Report to:

AMERICAN MANGANESE INC.

TECHNICAL REPORT AND PREFEASIBILITY STUDY

ARTILLERY PEAK PROJECT,

MOHAVE COUNTY, ARIZONA

DOCUMENT NO.: 1191630100-REP-R0001-06

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EFFECTIVE DATE: JUNE 28, 2012
AMENDED DATE: SEPTEMBER 12, 2012

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1191630100-REP-R0001-06

TABLE OF CONTENTS

LIST OF TABLES

LIST OF FIGURES

GLOSSARY

Units of Measure

Above mean sea level	amsl
Acre	ac
Ampere	A
Annum (year)	a
Billion	B
Billion tonnes	Bt
Billion years ago	Ga
British thermal unit	BTU
Centimetre	cm
Cubic centimetre	cm^3
Cubic feet per minute	cfm
Cubic feet per second	ft^3/s
Cubic foot	ft^3
Cubic inch	in^3
Cubic metre	m^3
Cubic yard	yd^3
Coefficients of Variation	CVs
Day	d
Days per week	d/wk
Days per year (annum)	d/a
Dead weight tonnes	DWT
Decibel adjusted	dBa
Decibel	dB
Degree	°
Degrees Celsius	°C
Diameter	ø
Dollar (American)	US$
Dollar (Canadian)	Cdn$
Dry metric ton	dmt
Foot	ft
Feet above mean sea level	famsl
Gallon	gal
Gigajoule	GJ
Gigapascal	GPa
Gigawatt	GW
Gram	g
Grams per litre	g/L
Grams per tonne	g/t
Greater than	>

Hectare (10,000 m^2)	ha
Hertz	Hz
Horsepower	hp
Hour	h
Hours per day	h/d
Hours per week	h/wk
Hours per year	h/a
Inch	"
Kilo (thousand)	k
Kilogram	kg
Kilograms per cubic metre	kg/m^3
Kilograms per hour	kg/h
Kilograms per square metre	kg/m^2
Kilometre	km
Kilometres per hour	km/h
Kilopascal	kPa
Kilotonne	kt
Kilovolt	kV
Kilovolt-ampere	kVA
Kilovolts	kV
Kilowatt	kW
Kilowatt hour	kWh
Kilowatt hours per tonne (metric ton)	kWh/t
Kilowatt hours per year	kWh/a
Less than	<
Litre	L
Litres per minute	L/m
Megabytes per second	Mb/s
Megapascal	MPa
Megavolt-ampere	MVA
Megawatt	MW
Metre	m
Metres above sea level	masl
Metres per minute	m/min
Metres per second	m/sec
Metres per hour	m/h
Metric ton (tonne)	t
Microns	µm
Milligram	mg
Milligrams per litre	mg/L
Millilitre	mL
Millimetre	mm
Million	M
Million bank cubic metres	Mbm^3
Million bank cubic metres per annum	Mbm^3/a
Million tonnes	Mt

Minute (plane angle)	'
Minute (time)	min
Month	mo
Ounce	oz
Pascal	Pa
Centipoise	mPa·s
Parts per million	ppm
Parts per billion	ppb
Percent	%
Pound(s)	lb
Pounds per square inch	psi
Revolutions per minute	rpm
Second (plane angle)	"
Second (time)	sec
Specific gravity	SG
Square centimetre	cm^2
Square foot	ft^2
Square inch	in^2
Square kilometre	km^2
Square metre	m^2
Standard Atmosphere	atm
Thousand tonnes	kt
Tonne (1,000 kg)	t
Tonnes per day	t/d
Tonnes per hour	t/h
Tonnes per year	t/a
Tonnes seconds per hour metre cubed	ts/hm^3
Volt	V
Week	wk
Weight/weight	w/w
Wet metric ton	wmt
Year (annum)	a

Abbreviations and Acronyms

Acid Base Accounting	ABA
Advance Minimum Royalty	AMR
Air Quality Index	AQI
ALS Chemex Laboratory	ALS
American Assay Laboratory	AAL
American Manganese, Inc.	AMI
Aquifer Protection Permit	APP
Arizona Department of Environmental Quality	ADEQ
Arizona Game and Fish Department	AGFD
Arizona Manganese Corporation	AMC

Arizona Pollutant Discharge Elimination System .... AZPDES
Artillery Peak Project.......... the "Project"
Artillery Peak Property .... the "Property"
Atomic Absorption.......... AA
Bankable Feasibility Study.......... BFS
Best Available Demonstrated Control Technology.......... BADCT
Best Management Practices .... BMPs
Bureau of Land Management .... BLM
Canadian Institute of Mining .... CIM
Chapin Exploration Company .... Chapin
Chemical Manganese Dioxide .... CMD
Clean Air Act.......... CAA
Closed-Circuit Television .... CCTV
Cobwebb Hill.......... CWH
Compound Annual Growth Rate .... CAGR
Corps of Engineers .... COE
Counter Current Decantation .... CCD
Distributed Control System .... DCS
Electrolytic Manganese Dioxide .... EMD
Electrolytic Manganese Metal .... EMM
Endangered Species Act .... ESA
Environmental Assessment .... EA
Environmental Impact Statement.......... EIS
Environmental Planning Group .... EPG
Federal Bureau of Land management .... FBLM
Finding of No Significant Impact .... FONSI
First World War.......... WWI
Heating, Ventilation and Air Conditioning.......... HVAC
Individual Permit .... IP
Inductively Coupled Plasma Mass Spectrometry.......... ICP-MS
Inverse Distance Squared.......... ID^2
Kemetco Research Inc.......... Kemetco
Life of Mine .... LOM
Loss on Ignition.......... LOI
M.A. Hanna Company.......... M.A. Hanna
Manganese Carbonate Dissolution.......... MCD
Manganese Carbonate Precipitation.......... MCP
Maximum Credible Earthquake.......... MCE
Maximum Probable Earthquake.......... MPE
Mechanical Vapour Recompression .... MVR
National Ambient Air Quality .... NAAQS
National Environmental Policy Act .... NEPA
National Hydrography Data.......... NHD
National Pollutant Discharge Elimination System .... NPDES
National Wetlands Inventory.......... NWI
Nationwide Permit.......... NWP

Nearest Neighbour	NN
Net Neutralization Potentials	NNP
New Source Performance Standards	NSPS
North American Datum	NAD
Notice of Intent	NOI
Operator Interface Stations	OIS
Ordinary Kriging	OK
Overflow	O/F
Piping and Instrumentation Diagram	P&ID
Pit/Tailings North And South Areas	PTNS
Pocock Industrial	Pocock
Prefeasibility Study	PFS
Pregnant Leach Solution	PLS
Primus Resources, LC	Primus
Process Research Associates Ltd.	PRA
Process Water Pond	PWP
Quality Assurance/Quality Control	QA/QC
Rocher deBoule Minerals Corp.	RDM
Rocher Manganese Inc.	RMI
Rotary Percussion Reverse Circulation	RPRC
Run-of-Mine	ROM
Second World War	WWII
Slope of Regression	SOR
Solid-Liquid Separation	SLS
Storm Water Pollution Prevention Plan	SWPPP
Synthetic Precipitation Leaching Procedure	SPLP
Synthetic Precipitation Leaching Procedure	SPLP
Three Dimensional Model	3DM
Three Dimensional	3D
Total Dissolved Solids	TDS
Underflow	U/F
UniSource Energy Services	UNSE
United States Bureau of Mines	USBM
United States Geological Survey	USGS
Universal Transverse Mercator	UTM
US Army Corps of Engineers	USACE
US Fish and Wildlife Services	USFWS
Value Added Tax	VAT
Water Management Plan	WMP
Western Area Power Authority	WAPA
Workplace Hazardous Materials Information Systems	WHMIS

List of Chemical Symbols

Aluminum Oxide	Al_2O_3
Aluminum	Al
Arsenic	As
Ammonium Sulphate	NH_4SO_4
Barium Oxide	BaO
Calcium Oxide	CaO
Cobalt	Co
Copper	Cu
Iron Oxide	Fe_2O_3
Magnesium Oxide	MgO
Manganese Carbonate	$MnCO_3$
Manganese Oxide	MnO
Manganese Oxide	Mn_3O_4
Manganese Sulphate	$MnSO_4$
Manganese	Mn
Nickel	Ni
Phosphorus Oxide	P_2O_5
Phosphorus	P
Pregnant Leach Solution	PLS
Potassium Oxide	K_2O
Silicon Dioxide	SiO_2
Silicon	Si
Sodium Oxide	Na_2O
Sulphide	S
Sulphuric Acid	H_2SO_4
Sulphur Dioxide	SO_2
Thallium	Tl
Zinc	Zn

1.0 SUMMARY

1.1 INTRODUCTION

American Manganese Inc. (AMI) commissioned Tetra Tech Wardrop (Tetra Tech) to prepare this Prefeasibility Study technical report on the Artillery Peak Project (the Project).

The following consultants were commissioned to complete the component studies for the National Instrument 43-101 (NI 43-101) technical report:

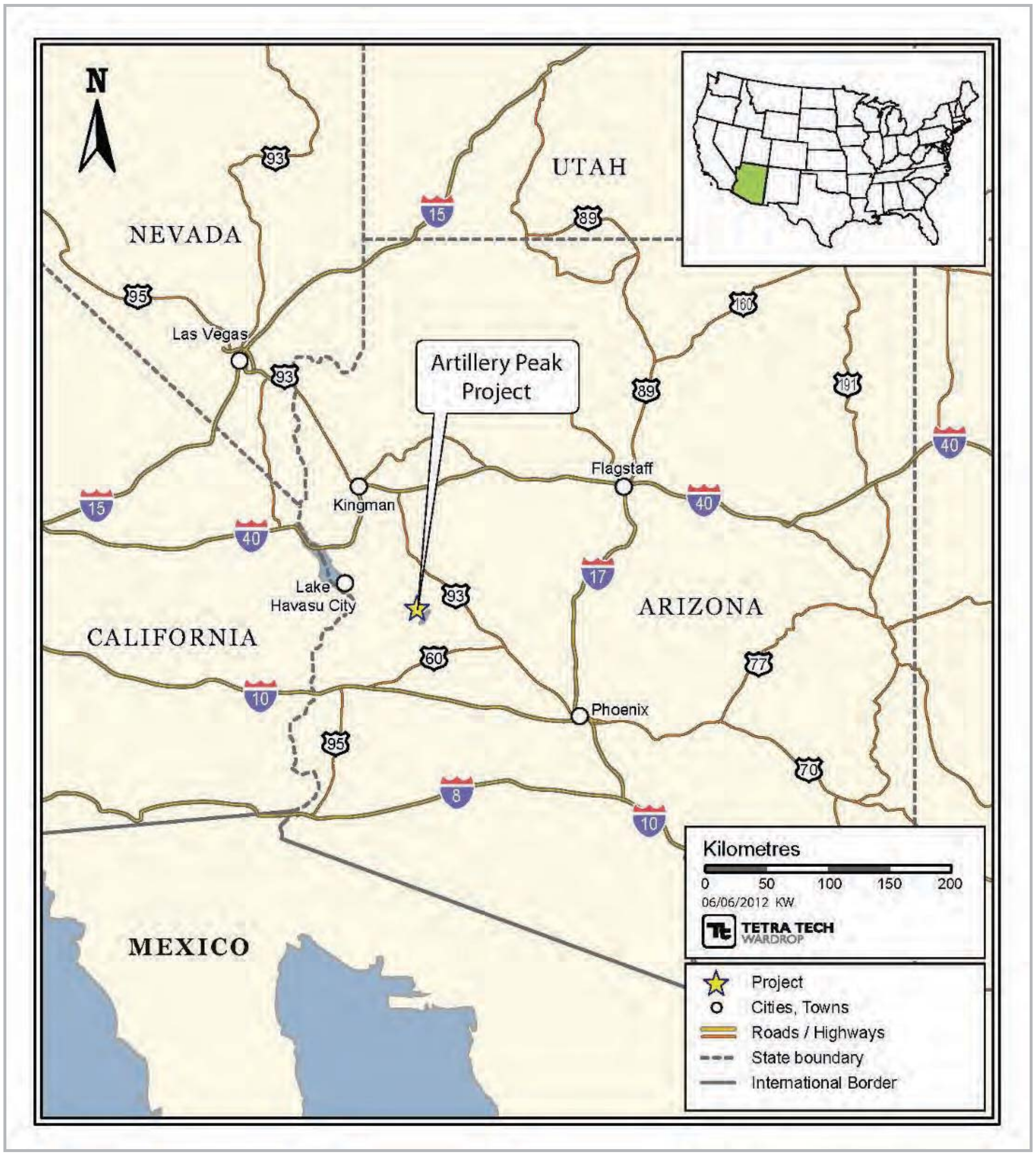
- Tetra Tech: resource estimate, mining, processing, infrastructure, environmental aspects, water and waste management, geotechnical design capital and operating cost estimate, and financial analysis
- Kemetco Research Inc. (Kemetco): metallurgical testing and process piloting
- CPM Group (CPM): electrolytic manganese market outlook, 2012.

1.2 PROPERTY LOCATION AND DESCRIPTION

The Property is located in the Artillery Mountains of Mohave County in northwestern Arizona (Figure 1.1). The Property is approximately 170 km (108 miles) northwest of Phoenix, Arizona, and 240 km (150 miles) southeast of Las Vegas, Nevada. The Property covers approximately 10,000 acres and comprises 471 mining claims, 85 of which are covered by lease agreements. Manganese was discovered on the Property in 1928 and was historically mined during the 1930s, 1940s, and 1950s.

The Property is located on the southeastern edge of the Mojave Desert, a desert climate with low precipitation and extreme hot temperatures in summer months, and mild temperatures in winter months. The Property is accessible by unpaved roads leading from Highways 40 or 93.

Figure 1.1 Property Location



1.3 GEOLOGY, MINERALIZATION, AND DEPOSIT TYPES

The Property is located within the Basin and Range province, which was formed by lithosphere extension that resulted in repeated normal faulting and downward displacement of blocks of the crust, as well as local volcanism. This formed a horst and graben landscape, or mountain ranges separated by basins. The faulting in the region juxtaposes Late Proterozoic to Early Paleozoic metamorphic rocks against much younger Cenozoic sedimentary and volcanic rocks.

On the Property, the older basement rocks are unconformably overlain by sedimentary and volcanic rocks of the Artillery formation. The age of the Artillery Formation is at least 20 Ma and it comprises a number of sandstone and conglomerate units, a basalt flow, and a megabreccia.

The Artillery Formation is unconformably overlain by the Chapin Wash Formation. This formation comprises a series of sedimentary rocks that have been interpreted to represent alluvial fans and playa deposits, and is the unit that hosts the stratiform manganese deposits. The Chapin Wash Formation includes sandstone, siltstone, shale and conglomerate, as well as minor beds of tuff and lacustrine limestone.

In most areas of the Property, the Chapin Wash Formation is overlain by the Cobwebb Basalt, which is dated at 13.3 Ma and is conformable with the underlying Chapin Wash Formation. Mineralization typically appears within a few metres to a few tens of metres below the Cobwebb Basalt. The Sandtrap conglomerate unconformably overlies the Cobwebb Basalt and Chapin Wash Formation, and is interbedded with basalt flows dated at 9.5 Ma. The mesas in the region are topped with a younger basalt flow up to 10 m thick.

The Property lies above a gently northeast-dipping, large displacement, normal fault known as the Buckskin-Rawhide detachment fault. Normal faulting was ongoing during deposition of the sedimentary and volcanic units described above, resulting in progressive tilting of older strata. Numerous faults are present on the Property and largely control the present-day topography; possible volcanic vents are present along these faults.

1.3.1 *Mineralization and Deposit Types*

Two distinct styles of manganese mineralization are observed on the Property: stratiform manganese deposits, and vein manganese deposits. The mineralization style responsible for the majority of manganese on the Property is stratiform manganese, which occurs within the Chapin Wash Formation. These deposits have typical grades of around 3% to 4% Mn and locally 6% to 10% Mn. Some workers interpret these deposits to be syngenetic with the host sediments, whereas others interpret remobilization of manganese and potassium by secondary metosomatic fluids. Some secondary redistribution of manganese occurred because of interaction with meteoric water.

Manganese also occurs within veins, breccias (some of which have been interpreted as possible volcanic vents), and fracture and fault zone cement, and overall this style of mineralization is referred as vein manganese. Mineralized veins are locally common within or near the stratiform deposits although they do occur in other units in addition to the Chapin Wash Formation, particularly within the Sandtrap Conglomerate. Vein manganese deposits are several million years younger than stratiform deposits, and although they are not obviously related, their spatial association does suggest some genetic link. Vein-type deposits are smaller, irregular and locally higher grade than the stratiform deposits and their size currently does not support large tonnage bulk mining practices.

1.4 EXPLORATION AND DRILLING

In 2008, nine channels were collected from the MacGregor open pit face at 50 m spacing, and were treated as vertical drillholes for resource estimation. Also in 2008, 17 NQ diamond drillholes (3,011 m) were completed, 8 (821 m) of which were drilled within the area included in the current resource estimate.

Exploration in 2010 consisted of drilling using a rotary percussion reverse circulation (RPRC) method. Fifty-three RPRC holes (4,649 m) were completed in 2010, 33 (2,615 m) of which were drilled within the area included in the current resource estimate.

Exploration in 2011 included further RPRC drilling. Eighty-one RPRC holes (10,003 m) were completed in 2011, 56 of (6,238 m) of which were drilled within the area included in the current resource estimate. Also in 2011, two bulk samples totalling 8 t of material were collected from the Cobwebb Hill escarpment and processed in a metallurgical study.

Detailed geological mapping was conducted in 2011 and 2012 at a 1:6000 scale.

1.5 RESOURCE ESTIMATES

1.5.1 2011 ESTIMATE

This estimate was originally presented in the report entitled "Mineral Resource Evaluation Report on the Artillery Mountain Manganese Property, Mohave County, Arizona, U.S.A.," by Norm Tribe, P.Eng., dated September 1, 2011 and amended December 1, 2011. It tabulated the estimated resources for all sites of known mineralization within AMI's land holdings.

The 2011 estimate remains in effect for the surrounding and outlying mineralized areas contained within AMI's total land holdings including, but limited to, Maggie Mine, Shannon Mine, Love's, Hurley and Planche Mines, and South Chapin, Burro, Price and Priceless Zones. The remaining resources estimated in these areas, at a cut-off grade of 0.90% Mn, include an Indicated Resource of 143,575,196 t at an average grade of 2.98% and an Inferred Resource of 54,700,239 t at an average manganese grade of 2.83%.

1.5.2 2012 Estimate

This resource estimate was based on a 3D geology wireframe constructed using the 2008, 2010, and 2011 drillhole data with a nominated grade threshold of 0.9% Mn. The wireframe was constrained by several faults and topography. Grades were interpolated into this solid using ordinary kriging (OK) on 1.5 m composites; no assay values were capped.

At a base-case cut-off of 1% Mn, this estimate includes an Indicated Resource of 62,201,000 t at an average manganese grade of 2.3%, and an Inferred Resource of 20,033,000 t at an average manganese grade of 2.5%. This resource estimate supersedes the estimate for the North Chapin/Lakes/MacGregor region from previous resource estimates, which is the area used to develop the commercial mining scenario described in this report.

1.6 Mining Operations

The deposit contains 45 Mt of mineral resource to be used for the mine plan with an average grade of 2.46% Mn, including an average allowance of 5% dilution.

The mineral resource was estimated through the selection of an optimum final pit, based on the block model data, which represents the portion of the Indicated Mineral Resource that falls within the selected final pit. Although the block model included some Inferred Mineral Resource, this portion of the model was not included in the mineral resource and remains classified as waste.

Tetra Tech developed an open pit, truck/shovel mine plan for the deposit, involving mining up to 7,000 t/d of mineral resource and an average of 17,220 t/d of waste over a 21-year LOM. The stripping ratio will average 2.46. For the duration of the LOM, mineral resource containing 1,106,000 t of manganese will be delivered to the mill, for a recovered total of 994,499 t of electrolytic manganese metal (EMM).

Tetra Tech developed a mining schedule that includes one year of preproduction, 21 years of mining operations, and three years of reclamation activities. Section 16 of this report shows the production schedule, indicating the total mined waste, total mined mineral resource, and total EMM produced in each year.

In the current schedule, mineral resource with marginal grades (low grade) will be dumped in waste dumps rather than stockpiled.

Tetra Tech selected a mine plan that involves mining the higher-grade mineral resource in the first years of operation, rather than a directional, strip-mining method. Tetra Tech tested a directional, strip-mining method using Whittle™ software. The results indicated that grade and topography variations, made the strip-mining method less desirable than the common open pit method.

Most of the benches have been designed to accommodate a loader/shovel and a 136 t truck. In the last bench at the bottom of the pit, there are areas where mining will be reduced to areas of 30 m x 30 m or less. Tetra Tech recommends the use of a hydraulic excavator to accommodate space limitations and reduce dilution.

The mine plan will include a haul backfill system for co-disposal of all waste products within previously mined-out areas of the open pit.

The Project will produce approximately 111 Mt of waste rock and 47 Mt of tailings, which will be co-disposed in the tailings waste storage facility (TWSF) and ultimately returned to mined-out pit, in order, among other reasons, to reduce the mine footprint.

In keeping with the mining sequence, one external waste dump southeast of the pit has been designed to accommodate the first six years of waste material, and another to the north to accommodate waste in later years. After Year 6, the co-disposed waste rock/tailings material will be immediately backfilled to the north and south sections of the mined-out pit, while a clear mining zone in the centre of the pit is maintained. At the end of the LOM, all co-disposed waste will be reclaimed to the south and north in-pit waste dumps.

1.7 METALLURGICAL TESTING AND PROCESS PILOTING

Treatment strategies for recovering manganese from lower grade resources of the Artillery Peak region were pioneered by the United States Bureau of Mines (USBM) from the 1930s to the late 1980s. Of key importance of the prior work was the discovery that higher valent manganese resources can be readily leached with sulphur dioxide dissolved in water. This eliminates the requirement for high temperature reduction roasting that is conventionally used in processing high-grade material. High temperature roasting is energy intensive, emits a large amount of carbon dioxide, and would not be economical for processing lower grade material. While the prior work provides important information to extract manganese from lower grade resources, the production of unwanted dithionate by-products, and high water and energy use has limited its commercial use.

With the goal to develop robust cost effective process to recover manganese from lower grade resources, AMI contracted Kemetco Research Inc. in 2009, to undertake an extensive bench scale metallurgical test program to address foregoing issues. The test program was partially funded by the Canadian Government through the National Research Council, Industrial Research Assistance Program (NRC-IRAP). The development work was successfully completed by Kemetco. The results of the bench scale research have led to the development of a flowsheet to process lower grade manganese resources. A PCT International Patent application (application number PCT/US2011/047916) was filed on the conceptual process in 2011.

During 2011 and 2012, Kemetco performed a much more intensive hydrometallurgical testing program. A thorough pilot plant study confirmed a processing scheme that can be modeled and scaled-up to a full size processing plant, utilizing commercially available technologies and equipment. This plant model was used as the basis for engineering design.

1.8 RECOVERY METHODS – PROCESSING

The proposed processing plant is designed to process mill feed containing 2.2% to 3.5% Mn in the range of 3,500 t/d to 7,000 t/d, to produce 50,000 t/a of EMM at a purity of higher than 99.7% Mn and an overall recovery of 88 to 93% Mn. Anhydrous sodium sulphate will be produced as a byproduct.

The ROM mill feed will be trucked from the open pit to the crushing facility located at the plant site. Two stages of crushing will reduce the ROM feed to a particle size of 100% passing 5 mm. The first stage of the crushing will use a sizer crusher in an open circuit. The second stage of crushing will use an impact crusher in a closed circuit with a dry screen. The product from the crushing circuit will be stored in two 7,000-t surge bins prior to being conveyed to the leaching circuit.

The crushed mill feed will be leached in two stages: pre-conditioning leaching with acidic solution containing sulphuric acid (mainly recycled from downstream washing circuits), and sulphur dioxide reductive leaching. The insoluble manganese in the 4^+ oxidation state can be readily reduced to the soluble 2^+ oxidation state by adding sulphur dioxide. Both leaching agents—sulphur dioxide and sulphuric acid—will be generated from burned liquid sulphur on-site.

The leached slurry after aeration will be directed to the counter-current decantation (CCD) circuit, where the manganese-bearing pregnant leach solution (PLS) will be separated from the leach residue. The leach residue will be subjected to multiple stages of CCD washing, followed by further dewatering in pressure filters to reduce water consumption and water content of the residue for waste rock/residue co-deposit. The washed leach residue will be back-hauled by truck to the excavated area within the open pit and blended for co-disposal with waste rock.

The PLS will be treated by two stages of purification to remove any impurities (such as iron, aluminum, zinc, nickel, etc.) which may have detrimental effects on the downstream operations.

The purified PLS containing mainly manganese sulphate and manganese dithionate is directed to the manganese carbonate precipitation circuit. The precipitation of manganese carbonate is achieved by mixing the purified PLS with sodium carbonate. In the precipitation process, soluble sodium sulphate and sodium dithionate are effectively eliminated from the manganese carbonate product. The resulting manganese carbonate precipitate will then be separated from the solution by thickening and filtration. The sodium sulphate and sodium dithionate solution will be sent to the nanofiltration/ mechanical vapour recompression (MVR) evaporation system to recover sodium sulphate and water.

The manganese carbonate solids will be dissolved by the acidic spent electrolyte solution produced from the manganese electrowinning circuit. This solution is further purified in two stages to remove any impurities and to generate the manganese electrolyte feed for subsequent use in the EMM electrowinning process.

The electrolyte feed (neutral manganese sulphate) will be pumped into the electrowinning cells where the anode and cathode compartments are separated by semi-permeable diaphragms. At the cathode, Mn^{2+} will be reduced to $Mn^{o}(s)$ and deposited onto the cathode plate. Depending on process conditions, cathodic hydrogen gas and anodic oxygen or manganese dioxide will be generated, respectively, due to competing side reactions.

Manganese metal obtained at the steel cathode plates will have an overall purity of over 99.7% Mn. The deposited manganese will be treated by washing and passivation prior to being peeled from the cathode plate. The manganese metal flakes will be transported to the manganese stock silo, and then bagged for shipment.

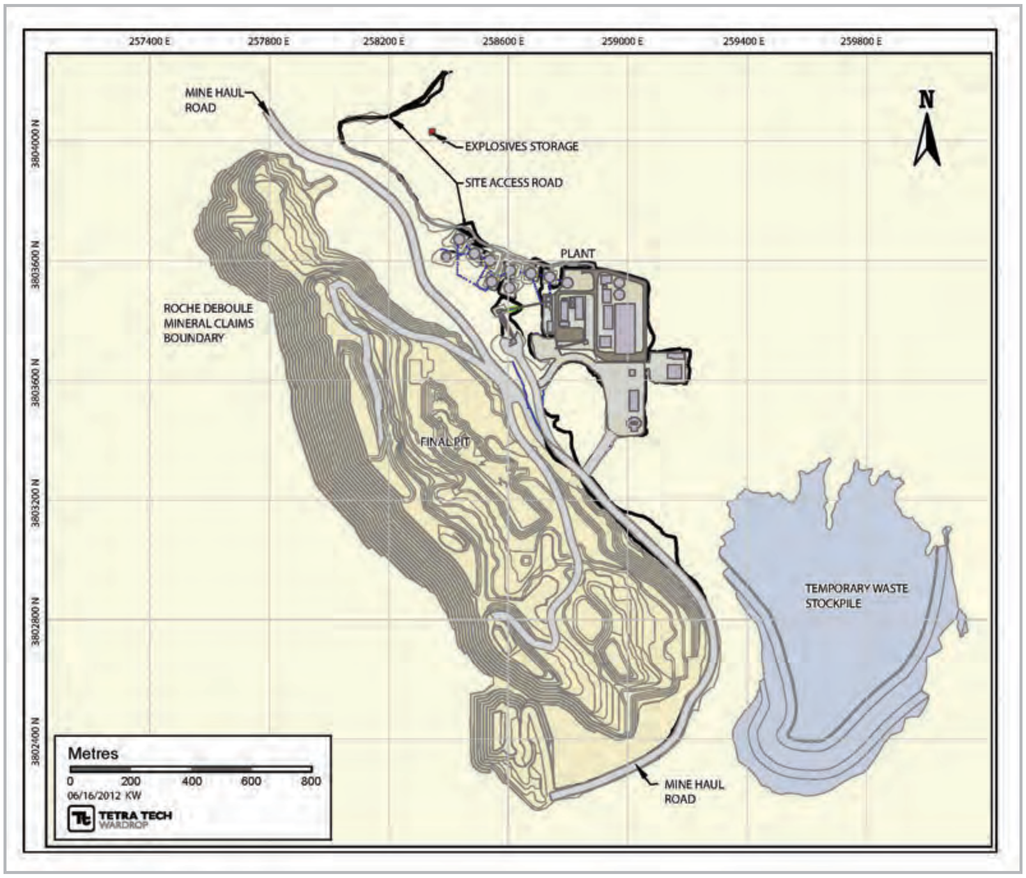
The sodium sulphate and sodium dithionate solution from the manganese carbonate precipitation circuit will be further processed to recover the sodium sulphate produced as a byproduct and water, which will be reused as process water.

The heat generated from the liquid sulphur burner will be recovered and used to generate electricity by a steam turbine, and/or provide direct heating in the form of intermediate steam, as needed in the process.

1.9 INFRASTRUCTURE

The Project will be a comprehensive greenfield development situated in low-lying mountainous terrain. The process site is located directly east of the open pit. The overall mine site arrangement is shown in Figure 1.2.

Figure 1.2 Mine Site



The on-site facilities are situated in a flat plain area, and include all key components of the Project, including:

- a process plant
- a fleet maintenance facility
- administration offices
- a construction camp
- access and on-site roads
- a power substation
- a TWSF.

1.9.1 Process Plant

The process plant will include:

- a crushing facility, including a screening transfer tower

- leach feed surge bins (a total live capacity of 14,000 t)
- leaching and related facilities, including leaching, CCD thickeners and tailings filtration
- PLS purification, including manganese precipitation
- an electrowinning process plant
- a sodium sulphate byproduct production facility
- a sulphur dioxide generation, sulphuric acid generation, heat recovery and electrical power generation facility.

A detailed facilities and site services description is provided in Section 18 of this report.

1.9.2 SITE ACCESS ROAD

The process plant site, project facilities and open pit will be accessible by a new permanent road, connected to the existing, well-maintained Alamo Lake Road. From the intersection with Alamo Lake Road, north from the Project, the proposed access will pass alongside mountainous washes, and wind through promontories and outcropping rocks, head southwest of the adjacent drainage basin, until its connects to the north boundary of the plant site.

1.9.3 POWER SUPPLY AND DISTRIBUTION

The incoming purchased power from the utility will terminate just outside of the plant substation. It will be a conventional substation with two 30 MVA step-down transformers for power redundancy. Distribution from the transformers will be from a redundant main-tie-main 15 kV switchgear (13.8 kV) feeding the main plant overhead power distribution system that will deliver energy throughout the facility and mine. The onsite 4 MVA power generation facility will be tied into the substation switchgear.

1.9.4 WATER SUPPLY

The Project site will require approximately 3.22 m^3/min (850 gal/min) of continuous water supply.

There may be usable underground water in the aquifers, located southeast of the mine area. These wells will produce a limited amount of water, and may form part or all of the solution to the required water supply.

Nine source wells will be drilled in the areas of expected higher permeability. Aquifer testing performed on each of the nine source wells will evaluate the suitability for the siting of a larger scale water-supply well field in selected areas.

A preliminary water distribution line was designed to convey the water from the furthest identified well source to the proposed mine facilities. The proposed water line was assumed to be constructed on grade, following existing roadways to the planned mine site.

1.9.5 WATER MANAGEMENT

Water management will be a critical component of the project area. The Project will require the use of process and non-process water. Process water is water that will be used in the plant site, or conveyed to various on-site areas via pipelines. Non-process water is surface drainage and runoff water that will require management at the site. Non-process water will comprise contact and non-contact water.

Contact water is water that will be exposed to facilities such as the open pit, TWSF and the plant site. This water will be collected in designated ponds for treatment, as necessary, prior to its release, and will be used in the process to the maximum extent possible. Contact water from direct run-off from the southeast TWSF (and for un-diverted runoff from contributing basins) and from the south backfilled pit area is diverted to a collection pond, for treatment (if required) and subsequently pumped to the contact water process pond. Contact water from direct run-off from the open pit and un-diverted run-off from contributing basins and from the north backfilled pit area is accumulated in the bottom of the operating pit and subsequently pumped directly to the contact water process pond located near the plant site, for use in the production process, should this be required.

Non-contact water will be directed around the facilities via diversion channels or culverts and discharged into the natural drainages upon achievement of sediment removal to acceptable levels.

The overall goal of the water management plan is to:

- minimize the amount of contact water (water resulting from interaction with mine waste and other elements from mining activities) produced during project construction, operation and following closure
- collect contact water to enable reuse, or discharge after treatment (unless compliance with water quality standards can be demonstrated)
- provide and retain water for mine operations
- provide a basis for management of the freshwater on the site.

1.9.6 WASTE MANAGEMENT

The Project will generate waste rock and waste filtered tailings. The tailings and waste rock will be placed back into the pit, eliminating any permanent waste storage facility. Waste generated from the first six years of production will be temporarily stored in a designated area located southeast of the pit. After year six, the waste will be placed directly into the mined sections of the pit. At end of operations, the material stored in the temporary facility will be relocated in the pit.

1.10 ENVIRONMENTAL CONSIDERATIONS

The environmental permitting and approvals necessary to bring the Project into production are complex. Various permitting programs require advance/baseline studies. The list of permits and approvals include federal and State of Arizona programs. Compliance with each of these programs is feasible; some compliance activities will require more effort and planning to streamline and avoid time delays and plan revisions. A full explanation of environmental permitting program requirements is provided in Section 20.

One of the most detailed and involved process is that which is required under the National Environmental Policy Act (NEPA). NEPA forms the basis of the federal government's decision-making process by requiring full and complete disclosure of the impacts of the proposed action on the human environment.

The NEPA process includes an initial review of the project. The NEPA agency, which in this case is the US Bureau of Land Management (BLM), will require an Environmental Impact Statement (EIS) with full public disclosure of the Project's environmental impacts. NEPA is the disclosure authority, not the decision-making authority. Other statutes provide the basis for deciding the approval of the Project, based on the NEPA analysis.

The EIS process requires intensive public involvement and a series of administrative and technical stages. Baseline study work is being conducted and planned in the following area, in order to investigate the potential environmental impacts for NEPA:

- biological impacts to wildlife, plants including any endangered or protected species including surveys which are currently being planned
- cultural impacts to archaeological and cultural resources including surveys which were initiated in October 2011 and are ongoing for the mine project
- impacts to the air from proposed mining operations through studies and/or modelling and utilizing baseline studies of existing conditions
- impacts to groundwater quality and quantity via hydrogeologic studies by conducting well installations which will occur in the Fall 2012
- social and economic impacts
- noise and visual impacts
- impacts of tailings and waste rock drainage via geochemical studies of the materials for acid rock drainage and leaching potentials.

These impacts require further study but preliminary studies in some areas, such as biological and cultural, indicate that any impacts can be mitigated.

The most detailed and involved Arizona Department of Environmental Quality (ADEQ) program is the Aquifer Protection Permit (APP) program. The APP program applies to facilities that may potentially discharge pollutants that may adversely impact ground water quality. Facilities and process units subject to the APP program will largely be designed and operated to contain contaminants and monitor for accidental releases.

As part of the preparation for an APP permit, waste rock and tailings characterization is required to establish any potential environmental impacts to groundwater. Preliminary samples were collected in August 2011 from eleven areas representing waste rock; nine tailings materials samples were collected during three separate runs of the pilot mineral resource process from October 2011 through December 2011.

As a result of this study, it was determined that the metallurgical processes will be re-evaluated during the feasibility stage to determine whether manganese recovery, and secondary isolation or removal of metal byproducts (e.g. arsenic and thallium) will be required. Based on initial geotechnical analyses of the tailings, additional changes and refinements to the mineral resource process are necessary and will require additional characterization and testing to establish results for permitting of the tailings.

Eleven waste rock samples were evaluated for ABA and SPLP. These initial samples indicated that the waste rock is not acid generating and limited in elevate metal content. Additional data is required for APP and further sampling of waste rock material is being conducted in June 2012.

The feasibility study effort will focus on improvements to metallurgical processes, as well as any innovations considered relevant to the metal recovery/control technologies applied at the Project. The results of these efforts and continued geochemical testing will help refine the waste disposal methods ultimately selected for the Project.

The State of Arizona's implementation and local accentuation of air regulations from the Environmental Protection Agency (EPA) includes the Clean Air Act (CAA). The EPA delegated the authority for air quality permitting to ADEQ. It is anticipated that the air permitting process in Arizona will take approximately one year to complete. The Project includes several processes with anticipated pollutant emissions. Air permitting requires details of the mine process in order to identify the potential emissions and required performance standards. Due to the preliminary stage of the process design determination and modelling of the impacts to air quality from the Project will be completed during the feasibility stage. Modelling of the emissions is likely in order to proceed with the air permitting.

The mine project has obtained a multi-sector general permit for stormwater discharges and a De Minimis Permit for discharges from groundwater wells to the ground surface.

The mine project is progressing with a comprehensive approach to the permits and regulatory requirements to ensure that the regulations that require the greatest effort in terms of preliminary and baseline study work are planned and prepared well in advance in order to maintain the Project timeline.

Several environmental permits or regulatory requirements are identified in Section 20.

1.11 CAPITAL COST ESTIMATE

Tetra Tech developed a capital cost estimate for the Project, with an expected accuracy range of ±25%. All costs are expressed in United States dollars.

The estimate consists of four main components:

- direct costs
- indirect costs
- contingency
- Owner's costs.

The initial capital cost for the Project is estimated to be $476,972,678. The capital cost summary is shown in Table 1.1.

Tetra Tech prepared this capital cost estimate with a base date of Q2 2012.

Table 1.1 Capital Costs Estimate

Description	Labour Cost ($)	Material Cost ($)	Construction Equipment Cost ($)	Equipment Cost ($)	Total Cost ($)
Direct Costs					
Overall Site	5,544,314	7,857,713	7,979,416	-	21,381,443
Mining	1,104,890	463,350	783,380	14,427,910	16,779,530
Milling	42,830,133	51,432,930	8,214,060	137,484,598	239,961,721
Mine Site Utilities	11,449,809	18,630,371	2,606,053	11,181,061	43,867,294
Mine Site Buildings	934,791	2,847,876	280,095	1,583,013	5,645,775
Tailings	767,793	995,202	217,281	-	1,980,276
Plant Mobile Equipment	49,350	-	-	1,276,590	1,325,940
Temporary Facilities	62,040	2,006,125	44,000	-	2,112,165
Subtotal Direct Costs					**333,054,144**

Description	Labour Cost ($)	Material Cost ($)	Construction Equipment Cost ($)	Equipment Cost ($)	Total Cost ($)
Indirect Costs	2,053,440	79,238,297	-	-	81,291,737
Owner's Costs	2,472,000	7,261,103	-	-	9,733,103
Contingency	-	-	-	-	52,893,694
Total					**476,972,678**

1.12 OPERATING COST ESTIMATE

The Project's average annual operating cost is estimated to be $1.012/lb of EMM produced. Process cost includes credits from in situ generation of electricity power and sodium sulphate anhydrous byproduct. All operating costs are expressed in United States dollars.

The average LOM operating cost is based on an average annual production rate of 50,000 t of EMM.

The average LOM unit cost distribution after credits is presented Table 1.2.

Table 1.2 Average LOM Operating Cost

Description	Operating Cost ($/lb EMM)
Mining	0.200
Processing	0.751
G&A	0.050
Surface Services	0.011
Total Operating Cost	**1.012**

The annual process operating cost for a maximum feed rate of 7,000 t/d, and an EMM production rate of 50,000 t/a, is estimated to be $95.9 million, or $37.75/t of milled mineral resource $0.870/lb EMM, excluding the credits from heat recovery (electricity generation) and sodium sulphate anhydrous by-product. After these credits, the unit process cost per pound of EMM produced is estimated to be $0.767. These unit production costs will vary with mill feed rates or mill feed grades.

1.13 ECONOMIC EVALUATION

Tetra Tech prepared an economic evaluation of the Project based on the assumption that 60% of EMM production will be sold domestically in the US and 40% overseas. The results of the pre-tax and the post-tax evaluations are as follows:

Pre-tax Evaluation

- The Base Case uses the weighted three-year historical average world and US EMM price of $1.54/lb.
- The Alternate Case 1 uses the EMM CPM expected price forecast for overseas sales with price increase by 14% for domestic sales, generating an average price of $1.93/lb.
- The Alternate Case 4 uses the three-year historical average EMM price of $1.54/lb reduced by 25% or $1.16/lb.

For the 21-year LOM at an average annual production of 50,000 t of EMM, the following pre-tax financial results were calculated:

- Base Case:
  - 7.28% internal rate of return (IRR)
  - 10.3 year payback on $477 million initial capital
  - ($23) million net present value (NPV) at an 8% discount rate.
- Alternate Case 1:
  - 19.95% internal rate of return (IRR)
  - 4.6 year payback on $477 million initial capital
  - $403 million net present value (NPV) at an 8% discount rate.
- Alternate Case 4:
  - no internal rate of return (IRR)
  - no payback on $477 million initial capital
  - ($371) million net present value (NPV) at an 8% discount rate.

Post-tax Evaluation

A post-tax financial analysis of the Base Case was prepared to evaluate the effect of taxes on future financial performance of the potential operations.

For the 21-year LOM at an average annual production of 50,000 t of EMM, the following post-tax financial results were calculated for the Base Case:

- 6.14% internal rate of return (IRR)
- 10.7 year payback on $477 million initial capital
- ($54.407) million net present value (NPV) at an 8% discount rate.

Tetra Tech conducted sensitivity analyses to establish the sensitivity of the Project merit measures (NPV, IRR and payback periods) to the main inputs.

The financial results described for Alternate Case scenarios presented in Section 22, did not factor in taxes, and the impact of taxes would reduce the financial results.

Details of the CPM EMM market outlook and price forecast are provided in Section 19.

Details of the EMM price calculations and the Project economic evaluation are provided in Section 22.

1.14 CONCLUSIONS AND RECOMMENDATIONS

Based on the results of the work presented in this Report, Tetra Tech recommends that AMI proceed with the next phase of work to identify potential cost savings and additional revenue generating opportunities and more completely assess the viability of the Project.

The overall processing concept has been proven by metallurgical bench and pilot plant tests. The diagnostic projections of the PFS can be used to focus further efforts in optimizing, designing and demonstrating an initial refined embodiment of the actual operations. The main risks and challenges to the Project are shared by many established producers worldwide and pertain to rising energy, environmental protection, labour and reagent costs.

1.14.1 FINANCIAL EVALUATIONS BASE CASE AND ALTERNATE 4 CASE

Financial evaluations of the Base Case and Alternate Case 4 EMM pricing regimes show that the Project has a negative NPV at an 8% discount rate. For these cases, the Project is deemed to be uneconomic and contains no mineral reserves that meet the Canadian Institute of Mining (CIM) definition.

2.0 INTRODUCTION

Tetra Tech has prepared this technical report on the Artillery Peak Prefeasibility Study in general accordance with the guidelines provided in National Instrument 43-101 (NI 43-101) Standards of Disclosure for Mineral Projects.

Tetra Tech, all qualified persons (QPs), and authors of this technical report, are independent of the issuer, American Manganese Inc. (AMI), its directors, senior management and advisers. Tetra Tech prepared this report at the request of American Manganese Inc.

This report is based on information provided by AMI to Tetra Tech, as well as observations and information that Tetra Tech obtained during the site visit to the Property.

All costs are expressed in United States dollars, unless stated otherwise.

The following technical experts and QPs visited the Property:

- Anoush Ebrahimi, P.Eng., on January 28, 2012
- Norm Chow, P.Eng., on February 25, 2011
- Marvin Silva, PhD, PE, P.Eng., on February 25, 2011
- Margaret Harder, P.Geo., on June 2, 2011.

A summary of the QPs responsible for each section of this report is detailed in Table 2.1.

Table 2.1 Summary o uali ied Persons

eport Section		Company	P
1.0	Summary	All	Sign off by Section
2.0	Introduction	Tetra Tech	John Huang, P.Eng.
3.0	Reliance on Other Experts	Tetra Tech	John Huang, P.Eng.
4.0	Property Description and Location	Tetra Tech	Margaret Harder, P.Geo.
5.0	Accessibility, Climate, Local Resources, Infrastructure, and Physiography	Tetra Tech	Margaret Harder, P.Geo.
6.0	History	Tetra Tech	Margaret Harder, P.Geo.
7.0	Geological Setting and Mineralization	Tetra Tech	Margaret Harder, P.Geo.
8.0	Deposit Types	Tetra Tech	Margaret Harder, P.Geo.
9.0	Exploration	Tetra Tech	Margaret Harder, P.Geo.
10.0	Drilling	Tetra Tech	Margaret Harder, P.Geo.

eport Section		Company	P
11.0	Sample Preparation, Analyses, and Security	Tetra Tech	Margaret Harder, P.Geo.
12.0	Data Verification	Tetra Tech	Margaret Harder, P.Geo./ Michael O Brien, M.Sc., Pr.Sci.Nat., FGSSA, FAusIMM, FSAIMM
13.0	Mineral Processing and Metallurgical Testing	Kemetco	Norm Chow, P.Eng.
14.0	Mineral Resource Estimates	Tetra Tech	Norm Tribe, P.Eng., B.A.Sc./ Michael O Brien, M.Sc., Pr.Sci.Nat., FGSSA, FAusIMM, FSAIMM
15.0	Mineral Reserve Estimates	Tetra Tech	Anoush Ebrahimi, P.Eng.
16.0	Mining Methods	Tetra Tech	Anoush Ebrahimi, P.Eng.
17.0	Recovery Methods	Tetra Tech	John Huang, P.Eng.
18.0	Project Infrastructure	Tetra Tech	Sign-off by Section
	18.1 Introduction	Tetra Tech	John Huang, P.Eng.
	18.2 Process Plant	Tetra Tech	John Huang, P.Eng.
	18.3 Maintenance and Storage	Tetra Tech	John Huang, P.Eng.
	18.4 Administration Building	Tetra Tech	John Huang, P.Eng.
	18.5 Building Services	Tetra Tech	John Huang, P.Eng.
	18.6 Site Access Roads	Tetra Tech	Anoush Ebrahimi, P.Eng.
	18.7 Power Supply and Distribution	Tetra Tech	Jerry Harris, PE, P.Eng.
	18.8 Temporary Waste Storage Facility	Tetra Tech	Marvin Silva, P.Eng
	18.9 Water and Waste Management	Tetra Tech	Marvin Silva, P.Eng.
19.0	Market Studies and Contracts	Tetra Tech	Sabry Abdel Hafez, Ph.D., P.Eng.
20.0	Environmental	Tetra Tech	John Huang, P.Eng.
21.0	Capital and Operating Costs	Tetra Tech	Sign-off by Section
	21.1 Capital Cost Estimate	Tetra Tech	Hassan Ghaffari, P.Eng.
	21.2 Operating Cost Estimate	Tetra Tech	John Huang, P.Eng.
	21.2.1 Mine Operating Costs	Tetra Tech	Anoush Ebrahimi, P.Eng.
22.0	Economic Analysis	Tetra Tech	Sabry Abdel Hafez, Ph.D., P.Eng.
23.0	Adjacent Properties	Tetra Tech	Margaret Harder, P.Geo.
24.0	Other Relevant Data and Information	Tetra Tech	Sabry Abdel Hafez, Ph.D., P.Eng.
25.0	Interpretation and Conclusions	All	Sign off by Section
26.0	Recommendations	All	Sign off by Section
27.0	References	All	Sign off by Section

3.0 RELIANCE ON OTHER EXPERTS

The authors wish to make clear that they are QPs only in respect of the areas in this report identified in their “Certificates of Qualified Persons” submitted with this report to the Canadian Securities Administrators.

The report has been reviewed for factual errors by AMI. Therefore, the statement and opinions expressed in this document are given in good faith and in the belief that such statements and opinions are neither false nor misleading at the date of this report. Tetra Tech has not independently verified the legal status or title of the claims or exploration permits, and has not investigated the legality of any of the underlying agreement(s) that may exist concerning the property.

The QPs who prepared this report relied on information provided by the following experts who are not QPs:

- Mr. Larry W. Reaugh, President and Chief Executive Officer, of AMI, has been relied on for advice on matters relating to Royalties.
- Mr. Tim Johnston and Mr. Bruce McGregor, of PricewaterhouseCoopers (PwC), have been relied on for advice on matters relating to taxes in the economic modelling. Mr. Tim Johnston is the engagement partner, a Mining Tax Partner with all his clients being mining companies. Mr. Bruce McGregor is a US Tax Partner and he conducted the partner level review of the model.
- Mr. James Marin, VP (Lands) for Rocher Manganese Inc (RMI), a wholly-owned subsidiary of AMI, has been relied on for matters relating to Mineral Dispositions and Tenure Rights.
- Ms. Catherine M. Virga, Director of Research of CPM Group, has been relied on for the study of the manganese market outlook and EMM flake prices used in the Project economic evaluation as provided in CPM Group’s report “Electrolytic Manganese Market Outlook,” 2012.

  CPM Group is the world s premier commodities research and consulting company. The firm’s primary focus is on precious, industrial, and specialty metals. Due to the lack of publicly available information on the EMM market, the QP relied on CPM Group’s EMM price forecast for the development of alternate case scenarios of the Project economic evaluation. In order to mitigate any potential risk associated with the valuation or pricing, the QP applied the conservative three-year trailing average pricing to the base case economic evaluation.
- Mr. Reese Hastings, JD, LPG, LHYD, Principal Scientist of Tetra Tech Inc., has been relied on for matters relating to environmental studies, permitting and social or community impact.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION

The Property is located in the Artillery Mountains of Mohave County, northwestern Arizona (Figure 4.1). The Property is approximately 170 km (108 miles) northwest of Phoenix, Arizona, and 240 km (150 miles) southeast of Las Vegas, Nevada. The nearest communities by road are Kingman, Arizona (approximately 140 km or 87 miles) and Lake Havasu, Arizona (145 km or 90 miles). The Property is centered at latitude 34 30’ N and longitude 113 50’ W, or at Universal Transverse Mercator (UTM) coordinates 257,000mE, 3,802,100mN (North American Datum (NAD) 83, Zone 12S).

The Property is located near the southern boundary of Mohave County. This area contains numerous manganese occurrences, and there are several small historical manganese mine workings on the Property. These include adits in Maggie Canyon and open pit mines in the MacGregor and Priceless deposits. Numerous smaller workings include the Rudy, Shannon and Alamo Queen deposits on the Maggie Wash; the Upper Chapin deposit on the Chapin Wash, the Love and Hurley deposits in the central Chapin Area; the Planche deposit on the eastern side of the Planche Mesa, and several small prospects in the Upper Burro Wash.

4.2 MINERAL DISPOSITIONS

The Property comprises 478 mining claims covering approximately 10,669 acres, of which approximately 2779 acres are private property Figure 4.2). The mining claims include 80 patented claims, two fee simple claims, and four unpatented claims which are under lease or option-to-purchase agreements, and 392 unpatented mining claims which are 100% owned by Rocher Manganese Inc. (RMI), a wholly-owned subsidiary of AMI In the past, the surface rights to 43 of the patented claims were sold separately, leaving split-title packages with mineral rights separate from the surface rights. The surface rights of 41 of these split-title packages have been redeemed and recombined with their mineral rights, and are now under the ownership of AMI The purchase of surface rights for the remaining two split-title claims is currently being negotiated. Surface rights of all unpatented claims are administered by the Bureau of Land Management (BLM) and permits are required for exploration activities, as summarized in Section 4.4.2. Claim agreements and tenure ownership are described in more detail in Section 4.2.

Figure 4.1 Property Location

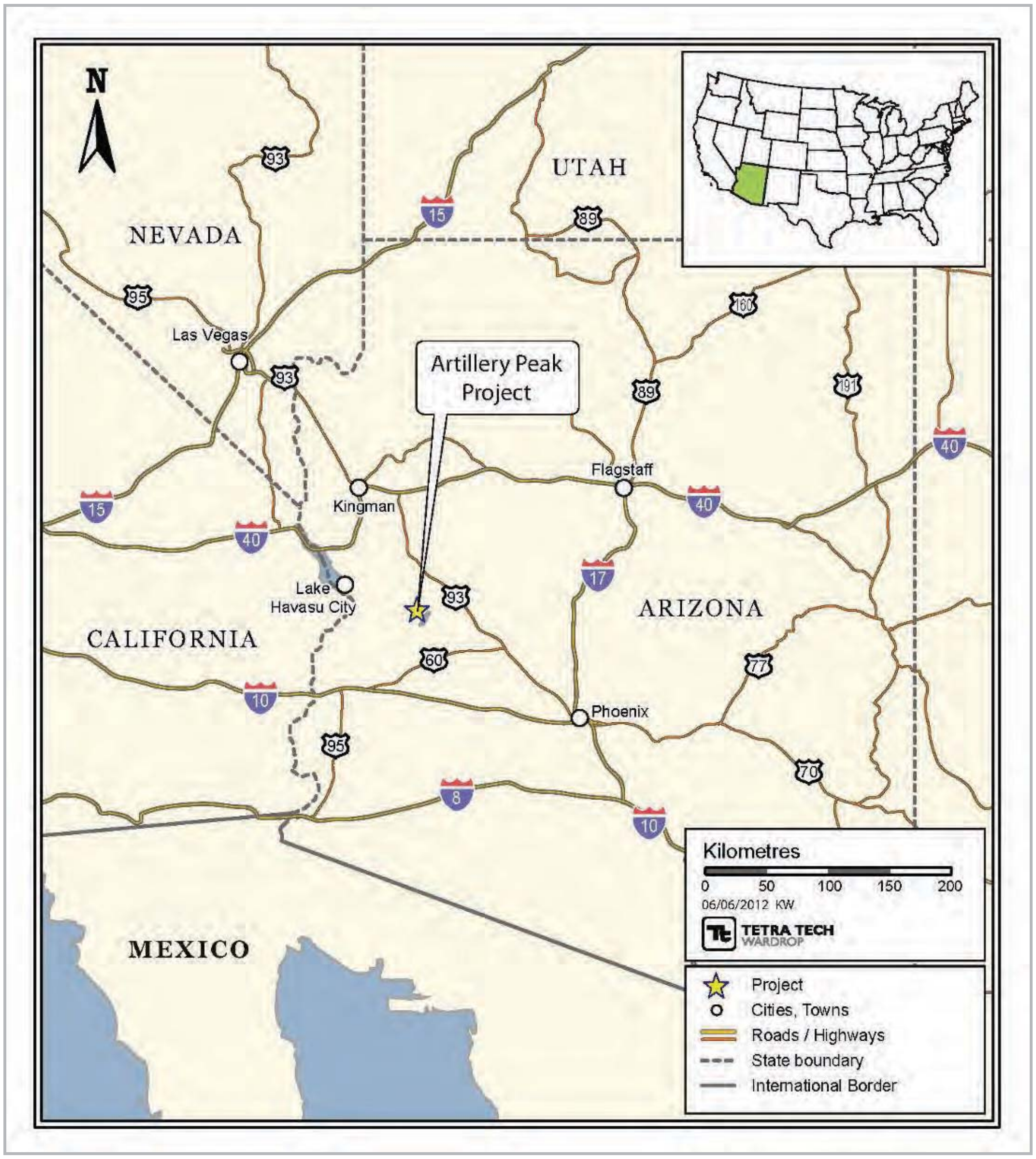
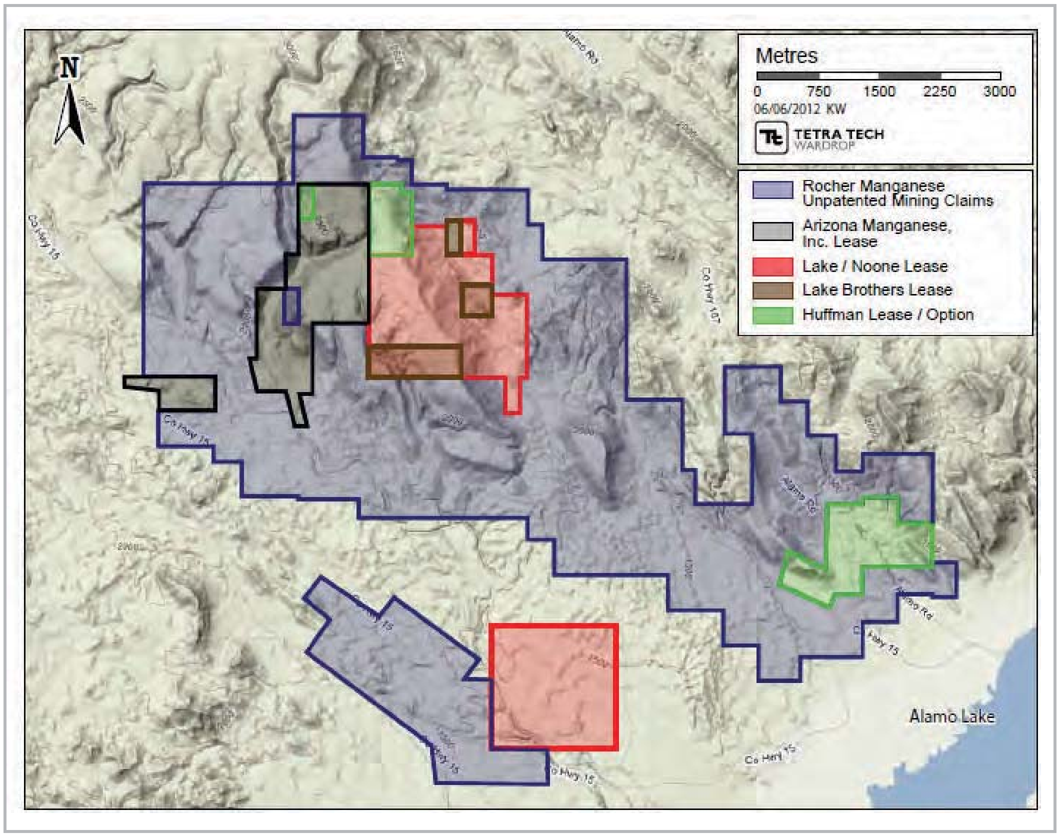


Figure 4.2 Property Tenure Map



The names and details of the claims covered by lease agreements are shown in Table 4.1 and the unpatented claims not covered by lease agreement in Table 4.2. All claims are in good standing and do not have expiry dates, pending annual payments required by the BLM and by Mohave County.

All mining claims in the United States fall under the Mining Law of 1872 (and any subsequent amendments therein) and require that every year the claimant file federal BLM fee documents to keep the mining claims active and in good standing.

These maintenance fee documents and attached annual fees ($140 per year for each claim) must be received and time stamped by the State Office of the BLM on or before August 31 of that assessment year. If the fees are not filed on or before this date, then the mining claims laps back into the public domain. Filing of these documents and fees are paid in advance for the upcoming assessment year, and therefore the next mandatory annual filing for claims on the Property will be for the assessment year of 2013.

Table 4.1 Mining Claims Covered nder Lease Agreements

Claim ame	Parcel o.	Claim Type	ecord Date	Lease Agreement	Mineral ig ts	Sur ace ig ts
Maggie 15-17 (odd numbers)	101-34-003	Patented	15-Jun-08	Huffman-Maggie Canyon	DJH	DJH
Maggie 22-25	101-34-003	Patented	15-Jun-08	Huffman-Maggie Canyon	DJH	DJH
Muroc	101-35-003	Patented	15-Jun-08	Huffman-Maggie Canyon	DJH	DJH
Muroc 1	101-35-003	Patented	15-Jun-08	Huffman-Maggie Canyon	AMC	DJH
Shannon-N2	101-13-003	Patented	15-Jun-08	Huffman-Maggie Canyon	AMC	RMI/AMI
Shannon-S2	101-13-002	Patented	15-Jun-08	Huffman-Maggie Canyon	AMC	DJH
Shannon 1-N2	101-13-003	Patented	15-Jun-08	Huffman-Maggie Canyon	AMC	RMI/AMI
Shannon 1-S2	101-13-002	Patented	15-Jun-08	Huffman-Maggie Canyon	AMC	DJH
Shannon 2-N2	101-13-003	Patented	15-Jun-08	Huffman-Maggie Canyon	AMC	RMI/AMI
Shannon 2-S2	101-13-002	Patented	15-Jun-08	Huffman-Maggie Canyon	AMC	DJH
Shannon 3-N2	101-13-003	Patented	15-Jun-08	Huffman-Maggie Canyon	AMC	RMI/AMI
Shannon 3-S2	101-13-002	Patented	15-Jun-08	Huffman-Maggie Canyon	AMC	DJH
Priceless 1	101-12-004	Patented	15-Jul-08	Huffman-Priceless	DJH	DJH
Priceless Annex	101-12-009	Patented	15-Jul-08	Huffman-Priceless	DJH	DJH
Priceless 5	101-12-009	Patented	15-Jul-08	Huffman-Priceless	DJH	DJH
Priceless Extension 3	101-11-017	Patented	15-Jul-08	Huffman-Priceless	DJH	DJH
Priceless Extension	101-11-013	Patented	15-Jul-08	Huffman-Priceless	DJH	DJH
Priceless Extension 6	101-11-013	Patented	15-Jul-08	Huffman-Priceless	DJH	DJH
Priceless 3	101-11-012	Patented	15-Jul-08	Huffman-Priceless	DJH	DJH
Priceless Extension 1	101-11-014	Patented	15-Jul-08	Huffman-Priceless	DJH	DJH
Priceless 11	101-11-010	Patented	15-Jul-08	Huffman-Priceless	DJH	DJH
Priceless 46	101-11-009	Patented	15-Jul-08	Huffman-Priceless	DJH	DJH
Priceless Extension 5	101-12-006	Patented	15-Jul-08	Huffman-Priceless	DJH	DJH
Priceless Extension 1A	101-11-011	Patented	15-Jul-08	Huffman-Priceless	DJH	DJH
Priceless Extension 4	101-11-018	Patented	15-Jul-08	Huffman-Priceless	DJH	DJH
Priceless Extension 3	101-11-016	Patented	15-Jul-08	Huffman-Priceless	DJH	DJH

Claim ame	Parcel o.	Claim Type	ecord Date	Lease Agreement	Mineral ig ts	Sur ace ig ts
Priceless	101-12-004	Patented	15-Jul-08	Huffman-Priceless	DJH	DJH
Priceless 45	101-12-004	Patented	15-Jul-08	Huffman-Priceless	DJH	DJH
Priceless 4	101-11-007	Patented	15-Jul-08	Huffman-Priceless	DJH	DJH
Priceless Extension 2	101-11-006	Patented	15-Jul-08	Huffman-Priceless	DJH	DJH
Price 10	AMC-392432	Unpatented	15-Jul-08	Huffman-Priceless	DJH	DJH
Price 14-16	AMC-392433-392435	Unpatented	15-Jul-08	Huffman-Priceless	DJH	DJH
Chapin 4	101-09-001	Patented	1-Aug-08	Lake-Noone	CEC	CEC
Chapin 15-20	101-37-001	Patented	1-Aug-08	Lake-Noone	CEC	CEC
Minnesota 2-4	101-37-001	Patented	1-Aug-08	Lake-Noone	CEC	CEC
Muroc 2-4	101-35-005	Patented	29-Sep-08	AMC-Maggie Canyon	AMC	RMI/AMI
Muroc 5-7	101-36-015	Patented	29-Sep-08	AMC-Maggie Canyon	AMC	RMI/AMI
Maggie 6	101-35-006	Patented	29-Sep-08	AMC-Maggie Canyon	AMC	RMI/AMI
Maggie 7	101-36-003	Patented	29-Sep-08	AMC-Maggie Canyon	AMC	RAH
Maggie 7A	101-35-004	Patented	29-Sep-08	AMC-Maggie Canyon	AMC	RMI/AMI
Maggie 13-14	101-35-004	Patented	29-Sep-08	AMC-Maggie Canyon	AMC	RMI/AMI
Maggie 31	101-35-004	Patented	29-Sep-08	AMC-Maggie Canyon	AMC	RMI/AMI
Maggie 8	101-36-015	Patented	29-Sep-08	AMC-Maggie Canyon	AMC	RMI/AMI
Maggie 9	101-36-017	Patented	29-Sep-08	AMC-Maggie Canyon	AMC	RMI/AMI
Maggie 12	101-36-016	Patented	29-Sep-08	AMC-Maggie Canyon	AMC	RMI/AMI
Maggie 16	101-36-016	Patented	29-Sep-08	AMC-Maggie Canyon	AMC	RMI/AMI
Maggie 26	101-36-015	Patented	29-Sep-08	AMC-Maggie Canyon	AMC	RMI/AMI
Mesa 1-2	101-36-012/013	Patented	29-Sep-08	AMC-Maggie Canyon	AMC	RMI/AMI
Mesa 3-4	101-36-009/008	Patented	29-Sep-08	AMC-Maggie Canyon	AMC	RMI-AMI
Mesa	101-36-011	Patented	29-Sep-08	AMC-Maggie Canyon	AMC	RMI/AMI
Rudy	101-36-014	Patented	29-Sep-08	AMC-Maggie Canyon	AMC	AMC
Rudy 1-2	101-36-015	Patented	29-Sep-08	AMC-Maggie Canyon	AMC	RMI/AMI
Rudy 5-8	101-36-015	Patented	29-Sep-08	AMC-Maggie Canyon	AMC	RMI/AMI

Claim ame	Parcel o.	Claim Type	ecord Date	Lease Agreement	Mineral ig ts	Sur ace ig ts
Rudy Fraction	101-36-015	Patented	29-Sep-08	AMC-Maggie Canyon	AMC	RMI/AMI
Rudy 3	101-10-001	Patented	29-Sep-08	AMC-Maggie Canyon	AMC	RMI/AMI
Shannon 4	101-13-003	Patented	29-Sep-08	AMC-Maggie Canyon	AMC	RMI/AMI
Shannon Annex 3-NS	101-13-003	Patented	29-Sep-08	AMC-Maggie Canyon	AMC	RMI/AMI
Shannon Annex 3-S2	101-13-002	Patented	29-Sep-08	AMC-Maggie Canyon	AMC	DJH
Priceless 2	101-12-008	Patented	29-Sep-08	AMC-Maggie Canyon	AMC	RAH
Section 16	101-08-003	Fee Simple	15-Mar-10	Lake Brothers	SL, JL, ML	SL, JL, ML
Minnesota 1	101-34-007	Patented	15-Mar-10	Lake Brothers	SL, JL, ML	SL, JL, ML
Section 32	101-33-003	Fee Simple	15-Mar-10	Lake Brothers	SL, JL, ML	SL, JL, ML

Notes: DJH David John Huffman
AMC Arizona Manganese Corporation
RMI/AMI Rocher Manganese/American Manganese, Inc.
CEC Chapin Exploration Company
RAH Robin A. Huffman
SL, JL, ML Steven R. Lake, James W. Lake, Mack C. Lake III

Table 4.2 npatented Mining Claims

Claim ame	Claim Serial umber (AMC)	Claim Status	Mineral ig ts	Sur ace ig ts
Artillery 1-14	383719-383732	Active	RMI/AMI	BLM
Artillery 21-25 (odd numbers)	383733-383735	Active	RMI/AMI	BLM
Artillery 27-40	383736-383749	Active	RMI/AMI	BLM
Artillery 49-54	383750-383755	Active	RMI/AMI	BLM
Artillery 67-73 (odd numbers)	383756-383759	Active	RMI/AMI	BLM
Artillery 78-81	383760-383763	Active	RMI/AMI	BLM
Artillery 84-95	383764-383775	Active	RMI/AMI	BLM
Artillery 99-107	383776-383784	Active	RMI/AMI	BLM
Artillery 109-112	383785-383788	Active	RMI/AMI	BLM
Artillery 177	383789	Active	RMI/AMI	BLM
Artillery 197	383790	Active	RMI/AMI	BLM
Artillery 200-201	383791-383792	Active	RMI/AMI	BLM
Mesa 5	383793	Active	RMI/AMI	BLM
Maggie 27	383794	Active	RMI/AMI	BLM
Mag 1-2	383795-383796	Active	RMI/AMI	BLM
Javelina 1-6	383797-383802	Active	RMI/AMI	BLM
Javelina 14-16 (even numbers)	383803-383804	Active	RMI/AMI	BLM
Javelina 21-27 (odd numbers)	383805-383808	Active	RMI/AMI	BLM
Artillery 191-196	391001-391006	Active	RMI/AMI	BLM
Artillery 202-207	391007-391012	Active	RMI/AMI	BLM
Burro 1-8	391013-391020	Active	RMI/AMI	BLM
Javelina 17-20	391021-391024	Active	RMI/AMI	BLM
Javelina 50-54	391025-391029	Active	RMI/AMI	BLM
M 1-6	391030-391035	Active	RMI/AMI	BLM
M 23	391036	Active	RMI/AMI	BLM
M 7-8	391993-391994	Active	RMI/AMI	BLM
Artillery 15-20	391995-392000	Active	RMI/AMI	BLM
Artillery 97-98	392001-392002	Active	RMI/AMI	BLM
Artillery 123-131 (odd numbers)	392003-392007	Active	RMI/AMI	BLM
Artillery 136-145	392008-392017	Active	RMI/AMI	BLM
Artillery 239-247	392018-392026	Active	RMI/AMI	BLM
Artillery 249-255	392027-392033	Active	RMI/AMI	BLM
Artillery 263-264	392034-392035	Active	RMI/AMI	BLM
Artillery 63-65 (odd numbers)	392422-392423	Active	RMI/AMI	BLM
Burro 0	392424	Active	RMI/AMI	BLM
Burro 15-17 (odd numbers)	392425-392426	Active	RMI/AMI	BLM
Burro 29	392427	Active	RMI/AMI	BLM
PR 1-4	392428-392431	Active	RMI/AMI	BLM
Artillery 146-147	393477-393478	Active	RMI/AMI	BLM
Artillery 256-257	393480-393481	Active	RMI/AMI	BLM
Burro 16	393482	Active	RMI/AMI	BLM

Claim ame	Claim Serial umber (AMC)	Claim Status	Mineral ig ts	Sur ace ig ts
Burro 19	393483	Active	RMI/AMI	BLM
Burro 30-34	393484-393488	Active	RMI/AMI	BLM
Burro 41	393489	Active	RMI/AMI	BLM
Javelina 15	393490	Active	RMI/AMI	BLM
MAG 3-19	393491-393507	Active	RMI/AMI	BLM
NM 1-10	393508-393517	Active	RMI/AMI	BLM
Artillery 133-135 (odd numbers)	393971-393972	Active	RMI/AMI	BLM
Artillery 148	393973	Active	RMI/AMI	BLM
Artillery 151	393974	Active	RMI/AMI	BLM
Artillery 265-267	393975-393977	Active	RMI/AMI	BLM
Burro 42-45	393978-393981	Active	RMI/AMI	BLM
Cob 1-3	393982-393984	Active	RMI/AMI	BLM
MAG 0	393985	Active	RMI/AMI	BLM
MAG 28	393986	Active	RMI/AMI	BLM
NM 11-13	393987-393989	Active	RMI/AMI	BLM
PR 7-17	393990-394000	Active	RMI/AMI	BLM
Artillery 115-122	408197-408204	Active	RMI/AMI	BLM
Artillery 124-130 (even numbers)	408205-408208	Active	RMI/AMI	BLM
Artillery 152-155	408209-408212	Active	RMI/AMI	BLM
Artillery 164-172	408213-408221	Active	RMI/AMI	BLM
Artillery 208-209	408222-408223	Active	RMI/AMI	BLM
MAG 29-48	408224-408243	Active	RMI/AMI	BLM
M 13-18	408244-408249	Active	RMI/AMI	BLM
M 24-55	408250-408281	Active	RMI/AMI	BLM
Javelina 22-24 (even numbers)	408282-408283	Active	RMI/AMI	BLM
Javelina 31-32	408284-408285	Active	RMI/AMI	BLM
Javelina 55-56	408286-408287	Active	RMI/AMI	BLM
Burro 9-11	408288-408290	Active	RMI/AMI	BLM
Burro 13	408291	Active	RMI/AMI	BLM
PR 18-24	408292-408298	Active	RMI/AMI	BLM
Artillery 41-48	Pending	Active	RMI/AMI	BLM
Artillery 55-61 (odd numbers)	Pending	Active	RMI/AMI	BLM
Artillery 68-74 (even numbers)	Pending	Active	RMI/AMI	BLM
Artillery 300-306	Pending	Active	RMI/AMI	BLM
South 1-12	Pending	Active	RMI/AMI	BLM
South 40-44	Pending	Active	RMI/AMI	BLM

Notes: RMI/AMI Rocher Manganese/American Manganese, Inc.
BLM Bureau of Land Management

4.3 TENURE RIGHTS

In 2007, AMI entered into a purchase agreement with Primus Resources, LC (Primus), to acquire 90 unpatented mining claims. The rest of the unpatented claims were staked following this purchase. The patented and fee simple lands on the Property were obtained through five separate mineral leases and/or lease option agreements with the relevant companies or individuals. The tenure rights and royalties for the unpatented, patented, and fee simply tenure types are summarized below. Section 4.3.4 provides a summary of the terms of agreements for leases on patented and fee simple lands. The names and details of the claims are also shown in Table 4.1 and Table 4.2, including lease agreements.

4.3.1 *UNPATENTED MINING CLAIMS*

The 396 unpatented mining claims are subject to the following royalty provisions:

- All 396 unpatented claims are subject to a 2.0% NSR to Primus, as agreed during purchase of the PRIMUS-90 claims. Half of this royalty can be purchased back from the vendor for $2,000,000.
- 392 of the unpatented mining claims are also subject to a $0.04/lb royalty on manganese production in favour of James L. Lake and Barton and Peter Noone.
- All 396 unpatented mining claims are subject to a $0.01/lb royalty on manganese production in favour of James L. Lake, or his heir or assigns.

Annual 2012 Federal Bureau of Land Management (FBLM) maintenance fees and county filings have been paid and submitted in a timely fashion. All unpatented claim locations are valid and in good standing.

4.3.2 *PATENTED MINING CLAIMS*

The patented mining claims are subject to the following royalty provisions:

- 80 patented claims are subject to a US$0.01/lb royalty on manganese production in favour of James L. Lake, or his heir or assigns.
- Ten of the patented claims are subject to a royalty of US$0.04/lb on manganese and a 1.5% NSR on other mineral production in favour of James L. Lake and Barton and Peter Noone.
- One patented claim is subject to a US$0.04/lb royalty on manganese and a 1.5% NSR on other mineral production in favour of the Lake Brothers (Lake Brothers' Lease Agreement).
- 43 patented claims are subject to a 2.25% NSR in favour of Arizona Manganese Corporation (AMC) (AMC-Maggie Canyon Lease Agreement).

- 19 patented claims have an exclusive right of option to purchase from the Lessor (David J. Huffman) for $2,250,000, with the purchase price increasing by 2% on each anniversary of the Lease (Huffman-Priceless Lease Agreement). There are no other royalties, obligations or adhesions in this buyout option.
- Seven patented additional claims have an exclusive right of option to purchase from the Lessor (David J. Huffman) for $1,000,000, with the purchase price increasing by 2% on each anniversary of the Lease (Huffman-Maggie Canyon Agreement). There are no other royalties, obligations or adhesions in this buyout option.

4.3.3 *Fee Simple Lands*

There are two fee simple primary parcels subject to the following royalty provisions:

- All parcels are subject to a $0.04/lb royalty on manganese and a 1.5% NSR on other mineral production in favour of the James Lake and the Lake Brothers and Barton and Peter Noone.
- All parcels are subject to a $0.01/lb royalty on manganese production in favour of James L. Lake, or his heirs or assigns.

4.3.4 *Terms of Agreements*

Five separate mineral leases or lease options exist for patented and fee simple lands on the Property. A brief summary of these five leases is provided below.

Huffman-Maggie Canyon Lease

In 2008, RMI entered into a ten-year lease-option agreement, the Huffman-Maggie Canyon Lease, with David John Huffman to lease 13 patented mining claims (totalling 203.35 acres). Only seven of the 13 patented claims included both the mineral and surface rights.

The annual terms of this lease option are minimum annual payments of $10,000 and a right to purchase the claims for $1,000,000 with the price increasing by $20,000 each year. No other royalties, obligations or adhesions exist on the lease.

Huffman-Priceless Lease

In 2008, RMI entered into a second, ten-year lease-option agreement, the Huffman-Priceless Lease, with David John Huffman to lease 19 patented mining claims (totalling 318.47 acres), along with four unpatented mining claims (totalling 80 acres).

The annual terms of this lease option are minimum annual payments of $20,000 and a right to purchase the claims for $2,250,000 with the price increasing by $50,000 each year. No other royalties, obligations or adhesions exist on the lease. The four unpatented claims also carry a 2% royalty to Primus.

Lake-Noone Lease

In 2008, RMI entered into a ten-year renewable lease agreement, the Lake-Noone Lease, with James L. Lake, Barton Noone and Peter Noone, to lease 10 patented mining claims and five parcels of fee simple lands (totalling 1,167 acres).

The annual terms of this lease agreement are payments of $60,000 in the first year and increasing annual payments by $20,000 up to the seventh year when the annual payments will be $200,000. An additional annual payment of $10,000 will be required if any manganese minerals from the Property are being processed. Royalties on the lease include $0.04/lb of manganese production and 1.5% of net returns on any metals

AMC-Maggie Canyon Lease

In 2008, RMI entered into a 20-year renewable lease agreement, the AMC-Maggie Canyon Lease, with AMC to lease the mineral rights of 42 patented mining claims. Surface rights of these claims were sold to third parties and RMI has subsequently purchased full surface rights for 33 of these claims and has obtained surface rights for six claims in other lease or option agreements. An additional patented claim was added to this lease in 2009.

The annual terms of this lease agreement are payments of $50,000 in Year 1, $55,000 in each of Years 2 through 4, $65,000 in each of Years 5 through 10, and $70,000 annually after Year 10.

Lake Brothers' Lease

In 2010, RMI entered into a renewable 10-year lease agreement, the Lake Brothers' Lease, with the Lake Brothers. This lease includes one patented claim (20.66 acres) and two parcels of fee simple lands (320 acres).

The annual terms of this lease are payments of one third of the total of the advance minimum royalty payments to each individual on the lease. The royalty includes $0.04/lb of manganese produced from the Property and 1.5% NSR from production of any non-manganese minerals, split equally between the three individuals.

4.4 PERMITS AND LICENCES

4.4.1 PRIVATE LANDS PATENTED AND FEE SIMPLE CLAIMS

AMI has obtained two permits required by the State of Arizona including a “Mining Multi-Sector General Stormwater Permit” and “De Minimis Discharge Permit.” Work is in progress for a number of the federal and state permits.

Detailed discussions of federal and state permits and licenses for the private lands are included in Section 20.

4.4.2 PUBLIC LANDS UNPATENTED CLAIMS

AMI has obtained two permits required by the State of Arizona including a “Mining Multi-Sector General Stormwater Permit” and “De Minimis Discharge Permit.” The important clearance for the project involves the NEPA, which will be led by the BLM. This will involve evaluation of the environmental impacts of the project. In order to complete the NEPA evaluation process, the Plan of Operations must be submitted along with a document of the evaluated environmental impacts. Work on the NEPA process and other federal and state permits and licences are in progress.

Detailed discussions of federal and state permits and licenses for the public lands are included in Section 20.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 PHYSIOGRAPHY

The Property is located on the southeastern edge of the Mojave Desert and is part of the Basin and Range physiographic province. Flat, arid valleys, or basins, between fault-bound mountain chains with common mesas and plateaus characterize the topography. The mountain chains are oriented sub-parallel and the elevation difference between the 'basins' and 'ranges' can be as high as 800 m. The Property lies within the Artillery Mountains, whose highest point is Artillery Peak at approximately 890 masl. Artillery Peak is located approximately 4.8 km (3 miles) north of the Property (Figure 4.2). Total elevation range on the Property is around 400 m.

Although bodies of water are rare, the southeastern edge of the Property is approximately 2 km (1.2 miles) from Alamo Lake. Vegetation on the Property is generally sparse and is dominated by Mesquite, Joshua Trees, Juniper, Palo Verde, Acacias, Ironwood, Ocotillo cactus, Saguaro cactus and Prickly Pear. An abundance of Tamarisk, an introduced species, is noted in Chapin Wash. Wild donkeys, deer, peccary and coyotes are present in the area. Chaparral Cock (Road Runners) are present in the valley desert, and California Quail and Mourning Doves are present near the water of Lake's Lake (named after Mr. James L. Lake) and near Alamo Lake.

5.2 ACCESSIBILITY

The Property is accessible by unpaved roads off Highway 40 or Highway 93. The unpaved roads are generally navigable year-round, although access to the Property does require crossing dry creek beds, which can be impassable for several days following rainstorms. The nearest communities by road are Kingman, Arizona (approximately 140 km or 87 miles) and Lake Havasu City, Arizona (145 km or 90 miles). The Property is located between the Alamo Road to the west and the Burro Wash to the east. From Highway 40, the Property can be accessed from the Alamo Road just south of the town of Yucca, Arizona. The distance along the Alamo Road to Highway 40 is approximately 100 km (62 miles). From Highway 93, the Property can be accessed via Chicken Springs Road at the town of Wikieup, Arizona, travelling west for approximately 25 km (15 miles), and joining the Alamo Road approximately 55 km (34 miles) north of the Property. The town of Yucca is approximately 40 km (25 miles) south of Kingman, and the town of Wikieup is approximately 90 km (55 miles) southeast of Kingman.

Access to the western portion of the property is from numerous mine roads branching off from the Alamo Road. Access to the eastern part of the property and those claims along Chapin Wash is by continuing south on the Alamo Road to the Huffman Road, travelling 3 km east to the Chapin Wash road, and then continuing north into the Property.

5.3 CLIMATE

The Property is located on the southeastern edge of the Mojave Desert, characterized by a desert climate with low precipitation and extreme hot temperatures in summer months, and generally mild conditions in winter months. The nearest community in a direct line that has regular weather data collection is Lake Havasu City, Arizona (located 65 km or 40 miles northwest of the Property), and these data are used as a proxy for typical conditions on the Property (averages for the years 1981-2010; Western Regional Climate Center). The coldest month is December, with an average daily temperature of 11.7°C (53.0°F), and the warmest month is July with an average daily temperature of 36.1°C (96.9°F). The wettest month is January, with an average precipitation of 19.3 mm (0.76") and the driest month is June, with an average rainfall of 0.3 mm (0.01"). The average yearly precipitation is 97.5 mm (3.84"). Torrential rainfalls can occur, which leave the washes flooded and roads impassable for several days at a time.

5.4 LOCAL RESOURCES

The nearest cities to the Property are Kingman and Lake Havasu City, which support populations of approximately 28,000 and 52,000, respectively. Both cities offer all basic amenities and have airports serviced by regional airlines.

5.5 INFRASTRUCTURE

The nearest power line follows Signal Road as far as Signal, a distance of about 21 km from the Property. The state high voltage power grid is located on Highway 93 approximately 40 km to the east. A local feeder line delivers power to the Alamo dam approximately 10 km to the west of the Property. Use of this local line would require upgrading the line as far as Highway 93.

During the 2008, 2010, and 2011 drilling programs, water was hauled from a small pond (Lake's Lake) on the patented claims of James L. Lake, located just west of the MacGregor Pit. A potable water well was drilled at the camp just south of Lake's Lake; limited water was encountered in this hole, and ground water is likely insufficient for use in a processing plant. Water source will be sought from groundwater wells in an area to the southeast of the mine area, close to the Bill Williams River, which is approximately 4 km to 6 km from the Property.

Roads accessing the Property are in good condition and well maintained; however, no other infrastructure is available at the property.

.0 HISTORY

Manganese was first discovered in the region around 1880, in outcrops along the Alamo to Signal road (Lasky and Webber, 1949). The first known claims staked in the region were by the Rodgers brothers in 1914, and the deposit on these claims was named the Graham deposit (which is now considered part of the McGregor deposit). Several other exposures had been explored by 1918 as a result of demand for manganese during the First World War (WWI). However, no manganese ore was shipped from the Property until 1928.

In 1928, the Chapin xploration Company (Chapin) acquired 1,700 acres of land, including the area staked by the Rodgers Brothers (Lasky and Webber, 1949; Farnham and Stewart, 1958). Private Chapin reports estimated the claims to contain approximately 66,000,000 short tons of ore averaging 8.5 Mn; however, it is not evident what exploration or estimation methods these values are based on. Chapin was reported to have produced ore from the Black Warrior, Graham, and Big Jim deposits in 1928 (Farnham and Stewart, 1958), and there were unconfirmed reports of shipments to Alabama of about four railcars of ore containing 41 to 45 Mn (Lasky and Webber, 1949).

In 1930, D.W. Woodbridge discovered manganese outcropping in Maggie Canyon, and AMC staked approximately 2,000 acres in this area; however, little work was done until 1936. In 1936, the M.A. Hanna Company (M.A. Hanna) staked 2,500 acres in the region of Maggie Canyon, and they conducted exploration from 1936 to 1941 in conjunction with the US Bureau of Mines. xploration included surface cuts, shallow underground workings and diamond drilling (Lasky and Webber, 1942). A total of 48 vertical drillholes were drilled by M.A. Hanna and 15 drillholes were drilled by the Federal Bureau of Mines from 1940 to 1942 (Farnham and Stewart, 1958). This work was a joint investigation of the strategic mineral resources of the nation by the Geological Survey and the Bureau of Mines. During this time, the Bureau of Mines built a tunnel in Maggie Canyon and removed several thousand short tons of ore for mining method evaluations and metallurgical testing (Lasky and Webber, 1949). From 1949 to 1951, the Bureau of Mines also completed 58 drillholes from underground adits in the Maggie deposit (Farnham and Stewart, 1958; Tribe, 2011).

Production on the Property resumed in 1940 and continued through 1945, in response to demand from the Second World War (WWII) (Farnham and Stewart, 1958). Authentic records of this production do not exist. From 1940 to 1942, Arizona Manganese explored the Property by an adit and branch drifts, and may have conducted mining. From 1940 to 1945, .W. McGregor staked numerous claims, and started mining the Black Warrior, Graham and Big Jim deposits (which were previously mined by Chapin). There are no reliable records of the amount of ore mined or the grade of the material, but Farnham and Stewart (1958) estimated that around 5,000 short tons of sorted high-grade ore was shipped.

Lasky and Webber (1949) provided a resource estimate of the total manganese content in the Artillery Peak region, based on the diamond drilling completed by M.A. Hanna and the US Bureau of Mines. Tonnages were estimated based on a density of approximately 2.00 g cm^3 (16 ft^3 ton), which was obtained from SG measurements on 32 samples of rock. Lasky and Webber (1949) estimated the Artillery Peak region contained 200,000,000 short tons averaging 3   to 4   Mn. This historical estimate is based on limited drilling, incomplete assay data and broad assumptions and therefore is not considered reliable.   o QP has done sufficient work to classify the historical estimate as a current mineral resource; the issuer is not treating the historical estimate as a current mineral resource.

In the post-WWII period, the Artillery Peak region was actively mined as part of the US Government's strategic stockpile, and better records of this mining exist than for the pre-WWII work. From 1952-1955, approximately 79,300 long tons of ore averaging 18.5   Mn was produced from the Property and shipped to government stockpiles in Wenden, Arizona and Deming,   ew Mexico (Farnham and Stewart, 1958). This crude ore produced close to 61,000 long tons of concentrate averaging 30   Mn. In addition to crude ore shipped directly to the stockpiles, an estimated 300,000 short tons of ore was mined and milled from 1951 to 1956 by various operators, producing approximately 61,000 short tons of concentrate. A summary of the production and exploration on the Property by various operators over the period 1950 to 1956 is provided in Table 6.1; the data is summarized from Farnham and Steward (1958). This summary required some interpretation of the description provided by Farnham and Stewart (1958), and Tetra Tech has not verified the data.

In 1958, the US Bureau of Mines completed a statistical analysis of results from diamond drillholes on the Maggie Canyon deposit (Hazen, 1958). The results are summarized by three different estimation techniques cross-section (25,129,693 short tons at 4.73   Mn), polygon (27,387,872 short tons at 5.45   Mn) and triangle (27,596,489 short tons at 5.35   Mn)[1]. These historical estimates are not considered reliable.   o QP has done sufficient work to classify the historical estimate as a current mineral resource; the issuer is not treating the historical estimate as a current mineral resource.

Mining works on the Property ceased in 1956; a summary of production from all mineral deposits in the State of Arizona estimated that, from 1946 to 1959, 243,335 long tons (95,108,000 lb) of manganese was mined from the Artillery Peak area (Keith et al., 1983). This summary differs from that of Farnham and Stewart (1958), indicating that the historical records are not entirely reliable and provide only a general estimate of production from the Property.

[1] Source http   www.americanmanganeseinc.com properties artillery-peak

Tribe (2011) reports that additional surface drillholes were completed in 1958; however, it is not stated who completed the drilling, or the location or total meterage of the holes.

To the knowledge of Tetra Tech, no further mining or exploration was conducted on the Property until 2007, when AMI acquired the McGregor and Chapin claims from Primus. It is not known when Primus obtained rights to these claims. Most of the claims on the Property were historically owned by individuals, and these claims were passed on to later generations. As a result, numerous separate leases with the individuals currently owning the claims were required for AMI to obtain rights to them; these leases are described in Section 4.3.

Table 6.1 Production from Private Companies on the Property, 1950-1956

Operator	Year	Activities	Deposit/Claims	Production/Grade
Mohave Mining and Milling Co.	1953-1955	Open pit mining	Price and Priceless claims	175,000 tons of raw ore mined producing 51,000 long tons of concentrate; average grade 29 Mn
Al Stovall; F.A. Sitton	1953-1955	Surface open-cuts and underground workings	McGregor claims including Black Warrior, Graham and Big Jim deposits	9,350 long tons of washed concentrate averaging 33.8 Mn
F.A. Sitton	1955	Open pit mining	Alamo Queen deposit	Several thousand tons of ore
C. eal and C.C. Patterson	1952-1955	Open pit and underground workings	Black Diamond claims	8,600 tons averaging 19.3 Mn
F.A. Sitton	1955	Open pit and underground workings	Black Diamond claims	600 tons averaging 19.3 Mn
J. Lewis and C. eal	1953-1954	Surface open-cuts and underground workings	Black agle claims	356 long tons averaging 28 Mn
Ike W. Kusisto	1954	Surface open-cuts and pits	Shannon claims	50 tons averaging 15 Mn
Arizona Manganese Corp.	1954	Surface open-cuts	Lone Star claims	79 long tons averaging 22.6 Mn
World Manganese Corp.	1954-1956	Open pit and underground workings	Alamo Queen, eedle ye, Lone Star, and Black Diamond deposits	eedle ye claims - 476 long tons of sorted ore averaging 21 Mn; 711 long tons of concentrate containing 31-40.2 Mn; Alamo Queen deposit mining rate of several hundred tons daily
Hewitt West	1953	Surface open-cuts	Rudy claims	15 tons averaging 36.5 Mn; reported but unrecorded earlier production by Arizona Manganese (Corp. or Inc.)
Hewitt West	1950-1954	Underground workings	Maggie claims	8,500 tons of selectively mined ore averaging 16.2 Mn

Operator	Year	Activities	Deposit/Claims	Production/Grade
Al Stovall	1953-1954	Underground Workings	Lake and Psilomelane claims	33,126 long tons averaging 18.8 Mn (Lake deposit) and 20,000 tons averaging 18.4 (Psilomelane deposit)
Arizona Metals Corp.	1954	Underground Workings	Black Jack deposit	1,230 long tons of sorted ore averaging 20.6 Mn
S.J. Love	1954	Adit development	Psilomelane and Plancha Mountain claims	o production recorded
Les eilson	1953-1954	Surface open-cuts	Black Crow claims	287 tons averaging 18.6 Mn
R.B. Hurley	1954	Adit development	Black Warrior claims	80 tons averaging 10.6 Mn
R.S. Rodgers	1953	Unknown mining method	Last Chance deposit	180 long tons averaging 34.4 Mn
Tate, Johnson, and McBride	1954-1955	Underground workings	Polianite Claims	1,600 long tons averaging 21 Mn; additional small undisclosed ore shipments
R.J. Carpenter	1953-1954	Surface open-cuts and underground workings	Black Mollie claims	614 long tons averaging 18.3 Mn
T.J. Rodgers and J.M. eal	1952-1954	Surface open-cut	Oversight claims	1,360 long tons averaging 10-20 Mn

.0 GEOLOGICAL SETTING AND MINERALI ATION

.1 REGIONAL GEOLOGY

The Property is located within the Basin and Range Physiographic Province of the southwestern United States. In the Oligocene, a major shift occurred in the plate tectonic setting off the southwestern coast of the United States. The major style of plate movement changed from subduction to strike-slip, resulting in the extension of the crust and development of the Basin and Range Province. The lithosphere extension resulted in repeated normal faulting and downward displacement of blocks of the crust, as well as local volcanism. This formed a horst and graben landscape characterized by upthrust horst blocks, largely forming the mountain ranges in the region and down-faulted graben or basins. The Property is on the edge of the Artillery Mountains, and lower-elevation areas of the Property form a basin between this range and the Rawhide Mountains to the southeast. The Basin and Range Province remains tectonically active today.

The Precambrian sedimentary rocks that are exposed in the Grand Canyon to the north of the Property once extended over the area of the Property, but remnants of these rocks in the region of the Property are rare. Paleozoic and Mesozoic sediments are observed in some areas of western Arizona, indicating that the region was part of a continental shelf setting prior to the subduction that has affected the western margin of orth America since the late Cretaceous. Uplift and extensive erosion subsequently removed most of these earlier sedimentary rocks. The younger Cenozoic sedimentary rocks present in the region are related to faulting and volcanism during formation of the Basin and Range Province and the resulting basin-fill sedimentation.

The faulting in the region juxtaposes Late Proterozoic to arly Paleozoic metamorphic rocks against much younger Cenozoic sedimentary and volcanic rocks (Figure 7.1). The metamorphosed basement rocks include metagranite, gneiss, schist, marble, slate and quartzite rocks (Lasky and Webber, 1949). Younger sedimentary rocks vary, from sandstones and shales deposited in lacustrine and alluvial settings in the basin floor, to alluvial fan and playa deposits along the basin margins. Volcanic rocks range from common basaltic to andesitic flows to ignimbrites produced by larger, more explosive volcanic eruptions.

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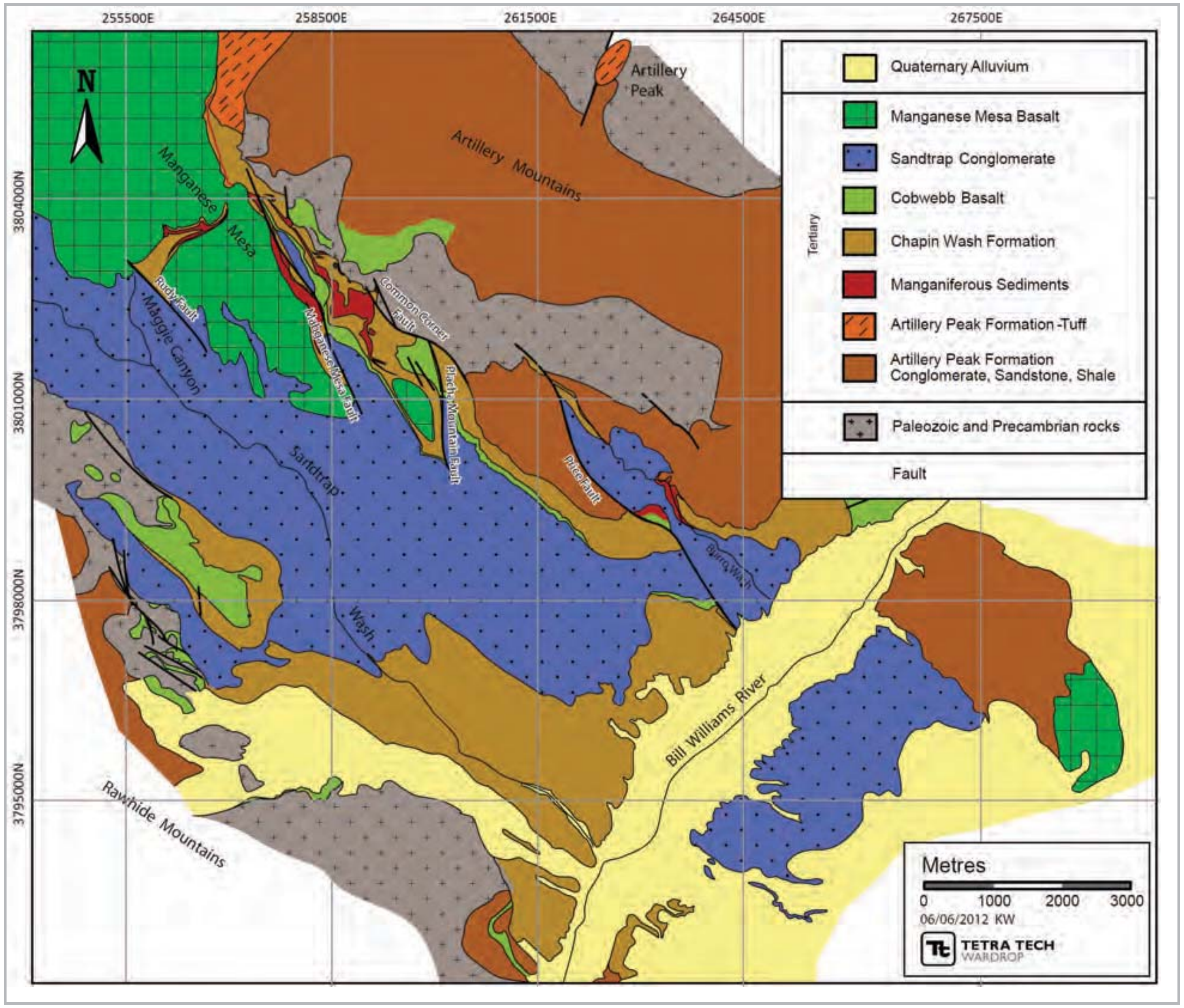


.2 LOCAL GEOLOGY

The Property is located on the southwestern edge of the Artillery Mountains and in the basin enclosed between the Artillery and Rawhide Mountains. On the Property, the oldest exposed rocks are Proterozoic-age, coarse-grained gneiss and metagranitic rocks (Lasky and Webber, 1949). These basement rocks are unconformably overlain by sedimentary and volcanic rocks of the Artillery Formation. The Artillery Formation is interpreted to comprise closed-basin deposits of shale, conglomerate, sandstone and minor lacustrine limestone and tuff, as well as a widespread basalt flow (Lasky and Webber, 1949). In the region of Manganese Mesa, the Artillery formation comprises a number of sandstone and conglomerate units, a basalt flow, and a megabreccia, which was interpreted as a catastrophic debris-avalanche deposit containing large, intact blocks of rocks that can be hundreds of metres in size (Spencer, 1991). The Artillery Formation is cut by the Santa Maria Peak intrusion, which has been dated at approximately 20 Ma and therefore provides a minimum age of the Artillery Formation. Both the Artillery Formation and Santa Maria Peak intrusion were then exposed by faulting and eroded prior to subsequent deposition.

The Artillery Formation is unconformably overlain by the Chapin Wash Formation. The Chapin Wash Formation comprises a series of sedimentary rocks that have been interpreted to represent alluvial fans and playa deposits, and is the unit that hosts the stratiform manganese deposits (Sanford and Stewart, 1948). The Chapin Wash Formation includes sandstone, siltstone, shale and conglomerate, as well as minor beds of tuff and lacustrine limestone. The unit is up to 450 m thick, and thins towards the margins of the basin, pinching out in other sedimentary units. The alluvial fans were interpreted to have formed within closed basins as a result of the faulting that formed the horsts and grabens that characterize the region. The sediments were likely mainly deposited during and following the periodic flooding that occurs during flash rainstorms (Sanford and Stewart, 1948). Although individual beds within the Chapin Wash Formation are inter-fingered, making correlation difficult in areas, the larger-scale alluvial fan is more homogeneous and continuous (Tribe, 2011).

In most areas of the Property, the Chapin Wash Formation is overlain by the Cobwebb Basalt, which is dated at 13.3 Ma (Spencer, 1991). This basalt is significant because it has reduced the impact of subsequent erosion on the underlying sediments. The Cobwebb Basalt is conformable with the underlying Chapin Wash Formation. Mineralization typically appears within a few metres to a few tens of metres below the Cobwebb Basalt, and it may have acted as a barrier to the mineralizing fluids (Tribe, 2011).

The Sandtrap conglomerate unconformably overlies the Cobwebb Basalt and Chapin Wash Formation, and is a generally soft and friable unit. The Sandtrap conglomerate is interbedded with basalt flows, which are dated at 9.5 Ma (Spencer, 1991). The mesas in the region are topped with a younger basalt flow up to 10 m thick, which does not appear to be associated with manganese mineralization. As with the Cobwebb basalt, this younger basalt flow has reduced the effect of erosion on the underlying sediments.

The Property lies above a gently northeast-dipping, large displacement, normal fault known as the Buckskin-Rawhide detachment fault (Spencer, 1991). During normal faulting, the younger units were displaced downwards relative to the older and more resistant Precambrian and Paleozoic rocks that form the core of the Rawhide and Artillery ranges. This faulting established the general position of the Artillery and Rawhide mountains and the basin between the two ranges (Lasky and Webber, 1949), and the Cenozoic sediments and volcanic rocks were deposited into this basin. ormal faulting was ongoing during deposition of these units, resulting in progressive tilting of older strata. Strata in the sedimentary sequence on the Property all dip to the southwest, but strata at the base of the sequence have dip angles of approximately 30° to 40° (and as high as 70°; Lasky and Webber, 1949), whereas those at the top of the sequence have dip angles of 10° to 20° (Spencer, 1991). Several faults are observed on the Property the Common Corner Fault, the Price Fault, the Burro Wash Fault, the Manganese Mesa Fault, the Rudy Fault, the MacGregor and MacGregor Wash Faults, the Cobwebb Cross Fault, the Planche and West Planche Faults, and the Shannon Fault (Tribe, 2011; Figure 7.2). Possible volcanic vents are present along these faults. These faults control the present-day topography to a large degree, and erosion is generally most pronounced along faults; some fault scarps are still sharply defined (Lasky and Webber, 1949).

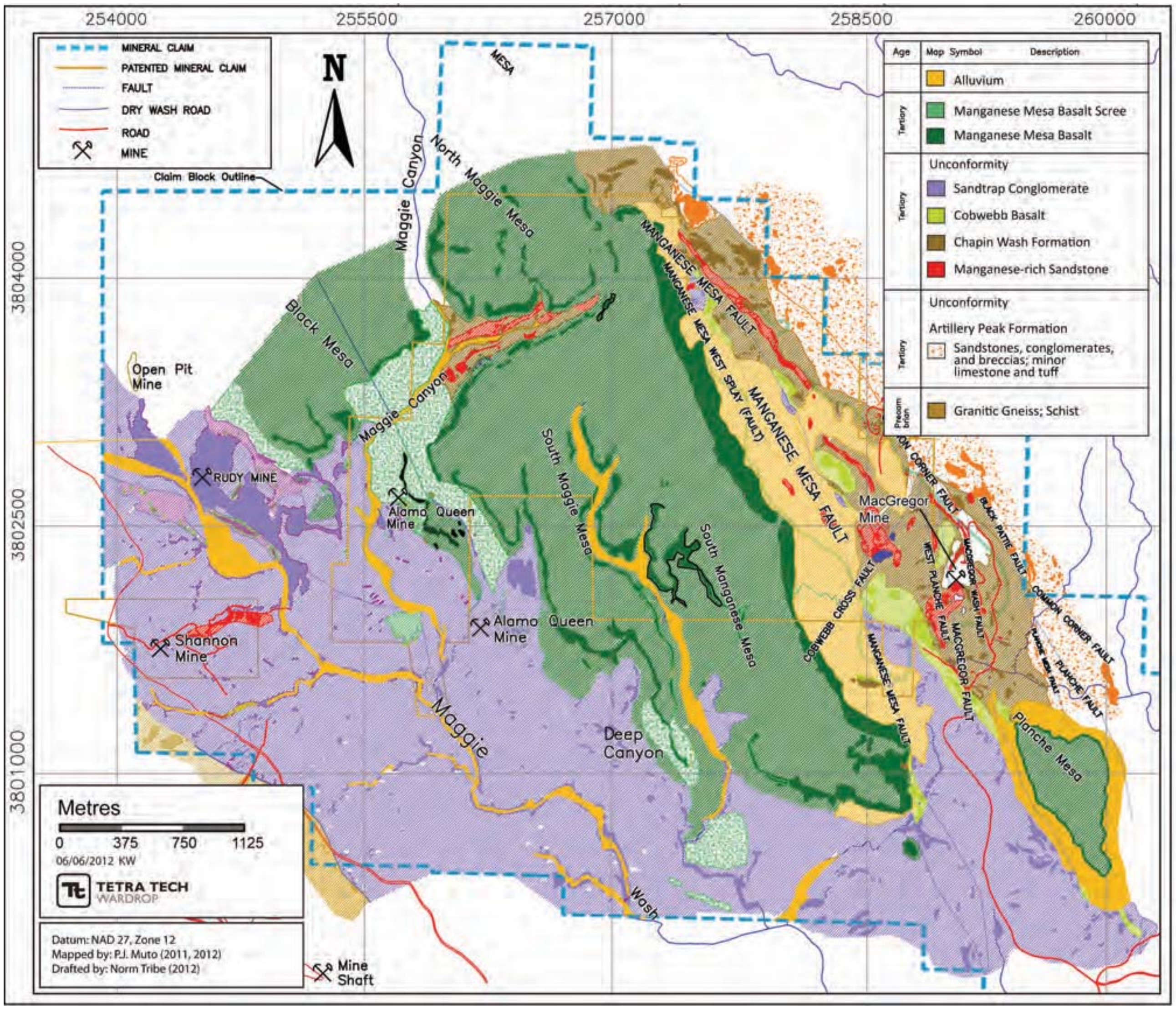
.3 MINERALI ATION

Two distinct styles of manganese mineralization are observed on the Property stratiform manganese deposits, and vein manganese deposits.

.3.1 *STRATI RM MA A ESE EP SITS*

The mineralization style responsible for the majority of manganese on the Property is stratiform manganese, which occurs within the Chapin Wash Formation. The stratiform deposits are relatively low-grade, and are exposed in two northwest-trending belts, with grades typically around 3 to 4 Mn and locally 6 to 10 Mn (Spencer, 1991). The southwestern belt contains lenses of mineralization that can be several tens of metres thick, separated by non-mineralized sediments, and the majority of manganese is hosted within sandstone. Little historical mining occurred within this belt, and most of the belt lies to the southwest of the Property boundary. The northeastern belt is approximately 5 km long, ranges in thickness from a few metres to many tens of metres, and manganese is hosted with sandstones and siltstones. The northeastern belt is more continuous and contains much more manganese than the southwestern belt, and has many historical mines including the Maggie and MacGregor Mines.

i ure . Property Geolo y



Manganese in the stratiform deposits is hosted as very fine-grained oxides in pore spaces in the sediments (Spencer, 1991). A detailed study of manganese minerals by Mouat (1962) used petrography and -ray diffraction to analyze the mineralization, and frequently found the minerals too fine-grained for proper identification. The manganese mineralization is of the type called wad manganese, an amorphous manganese oxide ($K_{22}Ba_{20}Sr_7Al_2Fe_{25}Mn_{315}O_{557}$) with some manganite ($MnO(OH)$), psilomelane ($(BaH_2O)_2Mn_5O_{10}$) and minor pyrolusite ($MnO_2$) (Tribe, 2011). The likely presence of needle-like cryptomelane ($K(Mn^{4},Mn^{2})_8O_{16}$) and hollandite ($Ba(Mn^{4},Mn^{2})_8O_{16}$) is also suggested by Mouat (1962). The manganese-bearing minerals form a coating on sedimentary particles, and locally occur as cement between the particles. Manganese minerals are also occasionally observed as fracture fill within the stratiform deposits. There is no evidence of mineral replacement by manganese minerals, and they appear to be present only as interstitial minerals.

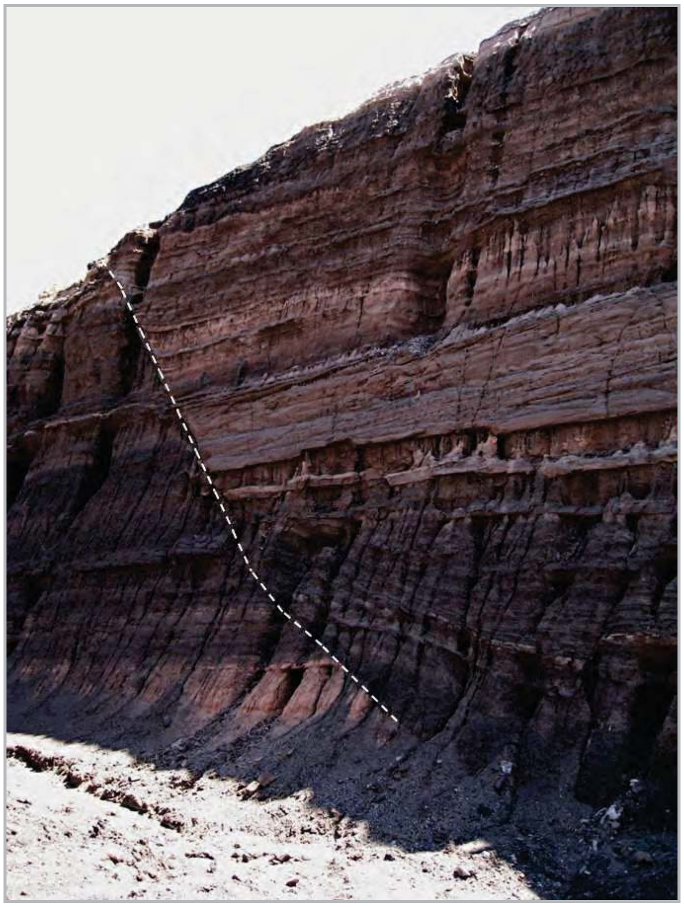
In some areas, interstitial material to the sediments has recrystallized, likely due to interaction with meteoric water, and formed what has historically been known as hard ore. The hard ore is associated with calcite, and locally silica cement, and there appears to have been movement and recrystallization of manganese associated with this process. Overall, the hard ore has a similar grade to the rest of the stratiform manganese, but locally the grades exceed 20 Mn (Lasky and Webber, 1949), indicating some local enrichment of manganese. Within hard ore material, Mouat (1962) identified hollandite, psilomelane and cryptomelane in thin sections, as well as possible ramsdellite (MnO_2), by -ray diffraction.

Figure 7.3 shows the Cobwebb Hill escarpment and illustrates stratiform manganese within the Chapin Wash Formation. The formation is cut by a small normal fault on the left side of the photo (marked by the dotted white line), and strata in the hangingwall are mildly deformed at the base of the visible stratigraphic section. Figure 7.4 shows a closer view of a manganese-rich sandstone boulder.

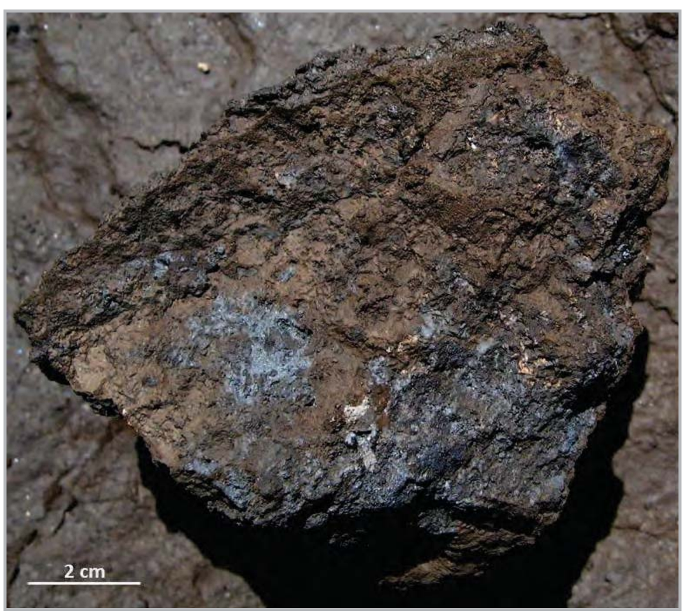
.3.2 EI MA A ESE EP SITS

Manganese also occurs within veins, breccias, and fracture and fault zone cement, and overall this style of mineralization is referred as vein manganese. Veins are locally common within or near the northeastern stratiform deposits (Spencer, 1991), where mineralization is observed along faults, fractures and bedding planes in some areas, and occurs within other units in addition to the Chapin Wash Formation, particularly within the Sandtrap Conglomerate. These veins also occur within breccia zones that have been interpreted to be possible volcanic vents (Tribe, 2011). The mineralized vein-fill is often characterized by colloform manganese oxide along with calcite and barite, and these veins are typically a few centimetres thick (Spencer, 1991).

i ure . Photo raph of tratiform an anese inerali ation



i ure . Photo raph of an anese- ich andstone



Both the Priceless and Price occurrences are vein-type deposits, where the veins fill breccias that may have been volcanic vents (Tribe, 2011). The Priceless occurrence is the largest such deposit observed on the Property, and is approximately 400 m long x 80 m wide and sits on a slight flexure in the Burro Fault (Tribe, 2011). The Price occurrence is located just southeast of the Priceless occurrence and is much smaller, at approximately 100 m x 50 m in size. Both of these occurrences were historically mined as open pits.

Mouat (1962) used petrology and -ray diffraction to determine the manganese minerals present in vein-style mineralization. These minerals include common hollandite, cryptomelane and psilomelane, as well as less common ramsdellite and lithiophorite ((Al,Li)Mn $^{4}O_2(OH)_2$).

.0 DEPOSIT TYPES

.1 DEPOSIT TYPES

The manganese deposits occur as two distinct types of deposits, as summarized in Section 7.3 stratiform manganese deposits and vein manganese deposits. Possible deposition mechanisms are briefly discussed below; however, the origin of both types of deposits, and in particular that of the stratiform manganese deposits, remains unclear (Spencer, 1991).

.1.1 STRATI RM MA A ESE EP SITS

arly workers on the Property suggested that the stratiform manganese deposits were syngenetic with the Chapin Wash Formation sediments hosting the deposit, and that the manganese oxides were transported and deposited by mechanical processes (Lasky and Webber, 1949). The presence of the manganese oxides within different types of sediments (sandstone, siltstone), the continuity of the deposits, the inter-fingering of barren and mineralized beds, and apparent sedimentary features were thought to be evidence of syngenetic deposition by sedimentary processes. arly interpretation of the manganese source was uncertain, but was inferred to be detrital and formed from erosion of pre-existing manganese mineralization occurring at or near the earth’s surface (Spencer, 1991). Tribe (2011) also interpreted the stratiform deposits to be syngenetic with sedimentation, and further suggested a source of the manganese to be volcanic vents, which were interpreted to have been present along faults on the Property.

Other workers have suggested that the stratiform manganese deposits are related to secondary metasomatism by potassium-bearing fluids (Roddy et al., 1988; Spencer, 1991). These workers identified evidence of potassium metasomatism over large areas of Arizona, which occurred in the Miocene and was locally pervasive in the Artillery Mountains. This type of alteration can affect large volumes of rock, and therefore the chemical and hydrological conditions producing the potassium alteration could feasibly be responsible for manganese transportation and precipitation within inter-granular spaces in the sediments (Roddy et al., 1988; Spencer, 1991). Replacement of pre-existing minerals is not necessarily required by this mechanism, particularly if the sediments were not cemented at the time of alteration.

Some redistribution of manganese occurred as a result of interaction with meteoric water, and this material is referred to as hard ore. The hard ore is primarily due to the presence of calcite and silica cement; however, some remobilization of manganese occurred, and locally resulted in higher concentrations of manganese. Compared to the rest of the stratiform mineralization, the hard ore is more resistant to weathering, and therefore crops out more prominently. This hard ore is a distinctive feature of Maggie Canyon, where it forms a cliff that resembles the overlying basalt (Lasky and Webber, 1949). Previous workers suggest that the hard ore was formed by redistribution of manganese as a result of meteoric fluids which were likely at or near atmospheric temperatures and pressures.

The Cobwebb basalt that immediately overlies the Chapin Wash Formation was likely an important factor for preservation of the mineralized sediments, and Spencer (1991) suggests that the basalt may have acted as a barrier for potassium metasomatic fluids. Mineralization typically occurs within a few metres to tens of metres below the Cobwebb basalt (Tribe, 2011), which may suggest a genetic link between the mineralization and this basalt flow. The younger basalt flow that caps the mesas in the region does not appear to be associated with manganese mineralization, except to protect the deposits from physical erosion. The mineralization likely occurred prior to eruption of this younger basalt.

.1.2 EI MA A ESE EP SITS

Vein-type deposits occur within or near the northeastern belt of stratiform deposits, and comprise manganese oxides, calcite and barite (Spencer, 1991). These deposits have been dated at several million years younger than the stratiform deposits (Spencer, 1991). Although the two types of deposits are therefore not obviously related, their spatial association does suggest some genetic link.

The occurrence of the veins as fracture, fault and breccia fill, as well as fluid inclusion studies indicate, a hydrothermal source of the fluids (Roddy et al. 1988; Spencer, 1991). These fluids may be related to the possible vents on the Property interpreted by Tribe (2011). Breccia deposits are observed along flexures in faults, which are possibly related to the formation the horst graben boundaries. The source of manganese in these deposits could be directly from the hydrothermal fluids. Alternatively, the hydrothermal fluids could have redistributed some manganese from the pre-existing stratiform deposits, which may explain the spatial relationship of the two types of deposits (Spencer, 1991).

Vein-type deposits are smaller, irregular and locally higher grade than the stratiform deposits. The size of vein deposits does not support modern large tonnage bulk mining practices. However, these deposits were mined historically in the Shannon, Priceless and Price mines.

.0 E PLORATION

.1 CHANNEL SAMPLING

Tribe (2011) describes several channel-sampling exercises conducted in 2007 and 2008. In 2007, chip channel samples were taken from the MacGregor pit face to evaluate the general grade of the deposits. These results are not provided in detail, but the report indicates a potential for large deposits of 4 Mn. The deposits were deemed of interest and a second round of sampling was completed in 2008, consisting of chip channel sampling at 50 m intervals, cut vertically on the pit face of the MacGregor open pit. ine channels were collected and analyzed for manganese content, and these channels are treated as vertical drillholes for resource estimation. The coordinates at the starting point of each channel are provided in Table 9.1, and a summary of results from these channel samples is provided in Table 9.2.

Table 9.1 tartin Coordinates of 00 Channel amples

ole D	astin m	orthin m	levation m	en th m
MPF-1	258965	3802622	650.00	9.14
MPF-2	259024	3802601	662.00	21.34
MPF-3	259048	3802560	669.00	27.43
MPF-4	259066	3802523	667.00	27.43
MPF-5	259102	3802489	648.00	18.29
MPF-6	259145	3802489	650.00	21.34
MPF-7	259099	3802357	674.00	24.38
MPF-8	259100	3802304	553.00	3.14
MPF-9	259068	3802280	563.00	18.29

Table 9. ummary of esults from 00 Channel amples

ole	rom m	To m	nterval m	Grade n
MPF-1	0.00	9.14	9.14	7.82
MPF-2	0.00	21.34	21.34	6.42
MPF-3	0.00	9.14	9.14	3.69
MPF-3	12.19	27.43	15.24	7.37
MPF-4	3.05	6.10	3.05	4.95
MPF-4	12.19	15.24	3.05	1.50
MPF-5	18.29	27.43	9.14	5.09

ole	rom m	To m	nterval m	Grade n
MPF-6	3.05	18.29	15.24	6.69
MPF-7	0.00	24.38	24.38	4.76
MPF-8	0.00	3.05	3.05	4.58
MPF-9	0.00	15.24	15.24	5.89

.2 METALLURGICAL SAMPLING

In 2011, two bulk samples were collected from the Cobwebb Hill escarpment. The samples were collected with a small backhoe in two vertical trenches, limited in size to the reach of the backhoe, or roughly 2 m deep x 3 m high. This material was sent to the Kemetco metallurgical laboratory in Richmond, British Columbia, for pilot plant scale processing tests. Approximately 8 t (17,500 lb) of material was shipped. Results of this testing are provided in Section 13.

.3 GEOLOGICAL MAPPING

Detailed geological mapping at a 1 6000 scale was conducted in 2011 and 2012, and the results of this mapping are presented in Figure 7.2.

.4 DENSITY SAMPLING

Fifteen density samples were collected in late 2011 from representative rock types on the Property. These samples were dominantly collected in the area of the MacGregor mine, which is within the area that is the focus of this technical report. Both mineralized and non-mineralized (country-rock or waste) samples were collected. The results of this sampling are presented in Table 9.3.

Table 9. ummary of esults from 011 Density amples

oc Type	G t/m
Mineralized material (MacGregor Mine area)	3.29
Mineralized material (MacGregor Mine area)	2.27
Mineralized material (MacGregor Mine area)	2.27
Hard ore material (MacGregor Mine area)	2.71
Mineralized material, bulk sample cliff face	1.96
Mineralized material, bulk sample cliff face	1.96
Waste (sandstone) inter-bedded with mineralization (MacGregor Mine area)	2.15
on-mineralized sandstone	2.59
on-mineralized sandstone	2.17

oc Type	G t/m
on-mineralized sandstone	2.44
Basalt (fresh)	2.63
Basalt (fresh)	2.71
Basalt (vesicular)	1.25
Basalt (amygdaloidal)	1.85
Basalt (weathered)	1.99
Avera e G inerali ed aterial	. 1
Avera e G aste andstone	.
Avera e G aste asalt	.09

10.0 DRILLING

AMI conducted the drilling on the Property in three separate programs in 2008, 2010 and 2011; each drilling program is briefly described below. The drillhole collar coordinates, as made available to Tetra Tech, required evaluation and adjustment to match the current topographic surface. Only holes that were within the area included in the resource estimate (Section 14) were fully re-evaluated, and collar locations lying outside the estimated area are not considered reliable without further evaluation. A total of 151 drillholes were completed on the Property by AMI from 2008 to 2011, for a total of 17,663 m.

10.1 200 DRILLING

Drilling in 2008 was completed by contractor Major Drilling from Denver, Colorado. All holes were diamond drillholes and the drillcore was Q diameter. Drilling was completed using a truck-mounted diamond drill rig at roughly 50 m centres lengthwise along the MacGregor pit, and staying 50 m behind the pit walls. Further sampling along strike or down dip from the MacGregor pit was not permitted in 2008, due to limitations imposed by the Bureau of Land Management (BLM) pending more complete archaeological and environmental studies.

A total of 17 drillholes were completed in the 2008 core-drilling program, for a total of 3,011 m. Of these holes, eight (totalling 821 m) were drilled in the MacGregor pit area and within the area of the current resource estimate. Figure 10.1 shows the location of 2008 drillholes, marked by green stars.

Table 10.1 provides a list of 2008 AMI drillhole collar locations and total hole lengths. Table 10.2 lists significant results for these holes; results are only provided for drillholes included in the current resource estimate.

Table 10.1 00 A Drillholes

ole D	astin m	orthin m	levation m	en th m
ADH1	258580	3803084	563.88	114.94
ADH2	259109	3802548	584.13	32.01
ADH3	258991	3802568	614.68	193.45
ADH4	259024	3802308	563.88	145.73
ADH5	259156	3802426	558.08	46.49
ADH6	258992	3801656	534.17	187.80
ADH7	258922	3801448	490.24	301.60
ADH8	260080	3801619	270.64	116.46
ADH9	260205	3801402	471.22	244.51
ADH10	259869	3801470	587.68	200.30
ADH11	260108	3800685	481.15	304.88
ADH12	260288	3801161	470.20	304.88
ADH13	260466	3800513	470.38	304.88
ADH14	261149	3800002	462.04	304.88
ADH15	259010	3802392	570.06	39.94
ADH16	259017	3802480	590.95	60.98
ADH25	261527	3799857	458.08	107.32

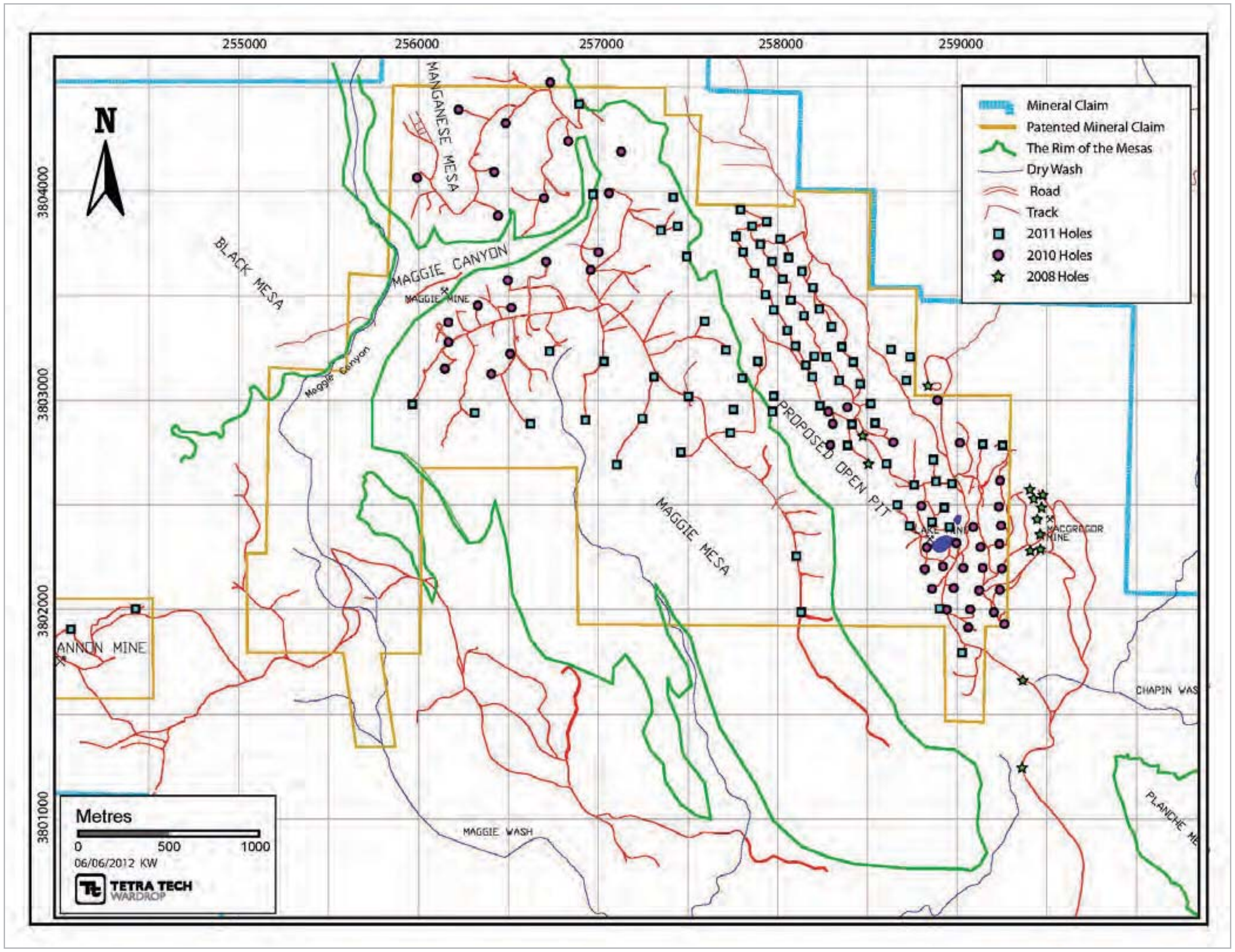
ote Collar locations have not been checked by Tetra Tech

Table 10. ummary of i nificant esults from 00 Drillholes ncluded in Current esource stimate

ole	rom m	To m	nterval m	Grade n
ADH1	3.05	12.19	9.14	1.23
ADH2	0.00	9.14	9.14	5.18
ADH3	0.00	9.14	9.14	4.41
ADH3	15.24	36.58	21.34	4.23
ADH3	57.91	67.06	9.14	1.91
ADH4	9.14	30.48	21.34	3.52
ADH6	48.77	60.96	12.19	2.14
ADH15	12.19	39.62	27.43	3.91
ADH16	30.48	57.91	27.43	4.02

ote only significant results over 4.5 m intervals reported here, at a 0.9 Mn cut-off.

i ure 10.1 ocation ap of 00 , 010 and 011 Drillholes



10.2 2010 DRILLING

Drilling in 2010 was completed by contractor Brown Drilling from Kingman, Arizona, and all drilling was by a rotary percussion reverse circulation (RPRC) method. The equipment used was a skid-mounted rotary percussion rig with a down-the-hole hammer. Most of the 2010 RPRC program was completed with dry drilling. If water was encountered in the hole, more water was introduced through the air stream to keep the sample fluid enough for the airlift to deliver the sample to the cyclone and sample collector. The diameter of all holes was 12.7 cm (5"), and the material produced was discharged into a 61 cm (24") cyclone. This configuration was chosen to minimize volume loss in dust. The sampling station was also designed to minimize the release of dust and drill cuttings at the drill site.

A total of 53 RPRC holes were drilled in 2010, for a total of 4,649 m. Of these holes, 33 holes (totalling 2,615 m) were drilled within the area included in the current resource estimate (Section 14). The other 20 holes (totalling 2,034 m) were drilled on the Maggie Canyon deposits. Figure 10.1 shows the location of 2010 drillholes, marked by purple circles.

Table 10.3 provides a list of 2010 AMI drillhole collar locations and total hole lengths. Table 10.4 lists significant results for these holes; results are only provided for drillholes included in the current resource estimate.

Table 10. 010 A Drillholes

ole D	astin m	orthin m	levation m	en th m
AP10-1	258765	3801910	522.60	100.58
AP10-2	258747	3802008	523.78	100.58
AP10-3	258643	3802004	532.50	91.44
AP10-4	258672	3802098	533.46	112.78
AP10-5	258576	3802108	533.40	73.15
AP10-6	258804	3802306	559.23	73.15
AP10-7	258922	3801924	518.16	74.66
AP10-8	258899	3802096	533.40	79.25
AP10-9	258857	3801996	526.41	128.02
AP10-10	259024	3802308	563.88	30.48
AP10-11	258721	3801800	534.07	112.78
AP10-12	258633	3802196	544.41	85.34
AP10-13	258697	3802298	556.30	73.15
AP10-14	258717	3802200	563.88	99.06
AP10-15	258773	3802406	573.74	60.96
AP10-16	258909	3802406	588.42	35.05
AP10-17	258813	3802200	545.75	82.30
AP10-18	258804	3802098	542.84	85.34

ole D	astin m	orthin m	levation m	en th m
AP10-19	258905	3802200	535.99	91.44
AP10-20	258900	3802312	559.27	76.20
AP10-21	258690	3802900	555.30	38.10
AP10-22	258708	3802800	553.94	72.54
AP10-23	258908	3802498	594.91	60.96
AP10-24	258897	3802596	595.63	60.96
AP10-25	258810	3802500	574.15	85.34
AP10-26	258801	3802604	570.80	62.48
AP10-27	258789	3802700	583.67	38.10
AP10-28	258886	3802704	613.71	27.43
AP10-29	258531	3802200	533.40	91.44
AP10-30	258546	3802298	539.38	103.63
AP10-31	258527	3802502	548.64	121.92
AP10-32	258381	3802800	570.97	106.68
AP10-33	256553	3803572	745.00	85.34
AP10-34	256570	3803439	745.00	82.30
AP10-35	256249	3803362	737.00	105.16
AP10-36	256265	3803275	740.00	134.11
AP10-37	256404	3803453	737.00	103.63
AP10-38	256131	3804087	741.00	121.92
AP10-39	256492	3803899	764.00	106.68
AP10-40	256482	3804084	778.00	115.82
AP10-41	256310	3804397	811.00	121.92
AP10-42	256544	3804331	806.00	147.83
AP10-43	256742	3804484	827.80	91.44
AP10-44	256720	3803488	776.00	80.77
AP10-45	256833	3804247	801.00	70.10
AP10-46	257095	3804206	782.73	73.15
AP10-47	257025	3803998	774.00	91.44
AP10-48	256946	3803623	769.00	64.01
AP10-49	256244	3803154	742.00	137.16
AP10-50	256462	3803126	732.00	92.96
AP10-51	256551	3803222	740.00	121.92
AP10-52	256733	3803664	757.00	86.87
AP10-53	256984	3803717	770.22	79.25

ote Collar locations have not been checked by Tetra Tech

Table 10. ummary of i nificant esults from 010 Drillholes ncluded in Current esource stimate

ole	rom m	To m	nterval m	Grade n
AP10-2	64.01	70.10	7.62	4.05
AP10-3	67.06	71.63	4.57	2.81
AP10-4	59.44	67.06	9.14	2.38
AP10-6	19.81	22.86	4.57	2.70
AP10-6	38.10	44.20	6.10	2.27
AP10-6	45.72	56.39	10.67	3.86
AP10-8	30.48	36.58	6.10	1.55
AP10-10	1.52	18.29	16.76	3.82
AP10-12	48.77	59.44	10.67	2.30
AP10-13	10.67	15.24	4.57	2.89
AP10-13	28.96	33.53	6.10	1.54
AP10-13	35.05	45.72	10.67	3.71
AP10-13	50.29	54.86	4.57	1.55
AP10-14	59.44	65.53	6.09	2.38
AP10-14	68.58	73.15	4.57	2.15
AP10-15	10.67	16.76	6.10	2.74
AP10-15	18.29	30.48	12.19	3.38
AP10-15	35.05	41.15	6.10	2.82
AP10-16	0.00	7.62	7.62	3.42
AP10-16	9.14	13.72	4.57	3.77
AP10-17	71.63	77.72	6.10	2.27
AP10-18	24.38	28.96	4.57	1.25
AP10-19	42.67	47.24	4.57	2.97
AP10-19	57.91	73.15	15.24	2.84
AP10-20	6.10	10.67	4.57	1.03
AP10-20	21.34	36.58	15.24	3.18
AP10-20	41.15	45.72	4.57	2.41
AP10-22	21.34	25.91	4.57	2.49
AP10-23	3.05	21.34	18.29	4.83
AP10-23	22.86	32.00	9.14	3.00
AP10-24	0.00	25.91	25.91	4.67
AP10-25	1.52	7.62	6.10	4.04
AP10-25	13.72	32.00	18.29	3.60
AP10-25	35.05	39.62	4.57	4.90
AP10-26	1.52	15.24	13.72	4.00
AP10-26	18.29	27.43	9.14	3.73
AP10-29	30.48	36.92	9.14	1.87
AP10-31	15.24	21.34	6.10	1.55
AP10-31	27.43	39.62	12.19	4.11
AP10-31	44.20	51.82	7.62	2.61

ole	rom m	To m	nterval m	Grade n
AP10-32	50.29	54.86	4.57	2.30
AP10-32	56.39	60.96	4.57	2.13
AP10-32	62.48	73.15	10.67	2.33
AP10-32	76.20	88.39	12.19	4.44
AP10-32	91.44	97.54	6.10	3.07

ote only significant results over 4.5 m intervals reported here, at a 0.9 Mn cut-off.

10.3 2011 DRILLING

Drilling in 2011 was completed by two drill rigs operated by two contractors Drill Tech Inc., from Chino Valley, Arizona, and an independent subcontractor from lko, evada. The 2011 drilling was completed with a truck-mounted rotary percussion rig, and drilling was wet using water pumped from Lake's Lake. All holes were RPRC holes, with a diameter of 12.7 cm (5"). The change to wet drilling following the 2010 program was primarily due to changes in safety rules forbidding dry drilling in mines. Although the Property is at the exploration stage and there is no current production from mining, mining regulations were followed in order to avoid any regulatory problems.

During the 2011 program, 81 RPRC holes were completed for a total of 10,003 m. Of these holes, 56 holes (totalling 6,238 m) were drilled within the area included in the current resource estimate (Section 14), and 25 holes (totalling 3,765 m) were drilled on the Maggie Canyon deposits. Figure 10.1 shows the location of 2011 drillholes, marked by blue squares.

Figure 10.2 shows the truck-mounted RPRC rig used during the 2011 program. Table 10.5 provides a list of 2011 AMI drillhole collar locations and total hole lengths. Table 10.6 lists significant results for these holes; results are only provided for drillholes included in the current resource estimate.

i ure 10. P C i – 011 Drillin Pro ram

Table 10.5   011 A   Drillholes

ole D	astin m	orthin m	levation m	en th m
AP11-54	258676	3802596	548.64	61.00
AP11-55	258446	3802300	542.92	103.70
AP11-56	258665	3802388	548.64	61.00
AP11-57	258491	3802594	554.14	106.70
AP11-58	258479	3802696	559.20	91.50
AP11-59	258484	3802798	560.95	82.30
AP11-60	258601	3803004	560.20	45.70
AP11-61	258806	3802798	564.64	45.70
AP11-62	258902	3802790	614.47	30.50
AP11-63	258380	3803236	563.88	61.00
AP11-64	258475	3803192	563.88	45.70
AP11-65	258406	3802482	559.89	61.00
AP11-66	258460	3803118	563.88	61.00
AP11-67	257954	3803610	619.60	61.00
AP11-68	258096	3803348	587.32	164.60
AP11-69	258044	3803440	593.78	140.20
AP11-70	258002	3803532	622.92	97.60
AP11-71	258190	3803192	582.51	73.20
AP11-72	258235	3803096	609.60	61.00
AP11-73	258278	3802990	607.70	73.20
AP11-74	258189	3802988	594.36	118.90
AP11-75	258121	3803094	596.94	152.40
AP11-76	258014	3803212	609.78	170.70
AP11-77	258280	3802804	582.27	128.00
AP11-78	258077	3803202	596.73	176.80
AP11-79	258360	3802694	576.00	134.10
AP11-80	257962	3803390	627.61	176.80
AP11-81	258199	3802898	600.21	140.20
AP11-82	257906	3803476	636.53	140.20
AP11-83	258406	3802598	564.56	109.80
AP11-84	257859	3803568	643.73	164.60
AP11-85	258475	3802402	545.10	176.80
AP11-86	257808	3803654	632.53	158.50
AP11-87	258148	3803278	583.33	103.63
AP11-88	257759	3803740	631.76	122.00
AP11-89	257885	3803696	613.74	60.96
AP11-90	257714	3803824	642.02	79.25
AP11-91	257854	3803776	610.25	54.86
AP11-92	257782	3803848	616.74	67.06
AP11-93	258305	3802892	587.86	40.88
AP11-94	257667	3803910	630.13	73.15

ole D	astin m	orthin m	levation m	en th m
AP11-95	258268	3802701	593.42	164.59
AP11-96	257630	3803778	673.38	79.25
AP11-97	258182	3802786	620.25	170.69
AP11-98	257663	3803716	660.82	91.44
AP11-99	258626	3802494	548.64	67.06
AP11-100	257725	3803600	663.95	109.73
AP11-101	257782	3803508	668.62	182.88
AP11-102	257823	3803412	670.34	140.21
AP11-103	257879	3803332	656.16	176.78
AP11-104	257922	3803256	639.84	170.69
AP11-105	257964	3803176	621.75	158.50
AP11-106	257991	3803084	627.61	213.36
AP11-107	258104	3802899	606.31	213.36
AP11-108	258579	3802410	548.64	115.82
AP11-109	258596	3802592	548.64	60.96
AP11-110	258577	3802690	551.50	54.86
AP11-111	258039	3802982	624.08	195.07
AP11-112	257735	3803184	717.00	115.82
AP11-113B	257608	3802842	752.00	128.02
AP11-114	257920	3802258	722.00	85.34
AP11-115	257377	3802746	732.00	128.02
AP11-116	257418	3802978	757.00	121.92
AP11-117	257332	3803791	759.00	140.21
AP11-118	256913	3802906	734.00	213.36
AP11-119	257358	3803703	770.00	128.02
AP11-120	257010	3803207	754.00	176.78
AP11-121	257342	3803912	762.00	134.11
AP11-122	256745	3803232	748.00	176.78
AP11-123	257063	3802695	735.00	146.30
AP11-124	257581	3803251	772.00	170.69
AP11-125	157198	3802947	745.00	170.69
AP11-126	257474	3803376	778.00	182.88
AP11-127	257230	3803118	758.00	134.11
AP11-128	256086	3802965	737.00	207.26
AP11-129	256957	3803926	752.95	109.73
AP11-130	256388	3802935	729.00	170.69
AP11-131	256890	3804440	797.00	109.73
AP11-132	256664	3802874	745.00	176.78
AP11-134	257946	3802047	726.00	213.36
AP11-W LL	258611	3802024	533.40	219.46

ote Collar locations have not been checked by Tetra Tech

Table 10.6 ummary of i nificant esults from 011 Drillholes ncluded in Current esource stimate

ole	rom m	To m	nterval m	Grade n
AP11-54	10.67	18.29	7.62	2.15
AP11-56	15.24	22.86	7.62	2.68
AP11-56	25.91	35.05	9.14	1.50
AP11-56	39.62	45.72	6.10	2.07
AP11-57	9.14	22.86	13.72	2.38
AP11-57	27.43	35.05	7.62	2.22
AP11-57	36.58	50.29	13.72	4.16
AP11-57	54.86	62.48	7.62	3.13
AP11-58	10.67	15.24	4.57	1.15
AP11-58	16.76	25.91	9.14	1.65
AP11-58	27.43	57.91	30.48	2.81
AP11-58	64.01	71.63	7.62	2.55
AP11-58	73.15	77.72	4.57	1.18
AP11-59	7.62	39.67	32.00	3.21
AP11-59	42.67	50.29	7.62	3.32
AP11-59	51.82	56.39	4.57	1.34
AP11-62	0.00	9.14	9.14	4.22
AP11-63	22.86	30.48	7.62	3.07
AP11-64	3.05	7.62	4.57	1.37
AP11-64	10.67	15.24	4.57	2.39
AP11-65	21.34	27.43	6.10	1.63
AP11-65	30.48	35.05	4.57	2.62
AP11-66	24.38	41.15	16.76	2.34
AP11-67	1.52	10.67	9.14	1.45
AP11-67	22.86	27.43	4.57	3.16
AP11-67	30.48	35.05	4.57	3.68
AP11-68	70.10	83.82	13.72	1.95
AP11-68	89.92	112.78	22.86	3.54
AP11-68	114.30	118.87	4.57	1.62
AP11-68	135.64	143.26	7.62	2.43
AP11-68	146.30	150.88	4.57	2.06
AP11-69	42.67	56.39	13.72	2.02
AP11-69	67.06	74.68	7.62	2.90
AP11-69	82.30	86.87	4.57	1.32
AP11-69	89.92	94.49	4.57	3.30
AP11-69	117.35	129.54	12.19	2.89
AP11-70	6.10	18.29	12.19	2.89
AP11-70	27.43	32.00	4.57	3.49
AP11-70	38.10	42.67	4.57	2.28
AP11-70	50.29	56.39	6.10	1.39

ole	rom m	To m	nterval m	Grade n
AP11-71	12.19	44.20	32.00	2.68
AP11-71	47.24	54.86	7.62	2.06
AP11-72	3.05	35.05	32.00	2.98
AP11-72	47.24	53.34	6.10	2.12
AP11-73	0.00	6.10	6.10	1.41
AP11-73	9.14	35.05	25.91	2.84
AP11-73	36.58	42.67	6.10	3.63
AP11-73	45.72	50.26	4.57	1.10
AP11-74	35.05	59.44	24.38	2.13
AP11-74	65.53	76.20	10.67	3.84
AP11-74	79.25	88.39	9.14	2.66
AP11-74	102.11	108.22	6.10	1.48
AP11-75	60.96	67.06	6.10	2.39
AP11-75	70.10	88.39	18.29	2.21
AP11-75	92.96	120.40	27.49	3.40
AP11-75	121.92	126.49	4.57	1.51
AP11-75	134.11	143.26	9.14	1.52
AP11-76	102.11	120.40	18.29	1.89
AP11-76	128.20	141.73	13.72	3.47
AP11-76	146.30	153.92	7.62	2.18
AP11-76	155.45	160.02	4.57	1.41
AP11-77	41.15	65.53	24.38	1.89
AP11-77	70.10	86.87	16.76	3.06
AP11-77	89.92	99.06	9.14	3.23
AP11-77	118.87	124.97	6.10	1.56
AP11-78	86.87	94.49	7.62	1.21
AP11-78	96.01	114.30	18.29	1.85
AP11-78	121.92	135.64	13.72	3.29
AP11-78	140.21	149.35	9.14	3.38
AP11-78	152.40	156.97	4.57	1.54
AP11-79	38.10	67.06	28.96	1.73
AP11-79	68.58	80.77	12.19	2.77
AP11-79	83.82	97.54	13.72	3.51
AP11-80	97.54	118.87	21.34	2.36
AP11-80	129.54	143.26	13.72	3.27
AP11-80	152.40	163.07	10.67	2.48
AP11-80	164.59	169.16	4.57	1.20
AP11-81	59.44	70.10	10.67	2.75
AP11-81	71.63	83.82	12.19	1.69
AP11-81	85.34	100.58	15.24	3.54
AP11-81	103.69	112.78	9.14	3.02
AP11-81	126.49	132.59	6.10	1.56

ole	rom m	To m	nterval m	Grade n
AP11-82	85.34	106.68	21.34	1.79
AP11-82	114.30	118.87	4.57	2.67
AP11-82	120.40	129.54	9.14	2.62
AP11-83	35.05	41.15	6.10	2.04
AP11-83	45.72	51.82	6.10	1.69
AP11-83	56.39	65.53	9.14	1.74
AP11-83	70.10	83.82	13.72	3.19
AP11-83	88.39	96.01	7.62	2.53
AP11-84	79.25	92.96	13.72	2.54
AP11-84	96.01	100.58	4.57	3.53
AP11-84	102.11	108.20	6.10	2.57
AP11-84	114.30	121.92	7.62	2.09
AP11-85	6.10	19.81	13.72	2.40
AP11-85	135.64	140.24	4.57	2.08
AP11-86	51.82	68.58	16.76	2.32
AP11-86	83.82	88.39	4.57	3.72
AP11-86	94.49	100.58	6.10	3.80
AP11-87	32.00	67.58	36.58	3.95
AP11-87	71.63	80.77	9.14	1.88
AP11-88	16.76	33.53	16.76	1.92
AP11-88	62.48	68.58	6.10	2.84
AP11-89	3.05	15.24	12.19	1.67
AP11-90	10.67	19.81	9.14	1.48
AP11-91	6.10	12.19	6.10	5.65
AP11-92	32.00	39.62	7.62	3.68
AP11-93	56.39	97.06	10.67	2.52
AP11-93	70.10	79.25	9.14	2.08
AP11-93	82.30	92.96	10.67	4.43
AP11-93	97.54	106.68	9.14	2.21
AP11-93	120.40	126.49	6.10	1.65
AP11-94	12.19	18.29	6.10	1.41
AP11-94	45.72	50.29	4.57	1.65
AP11-95	60.96	82.30	21.34	3.10
AP11-95	100.58	105.16	4.57	2.76
AP11-95	112.78	117.35	4.57	1.47
AP11-97	79.25	106.68	27.43	2.47
AP11-97	108.20	126.49	18.29	1.78
AP11-97	129.54	140.21	10.67	3.16
AP11-97	146.30	150.88	4.57	2.02
AP11-98	65.53	70.10	4.57	1.87
AP11-99	15.24	21.34	6.10	1.84
AP11-99	22.86	35.05	12.19	3.03

ole	rom m	To m	nterval m	Grade n
AP11-99	39.62	47.24	7.62	2.39
AP11-100	41.15	54.86	13.72	1.55
AP11-100	73.15	86.87	13.72	2.64
AP11-100	92.96	102.11	9.14	1.43
AP11-100	109.73	114.30	4.57	1.72
AP11-101	50.29	59.44	9.14	3.33
AP11-101	65.53	73.15	7.62	1.54
AP11-101	77.72	86.87	9.14	1.40
AP11-102	140.21	149.35	9.14	1.72
AP11-102	155.45	161.54	6.10	1.61
AP11-103	1.52	7.62	6.10	2.85
AP11-103	93.96	100.58	7.62	1.82
AP11-103	112.78	120.40	7.62	1.90
AP11-104	12.19	16.76	4.57	1.26
AP11-104	105.16	117.35	12.19	1.98
AP11-104	129.54	143.26	13.72	2.75
AP11-104	144.78	153.92	9.14	3.46
AP11-105	138.68	149.35	10.67	2.05
AP11-105	155.45	163.07	7.62	3.20
AP11-105	164.59	169.16	4.57	3.71
AP11-106	103.63	143.26	39.62	3.03
AP11-107	92.96	109.73	16.76	2.50
AP11-107	137.16	161.54	24.38	2.25
AP11-107	164.59	179.83	15.24	3.21
AP11-107	185.93	190.50	4.57	2.90
AP11-108	0.00	6.10	6.10	2.21
AP11-108	16.76	21.34	4.57	2.44
AP11-108	25.91	39.62	13.72	3.66
AP11-108	44.20	50.29	6.10	2.33
AP11-109	4.57	12.19	7.62	2.55
AP11-109	13.72	27.43	13.72	3.86
AP11-110	0.00	10.67	10.67	2.31
AP11-110	12.19	27.43	15.24	4.61
AP11-111	109.73	120.40	10.67	1.71
AP11-111	121.92	137.16	15.24	1.76
AP11-111	140.21	149.35	9.14	1.77
AP11-112	16.76	27.43	10.67	3.09
AP11-112	35.05	41.15	6.10	3.19
AP11-112	77.72	86.87	9.14	2.65
AP11-W LL	94.01	73.15	9.14	1.90
AP11-W LL	188.98	193.55	4.57	6.07

ote only significant results over 4.5 m intervals reported here, at a 0.9 Mn cut-off.

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

The information provided in this section is summarized from Tribe (2011).

11.1 SAMPLE PREPARATION

11.1.1 200 A E SAMP ES

The 2008 channel sampling was conducted in the MacGregor pit face, with nine vertical channels completed using 3 m sample intervals. The location of the start of each channel was measured using a GPS, and the intervals measured vertically downward relative to this point. The sampling was conducted by suspending a sampler over the pit wall using a climbing rope and bosun chair. The samples were bagged, marked and sent to the ALS Chemex Laboratory (ALS) in Sparks, evada.

11.1.2 200 RE RI I PR RAM

Drillcore from the 2008 program was halved with a rock saw and one-half of the core was sampled for assay analysis and the remaining half retained. For the first several drillholes, all drillcore was sampled in roughly 3 m lengths. After the first several holes, sampling was reduced to include only those portions of the core that, in the judgement of the field geologist, appeared likely to contain manganese mineralization. Sample intervals were typically 3 m, regardless of rock type or apparent mineralization. The samples were bagged, marked and sent to the ALS laboratory in Sparks, evada.

11.1.3 2010 RPR RI I PR RAM

Drill chip samples from the 2010 RPRC program were collected over 1.5 m intervals, and drilling was primarily conducted without the use of water. All chips from drilling were collected at the cyclone and stored in bags at the site. During the sample-collection process, a rotary sample splitter was used to collect a 2 kg to 4 kg sample for assay analysis. A subsample of chips produced over each interval were split at the cyclone using a Jones or a Gilson splitter, and a 2 kg split was bagged, marked and set aside for logging and delivery to the assay laboratory. Logging samples were taken from this sample and washed and placed into muffin tins, which were used as temporary chip trays for logging. At the drill site, the rejects were bagged in their entirety and transported to a local storage facility on the Property. These samples are available for check assaying or metallurgical work, as required. The samples were delivered to the ALS laboratory in Sparks, evada.

11.1. 2011 RPR RI I PR RAM

Drill chip samples from the 2011 RPRC program were collected over 1.5 m intervals, and drilling was conducted using water. The total volume of chips produced was collected and approximately 20 of the material over each interval (2 to 4 kg of material) was split out for an assay sample. All chips were collected at the cyclone using a rotary sample splitter and stored in marked bags at the drill site. All samples were then left to dry in the sun prior to transportation in order to minimize shipping weights. Assay samples showing any perceptible manganese mineralization were sent to the laboratory for analysis, and the samples not containing obvious mineralization were stored on the Property. Chip samples were collected for logging every 1.5 m using muffin tins as temporary chip trays and left to dry. The rejected material was also dried and then transported to a local storage facility on the Property, and are available for analysis as required.

11.2 SAMPLE ANALYSIS

11.2.1 200 A E A RI RE SAMP ES

The samples were analyzed at ALS laboratories in both Sparks, evada and orth Vancouver, British Columbia. At the Sparks ALS lab, the samples were crushed, pulverized and split with a 250 g sample reserved for analysis in the orth Vancouver lab. The analyses were done using a four acid, near-complete digestion and inductively coupled plasma mass spectometry (ICP-MS) analysis (package number M -MS61). All samples exceeding a level of 1 concentration for any metal, and were thus outside the analytical range for this method, were re-analyzed using ICP-MS package OG62 with an atomic absorption (AA) finish.

11.2.2 2010 RPR IP SAMP ES

The samples were analyzed at the ALS laboratories in both Sparks, evada and in orth Vancouver, British Columbia. At the Sparks ALS lab, the samples were dried, crushed and pulverized to 70 -200 mesh. The samples were prepared for assay using ALS prep 31 methods and a small sample (10-15 g) sent to orth Vancouver for analysis. The analyses were done using a four acid near complete digestion and ICP-MS analysis (M -MS61). All samples exceeding a level of 1 concentration for any metal, and were thus outside the analytical range for this method, were re-analyzed using ICP-MS package OG62 with an AA finish. very tenth sample was taken as a check sample and sent to American Assay Laboratory in Sparks.

11.2.3 2011 RPR IP SAMP ES

The 2011 sample analysis procedures were the same as followed in the 2010 program, with the exception of the location of the laboratory. Samples in 2011 were delivered to the ALS laboratory in Reno, evada, where they were dried, crushed and pulverized. A small sample of this material was then shipped to the orth Vancouver laboratory for analysis. The analytical procedures in orth Vancouver were the same as described in Section 11.2.2. very tenth sample was taken as a check sample and sent to American Assay Laboratory in Sparks.

11.2. 2011 E SIT SAMP ES

In late 2011, 15 density samples were collected on the Property. The samples chosen were considered representative of the deposit area. Once sampled, the sample location coordinates were recorded, they were briefly described geologically, and then bagged and labeled. The samples were transported by Mr. orm Tribe (a Qualified Person who is independent of AMI) to Kelowna, British Columbia where he also conducted the density measurements.

The density measurements were determined using a water-displacement method where they were weighed dry and then submerged in a pre-determined volume of water and weighed again. The scale used for these analyses has an accuracy of 10 g.

11.3 SAMPLE SECURITY

11.3.1 200 A E A RE RI I PR RAMS

AMI personnel transported and delivered the core and channel samples in a pickup truck to a core storage facility in Wikieup. The facility was a covered and fenced compound. An AMI geologist logged the core at this facility. Once assay samples were prepared, the samples were transported by AMI personnel to Kingman, where they were couriered to the ALS laboratory in Sparks.

11.3.2 2010 A 2011 RI I PR RAMS

AMI personnel transported the assay samples from the 2010 and 2011 programs in a private vehicle to a courier in Kingman. The samples were then shipped by courier to the ALS sample preparation laboratory in Sparks (2010 program) or Reno (2011 program). For the duration of drilling in both 2010 and 2011, AMI personnel camped at the access road on the Property, and access to the Property was monitored.

12.0 DATA ERIFICATION

12.1 SITE ISIT

Ms. Margaret Harder, P.Geo., visited the Property on June 2, 2011. During this visit, the manganese-bearing Chapin Wash Formation was examined in numerous locations, as well as the open pit workings from several historical mines. Drilling of the 2011 RPRC program was ongoing at the time of the visit, and the rig was observed during operation. Figure 12.1 shows a photograph taken on the Property during this site visit.

i ure 1 .1 Photo raph of the Property ho in an anese inerali ation dar material in fore round

12.2 DRILLING AND ASSAY ALIDATION

The drillhole collar coordinates, as made available to Tetra Tech, required evaluation and adjustment to match the current topographic surface. Only holes that were within the area included in the resource estimate (Section 14) were fully re-evaluated, and collar locations lying outside the estimated area are not considered reliable without further evaluation. The position of drillholes was reconciled with maps and topographic data.

12.2.1 T I E RI E

One diamond drillcore hole from the 2008 program (ADH 4) was twinned by one RPRC hole from the 2010 program (AP10-10), with the goal of providing a comparison of the core and RPRC drilling methods and to evaluate the reproducibility of the results by different drilling and sampling methods. These two holes each intersected approximately 30 m of manganese-bearing sandstone.

A comparison of the data from the twinned holes is ambiguous and the results are not considered to be a statistically meaningful analysis of the two drilling and sampling methods. The 2011 drilling program also used different drilling methods from either of the 2008 or 2010 programs, and the results from this program have not been compared against the others. The results of the 2008 and 2010 twinned holes are presented in Table 12.1.

Taken directly from the average assay value in each of the holes, the RPRC drilling returned a slightly lower manganese value than the diamond drilling (7 lower overall) through the mineralized zone. However, the variation between the same depth intervals in each hole is often considerable, with values typically differing by several percent manganese. Given that the holes are several metres apart and that only one hole was twinned, this variation is not considered a statistically representative comparison of the two drilling programs. Further drilling (twinning of previously completed holes) is warranted to evaluate the recovery of mineralized material and corresponding grades from each program. The assay results of the 2008, 2010, and 2011 are considered reliable (see Section 12.2.2), however different drilling methods may produce different recovery rates of mineralized material, and in some cases a correction factor may be applied to different drilling methods for more accurate comparison of separate drilling programs. These correction factors do not typically result in meaningful differences to overall grade results, but may help refine resource estimation procedures in the future.

Table 1 .1 Comparison of 00 Drillcore and 010 P C Assay esults

00 Diamond Drillhole				010 P C Drillhole			
ole D	rom m	To m	n	ole D	rom m	To m	n
ADH4	0	3	1.04	AP10-10	0	3	0.91
ADH4	3	6	0.45	AP10-10	3	6	4.53
ADH4	6	9	0.45	AP10-10	6	9	2.96
ADH4	9	12	1.45	AP10-10	9	12	2.86
ADH4	12	15	4.78	AP10-10	12	15	6.27
ADH4	15	18	3.32	AP10-10	15	18	3.87
ADH4	18	21	4.05	AP10-10	18	21	1.82
ADH4	21	24	5.83	AP10-10	21	24	0.59
ADH4	24	27	4.25	AP10-10	24	27	0.41
ADH4	27	30	0.94	AP10-10	27	30	0.55

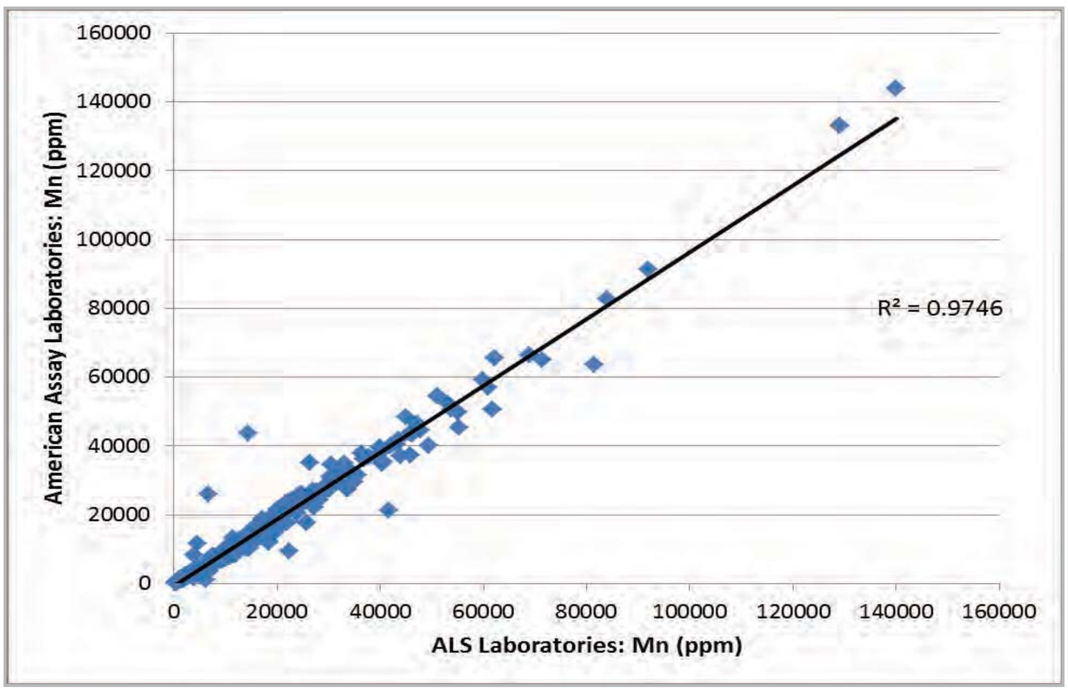
12.2.2 ASSA A I ATI

CHECK SAMPLES

Standard internal laboratory quality assurance quality control (QA QC) samples (duplicates and control samples) were analyzed during the sample stream by ALS and provided an internal check of the reproducibility of the analyses following sample preparation. For the 2010 and 2011 programs, 250 g of pulverized reject was set aside from every tenth sample and sent to American Assay Laboratories for independent external assay checks. Although this does not provide a check of the laboratory preparation or internal contamination during sample preparation, this does allow for comparison of results from different analytical equipment. In all, 524 samples were sent for check assaying, which represents approximately 12 of the total samples for these two drilling programs.

Tetra Tech evaluated the results of these check samples, and the results are illustrated in Figure 12.2. There is good correlation between the results from both labs, providing confidence that the analytical analyses of samples from the primary lab, ALS, are reliable.

i ure 1 . Comparison of Chec Assay amples



Results from internal laboratory QA QC samples have not been evaluated by Tetra Tech. However, this comparison of check samples by an external lab is considered to be a more meaningful and reliable method of evaluating the analytical results of the primary lab used.

ASSAY VERIFICATION

Assay results for drillholes completed on the Property during the 2008, 2010, and 2011 drilling programs were verified by Tetra Tech. Original assay certificates from the external lab (ALS Minerals) used for these analyses were compared against the database as provided by AMI (see Section 10). Tetra Tech verified assay results for approximately 10 of these drillholes, including ADH4, ADH16, AP10-2, AP10-8, AP10-14, AP10-20, AP10-29, AP11-57, AP11-63, AP11-69, AP11-79, AP11-87, AP11-94, AP11-102, AP11-111. The error incidence of all assay values was less than 3 and the errors that were noted are minor discrepancies which do not impact the overall grades or grade distribution in the current resource estimation. A record of any database discrepancies noted will be provided to AMI for correction.

Moving forward into a feasibility study, Tetra Tech recommends that a single, comprehensive database is adopted for all future drilling programs and that all data (including that of previously completed holes) is validated against lab assay certificates.

12.3 A C PROCEDURES

Standard QA QC samples (blanks, standards and field duplicates) were not submitted in the sample stream. However, check samples were routinely (every 10th sample) sent to an independent external laboratory (Section 12.2.2), providing confidence in the analytical procedures and results from ALS labs.

12.4 P OPINION

Tetra Tech considers the data used in this resource estimation to be reliable for the purpose of this study. The stratiform nature of the mineralization and the overall continuity of the mineralized zones mitigates the risk of concerns regarding less than ideal database management and QA QC procedures.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 SUMMARY

In 2009, Kemetco was commissioned by AMI to investigate sulphur dioxide leaching of manganese mineralization from the Artillery Peak deposits located at the Artillery Peak Region, Mohave County, Arizona. It was recognized that these low-grade resources were soft and not suitable for heap leaching or flotation, as the amorphous hydroxide rich pyrolusite derivatives (four-valent Mn) wad minerals consisted of thinly coated clayey particles, and the agglomerates were further cemented within a calcite matrix.

Key aspects of historical accounts are summarized in Section 13.2.

The basis for Kemetco's research resulted from

- detailed literature reviews and thermodynamic calculations aimed at technological breakthroughs
- bench-scale testing to develop an innovative conceptual process
- confirmatory variability studies and integration of various unit operations
- proof of concept for the recovery of MM from representative Cobwebb Hill (CWH) samples.

Continuous campaigning was preceded by batch tests to confirm that leaching of coarsely crushed feed at high densities was indeed viable. These conditions also verified where key improvements in the setup were needed to ensure trouble-free operations. Limited variations established sensitivity of leaching and settling to reagent types and dosages. Testing of sequential products in downstream treatment modules produced acceptable manganese carbonate, sodium sulphate and MM grades, and recycle streams and equipment options.

13.2 HISTORICAL DATA RE IE

A critical review of literature relevant to the project was key in developing the Kemetco concepts; and an abbreviated summary of historical documents is shown in Table 13.1. Attention focused on front-end leaching, dewatering and purification modules, primarily pertaining to the Artillery Peak deposits. The 1934 USBM metallurgical review (IC 6770), and the 1945 USGS study on geology and economic potentials of these particular resources (Bull.961), are comprehensive starting points. The objective was to establish the consistency of current findings in the light of selected historical data.

Metallurgical data from USBM test programs were relevant to efforts to bring domestic manganese resources into competition with higher-grade imports that had supplied the ferro-alloy and steel industries since the 1920s ((USBM RI 9399 (1994)). Many tests conducted on Artillery Peak materials typified fast-leaching wad resources, tending to be soft and clayey with poor responses to heap leaching or flotation. Direct sulphur dioxide leaching in atmospheric reactors compared favourably to the reductive roasting prior to acid leaching method that was used in the earliest MM plants in Knoxville, T , and Boulder City, evada. The M.A. Hanna Company commissioned the Pan American ngineering Co. to operate a 5 t d pilot plant in Berkeley, California, which resulted in the design of a 1,000 t d operation for the Manganese Ore Co. in Henderson, evada. Mr. M.C. Lake of Hanna Mining was directing the Las Vegas plant in conjunction with industrial figures and consultants, including the Hazen brothers (Kemetco 2009a).

For MM production (during the 1940s), reductively roasted Three Kids calcine in the Boulder City pilot plant (USBM Bull. 463 (1946)) was leached with anolyte at 3 to 5 pulp density and a pH 4, and neutralized with ammonia. The 3,000-gallon batches took over three hours of processing, and the leach pulp was settled in six 25 ft diameter thickeners for washing by counter-current decantation. Underflow pulp densities of up to 27 solids were achieved without any flocculant additions. As summarized in several reports (USBM RI 4077 (1947), RI 9150 (1988), IC 6770 (1934), RI 4163 (1947)), the ore from Three Kids, evada, was very similar to Artillery Peak materials; it was, however, of higher grade and more granular, and thus more amenable to flotation (Kemetco 2009a).

Operations at Berkeley and Las Vegas were refocused on Three Kids ores with production of oxides for ferro-manganese smelting during WWII. Pilot leaching using aqueous sulphur dioxide in agitated tanks with lead covers took more than four hours for feed ground finer than 65 mesh. Target manganese pregnant leach solution (PLS) levels of 130 g L were achieved at pulp densities of 20 solids with feed grades of 20 to 30 Mn. Leaching at pH 2 was found beneficial for lowering entrained sulphur dioxide and settling of the leach pulp, with dextrin or guar gum used as the preferred flocculants.

Table 1 .1 ist of istorical Publications Discussin Artillery Pea aterial Characteristics

Date	irst Author	Title	Description	ef. o.
1856	J. Marcou	RR from Mississippi to Pacific, vol.3, pt.4	Surveys for US War Department near 35th parallel	USGS Bull. 961, p.6
1908	W.T. Lee	Geologic Reconnaissance, part of western A	Primarily a study of water resources	USGS Bull. 352
1919	.S. Leaver	Apparatus for extracting metals from their ores	vertical compartmented drum design	USP 1,312,488
1920	C. . van Barneveld	Manganese Uses, Prep., Mining Costs Ferro alloys - Chapter	Leaching of manganese ores with SO_2	USBM Bull. 173
1920	.L. Jones, Jr. et al.	Deposits of manganese ore in A	Incl. "Artillery Mountains Williams River region"	USGS Bulll. 710-D
1930	C.W. Davis	Dissolution of various manganese minerals		USBM RI 3024
1934	R.S. Dean et al.	Manganese, its occurrence, milling metallurgy - Ch.6	pt.1 incl. "Kingman district, A "	USBM IC 6768
1934	.S. Leaver	Ibid, Ch.7 - Hydrometallurgy of manganese	pt.3 incl. grade, tonnage mineralogy	USBM IC 6770
1936	D.F. Hewett et al.	Mineral Resources of region around Boulder Dam	incl. Artillery Mountains resource estimates	USGS Bull. 871
1938	S.M. Shelton et al.	lectrolytic Manganese	incl. leach MM tests at plant in Boulder City, V	USBM RI 3406
1945	S.G. Lasky et al.	Mn resources of the Artillery Mountains region, Mohave, A	Detailed geology mineralogy resource estimates	USGS Bull. 961
1946	J.H. Jacobs et al.	Operation of lectrolytic Manganese pilot plant	pt.1-3 accounts for Boulder City, V	USBM Bull. 463
1947	W.F. Wyman et al.	SO_2 leaching tests on various western manganese ores		USBM RI 4077
1947	J.H. Jacobs	lectrowinning of manganese from domestic ores		USBM RI 4163
1948	D. . Vedensky et al.	SO_2 process for treatment of manganese ore	pt.1-3 accounts for Three Kids Plant, Las Vegas, V	PanAm. ng. Co.
1948	R.S. Sanford et al.	Artillery Peak manganese deposits, Mohave County, A	mining of Chapin Wash materials	USBM RI 4275
1949	S.F. Ravitz et al.	Semi-pilot plant tests on treatment of manganese-silver ores by the Dithionate Process		USBM TP 723
1952	A. . Back et al.	Formation of Dithionate and Sulphate in the oxidation of SO_2 by MnO_2 and air		USBM RI 4931
1957		Mining of Maggie Canyon materials		USBM RI 5292
1957	F. . Bender	Percolation Leaching of manganese ores with SO_2	incl. response of Arizona materials	USBM RI 5323
1957	J.B. Rosenbaum et al.	Pilot plant flotation of manganese ore from the Maggie Canyon deposit, Artillery Mountains region, A		USBM RI 5330
1959	C. Rampacek et al.	Operation of a Dithionate Process pilot plant for leaching manganese ore from Maggie Canyon deposit		USBM RI 5508
1962		Sintering and smelting manganese concentrates from Maggie Canyon ore, Artillery Mountains area, A		USBM RI 5939
1963	P.V. Fillo	Manganese mining milling methods and costs, Mohave,		USBM IC 8144

Date	irst Author	Title	Description	ef. o.
		Mining Milling Co, Mariposa County, A		
1964	D.A. lkins	stimated cost of exploiting hard manganese ore from the Maggie Canyon Deposit, Artillery Mountains region, A		USBM RI 6438
1968	J.J. Henn et al.	Review of major proposed processes for recovering manganese from US resources, pt.3 SO_2 Processes		USBM IC 8368
1981	S. . Khalafalla et al.	Selective extraction of metals from Pacific Sea nodules with dissolved sulphur dioxide		USBM RI 8518
1982	G.M. Potter et al.	Technological invesigation to determine the feasibility of in situ leaching metallic ores other than Cu and U		Proc. SM -AIM
1982	C.C. Kilgore et al.	Manganeses availability - domestic	minerals availability system appraisal	USBM IC 8889
1977-83	D. . Goens et al.	Investigations on the production of MM for Mr. James L. Lake		HRI PR 4267, 5517
1983	J.L. Lake	Feasibility study - Production of electrolytic manganese from Artillery Peak ore		Hazen Research
1987	J. . Pahlman et al.	Dual leaching method for recovering silver and Mn from domestic manganiferous silver deposits		USBM RI 9126
1988	J. . Pahlman et al.	Leaching of domestic manganese ore with dissolved sulphur dioxide		USBM RI 9150
1994	T.S. Jones	Manganese Material Flow Patterns	Domestic Data 1900-1990	USBM RI 9399
Sep-08	J.L. Lake	Proposal program for treatment of Artillery Peak, A - manganese ores for the Rocher deBoule Minerals Company		Kemetco 2009a HRI 2008
ov-08	R. Bhappu	Recovery of manganese from the Artillery Mountain resource, ArIzona		Kemetco 2009a MSRDI 2008
Dec-08	K.G. Tan	Artillery Peak manganese Phase-1 and Phase-2 Test Reports		Kemetco 2009a PRA - 2008
Feb-09	. Tribe	Mineral resource evaluation report on Artillery Mountain manganese property, Mohave County, A		. Tribe 2009a
Aug-09	. Tribe	Preliminary conomic Assessment for the Artillery Mountain manganese property Mohave County, A		. Tribe 2009b
Jul-10	. Chow et al.	The recovery of manganese from low grade resources bench scale metallurgical test program completed		Kemetco 2010b IRAP PR 712681

The 1,000 t d plant design was based on leaching in packed towers, which was judged less effective than agitation in tanks due to plugging issues with gypsum. Most of the sulphur dioxide was recycled from reductive roasting of manganese sulphate; makeup sulphur dioxide from sulphur burners at preferred sulphur dioxide oxygen (SO_2 O_2) ratios had difficulty overcoming severe fluctuations in the roaster gas output. The counter-current leaching in towers was largely self-regulated, with a limited degree of pulp recycling from inter-stage agitators that aided acid generation. The heat generated in operation at less than half plant capacity suggested that cooling below 77°C would be needed in the summer for protection of rubber liners and to reduce off-gas emissions to the stack. Primary thickening of aerated and cooled pulp (≤43°C) allowed adequate washing in seven 25 ft diameter CCD thickeners with underflow densities of up to 50 solids (P_{80} 200 mesh). Aeration stripped out residual sulphur dioxide and generated acidity (slow at pH ≥3), with elimination of some thiosalts. eutralization with raw ore or freshly produced manganese oxide (MnO) was preferred over the use of ammonia or lime, and precipitated iron cake was recycled to the CCD circuit to reduce soluble losses. Manganese recovery (78 to 94) relied on evaporating-neutralized PLS to produce manganese sulphate and manganese dithionate hydrates that were then roasted to produce manganese oxide with recovery of sulphur dioxide. The evaporation step is energy-intensive and highly corrosive, while the vapour phase pH had to be controlled with sodium hydroxide additions. However, calcining immobilizes key impurities, so the selective redissolution of anhydrous manganese sulphate in spent electrolyte produced very clean feed for the production of MM. Based on this process, overall sulphur consumption estimates were as low as 1 kg S per kilogram of manganese leached, with recovery of excess acid and or ammonium sulphate from electrolysis.

In the 1980s the USBM findings were confirmed and expanded upon by Mr. James L. Lake and Hazen Research, to the extent that the USBM projections for operating costs of 80 per long dry ton (LDT) of contained manganese (in the form of oxide nodules), was converted into the following estimates for Artillery Peak materials (Kemetco 2009a)

- Main Product..................... 20 t d MM
- Throughput........................ 305 t d of 7.0 Mn grade
- 1983 CAP 4,607,000
- 1983 OP .... 0.198 lb MM (priced at 0.33 lb).

The historical examples reveal that beneficiation of Artillery Peak materials is indeed promising if lower-strength PLS can be targeted. These and other findings also confirm

- abundant and massive formations in the arid Arizona climate favour open pit operations which are much cheaper than underground mining

- soft, self-slaking wad minerals in soils, sandstones or agglomerates incur very low comminution costs
- leaching with sulphur dioxide is exothermic and very fast, minimizing energy costs, tank sizes and retention times, even at coarser feed crush sizes
- the main operational challenges relate to solid-liquid separations, dithionate control and downstream recovery options, including manganese electrowinning.

It was therefore recognized that energy inputs, reagent management (including water) and environmental considerations would dominate the process economics. Mr. James L. Lake's perceptive insights are very helpful in guiding a rational development procedure that has led to the technological breakthroughs discussed in the next section.

13.3 EMETCO PROCESS DE ELOPMENT

Kemetco s 2009-2010 efforts reviewed the treatment strategies formulated by Mr. James L. Lake, and the USBM. These findings are of particular relevance to the beneficiation of Artillery Peak resources, and Mr. James L. Lake's files highlight the most critical issues that have prevented commercialization of these domestic materials to date. Selected project data were presented in a document for a grant application from the ational Research Council of Canada (RC) through its Industrial Research Application Program (IRAP) (Kemetco 2009a) and reviewed in a I 43-101 compliant Preliminary conomic Assessment (. Tribe 2009b).

Kemetco's concept is based on reducing processing costs by avoiding fine grinding and minimizing energy, reagent and water usages. Based on literature data analyses and thermodynamic calculations, several process improvement options were considered. These included autoclaving, multi-stage leaching, solvent extraction as well as bio-leaching routes, which were briefly evaluated. With feed grades around 10 Mn, multi-stage leaching would be needed to achieve a solution of greater than or equal to 40 g L Mn suitable for further processing. Settling rates of leached pulps, however, proved rather marginal, so extra solid-liquid separation steps were avoided. Dithionate in solution interferes with manganese recovery unless clean conversions as alkali species could be achieved. The need for concentrated PLS levels and corrosive control of dithionate can be overcome by producing manganese carbonate and using nanofiltration and chilling to recover neutral hydrates of sodium sulphate and sodium dithionate. Manganese carbonate is very suitable for the production of MM.

Kemetco's selected approach eliminates evaporative crystallization of manganese sulphate, as practiced by USBM, to reduce process energy requirements by one order of magnitude (Kemetco 2010b). Corrosive problems are eliminated, and a high-grade sodium sulphate byproduct is obtained after complete precipitation of the manganese. Water and sulphur dioxide can be recycled, and dithionate-free manganese carbonate is very amenable for dissolution for subsequent processing to MM. Manganese-free carbonated liquor will feed a conventional crystallization circuit where much of the water will be rejected as a clean F permeate. The brine product can be chilled to yield hydrated salt crystals to be harvested in a chilled centrifuge. This sodium sulphate production route has already found widespread industrial applications worldwide, but the effects of dithionate and impurity deportments needed further study. Calcining of the dithionate-containing solids at 400°C can recover additional water and sulphur dioxide and yield anhydrous sodium sulphate for sale. Multi-effect evaporation and crystallization methods were evaluated in combination with solubility studies and F.

Manganese carbonate readily dissolves in acidic, spent MM anolyte, removing the need for recovery of excess acid from electrowinning, as occurs if manganese sulphate is used (Kemetco 2010b).

Follow-up studies suggested that liquid-solid separations improved by leaching of coarse feed and providing gypsum seed crystals to the leach (Lehne Associates 2011, Kemetco 2011c . The survival rate of pebbles coarser than " dropped, even for the lower-grade CWH feed, when leached using calcium bisulphite with heating to 50°C to compensate for the lower exothermicity. The leached pulps settled to 45 solids in static settling tests with non-ionic flocculants (Kemetco 2011d). Hardness testing and mineralogical examinations were conducted, in support of pilot design and preparation work

13.3.1 SAMP E ARA TERISTI S

Bulk samples, collected from two locations across the face of CWH in the upstream part of the Chapin Wash, arrived at the start of 2011 in 25 steel drums, weighing about 7 t in total (Kemetco 2011). Three randomly selected drums were prepared for the test program; their contents were very consistent in grade and particle size assays. Typical blends of materials requiring size reduction contained approximately 50 mudstones and rocks up to 10" in size. Materials were crushed to a top size of ". Assays are compared in Table 13.2, showing that the three CWH drums selected for batch testing were homogeneous in makeup. Random samples of MacGregor pit (MGP) material that were also tested showed higher grades of manganese, but otherwise very similar feldspar-dominated compositions.

Test charges were subjected to dry milling, yielding pulps finer than 4 mm after very short grinding times. The Bond rod mill index of 6 kWh t confirmed the soft nature of the material, with a negligible abrasion index (0.09 g) on the more competent specimens that were used for hardness testing (Kemetco 2011c; Kemetco 2011d; Hazen 2011). Crushing tests were conducted with different types of equipment to verify that the materials were soft, non-abrasive and free-flowing (Kemetco 2012b).

Table 1 . elevant eed Assays of aterials Tested for Proof of Concept by

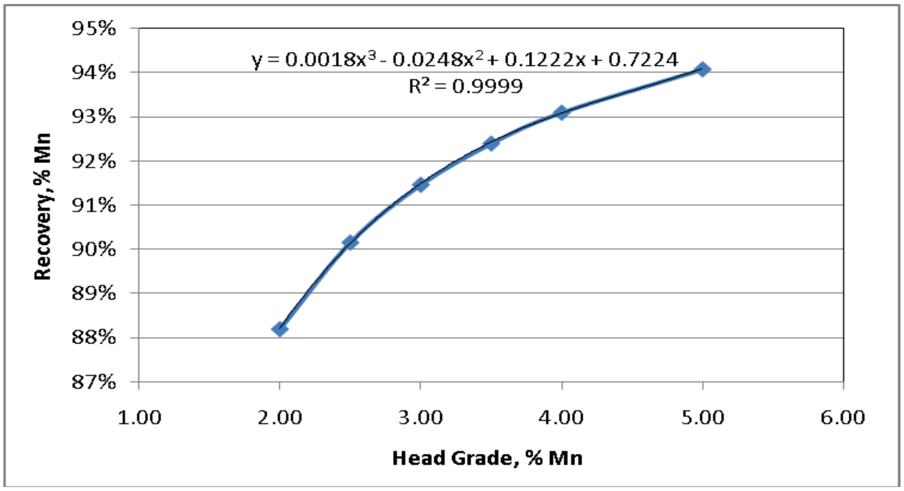
ample ame	iO	Al O	e O	CaO	O	a O	O	TIO	nO	P O_5	aO	O	Total
CWH Drum A	54.68	13.57	5.34	1.72	1.90	1.02	5.63	0.57	4.50	0.16	0.56	8.47	9 . 6
CWH Drum B	55.13	13.79	5.46	1.14	1.91	0.93	6.01	0.59	4.60	0.16	0.50	8.25	9 .6
CHW Drum C	55.65	12.07	4.34	3.03	1.76	1.36	4.38	0.48	4.07	0.14	0.98	10.00	9 .55
MGP Met 4	58.54	13.28	4.51	1.43	1.64	1.75	5.18	0.55	4.78	0.16	0.65	5.63	9 .
MGP Met 2	54.43	11.87	3.81	2.89	1.42	1.42	5.09	0.45	7.38	0.13	0.90	7.99	9 . 6
MGP Met 1	56.35	10.90	3.14	2.71	1.16	1.69	4.57	0.39	8.83	0.14	1.32	6.95	9 .

Unconfined Compressive Strength data (MSRDI 2008, Tetra Tech 2011a) were generally well below 5,500 psi on average. Mineralogical results on several material types from different zones in the deposit are also available (USGS Bull.961 (1945); Kemetco 2012b; Ammtec 2011; Lehne 2011a; Lehne 2011b) that show the presence of clay (montmorillonite) in varying amounts. All in all, a very consistent picture emerges that explains why the materials are less amenable to flotation but uniform in makeup, very soft in nature, and easy to leach with sulphur dioxide with concomitant removal of the calcite.

Similar in character to the medium grade MGP material (10.5 Mn) that was tested previously (Kemetco 2010b-d; Kemetco 2011a-c), the Chapin Wash mineralogy can be described as friable arkosic sandstone (USGS Bull.961 (1945);Lehne 2011a; Lehne 2011b) that consists of non-weathered feldspathic clay and silt particles cemented together by cryptomelane (wad manganese) and further consolidated by more recent calcite deposits. on-weathered material is self-slaking and tends to disintegrate upon leaching. Minor glassy volcanics or opalized specimens seem harder and more resistant to leaching. Only the presence of manganese, as indicated by darker colorations, distinguishes the resource from barren ferruginous materials that were less affected by hydrothermal intrusions.

In summary, testing since the 1930s has confirmed the remarkable uniformity of Artillery Peak materials that respond very well to aqueous sulphur dioxide leaching. Based on the Kemetco tests on Chapin Wash materials ranging from MGP grading 10.5 Mn down to CWH of 3.5 grade, a conservative depletion to 0.2 Mn in the leach residues is readily achieved. This forms a reliable basis for grade-sensitivity estimates as illustrated in Figure 13.1 where extractions 85 Mn can be expected for materials grading as low as 2 Mn under baseline non-optimized conditions.

i ure 1 .1 tractions at i ed esidue Grade of 0. n

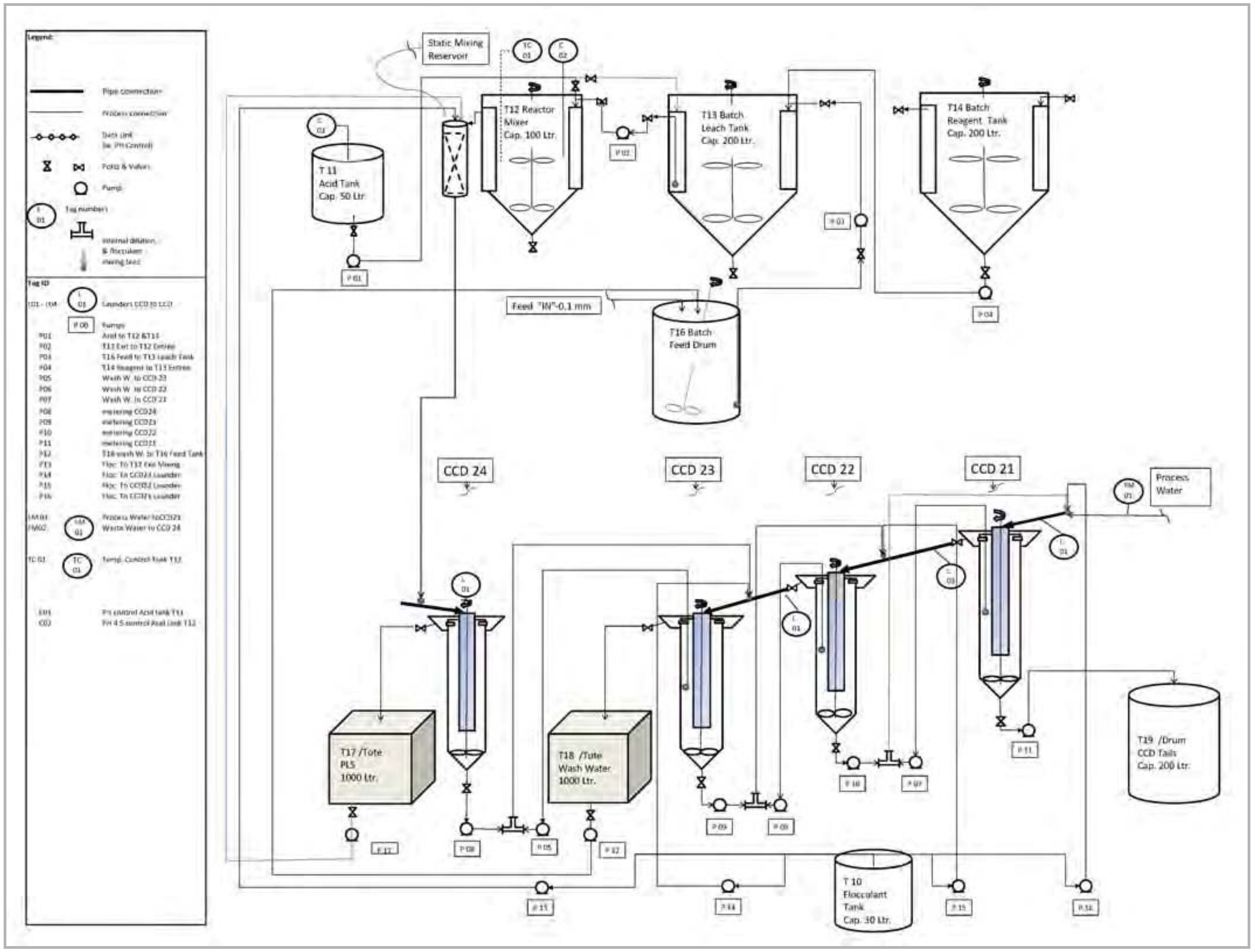


13.3.2 *PI T P A T ESI A TEST PR RAM*

Key aspects determining process viability lie primarily in the leach and solid-liquid separation modules, as downstream processing of the leached manganese is quite conventional and widely practised elsewhere. A conservative and dynamic mode of operation was selected to simplify setup and mitigate operational issues. The convenience of handling sodium metabisulphite in preference to using gaseous sulphur dioxide on a laboratory scale has long been appreciated. missions of sulphur dioxide gas were avoided by operating at relatively mild temperatures and slightly higher pH regimes. Very aggressive targets for levels of agitation, feed type (grade and particle size), pulp density and overall retention times were set in order to prove that bench-scale results can be matched through extensive steady-state dynamic processing. Piloting equipment was designed with flexibility and reliability in mind, in order to minimize the chances for accidents, breakdowns or failure. Once leaching and washing in an abbreviated train of four high-density, deep-cone thickeners was conducted, the products were tested batch-wise, in various downstream modules.

Interchangeable polypropylene leach tanks of 200 L, 100 L and 50 L nominal volumes were fitted with stainless steel agitators with double, opposing propellers, capable of operating on coarse crushed feed at high rpm and elevated pulp densities. Pumping by means of peristaltic units with only minimal process control was provided. Modest heating by means of hot water additions or immersion heating was installed, and acid additions were either manual or with a pH controller to the main leach tank. o provisions for sulphur dioxide emission controls were needed, and manual blending of the feed was conducted periodically in batches of up to 200 L. The nominal volume of each polypropylene thickener unit was 100 L. Picket rakes were installed, as well as adjustable feedwells, launders and appropriately located sight windows. A layout is schematically shown in Figure 13.2 for the continuous portion of the pilot campaign.

Figure 13.2 Schematic Piping and Instrumentation Drawing for Continuous Piloting



13.3.3 ARSE-PARTI E EA I

Piloting started batchwise on August 18, 2011, with a representative CWH composite sieved and crushed to 5 8" nominal size. Given the lower feed grade, pre-leaching at 50 solids with sub-stoichiometric levels of reagents was conceived as a mechanical and chemical sorting device to screen out either barren or refractory pebbles (Kemetco 2010). Batch test BPL1A at first showed only limited mechanical attrition above pH 6, with 10 of feed mass surviving as 16-mesh pebbles. Higher acid additions in BPL1B, still with sub-stoichiometric bisulphite levels, generated fewer and better-leached pebbles that were difficult to separate from the viscous fines without excessive washing. Similar results were obtained with 10-mesh feed in test BPL2, where grit accumulated in the reactor cone, particularly after further dilution to feed the ensuing dynamic tests SL1 and SL2 (Table 13.3). An increased reagent dosage in BPL3 yielded 90 Mn-extraction from " feed in a single-stage leach at 30 solids, 50°C, pH 3 and 60 minutes of retention. Based on the reagents used for treating 1,200 kg of feed during the entire campaign, an overall consumption of 1.42 kg of sulphur per kilogram of leached manganese was obtained for dynamic testing with bisulphites. Some sulphur dioxide was lost in the modest emissions that were occasionally detected; these losses would be captured in the field by scrubbing and decomposition of sodium dithionate. Most of the acid requirements for leaching are expected to be covered by the oxidation of sulphur dioxide with air and will be drastically reduced when operating with the gaseous reagent. For now, the sulphur dioxide requirements should be derived from extrapolated results and historical accounts, subject to verification for an optimized closed-loop process in the next phase of study.

Table 1 . elevant hole oc Assays of Coarser nitial Pilot each esidues, by

eed Top i e	ample D	Test Date	GT Type	ashed each esidue Assays,						60-min n- .	Dosa e, / n	
				Al O	CaO	nO	a O	iO			O	O
16-mesh	BPL1-T13	18 8	CaBS	13.0	6.20	2.04	0.97	54.14	2.69	51.6	1.1	1.35
10-mesh	BPL2-T14	22 8	CaBS	12.7	8.86	0.37	0.69	50.09	4.32	90.5	2.0	1.05
16-mesh	SL1-T24	23 8	CaBS	14.3	5.47	0.56	0.81	54.71	2.54	86.9	n.a.	n.a.
16-mesh	SL2-C21	25 8	MBS	13.2	2.41	1.11	1.21	64.38	0.46	77.9	n.a.	n.a.
1 4"	BPL3-T12	26 8	Ca SBS	10.6	12.5	0.36	0.83	46.03	6.48	90.0	3.5	0.95
-5 mm	*CWH Head*	Avg.	SO_2 gas	*13.9*	*2.02*	*4.35*	*1.14*	*55.88*	*0.11*	SiO_2-tie	2.2	1.12

otes measured acid consumptions were halved to correct for partly neutralized bisulphate reagents that will be replaced by gaseous SO_2

Based on atomic weights, total S consumption will equal that of SO_2 plus ⅓ that of H_2SO_4 or approximately 1.6 kg per kg of Mn leached.

It should be noted that partially neutralized bisulphites require extra acid to dissolve manganese, as compared to the use of gaseous sulphur dioxide only. Additional reagent is also consumed by the formation of dithionate and sulphuric acid in side reactions. Gypsum can be formed in the removal of calcite blocking access to the manganese, as well as by the use of calcium sulphite from off-gas scrubbing. A host of other reactions (such as dissolution of sulphur dioxide in water to form bisulphite ion), the production of thiosulphate or thiosalts, and insolubles such as manganese sulphite can best be ignored to preserve a simplified overview. The main reactions illustrating this complex leaching behaviour are shown below

- SO_2 MnO_2 - $MnSO_4$ ....(eq.13.1)
- 2 SO_2 MnO_2 - MnS_2O_6 ....(eq.13.2)
- SO_2 O_2 H_2O - H_2SO_4....................(eq.13.3)
- H_2SO_4 $CaCO_3$ H_2O - $CaSO_4.2H_2O$ CO_2....................(eq.13.4)
- $Ca(HSO_3)_2$ 2 MnO_2 H_2SO_4 - 2 $MnSO_4$ $CaSO_4.2H_2O$ ....(eq.13.5)
- $Ca(HSO_3)_2$ MnO_2 H_2SO_4 - MnS_2O_6 $CaSO_4.2H_2O$....................(eq.13.6)
- $Ca(HSO_3)_2$ O_2 H_2O - $CaSO_4.2H_2O$ H_2SO_4 ....(eq.13.7)

The production of manganese carbonate as an intermediate product may require additional acid for re-dissolution in acidic spent electrolyte for MM or electrolytic manganese dioxide (MD) production. Both methods of electrolysis, on the other hand, are net acid producers.

Continuous leaching was started on - " material, gradually filling up the thickeners to scope out mechanical settling and washing. While primary settling was readily achieved with a 30 g t flocculant dosage and an internal dilution to 10 solids, dispersion of the viscous and hydrophobic underflow was inadequate. Residual grit from leaching was sanding out, causing frequent stoppages in various locations. These bottlenecks were remedied and the pumping capacity was doubled; the decision was made to continue piloting on finer ground material only. Accordingly, six drums of -3" feed were prepared by Inspectorate Mining and xploration Services Ltd. of Richmond, British Columbia (Inspectorate) in record time to provide at least 1.5 t of 200-mesh material at a 60 pulp density for continuous operations.

Dissolved Mn^2 cation levels in the accumulated PLS were built up by its extensive recycling to the leach during the remedial period (Table 13.4). Once a modest 10 g L Mn^2 concentration was reached, recycling of 3 g L Mn^2 wash water maintained a steady PLS level at a leach pulp density target of about 30 solids.

Table 1 . nitial olution Assays at Decreasin Pulp Densities

CP-O	eads P 1, Av . m /	60-min P 1 1 -5/ m /	P -10 m /	ead P in P m /	60-min P -1/ P m /
olution Assays					
Al Aluminum	2	4	177	2	77
Ca Calcium	963	931	1,243	670	1,074
Fe Iron	1	1	541	1	218
K Potassium	33	170	282	58	132
Mg Magnesium	316	682	625	323	390
Mn Manganese	1,441	17,879	18,543	3,095	9,295
a Sodium	673	1,037	983	2,016	2,113
Si Silicon	23	59	218	37	111
n inc	0.8	76	88	2.1	28.1
Additional Assays Pertinent each Data					
pH	7.1	3.0	2.0	3.61	1.82
ORP, mV	243	750	320	140	243
SG, g cc	1.003	1.055	1.058	1.03	1.048
Acidity, mg L	-	1,580	8.120	ins.	ins.
SO_2, g L	-	-	7.52	-	-
SO_4, g L	4.10	18.8	28.5	8.4	21.1
S_2O_6, g L	7.30	38.8	36.0	7.1	19.3
Pulp Density,	50	65.0	50.0	30	30
Feed P_{80}, mesh	var.	5 8"	10	1 4"	1 4"
Mn Dissolved,	0	53.5	85.1	0	94.6
H_2SO_4, kg kg Mn	0.05	2.7	2.1	0.05	1.9

13.3. ES RIPTI TI S I E EA SE ME TS

Continuous piloting was initiated on milled feed by varying the reagent dosages as the work was underway. CCD retention times of about 1.5 h per 100 L thickener unit were established (0.22 m^2 t d unit area, 4 m h rise rate). This mode of operation was maintained over the first campaign shifts with elevated flocculant levels to achieve unstable sludge beds (Figure 13.3). A leach retention time of 15 min was more than adequate for feed pulps that were finer than 65-mesh, but fluctuating extractions were achieved due to cycling of reagent dosages (Table 13.5). Since the majority of residual manganese oxide levels were quite elevated, it would seem that leaching with sodium metabisulphite (MBS) is less effective than the use of calcium bisulphite (CaBS), even on these very fine materials.

i ure 1 . tart-up CCD evels Thic ener ≥ 11

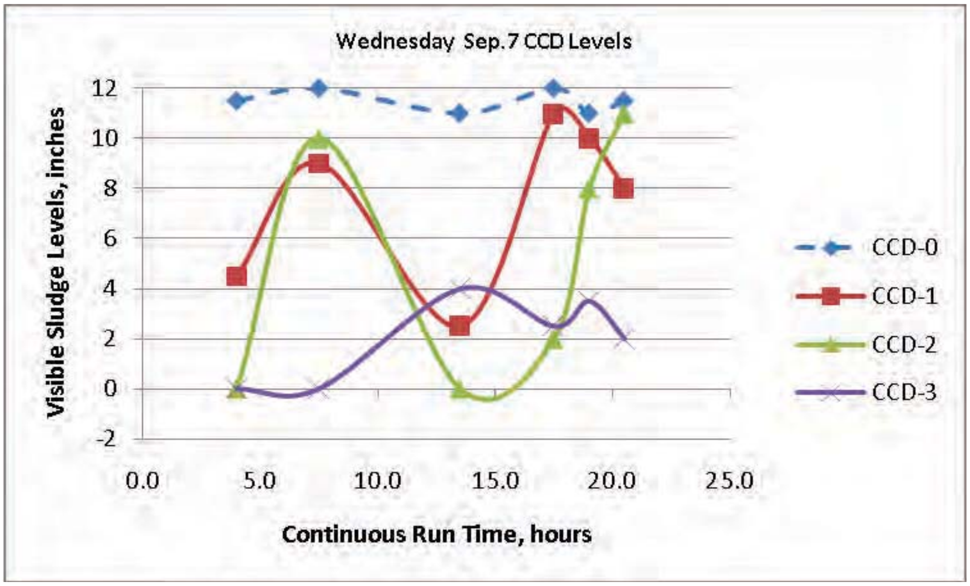


Table 1 .5 nitial ashed Continuous each esidue Assays by eco and

Sample Designation	S %	Al2O3 %	BaO %	CaO %	Fe2O3 %	K2O %	MgO %	MnO %	Na2O %	P2O5 %	SiO2 %	LOI %
WeD-13-T24I	0.48	14.69	0.85	1.29	5.67	6.32	1.96	0.31	1.07	0.08	62.90	3.81
WeD-15-T24R	0.51	13.42	0.86	2.30	5.14	5.44	1.88	4.63	1.09	0.17	58.68	4.83
WeD-15-T12O	0.55	14.24	0.91	1.37	5.36	5.88	1.91	0.24	1.08	0.08	62.00	3.89
WeD-15-T14O	0.65	13.10	0.85	2.34	5.08	5.34	1.80	4.78	1.11	0.17	57.48	4.99
WeN-01-T14O	0.22	13.02	0.80	2.68	5.29	5.34	1.78	4.65	1.12	0.18	57.93	4.83
WeN-08-T14O	1.13	12.73	0.82	2.84	5.10	5.16	1.75	3.43	1.14	0.17	57.06	4.77
WeN-01-T12O	1.05	12.87	0.81	2.51	5.39	5.33	1.73	4.28	1.04	0.18	56.67	4.52
WeN-05-C21U	0.97	13.58	0.86	2.27	5.23	5.68	1.77	0.46	1.09	0.17	60.50	4.19
WeN-05-T14O	1.17	13.02	0.81	2.81	5.29	5.32	1.74	1.65	1.14	0.18	58.67	4.13
WeN-05-T12O	1.29	13.30	0.84	2.78	5.27	5.49	1.72	2.23	1.06	0.13	59.60	4.54
WeN-05-C21U	0.65	13.76	0.86	1.89	5.41	5.69	1.82	2.61	1.07	0.18	60.22	4.09
WeN-08-T12O	1.17	13.50	0.85	2.81	5.48	5.48	1.84	1.41	1.12	0.18	59.85	4.40
WeN-08-C21U	0.93	13.42	0.82	2.34	5.31	5.59	1.79	2.96	1.08	0.18	59.28	4.35

The next continuous piloting test segments were operated at lower throughputs, more than doubling the retention to 4 h maximum per CCD unit. Sludge levels stabilized at lower flocculant dosages (60 g t Alfloc 90), for 5 to 15 internally-diluted thickener feeds, with underflow densities as high as 60 solids. Dissolved manganese occasionally oxidized in more neutral overflow samples, which could be avoided by pH adjustments with dilute acid in CCD-3. High CCD sequential dilution ratios, averaging 3.5 1, were evident throughout the continuous campaign (Figure 13.4, Table 13.6). Consistent manganese leach recoveries of over 93 were achieved based on the 4.35 MnO head assay, with residual whole rock assays dropping below 0.3 MnO (Table 13.7) when seeding with gypsum during the latter stages of the campaign.

i ure 1 . Continuous CCD Profiles

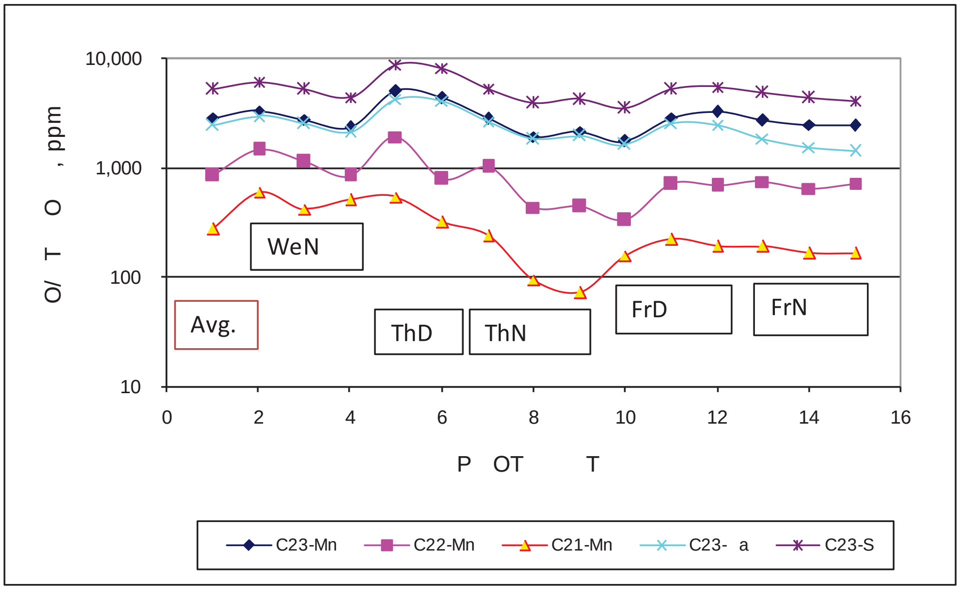


Table 1 .6 Avera e CCD ash atios over the ntire Campai n

Analyte	Avera e olution Tenors, m /					ndicated Dilution atios		
	T	C	C	C 1		T /C	C /C	C /C 1
Mn	10,831	2,881	856	281		3.76	3.36	3.05
a	8,853	2,434	711	213		3.64	3.42	3.34
S	11,615	5,280	1,930	945		2.20	2.74	2.04

Table 1 . inal ashed Continuous each esidue Assays by eco and

ample Desi nation		Al O	aO	CaO	e O	O	O	nO	a O	P O$_5$	iO	TiO	O	Total
ThD-12h-C21	1.02	13.42	0.84	2.52	5.25	5.49	1.82	1.96	1.11	0.17	59.58	0.61	4.39	100.66
ThD-18h-C21	0.84	13.97	0.93	2.26	5.32	5.60	1.92	0.74	1.11	0.17	61.84	0.63	4.29	101.81
Th -22h-C21	0.78	14.00	0.91	2.16	4.97	5.59	1.83	0.34	1.15	0.16	63.00	0.62	4.03	101.54
Th -02h-C21	0.78	14.11	0.95	2.02	5.06	5.63	1.87	0.36	1.14	0.17	63.00	0.64	4.02	101.58
Th -06h-C21	0.64	14.07	0.96	1.82	5.10	5.63	1.87	0.31	1.13	0.16	63.44	0.63	3.92	101.38
FrD-12h-C21	1.35	13.74	0.82	2.80	5.32	5.77	1.84	0.37	1.04	0.18	59.36	0.64	4.20	99.96
FrD-13h-C21	0.67	13.79	0.91	1.76	5.14	5.60	1.79	0.28	1.12	0.15	62.25	0.62	3.87	99.61
FrD-15h-C21	0.88	14.13	0.88	1.97	5.21	5.77	1.80	0.23	1.08	0.14	62.43	0.63	3.80	100.72
FrD-19h-C21	1.77	13.73	0.84	3.68	5.15	5.75	1.78	0.23	1.02	0.16	60.05	0.64	4.36	102.51
Fr -23h-C21	2.40	13.32	0.79	5.01	5.08	5.67	1.71	0.25	1.01	0.16	58.08	0.62	5.08	103.79
Fr -03h-C21	2.72	13.10	0.77	5.47	4.98	5.59	1.69	0.25	0.97	0.16	56.85	0.61	5.30	103.43
Fr -07h-C21	2.87	13.10	0.72	5.84	5.03	5.71	1.72	0.36	0.94	0.16	56.08	0.61	5.53	104.1

Provisions for gypsum seeding in the last phases of operation led to very stable and easily controlled sludge levels at moderate 2.5 h unit retention times. Settled underflow from each CCD unit reached the consistency of toothpaste, without difficulty in pumping, at slower discharge rates. Overflow clarity in all the thickeners was excellent, as the re-dispersion of pre-flocculated underflow into diluent overflow streams was achieved by intensive mixing with multiple pump heads. Once sludge from the entire CCD circuit had been systematically purged out, contents of the 100 L main reactor were discharged into a series of barrels. Samples from the non-flocculated leach pulp were shipped to Pocock Industrial (Pocock 2012) in Utah for independent SLS characterizations (Kemetco 2011c; Kemetco 2012b).

13.3.5 I TERPRETATI RES TS

Anticipated feed grades down to 2 Mn could be processed with maximum daily throughputs of 7,000 t, for a relatively constant manganese output of around 6.6 t h in the form of manganese carbonate or MM. Realistic requirements of 2.6 kg SO_2 per kilogram of manganese (Table 13.8) are likely, with up to 25 recycling via an off-gas scrubber. The non-optimized pilot predictions of sulphur consumption in this test work are up to 50 higher than expected with the use of sulphur dioxide reagent gas, based on comparison with data from available literature.

Table 1 . each ea ent Consumption stimates

Year/ ab.	Test/ ample	ead Assays,				each Consumptions / n				ea ent Types
		Ca	e	n	iO	O	O	Σ	p	
09 KRI	batch MGP	2.14	2.2	10.5	51.2	var.	var.	0.65	var.	SO_2 gas
08 PRA	batch MGP	2.21	2.34	9.82	-	1.43	2	0.98	1	bisulphites
11 KRI	batch CWH	-	-	-	-	2.20	2	1.37	2.5	bisulphites
Pilot	Overall	1.44	3.35	3.37	55.9	2.18	1.36	1.27	2.5	bisulphites
80 HRI	MGP	-	-	8.13	-	2.45	A	1.23	1	aq.SO_2
88 USBM	CWH 100	2.5	2.6	3.5	A	2.0	0	1.00	1	aq.SO_2
57 USBM	MGP	2.3	1.8	7.9	A	2.4	-0.19	1.20	1	SO_2 gas
RI 5323	Maggie	0.9	0.5	5.7	A	2.1	-0.15	1.05	1	SO_2 gas

ote Based on SO_2 gas mainly

Table 1 .9 Process Desi n stimates in a Conceptually nte rated ode

on-optimi ed ea ent stimates n- ecovery				Pro ected nte rative Details		
nit Operation	t.	/ n	n- ,	ecycle	in the form of	T, h
1. Leach	SO_2	1.5	-	-	Multi-stage Total	2
	H_2SO_4	0.5	-	-	Recirculating Load	
	H_2O	1	95	180 Mn	of Δ (feed), in WW	
2. SLS (5 CCD)	90	0.003	-	10 MM	from recycled sludge	6 2
	PW	41.9	-	50 H_2O	from F permeate	
3. Purification	Air MnO_2	tbd	-	1 of MM from anode sludge		2 2
2-stages	Lime	0.25	-	5 Mn	in iron cake to SLS	
	aHS	0.0005	94	10 Rgt	as MnS cement	
4. $MnCO_3$	aOH	negligible		incl. as SA	-	
2-stages	$aHCO_3$	if needed		incl. as SA	-	2 1
	a_2CO_3	1.5	90	10 MM	2nd cake to neutr.	
5. Crystallizer	Steam	20.5	MW	-	multiple stages, each	1
F	lectrical	1.5	MW	energy, steam and water		
(2.0 safety factor)	Water		recovery	bleed to leach	-	
	SO_2		recovery	vapour SO_2 to burner	-	
6. lectrowin	H_2SO_4	0.25	-	80 H_2	plating cycles	24 - 72
incl.	H_3	0.1	-	50 H_3	in Vapour to burner	
dissolution and	De-scale		-	100 O_2	-	
2nd purification	Oxalic		90	MnO_2 gypsum sludges		

In piloting, four CCD units were operated with variable, dispersed feed dilutions of 5 to 20 solids and ≤1.2 L min additions of wash water (approximately 20 in excess of overall feed throughput rate). Consistent underflow densities of ≥55 solids in pulp were achieved, with a unit area of 0.5 m^2 t d.

on-flocculated leach pulps, some weeks later, were tested at Pocock's laboratory in Utah. Their findings (Pocock 2012) were not as encouraging as observed during piloting. Pocock attributed this to either, segregation of the non-flocculated pulp in transit or changes in clay rheology with time. Combined test data agrees that filtration is slow due to blinding of the pores with fine clay particles. However, final cake moistures of 30 by weight are consistently achieved by conventional means. The prefeasibility results reported in this work are based on very conservative and non-optimized findings.

The manganese carbonate precipitate settled very well at 13 solids and 3.5 m h rise rate, with 40 g t AF308, to yield 59 solids underflow. A wash ratio of 1.0 produced 93.4 wash efficiency by vacuum filtration without any flocculant at a residual moisture of 40.5 . Pressure filtration showed similar wash efficiency with 27 of residual moisture.

13.3.6 E TRA I ATI A P RI I ATI

Acidic PLS (pH 2.5) consumed approximately 0.25 kg lime (as calcium oxide) per kilogram of feed manganese, with 3 h of aeration at pH 7.3 to remove about 0.55 g L iron and 0.19 g L aluminum, along with some arsenic, silicon, phosphorus and zinc by oxyhydrolysis. xpected lime requirements can be doubled to cover scrubber and environmental protection purposes. Manganese losses with the so-called iron cake (typically grading 15 Ca, 4 Fe and 6 Mn) can be reduced by the use of ammonia. Manganese oxides are unsuitable for neutralization, as the natural pH tends to be slightly acidic. It is suggested that iron cakes be blended with the leaching residue during the CCD washing stage to improve stability of the gangue. Other neutralizing reagents, including anodic manganese dioxide sludge and secondary manganese carbonate cake, should be investigated, with an eye on improving calcium control at elevated temperatures as well. Sequencing of the oxyhydrolysis to follow sulphide precipitation (at least partly to coagulate colloidal impurities) should also be tested for optimization purposes. Representative iron sludge was sent to Pocock for detailed SLS characterization, and dried iron precipitates were characterized for environmental evaluations.

The iron sludge, as tested in-house and at Pocock was difficult to dewater below 50 moisture, either by settling, vacuum or pressure filtration. Washing this sludge in the leach CCD was practiced in past USBM piloting operations. The ABA test showed a negative P, but SPLP had a lower arsenic level of 0.122 mg L as compared with iron-poor tailings. Fixation of the sludge was explored, using 20 additions of either cement or tailings. Both treatments lowered the arsenic and thallium to levels satisfying the Aquifer Water Quality Standards, although the tailings addition provided a much sturdier agglomerate after pelletizing and curing.

Residual metal impurities were removed by precipitation with sodium hydrosulphide at neutral pH, consuming approximately 1 g reagent per 30 to 50 L of PLS. Very fine and sticky zinc sulphide precipitates carrying significant amounts of manganese, aluminum, calcium, cobalt, copper, iron, nickel, silicon, thallium and arsenic will require separate disposal for environmental concerns. One proposed method would rely on encapsulating with higher-grade gypsum, obtained from the scrubber or even the electrowinning cells. Modest heating to 150°C produced hemi-hydrate (plaster of Paris) that could protect the metals from weathering effects; active carbon, use of coagulants or manganese sulphide cement could also be considered in optimization. Insufficient product was generated to allow more detailed testing of settling or filtration, or for environmental characterizations. It should be noted that similar solution purification is repeated to condition feed electrolyte for MM production (see below). Strict residual cobalt and nickel limits of less than 1 mg/L with ≤0.3 g L Ca will be needed for efficient manganese electrolysis in the re-dissolution of manganese carbonate with spent electrolyte. Difficulties in attaining these solution specifications delayed the MM production goals until such issues as gypsum fouling of membranes and passivation of the cathode were solved.

13.3. MA A ESE AR ATE PRE IPITATI

The intermediate production of manganese carbonate improves processing by addressing several key issues at the same time. Firstly, the product is easy to obtain even from rather dilute PLS at various purity specifications. It tends to be stable, and its grade is quite suitable for shipping. ntrainment of remaining calcium, magnesium, alkali and base metal impurities can to some extent be controlled by finely-tuned multi-stage precipitation recipes, while rejection of sodium sulphate and dithionate into the filtrate opens up a convenient byproduct recovery scheme. Test results and literature reviews indicate that pH or bicarbonate control at carefully designed dosages, as well as inert gas atmospheres and seed crystallization, play important roles, along with various recycle and refining strategies.

For present design purposes, it is assumed that a first stage precipitation of 90 of the manganese can be attained in fairly pure form by targeting enough bicarbonate to retain much of the calcium and magnesium in solution. Remaining cations are precipitated in a second stage with a slight excess of carbon dioxide at a pH 8 to eliminate residual bicarbonate. The impure secondary manganese carbonate cake is best recycled to the neutralization (oxyhydrolysis) stage for calcium control. Present overall reagent requirement, expressed in technical grade soda ash, typically amounts to 1.8 kg per kilogram of manganese feed, and is somewhat dependent on more precise operating conditions upstream.

A representative pulp sample was also sent to Pocock for more detailed SLS testing. Conventional thickening at Pocock achieved a 59 underflow density after dilution to 13 solids and flocculation with 40 to 45 g t AF308. A 10 m diameter thickener could provide the required unit area per 1,000 t d of throughput, and both vacuum or pressure filtration would be effective for washing the micro-crystalline product (finer than 500 mesh), if needed. Some sodium entrainment into the MM circuit would not be harmful, and the re-dissolved manganese carbonate would undergo additional purification in any case.

13.3. R STA I ATI A A I TRATI

As conceived and briefly explored in a previous program (Kemetco 2010), the recovery of sodium sulphate and sodium dithionate from carbonated liquors could be achieved by standard crystallization techniques, such as chilling and F. It was argued that near ambient conditions would yield the stable hydrated forms in contact with saturated solutions at designated temperatures and pressures. Solubility tests were conducted to establish empirical operating curves on synthetic and actually processed solutions. Crystal yields, morphologies and final impurity control measures are in progress, and initial results have confirmed that good quality crystals are formed with recovery of virtually potable water. Swenson International was retained to conduct assessments on recovery of sodium sulphate from the carbonated liquors by several different concentrative processes, including multiple effect deep vacuum crystallizers, chillers and steam jet compressors. It appears that the latter, known as mechanical vapour recompression (MVR) using available steam, could replace the chilling.

nergy costs and plant sizes can be drastically reduced if evaporative crystallization is considered. The recommended mode will produce anhydrous material, using ample waste heat available from the sulphur burner.

13.3.9 E E TR TI MA A ESE META

Conventional MM setups typically operate at 70 current efficiency if selenium is not used to stimulate conductivity. Ohmic losses give rise to heat that has to be removed from this stage. Techniques are essentially unchanged from USBM conditions derived in the 1940s. Plating of the electro-negative manganese from aqueous media is possible only at a non-acidic pH and 1 g L sulphur dioxide is added to help control favourable polarization at regions of high hydrogen over-potential. A diaphragm divides the cell to ensure that excess acid stays in the anolyte exiting the cell. Cathode deposits may be passivated with potassium permanganate to prevent re-dissolution of the manganese. Substantial hydrogen production competes with the plating of MM. However, the electrolysis of one mole of manganese sulphate regenerates one mole of acid. The initial tests suggest that gypsum sludge build-up on the diaphragm may obstruct flow of catholyte into the anode compartment, where the generation of oxygen or manganese dioxide will generate more acid. lectrolysis feed at 35°C, pH 6.5, contains 35 g L Mn and 130 g L of ammonium sulphate to buffer the catholyte below pH 9, with some losses of ammonia. Precipitation of salts is usually observed, and magnesium levels seldom need to be controlled over the longer term by purging an electrolyte bleed stream.

Tests on evaporated and calcined manganese sulphate showed few impurity control issues, but required rejection of excess acid released. Based on current observations and available literature and data, additions of 0.25 kg acid to the return anolyte (per kilogram of manganese) along with purification and reagent replenishments, could drive the manganese carbonate re-leach. Dissolution of the manganese carbonate in half of the spent electrolyte spiked with make-up acid would yield significantly lower gypsum saturation levels at elevated temperatures. After solution purification with removal of colloidal sulphides, the remainder of the anolyte is used to adjust proper concentrations and pH levels to feed the cell. In such a manner, two 24-hour operating cycles were successful in producing MM of 99.7 purity, but further optimization and longer term locked-cycle testing will need to be pursued. Currently, it should be assumed that 67.5 current efficiency may be routinely achieved, with most of the losses in the form of hydrogen production at moderate current densities that will avoid some of the need for cooling.

14.0 MINERAL RESOURCE ESTIMATES

14.1 2011 RESOURCE ESTIMATE

Mineral resources from the Property were disclosed in Tribe (2011). A summary of this disclosure is discussed below. The 2011 estimate comprises some ten areas of the Property that were modelled as individual deposits, although there is likely geological continuity between many of these. They comprise

- Maggie Mine
- Shannon Mine
- Love's, Hurley, and Planche Mines
- South Chapin, Burro, Price, and Priceless ones
- orth Chapin Lakes MacGregor.

The 2011 estimate for the orth Chapin Lakes MacGregor area is superseded by a new resource estimate that is disclosed in Section 14.2.

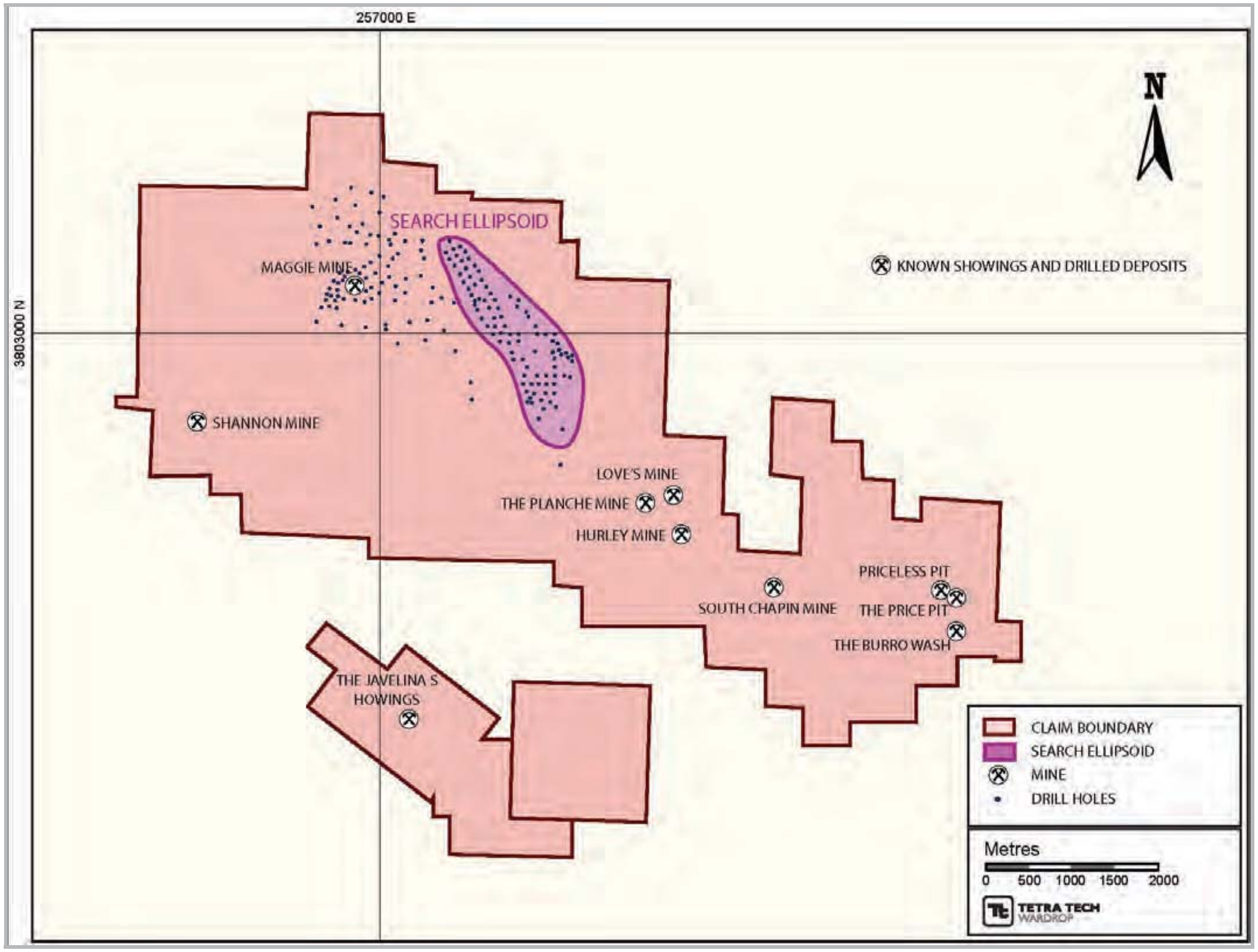
1 .1.1 RES R E A ATI PARAMETERS

The information in this section is summarized from Tribe (2011).

The 2011 resource calculations were based on a polygonal method where the area of influence for each drillhole was based on a polygon drawn around each hole and projected half way to the adjacent hole or a fixed distance in each direction away from the hole. The maximum distances used in this calculation were 100 m for an Indicated classification and 200 m for an Inferred classification. Grades were projected within established geological units and in many areas were controlled by faults. The area of the polygon was measured and the volume calculated based on the true thickness of the intercepts of material in the hole that exceeded a pre-established cut-off grade of 0.90 Mn. o internal waste was included in the intercept width and no external waste or dilution was included in the volume. The density used to determine the corresponding tonnage was 2.10 t m^3. The grade assigned to this tonnage is the assay value for that particular material recovered from the beds intersected by the hole over that width.

Figure 14.1 provides a map of the Property showing the general location of the various zones included in the 2011 estimate. The area outlined in purple is the approximate boundary of the orth Chapin Lakes MacGregor area that has been re-estimated in Section 14.2.

i ure 1 .1 Property ap ho in an anese-bearin ones ncluded in the 011 ineral esource stimate



The data support for each of the nine additional areas estimated in 2011 is summarized briefly below. Grade and tonnage values are grouped for many of these areas, as shown in Table 14.1. The previously reported values for the orth Chapin, Lake, and MacGregor ones are not discussed below as they are superseded by the new estimate presented in Section 14.2.

Table 1 .1 ummary of ineral esource stimates Tribe 011

Deposit/ e ion	esource t	Avera e Grade n
ndicated Grade Cut-off 0.90 n		
Maggie Mine	113,663,426	2.90
Shannon Mine	1,507,022	3.96
Love's, Hurley, and Planche Mines	3,616,579	5.16
South Chapin, Burro, Price, and Priceless ones	24,788,169	2.87
Total Tonnes/Grade	1 ,5 5,196	.9
nferred Grade Cut-off 0.90 n		
Maggie Mine	40,360,508	2.71
Shannon Mine	4,298,133	2.74
Love's, Hurley, and Planche Mines	4,110,834	4.00
South Chapin, Burro, Price, and Priceless ones	5,930,764	2.87
Total Tonnes/Grade	5 , 00, 9	.

MA IE MINE

The mineral resource estimate on the Maggie zone was based on 2011 drilling and on earlier work conducted by the US Department of the Interior and by M.A. Hanna. The majority of material in the region of the Maggie Mine was classified as an Indicated Resource.

PLANCHE MINE

The Planche Mine deposits were sampled historically by three adits driven into Planche Mountain below the mesa wall rim-rock. This mineralization was projected the full extent of the mesa, a total of 650 m. The majority of material in the region of the Planche Mine was classified as an Inferred Resource.

LOVE S MINE

The Love's Mine deposits outcrop in the Chapin Wash. Several historical adits and declines were completed into these deposits. Material in the region of Love's Mine was projected 175 m down-dip to where the Common Corner Fault's West Splay is interpreted to cut the mineralized beds. Some of this material is classified as an Indicated Resource and the remainder as an Inferred Resource.

H RLEY MINE

The Hurley deposits also outcrop in the Chapin Wash and are projected down dip to where the Common Corner Fault's West Splay is interpreted to cut the mineralized beds. Some of this material is classified as an Indicated Resource and the remainder as an Inferred Resource.

SO TH CHAPIN ONE

The South Chapin deposits were projected to distances of up to 500 m based on information provided by two drill holes intersecting this mineralization. This material is classified as an Inferred Resource.

RRO ASH ONE

The Burro deposits are observed outcropping in the Burro Wash and were projected 425 m to where the Price fault is interpreted to truncate this mineralization. This material is classified as Inferred Resource.

PRICELESS MINE

The Priceless deposit was estimated using the surface area of the structure and the published production grades from historical mining from the pit. Grades are projected 50 m below the bottom of the pit. This material is classified as an Indicated Resource.

PRICE MINE

The Price deposit is projected approximately 100 m to the west of the mine where the beds are interpreted to be cut by the Price Fault. Grades are based on historical mining data. This material is classified as an Indicated Resource.

14.2 2012 RESOURCE ESTIMATE

1 .2.1 S MMAR

The current Artillery Peak resource supersedes the Tribe (2011) estimate for the orth Chapin Lakes MacGregor area and is based on drill data to September 2011 provided by AMI.

The mineral resources has been estimated by Michael O'Brien, Pr. Sci. at., of Tetra Tech, a Qualified Person who is independent of AMI and responsible for the estimate in this report. The resource was estimated using Isatis software v11.0.2. Manganese metal has been estimated into the block model, and the model was populated with bulk density estimates based on literature and limited local sampling. Table 14.2 summarizes the current mineral resource estimate for the Property. At a base-case cut-off of 1 Mn (highlighted in grey in Table 14.2), this estimate includes an Indicated Resource of 62,201,000 t at an average manganese grade of 2.3 , and an Inferred Resource of 20,033,000 t at an average manganese grade of 2.5 .

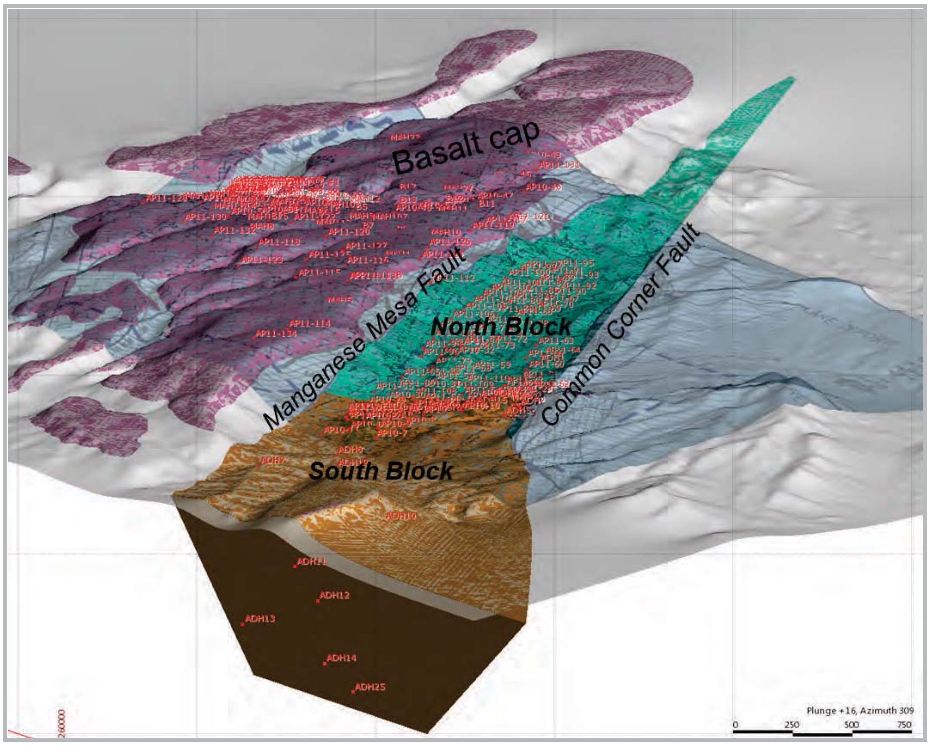
Table 1 . ovember 011 ineral esource stimate

Cut-off n	esource t	Contained n etal t	Avera e Grade n
ndicated			
0.5	65,658,000	1,474,000	2.2
1.0	62,201,000	1,445,000	2.3
1.5	53,938,000	1,339,000	2.5
2.0	38,093,000	1,062,000	2.8
nferred			
0.5	20,377,000	509,000	2.5
1.0	20,033,000	506,000	2.5
1.5	17,351,000	471,000	2.7
2.0	11,724,000	372,000	3.2

1 .2.2 ESTIMATI MAI T E ARTI ER PEA EP SIT

The geological setting of the deposit has been previously described and resources estimated for the Property (Tribe 2009; 2010; 2011). The mineralized zones were estimated within a graben block trending north-northwest to south-southeast, bounded by the Manganese Mesa Fault on the west (which dips at 55 to 60 degrees towards the east) and the Common Corner Fault on the east (which dips at 50 to 60 degrees towards the west). Within the graben, the rocks may be divided into north and south blocks which are separated by the sub-vertical, northeast-to-southwest-trending Lake Cobwebb Faults. The mineralization is concentrated in lenticular bodies, and to the west of the graben, the hills are capped by a resistant basalt flow, which is up to 10 m thick. These features are illustrated in Figure 14.2 and Figure 7.2.

i ure 1 . Perspective ie of the Artillery Pea Deposit, oo in orth- ortheast

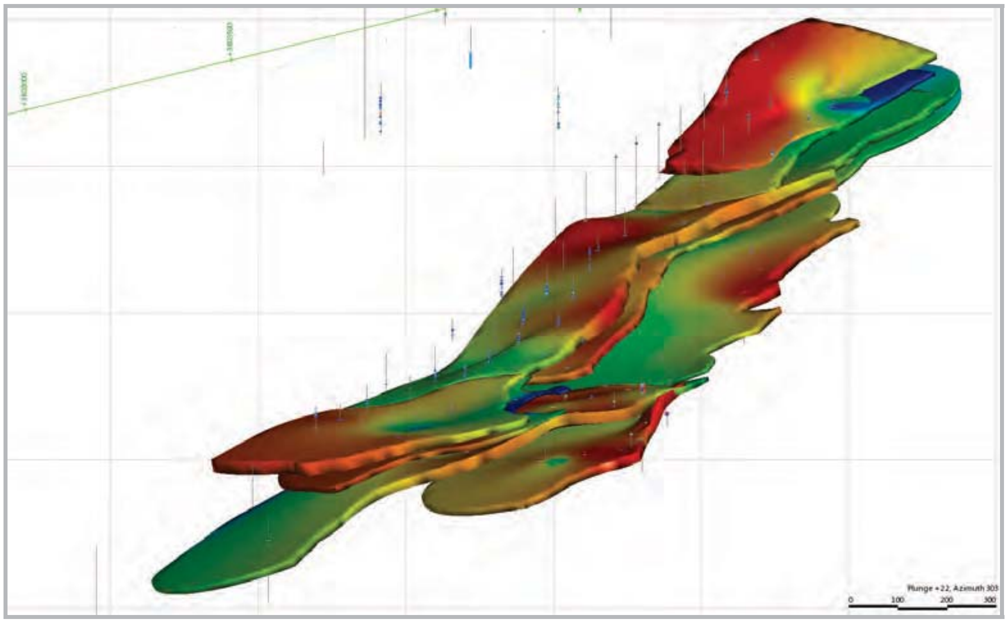


ote Top right is northeast; bottom left is southwest

A nominated grade threshold of 0.9 Mn was used to segregate mineralized from unmineralized material in the drillhole data within the graben structure between the Manganese Mesa Fault on the west and the Common Corner Fault in the east. Mineralized envelopes (nine in total) were modelled in three dimensions using Leapfrog software v2.4.109 and a 20 m triangle resolution between drillhole intercepts. The resulting geometry was similar to the local geological pattern of mineralized lenses and is an appropriate basis for mineral resource estimation. Material outside the mineralized envelopes was assumed barren, which is a conservative but prudent assumption. The mineralized lenses have been displaced by the Lake Cobwebb Fault.

The wireframes built for the mineralized lenses are 3D solids constructed progressively from drillhole data (Figure 14.3). A total of 106 drillholes fall within the geology solid, and these holes include 3,660 assay samples, 1,520 of which are mineralized. The best-fit hangingwall and footwall points constraining grades of greater than or equal to 0.9 Mn were identified in drillholes. Hangingwall and footwall surfaces were based on these points, and the hangingwall and footwall surfaces were then joined to form solid, lenticular units. Most manganese grades greater than or equal to 0.9 were constrained within these envelopes, but there are some isolated areas with grades greater than 0.9 outside the envelopes and, conversely, there are some low grade areas less than 0.9 Mn within the envelopes. The mineralized lenses were terminated against topography and faults.

i ure 1 . Perspective ie of the inerali ed ireframes, oo in orth- ortheast



ote Top right is northeast; bottom left is southwest.
Colour coding is thickness of the envelopes (dark blue 5 m, to red 25 m)

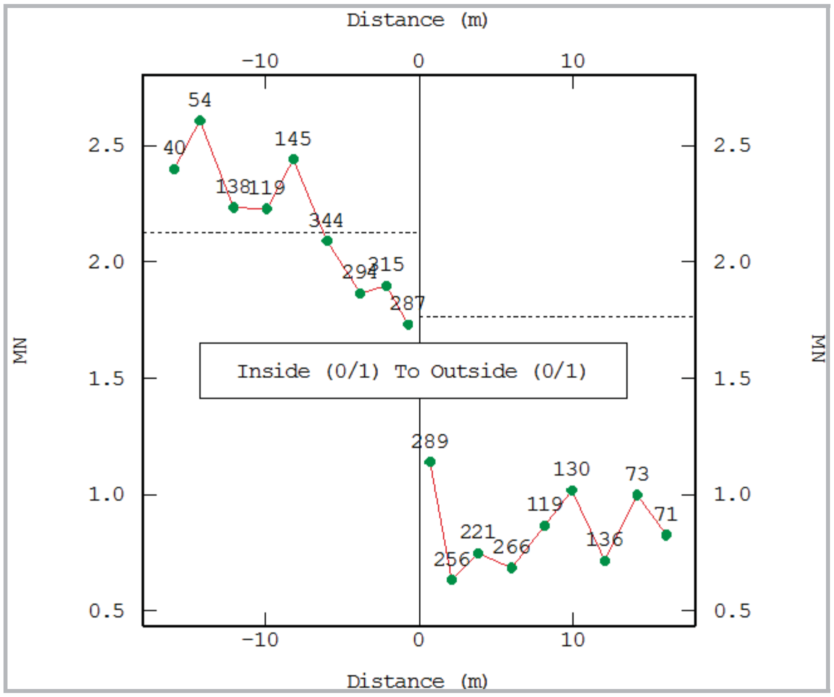
The geometrical characteristics of the manganese-rich zones are summarized in Table 14.3.

The 3D wireframes were verified by checking the disposition of manganese assay data inside and outside the wireframes. A boundary analysis was carried out by checking the average grade of samples at different distances from the wireframe surface. Averages of manganese sample grades within 2 m thick onion skin samples both inside and outside the manganese-rich envelope model are shown in Figure 14.4.

Table 1 . Geometrical Characteristics of an anese- ich ones

ame	olume m	nclined Area m	Thic ness m	urface Area m	ample Av . n	1.5 m Comp.	urface Area/ ncl. Area
Mn 01	2,851,200	223,760	12.74	475,120	3.97	152	2.12
Mn 02	5,979,300	862,140	6.94	1,763,900	1.67	120	2.05
Mn 02 S	8,276,100	567,070	14.59	1,180,000	2.80	338	2.08
Mn 03	6,573,500	658,600	9.98	1,366,000	2.37	180	2.07
Mn 03 S	2,315,900	247,720	9.35	516,170	1.16	87	2.08
Mn 04	10,273,000	552,770	18.58	1,187,900	2.55	373	2.15
Mn 04 S	4,396,100	237,150	18.54	514,940	2.12	85	2.17
Mn 05	2,480,200	203,620	12.18	432,430	2.11	69	2.12
Mn 06	4,055,600	182,410	22.23	407,860	2.21	116	2.24
Total	, 00,900	, 5, 0	-	, , 0	-	-	-
Avera e	-	-	1 .6	-	.	-	.10

Figure 1 . oundary Analysis Graph, sho in Grade ariation of amples Close to the ar ins of the ireframes



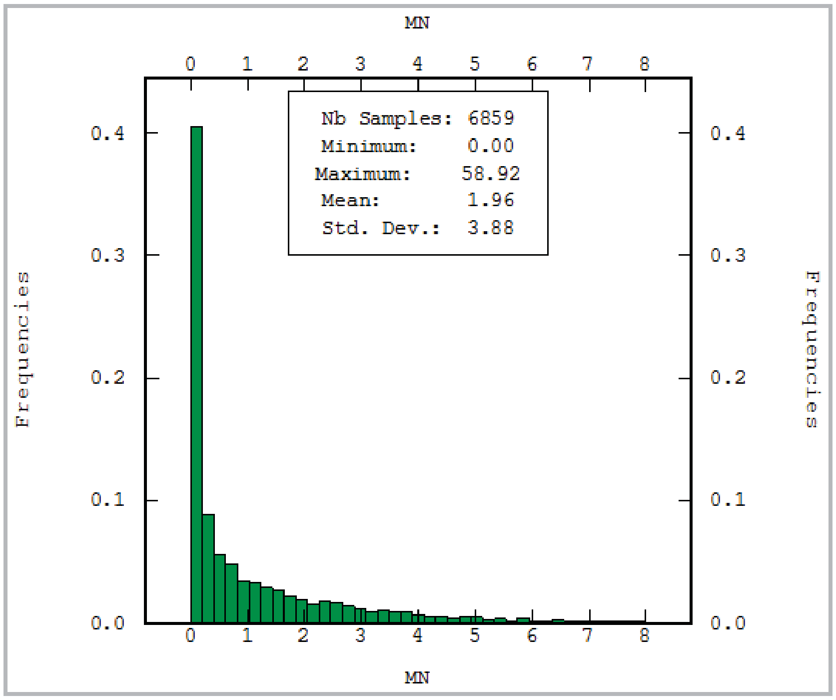
There is a pronounced step in grade from 0.7 Mn outside the wireframe model to 2 inside the model, which indicates that the wireframe modelling process was successful in consistently isolating the higher manganese grades in the deposit. It should be noted that manganese is present at levels higher than 0.5 for some distance outside the wireframes.

1 .2.3 E P RAT R ATA A A SIS

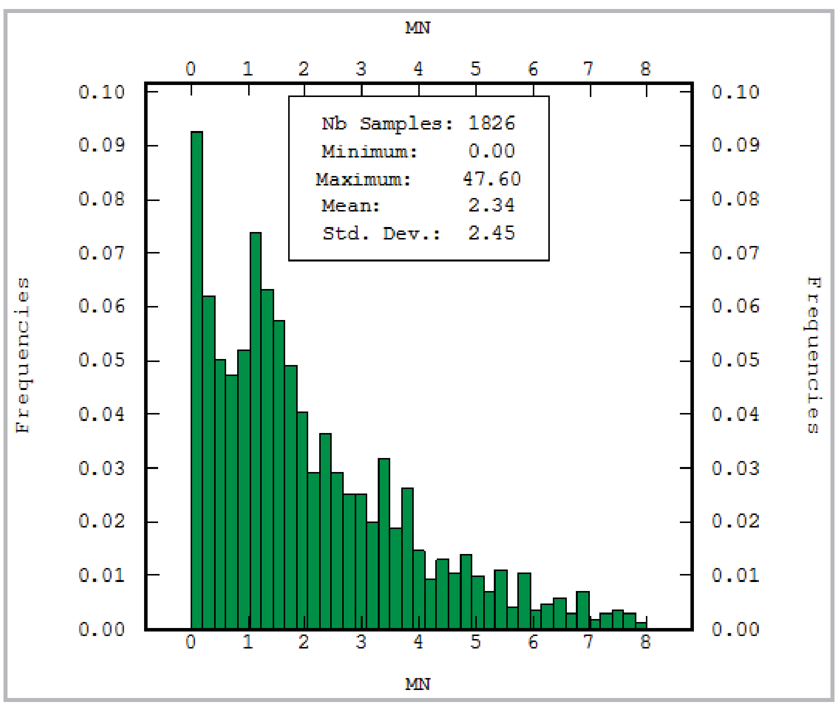
Drillhole data was imported into Isatis software (version 11.0.2) from text files and was used to generate the block estimates. Figure 14.5 shows the histogram of manganese grades weighted by length.

The histograms of the length-weighted samples and 1.5 m composites show two peak grade frequencies, one at less than 0.5 Mn, and a peak frequency of grades at 1.5 Mn (Figure 14.6 and Figure 14.7. The former represents thin intercalations of barren material included within the wireframes, and the latter represents the mode of samples within the better-mineralized portions of the deposit.

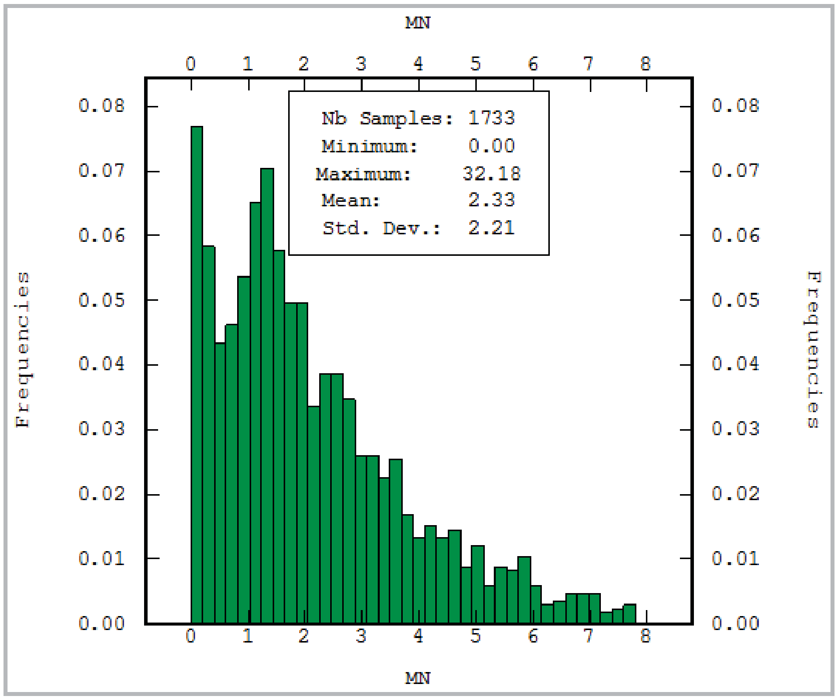
i ure 1 .5 an anese isto ram, ei hted by en th,
all amples nside and Outside the inerali ed ones



i ure 1 .6 an anese isto ram, ei hted by en th,
amples ithin an anese- ich ones D ireframes



i ure 1 . an anese isto ram, 1.5 m Composite amples ithin an anese- ich ones D ireframes

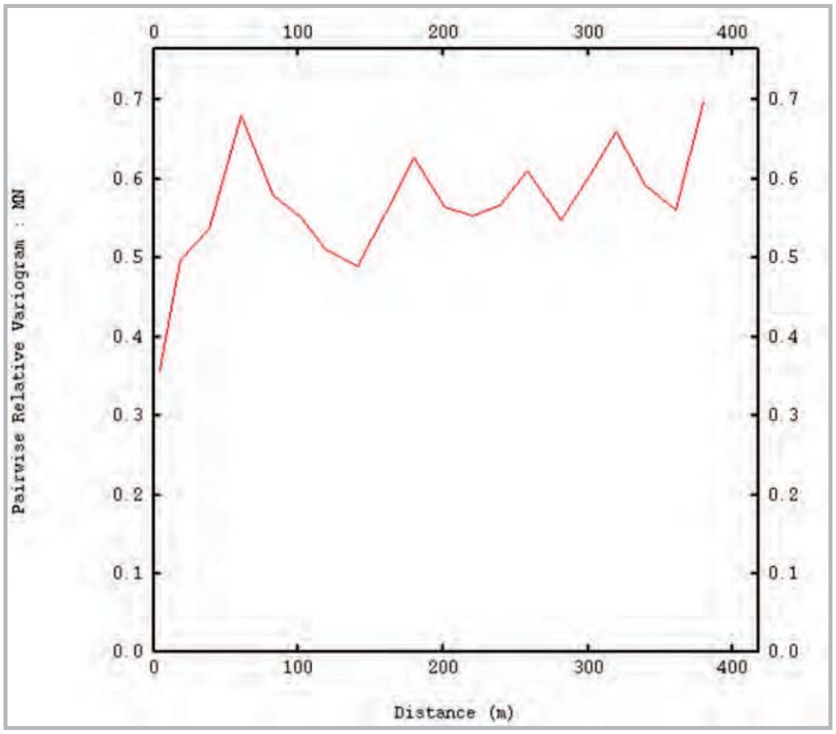


1 .2. SPATIA ATA A A SIS

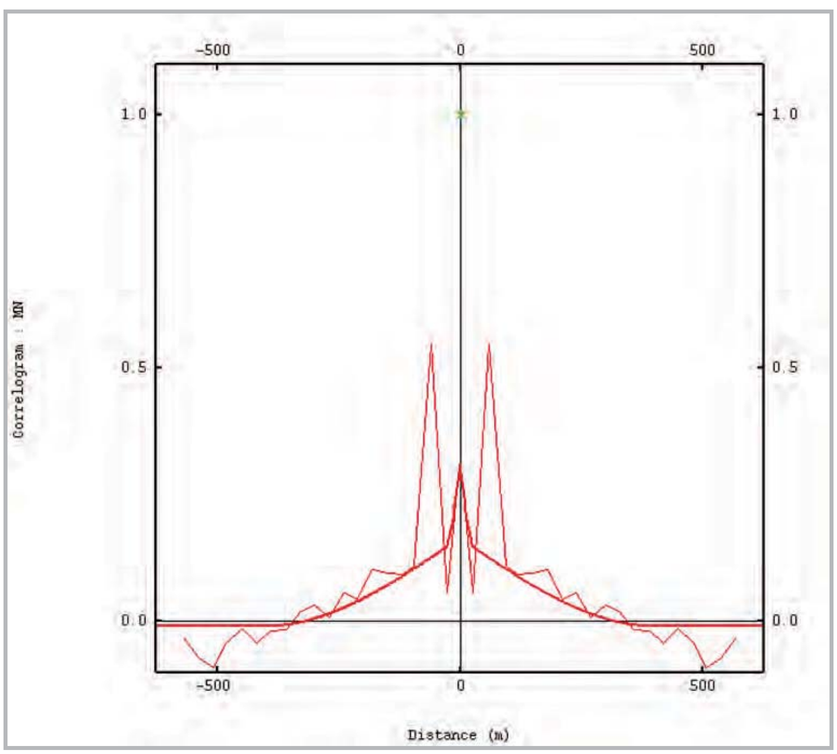
Geostatistical estimation by means of kriging requires an analysis of the spatial variability of the component to be estimated. This can be summarized in the form of an experimental variogram (or semi-variogram), which describes the expected difference in value between pairs of samples at different distances apart and at different orientations. The experimental variography can be used to model the theoretical difference between sample pairs, which can be used to inform the kriging-weighting scheme applied to samples in order to estimate grades in blocks from samples in the vicinity. The experimental variogram within the mineralized manganese zones is shown in Figure 14.8. Another graphical tool for the analysis of spatial variability is the correlogram, as shown in Figure 14.9.

The experimental variogram of the assay values show correlation to a range of 400 m within the plane of the stratigraphy, indicating a high degree of continuity, which is expected in this type of deposit. The nugget effect (or the inherent internal variability) is relatively high, indicating a comparatively high degree of uncertainty between closely spaced samples. The repeatability of the sample preparation and assay process may also be significant contributors to this uncertainty, as both sample variability and in situ geometric distribution of grades contribute to this factor.

i ure 1 . an anese, 1.5 m Composites perimental ario ram, ithin inerali ed nvelopes



i ure 1 .9 an anese, 1.5 m Composites perimental Correlo ram, ithin inerali ed nvelopes



Using the variogram and correlogram for guidance, the following variogram model was constructed Gamma(h) 0.26 0.6 Sph(30;30;3) 0.1 Sph(400;400;10). The plane of maximum elongation of the variogram model was 151° azimuth with a 17° dip and a 90° plunge. The ranges of variograms normal to stratigraphy are 10 of the ranges parallel to the stratigraphy. The behaviour of the experimental variograms calculated downhole or at right angles to stratigraphy is not clear-cut, due to the limited continuous sampling, but the variogram model is considered adequate for this level of study.

1 .2.5 RES R E M E

The block size was set at 40 m x 40 m x 1.5 m, with no sub-blocking and no rotation (Table 14.4). The size is adequate, considering the drillhole spacing (around 80 m) and the sample composite size of 1.5 m.

Variables in the block model included solidprop (proportion of block that is solid) and Mn prop (proportion of block that is part of the defined manganese-rich zones). In each case where solidprop and Mn prop of less than 100 exist, the remainder of the block should be considered air (bulk density 0) or waste, respectively. Blocks where at least 10 of the block volume was within the manganese rich wireframe (i.e. Mn prop 0.1) have been estimated for manganese percent. The remainder should be considered to have a manganese percentage grade of zero.

Table 1 . loc odel Geometry

		Y	
Origin (m)	257,400	3,801,520	452.5
Block Size (m)	40	40	1.5
Block Count	50	70	220
Model xtent (m)	2,000	2,800	330

1 .2.6 I TERP ATI P A

Ordinary kriging (OK) has been employed as the grade interpolation methodology. The model that was used is described by the formula

amma 0.2 0. 30 30 3 0.1 400 400 10 .

The plane of maximum elongation of the variogram model was 151° azimuth with a 17° dip and a 90° plunge.

The search ellipse was oriented parallel to the variogram model. The search distance in the plane of the stratigraphy and plane of the variography long ellipse was 400 m, while the search in the vertical plane was restricted to 10 m. A minimum of five samples were used for each block estimate, with additional samples used up to a maximum of 30 samples (Table 14.5). The estimation was hard-bounded, as only samples within the manganese-rich zones were used for estimation of the zones. o estimation was carried outside the manganese-rich wireframes, and no assay values were capped.

Table 1 .5 ineral esource[1]

Cut-off	esource t	Contained n etal t	Avera e Grade n
ndicated			
0.0	67,220,779	1,473,838	2.19
0.5	65,658,318	1,473,797	2.23
1.0	62,201,129	1,445,047	2.32
1.5	53,937,671	1,339,240	2.48
2.0	38,093,110	1,061,982	2.79
nferred			
0.0	20,384,577	508,948	2.50
0.5	20,376,715	508,911	2.50
1.0	20,033,032	505,874	2.53
1.5	17,351,110	470,957	2.71
2.0	11,723,602	372,000	3.17

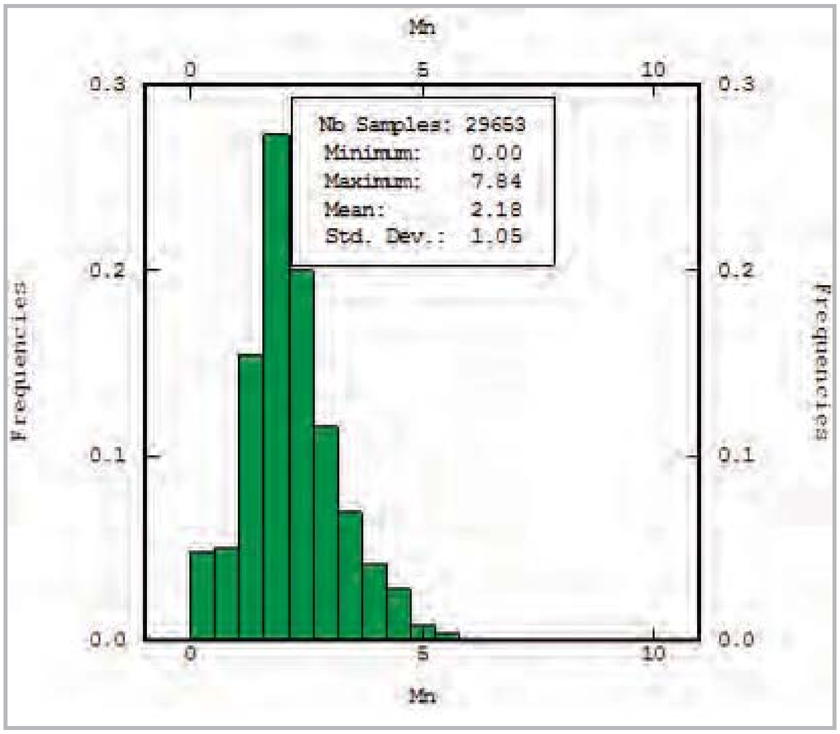
ote [1]Cut-off 0.0, includes all estimated blocks material within the 0.9 Mn grade shell.

1 .2. M E A I ATI

Block model validation of the OK interpolation involves direct comparison with a similar interpolation of the same sample data. These interpolation methods include nearest neighbour (polygonal estimation) and inverse distance squared (ID^2) block models. The OK model was completed using Vulcan software. Block statistics were used to interrogate global similarities and differences by domain. The model and samples were visually examined to ensure no obvious interpolation anomalies. Swath plots were used to investigate similarities and differences across identical volume references, as defined by the block size, across the entire deposit. The following sections present the results of the investigation. A histogram of block estimates is shown in Figure 14.10.

The block estimates within the mineralized envelopes range from less than 1 Mn to 7.8 Mn, with a mean grade of 2.18 Mn.

i ure 1 .10 isto ram and tatistics of loc stimates

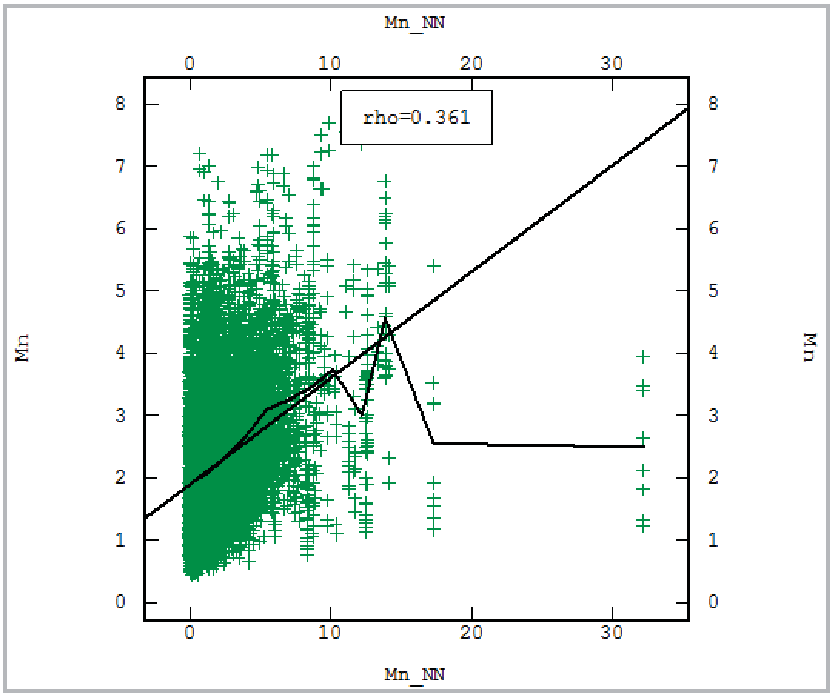


COMPARATIVE LOCK STATISTICS

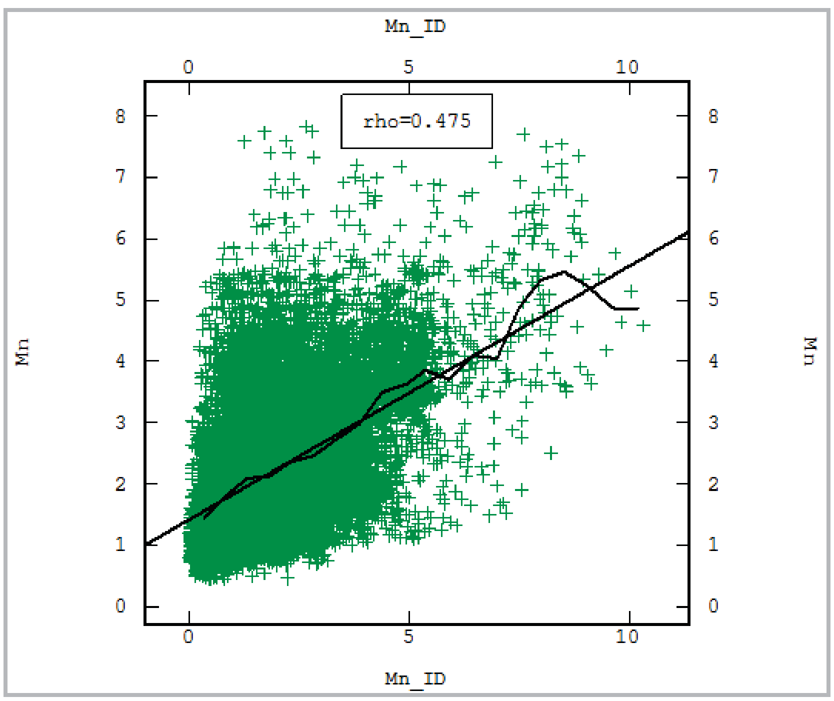
Using Isatis software version 11.0.2, parallel block interpolations using and ID^2 interpolations were conducted for comparison with the OK interpolation. Regression plots were produced to examine the relationship between the different types of estimates to ensure that corresponding grades were accurately reproduced by the different methods (see Figure 14.11 and Figure 14.12).

The method by definition captures extreme values, as it reflects the nearest sample to the block being estimated. Consequently, it completely overstates the variability of the estimates compared with a block interpolation, and has no change of support characteristics. The correlation coefficient (rho) of 0.361 is relatively low, but is significant considering the number of pairs being compared, and at least indicates that high and low grades in the OK reflect high and low grades in the samples. The ID^2 estimates are less variable than the approximations and show a more significant correlation to the OK estimates, with a correlation coefficient (rho) of 0.475.

i ure 1 .11 Comparison of an anese loc stimates O vs.



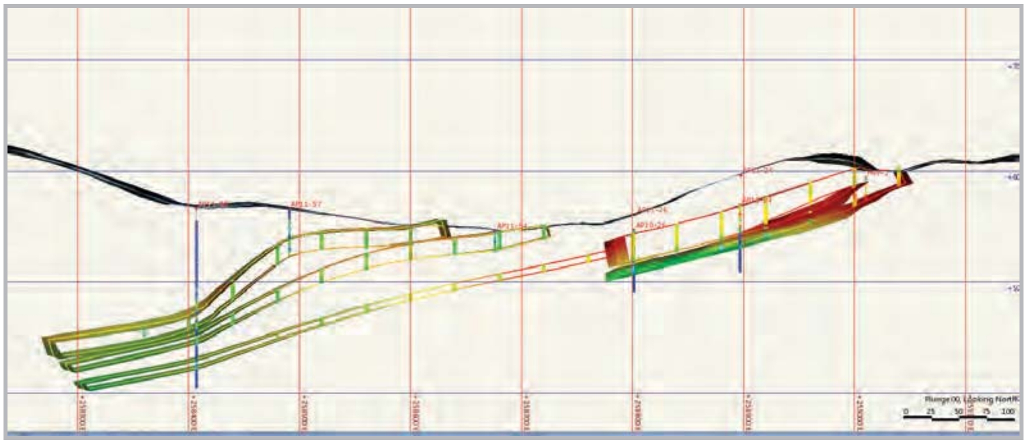
i ure 1 .1 Comparison of an anese loc stimates O vs. D



VIS AL E AMINATION

Figure 14.13 is a representative cross-section showing the block model interpolated grades against respective drillhole data.

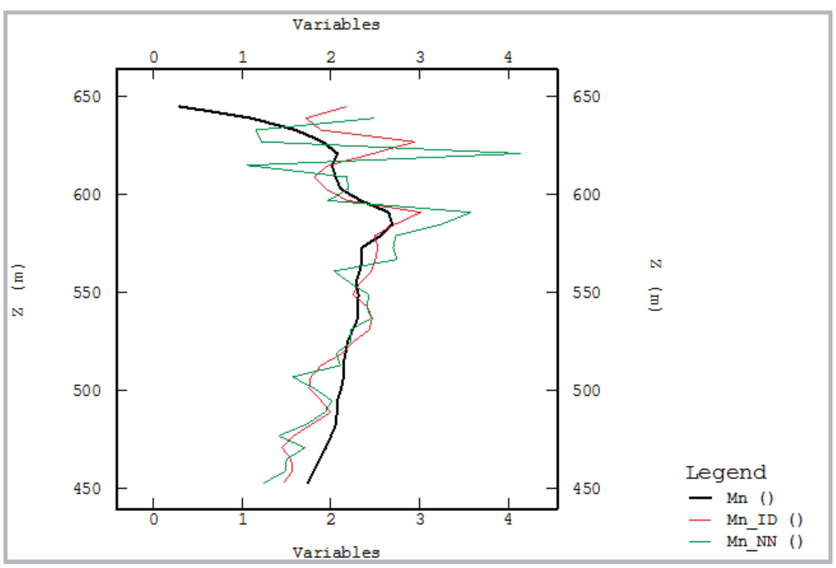
i ure 1 .1 loc odel Centroids, Drillhole amples, inerali ed nvelopes and Topo raphy on Y , 0 ,600



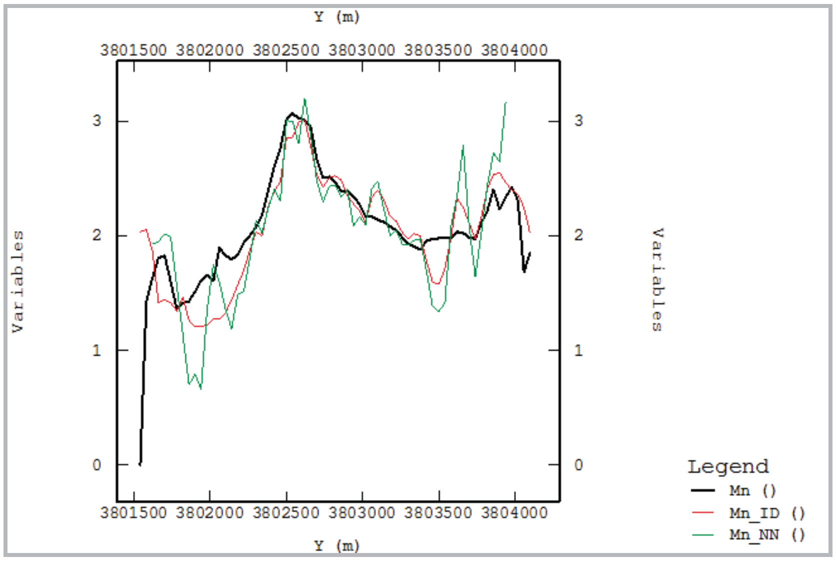
S ATH PLOTS

Comparative swath plots are presented in Figure 14.14 to Figure 14.16. The estimates are highly variable and preserve the extreme sample values. Overall, the , ID^2, and OK estimates show a very similar pattern and similar grades.

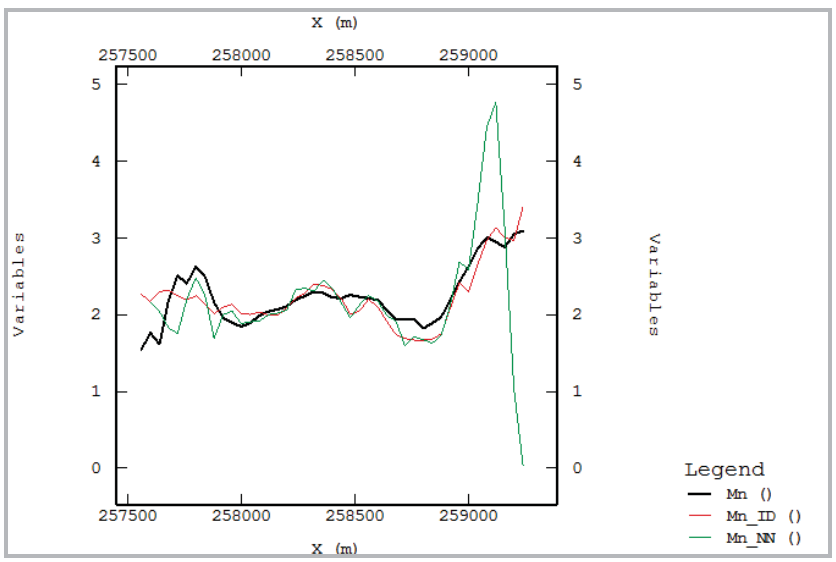
i ure 1 .1 an anese odel ath Plots by ench



i ure 1 .15 an anese odel ath Plots by o Y



i ure 1 .16 an anese odel ath Plots by astin



1 .2. E SIT

The bulk density values used in resource estimation for mineralized and non-mineralized or country rock material are described below. In the absence of a comprehensive and representative sampling program, conservative (low value for mineralized material and high values for country rock) values have been used in both instances.

MINERALI E ROCKS

The SG data follows on from Tribe (2010), which used two specimens (averaging 2.1 t m^3). The 2.1 t m^3 used in Tribe (2010) is used and considered appropriate for the current estimate, based on supporting evidence discussed below.

This mineralized density value is supported by a review of analogous deposits, together with historical SG sampling and sampling conducted in 2011. The historical SG figures are derived from the USBM and indicate a density for mineralized material of 2.13 t m^3 in the region of the Maggie Mine (Tribe 2010). Other studies of manganese wad material yield variable but overall higher density values such as 2.8 to 3.9 t m^3 (Mohapatra et al. 2010) and 3.0 to 4.2 t m^3 (Hoffman 1957).

Density sampling conducted by Tribe on the Property in 2011 (described in Section 9.4 and Section 11.2.4) focused on material in the area of the MacGregor mine and included a variety of mineralized samples (both relatively low and high grade) as well as sandstone and basalt country rock. Six samples of mineralized material were collected with values varying from 1.96 to 3.29 t m^3 and averaging 2.41 t m^3.

Together, the above discussion demonstrates that the SG of 2.1 t m^3 used for this estimation is a reasonable, conservative global SG value for the mineralized material. Tetra Tech is aware of the risk of overestimating the density of wad, a material with a history of variable bulk density in other deposits. Overestimation of the bulk density of the manganese-bearing material will directly lead to overestimation of the metal content of a volume of rock by the same margin. Overestimation of the bulk density of the manganese-bearing material, combined with underestimation of the overburden will lead to overestimation of the capacity to mine by open-pit methods, as costs are generally expressed on a tonnage bases. Moving forward into a feasibility study on the Property, further bulk density work is recommended to support future mineral resource estimates and classification (Sections 14.2.1 and 14.2.6).

NMINERALI E ROCKS

An average SG of 2.9 t m^3 value has been used for country rock basalt and 2.4 t m^3 has been used for country rock sediments. These values are considered appropriate for the current estimate based on supporting evidence discussed below.

Density sampling conducted in 2011 (Section 9.4 and Section 11.2.4) included non-mineralized and country rock material. Four samples of non-mineralized (waste) sandstone were collected, including one sample inter-bedded with mineralized sandstone. SG values for these samples range from 2.15 to 2.59 t m^3 and averaging 2.34 t m^3. Five samples of country-rock basalt were collected including fresh, weathered, vesicular, and amygdaloidal samples. SG values for these samples range from 1.25 t m^3 (vesicular) to 2.71 t m^3 (fresh), averaging 2.09 t m^3. Based on mapping of the basalt in the region, the majority of the basalt will likely be relatively fresh and poorly vesicular, and therefore an SG typical of the fresh basalt samples is more likely to be encountered.

The SG values obtained for country rock material on the Property provide confidence that the global SG values of 2.4 t m^3 (sandstone) and 2.9 t m^3 (basalt) are valid, conservative global SG values for the non-mineralized country rock material.

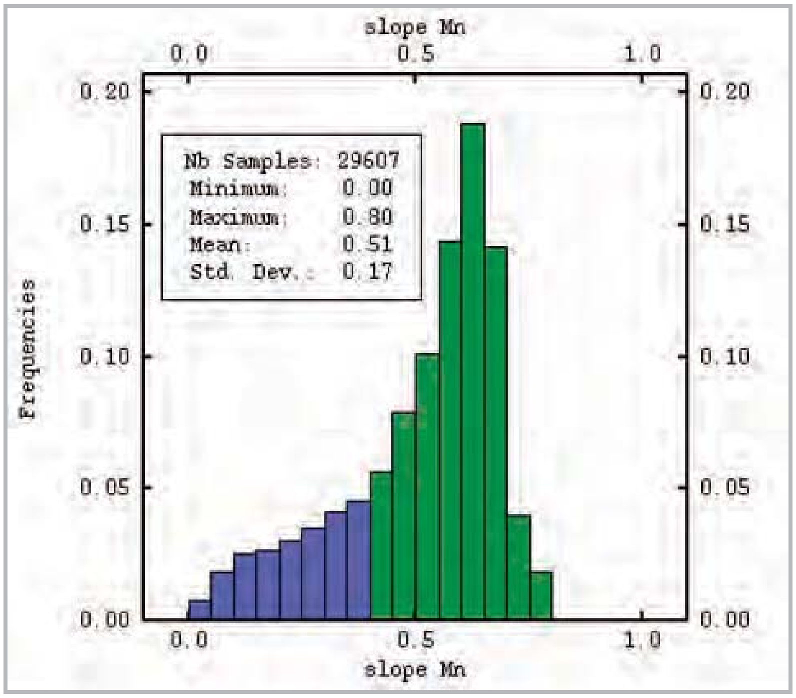
1 .2.9 MI ERA RES R E ASSI I ATI

The data supporting the Artillery Peak deposit consists of drillhole data and surface outcrop mapping. The level of confidence that can be ascribed to the block estimates is proportional to the amount of sampling information that exists in the vicinity, and thus inversely related to the drillhole spacing. By prescribed definition, Mineral Resources do not have demonstrated economic viability. Measured and Indicated Resources are that part of a mineral resource for which quantity and grade can be estimated with a level of confidence sufficient to allow the application of technical and economic parameters to support mine planning and evaluation of the economic viability of the project. Inferred Mineral Resources are that part of a mineral resource for which quantity and grade can be estimated based on geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. Inferred Mineral Resources are considered too speculative to allow the application of technical and economic parameters to support mine planning and evaluation of the economic viability of the project. One of the principal conditions for reasonable prospects for economic extraction is met by application of a cut-off grade. In the absence of operating experience or of a detailed feasibility study, it is generally accepted practice to use a cut-off grade from mines with similar characteristics (i.e. size, deposit type and grades). However, the mineralized material type, intended processing route and product value are rather different from those at existing manganese operations and consequently the cut-off grade (0.9 Mn , applied as a grade shell) is arbitrary at this time and will be explored during the remainder of this study. CIM definition standards for mineral resources and mineral reserves were followed for resource (CIM Definition Standards dated ovember 22, 2005, and adopted by CIM Council on December 11, 2005).

During the estimation process, a statistical measure of confidence in the estimate was generated. The slope of regression (SOR) parameter is a measure of local uncertainty attached to the estimate. The parameter assumes values between 0 (highly uncertain) to 1 (very confident). In the case of Artillery Peak, a minimum threshold SOR of 0.4 was considered to represent blocks with good confidence in the local estimates. Blocks with SOR less than 0.4 were considered to be of low confidence in local estimates. This criterion was used to separate Indicated mineral resources from Inferred mineral resources. The histogram of the SOR is shown in Figure 14.17, with the Indicated mineral resource blocks reporting to the green columns greater than or equal to 0.4 and the Inferred mineral resource blocks reporting to the lower blue columns less than 0.4. The resulting geographic distribution of blocks was simplified to avoid small outliers for each category. In practical terms of matching the SOR criterion to drillhole spacing, Indicated resources require a drillhole spacing less than or equal to 120 m and Inferred mineral resources are those blocks with drillhole spacing greater than 120 m but still able to be estimated using the current estimation parameters (see Section 14.2.5).

For this case, one can consider the ability to make effective local estimates with sufficient confidence in continuity to sufficient quality (SOR   0.4) to support a pre-feasibility study, as per the prescribed definition (CIM Definition Standards of 2005).

i ure 1 .1   Distribution of lope of e ression Parameter



15.0 MINERAL RESER E ESTIMATES

There are no Mineral Reserve estimates produced for this Project as defined by I 43-101 regulations.

1 .0 MINING METHODS

1 .1 SUMMARY

Tetra Tech developed an open pit, truck shovel mine plan for the Artillery Peak manganese deposit, involving mining an average 7,000 t d of mineral resource and 17,220 t d of waste over a 21-year LOM. The mine plan will include a haul backfill system for co-disposal of all waste products within previously mined-out areas of the open pit. The annual MM production is set at 50,000 t a, with maximum processing rate of 2.55 Mt a. For the duration of the LOM, mineral resource containing 1,106,000 t of manganese will be delivered to the mill, for a recovered total of 994,499 t of recovered MM.

The selected open pit contains 45 Mt of mineral resource with an average grade of 2.46 Mn, including an average allowance of 5 dilution. The average stripping ratio will be 2.46 requiring 111 Mt of waste to be mined. The quantity of mined material will vary by year, as shown in Table 16.5.

The 21-year LOM will be divided into four phases

- Phase 1 Years 0 (pre-production) through 6
- Phase 2 Years 7 through 10
- Phase 3 Years 11 through 15
- Phase 4 Years 16 through 21.

The various mining phases are discussed in greater detail in Section 16.5.2.

Two different rock types will be mined mineral resource with a specific gravity of 2.1, and waste with a specific gravity of 2.4. Most of the high-grade mineral resource located in the south area of the pit will be mined in the initial years of production (Phase 1). In addition to mining high-grade mineral resource, mining activities during the first mining phase will create space for an in-pit dumping area for co-disposal of tailing and waste rock, thereby reducing the external waste dump footprint. At the end of the LOM, mining will be limited to the centre-most and deepest part of the pit.

Tetra Tech considered stockpiling the low-grade mineral resource and processing it at the end of mine life; however, since this option provided negligible economic benefits, a stockpile was excluded from the mine plan. Both mineral resource and waste will be mined in 10 m benches. Where the manganese bed is smaller than 10 m, a smaller bench height was adopted, to minimize dilution.

The long axis of the final pit has an azimuth of 150°. The pit has a maximum length of 2,340 m, a maximum width of 660 m and a maximum depth of 250 m. Wall slopes are about 44° on the western side, which has no ramp. On the eastern side, the pit walls are formed by the footwall of the manganese deposit, which lies at an angle of less than 30°. All in-pit roads are located on the east wall.

1 .2 INTRODUCTION

The Project is located in northern Arizona, a relatively dry region with moderate terrain. The deposit consists of a series of manganese beds that vary in thickness and quality. Light coloured waste rocks both overburden and interburden can be easily distinguished from the mineral resource that contains dark manganese beds (Figure 16.1). This visual disparity will assist with grade control. The deposit is exposed on surface, and dips gently from a few degrees to up to 30° to the west. ase of access and moderate topography make the deposit ideal for open pit mining. A low amount of pre-stripping will be required to initiate mining production.

i ure 16.1 Dar Coloured an anese ed and i ht Coloured nterburden

1 .3 PIT OPTIMI ATION

Tetra Tech optimized the open pit using Gemcom Whittle software and Lerchs-Grossmann (LG) optimization algorithms. The pit optimization process defined the various pit shells for the resource model, completed in ovember 2011, which contains the most recent drillhole data. The input parameters that were used in the process of pit optimization were provided by AMI and other team members, including geotechnical and metallurgical groups. The input parameters are provided in Table 16.1.

Table 16.1 Pit Optimi ation Parameters

tems	nits	alue
Production Rate	t a (t a MM)	2,450,000 (50,000)
MM Price	lb	1.84
Process Recovery		90
SR Royalty as Percentage of MM Revenue		2
Operating Cost Mining	t mined	2.42
	t waste mined	2.30
Operating Cost Processing		for 2-6 Mn 0.763 lb MM 20.5 t milled
		for 2 Mn 0.763 lb MM 15 t milled
		for 6 Mn 0.763 lb MM 25 t milled.
Block Size	m	20 x 20 x 5
Density	t m^3	2.9 basalt, 2.4 sediment, 2.1 Mn one
Mining Recovery		95
Mining Dilution		5 with diluting material at 0.5 grade
Pit Slope Angles		44

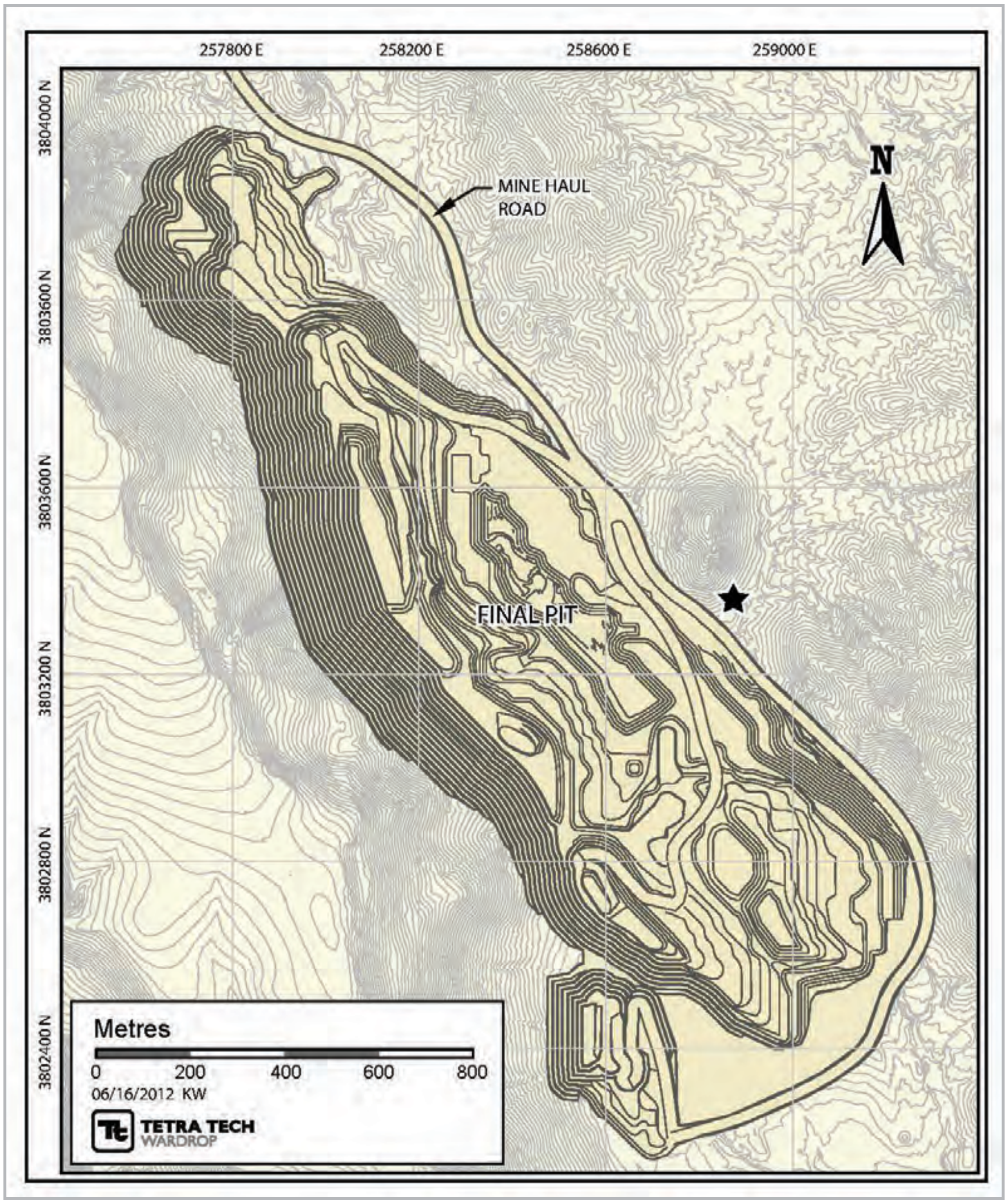
16.3.1 PIT PTIMI ATI I P TS

Pit design is sensitive to the inputs used in the process of the pit optimization. Therefore, in this section the most important parameters have been discussed and explained.

TOPO RAPHY

AMI provided topographic data as an AutoCAD (D F format) file, including a reference grid based on UTM coordinates. Figure 16.2 shows the topography with pit outline. The topography is generally of moderate relief, with higher grounds in the west and lower grounds in the south and east (Figure 16.3). levations range from as low as 510 m to as high as 780 m. ear the pit, the slopes run from northwest to southeast with an average grade of 3 . There are several manganese bed outcrops in the south and centre of the site. There was minor mining activity on the property on the 1940s and 1950s. A high wall located in the southwest portion of the deposit, a small waste dump, are legacy indicators of mining activity in this period (Figure 16.3).

i ure 16. General Topo raphy ith inal Pit Outline



i ure 16. General ie orth acin of the Central and orthern Part of the ite

16.3.2 M E S

Tetra Tech originally produced the block model using Leapfrog software. A text file was provided to the mining group in CSV format with all of the related information. This model was then converted to a format suitable for Gemcom Surpac software for mine design purposes. There is no rotation for the block model. The large axis of the block model is oriented south to north. The model contains regular blocks with 20 m x 20 m x 5 m dimensions. Information about the resource model can be found in Section 14. Table 16.2 summarizes the specifications of the block model, including its origin and extension.

Table 16. esource odel nformation

Type	Y		
Minimum Coordinates	3,801,530	257,410	452.5
Maximum Coordinates	3,804,270	259,350	782.5
User Block Size (m)	20	20	5
xtensions (m)	2,740	1,940	330
umber of Blocks	137	97	66
Rotation	0	0	0

Table 16.3 shows tonnages of Indicated resources with various manganese cut-off grades based on 20 m x 20 m x 5 m blocks. The model was developed by applying a cut-off grade of 0.9 Mn to the database. With a cut-off grade of 1 Mn, the model contains 62.2 Mt of Indicated resources at an average grade of 2.32 Mn. Table 16.3 shows the relationship of tonnage to grade obtained from the block model that was used in the pit optimization process.

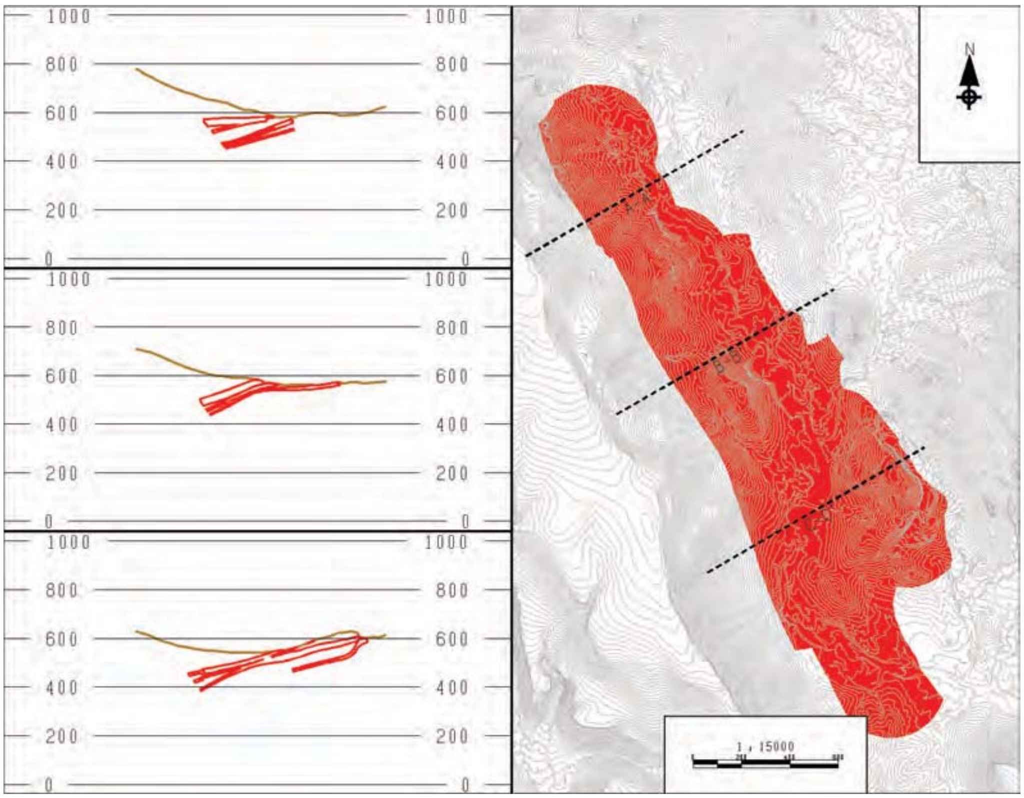
Table 16. Tonna es of ndicated esources ased on Different Cut-off Grades

n Cut-off	ndicated esource t	n t	n Av . Grade
0.1	65,667,340	1,473,838	2.24
0.2	65,667,340	1,473,838	2.24
0.3	65,667,340	1,473,838	2.24
0.4	65,667,340	1,473,838	2.24
0.5	65,658,318	1,473,797	2.24
0.6	65,506,339	1,472,944	2.25
0.7	65,194,715	1,470,889	2.26
0.8	64,373,055	1,464,697	2.28
0.9	63,387,937	1,456,306	2.30
1	62,201,129	1,445,047	2.32
1.1	60,929,877	1,431,733	2.35
1.2	59,598,142	1,416,414	2.38
1.3	58,031,774	1,396,806	2.41
1.4	56,198,391	1,372,001	2.44
1.5	53,781,580	1,336,899	2.49
1.6	50,808,488	1,290,768	2.54
1.7	47,655,587	1,238,721	2.60
1.8	44,448,502	1,182,606	2.66
1.9	41,176,517	1,121,980	2.72
2	37,688,226	1,053,885	2.80
3	10,705,559	395,120	3.69
4	2,487,719	115,814	4.66
5	490,957	28,096	5.72
6	140,767	9,342	6.64
7	28,627	2,109	7.37

The deposit consists of multiple beds, generally oriented southeast to northwest, with a long-axis orientation of 150° azimuth. The deposit dips to the southwest; the slope varies from a few degrees to less than 30°. A 3D model was interpreted and built for manganese mineralization beds using information obtained from drillholes and outcrops.

Figure 16.4 shows the 3D deposit model and three cross-sections located at the southern, central and northern areas of the model. The sections show how manganese beds vary in different areas both by dip and thickness. This deposit characteristic introduces challenges to mining operations that must be addressed by a good grade control program.

i ure 16. Deposit odel and Cross- ections



16.3.3 MA A ESE PRI ES R PIT PTIMI ATI

Twenty-four optimum pit shells were produced using a range of MM prices from 1.15 lb to 2.15 lb, in 0.046 lb increments. Pit 13 was selected for detailed pit design, production scheduling, and evaluation. Pit 13 corresponds to an average MM selling price of 1.70 lb, and contains sufficient mineral resources to support a 20-year mine life.

16.3. PR ESSI RE ER

In the pit optimization stage, a flat processing recovery of 90 for all grades was applied to the entire resource model.

16.3.5 ERA PIT S PES

o detailed geotechnical study has been completed on the property. Based on existing information, Tetra Tech recommends a universal inter-ramp angle of 44°, for all walls and directions. The east wall is generally formed by the footwall of the deposit; the angle of this wall will be less than 30°.

16.3.6 I TI A MI I RE ER

The mineral resource can be easily distinguished from waste rock by its dark colour, which will assist operations personnel with grade control and dilution reduction. The manganese deposit is formed in bed-like shaped structures; its thicknesses varies from a few metres to greater than 10 m in some places. A resource model has been constructed using regular blocks with a 5 dilution built into each block, assuming a background manganese value of 0.5 for waste rocks. For example, applying 5 dilution with 0.5 Mn waste rock to a block grading 2.26 Mn will reduce its grade to a grade of 2.18 Mn. The following formula has been used for the diluted grade calculation.

ed ade ade 0.05 0.5 1.05

Tetra Tech also assumed an overall mineral resource recovery of about 95 .

16.3. MI I A PR ESSI PERATI STS

Average mining operating costs of 2.30 t and 2.42 t were estimated for waste and mineral resource, respectively for the LOM. Processing operating costs vary based on feed grade as listed in Table 16.1.

16.3. T ER I P TS

A 2 royalty charge was applied to the manganese revenue.

16.3.9 PIT PTIMI ATI RES TS

The selected pit shell, Pit 13, provides the highest grade feed required for the desired nominal 20 year processing period at the lowest incremental ratio of waste to contained manganese. This pit shell, containing 46.4 Mt of mineral resource, was therefore selected for the detailed pit design production scheduling and evaluation.

A 5 dilution at 0 Mn grade was applied to all mineral resource blocks. There will be minor variations in the total amount of rock mined within the selected pit design to account for haul roads, catch berms, and minimum operating space criteria.

1 .4 FINAL PIT DESIGN

16. .1 E EI T A PIT A S PES

In February 2012, Tetra Tech completed a pit slope angle report-recommending bench and pit slope angles for the mine design as discussed in Section 16.3.5. Both waste and mineral resource will be mined in 10 m benches. For the final walls all benches are planned at a 10 m height. The face angle is designed using 75° slopes, with a minimum berm width of 8 m.

The eastern final wall is mainly formed by the deposit's footwall.

16. .2 MI IM M R I AREA

Most of the benches have been designed to accommodate a loader shovel and a 136 t truck. In the last bench at the bottom of the pit, there are areas where mining will be reduced to areas of 30 m x 30 m or less. Tetra Tech recommends the use of a hydraulic excavator to accommodate space limitations and reduce dilution.

16. .3 S MMAR T E TIMATE PIT ESI

Figure 16.5 shows a general view of the final pit and its dimensions. The trend of the ultimate pit is identical to that of the deposit, with an azimuth of 150°. The ultimate pit is 2,340 m long, with the width varying from 440 m to 660 m. The depth of the pit varies by section, the deepest part being in the centre where the high wall on the western side is 240 m tall. The northern and southern pit centres are shallower and provide greater ease of access. The lowest bench is located at 460 masl, and the highest at 710 masl.

i ure 16.5 inal Pit

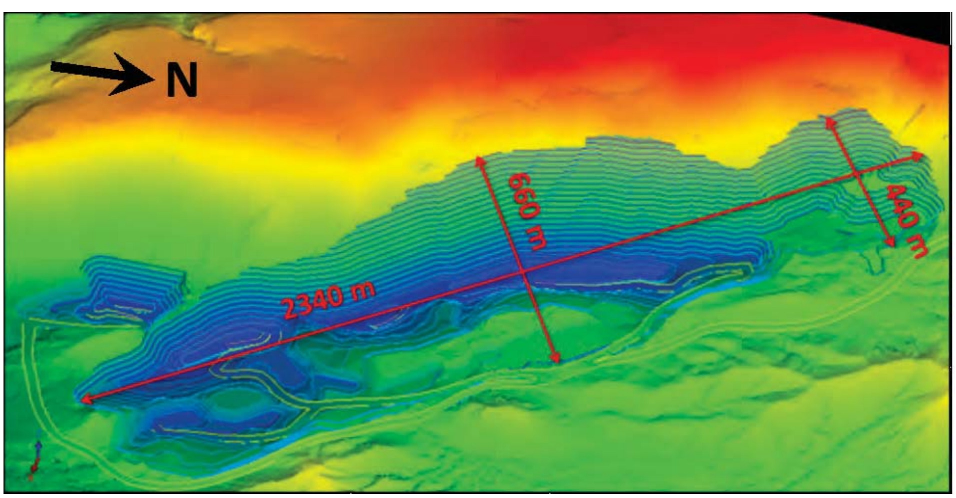


Table 16.4 summarizes tonnages and grades of mineral resource and waste mined from the final pit.

Table 16. ineral esource and aste ined

Type	Cate ory	uantity t	Grade n
Waste (Sent to Waste Dump)	Waste Rock	104,508,816	-
	Inferred	2,468,856	2.48
	Indicated below Cut-off	128,202	0.66
	Indicated Low Grade	3,816,852	1.28
	Total aste	110,9 , 6	-
Mineral Resource (Sent to Mill)	Indicated High Grade	39,413,640	2.58
	Indicated Marginal Grade	5,652,584	1.64
	Total ill eed	5,066,	. 6
Total ill eed and aste		155,9 ,950	
trip atio		. 6	

1 .5 PRODUCTION SCHEDULING

16.5.1 MI I PERATI A PR TI S E E

Tetra Tech developed a mining schedule that includes one year of preproduction, 21 years of mining operations, and three years of reclamation activities. Figure 16.6 shows the mineral resource production schedule, indicating the total mined waste, total mined mineral resource, and total MM produced in each year.

In Year 1 of mining operations, the mill is expected to operate at 75 capacity. The total amount of material mined will increase as the mine deepens. In early years of operation, mining capacity will not exceed 5.5 Mt. Mining capacity will gradually increase to a maximum of 9.5 Mt a in Phase 4 (Years 16 to 19). In the last year of mine life, mining capacity will decrease as the majority of the pushbacks for waste rock are completed. xtra capacity available in Year 22 will be used for reclamation.

Tetra Tech developed the mine production schedule to meet the following goals

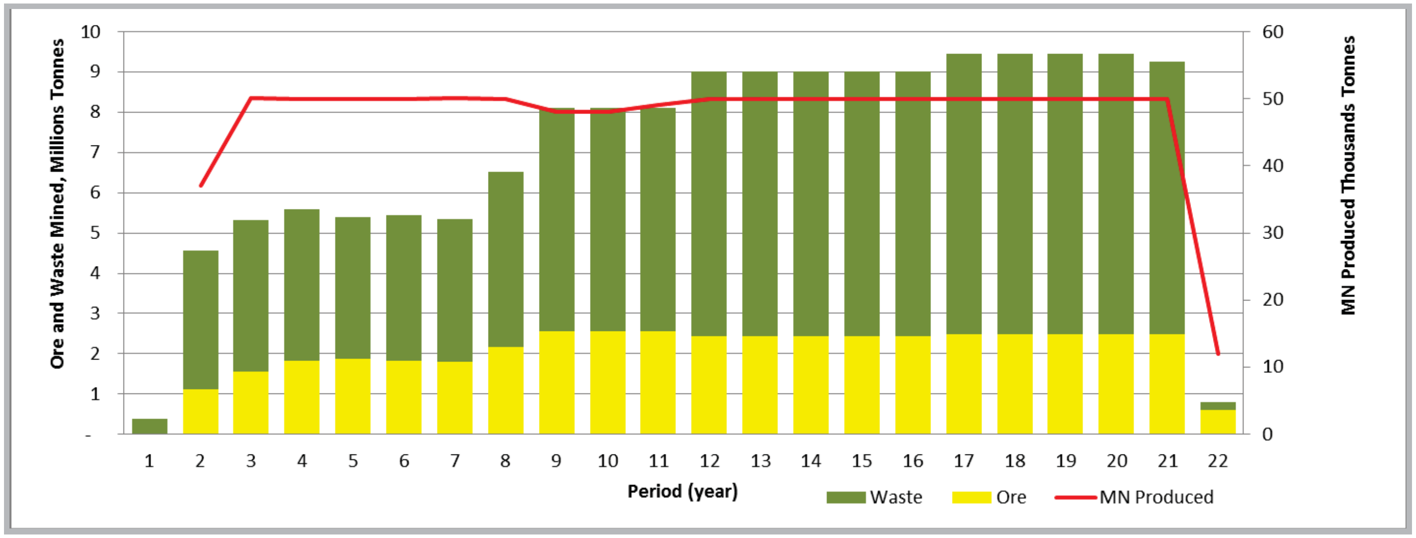
- produce about 50,000 t a of MM
- mining high-grade mineral resource as soon as possible
- improving the cash flow
- minimizing the stripping ratio
- backfilling both tailing and waste rocks into the pit.

The mine production schedule is constrained by the crusher's capacity of 2.55 Mt a and the maximum production of 50,000 t a of MM. In the current schedule, mineral resource with marginal grades (low grade) will be dumped in waste dumps rather than stockpiled. The maximum mining equipment fleet capacity required for the operation is about 9.5 Mt a.

Mineral resource will be mined in the direction of the strike of the deposit; mining activities will be restricted by topography and low-grade deposit zones. In some places, mineral resource will be mined using box cuts to minimize the stripping ratio and maximize mineral resource mining recovery. For these areas, smaller equipment will need to be employed.

Tetra Tech selected a mine plan that involves mining the higher-grade mineral resource in the first years of operation, rather than a directional, strip-mining method. Tetra Tech tested a directional, strip-mining method using Whittle software. The results indicated that grade and topography variations, made the strip-mining method less desirable than the common open pit method.

i ure 16.6 Production chedule



16.5.2 MI E P A S

In Year 0 (preproduction), mining activities will strip a total of 370,000 t of waste rock. A small stockpile of mineral resource is expected to accumulate before the crusher starts up; this stockpile will be placed close to the crusher, and for commissioning and testing purposes. There is a degree of flexibility in scheduling between the preproduction stage and Year 1 of operations.

The mine is scheduled in four phases. Phase 1 of operations encompasses 7 years, including the preproduction period. Phase 1 targets the high-grade area of the deposit, when the grade of mined mineral resource will vary from a minimum of 2.93 Mn in Year 4 to a maximum of 3.56 Mn in Year 1. During Year 1, the mill will operated at 75 of full capacity.

Phase 2 will be a 4-year period, from Year 7 through to the end of Year 10. During this phase, lower-grade mineral resource will be mined; some low-grade materials will be co-disposed with waste rock. The average grade of mineral resource mined in this phase will be 2.12 Mn.

Phase 3 will be 5 years in duration, from Year 11 through to the end of Year 15. The average grade of mined mineral resource for this period will be 2.29 Mn. In the last phase of mine life, from Year 16 to Year 21, mineral resource grade will average 2.26 Mn. Mining activities will end and reclamation activities will begin in Year 21, and end in 3 years. Waste dump reclamation will begin in Year 20, in order to minimize the footprint of the mine. All the material stored in the south external waste dump will be rehandled placed inside the pit.

Although the deposit is not geologically confined, it is assumed that no additional mineral resources will be discovered during the mine life that may be converted to mineral reserve. This is an important assumption because in this design most of the mined-out area will be backfilled with waste rock and tailings. There are areas inside and under the designed pit that contain Inferred materials; these may potentially be converted to Indicated resources with extra exploration.

Table 16.5 shows the mine production schedule for the preproduction period and 21 years of LOM. The cut-off grade varies. Based on the varying cut-off grades and tonnages available, mineral resource grades were divided into four different categories

- 0.85 Mn
- 1.5 Mn
- 1.75 Mn
- greater than 1.75 Mn.

Table 16.5 ine Production chedule

Type	Y0	Y1	Y	Y	Y	Y5	Y6	Y	Y	Y9	Y10	Y11
Mineral resource sent to Mill - (t)	13,198	1,126,568	1,553,407	1,816,773	1,868,909	1,815,992	1,811,650	2,163,485	2,555,000	2,555,000	2,555,000	2,442,834
Mn grade ( )	3.90	3.56	3.49	3.01	2.93	3.01	3.02	2.56	2.12	2.12	2.16	2.29
Recovery ( )	-	92.43	92.33	91.46	91.28	91.46	91.48	90.29	88.72	88.72	88.88	89.38
Total Produced t	-	,500	50,000	50,000	50000	50000	50,000	50,000	,000	,000	900	50,000
Waste sent to waste dump (t)	368,773	3,427,721	3,763,620	3,770,204	3,531,622	3,618,558	3,536,198	4,352,930	5,552,930	5,552,930	5,552,930	6,579,586
Total ined	1,9 1	,55 , 9	5, 1 ,0	5,5 6,9	5, 00,5 1	5, ,550	5, ,	6,516, 16	,10 ,9 0	,10 ,9 0	,10 ,9 0	9,0 , 19
Strip Ratio	-	3.04	2.42	2.08	1.89	1.99	1.95	2.01	2.17	2.17	2.17	2.69

	Y1	Y1	Y1	Y15	Y16	Y1	Y1	Y19	Y 0	Y 1	Total
Mineral resource sent to Mill (t)	2,442,834	2,442,834	2,442,834	2,445,486	2,474,766	2,474,766	2,474,766	2,474,766	2,474,766	590,595	45,016,228
Mn grade ( )	2.29	2.29	2.29	2.29	2.26	2.26	2.26	2.26	2.26	2.26	2.46
Recovery ( )	89.38	89.38	89.38	89.37	89.28	89.28	89.28	89.28	89.28	89.28	-
Total Produced t	50,000	50,000	50,000	50,000	50,000	50,000	50,000	50,000	50,000	11,900	99 ,500
Waste sent to waste dump (t)	6,579,586	6,579,586	6,579,586	6,579,586	6,987,343	6,987,343	6,987,343	6,987,343	6,787,343	200,000	110,863,063
Total ined	9,0 , 19	9,0 , 19	9,0 , 19	9,0 5,0	9, 6 ,109	9, 6 ,109	9, 6 ,109	9, 6 ,109	9, 6 ,109	90,595	155, 9, 91
Strip Ratio	2.69	2.69	2.69	2.69	2.82	2.82	2.82	2.82	2.74	0.34	2.46

All material grading below 1.5 Mn will be treated as waste and sent to dumps. Marginal-grade mineral resource (1.5 Mn to 1.75 Mn) will also be sent to the waste dumps if sending it to mill would exceed mill processing capacity. All mineral resource above 1.75 Mn will be sent to the mill.

The recovery percentage provided in Table 16.5 is based on the following formula

$$C \quad 0.0018^{3} - 0.0248^{2} \quad 0.1222 \quad 0. \ 224$$

where

e a e eed e e .

1 . ASTE ROC STORAGE

The Project will produce approximately 111 Mt of waste rock and 47 Mt of tailings, which will be co-disposed in the TWSF and ultimately returned to the mined-out pit in order to reduce the mine footprint.

In keeping with the mining sequence, one external waste dump southeast of the pit has been designed to accommodate the first six years of waste material, and another to the north to accommodate waste in later years. After Year 6, the co-disposed waste rock tailings material will be immediately backfilled to the north and south sections of the mined-out pit, while a clear mining zone in the centre of the pit is maintained. At the end of the LOM, all co-disposed waste will be reclaimed to the south and north in-pit waste dumps.

As a result of previous mining in the area and mineral resource outcroppings, overburden extracted in pre-production will be minimal and treated as waste rock. Low-grade mineral resource will not be stockpiled.

ASTE MP LOCATIONS

South E ternal aste ump

The south external waste dump will be located approximately 50 m southeast of the southern toe of the pit. Situating the waste dump in close proximity to mining operations for the first six years of mine life allows for short initial haul distances and minimum re-handling distances for final reclamation to the pit. The base of the dump skirts the south water pump runoff area, providing sufficient room for water collection, containment and flow off site. A waste rock berm will contain the co-disposed waste rock and tailings. By the end of Year 6, this dump will have a 33.5 Mt capacity, and a footprint of 609,677 m^2t will extend from the 525 m level to the 625 m level. The dump will then become dormant for the remainder of the LOM. The co-disposed waste is scheduled to be re-handled into the mined out open pit upon completion of MM production.

orth E ternal In-Pit aste ump

The north waste dump will be located 50 m north of the pit. This dump will store waste extracted from the north end of the pit, and will increase in size as this zone is developed. The footprint of this dump will extend into the pit as mining activities progress from the north and south ends towards the centre of the pit. Trucks will access the dump via existing external and temporary in-pit haul roads. By the end of the LOM, the total footprint of this dump will be 379,529 m^2; it will extend from the 480 m level to the 700 m level. Once mine production is complete, a portion of the material in this dump will be reclaimed to the north in-pit waste dump.

South In-Pit aste ump

The south in-pit waste dump will be created after Year 6, when waste material is backfilled into the previously-mined areas of the open pit. There exists a natural waste rock barrier between the high-grade mineral resource zone (mined in Years 1 through 6) and the main pit. This barrier will remain in place to ensure that waste material dumped into the south pit does not interfere with main pit mining activities. The waste dump will extend as south mining zones are completed and made available for backfilling. At the end of the LOM, the total south in-pit footprint will extend beyond the final pit rim by no more than 400 m to the south. The final waste dump footprint will be 811,444 m^2, and extend from the 460 m level to the 620 m level.

ASTE MATERIAL RECLAMATION

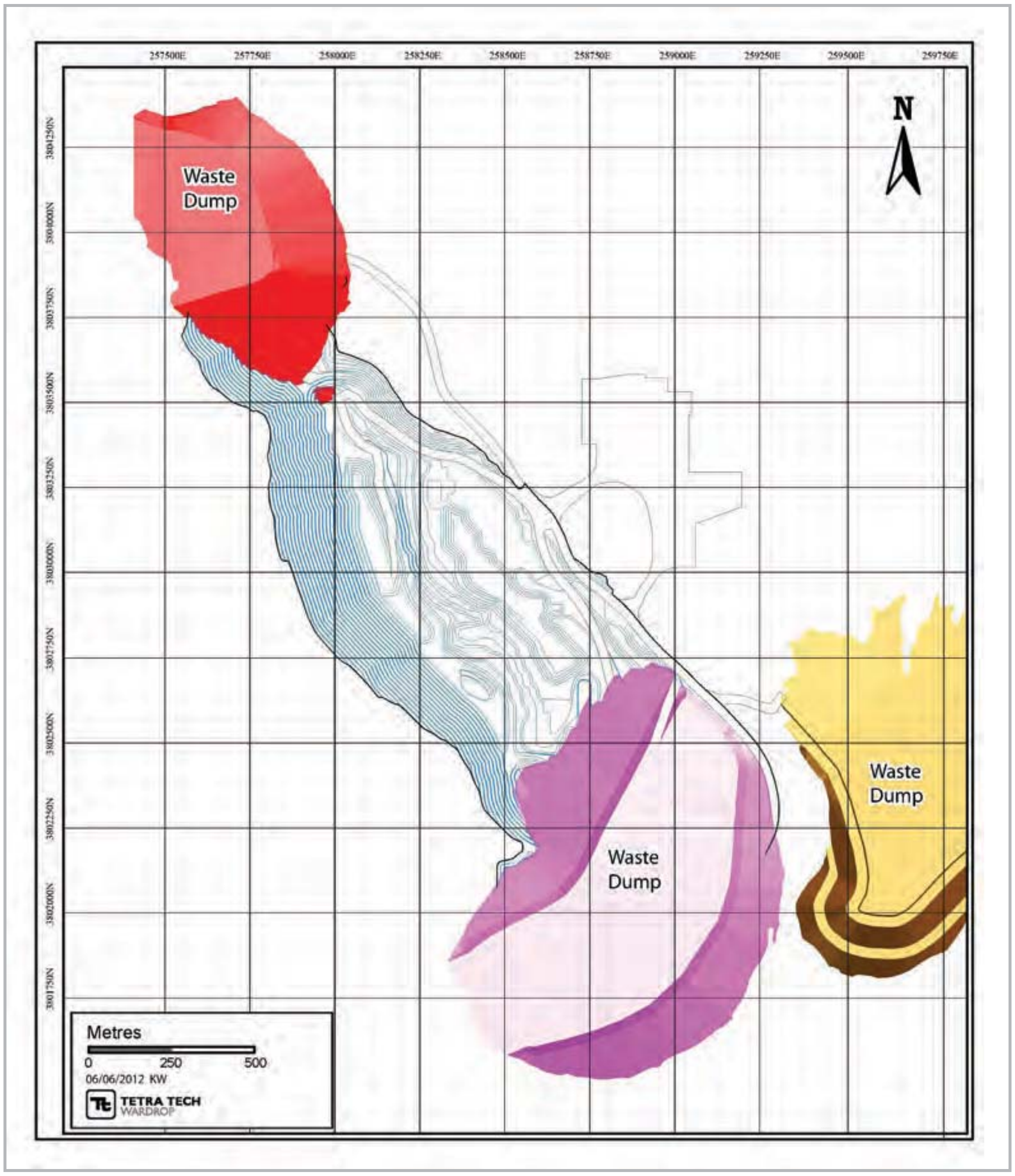
All waste material will be reclaimed to the pit once production is completed at the end of Year 21. This will increase the footprints of the north and south in-pit waste dumps to 457,475 m^2 and 1,066,923 m^2, respectively. The central section of the pit will remain clear of waste material to serve as a pool to collect water runoff (Figure 16.7 and Figure 16.8).

ASTE MP ESI N CRITERIA

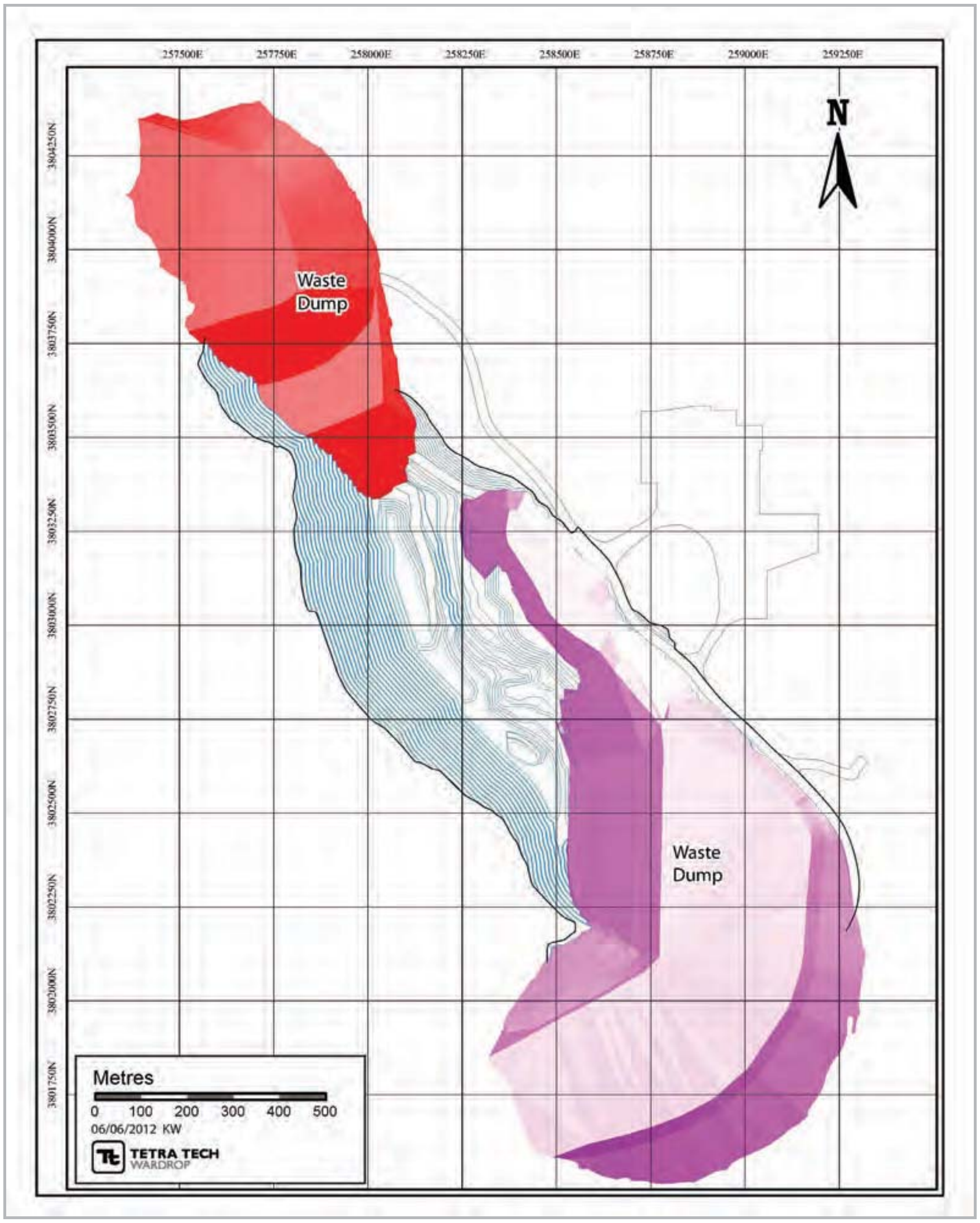
Surface waste dumps were designed to ensure interim slope stability for external dumps and long-term slope stability for all in-pit dumps. An average bulk density of 1.7 t m^3 was used for the co-disposed material waste dump volumetric design. This accounts for a 2 1 rock to tailings ratio, wherein the tailings reduce the porosity of the waste rock, thereby increasing the density.

The bottom-up external waste dump bench design consists of temporary lift heights of 50 m and berm widths of 30 m, with single lift slope angles of 38°, and a maximum height of 100 m. This ensures a factor of safety of 1.5 and an overall slope angle of 28°. After Year 6, and then again following mine closure, the in-pit waste piles will be dumped from the pit rim, and no interim berms will be necessary. These piles are designed with slope angles of 26°, below the angle of repose of the material creating a reclaimed slope angle of less than 50 (2H 1V) (Figure 16.9).

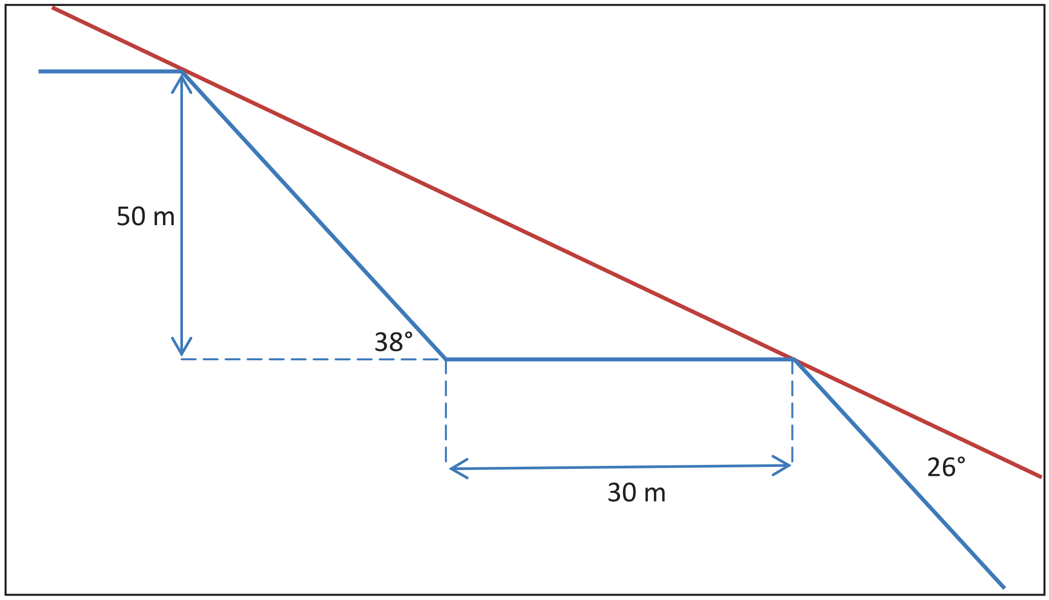
i ure 16. Plan ie of a imum aste tora e ootprint Year 1



i ure 16.   Plan ie of eclaimed aste Dumps



i ure 16.9 urface aste Dump lope Confi uration



1 . MINING E UIPMENT FLEET PRODUCTI ITIES

16. .1 E ERA

The proposed mining method is based on a conventional open pit operation, using trucks and shovels. All mining operations will be performed by the Owner.

The open pit will be accessed through two-lane, 31 m-wide roads or ramps, at 10 maximum grade, to accommodate 136 t class haul trucks.

quipment units were selected for operational flexibility, mineral resource grade control and variable bench height for mineral resource and waste. Off-highway rear-dump-haul trucks of 136 t capacity will be used for mineral resource, waste and tailings haul. One 11.0 m^3 diesel hydraulic face shovel will primarily be used for mineral resource mining; excess loading capacity will be used to strip waste. Two 11.5 m^3 wheel loaders will complement the hydraulic shovel in loading mineral resource, and waste and in-pit backfill. Mineral resource and waste rock material will require drilling and blasting.

The selected major and support equipment for mining operations are listed in Table 16.6 and Table 16.7, respectively.

Table 16.6 Avera e umber of a or inin uipment by Year

uipment	i e	Year								
		-1	1				5	6-10	11-15	16- 1
Diesel Rotary Drill	165 mm	1	1	1	1	1	1	1	1	1
Secondary Drill	114 mm	-	1	1	1	1	1	1	1	1
Hydraulic Shovel (diesel)	11 m^3	-	1	1	1	1	1	1	1	1
Wheel Loader	11.5 m^3	1	1	1	2	2	2	2	2	2
Haul Truck	136 t	2	2	3	3	3	3	4	5	5

Table 16. Avera e umber of ine upport uipment by Year

uipment	i e	Year								
		-1	1				5	6-10	11-15	16- 1
Track Dozer	306 kW	1	2	2	2	2	2	2	2	2
Track Dozer	231 kW	1	1	1	1	1	1	1	1	1
Motor Grader	193 kW	1	2	2	2	2	2	2	2	2
Water Truck	5,000 gal	1	1	1	1	1	1	1	1	1
Blast Stemmer	1.0 m^3	1	1	1	1	1	1	1	1	1
Small xcavator	2.3 m^3	1	1	1	1	1	1	1	1	1

16. .2 MA R MI E E IPME T ET PR TI E PERATI TIMES

The mine is expected to operate 365 d a, in two 12-h shifts per day. For all major equipment, the net operating hours (OH) were calculated using the following formula

H Ca e da H a a 1 a d e a a

Where

- Availability is the percentage of time that the unit is mechanically available (i.e. not at the truck shop).
- Standby is the percentage of time the unit is available but waiting for work.
- Use of Availability is the percentage of available time that the unit is doing productive work. Unproductive time includes all the various operating delays such as blasting, breaks, shift change, fuelling, shovel moves and weather delays.

Table 16.8 shows a sample of the OH calculations that were used for each piece of major equipment.

Table 16. ample O Calculations

Parameter	nits	alue
Calendar Days	d a	365
Scheduled Outages	d a	0
Shifts per Day	shifts d	2
Hours per Shift	h shift	12
Total Hours	h a	8,760
Availability		76
Available Hours	h a	6,658
Standby Idle		2
Gross Operating Hours	h a	6,524
Blasting	h shift	0.25
Breaks Meals	h shift	1.00
Shift Change	h shift	0.50
Miscellaneous (moves, fuel, power, cleanup)	h shift	0.25
Weather	h shift	0.33
Total Operatin Delays based on availability	h/shift	.
	h/a	1, 9
O	h/a	5,

1 . DRILLING

Blasthole drilling will be performed with one 165 mm diesel rotary drills as the primary drilling equipment. Blastholes will be drilled with 1.3 m of sub-grade, giving a total blasthole length of 11.3 m for the 10 m waste bench. A multipurpose crawler drill capable of drilling 127 mm holes will provide highwall sloping and secondary drilling.

16. .1 AST E RI PR TI IT

The primary drilling pattern for waste will use a burden of 5.6 m and a spacing of 6.7 m. Drill productivity was estimated by calculating the total time to drill one hole and move to the next one. The drill penetration rate was estimated at 60 cm min. Time was added for sampling, collaring and moving. The total required drilling hours were then determined by dividing the number of required holes by the drill cycle time per hole. Table 16.9 shows a sample of the productivity calculations for drilling in waste benches.

Table 16.9 ample Drill Productivity Calculations in aste

Parameter	nits	alue
Waste Production	t	6,579,586
Bench Height	m	10.0
Hole Diameter	mm	165
Burden	m	6.6
Spacing	m	9.9
Sub-grade Drilling	m	1.3
Hole Length	m	11.3
Stemming Length	m	4.6
Yield per Hole	bcm	523.0
SG of Ore	t bcm	2.40
Yield per Hole	t	1,255
umber of Holes Drilled	-	5,242
Annual Drilling Requirement	m	59,370
Penetration Rate	cm min	60
Moving, Align Time	min	2.1
Collaring	min	1.0
Grade Control Sampling Time	min	0.8
Total Cycle Time per Hole	min	22.7
Availability		85
Use of Availability		79
Maximum OH per Unit	-	5,783
Required OH	-	2,385
umber of Units Required	-	0.41

1 . BLASTING

16.9.1 E ERA ASTI ITI S R PR TI ES

An explosives supplier will supply and store explosives in a separate area northeast of the open pit. The supplier's office and service shop, the ammonium nitrate silos and the emulsion tank will be contained within an area secured by a fence.

An explosives magazine will house all detonators and primers. The magazine will be bullet-proof, fire-resistant and well-ventilated.

The Owner will contract an explosives supplier to provide all the necessary specialty equipment, including all bulk loading trucks, and to provide complete in-the-hole service. Bulk explosives will be delivered on a down-the-hole basis.

Blasting will be performed using bulk ammonium nitrate fuel oil (A FO) as the main explosive for 90 of the holes; emulsion will be used for the remaining holes that are expected to be wet. A blended powder factor of 0.11 kg t will be required.

The capital and operating costs of mobilizing and maintaining these facilities and equipment are included as part of direct annualized blasting operating costs.

1 .10 LOADING

16.10.1 E ERA A I ITI S

Primary loading equipment will include one 11.0 m^3 diesel hydraulic shovel, complemented by up to two 11.5 m^3 wheel loaders. The loading capacity will ensure mineral resource grade control and in-pit backfill capabilities as well as extra stripping capacity to provide continuous availability of exposed mineral resource.

16.10.2 A I PR TI IT

The maximum loading productivity was calculated based on the total loading time per truck, then de-rated to account for additional loading unit waiting time. The mining production rate will be constrained by the capacity of the truck fleet. Table 16.10 shows the productivity calculations for the loading equipment.

Table 16.10 ample aste oadin Productivity Calculations

Parameter	nits	ydraulic hovel	heel oader
Bucket Capacity (heaped)	m^3	11.0	11.5
Material Weight	dmt bcm	2.40	2.40
Bulk Factor		1.5	1.5
Material Weight	dmt lcm	1.60	1.60
Moisture		3	3
Material Weight	wmt lcm	1.65	1.65
Fill Factor		95	90
ffective Bucket Capacity	m^3	10.5	10.4
Tonnes Pass	wmt	17.3	17.1
Truck Size Capacity	wmt	136.0	136.0
Average umber of Passes		7.9	8.0
Truck Spot Time	sec	42	42
First Bucket Cycle Time	sec	30	42
Subsequent Bucket Cycle Time	sec	35	42
Load Time per Truck	min	5.22	6.28
Maximum Productivity	trucks h	11.5	9.6
	wmt h	1,565	1,300
Truck Availability to Shovel		90	90
ffective Productivity	wmt OH	1,408	1,170

1 .11 HAULAGE

16.11.1 E ERA A I ITI S

Primary hauling equipment will consist of 136 t haul trucks. The 136 t truck fleet will reach a maximum of six units in Year 15, when haul distances will reach over 2 km.

All mineral resource will be hauled to the primary crusher located at 570 masl. After dumping the mineral resource, the truck will proceed to the tailings loadout facility located due east of the crusher. An automatic loadout system will load the tailings into the truck. The tailings material is then hauled to the waste dump for disposal with waste rock. For the first five years of production, the round trip cycle to haul mineral resource and tailings material averages approximately 27 min. The round trip haul cycle will reach a maximum of 33 min in Year 16, when the tailings material is hauled for approximately 2.4 km to the waste dump.

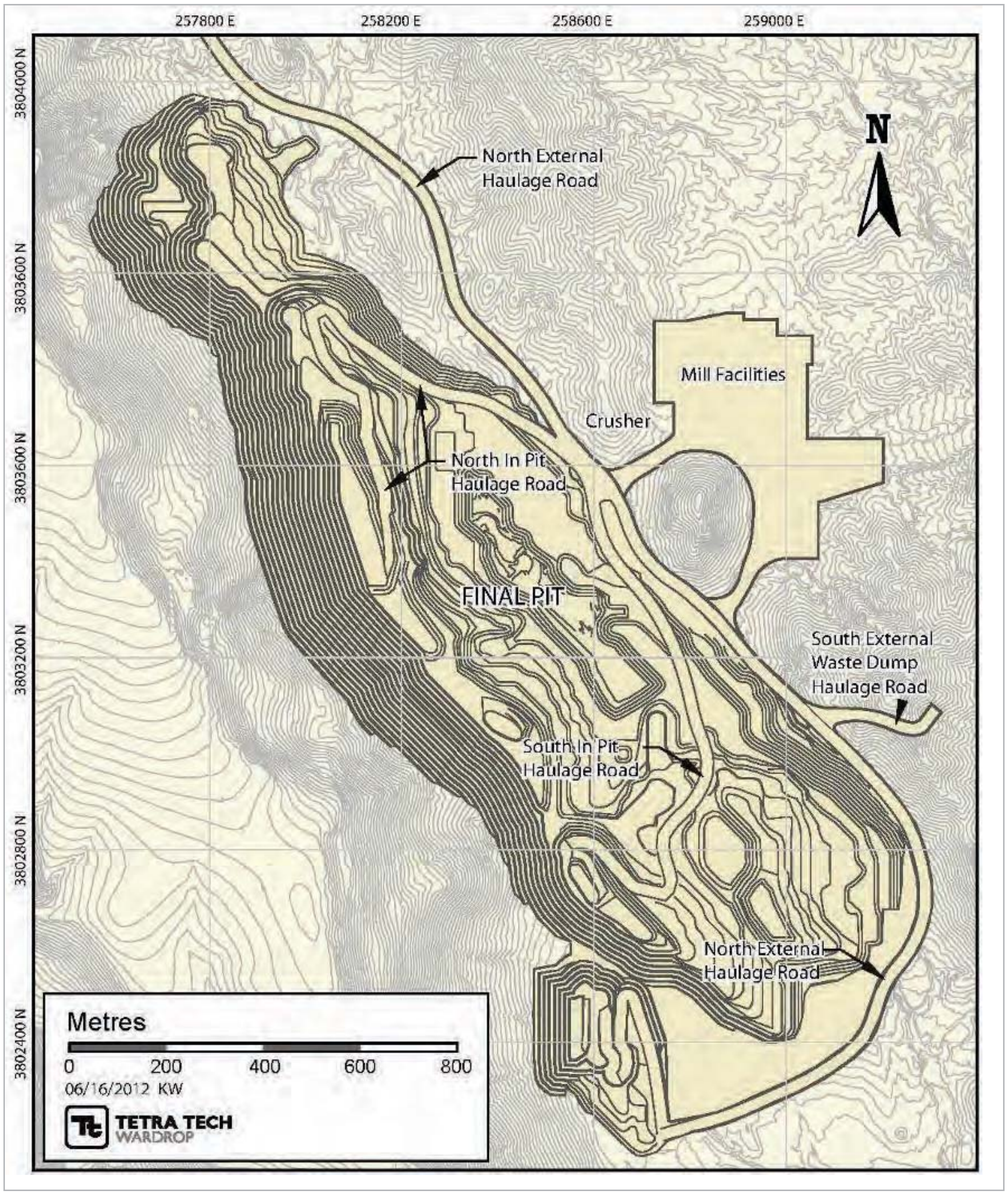
The waste material will be hauled to the two surrounding waste rock storage areas (WRSA) and will exit the pit at strategically positioned exit points. From each of these exit points, two 45 m lifts of the WRSA can be built. Overall, the WRSAs are to be built bottom-up, and much of the initial waste is required at lower elevations on the WRSA to develop the initial stabilizing lifts. Whenever possible, the elevation difference between source and destination was minimized to shorten the haul distance.

16.11.2 A R A S

Based on early production planning and equipment selection, Tetra Tech assumed that 136 t trucks (CAT 785D trucks, with an operating width of 7.1 m, or equivalent) will haul both mineral resource and waste from the pit. Tetra Tech calculated an optimum haul road width of 31 m for two-way traffic, and 21 m for one-way traffic. The overall haul road gradient will vary from 8 to 10 , and for very limited lengths may be greater than 10 . For most stretches of the road, the grade will be approximately 10 .

To minimize waste rock mining, all haul roads are designed on the east wall, where the pit dips to the west on the footwall of the deposit. The final pit has two distinct bottoms in the south and north-central zones. The haul road entrance to the pit will be situated in the centre from the east, which will provide easy access to both the south and north pit extents, to minimize haul distances. All the facilities, including the crusher, will be located near the haul road entrance. Figure 16.10 shows a plan view of the pit and haul roads.

i ure 16.10 aul oad ayout



A primary external haul road will extend along the eastern length of the pit, providing easy access to all mining activities. It will curve around the pit at the northern and southern ends to facilitate reclamation dumping from the pit rim. The main drainage ditch for on-site water control and collection will run alongside the main haul road. A 150 m road to the crusher will separate the main haul road into the 12.65 km long north and 1.91 km long south sections. This road will then loop around towards the mill, joining up with the main haul road 350 m further south. There will be a small access road of 265 m to the south external waste dump.

While there is no current need for a road on the west side of the pit rim, one may be required as mine planning and scheduling activities advance in future stages.

To minimize the number of switchbacks, some roads were designed to lie on low-grade ore and or Inferred materials. Should the blocks in these areas be upgraded from Inferred to Indicated categories, they could be recovered through excavation and then with waste.

16.11.3 A TR PR TI IT

To determine haul truck productivity, haul profiles were digitized for all possible source-to-destination combinations by year of mining production. Once the production schedule was finalized and the amounts of material for each source-destination combination were known, average cycle times per period were calculated.

Cycle times in minutes for the full round trip were estimated based on using haul trucks with a payload capacity of 136 t. Rimpull curves and speed bins were generated using Caterpillar's fleet production and cost (FPC) haul truck simulation tool. Rolling resistances used were 3 , 4 and 5 for the ramp routes, bench haul and waste dump hauls, respectively. Loading, dumping and spotting, and waiting times were estimated at 5.22, 1.0 and 0.5 min, respectively; these are fixed times that were added to the mineral resource haul cycle time.

16.11. A S E E

Haul profiles up to Year 6 all have only one dump location the south external waste dump. After Year 6, two waste dumps will be utilized interchangeably, depending on the mining location and haul distances. Haul distances to waste dumps in the final years of mining will be greatly reduced, since waste rock will be dumped to the north and southern in-pit zones as mining continues mid-pit.

Figure 16.11 through Figure 16.14 show the progressive development of the pit roads throughout the LOM.

Figure 16.11 Year 6 Select Haul Profiles

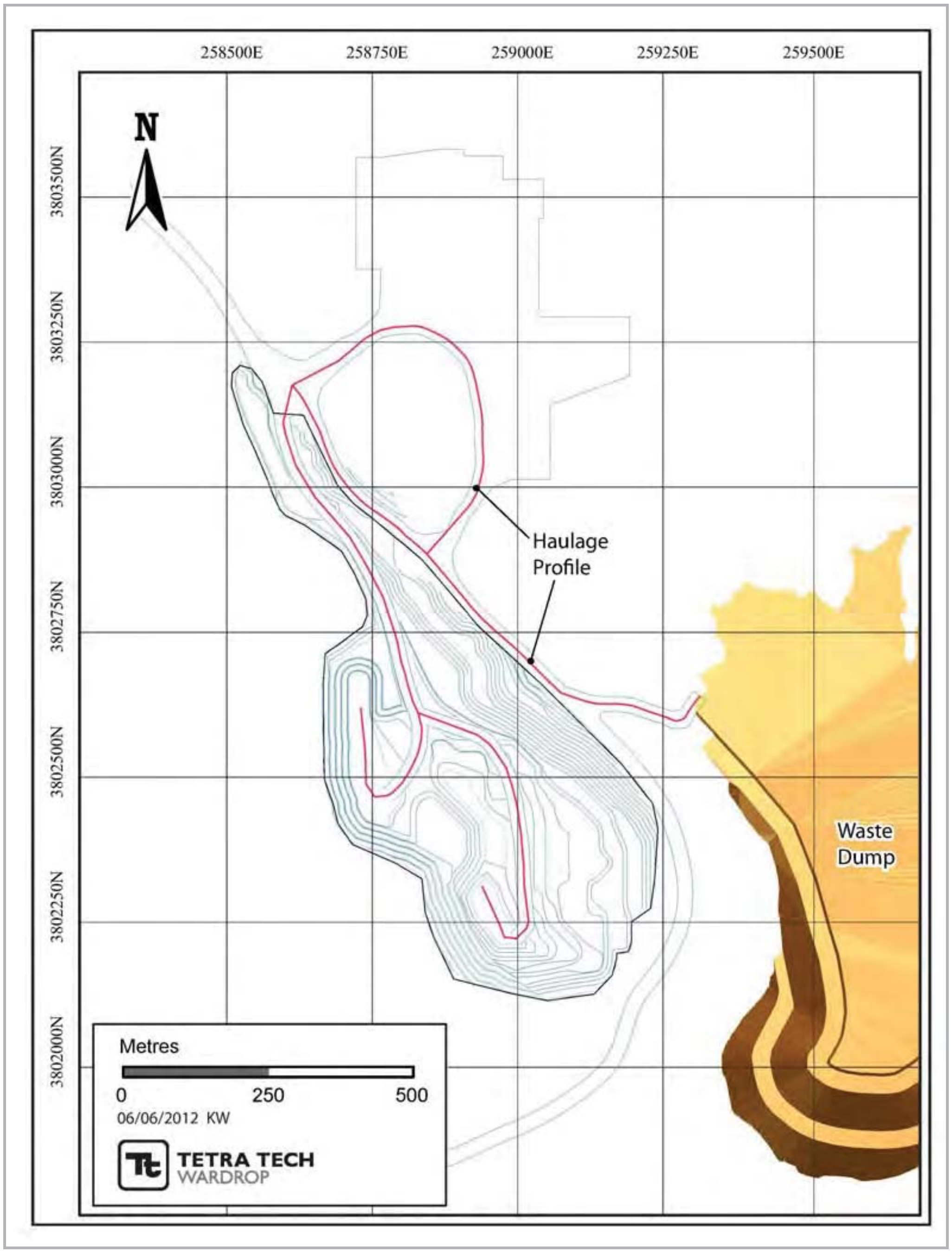


Figure 16.12 Year 7 Select Haul Profiles

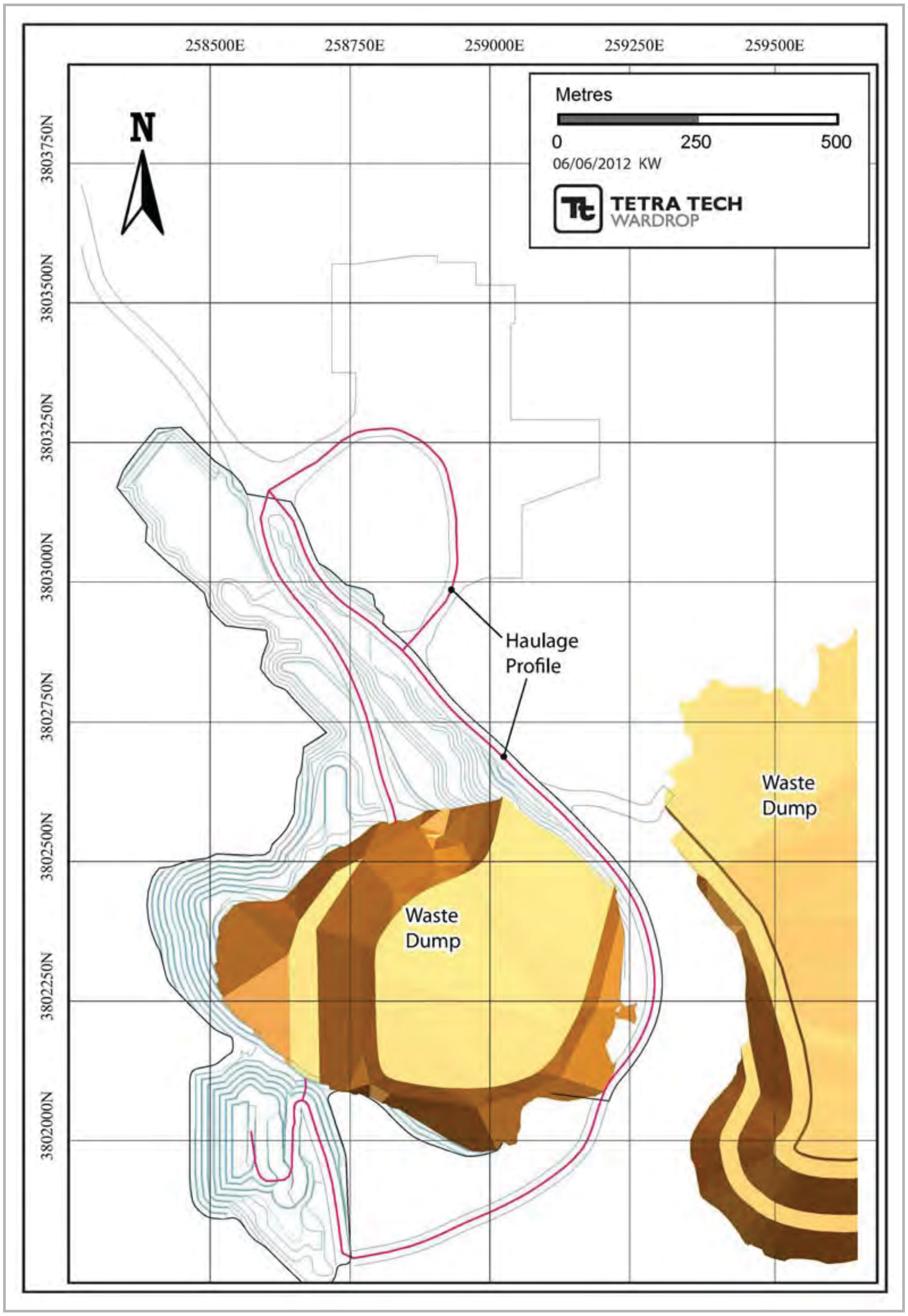


Figure 16.13 Year 15 Select Haul Profiles

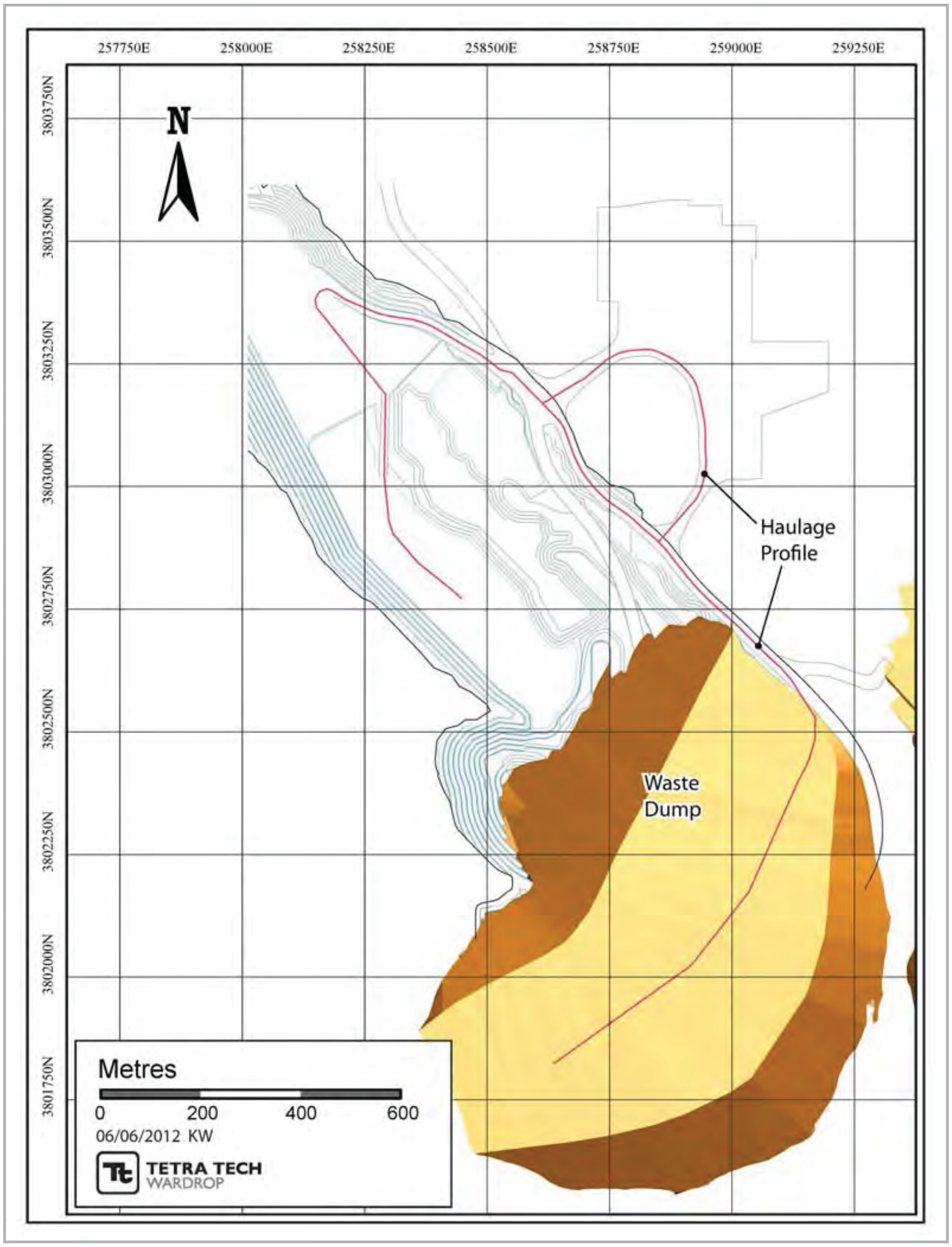
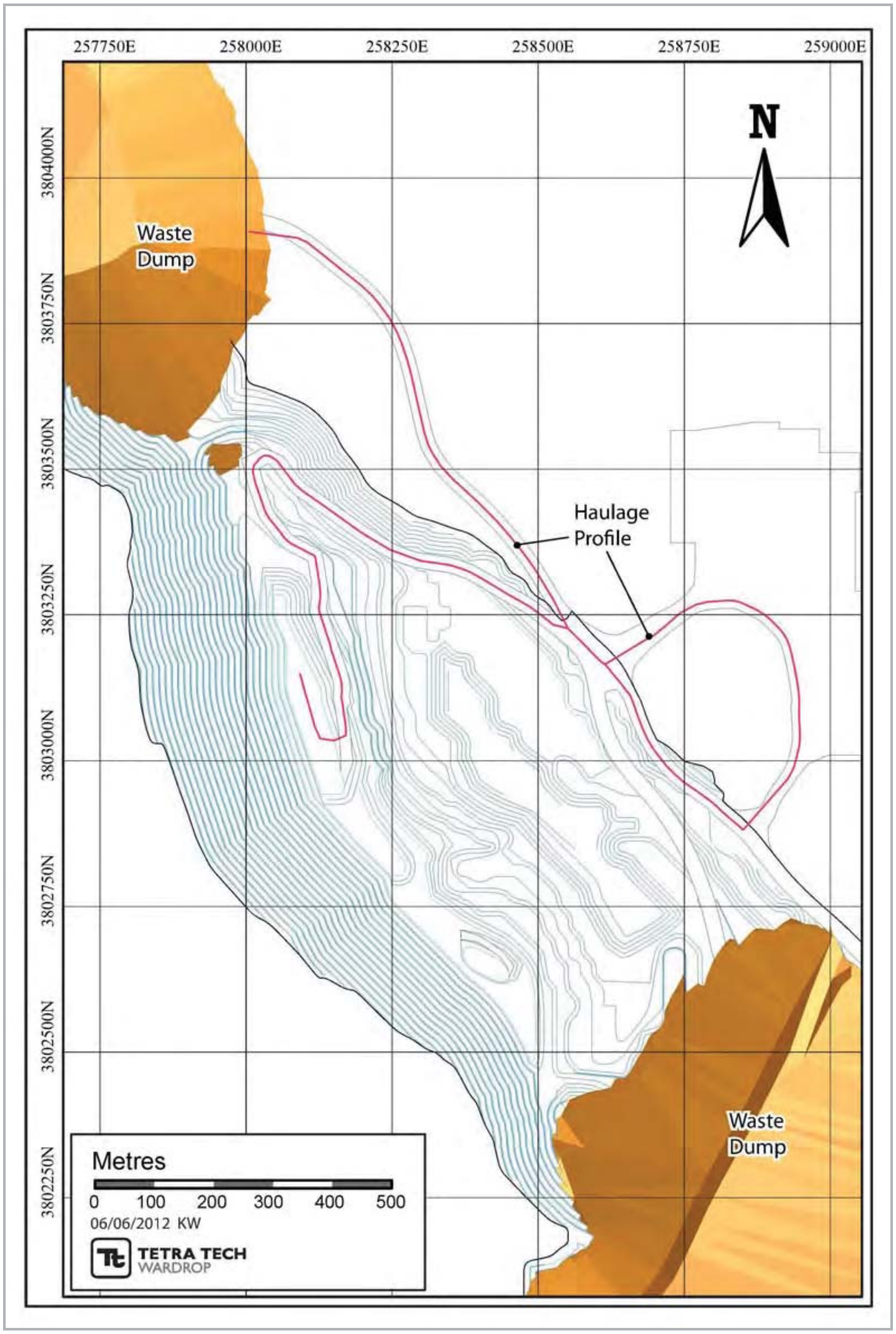


Figure 16.14 Year 20 Select Haul Profiles



1 .0 RECO ERY METHODS

1 .1 INTRODUCTION

The mineralization of the Project is in the form of four-valent manganese in pyrolusite (manganese dioxide), psilomelane (complex hydrous manganese), and wad manganese (hydrated manganese oxide). It is the largest known low-grade deposit of manganese in the southern United States. The mineralization is friable and soft, and amenable to reductive leaching at a coarse particle size of approximately 80 passing 2 mm. According to historical testwork results, Kemetco Research Inc. (Kemetco) further investigated the reductive leaching process using sulphur dioxide and other related processes. The testwork findings are summarized in Section 13.

The process and process conditions developed by Kemetco establish the design basis for the manganese extraction process for the project. This section outlines the metallurgical process, including major design criteria and process descriptions.

1 .2 SUMMARY

The proposed processing plant is designed to process mill feed containing 2.2 to 3.5 Mn in the range of 3,500 t d to 7,000 t d, to produce 50,000 t a of MM at higher than 99.7 Mn purity and an overall recovery of 88 to 93 Mn. Anhydrous sodium sulphate will be produced as a byproduct.

On average, the LOM mill feed grade is estimated to be 2.46 Mn, and will range from 2.12 Mn to 3.56 Mn, depending on the mining year.

The ROM mill feed will be trucked from the proposed open pit to the crushing facility located at the plant site. The proposed Project process will include the following main process facilities

- two stages of crushing to the top size of 5 mm
- reductive leaching circuit using sulphur dioxide
- leach residue washing and dewatering
- pregnant leach solution purification to remove metal and other impurities
- manganese carbonate precipitation and filtration to remove sodium sulphate and sodium dithionate
- precipitated carbonate re-dissolution in anolyte spiked with sulphuric acid

- selenium-free manganese electrowinning circuit
- nanofiltration and evaporative crystallization for anhydrous sodium sulphate production.

The proposed Project process will include the following main ancillary facilities

- Liquid sulphur burning and sulphuric acid (H_2SO_4) production system, which will produce sulphur dioxide (gas) and sulphuric acid for the manganese leaching and downstream operations. The heat generated from sulphur burning will be recovered for power generation and used for processing, such as mechanical vapour recompression (MVR) evaporation.
- nvironmental management that includes emission controls, water conservation and reagent recycling, and tailings disposal.
- Two stages of crushing will reduce the ROM feed to a particle size of 100 passing 5 mm. The first stage of the crushing will use a sizer crusher in an open circuit. The second stage of crushing will use an impact crusher in a closed circuit with a dry screen. The product from the crushing circuit will be stored in two 7,000-t surge bins prior to being conveyed to the leaching circuit.

The crushed mill feed will be leached in two stages pre-conditioning leaching with acidic solution containing sulphuric acid (mainly recycled from downstream washing circuits); and sulphur dioxide reductive leaching. The insoluble manganese in the 4 oxidation state can be readily reduced to the soluble 2 oxidation state by adding sulphur dioxide. Both leaching agents sulphur dioxide and sulphuric acid will be generated from liquid sulphur on-site.

The leached slurry after aeration will be directed to the counter-current decantation (CCD) circuit, where the manganese-bearing pregnant leach solution (PLS) will be separated from the leach residue. The leach residue will be subjected to multiple stages of CCD washing, followed by dewatering in pressure filters to reduce water consumption and water content of the residue for waste rock residue co-deposit. The washed leach residue will be back-hauled by truck to the excavated area within the open pit and blended for co-disposal with waste rock.

The PLS will be treated by two stages of purification to remove any impurities (such as iron, aluminum, zinc, nickel) which may have detrimental effects on the downstream operations.

The purified PLS containing mainly manganese sulphate and manganese dithionate is directed to the manganese carbonate precipitation circuit. The precipitation of manganese carbonate is achieved by mixing the purified PLS with sodium carbonate. In the precipitation process, soluble sodium sulphate and sodium dithionate are effectively eliminated from the manganese carbonate product. The resulting manganese carbonate precipitate will then be separated from the solution by thickening and filtration. The sodium sulphate and sodium dithionate solution will be sent to the nanofiltration MVR evaporation system to recover sodium sulphate and water.

The manganese carbonate solids will be dissolved by the acidic spent electrolyte solution produced from the manganese electrowinning circuit. This solution is further purified in two stages to remove any impurities and to generate the manganese electrolyte feed for subsequent use in the MM electrowinning process.

The electrolyte feed (neutral manganese sulphate) will be pumped into the electrowinning cells where the anode and cathode compartments are separated by semi-permeable diaphragms. At the cathode, Mn^{2} will be reduced to $Mn^{o}(s)$ and deposited onto the cathode plate. Depending on process conditions, cathodic hydrogen gas and anodic oxygen or manganese dioxide will be generated, respectively (hydrogen gas and anodic oxygen or manganese oxide), due to competing side reactions.

Manganese metal obtained at the steel cathode plates will have an overall purity of over 99.7 Mn. The deposited manganese will be treated by washing and passivation prior to being peeled from the cathode plate. The manganese metal flakes will be transported to the manganese stock silo, and then bagged for shipment.

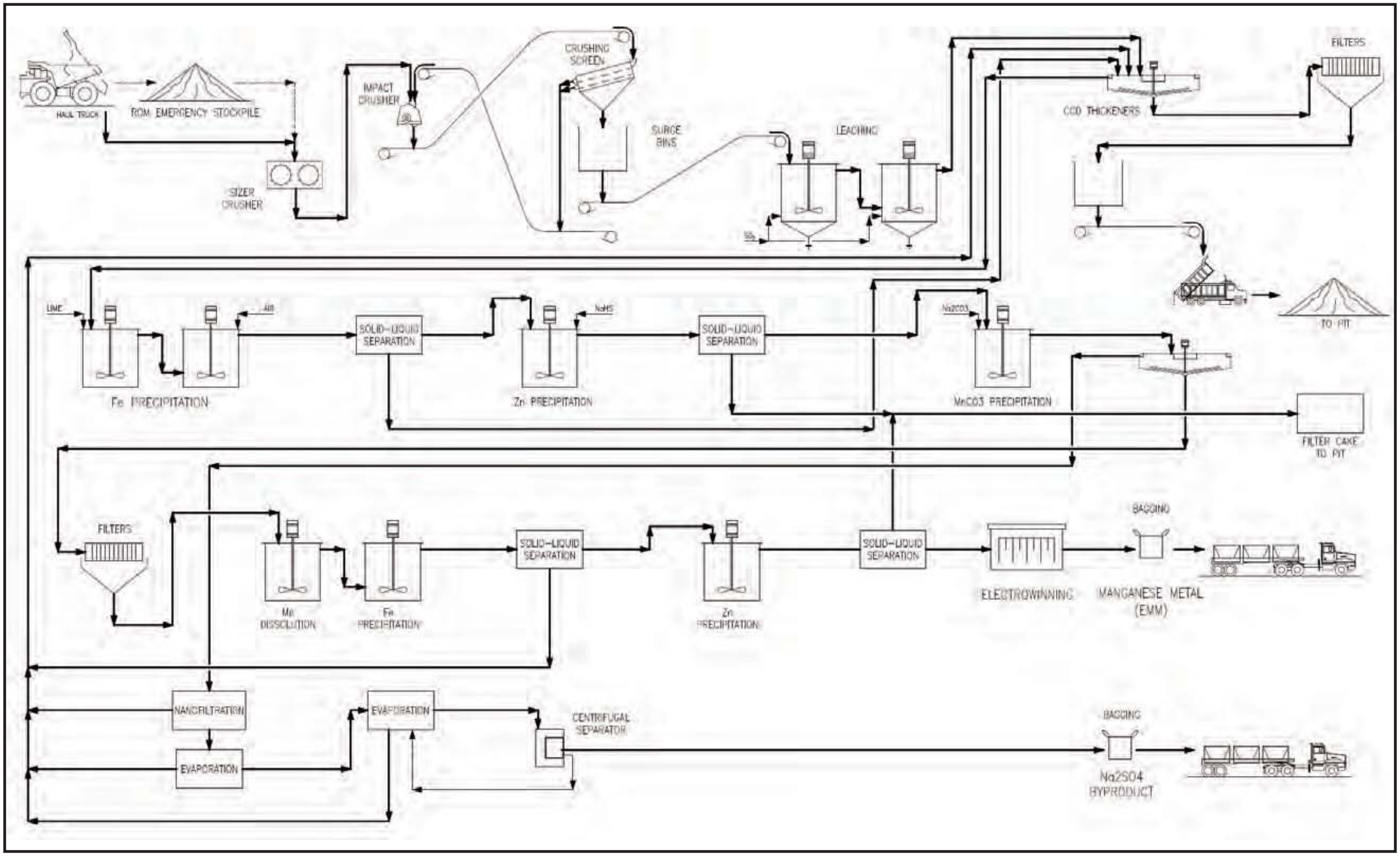
The sodium sulphate and sodium dithionate solution from the manganese carbonate precipitation circuit will be further processed to recover the sodium sulphate produced as a byproduct and water, which will be reused as process water. The process will include

- nanofiltration pre-concentration
- MVR evaporation and crystallization to produce partially hydrated sodium sulphate and sodium dithionate solids
- solid liquid separation by centrifuges
- calcination of the solids to produce anhydrous sodium sulphate and recover sulphur dioxide that can be recycled back to the reduction leaching circuit.

The heat generated from the liquid sulphur burner will be recovered and used to generate electricity by a steam turbine, and or provide direct heating in the form of intermediate steam, as needed in the process.

The simplified process flowsheet is shown in Figure 17.1.

i ure 1 .1 implified Processin lo sheet



1 .3 MA OR DESIGN CRITERIA

The processing plant is designed to process mineralized material at a rate ranging from 3,500 to 7,000 t d (calendar). The major criteria used in the design are outlined in Table 17.1.

Table 1 .1 a or Desi n Criteria

Criteria	nits	alue
General Desi n Criteria		
Operating Days per Year	d	365
Plant Overall Availability		92
Head manganese grade (average)		2.26
Average Process Rate		
Crushing	t h	417
Leaching Dewatering	t h	317
Purification lectrowinning	t a MM	50,000
Manganese metal content		99.7
Leaching		Pre-conditioning leaching Reductive Leaching with SO_2
PLS		Two Stages of Purification
Manganese Carbonate Precipitation		By sodium carbonate (a_2CO_3)
Manganese Re-dissolution		By sulphuric acid
a_2SO_4 Recovery		anofiltration MVR Calcination
lectrowinning Circuit		
Manganese Concentration Cell Feed Solution	g L	35 40
Manganese Concentration Catholyte Solution	g L	14 16
Manganese Concentration Anolyte Solution	g L	12 14
Cathode Current Density ominal	A m^2	300 400
Anode Current Density ominal	A m^2	450 650
Cell Voltage ominal	V	4.8 5.2
Feed Catholyte Solution pH		7.0 7.5
($H_4)_2SO_4$ Concentration	g L	125 150
SO_2 (g) Concentration	g L	0.1
Operating Cycle	h	24
Cell Operating Temperature	C	35 40

1 .4 PROCESS PLANT DESCRIPTION

The process to obtain MM from the Artillery Peak mineralized material essentially consists of these main steps

- crushing of the ROM mineralized material to 80 passing 2 mm
- reductive leaching of the crushed mineralized material
- purification of the PLS
- precipitation of manganese carbonate
- dissolution of manganese carbonate, followed by further PLS purification
- electrowinning of the purified manganese sulphate PLS solution
- sodium sulphate byproduct and process water recovery
- disposal of tailings paste as in-pit backfill co-disposed with waste rock.

The crushing, leaching and residue dewatering are designed based on the nominal process rate of 7,000 t ROM mineralized material per day, while the precipitation of manganese carbonate, electrowinning and downstream circuits are based on an annual production rate of 50,000 t MM.

1 . .1 R S I

A simple crushing circuit is designed to reduce the friable and clayey feed to a particle size suitable for downstream agitating leaching operation.

The ROM material will be transported from the open pit to the crushing facility by 136-t hauling trucks. The crushing circuit includes two crushers designed to process 417 t of mineralized material per hour to a particle size of 80 passing finer than 2 mm.

The ROM mineralized material is discharged onto a stationary grizzly. Material finer than 500 mm will flow by gravity into the crusher feed surge bin, and then be fed the sizer crusher at a controlled speed. The crushed material from the primary crusher will report to the secondary crushing feed bin, which will also receive the oversize of the secondary crushing screen. An impact crusher equipped with a 750 kW motor will be used for the secondary crushing to further crush the primary crusher discharge. The impact crusher will be in a closed circuit with a vibrating screen. The screen oversize material (coarser than 5 mm) will be recycled to the secondary crusher feed surge bin, while the screen undersize product will be directed by conveyors to two 7,000-t surge bins.

A dust collection system will be installed to control the spread of fugitive dust generated during the mineralized material crushing and transport at the area.

Magnets are provided over the surge bin feed conveyors to remove any tramp steels to protect the downstream crusher and conveyors.

In addition, a space at the primary crushing feed pad is provided for ROM mineralized material emergency stockpiling.

The main equipment and facilities in the crushing circuit are

- one sizer crusher (112 kW)
- one hydraulic rock breaker
- one secondary crusher feed bin (10 m^3)
- one secondary crusher belt feeder
- one impact crusher (750 kW)
- one vibrating screen
- two fine mineralized material surge bins, each 7,000-t capacity
- various belt conveyors.

1 . .2 EA I

The crushed mineralized material from the fine mineralized material surge bins will be leached in two stages

- pre-conditioning with acidic wash solution (recycled CCD washing solution and scrubber sludge)
- reductive leaching with sulphur dioxide.

The leaching agents used will be sulphuric acid and sulphur dioxide (SO_2 (gas)). Both the sulphur dioxide and sulphuric acid will be generated on site by burning liquid sulphur.

After leaching, the slurry will be directed to the CCD circuit for solid-liquid separation.

PRE CON ITIONIN LEACH

The crushed mineralized material will be pre-leached in a stirred tank with the recycled washing solution from the CCD circuit and the sludge from the off-gas scrubber. The make-up water contains the manganese recovered from the leach residue washing. The pre-leach agitation will further break down aggregated particles prior to the reductive leaching. The leach tank will be covered, and a ventilation system will be provided to direct any leaked gas to the scrubbing system, which will service the entire leaching area.

RE CTIVE LEACH ITH S LPH R IO I E

The reductive leaching circuit consists of a bank of four stirred and enclosed tanks. The slurry from the pre-leaching circuit will gravity flow to the first reductive leaching tank. Sulphur dioxide gas will be sparged into the leach tanks to reduce Mn^{4} in the form of manganese dioxide to soluble Mn^{2} in the forms of manganese sulphate and manganese dithionate. The leach tanks will be equipped with oxidation-reduction potential (ORP) probes to control reduction reactions.

The sulphur dioxide depleted off-gas from the first leaching tank will be directed to the two-stagelime scrubbing system to remove residual sulphur dioxide, while the off-gas from the rest of the leaching tanks will be forced to the preceding leaching tank. The scrubbed gas will be discharged into the air, and scrubbed slurry will be pumped to the pre-leaching tank.

The leached slurry will be pumped to the downstream CCD circuit for solid-liquid separation.

Key equipment components of the leaching circuit are

- one acid pre-leach tank (4.5 m diameter x 5 m high) and associated agitator
- four reductive leaching tanks (6.5 m diameter x 6.5 m high) and associated agitators
- one two-stage scrubbing system.

1 . .3 EA RESI E E ATERI

CC THICKENIN FILTRATION

The solid-liquid separation will occur in the CCD train, which consists of eight high-rate thickeners, each having a diameter of 36 m. The slurry from the reactive leaching circuit will report to the first thickener feed well, where the slurry will be mixed with flocculant solution. The overflow of the first thickener, or manganese-bearing PLS, will leave the CCD circuit and report to the PLS purification circuit.

The thickener underflow, containing approximately 55 solids, will be pumped to the second thickener feed well, where the slurry will be mixed with the overflow of the third thickener. The thickener overflow of the second thickener will be used as make-up water for the pre-leaching of the crushed mineralized material. The underflow of the second thickener will be pumped to the third thickener feed well for further washing. Similarly, the third thickener underflow will be sent to the next washing thickener feed well, while the third thickener overflow flows to the preceding thickener as washing solution.

The freshwater or recovered clean water will be added to the last thickener feed well as washing media. The washing ratio will be 2 1 (freshwater solids). The underflow from the last thickener will be neutralized by lime and pumped to the residue filtration feed surge tanks.

All the thickeners will be capped with covers to prevent any residual sulphur dioxide gas spreading into the surrounding environment. An exhaust pipe will guide residual sulphur dioxide gas away from the thickeners to the gas scrubbing system.

The underflow from the last stage of thickening will be pumped to two of the pressure filter feed stock tanks. Ten 1,400-m^2 frame-plate filter presses are proposed to further dewater the thickener underflow from approximately 55 solids to 67 solids.

The filtration cake will be discharged onto the filter discharge conveyors, and from there onto the cake out-loading conveyor, which will transport the cake to the out-loading surge bin. An automatic out-loading system will be incorporated to automatically load the cake onto the trucks which are used to deliver the mineralized material to the crushing facility from the open pit. The leach residue will be hauled back to the pit and co-disposed with waste rock in the excavated area within the pit.

The equipment used for the filtration and out-loading systems includes

- eight 36-m diameter high-rate thickeners
- two residue filter stock tanks (each 8.5 m diameter x 9 m high)
- ten residue filter presses (1,400 m^2 each) and associated pumps
- one residue filter cake load-out conveyor (1 m wide x 59 m long)
- other associated conveyors and feeders.

1 . . PRE A T EA S TI P RI I ATI E RE MA A ESE AR ATE PRE IPITATI

The PLS from the first CCD thickener will be purified prior to the manganese precipitation using sodium carbonate. The purification will be completed in two stages, each stage having a separate reactor and clarifier.

a a e e iron, aluminum, arsenic and suspended solids, including silicon dioxide, will be precipitated by aeration, and solution pH will be increased to 6 with the addition of hydrated lime in a mixing reactor.

The PLS will be treated by aeration with air and neutralized by lime milk to a pH higher than six in two reaction tanks. The aeration will oxidize soluble ferrous iron to less soluble ferric iron. The treated PLS will be pumped to a 12 m clarifier to separate precipitates from the solution. Flocculant will be added to assist the settlement of the precipitates. The clarifier underflow will be directed to a tube filter where the underflow is further dewatered. The resulting filtrate from the tube filter will be recycled to the clarifier, and the solids will be sent to the leach residue CCD circuit. The clarifier overflow is sent to the second stage of precipitation.

a a e The clarifier overflow from the first stage of precipitation will be further treated to remove soluble heavy metals. The treatment will include heavy metal sulphidation and solid-liquid separation.

Sodium hydrosulphide will be added to the sulphidation reaction tank where the heavy metals, such as n ^{2} ions, will be precipitated out in the form of sulphides. The reacted solution will be pumped to a 12 m clarifier. A tube filter will further dewater the underflow of the clarifier. The filtrate will be returned to the clarifier, while the retained solids will be solidified with cement prior to disposal.

The clarifier overflow or the treated PLS, will pass through a polishing tube filter to remove any solid impurities. The resulting PLS, containing mainly manganese sulphate and manganese dithionate solution, will be directed to the manganese carbonate precipitation stage.

The main equipment in the PLS purification circuits are

- three 6.5 m diameter x 5.5 m high reaction tanks and associated agitators
- two 12 m clarifiers
- two clarifier tube filters
- one PLS solution polishing tube filter.

1 . .5 MA A ESE AR ATE PRE IPITATI

The precipitation of Mn^{2} from the PLS solution will be achieved by mixing the purified PLS with sodium carbonate in two reaction tanks, each equipped with an agitator.

Most of the manganese will be precipitated in the first reactor, and the balance will react with sodium carbonate in the second reactor. The SO_4^{2-} and $S_2O_6^{2-}$ associated with the Mn^{2} in the PLS will be released and react with a to produce sodium sulphate and sodium dithionate. The solid manganese carbonate will be separated from the sodium sulphate- and sodium dithionate-bearing solution in a thickener.

The sodium sulphate- and sodium dithionate-bearing overflow will be sent to the sodium sulphate and sodium dithionate stock tank for recovering sodium sulphate produced as a byproduct and water.

The thickener underflow will then be dewatered by a pressure filter (with provision for abbreviated washing) to produce a higher-grade manganese carbonate cake for MM production feed.

The main equipment used for the manganese carbonate precipitation circuit are

- two manganese precipitation tanks (5 m diameter x 5.5 m high) and associated agitators

- one 10 m manganese carbonate thickener
- one manganese carbonate slurry stock tank (5 m diameter x 5 m high) and associated agitator
- two 76 m^2 manganese carbonate press filters
- one manganese carbonate storage bin, equipped with a manganese carbonate discharge screw feeder.

1 . .6 MA A ESE AR ATE ISS TI A PRE A T EA S TI P RI I ATI A TER MA A ESE AR ATE PRE IPITATI

The manganese, in the solid form of manganese carbonate, will be re-dissolved, and the resulting PLS solution will be subjected to further purification to remove impurities prior to the manganese electrowinning circuit.

The solid manganese carbonate precipitate will be re-dissolved with the spent electrolyte recycled from the electrowinning cells. Fresh sulphuric acid will be added to completely dissolve the precipitate, as required. The carbon dioxide gas generated from the dissolution will be directed away from the reaction tanks, and scrubbed or discharged into the air. The dissolution will be carried out in two stages in two agitated tanks, each 2 m in diameter x 2.5 m high.

Prior to the electrowinning, the resulting manganese PLS will be further treated in two stages of purification

- Stage One aeration and precipitation by addition of ammonium hydroxide
- Stage Two residual heavy metal sulphidation by adding sodium hydrosulphide.

After treatment, the PLS will be mixed with flocculant and the solids produced will settle in a 9 m clarifier. The clarifier underflow will be filtered. The filter cake will be sent to the cement mixer, where it will be mixed with cement prior to disposal. The filtrate will be pumped back to the clarifier feed well.

The clarifier overflow will pass through a carbon column to further remove any impurities that may adversely affect the electro-deposition of manganese, or have detrimental impacts on the quality of the   MM produced. Then, the cleaned solution will report to the manganese sulphate PLS stock tank, which is 36 m in diameter and 15 m high. The PLS solution that will feed to the electrowinning process will contain approximately 35 to 40 g Mn L.

1 . . E E TR I I

The electrolyte solution from the manganese sulphate PLS stock tank will be used for the electro-deposition. There will be three electrowinning lines each line will consist of one hundred and sixteen 6 m long x 1.48 m wide x 1.47 m high electrolytic cells made from polymerized vinyl compounds. The cathodes will be 1,000 mm high x 660 mm wide 316 stainless steel plates, and the anodes will be 900 mm long x 600 mm wide solid rolled lead silver alloy sheets. Cathodes and anodes will be separated by semi-permeable diaphragms.

The PLS solution will be introduced to the cathode zones. A portion of the manganese will deposit onto the cathode surfaces. Hydrogen gas will be generated on the surfaces of the cathodes, due to competing side reactions. The main reductive reactions at the cathodes are shown below

Mn^{2} $2e^{-} \rightarrow Mn$ .... (eq.17.1)
$2H$ $2e^{-} \rightarrow H_2$ .... (eq.17.2)

The manganese-depleted solution from the cathode zones will pass through diaphragms and enter the anode zones. Oxygen gas will be generated on the surface of the anode sheets. Mn^{2} may be oxidized as manganese dioxide precipitates, depending on operating conditions. The key oxidation reactions at the anodes are shown below

Mn^{2} $2H_2O$ $2e^{-} \rightarrow MnO_2$ $4H$ .... (eq.17.3)
$2H_2O$ $4e^{-} \rightarrow 4H$ O_2..................................... (eq.17.4)

The spent acidic anolyte solution will leave the electrolytic cells and enter the anodic solution collecting sumps, which will pump it to an anodic solution stock tank. The anodic (or barren) solution will be filtered to remove any solid impurities produced during the electrolytic process. The filtered barren solution will sent to the manganese carbonate dissolution circuit. The solids from the filtration will be leached with sulphur dioxide to dissolve manganese dioxide, and the resulting slurry will be pumped to the PLS solution purification circuits ahead of the manganese carbonate precipitation circuit.

Three separate rectifier systems will provide direct current for the three electrolytic lines. The rectifier system will include three 20 MW rectifier transformers and three 16.5 MW rectifiers.

The main operating parameters are listed below

- Mn Concentration Catholyte Solution .... 14 16 Mn g L
- Cathode Current Density nominal....................... 300 400 A m^{2}
- Anode Current Density nominal.......................... 450 650 A m^{2}
- Cell Voltage ominal .... 4.8 5.2 V
- Cell Operating Temperature .... 35 40°C
- Feed Catholyte Solution pH.................................. 7 7.5

Sulphur dioxide gas will be introduced to the electrolytic cells to improve electrolytic efficiency. As a buffering agent, ammonium sulphate (H_4SO_4) will be used to control electrolytic solution pH, and foam will be added to control mist generation on the surface of the electrolytic cells. Cooling water will be circulated through coils installed inside of the electrolytic cells to control the cell temperature. The cooling water will be in a closed circuit with two cooling towers. A ventilation system will be provided to direct gas generated from the electrolytic process to a scrubbing system.

The local control system will monitor and control cathode current density, cell voltage, solution pH, solution flowrate and bath temperature.

The deposition period of manganese ions will be approximately 23 to 24 hours. After the required deposition period, the cathodes will be removed from the cells and replaced with fresh cathode plates. The deposited cathodes will be subjected to multi-stage treatment prior to the deposited manganese being peeled. ach electrowinning line will have four post-deposition plate treatment systems, each with automatic plate washing, surface passivation, washing, drying and peeling, where the deposited manganese metal will be stripped from the cathode plate. The cathode plates will be further treated by polishing before they are reused in the electrowinning circuits.

Manganese metal stripped from the cathode plates would have an overall purity of higher than 99.7 Mn. The manganese metal will be conveyed to the MM product stock silo, from where it will be bagged in 1 t sack bags or custom sized bags prior to shipment.

The main equipment used in the electrowinning circuit and the deposited cathode plate peeling circuits are

- 348 electrowinning cells (6,000 mm long x 1,480 mm wide x 1,470 mm high) in three lines, each line116 cells)
- three 20 MW rectifier transformers
- three 16.5 MW rectifiers
- twelve automatic plate washing, surface passivation, washing, drying and peeling systems
- six cathode plate polishing machines
- two electrowinning cooling towers
- conveyors and one bagging machine.

1 . . A R S S I M S P ATE PR TI A ATER RE ER S STEM

The solution generated from the precipitation of manganese carbonate contains soluble sodium sulphate and sodium dithionate. This solution will be pumped to the sodium sulphate recovery facility to recover sodium sulphate produced as a byproduct and water. The processes used for the recovery will consist of

- nanofiltration
- two stages of evaporation by MVR
- solid-liquid separation
- solid calcination.

The solution generated from the precipitation of manganese carbonate will be forced through a nano-membrane filter. The filtration permeate will be reused as process water in the manganese leaching and CCD circuits. The retained sodium sulphate and sodium dithionate, which have been concentrated, will be further concentrated in the first stage of MVR evaporation, followed by additional MVR evaporation during which they will be crystallized. The vapour from the MVR evaporation will be condensed and reused as process water.

The sludge from the MVR process will be dewatered by a centrifugal separator to separate the crystals from the sludge. The solution produced will be recycled back to the second stage of MVR. The solid product, which contains sodium sulphate decahydrate and sodium dithionate dehydrate, will be calcined in two stages to convert these to sodium sulphate and water. In the second stage of the calcination, at 267^{o}C, sulphur dioxide will be released from the decomposition of sodium dithionate dehydrate. The released sulphur dioxide gas will be used for the manganese reductive leaching.

The sodium sulphate byproduct will be bagged prior to shipment off site.

The main equipment used in the facility will include

- two 550 m^2 nanofilters (vendor package)
- one MVR crystallization system (vendor package)
- one sodium sulphate calciner
- one centrifugal separator.

1 . .9 REA E T A I A ST RA E

The main reagents used for the project will include liquid sulphur, sodium carbonate, lime, flocculants, ammonium sulphate, and sodium hydrosulphide. To ensure containment in the event of an accidental spill, the reagent preparation and storage facility will be located within a containment area designed to accommodate 110 of the volume of the largest tank. The storage tanks will be equipped with level indicators and instrumentation to ensure that spills do not occur during normal operation. Appropriate ventilation, fire and safety protection, and material safety data sheet (MSDS) stations will be provided at the facility.

ach reagent line and addition point will be labelled in accordance with Workplace Hazardous Materials Information Systems (WHMIS) standards. All operational personnel will receive WHMIS training, along with additional training for the safe handling and use of the reagents.

S LPH R RNIN SYSTEM AN PO ER ENERATION

The sulphur dioxide gas and the sulphuric acid required for the process will be generated by the burning of liquid sulphur, which will be transported to the site by tanker trucks.

The liquid sulphur is burnt, or oxidized, to produce sulphur dioxide. Most of the sulphur dioxide gas will be sent to the manganese reduction leaching circuit, while the rest of the sulphur dioxide gas will be used to produce sulphuric acid in an integrated sulphuric acid plant. The sulphur dioxide will be catalytically oxidized to sulphur trioxide, and absorbed to produce sulphuric acid. The heat from the sulphur burner will be recovered and used for a steam turbine to generate electric power.

Storage tanks will be provided to receive liquid sulphur and to store the sulphuric acid produced.

SO I M CAR ONATE

Sodium carbonate in solid form will be dissolved and diluted with water in a mixing tank, and stored in a 1.5 m wide x 2.5 m high tank before being added to the addition points by metering pumps.

LIME

Lime will be delivered by bulk tanker trucks and stored in a dedicated silo with a capacity of 350 m^3. Lime will be retrieved from the silo by a screw conveyor and slaked in a tower mill. The slaked lime slurry, with 15 solids, will be stored in an agitated tank and distributed throughout the processing plant via a pressurized lime loop. The lime will mainly be used for off-gas scrubbing and leach residue neutralization.

FLOCC LANT

Two types of solid flocculant will be used for this project. They will be shipped in 25 kg bags, prepared in separate wetting and mixing systems, diluted to 0.5 strength and stored separately in holding tanks. The flocculant solutions will then be fed to the thickener feed wells by metering pumps.

AMMONI M HY RO I E

Ammonium hydroxide (25) will be added to the process circuits via individual metering pumps. This reagent will be delivered in bulk containers.

SO I M HY ROS LPHI E

Solid sodium hydrosulphide will be shipped to the mine site in drums or bags, diluted with water to the desired solution strength (approximately 10 to 20) in a mixing tank, and then stored in a holding tank before being added to the various addition points by metering pumps.

AMMONI M S LPHATE

Solid ammonium sulphate will be shipped to the mine site in drums or bags, diluted with water to the desired solution strength (approximately 10 to 20) in a mixing tank, and then stored in a holding tank before being pumped to the addition points by metering pumps.

OTHER REA ENTS

Anti-scale chemicals, as required, will be added to minimize scale build-up in the reclaim or recycle water lines. This reagent will be delivered in liquid form and metered directly into the process water line.

1 . .10 ATER S PP

Two separate water supply systems, for freshwater and process water, will be provided to support the operations.

FRESH ATER S PPLY SYSTEM

Freshwater will be supplied from off-site wells and pumped to a 7 m diameter by 11 m high freshwater storage tank, from which water will be distributed to the mine plant site and to the potable water storage tank. The potable water will be treated (by chlorination and ultraviolet lamps) before it is delivered to various service points.

Freshwater will primarily be used for the following

- firewater for emergency use
- reagent preparation
- dust suppression
- potable water supply

- process makeup water.

The freshwater tank is designed to be full at all times, and will provide at least two hours of firewater in an emergency. The freshwater supply source and system is described in Section 18.

PROCESS ATER

Process water will primarily consist of recovered water from both the leach residue dewatering treatment and the sodium sulphate water recovery system, augmented by freshwater as needed. Freshwater and recovered water will be directed to a process water storage tank, from where the water will be pumped to the processing plant.

1 . .11 AIR S PP

Plant air service systems will supply air to the following areas

- crushing circuit high pressure air for air seal, dust collection and maintenance
- leaching circuits purification high pressure air by dedicated air compressors
- filtration circuit high pressure air for filter pressing and drying of the leaching residue by dedicated air compressors
- electrowinning plant service air high pressure air for various services by two dedicated air compressors
- sodium sulphate recovery circuit high pressure air for various services
- instrumentation instrument air for the plant site will come from the plant air compressors and will be dried and stored in a dedicated air receiver.

1 . .12 ASSA A META R I A A RAT R

The assay laboratory will be equipped with all necessary analytical instruments to provide routine assays for the mine, process and environmental departments.

The metallurgical laboratory will undertake all necessary testwork to monitor metallurgical performance and to improve the process flowsheet and efficiency.

1 . .13 PR ESS TR A I STR ME TATI

The plant control system will consist of a distributed control system (DCS) with PC-based operator interface stations (OIS) located in the plant site central control room and the electrowinning plant local control room.

The plant central control room will be staffed by trained personnel 24 h d.

In addition to the plant control system, a closed-circuit television (CCTV) system will be installed at various locations throughout the plant, including various facilities and conveyor discharge points. The cameras will be monitored from both the local control room and the central control room.

Process control will be enhanced with the installation of an automatic sampling system, which will collect samples from various streams for on-line analysis and daily metallurgical balances.

For the protection of operating staff, a sulphur dioxide monitor alarm system will monitor the sulphur burning sulphuric acid production plant and reductive leaching areas.

1 . .1 META PR TI PR E TI

Projected annual metal production, according to the metallurgical performance projections developed by Kemetco based on test results and the proposed mine plan, is shown in Table 17.2.

Table 1 . etal Production Pro ection

Year	Y1	Y	Y	Y	Y5	Y6	Y	Y	Y9	Y10	Y11
Mill Feed, kt	1,140	1,553	1,817	1,869	1,816	1,812	2,163	2,555	2,555	2,555	2,443
Head Grade, Mn	3.56	3.49	3.01	2.93	3.01	3.02	2.56	2.12	2.12	2.16	2.29
Recovery, Mn	92.4	92.3	91.5	91.3	91.5	91.5	90.3	88.7	88.7	88.9	89.4
Mn Produced, t	37,500	50,000	50,000	50,000	50,000	50,000	50,000	48,000	48,000	48,900	50,000

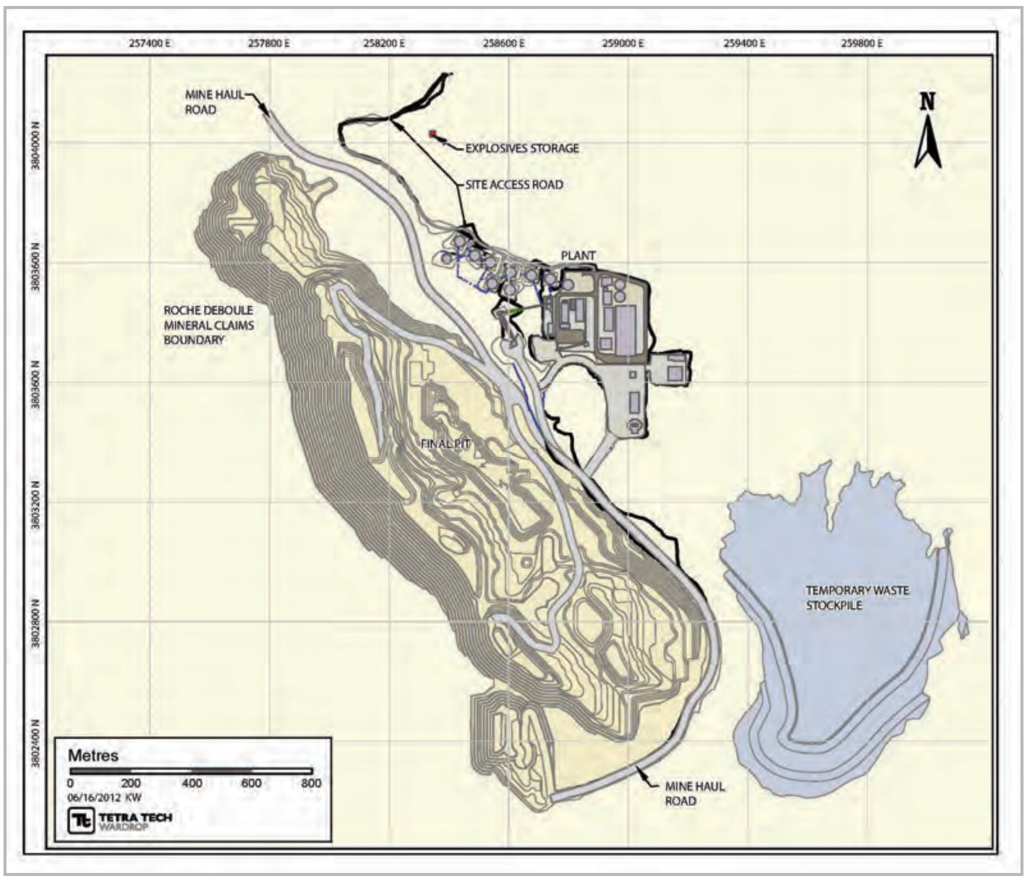
Year	Y1	Y1	Y1	Y15	Y16	Y1	Y1	Y19	Y 0	Y 1	Total
Mill Feed, kt	2,443	2,443	2,443	2,445	2,475	2,475	2,475	2,475	2,475	591	45,016
Head Grade, Mn	2.29	2.29	2.29	2.29	2.26	2.26	2.26	2.26	2.26	2.26	2.46
Recovery, Mn	89.4	89.4	89.4	89.4	89.3	89.3	89.3	89.3	89.3	89.3	90.0
Mn Produced, t	50,000	50,000	50,000	50,000	50,000	50,000	50,000	50,000	50,000	11,900	994,500

1 .0 PRO ECT INFRASTRUCTURE

1 .1 INTRODUCTION

The Project will be a comprehensive greenfield development situated in low-lying mountainous terrain. The process site is located directly east of the open pit. Overall mine site arrangement is shown in Figure 18.1.

i ure 1 .1 ine ite ayout



The on-site facilities are situated in a single, continuous, flat plain area, and include all key components of the Project, as shown in Figure 18.2 and Figure 18.2.

- a process plant
- a fleet maintenance facility
- administration offices
- a construction camp
- access and on-site roads
- a power substation
- a TWSF.

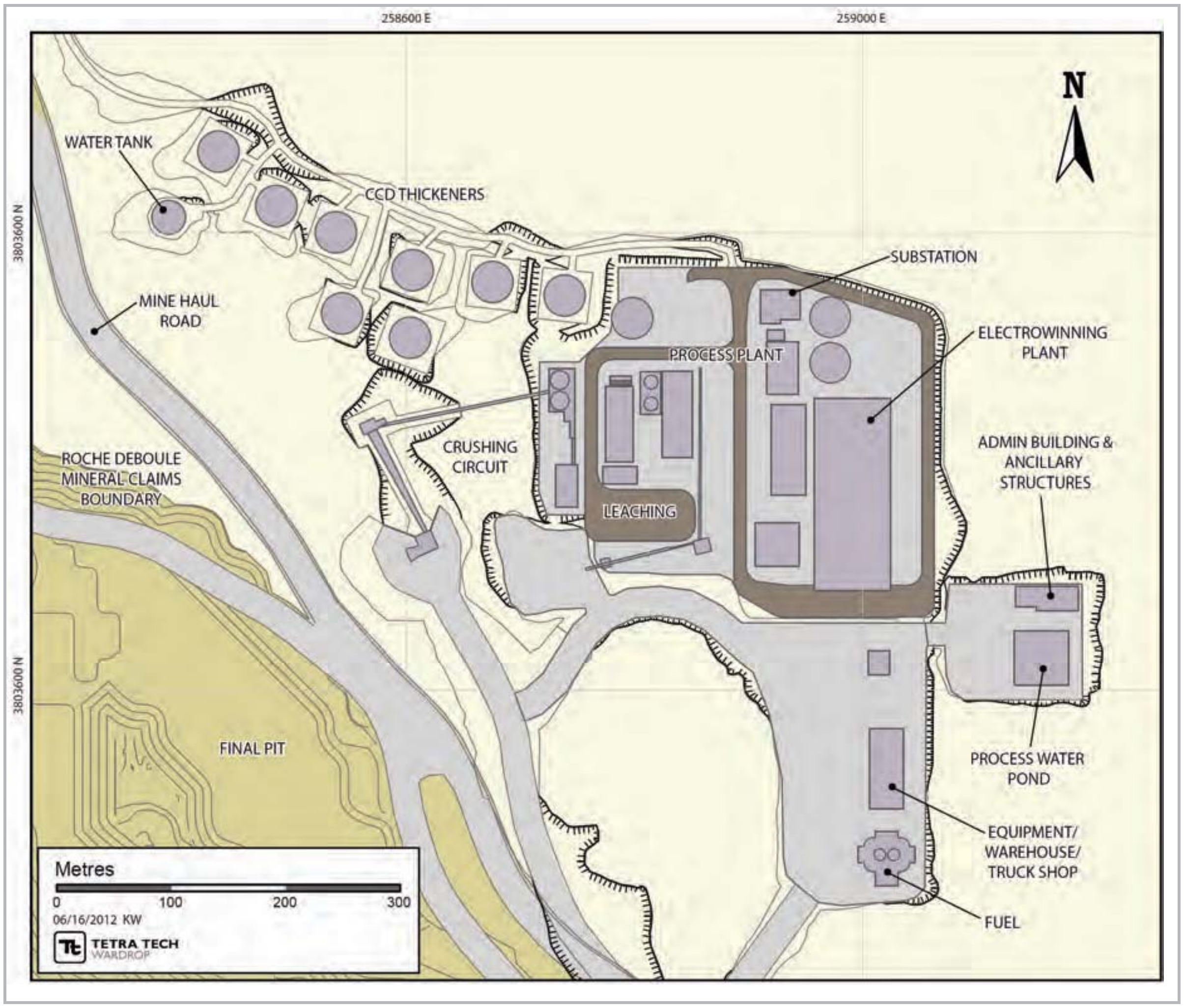
1 .2 PROCESS PLANT

The process plant will include

- a crushing facility
- a transfer tower (screening)
- leach feed surge bins (a total live capacity of 14,000 t)
- leaching and related facilities, including leaching, CCD thickeners and tailing filtration
- PLS purification, including manganese precipitation
- an electrowinning process plant
- a sodium sulphate byproduct production facility
- a sulphur dioxide generation, sulphuric acid generation, heat recovery and electrical power generation facility.

The plant site pad will be gravel-surfaced and graded to control water flow. The major process facilities are shown in Figure 18.2, and discussed in further detail in the following sections.

i ure 1 . General Plant ite Arran ement



1 .2.1 A I ITIES ES RIPTI S

CR SHIN

The crushing building will be of concrete construction, with multiple levels housing sizer crusher (primary crusher), impact crusher (secondary crusher), and associated feeders.

The structure will be earth retaining on three sides, stick-built, and enclosed up to the dump pocket. ROM material will be discharged into the dump pocket at the top. Interior steel platforms will be provided to support equipment for ongoing operations and maintenance.

The area will be equipped with a dust control system, to collect the fugitive dust generated during crushing and transports.

SCREEN FEE TRANSFER TO ER

The screen feed transfer tower building will be a pre-engineered steel structure with a roof and walls. The building foundation will consist of concrete spread footings, grade walls along the building perimeters and slab-on-grade. The building will be serviced by a mobile boom crane through a roof hatch.

The feed screen will be supported on an elevated steel platform over a concrete foundation. The area will be equipped with a dust control system.

LEACH FEE S R E IN

The surge bin will be an engineered post-and-beam structure connected to two bins, supported over a heavy concrete mat foundation. The two bins will be equipped with an air cannon system. The area will be equipped with a dust control system to control fugitive dust generated during transportation of feed material.

LEACH FACILITY

The leach facility will be an engineered post-and-beam steel structure with a steel roof and no walls. The foundation will consist of concrete spread footings, grade walls along the structure perimeters, a slab-on-grade floor, and containment curbing. The floor surfaces are acid protected, and will have localized areas that are sloped toward sumps for cleanup operations.

The building will house reduction leach tanks and CCD thickeners feed pumps. The area will be equipped with a scrubbing system, a ventilation system and an air compressor.

S LPH R IO I E ENERATION HEAT RECOVERY AN PO ER ENERATION FACILITY

The sulphur dioxide generation sulphuric acid generation, heat recovery and electrical power generation building will be a pre-engineered steel structure with steel roof and walls. The building foundation will consist of concrete spread footings, grade walls along the building perimeters and a slab-on-grade floor. The floor surfaces will be acid-protected where are required.

The building will house the sulphur dioxide and sulphuric acid production system including a sulphur burner, a heat recovery boiler and a steam power generator. Liquid sulphur storage tanks and sulphuric acid storage tanks will be placed external but adjacent to the building.

CC THICKENERS

The leach residue washing CCD thickeners will consist of a set of eight 36-m diameter thickeners, each with a removable fibreglass cover. The CCD thickeners will be supported by an engineered post-and-brace steel structure. The foundation will consist of concrete spread footings, grade walls along the perimeters, pedestals and piers, and a slab-on-grade floor. The slab will have containment curbing, and floor surfaces will have acid protection under the first three CCD thickeners. The gas collected from the fibreglass covers will be directed to the scrubbing system.

RESI E FILTRATION FACILITY

The residue filtration facility will be on a large concrete slab with containment curb. The facility will include a stick-built, steel column-and-beam structure with a standing seam steel roof and no walls. The facility will have an elevated steel operating floor platform throughout, and will be supported on concrete spread footings with concrete grade walls along its perimeters.

This facility houses the bank of residue filter presses and residue filter cake collecting conveyors and related filter feed pumps and air compressors.

PLS P RIFICATION FACILITY

The purification building will be a pre-engineered steel structure with a steel roof and walls, and roof hatches for access by the mobile boom crane. The floor surfaces will have localized areas that are sloped toward sumps for cleanup operations. The floor surfaces will have acid protection as required.

ELECTRO INNIN PLANT

The electrowinning building will be a pre-engineered steel structure with steel roof and walls. The building will house three overhead cranes via two rows of internal columns. The floor surfaces will have acid protection over 50 of the area.

The building will house three lines of electrowinning cells and related direct current power supply systems and cathode and anode plate treatment equipment, including rectifiers, automatic plate surface passivation, automatic plate washing, automatic plate dryer, automatic plate peeling, cathode plate polishing, MM surge silo and manganese bagging system. Anode and cathode plate storage areas and MM product storage load-out area are incorporated in the layout plan. The electrowinning building will also house an assay and metallurgical laboratory and a control room. The ventilation system will be installed to collect the gases produced from the eletrowinning cells.

YPRO CT PRO CTION SO I M S LPHATE RECOVERY FACILITY

The sodium sulphate by-product production building will be a concrete pad with a small pre-engineered building to house nanofilters and the sodium sulphate bagging system and load-out. A two-stage mechanical vapour recompression evaporation system will be installed outside of the nanofiltration building.

REA ENTS PREPARATION IL IN

The reagents preparation building will be a pre-engineered steel building with roof and walls and several separated areas, each with a separate fume extraction system. The building will be supported on concrete spread footings with concrete grade walls along its perimeters.

1 .3 MAINTENANCE AND STORAGE

1 .3.1 TR S P

The truck shop will be a pre-engineered steel building with roof and walls. The building will be supported on concrete spread footings with concrete grade walls along its perimeters. Sumps and trenches will be constructed to collect waste water and waste oils in the maintenance bays.

The building will consist of an exterior wash bay complete with six pressure washer cannons on two elevated platforms, repair bays, warehouse area, welding area, machine shop, emergency vehicle parking, first aid room, electrical room, compressor room and a lubricants storage room.

1 .3.2 MI I E IPME T ST RA E, ARE SE A EMER E IRST AI I I

This building will be a pre-engineered steel building with roof and walls. The building will be supported on concrete spread footings with concrete grade walls along its perimeters.

1 .3.3 E ST RA E A ISTRI TI

Diesel fuel for the mining equipment and ancillary facilities will be supplied from the aboveground diesel fuel storage tanks, located near the truck shop. The diesel fuel storage tanks will have a capacity sufficient for approximately three days of operations. Diesel storage will consist of aboveground tanks and a containment pad, complete with loading and dispensing equipment conforming to regulations.

1 .4 ADMINISTRATION BUILDING

1 . .1 MI E R A A MI ISTRATI MP E

The complex will be a prefabricated modular structure located in close proximity to the electrowinning plant building. This facility will house a mine dry and office areas. A mudroom will also be attached.

1 . .2 STR TI AMP

A temporary camp will be provided to house construction persons for the duration of mine construction.

1 .5 BUILDING SER ICES

1 .5.1 EATI , E TI ATI A AIR ITI I A

All process areas will be heated to a minimum temperature of 5°C. Large process buildings will be ventilated year-round to prevent a buildup of contaminants and humidity. Air conditioning will be limited to offices, control and electrical rooms, laboratories, and those rooms where heat gains are excessive.

1 .5.2 IRE PR TE TI

A firewater tank will be provided capable of sustaining two hours of firefighting at the design water flow rate. Sprinkler systems will be provided in lubricants rooms, air compressor rooms, blower rooms, truck shops, warehouses, laboratories, the elevated mill offices, the mining equipment storage building and the administration building. Sprinklers will also be used to protect conveyors located in enclosed areas.

Fire hose stations located such that all areas of each building will be within reach of a 30 m hose and a 15 m hose stream.

1 .5.3 ST TR

Dust control systems will be provided at the crushing and crushed material transport facilities and areas where fugitive dust may be generated. The dust generated from routine truck transport will be suppressed by spray water. The dust collection equipment will consist of dry baghouses, and the collected fines will be returned to the process stream.

1 . SITE ACCESS ROAD

The process plant site, project facilities and open pit will be accessible by a new permanent road, connected to the existing, well-maintained Alamo Lake Road. From the intersection with Alamo Lake Road, north from the Project, the proposed access will pass alongside mountainous washes, and wind through promontories and outcropping rocks, head southwest of the adjacent drainage basin, until its connects to the north boundary of the plant site.

The permanent site access road will also act as collector for the network of feeder roads from the CCD thickeners and CCD lay-down drop zones for the mobile boom crane required for operations and maintenance.

1 . PO ER SUPPLY AND DISTRIBUTION

1 . .1 PERATI S A

Approximately 4 MW of local generation is available at the mine site, and an additional 60 MW is being brought in from area electrical utilities, consisting of Western Area Power Authority (WAPA), Southwest Co-op and UniSource nergy Services (U S).

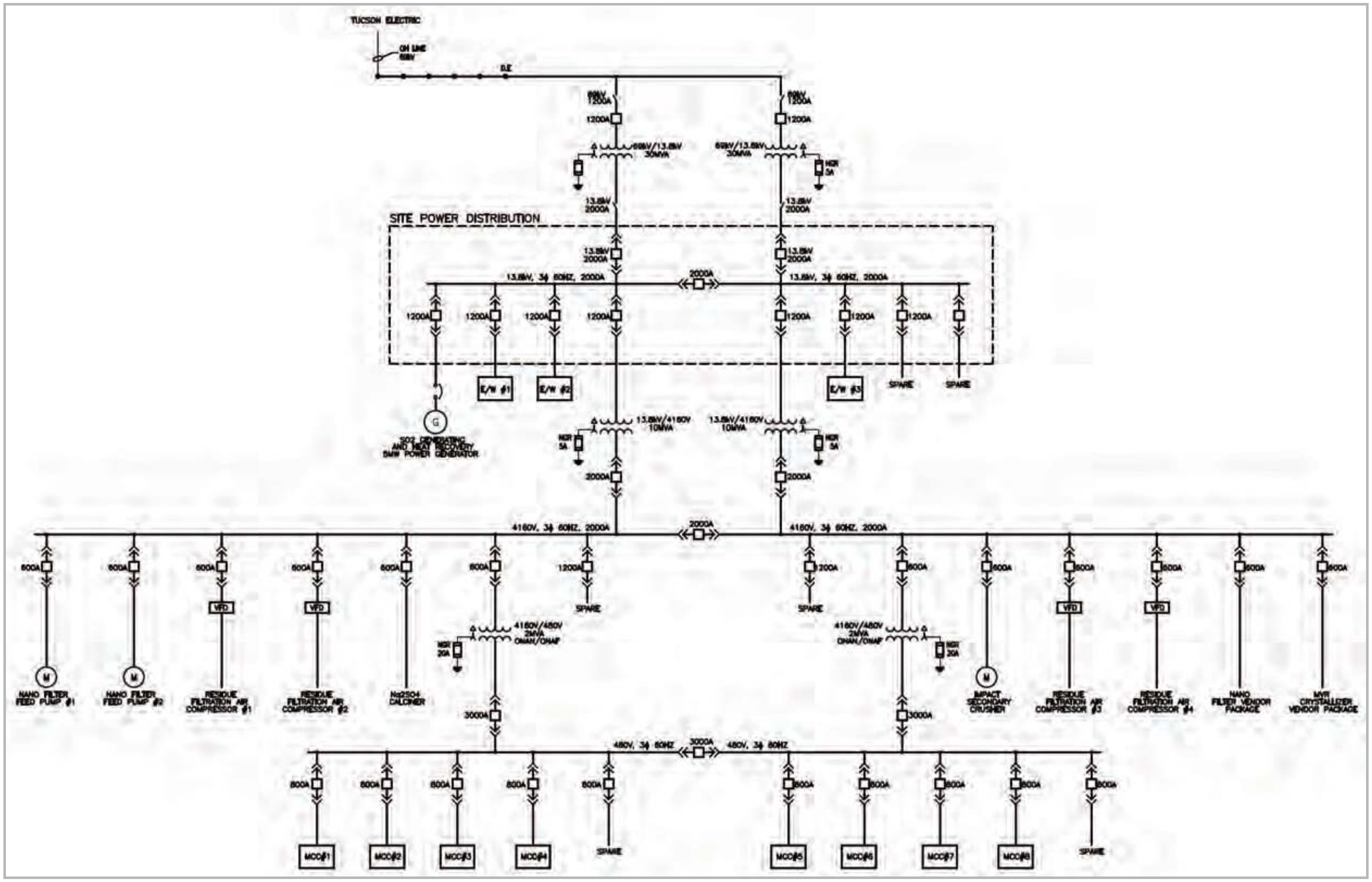
1 . .2 P ER S R E

For the purpose of this PFS, the recommended source of site power is through U S in coordination with WAPA.

1 . .3 P ER ISTRI TI

Power supply for distribution to the mine and process facilities will be brought into the facility at 69 kV where AMI's substation is located. The utility will bring the power distribution to an H-Frame located just outside of the AMI substation, where incoming voltage will be reduced to the plant distribution voltage of 13.8 kV. Distribution throughout the facility will be by overhead wooden pole and underground duct banks. The substation will consist of metering, steel and concrete structures, and two 30 MVA transformers connected to 13.8 kV switchgear. The switchgear will be in a main-tie-main configuration to ensure redundancy. The on-site 5 MVA power generator will connect into the electrical system through a circuit breaker on one of the electrical buss bars. Figure 18.3 shows the conceptual one-line diagram for the plant site.

i ure 1 . Conceptual One- ine Dia ram Plant ite



1 . . E E TRI P ER E I ER

Obtaining electric power from off-site sources is based on increasing delivery of energy at the Parker 69 kV substation to accommodate delivery of 66 MW of load (plus losses) from U S to site. A new transformer at the substation will be necessary, as will modifications to the 161 kV and 69 kV yards, and a rebuild of the first 13 miles of 69 kV line up to the planet tap located on the southwest transmission 69 kV line located near site. A portion of this line will need to be rebuilt from the Parker Substation to the tap point where the overhead transmission line departs to the mine site. WAPA has provided a cumulative construction cost that will also be included.

The AMI substation cost is estimated based on a design build substation, utilizing pricing from in-house project information, and, together with all power supply and distribution costs, is included in the capital cost estimate for the Project.

1 . TEMPORARY ASTE STORAGE FACILITY

The TWSF will be located southeast of the pit, and will be constructed with filtered tailings and waste rock generated from the beginning of mining through to Year 6. At the end of the LOM, the material stored in this TWSF will be relocated to the mined-out sections of the pit.

1 . ATER AND ASTE MANAGEMENT

1 .9.1 ATER MA A EME T

ATER S PPLY

The Project site will require approximately (0.05 m^3 sec (1.9 ft^3 sec), or 3.22 m^3 min (850 gal min) of continuous water supply.

There is a possibility of usable underground water in the aquifers southeast of the mine area. Aquifer testing of wells east and southeast of the mine produced a limited amount of water, and may form part or all of the solution to the required water supply.

ine source wells will be drilled in the areas of expected higher permeability. Aquifer testing performed on each of the nine source wells will evaluate the suitability for the siting of a larger scale water-supply well field in selected areas. There is a possibility of usable underground water from wells that will be drilled in areas southeast of the mine.

Based on the water resources information above, a preliminary water distribution line was designed to convey the water from the furthest identified well source to the proposed mine facilities. The proposed water line was assumed to be constructed on grade, following existing roadways to the planned mine site.

Preliminary calculations were run to determine the required pipe size, pump systems, and storage tanks for the mining operation

- The Hazen-Williams equation was used to determine the pipe size and the major losses for pipe system.
- The elevation difference and the major losses were used to determine the pump system required to deliver the required 850 gal min flow rate to the mine site.
- An estimated three days of storage along with the required 850 gal min flow rate was used to determine the required tank size for the mine site.

ATER MANA EMENT SITE FACILITIES

Many proposed site facilities will change over time as mining progresses. To detail these changes, a series of figures showing the evolution of these facilities (Figure 18.3 to Figure 18.10) have been produced. Six case scenarios options were evaluated for water management purposes and changes over the LOM open pit, plant site, TWSF, backfilled pit areas, roads and diversions.

OPEN PIT

The open pit is the facility that changes the most as mining progresses. Open pit mining will begin in Year 1 in an approximately 0.2 km^2 area in the southeast portion of the mine. Subsequent mining phases will increase the depth and area of the pit. For the water management plan (WMP), the open pit will be treated as a closed system, with all direct rainfall and local runoff treated as contact water and collected in a sump in the pit bottom. The groundwater table is anticipated to be located below the bottom of the pit, and a pit dewatering system will not be required. Any perched groundwater entering the pit will also be collected in the sump. The captured water will be conveyed to the process water pond (PWP), and then incorporated into the process circuit during active operation.

PLANT SITE

The plant site facility will not change over the LOM, so all water management structures will be built in Year 1 and remain in place for the entire LOM. Like the open pit, the plant site will be a closed system, with all precipitation and local runoff collected in the PWP and treated as contact water. The captured flows will then be incorporated into the process flows.

TEMPORARY ASTE STORA E FACILITY

The TWSF will be located southeast of the open pit. The general concept is to construct lifts of mixed tailings and waste rock. Concurrent reclamation and best management practices (BMPs) will be used to limit erosion on the sloped areas. The surface layer of filtered tailings and waste rock will be compacted, forming a fairly impervious zone, and will be sloped inward so that all precipitation that falls on top of the active area will remain on top and evaporate. Ponded water can also be pumped to the PWP as needed to limit infiltration into the commingled tailings mass.

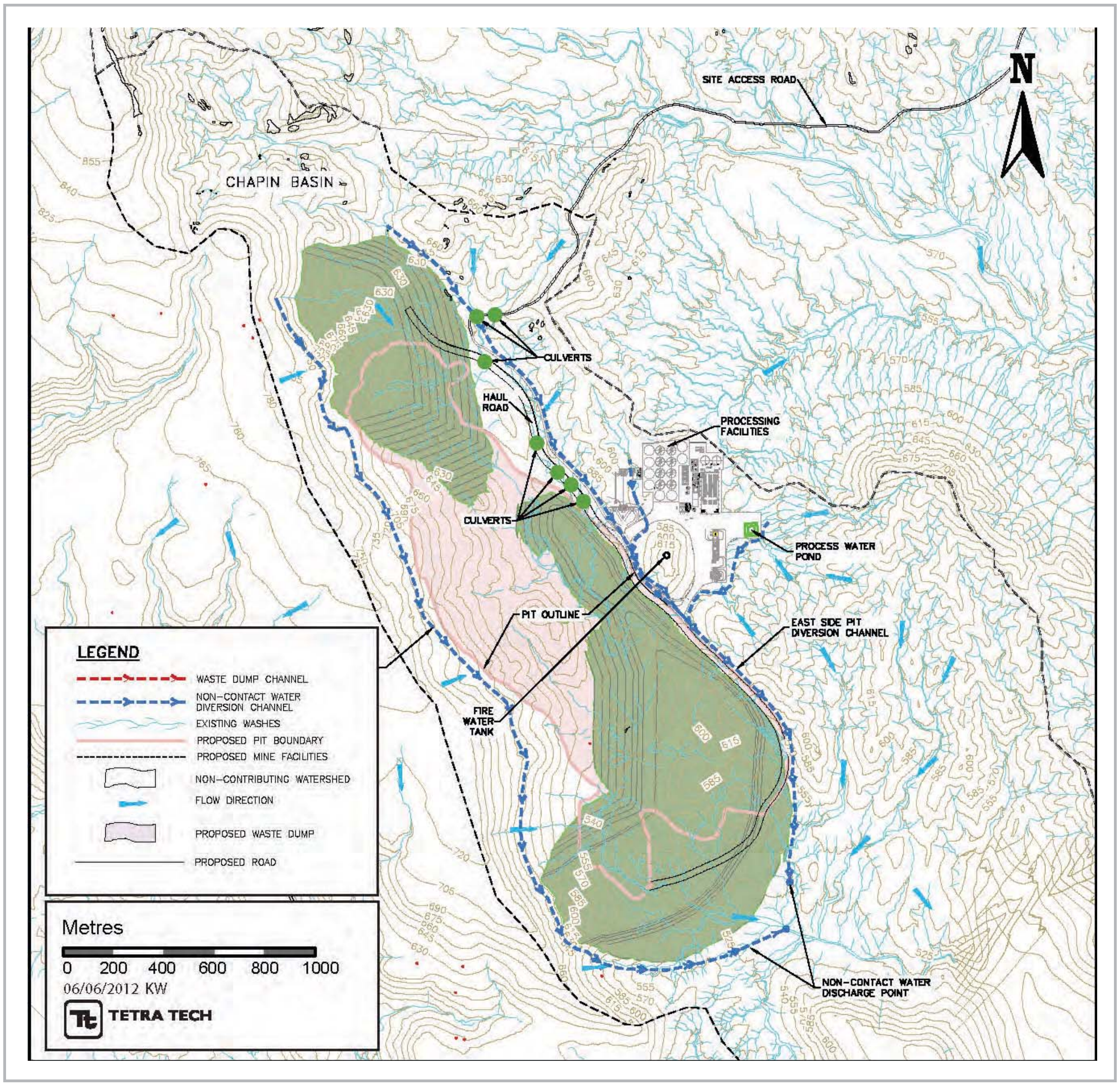
The TWSF will be constructed of a mixed filtered tailings and waste rock generated between the beginning of mining and Year 6. Alternate truckloads of filtered tailings and waste rock will be deposited and immediately mixed using a bulldozer. After Year 20 or prior to the end of the LOM, the TWSF will be moved back into the open pit for final reclamation of the mine. The material will be transported to the pit using dump trucks and then spread using a bulldozer. The location and progression of the TWSF is shown in Figure 18.4 to Figure 18.10.

ACKFILLE PIT AREAS

The two backfilled pit areas will be located south and north of the open pit. The general concept is to construct lifts of mixed tailings and waste rock, similar to the construction of the TWSF. Concurrent reclamation and BMPs will be used to limit erosion on the sloped areas. As with the TWSF, the commingled tailings mass surface layer will be compacted and sloped inward, so that all precipitation falling on the active area will remain and evaporate, and ponded water can be pumped to the PWP as needed to limit infiltration.

The pit will be backfilled after Year 6, starting at the south end. A channel will surround this backfilled zone to collect the contact water and store it in contact water ponds for testing before being discharged. At Year 10, backfilling of the north end of the pit will start. This backfilled zone will have a channel that collects the contact water and discharges it directly into the open pit, where it will be collected for use in the process plant. The location and progression of the backfilled pit areas are shown in Figure 18.4 to Figure 18.10.

i ure 1 . Year 1 ater ana ement Plan



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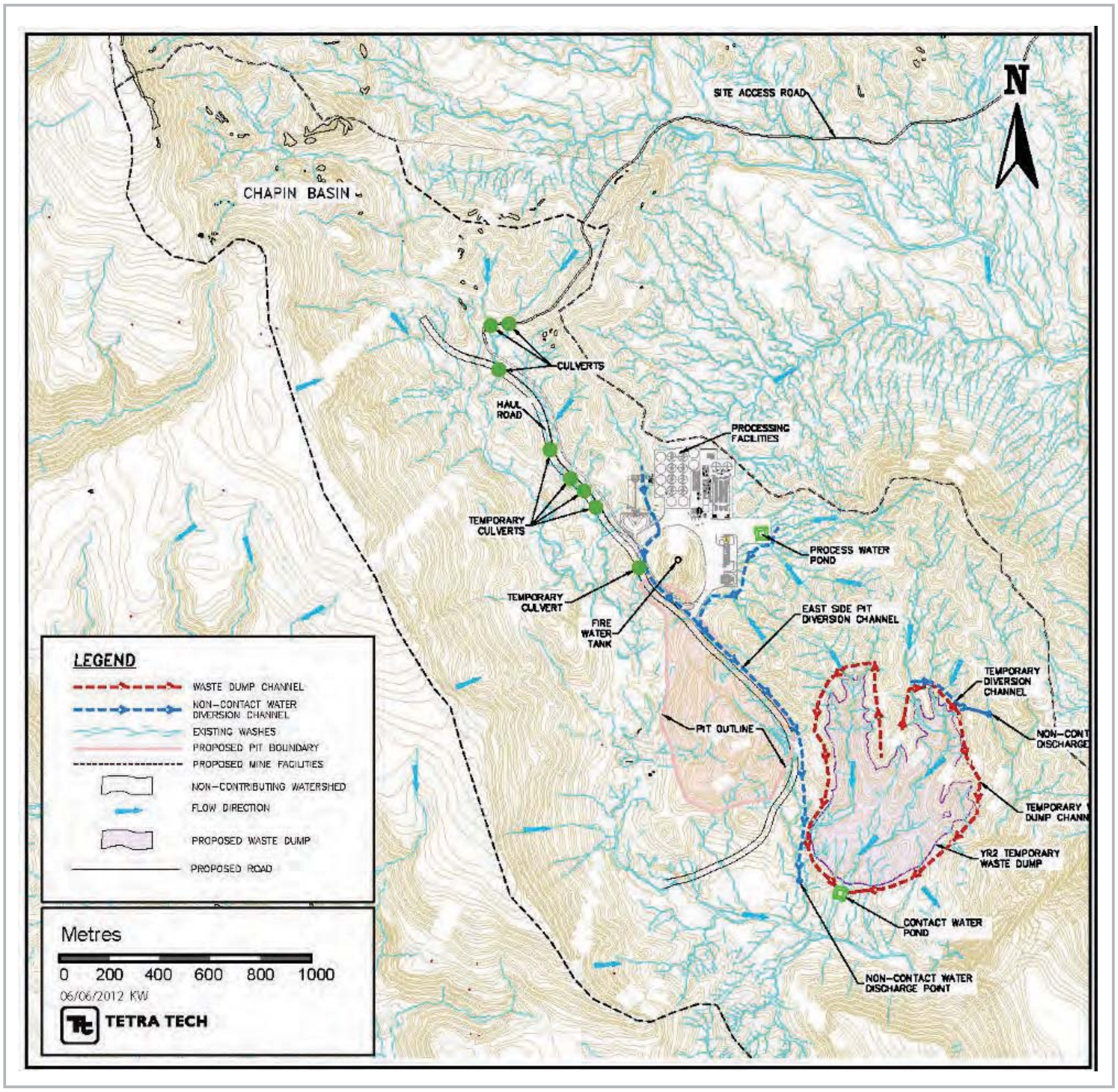
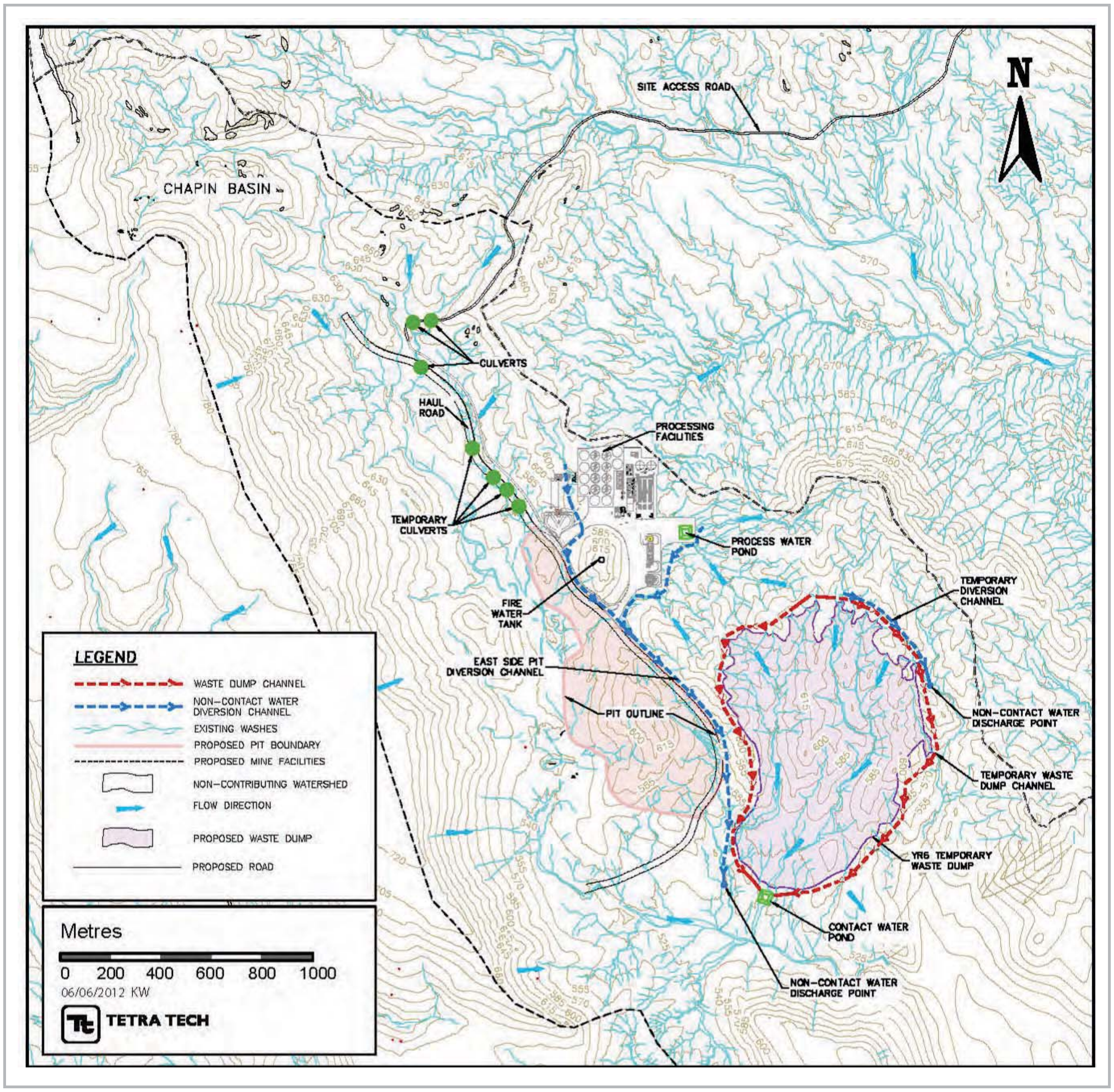
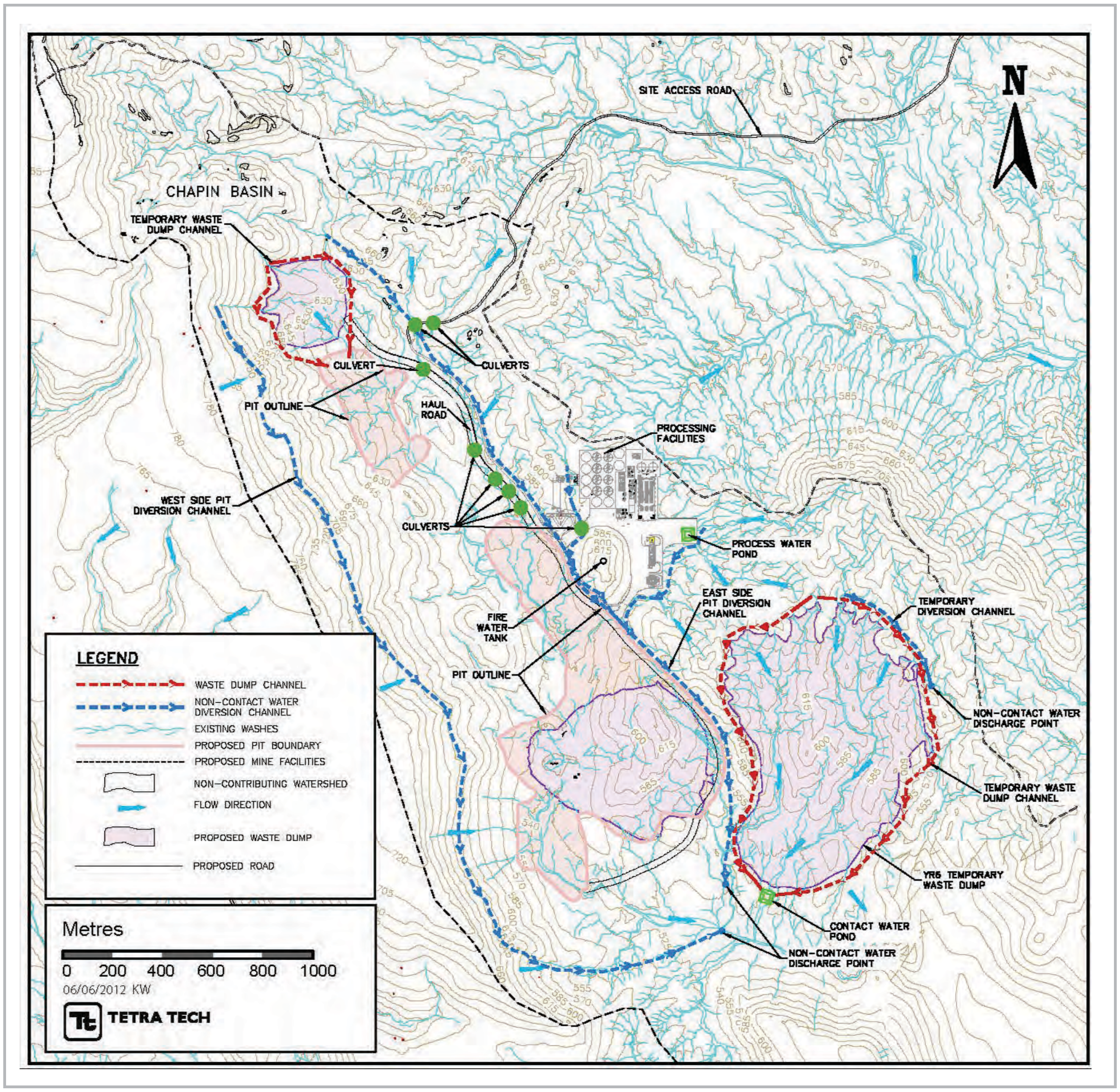


i ure 1 .6 Year 6 ater ana ement Plan



i ure 1 . Year 10 ater ana ement Plan



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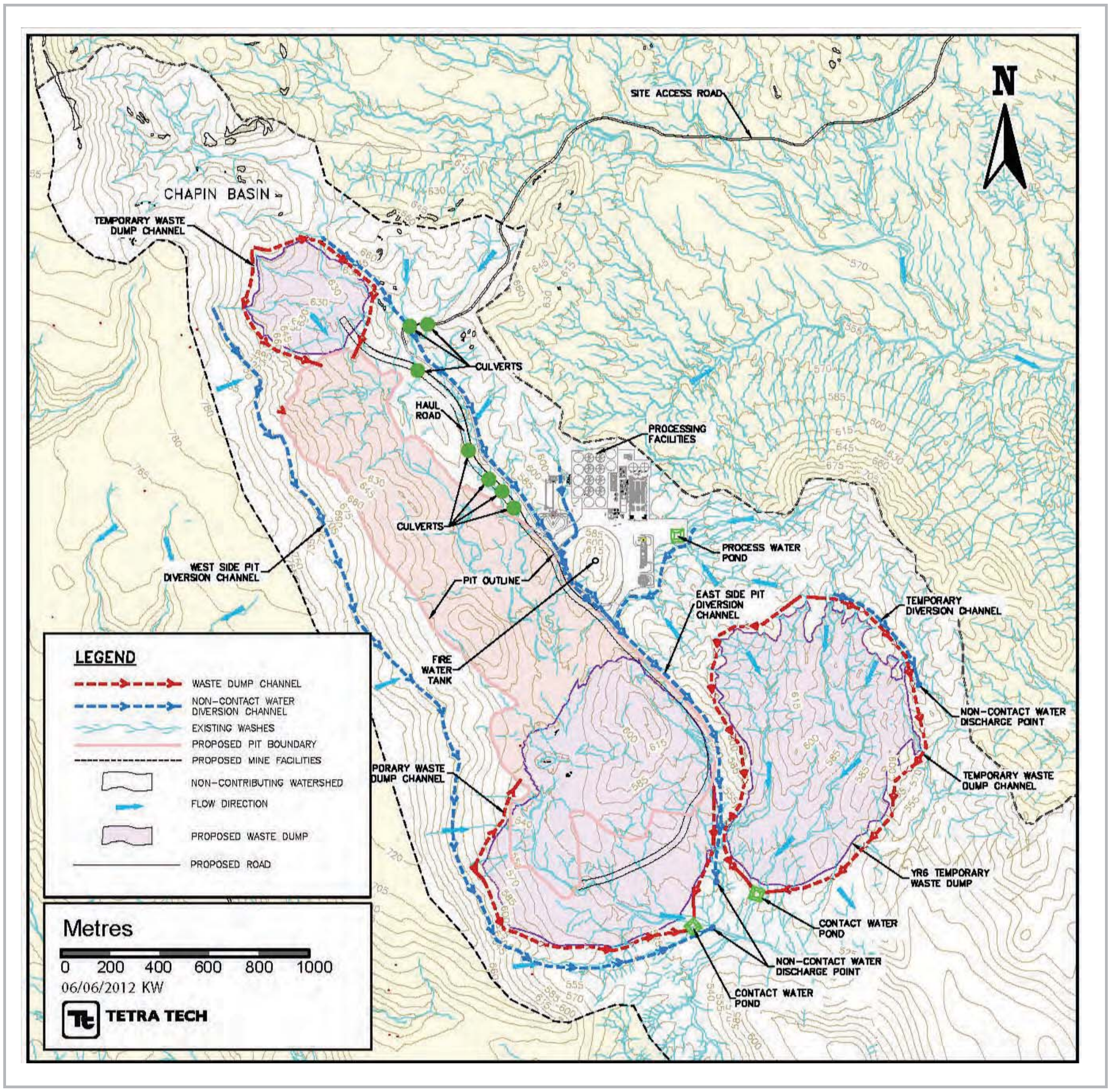
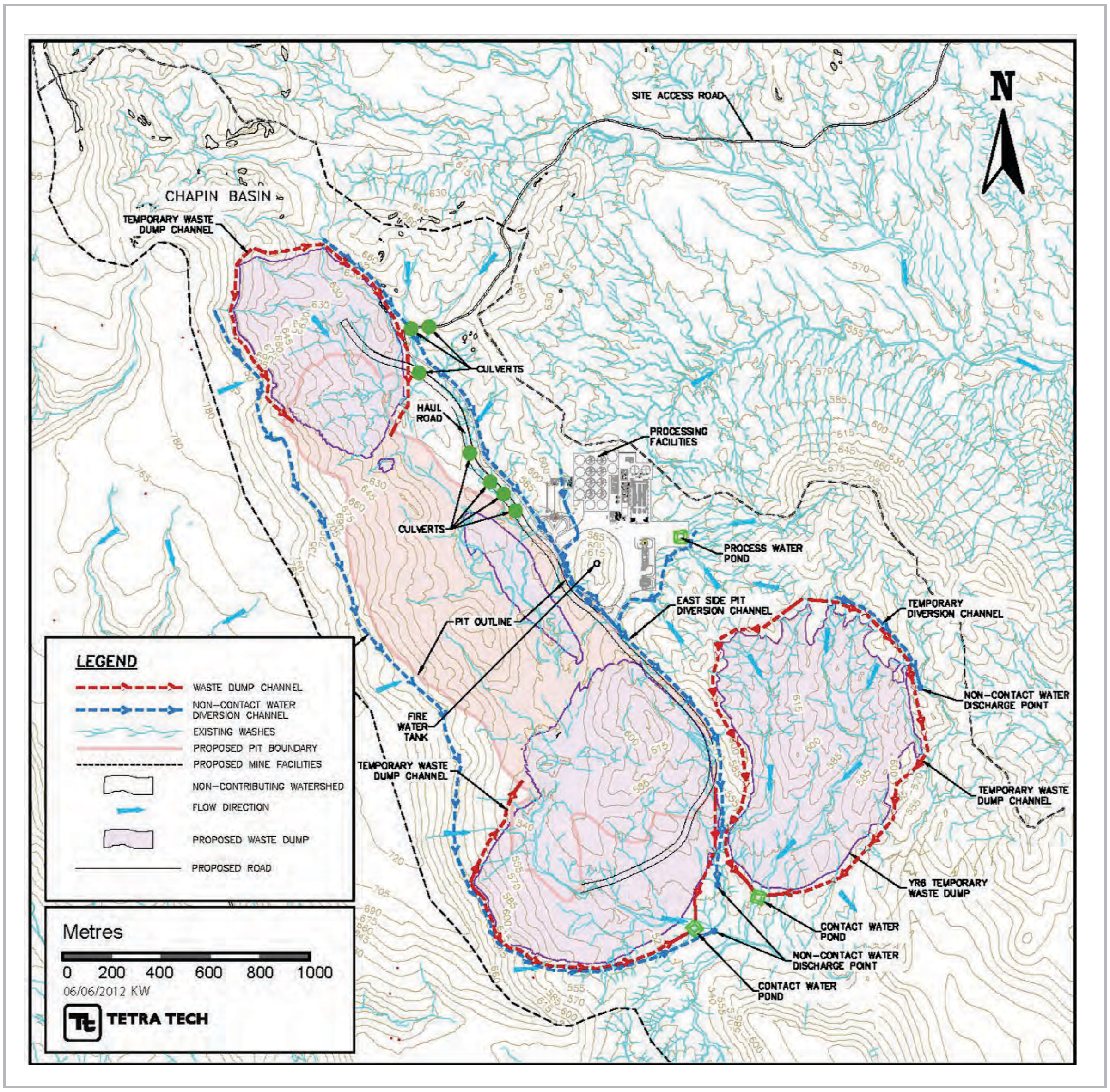
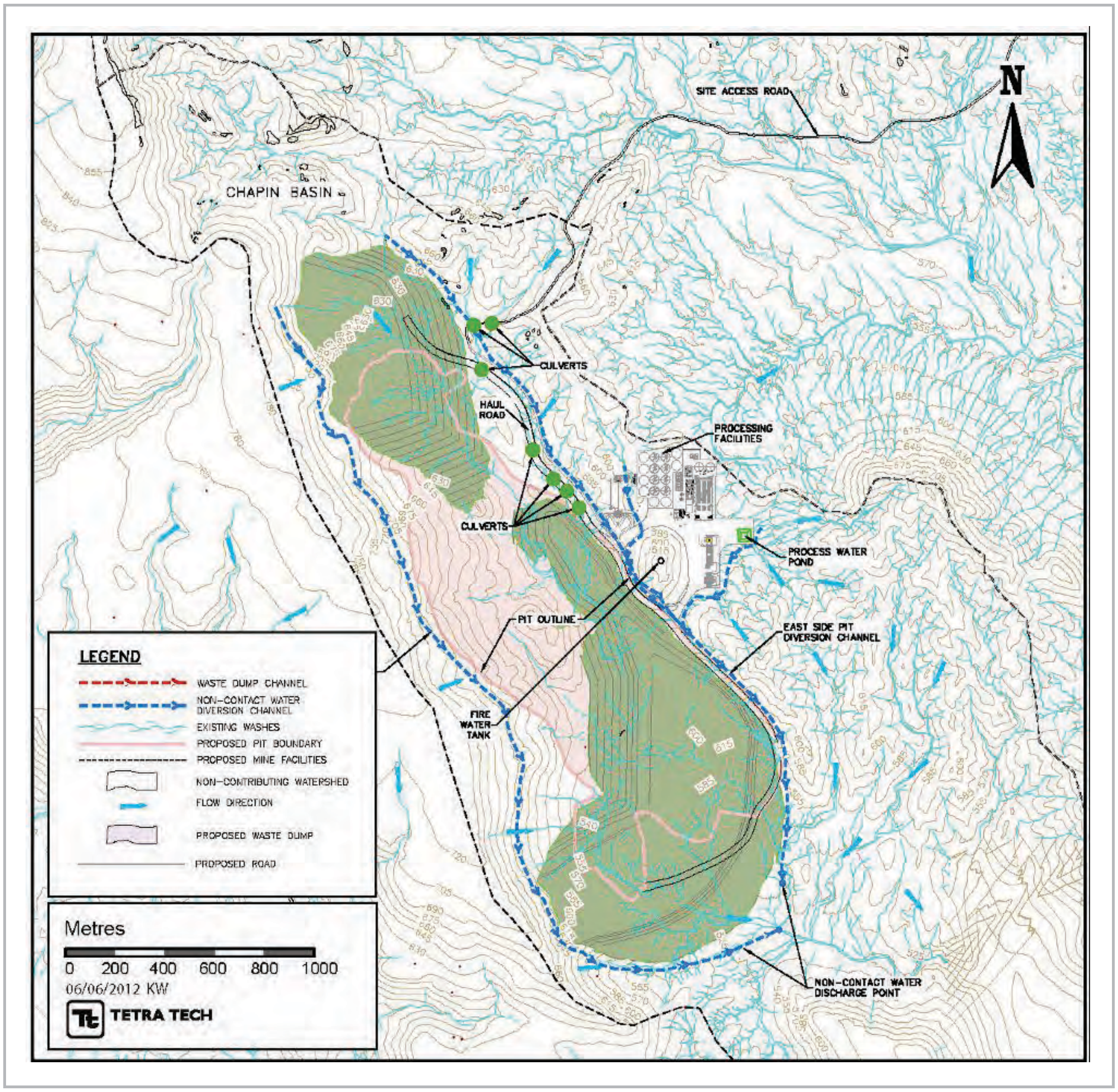


i ure 1 .9 Year 1 ater ana ement Plan



i ure 1 .10 Closure ater ana ement Plan



ROA S

The main access road to the mine site will cross several drainage ways. Culverts will be placed under the access road to allow the drainage to continue to flow as it does under existing conditions. The main haul road for the mine will be built on the east side of the open pit to connect the main facilities. For Year 1 through Year 6, culverts will be installed under the haul road to allow non-contact water to flow as it does under existing conditions. At Year 10, when the east side diversion channel is extended to the north end of the open pit, most of the non-contact water will be diverted around the open pit. However, the culverts under the haul road will remain to allow the small area between the east side diversion channel and the haul road to drain into the open pit.

BMPs will be used during construction to limit erosion potential. The roads should not be a significant source of sediment yield over the long term. The location and progression of the access and haul roads are shown in Figure 18.3 to Figure 18.9.

IVERSIONS

Diversions will be used throughout the LOM to minimize the amount of non-contact water draining into the open pit or contacting waste from the mining process. The diversion channels will run in parallel to the existing drainage patterns of the area, flowing from the northwest to the southeast.

Pit i ersion

For Year 1 through Year 6, the east side pit diversion channel will run from the south side of the plant site to the south side of the open pit and discharge into the existing drainage way. After Year 10, the open pit expands further north, and the east side diversion channel will need to be extended to prevent non-contact water from entering the open pit.

At Year 10, the west side diversion channel will be built to minimize the amount of non-contact water entering the open pit.

The diversion channels built after Year 10 will remain for the rest of the LOM to divert non-contact water away from the open pit.

Plant Site i ersion

Water draining from east to west near the plant site will be diverted through a diversion channel tying directly into the east side diversion channel.

T S i ersion

Another diversion channel will be constructed at the northeast upstream side of the TWSF to divert non-contact water around this facility, where it will be discharged into the natural drainage way. This channel will change location up to Year 6, when the TWSF reaches its final design capacity.

ackfilled Pit Areas i ersion

The east side diversion channel and west side diversion channel at Year 10 will divert non-contact water around the southern and northern backfilled pit areas.

PROCESS ATER STORA E PON

The lined process water storage pond will be constructed near the plant site to provide storage of contact water to be used in the processing of the mineralized material.

1 .9.2 ASTE MA A EME T

S MMARY

Two types of waste will be generated at the mine site waste rock and filtered tailings. The current plan is to place the tailings and waste rock back into the pit. o permanent waste storage facility will be required for the Project. The first six years of waste production will be temporarily stored in a TWSF located southeast of the pit. After Year 6, the waste will be placed directly into the mined-out sections of the pit. At, or before, the end of operations, the material stored in the TWSF will be relocated to the pit.

CRITERIA

Table 18.1 outlines the tonnages of filtered tailings and waste rock requiring disposal at the mine site.

Table 1 .1 aste Deposition Criteria

Criteria	Description
Mine Life	21 years
Waste Rock Tonnage	Total 111.9 Mt Deposited in TWSF (S of pit first 6 years) 21.9 Mt Deposited directly in the pit 90 Mt
Dry-Stacked Tailings Tonnage (generated from mine plant)	Total 47 Mt Deposited in TWSF (S of pit first 6 years) 12.9 Mt Deposited directly in the pit 34.1 Mt

ESI N ASIS

The co-disposal of filtered tailings and waste rock is based on the following design criteria

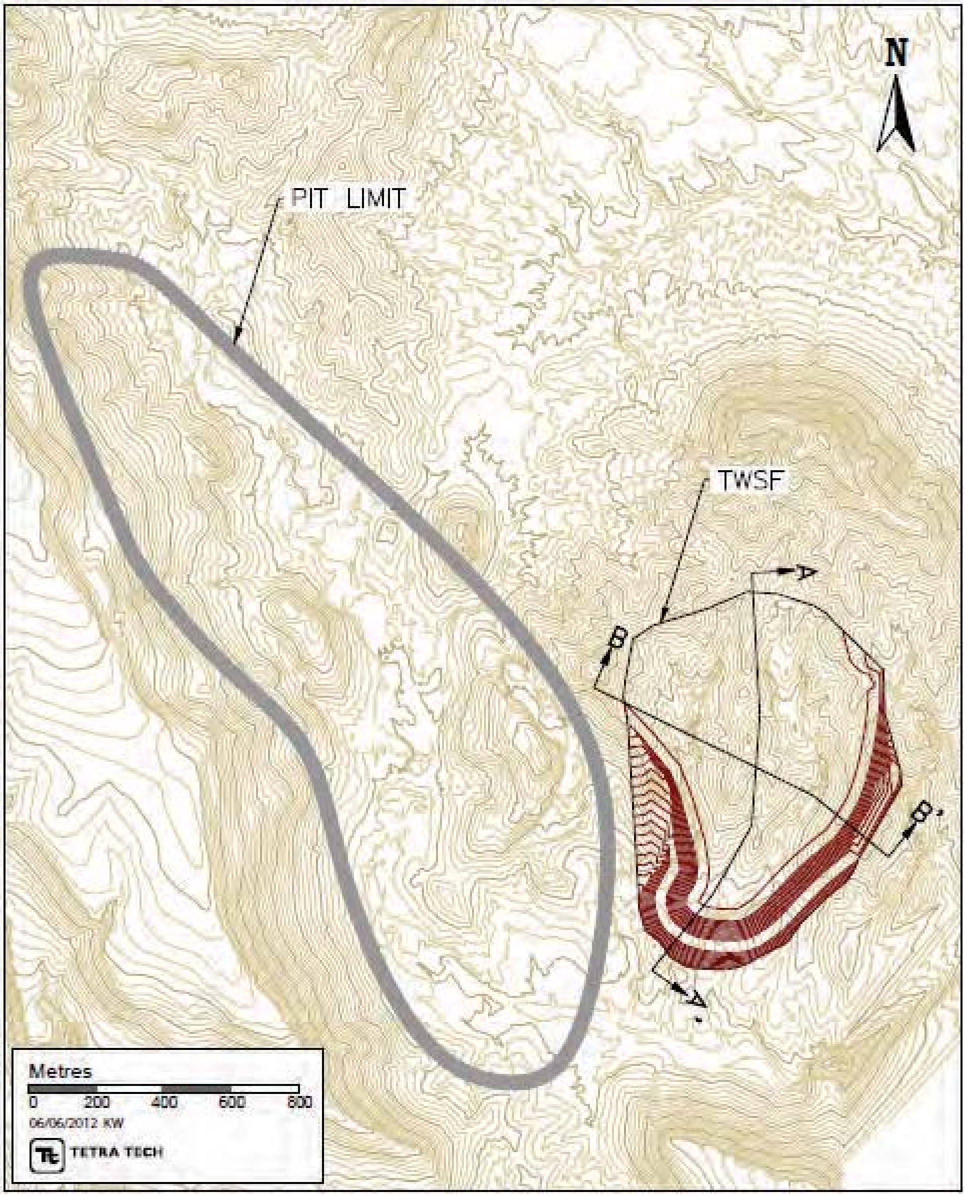
- The filtered tailings and waste rock generated by the mine will be placed and mixed in a TWSF.
- The bulk density of the mixture (filtered tailings and waste rock) is assumed to be 1.7 t m^3.
- The tailings will be filtered to achieve an optimum moisture content target of 22 ; 2 . The goal is to produce a mixture of tailings and waste rock at a moisture content near the optimum moisture content (moisture content is defined as the weight of the liquid divided by the weight of the dry solids).
- o liner system will be used beneath the mixture in the TWSF or in-pit backfill.
- The angle of repose of the waste rock material is assumed to be 37 degrees with zero cohesion.
- The water table is at significant depth.
- The material stored in the TWSF will be relocated to the pit; the filtered tailings and waste rock generated after Year 6 will be deposited directly in the pit.
- An evapotranspirative cover, consisting of 0.5 m of vegetated soil, will be used for closure of the backfilled pit.

FILTERE TAILIN S AN ASTE ROCK EPOSITION

In the first six years of mining, filtered tailings from the processing plant and waste rock from the pit will be hauled to the TWSF designated area. After Year 6, the two waste streams will be placed directly in the mined-out sections of the pit. The two waste streams will be mixed using a bulldozer making a homogeneous mixture. At the end of operations, the material stored in the TWSF will be moved to the pit. Figure 18.11 presents a plan view and two cross sections of the TWSF.

Surface layer compaction of this material is achieved by routing heavy equipment evenly over each lift. A minimum of five passes is recommended. This material must be placed in a manner that will minimize segregation of coarser particles. To avoid nesting, boulders greater than 750 mm in size will be removed from the fill as much as practically possible and pushed to the slope surface.

i ure 1 .11 Temporary aste tora e acility



Source Tetra Tech

ST CONTROL

The potential for dust generation due to wind erosion is considered for the waste rock and tailings stockpile created during the first six years of mining. BMPs for dust control will be implemented including, but not limited to

- restrict haul equipment travel over the tailings surface in order to avoid contamination of haul roads

- establish work guidelines for tailings haulage and placement to minimize dust generation
- place rip rap created from waste rock on the surface of tailings stockpiles
- apply dust suppressants on the active tailings surface, if required.

Air emissions from the commingled tailings will be monitored during the initial stages of construction by visual observation and air emission monitoring stations around the TWSF. If necessary as deposition progresses, a number of additional preventive measures will be evaluated, including

- increased application of dust suppressants and evaluation of alternative agents and measures
- smooth-drum compaction of the tailings surface where wind erosion potential is greatest
- installation of wind fences.

T SF RECLAMATION

The closure plan approach for the in-pit co-disposal is to return the land to pre-mining conditions. Potential impacts on groundwater during the operational, closure and post-closure periods of the facility will be limited by the low permeability of the mixed waste rock and tailings. A cover system will be employed to minimize erosion and promote landform stability and revegetation. The cover system will be designed to limit surface water infiltration into the tailings mass and to be sufficiently durable to withstand environmental and climate conditions, including extreme precipitation events.

The in-pit co-disposal landform will be constructed as backfill is placed into the mine open pit, with progressive reclamation as it becomes feasible. Once tailings and waste rock placement for a portion of the mined-out pit is complete, an evapotranspirative cover (0.5 m of overburden soil) will be placed over the surface of the compacted tailings facility to temporarily store runoff, allowing it to evaporate or be used by vegetation. The entire affected footprint of the in-pit co-disposal landform will be seeded and or re-vegetated with plants that promote soil evapotranspiration.

1 .0 MAR ET STUDIES AND CONTRACTS

This Section has been excerpted from CPM Group's lectrolytic Manganese Market Outlook, 2012.

1 .1 SUMMARY

Manganese is an essential industrial metal used as an additive in a wide range of steels, non-ferrous alloys and electronic components, as well as in specialty chemical applications. In the steel manufacturing process, the addition of manganese removes impurities such as sulphur and oxygen. It also optimizes the physical properties of the steel by improving its strength, hardness and abrasion resistance.

Of the roughly 15 Mt of manganese produced (metal content), roughly 89 is upgraded into ferromanganese and foundry products. High, medium and low carbon ferromanganese along with silicomanganese fall into this category. The remaining 11 of manganese ore is consumed in the production of metallurgical and chemical products, including electrolytic manganese metal, electrolytic manganese dioxide, lithium manganese oxide, manganese sulphate, and other chemicals. The primary focus of this report is on electrolytic manganese metal, which is produced through the electrolysis of a sulphate solution and sold as flakes or as powder. While Chinese producers and consumers dominate the MM market, structural shifts in supply fundamentals have transformed the metal's relative value. In addition to increased regulatory pressures in China, falling Chinese ore grades and rising production costs are opening up new opportunities for producers outside China.

The MM market is highly concentrated, with over 97 of global MM production sourced from China in 2010. South Africa, the only other MM-producing country, accounted for 2.1 of global supply. Rapid growth in Chinese MM capacity, tied to robust demand fundamentals and lenient government regulation, has crowded out producers in the United States and Japan over the past two decades. Over the past five years, MM prices have become more reflective of the tightening regulations governing the Chinese MM industry. At the same time, with improved steel technology in China, MM has grown to be increasingly attractive as a substitute for higher cost alloys in steel. While the environment for both MM supply and demand continues to evolve, the metal has established new structural price supports, partially based on cost inflation throughout the industry. Between 2012 and 2021, real MM prices are forecast to average nearly 2 lb, which will support project development activities and provide incentive pricing for bringing new projects online to meet continued growth in demand. In 2021 alone, annual MM demand may grow by more than 140,000 t, which is equivalent to the annual output from roughly four medium-scale operations in China.

1 .2 HISTORICAL AND RECENT PRICE TRENDS

In tandem with the increasing industrialization of China, structural changes in the MM market started to emerge in the early part of the last decade. In the 2000s, nominal monthly MM prices rose more than six-fold, from 0.36 lb in the second half of 2002 to 2.41 in June 2007. While a supply shock and speculation about Chinese regulation helped bolster prices to these highs, prices then remained at elevated levels, averaging roughly 1.76 until the global economic downturn in the fourth quarter of 2008. While market conditions for the industrial metal quickly shifted course in late September 2008 as the world economic environment turned decidedly negative, MM prices were relatively robust, holding largely above 1.00. Prices have been leveraged to the high growth Chinese market, with the country accounting for nearly 87 of global MM demand. After temporarily falling to a monthly low of 0.99 in ovember 2008, a choppy rise in prices followed over the next three years, buffeted by cyclical destocking and restocking activities in the steel industry. In the first ten months of 2011 manganese flake prices averaged 1.53. Concerns over uropean debt crises and slower growth in China weighed generally on commodity prices in the fourth quarter of 2011. Many of the underlying fundamentals that contributed to elevated MM prices remain intact going forward, namely rising production costs, more stringent Chinese regulations and strong demand growth.

1 .3 SUPPLY

While manganese is the twelfth-most abundant element in the earth’s crust, reserves are irregularly distributed. Data from the United States Geological Survey (USGS, 2008) showed that South Africa held the greatest share (77) of the 5.2 Bt in the global reserve base, followed by Ukraine with roughly 10 . Manganese ore reserves in South Africa are primarily high manganese grades (greater than 44 Mn), while ore grades in the Ukraine are typically lower (less than 30 Mn). Global reserves are more regionally diverse. early three-quarters of the world’s manganese reserves are in the Ukraine (22.2), South Africa (19.0), Brazil (17.5) and India (14.8), according to 2010 USGS estimates. Ore reserves in China account for roughly 7.0 of global reserves. However, Chinese manganese ore grades are low and production grades have been falling by 0.5 to 1.0 per year. China is becoming increasing dependent on overseas ore supplies. In 2010, imported manganese ore accounted for roughly 55 of China’s total manganese demand. Secondary supplies or scrap is not currently a commercially viable material source of manganese.

Manganese occurs in a variety of mineral compositions, including in the form of manganese oxides and hydroxides, manganese-bearing carbonates and silicates.

All types of manganese ore require processing to transform the feedstock into usable products. Both silicomanganese and MM are primarily produced from low-grade ore. However, falling ore grades and increased political pressure to reduce the environmental impact of the manganese industry have made MM production from high-grade ore more attractive, despite higher capital expenditures. In 2010 MM total cash costs for Chinese producers exporting MM fabricated from domestic low-grade carbonate ore (10 Mn) were roughly 1.32, nearly 13 more expensive than producing from imported high-grade ore (45 Mn). Currently, the vast majority of Chinese MM output is produced with domestic ore, while imported ore is used for alloy manganese products. The Chinese government has taken concerted actions to limit exports of materials with energy-intensive, polluting production processes. When selling to overseas markets, Chinese producers must pay a 17 value added tax (VAT) on most input costs and a 20 export tax.

Despite pressure from the central government to consolidate the industry, the manganese-mining sector remains highly fragmented, with roughly 600 manganese-mining companies. The MM market is slightly more concentrated, with less than 200 MM producers. To meet long-term strategic plans, the government will continue to promote industry concentration by eliminating small MM producers with outdated technologies. Chinese industry participants expect that regulators will target small MM producers with a single production line of less than 5,000 t in annual capacity during the current five-year plan.

Falling ore grades, increasing operating costs and low recoveries are weighing on Chinese production. China is estimated to have produced 1.38 Mt of MM in 2010. Between 2011 and 2021, Chinese production is forecast to grow at a compound annual growth rate (CAGR) of 5.4 , compared to a CAGR of 30.0 during the previous decade. The tightening of environmental standards and industry restructuring initiatives are expected to reduce surplus capacities in the long run. Capacity utilization stood at roughly 63 in 2010.

The South African producer Manganese Metal Company also has 30,000 t of MM production capacity. Despite South Africa's mineral wealth, increased demand for non-Chinese dependable MM supplies is not expected to be met by new MM capacity in South Africa, due to the country's inadequate electrical grid.

1 .4 EMM DEMAND

Manganese is the preferred alloying element for applications where cost-savings and moderate corrosion resistance is desired. Thus, MM is employed in a wide range of applications, including 200-series stainless steels (mostly used for consumer applications) other steels, non-ferrous alloys in the canning industry, electronic components and specialty chemical applications.

The widespread use of MM in steel-related applications is reflected in the breakdown of MM demand. early 74 of MM demand comes from the steel sector. Specialty alloys, such as aluminum alloys, account for 12 of global demand, while other end-uses, including electronics and chemicals, account for the remaining 14 .

MM demand is largely driven by fluctuations in Chinese steel production. China accounts for over 87 of global MM demand, with the steel sector accounting for nearly 82 of domestic consumption. Urbanization in many emerging economies and growth in Chinese industrial production over the past decade have resulted in strong MM demand growth rates, despite the 2008 to 2009 downturn in global economic activity. Improvements in economic sentiment over the course of 2010, following the economic recession, encouraged increased capital expenditures, fixed asset investments and personal expenditures over this period.

1 .5 EMM CONSUMPTION AND THE STEEL SECTOR

MM is mainly used as an alloying element in the steel sector. It would not be possible to achieve the necessary tensile strength, toughness, stiffness, wear resistance and hardness of steel in the absence of manganese additions. MM is specifically used in place of other manganese products, such as ferromanganese, as an alloy when the desired end-product requires very low levels of impurities. In ferromanganese, for instance, higher impurity levels of elements such as phosphorous degrade the weldability of steel. MM is preferred in other cases due to the specific requirements of certain steel production processes.

The need for MM from the steel industry accounts for nearly 74 of the total demand. MM can be found in stainless, carbon, construction, engineering, non-magnetic, Hadfield and HSLA steels. Steel-based MM demand has exhibited stronger growth rates than global crude steel production over the past decade, due to improvements in the quality of 200-series stainless steel, the MM market's leverage to the Chinese market in both production and consumption terms and nickel substitution. CPM Group expects MM demand from the steel sector to increase at a CAGR of 6.7 from 2011 to 2021, rising from 1.1 Mt in 2011 to 2.2 Mt in 2021.

19.5.1 STAI ESS STEE

The largest end-use of MM is in stainless steel, accounting for nearly 42 of global demand. In low- to moderate-corrosion applications such as home appliances, kitchenware and ornamental fittings, 200-series grades are increasingly used for their cost advantages over costlier, high nickel-content steels. MM is also employed in high-performance stainless steels requiring minimal carbon content. The increase in the 200-series' market share of stainless steel production also reflects improved production processes.

19.5.2 T ER STEE

Other steel types collectively account for 32 of global MM demand. Other steel grades in this category include construction, engineering, non-magnetic, Hadfield and HSLA steels. These steel grades are used in the automotive, construction, energy, infrastructure and transportation industries.

1 . OTHER EMM APPLICATIONS

19.6.1 A MI MA T ER A S

Aluminum and other alloys are the third largest consumer of MM by end-use, accounting for nearly 12 of global demand. Manganese is found in aluminum, magnesium, copper, nickel, titanium and zinc alloys. Of these, aluminum alloys are the largest consumer of MM in the form of briquettes, which can contain anywhere from 75 to 90 manganese. The addition of manganese to aluminum alloys is a low-cost method of increasing hardenability, corrosion resistance, deoxidization and castability properties, as well as imparting mechanical strength to the aluminum alloy. These alloys are used extensively in aluminum cans. MM demand from this end-use is expected to increase from 181,355 t in 2011 to 322,373 t in 2021, a 5.9 CAGR.

19.6.2 E E TR I S

lectronics account for almost 2 of MM consumption by end-use. MM is commonly used as a raw material for soft ferrite powder production. Soft ferrites are used in computers, communications equipment, automobiles, televisions and vending machines for their electromagnetic properties. MM consumption in this end-use is expected to continue to reflect global production trends in electronic equipment, growing from 30,676 t in 2011 to 50,376 t in 2021, a 5.1 CAGR.

19.6.3 T ER

Other end-uses collectively account for 12 of global MM demand. Of these end-uses, batteries are the largest consumer of MM. MM is sometimes used in dry-cell zinc-carbon, alkaline, and lithium-based batteries, though MD is commonly prevalent in the latter two. Other end-uses include dyes, disinfectants, catalysts for chlorination reactions, various agricultural uses, anti-bacterial and anti-fungicidal agents and in welding rod electrodes. Other end-uses are expected to grow at a CAGR of 5.0 , from 183,614 t in 2011 to 299,050 t in 2021.

1 . Focus on Chinese EMM Demand

O CD economies play a marginal role in the MM market, given that China overwhelmingly dominates consumption. It is therefore necessary to emphasize both the importance of China in the MM market and the growth in consumption that has occurred over the past decade. On a per-ton basis, Chinese MM consumption is nearly ten times higher than it was at the start of last decade. MM demand per-capita is expected to continue to grow as urbanization growth supports demand for consumer products, many using MM-containing 200-series stainless steel. Chinese manufacturing activity is expected to be buoyed by healthy fixed asset investment levels.

1 . Substitutes

CPM Group's analysis of historical market developments, technical considerations and relative economics concluded that manganese is used primarily as a cost-saving alloy, and the scope for MM substitution is moderate. Three main substitution threats may result in lower volumes of MM over the projection period, including substitution to other manganese products, switching between steel grades with different manganese contents and switching to other elements such as copper, titanium, and other metals.

Regarding the substitution of MM to other products, consumers have historically shifted some of their MM requirements to products such as ferromanganese. The major determinants for this type of substitution are relative pricing and availability of supplies. For instance, consumers in Japan have tempered demand for MM to reduce their exposure to Chinese MM exports. However, Chinese substitution has been limited, due to readily available MM supplies, Chinese stainless steel producers' preferences for MM and limited Chinese high-grade ore quantities necessary to domestically produce other manganese products.

Historically, MM has been employed in 200-series chromium-manganese low-nickel steels during times of high nickel prices. The market share of 200-series steels to total stainless steel production has increased significantly in recent years due to the cost advantages over pricier, high nickel-content steels. If lower nickel prices result in a greater share of 300-series steels relative to 200-series over the projection period, demand for MM would be adversely affected.

Given the importance of cost reduction in applications where manganese is used in lieu of nickel, copper is one of the main substitutes for manganese. Plastics may also continue to be an increasingly important substitute application for household steel items and aluminum canning. Lastly, titanium can be a substitute for manganese in some niche applications, though its higher cost would likely limit such substitution.

1 . EMM SUPPLY AND DEMAND OUTLOO

Global supply and apparent demand of electrolytic manganese have historically been closely aligned. Increased demand for electrolytic manganese metal between 2000 and 2007 ushered in a substantial and sustained shift in the market's dynamics, with the pace of new supplies accelerating as a result. This overcapacity effectively kept a ceiling on prices, which ultimately led to plant closures outside of China. As of 2010, Chinese MM capacity was up nearly 16-fold since the beginning of the last decade, at 2.2 Mt. Meanwhile, only one other country was producing MM, compared to three in 2000. Over this 11-year period, the market recorded deficits on an annual basis in six years; however, the scope of these supply shortfalls was fairly modest, averaging less than 2 weeks of demand over those six years.

In recent years, restrictive export Chinese policies have led to significant amounts of smuggling to evade export duties. The Chinese government is expected to continue to step up its enforcement of the MM industry. After over a decade depending primarily on Chinese MM production, CPM Group expects new MM capacity may come on stream between 2013 and 2016 from five probable projects in the Ukraine, Gabon, Kazakhstan, Mexico and the United States. Other possible sources of MM supply could come from development projects in Canada, Finland and the United States. During the second half of the forecast period, China could at times become a small net importer.

1 .10 EMM PRICE FORECAST

The need for new capacity outside China and the rising cost structure of existing capacity in China are the primary factors driving CPM Group's price forecast. Present operating costs and projected costs going forward are not supportive of prices reverting to their historical averages of 0.45 to 0.85 lb. While CPM Group's forward supply curve incorporates supplies from new producers outside of China, Chinese MM output is still expected to account for nearly 90 of global supplies in 2021. For Chinese producers to maintain current production levels, significant capital investments need to be made to upgrade existing technology and to preserve domestic ore resources.

Over the 10-year period, prices may rise at a CAGR of 4.8 , accelerating toward the end of the period, allowing for more supplies from marginal cost producers. Between 2012 and 2016, real prices may average 1.72, and rise roughly 23 between 2016 and 2021 to average 2.11 on an annual basis. Over the next 10 years, real electrolytic manganese metal prices are forecast to average 1.92 lb, reaching an annual high of 2.30 in 2021. This is nearly double the annual average price over the last 10 years.

1 .11 O ER IE OF ELECTROLYTIC MANGANESE DIO IDE EMD MAR ET

MD consumes roughly 2 of global manganese supplies. As with the MM market, the majority of global MD capacity is located in China. However, production capacity is more regionally diverse, with the United States (14.8), Japan (8.0), South Africa (7.1) and Greece (5.4) accounting for more than a third of total capacity in 2010.

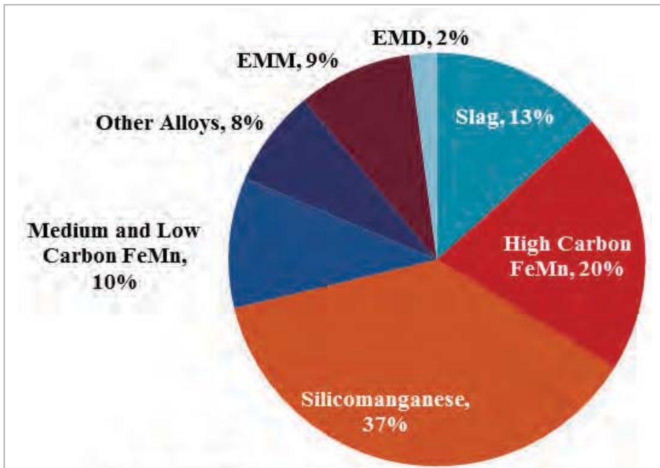
MD is mostly used in alkaline and other small-scale, consumer electronics batteries. Alkaline batteries are a low-growth end-use, expected to track well below GDP growth rates over the forecast period. In small-scale electronics, MD use is projected at historical growth rates of 4 . The highest potential growth segment for MD is in large-scale rechargeable batteries used in electric vehicles and electronics. At present, the rechargeable manganese battery segment accounts for less than 10 of total MD demand. As electric vehicles penetrate the auto market over the coming decades, MD demand stands to benefit. Demand from rechargeable batteries could reach 25 of total MD demand by 2021, according to CPM Group estimates. However, a large degree of uncertainty remains in the electric vehicle industry. CPM Group has accounted for this uncertainty by including an alternative case high-demand scenario, in which electric vehicles account for a greater share of total vehicle sales than in the base case.

MD demand is projected to rise from more than 349,000 t in 2011 to nearly 608,000 t in 2021, a 5.7 CAGR. In contrast to MM demand projections, MD demand growth is expected to accelerate after 2015, reflecting higher market penetration rates by electric vehicles. From 2016 to 2021, MD demand is expected to grow at a CAGR of 6.1 , compared to a 5.3 CAGR from 2011 to 2016.

Table 19-1 an anese Ore se

Description	Percenta e
Slag	13
High Carbon FeMn	20
Silicomanganese	37
Medium and Low Carbon FeMn	10
Other Alloys	8
MM	9
MD	2

i ure 19.1 an anese Ore se Comparison



Source: CPM Group

20.0 EN IRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 INTRODUCTION

The purpose of this section is to explain the process of environmental permitting and approvals necessary to bring the Project into production. The following sections explain the various permitting programs, their required advance studies, and the anticipated time required to secure approvals.

Table 20.1 lists the major permits required for construction.

Table 0.1 ist of A encies and Permits

A ency	tem	Description	Time e uired	Conditions
ederal Permits				
US Department of Interior, Bureau of Land Management (BLM)	Federal Land Policy and Management Act (FLPMA) Plan of Operations (Plan of Operations) Including Closure Plan and Reclamation Bond	Plan for mining operations on BLM lands	See PA time	Prepare a plan and manage according to the plan; update as required. Post reclamation bond. PA trigger see below
BLM	Closure Plan	Bonding requirements for operations in the ational Forest	Part of Plan of Operations	Prepare a plan and manage according to the plan; update as required
BLM	PA Review	Plan of Operations approval is a major federal action triggering PA review, including IS preparation	Process can take years estimate 3-5 in this case	Substantial public involvement, comprehensive baseline studies, evaluation of alternatives and multiple agency involvement, culminating in a ROD
PA. Arizona has primacy see below.	CWA PD S storm water permits construction and industrial operations	Regulates releases of storm water runoff through requirement for SWPPPs	A	Requires BMPs for erosion and sedimentation control, and control and removal of industrial contaminants, such as oil and grease or byproducts
PA	Hazardous Waste RCRA, RCRA ID umber	Waste activities and disposal of hazardous waste	One year unlikely to be required	Manifests, reporting and inspections

A ency	tem	Description	Time e uired	Conditions
USAC	CWA Section 404 Permit	Discharge of fill material to waters of the United States, including most major washes	One year	Various
Mine Safety and Health Administration (MSHA)	MSHA umber	Miner registration number	Immediate	Operate following MSHA rules
US Fish and Wildlife Service	ndangered species taking	Only required if endangered species will die or require relocation.	See PA	Can be covered by overall PA review
Department of Transportation	Hazardous Materials Transportation Registration	Shipment of hazardous materials	Immediate	Labelling, packaging and shipping
Bureau of Alcohol, Tobacco and Firearms	Blasting Operator Registration	Registration of all personnel that may handle blasting materials	Months	Background and fingerprint checks of all persons with access; update as required by Federal Agencies
Federal Communications Commission	Radio Licenses for Industrial Business Pool Conventional Use	Communications equipment must be licensed	Weeks to months	Follow license requirements
tate Permits				
AD Q	Aquifer Protection Permit	Dumps, tailings, leaching facilities and processing plant for ground water protection	2 years, including hydrogeologic study	Inspections, monitoring, maintenance, and reporting
AD Q	Air Quality Permit	Mobile and stationary emission sources	1 year	Inspections, monitoring, maintenance and reporting
AD Q	A PD S Construction Storm Water Permit General Storm Water Permit	Discharge of storm water	Immediate	Delineated in storm water management plan
AD Q	Solid Waste Management Inventory umber	Landfill and waste area requirements	Immediate	Monitoring, maintenance and operations
AD Q	Hazardous Waste Management umber	Management of hazardous waste	Immediate	Monitoring, maintenance and operations
AD Q	Waste Tire Cell Registration	Management of off-road tires greater than 1 m (3 ft) in diameter	Immediate	Annual reporting and cover requirements

A ency	tem	Description	Time e uired	Conditions
ADWR	Groundwater Withdrawal Permits	Groundwater withdrawal rights	6 months	Groundwater withdrawal annual reporting required
ADWR	Safety of Dams Permit	Requirements for dam construction	3 months	Monitoring, maintenance
ASMI	Reclamation Plan	Post-mining land uses and plans for regrading	3 months	Annual updates

ote IS nvironmental Impact Statement
PA ational nvironmental Policy Act
ROD Record of Decision
PA nvironmental Protection Agency
CWA Clean Water Act
PD S ational Pollutant Discharge limination System
SWPPPs Storm Water Pollution Prevents Plans
RCRA Resource Conservation and Recovery Act
USAC US Army Corps of ngineers
MSHA Mine Safety and Health Administration
AD Q Arizona Department of nvironmental Quality
ADWR Arizona Department of Water Resources
ASMI Arizona State Mine Inspector

20.2 LOCATION

The Project is located in Mohave County, Arizona, about 56 km (35 miles) over unpaved roads from the small community of Wikieup, which is located about 201 km (125 miles) northwest of Phoenix, Arizona. Table 20.2 identifies the project area legal descriptions.

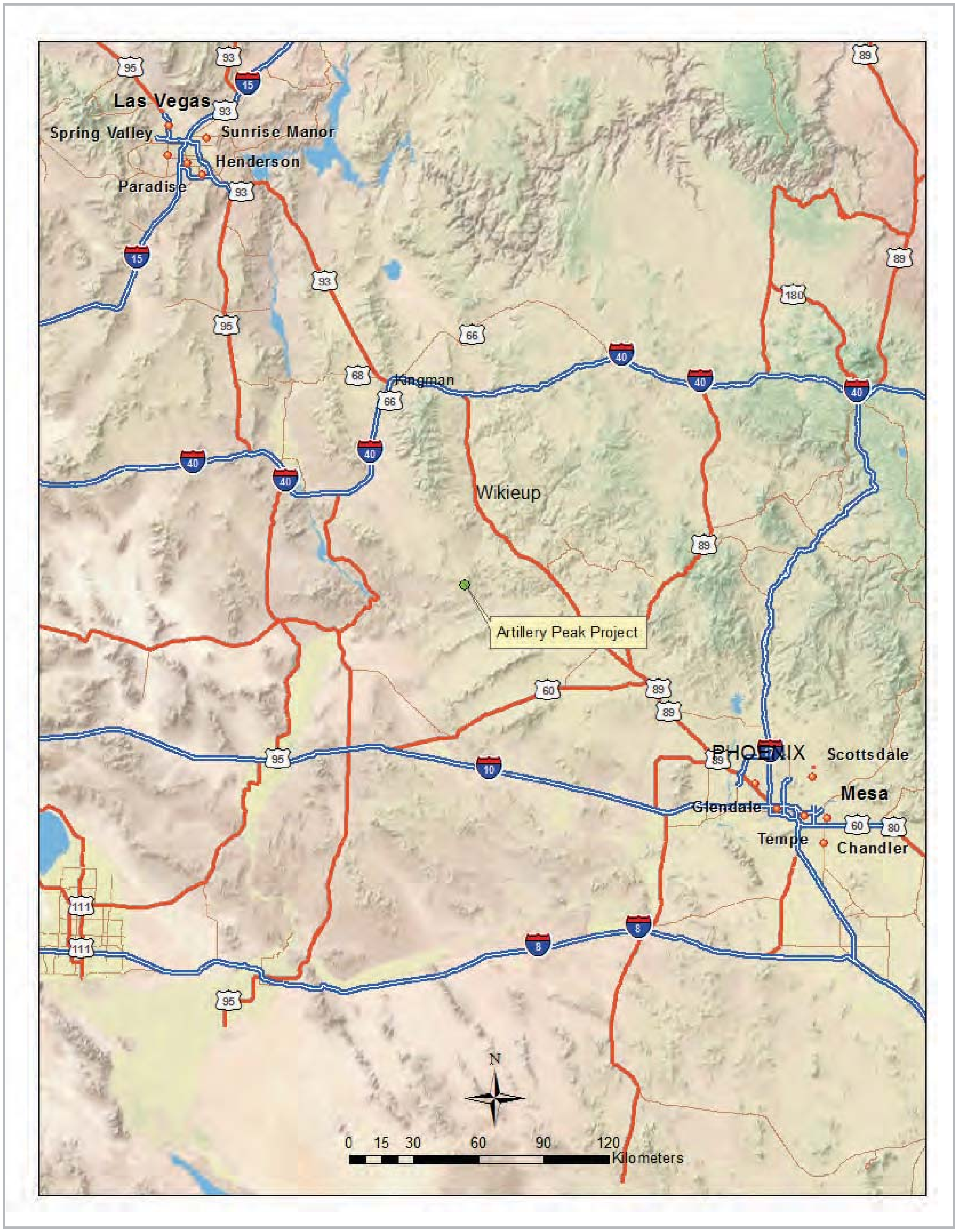
Table 0. Pro ect Area e al Descriptions

Topo raphic ection umbers	To nship and an e
Sections 22, 26, 27, 35, 36	Township 13 orth, Range 14 West
Sections 1, 5, 8, 9, 16, 21, 22, 26, 27, 35, 36	Township 12 orth, Range 14 West
Sections 6, 7, 8, 16, 17,19, 21, 22, 27, 28, 30, 31, 32, 33	Township 12 orth, Range 13 West
Section 1	Township 11 orth, Range 14 West
Sections 1, 2, 3, 4, 5, 6, 10, 11, 12, 14	Township 11 orth, Range 13 West

The UTM (AD 83) coordinates of the approximate centre of the patent claim block are 256,552 , 3,804,124 .

Figure 20.1 shows the general location of the Project.

i ure 0.1 General ocation ap



20.3 PERMITTING FEDERAL PROGRAMS

20.3.1 E ERA A P I A MA A EME T A T 19 6 PMA

The FLPMA provides the basis for the Bureau of Land Management (BLM) surface management regulations 43 C.F.R. 3809. It directs the Secretary of the Interior to take any action necessary by regulation or otherwise to prevent unnecessary or undue degradation of the public lands. The Act provides the basis for mining claim surface management regulations. It also sets forth the requirements for mining claim recordation.

Section 302 (Title 3) of the Act allows the Secretary of the Interior to authorize use, occupancy, and development of federal lands. This section provides the foundation for the BLM's 43 C.F.R. 3715 Mining Claim Use and Occupancy regulations.

Activities that ordinarily result in no or negligible disturbance of the public lands or resources are termed casual use. In general, the operator may engage in casual use activities without consulting, notifying or seeking approval from the BLM.

For exploration activity greater than casual use which causes surface disturbance of 2.02 ha (5 acres) or less of public lands, the operator must file a complete otice with the responsible BLM Field Office. otice is for exploration only, and a maximum of 1,000 tons may be removed for testing.

A Plan of Operations is required for surface disturbances affecting over 2.02 ha (5 acres).

20.3.2 ATI A E IR ME TA P I A T 1969

The ational nvironmental Policy Act (PA) plays a fundamental role in environmental protection; it applies to projects involving a federal decision, such as approval of a Plan of Operations. The purpose of the Act is to declare a national policy, which will encourage productive and enjoyable harmony between man and his environment, and to establish a Council of nvironmental Quality.

PA forms the basis of the federal government's decision-making process by requiring full and complete disclosure of the impacts of the proposed action(s) on the human environment.

Generally, the PA process begins with an initial review of the project. If the proposed project does not have readily apparent environmental consequences and is not categorically excluded from the PA analysis, the agency will prepare an nvironmental Assessment (A). This document analyzes the environmental impacts of the project, and ends with either a Finding of o Significant Impact (FO SI), or a finding that there are significant impacts. This latter finding then requires the preparation of an nvironmental Impact Statement (IS) with full public disclosure of those impacts (a FO SI would lead to the approval of the proposal without further PA analysis). If the agency anticipates that an undertaking may significantly affect the environment, or if a project is environmentally controversial, the agency may choose to prepare an IS without first preparing an A.

PA is the disclosure authority, not the decision-making authority. In this case, the BLM is the lead agency, with input and review from US PA and USFW as well.

A partial list of resources that will be evaluated in the Project's IS include

- air quality
- water quality and quantity transportation
- terrestrial wildlife, fisheries and migratory birds
- sensitive species
- socioeconomics
- wetlands
- visual resources, recreation, noise and vibration
- hazardous materials and hazardous waste
- historic trails
- cultural resources
- ative American concerns and traditional cultural places.

Other subjects may be evaluated based on public input into the process.

Requirements under PA pose the greatest permitting challenges in terms of cost and length of time required. The IS process requires intensive public involvement and a series of administrative and technical stages. The time required from beginning the PA process to issuance of a final Record of Decision (ROD) could be from one to five years, or more.

20.3.3 EA ATER A T

The discharge of solid material to waters of the US is regulated by the US Army Corps of ngineers (USAC), under Section 404 of the Clean Water Act (CWA). The State of Arizona plays a peripheral role in this program, providing certification of water quality in support of the permitting process. Because implementation and enforcement of the CWA is ultimately the responsibility of the PA, the PA can influence the Corps' permitting decisions.

Two primary permitting avenues are available to Project proponents under Section 404 the Individual Permit (IP), and the ationwide Permit (WP). WPs are available for certain specified categories and sizes of disturbance that result in only minimal impact to the aquatic environment. IPs are required for larger projects or projects whose activities are not covered by the WP program. The effort to obtain a Section 404 permit varies considerably both in time and cost, depending on the type and extent of the impacts. It should be noted that PA analysis, as either an A or IS, is required for an IP.

Typically, the project must be demonstrably the least environmentally damaging preferred alternative to obtain approval from the USAC . In addition, mitigation may be required to offset impacts to waters of the US.

Corps of ngineers (CO) Section 404 Dredge and Fill permits should also be reasonably straightforward and not particularly costly, since the PA process has already been triggered by the Plan of Operations approval requirement.

20.4 STATE OF ARI ONA PERMITS

The environmental permitting programs most critical for mine development in Arizona are the Aquifer Protection Permit (APP) program and the air quality permit program. Both fall within the jurisdiction of the Arizona Department of nvironmental Quality (AD Q). Arizona's mine reclamation program falls under the State Mine Inspector's jurisdiction. It is a lesser, but still important permit program.

20. .1 A I ER PR TE TI PERMITS

The AD Q is responsible for issuing an APP to facilities that may potentially discharge pollutants, which may adversely impact groundwater quality. As part of the permit process, applicants must demonstrate that their facilities are designed to be protective of groundwater quality, either through adoption of presumptive best available demonstrated control technology (BADCT) or equivalent facility-specific design.

The APP program also requires that the project proponent demonstrate its financial capability of to design, construct, operate, close and assure post-closure care of the facility. Further, hydrogeologic characterization of the site, groundwater quality monitoring, a contingency plan (in case of facility failure), a closure strategy and a post-closure monitoring and maintenance plan, are all required for the APP application.

APPs cover practices posing a hazard to ground water quality. Artillery Peak facilities subject to APPs would be the tailings and waste rock storage facilities, process water ponds, landfills and other surface impoundments.

20. .2 EA ATER A T SE TI 02 A P ES

Discharges of process water and storm water to waters of the US are regulated at the federal level by the PA under the ational Pollutant Discharge limination System (PD S), as outlined in Section 402 of the CWA. The PA delegated this program to the AD Q, which manages these discharges under the Arizona Pollutant Discharge limination System (A PD S). Individual A PD S permits must be obtained for each point source discharge from an operating mine site. Permits include effluent limitations, consisting of both numeric and narrative standards. The numeric limitations restrict quantities, rates and concentrations of pollutants that may be present in the discharge, and can be either technology- or water quality-based. Technology-based standards require usage of available pollution control technology, while water quality-based standards protect ambient water quality by requiring the discharger to achieve the applicable numeric standard as established by AD Q.

Storm water discharges from mining facilities require a multi-sector general permit (MSGP) for all project phases, including exploration and construction. The general storm water permit program requires a project proponent to prepare a Storm Water Pollution Prevention Plan (SWPPP), submit a notice of intent (OI) to discharge storm water, install appropriate best management practices (BMPs) and conduct regular inspections of the site in accordance with the SWPPP.

MSGP coverage also requires the establishment of discharge outfalls, and may require regular analytical monitoring of storm water discharges. Depending upon the nature of discharges from the project area, individual A PD S coverage may be required as well.

20. .3 AIR A IT

The PA under the Clean Air Act (CAA) regulates air quality at the federal level. ational Ambient Air Quality Standards (AAQS) have been established for each of the criteria pollutants of ozone, carbon monoxide, nitrogen dioxide, sulphur dioxide, particulate matter less than 2.5 m and less than 10 m aerodynamic diameter and lead. Authority for air quality permitting has been delegated by the PA to the A D Q.

Air emissions are regulated under the CAA in the context of the AAQS. The law and regulations differentiate between mobile and stationary sources, as well as between new and existing facilities. ew or modified existing stationary sources must meet performance standards (ew Source Performance Standards (SPS)) established by the PA for certain source categories. The standard of performance for a particular facility is based on the application of the best available system of emission reduction, taking cost into consideration. ew major sources are subject to preconstruction review, with different standards and levels of review applied to facilities proposed within attainment areas (Prevention of Significant Deterioration requirements) and non-attainment or non-classifiable areas (ew Source Review requirements).

missions of hazardous air pollutants (HAPs) are also regulated under the CAA. The PA has established HAP standards for both specific pollutants and for families of pollutants not emitted by a sufficient number of sources to justify development of a specific AAQS for that pollutant, but which can have serious human health implications. The CAA requires identification of major HAP sources, as well as area sources (sources below the volumetric thresholds for major sources). Sources are required to obtain permits for emitting any of the HAPs, again with variance between new and existing source standards.

The permitting components of the CAA for stationary sources are described in Title V of the CAA; thus, air emission operating permits are commonly referred to as Title V permits. These permits comprehensively address all relevant air emissions limitations, monitoring and reporting requirements, HAPs and SPS.

AD Q has established three other classes of permits

- Class I permits are required for major sources, solid waste incineration units, affected sources (a defined term) and any source in a category designated by the PA Administrator and adopted by the A D Q Director. Mining operations qualify as Class I major sources.
- Class II permits are required for construction or modification of sources that otherwise do not qualify for Class I permits, but that emit pollutants above certain thresholds, or for sources that are certain types of facilities.
- General Permits are pre-approved permits available for a specific class of sources, such as common types of facilities like gasoline stations.

In the arid southwest, fugitive emissions are a visual and environmental problem if not properly controlled. In an effort to conserve water and protect watershed areas, alternative forms of dust control are worth investigating. Capping, seeding and land management techniques will be used on waste rock piles and storage areas.

20.5 BASELINE CONDITIONS

20.5.1 IMATE, AIR A IT A ISE

CLIMATE

The site has a dry, desert climate that receives an average of less than 15.24 cm (6") of precipitation annually. During the summer months, temperatures can reach up to 49°C (120°F). Torrential downpours can also occur, mostly during the summer months, where washes become flooded and roads impassable. Annual pan evaporation exceeds 254 cm (100") per year, according to the Climate Atlas of the United States. During the winter months, frost is rare, and temperatures are fairly mild (T A 2010).

The nearest weather data station is in Wikieup. Table 20.3 shows average temperatures and rainfall data for the years 1981-2010 in Wikieup. The record high temperature was 48°C on July 28, 1995. The record low temperature was -9°C on February 27, 1986. The highest recorded daily rainfall was 76.2 mm on October 21, 2004.

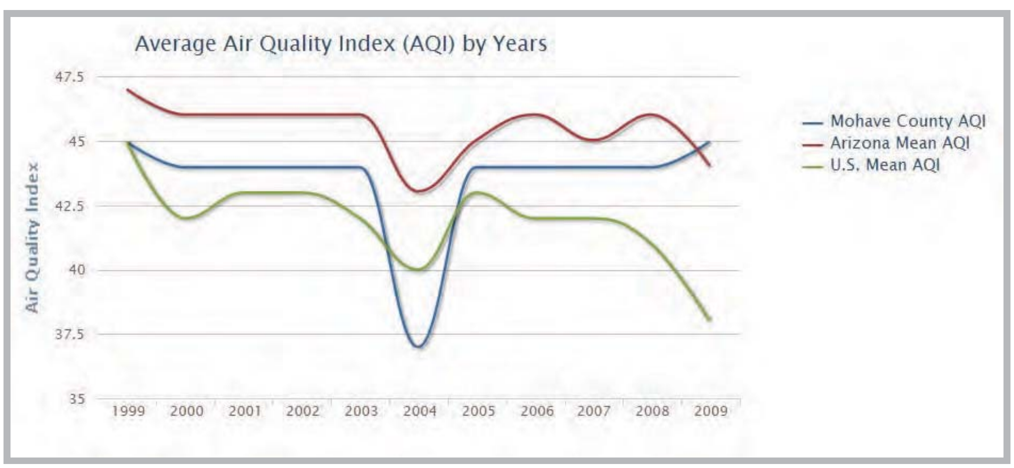
Table 0. Avera e Temperature in i eup, Ari ona

onth	Avera e Temperature C	Avera e i h C	Avera e o C	Avera e Precipitation mm
January	10	19	1	33
February	11	20	2	41
March	14	24	4	30
April	17	28	6	8
May	22	33	11	4
June	27	38	15	2
July	31	41	21	13
August	30	40	20	27
September	26	36	16	19
October	20	30	10	19
ovember	14	23	4	20
December	9	18	0	21
Avera e	19	9	9	Total 6

AIR ALITY

Air quality in this remote desert region is good. AD Q classifies the county as attainment for all priority pollutants. Figure 20.2 compares the Air Quality Index (AQI) for Mohave County to the mean AQIs for both the State of Arizona and the US. Any value below 50 is considered good.

i ure 0. ohave County Air uality nde



NOISE

o noise level data exists, but no permanent noise sources exist in the area. The site visit confirmed the low noise level.

20.5.2 ATER A IT

RO N ATER

Little historical groundwater quality information is available. Three locations near the Project site were sampled in July 2011. Sample sites included two wells (Huffman and Well 5), and a surface water pond known as the Adit Pond. The origin of the Adit Pond is being investigated; it may be a spring. The water sample from the Huffman well had the lowest total dissolved solids (TDS) and sulphate concentrations of the three samples, indicating that the water would be the least likely to produce gypsum scale, which could be difficult to remove from process vessels and piping. The water sample from Well 5 showed TDS concentrations similar to the Huffman well, but also contained higher concentrations of calcium and bicarbonate, indicating that this could produce gypsum and calcite scale. The water samples had concentrations of arsenic ranging from 0.24 to 0.70 mg L, which exceeds the AWQS for arsenic of 0.05 mg L. Other than these results, the water sample results indicated that the water is suitable for use in the process, though not necessarily suitable as potable water.

S RFACE ATER

o perennial streams exist in the project area. Surface waters within the area are characterized as washes or arroyos, with intermittent flow occurring only in response to rainfall events.

The site is located within the Bill Williams Watershed. Within the watershed, both the Big Sandy River and the Bill Williams River have been assessed by AD Q for attainment of stream quality standards. The Bill Williams River is classified as impaired by the AD Q due to ammonia, low dissolved oxygen and high pH. These contaminants are associated with farming and irrigation return flows. The Big Sandy River is not listed as impaired.

ETLAN S AN ATERS OF THE S

Tetra Tech completed an initial desktop evaluation, using readily available data from the ational Wetlands Inventory (WI) and the ational Hydrography Data (HD), in the summer of 2011. The results of this evaluation indicated the Patented (private land) portion of the Project area, along with the power line corridor; do not contain any perennial streams or springs. However, the evaluation indicates the presence of previously mapped wetlands, freshwater ponds and intermittent and ephemeral drainages within the assessed areas. In addition, the power line corridor crosses numerous drainages that support areas classified as riverine wetlands by the WI. Tetra Tech recommended and has initiated a protocol-level Waters of the US and Wetlands survey over the Project area to determine whether jurisdictional wetlands and waters of the US are present and could be impacted by proposed activities (Tetra Tech 2011).

20.5.3 TERRAI , S I S A E

The Project area topography consists of mesas separated by entrenched intermittent drainages, large alluvial fans, mountains and hills, encompassed by the Rawhide and Artillery Mountain Ranges. This area lies within two major desert zones the Lower Colorado River Valley, and the Arizona Upland subdivision. The Lower Colorado River Valley is the largest and most arid region of the Sonoran Desert, which generally consists of sparse and open vegetation with limited availability of water resources. Steep rocky slopes characterize the Arizona Upland subdivision. The drainage patterns in the area are dendritic, consisting of small channels that carry runoff to the Bill Williams and Big Sandy rivers. This also includes the Alamo Lake Reservoir (nvironmental Planning Group 2011).

The mineralization in the project area consists of exposed Paleozoic limestone and quartzite, with underlying Precambrian granite, schist and gneiss. The overlying geology contains several thousand feet of Tertiary sediments and igneous flows. The Tertiary deposits have been subdivided into formations Chapin Wash, Cobwebb Basalt, Artillery, Sandtrap Conglomerate, unnamed cap basalt and unnamed volcanics (nvironmental Planning Group 2011).

The Artillery bed is estimated to be 782 m (2,500 ft) at its thickest point in the formation. Artillery mineralization consists of widespread basalt, and closed basin deposits of sandstone, shale, conglomerate, tuff, clay and a portion of limestone. The mineralization of the Chapin Wash formation contains clay and alluvial-fan deposits comprising conglomerates, mudstone, sandstone and clay. Chapin Wash formation contains manganese-bearing beds and is up to 457 m (1,500 ft) in thickness. The beds of the formations range in color from black to reddish brown to pink. The black beds consist of manganese oxides and reddish-brown iron oxides. A layer of basalt up to 76 m (250 ft) thick, known as Cobwebb Basalt, caps the Chapin Wash formation. Cobwebb Basalt is overlain by a light to dark red conglomerate consisting of clay, sandstone and pebbles called the Sandtrap Conglomerate (nvironmental Planning Group 2011).

There are five major faults in the area, which strike northwest across the region and have a vertical displacement of as much as 91 m (300 ft). As a result, the underlying manganiferous outcrops and rock have been exposed. This combination of faulting and mineralization in the Artillery Mountain region leads to one of the largest reserves of manganese-bearing mineralized material in the United States (nvironmental Planning Group 2011).

20.5. I I E A E ETATI

ECOSYSTEMS

levations in the Project area range between 1,200 and 2,900 famsl. The Project area is located in two major desert vegetation biomes the Lower Colorado River Valley subdivision of the Sonoran desert scrub biotic community, which occurs in the lower portions of the Project area; and the Arizona Upland subdivision, which occurs at higher elevations (PG, 2011).

VE ETATION

Vegetation in the Project area is typical for these desert biomes, and varies by elevation and setting. Species observed during field investigations include creosote bush, bursage, paloverde, ocotillo, cholla, brittle brush, mesquite, Joshua trees, juniper, acacias, ironwood and various grasses. Cactus species include saguaro, beaver tail, hedgehog, prickly pear and pincushion. Tamarisk, an introduced species, was observed in several Project area drainages (PG 2011; T A 2010).

TERRESTRIAL IL LIFE

Wildlife expected to occur in the Project area includes various passerine birds, raptors, several bat species, reptiles, desert tortoise, rodents, small mammals (including rabbits, skunks and bobcats), and larger mammals (including wild donkeys, mule deer, peccary, coyotes, desert bighorn sheep and mountain lion). Chaparral cock (roadrunners), California quail and mourning doves were noted near Lake's Lake and Alamo Lake (PG 2011; T A 2010).

A ATIC IL LIFE

The Project area does not contain any established fisheries. However, several of the drainages that bisect the Project area flow towards Alamo Lake, located approximately 11 km (7 miles) to the southeast. The lake was created in 1968 with the construction of the Alamo Dam, which impounds the Bill Williams River. The lake supports a fishery of introduced species, including largemouth bass, channel catfish, crappie and bluegill. The lake fishery is managed by the Arizona Game and Fish Department (AGFD).

SPECIAL STAT S VE ETATION AN IL LIFE SPECIES

In August 2011, Tetra Tech and Harris nvironmental Group, Inc. (Harris) completed a limited evaluation of federal and state special status vegetation and wildlife species that may occur within the Project area. This evaluation included the completion of a desktop review of existing data, a limited site visit to identify habitats, and an assessment of vegetation and wildlife species listed and managed in accordance with the federal ndangered Species Act (SA) of 1973, as amended.

The evaluation indicated that a majority of the Federal or State listed species are unlikely to be present within the Project area, due to lack of suitable habitat or because the project area is outside the species geographic range. Federally listed avian species that may occur within the Project area on a transient basis include the yellow-billed cuckoo and the southwestern willow flycatcher. State listed avian species that may occur include the ferruginous hawk and the bald eagle. State listed amphibian species that may occur within the Project area include the lowland leopard frog. State listed mammals that may occur within the Project area include the spotted bat and the California leaf-nosed bat.

The reptilian species federally listed for Mohave County that will likely be of primary concern to US Fish and Wildlife Services (USFWS) within the Project area is the desert tortoise. The USFWS categorizes the Project area as general tortoise habitat, but not critical habitat. Tortoises were not observed during the August 2011 site visit; however, the prime tortoise survey period is generally during the spring season.

State listed salvage vegetation species that may occur within the Project area include the Bigelow onion, Missouri Cory cactus, clustered barrel cactus, Grand Canyon cottontop cactus, straw-top cholla, Kingman's prickly pear cactus, white-margined penstemon, cerbat beardtongue and Our Lord's Candle.

20.5.5 IS A A AEST ETI RES R ES

The Project site is located in a relatively remote and undisturbed area, characterized by rugged mountains and desert terrain. The nearest paved road is SR-93, located approximately 40 km (25 miles) east of the project site and a few miles from Alamo Lake. The nearest maintained road is Alamo Road, also known as County Road 15, a dirt road located approximately 1.6 km (1 mile) west of the Project site. The mine will be located in an isolated area that is not visible from the roadway. There is a concern that a portion of the mine may be visible from one portion of Alamo Lake, which is used for recreation.

20.5.6 RE REATI

Alamo Lake State Park is only about 11 km (7 miles) by air from the site. It appears accessible from the site on the northwest side by local dirt roads, but park facilities are on the opposite side; a drive of 102 km (64 miles), according to Google Maps.

The lake is located on the Bill Williams River where the Big Sandy River and Santa Maria River come together. It was created with the completion of the Alamo Dam in 1968. The USAC designed the earthen dam, primarily for flood control. During flood events, the lake basin is capable of capturing large amounts of water (including runoff from the project site) in a relatively short time.

The lake once recorded a vertical rise of 335 cm (11 ft) in one night. Unusually high flows during the late 1970s and through the 1980s increased the average size of the lake, helping to create one of Arizona s best fishing holes[1].

o other notable recreation facilities exist in the project area. However, like all rural desert areas in Arizona, dispersed recreation such as hunting and off-road vehicle use can be expected during certain seasons.

20.5. T RA RES R ES

A detailed Class III cultural resource survey completed by the nvironmental Planning Group (PG) during the spring and early summer of 2011 assessed potential impacts to cultural resources that could result from exploration drilling activities on lands administered by the BLM. These lands surround the area that will be mined. The results of this survey indicate that five sites are eligible for the ational Register of Historic Places and could be impacted by the proposed drilling activities. However, a proposed mitigation and treatment plan would allow these sites to be assessed and mitigated, with appropriate information and data collected such that exploration drilling could commence (PG, 2011).

20.5. S I E MI SETTI

The Project is located in a remote, relatively undeveloped portion of Mohave County.

The County includes five communities with populations over 4,000 the largest is Lake Havasu City (55,429), followed by Bullhead City (41,187), Kingman (28,823), ew Kingman Butler (4,922) and Colorado City (4,042). Kingman is the County seat.

Clearly, the Colorado River communities dominate the economy and social climate of the county. The closest towns by road to the project site are

- Wikieup .... 56 km (35 miles)
- Kingman.. 138 km (86 miles)
- Wickenberg (Maricopa County)........ 144 km (90 miles)
- Phoenix... 225 km (140 miles).

The 2010 census counted a population of 133 people in Wikieup. Table 20.4 shows the overall age distribution in Wikieup and in Mohave County.

[1] http azstateparks.com parks alla index.html

Table 0. i ieup Population by A e Group - 010 Census

	i ieup		ohave County	
	nhabitants		nhabitants	
Persons under 5 years	3	2.26	11,005	5.5
Persons under 18 years	24	18.05	30,260	15.12
Persons 18 to 64 years	81	60.90	112,263	56.08
Persons 65 years and over	25	18.80	45,658	23.31

The area has only minimal socioeconomic infrastructure. The Owens-Whitney lementary District has a single school with 29 students in grades K-8. High school students attend Kingman High School, a two-hour bus ride away.

The Mohave County Sheriff's office provides police service. There is no local police force.

The Kingman Regional Medical Center is the nearest hospital.

Unemployment is about 7 in the local census block, which includes Wikieup and the project site, and 11 in Mohave County overall.

20. BASELINE STUDIES

A multi-resource baseline study program would be implemented to collect the data required to support the completion of the federal and state agency permitting programs, and the anticipated environmental documentation process that will be required under PA. This baseline program could include, but is not limited to, studies for the following resources

- general vegetation
- general wildlife
- special status vegetation and wildlife species, including those species managed under the requirements of the SA of 1973, as amended
- invasive, non-native plant species including noxious weeds
- soils
- paleontology
- water quality and quantity, including surface and groundwater hydrology
- jurisdictional wetlands and waters of the US, as required by Section 404 of the federal CWA of 1977, as amended
- air quality, as required by the CAA of 1963, as amended

- cultural resources, as managed under the HPA of 1966, as amended, and the ARPA of 1979
- ative American traditional values, as regulated by various federal laws and regulations, including the AIRFA of 1978, as amended; the AGPRA of 1990; and xecutive Order 13175 Consultation and Coordination with Tribal Governments
- environmental justice, in accordance with xecutive Order 12898 Federal Actions to Address nvironmental Justice in Minority Populations and Low-Income Providers
- hazardous materials and solid waste, including waste rock, tailings and other materials expected to be generated during mining activities
- range management
- social and economic impacts
- recreation, particularly around use of Alamo Lake and surrounding camp and boat launch sites
- aesthetics, including noise and visual assessments
- geochemistry, including acid rock drainage.

This baseline program is being developed in consultation with the appropriate federal and state regulatory agency specialists, in order to ensure that the information is collected using approved procedures meeting appropriate data adequacy standards that support multi-federal and state agency permitting programs and the anticipated PA environmental documentation process.

20. EN IRONMENTAL EFFECTS AND MITIGATION

20. .1 AIR A IT

EFFECTS

The Project includes the following processes and anticipated pollutant emissions

- open pit mining operation air emissions, including fugitive emissions from mining vehicles' fuel combustion, and particulate (dust) emissions from mining, vehicle travel on unpaved roads, mineralized material stacking and stockpile operations
- mineralized material processing emissions, including particulate matter, VOCs and HAPs; the process burns sulphur to produce sulphur dioxide and sulphuric acid; emissions include sulphur oxides
- electrowinning process emissions, including sulphuric acid mist

- emergency generator (stationary source) combustion emissions, including nitrous oxides, sulphur dioxides, carbon monoxide, total suspended particulate (TSP), PM2.5 and PM10.

The project is not sufficiently advanced to allow estimation or prediction of potential or actual emissions. Based on other operations of similar size and complexity, it is safe to predict that potential emissions are significant, and that a substantial commitment of resources will be required for their control.

CONTROLS AN MITI ATION

Sufficient technology exists to eliminate, control and collect emissions from the various sources. Water sprays and collection systems can reduce potential fugitive dust to acceptable levels. Scrubbers and other control devices exist for control of aerosol and gaseous pollutants. Proper design, selection, operation and maintenance of air pollution controls will enable the project to proceed while maintaining acceptable air quality.

20. .2 ATER A IT

S RFACE ATER

Direct discharge of process-related waters, often termed impacted waters is forbidden by federal and state regulations. However, accidental releases or releases resulting from circumstances beyond the operator's control may result in discharges of process water to local waterways. Typically, this occurs as a result of extreme rainfall events that exceed design limits (typically the 24-hour, 100-year event).

Alamo Lake is about 11 km (7 miles) directly downstream along the main drainages from the site. Releases of process or other impacted waters could directly affect water quality in the lake. Good environmental practices dictate that operators prepare emergency response and contingency plans to minimize the potential for harmful releases.

Storm water permit conditions dictate controls and procedures to minimize the effects of non-point source releases.

RO N ATER

Groundwater could be affected by discharges from process and waste disposal units such as ponds, other surface impoundments, trenches, waste piles and other facilities where the potential exists for process or contaminated waters to enter the subsurface and, eventually, the groundwater.

Facilities and process units subject to the APP program will largely be designed and operated to contain contaminants and monitor for accidental releases.

20. .3 ASTE ISP SA

CO ISPOSAL OF ASTE AN TAILIN S

Open pit mine operations generate large quantities of waste material in the form of overburden, mine waste and tailings. The mine will excavate about 112 Mt of waste material and 56 Mt of filtered tailings at about 30 moisture.

e a a a ed e e de a m a e e ee C a e 13 e a e e e de ed m e e .

Material haracteri ation A A and SP P

An ongoing mineralized material-processing pilot program has generated three batches of tailings material. Samples of the tailings material have been collected and analyzed for acid-base accounting (ABA) and synthetic precipitation leaching procedure (SPLP) metals. Initially, three samples were collected from the first batch of tailings material processed in October 2011. The net neutralization potential (P) ratio indicated that the first three samples were slightly potentially acid generating. P range from -2.3 to -2.9 t calcium carbonate kt), which was also confirmed with the eutralization Potential Ratio (PR Acid eutralizing Potential divided by the Acid Generating Potential) (PRs ranged from 0.05 to 0.07). These results can be misleading because the samples contain no carbonate alkalinity, and limited sulphide sulphur. So, although the ABA testing results suggest that the samples could generate acid, it is unlikely that the acid generation would be significant. The samples contained arsenic ranging between 0.389 and 0.441 mg L and thallium ranging between 0.0122 and 00.0128, which are greater than the AWQS concentrations for arsenic (0.05 mg L) and thallium (0.002 mg L). It should be noted that the background groundwater quality samples have arsenic concentrations in the range of 0.24 to 0.70 mg L. These samples were expected to represent the final process; however, the process was refined after these samples were collected.

A second set of three samples was collected from the second batch of tailings material processed in ovember 2011. These samples were not acid generating (ABAs P ranged from 2 to 2.8 t calcium carbonate kt and PRs ranged from 3.2 to 5.7) because excess lime was introduced to the process. The arsenic concentrations ranged between 0.256 and 0.442 mg L, and thallium ranged between 0.00718 and 0.0114, which again exceeded the AWQS concentrations for arsenic (0.05 mg L) and thallium (0.002 mg L).

A third set of three tailings samples from a third batch of the process were collected and analyzed in January 2012. These samples were not acid generating (ABAs P ranged from 2.6 to 3.6 t calcium carbonate kt and PRs ranged from 2.4 to 3.1) as yet more lime was added in addition to other modifications. The arsenic concentrations ranged between 0.244 and 0.332 mg L, and thallium ranged between 0.0061 and 0.0065, which again exceeded the AWQS concentrations for arsenic (0.05 mg L) and thallium (0.002 mg L).

leven waste rock samples were tested in a similar manner.

ach of the waste rock samples were evaluated by ABA and one of the 11 samples, SS Sample 1, was identified as potentially acid-generating by the P criteria (P 0.4 t calcium carbonate kt), and PR criteria (PR 3). However, the sample did not contain more than 0.3 sulphur, reducing the potential for ARD, and the limited amount of carbonate present in the sample presents a false positive for potential acid generation. The remaining samples were identified as non-acid-generating using both the P and PR criteria.

The SPLP results for the waste rock samples indicated

- arsenic concentrations exceeded the AWQS in five samples
- beryllium concentrations exceeded the AWQS in two samples
- lead concentrations exceeded the AWQS in one sample.

Section 13 noted that the metallurgical processes intended for recovery of manganese and other by products would be evaluated during the feasibility stage for improvements in primary recovery of manganese and secondary isolation or removal of those metal by-products (arsenic, thallium, et. al.) that would be a concern as further refinements to disposal, containment or isolation of mineral waste streams. As the process is further evaluated during feasibility, additional process modifications, testing and samples will be collected and analyzed to establish additional characterization results for permitting of the tailings.

isposal Method

During the first six years of operation, filtered tailings and mine waste rock will be commingled and stored in a separate facility southeast of the pit. The TWSF will be an unlined pad in a roughly circular shape, covering about 60.83 ha (150.3 acres).

The TWSF will be located southeast of the pit and will be constructed with filtered tailings and waste rock generated from the beginning of mining and up to Year 6. The filtered tailings and waste rock will be alternately deposited and immediately mixed and compacted with a bulldozer. At or before Year 21, the material stored in this TWSF will be relocated to the mined sections of the pit. The disturbed area of the TWSF will be reclaimed to approximate pre-mining contours.

At end of mining and backfilling of the pit, a soil cover will be placed on top of the landform formed by the commingled tailings and waste rock. The soil cover will be designed to limit infiltration and erosion and to promote revegetation.

AMI regards this approach to offer an economical solution to managing site wastes at a level of environmental protection that may satisfy its permitting obligations. It will constitute managing a single waste stream that is intended to be returned to the open pit as part of the pit reclamation process. This particular approach has been selected for evaluation under the prefeasibility study effort, and will continue to be further evaluated during the feasibility study phase.

AD Q has considered a similar mine waste management system at the Rosemont Copper Mine and initially authorized the approach during the permitting phase. Although, the permit has been issued and later questioned by environmental protagonist groups, the AD Q is willing to consider such disposal practices, so long as the proposed methods demonstrate the ability to comply with the governing precepts of the BADCT waste management water resource protection requirements. The feasibility study effort will focus on improvements to the metallurgical processes and the mineral processing efforts under consideration, as well as any innovations later considered to be relevant to the metal recovery control technologies applied at the Artillery Peak Project. Further test work is required to develop a method of securing or stabilizing the material such that it will meet AD Q water quality criteria. AMI acknowledges that this disposal method may require slight modifications to minimize environmental and geotechnical risk as more data becomes available, but presents a reasonable estimate of the necessary operational costs for waste rock and tailings disposal. For this reason, the described method is a reasonable basis for the prefeasibility study cost estimate in order to help refine the waste disposal methods ultimately selected for use on-site.

20. . I I E A E ETATI

The operation will disturb over 202.42 ha (500 acres) of barren desert with limited vegetation and wildlife populations. Construction of the mine and facilities will remove vegetation and displace wildlife for the life of the mine from the 500 acre area.

Reclamation will restore nearly all the area to its pre-mining use for livestock grazing. ative plants will re-establish themselves, and native wildlife will return in roughly the original density.

20. .5 IS A A AEST ETI

Some Project facilities will be visible from Alamo Lake, a distance of about 11 km (7 miles). This lake is primarily used for fishing. The presence of the facility is not likely to discourage visitors or disturb any valuable view sheds. The area is not known for scenic beauty, and the presence of the mine will not subtract from aesthetic values to any appreciable degree.

20. .6 RE REATI

Alamo Lake is the only notable recreation resource in the area. Uncontrolled runoff and or accidental process water releases could reach the lake and potentially cause harm to fish populations and fishing activity. Proper design and operation of the facilities will minimize this possibility to an acceptable level.

20. . T RA RES R ES

Cultural resource surveys conducted in the area for the exploration-drilling program found several sites are eligible for the ational Register of Historic Places, and could be impacted by the proposed drilling activities. Presumably, more sites exist in the larger operations area. However, a proposed mitigation and treatment plan would allow these sites to be assessed and mitigated, with appropriate information and data collected such that the project can move forward without delay.

20. . S I E MI E E TS

By far the most significant socioeconomic effect will be the positive economic impact to the area.

Mine development can potentially place a considerable burden on local services and utilities; however, modern mines are careful to mitigate these affects by helping with education, medical facilities and police protection, as appropriate.

Additionally, mine development frequently results in upgrades to utility services for the entire community.

20. MINE CLOSURE

20. .1 RE AMATI P A I

A mine site reclamation and closure plan will be prepared and costed for the project as the review of mining rates, mineral processing methods, site facility layouts and disturbance levels reach a level of confidence where the site inventories of impacted or potentially affected resources can be reliably determined.

The document will be prepared to meet the requirements of the land management agencies, which in this case include the Bureau of Land Management (BLM) and Arizona State Mine Reclamation Mine Inspector’s Office.

The reclamation plan will address an inventory of soils and other materials necessary to achieve adequate reclamation of site disturbances including soils (revegetation growth medium, soil covers for waste rock, tailings, etc.), sub-soil rock materials (used for structural stabilization of slopes and surfaces, erosion protection, etc.) and local vegetation for potential use in a revegetation plan for site disturbance recovery. These inventories will be intended to meet reclamation needs of reclaiming, regarding, covering and revegetating surface disturbances associated with the proposed mining operation.

A segment of the reclamation plan will address the need to decommission, decontaminate, demolish and salvage dispose of the debris in a manner consistent with prevailing mining and environmental practices. For example, this effort would be

directed towards the mill process circuit (pipe, tank, storage
cleanout decontamination), any equipment maintenance facilities (haul truck, mining equipment, mine staff vehicles) where petroleum products are used, and storage facilities (fuels, bulk chemicals, explosives).

As mine planning (roads, dumps, impoundments, etc.); facility construction (buildings, tanks, etc.) and layout plans (road alignments, diversion structures, etc.) develop, a reclamation plan, schedule and cost estimate will be started and ultimately refined to the requisite feasibility level when the final plans are finalized. Along with this planning effort, a reclamation and closure cost estimate and cash flow analysis (CAP , OP) will be prepared to provide support for the reclamation bond estimates (BLM, A) and the project cash flow model.

These plans, estimates and documents would be prepared for use in the feasibility study, but not necessarily for purposes of supporting permitting efforts without further refinements and restructuring to suit individual permit needs.

The proposed reclamation closure mitigation elements for the Project include concurrent reclamation of the facilities. Therefore, reclamation obligations will be incrementally reduced as the operation progresses.

The facilities that will be reclaimed include

- TWSF
- pit tailings north and south areas (PT S)
- mine plant
- haul road
- access road.

T SF AN PTNS RECLAMATION

Reclamation activities for the TWSF and PT S areas will include subgrade preparation, capping and seeding. The PT S stacking areas will have a maximum horizontal to vertical slope ratio of 2H 1V. Additionally, collection channels will be constructed for the PT S to provide erosion control. The collection channels will be 1.8 m (6 ft) wide and 0.3 m (1 ft) deep, spaced 100 m apart and lined with riprap. The lined collection channels will direct rainwater runoff from the top of the PT S to the natural downstream channels. All surfaces will be graded prior to capping. The surfaces will be capped with 150 mm to 300 mm (6" to 12") of alluvial material. The alluvium will be transported from stockpile areas within 1.6 km (1 mile) of the mine site and placed on the surface using scrapers. It is assumed that the alluvium is from the upper 300 mm (6") of soil that will be stripped from the pit, mine plant, and road areas at the beginning of the mine. Areas and hauling distances are based on the proposed grading plan for the Project (Tetra Tech, 2012). The placed alluvium will be graded with a bulldozer, and the graded area will be seeded with a seed mix of native perennial species containing a variety of grasses, trees, shrubs, wildflowers and cacti approved for that geographic area.

MINE PLANT HA L ACCESS ROA RECLAMATION

The mine plant and haul and access road areas reclamation will involve scarification, sub-grade preparation and seeding. ach area will be scarified to a depth of 0.6 m (2 ft) to break up existing compaction. Sub-grade preparation, will involve shallow scarification to produce a seedbed conducive to revegetation. The scarified areas will be seeded with a native-species seed mix.

COST ESTIMATES

Based on the assumptions stated above, the average cost for reclaiming disturbed areas at the Project site was estimated and included in the capital cost estimate for the Project. The total reclaimed area is anticipated to be 267.5 ha (661 acres). The cost estimate was based on the 2011 R.S. Means Heavy Construction Cost Data, 25th Annual dition (R.S. Means, 2011)

Mine closure will proceed accordance to a mine closure and reclamation plan that will be developed during the feasibility and mine design phase. As a component of the overall environmental stewardship of AMI the mine closure and reclamation plan will be developed to exceed regulatory requirements by using a concurrent reclamation approach. This approach provides a template for development of ongoing mitigation measures that will be developed during the design and operations phases of the Project.

Major elements of the reclamation and closure plan are dictated by regulatory requirements contained in the Arizona Mined Land Reclamation Act, the Plan of Operations to be developed as part of environmental clearance for the BLM, the Arizona APP, and the mining SGP. Although other regulatory requirements may contribute mitigation elements, these regulatory programs constitute the framework for the reclamation plan, which will serve to reduce or eliminate environmental impacts from the mine during and after closure.

21.0 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COST ESTIMATE

21.1.1 S MMAR

Tetra Tech has developed a capital cost estimate for the Project, with an expected accuracy range of 25 . The estimate consists of four main components

- direct costs
- indirect costs
- contingency
- Owner's costs.

The Project capital cost is estimated to be 476,972,678. All capital costs are expressed in United States dollars, unless stated otherwise. The capital cost summary and its distribution by area are shown in Table 21.1.

Table 1.1 Capital Costs stimate

Description	abour Cost	aterial Cost	Construction uipment Cost	uipment Cost	Total Cost
Direct Costs					
Overall Site	5,544,314	7,857,713	7,979,416	-	21,381,443
Mining	1,104,890	463,350	783,380	14,427,910	16,779,530
Milling	42,830,133	51,432,930	8,214,060	137,484,598	239,961,721
Mine Site Utilities	11,449,809	18,630,371	2,606,053	11,181,061	43,867,294
Mine Site Buildings	934,791	2,847,876	280,095	1,583,013	5,645,775
Tailings	767,793	995,202	217,281	-	1,980,276
Plant Mobile quipment	49,350	-	-	1,276,590	1,325,940
Temporary Facilities	62,040	2,006,125	44,000	-	2,112,165
ubtotal Direct Costs					,05 ,1
ndirect Costs	2,053,440	79,238,297	-	-	81,291,737
O ner s Costs	2,472,000	7,261,103	-	-	9,733,103
Contin ency	-	-	-	-	52,893,694
Total					6,9 ,6

21.1.2 ESTIMATE ASE ATE, E A E RATE A A I IT PERI

Tetra Tech prepared this capital cost estimate with a base date of Q2 2012. o escalation beyond Q2 2012 was applied to the estimate.

The budget quotes used in this estimate were obtained in Q1 2012, and have a validity period of 90 days.

21.1.3 ESTIMATE APPR A

The capital cost estimate is based on the assembly and structure per the Project s work breakdown structure (WBS), consisting of the main areas as shown in Table 21.2.

Table 1. Pro ect

Description	Area	Area Description
11 Overall Site	111	Overall Site
21 Mining	211	Mining
31 Process Plant	311	Process Plant
	312	Leaching and Heat Recovery
	313	Purification, Precipitation and lectrowinning
	314	Storage Area
	315	Reagent
32 Mine Site Utilities	321	Power
	322	Fuel
	323	Water
	324	Waste
33 Mine Site Buildings		Buildings
34 Tailings		Tailings
35 Plant Mobile quipment		Surface Mobile quipment
37 Temporary Facilities		Camp
91 Indirects		Indirects
98 Owner's Costs		Owner's Costs
99 Contingency		Contingency

The capital cost estimate is based on

- budget quotations for all tagged major equipment
- Tetra Tech s in-house database for non-tagged major and other equipment
- preliminary material take-offs by discipline, as required
- electrical, platework, instrumentation and piping expressed as a percentage.

All equipment and material costs are included as free carrier (FCA) or free board marine (FOB) manufacturer plant, and exclusive of spare parts, taxes, duties, freight and packaging. These costs, if appropriate, are covered in the indirect section of the estimate.

quipment items valued under 100,000 may have been priced from in-house data and previous Project data if pricing was recently updated, unless the equipment is of a specialized nature.

stimated installation hours are based on in-house experience and cost references.

All equipment and material costs are based on FCA manufacturer plant (I COT RMS 2010), and exclude spare parts, taxes, duties, freight and packaging.

The freight and spares costs are covered in the indirect section of the estimate as an allowance, based on a percentage of the value of materials and equipment.

Tetra Tech assumed that the construction labour and workweek to be on a 10-h d, 3-week-on 1-week-off rotation schedule, and that the construction camp will be accessible by land.

21.1. E EME TS STS

IRECT COSTS

abour Rates, Producti ity and Tra el Allo ances

A blended labour rate of 62.00 h was used throughout the estimate.

The labour rate includes

- vacation and statutory holiday pay
- fringe benefits and payroll burdens
- overtime and shift premiums
- small tools
- consumables
- personal protection equipment
- contractor's overhead and profit.

Tetra Tech has assumed that the labour source is available as follows

- 10 locally
- 70 regionally
- 20 out of region.

The source and availability of labour should be verified in the next phase of the study.

A productivity factor of 1.2 was applied to the labour portion of the estimate, to allow for the inefficiency of long work hours, poor climatic conditions and the labour rotation schedule.

TIES AN TA ES

Duties and taxes are included in the Owner's costs.

COST ASIS Y ISCIPLINE

ulk Earth orks, Including Site Preparation and Access and aul Roads

Bulk earthwork quantities are based on information provided by the civil discipline.

All excavated material is deemed in-rock excavation, which will require blasting; the estimate assumes that all excavated material will be stockpiled on-site within 5 km of the point of excavation.

Mining

Mining pre-stripping costs and equipment costs were included in the direct costs.

oncrete

Concrete quantities are based on estimated quantities, with no allowance included for over-pour or wastage.

Typically, all concrete is based on a 28 d compressive strength of 30 MPa. The average installed concrete unit rates for 30 MPa concrete used in the estimate are 823 m . Concrete unit rates include formwork, reinforcing steel, placement and finishing of concrete.

The concrete price was supplied by the client.

Structural Steel

Structural steel quantities are based on estimated quantities, with no allowance made for growth and wastage. The estimate includes allowances for cut-offs, bolts and connections.

An average supply unit rate of 5,248 t for fabricated steel, based on quotations from recent similar projects, was used in this estimate. The cost of the use of cranes was included for all tonnages, at a rate of 250 t.

Plate ork and iners

Preliminary quantities for platework and metal liners for tanks, launders, pump-boxes and chutes were estimated using recent similar projects and in-house data.

Mechanical

The preliminary equipment estimate for the processing plant was prepared based on the Project equipment list and process flow diagrams. The mechanical pricing was based on budgetary quotes obtained for the processing plant major equipment. All other mechanical equipment was based on recent quotes from similar projects.

A and ire Protection

HVAC and fire protection was included as a percentage of the process equipment cost, based on experience with recent similar projects.

ust ollection

The cost of major dust collection equipment is included in the mechanical section of the estimate.

Piping and al es

Piping and valves allowances are included as a percentage of the process equipment, based on experience with recent similar projects.

Electrical

lectrical allowances are included as a percentage of the process equipment, based on experience with recent similar projects.

Major equipment is based on recent Project information.

Instrumentation

Instrumentation is included as a percentage of the equipment list allowance assigned to each area, based on experience with recent similar projects.

uildings

The estimates for the engineered steel-framed buildings are based on a square metre rate, and pricing based on recent similar projects.

Process Mobile E uipment

The estimate includes for the provision of mobile equipment.

21.1.5 E I EERI , PR REME T A STR TI MA A EME T

An allowance based on a percentage of the direct costs is included.

21.1.6 TA ES A TIES

Taxes and duties on materials were excluded from the estimate.

21.1. ISTI S A REI T

Although no logistics study has been conducted for this Project, a provision of 6 of the total capital cost of the process equipment was included in the estimate for freight.

21.1. SPARES

An allowance has been included for spares, based on a percentage of the process equipment costs.

21.1.9 ER S STS

An allowance for Owner s costs was included, based on a percentage of direct costs.

21.1.10 E SI S

The following are not included in the capital cost estimate

- force majeure
- schedule delays, including but not limited to those caused by
  - major scope changes
  - unidentified ground conditions
  - labour disputes
  - environmental permitting activities
  - abnormally adverse weather conditions
- receipt of information beyond the control of the PCM contractors
- cost of financing (including interest incurred during construction)
- royalties
- schedule acceleration costs
- working capital
- laydown areas.

21.1.11 ASS MPTI S

Tetra Tech assumed the following in the preparation of this estimate

- all material and installation subcontracts will be competitively tendered on an open shop, lump sum basis
- site work will be continuous and unconstrained by the Owner or others
- skilled tradespersons, supervisors and contractors will be readily available when they are required
- the geotechnical nature of the site is assumed to be sound, uniform and able to support the intended structures and activities; adverse or unusual geotechnical conditions requiring piles or soil densification have not been allowed for in this estimate.

21.1.12 TI E

A contingency allowance of 14 was included in the estimate. Tetra Tech believes this contingency will adequately cover minor changes to the current scope expected during the next phase of the Project.

21.2 OPERATING COST ESTIMATE

21.2.1 S MMAR

The Project's average annual operating cost is estimated to be 1.012 lb of MM produced. Process cost includes credits from in situ generation of electricity power and sodium sulphate anhydrous by-product. All operating costs are expressed in United States dollars.

The average LOM operating cost is based on an average annual production rate of 50,000 t of MM.

The average LOM unit cost distribution after credits is presented in Table 21.3. The credits for electrical power generation and the by-product are detailed in Section 21.2.3.

Table 1. ummary of Avera e O Operatin Cost

Description	Operatin Cost /lb
Mining	0.200
Processing	0.751
G A	0.050
Surface Services	0.011
Total Operatin Cost	1.01

21.2.2 MI I

S MMARY

The average annual mining operating cost for the LOM is estimated at 2.84 t mined, or 9.83 t processed.

The LOM operating costs were estimated from equipment productivity calculations and mining fleet suppliers' hourly equipment cost estimates. Annual equipment utilization hours were derived from calculated available hours, minus estimated operating delays, and then multiplied by the hourly equipment cost to derive direct mining operating costs. Minor support functions were estimated based on estimated annual usage, and then added to the overall costs to produce a mine operating cost.

The mine operating costs for the LOM, including the pre-production period, are summarized in Table 21.4.

Table 1. stimated O Avera e Operatin Costs for inin Cdn

Description	ine Operatin Cost		
	Operatin Cost	ined /t	Processed /t
uipment			
Drilling	11,467,832	0.07	0.25
Blasting	23,059,810	0.15	0.51
Loading	28,034,863	0.18	0.62
Hauling	98,790,741	0.63	2.20
Support quipment	61,968,833	0.40	1.38
uipment ubtotal	, ,0 9	1.	.96
abour			
Mine Technical Services	16,997,006	0.11	0.38
Production Support	16,804,236	0.11	0.37
Mine Operations	110,624,862	0.71	2.46
Maintenance Support	18,664,804	0.12	0.41
Mine Maintenance	56,072,269	0.36	1.25
abour ubtotal	19,16 ,1 6	1. 1	.
Total inin Cost	, 5, 55	.	9.

ENERAL ESTIMATION APPROACH

Approximate productivity and operating hours were calculated for all major equipment. To arrive at the total operating costs, the operating hours were multiplied by the hourly labour, maintenance, major component repairs, fuel and consumables costs

The operating cost includes labour, maintenance, major component repairs, fuel and consumables costs.

F EL

All mining and support equipment are equipped with diesel engines. The equipment fuel consumption amount was obtained from either suppliers' quotations or equipment handbooks. Fuel costs were calculated for each of the major and support equipment, based on a fuel price of 0.87 L. The total fuel cost over the LOM is estimated to be 77 million, which corresponds to a unit cost of 0.49 t mined.

MINE CONS MA LE S PPLIES AN MATERIALS

stimated costs for consumables such as tires, explosives and drill accessories were primarily obtained from supplier quotations, as well as Tetra Tech s in-house database.

The total consumable supplies and materials cost over the LOM is estimated at 66.5 million, which corresponds to a unit cost of 0.43 t mined.

MAINTENANCE

Maintenance, major component and wear item repairs were obtained from supplier estimates, based on major component replacement costs over the projected life of the equipment.

The total maintenance cost over the LOM is estimated to be 80 million, which corresponds to a unit cost of 0.51 t mined.

SALARY AN HO RLY LA O R

Labour requirements were determined for each labour category. In the case of operators, labour requirements were estimated based on the number of on-duty equipment items. The maintenance labour estimate is based on historical ratios between equipment operators and maintenance mechanics and electricians. All other labour and staff were estimated from experience with existing mines and anticipated operating conditions for the Project.

Staff and hourly operating rates are based on the base rates and burden as estimated for Arizona and the adjacent state of evada. A benefit package of 43.7 was applied to salaried staff and hourly labour base rates. The labour burden consisted mainly of vacation, public holidays, medical and health insurance, retirement benefits and disability insurance.

The total combined labour cost for salaried staff and hourly labour over the LOM is estimated at 219 million, or 1.41 t mined.

MINE PERSONNEL

The expected mine personnel requirements over the LOM is shown in Table 21.5.

Table 1.5 Annual ine Personnel e uirements

Cate ory	Y0	Y1	Y	Y	Y	Y5	Y6 10	Y11 15	Y16 0	Y 1
Chief Mine ngineer	1	1	1	1	1	1	1	1	1	0
Mine ngineer	1	1	1	1	1	1	1	1	1	1
Senior Geologist	1	1	1	1	1	1	1	1	1	0
Surveyor	3	4	4	4	4	4	4	4	4	2
Clerk ngineering	1	1	1	1	1	1	1	1	1	0
ubtotal	7	8	8	8	8	8	8	8	8	3
Production Superintendent	1	1	1	1	1	1	1	1	1	0
Mine Foreman	4	4	4	4	4	4	4	4	4	4
quipment Trainer	1	1	1	1	1	1	1	1	1	0
Clerk Operations	1	1	1	1	1	1	1	1	1	1
ubtotal	7	7	7	7	7	7	7	7	7	5
Mine Operations	33	57	61	61	61	61	66	74	75	54
Maintenance Superintendent	1	1	1	1	1	1	1	1	1	0
Maintenance Supervisors	4	5	6	6	6	6	6	6	6	4
Clerk Maintenance	1	1	1	1	1	1	1	1	1	0
ubtotal	6	7	8	8	8	8	8	8	8	4
Mine Maintenance	11	27	32	32	32	32	35	38	38	20
Total	6	106	116	116	116	116	1	1 5	1 6	6

21.2.3 MI PERATI ST

SMMARY

The annual process operating cost for a maximum feed rate of 7,000 t d, and an MM production rate of 50,000 t a, is estimated to be 95.9 million, or 37.75 t milled, or 0.870 lb of MM, excluding the credits from heat recovery (electricity generation) and sodium sulphate anhydrous by-product. After these credits, the unit process cost per pound of MM produced is estimated to be 0.767. These unit production costs will vary with mill feed rates or mill feed grades.

The operating cost estimate includes

- staffing and salary wage level estimates, based on a survey of comparable operations in the US in 2012
- power consumption estimates, based on main equipment power draw estimates for the processing plant equipment

- major crusher liner consumables estimates based on estimates from crusher suppliers or in-house database
- reagent consumable estimates, based on reagent dosages determined by metallurgical tests from the Kemetco's test programs, and the quoted prices for the reagents in Q1 Q2 2012; and Client quoted major reagent prices
- operating and maintenance supply cost estimates, based on approximately 5 of major equipment capital costs, benchmarked against comparable operations in the US and worldwide
- credits from production of sodium sulphate anhydrous by-product and the electricity generation from heat recovery.

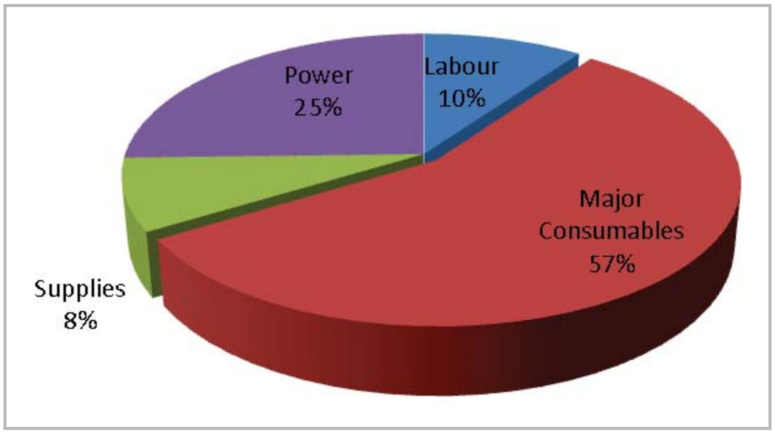
Table 21.6 provides a summary of estimated processing costs at a feed rate of 7,000 t d.

Table 1.6 ummary of Processin Costs ,000 t/d milled

Description	abour orce	Annual Cost	nit Cost	
			/lb n	/t milled
abour orce				
Operating Staff	10	1,105,000	0.010	0.435
Operating Labour	70	5,484,000	0.050	2.160
Maintenance	34	2,900,000	0.026	1.142
ubtotal abour orce	11	9, 9,000	0.0 6	.
a or Consumables				
Metal Consumables	-	100,000	0.001	0.039
Reagent Consumables	-	54,004,000	0.490	21.266
upplies				
Maintenance Supplies	-	5,774,000	0.052	2.274
Operating Supplies	-	2,150,000	0.020	0.847
ubtotal a or Consumables and upplies	-	6 ,0 ,000	0.56	. 6
Po er upply	-	24,360,000	0.221	9.592
Total Process	11	95, ,000	0. 0	. 55
Credits				
Sodium Sulphate Anhydrous	-	-	-0.086	-
lectricity Generation	-	-	-0.017	-
Total Process After Credit	-	-	0. 6	

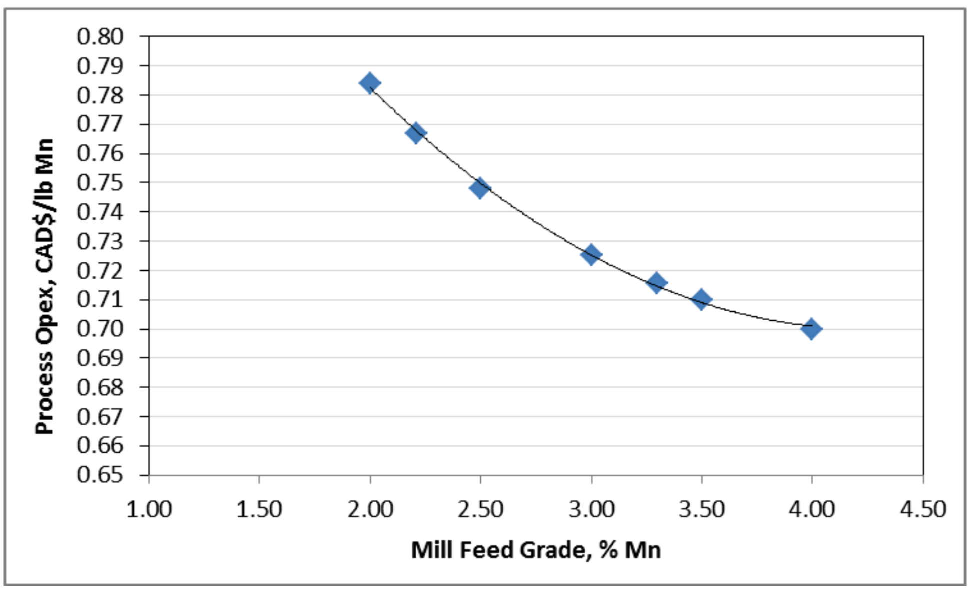
The major cost is from reagents that account approximately 57 of the total process operating cost. lectrical power consumption is the second major cost, which is approximately 25 of the total cost. Figure 21.1 shows the distribution of process operating costs at a process rate of 7,000 t d.

i ure 1.1 Process Operatin Cost Distribution



As indicated, unit manganese production cost (lb MM) is sensitive to feed grade. The cost reduces when mill feed grade is increased. The variation of unit process operating cost (lb MM) with feed grade (Mn) is illustrated in Figure 21.2.

i ure 1. Process OP vs. ill eed Grade



The following sections discuss the breakdown areas of the operating cost estimate based on 7,000 t d process rate.

LA O R COST

The annual total process labour costs are estimated to be 9.5 million, or 3.74 t milled, or 0.086 lb of MM. Salary estimates are based on 2012 labour rates for comparably sized operations in the area. The processing plant will be staffed with

114 personnel 10 in general supervisory and technical services, 70 in operational roles and 34 in maintenance roles.

The process labour cost breakdown is presented in Table 21.7 through Table 21.11. Loaded salary includes base salary and burden, which is estimated to be 43.7 of the base payment. The burden includes holiday and vacation payments, pension plan, life insurance plans, medical benefits, overtime payments and other benefits.

Table 1. Operatin Costs Processin abour orce

Description	abour orce	Annual Cost
General Management	5	654,000
Operators	68	5,356,000
Maintenance	34	2,900,000
Lab Technical Supports	7	579,000
Total abour	114	9,489,000

MILL MAINTENANCE AN OPERATIN S PPLIES OPERATIN COSTS

Annual maintenance supplies are estimated to cost 5.77 million, or 2.27 t milled, or 0.05 lb of MM. Annual operating supplies are estimated to be 2.15 million, or 0.85 t milled or 0.020 lb of MM. Table 21.8 shows the costs in detail.

Table 1. Operatin Costs aintenance and Operatin upplies

Area	Total Cost /a	nit Cost	
		/lb	/t milled
aintenance upplies			
Crushing	52,800	0.000	0.021
Leaching Thickening Filtration	2,485,000	0.023	0.979
Purification	100,000	0.001	0.039
W Peeling Plate Preparation	1,170,100	0.011	0.461
Sodium Sulphate Plant	816,500	0.007	0.322
Sulphur Dioxide Generation	850,000	0.008	0.335
Reagents	100,000	0.001	0.039
Miscellaneous Mill Supplies	100,000	0.001	0.039
Miscellaneous Building Complex Supplies	100,000	0.001	0.039
ubtotal aintenance upplies	5,774,400	0.052	2.274
Operatin upplies			
Filter Cloth	200,000	0.002	0.079
F Membranes	347,600	0.003	0.137
Anode Diaphragm Cloth	600,000	0.005	0.236
Cathode and Anode Plates	782,300	0.007	0.308
Assay Lab Supplies	100,000	0.001	0.039

Area	Total Cost /a	nit Cost	
		/lb	/t milled
(Mill CIL Mining Geological xploration)			
Mill Light Vehicle and Rental Vehicle Operations	20,000	0.000	0.008
Miscellaneous	100,000	0.001	0.039
ubtotal Operatin upplies	2,149,900	0.020	0.847
Total upplies	,9 , 00	0.0	.1 0

MA OR CONS MA LE COSTS

Annual total major consumables costs are estimated to be 54.1 million, or 21.3 t milled or 0.49 lb of MM. Table 21.9 summarizes the metal and reagent consumables operating costs.

The estimates of major consumables costs are based on the following

- consumption rate for crusher liners is based on data provided by the crusher suppliers or in-house data
- reagent consumable estimates, based on laboratory optimum dosages determined by metallurgical tests from test programs, and the quoted prices for reagents in Q1 Q2 2012 and also Client quoted major reagent prices.

Table 1.9 Operatin Costs a or Consumables Cdn

Consumables	Consumption /t milled	nit Cost /	Total Cost /a	nit Cost	
				/lb	/t milled
etal Consumables					
Liners	-	-	-	-	-
Primary Crusher Liners	-	-	50,000	0.000	0.020
Secondary Crusher Liners	-	-	50,000	0.000	0.020
ubtotal etal Consumables	-	-	100,000	0.001	0.039
Lime- eutralization	8.88	0.173	3,903,000	0.035	1.537
Lime-Purification	4.94	0.173	2,173,000	0.020	0.856
Sulphuric Acid (H_2SO_4) (included in liquid sulphur)	-	-	-	-	-
Sodium Carbonate (a_2CO_3)	40.10	0.266	27,050,000	0.25	10.65
Liquid Sulphur	25.10	0.185	11,743,000	0.107	4.62

Consumables	Consumption /t milled	nit Cost /	Total Cost /a	nit Cost	
				/lb	/t milled
Feed Tonnage Related	3.30	-	-	-	-
	21.70	-	-	-	-
Product Tonnage Related	-	0.185	1,555,000	0.014	0.61
	-	0.185	10,188,000	0.092	4.01
Ammonium Hydroxide (H_4OH)	2.95	0.450	3,375,000	0.031	1.329
Sodium Hydrosulphide (aHS)	0.01	1.400	43,000	0.000	0.017
Foam	0.01	4.000	81,000	0.001	0.032
Sodium Silicate (a_2SiO_3)	-	-	50,000	0.000	0.020
Flocculants	0.515	4.110	5,375,000	0.049	2.117
Flocculant 1	-	-	-	-	-
Flocculant 2	0.465	4.110	4,853,000	0.044	1.911
	0.050	4.110	522,000	0.005	0.206
Anti-scalant	0.010	6.300	160,000	0.001	0.063
Cement	-	-	50,000	0.000	0.020
ubtotal ea ents	-	-	54,004,000	0.49	21.266
Total a or Consumables	-	-	5 ,10 ,000	0. 9	1. 05

PO ER COST

Annual power supply is estimated to cost 24,359,645, or 9.592 t milled or 0.221 lb of MM.

21.2. E ERA A A MI ISTRATI E PERATI STS

Annual general and administrative (G A) expenses are estimated at approximately 5.5 million a, or 2.165 t milled or 0.050 lb of MM. The costs include

- personnel, general manager and staffing in accounting, purchasing and environmental departments, and other G A departments
- G A expenses, including insurance, administrative supplies, medical services, legal services, human resources related expenses, travelling, liaison committee sustainability, corporate office cost and external assay testing.

Table 21.10 shows a breakdown of the G A costs for personnel and general expenses.

Table 1.10 G A Operatin Costs Cdn

Description	abour orce	ase ate /a	oaded alary /a	Total Cost /a	nit Cost	
					/lb n	/t milled
G A abour orce						
G A	15	-	-	1,638,000	0.015	0.645
Warehouse First Aid	4			299,000	0.003	0.118
ubtotal abour	19	-	-	1,9 ,000	0.01	0. 6
G A penses						
General Office xpense	-	-	-	200,000	0.002	0.079
Computer Supplies, Including Software	-	-	-	50,000	0.000	0.020
Communications Tel, Fax, Internet, Postage	-	-	-	50,000	0.000	0.020
Travel	-	-	-	50,000	0.000	0.020
Consulting xternal Assays	-	-	-	200,000	0.002	0.079
nvironmental	-	-	-	200,000	0.002	0.079
Insurance	-	-	-	607,200	0.006	0.239
Regional Taxes Licences Allowance	-	-	-	100,000	0.001	0.039
Legal Services	-	-	-	50,000	0.000	0.020
Recruiting	-	-	-	50,000	0.000	0.020
ntertainment Membership	-	-	-	50,000	0.000	0.020
Medicals First Aid	-	-	-	25,000	0.000	0.010
Training Safety	-	-	-	100,000	0.001	0.039
Liaison Committee Sustainability	-	-	-	25,000	0.000	0.010
Corp. Office Cost	-	-	-	1,750,000	0.016	0.690
Others	-	-	-	50,000	0.000	0.020
ubtotal G A penses	-	-	-	,55 , 00	0.0	1. 0
Total	19	-	-	5, 95,000	0.050	.165

21.2.5 S R A E SER I ES PERATI STS

The site service cost is estimated at 0.493 t milled, or 0.011 lb of MM, or about 1.25 million per year. The estimate is summarized in Table 21.11.

The estimate includes

- personnel general surface services human power
- surface mobile equipment and light vehicle operations
- potable water and waste management
- general maintenance, including yards, roads, power line, fences and building maintenance
- hazardous waste disposal.

Table 1.11 urface ervices penses Cdn

Description	abour orce	ase alary /a	oaded alary /a	Annual Cost /a	nit Cost	
					/lb	/t milled
abour						
ubtotal Personnel		-	-	6.000	0.00	0.1
urface ervice penses						
Small Vehicles quipment	-	-	-	100,000	0.001	0.039
Potable Water Waste Management	-	-	-	175,000	0.002	0.069
Hazardous Waste Disposal	-	-	-	200,000	0.002	0.079
Supplies	-	-	-	100,000	0.001	0.039
Building Maintenance	-	-	-	150,000	0.001	0.059
Road Power Line Water Supply Maintenance	-	-	-	150,000	0.001	0.059
Surface Service Power	Included in process					
ubtotal penses	-	-	-	5,000	0.00	0. 5
Total urface ervice		-	-	1, 51,000	0.011	0. 9

22.0 ECONOMIC ANALYSIS

22.1 INTRODUCTION

Tetra Tech prepared an economic evaluation of the Project based on a pre-tax financial model that assumes 60 of MM production will be sold domestically in the US and 40 overseas. A three-case scenario was developed as follows

- The Base Case uses the weighted three-year historical average world and US MM prices as of April 30, 2012.
- The Alternate Case 1 uses the MM CPM expected price forecast for overseas sales, with price increased by 14 for domestic sales.
- The Alternate Case 4 uses the three-year historical average MM price reduced by 25 .

For the 21-year LOM at an average annual production of 50,000 t of MM, the following pre-tax financial results were calculated

- Base Case at MM price of 1.54 lb
  - 7.28 internal rate of return (IRR)
  - 10.3 year payback on 477 million initial capital
  - (23) million net present value (PV) at an 8 discount rate.
- Alternate Case 1 at MM average price of 1.93 lb
  - 19.95 internal rate of return (IRR)
  - 4.6 year payback on 477 million initial capital
  - 403 million net present value (PV) at an 8 discount rate.
- Alternate Case 4 at MM price of 1.16 lb
  - no internal rate of return (IRR)
  - no payback on 477 million initial capital
  - (371) million net present value (PV) at an 8 discount rate.

The weighted three-year historical average MM price was calculated as follows

- The three-year trailing average spot world price for manganese is 1.36 lb, and the three-year trailing average spot US price for manganese is 1.66 lb. The prices were obtained from the Metals Bulletin.

- Sixty percent of MM production will be sold in the US. Therefore, the three-year trailing average price results from taking the weighted average of the spot prices, as follows

1.36 x 0.4	0.544
1.66 x 0.6	0.996
Total	1.54

  The difference between the world MM price and the US MM price is primarily attributed to US demand for premium quality, selenium free product plus the application of the 14 tariff on MM imported into the US.

Price forecasts for Alternate Cases 1, 2, and 3 are based on the CPM Group MM market study report Manganese Metal Outlook, February 2012 . The CPM Group MM forecast prices are summarized in Table 22.1.

Table .1 CP orecast orld Prices /lb

Year	CP o er Case	CP pected	CP i her Case
2012	1.25	1.45	1.75
2013	1.35	1.65	2.00
2014	1.40	1.75	2.10
2015	1.45	1.85	2.05
2016	1.50	1.90	2.05
2017	1.65	1.95	2.15
2018	1.80	2.00	2.25
2019	1.90	2.10	2.35
2020	2.00	2.20	2.50
2021	2.10	2.30	2.60
2022 to 2035	1.45	1.68	1.90

The MM prices in the report are specified as projected world prices, and these prices were applied to 40 of MM production contemplated for overseas sales. The MM prices in the report, for the 60 of MM production contemplated for domestic sales, were increased by 14 to account for the tariff on MM imported into the US.

The price forecast used in Alternate Case 4 was determined from the Base Case price reduced by 25 .

Tetra Tech conducted sensitivity analyses to establish the sensitivity of the Project merit measures (PV, IRR and payback periods) to the main inputs.

22.2 PRE TA MODEL

22.2.1 MI E META PR TI I I A IA M E

The LOM mineralized material quantity, grade and MM production values are provided in Table 22.2.

Table . Production from the Artillery Pea Pro ect

Output ariable	alue
Total Tonnes to Mill (000 t)	45,016
Maximum Annual Tonnes to Mill (000 t)	2,555
Average Manganese Grades ( )	2.46
Total MM Production (Mlb)	2,192
Average Annual MM Production (Mlb)	104

22.2.2 ASIS I A IA E A ATI S

To evaluate the financial performance of the Project, the production schedule was incorporated into the 100 equity pre-tax financial model to develop annual recovered metal production from the tonnage processed, mill feed grades and estimated mill recoveries.

All costs and revenues were assumed to occur at the end of the year in which they were scheduled.

MM payable values (i.e. at-mine revenues) were calculated based on metal prices multiplied by quantities of MM produced over the LOM. Applicable Royalties were subtracted from the at-mine revenue to determine the net annual revenues. Unit operating costs for mining, processing, G A and surface services were applied to annual mined and processed tonnages to determine the overall operating cost, which was then deducted from net revenues to derive annual operating cash flow.

Initial and sustaining capital costs, as well as working capital, were incorporated on a year-by-year basis over the LOM. Mine reclamation costs were applied to the capital expenditure on a yearly basis, as applicable. Capital expenditures were then deducted from the operating cash flow to determine the net cash flow before taxes.

Initial capital expenditures include costs accumulated prior to first production of MM. Sustaining capital includes expenditures for mining, processing additions and equipment replacement.

The pre-production period is assumed to be 24 months in duration.

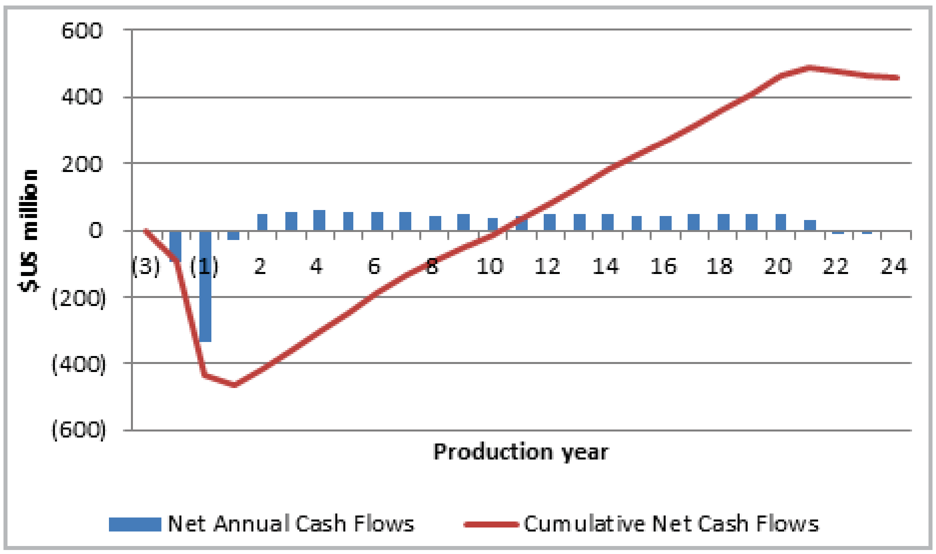
Working capital is set to be three months of the annual operating cost and fluctuates from year to year based on the annual cost. The working capital is recovered at the end of the mine life.

The salvage value of facilities and equipment at the end of mine life is assumed to be 2 of the initial and sustaining capital costs.

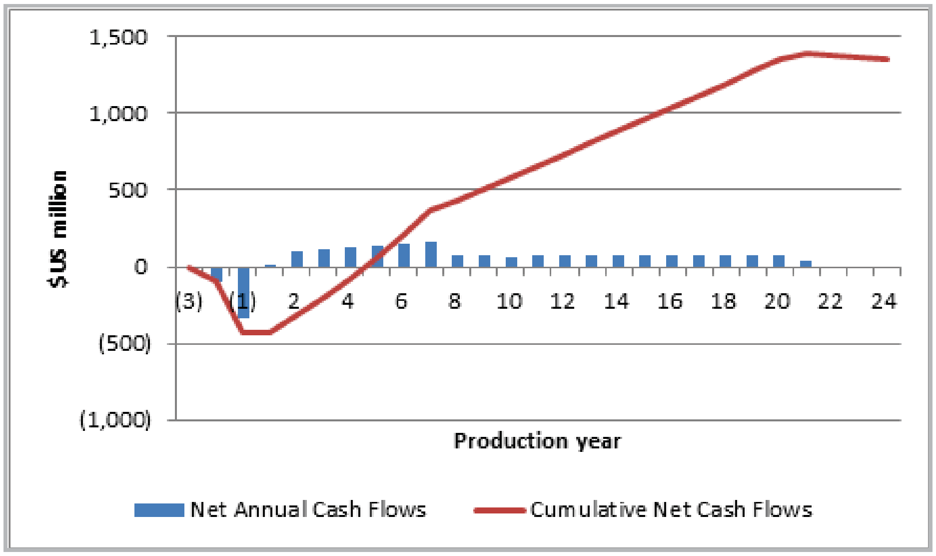
Mine closure and reclamation are estimated at 57.7 million.

The undiscounted annual net cash flows and cumulative net cash flows for the Base Case and Alternate Cases 1 and 4 are illustrated in Figure 22.1, Figure 22.2, and Figure 22.3 respectively.

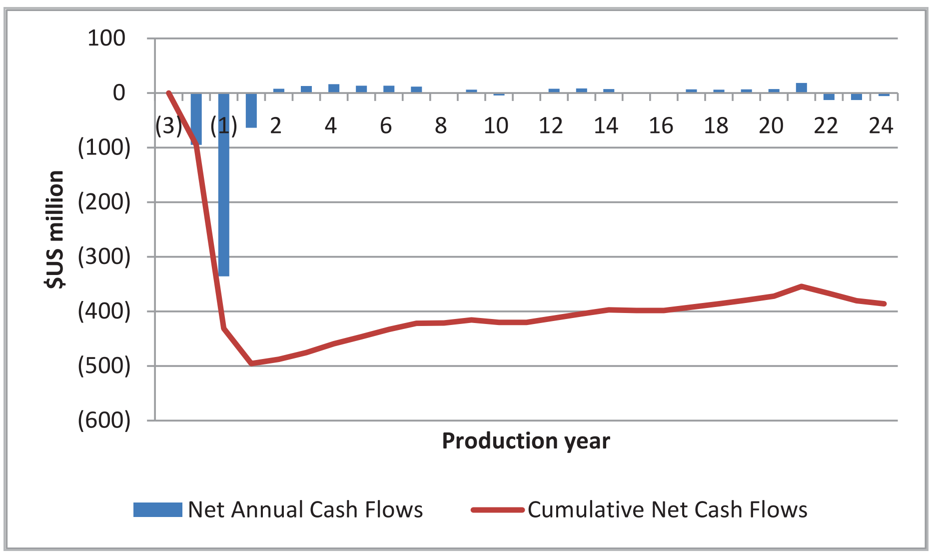
i ure .1 ndiscounted Annual and Cumulative et Cash lo s ase Case



i ure . ndiscounted Annual and Cumulative et Cash lo s Alternate Case 1



i ure . ndiscounted Annual and Cumulative et Cash lo s Alternate Case



22.3 SUMMARY OF FINANCIAL RESULTS

Tetra Tech evaluated the base case using the three-year historical average MM price. In addition to the base case, four other alternate cases were evaluated using the MM average prices, as follows

- Alternate Case 1 CPM expected price forecast
- Alternate Case 2 CPM upside price forecast

- Alternate Case 3 CPM downside price forecast
- Alternate Case 4 Three-year trailing average reduced by 25 .

Detailed MM market outlook and price forecast determined by CPM Group are presented in Section 19.

The pre-tax financial model was established on a 100 equity basis, excluding debt financing and loan interest charges. The base case and the four alternate cases assume that 60 of produced manganese will be sold domestically, and 40 will be sold overseas. The pre-tax financial results for the base case and alternate cases are provided in Table 22.3.

Table . ummary of Pre-ta inancial esults

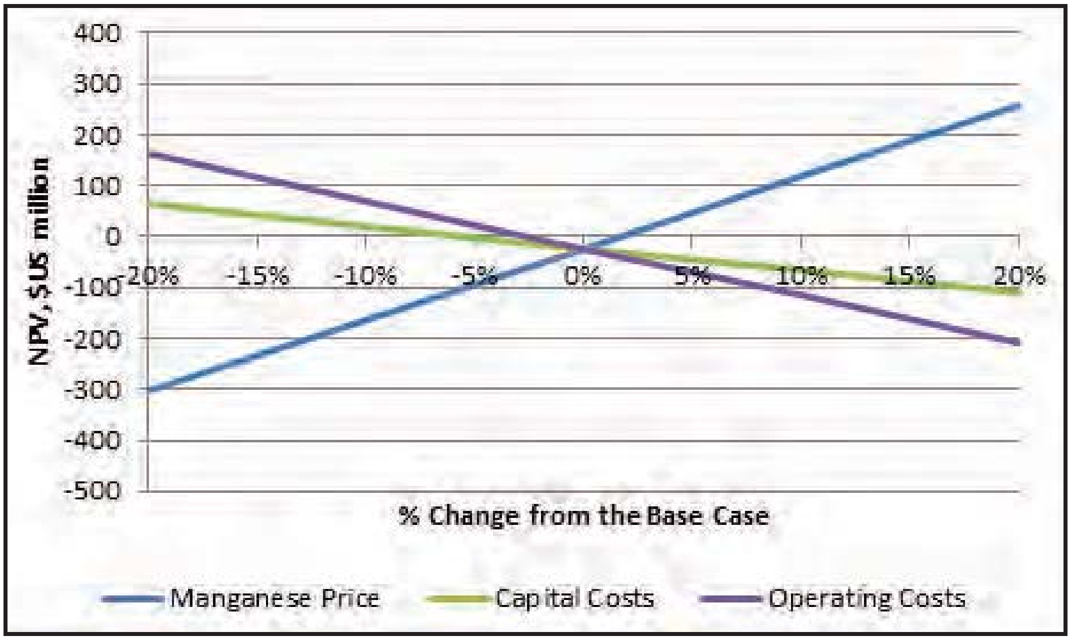
tem	ase Case -Year istorical Avera e Price	Alternate Case 1 CP pected Price orecast Avera e Price	Alternate Case CP pside Price orecast Avera e Price	Alternate Case CP Do nside Price orecast Avera e Price	Alternate Case - -Year istorical Avera e Price educed by 5
Average MM Price (lb)	1.54	1.93	2.18	1.68	1.16
Payable MM Value(At-Mine Revenue) (000)	3,386,317	4,291,178	4,827,780	3,714,977	2,539,738
Operating Costs (000)	2,223,823	2,223,823	2,223,823	2,223,823	2,223,823
Royalties (000)	117,900	121,187	122,490	119,525	115,831
Operating Cash Flow (000)	1,044,594	1,946,168	2,481,467	1,371,629	200,084
Initial Capital xpenditure (000)	476,973	476,973	476,973	476,973	476,973
Total Capital xpenditure (000)	586,191	586,191	586,191	586,191	586,191
et Cash Flow (000)	458,403	1,359,977	1,895,276	785,438	-386,108
PV 5.0 (000)	91,177	641,467	940,039	311,034	-380,022
PV 8.0 (000)	-22,901	402,945	623,234	153,606	-371,306
PV 10.0 (000)	-74,420	289,442	472,346	79,031	-364,337
IRR ( )	7.28	19.95	25.03	12.90	A
Payback (years)	10.3	4.6	3.9	5.9	A

22.4 SENSITI ITY ANALYSIS

Tetra Tech investigated the sensitivity of the Project's PV, IRR and payback period to the Project's key variables. Using the base case as a reference, each of the key variables (price of manganese, capital costs, and operating costs) were changed between -20 to 20 , at 5 intervals, while holding the other variables constant.

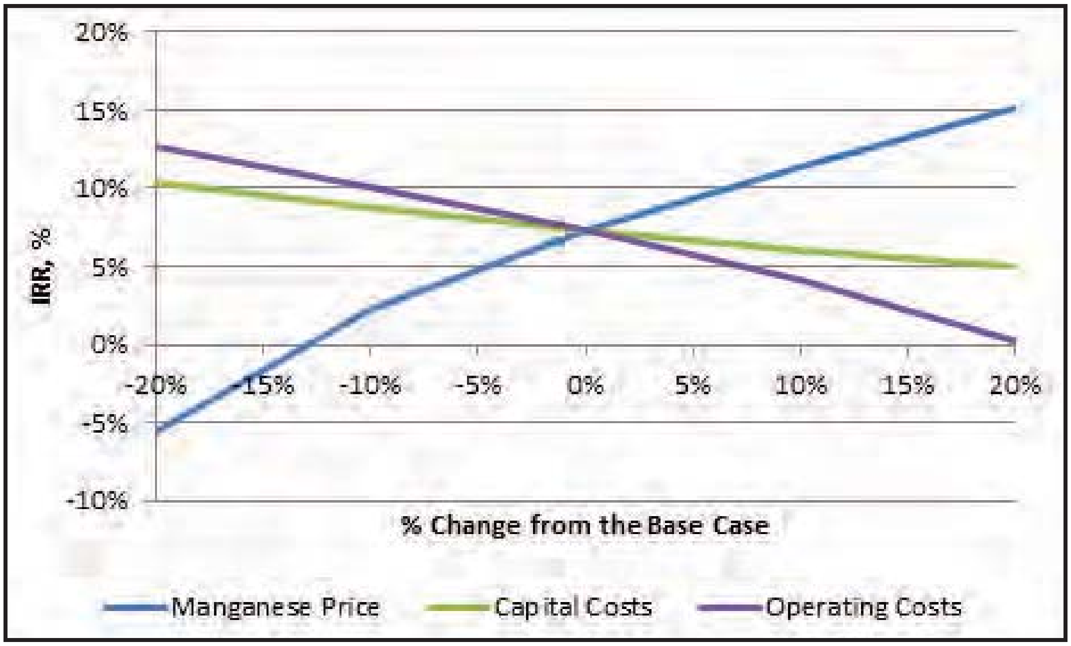
As shown in Figure 22.4, the Project PV, calculated at an 8 discount, is most sensitive to the price of manganese, followed in sensitivity by operating costs and capital costs.

i ure . P ensitivity Analysis



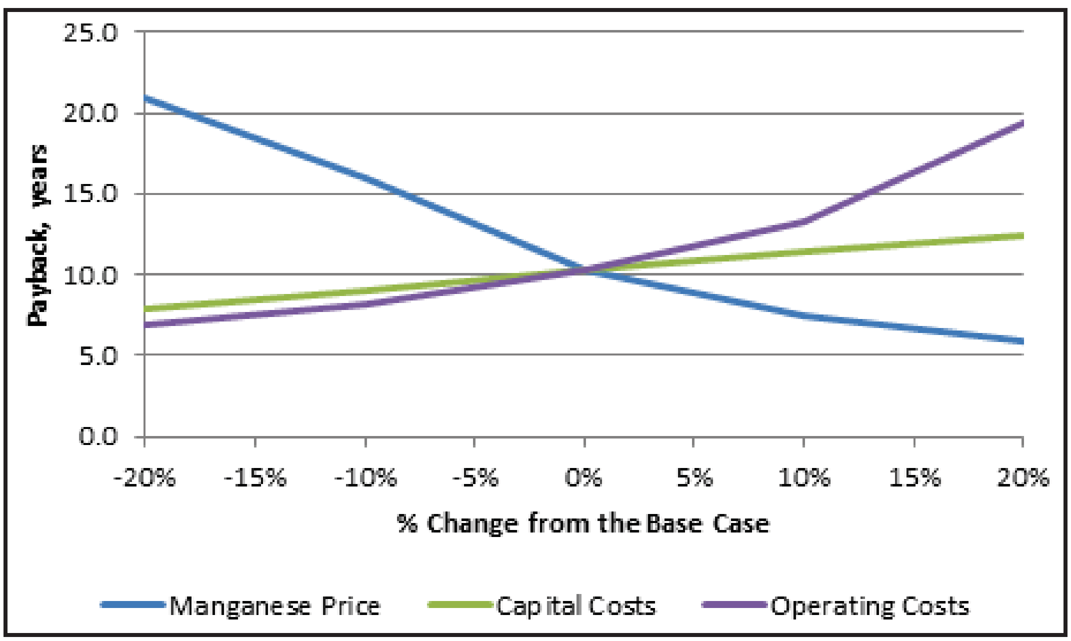
As shown in Figure 22.5, the Project IRR is most sensitive to the price of manganese, followed by operating costs and capital costs.

i ure .5 ensitivity Analysis



As shown in Figure 22.6, the payback period is also most sensitive to the price of manganese, followed by operating costs and capital costs.

i ure .6 Paybac Period ensitivity Analysis



22.5 ROYALTIES

Royalties of 1 on annual at-mine revenues, in addition to 0.05 lb metal, were applied to public land metal production. For private land production, royalties of 0.05 lb metal sold were applied.

22. TRANSPORTATION AND MAR ETING LOGISTICS

The electrolytic manganese metal flake will be picked up at the processing plant. Therefore, no additional marketing or transportation costs are included in the economic evaluation.

22. TA ES

AMI commissioned PwC to perform a post-tax analysis of the Project.

The economic evaluation of the Project, in this PFS, was based on a pre-tax financial model. The effect of taxes on future financial performance of the potential operations should be considered when evaluating merits of the Project. A post-tax financial analysis is recommended to be performed in the feasibility study.

The following general tax regime was recognized as applicable at the time of report writing.

22. .1 S E ERA A STATE TA ATI RE IME

The tax regime, as it may influence the operation of the Project, is shown in Table 22.4.

Ta able ncome	Ta
Arizona State Taxes	6.968
Federal Tax Rate	
0 - 50,000	x 15 - 0
50,001 - 75,000	x 25 - 5,000
75,001 - 100,000	x 34 - 11,750
100,001 - 335,000	x 39 - 16,750
335,001 - 10,000,000	x 34 - 0
10,000,001 - 15,000,000	x 35 - 100,000
15,000,001 - 18,333,333	x 38 - 550,000
18,333,334 over	x 35 - 0

22. .2 EP ETI

Depletion, like depreciation, is a form of cost recovery. Just as the Owner of a business asset is allowed to recover the cost of an asset over its useful life, a miner is allowed to recover the cost of mineral property. Depletion is taken over the period that mineral is being extracted. Two forms of depletion are allowed cost depletion and percentage depletion. The taxpayer is required to use the method which will result in the greatest deduction.

22. .3 ST EP ETI

The general method used for the calculation of depletion is the cost method. The first step of this method is to determine the number of units, which comprise the deposit. The units can be tonnes of ore, barrels of oil, board feet of timber, and other. The taxpayer must be consistent from year to year in the type of unit being calculated to insure uniformity. The second step takes the cost or adjusted basis of the property, which pertains to the deposit and divides this basis by the total number of units to obtain the depletion cost per unit. Once the total number of units extracted is determined for the tax year, it is multiplied by the cost per unit to obtain the amount of depletion available.

22. . PER E TA E EP ETI

Under the percentage depletion method, a flat percentage of gross income from the activity is used to calculate the depletion allowance. The deduction for depletion cannot exceed 50 of the taxable income from the activity. This limitation is computed without regard to the depletion allowance. Depletion percentages are found in IRC section 613(b) and Treas. Reg. section 1.613-2. The amount of the deduction allowable under percentage depletion is not limited by the basis of the property. Thus, even though the basis of the property is reduced by the amount of depletion taken, if the basis becomes zero, the depletion based on the percentage of gross income may continue. However, if cost depletion will yield a higher deduction, it must be used to calculate the amount deducted.

22. .5 ARI A SE ERA E TA

Arizona imposes a severance tax in lieu of sales tax on mining of metalliferous and non-metalliferous minerals. The amount of tax payable is 2.5 of one half of the difference between the gross value of production and the production cost.

22. .6 P ST-TA E MI E A ATI

A post-tax financial analysis of the Base Case was prepared to evaluate the effect of taxes on future financial performance of the potential operations.

Only pre-tax financial analysis was completed on all alternate case scenarios and when reviewing the results, a potential effect of taxes should be considered. A post-tax financial analysis, on all scenarios, is recommended to be performed in the next study of the Project.

For the 21-year LOM at an average annual production of 50,000 t of MM, the following post-tax financial results were calculated for the Base Case

- 6.14   internal rate of return (IRR)
- 10.7 year payback on   477 million initial capital
- (54.407) million net present value (PV) at an 8   discount rate.

The financial results described for Alternate Case scenarios 1 through 4 did not factor in taxes, and the impact of taxes would reduce the financial results.

23.0 AD ACENT PROPERTIES

There are no material properties adjacent to the Property.

24.0 OTHER RELE ANT DATA AND INFORMATION

There is no other relevant data and information available for the Project at this time.

25.0 INTERPRETATIONS AND CONCLUSIONS

Based on the results of the work presented in this Report, Tetra Tech recommends that AMI proceed with the next phase of work to identify potential cost saving and additional revenue generating opportunities and more completely assess the viability of the Project.

The overall processing concept has been proven by metallurgical bench and pilot plant tests. The diagnostic projections of this PFS can be used to focus further efforts in optimizing, designing and demonstrating an initial refined embodiment of the actual operations. The main risks and challenges to the project are shared by many established producers worldwide and pertain to rising energy, environmental protection, labour and reagent costs.

25.1 GEOLOGY AND MINERAL RESOURCES

Mineralization on the Property is stratiform and hosted within sandstones and siltstones of the Chapin Wash Formation. The mineralization typically occurs within a few metres to tens of metres below the Cobwebb Basalt, which conformably overlies the Chapin Wash Formation and has been dated at 13.3 Ma. This basalt flow acted as cap rock, reducing the impact of erosion, and may have acted as a barrier to secondary fluid movement. Faults and possible volcanic vents occur on the Property, and may have been the source of the manganese.

Confirmation of the stratiform nature of the mineralization and the continuity of the geological units and mineralized zones has resulted in an overall high confidence in the distribution of mineralization. However, numerous inconsistencies were noted in the geology database, particularly with respect to drillhole collar locations. In addition, QA QC samples were not collected and there were inconsistent sampling preparation procedures. Recommendations are provided with respect to these concerns.

25.1.1 2011 RES R E ESTIMATE

The 2011 estimate (Tribe, 2011) remains in effect for the surrounding and outlying mineralized areas contained within AMI's total land holdings including, but limited to, Maggie Mine, Shannon Mine, Love's, Hurley and Planche Mines, and South Chapin, Burro, Price and Priceless ones. The remaining resources estimated in these areas include an Indicated Resource of 143,575,196 t at an average grade of 2.98 and an Inferred Resource of 54,700,239 t at an average manganese grade of 2.83 .

25.1.2 2012 RES R E ESTIMATE

At a base-case cut-off of 1 Mn, this estimate includes an Indicated Resource of 62,201,000 t at an average manganese grade of 2.3 , and an Inferred Resource of 20,033,000 t at an average manganese grade of 2.5 . This resource estimate supersedes the estimate for the orth Chapin Lakes MacGregor region from previous resource estimates, which is the area used to develop the commercial mining scenario described in this report.

25.2 MINING

25.2.1 I ERRE RES R ES, S APE EP SIT A RT ER E P RATI

Within the ultimate pit, there are 2.5 Mt of Inferred resources at 2.46 Mn. Inferred resources are not included as mineral resource in the pit value estimation. Figure 25.1 shows the final pit and mineral resource block, coloured for different classes of mineral resource, with Inferred resources represented by red blocks. Inferred materials are more scattered on the outside edges of the pit than in the centre of the pit; however, the extension of the pit rim to the east is controlled by Inferred resources. These areas can be drilled and, if confirmed, upgraded to Indicated or Measured resources to be included in the mineral resource inventory. As shown in this figure, there is an area of waste material in the east-central part of the pit that is not mined. Most of this island consists of Inferred materials that, by exploration and upon improvement in classification, may ultimately be mined.

The footwall of the deposit shapes the east side of the mine. The current design is based on information obtained using exploration drilling. With more information becoming available in operation or with additional exploration, it may be possible to improve the design of the east wall.

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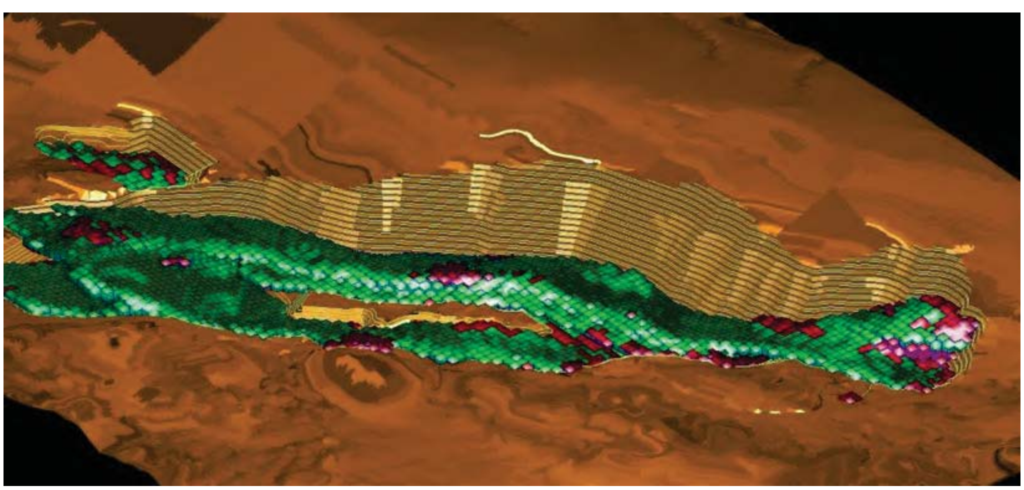

25.3 PERMITTING AND EN IRONMENTAL ASSESSMENTS

25.3.1 ASE I E ST IES, RE ATI S, A PERMITTI

The PA analysis program for the Project would take the form of an A with the FO SI or an IS if significant impacts are expected. From a conservative standpoint and based on preliminary conversations with the BLM, an IS will likely be the selected PA action. The IS process from otice of Intent to the Record of Decision is estimated to require up to three years in order to complete the necessary baseline studies, documentation and submittals. The duration of the process depends on public review and input, and final acceptance by the BLM. One of the crucial factors that could affect the outcome of the IS would be the impact of the mine operations water demand on local water resources for plants and wildlife.

One of the key permits that will be issued by the AD Q is the APP to prevent groundwater contamination. The APP will incorporate Best Available Demonstrated Control Technology (BADCT) guidance to demonstrate that the mine facility will not impact groundwater and violate aquifer water quality standards. Future feasibility work will need to demonstrate that the waste rock and tailings generated from mining activities are non-acid generating or metal leaching and do not pose a threat of contaminating the underlying groundwater. Although tailings testing to date show elevated arsenic and thallium concentrations, further research will be conducted in mineral resource process modifications to potentially remove or stabilize these metals. Furthermore, other important goals of the BADCT process will be to demonstrate that the bottom of the open pit will serve as an impermeable boundary to prevent infiltration of contaminants to groundwater and verify that the depth to groundwater is below the total depth of the open pit, ensuring that a pit lake will not develop. The APP process from application submittal to issuance of the permit by AD Q is estimated to require up to two years; however, baseline characterization has already been initiated to minimize this timeline.

During the feasibility study, a review of the mine processes considering available air pollution controls and reasonable operating limits should occur in order to create a strategy to lessen the air permit liability. The Project may require a Class I air quality permit, but if the appropriate controls and operating limits are implemented, the Project may qualify for a synthetic minor Class II air quality permit. In either case, air dispersion modelling will be necessary to demonstrate the effectiveness of controls and emissions compliant to AAQS. The air permitting process for obtaining a Class I Permit is anticipated to require 18 months to 2 years, whereas the timeframe for a Class II is between 9 and 12 months. The timing, similar to PA and APP, is dependent on the complexity and nature of public comments.

25.3.2 ATER RES R ES

A total of nine source wells will be drilled during Fall 2012 in the areas of expected higher permeability. Aquifer testing performed on each of the nine source wells will evaluate the suitability for the siting of a larger scale water-supply well field in selected areas; however, there is a risk that well yields may be insufficient to supply the required water needs. Further study will be required to determine the impacts of groundwater pumping on other nearby water resources, including springs seeps and Alamo Lake.

25.3.3 ATER A IT

Because little historical baseline water quality information is available for the Project site area, additional groundwater and storm water characterization will be implemented. A network of monitoring wells is planned for installation in and around the Project site in the fall of 2012. In accordance with APP requirements, groundwater sampling will be performed in selected wells to establish baseline groundwater quality prior to mining operations and to monitor for potential impacts to groundwater once mining operations commence. Four storm water monitoring stations have been established in washes downstream of the active project areas. In accordance with MSGP requirements, the stations will be used to monitor storm water discharges to establish baseline water quality of storm water prior to operations and to measure whether any mining operations are impacting storm water quality.

25.3. ST RM ATER MA A EME T

A otice of Intent and SWPPP containing the BMP and SAP were submitted to the AD Q as part of the application process. A Multi-Sector General Permit was issued. The SWPPP will require an amendment when the construction and operation phases of the Project begin.

25.3.5 A I R RAI A E

A total of eleven waste rock samples were collected and analyzed for a variety of geotechnical and analytical parameters. These initial samples indicated that the content of the waste rock is not acid generating and limited in elevated metal content, which is to be expected for an oxide deposit. Additional characterization of a broader base of waste rock material will be necessary to satisfy regulatory requirements. Samples can be from previous cuttings and or new samples collected in the Fall of 2012 during the new resource-exploration drilling program. Sampling methods will ensure that the material is fresh rock and has no signs of previous weathering.

25.3.6 ASTE MA A EME T

TAILIN S MANA EMENT

The mineral resource-processing pilot program generated three separate batches of tailings material, as the process was refined. Overall results from the samples indicated that the acid generating potential of the tailings was low and treatable. The tailings material contains a very limited amount of sulphide, though natural neutralization potential is also limited. The presence of arsenic and thallium was continuous through the three batches, and will require either additional research to establish the modification of the process, stabilization measures used to immobilize the metals in the tailings, or demonstration that infiltration and seepage is minimal due to compaction of the material during placement.

ASTE ROCK AN OVER R EN MANA EMENT

Cover for the waste rock and tailings will include any soil or organic matter that was present in the area during the start of mining and had been stockpiled. During the mining process, the waste rock and tailing dumps will be contoured on a progressive basis to blend into the natural topography of the area. Due to the lack of organic material in the general area, additional research will be conducted to identify other sources of growth media that can be imported to the site for reclamation. Imported growth media and any stockpiled topsoil will be placed over the contoured terrain and revegetated as per the reclamation plan.

HA AR O S ASTE MANA EMENT

AMI will manage the generation of hazardous wastes from the project and will have waste minimization practices in place. During mineral resource processing, a ferric sludge is generated that is acid-generating and contains elevated concentrations of arsenic, chromium and thallium that exceed the AWQS. This sludge, which also contains elevated manganese, is undergoing additional processing reviews to determine if there are treatment or stabilization options available that would remove additional metals and allow environmentally safe on-site disposal.

NON HA AR O S ASTE MANA EMENT

Utilizing a comprehensive waste management program, AMI will ensure that waste minimization practices are in place during construction and throughout mine operations.

25.3. ESI I A E

American Manganese's development and use of a comprehensive MS for the construction, operation and closure phases of the Project will ensure that all environmental concerns are identified, organized and managed for compliance with any regulatory or other requirements.

25.3. MI E S RE

A reclamation and mine closure plan will be required as part of the APP process, and will be developed during the feasibility and mine design phase. The reclamation and closure plan will identify how the mine area will be graded, covered and seeded, and how any environmental concerns will be managed for the long term once mining has ceased. A reclamation plan and closure approach will be completed concurrently with the completion of final mining operations. The plan for closure and reclamation will be to place all waste rock and tailings, mixed, in the completed pit in order to manage the wastes in one area and achieve a demonstrated impermeable layer and lack of groundwater connection.

25.4 METALLURGY AND PROCESS DESIGN

Kemetco's testwork has confirmed a processing scheme that can be modeled to a scalable processing plant that utilizes commercial equipment. This plant model was used as the basis for engineering design.

The main conclusions derived from this test work include

25. .1 RES R E MATERIA

- The resource material is soft, friable and breaks down easily. This was evidenced by unconfined compressive strength tests conducted by Speedie and Associates; Bond Rod Mill Index and Abrasion Index tests conducted at Hazen Research; crusher tests performed by Pennsylvania Crusher; observations of dry rod milling at very short retention times, by jaw crushing and by hammer mill tests at the University of British Columbia; and frequent observations before and after leaching.

25. .2 S P R R ER

- A maximum capacity of burning 135 t d of liquid sulphur will provide a heating value of 18 MW. Converting this heat to thermal energy in the form of high-grade steam is estimated to only generate 4 MW of electricity in a typical condensing turbine.

25. .3 EA IR IT

- Fine material (200 mesh) was tested in continuous mode in the pilot plant, and the manganese extractions achieved were consistently on the order of 90 to 93 . The leach times tested in continuous mode were adapted to the solid-liquid separation time requirements, which varied in the range of 2 to 4 hours.

- Semi-batch pilot leach tests performed on 6.35 mm material for 2 hours resulted in manganese extractions of greater than 90 .

25. . TAI I S AS I A E ATERI

- Solid liquid separation tests were conducted by Pocock Industrial using leached slurry samples collected from the pilot trial. The data generated by Pocock were used to model commercial plant requirements. Based on the testing by Pocock, an underflow density of 48 w w solids and wash ratio of 2 1 fresh water to dry solids was used to model the commercial plant. Flocculant dosage totalled 465 g t based on the plant model.
- Underflow densities achieved during continuous pilot operation reached 55 w w solids while still observed to be pumpable. Flocculant dosage during pilot plant operation was approximately 130 to 170 g t.
- The filtration scheme used in the process model is functional but is slow and expensive. Reducing or eliminating the filtration requirement beyond the CCD stages with the use of coarser feed material, paste thickeners and thermal drying of tailings to remove additional amounts of moisture may be possible with further test work.

25. .5 E TRA I ATI A P RI I ATI

- Purification of the pregnant leach solution (PLS) was successful and consisted of filtration through a polishing filter, hydroxide precipitation with lime in combination with aeration, and sulphide precipitation.
- Hydroxide precipitation with aeration effectively removed Fe, Al, As, Si, and Cu impurities from the PLS sample to below detection limit.
- Second-stage purification was conducted by sulphide precipitation of the treated PLS. Impurities such as n, Cd, Co, i, Tl, As, Mo, and Sb are removed by adding aHS followed by filtration.

25. .6 AR ATE PRE IPITATI

- Manganese carbonate precipitation for recovery of Mn from the PLS was successful. Manganese carbonate recovery was essentially stoichiometric with the addition of sodium carbonate.
- MnCO3 was produced that settled and filtered very well.

25. . ATER RE ER A A R S A_2S -PR T

- Single pass nanofiltration tests were successful in concentrating sodium sulphate and sodium dithionate mixtures from approximately 0.41 Molar to 0.93 Molar with high rejection of the sulphate and dithionate anions. Review of these data indicate that permeate from a complete nanofiltration system could be recycled as process water.

- Process modeling of the sodium sulphate crystallization circuit was conducted by Swenson Technology. The process model was based on crystallizing sodium sulphate from a feed solution concentration that would be produced by nanofiltration. The model recommends initial stage evaporation followed by crystallization with mechanical vapour recompression (MVR).

25. . EMM E E TR I I IR IT

- lectrolytic manganese metal (MM) was successfully produced from electrolyte produced from manganese carbonate generated in the pilot trial.
- The current efficiency for electrowinning was 67.5 without the use of selenium additives. This represents typical commercial performance.
- The purity of the MM was measured to be 99.7 .

25. .9 TAI I S ISP SA A E IR ME TA TESTI

- Detailed work on the tailings and rinse waters will be required for future environmental permitting.

25.5 ASTE MANAGEMENT

The co-disposal of filtered tailings and waste rock is a technology that has been successfully applied in other mines, and has been adopted for use for the Project. The laboratory analyses conducted in this prefeasibility study were performed on tailings samples generated from a bench-scale pilot test, and future tests are recommended as the Project advances to the next phase. It has been assumed that the filtered tailings will mix very well with the waste rock (at a ratio of about 2 1 waste rock to tailings, by weight) to form a workable mixture that can be spread and compacted to create a stable storage facility. The TWSF will probably require an APP, and should meet the requirements of BADCT in its design, construction and operation. Prescriptive BADCT criteria have not been established for filtered tailings storage facilities, and therefore an Individual BADCT design should be developed for the TWSF at the Artillery Peak site before advancing the project to the Feasibility Study. As per BADCT guidelines, important aspects of developing an Individual BADCT design for a tailings storage facility are

- discharge control technologies ordinarily constitute a discharge control system incorporating engineering features, operational measures and site characteristics to achieve BADCT
- alternative designs must be considered to arrive at a BADCT design.

In developing an Individual BADCT, the following steps are required:

- site selection
- development of individual site design based on demonstrated control technologies and site conditions
- estimation of aquifer loading for the design
- alternative design(s) selection
- estimation of aquifer loading for the alternative design(s)
- selection of BADCT design(s).

25.6 FINANCIAL EVALUATIONS BASE CASE AND ALTERNATE 4 CASE

Financial evaluations of the Base Case and Alternate Case 4 EMM pricing regimes show that the Project has a negative NPV at an 8% discount rate. For these cases, the Project is deemed to be uneconomic and contains no mineral reserves that meet the CIM definition.

2 .0 RECOMMENDATIONS

Based on conclusions and recommendations, the next phase of work for this Project is expected to include the following items with the estimated budgets

- additional infill and QA QC drilling   US 4.0 million
- bulk density studies   US 0.1 million
- geotechnical studies and hydrogeologic investigations   US 1.9 million
- metallurgical testing and pilot processing   US 5.0 million
- environmental investigations and studies   US 3.0 million
- subsequent study   US 5.0 million.

On a preliminary basis, the drilling, pilot processing and additional studies are estimated to cost approximately US 14 million, and production of the subsequent report is projected to cost approximately US 5 million, for a total of US 19 million.

2 .1 GEOLOGY AND MINERAL RESOURCES

26.1.1 ATA ASE A ATA ERI I ATI

Tetra Tech recommends that a single database be created using industry standard database management software (Microsoft Access or similar). All drillhole collar locations, geology logs, and assay and survey results should be entered into this database. Database verification should be conducted regularly to ensure accurate data entry. Before proceeding to another phase of evaluation on the Property, it is recommended that the existing data be consolidated into a database and verified for accuracy.

A component of drillhole collar verification is to contract an external surveyor to survey the locations of all drillhole collars. It is recommended that previously completed collars be surveyed, as well as all future drillholes.

26.1.2 E SIT

Tetra Tech recommends that bulk density be determined for representative samples within the area of the current resource estimate. These samples should include manganiferous material (at least 50 samples) and country rock material (at least 50 samples each of sedimentary and volcanic country rock). These samples should be extracted from outcrop and underground workings, and tested for bulk density using the method of weighing in air and weighing in water. The location and a brief geological description of each sample should be recorded.

Additional bulk density samples should be collected from material outside of the current resource area, in any areas that are targeted for inclusion in future estimates.

26.1.3 RES R E ASSI I ATI

There is a significant amount of Inferred resources reported for the Project in the current resource estimate. Areas identified with potential for resource re-classification can be targeted for infill drilling and, if mineralization grade and extent is confirmed, Inferred Resources can be upgraded to Indicated or Measured Resources. A drillhole spacing of less than 120 m is considered the minimum requirement for classification as an Indicated Resource. However, this depends on geological and grade continuity and it is possible that even further infill drilling (at tighter drillhole spacing) may be required locally to upgrade the classification. Therefore, it is recommended that processing of assay samples is ongoing during any future drilling programs, such that the results can be analyzed and further drillholes added to the program, if required.

The amount of infill drilling is at the discretion of the client and depends on their future plans for the Property. Further infill drilling is currently not required to proceed with the project to a feasibility study, although only material classified as an Indicated or Measured Resource can be included in mine planning.

26.1. A

In order to evaluate the reliability and accuracy of results from the three previous drilling programs, twinning of holes from each of the programs is recommended. Tetra Tech recommends that four previously completed holes be twinned, including at least one hole from each of the 2008, 2010, and 2011 programs. These holes should transect a reasonably continuous mineralized interval and (if applicable) to extend at least several metres into unmineralized material both above and below mineralization. All twinned holes should be completed using the same drilling technique (for example, RPRC drilling using water).

QA QC samples should be included during sampling of these four twinned holes, as well as during all future sampling programs (including core and RPRC drilling and channel sampling). QA QC involves insertion of standards, blanks, and field duplicates into the sample stream, in addition to the laboratory preparation and pulp duplicates and blanks that were analyzed in previous sampling programs. One QA QC sample should be inserted in the sample stream roughly every 20 samples.

26.1.5 A E -RA RES E E R

Tetra Tech recommends AMI consider obtaining a handheld RF instrument (available from vendors such as iton or Innov-). This instrument would be very useful as it could be used to produce preliminary corebox and coreshed analyses of manganese grade, and therefore aid decisions on whether or not to submit certain intervals of core for analysis. Provided consistent count times and distance from the core are maintained, manganese and iron content can be measured with good precision and accuracy at percentage level abundance and as such, this instrument can serve as a QA QC tool.

2 .2 METALLURGICAL TESTING AND INTEGRATI E PILOTING

Following are two major tasks that need to be completed according to strictest specifications and with a reliable degree of project management and coordination, with associated minimum budget expectations

- Bench-scale process definition work, including optimization of process flowsheet and conditions, systematic fine-tuning, variability and sensitivity studies, determining the data required for process design, as well as locked-cycle integration and pilot plant design.
- Demonstration piloting including gaseous SO_2-handling, more than five stages of CCD and recycling, multi-stage purification, multi-stage $MnCO_3$ precipitation, nanofiltration and crystallization, high level PLC and Data Logging, on site vendor testing.

2 .3 PERMITTING AND EN IRONMENTAL ASSESSMENTS

The following points summarize the recommendations and items that should be addressed to take this study to the next phase

26.3.1 ASE I E ST IES, RE ATI S A PERMITTI

- The otice of Intent needs to be filed for the BLM to start the PA process; however, the Mine Plan of Operations must be completed and submitted at the same time.
- More detailed baseline studies will need to be completed for biological, air, traffic, jurisdictional delineated wetlands and waters of the US, to address PA and BLM concerns.
- Additional samples representing the waste rock over the entire proposed mine area is needed to support the APP application process and to continue the demonstration of limited environmental risk from the waste rock.
- For APP, additional characterization of tailings samples from the process will need to be analyzed during feasibility to establish the final treatment methods to demonstrate the ability to immobilize any metals present.
- Drilling for groundwater and installing monitoring wells for baseline groundwater characterization including transmissivity and flow

characteristics for model parameters and constituent characterization will be needed for APP.

- A Reclamation Plan will need to be completed for the APP process.
- In order to make the air permitting efficient and the requirements streamlined, air permit experts will need to consult with the design engineers during the feasibility to define the best available air pollution controls and operating limits.
- Air dispersion modelling will be necessary for air permitting.

26.3.2 ATER RES R ES

- Completion of well drilling for the nine proposed locations in order to establish water supply source. Once these wells are drilled, delineation of water resources and hydrogeology south of the mine will be further defined.

26.3.3 ATER A IT

- Additional wells and ground water samples need to be collected from the immediate area of the mine for APP but also from the water source wells to characterize the groundwater and existing baseline water quality.

26.3. ST RM ATER MA A EME T

- Storm water sampling from each of the monitoring stations needs to be completed to establish baseline water quality of runoff in the area prior to any mining activities.
- A new storm water monitoring station will need to be identified downstream of the final mine layout to replace the station that is in the area of the mine.

26.3.5 A I R RAI A E

- Additional characterization of a broader base of waste rock material and a broader analyte list will be necessary to satisfy regulatory requirements for the adequacy of representative samples.

26.3.6 ASTE MA A EME T

- Additional research into the tailings process needs to be conducted to establish the method of removing or stabilizing the arsenic and thallium.
- Additional samples will need to be collected from the tailings process for APP demonstrations and characterization.

- Additional sources for organic material will need to be identified as part of reclamation planning to provide adequate cover and material for growth of vegetation.
- Additional research, testing and sampling analyses of the ferric sludge generated during mineralized material processing is needed to determine whether additional metals present can be removed, treated or stabilized to render the material more inert.

26.3. MI E S RE

- A detailed reclamation plan will need to be developed.
- Impermeability of the pit bottom geology and depth to groundwater will be important demonstrations of the feasibility phase for APP and closure planning.

2 .4 METALLURGY AND PROCESSING

The hydrometallurgical test work completed to-date has removed major process uncertainties and the project is ready to be advanced to the next phase of study. The recommended work includes

1. Process de-bottlenecking studies with the focus on opportunities to reduce capital and operating costs including
   - Review potential removal of 4 MW power plant in favour of more efficient uses of 18 MW thermal heat from sulphur burner.
   - Review the potential to reduce the number of CCD stages with confirmatory pilot test in the presence of an independent contractor.
   - Review the potential to reduce flocculant dosing with confirmatory pilot test work in the presence of an independent contractor.
   - Review the potential for removing all filter presses by leaching with larger particle size, replacing high rate thickeners with paste thickeners and use of surplus thermal heat for further moisture reduction in final paste tailings.
   - Review the potential to remove the first stage evaporator from the sodium sulphate crystallization circuit by concentration solutions further with nanofitration.
2. Confirming processing schemes for MD and CMD production.

The hydrometallurgical processing of manganese resources is conducive to the production of high purity electrolytic manganese dioxide (MD) and chemical manganese dioxide (CMD). This has presented an opportunity for additional revenues for supplying materials used in production of rechargeable lithium ion batteries. The following work is recommended to advance this opportunity

- Review all unit operations for MD production for integration into a complete flowsheet. This includes additional purification requirements, preparation requirements of the electrowinning solutions, conditions for electrowinning, reuse of spent electrolyte, harvesting of MD, and grinding of MD.
- Review all unit operations for CMD production for integration into a complete flowsheet. This includes additional purification requirements, preparation requirements of the appropriate solutions, conditions for chemical oxidation, reagent conservation, harvesting of CMD, and conditioning morphology of CMD.

3. Upgrade operation of the pilot plant for longer-term locked cycle testing of the complete process to develop a realistic simulation of the operation of the processing plant as a function of different operating conditions. Techniques and treatment protocols for stabilizing fixating residual metals in the various waste products need to evaluated and confirmed. These data will be required for detailed engineering and environmental permitting.

The recommended work in this section includes

- Upgrade of the pilot plant for longer-term locked cycle testing of the complete process.
- Conduct longer term locked cycle testing of the overall process within a range of practical process variables for each unit operation to generate data for detailed engineering.
- Analyze pilot plant samples from longer term locked cycle testing and defined processing requirements for environmental permitting.

2 .5 MINING

The following are mining related recommendation for the work to be completed before advancing project to the next phase

- An extensive dilution study should be completed to understand and measure the dilution.
- Dust control can become both a safety and a cost issue if not properly addressed. In this study, two water tanks have been included to control dust produced in mining operations. Since water resources are very limited in this region, it is recommended to study other types of dust control methods.
- Low-grade material was considered waste and excluded from processing. It is recommended to assess an inclusion of a stockpile in the next phase of the study to potentially process this material at the end of LOM to realize additional revenue stream for the Project.

2 . INFRASTRUCTURE

26.6.1 P ER S PP A ISTRI TI

Tetra Tech recommends that during the next phase of the project, utility permits and studies should be initiated to determine definitive costs related to the new 69 kV transmission distribution line. The amount of time required for the utilities design and procurement activities and inter-utility coordination may affect the development of the mine and process. egotiations with participating utilities should begin as soon as practical to obtain a power purchase agreement (PPA) so that reliable cost-effective power to site can be used for cost projections over the mine life.

26.6.2 ATER MA A EME T

The following points summarize the recommendations and items that should be addressed to take this report to the next phase

- Geochemical modelling must be carried out to establish potential acidity of surface run-off over the temporary waste dumps and pit, in order to assess the need for collecting and treating this water or discharging it directly to existing natural streams.
- Hydrology and hydraulic analyses, including a detailed water balance, will need to be completed to determine the types, locations and sizes of the water management structures as well as the times and quantities of discharges.
- Geotechnical investigations must be undertaken along the defined facilities in order to determine soil properties for foundations and slope stability, as well as to determine the construction aspects associated with diversion channels and ponds.
- A design criteria shall be developed to be used in the design of the permanent and temporary water management structures, as well as to optimize the hydrologic study conducted by Tetra Tech in 2011.
- Hydrogeological testing to confirm water flows and dewatering requirements for the open pit.
- Although the current topographic information available for the site is sufficient for this level of study, it is recommended that more detailed topographic information would be needed at some other places (South- ast corner).

26.6.3 ASTE MA A EME T

In order to take the waste management to the next phase the following recommendations are provided

- Conduct a meteorological data analyses to characterize the climate conditions.
- Conduct geotechnical field investigation with the following objectives
  - to define general subsurface conditions for use in evaluation of the TWSF stability
  - to identify suspect zones that could affect the performance of the TWSF
  - to quantify engineering characteristics of the materials incorporated into the TWSF
- Characterize the waste rock to determine engineering properties such as
  - grain size distribution
  - unconfined compressive strength
  - specific gravity
  - moisture content
  - unit weight
- Conduct Tailings Geochemical and Physical Characterization
  - Geochemical SPLP testing, MWMP testing, whole rock analysis. ABA testing, et Acid Generating testing, and Kinetic testing
  - Physical Index Properties, one-dimensional time consolidation, triaxial shear, flexible wall permeability, and Soil water characteristic curve (soil-water retention curves)
- Conduct a bench scale test mixing waste rock with tailings (at the proper rock to tailings ratio) to estimate the geotechnical characteristics of the co-mingled mixture to be followed by field trials as necessary.
- Conduct a Geological Hazards valuation at the site including
  - landslides rockfall hazards
  - collapsible soils
  - historic mining activities
  - earthquake Induced Ground Failure
  - seismic hazard assessment using deterministic method to estimate the maximum credible earthquake (MC) and probabilistic method to estimate the maximum probable earthquake (MP) for different return periods
- Conduct seepage and slope stability analyses.

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2 .0 CERTIFICATES OF UALIFIED PERSONS

2 .1 OHN HUANG, P.ENG.

I, Jianhui (John) Huang, P. ng., of Burnaby, British Columbia, Canada, do hereby certify

- I am a Senior Metallurgist with Tetra Tech W I Inc. with a business address at Suite 800 555 West Hastings Street, Vancouver, British Columbia, V6B 1M1.
- This certificate applies to the technical report titled Technical Report and Prefeasibility Study, Artillery Peak Project, dated June 28, 2012 and amended September 12, 2012 (the Technical Report).
- I am a graduate of orth- ast University (B. ng., 1982), Beijing General Research Institute for on-ferrous Metals (M. ng., 1988), and Birmingham University (Ph.D., 2000). I am a member in good standing of the Association of Professional ngineers and Geoscientists of British Columbia (License 30898). My relevant experience with respect to mineral engineering includes more than 28 years of involvement in mineral process for base metal ores, gold, silver and rare metal. I am a Qualified Person for purposes of ational Instrument 43-101 (the Instrument).
- I have completed a personal inspection of the Property on January 28, 2012.
- I am responsible for Sections 1.1, 1.8 to 1.10, 1.12, 1.14 (excluding 1.14.1), 2, 3, 17, 18.1 to 18.5, 20, 21.2, 25 (excluding 25.1, 25.2, 25.4, 25.5 and 25.6), 26 (excluding 26.1, 26.2, 26.4, 26.5 and 26.6), 27 and 28 of the Technical Report.
- I am independent of American Manganese Inc. as defined by Section 1.5 of the Instrument.
- I have had no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the parts of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 12th day of September, 2012 at Vancouver, British Columbia, Canada.

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John (Jianhui) Huang, P. ng.
Senior Metallurgist
Tetra Tech W I Inc.

2 .2 MARGARET HARDER, P.GEO.

I, Margaret Harder, P.Geo., of Vancouver, British Columbia, Canada, do hereby certify

- I am a Geologist with Tetra Tech W I Inc., with a business address at Suite 800 555 West Hastings Street, Vancouver, BC, V6B 1M1.
- This certificate applies to the technical report titled Technical Report and Prefeasibility Study, Artillery Peak Project, dated June 28, 2012 and amended September 12, 2012 (the Technical Report).
- I am a graduate of the University of Saskatchewan (B.Sc. in Geology, 2002) and of the University of British Columbia (M.Sc. in Geology, 2004). I am a member in good standing of the Association of Professional ngineers and Geoscientists of British Columbia, (Registration 32139). I have practiced my profession in geology for a total of seven years with experience in exploration, advanced evaluation and operations. My relevant experience includes diamond and large diameter drilling including logging and data evaluation, 3D modelling, and reporting. I am a Qualified Person for purposes of ational Instrument 43-101 (the Instrument).
- I have completed a personal inspection of the Property on June 2, 2011.
- I am responsible for Sections 1.2 to 1.4, 4 to 11, 12.1 to 12.2, 23, 27 and 28 of the Technical Report.
- I am independent of American Manganese Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the parts of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 12th day of September, 2012 at Vancouver, British Columbia, Canada.

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Margaret Harder, P.Geo.
Geologist
Tetra Tech W I Inc.

2 .3 MICHAEL F. O BRIEN, M.SC., PR.SCI.NAT., FGSSA, FAUSIMM, FSAIMM

I, Michael F. O'Brien, M.Sc., Pr.Sci. at., FGSSA, FAusIMM, FSAIMM of Vancouver, British Columbia, do hereby certify

- I am a Chief Geologist with Tetra Tech W I Inc. with a business address at 800-555 West Hastings Street, Vancouver, British Columbia, V6B 1M1.
- This certificate applies to the technical report titled Technical Report and Prefeasibility Study, Artillery Peak Project, dated June 28, 2012 and amended September 12, 2012 (the Technical Report).
- I am a graduate of the University of atal, (BSc (HO S) Geology, 1978). I am a registered Professional atural Scientist (Geological Scientist) in good standing of the South African Council for atural Scientific Professions (South Africa, 400295 87). My relevant experience is 33 years of experience in operations, mineral project assessment and I have the experience relevant to Mineral Resource estimation of metal deposits. I am a Qualified Person for purposes of ational Instrument 43-101 (the Instrument) under the Accepted Foreign Associations and Membership Designations (Appendix A).
- I did not complete a personal inspection of the Property.
- I am responsible for Sections 1.5.2, 12.3, 12.4, 14.2, 25.1 (except 25.1.1), 26.1, 27 and 28 of the Technical Report.
- I am independent of American Manganese Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 12th day of September, 2012 at Vancouver, British Columbia, Canada.

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Michael F. O'Brien, MSc, Pr.Sci. at.,
FGSSA, FAusIMM, FSAIMM
Chief Geologist
Tetra Tech W I Inc.

2 .4 NORMAN CHO , P.ENG.

I, orman Chow, P. ng., of Vancouver, British Columbia, do hereby certify

- I am a President with Kemetco Research Inc with a business address 445 5600 Parkwood Way, Richmond, BC.
- This certificate applies to the technical report titled Technical Report and Prefeasibility Study, Artillery Peak Project, dated June 28, 2012 and amended September 12, 2012 (the Technical Report).
- I am a graduate of the University of British Columbia (M.A.Sc. Metals and Materials ngineering, 1997). I am a member in good standing of the Association of Professional ngineers and Geoscientists of the Province of British Columbia (29340). My relevant experience with respect to Metals and Materials ngineering includes 17 years of experience in metallurgical and environmental projects in Canada. I am a Qualified Person for purposes of ational Instrument 43-101 (the Instrument).
- I have completed a personal inspection of the Property on February 25, 2011.
- I am responsible for Section 1.7, 13, 25.4, 25.5, 26.2, 26.4, 27 and 28 of the Technical Report.
- I am independent of American Manganese Inc. as defined by Section 1.5 of the Instrument.
- I have had no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the parts of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 12th day of September, 2012 at Richmond, British Columbia, Canada.

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orman Chow, P. ng.
President
Kemetco Research Inc.

2 .5 ANOUSH EBRAHIMI, P.ENG.

I, Anoush brahimi, P. ng., of Vancouver, British Columbia, Canada, do hereby certify

- I am a Principal Mining ngineer with Tetra Tech W I Inc. with a business address at Suite 800 555 West Hastings Street, Vancouver, British Columbia, V6B 1M1.
- This certificate applies to the technical report titled Technical Report and Prefeasibility Study, Artillery Peak Project, dated June 28, 2012 and amended September 12, 2012 (the Technical Report).
- I am a graduate of Kerman University, Iran (B.A.Sc., 1990); Poly Technique Tehran University, Iran (M.A.Sc., 1993); University of British Columbia, Canada (Ph.D., 2004). I am a member in good standing of the Association of Professional ngineers and Geoscientists of the Province of British Columbia (30166). My relevant experience with respect to Mining ngineering includes 23 years in mine design projects in Canada and abroad. I am a Qualified Person for purposes of ational Instrument 43-101 (the Instrument).
- I have completed a personal inspection of the Property on January 28, 2012.
- I am responsible for Sections 1.6, 15, 16, 18.6, 21.2.1, 25.2, 26.5, 27 and 28 of the Technical Report.
- I am independent of American Manganese Inc. as defined by Section 1.5 of the Instrument.
- I have had no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the parts of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 12th day of September, 2012 at Vancouver, British Columbia, Canada.

riginal document signed and sealed by
Anoush Ebrahimi, P.Eng.

Anoush brahimi, P. ng.
Principal Mining ngineer
Tetra Tech W I Inc.

2 . ERRY . HARRIS, PE, P.ENG.

I, Jerry W. Harris, of Golden, Colorado, USA, do hereby certify

- I am the Chief lectrical ngineer with Tetra Tech Inc. with a business address at 350 Indiana Street, Suite 500, Golden, Colorado, 80401.
- This certificate applies to the technical report titled Technical Report and Prefeasibility Study, Artillery Peak Project, dated June 28, 2012 and amended September 12, 2012 (the Technical Report).
- I am a graduate of University of Colorado, (BS , 1974). I am a member in good standing of the Association of Professional ngineers and Geoscientists of Alberta License 88444 and State Board of Technical Registration Arizona, License 53042. My relevant experience is lectrical ngineer responsible for Peebles Feasibility Study, KSM Pre-Feasibility, Freport MacMoran Bagdad Thickener Project, including numerous studies and engineering and construction projects over 37 years. I am a Qualified Person for purposes of ational Instrument 43-101 (the Instrument).
- I have not conducted a visit to the Property.
- I am responsible for Sections 18.7, 26.6.1, 27 and 28 of the Technical Report.
- I am independent of American Manganese Inc. as defined by Section 1.5 of the Instrument.
- I have only been involved in determining power supply for the property and preparing preliminary single line diagrams with the Property that is the subject of the Technical Report.
- I have read the Instrument and the parts of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the technical report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 12th day of September, 2012 at Golden, Colorado, USA

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Jerry Wayne Harris, P. ng., P
Chief lectrical ngineer
Tetra Tech Inc.

2 . MAR IN SIL A, PHD, PE, P.ENG.

I, Marvin Silva, PhD, P , P. ng., of Tucson, Arizona, do hereby certify

- I am a Senior Geotechnical ngineer with Tetra Tech Inc. with a business address at 3031 West Ina Road, Tucson, Arizona 85741.
- This certificate applies to the technical report titled Technical Report and Prefeasibility Study, Artillery Peak Project, dated June 28, 2012 and amended September 12, 2012 (the Technical Report).
- I am a graduate of the ational Autonomous University of icaragua (B.Sc Agricultural ngineering, 1981); Institute of Odessa in Ukraine (M.Sc. Water Resources ngineering, 1985); and University of Alberta (Ph.D. Geoenvironmental ngineering, 1999). I am a member in good standing of the Association of Professional ngineers and Geoscientists of the Alberta (52477). My relevant experience with respect to Geotechnical ngineering includes 21 years of experience in mining and infrastructure projects in Canada and the United States of America, including 7.5 years dedicated to research of mine waste tailings and design of tailings storage facilities. I am a Qualified Person for purposes of ational Instrument 43-101 (the Instrument).
- I have completed a personal inspection of the Property on February 25, 2011.
- I am responsible for Sections 18.8, 18.9, 26.6.2, 26.6.3, 27 and 28 of the Technical Report.
- I am independent of American Manganese Inc. as defined by Section 1.5 of the Instrument.
- I have had no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the parts of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 12th day of September, 2012 at Tucson, Arizona, USA.

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Marvin Silva, PhD, P , P. ng.
Senior Geotechnical ngineer
Tetra Tech Inc.

2 . SABRY ABDEL HAFE , PHD, P.ENG.

I, Sabry Abdel Hafez, PhD, P. ng., of Vancouver, British Columbia, Canada, do hereby certify

- I am a Senior Mining ngineer with Tetra Tech W I Inc. with a business address at 800-555 West Hastings Street, Vancouver, British Columbia, V6B 1M1.
- This certificate applies to the technical report titled Technical Report and Prefeasibility Study, Artillery Peak Project, dated June 28, 2012 and amended September 12, 2012 (the Technical Report).
- I am a graduate of Assiut University, (B.Sc Mining ngineering, 1991; M.Sc. in Mining ngineering, 1996; Ph.D. in Mineral conomics, 2000). I am a member in good standing of the Association of Professional ngineers and Geoscientists of British Columbia (34975). My relevant experience is in mine evaluation. I have more than 19 years of experience in the evaluation of mining projects, advanced financial analysis, and mine planning and optimization. My capabilities range from the conventional mine planning and evaluation to the advanced simulation-based techniques that incorporate both market and geological uncertainties. I have been involved in the technical studies of several base metals, gold, coal, and aggregate mining projects in Canada and abroad. I am a Qualified Person for purposes of ational Instrument 43-101 (the Instrument).
- I have not conducted a personal inspection of the Property.
- I am responsible for Sections 1.13, 1.14.1, 19, 22, 24 and 25.6 of the Technical Report.
- I am independent of American Manganese Inc. as defined by Section 1.5 of the Instrument.
- I have had no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the parts of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 12th day of September, 2012 at Vancouver, British Columbia, Canada.

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Sabry Abdel Hafez, PhD, P. ng.
Senior Mining ngineer
Tetra Tech W I Inc.

2 .10 HASSAN GHAFFARI, P.ENG.

I, Hassan Ghaffari, of Vancouver, British Columbia, Canada, do hereby certify

- I am a Manager of Metallurgy with Tetra Tech W I Inc. with a business address at 800 555 West Hastings Street, Vancouver, BC.
- This certificate applies to the technical report titled Technical Report and Prefeasibility Study, Artillery Peak Project, dated June 28, 2012 and amended September 12, 2012 (the Technical Report).
- I am a graduate of the University of Tehran (M.A.Sc., Mining ngineering, 1988) and the University of British Columbia (M.A.Sc., Mineral Process ngineering, 2004).
- I am a member in good standing of the Association of Professional ngineers and Geoscientists of the Province of British Columbia (30408).
- My relevant experience with respect to mineral process engineering includes 22 years' experience in mining and plant operation, project studies, management, and engineering.
- I am a Qualified Person for purposes of ational Instrument 43-101 (the Instrument).
- I have not visited the Property subject of this report
- I am responsible for Sections 1.11, 21.1, 27 and 28 of the Technical Report.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have had no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the Technical Report has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 12th day of September, 2012 at Vancouver, British Columbia, Canada.

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Hassan Ghaffari, P. ng.
Manager of Metallurgy
Tetra Tech W I Inc.

2 .11 NORM TRIBE, P.ENG., B.A.SC.

I, orm Tribe, P. ng., B.A.Sc., of Kelowna, British Columbia, Canada, do hereby certify

- I am a Geological Consultant with .Tribe Associates Ltd. with a business address at 2611 Springfield Road, Kelowna, BC, V1 1B9.
- This certificate applies to the technical report titled Technical Report and Prefeasibility Study, Artillery Peak Project, dated June 28, 2012 and amended September 12, 2012 (the Technical Report).
- I am a graduate of the University of British Columbia (degree in Geological ngineering, 1964). I am a member in good standing of the Association of Professional ngineers and Geoscientists of British Columbia, (Registration 11330). I have practiced my profession in geology for a total of seven years with experience in exploration, advanced evaluation and operations. My relevant experience is worldwide and in all phases of mineral exploration over the past 48 years. I am a Qualified Person for purposes of ational Instrument 43-101 (the Instrument).
- I have completed a personal inspection of the Property on ovember 18 to 22 2011.
- I am responsible for Sections 1.5.1, 14.1, 25.1.1, 27 and 28 of the Technical Report.
- I am independent of American Manganese Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the parts of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 12th day of September, 2012 at Kelowna, British Columbia, Canada.

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orm Tribe, P. ng., B.A.Sc.
Geological Consultant
.Tribe Associates Ltd.