Tasiast Project
Mauritania
National Instrument 43-101 Technical Report

Prepared for:
Kinross Gold Corporation

Prepared by:
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Effective Date: 31 March 2014

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1. SUMMARY

1.1 Executive Summary

Kinross has prepared a Technical Report for the wholly-owned Tasiast gold deposit (the Project) located in the Islamic Republic of Mauritania (Mauritania), Africa. Kinross is using this Technical Report to support disclosure of mineral resources and mineral reserves at the Project. The Technical Report conforms to National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) and has an effective date of March 31, 2014. Kinross will be using this Technical Report to support disclosure of mineral resources and mineral reserves at the Project.

Following a prefeasibility study (PFS) that was completed in May 2013, Kinross proceeded with a feasibility study (FS) on a 38,000 tonne/day (38 kt/d) carbon-in-leach (CIL) plant expansion, with an on-site heavy fuel oil (HFO) power plant. The Kinross FS was completed in March 2014. The Project is a brownfield expansion and the 38 kt/d CIL process plant would replace the existing 8 kt/d CIL plant. The existing process plant would continue operating during the construction of the new 38 kt/d CIL plant, but would cease to operate after the construction.

Although a construction decision on the 38 kt/d expansion will not be made until 2015 at the earliest, the FS supports an updated mineral reserve based on a reasonable expectation of proceeding with the expansion. It is important to note that the 2014 and 2015 mine plans for the 8 kt/d and 38 kt/d cases are identical.

The Project is located in north-western Mauritania and contains several gold deposits. The Tasiast mine and the mining permit are owned by Tasiast Mauritanie Limited S.A. (TMLSA). Two sister companies of TMLSA hold adjacent permits. Société d'Extraction du Nord de l'Inchiri S.A. (SENISA) holds two recently converted mining permits (for the Tmeimichat and Imkebdene areas) and Tasiast Mauritanie Limited (TML) holds two exploration permits (Tasiast Sud and N'Daouas-Est areas). All four are contiguous to the Tasiast mining permit lands (collectively, the Tasiast Lands). As part of the recently completed conversion process of two exploration permits, Kinross has undertaken to transfer to the Government of Mauritania a 10% carried interest in SENISA. Other than SENISA, all permit-holding affiliates of Kinross, including TMLSA, are wholly-owned indirect subsidiaries of Kinross. Kinross acquired TMLSA, including the Tasiast operation and exploration permits and lands, through its acquisition of Red Back in September 2010.

There are exploration prospects in the 312 km^2 El Gaîcha Mining permit and in the surrounding permits. Commercial production of gold at Tasiast began in January 2008, and a total of 1,139,275 oz. was produced by the end of 2013.

Most sections of this report are equally applicable to both the 8 kt/d and 38 kt/d cases. Where it is not appropriate to have both cases covered in the same section, the section applicable to 8 kt/d occurs first (with the exception of Sections 14 and 15). Additional sections on specific aspects of the 38 kt/d case are found in Section 24 - Other Relevant Data and Information, with subsection numbers corresponding to the main section number per NI 43-101. For example, Section 16 has Mining Methods for 8 kt/d, and Section 24.16 has Mining Methods for 38 kt/d. The report sections and applicable cases are shown in Table 1-1.

Table 1-1: Technical Report sections and applicability to the 8 kt/d and 38 kt/d cases

Item (section heading required under NI 43-101)	Applicable to 8 kt/d	Applicable to 38 kt/d
1 SUMMARY	●	●
2 INTRODUCTION	●	●
3 RELIANCE ON OTHER EXPERTS	●	●
4 PROPERTY DESCRIPTION & LOCATION	●	●
5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE & PHYSIOGRAPHY	●	●
6 HISTORY	●	●
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9 EXPLORATION	●	●
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11 SAMPLE PREPARATION, ANALYSES & SECURITY	●	●
12 DATA VERIFICATION	●	●
13 MINERAL PROCESSING & METALLURGICAL TESTING	●	●
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17 RECOVERY METHODS	●	
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21 CAPITAL & OPERATING COSTS	●	
22 ECONOMIC ANALYSIS	●	
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25 INTERPRETATION & CONCLUSIONS	●	●
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27 REFERENCES	●	●
28 DATE & SIGNATURE PAGE	●	●

1.2 Technical Summary

Property Description, Location and Land Tenure

The Project is located in northwestern Mauritania, approximately 300 km north of the capital Nouakchott and 250 km southeast of the major city of Nouâdhibou. The Tasiast Lands are accessed from Nouakchott by using the paved Nouakchott to Nouâdhibou highway for 370 km and then via 66 km of graded mine access road, which is maintained by TMLSA. There is an airstrip at the mine site that is used for light aircraft travelling to and from Nouakchott.

Mining operations commenced in 2007, with commercial production reached in January 2008. Infrastructure on site supports an open pit mining operation and associated processing facilities consisting of a 3 Mt/a carbon-in-leach (CIL) mill and a run of mine (ROM) dump leach.

TMLSA holds a valid mining permit, PE 229 (El Gaîcha), covering 312 km^2 granted in January 2004 and valid for a period of 30 years. The mining operations and infrastructure lie entirely within the lands subject to the mining permit. There are also four additional contiguous permits (3,118 km^2), each of which is in good standing. The Tasiast Lands fall within the administrative purview of the Inchiri and Dakhlet Nouâdhibou Districts, with PE 229 within the Inchiri District only.

Surface and Water Rights

Surface rights are granted along with permit PE 229 and are paid annually as determined by decree under Section 107 of the Mauritanian Mining Code Act No. 2008-011 (Mining Code). Surface rights for the permit are in good standing.

Three iron ore exploration permits overlap mining permit PE 229, entitling their holders to do exploration works, as long as they do not interfere with TMLSA's operations. TMLSA does not have any obligation to accommodate the holders of these permits.

These holders are not entitled to transform these explorations permits into mining permits on the overlapping area without TMLSA's prior written approval, and they are not entitled to any compensation from TMLSA.

The operation's water supply is located 64 km west of the mine and consists of a semi-saline underground aquifer, which is exploited by 47 wells. Water is pumped from the well field to the mine.

Royalties

A royalty equal to 3% of the gross revenue of TMLSA is payable to the government. In addition, Franco-Nevada Corporation (Franco-Nevada) holds a 2% net royalty on gold production in excess of 600,000 ounces. Production at Tasiast reached 600,000 ounces in July 2011 and the first royalty payment to Franco-Nevada was made in October 2011.

Permits

Exploration, development and mining activities to date have been performed under the appropriate permits, laws and regulations.

Environment

An environmental impact statement (EIS) was completed for the Project in 2004 and subsequently approved by the Director of Mines and Geology on 12 April 2005. To meet revised legislation, additional reporting was requested, which was completed in 2008. In 2009, an additional environment assessment was conducted for a second tailings storage facility, a dump leach facility and an expansion of the water bore field. The West Branch expansion was assessed in 2010, and subsequently approved in 2011.

Since acquiring Tasiast, in support of the expansion project, TMLSA has completed significant permitting activities including Phase 1 (2 Environmental Impact Notices (EINs) and 1 Environmental Impact Assessment (EIA)) in 2011 and Phase 2 (EIA for all on-site proposed expansion activities) in 2012. Phase 3 EIA for “off-site” sea water supply has been conditionally approved pending approval of additional information which has been submitted. A final EIA is under development to authorize the delivery of preassembled equipment to site.

Current environmental liabilities are those that would be expected from a mining operation, and include the mine, crushing and CIL processing plant, dump leach facilities, power plant, tailings and waste rock facilities, power grids, roads, accommodation camp, ancillary facilities and drill pads established to support mining and exploration activities.

Geology and Mineralization

The Tasiast Lands consist of three main Precambrian greenstone belts located in the western compartment of the Reguibat Shield. The Reguibat Shield consists of a series of west to east accreted, north-south trending Archaean and Lower Proterozoic metavolcano-sedimentary belts and domal basement gneiss complexes.

The Tasiast Lands are underlain by the Aouéouat greenstone belt, a north-south trending belt that is continuous for 75 km strike length on the Tasiast Lands and that may continue further to the north and south. The mine geology is characterized by a mafic to felsic metavolcano-sedimentary succession that is overlain by an iron stone formation and epiclastic units. The rocks have undergone deformation, were metamorphosed to greenschist and lower amphibolite grades and were cut by volumetrically minor younger mafic dikes. Three main prospective trends are recognized at the property with all known deposits spatially associated with the Tasiast trend. Other trends also contain gold occurrences but have been significantly under-explored relative to the Tasiast trend.

Known deposits are aligned along a north-trending corridor with a strike length greater than 10 km, with the Piment deposits at the northern half of the mine area and West Branch deposits at the southern half. At West Branch, first-order structural controls on mineralization include several subparallel anastomosing faults and several generations of veins developed predominantly in altered mafic meta-igneous and metavolcanic units locally called the Greenschist Zone. Mineralization at Piment is principally controlled by several anastomosing faults developed within the hanging wall block of iron formation, felsic metavolcanic and epiclastic rocks. Veins are spatially associated with mineralization over horizontal widths of up to 20 m.

Gold mineralization has been defined over a strike length of greater than 10 km and to vertical depths of at least 740 m. All of the significant mineralized bodies defined to date dip moderately (45º to 60º) to the east and have a south-southeasterly plunge. Most of the gold mineralization at West Branch is hosted in hydrothermally altered meta-igneous rocks (Greenschist zone) containing quartz-carbonate veins. The meta-igneous rocks are enveloped by felsic units known as felsites that occur on the footwall and hanging wall sides of the Greenschist zone. The Greenschist zone is characterized by consistently thick intervals of mineralization averaging 40 m to 100 m thick. Individual shoots are continuous over a strike length of at least 1,000 m. Mineralogy within the Greenschist package is dominated by pyrrhotite, pyrite and native gold that occur as vein infill or alteration spots commonly in and around the foliation. Pyrrhotite and pyrite occur together in many places but in variable ratios. Zones of pyrite-only and pyrrhotite-only sulphide facies are rare.

Piment mineralization is largely hosted along fault splays and within the adjacent altered and veined iron formation and epiclastic units. Individual mineralized shoots are continuous over 300 m and to vertical depths of at least 260 m. The minerals associated with gold at Piment are pyrrhotite and pyrite.

History and Exploration

Exploration programs have included geological and regolith mapping, satellite image interpretation, airborne and ground magnetic geophysical surveys, soil, rock chip, and grab geochemical sampling, trenching, reverse circulation (RC) and core drilling, engineering studies, metallurgical test work, and specialist geological studies such as ore and alteration petrography. Work was completed by the *Office Mauritanien de Recherches Géologiques* (OMRG), Normandy LaSource Development Ltd. (NLSD), Midas Gold plc. (Midas), Geomaque Explorations Inc. (Geomaque), Defiance Mining Corporation (Defiance), Rio Narcea Gold Mines Ltd. (Rio Narcea), Red Back and Kinross.

Exploration Potential

Mineralization remains open both laterally and vertically at the Tasiast deposits. Shallow reconnaissance RC drilling across district targets has detected gold mineralization that requires follow-up exploration. The potential to delineate additional resources both at the Tasiast mine and the district targets is considered good. Some of these district targets are described below.

Drilling uncovered encouraging gold results at the C67-Fennec sector and Tasiast Sud (previously named Charlize) targets, located 5 km north and 15 km south of Tasiast. Drilling has also intersected encouraging gold grades at the Tmeimichat and Imkebdene permitted areas, situated approximately 10 km north of the mine site.

At the Tasiast Sud targets, mineralization occurs near the surface in banded iron/magnetite (BIM) rocks that are part of the same mineralized BIM sequence further north at Tasiast. Mineralization has also been identified in sheared amphibolites that host quartz-carbonate veins where they are thrust over younger metasedimentary rocks. Gold mineralization at C67 is hosted by sheared and veined mafic metavolcanic rocks and by quartz diorites that may be intrusive into the metavolcanics. The gold is associated with quartz-carbonate veins that formed during the late stages of shearing. Besides quartz and carbonates, the vein mineralogy consists of pyrrhotite, pyrite, biotite and locally tourmaline. Pyrrhotite is generally more abundant than pyrite where significant gold grades are present. The structures controlling mineralization are moderately to steeply west-dipping shear zones that define a trend that is parallel to, but to the east of the Tasiast trend.

The Project area has considerable additional exploration potential as illustrated by the extensive gold anomalies in soil maps created by TMLSA. Several areas have been identified by TMLSA on the current Tasiast permits for near term drill testing. Targeting incorporates all available data sets including satellite imagery (Worldview-2), regional gold-in-soil data, airborne geophysical data (high resolution aeromagnetics and VTEM), regional scale geological maps of two generations, several generations of target scale geological maps, multi-element geochemical data derived from RC drill chips, assay results derived from historical and recent property scale drilling and trenching, reconnaissance geological observations and prospecting.

Drilling

The total number of drill holes completed on the Project totals 15,141 holes (43 rotary air blast (RAB), 14,133 RC and 762 core) for an aggregate total of 1,622,722 m. Resource drilling campaigns completed between 1999 and 2013 comprises 3,890 RC (620,106 m) and 473 core holes (207,803 m) for a total of approximately 827,909 m.

Drill programs were completed primarily by contract drill crews, supervised by geological staff of the Project operator. Where programs are referred to by company name, that company was the Project manager at the time of drilling and was responsible for the collection of data. Collar locations were surveyed by site surveyors using DGPS instruments. Down-hole surveys were mostly (+65%) completed using single shot reflex and north seeking gyroscope instruments. Core recoveries are typically greater than 93%.

Mineral Resource

The Tasiast mineral resource statement, as of year-end 2013, comprises Measured, Indicated and Inferred resources. Mineral resources were classified in accordance with the 2010 CIM Definition Standards for Mineral Resources and Mineral Reserves, incorporated by reference into NI 43-101. Mineral resources have an effective date of December 31, 2013.

Resources are stated at variable cutoff grades, dependent on the metallurgical type, mining operating cost and variable process recoveries which were based on metallurgical type. The cutoff grade for CIL ore types was 0.37 g/t Au and 0.22 g/t Au for leach ore types. The cutoff grades were determined using a gold price of US$ 1,400/oz. For the purposes of reporting the resource, the Leach ore types were reported within the Upper Transition and Oxide. The CIL ore types were reported within the Lower Transition and Fresh. The codes for these corresponding units are classified in the OX block model.

The mineral resources were reported below the projected December 31, 2013 mined surface and are constrained using the Lerchs-Grossman (LG) 38 kt/d pit shell designed by Kinross Technical Services. Kinross cautions that mineral resources that are not mineral reserves do not have demonstrated economic viability.

Table 1-2 shows the classified mineral resources, exclusive of reserves.

Table 1-2: 2013 Tasiast Mineral Resource Statement Exclusive of Reserves

Classification	Tonnes (000's)	Grade (Au g/t)	Ounces (000's)
Measured	53,889	0.64	1,103
Indicated	120,722	0.93	3,603
Subtotal M&I	**174,611**	**0.84**	**4,706**
Resource Stockpile	20,234	0.27	178
Inferred	14,146	1.46	664

1. Mineral Resources that are not Mineral Reserves are not required to demonstrate economic viability.
2. Mineral Resources are reported below the 31 December 2013 mined surface and are constrained using an optimized Lerchs-Grossman pit shell.
3. Mineral Resources are reported to a cut-off grade of 0.37 g/t Au for CIL and 0.22 g/t Au for dump leach based on a gold price of US$1,400/oz.
4. Stockpile balance above 1.1 g/t cut-off grade is considered as reserve, while below 1.1 g/t is considered as resource. Measured Resource includes the Resource Stockpile.
5. The above mineral resource estimate is classified in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum's "CIM Definition Standards - For Mineral Resources and Mineral Reserves" incorporated by reference into National Instrument 43-101 "Standards of Disclosure for Mineral Projects".

Estimation of the Tasiast mineral resources was based on a combination of development alternatives that assumed two potential processing options. These included CIL milling of fresh and high grade oxide resource and dump leaching for lower grade oxide material. The various processing recoveries and associated operating conditions were used to generate an optimized pit shell using an LG optimization algorithm.

Mineral Reserve

Mineral reserves for the Project incorporate appropriate allowances for mining dilution and mining recovery for the selected mining method. Mineral reserves have an effective date of December 31, 2013 and are summarized in Table 1-3.

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Table 1-3: 38 kt/d Reserve Estimate Effective December 31, 2013.

Classification	Tonnes (000's)	Grade (Au g/t)	Ounces (000's)
Proven	34,029	1.33	1,453
Probable	141,504	1.80	8,191
TOTAL	**175,533**	**1.71**	**9,644**
Reserve Stockpile	2,049	1.31	86

1. Mineral Reserves are reported to a cut-off grade of 0.6 g/t Au for CIL and 0.40 g/t Au for dump leach based on a gold price of US$1,200/oz less 5% gross royalty.
2. Stockpile balance above 1.1 g/t cut-off grade is considered as reserve, while below 1.1 g/t is considered as resource. Proven Reserve includes the Reserve Stockpile.
3. The Reserves Estimate assumes the construction of a 38 kt/d mill.
4. The above mineral reserve estimate is classified in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum's "CIM Definition Standards - For Mineral Resources and Mineral Reserves" incorporated by reference into National Instrument 43-101 "Standards of Disclosure for Mineral Projects".

Mining Operations

The Tasiast mine is a conventional open pit truck-and-shovel operation that would move up to 50-72 Mt/a over a 17 year mine life. The current mining operation sustains a 3 Mt/a ore feed to the existing carbon-in-leach (CIL) process plant, plus low-grade oxide ore feed to a dump leach process. Reclaiming stockpiles adds an additional two years of production through the CIL plant after mining is finished. The introduction of larger mining equipment has shifted the mining strategy from a highly selective mining practice to a combination of both bulk and selective mining. The design operating bench height is 10 m in West Branch and 5 m in the Piment pits. If required, sections of the footwall zone in West Branch may be mined on 5 m benches to improve ore selectivity.

Recovery Methods

Kinross would close down operation of the existing Tasiast 8 kt/d CIL plant before start-up of the new expansion plant. Operating and maintenance staff would be transferred to the new plant. The existing plant would be shut down in an orderly fashion, including cleaning out of all gold removed from the carbon and from the mills, tanks and other equipment.

The design capacities for the crushing plant and process plant use 70% and 92% effective operating time. The plant design life is 15 years.

Based on the proven and probable mineral reserve estimate presented in this Technical Report, the mine would continue production until 2027 at a processing rate of 38 kt/d. Once mining operations have been completed, the CIL plant would continue operating to process low grade stockpiles that will have been developed during the course of mining. This would be expected to continue until 2029.

Market Studies and Contracts

Kinross typically establishes refining agreements with third-parties for refining of doré. Kinross's bullion is sold on the spot market, by marketing experts retained in-house by Kinross. The terms contained within the sales contracts are typical and consistent with standard industry practice, and are similar to contracts for the supply of doré elsewhere in the world.

Capital and Operating Costs

For the 38 kt/d case, the total going forward capital cost estimate is $1,566 million. The scope for this expansion capital cost estimate includes mining; a new CIL process plant; expansion of the existing adsorption, desorption and recovery (ADR) plant; the Phase 2 power plant; site infrastructure facilities and utilities; and the tailings storage facility. The site closure cost estimate includes the 38 kt/d expansion facilities, existing facilities and known future work on the site.

A key result of the feasibility study was the shift to preassembly and precast concrete execution strategies. These strategies will facilitate higher productivities and more predictable and controllable costs. In addition, this will maximize opportunities for procurement to implement a global sourcing strategy to improve vendor quality, value pricing and delivery. Additional opportunities to improve schedule certainty and reduce capital costs will continue to be explored in the time leading up to the notice to proceed.

Operating cost estimates are shown in Table 1-4. The operating costs for each area include allocations for power plant operating costs.

Table 1-4: Operating Cost Estimates (Expansion Case)

Operating Cost	Unit	First 5 years 2018-2022	Life-of-Project 2018-2029	Life-of-Mine Average 2014-2029
Mining	US$/t mined[1]	$1.98	$2.11	$2.10
Processing (CIL Plant)	US$/t processed	$14.62	$14.49	$14.74
Processing (Dump Leach)	US$/t processed[2]	$39.21	$39.21	$10.90
Site Admin	million US$/a	$80.9	$65.1	$70.9

Notes:
1. Excludes capitalized maintenance.
2. Limited tonnes are placed over 2018-2019, while dump leach processing continues until 2020.

Economic Analysis

The economics of the Tasiast Expansion Project were evaluated using a real (non-escalated), after-tax discounted cash flow (DCF) model on a 100% project equity (unlevered) basis. Production, revenues, operating costs, capital costs and taxes were considered in the financial model. The main economic assumptions are a US$1,200/oz gold price and a 5% discount rate.

The valuation date for the financial analysis was set for January 1, 2014. All cash flows assumed for the purposes of this study are from this date onward. However, the notice to proceed decision for the expansion option is not expected to be made until 2015 at the earliest.

The cash flow analysis was used to estimate the economics of the 38 kt/d carbon-in-leach (CIL) plant expansion scenario using a heavy fuel oil (HFO) on-site power plant. This scenario assumes a new CIL plant starts up in 2018.

The results of the financial analysis, with sensitivities to gold price and discount rate assumptions, are shown in Table 1-5. The FS is based on $1,200/oz gold and a real discount rate of 5%. The project is economic using these assumptions.

This report does not include an Economic Analysis of the 8 kt/d case because Kinross is not required to do so for cases that do not include a material expansion of current production.

Table 1-5: Financial analysis results and sensitivities

Financial metric	Unit	Gold Price (US$/oz)			
		1,200	1,350	1,400	1,600
NPV at 0% discount rate	US$ billion	1.48	2.50	2.84	4.17
NPV at 5% discount rate	US$ billion	0.50	1.22	1.46	2.37
NPV at 10% discount rate	US$ billion	(0.02)	0.51	0.69	1.35
Internal rate of return	%	10	17	20	31
Payback year (undiscounted)	Year	2024	2020	2020	2019

Conclusions

Tasiast is viewed as a long-term strategic asset for Kinross, located in a district that is believed to have significant future potential. The expansion project is believed to provide an opportunity to capitalize on the full potential of the operation and to cement Tasiast as a cornerstone asset within the company.

At a 5% discount rate, the project is economic at a $1,200/oz gold price.

Recommendations

Leading up to the potential construction decision in 2015, Kinross will continue with a value improvement program focusing on:

- Continuous improvement and refinement of the execution strategy
- Procurement strategies and opportunities

- Continuous improvements to the mine plan
- Evaluation of requirements for the owners team, and engineering, procurement and construction management strategies and requirements

This will allow the project team to further improve and refine areas such as planning, procurement, and project execution.

2. INTRODUCTION

Kinross has prepared a Technical Report for the wholly-owned Tasiast gold deposit (the Project) located in the Islamic Republic of Mauritania (Mauritania), Africa (Figure 2-1). Kinross is using this Technical Report to support disclosure of mineral resources and mineral reserves at the Project. The Technical Report conforms to National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) and has an effective date of March 31, 2014.

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Although a construction decision on the 38 kt/d expansion will not be made until 2015 or later, the FS supports the mineral resource and mineral reserve estimates set forth in Sections 14 and 15, based on the economic viability conclusions of the FS and an assumed decision to proceed with the expansion for purposes of the estimates. The mine plans for 2014 and 2015 for the 8 kt/d and 38 kt/d cases are identical.

Most sections of this report are equally applicable to both the 8 kt/d and 38 kt/d cases. Where it is not appropriate to have both cases covered in the same section, the section applicable to 8 kt/d occurs first (with the exception of Sections 14 and 15). Additional sections on specific aspects of the 38 kt/d case are found in Section 24 - Other Relevant Data and Information, with subsection numbers corresponding to the main section number per NI 43-101. For example, Section 16 has Mining Methods for 8 kt/d, and Section 24.16 has Mining Methods for 38 kt/d. The report sections and applicable cases are shown in Table 1-1.

All measurement units used in this Technical Report are metric, and currency is expressed in US dollars unless stated otherwise. Mauritania uses the Ouguiya (MRO) as its currency.

Information used to support this Technical Report has been derived from the reports and documents listed in the References section of this Technical Report.

The use of the terms “we”, “us”, “our” or “Kinross” in this Technical Report refer to Kinross Gold Corporation.

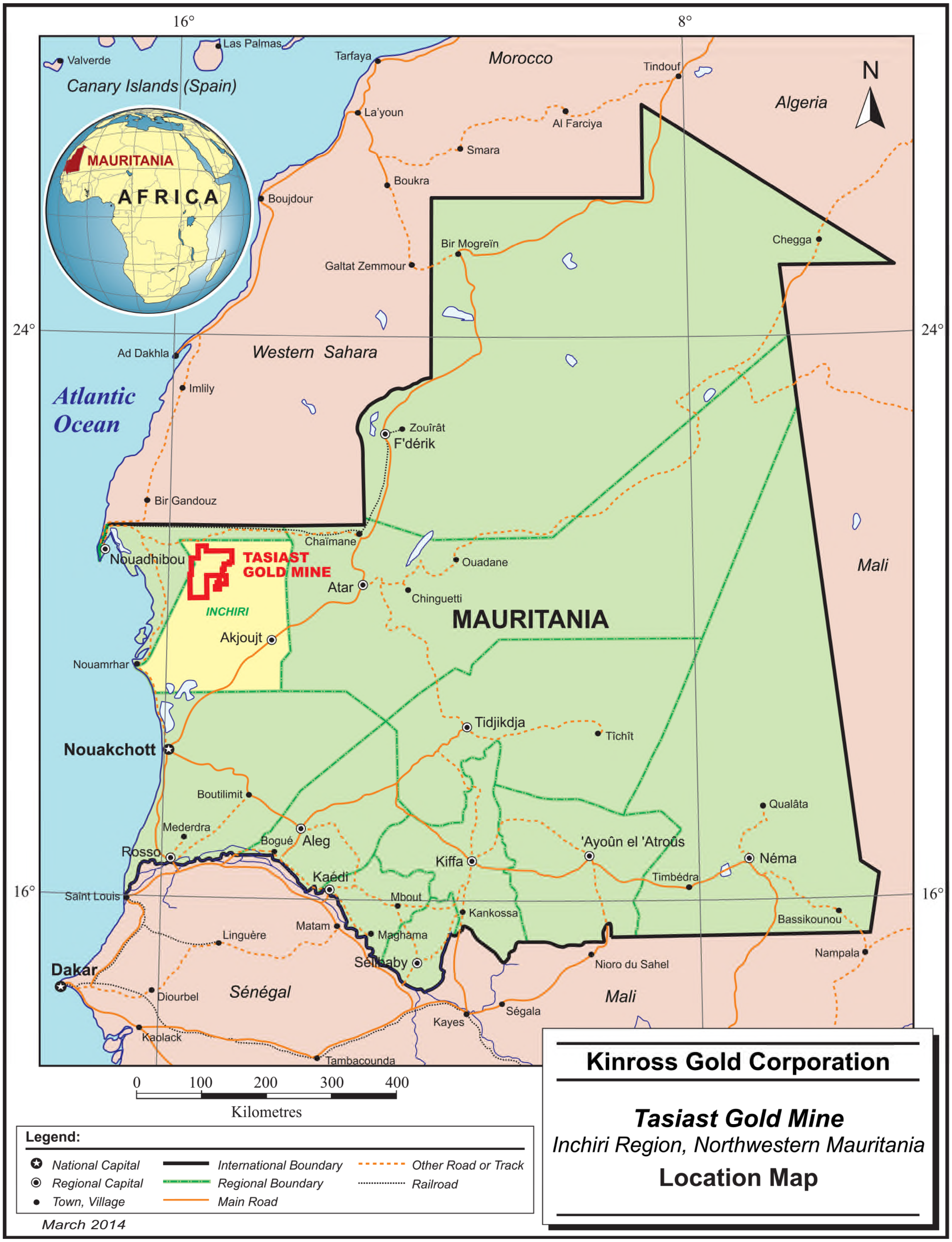

Figure 2-1: Project Location Plan

Table 2-1: Technical Report sections and applicability to the 8 kt/d and 38 kt/d cases

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28 DATE & SIGNATURE PAGE	●	●

2.1 Qualified Persons

The Qualified Person (QP) for this Technical Report is John Sims, AIPG Certified Professional Geologist and Vice President, Technical Services for Kinross.

Mr. Sims visited the site most recently in June 2012. During the site visit, Mr. Sims inspected core and surface outcrops, drill platforms and sample cutting and logging areas; discussed geology and mineralization with Project staff; reviewed geological interpretations with staff; and inspected the major infrastructure and current mining operations. All sections in this Technical Report have been prepared under the supervision of Mr. Sims.

2.2 Information Sources

Information used to support this Technical Report was derived from previous technical reports on the property, and from the reports and documents listed in the References section of this Technical Report.

Mineral Resources: The mineral resource estimates included in this report were prepared by Tim Maunula of T. Maunula & Associates Consulting Inc., an independent consulting firm. Mr. Maunula is a registered Professional Geoscientist in Ontario. Mr. Maunula visited the site in November and December of 2013.

Mineral Reserves / Mining: The mineral reserve estimate included in this report was prepared under the supervision of Todd Carstensen, Director, Mine Planning, Kinross Technical Services. Mr. Carstensen is a Society for Mining, Metallurgy & Exploration (SME) Registered Member. Mr. Carstensen visited the site in January 2013.

2.3 Effective Dates

The effective date of this Technical Report is March 31, 2014, and for the Tasiast expansion mineral resources and mineral reserves the effective date is December 31, 2013.

There were no material changes to the information on the Project between the effective date and the signature date of the Technical Report.

2.4 Previous Technical Report

Sedore, M. and Masterman, G., 2012: Tasiast Mine, Mauritania 43-101F1 Technical Report, effective date March 30, 2012.

2.5 List of Abbreviations

μ	micron	kPa	kilopascal
°C	degree Celsius	kWh/t	kilowatt-hour per tonne
°F	degree Fahrenheit	kW	kilowatt
μg	microgram	kWh	kilowatt-hour
A	annum	L	liter
Au	gold	LFO	light fuel oil
Bbl	barrels	L/s	liters per second
Btu	British thermal units	m	metre
C$	Canadian dollars	M	mega (million)
Cfm	cubic feet per minute	m^2	square meter
CIL	carbon-in-leach	m^3	cubic meter
Cm	centimeter	mbgl	metres below ground level
cm^2	square centimeter	min	minute
D	cay	masl	meters above sea level
dia.	diameter	mm	millimeter
Dmt	dry metric tonne	Mt/a	million tonne per year
Dwt	dead-weight ton	MTO	material take-off
Ft	foot	MW	megawatt
ft/s	foot per second	MWe	megawatt-electrical
ft^2	square foot	m^3/h	cubic meters per hour
ft^3	cubic foot	opt	ounce per short ton
G	gram	oz	Troy ounce (31.1035g)
G	giga (billion)	PAU	preassembly unit
Gal	Imperial gallon	ppm	part per million
g/L	gram per liter	psig	pound per square inch gauge
g/t	gram per tonne	RL	relative elevation
Gpm	Imperial gallons per minute	s	second
gr/ft^3	grain per cubic foot	st	short ton
gr/m^3	grain per cubic meter	stpa	short ton per year
Ha	hectare	stpd	short ton per day
HFO	heavy fuel oil	t	metric tonne
Hp	horsepower	t/a	metric tonne per year
In	Inch	t/d	metric tonne per day
in^2	square inch	US$	United States dollar
J	joule	USg	United States gallon
k	thousand (kilo)	USgpm	US gallon per minute
kg	kilogram	V	volt
km	kilometer	WBS	work breakdown structure
km/h	kilometer per hour	wmt	wet metric tonne
km^2	square kilometer	yd^3	cubic yard
kt/d	thousand tonnes per day	yr	year

3. RELIANCE ON OTHER EXPERTS

In the preparation of the Technical Report, the Qualified Person relied on information provided by internal Kinross legal counsel for the discussion of legal matters in Sections 4, 19, and 20.

Except for the purposes legislated under provincial securities law, any other use of this report by any third parties is at this party's sole risk.

4. PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Tasiast Lands are located in northwestern Mauritania, approximately 300 km north of the capital Nouakchott and 250 km southeast of the major city of Nouâdhibou. The Tasiast Lands fall within the Inchiri and Dakhlet Nouâdhibou Districts. The Tasiast mine is located at 446600E, 2275600N (UTM, WGS84, Zone 28N).

4.2 Mineral Tenure

The Tasiast mine is owned and operated by Tasiast Mauritanie Ltd. S.A. (TMLSA), a wholly owned subsidiary of Kinross, under exploitation Permit No. 229.

The Tasiast mine is located within the 312 km^2 Tasiast exploitation permit of El Gaîcha. The mining operations and infrastructure are located entirely within the lands subject to the mining/exploitation permit (*permis d'exploitation* or PE). Exploitation permit No. 229 is located centrally within a surrounding permit block of four contiguous exploration permits, totalling 3,118 km^2, as listed in Table 4-1 and shown in Figure 4-1. All these permits are in good standing. The table also indicates tenure expiry dates. The Tasiast mine and the exploitation permit are owned by TMLSA.

The adjacent four permits, the underlying lands of which are contiguous to the Tasiast exploitation permit lands, are held by two sister companies of TMLSA. Société d'Extraction du Nord de l'Inchiri S.A. (SENISA) holds two recently converted mining permits (for the Tmeimichat and Imkebdene areas) and Tasiast Mauritanie Limited (TML) holds two exploration permits (Tasiast Sud and N'Daouas-Est areas). As part of the recently completed conversion process of two exploration permits, Kinross has undertaken to transfer to the Government of Mauritania a 10% carried interest in SENISA. Other than SENISA, all permit-holding affiliates of Kinross, including TMLSA, are wholly-owned indirect subsidiaries of Kinross. Kinross acquired TMLSA, including the Tasiast operation and exploration permits and lands, through its acquisition of Red Back in September 2010.

Tenure coordinates are shown in Table 4-2. A permit boundary is defined by a list of the coordinates of its corners or pillar points. The boundaries are not physically marked on the ground, and have not been surveyed. However, extensive surveying has been conducted within both the exploitation permit No. 229 and adjoining exploration permits. To date, approximately 30,000 points have been located via formal surveying by qualified surveyors using Electronic Distance Meter total station instruments, and many additional points have been picked up by differential global positioning system (GPS) and GPS methods. All the known gold deposits are well inside the boundaries, and the size and shape of the exploitation permit are adequate for the intended exploration, mining and processing activities.

Table 4-1: Mineral Tenure Summary – Tasiast Property

Name	District	Type	No.	km^2	Granted	Expiry
Tasiast (El Gaîcha)	Wilaya de l'Inchiri	Mining Permit	PE 229	312	January 19, 2004	January 18, 2034
Imkebdene	Wilayas Dakhlet Noudhibou et Inchiri	Mining Permit	PE 2018C2	539	November 12, 2013	November 11, 2043
Tmeimichat	Wilaya de l'Inchiri	Mining Permit	PE 2019C2	746	November 12, 2013	November 11, 2043
Tasiast South	Wilayas Dakhlet Noudhibou et Inchiri	Exploration Permit	PRM 428	355	April 2, 2008	May 11, 2017
N'Daouas	Wilaya de l'Inchiri	Exploration Permit	PRM 437	1,478	April 2, 2008	May 11, 2017

Figure 4-1: Tenure Location Plan

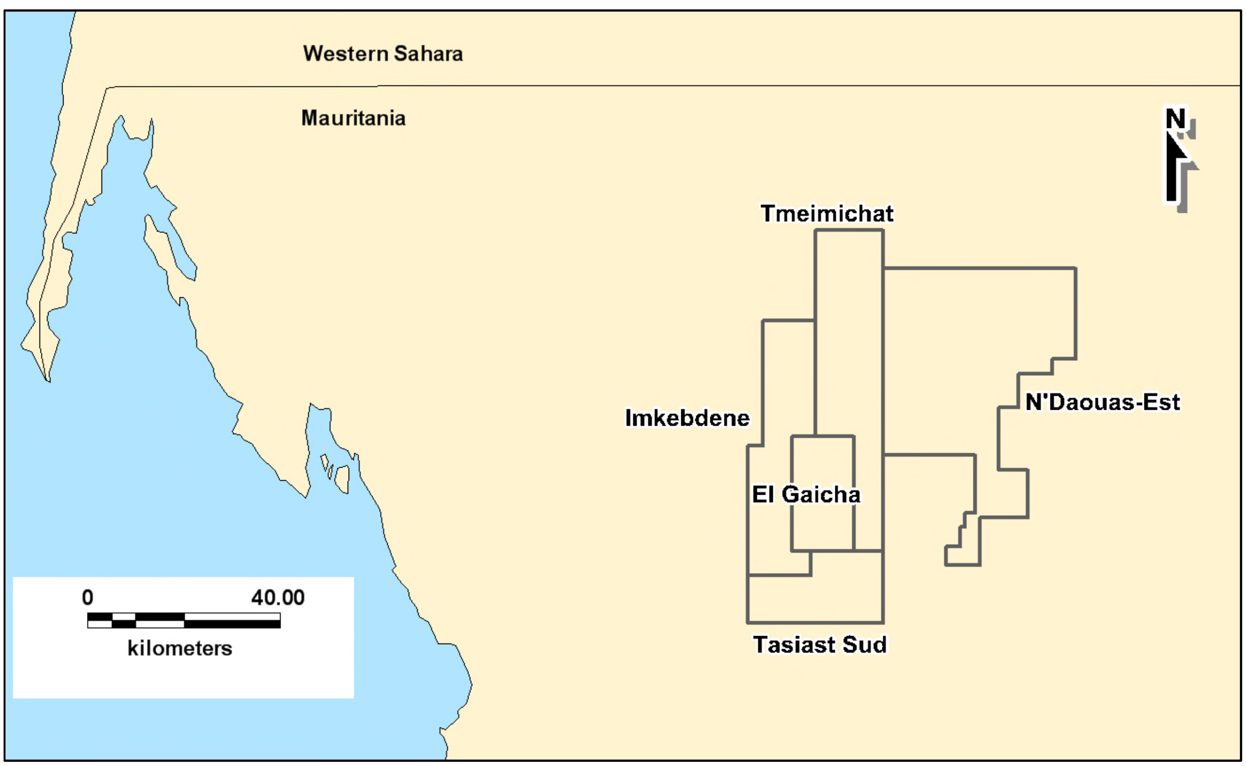


Table 4-2: Permit Boundary Coordinates

Name	Permit Number for Group 2 Minerals	Point	Coordinates UTM (E)	Coordinates UTM (N)
El Gaîcha	PE 229	A	441000	2287000
		B	454000	2287000
		C	454000	2263000
		D	441000	2263000
Imkebdene	PRM 237	A	435000	2311000
		B	446000	2311000
		C	446000	2287000
		D	441000	2287000
		E	441000	2263000
		F	445000	2263000
		G	445000	2258000
		H	432000	2258000
		I	432000	2285000
		J	435000	2285000
Tmeimichat	PRM 238	A	446000	2330000
		B	460000	2330000
		C	460000	2263000
		D	454000	2263000
		E	454000	2287000
		F	446000	2287000
Tasiast South	PRM 428	A	460000	2263000
		B	460000	2248000
		C	432000	2248000
		D	432000	2258000
		E	445000	2258000
		F	445000	2263000
N'Daouas East	PRM 437	A	460000	2322000
		B	500000	2322000
		C	500000	2303000
		D	495000	2303000
		E	495000	2300000
		F	488000	2300000
		G	488000	2293000
		H	484000	2293000
		I	484000	2280000
		J	490000	2280000
		K	490000	2270000
		L	480000	2270000
		M	480000	2260000
		N	473000	2260000
		O	473000	2264000
		P	476000	2264000
		Q	476000	2268000
		R	477000	2268000
		S	477000	2271000
		T	479000	2271000
		U	479000	2283000
		V	460000	2283000

Surface rights are granted along with Permit No. 229, and applicable fees are paid annually, as determined by decree under Section 107 of the Mining Code. Surface rights for the permit are in good standing, and there are no competing mining rights in the area, except for three iron-ore explorations permits that overlap mining permit PE 229. These permits entitle their holders to do exploration work, as long they do not interfere with TMLSA's operations. TMLSA does not have any obligation to accommodate the holders of these permits.

The iron-ore exploration permit holders are not entitled to transform their overlapping exploration permits into exploitation permits on the overlapping area without TMLSA's prior written approval, and they are not entitled to any compensation from TMLSA.

Exploration permits (*Permis de Recherche Minière* or PRM) grant exclusive exploration rights over a specific block (maximum 1,000 km^2) and are granted for a three-year period, renewable twice for up to three years at each renewal. Exploitation permits are granted for 30 years, and are renewable for periods of 10 years each. A condition of each permit is that the holder is required to hire Mauritanian tradespersons to provide services, and to contract with national suppliers and businesses in preference to foreign service providers, where the national suppliers and businesses can offer at least the same terms, quality and pricing. Table 4-3 summarizes the durations of exploration and mining permits in Mauritania. Operating permits are discussed in Section 20.2.

Current environmental liabilities are those that would be expected from a mining operation, and include the mine, crushing and CIL processing plant, dump leach facilities, power plant, tailings and waste rock facilities, power grids, roads, accommodation camp, ancillary facilities and drill pads established to support mining and exploration activities.

Table 4-3: Permit Durations in Mauritania

Licence Type	Duration	Renewal Period	Number of Permissible Renewals	Rules and mechanisms
Exploration Permit	3 years	3 years	Two After the two renewal periods lapse, the permit expires unless it is converted (in whole or in part) into an exploitation permit.	• Surface Area:1,000 km^2 blocks • Confers right to explore for resources to any depth within permit area • Number is limited to 20 exploration permits per holder; a holder must have the technical and financial capability to conduct the work • Permits taken under a joint venture are not taken into consideration for the calculation of the above limit, if the holder is not the controlling partner or the operator • Transferable under conditions established by the Decree on Mining Titles
Mining Permit	30 years	10 years	Several	• Necessary for operating a mine • Within an area initially covered by an exploration permit, for the same commodities, and on the basis of a feasibility study • Granted only to a legal entity incorporated under Mauritanian law and created by the holder of the exploration permit • Transferable under conditions established by the Decree on Mining Titles • Personnel health and safety reports to be lodged with the Ministry every six months, and environmental and activity reports every year • Land needs to be rehabilitated after mining

4.3 Fees, Royalties, Duties and Taxes

Mining activities in Mauritania are mainly governed by the Mining Code and its regulations, and by the Model Mining Convention Law, which provides the legal and tax framework for all mineral exploration and extraction activities.

The Mining Code establishes conditions and rules governing all phases of mining activity. The Model Mining Convention Law provides that each exploration permit (*Permis de Recherche Minière* or PRM) is subject to a mining convention with the State of Mauritania, which outlines the framework of customs, economic, financial, legal and tax terms and conditions under which the permit holder proceeds with its exploration or mining activities inside the perimeter of its permit. A Mining Convention is attached to a given permit. Table 4-4 summarizes provisions of TMLSA's Mining Convention relating to fees, royalties, duties and taxes.

The Mining Code is also complemented by the Decree on Mining Titles, which provides more details on the process governing the grant, renewal, expansion or reduction, division or merger, transfer, termination, suspension and cancellation of a permit for exploration or exploitation. It also governs the conversion of an exploration permit into an exploitation permit.

The conditions embodied in the Model Mining Convention (Law No. 2002/02) are designed to stimulate and encourage investment in both exploration and mining. Obtaining exploration permits is not difficult and the granting of mining permits is not expected to present difficulties in view of Mauritania's stated aim to promote the mining industry. The mining industry is seen as one of the main growth industries for the improvement of the country's economy.

Table 4-4: Fees, Royalties, Duties and Taxes

Applicable Obligation	Exploration Permit	Mining Permit
Compensatory fees (for the issuance, extension or reduction, renewal, early termination and transfer of a permit)	MRO 2,000,000	MRO 2,500,000
Annual surface fee	Initial period: MRO 2,000-6,000/km^2 First renewal period: MRO 10,000-14,000/km^2 Second renewal period: MRO 20,000-24,000/km^2	MRO 25,000/km^2
Royalty		For gold, 3% of the sales value of the metal at the final stage of processing within Mauritania is deductible from taxable income.
Customs duties and other taxes	Complete exemption on all equipment and supplies, including fuel	Complete exemption on all imported equipment and supplies, including fuel, for five years after the start of production (ended August 2012). Customs duties of 5% thereafter on equipment and supplies imported, except fuel, lubricants, mine supplies and spares that will continue to be exempt from duty.
Tax on business profits		TMLSA was exempt from this tax for the first three years after its first production. After three years, the rate was fixed and stabilized at 25%. Articles 8 to 17 inclusively of the Mining Convention establish the relevant deductions to determine taxable profits.
Fixed minimum tax		Exemption until the end of the third year following the year the exploitation permit was granted. Since the end of this period, the standard rate of 2.5% applies at a reduced rate (1.25%) for TMLSA.
Tax on salaries, wages and annuities of expatriates employed by TMLSA		The standard rate of 40% reduced to half (20%) for TMLSA.
Income tax from capital		Customary applicable rate is 10%. Dividends reinvested on Mauritanian territory are exempt from this tax.
General income tax		Exemption for the duration of the Mining Convention
Value-added tax (VAT)		Customary applicable rate is 14%, except for exports by TMLSA, which is 0% (contingent on export of at least 80% of production).
Tax on sales - consumption tax		Exemption for the duration of the Mining Convention
Housing tax		Applies according to the CGI (General Tax Code), as from the first exploitation permit
Land income tax (on properties built)		Rules of application are those of the CGI. The rate, as voted by the town council, is 3% to 10% of the taxable value, which is the rental value of the property minus 20%.
Trading tax		Applicable from the date the first exploitation permit was granted according to the CGI. This tax is a fixed duty based on turnover

Applicable Obligation	Exploration Permit	Mining Permit
Registration and stamp duties		Exemption for the duration of the Mining Convention
Tax on motor vehicles		Applies according to the CGI as from the first exploitation permit. The rate of the tax is based on the use of the vehicle and its tax power
Apprenticeship tax		Exemption for the duration of the Mining Convention

In addition to the 3% royalty payable to the government, Franco-Nevada Corporation holds a 2% net smelter return royalty on gold production at the Tasiast mine in excess of 600,000 ounces. Production at the Tasiast mine reached 600,000 ounces in July 2011, and the first royalty payment to Franco-Nevada was made in October 2011.

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

5.1 Accessibility

The Tasiast Lands are accessed from Nouakchott by using the paved Nouakchott to Nouâdhibou highway for 370 km and then via 66 km of graded mine access road which is maintained by TMLSA. An airstrip at the Mine Site is used for light aircraft primarily travelling to and from Nouakchott.

The principal ports of entry for goods and consumables are either Nouakchott or Nouâdhibou. Materials are transported by road to the mine site.

Routine access within the country is provided by an 11,000 km long road network, comprising approximately 3,000 km of paved highways and approximately 8,000 km of unpaved highways as well as numerous desert tracks. A paved 470 km long, two-lane highway runs between the cities of Nouakchott and Nouâdhibou.

A 717 km long rail line located along the border between Mauritania and Western Sahara is owned and operated by *Société Nationale Industrielle et Minière de Mauritanie* (SNIM). This rail line is primarily used to haul iron ore from SNIM's iron ore mines in Zouérate to the port of Nouâdhibou.

Access to the major urban centres of Mauritania is also possible via air. Nouakchott is accessible via international flights operated by numerous West and North African carriers; Air France also provides a direct connection to Paris.

5.2 Climate

Mauritania has an arid desert climate, with an average annual high temperature of above 45°C between May and August. Minimum temperatures may go below 10°C in December and January. From January to March, sandstorms frequently occur in the country; this causes sand build up and dune formation. Sandstorms vary in intensity, and visibility can be reduced to several metres. There is a rainy season, usually between July and September; however, the amount of rainfall and length of season varies spatially and temporally in the various regions of the country. Annual rainfall varies from a few millimetres in the desert regions to as high as 450 mm in the south along the Senegal River. During the last 20 years, the country has recorded two periods of drought, namely 1984-85 and 1991-92.

Average annual precipitation at the mine site is approximately 90 mm, and usually occurs during July to September. The average recorded monthly evaporation is approximately 320 mm/month (3840 mm/a).

Mauritania is located along the northwestern coast of Africa and is bordered by the Atlantic Ocean to the west. The country's land mass covers the western portion of the Sahara Desert. Mauritania's land mass consists mainly of flat and barren desert landscape surfaces that are cross cut by three large NE-SW trending longitudinal dune fields. In the central part of the country, near Adrar and Tagant, several hills and mountains rise up to 915 masl. In the desert regions, vegetation is sparse, consisting of various species of trees (e.g., acacia) and grasses.

5.3 Local Resources and Infrastructure

The mine is located in a remote area where there is no electric power grid. On-site power generation is discussed in Section 18.

The source of mine water supply is located 64 km west of the mine and consists of a semi-saline underground aquifer, which is exploited by 47 wells. Water is pumped from the well field to the mine (see Section 18).

In 2014, the Tasiast mine employs approximately 1,350 employees, of whom approximately 1,175 are Mauritanian nationals. Staff accommodation is provided at the mine site (see Section 18).

The terrain surrounding the Tasiast deposit is flat, and is adequate for construction and operation of the camp, mine, plant, tailings, and waste rock disposal facilities. Figure 5-1 shows the mine site layout.

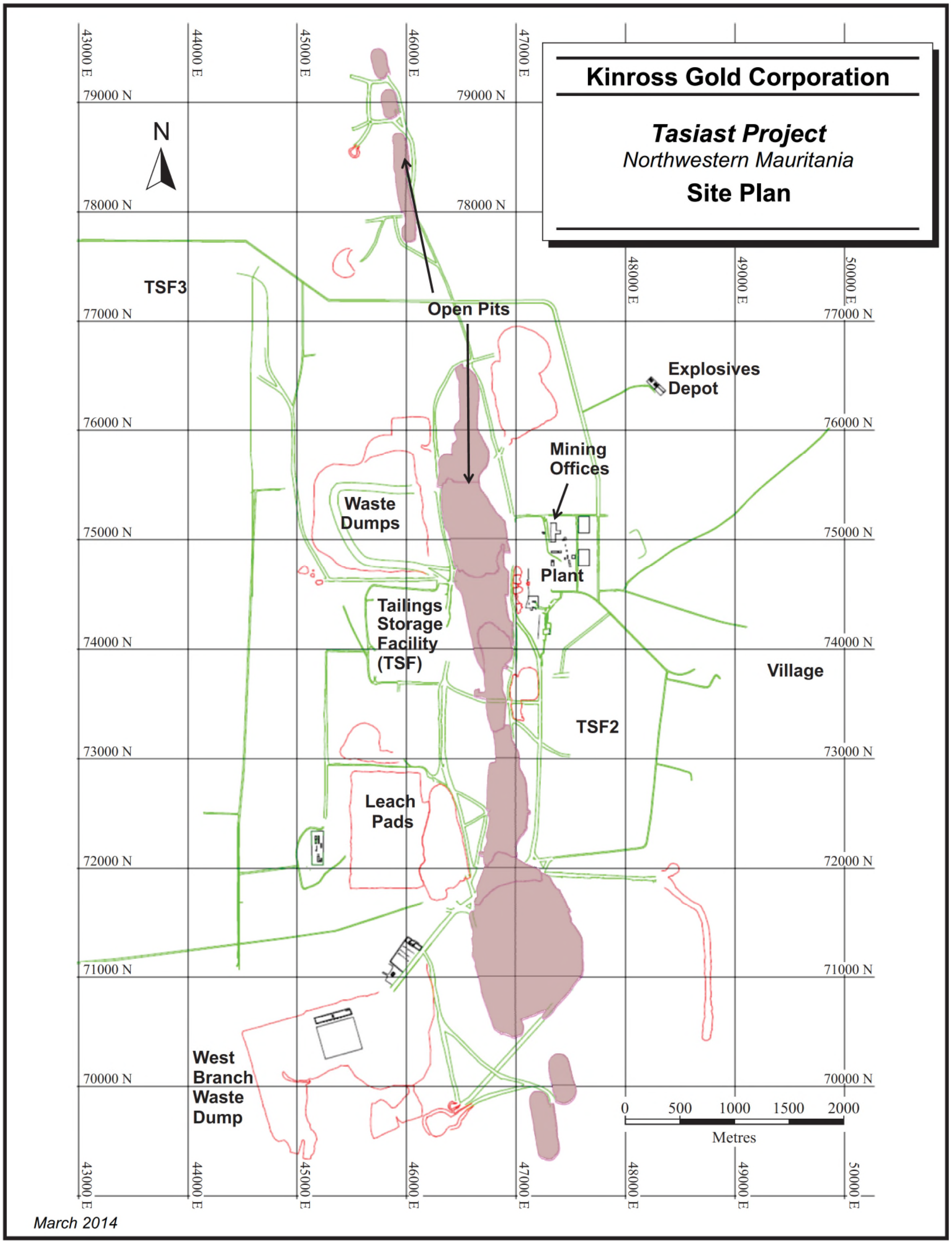


Figure 5-1: Mine Site Infrastructure Layout Plan

5.4 Physiography and Environment

The topography of the Tasiast Lands consists mainly of flat, barren plains which are primarily covered by regolith and locally by sand dunes, or eroded paleo-lateritic profiles. Elevation ranges from approximately 130 masl to 150 masl.

The drainage pattern around the Tasiast Lands consists of several intermittent dendritic first- and second-order streams that generally flow in a southwesterly direction. There are no permanent watercourses in the area. However, there are numerous, intermittent watercourses, known as “wadis”, which flow for only a few days per year. The largest wadi is the Khatt Ataoui wadi, which is located approximately 6 km from the mine site.

The Tasiast mine is located in the arid Saharan zone, where plant life is very scarce, consisting mainly of the low shrubs *Zygophyllum album*, the small tree *Maerua crassifolia* (atil) and the grass *Aristida pungens* (sbot). Acacias are also present along many of the wadis. The well field, water pipeline and road are almost exclusively colonized by *Zygophyllum album*.

Hares, hamsters and gerbils are the most common mammals at the mine site, and jackals, fennec fox and polecat can also be found in the well area. There are no protected species in the Project area. The eastern boundary of the Banc D’Arguin National Park is located about 2 km west of the well field area and 60 km from the mine site.

6. HISTORY

6.1 Tenure History

In 1996, the *Office Mauritanien de Recherches Géologiques* (OMRG) completed a regional reconnaissance exploration program within and around the lands hosting the Tasiast deposit and made this information available to third parties. As a result, NLSD (a subsidiary of Normandy Mining Ltd. of Australia) acquired the exploration rights to the Tasiast deposit.

In 2001, NLSD was acquired by Newmont Mining Corporation creating Newmont LaSource. Midas Gold PLC (Midas) was incorporated in England and Wales in 2002 for the purpose of acquiring Newmont LaSource's assets in Mauritania including exploration permits over lands hosting the Tasiast deposit, as well as various other permit areas. Midas completed its acquisition of the Tasiast deposit from Newmont LaSource on April 1, 2003 and, in April 2003, Geomaque Explorations Inc. (Geomaque) announced the acquisition of Midas. The merger of Geomaque and Midas ultimately created a new entity - Defiance Mining Corporation (Defiance). In June 2004, Rio Narcea Gold Mines, Ltd. (Rio Narcea) acquired Defiance and took ownership of the Tasiast deposit.

Red Back acquired the Tasiast deposit from Lundin Mining Corporation (Lundin) in August, 2007 following Lundin's acquisition of Rio Narcea. In September 2010, Kinross completed the acquisition of Red Back. Kinross, through TMLSA, holds 100% of the Project.

6.2 Project History

From 1962 to 1993, the Tasiast region was the subject of three regional exploration programs for pegmatites, iron ore, and nickel sulphides which were carried out by the BRGM and SNIM.

Three exploration programs were carried out in the Tasiast region between 1993 and 1996 as a European Development Fund project. Work completed included regional-scale reconnaissance geological mapping and geochemical sampling. Traverse lines for the mapping and geochemical sampling programs were oriented east–west with samples collected at 500 m centres; this work identified the Tasiast area as being anomalous in gold. More detailed soil sampling of the Tasiast area on 250 m spaced centres, and trenching was conducted.

NLSD, in the period 1996–2001 completed geological and regolith mapping, interpretation of satellite imagery, airborne and ground magnetic geophysical surveys, specialist petrographical, mineralogical, and geological studies, metallurgical test work, and auger, reverse circulation (RC) and core drilling.

Midas undertook a full review of all existing information in 2003, and prepared mineral resource estimates for the West Branch and Piment areas. From 2003 to 2004, Defiance completed mineralogical and metallurgical test work, hydrogeological studies, a preliminary pit slope design study, RC and core drilling, a mineral resource estimate, and a feasibility study.

Rio Narcea completed additional RC and core drilling during 2005–2006. Red Back also undertook RC and core drilling, re-estimated mineral resources, and updated engineering studies. Mine construction was completed in 2007, with the mine formally opened in early 2008.

Between August 2007 and September 2010 Red Back completed several large exploration campaigns in the Piment and West Branch areas, as well as at several district targets. Early drilling campaigns were directed at testing the lateral and vertical extents of the mineralization at Piment and drilling oxide resources at West Branch. In October 2009, Red Back discovered the Greenschist Zone at West Branch and commenced drilling the deposit.

From September 2010 to date, TMLSA has aggressively ramped up exploration with the majority of activities directed towards delineating the extents of the Greenschist Zone.

Mining at Tasiast commenced in April 2007 and the mine was officially opened by the President of Mauritania, His Excellency Sidi Mohamed Ould Cheikh Abdallahi, on July 18, 2007. A total of 1,139,275 oz has been produced by year-end 2013. A summary of gold production at Tasiast is included in Table 6-1. There has been no historical gold production from other deposits in the Tasiast area.

Table 6-1: Production Summary

Year	Carbon In Leach Tonnes Milled (Mt)	Grade (g/t)	Gold Produced ('000 oz.)	Recovery (%)	Dump Leach Gold Produced ('000 oz.)	Total Gold Produced ('000 oz.)
2007	0.22	4.36	21	68.6%	0	20,812
2008	1.49	3.07	140	95.4%	0	140,052
2009	1.68	2.88	142	91.4%	16,400	158,660
2010	2.14	2.52	150	86.8%	35,617	185,982
2011	2.60	2.04	153	89.4%	47,910	200,619
2012	2.55	1.54	114	90.2%	71,356	185,334
2013	2.50	1.96	144	91.3%	103,883	247,816
Total/Average	**13.18**	**2.27**	**864**	**89.9%**	**275,165**	**1,139,275**

7. GEOLOGICAL SETTING

7.1 Regional Geology

The Tasiast Lands lie in the southwestern corner of the Réguibat Shield (Figure 7-1), which is a large area underlain by Precambrian crystalline rocks that comprises the northern part of the West African Craton. In general, the western half of the Reguibat Shield contains rocks of Mesoarchaean age (~3.1 to ~2.9 Ga), whereas those in the eastern half are Palaeoproterozoic (~2.6 Ga). Large granitic intrusions and moderate- to high-grade metamorphic rocks including gneisses, amphibolites, and schists typify the basement rocks. Metamorphic grade in the Reguibat Shield generally increases from mid-greenschist facies in the southwest through to granulite facies in the central parts. Several north-south elongate greenstone belts occur over a broad area of the western Reguibat Shield. The greenstone belts consist of metavolcanic, meta-plutonic and metasedimentary rocks ranging from ultramafic to felsic compositions. The metasedimentary rocks were deposited within rift basins that formed on top of the metavolcanic rocks that rest upon the crystalline basement. Gold was deposited during deformation that was associated with inversion of the rift basins. The Tasiast lands fall within a cluster of several greenstone belts. The Réguibat Shield was cratonized by the end of the 2,100 to 2,000 Ma Eburnean orogeny and has been stable since at least 1,700 Ma. The Réguibat Shield is bounded on all sides by Pan African orogenic belts and covered in the south by the extensive intra-cratonic sediments of the Taoudeni Basin.

Three significant Archaean greenstone belts are exposed within the Tasiast Lands (LaSource-BRGM, 1997; Figure 7-2) and are termed, from east to west:

- N’Daouas (20 km long x 6 km wide);
- Aouéouat (+75 km long x 8 km wide); and
- Imkebdene-Kneffissat (+ 60 km long x 9 km wide).

The Aouéouat and Imkebdene-Kneiffissat greenstone belts are north–south trending linear belts, whereas the N'Daouas greenstone belt is arcuate in plan shape. The greenstone belts are surrounded by granitic intrusive rocks and gneissic domes of the types that comprise the bulk of the rocks within the southwest Réguibat Shield. The greenstone belts consist of ultramafic to felsic volcanic rocks and sedimentary packages that include ferruginous quartzite, greywacke, arkose, siltstone and turbidite sequences. Rock units within the belts have undergone mid-greenschist to lower amphibolite grade metamorphism and multiple deformation events. Two families of non-foliated mafic (gabbroic) dikes that strike predominantly north–northeast and east–

west crosscut all other rocks in the district, including undeformed pegmatite units that might be Proterozoic in age. The two main thrust systems that control gold mineralization are labelled in Figure 7-2, and the names of the mining and exploration permits are also indicated.

The Aouéouat greenstone belt hosts all of the known Tasiast gold deposits and most of the currently explored prospects. Other belts in the district contain gold occurrences, but remain under-explored. Uranium-lead (U-Pb) dates obtained from gneiss, granodiorite and pegmatite intrusive rocks sampled in the southwest Réguibat Shield indicate a range of ages from 3,070 Ma to 2,600 Ma (LaSource-BRGM 1997, Maurin et al, 1996).

The Precambrian lithostratigraphy established by Kinross for the Aouéouat belt is shown in Figure 7-3 along with several proprietary U-Pb dates for rocks of the Aouéouat and Tasiast assemblages and for granodiorite intrusions. The mafic to felsic volcanic and intrusive units that host the West Branch deposit belong to the Aouéouat assemblage that crystallized between 2,990 Ma and 3,000 Ma. Metasedimentary rocks of the Tasiast assemblage that overlay the mineralized West Branch units contain detrital zircons of similar ages and older populations derived from approximate 3,200 Ma orthogneiss basement. Granodiorites that crosscut the metavolcanic rocks are dated 2,960 Ma to 2,970 Ma. Although the gold deposition event has yet to be directly dated, it must have occurred following the intrusion and exhumation of some granodiorite plutons, as clasts of the granodiorites are identified within the metasedimentary rocks that are deformed and locally mineralized.

The principal north–south structural fabric in the greenstone belts is evident in satellite images (Worldview-2) and in geological maps over 70 km of the strike length. Steeply dipping foliations and isoclinals folds with north–south axial surface traces are common across the Aouéouat belt. Those structures formed through east–west transpressive shortening that occurred as a result of basin inversion. Strain was partitioned between tightly folded domains and north–south striking shear zones. Several families of cross-cutting faults with northeast and southwest strikes transect the folds and thrust structures.

Subtropical conditions in the tertiary formed a rather continuous laterite layer over the eroded Precambrian geology of the Aouéouat belt and other areas on the Tasiast lands. Locally, the duricrust has been stripped and re-deposited as a gravel lag or pavement that is locally referred to as reg. Depth of oxidation ranges from 10 m to 60 m, with an average of 40 m.

Figure 7-1: Geology of the West African Craton. Modified from Fabre (2005).

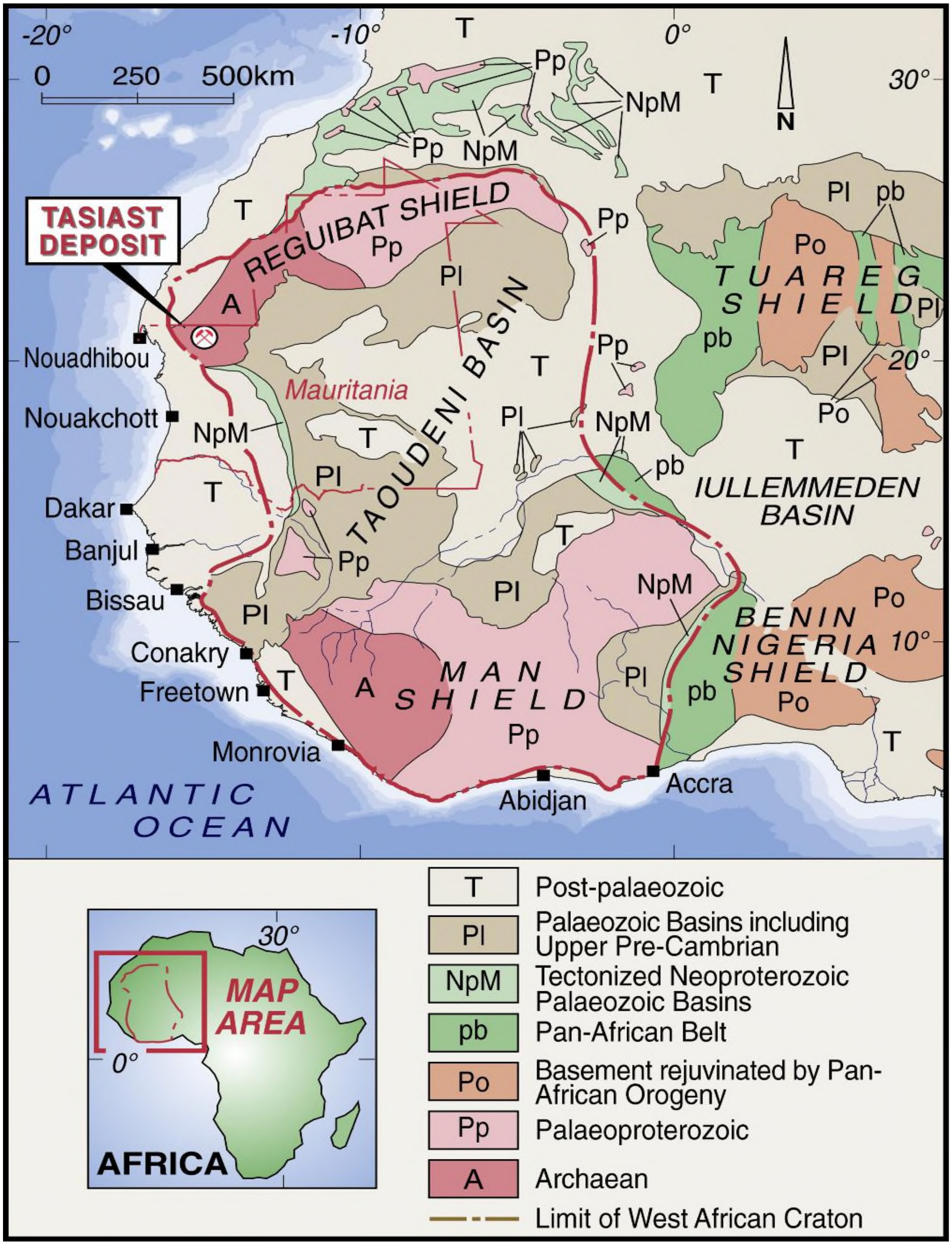


Figure 7-2: Tasiast Geology Overlain on Worldview-2 Satellite Imagery

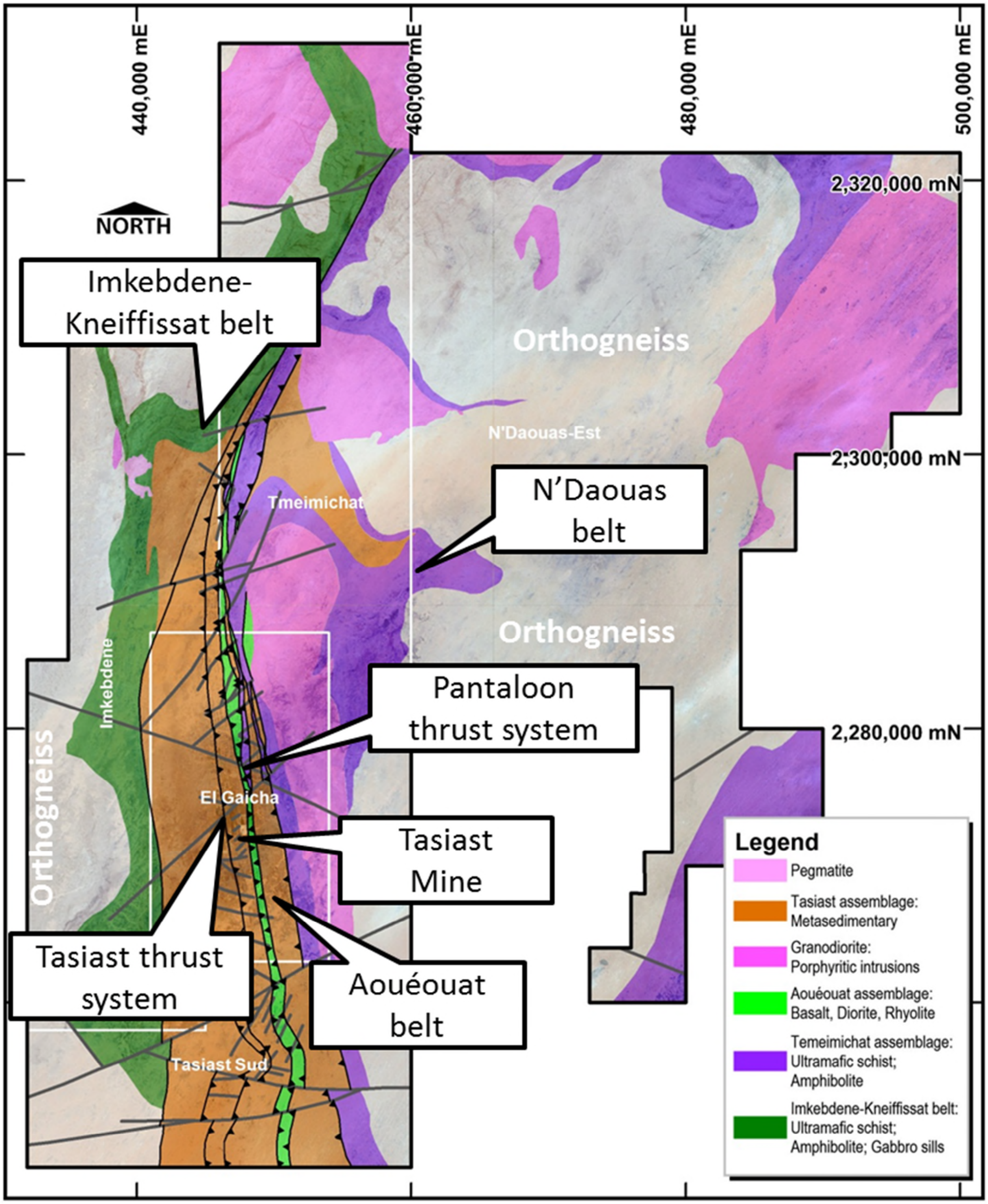
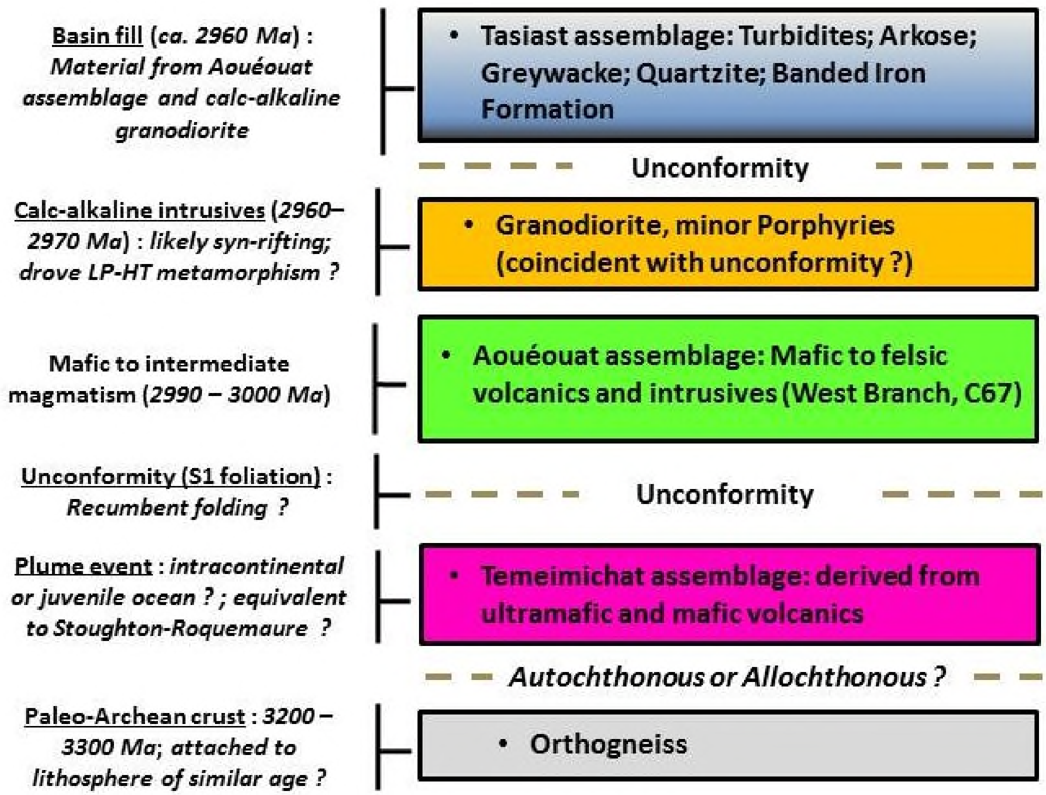


Figure 7-3 Precambrian Stratigraphy of the Aouéouat Greenstone Belt



7.2 Property Geology

The property geology (Figure 7-4 and Figure 7-5) is characterized by orthogneiss basement overlain by deformed north-striking metavolcanic and metasedimentary successions intruded by stocks and plutons of mafic to intermediate composition. All units are cut by unfoliated and post-mineral mafic (gabbroic) dikes. The orthogneiss occurs within the Tasiast lands, but is not present in the mine geology sequence.

The volcano-sedimentary stratigraphy has been tightly to isoclinally folded, and is cut longitudinally by sub-parallel shear systems that are also sub-parallel to the folds. The shear systems are oblique thrusts that are thought to be linked to a deeply rooted system of ductile faults, as illustrated by the structural profile in Figure 7-6. The deeply rooted faults are likely reactivated normal faults that were part of the architecture of the Tasiast rift basin. The most highly prospective gold trends are bounded by the Tasiast and Pantaloon zones (Figure 7-6).

The Tasiast gold trend includes the active Piment and West Branch pits. It is spatially associated with the west vergent Tasiast shear system that places mafic to felsic volcanic and intrusive rocks of the Aouéouat assemblage, including the host rocks of the West Branch deposit, on top of the younger metasedimentary rocks of the Tasiast assemblage. The Tasiast trend passes north–south through the El Gaîcha mining permit and extends over tens of kilometres to the north and south.

The Pantaloon gold trend is located about 3 km to the east of the Tasiast trend. It is spatially associated with the north to north-northwest striking Pantaloon shear system. Mineralization is hosted by quartz-carbonate veins within sheared metasedimentary rocks of the Tasiast assemblage in the hanging wall of the thrust, and sheared ultramafic schist in the footwall of the thrust. The Pantaloon trend is the locus of several exploration prospects on the El Gaîcha mining permit and also on the Tmeimichat exploration permit to the north.

Figure 7-6 shows an interpreted structural profile at 72,000N, based on mapped geology and on modelled gravity data.

The profiles display the geology (top panel) and main alteration assemblages (bottom panel). The principal shear zones are indicated by dashed lines. The view is looking to the north.

Some gold prospects lie between the Tasiast and Pantaloon shear systems and also to the east of the Pantaloon system where they might be aligned parallel to rift-related structures preserved in the underlying basement rocks.

Mineralization at the Piment deposits (South-South, South-North, Central, North, Mars, Prolongation (Figure 7-4) is hosted by iron-rich metasedimentary formations that belong to the Tasiast assemblage and that have been metamorphosed to upper Greenschist and lower amphibolite peak grades. Gold is mainly associated with the sulphide replacement of Fe-oxide minerals, such as magnetite. At the West Branch deposit, the mineralization is hosted by quartz-carbonate veins within strongly sheared, fine- to medium-grained intrusive rocks of diorite and quartz diorite compositions that belong to the Aouéouat assemblage.

Figure 7-4 Plan Map of the Main Structural Features

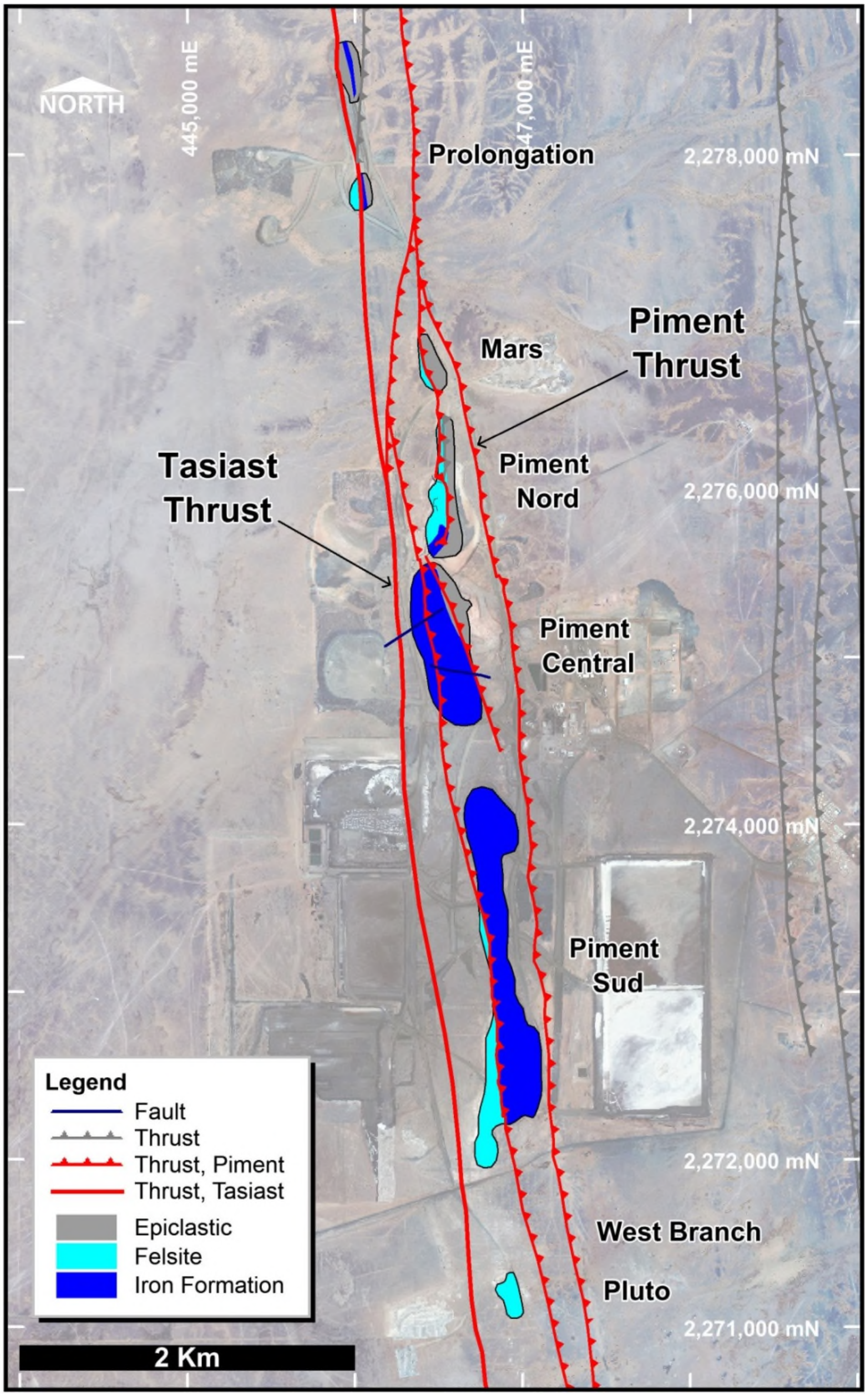


Figure 7-5 Geological Plan View of West Branch and Piment

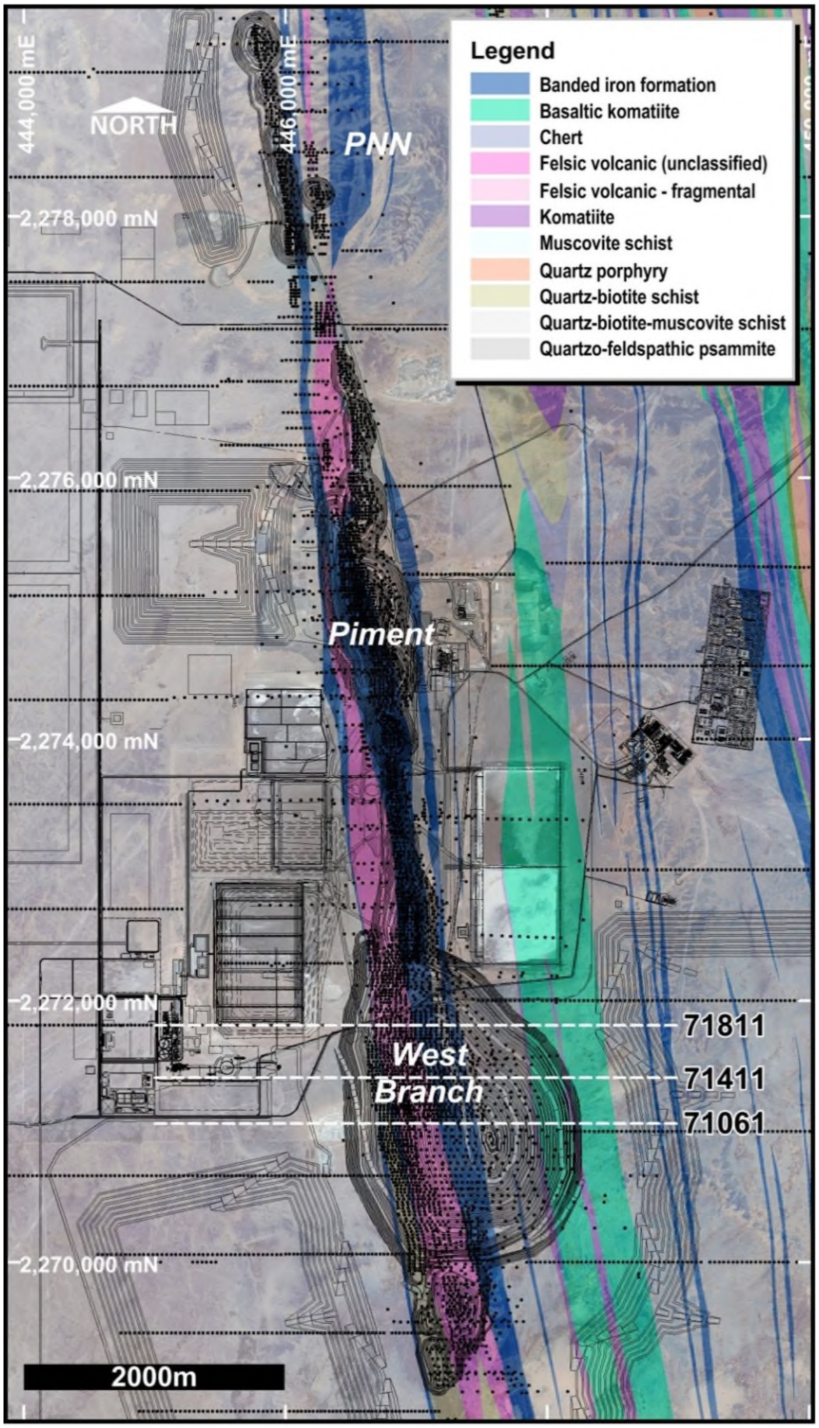
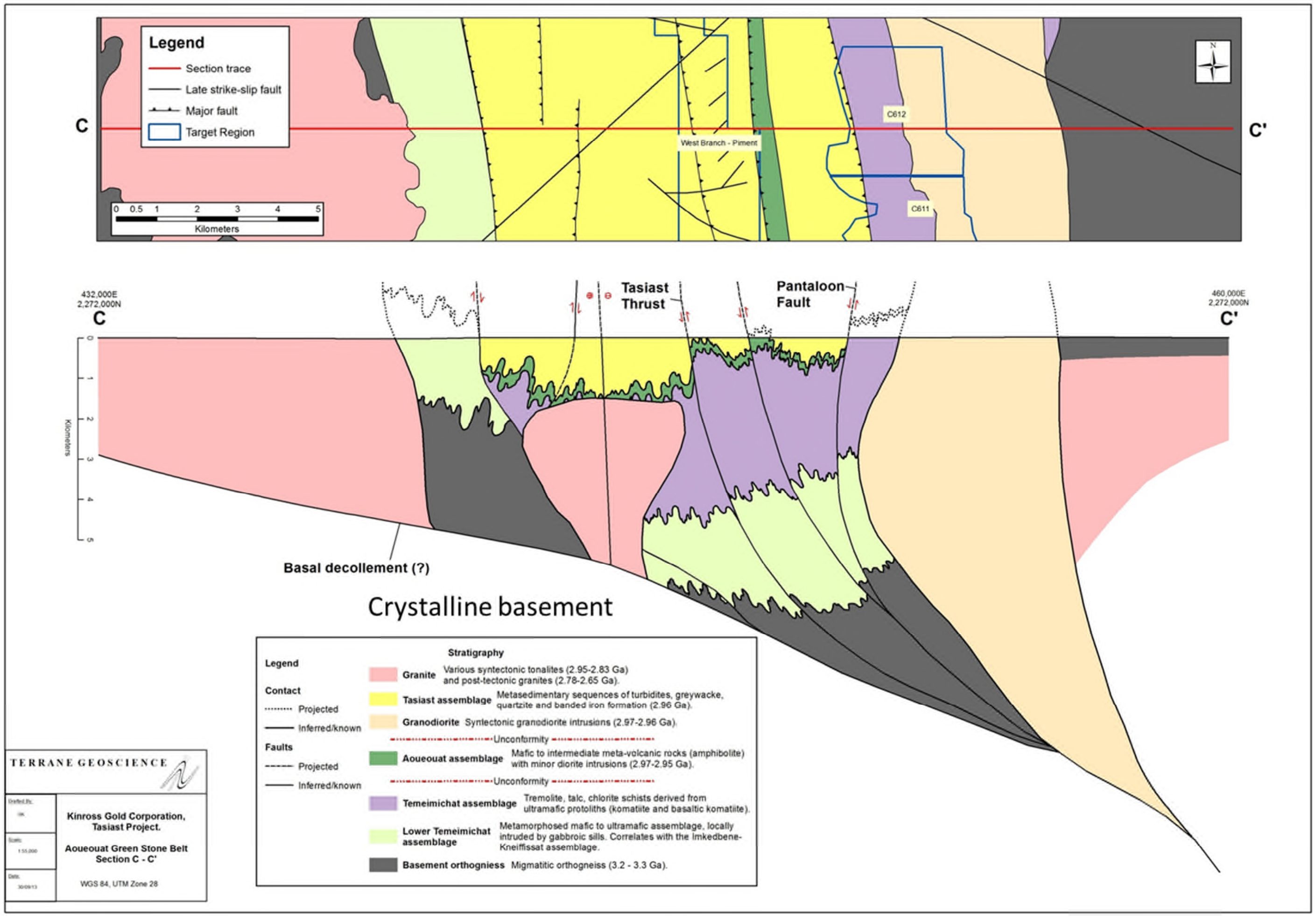


Figure 7-6 Interpreted Structural Profile at 72,000N



Lithologies

Greenschist Zone – Diorites and Basalts of the Aouéouat Assemblage

The Greenschist Zone hosts the economic mineralization at West Branch. It includes two units logged as mafic metavolcanic and meta-intrusive rock types. The contact between the two units ranges from sharp to gradational. At West Branch, the Greenschist Zone consistently averages 50 m to 100 m in thickness over a strike length that exceeds 2,000 m. The thicknesses of the main lithological units range from 5 m to 50 m for the meta-intrusive (average 35 to 40 m) and 20 m to 80 m for the mafic metavolcanic rocks. The Greenschist zone rock units have been intersected by deep drilling at Piment, but are generally thinner there (<20 m) than at West Branch. Between Piment Central and Piment North, the Greenschist zone is up to 80 m thick and dominated by mafic metavolcanic rocks.

Mafic Metavolcanic Rocks – Basalt

Mafic metavolcanic rock is interpreted to be the oldest rock unit within the mine stratigraphy. It is a dark green to brown, mafic, very fine to fine-grained rock that is composed of varying amounts of actinolite, biotite and plagioclase. It is locally garnet bearing. The rock preserves a moderately strong to strong foliation defined mainly by the alignment of biotite crystals. Previously, the unit was logged as an intermediate volcanic; it is now logged as basalt, which is the most likely protolith.

Mafic to Intermediate Meta-intrusive rocks – Diorite, Quartz Diorite

This light to medium grey, medium to fine-grained rock is dominantly composed of plagioclase, quartz and biotite with potassium feldspar present in amounts up to half of the plagioclase content. The unit typically comprises a distinctive penetrative foliation that is most strongly expressed by the alignment of biotite crystals. It was previously referred to as plagioclase-biotite schist and was logged as schist (SHT) and biotite schist (BST). The logging terms GDI (diorite) and GDQ (quartz diorite) have now been adopted. For geological modelling purposes, the GDI and GDQ can be grouped together. Limited geochemical data suggest the rocks are fractionated derivatives of basaltic magmas.

Felsite of the Aouéouat Assemblage

Stratigraphically overlying the Greenschist Zone is a cream coloured, microcrystalline quartzo-feldspathic rock that locally preserves quartz and/or plagioclase phenocrysts. This siliceous unit is logged as felsite (FVC) and occurs discontinuously throughout most of the Piment and West Branch deposits. At West Branch, the FVC unit is present both structurally above and below the Greenschist Zone. The unit is strongly sheared and preserves a well-developed phyllosilicate foliation. The rock is interpreted to represent pyroclastic formations and possibly flows of dacitic to rhyolitic compositions. Within the FVC, near to its contacts with other units, a cream coloured rock locally occurs that hosts fuschitic (chromium rich) mica. That rock is considered to represent bleached, highly altered mafic metavolcanics possibly derived from the basalt unit of the Greenschist Zone.

Banded Iron and Magnetite Formation of the Tasiast Assemblage

The banded iron and magnetite formation (BIM) unit is composed of alternating layers of dark greenish magnetite-grunerite and light gray quartz-feldspathic compositions. The rocks are generally strongly magnetic and they are interpreted to represent metamorphosed iron-rich siltstones and argilites that are the upper parts of complete or partial Bouma (turbidite) sequences. The rocks are logged as BIM, and they belong to the lower stratigraphic levels of the Tasiast assemblage.

Layers within the BIM vary in thickness from 0.2 cm to a few centimetres and typically have sharp contacts. Mesoscopic folds within the unit are typically very tight to isoclinal. The BIM unit has a thickness up to 200 m in the hanging wall of the Tasiast deposits and it is also intersected in drill holes in the footwall of the Tasiast thrust system where it is less than 15 m in thickness.

The contact between the hanging wall BIM and the FVC units is locally defined by the presence of a discontinuous conglomerate that contains abundant clasts derived from the FVC unit and a subordinate proportion of clasts derived from mafic metavolcanic units. Locally, the conglomerate presents a laminated texture where it is strongly sheared. The presence of the conglomerate at the base of the Tasiast assemblage suggests the presence of an erosional unconformity.

Siliciclastic Metasedimentary Rocks of the Tasiast Assemblage

The bulk of the Tasiast assemblage rocks that overlie the FVC in the Tasiast mine area are siliciclastic metasedimentary rocks (SVC), including several facies of which greywacke and siltstone are the most abundant. Conglomerate, microconglomerate, arkose and some feldspathic sandstone are also present. Numerous turbidite and partial turbidite sequences composed of the previous rock types are identified. Locally, the BIM type metasedimentary rocks are finely interbedded with these facies and can be seen to form the upper parts of turbidite sequences. The logging code SVC is inherited from early days of exploration when the rocks were interpreted to be volcaniclastic sediments.

Mafic Dikes

Mafic dikes that are post schistosity and post mineralization are dark olive green, fine to medium-grained and are locally plagioclase phyric. Dikes are typically less than 5 m wide, weakly magnetic and have locally developed hornfelsed and brecciated margins with a carbonate-chlorite assemblage. The dikes are dominantly barren and crosscut mineralized units.

7.3 Structural Geology

The West Branch and Piment deposits are hosted within a package of strongly folded and sheared rocks in the hanging wall block of the Tasiast thrust system. The Tasiast thrust is the master (lowermost) structure, and higher level shear zones are interpreted to be splays. Modelling and interpretation of high-resolution gravity data gathered by Fugro Ground Geophysics for Kinross in 2013 shows the deep geometry of the thrust system underlying the Aouéouat belt. It is very likely that the Tasiast thrust is linked to a system of deeply rooted ductile faults (Figure 7-6). The deeply rooted faults would have formed as normal faults during the basin forming rift event that led to the deposition of the Tasiast assemblage. The deep-seated faults served to channel mineralizing fluids from the deep crust and lithosphere to West Branch and Piment.

Fold repetition is evident within the Tasiast thrust system and to the east of it. The dominant foliation in the mine sequence is moderately to steeply east-dipping (40° to 65°). Overall, the dip angle of the thrusts and of the foliation steepens from the southern part of West Branch toward the north where it is subvertical near Prolongation.

The Tasiast thrust and its splays are typically 0.5 m to 10 m thick and are characterized by a laminated foliation with locally preserved mylonitic textures. The fault splays are commonly situated at the contacts of lithologic units. Hydrothermal alteration assemblages, sulphides and quartz veins are commonly spatially associated with the principal thrusts and splays.

Previous structural interpretations of the mine area suggested either a thrust system developed with a ramp-flat geometry (Davies, 2011) or shear zones developed along fold limbs (Goodman, 2011). In fact the two interpretations are not mutually exclusive. Kinross interprets that the vast bulk of thrust displacement on the Tasiast thrust system was accommodated by the main (basal) thrust and that some of the recognized splays represent shears that developed on the limbs of isoclinal folds in the hanging wall block of the main thrust.

At West Branch, first-order controls on the ore bodies include an interpreted periclinal antiformal fold in the hanging wall of the Tasiast thrust where the core of the antiform would be occupied by the Aouéouat assemblage rocks. The culmination of the interpreted antiform is located around Section 71,811N. The other principal structural controls on mineralization are the Tasiast thrust and its splays. The splays are second-order thrusts that served to channel fluids from the main thrust.

Lithological–structural profiles through the West Branch deposit at 71,411N, 71,811N and 71,061N are provided in the upper panels of Figure 7-7 to Figure 7-9. The profile lines are located in Figure 7-5. The profiles display the geology (top panel) and main alteration assemblages (bottom panel). The principal shear zones are indicated by dashed lines. The view is looking to the north.

The thrusts dip consistently eastward at moderate (45° to 55°) angles and are identified as zones of intense foliation, often situated on contacts between lithological units. This suggests the thrusts are the result of strain localization at the lithological contacts. Quartz-carbonate veins sets occur sub-parallel and oblique to foliation and range in style from boudinaged, buckled, folded to planar. The veins certainly formed in extensional and/or Riedel shear orientations and were progressively folded, rotated, locally boudinaged and partially or wholly transposed parallel to the foliation. In the core of the West Branch Greenschist Zone vein, densities are typically higher in the meta-intrusive dioritic unit (averaging between 2% to 5%) than in the meta-basalt (<2%). This higher density suggests the coarser-grained feldspar-rich dioritic facies focused stresses and readily developed brittle-ductile shears, as expected for quartzo-feldspathic rocks under retrograde Greenschist metamorphic conditions. Along the margins of the West Branch deposit, both the dioritic and meta-mafic volcanic units have a low vein density (<1%). Quartz-carbonate veins also developed locally within FVC that envelops the Greenschist Zone and within the footwall meta-sedimentary units.

Figure 7-7 Profiles through the Greenschist Zone at West Branch at 71,411N

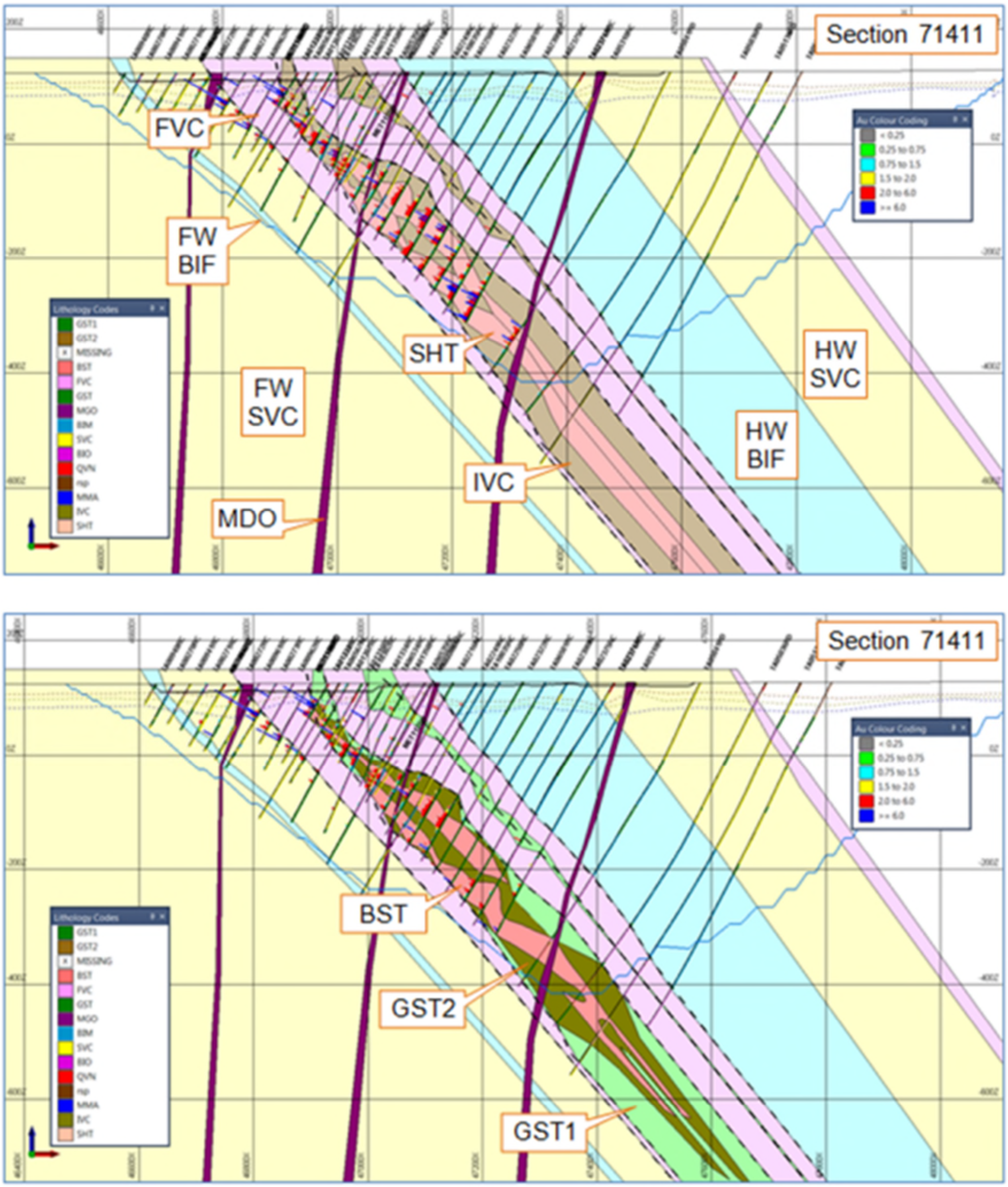
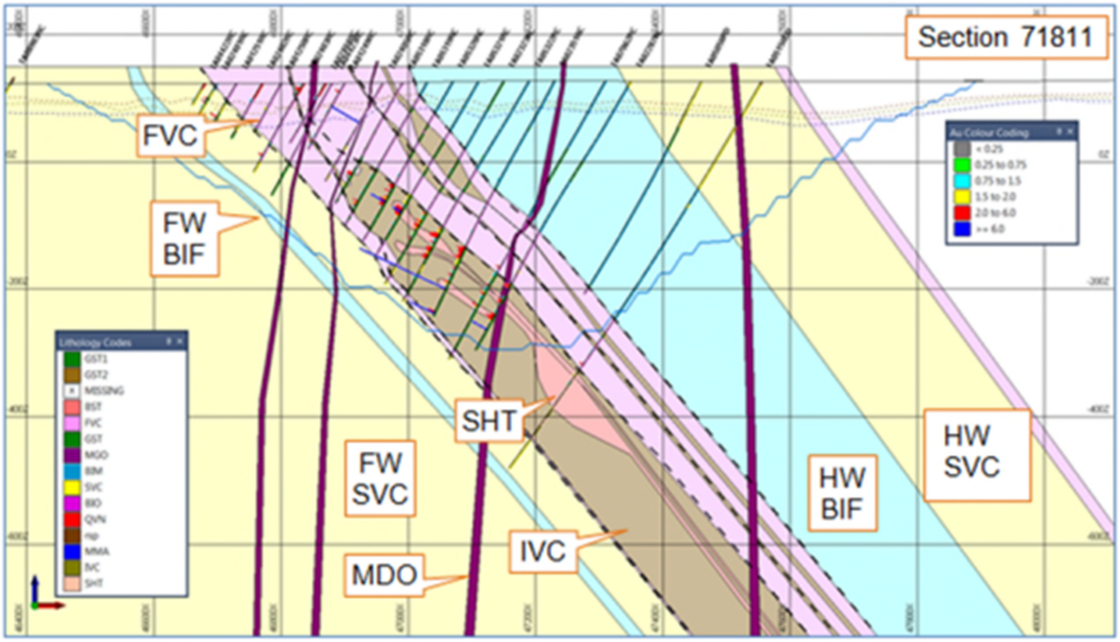


Figure 7-8 Profiles through the Greenschist Zone at West Branch at 71,811N



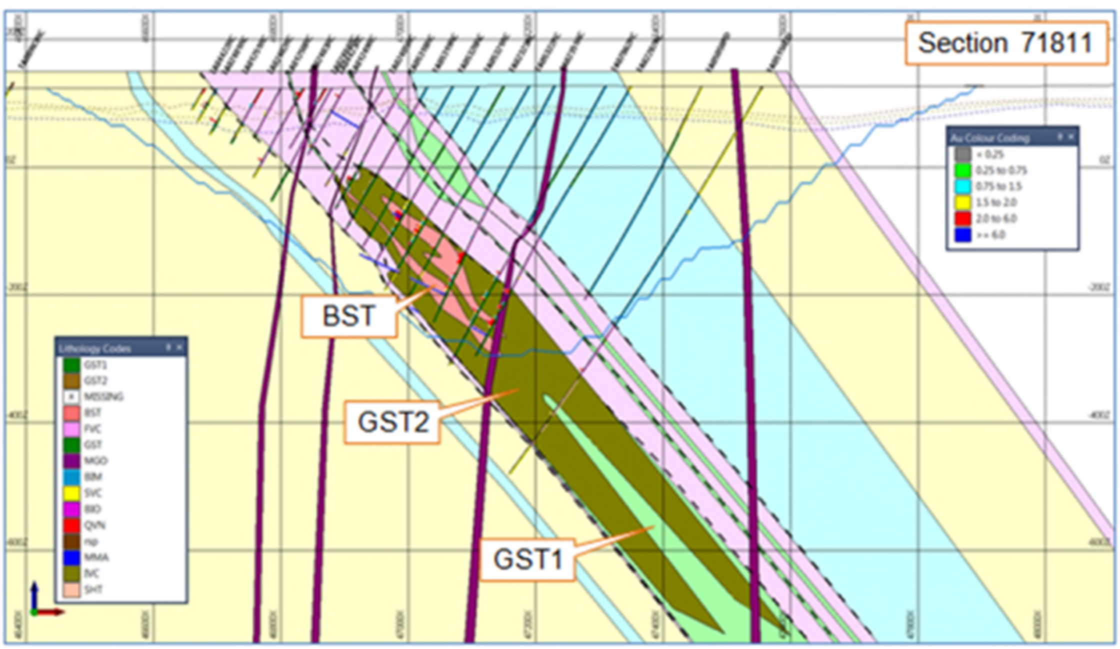
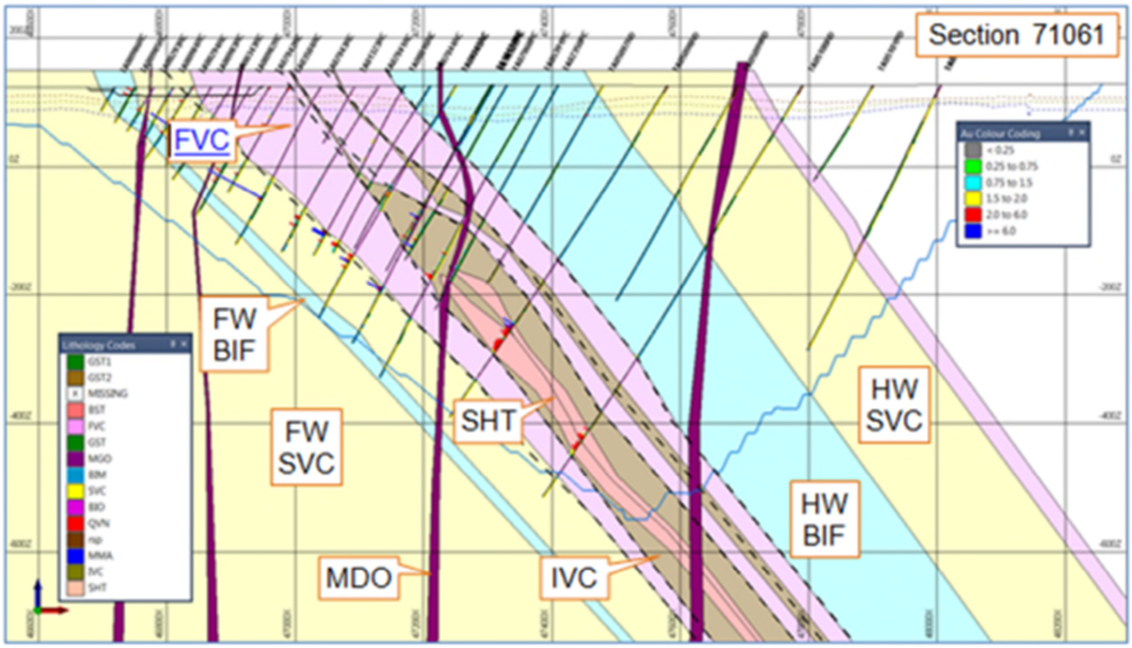
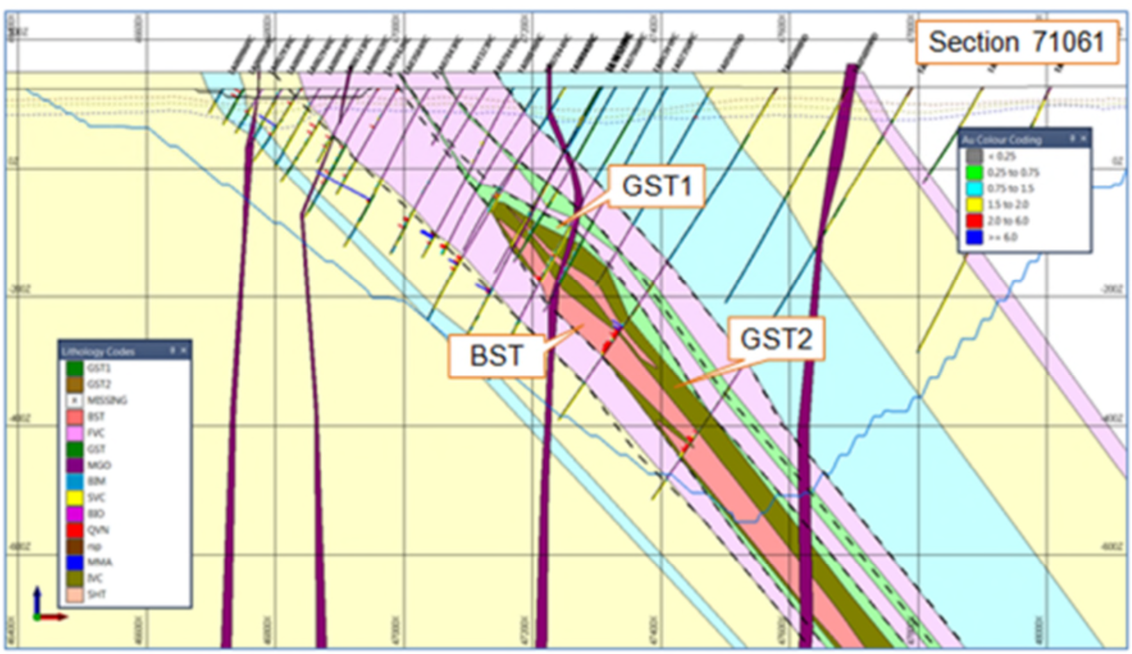


Figure 7-9 Profiles through the Greenschist Zone at West Branch at 71,061N





7.4 Mineralization and Alteration

Gold mineralization and alteration assemblages at the Tasiast deposits have been defined over a strike length of greater than 10 km and to vertical depths of at least 740 m. Alteration profiles through the West Branch deposit at 71,411N, 71,811N and 71,061N are provided in the lower panels of Figure 7-7 to Figure 7-9. Alteration mineral associations within the Greenschist Zone at West Branch include a central core of biotite-carbonate-quartz-pyrrhotite-pyrite ± magnetite ± garnet ± tourmaline ± leucoxene (coded as BST alteration zone) that is mainly developed within the mafic to intermediate intrusive rocks (Pollard, 2011). Intense silicification is also locally well developed in the same rocks where it is associated with sheeted quartz veins. A strong correlation exists between the +2 g/t Au grade shell and quartz vein densities of 1% to 5%, sulphide volume of 0.5% to 2.0% (pyrrhotite dominated) and a medium abundance of carbonate in both the veins and groundmass.

The BST alteration grades outward to a biotite-amphibole-carbonate-quartz-pyrite-pyrrhotite ± magnetite ± garnet alteration assemblage that itself grades into an amphibole-biotite-garnet ± carbonate ± quartz ± pyrite ± pyrrhotite assemblage. These two alteration assemblages are respectively coded as GST2 and GST1 and are hosted mainly in the mafic meta-volcanic unit, and to a lesser degree in the mafic to intermediate intrusive rocks.

Volumetrically minor mineral alteration associations include sericite-zoisite ± quartz ± carbonate detected on the margins of the GST2 and GST1 and a late overprinting albite-sericite-carbonate assemblage observed locally throughout the West Branch deposit. The FVC units have been variably altered by albite, biotite, muscovite, pyrite and quartz, and contain variable quartz vein densities.

At West Branch, several types of veins are recognized within the Greenschist package and include the following (in order of paragenesis):

i. Early milky white quartz that are commonly folded and boudinaged and oriented close to the foliation plane.

ii. Quartz-albite-tourmaline veins that are boudinaged, attenuated and oriented close to the foliation plane.

iii. Greenish gray to translucent quartz ± native gold veins with a silica selvedge that are oriented at low to moderate angles to the foliation plane.

iv. Quartz-amphibole-biotite-carbonate-pyrite-garnet veins that are oriented at low to moderate angles to the foliation plane.

v. Quartz-carbonate-biotite ± actinolite ± pyrite ± pyrrhotite ± tourmaline ± magnetite ± garnet veins that crosscut foliation at a high-angle.

vi. At Piment, the main mineral associations comprise magnetite-quartz-pyrrhotite ± actinolite ± garnet ± biotite. Sulphide replacement of magnetite is best observed within the banded iron magnetite units. The assemblage is spatially associated with the main shear zone as well as splays.

The deposits can be approximated as slightly flattened prolate ellipsoidal shoots that have moderate dips to the east and a shallow to moderate (approximately 30°) southerly plunge. Notable variations between the West Branch and Piment deposits are the rocks hosting mineralization and the continuity of individual mineralized shoots. The majority of mineralization at West Branch is hosted in the altered and veined Greenschist Zone that is bound by footwall and hanging wall FVC.

A strong correlation is established between the BST altered Greenschist Zone rocks and a continuous +2 g/t Au shell at West Branch. The +1 g/t Au grade shell correlates with the portions of the mafic meta-volcanic rocks, whereas the 0.6 g/t Au shell extends locally into the FVC. The low-grade halo (ranging from 0.25 g/t Au to 0.60 g/t Au) extends further into the FVC and is also developed over approximately 100 m horizontal widths in the SVC footwall.

At West Branch, individual shoots that define +2 g/t Au grade shells are continuous over a strike length of at least 1,000 m and have a down dip extent between 100 m and 400 m. In comparison, Piment mineralization is largely hosted along faults and within the adjacent altered rocks. Individual mineralized shoots at Piment have been continuously defined over a 300 m strike and range in thickness from 5 m to 40 m.

Metallic minerals within the Greenschist Zone are dominated by pyrrhotite, pyrite and native gold that occur as vein infill and alteration spots commonly in and around the foliation planes in proximity to veins. Pyrrhotite and pyrite occur together in many places, but in variable ratios. Zones of pyrite-only and pyrrhotite-only are rare.

Blake (2011a, b) studied the nature, grain size and mode of occurrence of native gold grains in seven composites from two drill holes that intersected Greenschist Zone mineralization. Results from the study concluded that the volume of coarse (>100 µm) gold grains account for the majority of contained gold and greater than 60% of grains occur in the liberated form and a subordinate portion associated with gangue minerals and micrometre grains detected in slimes. At Piment, the metallic minerals include native gold, pyrrhotite and pyrite. Other sulphide minerals recognized at both deposits in minor to trace abundances include electrum, chalcopyrite, arsenopyrite, sphalerite, covellite, pentlandite and petzite.

The oxide zone is characterized by leaching of quartz-carbonate veins and sulphides, and precipitation of calcrete and iron hydroxides in voids and fractures. No well-defined transition zone was identified as the strongly weathered upper portion of the deposits grades into fresh rock at depth. The depth of oxidation is in the order of 30 m to 60 m, at an average of 40 m. No supergene enrichment of gold is apparent in the upper profile.

8. DEPOSIT TYPES

The Tasiast deposits are hosted in Archaean volcanic-sedimentary sequences that have been deformed and metamorphosed to lower amphibolite peak metamorphic grade. Mineralization is both structurally and lithologically controlled, epigenetic in style and was coincident with early stages of post-peak metamorphic retrograde Greenschist P-T conditions.

The regional geological setting and deposit features at Tasiast are similar to other well-known Archaean cratons and greenstone belts that host major gold camps. Examples of analogue terranes of similar ages to the Aouéouat belt include the Kaapvaal craton in South Africa, the Pilbara craton in Australia and the Wyoming craton in the USA. The Aouéouat belt also shares some similarities with gold-rich Late Archaean terranes, such as the Yilgarn in Australia and the Abitibi in Canada.

9. EXPLORATION

Exploration has been undertaken by TMLSA, its precursor companies (e.g. gold exploration by NLSD), or by contractors (e.g. geophysical surveys).

9.1 Grids and Surveys

The Coordinate System used on site is UTM (WGS84) 28 North, however a translation constant is used (Easting of 400 000 and Northing of 2 200 000). The Original Control has been set out by IPH Engineering and ten control points are set out across the mine. Surveyors use a differential GPS for surveying at the mine.

9.2 Geological and Regolith Mapping

Numerous phases of geological and regolith mapping have been undertaken during the life of the project, and range from regional (1:150,000) to prospect (1:12,500) scale. Work was completed by the BRGM, SNIM, NLSD, Defiance Mining Corporation, Red Back and Kinross. Mapping was facilitated by good outcrop, RC drilling chips, high resolution satellite imagery and detailed airborne geophysical data. Results were used to identify areas of alteration, structural complexity, quartz-carbonate veining, and sulphide outcrop that warranted additional work.

9.3 Geochemistry

Soil, grab and rock sampling were used to evaluate mineralization potential and generate drill targets. Approximately 47,700 surface samples, including mostly soil and rock chip, have been taken over the life of the project area. From 2011 to date, TMLSA expanded the extent of the historical surveys and collected an additional 12,800 soil samples for both gold and multi-element analyses, and approximately 5,000 rock chip samples. Surface sampling was used as a first-pass exploration tool to identify areas of geochemical anomalism; some of these anomalies remain to be studied further.

9.4 Geophysics

Ground and airborne magnetic surveys were completed by NLSD and Red Back and used to delineate intrusive rocks, banded iron-formations, fault structures, and sulphide-rich zones at depth. Red Back also completed an electromagnetic survey (VTEM EM) in 2008. TMLSA completed a detailed airborne magnetic and radiometric survey across the mining permit and exploration permits in 2011. A small ground induced polarization (IP) survey was also conducted across a portion of the West Branch deposit, with subsequent IP surveys completed on near mine and district targets in 2013. In 2013, TMLSA also completed a ground gravity survey across the mining permit and exploration permits.

9.5 Pits and Trenches

Excavation of trenches as an exploration technique has been very successful and was extensively used during the NLSD phase of exploration, when 55 trenches (26,593 m) were excavated, and an additional 27 trenches (1,309 m) were hand-dug. Significant gold intersections in trenches typically overlay sub-surface zones of similar grade and width, as defined by subsequent drilling. TMLSA completed 18 trenches from 2011 to the end of 2013, for a total of 3,942 m.

9.6 Drilling

Drilling completed on the Project is discussed in section 10.0.

9.7 Bulk Density

Bulk density determinations are discussed in section 11.0.

9.8 Petrology, Mineralogy and Other Research Studies

In 1999, NLSD collected and carried out a petrographic and mineralogical study of 10 core samples selected from five drill holes that intersected Piment Central mineralization. The mineralogical and petrographic study noted the following (Bailly et al., 1999):

- Opaque minerals of the surrounding rocks consist of abundant magnetite crystals frequently associated with graphite;
- The mineralization assemblage consists of pyrrhotite with minor chalcopyrite and electrum ± arsenopyrite;
- Mineralization occurs as veinlets with carbonate, biotite and, locally, blue tourmaline impregnated along bedding planes within the host rock.

In 2010, Red Back submitted 10 core samples from West Branch for a petrological and mineralogical study. Results from the work indicated significant pyrrhotite mineralization developed along foliation planes and associated with accessory magnetite, chalcopyrite and pyrite (Strashimirov, 2010).

Further petrological studies were carried out for TMLSA in 2010 including work by Leitch (2010) and Larson (2011), followed by a gold characterization study in 2011

(Blake, 2011a, b). Transmitting and reflecting microscopes were acquired for the Project in 2011 and selected slabs are submitted for thin sections. From 2011 to date, approximately 220 thin sections have been generated by TMLSA for mineral and rock identification purposes.

9.9 Exploration Potential

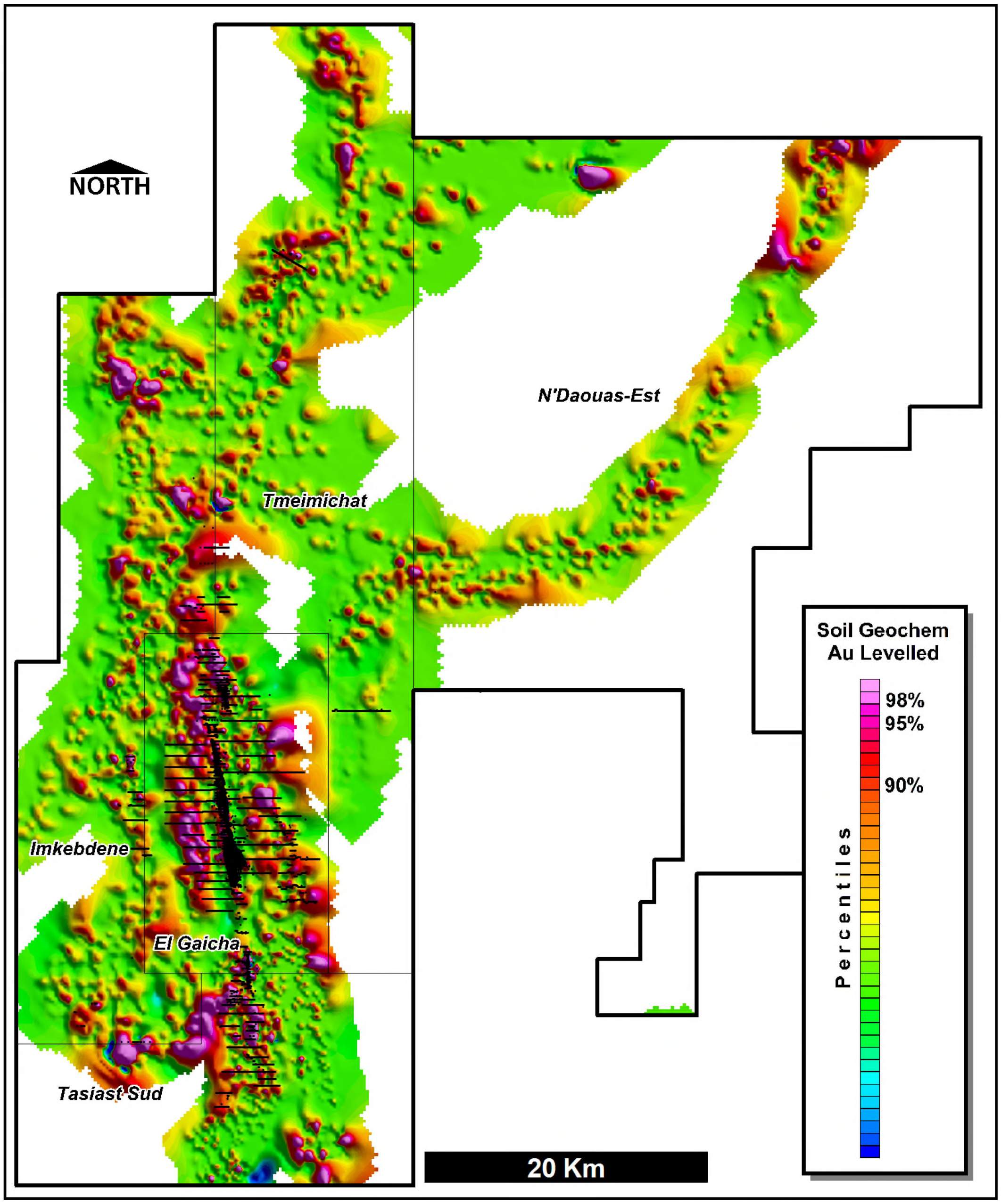
Mineralization remains open both laterally and vertically at the Tasiast deposits. Shallow reconnaissance RC drilling across district targets has detected gold mineralization that requires follow-up exploration. The potential to delineate additional resources both at the Tasiast mine and the district targets is considered good. Some of these district targets are described below.

Drilling uncovered encouraging gold results at the C67-Fennec sector and Tasiast Sud (previously named Charlize) targets, located 5 km north and 15 km south of Tasiast. Drilling has also intersected encouraging gold grades within the Tmeimichat and Imkebdene permitted areas, situated approximately 10 km north of the mine site.

At the Tasiast Sud targets, mineralization occurs near the surface in BIM rocks that are part of the same mineralized BIM sequence further north at Tasiast. Mineralization has also been identified in sheared amphibolites that host quartz-carbonate veins where they are thrust over younger metasedimentary rocks. Gold mineralization at C67 is hosted by sheared and veined mafic metavolcanic rocks and by quartz diorites that may be intrusive into the metavolcanics. The gold is associated with quartz-carbonate veins that formed during the late stages of shearing. Besides quartz and carbonates, the vein mineralogy consists of pyrrhotite, pyrite, biotite and locally tourmaline. Pyrrhotite is generally more abundant than pyrite where significant gold grades are present. The structures controlling mineralization are moderately to steeply west-dipping shear zones that define a trend that is parallel to, but to the east of the Tasiast trend.

The Project area has considerable additional exploration potential as illustrated by the extensive levelled gold in soil anomalies shown in Figure 9-1. Several areas have been identified by TMLSA on the current Tasiast permits for near term drill testing. Targeting incorporates all available data sets including satellite imagery (Worldview-2), regional gold-in-soil data, airborne geophysical data (high resolution aeromagnetics and VTEM), regional scale geological maps of two generations, several generations of target scale geological maps, multi-element geochemical data derived from RC drill chips, assay results derived from historical and recent property scale drilling and trenching, reconnaissance geological observations and prospecting.

Figure 9-1: Levelled Gold in Soil Grid



10. DRILLING

The total number of drill holes completed on the Project totals 15,129 holes (14,117 RC holes, 808 diamond core holes and 204 RC pre-collar with diamond tail holes) for an aggregate total of 1,613,919 m. Resource drilling campaigns completed between 1999 and 2013 comprise 3,890 RC (620,106 m) and 290 diamond core holes (89,735 m) and 163 RC pre-collar with diamond tails (118,068m) for a total of approximately 827,909 m (Table 10-1).

Drill programs were completed primarily by contract drill crews, supervised by geological staff of the Project operator. Where programs are referred to by company name, that company was the Project manager at the time of drilling and was responsible for the collection of data.

10.1 Drilling Methods and Equipment

Normandy LaSource Development Ltd. Drill Programs

NLSD completed two drill campaigns between 1999 and 2000, including 412 RC holes for 32,463 m (including 43 RAB holes totalling 2,135 m) and 47 core holes for 5,456 m.

Drilling was initially undertaken on 200 m spaced east-west sections with 50 m hole centres on section, to depths of 50 m to 100 m. Drilling methods were predominantly RC with lesser core drilling (HQ; 63.5 mm core diameter) and included core tails to some RC holes (NQ; 47.6 mm diameter core).

Defiance Drill Programs

From March 1 to June 18, 2003, a total of 303 RC drill holes (25,812 m) were completed on the Piment zone by Defiance. RC drill holes were drilled in between old NLSD RC holes along drill fences at 25 m spacing along east–west fences. The majority of the RC drill holes were drilled at an azimuth of 270° (grid orientation) and at an inclination of -60°. From March 2004 to October 2004, a total of 112 RC holes (8,947 m including 4 RC pre-collars of four deep core drill holes) were completed on the Piment Zone and to the west (as sterilization of the waste dumps and tailings dam areas). RC drill hole diameters were 130.4 mm (5 5/16”).

From March 1 to May 25, 2003, a total of 29 core drill holes (2,908 m) were completed on Piment Central, Piment South, and Piment North (southern extension) by Defiance. Core diameter used was HQ3 (61.1 mm core diameter) for 25 of the 29 drill holes, while one core hole (SC062) was drilled utilizing NQ core diameter. Seven of the core holes were drilled primarily for geotechnical purposes and three vertical PQ3 (83 mm core diameter) drill holes (SC059, SC060 and SC061) were drilled to collect samples for metallurgical test work.

Table 10-1: Global and Resource Drill Hole Summary

Global Drill Holes - Collar File

Company	Year	Reverse Circulation * Qty	Reverse Circulation * Metres	RC Pre-collar & Diamond Tail Qty	RC Pre-collar & Diamond Tail Metres	Diamond Qty	Diamond Meters	Total Holes	Total Metres
Normandy - La Source	1999	355	28,447	-	-	11	585	366	29,032
Normandy - La Source	2000	57	4,016	-	-	36	4,871	93	8,886
Defiance	2003	303	25,812	4	1,417	29	2,908	336	30,138
Defiance	2004	112	8,947	-	-	-	-	112	8,947
Rio Narcea	2006	9	1,435	-	-	-	-	9	1,435
Rio Narcea*	2007	242	25,110	1	173	72	7,927	315	33,210
Rio Narcea - Red Back*	2008	1,022	112,777	-	-	23	2,716	1,045	115,493
Red Back	2009	2,946	203,324	1	300	26	2,681	2,973	206,306
Red Back - Kinross*	2010	2,904	243,666	75	52,628	66	13,265	3,045	309,559
Kinross*	2011	2,811	288,543	99	68,516	176	98,410	3,086	455,469
Kinross*	2012	2,693	267,175	12	1,892	287	66,329	2,992	335,396
Kinross*	2013	663	63,341	12	2,300	82	14,406	757	80,047
Grand Total		**14,117**	**1,272,592**	**204**	**127,227**	**808**	**214,100**	**15,129**	**1,613,919**

* Diamond Drill Holes include Met & Geotech Holes 213 holes @ 43,932 m (from 2007 to 2013)
1999 Reverse Circulation Holes include 43 holes @ 2,135 Rotary Blast holes

Resource/Reserve Model Drill Holes - Collar File

Company	Year	Reverse Circulation Qty	Reverse Circulation Meters	RC Pre-collar with Diamond Tail Qty	RC Pre-collar with Diamond Tail Meters	Diamond Qty	Diamond Meters	Total Holes	Total Metres
Normandy - La source	1999	219	19,336	-	-	-	-	219	19,336
Normandy - La source	2000	-	-	-	-	31	4,358	31	4,358
Defiance	2003	299	25,644	4	1,417	28	2,798	331	29,859
Defiance	2004	28	6,011	-	-	-	-	28	6,011
Rio Narcea - Red Back	2007	235	23,890	1	173	61	7,611	297	31,674
Red Back	2008	872	97,506	-		-	-	872	97,506
Red Back	2009	759	113,045	1	300	-	-	760	113,345
Red Back - Kinross	2010	735	153,577	70	50,966	-	-	805	204,543
Kinross	2011	700	171,119	87	65,212	89	59,812	876	296,143
Kinross	2012	35	9,458	-	-	29	8,296	64	17,754
Kinross	2013	8	520	-	-	52	6,860	60	7,380
Subtotal		**3,890**	**620,106**	**163**	**118,068**	**290**	**89,735**	**4343**	827,909

From March 2003 to October 2003, four RC holes with core tails totalling 1,417 m were completed on the Piment Zone to check the down-dip extension of the northern Piment Central shoot.

Rio Narcea Drill Programs

During late 2006 and into 2007, Rio Narcea drill programs used an in-house CS2000 drill rig and an RC rig supplied by Drillcorp Sahara. During the period to August 10, 2007, before Red Back acquired the Tasiast Project, Rio Narcea drilled RC and diamond core holes. The RC drilling was specifically aimed at testing the northern extensions of the Piment Zone and infill drilling at the West Branch prospect.

2007-Sept 2010 Red Back Mining Drill Programs

Following the acquisition of the Project, Red Back commenced an aggressive program of RC drilling to fully define the mineral resources in the Piment and West Branch mineralized zones. Red Back also used a small RC rig to conduct shallow (40 m) open face RC-style drilling on district targets.

In 2010, Red Back initiated a diamond drilling program for deep exploration beyond the depth limit of the RC rigs, metallurgical test work relating to the dump and heap leach potential of the Project, and for quality control purposes. The majority of holes included RC pre-collars drilled to depths ranging from 200 to 300 m and were tailed with HQ core. From January to September 2010, Red Back completed 175,454 m of RC and 24,120 m of core.

From 2007 to September 2010, Red Back completed a total of 6,174 RC holes totalling 509,200 m and 139 core holes totalling 28,052 m.

Sept 2010-Present TMLSA Drill Programs

TMLSA commenced a ramp up in exploration activity following completion of the Red Back acquisition.

The majority of the drilling that TMSLA completed between September 17, 2010, and December 31, 2010, focused on the West Branch deposits to expand the mineral resource, to test the extents of mineralization, and to continue with metallurgical test work. The majority of the holes included RC pre-collars drilled to depths ranging from 200 m to 300 m and were tailed with HQ core. Drill rigs used included the following types: Schramm T685WS, EDM2000 RC, EDM2000 core, UDR200, UDR650, UDR1200, Golden Bear 1400, Silver Bear A5, Coretex YDX-1800 and YDX-3L, and LF230.

During 2011, TMLSA completed 2,811 RC holes (288,543 m), 99 RC collar with diamond tail holes totalling 68,516m, and 176 core holes (98,410 m). A total of 15 drills were operating on site by year end 2011.Logging Procedures

For the Defiance, Red Back and TMLSA RC drill programs, a field geologist described the rock chips (dry and wet), and then placed a representative sample into pre-labelled plastic RC chip boxes. Logging was performed on hard-copy sheets by Defiance and data recorded included drill hole ID, sample number and depth, oxidation state, colour, presence of water, sample weight, lithologies, grain size, structure, alteration, vein types, and sulphide types and percentages. Red Back and TMLSA logging was recorded directly in digital format at the rigs. Data recorded included drill hole ID, sample number and depth, oxidation state, colour, presence of water, and lithologies. Selected RC holes from the West Branch region have been relogged for alteration and mineralization.

The Defiance, Red Back and TMLSA core logging geologists completed all geological and geotechnical descriptions and these were recorded on separate geological and geotechnical hard copy log sheets. All geological and geotechnical descriptions were encoded and standard codes were utilized during the program.

Once a core drill hole was completed, the detailed geological and geotechnical core drill logs were then submitted to the Geology Department for data input in Microsoft® Excel files. This procedure was replaced (near the end of 2009) with the current system of digitally recording geological information from core holes into Tough Books computers and uploading these directly into a Century System Geological Database software. Logging recorded rock quality designation (RQD), lithology, oxidation state, structural orientation, mineral association, vein types and density, sulphide types and volume, specific gravity, and point load test (PLT).

Defiance geologists mounted a digital camera on a frame with fluorescent lights for photographing all core in the boxes. Red Back and TMLSA captured core images under natural light.

10.2 Collar Surveys

During the Defiance drill programs, the Defiance surveyor surveyed the collar of each RC and diamond drill hole upon completion, using a Geodimeter 510 total station instrument. The measurement was taken on the west side of the collar. The surveyor recorded the Cartesian coordinates digitally for each drill hole.

During Red Back and TMLSA programs, the surveyor surveyed the collar upon completion of the hole using a differential (GPS). Once completed, the Cartesian coordinates were digitally recorded and emailed to the database manager.

Since 2012, the survey data were recorded, validated and imported directly into the Fusion Remote Data Base by the surveyor.

10.3 Down-hole Surveys

During the Defiance drill programs, down-hole surveys of the drill holes were completed with a Humphries gyroscope due to the strong magnetism of the BIF units. Down-hole gyroscope measurements for drill holes drilled from 0 to 60 m involved a reading taken at 10 m intervals. Measurements were taken every 20 m down the hole for drill holes below 60 m depth. The gyroscope was oriented using the Geodimeter total station relative to the local NLSD 2500 E baseline grid. All of the survey data was digitally acquired, recorded and submitted to Defiance on a weekly basis on floppy diskette by Surtron survey personnel as Excel and Word files. Defiance geologists would review the raw collar and down-hole survey data for accuracy before uploading to the database.

Down-hole surveys completed during Red Back and TMLSA programs used north seeking gyroscope (continuous and multishot), single shot reflex and Mems gyroscope instruments. Measurements from the multishot tools were at 5 m intervals. Survey data was digitally acquired, recorded and submitted to Red Back and TMLSA on a daily basis via email. The tools were tested and calibrated on a weekly basis from a set station. Additionally, from mid-2012, Optical Televiewer (OPTV) and Acoustic Televiewer (BHTV) imagery were acquired for many diamond drill holes. The OPTV and BHTV data are mainly used for structural and geotechnical analyses.

10.4 Recovery

Sample recovery for RC drilling in the NLSD campaigns is reported to have been good (Guibal et al., 2003). SRK in 2003 noted that drill core recovery was variable in the oxidized zone and high (90% to 100%) in the fresh rock.

Diamond core recovery from Red Back and Kinross drill programs were collected on all core holes. Based on 17,718 measurements the average total recovery from core runs (in both oxide and fresh) is 98% and the RQD is greater than 93%. Measurements from downhole depths below 50 m (approximate oxide-fresh boundary) returned values of 99% and 95% for total recovery and RQD, in comparison to shallower depths where total recovery is 87% and RQD averages 43%.

Reverse circulation holes after September 2013 recorded the total weight of the samples for every metre drilled.

10.5 Deposit Drilling

The Piment and shallow portion of West Branch deposits have typically been drilled at approximate 25 m x 25 m spacing, whereas the deeper portion (below 430 m vertical) of West Branch has been drilled out at 70 m x 70 m spacing.

Drill hole inclinations dominantly range between -50° and -75°, azimuths are dominantly 270°, and the mineralized zones dip between 45° and 65° to the east. Most holes therefore intersect the mineralized zones at a high angle; however the drill hole intercept widths reported for the Project are not true widths. Depending on the dip of the drill hole, and the dip of the mineralization, drill intercept widths are typically greater than true widths.

An example of the relationship between true widths, drill intercepts, lithologies and gold grades for intervals in the drill holes is shown on the cross-section included as Figure 7-6.

10.6 Geotechnical, Hydrological and Metallurgical Drilling

Geomechanical drilling campaigns were conducted across the mine lease to identify potentially suitable locations for mine infrastructure. The campaigns required detailed knowledge of rock mass characteristics and ground-water conditions. Hydrological drilling was conducted in the area of the water bore field, to identify sufficient water for processing.

Large diameter core holes were completed to provide sufficient sample for metallurgical test work.

The various geotechnical studies completed, based upon the progress of pit development, include the following:

- Golder Associates (2003, 2004): This study provided preliminary pit slope design parameters for the oxide zone and fresh zone. Results were based on the analysis of geotechnical data collected in 2003 and were used in the preliminary mine planning and development of the project. The study focused on the BIF lithology as four open pits were proposed within this lithological unit. Rock testing was confined to this unit, and the drill-hole data was subject to a strong directional bias with almost all boreholes drilled at either 60° or 270° azimuths.

- Scott Wilson (2008, 2009): This study provided slope stability analysis and ultimate pit slope design parameters of the four open pits (Piment North, Central, South-North and South-South) for the oxide zone and fresh zone. Results were based on the analysis of the geotechnical data collected in 2008, which was based on an orthogonally oriented drill hole program. Geotechnical and discontinuity data were collected and processed to form eleven pit sectors based on geotechnical characteristics. The collected data contributed to filling the gaps of the 2003 study conducted by Golder Associates. The study also included a review of the seismicity and hydrogeology of the Piment site. The 2008 investigation also focused on logging the wider variety of lithologies and structural features encountered within the four open pits. Laboratory and field rock strength testing was also undertaken on representative samples to establish base design values. Each pit sector was compared to the discontinuity sets to identify kinematically feasible modes of failure. Slope designs were done for a base pit, as defined by the “$700 (ultimate) pit shell” provided by TMLSA. A series of recommendations included overall slope angle, bench stack angles, inter-ramp angles, and the structural and bedding controls based on operational assumptions adopted for safe operation.

- Scott Wilson (2010): This study provided slope stability analysis and ultimate pit slope design parameters for the West Branch area. Results were based on the analysis of geotechnical data collected in 2010, which was based on an orthogonal oriented drill-hole program. Geotechnical and discontinuity data were collected and processed to form four broad pit sectors based on geotechnical characteristics. The collected data was based on the configuration of drill holes of the West Branch proposed pit shell. The focus of the study was on logging a wider variety of lithologies and structural features encountered within the open pit footprint, based on experiences from the study conducted in 2008. Laboratory and field rock strength testing was also undertaken on representative samples to establish base design values. Each pit sector was compared to the collected discontinuity sets to identify kinematically feasible modes of failure. Slope designs were performed for a base of pit, as defined by a 30 kt/d CIL pit shell. A series of recommendations included overall slope angle, bench stack angles, inter-ramp angles, and the structural and bedding controls based on operational assumptions adopted for safe operation.

- Scott Wilson (2011): This study provided a slope stability analysis and refined the ultimate pit slope design parameters for the West Branch area. Results were based on analysis of the geotechnical data collected in 2010 and 2011. A total of 23 geotechnical and six hydrogeological drill holes were completed. Two pits at West Branch were designed (North and South pits). The northern pit assumes a 700 m pit depth, whereas the southern pit assumes a shallower depth.

- Golder and Schlumberger (2013): A geotechnical and hydrogeological drilling program for the West Branch pit was completed from March to June of 2013 (eight holes at 4,612 m). The purpose of the drilling and investigation program was to provide additional geotechnical and hydrogeological data where data are lacking and to complement those data that currently exist from previous investigation campaigns.

10.7 Comment on Drill Programs

In the opinion of the QP, the quantity and quality of the lithological, geotechnical, collar and down hole survey data collected in exploration and infill drill programs are sufficient to support mineral resource and mineral reserve estimation as follows:

- Core logging meets industry standards for gold exploration
- Collar surveys have been performed using industry-standard instrumentation
- Down hole surveys have been performed using industry-standard instrumentation
- Recovery data from core drill programs are acceptable
- Geotechnical logging of drill core meets industry standards for open pit operations
- Drilling is normally perpendicular to the strike of the mineralization. Depending on the dip of the drill hole, and the dip of the mineralization, drill intercept widths are typically greater than true widths
- Drill orientations for Tasiast are generally appropriate for the mineralization style, and have been drilled at orientations that are optimal for the orientation of mineralization for the bulk of the deposit area. Drill orientations are shown in the example cross-section (Figure 7-6), and can be seen to appropriately test the mineralization.

11. SAMPLE COLLECTION, PREPARATION, ANALYSES, AND SECURITY

Project staff of the operator at the time was typically responsible for the following:

- Sample collection;
- Core splitting;
- Delivery of samples to the analytical laboratory;
- Density (specific gravity) determinations;
- Sample storage; and
- Sample security.

11.1 Sampling Method and Approach

Geochemical and Trench Sampling

As the geochemical and trench data have been superseded by information from drilling and mining operations, these sample types are not discussed further in the Technical Report. This information is not relied on for use in geological modelling or resource estimations.

NLSD

Little information has been kept or is available regarding drilling procedures used in this drilling by NLSD.

2003-2004 Defiance and 2007 Rio Narcea

RC Sampling

All of the RC holes were sampled at one-metre intervals and each sample was collected in a large plastic sample bag that was held below the cyclone spigot by a drill helper. All samples were sent for assay except those that originated from the non-mineralized hanging wall at the start of each hole. To avoid sample contamination after a drill run was completed, blow-backs were carried out at the end of each 6.0 m run by the driller whereby the percussion bit was lifted off the bottom of the hole and the hole was blown clean. When water was encountered in the hole, the driller would dry out the hole by increasing air pressure into the hole and lifting and lowering the rods prior to continuing the drilling.

Throughout the Defiance RC drill program, logging of all RC drill holes was conducted by the field geologist at the drill site. After each drilled 1.0 m interval, the sample was weighed, sieved and split to give a 2 kg to 3 kg sample for analysis.

A representative sub-sample for geological logging was collected from the large sample bag by spearing a small diameter PVC pipe into the bag and emptying the contents of the PVC pipe into a hand sieve.

At the end of each day or at the completion of a RC hole, calico sample bags for RC drill holes completed that day were loaded onto a 4x4 pick-up truck by the field geologist and then delivered directly to the on-site sample preparation laboratory. Once the samples were unloaded from the pick-up truck and both the field geologist and laboratory technician confirmed receipt of all calico sample bags, the field geologist then registered the sample number sequence in the database.

Diamond Drill Core Sampling

Upon completion of geological and geotechnical core logging of a diamond drill hole, Defiance's core logging geologist identified the sections of core to be sampled and analysed for gold. Once identified, the core-logging geologist measured and marked out the sample intervals onto the uncut core down the right hand side of the orientation line. Individual sample intervals were recorded onto a core-sampling sheet. The core was sampled according to lithological boundaries and vein widths, but the maximum sample interval did not exceed 1.50 m in length.

At the core cutting facility the drill core boxes were stacked in ascending order so as to avoid sampling mix-ups. The core was cut on the orientation line marked by the geologist and the right hand side of the core (looking down hole) was placed in a numbered calico bag.

Once the core for a drill hole was cut and sampled, the core cutter and the core logging geologist then delivered the samples, with the core sampling sheet, to the preparation laboratory technician for sample preparation.

Red Back and TMLSA

RC Sampling Procedure

To minimize down-the-hole deviation, RC drilling is conducted with contract single and multi-purpose rigs using a standard 5½" face sampling hammer leading a 4½" rod string.

The entire sample is collected in a large plastic bag tightly clamped onto the cyclone base. The entire length of each RC hole is sampled. A one-meter sample length is used in all holes. Dry samples, of nominal 20 to 25 kg weight, are reduced in size by riffle splitting using a three stage Jones riffle splitter to about three to four kilograms, and then placed in pre-numbered sample bags for dispatch to the assay laboratory. A record is made at the drill site of the sample identity numbers and corresponding intervals, and this is also recorded in the geological log.

After September 2013, RC samples with a nominal weight from 36 kg to 40 kg (each 1 m) were collected in a large plastic bag, tightly clamped onto the cyclone base and reduced in the field by 50/50 manual riffle splitters. About 6 kg to 8 kg weight samples were placed into pre-numbered sample bags to dispatch to the laboratory. Every 20 samples, a field duplicate was collected as part of the quality assurance/quality control (QA/QC) procedure.

DH Sampling Procedure

For diamond drilling, core was transported from the drill rigs to the core facility where geological and geotechnical core logging was completed. The geologist marked one-metre intervals and orientation lines (bottom of hole) along the core axis for core cutting. A record was made at the core facility of the sample identity numbers and corresponding intervals. At the core cutting facility the drill core boxes were stacked in ascending order so as to avoid sampling mix-ups. The core was cut on the metre and orientation lines and the left hand side of the core looking down-hole is placed in a numbered plastic bag with sample ticket.

Once the core for a drill hole was cut, sampled and bag sealed, the core was then stored in a secure area (either locked 40 ft shipping container or fenced off area) for sample dispatch.

All the sampling processes for RC and diamond drilling were handled under TMLSA's chain of custody.

Quality Assurance/Quality Control Sampling Procedure

Before 2012, the QA/QC process was as follows:

- A routine analytical sample 'field duplicates' were collected every 20th sample, and submitted in blind sequence after every 20th and 21st interval of the sample stream.

- For RC samples, a further representative triplicate sample was routinely collected every 60th original sample in the sequence, and retained for later submission to a third-party, independent referee laboratory.

- Analytical ‘blanks’ were inserted every 20th original sample, and were taken from barren dune sand collected from a source distant from the mine.

- GANNET, ROCKLABS and GEOSTATS certified reference material, also known as standard reference material (SRM), in pulp form, were selected based on certain ‘resource thresholds,’ and inserted as standards every 20th sample.

- All QA/QC samples were inserted by the rig geologist at the rig.

- Grades of standards to be used were selected by the senior geologist and provided to the rig geologist in the rig box.

- TMLSA submitted 16% routine QC samples within the sample string.

- Holes were submitted by the rig geologist directly to the on-site laboratory as individual batch jobs, or dispatched from the site to Mali, Burkina Faso and South African laboratories.

In contrast to Defiance and Rio Narcea, Red Back and TMLSA routinely sampled every metre drilled for RC, and selectively sampled diamond holes, based on visual evidence of alteration and mineralization.

During 2012, a QA/QC team was established that was in charge of the sample procedure, sampling protocols, sampling transport, sampling tracking and reporting.

During 2012, the following quality control processes were implemented:

- Field duplicates were collected every 20th sample, and submitted in blind sequence after every 20th and 21st interval of the sample string. (This procedure was also performed before 2012.)

- Analytical ‘blanks’ were inserted every 20th original sample, and were taken from gold barren material (not sand) collected from a source distant from the mine. This material was submitted blind with the samples dispatched, like any other samples. RC blanks were crushed to simulate the RC sample grain size. Diamond drill blanks were used that resembled the diamond drill core size.

- GANNET, ROCKLABS and GEOSTATS certified reference material, also known as SRM, in pulp form, were selected based on certain resource thresholds, and inserted as standards every 20th sample. (This practice was also performed before 2012.)

- All QA/QC samples were inserted by the QA/QC team in the exploration yard, except the field duplicates that were generated at the drill. Grades of standards to be used were selected by the geologist that logged the RC or DD holes and they were provided by the QA/QC Team. For every batch, TMLSA inserted 16% QC samples within the sample string. Holes were submitted by the QA/QC team directly to the on-site laboratory as individual batch jobs, or dispatched from the site to Mali, Burkina Faso and South African laboratories.

Twice in 2013, 5% of the samples (pulp) were sent to a third laboratory to be checked.

Density/Specific Gravity

The results from 1,699 bulk density determinations completed by NLSD at Tasiast during previous drilling programs are available. The origin of the sample, its borehole number and sample depth was entered as an individual MS Access database file into NLSD's project database. However, information on the sample size/length, lithology and oxidation state was not recorded in the NLSD database. The bulk density measurement for each NLSD sample was derived by using the Weight in Air/Weight in Water (Archimedes) method. The oxidized core samples were sealed with molten wax and re-weighed to determine the weight of the paraffin coating, prior to weighing in water. The bulk density determinations were done on short (5 cm), half core specimens, taken at close intervals. The NLSD bulk density data were collected from one core hole in the Piment South area and from 13 core holes from the Piment Central area.

A total of 131 bulk density measurements were carried out on lengths of complete drill core by Defiance during their programs. Density determinations were undertaken prior to core sawing on 131 samples of about 8 cm to 15 cm in length and of both HQ and HQ3 diameter. The water displacement method was used.

From 2008 to December 2011, Red Back and TMLSA completed 24,702 specific gravity determinations of bulk density using the Archimedes method. The samples were selected to provide a representative suite of densities covering all major lithology types and from all oxidation levels.

Initial Red Back and TMLSA density determinations were done using wax-coated samples for both oxide and primary material. This procedure was changed by using uncoated core samples for only primary material to speed up the test work. Duplicate tests with one-wax coated samples for every lithology per hole were done to evaluate bias between the data pairs. About 650 duplicate tests were done up to December 2011. Initial analysis of the check samples showed very good correlation between the uncoated and coated density values. A total of 90% of the dataset shows a difference of 1% variability between the pairs samples (coated and uncoated). For the 2013 model, 26,940 density records were used to develop the new density values. The 2013 model used the same methodology of combining the rock type (or lithology) and state of oxidation to determine the appropriate density value. These values were assigned to the block model, based on these two values. If no data were available, the 2011 model density values were still used in the 2013 model.

Using specific gravity measurements taken on the various rock types, an average density value was calculated for each logged rock type and oxide type. These values were then assigned to the domains in the block model using simple manipulation scripts.

11.2 Analytical Laboratories

Sample preparation was undertaken on site by NLSD staff during their drill programs. Analytical laboratories used were the BRGM laboratory in Orleans, France and the OMAC laboratory in Ireland. QA/QC was undertaken by Genalysis Laboratories in Perth, Australia, and SGS Laboratories in France. Laboratory accreditations at the time are not known; all analytical laboratories were independent of NLSD.

During Defiance's RC and diamond drill programs, the analytical work was carried out by SGS Analabs in Kayes, Mali and by Abilabs located in Bamako, Mali. Analabs is an ISO accredited laboratory whereas Abilab is not ISO accredited. The laboratories were independent of Defiance.

Following Red Back's acquisition of the Tasiast deposit in August 2007, an on-site SGS Analabs assay facility became operational. Prior to that time, samples had been prepared on site by Red Back staff under supervision of senior geological staff. All drill samples since 2007 have been prepared and analysed under contract by SGS on site and by SGS Analabs in Kayes, Mali, SGS Analabs in Morilla, Mali, and SGS in Ouagadougou, Burkina Faso. Laboratories were independent of Red Back. The two SGS laboratories hold ISO9000 accreditations.

In December 2010, SGS constructed and commissioned a mobile sample preparation facility in Nouakchott, Mauritania, and selected samples were submitted to the facility for preparation. In late 2011, a new on-site SGS preparation and assay laboratory was commissioned at Tasiast, with a capacity of up to 2,000 samples per day. In mid-2012, TMLSA stopped sending exploration samples to the SGS Tasiast laboratory due to quality control concerns. Due to the large volumes of the samples and laboratory delays, TMLSA started sending samples to nine laboratories outside the country to different accredited laboratories. In April 2013, ALS Chemex took over the Tasiast laboratory facilities and initiated carrying on the sample preparation and analytical services. The Tasiast laboratory facilities were previously operated by SGS.

11.3 Sample Preparation

Midas, Defiance and Rio Narcea RC drill sample preparation involved the entire RC calico sample bag, which was oven-dried for 24 hours and then weighed before pulverizing the entire 2 kg to 3 kg subsample using a Labtecnics LM5 mill. Each core sample was crushed to -10 mm in a jaw crusher, and the entire sample was pulverized to P90 (90% passing) at 75 µm using a Labtecnics LM5 mill. Barren dune sand was used to clean the bowls after every sample. The pulverized material was sampled using a spatula, and two 120 g pulp sub-splits were taken; one packet was prepared for shipment to the assay laboratory and one packet remained on site for future reference. Blanks of dune sand and certified reference standard were then inserted with the field samples.

Sample pulp shipments were conducted on a weekly basis. The samples were transported in secured wood boxes to Nouakchott, where Mauritanian Custom inspected the shipment and released the proper documentation for exportation. The boxes included a sample submission sheet prepared by the laboratory manager. Samples were then shipped by airfreight to SGS Analabs.

At SGS Tasiast and SGS Nouakchott, the entire RC and core sample was oven-dried for 24 hours in a cleaned metal dish, weighed and then crushed to 75% passing at 2 mm. At SGS Tasiast, a 1.5 kg subsample was split using a Jones riffle splitter, and pulverized in a Labtech Essa LM2 ring pulverizer using a 2 kg bowl to 85% passing at 75 µm. At SGS Nouakchott, the sample was split once using a Jones riffle splitter and pulverized in a Labtech Essa LM2 ring pulverizer using a 2 kg bowl to 85% passing at 75 µm. Both laboratories took a 200 g subsample for gold (Au) fire assay.

At the SGS Tasiast laboratory and relocated mobile sample preparation facility, the procedure for sample analysis remains unchanged. However, subsample size at the Tasiast laboratory has been increased to 2 kg, to improve the precision of results.

For RC and core samples processed by SGS Analabs in Kayes, samples were stockpiled in a secure area within the Tasiast core facility, and collected by a truck contracted by either Analabs or TMLSA for shipment to Kayes. The samples were enclosed and secured in a large tarpaulin and transported directly from the site to the laboratory. The entire core or RC sample was oven-dried for 24 hours and then weighed before pulverization. Samples were crushed to 75% passing 2 mm, and two 1.5 kg subsamples were split using a Jones riffle splitter and pulverized in a Labtech

Essa LM2 ring pulverized using a 2 kg bowl to 85% passing at 75 µm. These two pulps were recombined before being subsampled (200 g) for an Au fire assay.

After ALS Chemex took over the laboratory facilities at Tasiast in early 2013, a few changes were introduced at the sampling preparation stage, including:

- Drying the entire RC or core samples for three to four hours at 105 ºC.
- Registering the dry weight.
- Crushing the entire samples 80% passing 2 mm. In November 2013, it was decided to increase the passing to 85% passing 2 mm. Every 20 samples must generate a preparation duplicate.
- Splitting the samples using a Jones riffle splitter to obtain 1 kg, and pulverizing the samples to 85% passing 75 µm.
- Taking 200 g for analysis. Every 20 samples must generate a pulp duplicate.

11.4 Sample Analysis

For the samples processed by BRGM on behalf of NLSD, the following methods were used:

- Roasting (77 Phase 1 samples and all Phase 2 samples)
- Total attack (hydrofluoric acid and aqua regia);
- Atomic absorption (AA) analysis, detection limit: 20 ppb Au (Phase 1) and 100 ppb Au (Phase 2).

SRK, in 2003, noted that check analysis of 74 Phase 1 samples showed no significant variations between roasted and non-roasted samples.

OMAC used the following methods on samples processed for NLSD:

- Ignition / Aqua Regia Digest / MIBK Extraction / AA on 30 g sample; detection limit: 10 ppb Au; 10% repeats.
- Fire assay (30 g sample): re-analysis of 903 mineralized samples (Phases 1 & 2) + all samples >1 g/t Au and those <1 g/t Au which were included in mineralized intersections (Phases 3 & 4); detection limit: 10 ppb Au.

All of the sample pulps from the Midas, Defiance and Rio Narcea drill programs were analysed for gold using a 50 g fire assay with an atomic absorption spectroscopy (AAS) finish at both laboratories. The Analabs 50 g fire assay/AAS method (FA50) has a lower detection limit of 0.005 g/t Au; Abilabs' lower detection limit is 0.010 g/t Au.

Analabs routinely ran random check assays in all batches. However, when the laboratory was notified of possible samples containing high values of gold for the core samples, Analabs carried out a fire assay/AAS method, with repeats in some case, as well as fire assay/gravimetric analysis for samples grading greater than 5.00 g/t Au. Analabs also provided Defiance with its internal QA/QC data during the analysis period.

For Red Back and TMLSA samples, sample pulps were analysed for gold using a 50 g fire assay with an AAS finish with a detection limit of 0.01 g/t Au. Results higher than 5 g/t Au were re-analyzed by fire assay technique and gravimetric finish. In 2012, TMLSA began gravimetric finishes for gold above 5 g/t, and also began screened metallic fire assays.

11.5 Quality Assurance and Quality Control

Most of the documented QA/QC cited by SRK in 2003 on NLSD samples related to measurements of the analytical errors through pulp duplicates, where two analytical methods (aqua regia: AAS and Fire assaying: FA) are compared. No significant problem was detected. In early 2003 a total of 429 pulp samples, collected by staff from TMLSA and representing close to 10% of the mineralized samples within the wireframed resources, along with 54 standards (of values 0.5, 1.66 and 3.22 g/t Au) and 18 blanks were re-assayed by Genalysis. SRK noted that the Genalysis results compared well with the database and standards and blanks were assayed within acceptable limits.

For the Defiance and Rio Narcea drill programs, a total of 21,686 RC sample pulps, including field duplicates, blanks and standards, and 904 diamond drill hole core sample pulps, including field duplicates, blanks and standards, were shipped in 16 batches, of which 14 went to Analabs and two went to Abilabs. Included within these sample batches were a total of 774 field duplicate samples, each one being a second split from a 1 m interval field sample bag, and 1,136 preparation duplicates, each one being a second split from the pulverized RC and core sample at the preparation laboratory.

The analytical QA/QC program implemented by Defiance was monitored by the routine submission of commercial SRMs purchased from Gannet Holdings Pty. Ltd. of South Perth, Western Australia. SRMs were inserted at every 20th sample and an internally prepared coarse blank sand inserted at every 10th sample within the RC and core sample stream. Field duplicates were collected by the field geologist after the completion of each RC hole and the number of field duplicates on a per RC hole basis was dependent on the length of the hole or equivalent to every 20th sample. Preparation duplicates were selected for every 20th sample number in a sequence and submitted as a separate sample number series on a per batch basis.

TMLSA commenced a systematic process for analysis of ‘Prep Duplicates’ and ‘Pulp Duplicates’ from mid-2011. The duplicates are collected every 20th sample within intervals of visual mineralization and resubmitted blind to the laboratory. In addition, other initiatives TMLSA implemented for QA/QC measures in August 2011 included using the standard deviation of the mean value for the standard reference material (previously ±10% of mean value), rules for batch pass and failures, and a log of all errors.

In 2011, TMLSA engaged an independent consultant to provide a regular review of the QA/QC data. Issues identified during the early review in September 2011 (Heberlein, 2011), such as switched standards and standard identification, have been corrected and control actions implemented. Other issues, such as analytical repeatability over time, are still being addressed, and since then control actions were implemented to reduce the number of errors.

Additional actions implemented in 2012 and 2013 to address other recognized issues, such as duplicate precision, include the following:

1. Conducted routine crushing and analyzed pulverized duplicate samples at the majority of the laboratories.

2. Conducted gradual replacement of three tier-riffle splitters and cone splitters on RC rigs by 50/50 manual riffle splitters.

3. Established a dedicated group to control and monitor sampling, dispatch and quality control analytical results.

4. Assigned a Tasiast technician permanently to the on-site Tasiast exploration preparation facility to monitor and control the workflow.

5. Conducted a routine independent-consultant review of data and laboratory audits.

6. Reviewed various sample volume and preparation methods that have resulted in larger samples (between 5 kg and 10 kg) collected from RC rigs in 2013.

SRK 2013 QAQC Review

In April 2013, SRK conducted a review of the analytical quality control procedures and results for the Tasiast gold project in Mauritania. The objective of the review was to provide an independent analysis of the sampling procedures and a review of the analytical quality control results for the data to be used in the current resource estimate.

SRK visited the Project site from October 11 to 15, 2013. SRK also visited a third-party preparation laboratory operated by SGS Minerals in Nouakchott, Mauritania. The purpose of the site visit was to audit project technical data and collect all relevant information for the compilation of the Sample Preparation, Analyses, and Security and Data Verification sections of a technical report. SRK was given full access to relevant data and conducted interviews with Kinross personnel to obtain information on past exploration work and understand the procedures used to collect, record and analyze historical and current exploration data.

SRK reviewed the field procedures and analytical quality control measures used in the Tasiast gold project. In SRK's opinion, Kinross personnel used care in collecting and managing field and assay exploration data. The sample preparation, security and analytical procedures used by Kinross are consistent with generally accepted industry best practices, and are therefore adequate to support the mineral resource estimation.

A summary of SRK's main conclusions are:

- The sampling procedures used meet industry best practices. All borehole sampling was conducted by appropriately qualified personnel under the direct supervision of appropriately qualified geologists. The sample preparation, security, and analytical procedures were consistent with generally accepted industry best practices and are therefore adequate to support Mineral Resource estimation.

- The paired data results are consistent with results expected from gold mineralization in an epigenetic deposit that is structurally and lithologically controlled.

- Results from the SRMs are acceptable.

- The non-certified field blanks consistently returned values at or below the detection limit at most primary laboratories.

- SRK concurred with Heberlein (2013) that sample preparation procedures are failing to properly homogenize the samples. SRK attributes part of that failure to a prominent nugget effect and notes that control charts display no apparent bias between original and duplicate samples.

- Much of the analytical data informing the Mineral Resource estimate was derived from several different unaccredited laboratories, including the mine laboratory operated by SGS.

11.6 Sample Storage

Sample pulps are returned from the laboratory in plastic vials or sealed paper envelopes, and these are stored in sealed containers at site. The majority of historic coarse reject samples were not stored, but TMLSA has commenced storing selected mineralized coarse reject material. The remaining half of the drill core is well stored in stacked wooden trays referenced by hole identification number and interval length. Some core intervals have been totally sampled for metallurgical or check (umpire) sampling.

11.7 Sample Security

Following TMLSA's acquisition of the Project in September 2010, all drill samples collected are under direct supervision of Project staff from the drill rig and remained within the custody of staff up to the moment the samples were delivered to laboratory or placed on contracted trucks for delivery to the Mali laboratory. Samples, including duplicates, blanks and certified reference materials are delivered daily from the drill rig to a secure storage area within the fenced Tasiast core facility.

Chain of custody procedures consist of filling out sample submittal forms that are sent to the laboratory with sample shipments to make certain that all samples are received by the laboratory.

11.8 Comment on Sample Collection, Preparation, Analysis and Security

In the opinion of the QP, the sampling methods are acceptable, meet industry-standard practice, and are adequate for mineral resource and mineral reserve estimation and mine planning purposes, based on the following:

- Data are collected following industry standard sampling protocol

- Sampling has been performed in accordance with industry standard practices

- Sample intervals in RC and core drilling comprise a maximum of 1.5 m for core and 1 m to 2 m for RC. Samples are broken at lithological and mineralization changes in the core. Sample intervals are typical of sample intervals used for gold mineralization in the industry, and are considered to be adequately representative of the true thicknesses of mineralization. Not all drill material was sampled in early drill programs

- The specific gravity determination procedure is consistent with industry-standard procedures
- There are sufficient specific gravity determinations to support the specific gravity values utilized in waste and mineralization tonnage interpolations
- Geochemical sampling covered sufficient area and was adequately spaced to generate first-order geochemical anomalies, and thus is representative of first-pass exploration sampling
- Drill sampling has been adequately spaced to first define, then infill, gold anomalies to produce prospect-scale and deposit-scale drill data. In general, the drill collar spacing ranges from 25 m x 25 m to 70 m x 70 m
- Sample preparation for RC and core samples has followed similar preparation procedures. These procedures are in line with industry-standard methods, and suitable for the greenstone- and BIF-hosted coarse-gold deposit style
- The QA/QC program comprised insertion of blank, duplicate and SRM samples. The QA/QC program results do not indicate any problems with the analytical programs, therefore the gold analyses from the RC and core drilling are suitable for inclusion in mineral resource and mineral reserve estimation
- Sample security has relied upon the fact that the samples were always attended or locked in appropriate sample storage areas prior to dispatch to the sample preparation facility. Chain of custody procedures consist of filling out sample submittal forms that are sent to the laboratory with sample shipments to make certain that all samples are received by the laboratory
- No factors were identified with the drill programs that could affect mineral resource or mineral reserve estimation
- Current sample storage procedures and storage areas are consistent with industry standards.

12. DATA VERIFICATION

A number of verification checks have been performed on data collected from the Project, either in support of technical reports, or as part of the Project feasibility study.

12.1 Verification in Support of Technical Reports

A number of external consultants and consultancies have reviewed Project data, and made recommendations for future work.

SRK 2003

SRK reviewed the data available in 2003, as part of supporting documentation for the acquisition of Midas by Geomaque, and commented:

- *Although SRK was not involved in the selection and collection of the check samples, the documentation supplied by [Midas] which includes all the analytical results show that the general quality of the sampling/assaying is acceptable and to industry standards;*
- *Although the density data originates from a relatively small number of drill holes, there are indications that a single tonnage factor for oxide and for primary, as used in the resource estimates, is a simplification of the true variation with depth.*

ACA Howe 2003 and 2007

Howe inspected Defiance's sample preparation facility, and considered the facility to be reasonably well equipped and maintained, in accordance with acceptable industry standards.

Howe reported that on the recommendations of RSG Global, Midas collected a total of 429 pulp samples of known NLSD drilled mineralized zones in early 2003. Midas inserted blanks and standards and submitted this sample batch to Genalysis. The Genalysis results compared well with the NLSD assays and the standards and blanks inserted by Midas assayed within acceptable limits.

A comparison of RC and core duplicate samples indicated no major bias across all grade ranges. Howe considered that the degree of scatter shown in graphed data was acceptable for resource estimation purposes. No bias occurred towards the higher-grade original or repeat assays.

A total of 134 one metre interval RC samples from six of Defiance's RC drill holes and 27 core pulp samples from two Defiance core drill holes were submitted by Howe to ALS Chemex Laboratories in Mississauga, Ontario for check analysis. Upon review of the results, Howe was of the opinion that its independent check assay results confirmed the presence of gold mineralization at Tasiast.

SNC-Lavalin 2004

Defiance selected mineralized intersections from 30 RC holes covering the four mineralized areas of the Piment Zone, which were sent to Canada for metallurgical test work. SNC-Lavalin (SNC) reviewed the drill hole information on the geological sections prepared by Howe and combined the sampled intersections of several drill holes to obtain nine samples considered to be more or less representative for the various mineralized zones and their high and low gold grades. These samples were sent to SGS Lakefield in Ontario, Canada; an ISO/IEC 17025 accredited laboratory for assay. The comparison of the assay results of the initial samples and those from Lakefield was acceptable and showed a reasonable correlation.

SNC representatives collected eight samples of RC drilling chips that had previously been assayed by Analabs. These samples were sent to Lakefield for assay. Results showed that gold was present in the indicated mineralized zones even though the correlation was rather erratic due to the statistically low number of samples.

Red Back 2008–2010

Red Back conducted an analysis of the available, historical QA/QC data from Defiance and Rio Narcea as part of the February 2008 resource update comparing all historical data with data generated by Red Back as at February 2008.

The following conclusions were noted (Stuart, 2008):

- Globally, all of the laboratories used to compile the Tasiast resource have reported the ore grade standards well. On average, 85% of the six +1.5 g/t Au internationally-accredited standard reference materials (SRMs) submitted reported to within an accuracy of +/-10%. The historical pre-Red Back database reported 86%. The Red Back database reported 84%
- A minor negative bias was repeated in each of the standards tested at each of the laboratories. The negative direction of the standards bias, however, results in a degree of conservatism in the assays reported
- The <1.0 g/t Au standards performed less well with a range of 67% to 75% of the standards submitted reporting to within +/-10%. The poorer precision and accuracy of the QA/QC data below 1.0 g/t Au is evident across the whole of the resource timeframe
- Both routine Red Back and pre-Red Back blank submissions performed well, exhibiting only a minor low level <50 ppb Au cross-contamination. Evidence

suggests a component of poor blank selection may have contributed to the Red Back higher bias and was noted for further attention

- The total operational precision (TOP) achieved by Red Back, demonstrated by a percent Median Absolute Half Difference (%MAHD) = ±14% of resource grade assays >0.2 g/t Au, is generally within acceptable limits of a coarse gold deposit such Tasiast. The coarse gold nature of the deposit is apparent within the range of errors expressed by the 90th percentile Absolute Half Difference (AHD) = ± 58%
- While analysing the historical pre-Red Back duplicate data it was observed that the historical resource data reported similar “nuggetty” duplicate assaying, closely comparative to the RBK data with %MAHD = ±14% and a P90 AHD = ± 50% >0.2 g/t Au. The datasets have equivalent coarse gold features and equivalent assaying precision
- The imprecision consequent of coarse gold is evident across the entire Tasiast grade profile from 0.2 g/t Au to 10 g/t Au. Clustering of “nuggetty assaying” is often observed in mesothermal greenschist facies, epigenetic, structurally-controlled deposits at the high-grade end of the profile due to the coarse gold being hosted dominantly in the quartz vein materials, the gold being finer, and closer to sulphide lattice within the disseminated selvages
- Red Back considered the extant of the assay data included in the Tasiast resource data to be accurate and precise to within the inherent, natural coarse grade variation observed in the grade profile of this structurally controlled, sub-amphibolite, BIF-hosted style of gold mineralization.

Review of the blank, duplicate and SRM submissions in 2009 and 2010 indicated no significant errors or biases in the analytical data. Prior to late 2009 the majority of the field duplicate analyses completed were from non-Greenschist mineralisation styles, e.g., Piment iron-formation and West Branch footwall. A total of 16,907 (2009) and 15,929 (2010) QA/QC samples were blindly inserted as part of the routine sample preparation and were submitted for analysis. Red Back concluded that the QA/QC data reported was of industry accepted standards and the assay data was considered reliable for inclusion in the December 2008, 2009 and 2010 resource estimations.

12.2 TMLSA QA/QC

The data presented below covers all QA/QC data for 2011. In 2011 a total of 22,525 QA/QC samples including standards (STD), blank (BLK) and field duplicates (FDups) were submitted routinely and blindly to the labs with resource related submissions. TMLSA maintained 10% total QC through the sample stream.

Because of the volume of samples, 18 different laboratories were used for the sample analysis. Table 12-1 presents the summary statistics of sample types submitted to the various laboratories for analysis.

Table 12-1: QA/QC Samples by Laboratory

Laboratory	Standard	Blank	RC Duplicate	Core Duplicate
Actlabs Burkina Faso	199	209	222	0
ALS Chemex Ouaga	447	454	189	262
ALS Johannesburg	599	621	403	251
ALS Kumasi	326	330	288	63
ALS Loughrea	108	86	82	42
ALS Nouakchott	4	4	0	5
ALS Romania	527	512	303	280
ALS Tasiast	94	90	4	86
ALS Vancouver	226	252	231	15
SGS Kayes	13,498	14,583	12,706	1,162
SGS Kayes Lab	68	77	14	160
SGS Lakefield	9	9		18
SGS Morila	6,180	6,583	5,968	364
SGS Ouaga	1,420	1,484	1,213	223
SGS Ouaga Lab	81	80		80
SGS TM Lab	386	400	54	338
SGS TML	4,714	5,215	4,416	809
Unknown	812	1,905	0	0
Total	29,698	32,894	26,093	4,158

Table 12-2 presents the data used for the 2011 QA/QC analysis after results that are likely to cause bias in the analysis have been removed. Assay results that were excluded in the QA/QC analysis are as follows:

- IS - insufficient samples with assigned values of -1 in the assay results.
- LNS - listed not received samples with assigned values of -2 in the assay results.
- Samples with gold values below the resource threshold of 0.1 g/t Au were excluded in the Half Absolute Relative Difference (HARD) analysis for the

duplicate data pairs because it does not have much relevance to the potential limit of economic mineralization.

- Assay pairs involving standards and duplicates with unrelated results were considered as results with gross errors. The threshold for assays with gross errors was fixed at HARD value of ±75%. Assay pairs (standards and duplicates) with values greater than this threshold were excluded from the QA/QC analysis.

Table 12-2: 2011 QA/QC Samples Analysed by Laboratory

Laboratory	Standard	Blank	RC Duplicate	Core Duplicate
ALS JHB	290	300	50	42
ALS Ouaga	308	316	15	25
SGS Kayes	3,570	3,638	552	159
SGS Morila	1,758	1,828	289	83
SGS Ouaga	570	598	25	73
SGS TML	1,075	1,101	113	100
Total	7,571	7,781	1,044	482

Each standard is rated on how accurately the laboratory has reported its value. Results of the standards are assessed against the inner (±2 x standard deviation) and outer limits (±3 x standard deviation). The blanks are also expected not to report above 0.05 g/t Au. Standards and blanks that failed accuracy controls were requested to be re-analyzed where a material impact on the reported results was considered. A precision analysis is conducted on the duplicates. Precision is defined as the level of error obtained at the 90th percentile in order to reproduce a sample or data.

An average accuracy of 93% was achieved for the internationally accredited SRM standards (see Table 12-3). This implies that 93% of the total SRM samples submitted to the various laboratories reported within acceptable limit of ±3 standard deviation. Routine field blank samples submitted to the various laboratories reported about 99% of the samples below 0.05 g/t Au.

Table 12-3: 2011 Resource QA/QC Results

Laboratory	Total QA/QC (Standards)	STD within 1 ∂	STD within 2 ∂	STD within 3 ∂	BLANK < 50 ppb
Actlabs Burkina Faso	199	91%	93%	94%	99%
ALS Chemex Ouaga	447	75%	93%	97%	99%
ALS Johannesburg	599	67%	88%	96%	99%
ALS Kumasi	326	70%	93%	96%	99%
ALS Loughrea	108	74%	90%	98%	99%
ALS Nouakchott	4	90%	100%	100%	100%
ALS Romania	527	90%	100%	100%	100%
ALS Tasiast	94	84%	91%	100%	100%
ALS Vancouver	226	81%	96%	98%	99%
SGS Kayes	13,498	78%	94%	98%	100%
SGS Kayes Lab	68	79%	97%	100%	100%
SGS LAKEFIELD	9	100%	100%	100%	100%
SGS Morila	6,180	81%	96%	99%	99%
SGS Ouaga	1,420	78%	12%	8%	98%
SGS Ouaga Lab	81	80%	99%	100%	99%
SGS TM Lab	386	78%	95%	99%	99%
SGS TML	4,714	65%	86%	93%	95%
Unknown	812	55%	83%	93%	99%
Total	**29,698**	**76%**	**89%**	**93%**	**99%**

The few deviations in the standards were mainly due to failed standards and swapping of identification numbers. Deviations in the blanks are related with low level cross contamination in the laboratories and sample switches. Example charts showing deviations in standards (Standard G305-2) and blanks plotting as spikes are presented in Figures 12-1 and 12-2.

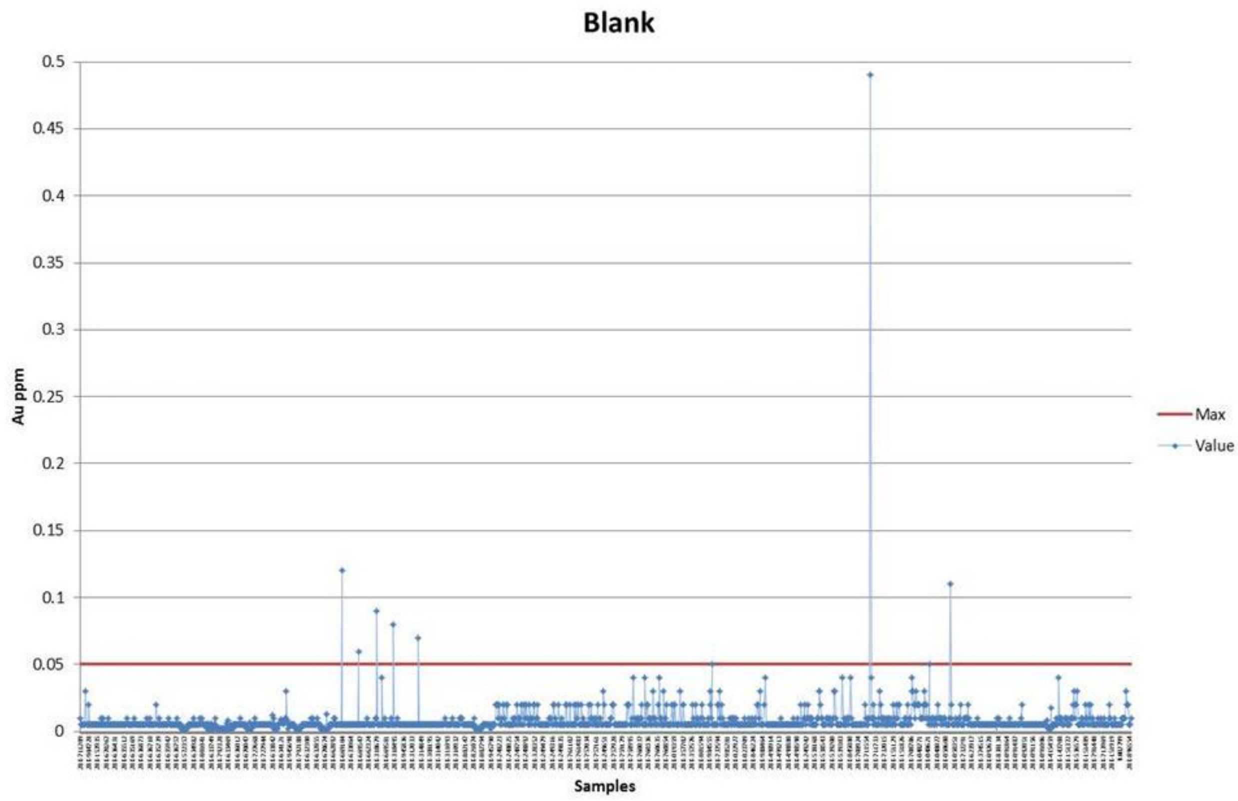
The TOP of the duplicates as expressed by a 50th %MAHD is between ±13.5% and ±17.6% for RC duplicates and between ±21.0% and ±31.6% for core duplicates for resource grade assays >0.1 g/t Au and is somewhat within acceptable industry limits for coarse gold deposit. Values for field duplicate core samples are on the high side and this is possibly a reflection of the decrease in sample homogeneity in moving from RC to core sample.

Figure 12-1: Standard Control Plot for Standard G305-2 Results from SGS Kayes

According to Heberlein (2013), measurable improvement was observed in the QA/QC procedures since his involvement in the Project.

The improvement has resulted in a measurable increase in the overall precision of the analytical results. Early precision estimates of field duplicate results (containing the total error of sampling, preparation and analysis) showed unacceptably high values for both core and RC duplicate samples. The initial analysis of duplicate results in 2011 determined precisions in the 80% to 90% range for core and 30% to 40% range for RC. Improvements to sampling and sample preparation procedures, particularly at the on-site laboratory (TML) have brought the drill core precision down to the 45% to 50 % range, which is reasonable for the nugget style of mineralization at Tasiast.

Figure 12-2: Blank control plot for Blank results from SGS Kayes.



Database

Red Back and TMLSA import and store drill related data in a Century database management system (CDMS) under the supervision of an experienced on-site database manager. All drill data has been imported into this system and has been re-validated. The drill data for resource estimation purposes was exported as comma delimited ASCII files. The CDMS was also used to generate monthly, quarterly and yearly QA/QC reports.

TMLSA carried out a 5% quality control analysis of the imported assay values in the resource database versus those from the certificate. The analysis included 53,469 samples from 235 holes and no major issues were identified. Eleven samples in the analysis had database assays values that did not correspond to the certificates and were from failed assay batches. Proper labelling of failed batches and re-analyzed batches has been recommended as part of ongoing quality control checks.

From May to July 2013, TMLSA and Kinross Technical Services (KTS) conducted an extensive data validation using a set procedure with guidance from CAE Mining. Geological data collected by the exploration team at Tasiast, including drilling and surface data, is stored in a Relational Database System called Fusion. CAE Mining is the company that develops, commercializes and maintains the software. The system was implemented in Tasiast by the previous owner of the property, Red Back, and Kinross migrated to a different system structure, but maintained the integrity of the existing data when the site was purchased in 2010.

The final database was handed over to KTS on July 17, 2013, and provided to Dominic Chartier from SRK for further verification on July 26, 2013. Additional checks and comparisons were completed to ensure the integrity of the export files (i.e., building and rebuilding queries, and exporting the same file twice to ensure all results carried over). No significant discrepancies were observed.

Data Validation

Data validation ensures that accurate data are used for preparation of the resource estimate. An initial analysis by KTS of the data extracted revealed that the majority of the data needed extra validation and verification. The major problem found was in the QA/QC status of many historical holes that were showing a “failed” status.

A total of 1,363 batches had a failed status. Those batches were reviewed by the Tasiast Exploration managers. The batches were grouped by laboratory, and their QA/QC charts were printed or regenerated to aid in the analysis done by the QP. From all the batches reviewed, the status of 58.2% of the batches could be changed from “failed” to “QP-Accepted”. In the majority of the cases, the failures were due to the definition of the standards in the database. By default, a batch would fail if it had at least one control sample with a result that was more than two standard deviations; however, Kinross uses three standard deviations instead of two for a single sample. Furthermore, 38.4% of the batches reviewed needed more review because the QA/QC team might have mistakenly switched the definition of the samples (i.e., a sample was defined as a standard when it should have been defined as a blank, or similar issues).

Other validation performed on the data at Tasiast included:

1. Collar coordinates, by comparing the data in the database with the data obtained by the surveyor.

2. Downhole survey, by plotting the holes in 3D software and checking for any anomalies in the deviations of the holes. Many different instruments for measuring hole deviation in downhole surveying were used at Tasiast. So a ranking of instrument was implemented, giving priority to the instrument with better precision, when multiple instruments were used to survey the same hole. Instruments giving the more accurate reading preceded all other surveys.

3. Lithology, alteration, mineralization, by comparing the data in the database with the data captured in the logs and correcting a few cases of gaps and/or overlaps.

4. Assays, by comparing 5% of the results in the database to the results in the original certificates provided by the laboratories.

5. The validation process was completed by importing the data into Micromine (mining software) to visually check the validity of the data and to generate a report.

Twin Holes

In 2010 and 2011 TMLSA also twinned three RC holes from the Greenschist zone. In 2010 two twin holes were completed on the lower portion of the Greenschist zone to test for due diligence and variability between RC and core drilling. Results from the work returned a strong to acceptable correlation between mineralized intervals in the RC and core holes. The purpose of the 2011 hole that drilled shallow in the Greenschist zone is to improve understanding of any potential sampling differences between RC and core material. Geological logs between the core and RC drilled in the shallow portion of the zone retuned similar data with lithological units within expectations.

12.3 Comment on Data Verification

The process of data verification for the Project has been performed by TMLSA, Red Back, and personnel of precursor companies, and external consultancies contracted by those companies.

The QP has reviewed the reports and is of the opinion that the data verification programs undertaken on the data collected from the Project adequately support the geological interpretations, the analytical and database quality, and therefore support the use of the data in mineral resource and mineral reserve estimation.

Data used to support mineral resource and mineral reserve estimates have been subjected to validation, using built-in software program triggers that automatically check data for a range of data entry errors. Verification checks on surveys, collar coordinates, lithology, and assay data have also been conducted. The checks are appropriate, and consistent with industry standards.

Ongoing sample preparation and analytical work is recommended to obtain more acceptable precision from the duplicate samples.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Mineralogy

The Tasiast mineralization is free-milling and amenable to gold extraction by simple cyanide leaching. The existing mill has been operating since 2008 initially treating oxide BIF hosted ore yielding a typical gold recovery of 93%. Gold recovery from fresh ore, which forms an increasing portion of the mill feed since 2010, varies between 91% and 93%. A proportion of the gold is coarse and responds well to gravity concentration. Gold mineralization is associated with structurally controlled faults and shears, quartz-veining and silica-flooding. Gold grains observed in the exploration core holes are seen in isolated grains in quartz veins and are closely associated with pyrrhotite. The mineralization has relatively low levels of sulphides approximately 1% to 5% S, predominantly represented by pyrrhotite and to lesser extents pyrite, arsenopyrite, and chalcopyrite. Other metal contents are low such as silver approximately 1 ppm to 2 ppm, copper approximately 100 ppm, arsenic approximately 10 ppm and very low levels of mercury, less than 0.3 ppm Hg.

13.2 Metallurgical Test Work

Bulk of the metallurgical test work has been done to evaluate the optimum process for the West Branch ore which is expected to be treated at a higher throughput. Major metallurgical sampling campaigns were conducted on the West branch mineralized zone and test work to optimize cyanide addition rate and grinding tests were completed.

Evaluation of West Branch Ore Processing

Four major metallurgical sampling campaigns were conducted on the West Branch mineralized zone as follows:

- Campaign 1: AMMTEC Pty. Ltd. (AMMTEC)
- Campaign 2: SGS Canada Inc. (SGS)
- Campaign 3: SGS, additional boreholes to test for variability
- Campaign 4: KHD Humboldt Wedag (KDH) and JKTech Pty. Ltd. (JKTech), high pressure grinding roll (HPGR) and comminution tests

A program of waste rock sampling and characterization was also undertaken with core samples selected to represent all rock lithologies and depths.

Test Work Program

Test work was conducted by multiple laboratories and the results from the different laboratories were comparable.

The information generated in the test work programs was sufficient for:

- Preparation of ore characterization
- Process selection
- Facility description
- Production scheduling
- Expenditure estimates

Summary of Comminution Characteristics

The comminution characteristics of West Branch samples Work was carried out by Ammtec, SGS and JKTech to:

- Assess the variation in comminution parameters for the deeper deposits.
- Confirm grinding energy requirements for the deeper ore.

All of the samples were checked for their correct lithologies and split into separate lithology tables for analysis. The average comminution parameters obtained from laboratory testwork are in Table 13-1.

Table 13-1: Comminution Characteristics

	Year	2018	2019	2020	2021	2022	2023	2024	2025	2026 to End	LOM
West Branch + Stockpiles	WB-fBIMFrW	0.98	1.86	2.07	5.49	0.21	5.43	1.32	0.23	6.58	3.23
	WB-FVC FrW	8.92	15	11.8	85.9	10.8	18.4	9.47	4.26	21.3	21.8
	WB-GST FrW	66.5	44.4	47.6	1.92	22.8	37.2	80.3	93.1	29.7	43.4
	WB-hBIMFrW	0.28	9.72	10.6	1.11	3.6	6.38	1.69	0.27	7.84	4.67
	WB-MDO FrW	-	-	-	-	-	-	-	-	-	-
	WB-SVC FrW	16.7	28.5	22.7	5.61	10.3	28.3	6.87	1.19	34.4	19
Piment + Stockpiles	PM-fBIMFrP	-	-	-	-	-	1.93	0.16	0.17	0.07	0.19
	PM-fBIMOxP	-	-	-	-	-	1.22	-	-	-	0.09
	PM-FVC FrP	1	0.03	0	-	1.95	0.26	0.01	0.08	0.03	0.34
	PM-GST FrP	2.45	0.02	0	-	10.6	-	-	-	-	1.21
	PM-hBIMFrP	3.04	0.47	4.98	-	39.2	0.78	0.17	0.74	0.11	5.99
	PM-MDO FrP	-	-	-	-	-	-	-	-	-	-
	PM-MDO OxP	-	-	-	-	-	-	-	-	-	-
	PM-SVC FrP	0.08	0.02	0.23	-	0.63	-	-	-	-	0.09
Existing Stockpiles		-	-	-	-	-	-	-	-	-	-
Total Milled		100	100	100	100	100	100	100	100	100	100

Metallurgical Testing

<u>Metallurgical Test Work Parameters</u>

Extensive metallurgical testing was completed on West Branch samples, twinned hole samples and deeper level variability samples. In general, test work indicated that the ore was amenable to gravity recovery and cyanide leaching, resulting in selection of a similar flow sheet to the existing plant. Some of the key parameters that resulted from the test work are:

- Grind size: 90 µm
- Gravity recovery: 30% to 50%
- Leach retention: 24 hours
- Ore is not preg-robbing
- Leach environment: seawater
- Cyanide consumption: 0.47 kg/t

Cyanide Concentrations and Grinding Test Work

The cyanide addition rate has been optimized to a low addition rate. Test work results indicate that cyanide concentrations as low as 0.5 g/L, corresponding to a cyanide addition rate of 0.6 kg/t, still provide adequate overall gold extraction after 24 hours, as shown in Figure 13-1.

The grinding test work results show that gold extraction increases with a finer grind size. Gold dissolution kinetics were enhanced at the finer 80% passing (P_{80}) grind sizes of 90 µm and 75 µm. At the selected grind of 90 µm, test work indicates that some dissolution still takes place after 24 hours of leach. However, the plant processing circuit will include grinding in process water containing cyanide recycled from the tailings thickener. Also, in the operating plant the coarser gold reports to the cyclone underflow and will be recovered by gravity, allowing smaller particles to overflow. Since this cannot be accurately simulated in a laboratory, 24 hours is concluded to be a suitable leach time. In the operating plant, the balance between finer grind (power cost) and gold recovery will be optimized.

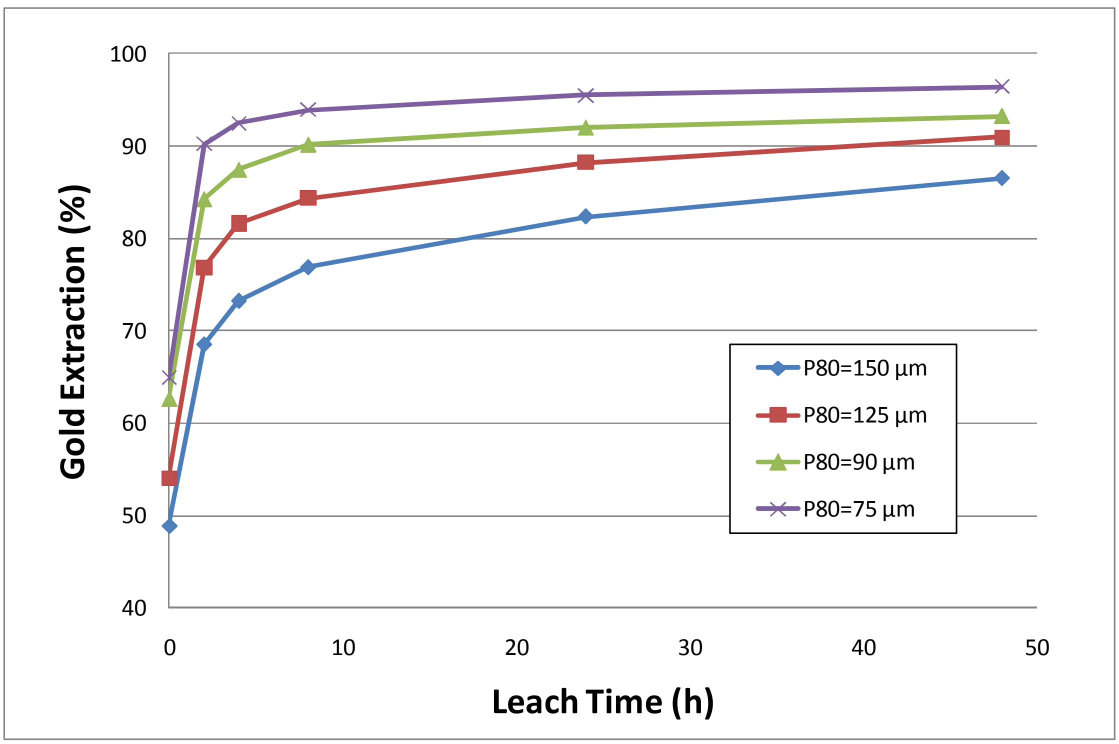


Figure 13-1: Gold Recovery as a Function of Grind and Leach Retention

Summary of Tests and Recoveries

A compilation of all the relevant tests done, limited to those samples within the currently defined resource, produced the recoveries shown in Figure 13-2.

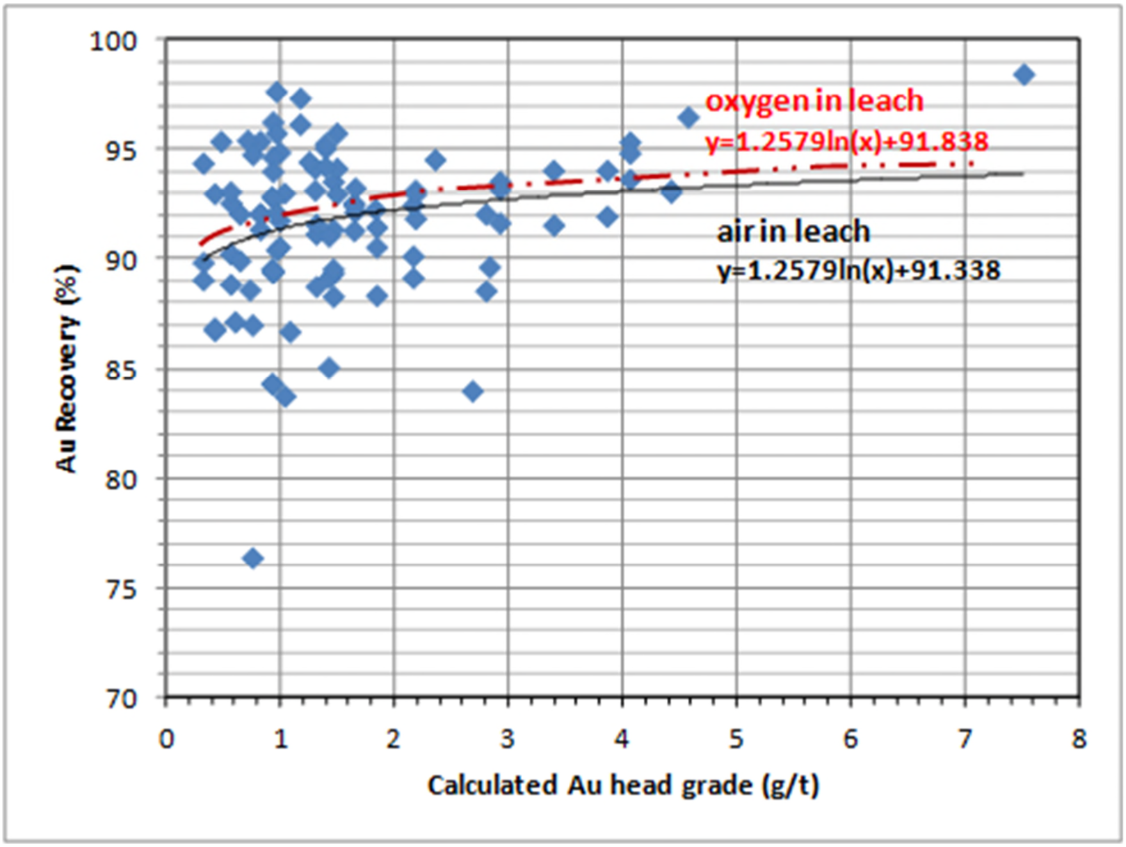


Figure 13-2: Gold Grade vs. Recovery Relationship

The figure shows that all of the selected samples leached well, that oxygen enhancement improved leach rate, and that recoveries are between 87% and 93% at a grade of 2 g/t. The recoveries are predominantly above 86%, with four exceptions that, from a metallurgical perspective, gives high confidence that all the sampled parts of the orebody are amenable to gravity and cyanidation. The trend line indicates a relationship between head grade and gold recovery, with higher recoveries achieved at higher gold head grades as expected. The mathematical relationship developed was used to estimate recovery based on the ore grade obtained from the mine plan.

Summary of Thickening Characteristics

AMMTEC performed flocculant screening tests on ground composite samples of West Branch ore using seawater obtained near Perth. Magnafloc MF336 flocculant was selected for subsequent settling tests to optimize flocculant consumption and develop thickener sizing criteria.

Thickening characteristics of deeper level variability samples were determined through Outotec test work in 2010, FLSmidth test work in 2011, SGS Lakefield test work in 2013 and FLSmidth test work in 2013. Outotec investigated the dynamic settling characteristics and determined the thickener sizing criteria. In 2011, FLSmidth conducted sedimentation and rheology testing. SGS conducted dynamic settling tests on a number of composite samples that had been prepared for leaching test work in 2012 and 2013. Based on SGS results, an average thickener unit area of 0.045 m^2/t/d was estimated. In 2013, FLSmidth determined sizing operating parameters for pre-leach and CIL tailing thickeners. Based on 2013 FLSmidth results, an average thickener unit area of 0.035 m^2/t/d was estimated.

Acid Mine Drainage Characteristics of West Branch Samples

Acid rock drainage (ARD) testing was completed on leach residue generated from the GST samples in the AMMTEC 2011 follow-up test work program to simulate plant tailings. Results indicated that the leach residues do not have potential acid generating characteristics, but have significant acid consuming capacity (likely due to the carbonate content of each ore composite).

In 2011, a waste rock material characterization program was conducted by URS Scott Wilson and supported by Kinross Tasiast and SRK Consulting. During the study, 154 samples were collected from exploration drilling core of different lithologies to assess the ARD potential. Study results showed that waste rock typically exhibits a significant residual neutralization potential for all the lithologies investigated.

The study results, coupled with the favorable arid climate, lack of surface water and very limited groundwater (no viable groundwater aquifer exists) and a Materials Management Plan (as outlined in Section 14.2 under “Acid Rock Drainage Potential” and Section 16 under “Waste Dumps”) indicate low potential for ARD or metal leaching to develop.

14. MINERAL RESOURCE ESTIMATE

Kinross Technical Services (KTS) and T. Maunula & Associates Consulting Inc. (TMAC) prepared the Tasiast resource estimate model using Gemcom GEMS™ 6.4.1 desktop software. The model was prepared using interpreted mineralized domains built on a 0.1 g/t Au cutoff grade. The project was divided into two parts at 72,325N, with WB (West Branch) south of that coordinate and Piment+Prolongation (PP) north of it.

The resource is stated using variable cutoff grades, which are dependent upon the ore type and pit shell. To report the resource, the project was further subdivided based on the location of the “land bridge” at 73,500N. The effective date of this 38 kt/d FS Mineral Resource is December 31, 2013.

14.1 Database

The database, which is maintained in a Century Systems platform, was supplied by site staff in the form of exported .csv files, which included the header, down-hole survey, lithology, density, structure, mineralization, alteration and assay data sets.

The database contained 4,857 drill holes, of which 93% were RC holes and 7% were diamond drill holes. A total of 4,343 holes (89.4%) in the database provided were used for grade estimation (Table 14-1). The previous resource estimate contained 4,038 holes, so an additional 819 additional holes were available for the FS2013 model.

Those holes (or assay values) that were not used in the FS2013 models were primarily excluded due to QA/QC failure, incomplete holes or because they were outside the interpreted mineralized domains.

Table 14-1: Summary of Drill Hole Database

Area	No. of Drill Holes	No. of Overlap Drill Holes*	No. of Assay Records	No. of Survey Records	Total Depth (m)
West Branch	2,481	123	493,327	60,765	589,356.94
Piment+Prolongation	2,653	154	304,075	27,488	351,165.82

* to avoid edge effects from the resource estimation process, drill holes were exported from the database with a 200 m overlap on either side of the initial model boundary of 72,325N.

Data Import

Before conducting statistical analyses, all data was imported into GEMS software and a check on the database was performed to search for any obvious errors, such as negative values and overlapping sample intervals.

KTS performed a visual check of the drilling against the most recent topographic surface, which revealed that the majority of collars are set to the surface. However, some drill holes were noted above the topographic surface, and this was deemed to be a consequence of poor topographic resolution and the variable elevation arising from the mining operations.

14.2 Wireframes

Four sets of wireframes were included in the Tasiast project:

- Lithology (see Table 14-2)
- Oxide (see Table 14-3)
- Mineralized domains (see Table 14-4)
- Acid rock drainage (ARD) potential (see Table 14-5)

Note these four tables document the wireframes using the GEMS nomenclature. The mineralized domains were the basis for grade estimation, the lithological wireframes were used for density, and the oxide surfaces were used to assist with density assignment and determine ore type for resource reporting. All wireframes used in the model are summarized in Table 14-2 through Table 14-5. The wireframes are listed in order of decreasing precedence, which is a standard set-up for volumetrics reporting in GEMS.

Table 14-2: Lithology – 2013 Lith Workspace

Workspace	Rock Code	Name 1	Name 2	Name 3	Lithcode	Precedence
2013Lith	DYKE	Piment	DYKE	May2013	10	1
2013Lith	RSPRLT	Piment	RSPRLT1	May2013	20	2
2013Lith	BIMHW	Piment	BIM1HW	May2013	40	3
2013Lith	BIMFW	Piment	BIM2FW	May2013	45	4
2013Lith	BIMHW	Piment	BIM2HW	May2013	40	5
2013Lith	BIMHW	Piment	BIM3HW	May2013	40	6
2013Lith	BIMHW	Piment	BIM4HW	May2013	40	7
2013Lith	BIMFW	Piment	BIM5FW	May2013	45	8
2013Lith	BIMHW	Piment	BIM5HW	May2013	40	9
2013Lith	BIMFW	Piment	BIM6FW	May2013	45	10
2013Lith	BIMHW	Piment	BIM6HW	May2013	40	11
2013Lith	FVC	Piment	FVC1	May2013	50	12
2013Lith	FVC	Piment	FVC2	May2013	50	13
2013Lith	SVC	Piment	SVC1	May2013	60	14
2013Lith	SVC	Piment	SVC2	May2013	60	15
2013Lith	SVC	Piment	SVC3	May2013	60	16
2013Lith	SVC	Piment	SVC4	May2013	60	17
2013Lith	SVC	Piment	SVC5	May2013	60	18
2013Lith	SVC	Piment	SVC6	May2013	60	19
2013Lith	SVC	Piment	SVC7	May2013	60	20
2013Lith	SVC	Piment	SVC8	May2013	60	21
2013Lith	DYKE	WB	MafDikes	2013	10	22
2013Lith	GST	WB	GSTMain	2013	30	23
2013Lith	GST	WB	GSTHWLeg	2013	30	24
2013Lith	GST	WB	GSTHW2	2013	30	25
2013Lith	GST	WB	GSTHW4	2013	30	26
2013Lith	GST	WB	GSTS2	2013	30	27
2013Lith	GST	WB	GSTHW	2013	30	28
2013Lith	GST	WB	GSTHW3	2013	30	29
2013Lith	GST	WB	GSTSouth	2013	30	30
2013Lith	BIMFW	WB	BIMFW	SJ2013	45	31
2013Lith	BIMHW	WB	BIMMnHW	2013	40	32
2013Lith	FVC	WB	FVC	2013	50	33
2013Lith	SVC	WB	SVC	2013	60	34
2013Lith	SVC	WB	SVCHost	2013	60	35
2013Lith	GST	Gap	GST	May2013	30	36
2013Lith	BIMHW	Gap	BIMMnHW	May2013	40	37
2013Lith	BIMFW	Gap	BIM2FW	May2013	45	38
2013Lith	FVC	Gap	FVC	May2013	50	39
2013Lith	SVC	Gap	SVC	May2013	60	40

Table 14-3: Oxide– 2011 Sulox Workspace

Workspace	Rock Code	Name 1	Name 2	Name 3	Oxcode	Precedence
2011Sulox		2011	2BOCO	NicJohnson		
2011Sulox		2011	2BDOX	NicJohnson		
2011Sulox		2011	2TOFR	NicJohnson		
2011Sulox	OXIDE	PP	BOCO	1 TO 2	1	1
2011Sulox	UT	PP	BDOX	2 TO 3	2	2
2011Sulox	LT	PP	TOFR	3 TO 4	3	3
2011Sulox	FRESH	PP	FRESH	4 TO 5	4	4
2011Sulox	OXIDE	WB	BOCO	1 TO 2	1	1
2011Sulox	UT	WB	BDOX	2 TO 3	2	2
2011Sulox	LT	WB	TOFR	3 TO 4	3	3
2011Sulox	FRESH	WB	FRESH	4 TO 5	4	4

Table 14-4: Mineralized Domains – 2013Min Workspace

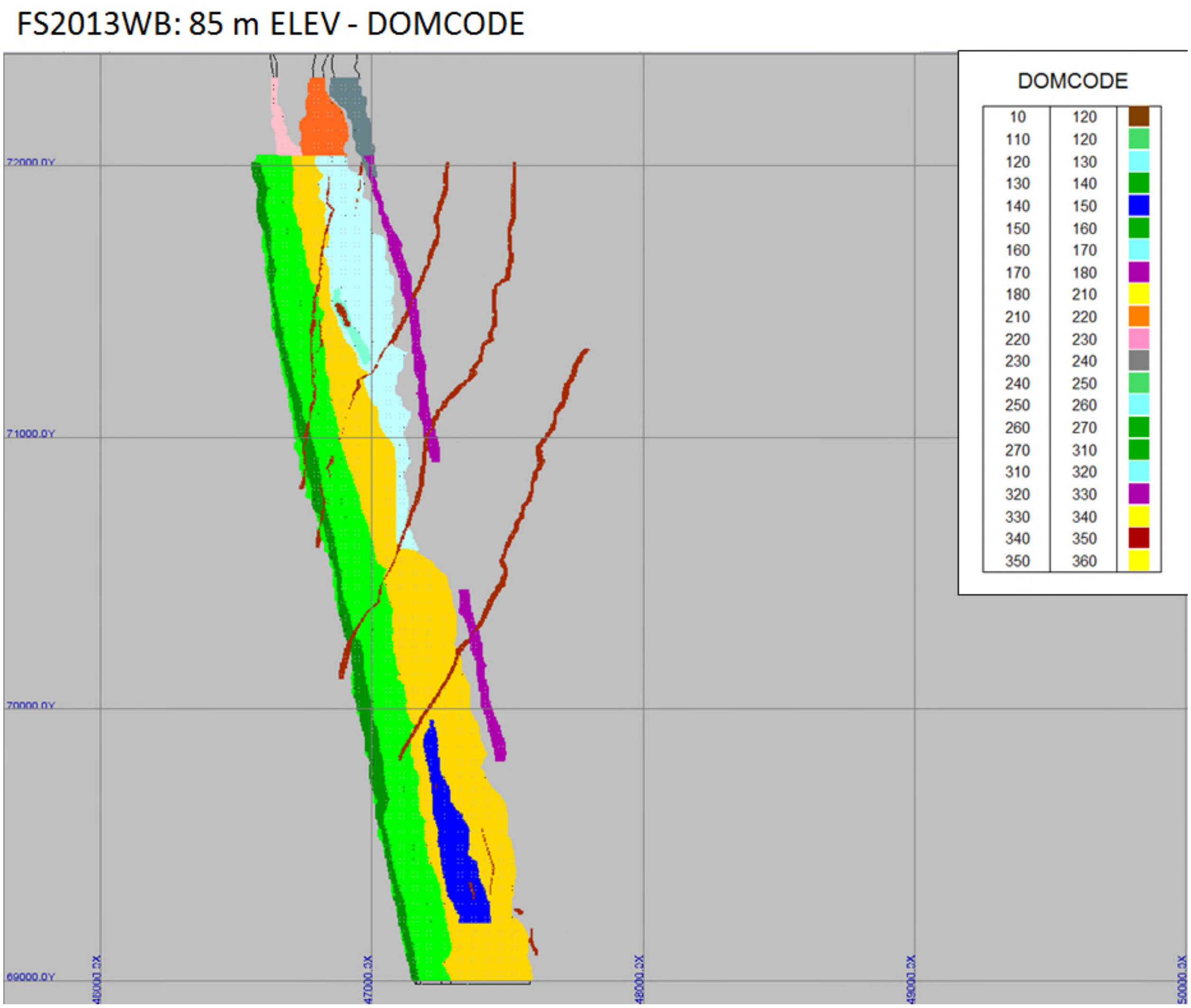
Workspace	Rock Code	Name 1	Name 2	Name 3	Domcode	Precedence
2013Min	DYKE	WB	MafDikes	2013	10	1
2013Min	DYKE	Piment	DYKE	May2013	10	2
2013Min	DOM6_1	WB	Dom6_1	May102013	110	3
2013Min	DOM6_2	WB	Dom6_2	May102013	120	4
2013Min	DOM2	WB	Dom2	May102013	130	5
2013Min	DOM7	WB	Dom7	May102013	140	6
2013Min	DOM3	WB	Dom3	May102013	150	7
2013Min	DOM8	WB	Dom8	May102013	160	8
2013Min	DOM9	WB	Dom9	May102013	170	9
2013Min	DOM5	WB	Dom5	May102013	180	10
2013Min	WBEXTEAS	Piment1	wbexteas	May102013	210	11
2013Min	WBEXTWES	Piment2	wbextwes	May102013	220	12
2013Min	PSSMAIN	Piment3	pssmain	May102013	230	13
2013Min	PN	Piment7	pn	May102013	240	14
2013Min	PCMAIN	Piment6	pcmain	May102013	250	15
2013Min	PCWEST	Piment5	pcwest	May102013	260	16
2013Min	PSNMAIN	Piment4	psnmain	May102013	270	17
2013Min	PRO2B	Pro2	b	May102013	310	18
2013Min	PRO2A	Pro2	a	May102013	320	19
2013Min	PRO1C	Pro1	c	May102013	330	20
2013Min	PRO1B	Pro1	b	May102013	340	21
2013Min	PRO1A	Pro1	a	May102013	350	22

Table 14-5: ARD Potential – ARD Workspace

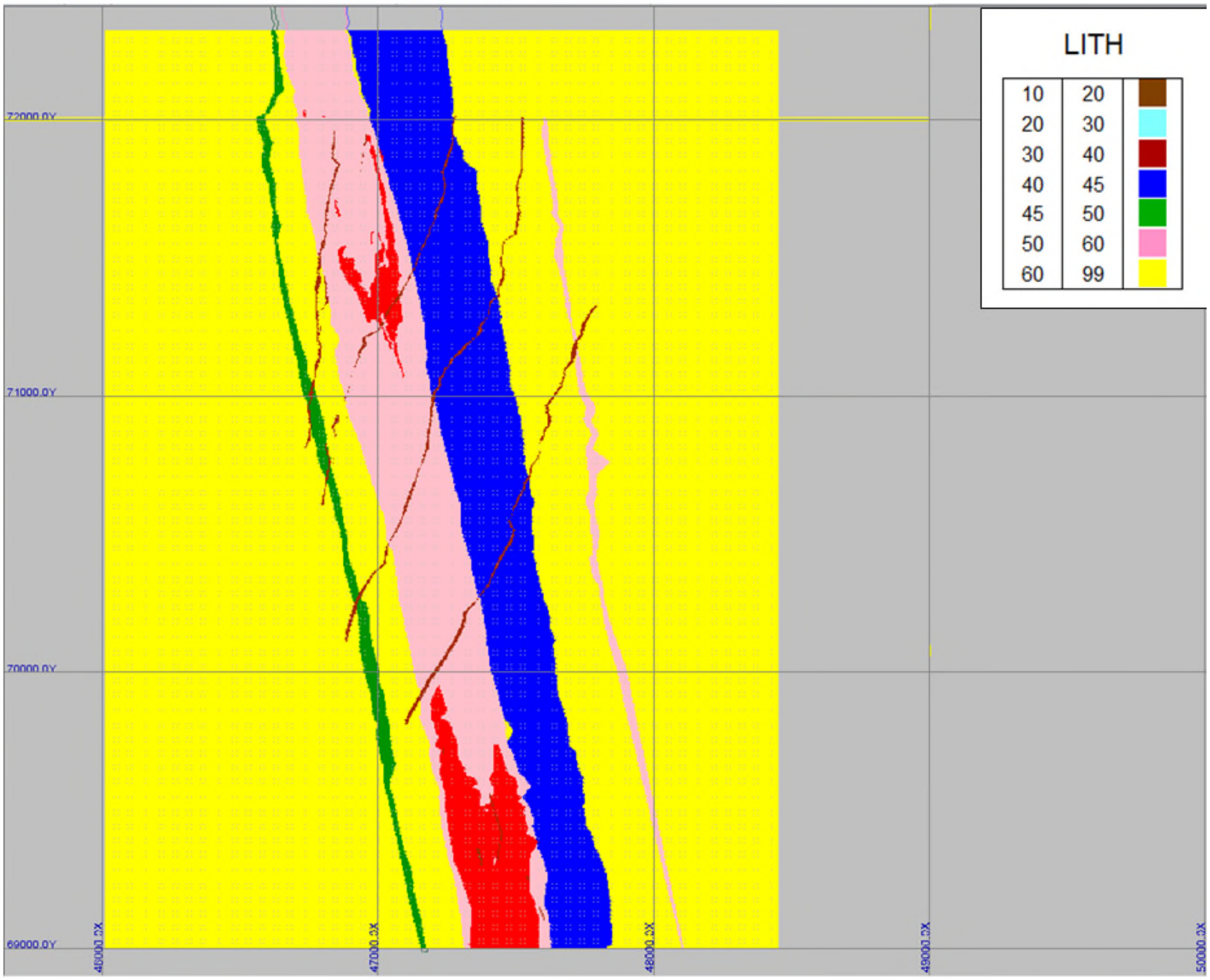
Workspace	Rock Code	Name 1	Name 2	Name 3	Code	Precedence
ARD	ARD	ARD	Charlie	July2013	555	

Figure 14-1 and Figure 14-2 show section and plan views of WB and PP, from the GEMS model.

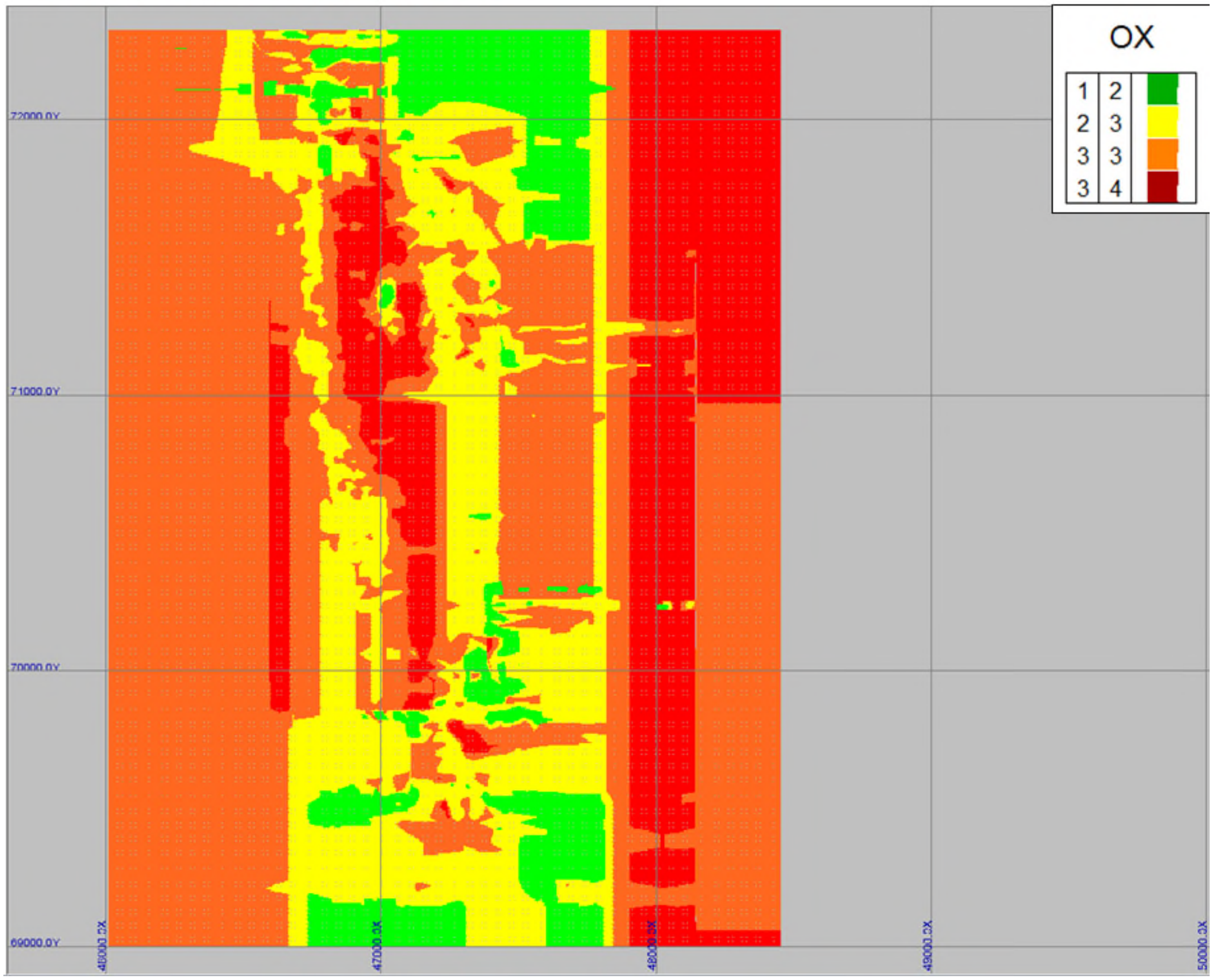
Figure 14-1: West Branch Plan and Section Views



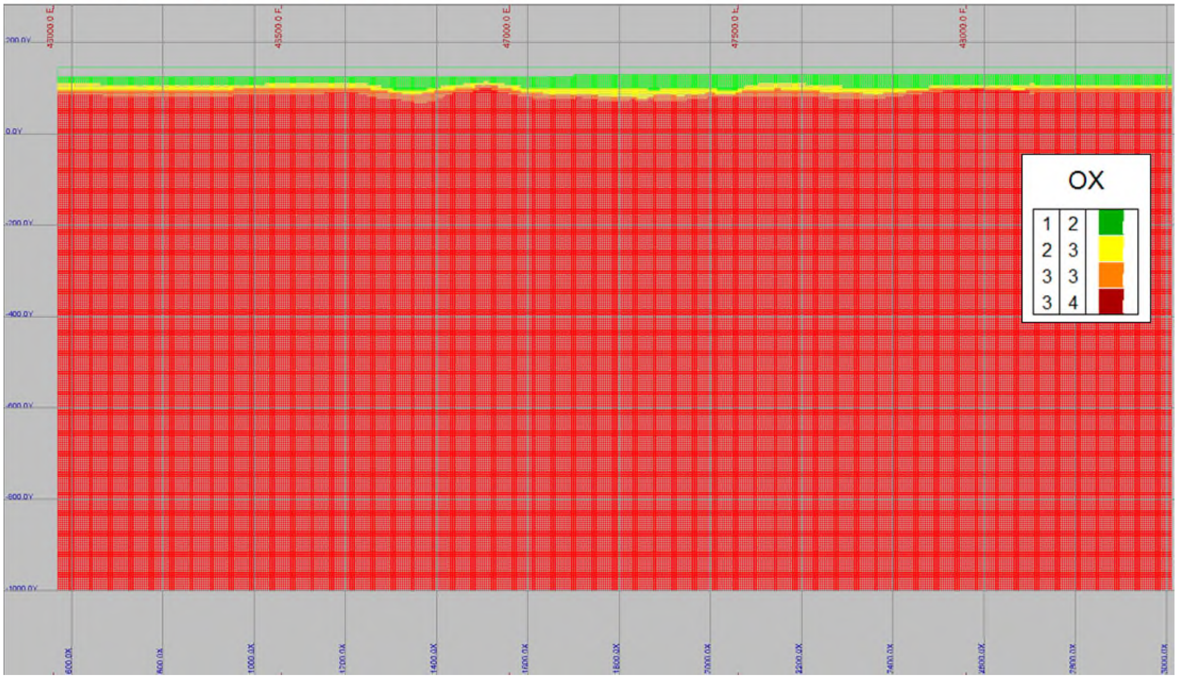
FS2013WB: 85 m ELEV - LITHCODE



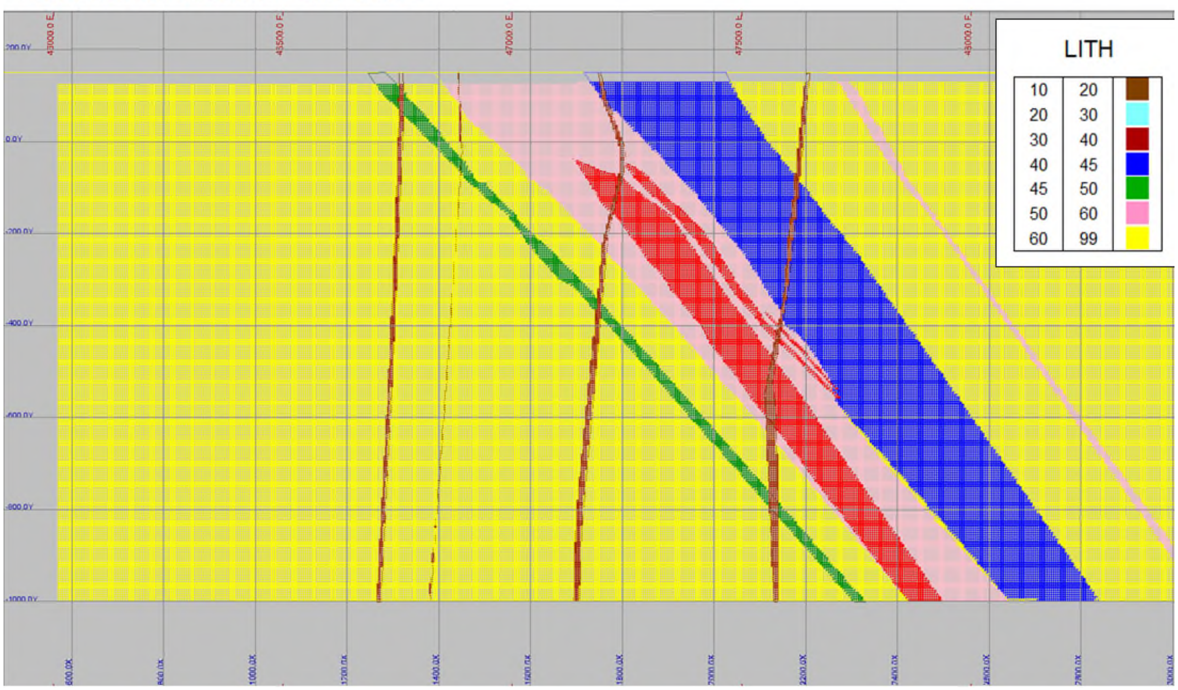
FS2013WB: 85 m ELEV - OXCODE



FS2013WB: 70998.5N - OXCODE



FS2013WB: 70998.5N - LITHCODE



FS2013WB: 70998.5N - DOMCODE

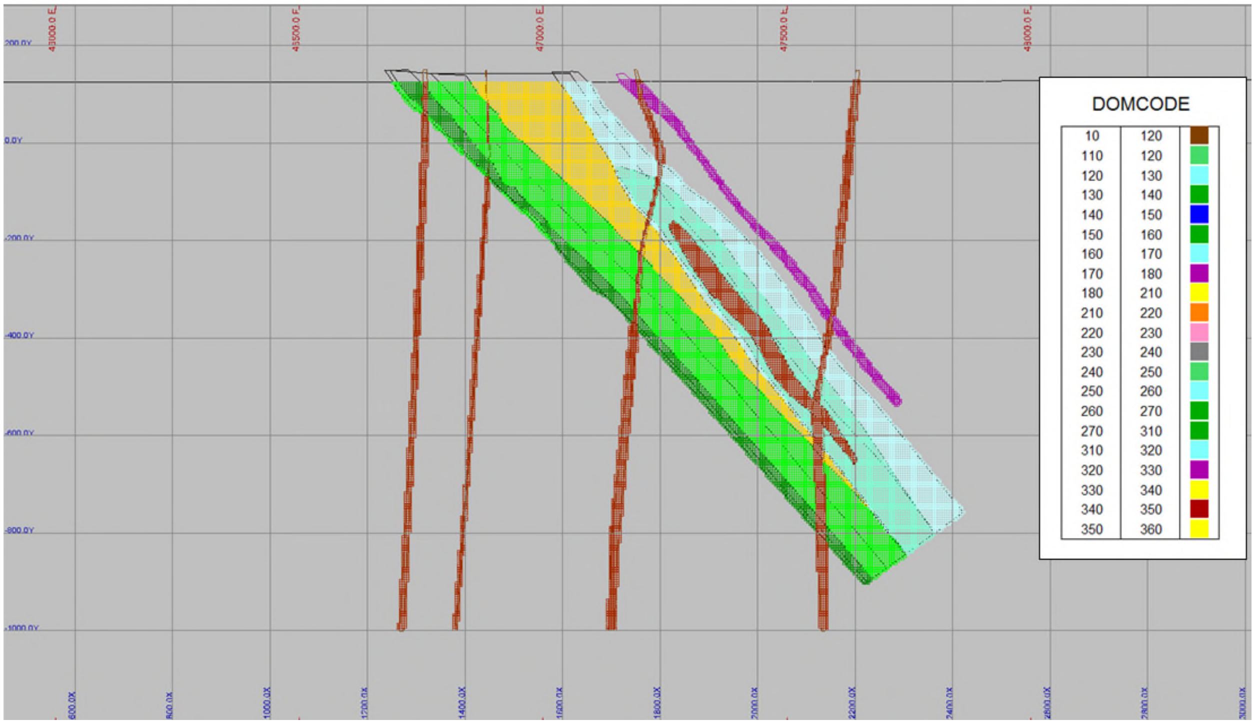
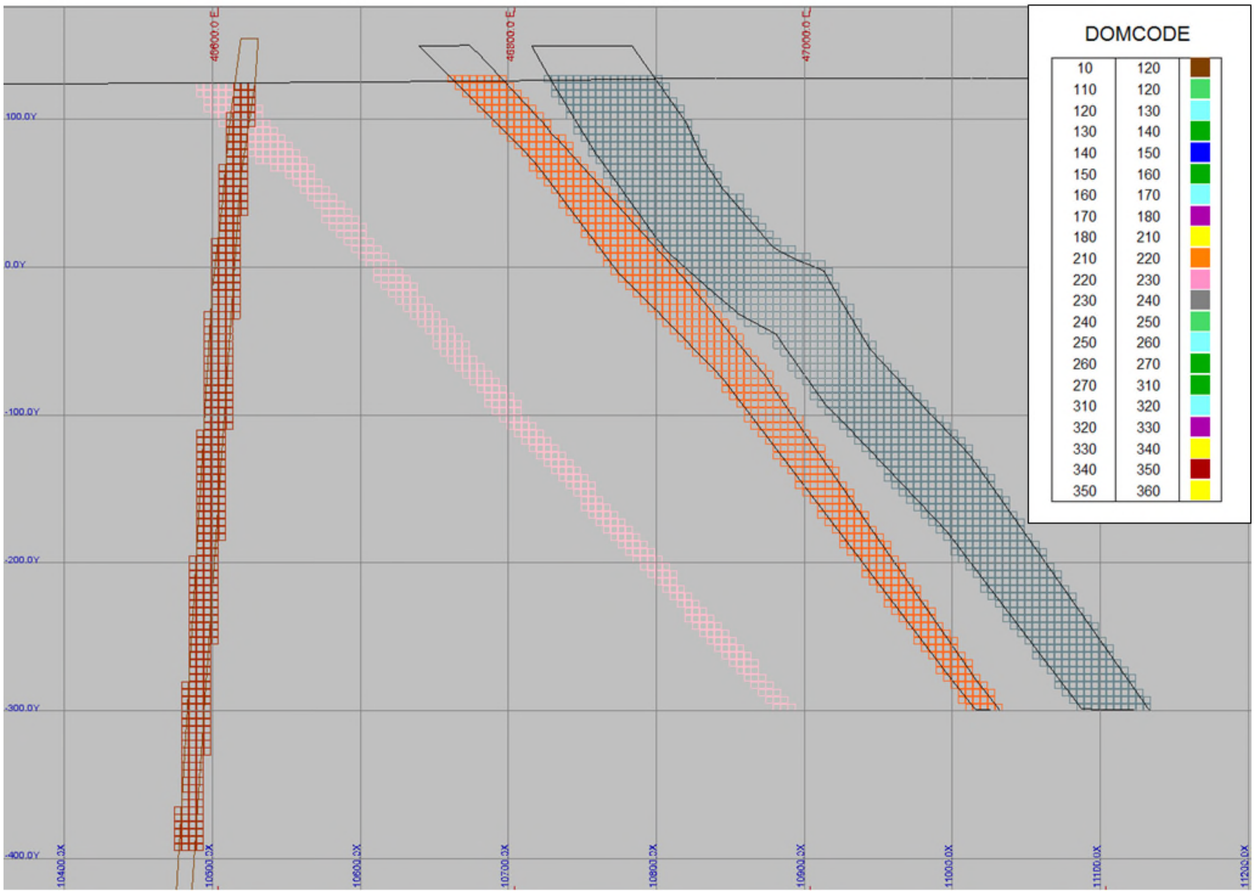
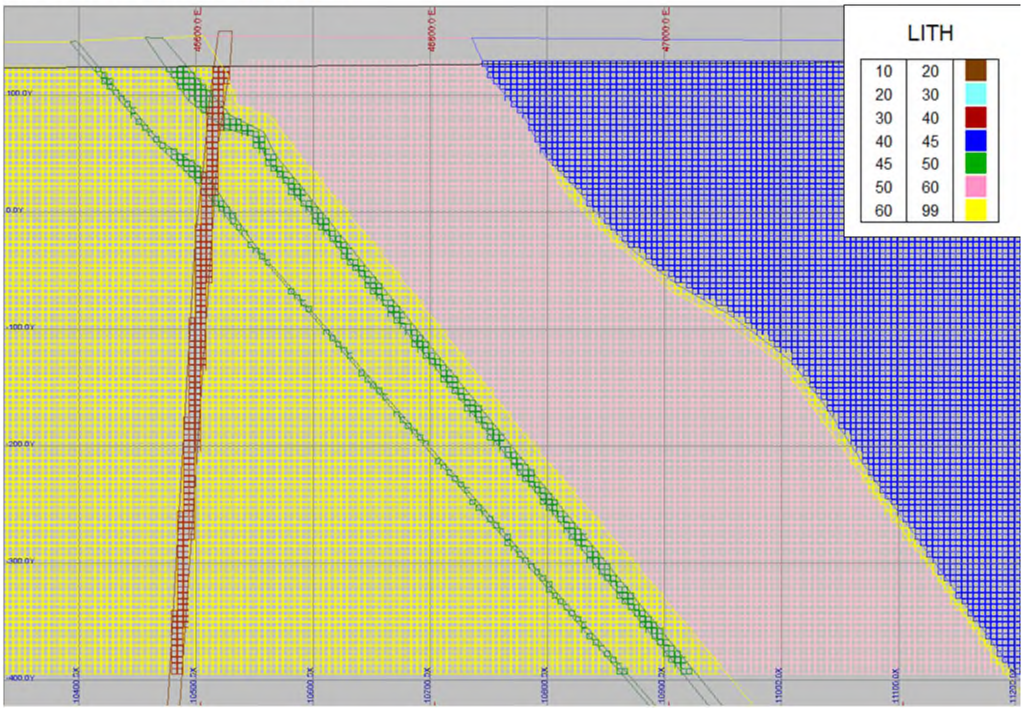


Figure 14-2: Piment Plan and Section Views

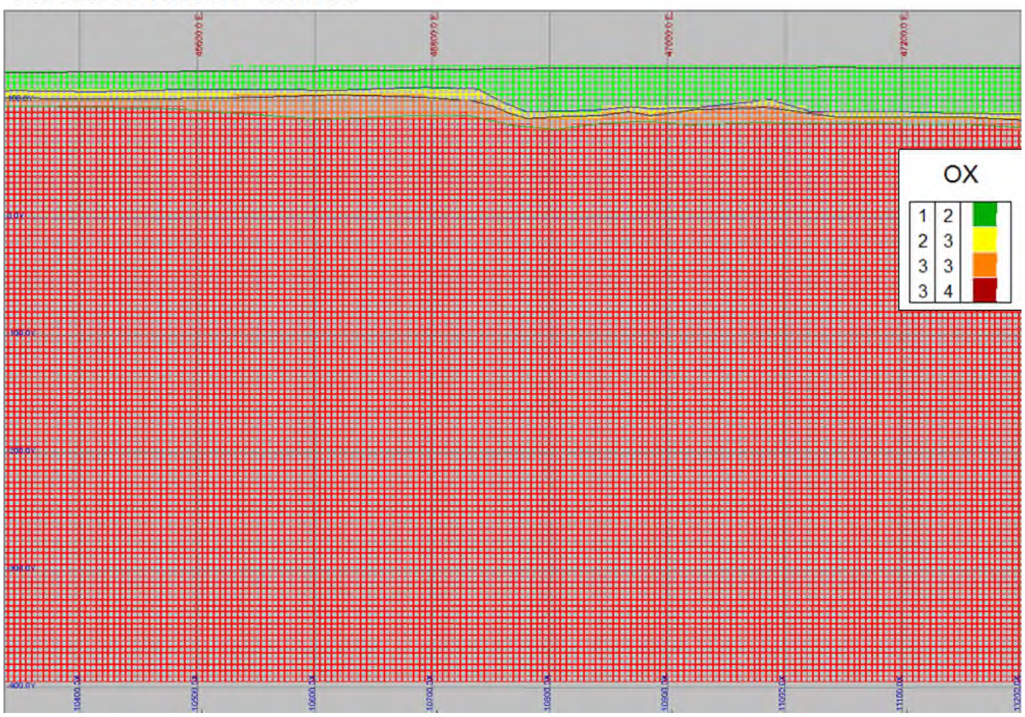
FS2013PP: 72356N - DOMCODE

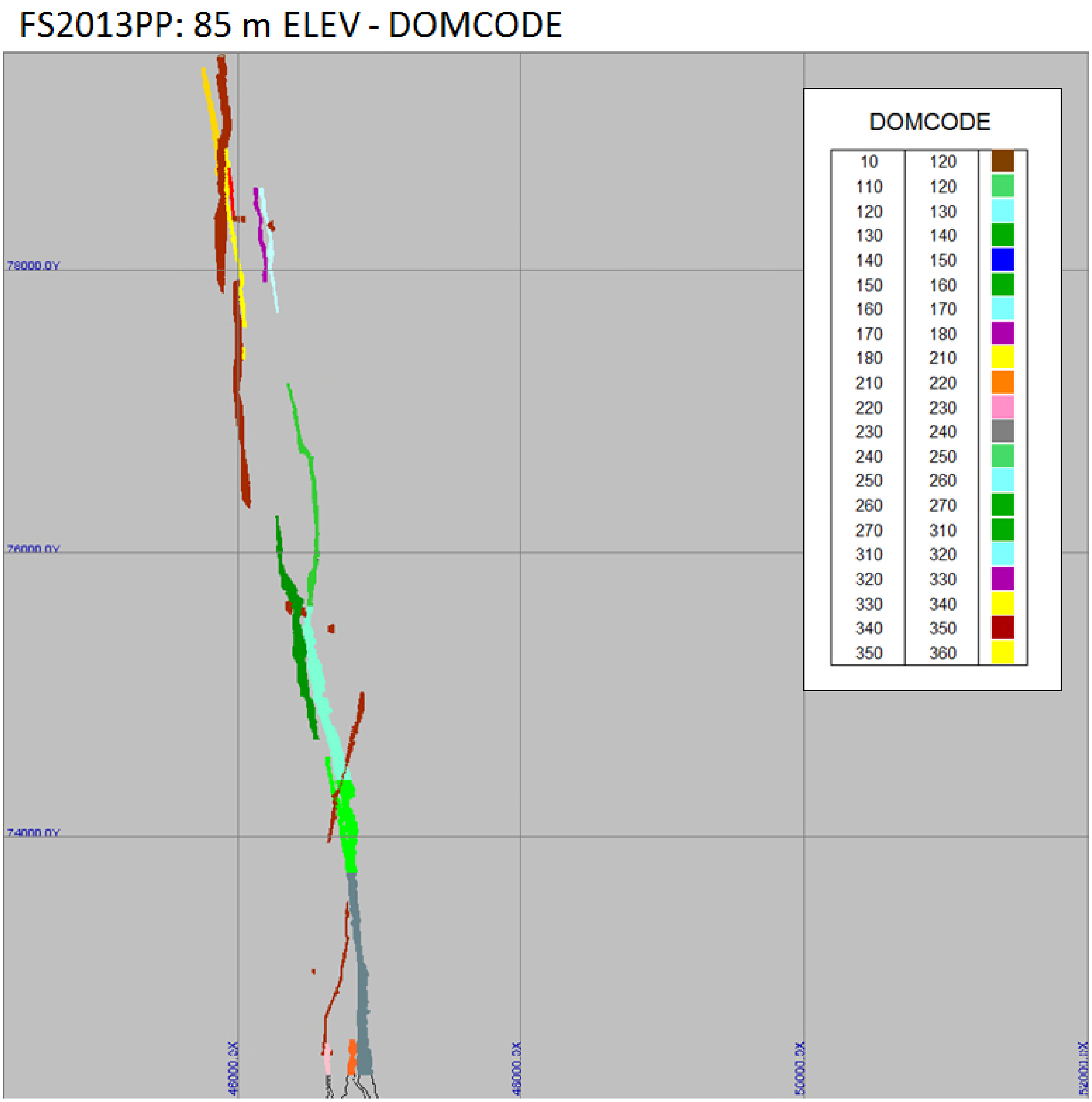


FS2013PP: 72356N - LITHCODE

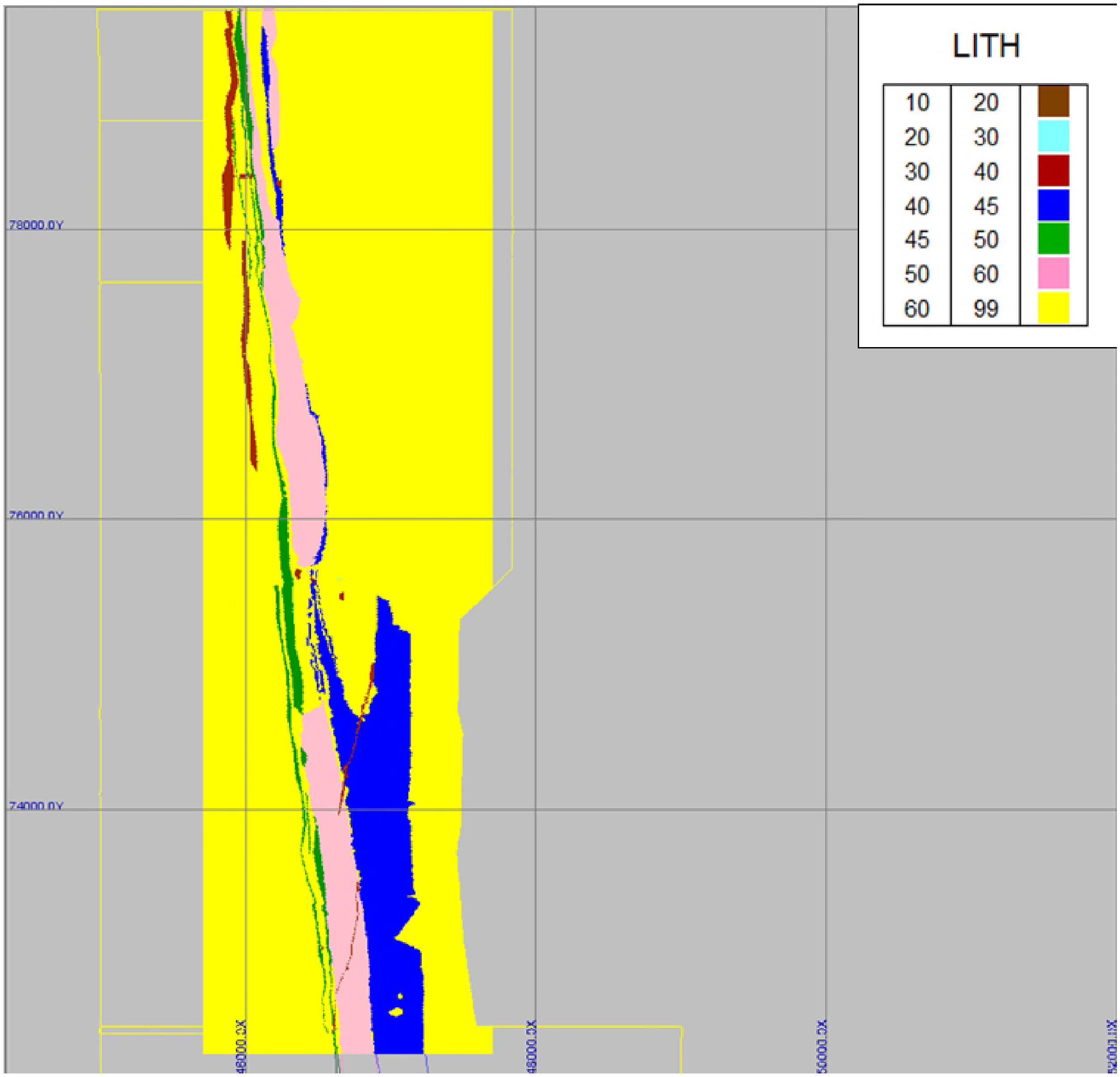


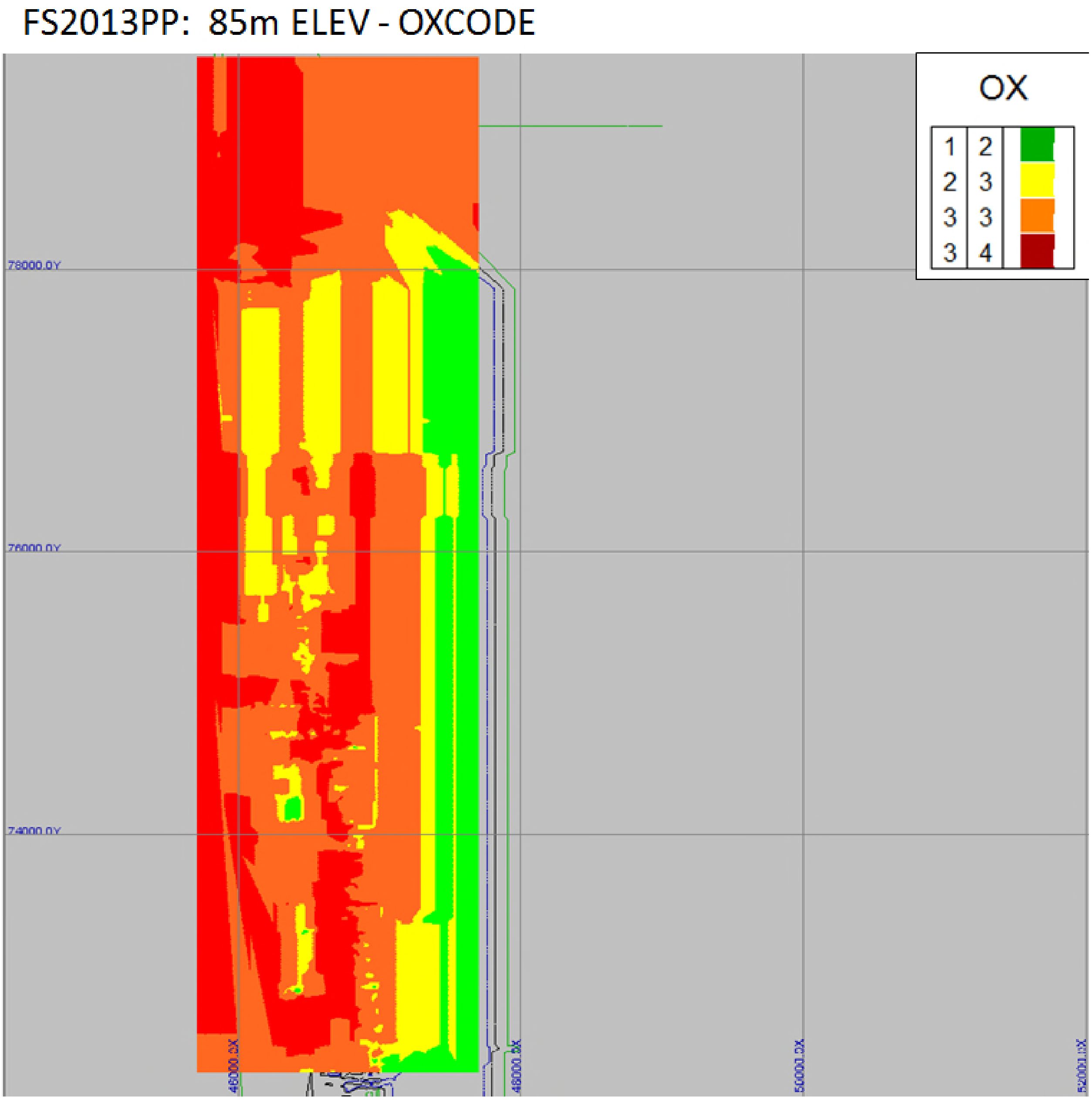
FS2013PP: 72356N - OXCODE




FS2013PP: 85 m ELEV - DOMCODE
78000.0Y
76000.0Y
74000.0Y
46000.0X
48000.0X
50000.0X
52000.0X
DOMCODE
10 120
110 120
120 130
130 140
140 150
150 160
160 170
170 180
180 210
210 220
220 230
230 240
240 250
250 260
260 270
270 310
310 320
320 330
330 340
340 350
350 360

FS2013PP: 85m ELEV - LITHCODE





Wireframe Construction Methodology

<u>Lithologies</u>

Geological wireframes were generated based on sectional interpretations every 25 m. Table 14-6 summarizes the codes and a brief description of each lithology.

Table 14-6: Lithology codes

Lithcode	Lithstring	Description
10	Dike	Mafic Dike
20	RSPRLT	Saprolite and Talus (overburden)
30	GST	Greenschist Zone
40	BIM	Banded Iron and Magnetite Formation
50	FVC	Felsite
60	SVC	Epiclastic Unit

<u>Mineralization</u>

Mineralized domains were constructed based on a greater than, or equal to, 0.1 g/t Au cutoff of the logged vein mineralization. For the high-grade zone associated with the GST lithology, a high-grade core was defined at greater than, or equal to, 2.0 g/t Au. Table 14-7 describes the mineralized domains based on cutoff grade and associated lithology.

Table 14-7: Mineralized Domains

Domcode	Domstring	Description
10	Dike	Mafic Dike, Barren
110	Dom6_1	GST, >2 g/t Au
120	Dom6_2	GST, 0.1<Au<2 g/t
130	DOM2	BIM (FW), >0.1 g/t Au
140	DOM7	South portion of GST (oxide), >0.1 g/t Au
150	DOM3	SVC (FW), >0.1 g/t Au
160	DOM8	FVC, >0.1 g/t Au
170	DOM9	Mineralized portion of BIM (HW), >0.1 g/t Au
180	DOM5	FVC/SVC (mineralized zones not included in other domains), >0.1 g/t Au
210	WBEXTEAS	SVC/FVC bridge between WB and PP, >0.1 g/t Au
220	WBEXTWES	SVC/FVC bridge between WB and PP, >0.1 g/t Au
230	PSSMAIN	East of WBEXTEAS/WBEXTWES, >0.1 g/t Au
240	PN	Piment North, >0.1 g/t Au
250	PCMAIN	Piment Central – Main Zone, >0.1 g/t Au
260	PCWEST	Piment Central – West Zone, >0.1 g/t Au
270	PSNMAIN	Piment Main extensions, >0.1 g/t Au
310	PRO2B	Prolongation, >0.1 g/t Au
320	PRO2A	Prolongation, >0.1 g/t Au
330	PRO1C	Prolongation, >0.1 g/t Au
340	PRO1B	Prolongation, >0.1 g/t Au
350	PRO1A	Prolongation, >0.1 g/t Au

<u>Oxide</u>

The same oxide surfaces generated for the 2011 resource estimate were used in the FS2013 model. Relogging and revised wireframes were completed for PP, but not for

WB. However, this revised interpretation was not used in order to maintain consistency in the oxide models between WB and PP. Metallurgical test work demonstrated that the upper transition is amenable to dump leaching, so the oxide surface is critical for ore typing and mine planning. Table 14-8 summarizes the oxide codes that were used to code the OX variable in the block model.

Table 14-8: Oxide Codes

Oxide Code	Oxide String	Description
1	Oxide	Oxide material
2	UT	Upper transition
3	LT	Lower transition
4	Fresh	Fresh material

Acid Rock Drainage Potential

Although the ARD test work results showed there is overall low potential for ARD concerns to develop, results for a very limited lithology and tonnage showed Potential Acid Forming conditions. In taking a precautionary approach, a wireframe for the ARD potential material at West Branch was created based on the contact between the hanging wall BIM and FVC unit combined with the presence of sulphide (pyrrhotite and pyrite) reported in the exploration mineralization and relogging files. If the sulphide presence was not reported for holes that crossed the lithological contact, a buffer of up to 5 m into the contact was applied. No ARD was interpreted for Piment and Prolongation. Figure 14-3 illustrates the positioning of the ARD wireframe within the ultimate pit.

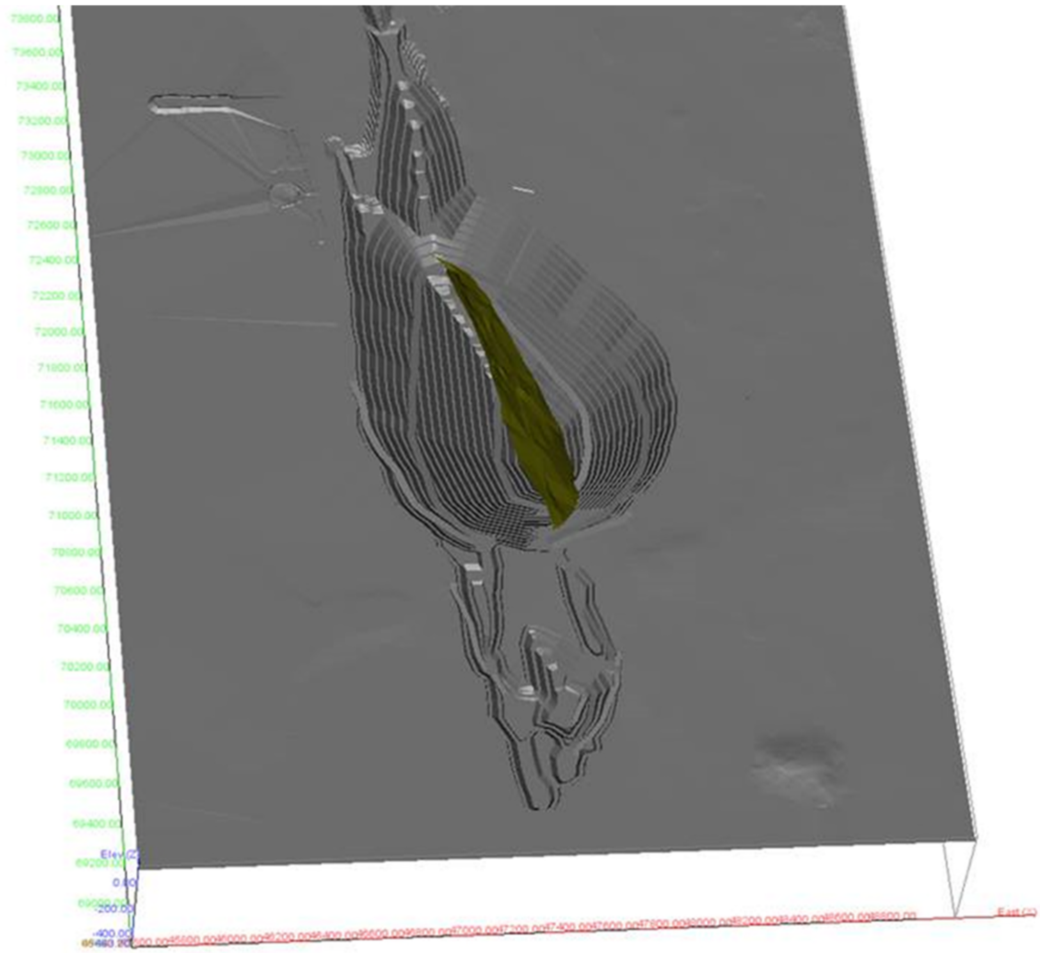


Figure 14-3: ARD Potential at West Branch

14.3 Exploratory Data Analysis

Exploratory data analysis (EDA) is the application of various statistical tools to explain the characteristics of a data set. In this case, the objective is to understand the population distribution of the grade elements using tools such as histograms, descriptive statistics and probability plots.

Outlier Management and Capping Strategy

When dealing with skewed populations and outlier assay samples relative to the normal population distribution, it is common practice to restrict the influence of these high-grade assays through capping. The capping limits were chosen as a function of the continuity–discontinuity of the high-grade tail of the gold assays.

The analysis included a visual review of the probability plots, a statistical assessment of the 97% and 99.9% percentiles, and decile analysis using the Parrish method. A summary of the analysis and the recommended capping levels are included in Table 14-9 and Table 14-10. As a result, 501 gold assays were capped. These values represent 0.13% of the entire sample population.

The capping was addressed by capping assay grades to the selected values before compositing.

Table 14-9: WB Au Capping Analysis

		Recommended Cap		Summary Statistics, Au g/t					Probability Plot, Au g/t			Decile Analysis
DomCode	**DomString**	**Au g/t**	**Count**	**Count**	**%Data**	**Mean**	**Maximum**	**CV**	**97%**	**99.9%**	**Outlier**	**Au g/t**
10	DYKE	**1.0**	**278**	10,174	3.3	0.147	101.000	9.14	1.0	8.0	50.0	
110	DOM6_1	**150.0**	**2**	15,661	5.1	3.411	176.000	1.33	12.0	50.0	150.0	70.0
120	DOM6_2	**60.0**	**3**	32,018	10.4	0.857	175.000	2.45	4.5	25.0	60.0	60.0
130	DOM2	**60.0**	**1**	16,871	5.5	0.427	77.200	2.49	1.8	10.0	60.0	20.0
140	DOM7	**10.0**	**9**	7,458	2.4	0.540	52.700	2.28	2.8	12.0	42.0	40.0
150	DOM3	**70.0**	**8**	79,244	25.7	0.435	496.000	5.61	2.8	15.0	70.0	70.0
160	DOM8	**45.0**	**6**	56,123	18.2	0.227	81.900	5.02	1.3	15.0	45.0	40.0
170	DOM9	**8.0**	**24**	8,067	2.6	0.308	15.800	2.85	2.4	10.0	8.0	8.0
180	DOM5	**70.0**	**6**	63,951	20.7	0.133	129.000	10.85	0.6	15.0	70.0	70.0
210	WBEXTEAS	**100.0**	**1**	7,725	2.5	0.517	154.000	5.46	3.0	35.0	150.0	150.0
220	WBEXTWES	**15.0**	**6**	2,954	1.0	0.460	100.000	4.86	3.5	18.0	15.0	15.0
230	PSSMAIN	**60.0**	**3**	8,137	2.6	0.492	83.500	3.75	3.0	20.0	65.0	60.0
*	GLOBAL		**347**	**308,383**		**0.522**	**496.000**	**4.15**	2.8	17.0	60.0	60.0

Table 14-10: PP Au Capping Analysis

		Recommended Cap		Summary Statistics, Au g/t					Probability Plot, Au g/t			Decile Analysis
DomCode	**DomString**	**Au g/t**	**Count**	**Count**	**%Data**	**Mean**	**Maximum**	**CV**	**97%**	**99.9%**	**Outlier**	**Au g/t**
10	DYKE	**1.0**	**16**	5,698	6.5	0.021	7.700	7.21	0.1	3.0	10.0	
210	WBEXTEAS	**100.0**	**1**	5,431	6.2	0.501	154.000	5.88	3.0	35.0	150.0	150.0
220	WBEXTWES	**15.0**	**3**	1,551	1.8	0.438	18.900	3.11	4.0	15.0	17.0	15.0
230	PSSMAIN	**60.0**	**6**	19,885	22.6	0.733	88.900	3.13	4.5	30.0	60.0	60.0
240	PN	**35.0**	**5**	2,809	3.2	1.555	119.000	2.78	10.3	40.0	35.0	35.0
250	PCMAIN	**35.0**	**64**	25,165	28.6	1.219	118.000	3.03	8.0	50.0	35.0	35.0
260	PCWEST	**35.0**	**29**	13,520	15.3	0.663	166.000	5.09	5.0	45.0	35.0	35.0
270	PSNMAIN	**20.0**	**16**	9,234	10.5	0.714	83.300	3.15	5.0	25.0	20.0	20.0
310	PRO2B	**18.0**	**6**	645	0.7	0.548	28.200	4.12	5.5	28.0	18.0	20.0
320	PRO2A	**9.0**	**2**	524	0.6	0.278	21.100	4.27	3.0	21.0	9.0	9.0
330	PRO1C	**35.0**	**3**	1,276	1.4	1.275	49.300	3.02	11.0	40.0	35.0	35.0
340	PRO1B	**10.0**	**1**	315	0.4	0.264	20.300	4.92	1.5	20.0	20.0	20.0
350	PRO1A	**50.0**	**2**	2,072	2.4	1.014	57.000	3.04	7.5	50.0	50.0	50.0
*	GLOBAL		**154**	**88,125**		**0.857**	**166.000**	**3.58**	2.5	25.0	65.0	65.0

Descriptive Statistics – Raw Data (Capped and Uncapped)

Table 14-11 and Table 14-12 summarize statistics by mineralized domain for uncapped and capped gold assay grades in WB. It is important to note that capping reduced the coefficient of variation of the various domains, particularly for the high grade domain of 6-1 in West Branch.

Table 14-11: WB Uncapped Gold Assay Statistics by Domain

DomCode	DomString	Count	Mean Grade (g/t)	Standard Deviation	Variance	Coefficient of Variation
10	DYKE	10174	0.147	1.342	1.801	9.144
110	DOM6_1	15661	3.412	4.554	20.737	1.335
120	DOM6_2	32018	0.857	2.097	4.398	2.448
130	DOM2	16871	0.427	1.065	1.134	2.491
140	DOM7	7458	0.540	1.234	1.522	2.284
150	DOM3	79244	0.435	2.439	5.947	5.605
160	DOM8	56123	0.227	1.137	1.294	5.021
170	DOM9	8067	0.308	0.876	0.767	2.845
180	DOM5	63951	0.133	1.442	2.080	10.851
210	WBEXTEAS	7725	0.517	2.818	7.943	5.456
220	WBEXTWES	2954	0.461	2.240	5.017	4.862
230	PSSMAIN	8137	0.492	1.845	3.403	3.751

Table 14-12: WB Capped Gold Assay Statistics by Domain

DomCode	DomString	Count	Mean Grade (g/t)	Standard Deviation	Variance	Coefficient of Variation	Number of Capped Assays
10	DYKE	10174	0.087	0.204	0.042	2.351	278
110	DOM6_1	15661	3.409	4.467	19.953	1.310	2
120	DOM6_2	32018	0.851	1.809	3.273	2.125	3
130	DOM2	16871	0.426	0.997	0.995	2.339	1
140	DOM7	7458	0.525	0.897	0.805	1.708	9
150	DOM3	79244	0.426	1.507	2.271	3.534	8
160	DOM8	56123	0.225	1.045	1.092	4.650	6
170	DOM9	8067	0.302	0.801	0.642	2.658	24
180	DOM5	63951	0.131	1.291	1.668	9.885	6
210	WBEXTEAS	7725	0.510	2.485	6.174	4.876	1
220	WBEXTWES	2954	0.428	1.266	1.603	2.959	6
230	PSSMAIN	8137	0.487	1.661	2.758	3.409	3

Table 14-13 and Table 14-14 summarize statistics by mineralized domain for uncapped and capped gold grades in PP.

Table 14-13: PP Uncapped Gold Assay Statistics by Domain

DomCode	DomString	Count	Mean Grade (g/t)	Standard Deviation	Variance	Coefficient of Variation
10	DYKE	5698	0.025	0.165	0.027	6.683
210	WBEXTEAS	5431	0.533	3.033	9.200	5.695
220	WBEXTWES	1551	0.469	1.404	1.972	2.994
230	PSSMAIN	19885	0.756	2.323	5.399	3.074
240	PN	2809	1.559	4.330	18.750	2.777
250	PCMAIN	25165	1.255	3.736	13.961	2.978
260	PCWEST	13520	0.669	3.388	11.476	5.065
270	PSNMAIN	9234	0.739	2.288	5.233	3.095
310	PRO2B	645	0.803	2.696	7.271	3.360
320	PRO2A	524	0.377	1.369	1.875	3.631
330	PRO1C	1276	1.330	3.925	15.404	2.950
340	PRO1B	315	0.347	1.480	2.191	4.264
350	PRO1A	2072	1.170	3.282	10.775	2.806

Table 14-14: PP Capped Gold Grade Statistics by Domain

DomCode	DomString	Count	Mean Grade (g/t)	Standard Deviation	Variance	Coefficient of Variation
10	DYKE	5698	0.021	0.073	0.005	3.512
210	WBEXTEAS	5431	0.523	2.586	6.685	4.947
220	WBEXTWES	1551	0.465	1.363	1.858	2.928
230	PSSMAIN	19885	0.752	2.211	4.887	2.939
240	PN	2809	1.517	3.645	13.285	2.403
250	PCMAIN	25165	1.213	3.125	9.763	2.576
260	PCWEST	13520	0.630	2.524	6.371	4.006
270	PSNMAIN	9234	0.710	1.733	3.005	2.443
310	PRO2B	645	0.768	2.415	5.833	3.146
320	PRO2A	524	0.352	1.080	1.165	3.063
330	PRO1C	1276	1.310	3.712	13.779	2.834
340	PRO1B	315	0.314	1.104	1.219	3.511
350	PRO1A	2072	1.165	3.208	10.290	2.753

Compositing and Descriptive Statistics (Capped and Uncapped)

Upon examination of the raw sample length statistics, a composite length of 2 m was chosen for the assay samples. Composites honoured the mineralized domain boundaries and were applied to the capped raw assays.

When compositing, assay values less than the detection limit were assigned a value half of the lower detection limit value, which depending on the laboratory, was commonly 0.003 g/t Au or 0.005 g/t Au.

Composite length statistics are provided in Figure 14-4 and Figure 14-5 for WB and PP. Table 14-15 and Table 14-16 summarize statistics by mineralized domain for uncapped and capped gold composite grades in WB.

Figure 14-4: Composite Length Statistics, West Branch

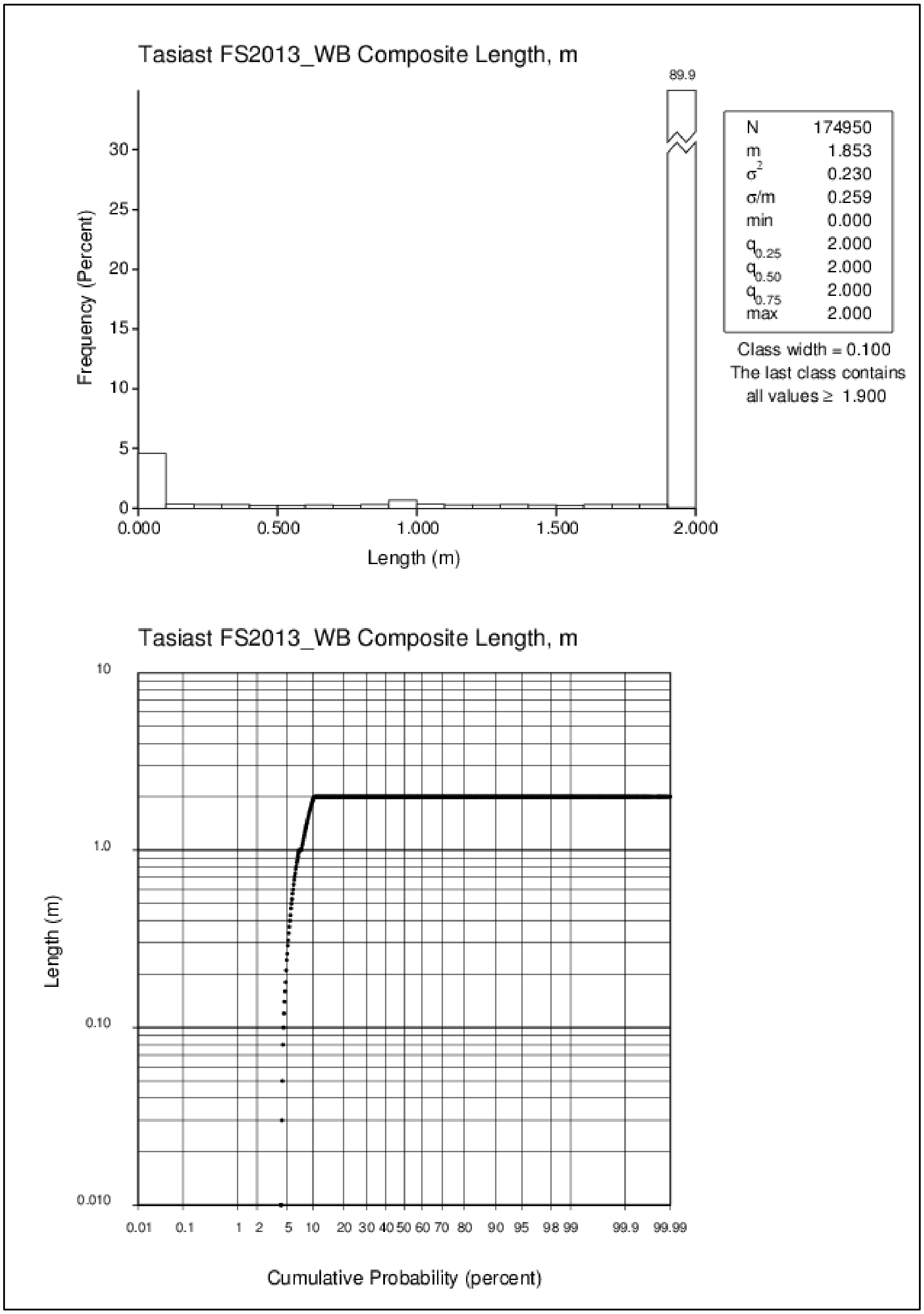
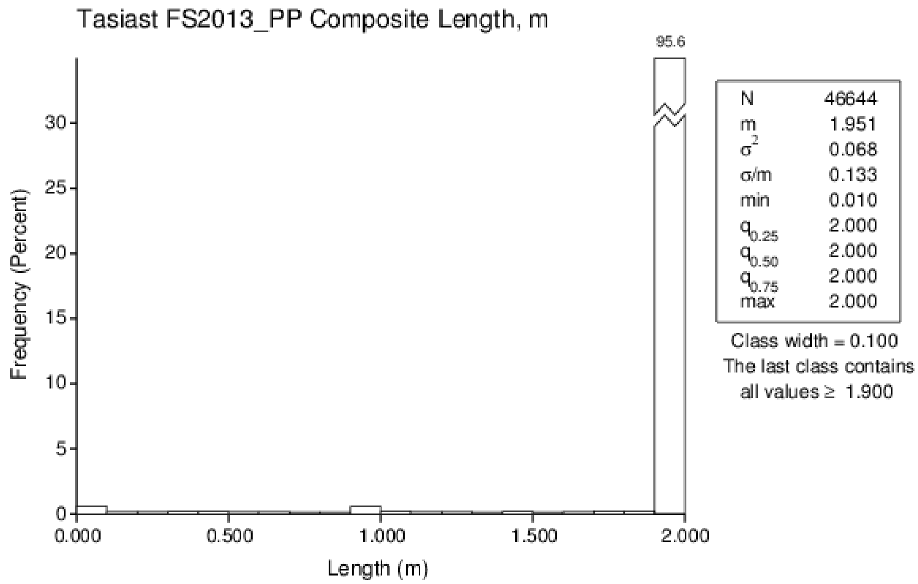


Figure 14-5: Composite Length Statistics, Piment



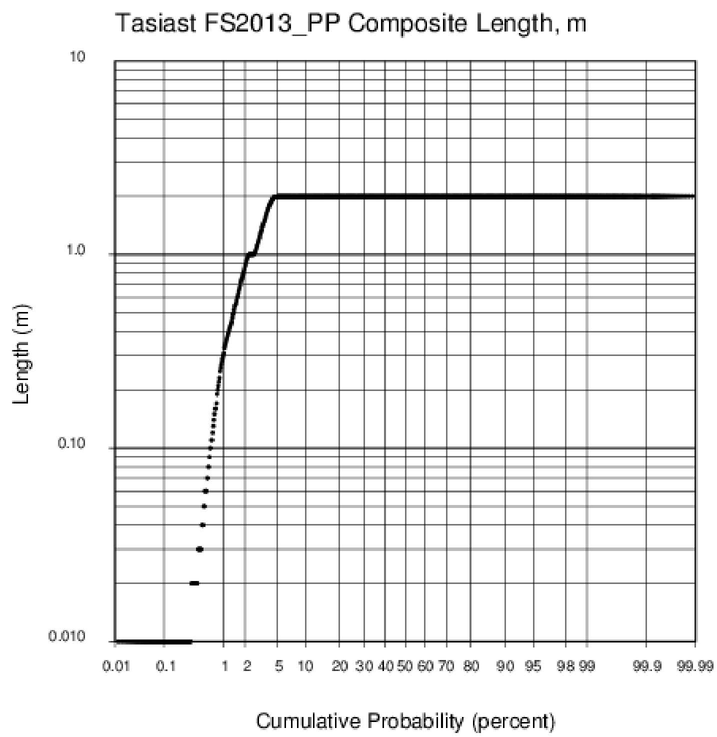


Table 14-15: WB Uncapped Gold Composite Grade Statistics by Domain

DomCode	DomString	Count	Mean Grade (g/t)	Standard Deviation	Variance	Coefficient of Variation
10	DYKE	6667	0.152	0.951	0.904	6.252
110	DOM6_1	8594	3.381	3.493	12.200	1.033
120	DOM6_2	17567	0.876	1.548	2.397	1.767
130	DOM2	9428	0.434	1.053	1.108	2.423
140	DOM7	4171	0.533	0.932	0.868	1.750
150	DOM3	43902	0.435	1.533	2.351	3.523
160	DOM8	32142	0.228	0.891	0.795	3.903
170	DOM9	5029	0.309	0.676	0.457	2.185
180	DOM5	36858	0.133	1.232	1.519	9.261
210	WBEXTEAS	4329	0.516	1.984	3.934	3.848
220	WBEXTWES	1741	0.461	1.557	2.425	3.380
230	PSSMAIN	4522	0.492	1.291	1.666	2.625

Table 14-16: WB Capped Gold Composite Grade Statistics by Domain

DomCode	DomString	Count	Mean Grade (g/t)	Standard Deviation	Variance	Coefficient of Variation
10	DYKE	6667	0.093	0.204	0.042	2.194
110	DOM6_1	8594	3.379	3.412	11.643	1.010
120	DOM6_2	17567	0.871	1.364	1.861	1.567
130	DOM2	9428	0.432	0.917	0.841	2.121
140	DOM7	4171	0.518	0.702	0.493	1.355
150	DOM3	43902	0.427	1.086	1.179	2.544
160	DOM8	32142	0.226	0.792	0.628	3.503
170	DOM9	5029	0.303	0.626	0.392	2.064
180	DOM5	36858	0.131	1.112	1.237	8.500
210	WBEXTEAS	4329	0.509	1.749	3.058	3.438
220	WBEXTWES	1741	0.429	0.999	0.998	2.332
230	PSSMAIN	4522	0.487	1.187	1.410	2.438

Table 14-17 and Table 14-18 summarize statistics by mineralized domain for uncapped and capped gold composite grades in PP.

Table 14-17: PP Uncapped Gold Composite Grade Statistics by Domain

DomCode	DomString	Count	Mean Grade (g/t)	Standard Deviation	Variance	Coefficient of Variation
10	DYKE	3735	0.025	0.121	0.015	4.843
210	WBEXTEAS	2997	0.531	2.120	4.492	3.994
220	WBEXTWES	916	0.481	1.093	1.194	2.274
230	PSSMAIN	11067	0.760	1.777	3.156	2.337
240	PN	1515	1.592	3.474	12.066	2.182
250	PCMAIN	14367	1.246	3.123	9.751	2.506
260	PCWEST	7082	0.668	2.532	6.410	3.790
270	PSNMAIN	5169	0.735	1.723	2.970	2.345
310	PRO2B	502	0.807	2.062	4.251	2.554
320	PRO2A	357	0.376	1.129	1.274	3.001
330	PRO1C	673	1.325	3.113	9.689	2.349
340	PRO1B	214	0.337	1.042	1.085	3.088
350	PRO1A	1346	1.272	2.679	7.179	2.106

Table 14-18: PP Capped Gold Composite Grade Statistics by Domain

DomCode	DomString	Count	Mean Grade (g/t)	Standard Deviation	Variance	Coefficient of Variation
10	DYKE	3735	0.022	0.065	0.004	2.977
210	WBEXTEAS	2997	0.521	1.801	3.243	3.458
220	WBEXTWES	916	0.477	1.070	1.145	2.243
230	PSSMAIN	11067	0.757	1.712	2.931	2.263
240	PN	1515	1.552	3.022	9.130	1.947
250	PCMAIN	14367	1.203	2.634	6.936	2.188
260	PCWEST	7082	0.629	1.985	3.941	3.154
270	PSNMAIN	5169	0.709	1.411	1.991	1.989
310	PRO2B	502	0.768	1.866	3.482	2.429
320	PRO2A	357	0.352	0.873	0.763	2.482
330	PRO1C	673	1.305	2.974	8.844	2.280
340	PRO1B	214	0.306	0.787	0.619	2.572
350	PRO1A	1346	1.268	2.640	6.969	2.082

Contact Profiles

Contact profiles were generated to confirm the grade interpolation limits along the domain contacts. Contact plots were generated between all domains. Contacts may be interpreted as:

- Soft (S) – soft contacts show no significant grade differential at, or near, the contact
- Firm (F) – firm contacts are present if the grade is transitional between two domains
- Hard (H) – hard contacts exhibit a sharp grade differential at the domain contact

Table 14-19 summarizes the contact relationships for West Branch. The default contact relationship was assumed to be hard. At WB, soft and firm contacts were identified. Firm contacts were implemented during grade interpolation as soft contacts.

Table 14-19: Summary of Domain Contact Relationship, WB

		10	110	120	130	140	150	160	170	180	210	220	230
DYKE	10		H	H	H	H	H	H	H	H	H	H	H
DOM6_1	110	H		H	H	H	H	H	H	H	H	H	H
DOM6_2	120	H	H		H	H	H	H	H	H	H	H	H
DOM2	130	H	H	H		H	S	H	H	H	H	H	H
DOM7	140	H	H	H	H		F	H	H	H	H	H	H
DOM3	150	H	H	H	S	F		S	H	H	H	S	H
DOM8	160	H	H	H	H	H	S		F	S	F	H	H
DOM9	170	H	H	H	H	H	H	F		H	H	H	S
DOM5	180	H	H	H	H	H	H	S			H	F	H
WBEXTEAS	210	H	H	H	H	H	H	F	H	H		H	H
WBEXTWES	220	H	H	H	H	H	S	H	H	F	H		H
PSSMAIN	230	H	H	H	H	H	H	H	S	H	H	H	

For PP, no soft boundaries were identified and only a single firm contact relationship was identified, as most of the domains were not in contact with adjoining domains. All contacts in PP were implemented as hard boundaries.

14.4 Variography

Geostatisticians use a variety of tools to describe the pattern of spatial continuity, or strength, of the spatial similarity of a variable with separation distance and direction. One of these tools is the correlogram, which measures the correlation between data values as a function of their separation distance and direction.

The approach to develop the variogram models used SAGE2001© software. Directional sample correlograms were calculated along horizontal azimuths of 0, 30, 60, 120, 150, 180, 210, 240, 270, 300 and 330 degrees. For each azimuth, sample correlograms were also calculated at dips of 30 degrees and 60 degrees, in addition to horizontally. Lastly, a correlogram was calculated in the vertical direction (-90 degrees). Using the 37 sample correlograms, an algorithm determined the best-fit model nugget effect and two-nested structure variance contributions. After fitting the variance parameters, the algorithm then fitted an ellipsoid to the 37 ranges from the directional models for each structure. The anisotropy of the correlation was given by the range along the major, semi-major and minor axes of the ellipsoids and the orientations of these axes for each structure. KTS reviewed the fitted model and made adjustments, as required, to reflect geological knowledge and grade continuity of the deposit.

All conventions follow those of the Cartesian coordinate system. For example, the Z axis will be vertical with values increasing upward, if the system of axes is oriented so that:

- The X axis runs east–west with values increasing to the east
- The Y axis runs north–south with values increasing to the north

A positive dip angle is measured upwards from the horizontal, whereas a negative dip angle is measured downwards from the horizontal.

The order and direction of the rotations around the three axes are given by the following (in each case the direction is given by the right-hand rule):

- The first rotation is around the Z axis.
- The second rotation is around the rotated Y axis.
- The third rotation is around the rotated Z axis.

Table 14-20 and Table 14-21 summarize the variogram results for the capped 2.0 m Au composites for WB and PP. All correlograms were spherical models consisting of a nugget and two structures.

Table 14-20: WB Gold Grade Correlogram Models

Domain	Nugget	Sill	Rot. Z	Rot. Y'	Rot. Z'	Range Z'	Range Y'	Range X'
10	0.450	0.524	4	-52	-3	7.0	135.8	21.6
		0.026	-97	119	75	322.3	245.4	136.1
110	0.200	0.700	54	125	34	14.2	28.4	7.4
		0.100	-57	-68	126	134.2	36.4	70.0
120	0.400	0.500	22	52	22	15.0	25.0	15.0
		0.100	58	-49	-35	300.0	150.0	50.0
130	0.400	0.490	-21	-22	43	25.0	30.0	25.0
		0.110	11	-41	92	250.0	50.0	250.0
140	0.400	0.402	-50	18	-1	4.8	35.2	35.5
		0.198	-18	-47	22	583.8	424.4	34.1
150	0.350	0.554	-19	45	-9	6.1	17.7	33.9
		0.096	-57	-69	31	516.9	408.8	54.4
160	0.350	0.600	9	55	-51	10.0	15.0	30.0
		0.050	59	-56	-33	111.3	184.8	31.7
170	0.250	0.539	23	11	55	4.5	39.3	41.0
		0.211	14	-40	-2	263.0	604.8	41.2
180	0.350	0.500	-2	-25	58	15.0	40.0	30.0
		0.150	-71	99	123	250.0	60.0	50.0
210	0.500	0.350	-7	-78	-3	30.0	45.0	10.0
		0.150	-32	126	66	250.0	35.0	45.0
220	0.200	0.650	-1	57	10	20.0	25.0	45.0
		0.150	-62	-72	52	350.0	250.0	60.0
230	0.400	0.492	-1	46	37	6.5	21.4	32.4
		0.108	3	-53	-3	141.5	247.0	64.2

Table 14-21: PP Gold Grade Correlogram Models

Domain	Nugget	Sill	Rot. Z	Rot. X'	Rot. Z'	Range Z'	Range X'	Range Y'
10	0.150	0.800	2	-50	0	15.0	25.0	20.0
		0.050	4	-33	-3	150.0	250.0	70.0
210	0.500	0.465	-9	49	9	5.9	8.5	20.1
		0.035	-12	-52	6	191.9	292.2	14.3
220	0.200	0.600	16	36	-41	10.0	20.0	60.0
		0.200	-45	-49	33	250.0	150.0	40.0
230	0.250	0.578	-18	19	13	7.4	23.7	29.6
		0.172	28	-43	-20	156.4	392.4	27.6
240	0.350	0.396	-14	48	94	3.2	24.3	16.2
		0.254	13	73	-3	97.8	73.9	7.3
250	0.150	0.689	4	-18	-59	8.5	18.5	11.1
		0.161	63	-23	-43	70.2	330.2	26.1
260	0.400	0.500	5	17	56	7.5	8.4	36.7
		0.100	-83	93	19	154.6	26.8	40.0
270	0.350	0.475	18	11	31	4.5	21.9	28.0
		0.175	36	-49	-24	83.8	177.9	27.8
310 320	0.250	0.600	34	49	32	10.0	10.0	15.0
		0.150	50	-96	71	125.0	200.0	100.0
330 340 350	0.300	0.570	15	-14	-15	10.0	15.0	25.0
		0.130	22	-26	-12	225.0	180.0	40.0

14.5 Block Model Workspace

Two block models were created to cover the known extents of the Tasiast deposit.

Table 14-22 provides the GEMS coordinates of the block model origin for WB and PP. These models were created and mapped to the appropriate subfolders and subsequently combined into a consolidated model using simple manipulation scripts in GEMS.

Table 14-22: Tasiast Block Model Workspace for WB and PP

	Origin				Block Size (m)			Number of Blocks		
Project	X	Y	Z	Rotation	Column	Row	Level	Columns	Rows	Levels
WB	46,010	69,000	145	0	5	5	5	488	665	229
PP	45,710	72,325	150	0	5	5	5	398	1,435	109

In addition to the two block models reported in the above section, KTS created three additional GEMS projects:

- TasiastPiment2012
- TasiastWB_Final
- TasiastPP_Final

The TasiastPiment2012 model was the Gemcom project used for developing the geological and mineralization interpretations. The drill hole workspace Feb2013 includes a geology table with the lithology code used for the interpretation. No block models were updated in this project.

Table 14-23 summarizes the Gemcom projects created for Tasiast and their primary use.

Table 14-23: Gemcom Project Summary and Usage

Gemcom Project	Usage Description
TasiastPiment2012	The entire Tasiast drill hole database was used to construct wireframes.
FS2013_WB	West Branch Resource Model, South of 72325N – was used for compositing, grade interpolation and resource classification.
FS2013_PP	Piment and Prolongation Resource Model, North of 72325N – was used for compositing, grade interpolation and resource classification.
TasiastWB_Final	West Branch Resource Model, South of 73500N * – was used for volumetrics.
TasiastPP_Final	Piment and Prolongation Resource Model, North of 73500N – was used for volumetrics.

Note: * 73,500 North is the location of the “land bridge” used to cross over. No mining is proposed for this area.

TasiastWB_Final and TasiastPP_Final were Gemcom models created relative to the land bridge at Tasiast. The land bridge is located at 73,500N, which is 1,175 m north of the West Branch FS2013 model. This area is used for equipment and services and is the site-based division point for WB and PP. The consolidated block model folder for these two projects was populated from block model export files derived from TasiastFS2013_WB and TasiastFS2013_PP projects.

14.6 Grade Interpolation

The interpolation methods used for populating the block models were Inverse Distance Squared (ID^2), Ordinary Kriging (OK) and Nearest Neighbour (NN).

For the ID^2 and OK interpolation methods, two passes were used. For the first pass, a minimum of 7 composites and a maximum of 18 composites were used. A constraint of three composites per drill holes was imposed so combined with the other composite selection parameters this means that all blocks were estimated by a minimum of two drill holes. For the second pass a minimum of 4 composites were used with the other composite selection parameters maintained the same.

For NN, one pass was used which reflected the second pass interpolation parameters use for ID^2.

Table 14-24 and Table 14-25 outline the parameters used for the search ellipses for WB and PP.

Table 14-24: WB Search Ellipse Profiles

Search Ellipse	Search Anisotropy	Principal Azimuth	Principal Dip	Inter-Mediate Azimuth	Aniso-tropy X	Aniso-tropy Y	Aniso-tropy Z	Search Type	Min. Octants	Max. Samples Per Octant
NNP2	Azimuth, Dip, Azimuth	80	40	0	50	150	150	Ellip-soidal	NA	NA
OK1	Azimuth, Dip, Azimuth	80	40	0	50	100	100	Octant	4	3
OK2	Azimuth, Dip, Azimuth	80	40	0	50	150	150	Octant	2	2
1OK90-45	Azimuth, Dip, Azimuth	90	45	0	20	100	100	Octant	4	3
2OK90-45	Azimuth, Dip, Azimuth	90	45	0	30	150	150	Octant	2	2

Table 14-25: PP Search Ellipse Profiles

Search Ellipse	Search Anisotropy	Principal Azimuth	Principal Dip	Inter-Mediate Azimuth	Aniso-tropy X	Aniso-tropy Y	Aniso-tropy Z	Search Type	Min. Octants	Max. Samples Per Octant
NN80-25	Azimuth, Dip, Azimuth	80	25	0	30	150	150	Ellip-soidal	NA	NA
NN80-45	Azimuth, Dip, Azimuth	80	45	0	30	150	150	Ellip-soidal	NA	NA
NN85-25	Azimuth, Dip, Azimuth	85	25	0	30	150	150	Ellip-soidal	NA	NA
NN90-45	Azimuth, Dip, Azimuth	90	45	0	30	150	150	Ellip-soidal	NA	NA
NN90-55	Azimuth, Dip, Azimuth	90	55	0	30	150	150	Ellip-soidal	NA	NA
1OK80-25	Azimuth, Dip, Azimuth	80	25	0	20	100	100	Octant	4	3
2OK80-25	Azimuth, Dip, Azimuth	80	25	0	30	150	150	Octant	2	2
1OK80-45	Azimuth, Dip, Azimuth	80	45	0	20	100	100	Octant	4	3
2OK80-45	Azimuth, Dip, Azimuth	80	45	0	30	150	150	Octant	2	2
1OK85-25	Azimuth, Dip, Azimuth	85	25	0	20	100	100	Octant	4	3
2OK85-25	Azimuth, Dip, Azimuth	85	25	0	30	150	150	Octant	2	2
1OK90-45	Azimuth, Dip, Azimuth	90	45	0	20	100	100	Octant	4	3
2OK90-45	Azimuth, Dip, Azimuth	90	45	0	30	150	150	Octant	2	2
1OK90-55	Azimuth, Dip, Azimuth	90	55	0	20	100	100	Octant	4	3
2OK90-55	Azimuth, Dip, Azimuth	90	55	0	30	150	150	Octant	2	2

Visual Confirmation of Search Ellipse

Figure 14-16 illustrates the OK1 search ellipse used for DOM2 mineralized domain. The composites and block grades are also displayed. Figure 14-7 illustrates the 2OK80-45 search ellipse for the PCWEST mineralized domain.

Figure 14-6: WB Search Ellipse OK1

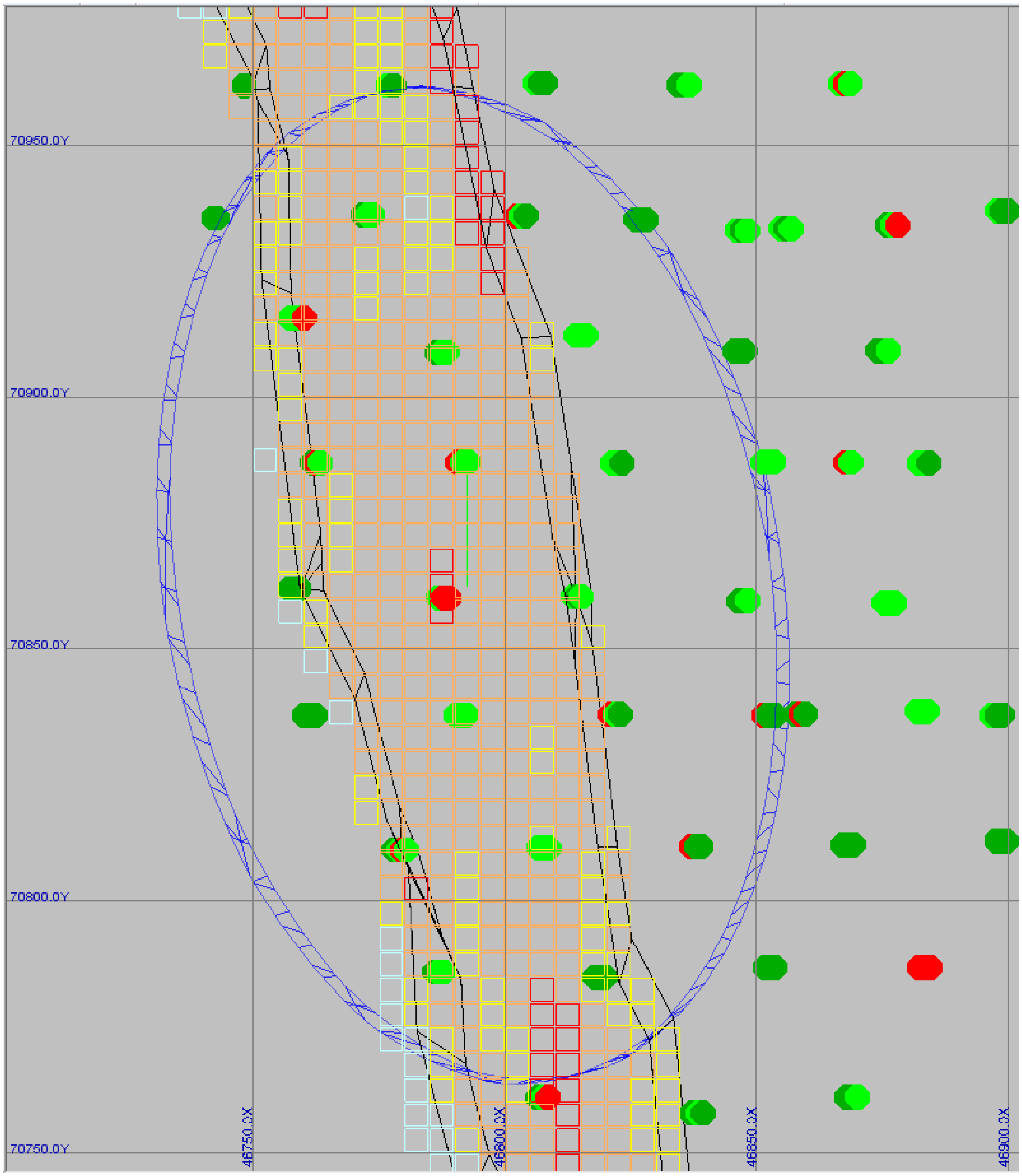
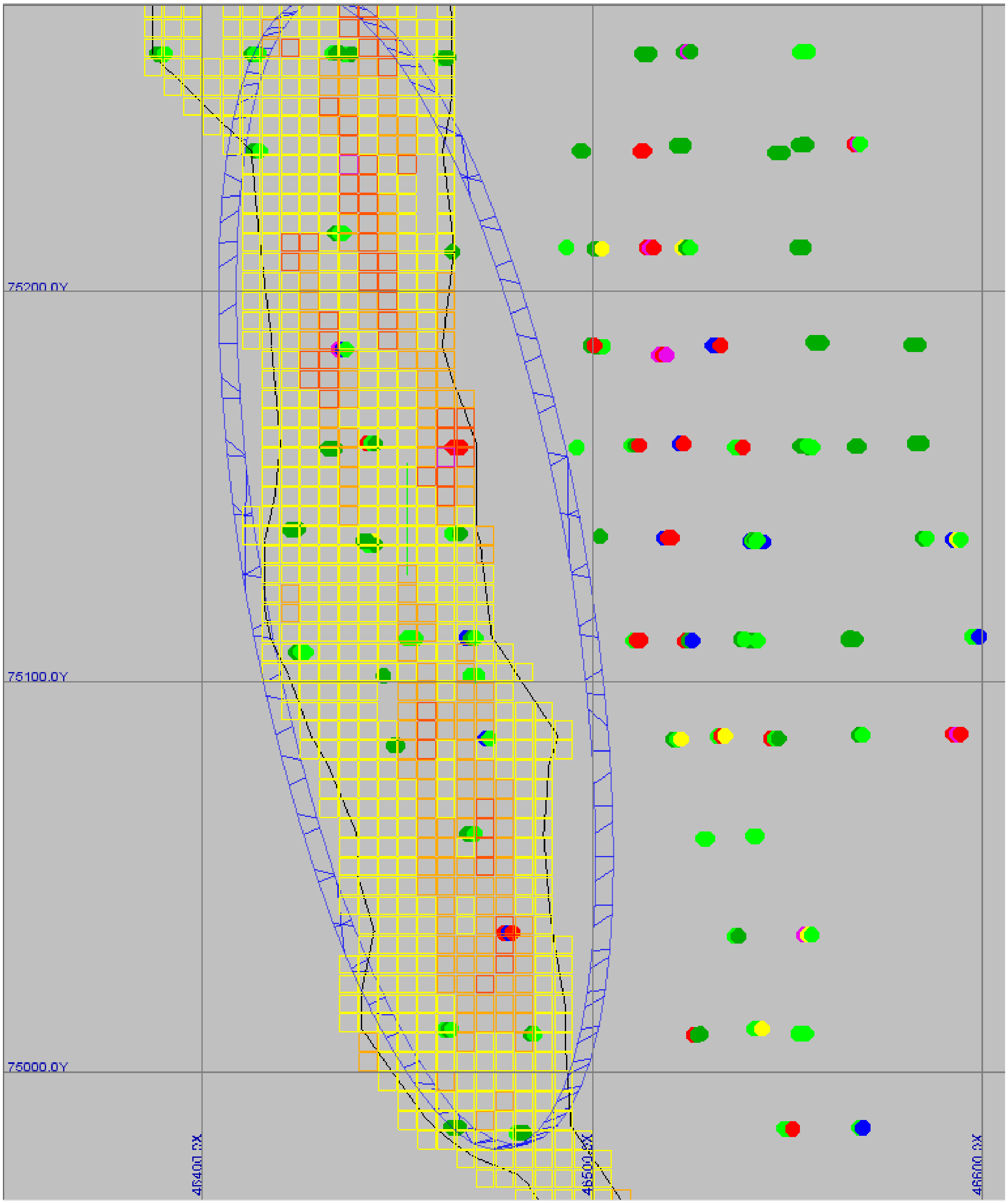


Figure 14-7: PP Search Ellipse 2OK80-45



14.7 Bulk Density Modelling

The density values used in the 2011 MIK model were compared with the data currently available. For the 2011 MIK model, 24,702 data values were used to assign density values based on the metallurgical zone. The metallurgical zone was referenced by rock type and state of oxidation.

There were 26,940 density records used to develop the new density values for the 2013 model. Table 14-26 summarizes the density analysis and new values to be assigned. The 2013 model used the same methodology of combining the rock type (or lithology) and state of oxidation to determine the appropriate density value. These values were assigned to the block model based on these two values. If no data was available, the 2011 model density values were still used in the 2013 model.

Using specific gravity measurements taken on the various rock types, an average density value was calculated for each logged rock type and oxide type. These values were then assigned to the domains in the block model using simple manipulation scripts in Gemcom

The differences in density values resulted from a change in geological interpretation based on relogging and updated lithological wireframes.

Table 14-26: Density Values Assigned to Models - g/cm^3

			FS2013 Calculations		2011 MIK Calculations		FS2013 Values Used	
Code	Lith	Oxide	PP Density {value(count)}	WB Density {value(count)}	PP Density {value(count)}	WB Density {value(count)}	PP Density {value(count)}	WB Density {value(count)}
10	DYKE	oxide	-	-	2.79	2.79	2.79	2.79
10		upper transition	-	-	2.84	2.84	2.84	2.84
		lower transition	-	2.67 (2)	2.91	2.91	2.89 (77)	2.89 (77)
		fresh	2.89 (77)	2.96 (394)	2.97	2.97	2.89 (77)	2.96 (394)
20	RSPRLT	oxide	-	n/a	n/a	n/a	2.54 (70)	n/a
20		upper transition	-	n/a	n/a	n/a	2.72 (49)	n/a
		lower transition	-	n/a	n/a	n/a	2.83 (73)	n/a
		fresh	-	n/a	n/a	n/a	n/a	n/a
30	GST	oxide	n/a	2.64 (1)	2.3	2.3	n/a	n/a
30		upper transition	n/a	2.87 (5)	2.59	2.59	n/a	n/a
		lower transition	n/a	2.92 (7)	2.74	2.74	n/a	n/a
		fresh	n/a	2.87 (3,105)	2.85	2.85	n/a	2.87 (3,105)
40	BIMHW	oxide	2.37 (26)	2.82 (47)	2.78	2.66	2.37 (26)	2.82 (47)
40		upper transition	2.88 (12)	2.86 (24)	2.9	2.96	2.88 (12)	2.86 (24)
		lower transition	2.88 (11)	2.94 (45)	2.91	3.14	2.88 (11)	2.94 (45)
		fresh	3.26 (969)	3.20 (5,964)	3.07	3.26	3.26 (969)	3.20 (5,964)
45	BIMFW	oxide	-	-	2.52	2.52	2.52	2.52
45		upper transition	-	-	2.77	2.77	2.77	2.77
		lower transition	2.79 (3)	-	2.85	2.85	2.85	2.85
		fresh	3.02 (70)	3.12 (311)	3.04	3.04	3.02 (70)	3.12 (311)
50	FVC	oxide	-	2.61 (3)	2.19	2.19	2.19	2.19
50		upper transition	2.7 (3)	2.57 (3)	2.57	2.57	2.57	2.57
		lower transition	2.64 (10)	2.77 (5)	2.64	2.64	2.64 (10)	2.64
		fresh	2.72 (765)	2.69 (4,866)	2.68	2.68	2.72 (765)	2.69 (4,866)
60	SVC	oxide	2.54 (70)	2.41 (90)	2.45	2.45	2.54 (70)	2.41 (90)
60		upper transition	2.72 (49)	2.61 (56)	2.57	2.57	2.72 (49)	2.61 (56)
		lower transition	2.83 (73)	2.70 (81)	2.69	2.69	2.81	2.70 (81)
		fresh	2.81 (1,686)	2.77 (5,675)	2.76	2.76	2.81 (1686)	2.77 (5,675)

14.8 Mineral Resource Classification

The Mineral Resources are classified under the categories of Measured, Indicated and Inferred, in accordance with CIM Definition Standards. Classification of the resources reflects confidence of grade continuity, as a function of many factors, including primarily, assay data quality, QA/QC procedures, quality of density data, and sample spacing relative to geological and geostatistical observations regarding the continuity of mineralization.

The resource model blocks were classified into Measured, Indicated and Inferred categories, based on the level of confidence in the grade estimate for each block. Measured resources were defined based on a nominal drill spacing of 25 m. Indicated resources were categorized based on a nominal drill spacing of 50 m, and the remaining interpolated blocks within 120 m of the nearest composite were classified as inferred resources. Blocks estimated using data beyond 120 m were not classified as mineral resources. A minimum of two drill holes were used to estimate all blocks. The interpolation pass was not used for developing the resource classification. The selection of the nominal spacing was based on a combination of the review of the variogram structures (which typically demonstrated short range for the first structure within 10 m to 30 m), the prior resource classification and the qualified person's experience. The classification was subsequently confirmed by reviewing the special model statistics reported below (Table 14-27).

Solids were constructed in Leapfrog using an isotropic distance interpolation around each drill hole. Then, the solids were contoured around contiguous solids and 3-D wireframes were constructed in Leapfrog for use in classification. Areas that demonstrated discontinuity and where the solids were not contiguous were excluded from the wireframe.

Blocks were assigned preliminary confidence categories, based on the Leapfrog wireframes. Special models were populated during the OK interpolation to assist with resource classification as follows:

- Actual distance to closest point (DISTN)
- Number of points used for the estimate (NCOMP)
- Number of octants (NOCTS)
- Mean distance for samples used (DISTA)
- Number of holes used (NDDH)

These special models were used to refine the initial block classification from the Leapfrog wireframes. Table 14-27 summarizes the average values for these special models.

Table 14-27: Special Model Average Values

Area	Model	Measured	Indicated	Inferred
WB	DISTN	11.4 m	22.1 m	45.8 m
	NCOMP	14	13	9
	NOCTS	5.5	5.1	4.2
	DISTA	26.3 m	45.1 m	73.7 m
	NDDH	6	5	4
PP	DISTN	10.5 m	19.8 m	56.1 m
	NCOMP	14	13	6
	NOCTS	5.6	5.0	3.2
	DISTA	25.1 m	40.9 m	77.5 m
	NDDH	6	5	3

Figure 14-8 illustrates the resource classification code (RCLASS) for 95 m elevation in the WB model. Figure 14-9 illustrates RCLASS for the same elevation in PP. The fair value pit outline is shown for reference.

The block model codes used for the resource classification model, RCLASS, are summarized in Table 14-28.

Table 14-28: Resource Classification Codes

Code	Description
1000	Measured resource
2000	Indicated resource
3000	Inferred resource
4000	Unclassified resource (within mineralized domain)
0	Unclassified (not within mineralized domain)

Figure 14-8: WB Resource Classification, 95 m Elevation

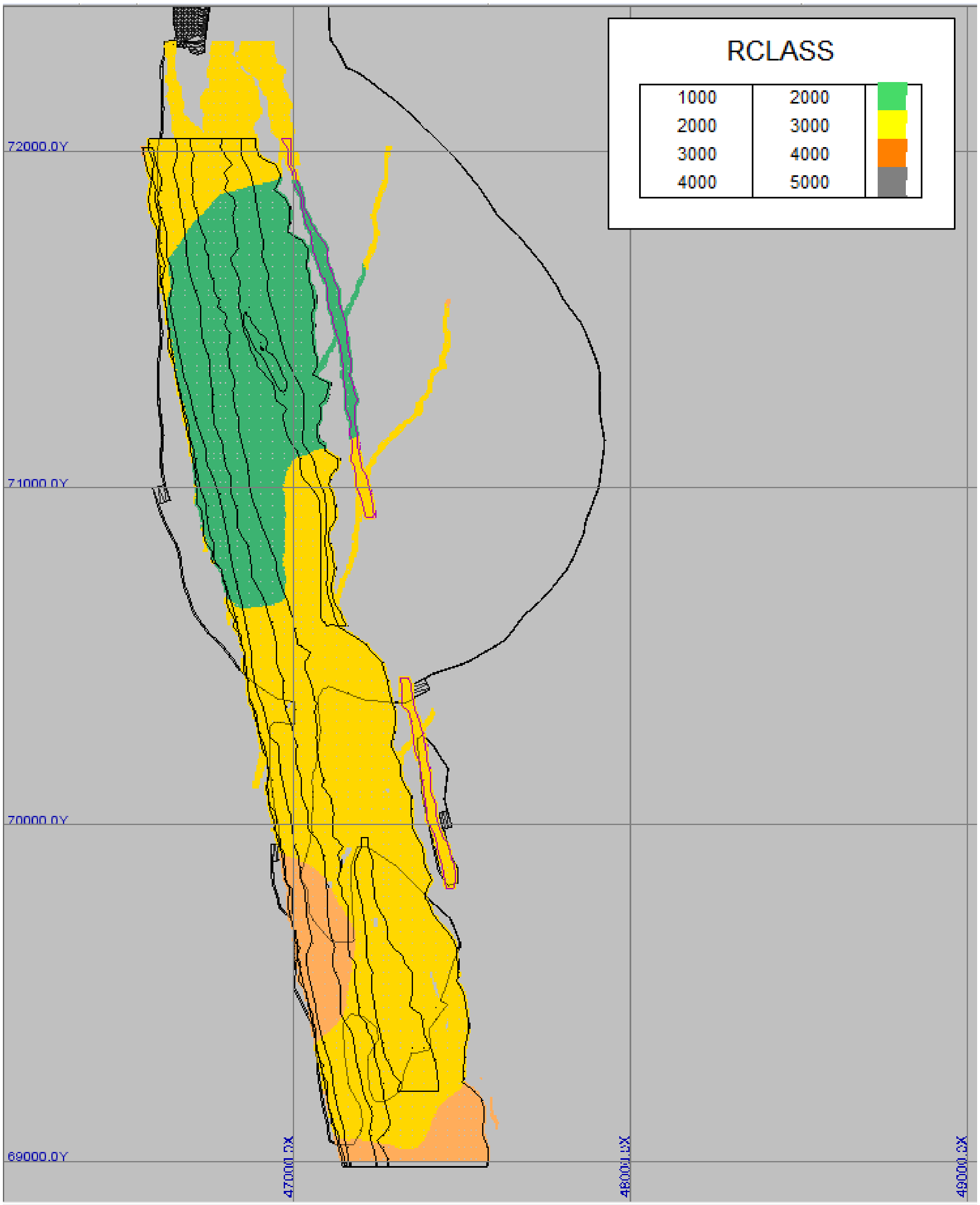
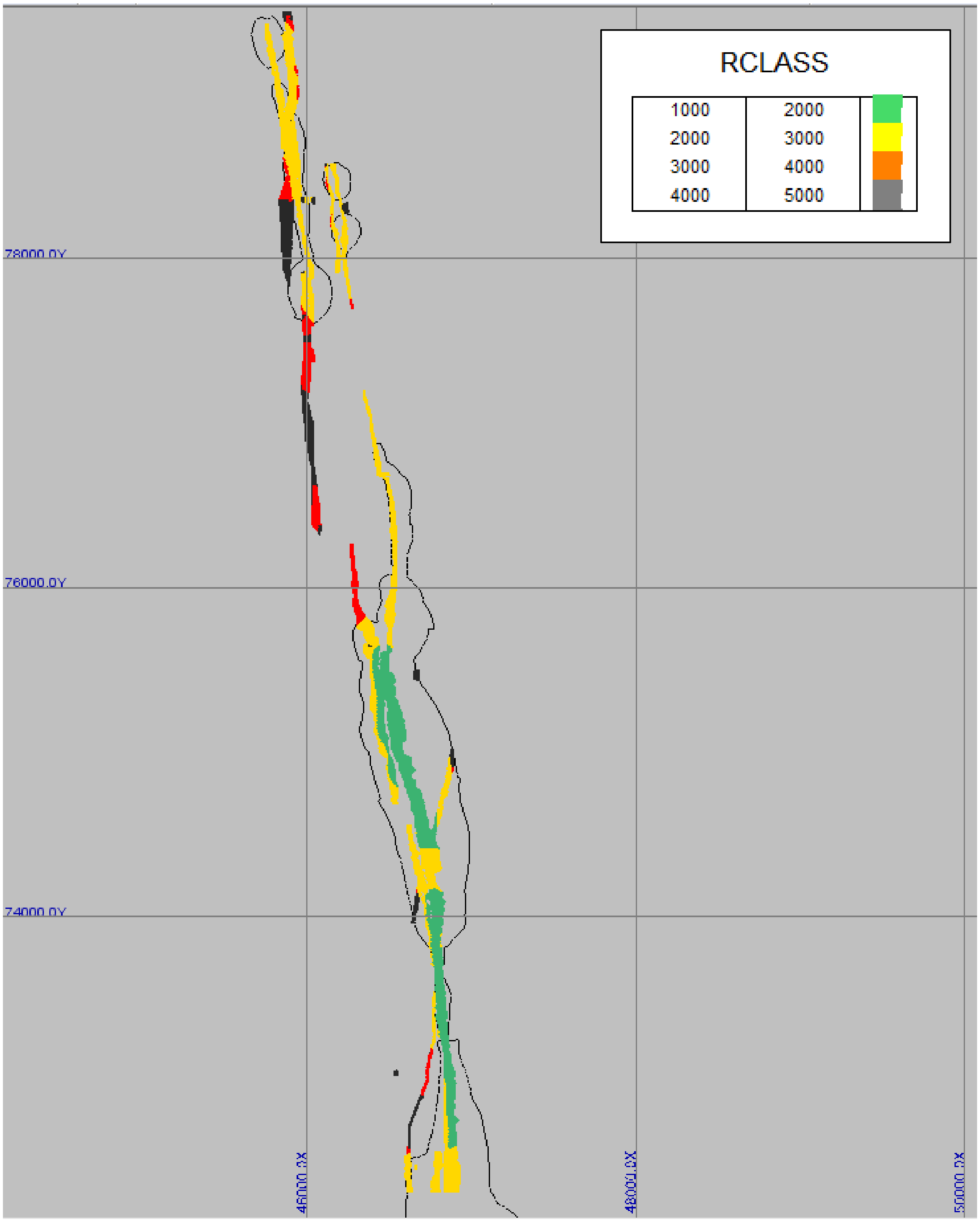


Figure 14-9: PP Resource Classification, 95 m Elevation



Grade Tonnage Curves

Grade tonnage curves were produced from the 2013 resource models to examine the sensitivity of tonnage to grade changes in the deposit. The grade tonnage curves are shown in Figure 14-10 for WB and Figure 14-11 PP.

Block Model Metal Loss from Capping

The amount of metal that was removed from the Au block model as a result of capping for Measured+Indicated (Mea+Ind) and Inferred (Inf) category is shown in Table 14-29 through Table 14-32. The metal reported was constrained within the 2013 year-end surface and 2013 $1400USD resource pit shell. The capped grade model (AUCK) was the basis for the reported tonnage. The capped and uncapped grade models reported were AUCK and AUK.

Table 14-29: WB Metal Loss from Capping, Mea+Ind

Mea+Ind Capped			Mea+Ind Uncapped		Metal Change
Tonnes (k)	Grade (g/t)	Ounces (k)	Grade (g/t)	Ounces (k)	Delta Au (%)
270,251	1.360	11,813	1.384	12,028	0.8

Table 14-30: WB Metal Loss from Capping, Inf

Inf Capped			Inf Uncapped		Metal Change
Tonnes (k)	Grade (g/t)	Ounces (k)	Grade (g/t)	Ounces (k)	Delta Au (%)
10,971	1.395	492	1.410	497	-1.1

Table 14-31: PP Metal Loss from Capping, Mea+Ind

Mea+Ind Capped			Mea+Ind Uncapped		Metal Change
Tonnes (k)	Grade (g/t)	Ounces (k)	Grade (g/t)	Ounces (k)	Delta Au (%)
57,610	1.226	2,271	1.274	2,360	-3.9

Table 14-32: PP Metal Loss from Capping, Inf

Inf Capped			Inf Uncapped		Metal Change
Tonnes (k)	Grade (g/t)	Ounces (k)	Grade (g/t)	Ounces (k)	Delta Au (%)
3,175	1.684	172	1.862	190	-10.5

Figure 14-10: WB Grade Tonnage Curve

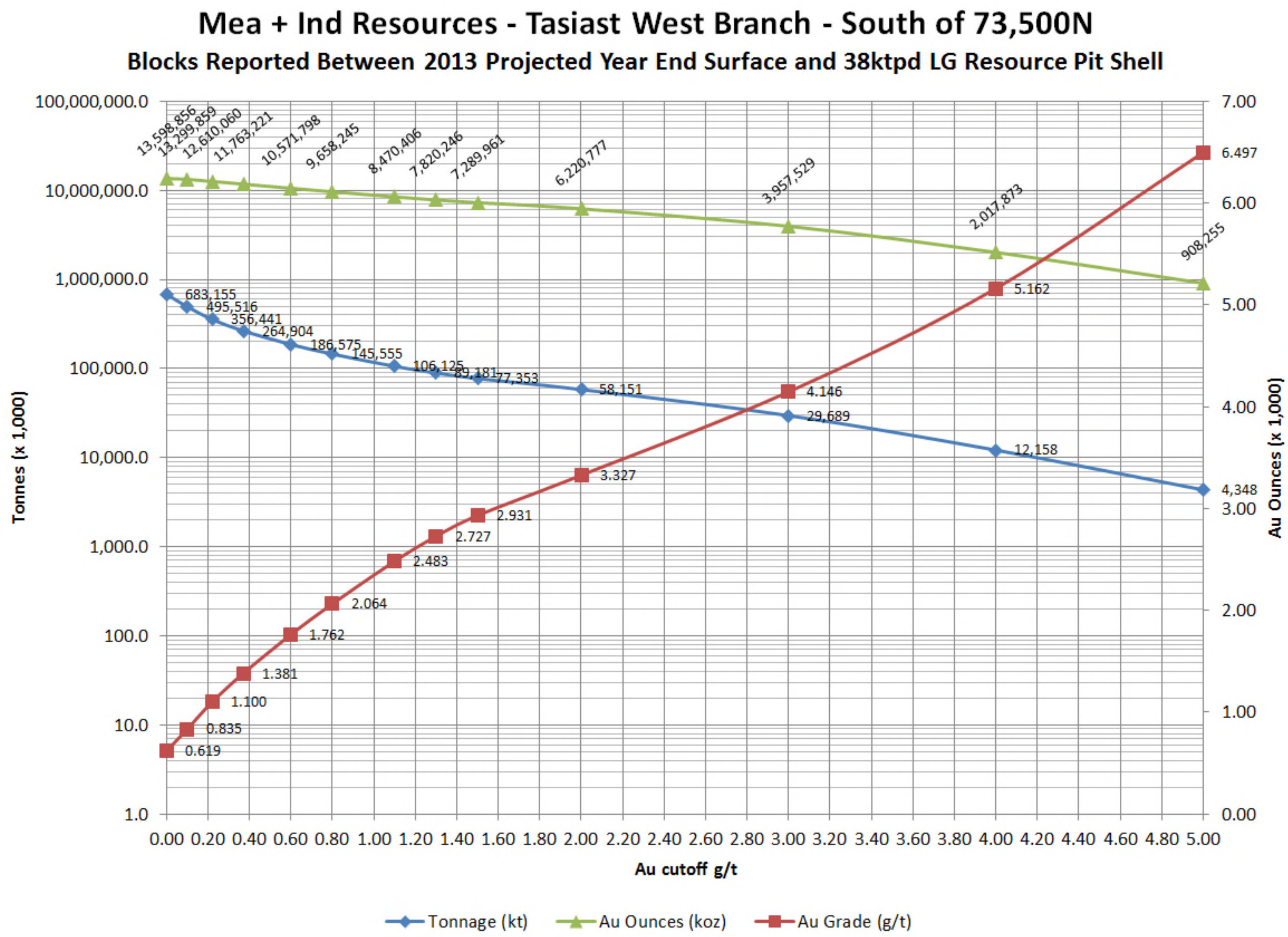
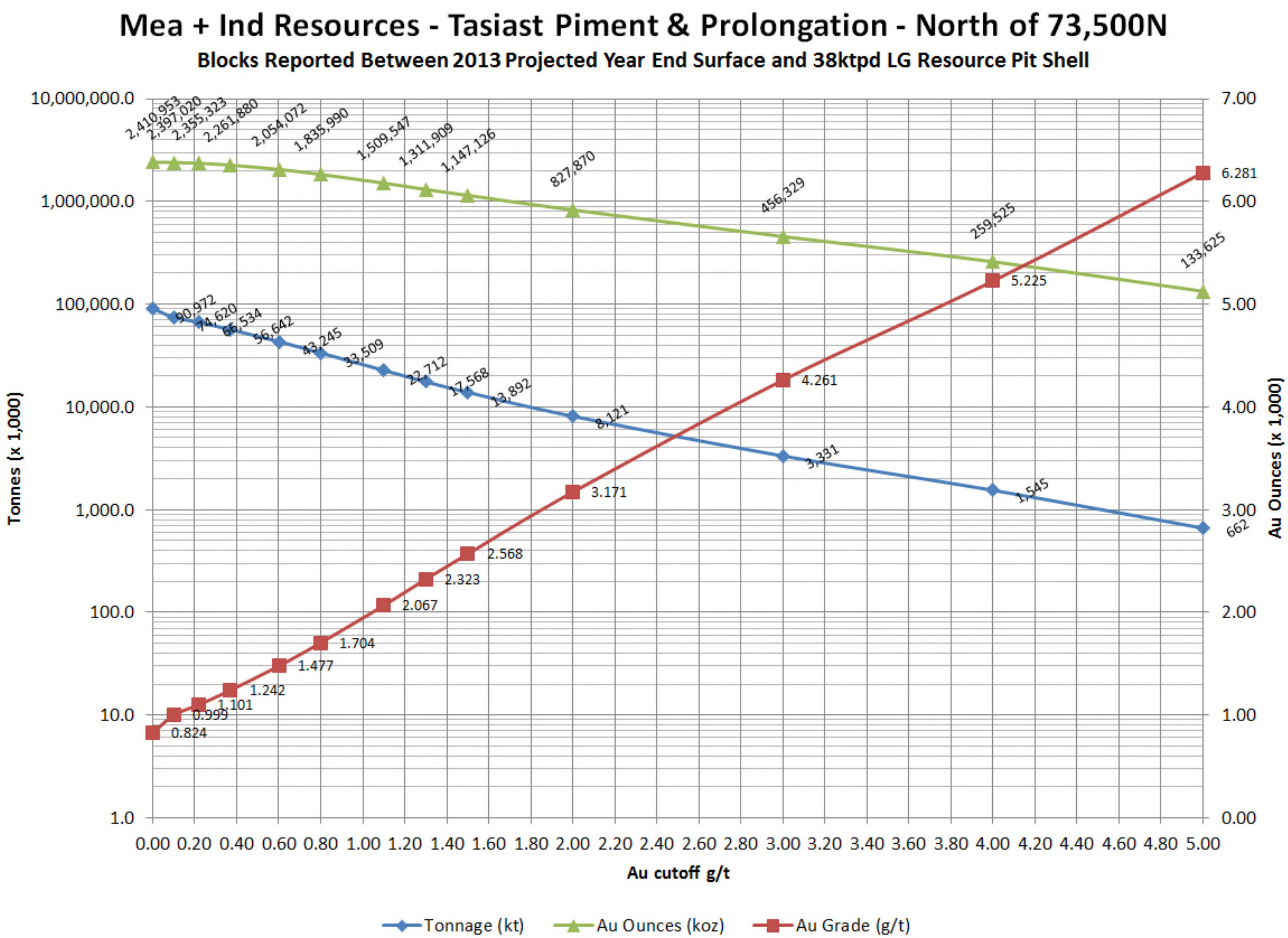


Figure 14-11: PP Grade Tonnage Curve



Consolidated Model

The Tasiast GEMS block model is a percent type with folders for each mineralized domain. Based on present weighted tonnage and grade, these subfolders were consolidated into a single folder named: CONSOLIDATED. The consolidated model was created to facilitate exporting to other mining software packages and reviewing within Gemcom. The consolidated model assigns the majority codes, such as DOM or ROCK TYPE or DENSITY, to the block. The grade was volume weighted as the density is the same for each mineralization domain. Simple manipulation scripts were used to populate block models grades in the consolidated model folder from the individual mineralized domain folders

14.9 Block Model Validation

The Tasiast FS2013 model was validated by:

1. Comparison of the global mean block grades for ID^2, OK, NN and composites
2. Visual comparison of colour-coded block model grades with composites on section and in plan
3. Swath plots along the row, columns and levels of the block model
4. Comparison of OK and NN block grades with Herco distribution.

Interpolation Validation—Global Comparison

The block model estimates were verified for global bias by comparing the average Au grade (with no cutoff value) for Measured, Indicated and Inferred resource category blocks from the model (OK) using ID^2 and NN estimates. The NN estimator produces a theoretical unbiased estimate of the average value when no cutoff grade is imposed, and is a good method for checking the performance of different estimation methods. The results show no evidence of bias in the estimate (Table 14-33).

Table 14-33: Comparison of Mean Au Grade for OK, ID^2 and NN Interpolations

AREA	AUNN (g/t)	AUCNN (g/t)	AUID (g/t)	AUCID (g/t)	AUK (g/t)	AUCK (g/t)
WB	0.393	0.380	0.390	0.380	0.390	0.379
PP	0.565	0.550	0.604	0.587	0.610	0.592

Table 14-34 and Table 14-35 summarize the composite versus block model statistics for WB and PP.

Table 14-34: Comparison of WB Statistics for Capped Composites and Model Grades

Statistic	Composite AU_CAP	Block Model AUCK
Count	134,471	5,954,679
Mean (g/t)	0.518	0.379
Standard deviation	1.46	0.72
Sample variance	2.14	0.51
Coefficient of variance	2.82	1.89

Table 14-35: Comparison of PP Statistics for Capped Composites and Model Grades

Statistic	Composite AU_CAP	Block Model AUCK
Count	49,940	1,091,644
Mean (g/t)	0.839	0.592
Standard deviation	2.11	0.86
Sample variance	4.44	0.73
Coefficient of variance	2.51	1.44

Sectional Validation—Blocks versus Composites

Interpolated block grades, resource classification, geological interpretation outlines and drill hole composite intersections were verified on screen in plan and in vertical section. Based on visual inspection, the block model grades appear to honour the data well. The estimated grades exhibit a satisfactory consistency with the drill hole composites (see Figure 14-12 for WB and Figure 14-13 for PP).

Figure 14-12: WB Section 71511N, Looking North

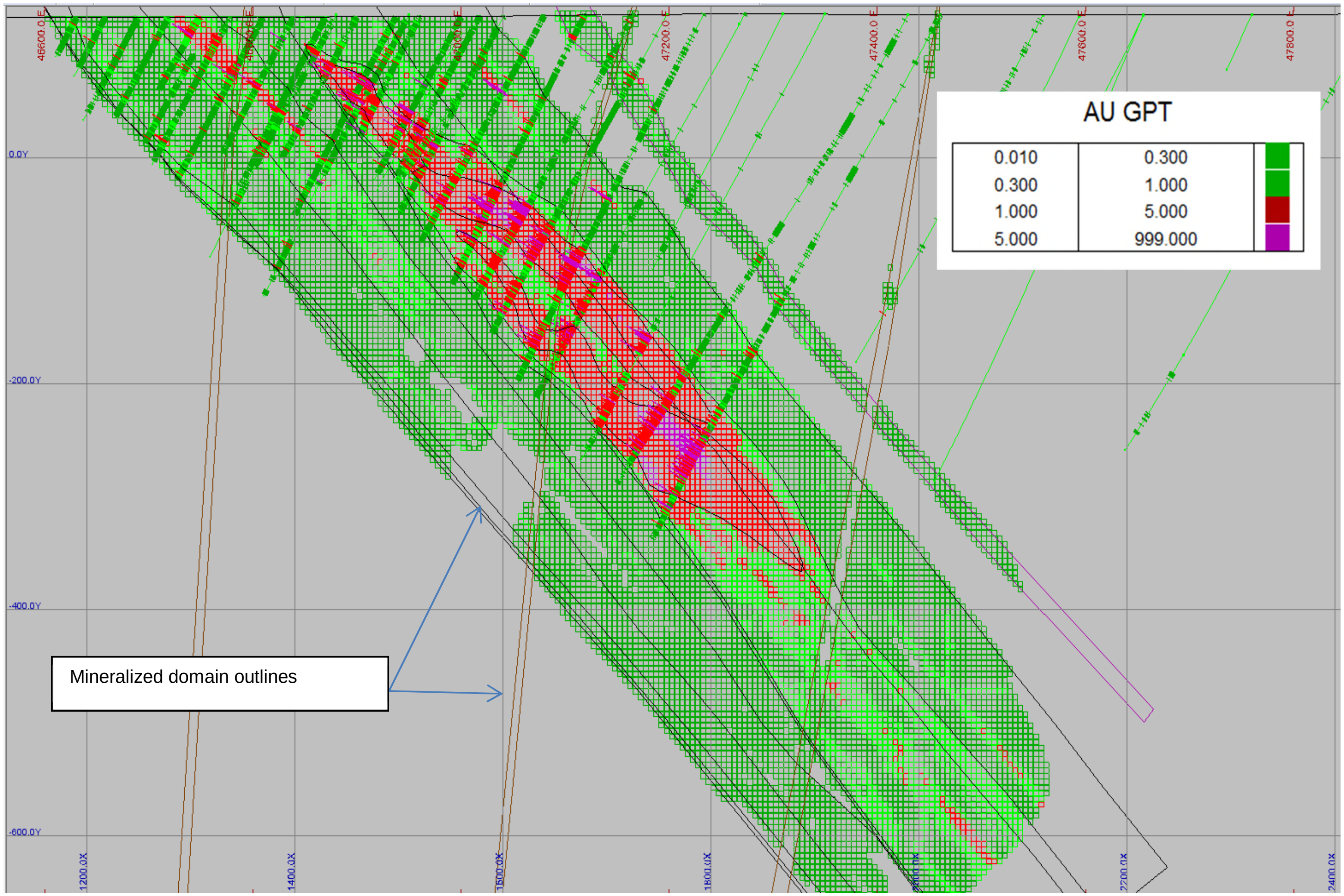
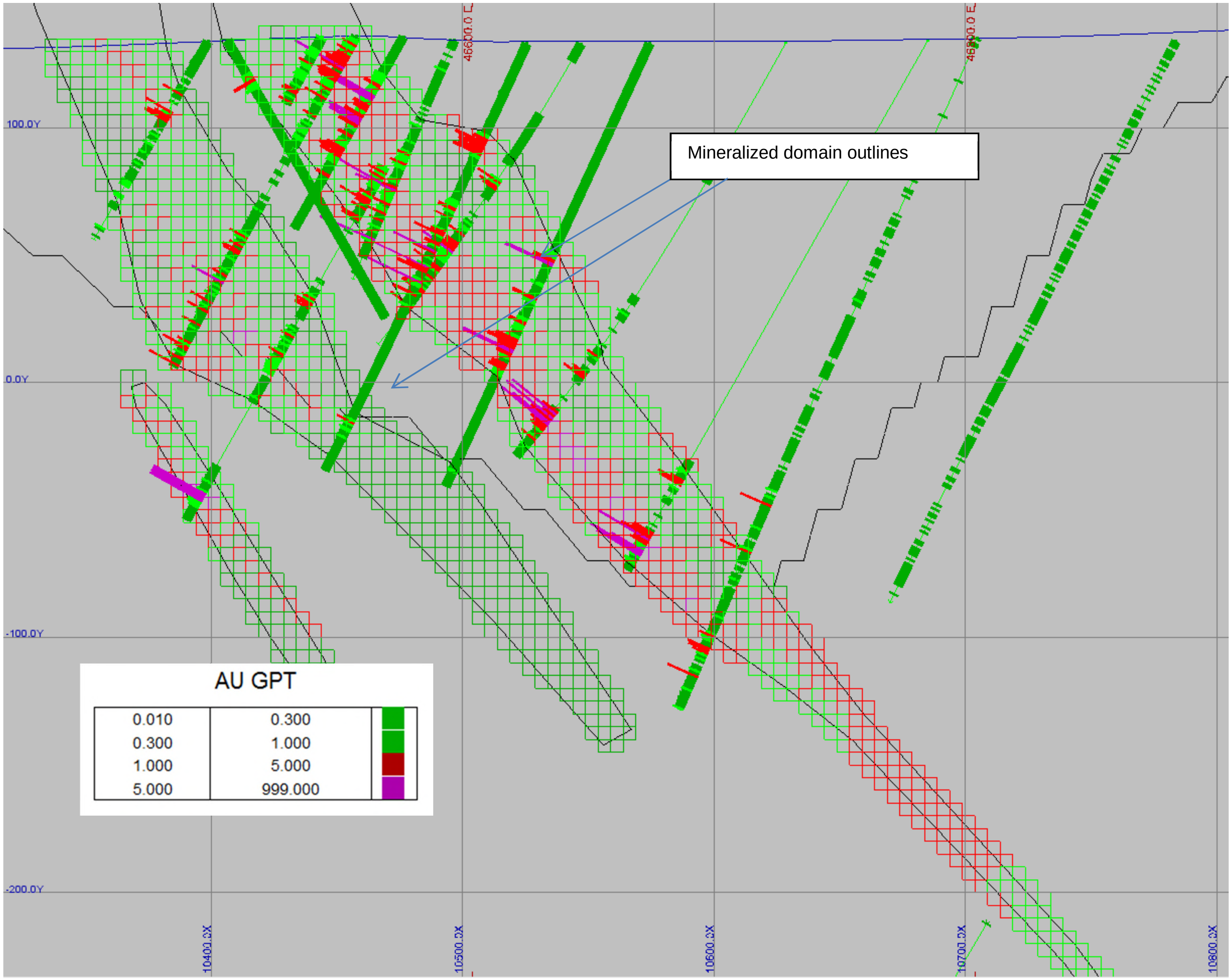


Figure 14-13: PP Section 75036N, Looking North



Swath Plots

Swath plots are commonly used as a block model validation tool as it provides a graphical comparison of key modeling outputs such as the number of composites, composite grades, raw grades, and block grades for various interpolation methods.

The graphical results are based on sectional slices or “swaths” through the block model space on a chosen corridor width. Within a given swath, each data type is averaged and plotted on a graph by easting, northing, and elevation. This provides a spatial comparison of the data so that estimation or compositing issues can be flagged and reviewed in 3D.

Swath plots for WB and PP, by northing, easting, and elevation are shown in Figure 14-14 through Figure 14-19. These swath plots were generated to compare results from the three estimation methods, and do not include the number of composites.

Generally, swath plots report the highest average grade for models estimated by NN. The OK grade models are usually more smoothed and report a lower average grade.

For WB, the OK and ID^2 models correlate well. In areas of lower data density (east of 47,500E and below -450 m elevation), the OK and ID^2 models report lower average grades than the NN model.

For PP, the grade curves appear more closely correlated in the swath plots by easting and northing. However, the grade by elevation swaths report OK and ID^2 models at higher average grades than the NN model in localized areas, which might reflect data density.

Figure 14-14: WB Swath Plot by Northing

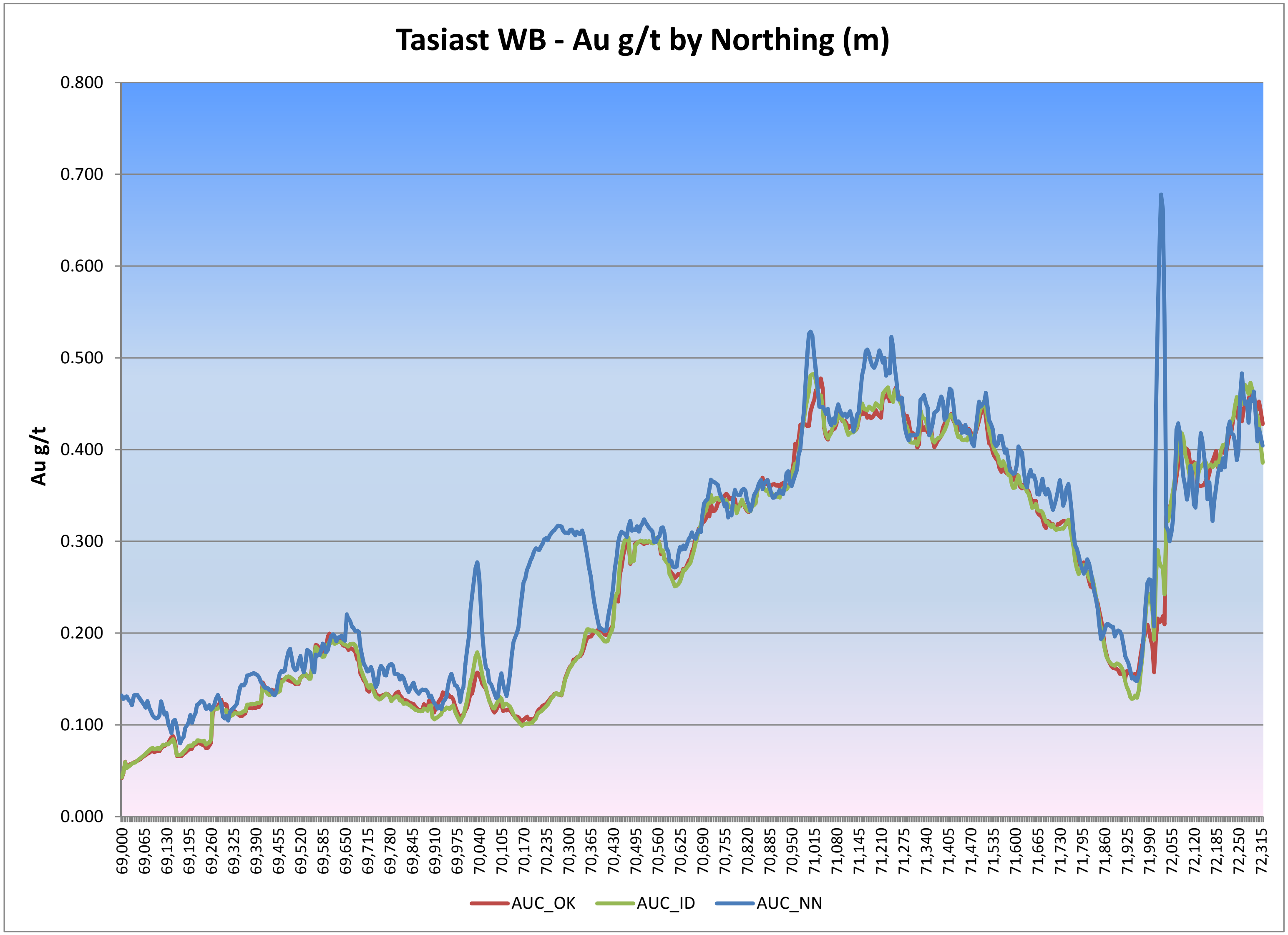


Figure 14-15: WB Swath Plot by Easting

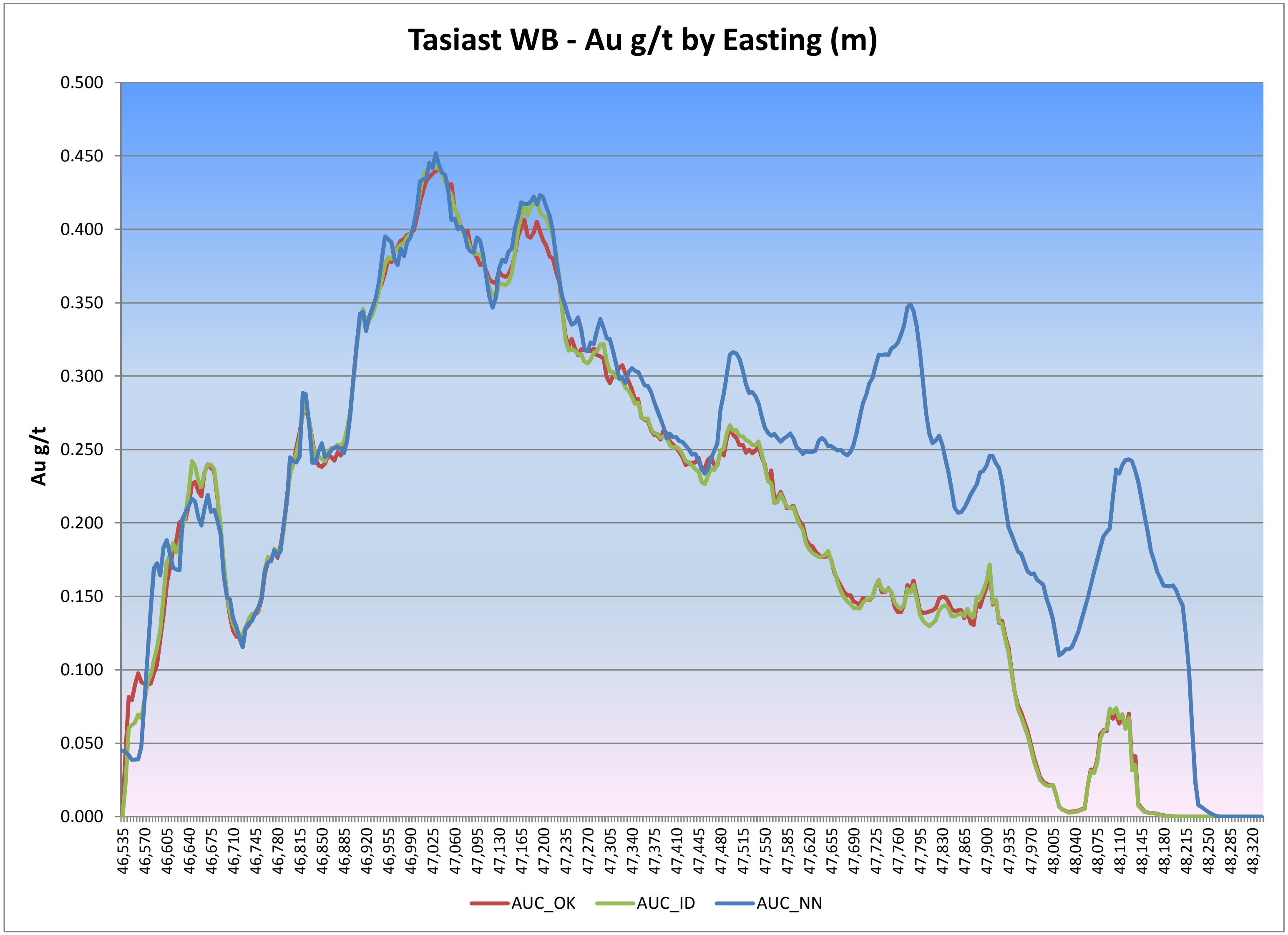


Figure 14-16: WB Swath Plot by Elevation

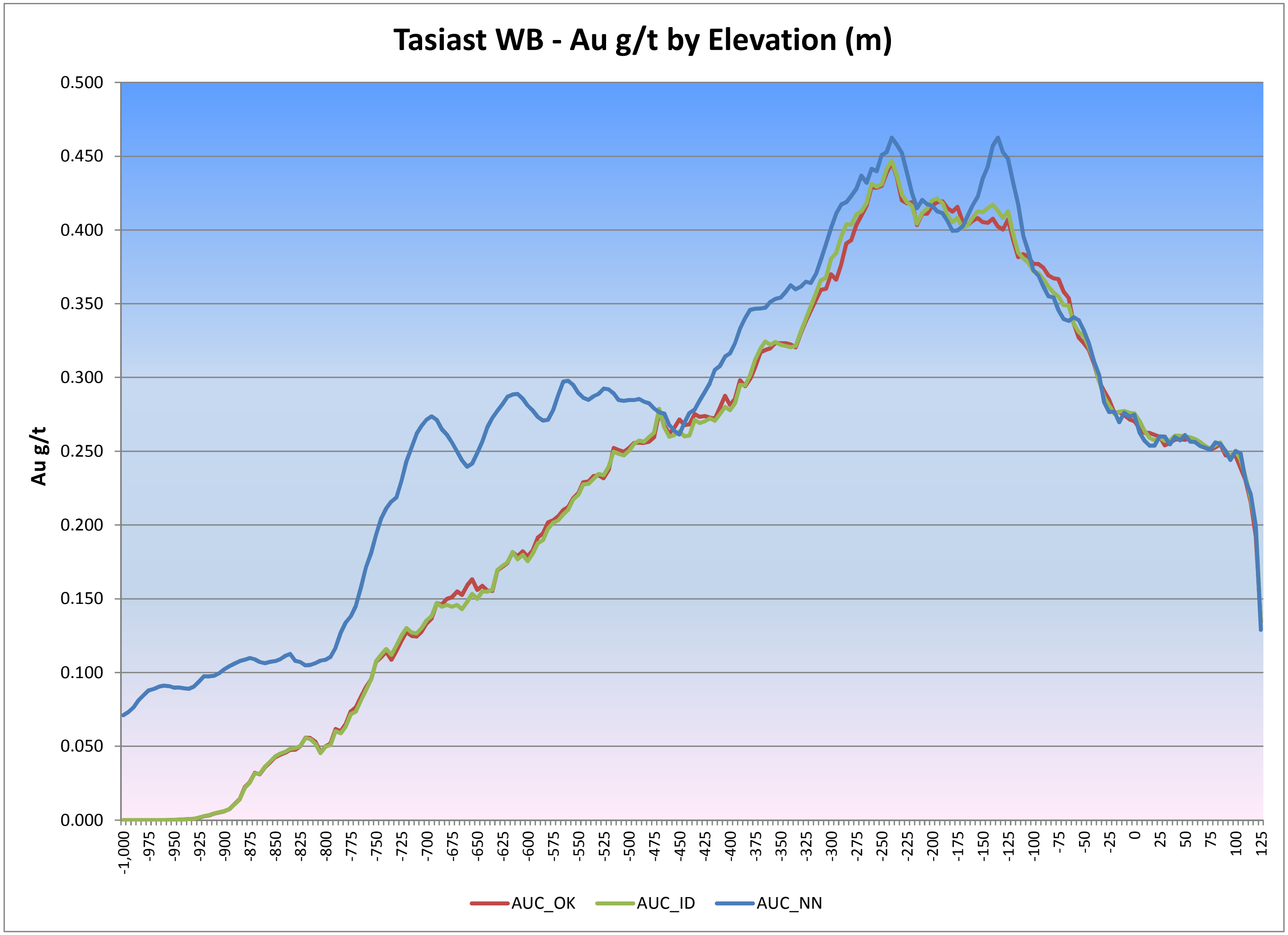


Figure 14-17: PP Swath Plot by Northing

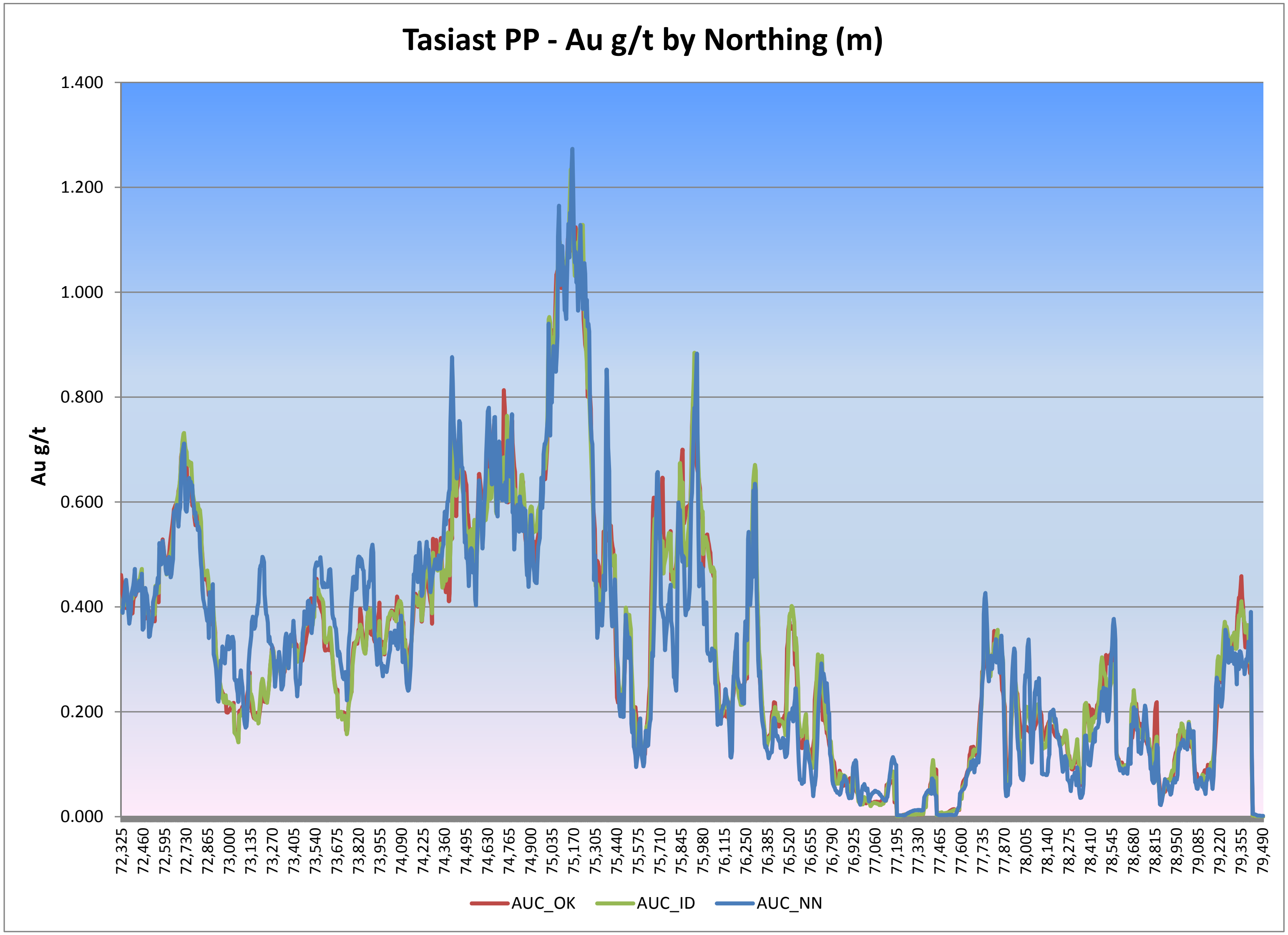


Figure 14-18: PP Swath Plot by Easting

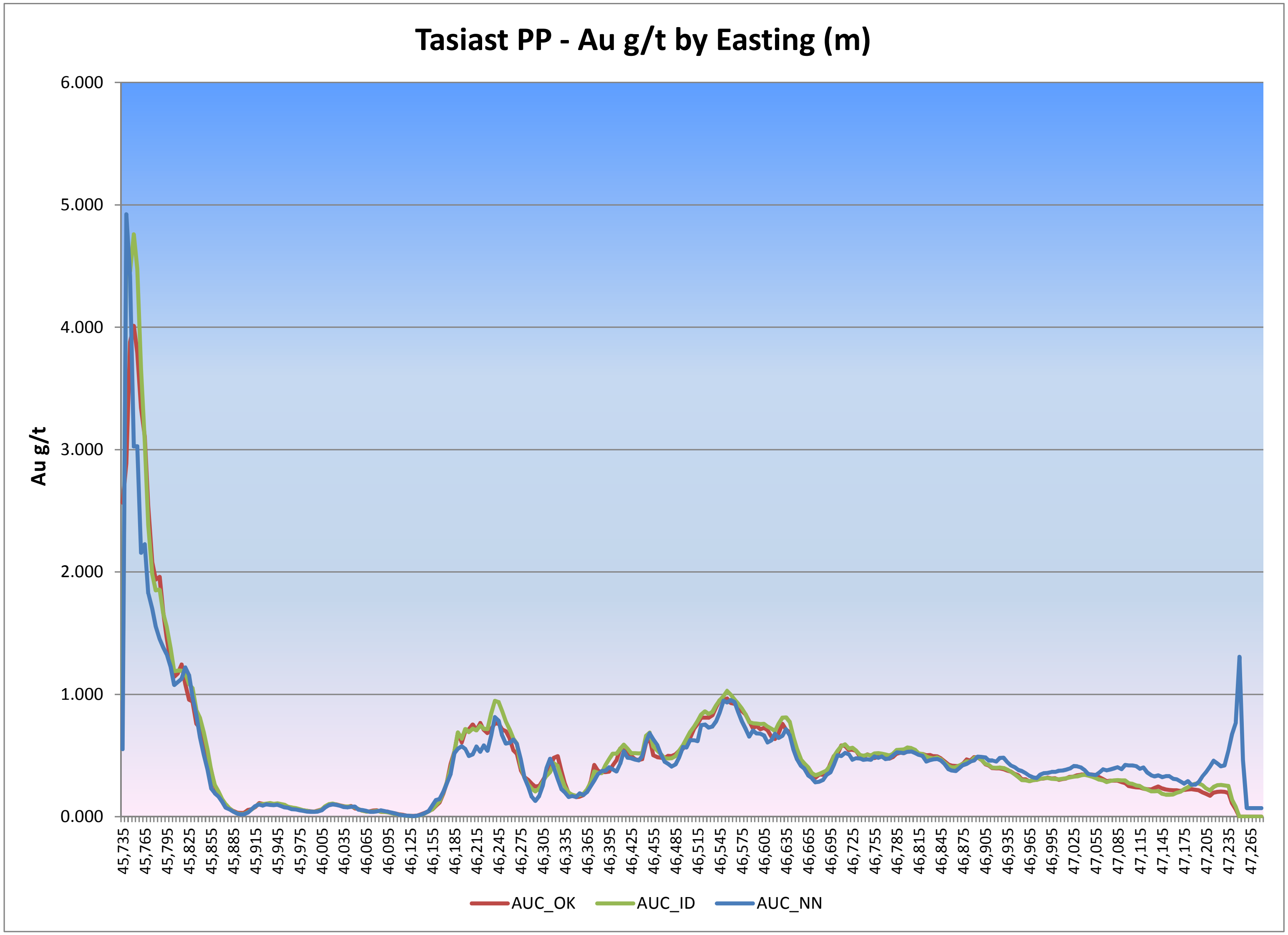
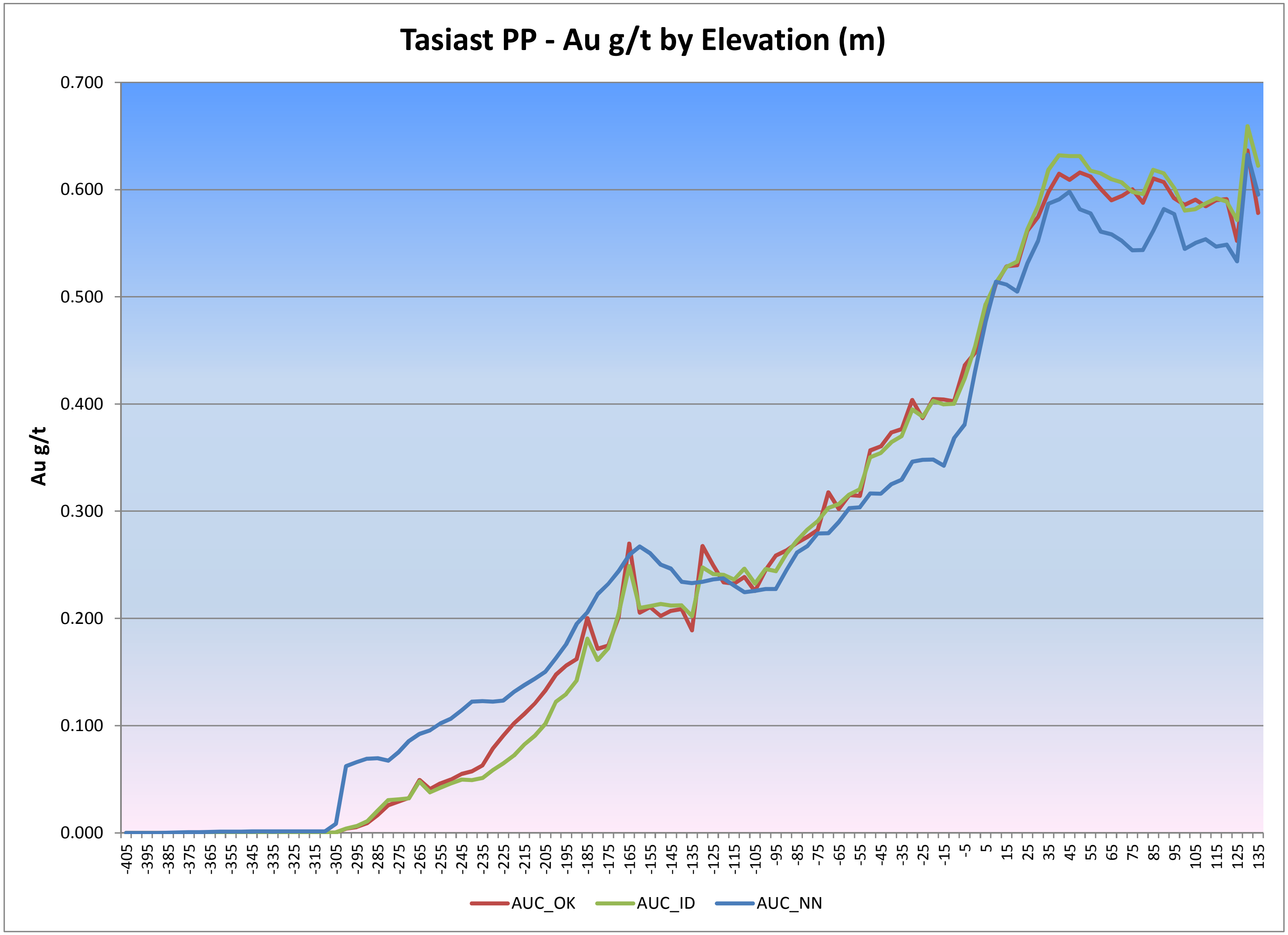


Figure 14-19: PP Swath Plot by Elevation



Swath Plots—GST

Additional swath plots (see Figure 14-20 and Figure 14-21) were created to confirm the interpolation in the area of the GST in the West Branch model. This is a higher grade gold zone in WB. The domain DOM6_1 [Code 110] is based on a 2 g/t Au grade shell and domain DOM6_2 [Code 120] is the surrounding lower grade halo based on 0.1 g/t Au cutoff.

In addition to the OK, ID2 and NN grades, these swath plots include the average grade and composite count as an indication of the data density.

These swath plots illustrate that in areas with good data density, the correlation between composite and block model Au grades is high. In areas along the edges with lower data density, the variability between grade interpolation methods is greater.

These swath plots confirm that within the GST unit at WB, the resource model interpolation performed well when compared to the composite grades.

Figure 14-20: DOM6_1 Swath Plot

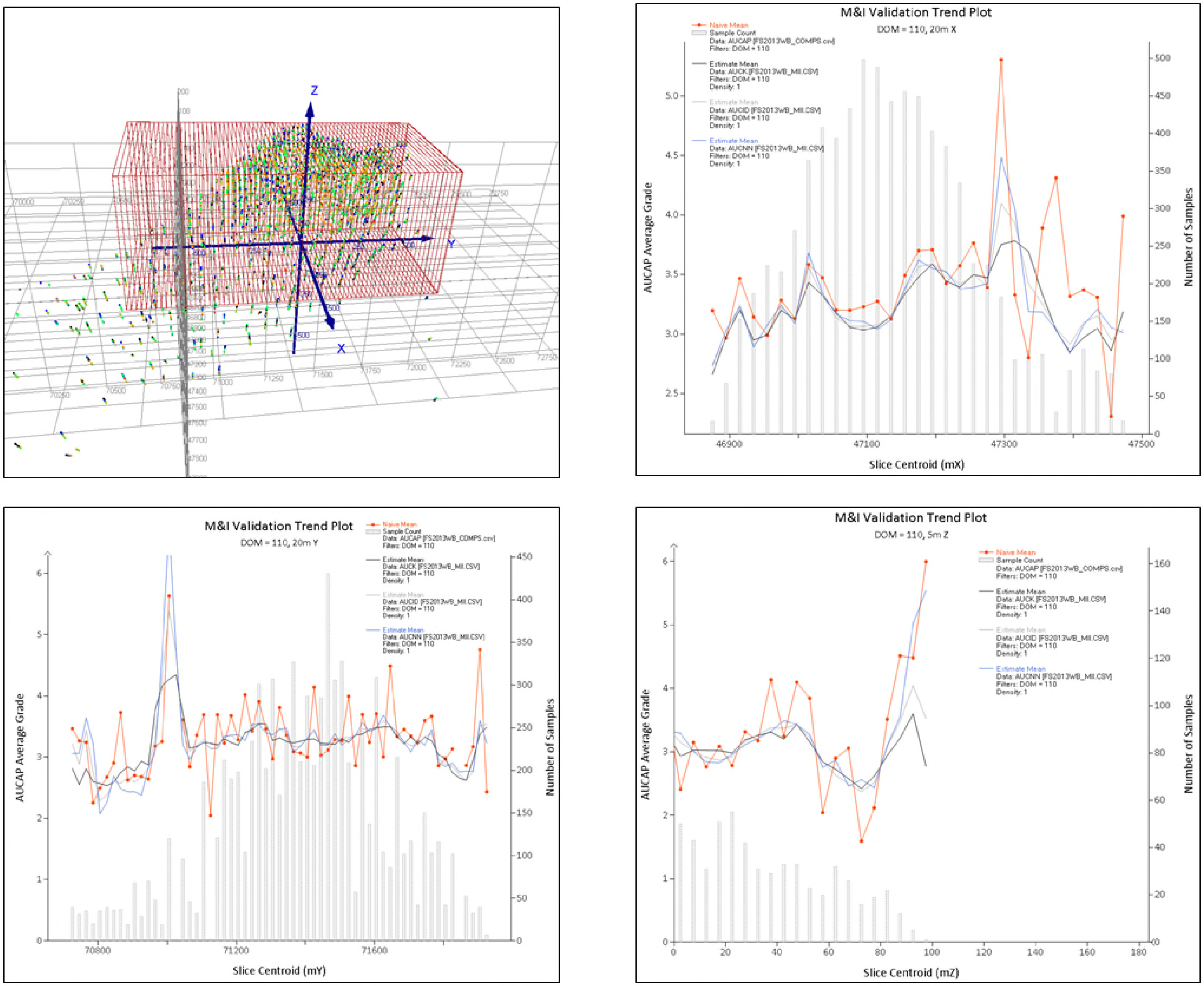
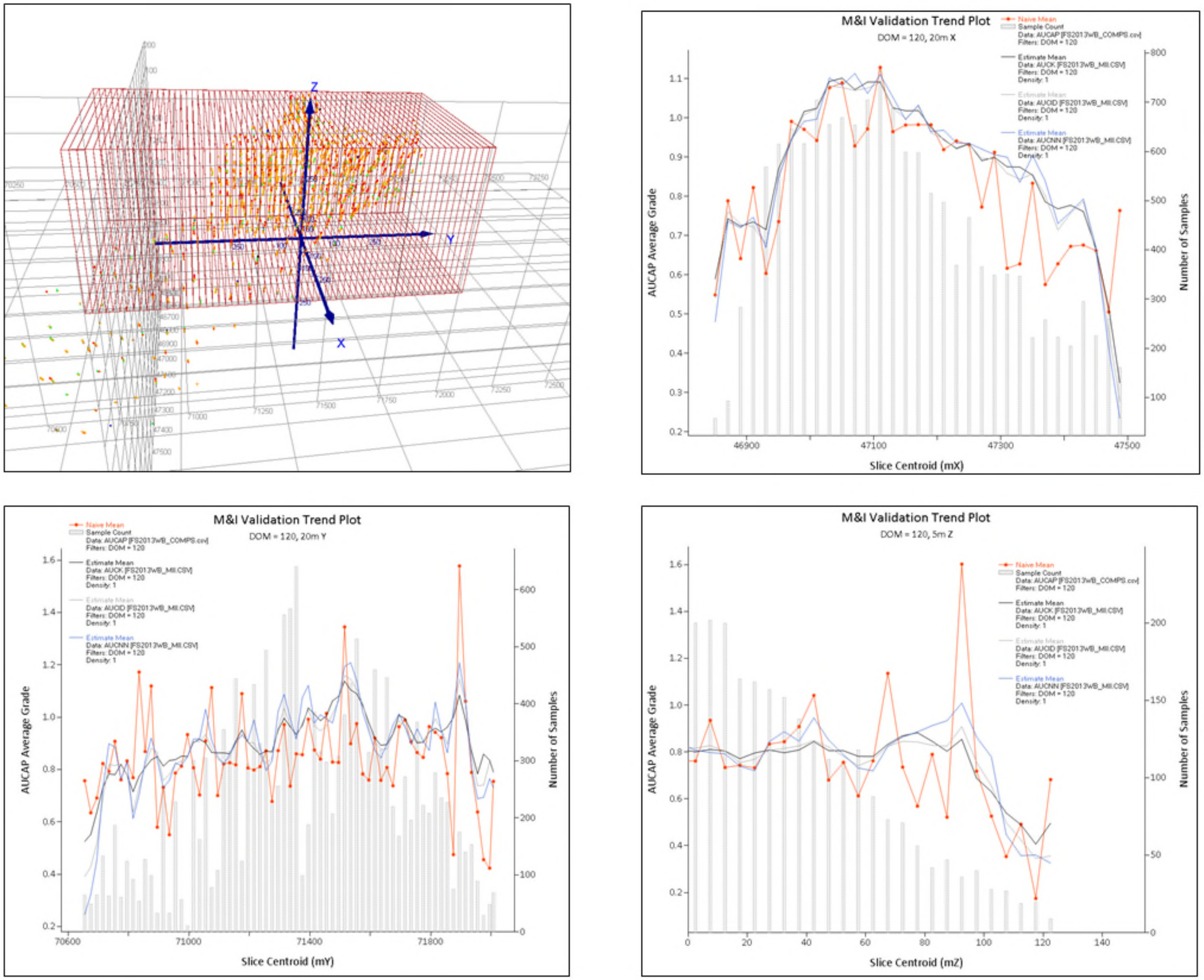


Figure 14-21: DOM6_2 Swath Plot



Hermitian Correction

The relative degree of smoothing in the block model estimates can be evaluated using the Discrete Gaussian or Hermitian Polynomial Change of Support method (described by Journel and Huijbregts, *Mining Geostatistics*, 1978).

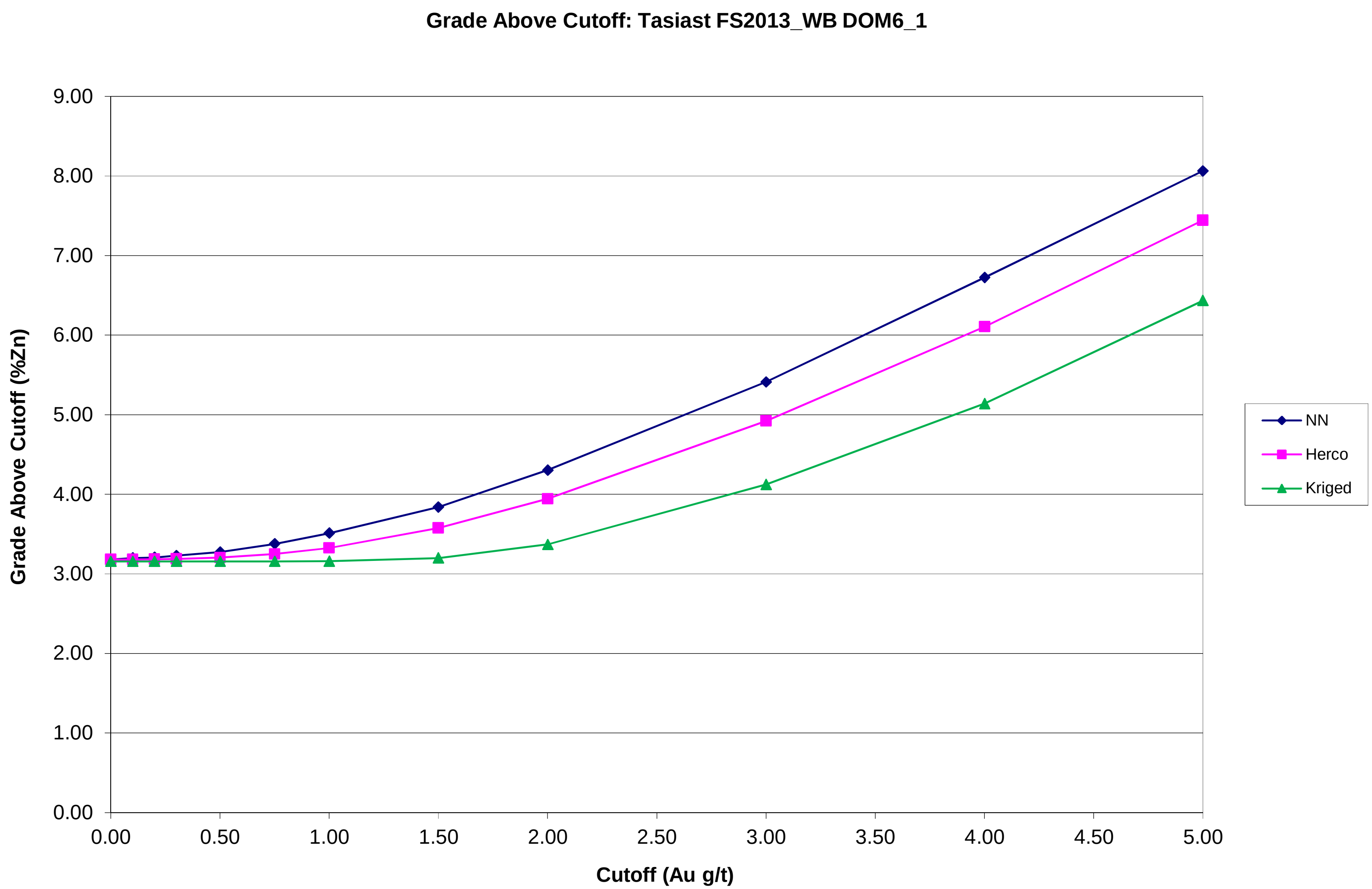
With this method, the distribution of the hypothetical block grades can be directly compared to the estimated (OK) model by using pseudograde/tonnage curves. Adjustments are made to the block model interpolation parameters until an acceptable match is made with the Hermitian Correction (Herco) distribution.

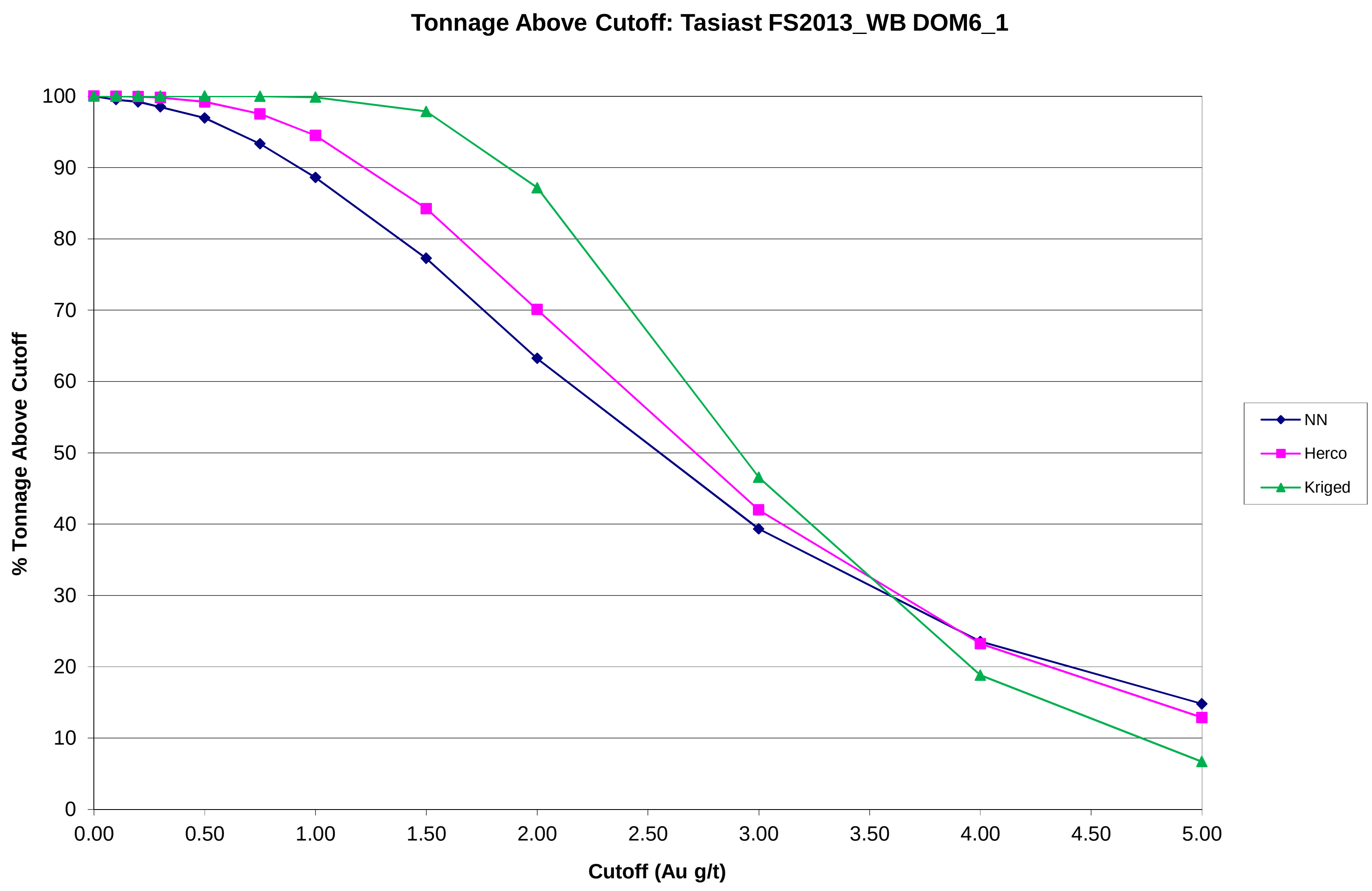
In general, the estimated model should be slightly higher in tonnage and slightly lower in grade, compared to the Herco distribution at the projected cutoff grade. These differences account for selectivity and other potential ore-handling issues that commonly occur during mining.

The Herco distribution is derived from the declustered composite grades that have been adjusted to account for the change in support from smaller drill-hole composite samples to the large blocks in the model. The transformation results in a less skewed distribution, but with the same mean as the original declustered samples.

Generally, the OK models are smooth relative to the Herco distribution. The key exception for WB is DOM6_1, which is the high-grade gold zone (Figure 14-22). At 0.3 g/t Au cutoff, the grade is slightly smoothed (-1%). At 1.00 g/t Au cutoff, the grade is slightly smooth (-5%) and the tonnage is slightly overestimated (5%).

Figure 14-22: WB DOM6_1 Herco Grade-Tonnage Curves




Tonnage Above Cutoff: Tasiast FS2013_WB DOM6_1
100
90
80
70
60
50
40
30
20
10
0
% Tonnage Above Cutoff
0.00
0.50
1.00
1.50
2.00
2.50
3.00
3.50
4.00
4.50
5.00
Cutoff (Au g/t)
NN
Herco
Kriged

14.10 Mineral Resource Statement

The Tasiast Mineral Resource statement, as of year-end 2013, comprises Measured, Indicated and Inferred resources. The Mineral Resources are in accordance with the definitions in national Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101), which became effective February 1, 2001. Mineral resources have an effective date of December 31, 2013. The Mineral Resource block model was prepared by Mr. Tim Maunula, P.Geo., Principal Geologist with T. Maunula & Associates Consulting Inc.

Mineral Resources are stated at variable cutoff grades, dependent on the metallurgical type, mining operating cost and variable process recoveries which were based on metallurgical type. The cutoff grade for CIL ore types was 0.37 g/t Au and 0.22 g/t Au for leach ore types. The cutoff grades were determined using a gold price of $1,400/oz. For the purposes of reporting the resource, the Leach ore types were reported within the Upper Transition and Oxide zones. The CIL ore types were reported within the Lower Transition and Fresh zones. The codes for these corresponding units are classified in the OX block model.

The Mineral Resources were reported below the projected December 31, 2013 mined surface and are constrained using the Lerchs-Grossman 38 kt/d pit shell designed by Kinross Technical Services. Kinross cautions that Mineral Resources that are not Mineral Reserves are not required to demonstrate economic viability. Table 14-36 shows the classified Mineral Resources, exclusive of Mineral Reserves.

Table 14-36: 2013 Tasiast Mineral Resource Statement Exclusive of Reserves

Classification	Tonnes (000’s)	Grade (Au g/t)	Ounces (000’s)
Measured	53,889	0.64	1,103
Indicated	120,722	0.93	3,603
Subtotal M&I	**174,611**	**0.84**	**4,706**
Resource Stockpile	20,234	0.27	178
Inferred	14,146	1.46	664

1. Mineral Resources that are not Mineral Reserves are not required to demonstrate economic viability.
2. Mineral Resources are reported below the 31 December 2013 mined surface and are constrained using an optimized Lerchs-Grossman pit shell.
3. Mineral Resources are reported to a cut-off grade of 0.37 g/t Au for CIL and 0.22 g/t Au for dump leach based on a gold price of US$1,400/oz.
4. Stockpile balance above 1.1 g/t cut-off grade is considered as reserve, while below 1.1 g/t is considered as resource. Measured Resource includes the Resource Stockpile.
5. The above mineral resource estimate is classified in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum’s “CIM Definition Standards - For Mineral Resources and Mineral Reserves" incorporated by reference into National Instrument 43-101 “Standards of Disclosure for Mineral Projects".

Basis for Mineral Resource Statement

Estimation of the Tasiast Mineral Resources was based on a combination of development alternatives that assumed two potential processing options. These included CIL milling of fresh and high grade oxide resource and dump leaching for lower grade oxide material. The various processing recoveries and associated operating conditions were used to generate an optimized pit shell using a Lerch-Grossman (LG) optimization algorithm.

Table 14-37 summarizes the parameters used for the calculation of the cutoff grade for a 38 kt/d LG pit shell.

Table 14-37: 38 kt/d LG Mineral Resource Shell Parameters

CIL all Ore Types

Gold price	45.01
Selling cost	2.47
	42.55
Mining cost	2.13
Milling cost	14.17
Tonnes	1.00
Au	0.37
Recovery	0.91
Recovered Au	0.34
Revenue	14.28
Cost	14.17
Profit	0.11
Calculated cutoff	0.37
Cutoff	**0.37**

Leach all Ore Types

Gold price	45.01
Selling cost	2.47
	42.55
Mining cost	2.13
Leach cost	5.50
Tonnes	1.00
Au	0.22
Recovery	0.60
Recovered Au	0.13
Revenue	5.62
Cost	5.50
Profit	0.12
Calculated cutoff	0.22
Cutoff	**0.22**

14.11 Comment on Mineral Resources

The QP is of the opinion that the Mineral Resources for the Project, which have been estimated using core drill data, have been prepared using industry best practices and conforms to the requirements of CIM Definition Standards. The Mineral Resources are adequate to support mine expansion studies.

Mineral Resources that are not Mineral Reserves do not have to demonstrate economic viability.

15. MINERAL RESERVE ESTIMATE

Although a construction decision on the 38 kt/d expansion will not be made until 2015 or later, the FS supports the mineral resource and mineral reserve estimates set forth in Sections 14 and 15, based on the economic viability conclusions of the FS and an assumed decision to proceed with the expansion for purposes of the estimates[1]. It is important to note that the 2014 and 2015 mine plans for the 8 kt/d and 38 kt/d cases are identical.

The Mineral Reserve for the Tasiast open pit mine was estimated using the FS2013 resource models for West Branch (WB) and Piment and Prolongation (PP), as discussed in section 14. These resource block models were used to develop a diluted mine planning model by applying a methodology that mimics the dilution and ore loss that is likely to occur during mining operations by re-blocking at cut-offs that match the grade control cut-off grades.

The 38 kt/d feasibility reserves, effective December 31, 2013, were estimated using the diluted mine planning block model and applying a gold price of $1200/oz, a CIL processing cost of $17.82/t, a $ 5.49/t processing cost for dump leach ore, and a base mining cost of $2.13/t to $2.17/t, excluding incremental haulage. The reserve estimates include material contained within the final pit design that can be extracted and processed economically and are therefore based on Proven and Probable reserves only, which correspond to Measured and Indicated mineral resource classifications (Table 15-1).

Table 15-1: Reserves Estimate Effective December 31, 2013

Classification	Tonnes (000's)	Grade (Au g/t)	Ounces (000's)
Proven	34,029	1.33	1,453
Probable	141,504	1.80	8,191
TOTAL	**175,533**	**1.71**	**9,644**
Reserve Stockpile	2,049	1.31	86

1. Mineral Reserves are reported to a cut-off grade of 0.6 g/t Au for CIL and 0.40 g/t Au for dump leach based on a gold price of US$1,200/oz less 5% gross royalty.
2. Stockpile balance above 1.1 g/t cut-off grade is considered as reserve, while below 1.1 g/t is considered as resource. Proven Reserve includes the Reserve Stockpile.
3. The Reserves Estimate assumes the construction of a 38 kt/d mill.
4. The above mineral reserve estimate is classified in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum's "CIM Definition Standards - For Mineral Resources and Mineral Reserves" incorporated by reference into National Instrument 43-101 "Standards of Disclosure for Mineral Projects".

1. As per the CIM Definition Standards, "The term 'Mineral Reserve' need not necessarily signify that extraction facilities are in place…"

15.1 Basis of Reserve Estimate and Pit Optimization

An economic pit shell generated with cost criteria, metallurgical recoveries, geologic and geotechnical considerations guides the final pit design. The economic pit shell used to define the final pit limits was created using CAE's NPV Scheduler software (NPVS). NPVS uses the Lerchs-Grossman (LG) algorithm to define blocks that can be mined at a profit. The program then creates an economic shell based on the following information:

- Starting topography
- Overall slope angles by rock type
- Metallurgical recoveries by mineralization and rock type and oxidation state
- Geologic grade model with gold grades, density, lithology and mineral types
- Process and mining costs
- Incremental vertical bench mining cost
- Downstream costs, such as gold refining, freight and marketing
- Sustaining capital for future equipment replacements
- Mining dilution and recovery

The Mineral Reserve Estimate was prepared using the December 31, 2013 topography and the parameters detailed in Table 15-2.

Table 15-2: LG Optimization Parameters

LG Parameter	Cost / Assumption	Unit
Piment (PM) Mining Cost	2.17	US$/t
PM Haulage Increment Bench (5 m)	0.029	US$/t
West Branch (WB) Mining Cost	2.13	US$/t
WB Haulage Increment Bench (10 m)	0.038	US$/t
Process Cost	14.37	US$/t ore
Site Admin	2.31	US$/t ore
Sustaining Capital	1.14	US$/t ore
Stockpile Re-handle	1.00	US$/t stock
Dump Leach	5.49	US$/t ore
Gold Price	1200	US$/oz
Selling cost	66.60	US$/oz
Other costs	6.00	US$/oz
Royalties	5.00	%
Payable	99.95	%
Discount rate	5	%
Sinking rate	10	Bench/year

Mineral Reserves are stated within an ultimate pit design at variable cut-off grades that are based on the process type, operating costs and metallurgical recovery.

A dilution allowance for the Mineral Reserve estimate is applied using diluted model grades. The diluted model, which was built from the resource block model, incorporates dilution and ore loss and eliminates the need for applying additional factors.

Slope parameters based on geotechnical considerations were applied to the pit design along with ramps and geotechnical catch benches, and subsequently used to generate overall slope angles. The overall slope angles used in pit optimization are shown in Table 15-3.

Table 15-3: Overall Slope Angles Used in Pit Optimization

Material	West Branch		Piment	
	Azimuth (°)	Slope (°)	Azimuth (°)	Slope (°)
Oxide	0	30	0	25
	45	36	90	29
	225	30	270	25
Fresh	0	33	0	34
	20	45	90	48
	210	33	270	34

Gold recovery is determined by ore type and process method. The gold recovery is calculated from the information in Table 15-4 where gold grade is expressed in grams per tonne (g/t).

Table 15-4: Process Recoveries

Facility	Ore Type	Gold Recovery (%)
Existing CIL	Oxide mineralization	93
	Fresh mineralization	Recovery % = 1.2579 x ln(head grade g/t) + 91.338
Dump leach	Hanging wall BIM oxide	75
	Footwall BIM oxide	54
	Greenschist oxide	70
	Sedimentary volcanic clastic oxide	70
	Felsic volcanic clastic oxide	72
New CIL	Oxide mineralization	93
	Fresh mineralization	Recovery % = 1.2579 x ln(head grade g/t) + 91.838

The mine operating costs used for pit optimization include ongoing major mine equipment capital costs. The mine equipment sustaining capital is used in the economic model to simulate mine capital expenditures when generating the economic pit.

The top-down discount method was used for pit optimization. This is a procedure based on multiplying the block value by a discount factor that is a function of the annual cost of money, an estimate of the average annual vertical advance rate of mining, and the relative depth of the block. This method helps to simulate the actual mine plans that are burdened with up front stripping costs and aids in the selection of a higher value pit.

15.2 Comment on Mineral Reserves

Roscoe Postle Associates Inc. conducted a review of the resource model to assess the model's suitability for pit optimization, design and resource estimation.

David Tutton, a independent consultant mining engineer, conducted an independent verification of the mineral reserves. The same party also reviewed the mine planning parameters used in the determination of the mineable resource.

John Sims, AIPG Certified Professional Geologist, has certified that, to the best of his professional judgment as a QP (as defined under NI 43-101), the Mineral Reserve and Resource estimates have been prepared in compliance with NI 43-101, including the CIM Definition Standards incorporated by reference, and conform to generally accepted mining industry practices.

The results of the economic analysis to support Mineral Reserves represent forward-looking information that is subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. Areas of uncertainty that may materially impact mineral reserve estimation include:

- Commodity price and exchange rate assumptions;
- Capital and operating cost estimates; and
- Geotechnical slope designs for pit walls.

16. MINING METHODS

This section describes Mining Methods for the current 8 kt/d operation. It is important to note that the 2014 and 2015 mine plans for the 8 kt/d and 38 kt/d cases are identical. Mining Methods for the 38 kt/d case are described in Section 24.16.

Mining Operations

Commercial production of gold at Tasiast began in January 2008, and a total of 1,139,275 oz was produced by the end of 2013.

Ore and waste rock is currently mined by conventional open pit methods from two pits (West Branch and Piment). Prior mining has taken place in West Branch, Piment and several other completed pits at Tasiast. Since Kinross acquired the property in late 2010, a total of 211 million tonnes of material have been mined from various pits, including 51 million tonnes in 2011, 78 million tonnes in 2012 and 82 million tonnes in 2013. Drilling and blasting is performed with regards to wall control and fragmentation using the same methods in both ore and waste material. The excavation fleet on site is made up of seven Bucyrus RH340 hydraulic excavators that are used to load 42 Caterpillar 793D 220 t haul trucks and six Komatsu PC1250 hydraulic excavators that are used to load 21 Komatsu 785 90 t haul trucks. The larger shovel and haul truck pair is used at West Branch (Figure 16-1), while the smaller shovel and truck combination is used at Piment. The introduction of larger mining equipment has shifted the mining strategy from a highly selective mining practice to a combination of both bulk and selective mining.

Figure 16-1: Excavators and Haul Trucks Working in West Branch Pit

The mining fleet present at site is detailed in Table 16-1.

Table 16-1: Summary of Current Mining Equipment

Equipment	Type	Quantity
Komatsu 785	Haul truck	21
CAT 793D	Haul truck	42
PC1250	Shovel	6
RH340[2]	Shovel	7
CAT 988H	Loader	3
CAT 988F	Loader	1
CAT 994H	Loader	2
5 m drills (Pantera 1500)	Production drill	5
10 m drills (Bucyrus SXFX)	Production drill	10
DR 580	GC and pre-split drills	2
CAT D11T	Bulldozer	2
CAT D10T	Bulldozer	8
CAT D9R	Bulldozer	3
CAT D8	Bulldozer	1
CAT D6	Bulldozer	1
CAT 854K	Wheel dozer	3
CAT 824G	Wheel dozer	1
CAT 24M	Grader	3
CAT 16M	Grader	4
Komatsu HD785	Water truck	4
CAT 773	Water truck	4
Benz Actros	Small water truck	3
Renut	Small water truck	1
Pegasol	Small water truck	3
Western Star	Small water truck	2

[1] For the 8 kt/d case, no mining equipment is purchased in 2014.
[2] RH340 is now referred to as Caterpillar 6060.

From the two operational pits, mineralized material is hauled to the run-of-mine (ROM) pad adjacent to the primary crusher and tipped to the stockpile for reclaim by a front-end-loader (CAT 988). Sub-grade material is stockpiled adjacent to the ROM pad for later treatment, while mineralized material goes directly to the 8,000 t/d plant for processing. The grinding circuit produces a product size of 80% passing 90 microns which is then processed in a conventional CIL circuit and ADR plant to produce doré bullion. Gold recovery varies from 91% to 93%. Low-grade oxide material is trucked directly to the dump leach pads. The dump leach facility consists of two separate leach pads with gold recoveries ranging from 54% to 75%. Tailings slurry from the CIL process is pumped to the tailings storage facility (TSF), a specifically engineered facility comprised of two imperviously-lined paddock dams located one kilometer southwest of the processing plant. After settling out the solids, the process solution is recovered and pumped to the plant for re-use.

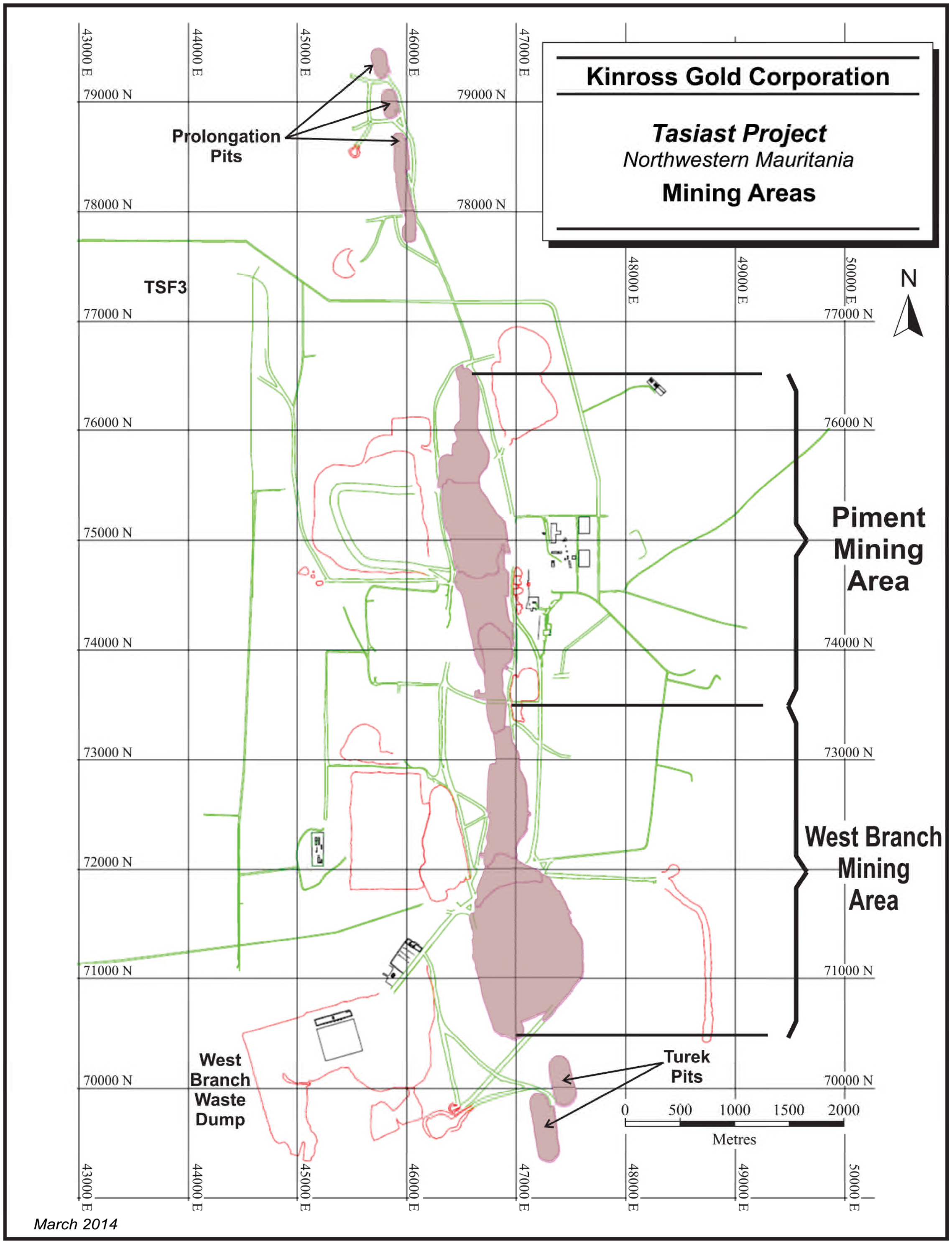
In July 2009, the Greenschist Zone mineralization was identified and subsequently confirmed by exploration drilling. A PFS was initiated in 2012 and completed in May 2013 to assess the economic viability of an expansion to the current Tasiast operations. The positive results indicated by the PFS prompted a subsequent FS that was completed in March 2014. The FS examines the case for expanding the Project to a throughput of 38 kt/d. The results of the FS regarding the expansion case are discussed in Section 24.

Mine Design

The Tasiast final pit design consists of a series of overlapping pits that extend along a strike length of approximately 8 km. The configuration of the mining area is shown in Figure 16-2. Only Piment and West Branch are actively being mined at present.

Most of the in-pit ramps are placed on the footwall (west) side of each staged pit to provide the shortest trucking distance for the majority of excavated material hauled to the waste dumps and dump leach pads that are on the west side of the pit. Ramps have a gradient of 10%. Bi-directional traffic occurs on 20 m wide roads at Piment and 35 m roads at West Branch. West Branch has a second haul road on the east pit wall for access to the east side waste dump. All intersections and switchbacks are designed without grade (flat) wherever possible. The processing plant is located on the east side of the Piment pit and access roads for ore haulage from west to east are located between the operational pits. This cross-pit access road will be maintained by backfilling portions of the mined out pits with waste rock.

Figure 16-2: Plan View of Tasiast pits



Waste Dumps

Waste rock is used for haul road and tailings dam construction as needed. The road network currently in place is well developed, but will require continued maintenance. Additional roads will also be required throughout the life of the mine.

When not needed for infrastructure purposes, waste material is disposed of in constructed waste rock dumps situated at least 100 m radially away from the final crest of an open pit and outside the zone where there is potential for dilation, cracking and subsidence related to the pit walls. Management of these dumps during the mine life cycle is important to protect human health and safety, along with the environment. As no specific national standard for waste rock storage exists, the Kinross Standard for Waste Rock is applied to the Tasiast waste rock storage facilities.

The sites of the waste dumps are characterized by flat ground with a thin cover of soil, alluvium and drifting sand, with minor low outcrop. The soil and alluvium are thought to have a low clay content, but there is potential for localized shallow clay pans. The rock below this is completely weathered and partially weathered (pervasive oxidation) to a depth of about 40 mbgl. Minor quantities of groundwater are expected in this area, but much of this will drain as the pits are developed nearby, which suggests that there is unlikely to be significant subsidence as drainage occurs. No known transects of defined streams or gullies are in this area that would facilitate run-off water flow, nor are there paleo-alluvial channels. However, depressions might occur in which sheeting run-off water could become concentrated.

Dumped waste material comprises weathered rock and unweathered rock. The weathered material ranges from pervasively to partially weathered and includes patches of completely weathered, clayey rock. Partial weathering predominates. Blasted weathered rock is finely graded, including clay-silt fines. The unweathered rock is strong and massive and, when blasted, is coarsely graded, including boulders that will require secondary blasting for loading and haulage. This material is suitable for the base of the dump, as it does not have a high clay content.

Static Acid Rock Drainage (ARD) tests performed on the range of lithologies encountered in the mining area indicate that 91.6% of the samples were considered to be non-acid forming (NAF). Where materials have been identified as potentially acid forming (PAF), the magnitude of acid generation is generally regarded as comparatively low. On balance, the acid neutralization capacity of host waste rock is currently considered to be significantly in excess of the total acid generating potential.

Leachate testing demonstrated that under the provision that waste rock dumps remain at circum-neutral pH, no potential toxic metal leaching is to be anticipated. However, the release of oxyanions (arsenic, boron, molybdenum, vanadium and tungsten) under circum-neutral conditions were marginally elevated.

As the climate is arid and there is no permanent surface water and very limited groundwater, there is low potential for ARD concerns to develop. However, any potential ARD issue will be mitigated by ensuring that material that is identified as PAF (per the material contained inside the wireframe for ARD – Section 14.2 under "Acid Rock Drainage Potential") will not be dumped on the outer shell of the waste dumps.

The dump design is based on 10 m high lifts with a dump face angle at the angle of repose of the dump material (37°). Each successive lift will be stepped back by approximately 30 m from the crest of the underlying lift to create a benched configuration for the final face of the dump. Total heights will be limited to 50 m to 100 m. The dumps will be accessed by 35 m wide dual-lane ramps at a 10% gradient. Track bulldozers will be used to assist the haulage fleet to facilitate proper dump construction, including grading the top of each lift away from the pit to direct any rainwater run-off and placing coarsely-graded, unweathered rock on final rill faces.

The dump footprint and height requirements were enhanced based on an analysis that optimized haulage from the pit. The waste dumps are located within the footprints of previous studies and require no modification to the current permitting.

Geotechnical Considerations

Historically the Project has been broken into two geotechnical zones where the Piment Zone is north of approximately 72,000 N and the West Branch Zone is south of this northing. Overall pit slope angles and inter-ramp angles for the Piment Zone were initially determined by Scott Wilson Mining UK (Scott Wilson) in 2009 and subsequently optimized by Stacey Mining Geotechnical Ltd (Stacey) in 2011. The slope angles that are currently applied at Piment are shown in Table 16-2, with catch benches based on the bench face angles and ultimate bench height.

Table 16-2: Geotechnical Design Parameters – Piment Zone

Zone	Depth (m)	Wall(s)	Bench Face Angle (°)	Inter-Ramp Angle (°)	Total Bench Height (m)	Berm Width (m)
Oxide	0 – 10	All walls	60	39	10	6.5
Transition	10 – 50	West wall (FW)	50	34	10	6.5
		All other walls	65	42		
Fresh	> 50	West walls (FW)	50	38.5	20	8.5
		East walls (HW) and end walls	75	55		

The pit slope angles and associated geotechnical parameters for West Branch were determined by Scott Wilson in 2009 and updated by the same party in August 2012. The 2012 Scott Wilson design has subsequently been supplemented by interim reports completed by Golder Associates based an advanced geotechnical investigation (GI) conducted in 2013. The final report detailing the findings of the 2013 GI has not yet been received, but two interim reports have been submitted and taken into consideration. The slope angles that are currently in use at West Branch are shown in Table 16-3.

Table 16-3: Geotechnical Design Parameters - West Branch Zone

Zone	Depth (m)	Azimuth	Bench Face Angle (°)	Inter-Ramp Angle (°)	Total Bench Height (m)	Berm Width (m)
Oxide	0 – 30	000° to 210°	60	39.2	10	6.5
		210° to 360°	45	31.2		
Transition	30 – 100	000° to 210°	65	48.3	20	8.5
		210° to 360°	45	35.1		
Fresh	>100	000° to 210°	65	50.8	30	10.5
		210° to 360°	45	36.5		

The designed operating bench height is 10 m in West Branch and 5 m in Piment. The final pit walls in West Branch will be triple benched where it is permissible, resulting in a bench height of 30 m. Piment will be quadruple benched, resulting in a 20 m bench height. Intervening catch benches vary based on the bench face angles and ultimate bench height.

The overall slope angles used for optimization purposes (Table 16-3) were adjusted to reflect allowances for haul ramps.

Groundwater ingress is not expected to interfere with operations as mining progresses into fresh rock. A pit floor sump is currently used to remove water during mining of the oxide and transition zones. This water is used for dust suppression. The need for horizontal drains for pit wall depressurization and/or dewatering wells is currently being determined by a hydrogeological study.

Surface run-off from any unexpected storm events will be diverted from the pit by a ditch and berm system.

Production Schedule

A mining rate of 50 Mt/a would be sustained through 2015 in both the 8kt/d and the 38 kt/d scenarios. In the 8 kt/d case, the current mining rate would ramp up to 72 million t/a through 2020 before leveling off at 50 million t/a through 2026, at which point it would decline until the end of mining in 2030. Between 2031 and 2045 the existing mill would continue to produce from stockpiles at the 8 kt/d rate.

A decision regarding the proposed expansion will be made in 2015 at the earliest. The mine life estimated for the 38 kt/d processing capacity and a ramp up to 120 million t/a is discussed in Section 24.

Personnel Requirements

The current mine operation at Tasiast is owner-operated and applies conventional open pit operational practices, with drilling, blasting, loading, hauling, support and administrative functions. The mine operates 365 scheduled days per year and 24 hours per day, primarily divided into two 12-hour shifts per day for mine operations and mine maintenance.

The mine organization includes functional groups for mine operations (drilling and blasting, loading and hauling), maintenance, mine technology and technical services (Table 16-4). The organization is staffed to support operational, safety and environmental requirements. Mining-related functional groups are organized under the mine manager or the director technical services. The mine manager is allocated functional groups for mine operations, maintenance and mine technology. Among the functional groups responsible for mine operations, drilling and blasting are managed together, as are loading and hauling. The director of technical services oversees functional groups for technical services, geology and geotechnical services. The technical services function for the existing operations includes technical engineers in mine planning, surveying, geotechnical engineering and mine operations, as well as geologists in grade control. The mine manager and director of technical services collaborate to manage mine operations, with each reporting to the general manager of Tasiast.

Table 16-4: Mining Personnel by Function by Year for the 8 Kt/D Case

Position	2014	2015
Mine Management	4	4
Mine Operations	328	312
Mine Maintenance	265	274
Mine Technology	13	12
Mine Technical Services	33	33
Geology	21	21
Mining Total	664	656

17. RECOVERY METHODS

17.1 Current Waste Rock Facility

Currently, two waste dumps along the east and west sides of the existing Piment Pit and the west side of West Branch pit are used to dispose of waste rock.

17.2 Current Water Management

A water supply bore field provides raw water for the mine, processing plant, dump leach and camp. The site potable water requirements are met by three reverse osmosis plants. For further detail on Water Supply, see Section 18.

The plant rejects in the form of tailings slurry is pumped to a tailings storage facility, where about 50% of the contained tailings water is recovered and pumped back to the plant or dump leach storage ponds.

17.3 Current Process Plant

The 8 kt/d capacity processing plant using the CIL process (approximately 3 Mt/a) produces approximately 70% of the gold shipped from Tasiast. The remaining gold is produced from the dump leach operations via a dedicated ADR plant. All produced gold is in the form of bullion and is transported regularly to a refinery for final refining and sale. The Tasiast process flow diagram based on 8 kt/d is shown on Figure 17-1. A project to upgrade and remove bottlenecks in the plant was completed in 2013, at a total cost of approximately US$ 23 million.

Crushing

Mined ore above plant feed cut-off grade is transported from the open pits to the plant by truck and deposited onto the ROM pad. The material is blended according to grade and competency. To aid blending, material is transferred to the crushing plant feed bin by front end loader. Lower grade weathered ore is hauled directly, without crushing, to the dump leach pads.

Crushing of the mineralized material takes place in three stages; a primary jaw crusher that reduces rock to less than 150 mm; a secondary cone crusher and two tertiary cone crushers. Screens located before the secondary crusher remove material that is at the final product size, nominally passing 16 mm. Material that is greater than 16 mm is fed to the secondary crusher.

Secondary crushed material is conveyed to a screening section before two tertiary stage crushers. Oversized rock is subjected to further crushing and returned to the screens, the tertiary screens and crushers forming a closed circuit. Material passing through the tertiary screens joins the secondary screen undersize as final product, of 80% passing 16 mm screen, which is then transferred to two fine ore bins.

Grinding

Crushed material is transferred from the fine ore bin at a controlled rate to each of two ball mills by means of a conveyor belt, and water is added to maintain a fixed slurry density in the mills. The ground ore slurry is pumped to hydrocyclones for classification. The target grind size is 80% passing 90 µm. Cyclone overflow containing solids of the required size flows to the leaching circuit. Coarser solids are returned to the mill for further grinding, via a gravity circuit to capture liberated, coarse gold particles.

The cyclone clusters were upgraded in 2013 to remove a capacity restriction, and new gravity concentrators installed in each grinding circuit to maximise recovery of coarse gold. The gravity concentrates flow periodically to a Leach Reactor where the gold is dissolved from the concentrates and the resulting solution pumped for gold recovery in the gold room.

CIL

The leaching circuit at Tasiast comprises a single pre-leach tank and 6 CIL (carbon-in-leach) tanks where the dissolved gold is recovered on activated carbon. Material that exits the grinding circuit has an approximate slurry density of 42% solids, by weight. The slurry gravitates to the first of the seven agitated tanks via a trash screen to remove oversize tramp particles. Lime is added to increase the slurry pH to 10.2 then dilute sodium cyanide solution is added to maintain a fixed cyanide concentration. Pure oxygen is injected into the first 4 tanks to enhance gold leaching kinetics. The oxygen demand in the remaining tanks is satisfied by compressed air injection.

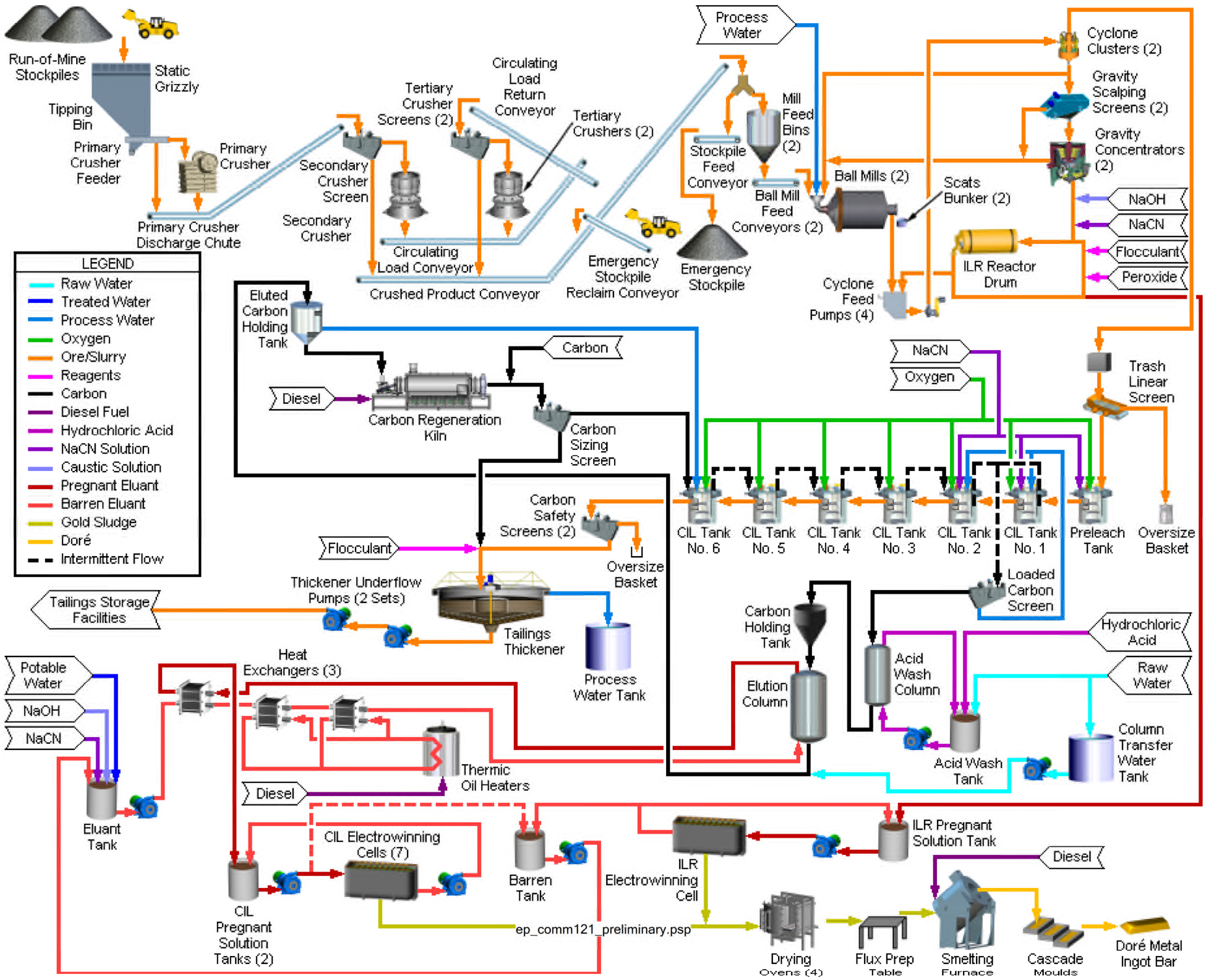
Activated carbon granules contact slurry in the CIL tanks to adsorb the dissolved gold from solution. Carbon that has achieved the target gold content, typically around 2500 g/t, is termed "loaded" and is transferred daily to the elution circuit for recovery of the gold on a batch basis. After each such transfer, carbon in the remaining tanks is advanced counter-current to the slurry flow on a sequential basis, and fresh or “barren” carbon is added to the last CIL tank to maintain the carbon inventory.

After maximising gold recovery from the solution and ore particles in the CIL process, the resulting slurry flows via carbon retention screens to the thickener, where the solids are settling to achieve a density of approximately 55% solids and residual solution is returned to the process. This existing thickener was modified, as part of the de-bottlenecking project, to achieve higher settling capacity and returned to service in 2013.

Carbon Elution and Electrolysis

Loaded carbon recovered from the CIL slurry by screening is first washed to remove entrained ore particles and then washed with hydrochloric acid solution, in a dedicated wash vessel, to remove lime scale from the carbon surfaces. The acid-washed carbon after being neutralized is transferred to the elution pressure vessel. To recover gold from the loaded carbon, batches of approximately 6 tonnes of carbon are subjected to a high pressure and temperature, leaching process, called elution. Tasiast is using the Anglo American Research Laboratories (AARL) strip process with no cyanide addition. A hot caustic soak under pressure followed by a rinse with demineralized water removes the gold from the carbon and into solution. After gold removal, the "barren" carbon is transferred to a regeneration kiln for thermal reactivation of the carbon. Reactivated carbon is returned to the last CIL tank. Gold is recovered from the caustic solution by electrowinning onto stainless steel wire wool cathodes in electrowinning cells, located within the Plant gold room. The gold is removed as a sludge by pressure washing the cathodes at intervals, The sludge is dried and mixed with fluxing materials and charged to a diesel-fired crucible furnace. After melting, the slag is poured and followed by pouring the gold into bullion moulds. Bullion or “doré” contains 94% or higher gold content together with a minor content of silver. The doré bars are transported by a security firm to a commercial refinery for further purification and sale.

Figure 17-1: 8 kt/d Plant Process Flow Sheet



Current Tailings Storage Facility

Tailings from the CIL process plant are pumped to tailings storage facility (TSF). A new facility (TSF3) was constructed and commissioned in 2012. The TSF is a specifically engineered facility, currently comprising a four-sided lined basin, located on the west side of the open pits. After settling of the solids, a proportion of the contained water drains to a collection area within the storage basin, from where it is pumped to the plant for makeup to the process water system. Solids are retained in the TSF.

Dump Leach

Dump Leach Pads (Heaps)

The dump leach operation has recently been re-designed to process up to 11 Mt/a of low-grade oxide mineralization utilizing two dump leach operations, one pad (Piment) with five cells separated by a raised berm for solution drainage control, and the other pad (West Branch) with 8 cells. The design of each pad allows for three 10 m lifts for a final stack height of 30 m. All solution collection ponds are plastic-lined with installed leak detection systems and bird netting protection. The gold-containing (pregnant) solution produced by the dump leach operations is pumped to a dedicated ADR plant to recover the gold and reactivate the carbon.

The initial construction involved earthworks to ensure correct drainage and following compaction a 1.5 mm HDPE plastic geomembrane liner was laid down. The liner was covered with 0.5 m layer of overliner (suitable crushed waste or low-grade mineralization) which acts as a cushion layer to protect the plastic when dumping. Slotted plastic drainage pipes are laid across the pad within the overliner to allow the solution percolating through the heap to be collected and passed to the main drainage pipe at the side of the pad and hence to the required process pond. In addition, leak detection systems were installed below the geomembrane liner. Three ponds adjacent to each of the two pads are provided for storage and management of the barren, intermediate and pregnant solutions.

Mine haulage trucks deliver low-grade ROM material directly to the pads, dump and the ore pushed as required by bulldozer to level the surface. Once sufficient material has been placed to an initial height of 10 m, a layer of lime is spread on top to the typical usage of 4.0 kg/t and the surface is ripped. The heap is then available for irrigation.

At each of the two pads, barren solution is pumped from the barren pond to irrigate the heap. The first drainage collection of “intermediate” solution is collected in a pond, then pumped to another section of the pad to contact freshly placed ore. The resulting drainage, or Pregnant solution is captured in the pregnant solution pond adjacent to each pad and pumped at a controlled fixed rate to the ADR plant for gold recovery.

Make-up water is added to the systems from the raw water pipeline system connecting the bore field with the CIL mill. The lime added during stacking maintains a minimum pH of 10. Irrigation is targeted at 10 L/h/m^2 and is applied using a system of plastic pipe headers and dripper pipe. Anti-scalant solution is added at pumping points to inhibit scale formation. Pregnant solution contains approximately 0.5 g/t Au.

ADR Plant

The pregnant solution produced by the dump leach operations is received in a single train of six carbon containing columns at the ADR plant, as seen in Figure 17-2. The pregnant solution flows by gravity from column to column, contacting carbon in each column as it rises from the bottom to overflow. The Activated carbon granules adsorb the dissolved gold from solution. The solution leaving the last column is barren solution and is pumped to the dump leach operations for re-use. When the carbon in the first column loads with gold to the target level, the first column contents are advanced by airlift to a screen and discharged to an acid wash vessel. Sequentially, each following column is advanced counter-current to solution flow.

After acid washing of the carbon it is advanced to the elution vessel, where using the same AARL process as in the Plant elution system, gold is eluted from the carbon and the gold rich “pregnant” solution stored in a surge tank prior to electrowinning. After elution of the gold, the barren carbon is re-activated thermally in a diesel fired rotary kiln, stored in a surge bin and eventually returned to the column train via the last tank in series.

Figure 17-2: West Branch Solution Ponds and ADR Plant

Gold is recovered from the pregnant solution by electro-winning onto stainless steel wire wool cathodes in a single electro-winning cell. The gold is washed from the cathodes after each elution cycle, dried in an oven, mixed with fluxes and melted in a crucible furnace, all within the Gold Room attached to the ADR plant. Doré bars of 94% or higher purity are produced and transported to a commercial refinery for further refining and sale.

Gold Recoveries

Unit gold recovery estimates for the existing plants are based on metallurgical test work and a review of historical performance. The recovery varies by mineralization type, lithology and treatment method as presented in Table 17-1.

Table 17-1: Metallurgical Recovery by Process

Process	Oxide	Primary
CIL Recovery	93%	91%
DL Recovery*	54% - 75%	

*Note: * Dependent on lithology*

18. PROJECT INFRASTRUCTURE

Tasiast Team Village (TTV)

The TTV is sized to accommodate a workforce of 3,540 personnel. It includes various facilities, such as clinic, laundry, kitchen and dining areas, gymnasiums, recreational rooms and various sports playgrounds.

Waste Management

Waste from plant and equipment maintenance, construction, offices, kitchens and accommodation is recycled or handled in an on-site landfill.

Sewage is disposed of through septic tanks fitted with soak away overflow systems. Currently there are septic tank systems at the mine camp and at the mine offices. Tanks are emptied on an ‘as required’ basis and the effluent is placed in a bunded area to dry. A waste water treatment plant was commissioned in 2011 and is treating approximately 50% of camp waste water. Treated effluent is disposed of through a spray field.

Water Supply

Raw water for the Tasiast site is from a water supply bore field, which is located 64 km west of the mine, and draws from a brackish aquifer using a system of 47 wells. Each well is able to produce approximately 500 m^3/d on a continuous basis. Individual wells are combined in a manifold to feed three different systems, each with a pumping station and downstream booster stations. In total, the well field and pipelines are capable of supplying up to 24,000 m^3/d of raw water to the site based on the availability of the pipelines and pumps. Upgrades to increase the reliability of the system to achieve continuous throughput of 24,000 m^3/d are being evaluated.

Reverse osmosis (RO) water treatment plants and storage basins/tanks are located at the mine site. Saline water produced from the RO plant is used to water the haul roads. Potable water is produced from RO water following additional disinfection steps. Potable water is also used for domestic purposes at the Tasiast site.

Power Supply

Electric power required is provided as follows:

- The Phase 1 plant consists of eight LFO Caterpillar 3512 MUI high-speed generator sets and three HFO Caterpillar MaK 6CM32C medium-speed generator sets, with a total capacity of 12.7 MWe.

- The Phase 1A plant consists of four LFO Caterpillar 3512 MUI high-speed generator sets, with a total capacity of 4.8 MWe.
- The Phase 1B plant consists of four HFO fired Wärtsilä 12V32 medium speed generator sets, with a total capacity of 18 MWe.
- The TTV plant consists of five LFO MTU Model 16V40000G23 high-speed generator sets, with a total capacity of 8.6 MWe.

Power from the Phase 1 and Phase 1B power plants is fed via a new 33 kV distribution system to supply required loads.

Service and Administration Buildings

Service and administration buildings include:

- Guardhouse
- Training and development campus
- Training centre
- Warehouses
- Kitchen and dining hall
- Plant office building
- Welcome centre

19. MARKET STUDIES AND CONTRACTS

Kinross typically establishes refining agreements with third-parties for refining of doré. Kinross's bullion is sold on the spot market or as doré, by marketing experts retained in-house by Kinross. The terms contained within the refining contracts and sales contracts are typical and consistent with standard industry practice, and are similar to contracts for the supply of bullion and doré elsewhere in the world.

20. ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Studies

Current mine operations and the expansion project are based on the formal approval of a number of Environmental Impact Assessment (EIA) studies completed before and since mine commissioning in 2007.

For all project areas (excluding the beach landing for import of preassembled equipment), environmental baseline conditions have been determined by reviewing existing published data, previous EIAs, satellite imagery and environmental reporting undertaken for the mine. Where appropriate, existing data for project areas was supplemented by primary data collected through environmental baseline surveys. Field-level baseline surveys were completed for project areas, including air quality, archaeology, flora, fauna, marine, water quality, traffic, and socioeconomics.

The baseline conditions formed the basis to assess the project through a series of EIAs and Environmental Impact Notifications (EINs). The environmental assessments utilized applicable Mauritania legislation, the International Finance Corporation Performance Standards, the International Cyanide Management Code and Kinross Health, Safety, Environment and Social Management Systems for project design and management, mitigation strategies and performance monitoring. The environmental assessments determined appropriate mitigation and management where impacts could not be avoided through project design.

The Tasiast facilities operate under an environmental management system (EMS) that specifies activities to be planned and implemented by the mine's environmental management team. The EMS incorporates the project design and management, mitigation strategies and performance monitoring commitments outlined in the environmental assessments, applicable legislation and specific permit requirements.

An element of each EIA prepared for the project is a preliminary reclamation and closure plan and associated cost estimate. The preliminary reclamation and closure plan outlines the measures that will be taken to reclaim and close the proposed activities assessed in each EIA. The preliminary reclamation and closure cost estimate forms the basis of the financial assurance which is required prior to commencing activities outlined in the EIA. The current financial assurance for the existing operation is approximately $6.2 million. At least two years before entering closure, a detailed reclamation and closure plan must be submitted to the appropriate ministries for approval.

20.2 Permitting

In addition to the exploitation permit No. 229 (Section 4.2) and the adjacent exploitation and exploration permits, other necessary permits for exploiting the Tasiast mine complex have been granted by the relevant Mines and Environment authorities. Phase 3 EIA for "off-site" sea water supply has been conditionally approved pending approval of additional information which has been submitted. A final EIA is under development to authorize the delivery of preassembled equipment to site. The key permits are shown in Table 20-1.

Table 20-1: Key Operating Permits and Environmental Assessments

Brief Name	Issue Date
Authorization to construct a water pipeline route to the mine	MMI Letter 090 – 23 May 2006
Authorization of water extraction (12 boreholes)	MHE Letter 560 – 24 July 2008
Original EIA permit for Tasiast Mine	MEDD Letter 407 – 27 August 2009 MIM Letter 264 – 27 August 2009
New developments EIA permit (dump leach, Tailings Storage Facility 2)	MEDD Letter 408 – 27 August 2009 MIM Letter 264 – 27 August 2009
Environmental authorization for Phase 1ai and Phase 1aii environmental impact notices	MEDD Letter 151 – 16 June 2011 MEDD Letter 166 – 10 July 2011
Environmental authorization for West Branch development (EIA)	MEDD Letter 665 – 10 October 2011 MPEM Letter 1209 – 25 October 2011
Environmental authorization for Phase 1b development (EIA)	MEDD Letter 713 – 18 October 2011 MPEM Letter 1210 – 25 October 2011
Environmental authorization for Phase 2 development (EIA)	MEDD Letter 556 – 19 July 2012 MPEM Letter 1049 – 25 July 2012
Conditional environmental authorization for Phase 3 development (EIA)	MEDD Letter 600 – 22 September 2013 MPEM Letter 844 – 6 October 2013

20.3 Socio-Economics

Mauritania is divided into 12 wilayahs (regions), one district (Nouakchott), 53 moughataas (counties) and 208 communes (communities).

The mine site is located in the Inchiri wilayah, which has a very low population density. The wilayah includes the Akjoujt moughataa and two main communities, Akjoujt and Bennichab, Akjoujt being the administrative capital with a population of approximately 8,500. The wilayah is administered by a council, directed by a governor (wali) who delegates responsibilities to ministers and administrative authorities in the regions. The basic administrative unit, the moughataa, is directed by a Prefect (Hakem) who exercises his power under the authority of the governor.

Inchiri is the least populated wilayah in the country, with the nomadic way of life being a key feature making up 20% of the total population. There tends to be a small number of nomadic people located within the vicinity of the Tasiast mine. The mine itself is located 110 km south west from the nearest permanent community of Louik, which is located in the Banc D'Arguin National Park.

The nearest industries to the site are in the towns of Boulanour, Akjoujt and Bennichab, which are respectively 120 km northwest, 150 km east–southeast and 130 km southeast from the mine site.

There are no permanent settlements within the vicinity of the Tasiast mine. However, within 30 km of the Tasiast mine, a number of isolated families have set up structures and reside, predominantly within three communities. Residents practice animal husbandry and other subsistence forms of livelihood. There are also nomadic groups that occasionally transit the area.

21. CAPITAL AND OPERATING COSTS

This section covers Capital and Operating Costs for the current 8 kt/d operation. Capital and Operating Costs for the 38 kt/d case are described in Section 24.21.

21.1 Capital Costs (8 kt/d)

The previous, 8 kt/d Proven and Probable Mineral Reserve mine plan assumed no major expansion to the existing mill and dump leach facilities; however capital would be required to sustain the existing operation. To improve operations, $23 million was spent on CIL plant modifications in 2012.

Life of mine sustaining capital for the 8 kt/d existing operation is estimated to be $792 million, for the period 2014 to 2042 (Table 21-1).

Table 21-1: Life of Mine Sustaining Capital Costs, 8 kt/d

Sustaining Capital Cost Item	Cost (US$M)
CIL Plant	$151
Dump Leach	$0
Seawater Pipeline	$247
Mine Fleet Capitalized Maintenance	$130
Mine Fleet / Non-Mine Fleet Replacement	$39
Site Infrastructure	$88
Tailings Storage Facility	$110
Processing Other (Mobile Equip)	$11
Withholding Tax on Sustaining Capex	$17
Total	$793

21.2 Operating Costs (8 kt/d)

Tasiast's LOM unit processing costs are estimated to be $21.30 per tonne processed CIL, and $17.22 per tonne processed dump leach. The cost estimate is based on mining a total of 787 million tonnes up to the year 2030. The average mining unit cost is $2.33 per tonne mined (excluding $0.19/t for capitalized maintenance).

Upon completion of mining activities, low grade stockpiles would continue to be processed by the 3 Mt/a mill until 2042. The mine plan assumes that a total of 81.8 million tonnes of CIL ore and 7.7 million tonnes of dump leach ore will be processed. The most significant component of the CIL processing cost is electric power which is approximately $6.18/t.

Royalty payments are included in the total operating costs and consist of 3% of the gross revenue of TMLSA payable to the government and a 2% net royalty on gold production payable to Franco-Nevada.

Table 21-2: Operating Cost Estimates (8 kt/d)

Operating Cost	Unit	Life-of-Mine Average 2014-2042
Mining	US$/t mined[1]	$2.33
Processing (CIL Plant)	US$/t processed	$21.30
Processing (Dump Leach)	US$/t processed	$17.22
Site Admin	million US$/a	$57.5

Notes:
1. Excludes capitalized maintenance. Mining ends in 2030 for the 8 kt/d case.

22. ECONOMIC ANALYSIS

Under NI 43-101 rules, a producing issuer may exclude the information required for Item 22 – Economic Analysis on properties currently in production for a case that does not include a material expansion of current production. Kinross is a producing issuer, the Tasiast mine is currently in production, and the 8 kt/d case does not include a material expansion. Kinross has carried out an economic analysis of the 8 kt/d case consistent with technical information in this report and confirms that the outcome is a positive cash flow.

See Section 24.22 for the economics of the 38 kt/d expansion.

23. ADJACENT PROPERTIES

There are no adjacent properties to be discussed in this Technical Report.

24. OTHER RELEVANT DATA AND INFORMATION

24.16 Mining Methods (38 kt/d)

Mining Operations

The Tasiast FS was initiated in 2013 to assess the economic viability of an expansion of the current operations to 38 kt/d following the completion of a successful PFS. The PFS considered three process plant throughput options: 8 kt/d (status quo), constructing a new 30 kt/d process plant, and constructing a new 45 kt/d process plant. The PFS also incorporated trade-off studies in which using the existing 8 kt/d process plant capacity at Tasiast in addition to a new 30 kt/d process plant was considered. These studies concluded that a single new 38 kt/d mill option would be expected to provide the optimum economics for an expanded project. Based on these results, Kinross proceeded to a full FS on an expanded Tasiast operation with 38 kt/d total throughput.

The current conventional open pit truck and shovel mining method discussed in Section 16 will continue to be used. Varied blasting techniques, such as presplit and buffer holes, will be employed to protect the pit slopes. Blasted material will be routed based on the application of cut-off grades. Cut-offs are initially based on the net block value at the pit optimization stage and later on the gold grades during scheduling. Applying cut-off controls ensures that the highest-value materials are routed to the CIL process, while lower-grade materials are routed to the stockpiles or, if appropriate, to the dump leach. Materials below the cut-off threshold are sent to the waste destinations. The material grades and potential stockpile destinations are shown in Table 24-1.

Table 24-1: Material Routing

Grade Bin	Identification	Potential Stockpile
2.0+	High grade	High grade (HG) stockpile
1.5-2.0	High grade	HG stockpile
1.0-1.5	Medium grade	Medium grade (MG) stockpile
0.6-1.0	Low grade	LG stockpile
0.4-1.0	Oxide material	Leach dump
Oxide < 0.4 Fresh < 0.6	Waste dump	Waste dump

Loading and hauling requirements will be met by maintaining the current operating mining fleet and purchasing additional mining equipment to support the processing expansion case to 38 kt/d. Equipment selection assumes extended use of the mobile mine equipment that is currently in operation at Tasiast. Estimates of future requirements are based on current operating practices, general and site experience, and take into consideration:

- Operation of several faces (shovel pits) simultaneously to meet the long-range schedule and requirements
- Use of existing and new large-scale mining equipment to lower operating costs
- Use of well-proven and advanced mine equipment technologies to improve performance
- Use of component replacement and preventive maintenance practices to minimize major equipment failures
- Design of constraints to accommodate the selected fleets in both direct production and support roles.

The current configuration of the existing shovel and haul truck fleets will be used for the duration of mining and no replacement of this equipment is anticipated for the remainder of the LOM in the 38 kt/d case. Equipment life has been projected from actual operating hours, with estimates of future usage based on the mine plan.

Mining at the 38 kt/d throughput rate will be performed by the current mining fleet and supported by new purchases shown in Table 24-2. There are plans to convert two of the existing and underused Komatsu HD785 haul trucks to HD785 water trucks when the two older existing water truck reach the end of their useful lives. The timing for the equipment purchases is informed by analysis of the LOM plans in the CAPEX model.

Table 24-2: 38,000 t/d Total Mobile Equipment Schedule

Mobile Equipment	On-site	Required	Additional
Haulage			
Cat 793	42	50	8
Komatsu 785	22	15	-
Loading			
Cat 6060/340	7	7	-
PC 1250	6	6	-
Drills			
Pantera 1500/1100 (5m)	5	5	-
Pantera DR580 (5/10m)	2	5	3
Bucyrus SKFX (10m)	8	15	5
Atlas PV 235 (10M)	2	-	-
Major Support Equipment			
D10T Bulldozer	8	8	-
D11T Bulldozer	2	2	-
D6R, 8T, & 9R	5	5	-
Cat 994 Loader	2	2	-
Cat 854K Dozer	3	3	-
Cat 773G Lube Truck	4	5	1
Cat 16M	4	4	
Cat 24M	3	3	
Water Truck	4	4	

Mine Design

The basis for the ultimate pit design is the economic LG pit shell generated using CAE's NPVS software package. The pit limit was generated by applying a $1,200/oz gold price, operating costs and geotechnical criteria to the Measured and Indicated resource. An economic model was created within the NPVS program and the final pit limit was generated using an industry standard LG algorithm. The resulting optimized economic pit shell does not include access ramps and is not restricted by equipment or mining limitations. The final design pit includes these considerations while maintaining as much of the LG guidance as is feasible. The final pit design includes compliance with the geotechnical recommendations for slope design parameters, catch benches, and haul road width. The bench height was selected to ensure that the existing fleet of Caterpillar 6060 and Komatsu PC1250 excavators are able to mine safely. The design also places pit exits close to the various material destinations, utilizes flat switchbacks wherever possible, maintains ramps with a gradient of 10% and continues the use of 20 m wide roads at Piment and 35 m wide roads at West Branch as with the 8 kt/d case. Figure 24-1 shows a typical section through the pit that compares the ultimate designed pit to the LG pit shell. The designs are well within acceptable design tolerance of the shells that informed them.

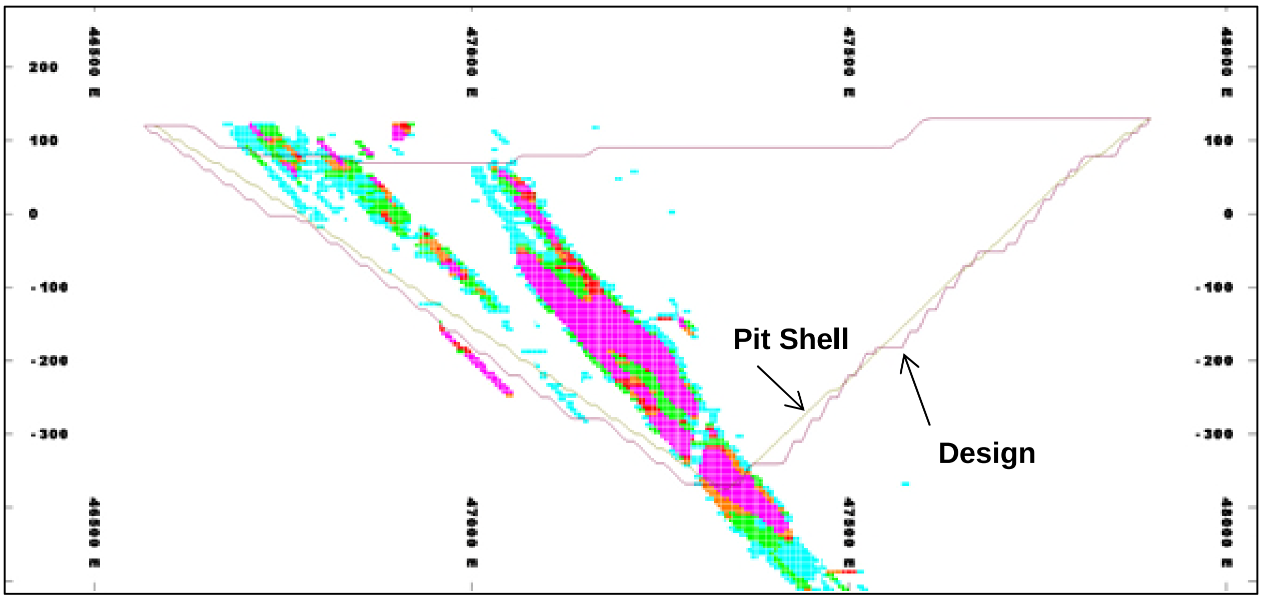
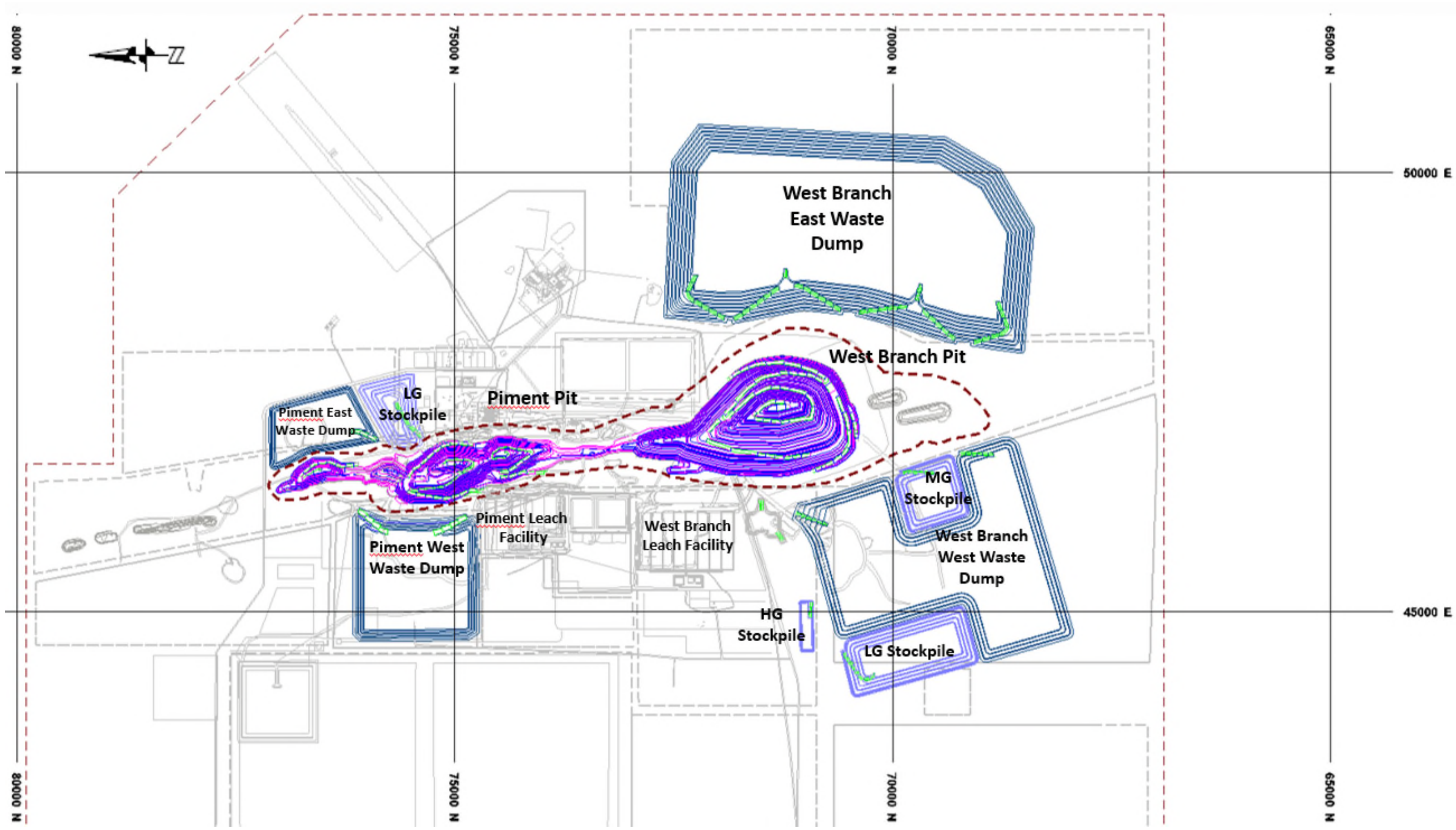


Figure 24-1: Comparison of Designed Ultimate Pit Versus LG Pit Shell

A new process facility and associated primary crusher will be located northwest of the West Branch pit. Prior to the commissioning of the new process plant, CIL ore will be routed to the existing facility located to the east of the Piment pit (Figure 24-2).

Figure 24-2: Piment and West Branch Mining Limits and Infrastructure Layout



Waste Dumps

Currently, two waste dumps along the east and west sides of the existing Piment Pit and the west side of West Branch pit are used to dispose of waste rock. Due to the expansion of mine operations and an increase in mined waste tonnage, the dumps will be expanded and an additional waste dump will be located east of the West Branch pit. Based on the feasibility study mine plan, the waste dumps will require 932 Mt of capacity after 2016. Excess capacity is available in all waste dumps.

Geotechnical Considerations

Slope design parameters and geotechnical considerations applied to final pit design in the 38 kt/d scenario are the same as those applied to the 8 kt/d case and detailed in Section 16.

Production Schedule

The current Proven and Probable Mineral Reserve at the end of 2013 was estimated to be 176 million tonnes at a grade of 1.71 g/t Au and containing a total of 9.6 Moz Au. The mining rate recommended for this project is a nominal ~120 Mt/a to a maximum of 10 benches per pushback. The mining schedule was optimized with a maximum of 50 Caterpillar 793 haul trucks in West Branch and a maximum of 14 Komatsu 785 haul trucks in Piment, such that the total material moved varies each year (Table 24-3). The 38 kt/d case requires approximately 100 Mt/a of waste stripping.

Table 24-3: 38 kt/d Case Mine Production Schedule

	Piment			West Branch			Total Mined
	Ore	Gold	Waste	Ore	Gold	Waste	
	Thousand t	g/t Au	Thousand t	Thousand t	g/t Au	Thousand t	Thousand t
2014	1388	2.09	4,972	6,101	0.97	42,630	55,092
2015	-	-	-	3,244	1.95	45,086	48,330
2016	338	1.3	11,479	14,594	1.34	105,199	131,610
2017	1001	1.28	9,650	13,632	1.24	91,746	116,029
2018	594	1.78	9,576	23,861	1.81	76,523	110,554
2019	1331	1.14	17,309	12,730	1.87	85,736	117,106
2020	3137	1.33	12,467	7,411	1.29	88,640	111,654
2021	2537	1.55	6,786	16,584	2.15	90,613	116,521
2022	2275	0.88	13,702	12,091	1.79	90,680	118,748
2023	4048	1.46	8,412	4,183	1.89	87,196	103,838
2024	2491	1.19	7,417	8,025	1.3	67,189	85,122
2025	2254	1.07	5,823	10,883	2.3	27,117	46,078
2026	3360	1.72	3,770	10,134	2.45	11,866	29,130
2027	- -	-	-	5,257	2.31	3,148	8,405
LOM	24,755	1.39	111,364	148,730	1.77	913,369	1,198,217

Notes:
1. Mine production schedule is based on the final pit design, diluted mine model, and the end of 2013 surface.
2. Excludes ore tonnes and gold ounces in starting stockpile mined pre-2014.

In 2013, the mining rate was increased to 80 million t/a by initiating advance stripping. Due to the drop in gold price experienced in 2013, the mining rate was decreased to 50 million t/a for 2014 and 2015, while waiting for the completion of the feasibility study and the go-ahead decision in 2015. High mining rates in 2016 and 2017 will compensate for these lower rates and will begin building high grade and medium grade stockpiles.

The mining rate was optimized to feed the existing process plant and to prepare for the process plant expansion in 2018. The existing 8 kt/d process plant will continue to operate until the end of 2017, when the new process plant is commissioned. The new process plant is anticipated to start up in January 2018 and achieve full production eight months later in September 2018. Table 24-4 shows the process plant feed schedule by year.

Table 24-4: 38 kt/d Case Process Plant Feed Schedule

	CIL Process Plant				Dump Leach			
	Ore Thousand t	Au Grade (g/t)	Contained Gold Thousand oz	Gold Recovery (%)	Ore Thousand t	Au Grade (g/t)	Contained Gold Thousand oz	Gold Recovery (%)
2014	2,794	2.10	189	91.4%	4,990	0.55	89	62.3%
2015	2,891	2.08	193	89.4%	-	-	-	-
2016	2,920	2.87	269	93.2%	2,798	0.89	80	69.4%
2017	2,920	2.57	241	93.1%	226	0.92	7	72.6%
2018	11,890	2.82	1,077	93.2%	235	0.87	7	74.7%
2019	13,880	2.04	911	92.9%	152	0.91	4	74.6%
2020	13,880	1.47	657	92.5%	-	-	-	-
2021	13,880	2.56	1,144	93.2%	-	-	-	-
2022	13,880	1.68	750	92.8%	-	-	-	-
2023	13,880	1.3	579	92.5%	-	-	-	-
2024	13,880	1.15	512	92.3%	-	-	-	-
2025	13,880	2.02	899	93.1%	-	-	-	-
2026	13,880	2.23	994	93.1%	-	-	-	-
2027	13,880	1.33	596	92.5%	-	-	-	-
2028	13,880	0.74	330	91.5%	-	-	-	-
2029	4,917	0.74	117	91.6%	-	-	-	-
LOM	**167,132**	**1.76**	9,457	**92.7%**	**8,401**	**0.69**	187	**66.4%**

Notes:
1. Existing CIL process plant feed is shown 2016-2017, 38,000 t/d process plant feed is shown 2018+.
2. Process plant feed schedule includes ore tonnes and gold ounces mined pre-2014, processed from stockpile.
3. Dump leach process feed excludes 0.15 M t of ore and 2.5 k oz of gold because it is mined after dump leach processing is shut down.

Once mining operations have been completed in 2027, the CIL plant will continue processing the low grade stockpiles that will have developed over the course of mining.

Personnel Requirements

The current mine operation at Tasiast is owner-operated and applies conventional open-pit operational practices, with drilling, blasting, loading, hauling, support and administrative functions. The mine operates 365 scheduled days per year and 24 hours per day, primarily divided into two 12-hour shifts per day for mine operations and mine maintenance.

The maximum mining rate planned under the expansion case mine plan is 132 million tonnes in 2016. For the first five years of new CIL plant operations, the expansion case mine plan requires a mining rate of between 110 and 120 million tonnes per year. The increased mining rate planned for the expansion case will be achieved by leveraging and expanding the existing mine operations and management discussed in section 16.

Mining cost estimates for the expansion case include appropriate staffing levels for mine operations, based on present staffing levels for the existing organization and future mine plan requirements. Expansion case requirements for equipment operators and mechanics, along with the requisite supervisors and support vehicle operators, have been projected in line with the vehicle operating hours required to realize planned tonnage as per the expansion case mine plan. Expansion case mining cost estimates have been benchmarked against mines similar in scope to the expanded Tasiast mine.

Table 24-5 outlines staffing levels for the expansion plant by functional group. The expansion of the plant requires a significant increase in personnel from 2015 to 2016 to accommodate the increase in mined tonnage. This personnel increase is concentrated in mine Operations (which includes drilling, blasting, loading and hauling) and mine maintenance, in which additional mechanics will be required to accommodate the larger mining fleet.

As indicated in Table 24-5, total staffing levels will continue to increase after 2016 to accommodate the increased vehicle hours required by the expansion case mine plan. For example, as the pit deepens and mining rates are sustained after 2017 (resulting in increased haulage distances and drilling requirements), mine operations personnel will rise from 400 in 2016 to 441 by 2019.

Table 24-5: Expansion Mining Personnel by Function by Year

	Site Budget		Planned Personnel for Expansion Mine Plan													
	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029
Tonnes Mined (million t)	**55**	**48**	**132**	**116**	**111**	**117**	**112**	**117**	**119**	**104**	**85**	**46**	**29**	**8**	**-**	**-**
Mine Management	4	4	4	4	3	3	3	3	3	3	3	3	3	-	-	-
Mine Operations	328	312	400	397	401	441	429	420	435	397	350	240	171	74	-	1
Mine Maintenance	265	274	368	377	379	403	393	388	398	378	333	220	149	72	21	22
Mine Technology	13	12	12	12	12	13	13	13	13	11	11	11	11	6	-	-
Mine Technical Services	33	33	36	35	33	45	45	45	45	31	31	31	16	8	-	-
Geology	21	21	21	21	21	24	24	24	24	19	19	18	12	6	-	-
Mining Total	**664**	**656**	**841**	**846**	**849**	**929**	**907**	**893**	**918**	**839**	**747**	**523**	**362**	**165**	**21**	**23**

As a result of the limited number of skilled mining personnel in Mauritania, it is expected that expatriate staff will be recruited to fill key senior management and technical roles, as well as key operational roles, such as excavator operators and operator trainers. Over time, as members of the national workforce develop mining skills and expertise, selected roles will no longer be filled with expatriate employees, and expatriate staff numbers will be progressively reduced. Table 24-6 presents the gradual replacement of expatriate personnel with national personnel as more roles can be filled by Mauritanians.

Table 24-6: Expansion Expatriate and National Mining Personnel by Year

	Planned Personnel for Expansion Mine Plan													
	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029
Expatriates (Non-operator, Non-mechanic)	57	65	62	58	45	45	33	20	20	12	8	4	1	1
Nationals (Non-operator, Non-mechanic)	142	145	145	180	193	193	205	172	167	153	92	50	20	20
Operators	349	346	350	375	364	355	370	351	304	194	143	61	0	1
Mechanics	293	290	292	316	305	300	310	296	256	164	119	51	0	1
Mining Total	**841**	**846**	**849**	**929**	**907**	**893**	**918**	**839**	**747**	**523**	**362**	**165**	**21**	**23**

24.17 Recovery Methods (38 kt/d)

General

Kinross would close down operation of the existing Tasiast 8 kt/d CIL plant before start-up of the new expansion plant. Operating and maintenance staff would be transferred to the new plant.

The existing plant would be shut down in an orderly fashion, including cleaning out of all gold removed from the carbon and from the mills, tanks and other equipment.

Design Criteria

The design capacities for the crushing plant and process plant use 70% and 92% effective operating time. Based on test work, the key nominal design criteria for the major process circuits are summarized in Table 24-7. It should be noted that design factors accounting for higher gold grades were applied when required to match the mine plan. The plant design life is 15 years.

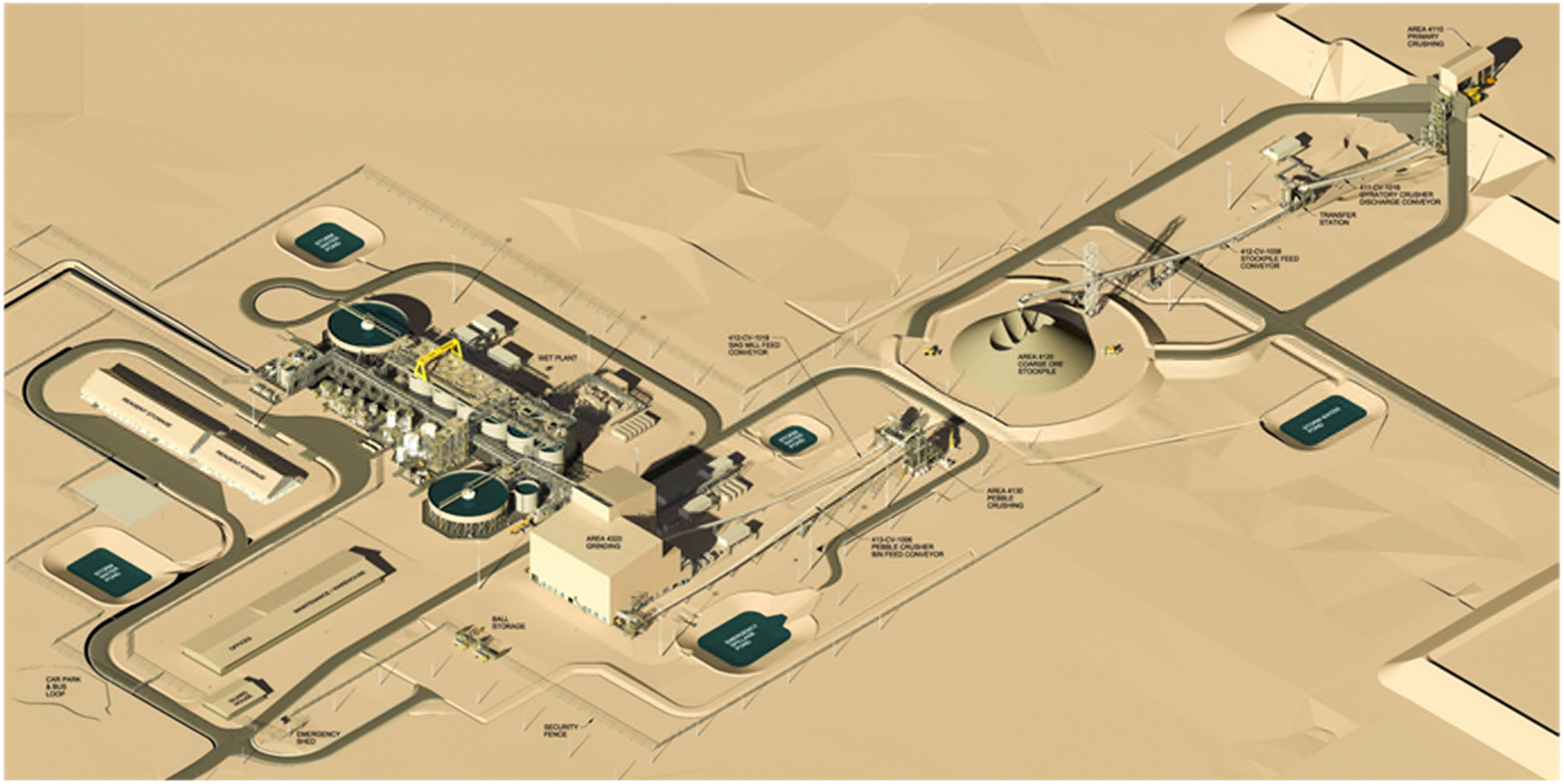
Table 24-7: Key Process Design Criteria

Area	Criteria	Unit	Nominal Value
General	Gold	g/t	2
	Daily throughput	t/d	38,000
	Process plant availability	%	92
	Annual gold production	oz/y	826,844
Primary crusher	Availability and utilities	%	70
	Crusher work index	kWh/t	14.40
	ROM top size	Mm	.1,100
	Crusher product size (P_{80})	mm	150
Ore storage	Capacity (live)	t	20,000
	Capacity (live)	h	12
Grinding and pebble crushing	Bond ball mill work index	kWh/t	13.3
	JKTech Axb		31.4
	SAG mill product size (P_{80})	µm	2,000
	Ball mill product size (P_{80})	µm	90
	Pebble crusher feed size	mm	70
	Nominal feed to pebble crusher	t/h	430
Gravity recovery	Gold recovery	%	31
	Feed % to gravity circuit (cyclone underflow)	%	27.0
Screening and thickening	Feed density	% w/w	30.3
	Thickener underflow density	% w/w	50
Leaching and CIL	Leach retention	h	12
	CIL residence time	h	12
	CIL carbon retention time	d	14
	CIL carbon concentration	g/L	12
Tailings thickening	Thickener underflow density	mg/L	65.0
Cyanide destruction	Discharge solution CN_{WAD}	mg/L	45.0
	Residence time	h	1
Carbon treatment	Stripping solution flow rate	BV/h	2
	Operating temperature	°C	140
	Operating pressure	kPa	450
	Number of elution vessels		2
	Elution batch size	t	15
Electrowinning (EW) and refining	EW recovery	%	98

Location Study

The Tasiast Expansion Project is a brownfield expansion; however, the new process plant is located in a separate location within the mine lease boundaries. Figure 24-3 shows the 38 kt/d CIL Plant Layout.

Figure 24-3: CIL Plant Layout (38 kt/d)



Engineering Design Basis

Material Selection

The environmental and plant conditions will be extreme and will require careful selection of construction materials to suit the 15-year design life of the facility. Erosive process slurries will cause wear to plant equipment and piping. Strong acids and bases, cyanide and the use of seawater as process water and wash water will expose most of the wetted equipment to corrosion conditions. Reference documents for guiding the selection of construction materials were provided in appendices in the 2014 Feasibility Study Report by Hatch on the Tasiast Expansion Project.

Solution and Slurry Containment

All wet areas of the process plant will be bunded with a containment volume equal to 110% of the volume of the largest tank in the containment area, or 25% of the total combined tank volume in the case of hazardous materials. All wet areas in the thickener, leach and CIL sections will be connected to optimize containment. An emergency spillage pond will be provided to the south of the grinding building to contain excess slurry spillage that cannot be contained within the grinding building bunded floor area.

Areas with specific requirements (design code requirements or incompatibility of solutions, such as hydrochloric acid and sodium cyanide) will be provided with separate containment.

Secondary containment will be provided for major spills. Spills will be collected in a separate process spillage pond adjacent to the process plant.

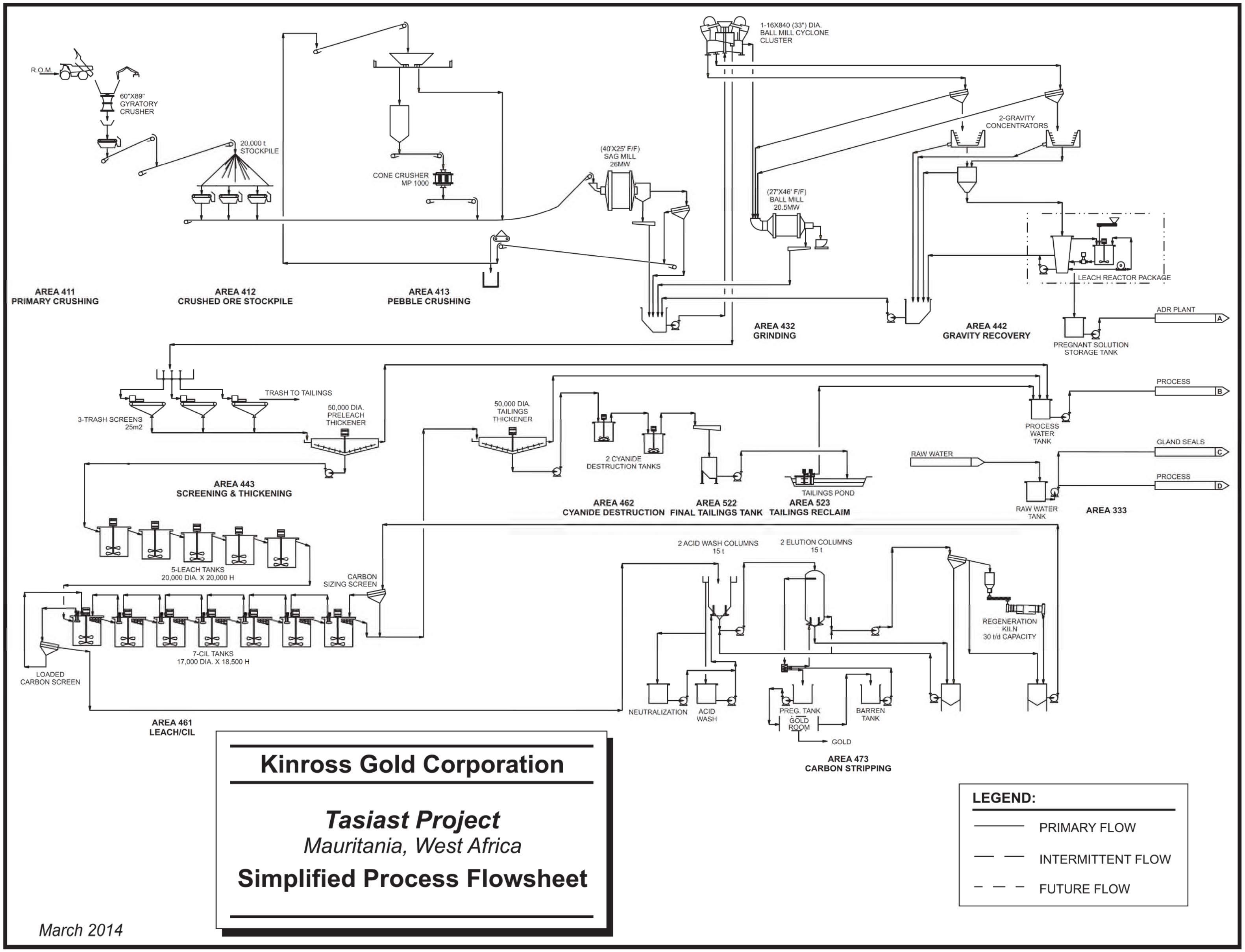
Interfaces Between Process Facilities

Gold containing pregnant solution generated by the leach reactor and the carbon treatment plant at the new CIL plant will be processed at the exiting ADR plant. The resulting barren solution from the ADR plant will be pumped back to the carbon treatment plant and the new CIL plant.

New CIL Plant

A simplified overall process flow diagram is illustrated in Figure 24-4.

Figure 24-4: Simplified Overall Process Flow Diagram (38 kt/d)



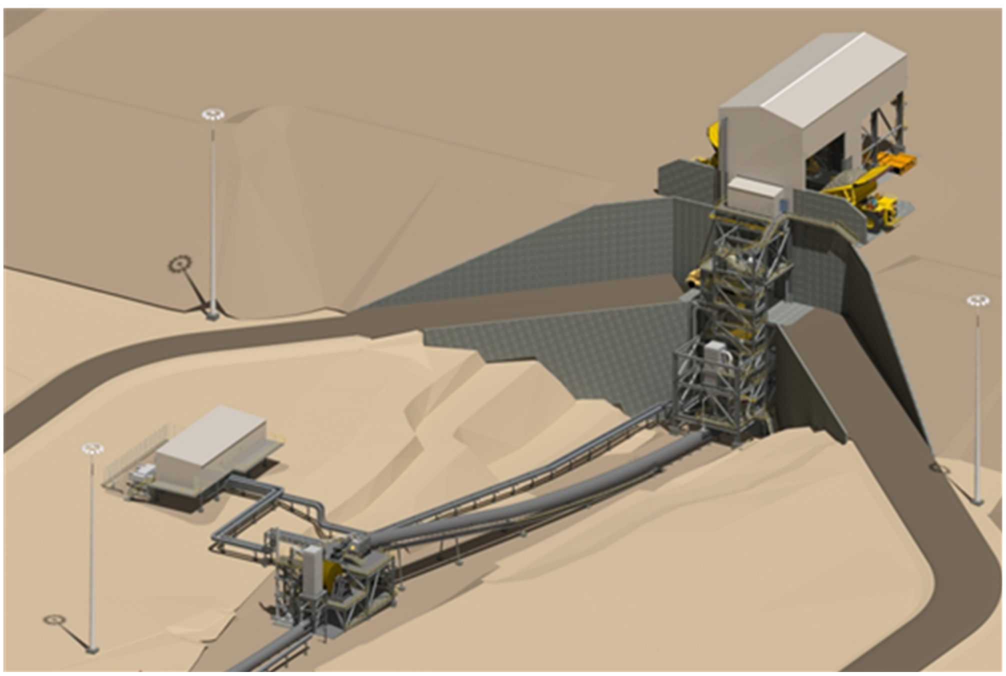
Primary Crushing

The primary crushing facility will include a gyratory crusher, rock breaker and an apron feeder with two belt conveyors in series to convey crushed ore to the ore stockpile.

ROM ore with a top size of approximately 1,100 mm will be delivered via 225 t capacity Caterpillar 793 rear-dump haulage trucks, or 90 t capacity Komatsu 785 rear-dump haulage trucks, to the gyratory crusher. The hopper layout will allow both types of trucks to dump from two sides to the gyratory crusher. Ore will be dumped into a 450 t capacity hopper directly over a gyratory crusher 60 in. x 89 in., 1,524 mm diameter by 2,261 mm long, 600 kW.

The primary crushing layout is shown in Figure 24-5.

Figure 24-5: Primary Crushing



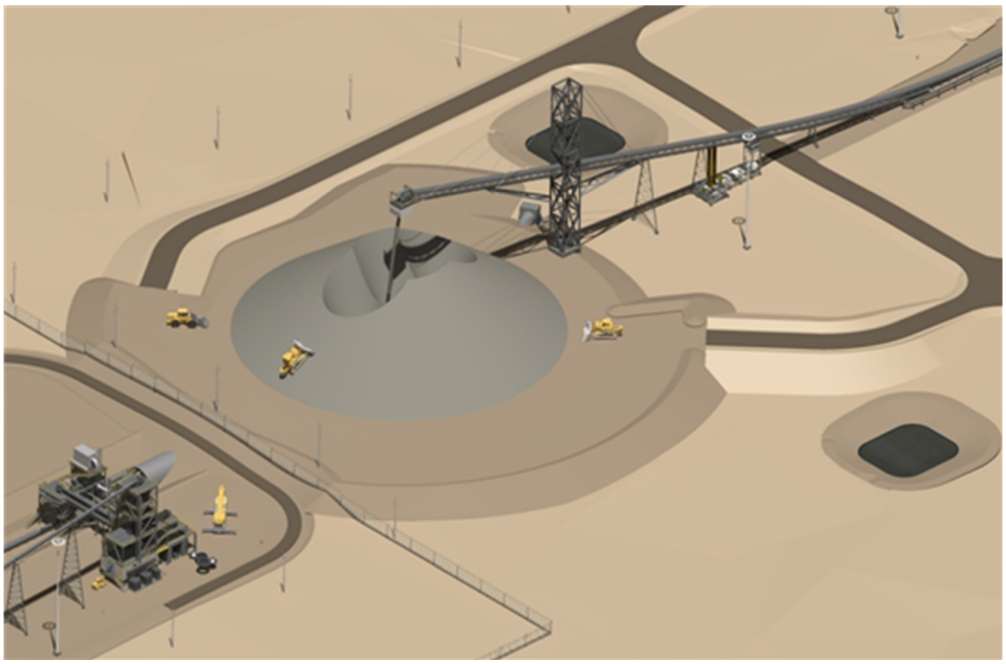
Coarse Ore Storage

The ore storage facility will consist of a crushed ore stockpile with three in-line apron feeders located in a tunnel under the stockpile. The apron feeders will transfer ore from the stockpile to the conveyor feeding the SAG mill.

The stockpile will have a live capacity that can support process plant operation for about 12 hours when the gyratory crusher is not operating. Dead ore will be recovered by a bulldozer and front-end loader and will be pushed to the apron feeders, if the gyratory crusher is unavailable for extended periods. The total capacity of the stockpile, including the dead ore, is approximately 100,000 t.

The ore storage layout is shown in Figure 24-6.

Figure 24-6: Ore Stockpile



Grinding Area

The grinding circuit will consist of a SAG mill and one ball mill operating in closed circuit with a hydrocyclone cluster. The ball mill and SAG mill will share a common pumpbox and arranged in a side-by-side configuration.

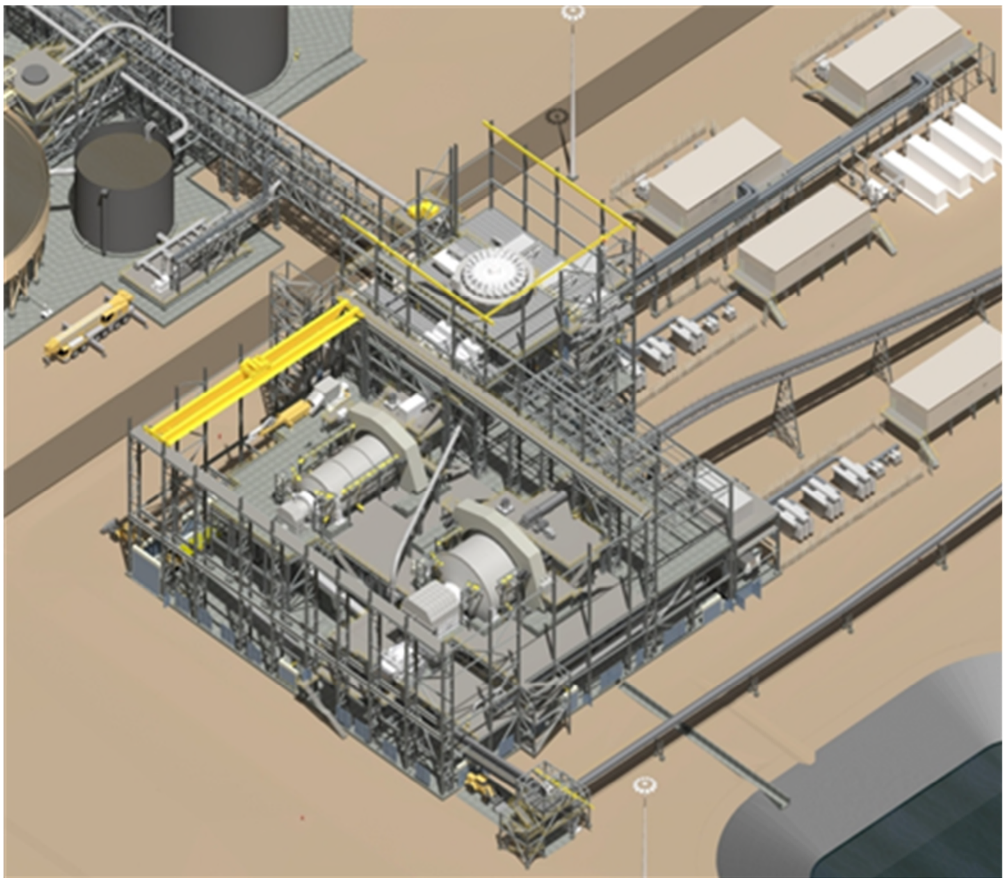
Grinding will be performed by a 12.2 m diameter by 7.6 m long (flange to flange), 26,000 kW grate discharge SAG mill and a 8.2 m diameter by 14.0 m long (flange to flange) 20,500 kW overflow discharge ball mill. Ore from the stockpile will be fed to the SAG mill via the SAG mill feed conveyor. Crushed pebbles from the pebble crushing plant also will be added to the SAG mill feed conveyor.

The SAG mill trommel screen oversize (pebbles too big to pass through the screen) will discharge onto a rail-mounted double deck vibrating screen where the remainder of the slurry will be removed, and pebbles will be sent to the pebble crusher.

The hydrocyclone cluster will classify the feed slurry into coarse and fine fractions.

The grinding circuit layout is shown in Figure 24-7, with the building sheeting removed for clarity.

Figure 24-7: Grinding Circuit Layout – Building Sheeting Removed for Clarity



Pebble Crushing

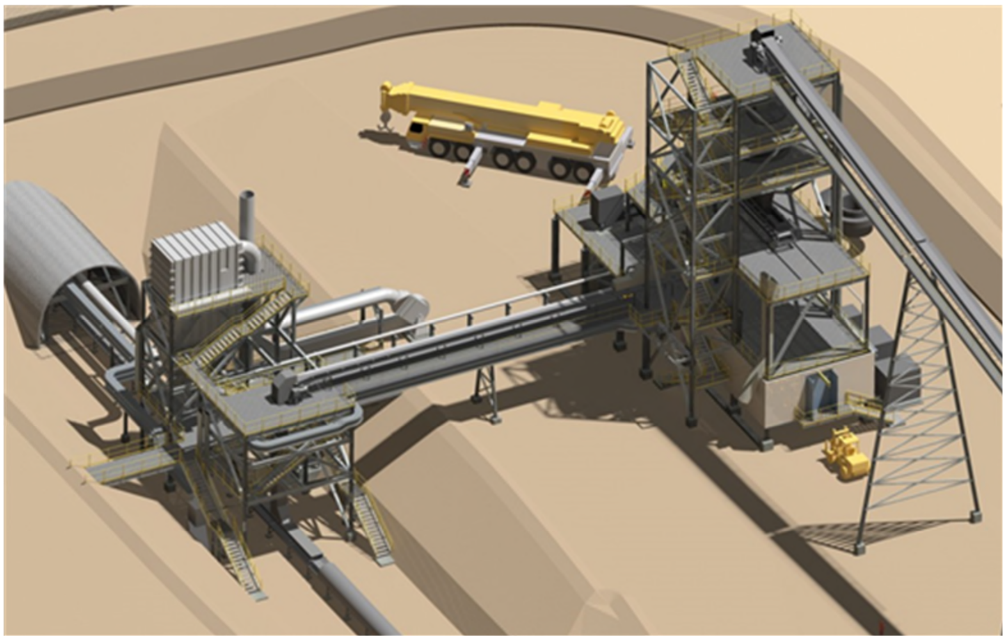
The pebble crushing plant will consist of a 60 m^3 capacity surge bin, a pebble crusher (MP1000 [746kW]) operating as an open crushing circuit, and associated belt.

Pebbles from the SAG mill will be conveyed by belt conveyor to the pebble crushing plant and feed the surge bin. A travelling rock box at the head of the bin feed conveyor and bypass chutes will allow alternative pebble flow either to the surge bin or to a belt directly feeding the SAG mill feed conveyor.

The pebble crusher product will discharge onto the crushed pebble return conveyor that will return the material to the SAG mill feed conveyor.

The pebble crushing layout is shown in Figure 24-8.

Figure 24-8: Pebble Crushing



Gravity Recovery and Intensive Leach Circuits

The gravity recovery and intensive leach circuits will consist of two XD-70 centrifugal gravity concentrators with feed-scalping screens, concentrate hoppers and a skid-mounted intensive leach unit. The equipment will be located on the east side of the grinding area.

A portion of the hydrocyclone underflow from the grinding area will be directed to two scalping screens to remove coarse particles.

Gravity concentrate from the centrifugal concentrators will be batch processed in the intensive leach circuit.

At the completion of the batch leach cycle, the resulting gold-rich pregnant solution will be pumped to a pregnant-solution tank, and from there pumped to the ADR plant to recover the gold.

Screening and Thickening

The screening and thickening area will include three parallel linear trash screens, a pre-leach thickener and a tailings thickener.

Hydrocyclone overflow will flow by gravity to a distributor where the stream will be split among three linear trash screens to remove tramp material. The linear screens will each have an area of 25 m^2 and have a screen opening of 700 µm. The oversize rock particles and trash from the screens will be collected in a bunker and periodically picked up and trucked to the tailings area for disposal. The undersize from the screens as slurry will be collected in a pipe and flow by gravity to the pre-leach high-rate thickener via a feed tank.

The pre-leach thickener will have a diameter of 50 m, a high-rate feed well design, and will produce a thickened slurry of 50% solids to feed CIL circuit.

CIL tailings slurry will flow by gravity to the tailings thickener via the thickener feed tank, together with reclaim water and make-up raw water to wash the solids and recover and recycle the cyanide remaining in the slurry.

Overflow from the thickeners will flow by gravity to the process water tank for reuse in areas of the process plant.

The layout of pre-leach and tailings thickening are shown in Figure 24-9 and Figure 24-10.

Figure 24-9: Pre-Leach Thickener and Process Water Tank

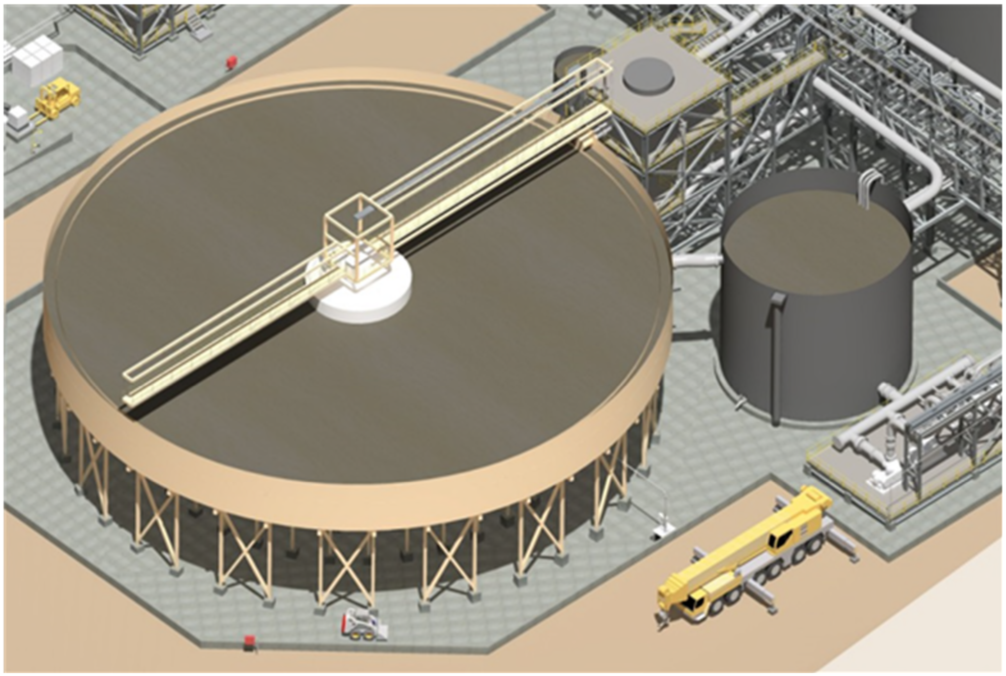
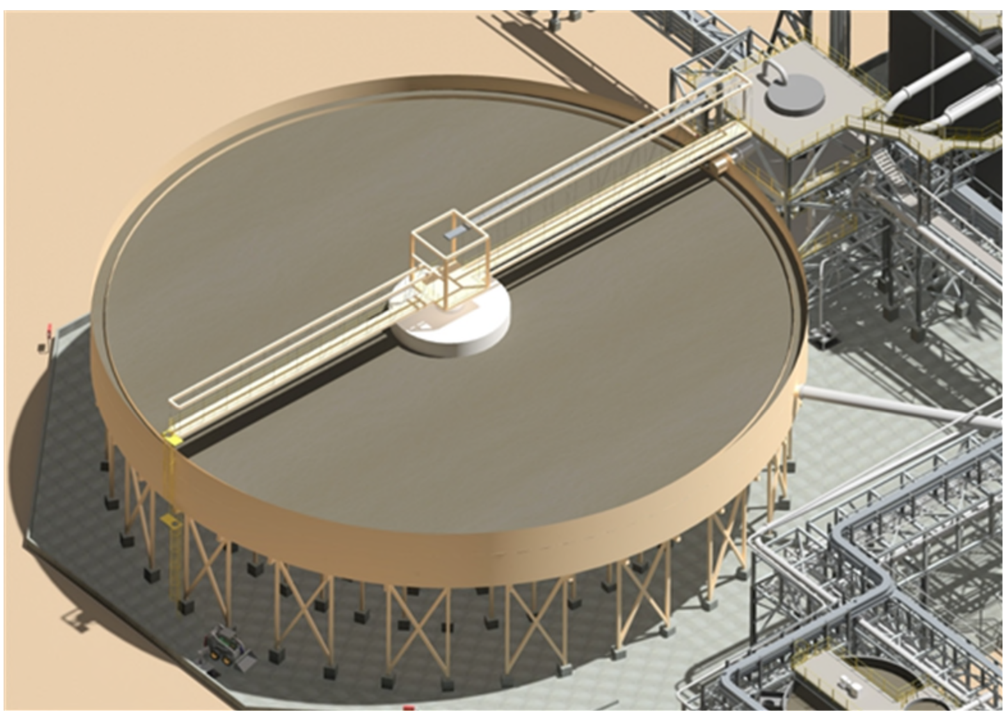


Figure 24-10: Tailings Thickener



Cyanidation - Leaching and CIL

The leach circuit will comprise five mechanically agitated leach tanks (20 m diameter by average 20 m high) and seven mechanically agitated CIL tanks (17.5 m diameter by 18.5 m high)

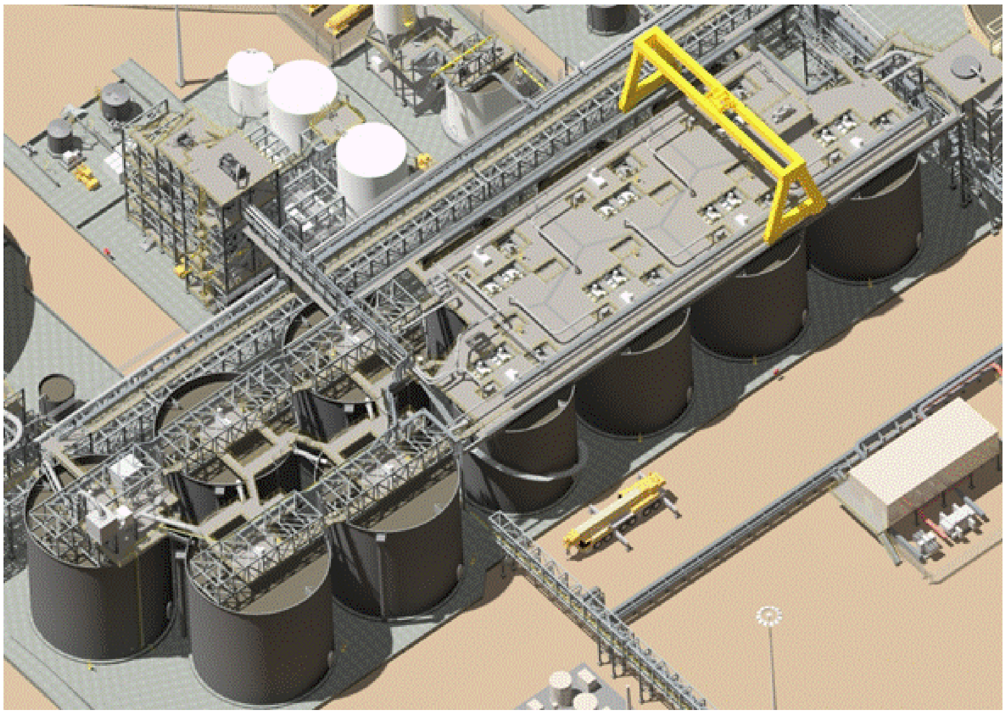
The pre-leach thickener underflow will be pumped to the first leach tank and leached in a weak cyanide solution to dissolve gold. The leach circuit will increase gold concentration in the solution before contact with activated carbon in the CIL circuit. The leach circuit has been designed to provide 12 hour leach retention time, with an additional 12 hours in the CIL tanks for a total of 24 hours. The process water returning to grinding will also contain cyanide, so additional, well-agitated and aerated leaching will occur before entering the leach tanks. Oxygen will be sparged at the bottom of the leach tanks.

In the leach section, the ore slurry will flow by gravity from tank to tank. The leach tanks will have the same diameter (20 m), but the height of the tanks will be varied in 500 mm increments to facilitate gravity flow between the tanks. The foundation for all tanks will be at the same elevation.

Leached ore slurry from the last leach tank will flow by gravity to the CIL circuit. Dissolved gold and silver will be adsorbed onto activated carbon particles in the CIL tanks. The CIL circuit has been designed to provide 12 hours of slurry retention time.

The layout of the leach and CIL facilities is shown in Figure 24-11.

Figure 24-11: Leach and CIL Circuits



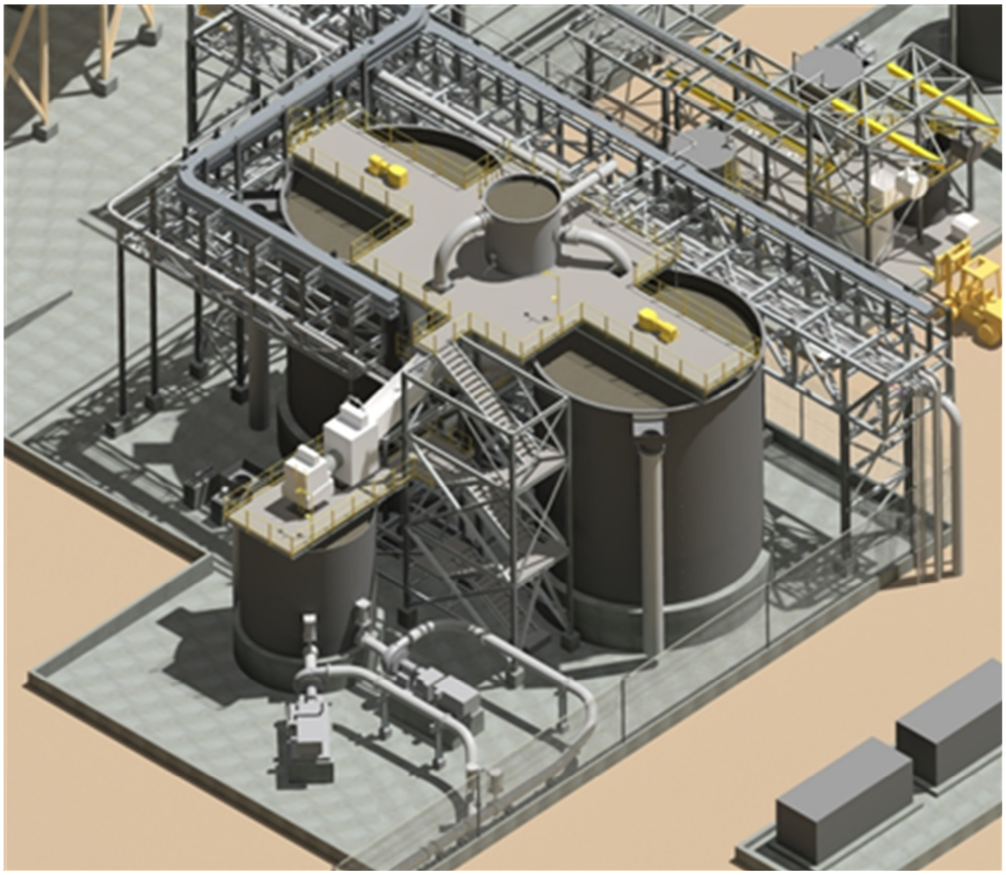
Cyanide Destruction and Final Tailings

The CND circuit will consist of two mechanically agitated tanks operating in series to provide a total residence time of one hour. The cyanide content of the slurry will be reduced to below International Cyanide Management Code effluent guidelines using the Cyanco CombinOx process. Treated slurry from the CND circuit will flow by gravity to a final tailings pump box, from where the tailings will be pumped to the tailings impoundment facility.

The CND circuit will treat thickened slurry from the tailings thickener, process spills from various contained areas and process bleed streams.

The CND layout is shown in Figure 24-12.

Figure 24-12: Cyanide Destruction Layout



Carbon Treatment

The carbon treatment circuit design has been based on a modified Zadra design, and will consist of acid wash, carbon elution (stripping) and carbon regeneration. This circuit will recover the gold from the carbon and reactivate the carbon for reuse in the CIL circuit. Two trains will be provided to operate in batch mode. The circuit has been designed to complete two carbon strips per day and process 30 t/d of carbon for the 38,000 t/d plant. Each train will treat 15 t batches of carbon.

Electrowinning and Refining

Electrowinning and refining equipment for the new CIL plant will be located in the existing dump leach ADR plant, and will consist of electrowinning cells with associated equipment to handle the barren solution and sludge from the electrowinning cells. A smelting furnace and dust scrubber will be added to the gold room in the ADR plant to handle the additional gold production.

Tailings Reclaim

Water will be reclaimed from the tailings impoundment facility for reuse in the process plant via a decant tower and submersible pump system.

Tailings reclaim water will be used as wash water in the tailings thickener.

Dump Leach and ADR Plant

Oxidized or weathered low-grade material not meeting the minimum gold content required for processing in the CIL plant is processed by heap leaching of ROM material (without crushing) in the dedicated Piment and West Branch dump leach pads. In this process, weathered, low-grade material from the mine as ROM material is treated with dilute cyanide solution.

According to the preliminary FS mine plan the placement of ore on dump leach is anticipated to cease in 2019. This process will be followed by a period of two to four years of continued irrigation, first to recover all of the available gold, and then with raw water to rinse residual cyanide solution from the leached ore before closure.

Dump Leach

The current dump leaching operation treats uncrushed ROM rock from the open pit on the dump leach facility on the west side of the open pit. The tonnage treated annually averages 5.5 Mt/a, although daily rates vary according to the availability of suitable material from the mine.

The facility comprises a graded and lined pad on which the ore is placed in layers 10 m high. Once the dumping cycle is finished, the dump is dosed with dry hydrated lime for pH control and irrigated with dilute cyanide solution through a surface network of pipes and drippers. Solution percolates through the dump, collects at the base and drains to a collection pond. Two-stage counter current leaching is used to maximize gold recovery and minimize water consumption. Three ponds are used for solution management:

- Barren (feed) solution and make-up water
- Intermediate leach solution
- Pregnant (final product) leach solution

A new dedicated ADR plant to process solutions from the dump leach pads was designed, constructed and commissioned in October 2011, and was operational by

January 2012. The ADR plant treats pregnant leach solution from both the Piment and West Branch dump leach pads and returns barren solution to these systems.

ADR Plant

The current ADR plant comprises a series of carbon contacting columns (each containing 6 t of carbon) where pregnant solution pumped from both dump leach operations is contacted with activated carbon to absorb the gold. Solution flows from one column to the next by gravity, ensuring gradual removal of gold. The barren solution (after gold adsorption) is pumped to the pond system at each of the two dump leach pads.

Periodically, loaded carbon from the first column is removed to the loaded carbon screen and then flows into the acid-wash vessel. Washed carbon is then transferred by pump to the stainless-steel pressure elution vessel. The caustic solution is indirectly heated using a diesel-fired boiler. After elution, the carbon is pumped from the vessel and regenerated through a rotating kiln fired by diesel oil. Treated carbon is sized and returned to the adsorption columns together with fresh carbon as a make-up.

Eluate is collected in a surge tank and then pumped to the gold room in the ADR plant for treatment in an electrowinning cell to recover gold as sludge. Periodically, the sludge is drained, filtered, mixed with fluxes and smelted to produce gold bullion, all within a secure and supervised area. The gold room is power ventilated to maintain a safe working environment.

Utilities and Reagents

Water Distribution

The following different types of water will be used in the new process plant:

- Process water: Overflow from the pre-leach thickener and tailings thickeners will be used as process water. The water will have a low gold concentration and a cyanide concentration of about 28 ppm. Process water will be used predominantly in grinding and widely in the process plant, except for areas that process tailings.

- Demineralized water: Demineralized water, or RO water, will be produced in a reverse osmosis plant located next to the raw water pond. Demineralized water will be used for flocculant preparation, in the carbon treatment circuit, in the intensive leach reactor and for sodium hydroxide solution makeup.

- Raw water: Raw water for the process plant will initially be brackish water from the water supply bore field and subsequently, after 2019, untreated seawater that will be delivered to a raw water pond on site. Water will be pumped from the raw water pond and will be introduced into the tailings thickener for wash water and make-up water. Raw water will also be used in areas that process tailings and in the slurry-pump gland seals. The permanent raw water will be supplied via a pipeline approximately 150 km long from an open seawater intake at the coast.

- Reclaim water: Water will be reclaimed from the tailings impoundment facility via a submerged pump decant system to the new process plant's water storage tanks. Reclaim water will be used as process water make-up.

- Carbon Transport Water: RO water will be used for carbon transport in the carbon treatment area. The transport water will be recirculated within the acid wash and elution circuits; carbon fines will be removed in a settling tank.

Air Distribution

The air distribution system for the new CIL plant will include instrument, plant and process air, and will be based on a decentralized system due to the significant distances between users. The following compressed air centres are planned:

- An instrument and plant air system with compressors and receivers will be provided and located in the grinding area. Instrument and plant air will be distributed to all users in the process plant.

- A smaller and separate air compressor in the crushing area will provide air for that area.

- Process air compressors will be located in a compressor house near the cyanide destruction tanks. The three compressors will supply low-pressure process air to the leach tanks (in case the oxygen plant is down) and CIL tanks.

Oxygen

A vacuum-pressure swing adsorption (VPSA) oxygen generation plant will provide oxygen gas to be used in the leach tanks. Oxygen will be sparged at the bottom of the leach tanks. The oxygen plant will be located adjacent to the process air compressors, and will be provided as a vendor package.

Reagent Mixing and Storage

<u>General</u>

In general, with the exception of lime, consumables are received in supersacs, totes and similar small containers after ocean transport in sea containers. With the exception of cyanide, the containers are emptied at the port and hauled to the site using typical highway-type trucks. Storage at the site is designed to provide continuous supply of the consumables after allowing for ocean shipping frequency, port disruptions and in-country delays. Generally, up to 45 days supply of each consumable is stored at the site.

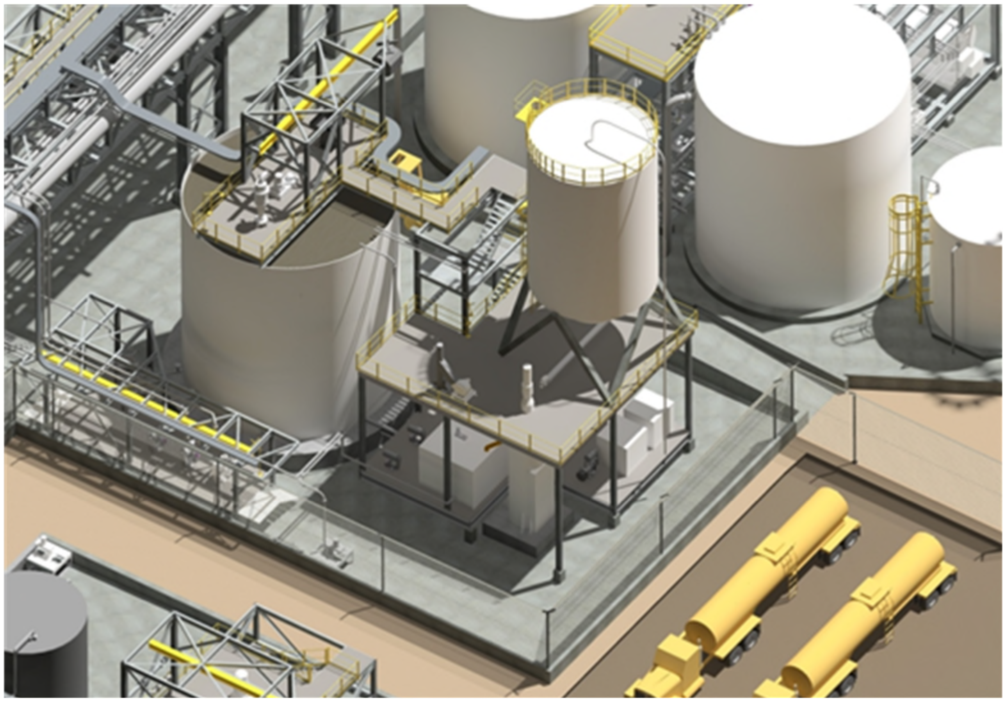
Process reagents for the new CIL plant will be stored in segregated covered structures in a separately fenced area, located northwest of the plant. A portion of this area will be used for temporary storage of empty containers.

Slaked Lime Slurry

Slaked lime slurry, produced from quicklime (CaO) received in bulk, will be used for pH control in the leach and cyanide destruction circuits. While the existing operation receives hydrated lime in one-tonne bags, the supplier has proposed to erect a facility at one of the ports to receive coarse quicklime in bulk, to store and mill it to provide a ground, 2 mm dry product and to deliver it on a just-in-time basis to the mine. The supplier will purchase and operate the pneumatic tanker trucks for delivery.

The lime slaking area layout is shown in Figure 24-13.

Figure 24-13: Lime Slaking System

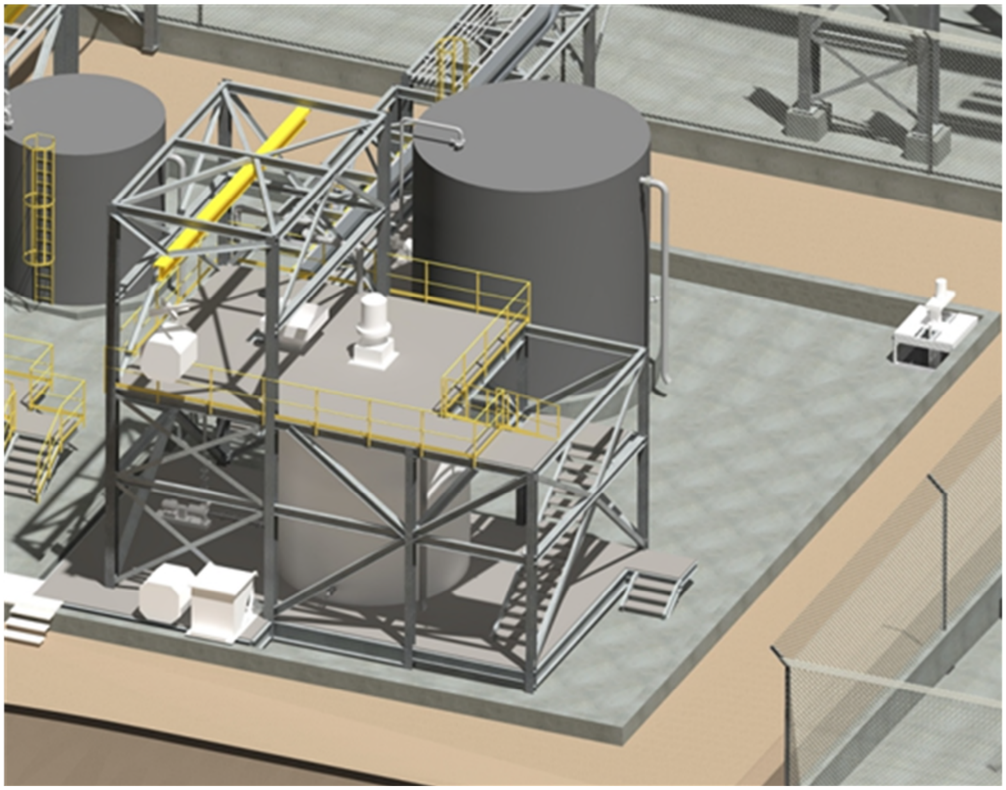


Sodium Cyanide

Sodium cyanide will be used to dissolve gold and silver in the leach circuit. Sodium cyanide will be supplied in 1 t boxed bags as solid briquettes and dissolved in process water to make a 20% w/w solution in the 57 m^3 mixing tank. Sodium hydroxide will be added to safely dissolve the solid sodium cyanide in a high pH solution. During dissolution, the solution will be maintained at a pH greater than 12 to avoid volatilization of hydrogen cyanide gas.

The sodium cyanide handling area layout is shown in Figure 24-14.

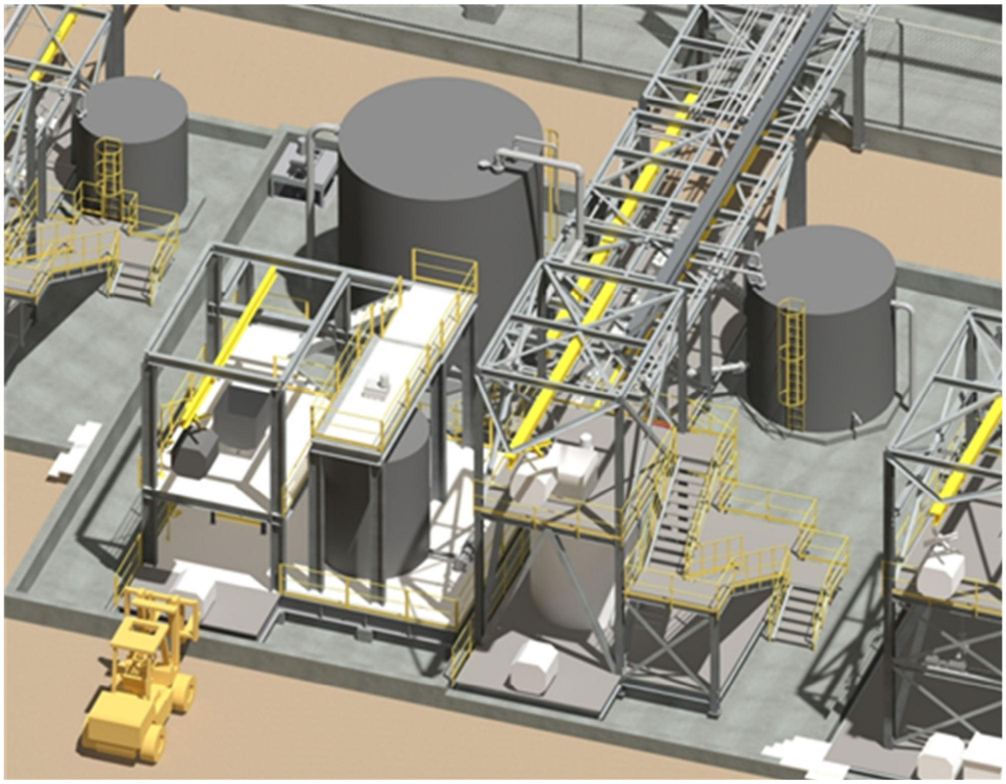
Figure 24-14: Sodium Cyanide Handling Layout



Flocculant

A packaged flocculant make-up system will be provided to supply flocculant to the thickeners. Flocculant will be supplied in solid form and mixed with demineralized water in a mix tank. The flocculant preparation system is shown in Figure 24-15.

Figure 24-15: Flocculant Preparation System



Sodium Hydroxide (Caustic)

Sodium hydroxide will be primarily used in the carbon treatment circuit. Sodium hydroxide will be supplied in solid form and dissolved in demineralized water to make a 20% w/w solution in a 20 m^3 mixing tank. The mixed solution will be pumped to the 48 m^3 supply tank, where it will be stored before being pumped to the carbon strip circuit. Both mixing and supply tanks will be located adjacent to the carbon treatment plant.

Hydrochloric Acid

Hydrochloric acid will be used in the carbon treatment circuit. The hydrochloric acid will be supplied in liquid form at 30% v/v strength in 1000 L totes. The hydrochloric acid will be pumped from the totes to a 38 m^3 mixing tank and will be diluted to a 3% v/v solution using demineralized water before being pumped to the acid-wash circulation tank at the carbon treatment circuit.

Activated Carbon

Activated carbon will be added to make up carbon losses and carbon fines removed in the carbon treatment circuit. The activated carbon will be supplied in solid form, and raw water will be added to make a mixing.

Copper Sulphate

Copper sulphate will be used as a catalyst in cyanide destruction. Copper sulphate will be supplied in solid form in 1 t bags and dissolved in raw water in a 10 m^3 mixing tank to make a 15% w/w solution. The mixed solution will be pumped to the 20 m^3 supply tank, where it will be stored before being pumped to the CND circuit.

Sodium Metabisulphite

Sodium metabisulphite (SMBS) will be used as the sulphur dioxide source for cyanide destruction. SMBS will be supplied in solid form and dissolved in seawater to make a 20% w/w solution in a 10 m^3 mixing tank. The mixed solution will be pumped to the 20 m^3 supply tank for the 38,000 t/d plant, where it will be stored before being pumping to the cyanide destruction circuit. Both mixing and supply tanks will be located close to the copper sulphate system.

Hydrogen Peroxide

Hydrogen peroxide will be used for cyanide destruction. The hydrogen peroxide will be supplied in liquid form at 70% (by volume) strength in 25 t bulk iso-containers and will be pumped from the iso-container to a 7 m^3 storage tank. Hydrogen peroxide will be pumped directly to cyanide destruction tanks.

Antiscalant

Antiscalant will be used in the carbon elution columns in the carbon treatment plant, and will be introduced via the elution barren tank. The antiscalant will be delivered to the plant in 1000 L capacity totes in liquid form, and will be pumped directly to the elution barren tank in the carbon strip circuit.

Facility Operations

Commissioning and Ramp-up

Commissioning usually begins during the final stages of construction. As the construction of a section of the plant is completed, the section is handed over to commissioning. Having both construction and commissioning underway at the same time minimizes the delay between final construction completion and the start-up of the plant.

The main phases of commissioning are:

- Completion of installation
- Inspection, testing and trials of equipment
- Systems handover to the owner by the contractor
- System testing
- Plant start-up
- Plant ramp-up

When all systems have been tested, the plant is handed over to the owners operating team. Acceptance criteria in terms of plant throughput and duration of test runs will have been agreed upon before beginning commissioning.

Start-Up

When all systems have been successfully tested and accepted, the plant is handed over to the operations manager who then takes responsibility for operating the plant.

Ramp-Up and Performance Testing

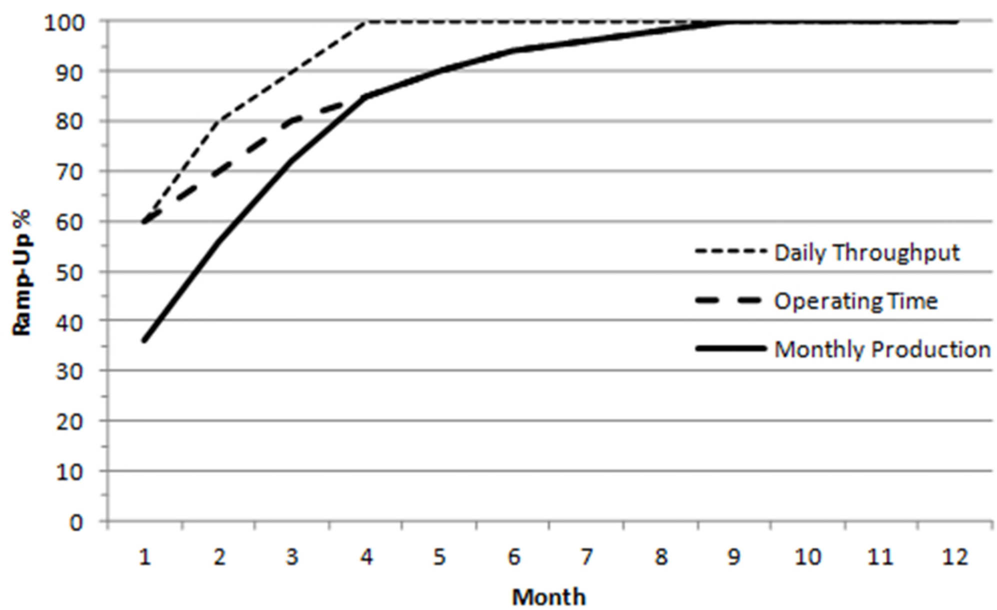
This stage of commissioning consists of the period after handover and start-up during which the plant throughput is increased to full capacity, through gaining operational familiarity and engineering out any constraints, operational or mechanical and electrical, which might occur.

This stage varies in duration, depending on the complexity of the plant. For this plant, a comparatively simple CIL gold plant, the duration is planned to be about three months.

Bringing the plant up to design metallurgical performance might take longer, and involves the optimization of mill and cyclone performance, reagent dosage and control, carbon management and operator training. This work is planned to take an additional six months.

Performance testing is usually done within an agreed period after start-up and once the plant has reached full capacity. The period varies depending on the complexity of the plant; in this case, likely twelve months. Figure 24-16 shows the ramp-up schedule for the 38,000 t/d new CIL plant with 92% operating time.

Figure 24-16: Commissioning Schedule



24.18 Project Infrastructure (38 kt/d)

This section describes significant additional infrastructure required for the expansion project.

Access Road

The access road from the highway to the site entrance would need to be upgraded with an additional 200 mm of sub-base gravel material over 11 km of the existing road, to support trailers carrying preassembly units for the expansion project.

Water Supply

The current Tasiast permit allows a total extraction of 54.8 Mm^3 of water from the water supply bore field. Based on the current demand and future forecast of the expansion project, this quota will be reached in Q2 2020. A new permanent seawater supply system will be implemented as a sustaining capital project to be executed in the future to meet the expansion project's demand beyond 2020. As such, it is not part of the expansion project but would be developed in support the expanded, 38 kt/d operation.

The seawater supply system would source raw water from the Baie du Levrier near Nouadhibou, through an open intake, and convey the raw water through a buried pipeline for approximately 140 km inland to the mine site. The system consists of the following subsystems:

- Open intake
- Main pumping station
- Power station
- Pipeline
- Raw water pond

The minimum and maximum flow rates for the system are 22,300 m^3/d and 39,600 m^3/d.

Phase 2 Power Plant

The Phase 2 power plant would be located adjacent to the Phase 1B plant and the fuel oil storage area. Power generation will be from simple cycle, reciprocating engines located in a totally enclosed engine hall with an overhead electric gantry crane for service and maintenance. Depending on the selected equipment manufacturer, medium-speed (500 rpm to 700 rpm) reciprocating units for this duty will have a site net output rating of 14 MWe to 18 MWe, running primarily on HFO with the flexibility to also operate on LFO continuously, if necessary, and for normal start-up and shutdown.

The power plant will be capable of delivering the required net peak power demands at any time and at any expected ambient condition, while accounting for equipment fouling, ageing effects and power plant parasitic loads. The plant configuration will also cater to the planned and forced outages of the generating sets expected during the design life of the plant. The power plant will not be connected to the national electric utility, and will operate 24 hours per day, seven days per week and 365 days per year as an islanded operation.

The existing Phase 1B power station will remain in service and will be integrated into the overall power distribution system to provide backup to the new Phase 2 power station on site. If one generator set (17 MW) in the Phase 2 power station is unavailable, then the Phase 1B (18 MW) power station will be used.

Additional Buildings

Additional service and administration buildings for the expansion include:

- Plant office building
- Welcome centre
- Plant gate and change house

24.21 Capital and Operating Costs (38 kt/d)

Capital Costs

Basis of Estimate – Capital Costs

The scope of the capital cost estimate for the expansion project includes:

- Mining
- New CIL process plant
- Expansion of the existing adsorption, desorption and recovery (ADR) plant
- Phase 2 power plant
- Site infrastructure facilities and utilities
- Tailings storage facility

The site closure cost estimate includes the 38 kt/d expansion facilities, existing facilities and known future work on the site.

A key result of the feasibility study was the shift to preassembly and precast concrete execution strategies. These strategies will facilitate higher productivities and more predictable and controllable costs. In addition, this will maximize opportunities for procurement to implement a global sourcing strategy to improve vendor quality, value pricing and delivery. Additional opportunities to improve schedule certainty and reduce capital costs will continue to be explored in the time leading up to the notice to proceed. The basis of estimate for each cost area is summarized in Table 24-8.

Table 24-8: Basis of Estimate – Capital Costs

Capital Cost Item	Estimate Basis
Mining	
Mine fleet equipment	• Mine fleet equipment costs were based on quotes from vendors • Mine fleet quantities were provided by Kinross • Indirect costs were estimated by Kinross
Seawater Supply System (included in Sustaining Capital)	
Seawater pipeline, pumping station, power station, seawater intake and raw water pond at the Tasiast site	• Pipeline, pumping station and power station costs were based on the awarded contract and adjusted to suit the reduced water flow rate requirements • Raw water pond was included in initial project scope of work • Indirects were estimated based on the requirements for managing the overall scope of work
Phase 2 Power Plant	
Power plant	• Based on the budget price (engineering, procurement and construction [EPC] basis) for a heavy fuel oil power plant located at the site • Indirects were estimated based on the requirements for managing the EPC scope of work
New CIL Process Plant Equipment	
Major equipment	• Firm prices and budget were obtained for selected equipment • Mechanical equipment list was used to identify equipment installed as part of PAUs and equipment installed on site
Minor equipment	• Budget prices were obtained for selected equipment • In-house database and costs from similar projects were used for the balance of equipment • Mechanical equipment list was used to identify equipment installed as part of PAUs and equipment installed on site
Ductwork and chutes	• Material unit costs were based on budget information and in-house data • Material take-offs (MTOs) were prepared by work breakdown structure (WBS) preassembly units (PAUs) • MTOs were used to identify plate work installed as part of PAUs and plate work installed on site

Capital Cost Item	Estimate Basis
New CIL Process Plant Bulk Materials and Site Works	
Concrete	• Precast material unit costs were based on budget pricing obtained from contractors • Concrete batch plant and material unit costs were based on budget pricing obtained from contractors • MTOs were prepared by WBS • MTO quantities were tabulated by precast and cast-in-place
Structural steelwork	• Material unit costs were based on budget pricing obtained from a Spanish supplier • MTOs were prepared by WBS • MTO quantities were tabulated by PAUs and site erect • MTO quantities for PAUs included additional steel amounts for grillage, stiffening and ocean transportation • MTOs were used to identify steel amounts to be removed after PAUs are positioned on site
Process and services piping	• Material unit costs were based on budget pricing obtained from European suppliers • MTOs were prepared by WBS • MTO quantities were tabulated by PAUs and site installation • MTO quantities included PAU interconnecting piping after PAUs are positioned on site
Electrical	• Budget pricing was obtained for electrical equipment • Material unit costs were based on budget pricing obtained from European suppliers, in-house data and actual cost data from completed Tasiast work • MTOs were prepared by WBS • MTO quantities were tabulated by PAUs and site installation
Instrumentation and control system	• Process Control System (PCS) hardware pricing was based on a vendor quote with updated quantities • Material unit costs were based on budget pricing obtained from European suppliers, in-house data and actual cost data from completed Tasiast work • MTOs were prepared by WBS • MTO quantities were tabulated by PAUs and site installation
Earthworks	• Budget pricing was obtained for a mobile crushing and screening plant to process mine waste rock as fill material • Material unit cost for fill was developed from first principles • Budget pricing was obtained for high-density polyethylene (HDPE) liner material • MTOs were prepared by WBS

Capital Cost Item	Estimate Basis
Infrastructure, non-process buildings	• Material unit costs were based on budget pricing obtained from suppliers • For selected buildings, building costs were based on budget pricing obtained from design and build contractors • MTOs were prepared by WBS • MTO quantities were tabulated by PAUs and site installation
Infrastructure, utilities	• Material unit costs were based on budget pricing obtained from European suppliers • MTOs were prepared by WBS
Tailings storage facility (TSF)	• MTO quantities were distributed on an annual basis for Cell 1 first lift, Cell 1 subsequent lifts and Cell 2 • Only Cell 1 first lift is included in the initial cost estimate, and the balance of the TSF scope was included in sustaining capital cost estimate • Material unit cost for fill was developed from first principles using a mobile crushing and screening plant • Budget pricing was obtained for the HDPE liner and geotextiles
New CIL Process Plant Indirect Costs	
Construction temporary facilities and support services	• Includes: temporary offices, warehousing, laydown areas, temporary electrical power supply and distribution, water, sewage, information and communication technology, services such as surveying, warehousing, soil and concrete testing, structural steel torque verification and instrument calibrations • Cost was developed from the first-principles approach using data from previous Tasiast construction work, pricing obtained from suppliers and in-house data
Camp	• The existing "Old Town" construction camp will be used for the expansion project (currently moth-balled) • Refurbishment cost of the camp was developed from first principles
Catering	• Rate per day per person based on current camp operations
First fills	• Cost of first fills of reagents was based on the reagent unit costs used for estimating operating costs • Cost was estimated for the first fills of lubricants • Costs for inventory and other consumables were excluded from the estimate
Spare parts	• Cost was based on budget pricing obtained from equipment vendor recommendations for critical spares, first year spares and commissioning spares • Allowances were included where vendors did not provide pricing

Capital Cost Item	Estimate Basis
Vendor representatives	• Cost was developed based on data from vendors
Freight and handling of freight on site	• Cost for transportation of PAUs was based on budget pricing from a heavy lift specialist contractor and included ocean freight, in-land freight and placement of PAUs on foundations • For the balance of equipment and materials, a detailed estimate of freight from the suppliers to the preassembly yard and/or site was developed • Handling of freight on site included freight laydown areas, labour, forklifts and cranes
Engineering, procurement, and construction management (EPCM)	• Cost of services estimate includes work-hours for services, based on deliverables, staffing plans, offices expenses, travel and site assignment costs
Additional consultants	• Cost estimates obtained from consulting firms for specific areas • Allowances were included for other areas
Pre-operational testing	• Detailed cost build-up was prepared
Other Project Costs	
Owner's cost	• Includes Kinross team salaries and expenses, project insurance, office and other overhead cost, legal fees, environmental services, commissioning (including operators and operations support), training, community support, interest and finance cost and the associated contingency
Contingency	• Contingency result was determined based on a quantitative risk assessment
Taxes and duties	• Includes payroll taxes, duties, withholding services tax and withholding goods tax
Escalation	• Escalation is excluded from the capital cost estimate (the financial model uses a real discount rate)

Capital Cost Estimate

The total capital cost estimate is shown in Table 24-9.

Table 24-9: Capital Cost Estimate

Category	Cost (US$M)
Direct Costs	
Mine Equipment	52
Site Infrastructure and Facilities	18
Site Utilities	170
Crushing and Storage	84
Grinding	194
Concentration	33
Leaching	73
Metal Production and Refining	18
Reagent Mixing and Storage	23
Other Processes	6
Tailings	81
Project Distributables	104
Total Direct Cost	858
Site indirects	105
Project indirects	224
Freight	100
Owner's cost	86
Contingency	182
Taxes and duties	84
Long lead equipment already purchased	-72
Total going forward capital cost	1,566

As a percentage of the total cost estimate, 78% was in US dollars, 19% in Euros, 1% Australian dollars and 1% Canadian dollars, with the balance from small amounts of other currencies.

Estimated annual sustaining capital is shown in Table 24-10.

Table 24-10: Annual Sustaining Capital with Witholding Tax

Year	Sustaining Capital (US$M)
2016	40,921,369
2017	38,995,558
2018	82,749,694
2019	146,300,663
2020	147,109,251
2021	37,589,304
2022	93,737,142
2023	73,824,600
2024	24,266,271
2025	28,939,407
2026	17,899,070
2027	15,311,103
2028	12,546,551
2029	-
2030	-
2031	-
2032	-
Total	**760,189,982**

Operating Costs

<u>Basis of Estimate – Operating Costs</u>

Operating costs were estimated for mining, processing, site administrative, and power plant costs. The basis of estimate is summarized in Table 24-11.

Table 24-11: Basis of Estimate – Operating Costs

Operating Cost Item	Estimate Basis

Mining	Developed from first principles by:
	• Developing a detailed mine plan schedule
	• Calculating a haulage network (specific to the detailed mine plan) to generate equipment hours and fuel consumption based on site conditions.
	• Applying key cost parameter inputs such as:
	  o Input prices (diesel, blasting explosives and tires) from existing site contracts
	  o Productivity – either existing productivity or expected 2014 productivity
	  o Labour rates – based on 2014 labour rates
	  o Fuel burn rates – based on existing site conditions
	  o Maintenance costs
	  o Other inputs, such as tire life and lubrication strategy – based on existing site strategy
	  o Drill consumables life – based on site experience to date

Processing	Estimation methodology varied by cost component, but was primarily built from first principles, relying on a combination of: • Knowledge from existing operations • Laboratory testing completed for the new CIL • Supplier reagent and consumable costs • Energy consumption estimates per motor • Mass and water balance Major categories include the following, which collectively result in a processing cost estimate for the expansion scenario: • Power • Consumables • Reagents • Labour • Maintenance • Water • Laboratory • Plant admin

Operating Cost Item	Estimate Basis
Site Administrative	Bottom-up approach applying labour, camp and other costs to various areas including: • Administration • Heath, safety and environment, including: • Tasiast Team Village accommodation and messing • Training • Recruiting • Operations readiness • Nouakchott accommodation • Site services • People mobility

Operating Cost Estimate

Operating cost estimates are shown in Table 24-12. The operating costs for each area include allocations for power plant operating costs.

Table 24-12: Operating Cost Estimates (Expansion Case)

Operating Cost	Unit	First 5 years 2018-2022	Life-of-Project 2018-2029	Life-of-Mine Average 2014-2029
Mining	US$/t mined[1]	$1.98	$2.11	$2.10
Processing (CIL Plant)	US$/t processed	$14.62	$14.49	$14.74
Processing (Dump Leach)	US$/t processed[2]	$39.21	$39.21	$10.90
Site Admin	million US$/a	$80.9	$65.1	$70.9

Notes:
1. Excludes capitalized maintenance.
2. Limited tonnes are placed over 2018-2019, while dump leach processing continues until 2020.

24.22 Economic Analysis (38 kt/d)

The economics of the Tasiast Expansion Project were evaluated using a real (non-escalated), after-tax discounted cash flow (DCF) model on a 100% project equity (unlevered) basis. Unless otherwise stated, all economic parameters are shown on an absolute basis (not incremental to existing operations). Production, revenues, operating costs, capital costs and taxes were considered in the financial model. The main economic assumptions are a US$1,200/oz gold price and a 5% discount rate.

The valuation date for the financial analysis was set for January 1, 2014. All cash flows assumed for the purposes of this study are from this date onward. However, the notice to proceed decision for the expansion option is not expected to be made until 2015 at the earliest.

The cash flow analysis was used to estimate the economics of the 38 kt/d carbon-in-leach (CIL) plant expansion scenario using a heavy fuel oil (HFO) on-site power plant. This scenario assumes a new CIL plant starts up in 2018.

Key assumptions in the financial model are summarized in Table 24-13.

Table 24-13: Financial model key assumptions

Parameter	Assumption	Description
General		
CIL plant throughput options	38,000 t/d	The throughput of the expansion plant is 38 kt/d, which assumes the closure of the existing 8 kt/d plant once the new plant is operational. The throughput assumption for the existing plant until start-up comes from current site operations. The throughput assumptions for the expansion scenario were defined by the project team.
Dump leach and adsorption, desorption and recovery timing	Year 1: 60% Year 2: 20% Year 3: 10% Year 4: 5% Year 5: 5%	This is the assumed recovery timing of the gold when the ore has been placed on the dump leach. Recovery timing assumptions were estimated by the current site operations in conjunction with the project team.
Power option (for expansion option)	HFO on-site	The expansion option was analyzed with the HFO on-site power plant.
Mining years	14 years	The number of years of mining operations has been set by the mineable inventory (2014 is considered Year 1).

Parameter	Assumption	Description
Processing years	16 years	The number of years of processing operations was set by the mineable inventory and stockpiles (2014 is Year 1).
Currency	US dollars	The model was constructed as a single-currency model using US dollars (US$).
Inflation	None – real basis	All projected revenue and costs were assumed to be in January 1, 2014 real terms, with no inflation applied.
Starting basis	January 1, 2014 go forward	All economic analyses were done on a January 1, 2014, “go-forward” basis. Spending before this date was considered to be sunk, and was not considered, except for opening balances.
Capital structure	Unlevered	The calculated financial results assume the project will be internally financed. No debt financing or interest payments were assumed.
Discount rate	5% real	All the NPVs shown in this report were calculated using a discount rate of 5%. Sensitivity analysis has been completed for 0% and 10% discount rates.
Commodity Prices		
Gold	US$1,200/oz	Commodity prices were assumed to be constant over the DCF timeframe.
Oil (West Texas Intermediate [WTI])	US$100/bbl	Commodity prices were assumed to be constant over the DCF timeframe.
HFO	US$ 0.226/kWh	Power cost based on HFO prices and generator characteristics
Refining, Transport and Other Charges		
Refining charge	US$ 0.25/oz Au	A refining charge of US$ 0.25/oz was applied before the royalties were calculated.
Doré transportation	US$ 3.50/oz Au	A doré transportation charge was applied before the royalties were calculated.
Location swap	US$ 0.25/oz Au	A location swap charge of US$ 0.25/oz was applied before the royalties were calculated.
World Gold Council Fees	US$ 2.00/oz Au	A world gold council fee of US$2.00/oz was applied before the royalties were calculated.

Parameter	Assumption	Description
Capital and Operating Costs		
Capital expenditures (CAPEX)	Described in Section 24.21.	Spending before January 1, 2014, is considered sunk cost and is not considered in the analysis, except for opening balances for tax depreciation. Sustaining capital costs: portions were derived using zero-based costing where possible. Provisional estimates for LOM values were made otherwise.
Operating costs (OPEX)	Described in Section 24.21.	Described in Section 24.21.
Accounts receivable	N/A	A balance of gold is held in accounts receivable and cashed out at the end of the LOM. This balance is equal to the accounts receivable at site (based on budget figures for 2014 and 2015), and it is assumed that it will not change beyond 2015 until it is recovered at the end of the mine's life.
Inventory	90 days OPEX	Working capital requirements for inventory, including mining fuel, were assumed to be 90 days of all non-power operating costs.
Accounts payable	30 days OPEX	Working capital requirements were assumed to be 30 days of all non-power operating costs.
Stockpiles	Expensed in the year that tonnes are processed	In the model, the mining cost associated with stockpiled tonnes was tracked in working capital and considered a cash cost in the year incurred. For tax purposes, mining costs were expensed in the year that the tonnes were processed.
Salvage value at the end-of-mine life	US$ 21.8 million	No salvage value was applied to the majority of the assets at the end-of-mine life except for an assumed salvage value related to the mine fleet (which depends on the remaining vehicle life).
Closure and rehabilitation costs	US$ 78 million	Total closure costs were estimated at US$ 78 million for the expansion scenario. The majority of the costs were assumed to be incurred in the three years immediately following the completion of processing operations.
Taxes and Duties		
Governing convention	2002 Mining Convention	Unless otherwise noted, the economic evaluation in this feasibility study was assumed to be governed by the Tasiast Mauritanie Ltd. S.A. (TMLSA) 2002 Mining Convention.
Income taxes	25%	Income taxes were included in this financial analysis with a marginal tax rate of 25%. The annual tax paid in the model is the *higher* of the income tax or the minimum tax (see below).

Parameter	Assumption	Description
Minimum tax	1.25%	The minimum tax was calculated on total metal revenues. The *higher* of the income tax or the minimum tax is the given tax paid in any year.
Royalties	5% total	Royalties were forecasted to be constant over the mine life at 3% of revenue to the Mauritanian government and 2% to a third party (Franco-Nevada Corporation).
Import duties	5%	Import duties for all non-resident goods were included in CAPEX and OPEX estimates. (Note that as part of the TMLSA Mining Convention, fuels are assumed to be exempt from import duties.)
Withholding tax	17.6%	Withholding tax is applicable to all non-resident services.
Tax loss carry forwards	5 years	Tax-loss carry forwards expire after five years.
Depreciation of Fixed Assets	Varies from 2 to 20 years depending on asset class	For most assets, depreciation was calculated on a straight-line basis. Depreciation was calculated according to cash-tax depreciation rates and it is always applied in the year it is incurred. Depreciation-generated losses can be carried forward indefinitely and applied against future earnings. Accelerated depreciation is applied if the mine life is less than the asset life.

The results of the financial analysis, with sensitivities to gold price and discount rate assumptions, are shown in Table 24-14. The FS is based on $1,200/oz gold and a real discount rate of 5%. Annual life-of-mine cash flows are shown in Table 24-15.

At a 5% discount rate, the project is economic at a $1,200/oz gold price.

Table 24-14: Financial analysis results and sensitivities

Financial metric	Unit	Gold Price (US$/oz)			
		1,200	1,350	1,400	1,600
NPV at 0% discount rate	US$ billion	1.48	2.50	2.84	4.17
NPV at 5% discount rate	US$ billion	0.50	1.22	1.46	2.37
NPV at 10% discount rate	US$ billion	(0.02)	0.51	0.69	1.35
Internal rate of return	%	10	17	20	31
Payback year (undiscounted)	Year	2024	2020	2020	2019

Table 24-15: Life of Mine Cash Flows, 38 kt/d with HFO power.

Year	Cash flow, $1,200/oz Au (US$M)	Gold recovered (Moz)
2014	(1,643)	268
2015	(378,588)	224
2016	(727,165)	301
2017	(765,970)	246
2018	512,280	1016
2019	281,694	853
2020	26,559	612
2021	644,545	1066
2022	189,167	696
2023	36,775	535
2024	43,497	473
2025	528,147	837
2026	680,537	925
2027	323,287	551
2028	102,961	302
2029	48,786	107
2030	(15,285)	0
2031	(24,256)	0
2032	(23,044)	0
2033	(3,225)	0
2034	(973)	0
2035	(836)	0
2036	(699)	0
2037	(699)	0

25. INTERPRETATION AND CONCLUSIONS

Tasiast is viewed as a long-term strategic asset for Kinross, located in a district that is believed to have significant future potential. The expansion project is believed to provide an opportunity to capitalize on the full potential of the operation and to cement Tasiast as a cornerstone asset within the company.

At a 5% discount rate, the project is economic at a $1,200/oz gold price.

26. RECOMMENDATIONS

Leading up to the potential construction decision in 2015, Kinross will continue with a value improvement program focusing on:

- Continuous improvement and refinement of the execution strategy
- Procurement strategies and opportunities
- Continuous improvements to the mine plan
- Evaluation of requirements for the owners team, and engineering, procurement and construction management strategies and requirements

This will allow the project team to further improve and refine areas such as planning, procurement, and project execution.

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28. DATE AND SIGNATURE PAGE

The effective date of this Technical Report entitled “Kinross Gold Corporation, Tasiast Project, Mauritania, NI 43-101 Technical Report” is March 14, 2014.

“Signed and sealed”

John Sims, AIPG Certified Professional Geologist

Dated March 31, 2014