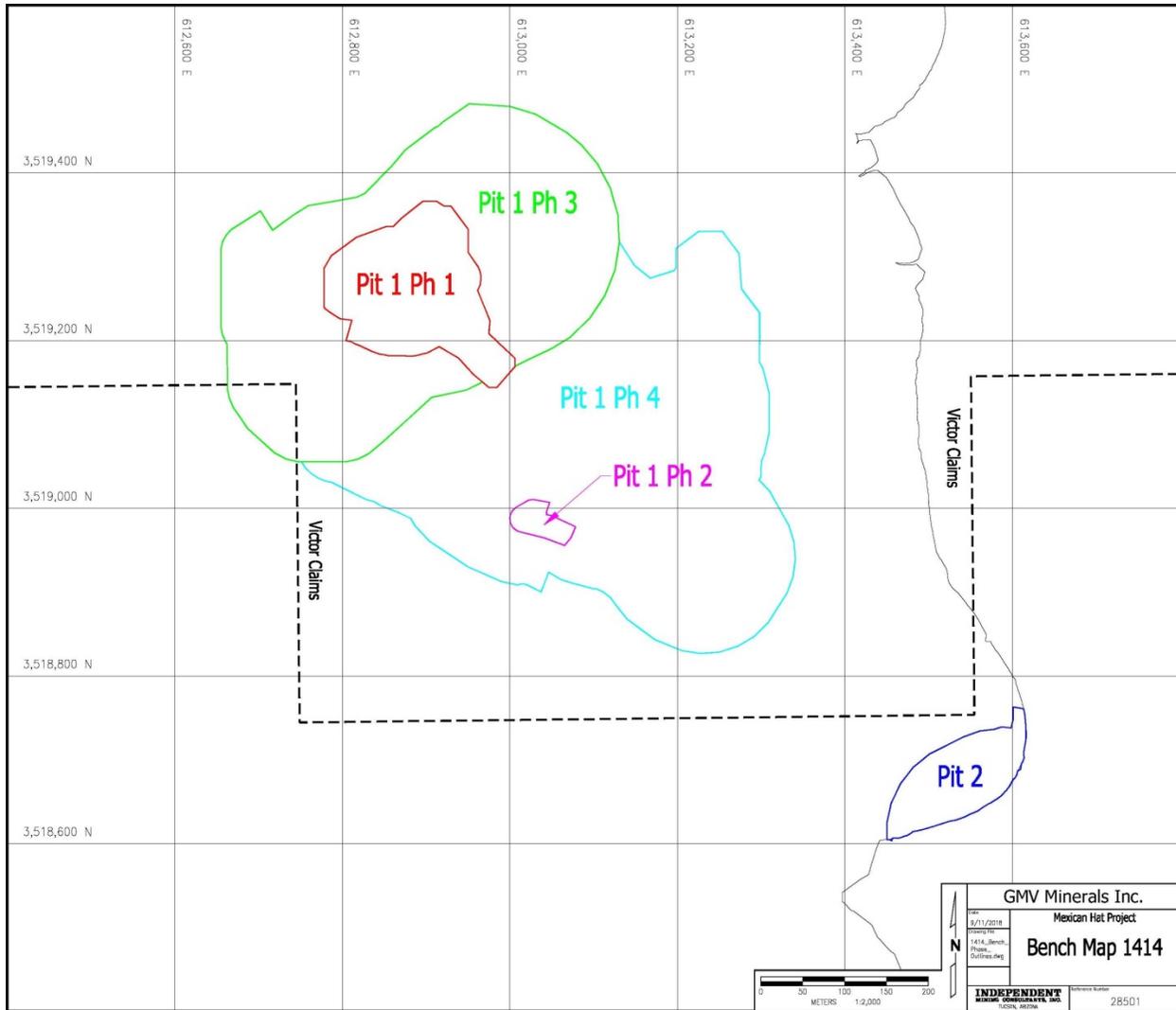
Figure 16-2: Relative Location of Pits and Phases, 1414 Elevation

16.2   MINE PRODUCTION SCHEDULE

The mine production schedule was developed with the goal of assuring ore availability at the required production rates and maximizing the project return on investment while respecting practical mine operational constraints. Multiple mine production schedules were developed that analyzed alternative cutoff grade strategies versus mine total material movement on a preliminary NPV basis. Total material rates are tied to the size and number of loading units so that the final selected schedule would provide efficient use of the capital equipment employed.

Multiple schedules were evaluated on a net present value basis at the project design prices. The best overall production schedule on an economic and practical basis was selected and is summarized on Table 16-2.

Preproduction stripping in Phase 1 of Pit 1 occurs in year 0 (preproduction) before the process plant construction is started. Road pioneering to the peak of Mexican Hat will also be a high priority during preproduction to build the infrastructure necessary for Pit 1. As Phase 1 of Pit 1 is mined, high grade feed material will be stockpiled in preproduction until the plant begins operation in year 1. The temporary high-grade stock pile is required because the early waste requirements that are necessary to sustain feed material release of 15,000 tpd incur small amounts of that

are stored for future processing. During preproduction, the mine will stockpile 1 Mtonnes (1,000 Ktonnes) of high-graded feed material and 199 Ktonnes of low-grade feed material.

In year 1, the mine will reach the nominal production capacity of 58,000 tpd. The mine will deliver the stockpiled 1Mtonnes of feed material and 4.475 Mtonnes of feed material produced from the pit.

The nominal production capacity of 58 ktpd is maintained until the mine finishes developing all phases in year 5. At the end of the pit life, the mine will deliver 1.165 Mtonnes of stockpiled low-grade feed material to the process plant.

Cost and recoveries on the pit optimization parameter Table 16-1 were used to calculate the internal, breakeven, and low-grade cut-off grades of 0.22 gpt-au, 0.23 gpt-au, and 0.25 au-gpt. These cut-off grades were used as guidance in the development of the mine production schedule.

The total feed material that is planned for processing on Table 16-2 is inferred mineralization. The reader is cautioned that this mine plan is based on a mineral resource which is entirely an inferred category mineralization. The reader is cautioned that mineral resources are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the mineral resources will be realized or that they will convert to mineral reserves.

There is no mineral reserve at Mexican Hat at this time. A component of the mineral resource, inclusive of inferred mineralization has been designated as potentially minable material for this PEA.

Figure 16-3 through Figure 16-8 illustrate the annual progression of the Mexican Hat pits and rock stockpiles based on the production schedule shown in Table 16-2.

Table 16-2: Mexican Hat Mine Production Schedule

Mexican Hat Mine Production Schedule								
	Direct Crusher Feed			Low-grade Feed >=0.25 Au-gpt				
Year	Cut-off Grade au-gpt	Feed Ktonnes	Gold au-gpt	Rec Gold au-gpt	LG Ktonnes	Gold au-gpt	Rec Gold au-gpt	Waste Ktonnes	Total Ktonnes
0	0.30	1,000	0.702	0.623	199	0.273	0.218	3,971	5,170
1	0.30	4,475	0.769	0.683	966	0.273	0.218	15,439	20,880
2	0.22	5,475	0.634	0.559				15,405	20,880
3	0.22	5,475	0.542	0.474				15,405	20,880
4	0.22	5,475	0.671	0.592				15,413	20,888
5	0.22	2,045	0.991	0.882				5,643	7,688
Total		23,945	0.680	0.601	1,165	0.273	0.218	71,276	96,386

Process Production Schedule				
Year	Cut-off Grade au-gpt	Feed Ktonnes	Gold au-gpt	Rec Gold au-gpt
0		0		
1	0.30	5,475	0.757	0.672
2	0.22	5,475	0.634	0.559
3	0.22	5,475	0.542	0.474
4	0.22	5,475	0.671	0.592
5	0.22-0.25	3,210	0.730	0.641
Total		25,110	0.661	0.583

Figure 16-3: End of Pre-Production

Figure 16-4: End of Year 01

Figure 16-5: End of Year 02

Figure 16-6: End of Year 03

Figure 16-7: End of Year 04

Figure 16-8: End of Year 05

16.3   LOW-GRADE STOCKPILE STORAGE

The low-grade stockpile storage facility has an initial capacity of 1.4 Mtonnes that is reduced by the pit footprint later in the mine life to 1.2 Mtonnes. In preproduction, the low-grade stockpile will also temporarily store high-grade feed material until it is reclaimed to the process plant in year 1. At the end of the mine life, the stockpile is reclaimed entirely and fed to the process plant. The stockpile facility is temporary in nature, so it has been designed at angle of repose (37 degrees).

The low-grade stockpile storage facility is built to capacity and can be seen in period map Year 01 Figure 16-4.

16.4   WASTE STORAGE

The annual mine schedule drawings illustrate the location and size development of the waste storage facilities. The end of mine life waste storage facilities can be in period map Year 05 Figure 16-8.

Mexican Hat will maintain 2 waste storage facilities, the East Dump and the West Dump. The East Dump over the mine life will hold 34.8 Mtonnes of waste while the West Dump will hold 13.4 Mtonnes of waste. The East Dump facility will be developed first due to an initial shorter haul distance. In the second half of the mine life, hauls to the two dumps are balanced to maintain a consistent truck haulage fleet.

The waste storage facilities are built at 2.5 to 1 slope (21.8 degrees) in order to facilitate reclamation at the end of the mine life. Dump lifts are 25 m high and are dumped at angle of repose (37 degrees) with 29.3 m setbacks left between lifts to maintain the 2.5 to 1 overall angle for future reclamation. The dumps were designed to avoid the property boundary and major drainage washes on the property. A 100' foot buffer zone was built into the designs to prevent encroachment on the drainage or claim boundary.

There is no provision for re-contouring of the waste dumps within the mine operating costs. Mine reclamation costs are not included within the mining costs.

16.5   MINE EQUIPMENT REQUIREMENTS

Mine equipment is comprised of standard off-the-shelf units. Three front end loaders were selected to match production requirements based on the mine production schedule.

Truck fleet requirements were developed from haul time simulation over profiles measured for each material type, by phase, for each year of the mine plan.

Table 16-3 summarizes the major mine equipment units that will be on site throughout the mine life. Additional minor units are included in the capital cost table in Section 21 that will be required to maintain and sustain mine operations. The mine will lease the mine equipment throughout the mine life.

Table 16-3: Mine Major Equipment Fleet on Hand
(Units owned based on fleet build up and replacement)

Equipment Type	Time Period					
	Preprod	Y01	Y02	Y03	Y04	Y05
CAT MD6290 Diesel Rotary Drill (17.1454 cm)	1	5	5	5	5	5
CAT 992 Front End Loader (11.5 CuM)	1	3	3	3	3	3
CAT 777 Haul Truck (87.9 tn)	2	11	11	11	11	11
CAT D8 Track Dozers (264 kW)	2	2	2	2	2	2
CAT 834K Wheel Dozer (12.232 CuM)	1	1	1	1	1	1
CAT 16M Motor Graders (16 ft blade)	1	1	1	1	1	1
CAT 745WT Water Truck (30,000 Ltr)	1	1	1	1	1	1
CAT 745 Aux Truck (41 tn)	1	1	1	1	1	1
CAT 390 Excavator (4.587329 CuM)	1	1	1	1	1	1
Total	11	26	26	26	26	26

16.6 MANPOWER REQUIREMENTS

Mine hourly manpower requirements were established to operate and maintain the mine mobile equipment. Laborers necessary for the mine will be equipment operators and drivers, blasting crews, maintenance crew, technicians, and general laborers. An hourly labor crew of about 125-135 will need to be kept on staff depending on the stage of the mine in the mine life. Preproduction will necessitate a smaller subset of the staff as the mine moves to nominal mine production – the similar workforce subset will also be maintained in the last year of the mine life.

Staff labor (Supervisory, Engineering, and Geology) requirements are estimated to be 46 persons for all years except preproduction and the late years of the mine life when a smaller staff can be utilized.

17   RECOVERY METHODS

17.1   PROCESS FLOW SHEET

The Mexican Hat gold deposit will be mined as an open pit. Precious metal recovery from the mine in this Preliminary Economic Analysis (PEA) is through conventional heap leaching and adsorption, desorption, regeneration (ADR) technology for metal extraction from crushed feed. Processing will involve feed passing through a three stage crushing plant, which will allow for belt conveyor stacking of the material onto a heap leach pad. The processing facilities accommodate a leachable resource of approximately 25.1 Mt of material at a gold grade of 0.66 g/mt and a process rate of 15,000 mtpd. The heap leach pad has been located and designed with expandability for a feed increase.

Over the life of the mine, feed will be delivered to the stockpile adjacent to the crushing plant. The material will be fed to the crushing plant using a front-end loader, and will be crushed and transported to the heap leach pad via an overland conveyor. The feed will be stacked onto the heap using a radial stacker and then leached with a weak cyanide solution to extract the precious metal values. The gold will then be recovered from the pregnant solution in the carbon plant by adsorbing the dissolved gold onto activated carbon followed by desorption, electrowinning, retorting and smelting to recover the gold as a final doré product.

The process flow sheet is shown in Figure 17-1.
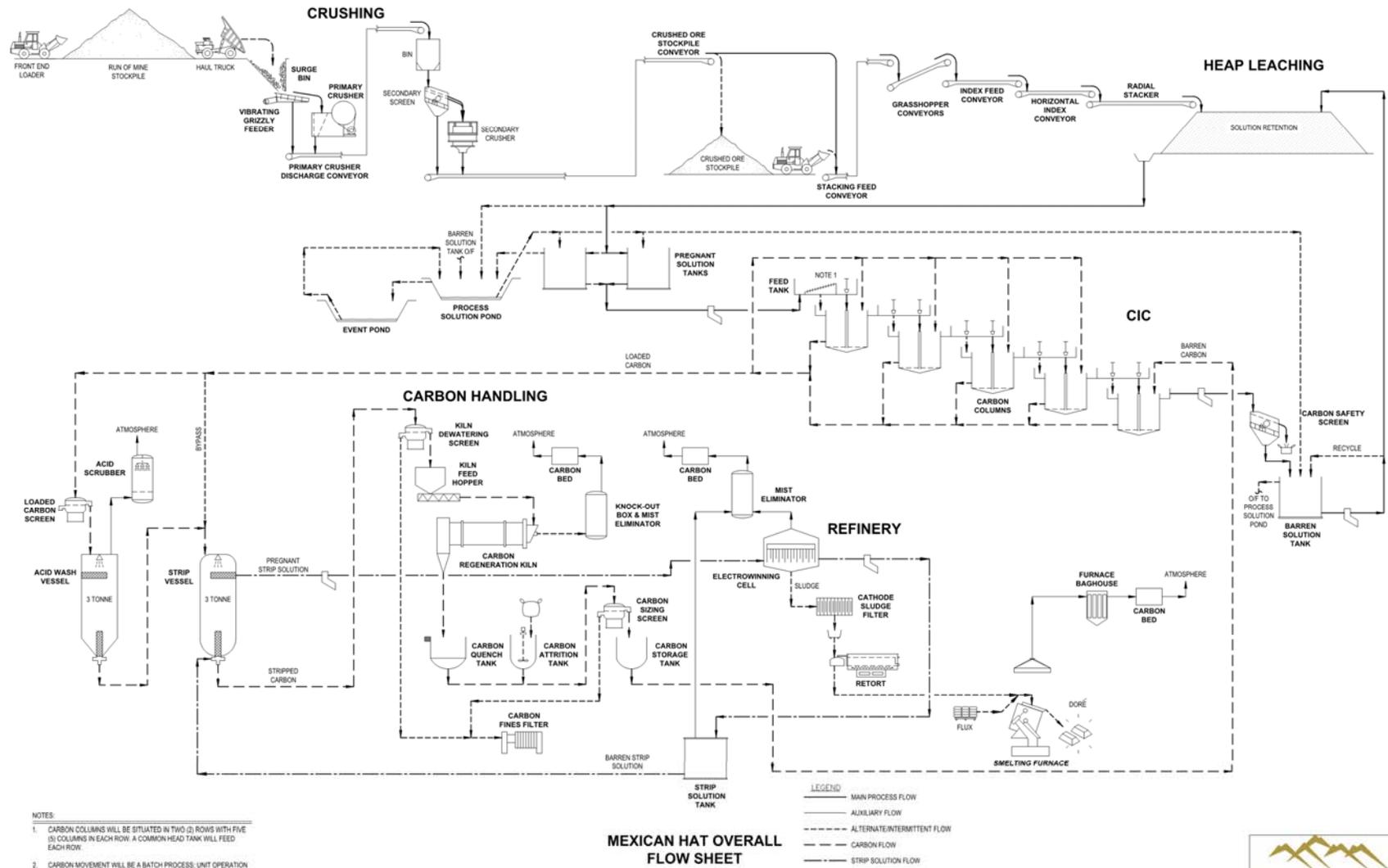
Figure 17-1: Process Flow Sheet

17.2 CRUSHING AND STACKING

The crushing and stacking circuits are designed for 833 dry tonnes per hour (mtph), with operations scheduled for 24 hours per day, seven days per week, 365 days per year. At an expected 75% operating availability, this will result in placing an average of 15,000 mtpd onto the heap leach pad, equivalent to 5,475,000 tonnes per year.

Run-of-mine (ROM) material will be trucked from the mine to a stockpile close to the primary crusher and subsequently reclaimed with a front-end loader (FEL). The FEL will dump feed into a surge bin. A vibrating grizzly feeder will draw feed from the surge bin, with the feeder oversize reporting to a jaw crusher. The grizzly feeder undersize material will bypass the primary crusher and will combine with the crusher product on the primary discharge belt conveyor.

The grizzly undersize and the primary crushed material will be conveyed to a bin that will feed the secondary screen. Screen oversize will report to the secondary crusher. The undersize fraction from the screen will be conveyed to the crushed stockpile. The secondary crusher product will be combined with the undersize fraction from the secondary screen on the conveyer that delivers to the crushed stockpile.

Crushed material will be reclaimed by a front end loader and transferred to a stacking feed conveyor. The stacking feed conveyor discharges onto a series of several mobile ramp conveyors and mobile grasshopper-type conveyors. Units of mobile conveyors and grasshopper-type conveyors will be added or removed as required, dependent upon the stacking location on the pad. The final conveyor will be a radial-type mobile stacker that will place material in lifts, up to 9 meters in height.

Table 17-1 provides the key crushing and stacking process design parameters.

Table 17-1: Key Crushing and Stacking Process Design Parameters

Crushing and Stacking	Unit	Design
Crushing and Stacking Process Rate	mtpd	15,000
Crushing and Stacking Throughput Rate	mtph	833
Primary Grizzly Aperture	mm	75
Secondary Screen Aperture – bottom deck	mm	25
Tertiary Screen Aperture	mm	25
Ore Bulk Density	mt/m^3	1.6
Crushing Work Index	kWh/t	15.4
Stacked Ore Height	m	9
Crushing Plant Operating Availability	%	75

17.3 HEAP LEACH PAD DESIGN

The following is a summary of the proposed heap leach pad design. Key heap leach process design parameters are provided in Table 17-2.

Table 17-2: Key Heap Leach Process Design Parameters

Heap Leaching	Unit	Design
Lift Height	m	9
Solution Application Rate	L/m²/hr	11.0
Leach Cycle	days	100
Tons Under Leach	ktonnes	1,500
Cyanide Concentration	kg/mt soln.	0.5
Leach Solution pH	pH	10.5 to 11.0
Pregnant Solution Flow Rate	m³/hr	1,022
Barren Solution Flow Rate	m³/hr	1,136
Pregnant Solution Tank Capacity	m³	100
Metal Recovery Plant Operating Time	%	98

The heap leach pad will be designed to be constructed in phases to Arizona Department of Environmental Quality (ADEQ) and Best Available Demonstrated Control Technology (BADCT) standards. The leach pad extends from an elevation of 1,398 meters at the toe of the process ponds to an elevation of 1,412 meters at the western edge of the leach pad. The grade of the lined base receiving feed will range from 1 to 2%. In total, the leach pad will have a total lined area of 46.1 hectares. The final reclaimed surface of the leach pad will be graded to 3H:1V (horizontal to vertical) side slopes and will be covered with 0.6 m of growth media. The leach pad design provides a total capacity of approximately 25.1 Mt using an average dry density of 1.5 tonnes/m³ for stacked feed.

Prior to development, the footprint of each facility will be cleared and grubbed of existing vegetation and topsoil. Cut-to-fill regrading will be utilized where possible to minimize earthworks requirements. Excess soil removed from the base of each phase will be stockpiled for later use as growth media to provide cover to be placed over the finished leach pad at the end of the Project.

17.4   PROCESSING PLANT DESIGN AND OPERATIONS

17.4.1   Metal Recovery Plant Design and Operations

A carbon ADR circuit will be used at the Mexican Hat Mine to recover gold and silver from the pregnant solution. The ADR plant will recover and send concentrated gold solution to the refinery where the final marketable doré bars are produced. Pregnant solution from the heap leach pad will flow by gravity to the pregnant solution tanks. From the pregnant solution tanks, the solution will be pumped to the ADR plant, where soluble gold and silver will adsorb onto activated carbon. Adsorption of the gold onto activated granular coconut shell carbon will be conducted in a five-stage counter current carbon-in-column (CIC) circuit. Carbon is advanced from column to column counter current to the pregnant solution flow so that the highest-grade carbon contacts the highest grade pregnant solution and the most active fresh carbon contacts the lowest grade solution.

Loaded carbon from the first adsorption column will be pumped to an acid wash vessel and will be acid washed by circulating dilute hydrochloric acid upwards through the bed of carbon to remove scale build-up (mainly calcium) to maintain the carbon's ability to recover gold and expose the surface to improve gold elution efficiency. Residual acid in the acid wash vessel will be neutralized with caustic before the loaded carbon is transferred to the strip (elution) vessel. Hot caustic cyanide solution will be pumped through the strip vessel to remove the gold and silver from the loaded carbon. Elution is conducted at 100 psi and 150°C for up to ten hours. Sodium hydroxide will be added to the stripping solution to aid stripping and provide electrolyte for the subsequent electrowinning stage.

Stripped carbon will be transferred to the regeneration circuit where carbon will be thermally regenerated in a horizontal rotary kiln or the carbon may be sent directly to the sizing circuit and returned to the carbon columns. Stripped carbon will be washed with water then screened to remove fines prior to being fed to the regeneration kiln. There it is heated

to approximately 760 ˚C in a moist, oxygen-free atmosphere to reactivate its surfaces before it is reused in the carbon columns.

Recovery of precious metals from the rich pregnant strip solution will be conducted in a single electrowinning (EW) cell. Electrowinning removes the precious metals from the pregnant solution by passing direct current through an electrowinning cell. The rich strip solution is transferred to the electrowinning cells. The precious metal ions transfer from the solution to the stainless-steel wool cathode and deposit onto the steel wool as a weakly bonded sludge. The barren EW solution is then returned to the stripping circuit, completing the elution cycle. The barren strip solution from the electrowinning cell will be collected in the EW barren return tank and pumped to the strip solution tank for reuse in the strip circuit. The sludge in the electrowinning cell will be washed off the cathodes in batches and recovered as a damp cake in the cathode sludge filter press.

The filtered cake will be retorted to remove and recover mercury prior to smelting the dried sludge to produce Doré bars. Filtered cake will be collected in pans. The pans will be placed in a mercury retort system for several hours. The retort will heat the filtered cake to approximately 650˚C to vaporize mercury. The retort temperature will be ramped up gradually to enable the sludge to dry completely before mercury is vaporized and to allow time for the mercury to diffuse to the solid surfaces. Retort vapor will be withdrawn from the retort by a vacuum pump, which will pull the vapor through a condenser where the mercury will condense and flow into a mercury collection compartment. Mercury will be removed as required.

Following a cooldown period, the dried (retorted) cake will be mixed with fluxes and charged to an electric induction furnace and heated to approximately 1,230°C. When the furnace charge is fully molten, it separates into two distinct layers: the slag (on the top) and metal (on the bottom). The slag layer, containing fused fluxes and impurities, will be poured first into conical pots. Once slag has been removed, the melted gold and silver (metal layer) will be poured into molds to form Doré bars.

Bars will be cooled, cleaned, weighed, and stamped with an identification number and weight. Doré bars will be the final product of the plant which will be shipped to the market at 90-95% Au/Ag purity.

Table 17-3 provides the PEA design parameters for the ADR Plant.

Table 17-3: Key ADR Process Design Parameters

ADR Plant	Unit	Design
CIC Adsorption Circuit		
Column Carbon Capacity	tonnes/column	3.0
Column Specific Flow Rate	$m^3/min/m^2$	0.97
Carbon Loading	g Au+Ag/mt carbon	4,000
Adsorption Efficiency	%	99
Operating Time	%	98
Desorption Circuit		
Column Carbon Capacity	tonnes/strip	3.0
Flow Rate	m^3/hr	15
Elution Temperature	deg. C	150
Elution Pressure	psig	100
Elution Time	hrs	10
NaOH Concentration	%	2.0
Efficiency	%	95
Acid Wash		
Column Carbon Capacity	tonnes	3.0
Hydrochloric Acid Concentration	%	3.0

ADR Plant	Unit	Design
Carbon Reactivation		
Throughput Rate	kg/hr	300
Temperature	deg. C	760
Mercury Retort		
Temperature	deg. C	650
Electrowinning		
Flow Rate	m³/hr	15
Rectifier	kW	18
Current Density	A/m²	200
Efficiency	%	90-97
Smelting		
Temperature	deg. C	1,230
Pour Per Month		8

17.5 CONSUMABLE REQUIREMENTS

17.5.1 Power

Power will be supplied by Sulfur Springs Valley Electric Cooperative at the cost of $0.08 per kWh. The capital costs to bring line power to site are included in the project financial model.

17.5.2 Water Supply

The make-up water requirement for the Project is 160 m³/hr. This is projected from the heap leach make-up requirement and mine dust suppression as the two major points of use. The water source for the Project will be from water wells on site, including pit dewatering wells.

Table 17-4: Water Consumption

Reagent	Use
Mine Dust Suppression	45 m³/hr
Heap Leach Make-Up	110 m³/hr
General Plant Usage	5 m³/hr
Total	**160 m³/hr**

17.5.3 Major Reagents

Major reagents and usage for the heap leach operation are provided in Table 17-5. Lime and Cyanide reagent consumption was determined during metallurgical test work performed by McClelland Laboratories Inc. in 2015 and 2016. The other reagent consumptions were projected by M3 using data from existing properties with similar processes.

Table 17-5: Major Reagent Consumption

Reagent	Use
Lime	1.5 kg/mt
Sodium Cyanide	0.30 kg/mt
Caustic Soda	0.025 kg/mt
Antiscalant	0.015 kg/mt
Hydrochloric Acid	0.04 kg/mt
Carbon	0.025 kg/mt
Refinery Fluxes	0.0125 kg/mt

18   PROJECT INFRASTRUCTURE

The proposed Mexican Hat Project includes open pits, waste rock storage, a heap leach pad, a conventional adsorption desorption regeneration (ADR) process plant with ancillary facilities.

Figure 18-1 shows an aerial view of the conceptual site infrastructure.
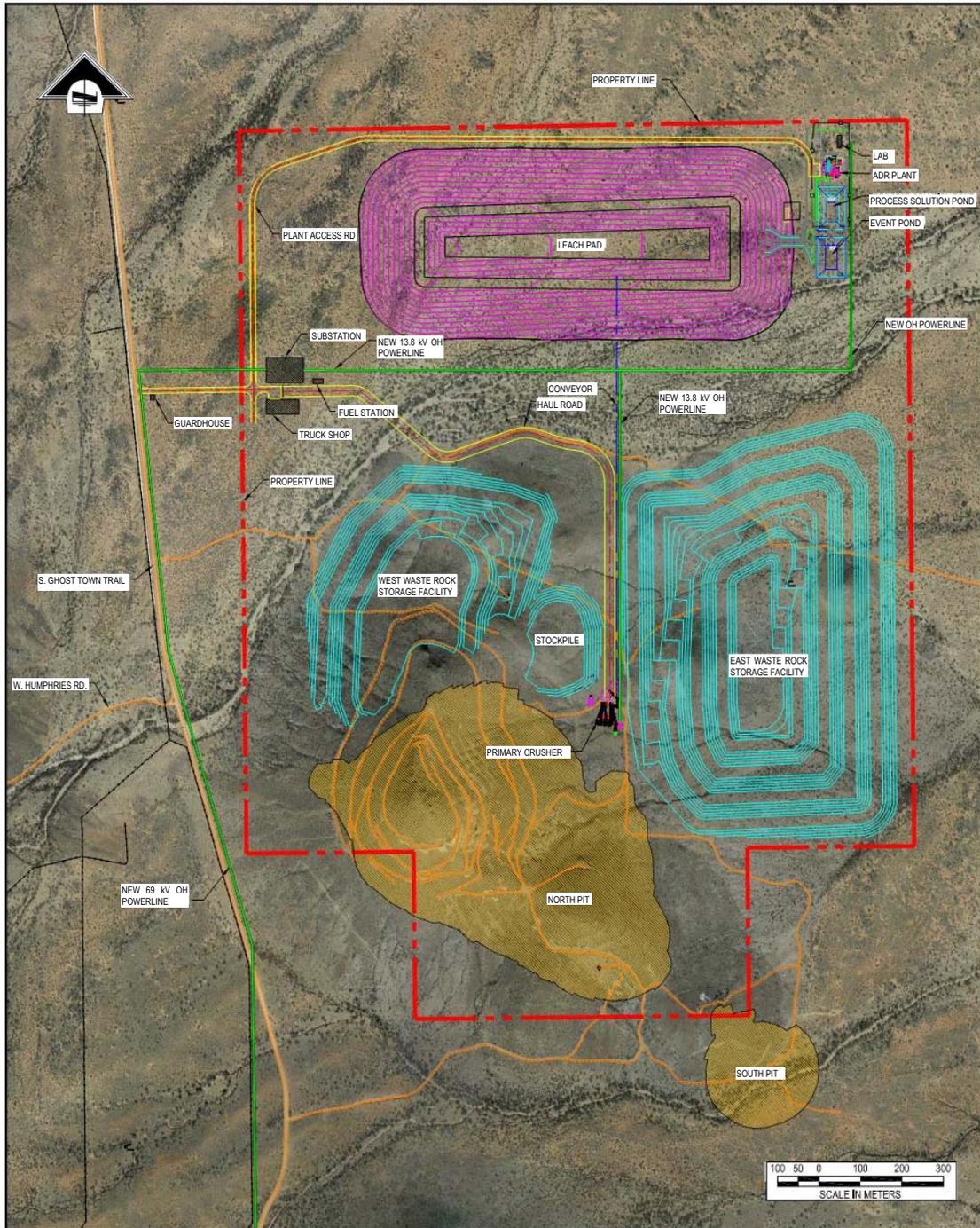
Figure 18-1: Overview of Mexican Hat Conceptual Site Infrastructure

18.1   ACCESS

The property is located in Cochise County, immediately north of the Gleeson Courtland district, 10 km (6 miles) south of Pearce, Arizona, and approximately 140 km (90 miles) by road from Tucson, Arizona. Road access to the site is all paved, other than the 10 km gravel section south of Pearce.

Wilcox and Benson are larger communities located about a 30 minute drive away from Pearce that offer amenities and accommodations.

Figure 18-2: Location of Mexican Hat Project, Cochise County, Arizona

18.2   POWER SUPPLY

Sulphur Springs Valley Electric Cooperative (SSVEC), located 30 km north of the project site, is not for profit, member-owned distribution cooperative providing electricity to more than 51,000 services over some 6,800 km of energized line. Located in southeastern Arizona, the cooperative's service territory covers parts of Cochise, Graham, Pima and Santa Cruz Counties and includes the communities of Sierra Vista, Huachuca City, Patagonia, Elfrida, Benson, St. David, Bowie, San Simon, Wilcox, Sonoita and Pearce-Sunsites.

SSVEC will extend a 69 kV powerline to the project site, for the 6 MW load. GMV will construct a substation on the project site to drop the voltage to 13.8 kV.

18.3   WATER SUPPLY

Groundwater has been identified as the best source for a water supply. There is a regional aquifer in the alluvial deposits that occur in the region. The aquifer is exploited by large-capacity wells that support extensive agricultural irrigation. Based on the pumpage rates of the wells, it is believed that water supply will be sufficient to meet the project demand. A formal hydrogeologic study will need to be conducted during the next phase of project advancement to characterize local water quality and supply.

19   MARKET STUDIES AND CONTRACTS

No market studies have been completed or planned for gold production.

20   ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1   INTRODUCTION

The Mexican Hat Project will require various state and federal authorizations, licenses and permits for project construction, operation, closure and post-closure. Comprehensive environmental and socio-economic baseline studies will be required. A description of the anticipated permitting process is described in this section of the PEA.

20.2   ENVIRONMENTAL SETTING

Southeastern Arizona is part of the Basin and Range physiographic province, which is characterized by northwest-southeast trending mounting ranges separated by broad alluvial valleys. The Mexican Highland section is a higher elevation area of the province with valleys ranging from 2,500 to 4,000 feet above sea level. A unique feature of the area is the presence of mountain ranges that are isolated from each other by valleys of desert grasslands and desert scrub. These "sky islands" are part of a complex of about 27 mountain ranges in Arizona, New Mexico, and the bordering Mexican states of Sonora and Chihuahua. The sky island encompasses climate zones from subtropical to temperate latitudes, a condition found nowhere else.

The isolation has significant implications for these natural habitats, including endemism, altitudinal migration and relict populations. Although the project is not within a sky island, the surrounding mountain ranges are of concern for biodiversity and conservation groups. At the project site, it is expected that some protected species will be present, such as desert tortoises, rattlesnakes and Gila monsters, which have already been observed.

Both the Willcox Playa and San Pedro River, natural water features in the region, are sensitive biological areas for migratory birds. It is unlikely that bird populations will be attracted to any future mining operations since the size and number of ponds will be relatively small.

Portions of the property have already been disturbed by previous exploration and/or mining operations. No evidence of mineral processing activities was noted. The property has not been given any known environmental designations that would preclude mining operations.

20.3   WATER SUPPLY HYDROLOGIC SETTING

The Arizona Department of Water Resources has identified a variety of groundwater basins in southeastern Arizona. The Mexican Hat Project is located in the Douglas Basin, a hydrologic feature that occupies the southern portion of a northwest-southeast trending, structural trough that extends from the central part of the Aravaipa Canyon Basin, through the Willcox Basin, to the northeastern part of Sonora, Mexico (Figure 20-1). The Douglas Basin comprises the southern part of the Sulphur Springs Valley, which is bounded by the Chiricahua and Mule Mountains to the east, and the Dragoon and Little Dragoon Mountains to the west. The Willcox Playa, a 50-square mile endorheic (closed) basin, is a well-known feature in the northern part of the Sulphur Springs Valley. The hydrologic basins surrounding the Douglas Basin are the Willcox Basin to the north and northeast, Upper San Pedro Basin to the west and San Bernardino Valley Basin to the east and south.
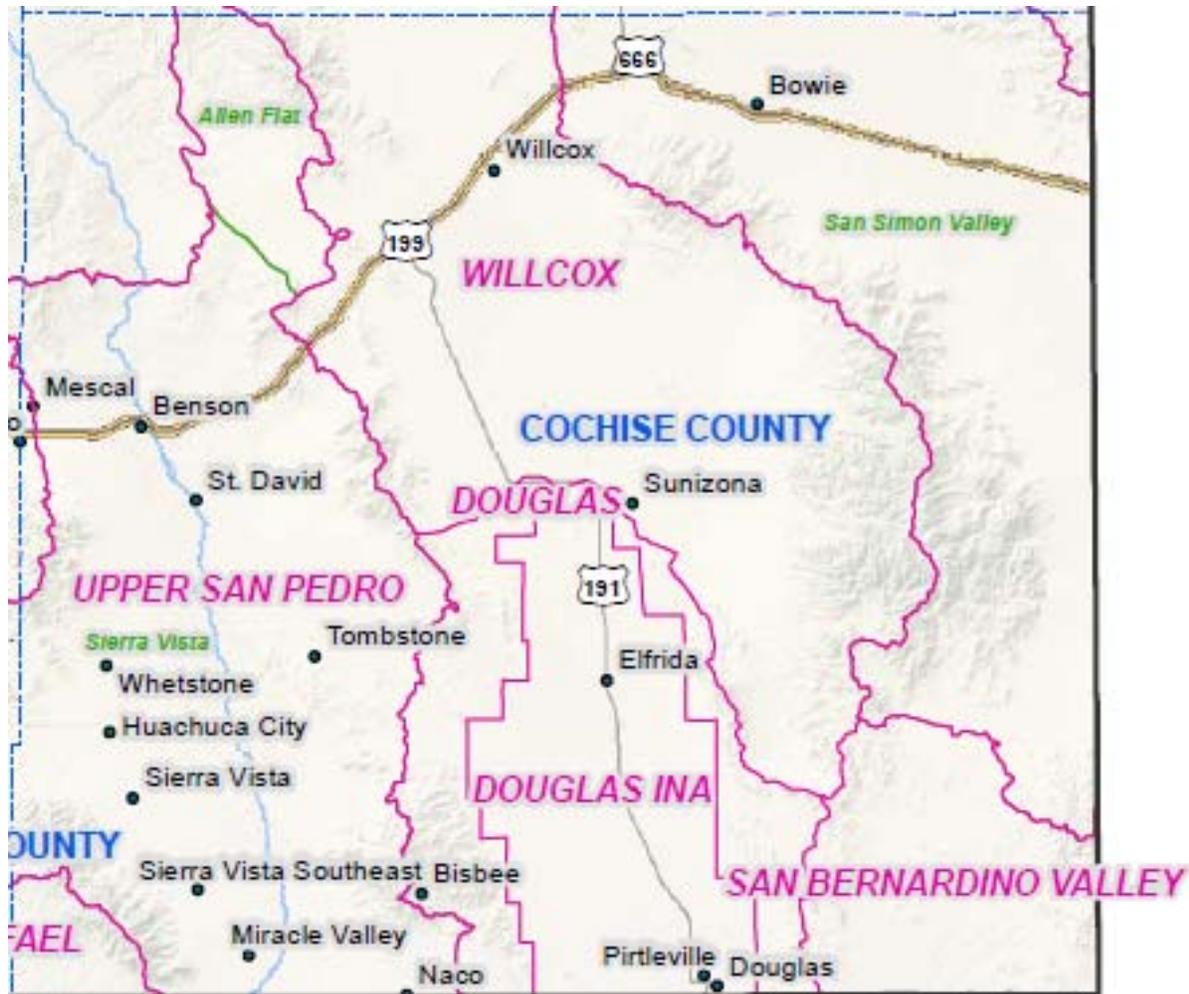
Source: Arizona Department of Water Resources, 2009

Figure 20-1: Location of the Douglas Basin

A long alluvial valley in the Douglas Basin contains its main aquifer, which is formed of basin fill. The basin fill is composed of sand and gravel lenses interbedded with silt and clay lenses. The sand and gravel lenses are the main source of water to most of the large-capacity wells in the region that support extensive agricultural irrigation. Groundwater is primarily unconfined, although artesian conditions were reported locally in the upper alluvial deposits in the early 1950s, prior to the start of heavy groundwater pumping. Groundwater is also found in the mountain bedrock, which provides relatively small amounts of water for stock and domestic use.

Groundwater flow is generally from north to south, although agricultural pumpage has altered flow directions in the vicinity of Elfrida where a cone of depression has developed. A recent groundwater flow model developed by the Arizona Department of Water Resources (2018) indicates that the regional aquifer system is not closed between the Willcox and Douglas basins. Saturated basin-fill deposits extend south from the Willcox Basin into the Douglas Basin, and groundwater flows from the Willcox Basin south to the Douglas Basin. The basin-fill materials are generally comprised of alluvial, lacustrine and volcanic rocks, whereas the floor and sides of the basin are composed of impermeable igneous, metamorphic and sedimentary rocks that outcrop in the surrounding mountains. The depth to bedrock in the Willcox Basin ranges from zero at the basin margins to over 4,000 feet below ground surface in the deeper, central portion of the basin. Data regarding the depth of the valley fill in the Douglas Basin was not encountered at the time of this report. It is assumed to be less than 400 feet on the project property, based on the available public

information regarding the Willcox Basin and the presence of rock outcrops (Figure 20-2). The exploration program results indicate that groundwater has been encountered in every drillhole completed and that the depth to groundwater is often less than 50 m (164 feet) below ground surface (Dave Webb, personal communication, 2018).
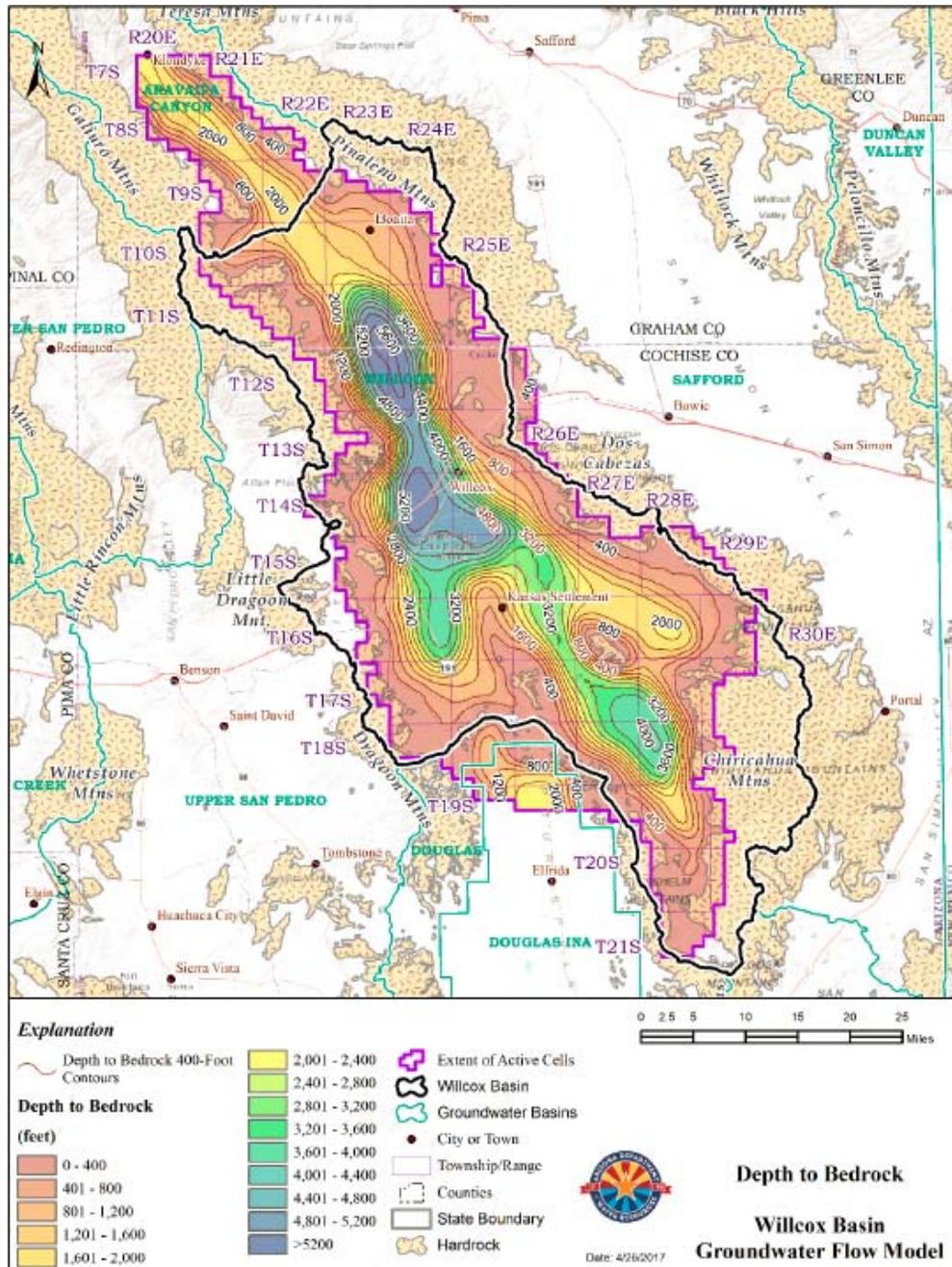
Source: ADWR, 2018.

Figure 20-2: Depth to Bedrock, Willcox Basin

Groundwater recharge occurs mainly in washes and along mountain fronts and is estimated at 15,500 to 22,000 acre-feet per year (AFA) (Arizona Department of Water Resources, 2009). Incidental recharge may also come from infiltration of agricultural irrigation.

Groundwater is relatively abundant in the Douglas Basin, and well yields are high. In 1994, the median well yield was 600 gpm and ADWR estimated that the basin water in storage was 32,000,000 acre-feet (ADWR, 2009, Table 3.5-5).

There are concerns about the long-term pumpage of groundwater and future groundwater supply availability. Groundwater in the basin is depleted at a rate faster than recharge, and water levels have declined in most wells measured in the basin (Figure 20-3). Annual basin losses from groundwater pumping were reported as almost 60,000 AFA (ADWR, 2018, Figure 20-3).
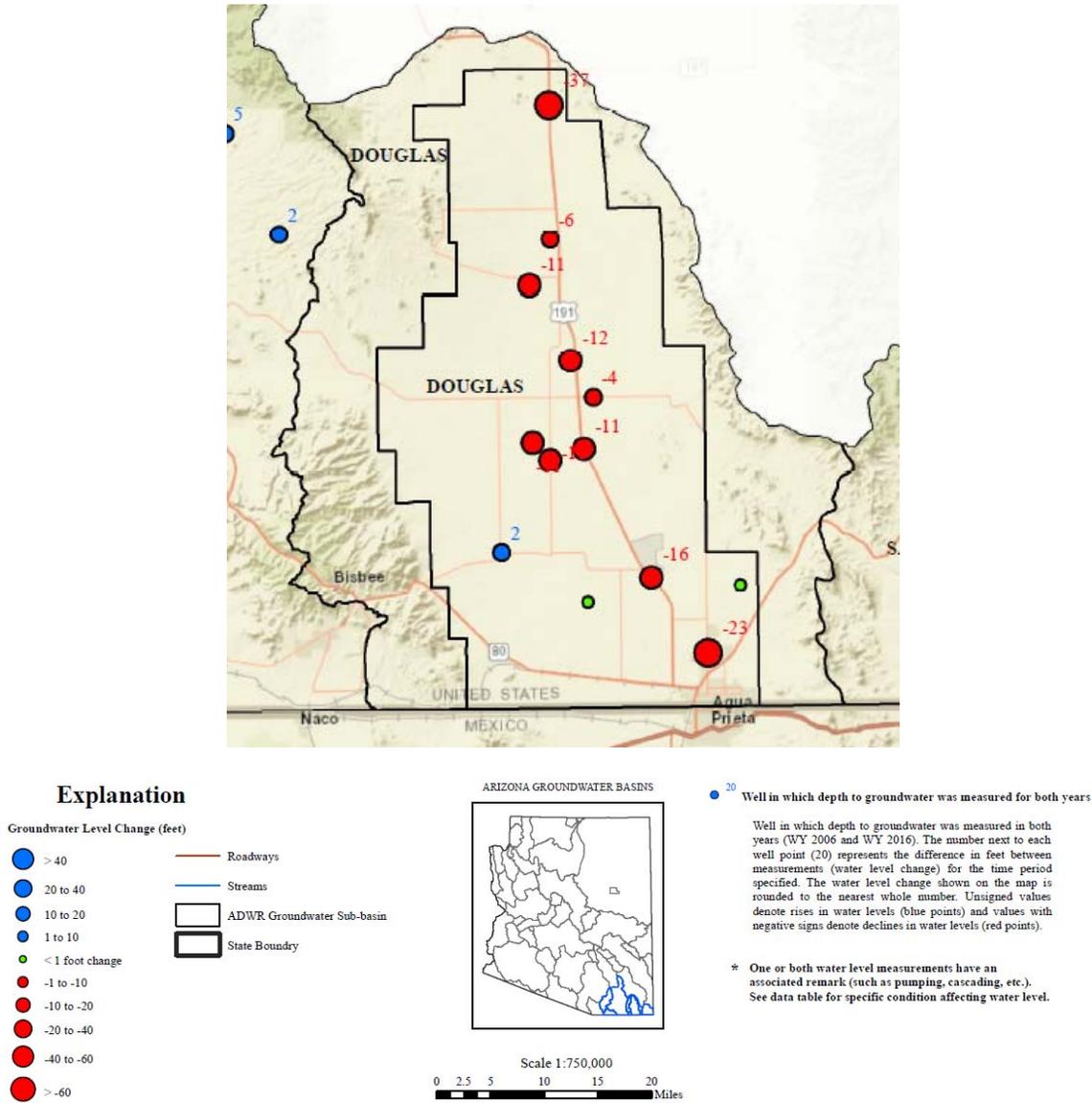
Source: Arizona Department of Water Resources, 2017.

Figure 20-3: Groundwater Level Changes Water Years 2006-2016 (Douglas Basin)

The Douglas Basin includes a portion of the basin where the use of groundwater is managed under the 1980 Groundwater Management Act, which established the Douglas Irrigation Non-Expansion Area (INA), as shown in Figure 20-3. All annual withdrawals of groundwater from non-exempt wells must be reported to ADWR. Exempt wells are those with less than 10 acre-feet withdrawal and for non-irrigation purposes. The Mexican Hat Project water supply

needs have not been evaluated as to the locations of future production wells, but a portion of the mining claims are within the Douglas INA; Claims located on T19S, R25E, sections 1, 2, and 3 are within the Douglas INA. The boundaries of the Douglas INA are shown in Figure 20-4. Assuming a water requirement of 700 gpm (1,130 AFA) for mining operations and being within the boundaries of the INA, the Mexican Hat Project will be considered non-exempt, and therefore subject to reporting requirements. The Douglas INA is administered by ADWR staff in the Tucson office.
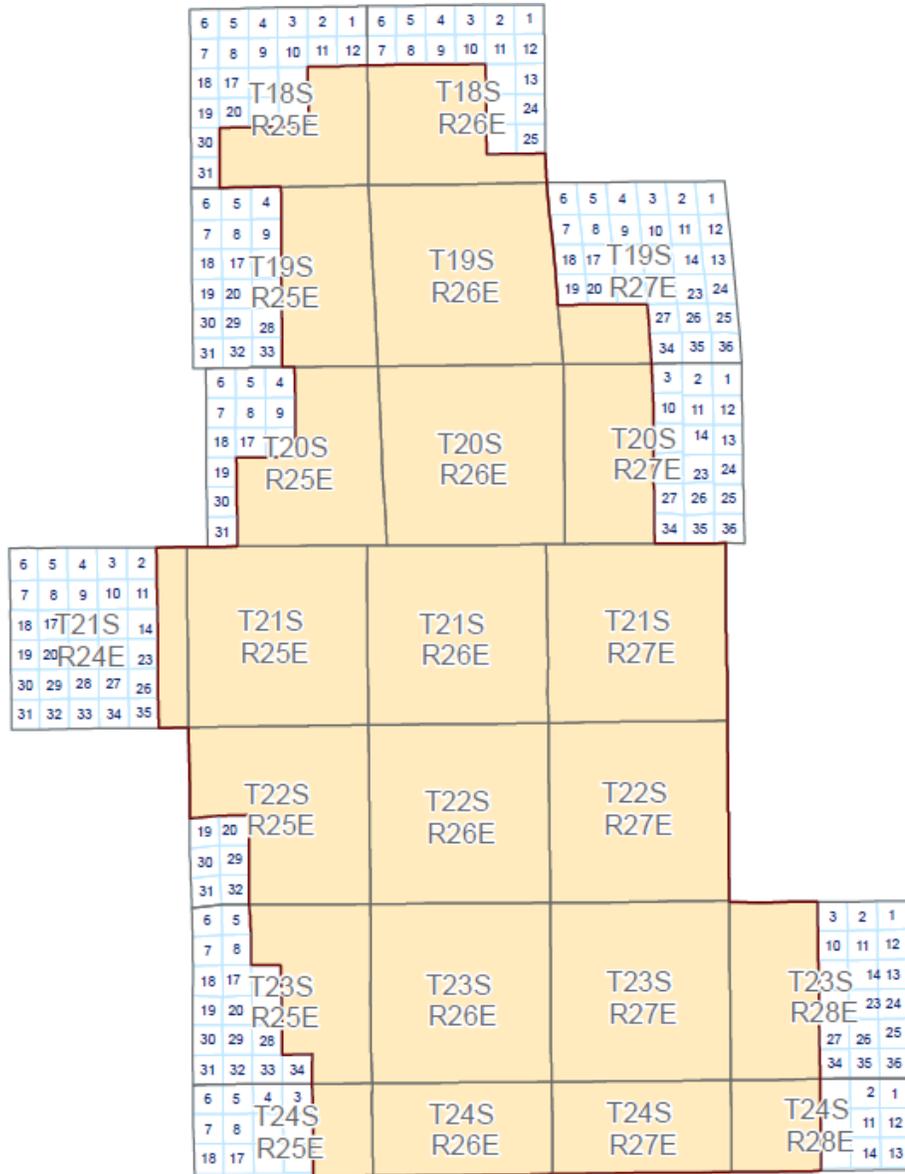
Source: Arizona Department of Water Resources
(http://www.azwater.gov/AzDWR/Watermanagement/AMAs/TucsonAMA/documents/DINA_TRS.pdf).

Figure 20-4: Douglas Irrigation Non-Expansion Area

Although there are concerns about long-term water availability within the Douglas Basin and Douglas INA, there is no restriction for applying for a non-irrigation well. It is believed that sufficient water supply will be available, and it is recommended that a hydrogeologic study be initiated in the next phase of project advancement.

20.4   BASELINE STUDIES AND POTENTIAL IMPACTS

Future mining operations will expose waste rock and mineralized material, and the potential to leach metals and metalloids or generate acid, will need to be evaluated. Acid rock drainage and metals leaching could affect runoff and seepage from the waste rock storage facilities, as well as the chemistry of a pit lake that forms during operations or after closure. A geochemistry testing program that characterizes rock types, plus metallurgical wastes (assumed to be spent feed), is required to evaluate the potential for future environmental impacts, especially on a long-term basis. Results of the geochemistry program should be used to guide the design of the mining facilities and to support the closure cost estimate.

Baseline studies associated with the environment (such as air, hydrology, meteorology, sediment and soils, terrestrial landforms, flora and fauna) are all required to document pre-mining conditions and for environmental permitting. If a weather station has not been installed at site, then it is recommended to install one as soon as possible. The climate data will be needed as input to the facility designs, especially for the surface water controls. Groundwater quality sampling generally requires two years of quarterly monitoring, and appropriate sampling locations will need to be identified (if existing). There are existing privately-owned and operated wells in the vicinity that can be used to obtain preliminary data on groundwater quality, but it is likely that new monitoring wells will need to be installed.

Socio-economic and land use baseline studies will also need to be carried out.

20.5   ENVIRONMENTAL MANAGEMENT

An environmental management plan to address the operational and environmental risks associated with the project will be developed as the project advances. Details of the environmental management plan are required for environmental permitting documents and are subject to a number of specific requirements under Arizona and Federal regulations. If the federal agency with authority over the project determines that the project may affect a listed species or designated critical habitat, consultation with U.S. Fish and Wildlife Service will be required. Likewise, should a cultural resource be identified, a variety of stakeholders would be invited to consult on the project.

20.6   PERMITTING

Permitting a mine in Arizona requires a variety of permits from exploration to closure planning. The permits are related to land use; water use; use of explosives, fuel and oil; air quality; water quality, protection of native plants; use of hazardous materials, waste disposal; drinking and waste water permits; flood control and building codes; mine health and safety; protection of wildlife and cultural resources; nuclear regulation; and communication.

Permitting a new mine is possible in Arizona, although the process is not deemed to be trivial. It is important to note that the permitting process varies based on the land ownership (private, state or federal).The major permits or approvals required by state and federal agencies are listed in Table 20-1. The applicability of each permit to the project has not been fully determined at this time.

Table 20-1: Environmental Permits

Required Permits	Issuing Agency	Regulatory Program or Statute	Purpose
Mine Plan of Operations	United States Bureau of Land Management	Federal Land Policy Management Act	Safe mining operations and protection of environment. Applies to lands patented under the Stock Raising Homestead Act.
Environmental Assessment/ Environmental Impact Statement	United States Bureau of Land Management	National Environmental Policy Act (NEPA)	Required analysis on potential environmental effects of proposed project (applicable when there is a federal permit)
Aquifer Protection Permit, Individual Permit	Arizona Department of Environmental Quality	Environmental Quality Act, APP Program	Protection of underground water quality; applies to mine leaching operations, plus surface impoundments, pits and ponds, and wastewater treatment facilities. Some minor facilities, such as the sewer facility, may be subject to the APP general permit program
Air Quality Permit	ADEQ	Clean Air Act	Protection of air quality, compliance with permissible limits
Drinking Water System Approval to Construct and Approval of Construction	Arizona Department of Environmental Quality	Safe Drinking Water Act	If applicable
Section 404 Permit (assuming that there are jurisdictional waters of the US)	US Army Corps of Engineers	Clean Water Act, Section 404	Disturbance of a federal waterway
Mined Land Reclamation Permit	Arizona State Mine Inspector	Arizona Revised Statues 27-901	Reclamation plan and financial assurance mechanism
Sewage System Permit	Cochise County, Department of Health and Social Services		Sanitary wastes authorization
Intent to Clear Land	Arizona Department of Agriculture	Arizona Revised Statues 3	
Clearance letter	United State Environmental Protection Agency (US EPA)	Endangered Species Act, Bald and Golden Eagle Protection Act, Migratory Bird Treaty	Identification and management of endangered species
Clearance letter	US EPA	National Historic Preservation Act, Archaeological Resources Protection Act, Native American Graves Protection and Repatriation Act	Identification and management of cultural resources
Water extraction permit	Arizona Department of Water Resources	Arizona Revised Statues 45	Notice of Intention to Drill and Groundwater Withdrawal Permit
Hazardous Waste Generator License	Arizona Department of Environmental Quality	Comprehensive Environmental Response, Compensation and Liability Act	Registration as hazardous waste generator, if applicable
Solid Waste Permit	Arizona Department of Environmental Quality	Resource Conservation and Recovery Act	Authorization for disposal of solid wastes
Construction of Dam	Arizona Department of Water Resources	Arizona Revised Statues 45	Construction of "dam" that requires a water diversion; could apply to waste rock storage facility and heap leach

The key permit issued by the State of Arizona is the Aquifer Protection Permit. It is understood that the mine design would be a "zero discharge" facilities, which means that no impacted water would be discharged from the site. A zero discharge mine, designed using Best Available Demonstrated Control Technology (BADCT) should not have any substantial impediment to obtaining an APP.

The key document to be approved by the federal government will be the Environmental Impact Statement, which would be developed under the National Environmental Policy Act (NEPA). The draft document requires public review and comments, with a minimum wait period but an undefined decision period. Mapping of the drainage boundaries will be required to determine whether designs impactdrainage systems that qualify as "waters of the U.S." per the criteria ofthe U.S. Corp of Engineers. Should the mine design indicate that facilities create a disturbance of a jurisdictional drainage, then a 404 permit will also be required.

The timing to collect to plan and collect baseline data and to develop models required by permitting authorities may take several years, with a minimum of 1 to 2 years of data collection. There can be an overlap in the collection of baseline data and proceeding with the preparation of the permit submittals. An additional period of time, on the order of two or more years, may be needed to complete regulatory review, public input and final issuance of the major permits. The timing for the review and public input under the federal government is less prescriptive than under the state of Arizona, and it can take longer to acquire federal permits than state permits. The key drivers to the permitting schedule are likely to be NEPA and the APP.

20.7   SOCIO-ECONOMIC STUDIES AND IMPACTS

The property is in the Turquoise Mining District, where historic land usage has been mining, agriculture and ranching. The closest community is Pearce, and there are several other small communities in the area (Sunsites), plus larger areas such as Willcox, Benson and Sierra Vista, which could be impacted by the development of the Mexican Hat Project. Within the region, concerns about sustainable water supplies, water level declines, increased agricultural demand and environmental protection activities have been identified by community water shed groups within the ADWR Southeastern Arizona Planning Area (ADWR, www.azwater.gov). It is anticipated that water will be a primary concern by stakeholders.

At this time, no community relation and stakeholder outreach program has been developed. If the project continues to advance, it is recommended that the stakeholders be identified and that a formal community relations program be developed to have consistent and ongoing communication with all stakeholders, and to provide opportunities for meaningful two-way dialogue and active public involvement.

20.8   CLOSURE AND RECLAMATION

Two separate closure regulations will apply for the project. One is the closure of facilities, such as the heap leach and its ancillary ponds, under the APP requirements to prevent long-term environmental impacts from post-closure facilities. The second is the closures of non-APP facilities, such as buildings and infrastructure, which are to be reclaimed in accordance with the Mined Land Reclamation Program (Arizona State Mine Inspector's Office). This program requires the development of reclamation plans that will ensure safe and stable post-mining land use. Re-grading and resurfacing needs, if any, will be completed with good engineering practices to minimize unwanted surface disturbances and to provide for surface water drainage. The closure and reclamation plans must include cost estimates and financial assurance. Acceptable financial assurance mechanisms are listed under ARS 27-973.

It is anticipated that closure methods and associated costs will be developed as part of the next phase of the project.

20.9   CONCLUSIONS

No known factors exist to preclude a successful permitting effort; however, the length and the effort of the permitting process can be difficult to predict due to the multiple agencies that will be involved, including the possibility of a NEPA process.

20.10   RECOMMENDATIONS FOR FUTURE WORK

To advance the project, baseline studies should be started to support the permitting process. Consultation with the community and the regulatory agencies should be initiated.

Baseline studies should overlap with the metallurgical testwork, geotechnical study and water supply study to optimize the project advancement. In particular, it is recommended that the geochemistry study to evaluate long-term environmental impacts be started as soon as possible, plus that drilling programs be combined to utilize exploration boreholes as groundwater monitoring points, and that any geotechnical drilling be combined with hydrogeologic requirements.

The company will need to contract or hire an environmental manager and a community relations manager to oversee activities.

21 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COST ESTIMATE

The capital cost for the initial development of the Mexican Hat Project is summarized in Table 21-1.

Table 21-1: Initial Capital Cost Summary ($M)

Site, Process and Infrastructure	$95.0
Initial Fleet Lease	$14.9
Pre-Production Waste Removal	$15.2
Owner's Costs	$5.0
Total	$130.1

21.1.1 Mine Capital Costs

Mine capital cost for mobile equipment was developed from the mine equipment list presented in Section 16. Unit costs for the major equipment and most of the minor equipment were based on equipment costs derived from other recent similarly sized projects to Mexican Hat.

A summary of capital costs can be seen in Table 21-2. Preproduction development expenses have been categorized here as capital costs.

Mine equipment capital costs do include:

1. All mine mobile equipment required to drill, blast, load, and haul the material from the pit to the appropriate destinations.

2. Auxiliary equipment to maintain the mine and material storage areas in good working order as well as construct the mine haul roads and maintain them.

3. Equipment to maintain the mine fleet such as tire handlers and forklifts.

4. Light vehicles for mine operations and staff personnel.

5. An allowance for shop tools to service the equipment in the shop facilities

6. Critical initial spare parts inventory.

7. Equipment replacements are included as required based on the useful life of the equipment.

Mine equipment capital costs DO NOT include:

1. Mine office buildings, or shop facilities. These costs are included in the Process Capital Costs.

2. An allowance for clearing and grubbing for mine site facilities. These costs are included in the Process Capital Costs.

3. Mine engineering equipment (computers, survey equipment etc.). These costs are included in the Owner's Costs.

4. Mobile equipment that is not required by the mine (i.e. no mobile units for the plant).

5. Infrastructure or process plant related costs.

6. Mine communication network & system.

Table 21-2: Mine Capital Costs Summary ($000)

| Year | Mine Equipment | | Salvage Value Benefit | Mine Preprod. Development | Total Mine Capital |
	Initial Capital Cost	Sustaining Capital Cost			
Preprod	5,784			15,209	20,993
Y01	9,078				9,078
Y02		6,830			6,830
Y03		6,830			6,830
Y04		6,830			6,830
Y05		10,468			10,468
Y06			-10,000		- 10,000
TOTAL	14,862	30,958	-10,000	15,209	51,029

21.1.2   Process Capital Costs

The capital cost for initial development of the process plant and infrastructure is summarized in Table 21-3.

Table 21-3: Process Capital Cost

Cost Item	Total ($M)
Process Plant and Infrastructure	$ 43.5
Heap Leach Pad	$ 9.0
Ancillaries and Mobile Equipment	$ 8.8
Power Supply	$ 2.0
Total Directs	$ 63.3
Project Indirects	$ 12.7
Subtotal	$ 76.0
Contingency (25%)	$ 19.0
Total Cost	$ 95.0

21.1.2.1   Contingency

Contingency is a cost that statically will occur based on historical data. The term is not used to cover changes in scope, errors, or lack of sufficient information to meet a desired accuracy range. Contingency is used to cover items of cost which fall within the scope of work, but are not known or sufficiently detailed at the time that the estimate is developed (e.g. geotechnical data).

21.1.2.2   Estimate Accuracy

The accuracy of this estimate for items identified in the scope of work is within the range of plus 30% to minus 25%; i.e. the costs could be 25% lower than the estimate or it could be 30% higher. Accuracy accounts for bidding climate variances from estimate date to actual construction date. Accuracy is an issue separate from contingency.

21.1.3   Process Sustaining Capital Costs

The process related sustaining capital costs includes the staged progression of the heap leach pad in Year 2 ($4.5 M) and Year 3 ($4.5 M).

21.2 OPERATING COST ESTIMATE

21.2.1 Mine Operating Costs

Mine operating costs were developed based on costs sourced from other recent similarly sized projects for the mine plan and on-hand equipment list presented earlier in Section 16. Mine operating costs are in US dollars. The mine is planned to work two shifts per day for 360 days per year. Five days (10 shifts) of loss time are assumed due to weather delays.

Table 21-4 summarizes the average mine operating costs estimated for the project. Preproduction will occur over a year period. The cost per tonne-moved in all periods is based on the total tonnage moved within the mine plan averaging $1.84/tonne-moved. On a per tonne-mined basis, the average mine operating cost is slightly higher at $1.88/tonne-mined.

The mine operating costs include:

1. Drilling, blasting, loading, and hauling of material from the mine to the crusher or waste storage facilities. Re-handling of material for the stockpile movements to the crusher in Year 01 and Year 05. Maintenance of mine mobile equipment is included in the operating costs.

2. Mine supervision, mine engineering, geology and ore control are included in the G&A category.

3. Operating labor and maintenance labor for the mine mobile equipment are included.

4. A general mine allowance is included that is intended to cover general operating supplies that cannot be assigned to one of the unit operations.

5. A general maintenance allowance is included that is intended to cover the general operating supplies of the maintenance group.

The mine operating costs do not include:

1. Crushing, conveying or processing

2. There are currently no reclamation or recontouring costs within the mine operating cost estimate.

Table 21-4: Life-of-Mine Mine Operating Costs

Summary of LOM Average Operating Cost (Moved tonnes)		Summary of LOM Average Operating Cost (Mined tonnes)	
Cost Category	Cost ($/t)	Cost Category	Cost ($/t)
Drilling	0.260	Drilling	0.266
Blasting	0.303	Blasting	0.310
Loading	0.167	Loading	0.170
Hauling	0.430	Hauling	0.440
Auxiliary	0.252	Auxiliary	0.258
General Mine	0.077	General Mine	0.079
General Maintenance	0.103	General Maintenance	0.105
G&A	0.249	G&A	0.254
Total	1.841	Total	1.883

The mine operating costs are comprised of labor and parts and consumables. The parts and consumables include four cost categories; mine major equipment, blasting, general mine, and general maintenance. Labor includes mine salary personnel, hourly mine operators and hourly maintenance personnel.

A diesel fuel cost of $0.66 per liter has been used in the calculation of the operating costs.

The general mine costs include the parts and consumables costs associated with maintaining the mine operations support equipment and supplying the mine engineering and operations offices. General mine also includes assaying costs and software license fees.

The general maintenance costs include the parts and consumables associated with maintaining the mine maintenance support equipment such as the dozers, graders, water trucks, fuel and lube trucks, cranes, etc.

Labor costs account for about 40% of the total operating cost. Labor includes the salaried staff, mine operators, and maintenance staff.

21.2.2 Process Operating Costs

21.2.2.1 Process Plant Operating & Maintenance Costs

The process plant operating costs are summarized by area and are further distributed by cost elements of labor, power, reagents, maintenance parts and supplies and services in Table 21-5.

Table 21-5: LOM Operating Cost Summary

Processing Units Base Rate (ktonnes)	25,110	
	LOM Cost $000	$ per tonne feed
Crushing & Stacking	$36,595	$1.46
Process Plant	$57,040	$2.27
ADR & Metal Recovery	$11,346	$0.45
Total Process Plant	$104,980	$4.18

21.2.2.1.1 Process Labor & Fringes

Process labor costs were derived from a staffing plan and based on prevailing daily or annual labor rates in the area. Labor rates and fringe benefits for employees include all applicable social security benefits as well as all applicable payroll taxes. The staffing plan summary and gross annual labor costs are shown in Table 21-6 below.

Table 21-6: Process Plant Labor & Fringes

Department	Number of Personnel	Annual Cost ($000)
Operations	47	$4,326
Maintenance	12	$1,218
Total	59	$5,544

21.2.2.1.2 Electrical Power

Power consumption was based on the equipment list connected kW, discounted for operating time per day and anticipated operating load level. The power rate is estimated at $0.0756 per kWh. A summary of the power cost and consumption are shown in Table 21-7.

Table 21-7: Process Plant Power Cost Summary

Area	Annual Power Consumption Year 2 (MWh)	Annual Cost ($000)
Crushing	5,029	$380
Leach Pad Stacking	6,173	$467
Heap Leach & Solution Transfer	7,104	$537
ADR	2,255	$170
Refinery	288	$22
Water Systems	2,353	$178
Reagents	388	$29
Total	23,589	$1,783

21.2.2.1.3   Reagents

Consumption rates were determined from the metallurgical test data or industry practice. Budget quotations were received for reagents supplied from local sources where available with an allowance for freight to site.

A summary of process reagent consumption and costs are included in Table 21-8.

Table 21-8: Reagents Consumption Summary

Reagent	kg/t feed	$/kg
Sodium Cyanide	0.30	$3.31
Lime	1.50	$0.24
Carbon	0.03	$3.86
Caustic Soda	0.03	$0.55
Antiscalant	0.02	$2.76
Hydrochloric Acid	0.04	$0.66
Refinery Fluxes	0.01	$2.43

21.2.2.1.4   Maintenance Wear Parts and Consumables

Crusher wear items (liners) were based on industry practice. These consumption rates and unit prices are shown in Table 21-9.

Table 21-9: Crusher Liners

	kg/t	$/kg
Crusher Liners	0.13	$2.98

An allowance was made to cover the cost of maintenance of all items not specifically identified and the cost of maintenance of the facilities. The allowance was calculated using the direct capital cost of equipment times a percentage factor, which totalled approximately $1.1 million for the process plant. An annual allowance was made for outside maintenance services to be performed at approximately $0.1 million.

21.2.2.1.5   Process Supplies and Services

Allowances were provided in process plant for lubricants, outside consultants, outside contractors, vehicle maintenance, and miscellaneous supplies. The allowances were estimated using M3's information from other operations and projects. Approximately $3.2 million will be spent annually.

21.2.2.2   General and Administration (G&A)

General and administration costs include labor and fringe benefits for the administrative personnel, human resources, accounting, purchasing and warehousing, community relations, safety and environmental. Also included are office supplies, communications, insurance, employee transportation and other expenses in the administrative area. Annual cost is estimated at $8.2 million.

Table 21-10: General & Administration

	$/t mineralization
G&A Costs	$1.50

22 ECONOMIC ANALYSIS

22.1 INTRODUCTION

The financial evaluation presents the determination of the Net Present Value (NPV), payback period (time in years to recapture the initial capital investment), and the Internal Rate of Return (IRR) for the project. Annual cash flow projections were estimated over the life of the mine based on the estimates of capital expenditures and production cost and sales revenue. The sales revenue is based on the production of gold doré. The estimates of capital expenditures and site production costs have been developed specifically for this project and have been presented in earlier sections of this report.

22.2 MINE PRODUCTION STATISTICS

Mine production is reported as feed and waste from the mining operation. The annual production figures were obtained from the mine plan as reported earlier in this report.

The life of mine feed and waste quantities and feed grade are presented in Table 22-1.

Table 22-1: Life of Mine Ore, Waste and Metal Grades

	Tonnes (000's)	Gold g/t
Feed	23,945	0.680
Low Grade	1,165	0.273
Waste	71,276	

22.3 PLANT PRODUCTION STATISTICS

Mineralized feed will be crushed, screened, conveyed and placed on a heap leach and processed in a ADR carbon plant and will produce a gold doré.

The estimated metal recoveries are presented in Table 22-2.

Table 22-2: Metal Recovery Factors

	Gold %
Gold Doré	88.2

Estimated life of mine gold doré production is presented in Table 22-3 with the approximate metal contained.

Table 22-3: Life of Mine Production Summary

	Gold (kozs)
Gold Doré	470

22.3.1 Smelter Return Factors

Gold doré will be shipped from the mine site to a refining company. Refining charges are negotiable at the time of agreement and are shown below.

Table 22-4: Refining Terms

Doré	
Payable gold	99.9 %
Refining/Selling charge – Au ($/oz)	$0.65
Freight/Insurance charge – Au ($/oz)	$0.50

22.4   CAPITAL EXPENDITURE

22.4.1   Initial and Sustaining Capital

The total capital carried in the financial model for initial and sustaining capital is shown in Table 22-5.

Table 22-5: Initial and Sustaining Capital Summary

Period	Initial Capital ($000)	Sustaining Capital ($000)
Year -2	$5,000	
Year -1	$110,991	
Year 1	$14,078	
Year 2		$11,330
Year 3		$11,330
Year 4		$6,830
Year 5		$10,468
Year 6		
Total	$130,070	$39,958

22.4.2   Working Capital

A delay of receipt of revenue (15 days) from sales is used for accounts receivables. A delay of payment for accounts payable of 30 days is also incorporated into the financial model. Inventory and parts are estimated at $1.0 million. All the working capital is recaptured at the end of the mine life and the final value of these accounts is $0.

22.4.3   Salvage Value

An allowance for salvage value has been included in the cash flow analysis and estimated to be $11.0 million.

22.5   REVENUE

Annual revenue is determined by applying estimated metal prices to the annual payable metal estimated for each operating year. Sales prices have been applied to all life of mine production without escalation or hedging. The revenue is the gross value of payable metals sold before treatment and transportation charges. Metal sales prices used in the evaluation are as follows:

- Gold $1,300.00/troy ounce

22.6   OPERATING COST

Table 22-6 shows the estimated life of mine on-site operating cost by area per tonne of feed processed and per ounce produced. Table 22-7 shows production costs by year of operations.

Table 22-6: LOM Operating Cost Summary

	Cost per Tonne of Feed Processed	Cost per Ounce Produced
Mining	$6.62	$353
Process	$4.18	$223
G&A	$1.50	$80
Total Site Operating Cost[1]	$12.30	$657

Notes:

1. Site Operating cost is calculated by dividing total life of mine on-site costs by total ounces produced.

Table 22-7: Production Cost by Year of Operation

Year	$/oz Gold Payable				$/Tonne of Feed			
	Mining	Process	G&A	Total	Mining	Process	G&A	Total
1	300.64	188.36	69.49	558.50	6.49	4.07	1.50	12.06
2	364.98	226.46	83.55	674.98	6.55	4.07	1.50	12.12
3	438.94	267.07	98.53	804.54	6.68	4.07	1.50	12.25
4	377.54	213.83	78.89	670.26	7.18	4.07	1.50	12.74
5	286.68	241.15	72.86	600.69	5.90	4.96	1.50	12.37
Average	353.69	223.35	80.13	657.18	6.62	4.18	1.50	12.30

22.6.1 Total Production Cost

The average Production Cost over the life of the mine is estimated to be $12.72/t of feed processed. Total Cash Cost includes Total Operating Cost, royalties, Severance Tax, salvage value and reclamation and closure costs.

The All-in Sustaining Cost is estimated to be $764 per ounce of gold. The All-in Sustaining cost is calculated by dividing the sum of the Total Production cost, the LOM sustaining capital cost and the Arizona Severence Tax by total ounces of gold produced.

The All-in Cost is estimated to be $1,076 per ounce of gold. The All-in cost is calculated by dividing the sum of all capital, operating, sustaining capital, tax, royalty, salvage and reclamation costs by total ounces of gold produced.

22.6.1.1 Royalty

Half of the Hernandez royalty will be purchased in year 1 for $1.5 million and the other half is subject to a 1.5% royalty based on the NSR and is estimated to be $10.4 million for the life of mine.

22.6.1.2 Depreciation

Depreciation is calculated using an unit of production method starting with first year of production for both the initial capital and sustaining capital.

22.6.1.3 Reclamation & Closure

An allowance for the cost of final reclamation and closure of the property has been included in the cash flow analysis and is estimated to be $8.5 million.

22.7   TAXATION

A severance tax is payable to the state of Arizona is approximately $2.1 million for the life of the mine.

The Arizona state income tax rate is 6.5% and the taxes payable for the life of mine is approximately $4.3 million.

Corporate income taxes paid is estimated to be $12.4 million using a tax rate of 21%.

22.8   PROJECT FINANCING

For the purposes of this study it is assumed investment in the Mexican Hat mine will be equity financed.

22.9   NET INCOME AFTER-TAX

Net Income after Tax is approximately $105.0 million for the life of the mine.

22.10   NPV/IRR

At a $1,300 per ounce gold price, the base case economic analysis of the Mexican Hat Project indicates an NPV at 5% discount rate (NPV_5) of $77.8 million and an after-tax IRR of 29.1%. Table 22-8 compares the base case project financial indicators with the financial indicators when different variables are applied.

Table 22-8: Sensitivity Analysis After-Tax ($M)

	NPV @ 5%	IRR%	Payback (yrs)
Gold Price			
+20%	$165.3	53.8%	1.3
+10%	$122.1	41.7%	1.7
Base Case	$77.8	29.1%	2.3
-10%	$32.9	15.5%	3.4
-20%	$-0.4	-0.1%	7.0
Initial Capital			
+20%	$55.5	19.4%	3.1
+10%	$66.7	23.9%	2.8
Base Case	$77.8	29.1%	2.3
-10%	$88.8	35.4%	1.9
-20%	$99.8	43.3%	1.6
Operating Cost			
+20%	$32.1	15.4%	3.4
+10%	$55.1	22.4%	2.9
Base Case	$77.8	29.1%	2.3
-10%	$100.3	35.4%	1.9
-20%	$122.3	41.4%	1.7

Table 22-9: Financial Model

Mining Operations	Total	-2	-1	1	2	3	4	5	6	7
Feed										
Beginning Inventory	23,945	23,945	23,945	22,945	18,470	12,995	7,520	2,045	-	-
Mined (kt)	23,945	-	1,000	4,475	5,475	5,475	5,475	2,045	-	-
Ending Inventory	-	23,945	22,945	18,470	12,995	7,520	2,045	-	-	-
Au Grade (g/t)	0.680	-	0.702	0.769	0.634	0.542	0.671	0.991	-	-
Contained Au (kozs)	523	-	23	111	112	95	118	65	-	-
Low Grade										
Beginning Inventory	1,165	1,165	1,165	966	-	-	-	-	-	-
Mined (kt)	1,165	0	199	966	0	0	0	0	0	0
Ending Inventory	-	1,165	966	-	-	-	-	-	-	-
Au Grade (g/t)	0.273	0	0.273	0.273	0	0	0	0	0	0
Contained Au (kozs)	10	-	2	8	-	-	-	-	-	-
Waste (kt)	71,276	-	3,971	15,439	15,405	15,405	15,413	5,643	-	-
Process Plant										
Feed Processed (kt)	25,110	-	-	5,475	5,475	5,475	5,475	3,210	-	-
Au Grade (g/t)	0.661	-	-	0.757	0.634	0.542	0.671	0.730	-	-
Contained Au (kozs)	534	-	-	133	112	95	118	75	-	-
Gold Recovery (%)	88.2%	0.0%	0.0%	88.8%	88.2%	87.5%	88.2%	87.8%	0.0%	0.0%
Recovered Gold (kozs)	470	-	-	118	98	83	104	66	-	-
Payable Metals										
Payable Gold (kozs)	470	-	-	118	98	83	104	66	-	-
Income Statement ($000)										
Gold Price ($/oz)	$1,300	$1,300	$1,300	$1,300	$1,300	$1,300	$1,300	$1,300	$1,300	$1,300
Revenues										
Gold	$611,029	$0	$0	$153,631	$127,790	$108,358	$135,334	$85,916	$0	$0
Operating Cost										
Mining	$166,244			$35,529	$35,878	$36,587	$39,303	$18,947	$0	$0
Process Plant	$104,980			$22,261	$22,261	$22,261	$22,261	$15,938	$0	$0
General Administration	$37,665			$8,213	$8,213	$8,213	$8,213	$4,815	$0	$0
Refining Charge	$541			$136	$113	$96	$120	$76	$0	$0
Total Operating Cost	$309,430	$0	$0	$66,139	$66,464	$67,156	$69,896	$39,775	$0	$0
Royalty	$10,280			$3,768	$1,829	$1,601	$2,022	$1,061	$0	$0
Salvage Value	-$11,000								-$11,000	
Reclamation & Closure	$8,500								$8,500	
Total Production Cost	$317,210	$0	$0	$69,907	$68,293	$68,757	$71,918	$40,836	-$2,500	$0
Operating Income	$293,818			$83,724	$59,497	$39,602	$63,416	$45,080	$2,500	$0
Depreciation	$170,027			$32,704	$30,368	$29,475	$40,990	$36,491	$0	$0
Net Income After Depreciation	$123,791	$0	$0	$51,021	$29,129	$10,127	$22,425	$8,590	$2,500	$0
Taxes	$18,819	$0	$0	$0	$7,934	$4,193	$1,520	$3,253	$1,265	$653
Net Income After Taxes	$104,972	$0	$0	$51,021	$21,195	$5,933	$20,905	$5,337	$1,235	-$653
Cash Flow										
Operating Income after Depreciation	$123,791	$0	$0	$51,021	$29,129	$10,127	$22,425	$8,590	$2,500	$0
Add Back Depreciation	$170,027	$0	$0	$32,704	$30,368	$29,475	$40,990	$36,491	$0	$0
Working Capital										
Account Receivable	$0	$0	$0	-$6,314	$1,062	$799	-$1,109	$2,031	$3,531	$0
Accounts Payable	$0	$0	$0	$5,436	$27	$57	$225	-$2,476	-$3,269	$0

Mining Operations	Total	-2	-1	1	2	3	4	5	6	7
Inventory - Parts, Supplies	$0		-500	-500				$1,000		
Total Working Capital	$0	$0	-$500	-$1,378	$1,089	$855	-$883	$555	$262	$0
Capital Expenditures										
Initial Capital										
Mine	$30,070	$0	$20,991	$9,078	$0	$0	$0	$0	$0	$0
Process Plant	$95,000	$4,750	$85,500	$4,750	$0	$0	$0	$0	$0	$0
Owners Cost	$5,000	$250	$4,500	$250	$0	$0	$0	$0	$0	$0
Sustaining Capital	$39,958	$0	$0	$0	$11,330	$11,330	$6,830	$10,468	$0	$0
Total Capital Expenditures	$170,027	$5,000	$110,991	$14,078	$11,330	$11,330	$6,830	$10,468	$0	$0
Cash Flow before Taxes	$123,791	-$5,000	-$111,491	$68,269	$49,256	$29,127	$55,702	$35,167	$2,762	$0
Cumulative Cash Flow before Taxes		-$5,000	-$116,491	-$48,223	$1,033	$30,160	$85,862	$121,029	$123,791	$123,791
Taxes	$18,819	$0	$0	$0	$7,934	$4,193	$1,520	$3,253	$1,265	$653
Cash Flow after Taxes	$104,972	-$5,000	-$111,491	$68,269	$41,322	$24,934	$54,182	$31,914	$1,496	-$653
Cumulative Cash Flow after Taxes		-$5,000	-$116,491	-$48,223	-$6,901	$18,033	$72,215	$104,129	$105,626	$104,972

Economic Indicators before Taxes	
NPV @ 0%	$123,791
NPV @ 5%	$93,805
NPV @ 10%	$69,602
IRR	33.1%
Payback	2.0

Economic Indicators after Taxes	
NPV @ 0%	$104,972
NPV @ 5%	$77,779
NPV @ 10%	$55,787
IRR	29.1%
Payback	2.3

23   ADJACENT PROPERTIES

There are no immediately adjacent properties.

The Commonwealth Gold and Silver property owned by Marlin Gold Mining Ltd. is located in Pearce, 9.7 km north of the Mexican Hat deposit. This is a more conventional epithermal precious metal deposit associated with northwest trending quartz veins cross-cutting Tertiary volcanic rocks. A 43-101 compliant Mineral Resource Estimate completed by Black, Z.J. (2016) reports the following:

Cutoff	Volume	Tonnage	Gold Equivalent		Gold		Silver	
(gpt)	cu. M	000 tonnes	gpt	t. oz.	gpt	t. oz.	gpt	t. oz.
Inverse Distance 2.5 Model In Pit Measured Resources								
0.4	1,662,900	4,069	1.380	180,800	0.57	74,800	48.6	6,357,700
0.3	1,841,200	4,504	1.280	185,700	0.53	77,200	45.0	6,516,900
0.2	2,047,000	5,007	1.18	189,800	0.49	79,000	41.3	6,648,500
Inverse Distance 2.5 Model In Pit Indicated Resources								
0.4	8,966,100	21,934	1.06	746,100	0.45	314,500	36.8	25,950,900
0.3	10,893,200	26,643	0.93	799,200	0.40	339,200	32.2	27,582,000
0.2	12,522,400	30,623	0.85	832,000	0.36	354,400	29.1	28,650,600
In Pit Measured and Indicated Resources								
0.4	10,629,100	26,003	1.11	926,900	0.47	389,300	38.6	32,308,700
0.3	12,734,400	31,147	0.98	984,900	0.42	416,400	34.1	34,098,900
0.2	14,569,400	35,630	0.89	1,021,700	0.38	433,500	30.8	35,299,100

*Notes:

(1) Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resources estimated will be converted into Mineral Reserves.

(3) Measured and Indicated Mineral Resources captured within the pit shell meet the test of reasonable prospect for economic extraction and can be declared a Mineral Resource.

(4) All resources are stated above a 0.2 g/t gold equivalent ("AuEq") cut-off.

(5) Pit optimization is based on assumed gold and silver prices of US$1,350/oz. and US$22.50/oz., respectively and mining, processing and G&A costs of US$7.25 per tonne. Metallurgical recoveries for gold and silver were assigned by lithologic unit in the optimization.

(6) Mineral resource tonnage and contained metal have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding.

(7) Gold Equivalent stated using a ratio of 60:1 and ounces calculated using the following conversion rate: 1 troy ounce = 31.1035 grams. Metallurgical recoveries are not accounted for in the gold equivalent calculation.

The QP (Dave Webb) has visited the property but has not completed anything but cursory examinations of the deposit from a distance.

The QP asserts that information on the adjacent properties is not necessarily indicative of the mineralization on the subject of this technical report; the Mexican Hat property.

24   OTHER RELEVANT DATA AND INFORMATION

There is no other relevant data to report.

25   INTERPRETATION AND CONCLUSIONS

The results of the preliminary economic analysis conclude:

- The PEA for the Mexican Hat Project indicates the potential economic viability of the gold deposit. At a $1,300 per ounce gold price, the base case economic analysis of the project indicates an NPV at 5% discount rate of $77.8 million and an after-tax IRR of 29.1%.
- Mine life of 5-years with a 1-year pre-production period, mined by conventional open pit hard rock methods.
- Crushed mineralized material will be conveyor stacked at a rate of approximately 15,000 tonnes/day on a conventional heap leach pad.
- Life of mine ("LOM") head grade of 0.66 g/t gold
- Low LOM Strip Ratio of 2.8
- Total amount of gold recovered is estimated at 470,000 oz
- Average annual gold production of approximately 94,000 oz
- Peak annual gold production of approximately 118,000 oz
- LOM direct operating cash cost is estimated at $658/oz of gold recovered
- All-in sustaining cost is estimated at $764/oz of gold recovered
- LOM sustaining capital costs is estimated at $40.0 million
- The Mexican Hat Mineral Resource is currently open in three directions and to depth
- No known factors exist to preclude a successful permitting effort; however, the length and the effort of the permitting process can be difficult to predict due to the multiple agencies that will be involved, including the possibility of a NEPA process.

While the authors have confidence in the level of study completed and the results of the Mexican Hat PEA, it is with the understanding that the Mexican Hat PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the preliminary economic assessment will be realized.

26   RECOMMENDATIONS

26.1   DRILLING AND EXPLORATION

Aspects of grade variation and geological interpretation must be further refined to reduce risk prior to advancing engineering and development of the Project. Additional drill programs should place an emphasis on replacing historical drillholes with modern drilling, where core recovery methods are used rather then RC recovery. A total of $350,000 should be allocated to drilling which could be raised to $850,000 if core drilling is used, and $100,000 allocated to relogging and mapping. Deep drilling, if warranted could add $900,000 to the program.

Additional exploration recommendations include the following:

- Mineralization is open along the Victoria Fault and to depth to the north. Evidence of mineralization within the footwall of the Victoria Fault should be investigated further to potentially increase the resource in each direction.

- Drill targets should focus initially on areas within the modeled pit that are sparsely drilled, and shallow extensions of known trends or zones adjacent to the modeled pit to increase confidence in the geological and grade distribution.

- All drilling should be carefully logged for geotechnical parameters as well as exploration details; development of a structural model should be considered to help identify presence and timing of structures relative to gold mineralization.

Geophysics has identified rocks that appear to be similar magnetically and electromagnetically in several locations and these should be examined. A total of $120,000 should be allocated to geophysics.

- Surficial geochemistry identifies Mexican Hat Mountain and the down-slope soils as coming from geochemically anomalous rocks. Likewise, similar anomalies can be identified coming from the Hernandez Hill area, and along the east-slope of Little Hat Mountain. These areas should be examined.

- A ground truthing study should be undertaken using a differential GPS to determine best as possible the location of previous drillholes.

26.2   DATABASE

- It's recommended that GMV implement an enhanced QAQC system, which targets a more representative range of assays for duplicate testing.

- It's recommended that GMV implement a database management system in order to better track and organize the large volumes of data which are collected and stored for the Project

26.3   METALLURGY

A single column test under the final leach conditions should be carried out to confirm the expected results. As exploration expands the resource, it would be prudent to confirm that additional resources are amenable to treatment by the designed plant operating conditions. Potentially, an additional 3 tests may be required depending on the success of the exploration programme.

Each test would cost about $12,000 including an allowance for freight, sample preparation and assaying, thus a budget of $48,000 should be allowed. Material left over from the exploration drill core can be used so no sample acquisition costs are needed.

26.4   ENVIRONMENTAL

To advance the project, baseline studies should be started to support the permitting process. Consultation with the community and the regulatory agencies should be initiated. The company will need to contract or hire an environmental manager and a community relations manager to oversee activities.

The specific baseline studies should include biological resources, cultural resources, hydrogeologic studies, geochemical studies, air and weather monitoring and surface hydrology study. The estimated cost for the baseline studies is between $2.5M to $3M. There is a potential significant cost savings opportunity to combine hydrogeologic studies with the exploration drilling and geotechnical investigations. The geochemistry study can also be coordinated in conjunction with exploration and geometallurgical work.

27　　　　REFERENCES

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APPENDIX A: FEASIBILITY STUDY CONTRIBUTORS AND PROFESSIONAL QUALIFICATIONS

CERTIFICATE OF QUALIFIED PERSON

Daniel Roth

I, Daniel Roth, PE, P. Eng. do hereby certify that:

1. I am currently employed as a project manager and civil engineer at M3 Engineering & Technology Corp. located at 2051 West Sunset Rd, Suite 101, Tucson, AZ 85704.

2. I graduated with a Bachelor's of Science degree in Civil Engineering from The University of Manitoba in 1990.

3. I am a registered professional engineer in good standing in the following jurisdictions:

    - British Columbia, Canada (No. 38037)
    - Alberta, Canada (No. 62310)
    - Ontario, Canada (No. 100156213)
    - Yukon, Canada (No. 1998)
    - New Mexico, USA (No. 17342)
    - Arizona, USA (No. 37319)
    - Alaska, USA (No. 102317)
    - Minnesota, USA (No. 54138)

4. I have practiced engineering and project management for 25 years. I joined M3 Engineering in November 2003.

5. I have read National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association as defined in NI 43-101 and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101.

6. I am one of the authors of this Technical Report titled *Mexican Hat Project, NI 43-101 Technical Report, Preliminary Economic Assessment* prepared for GMV Minerals Inc., with an effective date of December 6, 2018 and issue date of December 18, 2018. I am responsible for Sections 2, 18, 21.1.2, 21.1.3, 22, 27 and corresponding sections of 1, 25, and 26 of this Technical Report. This Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

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

8. I visited the property that is the subject of this Technical Report on 28 August 2018.

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

10. I am independent of GMV Minerals Inc. and all their subsidiaries as defined by Section 1.5 of NI 43-101.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their website accessible by the public, of the Technical Report.

Dated 18 December, 2018.

(signed) (sealed) Daniel Roth
Daniel Roth, PE, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

Dave R. Webb

I, Dave R Webb, Ph.D., P.Geol., P.Eng., do hereby certify that:

1. I am president of:

    DRW Geological Consultants Ltd.,
    1909 108 West Cordova St.,
    Vancouver, B.C., V6B 0G5

2. I graduated with a B.A.Sc. from the University of Toronto, Toronto, Ontario, an M.Sc. from Queen's University in Kingston, Ontario, and a Ph.D. from the University of Western Ontario, London, Ontario.

3. I am a Professional Geologist and a Professional Engineer in good standing with NAPEGG in the Northwest Territories, Canada.

4. I have worked as a geologist and engineer for more than 30 years. My experience includes mapping and Mineral Resource and Mineral Reserve work over this time frame.

5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

6. I am [the principal author OR a contributing author] for the preparation of the technical report titled "Mexican Hat Project, NI 43-101 Technical Report, Preliminary Economic Assessment", (the "Technical Report"), dated effective December 6, 2018 and issued December 18, 2018, prepared for GMV Minerals Inc.; and am responsible for Sections 2 to 11, 15, 19, 23 and 24 and a contributing author to Sections 1, 14, 25 and 26. I have visited the project site 100 days between 2014 and 2018.

7. I have prior involvement with the property that is the subject of the Technical Report as author to technical reports in 2015, and a contributing author to a technical report in 2018.

8. I also have provided independent services in the past to a company controlled by some of the same principals as GMV Minerals Inc.

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

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

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

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 18th day of December, 2018.

(Signed) (Sealed) *Dave R. Webb*
Signature of Qualified Person

Dr. Dave R. Webb, Ph.D., P.Geol., P.Eng.
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Justin Miles Black

I, Justin Black, P.E., do hereby certify that:

1. I am a Chemical/Process Engineer of:

    M3 Engineering & Technology Corp.
    2175 W. Pecos Road, Suite 3, Chandler, Arizona, USA, 85224

2. I graduated with a Bachelor of Science degree in Chemical Engineering from the University of Arizona in 2011.

3. I am a certified Professional Engineer in good standing in the state of Arizona in the areas of chemical engineering [Cert. No, 63607].

4. I have worked as Process Engineer and Project Manager for a total of 8 years in the states of Arizona and Nevada.

5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

6. I am a contributing author for the preparation of the technical report titled "Mexican Hat Project, NI 43-101 Technical Report, Preliminary Economic Assessment", (the "Technical Report"), dated effective December 6, 2018, prepared for GMV Minerals Inc.; and am responsible for Sections 17, 21.2.2 and corresponding subsections of 1, 25 and 26.

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

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

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

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

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 18th day of December, 2018.

(Signed) (Sealed) *Justin Miles Black*
Signature of Qualified Person

Justin Miles Black
Print Name of Qualified Person

CERTIFICATE OF QUALIFICATION

John R.W. Fox, P.Eng

I, John R.W. Fox, of 1677 Deep Cove Road, N. Vancouver, British Columbia, V7G 1S4, do hereby certify that:

1. I am a consulting mineral processing engineer with an office at 302-304 West Cordova Street Vancouver, British Columbia, V6B 1E8.

2. This certificate applies to the "Mexican Hat Project, NI 43-101 Technical Report, Preliminary Economic Assessment" of GMV Minerals Inc. in Arizona USA, dated 06 December 2018 prepared for GMV Minerals Inc., Vancouver, B.C.

3. I am a graduate of the University of Leeds in 1971 with a B.Sc in Applied Mineral Sciences. I am a member in good standing of the Association of Professional Engineers and Geoscientists of the Province of British Columbia License #12578. I have practised my profession as a mineral processing engineer throughout the world continuously since 1971.

4. I directed metallurgical testwork and participated in the August 2018 NI 43-101 report on the Mexican Hat property. I have not visited the property.

5. I am responsible for Section 13 and contributed to the metallurgical components in section 1, 25 and 26.

6. I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of education, experience, independence and affiliation with a professional association, I meet the requirements of an Independent Qualified Person as defined in National Policy 43-101.

7. I am not aware of any material fact or material change with respect to the subject matter of the technical report that is not reflected in the "Mexican Hat Project NI 43-101 Technical Report, Preliminary Economic Assessment".

8. I have read National Instrument 43-101, Standards for Disclosure of Mineral Properties. This technical report has been prepared in compliance with National Instrument 43-101.

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

10. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public files on their websites accessible by the public.

Dated this 18th day of December 2018 in Vancouver, B.C.

\Original Signed and Sealed by John R.W, Fox P.Eng.

CERTIFICATE OF QUALIFIED PERSON
John M. Marek

I, John M. Marek P.E. do hereby certify that:

1. I am currently employed as the President and a Senior Mining Engineer by:

    Independent Mining Consultants, Inc.
    3560 E. Gas Road
    Tucson, Arizona, USA 85714

2. I graduated with the following degrees from the Colorado School of Mines
    Bachelors of Science, Mineral Engineering – Physics 1974
    Masters of Science, Mining Engineering 1976

3. I am a Registered Professional Mining Engineer in the State of Arizona USA
    Registration # 12772
    I am a Registered Professional Engineer in the State of Colorado USA
    Registration # 16191
    I am a Professional Engineer, Yukon Territory, Canada
    I am a Registered Member of the American Institute of Mining and Metallurgical Engineers, Society of Mining Engineers, Registration # 2021600

4. I have worked as a mining engineer and reserve estimation specialist for more than 41 years. I have prepared mine plans for numerous gold heap leach projects over that time. I have acted as the Qualified Person on these topics for Technical Reports.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI43-101.

6. I am responsible for Sections 16, 21.1.1, 21.2.1, and corresponding sections of 1, 25 and 26, of the Technical Report titled "Mexican Hat Project, NI 43-101 Technical Report, Preliminary Economic Assessment" with an effective date of 06 December 2018.

7. I visited the Mexican Hat Project on 28 August 2018, during which times I reviewed the site conditions, surface geology, and selected drill core.

8. Independent Mining Consultants, Inc., and this author have not previously worked on the Mexican Hat Project.

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

10. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

11. I am independent of the issuer applying the tests in Section 1.5 of NI 43-101.

12. I have read national Instrument 43-101 and Form 43-101F1, and to my knowledge, the Technical Report has been prepared in compliance with that instrument and form.

13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated: 18 December 2018

(Signed) (Sealed) *John M. Marek*
John M. Marek, RM-SME

CERTIFICATE OF QUALIFIED PERSON
James Barr, P.Geo.

I, James Barr, P.Geo., of Kelowna, British Columbia, do hereby certify:

1. I am Senior Geologist and Team Lead with Tetra Tech Inc. with a business address at Suite 150 - 1715 Dickson Avenue, Kelowna, BC, V1Y 9G6.

2. I am a registered Professional Geoscientist with the Engineers and Geoscientists of British Columbia (#35150).

3. Since 2003 I have worked as an exploration and resource geologist for numerous precious and base metal projects in Canada, Africa and Mexico, and have been preparing mineral resource estimates since 2008, including for open pit and underground precious metal vein hosted deposits.

4. I graduated from the University of Waterloo in 2003 with a B.Sc. (Honours) in Environmental Science, Earth Science and Chemistry.

5. This certificate applies to the technical report entitled "Mexican Hat Project, NI 43-101 Technical Report, Preliminary Economic Assessment" (the "Technical Report") dated effective December 6, 2018.

6. I am responsible for Sections 12, 14, and corresponding sections of 1, 25 and 26 of this Technical Report.

7. I visited the Property that is the subject of the Technical Report from July 18-19, 2017.

8. I am independent of GMV Minerals Inc., as defined by Section 1.5 of the Instrument.

9. I have had prior involvement with this Property as co-author of the technical report entitled "2018 Technical Report and Mineral Resource Estimate on the Mexican Hat Project Cochise County, Arizona, USA" with effective date of June 22, 2018.

10. I confirm that I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with them.

11. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections within the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 18th day of December, 2018 at Kelowna, British Columbia

(signed) (Sealed)

_____
James Barr, P.Geo.
Senior Geologist and Team Lead - Geology
Tetra Tech Inc.

TETRA TECH

Dawn H. Garcia, PG, CPG

8261 East Placita del Oso
Tucson, Arizona 85750
USA
T (520) 471-9387
E dawnhgarcia@q.com

CERTIFICATE of AUTHOR

Dawn H. Garcia

This Certificate of Author has been prepared to meet the requirements of National Instrument 43-101 Standards of Disclosure for Mineral Projects as published 09 May 2016, Part 8.1. This certificate applies to the "Mexican Hat Project, NI 43-101 Technical Report, Preliminary Economic Assessment" herein referred to as the "Technical Report," dated effective December 6, 2018.

I, Dawn Garcia, hereby certify that:

1. I reside at 8261 East Placita del Oso, Tucson, Arizona 85750, USA.

2. I am a graduate from Bradley University in Peoria, Illinois, with a Bachelor of Art, Geological Sciences (1981) and from California State University with a Master of Sciences, Geology (Hydrogeology Emphasis) (1995). I have practiced my profession continuously since 1985. I am an independent consultant.

3. I am a licensed Professional Geologist in Arizona (License No. 26034), Alaska (License No. 610), and California (License No. 5425) and a Certified Professional Geologist registered with the American Institute of Professional Geologists (Membership No. CPG-8313). I am also a registered member of the Society for Mining, Metallurgy & Exploration (Membership No. 4135993).

4. I am a qualified person for the purpose of NI 43-101 with regards to Environmental, Permitting, Social, and water-related aspects. I have worked as an environmental Geologist and Hydrogeologist for 33 years since graduation. My relevant experience for the purpose of this Technical Report is:

   - Acted as the Qualified Person for the Environmental, Permitting, and Social sections for 8 NI 43-101-compliant technical reports and more than 16 detailed environmental and permitting reviews.

   - Conducted environmental, socio-economic, or water-related tasks for over 50 mineral development, mineral processing, and mining operations.

5. I visited the property once on November 5, 2018.

6. I was solely responsible for Section 20.0 of the Technical Report and for contributing to portions of Sections 1, 18.3, 25 and 26.

7. I am independent of GMV Minerals, applying the definition of independence set out in Section 1.5 of NI 43-101.

8. Neither I, nor any affiliated entity of mine, is at present under an agreement, arrangement, or understanding or expects to become an insider, associate, affiliated entity, or employee of GMV Minerals or any associated or affiliated entities.

9. Neither I, nor any affiliated entity of mine, own—directly or indirectly—nor expect to receive any interest in the properties or securities of GMV Minerals or any associated or affiliated companies.

10. Neither I, nor any affiliated entity of mine, have earned the majority of our income during the preceding 3 years from GMV Minerals or any associated or affiliated companies.

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

12. I have read NI 43-101, NI 43-101 CP, and Form 43-101F1 and have prepared the Technical Report in compliance with NI 43-101, NI 43-101 CP, and Form 43-101F1. I have prepared the Technical Report in conformity with generally accepted Canadian mining industry practice, and as of the date of the certificate, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 18th day of December 2018.

(Signed) (Sealed) *Dawn H. Garcia*
Dawn H. Garcia