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Prepared for:

GMV MINERALS INC.

NI 43-101 Technical Report
Updated Preliminary Economic Assessment ('PEA')
Mexican Hat Project
Cochise County, Arizona, USA

Report Issue Date: November 19, 2020; Revision 0

Effective Date: October 20, 2020

GMV MINERALS INC.

REPORT DATE AND QUALIFIED PERSON SIGNATURES PAGE

As required by National Instrument 43-101, the effective date of this report is October 20, 2020. The issue date of this report is November 19, 2020. See Appendix 28.1 for certificates and consent letters of the Qualified Persons (QP) or authors contributing to the information contained in this report. These QP certificates are considered the date and signature of this report in accordance with Form 43-101F1.

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Above mean sea level	amsl
Acre Foot	ac-ft
Acre-feet per year	AFA
Adsorption-Desorption-Recovery	ADR
Ampere	A
Annum (year)	a
Bank cubic meter	BCM
Copper	Cu
Cubic meter	m^3
Cubic meters per day	m^3/d
Cubic meters per hour	m^3/h
Day	d
Days per week	d/wk
Days per year (annum)	d/a
Degree	°
Degrees	deg
Degrees Celsius	°C
Diameter	ø
Dry metric tonne	dmt
Feet	ft
Gallon	gal
Gallon per Minute	gpm
Gold	Au
Gram	g
Grams per cubic centimeter	g/cc
Grams per liter	g/L
Grams per tonne	g/t
Greater than	>
Hectare (10,000 m^2)	ha
Hertz	Hz
Horsepower	hp
Hour	hr
Hours per day	h/d
Hours per week	h/wk
Hours per year (annum)	h/a
Internal Rate of Return	IRR
Joule (Newton-meter)	J
Kilometer	km
Kilowatt-hour	kWh
Kilowatt-hours per short ton (US)	kWh/t
Kilowatt-hours per tonne (metric tonne)	kWh/mt
Kilowatt-hours per year	kWh/y
Kilowatts adjusted for motor efficiency	kWe

Lead ..... Pb
Less than ..... <
Life-of-Mine ..... LOM
Liter L
Liters per day ..... L/d
Liters per minute ..... L/m
Liters per second ..... L/s
Loose cubic meters ..... LCM
Megawatt ..... MW
Megabytes per second ..... Mb/s
Meter ..... m
Meters above sea level ..... masl
Meters per second ..... m/s
Metric tonne ..... t
Metric tonne ..... mt
Micrometer ..... micron
Microsiemens (electrical) ..... µS
Miles per hour ..... mph
Million ..... M
Million metric tonnes (megatonne) ..... mmt
Minute (plane angle) ..... '
Minute (time) ..... min
Molybdenum ..... Mo
Month ..... mo
Newton ..... N
Net Present Value ..... NPV
Ohm (electrical) ..... Ω
Ounce (troy) ..... ozt
Parts per billion ..... ppb
Parts per million ..... ppm
Pascal (Newtons per square meter) ..... Pa
Pascals per second ..... Pa/s
Percent ..... %
Percent moisture (relative humidity) ..... %RH
Phase (electrical) ..... Ph
Potential of Hydrogen (*i.e.* acidity or alkalinity level) ..... pH
Pound (avoirdupois) ..... lb
Power factor ..... pF
Pregnant Liquor Solution ..... PLS
Preliminary Assessment ..... PA
Professional Engineer ..... PE
Revolutions per minute ..... rpm
Run-of-Mine ..... ROM
Samuel Engineering, Inc. ..... SE

Second (plane angle).."
Second (time)...s
Short ton (2,000 lb)...st
Short ton (US)...st
Short tons per day (US)..stpd
Short tons per hour (US) .... stph
Short tons per year (US)...stpy
Silver .... Ag
Specific gravity .... sg
Square meter...m^2
Square feet... ft^2
Tailings Storage Facility .... TSF
Tonne (1,000 kg) .... t
Tonnes per annum .... t/a
Tonnes per day .... t/d
Tonnes per hour...t/h
Total dissolved solids .... TDS
Total suspended solids..TSS
Volt V
Volt-Ampere ....VA
Waste Rock Storage Facility .... WRSF
Watt (Joules per second) ....W
Week .... wk
Weight/weight .... w/w
Wet metric ton .... wmt
Yard ....yd
Year (annum) ....a
Year (US)..yr

Measurement Units and Symbols

The units used in this report are the International System of Units (SI). The reference conditions for gas volume are 0°C and 101.325 kPa, corresponding with a molar (ideal) gas volume of 22.414 m^3/ (kg-mol). This is shown as "m^3 (normal)" or abbreviated to (non-SI) "Nm^3." The unit "t" rather than Mg is used for 1,000 kilograms mass. The dimensionally independent SI base units are shown in Table 1-1. The permitted base units are shown in Table 1-2.

Table 1-1
SI Base Units

Quantity	Unit	Symbol
Length	Meter	m
Mass	Kilogram	kg
Time	Second	s
Electric Current	Ampere	A
Thermodynamic Temperature	Kelvin	K
Amount of Substance	Mole	mol
Luminous Intensity	candela	cd

Table 1-2
Permitted Base Units

Quantity	Unit	Symbol	Definition
Time	Minute	min	60 seconds
	Hour	h	60 minutes
	Day	d	24 hours
	Calendar Year	y	365 days
Mass	Metric Tonne	t	1,000 kg

SI prefixes, as listed in Table 1-3, are used only with SI base units. It is incorrect to use these prefixes with the permitted base units shown in Table 1-2.

Table 1-3
SI Base Units

Power	Prefix	Symbol	Decimal Equivalent
10^{24}	yotta-	Y	1,000,000,000,000,000,000,000,000
10^{21}	zeta-	Z	1,000,000,000,000,000,000,000
10^{18}	exa-	E	1,000,000,000,000,000,000
10^{15}	peta-	P	1,000,000,000,000,000
10^{12}	tera-	T	1,000,000,000,000
10^{9}	giga-	G	1,000,000,000
10^{6}	mega-	M	1,000,000

Table 1-3
SI Base Units

Power	Prefix	Symbol	Decimal Equivalent
10^3	kilo-	K	1,000
10^2	hecto-	H	100
10^1	deca-	Da	10
10^0			1
10^{-1}	deci-	D	0.1
10^{-2}	centi-	C	0.01
10^{-3}	milli-	M	0.001
10^{-6}	micro-	M	0.000 001
10^{-9}	nano-	N	0.000 000 001
10^{-12}	pico-	P	0.000 000 000 001
10^{-15}	femto-	F	0.000 000 000 000 001
10^{-18}	atto-	A	0.000 000 000 000 000 001
10^{-21}	zepto-	Z	0.000 000 000 000 000 000 001
10^{-24}	yocto-	Y	0.000 000 000 000 000 000 000 001

The prefixes and prefix symbols are used with the SI base and derived units – except for kg. The base mass unit, kg, already has a prefix, hence the SI prefixes are applied to the unit gram (g). In this manner, the symbol for metric tonne is Mg; however, in this report the permitted alternate, t as listed above, is used.

1.0 SUMMARY

GMV Minerals Inc. ("GMV"), a junior gold development company listed on the TSX Venture Exchange (TSXV) where it trades under the symbol GMV, and listed on the OTCQB under symbol GMVMF, engaged the services of Samuel Engineering, Inc. ("SE"), in conjunction with Mine Development Associates ("MDA"), a division of RESPEC, Tierra Group International Ltd. ("Tierra Group"), Golder Associates Inc. ("Golder"), Tetra Tech Inc. ("Tetra Tech"), and DRW Geological Consultants Ltd. ("DRW") to prepare a Canadian National Instrument 43-101 ("NI 43-101") Technical Report as a Preliminary Economic Assessment (PEA) on its Mexican Hat property located in Arizona, USA.

This Technical Report addresses all technical and cost aspects of the Mexican Hat Project (the "Project") for geology, resource, mining, processing, environmental and infrastructure design, capital and operating costs, and economic analysis. Gold-bearing resources considered in this report would be mined by open pit, crushed, and loaded onto heap leaching pads for recovery of contained gold by conventional processing methods. Production data in the Technical Report are stated in metric units.

The effective date of this Technical Report is October 20, 2020.

1.1 KEY PROJECT INFORMATION

- GMV has 100% interest in the Project.
- The Project's inferred resources can be mined and processed using conventional technologies to produce gold doré.
- The Project is subject to a 3% net smelter returns royalty (NSR Royalty). GMV has the option to reduce this royalty to1.5% with a buy back payment of $1.5 M. This option has been included in the Project's economic analysis.
- Inferred resources are estimated at 36.733 Mt at a gold grade of 0.58 g/t using a cut-off grade of 0.20 Au g/t contained in the open pit deposits.
- The inferred resources will be mined by conventional open pit at a low, life of mine (LOM) stripping ratio of 1:87:1 waste to material leached.
- A total of 32.632 Mt will be mined from the inferred resources, crushed and placed on the heap leach pads for leaching with sodium cyanide and subsequent processing of the gold-bearing solution in an adsorption, desorption, recovery (ADR) plant for producing gold doré.
- The PEA is designed for contractor mining and crushing as opposed to owner operation.
- Contractor mining and crushing will be done at a nominal production rate of 10,000 tpd delivery to a crushing plant and lined heap leach pad.
- Gold recovery projected from preliminary metallurgical testing is 88% with an estimated sodium cyanide consumption of 0.3 kg/t of material leached.
- LOM gold production is estimated in the gold doré at 525,000 ounces.
- Initial capital cost of the Project is estimated at $67.8 M including mine, process plant, infrastructure, and heap leach pad construction. LOM sustaining capital is estimated at $13.0 M.

- Operating C1 cash cost is estimated at an average $951 per ounce of gold produced ($15.30 per tonne processed) and an all-in sustaining costs ("AISC") of $1,136 per ounce of gold produced.
- The Project will require various Arizona state and federal authorizations, licenses and permits for construction, operation, closure, and post-closure.
- Project economic analysis at gold price of $1,600/oz yields a pre-tax Internal Rate of Return ("IRR") of 39.3% (after tax 29.3%) and a pre-tax net present value ("NPV") at a 5% discount rate of 150.6 million (after tax $100.0 million) with a 2.85 year payback of invested capital. Below is a summary of the pre and after tax financial indicators.

Financial Indicators Pre Taxes	Values
NPV cash flow (undiscounted)	US$220.4 M
NPV @ 5%	US$150.6 M
IRR %	39.3%
Payback (years)	2.85

Financial Indicators After Taxes	Values
NPV cash flow (undiscounted)	US$153.0M
NPV @ 5%	US$100M
IRR %	29.3%
Payback (years)	2.85

- Engineering design analysis indicates the potential to increase pit size and contained ounces with increased gold prices.
- Based on the study results, it is recommended to advance the project to a Pre-feasibility Study.

1.2 PROJECT LOCATION

The Mexican Hat property is located in Cochise County, Arizona, 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) (Figure 1-1).

Figure 1-1: Mexican Hat Project Location

1.3 PROPERTY DESCRIPTION

The Mexican Hat property is in Cochise County in southeastern Arizona immediately north of the Gleeson Courtland district, 10 km (6 miles) south of Pearce, Arizona, and approximately 140 km (90 miles) by road southeast of Tucson, Arizona. GMV 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 (News Release; GMV Minerals Inc., dated May 30, 2014). No major environmental liabilities have been noted by the QP (Dave Webb).

The Mexican Hat Property is considered to host a low sulphidation alkaline epithermal gold deposit in Tertiary volcanic rocks. The Mexican Hat Project is currently an undeveloped gold project and not in commercial production. The proposed development of the Project would be by open pit mining, crushing of mined gold mineralized material with subsequent conveyance to heap leaching with cyanide solution for recovery of the contained gold into a pregnant liquor solution (PLS), The PLS from heap leaching would be pumped in an ADR plant for the production of gold doré. Contractors would be used for both mining and crushing operations.

1.4 HISTORY

There is a general lack of recorded information available on the project in the historical record prior to the 1980's. The area around and immediately south of the project area underwent mining activity during the 16th or 17th century by early Indians seeking turquoise, semi-precious gemstone used for decorative purposes. Later Spanish explorers apparently worked the area for gold. Early exploration at Mexican Hat was reportedly done during the 1930's on a portion of the present project area under the property name of the Gold Band prospect. In 1989, Oneida Resources Inc. ("ODI") of Vancouver, B.C. optioned a smaller portion of the present day Mexican Hat Gold Property ("MHGP") and conducted surface exploration work completing a 1,524 m percussion drill program comprised of 20 holes. ODI geologists collected rock chip samples from channel sample bulldozer trenches and had the samples analyzed for gold. Placer Dome (USA) Inc ("PDI") assumed operation of the project, in 1989 and had the pulp samples of the Oneida sampling program fire assayed for gold (Au) and silver (Ag) including a 14-element geochemical package. Subsequent exploration programs were conducted between 2008-2013 by various entities including Kalahari Resources, N.A. Pearson (lessee), Capitol Hill, and Auracle Resources.

GMV Minerals Inc. acquired 100% of the leasehold interest in the Mexican Hat Property in 2014.

1.5 GEOLOGIC SETTING 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 kilometres of ground magnetics, completion of three lines of audiomagnetotellurics geophysics, three gravity profiles, and drilling including 15 reverse circulation (RC) holes totaling 4,776.5 m in 2016, and in 2017 completed eight HQ core holes totaling 1,979.3 m and 15 RC holes totaling 4,032.9 m. The 2018 exploration program saw the completion of a further 11 RC holes totaling 3,250 m and the collection of 1,064 RC samples.

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, one west and one south of Mexican Hat are connected by a left lateral north dipping jog referred to as Zone 7.

Gold and silver mineralization is associated with moderate to strongly 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 Zone 7 Fault. Mapped gold mineralization is hosted in structures within all observed major rock units.

1.6 EXPLORATION

Recent exploration programs at Mexican Hat was initiated in 2014 by trench mapping. Continuous chip sample trenches were geologically mapped, and the results determined the existence of two types of structural controls on mineralization. GMV has completed four drill campaigns on the Mexican Hat Property since 2014 include 15 reverse circulation (RC) holes totaling 4,776.5 m in 2016, and in 2017 8 HQ core holes totaling 1,979.3 m and 15 RC holes totaling 4,032.9 m. In 2019, an additional 11 RC holes totaling 3,250 m were completed. Additional exploration programs and studies completed in 2016-2017 have included surficial geochemistry, airborne photogrammetry, geophysics, magnetics, and gravity.

1.7 MINERAL RESOURCE ESTIMATE

Resource modeling has been completed based on assay results from 45 core holes (totaling 2,650 assays), 38 reverse circulation (RC) holes (totaling 3,372 assays), 120 rotary holes (totaling 5,536 assays) and 149 trenches (totaling 1,864 assays). 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 seven (7) mineralized domains to represent the Zone 7 fault, the NE-SW trending fractured zones, and a higher grade core (>1.0 g.t) which appears to be present within a surrounding lower grade halo.

The Mexican Hat Deposit hosts an Inferred Mineral Resource of 36,733,000 tonnes grading 0.58 grams of gold per tonne, equivalent to 688,000 troy ounces of gold using a 0.20 g/t cut-off grade.

Table 1-1 Mineral Resource Statement, Mexican Hat Project, Arizona, USA, Effective Date June 22, 2020					
Category	**Cut-off (g/t Au)**	**Grade (Au, g/t)**	**Tonnes**	**Gold Oz**	**Strip Ratio**
Inferred	0.20	0.58	36,733,000	688,000	2.36

- 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 = $1,375 per troy ounce, throughput at 15,000 tpd, discount rate = 5%. A cost of $0.03 was added per bench to the mining cost below the existing level surface.
- A top cut of 32 gpt gold is applied to all zones except Zone 6 which has a top cut of 50 gpt gold.
- Mineral Resources have been calculated using the Inverse Distance Squared method.
- 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.
- The stated strip ratio of 2.36 in Table 1-1 represents the ratio of tonnes of gold resources (36.733 M t) above the gold cut-off grade (0.20 g/t Au) to the estimated tonnes of waste material below the cut-off grade. This ratio does not represent the inferred gold resources to waste that are calculated in the mine plan for extracting the gold materials for processing and producing gold at the stated mining parameters.

1.8 MINING

The PEA presented in this report considers open-pit mining of the Mexican Hat gold deposit. Note that a PEA is preliminary in nature and includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied that would enable them to be classified as mineral reserves. There is no certainty that the economic results of the PEA will be realized.

The methodology used for mine planning to define the economics for the PEA includes:

- Define assumptions for the economic parameters;
- Define geometric parameters and constraints;
- Run pit optimizations;
- Define road and ramp parameters;
- Create pit designs;
- Create dump designs;
- Produce mine and process production schedules;
- Define personnel and equipment requirements;
- Estimate mining costs, based on contractor mining and crushing as opposed to owner operation; and
- Perform an economic analysis (Completed by SE).

Economic parameters were used to generate optimized pits using a Lerches Grossman algorithm within Whittle™ software (Version 4.7). The economic parameters include mining costs, process cost, general and administrative costs ("G&A"), refining costs, royalties, and metal recoveries. The economic parameters used for the pit optimizations are shown in Table 1-2.

Table 1-2 Economic Parameters

	Base Case	Units
Mining Cost - OP	$ 2.68	$/tonne Mined
Processing Cost	$ 5.15	$/tonne Processed
G&A Cost	$ 2,730	K USD/yr
G&A Cost	$ 0.78	$/tonne Processed
Throughput	10,000	TPD
Throughput	3,500	K TPY
Refining Cost	$ 5.00	$/oz Processed
Recovery - Au	88%	
Payable - Au	100%	
Royalties - Hernandez	1.5%	
Royalties - Victor	0%	
Gold Price	$ 1,500	$/oz Au

A cut-off grade of 0.14 g Au/t was estimated based on the economic parameters. As this cut-off grade is very low with respects to detection limits for assays, a 0.17 g Au/t was used for the project. Pit optimizations were run using Whittle™ software (version 4.7). Inputs into Whittle included the resource block model along with the economic and geometric parameters previously discussed. Ultimate pit shells were selected from the Whittle results for final design. Detailed pit designs were completed for Mexican Hat using Surpac™ software (version 6.7). Each of the designs utilize 6.0 m benches with a catch bench installed every third bench, or 18 m. A bench face angle of 66° was used resulting in an inner-ramp angle of 45° when catch benches were included.

Five pit phases were used achieve the ultimate pit. Ramps of 28 m widths were used to allow access by 91-tonne (100-ton) class trucks. In-pit mineral resources were estimated for the Mexican Hat pit design and are tabulated in Table 1-3. The Mexican Hat pits have a total of 61.1 Mt of waste associated with the material to be processed, and thus have an overall strip ratio of 1.87 t of waste per tonne processed.

Table 1-3 Mexican Hat In-Pit Resources

	Units	Inferred	Waste
Mexican Hat	K Tonnes	32,632	61,115
	g Au/tonne	0.569	
	K Ozs Au	597	

Dump designs were created for the PEA to contain the waste material mined. A 1.3 swell factor was assumed which provides for both swell when mined and compaction when placed into the facility. The South pit was assumed to be mined first followed by the North pits. The South pit cuts across a major drainage which is typically dry. The South pit is assumed to be backfilled to re-establish the drainage. In addition, a single large dump to the east of the pit was planned for containment of the remaining waste material mined.

Production scheduling was completed using Geovia's MineSched™ (version 9.1) software. Inferred resources inside of the pit designs previously discussed were used to schedule mine production. The production schedule considers the processing of material by crushing followed by heap leaching. Monthly periods were used to create the production schedule with pre-stripping starting in Mexican Hat at month -6. The start of processing was assigned to month 1 though no gold production is realized until month 2. The maximum rate for processing was 10,000 tonnes per day or 3,500,000 tonnes per year on a 350-day basis.

The total mining rate would ramp up from 5,000 tonnes per day to about 26,500 tonnes per day over a period of 12 months. A maximum of 46,000 tonnes per day is used in later years when the stripping becomes more significant. The monthly mining production for Mexican Hat is summarized yearly in Table 1-4.

Table 1-4 Mexican Hat Mine Production Schedule

	Units	Yr_-1	Yr_1	Yr_2	Yr_3	Yr_4	Yr_5	Yr_6	Yr_7	Yr_8	Yr_9	Yr_10	Total
Pit to Stockpile	K tonnes	318	332	180	392	206	258	191	339	0	0	0	2,215
	g Au/tonne	0.543	0.222	0.207	0.229	0.221	0.206	0.218	0.269	0.208	0.209	0.213	0.273
	K Ozs Au	6	2	1	3	1	2	1	3	0	0	0	19
Pit to Crusher	K tonnes	-	2,956	3,339	3,249	3,107	3,341	3,306	3,173	3,500	3,500	947	30,418
	g Au/tonne	-	0.574	0.661	0.659	0.601	0.483	0.562	0.731	0.491	0.547	0.647	0.590
	K Ozs Au	-	55	71	69	60	52	60	75	55	62	20	577
Total Ore Mined	K tonnes	318	3,289	3,519	3,641	3,312	3,599	3,498	3,511	3,500	3,500	947	32,632
	g Au/tonne	0.543	0.539	0.638	0.612	0.578	0.463	0.543	0.687	0.491	0.547	0.647	0.569
	K Ozs Au	6	57	72	72	62	54	61	78	55	62	20	597
Total Waste	K tonnes	1,597	6,001	4,671	3,679	5,368	4,446	8,697	12,925	8,896	4,167	667	61,115
Total Mined	K tonnes	1,915	9,290	8,190	7,320	8,680	8,045	12,195	16,436	12,396	7,667	1,613	93,748
Strip Ratio	W:O	5.03	1.82	1.33	1.01	1.62	1.24	2.49	3.68	2.54	1.19	0.70	1.87
Rehandle	K tonnes	-	544	161	261	393	159	194	337	-	-	167	2,215
	g Au/tonne	-	0.413	0.206	0.240	0.215	0.207	0.217	0.269	-	-	0.209	0.273
	K Ozs Au	-	7	1	2	3	1	1	3	-	-	1	19

The process production schedule was created by MDA based on the mine production schedule and recoveries and lag times estimated by SE. The lag times were assumed over a 14-month period. A 0% recovery was assumed during the month placed followed by 67.5% of recoverable ounces during month 1 after placement. The remaining production of recoverable ounces was achieved over the remaining 13 months.

Longer term stockpiling of up to 250,000 t of material grading 0.21 g Au/t was assumed to increase the grade processed in the earlier years. This stockpile will be near the crusher. Equipment and personnel requirements for mine production were assumed to be provided by a mining contractor in order to reduce capital costs. 992-CAT type of loaders with 91-t capacity haul trucks are assumed. Production drilling would be done using 45,000-pound pulldown blast-hole drills and additional support equipment for road, pit, and dump maintenance would be provided by the contractor.

Additional mining personnel will be provided by the owner to provide supervision, engineering, and geology services. This would include mine planning, mineralized material control, and surveying.

1.9 METALLURGICAL TEST WORK

Two relevant metallurgical test programs have been performed on samples from the Mexican Hat Project. The first program, in 2015 was performed at McClelland Labs of Sparks, Nevada followed by a second program in 2016 at Bureau Veritas of Richmond, British Columbia, Canada. Both laboratories are accredited and considered as meeting industry standards for metallurgical test. Four types of mineralization were identified during preliminary geological assessments: latite comprising approximately 80% of the mineralization, with 8% each of andesite and basalt, and the remaining 4% dacite. Table 1-5 summarizes the metallurgical test results for McClelland and Bureau Veritas.

Table 1-5
Summary of Mexican Hat Metallurgical Test Work

Laboratory	Test Type	Range of Gold Recoveries, %
McClelland	Bottle Roll	82.2-97.6
McClelland	Column	77.1-95.0
Bureau Veritas	Bottle Roll	80.9-95.0
Bureau Veritas	Column	96.4-98.2

Based on these test results, a gold recovery of 88% for heap leaching has been used in this study.

1.10 RECOVERY AND PROCESSING METHODS

Gold production at Mexican Hat will be done using conventional heap leach recovery methods in the U.S. gold mining industry. The Mexican Hat gold deposit will be mined as an open pit. The nominal production capacity of processing is 10,000 t/d of mineralized gold resources to the crushing plant. Crushed material will be hauled from the open pit to a crushing circuit where material will be crushed to 80% passing 38 mm (P_{80}) and conveyed to a stockpile. From the stockpile, crushed material will be conveyed and stacked onto the heap leach pad (HLP). The crushed gold material will be conveyed by the discharge conveyor, grasshopper conveyors and a radial stacking conveyor for loading onto the HLP. A total of 32.6 M tonnes of crushed material will be placed on the HLP over the LOM. The HLF will be a single-use, multi-lift type HLF designed with a lining system in accordance with Best Available Demonstrated Control Technology (BADCT) criteria as described in Arizona Department of Environmental Quality (ADEQ) Arizona Mining BADCT Guidance Manual. Cyanide solution will be irrigated by wobbler sprinklers or buried driplines onto the HLP. Leachate will percolate through the heap to dissolve gold into the pregnant leach solution (PLS). The PLS from heap leaching will be collected in a lined pond from where the PLS will be pumped to an ADR and refinery plant for gold recovery and production of gold doré bars.

The LOM gold production in gold doré bars is estimated at 525,000 ounces based on the mine plan and 88% gold recovery.

1.11 INFRASTRUCTURE

Project infrastructure for Mexican Hat has been developed to support the mining, crushing, heap leaching and ADR operations. The infrastructure facilities would include project access, site guardhouse, site security fencing/cameras, truck scale, power supply, fresh water source/system, monitoring wells, and non-operational buildings. Groundwater will be used as the source of water for mining and processing operations.

1.12 ENVIRONMENTAL, STUDIES, PERMITTING, AND SOCIAL, OR COMMUNITY IMPACTS

The Project will require various state and federal authorizations, licenses and permits for Project construction, operation, closure, and post-closure. Comprehensive environmental and socioeconomic baseline studies will be required. No environmental baseline studies 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 both state and federal agencies. It is anticipated that the State of Arizona environmental permitting will be relatively straightforward because the discharging facilities will be designed and constructed using Best Available Demonstrated Control Technology (BADCT) standards, which allow for prescriptive design to facilitate permitting. Federal permitting is anticipated to be more complex due to the requirement to evaluate a range of alternatives. The National Environmental Policy Act (NEPA) will be triggered because the waste rock storage facility will be on federal Bureau of Land Management (BLM) land. Recent changes to NEPA include presumptive time limits, which will benefit the permitting timeline.

1.13 CLOSURE

The closure strategy involves returning the mine site and affected areas to viable and, wherever practicable, self-sustaining ecosystems that are compatible with a healthy environment. Key activities of closure will be decommissioning equipment and waste management; demolition of physical structures and management of infrastructure; characterization and mitigation of contaminated soils; regrading and contouring to allow for stormwater drainage; and revegetation of disturbed land. Facilities remaining after closure will be an open pit, the heap leach facility and waste rock storage facilities. The drain down of solution in the heap leach will need to be managed until discharges meets applicable environmental regulatory standards.

1.14 CAPITAL COSTS

The initial capital cost for the Project is estimated at $67.847 M as summarized in the below Table 1 6.

Table 1-6: Capital Cost Summary

Cost Components	Mine & Crushing	Leach Pad, Ponds & Pipelines	ADR, BOP & Infrastructure	Substation & Power	Total Capital Cost
Description	**Cost (USD)**	**Cost (USD)**	**Cost (USD)**	**Cost (USD)**	**Cost (USD)**
Directs					
Mechanical Equipment	-	2,712,000	6,741,000	-	9,453,000
Civil	-	7,370,000	584,000	83,000	8,037,000
Foundations	-	-	646,000	200,000	846,000
Structures	-	-	378,000	125,000	503,000
Buildings/Laboratories	-	-	1,359,000	-	1,359,000
Insulation	-	-	-	-	-
Piping	-	3,050,000	2,270,000	-	5,320,000
Electrical	-	-	706,000	1,838,000	2,544,000
Instruments	-	-	353,000	-	353,000
Miscellaneous	-	-	182,000	-	182,000
Subtotal Directs	**-**	**13,132,000**	**13,219,000**	**2,246,000**	**28,597,000**
Indirects					
Contractor Indirect	-	1,114,000	1,559,000	476,000	3,149,000
Construction Equipment	-	557,000	779,000	238,000	1,574,000
Surveying & Testing Svcs	-	139,000	225,000	60,000	424,000
EP Services	-	550,000	1,182,000	218,000	1,950,000
Construction Mgmt	-	446,000	934,000	135,000	1,515,000
Vendor Reps	-	68,000	151,000	21,000	240,000
Spare Parts	-	34,000	76,000	10,000	120,000
Initial Fills	-	25,000	250,000	10,000	285,000
Commissioning	-	104,000	146,000	45,000	295,000
Freight	-	137,000	501,000	45,000	683,000
Crushing Equipment-mob	3,000,000	-	-	-	3,000,000
Contractor Mining	2,430,000	-	-	-	2,430,000
Preproduction	4,300,000	-	-	-	4,300,000
Owner's Cost, incl Royalty	2,577,000	-	3,490,000	-	6,067,000
Taxes	157,000	217,000	529,000	47,000	950,000
Subtotal Indirects	**12,464,000**	**3,391,000**	**9,822,000**	**1,305,000**	**26,982,000**
Contingency	2,729,000	4,042,000	4,609,000	888,000	12,268,000
Total Cost (USD)	**15,193,000**	**20,565,000**	**27,650,000**	**4,439,000**	**67,847,000**

Project execution assumed for the capital estimate will follow a typical Engineering, Procurement, and Construction Management (EPCM) approach. The execution timeframe considered is approximately 24 months from notice to proceed through commissioning completion. Process equipment delivery and leach pad construction will drive the timeline for completion of the project. Project permitting represents the highest risk to the proposed schedule development plan.

The order of magnitude capital cost has been developed to a level sufficient to assess/evaluate the project concept and overall viability. The estimate can be classified as an AACE Class 5 estimate and after inclusion of the contingency of 22%, the estimate is thought be in the accuracy range of minus 20% to plus 35%.

Sustaining capital over the LOM for mining, processing, and expansion of the HLP is estimated at $12.363 M and included in the economic analysis.

1.15 OPERATING COSTS

The operating costs for the Mexican Hat Project were developed based on a combination of direct build up from production and metallurgical parameters, typical unit consumption and costs for similar operations and factoring. The operating costs for the Mexican Hat Project are summarized in Table 1-7. No contingency has been included in the operating costs presented. Taxes are considered in the financial analysis model.

Table 1-7
LOM Operating Costs

Production	Estimated Operating Costs ($000s)/Year				$/Au Oz
Year	Mining (1)	Process (1)	G&A	Total	Recovered
1	25,473	23,240	2,730	51,443	$1,152
2	21,510	23,240	2,730	47,480	$817
3	20,226	23,304	2,737	46,267	$716
4	23,354	23,240	2,730	49,324	$855
5	22,582	23,240	2,730	48,552	$1,037
6	31,348	23,240	2,730	57,318	$1,119
7	44,695	23,304	2,737	70,736	$1,034
8	32,427	23,240	2,730	58,397	$1,120
9	23,783	23,240	2,730	49,753	$1,009
10 (2)	5,420	12,400	1,505	19,325	$622
11 (2)	0	530	29	559	$509
Totals LOM Costs	**$250,817**	**$222,217**	**$26,119**	**$499,154**	**$951**
Total $/t leached	**$7.69**	**$6.81**	**$0.80**	**$15.30**	

(1) Includes contractor costs.

(2) Gold in Years 10 and 11 includes continued production from the leaching of crushed gold material placed on the leach pads in previous years.

1.16 FINANCIAL ANALYSIS AND METRICS

SE has prepared a discounted cash flow analysis of the Mexican Hat Project. Technical and cost inputs for the economic model were developed by SE and consultants with specific inputs provided by GMV. These inputs, which have been reviewed in detail by SE and consultants, are accepted as being reasonable.

The discounted cash flow analysis was performed on a stand-alone project basis with annual cash flows discounted on a beginning-of-period basis. The economic evaluation used a real discount rate of 5% and was performed at commencement of construction (denoted as Year -2 of the Project) using Q2 2020, U.S. dollars.

The economic analysis is a direct result of the capital cost estimate and is therefore considered to have the same level of accuracy, minus 20% to plus 35%.

Table 1-8 presents the economic summary results at a gold price of $1,600/oz. The Project's economics are highly sensitive to the gold price utilized in the calculations.

Table 1-8 Financial Analysis Summary		
Financial Results	**Units**	**Data (Base Case)**
Gold Price	$/oz	$1,600
Free Cashflow	$M	$220.4
Net Present Value (at 5% discount rate) – pre-tax	$M	$150.6
Net Present Value (at 5% discount rate) – after tax	$M	$100.0
Internal Rate of Return (IRR after-tax)	%	29.3%
Internal Rate of Return (IRR pre-tax)	%	39.3%
Payback (After Start of Production)	Years	2.85
C1 Cash cost (1)	$/oz	$973
AISC	$/oz	$1,136

(1) Refers to direct costs, which include costs incurred in mining and processing (labor, power, reagents, materials, etc.) plus local G&A, freight and realization and selling costs.

1.17 OPPORTUNITIES

GMV anticipates advancing the Project to the next stage of development for preparing a PFS Study. Several work programs and studies are recommended to advance the Project from PEA to PFS for improving the Project's opportunities and economics as listed below:

- Drilling:
  - Resources: Convert inferred to measured and indicated, and increase tonnage and grade for mineral reserves:
    - Hydrology: Characterization of hydrogeologic system for sources of water supply and characterization of the aquifer; water samples for permitting and project water balance; preliminary flow modeling to predict inflows to future open pits.
    - Metallurgy: Obtain representative samples for test work.
    - Mineralogy Study
    - Geotechnical:
      - Mine: Pit slope and waste dump designs
      - Heap Leaching; slope design
      - Foundations: Crushing and plant loads
- Labor Study: Availability and labor rates
- Metallurgical Test Program: Conducted on a representative composite basis to optimize process design parameters.
- Transportation Study
- Baseline environmental studies for characterization of environmental setting and mining wastes. These studies would be used for future permit submittals and would include:

- o Hydrologic study to evaluate the source of water supply, characterize the aquifer, and characterize ephemeral surface water
- o Biological studies
- o Jurisdictional water determination
- o Air quality monitoring
- o Cultural resources inventory
- o Socioeconomic baseline study
- o Community outreach program development
- o Geochemistry study of mining wastes
- o Climate study
- o Sediments and soils characterization

An integrated drilling program is an opportunity to reduce overall drilling costs compared to separate programs. An integrated drilling program can be developed to collect data for multiple purposes, such as metallurgical samples, geotechnical parameters, hydrogeological parameters, and water quality.

The results of the above recommendations will impact the technical parameters and economics of the PFS. There exist opportunities to reduce the capital and operating cost estimates used in this study. Based on the inputs used, the project shows merit with a 10-year mine life. Of note, initial designs were created using lower costs for processing and mining than the costs summarized in Section 21.0. Should the capital and operating costs be reduced, there is an opportunity to increase the resources that can be mined. For example, reductions could be made by eliminating contractor mining and processing. However, this will come at additional capital costs as an Owner operation.

1.18 RISKS

Key risks identified with the Project and development plan are as follows:

- The biggest mining risk will be the ability to effectively mine the upper portions of the Project's hill outcrop due to the steep nature of the terrain. In addition, mining of the South Pit is planned to be done first, which reduces the time to get into commercial production, but it will be important to mine the South Pit during the dry season as the pit is in a major drainage.
- Risk exists for the capital and operating cost, and the overall Project economics, should there be a substantial increase in unit costs (utility, fuel, labor, reagents, etc.).
- The Project's economics are very sensitive to gold price which has been highly variable in recent years.
- Risks associated with the project's infrastructure include the confirmation of available water sources for the proposed mining operations from on-site wells. Hydrological drilling and studies will be addressed in the next stage of study.
- No geotechnical drilling, test work or analysis has been conducted at the Project site. Technical and cost risk exist for determining the mine and heap leaching design parameters.

- The Project will require various state and federal authorizations, licenses and permits for Project construction, operation, closure, and post-closure. Comprehensive environmental and socioeconomic baseline studies will be required. No environmental baseline studies have been conducted. The long-term seepage and water management requirements have not been established, and these issues can impact closure costs.
- At this time, there are no known factors to preclude a successful permitting effort; however, the length and effort of the permitting process can be difficult to predict due to the multiple agencies that will be involved, including both state and federal agencies.
- A more detailed look at mining plans with upgraded resource estimates in the future may allow for advancement of higher-grade material early in the mine life.
- Metallurgical testing is preliminary in nature, as such, estimates for gold recovery from heap leaching and cyanide consumption may present risks. Additional investigation, including column testing on representative samples is required to fully assess the gold recovery and cyanide consumption estimates and timing included in the Report.

1.19 RECOMMENDATIONS

Further study is warranted based on the results of the preliminary economic analysis (PEA), technical information and associated risks with the Project. A pre-feasibility study (PFS) is recommended as the next level of investigation for the Project. For a complete summary of interpretations, conclusions, and recommendations to advance the Project, see Sections 25 and 26.

2.0 INTRODUCTION

2.1 PURPOSE OF THE TECHNICAL REPORT

GMV Minerals Inc. ("GMV") engaged the services of Samuel Engineering, Inc. ("SE"), in conjunction with Mine Development Associates ("MDA"), a division of RESPEC, and Tierra Group International Ltd. ("Tierra Group"), Golder Associates Inc. ("Golder"), Tetra Tech Inc. ("Tetra Tech"), and DRW Geological Consultants Ltd. ("DRW") to prepare a Technical Report as a Preliminary Economic Assessment (PEA) on its Mexican Hat property located in Arizona, USA. This Technical Report is based on the results of the PEA, and updated estimate of mineral resources developed since the last Technical Report prepared by M3 Engineering & Technology Corp, dated December 17, 2018. This report is prepared in accordance with the Canadian National Instrument 43-101 ("NI 43-101") Standards of Disclosure for Mineral Projects, including Companion Policy 43-101CP and Form 43-101F1.

GMV is a junior gold development company listed on the TSX Venture Exchange (TSXV), where it trades under the symbol GMV, and on the OTCQB under symbol GMVMF. GMV has 100% interest in the Mexican Hat Project (the "Project"), located about 115 km (72 miles) southeast of Tucson, Arizona, and is focused on its development.

This Technical Report addresses all technical and cost aspects of the Project for mining, processing, environmental and infrastructure design, capital and operating costs, and economic analysis. Gold-bearing resources considered in this report would be mined by open pit, crushed, and loaded onto heap leaching pads for recovery of contained gold by conventional processing methods. Production data in the Technical Report are stated in metric units.

As required in NI 43-101, the effective date of this Technical Report is October 20, 2020 and the issue date is November 19, 2020.

2.2 SOURCES OF INFORMATION

This Technical Report is prepared with contributions from GMV and the Qualified Persons (QP) listed in Table 2-1. The authors of this report have relied on historic, and recent information generated for the Project. Table 2-1 summarizes the section responsibilities of the QPs contributing to the Technical Report. As defined by NI 43-101, all QPs and their respective companies listed are independent of GMV.

Table 2-1
Qualified Persons Section Responsibilities

Qualified Person	**Company**	**Section Responsibility**
Dr. Dave Webb, PhD., P.Eng., P.Geo.	DRW Geological Consultants Ltd.	Sections: 4, 5, 6, 7, 8, 9, 10, 11, 15, 19, 23, 24 and corresponding sections of 1, 25 and 26
James Barr, P.Geo	Tetra Tech, Inc.	Sections: 12 and 14
Mr. Thomas L. Dyer, P.E.	Mine Development Associates a division of RESPEC	Sections: 16, 21.1.7, 21.2.1 and corresponding sections of 1, 25 and 26
Mr. Francisco J. Barrios, P.E.	Tierra Group International Ltd.	Sections: 17.5, 21.1.9, 21.1.10, 26.3 and corresponding sections of 1, 25 and 26
Ms. Dawn Garcia, CPG, PG	Golder Associates Inc.	Section: 20 and corresponding sections of 1, 25 and 26

Table 2-1 Qualified Persons Section Responsibilities		
Qualified Person	**Company**	**Section Responsibility**
Mr. Alva Kuestermeyer	Samuel Engineering, Inc.	Sections: 2, 3, 13, 17 (except 17.5), 18, 21.2 (except 21.2.1) and corresponding sections of 1, 25 and 26
Mr. Steven Pozder, P.E.	Samuel Engineering, Inc.	Sections: 21.1 (except 21.1.7, 21.1.9, 21.1.10 & 21.1.13), 22 and corresponding sections of 1, 25 and 26

2.3 PERSONAL INSPECTION OF THE MEXICAN HAT PROPERTY

A site visit to the Mexican Hat Property was conducted by Alva Kuestermeyer (SE), Tom Dyer (MDA), Francisco Barrios (Tierra Group) and Dawn Garcia (Golder) on July 13, 2020. The purpose of the visit was to gain a better understanding of the site's physical characteristics, environmental condition, equipment placement and layout options, and area logistics. Golder is contracted with GMV for Technical Report contributions regarding environmental, permitting, social aspects and closure cost.

3.0 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 GMV'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, because 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 has executed agreements to the property described in "Section 4: Property Description and Location"; however, this is not a legal opinion and as such, the authors of this Technical Report have relied upon GMV's due diligence. The location and nature of the lode claims, and exploration permits, and their validity are based on discussions with GMV's legal counsel, and documents provided by GMV and the state of Arizona. The QPs rely on these experts.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION

The Mexican Hat 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.

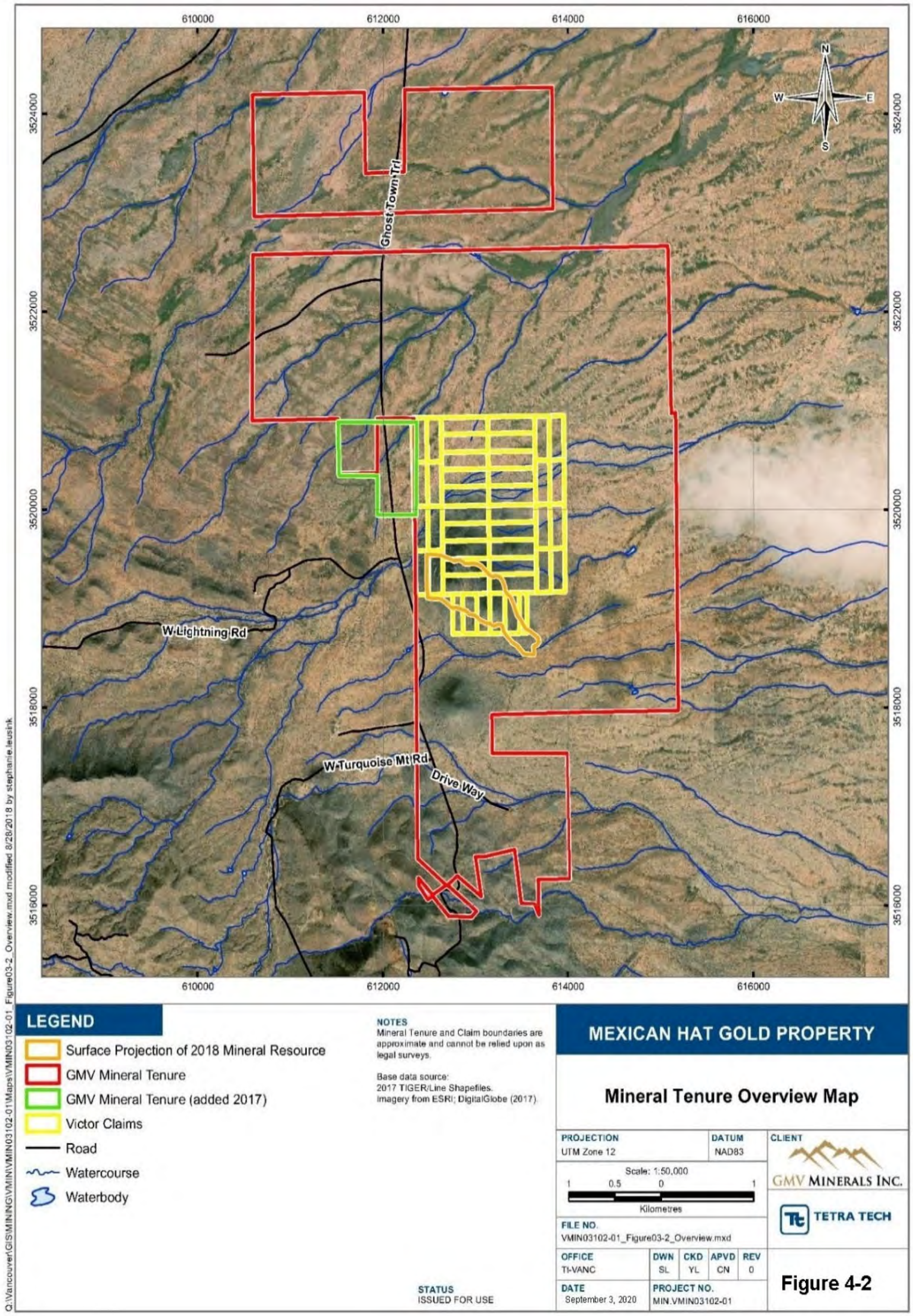


Figure 4-2: GMV Mineral Tenure, Overview Map

4.3 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

Table 4-1 Claims Leased to GMV			
No.	**Name of Claim**	**Fee #**	**BLM Serial No.**
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
	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	

4.3.1 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 in Table 4-2.

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 over staked fractions.

GMV also controls ten (10) Arizona Exploration Permits totaling 1,836.11 hectares (4,537.12 acres) as part of the Property as summarized below in Table 4-3.

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

Table 4-3 Arizona Exploration Permits held by GMV					
Permit No.	**Legal Description**	**Acres**	**Effective Date**	**Final Term. Date**	**Last Renewed Through Date**
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.

4.4 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 (News Release; GMV Minerals Inc., dated May 30, 2014).

4.5 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.

4.6 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 landowners to obtain and complete the necessary exploration and work programs on the Property.

4.7 ROYALTIES AND LIENS

The Mexican Hat Property is subject to a 3% net smelter returns royalty (NSR Royalty) in favor 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, GMV will be required to make advance royalty payments to Hernandez in the amount of US$4,500, payable quarterly. In addition, GMV 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.

5.0 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 have 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 below summarizes the average climate data for Pearce, Arizona (March 1950-December 2005.

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 may 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 northeasterly 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 GMV 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.

The Project site plan is shown in Figure 5-1.

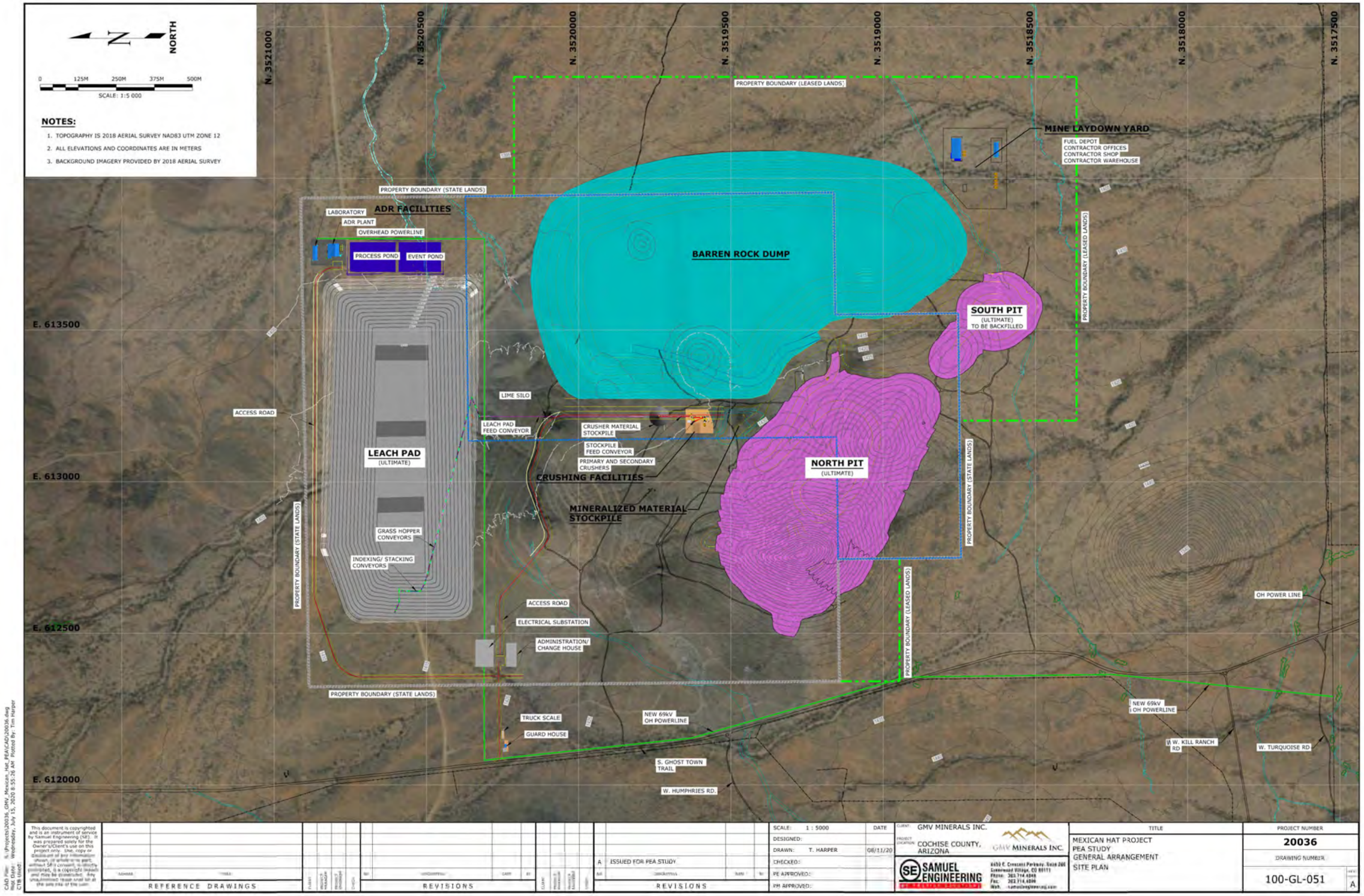


Figure 5-1: Mexican Hat Project Site Plan

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

GMV 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.

6.0 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 Zone 7 Fault zone 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 Zone 7 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 Zone 7 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 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 except for 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 most 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 western portion of Zone 7, 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.

Table 6-1 Twinned Drillholes (after McLeod, 2011)				
Percussion Hole	**Twin Core Hole**	**Twin Percussion**	**Twin Core Hole**	**Remarks**
89-4	13			To test the correlation in a high-grade section of Zone 7. 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 materials with feed grades between 1.097 and 33.394 g/t gold. Both cyanide 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 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).

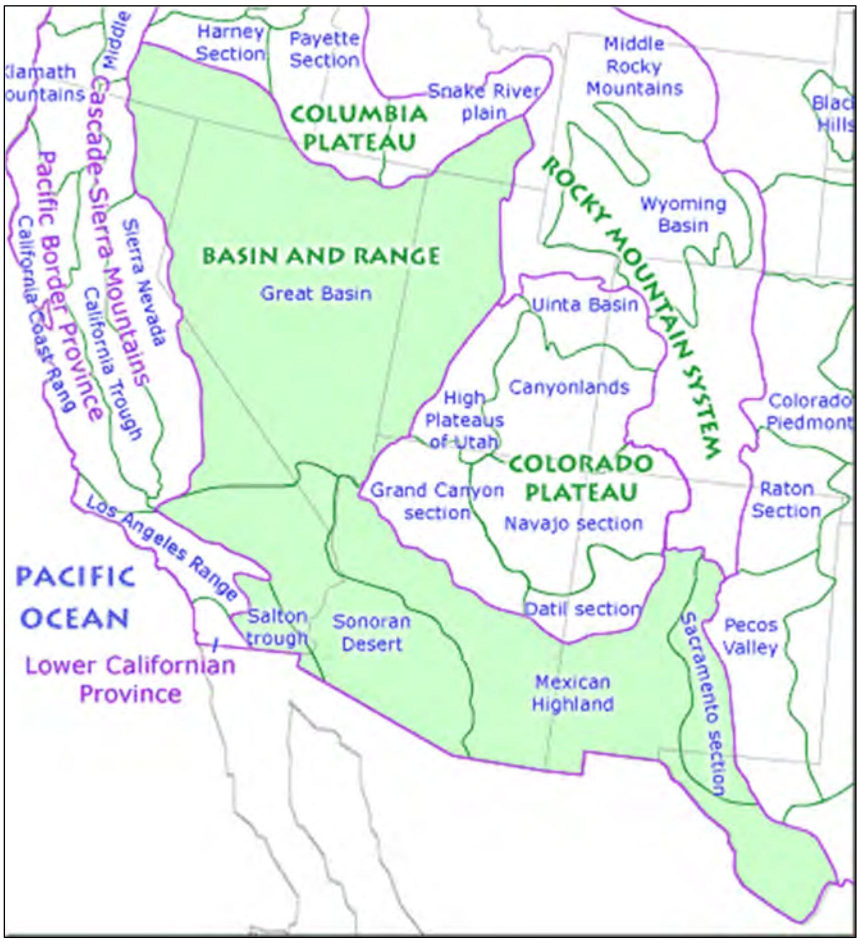
6.4.1 Geophysics

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

7.0 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. The Map Legend is on the on Following Page

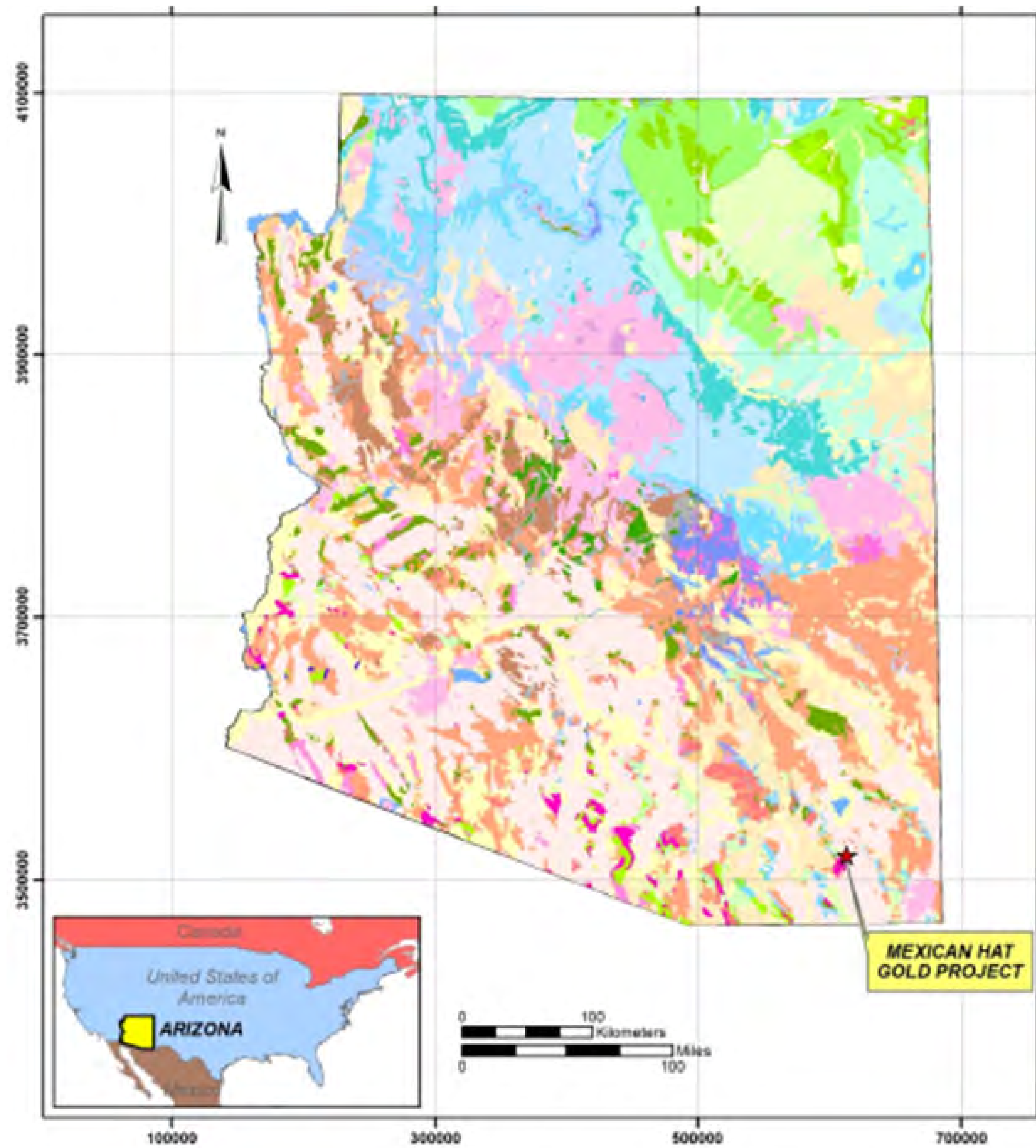
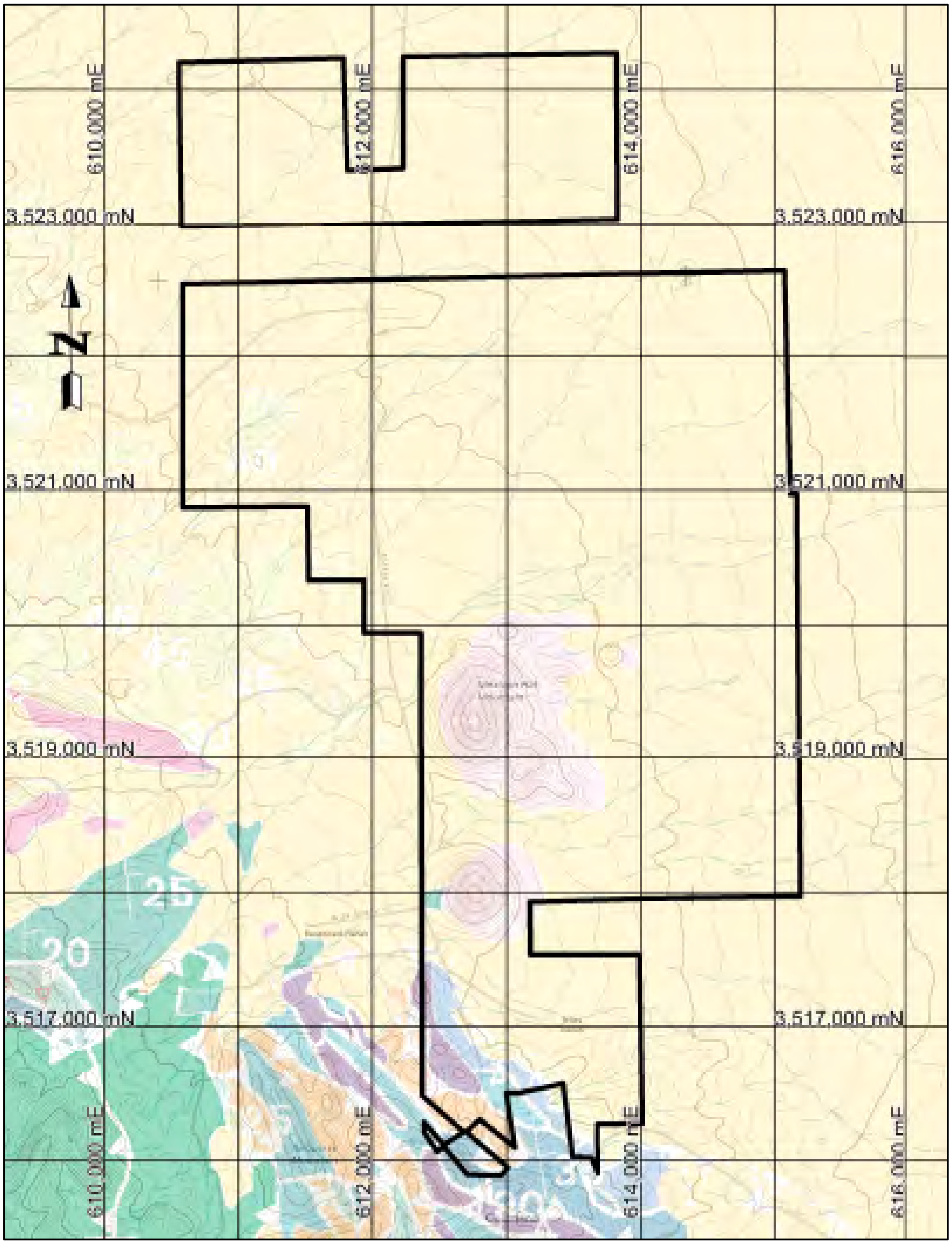


Figure 7-2: General Geology of Arizona showing Location of Mexican Hat Project, after Game (2013)

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)
- J| - 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)

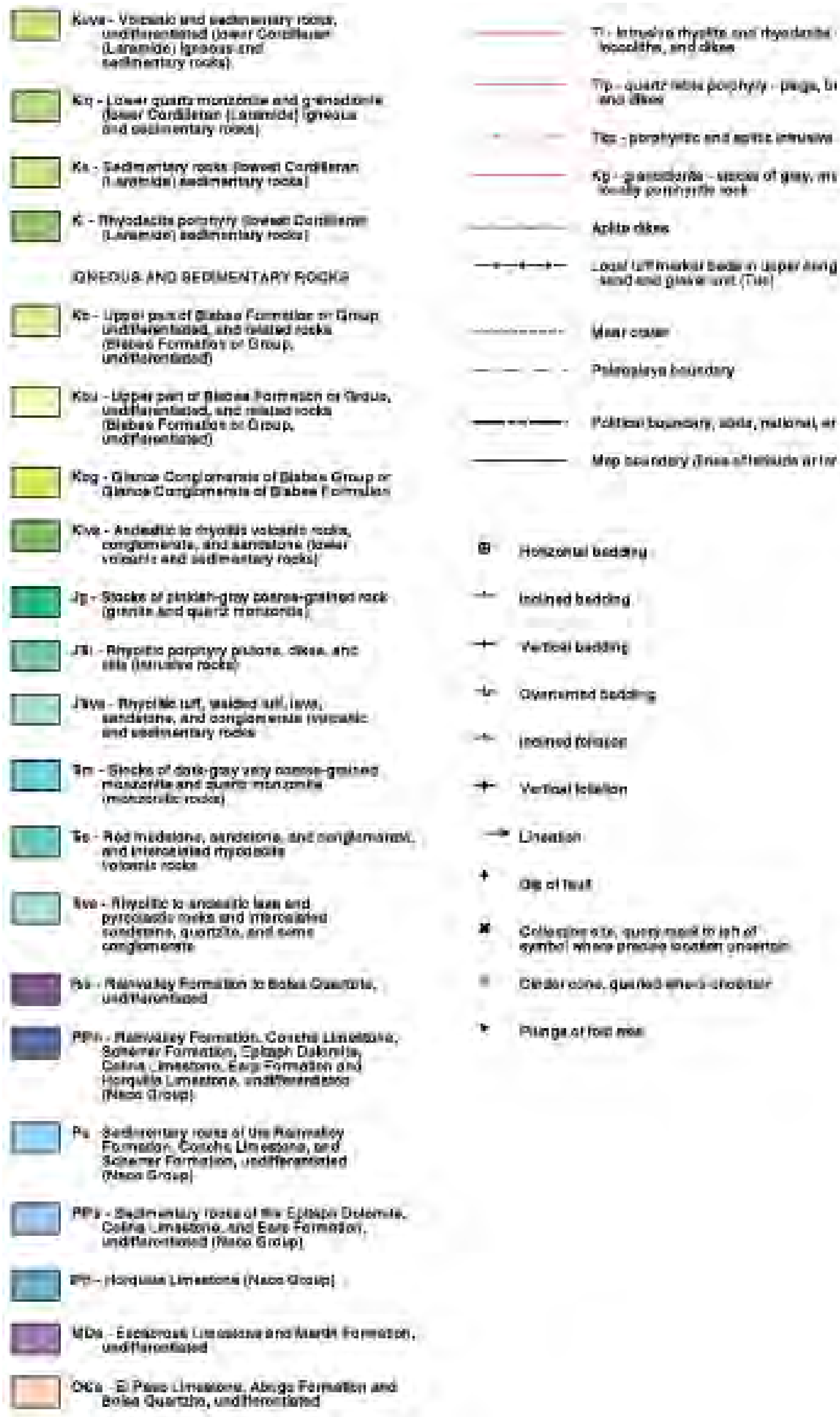
Calcareous or limey rocks exposed on the west-side of Pearce Road (Ghost town Trail) appear to be part of the Bisbee Group which underlies much of the nearby Turquois Mountains (Figure 7-3).



Legend following below

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

LIST OF MAP UNITS
SURFICIAL DEPOSITS
CORDILLERAN (LARAMIDE) IGNEOUS AND SEDIMENTARY ROCKS


IGNEOUS AND SEDIMENTARY ROCKS
Kbg - Glance Conglomerate of Bisbee Group or Glance Conglomerate of Bisbee Formation
Ps - Sedimentary rocks of the Rainvalley Formation, Concha Limestone, and Scherrer Formation, undifferentiated (Naco Group)
Aplite dikes
Paleoplaya boundary
Horizontal bedding
Inclined bedding
Vertical bedding
Overturned bedding
Inclined foliation
Vertical foliation
Lineation
Dip of fault
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).

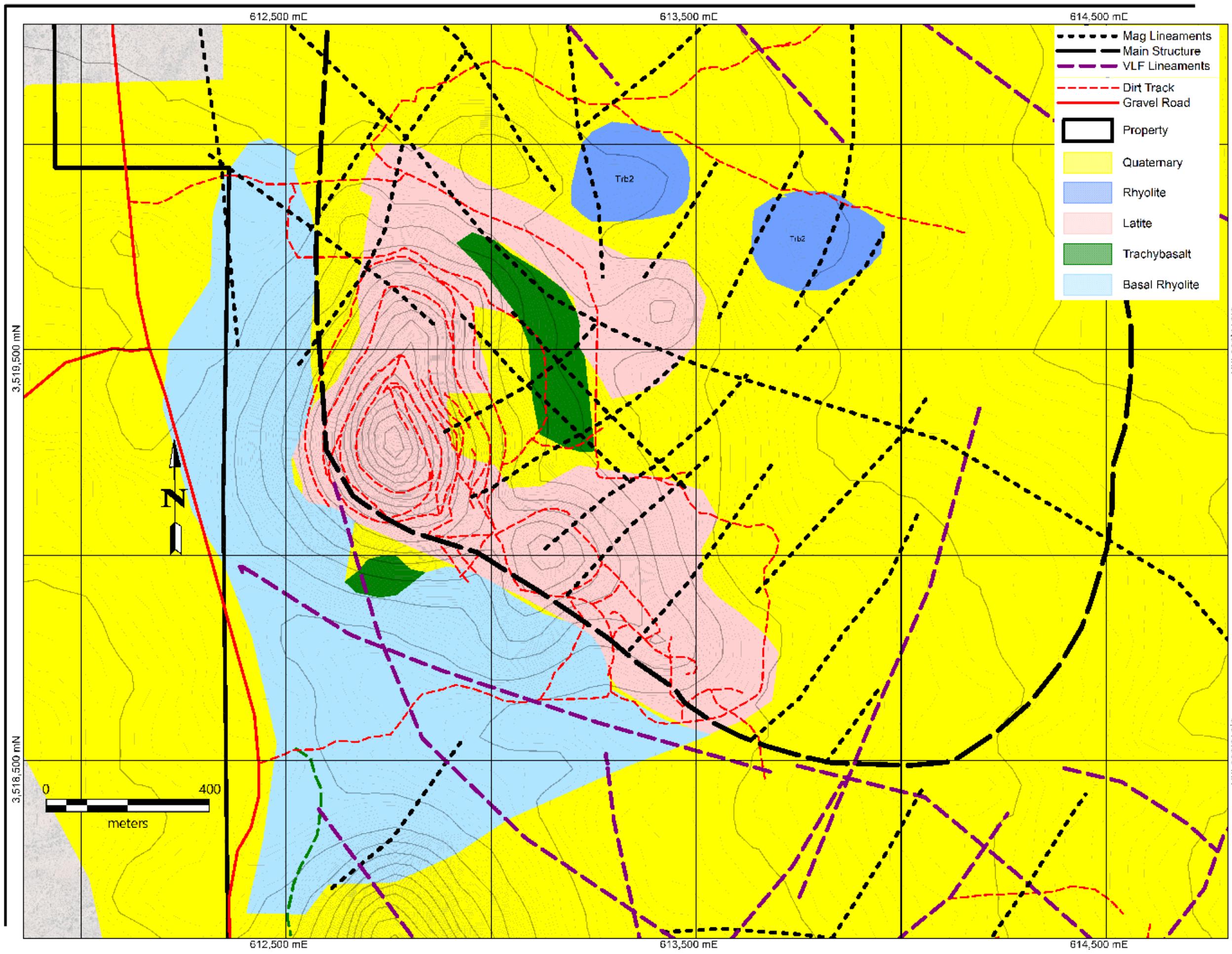


Figure 7-4: Simplified Local Property Geology, from Webb, 2019, UTM NAD83 Z.12

7.2.1 Mineralization

Gold 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. 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 were 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 accompanied by strong hematization. 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.

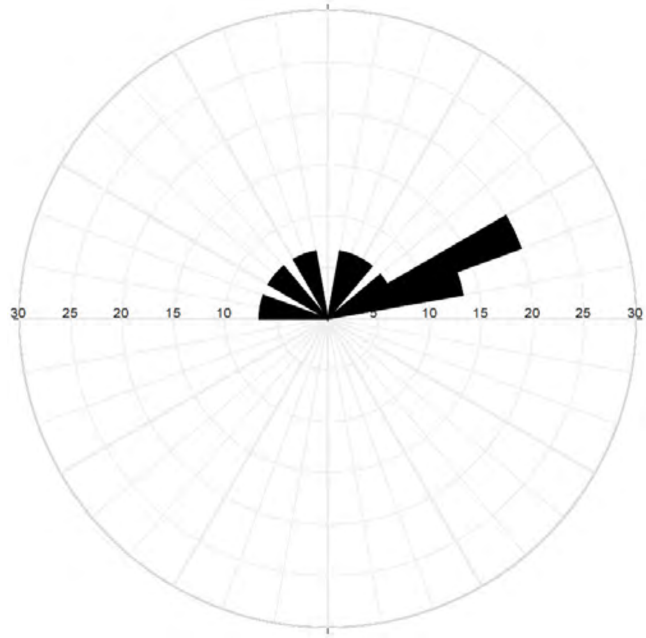


Figure 7-5: Rose Diagram of Strikes of all Faults Measured in Trenching Program (Webb and Malahoff, 2015)

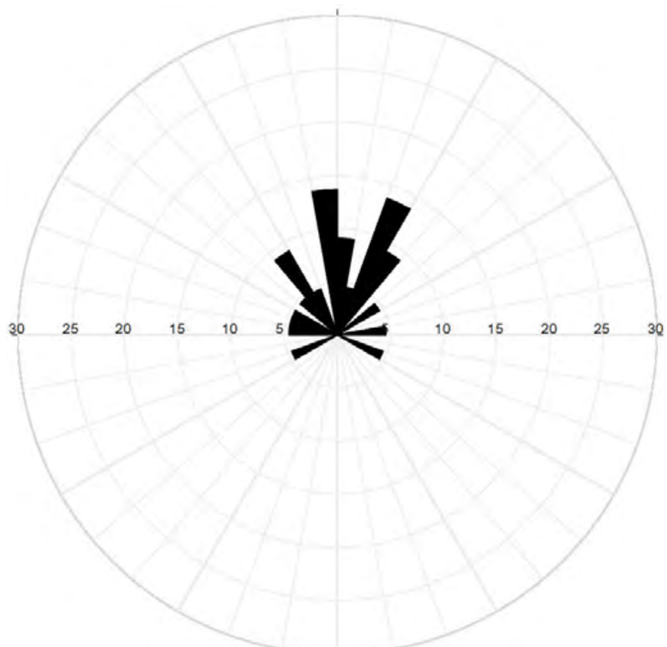


Figure 7-6: Rose Diagram of all Fractures with no Apparent Movement Measured in Trenching Program (Webb and Malahoff, 2015)

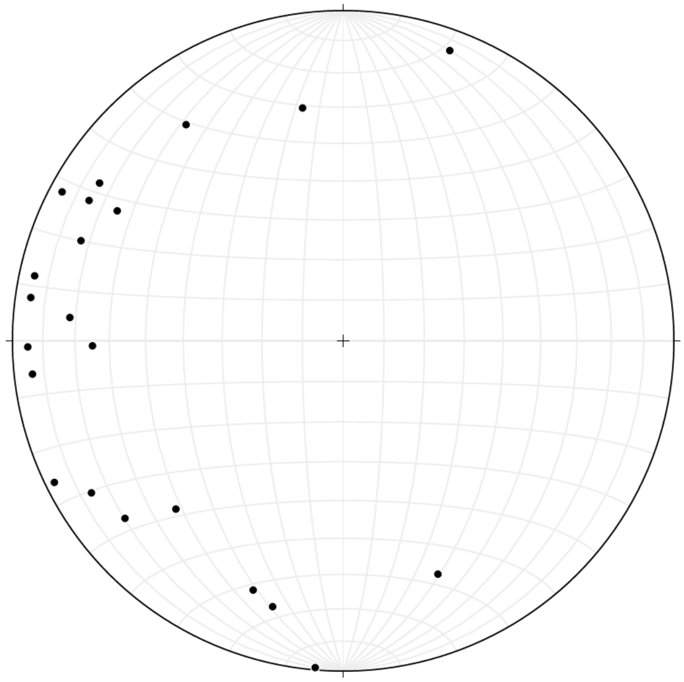


Figure 7-7: Stereonet Plot of Poles to all Fractures Measured in Trenches (Webb and Malahoff, 2015)

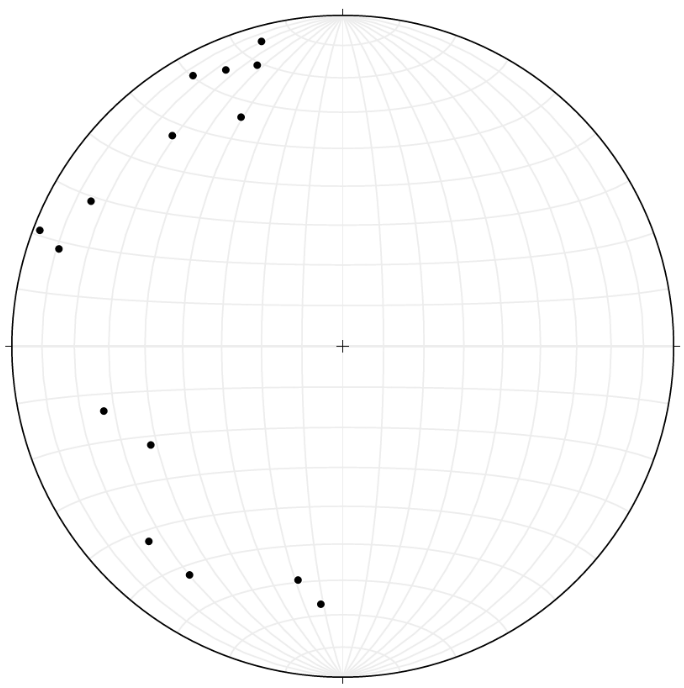


Figure 7-8: Stereonet Plot of Poles to all Faults Measured in Trenches (Webb and Malahoff, 2015)

Previous correlations and names of the faults on the property are not consistent. The following is 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 Zone 7, whereas Game (2013) shows a different location for Zone 7 Fault, and no structure corresponding to this geophysical lineament. In this area, Zone 7 strikes 296o, dips steeply to the northeast and joins geophysical lineaments correlating to Zone 7, 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 Zone 7 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 Zone 7 Fault.

The western portion of Zone 7 is exposed on the western flank of Mexican Hat Hill, striking northerly and dipping moderately to shallowly to the east and it apparently crosscuts the SMAG Faults.

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

8.0 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 in 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

9.0 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 like 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.

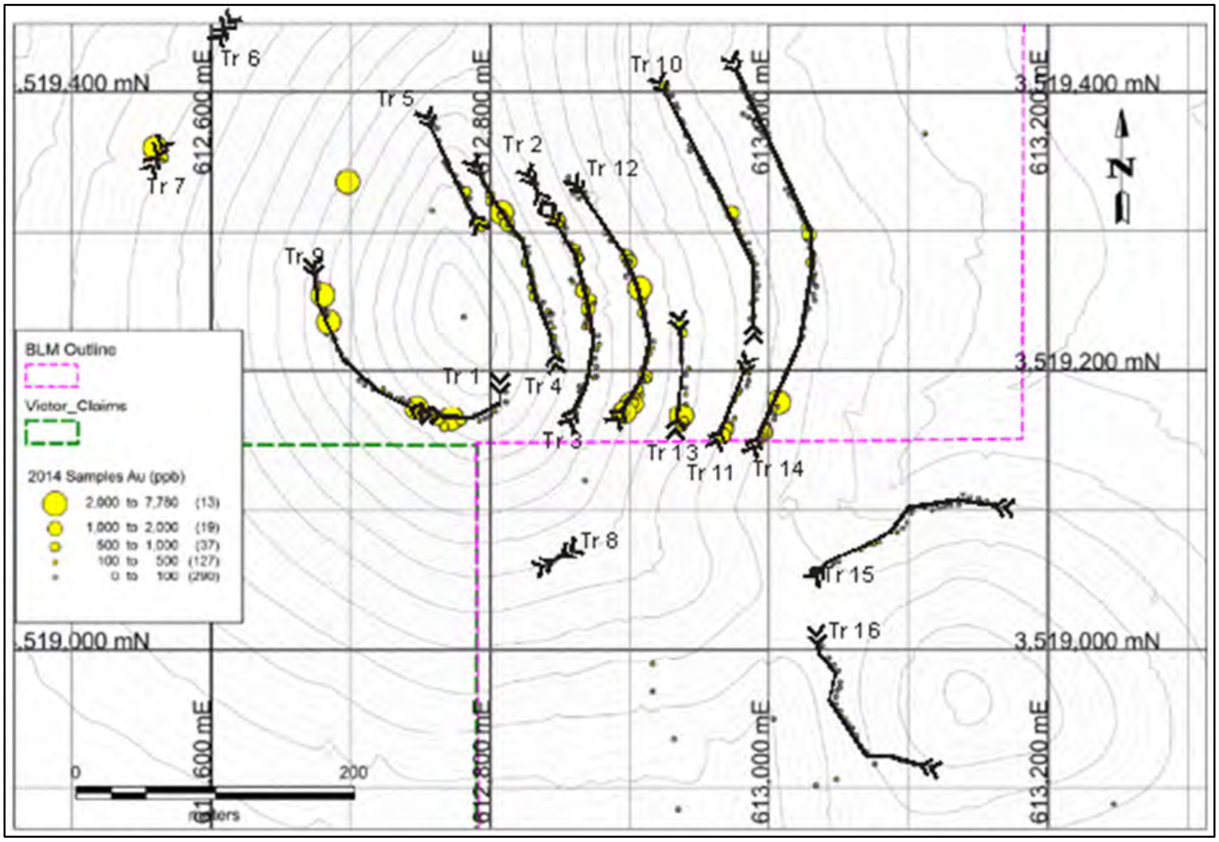


Figure 9-1: Location of Chip Sampling Program with Gold Values from 2014 Trench Program

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.

Historical Core Resampling

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 Placer Dome holes are within 30 meters and parallel to the Auracle drillhole and yields similar but not identical assays. Table 9-1 summarizes the proximity of other drillholes.

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

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.

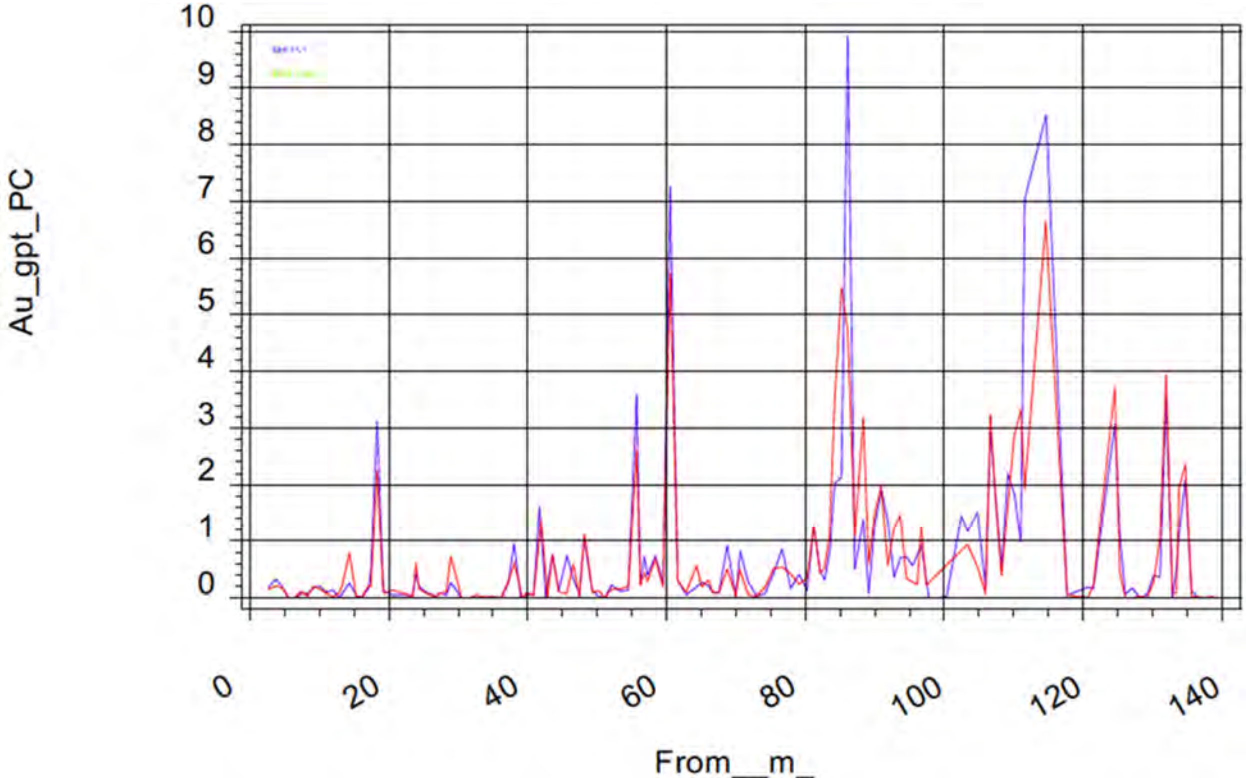


Figure 9-2: Line Plot of Gold Values in MH 11-1 (blue) and Re-assay Data as BTM 11-01 (red)

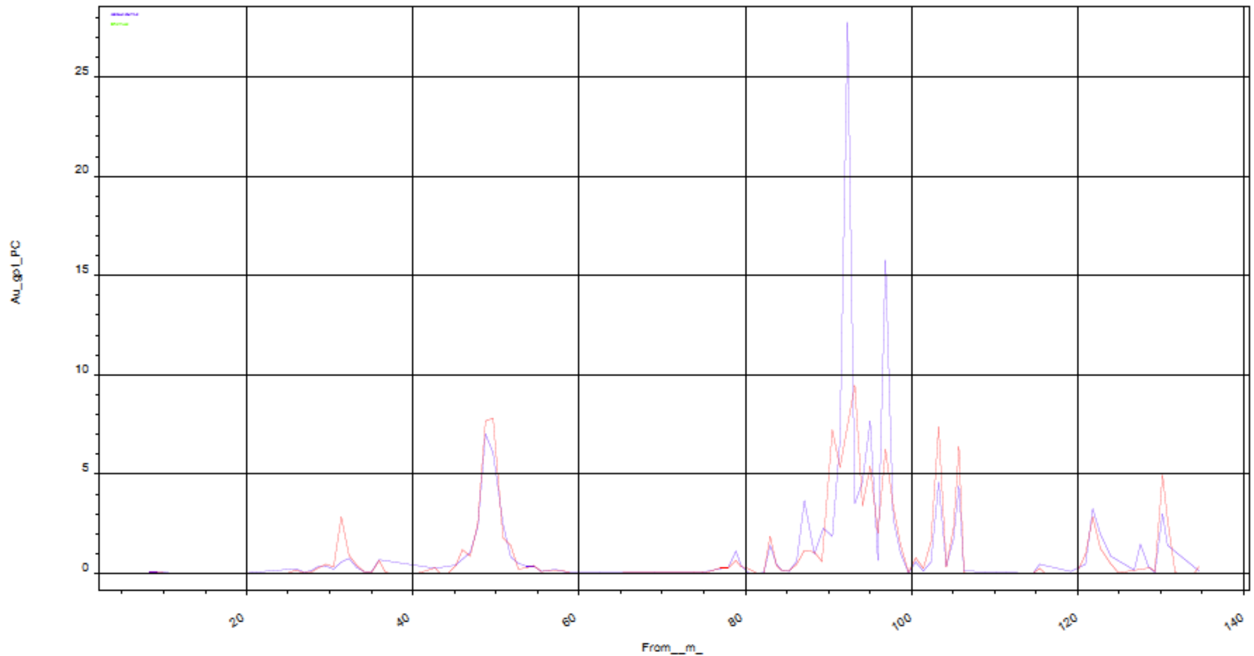


Figure 9-3: Line Plot of Gold Values in MH 11-2 (blue) and Re-assay Data as BTM 11-02 (red)

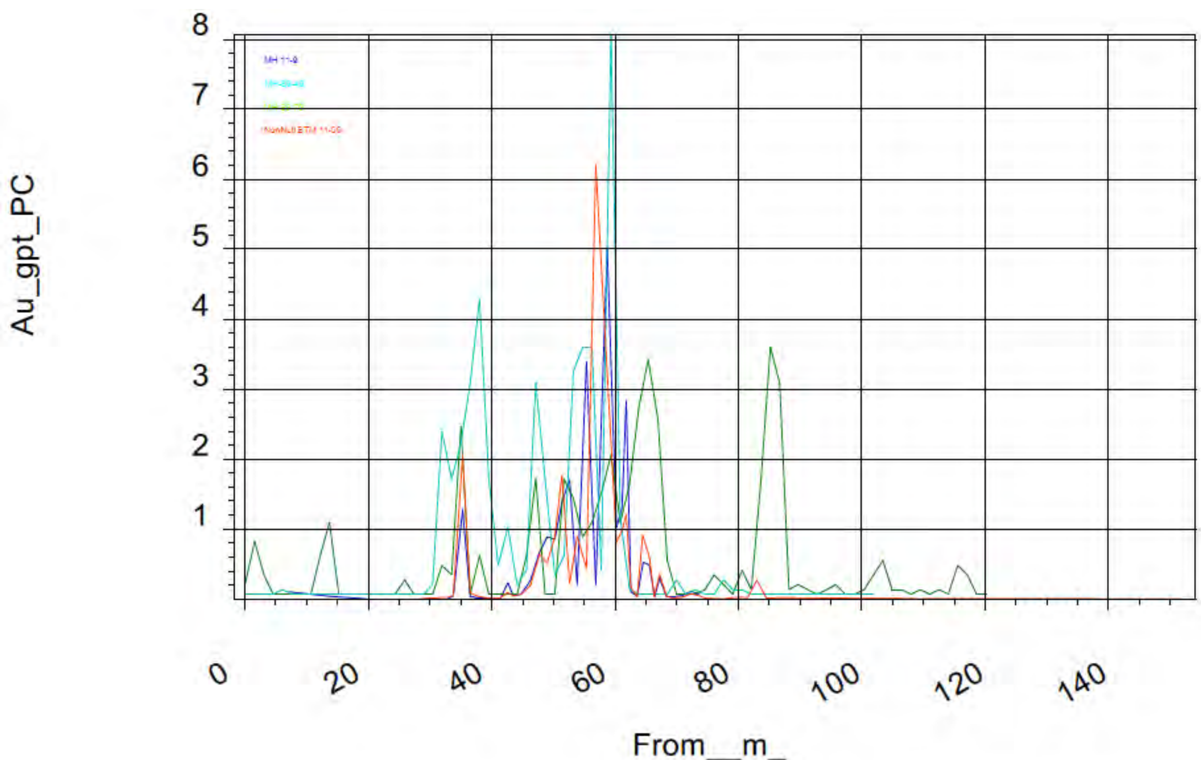


Figure 9-4: Line Plot of Gold Values in MH 11-9 (blue (dark), a Twin of MH 89-79 (light blue) and Nearby MH 89-41 (green) and Re-assay Data as BTM-09 (red)

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 9-1 and 9-2).

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

Photograph 9-2: Near continuous chip sample were collected in road cuts (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 (true 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 were 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 several Placer Dome drillholes, and DRW re-sampled several 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

Three phases of soil sampling were conducted in 2016, 2017 and 2019 covering the area shown on Figure 9-5.

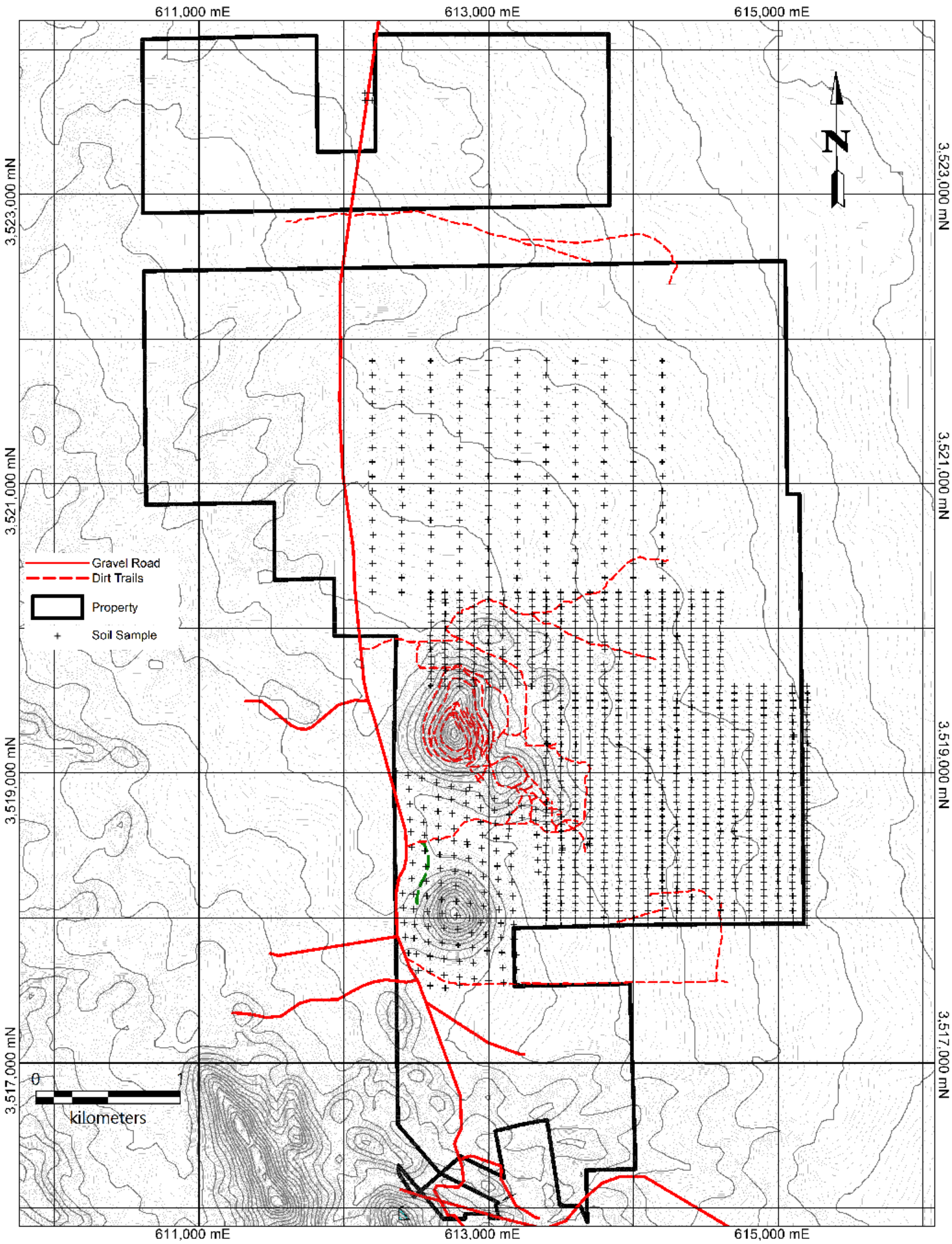


Figure 9-5: Location of 2016, 2017 and 2019 Soil Samples

Pearson correlation coefficients on the 1,253 soil samples are shown below on in Table 9-8 to Table 9-11.

Table 9-8
Pearson Correlation Coefficients on N=1252 Soil Samples 1 of 3

Field	Mo	Cu	Pb	Zn	Ag	Ni	Co	Mn	Fe	As	U	Au
Mo	1.0000	0.3155	0.4738	0.3294	0.3122	-0.0894	-0.0519	0.2992	0.1723	0.1112	0.3174	0.2974
Cu	0.3155	1.0000	0.6261	0.7197	0.4920	0.3618	0.4034	0.4716	0.4922	0.3692	0.2285	0.1087
Pb	0.4738	0.6261	1.0000	0.6505	0.4992	-0.0151	0.0882	0.5321	0.3261	0.0056	0.4781	0.0198
Zn	0.3294	0.7197	0.6505	1.0000	0.5631	0.3576	0.4296	0.7390	0.5798	0.1987	0.3717	0.0814
Ag	0.3122	0.4920	0.4992	0.5631	1.0000	0.2726	0.2155	0.4012	0.3417	0.3253	0.2997	0.4001
Ni	-0.0894	0.3618	-0.0151	0.3576	0.2726	1.0000	0.8494	0.1193	0.6079	0.5535	-0.2631	0.2345
Co	-0.0519	0.4034	0.0882	0.4296	0.2155	0.8494	1.0000	0.3505	0.6495	0.3837	-0.1677	0.1403
Mn	0.2992	0.4716	0.5321	0.7390	0.4012	0.1193	0.3505	1.0000	0.2574	0.0095	0.3125	-0.0017
Fe	0.1723	0.4922	0.3261	0.5798	0.3417	0.6079	0.6495	0.2574	1.0000	0.2780	0.0983	0.1619
As	0.1112	0.3692	0.0056	0.1987	0.3253	0.5535	0.3837	0.0095	0.2780	1.0000	-0.2870	0.3824
U	0.3174	0.2285	0.4781	0.3717	0.2997	-0.2631	-0.1677	0.3125	0.0983	-0.2870	1.0000	-0.0107
Au	0.2974	0.1087	0.0198	0.0814	0.4001	0.2345	0.1403	-0.0017	0.1619	0.3824	-0.0107	1.0000
Th	0.3148	0.0512	0.3752	0.1610	0.1615	-0.4380	-0.3766	0.1355	-0.0219	-0.3912	0.8769	-0.0194
Sr	-0.2505	0.1784	-0.1557	0.2158	-0.0098	0.5220	0.4892	-0.0112	0.2686	0.3391	-0.2059	0.0429
Cd	0.4178	0.7178	0.7342	0.7531	0.3916	0.1627	0.2851	0.6732	0.3377	0.1598	0.2316	0.0564
Sb	0.2450	0.2534	0.0494	0.2151	0.1380	0.2829	0.2092	0.1813	0.1804	0.6152	-0.2255	0.2945
Bi	0.4635	0.5309	0.8458	0.5958	0.4289	-0.2507	-0.1467	0.4428	0.2215	-0.2214	0.6423	-0.0560
V	-0.0711	0.3011	-0.0613	0.2478	0.1154	0.7574	0.7843	-0.0338	0.8084	0.4294	-0.1884	0.1741
Ca	-0.2245	0.0581	-0.1952	0.0847	-0.0660	0.2997	0.1914	-0.1019	0.0110	0.2951	-0.1771	0.0357
P	0.1078	0.5869	0.2371	0.7421	0.2451	0.5227	0.5728	0.4782	0.4804	0.3610	0.0531	0.0676
La	0.2005	0.3758	0.5595	0.5533	0.4231	0.0151	0.1491	0.6348	0.3133	0.1027	0.3237	0.0243
Cr	-0.0491	0.3015	-0.0463	0.2499	0.2332	0.9281	0.7740	0.0007	0.6659	0.4784	-0.2686	0.2463
Mg	0.0177	0.4916	0.1074	0.6080	0.3489	0.7901	0.6535	0.3210	0.4932	0.5851	-0.1397	0.2025
Ba	-0.1145	0.2063	0.1195	0.3584	0.2026	0.2701	0.2754	0.3138	0.1877	0.1712	0.0457	-0.0095
Ti	-0.1145	-0.0425	-0.3043	-0.0720	-0.0336	0.5282	0.4752	-0.1088	0.2412	0.3047	-0.2292	0.1281
B	-0.0234	-0.0433	-0.1307	-0.0975	0.0767	0.1120	0.0519	-0.1074	-0.0114	0.0430	-0.1413	0.1088

Table 9-8
Pearson Correlation Coefficients on N=1252 Soil Samples 1 of 3

Field	Mo	Cu	Pb	Zn	Ag	Ni	Co	Mn	Fe	As	U	Au
Al	0.0277	0.5754	0.2874	0.5869	0.4568	0.6937	0.5737	0.2329	0.6810	0.5909	-0.1601	0.1952
Na	-0.0633	0.1483	-0.0883	0.0664	0.0400	0.2691	0.3310	-0.0487	0.3061	0.1087	-0.1351	0.0137
K	0.2230	0.5147	0.5871	0.6859	0.5206	0.2768	0.1864	0.4264	0.4417	0.2637	0.1839	0.1218
W	0.3612	0.3113	0.1193	0.2528	0.1199	0.1542	0.1461	0.1731	0.1766	0.3071	0.0113	0.1735
Sc	-0.0261	0.4438	0.1635	0.4205	0.4522	0.8154	0.7636	0.1896	0.6833	0.5592	-0.1981	0.2333
Tl	0.1140	0.1977	0.4339	0.1199	0.4132	0.0182	-0.0469	0.0731	0.1645	0.2601	0.0999	0.1170
S	0.0982	-0.0751	0.0558	-0.0045	0.1438	-0.1000	-0.0989	0.0791	-0.1033	-0.1146	0.1148	-0.0109
Hg	0.0520	0.4338	0.3547	0.2365	0.3470	0.2510	0.3306	0.1875	0.2861	0.3605	-0.0511	0.0960
Se	0.0262	0.0958	0.1337	0.1024	0.1199	0.0552	0.0152	0.1158	0.0273	0.0926	-0.0779	0.0250
Te	0.4824	0.5908	0.7324	0.6262	0.3780	-0.1724	-0.1043	0.4541	0.2549	-0.0410	0.4411	0.0004
Ga	0.1193	0.6223	0.3422	0.7010	0.5790	0.7045	0.5876	0.3480	0.6899	0.6327	-0.0378	0.2484

Table 9-9
Pearson Correlation Coefficients of N=1,252 Soil Samples 2 of 3

Field	Th	Sr	Cd	Sb	Bi	V	Ca	P	La	Cr	Mg	Ba
Mo	0.3148	-0.2505	0.4178	0.2450	0.4635	-0.0711	-0.2245	0.1078	0.2005	-0.0491	0.0177	-0.1145
Cu	0.0512	0.1784	0.7178	0.2534	0.5309	0.3011	0.0581	0.5869	0.3758	0.3015	0.4916	0.2063
Pb	0.3752	-0.1557	0.7342	0.0494	0.8458	-0.0613	-0.1952	0.2371	0.5595	-0.0463	0.1074	0.1195
Zn	0.1610	0.2158	0.7531	0.2151	0.5958	0.2478	0.0847	0.7421	0.5533	0.2499	0.6080	0.3584
Ag	0.1615	-0.0098	0.3916	0.1380	0.4289	0.1154	-0.0660	0.2451	0.4231	0.2332	0.3489	0.2026
Ni	-0.4380	0.5220	0.1627	0.2829	-0.2507	0.7574	0.2997	0.5227	0.0151	0.9281	0.7901	0.2701
Co	-0.3766	0.4892	0.2851	0.2092	-0.1467	0.7843	0.1914	0.5728	0.1491	0.7740	0.6535	0.2754
Mn	0.1355	-0.0112	0.6732	0.1813	0.4428	-0.0338	-0.1019	0.4782	0.6348	0.0007	0.3210	0.3138
Fe	-0.0219	0.2686	0.3377	0.1804	0.2215	0.8084	0.0110	0.4804	0.3133	0.6659	0.4932	0.1877
As	-0.3912	0.3391	0.1598	0.6152	-0.2214	0.4294	0.2951	0.3610	0.1027	0.4784	0.5851	0.1712
U	0.8769	-0.2059	0.2316	-0.2255	0.6423	-0.1884	-0.1771	0.0531	0.3237	-0.2686	-0.1397	0.0457
Au	-0.0194	0.0429	0.0564	0.2945	-0.0560	0.1741	0.0357	0.0676	0.0243	0.2463	0.2025	-0.0095

Table 9-9
Pearson Correlation Coefficients of N=1,252 Soil Samples 2 of 3

Field	Th	Sr	Cd	Sb	Bi	V	Ca	P	La	Cr	Mg	Ba
Th	1.0000	-0.3837	0.0971	-0.2638	0.5762	-0.3426	-0.3151	-0.1611	0.1998	-0.3965	-0.3283	-0.1275
Sr	-0.3837	1.0000	0.0719	0.1128	-0.2583	0.4910	0.8384	0.5809	0.0136	0.4103	0.5634	0.6105
Cd	0.0971	0.0719	1.0000	0.2954	0.5961	0.0526	-0.0013	0.5843	0.4460	0.0943	0.3891	0.1754
Sb	-0.2638	0.1128	0.2954	1.0000	-0.1139	0.2001	0.1299	0.3016	0.0963	0.2781	0.3465	-0.0106
Bi	0.5762	-0.2583	0.5961	-0.1139	1.0000	-0.2174	-0.2601	0.1522	0.4625	-0.2493	-0.0751	0.0111
V	-0.3426	0.4910	0.0526	0.2001	-0.2174	1.0000	0.2150	0.4251	0.0035	0.8028	0.5264	0.1777
Ca	-0.3151	0.8384	-0.0013	0.1299	-0.2601	0.2150	1.0000	0.4857	-0.0666	0.1919	0.4412	0.5647
P	-0.1611	0.5809	0.5843	0.3016	0.1522	0.4251	0.4857	1.0000	0.2819	0.3810	0.7877	0.4375
La	0.1998	0.0136	0.4460	0.0963	0.4625	0.0035	-0.0666	0.2819	1.0000	-0.0743	0.2026	0.3333
Cr	-0.3965	0.4103	0.0943	0.2781	-0.2493	0.8028	0.1919	0.3810	-0.0743	1.0000	0.6644	0.1148
Mg	-0.3283	0.5634	0.3891	0.3465	-0.0751	0.5264	0.4412	0.7877	0.2026	0.6644	1.0000	0.3635
Ba	-0.1275	0.6105	0.1754	-0.0106	0.0111	0.1777	0.5647	0.4375	0.3333	0.1148	0.3635	1.0000
Ti	-0.3466	0.1753	-0.1473	0.1848	-0.4531	0.5267	0.0099	0.0735	-0.2568	0.5647	0.3299	-0.1194
B	-0.1861	0.1906	-0.0340	0.0053	-0.1810	0.0826	0.1671	0.0202	-0.0493	0.1067	0.1326	0.1654
Al	-0.2898	0.4893	0.4139	0.2983	0.0692	0.5834	0.3255	0.6180	0.3635	0.6215	0.7534	0.4288
Na	-0.1759	0.3701	-0.0599	-0.0373	-0.1307	0.4646	0.1868	0.1970	0.0171	0.2592	0.2269	0.2502
K	0.0948	0.1391	0.5329	0.1859	0.4302	0.1068	0.1062	0.4410	0.4710	0.1619	0.4182	0.4280
W	-0.0482	0.0729	0.2532	0.5714	0.0582	0.2269	0.0813	0.2969	0.0183	0.1475	0.2720	0.0200
Sc	-0.3348	0.3398	0.2425	0.1967	-0.0765	0.6957	0.0801	0.4339	0.2528	0.7728	0.6822	0.2482
Tl	0.0963	-0.1073	0.1007	0.0653	0.2838	-0.0288	-0.0794	-0.1953	0.4265	0.0021	-0.0969	0.1841
S	0.1235	-0.0648	-0.0064	-0.0684	0.0580	-0.1712	-0.0023	-0.0618	0.0398	-0.1026	-0.0595	0.1290
Hg	-0.1228	0.1104	0.3596	0.1936	0.1904	0.2316	-0.0232	0.1748	0.3710	0.2530	0.2243	0.0589
Se	-0.0669	-0.0370	0.1658	0.1136	0.0308	-0.0350	0.0185	0.1219	0.1804	0.0230	0.1357	0.0860
Te	0.3951	-0.1377	0.6868	0.1283	0.8412	-0.1587	-0.1195	0.3444	0.4047	-0.1635	0.1099	0.0401
Ga	-0.2116	0.4167	0.4710	0.3567	0.1454	0.5660	0.2539	0.6644	0.4139	0.6260	0.8167	0.3760

Table 9-10
Pearson Correlation Coefficients of n=1252 Soil Samples 3 of 3

Field	Ti	B	Al	Na	K	W	Sc	Tl	S	Hg	Se	Te
Mo	-0.1145	-0.0234	0.0277	-0.0633	0.2230	0.3612	-0.0261	0.1140	0.0982	0.0520	0.0262	0.4824
Cu	-0.0425	-0.0433	0.5754	0.1483	0.5147	0.3113	0.4438	0.1977	-0.0751	0.4338	0.0958	0.5908
Pb	-0.3043	-0.1307	0.2874	-0.0883	0.5871	0.1193	0.1635	0.4339	0.0558	0.3547	0.1337	0.7324
Zn	-0.0720	-0.0975	0.5869	0.0664	0.6859	0.2528	0.4205	0.1199	-0.0045	0.2365	0.1024	0.6262
Ag	-0.0336	0.0767	0.4568	0.0400	0.5206	0.1199	0.4522	0.4132	0.1438	0.3470	0.1199	0.3780
Ni	0.5282	0.1120	0.6937	0.2691	0.2768	0.1542	0.8154	0.0182	-0.1000	0.2510	0.0552	-0.1724
Co	0.4752	0.0519	0.5737	0.3310	0.1864	0.1461	0.7636	-0.0469	-0.0989	0.3306	0.0152	-0.1043
Mn	-0.1088	-0.1074	0.2329	-0.0487	0.4264	0.1731	0.1896	0.0731	0.0791	0.1875	0.1158	0.4541
Fe	0.2412	-0.0114	0.6810	0.3061	0.4417	0.1766	0.6833	0.1645	-0.1033	0.2861	0.0273	0.2549
As	0.3047	0.0430	0.5909	0.1087	0.2637	0.3071	0.5592	0.2601	-0.1146	0.3605	0.0926	-0.0410
U	-0.2292	-0.1413	-0.1601	-0.1351	0.1839	0.0113	-0.1981	0.0999	0.1148	-0.0511	-0.0779	0.4411
Au	0.1281	0.1088	0.1952	0.0137	0.1218	0.1735	0.2333	0.1170	-0.0109	0.0960	0.0250	0.0004
Th	-0.3466	-0.1861	-0.2898	-0.1759	0.0948	-0.0482	-0.3348	0.0963	0.1235	-0.1228	-0.0669	0.3951
Sr	0.1753	0.1906	0.4893	0.3701	0.1391	0.0729	0.3398	-0.1073	-0.0648	0.1104	-0.0370	-0.1377
Cd	-0.1473	-0.0340	0.4139	-0.0599	0.5329	0.2532	0.2425	0.1007	-0.0064	0.3596	0.1658	0.6868
Sb	0.1848	0.0053	0.2983	-0.0373	0.1859	0.5714	0.1967	0.0653	-0.0684	0.1936	0.1136	0.1283
Bi	-0.4531	-0.1810	0.0692	-0.1307	0.4302	0.0582	-0.0765	0.2838	0.0580	0.1904	0.0308	0.8412
V	0.5267	0.0826	0.5834	0.4646	0.1068	0.2269	0.6957	-0.0288	-0.1712	0.2316	-0.0350	-0.1587
Ca	0.0099	0.1671	0.3255	0.1868	0.1062	0.0813	0.0801	-0.0794	-0.0023	-0.0232	0.0185	-0.1195
P	0.0735	0.0202	0.6180	0.1970	0.4410	0.2969	0.4339	-0.1953	-0.0618	0.1748	0.1219	0.3444
La	-0.2568	-0.0493	0.3635	0.0171	0.4710	0.0183	0.2528	0.4265	0.0398	0.3710	0.1804	0.4047
Cr	0.5647	0.1067	0.6215	0.2592	0.1619	0.1475	0.7728	0.0021	-0.1026	0.2530	0.0230	-0.1635
Mg	0.3299	0.1326	0.7534	0.2269	0.4182	0.2720	0.6822	-0.0969	-0.0595	0.2243	0.1357	0.1099
Ba	-0.1194	0.1654	0.4288	0.2502	0.4280	0.0200	0.2482	0.1841	0.1290	0.0589	0.0860	0.0401
Ti	1.0000	0.0463	0.0878	0.1961	-0.2236	0.1601	0.3657	-0.2867	-0.1819	-0.0502	-0.1579	-0.4123
B	0.0463	1.0000	0.0689	0.1054	-0.0688	-0.0276	0.1380	-0.0512	0.2947	0.0136	0.0685	-0.1187
Al	0.0878	0.0689	1.0000	0.2886	0.6436	0.1638	0.8127	0.3656	-0.0717	0.4558	0.1778	0.2065
Na	0.1961	0.1054	0.2886	1.0000	0.0453	0.4504	0.2973	-0.0108	0.1486	0.1124	-0.0394	-0.1456

Table 9-10
Pearson Correlation Coefficients of n=1252 Soil Samples 3 of 3

Field	Ti	B	Al	Na	K	W	Sc	Tl	S	Hg	Se	Te
K	-0.2236	-0.0688	0.6436	0.0453	1.0000	0.1653	0.3992	0.5340	0.0768	0.2281	0.1832	0.4239
W	0.1601	-0.0276	0.1638	0.4504	0.1653	1.0000	0.0543	-0.0552	-0.0235	0.0484	0.1096	0.2880
Sc	0.3657	0.1380	0.8127	0.2973	0.3992	0.0543	1.0000	0.2701	-0.0716	0.4712	0.0873	-0.0535
Tl	-0.2867	-0.0512	0.3656	-0.0108	0.5340	-0.0552	0.2701	1.0000	0.0873	0.4368	0.1544	0.1659
S	-0.1819	0.2947	-0.0717	0.1486	0.0768	-0.0235	-0.0716	0.0873	1.0000	-0.0609	0.2792	-0.0028
Hg	-0.0502	0.0136	0.4558	0.1124	0.2281	0.0484	0.4712	0.4368	-0.0609	1.0000	0.1661	0.1791
Se	-0.1579	0.0685	0.1778	-0.0394	0.1832	0.1096	0.0873	0.1544	0.2792	0.1661	1.0000	0.1184
Te	-0.4123	-0.1187	0.2065	-0.1456	0.4239	0.2880	-0.0535	0.1659	-0.0028	0.1791	0.1184	1.0000
Ga	0.1706	0.0394	0.9511	0.2209	0.6505	0.2285	0.8115	0.3231	-0.0600	0.4227	0.1679	0.2735

Table 9-11
Basic Statistics for Soil Samples

Field	Count_n	Minimum	Maximum	Mean	Median	Range	Standard Deviation
Mo	1251	0.13	11.31	0.83	0.78	11.18	0.453
Cu	1251	9.69	189.36	30.97	27.88	179.67	14.780
Pb	1251	3.56	41.37	18.72	17.61	37.81	5.449
Zn	1251	15.60	112.10	44.51	42.00	96.50	16.739
Ag	1251	13.00	674.00	102.41	87.00	661.00	55.805
Ni	1251	3.20	53.20	9.06	7.40	50.00	4.425
Co	1251	2.90	22.40	7.54	6.80	19.50	2.478
Mn	1251	166.00	1400.00	524.68	478.00	1234.00	189.773
Fe	1251	1.03	4.40	2.04	1.96	3.37	0.441
As	1251	0.60	36.70	5.25	4.00	36.10	3.806
U	1251	0.40	6.10	1.40	1.20	5.70	0.642
Au	1251	0.10	2367.10	22.75	3.20	2367.00	122.744
Th	1251	0.70	74.80	12.61	11.00	74.10	8.055
Sr	1251	5.40	292.50	29.63	19.80	287.10	29.649
Cd	1251	0.04	0.70	0.21	0.18	0.66	0.095
Sb	1251	0.07	6.29	0.53	0.34	6.22	0.628
Bi	1251	0.06	2.67	0.84	0.70	2.61	0.429
V	1251	15.00	115.00	36.55	32.00	100.00	13.250
Ca	1251	0.04	16.10	0.73	0.26	16.06	1.655
P	1251	0.01	0.13	0.04	0.03	0.13	0.027
La	1251	6.60	60.10	28.43	27.80	53.50	4.917
Cr	1251	4.20	79.90	13.84	11.70	75.70	6.258
Mg	1251	0.09	2.19	0.34	0.27	2.10	0.226
Ba	1251	33.30	847.50	104.68	96.20	814.20	50.597
Ti	1251	0.00	0.12	0.03	0.03	0.12	0.016
B	1251	1.00	30.00	3.03	3.00	29.00	2.744
Al	1251	0.56	3.49	1.27	1.16	2.93	0.506
Na	1251	0.00	0.17	0.01	0.01	0.17	0.007
K	1251	0.08	0.49	0.21	0.20	0.41	0.067
W	1251	0.10	5.40	0.22	0.20	5.30	0.224
Sc	1251	1.50	7.30	3.05	2.80	5.80	0.972
Tl	1251	0.03	0.42	0.16	0.16	0.39	0.041
S	1251	0.02	0.11	0.03	0.03	0.09	0.014
Hg	1251	2.50	128.00	32.38	30.00	125.50	12.850
Se	1251	0.10	0.50	0.19	0.20	0.40	0.085
Te	1251	0.02	0.29	0.08	0.07	0.27	0.050
Ga	1251	1.90	12.60	4.43	4.10	10.70	1.723

Gold is distributed downslope from the Mexican Hat Deposit as shown on Figure 9-6.

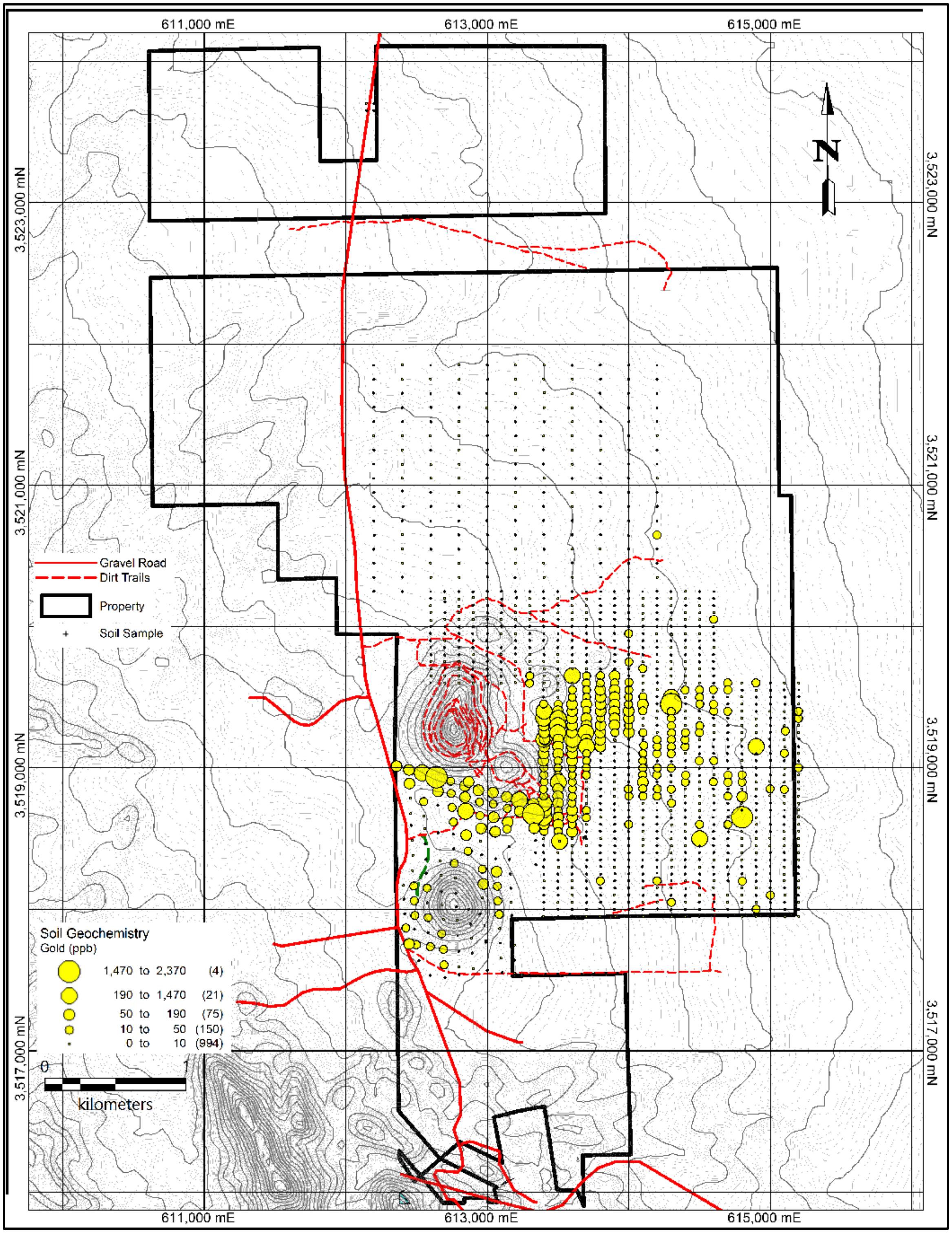


Figure 9-6: Distribution of Gold in Soils

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 9-10 below.

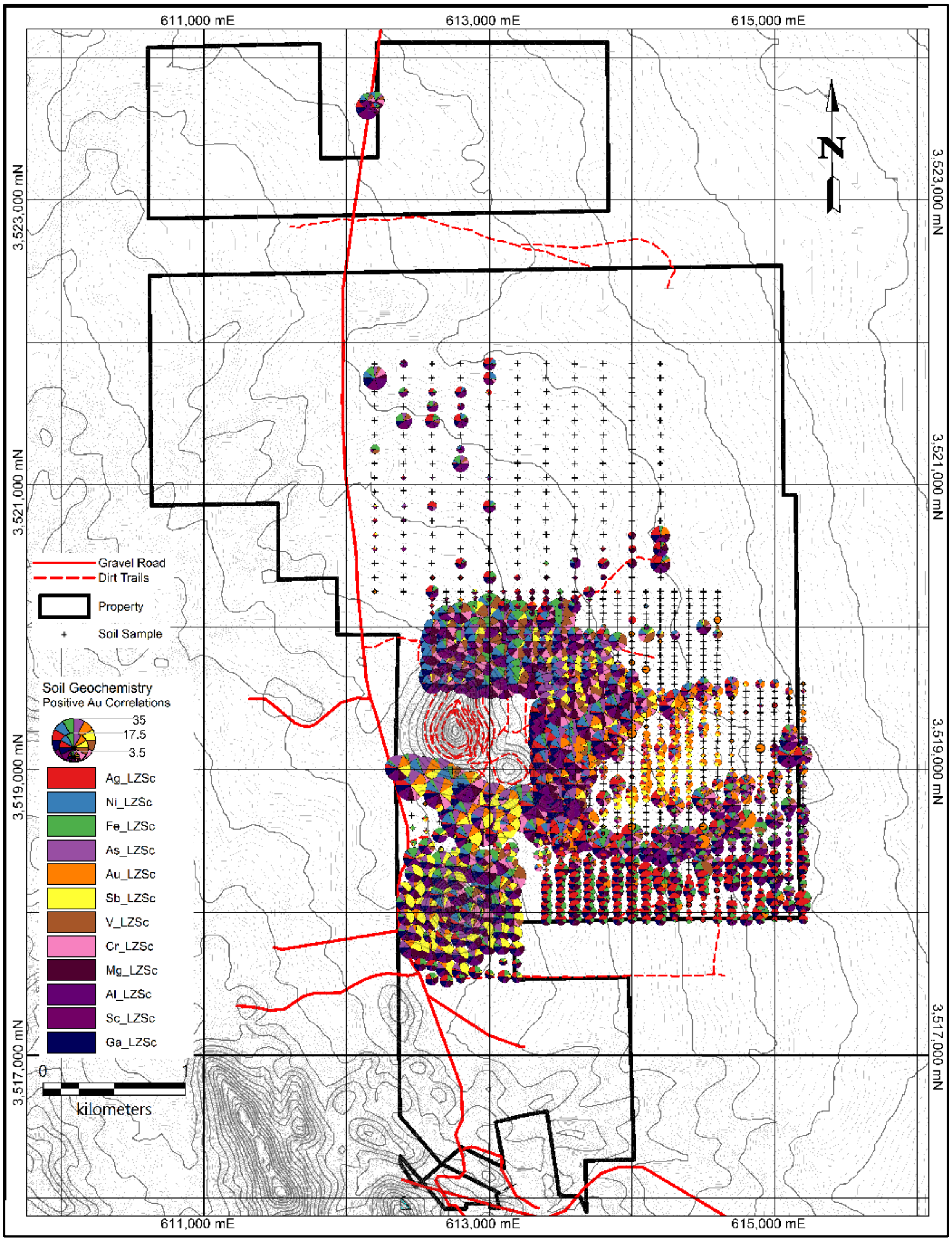


Figure 9-7: Distribution of all Elements Positively Correlating with Gold

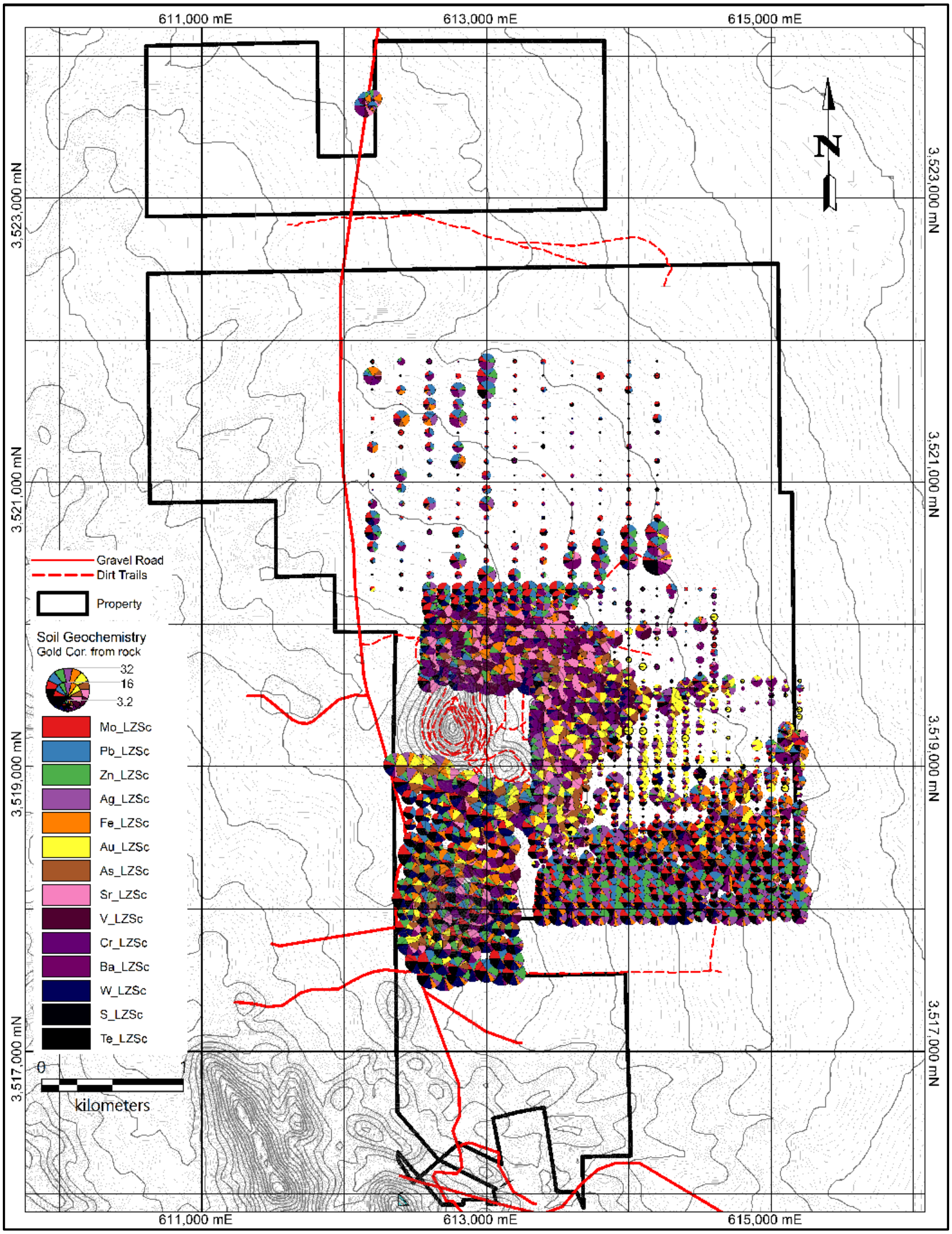


Figure 9-8: All Gold Pathfinders from Rock Geochemistry on Soils

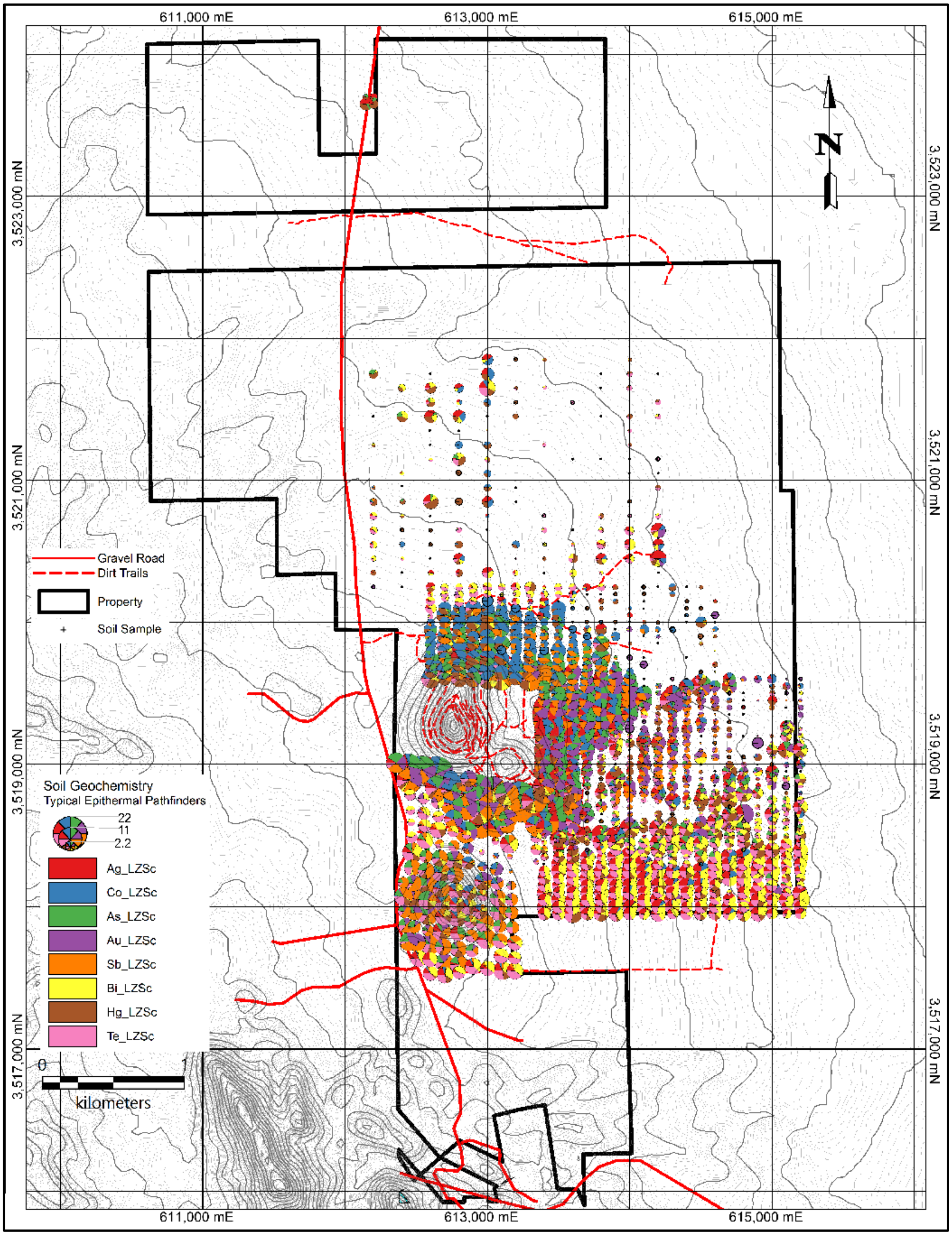


Figure 9-9: Distribution of Typical Epithermal Pathfinder Elements

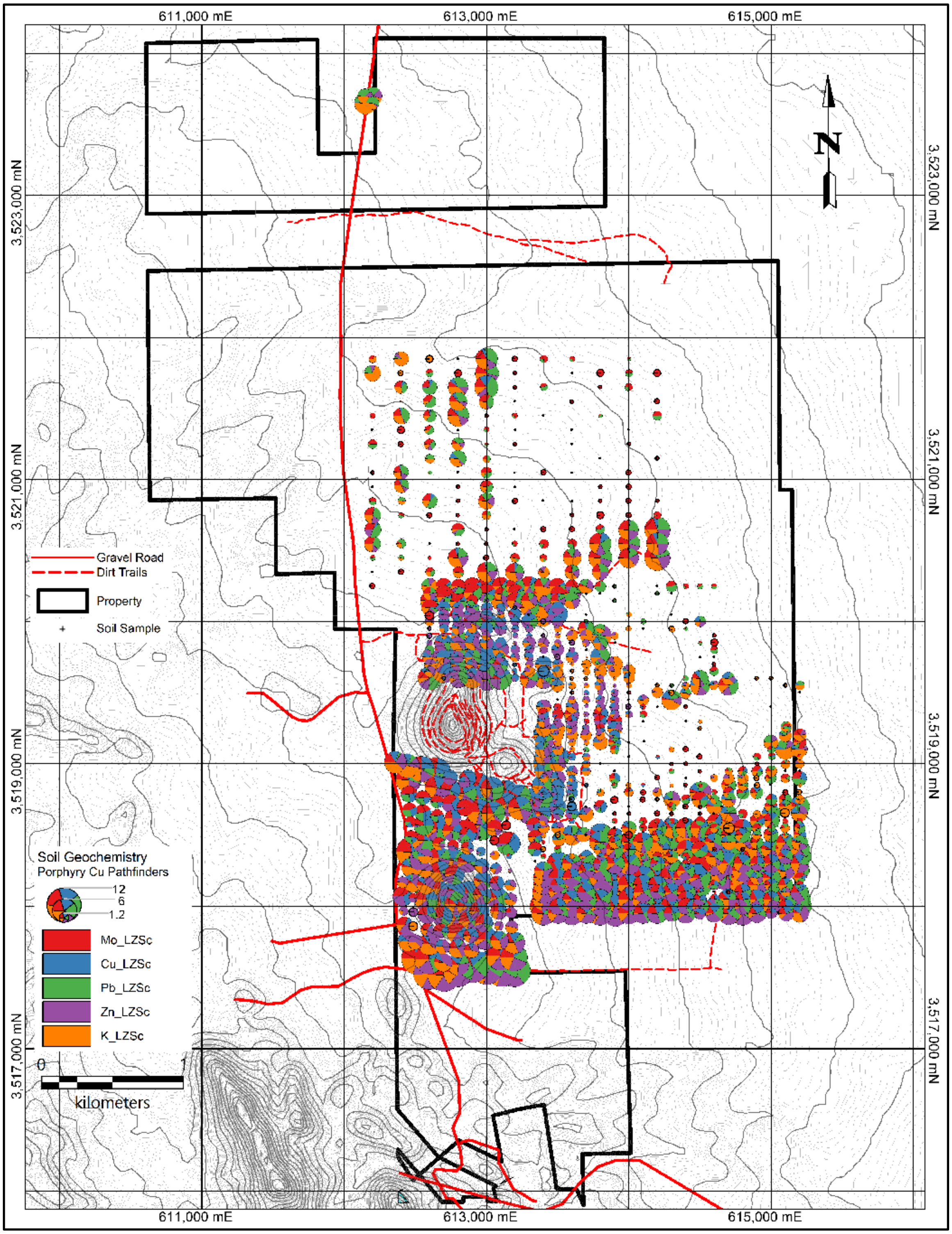


Figure 9-10: Distribution of Common Porphyry Copper Pathfinder Elements

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 9-11 and Figure 9-12.

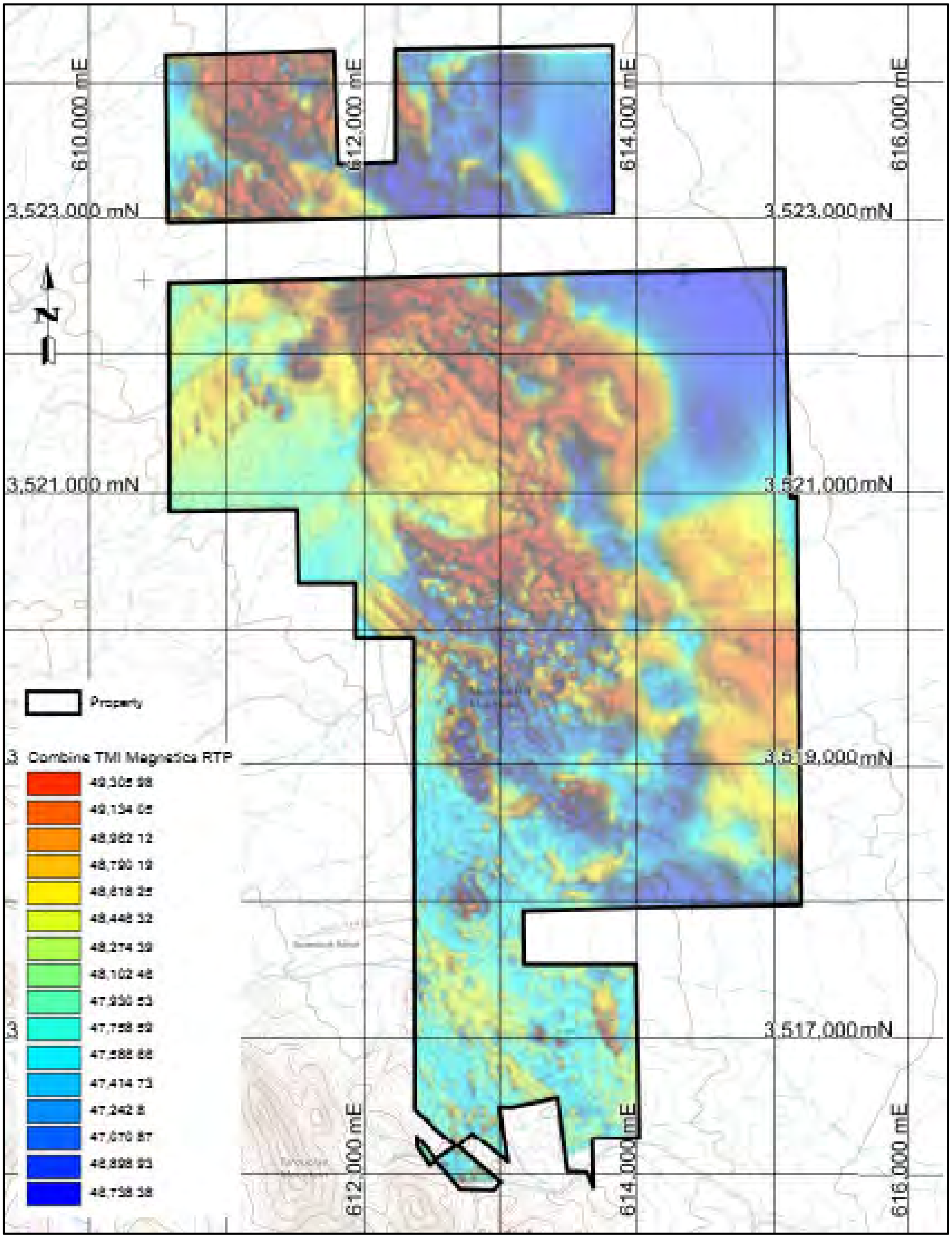


Figure 9-11: Combined TMI Survey RTP after Zonge (2017)

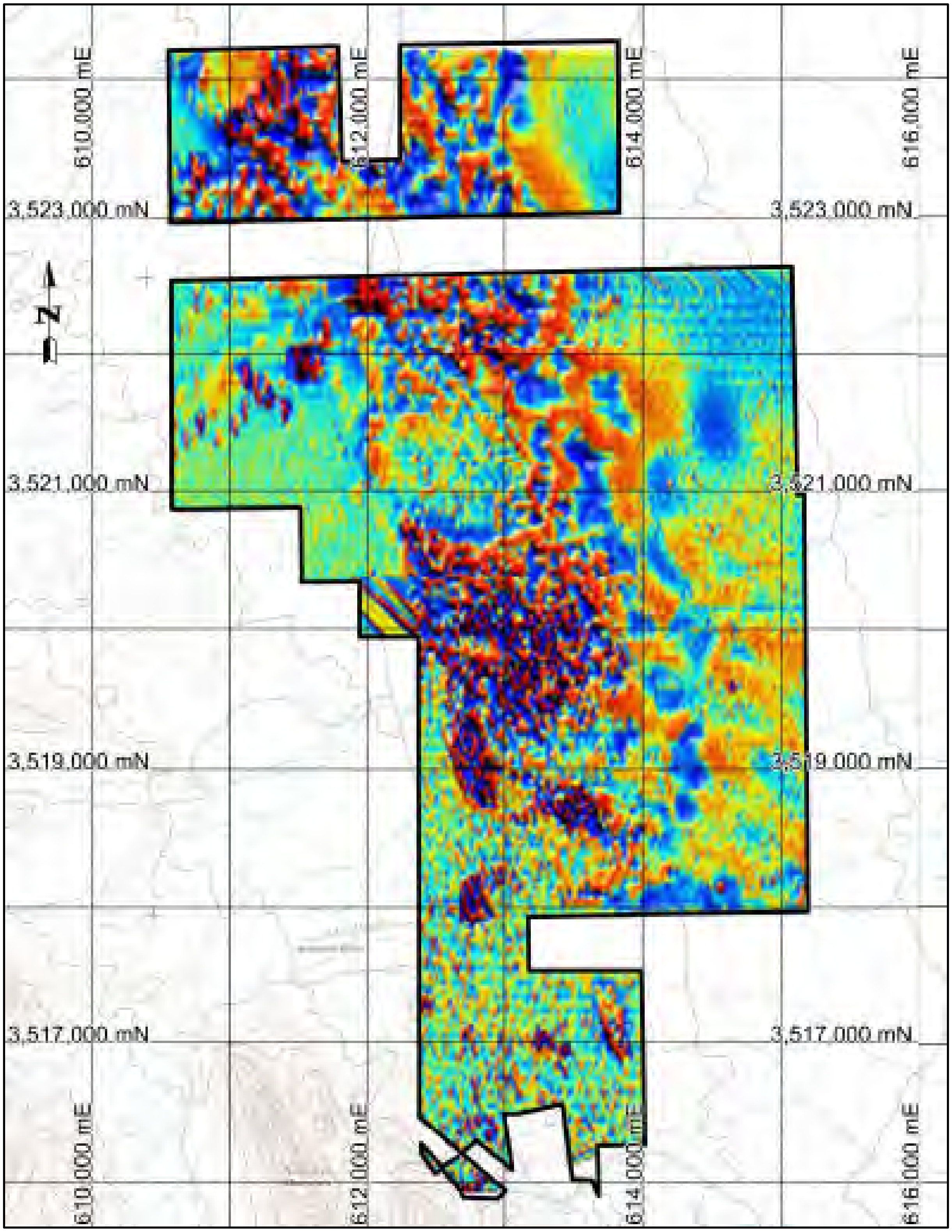


Figure 9-12: First Vertical Derivative of Combined TMI Survey, RTP after Zonge (2017)

9.2.3.2 *Audiomagnetotellurics*

Three lines of audiomagnetotellurics labeled AMT L2, L3 and L4, were tested across the eastern portion of the Mexican Hat Deposit as shown on Figure 9-13. Figures 9-14 through 9-19 below show the west-facing apparent resistivity and phase pseudo sections for Line L2, L3 and L4, respectively.

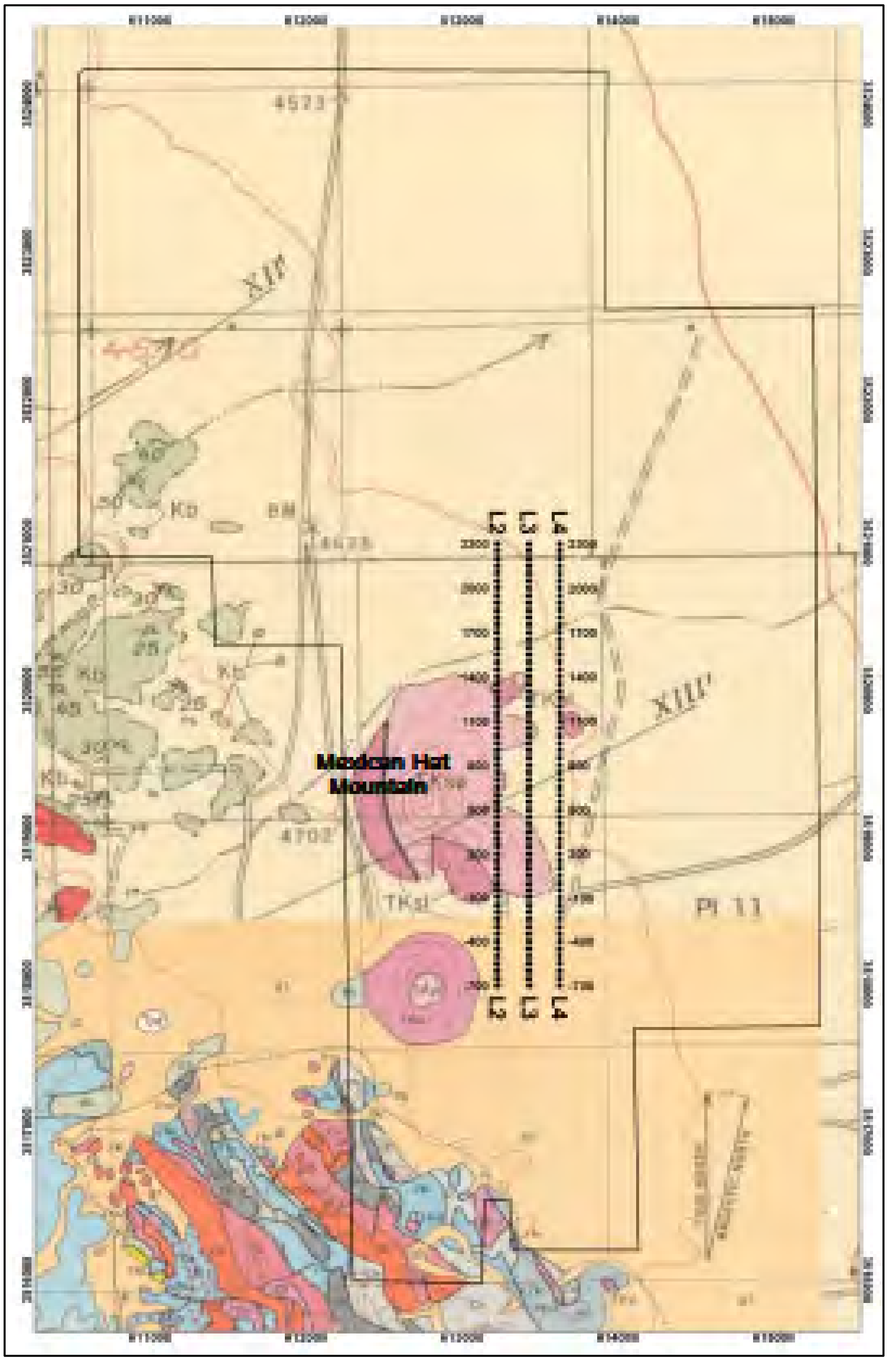


Figure 9-13: Location of AMT Test Lines

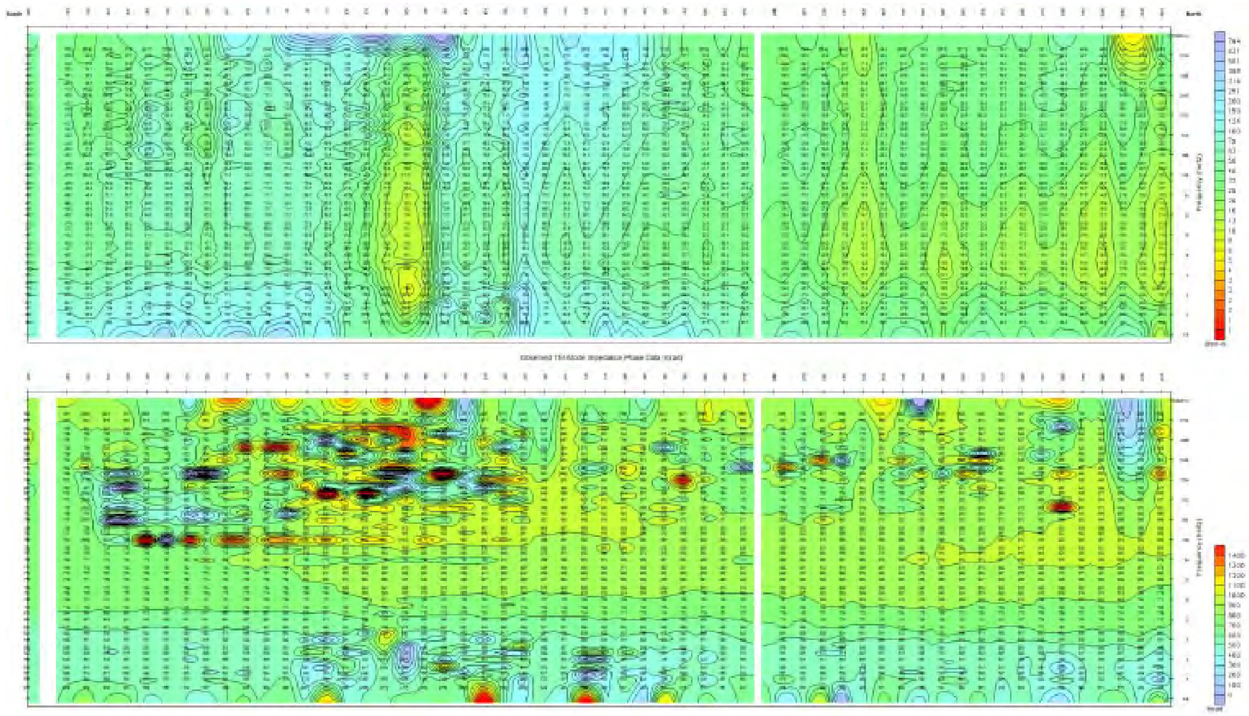


Figure 9-14: West-facing Apparent Resistivity and Phase Pseudo Section of Line 2

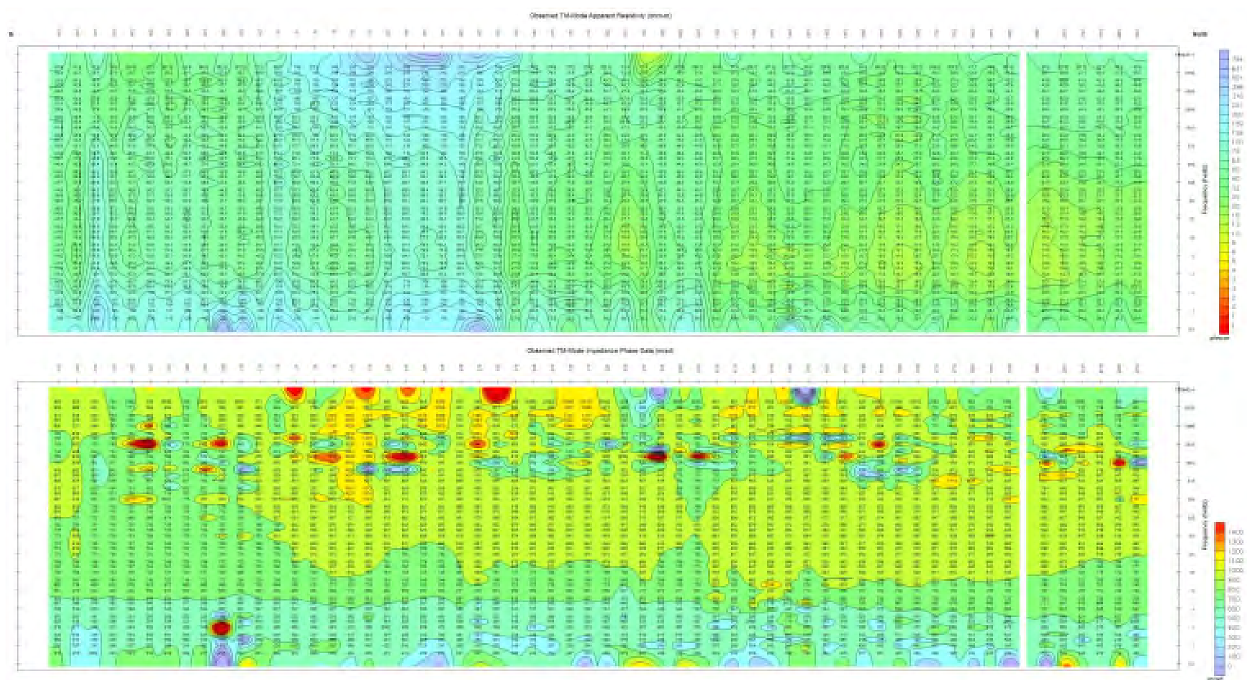


Figure 9-15: West-facing Apparent Resistivity and Phase Pseudo Section of Line 3

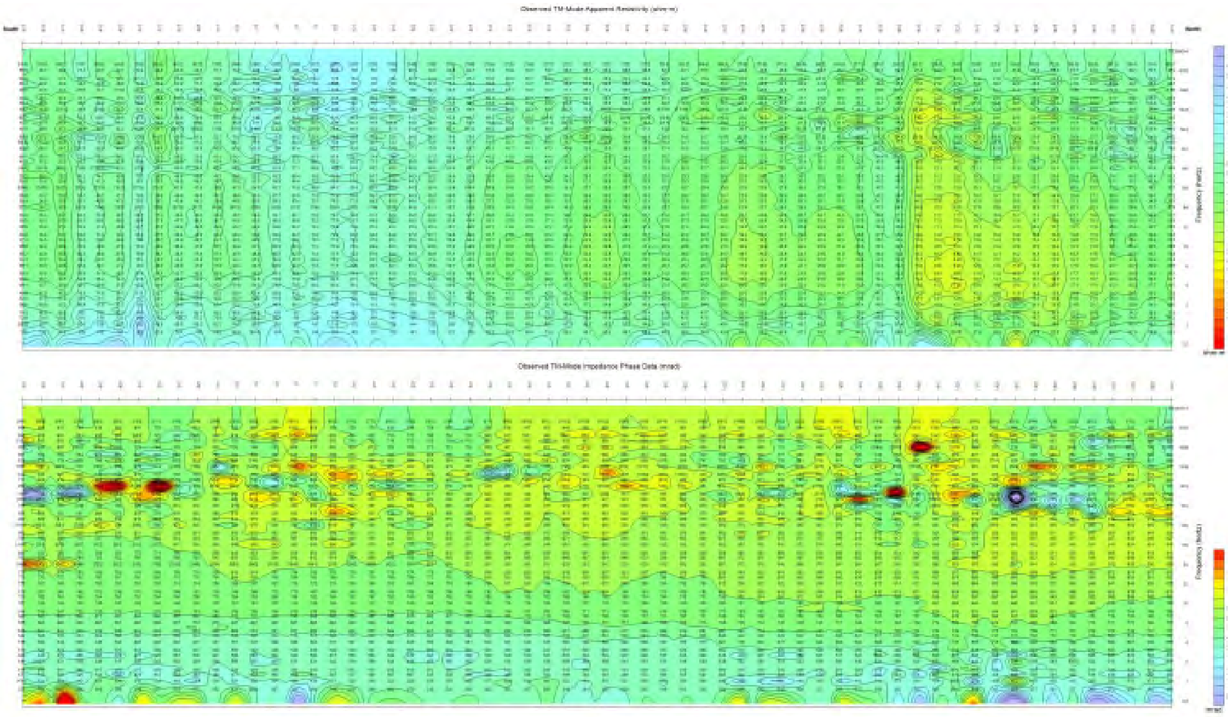


Figure 9-16: West-facing Apparent Resistivity and Phase Pseudosection of Line 4

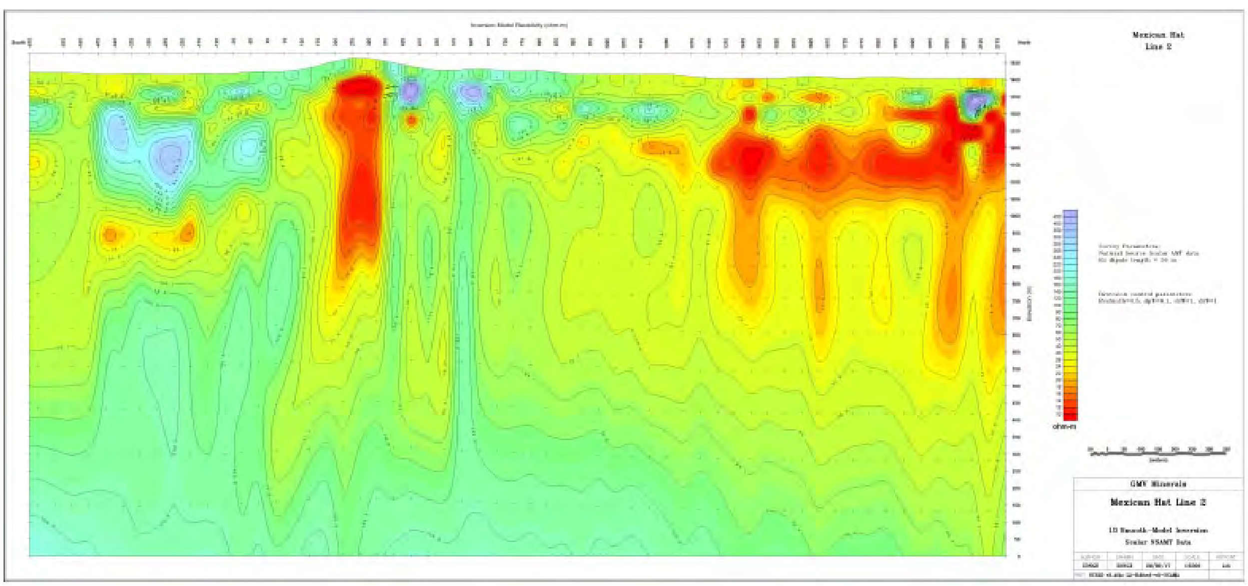


Figure 9-17: Section Showing West-facing 1D Inversion of Line 2 AMT Data

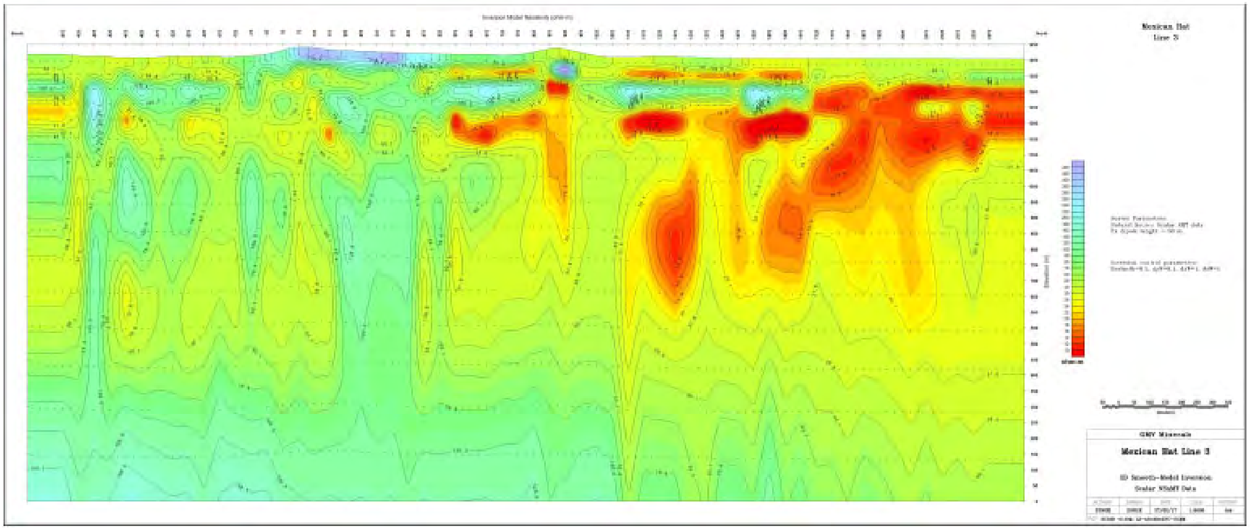


Figure 9-18: Section Showing West-facing 1D Inversion of Line 3 AMT Data

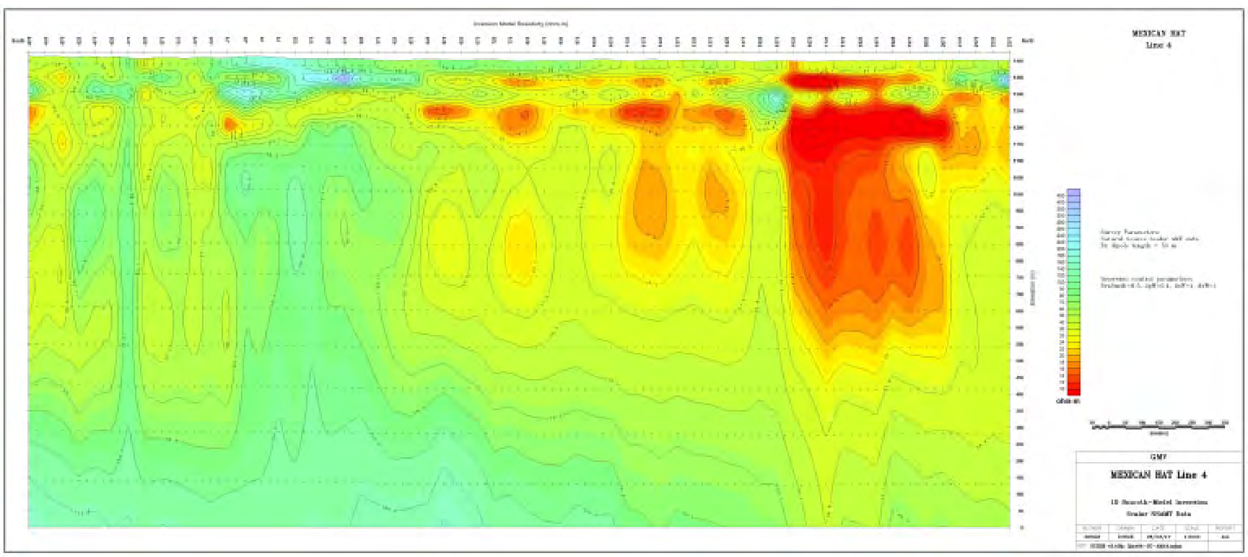


Figure 9-19: Section Showing West-facing 1D Inversion of Line 4 AMT Data

Zonge (2017) reported that, in general, the modeled resistivities are low over the entire cross-sections of the lines. The resistivity range of most 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 9-17). 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 9-18, stations pass over two lobes that extend eastward from the main body. Like 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 9-20 below shows the locations of the 3 gravity profile lines.

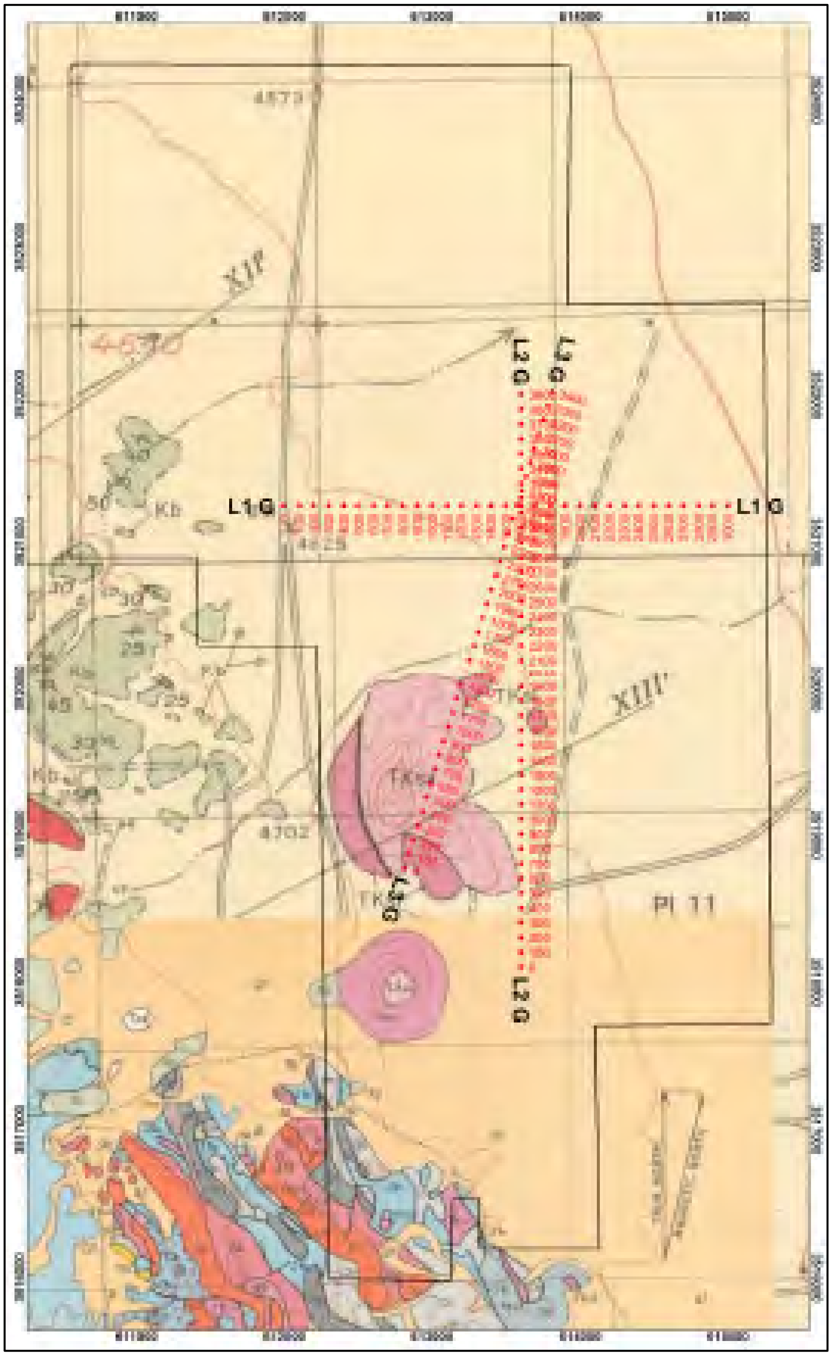


Figure 9-20: Location of Three Gravity Profile Lines

The following is drawn from Zonge (2017). Data from west-to-east gravity Line 1G is shown in Figure 9-21, 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 9-22. 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.

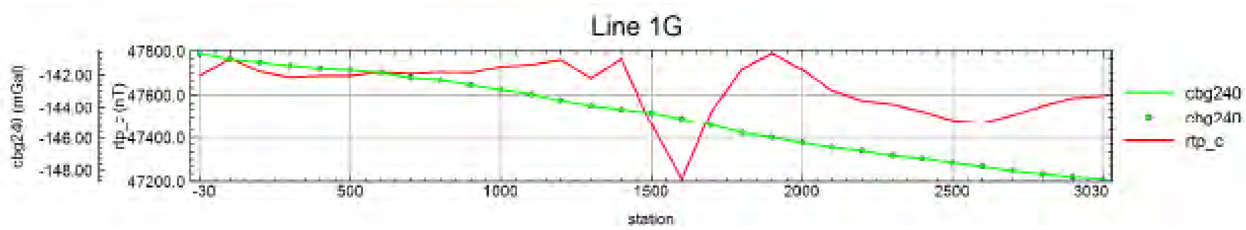


Figure 9-21: Section Along Gravity Line 1 with RTP Magnetic Data

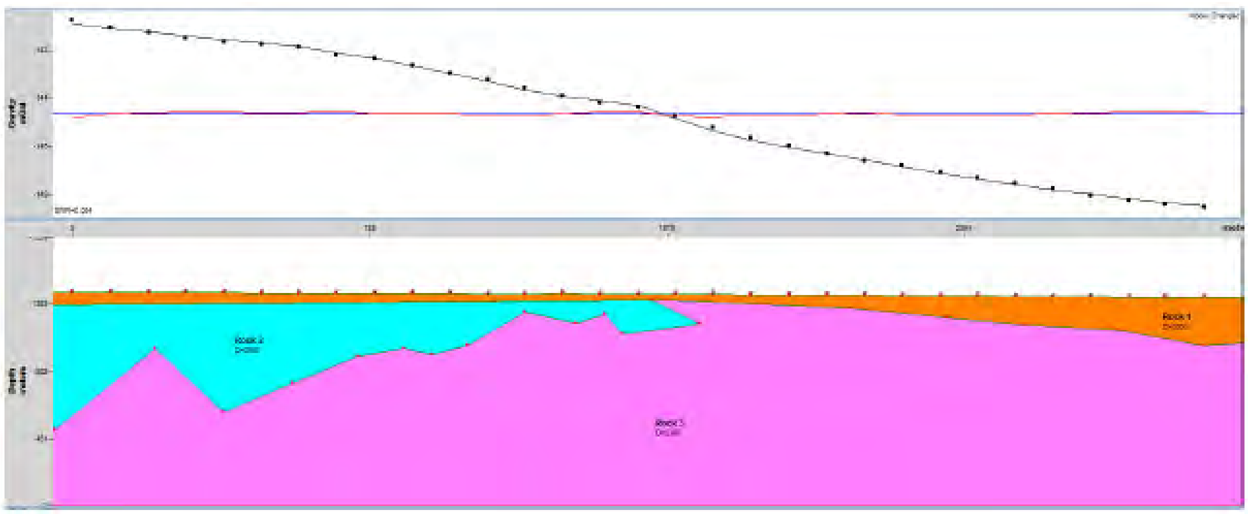


Figure 9-22: 1G Gravity Data Fit and Inverse Model for Line 1G. Upper Panel Shows Gravity Data (black dots). The Model Fit (solid black curve) and Fit Error (red curve)

Data from south-to-north gravity Line 2G is shown in Figure 9-23, 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 9-24. 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.

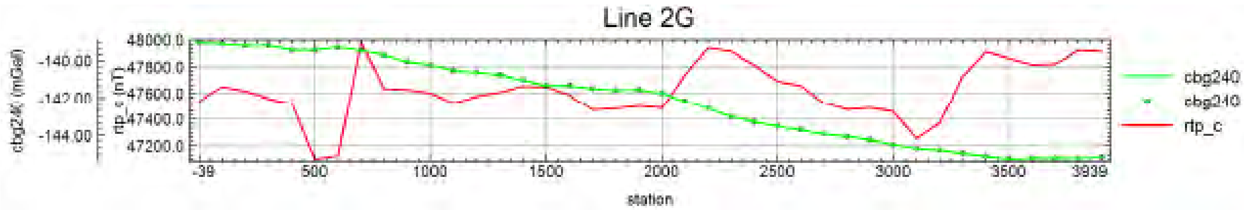


Figure 9-23: Terrain Corrected Gravity and RTP Magnetic Data along Line 2G

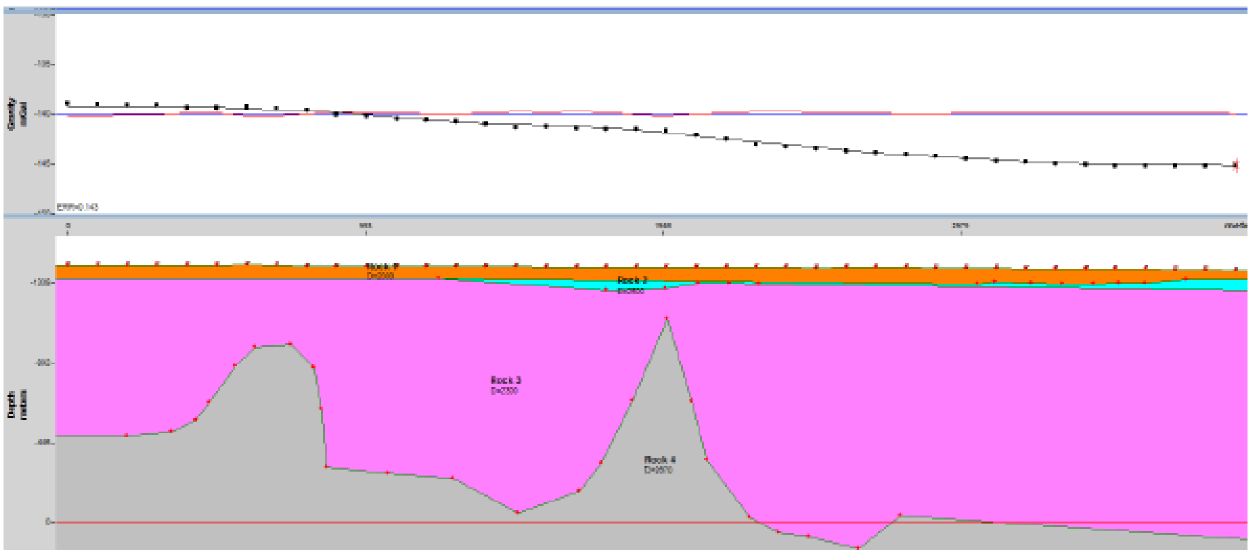


Figure 9-24: Gravity Data Fit and Inverse Model for Line 2G. Upper Panel Shows Gravity Data (black dots), the Model Fit (solid black curve), and Fit Error (red curve)

Data from southwest-to-northeast gravity Line 3G is shown in Figure 9-25, 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 9-26. 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.

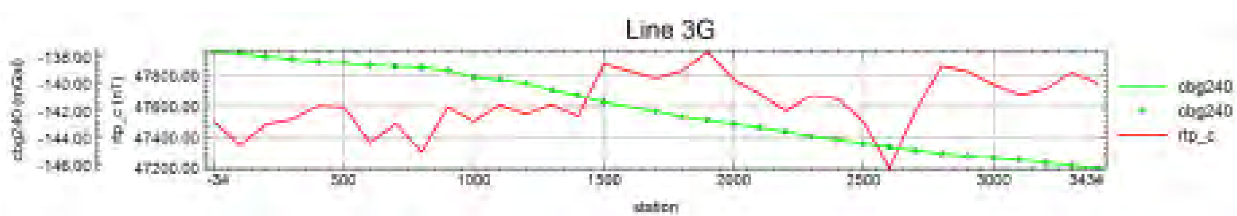


Figure 9-25: Terrain Corrected Gravity and RTP Magnetic Data along Line 3G

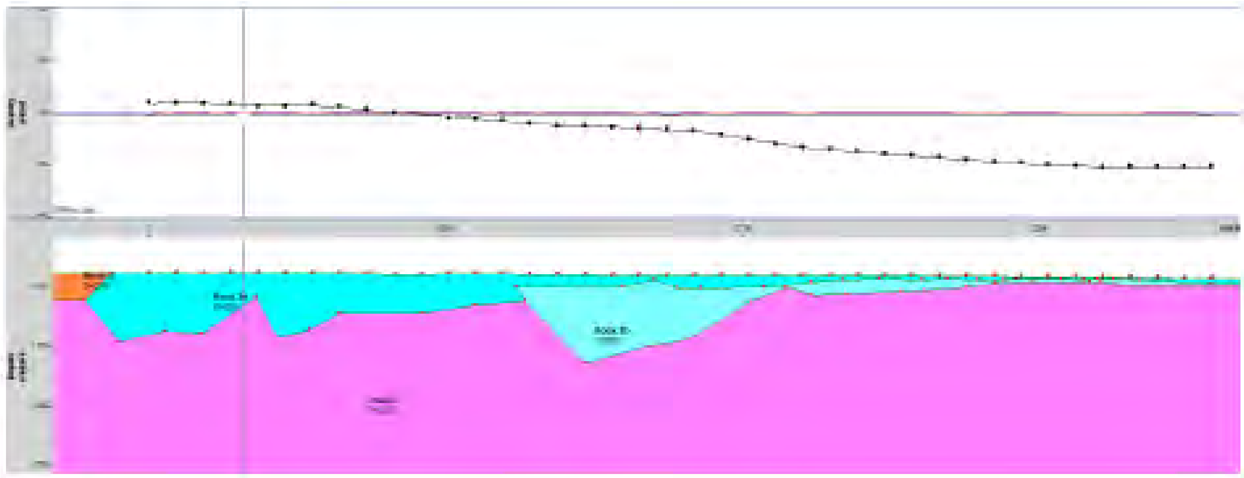


Figure 9-26 : Gravity Data Fit and Inverse Model for Line 3G. Upper Panel Shows Gravity Data (black dots), the Model Fit (solid black curve), and Fit Error (red curve)

10.0 DRILLING

GMV has completed four drill campaigns on the Mexican Hat Property since 2014, completing 15 reverse circulation (RC) holes totaling 4,776.5 m in 2016, and in 2017 completed 8 HQ core holes totaling 1,979.3 m and 15 RC holes totaling 4,032.9 m. In 2019, an additional 11 RC holes totaling 3,250 m were completed.

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	
2019					
	GMV	RC	11	3,250.0	
Grand Total			**223**	**39,337.3**	

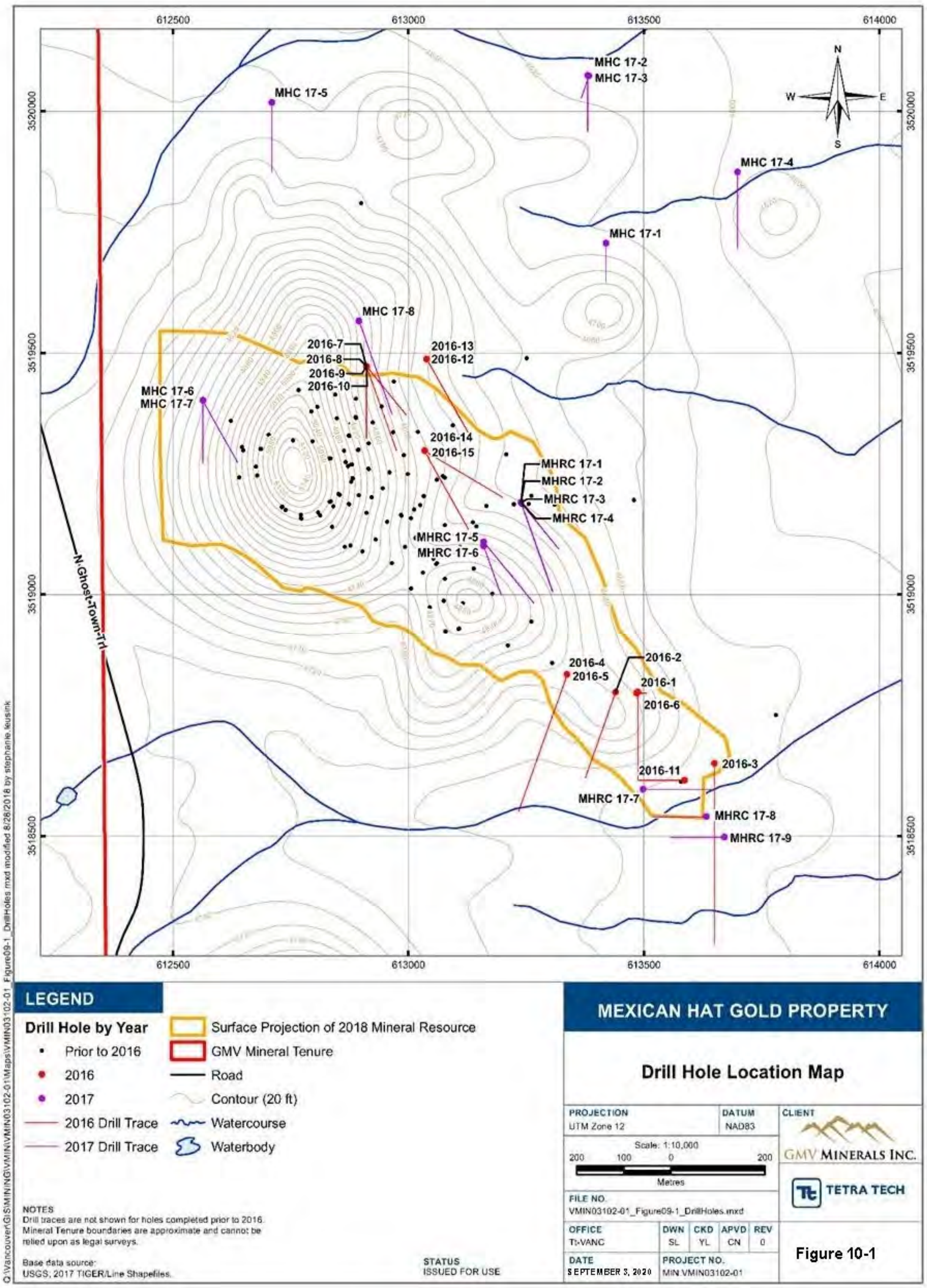


Figure 10-1: Drill Hole Location Map

10.1 2016 DRILLING

Fifteen RC holes were completed on the eastern portion of the Mexican Hat Deposit, specifically targeting the 120^0 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 with a nomenclature scheme which reflected the chosen drilling method for that hole. Diamond drillholes contained the prefix MHC, whereas 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).

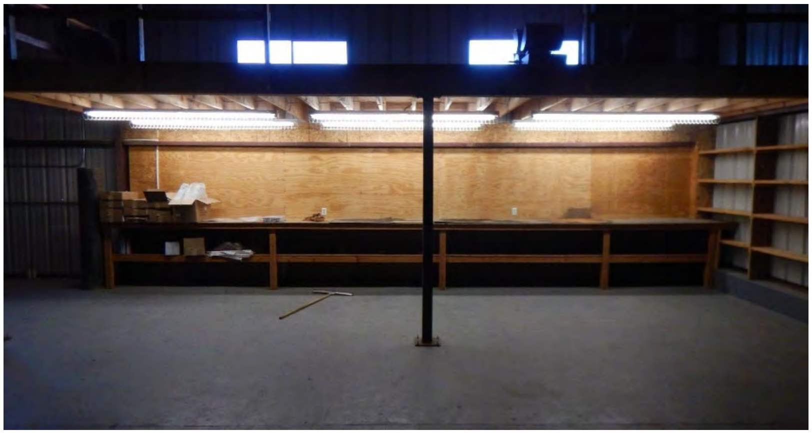
All holes were placed and oriented using field GPS and compass and checked upon completion for accuracy.

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. GMV's logging facility is shown in Photograph 10-1 below.



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

Table 10-3 Significant Results from 2016 RC Drillholes				
RC Hole	**From**	**To**	**Length**	**Gold**
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

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 with a nomenclature scheme which reflected the chosen drilling method for that hole. Diamond drillholes contained the prefix MHC, whereas 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

Table 10-4 2017 Mexican Hat Drilling Collar Information							
Hole_ID	**Azimuth**	**Dip**	**East_m**	**North_m**	**Elev_m**	**td__m_**	**Drill_Type**
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 GMV'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 GMV'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)	Drillhole	From	To	Length	Gold (g/t)
MHC 17-6	6.20	7.50	1.20	0.25	MHRC 17-1	18.30	27.40	9.10	0.80
MHC 17-6	6.20	11.00	4.70	0.12	MHRC 17-1	70.10	73.20	3.10	0.19
MHC 17-6	41.10	42.80	1.70	0.14	MHRC 17-2	42.70	48.80	6.10	0.46
MHC 17-6	54.90	63.70	6.90	0.48	MHRC 17-2	54.90	57.90	3.00	0.23
MHC 17-7	19.80	25.00	5.20	0.16	MHRC 17-2	67.10	73.20	6.10	0.24
MHC 17-7	103.60	105.60	2.00	1.30	MHRC 17-2	219.50	225.60	6.10	0.52
MHC 17-7	116.30	119.50	3.20	0.25	MHRC 17-3	24.40	42.70	18.30	0.57
MHC 17-8	199.50	200.30	0.80	0.12	MHRC 17-3	57.90	70.10	12.20	1.37
MHC 17-8	245.10	246.00	0.90	0.13	MHRC 17-3	225.60	234.70	9.10	0.20
MHC 17-8	284.80	288.40	3.60	0.24	MHRC 17-3	283.50	304.80	21.30	0.39
MHC 17-8	294.30	295.30	1.00	0.26	MHRC 17-4	3.00	12.20	9.20	0.23
MHC 17-8	297.90	327.50	5.00	0.78	MHRC 17-4	45.70	48.80	3.10	0.27
MHC 17-8	348.10	354.60	6.50	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

10.3 GMV DRILLING – 2019

The 2019 drilling campaign saw the completion of 11 drillholes on the Mexican Hat Property. The particulars of the drilling campaign are presented below in the following subsections.

10.3.1 Collar Surveys

All holes were numbered in accordance with a nomenclature scheme which reflected the chosen drilling method for that hole. Diamond drillholes contained the prefix MHC, whereas 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-19-02.

All holes were placed and oriented using field GPS and compass and checked upon completion for accuracy. Table 10-6 below summarizes the particulars of each hole completed during the 2019 program.

Table 10-6
2019 Mexican Hat Drilling Collar Information

Hole_ID	Azimuth	Dip	East_m	North_m	Elev_m	td__m_	Drill_Type
MHRC 19-1	200	-70	613,637.00	3,518,706.00	1,412.20	230	CORE
MHRC 19-2	180	-60	613,252.00	3,519,288.00	1,427.50	350	CORE
MHRC 19-3	140	-70	612,780.40	3,519,479.30	1,524.00	350	CORE

Table 10-6 2019 Mexican Hat Drilling Collar Information							
Hole_ID	**Azimuth**	**Dip**	**East_m**	**North_m**	**Elev_m**	**td__m_**	**Drill_Type**
MHRC 19-4	200	-70	612,780.40	3,519,476.00	1,524.00	350	CORE
MHRC 19-5	180	-60	613,000.00	3,519,361.00	1,458.80	350	CORE
MHRC 19-6	290	-75	612,879.10	3,519,268.20	1,515.60	300	CORE
MHRC 19-7	170	-55	613,386.00	3,519,004.00	1,450.20	250	CORE
MHRC 19-8	200	-88	613,019.00	3,519,202.00	1,456.70	300	CORE
MHRC 19-9	150	-65	612,773.00	3,519,423.00	1,533.80	320	CORE
MHRC 19-10	220	-70	612,837.00	3,519,129.00	1,522.30	200	CORE
MHRC 19-11	170	-75	612,837.00	3,519,129.00	1,522.30	250	CORE

10.3.2 Downhole Surveys

No downhole surveys were completed.

10.3.3 Logging

RC samples were collected, and chip trays were developed for each interval. In addition, sludge boards were generated for all RC holes. These have been preserved in GMV's logging facilities.

10.3.4 Recovery

Recovery measurements were not collected for the 2019 RC program; however, sufficient volume was collected for analysis for each drill run.

10.3.5 2019 Significant Drilling Results

A summary of the significant values intersected during the RC and core drilling program is shown below in Table 10-7. Significant intersections from the 2019 drilling are summarized in Table 10-7. The intercepts are listed as downhole widths and have not been estimated as true widths.

Table 10-7 2019 Significant Reverse Circulation Intersections. All Intersection Lengths are Drill Lengths and Not True Widths				
Drillhole	**From**	**To**	**Length (m)**	**Gold (g/t)**
MHRC 19-1	67	70	3	1.22
MHRC 19-2	61	64	3	1.09
MHRC 19-3	314	320	6	0.37
MHRC 19-4	119	128	9	0.82
MHRC 19-4	171	183	12	0.66
MHRC 19-4	207	223	16	0.97
MHRC 19-4	235	238	3	0.61
MHRC 19-5	67	73	6	0.30
MHRC 19-5	88	104	16	0.61

Table 10-7 2019 Significant Reverse Circulation Intersections. All Intersection Lengths are Drill Lengths and Not True Widths				
Drillhole	**From**	**To**	**Length (m)**	**Gold (g/t)**
MHRC 19-5	174	192	18	0.70
MHRC 19-5	241	262	21	0.42
MHRC 19-6	0	12	12	0.51
MHRC 19-6	58	155	97	0.52
MHRC 19-6	174	223	49	0.58
MHRC 19-6	268	271	3	1.00
MHRC 19-8	73	101	28	0.38
MHRC 19-8	192	250	58	0.53
MHRC 19-9	155	317	162	0.24
MHRC 19-10	15	24	9	0.39
MHRC 19-10	30	67	37	1.24
MHRC 19-10	113	137	24	0.26
MHRC 19-11	6	15	9	0.39
MHRC 19-11	70	79	9	0.85
MHRC 19-11	122	137	15	0.88

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 2016, 2017, AND 2019 SAMPLING METHODS

11.1.1 Sample Collection

The 2016, 2017, and 2019 drill programs were completed under the direct supervision of D.R. Webb, B.T. Malahoff, or S. Silaev, of DRW Geological. All samples remained in their possession 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.

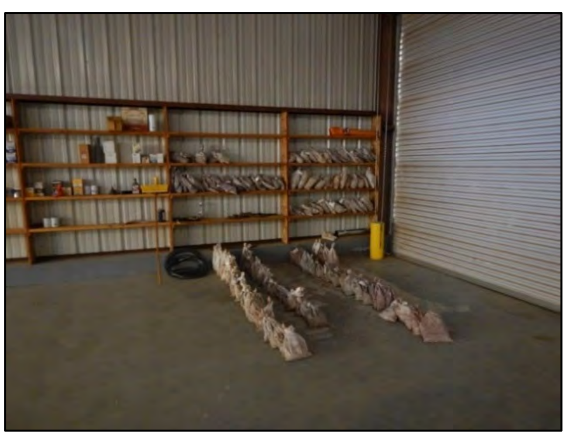
11.1.2 Sampling

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.

Drill logs 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 and results are displayed graphically in Figure 11-1 to Figure 11-5. Discussion of the QAQC control program is included below.

Table 11-1
Number of Analyses of Gold Completed in 2019 for each CRM and Blank

Standard or Blank	No. of Analyses
CDN-BL-10	36
CDN-GS-IT (Fire Assay)	11
CDN-GS-IW (Fire Assay)	20
CDN-GS-P5C (Fire Assay)	7
CDN-GS-P4H (Fire Assay)	17
Laboratory standard/blank	104

11.2.1.1 CDN GS-P5C

In total, 55 CRM standards with expected values of 0.571, 0.400, 1.063, and 1.080 g/t gold were submitted into the sample stream and assayed via ICP analysis, along with an additional 104 standards which tested the laboratories fire assay accuracy.

Figure 11-1: Certified Standard CDN GS P5C, fire Assay Results

11.2.1.2 CDN GS-1T

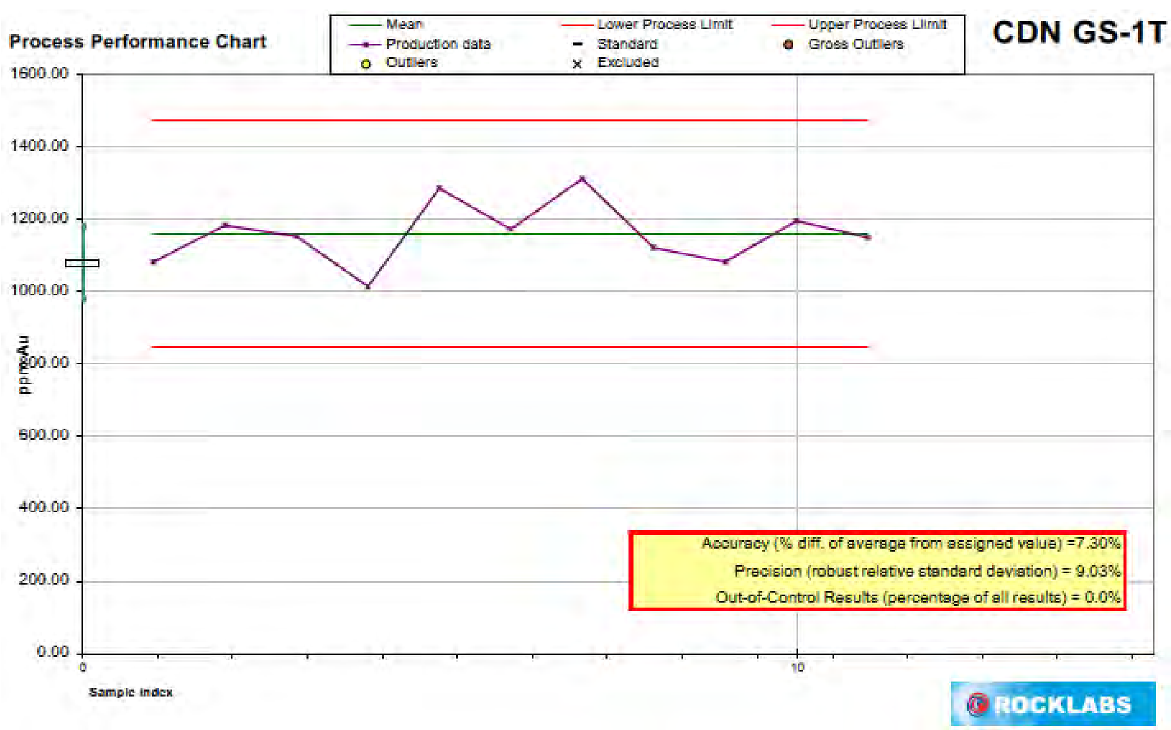


Figure 11-2: Certified Standard CDN GS 1T, Fire Assay Results

11.2.1.3 CDN GS-1W

Figure 11-3. Certified Standard CDN GS 1W, Fire Assay Results

11.2.1.4 CDN GS-1P4

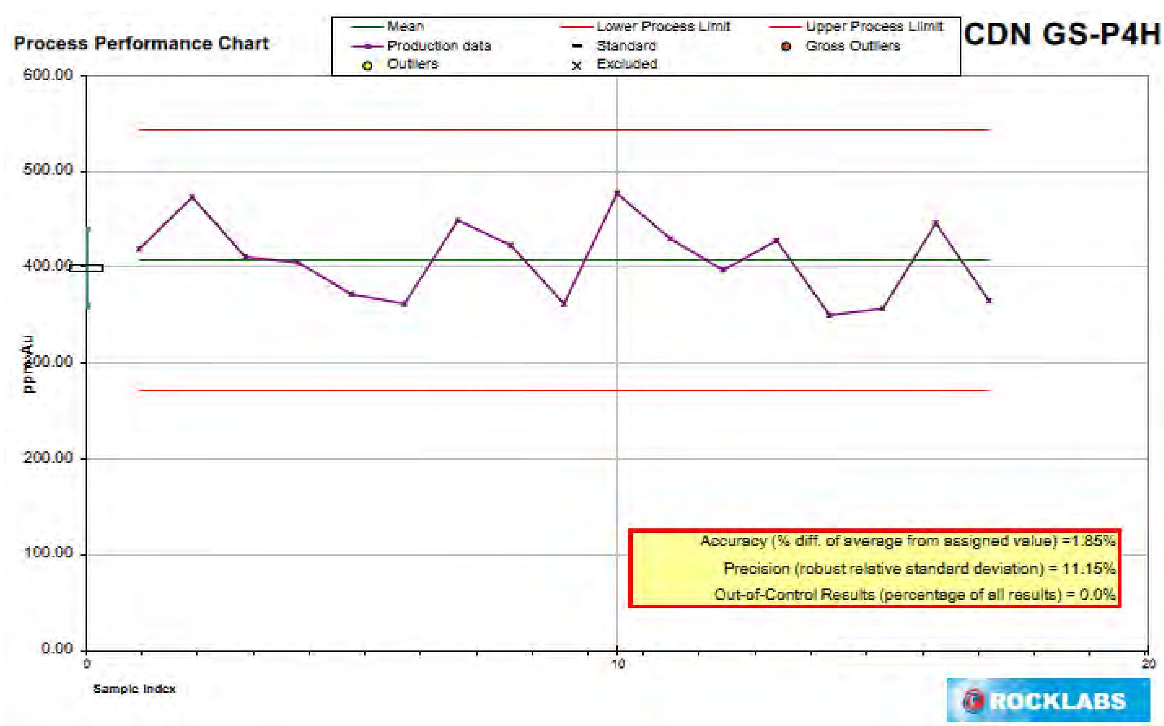


Figure 11-4. Certified Standard CDN GS-1P4, Fire Assay Results.

11.2.1.5 Analytical Blank

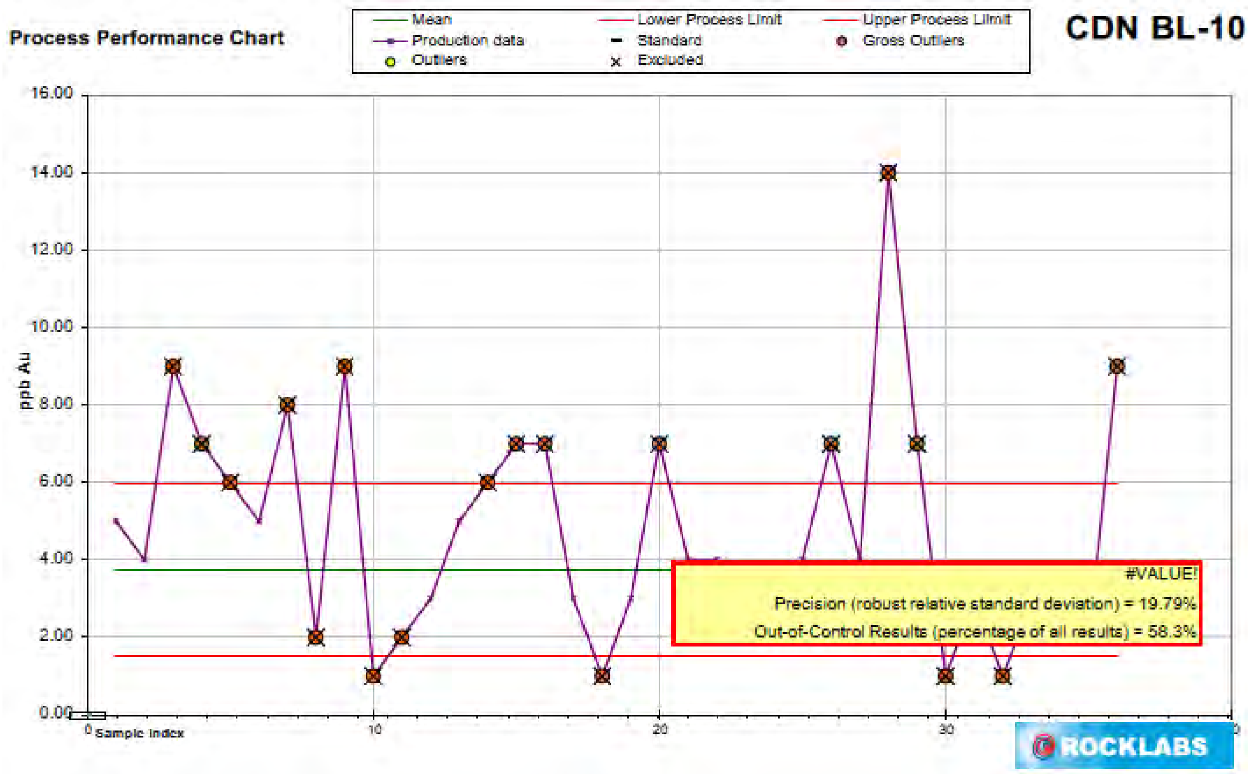


Figure 11-5. Plot of all CDN BL-10 ICP Results

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 11-6 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 11-7.

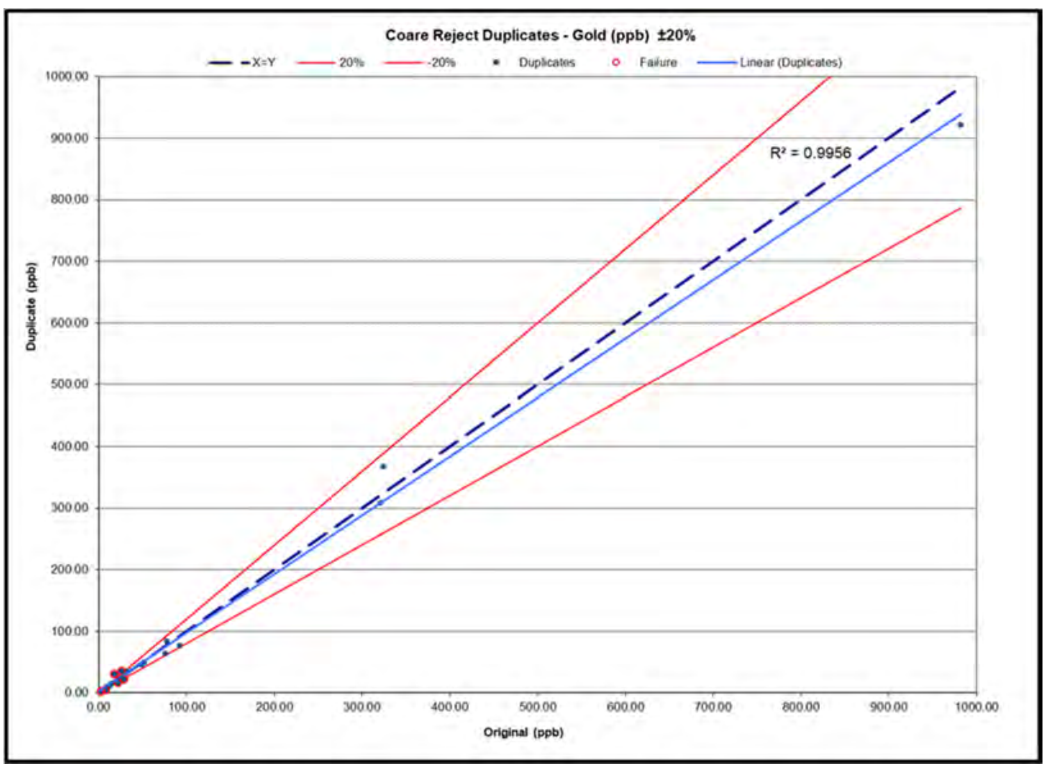


Figure 11-6: Coarse Reject Original Vs Duplicate Assays

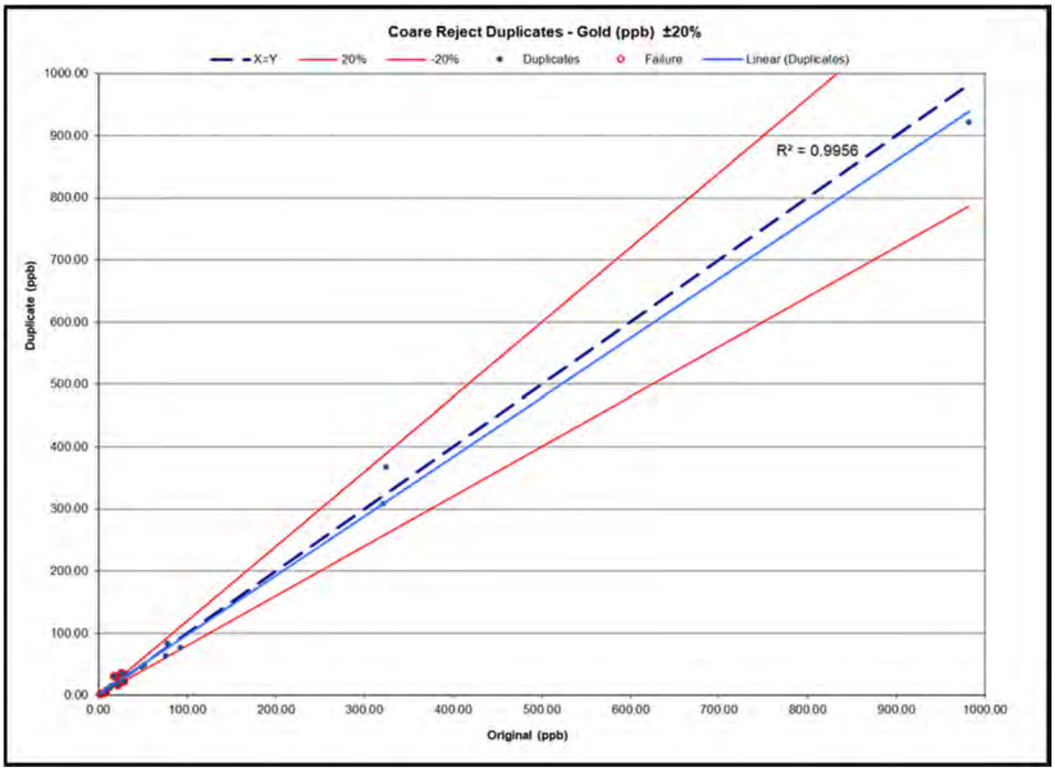


Figure 11-7: Mean Vs Duplicate Data for Au <100ppb

11.2.3 Pulp Duplicates

A total of 43 pulp duplicates were assayed during the 2016, 2017 field program. For pulp duplicates it is 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 11-8). 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

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
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

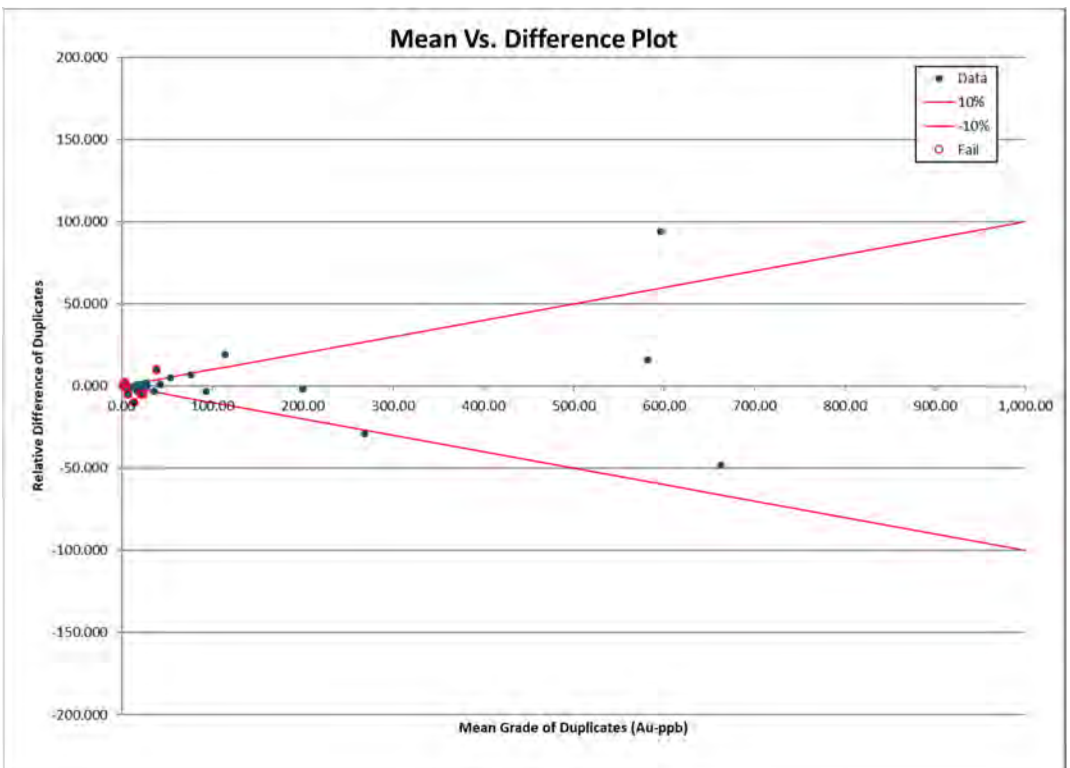


Figure 11-8: Pulp Duplicates Mean vs Difference. Samples with >10% RPD Circled in red (Gold grades presented in ppb)

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.

12.0 DATA VERIFICATION

12.1 TETRA TECH QP INDEPENDENT VERIFICATION

12.1.1 Database Audit

The Tetra Tech QP, James Barr, conducted a detailed review and audit of the Mexican Hat geological database provided by GMV in 2017 and 2018, in advance of a previous Mineral Resource Estimate with effective date of June 22, 2018. Data available at this time were included in the audit. 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.

Data collected from the 2019 drilling campaign were provided to Tetra Tech, were reviewed and merged into the existing drill hole database.

<u>Collar Locations</u>

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.

<u>Property Digital Elevation Model</u>

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 reflect the spatial location more accurately on the ground.

The identification of the shift in collar locations in a relatively 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.

Collar Elevations

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

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.

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

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.

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.

Review of QAQC Data

A review of the QAQC program that was implemented by GMV was undertaken, and 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, most 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 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.

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 Summarizes Results of Reconnaissance Grab Sample Validation						
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

Eleven samples (Table 12-2) were selected from the 2019 drilling to be analyzed at ALS Global in Nevada to check for interlaboratory errors.

Table 12-2 2019 Drilling Samples for Interlaboratory Errors								
Hole	**From (m)**	**To (m)**	**Length (m)**	**SAMPLE DESCRIPTION**	**ALS Au-AA25 Au ppm**	**BVI FA330 Au ppm**	**Difference**	**% Diff**
MHRC 19-9	176.78	179.83	3.05	xxx 245922	0.43	0.505	0.075	14.85
MHRC 19-9	225.55	228.60	3.05	245939	0.58	0.441	-0.139	-31.52
MHRC 19-9	240.79	243.84	3.05	245945	0.79	1.289	0.499	38.71
MHRC 19-4	213.36	216.41	3.05	380702	5.17	3.231	-1.939	-60.01
MHRC 19-4	234.70	237.74	3.05	380709	0.27	0.605	0.335	55.37
MHRC 19-8	213.36	216.41	3.05	380912	4.52	4.179	-0.341	-8.16
MHRC 19-6	109.73	112.78	3.05	398792	0.31	0.421	0.111	26.37
MHRC 19-1	36.58	39.62	3.05	398813	<0.01	<0.002	0	0
MHRC 19-1	45.72	48.77	3.05	398816	<0.01	<0.002	0	0
MHRC 19-1	51.82	54.86	3.05	398818	0.01	<0.002	0	0
MHRC 19-6	195.07	198.12	3.05	398898	1.34	1.489	0.149	10.01
				Average	**1.49**	**1.52**	**-0.11**	**-7.48**

Notes: Au-AA25 are the check assays from ALS Global Laboratories in Nevada
FA330 are the original assays from Bureau Veritas Laboratories in Vancouver

The check assays for this small sample population averages 0.11 gpt gold or 7.48% lower than the original assays with a high degree of variability indicating a nuggety gold distribution.

12.1.3 QP Opinion on Data Verification

The Tetra Tech QP 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.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

Two relevant metallurgical test programs have been performed on samples from the Project. The first program, in 2015 was performed at McClelland Labs of Sparks, Nevada followed by a second program in 2016 at Bureau Veritas of Richmond, British Columbia, Canada.

Four types of mineralization were identified during preliminary geological assessments: latite comprising approximately 80% of the mineralization, with 8% each of andesite and basalt, and the remaining 4% dacite.

13.1 MCCLELLAND TEST WORK

Metallurgical test work on two bulk composites comprising of trench samples, Tr-3, and Tr-12, were performed at McClelland Laboratories with the objective of determining heap leach amenability to extract gold and silver. Four total bottle roll cyanidation tests were conducted, one for each sample at 80% -1.7 mm and 80% -75 mm. Head samples were submitted for assays to determine gold, silver, cyanide soluble gold and multi-element ICP and bulk density measurements were taken on rock selections.

The average gold head grades for Tr-3 and Tr-12 were 1.53 and 1.33 g/t, respectively. Silver head grades and recoveries were low. A summary of the bottle roll and column tests is shown in Table 13-1 where the two tests run for 98 days represent the columns.

Table 13-1: McClelland Bench-Scale Test Work Summary

Sample	Leach Time Days	Size P80	Au % Rec.	Au g/t Tail	Au g/t Calc Head	Ag % Rec.	Ag g/t Calc Head	NaCN kg/t	Lime kg/t
Tr-3 Column	98	37.5 mm	95.0%	0.07	1.39	4.8%	1.4	0.92	1.4
Tr-3 Bottle Roll	4	1.7 mm	84.9%	0.24	1.59	23.1%	1.3	0.1	1.5
Tr-3 Bottle Roll	3	0.075 mm	96.9%	0.05	1.60	47.4%	<1.9	0.07	1.8
Tr-12 Column	98	37.5 mm	77.1%	0.30	1.31	3.2%	3.1	0.96	1.6
Tr-12 Bottle Roll	4	1.705 mm	82.2%	0.26	1.36	11.5%	2.6	<0.07	1.8
Tr-12 Bottle Roll	3	0.075 mm	97.6%	0.05	1.25	38.1%	2.1	0.16	2.1

The bottle roll test results showed that the samples were amenable to cyanidation at the particle sizes selected. Column tests showed 95.0% and 77.1% gold recovery after 98 days from the Tr-3 and Tr-12 samples, respectively. Gold recovery was deemed slow and was still increasing for Tr-12, which would likely have benefitted from a longer leach cycle. Cyanide consumption was moderate for both columns at 0.92 and 0.96 kg/t. The lime addition was less than optimal causing the pregnant solution pH to drop to 10.2 after 40 days, but that level held for the remainder of the tests.

The McClelland work was based on trench samples which may not be truly representative of the bulk of the mineralized material to be mined and it is unclear why core samples were not used. For a Preliminary Economic Assessment (PEA) samples should be indicative of the mineralized material to be mined and processed since the objective of these tests was to determine leach amenability, it likely presents a low risk as long as the results are not relied on heavily with regard to projecting recovery going forward.

13.2 BUREAU VERITAS TEST WORK

Based on the results of the McClelland testing, it was determined that a run-of-mine (ROM) leach may be possible. No summary laboratory report was provided for the Bureau Veritas work, but the data was provided in spreadsheet form. Column testing was conducted on material from an 18-tonne "bulk" sample, but it is unclear how the sample was collected. Samples for bottle roll tests were taken from some samples from the drilling program. The rhyolite and basalt samples subjected to bottle roll tests was not the bulk of the described mineralogy. Bottle roll and column test results are summarized in Table 13-2 below.

Table 13-2: Bureau Veritas Bottle Roll and Column Test Results

Sample	Leach Time Hour	Size P80 micron	Au % Rec.	Au g/t Tail	Au g/t Calc Head	Au g/t Assay Head	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

Gold recoveries show that the samples leached very well at finer particle sizes and compare well to the previous test work, however the reagent consumption is variable.

The column test was conducted on the bulk sample in a 1-meter diameter by 6-meter tall column at a nominal 150 mm (almost 6-inch) passing size. Leach progress appeared to stall after 40 days so the column was drained down and restarted a few days later. Leaching continued until Day 137 when it appeared to stop again, and the test was terminated at a gold recovery of approximately 57%.

ROM leaching is difficult to simulate in a column test and further work should be performed to refine this test procedure if this type of testing is planned in the future. Collecting representative sub-samples also proves to be challenging.

14.0 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 Mineral Resource Estimate, with Effective Date June 22, 2018, and originally documented in the report "2018 Technical Report and Mineral Resource Estimate on the Mexican Hat Project, Cochise County, Arizona, USA" dated August 29, 2018. This new work has been completed using Datamine Studio RM v 1.5.62.0.

Since the completion of the previous resource estimate, eleven additional reverse circulation (RC) drillholes, totaling 3,250 m, have been completed, along with the collection of 1,064 RC chip samples. The program tested and expanded upon the extent of the previously modeled deposit resources both at depth and along strike.

14.2 PREVIOUS ESTIMATES

A Mineral Resource Estimate was previously reported for the Mexican Hat Project in 2018 by Tetra Tech. The work incorporated the results from historical drilling and channel sampling, and the results of the GMV channel sampling up to and including the 2017 campaign. Both the geological modeling and block model development were completed using Datamine Studio RM software. The Mineral Resource Estimate was reported to be pit constrained at a 0.20 g/t cut-off and is summarized in Table 14-1 below.

The 2019 resource statement is superseded by the current resource statement and is no longer relied upon.

Table 14-1
Previous Mineral Resource Estimate, Effective Date June 22, 2018

Category	Cut-off (g/t Au)	Tonnes	Grams Au	Grade (Au, g/t)	Ounces Au
Inferred	0.20	32,876,000	20,252,000	0.62	651,000

- The Mineral Resource Estimate has been constrained to a preliminary optimized pit shell, using the following parameters: BD = 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 = $1,375 per troy ounce, throughput at 15,000 tpd., discount rate = 5%.
- 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.

14.3 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 (Table 14-2).

None of the drillholes completed by Kalahari Resources (1996) were included in the resource estimate due to lack of drilling logs and possible grade bias identified by geostatistical assessment. Trench MHT-133 (a single sample) was removed as it appeared to unduly influence the resource estimate. The holes and trenches listed in Table 14-3 were judged by Tetra Tech to be insufficiently supported and were not included in the mineral resource estimate as the assay data was either absent from the database or the holes were twinned by another drill hole. Moreover, any trench that was only represented by a single sample was further removed from the database.

Digital copies of the drillholeheader, survey, and assays data were provided to Tetra Tech. GMV does not have a detailed lithological database. It is strongly recommended that a lithology database be compiled to better understand, target, and model the stratigraphy and alteration zones present at Mexican Hat. Additionally, collection of structural geological information and measurements from future drilling will assist to interpret structural controls on mineralization and possible offset faulting along mineralized trends.

Table 14-2
GMV Database by Drilling

Core	45	52	2,650
RC	38	38	3,372
Rotary	120	120	5,536
Channel (trench)	149	183	1,864
Total	352	393	13,422

Table 14-3
Data Excluded from Mineral Resource Estimate

Hole Removed	Reason
BTM 11-09	Twinned by MH 89-41
MH 11-9	Twinned by MH-89-41 and BTM 11-09
MHC-5	Twin of MH 89-28
MH90-155	No assay data
MH90-156	No assay data
MH90-157	No assay data
MH90-158	No assay data
MH90-159	No assay data
MH90-160	No assay data
MH90-164	No assay data
MHT-108	Single Sample Trench
MHT-109	Single Sample Trench
MHT-11	No assay data
MHT-110	Single Sample Trench
MHT-111	Single Sample Trench
MHT-115	No Assay Data
MHT-133	Single high-grade sample, no shoulders, not appropriate for mineral resource estimation
MHT-2	No assay data

Table 14-3 Data Excluded from Mineral Resource Estimate	
Hole Removed	**Reason**
MHT-38	No assay data
MHT-4	No assay data
MHT-50	No assay data
MHT-51	No assay data
MHT-52	No assay data
MHT-53	No assay data
MHT-54	No assay data
MHT-58	No assay data
MHT-59	No assay data
MHT-6	No assay data
MHT-60	Single Sample trench
MHT-68	No assay data
MHT-72	Single Sample trench
MHT-79	Single Sample trench
MHT-8	No assay data
MHT-80	No assay data
MHT-81	No assay data
MHT-89	No assay data
MHT-9	No assay data
MHT-90	No assay data
MHT-91	No assay data
MHT-93	Single Sample trench
MHT-137	No collar value but has some nice assay data
MH 11-14	Twinned with 2016-11
MHC-13	Twinned with MH-89-4
MH 11-2	Twinned with BTM 11-02
MH 11-1	Twinned with BTM 11-01
MH-89-16	Twinned with MHC-2 (core hole)
MHC-6	Twinned with MH-89-38
MH-89-93	Twinned with MH90-99
MHC-7	Twinned with MH90-98
MHR Collars	Chip samples collected along roads. Very poor quality. It was left in to show trends for creating solids, but excluded from calculations
Kalahari Data	All removed

14.4 MODEL INPUT DATA ANALYSIS

14.4.1 Assays

Various analytical techniques and QAQC protocols were employed by each of the previous operators. 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 previous operator showed that Kalahari's dataset was statistically distinct from the other operators, so it was removed from further consideration within the database.

Overall, the Mexican Hat deposit has been sampled by 13,422 gold assays, of which, 8,224 samples were constrained by one of seven mineralization wireframes to support the current mineral resource. The assays outside of the mineralization wireframes were either below the mineralization cut-off grade or were not able to be assigned to a continuous mineralized zone. Assay intervals were used to define the mineralization wireframes. Any un-sampled intervals were assigned a gold grade of zero. Table 14-4 summarizes basic statistics for the assays captured within the various wireframe domains.

Table 14-4
Mexican Hat Mineral Domain Drillhole Statistics

Zone	Field	No. Of Samples	Minimum	Maximum	Mean	Variance	Standard Deviation
1	Length (m)	69	1.50	22.90	2.32	6.90	2.61
	Au (g/t)	67	0.00	7.02	0.58	1.14	1.07
2	Length (m)	187	1.50	21.40	2.36	4.08	2.02
	Au (g/t)	166	0.00	7.03	0.44	0.82	0.91
3	Length (m)	323	0.50	12.20	2.02	1.78	1.34
	Au (g/t)	290	0.00	13.20	0.43	1.05	1.03
4	Length (m)	735	0.30	45.20	2.02	6.19	2.49
	Au (g/t)	663	0.00	13.54	0.38	1.00	1.00
5	Length (m)	5,389	0.10	126.50	2.54	19.77	4.45
	Au (g/t)	4,602	0.00	30.69	0.38	0.84	0.92
6	Length (m)	1,536	0.20	27.50	1.85	3.90	1.98
	Au (g/t)	1,396	0.00	613.43	1.55	276.98	16.64
7	Length (m)	1,230	0.60	21.40	1.99	1.54	1.24
	Au (g/t)	1,130	0.00	80.23	0.62	9.54	3.09
All Zones	Length (m)	9,373	0.10	126.50	2.30	12.96	3.60
	Au (g/t)	8,224	0.00	613.43	0.61	49.12	7.01

14.4.2 Sample 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. Figure 14-1 presents a histogram of the sample lengths before compositing.

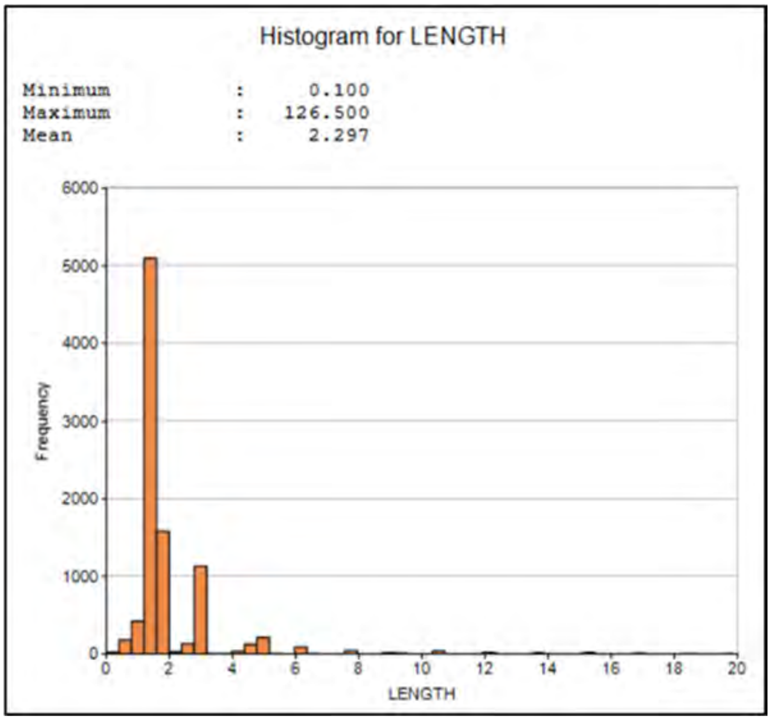


Figure 14-1: Mexican Hat Sample Length Histogram

Table 14-5 summarizes the statistics for the samples after compositing to 1.5 m lengths.

Table 14-5 Mexican Hat Composite Statistics								
Zone	**Field**	**Composite Length**	**No. Of Samples**	**Minimum**	**Maximum**	**Mean**	**Variance**	**Standard Deviation**
1	Length (m)	1.50	105	1.50	1.50	1.50	-	-
	Au (g/t)	1.50	105	0.00	7.02	0.54	1.31	1.14
2	Length (m)	1.50	290	1.50	1.50	1.50	-	-
	Au (g/t)	1.50	290	0.00	7.03	0.30	0.51	0.71
3	Length (m)	1.50	426	1.50	1.50	1.50	-	-
	Au (g/t)	1.50	426	0.00	12.40	0.30	0.66	0.82
4	Length (m)	1.50	978	1.50	1.50	1.50	-	-
	Au (g/t)	1.50	978	0.00	11.78	0.27	0.55	0.74
5	Length (m)	1.50	9,050	1.50	1.50	1.50	-	-
	Au (g/t)	1.50	9,050	0.00	29.76	0.21	0.38	0.62
6	Length (m)	1.50	1,865	1.50	1.50	1.50	-	-
	Au (g/t)	1.50	1,865	0.00	286.28	1.00	52.28	7.23
7	Length (m)	1.50	1,607	1.50	1.50	1.50	-	-
	Au (g/t)	1.50	1,607	0.00	74.95	0.45	4.82	2.19
All Zones	Length (m)	1.50	9,373	1.50	1.50	1.50	-	-

Table 14-5 Mexican Hat Composite Statistics								
Zone	**Field**	**Composite Length**	**No. Of Samples**	**Minimum**	**Maximum**	**Mean**	**Variance**	**Standard Deviation**
	Au (g/t)	1.50	9,373	0.00	286.28	0.35	7.77	2.79

14.4.3 Grade 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 implemented.

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

- Generation of a "high grade" mineralization shell (Zone 6) within a lower grade envelope to provide spatial constraint to interpreted continuous higher grade zone;
- Statistical capping of gold grades for samples which were deemed outliers.

14.4.3.1 Spatial Grade Shells Constraints

Several continuous mineralized zones were interpreted to exist at depth under the topographic promontory referred to as Mexican Hat. These zones are identified with elevated gold grades relative to the surrounding lower grade mineralization halo. To constrain the influence of this higher-grade material, a 1.0 g/t gold grade shell was generated, which was then 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.

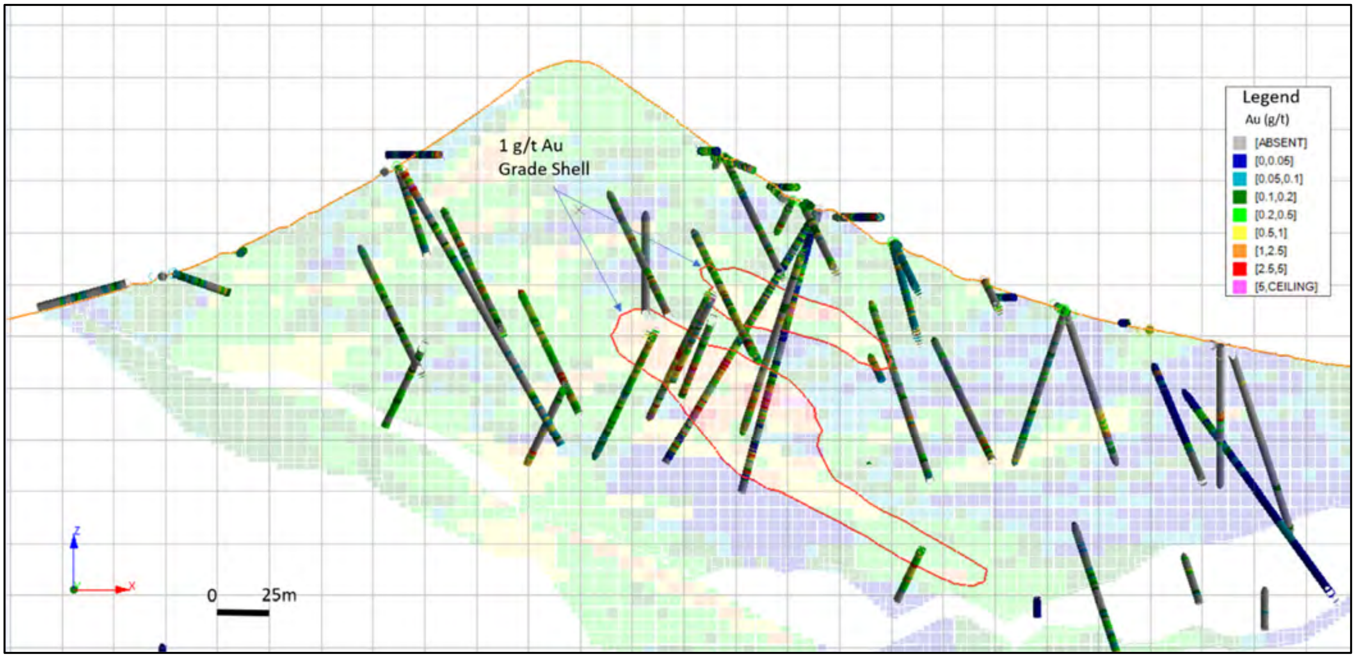


Figure 14-2: Mexican Hat Grade Shells

14.4.3.1 Capping of Gold Grade

Composite assay data for the lower grade mineralized wireframe domains (zones 1-5, and zone 7) were examined separately from the higher grade zone 6 domain in order to analyze the amount of metal that is at risk from high grade assays. This was completed by generating log histogram and log probability plots for the two domains, along with conducting quantile analysis.

The quantile function examines grade distribution based on the Parrish analysis method (Parrish, 1997). The Parrish analysis sets the following criteria to determine if capping is warranted.

- If the top decile has more than 40% of the total metal content;
- If the top decile has more than twice the metal content of the previous decile;
- If the top percentile has greater than 10% of the total metal content; and
- If the top decile has less than a full complement of samples.

If any of the above criteria outlined above by Parrish (1997) are met, then capping of grades may be required. The subsections below summarize the findings of the capping analysis on the two domains.

Zones 1-5, and 7 Capping Analysis

Based on a visual review of the spatial distribution of higher grade gold grade within the project, a the 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 within the 0.2 g/t Au grade domain zones. While the log histogram plot (Figure 14-3) shows a relatively log normal distribution of gold grades with a slight negative skew, and the log probability plot shows a linear distribution, the Parrish analysis (Table 14-6) indicates 50% of the total metal content is present in the top decile. A visual review of the spatial distribution of higher-grade mineralization within each zone did not identify any clustering of high grade. Therefore, the Parrish analysis indicates that capping is warranted in order lower the total metal content of the top decile to below 40%. This was achieved by establishing a cap of 32 g/t gold for zones 1-5 and zone 7 as the high grade encountered for these holes did display clustering.

One sample was capped within the 0.2 g/t gold domain zones (Table 14-8).

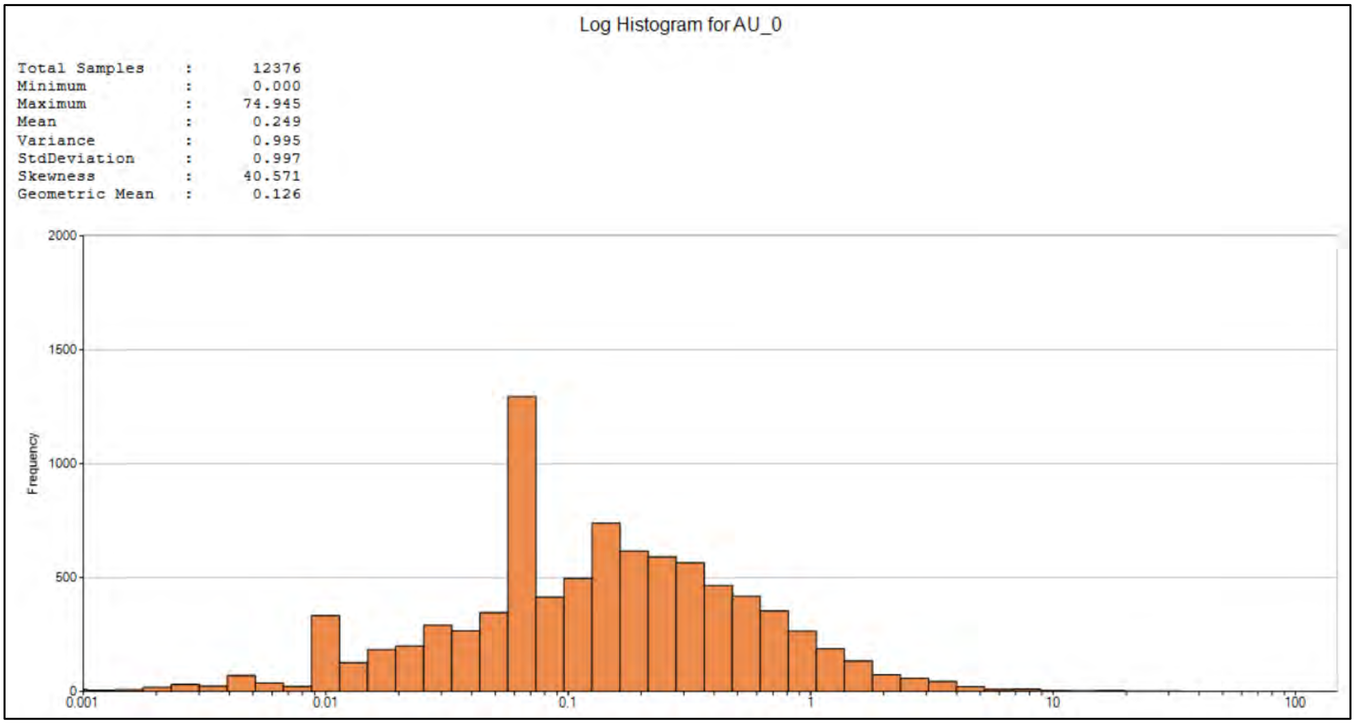


Figure 14-3: Log Histogram of Gold Assays (clustered) for Zones 1-5 and 7

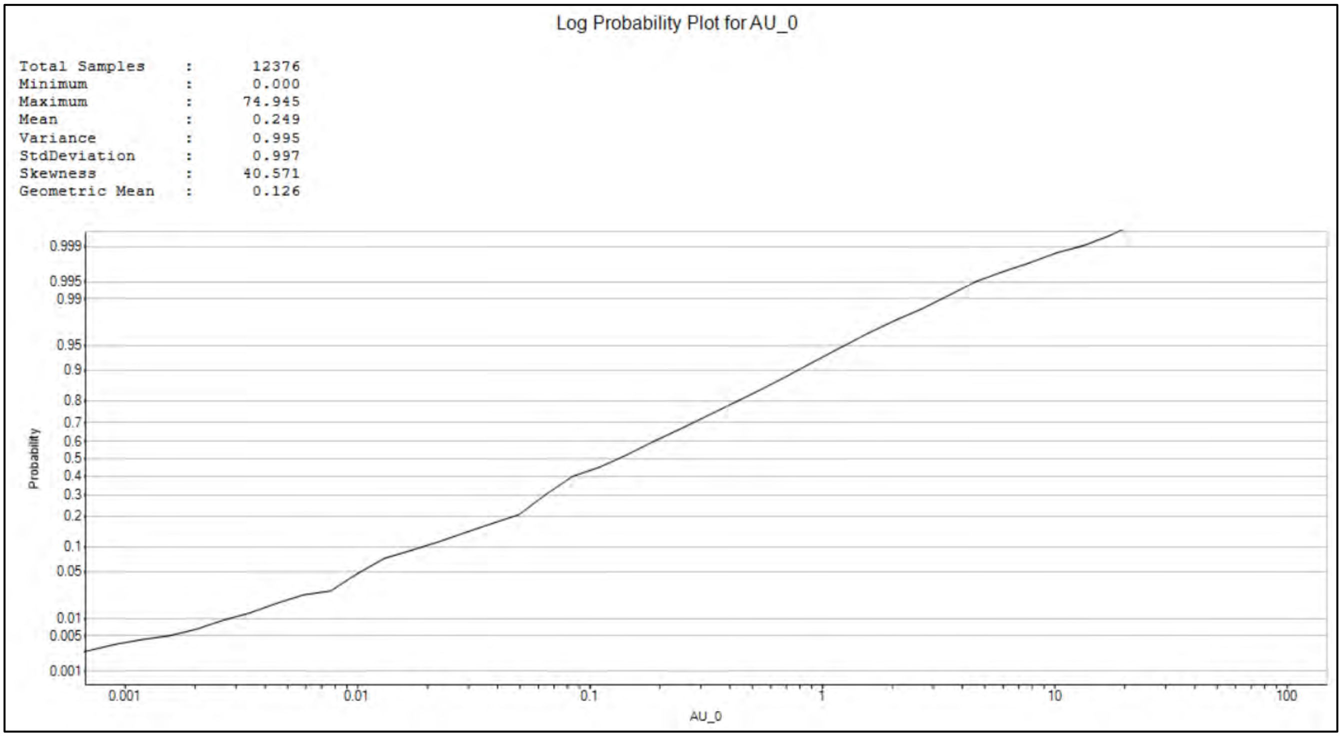


Figure 14-4: Log Probability Plot of Gold Assays (clustered) for Zones 1-5 and 7

Table 14-6
Parrish Decile Analysis for Capping of Gold Grades for Zones 1-5 and 7

Q%_FROM	Q%_TO	NSAMPLES	MEAN	MINIMUM	MAXIMUM	METAL	METAL%
0	10	698	0.06	0.05	0.07	43.60	1.43
10	20	699	0.07	0.07	0.07	48.93	1.61
20	30	699	0.09	0.07	0.11	60.04	1.98
30	40	699	0.13	0.11	0.14	87.94	2.89
40	50	699	0.16	0.14	0.20	114.97	3.78
50	60	699	0.23	0.20	0.26	157.72	5.19
60	70	699	0.31	0.26	0.36	213.41	7.02
70	80	699	0.43	0.36	0.54	303.76	10.00
80	90	699	0.69	0.54	0.91	480.31	15.81
90	100	699	2.19	0.91	74.95	1528.11	50.29
90	91	69	0.94	0.91	0.97	65.14	2.14
91	92	70	1.01	0.97	1.04	70.89	2.33
92	93	70	1.08	1.05	1.12	75.66	2.49
93	94	70	1.19	1.12	1.24	83.24	2.74
94	95	70	1.33	1.25	1.40	93.30	3.07
95	96	70	1.49	1.40	1.61	104.65	3.44
96	97	70	1.73	1.61	1.91	121.40	4.00
97	98	70	2.17	1.92	2.47	151.74	4.99

Table 14-6 Parrish Decile Analysis for Capping of Gold Grades for Zones 1-5 and 7							
Q%_FROM	Q%_TO	NSAMPLES	MEAN	MINIMUM	MAXIMUM	METAL	METAL%
98	99	70	2.97	2.49	3.51	208.03	6.85
99	100	70	7.92	3.53	74.95	554.07	18.23
0	100	6989	0.43	0.05	74.95	3038.79	100.00

Zones 6 Capping Analysis

Based on a review of probability (Figure 14-6) and histogram plots (Figure 14-5) of the composited assays, it was concluded that there was reasonable justification for capping of the Au values within the higher-grade zone 6 domain.

A review of the log histogram (Figure 14-5) shows a bimodal distribution of gold grades with a slight negative skew suggesting that the zone has captured some additional lower grade material that is otherwise below the 1.0 g/t Au domain threshold. A review of the log probability plot (Figure 14-6) shows a relatively linear grade distribution up to approximately 50 g/t Au, at which point a noticeable deviation in the grade distribution is observed indicating capping of the zone may be necessary. To establish an appropriate capping grade, zone 6 was further scrutinized via Parrish analysis (Table 14-7).

The results of the Parrish and log probability analysis indicated that a capping value of 50 g/t Au was appropriate for zone 6. At a 50 g/t Au capping grade, 3 assays with a mean grade of 144.88 g/t Au were capped (Table 14-8).

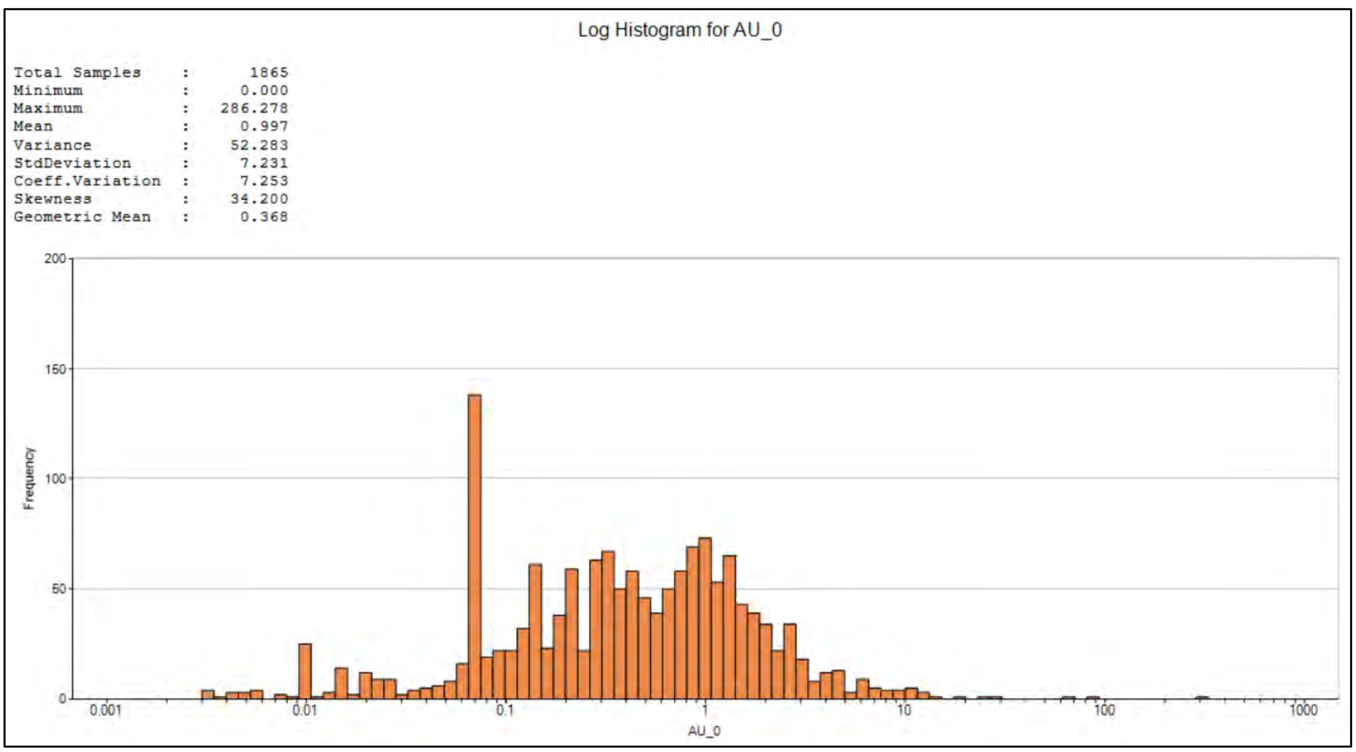


Figure 14-5: Log Histogram of Gold Assays (clustered) for Zone 6

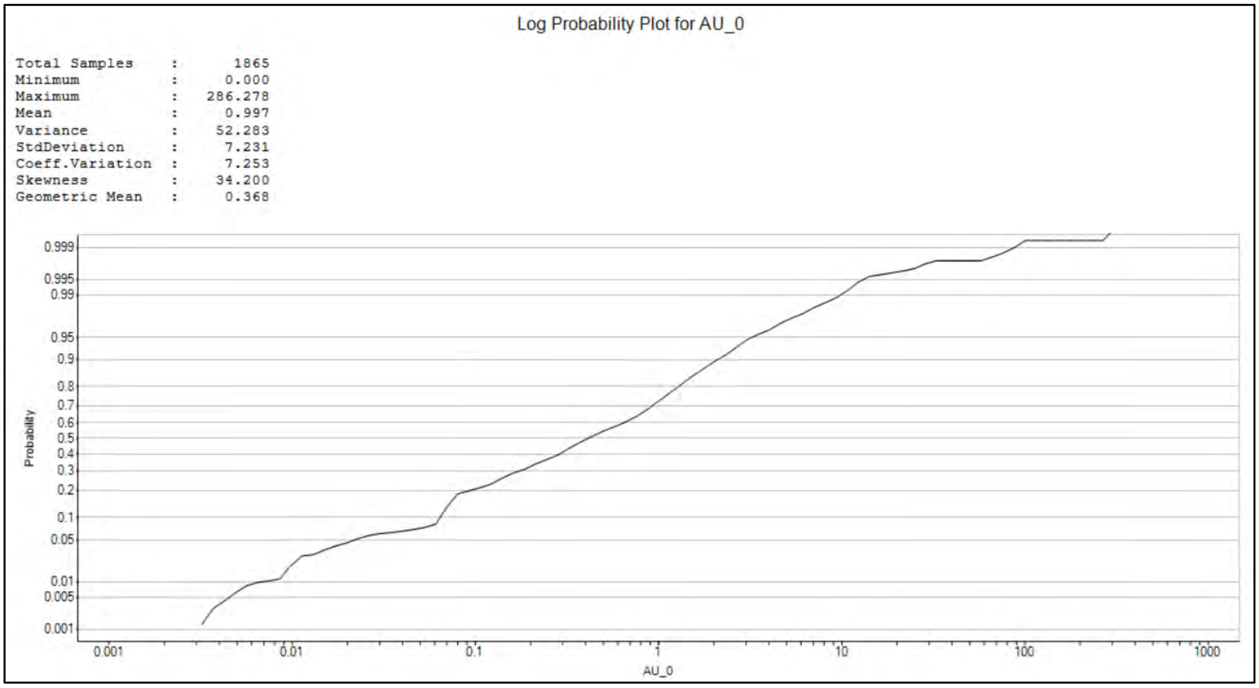


Figure 14-6: Log Probability Plot of Gold Assays (clustered) for Zone 6

Table 14-7
Parrish Decile Analysis for Capping of Gold Grades for Zone 6

Q%_FROM	Q%_TO	NSAMPLES	MEAN	MINIMUM	MAXIMUM	METAL	METAL%
0	10	141	0.07	0.05	0.07	9.55	0.51
10	20	141	0.11	0.07	0.14	14.87	0.80
20	30	142	0.18	0.14	0.22	25.50	1.37
30	40	141	0.28	0.22	0.34	39.56	2.13
40	50	142	0.40	0.34	0.48	56.23	3.03
50	60	141	0.59	0.48	0.73	83.73	4.51
60	70	141	0.85	0.73	0.97	119.69	6.44
70	80	142	1.15	0.97	1.36	163.27	8.79
80	90	141	1.68	1.37	2.19	237.49	12.79
90	100	142	7.80	2.19	286.28	1107.34	59.62
90	91	14	2.29	2.19	2.45	32.01	1.72
91	92	14	2.50	2.45	2.58	35.03	1.89
92	93	14	2.66	2.59	2.73	37.29	2.01
93	94	14	2.80	2.74	2.91	39.17	2.11
94	95	15	3.08	2.91	3.44	46.20	2.49
95	96	14	3.81	3.53	4.09	53.34	2.87
96	97	14	4.43	4.11	4.69	61.97	3.34
97	98	14	5.65	4.83	6.37	79.15	4.26
98	99	14	7.94	6.37	9.37	111.17	5.99

Table 14-7 Parrish Decile Analysis for Capping of Gold Grades for Zone 6							
Q%_FROM	Q%_TO	NSAMPLES	MEAN	MINIMUM	MAXIMUM	METAL	METAL%
99	100	15	40.80	10.16	286.28	612.01	32.95
0	100	1414	1.31	0.05	286.28	1857.23	100.00

Table 14-8 Post Capping Stats										
Zone	Field	Composite Length	No. Of Samples	Samples Capped	Average Capped Grade	Minimum	Maximum	Mean	Variance	Standard Deviation
1	Au (g/t)	1.5	76	0	-	0	7.02	0.52	1.67	1.29
2	Au (g/t)	1.5	239	0	-	0	7.03	0.25	0.47	0.68
3	Au (g/t)	1.5	413	0	-	0	12.40	0.30	0.67	0.82
4	Au (g/t)	1.5	966	0	-	0	11.78	0.27	0.55	0.74
5	Au (g/t)	1.5	9,044	0	-	0	29.76	0.21	0.38	0.62
6	Au (g/t)	1.5	1,886	3	144.88	0	50.00	0.84	6.54	2.55
7	Au (g/t)	1.5	1,632	1	74.95	0	32.00	0.42	1.95	1.40

14.4.4 Bulk Density

Bulk density (BD) data for the Mexican Hat property were collected from various lithology types within the 2017 GMV drill core and sent to ALS laboratories in North Vancouver for testing. BD 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 and mitigates water absorption into the sample's pores. The calculation used for the water displacement wax method is presented below:

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

Compiled BD values are shown in Table 14-9.

Table 14-9 Average BD 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 a method used to quantify the degree of variation two samples are expected to exhibit based upon varying spatial separations and direction. In general, it can be expected that samples located closer together will have less grade variation than samples collected at a greater distance. Conversely, the correlogram statistically measures a 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. Low sample density and/or wide spacing within the mineralized domains precluded reliable variographic assessment across the deposit; based upon the results of the assessment it was not possible to obtain meaningful variograms on the composited dataset.

14.5 LOCAL GRADE VARIABILITY

Local Grade Variability in reported gold mineralization at Mexican Hat is observed from twin holes and resampling programs. The variability is attributed primarily to the sampling stage where limitations and material from recovery methods are associated with RC drilling, and secondly to a nugget effect from gold grain size and/or distribution in clay associated mineralization within fracture network 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 14-10 below summarizes search distances and rotations for estimating a block as well as minimum and maximum number of composites required. Search distances were set to allow coverage greater than average sample spacing throughout the respective domains.

Table 14-10
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

Table 14-10 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**
		5	37.5	37.5	22.5	337	35	0	8	16	4
		6	35	35	15	30	20	0	8	16	4
		7	50	50	25	37	40	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	75	75	37.5	37	40	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	100	100	50	37	40	0	4	16	4

14.6 MODEL DEVELOPMENT

14.6.1 Geological Interpretation

A total of seven three-dimensional (3D) wireframe models of mineralization were modeled in Datamine. 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.

All mineralization has been interpreted to be oxidized with no apparent hypogene, or sulfide dominant, mineralization observed in drilling completed to date. Locally, some supergene enrichment may be present along structures or favorable horizons, such as the zone 6 mineralized domain. Mineralogical study is required to confirm this observation.

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

Zone 7 (Figure 14-7), oriented at approximately -40/310 (TN azimuth), represent a mineralized fault which is interpreted as a major "feeder" conduit for mineralizing hydrothermal fluids. All other zones were truncated to Zone 7 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 14-7). Encompassed within this zone is a structurally controlled high-grade gold core (zone 6). This relatively flat laying higher grade core was modeled using a minimum grade of 1.0 g/t Au over a minimum of two composited samples per drill hole.

To the south, mineralization occurs within 4 discreet, structurally hosted corridors, which are up to 20 m in width and display an average orientation of -35/245 (TN azimuth) (zones 1-4, Figure 14-7). The angle between the Zone 7 fault and the southern mineralized structures is approximately 50 degrees.

Interpretations were made in Datamine on a series of vertical cross sections and longitudinal cross sections. These interpretations were linked with tag strings and triangulated to build 3D solids. Table 14-11 tabulates the solids and their associated volumes. The solids were validated in Datamine software and no errors were found.

An overburden model was not developed for the geological model. Outcrop predominates the main Mexican Hat promontory. Insufficient data exists in historical logs to estimate the depth of colluvium and soils surrounding this topographic feature.

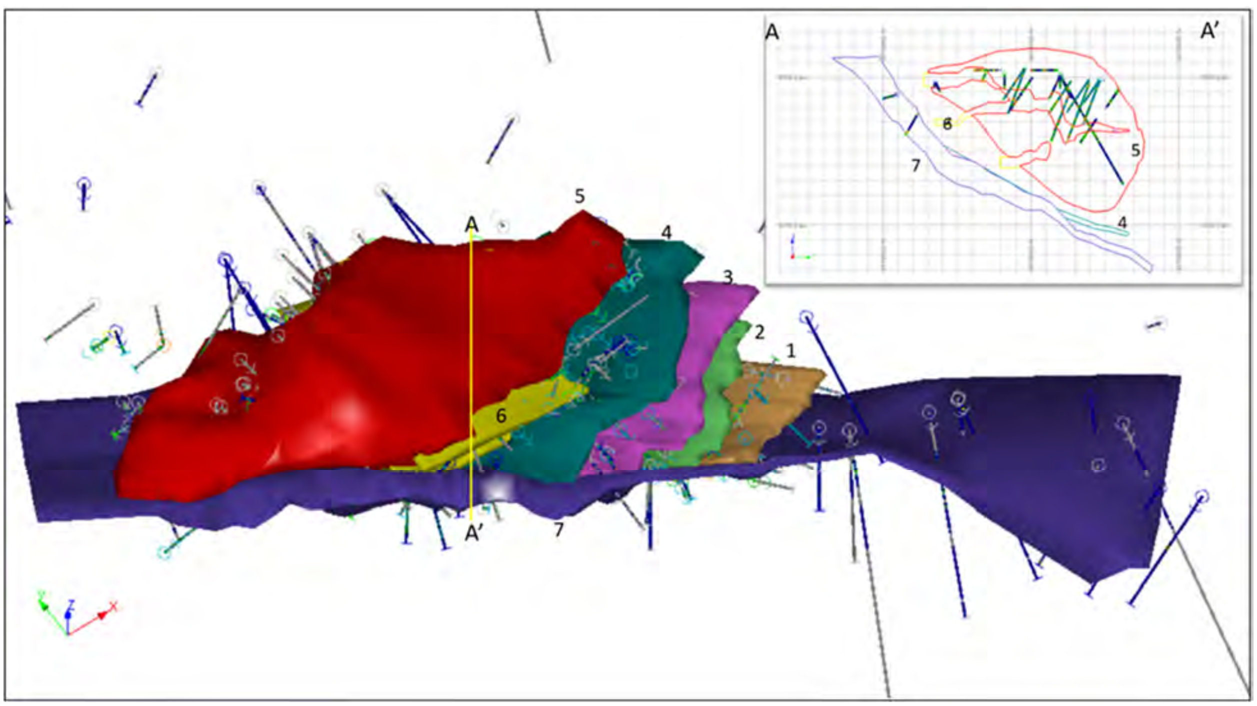


Figure 14-7: Oblique View of Mexican Hat Mineralized Domains, Looking Down to Northeast

Table 14-11 Mexican Hat Wireframe Volumes							
Zone	Minimum X	Maximum X	Minimum Y	Maximum Y	Minimum Z	Maximum Z	Volume (m3)
1	613,068.43	613,350.14	3,518,839.83	3,519,298.46	1,205.79	1,497.36	824,810.13
2	613,016.71	613,309.22	3,518,875.58	3,519,321.70	1,211.72	1,506.99	1,221,077.61
3	612,894.75	613,344.43	3,518,902.74	3,519,453.34	1,195.31	1,515.15	1,842,204.29
4	612,846.13	613,327.42	3,518,947.76	3,519,440.83	1,220.34	1,523.31	2,036,961.26
5	612,468.98	613,265.62	3,519,074.64	3,519,472.43	1,269.00	1,599.77	23,365,753.30
6	612,773.63	613,083.24	3,519,069.30	3,519,415.99	1,315.14	1,523.97	1,949,716.74
7	612,345.22	613,817.14	3,518,403.98	3,519,591.15	1,158.96	1,542.37	15,067,177.24

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 14-12.

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 could 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 Zone 7 allowed sub celling in the Y-Z direction, along with a variable width in the X direction. This allowed the blocks to fill the volume of the wireframes more accurately. Estimation on each block was completed using the parent block centroids and the grades assigned to the sub-cell blocks.

Occasionally, the trenching data did not align with the provided topography, with trenches occasionally plotting up to 5 metres 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 over top 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 14-8.

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

Table 14-12 Mexican Hat Parent Model Parameters								
Origin			Cell Size			Number of Cells		
X Origin	Y Origin	Z Origin	XINC	YINC	ZINC	NX	NY	NZ
612,255	3,518,310	1,144	6	6	6	291	265	76

Table 14-13 summarizes the Wireframe Volumes vs. Block Volumes.

Table 14-13 Mexican Hat Wireframe Volumes vs. Block Volumes				
Zone	**Wireframe Volume (m3)**	**Unfilled Block Volume (m3)**	**Volume Difference (m3)**	**Relative Difference (%)**
1	824,810.13	824,865	54.87	0.01
2	1,221,077.61	1,221,300	222.39	0.02
3	1,842,204.29	1,900,821	58616.71	3.13
4	2,036,961.26	2,036,444	-517.26	-0.03
5	23,365,753.30	23,366,754	1000.70	0.00
6	1,949,716.74	19,52780	3063.26	0.16
7	15,067,177.24	15,067,194	16.76	0.00

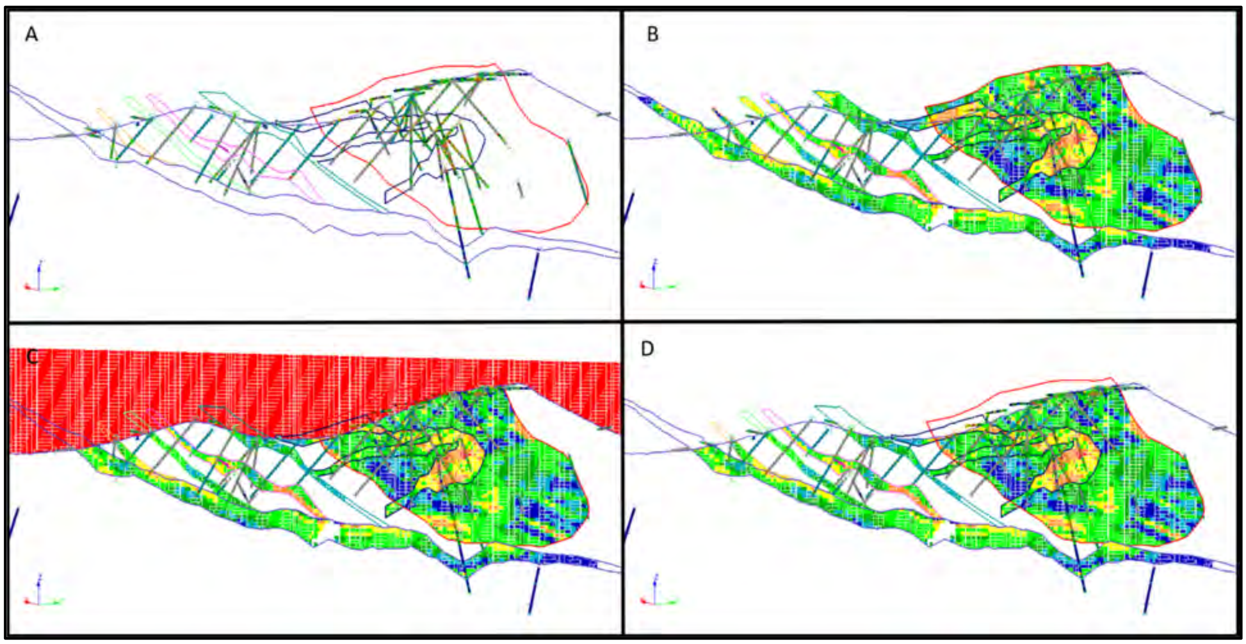


Figure 14-8: Process of Allowing Trenching Data to be Captured for Estimation and Removing any above Surface Mineralization. A: Section showing Trenches Plotted above Topography (orange line). B: Grade Filled Zones, with Blocks Extending Above Topography. C: Addition / Superimposing of "Air" blocks (Red). D: Removal of any Blocks Above Topography, Leaving Final Block Model

14.7 MINERAL RESOURCE STATEMENT

14.7.1 Classification

Classification of the Mineral Resource Estimate 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, based on:

- 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 through review of samples or analytical certificates,
- 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 applied a Lerchs Grossman pit optimizer algorithm and reasonable mining assumptions to evaluate the proportions of the block models that could be "reasonably expected" to be mined from an open pit (Table 14-14). 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.

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 14-9 shows a cross section of a designated Whittle shell within which the resources have been reported for the Mexican Hat deposit.

Table 14-15 represents the inferred resources for the Mexican Hat Project within the optimized pit shell at 0.2 g/t gold cut-off.

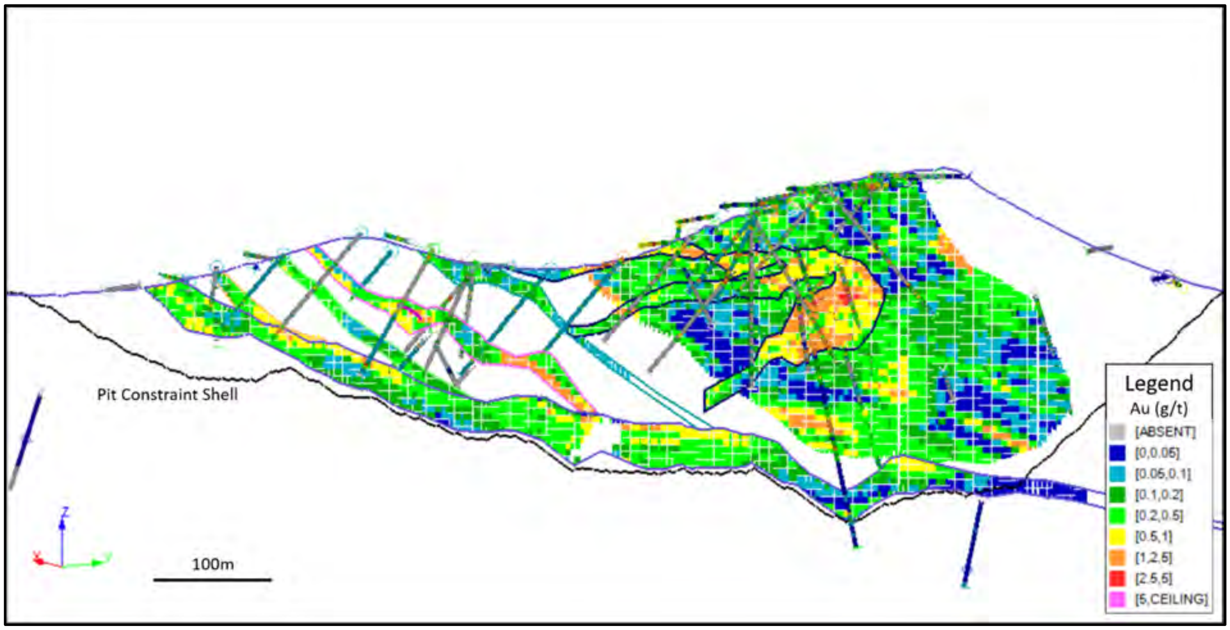


Figure 14-9: Mexican Hat Cross-section of the Pit Shell and Estimated Gold Block Grades

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 Datamine's NPVS pit optimization software with mining parameter 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 14-14).

Table 14-14
Datamine NPVSTM Optimization Parameters for Resource Estimation Constraint

NPVS 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
Increased cost of mining per bench below 1480.05 masl	$0.03
Gold Price ($/oz)	$1,375
Assumed Pit Wall Overall Slope Angle	45 degrees
Metallurgical Recovery of Gold	88.2%
Total Mining Limit (t/year)	25,000,000
Discount Rate (%)	5%

Table 14-15 Mineral Resource Statement, Mexican Hat Project, Arizona, USA, Tetra Tech Canada, Effective Date June 22, 2020					
Category	**Cut-off (g/t Au)**	**Grade (Au, g/t)**	**Tonnes**	**Gold Oz**	**Strip Ratio**
Inferred	0.20	0.58	36,733,000	688,000	2.36

- 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 = $1,375 per troy ounce, throughput at 15,000 tpd, discount rate = 5%. A cost of $0.03 was added per bench to the mining cost below the existing level surface.
- A top cut of 32 gpt gold is applied to all zones except Zone 6 which has a top cut of 50 gpt gold.
- Mineral Resources have been calculated using the Inverse Distance Squared method
- 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.
- The stated strip ratio of 2.36 in Table 14-15 represents the ratio of tonnes of gold resources (36.733 M t) above the gold cut-off grade (0.20 g/t Au) to the estimated tonnes of waste material below the cut-off grade. This ratio does not represent the inferred gold resources to waste that are calculated in the mine plan for extracting the gold materials for processing and producing gold at the stated mining parameters.

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 14-10. 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.

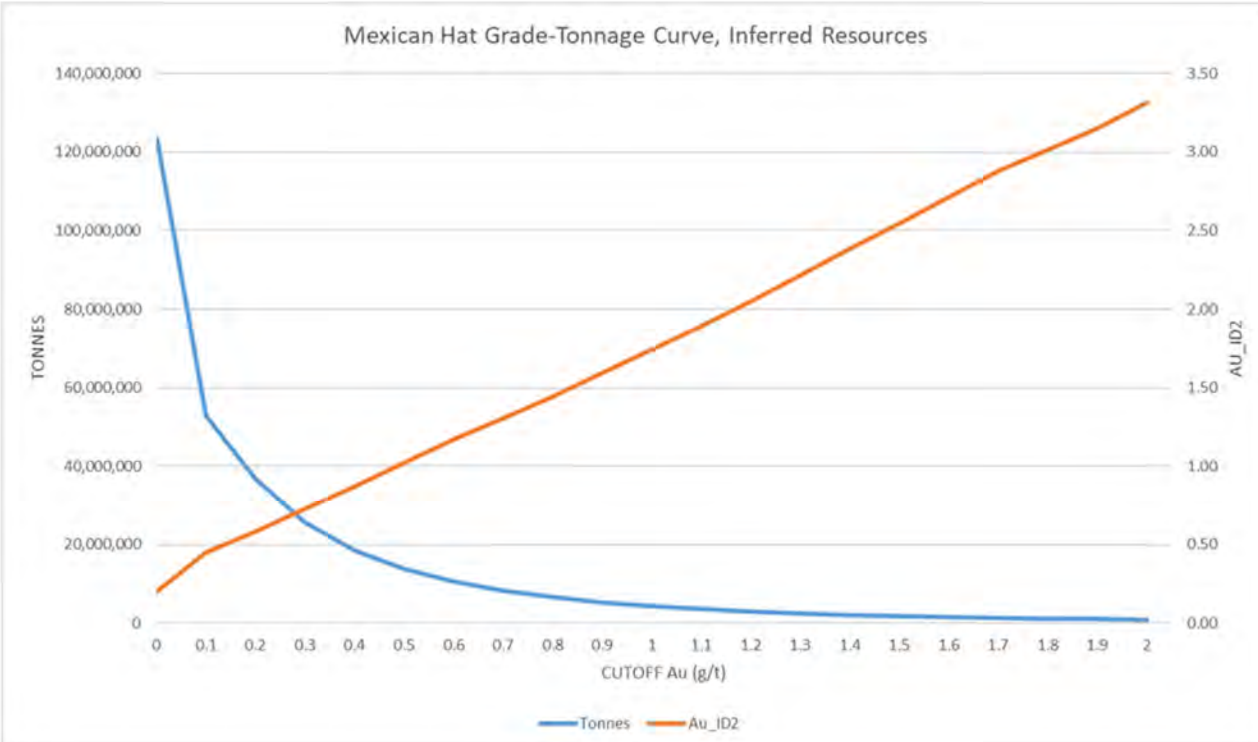


Figure 14-10: Mexican Hat Inferred Category Grade Tonnage Curves

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 ID2 was compared to that of the global ID3 and NN model values as well as the composite drillhole data. Table 14-16 shows this comparison of the global estimates for the three estimation method calculations. In general, there is agreement between the ID2 model, the ID3 model, and the NN model. Larger discrepancies are reflected because 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 14-16
Mexican Hat Comparison by Estimation Method

Estimation Method	Au g/t Cut-off	Tonnes	Au g/t	Contained Ounces	% Difference Total metal
Nearest Neighbour (NN)	0.2	27,579,000	0.84	749,000	8.78%
Inverse Distance Cubed (ID3)	0.2	35,503,000	0.61	694,000	1.16%
Inverse Distance Squared (ID2)	0.2	36,732,000	0.58	686,000	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.

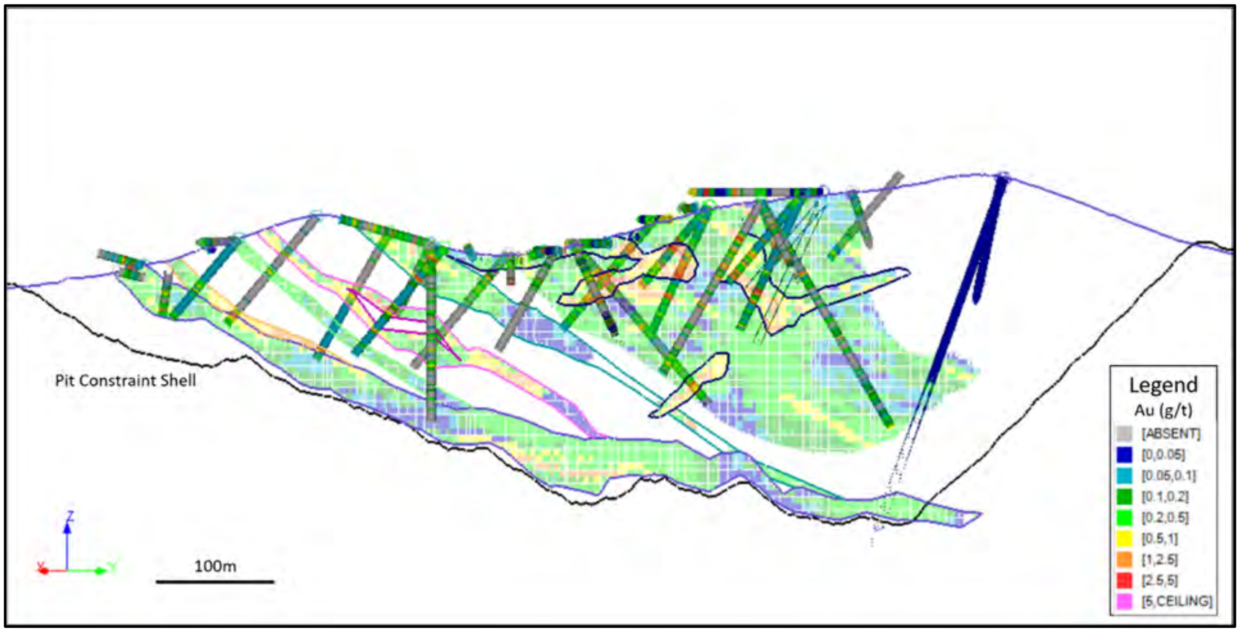


Figure 14-11: Mexican Hat Model, Looking West South West

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 a reasonable spatial estimation for the composite sample grades.

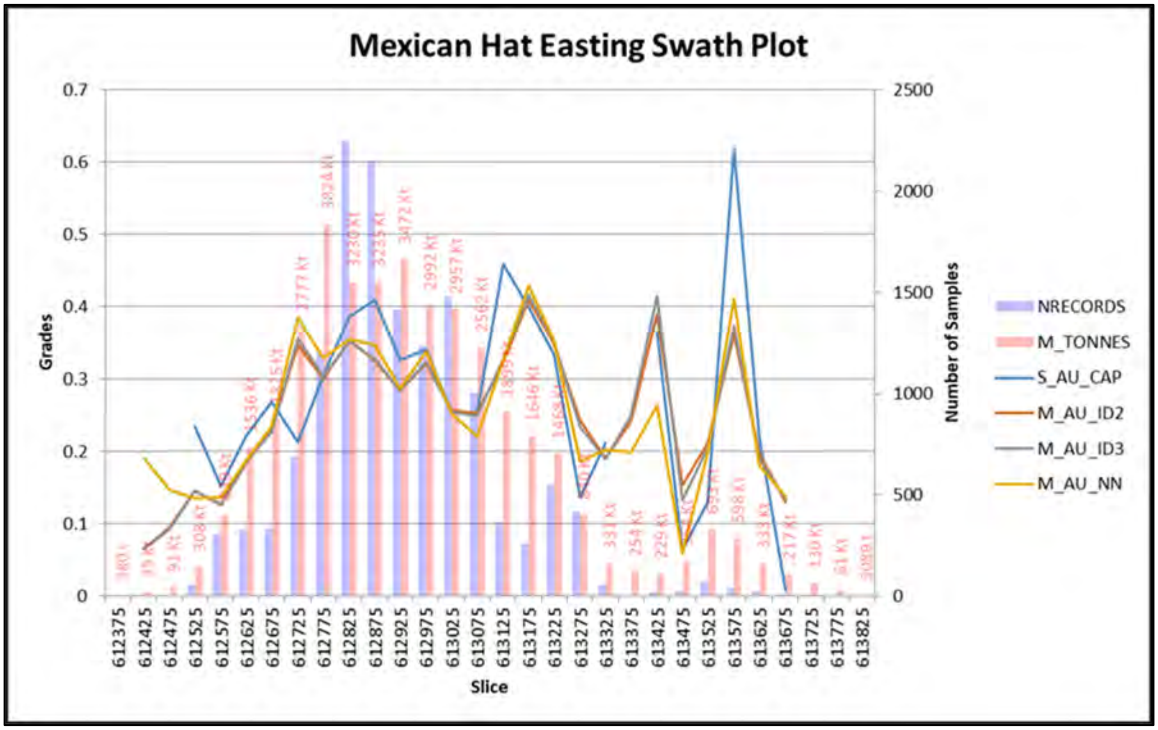


Figure 14-12: Swath Plot along Northings for the Mexican Hat Model

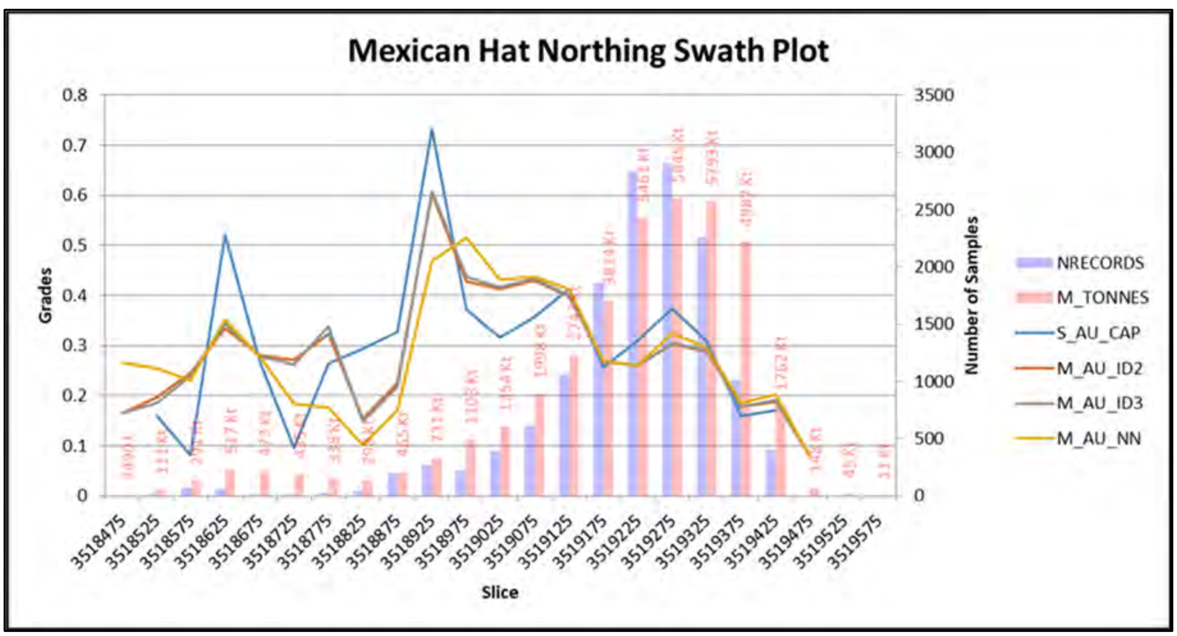


Figure 14-13: Swath Plot along Eastings for the Mexican Hat model

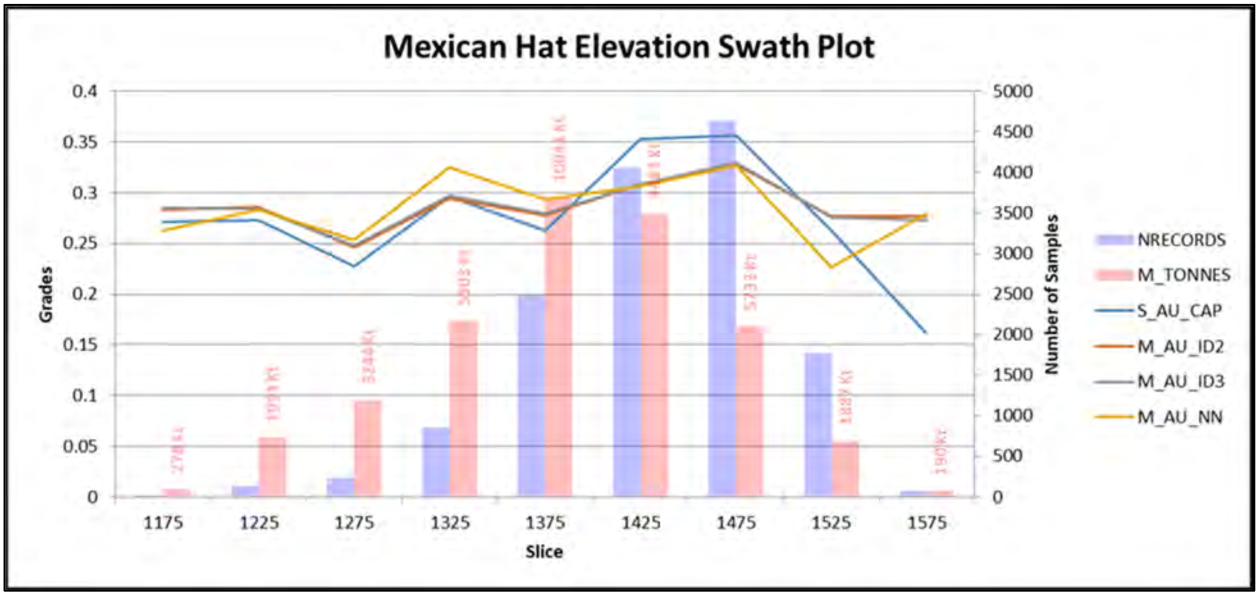


Figure 14-14: Swath Plot along Elevations for the Mexican Hat model

15.0 MINERAL RESERVE ESTIMATES

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

16.0 MINING METHODS

The PEA presented in this report considers open-pit mining of the Mexican Hat gold deposit. Note that a PEA is preliminary in nature and includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied that would enable them to be classified as mineral reserves. There is no certainty that the economic results of the PEA will be realized.

The methodology used for mine planning to define the economics for the PEA includes:

- Define assumptions for the economic parameters;
- Define geometric parameters and constraints;
- Run pit optimizations;
- Define road and ramp parameters;
- Create pit designs;
- Create dump designs;
- Produce mine and process production schedules;
- Define personnel and equipment requirements;
- Estimate mining costs; and
- Perform an economic analysis.

Section 16.0 summarizes the above topics, except for the mining cost estimates which are discussed in Section 21.0, and the economic analysis discussed in Section 22.

16.1 ECONOMIC PARAMETERS

Economic parameters were used to generate optimized pits using a Lerches Grossman algorithm within Whittle™ software (Version 4.7). The economic parameters include mining costs, process cost, general and administrative costs ("G&A"), refining costs, royalties, and metal recoveries. Mine planning is an iterative process, and initial costs and recoveries were assumed to determine how large pits would be. The economic parameters were developed based on previous experience with processing by crushing and leaching of oxide materials.

The economic parameters used are shown in Table 16-1. The overall process rate is assumed to be 3,500,000 tonnes per year. This assumption is only used to convert the fixed G&A component to a cost per tonne for the purpose of pit optimization. The G&A cost is later applied as a fixed cost in the cash-flow model.

Table 16-1: Economic Parameters for Pit Optimizations

	Base Case	Units
Mining Cost - OP	$ 2.68	$/tonne Mined
Processing Cost	$ 5.15	$/tonne Processed
G&A Cost	$ 2,730	K USD/yr
G&A Cost	$ 0.78	$/tonne Processed
Throughput	10,000	TPD
Throughput	3,500	K TPY
Refining Cost	$ 5.00	$/oz Processed
Recovery - Au	88%	
Payable - Au	100%	
Royalties - Hernandez	1.5%	
Royalties - Victor	0%	
Gold Price	$ 1,500	$/oz Au

16.2 CUT-OFF GRADES

Cut-off grades were calculated based on the economic parameters shown in Table 16-1 and are tabulated in Table 16-2 using a range of gold prices. Both internal and external break-even cut-off grades were calculated. The external cut-off grade uses all the operating costs shown in the economic parameters. The internal cut-off grade calculation eliminates the mining cost in the calculation, which is used for the determination of material to be processed. The pit designs are based on economical pits and the materials inside of the pits are assumed to be mined whether the material is waste or mineralized. Thus, the decision on whether to process the material is made at the point where the truck needs to turn either to the waste dump or the process facility. Thus, the mining cost is a sunk cost. The basic equation for the cut-off grade calculation is shown in Equation 1.

Equation 1 Breakeven Cut-off Grade Calculation (gAu/tonne)

$$\frac{Costs}{\left(\frac{Au\$}{oz} - RefCst\right)/31.10348 * (1 - Roy\%) * Rec\%}$$

Where costs are all processing costs plus G&A costs in $/tonne, plus the mining costs for the external cut-off grade calculation, RefCst is the refining cost in $/oz gold produced, Roy% is the NSR royalty, and Rec% is the calculated recovery at the cut-off grade.

Table 16-2: Cut-off Grades

Au Price ($/oz Au)	COG (g Au/tonne) Hernandez Royalty Internal	Hernandez Royalty External	Other Claims Internal	Other Claims External
$ 1,000	0.21	0.31	0.21	0.31
$ 1,100	0.19	0.28	0.19	0.28
$ 1,200	0.18	0.26	0.18	0.25
$ 1,300	0.16	0.24	0.16	0.23
$ 1,400	0.15	0.22	0.15	0.22
$ 1,500	0.14	0.21	0.14	0.20
$ 1,600	0.13	0.19	0.13	0.19
$ 1,700	0.13	0.18	0.12	0.18
$ 1,800	0.12	0.17	0.12	0.17
$ 1,900	0.11	0.16	0.11	0.16
$ 2,000	0.11	0.15	0.11	0.15

16.3 GEOMETRIC PARAMETERS

Geometric parameters include slope parameters which are shown in Table 16-3. The design slope parameters use an 18 m height between catch benches separated by three benches each of 6.0 m height, a 66° bench face angle, and 10 m catch benches or berms. This provides for a 45° inner-ramp angle. Table 16-3 also shows the design parameters used for waste dumps. The waste dump design includes 12 m lift heights with 12 m catch benches and a 34° slope between benches to achieve an overall slope of 2.5 horizontal to 1.0 vertical.

Table 16-3: Slope Parameters

Parameter	Units	Pit Designs	3:1 Dumps
Bench Height	m	18	12
Inner-Ramp Angle	deg	45	21.8
Bench Face Angle	deg	66	34
Berm Width	m	10	12

For pit optimizations, the slopes were flattened to account for ramp placement in the designs. The final design was compared with the Whittle pits used as a guide for the designs, and the pits compared well.

No land constraints were used during the pit optimization process.

16.4 PIT OPTIMIZATIONS

Pit optimizations were run using Whittle™ software (version 4.7). Inputs into Whittle included the resource block model along with the economic and geometric parameters previously discussed. Ultimate pit shells were selected from the Whittle results for final design.

The selections of ultimate pits and pit phases were done as a two-step process. The first step was to optimize a set of pit shells based on varying a revenue factor. This was done in Whittle using a Lerchs-Grossman algorithm. The revenue factor was multiplied by the recovered ounces and the metal prices, essentially creating a nested set of pit shells based on different metal prices. Revenue factors for the deposit were varied from 0.30 to 2.0 in increments of 0.025 with a base price of $1,000 per ounce of gold, so the resulting pit shells represent gold prices from $300 to $2,000 per ounce in increments of $25. This has the potential of generating up to 69 different pit shells that can be used for analysis. The resulting pit number one will be the first pit that optimizes, so if a pit is not viable until a given revenue factor, that will become the first pit. In addition, in some cases pit shells with increments are coincidental to other pits and reported as a single pit. Thus, the number of pits may vary for each deposit and run.

The second step of the process was to use the Pit by Pit ("PbP") analysis tool in Whittle to generate a discounted operating cash flow (note that capital is not included). These were done using a constant gold price of $1,500 per ounce. The PbP node uses a rough scheduling by pit phase for each pit shell to generate the discounted value for the pit. The program develops three different discounted values: best, worst, and specified. The best-case value uses each of the pit shells as pit phases or pushbacks. For example, when evaluating pit 20, there would be 19 pushbacks mined prior to pit 20, and the resulting schedule takes advantage of mining more valuable material up front to improve the discounted value. Evaluating pit 21 would have 20 pushbacks; pit 22 would have 21 pushbacks and so on. Note that this is not a realistic case as the incremental pushbacks would not have enough mining width between them to be able to mine appropriately, but this does help to define the maximum potential discounted operating cash flow.

The worst case does not use any pushbacks in determining the discounted value for each of the pit shells. Thus, each pit shell is evaluated as if mining a single pit from top to bottom. This does not provide the advantage of mining more valuable material first, so it generally provides a lower discounted value than that of the best case.

The specified case allows the user to specify pit shells to be used as pushbacks and then schedules the pushbacks and calculates the discounted cash flow. This is more realistic than the base case as it allows for more mining width, though the final pit design will have to ensure that appropriate mining width is available. The specified case has been used to determine the ultimate pit limits to design to, as well as to specify guidelines for designing pit phases.

The previously discussed parameters were used along with gold prices varying from $300 to $2,000 per ounce to create the pit optimization results. These results are shown in Table 16-4 at $100 gold price increments with the $1,500 pit shell highlighted as the base-case gold price used for the PEA study.

Table 16-5 shows the PbP results and these are also shown graphically in Figure 16-1. Pit 49 is highlighted as having the best discounted operating cash flow for the specified case. However, Pit 45 was chosen as the basis for the pit design to maximize the overall grade and net present value ("NPV").

Table 16-4: Mexican Hat Pit Optimization Results

		Material Processed			Waste	Total	Strip
Pit	Gold Price	K Tonnes	g Au/t	K Ozs Au	Tonnes	Tonnes	Ratio
1	$ 300	63	2.87	6	68	131	1.09
5	$ 400	170	2.06	11	240	410	1.41
9	$ 500	410	1.91	25	1,241	1,651	3.02
13	$ 600	1,305	1.51	63	3,979	5,284	3.05
17	$ 700	2,579	1.25	104	7,209	9,789	2.79
21	$ 800	5,956	1.05	201	16,791	22,747	2.82
25	$ 900	7,153	0.97	222	17,762	24,916	2.48
29	$ 1,000	10,621	0.90	306	29,597	40,218	2.79
33	$ 1,100	14,191	0.80	367	35,314	49,504	2.49
37	$ 1,200	17,980	0.75	431	43,248	61,228	2.41
41	$ 1,300	23,707	0.68	519	55,128	78,835	2.33
45	$ 1,400	26,068	0.65	546	57,117	83,185	2.19
49	$ 1,500	28,536	0.62	573	59,736	88,272	2.09
53	$ 1,600	30,466	0.60	591	60,447	90,914	1.98
57	$ 1,700	32,466	0.58	609	61,751	94,216	1.90
61	$ 1,800	36,828	0.56	658	71,879	108,707	1.95
65	$ 1,900	37,787	0.55	671	75,391	113,178	2.00
69	$ 2,000	38,518	0.55	681	78,388	116,906	2.04

Table 16-5: Mexican Hat Pit by Pit Results

	Material Processed			Waste	Total	Strip	Disc. Operating CF (M USD)			LOM
Pit	K Tonnes	g Au/t	K Ozs Au	Tonnes	Tonnes	Ratio	Best	Specified	Wosrt	Years
1	73	2.53	6	58	131	0.79	$ 6.75	$ 6.75	$ 6.75	0.02
2	98	2.29	7	93	191	0.95	$ 8.01	$ 8.01	$ 8.01	0.03
3	116	2.12	8	97	213	0.83	$ 8.74	$ 8.74	$ 8.74	0.03
4	180	1.88	11	200	379	1.11	$ 11.50	$ 11.50	$ 11.50	0.05
5	204	1.78	12	206	410	1.01	$ 12.28	$ 12.28	$ 12.28	0.06
6	239	1.65	13	219	457	0.92	$ 13.18	$ 13.18	$ 13.18	0.07
7	467	1.48	22	763	1,230	1.64	$ 21.48	$ 21.48	$ 21.48	0.13
8	530	1.43	24	846	1,376	1.60	$ 23.27	$ 23.27	$ 23.27	0.15
9	635	1.36	28	1,015	1,651	1.60	$ 26.12	$ 26.12	$ 26.12	0.18
10	1,464	1.18	56	2,506	3,970	1.71	$ 48.21	$ 48.21	$ 48.21	0.42
11	1,643	1.16	61	2,858	4,502	1.74	$ 52.49	$ 52.49	$ 52.49	0.47
12	1,720	1.15	63	2,933	4,653	1.70	$ 53.99	$ 53.99	$ 53.99	0.49
13	2,059	1.08	71	3,225	5,284	1.57	$ 59.03	$ 59.03	$ 59.03	0.59
14	2,163	1.06	74	3,321	5,484	1.53	$ 60.70	$ 60.70	$ 60.70	0.62
15	3,531	0.93	105	4,965	8,495	1.41	$ 79.87	$ 79.87	$ 79.87	1.01
16	3,650	0.92	108	5,038	8,688	1.38	$ 81.50	$ 81.49	$ 81.49	1.04
17	4,170	0.89	119	5,618	9,789	1.35	$ 88.77	$ 88.71	$ 88.66	1.19
18	4,266	0.89	122	5,793	10,060	1.36	$ 90.27	$ 90.20	$ 90.13	1.22
19	6,505	0.81	168	8,730	15,235	1.34	$ 115.44	$ 115.16	$ 113.79	1.86
20	9,043	0.77	225	13,089	22,132	1.45	$ 144.65	$ 144.08	$ 141.79	2.58
21	9,356	0.77	231	13,391	22,747	1.43	$ 147.67	$ 147.03	$ 144.64	2.67
22	9,476	0.77	234	13,637	23,113	1.44	$ 148.99	$ 148.31	$ 145.87	2.71
23	9,759	0.76	240	14,157	23,916	1.45	$ 151.76	$ 151.01	$ 148.49	2.79
24	9,991	0.76	245	14,511	24,501	1.45	$ 153.77	$ 152.94	$ 150.33	2.85
25	10,150	0.76	248	14,766	24,916	1.45	$ 155.03	$ 154.15	$ 151.48	2.90
26	10,322	0.76	252	15,242	25,564	1.48	$ 156.62	$ 155.71	$ 152.94	2.95
27	10,916	0.75	264	16,582	27,498	1.52	$ 161.38	$ 160.43	$ 157.45	3.12
28	11,091	0.75	267	16,738	27,829	1.51	$ 162.43	$ 161.47	$ 158.45	3.17
29	13,869	0.74	332	26,349	40,218	1.90	$ 185.12	$ 184.11	$ 179.28	3.96
30	14,151	0.74	337	26,849	40,999	1.90	$ 186.72	$ 185.70	$ 180.65	4.04
31	16,768	0.70	379	30,621	47,389	1.83	$ 199.53	$ 197.60	$ 190.84	4.79
32	16,891	0.70	382	30,865	47,756	1.83	$ 200.14	$ 198.17	$ 191.32	4.83
33	17,446	0.70	391	32,058	49,504	1.84	$ 202.69	$ 200.47	$ 193.27	4.98
34	19,367	0.69	427	37,261	56,627	1.92	$ 211.24	$ 208.03	$ 200.01	5.53
35	19,653	0.68	432	37,740	57,393	1.92	$ 212.23	$ 208.86	$ 200.66	5.62
36	20,702	0.67	449	39,774	60,476	1.92	$ 215.69	$ 211.88	$ 202.75	5.91
37	20,904	0.67	452	40,324	61,228	1.93	$ 216.37	$ 212.46	$ 203.23	5.97
38	22,426	0.66	476	43,442	65,868	1.94	$ 220.42	$ 216.36	$ 205.58	6.41
39	22,632	0.66	479	43,847	66,478	1.94	$ 220.92	$ 216.84	$ 205.84	6.47
40	22,793	0.66	482	44,206	66,998	1.94	$ 221.28	$ 217.19	$ 206.02	6.51
41	26,073	0.64	535	52,762	78,835	2.02	$ 228.03	$ 223.47	$ 209.41	7.45
42	26,278	0.64	538	53,241	79,519	2.03	$ 228.35	$ 223.75	$ 209.53	7.51
43	26,523	0.64	542	53,792	80,315	2.03	$ 228.64	$ 223.98	$ 209.49	7.58
44	26,713	0.63	545	54,475	81,188	2.04	$ 228.88	$ 224.15	$ 209.37	7.63
45	27,274	0.63	554	55,911	83,185	2.05	$ 229.34	$ 224.39	$ 209.00	7.79
46	27,603	0.63	558	56,666	84,268	2.05	$ 229.53	$ 224.46	$ 208.72	7.89
47	28,076	0.63	566	58,331	86,407	2.08	$ 229.72	$ 224.42	$ 208.18	8.02
48	28,403	0.63	571	59,253	87,656	2.09	$ 229.83	$ 224.41	$ 207.92	8.12
49	28,536	0.62	573	59,736	88,272	2.09	$ 229.84	$ 224.37	$ 207.80	8.15
50	28,804	0.62	577	60,515	89,319	2.10	$ 229.81	$ 224.20	$ 207.36	8.23
51	28,913	0.62	578	60,672	89,584	2.10	$ 229.78	$ 224.12	$ 207.18	8.26
52	28,946	0.62	579	60,717	89,663	2.10	$ 229.77	$ 224.10	$ 207.14	8.27
53	29,223	0.62	583	61,691	90,914	2.11	$ 229.55	$ 223.67	$ 206.39	8.35

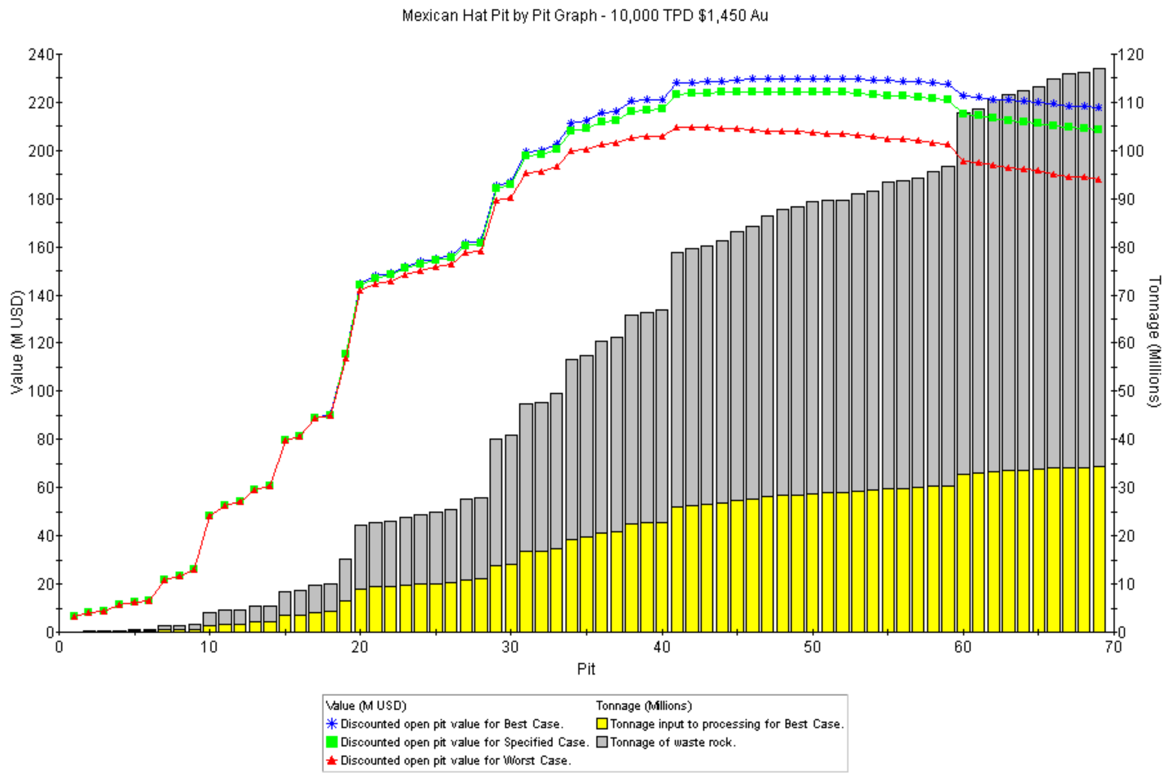


Figure 16-1: Mexican Hat Pit by Pit Graph

16.5 ROADS AND RAMP DESIGN

Road designs have been completed for the PEA to allow primary access for people, equipment, and consumables to the site. This includes haul roads between the designed pits, dumps, and proposed leach facility. Within the pit designs, ramps have been established for haul truck and equipment access. The in-pit ramps will only require a single berm. Ramps outside of the pit will require two safety berms. The design parameters for ramps and roads are shown in Table 16-6. Note that these also show parameters for one-lane traffic. These would be used near the bottom of pits where the strip ratio is minimal, and the traffic requirements are low.

The ramps and haul roads assume the use of CAT-777 91-tonne haul trucks with an operating width of 6.1m. For two-way access, the goal of the road design is to allow a running width of near 3.5 times the width of the trucks. MSHA regulations specify that safety berms be maintained at a height of at least ½ of the diameter of the tires of the haul trucks that will travel on roads. The ½ height of the CAT-777 tonne haul trucks tires are 1.35 m. An extra 10% was added to berm height design to ensure that all berms have sufficient height.

Safety berms assume a slope of 1.5 horizontal to 1.0 vertical. Considering that ramps in the pit only need one berm, the road width of 26 m was determined for two-lane traffic, which allows for 3.49 times the operating width of the haul trucks. Single-lane traffic roads are estimated to require 15.5 m which allows 1.77 times the operating width of the CAT-777 haul trucks.

Roads outside of the pit will require two berms and widths are estimated to be 30.5 m allowing 3.46 times the width of the CAT-777 haul trucks.

Road designs are intended to have a maximum of 10% gradient, though some may exceed this for short distances around inside turns. Where switchbacks are utilized, the centerline gradient is reduced to about 8%. This keeps the inside gradient approximately 12%. Switchback designs have not added the detail for super elevation through the curves, but is it assumed that this will be done when they are constructed.

Table 16-6: Road and Ramp Design Parameters

	In-Pit 2-Way Traffic In Pit Meters	Ex-Pit 2-Way Traffic Ex Pit Meters	In-Pit 1-Way Traffic In Pit Meters
Truck Width	6.10	6.10	6.10
Running / Truck Width Ratio	3.50	3.50	1.75
Road Running Width	21.35	21.35	10.68
Tire Size	27.00-R49	27.00-R49	27.00-R49
Tire 1/2 Height	1.35	1.35	1.35
Berm Height	1.48	1.48	1.48
Berm Top Width	0.25	0.25	0.25
Berm Slope	1.50	1.50	1.50
Berm Bottom Width	4.70	4.70	4.70
# Berms	1.00	2.00	1.00
Total Berm Width	4.70	9.40	4.70
Overall Width	26.05	30.75	15.38
Design Width	26.00	30.50	15.50
Running Width After Berms	21.30	21.10	10.80
Running Width / Truck Width	3.49	3.46	1.77

16.6 PIT DESIGN

A detailed pit design was completed for Mexican Hat using Surpac™ software (version 6.7). Each of the designs utilize 6.0 m benches with a catch bench installed every third bench, or 18 m, and the slope parameters shown in Table 16-3.

Mexican Hat pits were designed with five phases. Phase 1 mines the larger main portion of the deposit. Phase 2 continues to expand the main portion of the deposit. Phase 3 expands the main pit to the south and Phase 4 achieves the full depth and extents of the main pit. Phase 5 mines a satellite pit located to the south of the main deposit. Figure 16-2 shows the ultimate pit design. Figure 16-3, Figure 16-4, and Figure 16-5 show the North Pit designs for phases 1, 2, and 3, respectively, along with the South Pit design.

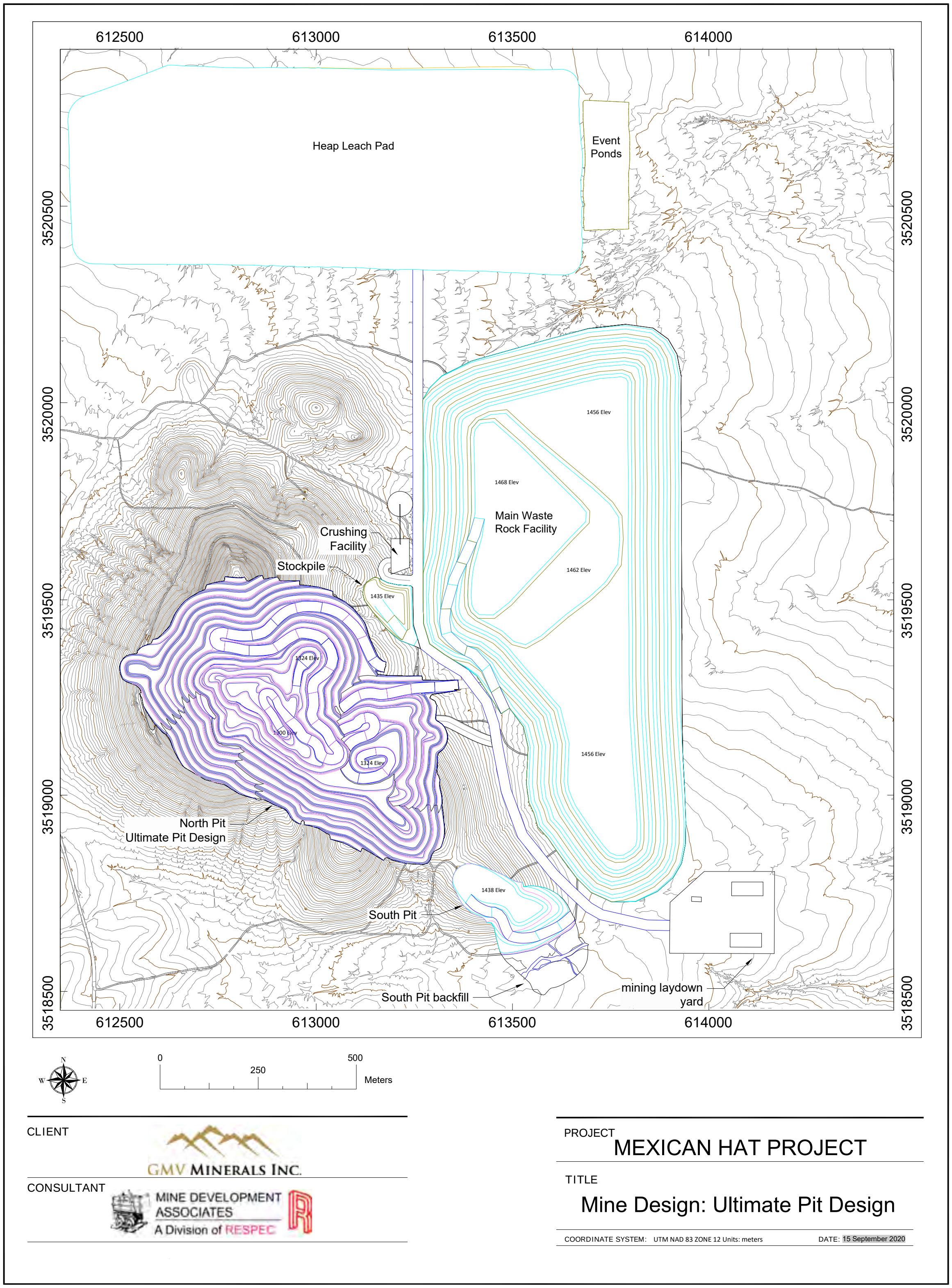


Figure 16-2: Ultimate Pit Design

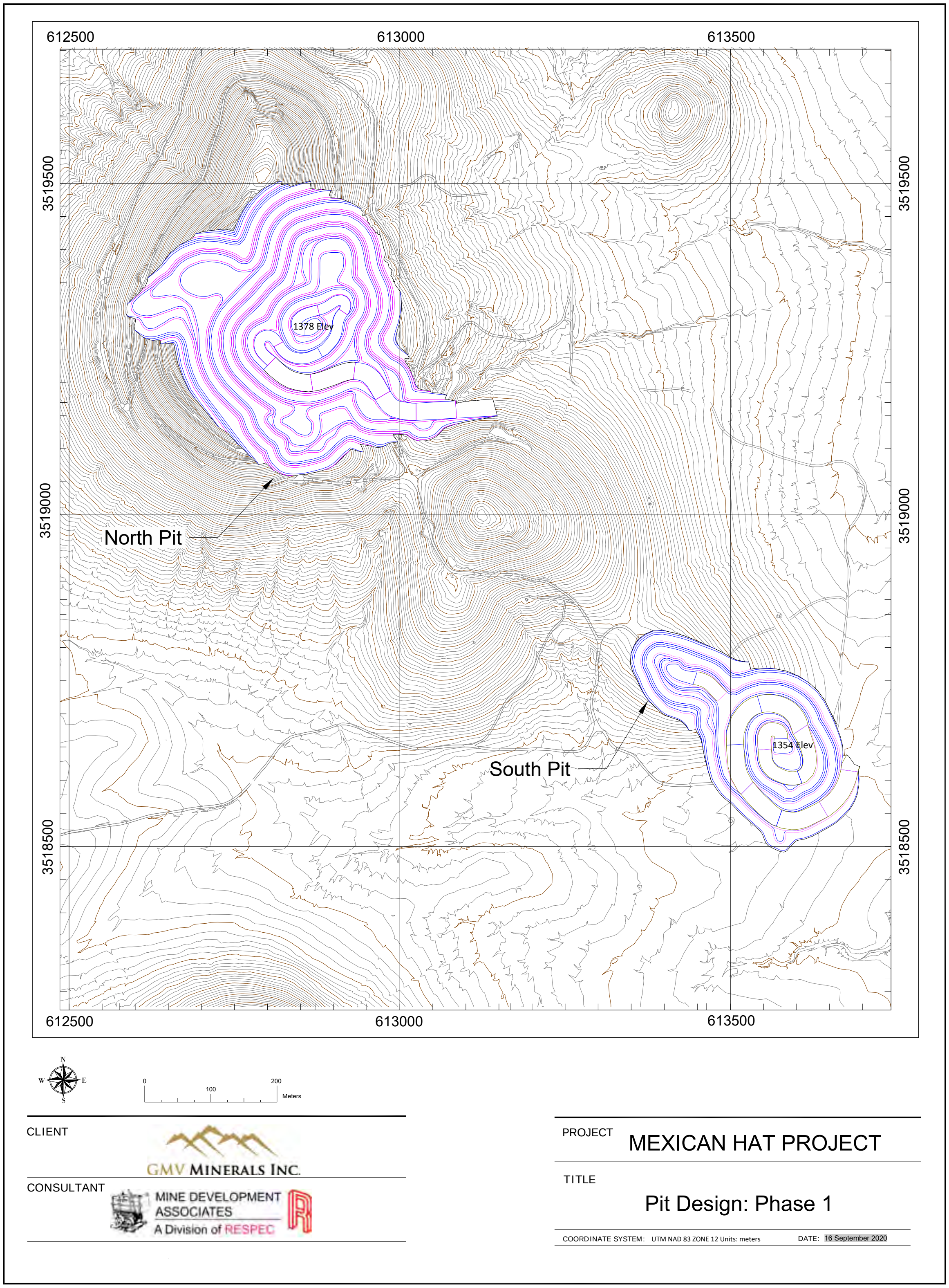


Figure 16-3: Phase 1 and South Pit Design

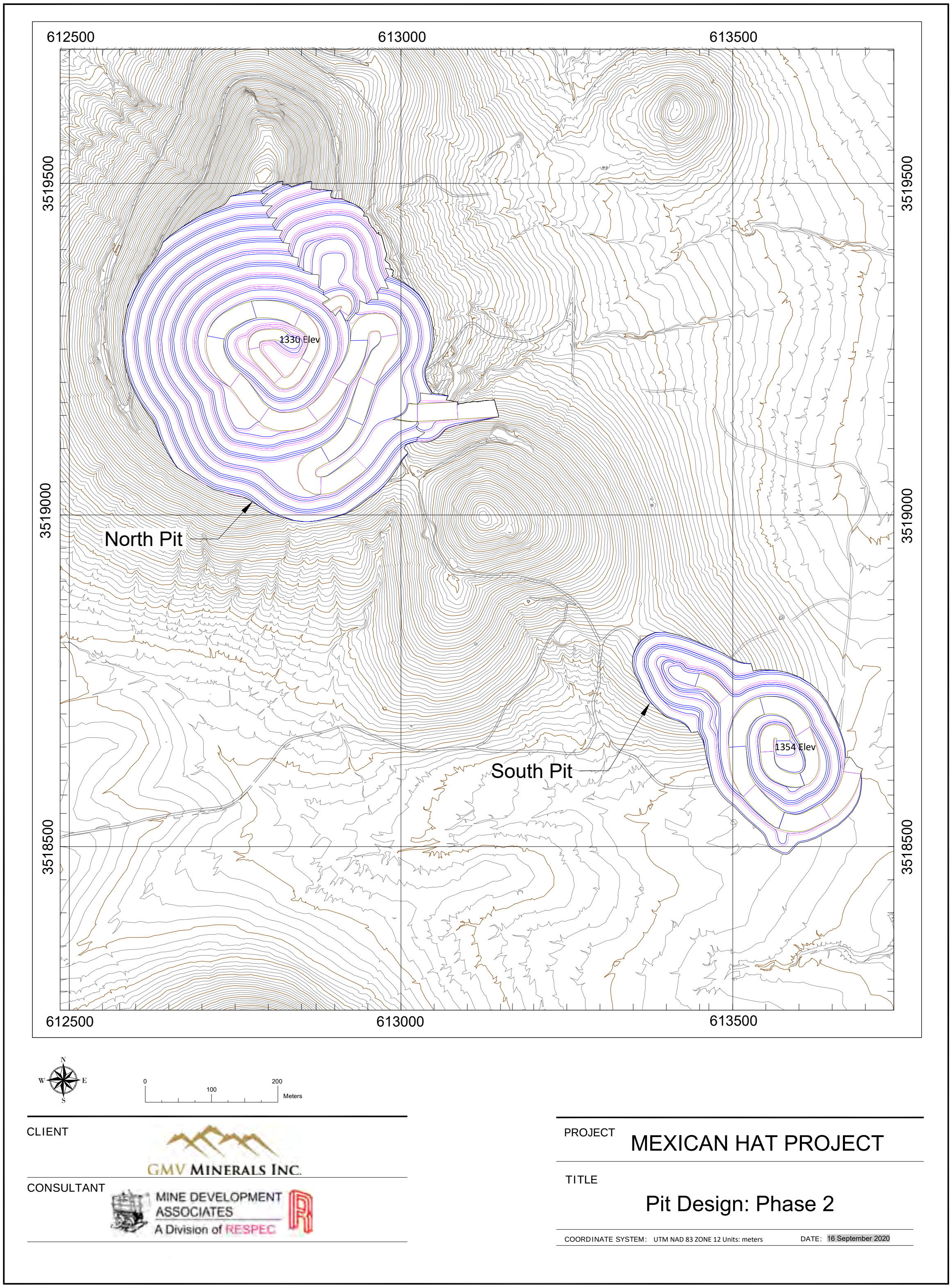


Figure 16-4: Phase 2 and South Pit Designs

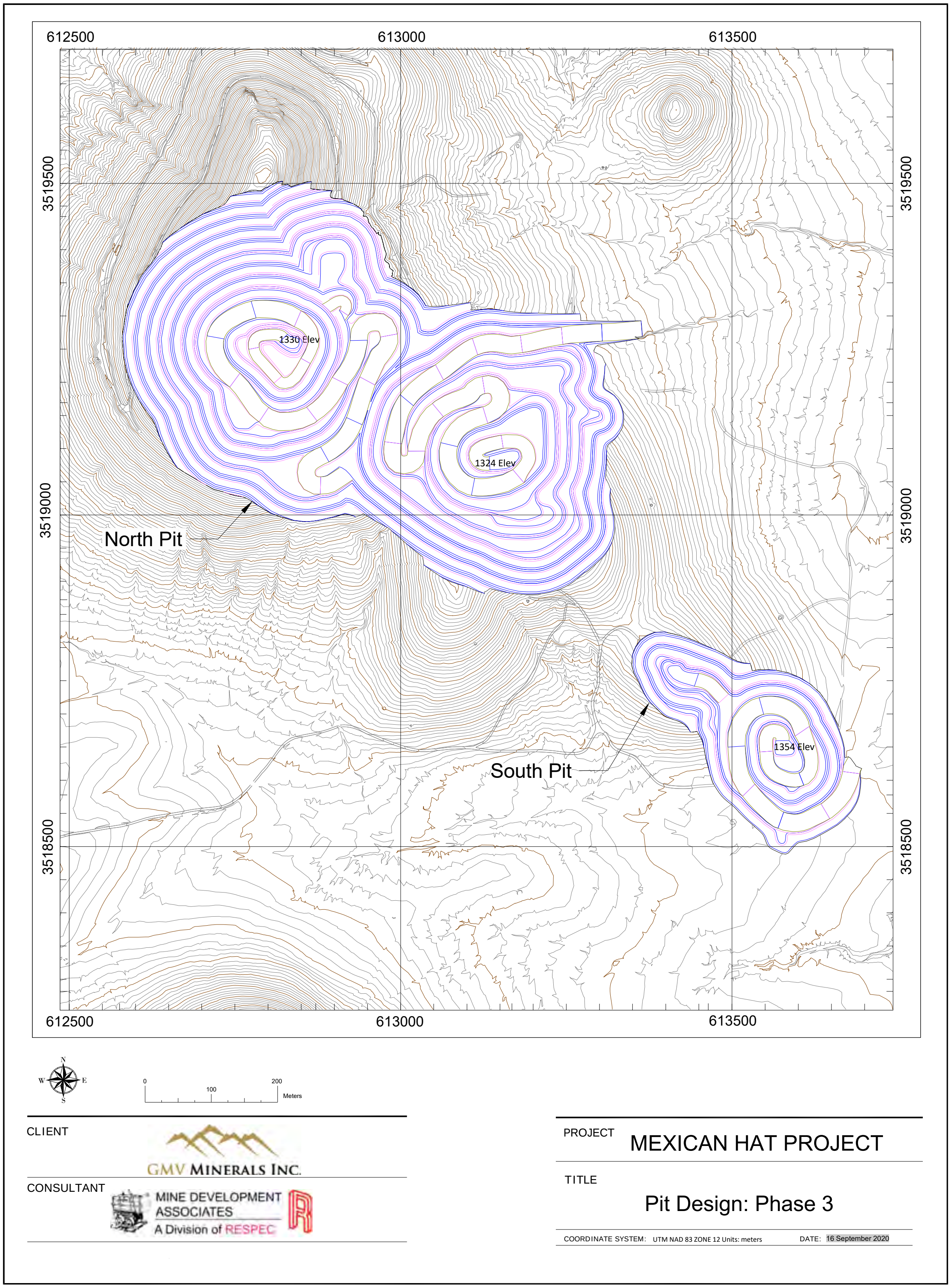


Figure 16-5: Phase 3 and South Pit Designs

16.7 IN-PIT RESOURCES

In-pit mineral resources were estimated for the Mexican Hat pit design and are tabulated in Table 16-7. The Mexican Hat pits have a total of 61.1 million tonnes of waste associated with the material to be processed, and thus has an overall strip ratio of 1.87 tonnes of waste per tonne leached.

Table 16-7: Mexican Hat In-Pit Resources

	Units	Inferred	Waste
Mexican Hat	K Tonnes	32,632	61,115
	g Au/tonne	0.569	
	K Ozs Au	597	

16.8 DUMP DESIGN

Dump designs were created for the PEA to contain the waste material mined. A 1.3 swell factor was assumed which provides for both swell when mined and compaction when placed into the facility. The total volume requirements for containment of waste material along with the intended destination for this material are shown in Table 16-8. The backfill will be placed into the South Pit earlier during the project and will be used to regrade the drainage channel to the south.

Table 16-8: Waste Containment Volume Requirements (K Cubic Meters)

	Destination		
Phase	Main Dump	Backfill Dump	Total
Phase 1	4,337	1,832	6,168
Phase 2	5,623	-	5,623
Phase 3	8,770	-	8,770
Phase 4	8,835	-	8,835
South Pit	1,518	-	1,518
Total	29,083	1,832	30,914

Waste dumps, the leach pad, and roads and facilities are shown in Figure 16-2.

16.9 MINE PRODUCTION SCHEDULE

Production scheduling was completed using Geovia's MineSched™ (version 9.1) software. Inferred resources inside of the pit designs previously discussed were used to schedule mine production.

Note that a PEA is preliminary in nature and it includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied that would enable them to be classified as mineral reserves, and there is no certainty that the PEA will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

The production schedule considers the processing of material by crushing followed by heap leaching. Monthly periods were used to create the production schedule with pre-stripping starting in Mexican Hat at month -6. The start of processing was assigned to month 1 though no gold production is realized until month 2.

The total mining rate would ramp up from 5,000 tonnes per day to about 26,500 tonnes per day over a period of 12 months. A maximum of 46,000 tonnes per day is used in later years when the stripping becomes more significant.

The monthly mining production for Mexican Hat is summarized yearly in Table 16-9.

Table 16-9: Mexican Hat Mine Production Schedule

	Units	Yr_-1	Yr_1	Yr_2	Yr_3	Yr_4	Yr_5	Yr_6	Yr_7	Yr_8	Yr_9	Yr_10	Total
Pit to Stockpile	K tonnes	318	332	180	392	206	258	191	339	0	0	0	2,215
	g Au/tonne	0.543	0.222	0.207	0.229	0.221	0.206	0.218	0.269	0.208	0.209	0.213	0.273
	K Ozs Au	6	2	1	3	1	2	1	3	0	0	0	19
Pit to Crusher	K tonnes	-	2,956	3,339	3,249	3,107	3,341	3,306	3,173	3,500	3,500	947	30,418
	g Au/tonne	-	0.574	0.661	0.659	0.601	0.483	0.562	0.731	0.491	0.547	0.647	0.590
	K Ozs Au	-	55	71	69	60	52	60	75	55	62	20	577
Total Above COG	K tonnes	318	3,289	3,519	3,641	3,312	3,599	3,498	3,511	3,500	3,500	947	32,632
	g Au/tonne	0.543	0.539	0.638	0.612	0.578	0.463	0.543	0.687	0.491	0.547	0.647	0.569
	K Ozs Au	6	57	72	72	62	54	61	78	55	62	20	597
Total Waste	K tonnes	1,597	6,001	4,671	3,679	5,368	4,446	8,697	12,925	8,896	4,167	667	61,115
Total Mined	K tonnes	1,915	9,290	8,190	7,320	8,680	8,045	12,195	16,436	12,396	7,667	1,613	93,748
Strip Ratio	W:O	5.03	1.82	1.33	1.01	1.62	1.24	2.49	3.68	2.54	1.19	0.70	1.87
Rehandle	K tonnes	-	544	161	261	393	159	194	337	-	-	167	2,215
	g Au/tonne	-	0.413	0.206	0.240	0.215	0.207	0.217	0.269	-	-	0.209	0.273
	K Ozs Au	-	7	1	2	3	1	1	3	-	-	1	19

The process production schedule was created by MDA based on the mine production schedule and recoveries and lag times estimated by SE. The recoveries are used to estimate recoverable gold. The lag time is generated by estimating the quantity of recoverable ounces on a month by month basis after placement of material. Table 16-10 shows the assumed rate of recovery of the recoverable ounces by month for leaching. The recovery of gold is 0% for the month placed/mined allowing material to be placed, ripped as required, and then start leach spraying. The second month after placement sees the most gold production. Leach recovery is assumed to take place between the month after placement through the 14th month after placement.

Table 16-10: Recovery of Recoverable Gold by Month

Mth Placed	Leach
Month Placed	0.0%
Month 1	67.5%
Month 2	8.4%
Month 3	4.8%
Month 4	3.5%
Month 5	2.7%
Month 6	2.1%
Month 7	1.9%
Month 8	1.6%
Month 9	1.4%
Month 10	1.3%
Month 11	1.3%
Month 12	1.3%
Month 13	1.3%
Month 14	0.9%
Total	100.0%

Table 16-11 shows the yearly process production summary. The rows labeled “K Au Rec” shows the thousands of recoverable ounces of gold and the rows labeled “K Au Prod” are the thousands of ounces of gold produced.

The PEA total life-of-mine (“LOM”) gold production is estimated to be 525,000 in doré with a LOM average recovery of 88%.

Table 16-11: PEA Process Production Schedule

Royalty	Units	Yr_-1	Yr_1	Yr_2	Yr_3	Yr_4	Yr_5	Yr_6	Yr_7	Yr_8	Yr_9	Yr_10	Total
Outside Victor Claims	K Tonnes	-	1,180	6	36	278	133	4	0	0	1	-	1,638
	g Au/tonne	-	0.59	0.20	0.34	0.63	0.72	0.88	0.21	0.21	1.68	-	0.60
	K Ozs Au	-	22	0	0	6	3	0	0	0	0	-	32
	K Au Rec	-	20	0	0	5	3	0	0	0	0	-	28
	K Au Prod	-	18	1	0	4	4	1	0	0	0	0	28
Inside Victor Claims	K Tonnes	-	2,320	3,494	3,473	3,222	3,367	3,496	3,510	3,500	3,499	1,113	30,994
	g Au/tonne	-	0.53	0.64	0.63	0.55	0.46	0.54	0.69	0.49	0.55	0.58	0.57
	K Ozs Au	-	40	72	70	57	50	61	78	55	62	21	565
	K Au Rec	-	35	63	62	50	44	54	68	49	54	18	497
	K Au Prod	-	26	57	65	54	43	51	68	52	49	31	496
Total Processed	K Tonnes	-	3,500	3,500	3,510	3,500	3,500	3,500	3,510	3,500	3,500	1,113	32,632
	g Au/tonne	-	0.55	0.64	0.63	0.56	0.47	0.54	0.69	0.49	0.55	0.58	0.57
	K Ozs Au	-	62	72	71	63	53	61	78	55	62	21	597
	K Au Rec	-	54	63	62	55	47	54	68	49	54	18	525
	K Au Prod	-	45	58	65	58	47	51	68	52	49	31	524

Table 16-12 shows the leach stockpile balance sheet. As previously mentioned, the leach stockpile is placed near the crusher. Rehandling of the leach stockpile will be done by the mining contractor.

Table 16-12: Total Stockpile Balance

Total - All StkPls	Units	Yr_-1	Yr_1	Yr_2	Yr_3	Yr_4	Yr_5	Yr_6	Yr_7	Yr_8	Yr_9	Yr_10
Added	K tonnes	318	332	180	392	206	258	191	339	0	0	0
	g Au/tonne	0.543	0.222	0.207	0.229	0.221	0.206	0.218	0.269	0.208	0.209	0.213
	K Ozs Au	5.6	2.4	1.2	2.9	1.5	1.7	1.3	2.9	0.0	0.0	0.0
Removed	K tonnes	-	544	161	261	393	159	194	337	-	-	167
	g Au/tonne	-	0.413	0.206	0.240	0.215	0.207	0.217	0.269	-	-	0.209
	K Ozs Au	-	7.2	1.1	2.0	2.7	1.1	1.3	2.9	-	-	1.1
Balance	K tonnes	318	106	125	256	69	167	165	167	167	167	-
	g Au/tonne	0.543	0.206	0.207	0.208	0.209	0.207	0.209	0.209	0.209	0.209	-
	K Ozs Au	5.6	0.7	0.8	1.7	0.5	1.1	1.1	1.1	1.1	1.1	-

16.10 EQUIPMENT REQUIREMENTS

The PEA has assumed contract mining in order to simplify the economic study. The contractor will be responsible for supplying equipment to be used for the project. It is anticipated that the equipment required for mining will be like:

- One enclosed/self-contained airtrack drill for pioneer mining;
- Up to two 45,000 pulldown blast hole drills used for production;
- One explosives truck and a skid steer loader for blasting operations;
- Up to two 992 Cat sized loaders for loading 100-ton capacity haul trucks;
- Between five and eight 100-ton capacity haul trucks;
- One D10 sized dozer to maintain waste dump faces;
- One D8 sized dozer for maintaining pit floors and pioneer mining;
- One 20,000-gallon sized water truck for dust control;
- One or two 16-foot moldboards graders for road maintenance; and
- Miscellaneous maintenance equipment.

16.11 PERSONNEL REQUIREMENTS

The mining contractor will be responsible for providing personnel for mining operations. Most personnel will likely be sourced from Tucson or the surrounding area. Thus, there is no need for any camp to house employees. The contractor will be responsible for transportation of employees to and from the site. It is expected that the contractor will have somewhere around 30 crew members per shift and operate two shifts per day. Assuming four shift rotations to provide seven-day per week operations, this would be about 120 employees provided by the contractor.

Additional employees will be provided by GMV for Mine General Services, which includes mine supervision and engineering and geology services. The Mine General Services employees will include one of each of the following:

- Chief Engineer to supervise mine planning activities and resource/reserve statements;
- Mine Engineer to assist the Chief Engineer with mine planning activities;
- Chief Surveyor to work with the surveying helper to lay out all drill patterns and mineralized material control along with recording as built drawings and verification of volumes mined and stockpiled;

- Surveyor's helper to assist the Chief surveyor;
- Chief Geologist to maintain mapping of geology, determine drilling program needs, supervise mineralized material control, and updates for resource modeling;
- Ore Control Geologist to maintain blast hole database integrity and provide mineralized material control boundaries for areas to be mined for processing and waste; and
- Samplers to pick up blast hole samples from the blast hole rig and deliver them to the assay lab on site. The Samplers will assist the Ore Control Geologist as available.

17.0 RECOVERY METHODS

17.1 PROCESS PLANT DESCRIPTION, DESIGN, AND FLOW SHEET

The Mexican Hat gold deposit will be mined by open pit. Run-of-mine material from the pit will be hauled to a two-stage crushing plant operated by a contractor with a nominal capacity rate of 10,000 t/d of mineralized gold material. The crushed material at 80% passing 38 mm will be conveyed to a crushed material stockpile with a live capacity of 24 hours. Crushed material will be reclaimed from beneath the stockpile by two feeders (4 ft by 15 ft each) and discharged onto an overland conveyor (1,000 m long by 42 inch wide) for conveying the crushed material to grasshopper conveyors at the heap leach pad. Grasshoppers and a radial stacking conveyor will be used to stack the crushed material onto the heap leach pads. The stacking schedule will be 12 hours per day at a rate of 833 t/hr. Lime will be added onto the reclaim conveyor from the stockpile ahead of heap leaching. The stacked crushed material will be irrigated with a cyanide solution to dissolve the contained gold into a pregnant leach solution (PLS). The cyanide solution will percolate through the heap crushed material and be collected in a lined pregnant solution pond. Pregnant solution will be pumped to an adsorption, desorption, recovery (ADR) plant. The ADR plant will be comprised of two trains of 5 carbon in column (CIC) tanks operated in series. The gold in the PLS solution will adsorb onto the carbon. Carbon will be advanced in the CIC circuit counter current to flow of the pregnant solution. Carbon will be transferred daily to a 3 t acid wash and elution circuit for carbon desorption. After carbon is acid washed, it will be rinsed then transferred to the elution column where the strip solution will be pumped to the bottom of the vessel and circulate the required number of bed volumes to recover gold. The eluate will be pumped to a series of two electrowinning cells where gold will be plated onto the steel wool. The electrowinning cell will be cleaned, and the recovered sludge will be filtered and dried prior to mercury retort prior to smelting in the furnace for producing the gold doré bars.

The process flow diagram is shown in the below Figure 17-1.

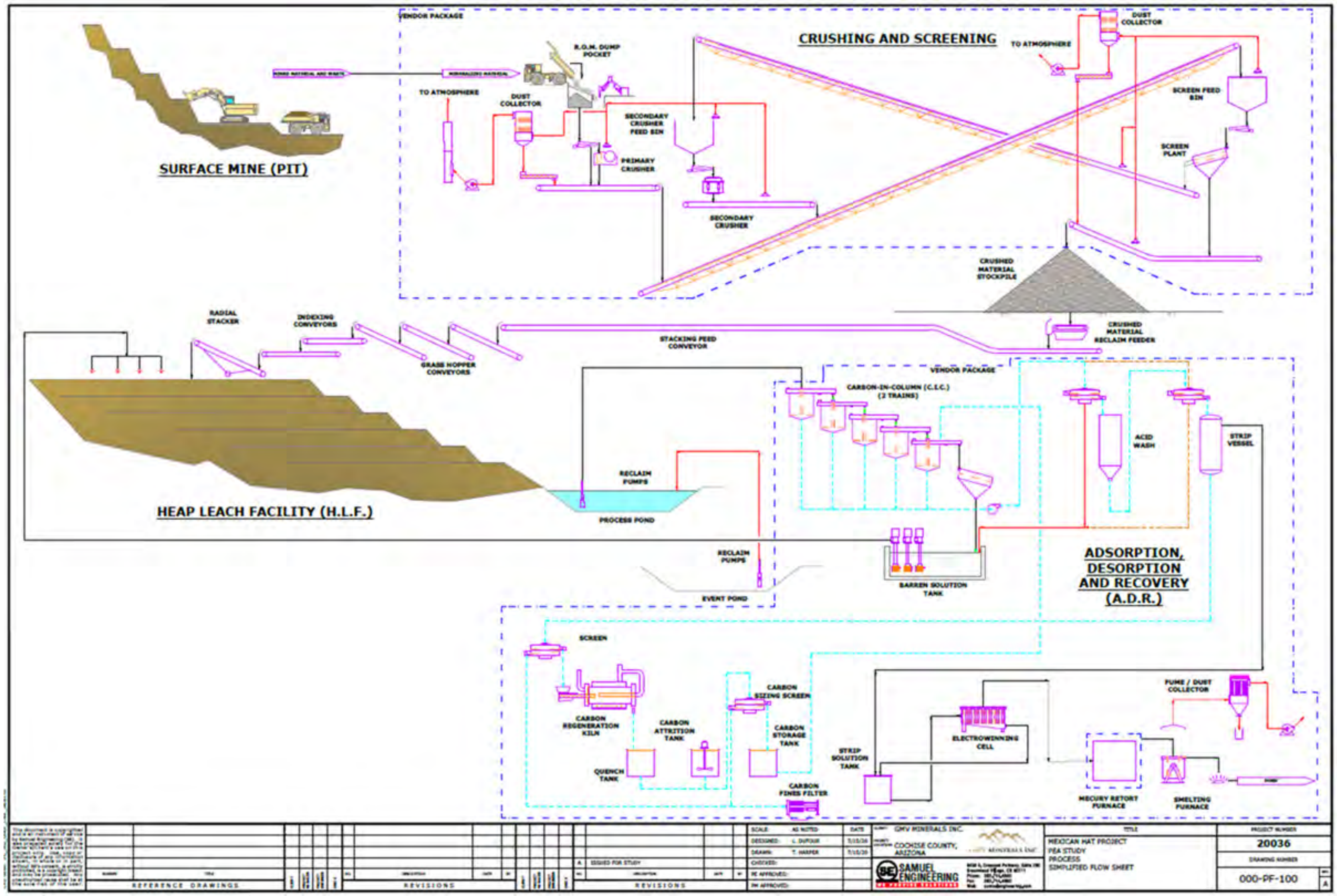


Figure 17-1: Mexican Hat Processing Facilities (Conceptual Flow Diagram)

17.2 CRUSHING DESIGN

The Mexican Hat project is designed to crush 3.5 million tonnes of resources per annum at a daily rate of 10,000 t/d for 350 days per year. The 2-stage crushing circuit is designed at a 75% availability, equivalent to a crushing rate of about 1,110 t/h. The crushing circuit is designed to crush run-of-mine material in 2 stages from 100% passing feed size of 1,400 mm down to a product at 80% passing 38 mm. The crushing plant will be supplied, operated, and maintained by the crushing contractor.

The crushed material will be conveyed from the crushing plant by a radial stacking conveyor to a crushed material stockpile. The stockpile will have 24 hour live capacity. Crushed material is reclaimed from beneath the stockpile by two reclaim feeders to an overland conveyor where lime will be added from a silo at the required dosage. The crushed material and lime will be conveyed to the heap leach pad for stacking onto the heap leach pad.

Table 17-1 below summarizes the key design parameters for crushing.

Table 17-1
Summary of the Key Design Parameters for Crushing

CRUSHING AND STACKING	UNIT	DESIGN
Crushing and stacking rate	tpd	10,000
Crushing Plant Availability	%	75%
Crushing throughput rate, nominal	t/h	1,110
Stockpile Live Capacity	h	24
Stacking Schedule	h/d	12
Stacking Rate, nominal	t/h	833
Grizzly screen aperture	mm	
Secondary Screen Decks	No.	2
Secondary Screen Bottom Deck Aperture	mm	
Crushed Material Bulk Density	Kg/m^3	1,600
Crusher Work Index	kWh/t	15.4
Heap Leach Stacked Crushed Material Height	m	9
Lime Consumption	Kg/t	1.5

17.4 PROCESS PLANT DESIGN AND OPERATION

17.4.1 METAL RECOVERY AND OPERATION

For Mexican Hat, the metal recovery from heap leaching will be in an ADR circuit, regeneration, and refinery plant. The plant will have an operation availability of 92% and will be supplied as a vendor package with all the equipment necessary to recover gold from the PLS.

Pregnant solution will gravity flow through the heap leach and be collected in a lined pregnant solution pond. Pregnant solution will be pumped to two trains of 5 CIC in series (total of 10 CIC tanks) where gold in solution is adsorbed onto the carbon. Loaded carbon is advanced in the CIC circuit counter current to the flow of solution. This ensures that the highest-grade gold in solution is in contact with the highest grade loaded carbon and that the lowest grade solution is in contact with the lowest grade carbon to ensure carbon lading efficiency. Loaded carbon is transferred daily from the CIC to the 3 tonne acid wash and elution circuit.

The loaded carbon acid wash is at atmospheric pressure and acid wash solution is circulated to the bottom of the tank and overflows back into the acid wash solution tank at a rate of 2 bed volumes per hour for 1.5 bed volumes. The carbon is then rinsed with water for 30 minutes. After rinsing, the carbon is transferred to the 3 tonne strip vessel where the carbon is stripped under pressure and about 145°C. Strip solution is pumped via the bottom of the strip vessel and flows to the electrowinning circuit where gold is plated onto steel wool. Typically, eleven bed volumes of strip solution are required at a pumping rate of 2 bed volumes per hour.

Stripped carbon will be transferred to the horizontal rotary kiln for regeneration at 750°C. The carbon should be regenerated in the kiln after every desorption stage otherwise carbon loading efficiency will decrease and gold losses in solutions in the CIC will increase.

The gold rich eluate is pumped to the electrowinning cells where a current is passed so that the gold will plate onto the steel wool cathode. The barren solution from electrowinning will recirculate back to the strip solution tank to be used for mixing strip solution. A 20% bleed of barren solution is recommended to avoid buildup of contaminants.

Gold contained sludge from the electrowinning cells will be washed off the cathode and pumped to a plate and frame filter press. Filter cake will be placed in trays into a drying oven. Dried filter cake will be put through mercury retort prior to mixing with smelting flux and put into the electric induction furnace. The slag layer containing impurities will be removed prior to pouring the gold into molds to produce gold doré bars. The doré bars will be cooled, cleaned, sampled, and shipped to market.

Table 17-2 summarizes the adsorption and desorption design parameters.

Table 17-2
Summary of Adsorption and Desorption Design Parameters

ADSORPTION & DESORPTION	UNITS	DESIGN PARAMETER
CIC Adsorption		
Number of Trains	No.	2
CIC per Train	No.	5
Carbon per CIC	t	5
Carbon Column Volume	m^3	11
Carbon Loading, nominal	g/t	2,523
Carbon Loading, maximum	g/t	4,000
Adsorption Efficiency	%	98
Operation Availability	%	92
Acid Wash		
Acid Wash Carbon Column Capacity	t	3
Acid Wash Solution	-	Hydrochloric Acid
Acid Wash Solution Concentration	%	3
Desorption		
Desorption Carbon Column Capacity	t	3
Desorption Carbon Column Capacity	m^3	6.38
Solution Flowrate	BV/h	2
Strip Solution	-	NaOH

Table 17-2 Summary of Adsorption and Desorption Design Parameters		
Strip Solution Concentration	%	1
Strip Solution Temperature	°C	145
Strip Solution Pressure	kPa	450
Strip Efficiency	%	95
Carbon Regeneration		
Kiln Throughput	Kg/h	150
Regeneration time	h	20
Regeneration Temperature	°C	750
Mercury Retort		
Mercury Retort Temperature	°C	650
Electrowinning		
Flowrate	m^3/h	10-15
Electrowinning Current	kW	18
Electrowinning Current Density	A/m^2	200
Electrowinning Efficiency	%	96-98
Smelting		
Smelting Temperature	°C	1,230
Smelting days Per Month	days	7

17.5 HEAP LEACH PAD DESIGN

The HLF is in an area of flat to gently sloping topography that will require some grading in the HLF footprint. The HLF surface is generally undisturbed with small shrubs, bushes, and desert cacti. All vegetative cover, organic soils, and growth media will be removed prior to construction. The HLF, which includes the HLF, PLS, and event pond is planned to be located north of the proposed pit. The HLF will be constructed in two phases and has been designed for a nominal production rate of 3,500,000 t of mineralized material per year (10,000 tpd) for a total heap capacity of 32.6 Mt assuming a heap bulk density of 1.5 t/m^3. The mineralized material will be mined by a standard open pit mining method, crushed to 80% minus 38 mm, and placed through transport and stacking on the HLF in 10-m-high lifts using a conveyor/stacking system. The HLF is anticipated to have a maximum height of 72 m and an overall slope of 2.5H:1V.

The HLF is designed to meet or exceed the prescriptive BADCT criteria as described in the ADEQ Arizona Mining BADCT Guidance Manual (ADEQ, 2004). Where appropriate, additional design criteria (not prescribed in BADCT) were included based on professional judgment, standard engineering practices, and site-specific conditions. Refer to Appendix 28.2 for the HLF Design Summary Memorandum.

The HLF will be constructed in two phases with approximate areas of 192,201 m^2 and 373,311 m^2 for Phase 1 and 2, respectively (or a total Phase 1 and 2 area of approximately 565,512 m^2).

The HLF will consist of:

Liner System: The liner systems provide a boundary to contain a PLS and protect the underlying groundwater. The composite geomembrane and geosynthetic clay liner (GCL) system is used where clay soils are not available for constructing liner system that meets Arizona's prescriptive BADCT requirements. Components of the liner systems are listed below (Table 17-3):

Table 17-3 Pad Liner System	
Tasks	**Description**
Bedding Fill or Prepared Subgrade	Bedding Fill for placement underneath the GCL will have 0.15 m of minus 1-inch maximum particle size liner bedding material on compacted and smoothed subgrade surface in preparation for geomembrane liner placement (Section 4.0). The Bedding Fill will be compacted to a minimum 95% of maximum dry density (ASTM D698) and moisture conditioned to within 2% of the optimum moisture content (ASTM D698). In some areas, the GCL can be placed over a prepared subgrade that meets the criteria for Bedding Fill, which consists of a minimum, 0.15 m native or natural materials scarified to a minimum depth of 0.15 m and compacted to 95% maximum dry density (standard Proctor; ASTM Method D698) within 2% of the optimum moisture content. The prepared subgrade should have a smooth surface with maximum particle size according to the specifications.
Clay Soil Liner	The design will utilize (double-sided non-woven) GCL or low permeability soil liner if available. The GCL or low permeability soil will have a hydraulic conductivity no greater than 1×10^{-6} cm/s (Section 4.0)
Geomembrane Liner	80-mil double-sided textured linear low-density polyethylene (LLDPE) as required for slope stability. Geomembrane liner must be protected from the additional loads that will occur adjacent to the drain piping system due to the arching loads.
Anchor Trench	0.6 m wide by 0.6 m minimum depth trench

Over-liner Drain Fill and Solution Collection System: Over-liner Drain Fill provides liner protection from exposure to the climate, vehicle tracks, and mineralized material placement via haul trucks. The Over-liner Drain Fill also reduces the hydraulic head on the pad liner when constructed in combination with supplemental drainpipes placed at a spacing determined by the leaching solution application rate and the permeability characteristics of the drain rock. Components of the Over-liner Drain Fill and Solution Collection System are listed below (Table 17-4):

Table 17-4 Pad Over Liner and Piping System	
Tasks	**Description**
Solution Collection System	100 mm diameter corrugated and perforated polyethylene (PE) N-12, or equivalent, primary pipes placed in a herringbone fashion placed on 6 m maximum centers. 450-mm diameter corrugated and perforated PE N-12, or equivalent, secondary pipes spaced as necessary to handle the solution application. 600-mm diameter corrugated, and perforated header pipes spaced as necessary to handle the solution application flows plus estimated flows from the design storm event. 600-mm diameter solid HDPE discharge pipes to route flows to the PLS Pond. Maximum allowable deflection under load of 20%
Over-liner Drain Fill	The HLF geomembrane liner will be covered by a minimum of 0.6 m of Over-liner Drain Fill, well graded and free-drainage granular material with less than 5 percent particles passing the No. 200 ASTM sieve size. No moisture conditioning or compaction of the Over-liner Drain Fill is required. Hydraulic Conductivity should maintain a minimum of one order of magnitude higher permeability compared to the overlying mineralized material heap.

Figure 17-2 shows the planned Phase 1 and Figure 17-3 the HLF final configuration.

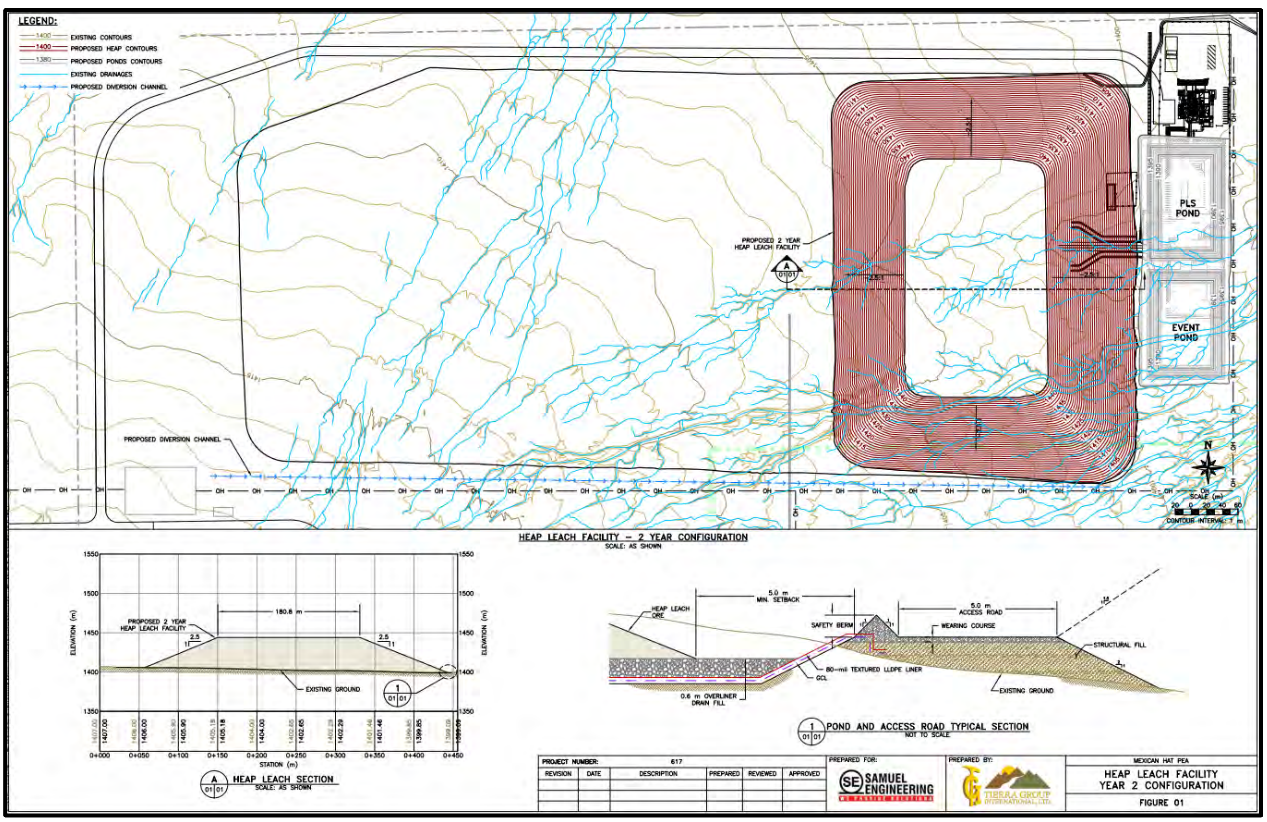


Figure 17-2: HLF Phase 1 Configuration

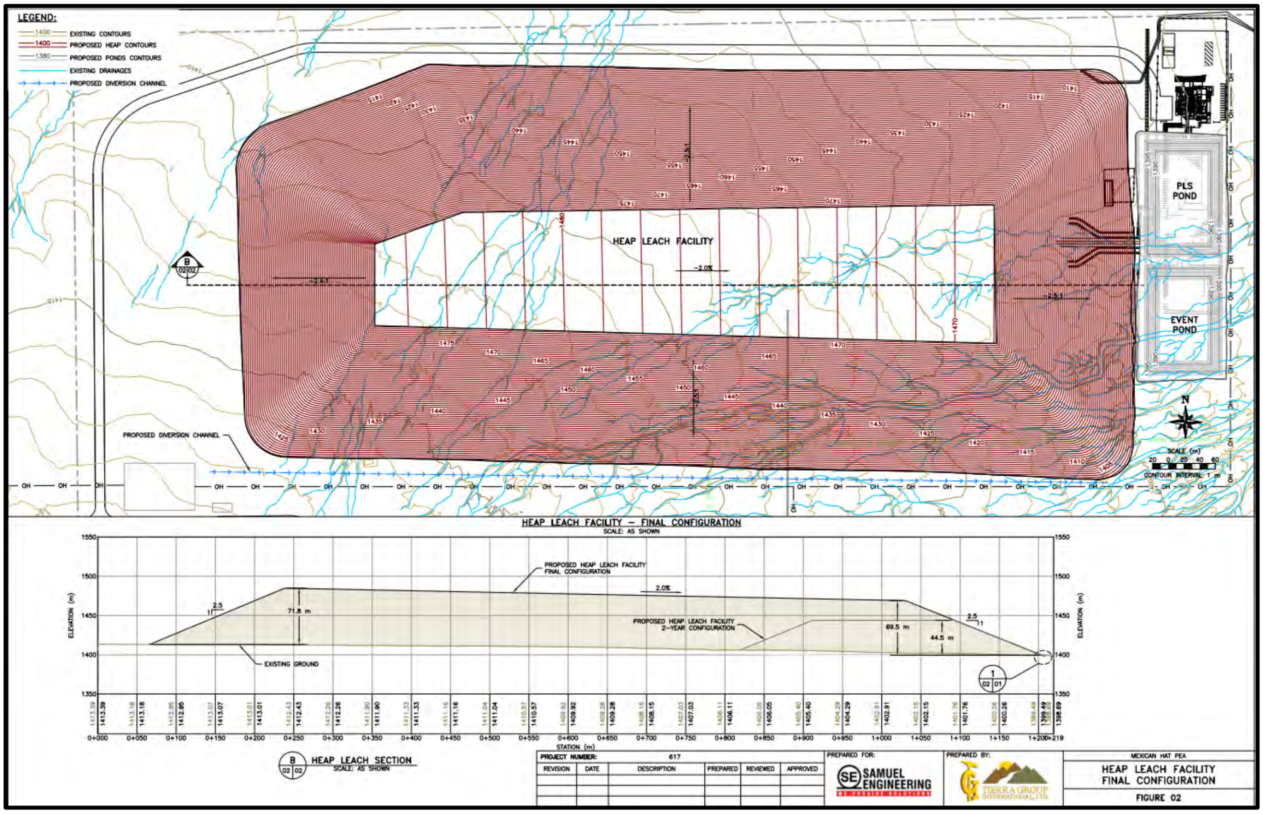


Figure 17-3: HLF Phase 1 and 2 Configuration

Crushed Material Stacking

The fully stacked HLF (Phase 1 and 2) will have a total capacity of 32.6 Mt of mineralized material stacked in approximately 9.3 years.

The Phase 1 will have the capacity to store mineralized material for 2 years, which will constitute the first four lifts to a nominal top surface elevation of approximately 1,444 m.

The Phase 2 expansion expands to the west providing capacity to store mineralized material for the LOM, which will constitute seven lifts to a nominal top surface elevation of approximately 1,472 m.

Collection Ponds

The HLF will include two (2) ponds: A PLS collection and a Storm/Upset Events pond. The PLS collection pond will collect and store the minimum operational volume, maximum average seasonal volume, and any temporary draindown. The Event Pond will collect the 100-year, 24-hour storm runoff volume.

The requirements specified in the ADEQ Mining BADCT Manual (ADEQ, 2004) were used to determine the required capacity for the Ponds. According to the Prescriptive BADCT guidance, the Ponds must provide storage volume for the following:

- Minimum Operating Volume;

- Average Seasonal Runoff Volume;
- Operational Upset Volume;
- 100-year, 24-hour Storm Runoff Volume (Design Storm); and
- Two (2) feet or 0.6 m of dry Freeboard Volume.

Prescriptive BADCT (ADEQ 2004) requires 2 feet or 0.6 m of dry freeboard above the required storage volume. A contingency was provided due to concerns regarding potential discharge from the Pond and recommends providing an additional 1 foot of freeboard (for a total of 3 feet or 1 m).

The PLS Pond will provide a storage volume of 78,381 m^3 for the operational volume, seasonal volume, and operational upset. A spillway will be constructed between the Event Pond and the PLS Pond (elevation and design will be part of the pre-feasibility study (PFS)). When the water surface elevation in the PLS Pond reaches the spillway crest, excess flows will be directed by the spillway to the Event Pond.

The Event Pond will provide a minimum storage volume of 68,559 m^3 to the spillway elevation. The combined storage volume of the ponds at the spillway elevation will be 146,940 m^3.

The crest of the combined Ponds is set at an elevation of 1,397 m amsl, providing 1 m of dry freeboard above the spillway (located at 1,396 m amsl). The provided volume is shown in Table 17-5.

Table 17-5
Provided Pond System Volume

Pond	Volume Requirements (m^3)	Actual Volume Capacity with Freeboard (m^3)	Actual Volume Capacity without Freeboard (m^3)
PLS Pond	78,381	78,381	100,123
Event Pond	68,559	68,559	92,843
Total	**146,940**	**146,940**	**192,966**

If flows entering the Ponds exceed the volumes generated by the design storm, the water level will rise in both Ponds simultaneously within the provided freeboard. Based on the configuration of the Ponds, the freeboard provides an additional 46,026 m^3 of capacity. The ultimate capacity of the Ponds with freeboard is 192,966 m^3. The pond configuration is shown in Figure 17-4.

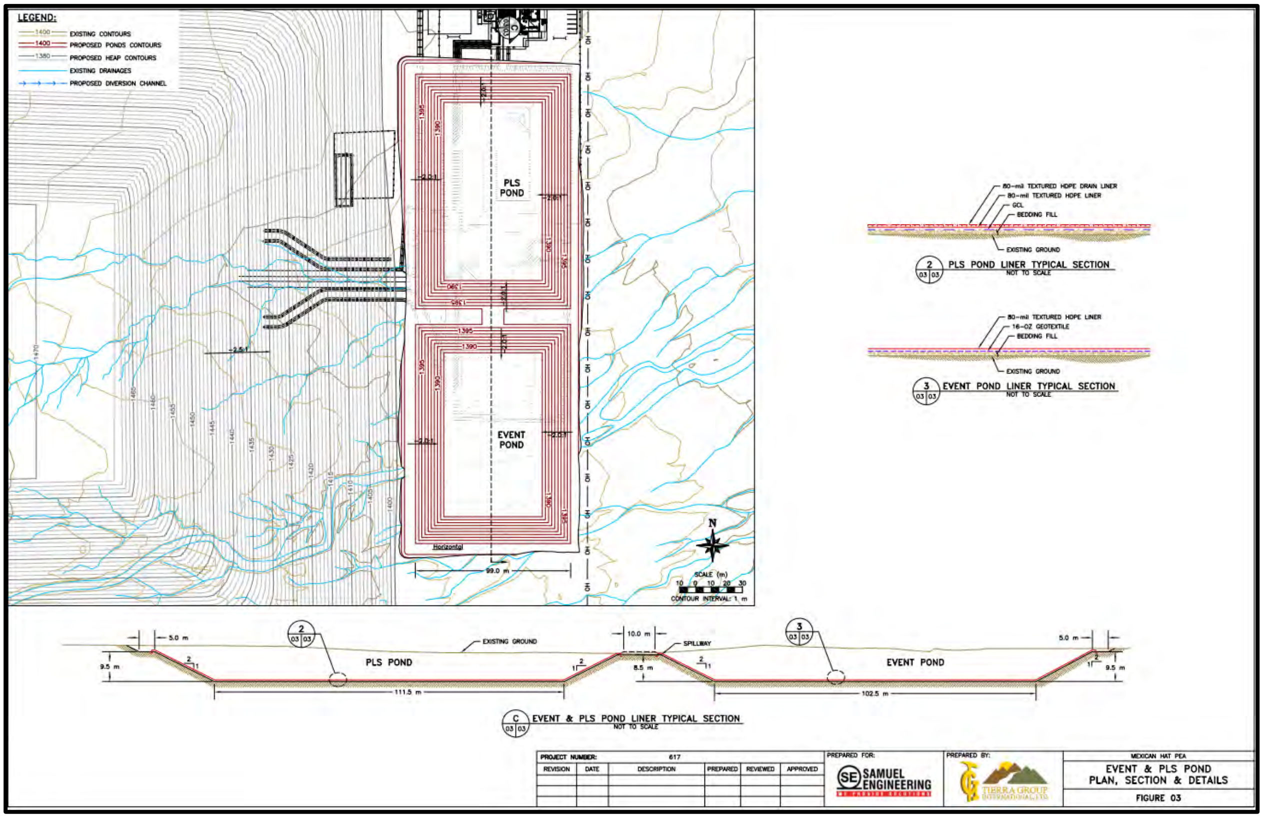


Figure 17-4: PLS and Event Pond Configuration

Drainage Control

Diversion channels will be constructed around the perimeter and upgradient of the HLF to divert stormwater runoff from contributing basins away from the HLF and collection ponds. Additionally, the HLP will have a perimeter berm to prevent solution and water within the HLF from overflowing.

17.6 LOM GOLD PRODUCTION

The LOM gold production from Mexican Hat is estimated at 525,000 ozs (about 71,200 ozs/y) contained in gold doré bars from the ADR plant based on the LOM mine plan for resource production, gold grade and processing recovery of 88%. Table 17-6 summarizes the LOM production data.

Table 17-6 LOM Gold Production													
Description	**Units**	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**Year 8**	**Year 9**	**Year 10**	**Year 11**	**Totals**
Leached Material	t 000's	3,500	3,500	3,510	3,500	3,500	3,500	3,510	3,500	3,500	1,113	-	32,632
Gold Grade	Au g/t	0.55	0.64	0.63	0.56	0.47	0.54	0.69	0.49	0.55	0.58	-	0.57
Contained Gold	000's Ozs	62	72	71	63	53	61	78	55	62	21	-	597
Recovered Gold	000's Ozs	54	63	62	55	47	54	68	49	54	18	-	525
Gold Produced (1)	000's Ozs	45	58	65	58	47	51	68	52	49	31	1	525

Note 1: LOM gold produced in production years have been adjusted to account for commissioning and leaching kinetics.

Note 2: Gold in Years 10 and 11 includes continued production from the leaching of crushed gold material placed on the leach pads in previous years.

18.0 PROJECT INFRASTRUCTURE

Project infrastructure for the Project has been developed to support the mining, crushing, heap leaching and ADR operations and will include the following:

1. Access.
2. Site Guardhouse with Site Security Fencing and Truck Scale.
3. Power Supply.
4. Fresh Water Source, System and Monitoring Wells.
5. Buildings (Equipped and Furnished with Communications).

18.1 ACCESS

The Project is in Cochise County in ESE Arizona. Road access to the site is by all paved Highway 191 from Pearce (about 10 km) to the Old Ghost Town Road, a gravel road. The Old Ghost Town Road runs along the south border of the Mexican Hat property. An access road from the Old Ghost Town Road would need to be constructed to Project operational areas, about 1 km.

18.2 SITE GUARDHOUSE WITH SECURITY CAMERAS, SITE SECURITY FENCING AND TRUCK SCALE

A site guardhouse will be constructed on the access road to the Project site. The guardhouse will be manned 24-hours a day for screening all vehicles and personnel entering the site and will be equipped with security cameras. Site security fencing will be built around the entire project site at an estimated total length of 8,800 m. A 60-t truck scale for weighing vehicles entering the project site will be located adjacent to the guardhouse.

18.3 POWER SUPPLY

Power will be supplied to the Project by the Sulphur Springs Valley Electric Cooperative (SSVEC). SSVEC services power in Cochise County for Mexican Hat region. SSVEC will extend an existing 69 kV powerline a distance about 12 km (7.5 miles) to the Project site for supplying a 6 MW load. SSVEC will be responsible for obtaining all right-of-way easements for the power line extension. GMV will need to construct its own substation at the Project site to drop the 69 kV incoming voltage to 4,160 V for its operations. Power cost is estimated at $0.08 per kWh in the PEA.

18.4 WATER SUPPLY AND MONITORING WELLS

Groundwater has been identified as the best source for a water supply to the Project operations. Successful water wells that are presently shut-in have apparently been drilled (Hernandez, pers. com, 2014) within the general project area. Water has been encountered in every drillhole completed on the Property by GMV, often at depths less than 50 m. Pumped water from the wells would be transferred by booster pumps to a freshwater tank for distribution to the various site operations. There would be separate fire and potable water tanks. A formal hydrogeologic study will need to be conducted during the next phase of project advancement to characterize local water quality and supply. Five monitoring wells would be installed around the Project site to check of the water quality for processing and environmental authorities.

18.5 BUILDINGS

Infrastructure buildings will be constructed on the Project site to support GMV's various operations (excludes contractors and process related) and include the following:

1. Administration (furnished - offices, meeting rooms, lunchroom, communications and first aid/training)
2. Employee Changehouse
3. Guardhouse at site front access gate

All buildings will be prefabricated structures.

19.0 MARKET STUDIES AND CONTRACTS

No contractual payable metal rates have yet been negotiated with smelters and/or refineries for treating the gold doré bars produced at Mexican Hat. SE has used typical rates based on industry experience or published guidelines. The payable rate for gold was set at 99.5%. A cost of $5.00 per payable troy ounce of gold was used for refining, transportation, and insurance.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 INTRODUCTION

The Project will require various state and federal authorizations, licenses and permits for Project construction, operation, closure, and post-closure. Comprehensive environmental and socioeconomic 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 762 to 1219 m (2,500 to 4,000 feet) above mean 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 because 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 (ADWR) has identified a variety of groundwater basins in southeastern Arizona. The 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 130 km^2 (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.

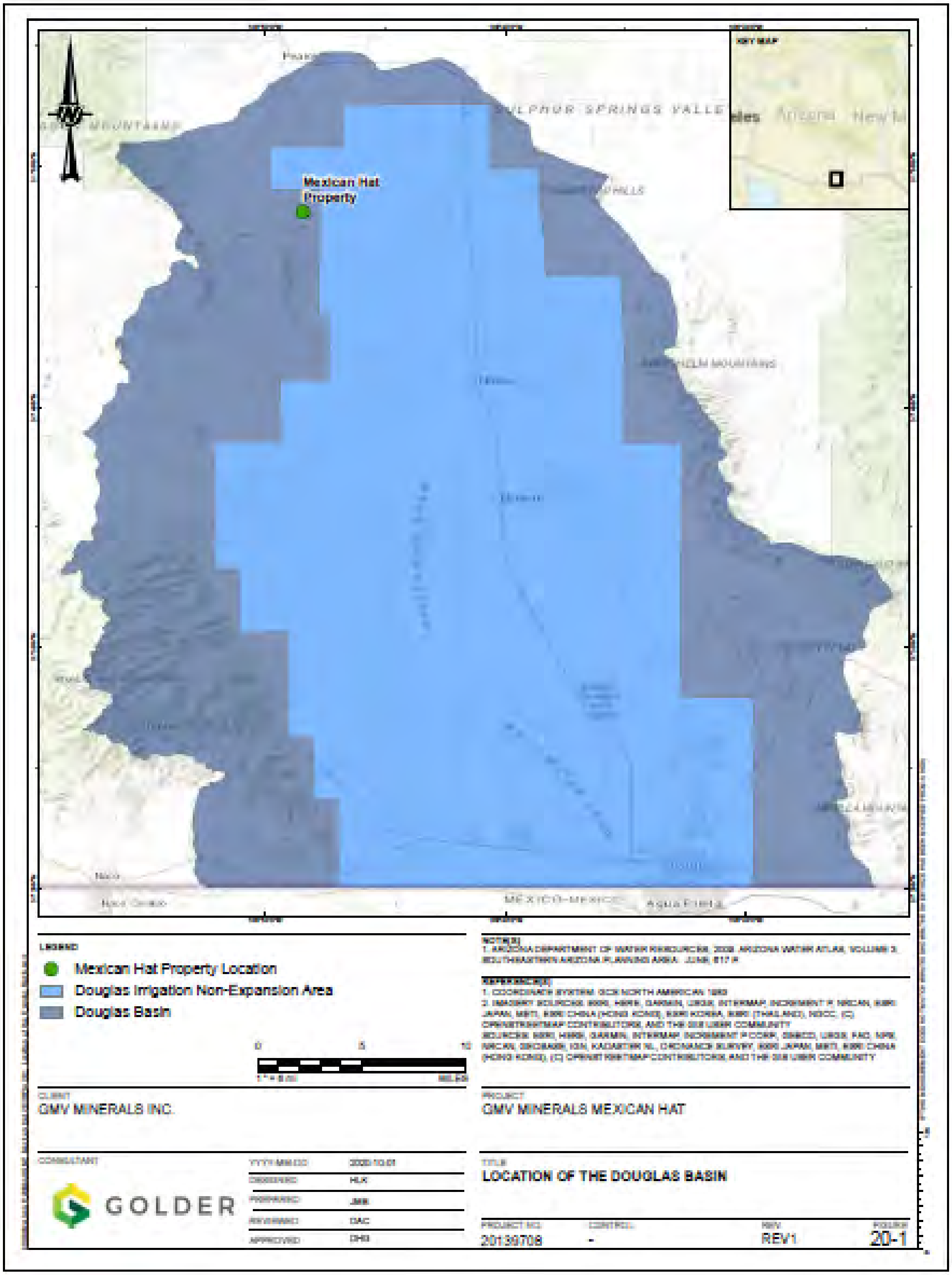


Figure 20-1: Location of the Douglas Basin

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 groundwater 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 pumping has altered flow directions in the vicinity of Elfrida where a cone of depression has developed. A recent groundwater flow model developed by ADWR (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 composed 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 1,219 m (4,000 feet) below ground surface in the deeper, central portion of the basin. Data regarding the depth of the basin fill in the Douglas Basin was not encountered at the time of this report. It is assumed to be less than 122 m (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 drill hole completed and that the depth to groundwater is generally less than 50 m (164 feet) below ground surface (Dave Webb, personal communication, 2018). The groundwater encountered in the exploration boreholes is believed to be structurally controlled (Clive Bailey, personal communication, 2020), and is not related to the alluvial basin that is the primary water source for the Douglas Basin.

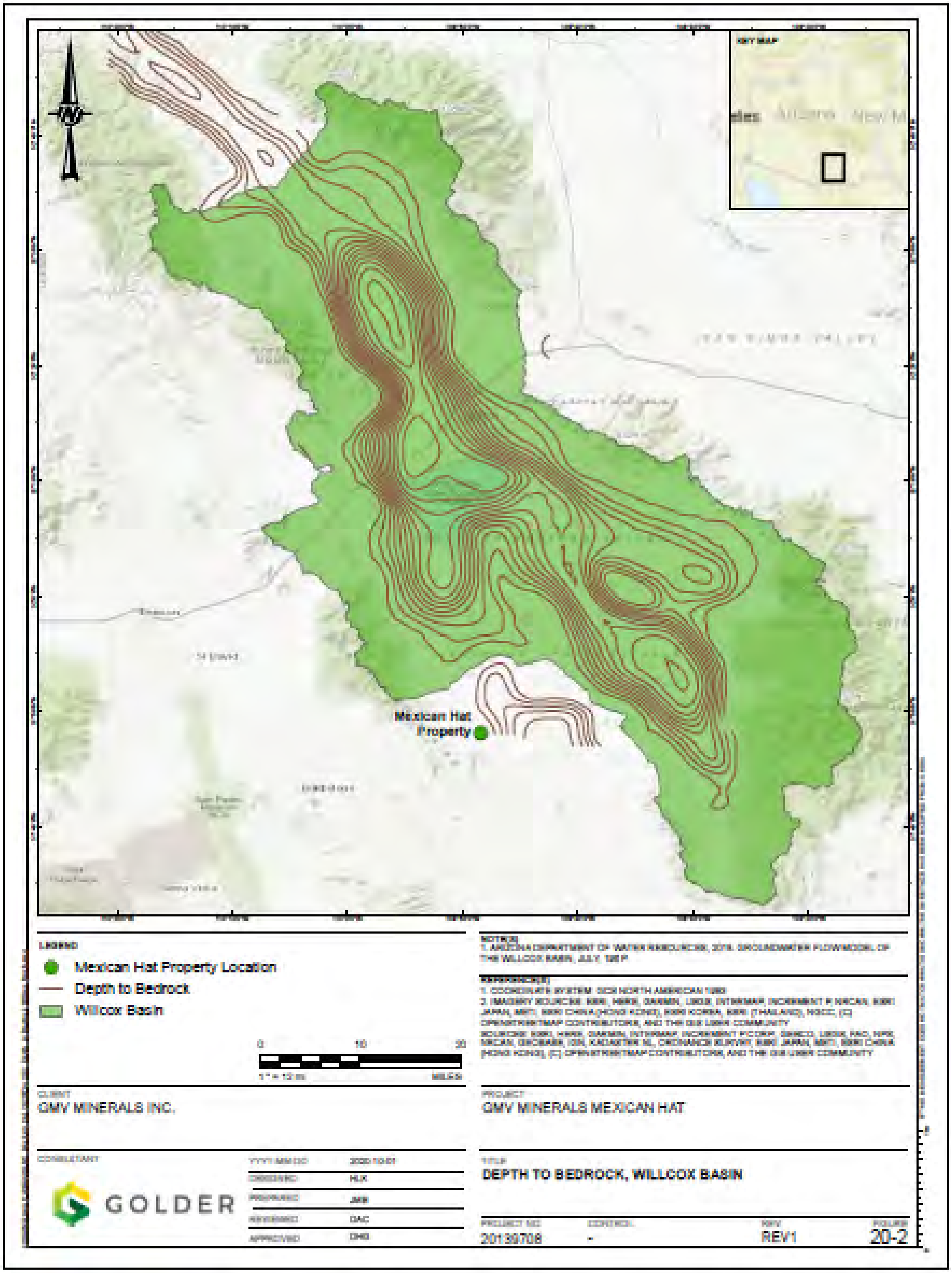


Figure 20-2: Depth to Bedrock, Willcox Basin

Groundwater recharge occurs mainly in washes and along mountain fronts and is estimated at 15,500 to 22,000 acre-feet per year (AFA) (ADWR 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 gallons per minute (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 impact of groundwater pumping 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).

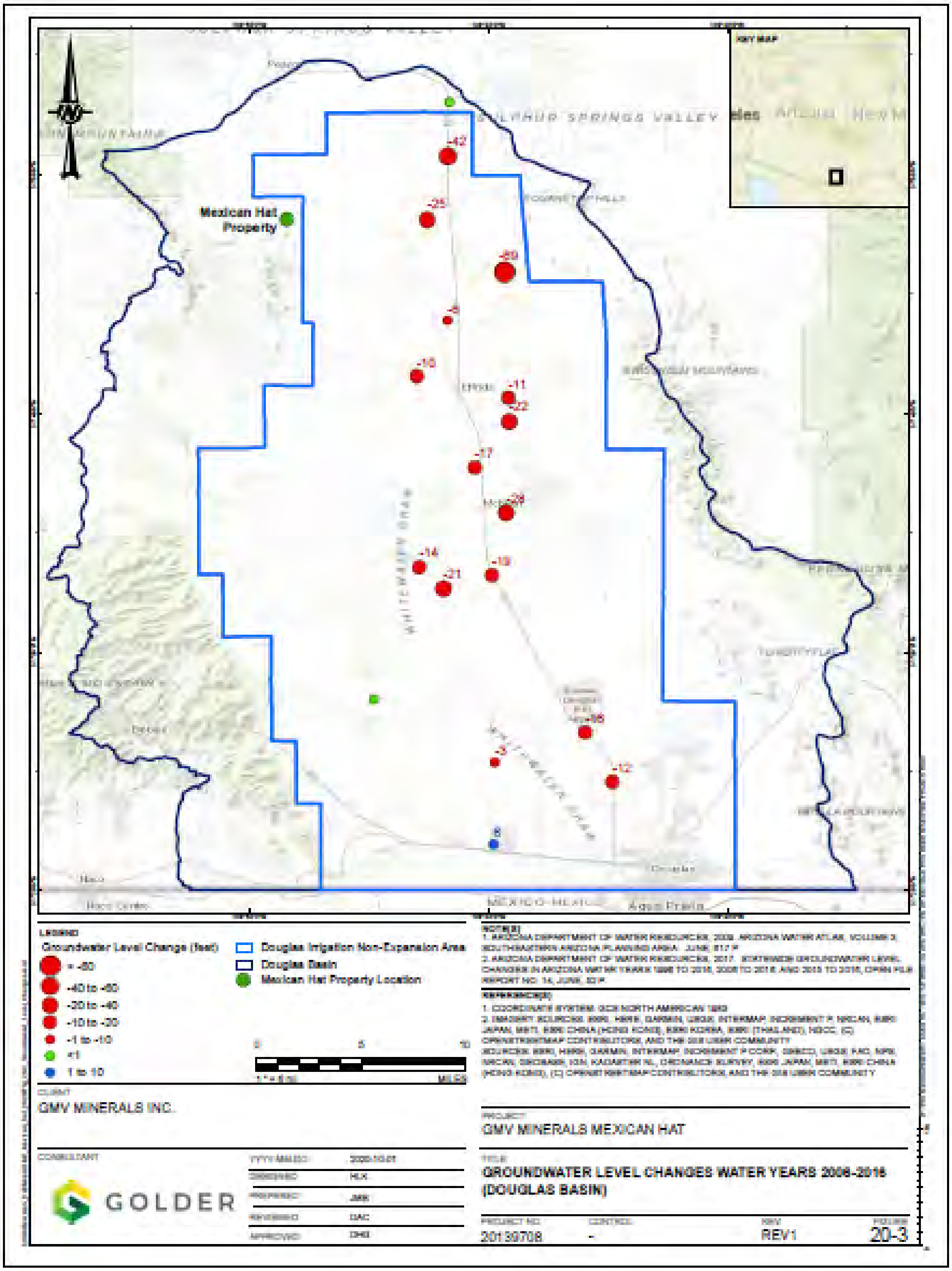


Figure 20-3: Groundwater Level Changes Water Years 2006-2016 (Douglas Basin)

The Douglas Basin (Figure 20-1) includes a portion of the basin where the use of groundwater is managed under the 1980 Groundwater Management Act, which established the Douglas INA, as shown on 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 of withdrawal and for non-irrigation purposes. The 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 in T19S, R25E, sections 1, 2, and 3 are within the Douglas INA. The boundaries of the Douglas INA are shown on Figure 20-4. Assuming a water requirement of 700 gpm (1,130 AFA) for mining operations and the possibility a future water supply well may need to be located within the boundaries of the INA, if an inadequate water supply is available at the Project, the Project will be considered non-exempt; and therefore, subject to reporting requirements. The Douglas INA is administered by ADWR staff in the Tucson, Arizona, office.

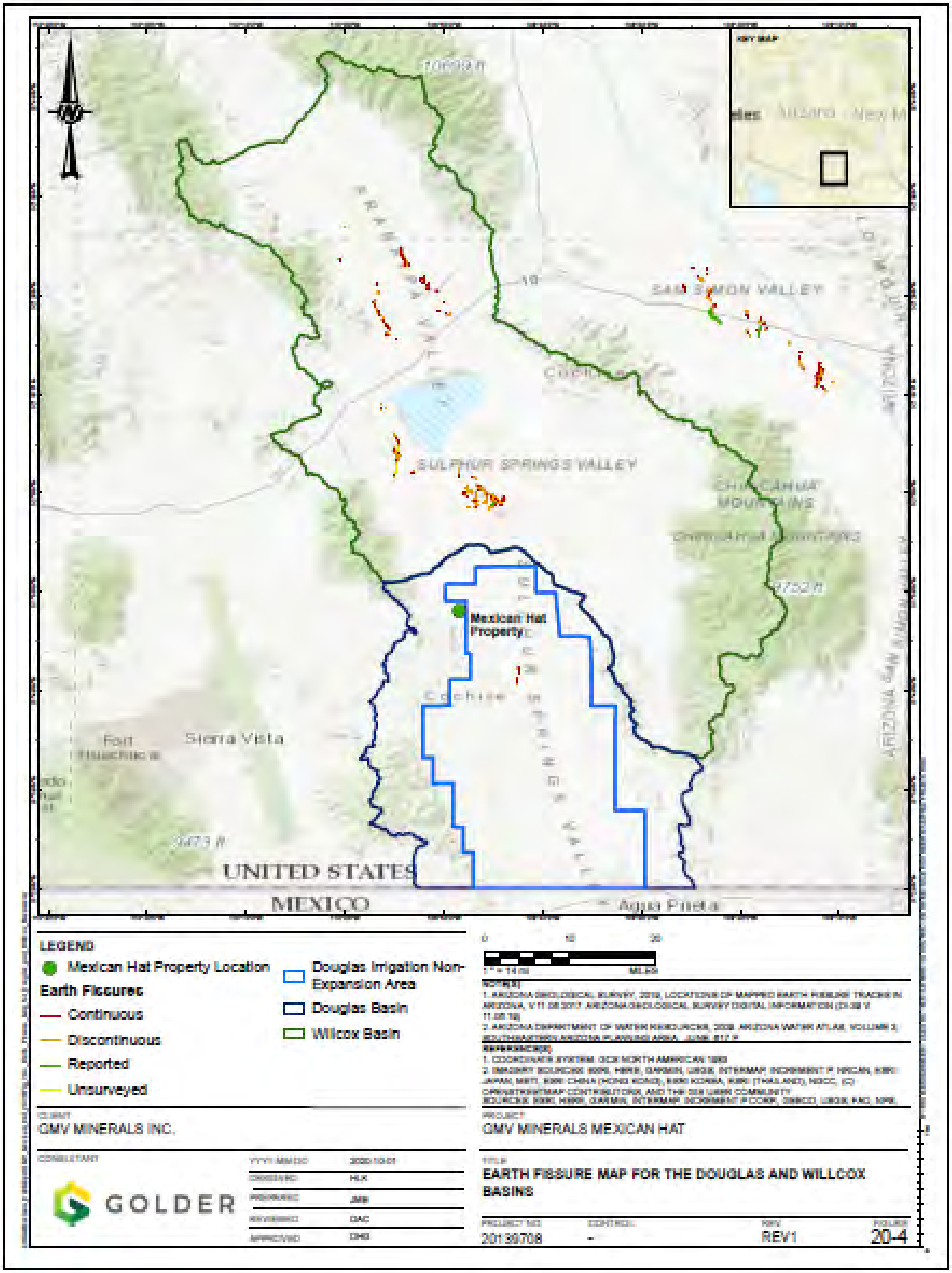


Figure 20-4: Earth Fissure Map for the Douglas and Willcox Basins

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 a sufficient water supply will be available locally (whether within the current mining claims property), and it is recommended that a hydrogeologic study be initiated in the next phase of Project advancement.

20.4 GEOLOGIC HAZARDS

A known geologic hazard in the region is earth fissures, which have resulted from ground subsidence caused by groundwater withdrawal. As the ground subsides unevenly, stress along the basin margins lead to earth fissure formation. Earth fissures are documented in Cochise County, especially in the Willcox Basin (Figure 20-4). None of the documented fissures are within the immediate vicinity of the Project site; however, the presence of fissures in the region should be monitored, especially fissures that occur near, or under, transportation routes.

20.5 BASELINE STUDIES AND POTENTIAL IMPACTS

Future mining operations will create waste rock and spent mineralized material and expose lithologies that could have the potential to leach metals and metalloids or generate acid. Acid rock drainage and metals leaching could affect the quality of 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 mineralized material), is required to evaluate the potential for future environmental impacts, especially on a long-term basis. Results of the geochemistry testing program should be used to guide the detailed 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, and flora and fauna) are all required to document pre-mining conditions and for environmental permitting. If a weather station has not been installed at the 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. Background groundwater quality 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.

To identify stakeholders, socioeconomic and land use baseline studies will also need to be performed.

20.6 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 several 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 US 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.

20.7 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; 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 environmental 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 Area-Wide 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~~ wastewater treatment 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.
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.
Drinking Water System Approval to Construct and Approval of Construction	Arizona Department of Environmental Quality	Safe Drinking Water Act	If applicable.
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 (APP), issued and administered by the Arizona Department of Environmental Quality (ADEQ). The two key requirements of the APP are to (1) meet numeric aquifer water quality standards (AWQS) at the point of compliance established in the APP, and (2) demonstrate that the discharging facilities subject to individual APP requirements have been designed to meet Best Available Demonstrated Control Technology (BADCT) standards, which reduces the discharge of pollutants to the greatest degree practicable before reaching the aquifer. The Project facilities subject to individual APP requirements include the heap leach pad, the process water and non-stormwater surface impoundments, and potentially the waste rock stockpiles. These facilities will be permitted under an area-wide APP, which allows multiple facilities to be included in a single permit. The onsite wastewater facilities can either be incorporated into the area-wide APP or permitted separately under one or more general APPs. If the Project's discharging facilities are designed to meet the prescriptive BADCT requirements established in the Arizona Mining BADCT Guidance Manual (ADEQ, undated), then the APP application review time by ADEQ will be expedited.

A portion of the proposed mining facilities will be located on federal property administered by the BLM. The BLM will be responsible for initiating the NEPA process and deciding whether the criteria of a significant impact to the environment is met. The BLM will decide to request an Environmental Assessment (EA) and/or an Environmental Impact Statement (EIS). The EA is a concise review document that includes the purpose and need of the Project, any alternatives, and a brief review of the impacted environment. The EA will either produce a "Finding of No Significant Impact" or, if significant environmental impacts appear likely, trigger an EIS. An EIS is a much more comprehensive document that requires everything an EA would require, plus a much more comprehensive discussion of the reasonable alternatives and cumulative impacts. The NEPA regulations have recently undergone a rule modernization under the White House's Council on Environmental Quality. A key change is a presumptive time limit of two years for completion of an EIS and one year for completion of an EA (www.whitehouse.gov/ceq/nepa-modernization). The final rule was announced on July 15, 2020.

The Project design is anticipated to impact surface water drainages on the site (Figure 20-5 in the PEA). Mapping of the drainage boundaries will be required to confirm whether designs and drainage features meet the "no disturbances" criteria of the regulatory requirements impact, especially drainage systems that qualify as "waters of the U.S" per the criteria of the US Corp of Engineers. Where facilities create a disturbance of a jurisdictional drainage, then a 404 permit will also be required.

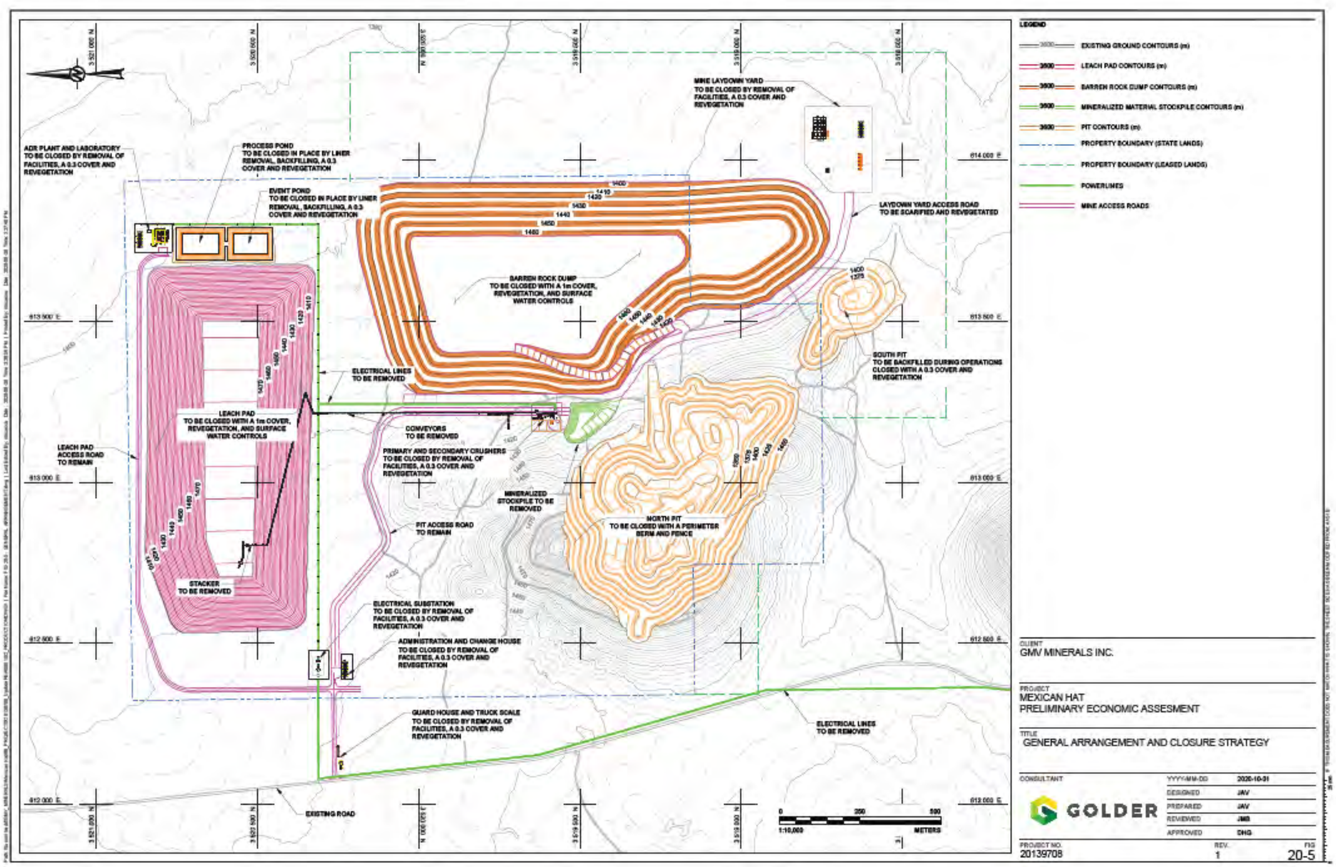


Figure 20-5: General Arrangement and Closure Strategy

The timing to plan and collect baseline data and to develop models required by permitting authorities may take several years, with a minimum of one to two 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, 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; however, the new NEPA ruling should improve the permitting timeframes. The key drivers to the permitting schedule are likely to be NEPA and the APP submittals and the approval process.

20.8 SOCIOECONOMIC STUDIES AND IMPACTS

The property is in the Turquoise Mining District, where historical land usage has been mining, agriculture and ranching. The closest community is Pearce, and there are several other small communities in the area, such as Sunsites, plus larger areas, such as Willcox, Benson, and Sierra Vista, which could be impacted by the development of the Project. Within the region, concerns about sustainable water supplies, water level declines, increased agricultural demand and environmental protection activities have been identified by community watershed groups within the ADWR Southeastern Arizona Planning Area (ADWR, www.azwater.gov /water-initiative/planning-area-process#cochise) that is part of the Arizona Water Initiative (ADWR 2014). It is anticipated that water will be a primary concern by stakeholders.

A baseline socioeconomic study will be required to advance to PFS. The baseline study can be primarily a desktop study based on federal and county websites with a visit to the Cochise County planning department. It is anticipated that the City of Tucson will be a primary source for workforce and supplies, with some minor contribution from the town of Benson and other surrounding communities.

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.

20.9 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, which is to prevent long-term environmental impacts from post-closure facilities. The APP requirements are published in the Arizona Administrative Code, Title 18, Chapter 9. Acceptable financial assurance mechanisms are required to cover closure and post-closure costs (R18-9-A203).

The second is the closure of non-APP facilities, such as buildings and infrastructure, which are to be reclaimed in accordance with the Mined Land Reclamation rules (Arizona Administrative Code, Title 11, Chapter 2). This rule requires the development of reclamation plans that will ensure safe and stable post-mining land use. Regrading and resurfacing needs, if any, will be completed in accordance with industry-standard 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.

In accordance with the general work schedule of the Mexican Hat Project, should no additional mineralization be found, the permanent closure phase will begin in year ten (that is, at the end of the ten year LOM) when no further extraction is planned from the open pits. In compliance with permitting regulations, a detailed closure plan will be developed prior to the closure period. It is expected that the land usage post-closure will be natural habitat for wild flora and fauna, and land for livestock grazing.

The closure strategy involves returning the mine site and affected areas to viable, and wherever practicable, self-sustaining ecosystems that are compatible with a healthy environment. Key activities of closure will be decommissioning equipment and waste management; demolition of physical structures and management of infrastructure; characterization and mitigation of contaminated soils; regrading and contouring to allow for stormwater drainage; and revegetation of disturbed land.

Conceptual-level closure methods and associated costs were developed based on the current facilities layout (Figure 20-5 in the PEA). All structures will be removed. The permanent impacts will be the North Pit (note that the South Pit will be backfilled during operations), the waste rock storage facilities and material placed in the Heap Leach Facility. The effects of mining are irreversible, although through planned restoration and revegetation methods, some effects will be improved. The closure costs assume that no impacts have occurred to the environment that will require long-term mitigation measures. The final site configuration after closure is presented on Figure 20-6 in the PEA, which shows the remaining facilities and areas of soil cover and revegetation.

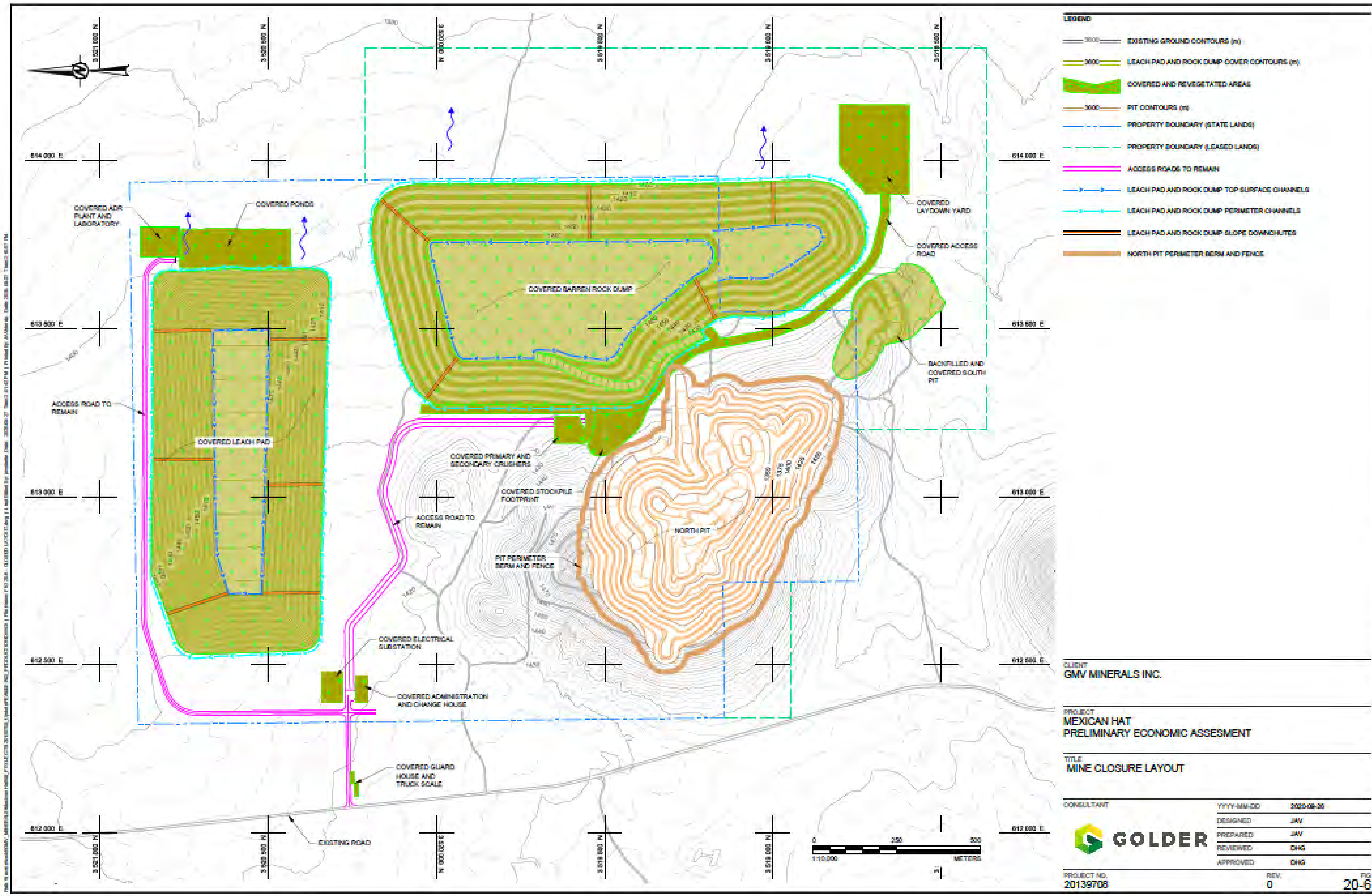


Figure 20-6: Mine Closure Layout

The closure methods are summarized as follows:

- **North Pit:** Berms will be constructed around the perimeter of the pit for safety and for surface water diversions. Warning signs will be installed. A chain-link fence will be installed around the facility. At this time, it is unknown whether a pit lake will form and whether access roads will be required for post-closure monitoring.
- **Heap Leach:** It is assumed that the facility will be allowed to drain for 5 years after cessation of active leaching. After it is no longer economically viable to process solution, the draindown solution will be treated or managed by recirculating, using active evaporation in the solution ponds and natural evaporation. The facility will be regraded and contoured to shed stormwater. The top will receive a closure cover to limit infiltration of surface water and generation of additional acidic water and metals leaching. Runoff and erosion controls will be installed (swales, v-ditches and downchutes) using low-permeability covers and riprap. The side slopes will be covered with a material that decreases erosion. Drainage channels will be constructed to direct stormwater off the top. Surface water diversions will be placed to minimize erosion at the base of the heap leach.
- **Waste Rock Storage Facilities:** The facilities will be graded and recontoured to prevent ponding with the final slopes designed to meet stability requirements. The facilities will be covered with a material that decreases erosion and subsequently revegetated. It is assumed that no seepage will be present that requires management.
- **Process Plant and Related Facilities:** All plant and related facilities will be dismantled, or demolished. Foundations will be removed, and excavated areas will be filled to restore drainage.
- **Roads:** Several roads will remain to access the property for closure and environmental monitoring. Internal roads will be leveled and graded to facilitate vegetation growth. Cover materials will be used, as needed.
- **Environmental monitoring:** Post-closure monitoring is assumed for 20 years or until non-hazardous conditions are achieved for any discharge from the remaining facilities, and groundwater and surface water quality meets applicable regulatory standards.
- **Covers:** At this time, the cover materials and revegetation strategy have not been defined. It is assumed that all affected areas will receive a 0.30-m cover, except the heap leach and waste rock storage facility will receive a 1-m cover. All affected areas, including the heap leach and waste rock storage facilities, will be revegetated with preferably native species, including the side slopes.
- Indirect costs of 45 percent were applied to the closure cost estimate. Indirect costs include engineering and design, administration, contractor profit, insurance, and contingencies.

The closure cost was estimated at $25,182,000 (USD) (see Table 20-2).

Table 20-2
Closure Costs Summary

Area	Closure Cost ($USD)
Plant and Auxiliary Facilities Closure	$426,200
North Open Pit Closure	$589,800
South Open Pit Closure	$91,800
Barren Rock Dump	$4,243,000
Leach Pad Closure	$3,751,200

Table 20-2 Closure Costs Summary	
Area	**Closure Cost ($USD)**
Infrastructure Closure	$379,800
Leach Pad Solution Management	$2,000,000
Waste Disposal	$182,500
Inspection and Maintenance	$1,200,000
Site Wide Additional Costs (Indirect Costs)	$5,788,800
Total Direct Costs	**$12,864,000**
Total Indirect Costs	**$5,789,000**
Total	**$18,653,000**
Contingency (PEA Level) ± 35%	**$6,529,000**
Grand Total	**$25,182,000**

Golder followed the recommendation of the US Forest Service (USFS) to apply 35 percent for a preliminary cost estimate (USFS 2004). Indirect costs were estimated at 45 percent, which follows the guidance of the US Bureau of Land Management (BLM 2014). A conceptual closure plan has not been prepared. The PEA text includes a description of the conceptual closure methods and closure cost estimate. If the Project is advanced to PFS and FS levels, the methods and costs should be refined.

20.10 CONCLUSIONS

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

The availability of a water source adequate for mining operations is unknown. This is identified as a Project risk.

20.11 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 also be initiated.

To optimize the Project advancement, baseline studies should overlap with the metallurgical testwork, geotechnical study, and water supply study. In particular, it is recommended that the geochemistry study to evaluate the long-term environmental impacts be started as soon as possible, that drilling programs be combined to utilize exploration boreholes as groundwater monitoring points, and that any geotechnical drilling be combined with hydrogeologic requirements. To establish an adequate water source, a high priority will be the hydrogeologic study.

To oversee these activities, the company will need to contract, or hire, an environmental manager as well as a community relations manager.

21.0 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COSTS

The estimated capital cost to design, procure, construct and commission the Mexican Hat Project facilities is $67.847 million. The initial capital cost estimate is summarized in Table 21-1.

Table 21-1: Initial Capital Cost Estimate for Mexican Hat Project

Cost Components	Mine & Crushing	Leach Pad, Ponds & Pipelines	ADR, BOP & Infrastructure	Substation & Power	Total Capital Cost
Description	**Cost (USD)**	**Cost (USD)**	**Cost (USD)**	**Cost (USD)**	**Cost (USD)**
Directs					
Mechanical Equipment	-	2,712,000	6,741,000	-	9,453,000
Civil	-	7,370,000	584,000	83,000	8,037,000
Foundations	-	-	646,000	200,000	846,000
Structures	-	-	378,000	125,000	503,000
Buildings/Laboratories	-	-	1,359,000	-	1,359,000
Insulation	-	-	-	-	-
Piping	-	3,050,000	2,270,000	-	5,320,000
Electrical	-	-	706,000	1,838,000	2,544,000
Instruments	-	-	353,000	-	353,000
Miscellaneous	-	-	182,000	-	182,000
Subtotal Directs	**-**	**13,132,000**	**13,219,000**	**2,246,000**	**28,597,000**
Indirects					
Contractor Indirect	-	1,114,000	1,559,000	476,000	3,149,000
Construction Equipment	-	557,000	779,000	238,000	1,574,000
Surveying & Testing Svcs	-	139,000	225,000	60,000	424,000
EP Services	-	550,000	1,182,000	218,000	1,950,000
Construction Mgmt	-	446,000	934,000	135,000	1,515,000
Vendor Reps	-	68,000	151,000	21,000	240,000
Spare Parts	-	34,000	76,000	10,000	120,000
Initial Fills	-	25,000	250,000	10,000	285,000
Commissioning	-	104,000	146,000	45,000	295,000
Freight	-	137,000	501,000	45,000	683,000
Crushing Equipment-mob	3,000,000	-	-	-	3,000,000
Contractor Mining	2,430,000	-	-	-	2,430,000
Preproduction	4,300,000	-	-	-	4,300,000
Owner's Cost, incl Royalty	2,577,000	-	3,490,000	-	6,067,000
Taxes	157,000	217,000	529,000	47,000	950,000
Subtotal Indirects	**12,464,000**	**3,391,000**	**9,822,000**	**1,305,000**	**26,982,000**
Contingency	2,729,000	4,042,000	4,609,000	888,000	12,268,000
Total Cost (USD)	**15,193,000**	**20,565,000**	**27,650,000**	**4,439,000**	**67,847,000**

21.1.1 Accuracy

The order of magnitude capital cost estimate has been developed to a level that is sufficient to assess the Project's concept, development options and overall potential. After inclusion of the PEA contributors' recommended contingencies, the capital cost estimate is considered to have an accuracy in the range of minus 20% to plus 35%.

21.1.2 Currency

The estimate is expressed in Q2 2020 United State Dollars. No provision is included to offset future escalation.

21.1.3 Scope

The Project consists of an open pit mine and, associated processing, infrastructure, storage and waste facility, and site services and utilities. The processing facilities, as currently designed, consist of a heap leach facility (HLF), ADR plant and an SX-EW plant. The facilities are designed to process 10,000 t/d of gold mineralized material. The expected LOM is 10 years.

The capital cost estimate is based on preliminary plant and facilities layout and design. The document used to prepare the estimate include:

- Preliminary process flow diagram
- Mechanical equipment list
- Preliminary site layout plan
- Budgetary quotations from vendors
- In-house historical data

SE provided capital cost for the ADR plant, infrastructure and utilities, Tierra Group provided the cost for the HLF and waste rock storage facility, and MDA provided pre-production mining cost.

21.1.4 Exclusions

Items not included in the capital cost estimate are:

- Crushing equipment cost
- Mining equipment cost
- Phase 2 HLF development
- Sunk cost
- Escalation beyond Q2 2020
- Feasibility study cost
- Closure and reclamation cost (included in cash flow)
- Mine roads
- Owner’s cost for environmental, land acquisition, permitting, etc.
- Allowance for special incentives (schedule, safety, etc.)
- Working capital, sustaining capital (included in cash flow)
- Interest and financing cost
- Force majeure occurrences, such as risk due to labor disputes, permitting delays, etc.

21.1.5 Estimating Methodology

Capital cost for the Project has used a “distributed percentage factoring” technique often employed when developing an estimate for a process facility at a preliminary stage where there is a lack of design data and specific requirements from which to base cost on. With this factoring technique, the supply cost of the mechanical equipment for the facilities is used as the basis for calculating the overall cost of the facility. Various percentages of the equipment cost are then applied to obtain values for each of the prime commodity accounts which include earthwork, concrete, structural steel, mechanical, piping, electrical and instrumentation.

All direct costs, other than the mechanical equipment cost, have been factored and distributed as percentages of the mechanical equipment cost.

Costs assume new equipment, material, and services will be purchased on a competitive basis with lump sum or unit rate contracts, and installation contracts will be awarded in well- defined packages.

21.1.6 Contingency

An overall project contingency of 22% or approximately $12.3 M is included in the capital cost in recognition of the degree of detail on which the estimate is based.

Contingency is an allowance to cover unforeseeable costs that may arise during the project execution and which reside within the scope-of-work but cannot be explicitly defined or described at the time of the estimate due to lack of information. It is assumed that contingency will be spent; however, it does not cover scope changes or project exclusions.

21.1.7 Mining

MDA has estimated the mining capital required for mining operations as shown in Table 21-2. The total initial mining capital is estimated to be $7,801,000 and sustaining mining capital is estimated to be $1,131,000. Total mining capital for the LOM is estimated to be $8,932,000.

This capital is minimized using a contract mining scenario. Most of the initial mining capital cost is due to preproduction mining during year -1. Preproduction includes operating costs incurred in year -1 for both contract mining ($4,899,000) and owner mining costs for personnel, supplies, and miscellaneous ($599,000).

Other mining related capital include:

21.1.7.1 Owner Mining Capital

- $105,000 for engineering and office equipment;
- $20,000 for a bladder which will contain dust suppression chemicals to be mixed with water in water trucks;
- $150,000 for GPS stations and surveying equipment; and
- Light vehicles at an initial capital cost of $196,000 and replacement costs of $196,000 in year 5.

21.1.7.2 Contract Mining Capital

- Pioneer mining is done to establish mining areas with $880,000 for initial mining to the top of the hill, and to start mining in year -1, and $220,000 to establish roads to phases 2 and 3 during year 2;
- Mobilization incurs initial capital of $550,000 in year -1 along with $65,000 in mobilization costs during year 5 due to additional haulage requirements. Demobilization costs of $650,000 are added to the end-of-mine ("EOM") life in year 10; and
- $1,000,000 in initial capital is added to year -1 for the establishment of the contractor maintenance facility on site.

Table 21-2: Mine Capital Summary

Owner Mining Capital	**Units**	**Yr_-1**	**Yr_1**	**Yr_2**	**Yr_3**	**Yr_4**	**Yr_5**	**Yr_6**	**Yr_7**	**Yr_8**	**Yr_9**	**Yr_10**	**Total**
Engineering & Office Equipment	KUSD	105	-	-	-	-	-	-	-	-	-	-	$ 105
Water Storage (Dust Suppression)	KUSD	20	-	-	-	-	-	-	-	-	-	-	$ 20
Base Radio & GPS Stations	KUSD	150	-	-	-	-	-	-	-	-	-	-	$ 150
Light Vehicles	KUSD	196	-	-	-	-	196	-	-	-	-	-	$ 393
Total Owner Mining Capital	KUSD	471	-	-	-	-	196	-	-	-	-	-	$ 668
Contractor Capital	**Units**	**Yr_-1**	**Yr_1**	**Yr_2**	**Yr_3**	**Yr_4**	**Yr_5**	**Yr_6**	**Yr_7**	**Yr_8**	**Yr_9**	**Yr_10**	**Total**
Pioneer Road	KUSD	$ 880	$ -	$ 220	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 1,101
Mob / Demob	KUSD	$ 550	$ -	$ -	$ -	$ -	$ 65	$ -	$ -	$ -	$ -	$ 650	$ 1,265
Shop	KUSD	$ 1,000	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 1,000
Total Capital	KUSD	$ 2,430	$ -	$ 220	$ -	$ -	$ 65	$ -	$ -	$ -	$ -	$ 650	$ 3,366
Preproduction Capital													
Owner Mining Personnel	K USD	$ 437	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 437
Owner Supplies and Misc.	K USD	$ 163	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 163
Total Owners Mining Costs	K USD	$ 599	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 599
Contractor Mining Cost	K USD	$ 4,300	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 4,300
Total Mine Preproduction Cost	K USD	$ 4,899	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 4,899
Total Mining Capital	***K USD***	***$ 7,801***	***$ -***	***$ 220***	***$ -***	***$ -***	***$ 261***	***$ -***	***$ -***	***$ -***	***$ -***	***$ 650***	***$ 8,932***

21.1.8 Processing

The capital costs for the processing areas are estimated at $48.3 M. These costs are inclusive from the reclaim feeders under the crushed material stockpile, conveyors from stockpile to the stacker conveyor at the heap leach pad area, heap leach pad construction and liners, ADR plant and associated building and facilities. The capital costs have been estimated into direct and indirect cost areas. A contingency has been applied to the cost estimate. Table 21-3 below summarizes the processing capital cost estimate. For processing.

Table 21-3
Processing Capital Cost Estimate Summary

Process Plant Cost Components	**Leach Pad, Ponds & Pipelines**	**ADR & Process Facilities**	**Total Capital Cost**
Description	**Cost (USD)**	**Cost (USD)**	**Cost (USD)**
Directs			
Mechanical Equipment	2,712,000	7,853,000	10,565,000
Civil	7,370,000	677,000	8,047,000
Foundations	-	765,000	765,000
Structures	-	448,000	448,000

Table 21-3 Processing Capital Cost Estimate Summary			
Process Plant Cost Components	**Leach Pad, Ponds & Pipelines**	**ADR & Process Facilities**	**Total Capital Cost**
Description	**Cost (USD)**	**Cost (USD)**	**Cost (USD)**
Buildings	-	1,452,000	1,452,000
Insulation	-	-	-
Piping	3,050,000	2,687,000	5,737,000
Electrical	-	836,000	836,000
Instruments	-	418,000	418,000
Miscellaneous	-	215,000	215,000
Subtotal Directs	**13,132,000**	**15,351,000**	**28,483,000**
Indirects			
Contractor Indirect	1,114,000	1,750,000	2,864,000
Construction Equipment	557,000	875,000	1,432,000
Surveying & Testing Svcs	139,000	225,000	364,000
EP Services	550,000	1,368,000	1,918,000
Construction Mgmt	446,000	1,079,000	1,525,000
Vendor Reps	68,000	179,000	247,000
Spare Parts	34,000	90,000	124,000
Initial Fills	25,000	250,000	275,000
Commissioning	104,000	164,000	268,000
Freight	137,000	593,000	730,000
Taxes	217,000	624,000	841,000
Subtotal Indirects	**3,391,000**	**7,197,000**	**10,588,000**
Contingency	4,042,000	5,208,000	9,250,000
Total Cost (USD)	**20,565,000**	**27,756,000**	**48,321,000**

The capital costs for the processing plant areas excludes the crushing plant.

21.1.9 Heap Leach Facility

Preliminary site infrastructure associated with the HLF and ponds were evaluated, and a conceptual arrangement was defined for the basis of the capital cost estimate. HLF construction costs were developed based on supplier quotes:

- Earthwork (Titan Construction);
- GCL (Cetco);
- Geomembrane (AGRU America); and
- Piping system (ADS).

Table 21-4 summarizes the initial capital cost for Phase 1.

Table 21-4 HLF Initial Cost (Phase 1)	
Tasks	**Cost (USD$)**
Phase 1 HLF	$7,369,724
Phase 1 Stacking System	$1,324,000
Phase 1 Contingency (30%)	$2,608,117
Phase 1 (Total)	**$11,301,841**

21.1.10 Sustaining Capital

The LOM sustaining capital is estimated at $13.027 M for mining, processing, administration, and expansion of the heap leach pad as summarized in the below Table 21-5. These costs include demobilization of the mining and crushing facilities as indicated in year 10 of operations.

Table 21-5
LOM Sustaining Capital (US$000's)

Description	Units	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Totals
Mining	$000's	0	220	0	0	261	0	0	0	0	650	1,131
Heap Leach Pad Expansion (Phase 2)	$000's	0	200	11,096	0	0	0	0	0	0	0	11,296
Administration	$000's	0	0	0	0	120	0	0	0	0	0	120
Processing (Crushing)	$000's	0	0	0	0	80	0	0	0	0	400	480
Total Sustaining Costs	**$000's**	**0**	**420**	**11,096**	**0**	**461**	**0**	**0**	**0**	**0**	**1,050**	**13,027**

Note: Year 10 costs are for demobilization of contractors.

21.1.11 Infrastructure

The capital cost for infrastructure is estimated at $1.235 M as summarized in Table 21-6 below:

Table 21-6 Infrastructure Capital Cost Summary	
Infrastructure	**Capital Cost $**
Access Road Improvement (0.5 km)	80,000
Pre-Fab Buildings (Admin: Furnished & Communications; Changehouse; Furnished)	450,000
Guard house, Site Security Fencing & Truck Scale	405,000
Fresh & Fire Water System with Pumps, Piping/Storage Tanks/Monitoring Wells	300,000
Total	**$1,235,000**

21.1.12 Owner's Costs

The Owner's costs are estimated at $4.567 M during the two year preproduction period as summarized in Table 21-7 below.

Table 21-7 Owner's Capital Cost Summary		
Owner's Cost Description	**Owner's Costs ($000s)**	
	Year -2	**Year -1**
Mining		
Engineering and Office Equipment	0	105
Water Storage (Dust Depression)	0	20
Base Radio and GPS Stations	0	150
Light Vehicles	0	196
Mining Personnel	0	437
Mining Supplies and Miscellaneous	0	169
Total Mining Owner's Cost	**0**	**1,077**
Processing and Administrative		
Owner's Project employees	985	1,460
Owner's expenses	150	315
Communication Systems	80	25
Hernandez Royalty Buy-Back	0	1,500
Mobile equipment/rentals/leases/contracts	120	355
Totals Processing and Administration	**1,335**	**3,655**
Totals Owner's Costs	**1,335**	**4,732**

21.1.13 Closure and Reclamation Costs

The conceptual closure cost is estimated at US$25,182,000 at the end of the mine life.

21.1.14 Assumptions & Exclusions

The following assumptions have been made in developing the Project's capital cost:

1. There are no capital costs for major equipment associated with mining.
2. No additional large equipment is required for the mining contractor.
3. There is water to the property line that can be used to supply fresh water to the plant.

21.1.15 Preliminary Project Execution and Schedule

Project execution will follow a typical EPCM approach. The execution timeframe considered is approximately 24 months from notice to proceed through commissioning completion. Project ramp-up will be commensurate with heap leaching pad development. Equipment delivery will drive the timeline for completion of the project.

21.2 OPERATING COSTS

The operating costs for the Project are based on a combination of direct build-up from mining and metallurgical parameters, typical unit consumption and costs for similar operations, and factoring. The direct operating costs average $15.30/t of material leached on the heap leach pad, equivalent to $951/Au oz recovered in doré as summarized in Table 21-8.

Table 21-8 LOM Operating Costs					
Production	**Estimated Operating Costs ($000s)/Year**				**$/Au Oz**
Year	**Mining (1)**	**Process (1)**	**G&A**	**Total**	**Recovered**
1	25,473	23,240	2,730	51,443	$1,152
2	21,510	23,240	2,730	47,480	$817
3	20,226	23,304	2,737	46,267	$716
4	23,354	23,240	2,730	49,324	$855
5	22,582	23,240	2,730	48,552	$1,037
6	31,348	23,240	2,730	57,318	$1,119
7	44,695	23,304	2,737	70,736	$1,034
8	32,427	23,240	2,730	58,397	$1,120
9	23,783	23,240	2,730	49,753	$1,009
10 (2)	5,420	12,400	1,505	19,325	$622
11 (2)	0	530	29	559	$509
Totals LOM Costs	**$250,817**	**$222,217**	**$26,119**	**$499,154**	**$951**
Total $/t leached	**$7.69**	**$6.81**	**$0.80**	**$15.30**	

(1) Includes contractor costs.

(2) Gold in Years 10 and 11 includes continued production from the leaching of crushed gold material placed on the leach pads in previous years.

21.2.1 Mine Operating Costs

Mine operating costs have been estimated based on owner's mine management needs and contractor proposals. The operating cost estimates are shown in Table 21-9. The total LOM mine operating cost is estimated to be $250,817,000 or $2.68/t, not including pre-stripping costs.

Table 21-9 Operating Cost Estimate

Total Mining Cost	Units	Yr_-1	Yr_1	Yr_2	Yr_3	Yr_4	Yr_5	Yr_6	Yr_7	Yr_8	Yr_9	Yr_10	Total
Owner Mining Personnel	K USD	$ -	$ 776	$ 776	$ 776	$ 776	$ 776	$ 776	$ 776	$ 776	$ 776	$ 259	$ 7,247
Owner Supplies and Misc.	K USD	$ -	$ 346	$ 346	$ 346	$ 346	$ 346	$ 346	$ 346	$ 346	$ 346	$ 116	$ 3,232
Total Owners Mining Costs	K USD	$ -	$ 1,123	$ 1,123	$ 1,123	$ 1,123	$ 1,123	$ 1,123	$ 1,123	$ 1,123	$ 1,123	$ 374	$ 10,478
Contractor Mining Cost	K USD	$ -	$ 24,350	$ 20,387	$ 19,103	$ 22,232	$ 21,459	$ 30,226	$ 43,573	$ 31,304	$ 22,660	$ 5,045	$ 240,339
Total Mine Operating Cost	K USD	$ -	$ 25,473	$ 21,510	$ 20,226	$ 23,354	$ 22,582	$ 31,348	$ 44,695	$ 32,427	$ 23,783	$ 5,420	$ 250,817
Owner Mining Personnel	$/t	$ -	$ 0.08	$ 0.09	$ 0.11	$ 0.09	$ 0.10	$ 0.06	$ 0.05	$ 0.06	$ 0.10	$ 0.16	$ 0.08
Owner Supplies and Misc.	$/t	$ -	$ 0.04	$ 0.04	$ 0.05	$ 0.04	$ 0.04	$ 0.03	$ 0.02	$ 0.03	$ 0.05	$ 0.07	$ 0.03
Total Owners Mining Costs	$/t	$ -	$ 0.12	$ 0.14	$ 0.15	$ 0.13	$ 0.14	$ 0.09	$ 0.07	$ 0.09	$ 0.15	$ 0.23	$ 0.11
Contractor Mining Cost	$/t	$ -	$ 2.62	$ 2.49	$ 2.61	$ 2.56	$ 2.67	$ 2.48	$ 2.65	$ 2.53	$ 2.96	$ 3.13	$ 2.56
Total Mine Operating Cost	$/t	$ -	$ 2.74	$ 2.63	$ 2.76	$ 2.69	$ 2.81	$ 2.57	$ 2.72	$ 2.62	$ 3.10	$ 3.36	$ 2.68

21.2.2 Owner's Mining Costs

Owner's mining costs are based on costs for personnel, supplies, and miscellaneous items required to supervise the mining operations. Mine general services costs will include personnel for Engineering and Geology. The Engineering department will assist with all mine production record keeping along with maintaining mine reserves information, short-term and long-term plans, and surveying services. Engineering will be staffed with a Chief Engineer, a senior level Mine Engineer, a Chief Surveyor, and a Surveyor Helper. Overlapping shifts will be utilized as needed to provide seven-day support for mining operations.

The Geology department will be responsible for maintaining estimated resource models and data, geology mapping, and mineralized material control. The department will be led by the Chief Geologist and have one Ore Control Geologist and a Sampler. Salary assumptions for Mine General personnel are shown in Table 21-10.

Table 21-10 Mine General Personnel Salaries

Mine General Personnel	Type	OT %	Hourly	Salary	Burden %	Net Salary
Chief Mine Engineer	Salary	NA		$ 120,000	38%	$ 165,600
Mine Engineer	Salary	NA		$ 100,000	38%	$ 138,000
Chief Surveyor	Salary	NA		$ 65,000	38%	$ 89,700
Surveyor	Hourly	5%	$ 25.00	$ 56,054	38%	$ 77,354
Chief Geologist	Salary	NA		$ 90,000	38%	$ 124,200
Ore Control Geologist	Salary	NA		$ 80,000	38%	$ 110,400
Samplers	Hourly	5%	$ 23.00	$ 51,569	38%	$ 71,166

Additional costs for mine general services assume:

- $2,000 per month in mine general services supplies;
- $15,000 per month of additional services for maintaining access roads and miscellaneous site maintenance utilizing contractor equipment;
- $6,000 per month for Engineering supplies, including surveying supplies ($3,000) and sampling supplies ($3,000); and
- $2,400 per month for software maintenance and support.

The total Mine General Services cost per tonne is estimated to be $0.11/t mined.

21.2.3 Contract Mining Costs

Contract mining costs have been based on contractor budgetary quotations received. The contract quotations were given based on the production schedules provided in Section 16. The quotations were provided along with ANFO and fuel consumption amounts so that adjustments for these costs could be done for sensitivity analysis. ANFO pricing assumes bulk product of ammonium nitrate ("AN") to be mixed with 6% fuel oil by weight. An cost of $0.595 per kilogram and fuel cost of $0.484 per liter was used for the operating cost estimate.

The contractor-specified operating unit costs were provided by year and pit phase. This was applied to the material mined. In addition, a $0.67/t rehandle cost was assumed for long-term stockpile handling and a charge of $0.54/t was quoted for feeding of the crusher. The total contract mining operating cost is estimated to be $2.56/t excluding preproduction mining.

21.2.4 Process Operating Costs

The process operating costs were estimated by first principles as summarized in Table 21-11 below. The total estimated cost is $6.64/t leached on an annualized basis. No contingency has been applied to the process operating costs.

Table 21-11
Annual Processing Operating Cost Estimate Summary

PROCESS PLANT SUMMARY	TOTAL ($/t leached)
Contract Crushing (Excludes power)	2.49
Reagents & Consumables	1.69
Maintenance	0.16
Labor	1.63
Power	0.59
Operating Supplies	0.08
Total Process Operating Costs	**6.64**

<u>Contract Crushing</u>

Bids for contract crushing were obtained from 5 contractors from which SE selected the contractor bid for $2.49/t crushed (leached). The contract operating excludes power which is paid for by the company. The power for crushing is included in the power unit cost of $0.59.

<u>Reagents and Consumables</u>

The reagents required for Mexican Hat are lime and cyanide. The consumption rate for these reagents is based on historical metallurgical testwork from 2015 and 2016 at McClelland Laboratory. Carbon consumption is based on industry standard at 30 g/t. The acid wash and elution reagents are based on the volume of the acid wash and elution columns. Smelting reagents are based on industry knowledge and experience. The reagent requirement is summarized in Table 21-12. Regent costs ae based on data base for recent process in the similar operations in the area.

Table 21-12
Summary of Reagents and Consumables

REAGENT	Unit	CONSUMPTION RATE	Cost/t Leached
Sodium Cyanide	Kg/t	0.3	1.02
Lime	Kg/t	1.5	0.53
Carbon	Kg/t	0.03	0.12
Caustic Soda	Kg/t	0.0001	0.00
Hydrochloric Acid	Kg/t	0.0004	0.00
Anti-Scalant	Kg/t	0.0002	0.00
Smelting Fluxes	Kg/t	0.0002	0.00
Diesel	Kg/t	0.042	0.01
Laboratory Supplies	NAp	NAp	0.01
Total	**$/t leached**		**1.69**

<u>Maintenance</u>

Maintenance costs were estimated for plant vehicles and the conveyors-plant at $560,000 per year. The maintenance cost for the conveyors and ADR plant is based on 5% of mechanical equipment capital cost. The maintenance cost for vehicles is calculated at $60,000 per year.

Labor

Process labor is based on a staffing plan and 4 twelve-hour shift rotation. Labor rates are based on similar operations in the area and a 35% burden rate. Table 21-13 shows a summary for process labor.

Table 21-13
Estimated Labor Operating Costs for Processing

DESCRIPTION	Number	Salary ($/month)	Burden (35%)	Total ($/annum)
Process Plant Labor				
Process Superintendent	1	7,300	2,555	118,260
Administrative Assistant	1	4,853	1,699	78,619
Plant Metallurgist	1	6,620	2,317	107,244
Plant Operations Foreman	4	6,587	2,305	426,838
Conveyor Operations Foreman	1	6,587	2,305	106,709
Conveyor Operators	3	6,070	2,125	295,002
Conveyor Helpers	3	5,255	1,839	255,393
Dozer Operator	1	6,300	2,205	102,060
Heap Leach Pad Operators	3	6,070	2,125	295,002
Heap Leach Pad Helpers	3	5,255	1,839	255,393
CIC Operator	4	6,070	2,125	393,336
Elution Operator	4	6,070	2,125	393,336
Refinery	2	6,070	2,125	196,668
Reagents/Water Operator	1	5,517	1,931	89,375
Laborers	2	4,333	1,517	140,389
Warehouse Clerks	2	4,853	1,699	157,237
Total Process Operations	**36**			**$3,410,861**
Process Plant Maintenance				
Maintenance Superintendent	1	6,933	2,427	112,315
Maintenance Planner	1	6,587	2,305	106,709
Day Foreman	1	6,587	2,305	106,709
Shift Maintenance Mechanics	4	6,070	2,125	393,336
Shift Maintenance Helpers	4	5,900	2,065	382,320
Shift Electrician	4	6,200	2,170	401,760
Millwright (days shift)	2	6,200	2,170	200,880
Instrumentation Technician	1	6,070	2,125	98,334
Total Process Maintenance	**18**			**$1,802,363**
Assay/Sample Preparation Laboratory				
Chief Chemist	1	6,070	2,125	98,334
Analytical Technicians	3	4,853	1,699	235,856

Table 21-13 Estimated Labor Operating Costs for Processing				
DESCRIPTION	**Number**	**Salary ($/month)**	**Burden (35%)**	**Total ($/annum)**
Sample Preparation	2	4,333	1,517	140,389
Total Laboratory	**6**			**474,579**
Totals	**60**			**$5,687,804**

Power

Power is based on connected mechanical equipment power and power load calculation based on hours per day a piece of equipment is operating and motor efficiency. The price for power is $0.08 kWh. Table 21-14 below summarizes the power costs by area. The highest power costs are for crushing $0.20/t leached, heap leach crushed material reclaim, conveying and stacking $0.09/t leached, heap leach irrigation (pumping) $0.11/t leached, and water systems 0.10/t leached.

Table 21-14 Process Power Costs		
Description	**Annual Cost**	**Cost/t Leached**
Crushing Plant	700,000	0.20
Reclaim Feeder and Conveyors	306,000	0.09
Heap Leaching Irrigation	374,000	0.11
PLS Pumping	34,000	0.01
Water Systems (process, raw, potable, reclaim)	350,000	0.10
ADR Plant	204,000	0.06
Buildings/Compressors/Reagents	68,000	0.02
Totals	**2,036,000**	**0.59**

Process Operating Supplies

Process operating supplies were calculated at 5% of the process labor costs.

21.2.5 General & Administrative (G&A) Operating Costs

G&A operating costs have been estimated based on the proposed mining and processing operations for Mexican Hat Project. The G&A operating costs are comprised of labor and expenses for project areas that are not directly related to the mine or process plant, and shared cost areas. The annual costs for G&A labor and expenses are estimated at $1.593 M ($0.45/t leached) and $1.147 M ($0.33/t leached), respectively. In total, the annual G&A operating cost is estimated at $2.7 M, or $0.78/t leached on an annualized basis. Tables 21-15 and 21-16 summarize the G&A operating cost estimates.

Table 21-15 G&A Labor Operating Costs						
Labor JobTitle	**Number**	**Annual Salary**	**Total Salary**	**Fringe Benefits @ 35%**	**Total Annual**	**Cost/T Processed**
General Manager	1	175,000	175,000	61,250	236,250	0.07
Accountant	1	125,000	125,000	43,750	168,750	0.05
Purchasing Agent	1	90,000	90,000	31,500	121,500	0.03
Clerks	2	50,000	100,000	17,500	135,000	0.04
Office/Change house Cleaner	1	35,000	35,000	12,250	47,250	0.01
Warehouse Manager	1	90,000	90,000	31,500	121,500	0.03
Site Maintenance	2	70,000	140,000	24,500	189,000	0.05
HR Manager	1	110,000	110,000	38,500	148,500	0.04
PR/Environmental Manager	1	120,000	120,000	42,000	162,000	0.05
Safety/Training Manager	1	100,000	100,000	35,000	135,000	0.04
Planner	1	70,000	70,000	24,500	94,500	0.03
Totals	**13**		**1,155,000**	**362,250**	**1,559,250**	**0.45**

Table 21-16 G&A Expenses (Includes Applicable Sales Taxes at 8.17%)		
Description	**Annual Cost**	**Cost/T Processed**
Audits/Legal	150,000	0.04
Leases	18,000	0.01
General Site Maintenance	200,000	0.06
Communications/Internet	100,000	0.03
Public Relations/Donations	50,000	0.01
Insurance	200,000	0.06
Light Vehicle Operations	50,000	0.01
Power	50,000	0.01
Profession Dues/Travel	20,000	0.01
Site Security	270,000	0.08
Office Supplies	39,000	0.01
Totals	**1,147,000**	**0.33**

22.0 ECONOMIC ANALYSIS

22.1 CAUTIONARY STATEMENT

Certain information and statements contained in this section and in the Report are "forward looking" in nature. Forward-looking statements include, but are not limited to, statements with respect to the economic and study parameters of the Project; Mineral Resource estimates; the cost and timing of any development of the Project; the proposed mine plan and mining methods; dilution and extraction recoveries; processing method and rates and production rates; projected metallurgical recovery rates; infrastructure requirements; capital, operating and sustaining cost estimates; the projected life of mine and other expected attributes of the Project; the net present value (NPV) and internal rate of return (IRR after-tax) and payback period of capital; capital; future metal prices; the timing of the environmental assessment process; changes to the Project configuration that may be requested as a result of stakeholder or government input to the environmental assessment process; government regulations and permitting timelines; estimates of reclamation obligations; requirements for additional capital; environmental risks; and general business and economic conditions.

All forward-looking statements in this Report are necessarily based on opinions and estimates made as of the date such statements are made and are subject to important risk factors and uncertainties, many of which cannot be controlled or predicted. Material assumptions regarding forward-looking statements are discussed in this Report, where applicable. In addition to, and subject to, such specific assumptions discussed in more detail elsewhere in this Report, the forward-looking statements in this Report are subject to the following assumptions:

1. There being no signification disruptions affecting the development and operation of the Project.
2. The availability of certain consumables and services and the prices for power and other key supplies being approximately consistent with assumptions in the Report.
3. Labor and materials costs being approximately consistent with assumptions in the Report.
4. Permitting and arrangements with stakeholders being consistent with current expectations as outlined in the Report.
5. All environmental approvals, required permits, licenses and authorizations will be obtained from the relevant governments and other relevant stakeholders.
6. Certain tax rates, including the allocation of certain tax attributes, being applicable to the Project.
7. The availability of financing for GMV Minerals planned development activities.
8. The timelines for exploration and development activities on the Project.
9. Assumptions made in Mineral Resource estimate and the financial analysis based on that estimate, including, but not limited to, geological interpretation, grades, commodity price assumptions, extraction and mining recovery rates, hydrological and hydrogeological assumptions, capital and operating cost estimates, and general marketing, political, business and economic conditions.

The production schedules and financial analysis annualized cash flow table are presented with conceptual years shown. Years shown in these tables are for illustrative purposes only. If additional mining, technical, and engineering studies are conducted, these may alter the Project assumptions as discussed in this Report and may result in changes to the calendar timelines presented.

The economic analysis is based on 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 PEA based on these Mineral Resources will be realized.

22.2 METHODOLOGY USED

SE has prepared a discounted cash flow analysis of the Mexican Hat Project. Technical and cost inputs for the economic model were developed by SE and its consultants with specific inputs provided by GMV Minerals. These inputs have been reviewed in detail by SE and are accepted as being reasonable.

The discounted cash flow analysis was performed on a stand-alone project basis with annual cash flows discounted on a beginning-of-period basis. The economic evaluation used a real discount rate of 5% and was performed at commencement of construction using Q2 2020, US dollars.

The exploration costs prior to pre-production of $4.7 million have been applied as a potential tax savings mechanism on a metal depletion basis in the economic analysis.

This economic analysis is a direct result of the capital cost estimate and is therefore considered to have the same level of accuracy, minus 20% to plus 35%.

22.3 FINANCIAL MODEL PARAMETERS

Technical-economic parameters used in the model are summarized in the following sections. Table 22-1 presents the model inputs used in the economic analysis based on Q2 2020, US dollars.

Table 22-1
Model Inputs

Description	**Values**
Construction Period	2 years
Mine Life (after preproduction)	10 years
LOM Resource Tonnage (Thousands)	32,632
LOM Gold Grade (g Au/t)	0.569
Avg. Annual Process Production Rate Gold (oz)	52,506
Metal Pricing	
Gold Price (US$/oz)	$1,600
Cost Criteria	
Estimate Basis	2nd Quarter 2020 USD
Inflation/Currency Fluctuation	None
Leverage	100% Equity
Taxes	

Table 22-1 Model Inputs	
Description	**Values**
Arizona Corporate	6.5% Profit
US Corporate	21% Profit
Arizona Mining Severance	2.5% (50% Net Revenue)
Royalties / Payments	
Hernandez Royalty (After Buy-back)	1.5% Victor Concession
Royalty Buy-back Payment	$1.5 million
Transportation and Refining Charges	
Shipping, Handling & Refining	$5 oz recovered
Gold Payfor	99.5%

Table 22-2 below summarizes gold price forecasts for 2021-2025 prepared by nine industry respected sources in the mining financial community. The consensus median gold prices range from a high of $1,755 in 2021 to a low of $1,600 in 2024-2025. These gold prices support the use of $1,600 in Mexican Hat PEA.

Table 22-2: Gold Price Forecasts Summaries for 2021 - 2025

	Q4 2020	2021	2022	2023	2024	2025
RBC Capital Markets		1893.00	1800.00	1800.00	1600.00	1600.00
Scotia Capital Markets		1750.00	1750.00	1750.00	1750.00	1750.00
Canaccord Genuity		1984.00	1976.00	2015.00	2015.00	2015.00
Haywood Securities		1700.00	1650.00	1650.00	1650.00	1650.00
Raymond James		1725.00	1600.00	1600.00	1600.00	1600.00
Toronto Dominion		1850.00	1850.00	1850.00	1850.00	1850.00
Eight Capital		1755.00	1650.00	1500.00	1500.00	1500.00
CIBC		1800.00	1600.00	1600.00	1600.00	1600.00
Beacon Securities		1600.00	1500.00	1500.00	1500.00	1500.00
Average Au Price July & Aug	1901.57					
Consensus Median Price	**1901.57**	**1755.00**	**1650.00**	**1650.00**	**1600.00**	**1600.00**

Source: GMV Minerals, October 2020.

22.3.1 Mineral Resource, Mineral Reserve, and Mine Life

The Mineral Resource estimate is provided in Section 14 of the Report. MDA provided a 10,000 t/d mine production schedule an annualized basis. There are no mineral reserves currently estimated for the Project.

The process schedule was prepared on an annualized basis by SE. It includes the mine production with gold grade from the mine production plan and adds plant processing data. The product for sales is reported as troy ounces of gold. Payables for gold from expected payment terms outlined in Section 17. No other payable metals were used in the economic analysis. Table 22-3 summarizes the production schedule.

Table 22-3 Production Schedule														
		LOM Total	Preproduction	Production Years										
Production Schedule			-1	1	2	3	4	5	6	7	8	9	10	11
Waste Material Mined	Tonnes (000)	61,115	1,597	6,001	4,671	3,679	5,368	4,446	8,697	12,925	8,896	4,167	667	
Stripping Ratio	-	1.972	-	2.587	1.337	1.059	1.666	1.320	2.488	3.683	2.542	1.191	0.599	
Mineralized Material Mined														
Victor Mineralized Material	Tonnes (000)	30,994		2,320	3,494	3,473	3,222	3,367	3,496	3,510	3,500	3,499	1,113	
Victor Mineralized Material Grade	Grams Au/tonne			0.531	0.641	0.630	0.551	0.461	0.543	0.687	0.491	0.547	0.581	
Other Mineralized Material	Tonnes (000)	1,638		1,180	5.59	36.3	278	132.7	4.17	0.03	0.00	0.72	0.00	
Other Mineralized Material Grade	Grams Au/tonne			374.6	374.6	374.6	374.6	374.6	374.6	374.6	374.6	374.6	374.6	
Total Mineralized Material Mined	Tonnes (000)	32,632		3,500	3,500	3,510	3,500	3,500	3,500	3,510	3,500	3,500	1,113	
Total Mineralized Material Processed	Tonnes (000)	32,632		3,500	3,500	3,510	3,500	3,500	3,500	3,510	3,500	3,500	1,113	
Mineralized Material Gold Grade	Grams Au/tonne	568.7		0.549	0.640	0.627	0.558	0.471	0.543	0.687	0.491	0.547	0.581	
Contained Gold	Kilograms	18,558		1,922	2,241	2,202	1,953	1,648	1,901	2,411	1,719	1,915	647	
	Ounces	596,661		61,790	72,050	70,798	62,779	52,971	61,115	77,521	55,277	61,560	20,799	
Gold Recovery	%	88.0%		88.0%	88.0%	88.0%	88.0%	88.0%	88.0%	88.0%	88.0%	88.0%	88.0%	
Recoverable Gold	Kilograms	16,331		1,691	1,972	1,938	1,718	1,450	1,673	2,122	1,513	1,685	569	
	Ounces	525,062		54,375	63,404	62,302	55,246	46,615	53,781	68,218	48,644	54,173	18,303	
Gold Recovered (delayed)	Kilograms	16,331		1,389	1,808	2,010	1,793	1,456	1,593	2,127	1,622	1,533	966	34
	Ounces	525,062		44,655	58,118	64,613	57,657	46,798	51,216	68,400	52,152	49,293	31,060	1,099

22.3.2 Refining Terms

The refining terms assumed in the financial analysis are summarized in Table 22-1. The product of the plant will be a gold doré.

22.3.3 Gold Price

Gold pricing in the economic analysis is assumed at $1,600/oz for the base case. Note that the price utilized for financial calculations does not match the $1500 / oz utilized for generation of the mine plan.

22.3.4 Capital Costs

The capital cost estimate basis was provided in Section 21.1 and are summarized in Table 22-4.

Table 22-4
Capital Cost Summary

Description	$US (000)
Direct Costs	
Leach Pad, Ponds & Pipelines	13,132
ADR, BOP & Site Facilities	13,219
Substation & Power	2,246
Indirects	
Engineering & Procurement	1,950
Construction Management	2,474
Contractor Indirects	4,723
Spare Parts and Initial Fills	405
Mine Preproduction & Contractor Assistance	9,730
Owner's Costs	6,067
Freight	683
Duties & Taxes	950
Contingency	12,268
TOTAL	**67,847**

The LOM sustaining costs by Project area is summarized below in Table 22-5.

Table 22-5
Summary of Sustaining Capital Costs (US$000)

Area	LOM	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10
Mining	1,131	-	220	-	-	261	-	-	-	-	650
Heap Leach Pad Expansion (Phase 2)	11,296	-	200	11,096	-	-	-	-	-	-	-
Administration	120	-	-	-	-	120	-	-	-	-	-
Processing (Crushing)	480	-	-	-	-	80	-	-	-	-	400
Total	**13,027**	**-**	**420**	**11,096**	**-**	**461**	**-**	**-**	**-**	**-**	**1,050**

22.3.5 Operating Costs

The operating costs were estimated based on the data in Section 21.3. A summary of the operating costs is provided in Table 22-6 below. Note that these costs are indicated on a LOM basis which include costs for operations during the final year after mining has concluded.

Table 22-6 Summary of Operating Costs		
Description	**LOM Total US$M**	**LOM Ave $ / t leached material**
Mining	$250.8	$7.69
Processing Costs	$222.2	$6.81
General Administration Costs	$26.1	$0.80
Total	**$499.2**	**$15.30**
		$ / oz. recovered
Total per recovered ounce		**$951**

22.3.6 Working Capital

Working capital is the amount of funds required during the initial operating period to offset expenses prior to the cumulative revenue offsetting the cumulative expenses; that is, when the operation becomes self-sustaining in its cashflow. Working capital is recovered at the end of a project operating life.

The working capital is estimated for use in the economic model as three months of operating expenses.

22.3.7 Taxes

U.S. and Arizona Income and Severance Taxes

Taxation for the Mexican Hat Project will be for gold sales income. Generally, the rates are as follows:

1. US Income Tax Rate is 21%
2. State of Arizona Income Tax Rate is 6.5%
3. Arizona Mining Severance Tax Rate is 2.5%

Rather than simply add the rates to get 30% total off of net income, we have calculated the tax for income and severance separately. The Severance Tax is based on 50% of the revenue less mining and processing costs and depreciation, while Income Taxes are based on Net Profit less depreciation and depletion. The Severance Tax was applied at a 2.5% rate and Income Tax at 27.5%.

22.3.8 Depreciation

All exploration expensed along with initial and sustaining capital costs have been depreciated on a metal depletion basis, thus reducing the tax burden on the project.

22.3.9 Closure Costs

Closure costs were estimated by Golder Associates at US$25.182 M.

22.3.10 Salvage Value

A salvage value of 15% of the initial processing facility capital cost has been considered in the economic analysis.

22.3.11 Financing

The financial model presents an unlevered case where no financing is assumed.

22.3.12 Inflation

Inflation is not included in the financial model or the capital and operating cost estimates.

22.4 ECONOMIC ANALYSIS

22.4.1 PEA Results

The Mexican Hat Project's post-tax economic results for the PEA evaluation are summarized in Table 22-7 and shows an after-tax net present value (NPV) of $100.0 M at a 5% discount rate, an internal rate of return (IRR after-tax) of 29.3% and a 2.85 year payback after project start-up on initial capital expenditures of $67.8 M. Table 22-7 presents the economic cashflow on an annualized basis.

Table 22-7
Post-Tax Financial Results Summary

Financial Results*	**Post-Tax**
Undiscounted Cash Flows (LOM)	$ 153.0 M
Net Present Value (5%)	$ 100.0 M
Net Present Value (8%)	$ 77.0 M
Net Present Value (10%)	$ 64.0 M
Internal Rate of Return (IRR)	29.3%
Payback	2.85 years
Initial Capital Cost	$ 67.8 M
Total Capital (LOM) Costs	$ 80.9 M

22.4.2 Cash Costs

The financial results include:

1. Post start-up C1 cash cost: $973 / oz gold.
2. Post start-up all-in sustaining costs (AISC): $1,136 / oz gold.

Cash cost includes all direct and indirect costs associated with the physical activities that would generate gold doré for sale to customers, including mining to gain access to mineralized materials for mining, mining of mineralized materials and waste, milling, third-party refining, insurance and transportation costs, on-site administrative costs and royalties. Cash cost does not include depreciation, depletion, amortization, exploration expenditures, reclamation and remediation costs, financing costs, income taxes, or corporate general and administrative costs not directly or indirectly related to the Mexican Hat Project.

All-in sustaining cost (AISC) includes cash cost plus on-site exploration, reclamation and sustaining capital costs. AISC is divided by the number of payable gold ounces generated by the operation for the period to arrive at AISC per payable gold ounce.

Cost of sales is the most comparable financial measure, calculated in accordance with generally accepted accounting principles (GAAP), to cash cost. As compared to cash costs, cost of sales includes adjustments for changes in inventory and excludes third-party related treatment, refining and transportation costs, which are reported as part of revenue in accordance with GAAP.

22.5 SENSITIVITY ANALYSIS

Table 22-8 presents sensitivities to copper price, capital cost, mineralized gold grade, metallurgical recovery, and operating costs (broken down). These sensitivities are illustrated in Figure 22-1 to Figure 22-4.

Table 22-8: Pre-Tax and Post-Tax Sensitivity Analysis

Pre-tax and Post-Tax Sensitivity to Gold Price									
	-40%	-30%	-20%	-10%	Base	+10%	+20%	+30%	+40%
Gold Price ($/oz)	960	1,120	1,280	1,440	1,600	1,760	1,920	2,080	2,240
Pre-Tax IRR		-14.1%	13.6%	27.6%	39.3%	49.7%	59.4%	68.6%	77.4%
Pre-Tax NPV @ 5% (US$M)	-93.8	-32.7	28.4	89.5	150.6	211.7	272.8	333.9	395.0
Post-Tax IRR			8.5%	20.1%	29.3%	37.6%	45.3%	52.5%	59.4%
Post-Tax NPV @ 5% (US$M)	-95.8	-40.8	11.2	56.2	99.9	143.7	187.5	231.2	275.0
Gold Recovery									
	-4%	-3%	-2%	-1%	Base	+1%	+2%	+3%	+4%
Average Annual Recovery	84%	85%	86%	87%	88%	89%	90%	91%	92%
Pre-Tax IRR	34.2%	35.5%	36.8%	38.0%	39.3%	40.5%	41.7%	42.9%	44.1%
Pre-Tax NPV @ 5% (US$M)	122.9	129.8	136.7	143.6	150.6	157.5	164.4	171.3	178.2
Post-Tax IRR	25.3%	26.3%	27.3%	28.3%	29.3%	30.3%	31.3%	32.2%	33.2%
Post-Tax NPV @ 5% (US$M)	80.1	85.1	90.0	95.0	99.9	104.9	109.8	114.8	119.8
Operating Cost									
	-40%	-30%	-20%	-10%	Base	+10%	+20%	+30%	+40%
Unit OPEX ($/oz)	9.18	10.71	12.24	13.77	15.30	16.83	18.36	19.89	21.41
LOM OPEX (US$M)	299.5	349.4	399.3	449.2	499.2	549.1	599.0	648.9	698.8
Pre-Tax IRR	64.3%	58.3%	52.2%	45.9%	39.3%	32.3%	24.7%	16.2%	5.7%
Pre-Tax NPV @ 5% (US$M)	299.1	261.9	224.8	187.7	150.6	113.4	76.3	39.2	2.1
Post-Tax IRR	49.2%	44.4%	39.6%	34.6%	29.3%	23.8%	17.7%	10.4%	0.1%
Post-Tax NPV @ 5% (US$M)	206.6	179.9	153.3	126.6	99.9	73.3	46.6	18.0	-13.7
Capital Cost									
	-40%	-30%	-20%	-10%	Base	+10%	+20%	+30%	+40%
CAPEX (US$M)	40.7	47.5	54.3	61.1	67.8	74.6	81.4	88.2	95.0
Sustaining Capital (US$M)	7.8	9.1	10.4	11.7	13.0	14.3	15.6	16.9	18.2
Pre-Tax IRR	63.5%	55.5%	49.0%	43.7%	39.3%	35.5%	32.1%	29.2%	26.7%
Pre-Tax NPV @ 5% (US$M)	179.9	172.6	165.2	157.9	150.6	143.2	135.9	128.6	121.2
Post-Tax IRR	48.4%	42.0%	37.0%	32.8%	29.3%	26.4%	23.8%	21.5%	19.5%
Post-Tax NPV @ 5% (US$M)	122.6	116.9	111.3	105.6	99.9	94.3	88.6	82.9	77.2
Gold Grade									
	-40%	-30%	-20%	-10%	Base	+10%	+20%	+30%	+40%
Au Grade (g/t)	0.341	0.398	0.455	0.512	0.569	0.626	0.682	0.739	0.796
Pre-Tax IRR	0.0%	-13.1%	13.7%	27.7%	39.3%	49.7%	59.4%	68.6%	77.3%
Pre-Tax NPV @ 5% (US$M)	-93.1	-32.2	28.7	89.7	150.6	211.5	272.4	333.3	394.2
Post-Tax IRR	0.0%	0.0%	8.6%	20.1%	29.3%	37.6%	45.2%	52.4%	59.3%
Post-Tax NPV @ 5% (US$M)	-95.1	-40.3	11.5	56.3	99.9	143.6	187.2	230.8	274.5

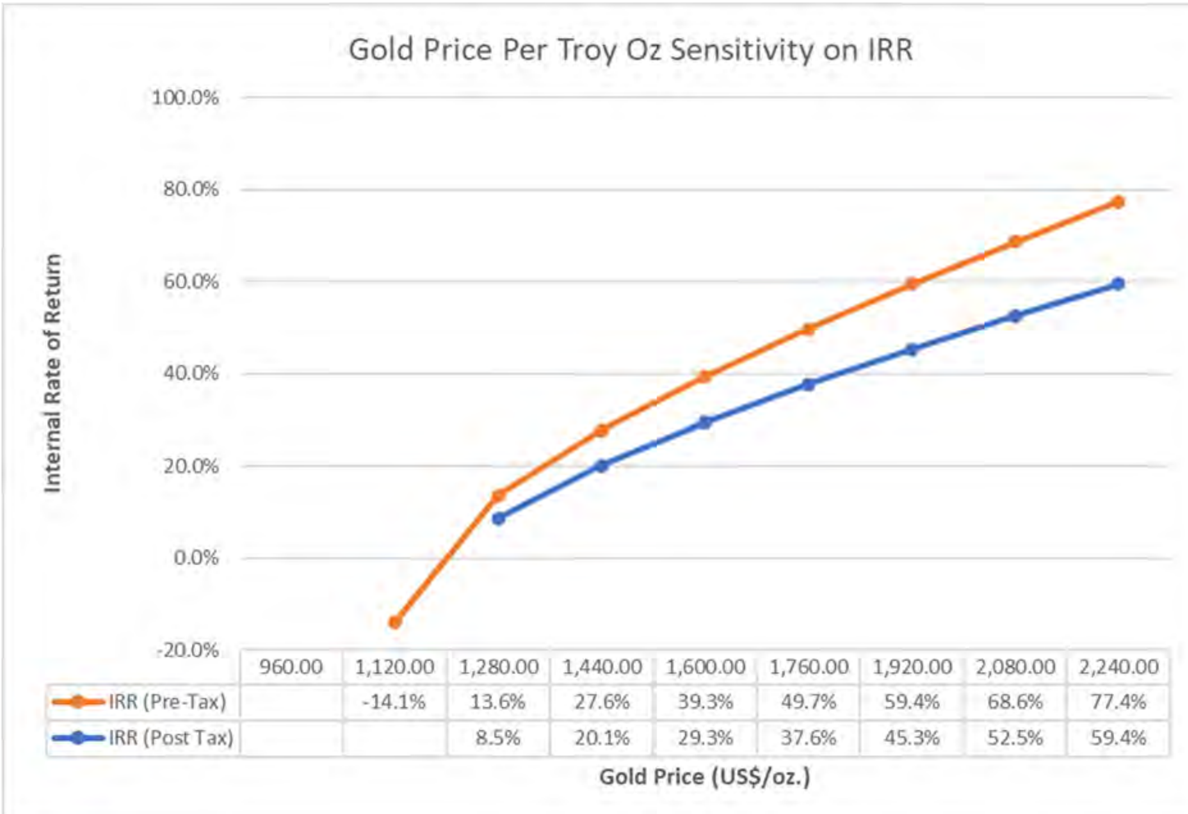
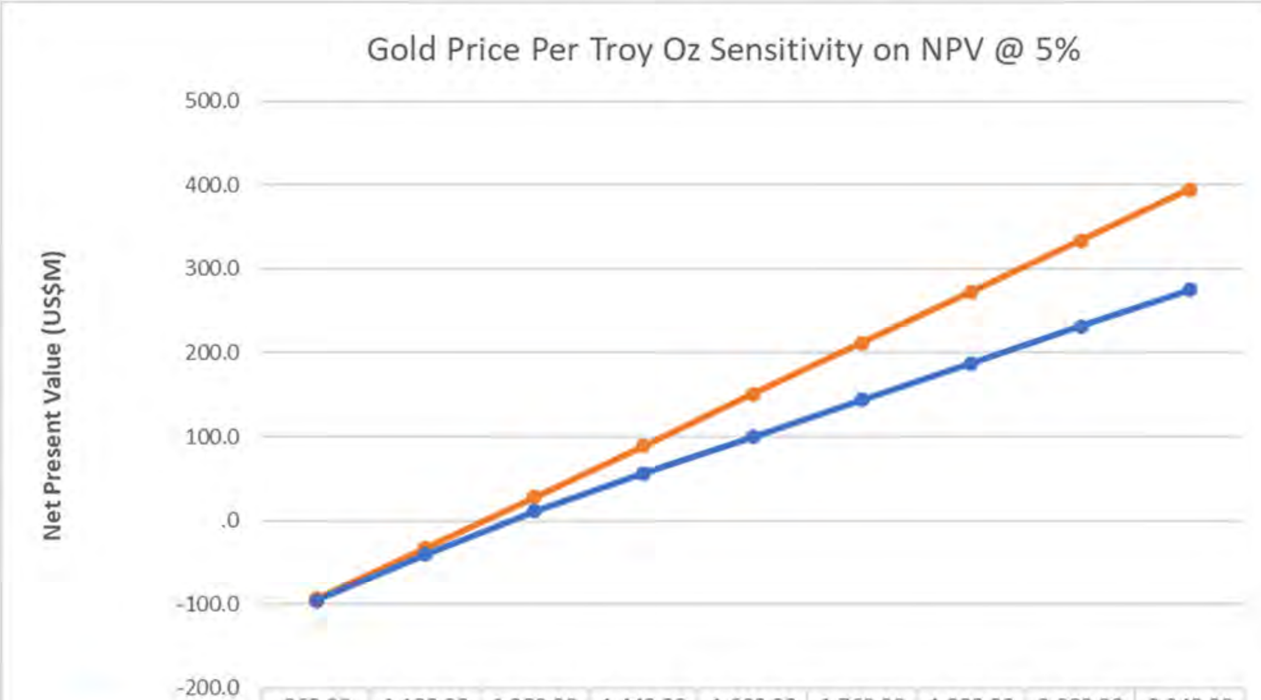


Figure 22-1: IRR Sensitivity to Gold Price



	960.00	1,120.00	1,280.00	1,440.00	1,600.00	1,760.00	1,920.00	2,080.00	2,240.00
NPV (Pre-Tax)	-93.8	-32.7	28.4	89.5	150.6	211.7	272.8	333.9	395.0
NPV (Post Tax)	-95.8	-40.8	11.2	56.2	99.9	143.7	187.5	231.2	275.0

Figure 22-2: NPV @ 5% Sensitivity to Gold Price

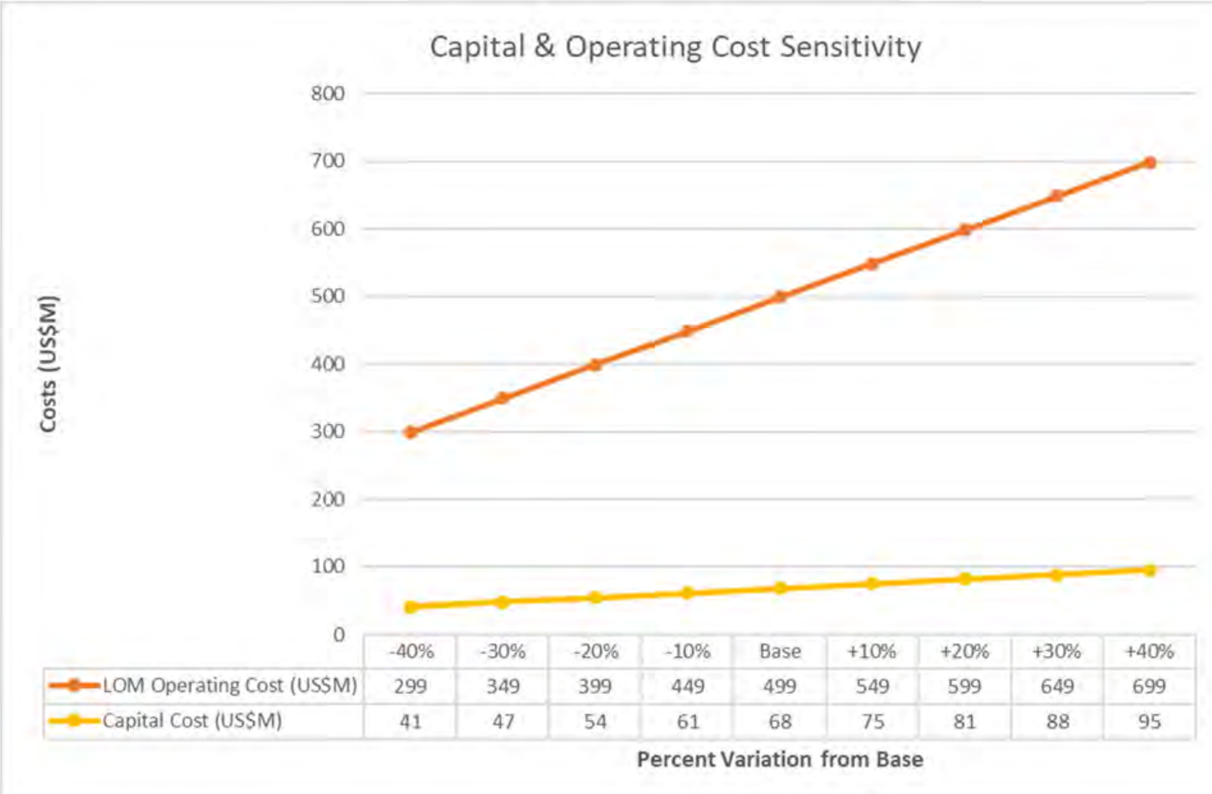


Figure 22-3: Sensitivity to Capex & Opex

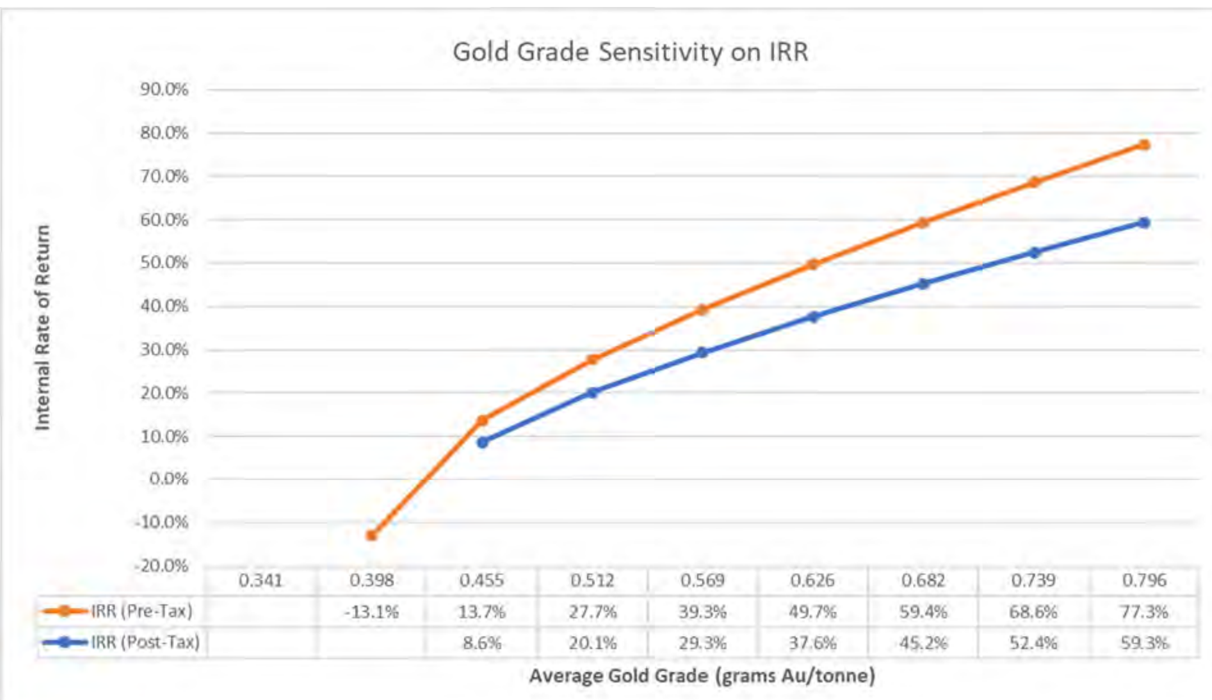


Figure 22-4: IRR Sens to Gold Grade

23.0 ADJACENT PROPERTIES

There are no immediately adjacent properties.

The Commonwealth Gold and Silver property owned by Wexford Capital Partners LLP is 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 (gpt)	Volume cu. M	Tonnage 000 tonnes	Gold Equivalent		Gold		Silver	
			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.

(2) 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 about this technical report, the Mexican Hat property.

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 OPPORTUNITIES

GMV anticipates advancing the Project to the next stage of development for preparing a PFS Study. Several work programs and studies are recommended to advance the Project from PEA to PFS for improving the Projects economics as listed below:

- Drilling:
  - Resources: Convert inferred to measured and indicated, and increase tonnage and grade for mineral reserves:
    - Hydrology: Characterization of hydrogeologic system for sources of water supply and characterization of the aquifer; water samples for permitting and project water balance; preliminary flow modeling to predict inflows to future open pits.
    - Metallurgy: Obtain representative samples for test work.
    - Mineralogy Study
    - Geotechnical:
      - Mine: Pit slope and waste dump designs
      - Heap Leaching; slope design
      - Foundations: Crushing and plant loads
- Labor Study: Availability and labor rates
- Metallurgical Test Program: Conducted on a representative composite basis to optimize process design parameters.
- Transportation Study
- Baseline environmental studies for characterization of environmental setting and mining wastes. These studies would be used for future permit submittals and would include:
  - Hydrologic study to evaluate source of water supply, characterize the aquifer, and characterize ephemeral surface water
  - Biological studies
  - Jurisdictional water determination
  - Air quality monitoring
  - Cultural resources inventory
  - Socioeconomic baseline study
  - Community outreach program development
  - Geochemistry study of mining wastes
  - Climate study
  - Sediments and soils characterization

An integrated drilling program is an opportunity to reduce overall drilling costs compared to separate programs. An integrated drilling program can be developed to collect data for multiple purposes, such as metallurgical samples, geotechnical parameters, hydrogeological parameters, and water quality.

The results of the above recommendations will impact the technical parameters and economics of the PFS. There exist opportunities to reduce the capital and operating cost estimates used in this study. Based on the inputs used, the project shows merit with a 10-year mine life. Of note, initial designs were created using lower costs for processing and mining than the costs summarized in Section **Error! Reference source not found.**. Should the capital and operating costs be reduced, there is an opportunity to increase the resources that can be mined. For example, reductions could be made by eliminating contractor mining and processing. However, this will come at additional capital costs as an Owner operation. Item 26.0 summarizes the costs for the recommendations.

24.2 RISKS

Key risks identified with the Project and development plan are as follows:

- The biggest mining risk will be the ability to effectively mine the upper portions of the Project's hill outcrop due to the steep nature of the terrain. In addition, mining of the South Pit is planned to be done first, which reduces the time to get into commercial production, but it will be important to mine the South Pit during the dry season as the pit is in a major drainage.
- Risk exists for the capital and operating cost, and the overall Project economics, should there be a substantial increase in unit costs (utility, fuel, labor, reagents, etc.).
- The Project's economics are very sensitive to gold price which has been highly variable in recent years.
- Risks associated with the project's infrastructure include the confirmation of available water sources for the proposed mining operations from on-site wells. Hydrological drilling and studies will be addressed in the next stage of study.
- No geotechnical drilling, test work or analysis has been conducted at the Project site. Technical and cost risk exist for determining the mine and heap leaching design parameters.
- The Project will require various state and federal authorizations, licenses and permits for Project construction, operation, closure, and post-closure. Comprehensive environmental and socioeconomic baseline studies will be required. No environmental baseline studies have been conducted. The long-term seepage and water management requirements have not been established, and these issues can impact closure costs.
- At this time, there are no known factors to preclude a successful permitting effort; however, the length and effort of the permitting process can be difficult to predict due to the multiple agencies that will be involved, including both state and federal agencies.
- A more detailed look at mining plans with upgraded resource estimates in the future may allow for advancement of higher-grade material early in the mine life.
- Metallurgical testing is preliminary in nature, as such, estimates for gold recovery from heap leaching and cyanide consumption may present risks. Additional investigation, including column testing on representative samples is required to fully assess the gold recovery and cyanide consumption estimates and timing included in the Report.

25.0 INTERPRETATION AND CONCLUSIONS

25.1 INTRODUCTION

The current study is considered scoping in nature and suitable for inclusion in a preliminary economic assessment as defined and allowed in NI 43-101 guidelines. The Project as contemplated in this study work to date presents the following attributes:

- GMV has 100% interest in the Project.
- The Project's inferred resources can be mined and processed using conventional technologies to produce gold doré.
- The Project is subject to a 3% net smelter returns royalty (NSR Royalty). GMV has the option to reduce this royalty to1.5% with a buy back payment of $1.5 M. This option has been included in the Project's economic analysis.
- Inferred resources are estimated at 36.733 Mt at a gold grade of 0.58 g/t using a cut-off grade of 0.20 Au g/t contained in the open pit deposits.
- The inferred resources will be mined by conventional open pit at a low, LOM stripping ratio of 1:87:1 waste to material leached.
- A total of 32.632 Mt will be mined from the inferred resources, crushed and placed on the heap leach pads for leaching with sodium cyanide and subsequent processing of the gold-bearing solution in an ADR plant for producing gold doré.
- The PEA is designed for contractor mining and crushing as opposed to owner operation.
- Contractor mining and crushing will be done at a nominal production rate of 10,000 tpd delivery to a crushing plant and lined heap leach pad.
- Gold recovery projected from preliminary metallurgical testing is 88% with an estimated sodium cyanide consumption of 0.3 kg/t of material leached.
- LOM gold production is estimated in the gold doré at 525,000 ounces.
- Initial capital cost of the Project is estimated at $67.8 M including mine, process plant, infrastructure, and heap leach pad construction. LOM sustaining capital is estimated at $13.0 M.
- Operating C1 cash cost is estimated at an average $951per ounce of gold produced ($15.30 per tonne processed) and an AISC of $1,136 per ounce of gold produced.
- The Project will require various Arizona state and federal authorizations, licenses and permits for construction, operation, closure, and post-closure.
- No known factors exist that could preclude a successful permitting effort; however, due to the multiple agencies that will be involved as well as the likelihood of a NEPA process, the length and the effort of the permitting process can be difficult to predict.
- Project economic analysis at gold price of $1,600/oz yields a pre-tax IRR of 39.3% (after tax 29.3%) and a pre-tax NPV at a 5% discount rate of 150.6 million (after tax $100.0 million) with a 2.85 year payback of invested capital.
- Engineering design analysis indicates the potential to increase pit size and contained ounces with increased gold prices.

- Based on the study results, it is recommended to advance the project to a Pre-feasibility study.
- The estimated cost for the next stage in development is $11.3 M.

Considering the above, and the absence of fatal or serious flaws, the Project is worthy of continued development to Pre-feasibility or Feasibility Study level of confidence and definition to advance the understanding of the technical risks associated with resource confidence, metallurgical performance and project development costs.

25.2 EXPLORATION, DRILLING & ANALYTICAL DATA COLLECTION SUPPORTING MINERAL RESOURCE ESTIMATION

At Mexican Hat, gold mineralization is hosted in fractures within peralkaline to subalkaline Tertiary igneous rocks ranging from basalt to rhyolite associated with iron oxide and carbonate alteration associated with elevated silver, arsenic, bismuth, antimony, mercury, sulfur, selenium and tellurium. This suite is commonly associated with epithermal systems and although elevated, are at lower than average levels for most epithermal deposits.

Several structural trends have been observed including generally brittle fractures +/- differential displacements. Northeast structures appear to be offset by the Zone 7 Fault, although given poor exposure, this may be tenuous. The dominant 120° azimuth fault appears to offset all the above-mentioned structures as is likely the dominant structure controlling mineralization.

Three principal mineralized structural trends have been observed, including:

1. Southwest striking, steeply northwest dipping structures up to 70 m wide and up to 415 m long.
2. Proximal to the Zone 7 Fault, striking 175 azimuth @ 055 degrees and dipping to the west
3. Within a late fault (120 degrees fault)

Mineralization occurs in every rock type and is hypothesized that the Zone 7 Fault acted as the conduit for the modeled mineralization. Gold grades are most robust within the Zone 7 Fault, and along with the hanging wall units to the fault, albeit mineralization has also been noted within the footwall units of the Zone 7 Fault. The tenor of this mineralization is not as fully understood.

Trench sampling, RC and core drilling completed by GMV confirms the location, form and tenure of mineralization and confirms that the mineralization is structurally controlled and present in each of the main lithological units.

25.2.1 Data QAQC

Blanks, duplicates, and standards confirm acceptable analytical results; however, some standards (SG5 and GSM1) appear to provide for lower than acceptable results when geochemical analysis is used.

25.2.2 Mineral Resource Estimate

Tetra Tech has updated the mineral resource estimate for the Mexican Hat Property. Gold mineralization is hosted within structurally controlled, or prepared, domains in association with clay alteration and hematite, hosted in predominantly Latite and Andesite rocks and associated with low sulphidation epithermal style geochemistry. A thorough review of new RC drilling information collected in 2019 has resulted in the generation of an updated 3D model, from which this resource was based.

The Mexican Hat Property represents a potentially economic gold deposit and warrants further work. The regional setting of the property is within the basin and range province which post-dates the Laramide copper mineralization most associated with Arizona.

Gold grades reported by previous operators, except that by Kalahari Resources, has been replicated by GMV with reasonable confidence, however, the historical database is lacking reliable geochemical, structural, and lithological data to develop detailed deposit models.

Local grade variability in reported gold mineralization at Mexican Hat is observed from twin holes and resampling programs. The variability is attributed primarily to the sampling stage where limitations and material from recovery methods are associated with RC drilling, and secondly to a nugget effect from gold grain size and/or distribution in clay associated mineralization within fracture networks. There may be a coarse gold issue that would necessitate larger samples to be collected, and that samples be collected from diamond drill core rather than from bulk RC samples. The QAQC program did evaluate duplicate samples, however, few were collected to represent the range of mineral grades of interest, above the mineral resource cut-off grade.

Reliable variograms were not able to be obtained for the Mexican Hat Project based on the existing data and geological model domains. Attempts were made to generate variograms for each of the zones, however, no distinct trends of mineralization were able to be extracted, attributed mainly to the spacing and sample density within the mineralized domains. ID^2, ID^3 and nearest neighbour estimates produce similar results and provide for confidence in the model when constrained to an optimized pit shell.

A 0.20 g/t cut-off has been recommended in reporting the resource estimates based upon preliminary metallurgy and discussions with independent mine contractors on mining costs. This is within current industry standards in this area. A preliminary pit model was generated to confirm a low strip ratio, supporting the recommended cut-off grade. Currently, all of the mineral resource is categorized as Inferred, in accordance with CIM Definition Guidelines based on reliance of the historical drillholes database, and uncertainties associated RC drill recoveries, and grade variability, and understanding of geological controls on the mineralization.

There is remaining opportunity to expand the resource by additional trenching and shallow drilling along strike of the Zone 7 Fault, along with the hanging wall units located within the eastern region of the fault zone.

25.3 MINING

Based on the inputs used, the project shows merit with a 9.3 year mine life. Of note, initial designs were created using lower costs for processing and mining than the costs summarized in Section **Error! Reference source not found.**. Should the costs be reduced, there is an opportunity to increase the resources that can be mined. For example, reductions could be made by eliminating contractor mining and processing. However, this will come at additional capital costs.

25.4 METALLURGICAL AND PROCESSING

Two relevant metallurgical test programs have been performed on samples from the Project at McClelland (2015) and Bureau Veritas (2016). Both laboratories are accredited facilities for conducting the selected test programs. Samples included both bulk from trenches and core composites. Four types of mineralization were identified during preliminary geological assessments: latite comprising approximately 80% of the mineralization, with 8% each of andesite and basalt, and the remaining 4% dacite. The test results were used for the determination of the processing design for the Project's processing plant. Based on the test results, a gold recovery of 88% for heap leaching has been used in this study.

25.6 INFRASTRUCTURE

Infrastructure required for the project is well understood, with much of the major components in place including power accessed from a nearby grid system, water from on-site wells, and project access from local roads and highways.

25.7 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACTS

The Project will require various state and federal authorizations, licenses and permits for Project construction, operation, closure, and post-closure. Comprehensive environmental and socioeconomic baseline studies will be required. No environmental baseline studies 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 both state and federal agencies. It is anticipated that the State of Arizona environmental permitting will be relatively straightforward because the discharging facilities will be designed and constructed using Best Available Demonstrated Control Technology (BADCT) standards, which allow for prescriptive design to facilitate permitting. Federal permitting is anticipated to be more complex due to the requirement to evaluate a range of alternatives. The National Environmental Policy Act (NEPA) will be triggered because the waste rock storage facility will be on federal Bureau of Land Management (BLM) land. Recent changes to NEPA include presumptive time limits, which will benefit the permitting timeline.

25.8 CAPITAL COST ESTIMATES

The methodology to develop the capital cost estimates are appropriate for this level of study and utilize budget pricing from vendors, cost data bases and recently estimated projects. As a further effort to confirm the accuracy ranges, the estimate information also compared to a more detailed recent project estimate for reasonableness on major components, materials, and costs.

The contingencies have been applied at a confidence level of confidence for the respective project areas – mining, processing, and infrastructure. The capital cost estimate is thought be in the accuracy range of minus 20% to plus 35% as stated.

25.9 OPERATING COST ESTIMATE

The methodology used to develop the operating cost estimates are appropriate for this level of study. An average operating cost of $951 per ounce of payable gold or $15.30 per tonne material leached is recommended for use at this time.

Mining costs were built up from first principles and include the mining contractor cost based on the LOM production plan. Mining haul distances a reasonably certain based on the design of the LOM mine plan.

Processing costs were developed from a combination of direct build-up of costs based on metallurgical parameters for sodium cyanide consumption. Electric power is based on the process motor loads and sourcing of power from a nearby grid system.

Manpower estimates were developed from first principles and payrates benchmarked to similar projects. A highly skilled workforce exists in the area familiar with mining and processing plant operations. The location of the Project should be favorable to attracting a highly qualified staff, and personnel for operations, maintenance, and administration.

25.10 ECONOMIC ANALYSIS

The Project's economics have been generated utilizing conventional economic analysis for the Project at the stated production parameters and costs. The gold price of $1,600 per ounce, used in the economic analysis, is based on the price forecast presented in Section 22.3. These results show a robust project with a 29.3% after-tax IRR and an NPV of $100.0 M @ a 5% Discount Rate. The project does appear to be quite sensitive to the gold price and capital cost.

26.0 RECOMMENDATIONS

The results of the updated Mineral Resource Estimate suggest that the project should be assessed at a Pre-feasibility level study to verify the input parameters used for pit constraints, and to assist in de-risking the project before proceeding with future drilling at advanced project stages. The following additional technical work and studies should be considered for the Pre-feasibility study:

- Mapping, geophysics, and drilling for updated resource estimate
- Metallurgical work and Mineralogy Study
- Engineering and drilling work for mining geotechnical, pit geotechnical and hydrogeological
- Environmental and permitting work

The total costs for producing a prefeasibility study are estimated at $11.3 M as detailed in the following discussions. Pre-Feasibility Study to be completed in 12-15 months. The Qualified Person(s) to this report make the following recommendations.

26.1 DRILLING AND EXPLORATION

Aspects of grade variation and geological interpretation must be further refined to reduce risk prior to advance the Project economic viability studies. Going forward, additional drill programs should place an emphasis on replacing historical drillholes with modern drilling, where diamond drill core recovery methods are used rather than RC recovery. Additional exploration recommendations include the following:

- Mineralization is open at depth and along the Zone 7 structure and drilling can test the structures to potentially increase the established resource in each direction.
- There is opportunity to infill large gaps in sample spacing along the Zone 7 structure to confirm continuity of mineralization and to test the mineralogical characteristics to confirm the interpretation that the structure was a main conduit for mineralizing fluids,
- 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.

3D modeling of the mineralization indicates the presence of potential additional unidentified controls on the mineralization. Within the generated 0.2 g/t gold grade shells, higher grade mineralization appears to develop oblique to the presently modeled zones. These relatively gently dipping trends indicate the potential presence of surficial enrichment, or the presence of un-identified flatter dipping gold bearing structures approximately 50m below topography, or subparallel splays off of the Zone 7 structure. A domain has been defined (zone 6) to constrain higher grade mineralization >1.0 gpt Au. Future drilling programs should place an emphasis on investigating these potential additional mineralization controls by carefully logging drill the alteration, sulphide constituents, and structures present in drill core.

Geophysics has identified rocks that appear to be similar magnetically and electromagnetically in several locations and these should be examined.

- 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.

A 3D geological model should also be developed with a focus on modeling the relationship between lithology and structure. A robust geological model will aid future exploration targeting, and it will also aid in any future geochemical and geotechnical studies which are required for mine planning and subsequent environmental and reclamation investigations.

26.1.1 Database for resources

- It is recommended that GMV implement an enhanced QAQC system, which targets a more representative range of assay grades for duplicate testing.
- A variability study should be undertaken to assess local grade variation and the nugget effect; the study may include collection of duplicate samples from twin drillholes and from sample splits; should evaluate gold grain studies using screen metallics, and test various sizes of sample charges for fire assay to determine optimal mass for use in future analytical programs.
- It is recommended that GMV implement a database management system to better track and organize the large volumes of data which are collected and stored for the Project.

26.1.2 Budget by Priority for Drilling and Resources

Further development of the property requires an improved understanding of the controls on mineralization. These would be called priority one issues and would be required for all work, whether development of exploration. Other priorities would allow for refinement and expansion as well as development of the resource. Table 26-1 below summarizes the budget for drilling and resource work for progressing to a pre-feasibility study.

Table 26-1
Summary of Costs for Drilling/Resources

Item	All in Cost	Priority One	Priority Two	Priority Three
Mapping	60,000	30,000	30,000	0
Relog core	55,000	30,000	25,000	0
3D Geological Modeling	30,000	30,000	0	0
Geophysics	120,000	0	120,000	0
Conversion drilling (core)	1,500,000	0	1,500,000	0
Shallow drilling (core)	850,000	0	0	850,000
Shallow drilling	350,000	0	350,000	0
Deeper drilling	900,000	0	0	900,000
Reporting	50,000	50,000	0	0
Totals	**3,915,000**	**140,000**	**2,025,000**	**1,750,000**

26.2 MINING

With additional resource drilling, the mineral resources would be upgraded from the Inferred to a Measured and Indicated status. With that, the project can be promoted to a prefeasibility study which would allow the statement of reserves. For the Pre-feasibility study, geotechnical work will be required for mine designs at an estimated cost of $150,000.

26.3 GEOTECHNICAL, HYDROLOGY AND HEAP LEACH PAD STUDIES

To evaluate the technical and economic viability of the HLF and advance the engineering design to a Pre-Feasibility level, the following recommendations should be considered:

- Conduct a geotechnical field investigation and laboratory testing program recommended to facilitate pre-feasibility design level of the HLF for the foundation and borrow materials. Subsurface conditions should be characterized in sufficient detail to provide a high level of confidence in the stratigraphy, ground water, structure, and preferential paths of flow in the upper 30 meters of the subsurface to satisfy standards of practice for design of heap leach facilities.
- Complete geotechnical drill holes to collect core for strength testing for open pit mine slope design.
- Conduct geotechnical testing of the leach material and proposed drain materials.
- Conduct liner interface strength testing of the proposed liner system.
- Evaluate the hydrogeological conditions that exist at the Mexican Hat Project specifically in the vicinities of the HLF.
- Conduct geological mapping and geohazard studies.
- Monitor groundwater levels to evaluate seasonal fluctuations.
- Monitor existing stream flows to measure sediment transportation in existing streams. This will provide valuable input for designing sediment control structures.
- Additional site-specific precipitation and evaporation measurements should be collected to better refine the predictions of the HLF process fluid water balance.
- Additional effort should be made to optimize geometry of the proposed facilities.
- Additional effort should be made to optimize geometry of the proposed stacking layout.
- Trade-off evaluation between conveyor stacking vs. truck stacking.

For pre-feasibility level study, Tierra Group estimates a budget of $750,000 to complete the above work tasks.

26.4 METALLURGICAL TEST WORK AND MINERALOGY STUDY

The deposit samples tested to date in columns to simulate heap leach have been on samples taken at or near surface level from two trenches and a bulk sample. It is recommended that representative samples from the Mexican Hat deposit be composited for the metallurgical programs to advance to a pre-feasibility level study. Test work from different areas of the Mexican Hat property should be tested to understand variability for different crushed material sizes to optimize the gold recovery and cyanide consumption rate. New test composites should use fresh drill core materials.

A minimum of eight column tests at a P_{80} of 38 mm (to be confirmed with bottle roll tests prior) is recommended on core from the top third of deposit, 2 column tests from the mid-portion of deposit and, 2 from the deepest part of the deposit.

It is recommended to conduct a mineralogical investigation and geometallurgical study to better understand the impacts of the crushed material characteristics and hardness such as abrasion tests. This information would also assist in optimizing the mine plan.

The budget for metallurgical test work and mineralogy study to advance Mexican Hat for a pre-feasibility study is estimated at $300,000.

26.5 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACTS

To advance the Project to a Pre-feasibility Study, baseline studies should be started to support the permitting process. Consultation with the community and the regulatory agencies should be initiated. To oversee these activities, the company will need to contract, or hire, an environmental manager as well as a community relations manager.

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

26.6 SUMMARY OF ESTIMATED BUDGETS

Table 26-2 summarizes the estimated budgets for preparing a Pre-feasibility Study for Mexican Hat.

Table 26-2
Recommendations For Pre-Feasibility Study

Description	PFS $000's (12-15 Months)
Mapping/Resource Drilling/Geophysics	3,915
Geotechnical Mining	150
Geotechnical/Hydrology for HLP Designs	750
Metallurgical Testwork/Mineralogy	300
Environmental Studies & Permitting	3,000
Owner's Personnel & Expenses	200
PFS Engineering Study	3,000
Total	$11,315

27.0 REFERENCES

Significant reference documentation reviewed in preparation of this report are listed below:

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2. Arizona Department of Environmental Quality, undated, Arizona mining guidance manual BADCT, available at **https://static.azdeq.gov/wqd/badctmanual.pdf**.
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5. Arizona Department of Water Resources. 2017. Statewide groundwater level changes in Arizona water years 1996 to 2016, 2006 to 2016, and 2015 to 2016, Open file report no. 14, June, 52 p.
6. Arizona Department of Water Resources. 2018. Groundwater Flow Model of the Willcox Basin, July, 196 p.
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8. Golder Associates Inc. 2020. Summary of environmental and hydrologic data for Mexican Hat PEA, technical memorandum prepared for GMV Minerals, 1 October, 31 pp.
9. Metallurgical Testing Report for DRW Geological Consultants. McClelland Laboratories, October 2015.
10. Mexican Hat Column Leach Results. Bureau Veritas, 2016.
11. Mexican Hat Project NI 43-101 Technical Report Preliminary Economic Assessment. M3, December 2018.
12. Ontario Securities Commission. 2011. Form 43-101F1, Technical Report, Table of Contents, available at **https://www.osc.gov.on.ca/en/SecuritiesLaw_ni_20110624_43-101_mineral-projects.htm**.
13. Sulphur Springs Valley Electric Cooperative, Inc. Letter dated September 20, 2018. Mexican Hat Project.

28.0 APPENDICES

28.1 CERTIFICATES AND CONSENT OF QUALIFIED PERSONS

28.2 HLF DESIGN SUMMARY MEMORANDUM

APPENDIX 28.1

CERTIFICATES & CONSENT OF QP's

8450 East Crescent Parkway, Suite 200
Greenwood Village, CO 80111

Phone: 303.714.4840
FAX: 303.714.4800

CONSENT OF QUALIFIED PERSON – Alva LeRoy Kuestermeyer

RE: GMV Minerals Inc.

I, Alva LeRoy Kuestermeyer, hereby consent to the public filing by GMV Minerals Inc. ("GMV" or "Client") of the technical report titled, "NI 43-101 Technical Report, Updated Preliminary Economic Assessment, Mexican Hat Project", dated November 19, 2020 (the "Technical Report").

I also consent to any extracts from, or a summary of, the Technical Report contained in GMV's press release ("PR") dated November 3, 2020.

I confirm that I have read the PR and that it fairly and accurately represents the information in the Technical Report for which I am responsible.

Dated this 17th day of November 2020

Alva LeRoy Kuestermeyer

SME
Society for
Mining, Metallurgy
& Exploration
Alva L. Kuestermeyer
SME Registered Member No. 4802010
Signature
Date Signed Nov 17, 2020
Expiration date Dec. 31, 2020

Alva LeRoy Kuestermeyer
Senior Process Engineer

Samuel Engineering, Inc.
8450 E. Crescent Parkway Ste. 200
Greenwood Village, CO, 80111

1-303-714-4840
akuestermeyer@samuelengineering.com

CERTIFICATE

I, Alva LeRoy Kuestermeyer, Registered Member (Society of Mining, Metallurgy and Exploration), of the City of Greenwood Village, Colorado, USA, do hereby certify that as the author (or co-author) of this Technical Report on the Mexican Hat Project, located in Cochise County, Arizona, USA, dated November 19, 2020, I hereby make the following statements:

(a) I am a Senior Process Engineer practicing at Samuel Engineering, 8450 E. Crescent Parkway, Suite. 200, Greenwood Village, CO 80111, USA.

(b) I graduated from South Dakota School of Mines and Technology with a Bachelor of Science degree in Metallurgical Engineering in 1973 and Colorado School Mines with a Master of Science degree in Mineral Economics in 1982.

(c) I am registered as a registered member (1803010RM) of the Society of Mining, Metallurgy and Exploration (SME).

(d) I have worked as a metallurgical engineer for more than 40 years since my graduation. Relevant experience includes providing process plant designs, metallurgical test programs, capital and operating cost estimates, and economic analyses of precious-and base metal projects in the United States and various countries of the world. During this period, I have worked as Chief Metallurgical Engineer of an operating mine in Arizona.

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

(f) I visited the Mexican Hat Project property and site facilities on July 13, 2020.

(g) I am responsible for the Technical Report Sections: 2, 3, 13, 17 (except 17.5), 18, 21.2 (except 21.2.1), and corresponding sections of 1, 25 and 26.

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

I have no prior involvement with the property that is the subject of this Report and I hold no interests in, nor do I expect to receive any interests, direct or indirect from GMV Minerals Inc. ("GMV") or any associated or affiliated company. I am independent of the issuer, GMV, applying the tests set out in Section 1.4 of NI 43-101. I have read NI 43-101 and this Report has been prepared in compliance with NI 43-101 and form 43-101F1.

I consent to the filing of this report with any stock exchange or other regulatory authority and any publication by them, including electronic publication in the public company files on their web sites accessible by the public of this Report.

Signed and dated this 17th day of November 2020, at 8450 E. Crescent Pkwy, Ste. 200, Greenwood Village, Colorado, USA.

Alva LeRoy Kuestermeyer
SME Registered Member (#1803010RM)

SME
Society for Mining, Metallurgy & Exploration
Alva L. Kuestermeyer
SME Registered Member No. 1802010
Signature
Date Signed Nov 17, 2020
Expiration date Dec 31, 2020

8450 East Crescent Parkway, Suite 200
Greenwood Village, CO 80111

Phone: 303.714.4840
FAX: 303.714.4800

CONSENT OF QUALIFIED PERSON – Steven Alan Pozder

RE: GMV Minerals Inc.

I, Steven Alan Pozder, hereby consent to the public filing by GMV Minerals Inc. ("GMV" or "Client") of the technical report titled, "NI 43-101 Technical Report, Updated Preliminary Economic Assessment, Mexican Hat Project", dated November 19, 2020 (the "Technical Report").

I also consent to any extracts from, or a summary of, the Technical Report contained in GMV's press release ("PR") dated November 3, 2020.

I confirm that I have read the PR and that it fairly and accurately represents the information in the Technical Report for which I am responsible.

Dated this 17th day of November 2020

Steven Alan Pozder

COLORADO REGISTERED PROFESSIONAL ENGINEER
STEVEN A. POZDER
29144

Expires 10/31/21

Steven Alan Pozder, P.E., QP

Samuel Engineering, Inc.
8450 E. Crescent Parkway Ste. 200
Greenwood Village, CO, 80111

1-303-714-4840, ext. 4828
spozder@samuelengineering.com

CERTIFICATE

I, Steven A. Pozder, Professional Engineer (Colorado #29144), do hereby certify that as the co-author of this Technical Report (Report) on the Mexican Hat Project, located in Cochise County, Arizona, USA, dated November 19, 2020, I hereby make the following statements:

(a) I am employed as a Senior Director, practicing at Samuel Engineering, Inc., 8450 E. Crescent Pkwy, Ste. 200, Greenwood Village, CO 80111, USA.

(b) I am a graduate of the University of Denver with a B.S. in Mechanical Engineering in 1988. I am a graduate of the University of Denver with an M.B.A. in General Business in 1994.

(c) I am registered as a Professional Engineer (P.E.) with the State of Colorado, Registration Number 29144.

(d) I have practiced my profession as a Mechanical Engineer and Project Manager in mineral processing and mining for over 30 years. My relevant experience for the purpose of the Technical Report is:

- I have worked as a consulting engineer on mining projects in roles such a mechanical engineer, project engineer, area manager, study manager, and project manager. Projects have included Scoping Studies, Prefeasibility Studies, Feasibility Studies, basic engineering, detailed engineering and startup and commissioning of new projects.
- In engineering positions, I have estimated and reviewed capital and operating costs and completed economic analyses including power requirements, reagent costs, labor requirements and costs, etc. for 24 years.

(e) I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101), and do certify that, by reason of my education, good standing as a registered Professional Engineer, and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

(f) I have not visited the Mexican Hat Project property in Cochise County, Arizona.

(g) I am co-author and take responsibility for Sections 21.1 (except 21.1.7, 21.1.9, 21.1.10 and 21.1.13), 22 and corresponding sections of 1, 25 and 26 of the Technical Report on the Mexican Hat Project, Cochise County, Arizona, USA issued on November 19, 2020.

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

I have no prior involvement with the property that is the subject of this Report and I hold no interests in, nor do I expect to receive any interests, direct or indirect from GMV Minerals Inc. ("GMV") or any associated or affiliated company. I am independent of the issuer, GMV, applying the tests set out in Section 1.5 of NI 43-101. I have read NI 43-101 and the Report has been prepared in compliance with NI 43-101 and form 43-101F1.

I consent to the filing of this report with any stock exchange or other regulatory authority and any publication by them, including electronic publication in the public company files on their web sites accessible by the public of this Report.

Signed and dated this 17th day of November 2020, at 8450 E. Crescent Pkwy, Ste. 200, Greenwood Village, Colorado, USA.

Steven A. Pozder, P.E., MBA
Colorado PE #29144

COLORADO REGISTERED PROFESSIONAL ENGINEER
STEVEN A. POZDER
29144

Expires: 10/31/21

CONSENT OF QUALIFIED PERSON – Dave R. Webb, Ph.D., P.Geol.

RE: GMV Minerals Inc.

I, Dave R. Webb, hereby consent to the public filing by GMV Minerals Inc. ("GMV" or "Client") of the technical report titled, "NI 43-101 Technical Report, Updated Preliminary Economic Assessment, Mexican Hat Project", dated November 19, 2020 (the "Technical Report").

I also consent to any extracts from, or a summary of, the Technical Report contained in GMV's press release ("PR") dated November 3, 2020.

I confirm that I have read the PR and that it fairly and accurately represents the information in the Technical Report for which I am responsible.

Dated this November 14, 2020

Dave R. Webb Ph.D., P.Geol.

1909 108 West Cordova St., Vancouver, B.C., Canada V6B 0G5
1 604 818-1400 www.drwgeological.com

Dave R. Webb Ph.D., P.Geol.

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

CERTIFICATE

I, Dave R. Webb Ph.D., P.Geol., of the City of Vancouver, B.C., do hereby certify that as the author (or co-author) of this Technical Report on the Mexican Hat Project, located in Cochise County, Arizona, USA, dated November 19, 2020, I hereby make the following statements:

(a) I am a Professional Geologist practicing at DRW Geological Consultants Ltd., 1909 108 W. Cordova St., Vancouver, B.C.
(b) I am a graduate of the University of Toronto with a B.A.Sc. degree in Applied Science and Engineering in 1981, Queens' University with an M.Sc. degree in Geological Sciences in 1983, and a Ph.D. in Geological Sciences in 1992.
(c) I am registered as a Professional Geologist with the Association of Professional Engineers and Geoscientists of B.C., Registration Number 49744.
(d) I have practiced my profession as a geologist in economic geology for over 35 years.
(e) I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101), and do certify that , by reason of my education, affiliation with a professional association (as defined by NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
(f) I have over 35 years of experience in the Mineral Resource industry, as President of DRW Geological Consultants Ltd.
(g) I have visited the Mexican Hat Project property and site facilities in 2014, 2015, 2016, 2017, and 2018 for exploration, trenching, and drill programs.
(h) I am responsible for the Technical Report Sections 4, 5, 6, 7, 8, 9, 10, 11, 15, 19, 23, 24 and corresponding sections of 1, 25 and 26

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

I have no prior involvement with the property that is the subject of this Report and I hold no interests in, nor do I expect to receive any interests, direct or indirect from GMV Minerals Inc. ("GMV") or any associated or affiliated company. I am independent of the issuer, GMV, applying the tests set out in section 1.4 of NI 43-101. I have read NI 43-101 and this Report has been prepared in compliance with NI 43-101 and form 43-101F1.

I consent to the filing of this report with any stock exchange or other regulatory authority and any publication by them, including electronic publication in the public company files on their web sites accessible by the public of this Report.

Signed and dated this 14 day of November, 2020, at Vancouver, B.C., Canada

Dr. Dave R. Webb, Ph.D., P.Geol.

TETRA TECH

150, 1715 Dickson Avenue
Kelowna, BC V1Y 9G6 CANADA

CONSENT OF QUALIFIED PERSON – James Barr, P.Geo.

RE: GMV Minerals Inc.

I, James Barr, P.Geo., hereby consent to the public filing by GMV Minerals Inc. ("GMV" or "Client") of the technical report titled, "NI 43-101 Technical Report, Updated Preliminary Economic Assessment, Mexican Hat Project", dated November 19, 2020 (the "Technical Report").

I also consent to any extracts from, or a summary of, the Technical Report contained in GMV's press release ("PR") dated November 3, 2020.

I confirm that I have read the PR and that it fairly and accurately represents the information in the Technical Report for which I am responsible.

Dated this November 18, 2020

James Barr, P.Geo.

James Barr, P.Geo.

I, James Barr, P.Geo., of Kelowna, British Columbia, do hereby certify that as the author (or co-author) of this Technical Report on the Mexican Hat Project, located in Cochise County, Arizona, USA, dated November 19, 2020, I hereby make the following statements:

- I am Senior Geologist and Team Lead with Tetra Tech Canada Inc. with a business address at Suite 150 - 1715 Dickson Avenue, Kelowna, BC, V1Y 9G6.
- I am a registered Professional Geoscientist with the Engineers and Geoscientists of British Columbia (#35150).
- I graduated from the University of Waterloo in 2003 with a B.Sc. (Honours) in Environmental Science, Earth Science and Chemistry.
- 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.
- I visited the Property that is the subject of the Technical Report from July 18-19, 2017.
- I confirm that I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with them.
    - I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101), and do certify that, by reason of my education, affiliation with a professional association (as defined by NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
    - I hold no interests in, nor do I expect to receive any interests, direct or indirect from GMV Minerals Inc. ("GMV") or any associated or affiliated company and state I am independent of GMV Minerals Inc., as defined by Section 1.5 of the Instrument,
- I was previously co-author and Qualified Person of the Technical Report " 2018 Technical Report and Mineral Resource Estimate on the Mexican Hat Project Cochise County, Arizona, USA", with effective date of June 22, 2018.
- I am responsible for Sections 12 and 14 of this Technical Report.

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 November, 2020 at Kelowna, British Columbia

PROVINCE OF BRITISH COLUMBIA
J. F. BARR
35150
PROFESSIONAL GEOSCIENTIST
2011/18

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

Thomas L. Dyer
Principal Engineer

Mine Development Associates

A Division of RESPEC

210 South Rock Blvd

Reno, NV 89502
775.856.5700 // c. 775.770.0249

CONSENT OF QUALIFIED PERSON – Thomas L. Dyer, PE

RE: GMV Minerals Inc.

I, Thomas L. Dyer, PE, hereby consent to the public filing by GMV Minerals Inc. ("GMV" or "Client") of the technical report titled, "NI 43-101 Technical Report, Updated Preliminary Economic Assessment, Mexican Hat Project", dated November 19, 2020 (the "Technical Report").

I also consent to any extracts from, or a summary of, the Technical Report contained in GMV's press release ("PR") dated November 3, 2020.

I confirm that I have read the PR and that it fairly and accurately represents the information in the Technical Report for which I am responsible.

Dated this 11/16/20

Thomas L. Dyer, PE

PROFESSIONAL ENGINEER - STATE OF NEVADA
THOMAS DYER
Exp. 12/31/20
MINING
No. 15729

Thomas L. Dyer, PE

Mine Development Associates
A division of RESPEC
210 Rock Blvd. Reno, NV 89502

775/856-5700
Tom.Dyer@RESPEC.com

CERTIFICATE

I, Thomas L. Dyer, PE, of the City of Reno, NV, do hereby certify that as an author of the technical report titled, "NI 43-101 Technical Report, Updated Preliminary Economic Assessment, Mexican Hat Project", dated November 19, 2020 (the "Technical Report") on the Mexican Hat Project, located in Cochise County, Arizona, USA. I hereby make the following statements:

(a) I, Thomas L. Dyer, P.E., do hereby certify that I am currently employed as Principal Engineer at Mine Development Associates, a division of RESPEC, whose address is 210 S. Rock Blvd., Reno, NV 89502

(b) I graduated from South Dakota School of Mines and Technology with a Bachelor of Science degree in Mine Engineering in 1996.

(c) I am registered as a Professional Engineer (P.E.) in the state of Nevada (#15729) and am a registered member (#4029995RM) of the Society of Mining, Metallurgy and Exploration.

(d) I have worked as a mining engineer for more than 24 years since my graduation. Relevant experience includes providing mine designs, reserve estimates and economic analyses, of precious-metals deposits and industrial minerals deposits in the United States and various countries of the world. During this period, I have worked as Chief Engineer of an operating gold mine in Nevada.

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

(f) I visited the Mexican Hat Project property and site facilities on July 13,2020.

(g) I am responsible for the Technical Report Sections: 16, 21.1.7, 21.2.1, and mining sections of 1, 25, and 26.

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

I have no prior involvement with the property that is the subject of this Report and I hold no interests in, nor do I expect to receive any interests, direct or indirect from GMV Minerals Inc. ("GMV") or any associated or affiliated company. I am independent of the issuer, GMV, applying the tests set out in section 1.4 of NI 43-101. I have read NI 43-101 and this Report has been prepared in compliance with NI 43-101 and form 43-101F1.

I consent to the filing of this report with any stock exchange or other regulatory authority and any publication by them, including electronic publication in the public company files on their web sites accessible by the public of this Report.

Signed and dated this 16th day of November 2020, at Reno, Nevada, United States of America.

Thomas L. Dyer, PE

1746 Cole Blvd., Suite 130
Lakewood, CO, 80401

CONSENT OF QUALIFIED PERSON – Francisco J Barrios

RE: GMV Minerals Inc.

I, Francisco J Barrios, hereby consent to the public filing by GMV Minerals Inc. (“GMV” or “Client”) of the technical report titled, “NI 43-101 Technical Report, Updated Preliminary Economic Assessment, Mexican Hat Project”, dated November 19, 2020 (the “Technical Report”).

I also consent to any extracts from, or a summary of, the Technical Report contained in GMV‘s press release (“PR”) dated November 3, 2020.

I confirm that I have read the PR and that it fairly and accurately represents the information in the Technical Report for which I am responsible.

Dated this 18th day of November 2020

Francisco J. Barrios, P.E., MBA
Arizona PE #50454

Registered Professional Engineer (Civil)
CERTIFICATE NO.
50454
FRANCISCO J.
BARRIOS
Date Signed
ARIZONA U.S.A.
Expires 12/31/21

Francisco J. Barrios, P.E.

Tierra Group International, Ltd.
1746 Cole Blvd., Suite 130
Lakewood, CO 80401

520.999.5188
fbarrios@tierragroupintl.com

CERTIFICATE

I, Francisco J. Barrios, P.E., Professional Engineer (Arizona #50454), do hereby certify that as the author (or co-author) of this Technical Report on the Mexican Hat Project, located in Cochise County, Arizona, USA, dated November 19, 2020, I hereby make the following statements:

(a) I am a Project Manager practicing at Tierra Group International, Ltd., 1746 Cole Blvd., Suite 130, Lakewood, CO 80401, USA.

(b) I am a graduate of the University of Colorado with a B.Sc. in Civil Engineering in 2003. I am a graduate of the University of Arizona with a M.Sc. in Civil Engineering in 2010. I am a graduate of Thunderbird School of Global Management with a Master of Business Administration (MBA) in 2013.

(c) I am registered as a Professional Engineer (P.E.) with the Arizona State Board of Technical Registration, Registration Number 50454.

(d) I have practiced my profession for 16 years as a Civil Engineer on mining projects. Experienced in the design, planning, construction, and project management for a variety of international mining projects (Canada, Chile, Colombia, Dominican Republic, Mexico, Nicaragua, Saudi Arabia, and USA). Project experience includes tailings storage facilities (TSF), heap leach facilities (HLF), and waste management at scoping, pre-feasibility, feasibility, and detail engineering level.

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

(f) I have visited the Mexican Hat Project property and site facilities in 13 July 2020.

(g) I am co-author and take responsibility for Sections 17.5, 21.1.9, 21.1.10, 26.3 and corresponding sections of 1, 25 and 26 of the Technical Report on the Mexican Hat Project, Cochise County, Arizona, USA issued on November 19, 2020.

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

I have no prior involvement with the property that is the subject of this Report and I hold no interests in, nor do I expect to receive any interests, direct or indirect from GMV Minerals Inc. ("GMV") or any associated or affiliated company. I am independent of the issuer, GMV, applying the tests set out in section 1.5 of NI 43-101. I have read NI 43-101 and this Report has been prepared in compliance with NI 43-101 and form 43-101F1.

I consent to the filing of this report with any stock exchange or other regulatory authority and any publication by them, including electronic publication in the public company files on their web sites accessible by the public of this Report.

Signed and dated this 18th day of November 2020, at 1746 Cole Blvd., Suite 130 Lakewood, CO 80401, USA.

Francisco J. Barrios, P.E., MBA
Arizona PE #50454

Registered Professional Engineer (Civil)
CERTIFICATE NO.
50454
FRANCISCO J.
BARRIOS
Date Signed
ARIZONA U.S.A.
Expires: 12/31/21

CONSENT OF QUALIFIED PERSON
Dawn Garcia, CPG, PG

I, Dawn Garcia, state that I am responsible for preparing and supervising the preparation of part of the technical report, titled "NI 43-101 Technical Report, Updated Preliminary Economic Assessment, Mexican Hat Project" with an effective date of October 20, 2020, as signed and certified by me (the "Technical Report").

Furthermore, I state that:

(a) I consent to the public filing of the Technical Report by GMV Minerals Inc.;

(b) The Technical Report supports the press release issued by GMV Minerals Inc., dated November 3, 2020, and titled "GMV Minerals Inc. Announces PEA Results at Mexican Hat Gold Project in S.E. Arizona" (the "Document");

(c) I consent to the use of extracts from, or a summary of, the Technical Report in the Document of GMV Minerals Inc.;

(d) I confirm that I have read the Document being filed by GMV Minerals Inc., and that it fairly and accurately represents the information in the Technical Report or part for which I am responsible.

Dated at Tucson, Arizona this 17th of November, 2020.

Dawn Garcia, CPG, PG

AMERICAN INSTITUTE OF PROFESSIONAL GEOLOGISTS
CERTIFICATE NUMBER
08313
AIPG
DAWN H. GARCIA
CERTIFIED PROFESSIONAL GEOLOGIST

CERTIFICATE OF QUALIFIED PERSON
Dawn Garcia, CPG, PG

I, Dawn Garcia, CPG, PG, state that:

(a) I am a Senior Geologist at :
Golder Associates Inc.
7458 N. La Cholla Blvd.
Tucson, Arizona, USA 85741

(b) This certificate applies to the technical report, titled "NI 43-101 Technical Report, Updated Preliminary Economic Assessment, Mexican Hat Project" with an effective date of October 20, 2020.

(c) I am a "qualified person" for the purposes of National Instrument 43-101 (the "Instrument"). My qualifications as a qualified person are as follows. I am a graduate of Bradley University with a bachelor's degree in Geological Sciences in 1982 and a graduate of California State University, Long Beach, with a master's degree in Geology in 1995. I am a licensed Professional Geologist in Arizona (License No. 26034) and am certified as a Professional Geologist (CPG) with the American Institute of Professional Geologists (Membership Number 08313). I am also a registered member of the Society for Mining, Metallurgy & Exploration (Membership No. 4135993). I have practiced my profession as an environmental geologist and hydrogeologist for over 35 years. I have over 20 years of experience in the mining industry. My relevant experience for the purpose of this Technical Report is:

- Acted as the Qualified Person for the Environmental, Permitting and Social section for 9 NI 43-101 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.

(d) My most recent personal inspection of each property described in the Technical Report occurred on July 13, 2020, and was for a duration of 1 day.

(e) I am responsible for Item 20 and corresponding parts of Items 1, 25 and 26 of the Technical Report.

(f) I am independent of the issuer as described in section 1.5 of the Instrument.

(f) My prior involvement with the property that is the subject of the Technical Report is as follows. I worked on the PEA that was submitted in 2018. I was QP for Item 20.

(g) I have read National Instrument 43-101. Item 20 and corresponding parts of Items 1, 25 and 26 have been prepared in compliance with this Instrument; and

(h) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the part of 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.

Dated at Tucson, Arizona, this 17th of November, 2020.

Dawn Garcia

AMERICAN INSTITUTE OF PROFESSIONAL GEOLOGISTS
CERTIFICATE NUMBER 08313
AIPG
DAWN H. GARCIA
CERTIFIED PROFESSIONAL GEOLOGIST

Dawn Garcia, CPG, PG

APPENDIX 28.2

HLF DESIGN SUMMARY MEMORANDUM

TIERRA GROUP INTERNATIONAL, LTD.

1746 Cole Blvd, Suite 130
Lakewood, CO 80401
P: 303.532.5300

Technical Memorandum

TO:	Project File
FROM:	Francisco Barrios P.E.
REVIEWED BY:	Troy Meyer P.E.
DATE:	1 October 2020
PROJECT NAME:	Mexican Hat PEA Heap Leach Facility
PROJECT NO.:	617
SUBJECT:	HLF Design Summary Memorandum
CC:	

1.0 Introduction

GMV Minerals is a junior gold development company focused on developing the Mexican Hat mine (Project) located in southeast Arizona. The Mexican Hat Preliminary Economic Assessment (PEA) requires constructing a new heap leach facility (HLF), process solution pond, and event pond.

The HLF is designed to meet or exceed the prescriptive Best Available Demonstrated Control Technology (BADCT) criteria as described in the Arizona Department of Environmental Quality (ADEQ) Arizona Mining BADCT Guidance Manual (ADEQ, 2004). Where appropriate, additional design criteria (not prescribed in BADCT) were included based on professional experience and judgment, standard engineering practices, and site-specific conditions.

2.0 Site Description

This section provides a summary of climatology, surface water hydrology, surface water control, and physiographic setting. This information was used, in part, to develop the Project design criteria presented in Section 3.0.

2.1 Climatology

Weather patterns at the Project will need to be studied to develop an understanding of the local climate. According to the Western Regional Climate Center (WRCC), the nearest weather station is Pearce Sunsites #026353. The data from the Pearce Weather Station has a period of record that extends from September 1913 through May 2016. The property experiences an average of 30 centimeters (cm) of annual precipitation and temperatures ranging from 25°C to 40°C during the summer and 5°C to 10°C during the winter. Table 2.1 summarizes the average monthly climate.

TABLE 2.1: PEARCE WEATHER STATION CLIMATE SUMMARY

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Tot.
Avg. Max. Temp. (°C)	16.2	18.1	21.1	25.4	30.1	34.9	34.4	32.8	31.5	27.0	20.5	15.9	25.7
Avg. Min. Temp. (°C)	-1.3	0.0	2.2	5.4	9.7	14.8	18.0	16.9	13.7	7.9	1.9	-1.4	7.3
Avg. Total Precip. (cm)	2.0	1.78	1.27	0.51	0.51	1.27	7.11	7.87	3.05	2.03	1.27	2.03	30.7
Avg. Total Snow Fall (cm)	1.0	0.76	0.25	0.25	0.0	0.0	0.0	0.0	0.0	0.0	0.25	0.0	2.5
Avg. Snow Depth (cm)	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

2.2 Surface Water Hydrology

Stormwater control structures will be designed around the HLF. These structures will be designed to contain or pass a specific storm event. National Oceanic and Atmospheric Administration (NOAA) data was used to develop storm event precipitation depths. Table 2.2. presents the flood frequency analysis rainfall depths obtained from NOAA.

TABLE 2.2: STORM EVENTS 90% CONFIDENCE INTERVAL (IN MILLIMETERS)

Duration	Average Recurrence Intervals (years)						
	1	2	5	10	25	50	100
1-Hour	22	29	37	44	52	59	65
3-Hour	27	34	43	51	61	69	78
6-Hour	31	38	48	57	69	78	88
24-Hour	38	48	59	68	80	90	99

2.3 Surface Water Control

Surface water control structures will involve diversion channels, culverts, erosion control structures, berms, and others. For the PEA, Tierra Group only included the main diversion channel located south of the proposed HLF. The proposed diversion channel is sized to convey the peak flow from the 100-year 24-hour storm, and will have the following characteristics: trapezoidal, 2H:1V (horizontal:vertical) side slopes, 2.5-meter (m) base width, 2.1-m height, 1% minimum slope, and an approximate length of 1.5 kilometers (km). Surface water control structure design will be further advanced during the next stage of studies.

2.4 Physiographic Setting

According to M3's NI 43-101 Technical Report issued on December 2018:

> *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.*

3.0 Heap Leach Facility Engineering Design

The following section contains a general description of the HLF and basis of design used to meet or exceed Prescriptive BADCT recommendations established by the ADEQ.

3.1 General Description

The final location for the HLF and ponds was selected considering the available area within the Project property and the location of other Project facilities. The HLF will be a single-use, multi-lift type HLF and has been designed with a lining system in accordance with BADCT criteria as described in ADEQ Arizona Mining BADCT Guidance Manual.

The HLF is located in an area of flat to gently sloping topography that will require some grading in the HLF footprint. The facility surface is generally undisturbed with small shrubs, bushes, and desert cacti. All vegetative cover, organic soils, and growth media will be removed prior to construction. The HLF, which includes the HLF, process solution pond, and event pond is planned to be located north of the proposed pit. The HLF will be constructed in two phases and has been designed for a nominal production rate of 3,500,000 tonnes of ore per year (10,000 tonnes/day (tpd)) for a total heap capacity of 32.6 million tonnes (Mt) assuming a heap bulk density of 1.5 metric tons per cubic meter (t/m^3). The ore will be mined by a standard open pit mining method, crushed to 80% minus 38 millimeters (mm), and placed through transport and stacking on the HLF (in 10-m-high lifts) using a conveyor/stacking system. The HLF is anticipated to have a maximum height of 72 m and an overall slope of 2.5H:1V.

The HLF will consist of:

- Pregnant Leach Solution (PLS) and event pond designed to meet or exceed the Prescriptive BADCT recommendations established by ADEQ (Tables 3.1 and 3.2);
- HLF designed to meet or exceed the BADCT recommendations established by ADEQ. Lined HLF with approximately 565,512 square meters (m^2) of lined area (Phase 1 construction includes 192,201 m^2 and Phase 2 covers approximately 373,311 m^2) with properties described in Table 3.3; and
- Pad Overliner Drain Fill and piping system are listed in Table 3.4. The pad Overliner Drain Fill provides liner protection from exposure to the climate, vehicle tracks, and ore placement via haul trucks. The Overliner Drain Fill also reduces the hydraulic head on the pad liner when constructed in combination with supplemental drain pipes placed at a spacing determined by the leaching solution application rate and the permeability characteristics of the drain rock. Additionally, a piping system distributed throughout the limits of the facility designed to collect and convey PLS in addition to stormwater.

TABLE 3.1: PLS POND

Tasks	Description
Pond Design Depth	9.5 m (from the lowest point).
Pond Bottom Grade	Grade to drain to corner sump.
Freeboard	1 m
Interior Berm Crest Width	5 m minimum.
Berm Slopes	2H:1V lined interior.
Clay Soil Liner	GCL or a minimum of 12-inch low permeability soil liner with a no greater hydraulic conductivity of 1×10^{-6} cm/s.
Top Geomembrane Liner	80-mil textured HDPE drain liner, or equivalent. Drain Liner TM is a typical geomembrane liner with a pattern of raised studs to provide a liner and drainage layer in one product. This eliminates the need for a separate geonet layer.
Bottom Geomembrane Liner	80-mil double-sided textured HDPE liner.
Leak Detection System	Liner sloped to sump at a minimum 3.0% slope. Leak detection system consisting of an 80-mil textured HDPE Drain LinerTM or equivalent on the pond slopes and bottom to corner leak detection sump and well system (150-mm typical HDPE pipe placed

Tasks	Description
	between liners). The drain liner can be replaced with a minimum 200-mil geonet between geomembranes. Leak detection system
Pond Liner Anchor Trench	0.6-m wide by 0.6-m minimum depth trench Backfill with Bedding Fill compacted in 0.15-m lifts to a minimum 95% of maximum dry density (ASTM D698).
Pond Sizing	Minimum 24-hour Operating Volume: 35,343 m3 Maximum Average Seasonal Volume: 28,758 m3 12-Hour Operational Upset Draindown Volume: 14,280 m3 Total Pond Capacity with 1m Freeboard: 78,381

TABLE 3.2: EVENT POND

Tasks	Description
Pond Design Depth	9.5 m
Pond Bottom Grade	Grade to drain to corner.
Freeboard	1 m
Interior Berm Crest Width	5 m minimum
Berm Slopes	2H:1V lined interior.
Components	Prepared subgrade, GCL or low permeability soil liner if available and 80-mil double-sided textured HDPE liner.
Pond Liner Anchor Trench	0.6-m wide by 0.6-m minimum depth trench Backfill with Bedding Fill compacted in 0.15-m lifts to a minimum 95% of maximum dry density (ASTM D698).
Pond Sizing	100-year, 24-hour Storm Runoff Volume: 62,324 m^3 Dead Volume (1m soil and debris buildup): 6,235 m^3 Total Pond Capacity with 1m Freeboard: 68,559
Perimeter Roads	5 m minimum road crest width along outside edge of ponds (with a safety berm)

TABLE 3.3: HEAP LEACH PAD

Tasks	Description
Expected Ore Tonnage (per PEA Mine Plan)	32.6 Mt
Ore Production	10,000 tpd (metric)
Ore Processing	P80: 38 mm
Ore Height	72 meters maximum height. Scarify the leached top lift for successive lift placement.
Heap Overall Slope	2:5H to 1V, 21.8° (to be verified based on stability analysis)
Stack/Lift Height	Individual ore lifts (10-m height) stacked at natural angle-of-repose (with benches width as required for design slope of 2.5H:1V).
Ore Setback	5-m minimum setback from the inside edge of perimeter berm limits.
Ore Density	Stacked ore density 1.5 t/m^3 (assumed)
Ore Moisture Content	To be determined
Ore Geotechnical Parameters	To be determined
Ore Angle of Repose	37° (1.3H:1V)
Seismicity	To be determined per BADCT guidance
Ore Stack Factor of Safety	Minimum static factor of safety = 1.3. Minimum pseudo-static factor of safety = 1.0. Minimum post-earthquake factor of safety = 1.2
Groundwater Flow	To be determined

TABLE 3.4: PAD OVERLINER AND PIPING SYSTEM

Tasks	Description
Drain Pipes	100-mm diameter corrugated and perforated polyethylene (PE) N-12, or equivalent, primary pipes placed in a herringbone fashion placed on 6 m maximum centers (to be confirmed based on ODF permeability testing). 450-mm diameter corrugated and perforated PE N-12, or equivalent, secondary pipes spaced as necessary to handle the solution application. 600-mm diameter corrugated and perforated header pipes spaced as necessary to handle the solution application flows plus estimated flows from the design storm event. 600-mm diameter solid HDPE discharge pipes to route flows to the PLS Pond. Maximum allowable deflection under load of 20%
Over-liner Drain Fill	The HLF geomembrane liner will be covered by a minimum of 0.6 m of Over-liner Drain Fill, well-graded, and free-drainage granular material with less than 5 percent particles passing the No. 200 ASTM sieve size. No moisture conditioning or compaction of the Over-liner Drain Fill is required. Hydraulic Conductivity should maintain a minimum of one order of magnitude higher permeability compared to the overlying ore heap.

3.2 Heap Leach Facility Design Criteria

The HLF is designed as a zero-discharge system with all process solutions and precipitation volumes over the lined areas being contained within the lined facility. Key heap leach design criteria are presented in Table 3.5.

TABLE 3.5: HEAP LEACH FACILITY DESIGN CRITERIA

Tasks	Description
Expected Ore Tonnage (per PEA Mine Plan)	32.6 Mt
Ore Production	10,000 tpd (metric)
Ore Processing	P80: 38 mm
Ore Height	72-m maximum height. Scarify the leached top lift for successive lift placement.
Heap Overall Slope	2:5H to 1V, 21.8° (to be determined based on stability analysis)
Stack/Lift Height	Individual ore lifts (10-m height) stacked at natural angle-of-repose (with benches width as required for design slope of 2.5H:1V).
Ore Setback	5 m minimum setback from the inside edge of perimeter berm limits.
Ore Density	Stacked ore density 1.5 t/m^3 (assumed)
Ore Angle of Repose	37° (1.3H:1V)
Seismicity	To be determined per BADCT guidance
Ore Stack Factor of Safety	Minimum static factor of safety = 1.3. Minimum pseudo-static factor of safety = 1.0.
Leaching Cycle	Normal 90 days leaching 120 Total (includes stacking rinsing)
Active Leach Surface	119,048 m^2
Solution Application Method	Buried Driplines or Wobbler Sprinklers
Solution Application Rate	10 L/h/m^2
Pad Liner System	Minimum of 0.15 m thick layer of properly compacted liner bedding fill (prepared subgrade) GCL which is equivalent to having a 0.3-m layer of compacted low permeability soil having a permeability no greater than 10^{-6} cm/s 80-mil LLDPE liner (double-side textured) Minimum 0.6 m of over-liner drain fill

3.3 Heap Leach Facility Stacking Design Criteria

The following section contains a general description of the HLF stacking and basis of design used to meet the stacking plan.

The HLF will be stacked in two phases, Phase 1 = years 1 - 2 and Phase 2 = years 3 - LOM. Ore will be crushed to 80% minus 38 mm and placed in 10-m-high lifts using a conveyor/stacking system. The HLF stacking rate is designed for 10,000 tpd for a total of 3,500,000 tonnes per year. Key heap leach stacking design criteria are presented in Table 3.6.

TABLE 3.6: HEAP LEACH STACKING DESIGN CRITERIA

Tasks	Description
Stack/Lift Height	Individual ore lifts (10-m height) stacked at natural angle-of-repose (with bench widths as required for design slope of 2.5H:1V).
Phase 1 Stacking Equipment	Grasshoppers + Radial Stacker
Grasshoppers	16 – 30 m length x 91 cm wide, 22 kW motor
Radial Stacker	1 – 37 m length x 107 cm wide, 15 m lift, self-drive, 60 kW motor
Phase 2 Stacking Equipment	Overland Conveyor + Grasshoppers + Radial Stacker
Overland Conveyor	1 – 300 M length x 91 cm wide, 112 kW motor
Additional Grasshoppers	4 – 30 m length x 91 cm wide, 22 kW motor
Existing Radial Stacker	1 – 37 m length x 107 cm wide, 15 m lift, self-drive, 60 kW motor

4.0 Cost Estimate

Preliminary site infrastructure associate with HLF and ponds have been evaluated and a conceptual arrangement was defined as the basis of capital cost estimate. The cost of construction of the HLF was developed based on supplier quotes. Cost were gathered for the following items: earthwork (Titan Construction), GCL (Cetco), geomembrane (AGRU America), and piping system (ADS). The estimated cost is considered to have an accuracy of +/- 40% and a 30% contingency (Phase 1) and 0% contingency (Phase 2 – per directions from Samuel Engineering) was used to account for unknown (water management works, access and perimeter road earthworks, and others). Operating cost estimate was developed by Samuel Engineering with input assistance from Tierra Group.

The HLF is expected to be constructed in 2 phases (Phase 1 built prior to beginning production and capacity to store ore for 2 years, and Phase 2 built after year 2). Table 4.1 present a summary of the estimated initial construction cost, and Table 4.2 present a summary of the estimated sustaining construction cost incurred after production starts as follow:

TABLE 4.1: HEAP LEACH FACILITY INITIAL COST (PHASE 1)

Tasks	Cost (USD$)
Phase 1 HLF	$7,369,724
Phase 1 Stacking System	$1,324,000
Phase 1 Contingency (30%)	$2,608,117
Phase 1 (Total)	**$11,301,841**

TABLE 4.2: HEAP LEACH FACILITY SUSTAINING COST (PHASE 2)

Tasks	Cost (USD$)
Phase 2 HLF	$10,631,850
Phase 2 Stacking System	$664,000
Phase 2 Contingency (0%)	Per direction from SE
Phase 2 (Total)	**$11,295,850**

5.0 Recommendations

In order to evaluate the technical and economic viability of the HLF and advance the engineering design to a Pre-Feasibility level, the following recommendations should be considered:

- Conduct a geotechnical field investigation and laboratory testing program recommended to facilitate pre-feasibility design level of the HLF for the foundation and borrow materials. Subsurface conditions should be characterized in sufficient detail to provide a high level of confidence in the stratigraphy, ground water, structure, and preferential paths of flow in the upper 30 meters of the subsurface to satisfy standards of practice for design of heap leach facilities;
- Conduct geotechnical testing of the leach ore and proposed drain materials;
- Conduct liner interface strength testing of the proposed liner system;
- Evaluate the hydrogeological conditions that exist at the Mexican Hat Project specifically in the vicinities of the HLF;
- Conduct geological mapping and geohazard studies;
- Monitor groundwater levels to evaluate seasonal fluctuations;
- Monitor existing stream flows to measure sediment transportation in existing streams. This will provide valuable input for designing sediment control structures;
- Additional site-specific precipitation and evaporation measurements should be collected to better refine the predictions of the HLF process fluid water balance; and
- Additional effort should be made to optimize geometry of the proposed facilities.
- Additional effort should be made to optimize geometry of the proposed stacking layout;
- Trade-off evaluation between conveyor stacking vs. truck stacking;
- Trade-off evaluation of stacking 24 hours per day vs day shift only; and
- For pre-feasibility level study, Tierra Group estimate a $500,000 to $1,000,000 budget will be necessary to complete tasks recommended in this report.