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NI 43-101 TECHNICAL REPORT
PRELIMINARY FEASIBILITY STUDY

TELSON RESOURCES
TAHUEHUETO PROJECT
DURANGO, MEXICO

EFFECTIVE DATE: DECEMBER 6, 2016
REPORT DATE: JANUARY 20, 2017

PREPARED BY
METAL MINING CONSULTANTS INC.

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CO-AUTHORS
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TIMOTHY J. GEORGE, P.E.

DATE AND SIGNATURE PAGE

NI 43-101 Technical Report, Preliminary Feasibility Study, Telson Resources, Tahuehueto Project, Durango, Mexico

The effective date of this report is December 6, 2016

Dated January 20, 2017

[Signed and Sealed] Scott E. Wilson

Scott E. Wilson, C.P.G.
Geologist

[Signed and Sealed] Carl E. Brechtel

Carl E. Bechtel, P.E.
Mining Engineer

[Signed and Sealed] Timothy J. George

Timothy J. George, P.E.
Mining Engineer

AUTHOR'S CERTIFICATE - SCOTT WILSON

I, Scott E. Wilson, of Highlands Ranch, Colorado, do hereby certify:

1. I am currently employed as President by Metal Mining Consultants Inc., 9137 S. Ridgeline Blvd., Suite 140, Highlands Ranch, Colorado 80129.
2. I graduated with a Bachelor of Arts degree in Geology from the California State University, Sacramento in 1989.
3. I am a Certified Professional Geologist and member of the American Institute of Professional Geologists (CPG #10965) and a Registered Member (#4025107) of the Society for Mining, Metallurgy and Exploration, Inc.
4. I have been employed as either a geologist or an engineer continuously for a total of 28 years. My experience included resource estimation, mine planning, geological modeling, geostatistical evaluations, project development, and authorship of numerous technical reports and feasibility studies of various projects throughout North America, South America and Europe. I have been involved in the evaluation and conceptual development of new and existing mining projects, managing studies to develop projects and managing due diligences for acquisitions. I have prepared capital and operating budgets and developed mining and haulage studies. I have overseen and developed operational schedules and implemented mine plans to meet targets. I have been involved with the installation of truck dispatch systems, drill and blast monitoring systems, mining equipment purchases, and construction projects. I have employed and mentored mining engineers and geologists continuously since 2003.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
6. I visited the Tahuehueto project and surrounding area the week of November 7, 2016.
7. I am responsible for sections 1-13, 14, 17, 20 and 23-27 of the technical report titled "Preliminary Feasibility Study, Telson Resources, Tahuehueto Project, Durango, Mexico," dated January 20, 2017 with an effective date of December 6, 2016 (the "Technical Report.").
8. I have had prior involvement with the property that is the subject of the Technical Report as an independent mining consultant.
9. As of the effective date of this report, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portion of the Technical Report for which I am responsible not misleading.
10. That I have read NI 43-101 and Form 43-101F1, and that this Technical Report was prepared in compliance with NI 43-101.
11. I am independent of the issuer as independence is described in Section 1.5 of NI 43-101.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated January 20, 2017

["Signed and Sealed"]
Scott E. Wilson, C.P.G.

AUTHOR'S CERTIFICATE - CARL E. BRECHTEL

I, Carl Brechtel, of Arvada, Colorado, do hereby certify:

1. I am currently Principal of Carl Brechtel Consulting LLC, 6439 Umber Circle, Arvada, Colorado 80007.
2. I am a co-author of the technical report titled "NI 43-101 Technical Report, Preliminary Feasibility Study, Telson Resources, Tahuehueto Project, Durango, Mexico" (the "Technical Report") dated January 20, 2017 with an effective date of December 6, 2016.
3. I graduated with a Bachelor of Science degree in Geological Engineering in 1973 and a Master of Science degree in Mining Engineering in 1978 from the University of Utah, Salt Lake City, Utah.
4. I am a Professional Engineer in Colorado (23212) and Nevada (8744) and a Registered Member (#353000) of the Society for Mining, Metallurgy and Exploration, Inc.
5. I have been employed as a mining engineer continuously for a total of 42 years. My experience included mining reserve estimation, mine planning and design, rock mechanics design, underground ventilation design and mine operations management in numerous projects throughout North and South America, Africa and Australia.
6. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
7. Due to scheduling conflicts I have not visited the property and have relied on the site visit of Mr. Scott Wilson. However, a site visit is planned for the first quarter of 2017.
8. I am responsible for section 15, section 16, and relevant portions of sections 1, 21 and 25 of the Technical Report.
9. As of the date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
10. I have had no prior involvement with the property that is the subject of the Technical Report as an independent mining consultant.
11. That I have read NI 43-101 and Form 43-101F1, and this Technical Report was prepared in compliance with NI 43-101.
12. I am independent of the Company. Applying all of the tests in Section 1.5 of NI 43-101.
13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

January 20, 2017

["Signed and Sealed"]
Carl Brechtel, PE

AUTHOR'S CERTIFICATE – TIMOTHY J. GEORGE

I, Timothy J. George, of Reno, Nevada, do hereby certify that:

1. I am a consulting mining engineer and Principal of Wildcat and Badger, LLC, 3690 Bozeman Drive, Reno, NV 89511.
2. I am a co-author of the technical report titled "NI 43-101 Technical Report, Preliminary Feasibility Study, Telson Resources, Tahuehueto Project, Durango, Mexico" (the "Technical Report") dated January 20, 2017 with an effective date of December 6, 2016.
3. I am a graduate of the University of Arizona, with a BS in Mining Engineering.
4. I am a licensed Professional Engineer in the States of Colorado, USA (No. 47109) and I am a member of the Society for Mining Metallurgy & Exploration.
5. I have been employed as a mining engineer continuously for a total of 9 years. My experience included mining reserve estimation, mine planning and design, in numerous projects throughout North and South America.
6. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
7. Due to scheduling conflicts I have not visited the property and have relied on the site visit of Mr. Scott Wilson. However, a site visit is planned for the first quarter of 2017.
8. I am responsible for section 21, and section 22 of the Technical Report.
9. As of the date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
10. I have had no prior involvement with the property that is the subject of the Technical Report as an independent mining consultant.
11. That I have read NI 43-101 and Form 43-101F1, and this Technical Report was prepared in compliance with NI 43-101.
12. I am independent of the Company. Applying all of the tests in Section 1.5 of NI 43-101.
13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

January 20, 2017

["Signed"] Sealed

Timothy J George, PE

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1 SUMMARY

Telson Resources Inc. ("Telson" or the "Company") requested that Metal Mining Consultants Inc. ("MMC") prepare a Pre-Feasibility Study ("PFS") for its 100% owned Tahuehueto Project. This report was authored by Scott E. Wilson of MMC along with contributions from other industry experts. This PEA has been prepared in compliance with Form 43-101F1 (Technical Report) and Companion Policy 43-101CP. The effective date of the report is November 18, 2016.

The Tahuehueto Project ("Tahuehueto" or the "Project) is an advanced exploration stage polymetallic project. The mineralization consists of epithermal Au-Ag veins and brecciated structures with lead, zinc and copper, and is located in the Durango State within the prolific Sierra Madre Mineral Belt which hosts a series of historic and producing mines and most of México´s active exploration and development projects.

From 1996 to present day, Telson and Real de la Bufa, S.A. de C.V., a Mexican subsidiary of Telson, have conducted surface and underground sampling and mapping, drilled 248 holes totaling 47,276 m into several mineralized bodies, and conducted metallurgical testing, as well as geophysics and other geological studies. The Project consists of 28 mining concessions that total 7,492.7889 ha.

The Project configuration evaluated in this PFS is an owner-operated 790 tpd underground mine that will utilize overhand cut and fill mining with conventional mining equipment in a blast/load/haul operation. Mill feed will be processed in a 550 tpd comminution circuit consisting of primary and secondary crushing, grinding in a single ball mill followed by three floatation circuits producing lead, copper, and zinc concentrates. The concentrates will be trucked from site for smelting and refining.

The highlights of this Pre-Feasibility Study report include:

- Post-tax Net Present Value ("NPV"), using an 8% discount, of $77M, with an internal rate of return ("IRR") of 36% and a payback period of three years.
- Pre-tax NPV, using an 8% discount, of $138M with an IRR of 56%.
- Financial Analysis completed on base case metal price forecasts of $0.87/lb for lead, $0.92/lb for zinc, $2.65/lb for copper, $1,180/oz for gold and $16.70/oz for silver.
- Metal Prices lower than 3-year averages.
- Average annual earnings before interest, taxes, depreciation, and amortization ("EBITDA") of $16.7M per year and $352M over the life of the Project.
- Probable Mineral Reserves of 3.3 million tonnes, grading 3.4 g/t gold, 41.8 g/t silver, 0.31% copper, 1.1% lead and 2.0% zinc.
- 21-year mine life with average annual production of 16,100 oz of gold, 177,100 oz of silver, 900 k-lbs of copper, 3,200 k-lbs of lead and 5,600 k-lbs of zinc.
- Pre-production capital costs of $32.2M including $17.2M surface site development including mill construction and $14.9M of mining equipment and preliminary underground development.

1.1 MINERALIZATION

Mineralization at Tahuehueto occurs as polymetallic epithermal veins with multiple mineralizing events overprinted on one another in the same vein structure. The primary host rock is andesite of the lower volcanic series, but in at least one case, the hydrothermal system penetrated felsic ignimbrite of the upper volcanic series. Breccias are an integral part of the Tahuehueto hydrothermal system and display several genetic styles. Many of the sulfide-mineralized zones display sulfide transport textures.

Overprinting of the lower-temperature, higher-level mineral assemblage onto the higher temperature, deeper-level mineral assemblage is referred to as telescoping. This telescoping may represent the progressive cooling of the hydrothermal system, although in some instances tectonic un-roofing of the cover rocks may also result in a decrease in overburden and progressive deposition of higher crustal level, lower temperature mineral assemblages. Increasing gold and silver grades in the later higher crustal level assemblages without significant base metals is an important element of this telescoping.

The uppermost portions of the mineralized structures are oxidized. In the oxide zone, mineralization consists of malachite, azurite, chalcocite, covellite, limonite, and hematite. Malachite overprints tetrahedrite, and chalcocite and covellite form coatings on sphalerite. The depth of the oxide-sulfide interface varies considerably, but is generally less than 100 m.

Sulfide mineralization lies below the oxidized zone and consists of sphalerite, galena, chalcopyrite, tennantite, tetrahedrite, and probably electrum. Gangue minerals are quartz, pyrite, chlorite, sericite, and calcite. Locally a light green phyllosilicate mineral interpreted to be celadonite forms as gangue and is closely associated with high-grade gold and silver mineralization.

1.2 MINERAL RESOURCE ESTIMATE

The mineral resource has been limited to mineralized material that occurs within the mineralized blocks and which could be scheduled to be processed based on the defined cut-off grade. All other material was reported as non-mineralized material.

Table 1.1 below, lists the current Mineral Resource estimate for the Project at cut-off grade of 2.5 g/t of gold equivalent (AuEq).

Table 1.1 Tahuehueto Project Measured, Indicated, and Inferred Mineral Resource Estimate

Classification	kTonnes	Au Grade (gpt)	Cont Au kOz	Ag Grade (gpt)	Cont Ag kOz	Cu Grade (%)	Cont Cu klbs	Pb Grade (%)	Cont Pb klbs	Zn Grade (%)	Cont Zn klbs
Total Measured	2,771	2.77	247	44.70	3,982	0.31	18,914	1.27	77,827	2.29	139,821
Total Indicated	3,343	2.23	240	41.26	4,435	0.30	22,466	1.15	84,455	2.04	155,687
Total Measured and Indicated	**6,114**	**2.48**	**487**	**42.82**	**8,417**	**0.31**	**41,380**	**1.20**	**162,282**	**2.15**	**295,508**
Total Inferred	3,501	1.31	147	37.59	4,230	0.27	20,469	1.34	103,080	2.44	188,409

1.3 MINING ENGINEERING

Mineralization at Tahuehueto occurs in different veins under a local mountainous landform. The near vertical dip of the veins and apparent rock mass quality, as demonstrated by the excavations formed by the previous mining and recent underground development in late 2016, makes the veins suitable for different sublevel mining methods. No trade-off studies have been performed and the Telson Management had based previous mining studies on the Overhand Cut-and-Fill mining method, which in the Author's opinion was suitable for the purposes of this PFS study.

The mining method used as the basis of this PFS design was, therefore, Overhand–Cut and–Fill mining with conventional drilling, blasting, mucking and hauling, scaling and ground support installation and backfilling with unconsolidated, mined waste materials. Full mechanization was assumed using diesel, rubber tired mining equipment and support vehicles. A summary of Project operating metrics are presented in Table 1.2.

Table 1.2 Tahuehueto Project Operating Metrics

Operating Metrics	Units	Value
Mill Throughput	t/year	155,000
Mine Life	Years	21
Pre-Production/Development Period	Years	2
Ore Mining Rate	t/year	155,000
Development Rate	t/year	77,000
Total Mining Rate	t/year	232,000
Development Tonnes to Ore Tonnes Ratio*	w/o	0.5

* waste:ore ratio

The mine design and Mineral Reserve estimate have been completed to a level appropriate for pre-feasibility studies. The Mineral Reserve estimate stated herein is consistent with the CIM Standards on Mineral Resources and Mineral Reserves and is suitable for public reporting. As such, the Mineral Reserves are based on Measured and Indicated Resources, and do not include any Inferred Resources.

A mine design was created in the Maptek Vulcan™ model to define access and mining of the stope shapes defined by the Stope Optimizer module within Vulcan™ software. The defined stope shapes and development excavations were scheduled to produce a basis for economic analysis. The resulting reserve is classified as Probable, and is listed in Table 1.3. No Proven Reserves were defined due to the limited definition resource drilling, limited definition by exploratory mining and the lack of geotechnical data that addresses underground mining.

Table 1.3 Tahuehueto Reserve Estimate Summary from Scheduled Stopes

Classification	kTonnes	Au Grade (gpt)	Cont Au kOz	Ag Grade (gpt)	Cont Ag kOz	Cu Grade (%)	Cont Cu klbs	Pb Grade (%)	Cont Pb klbs	Zn Grade (%)	Cont Zn klbs
Probable Reserves	3,264	3.40	356	41.80	4,387	0.35	25,028	1.19	85,762	2.24	161,314

1. Canadian Institute of Mining, Metallurgy and Petroleum standards were followed in the estimation of the Mineral Reserves.
2. Mineral Reserves are estimated using metal price forecasts of \$0.60/lb for lead, \$0.75/lb for zinc, \$2.10/lb for copper, \$1,000/oz for gold and \$19.12/oz for silver.
3. Totals may not add due to rounding.
4. The foregoing mineral reserves are based upon and are included within the current mineral resource estimate for the Project.

1.4 MINERAL PROCESSING

Based on metallurgical tests performed to date, the previous metallurgical campaigns provide sufficient data to reach a level of confidence that the flotation process chosen will work, flotation targets are attainable, and an economical concentrate can be produced. The proposed processing plant is a conventional crushing/milling/flotation/filtering process designed to process 165,000 tonnes per year in 300 operating days, equivalent to 550 tonnes per day through the grinding and flotation circuits producing lead, copper, and zinc concentrates. Concentrate recoveries are shown in Table 1.4 and life of mine metal production is shown in Table 1.5.

Table 1.4 Tahuehueto Average Metallurgical Recoveries

Product	kTonnes	Distribution % (Recoveries)				
		Au	Ag	Cu	Pb	Zn
Head	3,264	100%	100%	100%	100%	100%
Pb Concentrate	58	77.1%	62.8%	31.6%	85.5%	1.6%
Cu Concentrate	18	6.8%	10.3%	51.4%	0.6%	17.1%
Zn Concentrate	108	11.0%	11.7%	11.5%	6.1%	80.0%
Tails	3,096	5.4%	15.2%	5.4%	7.8%	1.3%

Table 1.5 Life of Mine Metal Production

LOM Metal Production	Units	Value
Gold	koz	340
Silver	koz	3,720
Lead	klbs	73,100
Copper	klbs	20,800
Zinc	klbs	128,700

1.5 CAPITAL AND OPERATING COST ESTIMATE

Capital and operating costs used for the Tahuehueto Project were developed from cost build up from first principles engineering along with vendor and contractor quotations. In addition, all available project technical data and metallurgical test work were considered to build up a processing operating cost estimate.

A project configuration which included the underground mines and a central process facility was developed as the basis for capital cost estimation. Preliminary site infrastructure alternatives (process plant, tails storage facility, and power) were examined as a basis to estimate costs. Generalized arrangements were evaluated to establish a physical basis for the capital costs estimates. Cost accuracy is estimated to be + or - 20%. The estimated capital costs are listed in Table 1.6 and operating costs are listed in Table 1.7.

Table 1.6 Tahuehueto Total Capital Costs

Capital Category	Initial ($M)	Sustaining ($M)	Total ($M)
Mine Mobile Equipment	8.8	4.4	13.2
Mine Fixed Equipment	1.1	0.3	1.4
Mine Development	5.1	0.7	5.8
Processing	16.1	-	16.1
Infrastructure	1.1	-	1.1
Total CAPEX	**32.2**	**5.4**	**37.6**

Table 1.7 Tahuehueto Unit Operating Costs

Operating Costs	LOM Cost $M	Unit Cost $/t mineralized
Mining	69.4	21.62
Processing	100.5	30.80
G&A	22.2	6.82
Total OPEX	**192.2**	**59.24**
Smelter	78.0	23.91
Freight & Marketing	15.7	4.80
Royalties	9.5	2.91
Total Operating Cash Cost	**295.4**	**90.86**

1.6 CONCLUSIONS

The work completed by Telson has resulted in sufficient drill sample density, and confidence in the geological interpretation, for MMC to reasonably estimate Mineral Resources and Mineral Reserves for Tahuehueto.

The economic performance of the Tahuehueto Project was evaluated with a cash flow based economic model using project costs and revenues as the financial basis. The revenue factors for the project are dependent on metal prices calculating into the net smelter return. Costs are in constant 2016 US$, no escalation of cost has been assumed. Operating costs are generated based on production physicals (tonnes) and unit rates. The Tahuehueto project is expected to yield an after-tax undiscounted LOM net cash flow of $209.7 million, and an NPV of $77.8 million at a discount rate of 8% per year. The results for the Tahuehueto Project economic analysis are summarized in Table 1.8.

Table 1.8 Tahuehueto Project Economic Results

Economic Metrics	Units	LOM Value
Total Ore Processed	kTonnes	3,264
Contained Gold Produced	kOz	340
Contained Silver Produced	kOz	3,720
Contained Lead Produced	kLb	73,100
Contained Copper Produced	kLb	20,800
Contained Zinc Produced	kLb	128,700
Gold Price	$/oz	1,180
Silver Price	$/oz	16.70
Copper Price	$/lb	2.65
Lead Price	$/lb	0.87
Zinc Price	$/lb	0.92
Gross Revenue	$M	590.8
Refining and Freight Costs	$M	91.3
Royalty (1.6%)	$M	9.5
Operating Costs	$M	192.2
Capital Costs	$M	37.6
Pre-Tax Cash Flow	$M	351.5
Special Mining Tax (7.5%)	$M	36.6
Special Mining Royalty (0.5%)	$M	2.2
Income Tax (30%)	$M	104.8
Post-Tax Cash Flow	$M	207.9
Pre-tax NPV (8%)	$M	137.8
Pre-tax IRR	%	56%
Post-tax NPV (8%)	$M	77.0
Post-tax IRR	%	36%

1.7 RECOMENDATIONS

This Tahuehueto Project review has indicated economic potential at current metal prices based on the economic analysis of the potential project. The project warrants advancement with the consideration of moving in the direction of an eventual feasibility study ("FS"). Infill drilling is warranted, with the goal of converting Probable Reserves to higher confidence categories. Underground development work is recommended to access ore in the lower El Creston zone and the Cinco de Mayo zone for additional bulk metallurgical test work and to provide access for underground infill drilling. This underground development work should incorporate the mine plan so as to reduce future development costs. Environmental data collection should continue to support environmental studies. In addition, geotechnical characterization investigations on surface outcrop exposures, in underground portals and with geotechnical drilling is recommended to support mine design and detailed cost estimates. It is also recommended that the Company investigate the possibility of installing an on-site processing plant capable of processing the ore that is to be collected during the industrial scale bulk sampling of the lower El Creston and Cinco de Mayo zone's. If the economics are positive this could be a more efficient and cost-effective way to process these bulk samples. A budget of $4.8 million is estimated for this work to further data collection in order support the advancement of the project as set out in Table 1.9.

These recommendations comprise several aspects of project development which are not necessarily successive phases that rely on positive results of previous phases. MMC believes that development of this information will support the execution of a FS. It is MMC's recommendation that Telson continue the proposed project advancement program.

Table 1.9 Proposed Tahuehueto Project Advancement Program

Budget Item	Description	Cost (1,000's)
Underground Development, Drilling and Geotechnical Studies	Haulage Level Portal Construction, Conduct Infill Drilling, Geotechnical Studies and Bulk Sample Collection and processing	$4,000
Geology	Resource Model Updates	$100
Geotechnical, Groundwater Hydrology and Tailings	Field and Engineering Work	$300
Rock Mechanics, Metallurgy and Economics	UG Mine Planning, Detailed Designs, Cost Estimations, and Reserves	$250
Other	Continuing Environmental, and Permitting	$150
Total		**$4,800**

2 INTRODUCTION

The Tahuehueto Project is an advanced exploration stage polymetallic project. The mineralization consists of epithermal Au-Ag veins and brecciated structures with lead, zinc and copper. The Project is located in the State of Durango within the prolific Sierra Madre Mineral Belt which hosts a series of historic and producing mines and most of México´s active exploration and development projects.

Telson Resources Inc. owns 100% of the project, but the company had undergone two name changes, Soho Resources Corp. (until January 2013) and Consolidated Samarkand Resources Inc. (until October 1999). Throughout this report, Soho Resources Corp. and Consolidated Samarkand Resources Inc. are considered to be the same as and referred to as Telson Resources Inc. ("Telson" or the "Company").

From 1996 to present day, Telson and Real de la Bufa, S.A. de C.V., a Mexican subsidiary of Telson, have conducted surface and underground sampling and mapping, drilled 248 holes totaling 47,276 m into several mineralized bodies, and conducted metallurgical testing, as well as geophysics and other geological studies.

2.1 PURPOSE OF TECHNICAL REPORT

This report was prepared as a National Instrument 43-101 (NI 43-101) Technical Report for Telson by MMC. The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in MMC's services, based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by Telson Resources Inc. subject to the terms and conditions of its contract with MMC and relevant securities legislation. The contract permits Telson to file this report as a Technical Report with Canadian securities regulatory authorities pursuant to NI 43-101, Standards of Disclosure for Mineral Projects. Except for the purposes legislated under provincial securities law, any other uses of this report by any third party is at that party's sole risk. The responsibility for this disclosure remains with Telson. The user of this document should ensure that this is the most recent Technical Report for the property as it is not valid if a new Technical Report has been issued.

This report provides Mineral Resource and Mineral Reserve estimates and classification of resources in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum Standards on Mineral Resources and Reserves: Definitions and Guidelines, November 27, 2010 (CIM).

2.2 DETAILS OF INSPECTION

Scott E. Wilson conducted a property inspection of the Property during the week of November 7, 2016. While visiting the property, Mr. Wilson inspected the Tahuehueto Project, the core logging facilities, industrial scale bulk sample mining and milling, as well as the overall project layout.

2.3 TERMS OF REFERENCE

The terms of reference were to prepare resource and reserve estimates and a Technical Report as defined in Canadian Securities Administrators' National Instrument 43-101, Standards of Disclosure for Mineral Projects, and in compliance with Form 43-101F1 (Technical Report).

All currency amounts in this report are stated in US dollars (US$), as specified, with commodity prices in US dollars (US$). Quantities are generally stated in SI units, the Canadian and international practice.

2.3.1 UNITS OF MEASURE – ABBREVIATIONS

Units of Measure - Abbreviations			
Metric		Imperial	
Unit	Description	Unit	Description
%	Percent	%	Percent
°C	Degrees Celsius	°F	Degrees Farenheit
cm	Centimeter (Centimetre)	in	Inch
m	Meter (Metre)	ft	Foot (12 Inches)
g	Grams	oz	ounce
g/t	grams per tonne	g/t	grams per tonne
ha	Hectare (10,000 M_2)	ac	Acres
kg	Kilogram	lb	Pounds
km	Kilometer (Kilometre)	mi	Miles
KW or kW	Kilowatt	hp	Horsepower
mm	Millimeters (Millimetres)	in	Inches
opt	Ounces Per Ton	opt	Ounces Per Ton
ppm	Parts Per Million	ppm	Parts Per Million
SG	Specific Gravity	SG	Specific Gravity
µm	Microns	in	Inches
ft^3	Cubic Feet	m^3	Cubic Meters (Metres)
in^3	Cube Inches	cm^3	Cubic Centimeter (Centimetre)

2.3.2 ACRONYMS AND SYMBOLS

Acronyms and Symbols	
Term	**Description**
Ag	Silver
As	Arsenic
Au	Gold
Ba	Barium
Bi	Bismuth
Cd	Cadmium
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
Co	Cobalt
Company	Telson Resources Inc.
Cr	Chromium
CRD	Carbonate Replacement Deposit
Cu	Copper
EMT	Emergency Medical Technician
ICP	Inductively Coupled Plasma
ID5	Inverse Distance to the Fifth Power
K	Potassium
Ma	Million Years
MMC	Metal Mining Consultants Inc
Mn	Manganese
Mo	Molybdenum
NAD	North American Datum
Ni	Nickle
NSR	Net Smelter Return
Pb	Lead
POO	plan of operations
Project	Tahuehueto Project
QA	Quality Assurance
QA/QC	Quality Assurance/Quality Control
QC	Quality Control
QP(s)	Qualified Person(s)
RC or RVC	Reverse Circulation
Rdi	Resource Development Inc
RQD	Rock Quality Designation
Sr	Strontium
tpy	Tons per Year
V	Vanadium
W	Tungsten
Zn	Zinc
ZnEq	Zinc Equivalent Grade

3 RELIANCE ON OTHER EXPERTS

Metal Mining Consultants has reviewed and analyzed exploration data provided by Telson, its consultants and previous explorers of the area, and has drawn its own conclusions therefrom, augmented by its examination. MMC has relied upon data presented by the Company, and previous operators of the project, in formulating its opinion while exercising all reasonable diligence in checking, confirming and testing it.

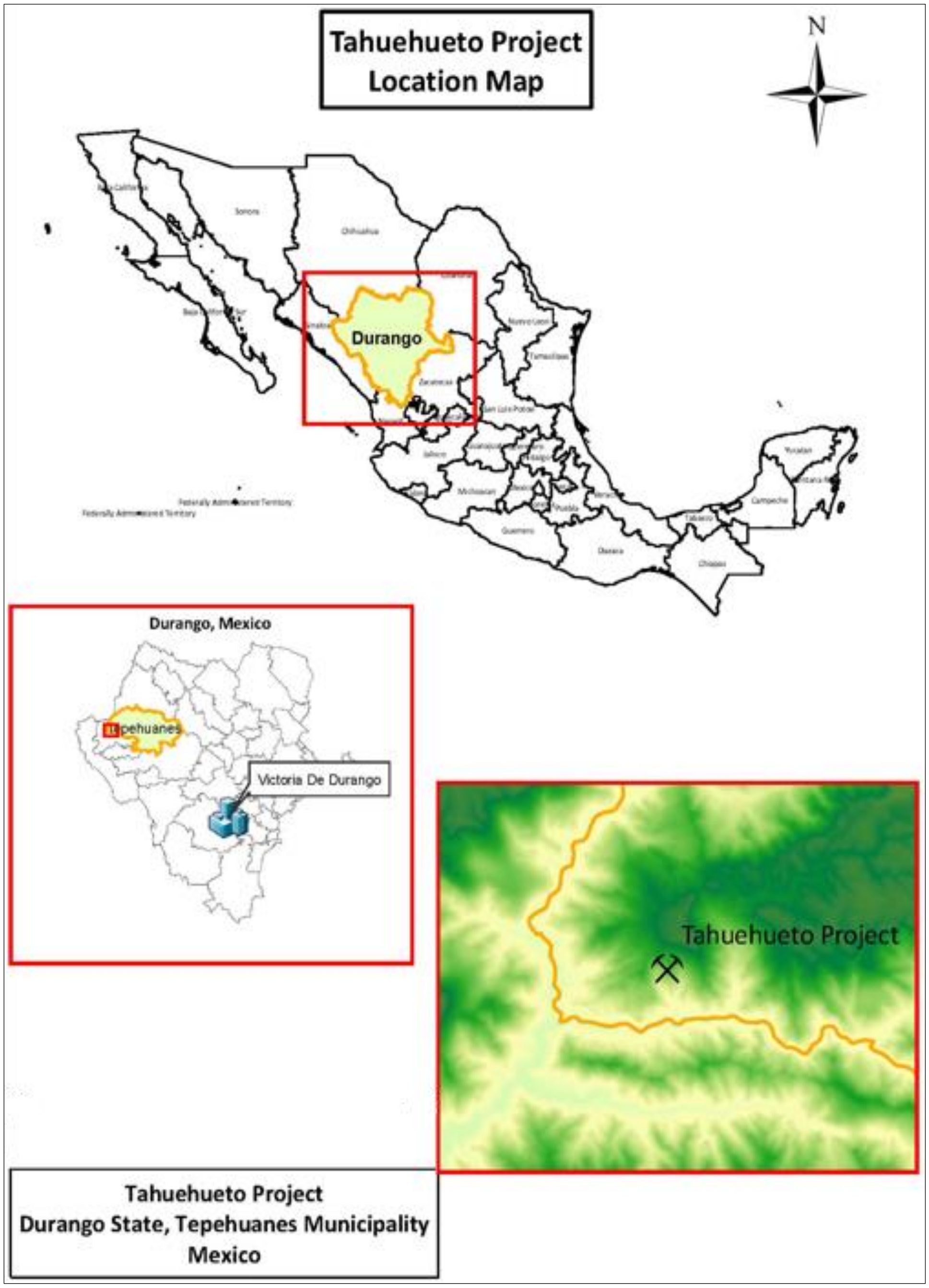
MMC relied on information provided by Telson, and the Company's legal counsel, as to the legal status of Telson and related companies, the title of the concessions and agreements comprising the Tahuehueto project, the terms of property agreements, and the existence of applicable royalty obligations, as well as all information concerning environmental issues and permitting. Section 4 in its entirety is based on information provided by Telson, and the authors offer no professional opinion regarding the provided information.

4 PROPERTY DESCRIPTION AND LOCATION

The Tahuehueto Project is located in the northwest portion of the state of Durango (Figure 4-1), about 250 km northwest of Durango, the state capital, and 160 km northeast of the city of Culiacan, Sinaloa. The Project is located about 25 km north of the Topia polymetallic-silver mine, 40 km northwest of the La Cienega gold, silver, base metal mine, 85 km southwest of the Guanacevi silver district, 280 km southeast of the Palmarejo silver and gold mine, and 150 km northwest of the San Dimas mining district, most notable for the Tayoltita silver and gold mine.

The project is approximately centered on UTM coordinates (WGS 84 Zone 13 for México) 337366 m E and 2812659 m N (106°37’1 longitude west and 25°25’19 latitude north).

Figure 4-1 Location of the Tahuehueto Project



4.1 LAND TENURE

The Tahuehueto property consists of 28 mining concessions that total 7,492.7889 ha. The concessions are shown in Figure 4-2 and listed in Table 4.1. The concessions are located in five non continuous blocks, shown in Figure 4-2. Some of them are subject to royalties of 1.6% of the NSR.

Figure 4-2 Tahuehueto Project Property Map

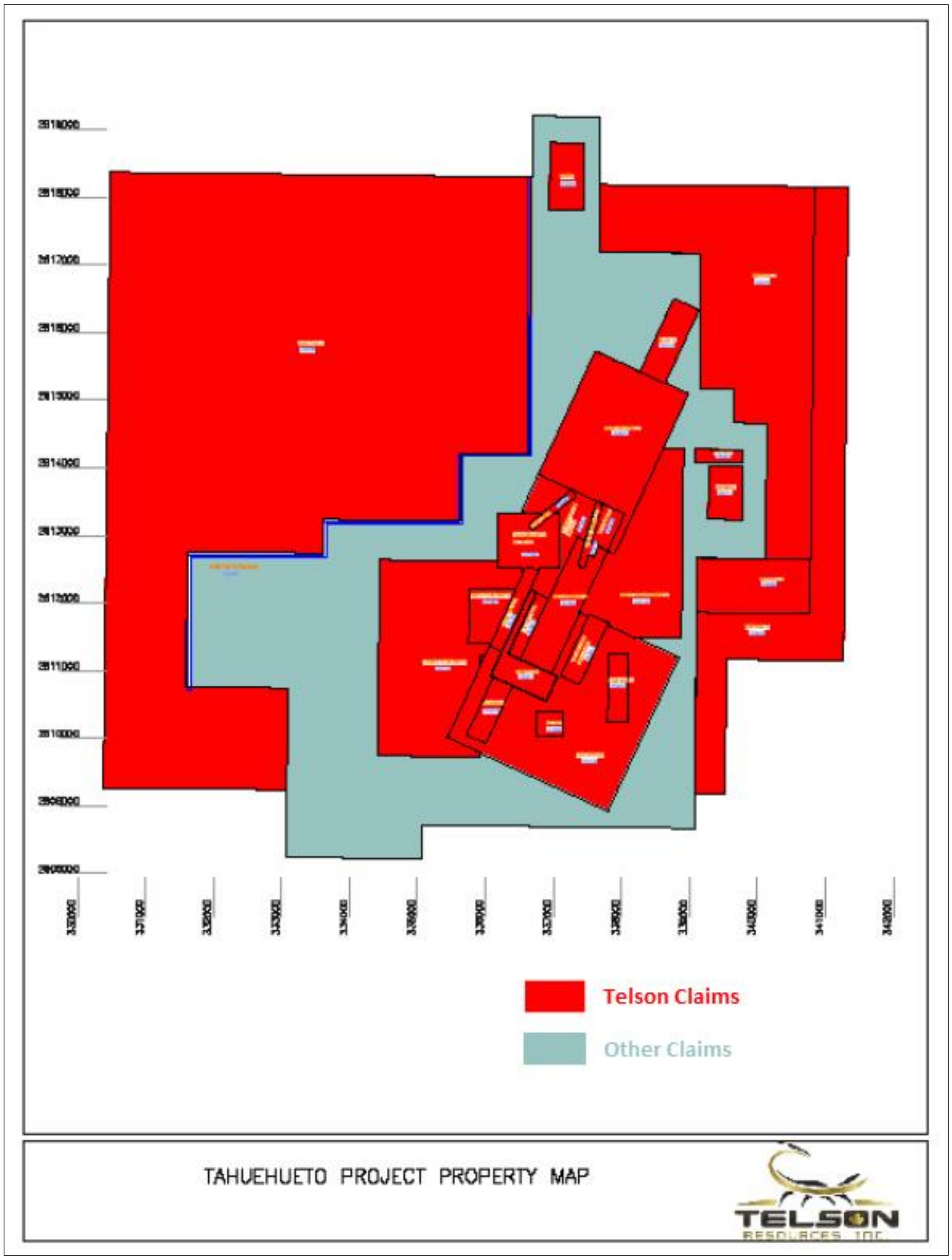


Table 4.1 Tahuehueto Mining Concessions

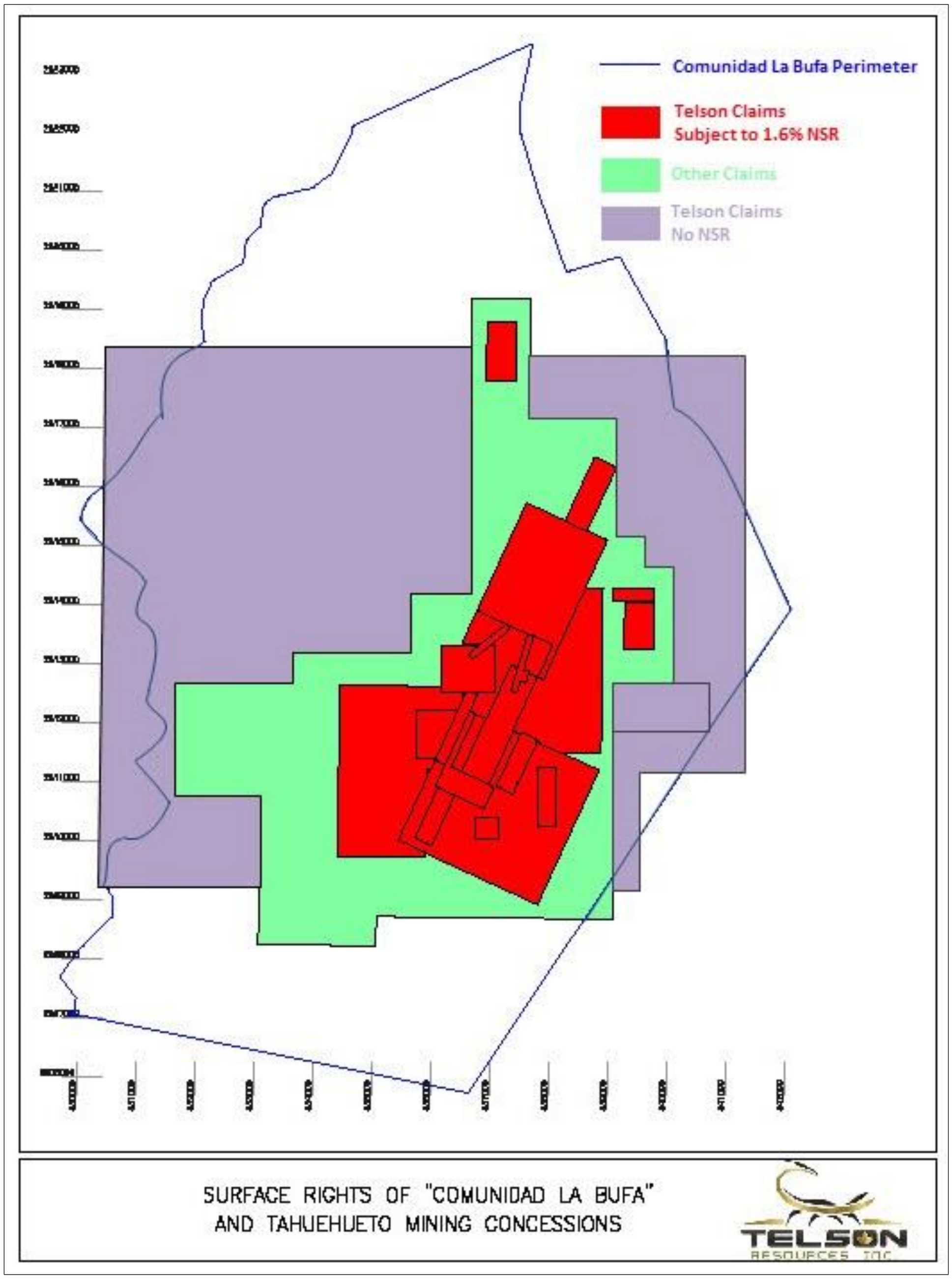
Tahuehueto Project-Mining Concessions						
Registered Owner	Mining Concession	Title	Granted	Expires	Hectares	Subject to Royalty 1.6%
Real	Dolores	153893	9/Jan/1971	8/Jan/2021	8.0000	No
Real	Colorado	160128	24/Jun/1974	23/Jun/2024	410.6622	No
Real	Tahuehueto El Alto	221990	27/Apr/2004	26/Apr/2054	68.5657	No
Real	Ampl Cinco De Mayo	221991	27/Apr/2004	26/Apr/2054	40.2384	No
Real	El Tres De Mayo	150452	26/Oct/1968	25/Oct/2018	30.0000	Yes
Real	Puerta De Oro Ii	151972	12/Nov/1969	11/Nov/2019	71.0475	Yes
Real	5 De Mayo	152274	20/Feb/1970	19/Feb/2020	25.8836	Yes
Real	Eugenia	152275	20/Feb/1970	19/Feb/2020	28.2288	Yes
Real	Guadalupe De Los Fresnos	152608	18/Mar/1970	17/Mar/2020	20.0000	Yes
Real	Sacramento	152634	18/Mar/1970	17/Mar/2020	94.3443	Yes
Real	Maria	152636	18/Mar/1970	17/Mar/2020	50.0000	Yes
Real	Sacramento	152716	18/Mar/1970	17/Mar/2020	12.0000	Yes
Real	Libertad	153872	9/Jan/1971	8/Jan/2021	46.0000	Yes
Real	La Gloria	153975	18/Jan/1971	17/Jan/2021	20.0000	Yes
Real	Montecristo	154675	12/May/1971	11/May/2021	305.9668	Yes
Real	La Reyna Del Oro	155213	10/Aug/1971	9/Aug/2021	30.0000	Yes
Real	Estela	156835	28/Apr/1972	27/Apr/2022	14.0000	Yes
Real	Yolanda	158064	17/Jan/1973	16/Jan/2023	18.6311	Yes
Real	Imperio	158112	19/Jan/1973	18/Jan/2023	40.0000	Yes
Real	Eloy Ii	160706	15/Oct/1974	14/Oct/2024	47.6740	Yes
Real	El 201	221992	27/Apr/2004	26/Apr/2054	14.4114	Yes
Real	Ampl Sacramento	222123	21/May/2004	20/May/2054	254.6345	Yes
Real	Ii Ampl 5 De Mayo	222124	21/May/2004	20/May/2054	411.8868	Yes
Real	El Espinal 3	228156	6/Oct/2006	5/Oct/2056	836.8595	No
Real	Vueltas 4	229396	17/Apr/2007	16/Apr/2057	3863.8992	No
Real	Vueltas 5 (Terrain Correction)	229397	17/Apr/2007	16/Apr/2057	53.7438	No
Real	El Espinal 5	229398	17/Apr/2007	16/Apr/2057	132.3710	No
Real	El Espinal 4	229438	19/Apr/2007	16/Apr/2057	543.7403	No
Total Hectares					7,492.7889	

All 28 mining concessions are owned by Real de la Bufa S.A. de C.V. ("Real"), a Mexican subsidiary of Telson Resources, and at present, all taxes owing to the Bureau of Mines of México are paid.

On April 26th, 2016 Real de la Bufa renewed and extended a temporary land use agreement (the "Agreement") with the Comunidad La Bufa (hereinafter "Comunidad"), holders of certain surface rights at Tahuehueto. The renewed Agreement allows the Company to explore, develop and produce minerals within an area of 2,062 ha over a period of 30 years beginning on the date of the Agreement and may be extended upon request by Real. This 30-year Agreement is obligatory to Comunidad, not for Real. Real will pay to Comunidad a fee of US $46,540.00 or the equivalent amount in Mexican Pesos for each 365-day period as compensation for the temporary land use for mining exploration and exploitation. Payments will be due yearly on the anniversary date of the Agreement and will be subject to annual increase of 5% on the value of the preceding year's payment.

The Surface Rights Agreement allows Real unrestricted access to explore, develop and mine metals within the area covered under the agreement and other areas are open to negotiate any agreement for further exploration as shown in Figure 4-3. The Agreement, applies to the internal 23 concessions. The 23 internal concessions are those subject to the 1.6% NSR as listed in Table 4.1.

Figure 4-3 Surface Rights of the Comunidad La Bufa and the Tahuehueto Mining Claims



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

5.1 ACCESS

Access to the Project by land is by paved Mexican Highway 45 from the city of Durango 53 km to the turnoff to Santiago Papasquiaro, then west on paved Mexican Highway 23 for 122 km through Santiago Papasquiaro and on to Tepehuanes. From Tepehuanes, an unnumbered paved road runs west through El Tarahumar.

The pavement ends approximately 65 km after Tepehuanes, and access is then by 120 km of unimproved dirt road to the project. The approximately 175 km trip by road from Tepehuanes to Tahuehueto takes about 5 to 6 hours of driving.

There is also access via fixed-wing aircraft from either Culiacan or Durango. A serviceable gravel airstrip is located 20 km by road north of Tahuehueto at El Purgatorio. This airstrip is maintained by Telson and is suitable for single-engine aircraft.

Narrow gravel roads in steep terrain provide access to various locations within the project limits.

5.2 CLIMATE

The climate of the region is moderate. Available climate data shows a warm-hot season from June through October, with 55 to 113 cm of precipitation possible; it is relatively dry from February to May. Annual precipitation ranges between 80 to 140 cm. Freezing temperatures were not recorded in the region between 1961 and 1990, although occasional snow has been reported (CONSEJO DE RECURSOS MINERALES, 1983a). Soho estimates that winter temperatures range from 5˚ to 24˚C, with summer temperatures in the range of 25˚ to 42˚C (Knight Piésold, 2005).

5.3 LOCAL RESOURCES AND INFRASTUCTURE

The nearest sizeable community to the project area is Tepehuanes, which is located approximately 175 km by road east of the property and has a population of approximately 15,000. A 34.5 kv power line and telephone service extends as far as Tepehuanes; diesel generators presently supply power to the project site.

Topia, also serviced with a power grid is located about 25 km southeast of the Project by air, is the nearest community of any size, with a population of about 1,200.

Telson first obtained water for the project from an underground adit above the camp; water is available from levels 16 and 14 of the El Creston underground workings, from all four levels of the El Rey underground workings and also from an artesian well recovered from one of Telson exploration drill holes. Since 2007, water has been pumped up to the project site from the Rio las Vueltas, which flows some 800 m year-round below the camp.

5.4 PHYSIOGRAPHY

The Tahuehueto project is on the western side of the Sierra Madre Occidental, a mountain range that forms the central spine of northern Mexico and is largely composed of Tertiary volcanic rocks. Tahuehueto is in a sub-province called Barrancas, which means ravines in Spanish and accurately describes the project area.

The terrain at the Project is very steep to precipitous in places. Elevations range from about 600 m in river valleys in the southern part of the property to over 2,500 m on high-level plateaus in the northern part of the property. El Creston, the most important of the mineralized zones identified to date, is located along a northeast-trending ridge that spans an elevation range of 1,400 to 1,800 m. Aerial views of the areas are shown in Figures 5-1 – 5-3.

In the treeless barrancas, scrub alpine bushes and cacti with minor underbrush make up the vegetation. Thicker underbrush, similar to willow, occurs in creek bottoms, while Ponderosa pine trees grow on the high plateaus.

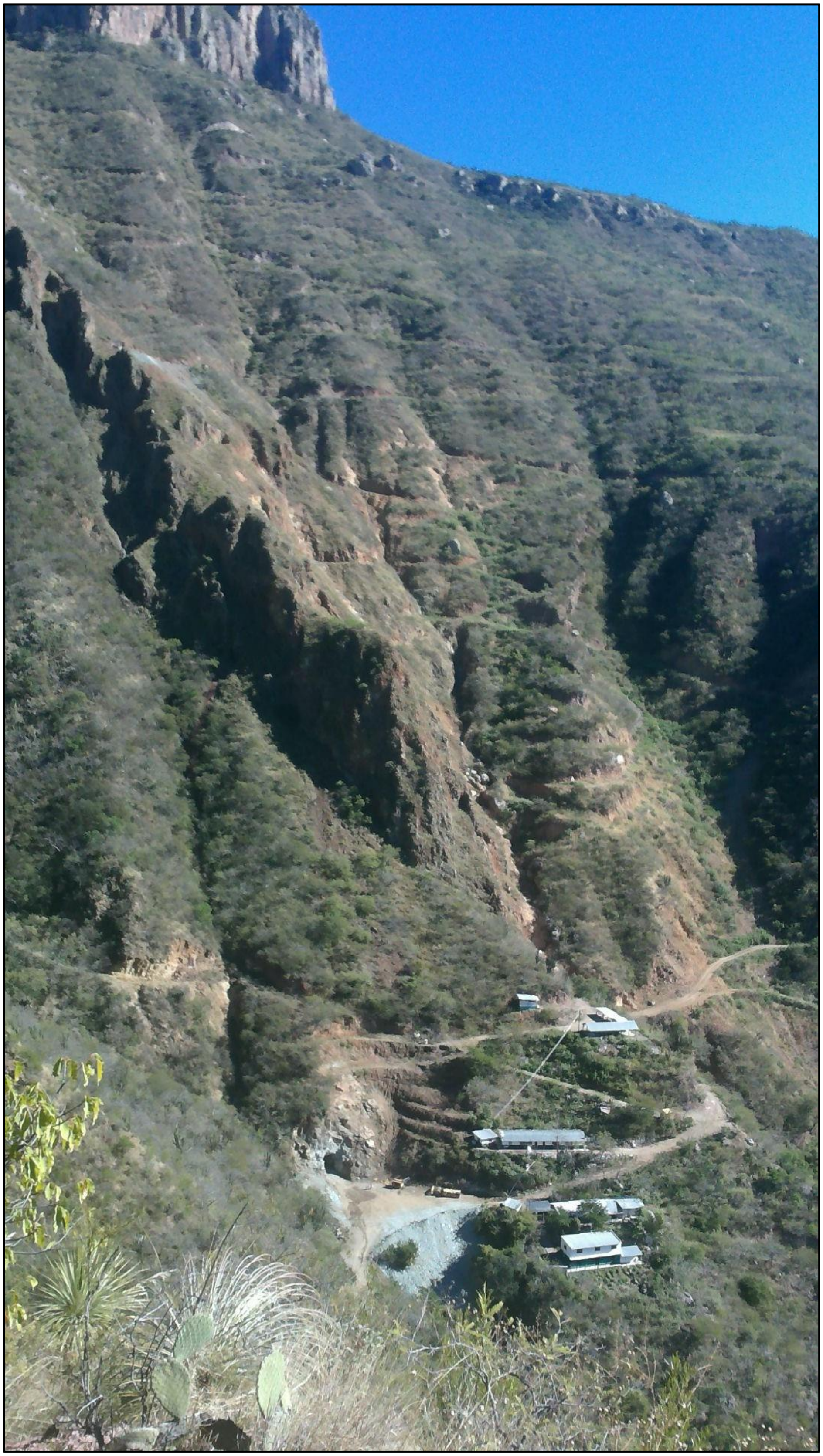
The region is drained by the Rio las Vueltas, which flows continuously all year from east to west and is located south of the camp at an elevation of 600 to 625 m. There is one major drainage basin in the Tahuehueto project area that feeds the Rio las Vueltas. Most streams in the area are seasonal.

The Project is in a relatively quiet seismic area that has seen no major earthquakes within about 400 km, based on the National Geophysical Center/NOAA’s Significant Earthquake Database that contains information on destructive earthquakes from 2150 BC to the present.

Figure 5-1 Aerial View of the Tahuehueto Project

Figure 5-2 View of the El Creston/El Perdido Area

Figure 5-3 View of the El Creston/El Perdido Area



6 HISTORY

Historic exploration has been focused along a series of exposed veins, silicified zones and color anomalies that are common within the Tahuehueto project area. The information in this subsection is taken from Brown (1998b, 2004) unless otherwise referenced. Gold and silver vein mineralization was discovered in the Tahuehueto area in the nineteenth century by Spanish explorers. The veins were examined and found to contain good gold and silver values hosted in sulfides, which at that time could not be processed.

The first recorded exploration was in 1904 (Cavey, 1994) when an English company began development on the Veta 20-93 (El Creston) at the Sacramento de la Plata mine. The actual starting date of the limited production is not recorded.

Companía Minera Sacramento de la Plata, a predecessor company of Real de la Bufa, was founded in 1966 and developed over 700 m of underground workings on the El Rey and El Creston structures in 1971. A 50 tpd plant was constructed to process the mined material and was operated in the 1970s. Concentrates from the mill were flown to Santiago Papasquiaro and then driven to the smelter at Torréon. Total production from Tahuehueto appears to have been limited. ASARCO sampled El Creston and other veins in the region. Tadmex, S.A. de C.V. developed Level 16 of the El Creston vein (Pedroza Cano, 1991). MMC has no detailed information on these programs.

A company called Emijamex, S.A. de C.V. (“Emijamex”) conducted geochemical and rock sampling, detailed geological investigations, drifting, and crosscutting at the Sacramento de la Plata mine from 1975 through 1977 (Kamono, 1978), including Levels 11, 12, 13 and 14 (Pedroza Cano, 1991). They also submitted an auriferous lead/zinc sample for metallurgical study that is described in Section 16 (Rios et al., 1977a, 1977b).

The Consejo de Recursos Minerales, a Mexican government geological organization that is currently called the Servicio Geológico Mexicano or “SGM”, drilled 28 surface and underground holes that tested the El Creston and Cinco de Mayo structures in the early 1980s. This appears to have been the only drilling done on the project prior to that of Soho, but no data from the Consejo de Recursos Minerales program are available. The Consejo de Recursos Minerales also conducted an induced polarization (“IP”) study over an area of about 3 by 0.4 km that included the El Creston, Cinco de Mayo, and Texcalama zones. The lines were 300m long and spaced 50m apart, and measurements were recorded every 20m at El Creston and 25m at Texcalama and Cinco de Mayo. The IP study identified anomalies that correspond to the continuation of the known structures (Consejo de Recursos Minerales, 1983b).

Castle Minerals Inc. of Vancouver (“Castle”; subsequently changed to Castle Rock Exploration Corp.) optioned Tahuehueto from Sacramento in 1994. At that point, the property consisted of 17 concessions totaling 1,261 ha. Cavey (1994) stated that Castle’s intention was to undertake a surface and underground exploration program to verify historic reserve estimates and to evaluate the potential for the existence of a much larger, lower grade open pit minable deposit.

Prior to the time that Castle acquired the property there were at least 15 documented mineralized zones. Many of these have since been determined to be parts of larger structures. The El Creston vein structure had been exposed in 10 horizontal levels, with 2.000m of total development in adits, drifts and crosscuts (Cavey, 1997) over a vertical distance of approximately 490m. The Cinco de Mayo vein system had been explored by 3 adits, one of which was inaccessible in 1997, if not so in 1994. The Texcalama structure had been explored by at least five separate single level adits along as much as 300m of exposure.

The report by Cavey (1997) presented a geological appraisal of the Tahuehueto project, describes the work completed by OREQUEST Consultants in 1994 for Castle, and made recommendations for further work.

Castle collected 459 rock samples, including 247 from the El Creston structure, 21 from the Cinco de Mayo structure, and 191 from other sites on the property. The samples included both surface chip samples and underground chip-channel samples.

At El Creston seven of the ten crosscuts examined were mapped and sampled in 1994 (Cavey, 1997) to determine the width of the vein and the general dimensions of the stockwork beyond the walls of the vein. Historic mining of the El Creston structure was over 3.0-6.0m widths, primarily within the zone of the most obvious sulfide mineralization, but the breccia/stockwork zone is locally up to 50m wide (Cavey, 1997). The best exposure of the mineralized vein system in 1994 was from level 11. The entire 39m of the vein exposure sampled averaged 5.50 g/t Au, 34.03 g/t Ag, 2.3% Zn and 0.92% Cu over an average width of 1.19m. Several of the samples were encouraging as they contained vein material as well as footwall and hanging wall material; these samples averaged 1.30 g/t Au, 5.0 g/t Ag, 0.7% Zn and 0.33% Pb over a true width of 13m.

Brown (2004) stated that the sampling by Castle in 1994 appeared to indicate that the El Creston vein is not continuous in the eight levels sampled as would be expected in a vein structure containing massive sulfides. Cavey (1997) did mention the presence of post-mineral faults within the El Creston structure creating up to 20m of offsets along the mineralization.

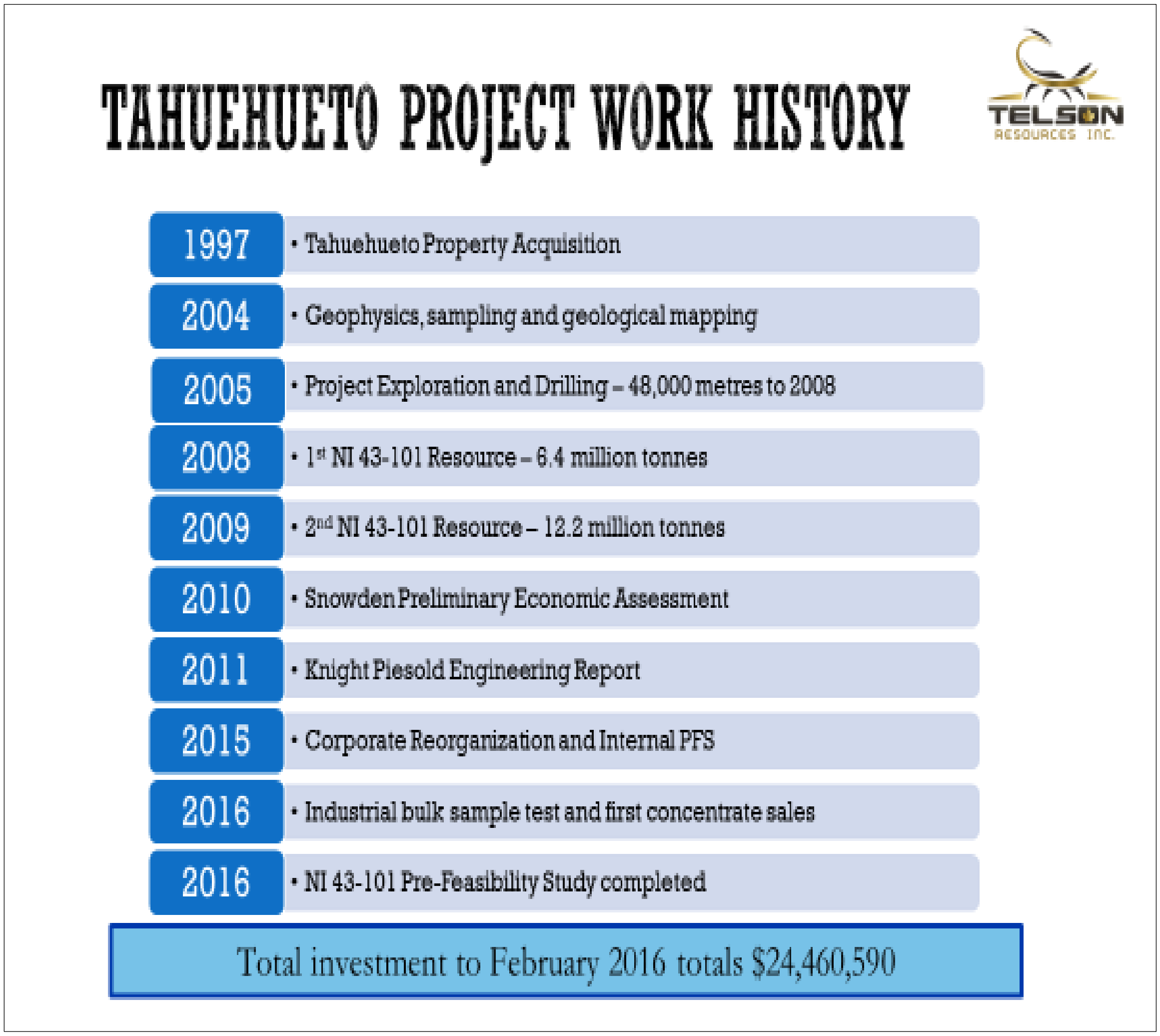
Sampling on the Cinco de Mayo structure by Castle in 1994 showed an average grade of 4.91g Au/t over an average width of 1.5m along 138m of the vein exposed in the underground workings; samples were taken approximately every 15-20 m (Cavey, 1997). One footwall sample returned a value of 9.73 g/t Au over 5.0m, and hanging wall samples returned gold values of 6.96 g/t over 1.5m, 0.74 g/t over 1.1m, 2.06 g/t over 5.0m, 0.56 g/t over 5.0m and 2.86 g/t over 5.0m. Several of the vein samples combined with either hanging wall or footwall samples resulted in nearly continuous chip samples that produced values of 4.78g Au/t over 11.3m, 1.63g Au/t over 6.3m, and 1.57g Au/t over 12.0m.

Cavey (1997) concluded that "*the 1994 sampling was unable to reproduce the grades obtained by others in the resource calculations done on the El Creston structure" but that "the 1994 Castle sampling confirmed the grades previously obtained in the Cinco de Mayo area.*"

Castle dropped their option within two years without having drilled any holes. Brown (1998b) reports that 5,900m of underground development and exploration workings at El Creston, Cinco de Mayo, and El Rey had been completed by previous operators, among whom he identified Asarco, Peñoles, Consejo de Recursos Minerales, and DOWA Mining Company.

A summary of the work completed on the Tahuehueto project since Telson's acquisition of the property is provided in Figure 6-1.

Figure 6-1 Telson Resources-Tahuehueto Work History



7 GEOLOGICAL SETTING

7.1 REGIONAL GEOLOGY

The Tahuehueto project lies near the western edge of the Sierra Madre Occidental, a 1,200 km long north-northwest-trending volcanic plateau that is 200 to 300 km in width. This mountainous plateau separates the southward extension of the Basin and Range Province of the southwestern United States into two parts; Sedlock et al. (1993) suggested calling these two areas of extension the eastern and western Mexican Basin and Range provinces. Tahuehueto is near the boundary between the Sierra Madre Occidental and western Mexican Basin and Range Province.

Basement rocks in the Sierra Madre Occidental are obscured by Cenozoic volcanic flows, tuffs, and related intrusions, but are inferred to include Proterozoic basement rocks, overlying Paleozoic shelf and eugeosynclinal sedimentary rocks, possibly scattered Triassic-Jurassic clastic rocks, and Mesozoic intrusions (Sedlock et al., 1993; Salas, 1991). These basement rocks are not exposed in the project area (Figure 7-1).

Cenozoic magmatic rocks in northern Mexico, including the Sierra Madre Occidental, are generally thought to reflect subduction-related continental arc magmatism that slowly migrated eastward during the early Tertiary and then retreated westward more quickly, reaching the western margin of the continent by the end of the Oligocene (Sedlock et al., 1993).

The eastward migration is represented in the Sierra Madre Occidental by the Late Cretaceous- Paleocene “lower volcanic series”, or Nacozari Group, of calc-alkaline composition. Over 2,000m of predominantly andesitic volcanic rocks, with some inter-layered ash flows and associated intrusions, comprise the lower volcanic series.

There was a period of approximately 10 million years between eruption of the lower volcanic series and the onset of the next phase of felsic volcanism, referred to as the upper volcanic series. A number of stocks intrude andesites of the lower volcanic series. These stocks are generally of granodiorite composition and are believed to be a late phase of the Sinaloa batholith (Henry et al., 2003). At Topia, it is during the hiatus in volcanism that the lower series was faulted, tilted, deeply dissected, and then intruded by the granodiorite stocks, and a northeast-trending set of faults was mineralized as Ag-Zn-Pb-Au-Cu rich fissure veins (Loucks et al, 1988).

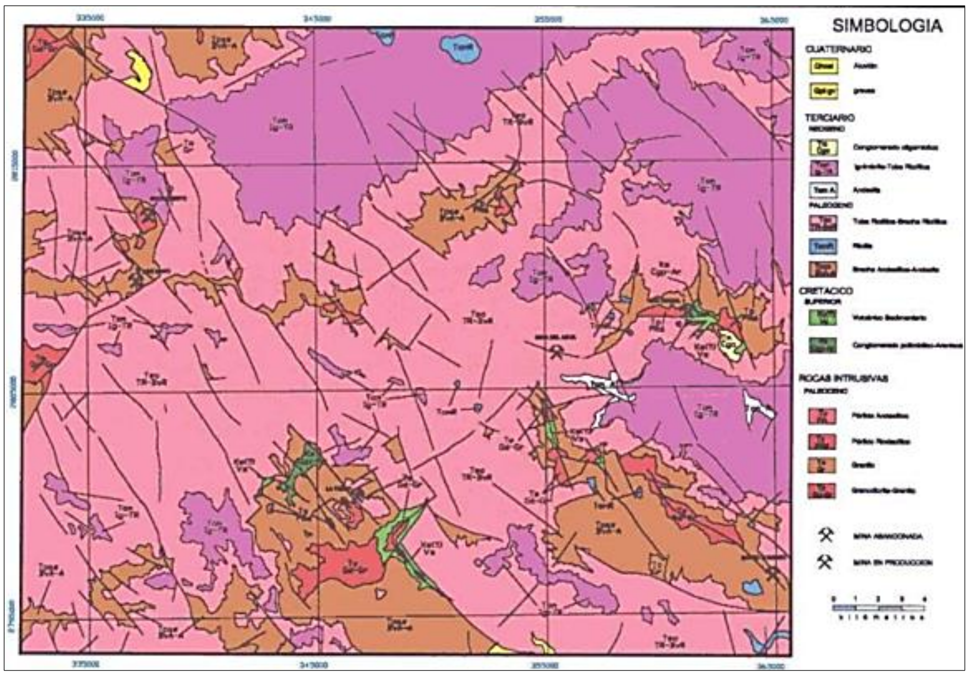
A similar scenario is envisioned at Tahuehueto. K-Ar dating at Topia of igneous rocks and mineralization yield an age of 46.1 Ma for one of the granodiorite stocks, ages between 43.5 Ma and 44.0 Ma for the hydrothermal system, and 37.9 Ma for the lowermost rhyolite welded tuff of the upper volcanic series (Loucks et al, 1988).

Rhyolitic ignimbrites and flows, with subordinate andesite, dacite, and basalt, formed during Eocene and Oligocene caldera eruptions. These volcanic rocks form a 1 km-thick unit that unconformably overlies the lower volcanic series andesitic rocks and constitutes the “upper volcanic super group” of the Sierra Madre Occidental (Sedlock et al., 1993), also commonly referred to as the upper volcanic series or Yecora Group.

The upper volcanic series ignimbrites are moderately west dipping in the Tahuehueto region. Loucks et al. (1988) report that the ignimbrites are warped into a broad north-south anticline. Tahuehueto lies in the western limb of this large regional structure. As the magmatic arc retreated to the western edge of the continent, becoming inactive by the end of middle Miocene time, late Oligocene to Miocene (24 to 17 Ma) basaltic andesites were erupted in a back-arc basin in the Sierra Madre Occidental. These basaltic andesites may have been deposited in a sub-aqueous environment.

Still younger alkalic basalts related to Basin and Range extension are found in and east of the range; these youngest basalts are present just north of the city of Durango. Although there appears to have been little late Cenozoic extension in the Sierra Madre Occidental itself, extensional Basin and Range-type structures and ranges formed to the east and west.

Figure 7-1 Regional Geological Setting of Tahuehueto Project



7.2 PROPERTY GEOLOGY

The property contains four main rock types: lower volcanic series andesite, granodiorite stocks, polymictic conglomerate, and felsic ash-flow tuffs of the upper volcanic series. The majority of the project area is underlain by andesite flows, tuffs, and volcaniclastic rocks of the lower volcanic series. A geologic map and stratigraphic column and geologic map of the Tahuehueto area are shown in Figure 7-2 and Figure 7-3, respectively.

The lower volcanic series remains generally undifferentiated. A volcaniclastic unit distinct from the andesite flows exists in the Texcalama and Cinco de Mayo areas and an andesite lithic lapilli tuff exists in the footwall of the El Creston structural zone. Granodioritic stocks intrude the andesites and are exposed at surface in the footwall of the El Creston structural zone and the El Rey mine area.

The andesites and granodiorite are overlain by a basal polymictic conglomerate unit that is tens of meters thick and marks the unconformity between the lower and upper volcanic series. Amygdaloidal basalt flows occur locally within the conglomerate unit. In some areas, thin units of ignimbrite were deposited before the conglomerate. Late Tertiary or Quaternary landslides obscure outcrop patterns in the El Creston-El Perdido area and are likely to be present in other areas of steep topography within the project area.

A series of northeast-striking veins that formed within a series of normal faults with subordinate left-lateral displacement hosts the Mineral Resources described in Section 14. The principal, through going veins have a general strike of 045° to 060° and dip between 65° and 80° to the southeast. This vein set includes Cinco de Mayo, El Catorce, and El Perdido and extends northeastward to Santiago. Other veins with the same orientation include El Rey, Dolores, Tahuehueto, Texcalama, El Espinal, and Tres de Mayo. Within the core area of the Mineral Resources, the El Creston series of veins, striking about 035° and dipping 60° to 80° east, formed in a strongly dilatant zone between the through-going El Perdido and El Rey structures.

Figure 7-2 Geology of the Tahuehueto Project Area

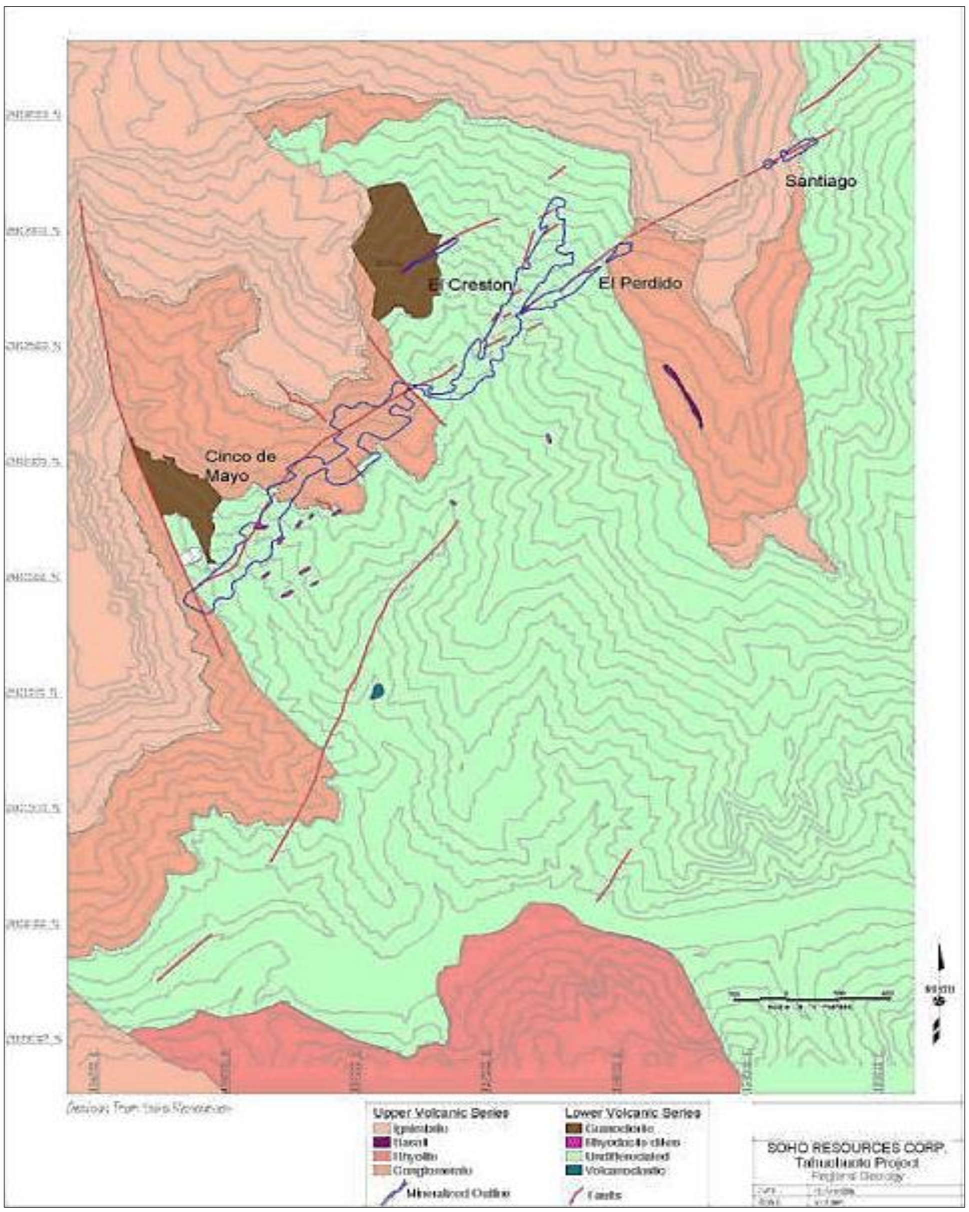
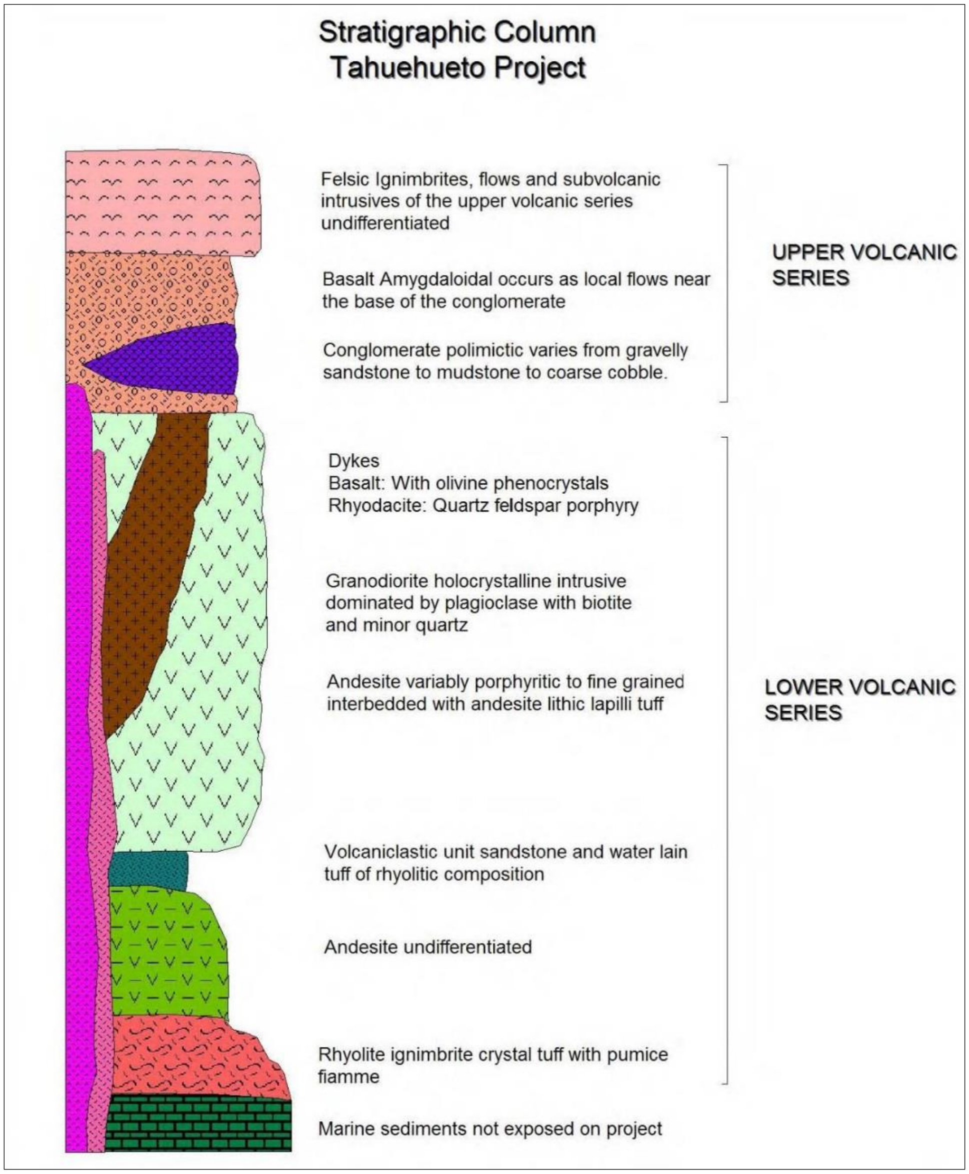


Figure 7-3 Stratigraphic Column of the Tahuehueto Project



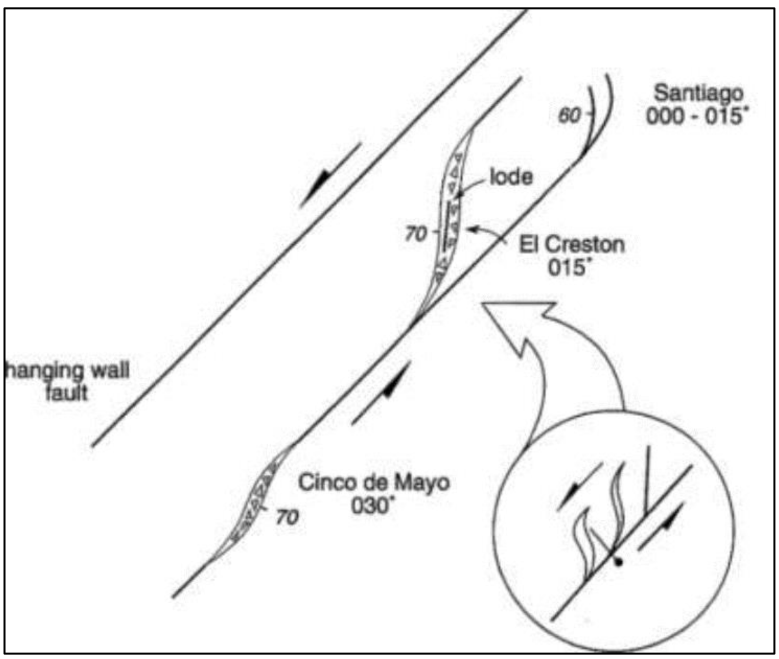
8 DEPOSIT TYPE

8.1 GEOLOGICAL MODEL

Mineralization at Tahuehueto is classified as intrusion related epithermal low sulfidation polymetallic Ag-Au style (Corbett, 2007), with Au and Ag accompanied by Cu, Pb, and Zn mineralization. These types of deposits are interpreted to have been derived from porphyry intrusion source rocks at depth.

A northeast-striking corridor of steep east dipping fractures and normal faults, traced for about 3 km from Cinco de Mayo in the south to Santiago in the north, represents the main control to mineralization at Tahuehueto. North-northeast trending subsidiary structures, such as at El Creston, are less continuous and commonly display more open vein textures typical of a dilational setting. Figure 8-1 schematically represents the interpreted structural elements present at Tahuehueto that localize mineralization at Cinco de Mayo, El Creston and at Santiago. In many vein systems much of the mineralization is confined to structural shoots that are commonly developed within dilational structures (Corbett, 2007).

Figure 8-1 Structural Elements Present at Tahuehueto



Mineralization at Tahuehueto is strongly telescoped, with early high temperature mineralization and alteration overprinted by intermediate temperature and then by younger epithermal mineralization and alteration assemblages. The multiple mineralizing events obscure vertical zonation patterns that are commonly found in other epithermal vein deposits.

Mineralized zones are characterized by pervasive silicification, quartz-filled expansion breccias, and sheeted veins. Multiple phases of mineralization produced several phases of silica, ranging from chalcedony to comb quartz (Corbett, 2007). The surface expression of known mineralization occurs over a vertical distance of at least 850m between Cinco de Mayo and Santiago. The El Creston mineralized zone has been developed by 10 levels over 490m vertical distance.

Cinco de Mayo occurs at the deepest crustal level, where alteration and breccias (below) are indicative of buried porphyry. The Santiago area occurs at the highest elevation where crystalline and chalcedonic quartz veins are consistent with the pronounced overprinting relationships recognized elsewhere on the property, and hypogene hematite in the chalcedony vein is indicative of lower temperature epithermal mineralization (Corbett, 2007).

8.2 MINERALIZATION

Mineralization at Tahuehueto occurs as polymetallic epithermal veins with multiple mineralizing events overprinted on one another in the same vein structure. The primary host rock is andesite of the lower volcanic series, but in at least one case, the hydrothermal system penetrated felsic ignimbrite of the upper volcanic series. Styles of mineralization identified by Corbett (2007) include:

- Initial pervasive propylitic-potassic alteration with local specular hematite develops as intrusion-related alteration.
- Early chalcopyrite-pyrite mineralization, locally with quartz-barite typically forms early and at deeper crustal levels in polymetallic Ag-Au vein systems.
- Polymetallic Ag-Au mineralization comprising pyrite-galena-sphalerite ± chalcopyrite ± chalcopyrite ±Ag sulfosalts ± barite represents the volumetrically most apparent mineralization and displays pronounced vertical variation discerned as changes in the sphalerite color from dark brown Fe-rich high temperature sphalerite formed early and at depth to red, yellow and less commonly white sphalerite as the Fe-poor low temperature end member that typically develops at higher crustal levels and as a later stage. Much of this mineralization occurs as sulfide lodes or as breccia infill. Bulk lower grade mineralization occurs as fine grained Au and Ag sulfosalts deposited within base metal sulfides as part of the main polymetallic mineralization, rising to higher grade Ag with increased base metal contents. These events evolve to mineralization with a more epithermal character and locally higher Au-Ag grades at later stages where base metal sulfides are overprinted by Ag-rich tetrahedrite (freibergite).
- Highest Ag-Au grades locally occur in the absence of Cu-Pb-Zn in ores described as the epithermal end member of polymetallic Ag-Au mineralization which is strongly structurally controlled. High grade Ag may occur as freibergite with celadonite in combination with white sphalerite and dark chlorite commonly with later stage opalchalcedony. Semi-massive to banded chlorite locally occurs with celadonite-pyrite-opal and displays elevated Au with significantly lower Ag: Au ratios. Hypogene hematite occurs with banded quartz as an epithermal assemblage which accounts for elevated Au grades overprinting earlier sulfide-rich mineralization.

Overprinting of the lower-temperature, higher-level mineral assemblage onto the higher temperature, deeper-level mineral assemblage is referred to as telescoping. This telescoping may represent the progressive cooling of the hydrothermal system, although in some instances tectonic un-roofing of the cover rocks may also result in a decrease in overburden and progressive deposition of higher crustal level, lower temperature mineral assemblages. Increasing gold and silver grades in the later higher crustal level assemblages without significant base metals is an important element of this telescoping (Corbett, 2007).

Breccias are an integral part of the Tahuehueto hydrothermal system and display several genetic styles. Corbett (2007) notes that many of the sulfide-mineralized zones display sulfide transport textures; typical of fluidized breccias. Milled breccias are those in which the clasts have undergone significant working while being transported from deeper to elevated crustal settings. These breccias typically contain rounded clasts in a matrix of milled rock flour which has undergone hydrothermal alteration. Expansion breccias, in which the fragments have been moved apart and filled in with carbonate or quartz in a jigsaw pattern, are typical in dilational structural settings. Magmatic hydrothermal breccias typically occur in near porphyry environments and contain clasts of porphyry intrusions and alteration in a milled matrix. Shingle breccias with elongate, parallel shingle-like fragments, are thought to have been formed by collapse following the explosive escape of volatiles from an underlying magma chamber. Figures 8-2 through 8-6 present examples of these breccias.

The uppermost portions of the mineralized structures are oxidized. In the oxide zone, mineralization consists of malachite, azurite, chalcocite, covellite, limonite, and hematite. Malachite overprints tetrahedrite, and chalcocite and covellite form coatings on sphalerite. The depth of the oxide-sulfide interface varies considerably, but is generally less than 100m.

Sulfide mineralization lies below the oxidized zone and consists of sphalerite, galena, chalcopyrite, tennantite, tetrahedrite, and probably electrum. Gangue minerals are quartz, pyrite, chlorite, sericite, and calcite. Locally a light green phyllosilicate mineral interpreted to be celadonite (Loucks, et al 1988) forms as gangue and is closely associated with high-grade gold and silver mineralization.

Corbett (2007) observed supergene enrichment in both mine workings and in drill hole DDH07- 081 from the upper part of the El Creston zone. The oxide-sulfide interface occurs at about 37m in depth in that hole. Corbett (2007) states that silver and zinc were leached from the oxide zone, with silver and copper being enriched below the base of oxidation. Silver increases from 39.1g Ag/t between 34.95 to 37m to 270g Ag/t at 37 to 40.05m in the hole. Gold is concentrated at the base of the zone of oxidation.

Figure 8-2 Fluidized Breccia

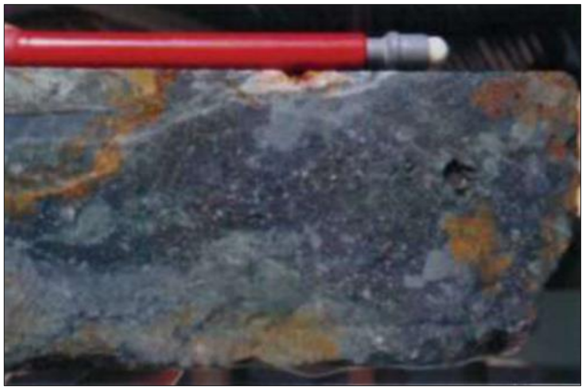


Figure 8-3 Milled Breccia

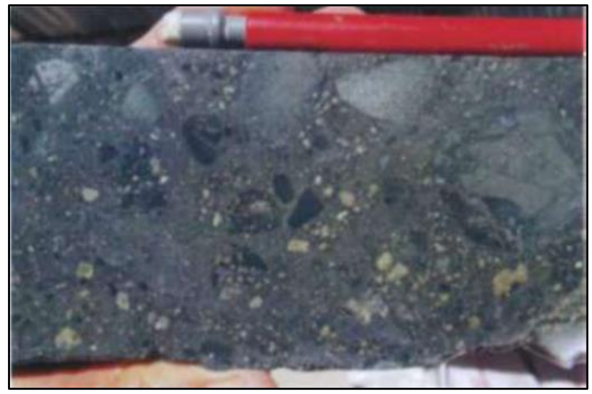


Figure 8-4 Carbonate Filled Expansion Breccia

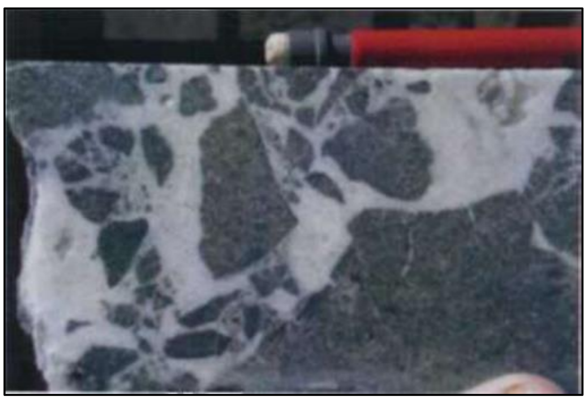


Figure 8-5 Magmatic Hydrothermal Breccia

Figure 8-6 Shingle Breccia

9 EXPLORATION

Telson entered into a "Promise to Contract" agreement in 1996, after Castle dropped their interest in Tahuehueto, whereby the owners of a majority of the outstanding shares of Sacramento agreed to enter into a Share Purchase Agreement. This agreement was executed in March 1997.

Telson conducted both surface and underground sampling in 1997 to verify historic mineral inventory estimates and to evaluate the potential for a much larger, lower grade open pit deposit (Brown, 1998b, 2004). The following summary of Soho's 1997 work is from Brown (2004):

The initial part of the work program consisted of both detailed rock channel sampling at the El Creston, along with camp construction. The second half of the work program was devoted to the continued channel sampling of the El Creston underground workings, and the preliminary geological mapping of the El Creston workings. Approximately 1,200 underground and surface channel samples were taken from the El Creston zone, with a few samples taken at Dolores, Cinco de Mayo and Los Burros. Channel samples taken in cross cuts were generally a 1.5 m width, while channel samples from drifts along the structures were from a 1.0-1.5 m width depending on the width of the drift. Along drifts, channel samples were taken at 2.5 m centers.

Telson geologists created a relational database to store and manipulate all of the sample location, description and analytical data. All the previous surveyed underground workings were digitized, and all the sample data has been plotted, level by level at El Creston, on sample number, gold, silver, copper, lead, and zinc maps. Telson geologists mapped the underground workings at El Creston, but Telson either in a reconnaissance or property scale manner did no geological mapping, this will have to be addressed in the following exploration programs.

Telson resumed exploration at Tahuehueto in the spring of 2004 (Soho, 2004). Initial focus was to prove continuity between the two existing highly mineralized El Creston and Cinco de Mayo zones and thereby demonstrate the potential for a large scale gold deposit (Soho, 2004b).

Subsequent exploration, as detailed below, has included continued exploration along the mineralized corridor between Cinco de Mayo and Santiago, while testing some of the additional mineralized zones on the property.

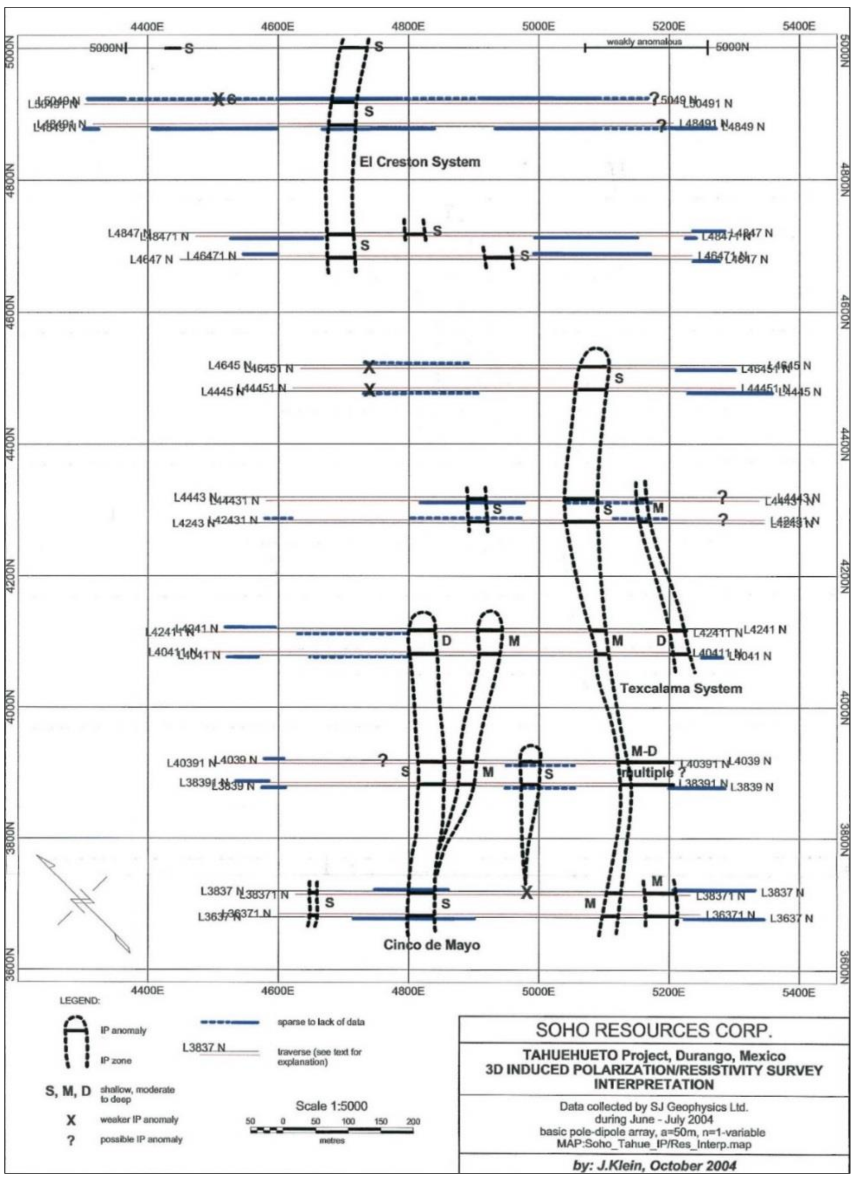
A geophysical survey was implemented in 2004 to prove continuity between the known mineralized zones at El Creston and Cinco de Mayo, followed by a drilling program to test El Creston, Cinco de Mayo, and any anomalies generated from the geophysical study.

SJ Geophysics Ltd. conducted the geophysical survey that included resistivity and IP measurements taken on approximately 18.5 km of grid using an Elrec 6 IP receiver and an Androtex 10 Kw transmitter (Visser, 2004). The configuration used for this survey was a 3Denhanced equivalent form of dipole-dipole IP with a 12 m by 50 m dipole array. Data were analyzed using the DCINV2D and 3D inversion program, which converts surface IP/resistivity measurements into a realistic "Interpreted Depth Section". The 3D IP survey was designed to examine the sulfide mineralization at El Creston, Cinco de Mayo, and Texcalama, and to test the intervening area for possible extensions of these mineralized zones. The following summarizes the results of this survey:

- The survey conditions were favorable, good electrical contact to the ground was established and high quality data was recorded across the entire survey grid.
- The portions of the El Creston, Cinco de Mayo and Texcalama mineralized zones surveyed returned significant anomalous chargeability responses. Discontinuously extending between these zones, which bracket the survey grid, run a suite of highly chargeable features.
- There does not appear to be a strong resistivity association with the known mineralization. The El Creston and Cinco de Mayo exhibit elevated resistivities. The Texcalama vein system is cross cut by a significant NW-SE trending resistive feature and may reflect a lithological contact between the Lower Volcanic Series and the "El Rey" Intrusive Suite (Visser, 2004).

Klein (2004) later reviewed the data from the geophysical program; his interpretive results are shown in Figure 9-1.

Figure 9-1 3D Induced Polarization/Resistivity Survey Interpretation



In November 2004 Dateline Internacional, S.A. De C.V. was contracted and drilled 34 reverse circulation ("RC") holes to test the induced polarization ("IP") geophysical chargeability anomalies located within and between the El Creston and Cinco de Mayo zones along the El Creston-Cinco de Mayo Trend as well as to test the chargeability responses along the Texcalama Trend. The RC drilling program commenced in December 2004 (Soho, 2004d).

Drill related activities had been the primary focus at Tahuehueto from 2005 thru 2008. Telson has undertaken a number of other exploration related activities since 2004 (Soho, 2004b,c, 2006a,c,d,e,f). Surface and underground sampling programs include:

- Underground at Cinco de Mayo South
- Surface and underground at Texcalama
- Surface at Santiago
- Surface at El Pitallo
- Underground at Espinal
- Underground at El Rey
- Surface along the northern Cinco de Mayo trend
- Follow up surface at Santiago
- Surface and underground at the numerous prospects & color anomalies on the property

Surface geologic mapping was initiated in 2004 and suggested that mineralization is closely related to coeval faulting, felsic volcanism, and sedimentation, and that mineralizing structures continue from the lower volcanic units, where they are most pronounced, into the uppervolcanic units (Soho, 2004b).

Several petrographic and fluid inclusion reports have been generated on samples from Tahuehueto. A total of 32 rock samples were sent to PetraScience Consultants Inc. for petrographic study in 2004, from which 30 were selected for petrographic analyses (Dunne, 2004b). Eight of these samples (2 from El Creston; 4 from Cinco de Mayo, 1 from Texcalama, and 1 from El Rey,) were selected for fluid inclusion petrography and micro-thermometry (Dunne, 2004a). The eight samples from the El Creston Zone comprise a variety of quartz veins, breccias and quartz vein breccias with primary and replacement vein textures and alteration assemblages indicative of the low sulphidation (adularia-sericite type) epithermal environment (Dunne, 2004b). The seven breccias/quartz vein breccias/vein stockwork samples from the Cinco de Mayo Zone showed alteration assemblages indicative of the lower crustiformcolloform and crystalline superzones of a low sulphidation epithermal environment and possibly transitional to polymetallic gold-silver veins or the quartz-sulphide-gold-copper environment describe by Corbett (2002). Two of the four samples from the Texcalama Zone may contain former porphyritic rock fragments. The two samples from El Rey were similar to those from level 3 at El Creston. One of four samples from the Tres de Mayo Zone contains "wispy quartz" texture characteristic of metamorphosed or deep vein systems (Dunne, 2004b).

Fluid inclusion evidence for boiling is present from two samples from Cinco de Mayo and one from Texcalama. Mineralogical evidence for boiling (lattice-bladed or ghost-bladed textures that pseudomorph lattice carbonate exist in samples from level 3 at El Creston, and from Texcalama and El Rey. Dunne

(2004a) stated that homogenization temperatures and salinities fall in the expected range for epithermal deposits and fall in the classification of shallow, boiling low-sulphidation epithermal deposits.

Five additional rock samples were submitted for petrographic analysis to PetraScience Consultants Inc. in 2005. An additional three samples were included for fluid inclusion analysis, but were deemed of little value so were not analyzed (Dunne and Thompson, 2005). The samples were described as variably altered volcanic or volcaniclastic rocks. Alteration consisted of early pervasive K-feldspar alteration followed by assemblages consisting of variable amounts of chlorite, calcite, hematite, sericite, and quartz. Spatial information was not included, so the data are of limited value. Eleven drill-core samples were sent to Kathryn Dunne for petrographic analysis in 2007.

Fourteen polished thin sections were prepared from these samples; and ten doubly polished fluid inclusion plates were prepared from eight of the samples (Dunne, 2007a; Dunne, 2007b).

The results from these samples were consistent with the observations and conclusions previously reported by Dunne (2004b). Drill cuttings samples from drill hole RC-018 from El Creston were submitted to Vancouver Petrographics Ltd in 2005. The samples were from six consecutive 1.5 m intervals from 67.06 to 76.2 m containing high-grade gold values (11.05-62.3 g/t Au). Both a screened fine grained sample and a coarse grained sample were included for all but the 67.1 to 68.6 m interval. The detailed descriptions of the samples focused on the distribution of native gold which occurred in almost all samples in a variety of textures (Payne, 2005). "*Most commonly it forms inclusions in pyrite, in part associated with other sulphides and in part alone*. *Less commonly it is associated with chalcopyrite or galena; in most of these occurrences, chalcopyrite and galena are associated with pyrite, either as inclusions or fracture-filling patches. Also widespread but not abundant are disseminated, isolated, small grains in sphalerite. A few free grains of native gold are present. One grain of native gold occurs in sericite. No native gold was seen in quartz. Grain size of native gold is mainly from 0.01- 0.05 mm, with a few grains up to 0.15 mm long. Grains smaller than 0.007 mm in size are not abundant and commonly occur near larger grains of native gold, mainly as inclusions in pyrite.*"

Six samples (type and locations not identified) were submitted to Vancouver Petrographics Ltd. in 2007. They are described by Leitch, (2007) as being "*strongly to intensely silicic/phyllic/advanced argillic altered and veined felsic volcanic rocks. Alteration locally obscures the original rock type, especially where it is associated with brecciation and significant to pervasive silicification and comb or cockade-textured, vuggy to drusy quartz veining.*"

A second suite of 14 samples, taken from core holes, was later submitted to Vancouver Petrographics Ltd. in 2007. Seven or eight of the samples were described as pre-mineral hypabyssal quartz latite porphyries; two as being "micro diorite" and four as "late" dikes of latite to trachyandesite composition (Leitch, 2007). Alteration ranges from propylitic through transitional propylitic/potassic to potassic.

A lithostructural Interpretation using satellite imagery was conducted by Technologies Earthmetric Inc., Montreal, Quebec, Canada on the Tahuehueto Project and surrounding areas for Soho in 2007 (Moreau, 2006). A series of maps at variable scales show interpreted regional and local structural features, interpreted veins and altered areas, along with target areas for exploration generated from the structural interpretation. Although of interest, Telson has not specifically targeted drill holes based on this data.

10 DRILLING

10.1 CONSEJO DE RECURSOS MINERALES DRILLING

The only drilling known to have been undertaken prior to Telson's involvement at Tahuehueto was conducted by the Consejo de Recursos Minerales. Although 28 surface and underground drill holes were reportedly drilled on the El Creston and Cinco de Mayo structures (Consejo de Recursos Minerales, 1983b), Telson was unable to obtain drill logs, collar locations, or results from this drilling.

According to the Consejo de Recursos Minerales (1983b), 15 angle holes totaling of 2,026.87 m were drilled from the surface using Longyear 34 and Longyear 24 rigs. Six of these holes, totaling 813.17 m, were drilled at El Rey; six more, totaling 858.15 m, were drilled at Cinco de Mayo; one 131.60m hole was drilled at El Creston; and two holes, totaling 223.95 m, were drilled at Tres de Mayo. An additional 13 holes, for a total of 4224.40 m, were reportedly drilled underground with a Pack Sack JKS25. Four of these holes (119.20 m) were drilled at El Rey; seven (234.50 m) were drilled at Cinco de Mayo; and two (70.70 m) were drilled at El Creston.

10.2 TELSON DRILLING

Telson first began drilling at Tahuehueto in December 2004 and completed 34 RC holes (RC-001 to RC034, including RC-006A, RC-008A, and RC-028A) during 2005; Twelve holes were drilled at Cinco de Mayo, two at Texcalama and the remainder at El Creston. Dateline Internacional, S.A. de C.V. of Hermosillo, Mexico was the drill contractor for this program. The RC rig was demobilized and replaced by an LF 70 core rig from Mexcore, S.A. de C.V. ("Mexcore") in June 2005 (Soho, 2005c). A total of 50 core holes were drilled with this rig from June 2005 to July 2006; 36 of these holes were drilled in 2005, with two holes drilled at Cinco de Mayo and the remainder at El Creston.

Telson expanded its core drilling to two rigs in August 2006, a UDR 200 and a JT 3000 rig from Major Drilling de Mexico, S.A. de C.V. ("Major"). The two Major rigs completed 76 holes before their contract terminated in July 2007 - DDH06-49 through DDH06-064 (including DDH06-051A), DDH07-065 through DDH07-121, and DDH07-123. A total of 32 holes were drilled in 2006 at Cinco de Mayo, El Creston, El Rey, and Santiago. Core drilling resumed in August 2007 with a Longyear 38 rig contracted through Tecmin Servicios, S.A. de C.V. ("Tecmin") of Zacatecas, Mexico. Tecmin drilled 13 holes through January 2008, including DDH07-122, DDH07-124 through DDH07-126, and DDH07-128 through DDH07- 136. A total of 72 holes were drilled at Cinco de Mayo, El Creston, and Santiago in 2007.

After attempts at establishing road access and drill sites at the intersection of the El Creston and El Perdido structures failed due to extremely steep topography, Telson developed ten remote drill sites and drilled one core hole (DDH07-127) using a fly-capable rig purchased by Telson (Soho, 2007d). The rig, which was operated by Telson, was transported to the drill pad by helicopter.

A total of 72 core holes were drilled in 2007. At the end of 2007, Telson contracted with Falcon Perforaciones Mexico, S.A. de C.V., who began core drilling in January 2008 (Telson, 2007g).

During 2008 an additional 34 core holes were completed for a total of 211 core holes for the project (includes 4 A holes), before drilling was shut down in August 2008.

Seventeen of the Mexcore holes, as well as the hole drilled by Telson's fly rig, were drilled using

NQ core. The remaining holes were drilled with HQ core, which was reduced to NQ when required by ground conditions. All of the core rigs were skid-mounted.

10.3 DRILL COLLAR SURVEYING

The drill-hole collar locations were surveyed by a variety of methods. A total of 123 of the holes were surveyed using a differential GPS instrument; the elevations for six of these holes were assigned by PhotoSat using photogrammetry, and the elevations for three of the holes were assigned by Telson using the project topography. Total station equipment was used to survey 30 holes. Seven holes (six used in the resource estimation) were surveyed using handheld GPS, and 17 (16 used in the resource estimation) were surveyed by chain-and-compass.

10.4 DOWN-HOLE SURVEYING

Core holes drilled in 2005 were surveyed with a Tropari, Reflex EZ-SHOT, or Flexit. All core holes drilled in 2006 were surveyed with a Flexit, and 2007 core holes were surveyed with either a Flexit or a Reflex EZ-SHOT. The down-hole survey data indicate that the hole deviations are typically minor, usually steepening by less than two degrees.

A total of 12 core holes have no down-hole survey data. Five of these holes were abandoned and not assayed, and one is located outside of the resource modeling area. No RC holes have down-hole surveys.

10.5 CORE HANDLING PROCEDURES

The core is laid out on logging tables that can accommodate up to 60 boxes of core. The core is re-assembled, washed by technicians, and a geologist reviews the core blocks for significant recovery or re-assembly problems. Technicians then measure RQD and recovery. Geologists log the core, mark sample intervals, and draw cut lines using a wax crayon. After logging, the core is photographed with the sample tags in place.

10.6 DRILL-HOLE DATABASE

MMC was provided with a drill hole database that included collar, survey, and geology data tables. The resources reported in this report were estimated using the Telson database, which includes a total of 248 holes drilled by Telson at Tahuehueto through the end of 2008, including 37 RC holes and 211 core holes (Table 11.1).

Table 10.1 Tahuehueto Resource Drilling Summary

RC		Core		Total Drill Holes	Total Meters
No	Meters	No	Meters		
37	3,668	211	43,608	248	47,276

Most of the holes were angled towards the northwest in order to cut the southeast-dipping mineralized structures, although the challenging topography hindered drill pad locations and many of these holes were not strictly orthogonal to the structures. Several holes, especially at El Creston, were collared in the footwall and angled back towards the structures, which yields intercepts significantly in excess of true thicknesses.

The drill hole database used for the Tahuehueto Mineral Resource estimation is further summarized in Table 10.2. An isometric view of the Project area with drill hole collars highlighted is presented in Figure 10-1. Figure 10-2 shows a plan map of the Tahuehueto project area with the surface projection of the drill holes. In addition to the drill hole information, the project database includes assays for a total of 1,788 underground samples, including 88 from the Cinco de Mayo workings, 450 from El Rey, and 1,250 from El Creston.

Table 10.2 Summary of Drilling Database

Item	Value
Number of Holes	248
Total Length (m)	47,276
Average Length (m)	190
Meters Sampled & Assayed	18,392
Drill hole Samples with Assays	18,339
Core Holes with Downhole Surveys	199
RC Holes with Downhole Surveys	0

Figure 10-1 Tahuehueto Drill Hole Locations

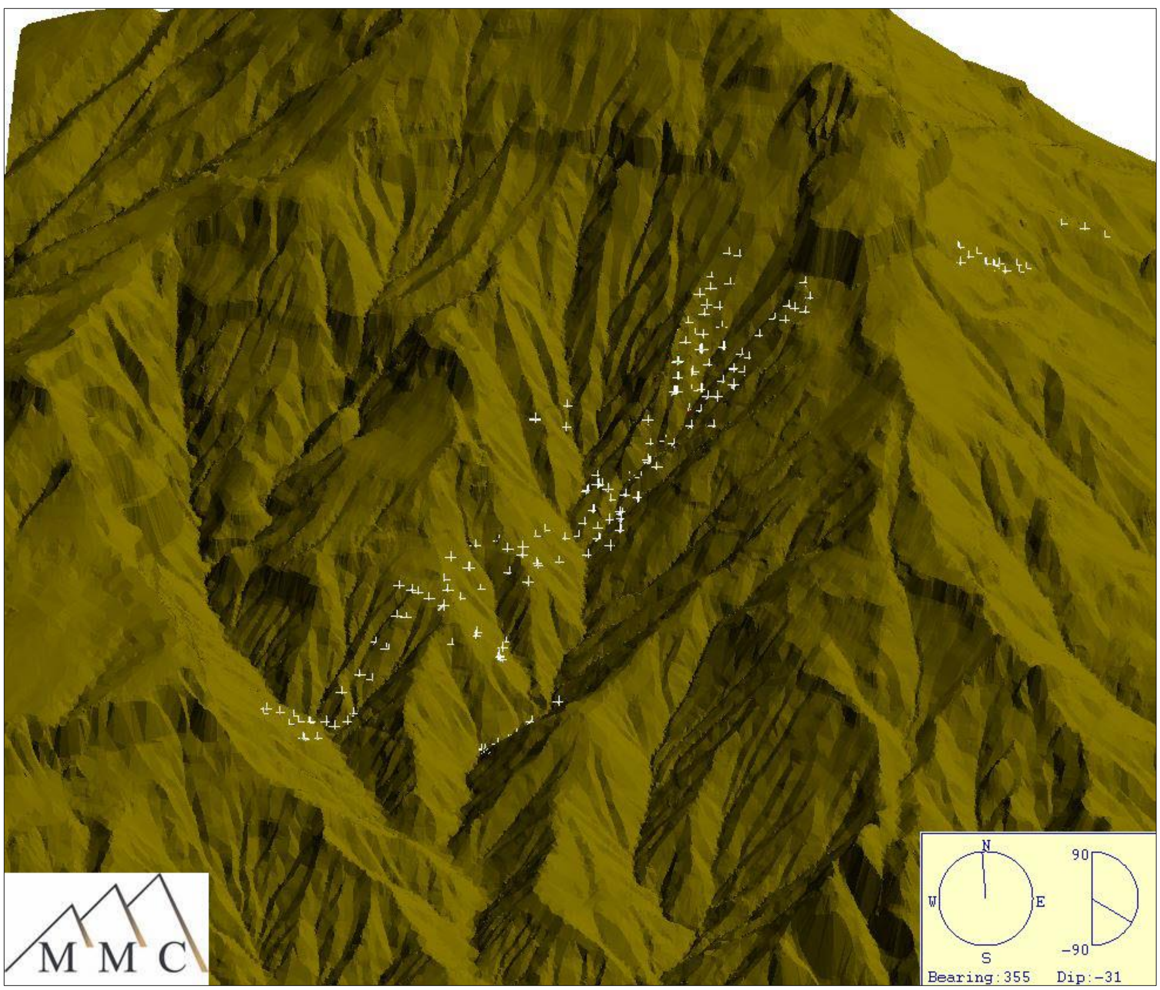
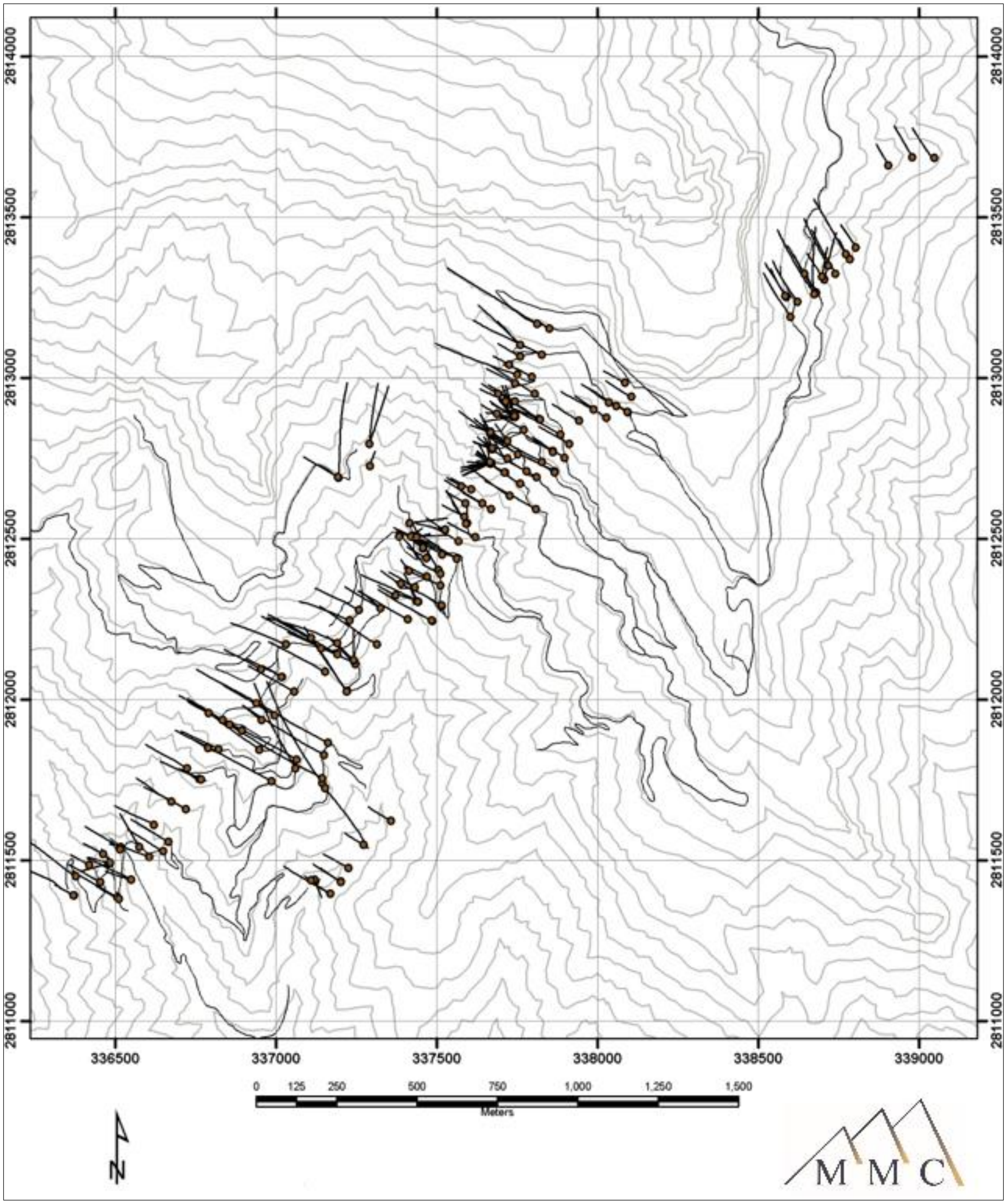


Figure 10-2 Tahuehueto Drill Hole Locations and Down Hole Traces



11 SAMPLE PREPARATION, ANALYSIS AND SECURITY

The Tahuehueto database includes Telson RC and core holes. MMC believes that the RC and core sampling procedures provided samples that are sufficiently representative and of sufficient quality for use in the Mineral Resource estimation discussed in Section 14. Results from Telson channel sampling of the underground workings are also included in the project database; channel sample data from El Rey were used in the Mineral Resource estimation.

11.1 HISTORIC SAMPLING

The Consejo de Recursos Minerales collected a total of 301 surface samples, 3,009 underground samples, and 116 drill samples during their exploration programs at the El Rey, Cinco de Mayo, El Creston, Texcalama, and Tres de Mayo zones (Consejo de Recursos Minerales, 1983b). No further details of these programs, including the drilling and sampling results, are known to Telson.

Castle collected 459 surface chip and underground chip-channel samples in 1994; 247 from the El Creston structure, 21 from the Cinco de Mayo structure, and 191 from other sites on the property (Brown, 2004). Telson does not have the results or any further details about the Castle sampling methods.

11.2 SAMPLING BY TELSON

11.2.1 TELSON CHANNEL SAMPLING

In 1997, Telson undertook channel sampling in ten of the underground levels on the El Creston vein. Approximately 1,200 underground and surface channel samples were taken from the El Creston zone, with a few samples taken at Dolores, Cinco de Mayo and Los Burros.

Channel samples were taken with chisel and hammer, and represent no more than a 1.5 meter sample width. Channel samples taken in cross cuts were generally a 1.5 meter width, while channel samples from drifts along the mineralized structure were from a 1.0-1.5 meter width depending on the width of the drift. Along the drifts, channel samples were taken at 2.5-meter centers. Forty-two check panel samples were taken over channel sample sites to confirm analytical results. Select channel and panel samples were then re-assayed from reject material to check the laboratory accuracy (Brown, 2004).

Drift channel samples were taken across the roof of the drift, perpendicular to the mineralized zone, while crosscut channel samples were taken at waist height on the crosscut wall.

Telson undertook detailed underground sampling in 2004 of the Cinco de Mayo South, Cinco de Mayo North 1, Texcalama 1, 2, and 3 adits to determine possible extensions of the El Creston zone (Soho, 2004b). The channel sampling program was been undertaken in the Cinco de Mayo main (southwest) adit, and the Texcalama 1, 2 and 3 adits. Sample locations, at 2.5 m intervals, were delineated by straight chain and demarked with spray paint to allow for further reference and repeat sampling. All samples within adits (as opposed to crosscuts) were acquired across the ceiling of the adit in a continuous hammer and chisel channel sample. The entire width of the adit was sampled. If the adit width exceeded 2 m the sample was split into 2 or more samples. Where crosscuts were encountered, several samples were collected across the entire crosscut width and each individual sample did not exceed 2 m in width. Each sample was a continuous hammer and chisel channel sample across the inward wall the crosscut.

The channel sampling technique for the adit sampling program is consistent utilizing a 4 lb short-handled sledge hammer and chisel to cut a channel continuously across the adit ceilings or cross cut walls. Attention and best effort was paid to acquiring consistent volumes of material across each sample. To ensure sufficient representative material was acquired, each sample averaged in the 2.5 to 3 kg range.

All samples were labeled, bagged and sealed (zap strapped) on location. The samples were then transported by burro to the camp office where they were sorted, grouped and sealed in rice bags for transportation to Durango by company truck. In Durango the samples were transferred to the company's subcontractor, Engineer Artemio Terrazas, for immediate delivery to ALS Chemex's sample prep lab in Guadalahara. Once prepped, ALS Chemex oversaw the shipment of the samples to its assay lab in North Vancouver, BC.

In 2005-2006, Telson undertook additional underground and surface sampling at the Santiago, Pitallo, Espinal, and El Rey mineralized zones (Soho, 2006a, 2006d). The channel samples did not exceed a maximum length of 1.5 m, with the limits of sampled material respecting geological contacts. The channels were cut across the structure at El Rey at 330°, and individual samples were collected across lengths of one meter or less. Over 150 m of the vein structure were sampled (Canova, 2006a).

Channels were cut every 4 m across the structure that trends 060° and dips 80°SE with widths of 1.0 m to 2.0 m. A total of 38 channels were cut across the structure. The structure is generally 1.8 m wide and consists of quartz-carbonate veining with visible mineralization of sphalerite, galena, and weak chalcopyrite.

The structure cuts across a grey, fine to medium grain granodiorite that is massive. The structure is narrow, linear, and contain gold, silver, lead and zinc. (Canova, 2006a).

Locations were surveyed in by straight chain and brunton, and tied to the adit portals. The resultant coordinates for the channel samples were then calculated based on the surveyed portal locations data at that time. The sampling conducted down the adit entry tunnels took place along the eastern walls of the adit at a height of approximately 1.4 m. Along the vein portion of the adits the sampling was along the adit ceilings. The width of the channel for the continuous chip / channel samples was approximately 15 cm. Sample size varied due to variable sample lengths. Unlike the first sampling conducted in Cinco de Mayo, which were channels across the entire adit ceilings unless the adit width exceeded 2 m, the El Rey sampling was broken into contiguous footwall, vein and hanging wall segments (Gustin, 2008).

A grid was established on the Santiago structure in early 2006 oriented with a 060° bearing along the Santiago structure and covering a strike length of more than 180 m (Canova, 2006b). Eight channels were cut across the structure. A total of 124 samples were collected, and the results indicated the width of the structure to be approximately 7 to 16.5 m.

11.2.2 TELSON REVERSE-CIRCULATION SAMPLING

Telson drilled 37 RC holes at Tahuehueto, all in the first half of 2005. Samples were collected every 1.524m from the rig's cyclone with about 30kg of material per sample. Every 1.524 m run was split into quarters with a sample splitter, with one-quarter bagged and sealed for shipment to the assay laboratory. The remaining three quarters were bagged, sealed, and stored at the project's field facilities. For every fifth sample, a duplicate sample (equal to one quarter of the total sample) was collected for quality-control analyses. At the field office, samples were recorded, batched, and sealed in large rice bags. Telson personnel drove the samples from the project site to Durango, where they were shipped by secure courier to the sample-preparation facilities of ALS Chemex ("Chemex") in Guadalajara, Mexico.

11.2.3 TELSON CORE SAMPLING

Telson began core drilling in mid-2005. Telson drilled with HQ and NQ core, depending on drilling conditions.

Samples varied from 0.5 to 2 m in length, averaging 1.0 m. Core samples were cut in half longitudinally with a rock saw, with one-half sent for assay and the remaining half boxed, sealed and stored at the project's field facilities. Samples were recorded, batched, and sealed in large rice bags at the field office, and then were shipped by tlson staff to the sample preparation facilities - SGS Minerals Services ("SGS") in Durango, Mexico in 2005 and 2006, and Inspectorate de Mexico S.A. de C.V. ("Inspectorate") in Durango in 2007 through to September 2007 (Soho, 2007a). Since September 2007, samples have been prepared at Chemex's preparation facility in Guadalajara, Mexico (Soho, 2007e).

Telson reports that the core was generally sampled over regular intervals that varied from 30 cm to 1.50 m, with sample intervals coinciding with major lithological boundaries and veins. In intervals where core recovery was less than 70%, samples within that 3.048 m run were sampled as a full 3.048 m interval. Samples were split lengthwise with a diamond saw, with one half taken for assay and the remainder retained for future reference. One blank sample was inserted at random every 25 samples and was placed after a highly mineralized zone, if possible. One standard sample was inserted into each batch of 24 core samples.

11.3 QUALITY ASSURANCE AND QUALITY CONTROL PROGRAMS

11.3.1 REVERSE-CIRCULATION SAMPLES

The Telson RC samples were prepped at the Chemex sample-preparation facilities in Guadalajara, Mexico, and the pulps were shipped by Chemex to their laboratory in North Vancouver for analysis (Soho, 2005a). Chemex is ISO 9001:2000 registered. Gold analysis was conducted by fire assaying a 30 g charge and utilizing a gravimetric finish (Chemex method Au-GRA21). Silver, copper, lead, and zinc were analyzed by ICP-AES ("inductively coupled plasma – atomic emission spectroscopy") following three-acid digestion and HCL leach (method ME-ICP61).

Over-limit silver, copper, lead, and zinc (100 ppm for Ag and 10,000 ppm for the base metals) results were re-assayed by three-acid digestion and HCL leach, with an AAS ("atomic absorption spectroscopy") finish (method AA62); approximately 2% of the samples were also analyzed for silver by fire assaying with gravimetric finish.

11.3.2 CORE SAMPLES

Core holes DDH05-001 through 05-031 and DDH05-033 through DDH06-048 were analyzed at the SGS lab in Toronto, Canada; SGS is ISO/IEC 17025 and ISO/IEC 9002 registered. The samples were first sent to the SGS sample-preparation facilities in Durango, Mexico, and then the pulps were shipped by SGS to the Toronto lab. Gold grade was determined by fire assaying of 30 g charges and finishing with AAS (SGS method FAA313); over-limit (>10g/t) analyses were completed by fire assaying 30 g charges and completing with gravimetric finish (method FAG303). Silver, copper, lead, and zinc grades for all samples were determined using four-acid digestion followed by ICP-AES analysis (method ICP40B). Silver over limits (>10ppm) for samples from holes DDH06-037 through 06-048 were determined by AAS after three-acid digestion (method AAS21E); over limits for the earlier holes were by method AAS40E, which no longer exists but may have been similar to AAS21E. Methods AAS21E and AAS40E both had an upper threshold of 300 ppm; samples exceeding this limit were analyzed by method AAA50, which is reported in units of g/t and has a 10 g/t detection limit, but is not described on the SGS website. AAS40E and AAS21E analyses were also completed on a number of the samples that were not subject to ICP40B over-limit assaying. Copper, lead, and zinc over-limit results were determined by method ICA50, which is reported in percent and has detection limits of 0.01% for all base metals, but is not listed as a current assaying method by SGS.

Inspectorate analyzed core samples from DDH05-032 and DDH06-049A through DDH07-121 in their Sparks, Nevada facility; pulps were first prepared at Inspectorate's preparation facility in Durango, Mexico. Gold was analyzed by 30 g fire assay with an AAS finish (Inspectorate method Au-FAA); all results of 3 g Au/t or greater were re-assayed by fire assaying with a gravimetric finish (method FAGRAV). Primary silver, copper, lead, and zinc determinations alternated between AAS analyses following aqua regia digestion (Soho, 2007d) and ICP; some primary silver analyses consisted of fire assaying with an AAS finish. The ICP and AAS methods had upper analytical limits of 200 ppm for Ag and 10,000 ppm for the base metals; over-limit analyses on these samples used the FAGRAV method for silver and the "AAS - Zinc by AA Assay" method for the base metals.

Samples from core holes DDH07-122 through DDH08-207 were also analyzed by Chemex; sample pulps were prepared at the Chemex facility in Guadalajara, Mexico. The pulps were first shipped by Chemex to their analytical laboratories in Lima, Peru for analysis (Soho, 2007e), then to the Chemex laboratory in Vancouver, Canada for analysis between September and December 2007; in January 2008 the pulps were once again being sent to the Lima laboratory (Soho, 2008a). Gold assays were first done on 30g charges by fire assaying with an AAS finish (method Au-AA23); over- limit (>10g/t) analyses were completed using the Au-GRA21 method. Silver, copper, lead, and zinc were analyzed by method OG62 (similar to AA62).

The following description of the custody procedures for the drill core and samples for holes DDH07-077 through DDH08-207 was provided by Telson. Core was in the custody of the drill crew until Telson geologists picked it up twice a day at about 9:00 AM and 6:00 PM. The core was taken to a fenced core-logging facility, where it was stacked until logging and sampling. At the end of each day, the bagged samples were moved into the portal of an adit near the core shed, which was secured with a locked gate. Samples were shipped from the project site to Durango in Telson vehicles by Telson personnel. In Durango, samples were shipped to Chemex in Hermosillo by Paqueteria y Mensajeria en Movimienito (a secure courier with a long-term contract with Chemex).

11.3.3 Underground Samples

Telson's surface and underground channel sampling in 1997 was conducted and supervised by three Canadian geologists, including Brown. Samples were prepared by Chemex at their facility in Hermosillo, Mexico, and then the pulps were sent to Chemex's lab in Vancouver, Canada for analysis (Brown, 2004, although Brown, 1998a, states that the 1997 channel samples were shipped directly to the Vancouver lab for both sample preparation and analysis). Brown (1998a) reports that samples were assayed for gold and a 30- element ICP package. Gold was initially assayed by fire assay with an AAS finish using a 30g charge. Samples with gold above 12g Au/t were re-assayed by one-assay-ton fire assay with a gravimetric finish. Samples with silver greater than 200ppm were re-assayed by fire assay with a gravimetric finish. Samples with lead or zinc exceeding 50,000ppm were re-assayed by atomic absorption using nitric-HCl-acid digestion.

For Telson’s 2004 sampling program, samples were prepared by Chemex at their sample prep lab in Guadalajara, Mexico and the pulps were shipped to Chemex’s lab in Vancouver for analysis.

According to an undated summary of the sampling of the Cinco de Mayo and Texcalama adits provided to MDA by Telson, primary gold analyses consisted of fire assaying with an AAS finish.

Samples with values exceeding this method’s upper limits of 10 g/t were then assayed by fire assay with a gravimetric finish. In addition, analysis of a suite of an additional 33 elements was done by ICP-AES, and where upper limits were exceeded for silver and/or base metals, samples were analyzed by aqua regia or acid digestion and AAS. A field-derived standard was inserted at regular intervals in the sample series, and the lab performed duplicate analyses on every 40th sample in a run. In addition, the lab inserted a blank at the beginning of each run as well as standards at random intervals. For the Cinco de Mayo and Texcalama 1 and 2 adits, blanks and standards were inserted by Telson into the sample sequence (every 20th sample) for assay quality control. For the Texcalama 3 adit, duplicate samples were taken every 20th sample (10th, 30th, 50th etc.) in addition to the above quality-control measures.

Pulps from the Telson 2005 and 2006 surface and underground channel samples from the Santiago, Pitallo, Espinal, and El Rey mineralized zones were prepared at the SGS facility in Durango and the pulps were sent to their Toronto laboratory for analyses.

12 DATA VERIFICATION

12.1 INTEGRITY OF DATABASE

In order to properly record all original assay data in the database, as well as to have unique fields for use in the resource estimation, gold, silver, copper, lead, and zinc fields were created in the database that are separate from the original assay data. These fields are assigned one of the assays for each of the five metals for any given sample based on a consistent hierarchy. For example, gravimetric analyses for gold and silver are given a higher priority than AAS analyses, and “ore grade” assays for copper, lead, and zinc are assigned a higher priority than ICP analyses. MMC verified that the databases were correct before any information was used in this report.

12.2 QUALITY CONTROL / QUALITY ASSURANCE PROGRAM

Quality-control samples were available for review and included duplicate samples, analytical standards, and blanks that were inserted into the sample stream by Telson; this discussion also includes analyses of some of the internal laboratory QA/QC results.

12.2.1 BLANKS

Blank samples are used to test for cross contamination between drill samples in the analytical laboratory, which is most common during sample-preparation stages. In order for the blanks to be meaningful, therefore, they must be sufficiently coarse to require the same crushing stages as the drill samples and should be placed immediately after mineralized drill samples (which would be the source of most cross-contamination issues) in the sample stream.

Telson has been inserting blank samples into the sample stream since drilling began at Tahuehueto in 2005. The coarse blank material is derived from an outcrop within the project area of post-mineral rhyolitic tuffs of the upper volcanic series that lies above the mineralized lower volcanic series rocks.

12.2.2 REFERENCE STANDARDS

To increase the integrity of the sample handling process, from collection to shipment to assay, standards are inserted in the sample stream at a rate of one standard and one blank for every 25 drill samples. The reference standards were prepared by WCM Minerals, a division of WCM Sales, Ltd. of Burnaby, BC, Canada. Reference standards are used to evaluate the analytical accuracy of the assay laboratory.

12.2.3 SURFACE AND UNDERGROUND CHANNEL SAMPLING

Telson completed an underground and surface sampling program at the Santiago, Pitallo, Espinal, and El Rey mineralized zones in 2005 and 2006. Telson reports that blank samples were inserted randomly within each series of 25 samples and standards were inserted every 25th sample during this program. MMC does not have the results from this QA/QC program.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 INTRODUCTION

Tahuehueto mineralization has been analyzed in several metallurgical testing programs. The first registered test was carried out in 1977 by the Comision de Fomento Minero (CFM) and the second was conducted by Westcoast Mineral Testing during 2009 and 2010. A third test was performed by Telson in order to verify the metallurgical parameters required to process, by flotation the El Creston and El Perdido mineralization.

13.2 HISTORICAL TESTING

13.2.1 COMISION DE FOMENTO MINERO TESTINGS

CFM, (currently referred to as Fideicomiso de Fomento Minero, FIFOMI) is a Government Institution that supports the mining industry with loans and technical assistance and with large and well equipped facilities for metallurgical test and process design. These metallurgic laboratories are now operated by the Mexican Geological Service.

The metallurgical testing was performed by CFM under request of the company Emijamex, S.A., a company with Japanese participation. This company tested a 150 kg sample from the El Creston and El Rey veins. The objective of this study was defining the parameters for designing a flotation process plant to process mineralization at 50 tpd.

The metallurgic research carried out included spectrographic qualitative assay, chemical qualitative assay, X-ray diffraction, chemical quantitative assay, mineralogy characterization, mill grinding indices development, and mill testing. In total 10 metallurgic tests were conducted using different parameters, reagents, PH, etc., in order to obtain the best recoveries and concentrates. The quantitative analysis of the sample is indicated in Table 13.1. The mineralogy obtained by CFM is shown in Table 13.2.

Table 13.1 Quantitative Analysis

Element	Assay	Unit
Au	3.00	g/t
Ag	53.00	g/t
An	6.40	%
Pb	3.50	%
Cu	0.24	%
Fe	4.80	%
S	4.64	%
SiO_2	68.19	%
Al_2O_3	4.67	%
CaO	1.18	%
MgO	6.29	%
$CaCO_3$	1.23	%

Table 13.2 Sample Mineralogy

Mineral	%
Sphalerite	9.55
Galena	4.04
Chalcopyrite	0.7
Pyrite	1.33
Calcite	1.23
Quartz, etc.	83.15
Total	100

The mill particle size test results, indicates that below -65 mesh 82.7% of the Zn, 82.8% of the Pb and 80.5% of the copper are free and that a – 100 meshe grind size is required to meet grind size parameters for flotation.

Table 13.3 provides a summary of the 10 metallurgic tests that were carried out by CFM in order to define the best parameters for processing material from Creston and El Rey. The parameters recommended by CFM are from test number 2 and are shown in Table 13.4.

Table 13.3 CFM Metallurgic Test Work Summary

Description		Test Number																			
		1		2		3		4		5		6		7		8		9		10	
Concentrate		Pb	Zn	Pb	Zn	Pb	Zn	Pb	Zn	Pb	Zn	Pb	Zn	Pb	Zn	Pb	Zn	Pb	Zn	Pb	Zn
Assays	Au g/t	32.0	1.7	31.8	2.4	24.5	4.2	12.4	3.3	17.1	2.0	32.5	3.7	34.5	4.1	24.0	5.6	22.4	3.5	19.1	3.0
	Ag g/t	602.0	123.0	782.0	72.0	480.0	108.0	273.0	70.0	345.0	115.0	739.0	120.0	746.0	50.1	554.0	145.0	488.0	94.0	473.0	117.0
	Pb%	43.6	0.1	45.0	0.4	28.0	0.4	11.5	0.3	22.0	0.3	41.0	0.4	44.0	0.4	30.0	0.7	26.0	0.4	28.0	0.5
	Zn%	7.4	56.0	8.0	55.8	8.0	65.0	8.0	52.5	7.8	60.5	8.0	49.4	7.5	49.3	9.0	52.5	9.5	48.8	8.5	60.0
	Fe%	6.4	3.0	6.5	11.3	7.0	11.5	7.0	11.8	7.0	8.3	7.5	15.5	7.0	19.7	6.4	17.0	6.8	13.0	7.2	9.6
Distribution %	Au	87.8	12.2	81.1	2.4	81.0	4.5	92.9	7.1	89.1	4.4	87.2	12.8	86.0	14.0	87.2	12.8	88.6	11.4	92.3	7.7
	Ag	77.2	22.8	87.6	70.0	86.0	10.8	92.2	7.8	85.2	13.8	81.7	18.3	79.7	20.3	84.8	15.1	85.5	14.5	85.9	14.1
	Pb	97.1	0.7	97.2	0.4	98.6	0.4	98.1	0.7	99.5	0.5	97.1	0.9	97.3	0.9	96.4	1.5	96.9	1.2	97.0	0.8
	Zn	9.3	85.5	10.5	55.8	14.9	65.0	28.7	70.5	19.8	78.7	9.7	88.0	9.0	88.7	16.6	78.4	17.2	80.2	15.9	81.1
	Fe	11.1	29.8	12.3	9.6	18.9	11.5	31.4	20.9	20.2	8.1	14.3	38.1	11.2	38.1	17.5	28.2	20.8	31.5	21.1	10.6
CR		12.8	10.5	11.9	10.1	7.9	11.6	4.3	11.6	6.0	11.6	13.3	9.1	12.9	8.6	8.9	12.0	7.7	8.5	8.5	11.8

Table 13.4 Number 2 Test Recommended by CFM

Product	WT %	Assays					Distribution %					CR
		Au g/t	Ag g/t	Pb %	Zn %	Fe %	Au	Ag	Pb	Zn	Fe	
Feed	100.0	3.3	75.0	3.9	6.4	4.4	100.0	100.0	100.0	100.0	100.0	
Pb Conc	8.4	31.8	782.0	45.0	8.0	6.5	81.1	87.6	97.2	10.5	6.5	11.9
Zn Conc	15.7	1.4	51.0	0.2	55.8	9.6	6.0	8.3	0.8	88.3	15.0	10.1
Tails	75.9	0.6	4.0	0.1	0.1	4.3	12.9	4.1	2.0	1.2	72.7	

13.2.2 WESTCOAST MINERAL TESTING

The second set of metallurgical tests were performed during 2009 and 2010 by Westcoast Mineral Testing Inc. from North Vancouver, B.C., Canada. Westcoast commentaries about the CFM test are as follows:

- There is very good recovery and distribution of metals into two flotation concentrates.
- Of particular significance and economic importance are the elevated recoveries of gold and silver into the lead-copper (actually "bulk") concentrate.
- The composite contains galena and chalcopyrite in a ratio of 5.8:1. The recovery of copper, although not reported, was likely to be quite high based upon subsequent testing by Westcoast Mineral Testing Inc. This partially explains the low 45 % Pb grade of the lead concentrate, since a clean lead concentrate typically grades > 65 % Pb.
- The deposit contains significant visible gold, so there will be some temptation to consider a gravity concentration stage. This needs to be investigated, but the potential for theft of a highly valuable gravity concentrate also needs to be considered.
- The zinc concentrate grade at 55.8 % Zn with a very modest 2.6 % Fe indicates that the zinc is not significantly marmatitic, and either the modest pyrite (Py) content in the feed (1.3 % Py) did not float with the zinc or the Py was easily depressed with lime. Both of these flotation characteristics will improve concentrate grade and metal recovery.
- The testing reported excellent concentrate values, in part because of the good payable grades of gold and silver in the lead concentrate.

From 1977 when Fomento Minero ran its metallurgical test to 2009, a number of changes in practices of mineral processing were adopted in the industry and these were undertaken by Westcoast for its investigation. The 2009 testing program carried out by Westcoast included ten bench scale flotation tests to investigate three process variables: collectors, frothers and pH.

In October 2009, Telson prepared five metallurgical composites, each of about 20 kg, all from diamond drill core assay rejects. From 40 to 69 individual samples were used to prepare each of four "zone" composites. An "overall composite" was prepared and was weight averaged to reflect the resource tonnage by zones, as shown in Tables 13.5 and 13.6

Table 13.5 Distribution of "Overall Composite"

Zone	% Mass
Cinco de Mayo	17
El Creston	42
El Perdido	14
El Catorce	27

Table 13.6 Composite Assays

Zone	Au g/t	Ag g/t	Cu %	Pb %	Zn %
Cinco de Mayo	1.70	58.00	0.45	0.92	2.29
El Creston	2.89	37.00	0.36	1.41	2.11
El Perdido	1.42	35.00	0.25	1.07	1.50
El Catorce	1.53	27.00	0.07	0.81	1.97
Overall Composite	**2.11**	**38.00**	**0.28**	**1.12**	**2.02**

Westcoast found that some of the composites exhibited modest oxidation of the base metals, averaging <5%. With that modest degree of oxidation, it is unlikely that economic justification exists to incur additional capital and operating costs to recover any oxidized base metals. Consequently, Westcoast did not develop processing performance characteristics for the oxidized material and Reyna agree with this consideration.

The composites were subjected to 15 bench scale flotation tests by Westcoast Mineral Testing of North Vancouver, BC (Hawthorn 2010). This test series evaluated flotation response to the following conditions including primary grind in the range P80 = 90 – 230 microns, several sphalerite depressants in the bulk Cu/Pb rougher stage, and Cu/Pb separation from the final bulk concentrate.

A summary of the copper – lead (bulk) rougher flotation stage results, showing only the "depressants" is shown in Table 13.7. Reagent additions are reported in g/t of feed. The best overall rougher stage recoveries and best selectivity, at nominal primary grinds of 60 % passing 200 mesh (P80 of 100 microns), are tabulated in Table 13.8.

Table 13.7 2009 Flotation Testwork Results

Test	Comp	Grind P_{80} (µm)	$ZnSo_4$	NaCN	NaMBS	Rec to Cu/Pb Rougher Conc		
						Cu %	Pb %	Zn %
09-11	Overall	230				61.9	74.9	52.5
09-12	Overall	105				64.0	65.8	41.3
10-12	Overall	230	1000	200		58.9	67.0	25.9
10-16	Overall	103	1000	200		77.8	84.5	23.6
10-17	Overall	91	1000	200		78.5	83.8	17.9
09-14	Cinco	230				74.6	83.4	48.7
09-15	Cinco	230	500	100		81.3	88.4	27.6
10-02	Cinco	230	1000			76.5	87.1	59.4
10-04	Cinco	117	1000	200		79.8	87.6	26.3
10-05	Cinco	117			2000	75.7	83.0	72.8
10-06	Cinco	98	500	100		88.6	98.8	16.4
10-03	Creston	92	1000			60.0	82.4	25.4
10-07	Creston	99	500	100		81.6	87.7	30.6
10-09	Creston	100	500	100		7.9	85.3	14.7
10-08	Perdido	92	500	100		84.8	93.6	56.0

Table 13.8 Best Westcoast Overall Rougher Flotation

Zone	Test	% Recovered				
		Au	Ag	Cu	Pb	Zn
Cinco de Mayo	10-06	91.5	91.4	88.6	98.8	97.2
El Creston	10-07	83.4	78.7	81.6	87.7	53.2
El Perdido	10-08	82.5	74.2	84.8	93.6	64.9
El Catorce	10-09	87.2	83.5	73.9	98.3	87.7
Arithmetic Average		86.1	82.0	82.2	94.6	75.8
Overall Composite	10-16	85.6	90.1	87.9	86.7	88.1

Although only preliminary in scope, the testing determined that a significant portion of the sphalerite is naturally floatable to the extent of 50 – 70 %.

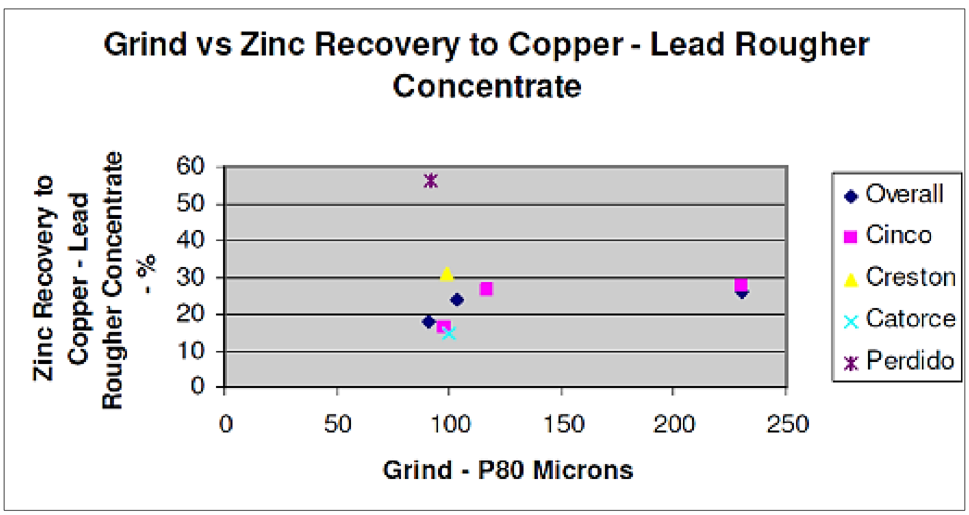
The addition of 2 kg/t of sodium metabisulphite (a source of sulphur dioxide) was not effective as a depressant, and neither was the addition of zinc sulphate without cyanide.

At this stage of the investigation, with the exception of El Perdido, 500 g/t of ZnSO4 and 100 g/t of NaCN is considered to be reasonably effective as a depressant, in that it reduces the zinc recovery into the bulk rougher concentrate to < 20 % at a grind of P80 of 90 – 100 microns.

Figure 13-1 shows grind size versus Zn recovery to Cu/Pb rougher concentrate for only those test in which both $ZnSO_4$ and NaCn were used. Note that primary grinds finer than P80 of 90 microns were not investigated, but it should be, since Figure 13-1 clearly shows decreasing zinc recovery into the copper rougher concentrate at finer grinds for both the "overall" and the Cinco de Mayo composites.

Although there are more data points for "Overall" and Cinco de Mayo, the data indicates that the selectivity is not as favorable for Creston and Perdido, consistent with the observations that were reported in Lehne (2009b).

Figure 13-1 Grind Size vs Zn Recovery to Cu/Pb Rougher Concentrate



Several tests included attempts to separate the copper and lead into two concentrates from the final bulk concentrate. The results were not consistent, but they suggest that NaCN is at least partially successful in depressing copper. Lehne (2009a) indicated that liberation is not an issue.

The typical best metallurgical balance from the "overall" composite, from test W-10-16, is shown in Table 13.9 and Table 13.10.

The most effective reagent usage to date is shown in Table 13.11. ICP analyses for trace elements have identified the significant metals in the three flotation concentrates shown in Table 13.12 that will be considered to estimate the concentrates payment conditions.

Table 13.9 Best Metallurgical Balance for Overall Composite (Grades)

Product	Mass %	Au g/t	Ag g/t	Cu %	Pb %	Zn %	Fe %	S %
Pb Conc	1.7	95.9	1007.0	7.9	53.9	8.1	12.4	24.1
Cu Conc	0.7	49.9	1154.0	10.0	4.6	22.0	15.1	26.1
Bulk 1 CC	2.4	82.4	1050.0	8.5	39.4	12.2	13.2	24.7
Bulk 1 CT	2.9	1.2	71.0	0.5	1.2	6.7	8.1	5.5
Bulk RC	5.4	37.6	511.0	1.1	18.4	9.2	10.4	14.1
Zn 2 CC	2.4	5.6	106.4	0.9	0.8	55.3	8.6	33.2
Zn 1/2 CT	4.7	1.9	37.9	0.3	0.6	0.9	18.8	14.9
Zn RC	7.1	3.2	61.3	0.3	0.8	19.5	15.3	21.2
O/A Rougher Conc	12.4							
Rougher Tail	87.6	0.4	4.0	0.0	0.2	0.3	3.8	1.0
Feed - Calc	100.0	2.6	35.2	0.3	1.2	2.1	5.0	3.1

Table 13.10 Best Metallurgical Balance for Overall Composite (Distribution)

Product	Au g/t	Ag g/t	Cu %	Pb %	Zn %	Fe %	S %
Pb Conc	62.2	48.5	46.0	77.0	6.4	4.2	13.0
Cu Conc	13.5	23.2	24.4	2.7	7.3	2.1	5.9
Bulk 1 CC	75.7	71.4	70.4	79.7	13.7	6.4	18.9
Bulk 1 CT	1.3	6.0	5.1	3.0	9.3	4.8	5.2
Bulk RC	77.0	77.7	75.5	82.7	23.0	11.2	24.1
Zn 2 CC	5.2	7.3	7.6	1.6	63.0	4.2	25.7
Zn 1/2 CT	3.4	5.0	4.8	2.5	2.0	17.7	22.2
Zn RC	8.6	12.4	12.4	4.1	65.0	21.9	47.9
O/A Rougher Conc	85.6	90.1	87.9	86.7	88.1	33.0	72.0
Rougher Tail	14.4	9.9	12.1	13.3	11.9	67.0	28.0
Feed - Calc	100.0	100.0	100.0	100.0	100.0	100.0	100.0

Table 13.11 Optimal Reagent Additions from Testwork

Event	Reagent (g/t feed)						
	DF250	3418A	CuSO5H$_2$0	ZnSo$_4$	Ca(OH)$_2$	NaCN	pH
Grind				1000		200	
Bulk Rougher	yes	14					
Zn Conc		4	500		2000		10.5
Zn Rougher							
Bulk Separation	yes	25				500-1000	
Zn Cleaning		10			yes		11.5

Table 13.12 Flotation Concentrate Detailed Assays

Element	Unit	Pb Conc	Cu Conc	Zn Conc
Au	g/t	66	125	4
Ag	g/t	1080	1450	100
Cu	%	10	10	0.7
Pb	%	50	10	0.6
Zn	%	5	18	62
Fe	%	13	18	5
S	%	23	23	31
As	ppm	230	270	70
Sb	ppm	1390	923	100
Hg	ppm	13	84	100
Bi	ppm	780	180	3
Cd	ppm	110	350	1350

The following parameters determined by Westcoast for the overall composite are as follows:

- The optimum primary grind will be P80 of 90 microns or finer.
- The role of regrinding, if there is any requirement at all, is unknown.
- Initial testing to separate the bulk copper / lead concentrate, was partially successful. Although the zinc grade was higher than that of the copper in the “copper concentrate” in test W-10-16 (Hawthorn 2010), it is anticipated that this product will be marketed as a copper concentrate to maximize the revenue from gold and silver.
- Future testing will need to focus on the optimum conditions for copper-lead separation.
- Future testing will need to evaluate the potential to divert the 7% of the zinc that reports to the copper concentrate, into the zinc flotation circuit.
- The overall composite responded reasonably well to the addition of 500 g/t of $ZnSO_4$ and 100 g/t of NaCN, suggesting that blending of fresh and supergene material may be satisfactory in the overall project.
- If El Perdido and to a lesser extent El Creston, represent a large portion of the plant feed at any time, the circuit may benefit from sequential feeding. This observation has not been evaluated, but it does represent a cautionary note for future testing.

The Westcoast recommendation for future testing was as follows:

- Alternative depressants and alternative addition rates in the bulk rougher circuit
- Alternative depressants in the copper-lead separation stage
- The role of finer primary grinding on zinc selectivity in the bulk rougher flotation stage. That may lead to the use of zinc depressants in the bulk cleaner circuit in an attempt to divert more of the zinc to the zinc concentrate.
- The role of regrinding
- Work Index testing
- Processing variability between the “fresh” and the “supergene” zones and the effect of randomly comingling of feed types.

13.2.3 EMIJAMEX TESTING

The following information is taken from the report of that study by Fomento (Rios, Castrejon, and Nieto, 1977a) and its English translation (Rios, Castrejon, and Nieto, 1977b). During Emijamex's exploration at the Sacramento de la Plata mine between 1975 and 1977, they sent a single 150 kg sample for flotation test work. The sample contained sphalerite and galena with minor chalcopyrite in a gangue of quartz, chlorite, hematite, pyrite, and limonite. Geochemical analysis of the sample is listed in table 13.13. According to the translation of the metallurgical report (Rios, Castrejon, and Nieto, 1977b), "The screening of pulverized material through a -65 mesh indicates a degree of recovery of 82.7, 82.8 and 80.5% of free zinc, lead and copper respectively". The report concluded that milling should be between -65 and -100 mesh and that "the studied ore adapts easily to the process of concentration by flotation" (Rios, Castrejon, and Nieto, 1977b).

Table 13.13 El Creston – Perdido Mineable Resource Grades

Au g/t	Ag g/t	Cu%	Pb%	Zn%
3.00	53.00	0.24	3.50	6.40

In 2009 Hawthorn of Westcoast Mineral Testing Inc reviewed the Fomento report and stated that the results of this met test were very encouraging. It was observed that subsequent to the 1970's a number of processing changes have been either adopted or are preferred. These include:

- The zinc sulphate and sodium cyanide combination, although technically effective, is no longer preferred for sphalerite (Sph) depression in lead rougher flotation, having been "replaced" by sulphur dioxide in one of several chemical forms. SO2 is generally as effective as zinc sulphate – cyanide, and its use will eliminate the transport of the significantly more hazardous sodium cyanide. At this stage of the testing, the requirement for a Sph depressant has not been determined in any case, so the first test in any future testing program should be performed without any Sph depressant to investigate the natural partitioning of galena (Ga) and Sph.
- The use of cresylic acid (cresilico) as a frother has been replaced by glycol and MIBC, so it would not be used in any future investigation.
- In the case of the various collectors that were investigated, there are now more recent introductions that will provide improved selectivity.

13.2.4 COMPAÑIA MINERA SACRAMENTO DE LA PLATA HISTORIC CONCENTRATE PRODUCTION

Compañia Minera Sacramento de la Plata constructed a 50 tpd flotation plant to process mined material from El Creston and El Rey and operated in the 1970s for around 2 years.

Concentrates obtained during this period were delivered to the Peñoles smelter at Torreon, Coahuila. Data from the Servicio Geologico Mexicano published in the "Monografia Geologica Minera del Estado de Durango" indicate the concentrates produced in this plant and are shown in Table 13.14.

Table 13.14 Historic Concentrates Production by Compañia Minera Sacramento de la Plata

Year	Au g/ton	Ag g/ton	Pb %	Cu %
1979	90	615	52	0.93
1979	35	754	38	3.05
1980	18	903	41	5.4

13.3 METALLURGICAL TESTWORK REVIEW

For the metallurgic testwork review, it is important to take into consideration that Telson is looking to initiate the exploitation of Tahuehueto in the El Creston – Perdido areas, based on the accessibility and quality of the resource. Table 13.14 shows a summary of the mineable resource defined for the El Creston and El Perdido areas. The metallurgic test performed by CFM comes from a composite taken from channel samples from different exposed old levels mainly in El Creston and El Rey. The Westcoast composite sample comes from core drilling of all the existing areas. This data is shown in Table 13.15.

Table 13.15 El Creston – Perdido Mineable Resource Grades

Au g/t	Ag g/t	Cu%	Pb%	Zn%
4.52	45.96	0.35	1.48	2.90

Table 13.16 Comparative Assays from Composites and Mineable Resources

Area	Au g/t	Ag g/t	Cu %	Pb %	Zn %
El Crestón Cutoff 6 Mineable Resources	5.11	44.88	0.31	1.40	2.96
El Perdido Cutoff 6 Mineable Resources	1.98	50.56	0.51	1.82	2.63
El Crestón – Perdido Cutoff 6 Mineable Resources	4.52	45.96	0.35	1.48	2.90
Composite El Crestón By Drilling Core	2.89	37.00	0.36	1.41	2.11
Composite El Perdido By Drilling Core	1.42	35.00	0.25	1.07	1.50
Crestón – El Rey CFM Sample	3.00	35.00	0.24	3.50	6.40

The assay of the sample used for CFM indicates much higher head grades for Pb and Zn in comparison to the 2009 resource estimated by Telson. However, gold and silver grades were much closer to estimated grades.

The metallurgic investigation performed by Westcoast was based on a sample mainly composited from Creston and Cinco de Mayo areas and these zones show a different mineralogy and operating parameters from one to another.

Table 13.16 shows the parameters of the independent rougher metallurgical test for El Creston and El Perdido areas. The best Westcoast metallurgical balance for overall composite (grades) is indicated in Tables 13.9 and 13.10.

Table 13.17 Westcoast Metallurgic Result from El Crestón and El Perdido Areas

Test	Area	Grind P_{80} (µm)	ZnSO4	NaCN	NaMBS	Rec to Cu/Pb Rougher Conc		
						Cu %	Pb %	Zn %
10-03	Creston	92	1000			60.0	82.4	25.4
10-07	Creston	99	500	100		81.6	87.7	30.6
10-08	Perdido	92	500	100		84.8	93.6	56.0

13.4 TESTWORK ANALYSIS

In order to confirm the previous metallurgical parameters, Telson prepared a new bulk sample from the El Creston area only. The sample was integrated with samples from the different existing levels and in selected zones of the vein that meet the cut off 6 g/t Au of the mineable resource grades in order to have a more accurate representation of the grades that will be mined in a selective way and processed in future mill.

The sample was sequentially reduced to 25 kg for the metallurgic test; the assay of this sample resulted in the highest average of the minable resources and is shown in Table 13.18

Table 13.18 Assay of the Composite Creston Sample to Confirm Previous Results

Au g/t	Ag g/t	Cu%	Pb%	Zn%
7.76	65.0	0.14	4.90	6.58

The mineralogy, mineral characteristics, grindability, along with the other pertinent parameters discussed in the previous tests were considered for inclusion in this test program to prove up and confirm that process parameters will be similar for the ores developed from the El Creston areas with similar recoveries, and marketable concentrates.

Table 13.19 provides the metallurgic results for the Telson confirmation Test 1. This test was carried out using the parameters and reagents defined by Comision de Fomento Minero, confirming that the material is suitable for a flotation process and that it is possible to produce a commercial Bulk Pb–Cu and Zn concentrates. The reagents used in the Test 1 are shown in Table 13.20.

Table 13.19 Metallurgic Test 1 (CFM Parameters)

PRODUCT	MASS	ASSAYS					RECOVERY				
		Au g/t	Ag g/t	Cu %	Pb %	Zn %	Au	Ag	Cu	Pb	Zn
Feed	4000.00	8.66	89.05	0.178	3.89	6.69	100	100	100	100	100
Conc Pb	219.75	96.00	682.50	1.264	56.80	9.40	68.08	47.70	35.47	76.93	7.13
Conc Zn	407.00	3.00	71.70	0.283	0.79	48.80	3.94	9.28	14.71	1.99	68.57
Medios Pb	229.20	17.40	213.80	0.701	7.60	11.57	12.87	15.59	20.52	10.74	9.15
Medios Zn	248.57	3.20	56.80	0.306	2.50	3.09	2.57	4.49	9.71	3.83	2.65
Tails	2895.48	1.34	24.91	0.053	0.37	1.25	12.55	22.94	19.60	6.79	12.49
Feed Calc		7.75	78.60	0.196	4.06	7.242	100	100	100	100	100
Rec Calc							87.45	77.06	80.40	93.49	87.51

Table 13.20 Reagents of Test 1

Reagent	Time	PH	ZnSO4	NaCN	AF-242	CAL	CuSO4	AF-211	X-343	Mibc 70
Molienda #1	30 min	7.5	4 g	0.5 g						
Molienda #2	30 min	7.5	4 g	0.5 g						
Flot PB-CU	10 min	7.5			20g/t					30 g/t
LIMPIA PB-CU#1	2 min	7.5								
FLOT ZN	10 min	11.5						40 g/ton	40 g/t	20 g/t
LIMPIA ZN #1	2 min	11.5								
ACOND PB-CU	5 min	7.5			40 g/t					
ACOND ZN	5 min	11.5				3 g/kg	2 g/kg			

Based on the results of the Test 1, a second test was performed, using the same sample in order to confirm the results of the Test 1 but using innovative reagents like Xantato, promoters AF-404 and AF-242, and canceling the use of NaCN for environmental considerations.

The metallurgical balance of the confirmation Test 2 is indicated in Table 13.21 and Table 13.22 lists the parameter and reagents used in the Test 2. No leaching test were run due to the minimum amount of lead concentrates remaining and available. NaCN will be cancelled with AF 404 and AF 242 promotors

Table 13.21 Metallurgic Test 2

Product	Weight	Assays					Content Tons			Content KG		Recuperations				
		Cu %	Pb %	Zn %	Au g/t	Ag g/t	Cu	Pb	Zn	Au	Ag	Cu	Pb	Zn	Au	Ag
Head	4000	0.178	3.890	6.690	8.660	89.050	7.120	155.600	267.600	34.640	356.200	100	100	100	100	100
Conc Pb	219.75	1.264	56.800	9.400	96.000	682.500	2.778	124.820	20.660	21.100	149.980	35.47	76.93	7.13	68.08	47.7
Conc Zn	407	0.283	0.790	48.800	3.000	71.700	1.152	3.232	198.620	1.220	29.180	14.71	1.99	68.57	3.94	9.28
Medios Pb	229.2	0.701	7.600	11.570	17.400	213.800	1.607	17.419	26.520	3.990	49.000	20.52	10.74	9.15	12.87	15.59
Medios Zn	248.57	0.306	2.500	3.090	3.200	56.800	0.761	6.214	7.680	0.800	14.120	9.7	0.83	2.65	2.57	4.49
Colas	2895.5	0.053	0.370	1.250	1.340	24.910	1.535	10.569	36.190	3.890	72.130	19.6	6.79	12.49	12.55	22.94
Calc Head		0.196	4.060	7.242	7.750	78.600	7.831	162.250	289.670	30.990	314.420	100	100.28	100	100	100
Recoveries Using Calculated Head												**80.4**	**93.49**	**87.51**	**87.45**	**77.06**

Table 13.22 Parameters of Metallurgic Test 2

Parameter		Time	pH	$ZnSO_4$	NaCN	Af-242	Cal	$CuSO_4$	AF-211	X-343	Mibc 70
Molienda #1	2000	30 min	7.5	4 g	0.5 g						
Molienda #2	2000	30 min	7.5	4 g	0.5 g						
Flot Pb-Cu		10 min	7.5			20 g/t					30 g/t
Limpia Pb-Cu#1		2 min	7.5								
Flot Zn		10 min	11.5						40 g/t	40 g/t	20 g/t
Limpia Zn #1		2 min	11.5								
Acond Pb-Cu		5 min	7.5			40 g/t					
Acond Zn		5 min	11.5				3 g/t	2 g/t			

13.5 RECOMMENDATIONS

Based on metallurgical tests performed to date, the following conclusions and recommendations were made for the mill design:

- The previous metallurgical campaigns provide sufficient data to reach a high level of confidence that the floatation process chosen will work, flotation targets are attainable, and an economical concentrate can be produced.
- For the economic analysis and plant design it is recommended to use the result of the last confirmation metallurgical test run by Telson, specifically for the use of new reagents and have extra capacity for processing high grade ores.
- Primary grind the material for processing to under 80% passing 200 mesh (P80 of 100 microns).
- Use reagents recommended by Telson Test 2, indicated in Table 13.22.
- Consider in the mill design the installation of a copper flotation section.
- Run a two more industrial tests of 3,500 to 5,000 tonne from the lower portion of the El Creston zone and the other from Cinco de Mayo Zone to test and determine recoveries for the different types of ore mineralization within those zones.
- Sell the produced concentrates of the industrial test, in order to prove the sales term for the Tahuehueto concentrates.
- If the industrial test result proves an economic benefit and cash generator for Telson, continue processing the material produced from the mine development until the Tahuehueto mill is ready to work.

Based on metallurgical tests performed to date, along with independent mass balance calculations, and production metrics from similar polymetallic milling operations, the following is the recommended distribution percentages for the mill design and economic analysis. The metallurgical recoveries are presented in Table 13.25.

Table 13.23 Tahuehueto Average Metallurgical Recoveries

Product	kTonnes	Distribution % (Recoveries)				
		Au	Ag	Cu	Pb	Zn
Head	3,264	100%	100%	100%	100%	100%
Pb Concentrate	58	77.1%	62.8%	31.6%	85.5%	1.6%
Cu Concentrate	18	6.8%	10.3%	51.4%	0.6%	17.1%
Zn Concentrate	108	11.0%	11.7%	11.5%	6.1%	80.0%
Tails	3,096	5.4%	15.2%	5.4%	7.8%	1.3%

14 MINERAL RESOURCE ESTIMATES

Mineral resources stated for the Tahuehueto Project conform to the definition standards adopted by the Canadian Institute of Mining, Metallurgy and Petroleum (CIM.) Mineral Resources were reported in accordance with the disclosure obligations under NI 43-101. Vulcan Software was used to quantify the Project mineral resources.

Mineral inventories are constrained and reported at appropriate cutoff grades to demonstrate the reasonable prospects of eventual economic extraction.

Mineral Resources for the Project are based on the statistical analysis of data from 248 drill holes totaling 47,276 m and 1,788 underground samples within a model area covering of 2,672 square km. A three dimensional geology model combining structural and stratigraphic units was used to constrain the Mineral Resource Estimate. Four resource models were built to encompass the six mining areas (El Creston, El Perdido, El Catorce, Cinco de Mayo, El Rey, and Santiago). The resource model uses block sizes ranging from 0.5 x 0.5 x 0.5 meters to 2 x 5 x 2 meters depending on geologic continuity and geometry. Resource classification into measured, indicated, and inferred categories was based on estimation variance, distances to nearest drill holes and visually.

14.1 RESOURCE DATABASE

A model was created for estimating the gold, silver, copper, lead, and zinc resources at Tahuehueto from data generated by Telson, including detailed geologic mapping, RC and core drilling data, underground sampling, and project topography derived from one-meter resolution IKONOS imagery. These data were incorporated into a digital database using the UTM Zone 13 NAD27 Mexico coordinate system, and all subsequent modeling of the Tahuehueto resource was performed using Maptek's Vulcan™ Software.

14.2 SOLID BODY MODELING

Various structures provide the primary controls of the mineralization at Tahuehueto. The Cinco de Mayo (including the El Catorce area), El Perdido, and Santiago mineralization lie along a structural zone that strikes 045˚ to 055˚ and dips 60˚ to 80˚ to the southeast. The El Rey deposit lies to the northwest along a subparallel structure striking 060˚ and dipping very steeply to the southeast, and mineralization at El Creston is primarily controlled by a structural zone that strikes 030˚ to 035˚ and dips to the southeast. The mineralization in most of these structures consists more of a zone of irregular veins and veinlets than single, well-defined veins. The strongest and most continuous zones of mineralization generally correlate positively with quartz veining and, in unoxidized zones, increases in sulfide minerals.

Telson provided MMC with a three-dimensional wireframe interpretation of the principal mineralized structures at Tahuehueto. These were used as hard boundaries for the extrapolation of mineralized grades.

The oxidized/partially oxidized zone was delineated from the unoxidized zone by means of a three-dimensional surface created by Telson. MMC used this surface in the modeling of the resources, as discussed below, after checking it against drill-hole oxidation codes in the project database.

There is a relatively minor amount of underground workings within the Cinco de Mayo, El Creston, and El Rey areas. These workings consist primarily of exploration-type drifts along mineralized structural zones and minor crosscuts. Telson reports that there is no stoping in any of the El Creston or Cinco de Mayo workings, but two raises have been developed between levels 9 and 10 in El Creston. There is one raise to surface from the Cinco de Mayo South adit, and there is a small stope in El Rey between levels 3 and 4.

The underground workings were either surveyed by Telson, or more commonly, Telson has digitized plan maps of the workings created by previous operators; the plan maps of the workings are located in UTM space based on Telson's surveying of the portals of the tunnels.

MMC created three-dimensional void wireframes from the data provided by projecting the outlines of the workings in plan 1 m vertically in both directions. The SEWC void wireframe suggests a total of about 62,000 tonnes was mined (assuming a specific gravity of 2.7).

Vertical cross sections oriented orthogonal to the average strike of each mineralized area at Tahuehueto were used to develop a 3-D geologic model. The sections were spaced at 50 m intervals at Cinco de Mayo and 25 m intervals at El Creston, El Perdido, and Santiago; some sections at Cinco de Mayo were skipped due to lack of drill data, leading to occasional 100 m-spaced sections. The drill-hole traces, underground sample data, topographic profile, surface structural mapping data, were all used in the definition of the Tahuehueto geologic model.

The models are meant to capture the gross geological sense of the mineral deposit. In this case the stockwork was modeled and then the veins internal to the stockwork were modeled. The interpretations of the two zones lead the creation of solid geologic models of the mineral body. These solid are used for coding a block model with the different material types and to ensure that geologic controls are used in the grade estimation process.

14.3 EVALUATION OF EXTREME ASSAY VALUES

Log probability plots were evaluated to determine if capping was necessary. Higher grades must be capped if they do not fit the distribution of the mineralization. This ensures that erroneously high grades are not used to overestimate the metal value of the deposit. Table 14.1 lists the grade caps used.

Table 14.1 Capping values

Au g/t	Ag g/t	Cu %	Pb %	Zn %
30	120	None	10	10

14.4 COMPOSITING

Taking consideration for block sizes used in the block model and after a review of core sample lengths, assays were composited to 2 m.

14.5 BLOCK MODEL

The Vulcan™ resource block model for the Tahuehueto Project subdivides the deposit into 2m by 2m by 2m cubed blocks. The contact between stockwork and veins was further subdivided into 0.5m by 0.5m by 0.5m blocks. All of the required information about the deposit is stored in each individual block. This includes estimated characteristics such as gold and silver grades. Statistical characteristics such as kriging variances, number of samples used in an estimate, distances to the nearest drill hole, etc., are also stored in each individual block for descriptive evaluations. Physical information stored in the blocks can include rock types, bulk densities, contained metal and alteration is stored in order to evaluate engineering, production and geotechnical parameters that might be utilized to determine the viability of mining the deposit.

Due to the nature and complexity of the geologic model of Tahuehueto, three separate block models were created to handle the deposit. The model dimensions are listed in Tables 14.2, 14.3 and 14.4.

Table 14.2 Cinco de Mayo/Catorce Model Dimensions

Item	X	Y	Z
Orientation/Bearing	120	30	Vertical
Model Origin	336,115	2,811,450	500
Minimum Block Offset	0	0	0
Maximum Block Offset	1,050	1,350	1,400
Minimum Block Size	0.5	0.5	0.5
Maximum Block Size	2	2	2
Minimum Number of Blocks	525	675	700
Maximum Number of Blocks	2,100	2,700	2,800
Number of Block in Stockworks and Veins			7,184,414

Table 14.3 El Creston/El Perdido Model Dimensions

Item	X	Y	Z
Orientation/Bearing	120	30	Vertical
Model Origin	336,115	2,811,450	500
Minimum Block Offset	0	1,350	0
Maximum Block Offset	1,050	2,450	1,400
Minimum Block Size	0.5	0.5	0.5
Maximum Block Size	2	2	2
Minimum Number of Blocks	525	550	700
Maximum Number of Blocks	2,100	2,700	2,800
Number of Block in Stockworks and Veins			5,124,350

Table 14.4 Santiago Model Dimensions

Item	X	Y	Z
Orientation/Bearing	150	60	Vertical
Model Origin	338,460	2,813,320	1,500
Minimum Block Offset	0	0	0
Maximum Block Offset	200	500	400
Minimum Block Size	0.5	0.5	0.5
Maximum Block Size	2	2	2
Minimum Number of Blocks	100	250	200
Maximum Number of Blocks	200	1000	400
Number of Block in Stockworks and Veins			138,019

14.5.1 EL REY MODELING

El Rey was modeled differently from the other areas due to the unique form of the mineralization, which consists of two discreet, thin, well-defined structures that are suggestive of classic veins; and the paucity of drill data, which led to the use of underground sampling as in the interpolation of grades. Underground workings at El Rey include drifts along the mineralized structures at several elevations, and these drifts have been systematically sampled across the structures at approximately four-meter intervals along strike. Two drill holes pierce the main structure below the workings. Vein solids were constructed from polygons interpreted on vertical cross sections at four-meter spacing. These solids were then used to constrain the estimation of mineralization at El Rey. El Rey Model statistics are shown in Table 14.5.

Table 14.5 El Rey Model Dimensions

Item	X	Y	Z
Orientation/Bearing	150	60	Vertical
Model Origin	337,060	2,812,920	1,200
Minimum Block Offset	0	0	0
Maximum Block Offset	150	350	450
Minimum Block Size	0.5	0.5	0.5
Maximum Block Size	2	2	2
Minimum Number of Blocks	75	175	225
Maximum Number of Blocks	30	700	900
Number of Block in Stockworks and Veins			45,151

14.6 RESOURCE ESTIMATION METHODOLOGY

Gold grade was estimated using Inverses Distance estimation techniques. The general procedure for creation of the gold-grade model was as follows:

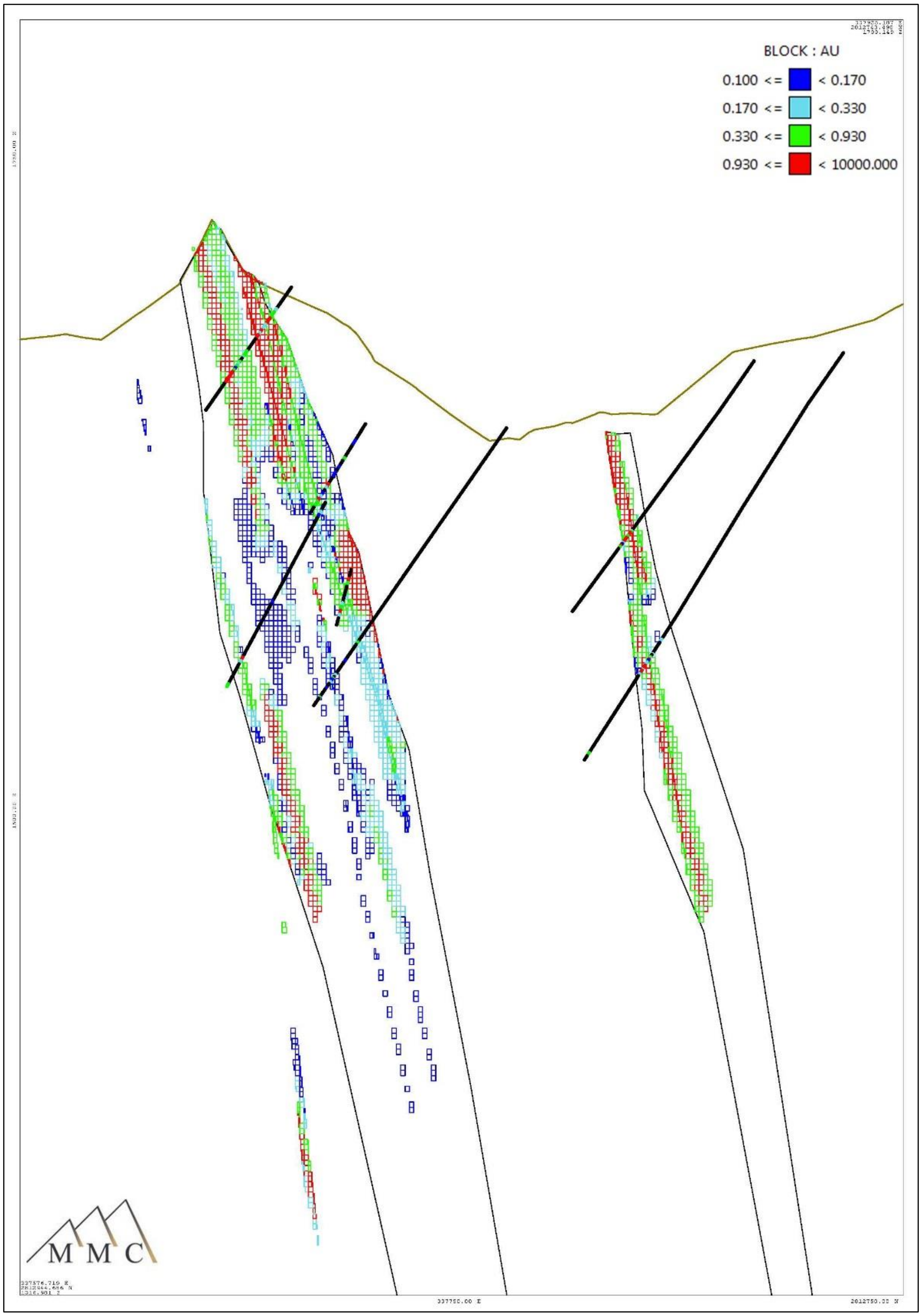
- The major axis of the search ellipse was oriented at the same angles as the geologic structures of each deposit.
- A composite had to be a minimum of 1 m, or half the SMU, in order to be used in the grade estimation run.
- Gold, Silver, Copper, Lead and Zinc were all estimated using the same parameters.

The grade estimation parameters are listed in Table 14.6. Figure 14-1 shows a typical section through Creston and El Perdido.

Table 14.6 Tahuehueto Estimation Parameters

Deposit	Rotations about Z Axis	Rotations about X Axis	Rotations about Y Axis	Major Axis Lengt	Semi Major Axis Length	Minor Axis Length	Minimum Samples	Maximum Samples	Maxmum Samples per Hole
Cinco de Mayo	120	-60	0	100	50	10	1	7	2
Catorce	120	-60	0	100	50	10	1	7	2
El Creston	115	-75	0	100	50	10	1	7	2
El Perdido	140	-75	0	100	50	10	1	7	2
Santiago	140	-75	0	100	50	10	1	7	2
El Rey	120	-85	0	100	50	10	1	7	2

Figure 14-1 Cross Section through Section 3075 Looking 30 Northeast



14.7 MINERAL RESOURCE CLASSIFICATION

The mineral resources in this report were estimated in accordance with the definitions contained in the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards on Mineral Resources and Reserves Definitions and Guidelines that were prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council.

14.8 MINERAL RESOURCE ESTIMATE

The Mineral Resource has been limited to mineralized material that occurs within the mineralized blocks and which could be scheduled to be processed based on the defined cut-off grade. All other material was characterized as non-mineralized material. The current mineral resource block models remain unchanged from the 2009 mineral resource estimate.

The "reasonable prospects for eventual economic extraction" requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at an appropriate cut-off grade that takes into account extraction scenarios and processing recoveries. The deposit gold mineralization is amenable for extraction by conventional underground mining methods. In order to determine the quantities of material offering "reasonable prospects for eventual economic extraction" for mining, MMC determined an internal economic cut-off grade based on gold equivalence. Metal values were calculated to a gold price per gram basis to be added combined with gold grades for a gold equivalent (AuEq) grade. The calculations and metal prices used to determine AuEq grades are listed in Tables 14.7 and 14.8, respectively.

Table 14.7 Gold Equivalence Grade Calculations

Gold Equivalence Grade Calculations
Ag Value = (Ag gpt*Ag Price)/Au Price
Cu Value = ((((Cu%/100)*sg*volume)*1,000,000)*Cu Price)/Au Price/(sg*volume)
Pb Value = ((((Pb%/100)*sg*volume)*1,000,000)*Pb Price)/Au Price/(sg*volume)
Zn Value = ((((Zn%/100)*sg*volume)*1,000,000)*Zn Price)/Au Price/(sg*volume)
AuEq gpt = Au gpt + Ag Value + Cu Value + Pb Value + Zn Value

Table 14.8 Metal Prices Used for Mineral Resource Estimates

Metal	Units	Value
Gold	$/oz	1,200
Silver	$/oz	20.00
Copper	$/lb	3.10
Lead	$/lb	0.90
Zinc	$/lb	0.97

The AuEq cut-off grade used is based on the financial model metal price of $1,180/oz gold, which is below the 3 year trailing average and is close to spot prices as of the effective date (November 18, 2016) of this report.

The Mineral Resources for the Tahuehueo Project are reported at an AuEq grade of 2.5 g/t. Table 14.9 lists the current Mineral Resource Estimate by classification and mineralized inventory for the Project.

Table 14.9 Tahuehueto Project Measured, Indicated, and Inferred Mineral Resource Estimate

Classification	kTonnes	Au Grade (gpt)	Contained Au kOz	Ag Grade (gpt)	Contained Ag kOz	Cu Grade (%)	Contained Cu klbs	Pb Grade (%)	Contained Pb klbs	Zn Grade (%)	Contained Zn klbs
Total Measured	2,771	2.77	247	44.70	3,982	0.31	18,914	1.27	77,827	2.29	139,821
Total Indicated	3,343	2.23	240	41.26	4,435	0.30	22,466	1.15	84,455	2.04	155,687
Total Measured and Indicated	**6,114**	**2.48**	**487**	**42.82**	**8,417**	**0.31**	**41,380**	**1.20**	**162,282**	**2.15**	**295,508**
Total Inferred	3,501	1.31	147	37.59	4,230	0.27	20,469	1.34	103,080	2.44	188,409

Mineral resources are inclusive of mineral reserves. Mineral resources that are not mineral reserves do not have demonstrated economic viability. Mineral resource estimates do not account for mineability, selectivity, mining loss and dilution. These mineral resource estimates include inferred mineral resources that are normally considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. There is also no certainty that these inferred mineral resources will be converted to measured and indicated categories through further drilling, or into mineral reserves, once economic considerations are applied.

Sections 14.8.1 through 14.8.6 includes tables of Mineral Resource Estimates by classification and mineralized inventory for the different deposits: El Creston, El Perdido, El Catorce, Cinco de Mayo, El Rey, and Santiago.

14.8.1 EL CRESTON MINERAL RESOURCES

The following Mineral Resources are reported at an AuEq grade of 2.5 g/t. Table 14.10 lists the deposits Mineral Resource estimate. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

Table 14.10 El Creston Measured, Indicated, and Inferred Resources

Classification	kTonnes	Au Grade (gpt)	Contained Au kOz	Ag Grade (gpt)	Contained Ag kOz	Cu Grade (%)	Contained Cu klbs	Pb Grade (%)	Contained Pb klbs	Zn Grade (%)	Contained Zn klbs
Total Measured	1,664	3.40	182	41.09	2,198	0.28	10,272	1.19	43,655	2.28	83,642
Total Indicated	1,594	2.89	148	38.73	1,985	0.27	9,489	0.98	34,442	1.93	67,831
Total Measured and Indicated	**3,258**	**3.15**	**330**	**39.94**	**4,183**	**0.28**	**19,761**	**1.09**	**78,097**	**2.11**	**151,472**
Total Inferred	768	2.14	53	40.32	996	0.30	5,080	0.90	15,240	1.97	33,359

14.8.2 EL PERDIDO MINERAL RESOURCES

The following Mineral Resources are reported at an AuEq grade of 2.5 g/t. Table 14.11 lists the deposits Mineral Resource estimate. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

Table 14.11 El Perdido Measured, Indicated, and Inferred Resources

Classification	kTonnes	Au Grade (gpt)	Contained Au kOz	Ag Grade (gpt)	Contained Ag kOz	Cu Grade (%)	Contained Cu klbs	Pb Grade (%)	Contained Pb klbs	Zn Grade (%)	Contained Zn klbs
Total Measured	351	1.65	19	44.64	504	0.41	3,172	1.39	10,755	1.84	14,237
Total Indicated	484	1.50	23	42.32	658	0.40	4,265	1.22	13,009	1.59	16,954
Total Measured and Indicated	**835**	**1.56**	**42**	**43.30**	**1,162**	**0.40**	**7,438**	**1.29**	**23,764**	**1.70**	**31,191**
Total Inferred	443	1.46	21	40.51	577	0.43	4,201	1.14	11,139	1.82	17,783

14.8.3 EL CATORCE MINERAL RESOURCES

The following Mineral Resources are reported at an AuEq grade of 2.5 g/t. Table 14.12 lists the deposits Mineral Resource estimate. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

Table 14.12 El Catorce Measured, Indicated, and Inferred Resources

Classification	kTonnes	Au Grade (gpt)	Contained Au kOz	Ag Grade (gpt)	Contained Ag kOz	Cu Grade (%)	Contained Cu klbs	Pb Grade (%)	Contained Pb klbs	Zn Grade (%)	Contained Zn klbs
Total Measured	301	1.63	16	47.39	458	0.12	796	1.08	7,161	2.36	15,648
Total Indicated	643	1.48	31	38.95	806	0.16	2,270	1.21	17,163	2.68	38,014
Total Measured and Indicated	**944**	**1.53**	**46**	**41.64**	**1,264**	**0.15**	**3,065**	**1.17**	**24,324**	**2.58**	**53,663**
Total Inferred	1,604	0.86	44	31.53	1,626	0.16	5,659	1.36	48,101	2.81	99,385

14.8.4 CINCO DE MAYO MINERAL RESOURCES

The following Mineral Resources are reported at an AuEq grade of 2.5 g/t. Table 14.13 lists the deposits Mineral Resource estimate. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

Table 14.13 Cinco de Mayo Measured, Indicated, and Inferred Resources

Classification	kTonnes	Au Grade (gpt)	Contained Au kOz	Ag Grade (gpt)	Contained Ag kOz	Cu Grade (%)	Contained Cu klbs	Pb Grade (%)	Contained Pb klbs	Zn Grade (%)	Contained Zn klbs
Total Measured	327	2.20	23	50.67	533	0.51	3,681	1.27	9,165	2.26	16,310
Total Indicated	544	1.96	34	47.85	836	0.50	5,993	1.36	16,300	2.30	27,566
Total Measured and Indicated	**871**	**2.05**	**57**	**48.91**	**1,370**	**0.50**	**9,673**	**1.33**	**25,465**	**2.28**	**43,875**
Total Inferred	590	1.37	26	43.65	829	0.39	5,076	1.83	23,820	2.33	30,328

14.8.5 EL REY MINERAL RESOURCES

The following Mineral Resources are reported at an AuEq grade of 2.5 g/t. Table 14.14 lists the deposits Mineral Resource estimate. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

Table 14.14 El Rey Measured, Indicated, and Inferred Resources

Classification	kTonnes	Au Grade (gpt)	Contained Au kOz	Ag Grade (gpt)	Contained Ag kOz	Cu Grade (%)	Contained Cu klbs	Pb Grade (%)	Contained Pb klbs	Zn Grade (%)	Contained Zn klbs
Total Measured	99	1.23	4	77.12	245	0.18	391	3.09	6,719	4.30	9,350
Total Indicated	58	1.00	2	69.42	130	0.18	232	2.63	3,384	0.00	5,070
Total Measured and Indicated	**157**	**1.14**	**6**	**74.26**	**375**	**0.18**	**623**	**2.92**	**10,103**	**2.70**	**14,420**
Total Inferred	87	0.86	2	70.26	197	0.20	385	2.46	4,730	3.89	7,480

14.8.6 SANTIAGO MINERAL RESOURCES

The following Mineral Resources are reported at an AuEq grade of 2.5 g/t. Table 14.15 lists the deposits Mineral Resource estimate. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

Table 14.15 Santiago Measured, Indicated, and Inferred Resources

Classification	kTonnes	Au Grade (gpt)	Contained Au kOz	Ag Grade (gpt)	Contained Ag kOz	Cu Grade (%)	Contained Cu klbs	Pb Grade (%)	Contained Pb klbs	Zn Grade (%)	Contained Zn klbs
Total Measured	29	3.59	3	47.13	44	0.94	603	0.58	372	0.99	635
Total Indicated	20	2.84	2	30.74	20	0.50	218	0.36	157	0.58	253
Total Measured and Indicated	**49**	**3.29**	**5**	**40.49**	**64**	**0.76**	**821**	**0.49**	**529**	**0.82**	**888**
Total Inferred	7	2.39	1	23.53	6	0.42	68	0.31	50	0.46	74

14.9 GRADE SENSITIVITY ANALYSIS

The mineral resources are highly sensitive to reporting cut-off grade. To illustrate this sensitivity, the block model quantities and grade estimates are presented at various cut-off grades by resource classification by deposit in Tables 14.16 through 14.33. The reader is cautioned that these figures should not be misconstrued as a mineral resource. The reported quantities and grades are only presented as a sensitivity of the resource models to the selection of cut-off grade.

Table 14.16 El Creston Measured Resource by Cut-Off

AuEq (gpt)	kTonnes	Au Grade (gpt)	Contained Au kOz	Ag Grade (gpt)	Contained Ag kOz	Cu Grade (%)	Contained Cu klbs	Pb Grade (%)	Contained Pb klbs	Zn Grade (%)	Contained Zn klbs
2	5,143	2.36	409	35.74	5,915	0.25	27,897	0.92	107,177	1.84	211,516
3	3,293	3.22	360	43.16	4,570	0.30	21,639	1.12	83,601	2.25	165,505
4	2,285	4.10	321	47.25	3,484	0.34	16,768	1.26	65,368	2.61	131,626
5	1,668	4.99	286	49.57	2,689	0.35	12,733	1.41	52,650	2.95	106,903
6	1,220	6.11	253	53.55	2,106	0.39	9,951	1.51	41,546	3.09	82,505
7	942	6.99	226	56.30	1,710	0.40	8,037	1.57	33,629	3.24	67,138
8	681	8.71	198	56.17	1,245	0.41	5,657	1.52	24,490	3.15	49,216
9	554	9.74	181	56.53	1,024	0.40	4,660	1.58	20,387	3.24	40,290
10	473	10.58	167	57.01	888	0.36	3,842	1.66	17,937	3.36	34,812

Table 14.17 El Creston Indicated Resource by Cut-Off

AuEq (gpt)	kTonnes	Au Grade (gpt)	Contained Au kOz	Ag Grade (gpt)	Contained Ag kOz	Cu Grade (%)	Contained Cu klbs	Pb Grade (%)	Contained Pb klbs	Zn Grade (%)	Contained Zn klbs
2	2,042	2.42	159	34.66	2,276	0.24	10,806	0.89	40,073	1.74	78,344
3	1,278	3.36	138	41.87	1,720	0.29	8,168	1.08	30,418	2.12	59,710
4	878	4.32	122	46.17	1,303	0.32	6,194	1.21	23,420	2.41	46,647
5	640	5.25	108	49.00	1,008	0.34	4,797	1.32	18,625	2.64	37,251
6	463	6.33	94	51.98	774	0.36	3,678	1.41	14,404	2.77	28,298
7	344	7.45	82	54.58	604	0.37	2,806	1.47	11,149	2.88	21,843
8	246	9.02	71	54.61	432	0.35	1,899	1.47	7,975	2.91	15,787
9	206	9.92	66	54.74	362	0.35	1,588	1.50	6,804	2.88	13,064
10	176	10.62	60	55.46	313	0.34	1,317	1.56	6,041	2.90	11,230

Table 14.18 El Creston Inferred Resource by Cut-Off

AuEq (gpt)	kTonnes	Au Grade (gpt)	Contained Au kOz	Ag Grade (gpt)	Contained Ag kOz	Cu Grade (%)	Contained Cu klbs	Pb Grade (%)	Contained Pb klbs	Zn Grade (%)	Contained Zn klbs
2	1,052	1.76	60	35.60	1,204	0.25	5,799	0.79	18,325	1.73	40,129
3	627	2.40	48	43.35	875	0.31	4,288	0.99	13,695	2.16	29,879
4	385	3.11	38	46.87	580	0.37	3,138	1.12	9,500	2.66	22,562
5	254	3.84	31	47.92	391	0.35	1,957	1.35	7,548	3.17	17,724
6	150	5.13	25	53.75	260	0.44	1,460	1.45	4,810	3.24	10,747
7	124	5.56	22	56.60	225	0.46	1,253	1.49	4,060	3.41	9,291
8	69	7.79	17	55.12	123	0.51	779	1.27	1,939	2.96	4,518
9	53	8.73	15	54.92	94	0.46	539	1.39	1,628	3.22	3,772
10	41	9.71	13	54.66	72	0.35	315	1.57	1,413	3.60	3,241

Table 14.19 El Perdido Measured Resource by Cut-Off

AuEq (gpt)	kTonnes	Au Grade (gpt)	Contained Au kOz	Ag Grade (gpt)	Contained Ag kOz	Cu Grade (%)	Contained Cu klbs	Pb Grade (%)	Contained Pb klbs	Zn Grade (%)	Contained Zn klbs
2	460	1.42	21	39.56	585	0.35	3,548	1.20	12,165	1.62	16,423
3	295	1.81	17	48.05	455	0.44	2,858	1.50	9,742	1.97	12,795
4	204	2.09	14	54.30	357	0.51	2,298	1.79	8,065	2.29	10,318
5	140	2.43	11	60.04	271	0.60	1,857	1.93	5,973	2.40	7,427
6	84	2.71	7	71.60	194	0.76	1,409	2.31	4,281	2.35	4,356
7	45	2.97	4	87.97	128	0.95	946	2.80	2,788	2.41	2,400
8	31	3.23	3	97.22	96	1.03	696	3.02	2,041	2.44	1,649
9	24	3.50	3	105.22	80	1.09	569	2.88	1,503	2.31	1,205
10	12	3.51	1	108.10	43	1.19	324	2.99	814	2.32	631

Table 14.20 El Perdido Indicated Resource by Cut-Off

AuEq (gpt)	kTonnes	Au Grade (gpt)	Contained Au kOz	Ag Grade (gpt)	Contained Ag kOz	Cu Grade (%)	Contained Cu klbs	Pb Grade (%)	Contained Pb klbs	Zn Grade (%)	Contained Zn klbs
2	665	1.27	27	37.37	799	0.35	5,131	1.05	15,392	1.39	20,376
3	385	1.68	21	45.58	564	0.43	3,648	1.33	11,283	1.74	14,762
4	242	2.02	16	51.35	400	0.50	2,673	1.61	8,607	2.04	10,906
5	157	2.34	12	56.97	288	0.57	1,977	1.74	6,035	2.15	7,457
6	77	2.58	6	70.84	176	0.73	1,243	2.17	3,694	2.20	3,745
7	37	3.00	4	85.44	102	0.85	695	2.48	2,029	2.23	1,824
8	21	3.47	2	101.61	67	0.99	450	2.46	1,119	2.09	951
9	15	3.84	2	108.15	51	1.03	334	2.16	699	1.98	641
10	2	3.40	0	107.12	6	1.08	44	3.06	125	2.39	98

Table 14.21 El Perdido Inferred Resource by Cut-Off

AuEq (gpt)	kTonnes	Au Grade (gpt)	Contained Au kOz	Ag Grade (gpt)	Contained Ag kOz	Cu Grade (%)	Contained Cu klbs	Pb Grade (%)	Contained Pb klbs	Zn Grade (%)	Contained Zn klbs
2	607	1.25	24	37.37	730	0.37	4,953	0.98	13,119	1.49	19,946
3	380	1.56	19	41.81	511	0.45	3,772	1.18	9,891	2.01	16,847
4	238	1.98	15	47.42	363	0.51	2,676	1.42	7,451	2.26	11,859
5	160	2.25	12	51.12	263	0.54	1,908	1.57	5,549	2.44	8,623
6	81	2.41	6	59.94	156	0.63	1,125	1.67	2,981	2.60	4,641
7	22	3.53	3	84.44	60	0.88	431	1.14	559	1.51	740
8	8	4.28	1	104.08	28	0.92	171	1.13	210	1.51	280
9	5	4.50	1	109.80	18	0.97	108	1.07	119	1.51	168
10	0	4.17	0	120.00	0	1.79	5	1.06	3	1.43	4

Table 14.22 El Catorce Measured Resource by Cut-Off

AuEq (gpt)	kTonnes	Au Grade (gpt)	Contained Au kOz	Ag Grade (gpt)	Contained Ag kOz	Cu Grade (%)	Contained Cu klbs	Pb Grade (%)	Contained Pb klbs	Zn Grade (%)	Contained Zn klbs
2	382	1.39	17	42.43	521	0.10	842	1.02	8,583	2.17	18,261
3	204	2.25	15	54.40	356	0.15	673	0.92	4,130	2.64	11,851
4	97	3.90	12	62.46	195	0.21	450	0.83	1,778	3.04	6,513
5	52	6.01	10	75.72	127	0.33	381	0.94	1,085	3.05	3,519
6	27	9.34	8	72.15	62	0.53	311	1.02	598	4.59	2,692
7	22	10.75	8	69.45	50	0.48	236	0.91	448	5.22	2,570
8	20	11.49	7	72.28	46	0.52	229	0.88	387	5.54	2,437
9	17	12.67	7	77.34	42	0.59	221	0.78	292	5.91	2,210
10	16	13.16	7	78.95	40	0.63	218	0.79	273	6.25	2,158

Table 14.23 El Catorce Indicated Resource by Cut-Off

AuEq (gpt)	kTonnes	Au Grade (gpt)	Contained Au kOz	Ag Grade (gpt)	Contained Ag kOz	Cu Grade (%)	Contained Cu klbs	Pb Grade (%)	Contained Pb klbs	Zn Grade (%)	Contained Zn klbs
2	841	1.24	34	34.90	944	0.13	2,411	1.11	20,585	2.42	44,878
3	471	1.89	29	42.12	637	0.19	1,972	1.14	11,830	2.99	31,027
4	239	2.98	23	49.27	379	0.27	1,425	1.36	7,176	3.45	18,203
5	140	4.06	18	57.41	259	0.40	1,238	1.70	5,263	3.64	11,268
6	87	5.58	16	56.08	157	0.44	847	1.65	3,175	4.37	8,410
7	65	7.09	15	51.34	107	0.31	441	1.46	2,077	5.10	7,255
8	45	8.80	13	58.71	85	0.39	386	1.21	1,197	5.34	5,280
9	30	11.25	11	69.59	67	0.54	355	0.84	552	5.71	3,751
10	27	11.79	10	72.09	64	0.58	351	0.77	466	5.79	3,506

Table 14.24 El Catorce Inferred Resource by Cut-Off

AuEq (gpt)	kTonnes	Au Grade (gpt)	Contained Au kOz	Ag Grade (gpt)	Contained Ag kOz	Cu Grade (%)	Contained Cu klbs	Pb Grade (%)	Contained Pb klbs	Zn Grade (%)	Contained Zn klbs
2	2,079	0.76	51	28.17	1,883	0.13	5,958	1.23	56,369	2.57	117,779
3	1,213	1.03	40	32.56	1,269	0.19	5,079	1.30	34,751	3.15	84,204
4	516	1.65	27	45.63	757	0.27	3,072	1.73	19,683	3.19	36,295
5	292	1.90	18	55.31	519	0.39	2,509	2.07	13,316	3.24	20,843
6	115	2.71	10	50.45	186	0.45	1,139	2.16	5,468	4.05	10,252
7	68	3.82	8	33.92	74	0.13	194	2.15	3,205	5.58	8,319
8	43	4.54	6	37.48	52	0.16	152	2.02	1,920	5.40	5,134
9	10	8.73	3	60.96	20	0.43	97	0.86	193	4.55	1,022
10	8	9.35	3	61.69	17	0.44	81	0.86	159	4.95	917

Table 14.25 Cinco de Mayo Measured Resource by Cut-Off

AuEq (gpt)	kTonnes	Au Grade (gpt)	Contained Au kOz	Ag Grade (gpt)	Contained Ag kOz	Cu Grade (%)	Contained Cu klbs	Pb Grade (%)	Contained Pb klbs	Zn Grade (%)	Contained Zn klbs
2	382	1.39	17	42.43	521	0.10	842	1.02	8,583	2.17	18,261
3	204	2.25	15	54.40	356	0.15	673	0.92	4,130	2.64	11,851
4	97	3.90	12	62.46	195	0.21	450	0.83	1,778	3.04	6,513
5	52	6.01	10	75.72	127	0.33	381	0.94	1,085	3.05	3,519
6	27	9.34	8	72.15	62	0.53	311	1.02	598	4.59	2,692
7	22	10.75	8	69.45	50	0.48	236	0.91	448	5.22	2,570
8	20	11.49	7	72.28	46	0.52	229	0.88	387	5.54	2,437
9	17	12.67	7	77.34	42	0.59	221	0.78	292	5.91	2,210
10	16	13.16	7	78.95	40	0.63	218	0.79	273	6.25	2,158

Table 14.26 Cinco de Mayo Indicated Resource by Cut-Off

AuEq (gpt)	kTonnes	Au Grade (gpt)	Contained Au kOz	Ag Grade (gpt)	Contained Ag kOz	Cu Grade (%)	Contained Cu klbs	Pb Grade (%)	Contained Pb klbs	Zn Grade (%)	Contained Zn klbs
2	841	1.24	34	34.90	944	0.13	2,411	1.11	20,585	2.42	44,878
3	471	1.89	29	42.12	637	0.19	1,972	1.14	11,830	2.99	31,027
4	239	2.98	23	49.27	379	0.27	1,425	1.36	7,176	3.45	18,203
5	140	4.06	18	57.41	259	0.40	1,238	1.70	5,263	3.64	11,268
6	87	5.58	16	56.08	157	0.44	847	1.65	3,175	4.37	8,410
7	65	7.09	15	51.34	107	0.31	441	1.46	2,077	5.10	7,255
8	45	8.80	13	58.71	85	0.39	386	1.21	1,197	5.34	5,280
9	30	11.25	11	69.59	67	0.54	355	0.84	552	5.71	3,751
10	27	11.79	10	72.09	64	0.58	351	0.77	466	5.79	3,506

Table 14.27 Cinco de Mayo Inferred Resource by Cut-Off

AuEq (gpt)	kTonnes	Au Grade (gpt)	Contained Au kOz	Ag Grade (gpt)	Contained Ag kOz	Cu Grade (%)	Contained Cu klbs	Pb Grade (%)	Contained Pb klbs	Zn Grade (%)	Contained Zn klbs
2	2,079	0.76	51	28.17	1,883	0.13	5,958	1.23	56,369	2.57	117,779
3	1,213	1.03	40	32.56	1,269	0.19	5,079	1.30	34,751	3.15	84,204
4	516	1.65	27	45.63	757	0.27	3,072	1.73	19,683	3.19	36,295
5	292	1.90	18	55.31	519	0.39	2,509	2.07	13,316	3.24	20,843
6	115	2.71	10	50.45	186	0.45	1,139	2.16	5,468	4.05	10,252
7	68	3.82	8	33.92	74	0.13	194	2.15	3,205	5.58	8,319
8	43	4.54	6	37.48	52	0.16	152	2.02	1,920	5.40	5,134
9	10	8.73	3	60.96	20	0.43	97	0.86	193	4.55	1,022
10	8	9.35	3	61.69	17	0.44	81	0.86	159	4.95	917

Table 14.28 El Rey Measured Resource by Cut-Off

AuEq (gpt)	kTonnes	Au Grade (gpt)	Contained Au kOz	Ag Grade (gpt)	Contained Ag kOz	Cu Grade (%)	Contained Cu klbs	Pb Grade (%)	Contained Pb klbs	Zn Grade (%)	Contained Zn klbs
2	107,963	1.15	3,992	73.81	256,202	0.17	404,631	2.93	6,973,928	4.01	9,544,523
3	93,051	1.28	3,829	78.91	236,072	0.18	369,257	3.19	6,544,048	4.47	9,169,873
4	83,451	1.37	3,676	81.95	219,873	0.20	367,956	3.35	6,163,267	4.77	8,775,756
5	70,990	1.48	3,378	85.73	195,669	0.21	328,663	3.62	5,665,529	5.11	7,997,473
6	58,647	1.62	3,055	89.06	167,926	0.23	297,377	3.87	5,003,691	5.44	7,033,612
7	44,339	1.82	2,594	91.55	130,506	0.24	234,600	4.22	4,125,049	5.83	5,698,824
8	31,334	2.02	2,035	95.79	96,500	0.25	172,698	4.62	3,191,466	6.27	4,331,276
9	22,935	2.22	1,637	97.89	72,181	0.26	131,463	4.88	2,467,462	6.60	3,337,141
10	13,401	2.34	1,008	100.58	43,336	0.26	76,817	5.40	1,595,429	7.14	2,109,511

Table 14.29 El Rey Indicated Resource by Cut-Off

AuEq (gpt)	kTonnes	Au Grade (gpt)	Contained Au kOz	Ag Grade (gpt)	Contained Ag kOz	Cu Grade (%)	Contained Cu klbs	Pb Grade (%)	Contained Pb klbs	Zn Grade (%)	Contained Zn klbs
2	61,854	0.96	1,909	67.12	133,477	0.17	231,818	2.57	3,504,549	3.78	5,154,551
3	54,333	1.05	1,834	71.66	125,180	0.18	215,612	2.72	3,258,136	4.12	4,935,118
4	46,611	1.14	1,708	75.91	113,756	0.20	205,518	2.93	3,010,833	4.43	4,552,216
5	36,015	1.26	1,459	80.76	93,513	0.22	174,679	3.30	2,620,187	4.85	3,850,880
6	26,105	1.44	1,209	86.11	72,270	0.23	132,366	3.67	2,112,108	5.29	3,044,428
7	17,748	1.62	924	88.06	50,247	0.25	97,816	3.95	1,545,498	5.94	2,324,116
8	10,662	1.96	672	93.39	32,013	0.26	61,114	4.48	1,053,049	6.40	1,504,355
9	8,240	2.12	562	95.45	25,287	0.28	50,865	4.57	830,196	6.69	1,215,320
10	4,628	2.37	353	96.15	14,306	0.28	28,566	4.85	494,812	7.13	727,425

Table 14.30 El Rey Inferred Resource by Cut-Off

AuEq (gpt)	kTonnes	Au Grade (gpt)	Contained Au kOz	Ag Grade (gpt)	Contained Ag kOz	Cu Grade (%)	Contained Cu klbs	Pb Grade (%)	Contained Pb klbs	Zn Grade (%)	Contained Zn klbs
2	92,340	0.82	2,434	67.81	201,313	0.19	386,790	2.41	4,906,129	3.74	7,613,661
3	82,867	0.88	2,345	71.82	191,346	0.20	365,382	2.50	4,567,281	4.02	7,344,188
4	72,678	0.94	2,196	75.66	176,792	0.22	352,502	2.67	4,278,091	4.23	6,777,650
5	54,155	0.99	1,724	82.53	143,694	0.25	298,477	3.01	3,593,662	4.66	5,563,609
6	40,058	1.07	1,378	88.70	114,236	0.27	238,444	3.21	2,834,835	5.03	4,442,125
7	21,010	1.12	757	91.79	62,002	0.33	152,852	3.49	1,616,525	5.95	2,755,966
8	9,987	1.30	417	95.38	30,625	0.34	74,859	4.59	1,010,594	6.65	1,464,150
9	8,013	1.36	350	96.44	24,844	0.37	65,360	4.67	824,950	6.88	1,215,344
10	3,662	1.56	184	95.30	11,219	0.35	28,254	4.79	386,676	7.14	576,382

Table 14.31 Santiago Measured Resource by Cut-Off

AuEq (gpt)	kTonnes	Au Grade (gpt)	Contained Au kOz	Ag Grade (gpt)	Contained Ag kOz	Cu Grade (%)	Contained Cu klbs	Pb Grade (%)	Contained Pb klbs	Zn Grade (%)	Contained Zn klbs
2	40,327	3.03	3,929	37.18	48,205	0.69	613,450	0.47	417,857	0.85	755,699
3	22,013	4.16	2,944	57.84	40,936	1.21	587,227	0.68	330,012	1.09	528,989
4	16,607	4.63	2,472	71.16	37,995	1.57	574,818	0.87	318,530	1.35	494,271
5	14,148	4.84	2,202	78.87	35,876	1.82	567,681	0.99	308,794	1.54	480,346
6	12,184	5.06	1,982	82.85	32,455	2.06	553,345	1.10	295,475	1.71	459,330
7	11,326	5.11	1,861	85.74	31,221	2.19	546,833	1.15	287,150	1.80	449,452
8	10,471	5.21	1,754	87.47	29,445	2.29	528,614	1.19	274,694	1.87	431,663
9	9,641	5.23	1,621	89.31	27,683	2.41	512,243	1.24	263,561	1.94	412,345
10	8,000	5.54	1,425	92.21	23,717	2.55	449,740	1.29	227,515	1.98	349,210

Table 14.32 Santiago Indicated Resource by Cut-Off

AuEq (gpt)	kTonnes	Au Grade (gpt)	Contained Au kOz	Ag Grade (gpt)	Contained Ag kOz	Cu Grade (%)	Contained Cu klbs	Pb Grade (%)	Contained Pb klbs	Zn Grade (%)	Contained Zn klbs
2	31,193	2.40	2,407	24.02	24,089	0.34	233,810	0.29	199,427	0.49	336,962
3	13,339	3.21	1,377	39.48	16,931	0.71	208,789	0.45	132,331	0.67	197,027
4	8,695	3.60	1,006	49.54	13,849	0.99	189,777	0.60	115,016	0.90	172,524
5	5,719	3.86	710	58.82	10,814	1.37	172,719	0.78	98,336	1.17	147,504
6	4,446	4.02	575	64.11	9,165	1.61	157,820	0.87	85,282	1.33	130,373
7	3,609	4.19	486	68.05	7,895	1.77	140,818	0.94	74,784	1.46	116,155
8	2,529	4.46	363	74.48	6,055	2.05	114,284	1.04	57,978	1.60	89,197
9	2,065	4.69	311	77.18	5,124	2.15	97,884	1.08	49,170	1.65	75,120
10	1,560	4.90	246	80.24	4,024	2.30	79,092	1.12	38,514	1.68	57,772

Table 14.33 Santiago Inferred Resource by Cut-Off

AuEq (gpt)	kTonnes	Au Grade (gpt)	Contained Au kOz	Ag Grade (gpt)	Contained Ag kOz	Cu Grade (%)	Contained Cu klbs	Pb Grade (%)	Contained Pb klbs	Zn Grade (%)	Contained Zn klbs
2	13,446	2.13	921	16.84	7,280	0.26	77,071	0.22	65,214	0.32	94,857
3	4,124	2.67	354	33.72	4,471	0.62	56,366	0.45	40,911	0.66	60,002
4	2,578	3.19	264	37.38	3,098	0.71	40,352	0.53	30,122	0.77	43,762
5	1,510	4.35	211	37.58	1,825	0.57	18,979	0.60	19,978	0.78	25,971
6	918	5.54	163	36.54	1,078	0.36	7,284	0.65	13,152	0.77	15,580
7	636	6.20	127	35.26	721	0.22	3,085	0.68	9,536	0.77	10,798
8	305	6.62	65	36.80	361	0.24	1,615	0.72	4,844	0.83	5,584
9	78	7.20	18	38.95	97	0.26	446	0.78	1,338	0.90	1,544
10	3	7.83	1	41.53	4	0.28	17	0.85	53	0.97	60

15 MINERAL RESERVE ESTIMATE

15.1 INTRODUCTION

The mine design and Mineral Reserve estimate have been completed to a level appropriate for pre-feasibility studies. The Mineral Reserve estimate stated herein is consistent with the CIM Standards on Mineral Resources and Mineral Reserves and is suitable for public reporting. As such, the Mineral Reserves are based on Measured and Indicated Resources, and do not include any Inferred Resources.

15.2 MINERALIZED ZONES

The Tahuehueto Resource Block Models were used in conjunction with the Stope Optimizer module within Maptek Vulcan™ software to identify mineable shapes within six mineralized vein systems in the four Tahuehueto resource block models. In this approach, the resource model blocks have attributes of volume, density, and metal grades for gold, silver, lead, zinc and copper. Other inputs for the analysis were the estimated production unit cost ($62.39 per tonne) and a Net Smelter Recovery model that calculates the economic value of the block based on input parameters of metal recovery, smelter tariffs, royalties and metal price assumptions. The software outlined mineable volumes considering input parameters of vertical stope interval (5 m), minimum mining width (3 m) and mining continuity, and tests production cost against NSR economic value. The model developed stope geometries on 5 m vertical intervals. Mineable shapes were defined in six of the veins at Tahuehueto; El Creston, El Perdido, El Catorce, El Rey, Cinco de Mayo and Santiago.

The NSR calculations coded to the resource model and used as inputs for the Stope Optimizer runs are not the same calculations as the NSR used for the economic analysis of the project. Simplified operating costs and freight and refining costs were used to develop these NSR values when compared to the detailed economic analysis as detailed in Section 22 and 23. The NSR was based on the inputs and calculations presented in Tables 15.1 through 15.5.

Table 15.1 Metal Prices Used for NSR Calculations

Metal	Units	Value
Gold	$/oz	1,000
Silver	$/oz	15.50
Copper	$/lb	2.10
Lead	$/lb	0.60
Zinc	$/lb	0.75

Table 15.2 Lead Concentrate NSR Input Parameters

Item	Au	Ag	Cu	Pb	Zn
Distribution % (Recoveries)	73.00%	71.00%	9.00%	82.00%	9.00%
Payable %	95.00%	95.00%	0.00%	95.00%	0.00%
Penalties $/%	$3 per % of Zn% > 3%				
Treatment Charge $/t	215.00				

Table 15.3 Copper Concentrate NSR Input Parameters

Item	Au	Ag	Cu	Pb	Zn
Distribution % (Recoveries)	12.00%	11.00%	75.00%	6.00%	4.00%
Payable %	95.00%	95.00%	97.00%	0.00%	0.00%
Penalties $/%	$3 per % of (Pb% + Zn%) > 3%				
Treatment Charge $/t	180.00				

Table 15.4 Zinc Concentrate NSR Input Parameters

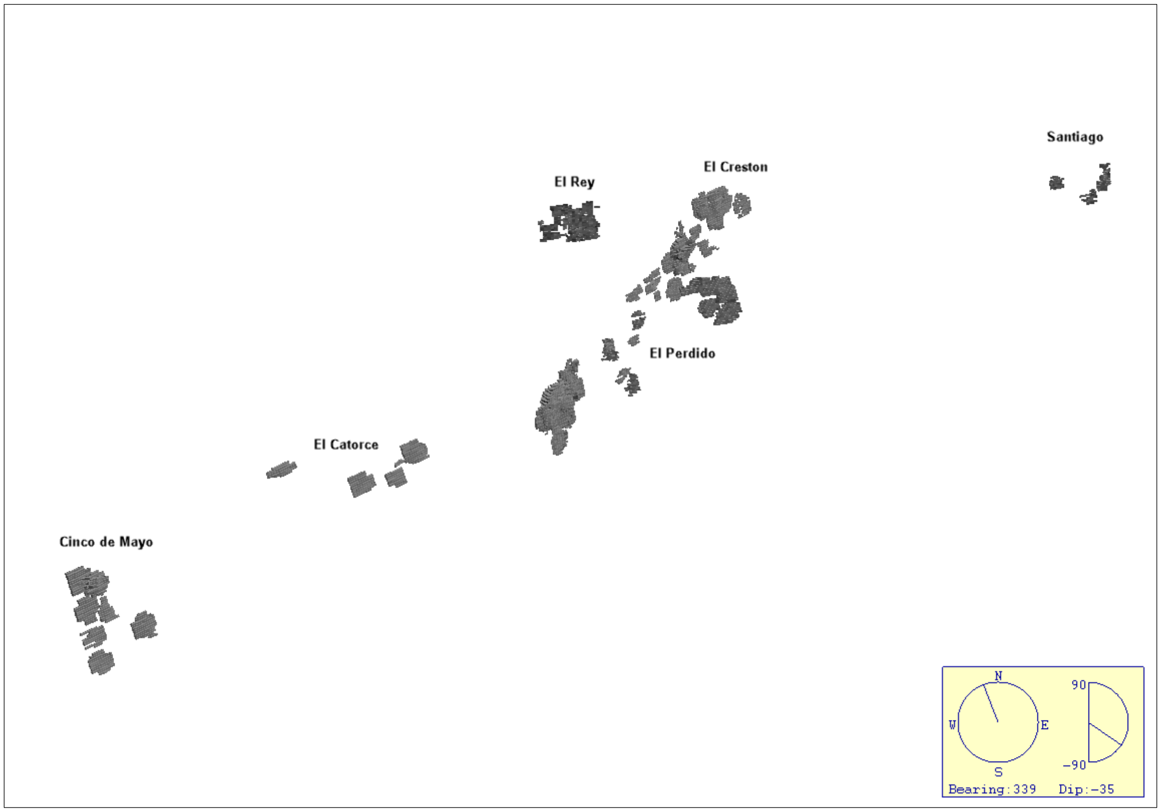
Item	Au	Ag	Cu	Pb	Zn
Distribution % (Recoveries)	2.00%	5.00%	3.00%	2.00%	76.00%
Payable %	80.00%	70.00%	0.00%	0.00%	85.00%
Penalties $/%	-				
Treatment Charge $/t	250.00				

Table 15.5 NSR Calculations

Item	Calculation
Value $/t	Cont. Metal/t * Dist. % * Payable % * Metal Price - Penalties - Treatment Charge
Value $	Value $/t * t concentrate
Value Added Tax (VAT) $	Value $ * 16.0%
Net Smelter Return (NSR) $	Value $ - VAT $

Mining shapes defined by the Stope Optimizer software have vertical and horizontal continuity, and are illustrated in Figure 15-1 for the six veins. Output for each 5 m high stope shape consisted of volume, tonnage, and density for the Measured and Indicated portion of the individual stope shapes. The graphical capabilities of the Vulcan™ design software were used measure the length (along the vein strike direction) of each 5 meter high stope shape. The data were then tabulated in an inventory spreadsheet, and an average stope width was calculated for each stope shape from tonnage, stope height (5 m), stope length and stope rock density.

Figure 15-1 Stope Shapes for the Six Veins Based On Economic Testing with the Stope Optimiser



15.2.1 EXTRACTION RATIO AND DILUTION ASSUMPTIONS

A group of design rules were developed for the individual 5 m stope shapes to adjust the contained Measured and Indicated Resource for the effects of mining dilution and extraction ratio. The rules were applied to individual 5 m high stope shapes and calculated a volume of dilution material with zero (0) metal grade that was equivalent to a minimum 0.25 m thick zone along the both stope walls (stope strike direction). If the average stope width was less than 3 m, it was set to 3 m to calculate the diluting volume. Diluting material was assumed to have the average density of the stope. An extraction ratio was calculated based on the average stope width, to account for the current uncertainty in geotechnical data and lack of rock mechanics analysis. The extraction ratio ranged from 100% for average stope widths less than or equal to 6 m, and gradually reduced to approximately 55% for the largest stope average widths. Details of the dilution and extraction ratio rules are described in Section 16.

For each 5 m high stope shape, dilution tonnage at zero (0) grade was calculated for the original stope dimensions and added to the original stope tonnage. The extraction ratio was then applied to calculate the total mined ore tonnage. The extraction ratio was also applied to the original stope metal content, and the diluted metal grades were calculated as the extracted metal content divided by the total mined tonnage.

Mining dilution averaged 15% of the original mining shape tonnage as defined by the Stope Optimizer and the extraction ratio averaged 94%.

The resulting dilution tonnage and extraction ratio for each of the six vein areas are listed in Table 15.6.

Table 15.6 Resource Model Ore Tonnage, Dilution Tonnage, and Average Extraction Ratio for The Tahuehueto Scheduled Production

Item	El Creston	El Perdido	El Catorce	Cinco de Mayo	El Rey	Santiago	Total
Model kTonnes	1,836	337	481	216	103	33	**3,006**
Dilution kTonnes	164	42	83	29	124	28	**470**
Average Extraction Ratio (%)	89.60	99.65	99.50	100.00	99.94	100.00	**93.89**
Mined kTonnes	1,792	377	560	245	227	62	**3,264**

15.3 TAHUEHUETO RESERVES

A mine design was created in the Vulcan™ model to define access and mining of the stope shapes defined by the Stope Optimizer. The resulting design is shown in Figure 15-2. The defined stope shapes and development excavations were scheduled to produce a basis for economic analysis. The resulting reserve is classified as Probable, and is listed in Table 15.7. No Proven Reserves were defined due to the limited definition resource drilling, limited definition by exploratory mining and the lack of geotechnical data that addresses underground mining.

Figure 15-2 Tahuehueto Reserve Mining Shapes and Developments (Looking North, Not to Scale)

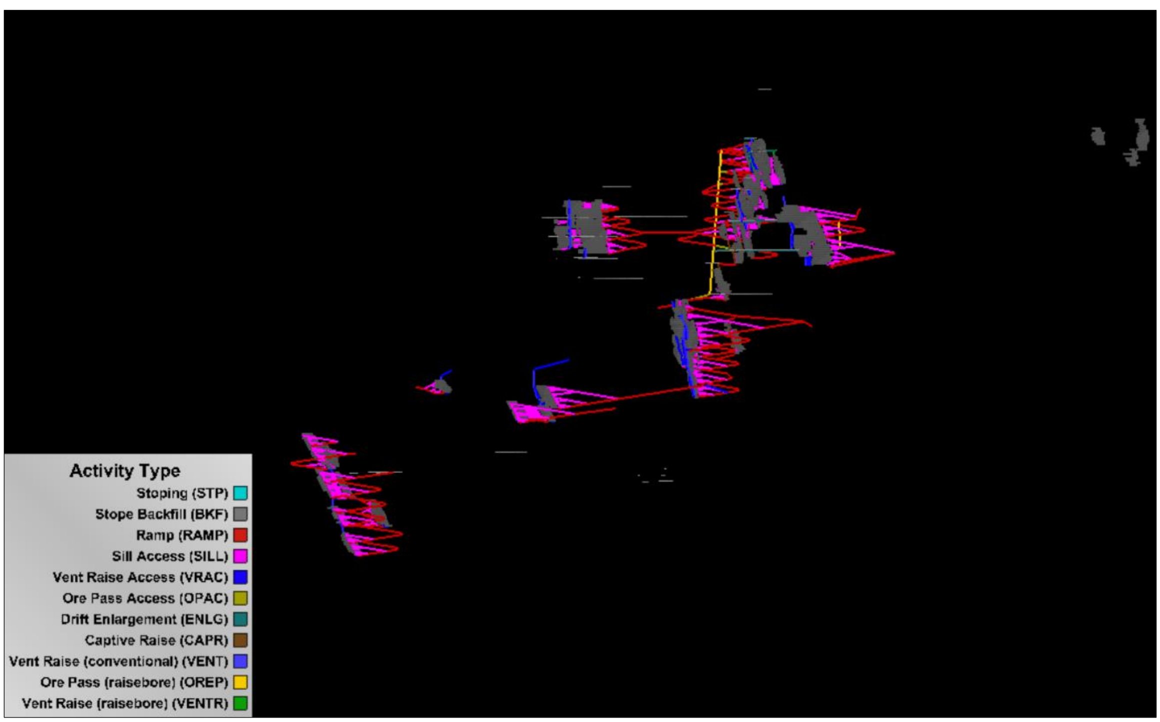


Table 15.7 Tahuehueto Probable Reserves Listed By Vein Zone

Probable	kTonnes	Au Grade (gpt)	Contained Au kOz	Ag Grade (gpt)	Contained Ag kOz	Cu Grade (%)	Contained Cu klbs	Pb Grade (%)	Contained Pb klbs	Zn Grade (%)	Contained Zn klbs
El Creston	1,792	4.18	241	39.16	2,256	0.29	11,371	1.23	48,601	2.32	91,458.72
El Perdido	377	2.02	24	44.63	541	0.53	4,439	1.31	10,867	1.55	12,912.35
El Catorce	560	2.77	50	46.45	837	0.56	6,966	1.03	12,738	2.47	30,473.97
Cinco de Mayo	245	4.08	32	51.70	408	0.16	880	0.79	4,292	2.49	13,488.96
El Rey	227	0.65	5	39.56	289	0.10	524	1.75	8,780	2.44	12,223.19
Santiago	62	2.02	4	28.00	56	0.62	849	0.35	484	0.55	756.93
Total Probable	**3,264**	**3.40**	**356**	**41.57**	**4,387**	**0.35**	**25,028**	**1.19**	**85,762**	**2.24**	**161,314**

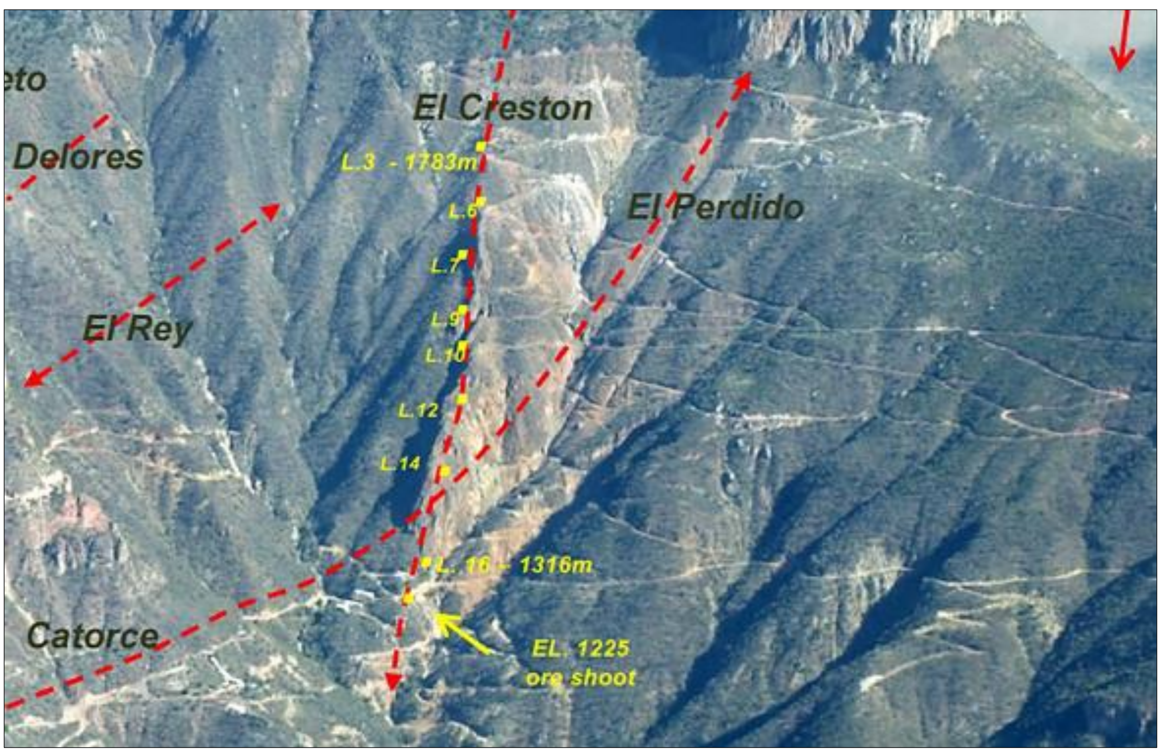
1. Canadian Institute of Mining, Metallurgy and Petroleum standards were followed in the estimation of the Mineral Reserves.
2. Mineral Reserves are estimated using metal price forecasts of $0.60/lb for lead, $0.75/lb for zinc, $2.10/lb for copper, $1,000/oz for gold and $19.12/oz for silver.
3. Totals may not add due to rounding.
4. The foregoing mineral reserves are based upon and are included within the current mineral resource estimate for the Project.

16 MINING METHODS

Mineralization at Tahuehueto occurs in different veins under a local mountainous landform as shown in Figure 16-1. The Figure 16-1 photograph shows the mountain terrain, the network of surface roads developed to reach various underground portal sites, and the locations of some of the surface projections of mineralized veins. Test mining was performed by Compania Minera Sacramento de la Plata in 1971 on the El Creston and El Rey vein structures. A total of 700 m primarily of development drift excavations and some stope type excavations were constructed from different levels along the mountain slope (identified as levels L.3-L.16 in Figure 16-1). The portions of the mineralized vein structures that passed the economic test applied using the Vulcan Stope Optimiser™ were:

- El Creston
- El Perdido
- El Catorce
- Cinco de Mayo
- El Rey
- Santiago

Figure 16-1 Photograph of the Mountainous Topography at Tahuehueto (Looking North East) With Projected Surface Traces of Some of the Mineralized Vein Structures and Locations of Previous Mining Developments (L.3 –L.16)



The near vertical dip of the veins and apparent rock mass quality, as demonstrated by the excavations formed by the previous mining, makes the veins suitable for different sublevel mining methods. No trade-off studies have been performed and the Telson Management had based previous mining studies on the

Overhand Cut-and-Fill mining method, which in the Author's opinion was suitable for the purposes of this PFS study.

The mining method used as the basis of this PFS design was, therefore, Overhand–Cut and–Fill mining with conventional drilling, blasting, mucking and hauling, scaling and ground support installation and backfilling with unconsolidated, mined waste materials. Full mechanization was assumed using diesel, rubber tired mining equipment and support vehicles.

16.1 GEOTECHNICAL ASSUMPTIONS

No rock mechanics studies have been performed to support the mine design and layout and no geotechnical data was available as part of this study. In the Author's opinion, the apparent rock stability indicated by the previous test mining is sufficient as a preliminary design basis, but confirming rock mechanics and geotechnical studies should be performed to verify the mining approach and details of the PFS layout and design before construction.

Early test mining at Tahuehueto was performed using typical drift dimensions of 2.0 m x 2.0 m and conventional drill and blast mining methods. In general, the mining excavations had no rock support installed during construction. Figure 16-2 is a recent photograph of the portal of one of the previously excavated drift excavations, reflecting strong rock and generally good local rock conditions at the portal access. No rock bolting or surface support system was employed for the excavations, and the excavations are reported by Telson to remain in generally good condition.

Figure 16-2 Photograph of the Rock Conditions in the Portal Excavation at Level 10

In 2016, the pre-existing excavation at Level 10 was enlarged to 5 m x 5 m to produce the mill test sample discussed in Sections 13.5 and 24 of this report. A photograph of the rock conditions in the enlarged drift is shown in Figure 16-3, and indicated generally good rock conditions at the excavated dimensions. No rock bolting or surface support was installed inside the excavation during or after construction. Chain-link mesh had been installed to protect the entryway from loose rock on the exterior portal face, however, no rock bolts or other surface support was installed in the rock surfaces during enlargement of the drift to 5 m x 5 m. Figure 16-4 is photograph showing rock conditions in the interior of the enlarged drift beyond the portal area.

Figure 16-3 Photograph Showing Rock Conditions at the Recently Enlarged Portal Area at Level 10

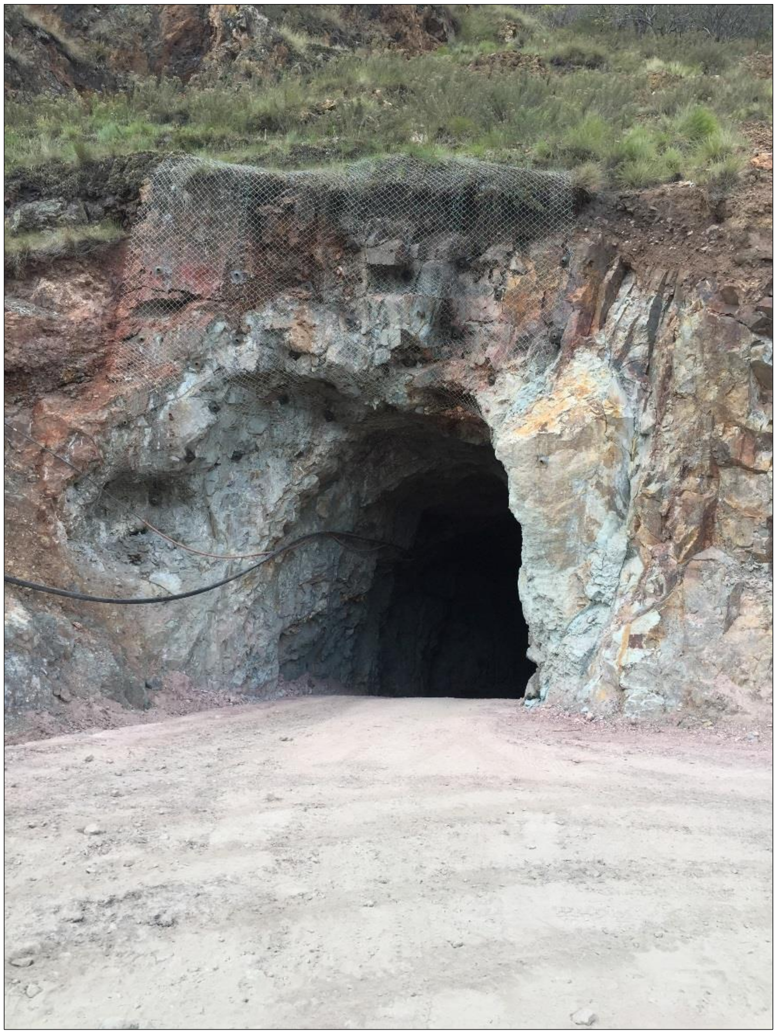


Figure 16-4 Exposed Rock Conditions within the Enlarged Portal Area at Level 10

A few production mining excavations were created in the early test mining, and the spans have measured in level maps that have been imported into CAD. The maximum dimensions created, according to these maps, had a square foot print in plan measuring 14 m x 14 m.

16.1.1 EMPIRICAL GEOTECHNICAL DESIGN ASSUMPTIONS

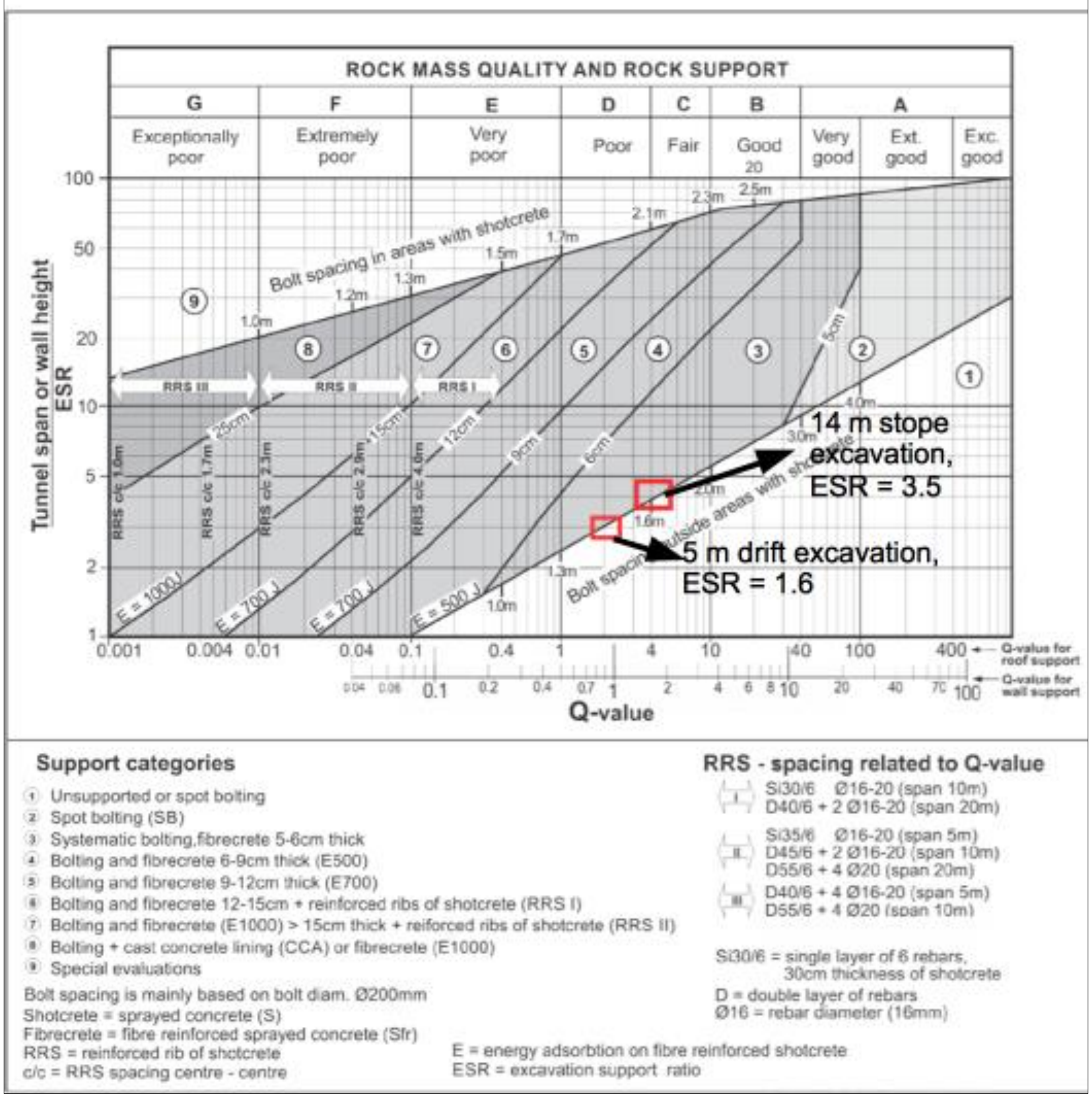
An empirical design approach from the Norwegian Geotechnical Institute (NGI – Q System (Barton, 1974)) has been used to estimate the rock mass quality indicated by the Tahuehueto existing mining excavations. In this approach, a rock mass quality parameter (Q) is correlated with a dimension index (Equivalent Span) and also correlated with ground support installed as indicated by case history data for tunnel excavations used in the system. The estimated rock mass quality based on the dimensions and performance of the existing excavations that have been constructed at Tahuehueto are shown in the design chart in Figure 16-5 (Barton, 1974). The design points are plotted at the boundary between the "No Support Required", Region 1, and the regions requiring support.

In Figure 16-5, the maximum span produced in existing production mining at Tahuehueto was approximately 14 m square, which assuming a ESR value of 3.5 for temporary mining excavations, gives an Equivalent Span of 4 m. The design point for standard development drifts assumed for the Tahuehueto PFS of 5 m spans, adjusted for an ESR value of 1.6 for permanent mine openings would be 3.1 m. The indicated range in Q for stable excavations without support or spot bolting would be Q =2-4, or relative rock quality of “poor”.

The geotechnical design for Tahuehueto was guided by the mining excavation dimensions indicated by the design chart in Figure 16-5, with the additional condition that systematic rock support would be applied on a proportion of the excavated roof area based on the excavation span and by adjustment of the extraction ratio based on the average width of mineralization on each 5 m vertical lift.

Figure 16-5 NGI Q-System Empirical Tunnel Support Design System showing Maximum Size Stope Excavation from Tahuehueto Existing Mining and PFS Drift Dimensions

Type or use of underground opening	ESR
Temporary mine openings	3.5
Vertical shafts, rectangular and circular respectively	2.0 - 2.5
Water tunnels, permanent mine openings, adits, drifts	1.6
Storage caverns, road tunnels with little traffic, access tunnels, etc.	1.3
Power stations, road and railway tunnels with heavy traffic, civil defence shelters, etc.	1.0
Nuclear power plants, railroad stations, sport arenas, etc.	0.8



16.1.2 EXTRACTION RATIO ADJUSTMENT

The mining method of overhand cut-and-fill requires that operating personnel are exposed to the rock surface in the stope excavation at all times during mining excavation. The mining sequence begins at the base of a stope block by development of a drift (nominally 5 m span by 5 m high) in mineralization along the entire local strike length of the mineralization. The stope mining would then proceed vertically upward in 2.5 m increments using slabbing of the overlying mineralization. If the stope length increased as the mining proceed upward, the length would be increased by extending the drift length on the particular level. The minimum mining width was assumed to be 3 m, so diluting material would be added to increase any mining width under 3 m. For all other mineralization widths, it was assumed that 0.25 m of waste material (at zero (0) grade) would be extracted on both walls (ribs) of the stope advance.

An extraction ratio was calculated based on the width (span) of the stope for 5 m vertical lifts. The extraction ratio was based on a transition of the stope from full stope width (for mining widths up to 6 m), to stopes with 5 m long (along stope strike) alcoves between 5 m strike pillars. The alcove was assumed to be excavated to the mineralization boundary perpendicular to the stope strike axis. If the average width of the stope was greater than 6 m, the extraction ratio was calculated assuming a 5 m strike drift, with 5 m alcove cuts perpendicular to strike between 5 m rib pillars. Histograms showing the resulting distribution of the stope widths and stope extraction ratios are presented in Figure 16-6. Figure 16-7 shows that nominally 84% of the individual 5 m stope lifts were less than 6 m wide, and that the extraction ratio was adjusted below 100% in approximately 10% of the individual stope lifts.

Figure 16-6 Distribution of Average Stope Widths for Tahuehueto Vulcan Stope Optimizer Output

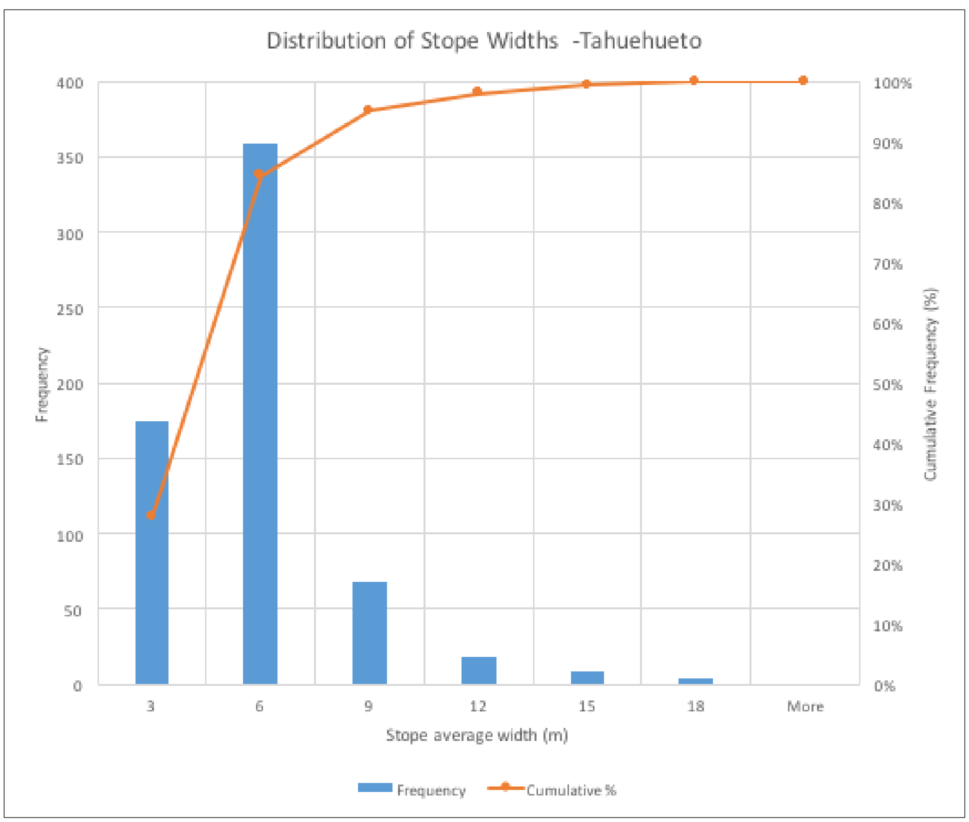
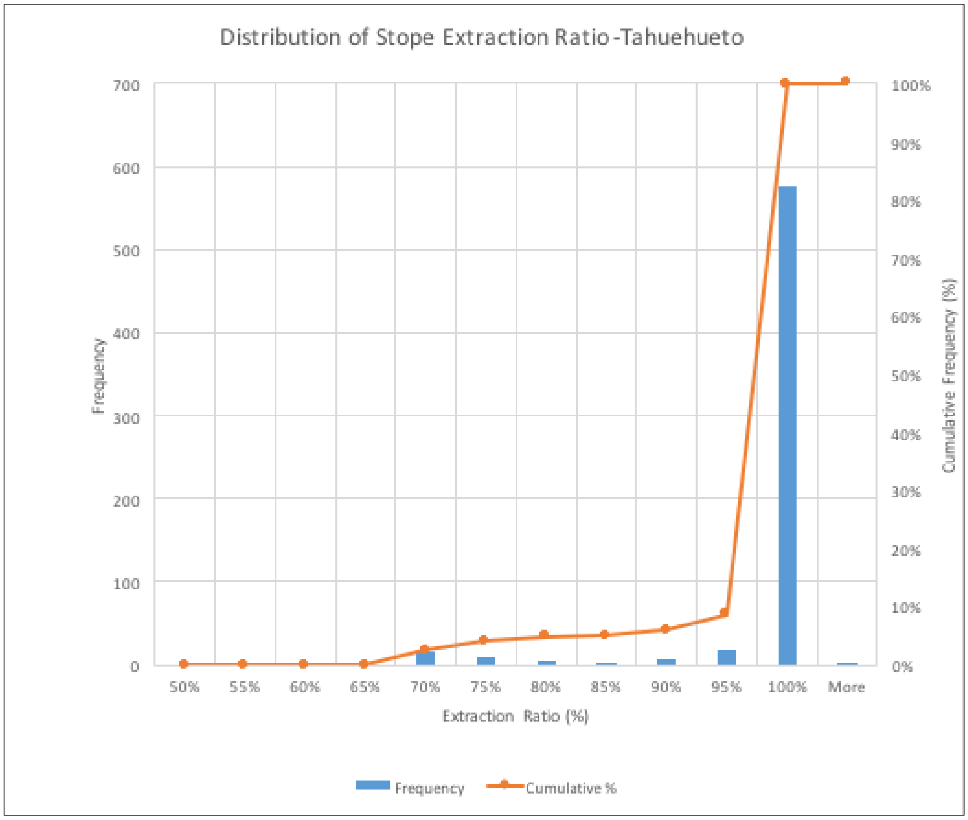


Figure 16-7 Distribution of Calculated Stope Extraction Ratios



16.1.3 GROUND SUPPORT ASSUMPTIONS

The rock stability in the existing Tahuehueto excavations of limited extent was not assumed to be representative of the conditions that will occur with the progress of vertical overhand stoping at Tahuehueto, or of the more extensive drift development required for the mine development. Ground support in the form of 2.4 m friction rock stabilizers (rock bolts) on a 1.3 m pattern, with welded wire mesh and steel straps was assumed to be required on a specified portion of the development drifts and stope excavations. The proportion of the roof surface requiring rock support was estimated based on stope or excavation width using the following rule:

- if the average stope or drift width was less than 5 m, 35% of the roof area was assumed to require rock bolts, mesh and steel straps;
- if the average stope or drift width was greater than 5 m but less than 8 m, 40% of the roof area was assumed to require rock bolts, wire mesh and steel straps;
- and if the average stope with or drift width was greater than 8 m, 100% of the roof area was assumed to require rock bolts, wire mesh and steel straps.

In the operating assumptions used for cost, manpower and equipment, ground support was assumed to be installed after each blast event (either drift round or slab round). The ground support operation would consist of inspection and scaling of the freshly blasted area during and after loading (mucking) the blasted material. Scaling was assumed to be performed with the LHD bucket, at assumed rates (m^2/hr) of the surface area roof and sidewalls.

Vertical progression of the stoping overhead or of stope accesses in waste by slabbing would expose freshly blasted rock surfaces, which were also assumed to require scaling and rock support installation according to the width based support rules.

16.2 WASTE BACKFILLING

As production mining proceeded overhead, existing mined void was assumed to be filled with blasted waste materials to provide a working platform for the next overhead lift. Blasted waste was assumed to have a swell factor of 50%. Mined waste material was assumed to be produced by development and stope access drifting in waste, vertical raise mining for ventilation and ore pass development, and as slabbed material produced from the roof in the stope access crosscuts as they migrate from -15% (minus) slope to a +15% slope as the stoping in mineralization progressed vertically upward. Another source of blasted waste was assumed to be produced locally where stope widths were less than 5 m. At these locations, the stope was assumed to be widened to 5 m width by blasting waste material after selective mining of the mineralized material.

All of the blasted waste material was assumed to be loaded into underground haul trucks and transported to the portal stockpile area. From the portal stockpiles, waste materials would be reloaded into surface haul trucks, then hauled to central stockpiles assumed to be located within 1500 m of the portal areas. Surface stockpiled waste material was then assumed to be rehandled into haul trucks, taken back to the required portal area, and then reloaded into underground haul trucks for transport into the stopes for backfilling. Backfill would be dumped at the required point, then distributed by LHD. A 10% volume reduction due to settlement upon final placement was assumed.

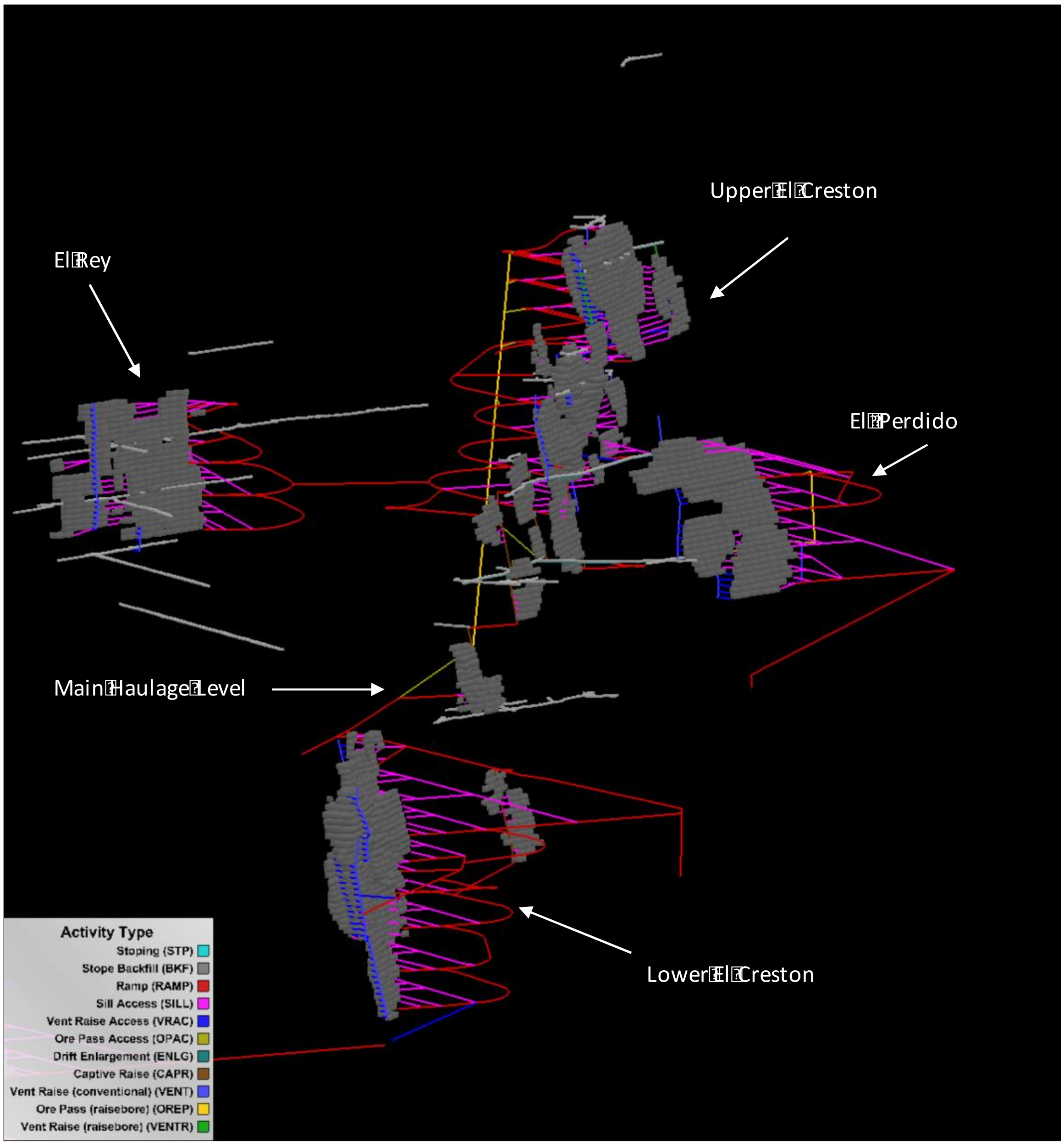
Over the life of mine assumed in the PFS, the volume of mined waste was sufficient to backfill the stope void with the assumed swell factor of 50%. This material balance was favorable because the average density of the mined mineralization was approximately 4.0 t/m^3, whereas the assumed density of the waste materials was 2.75 t/m^3.

16.3 MINING LAYOUT

The mining layout is illustrated in Figure 16-8, which is an isometric view of the mining and development layout in the El Creston, El Perdido and El Ray veins. Individual stope blocks, defined by the Vulcan Stope Optimizer™, are shown as groupings of 5 m high shapes, with long axis along the vein strike. These stope shapes have attributes of length, volume and average density. The length along strike of each stope shape is determined in Vulcan and is then used in conjunction with the shape volume and average density to calculate the average stope width.

The development layout in the Vulcan model is described by strings (lines), no volumes are assigned to these strings in the Vulcan model, only length and inclination are assigned as attributes. Cross sectional areas and density were assigned to the stings in spreadsheet inventories of the strings to allow calculation of waste tonnes. Main development drifts are shown as red lines and provide the routes to get equipment from the surface to the stope excavations. Pink lines are used to denote access crosscut ramps that allow the equipment to gain entry to the stopes, and are assumed to be slabbed vertically between (minus) -15% and +15% to maintain access to mineralization as the stoping proceeds from the local bottom to the local top of the stoping block.

Figure 16-8 Isometric View of Tahuehueto Underground Layout in the Area of El Creston, El Perdido and El Rey



Connection to the surface is via existing excavations from, previous mining (shown as light grey) or by new portals excavated at various locations on the mountain side. The location of the new portals is governed by the existing road network on the mountain side.

For upper El Creston, El Perdido and El Rey, mined mineralization would be hauled by 20 tonne capacity underground haul trucks from the stopes to a central ore pass (shown as yellow) and dropped to a bin at the bottom of the ore pass at a main haulage level which is at the general elevation of the mill and tails facility. For Lower El Creston, El Catroce and Santiago, the mineralization would be hauled by underground truck to the portal access stockpile and the re-handled to surface haul trucks for transport to the mill.

Ventilation air would be drawn into the mine from surface portals (either existing mined excavations or newly developed ramps) and into the stopes. Ventilation raises (blue lines) would be excavated vertically along the stope blocks and would be connected between the main ventilation levels (intake and exhaust) before production could begin in the stope block. Because of the length of the ventilation raises, twin raises (1.6 m x 1.6 m) would be constructed at each location, with a short ventilation access drift connecting to the stope. The ventilation access drift roof would be slabbed to maintain connection to the stope as the mining proceeds overhead. The ventilation raises would provide conduits for electrical power, compressed air, make up water into the stope blocks, and a secondary escape way ladder network would also be constructed. The ventilation raises would be backfilled to keep progress with the overhead progress of the stope mining.

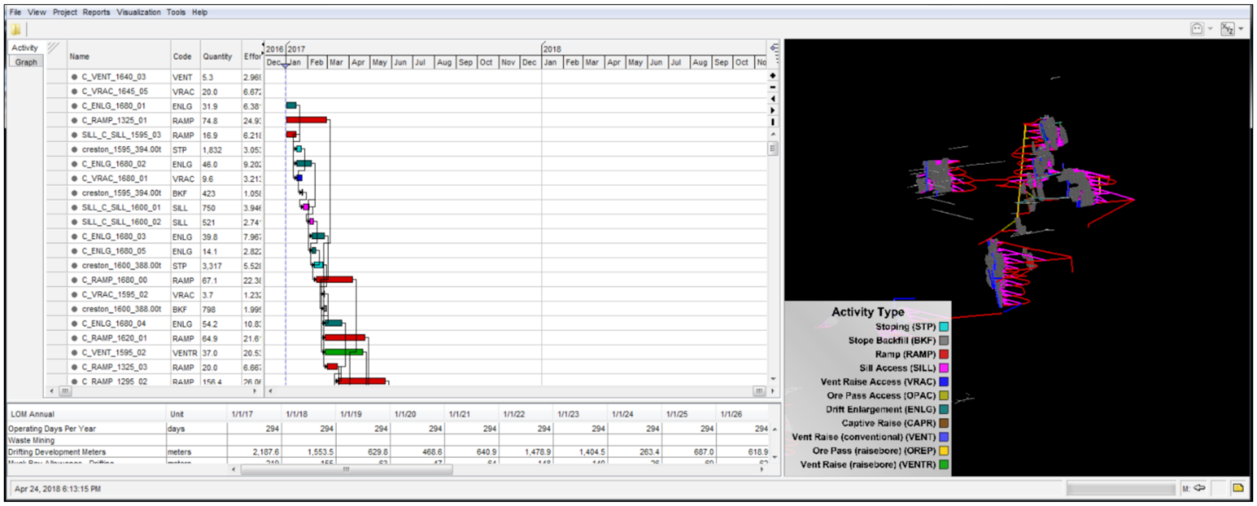
Rock mechanics analysis of the layout should be conducted to project rock stability impacts of the planned geometry. No analysis of ramp locations in the hanging wall has been conducted, and interactions between upper parts of El Creston and El Perdido need to be simulated. Stand-off distances of ramps and ventilation raises need to be evaluated to assure stability of the raise and infrastructure as the mining proceeds vertically.

16.4 MINING PRODUCTION

The Vulcan™ Stope Optimizer inventory of stope shapes was linked with the development ramp, drift and raise objects to create an inventory for scheduling the production. String elements denoting waste developments were assigned cross sections and the average density of waste material was assumed to be 2.75 tonne/m^3 to estimate waste tonnage. The production schedule was created using the iGantt underground scheduler software by Minemax of Centennial CO, USA, which is illustrated by Figure 16-9.

Mine scheduling was controlled by the development of ventilation circuits to provide fresh air to the working areas. For individual stoping blocks, a ramp would be developed between the lower and upper extents of the contiguous stope areas, which would be connected by a vertical raise. Twin ventilation raises would be constructed to facilitate the excavation over the relatively long vertical distances. These ventilation raises would also provide the secondary egress from the stope working area and provide the pathways for electrical, compressed air and make-up water services required for the stope mining. Sustaining development to the various stope production areas was scheduled to maintain a sufficient buffer of stopes ahead of required ore production.

Figure 16-9 View of the Igantt Production Schedule for the Tahuehueto Underground Production with Linked Image of the Tahuehueto Underground Layout



Production scheduling was driven by the selected peak mill throughput of 165,000 tonnes per year (nominally 155,000 tonnes per year over the LOM). The mine schedule delivered 165,000 tonnes per year by annual production based on using a single 12 hour shift per day, operating 6 days per week and 294 days per year, and by a mill stockpile created during the year -1 construction of the mill.

The development rates of advance per underground operating day used for the scheduling were:

- Ramps and access drifts = 3 meters/day
- Stope Access Slabbing = 190 waste tonnes/day
- Conventional Raising = 1.2 meters/day
- Raise boring = 1.8 meters/day

The pre-production development for the mine would focus on creating the initial access and ventilation circuit for the first stoping block in the upper El Creston, extending the ramp system for mining of the upper El Creston, and on creating the drift development for the haulage level. Figure 16-10 illustrates the sequence of development and establishment of production mining in the first 3 years of mining. Initial ore production would begin in the Upper El Creston vein at a nominal rate of 180 tonnes/day, with the first year's ore production being stockpiled as the mill was being constructed. Expansion of the number of stoping blocks would allow ore production to reach its nominal steady state rate of 560 tonnes/day (underground operating day). Ore production was predominantly from the Upper Creston until year 6, when development and mining of the lower El Creston and El Perdido veins would begin. Upper El Creston would be completed in year 7, and development and mining would begin in El Catorce. In year 10, development and mining begins in El Rey, and Santiago production comes near the end of the LOM. The shifts to the new areas of production in year 6 and year 10 are shown in Figure 16-11.

The physical units in the underground production schedule are listed in Table 16.1 , 16.2 and 16.3, for the detailed mine output, the schedule of mined waste to backfilling and backfill stockpile, and the mine ore output and mill stockpile to mill schedule, respectively.

Figure 16-10 Isometric View of Tahuehueto Underground Mining and Development for Years 1-3, Existing Mining Levels Shown In Light Grey

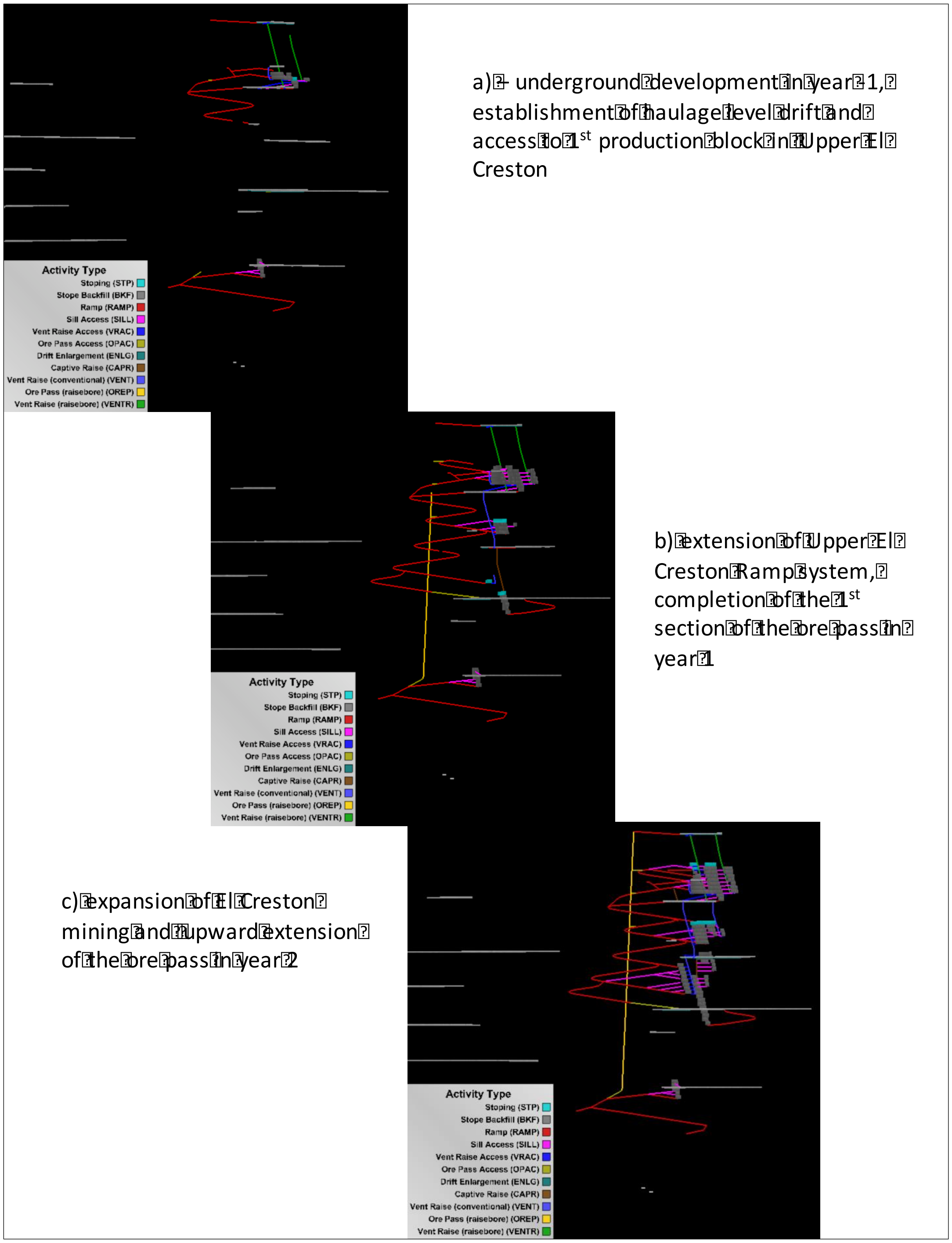


Figure 16-11 View of Tahuehueto Underground Layout In Year 6 and Year 10, With Existing Mining Levels Shown As Light Grey

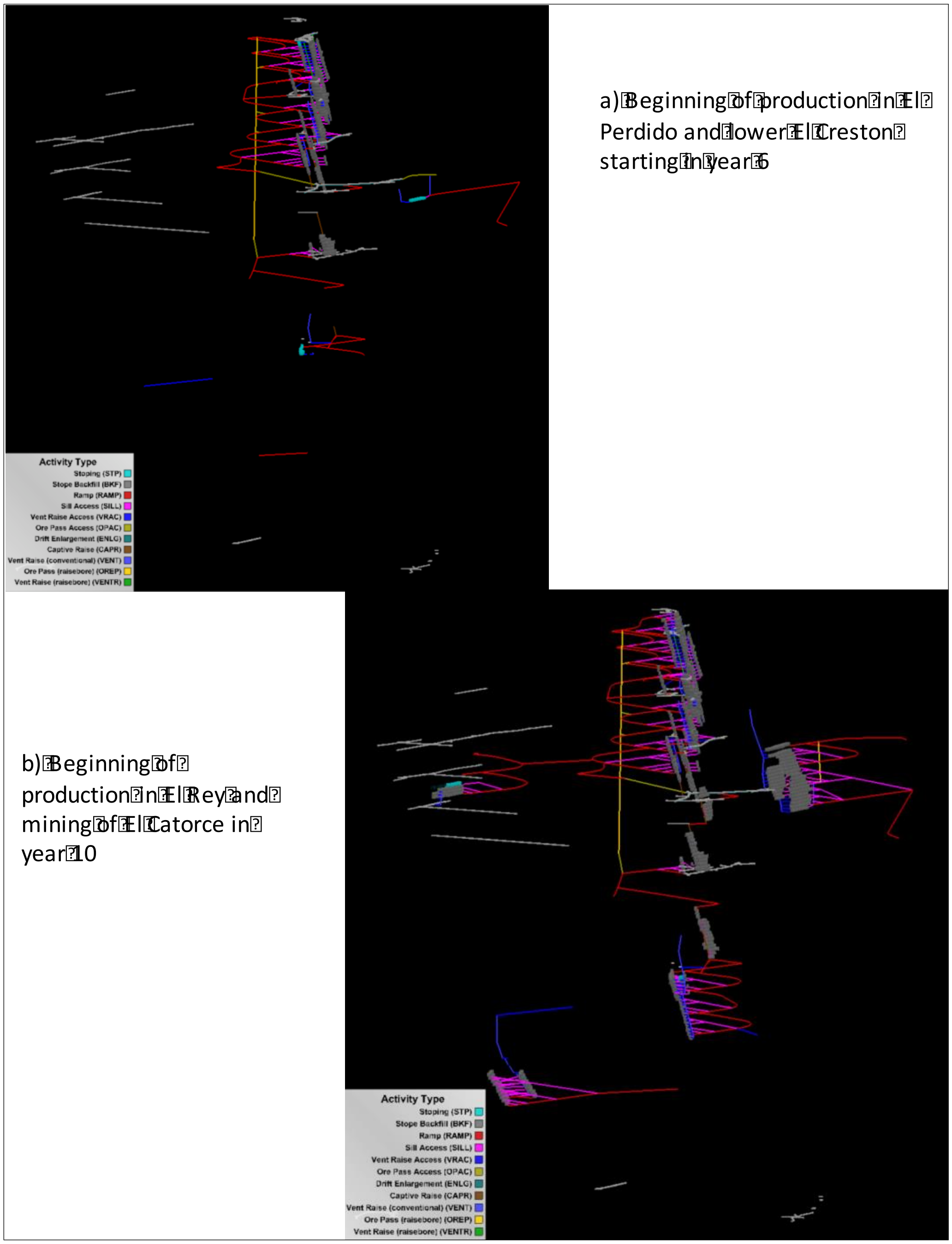


Table 16.1 Tahuehueto Underground Production Schedule by Year

Item	Year	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	Total
Drift Development Waste	m	2,410	1,710	690	520	700	1,630	1,540	290	760	680	210	520	1,020	1,370	1,790	600	550	100	140	120	90	30	**17,460**
Drift Development Waste	t	165,440	117,480	47,630	35,430	48,470	111,840	106,210	19,920	51,950	46,800	14,560	35,440	69,900	94,290	123,370	41,210	37,590	6,720	10,030	9,090	6,160	1,990	**1,201,530**
Waste Slabbing	m	30	50	90	60	30	10	30	90	60	70	50	50	60	50	30	50	70	60	30	60	20	20	**1,040**
Waste Slabbing	t	10,670	17,440	33,260	20,720	11,200	3,800	11,800	33,310	22,230	28,050	17,750	19,720	21,410	18,230	10,860	18,780	24,520	20,910	9,270	24,760	8,320	7,930	**394,940**
Vent Raise Waste	m	190	620	300	280	290	340	600	350	170	70	50	120	150	190	320	260	110	-	40	-	-	-	**4,440**
Vent Raise Waste	t	1,680	8,150	3,660	2,490	2,600	3,010	5,320	3,100	2,210	590	440	1,090	1,370	1,700	2,880	2,300	960	-	350	-	-	-	**43,900**
Ore Stoping	m	310	110	250	500	560	300	230	70	50	130	50	90	120	150	650	300	480	550	320	330	140	10	**5,680**
Ore Stoping	t	30,760	10,010	25,350	54,890	58,800	28,510	19,330	5,520	3,660	9,720	3,480	7,250	8,230	15,810	63,840	29,570	50,730	57,440	34,000	35,010	14,810	1,160	**567,850**
Ore Slabbing	m	250	1,020	1,520	1,240	1,100	1,180	1,380	1,700	1,500	1,400	1,430	1,510	1,500	1,440	1,060	1,590	1,180	1,110	1,120	860	640	560	**26,310**
Ore Slabbing	t	22,080	115,310	139,780	109,010	105,670	136,760	144,210	156,410	156,480	154,840	159,580	158,130	151,700	151,250	99,600	135,340	112,740	107,680	135,120	101,730	81,330	61,550	**2,696,270**
Total Ore Mined	t	52,840	125,310	165,130	163,890	164,470	165,270	163,530	161,920	160,140	164,560	163,060	165,380	159,930	167,070	163,440	164,910	163,470	165,120	169,110	136,740	96,140	62,710	**3,264,120**
Au Grade	g/t	3.9	4.4	2.8	3.7	3.3	3.0	3.2	3.1	2.8	2.5	2.3	2.8	3.4	3.7	3.4	3.0	3.4	4.5	3.9	3.9	4.5	6.1	**3.4**
Ag Grade	g/t	39.6	40.7	37.7	35.6	30.0	37.9	42.8	37.7	37.8	38.0	44.2	45.0	49.3	43.1	39.3	41.1	61.9	48.0	41.3	42.8	40.6	45.8	**41.8**
Cu Grade	%	0.2	0.2	0.2	0.2	0.1	0.2	0.3	0.4	0.4	0.3	0.4	0.4	0.4	0.3	0.4	0.5	0.4	0.4	0.5	0.5	0.5	0.5	**0.3**
Pb Grade	%	1.3	1.0	0.8	1.0	1.3	1.2	1.6	1.5	1.5	1.3	1.3	1.3	1.4	1.4	1.3	1.0	1.0	0.9	1.0	1.0	0.9	1.0	**1.2**
Zn Grade	%	3.1	2.9	3.0	3.3	3.0	2.0	1.9	1.9	2.0	1.8	1.9	2.2	2.9	2.5	1.9	1.6	1.6	1.9	2.3	2.4	1.6	1.9	**2.2**
Total Waste Mined	t	177,780	143,080	84,550	58,650	62,260	118,650	123,330	56,330	76,400	75,450	32,750	56,250	92,690	114,220	137,110	62,290	63,070	27,630	19,650	33,850	14,480	9,920	**1,640,370**
Total Mined	t	230,620	268,390	249,680	222,540	226,730	283,920	286,870	218,250	236,530	240,000	195,800	221,620	252,620	281,290	300,540	227,200	226,540	192,750	188,770	170,590	110,610	72,620	**4,904,490**
Total Backfill Placed	t	5,740	14,680	19,750	22,650	20,200	17,010	23,240	21,740	15,020	22,810	18,140	20,100	19,450	17,720	21,410	23,980	23,820	18,580	20,600	19,020	14,760	8,440	**408,870**

Table 16.2 Tahuehueto Mined Waste to Underground Backfilling Schedule

Item	Year	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	Total
Mine to Backfill Stockpile	t	177,780	143,080	84,550	58,650	62,260	118,650	123,330	56,330	76,400	75,450	32,750	56,250	92,690	114,220	137,110	62,290	63,070	27,630	19,650	33,850	14,480	9,920	**1,640,370**
Mine to Backfill Stockpile	m^3	80,400	64,710	38,240	26,530	28,160	53,660	55,780	25,480	34,550	34,120	14,810	25,440	41,920	51,660	62,010	28,170	28,520	12,500	8,890	15,310	6,550	4,490	**741,870**
Backfill Required	m^3	11,420	29,210	39,300	45,080	40,200	33,860	46,240	43,270	29,900	45,400	36,100	40,000	38,710	35,260	42,610	47,710	47,410	36,980	40,990	37,840	29,370	16,790	**813,650**
Waste Slabbed Locally	m^3	2,370	5,120	11,320	10,470	8,150	3,780	12,050	10,360	6,940	7,930	7,550	9,660	10,700	8,170	5,130	3,740	7,020	8,320	7,560	10,910	1,920	1,190	**160,380**
Total Backfill Rehandled	m^3	9,050	24,090	27,980	34,600	32,040	30,070	34,190	32,910	22,960	37,470	28,550	30,340	28,010	27,090	37,480	43,970	40,390	28,650	33,430	26,940	27,450	15,600	**653,260**
Total Backfill Placed	m^3	11,420	29,210	39,300	45,080	40,200	33,860	46,240	43,270	29,900	45,400	36,100	40,000	38,710	35,260	42,610	47,710	47,410	36,980	40,990	37,840	29,370	16,790	**813,650**
Total Backfill Placed	t	5,740	14,680	19,750	22,650	20,200	17,010	23,240	21,740	15,020	22,810	18,140	20,100	19,450	17,720	21,410	23,980	23,820	18,580	20,600	19,020	14,760	8,440	**408,870**
Total Waste in Backfill Stockpile	t	173,230	130,970	70,490	41,260	46,160	103,540	106,150	39,790	64,860	56,620	18,400	41,000	78,610	100,610	118,270	40,200	42,770	13,230	2,850	20,320	680	2,080	**59,640**

Table 16.3 Tahuehueto Mine Ore Production and Mill Stockpile to Mill Schedule

Item	Year	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	Total
Mine to Mill	t	-	125,310	165,000	163,890	164,470	165,000	163,530	161,920	160,140	164,560	163,060	165,000	159,930	165,000	163,440	164,910	163,470	165,000	165,000	136,740	96,140	62,710	**3,204,200**
Mine to Stockpile	t	52,840	-	130	-	-	270	-	-	-	-	-	380	-	2,070	-	-	-	120	4,110	-	-	-	**59,920**
Total Ore in Stockpile	t	52,840	13,150	13,280	12,170	11,640	11,910	10,450	7,370	2,510	2,060	120	500	-	2,070	500	410	-	120	4,240	-	-	-	**145,320**
Stockpile to Mill	t	-	39,690	-	1,110	530	-	1,470	3,080	4,860	440	1,940	-	500	-	1,560	90	410	-	-	4,240	-	-	**59,920**
Total Tonnes to Mill	t	-	165,000	165,000	165,000	165,000	165,000	165,000	165,000	165,000	165,000	165,000	165,000	160,430	165,000	165,000	165,000	163,880	165,000	165,000	140,970	96,140	62,710	**3,264,120**

16.5 MINE MOBILE EQUIPMENT

No productivity data was available from similar underground mining operations in the area of Tahuehueto, therefore first principal modeling of the mining sequence was performed using the mine layout, equipment operating performance specifications, design drift and slab sizes, haul distances and inclinations and general mining consumables assumptions. Spreadsheet models were developed for the stope mining operations and the drift development mining operations based on the following unit operations:

- Drill and blast – assumes drilling by single boom jumbo, blasting using ANFO explosive loading truck;
- Load and haul – assumes 10 tonne capacity LHD loading in excavation and transporting locally to a 20 tonne capacity underground haul truck in a stope access cross cut or muck bay, haul truck transport of the blasted material to an ore pass or surface portal stockpile;
- Scaling – assumes blasted rock surface scaled by LHD and scaled material hauled by the haul truck;
- Ground support installation – assumes rock bolt installation by Jumbo with welded wire mesh covering the assumed portion of the excavation area, 1 steel strap installed per 4 rock bolts;
- Backfilling with blasted waste material – assumes local blasting of waste with associated drilling, blasting, distribution of blasted material, scaling and ground support installation, LHD loading of the required additional fill material and hauling it to portal site and underground to stope location
- Ore hauled from ore pass bottom bin or portal stockpile to mill – assumes 20 tonne surface haul truck from ore pass bin to mill or portal ore stockpile to mill.

The unit operations were simulated for each design unit in the underground layout, drift or stope, and included the estimated effects of move in, move out, and equipment cycles. Average equipment productivities and materials consumptions were then calculated from the unit operation estimates of the cycles, and were normalized to dry tonnes in-place for each design unit. These average equipment productivities and supplies/materials consumption rates were used to estimate the operating costs, numbers of equipment required and labor required.

Contractors were assumed for conventional and bored raise development, and Telson supplied copies of the estimates were accepted for use in the PFS. Material broken by the contract raising was assumed to be moved by the Tahuehueto owner operation.

Average material densities and loose density assumptions were used to constrain the haulage payloads, and are listed in Table 16.4.

Table 16.4 Assumed Material Densities, Moisture Contents and Loose Densities

Parameter	Average Mineralization (Ore)	Waste
Dry, bank density (t/m^3)	4.02	2.75
Loose, wet density at 50% swell (t/m^3)	2.91	1.99

Productivity assumptions for the major components of jumbo drilling, LHD loading, underground haul truck operation and surfaced haul truck operation are listed in Table 16.5. Other underground equipment included supporting functions for underground road maintenance by low profile motor grader, UG water truck, scissor lift truck, fuel/lube/utility truck, main ventilation fans and ducted auxiliary fans, mobile power centers, mobile air compressors and jack leg drills. Table 16.6 lists the underground equipment and assumed numbers determined for the operating schedule and associated costs. Unit prices for the equipment and mining consumables were based primarily on vendor quotations supplied by Telson. For equipment that was not anticipated by Telson, equipment costs were derived from the Mine and Mill Equipment Costs, Estimators Guide 2015 (Mine and Mill, 2015), and equipment operating costs, materials consumptions and maintenance materials were also derived from the Estimators Guide 2015. Non-equipment capital costs such as spares, parts, tools, and underground shop facilities are presented in Table 16.7. USA costs were adjusted to consider the value added tax (IVA) of 16% required in Mexico. Equipment quotations supplied by Telson were variously in Mexican Pesos, US dollars and Euros. A peso/US dollar exchange ratio of 19 pesos per 1 US dollar was assumed, and a 1.0 Euro/1.07 US dollar exchange rate was assumed for costing purposes.

Table 16.5 Average Major Equipment Productivities Assume For Tahuehueto Underground Mining

Equipment Type	Availability (%)	Utilization of Availability (%)	Assumed Utilization (%)	Equipment Productivity (tonnes/man-hour)
Single Boom Jumbo	85%	77%	65%	36.0*
10 tonne LHD	90%	85%	77%	59.4
20 tonne UG Haul Truck	90%	87%	78%	40.0
20 tonne Surface Haul Truck	90%	89%	80%	28.0

*Represents a blend of drift face drilling, slab drilling and roof bolt drilling

Table 16.6 Tahuehueto Underground Mine Mobile Equipment Requirements

Equipment Type	Required at Start-up	Total Required LOM	Unit Price ($kUS)*	Total Price ($kUS)
Mobile Mine Equipment				
Single Boom Drill Jumbo	3	3	576.1	1,728.2
10 tonne LHD	2	2	581.4	1,162.9
10 tonne UG Haul Truck	2	2	548.5	1,097.1
UG Water Truck	1	1	548.6	548.6
Low Profile Grader	1	1	448.6	448.6
Scissors Lift Truck	1	1	351.5	351.5
Fuel/Lube/Utility Truck	2	2	376.8	753.5
ANFO Charge Truck	1	1	421.0	421.0
20 tonne Surface Haul Truck	3	3	156.6	469.8
Jack Leg Drills	4	4	102.8	205.6
Light Vehicles	4	4	90.5	181.0
Fixed Mine Equipment				

Main Ventilation Fans	2	7	13.8	55.2
Duct Ventilation Fans	2	5	24.7	98.6
UG Power Centers	3	5	24.8	173.7
Portable Generators	2	2	24.8	124.1
Portable Compressors	2	2	50.8	254.1
Total Mine Equipment				**8,073.5**

* Includes delivery and IVA of 16%

Table 16.7 Tahuehueto Underground Mine Non-Equipment Capital Requirements

Equipment Type	Total Price ($kUS)
Mine Mobile Equipment Spares	194.9
Mine Equipment Rebuild	3,669.1
Underground Shop and Tools	250.0
Total CAPEX	**4,114.0**

16.6 UNDERGROUND MINE MANPOWER

Underground operating manpower includes personnel involved in direct supervision and operation of the mining. This includes mine administration, operating labor, and direct maintenance. The grouping and the numbers of underground personnel are summarized in Table 16.8. Underground maintenance requirements were estimated based on ratios of maintenance manpower to operating manpower on each type of equipment, using the following ratios; 3 operator hours per 1 mechanics hour, 2 mechanics hours per 1 electrician hour and 1 mechanics assistant hour per 1 mechanics hour.

Operating equipment hours were determined by estimated equipment productivities. Equipment operating hours in a given 12 hour operating shift were based on the assumed utilization (for example, for major equipment see Table 16.6). Operating man hours were then factored up by a ratio to account for the 12 hour paid shift, no reassignment to other tasks was assumed. The operation was scheduled for 294 operating shifts per year on a 6 day work week. By regulation, maximum worked hours per week were 48 per man, so 4 - 12 hour shifts could be worked per man per week. Two crews would be required to sustain the 6 day operating schedule, and it was estimated that the maximum annual work hour per individual was 1,764.

The working manpower was variable on an annual basis, so that total mine personnel (considering 2 operating crews) would vary between 58 and 101 depending on the schedule year, which reflects the mixture of types of mining production. The distribution of manpower for the maximum manpower which would be required in the year 14 of the production schedule is listed in Table 16.8.

Table 16.8 lists partial personnel in the operator and maintenance functions which are derived from productivities. For costing, the man-hours were rounded up to the totals in each group, and the partial hours to get to a whole person were accounted for in the Mine Admin costing. Telson supplied labor rate assumptions were used for labor cost estimation.

Table 16.8 Underground Manpower by Group and Function for Year 14

Assigned Group	Function	Number of Personnel
Mine Administration	Mine Manager	1
	Mine Shift Supervisor	2
	Chief Engineer	1
	Mine Surveyor	1
	Surveyor Assistant	1
	Mine CAD Operator	1
	Mine Secretary	1
	Geology & Exploration Manager	1
	Mine Geologist	1
	Geotech	1
	Ore Control Supervisor	1
	Ore Control Tech	2
	Total Mine Admin	**15**
Mine Operators*	Jumbo	5.7
	Blaster	2
	LHD	4.5
	UG Haul Truck	4.6
	Surface Haul Truck	6.4
	Auxiliary Equipment	16.2
	Jack Leg	4.0
	Assistants	7.7
	Total Mine Operators	**52**
Mine Maintenance*	Mechanics	13.5
	Electricians	6.6
	Mechanics Techs	13.5
	Total UG Maintenance	**34**
Total Underground Personnel *		**101**

- * Includes 2 crews for mine operators and mine maintenance

16.7 UNDERGROUND VENTILATION

The Tahuehueto mine layout is based on overhead cut-and-fill stoping of different vein bodies which are located relatively close to the hillside topographic surface. Existing mining excavations have been constructed at various elevations to allow earlier sampling of these veins, and have been incorporated into the mine access and ventilation system concepts for the PFS. The general ventilation system concept is the same for development of all of the vein systems and would rely on multiple intake and exhaust surface points controlled by fans in bulkheads at the top of the vent raises. The establishment of individual ventilation circuits would follow the steps of:

- Develop access to the bottom level and top level of a local stoping block;
- Construct twin ventilation raises (1.6 m x 1.6 m) at a nominal setback distance from the vein to transport ventilation air from the bottom stope excavation to the top level access;
- Install relatively small diameter (1.4 m) fans in bulkheads on the ventilation raises at the top level to draw air from the surface along the bottom level stope access drifts, up the ventilation raises and exhausting into the surface connection of the top access.

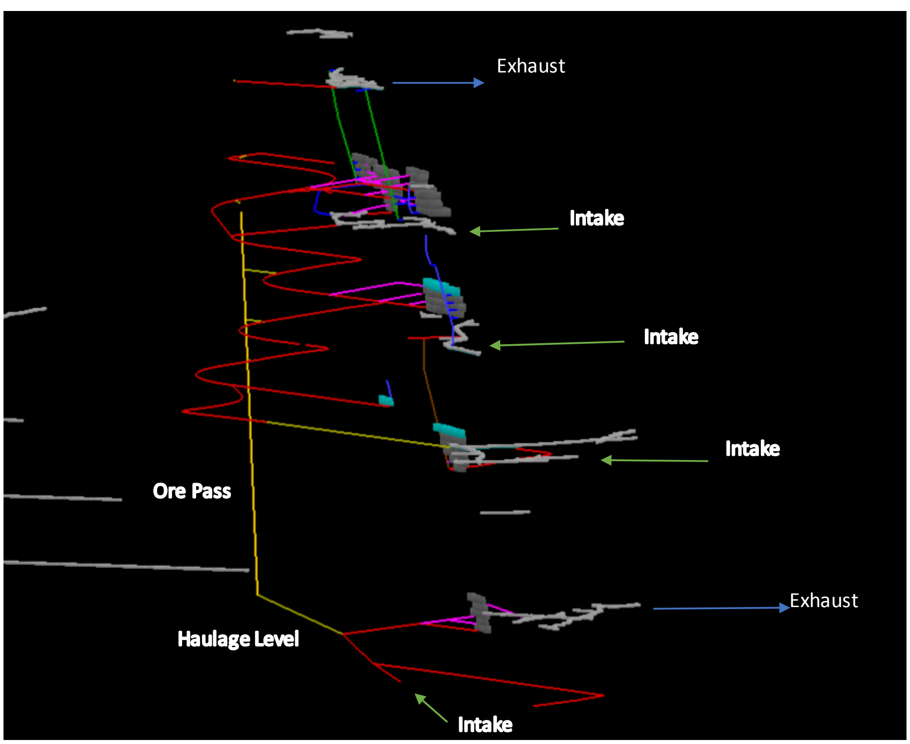
Based on the current estimates these main ventilation raise fans would be specified to supply 3540 m^3/min (125,000 cfm) at a nominal total pressure of 1 kilopascal (4 inches water gage). Quoted motor specification was 149 kW (200 hp), which was projected to be more than sufficient for the pressure drops estimated along the access drift, twin raises, and outlet drift courses.

As mining develops in the stope areas, several raise fan installations would be working in parallel so the total intake ventilation air will substantially exceed the requirements for the equipment operating. Figure 16-12 illustrates ventilation system at year 3 in the upper Creston area. The ventilation raises would be up cast with the individual raise fans controlling the split of air flow, and serial flow mixing as the air moved upward towards the exhaust level. The fans would always be located at the top of stope block, because the raise is filled as the stope moves upward. Intake air would be drawn to the stopes along the ramp system and stope access drifts.

On the haulage level, a ducted fan would be installed at the portal with rigid duct connecting the fan to the base of the ore pass. The fan-duct combination would be operated on negative pressure so that fresh air would be drawn to the base of the ore pass, collecting any dust created by the ore pass in the duct and exhausting it outside the haulage portal. This configuration would create a negative pressure in the ore pass to minimize the propagation of dust associated with dumping the ore in the ore pass on upper levels. Fan total pressure on this system is projected to be nominally 1.1 kPa (4.6 inch water gage) at 1,133 m^3/min (40,000 cfm) for a 1.3 m diameter duct.

Access development drifting would rely on fan and flexible duct systems until the drift is broken thorough to a main air intake level, at which point it would be integrated into the main ventilation system.

Figure 16-12 Upper El Creston Area Showing Intake and Exhaust Ventilation Air Flow Using Bulkhead Fans on Stope Ventilation Raises



16.8 UNDERGROUND INFRASTRUCTURE

Tahuehueto would be a relatively small underground mine operation. Dry mining conditions are assumed based on the existing mine excavations, the proximity of the veins to the mountain side, and the elevation of the mining above the local river drainage. No extensive mine pumping system has been included in the design.

The production mining requires electrical power and make-up water for the Jumbo drills and electrical power for the ventilation fans. For the majority of the mining, in El Creston and El Perdido, it has been assumed that mine electrical power will be provided by the generator array used to power the mill. The main electrical service would be run in through the haulage level, and pulled up boreholes to the upper El Creston. It would be run vertically in the ventilation raises from the top – down to preserve the connection as the mining proceeds upward. Power centers would be located in the stope access cross cuts, near the stope alignment. Make up water would be supplied by a pipe network that connects to tanks located at upper portal areas through boreholes and piping in the ramps.

16.9 MINING OPERATING COST

Mining operating costs have been built up from first principle analysis of the stoping units and development units. Average equipment productivities have been estimated from unit operation simulation spreadsheets. Materials consumptions and equipment operating costs have been derived from vendor quotations supplied by Telson and from equipment operating costs and materials consumptions in the Mining Cost Estimator for US operations. Labor rates for Mexico and overheads have been supplied by Telson. For costing and economic analysis, the mine maintenance man hours were removed from the mine labor and incorporated with mill labor. Similarly, the power requirements for the underground mine were accounted for as a process unit cost. This was done to reflect the power distribution of the Project site and the management structure, respectively. Costs for preliminary development and mine infrastructure such as raise boring have been taken out of the mining unit costs and are accounted for under capital mine development. The operating cost required is indicated in detail in Table 16.9.

Table 16.9 Mine Operating Cost

Costing Area	LOM Cost $k	Unit Cost $/t
Salaried Labor	6,231	1.91
Hourly Labor	14,111	4.32
Drilling	3,703	1.16
Blasting	7,856	2.46
Loading and Scaling	5,052	1.58
UG Hauling	3,727	1.17
Ground Support	1,388	0.44
Surface Hauling	4,658	1.46
Auxiliary	10,895	3.42
Mine Administration	11,794	3.70
Total Mining Operating Costs	**69,415**	**21.62**

17 RECOVERY METHODS

17.1 MAJOR PROCESS DESIGN CRITERIA

The flotation circuit is designed to process a peak of 165,000 tpy in 300 operating days, equivalent to 550 tonnes per day and 22.9 tph through the grinding and flotation circuits. More information of the operating parameters and reagents for optimizing recoveries and concentrate quality will be obtained from the 3,500 tonne industrial scale bulk testing discussed in Section 24. The Projects key drivers are presented in Table 17.1.

Table 17.1 Tahuehueto Project Key Drivers

Description	Units	Value
Operating Metrics		
Operating Days	dpy	300
Capacity	tpd	550
Peak Tonnes per Year	tpy	165,000
Total Ore Processed	kt	3,264
Head Grade		
Gold	gpt	3.504
Silver	gpt	41.931
Copper	%	0.348%
Lead	%	1.192%
Zinc	%	2.242%
Contained Metal		
Gold	kg	11,083
Silver	kg	136,443
Copper	kt	11.35
Lead	kt	38.90
Zinc	kt	73.17

The life of mine concentrate production metallurgic balance is indicated in Table 17.2. This metallurgic balance was prepared from the different metallurgic research discussed in Section 13 and is based on the production schedule detailed in Section 16.4 using the basic plant design criteria. The annual mill production schedule is presented in Table 17.3.

Table 17.2 Tahuehueto Average Metallurgical Recoveries

Product	kTonnes	Distribution % (Recoveries)				
		Au	Ag	Cu	Pb	Zn
Head	3,264	100%	100%	100%	100%	100%
Pb Concentrate	58	77.1%	62.8%	31.6%	85.5%	1.6%
Cu Concentrate	18	6.8%	10.3%	51.4%	0.6%	17.1%
Zn Concentrate	108	11.0%	11.7%	11.5%	6.1%	80.0%
Tails	3,079	5.4%	15.2%	5.4%	7.8%	1.3%

Table 17.3 Tahuehueto Mill Production Schedule

Item	Years	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	Total
Ore Tonnes to Mill	kt	-	165	165	165	165	165	165	165	165	165	165	165	160	165	165	165	164	165	165	141	96	63	**3,264**
Au Head Grade	g/t	-	4.4	2.8	3.7	3.3	3.0	3.2	3.1	2.8	2.5	2.3	2.8	3.4	3.7	3.4	3.0	3.4	4.5	3.9	3.9	4.5	6.1	**3.4**
Ag Head Grade	g/t	-	40.7	37.7	35.6	30.0	37.9	42.7	37.8	37.8	38.0	44.2	45.0	49.3	43.1	39.3	41.1	61.9	48.0	41.3	42.8	40.6	45.8	**41.8**
Cu Head Grade	%	-	0.2	0.2	0.2	0.1	0.2	0.3	0.4	0.4	0.3	0.4	0.4	0.4	0.3	0.4	0.5	0.4	0.4	0.5	0.5	0.5	0.5	**0.3**
Pb Head Grade	%	-	1.0	0.8	1.0	1.3	1.2	1.6	1.5	1.5	1.3	1.3	1.3	1.4	1.4	1.3	1.0	1.0	0.9	1.0	1.0	0.9	1.0	**1.2**
Zn Head Grade	%	-	2.9	3.0	3.3	3.0	2.0	1.9	1.9	2.0	1.8	1.9	2.2	2.9	2.5	1.9	1.6	1.6	1.9	2.3	2.4	1.6	1.9	**2.2**
Au Contained Metal	koz	-	20	10	20	20	20	20	20	20	10	10	10	20	20	20	20	20	20	20	20	10	10	**360**
Ag Contained Metal	koz	-	220	200	190	160	200	230	200	200	200	230	240	250	230	210	220	330	250	220	190	130	90	**4,390**
Cu Contained Metal	klb	-	730	730	730	370	730	1,090	1,450	1,460	1,090	1,460	1,460	1,410	1,090	1,460	1,820	1,450	1,460	1,820	1,550	1,060	690	**25,080**
Pb Contained Metal	klb	-	3,640	2,910	3,630	4,720	4,370	5,810	5,460	5,460	4,730	4,730	4,730	4,950	5,090	4,730	3,640	3,610	3,270	3,640	3,110	1,910	1,380	**85,520**
Zn Contained Metal	klb	-	10,550	10,910	12,000	10,920	7,280	6,910	6,910	7,280	6,550	6,910	8,000	10,260	9,090	6,910	5,820	5,780	6,910	8,370	7,460	3,390	2,630	**160,830**
Au Recovered Metal	koz	-	20	10	20	20	20	20	20	10	10	10	10	20	20	20	20	20	20	20	20	10	10	**340**
Ag Recovered Metal	koz	-	180	170	160	140	170	190	170	170	170	200	200	220	190	180	190	280	220	190	170	110	80	**3,720**
Cu Recovered Metal	klb	-	600	600	600	300	600	910	1,210	1,210	910	1,210	1,210	1,170	910	1,210	1,510	1,200	1,210	1,510	1,290	880	570	**20,820**
Pb Recovered Metal	klb	-	3,330	2,670	3,330	4,330	4,000	5,330	5,000	5,000	4,330	4,330	4,330	4,540	4,670	4,330	3,330	3,310	3,000	3,330	2,850	1,750	1,270	**78,370**
Zn Recovered Metal	klb	-	8,440	8,730	9,600	8,740	5,820	5,530	5,530	5,820	5,240	5,530	6,400	8,210	7,280	5,530	4,660	4,630	5,530	6,690	5,970	2,710	2,100	**128,680**

17.2 PLANT DESIGN

Based on metallurgical tests performed to date, the previous metallurgical campaigns provide sufficient data to reach a level of confidence that the flotation process chosen will work, flotation targets are attainable, and an economical concentrate can be produced. For the economic analysis and plant design it is recommended to use the result of the last confirmation metallurgical test run by Teslon, specifically for the use of new reagents and have extra capacity for processing high grade ores. The basic design criteria is shown in Table 17.4.

Table 17.4 Basic Design Criteria

Item	Value
PeakTonnes of dry ore processed by year	165,000
Days per year	300
Tonnes by day	550
Crushing hours by day	12
Tonnes per hour crushing	45.8
Grinding and Filtering hours per day	24
Tonnes per hour grinding and filtering	22.9
Pb Concentrate dry tonnes per year	2,800
Zn Concentrate dry tonnes per year	5,200
Cu Concentrate dry tonnes per year	860
Life of operation (years)	21

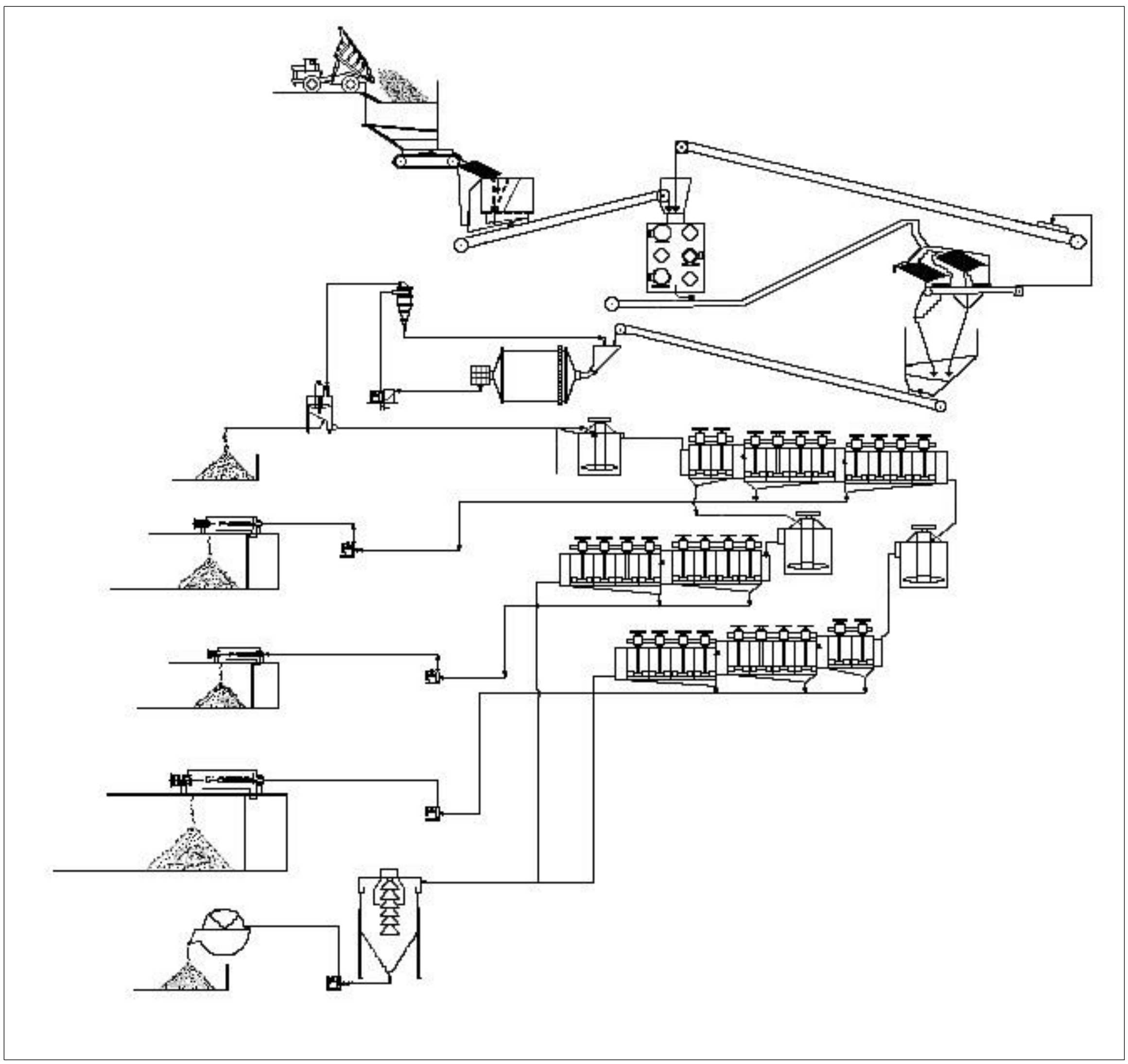
The proposed processing flow sheet is a conventional crushing/milling/flotation/filtering process and is described in continuation and in the diagram on Figure 17-1.

- One coarse ore bin with capacity to receive 20 ton trucks and total storage capacity of 80 tons and a mesh to control the rock size of 50 x 80 cm
- One Apron feeder with capacity to convey 40 to 60 tonnes per hour of rocks of 50 cm x 80 cm, with a mesh at the end to separate the -5 cm ore.
- One Primary Jaw crusher 76 cm x 90 cm to receive 42 tonnes per hour and a maximum rock dimension of 60 cm x 90 cm, which produces -5 cm rock size
- Secondary multi impact crusher for reduce the size of the ore to minus 10 mm
- Two screens 10 cm X 20 cm´ in parallel in close circuit with the Multi Impact crusher to ensure an ore size production of minus 10 mm
- Fine ore bin with capacity for a minimum of 550 tons of fine ore
- A single nominal 2.40 m x 2.40 m ball mill, with 300 HP for mill the ore to a P80 or 90 microns at a rate of 20.83 tonnes per hour.
- Cyclone classification system in closed circuit with the ball mill to guarantee an ore product grinded to -90 microns (80% - 200 mesh) for the next stage.
- One gravimetric concentrator with capacity to process 25 tonnes per hour
- One conditioner tank for Bulk Pb/Cu
- One rougher flotation circuit for Bulk Pb/Cu with a nominally 30 minute retention time.
- One cleaner flotation circuit for Bulk Pb/Cu with a nominally 30 minute retention time.
- One flotation circuit for Pb/Cu separation with a nominally 30 minute retention time.

- One cleaner flotation circuit for Pb/Cu separation
- One conditioner tanks for Zn
- One rougher flotation circuit for Zn with a nominally 30 minute retention time.
- One cleaner flotation circuit for Zn with a nominally 30 minute retention time.
- One decanter for Pb concentrate.
- One decanter for Cu concentrate.
- One decanter for Zn concentrate.
- One decanter for tailings.

All the concentrates will be shipped in bulk and trucked in 30 tons trucks. The Cu an Pb concentrates will be send to the port of Manzanillo, Colima and the Pb concentrates to Matehuala, San Luis Potosí.

Figure 17-1 Plant Flowsheet Section



17.3 PROCESS PLANT DESCRIPTION

17.3.1 GRINDING SECTION

This section of the plant works 12 hours per day and will receive ore from the mine in 20 t capacity dump trucks and discharge to the coarse ore bin with a capacity of 60 t. The coarse bin will have a 46 cm grizzly. The ore rocks greater than 46 cm will be reduced with the use of pneumatic hammer or by hand.

The ore from the coarse ore bin, will be fed to a primary jaw crusher by a 61 cm apron feeder, before passing through a fixed 5 cm screen, to send directly to the secondary crusher, using the 61 cm conveyor belt.

The ore fed to the primary crusher, is crushed to a size -5 cm and also passed to the 61 cm conveyor belt, which is fed along with the product -5 cm of the 5 cm screen to the secondary crusher.

The secondary crusher receives the -5 cm ore and reduces it to a -10 cm size, which is passed to a second 91 cm wide conveyor belt, which carries the ore to -10 cm vibrating screens, where the ore -10 cm is separated and passed to the fine ore bin with 500 t capacity. The ore +10 cm size, is transported by 61 cm belt conveyors and returned to the secondary crusher. The ore is then passed to the 91.4 cm conveyor belt which returns the ore to the -10 cm vibrating screens, thus creating a closed circuit to ensure having the fine ore bin ore with a size -10 cm for the next stage of the process.

17.3.2 MILL SECTION

The -10 mm crushed dry ore is fed to a 2.4 m x 2.4 m ball mill with a 300 HP motor, at an average of 20.83 tph. In this stage recycled and fresh water is added at a rate of 23 m^3/h, (80% is recycled water (304 l/m) and 20% (76 l/m) is fresh water). Also at this point the first reagents for the flotation process is added, AF 404 and AF 242 promotors and 1,100 g/t of $ZnSO_4$.

The download of the ore as pulp mill, is sent to a pair of cyclones, which return the ore with one size greater than 200 mesh to the mill to ensure that ore that will sent to the next stage in the gravimetric concentration and flotation section, reach a size to 80 % - 200 mesh.

17.3.3 GRAVIMETRIC CONCENTRATION SECTION

The pulp from the cyclones at a size of 80% -200 mesh, is fed into a gravimetric concentrator Falcon or Nelson type, which by the use of centrifugal force and difference of specific gravity of the mineral components, concentrates free heavy metals such as gold, silver, and iron. This concentrate is sent to lead concentrate decanter to be mixed and filtered with it. The tailings of the gravimetric concentrator are then sent to the flotation section.

17.3.4 FLOTATION SECTION

The ore pulp is received into the conditioning tank for lead and copper, where the ore sits for a period of 30 minutes. Additional reagents will be added at this point, if required.

Already conditioned ore pulp, is added to the first bank of cells, where is obtained the first bulk concentrate with lead and copper and precious metals as gold and silver, which passes to another group of cleaning flotation cells where is added AF-404 and $ZnSO_4$ to separate the Pb concentrate. Tails of the lead clean flotation cells, are sent to another group of cells where the AF-242 is added to separate the Cu concentrate.

The bulk of Pb/Cu flotation tails, are submitted to the conditioning tank for Zn, where 1 kg/t of $CuSO_4$ and 2 kg/t of CaO is added. Conditioning time is 30 minutes at which time the ore pulp is sent to the first flotation cells for zinc and this product is then sent to the second and third clean cells to finally get the Zinc concentrate. The tailings from the zinc flotation are sent to the tailings decanter for water recovery.

A summary of the Reagents type, application and amount used in the flotation process are indicated in Table 17.5

Table 17.5 Reagents Type and Consumption

Reagent	Aplication	Quantity
CaO	(pH control)	2,000 g/t
$ZnSO_4$	(Zn depress)	1,100 g/t
AF-242	(Promoter - Collector Pb - Cu)	70 g/t
AF-404	(Promoter- Collector Pb – Cu)	70 g/t
Mibc-70	(Floater)	50 g/t
$CuSO_4$	(Zn activator)	1,000 g/t
AF-211	(Zn promotor)	80 g/t
X-343	(Zn colector)	30 g/t.
Xantato	(Collector)	30 g/t

17.3.5 SECTION OF DECANTER AND FILTERING

Concentrates of Pb, Cu, and Zn, are sent to decanters, where the percentage of solids is increased to 90%. The recovered water is sent to the recycled water tank and returned to the process.

The concentrated with 10% water, percentage necessary for handle and avoid losses and pollution is send by gravity to the concentrate storage building and from here are loaded onto 30 t trucks, sampled, weighed, packed, and sent to the point of sale in the port of Manzanillo for the Zn and Cu concentrate and to Matehuala, S.L.P. the Pb concentrate.

The flotation tailings are decanted in a decanter which recovers 90% of the water, which is sent to the tank of recycled water for re-use in the process and the resulting tailings pulp with a 90% solids, composed for the milled original rock that hosted the minerals, is sent to the tailings dam with 10% of water, which will be deposited using the same 20 t trucks for ore haulage, according to the design in the area assigned to the tailings dam.

Water losses are caused by evaporation and moisture remaining in the concentrate since the process is closed.

17.4 MASS BALANCE

Based on the metallurgical testing and average annual metallurgical balance, the mass balance was prepared for the equipment selection for the 550 tpd flotation plant as indicated in Table 17.6.

Table 17.6 Mass Balance

		Solids				Liquids			Pulps			
Equipment Type	**OpTime hours**	**t/h**	**SG**	**m³/h**	**% Solids**	**t/h**	**SG**	**m³/h**	**t/h**	**SG**	**m³/h**	**gpm**
Crushing Circuit												
Coarse Ore Hopper 80 t	12	45.8	2.8	16.4	100.0	-	-	-	-	-	-	-
Apron Feeder 61 cm	12	45.8	2.8	16.4	100.0	-	-	-	-	-	-	-
Grizzly Rail Separation	12	45.8	2.8	16.4	100.0	-	-	-	-	-	-	-
Jaw Crusher 76 cm x 91 cm	12	45.8	2.8	16.4	100.0	-	-	-	-	-	-	-
Conveyor Belt 46 cm x 12 m	12	45.8	2.8	16.4	100.0	-	-	-	-	-	-	-
Multi-impact (Secondary) Crusher	12	45.8	2.8	16.4	100.0	-	-	-	-	-	-	-
Conveyor Belt 46 cm x 22 m	12	45.8	2.8	16.4	100.0	-	-	-	-	-	-	-
Vibrating Screen 1.2 m x 2.4 m	12	45.8	2.8	16.4	100.0	-	-	-	-	-	-	-
Conveyor Belt 46 cm x 7 m	12	45.8	2.8	16.4	100.0	-	-	-	-	-	-	-
Conveyor Belt 46 cm x 24 m	12	45.8	2.8	16.4	100.0	-	-	-	-	-	-	-
Fine Ore Hopper 550 t	12	45.8	2.8	16.4	100.0	-	-	-	-	-	-	-
Grinding Circuit												
Conveyor Belt 46 cm x 7 m	24	22.9	2.8	8.2	100.0	-	-	-	-	-	-	-
Continuous Scale	24	22.9	2.8	8.2	100.0	-	-	-	-	-	-	-
300 HP Ball Mill (Used) 2.4 m x 2.4 m	24	57.3	1.0	57.3	96.9	1.8	1.0	1.8	59.1	1.0	59.1	260.3
Horizontal Centrifugal Pump 2.4 m x 1.8 m	24	57.3	1.0	57.3	88.6	7.3	1.0	7.3	64.6	1.0	64.6	284.6
Classifiers	24	57.3	1.0	57.3	86.2	9.2	1.0	917.0	66.5	1.0	66.5	292.7
Gravimetric Concentrator	24	20.8	2.8	7.4	62.5	12.5	1.0	12.5	33.3	1.7	20.0	87.9
Zinc Circuit												
Conditioner Tank 2.4 m x 2.4 m	24	20.8	2.8	7.4	30.0	48.5	1.0	48.5	69.3	1.2	55.9	246.1
3 Flotation Cells Bank 10 t	24	20.8	0.0	1.2	30.0	48.5	1.0	48.5	69.3	-	55.9	246.1
4 Flotation Cells Bank 15 m	24	3.5	2.8	1.2	30.3	8.0	1.0	8.0	11.5	1.2	9.2	40.7
2 Flotation Cells Bank 15 m	24	3.5	2.8	1.2	6.0	0.0	-	0.0	0.0	-	0.0	0.1
Horizontal Centrifugal Pump 6.4 cm x 5.1 cm	24	3.5	2.8	1.2	30.2	8.0	1.0	8.0	11.4	1.2	9.2	40.5
Clarifying Decanter	24	0.6	2.8	0.2	90.1	0.1	1.0	0.1	11.4	1.2	9.2	40.5
Lead Circuit												
Conditioner Tank 2.4 m x 2.4 m	24	2.9	3.0	1.0	30.3	6.7	1.0	6.7	9.6	1.2	7.7	34.1
3 Flotation Cells Bank 10 t	24	2.9	3.0	1.0	30.3	6.7	1.0	6.7	9.6	1.2	7.7	34.1
4 Flotation Cells Bank 15 m	24	0.5	3.2	0.2	18.2	2.2	1.0	2.2	2.7	1.2	2.2	9.5
2 Flotation Cells Bank 15 m	24	0.5	3.2	0.2	18.2	2.2	1.0	2.2	2.7	1.2	2.2	9.5
Horizontal Centrifugal Pump 6.4 cm x 5.1 cm	24	0.2	3.2	0.1	50.0	0.2	1.0	0.2	0.3	1.5	0.2	0.9
Clarifying Decanter	24	0.3	3.2	0.1	90.9	0.0	1.0	0.0	0.3	2.7	0.1	0.5
Copper Circuit												
Conditioner Tank 1.5 m x 1.5 m	24	20.0	3.2	6.2	40.8	29.0	1.0	29.0	48.9	1.4	35.2	155.0
1 Flotation Cells Bank 10 t	24	20.0	3.2	6.2	40.8	29.0	1.0	29.0	48.9	1.4	35.2	155.0
2 Flotation Cells Bank 15 m	24	3.3	3.2	1.0	40.8	4.8	1.0	4.8	8.2	2.0	4.1	17.9
2 Flotation Cells Bank 15 m	24	3.3	3.2	1.0	40.8	4.8	1.0	4.8	8.2	1.4	5.9	25.8
Horizontal Centrifugal Pump 3.8 cm x 2.5 cm	24	3.3	3.2	1.0	50.8	3.2	1.0	3.2	6.6	1.5	4.3	18.8
Clarifying Decanter	24	0.9	3.2	0.3	90.2	0.1	1.0	0.1	1.0	2.6	0.4	1.7
Clarifying Decanter	24	19.1	3.2	6.0	90.1	2.1	1.0	2.1	21.2	2.6	8.0	35.4
Tails												
Horizontal Centrifugal Pump 8’x6’	24	19.1	4.2	4.5	40.8	27.6	1.0	27.6	46.7	1.4	33.6	147.9

17.5 PLANT ENGINEERING

The basic engineering was performed using the material volumes indicated in the Mass Balance described in 20.6 section. The size and type of equipment selection was determined by experience with the construction of other similar processing plants. A consideration in addition to mass in the equipment selection was the energy efficiency and low water consumption. Table 17.7 indicates the selected plant equipment.

Table 17.7 Plant Equipment

Equipment Type	Required
Coarse Ore Hopper 80 t	1
Apron Feeder 61 cm	1
Grizzly 46 cm Rail Separation	1
Jaw Crusher 76 cm x 91 cm	1
Conveyor Belt 46 cm x 12 m	1
Multi-impact (Secondary) Crusher	1
Conveyor Belt 46 cm x 22 m	1
Vibrating Screen 1.2 m x 2.4 m	2
Conveyor Belt 46 cm x 7 m	2
Conveyor Belt 46 cm x 24 m	1
Fine Ore Hopper 550 t	1
Conveyor Belt 46 cm x 7 m	1
Continuous Scale	1
300 HP Ball Mill (Used) 2.4 m x 2.4 m	1
Horizontal Centrifugal Pump 2.4 m x 1.8 m	3
Classifiers	2
Gravimetric Concentrator	2
Conditioner Tank 2.4 m x 2.4 m	2
3 Flotation Cells Bank 10 t	2
4 Flotation Cells Bank 15 m	2
2 Flotation Cells Bank 15 m	4
Horizontal Centrifugal Pump 6.4 cm x 5.1 cm	4
Clarifying Decanter	4
Conditioner Tank 1.5 m x 1.5 m	1
1 Flotation Cells Bank 10 t	1
Horizontal Centrifugal Pump 3.8 cm x 2.5 cm	1
Crushing and Mill Building	1
Flotation and Filtering Building	1
Lab Equipment	1
Fresh Water Tank	1
Recycled Water Tank	1
Reagents Tanks and Buildings	1
Auxiliary Plant Equipment	1

17.6 PLANT GENERAL LAYOUT

Based on the major equipment selection, a general layout was prepared to consider the installation of the crushing and milling section into the existing construction walls of the old mill. For preparation of the general layout of the plant, factors considered for the placement of the mill included proximity to the haulage level portal for the mine, surface conditions, access roads and existing infrastructure, and trying to reduce environmental impacts. Figures 17-11 and 17-12 shows the mill's preliminary general layout.

Figure 17-2 Preliminary Mill General Layout

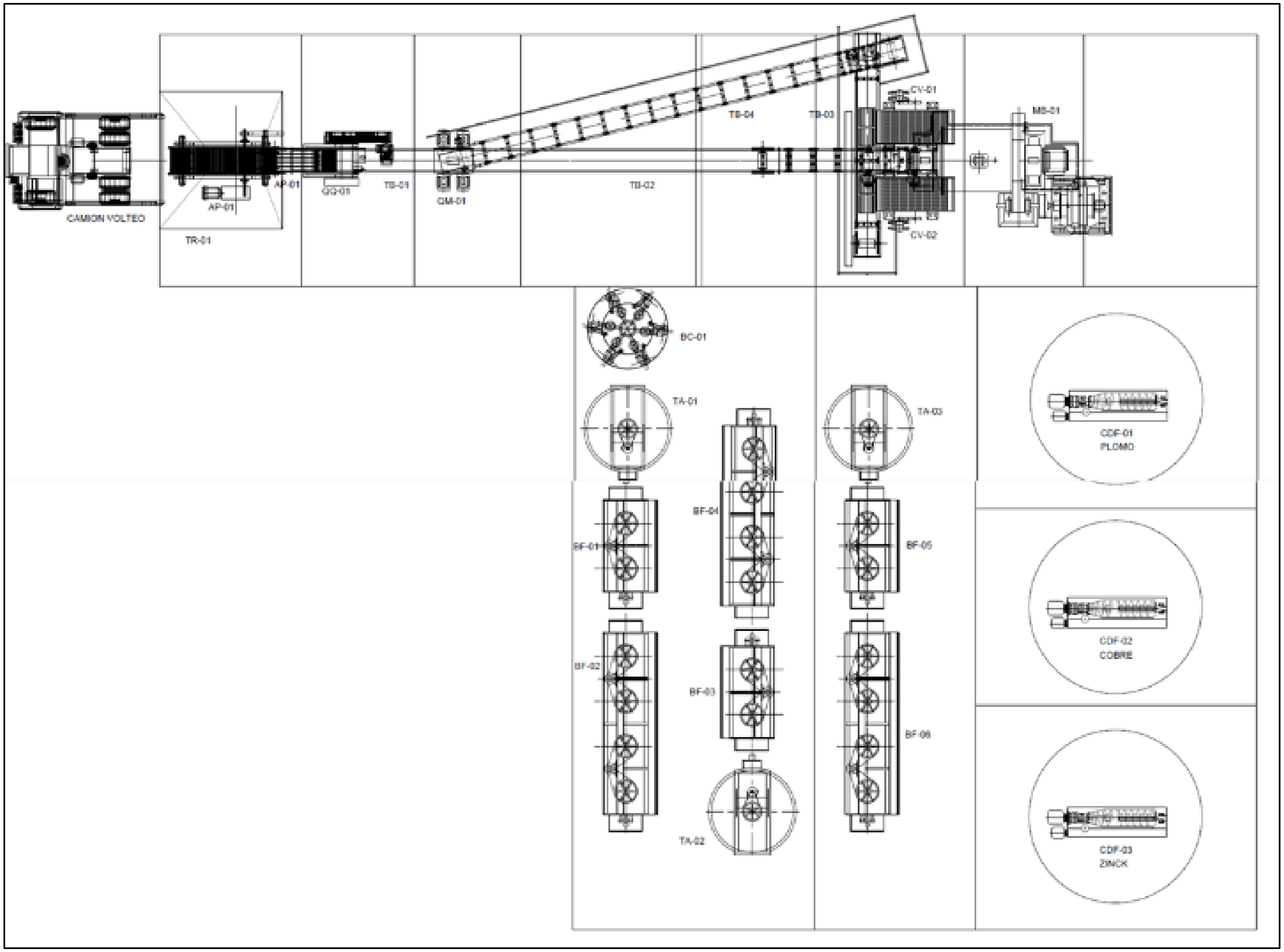
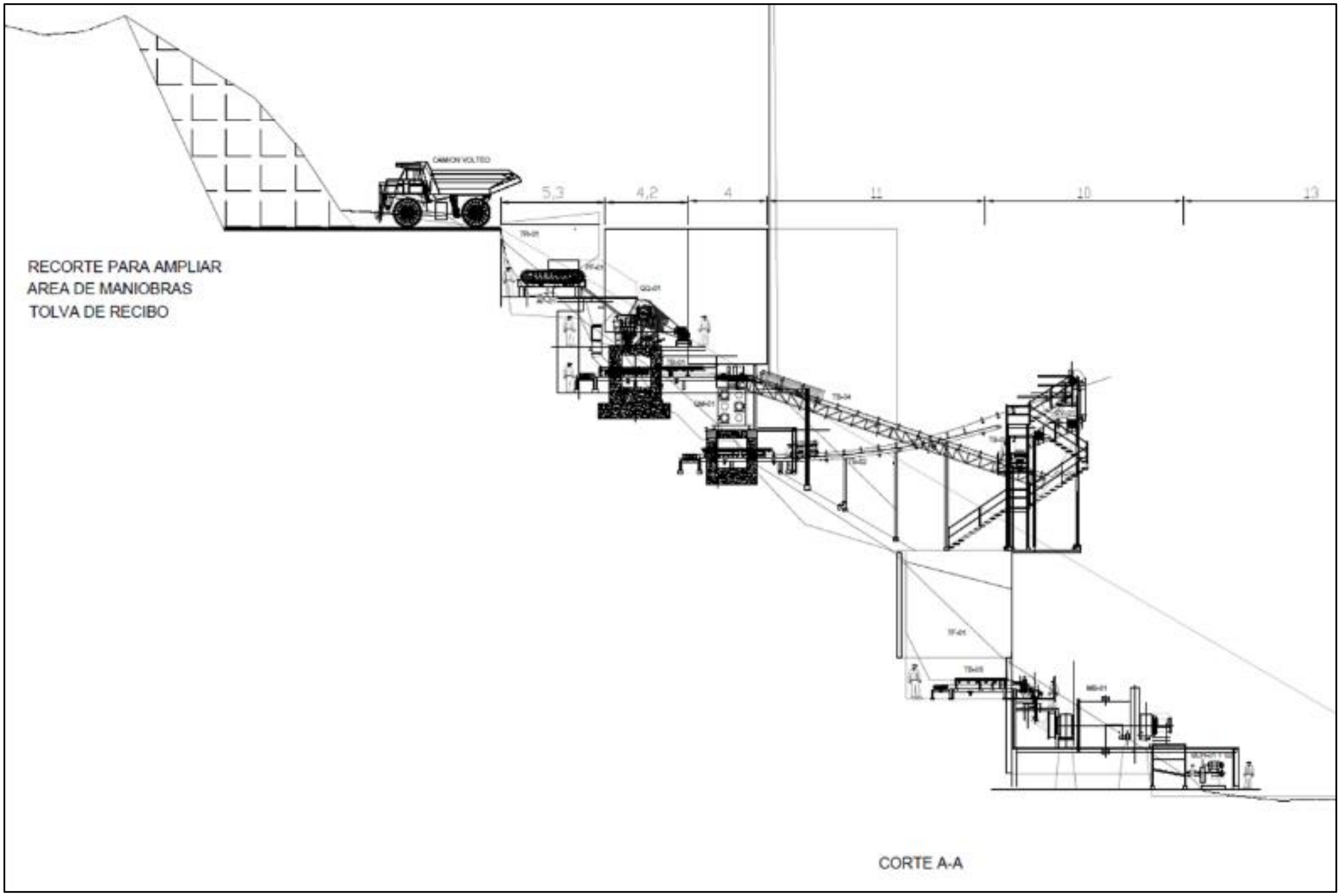


Figure 17-3 Preliminary Crushing and Mill Section Layout



17.7 TAILINGS DISPOSAL

Water is an important element and for that reason the tailings will be sent first to a decanter in order to reduce the water in the tailings from 70% to 10%. The recovered water will be sent to the recycled water tank for re-use in the process.

The dry tailings with 10% moisture will be trucked to the tailing dam in the same 20 tons trucks used for hauling the ore from the mine. The tailings dam has an estimated capacity of above the 38 million tonnes required from the production schedule.

17.8 PROCESS PLANT MANPOWER

The plant is going to operate two; 12 hours shifts each day, on the basis of 20 working days by 6 off. Table 17.8 indicates the 58 personnel required to operate and manage the processing plant at Tahuehueto.

Process manpower includes personnel involved in direct supervision and operation of the mill and processing operation. This includes administration, operating labor, and direct maintenance. The grouping and the numbers of personnel are summarized in Table 17.8. Maintenance requirements were estimated based on ratios of maintenance manpower to operating manpower.

Operating hours were determined by estimated equipment productivities. Equipment operating hours in a given 12 hour operating shift were based on the assumed utilization. Operating man hours were then factored to account for the 12 hour paid shift, no reassignment to other tasks was assumed. The operation was scheduled for 300 operating days per year on a 6 day work week.

For costing, the man hours were rounded up to the totals in each group. Telson supplied labor rate assumptions were used for labor cost estimation. For the economic analysis of the project, general site administration has been added to the processing plants manpower estimates. These general and administration (G&A) personal requirements are presented in Table 17.9.

Table 17.8 Process Plant Manpower by Group and Function

Assigned Group	Function	Number of Personnel
Mill Administration	Mill Manager	1
	Mill Superintendent	1
	Metallurgist	2
	Mill Operations Foreman	4
	Maintenance Superintendent	1
	Maintenance Foreman	3
	Plant Electrical Foreman	1
	Metallurgical Accountant	2
	Mill Administrative Assistant	1
	Total Mill Admin	**16**
Plant Operators	Crushing Control Room	4
	Crush and Convey	8
	Grinding and Flotation	4
	Regrind/Cleaner Float	4
	Thickener and Filtration	4
	Sampler	4
	Packing/Shipping/Reagents	2
	Tailings/Water Treatment	2
	Total Plant Operators	**32**
Mill Maintenance	Crusher	8
	Grinding and Flotation	4
	Regrind/Cleaner/Filter/ Packing	8
	Electrician	4
	Interment Technician	4
	Total Mill Maintenance	**10**
Total Mill Personnel		**58**

Table 17.9 Project General and Administration Manpower

Assigned Group	Function	Number of Personnel
General Administration	General Manager	1
	Administrative Assistant	1
	Controller	1
	Accountant	2
	Human Resources Manager	1
	Human Resources Generalist	1
	Regional Supply Manager	1
	Contracts Manager	1
	Supply Superintendent	1
	Warehouse Supervisor	1
	Warehouse Clerks	4
	Buyer/Coordinator	2
	Plant Engineer	1
	Environmental Engineer	1
	Industrial Hygienist	1
	Health and Safety Technician	2
Total G&A Personnel		**22**

17.9 PLANT CAPITAL COST

The capital cost of the plant is made up of the plant equipment, plant and service buildings, and plant construction. The total capital cost required is USD $14,350,000 without contingency and is indicated in detail in Table 17.10.

Table 17.10 Plant Equipment and Buildings Capital Cost

Costing Area	Cost ($k)
Direct Costs	
Crushing, Conveying, and Storage	2,840
Grinding	1,600
Flotation	1,520
Filtration, Packing, and Shipping	1,180
Fresh and Process Water	1,350
Ancillary Facilitates	1,000
Tailings Storage Facility	300
Main Plant Substation	940
General Site Earthwork and Grading	300
Indirect Costs	
Mobilization	100
Freight	250
Operations Overhead	120
Construction Overhead	620
Engineering, Procurement, and Construction Management	1,860
Vendor Support, Spares	250
Capital Spares, First Fills	120
Total Processing Capital Costs	**14,350**

17.10 PLANT OPERATING COST

The operating costs are based on first principle engineering cost buildups for processing operations as shown in Table 17.11. These costs were derived from materials, power, and labor for each unit operation. For costing, the mine maintenance man hours were incorporated with mill labor along with the power requirements for the underground mine. This was done to reflect the power distribution of the Project site and the management structure, respectively. Operating consumables and maintenance consumables were based on costing services.

Table 17.11 Plant Operating Cost

Costing Area	LOM Cost ($k)	Unit Cost ($/t)
Salaried Labor	10,264	3.14
Hourly Labor	37,552	11.50
Crushing and Grinding	23,795	7.29
Flotation	15,607	4.78
Filtration, Packing, and Shipping	3,447	1.06
Tailings Storage Facility	5,056	1.55
Mine Power Requirements	4,808	1.47
Total Processing Operating Costs	**100,529**	**30.80**

18 PROJECT INFRASTRUCTURE

Site infrastructure and ancillary facilities will comprise of the following components complying with all municipal, state and federal government regulations, and adhering to excellent engineering and construction practices.

18.1 SITE ACCESS ROAD

Access to the Tahuehueto project by land from Tepehuanes is by paved road for approximately 40 km to Ciniega de Los Frailes and then an additional 80 km of unpaved road and the last 40 km is a dirt road that needs new design in some areas to accommodate large trucks. The average time from Tepehuanes to the property is 5 hours.

According to the municipality authorities there is an approved budget for paving another 10 km of the Tepehuanes - Tahuehueto road, and there is the possibility that the State Government and the communication minister of the Federal Government, can further invest in the road to the property.

For the construction stage of the project, it will be necessary to invest in improving some curves of the roads, in order to give access to large trucks that will supply equipment and materials as gas, diesel and gasoline and also for the transport of the future concentrate production to the customers.

18.2 AIRPLANE ACCESS

There are 2 gravel airstrip sites for fixed-wing aircraft, El Purgatorio and Mesa de Toros, but both require renewal of the permit for use of the land. Mesa de Toros is also in need of repair to the access road.

The Company is working on renewing the permit for El Purgatorio, which will be used for the Community and Telson personnel, to have the ability to receive small planes like a Cessna 206 for 6 passengers.

Access can be from either Culiacan or Durango. El Purgatorio airstrip is located 20 km by road north of Tahuehueto and Mesa de Toros is located 5 km by road west of Tahuehueto. The flight from Durango is 45 minutes and from Culiacan 30 minutes. These airstrips will be maintained by Telson.

18.3 POWER SUPPLY AND DISTRIBUTION

The electricity required in the Tahuehueto project for a 500 tpd operation is divided into two 12 hours shifts. The first or daily shift requires power for the full operation of the mine. During the second shift the water pumping and crushing section in the plant as well as the mine are not operating, reducing the demand.

Power lines from CFE (Comision Federal de Electricidad) are not available to provide electricity in Tahuehueto, and the closer lines in Topia 25 km south of Tahuehueto have limited capacity. The investment needed to build a power line to the project at this stage is not an economic option. The options evaluated to supply the 2,500 kW of electricity required for the project has indicated the use of gas engine generators as the method of power generation for this study. The gas generators option is more expensive than the diesel generators in terms of capital cost, but in the short term this difference is compensated for by the fuel saving and in the long term is a better option than diesel engine generators.

Gas engine generators are a common option when a gas pipeline is available and the cost for gas is less than diesel. Gas has a lower price in relation to the diesel and is more efficient. There are no gas pipelines available in Tahuehueto or nearby areas, therefore, the gas would be delivered to the project by tanker truck from Durango.

This option considers two Caterpillar 1,827 kW gas generators, switchgear and synchrony system at 13.8 kV and transformation to 0.48 kV.

18.3.1 HYBRID SOLAR PANEL GENERATORS WITH GAS GENERATORS

There is an option that combines the use of solar panels with the gas generators. During the daytime 30% of the power required can be supplied by solar panels and during nighttime, the power is supplied by the fuel generators.

Solar panels can only supply power during the day and only for a maximum of 30% of the total demand and also require a large piece of land for their installation. The investment on this option is high in terms of units of power generated, but the fuel consumption, operating cost, and pollution are reduced. This option must be considered for long term operation.

The power in any option will be generated at 4,160 volts and transported by power lines to the operating area. In each are by the use of electric substations will be reduced to 440, 220 and 110 volts.

The larger electric engines of the mine, mill and water pumps equipment will use 440 volts, some maintenance and service motors will use 220 volts and general services for housing and camp will be at 110 volts.

18.4 WATER SUPPLY AND DISTRIBUTION

During the exploration stage, Telson was supplying water for drilling and services from the Vueltas River, that is located at the bottom of the canyon at an elevation of 660 m, the plant at 1300 m and the main fresh water tank at 1700 m. The estimated water demand for the 500 tpd operation is 200 liters per minute or 288,000 liters per day.

The Las Vueltas River is the closest and only continuous fresh water source in the area for the operation. The Company has a water concession that can be increased at the required volumes for the planned operation.

The water will be pumped to the water tanks close to the mill and services areas, from the Las Vueltas River at a total length of 2,830 m and a vertical distance of 700 m. The pump station under water concession will be located in the same place used for pump in the exploration stage, the coordinates are X = 337309.834, Y = 2809843.376 and Z = 660. The plant fresh water tank will be located at X = 337128.59, Y = 281237.159 and Z = 1400.

The second pump will be installed in the exploration adit development at river level in an existing outcrop of the mineralization trend of Creston – 5 de Mayo trend. For transport the water a 6” PVC Extrupak pipe will be used. The water from the Las Vueltas River will be pumped to the main mill fresh water tank using a 100 hp pump and from this tank to the 500,000 lts tank with a 60 hp pump.

18.5 ADMINSTATION BUILDING

Designed and constructed to provide sufficient working space for management, engineering, mine supervisors, geology, and all the operations support staff. This facility will be built in the called Industrial area between the mill and tailings dam where there is enough space for this purpose.

This 300 m^2 surface area building will have working space for 8 private offices and a general working area for 18 persons, along with 3 meeting rooms for 20, 12 and 6 persons that will be used for operation meetings and capacitation. The private offices will be for the General Manager and for the head people of each area of the mine.

18.6 MAINTENANCE

A maintenance warehouse facility is provided to service mobile equipment and for storage of equipment spares. The maintenance area includes an overhead crane and is fully equipped including lube racks, washer system, tools control, etc. This maintenance warehouse will include areas for electric and combustion engines and also an office for the head of maintenance.

18.7 WAREHOUSE

A general warehouse will be built for the storage and control of all materials and spare parts required for the total operation of the Mine. Due to the remote location of the property it will be necessary have this facility permanently stock the materials and spare parts for all the equipment and installation.

This warehouse will have a 500 m^2 surface and will include an office for the warehouse personnel. This area will also include the plant, metallurgic lab, hospital and truck wash.

18.8 BUTANE GAS STORAGE

An open area with a slab on grade has been provided to accommodate butane gas and built following all regulatory requirements. It must be installed near the actual Diesel storage and from this place, will be conducted to the power plant facility and other areas by pipe. The installation will have 4 gas tanks for storage 30 m^3 each one for a total capacity of 120 m^3.

18.9 OPEN AREA STORAGE

A chain-link fence with gate and concrete posts, this open storage area is provided for equipment and materials that can be stored outside. It will be necessary to cover 8 weeks of supplies plus all the “C” materials, due to the remoteness of the mine. This open area will be ready for the construction stage in order to have control of the equipment and materials that will be installed on the Mine Unit.

18.10 TRUCK WASH

Trucks and equipment will be washed in an open area near the maintenance building, with the facilities required for a thorough job. It will include a concrete slab and installations to separate water from fuel and lube oils.

18.11 MINE DRY

A separate mine dry, plant and maintenance facility will be provided, that will include showers and lockers to be used by mine, plant and maintenance personnel. The facilities will also have a water treatment system for recycling.

18.12 CAMP

In the main area there isn't specialized manpower, local people of the community will be trained, but some professionals and specialist persons will come from other areas and in the mine zone there are no facilities for stay. Accommodations will be needed for the estimated 55 professionals and specialists required for the full mine as well as the suppliers and contractors that will be visiting and doing temporary work for the operation.

Three accommodation areas will be built for these personnel. The first one will have accommodations for 25 persons for the senior management of the mine, the second camp will have accommodations for another 30 persons from specialized areas of the operation and the third camp, will be the actual accommodations that will be redesigned and will operate like a hotel for contractors and suppliers.

The 3 camp areas will have cooking and dining facilities. The community personnel will supply his lunch. The restaurants will have a complete kitchen and eating and entertainment areas. Breakfast will be served from 6 to 9 am, lunch from 12:30 to 3:30 pm and dinner from 6 to 9 pm. Staff will work in 2 shifts with 2 cooks, and 4 cooks assistants and will serve food to an average of 55 persons. The camps will also have a laundry rooms. Figure 20-13 shows the camp areas and the Figure 20-14 the existing camp that will be redesigned to accommodate visitors.

18.13 ASSAY LABORATORY

Facilities for the assay laboratory will include sample preparation with drying, crushing and pulverizing. An AA spectrometer for multi metal analysis, fire assaying for gold-silver and complete equipment for metallurgical testing is considered. Staff will include 5 people. This lab will have the capacity to assay 150 samples per day from exploration and mine control, from the plant for operation control, from concentrates shipments, water, tailings, etc.

18.14 FUEL STORAGE

Diesel fuel for the equipment will be transported to the mine from PEMEX distributors and stored in 2 existing tanks with 20,000 l of capacity each.

Gasoline will be stored in a 10,000 l tank near the warehouse for use in some gasoline vehicles. Most of the vehicles, pickups, trucks and equipment will use diesel engines.

The natural gas consumption for the power plant generators will be 250,000 l per month and around 10,000 more for the camp and lab. The gas storage will be in 4 tanks with 30,000 l capacity each, for a total storage capacity of 120,000 l.

All the fuel storage tanks and fuel transport equipment will meet existing regulations and safety requirements.

18.15 COMMUNICATION SYSTEMS

A fiber optic network will be installed on site with hot-spots that will handle data, radio, RFID for personnel and equipment. Such system will include underground workings with monitoring ventilation gases etc. thus providing real time control. Everything linked to a satellite communication facility linked to corporate offices.

All the offices, camp and inclusive surrounding areas to the mine will have access to internet and cellular phone services. This will be a contribution of the company to the community.

18.16 WASTE DISPOSAL

The waste water will be treated in a plant and will be used as process water for significant savings in fresh water use.

The solid waste, like oil and maintenance shop discards, will be treated following environmental regulation. Some of this material will be shipped out of the property by a licensed corporation. Other hazardous wastes like batteries will be handled by a third party certified by federal authorities.

The waste produced by the camp and living activities, will be stored in the existing storage facility that meets all existing regulations.

19 MARKET STUDIES AND CONTRACTS

The concentrates produced in the Tahuehueto mine will be tendered to the principal traders in Mexico such as Mercuria Commodities Trading and Trafigura to receive the best sale terms.

Final purchases and delivery terms and conditions will be agreed upon at the beginning of production but standard terms in the industry have been used to estimate revenues for the operation.

20 ENVIRONMENTAL STUDIES, PERMITING AND SOCIAL OR COMMUNITY

The Tahuehueto Project will consist of the exploitation of the underground mineral deposits. All the exploration work completed by Telson has been carefully constructed and adequate closure and restoration measures, as well as erosion control measures, were observed in the exploration areas.

There have been past mining activities at the Project site including the 50 tpd flotation mill operation, tailings dam for this operation, offices, houses, and more than 5000 m in underground exploration developments and exploitation works. That is why the Tahuehueto Project is an impacted terrain and a preventive analysis must be presented to the environmental authorities in parallel with the preparation of the MIA the (Environmental Impact Study) presentation.

Telson is working with the authorities to demonstrate that the new 550 tpd will be a continuation of the previous activities following the current environmental rules, in order to move ahead with the activities at the same time that the MIA is concluded.

The development of the deposit is planned for underground cut and fill mining. The mineralized material from the deposit will be processed at a central process plant. With the development of this project, the main permanent impacts will be the mineral processing facility, the surface portals, the waste rock facility and the tailing disposal facility. The underground cut and fill mining method will eliminate the need for permanent surface waste rock facilities as all waste is schedules to be used in the mine excavation as backfill. The waste rock will need temporary stockpiles near portals until it is needed to backfill the stoped out production areas.

The project site is not located within or near any natural protected area or fragile environment. The environmental controls that will be applied during the development and abandonment of the project will considerably diminish the level of risk and impacts to the natural environment.

Tailing characterization test will be performed to define the environmental protection measures during the construction, operation, maintenance and abandonment of the tailing facility.

The Project will be subject mainly to regulation from the Federal level, through its different stages of development, especially those related to environmental permits for construction and operation, which involves the environmental impact, environmental risk and land use change studies and resolutions. Air emissions, water use, water discharge and hazardous residues handling are also regulated by Federal laws.

Within the state and municipal scope there will be some important formalities to conduct before the construction of the project, such as the license for construction and land use, an endorsement letter for the use and storage of explosives and the permit to dispose the non-hazardous residues at the municipal landfills.

20.1 ENVIRONMENTAL CONSIDERATIONS

20.1.1 REGULATORY REQUIREMENTS

Mine permitting in Mexico is primarily administered by the federal government body, the Secretaría de Medio Ambiente y Recursos Naturales (SEMARNAT). Following from objectives outlined in the Ley General del Equilibrio Ecológico y la Protección al Ambiente, an Environmental Impact Assessment called a

Manifestación de Impacto Ambiental (MIA) will be required in order to gain approvals for construction and operation. SEMARNAT will decide if separate MIAs will be required for the road infrastructure, electrical transmission, and infrastructure. Additional regulatory triggers may require that the MIA for the infrastructure includes a Risk Assessment. Additionally, as a part of the land tenure process, a Change in Land Use Application called a Cambio de Uso de Suelo (CUS), supported by a Technical Supporting Study called an Estudio Técnico Justificativo may be required for all areas of land disturbance.

20.1.2 PREVIOUS STUDIES

To date a study has been carried out by Knight Piésold titled Preliminary Project Development Options and Baseline Data Collection, March 2005. The study considered various options for storage of tailings at the project site. Additionally there were some recommendations for follow up work. As part of the recommendations, an automated weather station has been installed at site to collect relevant weather data.

The metallurgical studies included limited studies for acid rock drainage. The samples were acid generating but had a neutralizing potential as well.

20.1.3 PROPOSED STUDIES

The MIA and CUS required for authorization will require studies to support exploration permitting. Snowden recommends that baseline studies for the MIA be started.

As part of that study, a baseline database should be continued for, climate, soils, surface hydrology, flora, fauna, and socio-economic factors. An environmental monitoring program should be initiated during initial development and throughout mining and milling operations.

More detailed studies including acid base accounting (ABA) and humidity cell testing to determine if the overall samples are acid generating will be needed to determine storage requirements for the tailings and waste rock.

As part of ongoing QC/QA of the climatological data collected at site by the automated weather station, readings could be taken manually for temperature, humidity and rainfall. Manual readings are simple and inexpensive to collect yet give warning of any component failure in an automated weather station. Manual readings can be taken with a simple thermometer, hygrometer and clear view rain gauge.

In addition to the data currently collected, it is recommended that a fauna sighting register begin at the site. The register would consist of a log book and list all animals viewed in the field listing species, location, date and time, observer, weather conditions, period of observation and comments. This can provide good background for the baseline fauna study to be completed.

20.1.4 IDENTIFIED ISSUES

No other environmental issues have been identified that would alter or compromise Project planning. The project is located outside of any protected natural areas or some other conservation program at the municipal, state or federal level.

20.2 ENVIRONMENTAL PROGRAM

The Tahuehueto Project will comply with all the environmental regulations and standards in place in Mexico as well as applicable international criteria. The mining works and supporting facilities will be designed, constructed and operated in such a way as to minimize the impact to the natural environment. An environmental management plan, an emergency response plan, a residues management plan and a closure and reclamation plan will be some of the most important documents to develop and implement early in the development of this project. A systematic environmental monitoring program of surface and underground water, creek sediments, soil, air, flora and wildlife conditions will be implemented, before, during and after mine operations.

20.2.1 SURFACE WATER MANAGEMENT

All the mining facilities will be protected from pluvial waters by means of a system of diversion channels such as channels and/or road-gutters constructed around the different facilities footprint to direct water runoff further downstream.

Fine particle migration will be the main potential impact to surface water from the waste storage facilities. Surface drainage from the different mine areas, such as waste rock facilities, tailings or other disturbed area, will be directed into sedimentation basins or ponds for sediment settlement before discharging into natural water courses. Ponds and process areas will be constructed on an impermeable double layer system and will operate on a close circuit with no discharge to the natural environment. The capacity of the containment ponds will be estimated based on a 100 yr-24 hr rain event.

A sampling program of main surface streams at the project area should be implemented on a quarterly basis as part of the environmental baseline studies.

20.2.2 GROUNDWATER MANAGEMENT

A hydrogeology study should be done to investigate the depth of the underground water and the aquifer vulnerability at the area where the tailing site is projected. The hydrogeology evaluations should also support the installation of wells for monitoring purposes. A monitoring program of underground water should be implemented, at least twice a year, on wells located close and upstram and downstream from the tailing site, leach pad, process plant and the waste rock dumps.

20.2.3 AIR QUALITY MANAGEMENT

No air quality data is available for the project area. The type of air pollution expected from the mining development will be the emission of particles (dust) mainly from the material handling operations, the service road traffic and the rock crushing therefore the concentration of PST (total suspended particles) and PM-10 (suspended particle less than 10 microns) should be monitored in the surrounding environment before and during the mine operations. Potential emissions from the Dore production furnace should also be monitored, if produced on site.

20.2.4 WILDLIFE MANAGEMENT

Prior to any vegetation clearing a rescue and protection program will be executed to relocate any flora species of interest. The wood to be produced from the clearing of the areas may be provided to the nearby communities for its management. No hunting will be allowed to workers and contractors of the Project, adequate fencing will be maintained on all operational areas, to protect wild life.

20.2.5 SOIL MANAGEMENT

Previous to the stripping and excavation of the land, the fertile layer of soil will be recovered and stored for future ecological restoration of the disturbed areas. Hydraulic works will be constructed for erosion control and soil conservation purposes. The works may consist of ditches, gabions, berms and/or rock mulch structures.

20.3 RECLAMATION AND CLOSURE ACTIVITIES

A detailed Closure and Reclamation Plan (CRP) will be prepared should this project proceed with the construction. In general terms, the CRP for the Tahuehueto Project will address the principles and guidelines of environmental regulation in Mexico as well as international best practices for the closure of this type of project. Concurrent reclamation will take place during the mine operation, although the majority of the work will occur after completion of mining and processing.

The objectives of the CRP will be based on the post-mining use of the land in addition to the following objectives: protection of public health and safety, protect the water resources, landforms stabilization and revegetation, preparation of the land for long term productive use and/or establishment of wildlife habitat.

20.3.1 MINE OPENINGS

Surface water diversion structure will be constructed around the mine openings. The end of mine design will leave stable pit walls. A protection fence and/or berms will be installed around mine openings to restrict the access of persons and fauna. Land scarification and seeding will be done on the surface disturbances to promote the development of vegetation cover. Partial backfilling of the openings and portals with waste rock produced during mining will be evaluated.

20.3.2 WASTE ROCK AND TAILINGS STORAGE FACILITIES

Slopes should be adjusted to provide a static and dynamic stability conditions. Erosion control structures will be constructed as necessary, such as rock armoring on the exterior slopes of the embankment and slope grading for stability.

The waste rock and tailings storage facilities will be contoured and graded for adequate surface water drainage. Water diversion features to control runoff around and within the tailing facility will be enhanced to avoid long term impact on the surface and underground water.

The waste rock and tailings should be characterized to determine the potential for acid drainage generation and metal leaching. Based on the results of the geochemical characterization of the material, a cover with inert and impermeable material may be necessary to avoid water infiltration and formation of toxic leachate to the natural environment. On top of this layer, another layer of top soil or organic material will be placed and native seeds planted to promote the development of a vegetation cover. Downstream sedimentation ponds will function as sediment traps for runoff water, before reaching the natural water courses.

20.3.3 PONDS

The solutions remaining in the ponds should be unloaded or evaporated in a safe manner. After the removal of pumping equipment, the plastic liner should be removed and the pond should be filled back with inert material and soil, allowing a good drainage out of this area. Land scarification, seeding and reforestation will complete the restoration of the pond sites

20.3.4 PROCESS FACILITY

All remaining substances and residues will be removed from the process areas. The Hazardous residues generated will be delivered to an authorized firm for final disposal or recycling. Tanks containing chemical substances and fuels will be cleaned before removing them from the site. All facilities and equipment will be dismantled and buildings demolished.

20.3.5 POST CLOSURE ACTIVITIES

After closure of the mine, the reclamation activities may be completed in short order, a periodic monitoring and maintenance program will continue for the necessary years until stable and safe conditions are achieved in all disturbed areas. Mine design, construction and operation should incorporate measures to minimize the requirements for care and maintenance at the abandonment stage.

The post-closure monitoring actions will include: underground opening closures, safety conditions, surface and underground water quality, air quality, vegetation development and wildlife reestablishment.

The quality of underground water and surface water bodies should be done on a quarterly basis. The monitoring sites should be within the drainage paths influenced by the tailing facility and the waste rock dumps.

Drainage and erosion control features should be maintained and/or constructed until stable conditions are observed on the soil and vegetation cover that will prevent from further erosion.

As the restored areas are functioning and the environmental monitoring indicates no harmful conditions, the frequency and intensity of the inspections and monitoring will be reduced until delivering the land for a post-mining use. An important goal of the mine reclamation will be the re-establishment of the vegetation cover, encouraging the natural succession of native species.

20.4 SOCIAL AND COMMUNITY

The closest important community to Tahuehueto is Tepehuanes (10,745 inhabitants), which has infrastructure and public services coverage such as: hotels, restaurants, electricity, cellular phone service, potable water, sewage system, landfill and airstrip for small planes.

Main economic activities at the region where the Tahuehueto project is located are cattle raising, agriculture, and forestry. The Project seems to be very well accepted and expected by most of the people living in the region. Job opportunities, services infrastructure improvement and technical training of local people are some of the most important benefits that the development of the Project will provide to the region. The local populace will also have access to potential employment opportunities, subject to the evaluation of individual qualifications, education, experience, as well as expectations for strict compliance with the code of conduct established for the Telson workforce and contractor staff.

There are no known archaeological sites or areas of significant cultural interest within the Project concession. Implementation of a Community Relations Management Plan for the Project will also provide the means of detecting and appropriately responding to any changing stakeholder views with respect to cultural heritage concerns, as well as employment or contracting opportunities, health and safety, and other social considerations.

20.5 CURRENT PROJECT ENVIRONMENTAL PROCEDURES

Telson, on April 13, 2016, filed a preventative report named “Informe Preventivo – Exploracion Nom 120” with SEMARNAT, the Mexican government responsible authority for filing of notices of work. This document notified and established Telson’s plans to commence underground exploration activities, as well as the rehabilitation of existing roads and other additional infrastructure. Included in the company’s work plans was the rehabilitation of the on-site mill processing and benefit plant. The rehabilitation of the benefit plant does not require authorization in environmental matters, according to the article 5, section L, subsection III, of the regulation of the Ley General del Equilibrio Ecologico y Protección al Ambiente and, for this reason, those rehabilitation activities were able to commence at any time.

On October 12, 2016, Telson submitted an environmental impact study named a “Manifestacion de Impacto Ambiental” (MIA) to SEMARNAT, the responsible Mexican governmental authority. This MIA outlined the company’s plans to recommence commercial exploitation/production at Tahuehueto with the construction of new portals, access roads, establishment of waste rock and ore storage patios, construction of a new camp, rehabilitation of the existing tailing disposal and other infrastructure items. To date, no comments have been returned to the Company by SEMARNAT. Telson anticipates a positive authorization to be granted prior to end of March 2017. Upon final authorization of this MIA, mining and mill processing operations can begin using the previously impacted area of the former mine which include the existing tailings storage facilities reported to have capacity, at 500 tonnes per day, for approximately 1.5 to 2 years.

Telson’s environmental consultant has prepared a second draft Informe Preventivo report based on NOM-141-SEMARNAT- 2003, seeking to permit the construction of the new tailings storage facility. In conjuction with this Informe Preventivo report the consultant has also prepared a draft technical study for the change of use of soil in areas where removal of vegetation is necessary (new waste rock storage areas, new tailings storage facility etc.), named Estudio Tecnico Justificative Para Cambio De Uso de Suelo. Filing of these two documents are awaiting the completion of this PFS to include as a supporting document to be included in the filing. Telson intends to immediately proceed with the filing of these documents with the authorities to permit the planned expansion of the mine past the 1.5 to 2 year capacity of the current tailings storage facility

21 CAPITAL AND OPERATING COST

Capital and operating costs used for the Tahuehueto Project were developed from cost build up from first principles engineering along with vendor and contractor quotations. In addition, all available project technical data and metallurgical test work were considered to build up a processing operating cost estimate.

A project configuration which included the underground mines and a central process facility was developed as the basis for capital cost estimation. Preliminary site infrastructure alternatives (process plant, tails storage facility, and power) were examined as a basis to estimate costs. Generalized arrangements were evaluated to establish a physical basis for the capital costs estimates. Cost accuracy is estimated to be + or - 20%.

21.1 CAPITAL COSTS

Capital costs were developed based production rates and from design assumptions. The costs are collected in two separate categories; initial capital (construction costs to initiate mining operations including Engineering, Procurement, and Construction Management (“EPCM”), mining and processing equipment, and contingency), and sustaining capital (additional equipment and equipment rebuilds). The estimated capital costs are listed in Table 21.1. Contingency was calculated on applicable items at a rate of 20%. Contingency was applied to all direct initial capital cost items. The contingency rate was determined based on confidence levels on capital used in the cost build up.

Table 21.1 Tahuehueto Total Capital Costs

Capital Category	Initial ($M)	Sustaining ($M)	Total ($M)
Mine Mobile Equipment	8.8	4.4	13.2
Mine Fixed Equipment	1.1	0.3	1.4
Mine Development	5.1	0.7	5.8
Processing	16.1	-	16.1
Infrastructure	1.1	-	1.1
Total CAPEX	**32.2**	**5.4**	**37.6**

21.1.1 INITIAL CAPITAL COSTS

The initial capital costs are listed in Tables 21.2 and 21.3 for mining and processing respectively. They consist of costs to be incurred after project approval, and after construction and operating permits have been received. The costs were assumed to occur in years -1 and -2, and include all capital costs up to the start of production. The scope of the initial capital includes direct capital costs and indirect costs. Direct capital costs include construction process facilities, establishment of the mining facilities, and purchase of fixed and mobile mining equipment. Mining and milling equipment costs are based on quotations for specific units and mining cost services where specific quotes were not available. Major equipment is brand new and only the ball mill is used refurbished equipment. Building and facilities procurement and construction costs are estimated by contractor quotations based on dimensions, average construction costs, plus the extra cost for building in a remote area.

Table 21.2 Mine Initial Capital Costs

Mine Capital Category	Cost $M
Mine Mobile Equipment Purchases	7.3
Mine Fixed Equipment Purchases	0.9
Mine Development	4.2
Contingency (20%)	2.5
Total CAPEX	**14.9**

Table 21.3 Processing Initial Capital Costs

Process Capital Category	Cost $M
Mill Equipment Purchases	9.5
Mill Construction	3.3
Tailings and Earthwork	0.6
Infrastructure	0.9
Contingency (20%)	2.9
Total CAPEX	**17.2**

21.1.2 SUSTAINING CAPITAL COSTS

Sustaining capital costs include capital expenditures during production years required to maintain capacity and meet mine production and mining condition needs. Mobile equipment was assumed to require re-build at 25% of original cost approximately every 15,000 hours of operation. Mining fixed equipment sustaining capital was primarily related to purchase of ventilation fans and mine development sustaining capital was for extension of the ore pass system. Sustaining capital costs are shown in Table 21.4.

Table 21.4 Sustaining Capital Costs

Mine Capital Category	Cost $M
Mine Mobile Equipment Rebuilds	3.7
Mine Fixed Equipment Purchases	0.3
Mine Development	0.6
Contingency (20%)	0.9
Total CAPEX	**5.4**

21.2 OPERATING COSTS

Operating unit costs are based on first principle engineering cost buildups for mining and processing operations. These costs were derived from materials, power, fuel consumption, and labor for each unit operation. Labor costs for the Project were estimated based on actual labor costs plus labor overheads for mine workers in Mexico supplied by Telson. Consumables and fuel were primarily based on vender quotes, however equipment operating consumables and maintenance consumables were based on costing services. The mining cost build up on a total tonne basis (ore or waste) was similar to a comparable cost structure supplied by a contractor estimate for drift development mining of 5 m drifts at Tahuehueto. Operating costs are listed in Table 21.5.

Table 21.5 Tahuehueto Unit Operating Costs

Operating Costs	LOM Cost $M	Unit Cost $/t mineralized
Mining	69.4	21.62
Processing	100.5	30.80
G&A	22.2	6.82
Total OPEX	**192.2**	**59.24**
Smelter	78.0	23.91
Freight & Marketing	15.7	4.80
Royalties	9.5	2.91
Total Operating Cash Cost	**295.4**	**90.86**

22 ECONOMIC ANALYSIS

The economic performance of the Tahuehueto Project was evaluated with a cash flow based economic model using project costs and revenues as the financial basis. The revenue factors for the project are dependent on metal prices calculating into the net smelter return. The metal prices used for analysis are shown in Table 22.1. The NSR of the three concentrates produced in the mill are summarized in Table 22.2

Table 22.1 Metal Prices for Economic Analysis

Economic Metrics	Units	LOM Value
Gold Price	$/oz	1,180
Silver Price	$/oz	16.70
Copper Price	$/lb	2.65
Lead Price	$/lb	0.87
Zinc Price	$/lb	0.92

Table 22.2 Net Smelter Return Summary

Item	Gross Revenue (k$)	Deductions (k$)	Freight & Marketing (k$)	NSR (k$)
Lead Concentrate	454,900	39,800	4,700	410,400
Copper Concentrate	69,000	4,700	400	64,000
Zinc Concentrate	172,000	43,800	11,800	116,400
TOTAL	**695,900**	**89,300**	**17,000**	**590,800**

- Totals may not sum due to rounding.

22.1 NET SMELTER RETURN

The economic driver of the project is based on the net smelter return. The costs and payable metal values from the produced concentrates are calculated to give the NSR. The detailed NSR from the Tahuehueto mill productions is detailed in Tables 22.3 through 22.5.

Table 22.3 Lead Concentrate NSR

Item	Au	Ag	Cu	Pb
Distribution (% Recoveries)	77.1%	62.8%	31.6%	85.5%
Payable (%)	94.9%	94.9%	83.6%	94.5%
Gross Revenue (k$)	324,300	46,000	21,000	63,600
Deductions (k$)	18,400	3,300	4,200	13,900
Revenue (k$)	305,900	42,700	16,800	49,700
Freight & Marketing (k$)	4,700			
NSR (k$)	**410,400**			

Table 22.4 Copper Concentrate NSR

Item	Au	Ag	Cu
Distribution (% Recoveries)	6.8%	10.3%	51.4%
Payable (%)	94.8%	94.9%	95.8%
Gross Revenue (k$)	28,800	7,500	32,700
Deductions (k$)	1,500	400	2,800
Revenue (k$)	27,300	7,100	30,000
Freight & Marketing (k$)	400		
NSR (k$)	**64,000**		

Table 22.5 Zinc Concentrate NSR

Item	Au	Ag	Zn
Distribution (% Recoveries)	11.0%	11.7%	80.0%
Payable (%)	93.5%	89.9%	84.8%
Gross Revenue (k$)	46,200	7,500	118,400
Deductions (k$)	3,200	800	39,800
Revenue (k$)	42,900	6,700	78,600
Freight & Marketing (k$)	11,800		
NSR (k$)	**116,400**		

22.2 CASH FLOW

The production schedules presented in Section 16 and 17 have been used in conjunction with the cost data discussed in Section 21 to create a model for the Telson's Project's economic performance. Costs are in constant 2016 US$, no escalation of cost has been assumed. Operating costs are generated based on production physicals (tonnes) and unit rates. The detailed cash flow model for the Tahuehueto Project is presented in Table 22.6. The results for economic analysis are summarized in Table 22.7.

Table 22.6 Tahuehueto Project Cash Flow

Costs ($k)	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	Total
Mining Operating	-	-	2,700	3,600	3,500	3,600	3,600	3,500	3,500	3,500	3,600	3,500	3,600	3,500	3,600	3,500	3,600	3,500	3,600	3,700	3,000	2,100	1,400	-	69,400
Processing Operating	-	-	4,900	4,900	4,900	4,900	4,900	4,900	4,900	4,900	4,900	4,900	4,900	4,900	4,900	4,900	4,900	4,900	4,900	4,900	4,600	3,800	3,300	-	100,500
General and Administrative	-	-	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,000	800	600	-	22,200
Total OPEX	**-**	**-**	**8,800**	**9,600**	**9,600**	**9,600**	**9,600**	**9,600**	**9,500**	**9,500**	**9,600**	**9,600**	**9,600**	**9,400**	**9,700**	**9,600**	**9,600**	**9,600**	**9,600**	**9,700**	**8,500**	**6,700**	**5,300**	**-**	**192,200**
Mining Initial Capital	-	12,400	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	12,400
Mining Sustaining Capital	-	-	500	200	-	-	800	700	300	-	100	-	-	900	700	-	300	-	-	-	-	-	-	-	4,500
Processing Initial Capital	1,200	13,100	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	14,400
Total CAPEX	**1,200**	**25,600**	**500**	**200**	**-**	**-**	**800**	**700**	**300**	**-**	**100**	**-**	**-**	**900**	**700**	**-**	**300**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**31,300**
Contingency (20%)	200	5,100	100	-	-	-	200	100	100	-	-	-	-	200	100	-	100	-	-	-	-	-	-	-	6,300
Working Capital	-	-	4,600	(1,700)	900	(500)	(300)	400	-	(200)	(600)	-	500	700	100	(300)	(500)	500	900	(400)	(600)	(1,100)	(400)	(2,200)	2,200
Total Cost	**1,500**	**30,700**	**13,900**	**8,100**	**10,600**	**9,200**	**10,300**	**10,900**	**9,900**	**9,300**	**9,100**	**9,600**	**10,100**	**11,200**	**10,600**	**9,300**	**9,500**	**10,100**	**10,500**	**9,300**	**7,900**	**5,600**	**4,900**	**(2,200)**	**231,900**
Revenue-NSR	**-**	**-**	**35,000**	**25,200**	**30,800**	**27,900**	**26,000**	**28,700**	**28,500**	**27,100**	**23,600**	**23,700**	**27,000**	**31,100**	**31,900**	**30,000**	**27,300**	**30,300**	**36,100**	**33,500**	**29,100**	**20,700**	**17,300**	**-**	**590,800**
Royalty (1.6%)	-	-	600	400	500	400	400	500	500	400	400	400	400	500	500	500	400	500	600	500	500	300	300	-	9,500
Pre-Tax Cash Flow	**(1,500)**	**(30,700)**	**20,500**	**16,700**	**19,800**	**18,300**	**15,300**	**17,300**	**18,200**	**17,300**	**14,100**	**13,700**	**16,400**	**19,500**	**20,900**	**20,300**	**17,300**	**19,800**	**25,000**	**23,700**	**20,700**	**14,800**	**12,100**	**2,200**	**351,500**
Special Mining Tax (7.5%)	-	-	1,900	1,600	1,900	1,700	1,400	1,600	1,700	1,600	1,300	1,300	1,600	1,900	2,000	2,000	1,700	1,900	2,400	2,300	2,000	1,400	1,200	200	36,400
Special Mining Royalty (0.5%)	-	-	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	-	2,200
Depreciation (10%)	-	-	3,200	3,200	3,200	3,200	3,200	3,200	3,200	3,200	3,200	3,200	-	-	-	-	-	-	-	-	-	-	-	-	32,200
Income Tax (30%)	-	-	5,100	4,000	4,900	4,500	3,600	4,200	4,500	4,200	3,200	3,100	4,900	5,800	6,200	6,000	5,200	5,900	7,400	7,100	6,200	4,400	3,600	600	104,800
Post-Tax Cash Flow	**(1,500)**	**(30,700)**	**13,300**	**11,000**	**12,900**	**12,000**	**10,200**	**11,400**	**11,900**	**11,400**	**9,500**	**9,200**	**9,800**	**11,700**	**12,500**	**12,100**	**10,400**	**11,800**	**14,900**	**14,200**	**12,400**	**8,900**	**7,300**	**1,300**	**207,900**
Cumulative Cash Flow	**(1,500)**	**(32,200)**	**(18,900)**	**(7,800)**	**5,000**	**17,000**	**27,200**	**38,600**	**50,600**	**62,000**	**71,500**	**80,700**	**90,500**	**102,200**	**114,700**	**126,800**	**137,200**	**149,000**	**163,900**	**178,100**	**190,500**	**199,300**	**206,600**	**207,900**	**207,900**

Table 22.7 Tahuehueto Project Economic Results

Economic Metrics	Units	LOM Value
Total Ore Processed	kTonnes	3,264
Contained Gold Produced	kOz	340
Contained Silver Produced	kOz	3,720
Contained Lead Produced	kLb	73,100
Contained Copper Produced	kLb	20,800
Contained Zinc Produced	kLb	128,700
Gold Price	$/oz	1,180
Silver Price	$/oz	16.70
Copper Price	$/lb	2.65
Lead Price	$/lb	0.87
Zinc Price	$/lb	0.92
Gross Revenue	$M	590.8
Refining and Freight Costs	$M	91.3
Royalty (1.6%)	$M	9.5
Operating Costs	$M	192.2
Capital Costs	$M	37.6
Pre-Tax Cash Flow	$M	351.5
Special Mining Tax (7.5%)	$M	36.6
Special Mining Royalty (0.5%)	$M	2.2
Income Tax (30%)	$M	104.8
Post-Tax Cash Flow	$M	207.9
Pre-tax NPV (8%)	$M	137.8
Pre-tax IRR	%	56%
Post-tax NPV (8%)	$M	77.0
Post-tax IRR	%	36%

22.3 SENSITIVITY ANALYSIS

Post-tax sensitivity of the projected economic performance was evaluated by varying the revenue, operating cost and capital cost over a range of 80% - 120% of the base case assumptions. The sensitivity of projected economic performance to variation in metal prices and metallurgical recovery is listed in Tables 22.8. Tables 22.9 and 22.10 lists the variation of post-tax NPV and post-tax IRR for ranges for revenue operating cost and capital cost. The results are shown graphically in Figure 22-1 and 22-2, for post-tax NPV at a discount rate of 8% and post-tax IRR, respectively.

Table 22.8 Sensitivity of Post-Tax NPV and IRR to Variation of Revenue

Total Revenue Sensitivity					
Factor	NPV (US$M)				IRR
	10%	8%	5%	0%	
120%	106.3	129.7	178.7	326.1	55%
116%	97.3	119.2	164.9	302.4	51%
112%	88.3	108.6	151.0	278.8	48%
108%	79.4	98.1	137.2	255.2	44%
104%	70.4	87.5	123.4	231.5	40%
100%	**61.4**	**77.0**	**109.5**	**207.9**	**37%**
96%	52.5	66.4	95.7	184.3	33%
92%	43.5	55.9	81.8	160.7	29%
88%	34.5	45.3	68.0	137.0	25%
84%	25.6	34.8	54.2	113.4	22%
80%	16.6	24.2	40.3	89.8	18%

Table 22.9 Sensitivity of Post-Tax NPV IRR to Variation of Operating Cost

OPEX Sensitivity					
Factor	NPV (US$M)				IRR
	10%	8%	5%	0%	
120%	46.7	59.6	86.8	169.5	31%
116%	49.7	63.1	91.4	177.2	32%
112%	52.6	66.6	95.9	184.9	33%
108%	55.5	70.0	100.5	192.5	34%
104%	58.5	73.5	105.0	200.2	35%
100%	**61.4**	**77.0**	**109.5**	**207.9**	**37%**
96%	64.4	80.4	114.1	215.6	38%
92%	67.3	83.9	118.6	223.3	39%
88%	70.3	87.4	123.1	231.0	40%
84%	73.2	90.8	127.7	238.7	41%
80%	76.1	94.3	132.2	246.4	42%

Table 22.10 Sensitivity of Post-Tax NPV and IRR to Variation of Capital Cost

CAPEX Sensitivity					
Factor	NPV (US$M)				IRR
	10%	8%	5%	0%	
120%	56.1	71.5	103.8	201.6	31%
116%	57.2	72.6	105.0	202.9	32%
112%	58.2	73.7	106.1	204.2	33%
108%	59.3	74.8	107.2	205.4	34%
104%	60.4	75.9	108.4	206.7	35%
100%	**61.4**	**77.0**	**109.5**	**207.9**	**37%**
96%	62.5	78.1	110.7	209.2	38%
92%	63.6	79.1	111.8	210.4	39%
88%	64.6	80.2	112.9	211.7	41%
84%	65.7	81.3	114.1	212.9	43%
80%	66.7	82.4	115.2	214.2	44%

Figure 22-1 Post-Tax 8% NPV Sensitivity to Variation of Revenue, Operating Cost and Capital Cost

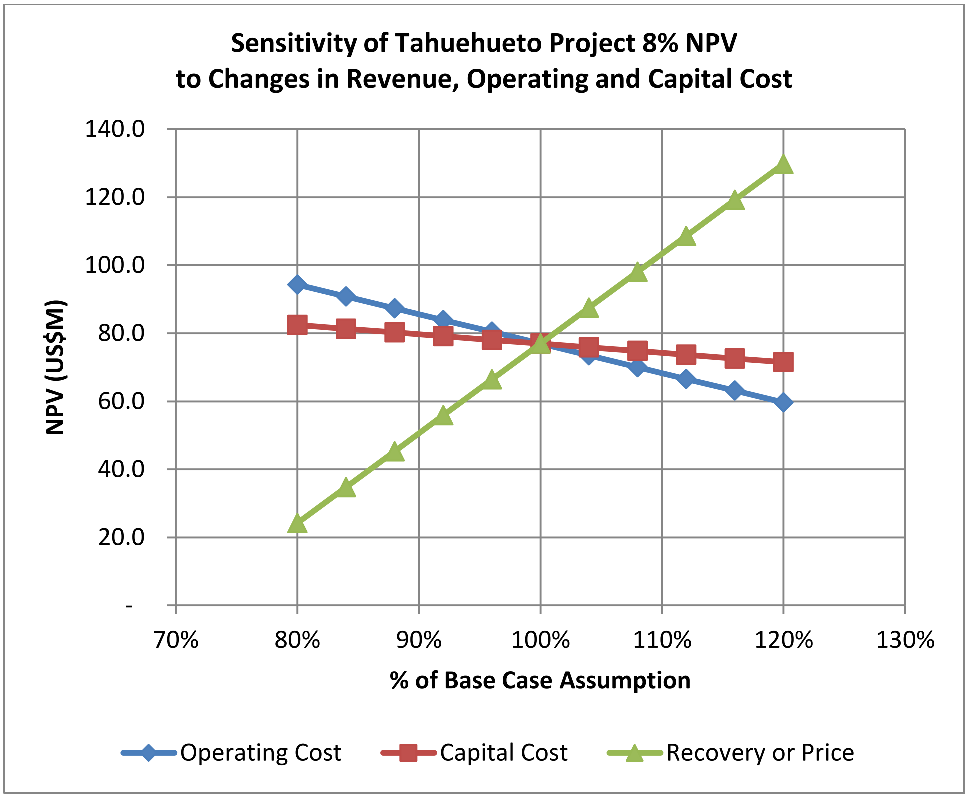
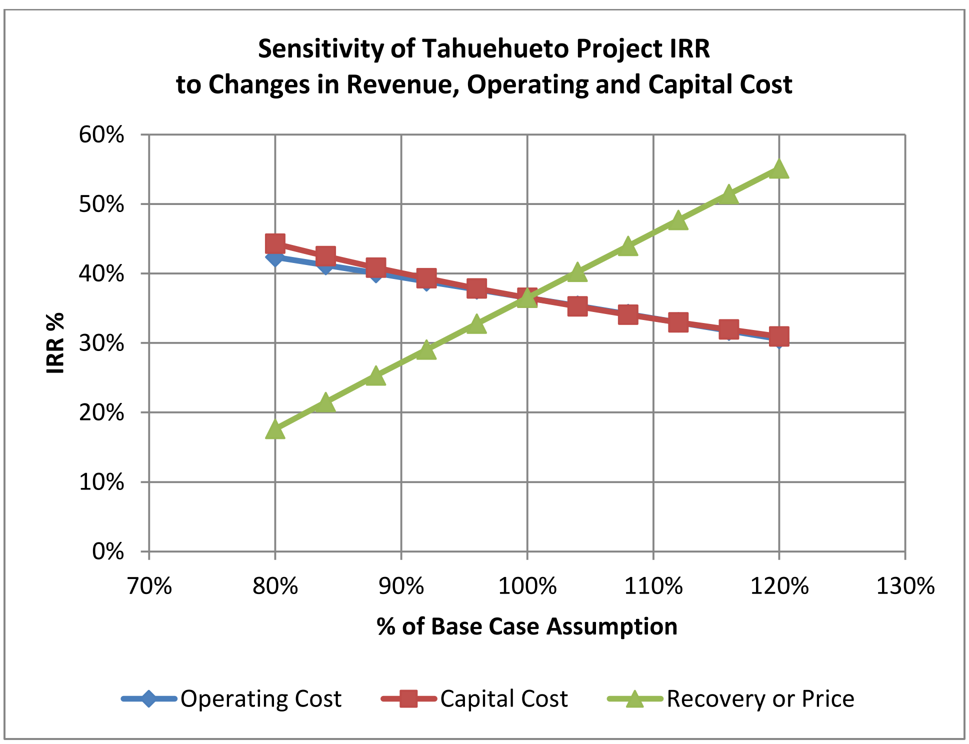


Figure 22-2 Post-Tax IRR Sensitivity to Variation of Revenue, Operating Cost and Capital Cost



23 ADJACENT PROPERTIES

There is no relevant information of adjacent properties which materially affect the opinion offered in this report.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 INDUSTRIAL SCALE BULK SAMPLE METALURGICAL TESTING

In order to confirm the previous results of the flotation processing an industrial scale, Telson implemented a 3,500 t industrial metallurgic test, in an existing flotation 120 tpd mill near Guanacevi, Durango for processing mineralized material from Tahuehueto.

The material from Tahuehueto came from the planned mine development Level 10 and was trucked to the Andes Mill near Guanacevi. Based on previous metallurgical testing, it was estimated that the produced test concentrates would be marketable and saleable. The revenue would help offset the cost of the industrial bulk metallurgical test. The estimated costs using contractors for the test are provided in Table 24.1.

Table 24.1 Estimated Bulk Metallurgical Test Costs

Description	Cost (US$)
Access Road Improvements	37,000
Mine Prep and Development of Levels 11 and 12	261,069
Mine Sample Material Production (5,000 t)	108,000
Haulage/Freight to Mill	285,714
Material Processing Cost	225,000
Concentrates Freight and Transportation Costs for Sale	11,412
Estimated Cost for Industrial Bulk Met Test	**928,195**
Estimated Sales Revenue of Test Concentrate Sales	**919,430**

In August 2016, Telson initiated the collection of the planned 2,000 bulk tonnes sample from the El Creston area on its Tahuehueto Project. The final test consisted of a total of 3,500 t to be processed. The material was shipped in conventional dump trucks for processing at a new 120 t/d flotation mill facility. Processing of this industrial scale bulk sample will allow fine tuning of grinding parameters, reagent types and reagent concentrations to be utilized, and allow the refining of milling techniques to improve on the recoveries that have already been demonstrated during bench scale metallurgical testing conducted by Telson in 2010. Other information that this test will provide include proof of concept of the selective mining method, ground support, and mining costs.

24.1.1 BULK SAMPLE COLLECTION

During the collection of the bulk sample a staff member from MMC was on the project site and was able to witness the bulk sample collection procedures. Extra samples were collected to verify the head grade and overall diluted grades being shipped to the processing facilities. The samples were collected following Telson's sampling procedures and were sent to a laboratory in Matehuala, San Luis Potosi, Mexico. After assays were received, independent samples were sent to be processed at a certified lab in Reno, Nevada to compare results. The results were collected and verified by MMC. A summary is described in the following subsections.

24.1.1.1 EL CRESTON LEVEL 10 PORTAL

Thirteen (13) samples, some of which are shown in Figure 24-1, were found to be logged in from the metallurgical test material. It was observed that all of the material that had been mined for testing was in piles outside of the portal waiting to be transported to a holding pad. Figure 24-2 is a photograph illustrating the type of information recorded on the sample bags.

One sample per shift was taken and logged in to the data record. An EJC LHD, illustrated by the photograph in Figures 24-3 and 24-4, was used to remove the sample material from the mine and place it in piles outside the portal.

Each shift produced about 20 bucket loads. The miners were observed at work, staging the piles of ore and sampling each one. The sampling process, shown in Figures 24-5 and 24-6, began with one of the workers taking a shovel and collecting material from various parts of each fresh pile removed from the mine. Each shovel of the sample material was placed on a large rubber mat.

Figure 24-1 Photo of Samples Collected from El Creston Level 10 for Metallurgical Bulk Test

Figure 24-2 Example of Data collected for samples from El Creston Level 10

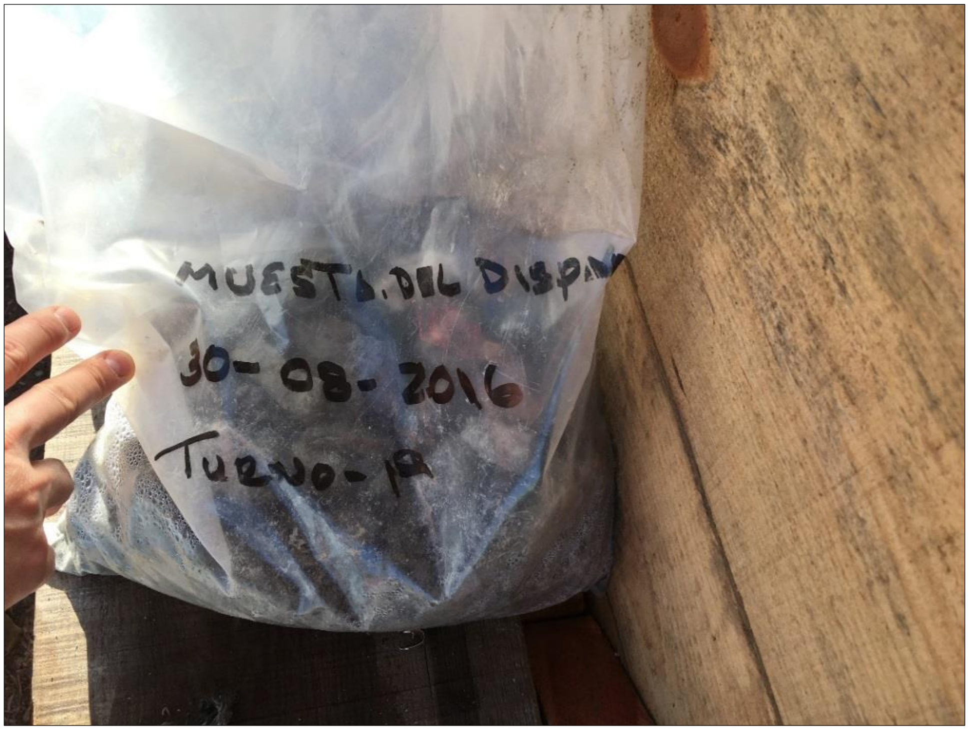


Figure 24-3 EJC Loader at El Creston Level 10

Figure 24-4 EJC Loader Working the El Creston Level 10 Metallurigical Bulk Test

Figure 24-5 Photo of Miner Taking Samples from Fresh Muck, El Creston Level 10

Figure 24-6 Photo of Miner Taking Samples from Fresh Muck, El Creston Level 10

Large pieces of the sample material were broken down with a hammer so that all of the pieces are roughly uniform in size of pebbles or smaller. (~4-5 cm), as shown in the photograph in Figure 24-7. Once all material for the shift had been brought out and sampled, the sample material on the mat was hand mixed to homogenize the entire sample. After the material is mixed together it was separated into quarters with one quarter chosen for lab analysis.

Each shift sample weighed between 8 and 9 kilograms. Two additional quarter sample were collected with two documented. A third quarter of the material was also retained for any required follow-up.

The waste rock in the samples was very recognizable and was separated from the ore piles. The miners are shown picking out large chunks of waste and removing them from the ore in Figure 24-8. However, some waste rock was retained in the samples to represent dilution remaining in the piles.

Figure 24-7 Photo of Miner Breaking Down Material from Samples, El Creston Level 10

Figure 24-8 Photo of Miners Removing Waste Rock from Muck Pile, El Creston Level 10

The shift samples to measure the head grade were sent to the lab of Altiplano Goldsilver in Matehuala SLP. It was requested that the pulps be sent back to Denver so that the assays could be checked by a certified lab to compare the results.

The holding pad where the 2,000 tons of material for the testing was placed and then loaded into larger haul trucks for transport to the mill was inspected for any abnormalities. It was an open, flat clearing that would accommodate the total amount of ore, and is shown in the photographs in Figures 24-9 and 24-10. The destination mill was called Planta Andes and is located in Aguacaliente, Municipality of Guanaceví.

Figure 24-9 Holding pad for Metalurgical Bulk Test Material

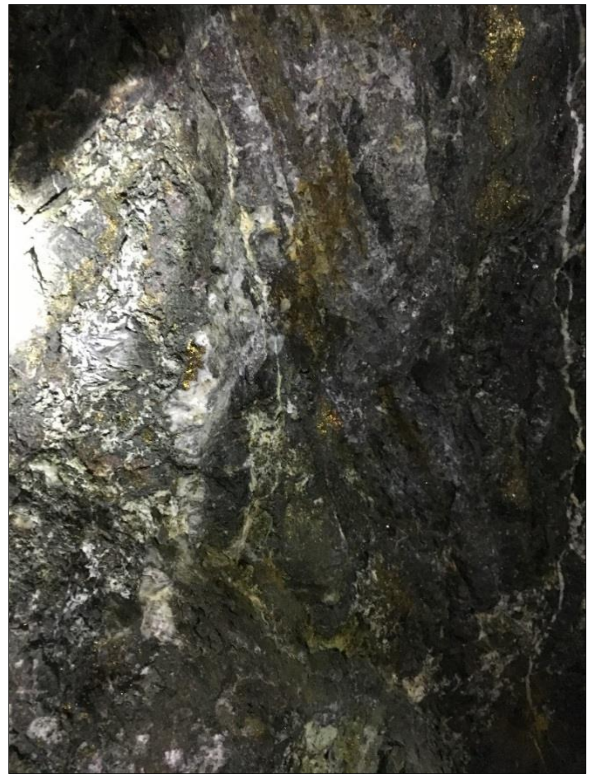
Figure 24-10 Holding pad for Metalurgical Bulk Test Material

24.1.1.2 CINCO DE MAYO

In the Cinco de Mayo vein, a 25 kg test sample has been excavated for metallurgical purposes. The ore veins were easily observed within the drift walls. They were observed to be rich in the various base metal mineralization and appeared to contain significant amounts of silver mineralization, as illustrated by the photographs in Figure 24-11 and 24-12. No lab sample was taken from this vein, and no results were provided at the time of this report.

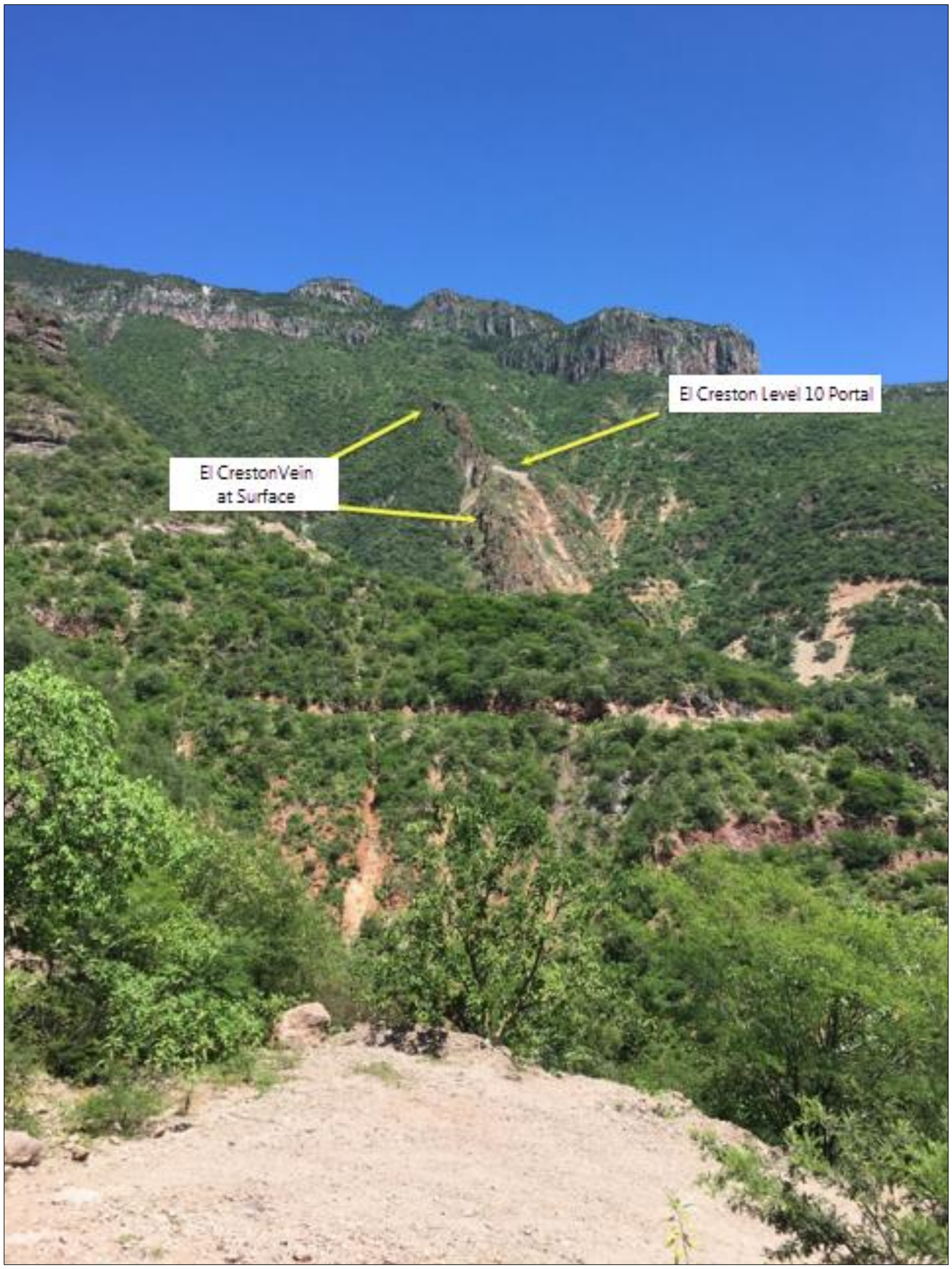
Figure 24-11 Copper Veins at Cinco de Mayo

Figure 24-12 Copper Veins at Cinco de Mayo



The El Creston vein was impressively continuous through a large area of the valley, as shown by the photograph in Figure 24-13.

Figure 24-13 Photo of El Creston Vein Outcrop and Portal



Two Certified Reference Materials (CRM) were received from Natural Resources Canada. These two samples were purchased to accompany the Thuehueto samples being sent to Denver from Mexico. The samples and CRMs were then sent to ALS Global in Reno, Nevada to check assays from the lab in Alitplano Goldsilver Mexico. One standard was a gold ore standard, while the second standard was a base metal standard containing Zinc-Tin-Copper-Lead ore. The two standards were accompanied by two sand samples to test the check of the lab and be sure that they are being tested properly.

Of the samples received from Mexico, five of the samples were independently collected while the other 5 were various other shift samples collected by Telson. These will be sent to ALS Global to compare results.

24.1.1.3 RESULTS

Table 24.15 lists and compares the assay data from the samples taken independently and the regular samples taken by the miners on shift.

Figure 24-14 Tahuehueto Samples comparing assays from AGS and ALS

Sample #	N-10 Creston	AGS Au g/t	ALS Au g/t	Variance	AGS Ag g/t	ALS Ag g/t)	Variance	AGS Pb %	ALS Pb %	Variance	AGS Zn %	ALS Zn %	Variance	AGS Cu %	ALS Cu %	Variance
16	Muestra de Tumbe	14.54	17.60	-17%	229.7	286.0	-20%	4.5	4.4	2%	8.0	9.3	-14%	0.15	0.31	-52%
18	Muestra de Tumbe	13.80	16.15	-15%	131.5	184.0	-29%	12.6	12.0	5%	10.8	11.7	-8%	0.15	0.31	-52%
20	Muestra de Tumbe	10.80	9.85	10%	69.4	81.2	-15%	18.1	17.9	1%	17.4	16.3	7%	0.15	0.33	-55%
21	Muestra de Tumbe	4.73	7.11	-33%	74.2	93.6	-21%	12.3	11.6	6%	12.8	13.3	-3%	0.10	0.35	-71%
22	Muestra de Tumbe	5.47	6.95	-21%	49.9	71.8	-31%	4.4	4.6	-4%	10.7	12.8	-16%	0.20	0.22	-9%
23	Muestra de Tumbe	17.93	19.70	-9%	146.1	171.0	-15%	9.5	9.6	-1%	9.6	10.0	-4%	0.15	0.38	-61%
24	Muestra de Tumbe	10.60	14.15	-25%	75.4	115.0	-34%	7.4	6.6	12%	13.3	14.1	-5%	0.25	0.41	-39%
25	1 de 2	5.87	7.98	-26%	313.1	309.0	1%	9.4	7.8	21%	8.7	9.0	-3%	0.40	0.69	-42%
26	2 de 3	10.33	12.05	-14%	119.6	159.0	-25%	9.5	8.4	13%	9.9	11.1	-11%	0.35	0.45	-22%
27	3 de 3	10.67	11.70	-9%	58.7	77.6	-24%	4.9	4.1	20%	9.3	10.6	-12%	0.25	0.27	-7%
	Total	**10.47**	**12.32**	**-15%**	**126.76**	**154.82**	**-18%**	**9.26**	**8.70**	**6%**	**11.05**	**11.81**	**-6%**	**0.22**	**0.37**	**-42%**

In Table 24.16, "AGS" compares assays from the Altiplano Gold Siver lab to "ALS" assaying of standards and blanks samples. The averages of the gold and silver assays are within 84% and 79%, respectively. Given that some visible Au and Ag is occurs within the ore the variance in grade may be due to a nugget effect. The Pb and Zn results are both within 93% agreement between the two labs. The Cu results, however, showed much higher disagreement, averaging 59%.

The highlighted rows for samples # 018 and 023 are samples from the same shift round but separate quarters of the muck pile as described above. The comparisons of these two samples shows that the quarter sample that is being taken per shift can still vary in grades.

Comparison of the check samples suggest that the AGS lab may be under representing the grades of the ore. Pb is the only result that was higher in the AGS lab in comparison to ALS

All of the standards and blanks used in the testing of the ALS lab passed. These were CRMs used to test the quality of the lab's ability to reproduce known values for known materials.

Figure 24-15 Standards and Blanks used to test abilities of ALS

Sample #	N-10 CRESTON	AGS Au (g/t)	ALS Au (g/t)	AGS Ag (g/t)	ALS Ag (g/t)	AGS Pb (%)	ALS Pb (%)	AGS Zn (%)	ALS Zn (%)	AGS Cu (%)	ALS Cu (%)
001	MP-1b	-	<0.05	47.000	51.100	2.1	2.1	16.67	17.05	3.07	3.13
002	MP-1b	-	<0.05	47.000	51.100	2.1	2.1	16.67	17.2	3.07	3.15
003	MA-2c	3.020	3.14	0.510	0.900	25 ppm	41 ppm	93 ppm	189 ppm	95 ppm	114 ppm
004	MA-2c	3.020	3.02	0.510	<0.5	25 ppm	34 ppm	93 ppm	122 ppm	95 ppm	102 ppm
005	Sand	-	<0.05	-	<0.5	-		-		-	
006	Sand	-	<0.05	-	<0.5	-		-		-	

* The highlighted results are 'recommended values' for that particular standard and should only be used as reference and not for "Pass/Fail" results.

24.1.1.4 SUMMARY

The procedures observed at the mine site are being followed consistently. The method of breaking the samples, mixing the material together and then quartering them is a good practice for homogenizing the material to be sampled. The results show that the assays can vary a little bit, but can still be trusted. The head grade that is being reported based on AGS assays may be under representing of the actual grade of the material mined.

24.1.2 RECOMMENDATIONS

MMC believes that the practices and procedures being followed at Tahuehueto are adequate enough to make a good estimate of the material being mined. It is recommended that the miners take a few more samples throughout the shift. Based on the results from the two separate quarters of the same round that show a bit of variance in the grade, 3 samples should be taken per round which would create a better representation of the material for the head grade of that round. It is also recommended that Telson purchase CRMs that they can send to AGS with the muck samples to test the ability to reproduce assay results of a known material, or switch to an ISO-17025 assay lab. Bulk Sample Testing Results

24.1.3 BULK SAMPLE SHIPPED TO ANDES MILL

The first results from the 3,500 t bulk sample tests produced concentrates containing 247 oz of gold, 1,827 oz of silver, 27 t of lead, and 49 t of zinc. As of publishing date, Telson had secured a buyer and has received USD $465,539 for the sale of these first concentrates. The results of the bulk testing was overall positive. The processing of this large industrial scale bulk sample has allowed the test of various reagents, grindings sizes and recovery methods in order to work towards optimizing recoveries of gold, silver, lead and zinc. The first shipments of lead and zinc concentrates have produced cash sales, subsequent shipments of lead and zinc concentrates are expected to command higher sale prices per tonne based upon the assay results obtained in the Andes Mill laboratory as recovery techniques were improved towards the end of processing the complete 3,500 t industrial scale bulk sample (Telson, 2016a, b, c).

The recovery results to date (January 20, 2016) from this test are presented in Tables 24.2 and 24.3. Based upon the results of the industrial scale test, Telson's management indicates that they are intending to continue with bulk sampling and processing of ore from other zones of slightly different ore characteristics within the deposit and are considering to also process ore obtained during underground development of the mine. The Company intends to sell all concentrates produced from these efforts under similar procedures to those concentrate sales already completed.

Table 24.2 Industrial Scale Bulk Metallurgical Testing Results to Date (January 20, 2016)

Lot	Date	Dry Tonnes	Concentrate Grades									
			Au (g/t)	Ag (g/t)	Pb %	Zn %	Cu %	Cd %	Fe %	SiO_2 %	As %	Sb %
Zinc Concentrate												
1	11/22/16	29.50	12.1	148	3.7	48.16	ND	0.59	6.06	4.6	ND	ND
2	12/3/16	34.09	9.7	130	3.86	48.6	ND	0.61	7.8	2.9	ND	ND
3-1	12/21/16	33.56										
3-2	12/21/16	31.35										
3-3	12/22/16	33.63										
4-1	1/16/17	35.83										
4-2	1/17/17	37.83										
5-1	1/20/17	22.82										
Lead Concentrate												
1A	11/22/16	30.93	97.12	728	36.21	27.22	ND	ND	ND	2.7	0.02	0.02
1B	12/1/16	32.93	68.13	584	29.04	33.6	ND	ND	ND	2.4	0.01	0.02
2	12/3/16	35.41	112.53	720	39.22	27.34	ND	0.61	7.8	1.54	0.02	0.06
3-1	12/22/16	36.96	165.93	869.73	46.4	20.4	2.3					
3-2	12/22/16	39.33	154.13	875.93	46.8	20.4	2.3					
3-3	12/22/16	30.32	142.23	787.43	45.2	19.65	1.9					
4	1/20/17	2.42	165.93	715.07								

Table 24.3 Industrial Scale Bulk Metallurgical Testing Payable Results to Date (January 20, 2016)

Lot	Dry Tonnes	Concentrate Grades										Payables $US
		Au (g/t)	Ag (g/t)	Pb %	Zn %	Cu %	Cd %	Fe %	SiO_2 %	As %	Sb %	
Zinc Concentrate												
1	29.50	12.1	148	3.7	48.16	ND	0.59	6.06	4.6	ND	ND	33,726.91
2	34.09	9.7	130	3.86	48.6	ND	0.61	7.8	2.9	ND	ND	37,380.78
Lead Concentrate												
1A	30.93	97.12	728	36.21	27.22	ND	ND	ND	2.7	0.02	0.02	199,228.85
2	35.41	112.53	720	39.22	27.34	ND	0.61	7.8	1.54	0.02	0.06	195,202.54
Totals	**129.93**	**59.08**	**437.24**	**21.16**	**37.62**	**ND**	**0.60**	**7.28**	**2.87**	**0.02**	**0.04**	**465,539.08**

As of January 20, 2017 final sale terms are pending for six lots of Zinc Concentrate; Lots 3-1, 3-2, 3-3 4-1, 4-2, 5-1 and four lots Lead Concentrate; Lots 3-1, 3-2, 3-1, 4.

25 INTERPRETATION AND CONCLUSIONS

The Tahuehueto project is expected to yield an after-tax undiscounted LOM net cash flow of $207.9 million, and an NPV of $77 million at a discount rate of 8% per year.

The results of post-tax sensitivity analyses show that the project can withstand increases in capital and operating costs. A 20% decrease in revenue still results in a positive NPV at 8% of $24 million.

The conclusions and recommendations of this report are based on the use of Mineral Reserves that have been classified as Probable Reserves for the Tahuehueto project. These Mineral Reserves have been used in the economic evaluation of the Project. The Mineral Resources estimated for the Project are not Mineral Reserves and as such do not have demonstrated economic viability. There is no certainty that Mineral Resources can be converted to Mineral Reserves.

25.1 GEOLOGY, MINERALIZATION AND RESOURCE

The exploration work completed by Telson has demonstrated the existence of an epithermal deposit. Historic exploration has been focused along a series of exposed veins, silicified zones and color anomalies that are common within the Tahuehueto project area. The veins were examined and found to contain good gold and silver values hosted in sulfides. Mineralization at Tahuehueto is classified as intrusion related epithermal low sulfidation polymetallic, with Au and Ag accompanied by Cu, Pb, and Zn mineralization. These types of deposits are interpreted to have been derived from porphyry intrusion source rocks at depth.

Mineral Resources for the Project are based on the statistical analysis of data from 248 drill holes totaling 47,276 m and 1,788 underground samples within a model area covering of 2,672 square km. Sampling of the deposit indicates that its grade is potentially economic under an underground mining scenario.

The work completed by Telson has resulted in sufficient drill sample density, and confidence in the geological interpretation, for MMC to reasonably estimate Mineral Resources and Mineral Reserves for Tahuehueto.

25.2 MINING AND PROCESSING

MMC concludes that the Project resources provide a suitable basis for a project configuration that would include an owner-operated 790 tpd underground mine that will utilize overhand cut and fill mining with conventional mining equipment in a blast/load/haul operation. Mill feed will be processed in a 550 tpd comminution circuit consisting of primary and secondary crushing, grinding in a single ball mill followed by three floatation circuits producing lead, copper, and zinc concentrates. The concentrates will be trucked from site for smelting and refining.

A project configuration which included the underground mine and a central process facility was developed as the basis for capital and operating cost estimation. Generalized infrastructure arrangements were evaluated to establish a physical basis for the capital costs estimates. Capital and operating costs used for the Tahuehueto Project were developed from cost build up from first principles engineering along with vendor and contractor quotations. In addition, all available project technical data and metallurgical test work were considered to build up a processing operating cost estimate. An economic analysis of the potential project performance indicated strong economic potential at current metal prices.

26 RECOMMENDATIONS

This Tahuehueto Project review has indicated economic potential at current metal prices based on the economic analysis of the potential project. The project warrants advancement with the consideration of moving in the direction of an eventual feasibility study ("FS"). Infill drilling is warranted, with the goal of converting Probable Reserves to higher confidence categories. Underground development work is recommended to access ore in the lower El Creston zone and the Cinco de Mayo zone for additional bulk metallurgical test work and to provide access for underground infill drilling. This underground development work should incorporate the mine plan so as to reduce future development costs. Environmental data collection should continue to support environmental studies. In addition, geotechnical characterization investigations on surface outcrop exposures, in underground portals and with geotechnical drilling is recommended to support mine design and detailed cost estimates. A budget of $4.8 million is estimated for this work to further data collection in order support the advancement of the project as set out in Table 26.1.

These recommendations comprise several aspects of project development which are not necessarily successive phases that rely on positive results of previous phases. MMC believes that development of this information will support the execution of a FS. It is MMC’s recommendation that Telson continue the proposed project advancement program.

Table 26.1 Proposed Tahuehueto Project Advancement Program

Budget Item	Description	Cost (1,000’s)
Underground Development, Drilling and Geotechnical Studies	Haulage Level Portal Construction, Conduct Infill Drilling, Geotechnical Studies and Bulk Sample Collection and processing	$4,000
Geology	Resource Model Updates	$100
Geotechnical, Groundwater Hydrology and Tailings	Field and Engineering Work	$300
Rock Mechanics, Metallurgy and Economics	UG Mine Planning, Detailed Designs, Cost Estimations, and Reserves	$250
Other	Continuing Environmental, and Permitting	$150
Total		**$4,800**

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