Report to:

East Africa Metals Inc.

East
Africa
Metals

National Instrument 43-101 Technical Report and Preliminary Economic Assessment for the Terakimti Oxide Deposit, Harvest Project, Tigray National Regional State, Ethiopia

Project No. 704-ENG.VMIN03019-01

TETRA TECH

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EAST AFRICA METALS INC.

NATIONAL INSTRUMENT 43-101 TECHNICAL REPORT AND PRELIMINARY ECONOMIC ASSESSMENT FOR THE TERAKIMTI OXIDE DEPOSIT, HARVEST PROJECT, TIGRAY NATIONAL REGIONAL STATE, ETHIOPIA

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GLOSSARY

Units of Measure

ampere .... A
centimetres .... cm
coefficient of variation CV
cubic centimetre .... cm^3
cubic metre .... m^3
cubic metres per second .... m^3/s
day .... d
degrees Celsius .... °C
degrees .... °
gram .... g
grams per cubic centimetre .... g/cm^3
grams per litre .... g/L
grams per tonne .... g/t
hectares .... ha
hectares .... ha
hour .... h
kilogram .... kg
kilograms per tonne .... kg/t
kilograms per tonne .... kg/t
kilometres .... km
kilopascal .... kPa
kilotonne per year .... kt/a
kilotonne .... kt
kilotonnes per annum .... kt/a
kilovolt ampere .... kvA
kilovolt .... kV

kilovolt	kV
kilowatt hour per tonne	kWh/t
kilowatt hour per tonne	kWh/t
kilowatt hour	kWh
kilowatt hour	kWh
kilowatt	kW
kilowatt	kW
litres per hour per square metre	L/h/m^2
litres	L
metres above seal level	masl
metres	m
microns	µm
microns	µm
millimetres per annum	mm/a
millimetres per day	mm/d
millimetres	mm
million tonnes	Mt
millions of years	Ma
month	mo
parts per million	ppm
pascal	Pa
percentage	%
plus/minus	±
second	s
square kilometre	km^2
square kilometres	km^2
square metre	m^2
square metres	m^2
three dimensional	3D
tonne per day	t/d
tonnes per annum	t/a
tonnes per cubic metre	t/m^3
tonnes per month	t/mo
tonnes	t
troy ounce	tr oz
US dollar	US$
volt	V
year (annum)	a

ABBREVIATIONS AND ACRONYMS

Abrasion Work index	Ai
ammonium nitrate and fuel oil	ANFO
Arabian Nubian Shield	ANS
atomic absorption spectroscopy	AAS

Ball Work Index	BWi
Banded Iron Formation	BIF
barren leach solution	BLS
Beijing Donia Resources Co. Ltd	Beijing Donia
Beles Engineering Ltd. Pty. Co.	Beles
Beles Engineering Pvt. Ltd. Co.	Beles
Blue Coast Research Ltd.	BCR
Canaco Resources Inc.	Canaco Resources
Canadian Institute of Mining, Metallurgy and Petroleum	CIM
carbon-in-leach	CIL
carbon-in-pulp	CIP
CDN Resource Laboratories	CDN Labs
Certified reference material	CRM
copper	Cu
cost, insurance, and freight	CIF
differential global positioning system	DGPS
downhole electromagnetics	DHEM
East Africa Metals Inc.	EAM
east	E
Engineering, procurement and construction management	EPCM
Environment and Community Directorate Director	ECDD
Environmental and Social Impact Assessment	ESIA
Environmental, Health and Safety	EHS
Ethiopian Birr	ETB
Ethiopian Environmental Impact Assessment	EIA
Exploratory data analysis	EDA
Ezana Mining Development	Ezana
free board marine	FOB
free carrier	FCA
front-end loader	FEL
GEMS™	GEMS™
general and administrative	G&A
geographic information system	GIS
gold	Au
ground time-domain electromagnetic	TDEM
Harvest Mining PLC	Harvest Mining
induced polarization	IP
inductively coupled plasma-emission spectroscopy	ICP-ES
inductively coupled plasma-mass spectrometry	ICP-MS
internal rate of return	IRR
International Electrotechnical Commission	IEC
International Organization for Standardization	ISO
International Union for Conservation of Nature and Natural Resources	IUCN
inverse distance weighted to the third power	ID^3

lead	Pb
leak detection and recovery system	LDRS
life-of-mine	LOM
lime	CaO
McClelland Laboratories, Inc.	McClelland
Ministry of Mines, Petroleum and Natural Gas	MoMPNG
nearest neighbour	NN
net present value	NPV
net smelter return	NSR
north	N
Nubian Drilling Limited	Nubian Drilling
one barren leach solution	BLS
Ordinary Kriging	OK
Ore Research and Exploration Pty Ltd.	Ore Research
polyvinyl chloride	PVC
pregnant leach solution	PLS
Preliminary Economic Assessment	PEA
Qualified Person	QP
Quality assurance	QA
quality control	QC
Quantile-Quantile	Q-Q
rock quality designation	RQD
run-of-mine	ROM
SGS South Africa (Pty) Ltd.	SGS
sodium cyanide	NaCN
south	S
specific gravity	SG
Tetra Tech Canada Inc.	Tetra Tech
the Terakimti oxide and volcanogenic massive sulphide deposit	the Terakimti oxide deposit
Tigray Resources Inc.	TRI
US dollar	US$
volcanic-hosted massive sulphide	VHMS
volcanogenic massive sulphide	VMS
waste rock dump	WRD
west	W
Whittle™	Whittle™
work breakdown structure	WBS
World Health Organization	WHO
x-ray fluorescence	XRF
x-ray fluorescence	XRF
zinc	Zn

1.0 SUMMARY

1.1 INTRODUCTION

East Africa Metals Inc. (EAM) through its wholly-owned subsidiary Tigray Resources Inc. (TRI), holds a 70% interest in the Harvest Project, which includes the Terakimti oxide and volcanogenic massive sulphide (VMS) deposit (the Terakimti oxide deposit or the Terakimti Gold Heap Leach Project). The remaining 30% ownership is held by Ezana Mining Development (Ezana), a private Ethiopian company.

The Harvest Project is located in the southern part of the Arabian Nubian Shield (ANS) in the Tigray region of northern Ethiopia.

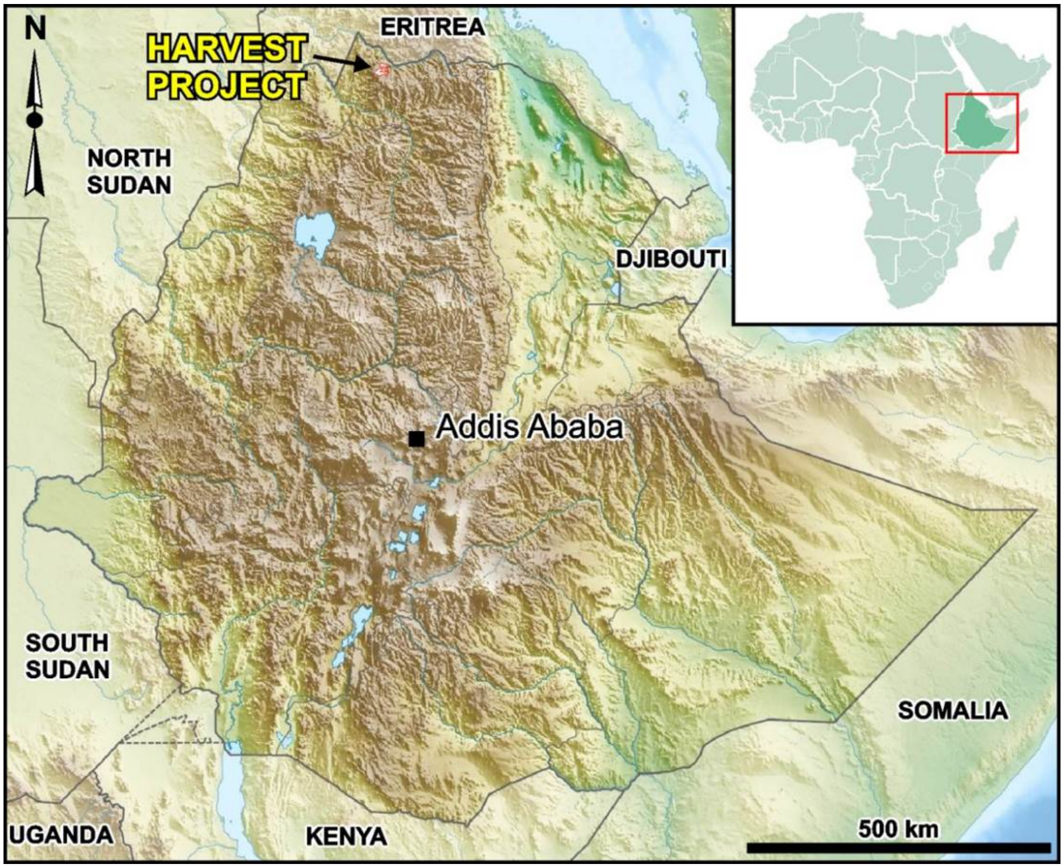
EAM commissioned Tetra Tech Canada Inc (Tetra Tech) to complete a Preliminary Economic Assessment (PEA) on the Terakimti Gold Heap Leach Project, which follows the Minerals Resource estimate prepared by Qualified Person (QP) David G. Thomas, P.Geo., of Fladgate Exploration Consulting Corporation (Fladgate).

The effective date of this PEA is April 30, 2018 and the effective date of the updated Terakimti oxide deposit Mineral Resource estimate is October 18, 2015.

1.2 PROPERTY DESCRIPTION AND OWNERSHIP

The Harvest Property is located in the Tigray National Regional State of the Federal Democratic Republic of Ethiopia, approximately 600 km (1,100 km ground distance) north-northeast of the capital city of Addis Ababa (population 3,385,000 in 2008), and 25 km north of the town of Shire (formerly Indaselassie) (Figure 1.1). The Federal Democratic Republic of Ethiopia comprises a total area of 1,104,300 km^2 and is located between longitudes 33°E to 48°E and latitudes 3°N to 15°N. The country is bounded by Eritrea to the north, Djibouti and Somalia to the east, Somalia and Kenya to the south, with North Sudan and South Sudan to the west.

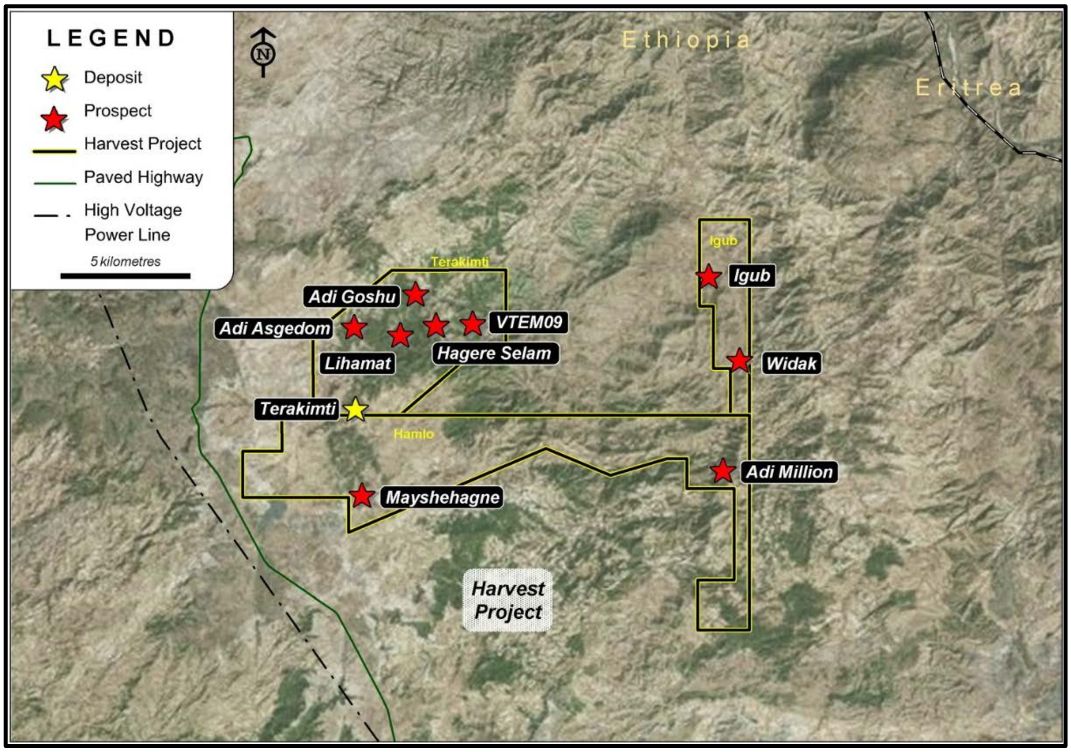
Figure 1.1 Harvest Property Location



Source: Archibald et al. (2014)

EAM and Ezana, through its Ethiopian subsidiary Harvest Mining PLC (Harvest Mining), currently holds one exploration licence for precious and base metal exploration (No. MOM\326-354\1999) which covers an area of 86.493 km². This exploration licence is in its seventh renewal period. The concessions are shown in Figure 1.2. The concession corners were established by geographic information system (GIS) co-ordinate points, and have not been surveyed or marked on the ground. The concessions were acquired from Ezana, as part of the formation of Harvest Mining. The four concessions were granted on January 11, 2007.

Figure 1.2 Harvest Property Concessions Map



Source: Harvest Mining (2016)

1.3 GEOLOGY AND MINERALIZATION

The Harvest Property is located in the Nafka Terrane within the prolific Arabian-Nubian Shield, a crustal remnant of the Neoproterozoic East African Orogeny. The geological setting includes multiple island arcs, back-arc basins, accreted oceanic crust, and late orogenic felsic intrusive bodies which are prospective for both polymetallic volcanic hosted massive sulphide and orogenic lode gold mineral deposits.

The geological setting in proximity to the Harvest Property is interpreted to comprise of a collapsed back-arc basin with the identification of both deep and shallow water sediments, basalts and mafics (some clastic) with some minor felsic volcanics. The entire terrane has undergone significant deformation during the collapse of the basin with isoclinal folding, recumbent folding, thrust and shear faults developing. Together all four of the concessions capture the majority of the stratigraphy succession during the life of the passive margin.

The overall structural trend of the Tigray region is northeast with multiple phases of folding and faulting observed across the belt including; isoclinal folding, recumbent folding, thrust and shear faults all of which have helped shape and influence deposition of the mineral occurrences found on the Harvest Property today.

To date the company has defined one deposit, the polymetallic Terakimti copper-zinc-gold-silver-lead VHMS deposit. The Terakimti oxide deposit is comprised of both near surface mineralized gossan (gold-silver oxide deposit), and deeper weathered sulphide bearing transitionary zone (supergene) and fresh sulphide hosted zone (primary). This near surface oxide deposit remains the near-term development goal for Harvest Mining PLC.

1.4 DRILLING

Exploration has been conducted in the region since the 1970's, with an extensive exploration effort dedicated to the Harvest Property and the Terakimti oxide deposit since 2004 by Ezana and their partner companies, and in 2007 by Harvest Mining. The work has included a systematic approach with remote sensing, regional and local scale geophysics, soil sampling, geological mapping, trenching, reverse circulation and diamond drilling. Nearly 40 significant mineral prospects have been identified for follow-up work. A total of 131 reverse circulation drillholes (6,611 m) and 112 diamond drillholes (21,036 m) have been completed on the Harvest Property, of which 127 reverse circulation drillholes (6,190 m) and 87 diamond drillholes (17,191 m) have been completed at the Terakimti oxide deposit.

In 2014, significant site work was completed that included upgrading and construction of numerous access and exploration roads, construction of a dedicated exploration camp at Terakimti including storage, office, dining and cooking sheds, a fuel depot (holding up to 3,000 L), a reverse circulation sample farm, a driller's storage shed, and tented accommodation.

The gossans at Terakimti have been known about since the 1960's. The government conducted extensive geophysical test work over the gossan zone at Terakimti in 1995, including induced polarization (IP) (resistivity and conductivity), and magnetic and radiometric surveys. In January 2007 an exploration licence was granted to Harvest Mining and exploration to 2009 included rock geochemical sampling, mapping, trenching, geophysical survey (magnetics, IP/resistivity) and diamond drilling over the gossan. This phase of exploration was managed by Beijing Donia. The prospect was first diamond drilled in late 2009 with the discovery of significant gold-copper-zinc-silver rich massive sulphide. In 2011, management of the project was transferred to Harvest Mining. From 2011-2014 the prospect was mapped at 1:500 scale, soils were geochemically sampled at 10 m x 10 m using hand-held x-ray fluorescence (XRF) and 40 m spaced soils for gold-silver assay, a microgravity survey and ground electromagnetic was conducted and 69 diamond drillholes were drilled (15,347 m) at 40 m x 40 m to 80 m x 40 m spacing. Holes were surveyed, petrology was completed, and five holes were cased for downhole electromagnetics (DHEM). In 2015 a total of 127 reverse circulation drillholes were completed (6,190 m), as were six diamond drillholes for metallurgical test work (271 m).

1.5 MINERAL RESOURCE ESTIMATE

In October 2015, Fladgate Exploration Consulting Corporation (Fladgate), an independent mineral resource consultant, completed an updated Mineral Resource estimate for the Terakimti oxide deposit which was publicly released on October 27, 2015. The previous sulphide Mineral Resource estimate is unchanged from January 2014.

A total of 81 core drill holes, 127 reverse circulation drillholes and 41 trenches for a total of approximately 25,970 m within the Terakimti database were used to support the Mineral Resource estimation. The drilling database comprises 12 core drillholes from the 2009-2010 due diligence drill campaign, 69 core drillholes from the 2013 drill campaign and 127 reverse circulation drillholes completed during 2014 and 2015. In addition, there are 41 trenches completed during 2014-2015. The model was completed using the development of three-dimensional solids representing the oxide and transition mineralization to constrain a block model in Minesight® software through Ordinary Kriging (OK) methods.

The Mineral Resource estimate was prepared in accordance with NI 43-101 and incorporates the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resource and Mineral Reserves. A summary of the oxide Mineral Resource estimate is shown in Table 1.1 and a summary of the sulphide Mineral Resource estimate is shown in Table 1.2.

Table 1.1 Terakimti Oxide Deposit Mineral Resource Estimate at a 0.5 g/t Gold Equivalent Cut-off, David G. Thomas, P.Geo. (Effective Date: October 18, 2015)

Classification	Tonnes (t)	Gold Equivalent (g/t)	Gold Grade (g/t)	Silver Grade (g/t)	Copper Grade (%)	Gold Metal (tr oz)	Silver Metal (tr oz)
Indicated	1,110,000	3.41	3.20	23.6	0.08	114,000	841,000
Inferred	15,000	2.06	1.94	13.5	0.04	1,000	7,000

Notes: Fladgate undertook data verification, and reviewed EAM's quality assurance and quality control programs on the Mineral Resource data. Fladgate concluded that the collar, survey, assay, and lithology data were adequate to support Mineral Resources estimation.

Domains were modelled in 3D to separate oxide, transition, supergene and primary sulphide rock types from surrounding waste rock. The domains conformed to lithological contacts logged in diamond drill core. Sub-domaining was further warranted to separate different grade populations and zones with differing strike and dip orientation within domains.

Raw drillhole assays were composited to 3 m lengths broken at domain boundaries.

High-grade assays were capped prior to compositing. Capping thresholds were assessed within each domain independently.

Block grades for copper, gold, and silver were estimated from the composites using Ordinary Kriging and into 2.5 m x 5.0 m x 2.5 m blocks coded by domain. The block model was re-blocked to a selective mining unit size of 2.5 m x 5.0 m x 5.0 m blocks for reporting of the Mineral Resource.

An average dry bulk density of the oxide zone was derived from specific gravity (SG) measurements on drill core and trench samples. Fladgate weighted the SG measurements by the proportion of each rock type within the oxide mineralization.

Blocks were classified as Measured, Indicated and Inferred in accordance with CIM Definition Standards.

Gold equivalent was estimated using undiluted grades, metal prices and heap leach process recoveries. The formula used is: gold equivalent = gold + (((silver price/31.103477) x (silver recovery))/(gold price/31.103477) x (gold recovery)

Metal prices used for gold and silver were US$1,300/tr oz, and US$17.50/tr oz respectively.

Metallurgical recoveries, supported by metallurgical test work were applied as follows: recoveries of 73.1% were applied for gold and 50.0% for silver. Copper and zinc are not recovered during the oxide phase and therefore are not considered a part of the oxide Mineral Resources.

The contained metal figures shown are in situ. No assurance can be given that the estimated quantities will be produced. All figures have been rounded to reflect accuracy and to comply with securities regulatory requirements. Summations within the tables may not agree due to rounding.

Table 1.2 Terakimti Sulphide Mineral Resource Estimate David Thomas, P. Geo. (Effective Date: January 17, 2014)

Classification	Material Type	NSR Cut-off ($/t)	Tonnes ('000s)	Cu (%)	Zn (%)	Au (g/t)	Ag (g/t)	Contained Metal Cu ('000 lb)	Zn ('000 lb)	Au ('000 tr oz)	Ag ('000 tr oz)
Indicated	Sulphide	23.9	1,841	2.20	1.65	1.06	17.5	89,477	66,871	63	1,033
	Subtotal Indicated	-	-	-	-	-	-	89,477	66,871	86	1,130
Inferred	Sulphide	23.9	2,583	1.09	1.42	0.96	20.6	62,187	77,101	80	1,712
	Underground Primary	63.9	939	0.69	2.92	0.84	15.2	14,198	60,358	25	459
	Subtotal Inferred	-	-	-	-	-	-	76,385	137,459	166	2,264

Notes: Fladgate undertook data verification, and reviewed the Harvest Project's QA/QC programs on the mineral resources data. Fladgate concluded that the collar, survey, assay, and lithology data were adequate to support Mineral Resources Estimation.

Domains were modelled in 3D to separate oxide, supergene and primary sulphide rock types from surrounding waste rock. The domains conformed to lithological contacts logged in diamond drill core. Sub-domaining was further warranted to separate different grade populations and zones with differing strike and dip orientation within domains.

Raw drillhole assays were composited to 5 m lengths broken at domain boundaries.

High-grade assays were capped prior to compositing. Capping thresholds were assessed within each domain independently.

Block grades for copper, zinc, gold, and silver and lead were estimated from the composites using a combination of OK and inverse distance weighted to the third power (ID^3) into 5 x 5 x 5 m blocks coded by domain.

Dry bulk density of the oxide, supergene and primary sulphide was estimated by ID^3 interpolation of SG measurements.

Blocks were classified as indicated and inferred in accordance with CIM Definition Standards.

Net smelter return (NSR) was estimated using undiluted grades, metal prices, recoveries, smelter treatment and refining costs.

Metal Prices used for copper, zinc, gold and silver were US$3.50/lb, US$0.9/lb, US$1,400/tr oz, and US$25/tr oz respectively.

Metallurgical recoveries, supported by metallurgical test work were applied as follows:

- Supergene zone: recoveries to copper concentrate of 87%, 36%, and 78% were applied for copper, gold and silver. Zero recovery of zinc from the supergene zone has been assumed. The supergene zinc metal content has not been included in the mineral resource tabulation.
- Primary zone: recoveries to copper concentrate of 89%, 45%, and 39%, were applied for copper, gold, and silver respectively. Recoveries to zinc concentrate of 85% and 10% were applied for zinc and silver.

A Lerchs-Grossman pit shell was generated from the NSR and using open pit mining costs of US$1.75/t. The total ore based costs (process and G&A) are US$25.9/t for oxide, and US$23.9/t for the supergene and primary rock types. A constant pit slope of 45° was used in the pit optimization.

Open Pit Mineral Resources were reported within the Lerchs-Grossman pit shell above an NSR cut-off equivalent to the total ore based costs stated above.

Underground Mineral Resources were reported within a grade shell generated at an NSR cut-off of US$63.90/t, assuming a US$40.00/t underground mining cost in addition to the ore based costs stated above. Isolated blocks were removed prior to tabulation.

The contained metal figures shown are in situ. No assurance can be given that the estimated quantities will be produced. All figures have been rounded to reflect accuracy and to comply with securities regulatory requirements. Summations within the tables may not agree due to rounding. The sulphide summation for contained zinc does not agree due to exclusion from the mineral resource of the contained zinc metal within the supergene zone.

Fladgate reviewed the impact of the reverse circulation drilling completed subsequent to the sulphide Mineral Resource estimate. The additional drilling did not result in any significant change to the volume or grade of the sulphide mineralization.

1.6 METALLURGY AND PROCESS

Several metallurgical test programs have been conducted since 2013. The test work includes tank and bottle leach and column leach tests, Bond work index tests and stacked material permeability (in columns) tests. The test work used various lithological samples collected from various space locations. In general, the mineralization from the Terakimti oxide zones is expected to be amenable to the conventional heap leach treatment. The test results produced by SGS South Africa (Pty) Ltd. (SGS) show an average gold extraction of 75.5% using bottle roll cyanide leach procedure at a grind size of 80% passing 106 µm. The gold dissolution rates vary varying from a low of 56.6% to a high of 91.2% for various individual samples. This implies that some of the gold may associate closely with its bearing minerals. In general, the silver extraction is low, averaging 47.1%.

Using the bottle test procedure, the effect of crushing particle size on the gold and silver extraction was tested on the composite blended from various lithological samples. At the crushing particle sizes of 100% passing 6, 11, and 16 mm, the extractions vary from 70.3 to 74.3% for gold and 32.2 to 34.4% for silver. Further tests were conducted on various lithological samples at a particle size of 100% passing 16 mm. The average gold extraction achieved was 75.9%, ranging 66.0 to 91.6%. The average silver extraction was 29.9%. All the test results indicate that the initial leach kinetics are rapid.

SGS also conducted two column leach tests on the composite sample identified as GOS Composite at a particle size of 100% passing 16 mm. The test results show that approximately 71% of the gold and 38% of the silver were extracted after the column leach for 47 days. A separate column cyanide leach testing was conducted by McClelland Laboratories, Inc. (McClelland) at coarse particle sizes of 100% passing 25 and 50 mm. Compared to the previous test results, both the gold and silver extractions are lower, ranging from 50 to 73% for gold and 7 to 38% for silver. On average, McClelland projected that the average extractions on the finer than 50 mm master composite sample could be approximately 65% for gold and 20% for silver. Compared to the tests conducted by SGS, the lower extractions may result from much coarse feed particle sizes and poor permeabilities. McClelland indicates that the low poor permeability may be due to high amount of fine material.

Both SGS and McClelland conducted the stack permeability tests. SGS tests suggest that the cement dosage for the mineralization is expected to 10 kg/t. McClelland suggests increasing cement dosage and limiting the leach pad height to improve the pad permeability. Further test work should be conducted to optimize the cement or bonding material type and dosage.

The grindability test results show that the material is very soft to ball mill grinding with an average Bond work index of 7.6 kWh/t. Also, the mineralization shows low abrasive characteristics.

According to the test results, a cyanide heap leach treatment is proposed for gold and silver recovery from the oxide mineralization. The processing will consist of:

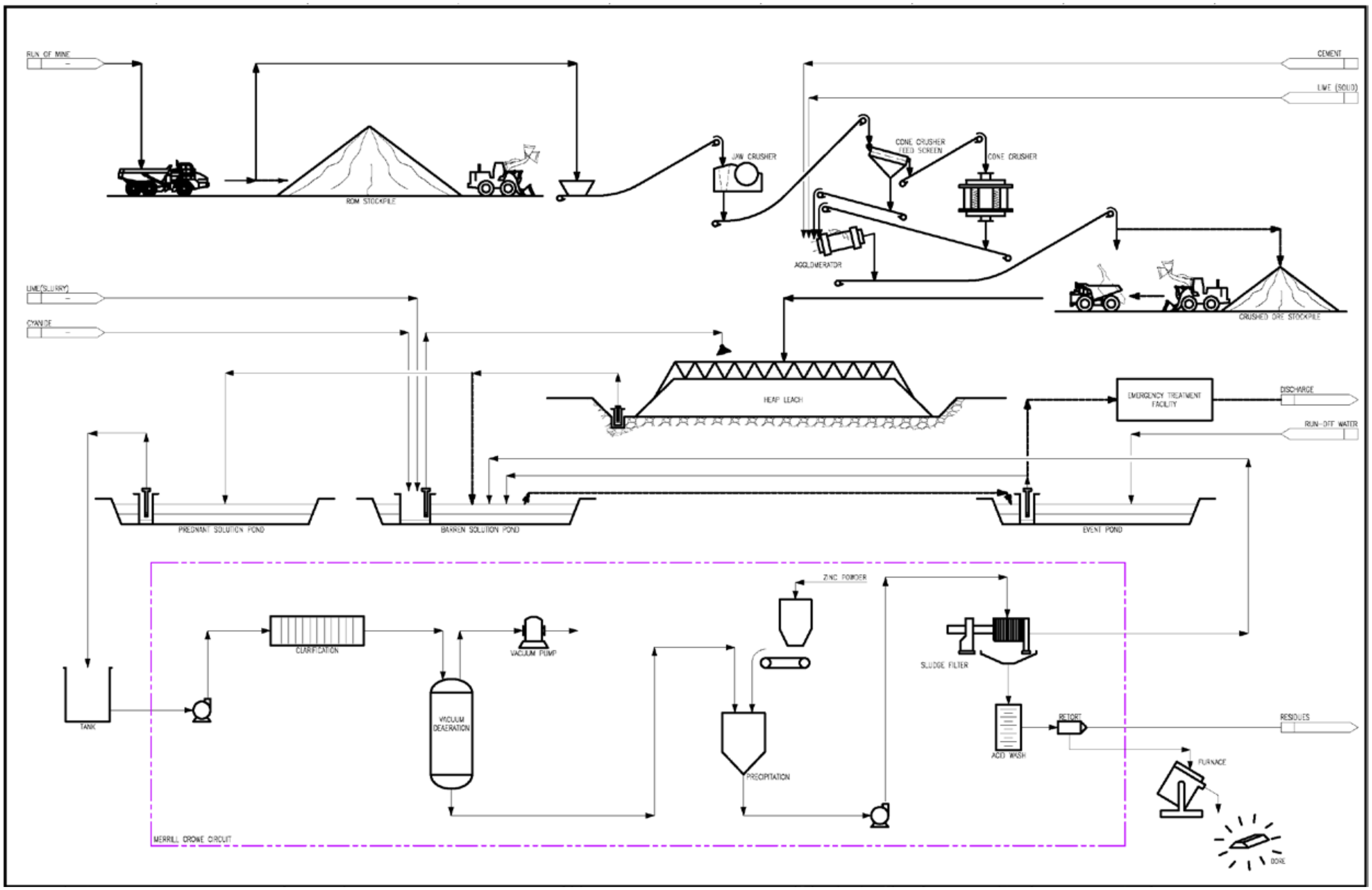
- crushing (two stages), screening and agglomeration
- heap stacking and leaching
- gold and silver recovery by Merrill Crowe processing.

A simplified flowsheet for the heap leach processing is shown in Figure 1.3

The heap leach feed will be crushed to approximately 100% 20 mm. The heap leach has been designed to process approximately 260 kt/a of the oxide mineralization. This would be equivalent to a throughput rate of 715 t/d. The crushing/leaching/gold recovery will operate 24 hours per day, seven days per week. The material that has been stacked on the heap leach pad will be continuously leached year-round. The availabilities will be 70% for the crushing and agglomeration circuits and 90% for the leaching and Merrill Crowe treatment circuits for planned downtimes, such as maintenances and shift changes, and unplanned downtimes.

Four water ponds have been planned for the heap leach processing: pregnant leach solution (PLS) pond, barren leach solution (BLS) pond, event pond, and polishing pond.

Figure 1.3 Simplified Process Flowsheet



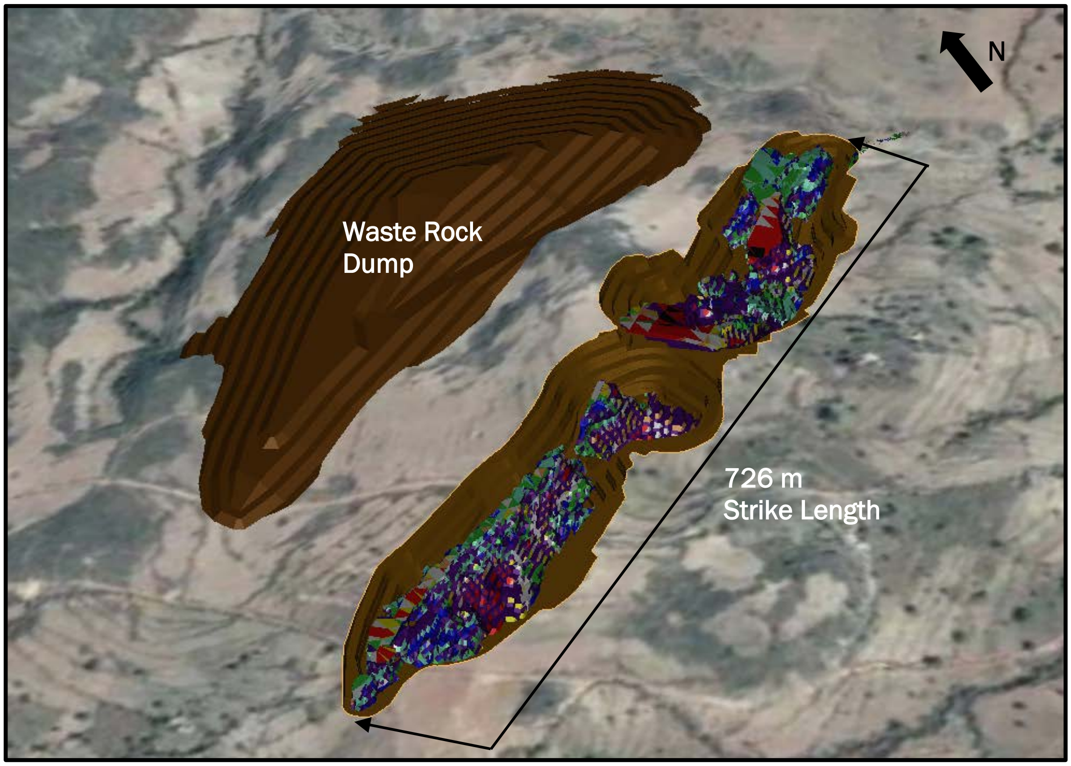
1.7 MINING METHODS

Tetra Tech completed an open pit mine plan for the Terakimti oxide deposit. An overall 3D view of the planned open pit is shown in Figure 1.4. The mine design and optimization were performed using GEOVIA GEMS™ (GEMS™) and Whittle™ (Whittle™) software.

The PEA open pit contains the following oxide resources:

- 1,086 kt of potential mill feed
- 4,093 kt of waste rock (including mineralized material below cut-off grade)
- 110,000 tr oz of gold
- 799,000 tr oz of silver.

Figure 1.4 Terakimti Oxide Deposit Open Pit



The breakdown of Mineral Resource categories included in the mine plan is shown in Table 1.2. Only 0.4% of Mineral Resources included in the mine plan are Inferred Resources.

Table 1.3 Breakdown of Open Pit Tonnage into Mineral Resource Categories

Resource Category	Percentage (%)
Inferred	0.4
Indicated	77.3
Measured	22.3
Total	100.0

Note: Totals may not match due to rounding.

Mining is planned to meet a design heap leach throughput of 715 t/d.

The pit shape follows the outcropping mineralised zone in a northeast-southwest orientation. The final pit designed for the PEA has three access areas, two on the eastern side of the pit as the main access from the process facility and one on the western side of the pit, which will be used for delivering waste to the waste rock dump (WRD).

1.8 PROJECT INFRASTRUCTURE

The Project infrastructure discussed in this PEA is limited to that required to mine and process the oxide zone of the Terakimti deposit. The infrastructure required includes a mine equipment maintenance facility, explosives storage facilities, processing facilities, crushing area, heap leach pad and ponds, general office infrastructure, and mine camp facilities.

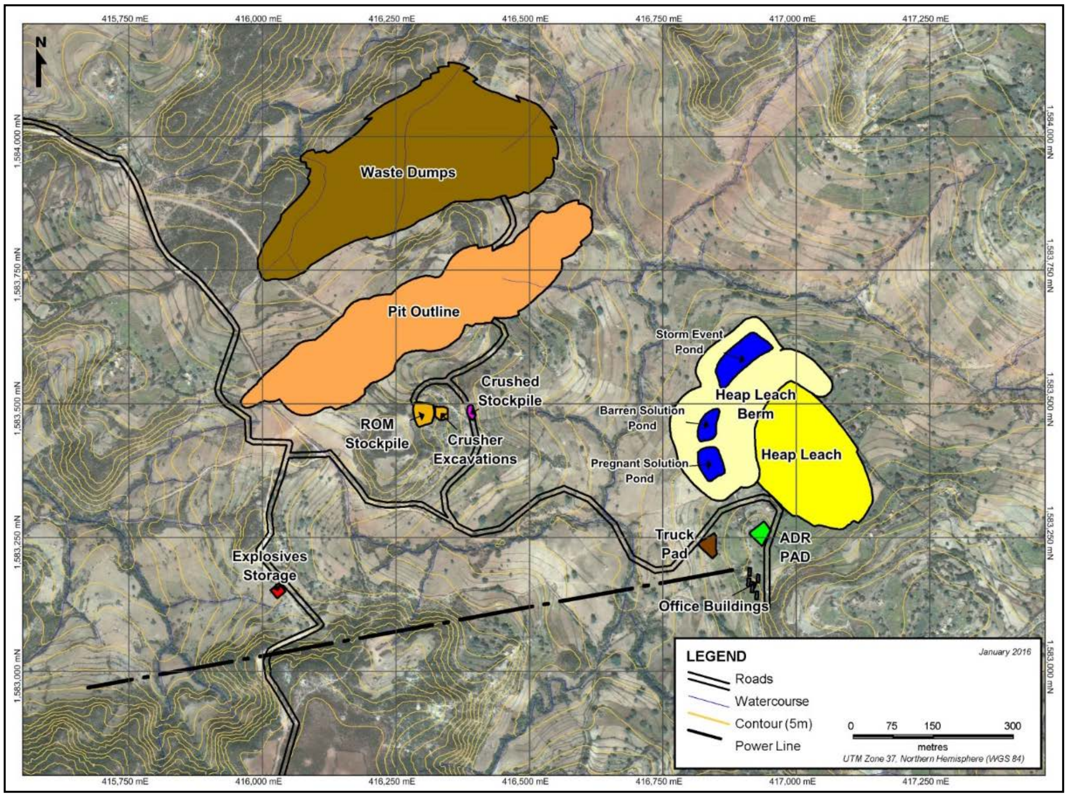
This PEA considers the use of existing utility grid power for the Terakimti deposit, for which the nearest high-tension power line is approximately 7 km away near Adi Dairo. A 230 kVA power station exists at Shire, approximately 30 km from the Terakimti Gold Heap Leach Project site and a 33 kV power line passes within 16 km of the site. A substation would be constructed at site to provide the required voltage for the mine site requirements.

Water required for mine, leach and process operation will be sourced through damming of surface drainages, dewatering of the pit and ground water wells. On-site water requirements will include dust control in the open pit and rock haulage/conveyor routes, process top-up water and, domestic and potable water for the camp and office. Process water will be recycled to the maximum practical extent with top-up water replacing evaporative and other losses.

Site roads will be constructed, as far as possible following existing roads serving the local population. The main access road will be from the south to provide for equipment and materials coming to site from Shire. Additional existing access roads from Adi Dairo and surrounding communities will be used by smaller vehicles only.

A conceptual site layout is shown in Figure 1.5.

Figure 1.5 Conceptual Mine Site Layout



1.9 WASTE MANAGEMENT, TAILINGS CONTAINMENT, AND WATER MANAGEMENT

Due to the nature of the proposed Terakimti mine, waste products are limited to waste rock from mining, small amounts of packaging, and other waste from processing. Heap leach operations do not include tailings. Waste rock will be placed in a WRD for reclamation at the end of the economic life of the operation. Waste including potentially hazardous materials from the processing will be separated and safely stored in a purpose built long term containment facility. The packaging materials will be disposed of in accordance with Ethiopian regulations and acceptable safe practice.

A comprehensive water management program will be developed for the construction, operation and post-closure stages of the Terakimti. The overall strategy will be to minimize water resources that come into contact with and to minimize water consumed by Terakimti. The Terakimti footprint will be designed to minimize impact on existing natural surface water courses. Should the Terakimti footprint require impact to an existing waterway, then an engineered diversion channel will be constructed so that the water way is safely diverted around Terakimti facility thereby maintaining the pre-existing natural quality of the water.

Similarly, berms will be designed to prevent surface run off (rainfall) from running into project facilities. Rainfall that falls within the process facility will be contained and collected so that it does not enter nearby natural waterways.

1.10 ENVIRONMENTAL AND SOCIAL IMPACT ASSESSMENT

Scoping and full-scale Environmental and Social Impact Assessment (ESIA) studies of the Terakimti oxide deposit area were undertaken by independent consultants and completed in October 2015 and April 2015, respectively. The study has established the biological and socio-economic baseline conditions; and identified impacts of the Project on the biological, physical and social environments and proposed mitigation measures. The study is conducted as per as per the requirements prescribed under the Ethiopian Environmental Impact Assessment (EIA) Proclamation (No. 299/2002) to proceed to the mining activities.

1.11 CAPITAL AND OPERATING COST ESTIMATES

1.11.1 CAPITAL COST ESTIMATE

The capital cost estimate for mining and processing the Terakimti Gold Heap Leach Project is shown in Table 1.3.

Table 1.4 Capital Cost Summary

Description		Cost (US$ million)
Direct Costs		
10	Overall Site	0.85
30	Mining	2.99
40	Process	4.31
50	Heap Leach Pads and Pond	1.29
70	On-site Infrastructures	1.12
Direct Cost Subtotal		**10.56**
Indirect Costs		
90	Project Indirects	3.37
98	Owner's Costs	0.65
99	Contingencies	2.60
Indirect Cost Subtotal		**6.62**
Total		**17.18**

Note: *Sustaining capital costs of $1.72 million include deferred construction of heap leach facility, replacement mobile equipment costs are not included in the table above.

1.11.2 OPERATING COST ESTIMATE

On average, the life-of-mine (LOM) on-site operating costs for the Project were estimated to be US$34.1/t of material processed, including the direct operating costs for mining, processing, site servicing, and general and administrative (G&A). Table 1.4 shows the cost breakdown for these areas. The expected accuracy range of the operating cost estimate is +35%/-25%. All the costs have been estimated in US dollars, unless specified.

Table 1.5 LOM Average Operating Cost Summary

Description	Cost (US$/t processed)
Mining (excluding pre-stripping)	15.10
Process	12.90
G&A and Site Services	6.11
Total Operating Cost	**34.11**

There will be no accommodation or catering services to be provided at the site. Personnel would commute to the site on a daily basis. The operating costs exclude shipping and marketing charges for gold and silver doré, which are included in financial analysis.

1.12 ECONOMIC ANALYSIS

A financial model for the development and operation of the Terakimti oxide deposit is presented in Table 1.5. The financial model is based on open pit mining followed by heap leaching to extract gold and silver from mined material. In summary, a mine with an operating life of four years, processing 1.0 Mt of oxide material is estimated to produce 71,000 tr oz of gold and 289,000 tr oz of silver resulting in a post-tax net present value (NPV) of $13.2 million and a post-tax internal rate of return (IRR) of 30.1%

Table 1.6 Summary Financial Model (Pre-tax)

Financial Summary	Units	Base Case Heap Leach
Tonnes Ore Mined	kt	1,086
Total Ounces Gold Recovered	'000 tr oz	71
Total Ounces Silver Recovered	'000 tr oz	229
Total Sales Revenue	US$ million	97
Royalties Paid to Ethiopian Government	US$ million	6.8
LOM Total Operating Costs	US$ million	37
Pre-production Capital	US$ million	17.2
Sustaining Capital	US$ million	1.7
Net Cash Flow (post-tax)	US$ million	21
NPV at 8% Discount (post-tax)	US$ million	13.2

1.13 Recommendations

1.13.1 General

Tetra Tech recommends that EAM advance the Terakimti Gold Heap Leach Project through completion of a Feasibility Study and detailed design of process and heap leach facilities.

Summary recommendations are noted below; further recommendations and costs can be found in Section 26.0.

1.13.2 Geology

Infill drilling of the Terakimti supergene and primary mineralization are recommended to fully assess the Terakimti oxide deposit. Other prospects on the property remain to be fully investigated and exploration work should continue to try and identify additional resources to complement those now identified. Supplementary work is required at the known Mayshehagne and VTEM09 VMS prospects to realize the potential of the existing mineralization. This would include metallurgical work and the assessment of potential mineralization extension. Additionally, the VMS trends identified should be tested for additional target using a deep electromagnetic (EM) testing system (both downhole and ground based).

1.13.3 Mining

Tetra Tech recommends that a geotechnical study is completed for the open pit to better understand the final pit wall slopes required to ensure stability.

1.13.4 Processing

Further tests are recommended to evaluate the metallurgical performances of the mineralization, including the variability tests and column leach tests. The test work should be conducted on the samples that better represent the oxide mineralization of the Terakimti oxide deposit, which is planned for the heap leach treatment.

1.13.5 Infrastructure

Tetra Tech recommends that EAM further investigate sources of electrical power and water.

1.13.6 Costs

Tetra Tech recommends that EAM approach additional vendors of equipment to look for opportunities to reduce costs.

2.0 INTRODUCTION

2.1 INTRODUCTION

EAM through its wholly-owned subsidiary TRI, holds a 70% interest in the Harvest Project, which includes the Terakimti oxide deposit. The remaining 30% ownership is held by Ezana, a private Ethiopian company.

The Harvest Project is located in the southern part of the ANS in the Tigray region of northern Ethiopia.

EAM commissioned Tetra Tech to complete a PEA on the Terakimti oxide deposit, which follows a Minerals Resource estimate prepared by QP David Thomas, P.Geo. of Fladgate.

The effective date of this PEA is April 30, 2018 and the effective date of the Terakimti oxide deposit Mineral Resource estimate is October 18, 2015.

2.2 QUALIFIED PERSONS

A summary of the QPs responsible for this report is provided in Table 2.1. The following QPs conducted site visits of the Property:

- David G. Thomas P.Geo., visited the Terakimti site from March 22 to 25, 2015.
- Mark Horan, P.Eng. visited the Terakimti site on April 6, 2017.

Table 2.1 Summary of QP Responsibilities

Report Section		Company	QP
1.0	Summary	All	Sign-off by Section
2.0	Introduction	Tetra Tech	Mark Horan, P.Eng.
3.0	Reliance on Other Experts	Tetra Tech	Mark Horan, P.Eng. Hassan Ghaffari, P.Eng.
4.0	Property Description and Location	Tetra Tech	Mark Horan, P.Eng.
5.0	Accessibility, Climate, Local Resources, Infrastructure and Physiography	Tetra Tech	Mark Horan, P.Eng.
6.0	History	Tetra Tech	Mark Horan, P.Eng.
7.0	Geological Setting and Mineralization	DKT	David G. Thomas, P.Geo.
8.0	Deposit Types	DKT	David G. Thomas, P.Geo.
9.0	Exploration	DKT	David G. Thomas, P.Geo.

table continues...

Report Section		Company	QP
10.0	Drilling	DKT	David G. Thomas, P.Geo.
11.0	Sample Preparation, Analyses and Security	DKT	David G. Thomas, P.Geo.
12.0	Data Verification	DKT	David G. Thomas, P.Geo.
13.0	Mineral Processing and Metallurgical Testing	Tetra Tech	Jianhui (John) Huang, Ph.D., P.Eng.
14.0	Mineral Resource Estimates	DKT	David G. Thomas, P.Geo.
15.0	Mineral Reserve Estimates	Tetra Tech	Mark Horan, P.Eng.
16.0	Mining Methods	Tetra Tech	Mark Horan, P.Eng.
17.0	Recovery Methods	Tetra Tech	Jianhui (John) Huang, Ph.D., P.Eng.
18.0	Infrastructure	Tetra Tech	Mark Horan, P.Eng.
19.0	Market Studies and Contracts	Tetra Tech	Mark Horan, P.Eng.
20.0	Environmental Studies, Permitting and Social or Community Impact	Tetra Tech	Hassan Ghaffari, P.Eng.
21.0	Capital and Operating Costs	Tetra Tech	Mark Horan, P.Eng. Hassan Ghaffari, P.Eng. Jianhui (John) Huang, Ph.D., P.Eng.
22.0	Economic Analysis	Tetra Tech	Mark Horan, P.Eng.
23.0	Adjacent Properties	Tetra Tech	Mark Horan, P.Eng.
24.0	Other Relevant Data and Information	Tetra Tech	Mark Horan, 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

Note: DKT – DKT Geosolutions Inc.

2.3 SOURCES OF INFORMATION

All sources of information for this report are noted in Section 27.0.

2.4 UNITS OF MEASUREMENT AND CURRENCY

All measurements are reported in metric, unless otherwise noted.

All currency is reported in US dollars, unless otherwise noted.

3.0 RELIANCE ON OTHER EXPERTS

The authors followed standard professional procedures in preparing the contents of this report. Data used in this report has been verified where possible and the authors have no reason to believe that the data was not collected in a professional manner.

Technical data provided by EAM for use by the authors in this PEA is the result of work conducted, supervised, and/or verified by EAM professional staff or their consultants.

Mark Horan, P.Eng. relied on the legal audit and opinion with regard to the title, mining concessions, and registration issues provided by Worku Fantahun Shumiye, Legal Affairs Consultant and Attorney at Law (Addis Ababa, Ethiopia) in a letter dated October 7, 2013. This information pertains to Section 4.0.

Mr. Horan also relied on Peter Granata, C.P.A., C.A., Chief Financial Officer of East Africa Metals, concerning tax matters relevant to this report. The tax, royalty, and government participation rates applied to the economic analysis in this PEA were taken from the current Ethiopian federal government tax proclamation for mining. This information pertains to Section 22.0.

Hassan Ghaffari, P.Eng. relied on Beles Engineering Pvt. Ltd. Co. (Beles) concerning environmental matters relevant to this report. Beles is one of the very few firms in Ethiopia which provides a wide spectrum of services in water, land and environment including consultancy, training, and construction. They provide services pertaining to consultations, investigations of natural resources, feasibility studies, design and design review, environmental and social impact assessment studies, and baseline surveys. This reliance is based on the report titled *Environmental and Social Impact Assessment of the Proposed Terakimti Gold Mining Project, Northwestern Tigray, Ethiopia* and dated April 2016. This information pertains to Section 20.0.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 PROJECT LOCATION

The Harvest Property is located in the Tigray National Regional State of the Federal Democratic Republic of Ethiopia, approximately 600 km (1,100 km ground distance) north-northeast of the capital city of Addis Ababa (population 3,385,000 in 2008), and 25 km north of the town of Shire (formerly Indaselassie) as shown in Figure 4.1. The Federal Democratic Republic of Ethiopia comprises a total area of 1,104,300 km^2 and is located between longitudes 33°E to 48°E and latitudes 3°N to 15°N. The country is bounded by Eritrea to the north, Djibouti and Somalia to the east, Somalia and Kenya to the south, with Sudan and South Sudan to the west.

Figure 4.1 Harvest Property Location

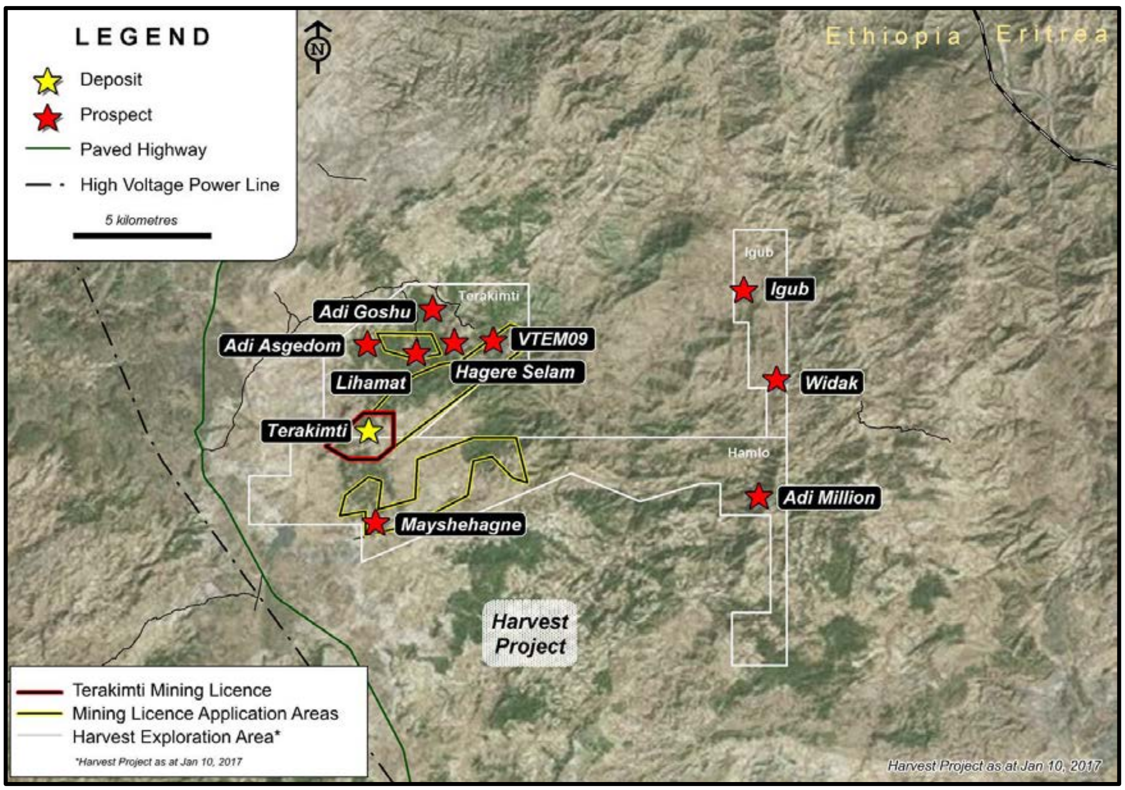
Source: Archibald et al. (2015)

4.2 MINERAL TENURE

As of January 10, 2017, Harvest Mining PLC held one exploration licence for precious and base metal exploration (No. MOM\326-354\1999) which covers an area of 86.493 km^2 (Figure 4.2). On August 11, 2016, Harvest Mining applied for a mining licence to cover the Terakimti deposit portion of the concession; the mining licence (covering 2.7682 km^2) was granted on December 6, 2017. An additional 17.126 km^2 is currently under application for a mining licence covering satellite targets outside the Terakimti deposit.

The concession corners were established by geographic information system (GIS) co-ordinate points, and have not been surveyed or marked on the ground. The concessions were acquired from Ezana, as part of the formation of Harvest Mining. The original four concessions were granted on January 11, 2007.

Figure 4.2 Harvest Property Concessions Map



Source: Harvest Mining (2017)

A joint venture was established between Ezana and Beijing Donia Resources Co. Ltd (Beijing Donia) in 2007, which included the transfer of ten exploration and two prospecting licences to three companies: Makeda Mining PLC, Donia Mining PLC, and Harvest Mining. Harvest Mining was a joint-venture company owned 70% by Beijing Donia and 30% by Ezana. Canaco Resources Inc. (Canaco Resources) acquired the 70% interest in Harvest Mining from Beijing Donia. On July 4, 2011, pursuant to a plan of arrangement, Canaco Resources transferred its interest in Harvest Mining to TRI (a previous wholly-owned subsidiary) along with working capital of CDN$4,000,000 in exchange for common shares of TRI, which common shares were then distributed to the Canaco Resources' shareholders. In February 2014, TRI entered into an agreement whereby EAM acquired all issued and outstanding common shares of TRI, along with the 70% interest in Harvest Mining. The remaining 30% is still held by Ezana. The Government of Ethiopia is entitled to a 5% free-carried interest.

There are no royalties, back-in rights, payments or encumbrances to Ezana in the current joint venture.

Table 4.1 Harvest Mining Exploration Concession Coordinates (as at Jan 10, 2017)

Tenement	Order	Long (X) (°)	Long (X) (min)	Long (X) (s)	EW	Lat (X) (°)	Lat (X) (min)	Lat (X) (s)	NS	License Size in 2017 (km^2)
Terakimti	1	38	12	31.66	E	14	19	18.81	N	29.140
Terakimti	2	38	12	31.32	E	14	20	56.40	N	
Terakimti	3	38	14	9.82	E	14	22	13.50	N	
Terakimti	4	38	16	32.04	E	14	22	13.95	N	
Terakimti	5	38	16	32.34	E	14	20	40.74	N	
Terakimti	6	38	15	59.00	E	14	20	40.60	N	
Terakimti	7	38	14	20.00	E	14	19	18.80	N	
Hamlo	1	38	11	52.49	E	14	19	18.80	N	46.950
Hamlo	2	38	21	36.00	E	14	19	18.80	N	
Hamlo	3	38	21	36.00	E	14	17	50.00	N	
Hamlo	4	38	21	17.00	E	14	17	50.00	N	
Hamlo	5	38	20	18.00	E	14	17	50.00	N	
Hamlo	6	38	20	18.00	E	14	18	26.00	N	
Hamlo	7	38	19	53.00	E	14	18	26.00	N	
Hamlo	8	38	18	42.81	E	14	18	5.86	N	
Hamlo	9	38	17	21.99	E	14	18	38.17	N	
Hamlo	10	38	13	15.99	E	14	16	57.00	N	
Hamlo	11	38	13	16.00	E	14	17	39.10	N	
Hamlo	12	38	11	3.80	E	14	17	39.10	N	
Hamlo	13	38	11	3.80	E	14	18	35.12	N	
Hamlo	14	38	11	52.65	E	14	18	35.28	N	
Igub	1	38	21	12.20	E	14	19	18.80	N	10.403
Igub	2	38	21	12.20	E	14	20	15.00	N	
Igub	3	38	20	51.00	E	14	20	15.00	N	
Igub	4	38	20	51.00	E	14	21	29.90	N	
Igub	5	38	20	33.70	E	14	21	29.90	N	
Igub	6	38	20	33.70	E	14	23	15.00	N	
Igub	7	38	21	36.00	E	14	23	15.00	N	
Igub	8	38	21	36.00	E	14	19	18.80	N	

Note: The "Harvest Licence" is comprised of Terakimti, Hamlo, and Igub
All coordinates shown are in Latitude/Longitude Adindan Projection

Table 4.2 Terakimti Mining Licence Coordinates

Mining Licence	Order	Long (X) (°)	Long (X) (min)	Long (X) (s)	EW	Lat (X) (°)	Lat (X) (min)	Lat (X) (s)	NS	License Size (km^2)
Terakimti	1	38	13	18.00	E	14	19	46.50	N	2.7682
Terakimti	2	38	13	53.50	E	14	19	46.50	N	
Terakimti	3	38	13	53.50	E	14	19	9.00	N	
Terakimti	4	38	13	35.00	E	14	18	54.00	N	
Terakimti	5	38	13	1.00	E	14	18	54.00	N	
Terakimti	6	38	12	34.00	E	14	19	9.00	N	

Note: All coordinates shown are in Latitude/Longitude Adindan Projection

Table 4.3 Harvest Mining Licence Application Coordinates

Corner Point	Latitude (°)	Longitude (°)	Application Area Size (km^2)
A	14.3550	38.2260	1.474
B	14.3546	38.2440	
C	14.3483	38.2468	
D	14.3469	38.2381	
E	14.3495	38.2286	
G	14.3296	38.2315	7.158
H	14.3296	38.2243	
I	14.3370	38.2313	
J	14.3418	38.2387	
K	14.3450	38.2467	
L	14.3461	38.2535	
M	14.3577	38.2695	
N	14.3556	38.2756	
O	14.3513	38.2755	
P	14.3361	38.2549	
Q	14.3150	38.2264	
R	14.3192	38.2315	
S	14.2971	38.2140	8.494
T	14.3036	38.2157	
U	14.3094	38.2234	
V	14.3080	38.2276	
W	14.2992	38.2264	
X	14.3024	38.2385	
Y	14.3150	38.2392	
Z	14.3218	38.2573	
AA	14.3218	38.2715	
AB	14.3076	38.2752	

table continues…

Corner Point	Latitude (°)	Longitude (°)	Application Area Size (km^2)
AC	14.3089	38.2706	8.494
AD	14.3158	38.2636	
AE	14.3145	38.2575	
AF	14.3036	38.2557	
AG	14.2939	38.2365	
AH	14.2910	38.2224	
AI	14.2985	38.2224	

Note: All coordinates shown are in Latitude/Longitude Adindan Projection

4.2.1 AGREEMENTS

Harvest Mining has been working with local communities to establish relationships and to consult with them regarding exploration activities since 2011. Land access agreements have been established with those land owners that have been impacted by exploration drilling programs undertaken to date.

Land ownership has been identified and mapped over a 365 ha area surrounding the Terakimti deposit. Following receipt of the Terakimti mining licence in December 2017, a relocation and resettlement plan for those affected by the mine site area has been initiated. This process is undertaken by the Ethiopian government and is currently underway.

As understanding of baseline socio-economic conditions has increased during the ESIA process, and Harvest Mining will continue to consult the local communities and establish land access agreements with land owners who will be affected by the development of the mine.

4.2.2 ROYALTIES

As outlined in Mining Operations Proclamation No. 678/2010 (as amended), Harvest Mining will pay government royalties of 7% based on sales price of the commercial transaction of previous metals produced. All operating cost estimates compiled for this PEA have been done on the basis that the Harvest Project is import tax and duty exempt.

4.2.3 PERMITTING

Table 4.3 shows the permits that have been received for the Terakimti Gold Heap Leach Project.

No other permits are expected to be required.

Table 4.3 Terakimti Gold Heap Leach Project Permitting Requirements

Statue	Authorization	Agency	Purpose
Mining Operations Proclamation (as amended) No. 678/2010	Large Scale Mining License	Ministry of Mines, Petroleum and Natural Gas (MoMPNG)	Authorization to undertake mining operations. In addition to outlining environmental, health, and social conditions, the license will specify the license area, type of mineral/s that can be extracted, the equipment and resources to be used, and the work program. The mining license includes the right to use water and timber found in the license area for the mining operation.
Environmental Impact Assessment Proclamation No. 299 of 2002	Authorization of ESIA Report	Ministry of Environment, Forest, and Climate Change	Approval of potential environmental and social impacts and proposed management measures.
Water Resources Management Proclamation No. 197/2000	Waste Water Discharge Permit	Ministry of Water, Irrigation and Electricity, Ministry of Environment, Forest, and Climate Change	Authorization to release or discharge waste into a waterbody.
Solid Waste Management Proclamation No. 513/2007	Solid Waste Management Permit	Ministry of Environment, Forest, and Climate Change	Authorization for the generation, storage, transportation, treatment, and disposal of non-hazardous solid waste.
Environmental Pollution Control Proclamation No. 300/2002	Hazardous Waste Permit	Ministry of Environment, Forest, and Climate Change	Authorization for the generation, storage, transportation, treatment, and disposal of hazardous waste.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESSIBILITY

The Harvest Property can be accessed directly by scheduled flights from Addis Ababa to Shire (population 43,967) during the dry season or Axum (population 47,320) year-round. Alternatively, the Harvest Property would be reached with a two-day drive from Addis Ababa.

The company head office is maintained in Shire Indaselassie. The Project area can be accessed from the office via a 20 km drive along a paved highway that passes along the western side of the concession area, followed by 10 km dirt road access to individual exploration concessions.

From Axum, the Harvest Property can be reached via a 70 km drive westward along a recently completed (2012) paved highway.

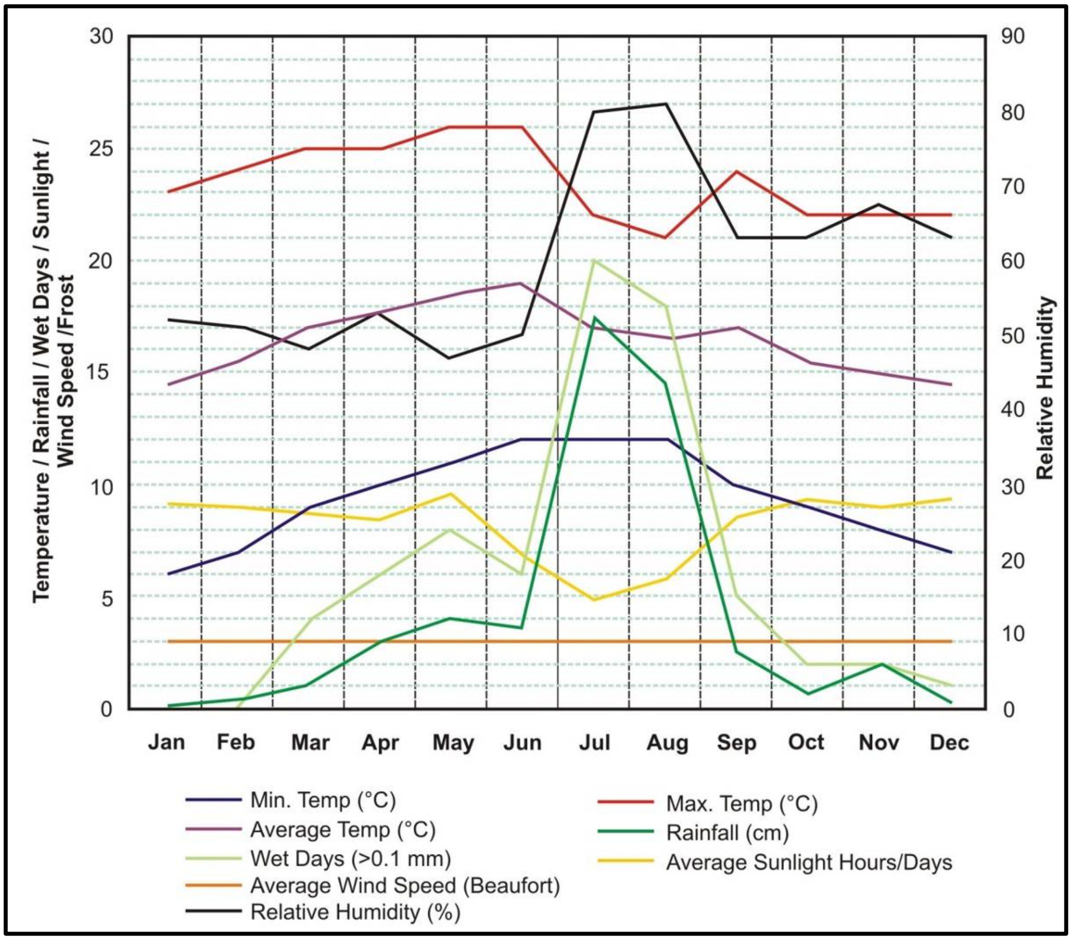
5.2 CLIMATE

The Project region is characterized by a temperate to hot climate and has both dry and wet seasons. The rainy season extends from mid-June to mid-September with average rainfall of 800 to 1,000 mm/a. Mean daily temperatures range from a high of 32.5°C in March to a minimum of 13°C in January. Most of the region is devoid of natural vegetation, with minor areas of shrub brush and trees most commonly located along tributaries and main drainages. Farming is the main land use and the growing season coincides with the rainy season. During this time crops such as teff and maize are grown for harvest in November (Archibald at al. 2014)

The climate graph below for Asmara (Figure 5.1) typifies weather in south-central Eritrea, 130 km northwest of the Harvest Property location. Extremes of heat are tempered by the elevated plateau present throughout much of Ethiopia.

Exploration activity can be conducted year-round, although extra caution must be exercised on the roads and while crossing streams in the wet season (June to September).

Figure 5.1 Climate Chart for Asmara, Eritrea (2,325 m)



5.3 PHYSIOGRAPHY

The Harvest Property is located in an area of varying relief ranging from 1,240 to 1,910 masl. Rivers in the area drain north-eastwards into the Mereb River. The Mereb River demarcates the northern Ethiopian border in the region, and is located 10 to 30 km to the northeast of the licence. Water is predominantly sourced through wells since tributary drainages are mostly seasonal. However, the larger rivers on the concessions (such as the Terakimti, Seye and Widake Rivers) rarely run dry and provide limited water for irrigation during the dry season. The Mereb can be dry for much of the year, and is subject to flash floods during the rainy season.

5.4 LOCAL RESOURCES AND INFRASTRUCTURE

Ethiopia is a landlocked country. Addis Ababa, the capital city has international air service to major cities in Africa and other global centres. There is good paved highway access from Addis Ababa to the port of Djibouti, a distance of 560 km. The highway is suitable for the movement of containerized and heavy lift cargo, and it is the main import and export route of the country. There is a paved highway from Addis Ababa to Axum and to Shire in northern Ethiopia, and this highway will be used to transport equipment and goods to Terakimti. In addition, the Government of Ethiopia has undertaken a major transportation infrastructure program for the country, including significant new railway routes. A new railway route from Addis Ababa is scheduled for completion in 2016. Although numerous rivers drain Ethiopia's diverse topography, the only navigable waterway is the Baro River (a tributary of the Nile), located on the country's western border with Sudan. Historically, Gambela in the southern part of the country has served as a port along the Baro River.

Approximately 50 civil airports exist in the country, including five in the Tigray Region (viz., Axum, Dansha, Humera, Makale, and Shire), along with two major military airports. In June of 2009, the Ethiopian government announced plans to construct 5,000 km of new railway for the country, primarily to facilitate the transportation of goods.

In the Harvest Property area, a major infrastructure initiative has recently taken place. This initiative included the construction of the main highway through the region, which is now paved between Axum and Shire, and continues northward to the Harvest Property. Shire is a university town with a population of 47,284 (2007 census). Many districts of the town have modern amenities such as running water, sewerage, and a hospital. A scheduled air service is operated during the dry season and a variety of commercial premises are located in the town. However, a subsistence lifestyle is evident in the villages in proximity to the Harvest Property, and only limited power and water is available for the inhabitants. Livestock and agriculture are emphasized.

High-voltage power lines are located along the Shire to Adi Dairo and Adi Nebried to Shiraro Roads (Figure 5.2), the voltage of the line is 33 kV.

Other significant infrastructure includes a large reservoir located 3 km southeast of Adi Dairo (Figure 5.3), a heliport 1 km south of Adi Dairo, and an airport at Shire. The cellular network and internet are reliable over the majority of the concession area.

Figure 5.2 Paved Road, Telephone Lines, and High-tension Power Lines near the Western Boundary of the Harvest Property

Source: Archibald et al. (2014)

Figure 5.3 Reservoir Approximately 3 km Southeast of Adi Dairo

Source: Archibald et al. (2014)

6.0 HISTORY

6.1 PROJECT OWNERSHIP AND EXPLORATION HISTORY

The Terakimti Project falls within the Harvest Property, owned by EAM and Ezana.

Historical ownership and exploration of the Harvest Property is summarized as follows:

- Exploration in the Harvest Project area during the 1970s was conducted by the Ministry of Mines, with work suspended from 1975 to 1993 due to civil war. Upon cessation of hostilities (1994 to 1995) Ministry of Mines fieldwork resumed in the form of regional and follow-up geological and geochemical surveys. Additional fieldwork, including geological mapping, trenching and IP geophysical surveys in selected areas was undertaken by the then Ethiopian Geological Survey.
- In 1996 the concessions were licensed to Ezana, an Ethiopian based company, who conducted additional stream sediment, soil and rock chip geochemical surveys. The surveys were successful in identifying targets prospective for base metals and gold.
- In 2004, Ezana contracted Fugro to carry out a 1:20,000 scale airborne electromagnetic survey covering the majority of the mapped surface sulphidic zones in the northern Shire area, outlining over 10 aerial electromagnetic anomalies with potential for volcanic-hosted massive sulphide (VHMS)-type copper-zinc-gold-silver-lead targets. These targets were then tested by drilling confirming the mineralization potential of the area.
- In 2007, Ezana completed a reverse circulation drill program, drilling one reverse circulation drillhole at Medadib (located in the eastern part of the Nefasit concession), encountering 20 m of gold enriched gossan, with the top 10 m grading 4.6 g/t of gold.
- Given the positive results of regional exploration, and similarities in geology to VHMS mineral discoveries in Eritrea, the Adi Nebried, Hamlo, Igub, Nefasit and Terakimti concessions were licensed on January 11, 2007, to Harvest Mining and Ezana under a joint-venture agreement. Other licences owned by Ezana were joint ventured to Makeda Mining (including the Terer property), and Donia Mining.
- In 2008, 1,920 rock chip samples were collected over the concession areas as part of a regional program including trenching. All samples were analysed by handheld x-ray fluorescence (XRF) machine, with 1,138 selected for laboratory assay. Results indicated anomalous gold, copper, zinc and lead. A regional

stream sampling program of 1,714 samples was conducted across the concessions.

- During 2009, Landsat and Quickbird image interpretation was conducted in addition to mapping and rock chip sampling. 4,274 rock chip samples were collected, of which 3,593 were collected as part of a regional program on 400 m spaced traverses and 681 trench samples were collected. All samples were analysed by XRF and 2,343 samples were selected for laboratory assay.
- In 2010, Harvest Mining conducted a regional rock chip program, mapping, soil sampling and trenching. 1,845 regional rock chip samples were collected on 200 m spaced traverse lines over the east to northeast portion of the Terakimti concession. These rock chips were analysed by handheld XRF. 8,018 soil samples were collected over Terakimti and samples were analysed by handheld XRF. Trenching (287 m in 6 trenches) was conducted in the northeast of the Terakimti concession, with a total of 153 samples collected and sent for laboratory analysis. Results included defined copper anomalies. Diamond drilling on the Property included 1,814 m in 17 holes, including 1,372 m in 12 holes at the Terakimti VHMS prospect and 442 m in 5 holes at the Adi Angoda prospect on the Nefasit concession.
- During 2011 exploration achievements included: purchase and interpretation of Worldview colour satellite imagery, 50 km^2 of 1 m topographic contour generation, 1:10,000 to 1:500 scale mapping totalling 15.68 km^2, 3,017 sieved soil samples collected and analysed by laboratory, 15,996 soil samples collected and analysed by handheld XRF, 233 stream sediment samples collected and analysed by laboratory, ground time-domain electromagnetic (TDEM) and gravity surveys over targets in Terakimti, Nefasit and Hamlo concessions; 9,800 m of diamond drilling on the Terakimti concession and 480 m on the Hamlo concession.
- 2012 exploration achievements included: 30 diamond drillholes totalling 7,199.95 m, 7 trenches totalling 434 m, 1,415 rock chip samples, 31,459 stream, soil and handheld XRF soil samples, 1,531 line-kilometres of airborne geophysics, and 101.38 km^2 of fact mapping at regional, local and detailed scales. The Midre Felasi license was relinquished as there were no further exploration targets identified to explore.
- 2013 exploration achievements included: 427 m of diamond drilling at the Terakimti and VTEM09 VHMS targets, 1398.5 m of trenching and 768 channel samples, first pass metallurgy and resource work for the Terakimti deposit, 18,780 soil samples for laboratory assay and/or handheld XRF analysis, 1,296 rock chip samples for laboratory assay and/or handheld XRF analysis and 39.7 km^2 of detailed mapping and 46.06 km^2 of regional mapping.

A summary of exploration work conducted in 2014 by Harvest Mining is presented in Section 9.0.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 GEOLOGICAL SETTING

7.1.1 REGIONAL GEOLOGY

Northern Ethiopia is located on the southern portion of the highly prospective Neo-Proterozoic ANS, an area of exposed continental crust formed during the prolific East African Orogeny during formation of the Gondwana supercontinent (Figure 7.1). The ANS formed over a period of approximately 300 Ma and is a coalescence of multiple island arcs, back-arc basins, accreted oceanic crust, and late orogenic felsic intrusive bodies. The oldest known rocks associated with the craton are 870 Ma and are found in eastern Sudan and southeast Arabia. The age of zircons dated from quartz porphyry intruding the Terakimti mineralized body is 884±19 Ma (Fengwei 2011). However, this age is in conflict with the accepted 870 to 650 Ma of age of formation of the belt. If the age is correct then the district is one oldest parts of the ANS.

Harvest is located on the Nakfa Terrane (Takar/Barka) which extends north of the Ethiopian border and to the South East of Sudan. The dominant structural trend is northeast swinging north-northeast and north in places. The terrane comprises of a fore arc to the east of the Haya/Nafka suture, a back-arc basin, or in the middle and stable continental terrane to the east. The amalgamation of the terranes occurred between 780 to 620 Ma, when collision and closure of back arc basins led to suturing. These suture zones are marked by ophiolites and intense deformation. The newly formed ANS underwent crustal thickening accompanied by melting and magma fractionation emplacing late cross cutting granite plutons (Ghebreab et al. 2009).

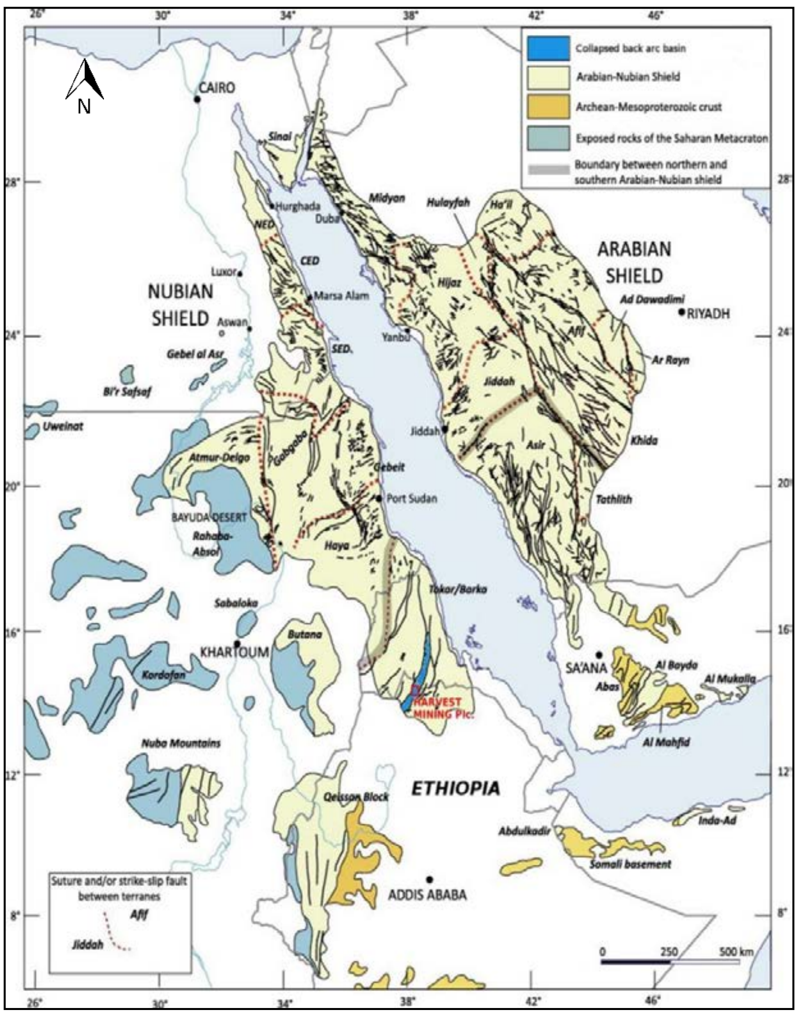
The Shiraro and Adi Hageray blocks are interpreted as the eastern margin of an island arc. The sequence contains many intrusive and extrusive mafic units (hornblende mafic, mafics), which are intercalated with sedimentary rocks. Late granodiorite and granite intrusions have deformed the dominant northeast trending stratigraphic zones into new geometries, typically into east-west or north trending structures. Late felsic dykes and quartz veins are common in the area.

The eastern margin of the Adi Hageray block is flanked by a deformed ultramafic-mafic belt known as the Zagar Mafic and Ultramafic belt, with ultramafic rocks including talc schists, peridotite, coarse grained pyroxenite and gabbro. The ultramafic package and larger gabbro bodies are interpreted to be thrusted ophiolite suite rocks.

The Adi Nebried block is interpreted as a collapsed back arc basin. The area contains intercalated felsic, intermediate and mafic extrusive rocks. Recumbent folding occurred during basin collapse. The dominant trend is northeast with moderate to steep dip northwest. During folding, shear zones and thrust zones developed creating duplication of the stratigraphy. Granites have been emplaced into the sequence at multiple stages; pre-deformation granites are deformed by the basin collapse, syn-deformation granites have ring dyke structures and/or shear fabric, and post deformation granites are circular and massive. Large porphyry intrusions are irregular and are interpreted to be related to the end of deformation and possibly intrude along shears and thrust zones.

The Chila block is inferred to be a sedimentary package within the collapsed back arc. The package includes a metamorphosed sequence of quartzite, meta-greywacke, amphibole schists, marble bands and graphitic schists, which are interpreted to be basin sediments. This sequence has undergone the same granite intrusive event as the Adi Nebried block.

Figure 7.1 The Arabian Nubian Shield Showing Suture Zones and Major Faults



Source: Diagram modified from Johnson et al. (2011)
Notes: The location of Harvest Mining exploration area is shown with a red box. The Adi Nebried (Asmara) back arc basin is shown as dark blue.

7.1.2 STRUCTURAL SETTING

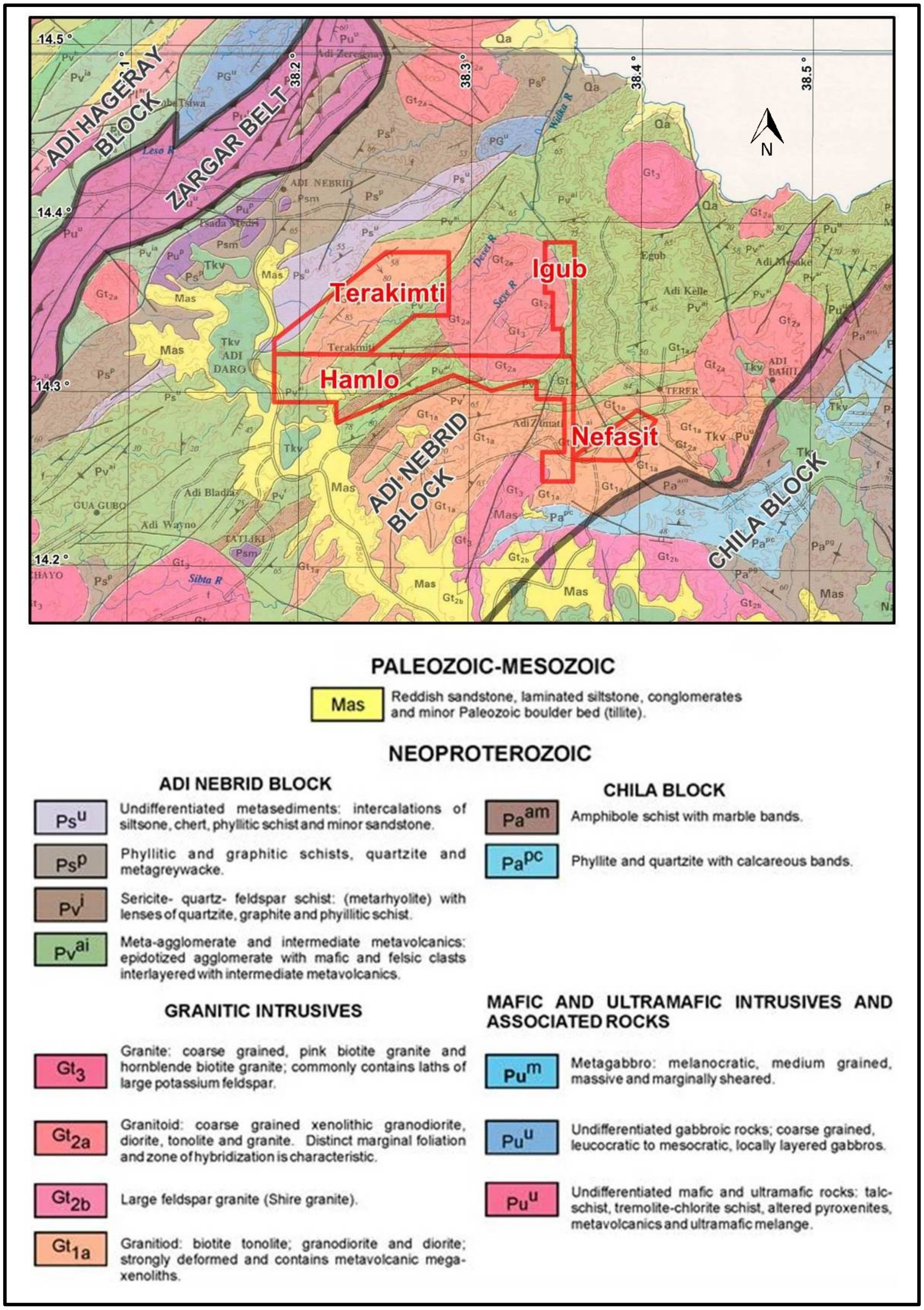
Numerous accreted terranes and microcontinents reflect episodic terrane collision and closure of the Mozambique Ocean from about 800 to 550 Ma as part of the East African Orogeny. Inner terranes collided against and deformed older Archaean to Mesoproterozoic crust of eastern Africa that were mainly high metamorphic grade. The final 100 Ma of the orogeny was dominated by strike-slip movement along suture zones between the obliquely colliding terranes. The ANS is characterized by an extremely broad group of such strike slip faults.

The overall structural trend of the Tigray region is northeast directed with multiple phases of folding and faulting observed across the belt including; isoclinal folding, recumbent folding, and thrust and shear faults. A recent detailed structural study has concluded that the Adi Nebried assemblage represents an intra-arc, paleo-oceanic trough or basin that was subject to transpressional deformation, followed by retrogressive lateral tectonic escape, leading to the formation of steeply dipping fold-shear structures, and sub-horizontal shears. Both massive sulphide zones and gold mineralized quartz veins are sheared and folded, and are often spatially associated. Where spatially associated, gold bearing veins cross-cut sulphide mineralization (Ghebreab et al. 2009).

7.1.3 LOCAL GEOLOGY

The geological setting in proximity to the Harvest Property is interpreted to comprise of a collapsed back-arc basin with the identification of both deep and shallow water sediments, basalts and intermediate volcanics, with some minor felsic volcanics. The entire terrane has undergone significant deformation during the collapse of the basin with isoclinal folding, recumbent folding, and thrust and shear faults developing. Together all four of the concessions capture the majority of the stratigraphic succession during the life of the passive margin. Figure 7.2 presents the location of the Harvest Licence concessions overlain on the 1:250,000 scale Bedrock Map from the Geological Survey of Ethiopia.

Figure 7.2 Local Geology from the 1:250 k Geological Survey of Ethiopia (1999) Axum Map Sheet



A summary of the dominant rocks, from interpreted oldest to youngest are described as follows and some examples are shown in Figure 7.3

Basement Rocks

The basement stratigraphy does not outcrop on the Harvest Property but does outcrop to the east and west. Both outcrops are sheared ultramafic complexes that have been thrust to the surface during the collapse of the basin. Both ultramafic complexes are interpreted to be oceanic crust. The western complex is interpreted as older basement crust that later became forearc. Whereas, the eastern complex is interpreted to be more juvenile back-arc basement crust that formed during the growth of the back-arc basin.

Sedimentary Package

During extension of the crust, before rifting was initiated and any volcanic activity begun, sedimentation is thought to have occurred. However, the expected sedimentary package has not been located. This package of rocks could have included conglomerates, sandstones and quartzites. Crucial to resolving this issue is understanding the metamorphic sedimentary rocks on the eastern edge of the Nefasit concession. At the time of compiling this report, these rocks have not been mapped sufficiently to resolve two possible theories about the nature of the eastern Nefasit rocks, these theories include: 1) the rocks are the pre-rifting sedimentary package, which are occurring along a thrust zone that has brought the deeper eastern ultramafic sequence to the surface; or, 2) these rocks outline the basement closure and are the sediments forming during the end of the rifting event.

Felsic and Mafic Dominant Volcanics (typical VHMS host rock)

Felsic and mafic dominant volcanics are the first sequence of rocks to have occurred during back-arc basement development and during the entire rifting event. From east to west, these rocks are found on the Nefasit, Hamlo, Igub and eastern Terakimti concessions. Typically, the rocks are commonly light green, green-grey to grey fine to medium grained. When weathered, the mafics can display remnant pillow structures and the felsics can be crusted with red-brown oxide mineralization. They are weakly deformed and can contain chlorite, amphibole, feldspar and sericite. The true thickness of this package is difficult to gauge, as the rocks are interpreted to be folded and perhaps structurally repeated across the concessions. The inferred thickness of the assemblage is between 2,000 and 3,000 m.

Regional mapping has identified that there is a repetition in the stratigraphy from felsic to mafic dominant lithology. Detailed mapping at prospective localities has identified that the packages are a complex sequence of inter-bedded extrusive and some intrusive rocks, that on occasion, include minor chert horizons. The extrusive rocks can include tuffaceous and volcaniclastic sequences. These rocks are very sparsely intruded by late quartz veins and very few felsic dykes. Chlorite or epidote alteration is common and appears to be spatially associated with inferred faults and possible shear zones.

The mafic dominant package is associated with thin Banded Iron Formations (BIFs), quartz veins and felsic dykes. The BIF layers are rare and are typically 10 to 30 m long in outcrop and less than 1 m thick. The quartz veins are common and are often parallel to stratigraphy and are sometimes 50 to 100 m long. Around the Terakimti deposit, quartz veins trend northwest and represent a different quartz veining event or reflect structural modification by growth faults. Felsic dykes, or sills, are rare and typically follow stratigraphy.

The significant VHMS prospects mapped to date on the Property occur within the felsic and mafic dominant package of rocks. This includes the Terakimti VHMS deposit and the Mayshehagne, VTEM09 and Adi Million prospects. Locally at the Terakimti and Mayshehagne VHMS prospects, BIF and chert horizons have also been found and jasperoid alteration is common.

Mafic Dominant Volcanics

These occur throughout the western and central part of the Terakimti concession and folded around the northern part of the Igub granite and onto the very northern part of the Igub concession. The unit compromises numerous volcanic sequences but is comprised dominantly of mafic (basalt) and mafic volcaniclastic rocks. The sequence is inferred to be 2,000 to 2,500 m thick, is displaced by late cross cutting faults and is interpreted to have been folded around the Igub syenite granite intrusion during the collapse of the bark-arc basin. Some intermediate volcanics are present and this may be related to bimodal volcanism at the time.

Typically, the mafics are fine grained, chloritic and weathered on the surface. However, the variety of mafics is quite complex and there are chlorite altered mafics with pyrite, mafics with 1 to 5 cm volcaniclastic clasts and coarse grained unaltered mafics. Chlorite is the dominate alteration, with epidote and silica alteration commonly associated with interpreted faults. In the mafic volcaniclastic rocks the clasts can often be epidote rich. The rocks appear to have undergone little deformation, however; folded quartz and epidote veining suggests ductile deformation occurred within the sequence. Cross cutting quartz veins, felsic and aplite dykes are common. BIF/Ironstone has been found as float across the package. The mafic dominant units form topographic highs, unless the sequence is dominated by volcaniclastic rock which is more susceptible to erosion.

Shale and Intermediate Volcanic Dominant Sequence

This sequence occurs on the western side of the previously held Adi Nebried concession. The sequence of rocks is between 1,500 to 2,500 m thick and is comprised of shale with silicified chert horizons inter-bedded with intermediate volcanics. Some mafic flows are present and one gabbro intrusion has been mapped to the southwest of the concession. The area is intruded by a few small granite plugs. The shale units are soft and light grey in colour with some units containing coarse pyrite. The shale, sometimes altered to sericite, contains silicified chert, banded chert and brecciated chert zones. The nature of the breccia is unknown but in places it appears to have a jigsaw texture suggesting possible hydrothermal association, if so this could suggest epithermal type mineralisation potential in the region. The basalts and mafic rocks in the area exhibit jasperoid, silica and epidote alteration to varying degrees and are sulphidic in places with several exposures exhibiting pyrite rich "pods". Quartz veining is observed, variably sulphidic and heavily oxidized with occasional boxwork texture. Topographically the silicified cherts form high ridges sometimes with steep cliffs.

Shale Dominant Sequence

This sequence of rocks does not occur on the Harvest Property but is the next stratigraphic unit and the last in the basin formation. It is included in this report for completeness. This sequence of rocks, formed in a deep quiet basin away from the rift zone and any other volcanic activity. The rocks outcrop to the west of the concessions and are interpreted to be between 500 to 5,000 m thick and trend roughly northeast. The variability in the thickness of the shale may be due to growth faults and structural repetition. The soft rheology of the shales has accommodated the majority of the strain during the basin collapse, and are likely to be highly contorted and sheared, and duplex complexes are possible. The shale occurs in fine grained massive sequences that rarely outcrop but form a white-grey powder clay-rich soil due to weathering. Outcrops of the shale are typically preserved as sericitized shear zones that form thin ridges that often extend for hundreds of metres. Topographically, the shale sequence forms low undulating ridges. It is believed that mapped structural thickening of this unit is of importance to the district as potential indicator of southeast directed faulting within the basin.

Syn-tectonic Granite Intrusions

Syn-tectonic syenite intrusives are alkaline feldspar dominant, coarse grained and pink in colour. Some smaller intrusions of coarse grained, white tonalities are also associated with the syenite, but their relationship has not yet been defined and could represent mineral segregation, or later intrusions as cross cutting dykes. The largest syenitic intrusion, known as the Igub granite, is 20 km in diameter, and overlaps portions of the Terakimti, Hamlo and Igub concessions. Its emplacement has deformed the older surrounding rocks, and has overturned the stratigraphic package to the northwest, thrusting out the felsic and mafic volcanics between the Terakimti and Igub concession and folding the Hamlo stratigraphy around the southern margin of the granite. Late faulting mapped on the south western and eastern margins of the Igub granite is interpreted to have occurred late in the basin collapse as the volcanic packages were deformed around the rigid granite. Interestingly, the Igub granite is possibly a ringed granite associated with a volcanic center, where rafts of country rock are still preserved within the granite. This is evident from local scale mapping on the Igub concession and regional satellite interpretation.

Several small syenite intrusions are mapped south of Hamlo and another east of Nefasit. Locally the stratigraphy has been deformed around these intrusions. Another interesting feature of the syenitic granites is that they are associated with later gabbroic intrusions that that have formed along their margins in pressure shadows.

Late Quartz Porphyry Intrusions

The rocks are coarse-grained and light grey to light green in colour. The porphyries intruded the mafic volcanic during fracturing and deformation. At the Terakimti deposit, the porphyries are possibly intruding along growth faults that formed during the deposit formation. Porphyry intrusions are key to the Ruwa-Ruwa gold trend with multiple gold targets hosted within the porphyry. Porphyries are also mapped in south Hamlo and in central Nefasit. These porphyries are parallel to stratigraphy and normal to principle stress during the collapse of the basin. However, the contacts are cross cutting and irregular.

Late Gabbro

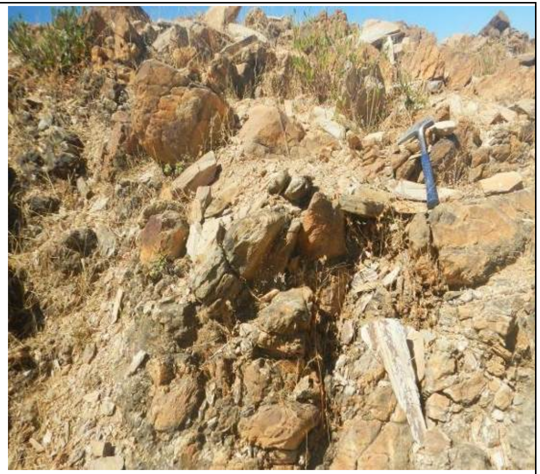
Gabbroic intrusions are mapped on the Nefasit and Hamlo concessions and are interpreted to relate to a mantle plume event. Intrusions are 100 to 500 m in diameter and often occur within granite pressure shadows. The rocks are typically fresh, coarse grained and outcrop as topographic highs. No sulphides have been identified in any of the intrusions mapped to date.

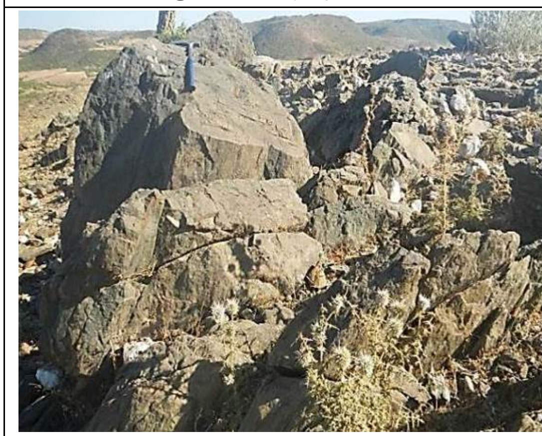
Cover

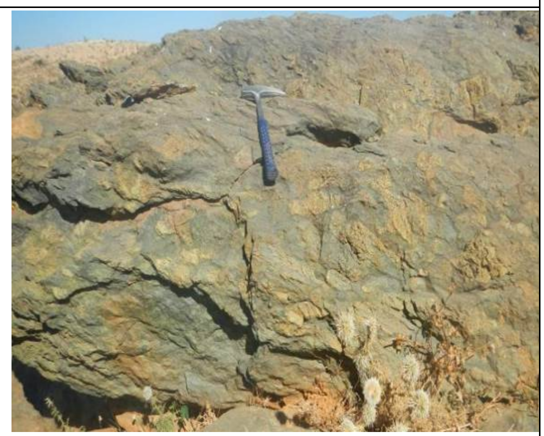
Late Palaeozoic to Mesozoic ironstone cover is found in the southeast corner of the Hamlo concession.

Figure 7.3 Photo Plate of Example Rock Types


BIF with magnetite and jasperoid, Terakimti


Felsic volcanic rock, Terakimti


Mafic rock with chlorite alteration, Terakimti


Mafic rock with epidote alteration, Terakimti

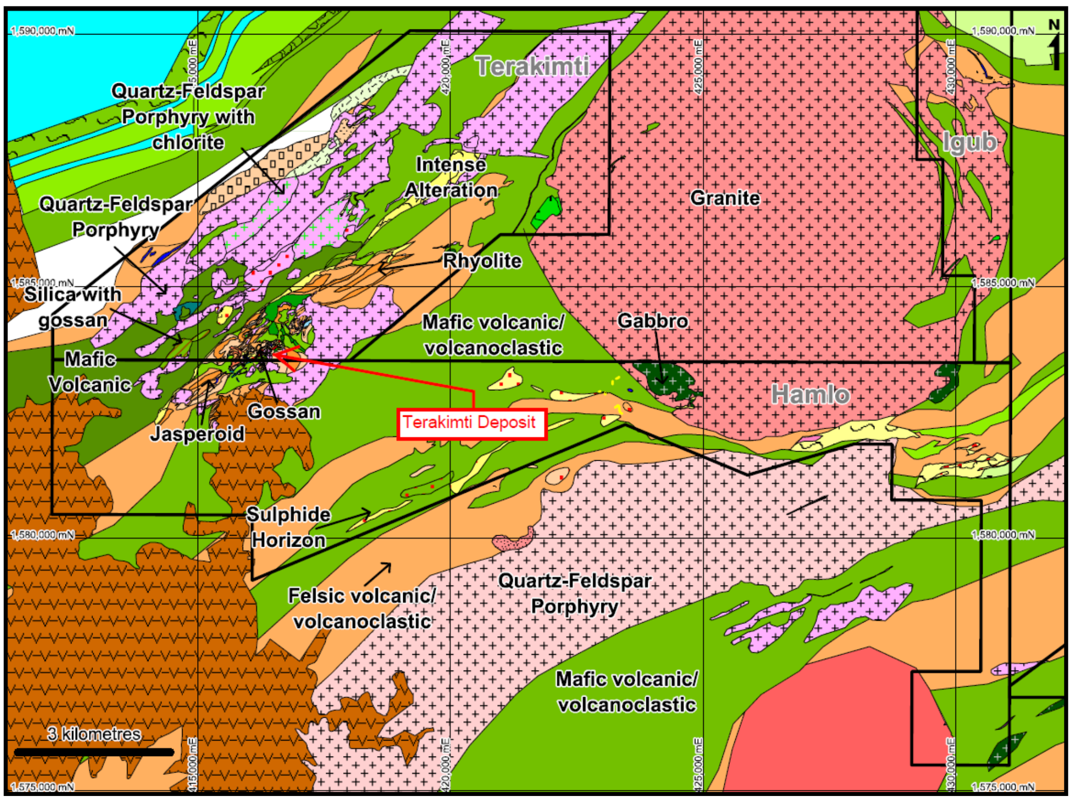
Intense sericitic alteration in felsic volcanic rock, Terakimti


Haematite-rich gossan, Terakimti

7.1.4 TERAKIMTI DEPOSIT GEOLOGY

The geology proximal to the Terakimti deposit comprises a northeast trending, central belt of intermediate porphyritic metavolcanic rocks flanked to the north by Neoproterozoic metasedimentary rock and granodiorite, and to the south by intermediate to mafic volcanic rocks (Figure 7.4). Significant chlorite, sericite, and silica alteration is associated with conformable gossanous horizons associated with the contact area of the intermediate and felsic volcanic rock packages, quartz-eye volcanic rocks and intrusive rocks are also present in this altered zone. The gossans are associated with polymetallic massive sulphide (gold-silver-copper-lead-zinc) mineralization at depth. Magnetic cherts are noted in the gossan area. The rocks have been affected by intense deformation, resulting in the development of a penetrative fabric in all lithologies, but in particular those rich in sericite. Local folding is present, but large-scale folds have not been identified.

Figure 7.4 Terakimti Deposit Geology



Source: Harvest Mining (2015)

TERAKIMTI DEPOSIT STYLE

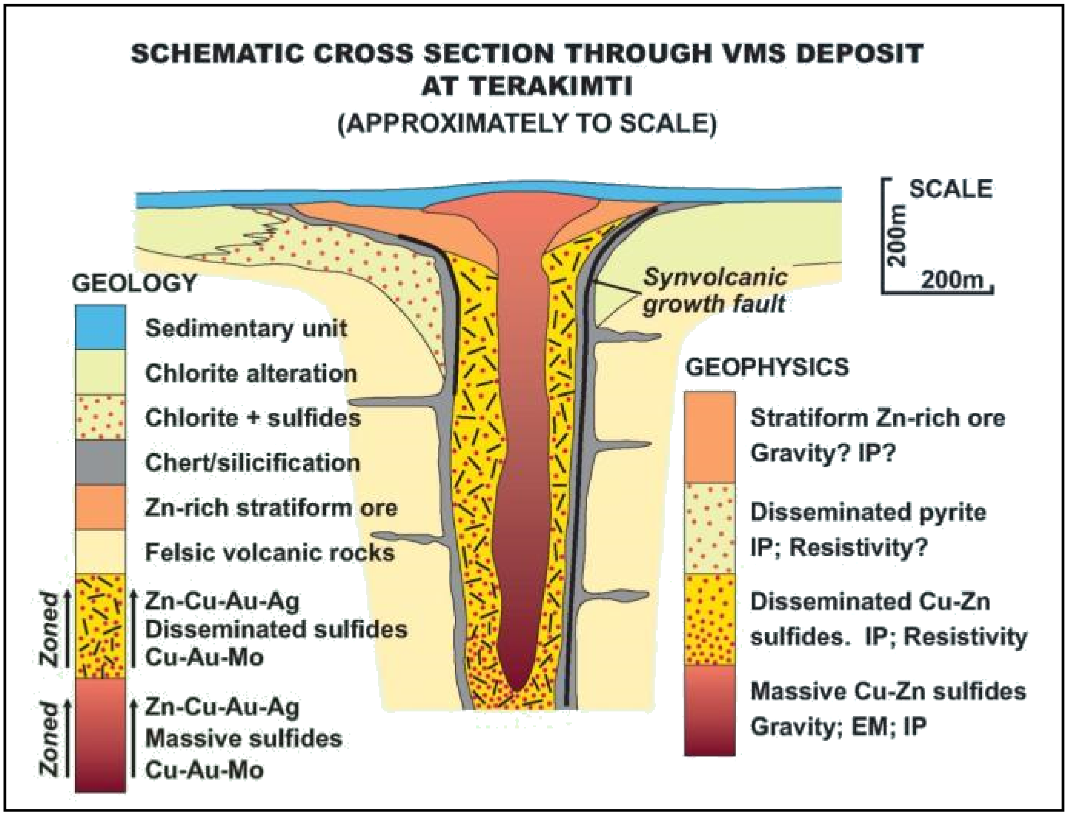
The largest known economic occurrence on the Harvest property is the Terakimti deposit, a Neoproterozoic volcanogenic hosted massive sulphide discovery. Extensive drilling at 40 m by 40 m to 40 m by 80 m drill spacings (20 m by 20 m in oxide), it is currently defined as a moderate-sized relatively high-grade copper-gold-silver-zinc-lead (Figure 7.5) occurrence containing multiple stacked lenses over 800 m strike and defined to depths of at least 260 m below surface. It is hosted within a bimodal volcanic sequence of intermediate and mafic volcanic (including pillow basalt) to volcanoclastic rocks. Numerous quartz-eye porphyry dykes intrude the centre of the mineralized system but the timing relationship is unclear (coeval to postdating mineralization). The VHMS is interpreted to be located along a syn-sedimentary fault, reactivated during regional compression, which cross cuts stratigraphy (Figure 7.6). The fault zone is defined in surface mapping as a zone of shearing and brittle faulting further northeast. In section, the interpreted fault zone includes sulphide breccia, porphyry and aplite dykes, jasperoid alteration of host rocks and laminated ore types.

Figure 7.5 Terakimti VHMS Mineralization (10HTD001)

Notes: Fine-grained pyrite, sphalerite and galena, cut by diffuse chalcopyrite veins from the Northern Lens, 10HTD-001.
Source: Archibald (2011)

Mineralization along the fault zone dips southeast at 40° to 90° and plunging at 20° to 40° degrees toward 073° to 090°. Bedding in the surrounding rocks generally dips 40° to the east, with open to tight folding observed. The Terakimti VHMS is interpreted to lie on the west limb of a regional tight syncline.

Figure 7.6 Model Highlighting Terakimti as a VHMS Crosscutting Stratigraphy Rather than a Stratabound Deposit



Notes: This indicates potential along the currently unexplored stratigraphic targets, especially in the south

The Terakimti VHMS system consists of at least four stacked lenses containing copper-gold-silver and variable zinc-lead:

- The Southern Lens is up to 50 m in true thickness, at least 360 m in strike, and up to 170 m high (Figure 7.7). It has a massive pyrite base up to 5 m thick and is a mound shaped lens. It is significantly supergene affected (upgraded) at the southern end of the north-northeast plunging Terakimti System. The peak primary sulphide intercept includes 73.85 m grading 3.77% copper, 1.31 g/t gold, 14 g/t silver, and 0.72% zinc in hole TD004. Gold enrichment has occurred in the weathered zone with a peak diamond drillhole intercept of 8.8 m grading 9.19 g/t gold and 78 g/t silver in drill hole TD029. The Southern Lode consists of massive fine-grained pyrite, with coarser grained chalcopyrite and lesser sphalerite as interstitial fill, pyrite replacement and fracture fill. Its average density is 4.07 (SG), including barren thin felsic intervals. It plunges at 20 to 45°.
- The Central Lens sits structurally above and flanks the Southern Lens, is up to 150 m high (dip component), up to 15 m in true width (averages 8 to 10 m) and is currently defined over 480 m down plunge (open down plunge) (Figure 7.8). This lens is well banded in places, somewhat tabular and reasonably predictable. The Central Lens is directly overlain by the hanging wall basalt,

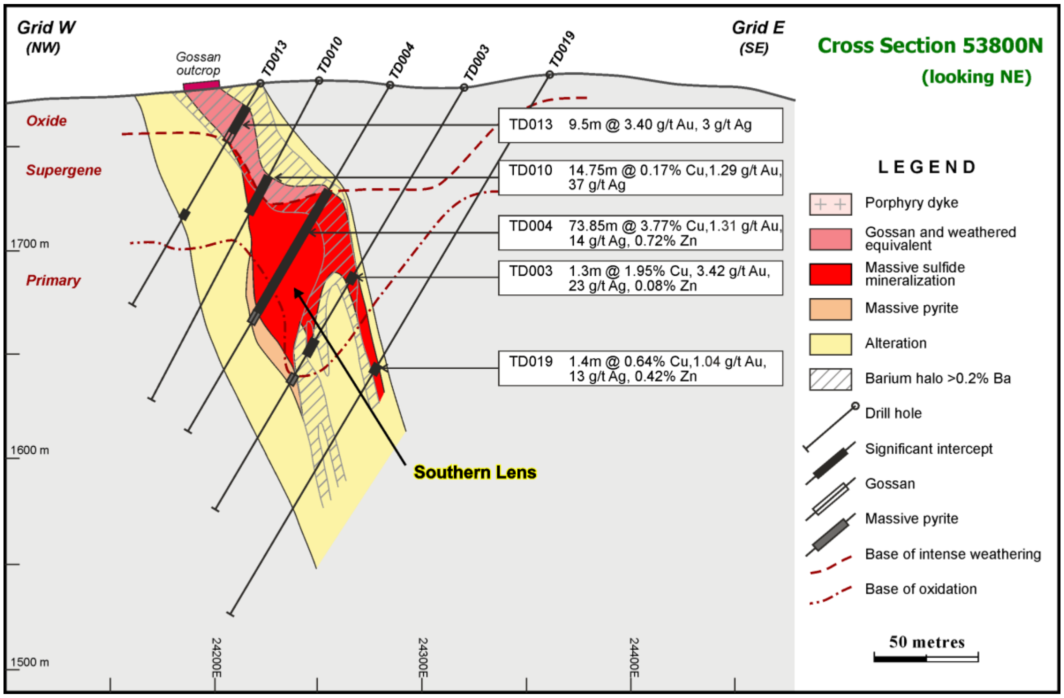
which is carbonate and jasperoid altered near sulphide mineralization. The peak diamond drillhole intercept is 15.2 m at 2.61% copper, 1.84 g/t gold, 43 g/t silver and 6.77% zinc; there are several laminated thin zones of galena rich sulphide in the northern section drilled. The peak reverse circulation oxide intercept is 33.0 m grading 7.198 g/t gold and 1.3 g/t silver in TRC058. The Lens has a shallow plunge in the southwest, steepening down plunge to the northeast and is roughly capped by the central porphyry intrusion. It is interpreted to lie within the main fault in the southwest and is located proximal to the main fault in the northeast.

- The Northern Lens is separated from the Southern Lens by a porphyry dyke swarm but the two lenses were unlikely to have joined (Figure 7.9). The northern lens is open down dip and plunge, strikes for at least 400 m, is up to 20 m true thickness and has a maximum down dip extent of 120 m thus far defined. This lens is also slightly banded and yields very high-grade gold gossans in the oxide zone (6.12 m at 27.2 g/t gold and 13 g/t silver) above the main high-grade shoot. Peak results from primary sulphide include 20.85 m at 5.67% copper, 1.48 g/t gold, 17.59 g/t silver and 0.77% zinc in TD008.
- A Lower Zinc Lens, has been intersected in several drill holes over a strike of 400 m with a vertical height of 30 m and is up to 10 m thick (overall cigar shaped lens). Very high-grade zinc occurs over 3.5 m at 1.41% copper, 2.09 g/t gold, 31 g/t silver and 23.03% zinc (TD040). This lens is interpreted to be on a separate structure to the east of the Central, Southern and Northern Lenses. The mineralization is typically brecciated with jasperoidal alteration and porphyry dykes intruding along the structure.

The near-surface part of the Terakimti system has been affected by supergene processes with distinctive vertical mineral zonation developed. These are:

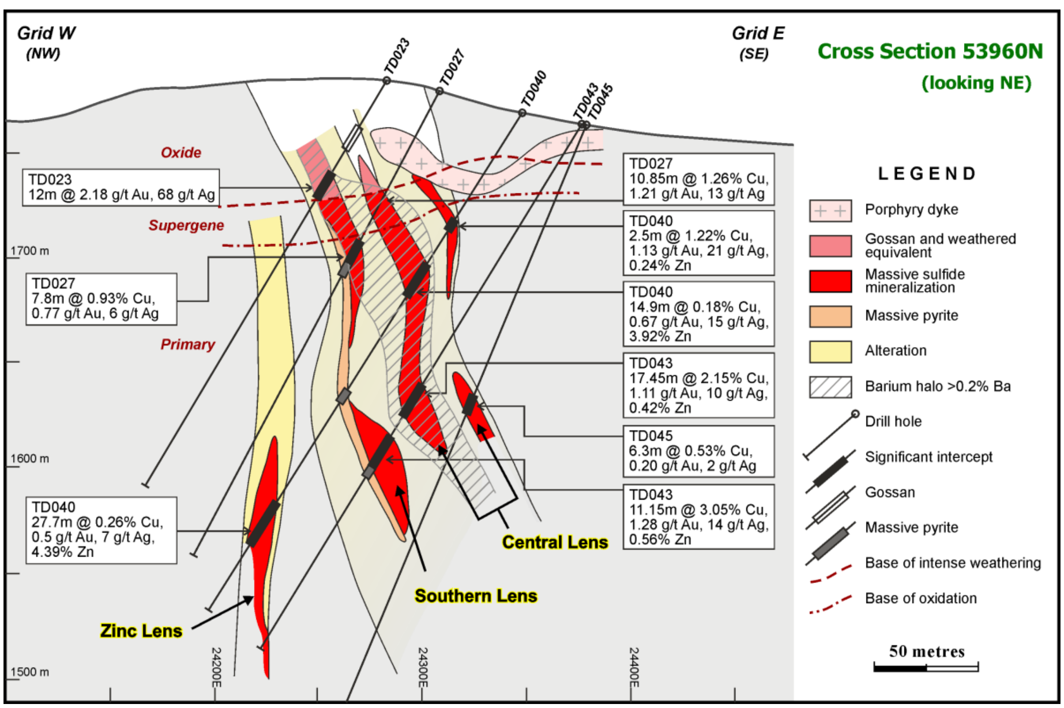
- Surficial gold enriched Oxide Zone (gossan). Gold is enriched with little to no sulphide, minor to no copper, zinc or silver and elevated lead. There may be a weak leached zone before the transition.
- Silver Enriched Transition Zone with variable gold. Pyrite remains but no other primary ore minerals are present. There is weak copper as covellite, pyrite, high gold and high silver (300 g/t gold).
- The Supergene Copper Zone which is largely primary with 5 to 20% secondary minerals (mainly covellite, minor chalcocite) with chalcopyrite present. Sphalerite locally remains and only chalcopyrite is significantly affected.
- Primary Zone of which there are several different lenses with different characteristics. The main lenses are massive to sub-massive fine-grained pyrite with overprinting, interstitial and fracture-related chalcopyrite and low-iron sphalerite, with gold and silver (rarely galena) in pyrite or as banded sulphide layers and occasional high-grade stringer zones.

Figure 7.7 Diamond Drill Section 53,800N through Southern Lens at Terakimti



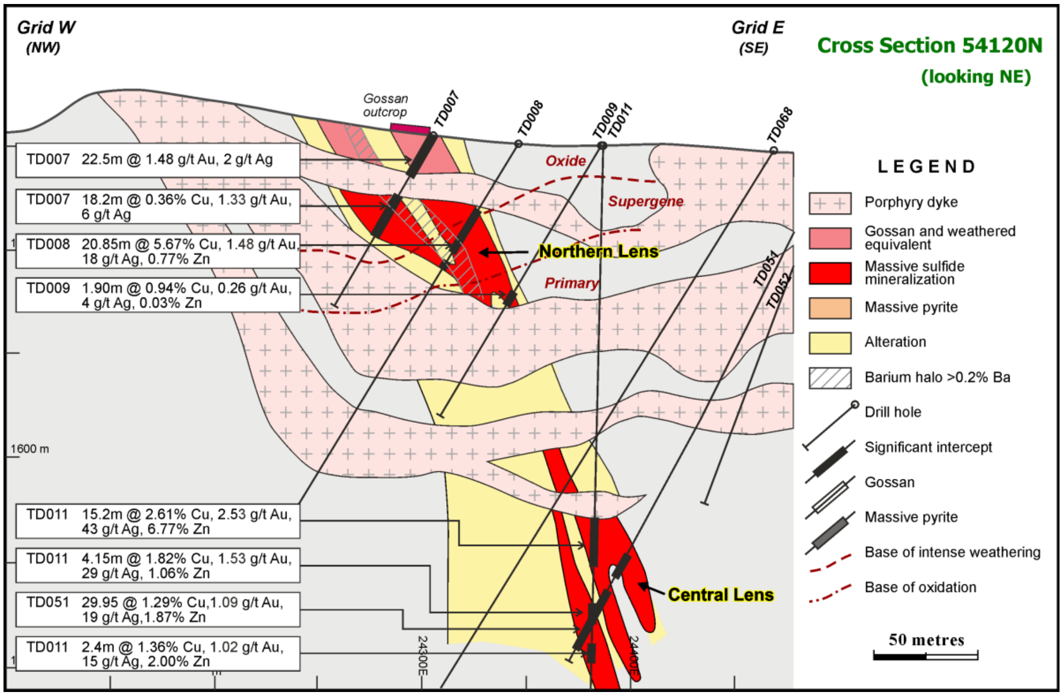
Source: Groves et al. (2013)

Figure 7.8 Diamond Drill Section 53,960N through Southern and Central Lenses at Terakimti



Source: Groves et al. (2013)

Figure 7.9 Diamond Drill Section 54,120N through Central and Northern Lens at Terakimti



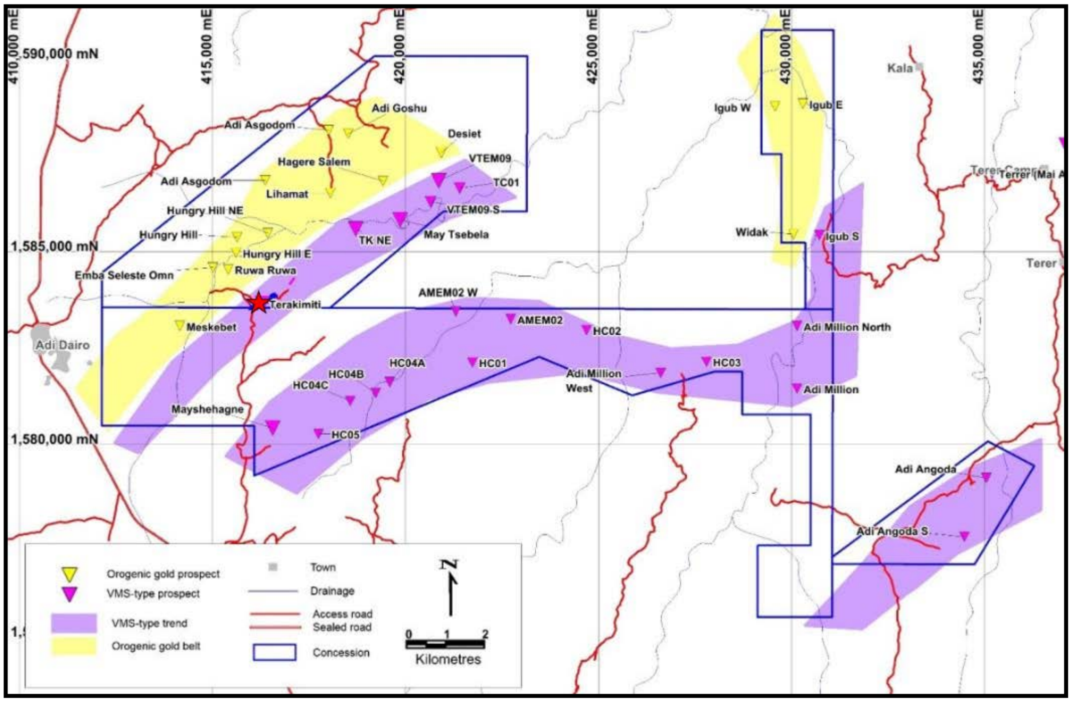
Source: Groves et al. (2013)

7.2 MINERALIZATION

Exploration efforts on the Harvest Property are currently targeting two deposit types, these being VHMS and orogenic lode-gold mineralization. A spatial relationship between these deposit types is noted on the property and may be related to reactivation of hydrothermal pathways or redistribution of deposited mineralization during orogenesis. Further discussion of orogenic gold style of mineralization is not included in this report. Figure 7.10 highlights the distribution of the current targets on the Property in context of regional deposit style trends.

To date the company has defined one deposit, the polymetallic (copper-zinc-gold-silver-lead) VHMS Terakimti deposit, by completing a Mineral Resource Estimate in accordance with the CIM Definition Standards for Mineral Resources and Reserves (2014). The deposit is comprised of both near surface mineralized gossan (oxide), and deeper sulphide bearing transitionary (supergene) and sulphide hosted zones. This deposit remains the near-term development goal for the Property. Mining engineering work for the near surface oxide resources are discussed in further detail in Section 16.0 of this report. The deposit is located along a favourable VHMS belt at the current boundary between the Terakimti and Hamlo concessions (shown as red star in Figure 7.10).

Figure 7.10 Harvest Mining Exploration Target Locations by Deposit Type

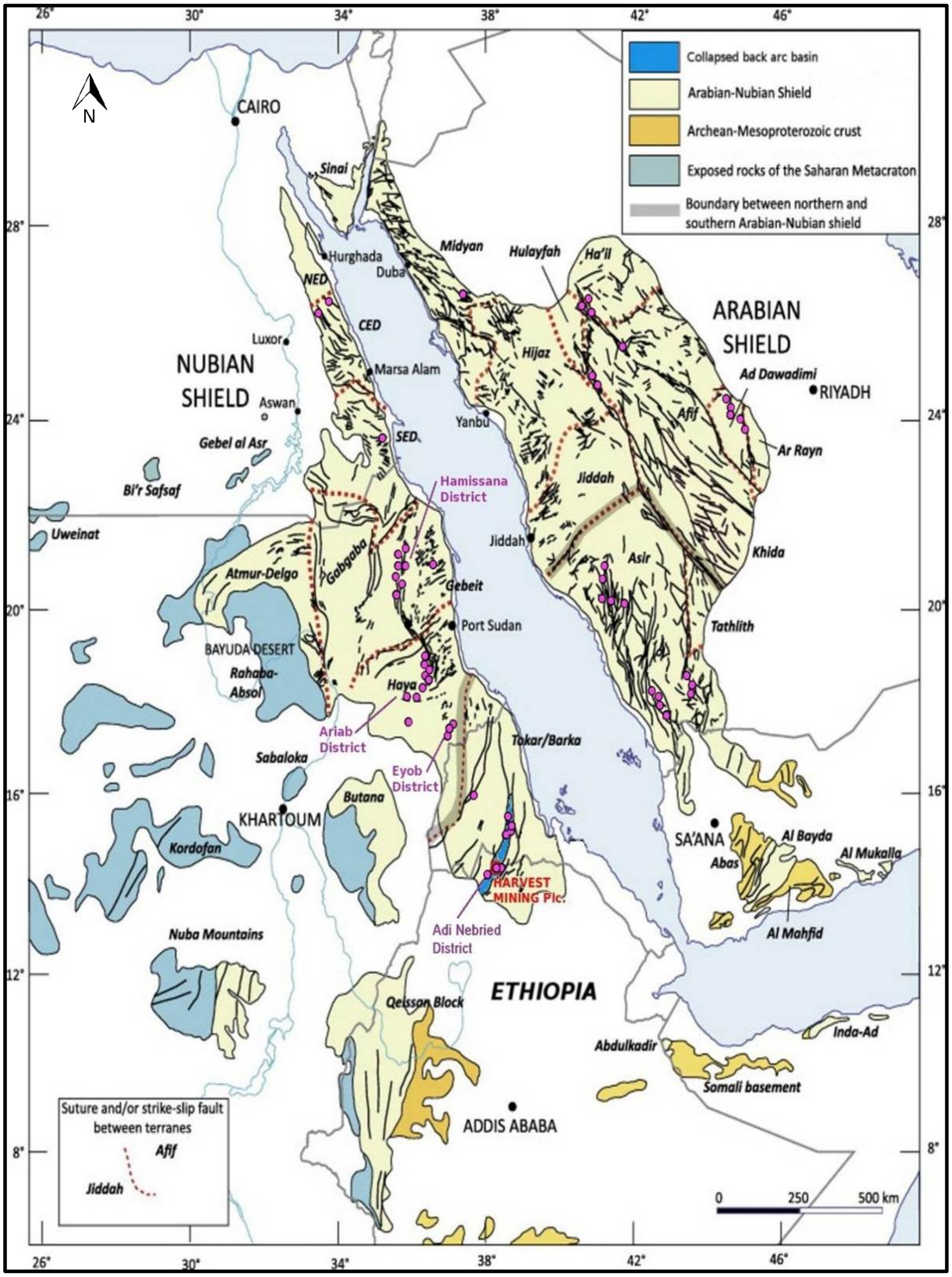


8.0 DEPOSIT TYPES

8.1.1 REGIONAL VHMS DEPOSITS STYLE

The ANS hosts numerous VHMS deposits (Figure 8.1). These deposits in the district have been variably described as Kuroko type (Chewaka and DeWit 1981) and as bi-modal mafic type following the classification of Hannington (2009), (Sunridge Gold Corp. 2006a, 2006b). Generally, VHMS deposits contain footwall mineralization consisting of quartz-chalcopyrite stringers (stockwork), overlain by primary bedded (stratiform) sulphides composed of pyrite, chalcopyrite, ± sphalerite, ± galena, ± barite, ± tetrahedrite/tennanite. In some deposits the stratiform massive sulphide lens makes up the entire economic deposit, whereas in other deposits large quantities of ore are also mined from the stockwork zone (Figure 8.2). The stratiform sulphides are typically overlain, or grade into, an iron-rich silica facies that is usually manifested as a BIF. Surficial weathering results in the primary sulphides forming secondary, supergene minerals such as chalcocite, covellite, digenite, and bornite. The surface manifestation of a VHMS system in this geological setting is the total leaching of metals with the exception of silica and iron to produce a hematite-goethite gossan. In an undisturbed horizontal system, vertical thicknesses can range up to 50 m and maximum strike lengths can be up to 1,500 m. Barite is commonly associated with mineralization.

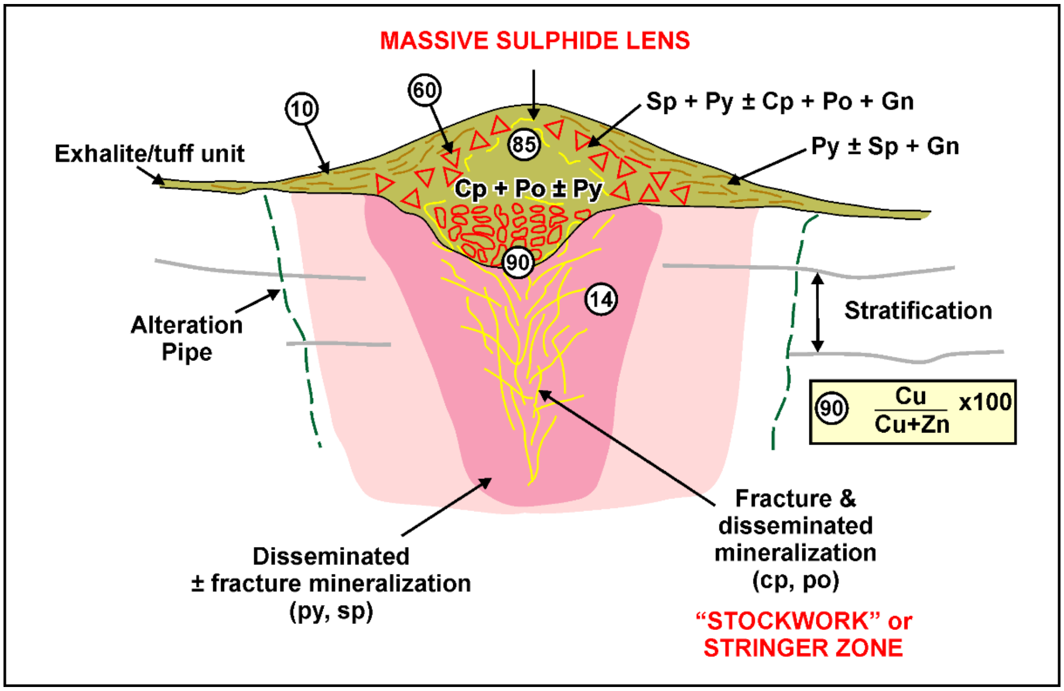
Figure 8.1 Known VHMS Deposits within the Arabian Nubian Shield



Source: Diagram Modified from Johnson et al. (2011)

Notes: Location of the Project is located within the Adi Nebried District, which hosts the known Ethiopia VHMS deposits of Terakimti, Rahwa and Terer

Figure 8.2 Schematic Model for Active VHMS Mineralization Showing Principal Alteration and Mineralization Types



Notes: Idealized VHMS deposit showing a stratiform lens of massive sulphide overlying a discordant stringer sulphide zone within an envelope of altered rock (alteration pipe). Base metal zonation indicated by numbers in circles with the highest numbers being Cu-rich and the lower numbers more zinc-rich (Py = pyrite, Cp = chalcopyrite, Po = pyrrhotite, Sp = sphalerite, and Gn = galena.

Source: After Gibson (2005)

The geological setting of the Terakimti deposit is considered analogous with that of the precious and base metal VHMS deposits of the Bisha Project of Nevsun Resources Ltd. (Eritrea), and particularly Debarwa and Emba Derho of Sunridge Gold Corp. (Eritrea), both located in the Nafka Terrane, and interpreted to be directly along strike from the Terakimti deposit and the Mayshehagne and VTEM09 targets located approximately 120 km to the south-southwest.

9.0 EXPLORATION

Exploration conducted on the Harvest Property, up to and including 2013, is summarized in Section 6.0. The following section outlines recent work on the Terakimti deposit. A full description of regional exploration (including soil geochemical surveys and geophysical surveys) is included in Archibald et al. (2014) and has been captured in detail within the Harvest Mining assessment reports (Gardoll et al. 2012; 2013; 2014). Complete descriptions of sampling procedures, for exploration samples are documented in Archibald et al. (2014).

Recent exploration efforts on the Property were focussed on advancing the Terakimti target to resource development stage through high-level studies on specific gravity, structure, metallurgy work, substantial trenching to test the oxide resource and infill reverse circulation drilling to 20 m x 20 m spacings to define the near surface oxide gold resource. This drilling is summarized in Section 10.0.

9.1 ROAD MAINTENANCE AND UPGRADES

Routine maintenance was conducted on the Terakimti road from the highway, with 10 km of road repaired after wet seasons during exploration years. At the Terakimti VHMS deposit dozing of drill lines was required (Figure 9.1), where steep terrain dictated the requirement to make a flat platform for drilling and access. In total, nine lines were dozed for a total of 650 m.

Figure 9.1 Reverse Circulation Drill Line Being Prepared Using the Dozer at the Terakimti Deposit

9.2 CAMP ESTABLISHMENT

A small camp was established at Terakimti, including a storage facility, office, dining and cooking sheds, a fuel depot (holding up to 3,000 L), a reverse circulation sample farm, a driller's storage shed and tented accommodation, as shown in Figure 9.2 and Figure 9.3. The camp provided accommodation for up to 25 persons.

Figure 9.2 Terakimti Camp and Fenced Sample Farm (in background)

Figure 9.3 **Cleared and Fenced Sample Farm, where all Coarse Samples from the Reverse Circulation Drilling were Stored and Prepared for Transport**

9.3 REPORT WRITING AND PRESENTATIONS

Reporting on project activities was on-going throughout the year with monthly internal reports, short geological summary reports, prospect summary reports and the Harvest annual and renewal reports. In addition, time was spent on detailed multi-element analysis and structural analysis for Terakimti.

As part of the transfer of knowledge, Harvest Mining presented several talks in Addis Ababa on July 14, 2014, at the EAM Exploration conference. Included in these presentations was *Terakimti Cu-Zn-Au-Ag-Pb VHMS Deposits from Discovery to Resource*, by Stephen Gardoll.

9.4 PETROGRAPHIC STUDIES

Two petrographic studies were performed on rocks collected from the Terakimti deposit. Three samples were studied by Vancouver Petrographic in 2011 and 18 samples by Dr. Craig Leitch in 2011.

9.4.1 VANCOUVER PETROGRAPHICS LTD. (2011)

Three samples from the Terakimti VHMS prospect were submitted for petrographic analysis by Dr F. Colombo, P.Geo., at Vancouver Petrographics Ltd. in June 2011. The samples were made into polished thin sections and analysed under transmitted and reflected light. The study showed the samples were comprised sulphide-rich, pyrite-dominated units from the replacement zones. The interstices of the fine-grained pyrite aggregates were filled with copper-rich minerals (chalcopyrite, covellite), with chalcopyrite being replaced by covellite in one of the samples, and sphalerite occurring as an accessory mineral. No gold was observed in the submitted samples.

9.4.2 CRAIG LEITCH (2011)

Eighteen samples were submitted to Dr Craig Leitch, P.Eng, in December 2011 for petrographic analysis of polished thin sections. The samples were comprised of massive/semi-massive sulphides (12), felsic/intermediate meta-volcanics (4), intermediate/mafic meta-volcanics (2), and were all collected from drillhole TD004 with the exception of one oxide sample collected from drillhole TD008. The massive sulphides are interpreted as being syngenetic within a VHMS setting and some evidence existed for stockwork veining. Mineralogy was relatively simple, with the sulphides composed of pyrite, chalcopyrite, sphalerite, galena and arsenopyrite, and the presence of covellite after chalcopyrite. A sulphosalt, possibly tetrahedrite, was present in most of the sulphide-bearing samples. No gold or electrum was observed during petrographic analysis.

10.0 DRILLING

Drilling was undertaken to define polymetallic mineralization identified by earlier prospecting, mapping, trenching and ground geophysics. The first drilling at the Terakimti deposit was in late 2009 and early 2010 where 12 holes drilled a total of 1,572.75 m. Drilling performed by Harvest Mining ran from August 2011 to June 2012, totalling 68 diamond holes drilled for a total length of 15,007.51 m. All core holes were collared using HW diameter rods before utilizing HQ diameter rods. Where necessary due to depth or ground conditions the rod diameter was further reduced to NQ. Despite several attempts to keep the holes open with polyvinyl chloride (PVC) piping (for future downhole geophysics), only five holes were successfully lined.

In 2014, a reverse circulation drilling campaign commenced at Terakimti to further define the oxide gold resource. Drilling was planned on a 20 m x 20 m grid. The program was completed in 2015 after 127 holes were drilled totalling 6,189.5 m. Additionally in late 2015 another six diamond drillholes were drilled for metallurgical test work.

A summary of the drilling at Terakimti is tabulated in Table 10.1.

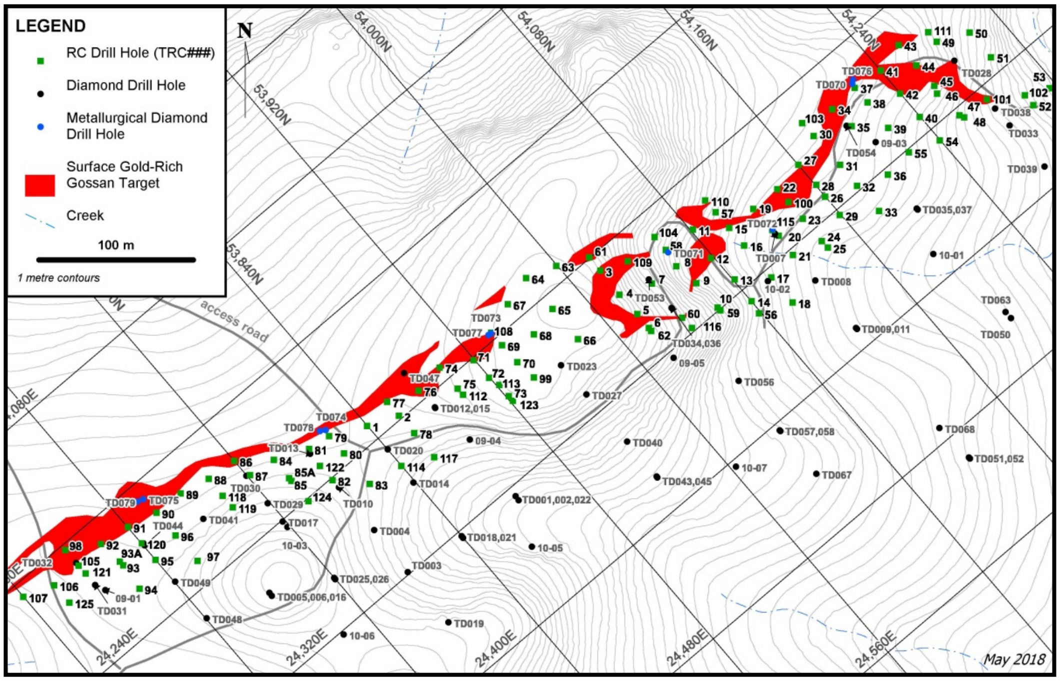
Table 10.1 Summary of Drilling at the Terakimti Deposit

		RC		Diamond	
Year	Prospect	Holes	Metres	Holes	Metres
2009	Terakimti	-	-	3	413.2
2010	Terakimti	-	-	9	1,159.55
2011	Terakimti	-	-	51	10,398.26
2012	Terakimti	-	-	17	4,609.25
2013	Terakimti	-	-	1	339.85
2014	Terakimti	18	828.0	-	-
2015	Terakimti	109	5,361.5	6*	270.73
2017	Terakimti			4*	174
Total	-	**127**	**6,189.5**	**87**	**17,190.84**

Note: *metallurgical holes

A plan of all of the drillholes completed to date is presented in Figure 10.1.

Figure 10.1 Terakimti Drillhole Plan Showing Collar Locations



10.1 DIAMOND DRILLING

10.1.1 DRILL CONTRACTORS

Diamond drilling at the Harvest Property was conducted by three contractors: Jintai Drilling (Shanghai, China) from December 12, 2009 to June 18, 2010 and also August 20, 2011 to March 29, 2012; Nubian Drilling Limited (Nubian Drilling) (Geneva, Switzerland) from October 30, 2011 to May 3, 2012; and Kluane Drilling (Yukon, Canada) for one hole in December 2013, six holes from November 2, 2015 to November 6, 2015, and four holes from May 20, 2017 to May 30, 2017.

In 2011, drilling was initiated at Terakimti on August 7 with the use of one electric powered Jintai Drilling rig (model HXY-42). A second Jintai Drilling rig started drilling on October 8th, working on a double shift basis. The Jintai Drilling contract was completed in January 2012. Jintai Drilling used a variety of different length rods and subs that were non-standard and calculated down-hole depths by measuring each rod (minus the thread), and as a result the core was recovered in non-standard intervals.

Nubian Drilling began drilling on October 30, 2011, with one rig (Longyear LF 90D) and completed drilling on May 3, 2012. Nubian Drilling utilized standard equipment including 3 m rods and 3 m and 1.5 m core barrels.

Kluane Drilling, using a man portable diamond drill rig, completed one deep hole in December 2013, using NTW diameter drill rods. In November of 2015, Kluane Drilling drilled six metallurgical holes for 271 m, with HTW triple tube diameter rod equipment. In

May 2017, Kluane Drilling completed 274 m of metallurgical drilling with HTW rod equipment.

10.1.2 DRILL SURVEYING

Drilling collar locations post 2010 were accurately surveyed in using a Trimble® differential global positioning system (DGPS), where drilling pads were cleared and built up.

Fast static DGPS equipment was hired in May 2015 to locate trenches and drillholes at the Terakimti deposit. Using sub-centimetre accuracy, the DGPS equipment allows very accurate plotting of drillhole collars, which assists in the ultimate delineation of the resource model.

Most diamond drillholes were completed on a 40 m to 80 m grid spacing and were drilled at an azimuth of 270°. This proposed drill spacing was chosen to facilitate resource estimation, and to allow infill drilling to maintain an even spaced grid. Drilling down dip was planned to intersect the mineralization at 40 m down dip intervals. Pierce points for the drilling targets were perpendicular to the interpreted strike of targeted mineralization to try and ensure true thicknesses of target rocks were intercepted.

Downhole survey measurements were taken at various depths, ranging from 6 to 50 m intervals down each drill hole using a Reflex EZ-Shot orientation instrument. A final reading was taken approximately 6 m from the bottom.

During exploration, Harvest Mining and their drilling contractors conducted the drilling program according to industry best practices. Drillhole collar coordinates were surveyed and at completion the holes were capped with concrete monuments. These coordinates were surveyed prior to drilling and again after drilling, by a qualified Harvest Mining surveyor using a DGPS Epoch 25 with a measurement accuracy of ±1 cm. Jintai Drilling used a Tropari Gimble for down-hole survey for all holes in 2009 and 2010 and for drillhole TD001 to 11 and TD013. All contractors used a multi-shot Reflex EZ-Shot orientation instrument for downhole surveys for all other drillholes including TD012 and TD014 to 68. Azimuth and dip information was recorded downhole at roughly 30 m intervals, and a final reading approximately 6 m from the bottom of every hole.

Jintai Drilling utilized a down hole spear for core orientation purposes (in an effort to facilitate structural studies on the core), and Nubian Drilling used a state of the art Acer Reflex tool. However, the poor training of the operators resulted in the failure to calibrate of one of the Acer reflex tools effectively rendered much of the core orientation data for structural work useless.

For the metallurgical drill holes, Reflex EZ-Shot downhole surveys were recovered every 10 m.

Core orientation information was initially taken at approximately 30 m intervals, and reduced to 6 m intervals in areas of mineralization to determine the orientation of mineralization and structures (e.g., foliation) within the rock. Core orientation measurements were collected every 6 m during drilling in 2011 and 2012 to increase the confidence and reliability of the measurements.

10.1.3 Core Recovery

Drill core recoveries for the project were acceptable, and the samples collected were representative of the observed mineralization. Determining the exact true thickness from individual drill holes is difficult, but generally based on the consistent drillhole orientation, the intercepts represent 60 to 70% of the true thickness. Diamond drill core recoveries were lesser in the oxide metallurgical drillholes and were more closely monitored and measured.

10.1.4 Drill Core Logging and Sampling Procedures

Geotechnical logging was performed on the core samples at the drill site to avoid unnecessary breaks that might occur during transport and therefore affect the apparent rock quality designation (RQD) of the core. Core orientation marks were taken every 6 m using a spear and the core was oriented and marked. At the end of every shift the core was transported by pick-up truck to Harvest Mining's Guna core logging and storage facility in Shire. The core logging process involved an initial cleaning of the core and checking of the core tags, and mark-ups on the individual boxes. Any discrepancies noted were addressed with the driller who was responsible for the core. At the camp all core was photographed prior to being logged by the geologist with an emphasis on structure, lithology, alteration and mineralization. Completed drill core logs were reviewed by a senior geologist to check consistency in logging.

Sample intervals were marked-up by the geologist logging the core and were based on sample intervals of either 0.7 m for mineralized core or 1 m for non-mineralized core. Sample intervals did not cross geological contacts. The physical sampling of the core was done with a diamond blade core cutting saw. The core was sawn in half along the line marked by the geologist to ensure a representative sample was taken.

Cloth sample bags were pre-numbered by a technician and the split core was moved to the sampling area for final preparation. Individual samples were then bagged and the ticket book filled out with tickets added to the sample and to the core box. The "side" of the split core was chosen systematically by reference to the orientation line and foliation in order to prevent any bias in sample selection. The samples from each drill hole were laid out in succession within the sampling area and loosely tied before being taken to the specific gravity station, where the specific gravity is determined and recorded by emersion of the sample in water. The samples are patted dry and then re-bagged, whereupon they are securely tied using the draw string and a final weight of the sample, and bag, is recorded for export purposes.

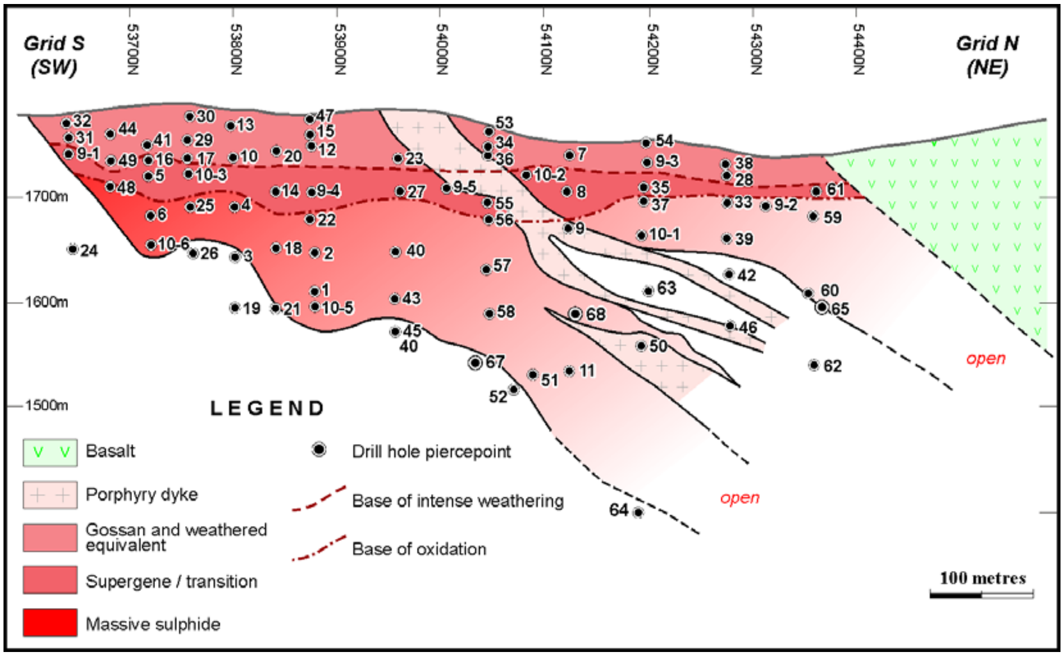
10.1.5 Significant Drill Core Mineralization Intersections

Selected significant mineralized intercepts at Terakimti are presented in Table 10.2. A general description of the mineralization encountered is presented in previous sections of this report, but in summary it consists of four lenses of bedded polymetallic (copper-gold-silver-zinc-lead) massive sulphide over a strike length of 800 m. These lenses plunge to the northeast and the structure remains open down-plunge (Figure 10.2).

Table 10.2 Terakimiti Diamond Drillhole Results

Hole ID		From (m)	To (m)	Interval (m)	Copper %	Gold g/t	Silver g/t	Zinc %	Local Azimuth	Dip
10HTD003		45.60	97.70	52.10	4.10	1.55	26	0.13	261	-76
	Incl	54.20	93.70	39.50	5.35	1.74	21	0.15		
TD004		57.45	131.30	73.85	3.77	1.31	14	0.72	270	-60
	incl	78.65	115.10	36.45	6.01	1.70	19	1.31		
TD008		38.75	59.60	20.85	5.67	1.49	18	0.77	270	-60
	incl	40.70	54.45	13.75	7.49	2.07	24	1.09		
TD011		181.75	196.95	15.20	2.61	2.53	43	6.79	270	-90
TD014		57.45	94.45	37.00	3.53	1.21	26	1.32	270	-65
	incl	58.20	80.50	22.30	5.35	1.65	40	2.08		
TD016		80.40	98.15	17.75	2.98	1.61	20	0.03	270	-45
TD018		124.05	154.80	30.75	2.55	0.99	9	1.52	270	-58
	incl	135.63	154.80	19.17	3.68	1.40	13	2.37		
TD022		87.35	116.30	28.95	2.99	0.83	23	3.56	270	-51
TD025		93.85	125.60	31.75	1.84	0.86	17	7.03	270	-58
	incl	110.50	122.40	11.90	3.08	1.11	20	9.54		
TD029*		36.45	45.25	8.80	0.01	9.19	78	0.00	270	-60
	incl	38.30	41.90	3.60	0.01	21.88	168	0.00		
TD034*		0.00	29.00	29.00	0.14	3.40	11	0.08	270	-61
	incl	20.20	29.00	8.80	0.29	8.77	34	0.12		
TD051		233.35	263.30	29.95	1.29	1.09	19	1.87	270	-67
TD053*		10.88	17.00	6.12	0.19	27.17	13	0.08	270	-60

Figure 10.2 Diamond Drilling Terakimti Long Section



Source: Harvest Mining, 2014

10.2 REVERSE CIRCULATION DRILLING

Reverse circulation drilling was completed with a track–mounted KL900 RC unit. Holes were completed to a maximum depth of approximately 200 m with a minimum dip angle of –55°. The drill rig was fitted with a cyclone and sample splitting was conducted separately. Drilling utilized a 4.5 inch rod string with a 5.25 inch Sandvik RE40 hammer. The Compressor and Auxiliary were attached to a separate track mounted vehicle, with 1170 CFM and 435 PSI capacity on the Compressor, with a booster with an additional 750 CFM and 350 PSI was available. Stainless steel rods (6 m) were utilized behind the hammer to allow down hole surveying within the rod string. A standard multi-shot down-hole survey tool was used.

A total of 127 reverse circulation drillholes (6,189.5 m) have been completed at Terakimti to define the oxide gold resource. Sample photos from this drill campaign are shown in Figure 10.3.Summary results are shown in Table 10.3 and are incorporated into the Terakimti oxide Mineral Resource estimate (Section 14 and Table 14.11).

Figure 10.3 Reverse Circulation Drilling and Terakimti Copper-Gold-Silver-Zinc-Lead Deposit

RC Drilling showing Geosearch RC drill rig in action

Splitting the RC drill sample for sampling

Logging the RC drill samples

Hand held XRF analysis of the RC drill samples

10.2.1 REVERSE CIRCULATION LOGGING AND SAMPLING PROCEDURES

All reverse circulation samples are collected on 1 m intervals in large green plastic bags. Sampling occurs at the drill site and samples are collected from within mineralised zones (gossan, massive sulphides, and stringer zones) at 1 m intervals. All 1 m samples are split to ensure homogeneity and repeatability as they may be required to be re-assayed at a later date. Samples collected from outside of mineralised zones are composited into 3 or 4 m samples and placed in bags. These composite samples are then potentially sub-sampled with a spear type sampler. Following spear-sampling, representative samples were sieved prior to washing chips for separate logging. The chips were placed into pre-numbered plastic chip trays with the Hole ID and intervals labelled, after completion of logging.If these samples return a high result the original 1 m samples are split and assayed.

Table 10.3 Reverse Circulation Sample Results

Hole ID		From (m)	To (m)	Interval (m)*	Copper (%)	Gold (g/t)	Silver (g/t)	Zinc (%)	Local Azimuth (°)	Dip (°)	Prospect
TRC003		2.00	4.00	2.00	0.03	0.856	0.0	0.01	269	-66	Terakimti Oxide
TRC004		0.00	14.00	14.00	0.15	7.626	4.8	0.03	272	-67	Terakimti Oxide
TRC005		5.00	13.00	8.00	0.19	3.227	4.0	0.05	272	-66	Terakimti Oxide
	including	6.00	9.00	3.00	0.32	7.258	9.2	0.06			
TRC006		0.00	8.00	8.00	0.20	8.660	4.1	0.08	277	-66	Terakimti Oxide
	including	3.00	7.00	4.00	0.32	16.126	7.0	0.11			
TRC007		12.00	18.00	6.00	0.23	11.211	8.5	0.08	270	-63	Terakimti Oxide
TRC008		16.00	29.00	13.00	0.07	1.709	4.4	0.05	270	-67	Terakimti Oxide
TRC009		13.00	23.00	10.00	0.11	4.943	54.0	0.06	274	-67	Terakimti Oxide
	including	18.00	23.00	5.00	0.16	9.370	107.2	0.09			Ag-Rich Oxide
TRC010		13.00	28.00	15.00	0.15	4.124	5.3	0.10	272	-66	Terakimti Oxide
	including	13.00	15.00	2.00	0.08	11.845	2.5	0.05			
	including	20.00	25.00	5.00	0.16	6.680	10.9	0.13			
TRC011	No Significant Results - Not Drilled to Sufficient Depth								272	-60	Terakimti Oxide
TRC012		11.00	16.00	5.00	0.12	0.611	2.8	0.06	272	-63	Terakimti Oxide
		30.00	34.00	4.00	0.01	2.704	4.3	0.00			
TRC013		0.00	2.00	2.00	0.08	3.194	0.7	0.09	272	-64	Terakimti Oxide
		28.00	43.00	15.00	0.03	12.364	38.2	0.02			
	including	33.00	43.00	10.00	0.01	18.218	51.2	0.01			
	and including	33.00	37.00	4.00	0.01	34.245	101.4	0.01			Ag-Rich Oxide
TRC014		30.00	38.00	8.00	0.04	4.072	28.7	0.04	270	-62	Terakimti Oxide
	including	30.00	34.00	4.00	0.04	7.695	52.9	0.06			
		41.00	43.00	2.00	0.01	0.344	53.2	0.01			Ag-Rich Oxide
TRC015		15.00	17.00	2.00	0.05	5.863	4.3	0.02	273	-68	Terakimti Oxide

table continues…

Hole ID		From (m)	To (m)	Interval (m)*	Copper (%)	Gold (g/t)	Silver (g/t)	Zinc (%)	Local Azimuth (°)	Dip (°)	Prospect
TRC016		13.00	26.00	13.00	0.06	4.653	5.6	0.04	274	-67	Terakimti Oxide
	including	19.00	23.00	4.00	0.08	13.543	12.9	0.06			
		35.00	45.00	10.00	0.99	1.296	2.5	0.00			Terakimti Supergene
	including	41.00	44.00	3.00	2.07	2.281	5.0	0.01			
TRC017		29.00	33.00	4.00	0.18	9.639	1151.9	0.07	275	-65	Terakimti Oxide
	including	30.00	33.00	3.00	0.17	12.748	1533.1	0.05			Ag-Rich Oxide
		33.00	46.00	13.00	2.57	1.559	292.6	0.02			Terakimti Supergene
TRC018		42.00	75.00	33.00	1.42	4.452	6.3	0.38	268	-65	Terakimti Supergene
	including	42.00	47.00	5.00	2.53	4.492	15.7	0.97			
	including	65.00	75.00	10.00	1.96	10.643	6.8	0.17			
TRC019		8.00	17.00	9.00	0.03	5.943	1.6	0.03	270	-64	Terakimti Oxide
		27.00	41.00	14.00	0.69	0.063	0.0	0.06			Terakimti Supergene
TRC020		31.00	35.00	4.00	0.03	2.495	2.7	0.01	270	-63	Terakimti Oxide
		39.00	46.00	7.00	1.95	0.026	0.0	0.09			Terakimti Supergene
		48.00	58.00	10.00	0.63	1.002	1.3	0.02			Terakimti Sulfide
TRC021		24.00	39.00	15.00	0.03	7.339	324.5	0.01	275	-63	Terakimti Oxide
	including	31.00	39.00	8.00	0.02	12.026	604.6	0.01			Ag-Rich Oxide
		40.00	49.00	9.00	1.03	1.079	12.1	0.07			Terakimti Supergene
		55.00	61.00	6.00***	0.17	0.550	790.0	0.08			Ag-Rich Oxide
TRC022	No Significant Results								276	-67	
TRC023		20.00	37.00	17.00	0.03	1.757	18.6	0.01	274	-61	Terakimti Oxide
		37.00	49.00	12.00	0.65	1.598	7.6	0.01			Terakimti Supergene/Sulfide
TRC024		24.00	36.00	12.00	0.04	2.252	45.7	0.02	270	-65	Terakimti Oxide
	including	29.00	34.00	5.00	0.03	2.278	83.9	0.01			Ag-Rich Oxide
		36.00	48.00	12.00	6.74	2.428	20.5	1.91			Terakimti Supergene/Transition
TRC025		28.00	32.00	4.00	0.03	0.537	4.9	0.04	269	-69	Terakimti Oxide
		32.00	79.00	47.00	2.09	0.781	23.1	0.34			Terakimti Supergene/Transition/Sulfide

table continues...

Hole ID		From (m)	To (m)	Interval (m)*	Copper (%)	Gold (g/t)	Silver (g/t)	Zinc (%)	Local Azimuth (°)	Dip (°)	Prospect
TRC026		10.00	24.00	14.00	0.08	7.253	8.7	0.10	276	-63	Terakimti Oxide
		32.00	43.00	11.00	0.70	0.933	6.7	0.15			Terakimti Sulfide
TRC027		28.00	31.00	3.00	1.36	0.975	14.0	0.09	272	-63	Terakimti Sulfide
TRC028		0.00	10.00	10.00	0.09	11.302	4.9	0.09	270	-62	Terakimti Oxide
		28.00	34.00	6.00	0.05	0.478	70.8	0.01			
	including	28.00	31.00	3.00	0.03	0.722	113.7	0.01			Ag-Rich Oxide
TRC029		25.00	29.00	4.00	0.02	0.913	3.0	0.01	277	-64	Terakimti Oxide
		35.00	40.00	5.00**	0.13	2.206	100.5	0.00			
	including	36.00	38.00	2.00	0.30	4.756	237.6	0.00			Ag-Rich Oxide
		41.00	61.00	20.00**	1.79	2.280	7.9	0.32			Terakimti Supergene
TRC030		17.00	26.00	9.00	0.10	3.316	9.0	0.01	270	-65	Terakimti Oxide
TRC031		6.00	18.00	12.00	0.09	3.586	10.2	0.12	273	-65	Terakimti Oxide
		32.00	43.00	11.00	1.20	1.597	7.9	0.72			Terakimti Transition/Sulfide
TRC032		27.00	33.00	6.00	0.03	5.827	28.4	0.02	270	-65	Terakimti Oxide
	including	30.00	33.00	3.00	0.05	3.675	48.6	0.03			Ag-Rich Oxide
		50.00	53.00	3.00	0.43	1.023	3.6	0.01			Terakimti Sulfide
TRC033		29.00	93.00	64.00***	1.95	2.409	9.9	1.12	270	-65	Terakimti Supergene/Transition/Sulfide
	including	39.00	71.00	32.00	3.08	4.398	15.8	1.81			
TRC034		2.00	4.00	2.00	0.18	1.878	0.4	0.12	272	-62	Terakimti Oxide
		10.00	17.00	7.00	0.11	1.325	2.1	0.05			
TRC035		0.00	7.00	7.00	0.16	6.085	3.3	0.09	276	-65	Terakimti Oxide
		30.00	38.00	8.00**	3.16	2.414	32.4	2.57			Terakimti Transition
TRC036		28.00	37.00	9.00	3.71	3.079	26.8	3.29	276	-64	Terakimti Sulfide
		55.00	67.00	12.00	0.37	0.433	3.8	0.15			
TRC037		0.00	11.00	11.00	0.11	1.561	2.9	0.07	273	-63	Terakimti Oxide
TRC038		9.00	29.00	20.00	0.04	1.989	18.4	0.03	276	-65	Terakimti Oxide
	including	20.00	27.00	7.00	0.01	1.887	36.1	0.00			Ag-Rich Oxide
		29.00	31.00	2.00	0.96	1.171	17.0	0.28			Terakimti Sulfide

table continues...

Hole ID		From (m)	To (m)	Interval (m)*	Copper (%)	Gold (g/t)	Silver (g/t)	Zinc (%)	Local Azimuth (°)	Dip (°)	Prospect
TRC039		15.00	25.00	10.00	0.02	3.047	5.3	0.03	276	-65	Terakimti Oxide
		40.00	51.00	11.00	3.01	3.132	33.2	2.53			Terakimti Transition
TRC040		3.00	22.00	19.00	0.04	1.403	6.1	0.03	272	-60	Terakimti Oxide
		22.00	40.00	18.00	1.45	0.596	16.6	0.31			Terakimti Supergene/Transition/Sulfide
		48.00	54.00	6.00	0.90	0.975	10.1	0.27			Terakimti Sulfide
TRC041		0.00	7.00	7.00	0.08	0.958	0.6	0.05	266	-62	Terakimti Oxide
TRC042		6.00	16.00	10.00	0.09	1.083	2.3	0.08	269	-63	Terakimti Oxide
		19.00	27.00	8.00**	0.04	1.575	11.4	0.02			Terakimti Oxide
	including	25.00	27.00	2.00	0.08	0.332	38.6	0.01			Ag-Rich Oxide
TRC043		0.00	7.00	7.00	0.04	2.587	0.4	0.05	268	-65	Terakimti Oxide
TRC044		9.00	20.00	11.00	0.04	0.890	3.7	0.03	270	-68	Terakimti Oxide
TRC045		0.00	25.00	25.00	0.05	1.696	30.4	0.05	270	-73	Terakimti Oxide
	including	20.00	25.00	5.00	0.02	1.201	133.9	0.02			Ag-Rich Oxide
TRC046		8.00	26.00	18.00	0.05	1.342	3.8	0.04	1	-60	Terakimti Oxide
TRC047		18.00	25.00	7.00^^	0.04	0.152	3.5	2.85	273	-66	
		34.00	40.00	6.00^^	0.26	0.198	5.9	1.56			
		48.00	57.00	9.00	0.65	1.446	21.2	1.26			Terakimti Transition
TRC048		39.00	49.00	10.00	0.19	0.201	5.4	3.53	343	-66	Terakimti Sulfide
TRC049		13.00	22.00	9.00	0.01	1.466	2.9	0.02	269	-62	Terakimti Oxide
TRC050	No Significant Results								273	-59	Terakimti
TRC051	No Significant Results #								272	-60	Terakimti
TRC052		26.00	66.00	40.00	0.48	0.383	15.1	3.67	274	-61	Terakimti Sulfide
TRC053	No Significant Results #								271	-59	Terakimti
TRC054	No Significant Results #								270	-65	Terakimti
TRC055		34.00	43.00	9.00	2.21	1.858	19.9	1.11	276	-66	Terakimti Transition
		43.00	53.00	10.00	0.81	0.712	5.0	0.24			Terakimti Sulfide
		53.00	69.00	16.00	0.46	0.440	2.0	0.30			
		69.00	81.00	12.00	0.29	0.323	1.8	0.12			

table continues...

Hole ID		From (m)	To (m)	Interval (m)*	Copper (%)	Gold (g/t)	Silver (g/t)	Zinc (%)	Local Azimuth (°)	Dip (°)	Prospect
TRC056		39.00	51.00	12.00	3.30	4.434	19.5	1.26	270	-62	Terakimti Transition
TRC057		0.00	22.00	22.00	0.05	1.514	0.3	0.04	270	-68	Terakimti Oxide
TRC058		5.00	38.00	33.00	0.09	7.198	1.3	0.08	270	-67	Terakimti Oxide
TRC059	No Significant Results								141	-89	Terakimti
TRC060		1.00	5.00	4.00	0.15	6.675	6.8	0.06	272	-83	Terakimti Oxide
		14.00	21.00	7.00	0.14	11.667	38.1	0.06			Terakimti Oxide
	including	16.00	19.00	3.00	0.10	14.996	77.3	0.06			Terakimti Oxide
		25.00	29.00	4.00	0.14	4.709	4.7	0.14			Ag-Rich Oxide
		32.00	38.00	6.00	0.06	2.983	14.3	0.02			Terakimti Oxide
TRC061		0.00	7.00	7.00	0.07	5.174	0.9	0.06	268	-67	Terakimti Oxide
TRC062	No Significant Results								313	-89	Terakimti
TRC063	No Significant Results								268	-65	Terakimti
TRC064	No Significant Results								270	-61	Terakimti
TRC065	No Significant Results								273	-67	Terakimti
TRC066		43.00	46.00	3.00**	0.00	0.893	19.6	0.00	275	-68	Terakimti Oxide
TRC067		6.00	11.00	5.00	0.04	0.793	0.3	0.03	273	-62	Terakimti Oxide
TRC068		22.00	27.00	5.00	0.03	0.997	7.1	0.02	274	-62	Terakimti Oxide
TRC069		7.00	17.00	10.00	0.07	5.366	2.8	0.08	274	-65	Terakimti Oxide
		28.00	35.00	7.00	0.02	0.662	3.1	0.02			
TRC070		22.00	32.00	10.00	0.03	7.688	11.0	0.03	273	-65	Terakimti Oxide
		37.00	47.00	10.00	0.01	1.436	7.5	0.01			
TRC071		0.00	8.00	8.00	0.05	1.504	1.1	0.03	270	-65	Terakimti Oxide
TRC072		21.00	24.00	3.00	0.05	1.229	4.0	0.02	270	-60	Terakimti Oxide
		35.00	39.00	4.00	0.01	2.977	108.2	0.01			Ag-Rich Oxide
TRC073		31.00	38.00	7.00	0.00	8.000	15.5	0.00	270	-73	Terakimti Oxide
		54.00	69.00	15.00	1.03	0.188	10.2	2.93			Terakimti Sulfide
		69.00	78.00	9.00	3.02	1.069	31.3	0.26			Terakimti Supergene
		78.00	83.00	5.00	0.86	0.383	12.6	0.12			Terakimti Sulfide

table continues…

Hole ID		From (m)	To (m)	Interval (m)*	Copper (%)	Gold (g/t)	Silver (g/t)	Zinc (%)	Local Azimuth (°)	Dip (°)	Prospect
TRC074	No Significant Results								273	-65	
TRC075		14.00	20.00	6.00	0.05	4.790	2.0	0.02	276	-65	Terakimti Oxide
		26.00	31.00	5.00	0.04	1.970	2.8	0.02			
TRC076		17.00	22.00	5.00	0.05	0.758	1.1	0.02	270	-60	Terakimti Oxide
TRC077		3.00	14.00	11.00	0.08	0.694	0.9	0.04	273	-64	Terakimti Oxide
TRC078		26.00	32.00	6.00	0.04	2.714	2.4	0.00	270	-65	Terakimti Oxide
		37.00	49.00	12.00**	0.02	0.708	93.0	0.00			Silver Interval
	including	44.00	49.00	5.00	0.02	1.264	139.7	0.01			Sulfide Rich Ag Oxide
TRC079		4.00	12.00	8.00	0.14	1.712	2.2	0.07	270	-60	Terakimti Oxide
TRC080		23.00	44.00	21.00	0.06	5.997	30.2	0.03	270	-65	Terakimti Oxide
	including	38.00	44.00	6.00	0.01	1.363	76.5	0.00			Ag-Rich Oxide
TRC081		5.00	18.00	13.00	0.11	5.363	8.2	0.05	272	-60	Terakimti Oxide
TRC082		46.00	58.00	12.00**	0.01	12.179	39.0	0.00	270	-62	Terakimti Oxide
	including	52.00	58.00	6.00	0.02	6.045	56.6	0.00			Ag-Rich Oxide
TRC083		49.00	56.00	7.00	0.03	2.576	37.5	0.01	271	-63	Terakimti Ag-Rich Oxide
		56.00	63.00	7.00	4.74	1.155	31.1	0.04			Terakimti Supergene
		63.00	81.00	18.00	0.80	0.807	15.2	0.07			Terakimti Sulfide
TRC084		5.00	12.00	7.00	0.05	3.897	3.0	0.02	274	-65	Terakimti Oxide
TRC085		27.00	31.00	4.00	0.05	3.763	26.3	0.01	271	-62	Terakimti Oxide
TRC085A		24.00	44.00	20.00	0.03	8.922	24.3	0.01	270	-64	Terakimti Oxide
	including	35.00	39.00	4.00	0.01	21.600	49.1	0.00			Ag-Rich Oxide
TRC086		0.00	3.00	3.00	0.05	2.377	3.9	0.03	272	-65	Terakimti Oxide
TRC087		11.00	20.00	9.00	0.09	5.091	19.0	0.05	272	-61	Terakimti Oxide
	including	18.00	20.00	2.00	0.09	0.668	37.6	0.04			Ag-Rich Oxide
TRC088		6.00	12.00	6.00	0.03	7.137	7.0	0.02	272	-60	Terakimti Oxide
TRC089		2.00	12.00	10.00	0.04	2.745	4.6	0.02	270	-61	Terakimti Oxide
		22.00	43.00	21.00	0.36	0.036	0.2	0.52			
TRC090		12.00	18.00	6.00	0.09	2.951	11.1	0.03	270	-64	Terakimti Oxide

table continues…

Hole ID		From (m)	To (m)	Interval (m)*	Copper (%)	Gold (g/t)	Silver (g/t)	Zinc (%)	Local Azimuth (°)	Dip (°)	Prospect
TRC091		16.00	18.00	2.00	0.06	1.545	2.8	0.02	272	-62	Terakimti Oxide
TRC092		1.00	5.00	4.00	0.19	0.971	0.4	0.03	271	-60	Terakimti Oxide
		12.00	24.00	12.00	0.10	7.541	7.8	0.02			
	including	17.00	19.00	2.00	0.10	40.600	41.2	0.03			Ag-Rich Oxide
TRC093**		16.00	21.70	5.70	0.09	5.366	21.1	0.02	270	-65	Terakimti Oxide
TRC093A		18.00	35.00	17.00	0.05	1.208	20.5	0.01	270	-57	Terakimti Oxide
	including	28.00	32.00	4.00	0.02	1.226	38.9	0.00			Ag-Rich Oxide
TRC094	No Significant Results								270	-65	Terakimti
TRC095		33.00	47.00	14.00	0.02	5.290	169.7	0.01	270	-60	Terakimti Oxide
	including	39.00	47.00	8.00	0.01	6.304	280.6	0.01			Ag-Rich Oxide
TRC096		23.00	45.00	22.00	0.02	5.946	23.8	0.01	274	-64	Terakimti Oxide
	including	37.00	45.00	8.00	0.01	4.803	46.9	0.00			Ag-Rich Oxide
TRC097		36.00	46.00	10.00	0.02	4.039	37.9	0.01	270	-62	Terakimti Oxide
	including	45.00	46.00	1.00	0.01	6.203	273.7	0.00			Ag-Rich Oxide
		46.00	67.00	21.00	4.92	0.940	25.4	0.04			Terakimti Supergene
TRC098		0.00	13.00	13.00	0.18	4.437	3.4	0.03	271	-61	Terakimti Oxide
TRC099		40.00	57.00	17.00	0.01	3.722	43.5	0.00	270	-70	Terakimti Oxide
	including	39.00	57.00	18.00	0.01	3.523	50.3	0.00			Ag-Rich Oxide
		57.00	63.00	6.00	0.06	1.480	48.3	0.01			Terakimti Sulfide
TRC100		0.00	10.00	10.00	0.03	13.034	4.0	0.04	270	-67	Terakimti Oxide
		15.00	28.00	13.00	0.04	1.291	3.2	0.03			
TRC101	No Significant Results								271	-55	Terakimti
TRC102		15.00	23.00	8.00	0.77	0.455	21.9	4.18	270	-60	Terakimti Sulfide
TRC103		13.00	17.00	4.00	0.04	1.067	1.3	0.02	269	-67	Terakimti Oxide
TRC104		16.00	17.00	1.00	0.04	3.521	0.0	0.03	268	-64	Terakimti Oxide
TRC105		6.00	8.00	2.00	0.05	0.861	3.7	0.01	273	-60	Terakimti Oxide
TRC106		12.00	21.00	9.00	0.12	2.277	12.0	0.04	270	-65	Terakimti Oxide
TRC107		7.00	9.00	2.00	0.07	4.058	16.7	0.01	270	-62	Terakimti Oxide

table continues...

Hole ID		From (m)	To (m)	Interval (m)*	Copper (%)	Gold (g/t)	Silver (g/t)	Zinc (%)	Local Azimuth (°)	Dip (°)	Prospect
TRC108	No Significant Results								270	-61	
TRC109		0.00	22.00	22.00	0.11	7.341	1.3	0.09	270	-60	Terakimti Oxide
TRC110	No Significant Results								270	-65	
TRC111		5.00	6.00	1.00	0.02	1.007	0.0	0.06	270	-60	Terakimti Oxide
TRC112		25.00	33.00	8.00	0.01	0.600	4.1	0.01	270	-83	Terakimti Oxide
		41.00	68.00	27.00	1.29	0.503	9.3	0.03			Terakimti Supergene
TRC113		10.00	19.00	9.00	0.06	6.436	1.7	0.04	270	-64	Terakimti Oxide
		23.00	34.00	11.00	0.03	0.667	1.3	0.01			Terakimti Oxide
		38.00	48.00	10.00	0.01	1.232	21.8	0.00			Terakimti Oxide
	including	43.00	45.00	2.00	0.01	3.607	37.2	0.00			Ag-Rich Oxide
		48.00	60.00	12.00	0.53	0.284	9.1	1.77			Terakimti Transition
TRC114		37.00	48.00	11.00	0.02	2.133	21.0	0.00	270	-65	Terakimti Oxide
	including	40.00	43.00	3.00	0.01	5.614	52.7	0.00			Ag-Rich Oxide
		48.00	55.00	7.00	6.11	1.455	40.7	0.05			Terakimti Supergene
		55.00	65.00	10.00	7.54	1.495	38.9	1.27			Terakimti Transition
		67.00	73.00	6.00	0.45	0.252	3.5	0.03			Terakimti Sulfide
TRC115		0.00	2.00	2.00	0.11	0.824	3.4	0.10	270	-60	Terakimti Oxide
		30.00	32.00	2.00	0.01	1.550	31.1	0.01			Terakimti Oxide
		36.00	51.00	15.00	0.57	0.311	0.3	0.01			Terakimti Supergene
TRC116	No Significant Results								266	-80	
TRC117	No Significant Results								273	-55	
TRC118		20.00	27.00	7.00	0.03	14.314	22.8	0.01	274	-60	Terakimti Oxide
TRC119		28.00	40.00	12.00	0.01	3.867	13.9	0.00	274	-71	Terakimti Oxide
	including	28.00	32.00	4.00	0.01	7.694	30.5	0.00			Ag-Rich Oxide
		55.00	60.00	5.00	0.07	0.872	4.6	0.03			Terakimti Sulfide
TRC120		17.00	38.00	21.00	0.08	3.485	5.5	0.02	270	-60	Terakimti Oxide
TRC121		33.00	37.00	4.00	0.01	4.924	34.6	0.00	270	-61	Terakimti Oxide
	including	33.00	36.00	3.00	0.01	6.460	44.7	0.00			Ag-Rich Oxide

table continued...

Hole ID		From (m)	To (m)	Interval (m)*	Copper (%)	Gold (g/t)	Silver (g/t)	Zinc (%)	Local Azimuth (°)	Dip (°)	Prospect
TRC122		25.00	41.00	16.00	0.02	2.115	16.6	0.00	269	-61	Terakimti Oxide
TRC123		43.00	46.00	3.00	0.47	0.706	16.6	0.15	270	-90	Terakimti Transition
TRC124		53.00	62.00	9.00	0.04	3.829	34.5	0.00	267	-65	Terakimti Oxide
	including	55.00	58.00	3.00	0.02	6.618	82.8	0.01			Ag-Rich Oxide
		62.00	73.00	11.00	0.25	0.461	7.6	0.01			Terakimti Sulfide
TRC125		30.00	33.00	3.00	0.01	0.392	1.1	0.02	275	-64	Terakimti Oxide

Notes: *Intervals stated are 40 to 100% true thickness.
** intervals restated after QA/QC review
***intervals restated after additional sub-sampling completed
Calculated intervals for gold are based on rounding to two decimal places.
Intervals use a 0.3 gram per tonne gold cut-off value, for gold only intervals. No top cut is used on assay values.
^^Zinc interval, not subject to gold cut-off criteria. Zinc intervals use a 1.00% cut-off value.
#Hole targeted oxide mineralization only. Hole of insufficient length to test potential for deeper mineralization.

10.2.2 REVERSE CIRCULATION SAMPLE QUALITY AND SAMPLE RECOVERY

Fladgate completed analyses of the RC sample quality from logged estimates of sample recovery and measurements of sample weight. Fladgate received a total of 1,806 weight measurements representing 40% of the RC drilling assay database.

An analysis of the bulk sample weight data shows the following.

- The average weight of all samples is 26.04kg.
- The average weight of all samples reported to have 100% recovery, is 33.34kg (547 samples total). The average weight of all samples with 100% recovery, removing all moist or wet samples, is 32.56 kg. However 85 of these samples are very heavy (sulphide with average weights of almost 50kg, strongly biasing the average). The average weight of all samples with 100% recovery minus moist or wet and $S, $D, $SM and $M, is 24.07kg.
- The average weight for all samples with 100% recovery shows the SG variability very well, ranging from GOSL (25.81kg average) to $M (49.63 kg average). The gossans are consistently lighter than the other rocks, largely due to the removal of elements during acid leaching and creation of porosity, and possible increased loss of material to the outside return.
- A total of 278 samples out of 1,608 samples were logged as being wet. This represents approximately 15% of the samples.

A comparison of the visual estimates of sample recovery with estimates of sample recovery derived from the sample weights is shown in Table 10.3. There is good agreement between the recovery estimates except in the vuggy gossan rock type (GosV). The vuggy gossan has lower visual estimates of sample recovery than those derived from the sample weights due to uncertainty in the amount of porosity. The sample recoveries in the gossans are significantly lower than those in the surrounding rock types and the underlying sulphides. Sample recoveries in the gossans range from 56% (GosV) to 66% (GosL) while in the other rock types the recoveries range from 74% (QFP) to 93% (IV).

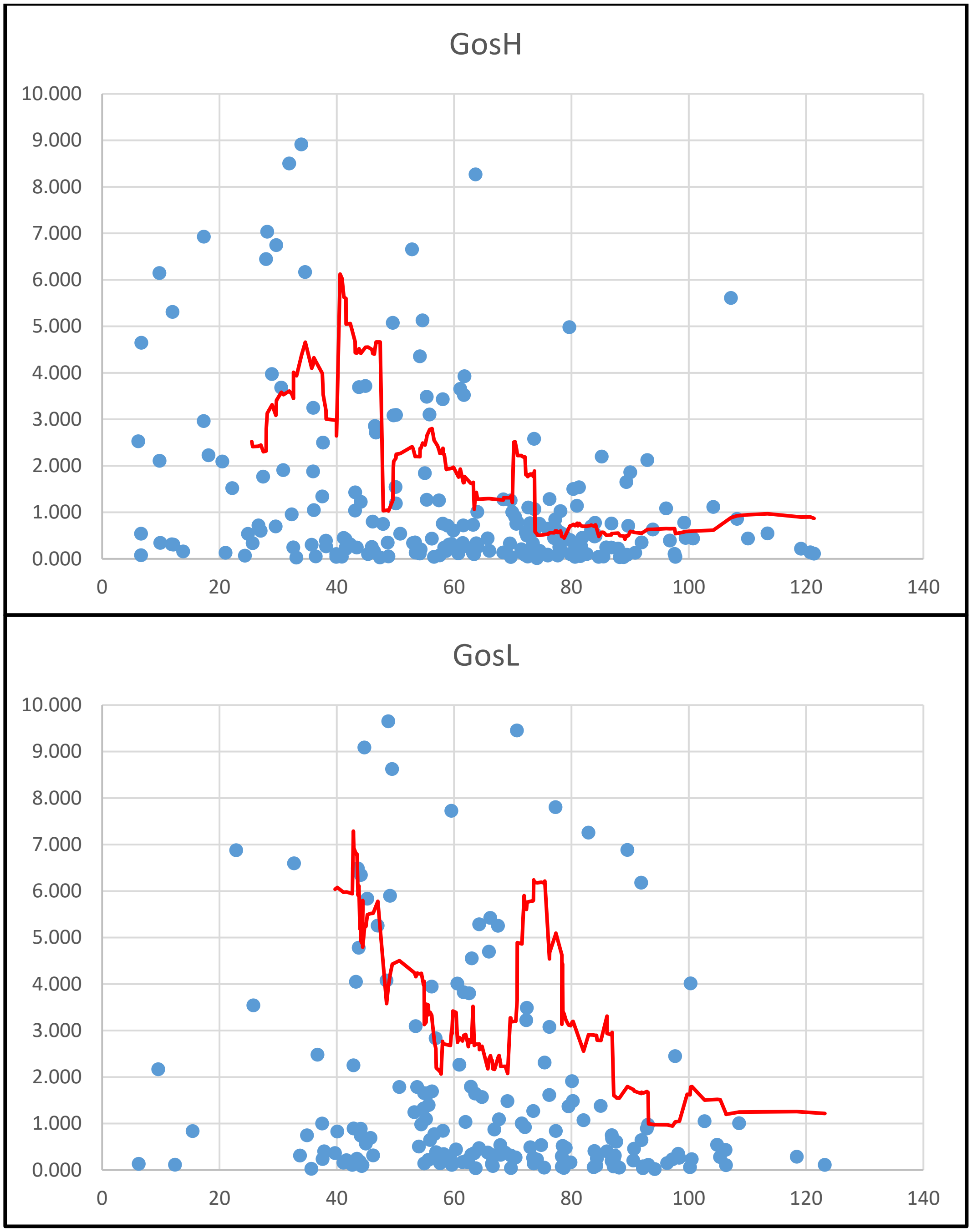
Table 10.4 Comparison of Visual Estimates of Recovery with Sample Weight Estimates

Lithology	Number	Minimum (%)	Maximum (%)	Sample Weight Mean (%)	Standard Deviation	Visual Estimate Mean	Difference (%)
Alt	340	4.07	210.06	79.77	30.29	80.2	-0.5
Massive	189	19.86	170.29	75.64	27.35	80.9	-6.9
Semi-Massive	66	25.48	153.40	78.43	29.10	83.6	-6.6
GosF	88	18.88	99.54	62.71	20.66	67.3	-7.3
GosH	220	6.19	121.36	60.40	25.11	68.4	-13.2
GosL	189	6.25	123.21	65.57	21.67	72.0	-9.8
GosM	81	2.75	97.86	52.10	23.48	55.5	-6.5
GosS	136	6.35	133.17	56.93	27.66	56.6	0.6
GosV	92	7.40	122.89	55.53	24.97	42.7	23.1
IV	61	33.08	156.82	92.97	29.16	86.4	7.1
QFP	214	16.66	162.99	74.34	26.27	75.8	-2.0
All Data	1676	2.75	210.06	69.48	-	71.8	-3.3

Fladgate plotted grades against the sample recoveries. Two of the six identified types of gossan (GosH and GosL) show evidence of a correlation between gold grade and sample recovery. Figure 10.4 shows scatter plots of the gold grades against sample recovery. There is a trend of higher grades with lower sample recoveries.

Fladgate concludes that there is uncertainty whether higher gold grades are associated with more friable zones or whether more gold is preferentially recovered in intervals with lower recovery (i.e. the samples are biased). The comparison of the grade of reverse circulation samples with the grades of core drillhole samples suggests the former may be the case.

Figure 10.4 Scatter Plots of Gold Grade Against Reverse Circulation Sample Recovery



Note: The red line shows a moving average window of 20 samples.

10.2.3 Assessment of Down-the-Hole Sample Contamination

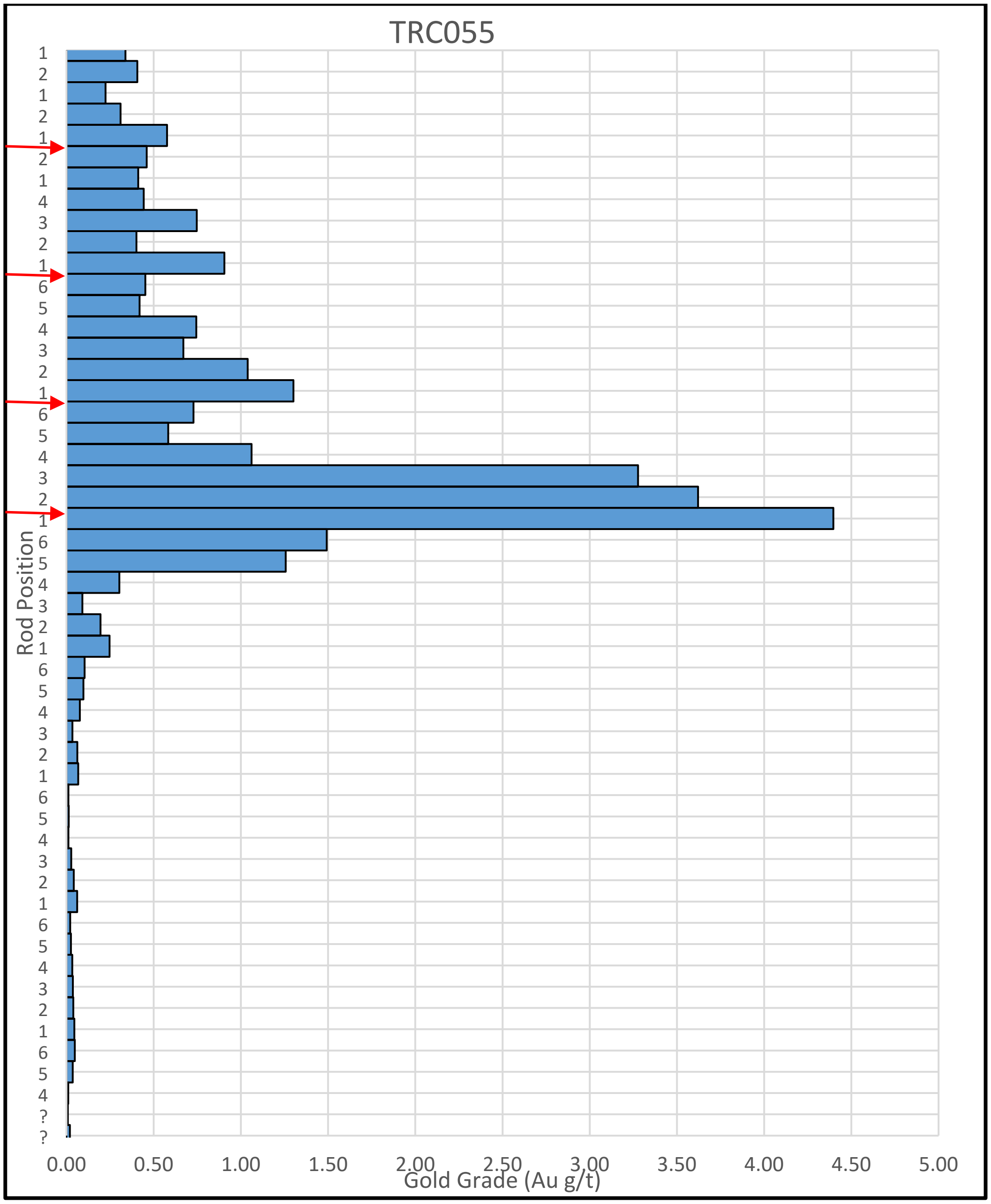
Fladgate completed an assessment of the risk of down-the-hole sample contamination during reverse circulation drilling. Fladgate examined the assay data for evidence of asymmetric grade profiles that are skewed downward and examined the reverse circulation assay data for evidence of cyclicity, where samples at a particular rod position (usually the first or last sample of a rod) have an unusual proportion of the highest grades.

Fladgate identified 20 reverse circulation drillholes with significant asymmetric grade profiles. A similar analysis was completed on the core drillholes which showed that asymmetric grade profiles are also present. Three of the drillholes with asymmetric grade profiles are located in areas with adjacent reverse circulation holes with asymmetric grades. In addition, the proportion of drillholes with asymmetric grades are similar (44.5% of the reverse circulation drillholes and 40% of the core drillholes).

Fladgate therefore concludes that the asymmetry in the grade profiles is naturally occurring and is not a result of contamination.

Two drill holes were identified with evidence of cyclicity, TRC052 and TRC055 where the highest grade in a sequence of rods is found at the first sample position. The down-hole grade profile of TRC055 is shown in Figure 10.5. Hole TRC055 is located in the northern zone of mineralization in the sulphide and therefore does not have any impact on the oxide mineral resource estimate.

Figure 10.5 Cyclicity in Hole TRC055



Fladgate concludes that there is no evidence of significant down-the-hole contamination in the reverse ciculation drilling.

10.2.4 COMPARISONS OF GRADES BY DATA TYPE

Fladgate compared the gold grades of twin holes and paired composites for the different drillhole types. The comparisons were performed to evaluate the risk of any systematic bias resulting from the different methods of drilling and sampling.

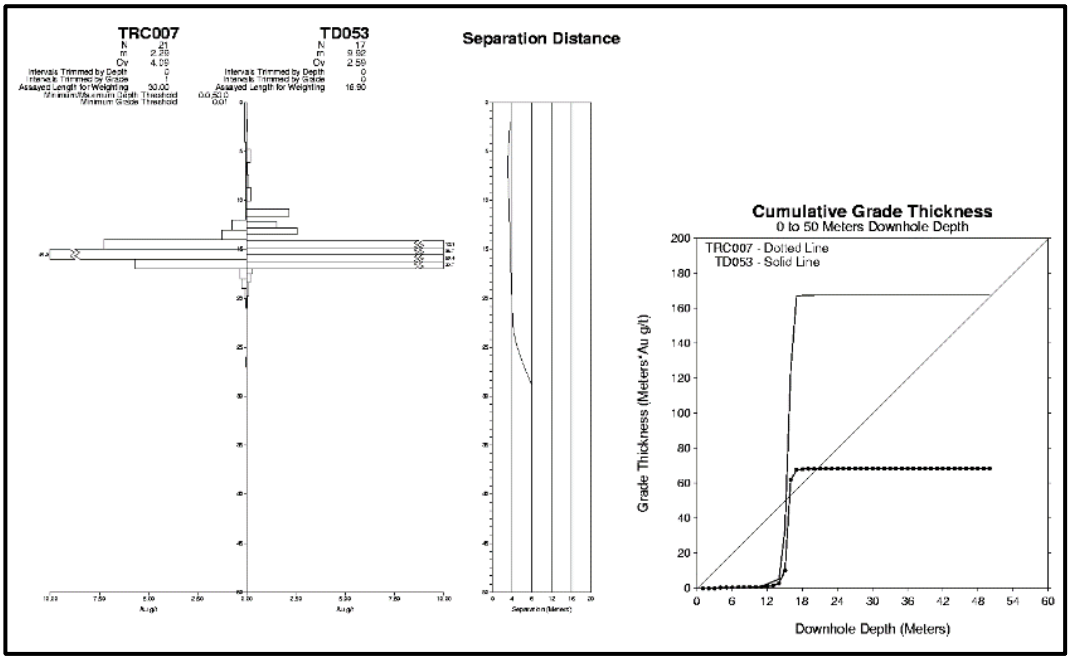
10.2.5 Core and RC Twin Holes

EAM drilled eight reverse circulation holes to twin pre-existing core drillholes within the oxide mineralization. Fladgate completed a comparison of the grades of the twin holes.

Out of the eight twin holes, four times the core drill holes are higher in grade than the corresponding reverse circulation hole, three times the reverse circulation holes is higher. One pair of holes showed similar grades. Fladgate concludes that neither of the drillhole types is systematically higher grade than the other.

An example of a graphical plot of twin holes is shown below in Figure 10.6.

Figure 10.6 Twin Hole Plot, TD053 and TRC007

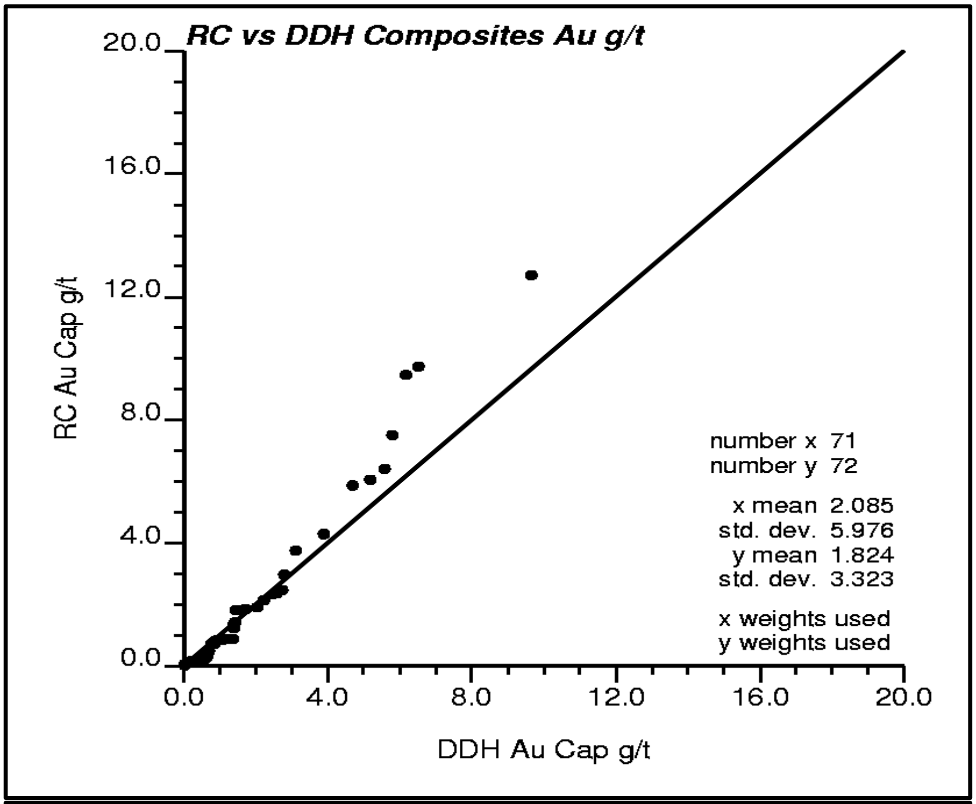


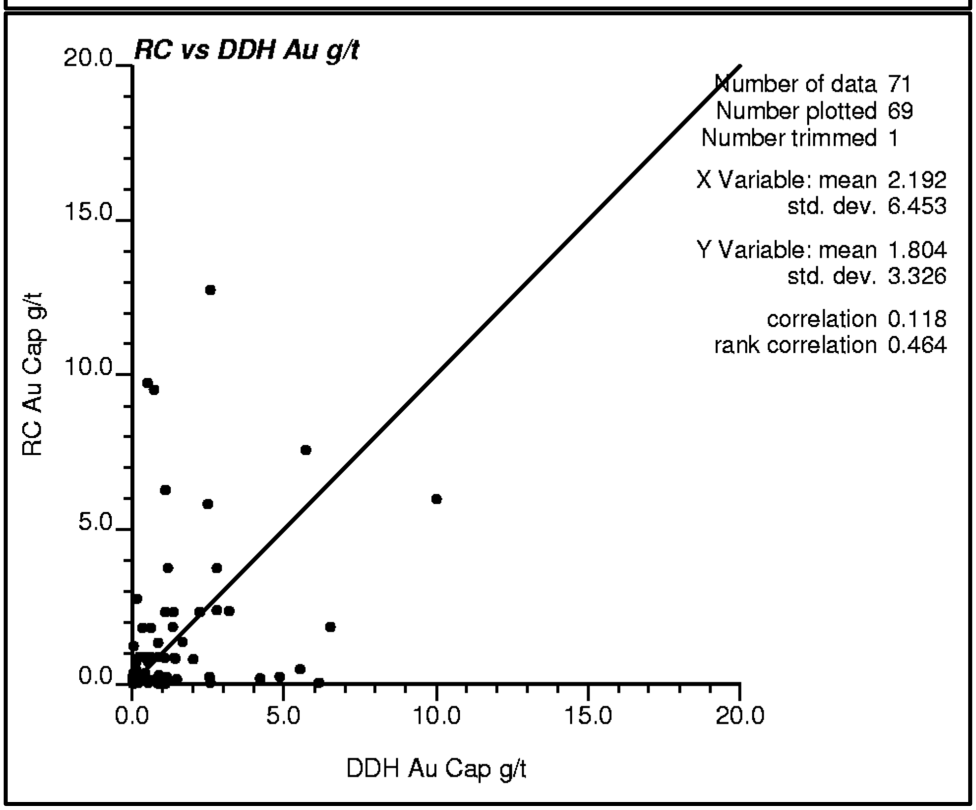
10.2.6 Paired Comparisons of Reverse Circulation, Core

Fladgate compared capped 3 m composites of each data type using distance thresholds to find the closest samples.

A Quantile-Quantile (Q-Q) plot and scatter plot of the paired core drillhole composites and reverse circulation composites with a separation distance of less than 7.5 m are shown below in Figure 10.7. The correlation between the composites are poor, however the mean gold grades are similar. The variance of the core composites is higher, which is to be expected based on the volume-variance relationship. The volume of the reverse circulation composites is larger therefore their variance is lower compared to the lower volume and higher variance of the core composites.

Figure 10.7 Scatter Plot and Q-Q Plot of Paired Reverse Circulation and Core Composites





11.0 SAMPLE PREPARATION, ANALYSIS AND SECURITY

The following sections summarize sample preparation, analyses and security protocol used in the drilling and trench programs at the Harvest Property. Quality assurance (QA) and quality control (QC) procedures are described in further detail in Section 11.4.

11.1 DRILL CORE AND REVERSE CIRCULATION SAMPLE HANDLING AND SECURITY

Geotechnical logging was performed on the core samples at the drill site to avoid unnecessary breaks that might occur during transport and therefore affect the apparent RQD of the core. Core orientation marks were taken every 6 m using a spear and the core was oriented and marked. At the end of every shift the core was transported by pick-up truck to Harvest Mining's Guna core logging and storage facility in Shire. The core logging process involved an initial cleaning of the core and checking of the core tags, and mark-ups on the individual boxes. Any discrepancies noted were addressed with the driller who was responsible for the core. At the camp all core was photographed prior to being logged by the geologist with an emphasis on structure, lithology, alteration and mineralization. Completed drill core logs were reviewed by a senior geologist to check consistency in logging.

Sample intervals were marked-up by the geologist logging the core and were based on sample intervals of either 0.7 m for mineralized core or 1 m for non-mineralized core. Sample intervals did not cross geological contacts. The physical sampling of the core was done with a diamond blade core cutting saw. The core was sawn in half along the line marked by the geologist to ensure a representative sample was taken.

Cloth sample bags were pre-numbered by a technician and the split core was moved to the sampling area for final preparation. Individual samples were then bagged and the ticket book filled out with tickets added to the sample and to the core box. The "side" of the split core was chosen systematically by reference to the orientation line and foliation in order to prevent any bias in sample selection. The samples from each drill hole were laid out in succession within the sampling area and loosely tied before being taken to the specific gravity station, where the specific gravity is determined and recorded by emersion of the sample in water. The samples are patted dry and then re-bagged, whereupon they are securely tied using the draw string and a final weight of the sample, and bag, is recorded for export purposes. All bags were sealed at the end of each shift. All sample preparation, and in particular the selection and insertion of QC samples, was undertaken under the direct supervision of the logging/project geologist. The remaining core was retained in the core trays and taken to the storage area. The individual sealed sample bags were placed in 60 L polypropylene barrels and sealed with tape for shipment to the preparation laboratories. All samples are inspected MoMPNG in Addis

Ababa to obtain an export permit. This requires the local Harvest Mining representative to be present while the barrels are opened and bags removed. The MoMPNG official opens each sample to visually inspect the contents and weighs each sample to ensure no extra material is being exported. Once an export permit has been obtained the MoMPNG seals the samples in the container, and takes them directly to the Ethiopian Revenue and Customs Authority, Airport Branch Office with the representative of the company where all customs formalities will be fulfilled before handing over the samples to the shipping company. The samples remain sealed until they arrive at the ACME sample preparation laboratory in Ankara, Turkey. The sample material is crushed in Ankara, with a 1. kg sample further pulverized on site. A pulversized split sample is sent to ACME in Vancouver for inductively coupled plasma-mass spectrometry (ICP-MS) and fire assay analysis, with the master coarse reject and pulp samples retained in ACME storage in Ankara.

Certified reference materials are stored in Harvest Mining's main office building in clearly marked plastic bags. The individual 100 g standards are held in clear plastic bags with removable identification labels to minimise the potential for insertion of an incorrect standard. Certified blanks are also stored in 100 g sealed plastic bags in the Harvest Mining office.

Duplicate soil sample paper bags are sorted and placed in cloth calico bags before being catalogued and stored in the core shed at Guna. Similarly, diamond drill core is stored inside in one of two sheds at Guna. The core shed is clean and well organized.

All reverse circulation samples are collected on 1 m intervals in large green plastic bags. Sampling occurs at the drill site and samples are collected from within mineralised zones (gossan, massive sulphides, and stringer zones) at 1 m intervals. All 1 m samples are split to ensure homogeneity and repeatability as they may be required to be re-assayed at a later date. Samples collected from outside of mineralised zones are composited into 3 or 4 m samples and placed in bags. These composite samples are then sub-sampled with a spear type sampler. If these samples return a high result the original 1 m samples are split and assayed. As part of the company's QA/QC protocol, standards (gold and base metal) and blanks are inserted every 20 samples. Replicates are also collected every 20 samples and replicate samples are split to ensure they closely match the original sample.

Reverse circulation drilling (644) samples collected in 2014 were sent to ACME in Turkey for aqua regia digest followed by fire assay for gold and silver and ICP-MS analysis for multi-elements. Samples are pulped to 75 µm. High-grade gold and silver values are analysed by fire assay and gravimetric finish.

11.2 LABORATORY PROCEDURES

Initial sampling of 17 Phase 1 boreholes was limited to high priority zones where mineralization and/or alteration were recorded during the core logging procedure. The resultant samples were submitted to ALS Laboratories in Vancouver, Canada, for analyses. Later in the exploration program, sampling was conducted on lower priority sections of these drill holes, and these samples were processed at ACME laboratory facility in Ankara, Turkey. Consequently, the situation exists where sample suites from 10 of the Phase 1 drill holes were sampled, prepared and analysed in different laboratories at different times (68% of Phase 1 samples at ALS and 32% at ACME). In contrast, all samples taken from the Phase 2 and later boreholes drilled by Harvest Mining were prepared at ACME Laboratories Ankara and analysed at ACME Analytical Laboratories (Vancouver) Ltd. (Note ACME is now known as the Bureau Veritas Mineral Group.)

The ALS Laboratory Group in Vancouver carries current International Organization for Standardization (ISO) 9001:2008 and ISO/International Electrotechnical Commission (IEC) 17025:2005 accreditation. ACME Analytik Ankara has ISO 9001:2008 Quality Management System accreditation. ACME Analytical Laboratories (Vancouver) Ltd. carries current ISO 9001:2008 accreditation for the provision of assays and geochemical analyses.

Bureau Veritas Minerals (formerly known as Ultratrace) in Perth, Australia, analyzed soil and stream samples. Rock and trench samples were analyzed by ACME, with preparation in Ankara, Turkey and analysis in Vancouver, Canada.

The sample preparation and assay methodologies used for rock and core samples at each of the laboratories employed by Harvest Mining, are comparable. A summary of the preparation and analytical procedures at ACME and Ultratrace is detailed in Table 11.1.

Table 11.1 Preparation and Assay Methods used During Analysis of Exploration Samples at ALS, ACME and IUntratrace

Sample Type	Laboratory	Preparation	Analytical Technique	Analyses	Detection Limit
Soil and Stream	Ultratrace	Samples pulverized	Aqua regia digest and ICP-MS analysis	Au	1 ppb
				Ag	0.05 ppm
Drilling (Phase 1)	ALS	Samples crushed to 70% <2mm, split and pulverized to 85% <75 µm	Fire assay and AAS	Au	0.005 ppm
			Aqua Regia digestion	Cu, Pb, Zn, Ag plus other multi-elements; Al, As, Au, B, Ba, Be, Bi, Ca, Cd, Ce, Co, Cr, Cs, Fe, Ga, Ge, Hf, Hg, In, K, La, Li, Mg, Mn, Mo, Na, Nb, Ni, P, Pd, Pt, Rb, Re, S, Sb, Sc, Se, Sn, Sr, Ta, Te, Th, Ti, Tl, U, V, W, Y, Zr	Ag 0.001 ppm Cu 0.01 ppm Pb 0.005 ppm Zn 0.1 ppm
Drilling, (Reverse Circulation and Diamond), Rock Chip, Trenching and Channel Sample	ACME	Samples crushed 1 kg to 80% passing 10 mesh, split 1000g and pulverized to 85% passing 200 mesh	Hot Aqua Regia digestion for base-metal	Cu, Pb, Zn, Ag plus other multi-elements; Al, As, Bi, Ca, Cd, Co, Cr, Fe, Hg, k, Mg, Mn, Mo, Na, Ni, P, Pb, S, Sb, Sr, W.	Ag 2 ppm
			sulphide and precious-metal ores. ICP-ES		Cu 0.001%
			analysis.		Pb 0.01%
					Zn 0.01%
			Fire assay	Au	0.005 ppm
			Gravimetric fire assay if Au is >10 ppm	Au	>10 ppm
			Gravimetric fire assay if Ag is >300 ppm	Ag	>300 ppm
			Volumetric titration if Cu > 20%	Cu	>20%
			Titration if Pb is >10%	Pb	>10%
			Titration if Zn is >40%	Zn	>40%

Note: AAS – atomic absorption spectroscopy; ICP-ES inductively coupled plasma-emission spectroscopy

11.3 SECURITY

The chain of custody procedure from the extraction of the core from the core barrel or reverse circulation rig, through logging and sampling up to the point of dispatch to the laboratory is described above. Through all of these stages the responsibility for security lies with Harvest Mining and their on-site personnel. Samples are transported from Shire to the MoMPNG in Addis Ababa, then onwards to Bole International Airport by an Ethiopian haulage company. After this the samples are in the care of airline cargo companies and international courier companies when shipped to the overseas laboratories. The security of the sample during transit cannot be guaranteed as tamper proof seals are not used on the sample bags. Upon receipt at the laboratory, the chain of custody passes to the assayer. Following assay, the remaining material is stored under secure conditions at the laboratory facilities. Approximately 1 kg of pulp is created from each drill core sample, with 100 g sent to ACME's Vancouver laboratory for analysis and the remaining 900 g stored in Turkey for potential follow-up work. The chain of custody reverts to Harvest Mining if the samples leave the assay laboratory storage facilities. This is the case with remaining pulp material following analyses, which is transferred back to the EAM corporate office in Vancouver, Canada, who then sent to a secured warehouse location for storage.

In general, industry best practices with respect to chain of custody procedures are followed on site. However, the weakest point in any chain of custody is during transport. The absence of tamper proof fastenings on the samples has been noted and their introduction would greatly improve the chain of custody between the site and laboratory. However, the physical inspection and weighing of all exported material by the MoMPNG in Addis Ababa adds complexity to this solution.

11.4 QUALITY ASSURANCE/QUALITY CONTROL REVIEW

Diamond drilling at the Harvest Property was initially supervised by Chinese geologists before the Harvest Mining geologists took responsibility of exploration in 2011. The geologists directed and managed the preparation, logging and sampling of core. With several geologists having logged the drill core, variation in lithologies and interpretation invariably occurred. In May 2012, Paul Cranney, an independent geological consultant, visited the property and spent 19 days re-logging all core drilled on the Harvest Property, with the exception of 10HTD004. This re-logging was necessary to rationalise the variation that resulted from multiple geologists logging core. By performing this study, it significantly improved correlations between various lithologies, and the corresponding geochemical signatures.

During sampling, QC standards and blanks were inserted to confidentially monitor laboratory performance. The progressive introduction and refinement of QA/QC procedures at Harvest included the implementation of field, reject and pulp duplicates, as well as specific programs of re-analysis and umpire laboratory assaying, consistent with industry best practice.

As previously described, three distinct drilling programs have been undertaken at the Terakimti deposit on the Harvest Property. Phase 1 was completed during 2009-2010 and consisted of 12 drillholes. Phase 2 drilling comprised of some 69 drillholes completed between 2011 and 2013. Phase 3 drilling was completed in 2014-2015 and included 127 reverse circulation drillholes, and 6 diamond drillholes. The majority of samples from the Phase 1 boreholes were analysed at ALS Vancouver whereas the subsequent Phase 2 and 3 samples were processed at ACME. For this reason, QA/QC procedures and results for each individual phase of drilling are presented in the following sections.

Certified reference materials (CRMs) are stored in Harvest Mining's main office building in clearly marked plastic bags. The individual 100 g standards are held in clear plastic bags with removable identification labels to minimise the potential for insertion of an incorrect standard. Certified blanks are also stored in 100 g sealed plastic bags in Harvest Mining's office. Coarse reject pulverized rock samples and the remainder of the base pulps (the remainder of the initial 1 kg of pulverized sample) are stored at the initial ACME preparation facility in Turkey, and analytical pulps are stored at the ACME Laboratory in Vancouver.

11.4.1 Phase 1 Quality Assurance/Quality Control

Certified Reference Materials

A variety of CRMs derived from certified laboratories in Australia and Canada were used during the initial sampling of the Phase 1 drillholes. Specifically, certified laboratory standards were obtained from CDN Resource Laboratories (CDN Labs) and Ore Research and Exploration Pty Ltd (Ore Research) for incorporation into the sampling sequence. These samples were inserted at a rate of approximately 9% and were analyzed at ALS.

The results of gold, copper, zinc and lead standards were generally within acceptable limits. CDN-ME-4 produced some variable results. Copper results for CDN-ME-2 returned results which were higher than recommended values. CDN-ME-11 returned results for zinc which were consistently below established values.

Blanks

Un-mineralized basalt was used as a blank control sample during sampling of the Phase 1 drilling. In addition, another blank sample (DD-Blank) was also used during the initial sampling. This blank was a barren gravel collected from a Vancouver gravel pit. Together, these were inserted at a rate of approximately 6% for samples analysed at ALS. Un-mineralized basalt was used consistently during the sampling of low priority sections of these Phase 1 drillholes analysed at ACME. These were inserted at a rate of approximately 6%.

No evidence of systematic contamination was found.

Check Assays

During 2013, EAM sent a representative selection of 274 pulp duplicate samples for check assaying of gold at the ALS Chemex laboratory in Vancouver. Results indicated that the gold assays from the Phase 1 drilling campaign are suitably accurate to support mineral resource estimation.

11.4.2 Phase 2 Quality Assurance/Quality Control

Phase 2 drilling includes all drillholes completed at Harvest between 2011 and 2013. All samples taken from these drillholes were prepared at the ACME Laboratory in Ankara and subsequently analysed at ACME Vancouver. Quality control standards and blanks were inserted routinely into the sample stream. Field, reject and pulp duplicates were also introduced during this drilling campaign.

Certified Reference Materials

Certified reference materials were obtained from CDN Labs and Ore Research. for incorporation into the sampling sequence. These samples were inserted at a rate of approximately 6% and were analyzed at ACME.

In general, the overall performance of the standards was within acceptable limits for base metals (copper, lead, zinc) and silver. On an individual basis, there were samples which registered caution or fail warnings for various elements. EAM undertook remedial action through re-analysis of samples surrounding the failing standard, particularly when located within or adjacent to a mineralized section within a drill hole.

Due to use of Geostats multi-element SRMs (GBM series), samples originally assayed from December 2011 to February 2012 were essentially uncontrolled for gold, with only nominal Neutron Activation values supplied for gold contents. Once this oversight was identified, EAM immediately introduced gold CRMs into their standard stream. From February 2012, dedicated gold standards were included in addition to base metal standards in areas of suspected mineralization considered to be potentially gold-bearing, and a systematic retest of the gold values obtained between December 2011 and February was additionally conducted.

Generally, results from the CRMs were acceptable.

Blanks

As per the Phase 1 drilling, un-mineralized basalt was used as a blank control sample during sampling of the subsequent drilling. The results of the blank analyses were generally acceptable and did not show evidence of contamination.

Field Duplicates

Field duplicates were not taken routinely as part of the Phase 2 drillhole sampling procedures. However, a series of quarter core duplicate samples were taken retrospectively from each of the Phase 2 drillholes for a total of 404 samples (approximate rate of 5% or 1 sample in 20). These samples were prepared at ACME in Ankara and analysed at ACME in Vancouver using the same techniques as the original samples.

90th percentile precision values are shown in Table 11.2. In general, the filed duplicates show reasonable reproducibility.

Table 11.2 90th Percentile Precision of Field Duplicates

Element	90th Percentile Precision (%)	Number	Insertion Rate (%)
Gold	±45.4	306	4.8
Silver	±27.4	79	1.2
Copper	±53.5	307	4.8
Lead	±45.3	99	1.6
Zinc	±36.9	212	3.3

Coarse Reject Duplicates

Reject duplicates (also known as coarse reject duplicates) are splits of a sample taken after the coarse crush but before pulverizing and then assayed as a separate, duplicate sample. Coarse reject duplicates measure the homogeneity of the sample at the coarse reject stage and assesses combined preparation and analytical precision.

A total of 288 reject duplicate samples were taken from the Phase 2 drilling program. These samples underwent further preparation at ACME in Ankara and analysis at ACME in Vancouver using the same techniques as the original samples. The 90th percentile precision values, number of samples and insertion rates are shown in Table 11.3.

Table 11.3 90th Percentile Precision of Coarse Reject Duplicates

Element	90th Percentile Precision (%)	Number	Insertion Rate (%)
Gold	±30.7	215	3.4
Silver	N/A	30	0.5
Copper	±18.2	226	3.6
Lead	±15.3	50	0.8
Zinc	±18.2	162	2.6

In general, the reject duplicates show good reproducibility.

PULP DUPLICATES

Pulp duplicate samples are taken from the unused analytical pulp returned from the laboratory and are then sent for analysis at the same laboratory. This sample type is used to assess analytical precision.

A total of 335 pulp duplicate samples were taken from the Phase 2 drilling program. These samples were analysed at ACME in Vancouver using the same techniques as the original samples. In general, the reject duplicates show good reproducibility. The 90th percentile precision values are shown in Table 11.4.

Table 11.4 90th Percentile Precision of Pulp Duplicates

Element	90th Percentile Precision (%)	Number	Insertion Rate (%)
Gold	±21.3	264	4.2
Silver	±20.5	70	1.1
Copper	±9.7	285	4.5
Lead	±13.3	103	1.6
Zinc	±10.6	218	3.4

CHECK ASSAYS

A suite of check assay samples were collected from ACME Analytical Laboratories (Vancouver) Ltd. and submitted to ALS Global (Vancouver) for assay. A total of 24 check assay samples were selected to cover a range of assay grades. No evidence of analytical bias was found.

11.4.3 PHASE 3 QUALITY ASSURANCE/QUALITY CONTROL

Phase 3 drilling includes all reverse circulation drillholes completed at the Harvest Property between 2014 and 2015 and specifically drillholes TRC001 to TRC 125 (127 holes). All samples taken from these drillholes were prepared at ACME in Ankara and subsequently analysed at ACME in Vancouver. Quality control standards and blanks were inserted routinely into the sample stream. Field, reject and pulp duplicates were also introduced during this drilling campaign.

CERTIFIED REFERENCE MATERIALS

Certified reference materials were obtained from CDN Labs and Ore Research for incorporation into the sampling sequence. These samples were inserted at a rate of approximately 5.9% and were analyzed at ACME. Summary statistics of the results of the CRM analyses are presented in Table 11.5.

Table 11.5 CRM Results

Standard	Gold				Copper				Silver			
	Number	Expected Value (g/t)	Mean (g/t)	Bias	Number	Expected Value (%)	Mean (%)	Bias	Number	Expected Value (g/t)	Mean (g/t)	Bias
CDN-GS-20A	2	21.12	22.30	6%	0				0			
CDN-CM-19	3	2.11	2.04	-3%	3	2.04	2.07	1%	0			
CDN-CM-35	22	0.32	0.33	2%	22	0.25	0.25	0%	0			
CDN-GS-10D	12	9.50	9.28	-2%	0				0			
CDN-GS-2M	20	2.21	2.24	1%	0				0			
CDN-GS-7F	19	6.90	6.94	1%	0				0			
CDN-ME-1204	21	0.98	0.97	0%	21	0.52	0.52	1%	21	58	61.37	6%
CDN-ME-1304	20	1.80	1.81	0%	20	0.27	0.27	2%	20	34	36.59	8%
CDN-ME-1305	17	1.92	1.94	1%	17	0.62	0.62	1%	17	231	241.51	5%
CDN-ME-16	4	1.48	1.35	-9%	4	0.67	0.69	2%	4	30.8	34.15	11%
G398-2	13	0.42	0.52	23%	0				0			
G900-7	26	3.19	3.20	0%	0				0			
G901-7	23	1.53	1.50	-2%	0				0			
G903-10	20	0.21	0.20	-7%	0				0			
G995-4	16	8.48	8.52	0%	0				0			
GLG305-1	11	0.10	0.11	6%	0				0			
GLG911-5	15	0.21	0.21	2%	0				0			
Bias in Slope	264			4.3%	87			1.3%	62			3.9%
Bias in Overall Mean	264	2.68	2.69	0.3%	87	0.47	0.48	1.2%	62	95.94	101.01	5.3%

Blanks

As per Phase 1 and 2 drilling, un-mineralized basalt was used as a blank control sample during sampling of the subsequent drilling. Blanks were inserted at a rate of approximately 5.8%.

Out of 261 blanks, 7 failed for gold (a failure rate of 2.7%) and 5 failed for copper (a failure rate of 1.9%).

The results of the blank analyses were generally acceptable and did not show evidence of contamination.

Field Duplicates

Field duplicates were collected routinely during the Phase 3 drilling program. A total of 263 samples were taken (approximate insertion rate of 5.9%). These samples were prepared at ACME in Ankara and analysed at ACME in Vancouver using the same techniques as the original samples. The 90th percentile precision values are shown below in Table 11.6. Generally, the field duplicates show reasonable precision for this type of duplicate sample.

Table 11.6 90th Percentile Precision of Field Duplicates

Element	90th Percentile Precision (%)	Number	Insertion Rate (%)
Gold	±37.7	226	5.00
Silver	-	-	-
Copper	±16.0	226	5.00
Lead	-	-	-
Zinc	-	-	-

Coarse Reject Duplicates

A total of 248 reject duplicate samples were collected during the Phase 3 drilling program (for an insertion rate of 5.5%). These samples underwent further preparation at ACME in Ankara and analysis at ACME in Vancouver using the same techniques as the original samples.

The 90th percentile precision values are shown below in Table 11.7. The coarse reject duplicates show reasonable precision for copper and somewhat lower precision for gold.

Table 11.7 90th Percentile Precision of Coarse Reject Duplicates

Element	90th Percentile Precision (%)	Number	Insertion Rate (%)
Gold	±36.8	248	5.50
Silver	-	-	-
Copper	±21.4	248	5.50
Lead	-	-	-
Zinc	-	-	-

PULP DUPLICATES

A total of 248 pulp duplicate samples (for an insertion rate of 5.5%) were collected during the Phase 3 drilling program. These samples were analysed at ACME in Vancouver using the same techniques as the original samples. The 90th percentile precision values are shown in Table 11.8. The coarse reject duplicates show reasonable precision for copper and somewhat lower precision for gold.

Table 11.8 90th Percentile Precision of Pulp Duplicates

Element	90th Percentile Precision (%)	Number	Insertion Rate (%)
Gold	±27.6	248	5.50
Silver	-	-	-
Copper	±10.2	248	5.50
Lead	-	-	-
Zinc	-	-	-

CHECK ASSAYS

29 check assay samples were collected from ACME Analytical Laboratories (Vancouver) Ltd. and submitted to ALS Global (Vancouver) for assay. The results are shown in Table 11.9.

Table 11.9 Accuracy of ALS Check Assays

	ACME Cu (%)	ALS Cu (%)	ACME Au (g/t)	ALS Au (g/t)	ACME Ag (g/t)	ALS Ag (g/t)
Mean	0.05	0.05	4.70	4.52	161.52	161.23
Bias in Mean	-	-6.9	-	-3.9	-	-0.2
Bias in Slope	-	3.0	-	5.8	-	-0.4

The results show no evidence of significant analytical bias.

11.4.4 TERAKIMTI DRILLING QUALITY ASSURANCE/QUALITY CONTROL SUMMARY

Throughout the drilling programs on the Harvest Project, Harvest Mining has progressively monitored and improved their QA/QC procedures. This is reflected in the introduction of supplementary quality control samples such as field, reject and pulp duplicates in addition to umpire samples. Where issues have been identified, the company has been proactive in seeking a resolution to the problem through introducing new CRMs, re-analysis of failing samples and running check sample program to determine the veracity of the laboratory techniques used. An obvious lapse in the QA/QC procedures occurred over the period from December 2011 to February 2012 when the GBM series (base metal) standards were used as the sole discriminator of gold performance. This oversight was rectified through the introduction of additional gold standards in subsequent sample batches, and also through the retroactive statistical review of GBM series gold results, and re-analyses of questionable results.

In the author's opinion, the sample preparation, security and analytical procedures are adequate to support Mineral Resource estimation.

12.0 DATA VERIFICATION

In consideration of the data summarized below, as well as information provided elsewhere in this report, the author of this section believes the current Terakimti Project data are acceptable for the purposes used in this report.

12.1 ELECTRONIC DATABASE

Initially, a Microsoft® Access database and related Microsoft® Excel spreadsheets were provided by Tigray to Fladgate as a universal project dataset along with a full set of assay certificates. Additional GIS data was provided in the form of MapInfo data files.

Information recorded from diamond drill core logging and assaying was integrated using industry standard data management software (Maxwell DataShed). The resultant data was reviewed, including validation of a random selection of data against the source information, and it is considered acceptable for the purpose of this report.

12.1.1 FLADGATE DATABASE VERIFICATION

Fladgate selected assay certificates containing the highest 5% of the copper, gold, silver and zinc grades in the database from each of the Phase 1 and Phase 2 drilling campaigns. Fladgate requested and received the original assay certificates from the ALS Chemex (Phase 1) and ACME (Phase 2) assay laboratories. From the Phase 1 drilling campaign, a total of 348 assays out of a total of 794 assays were verified, representing 44% of the Phase 1 assay database. In the Phase 2 drilling campaign, a total of 4,100 assays out of a total of 7,691 assays were verified, representing 53% of the Phase 2 assay database.

A comparison was made between the assay data in the database and the assay data in the original assay certificates. No errors were found.

12.2 DRILL HOLE COLLAR CHECKS

Nine drillhole collar checks were undertaken by Fladgate during the site visit using a hand-held Garmin GPS unit. The average deviation was 1.27 m for the easting and 2.15 m for the northing, with the largest deviation recorded being 2.94 m in the easting and 5.15 m in the northing components for TD020. It was noted that several concrete slabs (e.g., TD50) used to mark the location of the collar positions were absent, probably by local farmers to enable cultivation of their crops. Absent drillhole collars slabs should be replaced for future reference.

12.2.1 FLADGATE DRILL HOLE COLLAR CHECKS

During the Fladgate site visit, eight drill hole collar checks were completed using a hand-held Garmin GPS unit. Fladgate found no significant differences between the coordinates in the database and the coordinates collected during the site visit. The results are tabulated in Table 12.1.

Table 12.1 Fladgate Verification of Drill Hole Collars GPS

	Database			Difference (m)		
DHID	Easting	Northing	Easting	Northing	Easting	Northing
TD53	416,406.0	1,583,703.0	416,395.1	1,583,711.3	-10.9	8.3
10HTD02	416,471.0	1,583,713.0	416,472.0	1,583,710.0	1.0	-3.0
TD07	416,477.0	1,583,749.0	416,476.8	1,583,740.6	-0.2	-8.4
TD048	416,111.0	1,583,508.0	416,108.6	1,583,494.8	-2.4	-13.2
TD49	416,091.0	1,583,519.0	416,088.3	1,583,518.4	-2.7	-0.6
TD30	416,141.0	1,583,582.0	416,134.6	1,583,586.1	-6.4	4.1
TD10	416,195.0	1,583,583.0	416,194.3	1,583,578.4	-0.7	-4.6
TD20	416,222.0	1,583,601.0	416,225.9	1,583,602.9	3.9	1.9

12.2.2 FLADGATE DOWNHOLE SURVEY CHECKS

During the site visit, Fladgate examined original downhole survey documents and made a comparison with the downhole survey records found in the Terakimti database.

Fladgate verified a total of 34 downhole survey records or 5% of the entire downhole survey database. The verified records originated from five drillholes, representing 7% of the total number of drill holes in the database.

No discrepancies were found between the original documents and the database used for Mineral Resource estimation.

12.3 FLADGATE DRILL CORE LOGGING VERIFICATION

During the site visit, Fladgate examined drill core from four drillholes and verified the drillhole logging. Fladgate made a comparison of the logged intervals of sulphide mineralization in the database with Fladgate's own observations of the sulphide mineralization.

Fladgate found no significant differences.

12.4 VERIFICATION OF TRENCH SAMPLE QUALITY

Fladgate compared the gold grades of paired composites for drillholes and trenches. The comparisons were performed to evaluate the risk of any systematic bias resulting from the different methods of drilling and sampling.

Fladgate compared capped trench composites with core drill hole composites within a separation distance of 30 m. There is a poor correlation between the composite pairs however the mean grades are similar. A comparison of trench composites with reverse circulation composites within a separation distance of 15 m shows that the reverse circulation data are significantly higher in grade than the trench composites.

Fladgate concludes that there is equivocal evidence for a bias in the grades of the trenches with respect to the drillhole composites. If a bias exists then it is likely that the trench composites are lower in grade than the drillholes and therefore the grades in the mineral resource model may be somewhat conservative.

12.5 INDEPENDENT VERIFICATION OF MINERALIZATION (AURUM)

12.5.1 CORE SAMPLES

As part of the review of mineralized intervals within the Tigray drill core, five independent samples were collected for assaying. The samples consisted of quartered core from selected intervals from drillholes TD025, TD040, TD057 (all Terakimti), HD002 (and Mayshehagne), and TVD001 (VTEM09). Two CRM were included with the core samples which were then submitted to OMAC Laboratories Limited, a subsidiary of ALS Minerals based in Loughrea, Republic of Ireland for fire assay, using the Au-GRA22 technique, and base metal assaying using the ME-ICPORE technique.

The results of the independent samples analyzed at ALS are presented alongside the original ACME assay results for the selected intervals in Table 12.2.

Table 12.2 Independent Quartered Core Sample Assay Results

Hole ID	Sample No.	From (m)	To (m)	ACME (Au ppm)	ALS (Au ppm)	ACME (Cu %)	ALS (Cu %)	ACME (Pb %)	ALS (Pb %)	ACME (Zn %)	ALS (Zn %)
HD002	69,429	40.05	40.75	1.16	1.23	4.17	3.93	3.17	2.55	17.03	18.65
TD025	69,430	75.40	76.25	2.76	2.52	11.64	12.60	0.05	0.06	1.64	1.19
TD040	69,431	239.90	240.60	0.19	0.22	0.94	1.22	1.22	1.55	29.72	28.4
TD057	69,432	141.60	142.30	2.30	1.83	2.21	2.60	0.06	0.05	0.54	0.69
TV D001	69,433	29.30	29.85	3.85	8.41	2.40	2.53	0.81	0.97	7.77	8.19

Given the strong nugget effect typically seen with gold, the variation between the original ACME assay and the ALS verification results is in good agreement. Base metal concentrations are also in reasonably good agreement with the historic data.

12.5.2 Soil Samples

Two independent soil samples were collected for gold assaying to verify the validity of the soil geochemistry program in the Ruwa Ruwa trend. Owing to access problems encountered on the property, caused by the maturation of the grain crop, the author was unable to collect field samples. Instead, two Niton XRF samples from Adi Goshu, which corresponded to sites previously sampled for gold soil analyses, were selected for re assay.

The samples were submitted to OMAC Laboratories Limited, a subsidiary of ALS Minerals based in Loughrea, Republic of Ireland for fire assay, using the Au-GRA22 technique. The results of the independent samples analyzed at ALS are presented alongside the original Ultratrace assay results in Table 12.3.

Table 12.3 Independent Soil Sample Assay Results

Original Sample ID	New Sample ID	Ultratrace (Au ppm)	ALS (Au ppm)
W24239	NX25802	1.28	0.60
W24261	NX25854	1.20	0.82

Given the strong variability of gold in soil and the fact the XRF soils were collected at a depth of approximately 10 cm, rather than 20 to 30 cm for soil geochemistry samples, the ALS verification results indicate the samples clearly contain gold and correlate with the Ultratrace results. The results independently verify that the Ruwa Ruwa Trend contains gold soil anomalism.

12.5.3 UMPIRE SAMPLES

A suite of Umpire Samples were collected from ACME Analytical Laboratories (Vancouver) Ltd. and submitted to ALS Global (Vancouver) for assay. These samples represent pulp duplicates which are prepared at the primary laboratory and then set aside for later submission to a second laboratory for pulverization and assay. This approach provides a test of sample preparation and splitting procedure in the laboratory in addition to analytical variation. A total of 24 Umpire Samples were selected to cover a range of assay grades.

Scatterplot of the Umpire Sample results plotted against the original ACME assays for gold and copper are presented in Figures 12-1 and 12-2, respectively.

The Umpire Sample results show an excellent correlation (r2= 0.996) with the ACME assay results up to the 8 ppm gold level. Similar correlations are seen for copper (r2= 0.995), silver (r2= 0.992), lead (r2= 0.997) and zinc (r2= 0.997). Above this concentration a tendency towards higher grades for the original ACME assay is observed.

The ALS results for the Umpire Samples confirm the presence of gold and base metals within the samples and the levels of correlation are consistent with other coarse crush duplicate datasets.

Figure 12.1 Scatterplot of Pulp Duplicate Sample Pairs (Au)

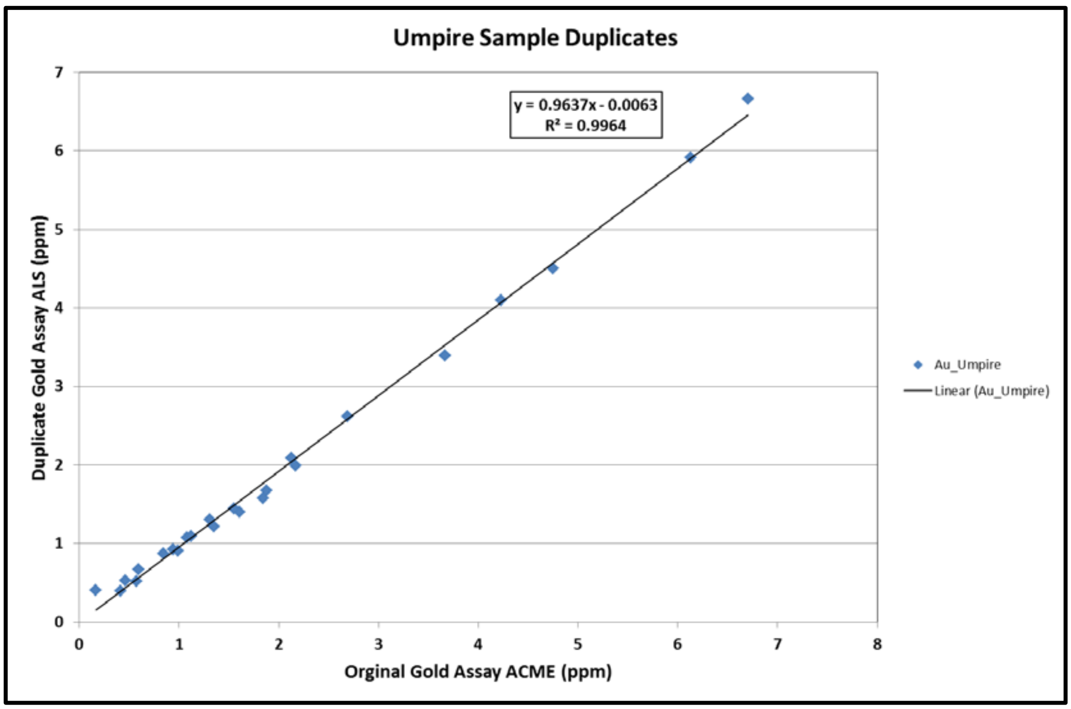
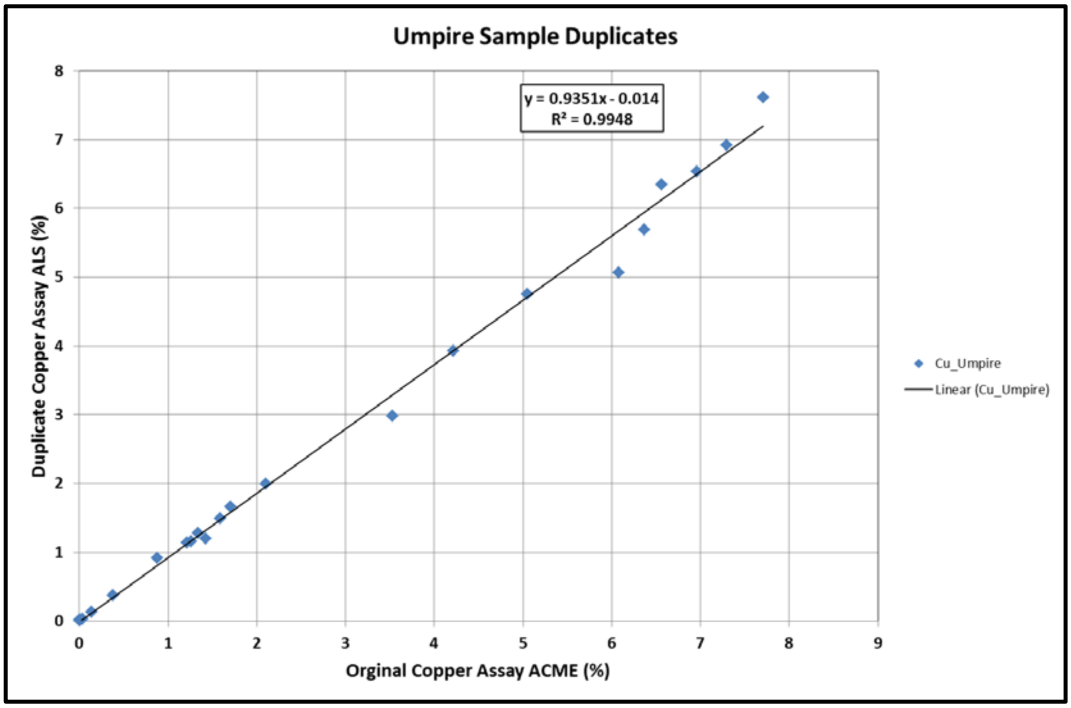


Figure 12.2 Scatterplot of Pulp Duplicate Sample Pairs (Cu)



12.6 QP Comments on Section 12.0

As a result of the data verification completed by Fladgate, the QP concludes that the drillhole data collected by Tigray is of sufficient quality to support Mineral Resource estimation.

Data verification on other geochemical (e.g., XRF and soil geochemistry) and geophysical surveys databases on the property was not conducted, since the Mineral Resource estimate focused on the drill core geochemistry.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 TEST WORK BY BLUE COAST RESEARCH LTD

In 2013 Blue Coast Research Ltd. (BCR) conducted a test program to investigate the amenability of cyanide leach technology for the extraction of gold and silver from two oxide composite samples collected from the north and south oxide zones of the Terakimti deposit.

The drill-hole samples used for constructing the two composites and head assays on the composite samples are presented in Table 13.1.

Table 13.1 Sample Composition and Head Assay

Composite	Drill Hole ID	Intercept (m)		Weight (kg)	Head Assay				
		From	To		Au (g/t)	Ag (g/t)	Fe (%)	Cu (%)	Zn (%)
15	TD29	36.5	45.3	9.72	3.41	30.3	9.55	0.05	0.03
	TD30	7.2	17.7	10.74					
	TD41	28.5	31.7	5.39					
	TD44	17.2	30.6	21.39					
	TD49	38.9	58.0	25.71					
16	TD007	0	22.5	48.01	2.26	7.17	13.06	0.06	0.05
	TD038	14.5	20.5	11.17					

Composites 15 and 16 were both subjected to bottle roll cyanide leach testing for 24 hours at pH 10.5 to 11. The concentration of sodium cyanide used for the tests was 1 g/L. The feed particle sizes varied from 80% passing 1,700 µm to 100 µm. The test results are shown in Table 13.2 and leach kinetics are presented in Figure 13.1 and Figure 13.2.

For Composite 15, gold extraction was 75% at a feed particle size of approximately 1,700 µm. The extraction improved to 82% when the sample was further ground to 80% passing 100 µm. Silver recoveries were lower, ranging from 35 to 45% at the respective sizes. Leach kinetics were rapid. Within five hours of leach retention time, most of the leachable gold and silver were extracted.

Figure 13.1 Gold and Silver Leach Kinetics- Composite 15

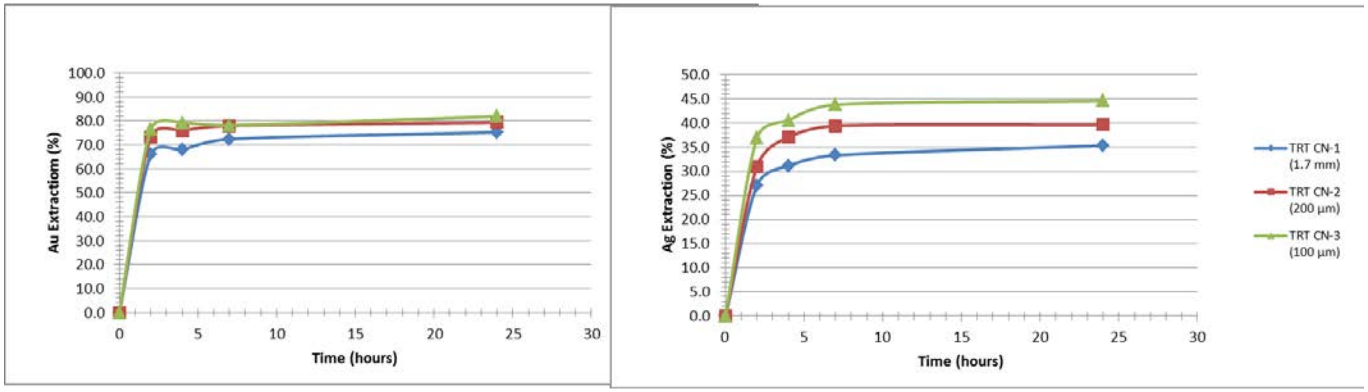


Figure 13.2 Gold and Silver Leach Kinetics – Composite 16

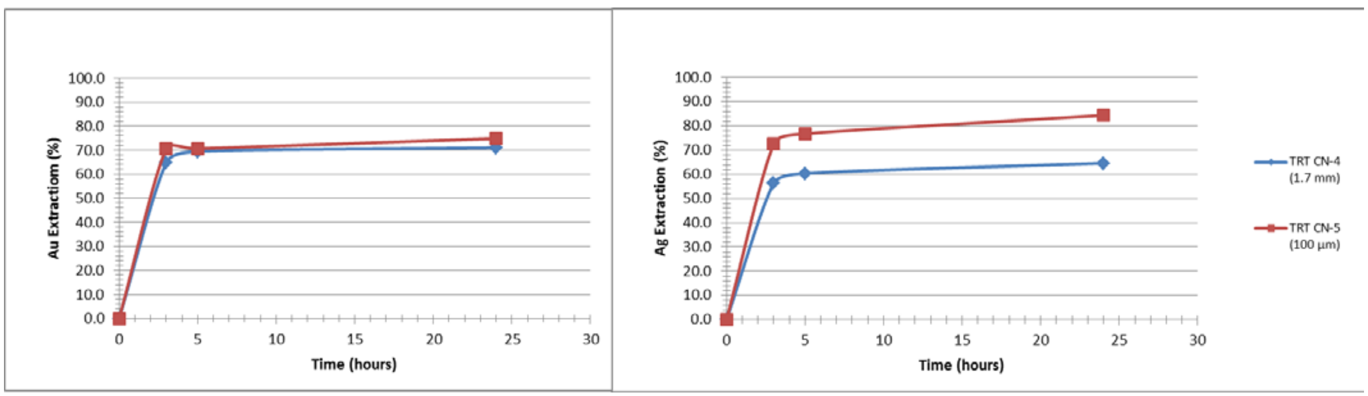


Table 13.2 Gold and Silver Cyanidation Test Results

Test ID	Sample	Particle Size 80% Passing (µm)	Reagent Consumption (kg/t of Feed)		Head Grade (g/t)		Extraction (%)	
			NaCN	CaO	Au	Ag	Au	Ag
TRT CN-1	Comp 15	<1,700	0.57	1.00	4.0	32.7	75.2	35.3
TRT CN-2	Comp 15	200	0.41	0.98	3.9	33.9	79.3	39.6
TRT CN-3	Comp 15	100	0.55	0.96	4.0	34.0	81.9	44.6
TRT CN-4	Comp 16	<1,700	0.74	1.36	2.3	6.3	71.0	64.7
TRT CN-5	Comp 16	100	0.65	1.38	2.4	6.5	74.9	84.4

For Composite 15, gold extraction was 75% at a feed particle size of approximately 1,700 µm. The extraction improved to 82% when the sample was further ground to 80% passing 100 µm. Silver recoveries were lower, ranging from 35 to 45% at the respective sizes. Leach kinetics were rapid. Within five hours of leach retention time, most of the leachable gold and silver were extracted.

For Composite 16, gold extractions were slightly lower, in comparison to Composite 15. The gold extractions were 71% at the feed particle size of 80% passing 1,700 µm and 75% at 80% passing 100 µm. Silver extractions were 65% and 84%, respectively, which are notably higher than the results generated from Composite 15. Composite 16 also showed similarly rapid leach kinetics.

The metallurgical test work shows that the two mineralization samples are amenable to conventional cyanide leaching for extraction of gold and silver.

13.2 TEST WORK BY SGS SOUTH AFRICA (PTY) LTD.

In November 2015, EAM initiated a comprehensive test work program to assess the metallurgical characteristics of the oxide zone of the Terakimti deposit. The objective of this program was to test the amenability of the Terakimti oxide and transition material to leach extraction of gold and silver utilizing industry standard heap leaching or agitated leaching technology. This program included the drilling of six dedicated metallurgical holes to obtain representative samples and selection of SGS, an internationally recognized metallurgical laboratory to perform the test work.

The key results of the test work are as follows:

- Gold extractions averaged 70% and 75% for simulated heap leach tests and 78% for agitated leach tests indicating that the Terakimti oxide mineralization will be amenable to either heap leaching or agitated leaching with cyanide for the recovery of gold and silver.
- The Terakimti material is expected to achieve satisfactory solution percolation rates with moderate agglomeration.
- The Terakimti oxide mineralization is relatively soft.

13.2.1 SAMPLE COLLECTION

In October and November of 2015 EAM metallurgical and geological personnel conducted a detailed review of the Terakimti deposit geological data including an assessment of grade, mineralogy, lithology, zonation and distribution. The objective was to develop a sample collection program that would provide a suite of samples for test work that would be representative of the overall oxide zone of the Terakimti deposit, taking into account mineralogical variation and geographic distribution.

The metallurgical sampling program included six drillholes completed along the strike of the deposit with each hole drilled through the entire depth of the oxide and transition zones. The cores from each hole were logged, weighed and photographed at the Terakimti site. A total of 1,887 kg of sample were collected. The complete, whole core from each hole was individually packaged, sealed and prepared for transport to the SGS laboratory in Johannesburg, South Africa. The core drilling, logging and preparation for transport was performed under the supervision of Iain Groves, and Steve Gardoll, EAM's geologists at Terakimti.

Table 13.3 Metallurgical Drill Hole Sample Details

Hole ID	Depth (m)	Core Recovery (m)	Core Recovery (%)	Weight (kg)	Geographic Zone
TD070	44.2	41.31	93.5	367.4	Northeast
TD071	39.62	32.56	82.2	289.7	Central
TD072	36.58	31.21	85.3	340.0	Northeast
TD073	55.23	44.72	81.0	343.9	Central
TD074	45.72	29.43	64.4	283.2	Southwest
TD075	49.38	31.26	63.3	263.4	Southwest

SAMPLE CLASSIFICATION

The initial geological review identified six mineralized lithologies that would be recognized for metallurgical testing as well as an estimate of the amount of each lithology relative to the total sample mass collected:

Table 13.4 Sample Lithologies

No.	Lithology	Sample (%)
1	Gossan F (GOS F)	10
2	Gossan H (GOS H)	16
3	Gossan H-Cl (GOS HCl)	9
4	Gossan L (GOS L)	20
5	Silica (SIL)	33
6	Transition (TRANS)	12

In addition, the samples were classified as to the geographic area of the oxide zone of the Terakimti deposit from which they were obtained. Three geographic zones were identified: Northeast, Central and Southwest. Two drillholes were located in each zone: TD 070 and 072 in Northeast; TD 071 and TD 073 in Central; TD 074 and TD 075 in Southwest.

13.2.2 SAMPLE PREPARATION AT SGS

Upon arrival at the SGS laboratory the sample containers were found to be secure and in good condition.

Samples were coded for identification as per the drillhole number, the lithology, and the geographical area of the oxide zone of the deposit from which the sample was collected—Northeast, Central or Southwest.

The initial stage of sample preparation was to split the rock core from each drill hole into the six lithologies previously identified and to retain these lithology samples separately. Then, the individual lithology samples from the two drill holes in each geographic zone were combined to create six lithology samples for each of the three geographic zones. This resulted in a total of eighteen master lithology samples created and retained for test work (three geographic zones x six lithologies). This was done under the direction of EAM's staff geologist. In advance of test work each master sample was crushed to –16 mm.

13.2.3 SCREEN ANALYSIS AND GRADE ANALYSIS

A single composite head sample was created by combining portions from each of the eighteen crushed master samples for a particle size distribution and gold grade analysis. The results of the screen size and grade analysis are presented in Table 13.5.

Table 13.5 Head Sample Screen and Grade Analysis

	Mass/Distribution				Gold Grade	Gold Distribution
Size	(g)	(%)	(% Retained)	(% Passing)	(g/t)	%
+16 mm	0	0.0	0.0	100		
+10 mm	562.3	12.7	12.7	87.3	7.93	15.4
+6 mm	425.8	9.6	22.2	77.8	6.36	9.4
+3.35 mm	600.6	13.5	35.8	64.2	7.92	16.4
+1.7 mm	469.7	10.6	46.3	53.7	6.41	10.4
+850 µm	397.8	9.0	55.3	44.7	5.76	7.9
+425 µm	366.2	8.2	63.5	36.5	6.21	7.8
+212 µm	523.7	11.8	75.3	24.7	0.86	1.5
+106 µm	622.8	14.0	89.3	10.7	8.20	17.6
+75 µm	301.8	6.8	96.1	3.9	7.68	8.0
-75 µm	171.6	3.9	100	0.0	9.29	5.5
Total	**4,442**	**100.0**	-	-	-	**100.0**

13.2.4 BOND GRINDING TEST WORK

Three composite samples were subjected to Bond Ball Work Index (BWi) test work to determine the hardness of the Terakimti material to ball mill grinding. The BWi averaged 7.6 kWh/t which indicates the material is very soft when compared to other deposits.

Table 13.6 Bond Ball Work Indices

Sample Composite	Lithologies	Bond BWI (kWh/t)	Product Particle Size (µm, 80% passing)
Composite 1	GOS F & GOS L	8.6	113
Composite 2	GOS H & GOS H-Cl	5.5	101
Composite 3	Silica	8.7	111
Average	-	7.6	-

An Abrasion Work index (Ai) test was performed. The measured Ai for the tested sample is approximately 0.0429, showing low abrasive characteristics.

13.2.5 AGITATED LEACH TESTS

A total of 24 agitated cyanide leach tests (bottle rolls) were performed in two stages on separate samples representing the six main lithologies and geographic zones within the oxide zone of the Terakimti deposit.

In the first stage eighteen bottle roll tests were performed, one for each of the six lithologies from each of the Northeast, Central and Southwest areas of the oxide zone of the deposit. The objective was to determine the gold and silver extraction from each of the six lithologies and to assess if the extractions varied within the three geographic zones of the oxide area of the deposit. All samples were ground to 80% passing 106 µm and the leach period was 48 hours. Lime and cyanide consumptions were measured; however, leach kinetics were not determined. The average gold extraction was 75.5%, varying from a low of 56.6% to a high of 91.2%.

The silver extraction averaged 47.1%, however, the silver extractions for the individual tests were highly variable. The test results are presented in Table 13.7 and Table 13.8.

Table 13.7 Gold Extraction – Bottle Roll Leach Tests at 80% Passing 106 µm

Lithology	Gold Extraction (%)* Geographic Zone Northeast	Southwest	Central	GOS F Surface	Average
GOS F	88.5	71.7	75.9	-	78.7
GOS H	84.3	56.6	80.5	-	73.8
GOS H-CL	91.2	80.2	56.8	-	76.1
GOS L	70.7	75.2	78.1	-	74.7
SILICA	61.0	82.8	79.8	-	74.5
TRANSITION	56.6	80.5	89.1	-	75.4
GOS F Surface	-	-	-	67.3	67.3
Average	75.4	74.5	76.7	67.3	75.5**
Test Parameters: Grind Size: 80% -106 µm; Leach Retention Time: 48 hours; NaCN Dosage: 2.0 kg/t, pH: 10.5					

Note: *Silver extractions calculated using average of residue assays and solution assays.
**Excludes data from Sample GOS F Surface

Table 13.8 Silver Extraction - Bottle Roll Leach Tests at 80% Passing 106 µm

Lithology	Silver Extraction (%)* Geographic Zone Northeast	Southwest	Central	GOS F Surface	Average
GOS F	21	14	119	-	51
GOS H	64	70	119	-	84
GOS H-CL	36	21	36	-	31
GOS L	33	44	53	-	43
SILICA	64	1	86	-	50
TRANSITION	10	37	27	-	25
GOS F Surface	-	-	-	10	10
Average	38	31	73	10	47**
Test Parameters: Grind Size: 80% -106 µm; Leach Retention Time: 48 hours; NaCN Dosage: 2.0 kg/t, pH: 10.5					

Note: *Silver extractions calculated using average of residue assays and solution assays.
**Excludes data from Sample GOS F Surface

As reported by SGS, due to the silver assay issues, poor silver accountabilities (less than 90% and more than 110%) were found for the initial cyanidation leach tests. After reviewing the assay procedures, SGS was able to produce consistent results which resulted in better accountabilities for the late test work.

The consumption of sodium cyanide and lime was measured for each test. The consumptions are within typical ranges for oxide gold material.

Table 13.9 Reagent Consumptions – Bottle Roll Leach Tests at 80% Passing 106 µm

Lithology	Reagent Consumptions					
	NaCN (kg/t)			CaO (kg/t)		
	Northeast	Southwest	Central	Northeast	Southwest	Central
GOS F	0.50	0.73	0.73	1.88	1.58	1.00
GOS H	0.44	1.04	0.80	1.18	1.27	0.82
GOS H-CL	1.85	0.63	0.84	1.78	0.87	1.00
GOS L	0.33	0.71	0.75	1.02	1.34	0.89
SILICA	0.44	1.61	0.67	1.27	0.87	0.40
TRANSITION	1.67	1.44	0.96	0.45	0.63	0.83
GOS F Surface	0.92			1.29		
Average	0.90			1.07		

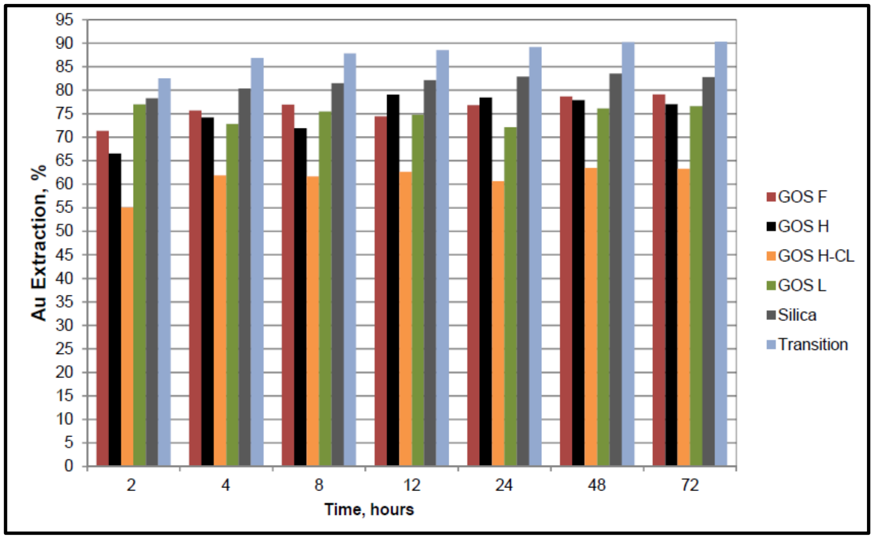
In the second stage six bottle roll leach tests were performed. These tests were performed on lithology composite samples. Portions of each lithology were combined according to the source zone (Northeast, Central and Southwest) thereby providing six separate lithology samples representing the entire oxide zone of the deposit. The samples were ground to 80% passing 75 µm and the tests run for 72 hours. Lime and cyanide additions were measured and leach kinetics were determined.

The average gold extraction from the six tests was 78.3% at 48 hours with no additional extraction achieved at 72 hours. The extractions for various individual samples at 72 hours ranged from 63.3 to 90.3%. The leach kinetics were rapid with an average gold extraction of 76.9% achieved at 12 hours. Silver extractions were variable, with an average of 47%. The test results are presented in Table 13.10.

Table 13.10 Gold and Silver Extractions - Bottle Roll Leach Tests at 80% Passing 75 µm

Lithology Type	Head Assays (g/t)		Extraction (%)				
			Leach Retention Time (h)				
			Gold				Silver
	Gold	Silver	12	24	48	72	72
GOS F	15.9	19.5	74.4	76.8	78.7	79.1	19
GOS H	8.31	37.9	79.1	78.4	77.9	77.0	73
GOS H-CL	4.16	5.6	62.6	60.6	63.5	63.3	30
GOS L	2.26	7.6	74.8	72.1	76.1	76.6	46
SILICA	6.82	27.0	82.1	82.9	83.5	82.8	60
TRANSITION	3.49	114.0	88.5	89.2	90.2	90.3	54
Average	6.8	35.3	76.9	76.7	78.3	78.2	47
Test Parameters: Grind Size: 80% -75 µm; NaCN Dosage: 2.0 kg/t, pH: 10.5-11.0							

Figure 13.3 Gold and Silver Leach Kinetics - 80% passing 75 µm



The leach tests done on the six lithology composites at different grind sizes did not indicate significant improvement in gold dissolutions at finer grinds sizes. The higher residual gold grades may be due to gold being locked in gangue minerals.

The consumption of sodium cyanide (NaCN) and lime (CaO) was measured for each bottle roll test. The results indicate consumptions that are within typical ranges for oxide gold material.

Table 13.11 Reagent Consumptions - Bottle Roll Leach Tests at 80% passing 75 µm

Lithology	Reagent Consumptions	
	NaCN (kg/t)	CaO (kg/t)
GOS F	1.59	0.84
GOS H	1.61	0.53
GOS H-CL	1.54	0.57
GOS L	1.57	0.36
SILICA	1.48	0.50
TRANSITION	1.59	0.66
Average	**1.56**	**0.58**

The results of the Bond rock hardness tests and agitated leach tests indicate that the oxide mineralization of the Terakimti deposit would be soft to ball mill grinding and would respond well to standard agitated cyanide leach treatment, such as carbon-in-pulp (CIP), carbon-in-leach (CIL) or agitated cyanide leaching followed by Merrill Crowe precipitation.

13.2.6 SIMULATED HEAP LEACH TESTS

Two series of simulated heap leach tests were performed as a first step to assess if the Terakimti oxide mineralization may be amenable to heap leach extraction using bottle leach procedure. The objective of the first series of simulated heap leach tests was to assess the effect of crush size on metal extractions and were performed on a composite sample. The second series of tests assessed the extraction to be achieved from each of the six lithologies, at a fixed crush size.

The composite sample for this first series was comprised of material from each of the six lithologies taken from each of the three geographic zones. Three simulated heap leach tests were performed on material crushed to 100% passing 6 mm, 100% passing 11 mm and 100% passing 16 mm. Each test was run for seven days.

The gold extraction for these three tests averaged 70.4%.

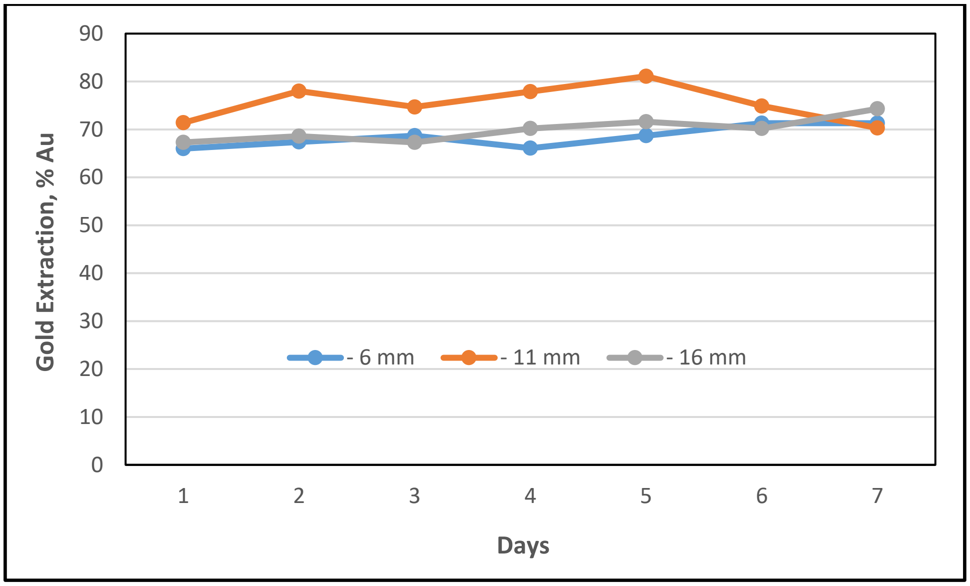
Table 13.12 Simulated Heap Leach vs Crush Size Results

Crush Size (100% passing)	Head Grade		Extraction		Reagent Consumption	
	Au (g/t)	Ag (g/t)	Au (%)*	Ag (%)**	NaCN (kg/t)	CaO (kg/t)
6 mm	7.4	28.2	71.3	32.2	1.97	0.46
11 mm	6.0	29.4	70.3	34.4	2.00	0.56
16 mm	6.9	29.8	74.3	34.3	1.98	0.45
Average	6.8	29.1	72.0	33.6	1.98	0.49
Test Parameters: NaCN Dosage: 5 kg NaCN/t; Leach Time: 7 days						

Notes: *Gold extractions calculated from leach residue assays.
**Silver extractions calculated using average of residue assays and solution assays.

The gold leach kinetics were rapid for each of the three crush sizes with most of the extraction achieved with 24 hours. The test results for the 100% passing 11 mm sample show extraction decreasing noticeably after five days. The reason for this is uncertain, however it is not consistent with the other tests, and therefore may be attributed to a problem with the test or assay.

Figure 13.4 Simulated Heap Leach Kinetics vs Crush Size



For the second series of simulated heap leach tests a separate test was performed on each of the six different lithology composite samples. Each lithology sample was composited with material from each of the three geographic zones of the oxide area of the deposit. These tests were performed at a crush size of 100% passing 16 mm and the tests were run for seven days. Gold extractions averaged 75.9%, and the leach kinetics were rapid with most of the gold extraction achieved within 24 hours. Silver extractions were poor, averaging at approximately 30%.

The consumption of sodium cyanide and lime for all the simulated heap leach tests are within typical ranges for gold oxide deposits. The test results including gold extraction and reagent consumptions are summarized Table 13.13.

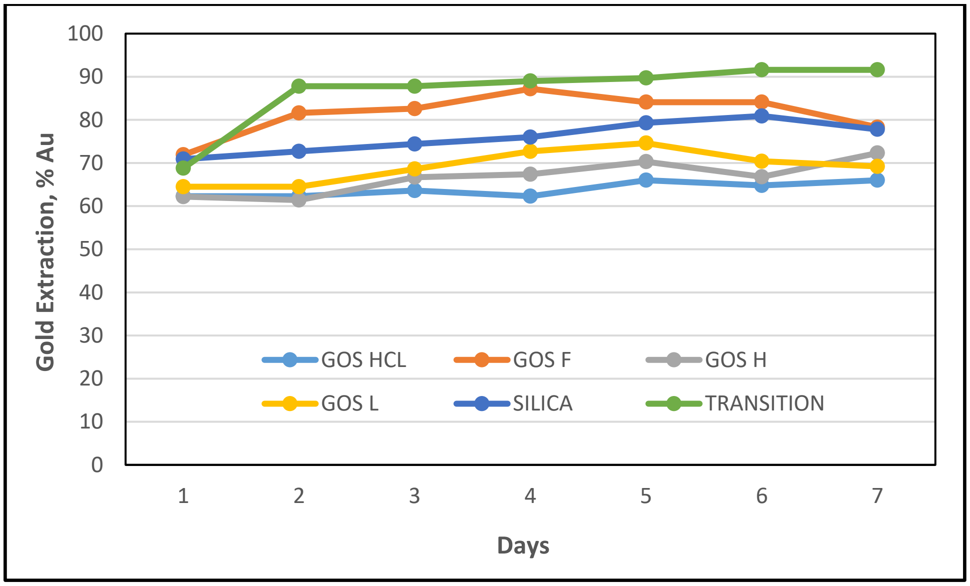
Table 13.13 Simulated Heap Leach Results by Lithology

	Calculated Head Grade		Extraction		Reagent Consumption	
Lithology	Au (g/t)	Ag (g/t)	Au (%)*	Ag (%)**	NaCN (g/t)	CaO (g/t)
GOS F	12.0	16.7	78.3	14.4	2.82	0.33
GOS H	12.7	45.1	72.3	62.3	2.39	0.30
GOS HCL	7.3	4.9	66.0	24.4	2.39	0.37
GOS L	2.35	13.8	69.2	26.1	1.52	0.30
SILICA	5.62	24.4	77.8	26.3	1.52	0.32
TRANSITION	4.82	125	91.6	26.1	1.37	0.32
Average	7.5	38.3	75.9	29.9	2.00	0.32
Test Parameters: NaCN Dosage: 5 kg NaCN/t; Leach Time: 7 days						

Notes: *Gold extractions calculated from leach residue assays.
**Silver extractions not available at date of this report.

The leach kinetic graphs in Figure 13.5 show that each of the lithologies demonstrates rapid extraction for gold with most of the gold extracted within 24 hours. The gold extraction from Sample GOS F decreases significantly after Day 4. Sample GOS L shows a significant decrease in the gold extraction after Day 5, and the Silica sample shows a lesser decrease after Day 6. The reason for these decreases in extraction was uncertain and should be assessed as test work continues.

Figure 13.5 Simulated Heap Leach Kinetics by Lithology



13.2.7 PERCOLATION AND AGGLOMERATION TESTS

Three percolation and agglomeration tests were performed on composite samples to assess the percolation rates through the samples. One composite sample was created for these tests with material from each of the six lithologies and from each of the Northeast, Central and Southwest zones of the oxide area of the deposit. The sample particle size for each test was 100% passing 16 mm. The samples were agglomerated with cement prior to the start of the tests. The test results are summarized below;

- Percolation Test #1 – 5 kg/t cement addition
  - In Test #1, with an initial solution addition rate of 10 L/h/m^2, ponding started on Day 3 and therefore this test was discontinued.
- Percolation Test #2 – 10 kg/t
  - In Test #2, the initial solution rate was 10 L/h/m^2 and increased gradually over the 11-day test period to a final solution addition rate of 100 L/h/m^2 and no ponding occurred. During the test the column height slumped by 29.5%.
- Percolation Test #3 – 15 kg/t

- In test #3 the solution rate started at 10 L/h/m^2 and increased gradually over the 11-day test period to a final addition rate of 100 L/h/m^2 and no ponding occurred. During the test the column height slumped by 16.7%.

The solution discharging from the bottom of the column was clear for each test.

The test results indicate satisfactory percolation rates can be achieved with the addition of 10 kg/t of cement, which is within industry norms for agglomeration in heap leach projects.

A pressure percolation test was conducted on the agglomerated GOS composite sample with a top particle size of 16 mm. A maximum corrected pressure of 247 kPa was obtained at a heap height of 18.1 m before ponding occurred at a water feed flowrate of 10 L/h/m^2.

13.2.8 COLUMN LEACH TEST WORK

The GOS composite sample was also tested for gold and silver extraction using column leach procedure to simulate the heap leach treatment. The crushed sample with a top particle size of 16 mm was agglomerated with 10 kg/t and 15 kg/t cement prior to the column leach testing. The gold and silver dissolution curves are shown in Figure 13.6 and the results are summarised in Table 13.14.

Figure 13.6 Gold and Silver Extraction Curves - Column Leach Test

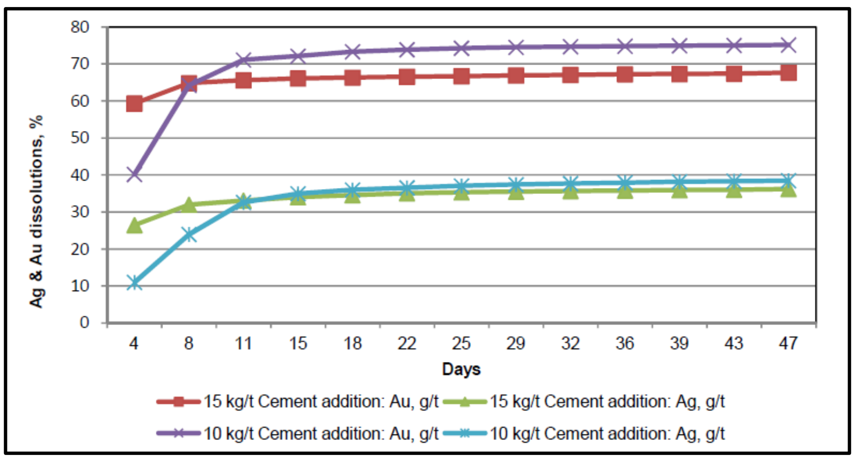


Table 13.14 Gold and Silver Extractions – Column Leach Tests

Cement Dosage (kg/t)	Grade		Extraction		Accountability (%)
	Head (g/t)	Residue (g/t)	Assayed* (%)	Calculated** (%)	
Gold					
10	7.9	2.04	74	75	104
15	7.9	2.21	73	68	85
Silver					
10	38.1	23.3	38	39	101
15	38.1	22.1	42	36	91

Notes: *based on assayed head & residue;
**based on assayed head & calculated residue

At the cement addition of 10 kg/t, approximately 75% of the gold was dissolved based on assayed head and solution assays after 47 days. The column heap leach tests indicated that this sample is amenable to heap leaching.

13.3 TEST WORK BY MCCLELLAND LABORATORIES, INC.

Starting from November 2017, McClelland conducted column leach tests on four different composite samples, identified as Composites GOS H, GOS H-CL, GOS L, Silica and Master. The composite samples were developed to represent the major lithologies identified in the oxide zone of the deposit. By conducting leach test work on samples of each different major lithology, a better understanding of the metallurgical response and metallurgical variability of each lithology could be achieved. In addition, the leach test work on two different crush sizes provided an assessment of the effect of particle size on leach response. Two particle sizes, 100% passing 25 mm and 100% passing 50 mm, were tested. Cement dosage used for these tests was 10 kg cement per one tonne mineral material. Applied sodium cyanide concentration was 1 g/L. Pregnant solution pH was in the range of 10.5 to 11.5. The key test parameters and results are shown in Table 13.15. These data are based on the interim data report, the final report is not yet available.

Table 13.15 Column Leach Test Results

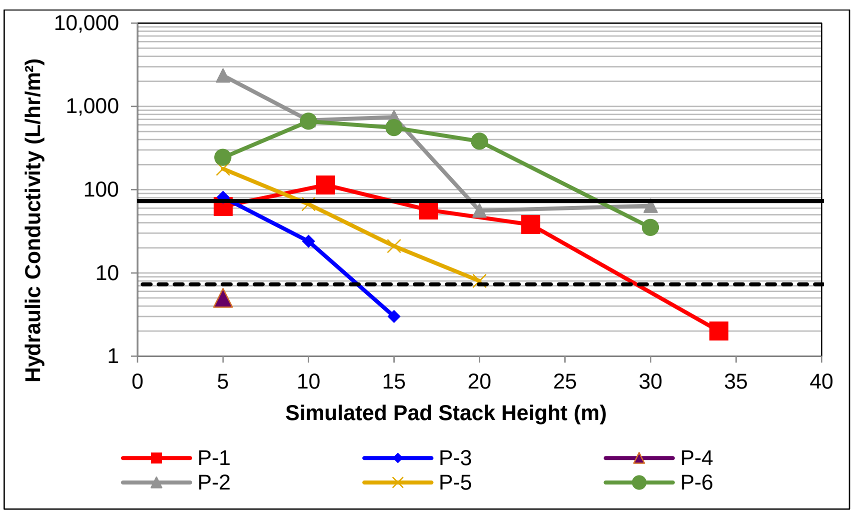
Composite	Test No.	Feed Particle Size (P100, mm)	Column Load Height (m)	Column Diameter (m)	Leach/ Rinse Time (d)	Solution Applied (Mt/Mt feed)	Calculated Head Grade		Extraction		NaCN Consumption (kg/t)
							Au (g/t)	Ag (g/t)	Au (%)	Ag (%)	
GOS H	P-1	25	1.70	0.15	97	7.31	3.39	19	50.4	10.5	1.52
GOS H	P-3	50	1.94	0.20	88	6.09	3.34	20	50.9	10.0	1.49
GOS H-CL	P-4	50	1.59	0.20	102	11.70	5.26	11	50.4	18.2	3.50
GOS L	P-2	25	2.06	0.15	97	7.09	2.87	22	69.3	9.1	1.18
GOS L	P-5	50	1.85	0.20	88	6.07	2.90	28	67.9	7.1	1.38
Silica	P-6	50	1.85	0.20	102	8.34	6.14	29	73.0	37.9	2.32
Master	P-7	50	0.97	0.20	115	14.67	4.32	21	64.6	19.0	4.08

Similar to the results of the SGS test work, gold and silver recoveries in the McClelland test work were variable for the different lithological mineralization. Overall gold and silver extractions in the McClelland column leach test work are lower, ranging from 50 to 73% for gold and 7 to 38% for silver. On average, McClelland projected that the average extractions on the master composite sample with a particle size of 100% passing 50 mm could be approximately 65% for gold and 24% for silver. The SGS column leach test on master composites achieved 68% and 75 % gold extraction (calculated). Compared to the tests conducted by SGS, the lower extractions may result from the much coarse feed particle sizes used in the McClelland test work and from the lower permeabilities achieved. McClelland indicated that the low permeability may be due to the relatively high amount of fine material present, especially in the GOS H-Cl portion of the composite sample. Hydraulic conductivity tests were conducted and potential maximum stack pad depths were simulated. The test and simulation results are shown in Table 13.16 and Figure 13.7. The permeability falls below a typical heap leach solution application rate. McClelland suggests increasing cement dosage and limiting the leach pad height to improve the pad permeability.

Table 13.16 Hydraulic Conductivity Testing Results - Column Test Residues

Sample	Test No.	Feed Particle Size (100% Passing, mm)	Fine Content (% Passing 75 µm)	Maximum Stack Depth 1 x App. Rate	Maximum Stack Depth 10 x App. Rate
GOS H	P-1	25	23	32	15
GOS H	P-3	50	20	14	6
GOS H-CL	P-4	50	39	<5	<5
GOS L	P-2	25	27	>30	20
GOS L	P-5	50	24	20	10
Silica	P-6	50	18	>30	29

Figure 13.7 Hydraulic Conductivity Simulation Results – Column Test Residues



Given the good permeabilities achieved in the SGS test work, the lower permeabilities in the McClelland test work were not anticipated, especially in light of the coarser crush size used. Further tests should be conducted to determine the optimum heap leach operating parameters.

14.0 MINERAL RESOURCE ESTIMATES

In 2013, Fladgate Exploration Consulting Corporation (Fladgate) undertook independent quality assurance and quality control studies on the drilling and assay database, and undertook independent sampling to provide confidence and support for mineral resource estimation. The Mineral Resource estimate was completed and filed with the Canadian Securities Administration (CSA) on February 14, 2014 and titled *NI 43-101 Technical Report on a Mineral Resource Estimate at the Terakimti Prospect, Harvest Property (centred at 38°21'E, 14°19'N), Tigray National Region, Ethiopia* by Dr. Sandy Archibald, P.Geo., Christopher Martin, C.Eng., and David G. Thomas, P.Geo.

In 2015, EAM requested that Fladgate update the oxide mineral resource estimate using the additional reverse circulation drilling and trench sampling information collected during 2014 and 2015. The updated Mineral Resource estimate was disclosed publicly in an EAM press release on October 27, 2015. Only the oxide Mineral Resource estimate is discussed in this report since the supergene and primary sulphide Mineral Resources are unchanged.

14.1 KEY ASSUMPTIONS AND PARAMETERS

Fladgate undertook quality assurance and quality control studies on the Mineral Resource data for the Harvest project. Fladgate concludes that the collar, downhole survey, assay and lithology data are adequate to support Mineral Resource estimation.

There is a total of 81 core drillholes, 127 reverse circulation drillholes and 41 trenches for a total of approximately 25,970 m of drilling within the Terakimti database used to support Mineral Resource estimation. The drilling database comprises 12 core drillholes (1,573 m) from the 2009-2010 due diligence drill campaign, 69 core drillholes (15,007 m) from the 2013 drill campaign and 127 reverse circulation drillholes completed during 2014 and 2015. In addition, there are 41 trenches completed during 2014 and 2015.

The database cut-off date for Mineral Resource estimate purposes was September 22, 2015.

Fladgate imported the collar, survey, lithology, alteration, and assay data into MineSight®, a commercial mining software program. Topographic contour limits were based on a surface supplied by EAM. The topography is based upon gravity geophysical survey stations and ortho-rectified stereographic images. The topography has an accuracy of ±60 cm. Fladgate checked that the drillhole collars matched the topographic surface. All data used the local grid coordinate system.

14.2 GEOLOGICAL MODELLING

Geological interpretations of the oxide, transition and supergene mineralization were completed by EAM based on lithological, mineralogical and alteration features logged in drill core, and were digitized by Fladgate to form three-dimensional solids representing the VHMS mineralization. Fladgate updated the sulphide mineralization wireframes from the 2013 mineralization wireframes using the new drill hole information. Fladgate reviewed the resulting wireframe models, minor adjustments were made to snap the wireframe boundaries to drill hole intercepts. Fladgate coded each zone separately. The zone codes are shown in Table 14.1.

Table 14.1 Terakimti Zone Domain Codes

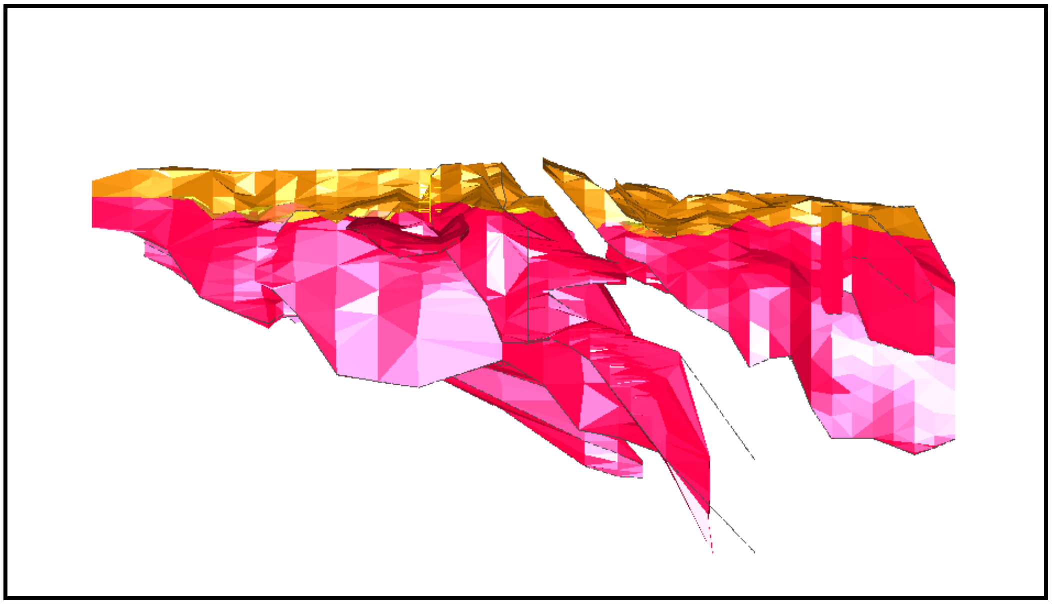
Zone	Code
C1	5
S1	10
LZn	15
L Cu	20
U1	25
N1	30
N2	35
N3	40
N4	45
N5	50

Two surfaces were constructed to represent the base of oxide mineralization and the base of the oxide transition-to-sulphide (transition) which are significant controls on the grade of gold and silver mineralization and are also significant for metallurgical processing and recovery. The surfaces were used to subdivide the VHMS mineralization into oxide, transition and sulphide domains. The geological models used to constrain mineral resource estimation are shown below in Figure 14.1. Locally there are areas of the mineralization which have a more moderate dip. Fladgate coded the more significant of these areas separately. A list of oxidation and dip subdomain codes is shown below in Table 14.2.

Table 14.2 Terakimti Oxidation and Orientation Subdomain Codes

Domains	Oxidation Code	Steep Dip Code	Moderate Dip Code
Oxide	5	10	5
Transition	10	10	5
Primary	15	10	5

Figure 14.1 Three-dimensional View of VMS Wireframes, Looking West



14.3 EXPLORATORY DATA ANALYSIS

Exploratory data analysis (EDA) comprised basic statistical evaluation of assays and 3 m composites for length, copper, gold and silver.

14.3.1 ASSAYS

Histograms and probability plots within the oxide and transition zones show some evidence of mixed populations for gold, silver and copper.

14.3.2 GRADE CAPPING/OUTLIER RESTRICTIONS

Length-weighted, normal-scale and log-scaled histograms and probability plots of the assays were used to evaluate and define grade outliers for gold, silver and copper within the oxide and transition zones.

Assay summary statistics, capping grade thresholds and the amount of metal to remove within the oxide and sulphide-transition domains are shown below in Table 14.3.

Capping was completed on the assays prior to compositing.

Table 14.3 Oxide and Sulphide-Transition Domains Summary Statistics and Capping Thresholds

	Gold (g/t)		Silver (g/t)		Copper (%)	
	Oxide	Transition	Oxide	Transition	Oxide	Transition
Number	2,397	322	2,401	322	2,401	322
Minimum	0.00	0.01	0.0	0.0	0.00	0.00
Maximum	98.70	59.10	438.0	4465.0	6.00	8.45
Average	2.09	3.12	5.4	97.0	0.09	0.16
Standard Deviation	6.50	6.53	16.1	323.6	0.16	0.72
CV	3.10	2.09	3.0	3.3	1.86	4.49
Capped Mean	2.07	3.06	5.2	66.6	0.08	0.11
Capped CV	2.99	1.99	2.20	1.42	1.13	2.91
Number Capped	7	2	2	7	10	7
Metal-to-Remove	-1.2%	-1.9%	-4.0%	-31.3%	-4.7%	-34.5%

Note: CV – coefficient of variation

14.3.3 Contact Plots

Contact plots show that there are generally sharp changes in grades from one side of the contact to the other. Therefore, the contacts are treated as hard boundaries during grade estimation.

14.3.4 Composites

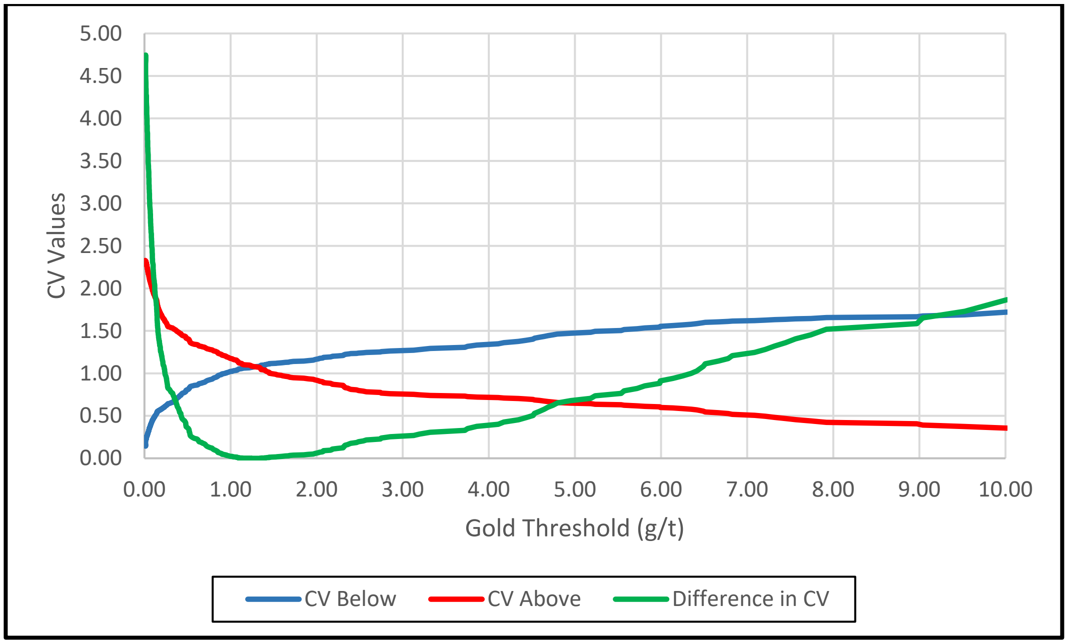
The assay intervals were regularized by compositing the drill hole data into 3 m lengths using the mineralization zone and oxidation domain boundaries to break the composites. A composite length of 3 m was chosen to reflect the width of a selective mining unit for a small-scale open pit operation. Capped and uncapped composites were calculated and the means of the composites were checked against the mean of the assays. No difference was observed in the means. Within the oxide and transition domains, the CV values of the gold composites are moderate to high (1.7 to 2.3). The CV values of the silver composites are low to moderate (1.2 to 1.8). The CV values indicate that further domaining is warranted for gold and silver in oxides and for gold within the transition zone.

14.3.5 Indicator Probabilistic Models

Gold Model

As a result of the high gold composite CV values within the oxide and transition domains identified by EDA, Fladgate created a probabilistic indicator model within the oxide and transition domains using a nominal threshold of 1.3 g/t gold. The threshold was chosen to minimize the CV of the composites above and below the threshold. The threshold was selected by ranking the composites in order of increasing grade and calculating a cumulative CV. A plot showing the difference between the CV values above and below threshold, the CV of composites above threshold and CV of the composites below threshold against grade is shown in Figure 14.2.

Figure 14.2 Ranked Cumulative CV Plot



The gold indicator model was validated visually on section and in plan. The model adequately reflects the input composite data. Further validation was completed by comparison with an nearest neighbour (NN) model (global means and swath plots).

Fladgate used the gold indicator probabilities in the blocks to control the smoothing of the model during the gold estimation process.

Silver Model

As a result of the multiple silver composite populations and high CV within the oxide domain identified by EDA, Fladgate created a probabilistic indicator model within the oxide domain using a nominal threshold of 12.5 g/t silver. The threshold was chosen based on an inflection point observed in a log-probability plot of the composites.

The gold indicator model was validated visually on section and in plan. The model adequately reflects the input composite data. Further validation was completed by comparison with a NN model (global means and swath plots).

Fladgate used the silver indicator probabilities in the blocks to control the composite and block matching during the silver estimation process. An indicator threshold of 0.41 was chosen to give an unbiased volume estimate of the number of blocks above the 12.5 g/t silver threshold (by comparison with the NN model).

14.3.6 VARIOGRAPHY

Fladgate constructed down-hole and directional correlograms for the gold grade and silver grade variables within the mineralized subdomains. Indicator correlograms were modelled for gold and silver, the indicator correlograms were used to interpolate the respective indicator probabilities for each respective element.

Fladgate plotted variogram maps of the data and checked that the selected anisotropy directions matched by plotting and comparing variogram maps of the variogram models.

Fladgate used a single spherical model and a nested exponential model and a nugget effect to fit the experimental correlograms. Table 14.4 shows the correlogram models.

The same correlogram model was applied to the areas of the mineralized domains with a more moderate dip angle. The final rotation angle was changed by 30° to rotate the variogram into the plane of the mineralization.

Table 14.4 Grade and Indicator Variogram Models

Grade Element	Nugget Effect	Sill		Structure Type		Ranges First Structure			Ranges Second Structure			Rotations		
		1st Structure	2nd Structure	First	Second	Y	X	Z	Y	X	Z	Z	X	Y
Au	0.45	0.47	0.08	Spherical	Exponential	30	10	10	66	25	20	10	0	0
Ag	0.18	0.50	0.32	Spherical	Exponential	40	10	10	200	30	20	10	0	20
Cu	0.05	0.34	0.61	Spherical	Exponential	5	5	5	25	25	25	0	0	0
Au Indicator	0.30	0.35	0.35	Spherical	Exponential	15	10	10	40	30	15	15	0	30
Ag Indicator	0.34	0.40	0.26	Spherical	Exponential	50	10	10	200	50	20	0	0	0

14.4 Grade Estimation

The block model consists of regular blocks (5 m along strike by 5 m across strike by 5 m vertically). The block size was chosen such that geological contacts are reasonably well reflected and to support an open pit mining scenario.

Fladgate checked that the volume resulting from coding of blocks from the wireframes matched the original volume of the wireframes.

Fladgate used an ordinary kriging grade interpolation method to estimate block grades.

For passes one and two, a minimum of three and a maximum of 16 composites were used for grade interpolations. In all passes, a maximum of two composites per hole was used to show the search distances and search ellipse orientations for the estimation domains. The orientation of the oxide search ellipse was adjusted to a flatter dip (by using an additional 30° X-axis rotation) where the wireframe also had a flatter dip orientation.

Gold grade estimation was completed for lower-grade mineralization and higher-grade mineralization based on the respective indicator codes of the composites. Silver grade estimation used a composite and block sharing scheme based on the respective indicator subdomains.

Grade estimation used a composite and block matching scheme based on the oxide, supergene and primary mineralization type domain codes and based on the respective indicator subdomain codes of the indicator models. Grade interpolation in the primary and supergene domains used matching of blocks and composites by the zone codes incorporated into the blocks.

The orientation of the search ellipse was adjusted to a dip 20° flatter where the wireframe also had a flatter orientation.

14.4.1 Model Re-blocking

EAM anticipate mining of the oxide and transition zone gold and silver mineralization over a 2 to 3-year period, Fladgate therefore re-blocked the model to a 2.5 m (along easting) by 5 m (along northing) by 5 m (bench height) block size.

14.5 Density Assignment

14.5.1 Mineralized Zones

EAM collected 79 SG determinations from drill core samples and 53 SG determinations from trench samples from gossan material within the mineralized zones. The determinations were performed by EAM personnel using unsealed immersion techniques to measure the weight of each sample in air and in water.

Summary SG statistics are shown in Table 14.5 and Table 14.6.

Table 14.5 Summary Specific Gravity Statistics, Combined Gossan Data

Rock Type	Number	Minimum (g/cm^3)	Maximum (g/cm^3)	Average (g/cm^3)	Standard Deviation	90% Confidence Interval in Mean	Confidence Interval % of Mean	Difference with DH (%)
GOS	3	1.90	2.39	2.06	0.28	0.27	13.0	3
GOSF	22	2.00	2.79	2.45	0.25	0.09	3.5	22
GOSH	55	1.82	2.65	2.21	0.21	0.05	2.1	55
GOSL	27	1.67	2.51	2.17	0.23	0.07	3.4	27
GOSM	11	2.20	2.70	2.44	0.15	0.08	3.1	11
GOSS	5	2.03	2.30	2.21	0.10	0.08	3.5	5
GOSV	9	2.20	2.70	2.46	0.19	0.10	4.3	9
All	132	1.67	2.79	2.28	0.24	0.03	1.5	132

A NN model of lithology groups within the oxide mineralization was used to estimate the proportions of each rock type. The proportions were used to estimate an average SG (2.3 g/cm^3) for the oxides weighted by the proportion of rock types.

EAM collected 59 SG measurements from drill core samples in the transition zone to estimate an average SG of 2.72 g/cm^3. Anomalously low and high SG measurements were removed to provide a more robust estimate of the mean SG. Summary statistics of the SG measurements in the transition zone are shown in Table 14.6.

Table 14.6 Transition Domain SG Suummary Statistics

Number	Minimum (g/cm^3)	Maximum (g/cm^3)	Uncapped Mean (g/cm^3)	Capped Average (g/cm^3)	Standard Deviation	90% Confidence Interval	Confidence Interval % of Mean
59	1.52	6.04	2.76	2.72	0.29	0.06	2.3

14.5.2 WASTE DENSITY ASSIGNMENT

Fladgate grouped together measurements of oxide waste SG into two groups of rock types; intrusives which cross-cut mineralization and the volcano-sedimentary package, which hosts mineralization at Terakimti. Summary statistics for the intrusives and volcano-sedimentary package are shown in below in Table 14.7.

Table 14.7 Waste SG Means

Lithology	Number of Measurements	Average SG (g/cm^3)	Minimum SG (g/cm^3)	Maximum SG (g/cm^3)
Quartz Porphyry	45	2.29	1.72	3.27
Volcano-Sedimentary	366	2.44	1.74	3.04

14.6 BLOCK MODEL VALIDATION

Fladgate validated the Terakimti block model to ensure appropriate honoring of the input data. NN grade models were created to validate the OK grade models.

The validation comprised:

- A comparison between the OK and NN estimates was completed to check for global bias in the grade estimates. Differences were within acceptable levels (less than 10%)
- Swath plot validation compared average grades from OK and NN models along different directions. Except in areas where there is currently limited drilling, the swath plots indicated good agreement for all variables.
- Detailed visual inspection of block grade versus composited data in section and plan view. The visual inspection of block grade versus composited data showed a good reproduction of the data by the model.
- A check on grade smoothing (model selectivity) for potential open pit mining using a global change-of-support correction (a discrete Gaussian model) to the NN model. The check was completed for gold in the oxide and transition domains for gold. The results show that the amount of smoothing is acceptable for a block size of 2.5 m by 5 m by 5 m around the cut-off grades of interest and are generally less than 5%.

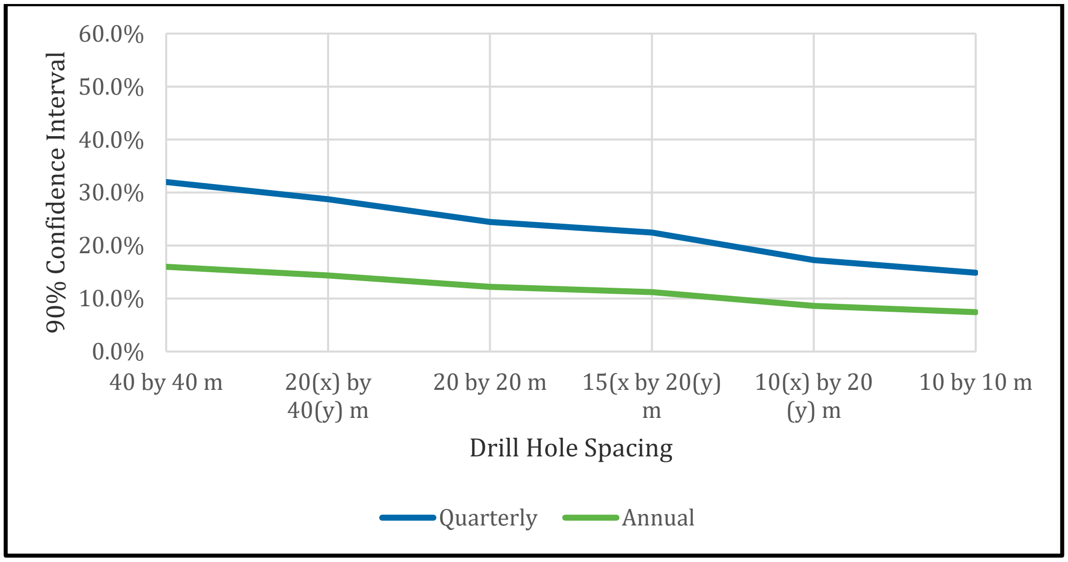
Fladgate evaluated the impact of capping by estimating uncapped and capped grade models. Generally, the amounts of metal removed by capping in the models are consistent with the amounts calculated during the grade capping study on the composites. The amount of metal removed by capping is calculated by the following formula:

$$\%\ Metal = \frac{(Mean\ Uncapped - Mean\ Capped)}{Mean\ Uncapped}$$

14.7 CLASSIFICATION OF MINERAL RESOURCES

Fladgate conducted an analysis of confidence limits using quarterly panels of production for a 1,100 t/d open pit mine operation. The accuracy of grade estimates was then scaled to annual production. Accuracy of ±15% or better at a 90% confidence limit on quarterly production was used as the criteria to select a drillhole spacing to be used to classify Measured Mineral Resources. The same accuracy and confidence limit on annual production was used to select a drillhole spacing to classify Indicated Mineral Resources. The results (Figure 14.3) show that a drillhole spacing of 10 m (along the easting) by 20 m (along the northing) is sufficient to classify Measured and a spacing of 20 m (along the easting) by 40 m (along the northing) is sufficient to classify Indicated Mineral Resources.

Figure 14.3 Drillhole Spacing Study Results, Gold Grades



Fladgate also completed an analysis of the classification categories using conditional simulation of grades within the oxide domain. Fladgate selected a confidence limit of ±20% or better at an 80% confidence limit to select blocks as potential candidates for the Measured category. The results of the conditional simulation identified similar areas of the mineralized zones as those identified by the drillhole spacing study.

The two Mineral Resource classification methodologies were merged together to produce the final set of blocks which could be classified to the Measured category. Fladgate manually modified the classification to remove isolated areas of Measured category blocks. Fladgate classified blocks with a minimum of two holes falling within 25 m and the closest hole within 25 m (i.e. with a 20 m by 40 m spacing) to the Indicated category.

Fladgate classified blocks all other blocks falling within the mineralization wireframes into the Inferred Mineral Resource category. The mineralization solids represent the limit at which grade continuity can reasonably be assumed while permitting a reasonable local estimate of grades (as demonstrated by model validation).

Fladgate is of the opinion that the geological model, data quality and geological continuity are sufficiently well known to allow classification of Measured and Indicated Mineral Resources. However, the metallurgical characteristics of the mineralization are insufficiently well known to permit the classification of Measured Mineral Resources at this time. Fladgate therefore only classified Indicated and Inferred Mineral Resources.

14.8 REASONABLE PROSPECTS OF ECONOMIC EXTRACTION

Fladgate assessed the classified blocks of oxide and transition mineralization for reasonable prospects of economic extraction by applying preliminary economics for potential open pit mining methods. Preliminary metallurgical test-work has been completed for the supergene and primary VHMS mineralization. Additional metallurgical test-work has been completed for the oxide and transition mineralization to support upgrades to the oxide mineral resource classification; however, there is still insufficient metallurgical information to support a Measured Mineral Resource category.

For the purpose, Fladgate used input process and operating costs, metal prices, metallurgical recovery to estimate a cut-off grade. The updated oxide and transition mineral resource is not constrained within a pit shell since the mineralization is less than 100 m below surface and is therefore considered to be amenable to open pit extraction.

The assessment does not represent an economic analysis of the deposit, but was used to determine reasonable economic assumptions for the purpose of estimating the mineral resource. The assumed long-term metal prices used by Fladgate for reporting mineral resources are shown in Table 14.8. Although the long-term metal prices are optimistic, the prices are suitable for the purposes of a Mineral Resource estimate.

Table 14.8 Fladgate Long-term Metal Price Assumptions

Metal	Price (US$/tr oz)
Gold	1,300
Silver	17.50

14.9 MARGINAL CUT-OFF GRADE CALCULATION

Fladgate estimated a gold equivalent cut-off of 0.52 g/t for oxide material based on the total costs shown in. The gold equivalent formula used is:

Gold equivalent = Gold Grade + Silver Grade x (((Silver Price/31.103477) x (Silver Recovery)) / (Gold Price / 31.103477) x (Gold Recovery)

The marginal cut-off is based on the generally accepted practice that a decision is made at the pit rim if mined material above the marginal cut-off grade will lose less money if it is sent to the mill rather than if it is sent to the waste dump. It is considered for further processing if it contains a value that is greater than the costs to process it. The assumed metallurgical recoveries are shown in Table 14.10.

Based upon the marginal gold equivalent cut-off grade, Fladgate have chosen a cut-off grade of 0.5 g/t gold equivalent for reporting the oxide Mineral Resources potentially amenable to an open pit mining method.

Table 14.9 Mining Costs and Ore-Based Costs Used for NSR Calculations

Mining Costs	Unit	Value
Waste Mining Reference Cost	US$/t mined	1.75
Total Reference Mining Costs	US$/t mined	1.75
Ore Based Costs		
Process Cost (Heap Leach)	US$/t ore	9.74
Process Cost (Floatation)	US$/t ore	17.9
G&A Cost (Heap Leach)	US$/t ore	6.0
G&A (Floatation)	US$/t ore	6.0
Total Ore Based Costs Oxide	US$/t milled	15.74
Total Ore Based Costs Supergene/Primary	US$/t milled	23.9

Table 14.10 Gold Metallurgical Recovery for Cut-Off Estimation

	Metallurgical Recoveries (%)			
	Gold	Silver	Copper	Zinc
Oxide	73.1	50	N/A	N/A
Transition	36.0	78	N/A	N/A

14.10 MINERAL RESOURCE STATEMENT

Mineral Resources for the Project were classified under the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves by application of a cut-off grade that incorporated mining and metallurgical recovery parameters. Mineral Resources are reported above a cut-off grade which uses commodity prices, metallurgical recoveries and operating costs.

Mineral Resources are tabulated in Table 14.11. The QP for the Mineral Resource estimate is David G. Thomas, P.Geo. Mineral Resources are reported using the long-term metal prices shown in Table 14.8 and have an effective date of October 18, 2015.

Table 14.11 Terakimti Oxide Mineral Resource Estimate at a 0.5 g/t Gold Equivalent Cut-off, David Thomas, P. Geo. (Effective Date: October 18, 2015)

Classification	Tonnes (t)	Gold Equivalent (g/t)	Gold Grade (g/t)	Silver Grade (g/t)	Copper Grade (%)	Gold Metal (tr oz)	Silver Metal (tr oz)
Indicated	1,110,000	3.41	3.20	23.6	0.08	114,000	841,000
Inferred	15,000	2.06	1.94	13.5	0.04	1,000	7,000

Notes: Fladgate undertook data verification, and reviewed EAM's quality assurance and quality control programs on the Mineral Resource data. Fladgate concluded that the collar, survey, assay, and lithology data were adequate to support Mineral Resources estimation.
Domains were modelled in 3D to separate oxide, transition, supergene and primary sulphide rock types from surrounding waste rock. The domains conformed to lithological contacts logged in diamond drill core. Sub-domaining was further warranted to separate different grade populations and zones with differing strike and dip orientation within domains.
Raw drillhole assays were composited to 3 m lengths broken at domain boundaries.
High-grade assays were capped prior to compositing. Capping thresholds were assessed within each domain independently.
Block grades for copper, gold, and silver were estimated from the composites using Ordinary Kriging and into 2.5 m by 5.0 m by 2.5 m blocks coded by domain. The block model was re-blocked to a selective mining unit size of 2.5 m by 5.0 m by 5.0 m blocks for reporting of the Mineral Resource.
An average dry bulk density of the oxide zone was derived from SG measurements on drill core and trench samples. Fladgate weighted the SG measurements by the proportion of each rock type within the oxide mineralization.
Blocks were classified as Measured, Indicated and Inferred in accordance with CIM Definition Standards.
Gold equivalent was estimated using undiluted grades, metal prices and heap leach process recoveries. The formula used is: gold equivalent = gold + (((silver price/31.103477) x (silver recovery))/(gold price/31.103477) x (gold recovery)
Metal Prices used for gold and silver were US$1,300/tr oz, and US$17.50/tr oz respectively.
Metallurgical recoveries, supported by metallurgical test work were applied as follows: recoveries of 73.1% were applied for gold and 50.0% for silver. Copper and zinc are not recovered during the oxide phase and therefore are not considered a part of the oxide Mineral Resources.
The contained metal figures shown are in situ. No assurance can be given that the estimated quantities will be produced. All figures have been rounded to reflect accuracy and to comply with securities regulatory requirements. Summations within the tables may not agree due to rounding.

Table 14.12 Terakimti Sulphide Mineral Resource Estimate David Thomas, P. Geo. (Effective Date: January 17, 2014)

Classification	Ore Type	NSR Cut-off ($/t)	Tonnes ('000s)	Cu (%)	Zn (%)	Au (g/t)	Ag (g/t)	Contained Metal Cu ('000 lb)	Contained Metal Zn ('000 lb)	Contained Metal Au ('000 tr oz)	Contained Metal Ag ('000 tr oz)
Indicated	Sulphide	23.9	1,841	2.20	1.65	1.06	17.5	89,477	66,871	63	1,033
	Subtotal Indicated	-	-	-	-	-	-	89,477	66,871	86	1,130
Inferred	Sulphide	23.9	2,583	1.09	1.42	0.96	20.6	62,187	77,101	80	1,712
	Underground Primary	63.9	939	0.69	2.92	0.84	15.2	14,198	60,358	25	459
	Subtotal Inferred	-	-	-	-	-	-	76,385	137,459	166	2,264

Notes: Fladgate undertook data verification and reviewed the Harvest Project's QA/QC programs on the mineral resources data. Fladgate concluded that the collar, survey, assay, and lithology data were adequate to support Mineral Resources Estimation.

Domains were modelled in 3D to separate oxide, supergene and primary sulphide rock types from surrounding waste rock. The domains conformed to lithological contacts logged in diamond drill core. Sub-domaining was further warranted to separate different grade populations and zones with differing strike and dip orientation within domains.

Raw drillhole assays were composited to 5 m lengths broken at domain boundaries.

High-grade assays were capped prior to compositing. Capping thresholds were assessed within each domain independently.

Block grades for copper, zinc, gold, and silver and lead were estimated from the composites using a combination of OK and inverse distance weighted to the third power (ID^3) into 5 x 5 x 5 m blocks coded by domain.

Dry bulk density of the oxide, supergene and primary sulphide was estimated by ID^3 interpolation of SG measurements.

Blocks were classified as indicated and inferred in accordance with CIM Definition Standards.

Net smelter return (NSR) was estimated using undiluted grades, metal prices, recoveries, smelter treatment and refining costs.

Metal Prices used for copper, zinc, gold and silver were US$3.50/lb, US$0.9/lb, US$1,400/tr oz, and US$25/tr oz respectively.

Metallurgical recoveries, supported by metallurgical test work were applied as follows:

- Supergene zone: recoveries to copper concentrate of 87%, 36%, and 78% were applied for copper, gold and silver. Zero recovery of zinc from the supergene zone has been assumed. The supergene zinc metal content has not been included in the mineral resource tabulation.
- Primary zone: recoveries to copper concentrate of 89%, 45%, and 39%, were applied for copper, gold, and silver respectively. Recoveries to zinc concentrate of 85% and 10% were applied for zinc and silver.

A Lerchs-Grossman pit shell was generated from the NSR and using open pit mining costs of US$1.75/t. The total ore based costs (process and G&A) are US$25.9/t for oxide, and US$23.9/t for the supergene and primary rock types. A constant pit slope of 45° was used in the pit optimization.

Open Pit Mineral Resources were reported within the Lerchs-Grossman pit shell above an NSR cut-off equivalent to the total ore based costs stated above.

Underground Mineral Resources were reported within a grade shell generated at an NSR cut-off of US$63.90/t, assuming a US$40.00/t underground mining cost in addition to the ore based costs stated above. Isolated blocks were removed prior to tabulation.

The contained metal figures shown are in situ. No assurance can be given that the estimated quantities will be produced. All figures have been rounded to reflect accuracy and to comply with securities regulatory requirements. Summations within the tables may not agree due to rounding. The sulphide summation for contained zinc does not agree due to exclusion from the mineral resource of the contained zinc metal within the supergene zone.

Fladgate reviewed the impact of the reverse circulation drilling completed subsequent to the sulphide Mineral Resource estimate. The additional drilling did not result in any significant change to the volume or grade of the sulphide mineralization.

14.11 SENSITIVITY OF THE OXIDE MINERAL RESOURCE TO CUT-OFF GRADES

The sensitivity of the oxide mineral resource estimate to changes in cut-off grade is shown in Table 14.13. The tabulations above varying cut-off grades show that the mineral resource estimate is not sensitive to changes in cut-off grade.

Table 14.13 Terakimti Oxide Mineral Resource Sensitivity to Cut-off Grade

Gold Equivalent Cut-off (g/t)	Tonnes (t)	Gold Equivalent (g/t)	Gold Grade (g/t)	Silver Grade (g/t)	Copper Grade (%)	Gold Metal (tr oz)	Silver Metal (tr oz)
Indicated							
0.10	1,485,000	2.62	2.45	18.56	0.08	117,000	887,000
0.20	1,420,000	2.75	2.57	19.37	0.08	117,000	882,000
0.30	1,305,000	2.96	2.77	20.72	0.08	116,000	869,000
0.40	1,195,000	3.21	3.01	22.34	0.08	115,000	856,000
0.50	1,110,000	3.41	3.20	23.60	0.08	114,000	841,000
0.60	1,070,000	3.53	3.31	24.25	0.08	113,000	832,000
0.70	1,050,000	3.57	3.35	24.39	0.08	113,000	824,000
0.80	1,030,000	3.62	3.40	24.56	0.08	112,000	814,000
0.90	1,010,000	3.68	3.45	24.80	0.08	112,000	806,000
1.00	990,000	3.74	3.51	25.17	0.08	111,000	800,000
Inferred							
0.10	40,000	1.00	0.93	7.6	0.07	1,000	10,000
0.20	40,000	1.01	0.94	7.6	0.07	1,000	10,000
0.30	25,000	1.60	1.49	11.2	0.06	1,000	9,000
0.40	15,000	2.01	1.88	13.4	0.04	1,000	7,000
0.50	15,000	2.06	1.94	13.5	0.04	1,000	7,000
0.60	15,000	2.44	2.31	13.2	0.03	1,000	7,000
0.70	15,000	2.44	2.31	13.2	0.03	1,000	7,000
0.80	15,000	2.44	2.31	13.2	0.03	1,000	7,000
0.90	10,000	2.65	2.55	11.0	0.03	1,000	7,000
1.00	10,000	2.69	2.59	10.8	0.03	1,000	5,000

14.12 FACTORS THAT MAY AFFECT THE MINERAL RESOURCE ESTIMATE

Areas of uncertainty that may materially impact the Mineral Resource estimates include:

- long-term commodity price assumptions
- long-term exchange rate assumptions
- mineralized materal-based cost assumptions used
- metal recovery assumptions used

- changes to the tonnage and grade estimates as a result of new assay and bulk density information, in particular oxide copper assays
- future tonnage and grade estimates may vary significantly as more drilling is completed
- changes to the metallurgical recovery assumptions as a result of new metallurgical test work
- any changes to the slope angle of the pit wall as a result of geotechnical information would affect the pit shell used to constrain the Mineral Resources.

14.13 COMMENTS ON SECTION 14.0

The QP is of the opinion that the Mineral Resources for the Harvest Project, which have been estimated using core drilling, have been performed to industry practices, and conform to the requirements of CIM Definition Standards (2014).

15.0 MINERAL RESERVE ESTIMATES

A Mineral Reserve is the economically mineable part of a Measured or Indicated Mineral Resource and has not been estimated for the Terakimti oxide deposit as part of this PEA.

16.0 MINING METHODS

16.1 INTRODUCTION

An open pit mine plan has been completed for the Terakimti Oxide Project. An overall 3D view of the planned open pit is shown in Figure 16.1. Mine design and optimisation has been performed using GEOVIA GEMS™ and Whittle™ software.

The PEA open pit contains the following oxide resources

- 1,086 kt of potential mill feed
- 4,093 kt of waste rock (including mineralized material below cut-off grade)
- 110,000 tr oz of gold
- 799,000 tr oz of silver.

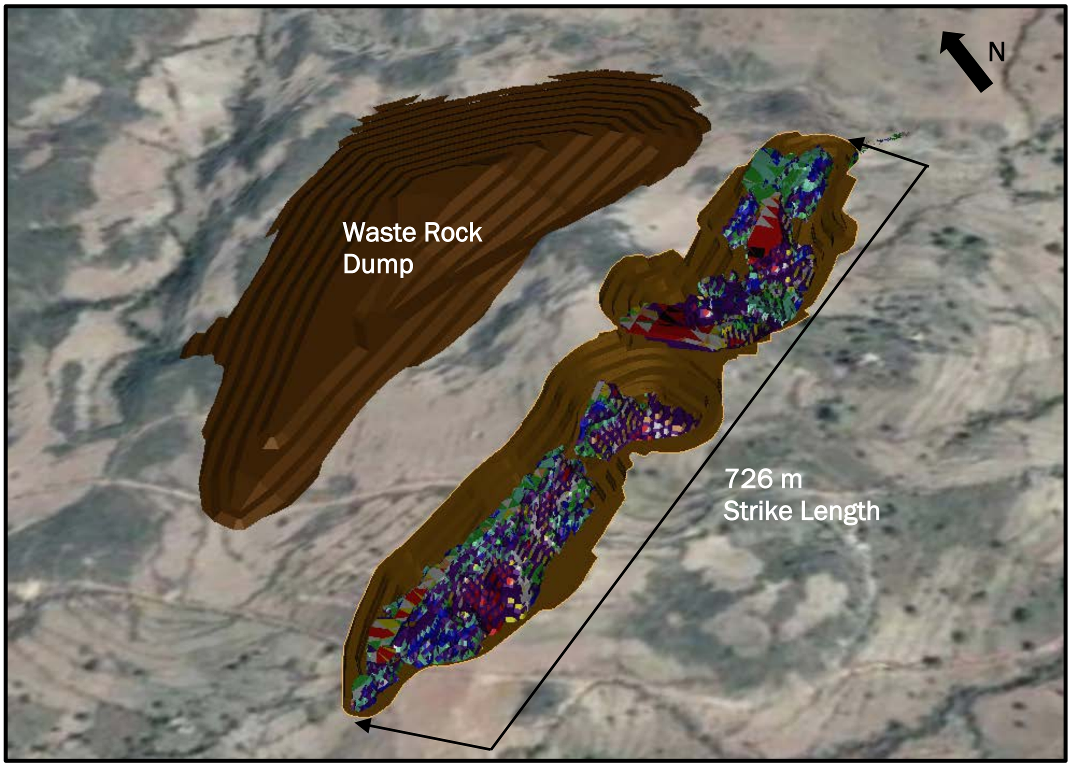
The breakdown of Mineral Resource categories included in the mine plan is shown in Table 16.1. Only 0.4% of the Mineral Resources included in the mine plan are Inferred Resources.

Table 16.1 Breakdown of Open Pit Tonnage into Mineral Resource Categories

Resource Category	Tonnes (kt)	Percentage (%)
Inferred	4	0.4
Indicated	840	77.3
Measured	242	22.3
Total	**1,085**	**100.0**

Note: Totals may not match due to rounding.

Figure 16.1 Terakimti Oxide Open Pit



Mining has been planned to meet a design heap leach throughput of 715 t/d, averaging approximately 690 t/d of leach feed for a period of four years and four months.

Total waste mined per day will average approximately 2,580 t/d, for a total mining rate of approximately 3,270 t/d over the LOM. The LOM stripping ratio is approximately 3.74:1 (waste to leach feed). Total material mined will average approximately 250,000 t/a of mineralized leachable material plus 944,000 t/a of waste material.

The pit shape follows the outcropping mineralised zone in a northeast-southwest orientation. The final pit designed for the PEA has three access areas, two on the eastern side of the pit as the main access from the process facility and one on the western side of the pit, which will be used for delivering waste to the WRD.

The Mineral Resource will be ripped where possible and with blasting expected to be required beyond a shallow free digging depth.

Hydraulic excavators (backhoe) and wheeled loaders will be used to load ore into trucks. The trucks will haul the material to a crusher on the south-east side of the pit. After crushing, the material will be loaded into a haul truck and transported to the heap leach facility.

Waste rock will be ripped where possible or blasted. Broken waste rock will be loaded into haul trucks and transported to an engineered WRD structure located approximately 120 m northwest of the pit as shown in Figure 16.1.

16.2 OPEN PIT OPTIMIZATION

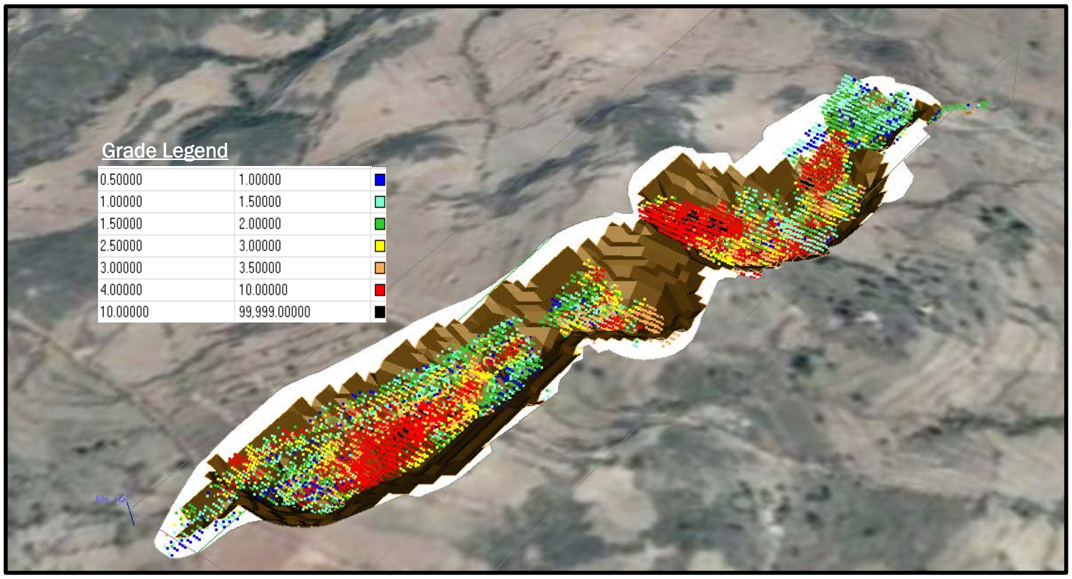
Open pit optimization was completed prior to developing an open pit mine design. The optimization used the Lerchs-Grossman algorithm in Whittle™ software using Measured, Indicated and Inferred Mineral Resources. The parameters shown in Figure 16.2 were used in the pit optimisation.

Table 16.2 Open Pit Mine Design Optimization Parameters

Item	Units	Value
Block Dimension X	m	7.5
Block Dimension Y	m	5
Block Dimension Z	m	5
Maximum Slope Above 1,765 m Elevation	degrees	35
Maximum Slope Angle (north side of pit)	degrees	50
Minimum Slope Angle (east side of pit)	degrees	45
Minimum Slope Angle (south side of pit)	degrees	50
Minimum Slope Angle (west side of pit)	degrees	55
Mining Cost	US$/t	4
Mining Recovery	%	98
Mining Dilution	%	8
Processing + G&A Costs	$/t	21
Gold Recovery from Heap Leaching	%	72
Silver Recovery from Heap Leaching	%	40
Gold Price Less Selling Costs	US$/tr oz	1,211
Silver Price Less Selling Costs	US$/tr oz	15.36

The resulting optimised pit was produced using Whittle™ software based on the parameters above is shown in Figure 16.2.

Figure 16.2 Whittle™ Optimisation Result



16.3 OPEN PIT OPTIMIZATION RESULTS

The results of the pit optimization exercise are presented in Figure 16.3. Pit shell no. 42 was selected as the basis for the engineered pit design and mine planning moving forward. Pit shell no. 30 was included as a push back for scheduling. The results of pit shell no. 42 are shown in Table 16.3

Figure 16.3 Pit Selection Chart for the Terakimti Oxide Zone; Pit No. 42, the Selected Pit, is Shown in Red

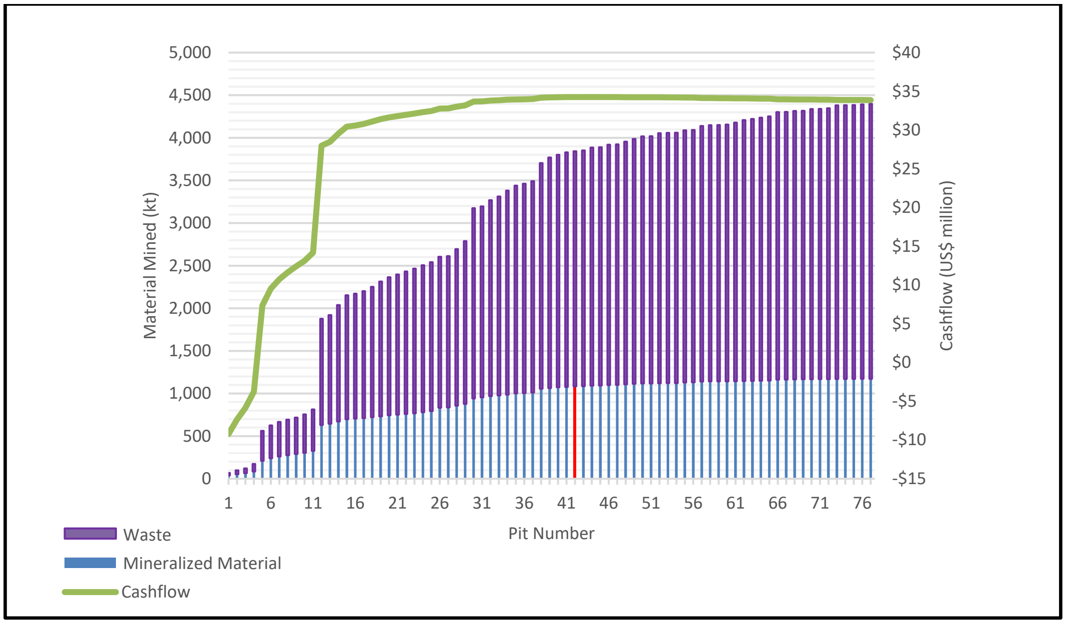


Table 16.3 Tonnes and Grade in Selected Whittle™ Shell

Item	Units	Value
Mineralized Material	t	1,084,773
Diluted Gold Grade	g/t	3.077
Contained Gold	tr oz	107,297
Diluted Silver Grade	g/t	21.2
Contained Silver	tr oz	740,427
Waste rock	t	2,757,576
Strip Ratio	degrees	2.54

16.4 MINE DESIGN

This PEA considers open pit mining of the Terakimti deposit oxide zone. Mining will commence with stripping and stockpiling of topsoil and overburden, which will subsequently be used for rehabilitation of the mine site after mining, is complete.

After stripping of topsoil and overburden, an initial zone of softer rock is expected to be amenable to free digging or ripping. Currently it is expected that as the pit deepens drill and blast would be required. Site geologists and mine planning personnel will demarcate mineralized material prior to excavation. Waste rock will be stored at the WRD. Heap leach feed will be transported to the crusher for sizing prior to transport to the heap leach pad.

The optimised pit shell shown in Figure 16.1 has been subjected to pit design work, which includes creation of benched pit walls and inclusion of pit haul roads. The specifications of the pit design are shown in Table 16.4.

Table 16.4 Design Parameters for the Terakimti Open Pit

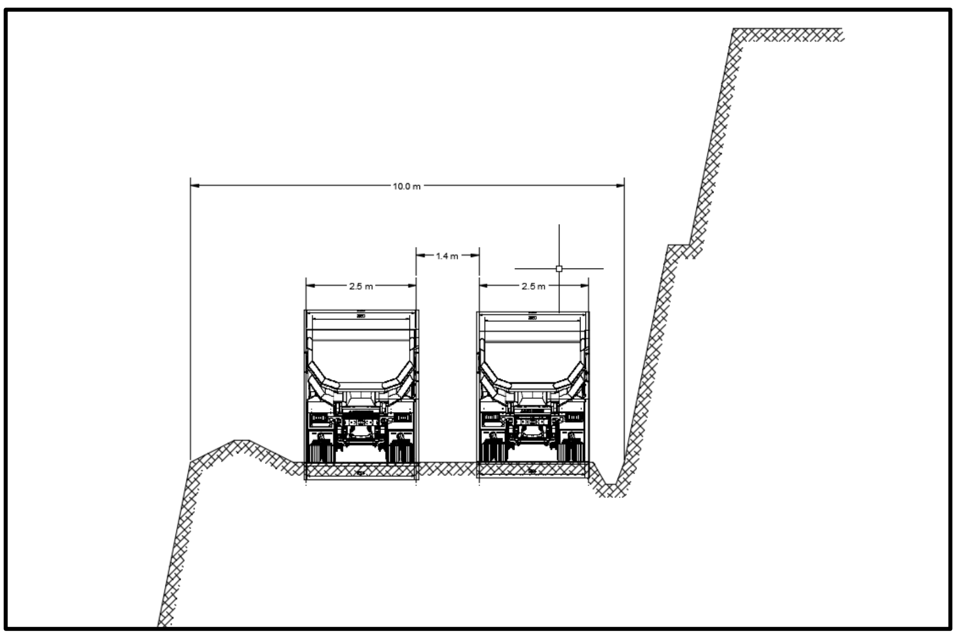
Item	Unit	Value
Pit Walls		
Bench Height	m	5
Bench Stack Height	m	10
Bench Widths	m	7
Bench Face Angle	degrees	80
Resulting Pit Slope Angle (maximum)	degrees	45
Hal Road		
Width of Main Haul Road (minimum)	m	10
Maximum Grade of Main Haul Road	%	10
Width of Slots or Bench Access Roads (minimum)	m	9
Maximum Grade of Slots or Bench Access Roads	%	15

The resulting pit has a maximum projected depth of 58 m and is shown in Figure 16.1.

16.4.1 Ramp Design

Ramp width for the PEA was based on a Volvo FMX tipper with a width of 2.5 m. Tetra Tech applied a multiplier of 3.5 to obtain a road width of 9 m, but elected to use 10 m to allow for safety berms and a toe drain (Figure 16.4).

Figure 16.4 Main Ramp Section Showing Two-way Traffic, Safety Berm and Bench Slopes



16.5 MINING OPERATIONS

Mining will be conducted on 5 m benches and will incorporate the concept of staged push backs to minimize and delay increases in waste stripping until required.

Mine planning personnel will plan the mining tactics, including ensuring the timely pre-stripping of waste material to expose ore. Mining will be planned to achieve the average of 250,000 t/a of heap leach material. However, it is expected that daily tonnage from the mine will fluctuate and as such a stockpile may be used to ensure consistent feed to the crusher and subsequent heap leach during operations.

16.5.1 DRILLING AND BLASTING

Drilling and blasting activities will be under the direct supervision of a certified blasting ticket holder.

Drilling will be done using track mounted percussion drill rigs capable of drilling hole diameters of up to 100 mm. Holes will be charged with ammonium nitrate and fuel oil (ANFO) and/or packaged emulsion explosives. Detonation of explosive will be achieved using pentolite boosters and non-electric downhole and hole trunk line connectors. Primary blast ignition will be done using a single electric detonator.

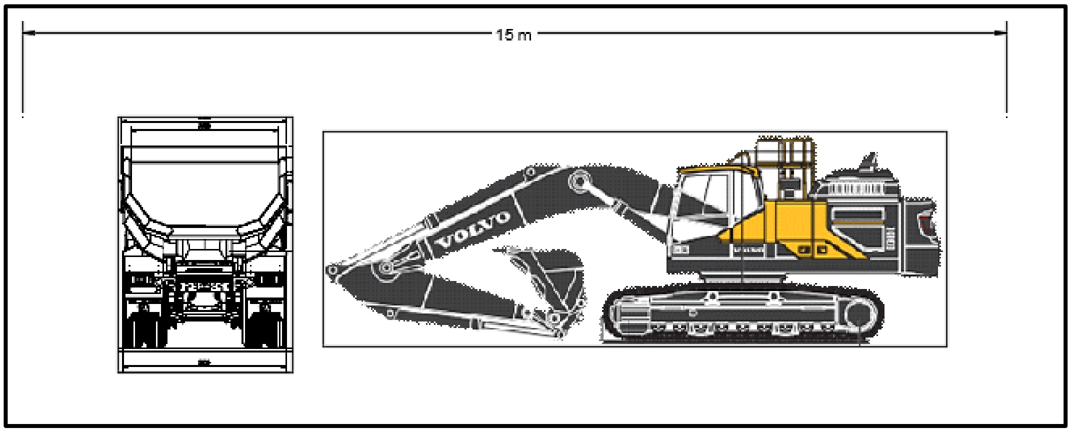
Explosives will be stored in purpose built magazines located at a safe distance from all project facilities. The explosives magazines will be secure structures with restricted access. Explosives will be withdrawn only as required for the next planned blast and only in the amount required for the next blast. Inventories of explosives will be strictly monitored.

Explosives for the project will ideally be sourced from established and recognized local suppliers.

16.5.2 LOADING AND HAULING

For this PEA, loading was assumed to be done by a crawler excavator with support from a wheeled loader. The minimum mining width has been included in the design as 15 m based on selected equipment operating widths (Figure 16.5)

Figure 16.5 Selected Truck and Shovel Mining Operating Width



16.5.3 MINE EQUIPMENT REQUIREMENTS

Mining equipment requirements have been evaluated to be in line with the production schedule. Table 16.5 shows the mining equipment list.

Table 16.5 Mining Equipment Selected for the Harvest Project

Equipment Type	Usage	Size	Engine (kW)	Number Required Over LOM
Trucks	Hauling ore and waste rock from the pit, and hauling ore from crusher to the heap leach	16 m^3	304	4
Excavator	Loading ore and waste rock in the pit	2.5 m^3 bucket	206	1
FEL	Loading crusher from ROM stockpile and loading trucks from crusher stockpile	3.5 m^3	97	2
Drill	Drilling blastholes for ore and waste	2.5 to 4 inch	142	1
Dozer (D7)	Levelling rock tipped at WRD and pit services	4 m blade	150	1
Dozer (D6)	Levelling ore delivered to heap leach	3.3 m blade	104	1
Mechanic Truck	Mechanical breakdown assistance	N/A	62	1
Tractor (Fuel/Water)	Delivery of fuel by bowser to pit and pulling water bowser for dust suppression	55 kW tractor	53	1
Pumps	Dewatering of open pit	30 kW pump	30	2
Light Delivery Vehicles	Used by maintenance and supervisors to access open pit	-	84	3
Bus	Transport of staff from local town to site	-	61	1
Grader	Haul and access road maintenance	3 m blade	104	1

16.5.4 ANCILLARY MINING OPERATIONS

To support core mining operations (drilling, blasting, loading and hauling), Harvest Mining will provide mining support operations which will include:

- equipment refuelling
- haul road maintenance
- supervision
- grade control
- geology
- equipment maintenance
- dewatering
- environmental services
- dust suppression.

The cost of the services above has been estimated and included in the mine operating cost.

16.5.5 GRADE CONTROL

Grade control activities considered in the PEA include:

- assaying of blast hole drilling chips
- employment of grade control technicians and geologists to work with mining
- crews
- use of the mill facility assay lab to conduct daily assaying of blast holes.

16.6 MINE SCHEDULING

A preliminary mine schedule was generated using Whittle™ software and Microsoft® Excel. Mine scheduling was based on use of the Whittle™ Milawa algorithm to optimize project value. The criteria used to generate the mine schedule is shown in Table 16.6 and the mine schedule is shown in Table 16.7.

Table 16.6 Mine Scheduling Criteria

Area	Unit	Value
Mining Recovery	%	98
Mining Dilution	%	5
Minimum Mining Width	m	15
Mining Cost	US$/t	3.30
Processing Cost	US$/t	12.90
G&A Cost	US$/mo	201,072
Maximum Mining Throughput	t/mo	150,000
Maximum Mill Throughput	t/mo	21,748
Gold Recovery	%	65
Silver Recovery	%	40
Gold Price	US$/tr oz	1,275.00
Gold Selling Costs	US$/tr oz	97.00
Silver Price	US$/tr oz	16.50
Silver Selling Cost	US$/tr oz	1.65

Table 16.7 Mine Schedule

Mine Schedule	Units	Years						Total/ Average
		-1	1	2	3	4	5	
Total Tonnes Mined	kt	345	1,785	1,547	846	625	32	5,180
Heap Leach Feed (diluted)	kt	46	246	261	261	261	12	1,086
Waste Mined	kt	299	1,538	1,286	585	364	20	4,093
Au grams as Mined to Mill	kg	247	787	742	777	799	63	3,414
Ag grams as Mined to Mill	kg	183	1,237	2,278	7,193	12,286	1,677	24,855
Au Head Grade	g/t	5.4	3.2	2.8	3.0	3.1	5.5	3.14
Ag Head Grade	g/t	4.0	5.0	8.7	27.6	47.1	145.3	22.9
Tonnes Under Leach	kt	-	227	261	261	261	77	1,086
Au Under Leaching	kg	-	881	714	783	793	243	3,414
Ag Under Leaching	kg	-	1,070	1,827	5,475	11,634	3,730	23,736
Au Recovered	kg	-	516	479	525	529	171	2,219
Ag Recovered	kg	-	301	550	1,632	3,474	1,164	7,121
Au Ounces Recovered	'000 tr oz	-	17	15	17	17	6	71
Ag Ounces Recovered	'000 tr oz	-	10	18	52	112	37	229

16.7 WASTE ROCK MANAGEMENT

Waste rock from mining will be placed in a waste rock dump (WRD) located approximately 120 m to the west of the open pit as shown in Figure 16.1. Waste rock will be used for haul road construction, heap leach pad berms, and other infrastructure pads at the mine site. The WRD has been designed to a capacity of 2 Mm³ based on an assumed density of 2.66 t/m³ and a 30% swell factor.

Table 16.8 Waste Dump Design Parameters

Item	Unit	Value
WRD		
Dump Bench Height	m	5
Dump Bench Width	m	7
Dump Face Angle	degrees	32
Overall Pit Slope Angle	degrees	20
WRD Haul Roads		
Minimum width	m	10
Maximum Grade	%	8

16.8 MINE PERSONNEL REQUIREMENTS

The mine will operate 24 hours per day, 7 days per week, 350 days per year. Operators and mine personnel are currently planned to work 12-hour shifts. A summary of the mine labour requirements is shown in Table 16.9.

Table 16.9 Mine Labour Requirements at peak labour requirements

Role	No. Required
Mining Management	
Mine Manager	1
Chief Engineer	1
Mining Engineer	1
Geologist	1
Pit Geologist	2
Surveyor	2
Grade Control	4
Administration	2
Safety Personnel	2
Clerks	4
Total Mining Management	**20**
Production and Operators	
Production Supervisor	4
Blasters	2
Blast Crew	4
Drill Operator	4
Drill Assistants	4
Excavator Operators	4

table continues...

Role	No. Required
Loader Operators	4
Truck Drivers	16
Dozer Operator	8
Pit Loader Operator	2
Grader Operator	2
Fuel Truck	4
Water Truck	2
Pump Crew	2
Total Production	**62**
Maintenance	
Maintenance Manager	1
Maintenance Supervisor	4
Mechanics	18
Welders	6
Tire Man	6
Store Man	6
Fuel Clerk	6
Maintenance Planner	5
Total Maintenance	**52**
Total Mining Staff	**134**

17.0 RECOVERY METHODS

17.1 INTRODUCTION

Based on the test work conducted by BCR in 2013 and SGS in 2016, the Terakimti deposit oxide zone appears to be amenable to conventional cyanide leaching utilizing either agitated tank leach or heap leach technology. For this study, the proposed gold extraction treatment is heap leaching. The following key factors have been considered to utilize heap leaching rather than agitated leaching:

- Ttest work indicates satisfactory gold extraction and rapid leach kinetics
- lower capital and operating costs
- reduced project complexity and shorter time required for project construction and implementation.

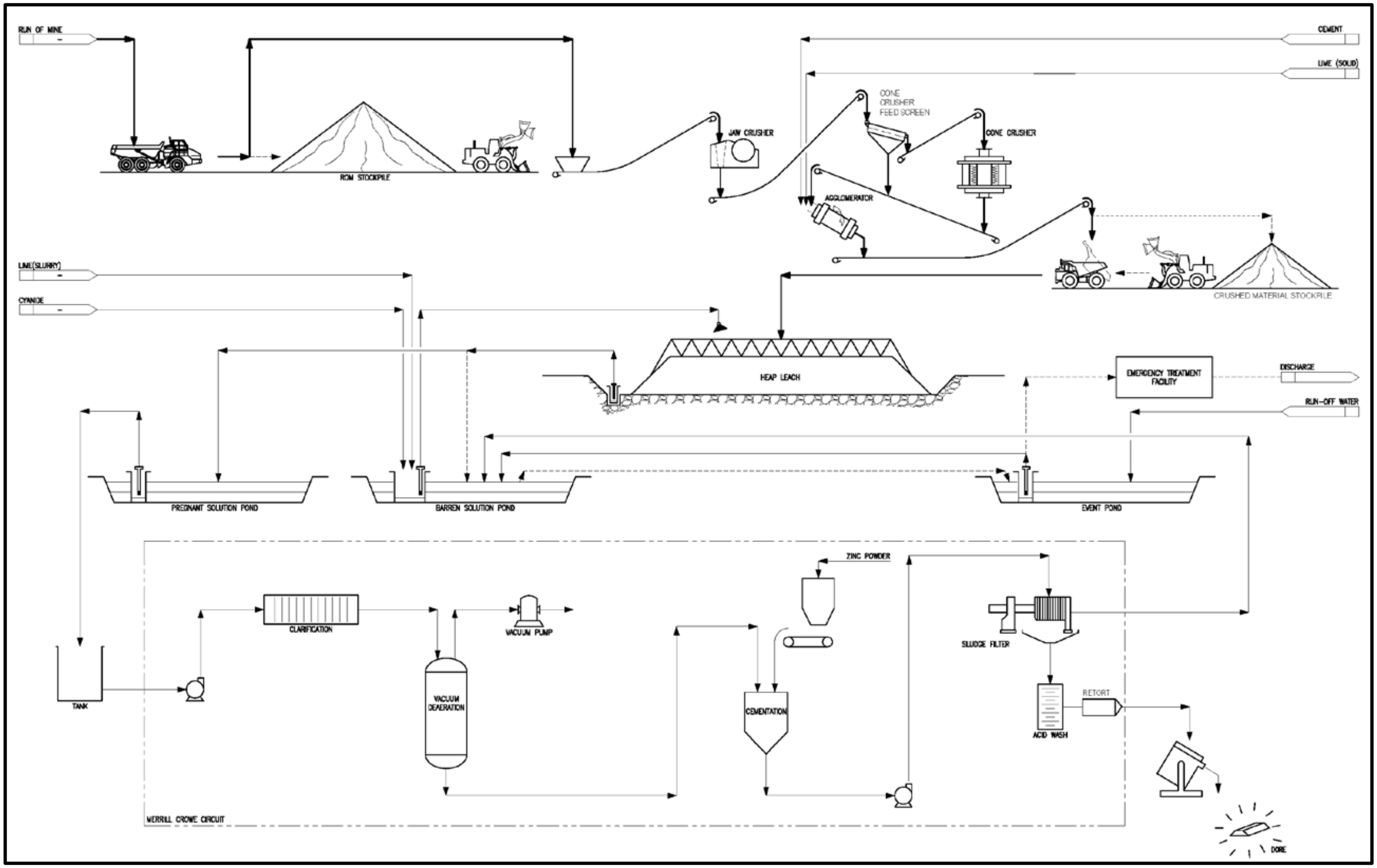
Due to the significant quantity of silver present in the oxide mineralization of the Terakimti deposit, Merrill Crowe precipitation will be used to recover dissolved gold and silver from the PLS.

The processing facilities proposed for Terakimti include:

- two-stage crushing, screening, and agglomeration
- heap stacking and leaching
- gold and silver recovery by Merrill Crowe processing.

A simplified flowsheet for the heap leach process is shown in Figure 17.1

Figure 17.1 Heap Leach Processing Flowsheet



17.2 MAJOR DESIGN CRITERIA

The heap leach has been designed to process 260 kt/a of oxide tailings. This would be equivalent to a throughput rate of 715 t/d. The crushing/leaching/gold recovery will operate 24 hours per day, seven days per week. The material that has been stacked on the heap leach pad will be continuously leached year-round. The availabilities will be 70% for the crushing and agglomeration circuits and 90% for the leaching and Merrill Crowe treatment circuits to allow for planned downtimes, such as maintenances, shift changes, and unplanned downtimes.

The major criteria used in the design are outlined in Table 17.1.

Table 17.1 Major Design Criteria

Criteria	Unit	Value
Operating Year	d	365
Crushing/Agglomeration Circuit Availability	%	70
Overall Plant Availability	%	90
Annual Processing Rate	kt	260
Daily Processing Rate	t/d	715
Agglomerated Bulk Density - Initial	t/m³	1.40
Crushed Particle Size, 100% Passing	mm	20
Moisture Content of Agglomerated Feed	%	8
Total Loading/Curing/Leaching/Rinsing Cycle	d	100
Cyanide Solution Strength	g/L	0.5-1.0

17.3 PROCESS PLANT DESCRIPTION

The process for the extraction of gold and silver from the Terakimti oxide material will utilize heap leach technology currently used globally for gold and silver mining operations and Merrill Crowe processing for gold and silver from the PLS.

17.3.1 CRUSHING

A two-stage crushing circuit has been proposed to reduce the run-of-mine (ROM) oxide material to finer than 20 mm. The crushing circuit will include a jaw crusher, a vibrating screen, a cone crusher, an agglomerator and related mobile conveyors.

The ROM material will be transported from the open pit to the crushing plant by haul trucks. The haul trucks will dump the ROM into a dump pocket which will feed the ROM material onto a belt conveyor for transport to the primary jaw crusher. A front-end loader (FEL) will also be used to reclaim the stocked ROM material into the dump pocket according to the mine plan. The jaw crusher product will be discharged onto a second belt conveyor and fed onto a vibrating screen to remove undersize material from the feed to the secondary cone crusher. Screen oversize will report into a surge bin ahead of the secondary cone crusher. The surge bin enables feeding of the cone crusher at a steady rate, which improves the crusher operating efficiency. The discharge of the cone crusher is planned to be finer than 20 mm.

The undersize material from the vibrating screen and the crushed material from the cone crusher will feed onto a common conveyor that will transport the material to an agglomerator. Cement will be added to the material on the conveyor as it feeds into the agglomerator which will cause the fine particles to agglomerate with the coarser particles. Spray water will be added into the agglomerator to improve agglomeration efficiency. The agglomeration prior to placing the crushed material onto the heap leach pad will improve the permeability of the heap and therefore improve the efficiency of gold and silver extraction in the heap leach operation. Lime will also be added to the crushed material prior to placement to control the alkalinity of the heap leach. The agglomerated material will discharge onto a crushed material stockpile adjacent to the crushing plant for curing.

17.3.2 HEAP LEACHING

The heap leach pad will be an engineered structure that will consist of a gravel or sand base covered with a clay liner, which is then covered with an impermeable synthetic geomembrane. The perimeter of the leach pad will be surrounded by impermeable berms. The pad will be gently sloped to a central PLS collection pond on the downside of the pad. The solution collection pond will also be lined with an impermeable geotextile liner. The leach pad will be designed to withstand the loading of crushed leach material and the movement of heavy equipment on top of the crushed material on the pad. A leak detection and recovery system (LDRS) including ground wells will be installed for the heap leach pad and the solution ponds to detect any solution leakages.

The crushed and agglomerated material will be reclaimed from the stockpile with a FEL and loaded into a haul truck for transport to the heap leach pad. Material will be stacked onto the heap pad in 5 m lifts. A bulldozer will be used to spread the material evenly on the leach pad. The final heap is expected to cover an area of approximately 5.0 ha, or 50,000 m^2, with an average vertical height of 20 m.

Barren leach solution will be pumped to the top of the heap leach pad and distributed through pipe headers and drip emitters onto the surface of the active heap. The cyanide solution will percolate down through the heap, dissolving the gold and silver. The gold and silver bearing PLS will be collected in the pregnant leach solution pond.

Four lined solution ponds are planned for the project, including one PLS pond, one barren leach solution (BLS) pond, one event or overflow solution pond and one polishing pond. The PLS pond will collect the gold and silver bearing pregnant solution draining from the heap leach pad. The BLS pond will be used to store the gold-silver depleted solution return from the Merrill Crowe facility. The event pond will be constructed to temporarily store excess process solution that may occur during upset conditions, freshet, and excess precipitation events. The solution contained in the event pond will be recycled back into the heap leach circuit when normal operation resumes.

Solution from the barren solution pond will be pumped to the leach heap. Concentrated cyanide solution will be added to the barren solution pond where it will be mixed to give a controlled cyanide concentration of approximately 0.5 to 1.0 g/L sodium cyanide strength. The pH will be maintained at 10.5 or higher. This solution will be distributed over the leach pad at an overall solution feeding rate of approximately 8 to 10 L/h/m^2. A total leaching/wash/rinse duration of 100 days will be allowed.

Solution from the PLS pond can overflow into the BLS pond should this be required. Solution from the BLS pond can also overflow into the event or overflow solution pond. This event pond will also collect excess water and drainage solution from the heaps and the plant areas. The overflow solution pond will also supply makeup water to the process by pumping the water back to the BLS pond. Alternatively, excess solution from this pond which is collected during June and September will be treated with calcium hypochlorite and discharged into the polishing pond to reduce the cyanide levels to acceptable limits prior to discharging this water to the environment, or re-using this water as process water.

17.3.3 MERRILL CROWE RECOVERY AND REFINING

The PLS will be pumped from the PLS pond to the Merrill Crowe facility for gold and silver recovery. The resulting BLS discharged from the Merrill Crowe facility is pumped back to the BLS pond for re-use in the heap leaching.

The PLS is pumped from the PLS pond to a PLS holding tank at the Merrill Crowe facility. The PLS is then pumped from the holding tank to a pressure clarifier to remove suspended solids. The clear PLS from the clarifier will be sent to a de-aeration tower operating under vacuum to remove oxygen. From the de-aeration tower the PLS will flow into a mixing cone where a slurry of zinc dust, lead nitrate, cyanide and filter aid will be pumped into the de-aerated solution. The cementation reaction occurs at the point of introduction of the slurry to the de-aerated solution. This reaction normally requires approximately 2 to 5 minutes for completion. The lead nitrate addition is used to improve the gold and silver precipitation efficiency. The PLS is then pumped to a plate and frame type filter press where the gold-silver-zinc precipitate will load onto the filter cloths. At regularly scheduled intervals the filter press will be taken off line, open and the loaded filter clothes removed. Clean filter clothes will be installed, the filters closed and put back into operation. The filter cloths, loaded with gold-silver-zinc precipitate containing excess zinc, will be washed in sulfuric acid solution. The acid washed slurry will be pumped to a digest precipitate filter press. This precipitate from the filter press will be dried in an oven.

In the on-site refinery, the dried gold-silver-zinc precipitate will be mixed with flux materials and then placed in a high temperature furnace for smelting. During the smelting process the gold and silver will be separated from the zinc and other materials and once separation is complete, the molten gold and silver will be poured from the smelting furnace into molds to produce doré bars. This doré is the final product from the minesite and the doré bars will be sent to an off-site refinery to produce separate high purity gold and silver products. The slag from the furnace is collected, and occasionally will be re-smelted to recover small amounts of gold and silver that collects in the slag.

The filtrate or barren solution, will flow into the barren solution pond where the pH will be adjusted to 10.5 with lime if necessary and then be pumped back to the heap pad for leaching after cyanide concentration is adjusted to approximately 0.5 to 1.0 g/L sodium cyanide.

The BLS from the Merrill Crowe facility will be pumped to the BLS pond where the solution will be conditioned with lime and cyanide prior to being recycled to the heap leach. The cyanide concentration will be adjusted to approximately 0.5 to 1.0 g/L sodium cyanide while the solution pH value to 10.5 or higher. The BLS will closely monitored to ensure levels are maintained at the specified design criteria.

17.3.4 WATER SUPPLY

Two separate water supply systems for fresh water and process water will be provided to support the operation.

Fresh Water Supply System

Fresh water will be supplied to a fresh/fire water storage tank from the pits, nearby river and/or from wells. Fresh water will be used primarily for:

- fire water for emergency use
- gland service for the slurry pumps
- reagent makeup.

The fresh/fire water tank will be equipped with a standpipe which will ensure that the tank is always holding sufficient fresh water, equivalent to a 2-hour supply of fire water.

Potable water supply will be supplied from nearby river separately and will be treated and stored in the potable water storage tank prior to delivery to various service points.

Process Water Supply System

Process water will be required for agglomeration and heap leach pad irrigation. The barren solution from the Merrill Crowe circuit will be reused for the heap leach irrigation. Due to void water loss and evaporation water loss, makeup water will be required, especially during the dry months. The makeup water is expected to be from the pit, runoff catchment, and wells.

17.3.5 AIR SUPPLY

Separate air service systems will supply air to the following areas:

- high-pressure air from a potable air compressor for the crushing area
- high-pressure air by dedicated air compressors for the Merrill Crowe circuit

- instrument air will come from the air compressors at the Merrill Crowe facility and will be dried and stored in a dedicated air receiver.

17.3.6 HEAP LEACH LAYOUT

The heap layout, heap lift height, and number of lifts have been assumed for the purposes of this study and are detailed in Section 17.3.2. The maximum height has been restricted to 20 m to prevent potential permeability issues in the leach pad. This proposed height for the heap would require geotechnical verification and further metallurgical test work confirmation. The preliminary site layout and available space, site drainage, and pad size have been designed according to the area topography and the best available information.

18.0 PROJECT INFRASTRUCTURE

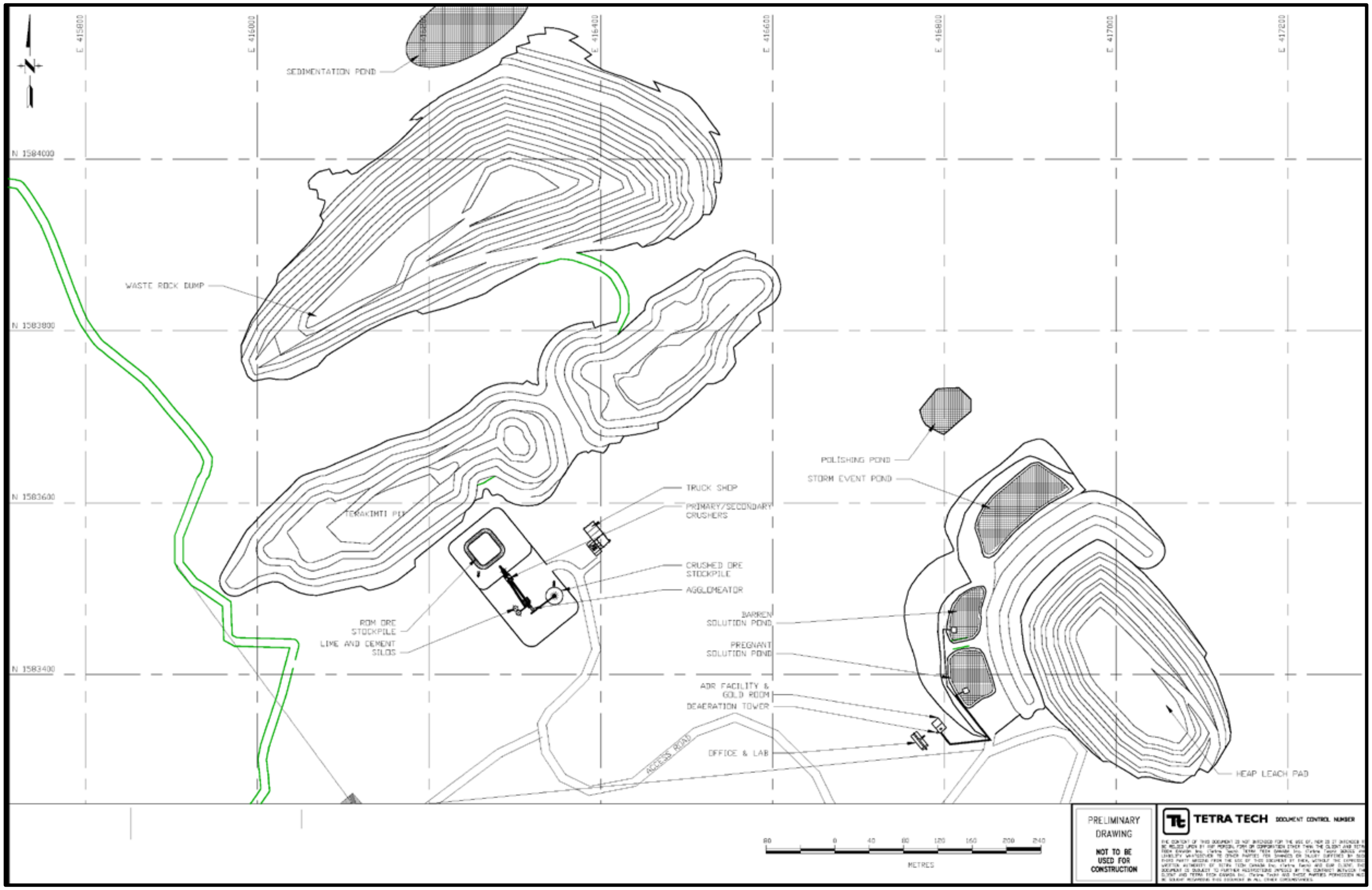
Tetra Tech completed and prepared preliminary infrastructure layouts and estimated the cost of infrastructure to support the Terakimti deposit oxide zone.

The infrastructure required to support the mining and processing operations will include the following:

- maintenance workshop, steel frame structure including container storage and blacksmith shops
- explosive storage magazines
- administrative building with emergency first room
- lunch room, washrooms and change rooms for employees
- assay laboratory and supplies warehouse
- lighting equipment
- substation and power distribution
- heap leach pad, adsorption/desorption recovery plant and solution ponds
- Access and site roads.

The layout of the mine site and associated infrastructure is shown in Figure 18.1.

Figure 18.1 Site Layout for the Terakimti Open Pit and Heap Leach Mining Operation



18.1 MINING FACILITIES

In addition to the open pit, waste rock dump and heap leach facilities, Harvest Mining will provide ancillary facilities to support mining operations. These are further detailed below.

18.1.1 EQUIPMENT LAYDOWN AREAS

The equipment laydown area is required for storage of large and/or bulk equipment and supplies used in operations.

The laydown area may include a shift change parking area for mine mobile equipment. The equipment laydown area will include a brake check facility, such that prior to each shift a brake checks can be performed on the mobile equipment, prior to the equipment descending into the pit.

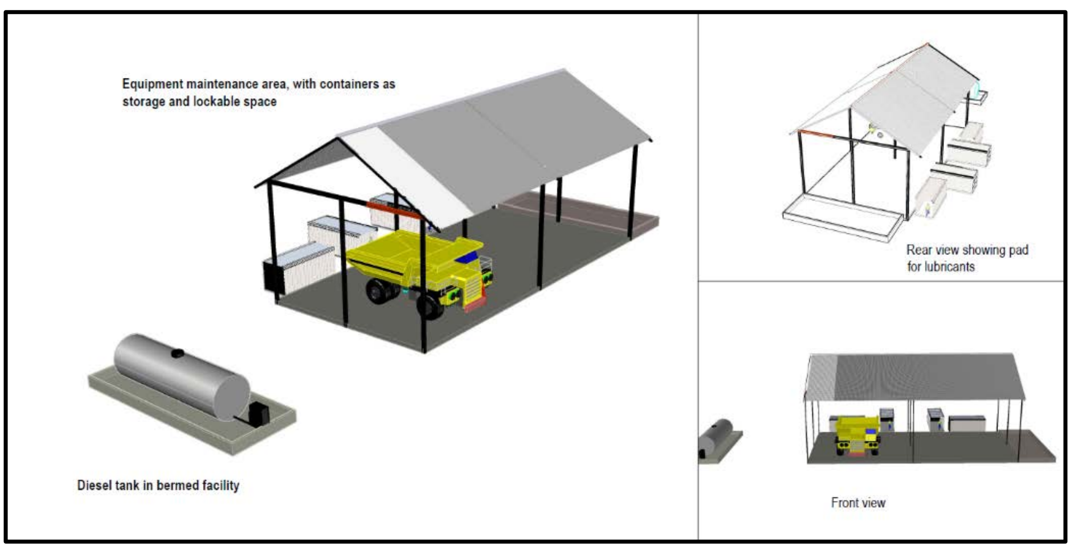
18.1.2 EQUIPMENT MAINTENANCE FACILITIES (TRUCK SHOP)

Harvest Mining will provide a mobile equipment maintenance facility which will comprise of a roofed structure, with a concrete floor. This facility will be used for mobile equipment maintenance and miscellaneous equipment repairs, and will house tools, steel working equipment and a tyre replacement and compressor facility.

The facility will include a lined area for storage of lubricants and other fluids used in equipment maintenance. Storage of parts and office supplies will be provided in customised containers, which will be placed at the facility.

The facility will include washrooms and change rooms for employees (Figure 18.2).

Figure 18.2 Mobile Equipment Maintenance Building (Truck Shop)



18.1.3 FUEL STORAGE AREA

The fuel for mobile equipment will be stored in a tank which will be placed on a curbed (bunded) concrete pad. The curbed concrete pad will be sized to contain 110% of the full volume of the fuel tank. The fuel tank is planned to be located adjacent to the equipment maintenance facility (truck shop) as shown in Figure 18.2.

18.1.4 EXPLOSIVES STORAGE AND MAGAZINES

Explosives will be stored in purpose built structures in a designated area a minimum of 500 m away from any inhabited or active area. The explosive storage will include two areas, namely ammonium nitrate prill storage and magazines for detonators and cartridges. No sensitive explosives including heat sensitive or flame sensitive explosives will be used on site. All explosives will be of the type that requires a detonation charge for detonation.

Explosive storage will be carried out as per Ethiopian regulations and/or international standards. This will include the separation of storage facilities and the placement of berms around explosive storage facilities. The explosives magazines will be secure structures with restricted access. Explosives will be withdrawn only as required for the next planned blast and only in the amount required for the next blast. Inventories of explosives will be strictly monitored.

Ammonium nitrate prill will be stored in 25 or 50 kg bags within the explosive storage facilities, while cartridges, boosters, and detonators will be stored in manufacturer packaging on shelves in a purpose-built magazine.

Magazines and/or ammonium nitrate storage will be spaced a minimum of 90 m from each other and the explosive storage site will be positioned 500 m from inhabited areas and other operational sites.

18.2 LUBRICANT STORAGE

Lubricant storage for mining equipment and processing equipment will be stored in a curbed concrete facility, adjacent to the equipment maintenance workshop. The facility may be enclosed with a lean-to structure if deemed necessary.

18.3 PROCESSING PLANT AND ADMINISTRATION FACILITIES

The processing plant consists of distinct areas as identified below:

- crushing and grinding facility
- Merrill Crowe process system and refinery
- PLS and BLS ponds, storm event pond, and pumping systems
- assay and metallurgical laboratories

- reagent storage
- hazardous waste storage.

18.3.1 HEAP LEACH FACILITY AND PONDS

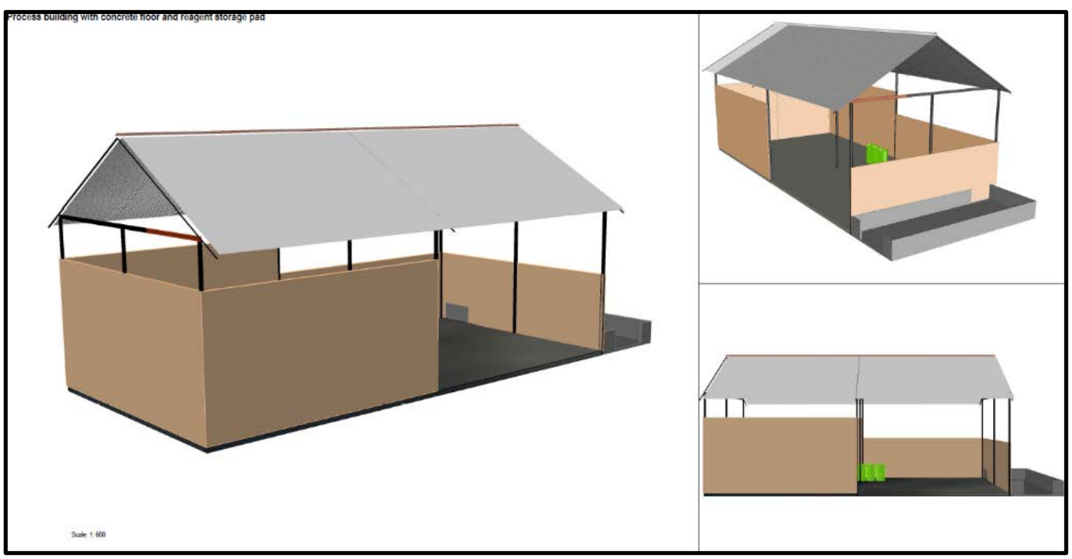
A key component of the mine infrastructure is the heap leach facility. For the PEA, a double lined heap leach facility with a toe berm has been proposed. The conceptual plan includes a 650,000 m^3 heap leach facility, 5,000 m^3 capacity PLS and BLS ponds, and a 10,000 m^3 capacity storm water (overflow) pond. These facilities will include pumping systems for the various leach solutions. The heap leach will be created by trucking the ore to the heap leach facility, where a bull dozer will spread the mineralized material. The toe berm provides for containment of leach solutions and potential geotechnical stability (subject to geotechnical analysis).

The heap leach has been designed with a slope no greater than 20° from horizontal with bench face angle of 30°. The lifts and berm widths were both designed to be 5 m.

18.3.2 MERRILL CROWE FACILITY

The Merrill Crowe facility will house the Merrill Crowe gold and silver recovery system and refinery room, including the furnace. This will be housed in a steel structure with a concrete floor, designed to contain any spills. Figure 18.3 shows a conceptual view of a process building. Part of the building would be secured for the refinery. Reagents would be stored on lined concrete pads.

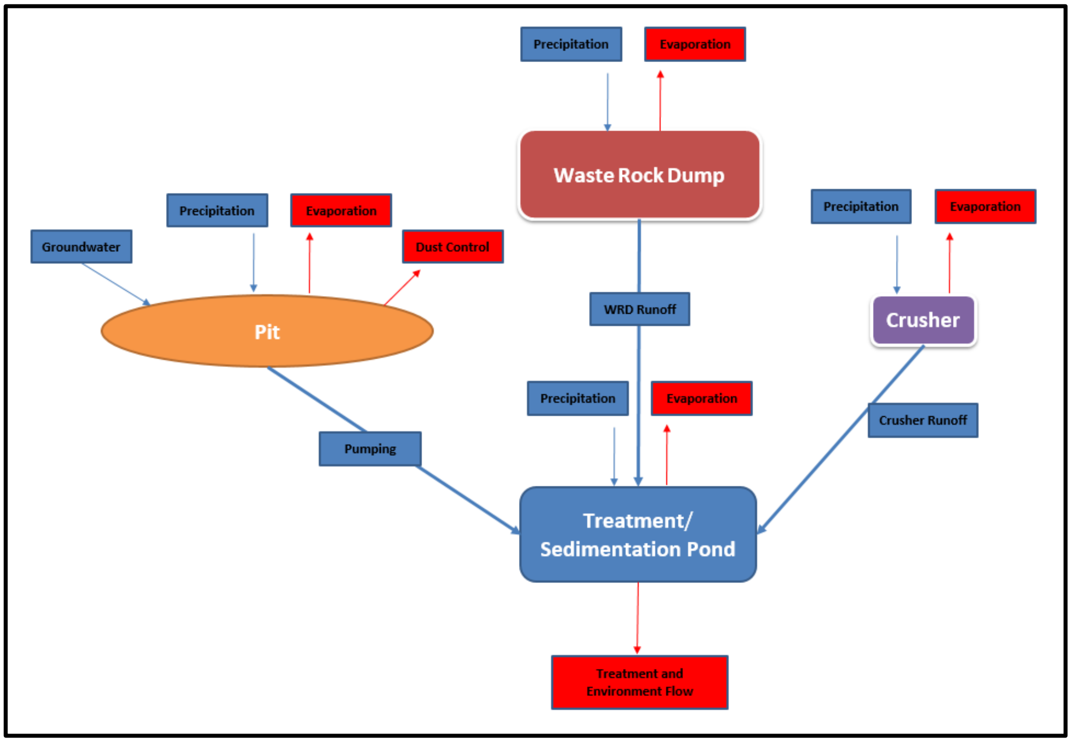
Figure 18.3 Design of Process Building



18.4 SITE WATER MANAGEMENT

The overall water balance at the mine site considered the interaction between the pit, waste rock dump, crushing facility and the treatment/sedimentation pond. Figure 18.4 shows a schematic for the overall mine site water balance.

Figure 18.4 Overall Site-wide Water Balance Processes



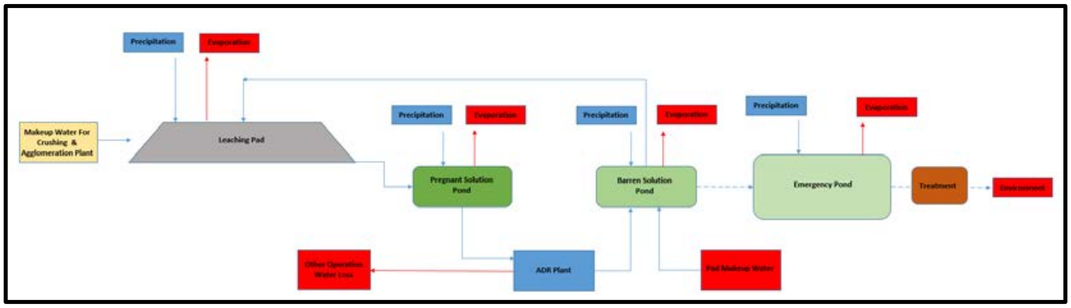
The water balance will be further defined in the detailed design phase for the proposed Terakimti mine. The key aspects to be defined include:

- perform a complete groundwater study to assess the groundwater inflow to the pit
- complete a contact and clean water diversion design for the various sub-catchments at the site
- finalize design of the sedimentation pond for proper settling and extreme storm events
- finalize a detailed water balance analysis for the pit by considering the pit stage storage curves to determine the required pumping rates throughout the LOM.

It is anticipated that the makeup water will be sourced from water wells, the final water sources will be confirmed in the detailed design stage.

In order to illustrate the water balance processes, a schematic was generated showing the sources and sinks of the flow into and out of the system. Figure 18.5 shows the water inputs and outputs at the Terakimti heap leach.

Figure 18.5 Heap Leach Process Water Flow Chart



18.4.1 Heap Leach Water Management

In relation to water management, the heap leach facility is the most significant aspect. The heap leach facility will include a pad with an impermeable synthetic liner onto which all the mined material will be placed. The heap leach works by circulating a leachate solution such that minimal makeup water is required and that minimal contact water will be treated and discharged. Figure 18.6 shows the catchment delineation for contact and non-contact water based on minimizing the contact water area. A summary of the catchment sizes is shown in Table 18.1.

Figure 18.6 Terakimti Mine Project Setting

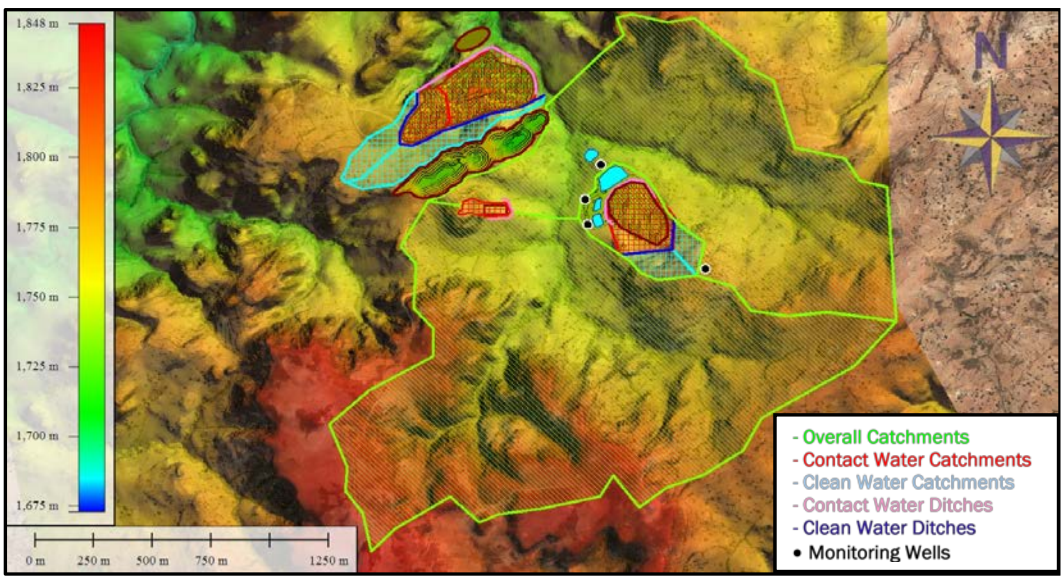


Table 18.1 Catchment Delineation

ID	Catchment Name	Area (km^2)
1	Overall Catchment (North)	1.200
2	Overall Catchment (South)	2.000
3	WRD Clean Water (South)	0.065
4	WRD Clean Water (North)	0.036
5	WRD Contact Water (West)	0.037
6	WRD Contact Water (East)	0.094
7	Pit	0.085
8	Crushing Area	0.012
9	Heap Leach Contact Water	0.090
10	Heap Leach Clean Water (East)	0.019
11	Heap Leach Clean Water (West)	0.026

18.4.2 STORM WATER MANAGEMENT

The storm event pond has been designed to provide sufficient capacity to contain the 1-in-100-year, 24-hour precipitation storm event, consistent with current Industry best practice. The daily maximum rainfall data for a period between 1997 and 2016 from the Shire station was used for frequency and extreme flow analysis.

The methods used to complete the frequency analysis included the GEV methods of moments and the log Pearson type III. Table 18.2 summarizes the extreme rainfall depth estimated for various return periods by averaging the values taken from the two mentioned frequency analysis methods.

The proposed storm event pond has a capacity of 12,500 m^3. The capacity should be sufficient to handle the operational precipitation provided that regular treatment will be in place during the wet season.

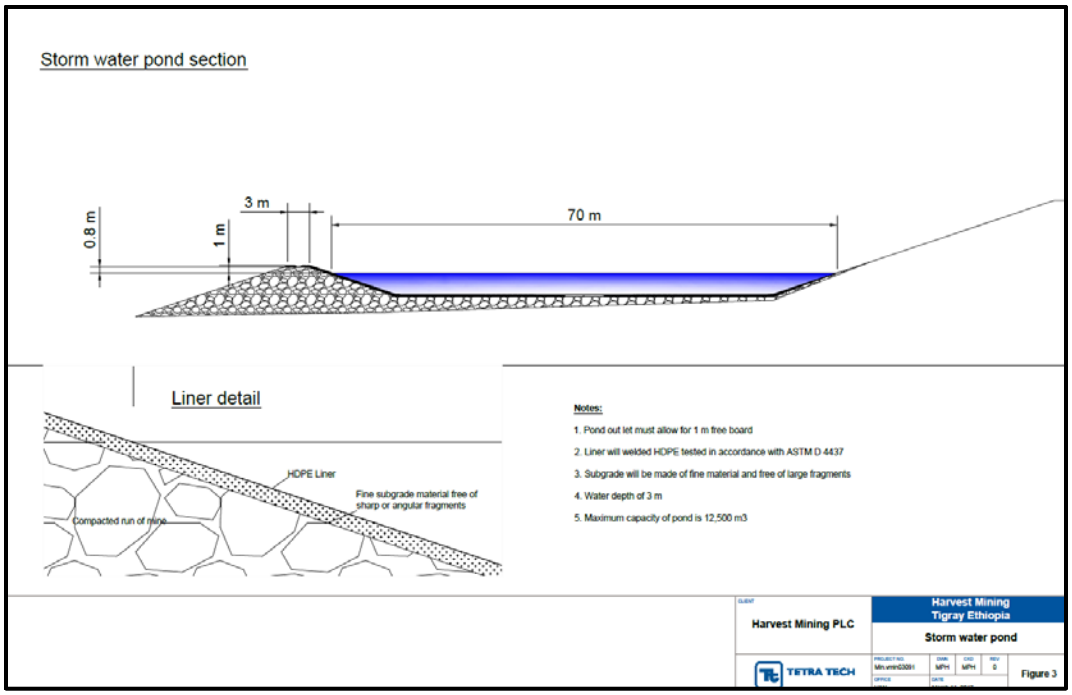
As shown in Table 18.2, the 1-in-100-year, 24 hour rainfall at the Shire is estimated to be 115 mm. This rainfall depth corresponds to a flood inflow of approximately 6,500 m^3 from the heap leach, PLS pond, BLS pond and the storm event pond.

Considering a surface area of 5,500 m for the storm event pond (Tetra Tech 2017), the corresponding flood depth will be approximately 1.2 m. The management of the storm event pond should therefore be such that there is always a 1.7 m of freeboard within the storm event pond to accommodate the 1-in-100-year, 24-hour design flow plus an additional 0.5 m freeboard allowance. Figure 18.7 shows a typical cross section of the storm event pond.

Table 18.2 Storm Frequency Analysis

Return Period (years)	Rainfall Depths (mm/d)
2	57.3
3	64.1
5	71.9
10	81.9
20	91.7
50	105
100	115
200	125

Figure 18.7 Typical Cross Section of Storm Event Pond



18.5 TAILINGS CONTAINMENT FACILITY

The Terakimti Heap Leach Project scope of development envisions utilizing a heap leach processing operation and as such no tailings would be produced from the operation. Therefore, a tailing containment and storage facility is not required.

18.6 EARTHWORKS

For various infrastructure aspects at Terakimti, cut-and-fill is required to form level working areas for building sites and vehicle traffic. Cut and fill slopes have been designed with 3:1 slopes (20° from horizontal) where excavations or pads need to be created for the different infrastructure around the mine site. Fill material will be sourced from pre-stripping of the open pit or borrow sites, based on suitability assessment at the time of construction.

18.7 POWER SUPPLY AND DISTRIBUTION

Power for Terakimti will be sourced from the Ethiopian national grid. Harvest Mining would request from the Ethiopian Electricity Authority for a connection point roughly 1.5 km south of Adi Dairo. From this point a line would be constructed 6.5 km towards the northeast to the mine site. The following infrastructure would be required for power supply to the mine:

- a tap point near Adi Dairo
- 7.5 km, 34 kV line from tap point to mine site
- substation at mine site including:
  - mine site disconnect
  - three-phase stepdown transformer from 34 kV to 380 V mine voltage
  - main switch gear and metering equipment
  - grounding protection
- mine site power distribution cabling
- Merrill Crowe facility distribution board
- crushing facility distribution board
- office and laboratory complex distribution boards.

18.8 CAMP AND DINING AREA

Harvest Mining has not commissioned studies to determine whether all, some or no workers will be accommodated on site during working rotations. This report considers housing of only a small portion of the workforce on site which will include:

- critical plant staff required for solution circulation pumps and systems during operations
- accommodation for fly-in/fly-out expatriate employees
- site security staff on duty.

The camp will consist of mobile units placed near the mine offices. All staff will be provided with meals during working hours and the kitchen provided for this purpose will be usable by staff staying in camp.

18.9 Security

Harvest Mining will provide security services for the mine site facilities, personnel, and key activities. It would be the intention of Harvest Mining to contract an experienced and highly respected Ethiopian-based security service.

18.10 Emergency Response and Fire Fighting

Harvest Mining will ensure that emergency response capabilities are provided at the mine site. This will include the following key tasks:

- emergency first-aid training for dedicated site safety personnel
- fire response training for dedicated safety personnel
- completion of an emergency response plan, including incident training for potential site incidents
- hazardous materials safe handling
- provision of equipment to support first aid, safety, fire control and other emergency response procedures.

Site firefighting equipment will include fire extinguishers, fire water pumps, sprinklers and other systems onsite. Water storage for the mine site operations will include a water reserve for firefighting which is not accessible for operations.

18.11 Office Infrastructure

Harvest Mining will provide a site office in the form of mobile office units and built structures. The offices and camp will include the following infrastructure:

- reception, meeting room, and training room
- roughly five offices, 10 workstations, and data storage areas
- kitchen and dining areas
- toilets and change rooms
- parking area
- rooms for overnight stays (used for expats staff, staff on standby and visitors)
- additional tented accommodation for temporary workers
- potable water treatment

- sewage treatment facilities
- training room/class room.

18.12 OFF-SITE INFRASTRUCTURE

Harvest Mining will maintain minimal off-site infrastructure. However, some staff will be sourced from Shire and other local areas, and expatriate staff will be housed in Shire. Depending on the status of the power supply, Harvest Mining may need to install power connection infrastructure outside Adi Dairo for a line to the mine site.

18.13 ROADS

Harvest Mining plans to access the mine site from two main routes namely, a southern route and a western route. The southern route will join the main tarred road approximately 20 km north of Shire. The western route leads directly to Adi Dairo. Harvest Mining is expected to use both routes during operation of the mine. The southern route will be upgraded where required for increased traffic. The western route will have multiple stream crossings and as such may be impassable in the rainy season.

The southern route will be the main route used by Harvest Mining to access the mine on a daily basis from Shire. The current plan is to bus workers in from Shire and Adi Dairo (and other areas) to the mine site daily and to have only critical staff housed at the mine site.

18.13.1 ROAD MAINTENANCE AND UPGRADE

Harvest Mining, with approval from local government, will undertake an upgrade to roads accessing the mine site. This upgrade will include widening and grading and potential placement of surfacing gravels if available from the mine site. During the operations Harvest Mining will continue to maintain the access roads through periodic grading as required.

19.0 MARKET STUDIES AND CONTRACTS

Once in production, there will be a number of markets available to Harvest Mining for the sale of the gold and silver produced. Typically, mining companies will send gold and silver doré to recognized international refineries. There are no gold refineries in Ethiopia. Refineries would charge a fee per ounce produced and retain a small portion of gold as part of the fee. Typically, doré is delivered to an international airport through a contract with a security firm. The PEA includes costs of selling doré.

For this PEA for the Terakimti oxide zone the following selling terms have been considered:

- 99.5% payable gold after refining (0.5% deduction by refineries)
- 95% payable silver (5% deduction by refineries)
- US$5/tr oz fee for gold refining
- US$0.7/tr oz for silver refining
- US$2/tr oz fee for transportation and insurance
- an additional Ethiopian government royalty deduction of 7% of saleable gold and silver, which has been included in the PEA economic model.

19.1 MARKETING

Harvest Mining will employ standard gold mining industry practice to sell the gold and silver produced to local and/or international gold and silver buyers.

20.0 ENVIRONMENTAL STUDIES, PERMITTIING AND SOCIAL OR COMMUNITY IMPACT

20.1 KEY FINDINGS

The environmental and social impact assessment identified the following key findings:

- There are no records of registered endemic, threatened or endangered species in the Harvest Project area.
- Community members state that a tomb of the martyrs exists at the St. Kirkos church within the Harvest Project area, both of which may require relocation.
- Subsistence farming is the primary livelihood of the residents in the Harvest Project area.
- Relocation of a relatively small number of affected residents will be required, and consultation has indicated the majority of affected residents are agreeable to relocation on the basis of land to land compensation and provision of basic public services equivalent to their current area.
- The majority of the community stated that they are in favour of the implementation of the Harvest Project. A small number of individuals stated that they did not want to leave their land.

20.2 LOCATION, ENVIRONMENTAL, AND PHYSICAL SETTING

The Terakimti deposit is located in the Tigray region of Northern Ethiopia approximately 40 km from the town of Shire, a regional service and supply centre.

The Tigray region is characterized by a temperate to hot climate and has both dry and wet seasons. The wet season extends from mid-June to mid-September with average rainfall of 800 to 1,000 mm/a, with high intensity rainfall events. Mean daily temperatures range from a high of 32.5°C in March to a minimum of 13°C in January.

Topography in the project area is characterized by undulating plateaus, with an average elevation of 1,700 m, cut by deeply incised valleys that commonly host seasonal tributaries and shallower valleys hosting north- and northeastern flowing rivers. Rivers in the Terakimti Project area flow into the Mereb River, 10 to 30 km northeast of the property. Vegetation consists of open grassland and arable fields on the plateaus and man-made terracing on steeper inclines, whereas river valleys and steep hills are typically covered in small shrubs or succulents. Soil cover is typically less than 1 m.

Most of the region is devoid of vegetation, particularly during the dry season, with small areas of shrub brush and trees most commonly located along rivers and their tributaries or ephemeral drainage.

The closest protected areas to the Harvest Project are the Shire Wildlife Reserve and the Simien Mountains National Park. The Shire Wildlife Reserve, covering an area of 75,300 ha, is located approximately 55 km south of the Harvest Project area. The Simien Mountains National Park covers an area of 17,900 ha and is located approximately 110 km to the south east.

20.3 ENVIRONMENTAL AND SOCIAL IMPACT ASSESSMENTS AND MINE PERMITTING

20.3.1 COUNTRY OVERVIEW

Mine permitting in Ethiopia falls under the jurisdiction of the MoMPNG. ESIAs are a key requirement of the mine permitting process. ESIAs are reviewed by the Environment and Community Directorate Director (ECDD) which reports to the MoMPNG. ESIAs must be completed by suitably qualified environmental and social consultants licenced by the government of Ethiopia. The ESIA completed for the Terakimti Project was performed by Beles Engineering Ltd. Pty. Co. (Beles), a qualified Ethiopian based environmental and social consultancy.

The ESIA was prepared in accordance with the Environmental Policy of Ethiopia (1997), Environmental Impact Assessment Proclamation No. 299/2002, Environmental Impact Assessment Guideline for Mineral and Petroleum Operation Projects (2003), and Environmental Assessment Reporting Guide by the Federal Environmental Protection Authority (EPA) (2004). The ESIA describes the baseline environmental and social conditions in the Harvest Project area, presents an assessment of potential impacts associated with the Terakimti Project, and proposes mitigation and management measures to minimize potential impacts.

Under Ethiopian regulation, the Terakimti Project requires a full ESIA assessment. Under World Bank guidelines, the Terakimti Project would be designated under Category A, which requires full environmental assessment followed by Independent Environmental Review. This ESIA for the Terakimti Project falls under Category A.

The ESIA Study was completed by Beles and submitted to Harvest Mining in October 2015. Subsequently the ESIA study was submitted to the MoMPNG in November August 2016 in support of an application for a mining licence for the Terakimti Project. The ESIA was reviewed and accepted by the MoMPNG, and was a key component of a formal approval and award of mining licence agreement to Harvest in December 2017.

20.3.2 SOCIO-ECONOMIC CONDITIONS

OVERVIEW

A preliminary socio-economic baseline study of Terakimti Project area was undertaken by Harvest Mining in December 2014. The study included mapping of existing community infrastructure and interviewing local people to gain an understanding of their culture, social organization, as well as to assess social utilities such as education, health, and gender participation. Public consultation was also used as a means of assessing public opinion regarding potential project impacts and mitigation measures. Interviews were carried out at household, community and institutional levels. This section presents a summary of the preliminary findings.

The people living in proximity to the Terakimti Project area are Tigray people who have similar culture, language affiliation, ethnicity, inter-marriage, and decent as the wider Tigray Society. Tigrigna is widely spoken along with the Saho language which is spoken by some Muslim minority groups in rural areas. The pattern of human settlement has two separate but related components: the way in which the land is divided among its owners and the way in which the owners arrange buildings on their land.

There are several schools in the area with the highest percentage of children living in proximity to the Terakimti Project area being enrolled in Aye Elementary School. The dominant religion in this area is Orthodox Tewohdo (Christian).

Agriculture forms the basis of livelihood of a large part of the inhabitants of the area. Maize and finger millet are by far the most common crops under cultivation.

Livestock production is also a very large component of the agricultural livelihood. Oxen are used for plowing and threshing, donkeys and camels for transportation, cows for the production of replacement stock and milk for household consumption, and sheep and goats are kept as assets, which can be exchanged into cash at times of need.

Artisanal mining is another economic activity practiced at specific locations within the general area. According to information from local water and energy personnel in the area, there may be approximately 3,500 artisanal miners that are organized in self-help associations and engaged in small scale gold production.

Consistent with a small scale agricultural livelihood, most of the farmers/residents of the study area use family labour for their farms, while a lesser amount (16%) use hired labour.

Educational levels in the area are low by western standards. The findings of this assessment indicate that only 6% of rural residents have primary education, and that a significant number of children are unable to go to school due to the inability to pay for school expenses, the local demand for child labour and lack of awareness on behalf of parents regarding education. Very high rates of illiteracy exist within the area, with illiteracy as a high as approximately 80%.

20.4 COMMUNITY CONSULTATION

Public consultation with the community members and officials was conducted. The discussion revealed that most members of the community were aware of the Terakimti Project although they did not have detailed information or knowledge. An important point of discussion was the resident's understanding regarding the benefits and impacts of the proposed project.

The consultation included discussion on the possible adverse impacts of the project development. Residents voiced concern regarding the potential adverse effects of the Terakimti Project including air pollution, loss of fertile soil, deforestation, soil degradation, noise pollution, risk of airborne diseases due to dust particles, decline of agricultural productivity, problems associated with loss of settlement, animal migration and insecurity of the community. It was recorded that the community members have different opinions and concerns regarding the project and their willingness to be relocated if need be to allow project development. Community members agreed that they are ready to leave their land and be relocated so long as the issue of compensation is properly settled.

The majority of the community stated that they are in favour of the implementation of the Terakimti Project. However, they demanded land to land compensation and basic public services equivalent to their current area. A small number of individuals stated that they did not want to leave their land.

20.5 FLORA AND FAUNA

The study area is primarily characterized by scattered vegetation (wood land species) dominated by Acacia spp. and with considerable densely vegetated land with shrubs dominated by Dodonaea Angustifolia. In the region, woody and herbaceous climbers are rarely found. The local Terakimti Project area is dominated by shrubs Tahises (Dodonaea Angustifolia) and woodland species. The hill area immediately to the north of the project area is covered by dense shrub vegetation. There are protected vegetated areas in the project site related to local farmers.

Figure 20.1 Woodland Vegetations Dominated by Acacia spp.

An assessment of the wildlife within the Terakimti Project area was obtained as follows:

- Formal and informal discussions with the local communities, elders and experts living in the Terakimti Project area based on their daily experiences and historic observations enabled them to identify the animals that frequent the Terakimti Project area.
- Numerous visual observations were made by traversing the study area and very common vertebrates and invertebrates were observed and recorded. In additional, evidence of their presence was studied in the form of tracks, nests, burrows, and feathers etc.

This baseline study documented thirteen (13) mammals, one (1) amphibian, three (3) reptiles, fifteen (15) bird species and six (6) species of invertebrate (insects) within the project area. The assessment did not identify any endangered, endemic or rare species in the Terakimti Project area. There are no records of registered endemic, threatened or endangered species in the Terakimti Project area. Several of the species are listed as Least Concern and Vulnerable on the International Union for Conservation of Nature and Natural Resources(IUCN) Red List data.

20.6 PROTECTED AREAS AND NATIONAL PARKS

The closest protected areas to the Terakimti Project is the Kafta Sheraro National Park which is located approximately 55 km south of the Terakimti Project area, and covers an area of 75,300 ha or 5,000 km^2. A total of 167 mammal species, 95 bird species and 9 reptile species have been recorded within the park. The park is home to a transboundary African elephant (nay Africa harmaz) population of about 100-150 individuals, which it shares with Eritrea's Gash-Setit, and which constitutes the northernmost elephant population in Eastern Africa. Kafta-Sheraro is also an important wintering site for demoiselle cranes (birds). Other notable wildlife species include lion, leopard (nebri), caracal(naybereka dimu), aardvark (tsihira), greater kudu (agazen), roan antelope(agazen), red-fronted gazelle (midaqa) and red-necked ostrich (segon). The Simien Mountains National Park protected area covers an area of 17,900 ha and is located approximately 110 km to the southeast of the Terakimti Project site.

20.7 HISTORICAL AND ARCHAEOLOGICAL SITES

The participants of the community consultation disclosed that there is a tomb site of martyrs at St. Kirkos church within the Terakimti Project site.

20.8 WATER RESOURCES AND GROUND WATER HYDROLOGY

The Harvest Property license is located in an area of varying relief ranging from 1,240 to 1,910 masl. Rivers in the area drain north-eastwards into the Mereb River approximately located 10 to 30 km to the northeast of the license area. Most of the streams in the region are seasonal. The drainage pattern is dominantly dendritic. The drainage density is moderate. There are few perennial rivers including the Terakimti. The most important rivers in the area are Terakimti, Seye and Widake. These rivers rarely run dry and provide limited water for irrigation during the dry season. In the mining area the only perennial river is the Terakimti. There is also seasonal stream called Sensela.

The climate of the region is semi-arid. Due to relatively low rainfall intensity the discharge of the streams is low for most of the year. Many streams are dry during the dry season or have very low discharge. However, in July and August the discharge of the rivers can be relatively high due to the combination of high precipitation and the low permeability of the basement rocks, which enhances surface runoff.

The only perennial river in the Terakimti Project area is the Terakimti River. The discharge of the river very much depends on the level of the Tarkimit dam constructed for irrigation purposes upstream of the proposed mining area. The Terakmiti River is considered to be one of the possible sources of water for mining. This river is being used for local irrigation and drinking downstream.

Attempt was made to measure the discharge in August. The discharge measurement indicates that the average discharge of the river is 1.034 m^3/s. The discharge was measured using a current meter. This measured discharge is expected to be lower during the dry season.

Both groundwater and surface waters were tested for water quality. In general, the surface water quality is better than the groundwater water quality. Due to the presence of sulphide minerals, the sulphate content in the groundwater is high which resulted in very low pH value. The groundwater from exploratory drilling boreholes is highly acidic. Deep wells in this area cannot be used for drinking water supply based on Ethiopian and World Health Organization (WHO) water quality standards.

The high sulphate content in the groundwater indicates the potential for acid mine drainage during and after mine operation. The Terakimti Project technical design will need to take into account the potential for mine acid water drainage.

20.8.1 COMMUNITY WATER SUPPLY

The rural community in the area gets water from hand dug wells and springs along the river courses. More recently, boreholes fitted with hand pumps are becoming more common for water supply to the communities, however these wells are still under distributed in the region. Many residents must travel long distances to collect water for domestic use. Water quality test results in some of these wells demonstrate relatively high heavy metals and the sulphate content due to the existence of sulphide minerals.

20.8.2 MINE PROJECT WATER SUPPLY

As part of the assessment, limited hydrogeological, hydrological and geophysical investigation was completed. Based on the assessment there is the possibility of using both groundwater and surface water resources in the area for the Terakimti mine requirements. Boreholes can be drilled along the course of the Terakimti River, preferably downstream of the Terakimti reservoir where there is sustained recharge to the shallow aquifer from the dam. In addition, a small diversion from the perennial Terakimti River may also contribute to mine water supply. It is projected that the mine water usage should not significantly impact the downstream water users.

20.8.3 LAND USE AND LAND COVER

According to the data obtained from *Adi Gedenakebele*, the total area of the *kebele* is about 6,011 ha, of which 2,087 ha is arable land, 1961 ha bare land, 1,450 ha covered by vegetation, and 513 ha is inaccessible land. Most of the arable land is covered by annual crops with very little permanent crops. The common agricultural cereal cultivation forms the core component of the local farming system. Typically, the region is sparsely vegetated with areas of scrub and brush vegetation located along river courses.

Most of the proposed mining area is farming land and bush land. The bushes are localized in hilly areas and along the course of the two main rivers which are Terakimti and Sensela. There are few hilly protected areas. The hilly areas were protected under a closure program for the rehabilitation of indigenous trees.

Land use in the project area is predominantly subsistence agriculture. A mixed farming system with predominantly crop production is practiced. The growing season coincides with the rainy season. During this time crops such as maize, millet, maize, sorghum, teff and different beans are grown. Typical land use pattern of the project area is shown below during the wet and dry season with illustrative plates.

Land is also used for habitation and artisanal mining (mainly near river banks). The houses are scattered and made of stone covered with stone (slates) and iron sheet.

Figure 20.2 Typical Land Cover and Vegetation During the Wet Season

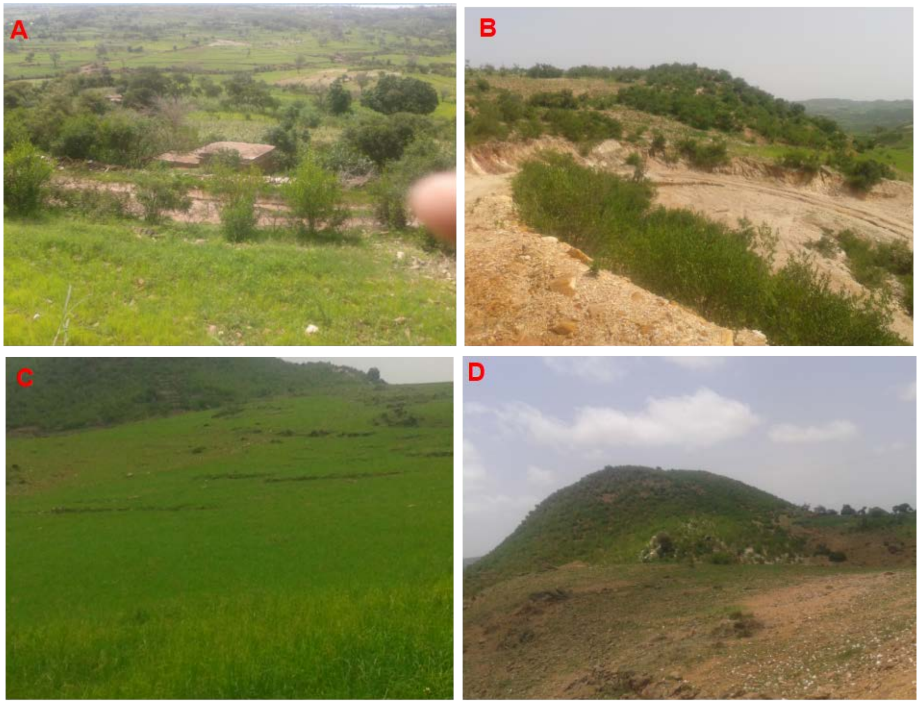


Figure 20.3 Typical Land Cover Features of the Terakimti Project Area During the Dry Season

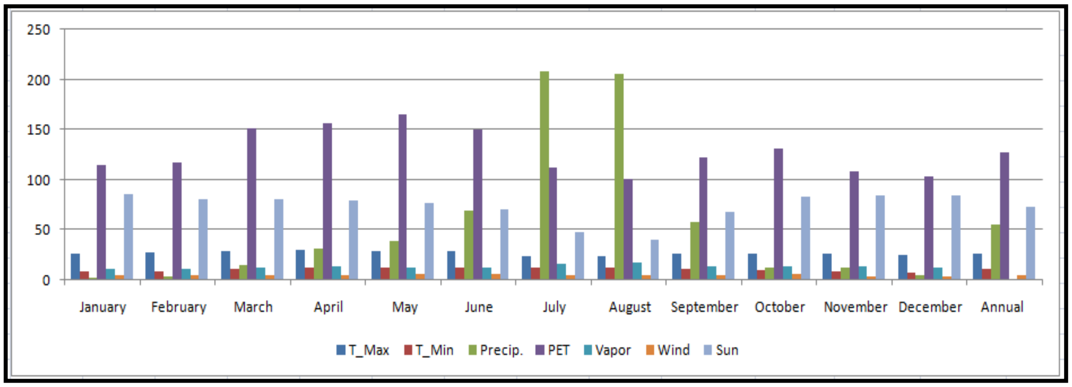
The natural vegetation that exists within the project area provides various ecosystem supplies including wood for harvesting, charcoal burning and firewood. The vegetation cover is characterized by scattered woodland and some densely vegetated shrubs dominated by Acacia spp. and *Dodonaea Angustifolia,* repectively.

20.8.4 CLIMATE

The nearest meteorological station to the Terakimti Project site is located at the town of Shire. The summary of the meteorological records for long term monthly data shows that the climate is semi-arid. The annual precipitation and potential evapotranspiration is 55.23 mm and 127.91 mm, respectively. The average annual temperature is 21°C.

The meteorological records show wide seasonal variations in meteorological parameters. The area receives significant rainfall from June to September. From December to March it is dry season. Major water stress exists in January, February and March. The region has a unimodal rainfall pattern. The main groundwater recharge and surface runoff occurs from June to September, the peak rainfall is in July. Figure 20.4 shows the seasonal variation of main meteorological parameters.

Figure 20.4 Long-term average meteorological data at Shire



20.8.5 Air Quality

The mining project site is located in a rural environment that is largely characterized by scattered households and subsistence agriculture. No industrial activities occur in the area and ambient air is expected to be clean and unpolluted and therefore within permissible limits of the air quality standards set by EPA and Environmental, Health and Safety (EHS) guidelines.

Air quality assessment results show that during the wet season the area is free of dust pollution. This is to be expected since the area is purely agricultural and the sampling period was in August 2015, during the rainy season. It is expected the baseline dust level to be significantly higher during the dry season, when more human activities and higher temperatures effects would generate more dust.

20.9 Compensation and Access

During the exploration period, Harvest Mining has implemented a policy of compensation for local farmers and landholders who are directly impacted by Harvest Mining's exploration activities, whether the impact is a result of access across their property or actual exploration activities on their properties. Compensation is determined through consultation with the Woreda Council representative and agreed with the local farmers whose land the exploration affects. Compensation was carried out on over one hundred drill sites and 29 trenches.

However, in Ethiopia the responsibility for formal relocation and resettlement of citizens that results from Government approved industrial development rests with the

Government, which has an established policy for relocation. In the case of development of the Terakimti mine, the relocation and resettlement of citizens will be managed by the Government.

21.0 CAPITAL AND OPERATING COST ESTIMATES

21.1 SUMMARY

The capital and operating costs estimated for the Terakimti Gold Heap Leach Project herein presented. The capital cost is estimated at US$17.2 million, with US$1.7 million in sustaining capital costs. Operating costs are estimated at US$34.10/t processed over the LOM.

All costs are reflected in Q1 2018/Q4 2017 US Dollars unless otherwise specified. The expected accuracy range of the cost estimates is –20% to +35%. When required, costs in this report have been converted using currency exchange rates of US$1.00:Ethiopian Birr (ETB) 23.00.

21.2 CAPITAL COST ESTIMATES

Tetra Tech developed and prepared the capital cost estimate based on preliminary design work, vendor budgetary estimates and quotes, Tetra Tech project data and industry cost publications.

Tetra Tech established the capital cost estimate using a hierarchical work breakdown structure (WBS). The accuracy range of the estimate is -20% to +35%. The base currency of the estimate is US dollars. A blended labour rate of US$8.00/h was used throughout the estimate.

The total estimated initial capital cost for the design, construction, installation, and commissioning is US$17.18 million. The estimated sustaining capital, which includes additional mining equipment and expansion of the heap leach pad totals US$1.7 million.

A summary breakdown of the initial capital cost is provided in Table 21.1.

Table 21.1 Capital Cost Summary

Description		Cost (US$ million)
Direct Costs		
10	Overall Site	0.85
30	Mining	2.99
40	Process	4.31
50	Heap Leach Pads and Pond	1.29
70	On-site Infrastructures	1.12
Direct Cost Subtotal		**10.56**
Indirect Costs		
90	Project Indirects	3.37
98	Owner's Costs	0.65
99	Contingencies	2.60
Indirect Cost Subtotal		**6.62**
Total		**17.18**

21.2.1 Mining Capital Cost Costs

The capital cost for mining is based on purchase of mining equipment. A total of $3 million was estimated for mining equipment. Waste rock pre-stripping cost has been allocated to the financial model and not included in the capital cost estimate.

21.2.2 Process Capital Costs

A total of $4.31 million was estimated for procurement, shipping, installation and commissioning of all process equipment. The cost includes crushing, agglomeration, Merrill Crowe circuit, leaching, reagent systems and the assay lab.

All process equipment and material costs are included as free carrier (FCA) or free board marine (FOB) manufacturer plants and are exclusive of spare parts, taxes, duties, freight, and packaging. These costs, if appropriate, are covered in the indirect cost section of the estimate.

21.2.3 Overall Site Infrastructure Capital Costs

A total of US$2.39 million was estimated for site preparation, earthworks, foundations, procurement, shipping, installation and commissioning for the overall site infrastructure, including heap leach pads and pond and their pipe work and liner systems. The cost includes administration building, truck shop, explosive storage, laydown area, water supply, sewage, on-site roads and the plant mobile fleet.

21.2.4 INDIRECT CAPITAL COSTS

The estimated indirect costs of US$3.37 million include construction indirects, spare parts, and freight and logistics, are calculated on a percentage basis based on Tetra Tech's experience. Allowances for initial fills are provided for reagents, lubricants and fuel. The engineering, procurement and construction management (EPCM) allowance is calculated on a percentage basis based on Tetra Tech in-house experience. Commissioning and start-up, and vendor assistance allowances are calculated based on the number of engineers required on site, estimated duration, and the average man-hour rates.

21.2.5 OWNER'S COSTS AND CONTINGENCIES

The US$0.65 million Owner's Costs were provided by EAM. The estimated contingencies, totalling $2.6 million, are allowances for undefined items of work which is incurred within the defined scope of work covered by the estimate. Each discipline was allocated different contingency factors due to the varied risk level. The average contingency for the Project is 24.6% of the total direct costs.

21.2.6 EXCLUSIONS

The following items are excluded from the capital cost estimate:

- pre-production mine pre-stripping (included in the financial model)
- working or deferred capital
- financing costs
- refundable taxes and duties
- land acquisition
- currency fluctuations
- lost time due to severe weather conditions
- lost time due to force majeure
- additional costs for accelerated or decelerated deliveries of equipment, materials, or services resultant from a change in project schedule
- warehouse inventories, other than those supplied in initial fills
- any project sunk costs (studies, exploration programs, etc.)
- mine reclamation costs (included in financial model)
- mine closure costs (included in financial model)
- escalation costs
- community relations.

21.3 OPERATING COST ESTIMATES

21.3.1 SUMMARY

On average, the LOM on-site operating costs for the Terakimti Heap Leaching Project are estimated to be US$34.10/t of material processed. The operating costs are defined as the direct operating costs including mine pre-stripping, mining, processing, site servicing, and G&A costs, including related freight costs. Table 21.2 shows the cost breakdown for various areas.

Table 21.2 LOM Average Operating Cost Summary

Description	Cost (US$/t processed)
Mining (excluding pre-stripping)	15.10
Process	12.90
G&A and Site Services	6.11
Total Operating Cost	**34.11**

The cost estimates in this section are based on information from Tetra Tech's in-house database or experience in similar projects. The expected accuracy range of the operating cost estimate is –25%/+35%. All the costs have been estimated in US dollars, unless specified.

It is assumed that operation personnel will reside in towns or villages nearby, excluding the management and senior technical team. There will be no accommodation or catering services to be provided at the site.

The operating costs exclude shipping and marketing charges for gold and silver, which are included in financial analysis.

21.3.2 MINING OPERATING COSTS

Tetra Tech estimated mining costs for each period of the LOM. Table 21.3 summarizes the mining costs over the LOM. The key assumptions used in the estimate of the mining costs include:

- fuel cost of US$0.85/L
- explosive cost of US $0.90/kg (based on quote by AEL Mining Services of $0.70 cost, insurance, and freight (CIF) Djibouti)
- mining labour rates varying from US$1.05 to US$30.00/h, averaging $7.50/h excluding burden costs, or $2.25 to $45.00/h including burden costs

Table 21.3 Mining Operating Costs

Description	Total Costs (US$ 000) Pre-stripping	Operations	Total LOM	Unit Costs (US$/t mined) Pre-stripping	Operations	Total
Management and labour	379	6,177	6,556	1.10	1.28	1.27
Labour burden cost	189	3,089	3,278	0.55	0.64	0.63
Explosives	108	1,539	1,647	0.31	0.32	0.32
Fuel	247	3,688	3,935	0.72	0.76	0.76
Maintenance	130	1,901	2,032	0.38	0.39	0.39
Total Costs	**1,054**	**16,394**	**17,448**	**3.05**	**3.39**	**3.37**

21.3.3 PROCESS OPERATING COSTS

The average LOM unit process operating cost is estimated at US$12.90/t processed, at a nominal processing rate of 715 t/d, or 261,000 t/a, including the power cost for the processing plant. The estimate is based on 12-hour shifts, 24 h/d, and 365 d/a.

The breakdown for the estimated process operating cost is summarized in Table 21.4.

Table 21.4 Process Operating Costs

Description	Unit Cost (US$/t processed)
Manpower	3.19
Metal Consumables	0.14
Reagent Consumables	5.38
Maintenance Supplies	1.24
Operating Supplies	0.90
Power Supply	0.70
Plant G&A	0.19
Others	1.16
Total Process Operating Cost	**12.90**

The process operating cost estimate includes:

- personnel requirements including supervision, technical supports, operation and maintenance and salary/wage levels, including burdens;
- Jaw crushing, cone crushing and agglomeration, estimated from Tetra Tech's experience;
- maintenance supplies, based on approximately 10% of major equipment capital costs or estimated based on the information from the Tetra Tech's database/experience;

- reagent consumptions, based on test results and reagent prices from Tetra Tech's database, the main reagents including cement, lime, sodium cyanide, zinc powder, lead nitrate, flux, and sulfuric acid;
- other operation consumables, including laboratory and service vehicles consumables;
- power consumption for the processing plant based on the preliminary plant equipment load estimates and a power unit cost of US$0.07/kWh.

All operating cost estimates exclude taxes unless otherwise specified.

21.3.4 General and Administrative and Site Services Operating Costs

G&A and site service costs include the expenditures that do not relate directly to the mining or process operating costs. A summary of the G&A and site service cost estimates is shown in Table 21.5.

Table 21.5 G&A and Site Service Operating Costs

Description	Unit Cost (US$/t processed)
Management and general labour salaries	3.90
Site maintenance and office supplies	0.20
Insurance, accounting	0.60
IT, communications and computing	0.30
Consulting services	0.10
Government fees and permit expenses	0.10
Expat and other travel	0.60
Other	0.30
Total	6.11

22.0 ECONOMIC ANALYSIS

A PEA should not be considered a Prefeasibility or Feasibility study, as the economics and technical viability of the Project have not been demonstrated at this time. The PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. Furthermore, there is no certainty that the conclusions or results reported in the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Tetra Tech completed a pre-tax preliminary economic analysis based on estimated costs and revenues for mining and processing the Terakimti oxide deposit.

The economic analysis was conducted in US dollars.

The economic analysis concluded the following financial results:

- pre-tax NPV of US$20.89 million at an 8% discount rate
- pre-tax IRR of 37.4%
- pre-tax payback period of two years

Tetra Tech was assisted by EAM to prepare a tax model for the LOM of the Terakimti Heap Leach Project, which produced the following financial results:

- post-tax cash flow of US$20.9 million
- income tax payable of US$7.4 million over the LOM
- government participation deductions of US$1.1 million over the LOM
- post-tax NPV of $13.18 million at an 8% discount rate
- post-tax IRR of 30.1%
- post-tax payback period of 2.4 years
- all in sustaining cash costs of US$649/tr oz of gold produced (net of by-product credits).

22.1 ECONOMIC ANALYSIS ASSUMPTIONS

For the purpose of the PEA, the following assumptions were applied to financial modelling:

- base case gold price of US$1,325/tr oz
- production rate of 715 t/d
- gold recovery of 65% and a silver recovery of 30%
- pre-production capital costs of US$17.177 million and sustaining capital costs of US$1.72 million
- average LOM costs of US$34.10/t processed
- Ethiopian government net sales royalty of 7%
- reclamation costs of US$1.25 million
- pre-production operating losses of US$1.2 million
- pre-stripping cost of US$1.05 million
- government mandated social and community responsibility expenses of US$1.9 million over the LOM (2% of revenue)
- a portion of in-country G&A expenses of US$863,000 over the LOM
- working capital of US$1.3 million dollars recovered over the LOM.

22.2 BASIS OF FINANCIAL EVALUATIONS

The production schedule has been incorporated into the 100% equity pre-tax financial model to develop monthly recovered metal production from the relationships of tonnes processed, head grades, and recoveries. Gold and silver payable values were calculated based on base case metal prices. Net invoice value was calculated each year by subtracting the applicable refining charges from the payable metal value. At-mine revenues are then estimated by subtracting transportation and insurance costs. Government royalties were deducted from net revenues from sales of doré.

Operating costs were then deducted from the remaining revenue to derive operating cash flow. Allowable capital cost depreciation, sustaining costs and other cash expenses were then deducted from operating cash flow to derive taxable income.

Capital costs were deducted in the year of expense, with non-cash costs (depreciation added back) to get pre-tax cash flow.

Taxes were calculated based on allowable deductions, and then deducted from pre-tax cash flow to derive post-tax cash flows.

22.3 SUMMARY OF FINANCIAL RESULTS

A summary of key financial results is shown in Table 22.1. Various gold prices were applied to evaluate project sensitivities.

Table 22.1 Summary of Financial Results

	Unit	Base Case	Lowest	5-year Average	Long Term
Gold Price	US$/tr oz	1,325	1,200	1,250	1,379
Tonnes Processed	t	1,086,329			
Waste Rock	t	4,093,444			
Strip Ratio	-	3.77			
Gold Ounces Mined	tr oz	109,773			
Silver Ounces Mined	tr oz	799,091			
Gold Ounces Recovered	tr oz	71,352			
Silver Ounces Recovered	tr oz	305,258			
Payable Gold	tr oz	70,996			
Payable Silver	tr oz	289,995			
Revenue					
Gross Revenue from Gold	US$000	94,069	85,195	88,745	97,903
Gross Revenue from Silver	US$000	3,697	3,697	3,697	3,697
Refining and Selling Cost for Gold	US$000	499	499	499	499
Refining and Selling Cost for Silver	US$000	214	214	214	214
Net Revenue from Gold	US$000	93,570	84,695	88,245	97,404
Net Revenue from Silver	US$000	3,537	3,537	3,537	3,537
Total Project Revenue	US$000	97,107	88,233	91,782	100,941
Precious Metals Royalty (7%)	US$000	6,797	6,176	6,425	7,066
Operating Costs					
Mining Costs	US$000	17,448	17,448	17,448	17,448
Process Costs	US$000	14,014	14,014	14,014	14,014
G&A Costs	US$000	6,641	6,641	6,641	6,641
Total Operating Costs	US$000	38,103	38,103	38,103	38,103
Other Costs	US$000	25,102	24,937	25,003	25,174
Net Operating Income	US$000	27,105	19,016	22,252	30,599
Capital Costs					
Initial Capital	US$000	17,177	17,177	17,177	17,177
Sustaining Capital	US$000	1,724	1,724	1,724	1,724
Cashflow from Operations	US$000	29,359	21,270	24,506	32,853
Pre-tax NPV at 8%	US$000	20,260	13,652	16,295	23,115
Pre-tax IRR	%	37.4	27.7	31.6	41.5
Post-tax Cashflow	US$000	20,891	15,128	17,433	23,380
Post-tax NPV at 8%	US$000	13,756	8,913	10,850	15,849
Post-tax IRR	%	30.1	21.9	25.2	33.7
Post-tax Payback Period	Years	2.42	2.82	2.64	2.28
C1 Cash Cost	US$/tr oz Au	465	465	465	465
AISC Cost	US$/tr oz Au	649	638	642	653

Note: AISC – all-in sustaining cost

22.4 POST-TAX ANALYSIS

Tetra Tech was assisted by EAM in estimating of taxes payable over the LOM of the Terakimti Heap Leach Project.

The following assumption were made for tax purposes.

- It has been assumed that the Terakimti Heap Leach Project will be 100% financed through equity.
- The only taxable jurisdiction has been assumed to be Ethiopia.
- A statutory income tax rate of 25% has been assumed with the following additional deductions payable to the Ethiopian government:
  - 7% net sales royalty on the value of metal doré sold
  - 5% net profit deduction.
- A portion of operating losses incurred during exploration have been carried forward to operations.
- Is has been assumed that taxes are payable in the month of April in the year after incurrence of revenues.

22.4.1 TAXES INCLUDED IN THE FINANCIAL MODEL

Table 22.2 summarises post tax results for the Terakimti Gold Heap Leach Project:

Table 22.2 Post-tax Results for the Terakimti Gold Heap Leach Project

Item	Cost (US$ million)
Income from Sales of Doré	97.1
Ethiopian Government Royalty	6.8
Net Operating Income after Operating Expenses	53.2
Other Expenses	25.1
Taxable Income	28.2
Income Tax	7.4
Government Proclamation (5% of Net Profits)	1.1
Post-tax Cashflow	20.9

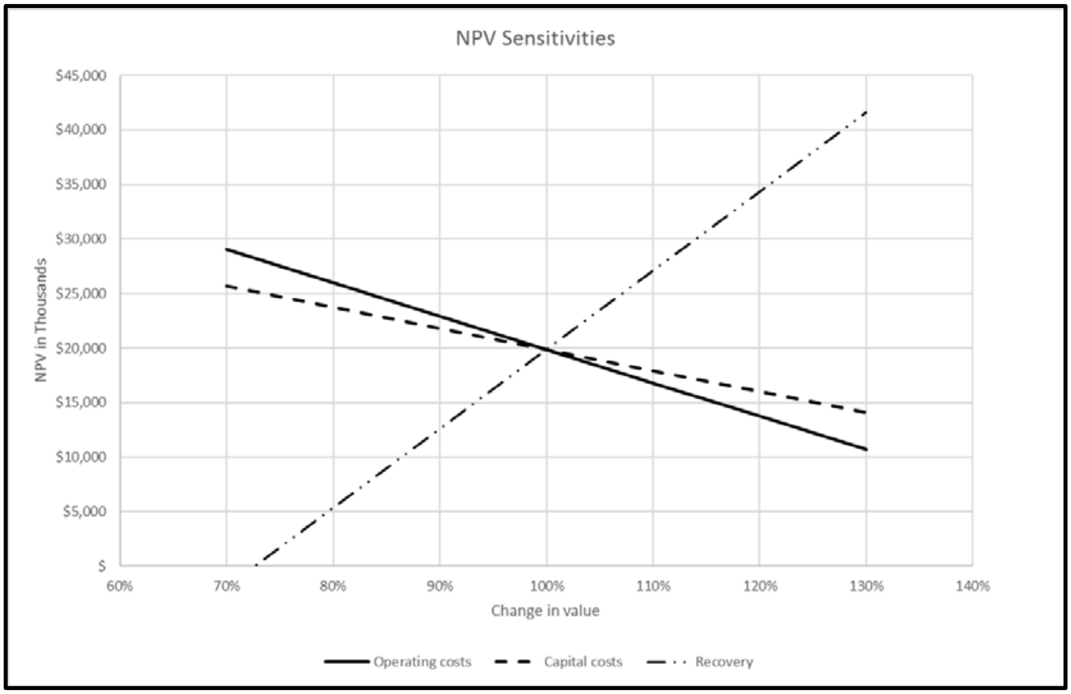
22.5 SENSITIVITY ANALYSIS

Tetra Tech conducted sensitivity analysis of a number of key financial inputs into the financial model (Figure 22.1)

Between recovery (or metal price), capital costs and operating costs, the financial results are most sensitive to variations in recovery (or metal price).

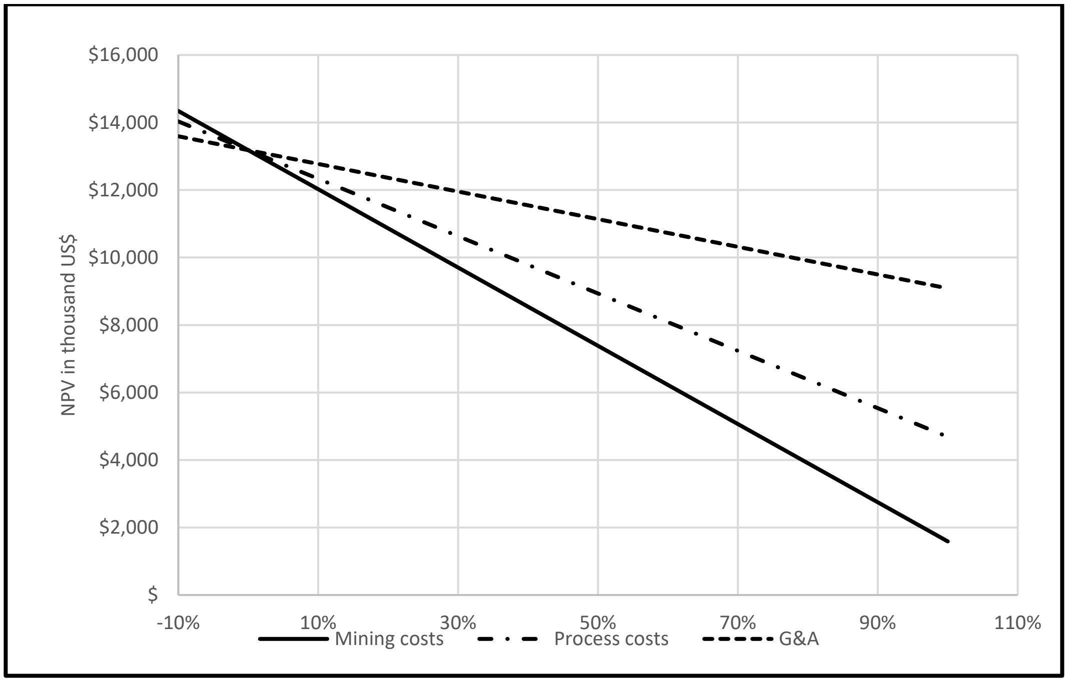
The project is more sensitive to changes in operating cost versus capital cost. The project has a positive cash flow, but a negative NPV (discount rate of 8%), at a gold price of US$960/tr oz.

Figure 22.1 Project Financial Sensitivities



As shown in Figure 22.2, the Terakimti Gold Heap Leach Project is most sensitive to mining operating costs, and least sensitive to G&A. The Terakimti Heap Leach Project would have a NPV of zero if the mining costs were to exceed US$7.20/t.

Figure 22.2 NPV Sensitivity to Operating Costs



22.6 CASH FLOW MODEL

An annualised and simplified cash flow model has been included in Table 22.3 to show how the financial results were generated.

Table 22.3 Terakimti Cash Flow

Mine Production Schedule	Unit	Total / Average	Quarter -6	Quarter -5	Quarter -4	Quarter -3	Quarter -2	Quarter -1	Quarter 1	Quarter 2	Quarter 3	Quarter 4	Quarter 5	Quarter 6
Process Feed	kilotonnes	1,086						46	51	65	65	65	65	65
Waste	kilotonnes	4,093						299	294	415	415	415	415	415
Total Tonnes Moved	tonnes	5,180						345	345	480	480	480	480	480
Mill Feed Grades														
Gold	grams/tonne	3.14						5.41	4.80	3.18	2.79	2.34	2.56	3.08
Silver	grams/tonne	22.88						4.01	6.10	5.01	3.83	5.37	6.57	6.57
Total Recovered Metals														
Gold	oz.	71,352							2,990	5,185	4,489	3,915	3,327	3,600
Silver	oz.	228,943							1,024	2,984	3,191	2,463	3,385	4,149
Metal Payment Factors														
Gold	%	99.5%						99.5%	99.5%	99.5%	99.5%	99.5%	99.5%	99.5%
Silver	%	95.0%						95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%
Offsite Costs														
Transport and Freight for Gold		$ (499)							$ (21)	$ (36)	$ (31)	$ (27)	$ (23)	$ (25)
Transport and Freight for Silver		$ (160)							$ (1)	$ (2)	$ (2)	$ (2)	$ (2)	$ (3)
Ethiopia Government Royalty		$ (6,797)							$ (276)	$ (479)	$ (416)	$ (362)	$ (309)	$ (335)
Operating Costs														
Total Mining		$ (17,448)						$ (1,054)	$ (1,045)	$ (1,259)	$ (1,274)	$ (1,275)	$ (1,272)	$ (1,276)
Processing		$ (14,014)							$ (588)	$ (660)	$ (842)	$ (836)	$ (841)	$ (841)
G&A		$ (6,641)							$ (409)	$ (377)	$ (377)	$ (377)	$ (368)	$ (368)
Total Operating Costs		$ (38,103)						$ (1,054)	$ (2,042)	$ (2,296)	$ (2,493)	$ (2,488)	$ (2,481)	$ (2,485)
Other Expenses														
Reclamation costs		$ (1,250)												
Contribution Expense		$ (1,835)			$ (29)				$ (73)	$ (127)	$ (110)	$ (96)	$ (82)	$ (89)
Ethiopia Head Office / In Country G&A		$ (863)	$ (37)	$ (33)	$ (38)	$ (33)	$ (37)	$ (33)	$ (38)	$ (33)	$ (37)	$ (33)	$ (38)	$ (33)
Total Other Expenses		$ (25,102)	$ (37)	$ (33)	$ (67)	$ (33)	$ (37)	$ (1,087)	$ (301)	$ (161)	$ (148)	$ (4,724)	$ (634)	$ (478)
Capital Investment														
Initial Capital		$ (17,177)	$ (1,494)	$ (1,771)	$ (2,716)	$ (5,025)	$ (1,110)	$ (5,062)						
Pre-Stripping		$ (1,054)						$ (1,054)						
Sustaining Capital		$ (1,724)							$ (190)				$ (514)	$ (356)
Working Capital		$ -							$ (4,082)	$ 1,279	$ 943	$ 612	$ (612)	$ (1,136)
Total Capital Investment		$ (19,954)	$ (1,494)	$ (1,771)	$ (2,716)	$ (5,025)	$ (1,110)	$ (6,116)	$ (4,272)	$ 1,279	$ 943	$ 612	$ (1,126)	$ (1,492)
Pre-Tax Cashflow Analysis														
Total Operating Income		$ 28,159	$ (37)	$ (33)	$ (67)	$ (33)	$ (37)	$ (1,087)	$ 1,318	$ 3,910	$ 2,880	$ (2,402)	$ 992	$ 1,487
Pre-tax Net Present Value (8%)		$ 19,467												
Pre-tax Internal Rate of Return	%	37.4%												
Post Tax Cashflow Analysis														
Income Tax Payable		$ (7,363)												$ (1,426)
Post Tax Net Annual Cash Flow		$ 20,941	$ (1,531)	$ (1,805)	$ (2,782)	$ (5,058)	$ (1,147)	$ (7,203)	$ (2,764)	$ 5,189	$ 3,823	$ 2,804	$ 380	$ (1,076)
Government Participation (Proclamation No. 816/2013)		$ (1,105)												$ (214)
Net Cash Flow - After Tax & Govt Participation		$ 19,837	$ (1,531)	$ (1,805)	$ (2,782)	$ (5,058)	$ (1,147)	$ (7,203)	$ (2,764)	$ 5,189	$ 3,823	$ 2,804	$ 380	$ (1,290)
Post Tax Net Present Value (8%)		$ 13,182												
Post Tax Internal Rate of Return	%	30.1%												
Post Tax Operating Payback	quarters	7.27												

Mine Production Schedule	Quarter 7	Quarter 8	Quarter 9	Quarter 10	Quarter 11	Quarter 12	Quarter 13	Quarter 14	Quarter 15	Quarter 16	Quarter 17	Quarter 18	Quarter 19
Process Feed	65	65	65	65	65	65	65	65	65	65	12		
Waste	285	171	138	154	167	126	114	81	62	107	20		
Total Tonnes Moved	351	237	203	220	232	192	179	146	127	172	32		
Mill Feed Grades													
Gold	2.97	2.76	3.45	3.35	2.44	2.67	2.72	2.97	3.80	2.76	5.49		
Silver	9.52	12.27	16.76	22.07	32.82	38.60	36.31	38.34	65.06	48.60	145.32		
Total Recovered Metals													
Gold	4,288	4,181	3,901	4,790	4,697	3,494	3,747	3,830	4,153	5,262	3,927	1,572	
Silver	4,167	5,972	7,703	10,496	13,817	20,470	24,213	22,940	24,124	40,425	30,968	6,453	
Metal Payment Factors													
Gold	99.5%	99.5%	99.5%	99.5%	99.5%	99.5%	99.5%	99.5%	99.5%	99.5%	99.5%	99.5%	99.5%
Silver	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%
Offsite Costs													
Transport and Freight for Gold	$ (30)	$ (29)	$ (27)	$ (34)	$ (33)	$ (24)	$ (26)	$ (27)	$ (29)	$ (37)	$ (27)	$ (11)	
Transport and Freight for Silver	$ (3)	$ (4)	$ (5)	$ (7)	$ (10)	$ (14)	$ (17)	$ (16)	$ (17)	$ (28)	$ (22)	$ (5)	
Ethiopia Government Royalty	$ (398)	$ (390)	$ (366)	$ (451)	$ (446)	$ (343)	$ (370)	$ (376)	$ (407)	$ (527)	$ (394)	$ (151)	
Operating Costs													
Total Mining	$ (1,087)	$ (915)	$ (862)	$ (881)	$ (912)	$ (855)	$ (831)	$ (776)	$ (755)	$ (855)	$ (263)		
Processing	$ (840)	$ (842)	$ (842)	$ (842)	$ (842)	$ (842)	$ (842)	$ (842)	$ (842)	$ (842)	$ (842)	$ (149)	
G&A	$ (368)	$ (368)	$ (368)	$ (368)	$ (368)	$ (368)	$ (362)	$ (362)	$ (362)	$ (362)	$ (355)	$ (355)	
Total Operating Costs	$ (2,295)	$ (2,125)	$ (2,072)	$ (2,091)	$ (2,121)	$ (2,065)	$ (2,034)	$ (1,979)	$ (1,959)	$ (2,059)	$ (1,460)	$ (504)	
Other Expenses													
Reclamation costs											$ (1,250)		
Contribution Expense	$ (106)	$ (104)	$ (97)	$ (120)	$ (119)	$ (91)	$ (98)	$ (100)	$ (108)	$ (140)	$ (105)	$ (40)	
Ethiopia Head Office / In Country G&A	$ (37)	$ (33)	$ (38)	$ (33)	$ (37)	$ (33)	$ (38)	$ (33)	$ (37)	$ (33)	$ (38)	$ (33)	$ (11)
Total Other Expenses	$ (143)	$ (4,731)	$ (467)	$ (485)	$ (156)	$ (4,719)	$ (136)	$ (133)	$ (146)	$ (4,767)	$ (1,393)	$ (74)	$ (11)
Capital Investment													
Initial Capital													
Pre-Stripping													
Sustaining Capital			$ (332)	$ (332)									
Working Capital	$ 23	$ (94)	$ (1,075)	$ 493	$ 1,229	$ (431)	$ (294)	$ (926)	$ (973)	$ 816	$ 3,738	$ 1,505	$ (1,015)
Total Capital Investment	$ 23	$ (94)	$ (1,407)	$ 161	$ 1,229	$ (431)	$ (294)	$ (926)	$ (973)	$ 816	$ 3,738	$ 1,505	$ (1,015)
Pre-Tax Cashflow Analysis													
Total Operating Income	$ 2,852	$ (1,671)	$ 2,328	$ 3,416	$ 3,650	$ (2,228)	$ 2,747	$ 2,889	$ 3,308	$ 172	$ 2,381	$ 1,432	$ (6)
Pre-tax Net Present Value (8%)													
Pre-tax Internal Rate of Return													
Post Tax Cashflow Analysis													
Income Tax Payable				$ (915)				$ (1,792)				$ (2,279)	$ (952)
Post Tax Net Annual Cash Flow	$ 2,875	$ 2,829	$ 1,253	$ 2,995	$ 4,880	$ 1,935	$ 2,453	$ 171	$ 2,335	$ 5,582	$ 6,119	$ 659	$ (1,973)
Government Participation (Proclamation No. 816/2013)				$ (137)				$ (269)				$ (342)	$ (143)
Net Cash Flow - After Tax & Govt Participation	$ 2,875	$ 2,829	$ 1,253	$ 2,857	$ 4,880	$ 1,935	$ 2,453	$ (98)	$ 2,335	$ 5,582	$ 6,119	$ 317	$ (2,116)
Post Tax Net Present Value (8%)													
Post Tax Internal Rate of Return													
Post Tax Operating Payback													

23.0 ADJACENT PROPERTIES

As discussed in Section 4.0 and shown in Figure 4.2, the Harvest Project consists of concessions which cover an area larger than has been included in the Terakimti Gold Heap Leach Project.

In addition, EAM owns the Adyabo Project, which includes the Mato Bula and Da Tambuk deposits. The 100% owned Mato Bula and Da Tambuk deposits are high sulphidation gold rich VMS-submarine porphyry-related systems located roughly 18 km to the west of Terakimti.

No other information is available to EAM or Tetra Tech regarding mineral concessions or properties adjacent to or in the vicinity of the Terakimti deposit.

24.0 OTHER DATA AND RELEVANT INFORMATION

24.1 PROJECT DEVELOPMENT PLAN

To achieve the Terakimti Heap Leach Project schedule, the long-lead process equipment will need to be identified at the beginning of the detailed engineering stage. The critical path will be the supply and delivery of this equipment.

The early-start date is driven by the civil construction work. To achieve this schedule, several construction packages will need to be issued as unit rate packages. The unit rate packages will include rough grading, concrete and structural steel buildings, and interior steel platforms.

Upon construction commencement, the temporary construction facilities will be mobilized, including the batch plant and aggregate plant. Site preparation, grading, and road construction will commence immediately upon receipt of permits and approvals. Modular construction will be utilized wherever practical to reduce field construction.

Upon completion of foundation preparation, the concrete for the main process building, truck shop, and powerhouse building foundations will be poured to allow the buildings to be erected. Once the buildings are erected, the concrete inside the buildings (including equipment supports) can be poured.

Electrical and mechanical installation contracts will be bid lump sum to qualified contractors. A start-up and commissioning period has been allowed at the completion of construction in order to complete mechanical check out and acceptance and commissioning of the facilities.

A conceptual summary schedule for the Terakimti Heap Leach Project is shown in Figure 24.1.

Figure 24.1 Conceptual Project Summary Schedule

	Year -2		Year -1				Year 1			
Quarter	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4
Detailed Engineering and Procurement	<<		>>							
Construction			<<		>>					
Mine prestripping and initial heap leach feed					<<	>>				
Mechanical completion						<<>>				
Commission heap leach						<<>>				
Commission Merril-Crowe						<<>>				
Start operations						<<				
First Pour of Doré							<<>>			

25.0 INTERPRETATIONS AND CONCLUSIONS

25.1 GEOLOGY

The Harvest Property has undergone extensive exploration using traditional and modern exploration techniques since the concessions were granted. During TRIs initial involvement at the Harvest Property (2011 through 2013) and subsequently EAMs, the volume and quality of the data has grown considerably. Detailed mapping combined with traditional gold soil geochemistry and portable XRF soil sampling is particularly successful in identifying anomalous areas with respect to gold and base metals in all of the concessions.

An airborne EM geophysical survey carried out in 2012 identified VTEM anomalies and magnetic lineaments that, when interpreted using geological mapping information, resulted in the production of a more accurate geology map, and most importantly the identification of new exploration targets. These targets are focus of the company's grassroots activities while the more advanced projects at Terakimti, VTEM09 and Mayshehagne see exploration in the form of core drilling and ground geophysics.

The majority of expenditure at the Harvest property related to the drilling testing at VMS prospects of Terakimti, Mayshehagne and VTEM09. The Mayshehagne and VTEM09 prospects appear to be isolated VMS lenses of limited surface extent, although both structures remain open at depth. The most substantial copper-gold-zinc-silver mineralization on the property is located at Terakimti, where four massive sulphide lenses are present. Typical primary sulphide grades are 0.5 to 3% copper, 0.5 to 2 g/t gold, 15 to 30 g/t silver, and 0.5 to 4% zinc (see Table 10.3 for precise grades). Oxide zone grades are much higher and can attain grades up to 27.2 g/t gold and 157 g/t gold. Drilling indicates that the mineralization is open down-plunge, although later northwest-trending quartz-feldspar porphyry intrusions have cut through several of the lenses in one part of the system.

In addition to the VMS mineralization, a major zone of gold mineralization was identified approximately 1.5 km west of Terakimti at Ruwa Ruwa. The Ruwa-Ruwa Trend contains several bedrock gold prospects and abundant artisanal eluvial and alluvial working over a distance of 7 km. These include Ruwa Ruwa, Adi Goshu, and Lihamat and are all classified as orogenic lode gold mineralization. The Lihamat area targets have been drill tested and have yield erratic localized high gold grade intersections.

The author concludes that the Harvest Property hosts a VMS system at Terakimti that has not been fully delineated. The discovery of VMS mineralization on the same scale as Terakimti is likely since VMS systems generally occur in clusters. However, to date, the prospects on gossanous trends other than Terakimti/VTEM09, and Mayshehagne, have not produced similar grades and widths. Orogenic lode-gold occurrences on the Ruwa

Ruwa Trend indicate the potential for economic gold mineralization on the Harvest Property, however surface expressions of mineralization appear to have limited spatial extent. The geologic understanding of the deposit settings, lithologies, and structural and alteration controls on mineralization is sufficient to support estimation of Mineral Resources. The mineralization style and setting is well understood and is sufficient to support Mineral Resource estimation. The exploration programs completed to date are appropriate to the style of mineralization found in the deposit.

Sampling methods are acceptable, meet industry-standard practice, and are acceptable for Mineral Resource estimation purposes. The quality of the Terakimti drill core, and channel analytical data is reliable and sample preparation, analysis, and security are generally performed in accordance with exploration best practices and industry standards

The Terakimti Mineral Resource has reasonable prospects of economic extraction and therefore further exploration is warranted. Mineral Resources, which were estimated using core drilling, reverse circulation drilling and trenches have been performed to industry practices, and conform to the requirements of CIM Definition Standards (2014).

Metallurgical recovery assumptions used to support reasonable prospects of economic extraction were based on the recoveries estimated from material collected from the Terakimti deposit. These were based on a small program of tests conducted on 16 sub-composites, two being oxide and fourteen being sulphide.

25.2 MINING

The preliminary assessment of mining the Terakimti open pit shows that the mining will be a relatively simple open pit for which only limited further work is required to advance the Terakimti Heap Leach Project further.

Some opportunities that may improve the economic results that could be further investigated include:

- campaign mining with the aim of reducing mining costs
- steepening pit walls, if geotechnical conditions allow
- further work in evaluating the amount of rock that could be free dug as opposed to blasting.

25.3 METALLURGY AND PROCESS

For the mineralization of the oxide zones of the Terakimti deposit, the preliminary test results indicate that the mineral samples responded well to cyanide leaching, including column leach treatment. In general, the initial leach kinetics for the samples tested are rapid. The gold in the mineralization shows better metallurgical performance, compared to the silver. There are significant variations in gold and silver extractions among the samples collected from different spatial locations and with lithological types. The mineralization is soft to ball mill grinding and has low abrasion characteristics. Due to a potentially high content in the fine material, cement or bonding material dosage needs to be further confirmed to avoid the potential issues associated with a low permeability when the leach pad is high.

The processing method proposed for the project is gold and silver extraction by heap leach processing following by Merrill Crowe treatment to recover the dissolved gold and silver from the pregnant solution. The processing has been widely used in gold and silver recovery from various oxide gold ores. The equipment proposed for the project are relatively small and can be supplied globally. It is anticipated that the leading time for the major processing equipment would be reasonable.

25.4 INFRASTRUCTURE

The infrastructure required to support the mining and processing operations will include the following:

- maintenance workshop, steel frame structure including container storage and blacksmith shops
- explosive storage magazines
- administrative building with emergency first room
- lunch room, washrooms and change rooms for employees
- assay laboratory and supplies warehouse
- lighting equipment
- substation and power distribution
- water supply
- heap leach pad, adsorption/desorption recovery plant and solution ponds
- WRD
- access and site roads.

25.5 COSTS

Tetra Tech estimated a total capital cost of US$18.9 million, a total operating cost of US$37 million, with other costs such as country management, reclamation and community and social responsibility expenses totalling US$5 million.

The Terakimti Heap Leach Project is most sensitive to operating costs and less sensitive to capital costs.

25.6 ECONOMICS

The financial results show that the Terakimti Heap Leach Project has potential to produce positive economic results, with a post-tax cash flow of US$22 million, a post-tax NPV of US$13 million and post-tax IRR of 30.5%.

While the financial results will vary with commodity pricing or assumptions used in the cost estimate. The economic assessment indicates that the Terakimti Heap Leach Project has the potential to be relatively robust.

26.0 RECOMMENDATIONS

26.1 GENERAL

Tetra Tech recommends that EAM advance the Terakimti Heap Leach Project through completion of a Feasibility Study.

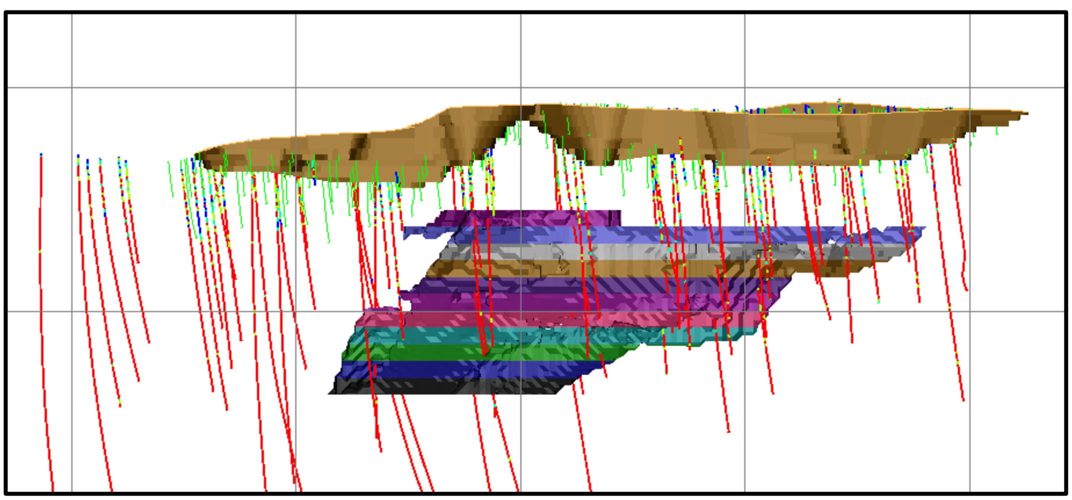
26.1.1 POTENTIAL EXPANSION INTO SUPERGENE AND HYPOGENE

Tetra Tech conducted a brief review of potential mining of the supergene and hypogene zones of the Terakimti deposit. This brief review included expansion of the planned oxide pit considered for this PEA as well as underground mining. The initial results show that further work is justified.

The preliminary evaluation included the use of flotation milling to recover gold, copper, silver, zinc and lead.

The preliminary underground areas evaluated are shown in Figure 26.1

Figure 26.1 Supergene and Hypogene Potential below the Terakimti Oxide Deposit



26.2 GEOLOGY

It is recommended that the path forward for the Harvest Property should include the following main activities during the next two phases of the Terakimti Heap Leach Project. These contingent phases are:

26.2.1 PHASE I

Infill drilling of the Terakimti supergene and primary mineralization are recommended to fully assess the Terakimti deposit. Key prospects on the property remain to be fully investigated and exploration work should continue to try and identify additional resources to complement those now identified. Supplementary work is required at the known VMS prospects to qualify the potential of the existing mineralization. This work would include deep EM testing and metallurgical work. Specifically, the recommended work program will include:

- infill diamond drill test primary and supergene mineralization at Terakimti
- carry out detailed metallurgical testing of Terakimti primary and supergene mineralization
- conduct both downhole EM (Terakimti/Mayshehagne/VTEM09), and deep ground EM testing on the VMS corridors identified at Terakimti/VTEM09 and Mayshehagne, to fully assess additional resource potential to depth
- perform detailed sequential leach analyses on existing drill samples, to assess supergene zonation
- conduct metallurgical testing at Mayshehagne and VTEM09, to ascertain resource assessment potential.

26.2.2 PHASE II

If positive results are achieved in Phase I, a second phase of work should be undertaken to further refine the mineral resources identified. This program will include:

- An update to the current Mineral Resource estimate at Terakimti
- Resource assessments at Mayshehagne, VTEM09.
- Assess potential for extension drilling at Mayshehagne, VTEM09, and any targets potentially identified through EM work.

In total, the cost of this work is expected to be up to approximately US$2,910,000. A summary of the expenditure break-down is presented in Figure 26.1.

Table 26.1 Summary of Expenditure

Phase	Description of Work	Cost (US$)
1	Infill diamond drilling of Terakimti primary and supergene mineralization	1,700,000
	Detailed metallurgical testing, Terakimti	75,000
	Downhole and Deep EM testing	400,000
	Sequential Copper leach analyses	25,000
	Metallurgical testing, VTEM09, Mayshehagne	40,000
	Subtotal Phase 1	**2,240,000**
2	Terakimti Resource update	40,000
	Mayshehagne, VTEM09 resource assessment	30,000
	EM identified extension/target drill testing	600,000
	Subtotal Phase 2	**670,000**
Total		**2,910,000**

26.3 MINING

Tetra Tech recommends that a geotechnical study is completed for the open pit to better understand the final pit wall slopes required to ensure stability.

To conduct geotechnical drilling and geotechnical analysis is expected to cost US$50,000.

26.4 METALLURGY AND PROCESSING

Further tests are recommended to evaluate the metallurgical performances of the mineralization, including the variability tests and column leach tests. The test work should be conducted on the samples that better represent the mineralization of the Terakimti deposit, which is planned for the heap leach treatment. The test work should include:

- head characterization and mineralogical determination
- leaching condition optimization, including cyanide concentration, leaching retention time, agglomeration binding material types and dosages
- determination of the effect of the particle size distribution on gold and silver extraction and on the leach pad permeability
- residual cyanide management tests, including residual cyanide management and valuable metal recoveries from the barren solution
- design related parameters should be determined, such as bulk density, specific gravity, agglomerate strength measurement.

The estimated costs for the test work, excluding sampling, are approximately US$150,000.

The leach pad arrangement, equipment sizing and overall plant layout should be further optimized. Overall water balance, especially water balance at the leach pad areas, and makeup water resources should be investigated, especially the water supply during the dry seasons. These studies will be a part of next phase studies.

26.5 INFRASTRUCTURE

Tetra Tech recommends that EAM further investigate the sources of electrical power and water. The site visit (by QP M. Horan) indicated that there is power along the paved road between Shire and Adi Dairo, however further details of tap points and alignment of power lines has not yet been concluded. Similarly, with water sources, the site visit indicated the presence of water source to the southwest of the mine site but water rights, accessibility and ideal sources has not yet been determined.

There is no estimate for this work, though EAM may need to engage with a local electrical company to conduct engineering of power lines, this may cost as much as US$50,000 for more detailed work.

26.6 CAPITAL COSTS

Tetra Tech recommends that EAM approach additional vendors of equipment to look for opportunities to reduce costs. The cost of this work is not expected to exceed US$20,000.

27.0 REFERENCES

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DAVID G. THOMAS, P.GEO.

I, David G. Thomas, P. Geo., of #601 – 1788 West Georgia Street, Vancouver, British Columbia, Canada, do hereby certify that:

- I am the principal mineral resource geologist and owner of the geological consulting firm DKT Geosolutions Inc.

- This certificate applies to the technical report entitled 'Technical Report and Preliminary Economic Assessment for the Terakimti Oxide Deposit, Harvest Project, Tigray National Regional State, Ethiopia' with an effective date of April 30th, 2018 (this "Technical Report") that was prepared for the Issuer.

- I am a graduate of Durham University, in the United Kingdom with a Bachelor of Science degree in Geology and am a graduate of Imperial College, University of London, in the United Kingdom with a Master of Science degree in Mineral Exploration.

- I have practiced my profession for over 23 years. In that time, I have been directly involved in reviews of exploration programs, geological models, exploration data, sampling, sample preparation, quality assurance-quality control, databases, and mineral resource estimates for a variety of mineral deposits, including VMS deposits and other copper-gold deposit types (Canada, Ethiopia and Eritrea).

- I am a member in good standing of the Association of Professional Geoscientists of British Columbia (APEGBC NRL # 149114). I am also a member of the Australasian Institute of Mining and Metallurgy (MAusIMM # 225250).

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

- I most recently visited the subject property from March 22 to March 25, 2015.

- I am responsible for Sections 1.3, 1.4, 1.5, 1.13.2, 7.0, 8.0, 9.0, 10.0, 11.0, 12.0, 14.0, 25.1, 26.2, and 27.0 of the Technical Report.

- I am independent of the Issuer applying all the tests in Section 1.5 of NI 43-101.

- I have had prior involvement with the property that is the subject of this Technical Report. I completed the initial Mineral Resource estimate at Terakimti (filed on SEDAR, February 14, 2014) and updated the oxide Mineral Resource estimate (disclosed in an EAM press release dated, October 27, 2015)

- I have read NI 43-101 and NI 43-101F1 and this Technical Report has been prepared in compliance with that instrument and form.

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

// Signed and Sealed //

David G. Thomas, P.Geo.

DATED at Bogota, Colombia, this 11th day of June 2018.

HASSAN GHAFFARI, P.ENG.

I, Hassan Ghaffari, P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am a Director of Metallurgy with Tetra Tech Canada Inc. located at Suite 1000, 10th Floor, 885 Dunsmuir Street, Vancouver, British Columbia, V6C 1N5.
- This certificate applies to the technical report entitled "Technical Report and Preliminary Economic Assessment for the Terakimti Oxide Deposit, Harvest Project, Tigray National Regional State, Ethiopia" dated April 30th, 2018 (the "Technical Report").
- I am a graduate of the University of Tehran (M.A.Sc., Mining Engineering, 1990) and the University of British Columbia (M.A.Sc., Mineral Process Engineering, 2004). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#30408). My relevant experience includes 27 years of experience in mining and plant operation, project studies, management, and engineering. As the lead metallurgist for the Pebble Copper/Gold Moly Project in Alaska, I am coordinating all metallurgical test work and preparing and peer reviewing the technical report and the operating and capital costs of the plant and infrastructure for both the scoping and prefeasibility studies. For the Ajax Copper-Gold Project in BC, I was the Project Manager responsible for the process, infrastructure and overall management of the 60,000 t/d mill. As well, I was the Project Manager responsible for ongoing metallurgical test work and technical assistance for the La Joya Project Copper/Silver/Gold Project in Durango, Mexico. I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- I have not completed a personal inspection of the Property that is the subject of this Technical Report.
- I am responsible for Sections 1.10, 3.0, 20.0, 21.2, and 27.0 of the Technical Report.
- I am independent of East Africa Metals Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property.
- 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 contains all of the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 8th day of June 2018 at Vancouver, British Columbia.

"Original document signed and sealed by
Hassan Ghaffari, P.Eng."

Hassan Ghaffari, P.Eng.
Director of Metallurgy
Tetra Tech Canada Inc.

JIANHUI (JOHN) HUANG, PH.D., P.ENG.

I, Jianhui (John) Huang, Ph.D., P.Eng., of Coquitlam, British Columbia, do hereby certify:

- I am a Senior Metallurgist with Tetra Tech Canada Inc. located at Suite 1000, 10th Floor, 885 Dunsmuir Street, Vancouver, British Columbia, V6C 1N5.
- This certificate applies to the technical report entitled "Technical Report and Preliminary Economic Assessment for the Terakimti Oxide Deposit, Harvest Project, Tigray National Regional State, Ethiopia" dated April 30th, 2018 (the "Technical Report").
- I am a graduate of North-East University, China (B.Eng., 1982), Beijing General Research Institute for Non-ferrous Metals, China (M.Eng., 1988), and Birmingham University, United Kingdom (Ph.D., 2000). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#30898). My relevant experience includes over 35 years involvement in mineral processing for base metal ores, gold and silver ores, and rare metal ores. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I have not completed a personal inspection of the Property that is the subject of this Technical Report.
- I am responsible for Sections 1.6, 1.13.4. 13.0, 17.0, 21.3.3, 25.3, 26.4 and 27.0 of the Technical Report.
- I am independent of East Africa Metals Inc. as defined by Section 1.5 of the Instrument.
- My prior involvement with the Property includes involvement in some internal studies since 2015.
- 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 contains all of the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 8th day of June 2018 at Vancouver, British Columbia.

"Original document signed and sealed
Jianhui (John) Huang, Ph.D., P.Eng."

Jianhui (John) Huang, Ph.D., P.Eng.
Senior Metallurgist
Tetra Tech Canada Inc.

MARK HORAN, P.ENG.

I, Mark Horan, P.Eng., of North Vancouver, British Columbia, do hereby certify:

- I am a Senior Mining Engineer with Tetra Tech Canada Inc. located at Suite 1000, 10th Floor, 885 Dunsmuir Street, Vancouver, British Columbia, V6C 1N5.
- This certificate applies to the technical report entitled "Technical Report and Preliminary Economic Assessment for the Terakimti Oxide Deposit, Harvest Project, Tigray National Regional State, Ethiopia" dated April 30th, 2018 (the "Technical Report").
- I have a BSc. Mining Engineering degree from the University of Witwatersrand, South Africa and a MSc. from Rhodes University, South Africa. I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#170768). I have 18 years' experience including working in precious and base metal operations and in consulting. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property that is the subject of this Technical Report was on April 6th, 2017, for one day.
- I am responsible for Sections 1.1, 1.2, 1.7, 1.8, 1.9, 1.11, 1.12, 1.13.1, 1.13.3, 1.13.5, 1.13.6, 2.0, 3.0, 4.0, 5.0, 6.0, 15.0, 16.0, 18.0, 19.0, 21.1, 21.3.1, 21.3.2, 21.3.4, 22.0, 23.0, 24.0, 25.2, 25.4, 25.5, 25.6, 26.1, 26.3, 26.5, 26.6 of the Technical Report.
- I am independent of East Africa Metals Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property.
- 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 contains all of the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 8th day of June 2018 at Vancouver, British Columbia.

"Original document signed and sealed by
Mark Horan, P.Eng."

Mark Horan, P.Eng.
Senior Mining Engineer
Tetra Tech Canada Inc.