Paracatu Project
Brazil
National Instrument 43-101 Technical Report

Prepared for:
Kinross Gold Corporation

Prepared by:
John Sims, AIPG Certified Professional Geologist

Effective Date: March 31, 2014

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1. SUMMARY

1.1 Executive Summary

Kinross Gold Corporation (Kinross) has prepared a Technical Report (the Technical Report) for the wholly-owned and operated Paracatu Project (Paracatu or the Project), located in the northwestern region of the state of Minas Gerais, Brazil. Kinross Brasil Mineração S.A. (KBM or the Company) is Kinross' operating entity for the Paracatu Project. KBM was previously known as Rio Paracatu Mineração S.A. (RPM), before the name was changed in 2010.

Paracatu is a large open pit gold mine located in the Minas Gerais region of Brazil. Operations include conventional shovel/truck open pit mining, two process plants with extraction of gold using gravity/ flotation/ carbon-in-leach (CIL) recovery processes. Since production started in 1987, Paracatu has processed approximately 548 million tonnes of material and has produced 5.95 million ounces of gold as of the end of 2013.

Paracatu will process at an average of approximately 55 million tonnes per annum (Mt/a) over the next four years, and at 38 Mt/a thereafter with mine operations concluding in 2030. Ore will be fed to the processing plants from the stockpile in 2031 and 2032.

The Mineral Resource and Mineral Reserve estimates as of year-end 2013 are summarized in Tables 1-1 and 1-2.

Table 1-1: Mineral Resources - December 31, 2013

	Tonnes (000s)	Gold (g/t)	Gold Ounces (000s)
Measured (M)	215,040	0.31	2,111
Indicated (I)	325,135	0.39	4,069
M+I	**540,175**	**0.36**	**6,180**
Inferred	3,239	0.27	28

Notes:
1. Mineral Resources are exclusive of Mineral Reserves.
2. Mineral Resources estimated according to CIM Definitions.
3. Mineral Resources estimated at $1,400/oz Au.
4. Mineral Resources are estimated at gold cut-off grades that vary by material type from approximately 0.139 g/t Au to 0.208 g/t Au.

Table 1-2: Proven and Probable Mineral Reserves – December 31, 2013

	Tonnes (000s)	Gold (g/t)	Gold Ounces (000s)
Proven	556,292	0.41	7,371
Probable	207,416	0.45	3,030
Total	**763,708**	**0.42**	**10,401**
Stockpile	2,915	0.24	23

Notes:
1. Mineral Reserves estimated according to CIM Definitions.
2. Mineral Reserves estimated at $1,200/oz Au.
3. Proven Reserve includes Stockpile.
4. Mineral Reserves are estimated at gold cut-off grades that vary by material type from approximately 0.163 g/t Au to 0.244 g/t Au.

Conclusions

Mineral Resource Estimation

- The 2013 year-end (EOY2013) Measured and Indicated Mineral Resources, exclusive of Mineral Reserves, total 540.2 million tonnes averaging 0.36 g/t Au and contain 6.2 million ounces of gold.

- The EOY2013 Inferred Mineral Resources total 3.2 million tonnes averaging 0.27 g/t Au and contain 28,200 ounces of gold.

- The resource estimate gold cut-off grades range from 0.139 g/t for B1 ore material (feed to Plant I) to 0.208 g/t for B2 ore material (feed to Plant II), which requires higher grinding energy.

- Mineral Resource estimates have been prepared using acceptable estimation methodologies. The classification of Measured, Indicated, and Inferred Resources conform to Canadian Institute of Mining, Metallurgy and Petroleum Definition Standards for Mineral Resources and Mineral Reserves dated November 27, 2010 (CIM Definitions).

- Protocols for drilling, sampling, analysis, security, and database management meet industry-accepted practices. The drill hole database was verified and is reasonable for supporting a resource model for use in Mineral Resource and Mineral Reserve estimation.

- Kinross is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other modifying factors which could materially affect the open pit Mineral Resource estimates.

Mining and Mineral Reserves

- The EOY2013 open pit Proven and Probable Reserves, including existing stockpiles scheduled for processing, are estimated to be 764 million tonnes at 0.424 g/t Au and contain 10.4 million ounces of gold.

- The reserve estimate gold cut-off grades range from 0.163 g/t for B1 ore material to 0.244 g/t for B2 ore material, which requires higher grinding energy.

- The Mineral Reserve estimates have been prepared using acceptable estimation methodologies and the classification of Proven and Probable Reserves conforms to CIM definitions.

- Recovery and cost estimates are based on actual operating data and engineering estimates.

- Economic analysis of the Paracatu Life-of-Mine (LOM) plan generates a positive cash flow and meets the requirements for statement of Mineral Reserves. In addition to the Mineral Reserves in the LOM plan, there are Mineral Resources that represent opportunities for the future.

- One major change between EOY2013 and EOY2012 is the inclusion of sustaining capital costs as operating costs for the purposes of pit shell determination. Some examples of these costs are: waste pre-stripping, mine fleet replacement costs, and tailings dam construction, for which additional investment is required in order for the operation to continue or for the reserve base to be enlarged. This change has a large impact on the overall reserve estimation since the Lerchs-Grossman (LG) calculation takes these added costs into consideration when sizing the overall economic pit. The end result is a smaller overall pit with a higher contained gold grade and improved economics.

Process

- Paracatu has two mineral processing plants known as Plant I and Plant II. Plant I focuses on treating the softer near-surface B1 (oxide) ore at a throughput of 20 Mt/a, whereas Plant II focuses on treating the harder B2 (sulphide) ore at a throughput of 41 Mt/a.

Environmental Considerations

- KBM is constantly seeking excellence in its operational health and safety, environmental and social responsibility system. The site maintains Certifications in ISO 14001, OHSAS 18001, SA 8000 and compliance with the Cyanide Management Code.

- KBM's Environmental Impact Control Plan and Closure Plan were updated in 2010 to reflect changes in mining activities, processing, and final disposal of waste and tailings.

Recommendations

There are no recommendations at this time as Paracatu is a fully operational mine.

1.2 Technical Summary

Property Description, Location and Land Tenure

The Paracatu property consists of five mining leases, nineteen exploration permits, and eleven exploration permit applications grouped into four non-contiguous blocks totalling approximately 25,986 ha. Located in northwestern region of the Minas Gerais State, the property is located approximately 230 km southeast of the national capital, Brasilia, and 480 km northwest of the state capital Belo Horizonte. It comprises an open pit mine as well as processing plants, tailings facilities and associated infrastructure. The property is easily accessible by road from the nearby municipality of Paracatu. It is centred at approximately 17°13'15"S latitude and 46°52'30"W longitude.

Land Tenure

Kinross first acquired a 49% interest in the mine upon completion of the merger with TVX Gold Inc. (TVX) and Echo Bay Mines Ltd. (Echo Bay) on January 31, 2003. On December 31, 2004, Kinross purchased the remaining 51% interest in the mine from Rio Tinto for US$260 million. Kinross' interest in the property is subject to a royalty of 1% of net sales due to Departamento Nacional da Producão Mineral (DNPM) and an additional royalty of 0.5% is due to the holders of surface rights in the mine area not already owned by Kinross.

History

The mining history of the Paracatu region is closely associated with the activities of the Portuguese *bandeirantes* expeditions who prospected for gold in Brazil's interior, arriving in the Paracatu region in 1722 after the discovery of gold in alluvial deposits.

Alluvial mining peaked during the second half of the 18th century. Such mining activities were not limited to the placer deposits along Rico Creek, but also extended to the oxidized ore outcrop on the top of Morro do Ouro hill or the "Hill of Gold".

Gold production declined sharply during the first decade of the 19th century. From this point forward, production was limited to subsistence mining practiced by local inhabitants known as *garimpeiros*. Various prospectors explored the region, but economically viable operations were limited as a result of the low-grade nature of the deposits.

Beginning in 1970, Paracatu attracted some attention from mineral exploration companies looking for lead and zinc deposits in the area. In 1984, Riofinex do Brasil (Riofinex) embarked on a surface exploration program that focused on the oxidized and weathered horizons of the Moro do Ouro area. Work by Riofinex and its various partners from 1984 to 2001 included 458 trenches (5,070 linear metres) and 674 drill holes (27,569 m).

At the end of 1984, based on the data from hundreds of test pits and further supported by a total of 44 drill holes, a "reserve" of 97.5 million tonnes grading 0.587 g/t Au was estimated by Riofinex for what is currently known as the Paracatu Mine. Kinross notes that this estimate pre-dates NI 43-101, cannot be relied upon, and is quoted here for historical purposes only.

Production at Paracatu commenced in October 1987 with the treatment of oxidized and highly weathered ore. The first gold bar was poured in December 1987. The following year, mine throughput reached the design capacity of 6.1 Mt/a. As a result of a series of expansion projects, the design capacity has increased to its current nominal throughput of 61 Mt/a.

Geology and Mineralization

The Paracatu Property is hosted within the Brasília Belt, a north-south trending Neoproterozoic belt that extends along the western side of the São Fransisco-Congo Craton. Sedimentary units are mostly preserved in the northern part of the belt, whereas in the southern part where Paracatu is located, there is intense deformation and metamorphism, and contacts between metasedimentary units are primarily tectonic. A series of east-northeast trending thrust faults is developed extensively along the belt. Metamorphic grade increases towards the west as the thickness of the fold belt increases. The timing of deformation is estimated at 800 Ma to 600 Ma which coincides with the Brasiliano orogenic cycle.

The property is underlain by a thick sequence of phyllites belonging to the basal Morro do Ouro Member of the Paracatu Formation of the Upper Proterozoic Canastra Group.

The Canastra Group is exposed along the south-central portion of the Brasília Belt, and is composed of sandy and shaley metasedimentary rocks. Due to intense deformation, the stratigraphic organization of the Canastra Group is not fully understood. The Canastra Group was metamorphosed to greenschist grade, although locally amphibolite grade assemblages have been reported.

The Paracatu Formation is subdivided into the basal Morro do Ouro Member, a 100 m thick layer of dark carbonaceous phyllite, and the overlying Serra da Anta Member, a sericitic phyllite. Both phyllites display fine-grained quartzite intercalations.

The host phyllites of the Paracatu Formation exhibit extensive deformation and feature well-developed quartz boudins and associated sulphide mineralization. Sericite minerals are common, likely as a result of extensive metamorphic alteration of the host rocks. Primary sedimentary features and bedding planes are easily recognizable, but are intensively deformed by thrusting, particularly along bedding planes, and the development of sygmoidal and boudinage structures.

Mineralization at Paracatu is closely related to a period of ductile deformation associated with shearing and thrust faulting. Overall, the Morro do Ouro sequence has been thrust to the northeast. Intense, low angle isoclinal folds are commonly observed. The mineralization appears to be truncated to the north by a major normal fault trending east-northeast. The displacement along this fault is currently unknown. The current interpretation is that the fault has displaced the mineralization upwards and erosion has removed the mineralization in the up-thrown block.

The Paracatu mineralization is subdivided into four horizons defined by the degree of oxidation, surface weathering, and sulphide mineralization. The contact between unmineralized host rock and the various mineralized horizons is gradational, occurring over a 10 m thick interval that is characterized by arsenic values of 200 ppm to 500 ppm and gold grades of up to 0.2 g/t. The sulphides content typically does not exceed 3% to 4%. The most common sulphides observed are arsenopyrite, pyrite and pyrrhotite. Galena is relatively common and may be accompanied by sphalerite. Chalcopyrite occurs locally in fractures within the main sulphide minerals listed above.

The mineralization at Paracatu exhibits distinct mineralogical zoning with the arsenopyrite content increasing towards the centre and west and in the zones of intense deformation. Gold grade increases with increasing arsenopyrite content. Pyrrhotite occurs in the western part of the deposit and gold grades are elevated where higher pyrrhotite content is observed. The deposit formation model proposed for Paracatu suggests that gold and arsenopyrite were introduced concurrently during the deformation event. Gold occurs either as free gold or electrum.

Exploration

Since Kinross acquired the Project in 2003, exploration efforts have been focused primarily on the main mining area. Exploration outside of the immediate mine area was initiated in 2006.

In the licensed exploration areas immediately bordering the mine leases, exploration activities were concentrated on soil and termite-mound geochemical sampling and interpretation of airborne magnetic survey data to look for nearby features similar to Paracatu. Some target areas were generated, mostly located west and west-northwest of the mine. Follow-up exploration returned no significant results.

A near-pit geophysical survey was performed in 2008 to define the induced polarization (IP) and resistivity geophysical signature for the known buried mineralization of the down-dip southwest extension of the B2 ore zone below and west of Rico Creek. A pattern was identified indicating higher chargeability in the non-mineralized zone above the ore zone, and high resistivity at depth within the ore zone.

Geophysical data were the primary driver of exploration in the licensed exploration areas located 10 km or more from the mine. Definition of favourable structural zones using regional airborne magnetic data yielded three targets which were then surveyed for IP and resistivity. Two targets were located approximately 50 km to 60 km from the mine and the third target was 10 km from the mine. Carbon-rich phyllites with quartz boudins and pyrite similar in lithologic character to the Paracatu deposit, but without gold and arsenopyrite, were identified in one of the targets located further from the mine.

Drilling

Since acquiring the Project in early 2003, Kinross has completed 1,186 drill holes for a total of 96,823 m. Core diameters for holes drilled by Kinross include HX (76.2 mm), HQ (63.5 mm), HTW (70.9 mm), and NQ (47.6 mm). Substantially all of this drilling has been completed on the mining leases.

Mineral Processing and Metallurgical Testing

The original processing plant at Paracatu (Plant I) has operated continuously since 1987 and received expansion upgrades in 1997 and 1999. The plant includes primary and secondary crushing, grinding, gravity and flotation circuits. A hydrometallurgical circuit leaches the flotation concentrates and produces gold bullion.

Samples for Bond Work Index (BWI) testing were collected and the BWI composite data were used to interpolate BWI into the Paracatu block model. Metallurgical recovery is also estimated for each block in the model and, along with BWI, is used for mine planning.

Plant recoveries are estimated on the basis of sulphur and arsenic content in each block. The maximum possible flotation plant recovery is 86% and this decreases linearly with increasing sulphur and arsenic assays. Hydromet gold recovery is modelled at a constant 96.5%.

Plant II was developed as part of the Paracatu Expansion III Project to counter the gradually increasing work index of the ore with increasing depth from surface.

The Mineral Resource and Reserve estimates summarized in this report assume operation of Plant I and Plant II. Plant I focuses on treatment of the softer near-surface B1 ore at a design throughput of 20 Mt/a, whereas Plant II focuses on treatment of the harder B2 ore at a design throughput of 41 Mt/a. The availability of B1 ore for processing will continue to be assessed by Kinross. Capacity may be limited to a design throughput of 41 Mt/a in 2016 due to lack of B1 ore.

The Expansion III Project proceeded in several stages beginning in 2006. The aim of the first stage was to increase plant capacity from 18 Mt/a to 50 Mt/a with the addition of a new 32 Mt/a Semi-Autogenous Grinding (SAG) mill. Further Plant II expansions were carried out from 2009 to 2012 to increase grinding capacity and recovery. By June 2011, a third ball mill and flash flotation for ball mills 1 and 2 were installed. In 2010, an optimization study concluded that expected ore work indices in 2012 would require a fourth ball mill to maintain the 61 Mt/a throughput rate. An increase in processing capacity was provided by the completion of a fourth ball mill in September 2012.

In 2013, the Paracatu plant processed 55.7 Mt/a and achieved average gold recovery of 75.8%.

Studies were undertaken by KBM to evaluate the substitution of the existing Acidification, Volatilization and Reneutralization (AVR) cyanide recovery process with the more modern and widely used SO_2/Air Cyanide Destruction Process. Laboratory testing of the SO_2/Air Cyanide Destruction Process on KBM cyanidation tailings demonstrated that the process had a series of advantages over the AVR process. A new SO_2/Air Cyanide Destruction plant was installed as part of the Expansion III Project.

With the implementation of the Expansion III Project, a new tailings facility was required. Bench scale and industrial scale test work on desulphurization of flotation tailings with sulphide recovery were completed and reported in the Feasibility Study for the Expansion III Project. Construction of the new Eustáquio Tailings Storage Facility (TSF) began in 2009 and tailings have been discharged into the new basin since 2012.

Mineral Resources and Mineral Reserves

Mineral Resources are reported using two cut-off grades depending on where the material is processed. Resources to be processed in Plant I were reported at a 0.137 g/t Au cut-off grade, whereas resources to be processed in Plant II were reported at a cut-off grade of 0.208 g/t Au. Cut-off grades were determined using a long-term gold price of US$1,400/oz. Processing recoveries and associated operating conditions were used to generate an optimized pit shell using the LG algorithm. Mineral Resources exclusive of Mineral Reserves are reported between the EOY2013 ultimate pit design and the $1,400/oz LG pit shell. The Mineral Resources also include Inferred and some incremental Measured and Indicated material situated in the pit design (see Table 1-1).

The Proven and Probable Mineral Reserves as of December 31, 2013 are estimated to be 763.7 million tonnes at 0.424 g/t Au, containing a total of 10.4 million ounces of gold (Table 1-2).

Mineral Reserves are reported using two cut-off grades depending on where the material is processed. Reserves to be processed in Plant I were reported at a 0.163 g/t Au cut-off grade, whereas reserves to be processed in Plant II were reported at a 0.244 g/t Au cut-off grade. Cut-off grades were determined using a long-term gold price of US$1,200/oz.

Due to the geometry and extension of the mineralization, mining recovery and mining dilution were considered to be 100% and 0% respectively. The Paracatu deposit is a massive ore body with very few lenses of internal waste.

Mining Method

The Paracatu Mine is a traditional open pit truck/shovel operation that has been in continuous operation since 1987. Paracatu plants have processed approximately 548 million tonnes of material and produced 5.95 million ounces of gold up to the end of 2013.

Mining is planned at an average rate of approximately 67 Mt/a over the next four years, followed by 59 Mt/a from 2018 to 2023, then increasing to an average of 73Mt/a through 2028, after which the rate will decline and mining will conclude in 2030. Ore will be fed to the mill from stockpile in years 2031 and 2032. Waste rock mining remains at 7.5 Mt/a through 2019 and then continually rises each year to a high of 43 Mt in 2027 before dropping off in the final year of operations.

Recovery Methods

Plant I at Paracatu has operated continuously since 1987, with expansion upgrades completed in 1997 and 1999. The plant consists of primary and secondary crushing, ball milling to 80% passing 75 microns, gravity recovery using jigs, rougher and cleaner flotation, concentrate regrinding, and cyanide leaching (Hydromet Plant). Final gold bullion is produced from the dedicated carbon adsorption, desorption and electrowinning circuit.

Plant II started production in September 2008, and achieved commercial production in December 2008. Currently, Plant II includes an in pit MMD crusher, a 1.8 km conveyor to a covered stockpile area, a 38 ft. SAG mill, and four ball mills. The recovery process uses gravity and flotation to produce concentrate, which leached in a carbon-in-leach (CIL) circuit in the hydromet plant. Gold is recovered by a carbon elution and electrowinning process and refined to gold bars.

The plant has a nominal capacity of 41 Mt/a when processing ore with a work index below 8.7 kWh/t. It is important to note that tonnage throughput will decrease as the work index increases.

Site Infrastructure

Paracatu infrastructure and services have been designed to support an operation of 61 Mt/a. The mine site consists of two processing plants, related mine services facilities (truck shop, truck wash facility, warehouse, fuel storage and distribution facilities, reagent storage and distribution facilities), and other facilities to support operations (safety/security/first aid/emergency response building, assay laboratory, plant guard house, dining facilities, offices etc.).

In 2011, a road was constructed directly from the highway west of the mine exclusively for access to the mine and plants. This four lane paved road is separated by a median and is 3.4 km long.

The mine draws its power from the Brazilian national power grid which relies mainly on hydroelectric power generation with outstanding reliability. KBM is furnished with a 230 kV connection by a substation that converts power from 500 kV transmission lines. A 34 km overhead transmission line connects the substation to the mine site substation which feeds 13.8 kV electricity to Plant II and 138 kV to Plant I.

Approximately 33% of KBM's power needs are fed by a self-generation structure located approximately 2,000 km from the mine in the state of Maranhao. KBM maintains a 95% stake in the Parnaiba IV thermal power plant and was granted self-generation status by the Brazilian Electricity Regulatory Agency (ANEEL) in January 2014.

The main sources of water for KBM operations are run-off water collected in mine sumps, run-off water collected in the tailings dam catchment basins, recirculated effluent from the process, and makeup water from three local surface water streams. The majority of process water is captured and maintained in the mine sumps and tailings catchment basins during the rainy season for use during the dry season.

Environmental, Permitting and Social Considerations

Paracatu is constantly seeking excellence in its occupational health and safety, environmental and social responsibility system. The site maintains Certifications in ISO 14001, OHSAS 18001, SA 8000 and compliance with the Cyanide Management Code.

Environmental analyses focus on operations in the KBM mining licence area and related facilities. An Environmental Impact Control Plan (PCIAM) has been implemented to ensure compliance with environmental permits. This plan focuses on such topics as air and soil quality, noise and vibration control, preservation of waterways, re-vegetation and tree planting, the safety and hygiene of workers, and reclamation projects as well as occupation proposals and future use of the areas affected by mining. The key aspects and environmental impacts addressed in the plan include deforestation, relocation of mammals and birds, soil removal and compacting, dust mitigation, fossil fuel emissions, rainwater runoff, noise and vibrations from operations and blasting, and changes to the landscape.

KBM’s Environmental Impact Control Plan and Closure Plan were updated in 2010 to reflect changes in mining activities, processing, and final disposal of waste and tailings.

Capital and Operating Cost Estimates

Planned capital costs at Paracatu are primarily sustaining capital, which includes mine equipment replacement and $793 million for the tailings dam expansions. Total sustaining capital costs are $1,453 million in real terms. Mine pre-stripping capital has been treated as an operating cost for the purposes of this Technical Report.

Operating costs are tracked and well understood. Unit operating costs for the LOM production schedule total US$8.79 per tonne processed (Table 1-3).

Table 1-3: Paracatu Production and Cost Summary

Area	Unit	Cost
Mining	(US$/t)	2.36
Processing	(US$/t)	5.35
Site Admin	(US$/t)	1.07
Total	**(US$/t)**	**8.79**

2. INTRODUCTION

Kinross Gold Corporation (Kinross) has prepared a Technical Report (the Technical Report) for the wholly-owned and operated Paracatu Project in the northwestern region of the state of Minas Gerais, Brazil. Kinross Brasil Mineração S.A. (KBM or the Company) is Kinross' operating entity for the Paracatu Project. KBM was previously known as Rio Paracatu Mineração S.A. (RPM), before the name was changed in 2010.

The purpose of this Technical Report is to support disclosure of end-of-year 2013 Mineral Resources and Mineral Reserves. The Technical Report conforms to NI 43-101 Standards of Disclosure for Mineral Projects and has an effective date of March 31, 2014.

Currency is expressed in US dollars unless stated otherwise. The currency of Brazil is the Real.

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.

2.1 Qualified Person

The Qualified Person for this Technical Report is:

- John Sims, AIPG Certified Professional Geologist, Vice-President, Technical Services, Reserves and Resources for Kinross

Mr. Sims visited the site most recently in October 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; and inspected the major infrastructure and current mining operations.

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.

Preparation of the revised Mineral Resource estimates included in this report was supervised by John Sims, Vice-President, Technical Services, Reserves and Resources at Kinross, and Eldrick L. Esper, Manager of Geology at Paracatu.

The revised Mineral Reserve estimate included in this report was prepared by Todd Carstensen, Director, Mine Planning, Kinross Technical Services.

2.3 Effective Dates

Several effective dates (cut-off dates for the information prepared) are appropriate for information included in this Technical Report. The effective date for the Mineral Resources and Mineral Reserves was December 31, 2013 (EOY2013). 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 List of Abbreviations

μ	micron	kt/d	thousand tonnes per day
°C	degree Celsius	kPa	kilopascal
°F	degree Fahrenheit	kWh/t	kilowatt-hour per tonne
μg	microgram	kW	kilowatt
a	annum	kWh	kilowatt-hour
Au	gold	L	litre
Bbl	barrels	LFO	light fuel oil
BRL	Brazilian Real	L/s	liters per second
Btu	British thermal units	m	metre
C$	Canadian dollars	M	mega (million)
CIL	carbon-in-leach	m^2	square meter
cm	centimeter	m^3	cubic meter
cm^2	square centimeter	mbgl	metres below ground level
CV	coefficient of variation	min	minute
d	day	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	s	second
gpm	Imperial gallons per minute	st	short ton
gr/ft^3	grain per cubic foot	stpa	short ton per year
gr/m^3	grain per cubic meter	stpd	short ton per day
ha	hectare	T	metric tonne
HFO	heavy fuel oil	t/a	metric tonne per year
hp	horsepower	t/d	metric tonne per day
in	Inch	US$	United States dollar
in^2	square inch	USg	United States gallon
J	joule	USgpm	US gallon per minute
k	thousand (kilo)	V	volt
kg	kilogram	WBS	work breakdown structure
km	kilometer	wmt	wet metric tonne
km/h	kilometer per hour	yd^3	cubic yard
km^2	square kilometer	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.

4. PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Paracatu Mine, also referred to as the Morro do Ouro Mine, is located immediately north of the city of Paracatu and 230 km southeast of the national capital of Brasilia in northwestern Minas Gerais State, Brazil (Figure 4-1). The Project comprises an open pit mine as well as processing plants, tailings facilities and associated infrastructure. The mine commenced production in 1987 and currently processes ore at a nominal plant throughput rate of 61 Mt/a.

4.2 Mineral Tenure

The Paracatu mine property consists of five grouped mining leases, nineteen exploration permits, and eleven exploration permit applications grouped into four non-contiguous blocks and totalling approximately 25,986 ha (Figure 4-2). The Universal Transverse Mercator (UTM) co-ordinates for the approximate centre of the Property are 8,098,500 mS and 298,000 mE (WGS84, Zone 23). The geographic coordinates are approximately 17°13’15”S latitude and 46°52’30”W longitude

As of the effective date of this report, all the leases and permits comprising the property are in good standing.

Kinross pays R$2.36 per hectare in annual exploration permit renewal fees to the Departamento Nacional da Producão Mineral (DNPM) during the initial three years of exploration. Renewed exploration permits beyond the initial three year exploration period require an annual renewal fee of R$3.58 per hectare.

Kinross first acquired a 49% interest in the Project upon completion of the merger with TVX Gold Inc. (TVX) and Echo Bay Mines Ltd. (Echo Bay) on January 31, 2003. On December 31, 2004, Kinross purchased the remaining 51% interest in the mine from Rio Tinto for US$260 million.

The grouped mining leases are located within the municipality of Paracatu. They are licensed as Mining Group number 238/2010 and are tied to the DNPM process number 931.299-2009 which is a combination of DPMN process numbers 800.005-1975, 830.241-1980, 832.225-1993, 832.228-1993 and 830.907-1999. The mining leases are confirmed by legal survey.

Figure 4-1: Paracatu Mine Location Map

The Paracatu mine area is located on properties that are owned by KBM, or on easements that are on a planned acquisition schedule. The current tailings impoundment is located on lands to which KBM has negotiated surface rights with the former landowner(s).

In addition, KBM holds title to nineteen exploration permits totalling approximately 12,033 ha and has applications before the DNPM for an additional eleven concessions totalling approximately 12,037 ha. These exploration permits and applications for exploration permits comprise a significant land package around the Paracatu Mine, covering the northern and southern strike extensions of the Morro do Ouro Member of the Paracatu Formation.

Table 4-1: Paracatu Mineral Rights

DNPM #	Type	Date Requested	Date Granted	Maturation Date	Area (ha)
931.299/2009	Grouped Mining Lease	19-May-09	25-Mar-10		
830.241/1980	Mining Lease (Suspended by 931.299/2009)	11-Mar-80	09-Aug-85		827.56
800.005/1975	Mining Lease (Suspended by 931.299/2009)	02-Jan-75	22-Jun-95		430.40
830.907/1999	Mining Lease (Suspended by 931.299/2009)	17-Feb-99	06-Mar-07		45.88
832.228.1993	Mining Lease (Suspended by 931.299/2009)	21-Jun-93	13-May-09		402.25
832.225/1993	Mining Lease (Suspended by 931.299/2009)	21-Jun-93	04-May-09		210.57
Subtotal					**1,916.66**
832.229/1993	Exploration Permit	21-Jun-93	22-Jan-03	19-Nov-08	256.30
835.561/1993	Exploration Permit	18-Oct-93	01-Sep-97	08-Dec-99	131.00
830.742/2005	Exploration Permit	04-Apr-05	01-Jun-09	01-Jun--11	376.64
831.537/2005	Exploration Permit	04-Jul-05	10-Jun-09	10-Jun-11	402.70
831.358/2005	Exploration Permit	13-Jun-05	24-Apr-09	24-Apr-12	138.64
831.892/2005	Exploration Permit	17-Aug-05	23-Mar-09	23-Mar-12	0.97
831.896/2005	Exploration Permit	17-Aug-05	23-Mar-09	23-Mar-12	1,879.09
830.140/2006	Exploration Permit	24-Jan-06	30-Jun-10	30-Jun-13	21.70
831.942/2005	Exploration Permit	22-Aug-05	02-Mar-11	03-Mar-14	411.90
831.944/2005	Exploration Permit	22-Aug-05	27-Apr-06	27-Apr-09	613.12
831.945/2005	Exploration Permit	22-Aug-05	23-Mar-09	23-Mar-12	1,366.63
830.949/2011	Exploration Permit	01-Apr-11	18-May-11	18-May-14	587.50
830.558/2007	Exploration Permit	15-Feb-07	07-Mar-12	07-Mar-15	1,446.09
830.557/2007	Exploration Permit	15-Feb-07	07-Mar-12	07-Mar-15	502.91
834.687/2007	Exploration Permit	27-Nov-07	07-Mar-12	07-Mar-15	1,458.97
831.900/2005	Exploration Permit	17-Aug-05	17-Mar-10	17-Mar-13	385.44
831.943/2005	Exploration Permit	22-Aug-05	17-Mar-10	17-Mar-13	48.35

DNPM #	Type	Date Requested	Date Granted	Maturation Date	Area (ha)
831.894/2005	Exploration Permit	17-Aug-05	31-May-10	31-May-13	1,776.30
830.801/2995	Exploration Permit	11-Apr-05	30-Jun-10	30-Jun-13	228.60
Subtotal					**12032.85**
833.477/2006	Exploration Permit Application	13-Nov-06			1,030.13
830.906/2012	Exploration Permit Application	26-Mar-12			1,750.24
830.854/2012	Exploration Permit Application	20-Mar-12			730.45
831.232/2012	Exploration Permit Application	06-Apr-12			1,000.06
833.345/2012	Exploration Permit Application	17-Oct-12			987.43
831.827/2007	Exploration Permit Application	11-Jun-07			1,469.99
832.896/2012	Exploration Permit Application	24-Sep-12			907.67
831.002/2006	Exploration Permit Application	21-Jan-12			743.58
831.003/2006	Exploration Permit Application	24-Jan-12			1,630.74
831.628/1985	Exploration Permit Application	09-Feb-09			800.00
834.827/2008	Exploration Permit Application	27-Dec-13			986.43
Subtotal					**12,036.72**
TOTAL					25,986.23

4.3 Mineral Rights

In Brazil, the DNPM issues all mining leases and exploration concessions. Mining leases are renewable annually and have no set expiry date. Each year KBM is required to provide information to the DNPM summarizing mine production statistics.

Exploration concessions are granted for a period of three years. Once a company has applied for an exploration concession, the applicant holds a priority right to the concession area as long as there is no previous ownership. The owner of the concession can apply to have the exploration concession successively renewed. Renewal is at the sole discretion of DNPM. Granted exploration concessions are published in the Official Gazette of the Republic (OGR), which lists individual concessions and their change in status. The exploration concession grants the owner the sub-surface mineral rights. Surface rights can be applied for if the land is not owned by a third party.

The owner of an exploration concession is guaranteed, by law, access to perform exploration field work, provided adequate compensation is paid to third party landowners and the owner accepts all environmental liabilities resulting from the exploration work.

In instances where third party landowners have denied surface access to an exploration concession, the owner maintains full title to the concession until such time as the issue of access is negotiated or legally enforced by the courts. Access is guaranteed under law and the owner of an exploration concession will eventually gain easements to access the concession. KBM has previously used the easement process to obtain surface rights from landowners during development of the Paracatu mine.

Once access is obtained, the owner has three years to submit an Exploration Report (ER) on the concession. The owner of a mineral concession is obligated to explore the mineral potential of the concession and submit an ER to the DNPM summarizing the results of the fieldwork and providing conclusions as to the economic viability of the mineralization. The content and structure of the report is dictated by the DNPM and a person with suitable professional qualifications must prepare the report.

The DNPM will review the ER for the concessions and will either:

- approve the report provided DNPM concurs with the report's conclusions regarding the potential to exploit the mineralization;
- dismiss the report should the report not address all requirements, in which case the owner is given a term in which to address any identified deficiencies in the report; or
- postpone a decision on the report should it be decided that exploitation of the deposits are temporarily non-economic.

Approval, dismissal or postponement of the ER is at the discretion of the DNPM. There is no set time limit for the DNPM to complete the review of the ER. The owner is notified of the DNPM's decision on the ER and the decision is published in the OGR.

On DNPM approval of the ER, the owner of an exploration concession has one year to apply for a mining lease. The application must include a detailed Development Plan (DP) outlining how the deposit will be mined.

DNPM reviews the DP and decides whether or not to grant the application. The decision is at the discretion of DNPM, but approval is virtually assured unless development of the project is considered harmful to the public or the development of the project compromises interests more relevant than industrial exploitation. Should the application for a mining lease be denied for exploration concessions for which the ER has been approved, the owner is entitled to government compensation.

On approval of the DP, DNPM grants the mining licence, which remains in force until the depletion of the mineral resource.

4.4 Royalties and Other Encumbrances

KBM must pay a royalty equivalent to 1% of net sales to the DNPM. An additional royalty of 0.5% is due to the holders of surface rights in the mine area not already owned by KBM.

4.5 Permitting

Kinross has obtained required permits for current activities and expects to obtain permits for future proposed work on the property as necessary. Permitting is discussed further in Section 20.2.

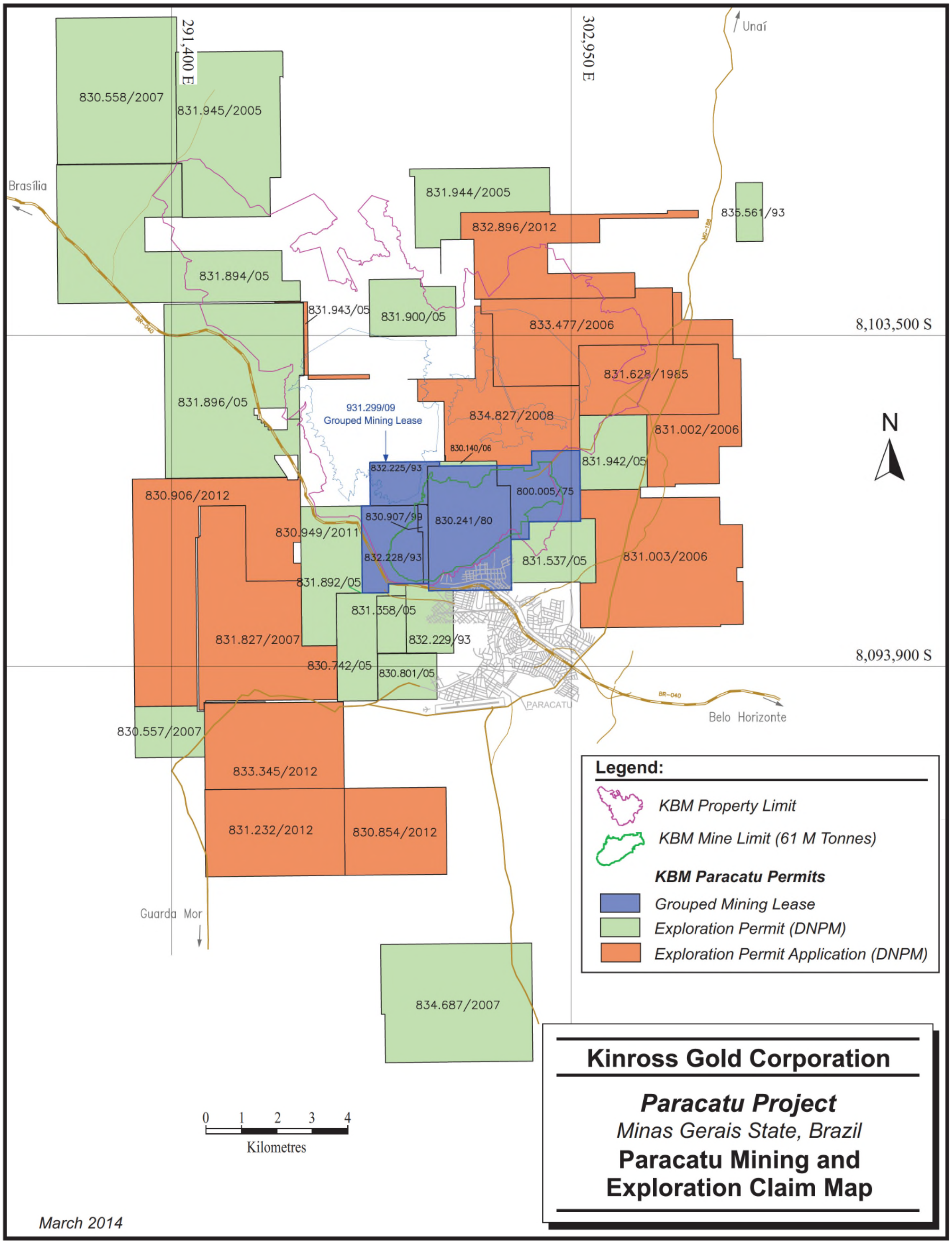

Figure 4-2: Paracatu Mining and Exploration Claim Map

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

5.1 Accessibility

The Paracatu Property is located immediately north and west of the Municipality of Paracatu, Minas Gerais, which has a population of approximately 85,000. Access from Paracatu is by vehicle via a four lane paved mine access road.

Paracatu is located approximately 230 km southeast of the national capital, Brasilia (population 2.5 million), and 480 km northwest of the state capital Belo Horizonte (population 2.5 million). Both are modern cities with industrial and manufacturing facilities.

A small paved airstrip that can accommodate small, charter aircraft also services Paracatu.

5.2 Climate

The climate is tropical sub-humid with a mean temperature of around 21°C that typical ranges from 17°C to 28°C. A peak temperature of 30°C usually occurs in August or September, while a low of 14°C is usually experienced in June or July. The rainy season is from October to March, although there is precipitation throughout the year. The relative humidity is at least 75% at least six months of the year.

The mine operates year round in this climate.

5.3 Local Resources and Infrastructure

Various services are available at Paracatu including housing, temporary accommodations, health services and police services. Building supplies and fuel are also available. A greater range of services, including mining equipment suppliers, mining contractors and trained manpower, can be obtained at Belo Horizonte.

The mine is connected to the national power grid, which relies mainly on hydroelectric generation. Electricity is subject to a free market environment with consumers able to select their supplier of choice. KBM obtains electricity from Centrais Elétricas Minas Gerais (CEMIG).

5.4 Physiography and Environment

Paracatu is located in the Brazilian savannah, a region characterized by low rolling hills that have been largely cleared of vegetation to support farming and cattle ranching. The elevation at the mine site is approximately 780 masl. The region is largely dependent on agriculture, with soya beans being the predominant crop.

6. HISTORY

6.1 Prior Ownership

Billiton acquired the original licences in what is now the Project and in 1980 entered into a joint venture partnership with Riofinex do Brasil (Riofinex), a subsidiary of Rio Tinto. In 1984, Riofinex acquired Billiton's interest in the property.

In 1985, RTZ Mineracão, a successor company to Riofinex, entered into a joint venture with Autram Mineracão e Particpaçoes (Autram). A new entity, Rio Paracatu Mineracão (RPM), was formed to hold the joint venture interests, with Rio Tinto holding a 51% interest and Autram holding a 49% interest in RPM.

Subsequently, Autram's interest in RPM was acquired by TVX Particpaçoes which later became TVX. TVX then entered into an agreement with Newmont Mining Corporation (Newmont) that resulted in Newmont and TVX each holding a 24.5% interest in RPM. In early 2003, TVX acquired Newmont's 24.5% interest to hold a 49% interest in RPM. In late January 2003, Kinross acquired its interest in the property by merging with TVX and Echo Bay. On December 31, 2004, Kinross purchased the remaining 51% interest in RPM from Rio Tinto for US$260 million. In 2010, the name of the operating entity was changed to KBM.

6.2 Exploration and Development History

The mining history of the Paracatu region is closely associated with the activities of the Portuguese *bandeirantes* who prospected for gold in Brazil's interior, arriving in the Paracatu region in 1722 after the discovery of gold in alluvial deposits.

Alluvial mining peaked during the second half of the 18th century. Such mining activities were not limited to the placer deposits along Rico Creek, but also extended to the oxidized ore outcrop on the top of Morro do Ouro hill or the "Hill of Gold".

Gold production declined sharply during the first decade of the 19th century. From this point forward, production was limited to subsistence mining practiced by local inhabitants known as *garimpeiros*. Various prospectors explored the region but economically viable operations were limited as a result of the low-grade nature of the deposits.

Beginning in 1970, Paracatu attracted some attention from mineral exploration companies looking for lead and zinc deposits in the area. Interest in the gold of Morro do Ouro was limited as the majority of the companies were not attracted by low grades that were initially considered to be too low to be extracted economically.

In 1984, Riofinex embarked on a surface exploration program that focused on the oxidized and weathered horizons of the Moro do Ouro area. At the end of 1984, based on the data from hundreds of test pits up to 25 m deep and further supported by a total of 44 drill holes, a “reserve” of 97.5 million tonnes at 0.587 g/t Au was estimated for what is currently known as the Paracatu Mine. Kinross notes that this estimate pre-dates NI 43-101, cannot be relied upon, and is quoted for historical purposes only.

Work by Riofinex and its various partners from 1984 to 2001 included 458 trenches (5,070 linear metres) and 674 drill holes (27,569 m) as listed in Table 6-1.

Table 6-1: Paracatu Historical Drilling

Year	Campaign	Hole Type (Diameter)	Number of Holes	Total (m)
1984	PMO	6"	44	2,462
1988	PAR	6"	26	1,014
1989	PRF	RC	67	2,791
1990	PRI	6"	15	652
1992-1997, 1999, 2000	PMP	6"	275	7,958
1993, 1996, 1997	PB2	6"	36	1,857
1993-1996	FPA	6"	97	3,405
1996	ALB	6"	11	335
1996	RAB	6"	20	583
2000	MA	HX (3")	2	35
2000	PEC	HX (3")	32	2,658
2000, 2004	WCR	HX (3")	9	2,031
2001	PPC	HX (3")	38	1,732
2001	PTE	HX (3")	2	56
Totals			**674**	**27,569**

The 1984 “reserve” estimate only included the superficial oxidized ore, then categorized as type C or T ore. Despite the low gold grade, Riofinex’s geologists believed that profitable extraction of the ore could be realized. In 1985, this was confirmed by a feasibility study. Total investment up to that period was $7.3 million including ground acquisition costs, exploration costs, and the cost of the feasibility study. Approval was granted by Rio Tinto to construct a mining project at a capital cost of approximately $65 million.

Production at Paracatu commenced in October 1987, treating oxidized and highly weathered ore from the C and T ore horizons described in Section 5.0 of this report.

The first gold bar was poured in December 1987. The following year, mine throughput reached the design capacity of 6.1 Mt/a.

After start-up, the throughput rate was progressively increased to 13 Mt/a, as a result of a number of improvement programs. In 1993, an $18.3 million Optimization Project was commissioned to provide extra water and flotation capacity to the circuit.

Throughput at Paracatu was increased again to 16 Mt/a in 1997 after the completion of Expansion Project I with a capital cost expenditure of $47.3 million.

Expansion Project II (1999) increased the plant design throughput to 20 Mt/a after a capital investment of $6.2 million. Due to an increase in ore hardness, throughput fell to 17.0 Mt/a at this time.

Expansion projects including construction of Plant II have increased mill throughput to the current design throughput of 61 Mt/a.

6.3 Historical Resource Estimates

At the end of 1984, based on the data from hundreds of test pits with depths of up to 25 m deep, and further supported by a total of 44 drill holes, a "reserve" of 97.5 million tonnes at 0.587 g/t Au was estimated at what is currently known as the Paracatu Mine. This estimate only included the superficial oxidized ore, then categorized as type C or T ore. Kinross notes that this estimate pre-dates NI 43-101, cannot be relied upon, and is quoted for historical purposes only.

6.4 Past Production

The property has been in continuous operation since 1987. Table 6-2 summarizes the historical life of mine production at Paracatu since commencement of commercial production.

Table 6-2: Paracatu Life of Mine Production Summary

Year	Tonnes milled (million)	Feed grade (Au g/t)	Gold Produced (oz)
1987	0.5	0.78	3,884
1988	6.2	0.77	113,257
1989	8.2	0.67	145,844
1990	9.3	0.64	160,258
1991	10.1	0.61	166,053
1992	10.5	0.58	167,000
1993	13.0	0.50	174,699
1994	13.4	0.50	169,003
1995	13.6	0.49	162,844
1996	13.5	0.50	165,646
1997	15.3	0.47	156,687
1998	15.6	0.48	181,305
1999	17.5	0.45	188,938
2000	19.7	0.47	228,866
2001	16.5	0.45	186,915
2002	18.4	0.48	224,539
2003	18.4	0.44	200,691
2004	17.3	0.44	188,574
2005	17.2	0.42	180,522
2006	18.1	0.39	172,683
2007	19.3	0.37	173,720
2008	20.5	0.39	190,810
2009	39.7	0.42	352,693
2010	42.7	0.45	479,575
2011	44.5	0.42	449,436
2012	53.0	0.38	463,393
2013	55.7	0.38	497,940
Total	**547.7**	**0.445**	**5,945,775**

7. GEOLOGICAL SETTING

7.1 Regional Geology

The Paracatu Property is hosted within the Brasília Belt, a north-south trending Neoproterozoic belt that extends along the western side of the São Fransisco-Congo Craton (Figure 7-1).

The Brasilia Belt resulted from the collision between three cratonic blocks: the Amazonian, the São Fransisco-Congo and a third block concealed under Phanerozoic sediments of the Paranã Basin to the south. Sedimentary units are mostly preserved in the northern part of the belt, whereas in the southern part where Paracatu is located, there is intense deformation and metamorphism, and contacts between metasedimentary units are primarily tectonic (Rodrigues et al., 2010).

The Brasília Belt has four main components (Rodrigues et al., 2010 and references therein):

- A continental block of Archaean rock units (the Crixás-Goiás region).
- Reworked sialic basement of Paleoproterozoic age, exposed mainly in the Almas-Cavalcante region.
- The Goiás Magmatic Arc, consisting of volcano-sedimentary rocks and tonalite/granodiorite gneisses.
- Thick sedimentary and metasedimentary sequences, including coarse and fine grained sediments with some carbonates, volcanic layers, phyllites, quartzites, and schists.

A series of east-northeast trending thrust faults is developed extensively along the belt. Metamorphic grade increases towards the west as the thickness of the fold belt increases. The timing of deformation is estimated at 800 Ma to 600 Ma during the Brasiliano orogenic cycle.

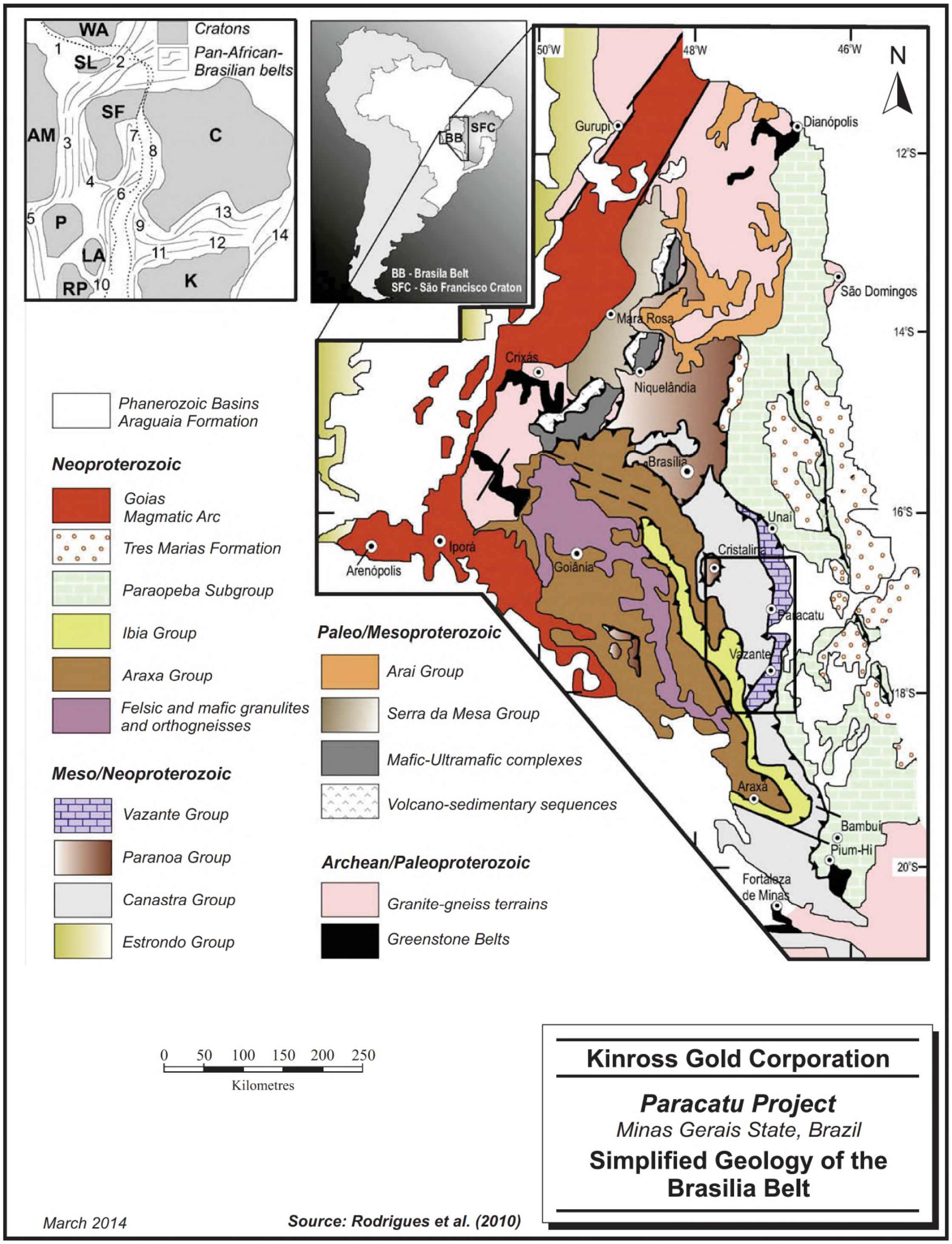

Figure 7-1: Simplified Geology of the Brasilia Belt

7.2 Local Geology

The Project is underlain by a thick sequence of phyllites belonging to the basal part of the Paracatu Formation of the Upper Proterozoic Canastra Group, which is exposed along the south-central portion of the Brasília Belt (Rodrigues et al., 2010; Figure 7-2). The Canastra Group is composed of sandy and shaley metasedimentary rocks, metamorphosed to greenschist grade. Because of intense deformation, its stratigraphy is not fully understood.

Figure 7-3 shows the stratigraphic column for the Canastra Group and the overlying Ibiá Group. The Canastra Group is made up of the following lithostratigraphic units, from base to top, which are separated from each other by thrust faults (Rodrigues et al. 2010).

<u>Serra do Landim Formation</u>

This unit mainly consists of calciferous shales and schists, with marble and limestone lenses.

<u>Paracatu Formation</u>

The Paracatu Formation includes the basal Morro do Ouro Member, a 100 m thick layer of dark carbonaceous phyllite, and the overlying Serra da Anta Member, a sericitic phyllite. Both phyllites display fine-grained quartzite intercalations.

<u>Chapada dos Pilões Formation</u>

This unit includes the basal Serra da Urucânia Member, a succession of quartzite and phyllite, and the upper Hidroelétrica Batalha Member, consisting of fine-grained quartzite and thinly-bedded phyllite.

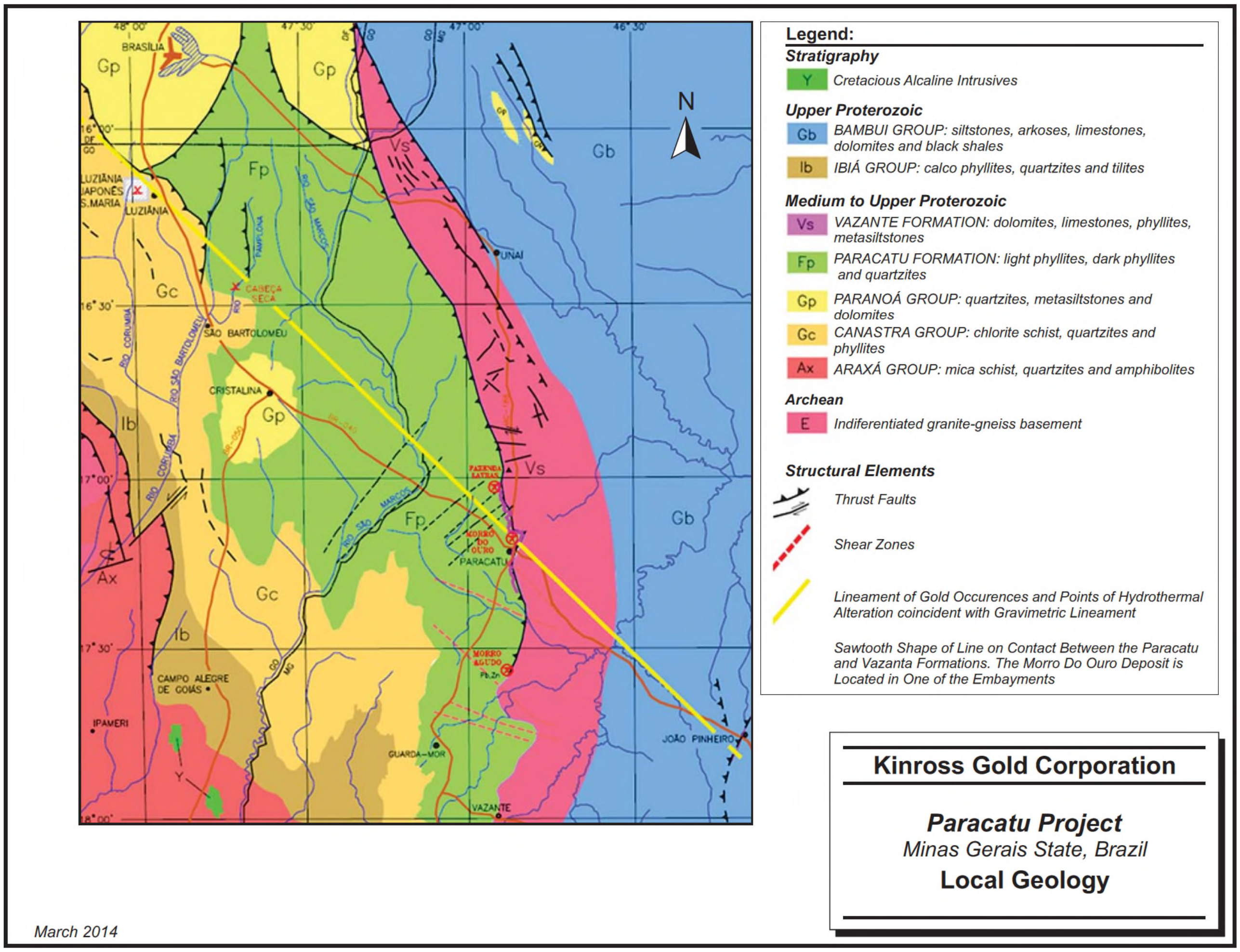

Figure 7-2: Local Geology

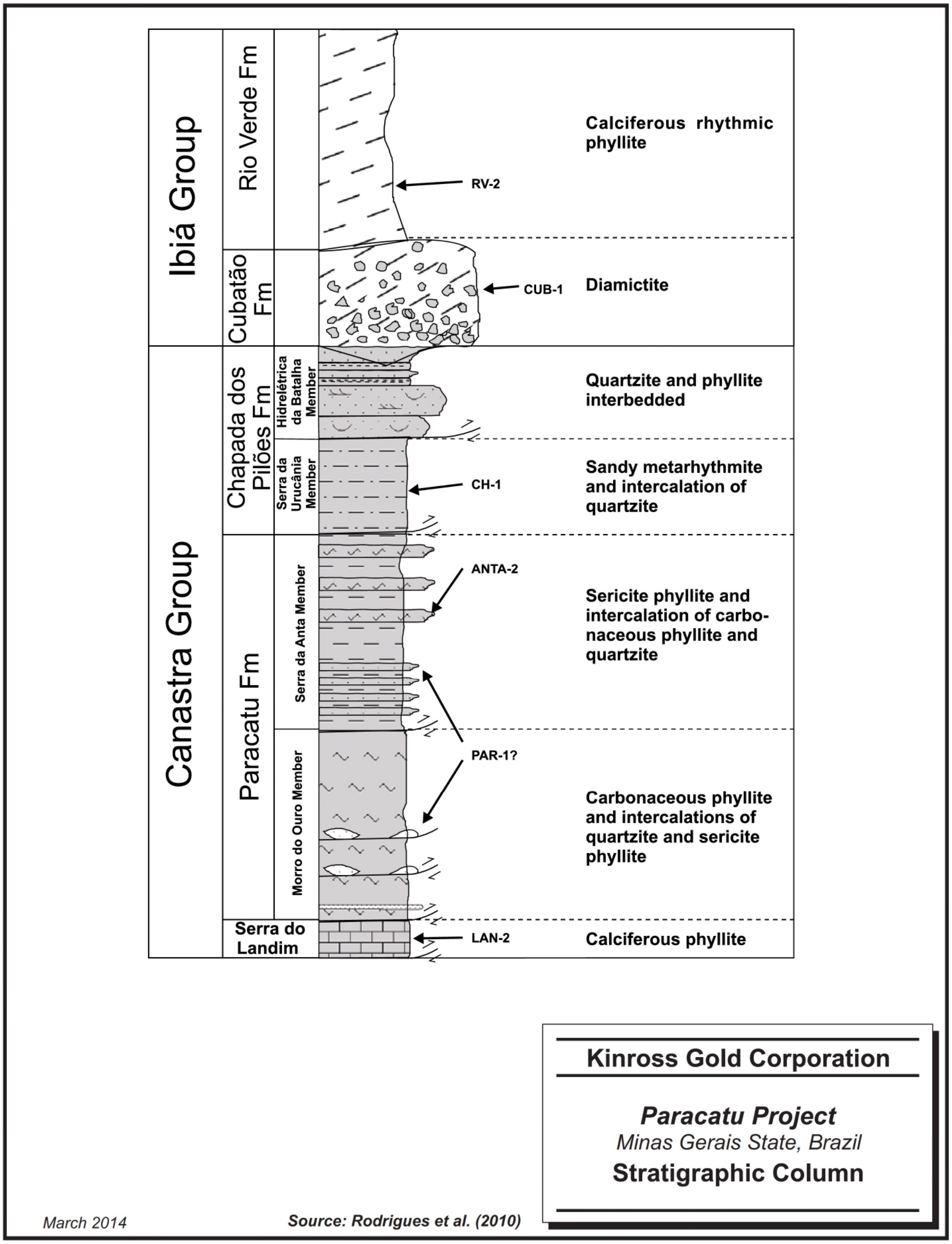

Figure 7-3: Stratigraphic Column

7.3 Property Geology

The phyllites at Paracatu lie within a broader series of regional phyllites of the Morro do Ouro Member of the Paracatu Formation. These phyllites exhibit extensive deformation and feature well-developed quartz boudins and associated sulphide mineralization. Sericite minerals are common, likely as a result of extensive metamorphic alteration of the host rocks. Primary sedimentary features and bedding planes are recognizable, but are intensely deformed by sygmoidal and boudinage structures, and by thrusting, particularly along bedding planes.

Mineralization at Paracatu is closely related to a period of ductile deformation, shearing and thrust faulting. Overall, the Morro do Ouro sequence has been thrust to the northeast. Intense, low angle isoclinal folds are commonly observed. The mineralization plunges to the west-southwest at 15° to 20° and there is secondary folding with axial planes striking to the northwest resulting in kink bands and egg box folds in some areas.

The mineralization appears to be truncated to the north by a major normal fault trending east-northeast as shown in Figure 7-4. The displacement along this fault is currently unknown but the fault is used as a hard boundary during mineral resource estimation. The current interpretation is that the fault has displaced the mineralization upwards and erosion has removed the mineralization in the up-thrown block.

Figure 7-5 presents a conceptualized geological cross section looking to the northwest through the deposit. The section shows the high strain zone in red surrounded by the weakly mineralized phyllites of the Morro do Ouro Member.

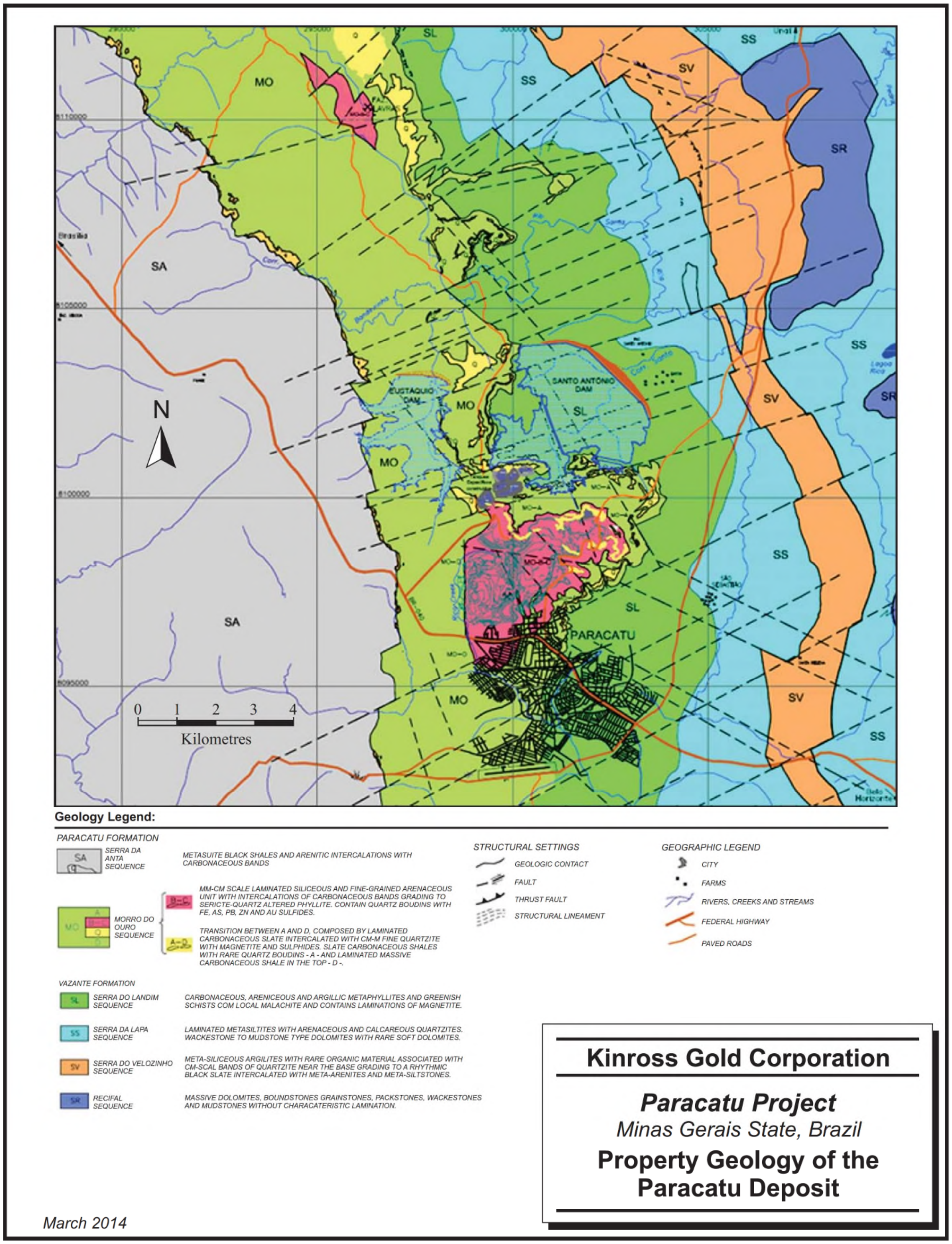

Figure 7-4: Property Geology of the Paracatu Deposit

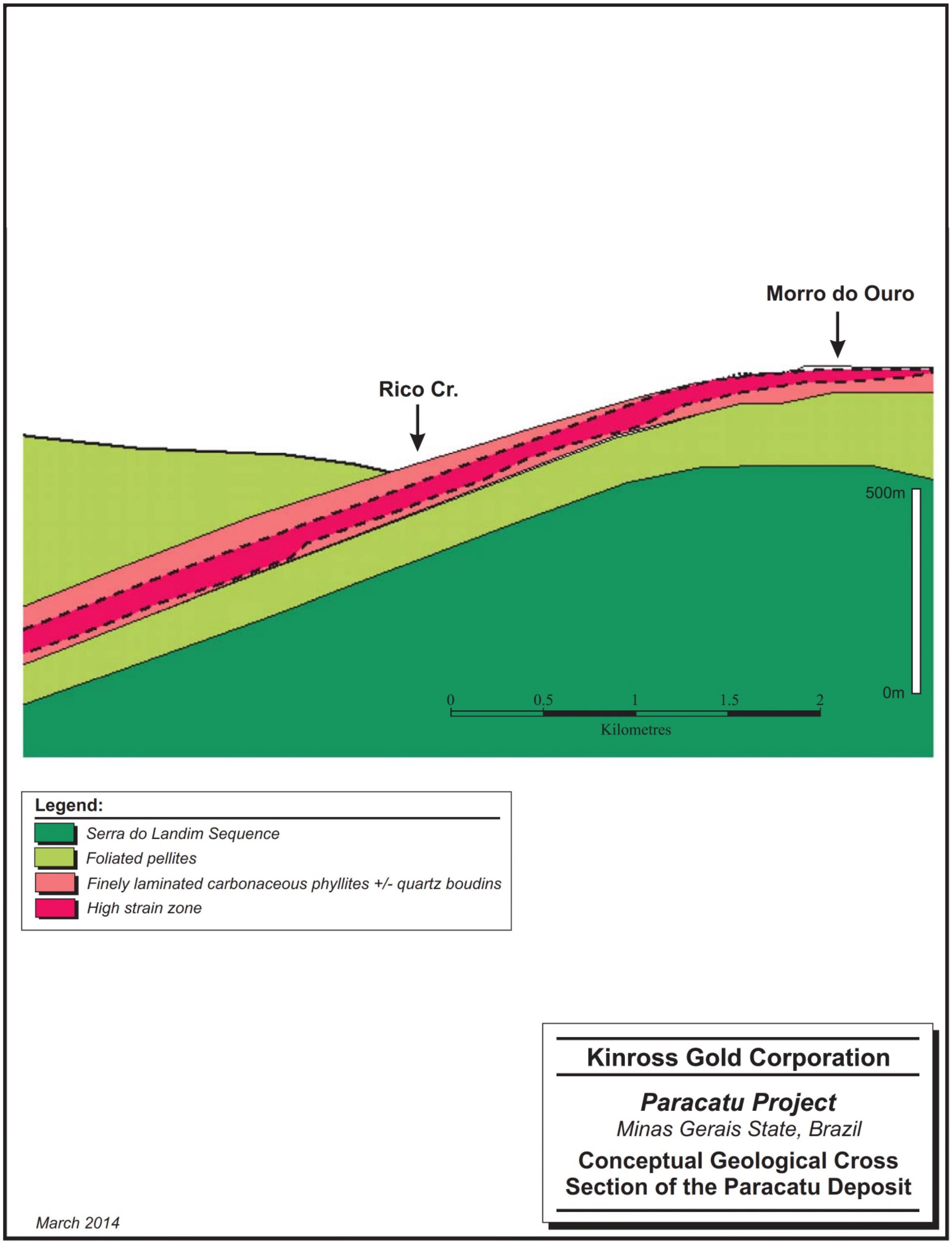

Figure 7-5: Conceptual Geological Cross Section of the Paracatu Deposit

7.4 Mineralization

The Paracatu mineralization is subdivided into four horizons defined by the degree of oxidation, surface weathering and sulphide mineralization. These units are, from surface, the C, T, B1 and B2 horizons. Figure 7-6 presents the conceptual pre-mining weathering surface and establishes the relative relationship between the various zones. Mining to date has exhausted the C and T horizons. The remaining mineral reserves are hosted exclusively in the B1 and B2 horizons. For this geological model, the interpreted geologic profiles combines the C, T and B1 horizons into a single unit.

Type C mineralization occurs at surface and extends to a depth of 20 m to 30 m. Type C mineralization is completely altered with no remaining sulphides. It also features localized laterite development.

The T horizon is generally only several metres thick, and marks the transition from the C horizon to the B1 horizon.

The B1 horizon is dark in colour and carbonaceous, with less oxidation than the C horizon. Sulphides have been completely oxidized but some fresh sulphide material is visible in the quartz boudins.

B2 mineralization was originally described as unweathered or fresh mineralization with primary sulphides.

The contact between unmineralized host rock (Type A – Oxide and Fresh) and the various mineralized horizons is gradational, occurring over a 10 m thick interval that is characterized by arsenic values of 200 ppm to 500 ppm and gold values of up to 0.2 g/t.

The mineralization at Paracatu is indicative of metamorphic alteration of lower greenschist grade. Petrographic studies of the B1 mineralization indicate that quartz and sericite make up 80% of the rock mass. Carbon occurs in the form of a fine opaque dust disseminated within the individual sericite bands. Carbon content varies from 5% to 20%. Minor amounts of ilmenite, tourmaline, anatase, rutile and limonite are also commonly observed.

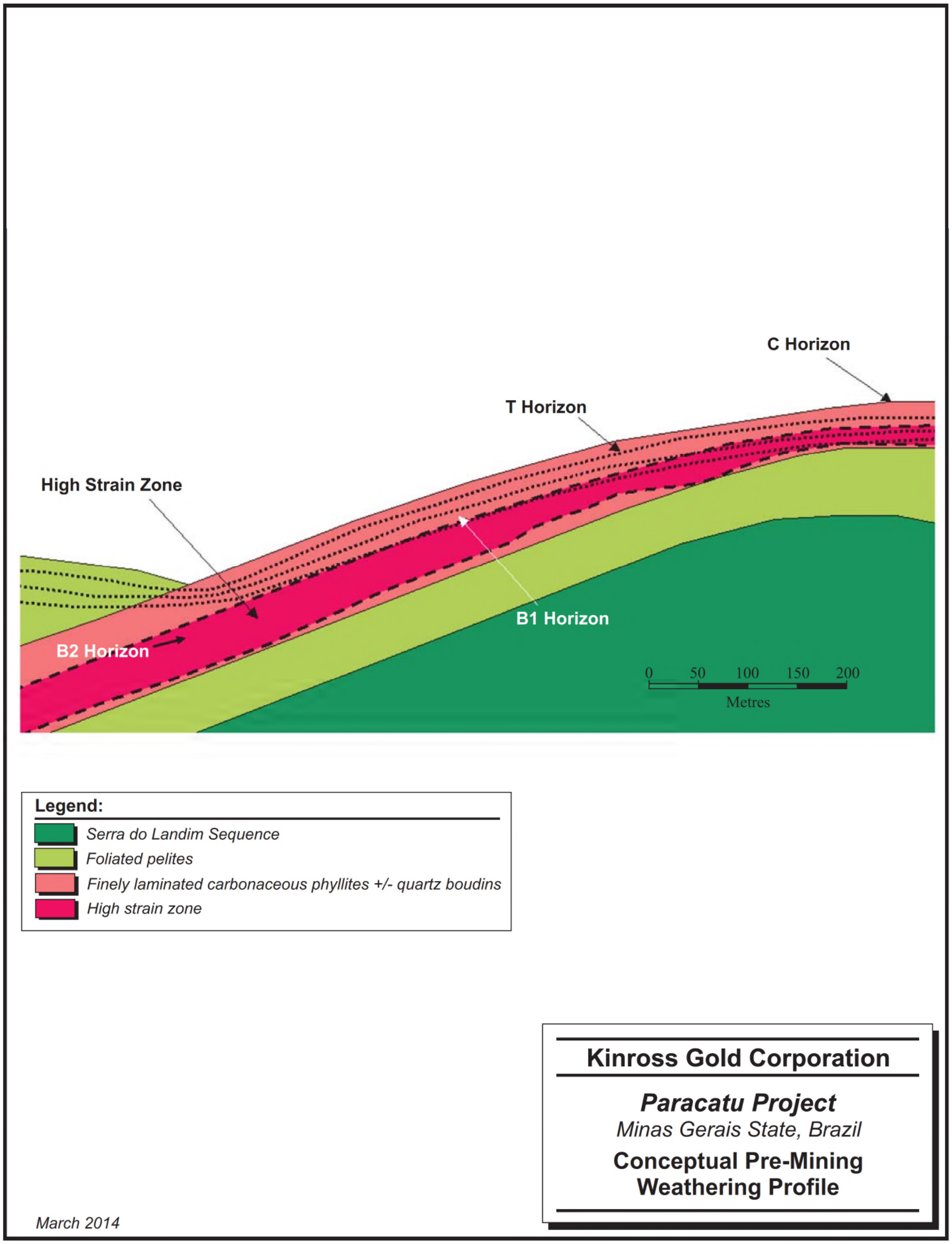

Figure 7-6: Conceptual Pre-Mining Weathering Profile

Petrography indicates that 60% to 90% of unoxidized phyllites were composed of variable amounts of quartz and sericite to produce a distinctive banding. Individual bands are typically less than two centimetres thick. The phyllites also contain up to 20% carbonate (calcite and ankerite). Fine grained carbon was also observed in the less weathered samples. Accessory minerals included muscovite, biotite, albite, tourmaline, ilmenite, chlorite, zircon and rutile.

The amount of sulphides present does not typically exceed 3% to 4%. The most common sulphides observed are arsenopyrite, pyrite and pyrrhotite. Galena is relatively common and may be accompanied by sphalerite. Chalcopyrite occurs locally in fractures within the main sulphide minerals listed above. The sulphides typically occur as individual crystals or coarse crystalline aggregates. Arsenopyrite is the most common sulphide and occurs as a fine grained (<1 mm) to coarse grained (>3 mm) aggregates. Crystals up to one centimetre in size are not uncommon. Arsenopyrite crystals increase in size to the southwest.

The mineralization at Paracatu exhibits distinct mineralogical zoning with the arsenopyrite content increasing towards the center and west and in the zones of intense deformation. Gold grade increases with increasing arsenopyrite content.

Pyrrhotite occurs in the western part of the deposit and gold grades are elevated where pyrrhotite increases. There is evidence for a pyrrhotite-rich deposit at depth, which has been intersected in a number of drill holes.

The deposit formation model proposed for Paracatu suggests that gold and arsenopyrite were introduced concurrently during the deformation event.

The quartz boudins typically observed in the higher grade portions of the Paracatu deposit represent original, attenuated quartz veins. The boudins crosscut bedding at a shallow angle. The boudin thickness likely represents the original thickness of the quartz veins, which have been stretched considerably, implying moderately high to very high strain in the system (Holcombe, 2005). The boudin formation has been interpreted as a two-stage process. First, quartz veins are emplaced early in the deformation event. As stress builds, these veins are folded, boudinaged and separated. Mineralized boudins are parallel to foliation. A final barren quartz stockwork phase cross cuts foliation in the low grade hanging wall.

Gold occurs either as free gold or electrum. Microscopic analysis indicates that 92% of the gold at Paracatu is free-milling with less than 8% encapsulated by sulphide grains or silica.

8. DEPOSIT TYPES

The Paracatu deposit is a metamorphic gold system with finely disseminated gold mineralization hosted within metasedimentary rocks. Very fine and evenly distributed gold is disseminated throughout a thinly bedded, highly deformed phyllite of Upper Proterozoic age. The deposit is part of a northwest-southeast lineament of gold occurrences including Cabeça Seca and Luziânia.

Gold mineralization was introduced syn-tectonically as the result of metamorphic alteration during thrusting of the Morro do Ouro Sequence over the rocks of the younger Vazante Formation. Structural interpretation suggests that mineralization was precipitated within a high strain zone where silica and carbonate were depleted from host phyllites, resulting in an increase in graphite content that may have acted as a chemical trap, precipitating out gold and sulphide mineralization remobilized during the metamorphic alteration of the Morro do Ouro Sequence.

The deposit has extraordinary lateral and longitudinal continuity. The majority of exploration efforts have sought to better define the continuous longitudinal continuity of mineralized phyllites at depth west of Rico Creek and the lateral limits of the economic mineralization.

9. EXPLORATION

Rio Tinto was the first company to apply modern exploration methods at Paracatu.

The exploration history at Paracatu has evolved concurrently with mine development. Initially, the exploration effort was focused only on defining mineral resources within the C and T horizons. As a result, the majority of the supporting sample data were limited to within 25 m to 30 m of surface. As mining of the C and T horizons advanced and the initial capital investment was recovered, the decision was made to evaluate the B1 horizon. Subsequent exploration drilling was focused on defining the deposit through drilling to the bottom of the B1 horizon.

As more knowledge was gained by mining the B1 horizon, the potential of the B2 horizon became increasingly important and exploration drilling was extended to test the entire thickness of the C, T, B1 and B2 horizons.

9.1 Mining Leases

Since acquiring the Project in 2003, Kinross has completed a significant amount of drilling within the main mining area. Section 10 of this report documents drilling activities on the property.

9.2 Exploration Licences

Exploration outside of the immediate mine area was initiated in 2006 and can be divided into two programs based on proximity to the mine site.

In the licensed exploration areas immediately bordering the mine leases, exploration activities were concentrated on soil and termite-mound geochemical sampling and interpretation of airborne magnetic survey data to look for nearby features similar to Paracatu. Some target areas were generated, mostly located west and west-northwest of the mine. Follow-up exploration returned no significant results.

A near-pit geophysical survey was performed in 2008 to define the induced polarization (IP) and resistivity geophysical signature for the known buried mineralization of the down-dip southwest extension of the B2 ore zone below and west of Rico Creek. A pattern was identified indicating higher chargeability in the non-mineralized zone above the ore zone, and high resistivity at depth within the ore zone.

Geophysical data were the primary driver of exploration in the licensed exploration areas located 10 km or more from the mine. Definition of favourable structural zones using regional airborne magnetic data yielded three targets which were then surveyed for IP and resistivity. Two targets were located approximately 50 km to 60 km from the mine and the third target was 10 km from the mine. Carbon-rich phyllites with quartz boudins and pyrite similar in lithologic character to the Paracatu deposit, but without gold and arsenopyrite, were identified in one of the targets located further from the mine. Although the samples did not contain gold mineralization, the program was successful in detecting litho-structural controls similar to those associated with the Paracatu deposit.

10. DRILLING

Since acquiring the Property in early 2003, Kinross has completed 1,186 diamond drill holes for a total of 96,823 m, as documented in Table 10-1.

Table 10-1: Kinross Drilling Campaigns Since Property Acquisition

Year	Campaign	Hole Type (Diameter)	Number of Holes	Total (m)
2004	PE	HX (3")	60	1,997
2005	K	HQ, HTW, NQ	267	48,660
2006	KAB	HQ	36	3,786
2006	KMA	HQ	5	574
2007	KOP	HQ	93	2,365
2008	KGM	HQ	36	2,445
2009	K09	HQ	64	2,001
2010	FRG	HQ	2	49
2010	KES	HQ	1	115
2010-2011	K10	HTW	129	6,843
2011	K11	HTW	80	5,192
2012	K12	HTW	307	16,774
2013	K13	HTW	106	6,022
Totals			**1,186**	**96,823**

Figure 10-1 is a plan map of the drill holes completed on the mining leases. Representative examples of drill sections are provided in Section 14.

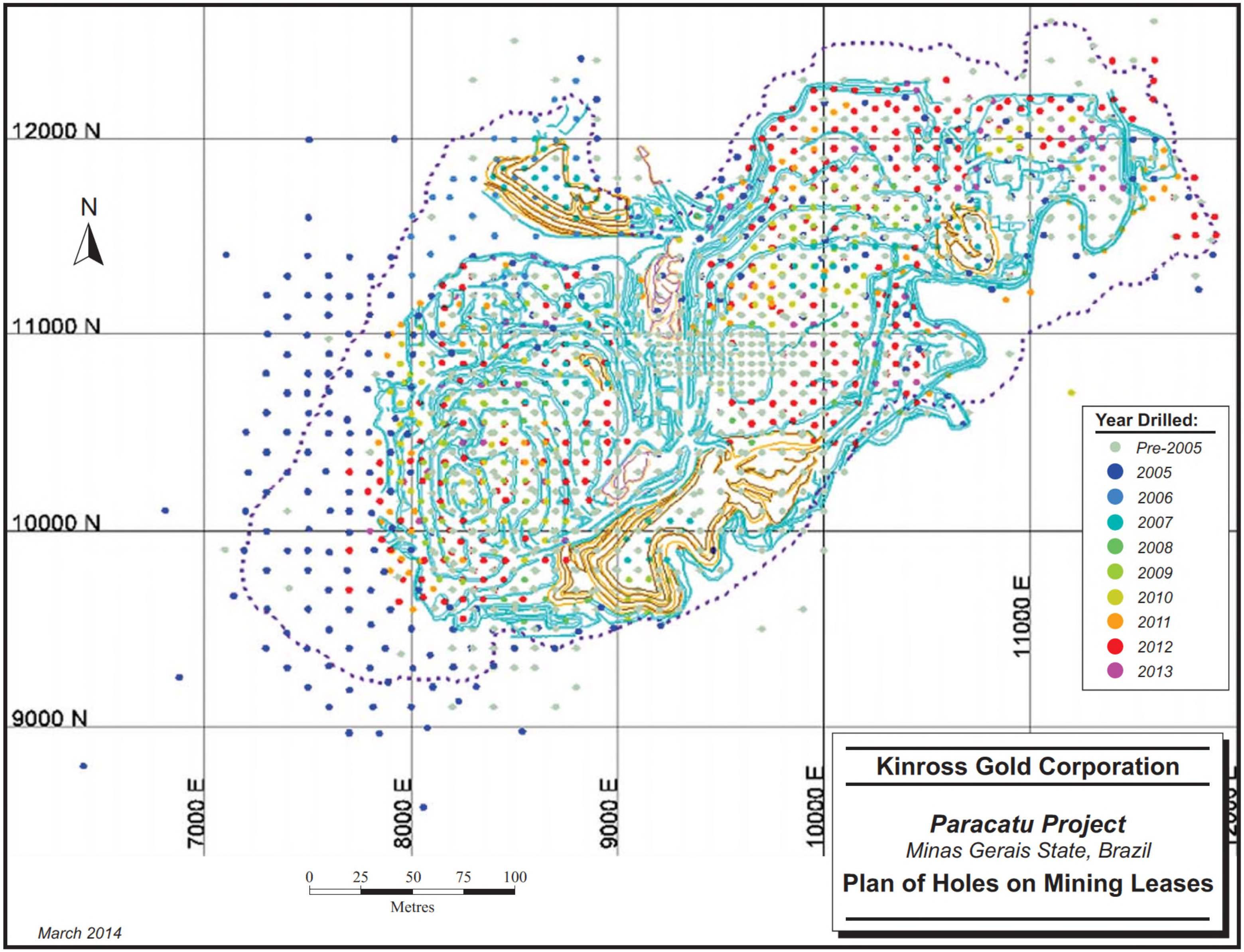

Figure 10-1: Plan View of Drill Holes Completed on Mining Leases

Core diameters for holes drilled by Kinross between 2005 and 2013 range from HX (76.2 mm), HQ (63.5 mm), HTW (70.9 mm), and NQ (47.6 mm).

All drill hole collars were established in the field by KBM's mine surveyor using standard Topcon GPS system. The drill hole is collared as close as possible to the collar coordinates established by the surveyors with most holes collared within five metres of their planned location.

All drill setups (-90^o) are checked by KBM geologists before beginning drilling. KBM geologists controlled the hole completion depths. A minimum of 20 m of barren core (no arsenopyrite and no boudins) beyond the interpreted footwall contact was the criteria used to terminate drilling.

Several holes west of Rico Creek were surveyed using a downhole instrument. The initial drill holes were surveyed using acid tube tests and a tropari instrument. Deviation was typically 2° per 100 m. Azimuth readings from the tropari tool were often suspect.

Later in the program, an E-Z shot system was used for down-hole directional surveying. Results from the E-Z shot instrument confirmed that some of the tropari readings were erroneous. Generally pyrrhotite content was low enough that magnetic error is thought to be marginal. Given the continuity and homogeneity of the mineralized zone and the wide spacing of drilling, inclinometry variance is thought to have marginal effect.

Hole collars were surveyed again by the mine surveyor after drilling. A 6 m PVC casing rod was placed downhole in as many collars as possible and collars were cemented into a cairn, labelled, and photographed with landmark backgrounds. All drill sites were cleaned, including removing the drill cuttings for storage at the KBM waste dump site, and backfilling the water sumps.

Core was collected continuously from the collar. Wooden tags were placed in the core trays and labelled according to the drill run. All core boxes were clearly labelled with the hole number and drilled interval. Lids were nailed on each core box at the drill site to facilitate transport to the KBM logging facility.

Drill reports identified all zones of broken ground, fault zones and water gain or loss. Water gain or loss was almost non-existent. Rusty water seams in the B2 horizon were extremely rare, suggesting that active hydrology occurs almost exclusively in the weathered zone only.

Core recovery is typically greater than 95%. Global average recoveries are reported to be greater than 99%. Kinross is not aware of any drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of the results.

Drill core logging is recorded on paper and later transcribed to Excel, exported as a comma delimited (csv) file and finally imported into acQuire. All pertinent features are logged, including lithology, alteration, weathering, structure, boudin structures, percent sulphides, etc. Currently the transcription is checked by the Senior Geologist prior to entry into acQuire to ensure that logged fields match expected codes. Any changes made must be noted as revisions in the logs so that they can be checked against the acQuire database.

11. SAMPLE COLLECTION, PREPARATION, ANALYSES, AND SECURITY

Kinross uses industry standard sample preparation, analysis, data management and security procedures for its drill programs. Kinross is of the opinion that the adequacy of the samples taken, the security of the storage and shipping procedures, the sample preparation, and analytical procedures used meet industry standard practices and that the results are suitable to estimate mineral resources and mineral reserves.

11.1 Sampling Method and Approach

Drill core is transported by KBM personnel from the drill site to the core logging facility for logging and sampling. Technicians check depth markers and box numbers, reconstruct the core, and calculate core recovery. The core is logged descriptively and marked for sampling by KBM geologists. Logging and sampling data are recorded on hardcopy logs which are later entered into Excel and imported into acQuire software. Core is photographed prior to sampling.

Core recovery from all diamond drill programs is excellent, averaging greater than 95%. The greatest areas of core loss were from the collar to 15 m downhole in laterite zones. KBM employed a systematic sampling approach where drill core was sampled using standard one metre sample lengths.

Whole core was submitted for analysis after the core had been logged and photographed. Reference pieces are 8 mm cores used for density and point load testing (PLT). These pieces are labelled and stored at the core logging facility. This practice of whole core sampling is acceptable for deposits with a low average grade and good grade continuity. Kinross does not consider the sampling of whole core to be a concern considering the property’s production history.

Only mineralized zones were sampled. The remaining non-mineralized core is stored in labeled metal boxes both at the logging facility and an enclosed secured storage building near the plant. Some core that was assessed to be low grade was chip sampled every one metre and composited to eight metres. In the few cases where the sample returned assay values close to 0.2 g/t Au, the entire eight metres was re-sampled in the traditional one metre interval pattern.

Prior to the start-up of the mine, all samples were shipped to independent analytical laboratories in-country for analysis. After construction of the mine, most samples were processed at the on-site laboratory by KBM employees.

11.2 Sample Security and Chain of Custody

Core samples for analysis are stored in a secure warehouse at site prior to sample preparation. The warehouse is either locked or under direct supervision of the geological staff. Prior to shipping, drill core samples are placed in large rice bags and sealed. A sample transmittal form that identifies each batch of samples is prepared. The samples are transported directly to the laboratory for sample preparation and analyses.

All core boxes are covered with wooden lids and nailed shut before being transported by KBM personnel from Geoserve or Geosol rigs to the logging facility located inside the fenced mine gates. After photographing, logging and marking one metre sample intervals, the whole core is placed in heavy gauge plastic bags with a unique sample tag. The sample tag number is also written in indelible marker on the outside of each sample bag.

Samples to be analyzed at the KBM laboratory are loaded by KBM personnel onto pickup trucks and transported to the KBM crushing facility. After crushing, samples are again transported by pickup truck to the RPM preparation laboratory where samples are riffle split. Approximately 6 kg are stored as coarse rejects and 2 kg are transported by pickup truck to the RPM assay laboratory for pulverization and analysis.

Samples that are to be analyzed by either Lakefield or ALS Chemex are loaded onto transport trucks operated by their respective laboratories and delivered to the appropriate sample preparation facilities in Belo Horizonte or Luziânia.

Analytical results are received electronically from the laboratories and imported into acQuire. Assay batches are reviewed for acceptance by the database administrator.

11.3 Sample Preparation, Analyses and Security

Prior to the start-up of the mine, all samples were shipped to independent analytical laboratories in Brazil for analysis. After the construction of the mine, most samples were processed at the on-site laboratory. However, in order to meet the demands of the extensive 2005 drill program, Kinross used the following three independent laboratories to perform the analyses:

- ALS Chemex sample preparation facility in Luziânia and ALS Chemex analytical facility in Vancouver, Canada (ISO 9001 Certified).
- SGS Lakefield laboratories – Belo Horizonte, Brazil (ISO 17025 Certified)

- KBM sample preparation and analytical facility, Paracatu. (ISO 14001 Certified).

The on-site laboratory is a not a certified analytical facility.

Most samples were prepared by crushing to 95% passing 2.0 mm to 3.5 mm depending on the lab. Two kilogram splits of crushed material were then pulverized to 95% passing 100 to 150 mesh. The remaining coarse reject was stored.

Until 2005, Kinross reduced the nugget effect by combining results from six separate fire assays of 50 g sample aliquots. Each sub-sample was fire assayed followed by an Atomic Absorption (AA) finish. In June 2005, Kinross commissioned Agoratek International to conduct a review of exploration sampling procedures and to assess the requirements for six 50 g aliquot assays per sample (Bongarcon, 2005). Agoratek, led by Dominique Francois-Bongarcon, a recognized expert in sampling, reviewed the sampling procedures and concluded that three 50 g analyses would be sufficient for the purposes of the exploration program. Since then, three sub-samples have been used.

Kinross standardized sample preparation and analytical procedures for all three labs as closely as possible, given equipment limitations and differences in internal lab Quality Assurance/Quality Control (QA/QC) protocols.

11.4 Background on Quality Assurance and Quality Control

Quality assurance (QA) measures involve collecting evidence to demonstrate that the precision and accuracy of the assay data is within generally accepted limits for the sampling and analytical methods used in order to have confidence in the resource estimation. Quality control (QC) consists of procedures used to ensure that an adequate level of quality is maintained throughout the process of sampling, preparing, and assaying the samples. In general, QA/QC programs are designed to prevent or detect contamination and allow analytical precision and accuracy to be quantified. In addition, a QA/QC program can disclose the overall sampling and assaying variability of the sampling method itself.

Accuracy is assessed by a review of results of certified reference material (CRM) standards, and by check-assaying at outside accredited laboratories. Assay precision is assessed by reprocessing duplicate samples from each stage of the analytical process from the primary stage of sample splitting, through the sample preparation stages of crushing/splitting, pulverizing/splitting, and assaying.

QA/QC programs were limited during early exploration at Paracatu. The primary QC procedure was the use of inter-laboratory check assays where results from RPM's analytical lab were compared with results from Lakefield Research in Canada. Additional check assay work was carried out at the Anglo Gold laboratories in Brazil (Crixás and Morro Velho).

The KBM lab procedure includes insertion of certified analytical standards and blanks. At least one blank and standard is inserted with each analytical batch of 30 samples. Results are statistically analysed and if they lie outside the performance gates, all the samples within the batch are repeated. Other checks are also conducted throughout the fire assay process, such as lead recovery to the buttons and silver recovery for the prills. The analyses are repeated if recoveries are below the criteria.

Kinross compiles, analyses, and documents results of the QA/QC programs on a campaign by campaign basis. Details on the methods and results from two of the more extensive drill campaigns are described below.

11.5 Quality Assurance and Quality Control for 2005 Drill Program

A suitable QA/QC program was implemented for the three labs used during the extensive 2005 drill program. The program consists of standards and blanks inserted in the sample streams. All three labs also reported using round robin checks. The labs were visited on an infrequent and unannounced basis by RPM representatives. No major sample preparation discrepancies were noted. The ALSC analytical facility in Vancouver was not visited.

Kinross purchased certified standard material from Rocklabs (New Zealand) in two lots. The standards were selected to meet typical Paracatu grade ranges. These standards were OXA26, OXC30, OXD27, SE19, and SF12.

For blanks, a local crushed gravel (1 cm to 2 cm) of calcareous metasiltstone was used but was clearly identifiable by its white color. In order to make the blanks less obvious to lab employees, samples of barren hanging wall phyllite with similar characteristics as regular samples were used in the latter part of the drilling program.

A model numbering code system was generated that could accommodate the three different batch sizes of the three labs. Table 11-1 presents a comparison between internal QA/QC for the labs and the QA/QC system implemented by Kinross for the 2005 exploration-drilling program.

Table 11-1: Summary of QAQC Insertion Rates by Laboratory

Laboratory	Batch Size	Internal Lab QA/QC Standards	Internal Lab QA/QC Blanks	Internal Lab QA/QC Duplicates	Client QA/QC Standards	Client QA/QC Blanks	Samples/ batch
Chemex	84	2	1	3	2	3	73
Lakefield	50	1	1	2	1	2	43
RPM	30	1	1	0	1	1	26

Each batch contained a minimum of one standard and one blank per analytical furnace tray. Standards were numbered according to the number model and were shipped in a separate bag to be inserted into the sample stream at the preparation facilities. The standards were not inserted in a manner that prevented the analytical lab from distinguishing standards from samples. However, five different standards were used and it is reasonable to assume that the standards were blind.

Results of the certified standards indicated that both ALS Chemex and KBM returned results mostly within the +1 to -1 standard deviation limits, while Lakefield returned results within +0.5 and -1.5 standard deviation, showing a consistent bias of -0.5 standard deviation. As sample lots were shipped to all three labs throughout the program, no one lab significantly dominates a spatial area of the mineralized resource.

Overall results returned from all labs were within industry accepted tolerances with failure rates of 0.9% to 2.7% for the analyses performed. A failure on a standard is classified as ±2.5 standard deviations from the certified mean for each standard. All failures occurring within the identified mineralized horizon were re-analyzed.

No re-runs were requested due to blank failures. The number of blank failures within the mineralization was regarded as minimal.

A significant number of swaps between standards were noted, possibly due to sample numbering mistakes by the geologists inserting the standards or transcription errors at the receiving labs. Sample swaps were readily identifiable when plotting standard performance.

11.6 Quality Assurance and Quality Control for 2012 Drill Program

Kinross operated an extensive drill program in 2012 consisting of 307 holes totalling 16,774 m, drilled in two campaigns referred to as K12_15000m and K12_3000m. QA/QC results for each program are summarized separately below.

The K12_3000m drill hole program consisted of 57 HQ diameter holes totalling 2,835.5 m of drilling. A total of 135 coarse blanks of crushed limestone, 100 geostats standards and 335 KBM standards were inserted with the samples sent to the SGS laboratory, representing insertion rates of 4.8% for coarse blanks and 15.3% for the standards. In addition, 139 coarse reject duplicates were analyzed.

There was a 2.2% failure rate in the blanks, no failures of the geostats standards and 10.7% failure rate in the KBM standards for an overall failure rate of 6.8% (Table 11-2). Good laboratory performances were observed with the blanks and geostats standards. The majority of the failures occurred with the KBM standards and this was primarily due to the poor quality of the standard itself.

The coarse reject duplicates show an absolute mean relative percent difference (AMRPD) of 28.6 % which is similar to the results subsequently discussed in the K12_15000m program below. Additional review suggests reasonable repeatability without bias for grade ranges supported by adequate data.

Table 11-2: SGS Geosol Blank and Standard QAQC Results – 3,000 m Program

QC ID	Min (g/t Au)	Max. (g/t Au)	Accepted Value (g/t Au)	+/- 2.5 SD %Limit	No. Submitted	No. Pass	% Pass	No. Fail	% Fail
BLANK	0	0.051	0.015	n/a	135	132	97.8	3	2.2
G308-1	0.18	0.28	0.23	22%	50	50	100.0	0	0.0
G905-3	2.145	2.595	2.37	9%	50	50	100.0	0	0.0
STD-KP-SF-01	0.068	0.648	0.358	81%	114	103	90.4	11	9.6
STD-KP-SF-02	0.384	0.709	0.547	30%	145	136	93.8	9	6.2
STD-KP-SF-04	0.627	1.9835	1.226	62%	76	60	78.9	16	21.1
Total					**570**	**531**	**93.2**	**39**	**6.8**

The K12_15000m drill hole program consisted of 250 HQ diameter holes totalling 13,938.6 m of drilling. The samples for this program were sent to SGS Geosol (11,772.5 m) and Intertek (2,166 m).

For the QA/QC of the SGS Geosol sample preparation and assaying program (Table 11-3), a total of 891 standards and 502 coarse blanks of crushed limestone were inserted with the samples sent to the laboratory, representing insertion rates of approximately 7.6% for the standards and 4.3% for coarse blanks. In addition, 555 coarse reject duplicates were also analyzed (4.7% of the data).

Table 11-3: SGS Geolsol Blank and Standard QAQC Results – 15,000 m Program

QC ID	Min (g/t Au)	Max. (g/t Au)	Accepted Value (g/t Au)	+/- 2.5 SD %Limit	No. Submitted	No. Pass	% Pass	No. Fail	% Fail
BLANK	0.005	0.015	0.015	n/a	502	493	98.20	9	1.80
G998-6	0.65	0.95	0.8	19	46	45	97.80	1	2.20
OXA89	0.064	0.104	0.0836	24	194	185	95.40	9	4.60
OXC102	0.18	0.234	0.207	13	93	79	84.90	14	15.10
OXD87	0.384	0.449	0.417	8	10	9	90.00	1	10.00
OXE101	0.567	0.647	0.607	7	42	35	83.30	7	16.70
OXE86	0.561	0.666	0.613	9	10	9	90.00	1	10.00
OXF85	0.742	0.867	0.805	8	10	8	80.00	2	20.00
SE58	0.56	0.654	0.607	8	10	10	100.00	0	0.00
SE68	0.566	0.632	0.599	6	61	42	68.90	19	31.10
SF57	0.773	0.923	0.848	9	100	76	76.00	24	24.00
SH65	1.278	1.418	1.348	5	56	39	69.60	17	30.40
STD-KP-SF-01	0.068	0.648	0.358	81	110	101	91.80	9	8.20
STD-KP-SF-02	0.384	0.709	0.547	30	40	39	97.50	1	2.50
STD-KP-SF-04	0.627	1.9835	1.226	62	109	92	84.40	17	15.60
Total					**1,393**	**1,262**	**90.60**	**131**	**9.40**

There was a 1.8% failure rate in the blanks, and a 13.7% failure rate in the standards. These failure rates are considered high and are primarily a function of lab performance and sample swaps.

Statistical summaries of the coarse reject duplicate results suggest poor precision. This has always has been the case with Paracatu assays because of the variability of the mineralized material. There are indications of an analytical bias for the grade range above 1 g/t Au. Although this potential bias is based on a small number of data points, KBM should nevertheless visit the lab and examine protocols to determine what the potential cause may be.

For the Intertek QA/QC program (Table 11-4), a total of 89 coarse blanks of crushed limestone and 141 Geostats and Rocklabs standards were inserted with the samples sent to the laboratory, representing insertion rates of approximately 4.1% for coarse blanks and 6.5% for the standards. In addition, 98 coarse reject duplicates were also analyzed (4.5% of the data).

There was a 1.1% failure rate in the blanks, no failures of the Geostat OX89A standard and a very poor failure rate of 47% in the remaining standards for an overall failure rate of 26%. There was good performance from the labs on the blanks and Geostat OX89A standard. The coarse reject duplicates also indicate poor precision but are similar to other previous results for Paracatu samples.

Analytical results of standards submitted to Intertek indicate poor lab performance. The laboratory was notified of such results. KBM geology discontinued using Intertek and only one month's worth of data was compromised.

Table 11-4: Intertek Blank and Standard QA/QC Results – 15,000 m Program

QC ID	Min (g/t Au)	Max. (g/t Au)	Accepted Value (g/t Au)	+/- 2.5 SD %Limit	No. Submitted	No. Pass	% Pass	No. Fail	% Fail
Blank	0.005	0.015	0.015	n/a	89	88	98.9	1	1.10
OXA89	0.064	0.104	0.0836	24	21	21	100	0	0.0
SE68	0.566	0.632	0.599	6	55	24	43.6	31	56.4
SF57	0.773	0.923	0.848	9	40	22	55.0	18	45.0
SH65	1.278	1.418	1.348	5	25	15	60.0	10	40.0
Total					**230**	**170**	**73.9**	**60**	**26.1**

11.7 Bulk Density Measurements

Bulk density analyses have been completed at various times throughout the exploration and development of the project. The original values were based on the results of samples collected from the surface test pits. Mining of the deposit indicated that these bulk density values were comparably low, so efforts were made to obtain a more representative value.

Changes were made to the bulk density calculation methodology and a linear regression method was employed until 1999. Reconciliation to actual production statistics indicated problems with the density calculations and a study was commissioned to examine the bulk density estimates.

Rio Tinto Technical Services Ltd (RTTSL) developed a new method that combined statistical evaluation of near surface sampling for the C, T and B1 horizons with a linear regression approach for the data within the B2 horizon in those areas where deep drill coverage was limited. This new method has improved reconciliation relative to the actual mill production to within 1.5% of predicted tonnage figures.

At the mine, in situ density measurements are taken by extracting a 30 cm cubic block from the upper level of a bench. Generally two samples are taken and averaged to give a value for the bench. The density determination at the top of the bench is applied through the entire height of that bench (8.0 metres). The in situ density measurements that are made and applied do not take into account any variation with depth.

For the core samples, specific gravity is measured using the water displacement method. This method is considered appropriate for the B2 horizon.

For the core samples in the B1 horizon, the recent specific gravity measurements were factored down by 5% based on the average moisture content measured by the process plant. In B2 material, the dry and wet density measurements on the core samples showed no significant differences. It was therefore decided that the B2 values need not be modified.

11.8 Conclusion

In Kinross' opinion, the sample preparation, security measures and analytical procedures meet industry best practices

12. DATA VERIFICATION

The resource database was reviewed and verified during site visits, a series of verification exercises by different entities and a review of QA/QC results. Results from the QA/QC exercises are presented in Section 11. All other checks are described in this section.

Rio Tinto applied a rigorous data verification process at Paracatu where the database was manually verified against original assay and field certificates. Rio Tinto also completed biennial reviews of RPM's procedures and methodology. The 1998, 2000 and 2002 reviews concluded that the procedures met Rio Tinto's corporate guidelines for resource modelling and reserve estimation.

Kinross independently verified 10% of the data collected between 1999 and 2004 against original source documents. The holes were chosen at random and any errors against original sources were documented. Results identified a single transcription error that was made in the arsenic values for an entire hole. No other errors were identified. The Kinross geology department recently verified 5% of the data collected between 2010 and 2012 against original source documents. The verification did not identify concerns regarding the quality or accuracy of the database.

All data generated during the extensive 2005 drill program were verified by Kinross' exploration geologists. Gold grades were all double entered and weight averaged per sample, then the two databases were cross-checked, with no significant errors or differences detected. The summary database spreadsheet was compared to the individual digital assay certificate files sent by the different laboratories.

The site performed several database checks, including tests for unreasonable grades and sample lengths, from/to mix-ups, missing sample numbers, duplicate sample numbers, unusual maximum or minimum values, etc. Collar locations were verified visually with respect to the topographic surface and drill hole traces were inspected for unreasonable bends and orientations. No significant issues were identified.

As part of external auditing in 2006, 2009, and 2012, RPA verified the gold values in the database with the assay certificates for a total of 1,192 assays from 13 drill holes. No significant errors were identified. RPA also checked the downhole survey values and found no significant errors.

In Kinross' opinion, the data is suitable for use in Mineral Resource and Mineral Reserve estimation.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

The resource and reserve estimates summarized in this report are based on the operating conditions of Plant I and Plant II. Plant I focuses on processing the softer near-surface B1 ore at a design throughput of 20 Mt/a, whereas Plant II focuses on processing the harder B2 ore at a design throughput of 41 Mt/a. The availability of B1 ore for processing will continue to be assessed by Kinross. Total processing capacity may be limited to 41 Mt/a in 2016 due to lack of B1 ore.

13.1 Plant I

Plant I at Paracatu has operated continuously since 1987 with expansion upgrades in 1997 and 1999. In 2007, the plant processed 19.3 Mt/a and achieved an average gold recovery of 76%. A detailed discussion on Plant I is presented in Section 17.0 of this report. In summary, the plant consists of primary and secondary crushing, ball milling to 80% passing 75 microns, gravity recovery using jigs, rougher and cleaner flotation, concentrate regrinding and gold leaching in the carbon-in-leach plant (Hydromet Plant). Final gold bullion is produced from the carbon adsorption, desorption and electrowinning circuit.

13.2 Plant II

Plant II started production in September 2008, and achieved commercial production in December 2008. A detailed discussion of Plant II is presented in Section 17.0 of this report. Currently, Plant II comprises an in pit MMD crusher, a 1.8 km conveyor to a covered stockpile area, a 38 ft. diameter semi-autogenous grinding (SAG) mill, and four ball mills. The recovery process uses gravity and flotation to produce concentrate, which leached in a carbon-in-leach (CIL) circuit in the hydromet plant. Gold is recovered by a carbon elution and electrowinning process and refined to gold bars.

The plant has a nominal capacity of 41 Mt/a when processing ore with a work index below 8.7 kWh/t. Tonnage throughput will decrease as work index increases.

In 2013, the Paracatu plant processed 55.7 Mt/a and achieved average gold recovery of 75.8%.

13.3 Bond Work Index

Samples for Bond Work Index (BWI) testing were collected during sample preparation of the 1 m raw samples. Composite samples were originally based on an 8 m down hole length representing the current mining bench height. The current model is based on a 12 m bench height which required re-compositing of the historical 8 m data to reflect the change in bench height.

Each composite is composed of a fraction of each metre after initial sample crushing to 2 mm. The BWI test is completed at the RPM process lab according to the Bond Work Index standard test methodology.

13.4 Cyanide Destruction

KBM uses cyanide to dissolve gold in the Hydromet plant CIL circuit. In the current process, the CIL tailings containing sulphide concentrate residue and residual cyanide are sent to sumps located in the mine area. Here the sulphide concentrate solid residues are settled and the cyanide solution is returned to an AVR (Acid, Volatization and Recovery) plant which recovers approximately 60% of the cyanide and recycles back it into the process. The residual solution after AVR treatment is discharged with the flotation tailings into the tailings storage facility, where any residual cyanide is naturally degraded by volatilization and ultraviolet rays. KBM rigorously monitors the residual cyanide discharge and degradation in the tailings dam pond.

Over the last several years, KBM has also implemented a series of improvements to reduce cyanide consumption, such as the implementation of automatic feed systems, which has reduced the consumption per tonne of ore processed by more than 50%. A further improvement that has been introduced in the expansion is a pre-aeration step prior to leaching, which will enable the current cyanide consumption levels to be maintained (and not increased) despite the higher amount of concentrate that will be processed.

Studies were undertaken by KBM to evaluate the substitution of the existing AVR cyanide recovery process with the more modern and widely used SO_2/Air Cyanide Destruction Process. Laboratory testing of the SO_2/Air Process on KBM cyanidation tailings has demonstrated that the process has a series of advantages over the AVR process. These advantages include the following:

- Lower operating cost
- Less risk of personnel exposure to gaseous cyanide products
- Lower residual cyanide effluent
- No necessity for maintaining open settling ponds with high CN contents
- Operational simplicity and ease of control

A distinct advantage of the process is the ability to treat pulps without substantial loss of kinetics. The treatment facility has been designed to treat the entire tailings stream, composed of a slurry with 50% solids, before being pumped out to the settling ponds. The solid residues will settle in the ponds and the solution can be recycled to the plant or blended with the flotation tailings for disposal in the main tailings dam. The SO_2/Air Cyanide Destruction plant was installed as part of the Expansion III Project.

13.5 Desulphurization Testing

Historically, rougher tailings generated from Plant I and Plant II were discharged into the Santo Antônio tailings facility. However, with the implementation of Expansion III Project a new tailings facility was required. Construction of the new Eustáquio tailings facility began in 2009 and tailings have been discharged into the new basin since 2012.

Commitments were made to environmental regulators to discharge tailings containing 0.4% sulphur or less to the new Eustáquio Dam. Since the tailings contain an average concentration of 1.0% sulphur, desulphurization was needed to meet target values. The implemented desulphurization system makes use of the existing rougher flotation and cleaner flotation equipment, with the addition of a new flotation building to house gold cleaner flotation cells and sulphide re-cleaner flotation cells.

Bench scale and industrial scale test work on desulphurization of flotation tailings with sulphide recovery were completed and reported in the Feasibility Study for the Expansion III Project. Pre-commissioning of the desulphurization flotation plant was completed on November 26th, 2011 and operations began in 2012.

13.6 Recovery

The metallurgical recovery of gold decreases with increasing sulphur and arsenic content. Laboratory test work has been conducted on core samples to replicate the flow sheet. The data have been factored to correspond with actual plant operation and the calculated recovery entered into the block model.

For Plant I, process recovery is based on maximum recovery from both the plant and hydromet circuit (CIL), a combined value of 78.6%.

For Plant II, the recovery is seen to be a function of sulphide and arsenic content of the ore applied to the maximum plant and hydromet fixed recovery values.

14. MINERAL RESOURCE ESTIMATE

14.1 Summary of Mineral Resources

Mineral Resources are reported using two cut-off grades depending where the material is processed. Resources to be processed in Plant I were reported at a 0.139 g/t Au cut-off whereas resources to be processed in Plant II were reported at a cut-off of 0.208 g/t Au. Cut-off grades were determined using a long-term average gold price of US$1,400/oz. Processing recoveries and associated operating conditions were used to generate an optimized pit shell using the LG algorithm. Mineral Resources exclusive of Mineral Reserves were reported between the EOY 2013 ultimate pit design and the LG pit shell (Table 14-1). The Mineral Resources also include Inferred material and some incremental Measured and Indicated material situated in the pit design.

Table 14-1: Mineral Resources Exclusive of Mineral Reserves – December 31, 2013

	Tonnes (000s)	Gold (g/t)	Gold Ounces (000s)
Measured (M)	215,040	0.31	2,111
Indicated (I)	325,135	0.39	4,069
M+I	**540,175**	**0.36**	**6,180**
Inferred	3,239	0.27	28

Notes:
1. Mineral Resources are exclusive of Mineral Reserves.
2. Mineral Resources estimated according to CIM Definitions.
3. Mineral Resources estimated at $1,400/oz Au.
4. Mineral Resources are estimated at gold cut-off grades that vary by material type from approximately 0.139 g/t Au to 0.208 g/t Au.

14.2 Resource Database

Drill hole data is stored and maintained in an acQuire software database. The database is maintained at site and updated as new validated data become available. The drill hole data for the 2013 model were exported as comma separated files and imported into Datamine software. A total of 115,578 m of diamond core samples and 2,791 m of RC samples were available to estimate mineral resources. Channel and trench samples were not used for the estimation process.

Sections 11 and 12 describe the steps to validate the acQuire database. Kinross is of the opinion that the drill hole database is valid and suitable to estimate mineral resources.

14.3 Geological Model and Estimation Domains

Wireframe models used to constrain the mineralization and metallurgical relationships were reinterpreted in 2012 and 2013 based on additional infill drilling in 2011 and 2012.

The Morro do Ouro phyllite was subdivided into geo-metallurgical domains based on mineral processing criteria including quantities of Au, As, Pb, and Zn, S%, pyrrhotite mineralization and oxidation. Wireframe models were built for the following geologic model domains:

- **A** – This unit is defined as a fresh, weakly to non-mineralized phyllite unit located above and below the mineralization.
- **AOX** – This unit is defined as an oxidized, barren to weakly mineralized phyllite unit located in the hanging wall above the B1 domain.
- **B1** – This unit is defined as an oxidized, gold mineralized phyllite unit.
- **B2** – This unit is a fresh, sulphide-bearing, gold-mineralized and partially deformed phyllite with a gold grade of less than 0.5 g/t and an arsenic content of less than 2,500 ppm. The B2 domain is the dominant ore type at Paracatu.
- **B2 Calha** – This B2 sub-unit has a grade of approximately 0.5 g/t Au and elevated arsenic content between 2,500 ppm and 4,000 ppm. It is distributed along the preferential direction of mineralization from northeast to southwest.
- **B2 Calha IDS** – This B2 sub-unit contains a gold content greater than 0.5 g/t and has an arsenic content greater than 4,000 ppm. This sub-unit is domained within the lower arsenic-bearing B2 Calha sub-unit. In diamond core, this domain was classified as B2 “Calha Intensive Deformation and Silicification (IDS)”. This sub-unit commonly contains coarse textured arsenopyrite.
- **B2 with Pyrrhotite** – This B2 sub-unit contains pyrrhotite and is characterized by magnetic susceptibility values of > 1.8 x 10-4 SI associated with this mineral.
- **B2 with Lead and Zinc** – This B2 sub-unit contains > 200 ppm Pb or Zn. Galena and sphalerite were combined into a single domain as these two minerals usually appear coincident.

- **Quartzite** – Defined as a hard, non-foliated metamorphic rock located within the larger phyllite package. This unit occurs typically as the discontinuous contact between the B1 and B2 domains in the eastern portions of the deposit.

Geological units and the mineralized domains were flagged in the block model. These wireframes were filled either above or below Digital Terrain Model (DTM) surfaces, or inside solid models.

In general, internal waste and quartzite which define poorly mineralized material and poorly mineralized durable lenses within the phyllite accounts for about 5% of the material within the Life of Mine (LOM) pit. Remaining B1 mineralization accounts for approximately 11% of remaining material. Approximately 83% of remaining material is defined as B2, B2 Calha and B2 Calha IDS (Table 14-2).

Table 14-2: Proportion of Model Domains Located within the Life-of-Mine Pit.

Domain Code	Name	Description	% of Pit Material
5	Waste	Internal Waste	3%
6	Quartzite	Quartzite	2%
7	B1	Oxide ore material	11%
8 and 9	B2	Sulphide ore material	69%
10 and 11	B2 Calha	Sulphide ore with >2,500 ppm As	12%
12 and 13	B2 Calha IDS	Sulphide ore with >4,000 ppm As	3%

Eleven surfaces, seven wireframes and one string were used to assign mineralized domains to the drill hole data and block model. The west and east designations are used to segregate the mine into two transitionally distinct ore bodies to the east and west of Rico Creek. To the east of Rico Creek, the deposit is more shallowly dipping, while to the west it is more steeply dipping. Domain 7 represents B1 mineralization. Domain 8 and 9 represent B2 mineralization east and west of Rico Creek respectively. Domain 10 and 11 represent B2 Calha east and west of Rico Creek respectively. Domains 12 and 13 encapsulate B2 Calha IDS mineralization east and west of Rico Creek, respectively (Figure 14-1).

Modelled surfaces were used to define B1 and B2 domains with solids used to classify the B2 “Calha” and B2 Calha IDS domains (Figure 14-2). Solids are used to define the bodies of internal waste and quartzite within the mineralized package. Modelled solids of pyrrhotite enrichment and lead and zinc enrichment were also defined.

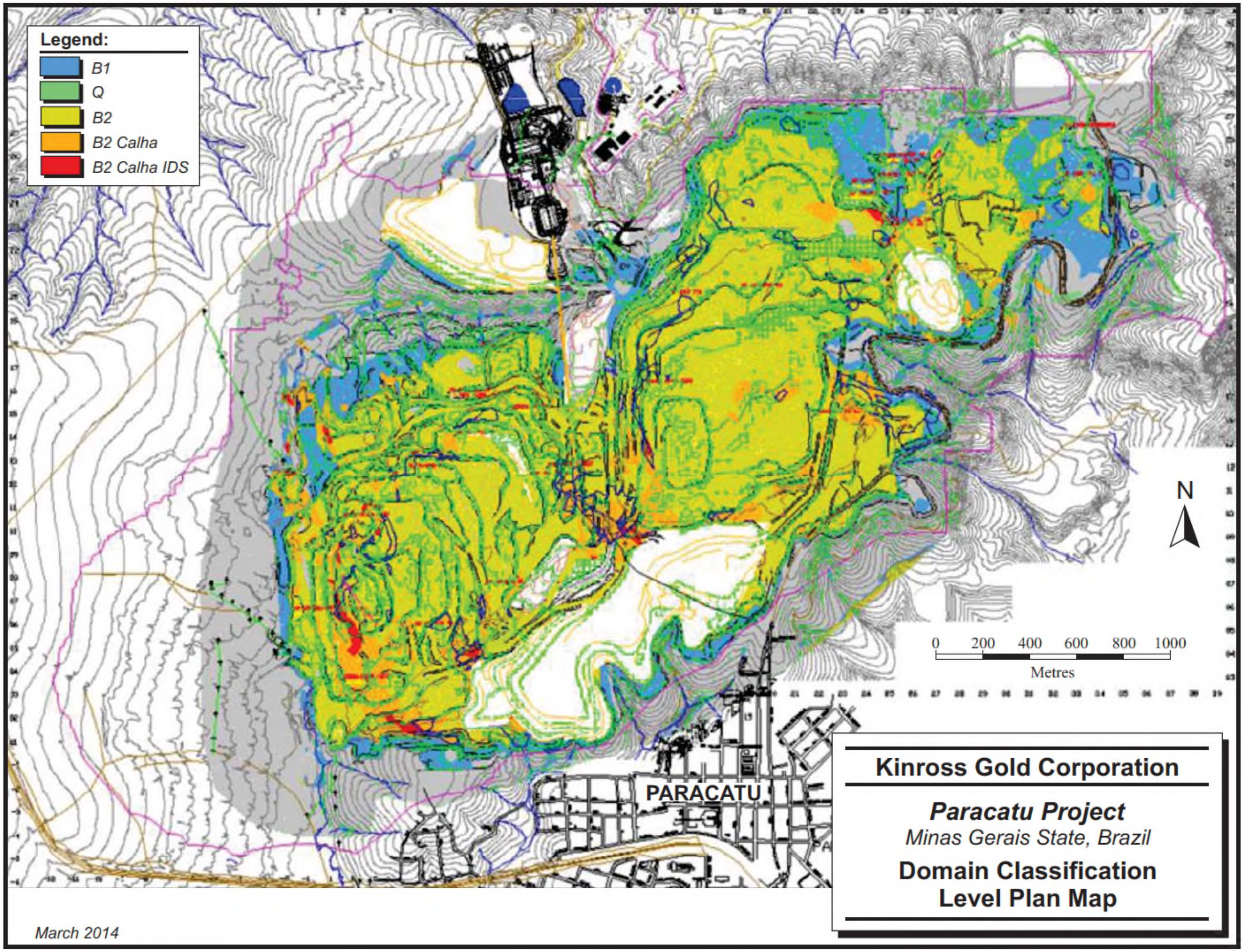

Figure 14-1: Domain Classification Level Plan Map

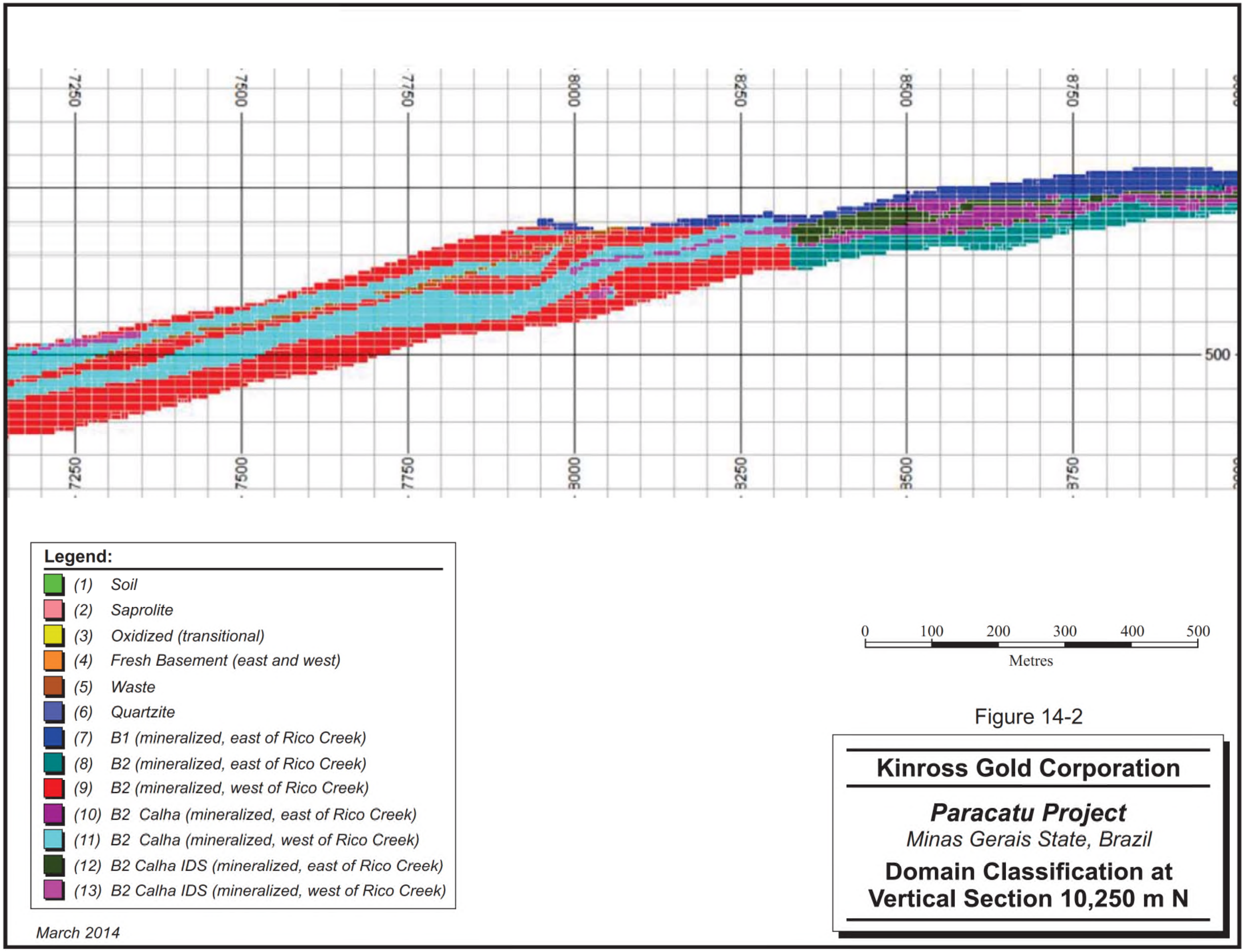

Figure 14-2: Domain Classification at Vertical Section 10,250 m N

14.4 Density

Kinross has a large density dataset comprised primarily of drill core measurements. In-situ bulk density measurements were excluded from the current model update.

The majority of the measurements were completed during the 2006 feasibility study. Measurements were made on 8 cm long pieces of drill core every two to six metres along the hole. These data were subsequently composited into four to six meter-long intervals. Density measurements for recent drilling use one 8 cm to 10 cm long sample collected every four metres. Samples were neither wax-sealed nor adjusted for temperature.

Density measurements segregated by primary oxidation domain (B1 and B2) demonstrate a significant decrease in variance when compared to the complete density database. Similarly, histogram plots of the entire population show a bimodal normal distribution reflecting contrasting densities of oxidized and fresh rock.

Density assignments are summarized in Table 14-3.

Table 14-3: 2013 Year End Density Database Summary

Rock Code	Name	Description	2013 Density (g/cm³)			
			nº Samples	Min	Max	Mean
1	Soil	Soil	41	1.732	2.839	2.418
2	AOX	Soil west of Rico creek	80	1.940	2.889	2.419
4	Waste	"A" Waste	5326	1.910	3.150	2.772
5	Waste	Internal Waste	1026	2.021	3.070	2.696
6	Quartzite	Quartzite	72	2.132	2.930	2.636
7	B1	Oxide Material	1015	2.000	3.200	2.414
8	B2 (including Pyrrhotite and Lead and Zinc)	Sulphide Material	12381	1.732	3.630	2.752
	B2 (Excluded Pyrrhotite)	Sulphide Material	10954	1.732	3.630	2.744
	B2 (Excluded Lead and Zinc)	Sulphide Material	11558	1.732	3.630	2.754
	B2 (Excluded Lead and Zinc and Pyrrhotite)	Sulphide Material	10349	1.732	3.630	2.746
9	Pyrrhotite	Pyrrhotite ore body	1427	2.146	3.050	2.818
10	Lead and Zinc	Lead and Zinc Ore body	834	2.100	3.000	2.725

14.5 Bond Work Index

The hardness of the mineralization is a key criterion used in estimating resources at Paracatu. In general, softer ore is sent to Plant I and harder ore is sent to Plant II. The mine planning teams decide the destination based on hardness, oxidation state and grade. Both hardness and Bond Work Index (BWI) are built into the block model.

Samples for BWI testing were collected during sample preparation of the one meter raw samples. Composite samples are based on a 12 meter downhole length representing the current mining bench height. The BWI test is completed at the Kinross process lab according to the BWI standard test methodology.

A total of 7,966 BWI tests were available in the database. The composite data is used to interpolate the BWI for individual blocks in the model using Ordinary Kriging. Interpretations of soft, medium and hard material based on threshold BWI criteria were created. Hardness surfaces are currently defined as:

- Soft – BWI < 7 kWh/t
- Medium – BWI between 7 kWh/t and 10 kWh/t
- Hard – BWI > 10 kWh/t

Results of the BWI block interpolation and hardness classification are shown in Figure 14-3.

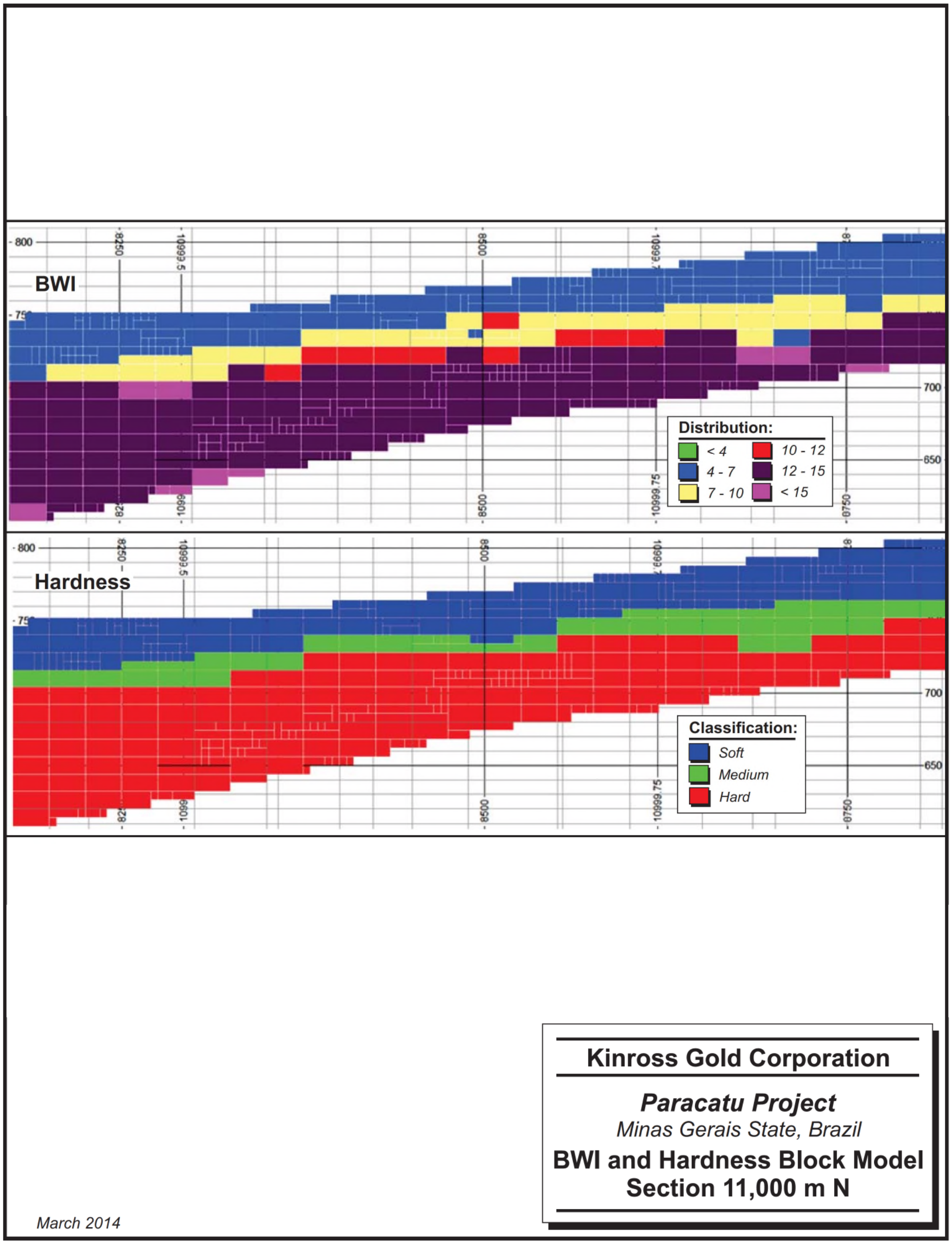

Figure 14-3: BWI and Hardness Block Models - Section 11,000 m N

14.6 Cutting High-Grade Values

Gold assays are capped at the 99th percentile prior to compositing (Table 14-4). Summary statistics are provided in Table 14-5. Capping is only applied to gold grades and was not applied to arsenic, sulphur, copper, manganese, lead, zinc and mercury values.

Table 14-4: Summary of High Grade Cutting Levels

Domain	Capping Level (g/t Au)	No. Capped	Percent Capped of Total
B1	1.459	199	0.01%
B2	1.533	581	0.01%
B2 Calha	1.745	144	0.01%
B2 Calha IDS	1.804	33	0.01%

Table 14-5: Assay and Composite Statistics by Domain

Item	Assay				Composite			
Au	b1-d	b1_tc-d (cap)	b2-d	b2_tc-d (cap)	b1-c	b1_tc-c (cap)	b2-c	b2_tc-c (cap)
Samples	17286	17286	47356	47356	3653	3653	9056	9056
Minimum	0.001	0.001	0.001	0.001	0.004	0.004	0.001	0.001
Maximum	29.251	1.465	94.661	1.517	5.028	1.33	16.0042	1.517
Mean	0.416	0.404	0.389	0.378	0.415	0.404	0.389	0.378
Standard deviation	0.414	0.274	0.558	0.270	0.242	0.187	0.264	0.168
CV	0.996	0.677	1.436	0.714	0.583	0.464	0.679	0.445
Variance	0.171	0.075	0.311	0.073	0.059	0.035	0.070	0.028
Au	ca-d	ca_tc-d (cap)	cai-d	cai_tc-d	ca-c	ca_tc-c (cap)	cai-c	cai_tc-c
Samples	13345	13345	3053	3053	2754	2754	656	656
Minimum	0.003	0.003	0.007	0.007	0.069	0.069	0.088	0.088
Maximum	14.192	1.782	3.623	1.795	2.902	1.782	2.573	1.795
Mean	0.506	0.499	0.525	0.521	0.506	0.499	0.526	0.522
Standard deviation	0.379	0.321	0.363	0.342	0.237	0.216	0.257	0.248
CV	0.749	0.643	0.691	0.656	0.468	0.432	0.489	0.475
Variance	0.144	0.103	0.132	0.117	0.056	0.047	0.066	0.061

An independent review (TRCI, 2013) notes that the capping levels used are similar to the 2012 values and they appear reasonable, but may be conservative. Cumulative frequency plots and sensitivity analysis indicate the capping levels may be closer to 2 g/t Au to 3 g/t Au. An independent review suggested that KBM review the cumulative frequency distributions to confirm the value of the outliers (TRCI, 2013).

14.7 Compositing and Statistical Analysis

A six metre downhole composite length was applied to all datasets. This length represents half of the planned mining bench height. Assays were sampled on 1 m lengths for Au, As, S, Cu, Mn, Pb, Zn, Hg, density, and BWI, so the minimum sample length resulting from compositing was one metre.

Assay and composite values inside the domain wireframes were back-coded to validate the domaining process and review the descriptive statistics by domain. Table 14-6 summarizes the basic statistics by domain.

Table 14-6: Assay and Composite Statistics by Domain

Domain / Item	Assay Length	Composite Length	Assay Grade	Composite Grade	Unit
B1					
Au			0.416	0.415	g/t
As	1.021	4.909	1211	1214	ppm
S			0.061	0.061	%
Count	18,062	3,753	17,286	3,653	Unit
B2					
Au			0.389	0.389	g/t
As	1.010	5.332	1236	1237	ppm
S			0.913	0.914	%
Count	48,265	9,142	47,356	9,056	Unit
B2 Calha					
Au			0.506	0.506	g/t
As	1.012	4.929	3324	3327	ppm
S			1.266	1.267	%
Count	13,521	2,775	13,345	2,754	Unit
B2_Calha_IDS					
Au			0.525	0.526	g/t
As	1.015	4.755	4562	4572	ppm
S			1.371	1.372	%
Count	3,096	661	3,053	656	Unit

14.8 Variography

Variographic analysis was carried out by domain, and further subdivided to honor the spatial and geologic differences encountered on either side of Rico Creek.

The nugget was interpreted for each domain using downhole variograms and then applied to subsequent spatial variography by domain. A 100 m lag distance was used for variography since the average distance between exploration drill holes is approximately 100 m. It should be noted that different lag distances generated poor variograms. All experimental variograms used 30° bandwidths, 30° directional increments and 0.5 (50%) tolerance to optimize orientations.

Experimental variography was subsequently used to calculate best-fit modeled variography. Data points with less than 100 sample pairs in down hole variography or less than 350 pairs for spatial variography were ignored. Similarly, calculations were weighted by pairs. Two spherical structures were used for both down hole and directional modelling. The variogram parameters used for resource estimation are listed in Table 14-7

Search lag distances and search angles were adjusted to best fit the data and maintain the highest resolution. For example, Au estimation generally used 60° search angles and 65 m lags for the first interval and 130 m for the second interval for the horizontal continuity. Domain 7 (B1) for As was better suited to 40 m lags for the first interval and 150 m lags for the second interval and 45° searches for the horizontal continuity. This is a reflection of the spatial distribution of the data. Sulphur in Domain 7 required a 200 m lag distance with 15° search angles. Variography with B2E (Domain 8) required 50 m and 45° searches.

Table 14-7: Variogram Interpretation

VDESC (A16)	VREFNUM	VANGLE1	VANGLE2	VANGLE3	VAXIS1	VAXIS2	VAXIS3	NUGGET	ST1	ST1PAR1	ST1PAR2	ST1PAR3	ST1PAR4	ST2	ST2PAR1	ST2PAR2	ST2PAR3	ST2PAR4
B1_AU_vp	7	0	0	-30	3	1	3	0.20	1	51.5	57	16.5	0.44	1	988	591.5	46.5	0.36
B2E_AU_vp	8	160	1	170	3	1	3	0.30	1	55	75	11	0.56	1	848.5	450.5	34.5	0.14
B2W_AU_vp	9	-90	20	-127	3	1	3	0.23	1	83	80	16	0.72	1	369.5	302	67	0.05
E10_AU_vp	10	0	0	-30	3	1	3	0.10	1	111.5	34	12.5	0.76	1	817.5	1012	22.5	0.14
W11_AU_vp	11	-100	10	-135	3	1	3	0.10	1	169.5	96.5	29.5	0.72	1	1628	461.5	65	0.18
E12_AU_vp	12	0	0	-30	3	1	3	0.25	1	103.5	73	17.5	0.65	1	484	301.5	22.5	0.1
W13_AU_vp	13	-90	20	-125	3	1	3	0.15	1	110.5	127	14.5	0.36	1	1386	1246	41	0.49
B1_AS_vp	7	0	0	-40	3	1	3	0.30	1	58	87.5	10.5	0.12	1	138	138	26.5	0.58
B2E_AS_vp	8	0	0	-50	3	1	3	0.25	1	79.5	19	15.5	0.55	1	305	228	52	0.2
B2W_AS_vp	9	-90	20	170	3	1	3	0.25	1	83	84	5.5	0.58	1	382.5	319	48.5	0.17
E10_AS_vp	10	0	0	-60	3	1	3	0.30	1	33	75	7	0.08	1	125.5	115	56	0.61
W11_AS_vp	11	-90	20	-110	3	1	3	0.20	1	104.5	79.5	14	0.73	1	204.5	151.5	49	0.07
E12_AS_vp	12	0	0	-60	3	1	3	0.15	1	69	91.5	11.5	0.81	1	152.5	265.5	36	0.04
W13_AS_vp	13	-90	20	-140	3	1	3	0.35	1	533	358	9	0.51	1	1032	661.5	18	0.14
B1_S_vp	7	0	0	-50	3	1	3	0.07	1	52	58.5	10	0.73	1	151	134	37	0.2
B2E_S_vp	8	0	0	-30	3	1	3	0.10	1	80.5	85.5	28	0.41	1	2268	1168.5	179.5	0.49
B2W_S_vp	9	-90	20	-127	3	1	3	0.10	1	163	121.5	21	0.57	1	914	861	57	0.33
E10_S_vp	10	150	10	180	3	1	3	0.10	1	221	42	39	0.62	1	1559	880.5	63.5	0.28
W11_S_vp	11	-140	15	0	3	1	3	0.20	1	193.5	241	40.5	0.53	1	890.5	885	108	0.27
E12_S_vp	12	0	0	-20	3	1	3	0.20	1	117.5	114.5	6	0.4	1	680	700	32.5	0.4
W13_S_vp	13	-90	20	-130	3	1	3	0.20	1	211.5	150	6.5	0.43	1	1518.5	602.5	190.5	0.37
B1_DENSITY_vp	7	0	0	-40	3	1	3	0.00	1	149	102	16	0.59	1	719	528.5	54.5	0.41
B2E_DENSITY_vp	8	0	0	-60	3	1	3	0.03	1	182.5	162	42.5	0.74	1	2059.5	996.5	72	0.23
B2W_DENSITY_vp	9	-30	10	20	3	1	3	0.20	1	134	65	86.5	0.64	1	599.5	335.5	99	0.16
SOFT_vp	7	0	0	-40	3	1	3	0.00	1	98	114.5	34	0.86	1	351	176	42.5	0.14
MEDIUM_vp	8	0	0	-100	3	1	3	0.00	1	177	121	26.5	0.73	1	2818	2631	43	0.27
HARD_vp	9	110	10	180	3	1	3	0.10	1	147.5	39.5	32	0.58	1	1198	661	188.5	0.32
I_WASTE_vp (AU)	9	0	0	-130	3	1	3	0.20	1	31	135.5	10	0.51	1	458	412.5	43.5	0.29
I_WASTE_vp (AS)	8	0	0	-30	3	1	3	0.18	1	402.5	365.5	4	0.5	1	1152.5	1186.5	80	0.32
I_WASTE_vp (S)	7	0	0	-120	3	1	3	0.08	1	163.5	233.5	26	0.27	1	1505	1372.5	114	0.65
QTZ_vp (AU)	9	0	0	-60	3	1	3	0.00	1	191	209.5	3	0.31	1	383.5	375	10	0.69
QTZ_vp (AS)	8	0	0	-50	3	1	3	0.30	1	102.5	10	3.5	0.51	1	239.5	245	8.5	0.19
QTZ_vp (S)	7	0	0	-20	3	1	3	0.25	1	531.5	160	8.5	0.13	1	786.5	543	20.5	0.62

14.9 Estimation Methodology

The 2013 block model was constructed using Datamine software (version 3.21.6885). The parent block size is 25 m (northing) by 25 m (easting) by 12 m (elevation). The block model origin is defined as the lowest south-western block edge and is 6,000 m E, 8,200 m N and 56 m elevation. The numbers of blocks in the model space are 256 (east), 196 (north), and 70 (elevation). Sub-cells are used to provide resolution between geological units. The smallest sub-cells used are 5 m (easting), 5 m (northing) and 3 m (elevation). Where possible, the sub-cells are grouped into larger units, with the maximum cell size defined by the parent cell.

Resource model estimations were completed for the B1 and B2 domains separately and then combined. A hard boundary was assumed between the two domains. Search parameters assumed a northeast-southwest orientation controlling the grade continuity. The final search orientation for resource estimation used an average of the results of the variography (e.g., density, BWI), or corresponded to the overall orientation of the mineralized structure (e.g., B1).

Separate Datamine estimation parameter files were used for each of the elements estimated (Au, As, S, Cu, Mn, Pb, Zn, Hg), as well as density and BWI. Three estimation types were created for each of these variables, including an Ordinary Kriged estimate (OK), an Inverse Distance Cubed estimate (ID3) and a Nearest Neighbor estimate (NN). Cu, Mn, Pb, Zn and Hg estimation used NN estimation instead of OK in the final model.

Datamine search parameter files were used to specify sample search strategies to satisfy estimation requirements. Search distances for the first pass are generally 160 m radius (long axis), 20 m radius (short axis), and 80 m radius (middle axis). This provides a search volume that, given the approximate 100 m drill grid, can accommodate at least 10 drill holes to satisfy the sample selection requirements for block estimation. Search orientations were determined by variography.

Table 14-8 lists the various block model attributes used for resource estimation.

Table 14-8: Variables used to build the 2013 MRMR block model

Variable	Description	Details
IJK	Cell location	Block sorting always completed on IJK
DOMAIN	Domain number	1: Soil. 2: AOX. 4: Fresh rock. 5: Waste. 6: Quartzite. 7: B1. 8: B2 east of Rico Creek (B2E). 9: B2 west of Rico Creek (B2W). 10:Calha E, 11: Calha W, 12: Calha IDS E, 13: Calha IDS W
MSTATUS	Mine status	1: Mined. 2: Not mined, within LOM pit shell, 3: Not mined, outside LOM pit shell. 0: Air
XC	Cell centre - x axis	Block centroid X direction
YC	Cell centre - y axis	Block centroid Y direction
ZC	Cell centre - z axis	Block centroid Z direction
XINC	Cell dimension - x axis	Volume = XINC*YINC*ZINC
YINC	Cell dimension - y axis	Volume = XINC*YINC*ZINC
ZINC	Cell dimension - z axis	Volume = XINC*YINC*ZINC
ROCK	Rock code	1-Soil, 2- AOX, 4 - fresh rock (east and west), 5 - Waste, 6 - quartzite, 7 - B1, 8 - B2 , 9 - Pyrrhotite, 10 - Lead and Zinc
ZONE	Field controlling estimation by Zone	Not used
AU	Estimated gold (g/t)	Ordinary Kriging Method used
DENSITY	Estimated density (g/cm3)	Ordinary Kriging Method used
BWI	Estimated Bond Work Index	Range: 3.3 to 24
CLASS	Resource Classification	1: Measured. 2: Indicated. 3:Inferred. 4:Potential
AS	Estimated arsenic (g/t)	Ordinary Kriging - reported with AS_NN and AS_ID in as_ore-m.dm
S	Estimated sulphur (%)	Ordinary Kriging - reported with S_NN and S_ID in s_ore-m.dm
Cu	Estimated Cooper ( )	Nearest Neighbour Method used
Mn	Estimated Manganese ( )	Nearest Neighbour Method used
Zn	Estimated Zinc ( )	Nearest Neighbour Method used
Pb	Estimated Lead ( )	Nearest Neighbour Method used
Hg	Estimated Mercury ( )	Nearest Neighbour Method used
MAG	Estimated Susceptibility Magnetic	Ordinary Kriging Method used
DM	Point Load Test axis 1	Not used
AX	Point Load Test axis 2	Not used
PCOST	Process cost	empty
MCOST	Mining Cost	empty
XMORIG	Block model origin (X)	Lowest southwestern corner of block model
YMORIG	Block model origin (Y)	Lowest southwestern corner of block model
ZMORIG	Block model origin (Z)	Lowest southwestern corner of block model
NX	Number of parent cell block (X)	Defines limit of model
NY	Number of parent cell block (Y)	Defines limit of model
NZ	Number of parent cell block (Z)	Defines limit of model
HARDNESS	Hardness	1: Soft, 2: Medium and 3: Hard

14.10 Cut-off Grade and Pit Constraint

Resources and reserves are reported above a minimum cut-off grade that represents the incremental cut-off. Mining costs are considered during pit optimization to value each block in the resource model when determining the optimum pit. The internal cut-off grade, which does not include mining costs, represents the cut-off grade once the ore reaches the pit rim and the decision must be made to process it or send it to the waste dump.

The 2013 Paracatu resource model was confined by the $1400/oz Au optimized pit shell. The majority of resource blocks within this pit shell are classified as Measured or Indicated using the criteria developed by Kinross.

Ore is processed in one of two plants: Plant I and Plant II. Descriptions of each plant are provided in Section 17 of this report. In summary, Plant I processes softer material at lower costs and Plant II processes harder material at higher costs. For determining the resource pit shell a cut-off grade of 0.139 g/t Au was calculated for material to be processed at Plant I using the following criteria:

Gold prices:	US$1,400/oz
Exchange Rate:	2 BRL to US$1
Mining Cost:	US$1.35/t ore
Process Cost:	US$2.73/t ore
Site Admin Cost:	US$0.69/t ore
Sales and Refining Cost:	US$9.13/oz Au
Royalty:	US$23.22/oz Au
Recovery:	78.57%

A cut-off grade 0.208 g/t Au was calculated for material to be processed at Plant II using the same criteria as above, with the exception of:

Mining Cost:	US$1.44/t ore
Process Cost:	US$5.02/t ore
Recovery:	78.7% average (formula based on grade)

14.11 Classification

The Paracatu deposit has been assigned resource classifications of Measured, Indicated and Inferred using the CIM Definition Standards for Mineral Resources and Mineral Reserves. For the Paracatu deposit, Kinross has defined the specific criteria required for resource classification as follows:

Measured: A Measured block has three drill holes with at least three assays within a distance of 100 m. A Measured block is assigned to the block model variable “CLASS” equal to 1.

Indicated: An Indicated block has at least one drill hole with at least three assays within a distance of 200 m. An Indicated block is assigned to the block model variable “CLASS” equal to 2.

Inferred: An Inferred block has at least one drill hole with at least three samples within a distance of 400 m. An Inferred block is assigned to the block model variable “CLASS” equal to 3.

Unclassified: All remaining blocks within the model extents without supporting drill hole data within 400 m are assigned the variable “CLASS” equal to 4. These blocks are not classified as mineral resources.

The criteria detailed above were used to classify blocks in Datamine using a NN estimation method (Figure 14-4).

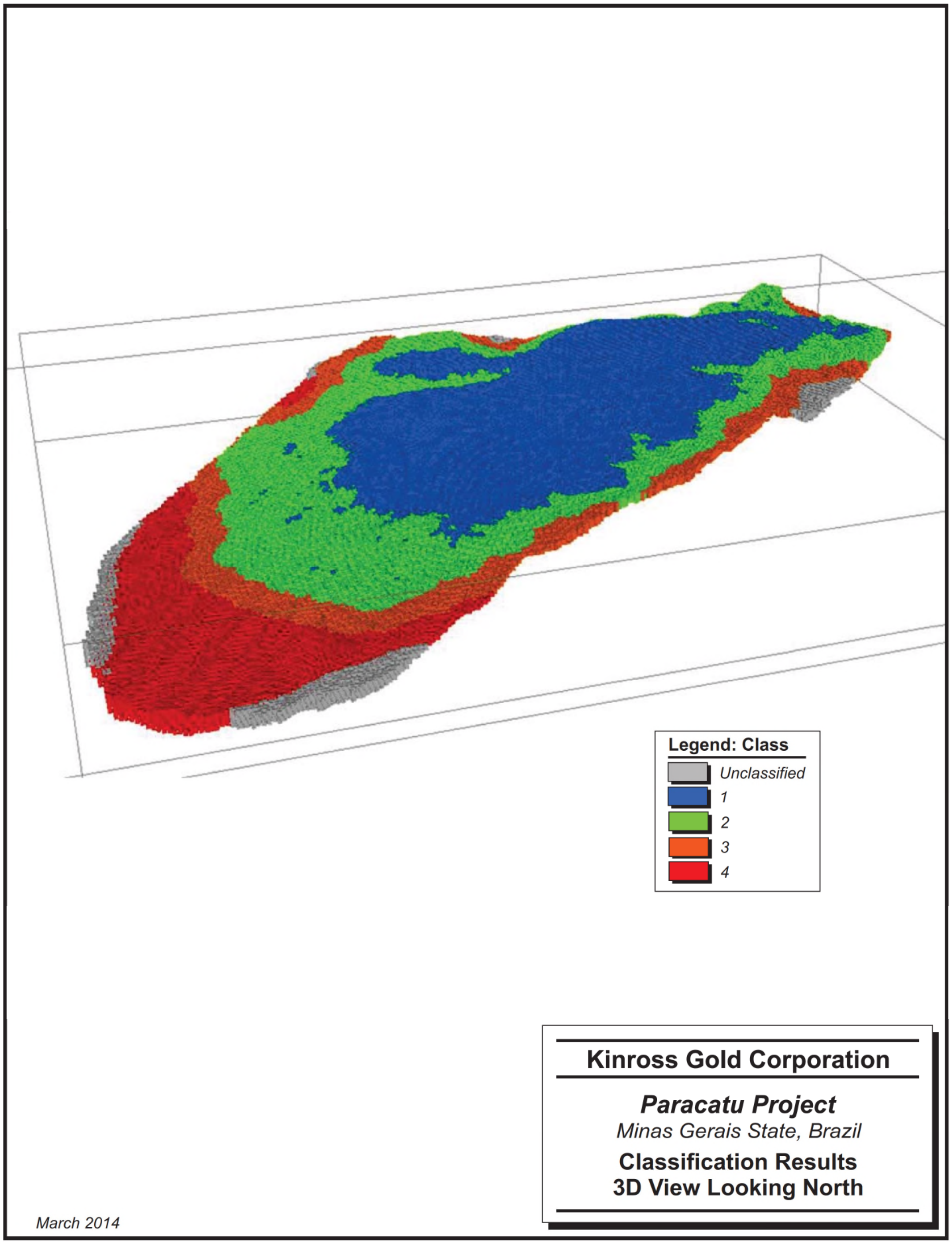

Figure 14-4: Classification Results – 3D View Looking North. 1- measured (blue), 2- indicated (green), 3- inferred (orange) and 4- potential (red). Blocks without classification are grey.

14.12 Mineral Resource Validation

The block model was validated using a number of techniques to confirm the assignment of appropriate variables and grade estimation. These included visual confirmation of block grades with respect to assay grades, statistical comparison of blocks and composite data, and comparisons between the ID3 and NN models. Geostatistics were used to compare global variations in block means, standard deviations, variance and covariance. Swath plots were used to show local differences in tonnes and grade between models with respect to composited samples used for the estimation. No significant discrepancies were identified.

14.13 Reconciliation

Kinross corporate guidelines state that production reconciliation should be within approximately 10% of the long range model predictions. Kinross reviewed the production reconciliation for the last three years.

Reconciliations compare the tonnage, grade and contained metal data from one data source to another, such as the resource model to production records. Data sources are compared by means of ratios or factors.

The F1 factor measures the accuracy of orebody knowledge in the ore reserves to the demarcation of ore and waste by ore control practices (short-term model). The F1 factor may be used to check and calibrate the selectivity of mineral resource models and/or planned dilution assumed in the transfer from mineral resources to ore reserves. The F2 factor enables a check on unplanned dilution entering the ore stream between ore control and the mill. Using the F1 and F2 factors, it is possible to calculate a monetary value on improvements in the accuracy of orebody knowledge, selectivity and the effects of dilution and ore loss. The F3 factor is the product of the F1 and F2 factors and therefore assesses the ability to recover the tonnage, grade and metal content as estimated in the resource model. The F3 factor provides a good indication of the overall reliability of the resource model.

Over the three year period from 2011 to 2013, the F3 tonnage, gold grade, and contained gold factors were 99%, 91%, and 90%, respectively. This indicates that over the last three years, the resource model was within 1% of the plant on tonnage but was overestimating the gold grade by 9% relative to the plant's back-calculated head grade, resulting in an overestimation of the contained ounces by 10%.

Kinross is not aware of any environmental, permitting, legal, title, location, socio-economic, marketing, political or other modifying factors which could materially affect the open pit Mineral Resource estimate.

15. MINERAL RESERVE ESTIMATE

The Proven and Probable Mineral Reserves as of December 31, 2013 are estimated to be 763.7 million tonnes at 0.424 g/t Au and containing 10.4 million ounces of gold, as presented in Table 15-1.

One major change between EOY2013 and EOY2012 is the inclusion of sustaining capital costs as operating costs for the purposes of pit shell determination. Some examples of these costs are: waste pre-stripping, mine fleet replacement costs, and tailings dam construction, for which additional investment is required in order for the operation to continue or for the reserve base to be enlarged. This change has a large impact on the overall reserve estimation since the Lerchs-Grossman (LG) calculation takes these added costs into consideration when sizing the overall economic pit. The end result is a smaller overall pit with a higher contained gold grade and improved economics.

Table 15-1: Proven and Probable Mineral Reserves – December 31, 2013

	Tonnes (000s)	Gold (g/t)	Gold Ounces (000s)
Proven	556,292	0.41	7,371
Probable	207,416	0.45	3,030
Total	**763,708**	**0.42**	**10,401**
Stockpile	2,915	0.24	23

Notes:
1. Mineral Reserves estimated according to CIM Definitions.
2. Mineral Reserves estimated at $1,200/oz Au.
3. Proven Reserve includes Stockpile.
4. Mineral Reserves are estimated at gold cut-off grades that vary by material type from approximately 0.163 g/t Au to 0.244 g/t Au.

Mineral reserve only considers Measured and Indicated ore in the final pit design. Inferred ore that has to be mined is considered as waste in the reserves. The final pit design is based the economic LG shell generated using CAE's NPV Scheduler (NPVS) software package. The final pit design is completed in Maptek's Vulcan® software using the NPVS LG optimized pit shell as a guide. These are well-recognized software packages and are commonly used for open pit mine optimization. Cut-off grades are based on ore types B1 and B2 which are categorized based on Bond Work Index values.

Mineral reserves only consider Measured and Indicated ore in the pit optimization. Inferred ore that has to be mined is considered as waste in the reserves declarations. For the resource calculations, Inferred ore is considered and a 1,400US$/oz gold price is used in the pit optimizations. This has an impact on the cut-off grade used in optimizations.

15.1 Dilution and Extraction

Due to the geometry and extension of the ore body mining recovery and dilution were considered to be 100% and 0% respectively. The Paracatu deposit is a massive ore body with very few lenses of internal waste.

15.2 Recovery

The process recovery was calculated in the block model and used different formulas for Plant I and Plant II. The formulas used are as follows:

Plant I Overall Recovery = *0.81 * 0.97 = 78.6%*

Where 0.81 is plant recovery and 0.97 is the hydrometallurgical recovery.

Plant II Overall Recovery =

$$\frac{[(Grade - (Grade \times 0.0359 + 0.059) \times 0.96]}{(Grade \times 0.985 + 0.032)} + 0.04 = 78.7\%$$

Where *(Grade x 0.0359 + 0.059)* = tailings grade of the concentration plant.

(Grade – (Grade x 0.0359 + 0.059)/ Grade = recovery of the concentration plant without a correction factor

0.985 = correction factor

0.032 = 3.2% gain of the desulfurization

0.96 = hydrometallurgical recovery

15.3 Mining Costs

Mining costs are based on the budget developed in 2012 for 2013 through 2018.

15.4 Process Costs

The process costs were calculated only for ore blocks above the cut-off grade as per Kinross Corporate guidance. The costs were extracted from 2012 Budget corrected by the new Exchange Rate. The process cost also includes the costs for limestone, contracted services and power. The process costs are calculated as follows:

Process Cost (Plant I) = 1.101+1.108* (BWI^0.23)

Process Cost (Plant II) = 2.067+0.656/exp((-0.126*BWI)*95%

15.5 Selling Costs

Royalty and Sales Taxes represent 1.5 % of the net revenue. Refining costs are based on budget figures developed in 2012 for 2013 through 2018.

15.6 Cut-Off Grade

A cut-off grade (COG) was calculated for both plants as shown in Table 15-2.

Table 15-2: Mineral Reserve Cut-Off Grade Calculation

Area	Units	Plant I	Plant II
Mining Cost	US$/t	1.35	1.44
Process Cost	US$/t	2.73	5.02
Site Admin Cost	US$/t	0.69	0.69
Total	**US$/t**	**4.78**	**7.15**
Refining/Sales	US$/oz	9.13	9.13
Royalty	US$/oz	23.22	23.22
Total	**US$/oz**	**32.35**	**32.35**
Total	**US$/g**	**1.04**	**1.04**
Gold Price	US$/oz	1,200	1,200
Gold Price	US$/g	38.58	38.58
Recovery (at COG)	%	78.6%	78.7%
Effective Revenue	US$/g	30.31	30.38
Less per gram Costs	US$/g	-1.04	-1.04
Realized Revenue	US$/g	29.27	29.34
Costs to Produce	US$/t	4.78	7.15
Reserve COG	**g/t**	**0.163**	**0.244**

16. MINING METHODS

16.1 Mining Operations

The Paracatu operation is composed of an open pit mine, two process plants, two tailings facilities, and related surface infrastructure and support buildings. Mine production from 1987 to 2013 is shown in Table 6-2.

At Paracatu, ore hardness increases with depth and, as a result, modelling the hardness of the Paracatu is important for costing and process throughput parameters. KBM modeled ore hardness based on Bond Work Index (BWI) analyses of diamond drill samples. KBM estimated that blasting of the Paracatu ore would be necessary for blocks with a BWI greater than 8.5 kWh/t. BWI for ore is shown in the production schedule in Table 16-4.

Expansion Project III (2006) increased the mill throughput to 61 Mt/a through the installation of Plant II. This initiative was undertaken to handle harder ore. In September 2010, Kinross installed a third ball mill in Plant II. To further augment processing and grinding capacity, the Company approved the addition of a fourth ball mill in 2010.

In 2011, Kinross installed a desulfurization circuit, received permit approval for the new Eustáquio tailings facility and purchased a second Bucyrus BH495 shovel.

Kinross purchased two 793 CAT haul trucks to add to the existing fleet of eleven 793 CAT haul trucks and started operating the Eustáquio tailings facility in 2012. Start-up of the fourth ball mill occurred in the third quarter of 2012.

16.2 Mine Design

The design process for the open pit mine at Paracatu began by completing a series of pit optimizations in order to create a pit shell that would form the basis, or template, for the open pit design.

Pit optimization for the Paracatu open pit was completed using CAE NPV Scheduler (NPVS), software that uses the Lerchs-Grossman (LG) algorithm. NPVS software produces computer generated phases that direct mining to the parts of the deposit that have higher value. Phase designs based on the NPVS suggested phase sequence are developed using Maptek's Vulcan® software.

Table 16-1 shows the grade and tonnages at a range of cut-off values. The Vulcan® model represents the entire block model. The exported model shows the actual tonnage and grade contained within the design pit.

Table 16-1: Grade and Tonnage Values for Full Model and Pit Design Model

Cutoff	Full Model (Vulcan)			Exported Model (NPVS)		
Grade (Au g/t)	Tonnes (000s)	Grade (Au g/t)	Au (Ounces)	Tonnes (000s)	Grade (Au g/t)	Au (Ounces)
0.1	3,097,927	0.337	33,565	962,656	0.406	12,565
0.2	2,780,368	0.356	31,822	941,046	0.411	12,435
0.3	1,649,097	0.43	22,798	772,193	0.445	11,048
0.4	882,606	0.502	14,245	465,715	0.504	7,546
0.5	334,879	0.596	6,417	181,642	0.596	3,481
0.6	113,956	0.699	2,561	59,690	0.708	1,359
0.7	41,236	0.805	1,067	24,574	0.805	636
0.8	16,811	0.892	482	9,503	0.902	276
0.9	4,975	1.005	161	2,706	1.048	91

Open pit design parameters included the operating costs, process recovery, metal price and pit slope angles. The optimization parameters used for this design exercise are presented in Table 16-2 and represent the Base Case. Parameters for EOY2013 are showed in comparison with EOY2012 to highlight the differences between years.

Table 16-2: Base Case Open Pit Design Parameters

Assumptions and Inputs	Unit	EOY2012	EOY2013	% Change
Economic Assumptions				
Au Reserve Price	US$/oz	1,200	1,200	0%
Brazilian Reals to US$	BRL	2.10	2.27	8%
Production Assumptions				
Production Rate / Day - Plant I	t/day	50,456	49,315	-2%
Production Rate / Day - Plant II	t/day	98,324	104,110	6%
Total Material- Mine	t/day	199,893	171,958	-14%
Mining Dilution	%	0%	0%	0%
Mining Recovery	%	100%	100%	0%
Recovery Assumptions - Plant I	%	78.6%	78.6%	0%
Recovery Assumptions - Plant II	%	80.4%	78.7%	-2%
Mining Cost Assumptions				
Base Mining Cost - B1 Ore	US$/t	1.556	1.351	-13%
Base Mining Cost - B2 Ore	US$/t	1.499	1.436	-4%
Increment/Bench Below Reference Bench	US$/t	0.015	0.026	75%
Estimated Average - Ore	US$/t	1.522	1.422	-6%
Estimated Average - Waste	US$/t	0.603	1.436	102%
Estimated Average Overall	US$/t	1.155	1.426	21%

Assumptions and Inputs	Unit	EOY2012	EOY2013	% Change
Processing Operating Cost Assumptions				
B1 Ore - 'Plant I'				
Processing Costs	US$/t	3.51	2.73	-22%
B2 Ore - 'Plant I'				
Processing Costs	US$/t	5.97	-	-100%
B2 Ore - 'Plant II'				
Processing Costs	US$/t	4.67	5.12	10%
Other Operating Cost Assumptions				
Site Admin Costs	US$/t	0.83	0.69	-16%
Refining Costs	US$/oz	4.07	9.13	124%
Royalty Costs @ $1,200 Au	US$/oz	17.89	23.22	30%
Production Tax	US$/t	0.02	0.04	57%
Sustaining Capital Cost Assumptions				
Mining Cost (i.e. Fleet Replacement)	US$/t		0.28	New
Process Cost (i.e. Tailings Pond)	US$/t		1.33	New
Total Sustaining Capital Cost	US$/t		0.72	New
Geotechnical*				
Slope Sector Name/Number				
1	Degrees	26.6°	26.6°	0%
2	Degrees	38.8°	38.8°	0%
3	Degrees	Profile 3	Profile 3	0%

(* See Geotechnical Section for more detail)

16.3 Differences Between EOY2013 and EOY2012 Reserve Estimates

It should be noted that one major change between EOY2013 and EOY2012 is the inclusion of sustaining capital costs as operating costs for the purposes of pit shell calculation. Some examples of these costs, which were traditionally considered as capital costs, but are now included as operating costs, are:

- Waste pre-stripping,
- Mine fleet replacement costs, and
- Tailings dam construction.

This change has a large impact on the overall reserves calculation since the LG cone calculation takes these added costs into consideration when sizing the overall economic pit. The end result is a smaller overall pit with higher grade.

Process recoveries were estimated during the modelling process with a unique process recovery estimated for each 50 m by 50 m by 12 m block in the model. The process costs were calculated within the block model based on the BWI that was also estimated during resource modelling.

Haul roads and in-pit ramps were designed to be 40 m wide with a gradient of 10%. The selected width is approximately three times the width of a CAT 793 haul truck (~7.4 m) with a safety allowance of an additional 5 m. The total running surface is 28 m wide with berms on either side of the haul road. The total allowance for the berms on each side of the running surface is 6 m, comprising a 3 m berm and a 3 m offset from both pit wall and edge of slope for a total measured width of 40 metres. A typical road cross-section is shown in Figure 16-1.

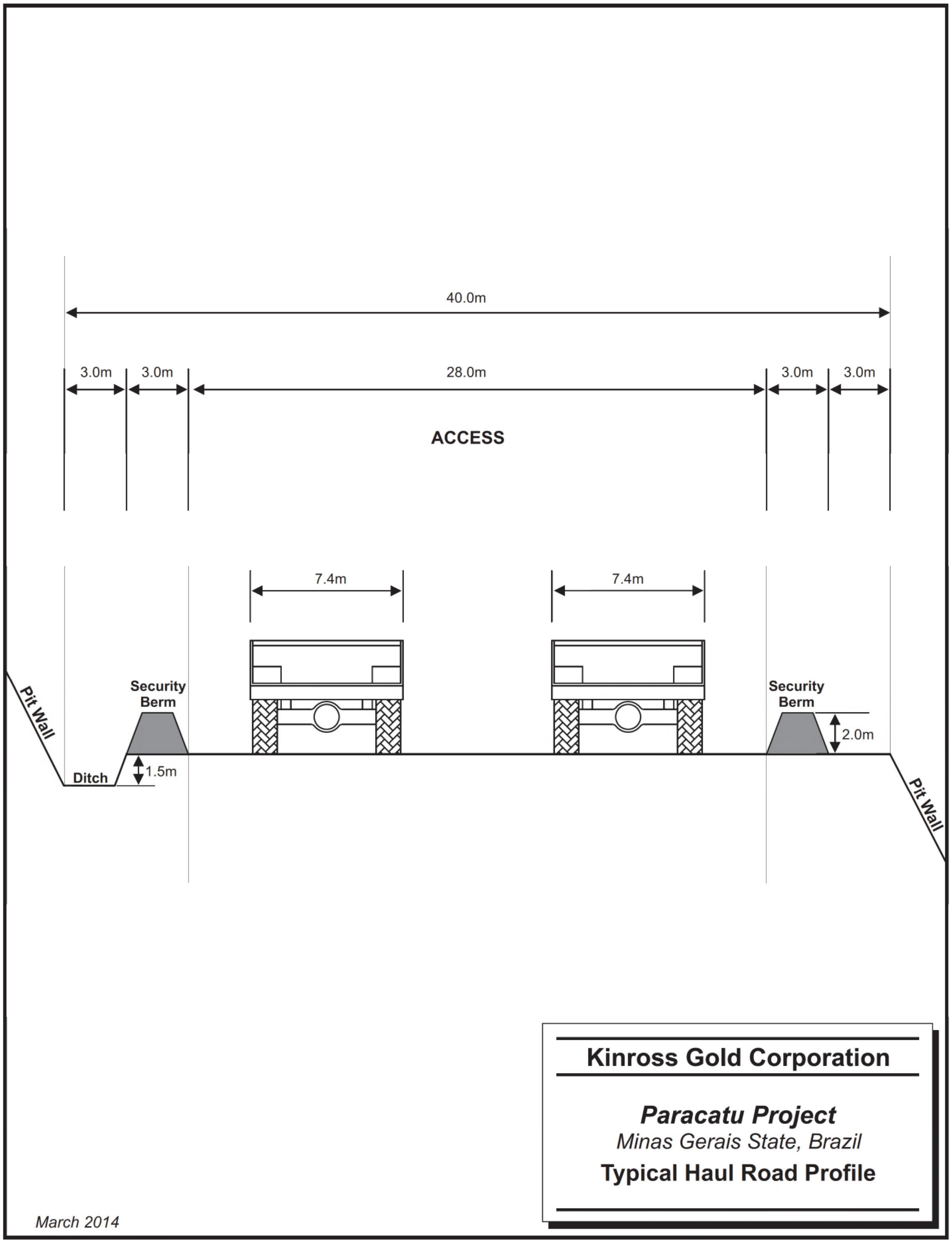

Figure 16-1: Typical Haul Road Profile

16.4 Geotechnical Considerations

The pit slope angles used in the pit optimization are based on an independent geotechnical review (Golder, 2006). These pit slope angles are presented in Table 16-3. For reference, Figure 16-2 gives a visual indication of the different slope regions in the Paracatu mine pit and their corresponding azimuths.

Table 16-3: Pit Slope Angles Used in Open Pit Mine Optimization

Geotechnical Zone	Rock Mass	Inter-ramp Angle (^{o})	Face Angle (^{o})	Berm (m)	Ramp Width (m)
Slope Region 1	Soil	26.6	45	6	40
Slope Region 2	Saprolite	38.8	60	8	
Slope Region 3	North Sulphide Rock	49	60	7	
	West Sulphide Rock	56.7	70	7	
	South Sulphide Rock	52.8	65	7	

Azimuth	Overall Angle (^{o})		
	No Ramp	One Ramp	Two Ramps
315 to 060	49.0	44.2	40.1
220 to 315	56.7	51.9	47.5
060 to 220	52.8	47.6	43.0

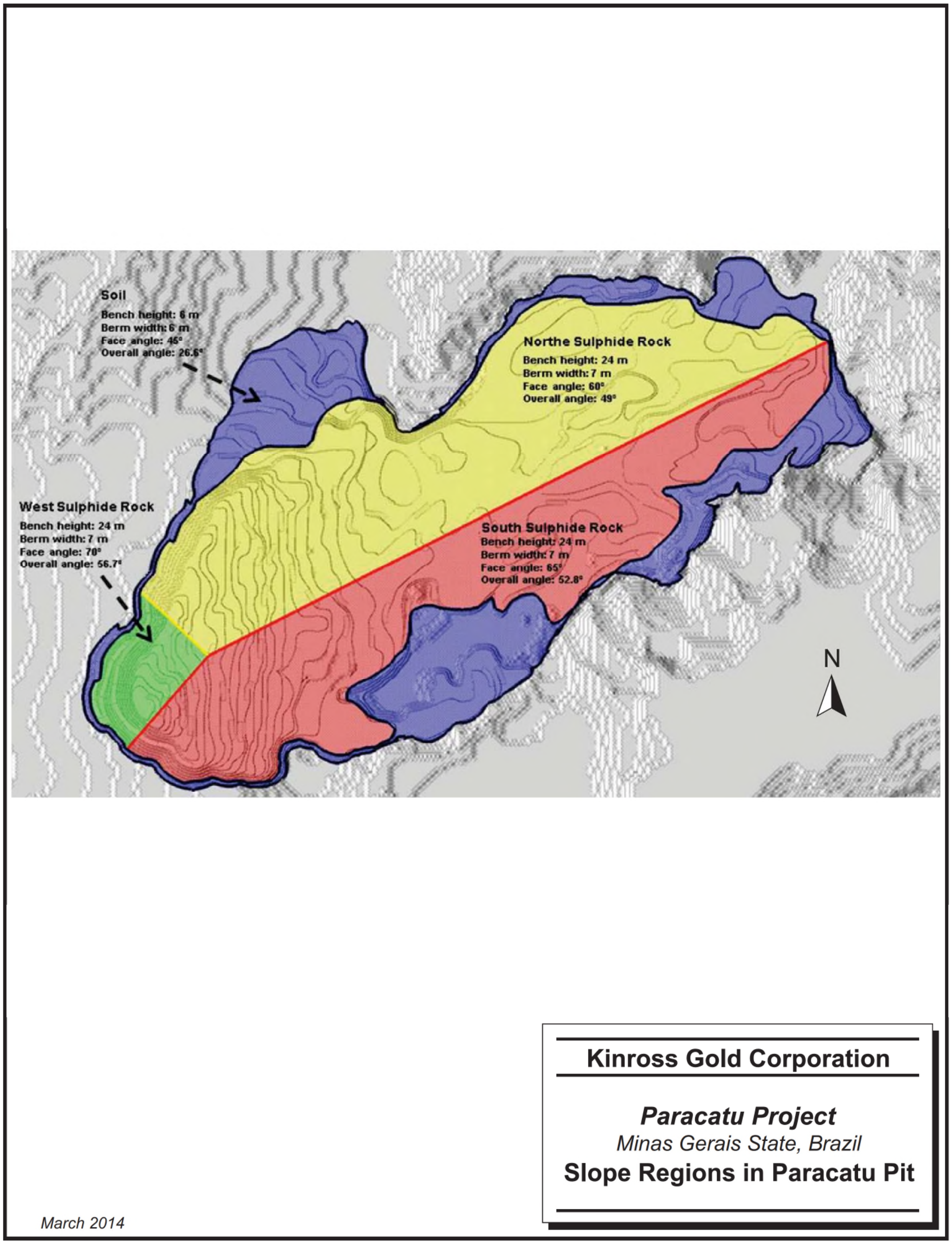

Figure 16-2: Slope Regions in Paracatu Pit

16.5 Production Schedule

The mine operates 24 hours a day, 365 days a year. Production rates vary by material type. Currently, the B1 ore is scheduled at a throughput rate of 17.5 Mt/a, and the B2 ore is scheduled at a rate of 38.0 Mt/a. The life-of-mine (LOM) schedule is presented in Table 16-4.

The LOM schedule in Table 16-4 shows that the final pit is depleted in 2030. The plant would continue to be fed from stockpiles until year 2032. The total mine life, including stockpile reclaim is 19 years as of the end of 2013. Table 16-5 presents the processing schedule for the Project.

Table 16-4: Paracatu LOM Mining Schedule

Year	B1 Ore (000 t)	B2 Ore (000 t)	Stockpiled (000 t)	Waste (000 t)	Mined (000 t)	Rehandled (000 t)	Moved (000 t)
2014	17,500	37,998	4,788	7,116	67,402	4,440	71,842
2015	17,500	37,995	4,548	8,283	68,326	4,440	72,766
2016	17,288	38,010	8,699	6,718	67,191	7,666	74,857
2017	17,364	37,988	4,858	6,061	66,270	4,428	70,698
2018	-	38,003	9,873	8,889	56,765	3,040	59,805
2019	-	37,989	12,091	7,903	57,983	3,039	61,022
2020	-	37,988	18,826	10,999	60,156	10,083	70,239
2021	-	38,002	15,524	12,146	62,167	6,264	68,431
2022	-	38,001	11,510	20,715	70,226	3,040	73,266
2023	-	37,979	10,444	25,786	74,208	3,038	77,247
2024	-	38,000	94	37,572	73,028	5,467	78,495
2025	-	38,000	-	41,646	75,843	6,539	82,382
2026	-	37,994	692	35,461	74,147	3,040	77,187
2027	-	38,000	179	43,047	77,275	6,675	83,950
2028	-	38,000	-	37,648	71,851	6,533	78,384
2029	-	38,000	128	25,262	59,467	6,649	66,116
2030	-	38,000	1,559	505	17,158	24,114	41,271
2031	-	38,000	-	-	-	38,000	38,000
2032	-	10,109	-	-	-	10,109	10,109
Total	**69,652**	**694,056**	**103,813**	**335,757**	**1,099,463**	**156,604**	**1,256,067**

Table 16-5: Paracatu Life of Mine Processing Schedule

Plant I					
Year	Tonnes (000s)	Recovery (%)	Grade (g/t)	BWI (kWh/t)	Au Produced (ounces)
2014	17,500	78.57	0.34	5.25	150,346
2015	17,500	78.57	0.31	5.24	135,879
2016	17,288	78.57	0.32	5.52	140,001
2017	17,364	78.57	0.33	5.41	146,212

Plant II					
Year	Tonnes (000s)	Recovery (%)	Grade (g/t)	BWI (kWh/t)	Au Produced (ounces)
2014	37,998	78.75	0.43	11.23	412,772
2015	37,995	78.38	0.42	12.23	398,836
2016	38,010	76.77	0.38	11.87	360,622
2017	37,988	77.18	0.39	12.36	369,067
2018	38,003	76.45	0.38	12.26	351,346
2019	37,989	79.44	0.45	12.25	440,376
2020	37,988	76.80	0.39	12.24	368,022
2021	38,002	78.12	0.42	11.48	405,150
2022	38,001	78.66	0.45	13.37	428,643
2023	37,979	77.86	0.44	13.56	422,521
2024	38,000	78.46	0.47	13.89	451,682
2025	38,000	78.96	0.46	14.55	440,771
2026	37,994	78.27	0.43	11.89	412,237
2027	38,000	78.99	0.49	14.66	474,819
2028	38,000	78.38	0.53	14.78	503,574
2029	38,000	79.19	0.50	14.61	480,823
2030	38,000	77.04	0.44	12.42	415,316
2031	38,000	75.00	0.35	10.39	321,134
2032	10,109	75.00	0.35	10.39	85,433

Total Combined					
Year	Tonnes (000s)	Recovery (%)	Grade (g/t)	BWI (kWh/t)	Au Produced (ounces)
2014	55,498	78.69	0.40	11.23	563,118
2015	55,495	78.44	0.38	12.23	534,716
2016	55,298	77.33	0.36	11.87	500,623
2017	55,352	77.62	0.37	12.36	515,279

Total Combined					
Year	**Tonnes (000s)**	**Recovery (%)**	**Grade (g/t)**	**BWI (kWh/t)**	**Au Produced (ounces)**
2018	38,003	76.45	0.38	12.26	351,346
2019	37,989	79.44	0.45	12.25	440,376
2020	37,988	76.80	0.39	12.24	368,022
2021	38,002	78.12	0.42	11.48	405,150
2022	38,001	78.66	0.45	13.37	428,643
2023	37,979	77.86	0.44	13.56	422,521
2024	38,000	78.46	0.47	13.89	451,682
2025	38,000	78.96	0.46	14.55	440,771
2026	37,994	78.27	0.43	11.89	412,237
2027	38,000	78.99	0.49	14.66	474,819
2028	38,000	78.38	0.53	14.78	503,574
2029	38,000	79.19	0.50	14.61	480,823
2030	38,000	77.04	0.44	12.42	415,316
2031	38,000	75.00	0.35	10.39	321,134
2032	10,109	75.00	0.35	10.39	85,433
Total	**763,708**	**77.95**	**0.42**	**12.67**	**8,115,583**

16.6 Waste Rock

There are three main waste dumps in the LOM plan. The first waste dump is an in-pit waste dump that is used for the initial 5 years of the plan. A backfill analysis was performed on the waste dump to evaluate the quantity of ore that would be sterilized from the in-pit dumping. The tonnes sterilized are minimal because the location of the waste dump is in a region of the deposit that is flat, shallow and poorly mineralized.

The second waste dump is to the northwest of the ultimate pit. The physical constraints for the dump include the ultimate tails limit, mine access road, major highway and $2,000/oz Au pit limit. This dump serves to reduce the haulage requirement in the middle years of the LOM plan (years six to ten) when the waste stripping rate doubles.

The third waste dump is an in-pit waste dump situated in shallow, flat portions of the deposit that will have been depleted of ore near the end of the mine life.

A semi-permanent stockpile location was developed in this LOM plan to accommodate the cut-off and blending strategy created in 2013. This stockpile is typically one large stockpile used for low grade (and/or high BWI) material over time. The stockpile location is in the pit, so the material must eventually be moved in order to gain access to the ore below.

16.7 Mine Equipment

The production fleet consists of ten Caterpillar 777 haul trucks and thirteen Caterpillar 793 haul trucks. In 2014, the Caterpillar 793 fleet will expand to fifteen haul trucks.

The loading fleet consists of three Caterpillar 992 loaders, three Caterpillar 994 loaders and two Bucyrus 495HD shovels.

The diesel-powered drill fleet is comprised of two Atlas Copco DM45 rigs, two Atlas Copco DM50 rigs, one Bucyrus 35R and one Atlas Copco ROC D7. Shovel and drill fleets will be replaced with comparable units upon retirement.

A summary of the mine equipment currently in use is displayed in Table 16-6.

Table 16-6: Paracatu Mine Equipment Schedule

Equipment Type	Quantity	Model
Haul Truck	8	Caterpillar 777C
Haul Truck	2	Caterpillar 777D
Haul Truck	13	Caterpillar 793D
Loader	1	Caterpillar 966H
Loader	2	Caterpillar 992G
Loader	1	Caterpillar 992K
Loader	3	Caterpillar 994F
Shovel	2	Bucyrus 495HD
Blasthole Drill	1	Atlas Copco Roc-D7
Blasthole Drill	1	Bucyrus 35HR
Blasthole Drill	1	Ingersoll-Rand DM45
Blasthole Drill	2	Ingersoll-Rand DM50
Dozer	2	Caterpillar D10R
Dozer	2	Caterpillar D10T
Dozer	1	Caterpillar D11R
Dozer	1	Caterpillar D11T

16.8 Personnel Requirements

Paracatu operates 24 hours per day, 365 days per year using two rolling schedules that rotate personnel through the day shift (6:00 am to 3:00 pm), afternoon shift (3:00

pm to 12:00 am) and evening shift (12:00 am to 6:00 am). In the first schedule, crews work six days on the day shift, followed by one day of rest, three days on the afternoon shift, three days on the night shift and ending with three days of rest. The second schedule starts with three days on the afternoon shift, followed by three days on the night shift, and two days of rest, at which point the crew returns to the first schedule.

The administrative team works Monday through Friday from 7:20 am to 5:20 pm and observes national holidays (approximately 12 per calendar year).

Personnel requirements for the LOM schedule is presented in Table 16-7 below.

Table 16-7: Operations Personnel Requirements

	Years		
Area	**2014-2018**	**2019-2023**	**2024-2030**
Mining	483	436	413
Processing	647	455	399
Site Admin	159	147	144
Total	**1,289**	**1,038**	**956**

17. RECOVERY METHODS

17.1 Process Description

Paracatu has two mineral processing plants known as Plant I and Plant II. Plant I treats the softer near-surface B1 ore at a design throughput of 20 Mt/a, while Plant II treats the harder B2 ore at a design throughput of 41 Mt/a.

17.2 Paracatu Expansion III Project

Plant II was developed as part of the Paracatu Expansion III Project to counter the gradually increasing work index of the deposit. The original Plant I circuit was not designed for hard ore, and capacity and operating costs would have been significantly affected unless additional grinding capacity was installed.

The Plant Capacity Scoping Study completed in June 2005 recommended that production be increased from 18 Mt/a to 50 Mt/a by installing a new plant with a capacity of 32 Mt/a.

As part of the engineering study SNC/Minerconsult (SNC) investigated the possibility of upgrading the 50 Mt/a design to 61 Mt/a. The study demonstrated that, with upgrades to some of the equipment, the bottlenecks restricting production to 50 Mt/a could be removed. The new design had Plant I operating at a throughput of 20 Mt/a, and Plant II at a throughput of 41 Mt/a, for a total of 61 Mt/a.

At the end of 2008, the installed Plant II consisted of one in-pit crusher (MMD toothed roll type), a 1.8 km conveyor to a covered stockpile area, one 20 MW semi-autogenous grinding (SAG) mill and two 13 MW ball mills.

17.3 Further Plant II Expansion Work

After the expansion, Plant II performed under target in both throughput and recovery. In November 2009, Kinross announced its intention to increase grinding capacity by commissioning a new ball mill. Installation of the 15 MW third ball mill was concluded in June 2011. Flash flotation cells for ball mills 1 and 2 were also installed during this period.

In early 2010, the Kinross Corporate Project Development Group, along with SNC and Paracatu site personnel, performed an optimization study on Paracatu. The study concluded that with the expected ore work indexes beginning in 2012, a fourth ball mill would be needed to maintain the 61 Mt/a throughput rate. The first ore was fed into the fourth ball mill on August 28th, 2012 and full capacity was reached on September 28th,

2012. Two flash flotation cells for ball mills 3 and 4 were also installed as part of this expansion.

17.4 Desulphurization

The implementation of Expansion Project III necessitated a new tailings facility. Construction of the new Eustáquio tailings facility began in 2009 and tailings have been discharged into the new basin since 2012.

The project was designed on the basis of tailings to the new Eustáquio Dam containing less than 0.4% sulphur. Since the tailings contain an average concentration of 1.0% sulphur, desulphurization was needed to meet target values. The desulphurization system makes use of the existing flotation equipment, with the addition of a new flotation building to house gold cleaner flotation cells and sulphide re-cleaner flotation cells. Operation of the desulphurization flotation plant began in 2012.

17.5 Plant I

In Plant I, ore is crushed through two stages and ground in ball mills prior to gold recovery by jigs and flotation. The concentrate is treated by gravimetric methods first and the coarser gold is recovered. The flotation and gravity concentrate is then leached with cyanide in a carbon-in-leach (CIL) circuit, followed by carbon elution and electrowinning to recover gold which is then smelted to form gold bars.

Plant I has operated continuously since 1987 and has had expansion upgrades in 1997 and 1999. In 2007, the plant processed 17.2 Mt/a and achieved an average gold recovery of 78.2%. Plant I has a nominal capacity of 20 Mt/a when processing ore with a Bond Work Index (BWI) of less than 8 kWh/t. Figure 17-1 is a simplified flow sheet of Plant I.

17.6 Plant II

Plant II initiated production in September 2008, and achieved commercial production levels in December 2008. Figure 17-2 is a simplified flow sheet of Plant II. Currently, Plant II consists of an in-pit MMD crusher, a 1.8 km conveyor to a covered stockpile area, an 11.6 m diameter SAG mill, and four ball mills. The ore recovery process uses gravity flotation to produce concentrate which is leached with cyanide in a CIL circuit, followed by carbon elution, electrowinning and smelting into gold bars.

The plant has a nominal capacity of 41 Mt/a when processing ore with a BWI below 8.7 kWh/t. Tonnage throughput decreases as the BWI increases.

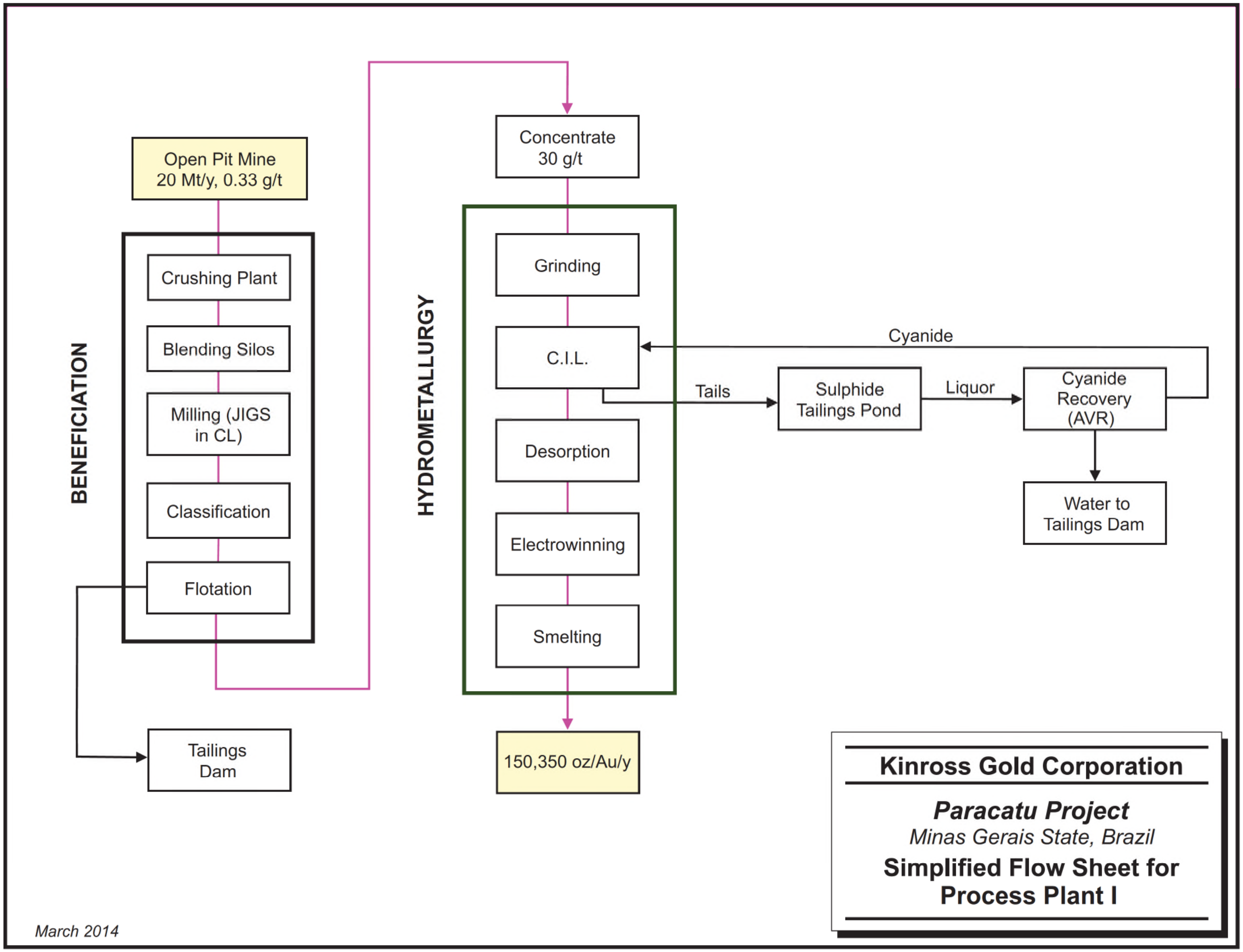

Figure 17-1: Simplified Flow Sheet for Plant I

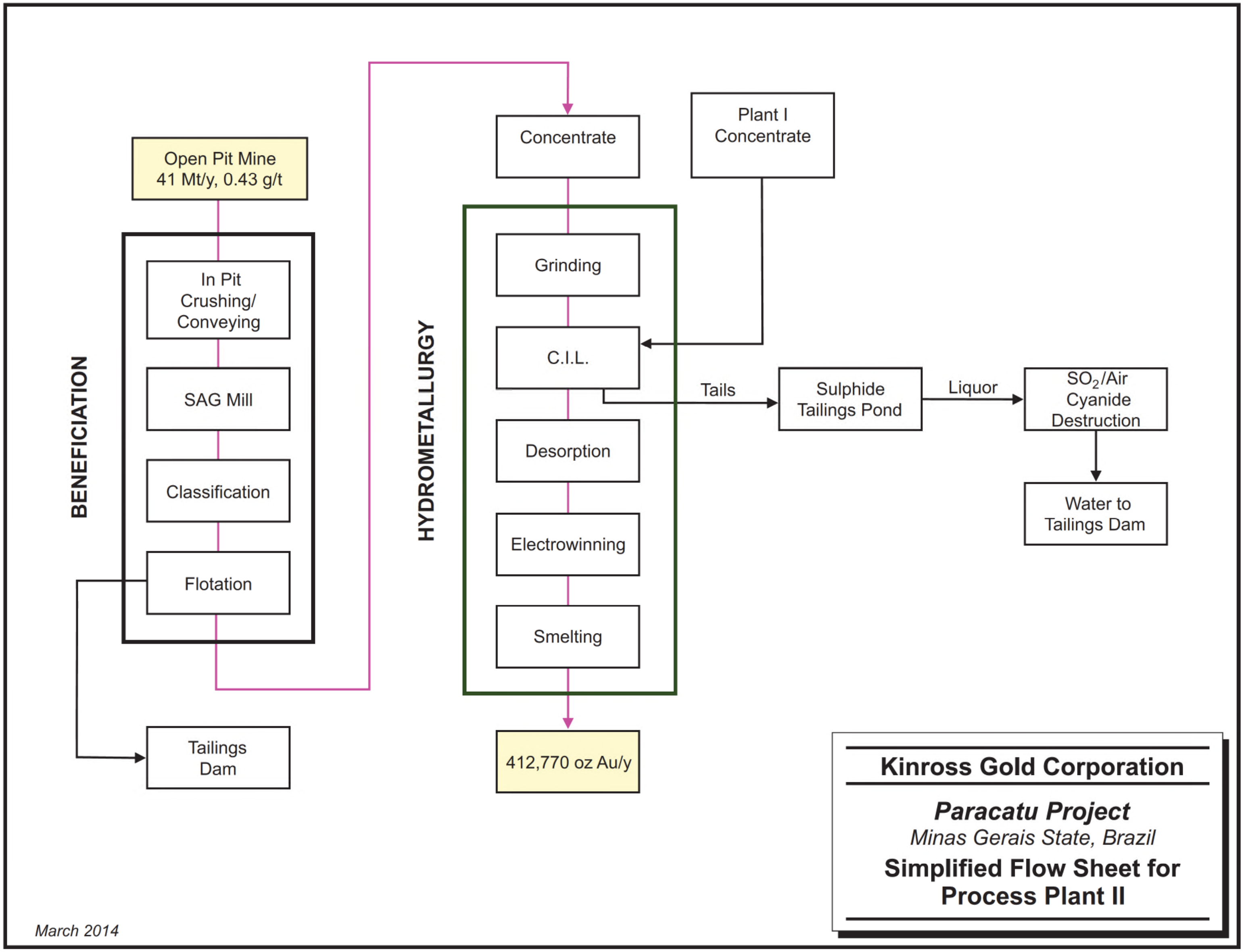

Figure 17-2: Simplified Flow Sheet for Plant II

17.7 Primary Crushing

The primary crusher is located within the open pit and is scheduled to operate 18 hours per day. Run-of-mine (ROM) ore is delivered by 240 t rear-dump, haul trucks to the 480 t crusher dump hopper. An apron feeder withdraws ROM ore from the dump hopper and feeds it at a controlled rate of approximately 7,000 t/h to an MMD 1300 Series Twin Shaft Sizer. The MMD sizer crushes the rock from a maximum size of 1,300 mm to a nominal size of 350 mm and discharges material directly onto a sacrificial conveyor, which in turn discharges onto the overland conveyor.

Three truck positions facilitate dumping to allow the MMD sizer to operate at its maximum instantaneous capacity. The middle dumping position is smaller than the two side positions and can accommodate up to 150 t trucks.

A stationary hydraulic rock breaker located at the MMD sizer feed chamber is used to break oversize rock that may be delivered. The MMD sizer can be removed from its operating position to a maintenance position by use of a winch and slide rails. A tower crane is used for maintenance of heavy components.

17.8 Stockpile

The crushed ore stockpile is a rectangular “A” frame with a dust and rain cover. Ore is delivered to the stockpile by a tripper conveyor. The stockpile provides 45,000 t of live volume. The total storage volume of 282,000 t can be accessed by a dozer.

The reclaim tunnel has six variable speed belt feeders. During normal operation, five of these feeders feed the SAG mill feed conveyor. The reclaim tunnel has a bag house, dust collector and escape tunnel.

17.9 Grinding

The Plant II grinding circuit is designed to operate at 5,400 t/h with an availability of 92%. The grinding circuit consists of one 11.6 m diameter by 6.7 m long Effective Grinding Length (EGL) 20 MW gearless drive SAG mill, two 7.3 m diameter by 12.0 m long EGL 13 MW ball mills and two 8 m diameter by 12.8 m long EGL 15 MW ball mills. Ball mills are equipped with dual pinion gear drives. The SAG mill operates in closed circuit with a trommel screen and vibrating screen, and the ball mills operate in closed circuit with hydrocyclones and flash flotation cells. Plant capacity has been selected to give a nominal flotation feed grind of 80% passing 75 µm.

Oversize rejects from the SAG mill are transferred to the SAG mill feed conveyor by three pebble conveyors in series. A pebble crusher is not provided for crushing SAG mill oversize, although provision is made for a possible future installation.

SAG mill discharge screen undersize and trommel screen undersize flow to a pump box from where it is pumped to the ball mill circuits. Each ball mill operates in closed circuit with a set of hydrocyclones. Cyclone underflow, at 73% solids, flows to a flash

Two liner handlers are provided for the SAG mills and the ball mills. One liner handler services the SAG mill and the other services the four ball mills. Jib cranes are located at each mill feed end to transfer new liners and scrap liners between the floor area and the liner handlers.

A 100 t capacity overhead crane with a 25 t auxiliary hoist is provided for the grinding bay.

17.10 Flotation, Desulphurization and Regrinding

The flotation circuit consists of rougher flotation followed by desulphurization cells and cleaning cells. Cleaner tails are recirculated to the rougher flotation cells and the cleaner concentrate is ground using a vertical stirred regrind mill operating in closed circuit with hydrocyclones. Rougher tailings are discarded to the tailings mix tank.

Flotation collector and frother are added to the slurry as it enters the four-way distribution box. A total of 24 rougher flotation cells are included, arranged in four rows of six cells each. The cells are 160 m^3 tanks fitted with self-aspirating mechanisms.

Reagents (collectors and frother) required for gold rougher flotation are supplied from the existing reagent tanks to head tanks in the grinding circuit, fitted with metering pumps adding reagents to the grinding hydrocyclone overflow launder. Collector required for sulphide rougher flotation is supplied via a loop fed from the retrofitted lime preparation system. Frother for sulphide flotation is supplied via a loop from an existing reagent tank at the flotation plant.

Concentrate from the rougher flotation cells is pumped to the cleaner flotation stage, consisting of three 40 m^3 capacity mechanically-agitated, forced-air cells in series. Cleaner concentrate is pumped to the existing final concentrate pump box.

Concentrate from the sulphide rougher flotation cells, along with gold cleaner tailings and sulphide re-cleaner tailings, is pumped to the existing cleaner circuit and renamed sulphide cleaner. The existing cleaner flotation cells, cleaner flotation area sump pumps and cleaner tailing pump box are required for sulphides cleaning.

The existing cleaner concentrate pump box and pumps is used as the final concentrate point, collecting concentrate from gold cleaners and sulphide re-cleaners.

All pumping systems, except for the gold cleaner and sulphide re-cleaner concentrate pumps, are horizontal centrifugal pumps with an operating pump and an installed standby pump. Both gold rougher concentrate pumps operate in parallel when using the 2-4 split. Gold cleaner and sulphide re-cleaner concentrate is pumped using tank pumps (vertical centrifugal pumps with integrated pump box; one is in operation and a second is an uninstalled spare).

Concentrate from the sulphide cleaner flotation cells is combined and pumped to the sulphide re-cleaner flotation cells, consisting of six 40 m^3 capacity mechanically agitated, forced-air cells in series. Sulphide re-cleaner concentrate is pumped to the final concentrate pump box.

Sulphide cleaner flotation tailings are pumped to the rougher tailings sampler, opening the flotation circuit. The option of pumping back to the rougher feed distributor box, closing the flotation circuit, was implemented into the design.

Flotation air blowers (one operating, one standby) supply compressed air to the gold cleaner and sulphide re-cleaner flotation cells.

The cleaner cells consist of two rows of five self-aspirated 60 m^3 tank cells. Cleaner tailings flow to a pump box and are pumped by a horizontal slurry pump to the rougher flotation distribution box. Cleaner concentrate is collected in a single pump box and pumped by horizontal slurry pumps to the solution removal thickener.

Both the gravity concentrate and the cleaner concentrate report to the solution removal thickener. Sieve bends are used to remove the coarse heavy particles in the gravity concentrate and direct them to the regrind mill. The purpose of the thickener is to remove solution containing flotation reagents and also to prepare the slurry to the optimum density for regrinding.

The concentrate regrind mill (13.5 m high, 931 kW vertical stirred type) grinds the combined cleaner and gravity concentrate to approximately 80% passing 40 µm. The mill operates in closed circuit with ten 254 mm diameter hydrocyclones to affect the classification.

17.11 Carbon-In-Leach Circuit

Plant I was upgraded from 35 t/h to 100 t/h by installing a new pre-aeration tank and four new CIL tanks. Pre-aeration residence time will be four hours and CIL residence time will be 36 hours.

Concentrate thickener underflow is pumped to a new trash screen located over the pre-aeration tank at the CIL plant. The cleaned slurry passes through the screen to the 750 m^3 agitated pre-aeration tank where milk of lime slurry is added. This tank overflows to four 750 m^3 CIL tanks. The flow from the fourth tank splits into two lines that flow into four 300 m^3 CIL tanks. Carbon is retained in each tank by swept North Kalgoorlie Mines (NKM) type screens. Cyanide and lime from existing make-up systems are staged and added to each tank train. On an intermittent basis, loaded carbon is pumped countercurrent to the slurry flow in order to increase the gold loading. Loaded carbon is removed from the first CIL tank, drained and washed on two vibrating screens and transferred to the loaded carbon bin. A carbon safety screen is located over the CIL tails pump box.

17.12 Carbon Elution and Regeneration

The carbon elution and regeneration plant is located in a building previously occupied by a redundant filtration plant. The plant is designed to handle a carbon batch size of 14 tonnes at 4,500 grams of gold per metric tonne of carbon.

The pregnant and barren eluate tanks are located in an area previously occupied by two small regeneration kilns. The new 600 kg/h kiln will be constructed and commissioned prior to demolishing the smaller existing kilns.

Loaded carbon is pumped to an acid wash vessel. After acid washing with 3% HCl is complete, the spent acid is neutralized with sodium hydroxide from the existing make-up system before discarding it to the tails pump box.

The elution cycle operates with a 0.2% sodium cyanide and 1% sodium hydroxide solution at a temperature of 145°C and a pressure of 450 kPa for approximately eight hours.

Stripped carbon is evacuated from the bottom of the elution vessel and this is processed in a new regeneration kiln to restore carbon activity.

17.13 Electrowinning and Refining

Pregnant solution is pumped to four electrowinning cells located adjacent to a single cell on the upper floor of the existing refinery. Gold is electrowon on the stainless steel wool cathodes. At the end of each cycle, the cathodes are removed from the cells and the gold-bearing sludge is recovered in a filter press. The resulting filter cake is dried, mixed with fluxes, usually borax, soda ash and occasionally sodium nitrate and fed to an existing electric induction furnace. The doré metal and slag separate in the furnace, and the slag is poured off into slag pots. Then the doré metal is poured into bars for shipment.

18. PROJECT INFRASTRUCTURE

Paracatu infrastructure and services have been designed to support an operation of 61 Mt/a.

The mine site consists of two processing plants, related mine services facilities (truck shop, truck wash facility, warehouse, fuel storage and distribution facilities, reagent storage and distribution facilities), and other facilities to support operations (safety/security/first aid/emergency response building, assay laboratory, plant guard house, dining facilities, offices etc.).

The site infrastructure layout is illustrated in Figure 18-1.

18.1 Access

Access to the mine site was initially through the town of Paracatu, which sits on the border of the southern side of the pit. However, as traffic to the mine intensified over time, the town requested a change to the access road route. After receiving governmental authorization in 2011, KBM constructed a road directly from the BR-040 highway west of the mine to the mine and plants. The paved road has four lanes that are separated by a median and its 3.4 km length is exclusively for mine traffic.

18.2 Power

The mine draws its power from the Brazilian national power grid which is largely based on hydroelectric power generation. KBM is connected to the 500 kV national grid via a 500 kV/230 kV substation owned by the mine. A 230 kV transmission line, approximately 34 km long, feeds the mine from this substation. This transmission line is connected to substation 43-SE-501 located at the mine site which subsequently feeds the Pant II distribution system at 13.8 kV and Plant I transmission line at 138 kV. The 138 kV Plant I transmission line feeds a 138 kV/13.8 kV substation located at Plant I, which subsequently feeds the Plant I distribution system.

Electricity purchase is subject to a free market environment with consumers able to select their supplier of choice. In order to reduce risk and provide flexibility, KBM procures power based on a portfolio approach with a combination of short and long term contracts from a number of suppliers. Part of KBM's power needs are now fed by a self-generation project that provides lower cost electricity from its 95% stake (46 MW) in a thermal power plant in a joint venture with Eneva Energy. The project, which was granted self-generation status in January 2014, has an innovative commercial structure and the estimated benefit to KBM resulting from the project is US $49 million over five years. KBM continues to pursue options for electricity supply and energy efficiency in order to reduce production costs.

The mine has a small emergency power capability, used for critical process equipment that cannot be suddenly stopped, such as thickeners and CIL tank agitators.

18.3 Water

The main water sources for KBM operations are run-off water collected in the mine sumps, run-off water collected in the tailings dam catchment basins, recirculated effluent from processing activities, and make-up water from three local surface water streams. The majority of process water is captured and maintained in the mine sumps and tailings catchment basins during the rainy season for use during the dry season. The current operating plan has all water in mine sumps pumped to the plants continuously with Eustáquio recycle water pumping set to the desired rate to maintain total demand.

Main water losses outside of process uses are from evaporation, percolation to groundwater, seepages from the tailings dam toe drains which maintain the ecological water flows of respective creeks, pumping to Rico Creek to maintain ecological flow and using water for dust control around the mine and surface infrastructure.

Make-up water is pumped from three local streams when flows exceed nominal stream flows, as specified in the extraction permit. The São Pedro stream discharges into the São Domingos catchment basin and can be pumped at a rate of 1,497 m^3/h over a 24 hour period. Additionally, the Santa Rita and São Domingos streams are permitted for 24 hour pumping rates of 300 m^3/h and 800 m^3/h. A pipeline connects the catchment at the Santa Rita stream to the existing reservoir at the São Domingos catchment. Water from the São Domingos catchment is pumped to the Santo Antônio Tailings Dam for storage until it is used.

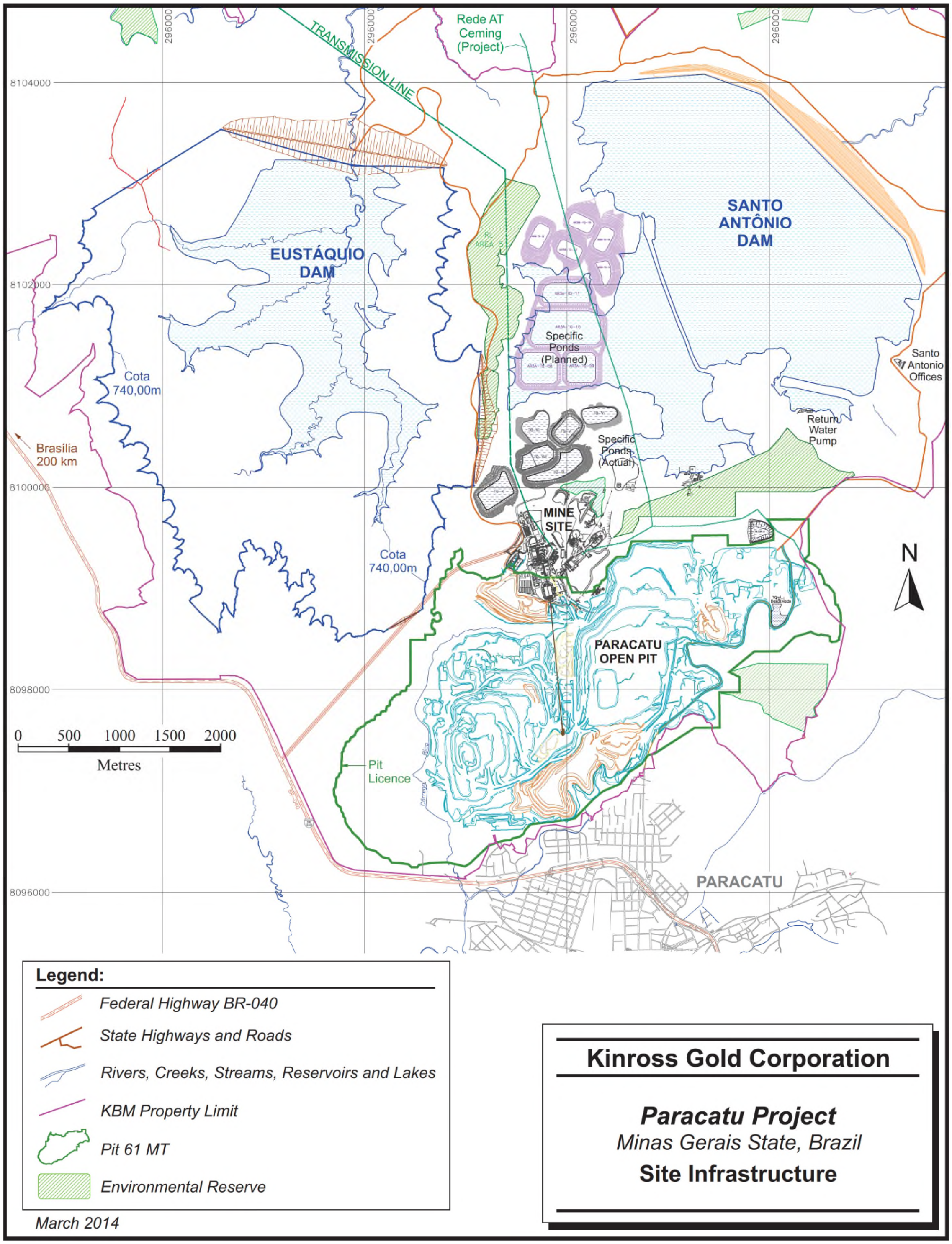

Figure 18-1: Site Infrastructure

19. MARKET STUDIES AND CONTRACTS

Kinross typically establishes refining agreements with third parties for the refining of doré bars. The Company retains marketing experts in-house to facilitate the sale of bullion on the spot market or as doré. 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 and Management

Kinross Paracatu is constantly seeking excellence in its occupational health and safety, environmental and social responsibility system. The site maintains certifications in ISO 14001, OHSAS 18001, SA 8000 and is in compliance with the Cyanide Management Code.

Environmental analyses focus on operations in the KBM mining licence area and related facilities. An Environmental Impact Control Plan (PCIAM) has been implemented to ensure compliance with environmental permits. This plan focuses on such topics as air and soil quality, noise and vibration control, preservation of waterways, re-vegetation and tree canopy planning, the safety and hygiene of workers, and reclamation projects as well as occupation proposals and future use of the areas affected by mining. The key aspects and environmental impacts addressed in the plan include deforestation, relocation of mammals and birds, soil removal and compaction, dust mitigation, fossil fuel emissions, rainwater runoff, noise and vibrations from operations and blasting, and changes to the landscape.

KBM's Environmental Impact Control Plan and Closure Plan were updated in 2010 to reflect changes in mining activities, processing, and final disposal of waste and tailings.

Diversion of Rico Creek

The Expansion Project III operation permit approved pit expansion through a two kilometre length of Rico Creek. In order to allow mining in the western portion of the pit, KBM intercepted and diverted a branch of Rico Creek in 2010. This process required KBM to obtain the relevant water grant from the State Water Agency (IGAM). Pertinent hydrogeological studies were performed by KBM and a minimum ecological flow is being maintained in accordance with the recommendation of 32 m^3/h. A second diversion will be implemented by 2015 ahead of continued mine expansion to the southeast.

The course of the Rico Creek has been deeply affected by siltation and mercury contamination resulting from artisanal mining activities. Historically, Rico Creek has also received domestic sewage from the city of Paracatu. A local water and sewage company installed a sewage collection and treatment system that has been in operation since October 2005. Most environmental impacts have been observed downstream of the mine site. To compensate its impacts on Rico Creek, Kinross agreed to implement a revitalization project of these downstream areas with the support of the Environmental Agency, local authorities, the Paracatu mayor and the local Public Attorney. The revitalization project includes improving the local drainage system, controlling erosion, and re-establishing nearby woodland areas in order to enhance downstream areas of the creek that cross the city. A park was also built and a second park is planned.

At the end of the mine life, a pit lake will be formed and integrated into the landscape and the ecological flow of Rico Creek will be discharged from the pit lake. The overall area will be reclaimed and vegetated.

Tailings

The main tailings storage facilities are subject to a rigorous audit and review process. In 2010, external consultants identified the need to install a buttress to increase the Santo Antônio dam's overall factor of safety as it entered its final phase of dam raise construction. Currently the Santo Antônio dam has completed all the dam raises planned for its operational life and is initiating closure plan measures. A concrete emergency spill way was installed between 2012 and 2013. Re-vegetation of the dam area began in 2013. A final drainage system is planned to be installed in the coming years. The Santo Antônio tailings dam has storage capacity for the disposal of Plant I tailings through October 2014 with possible capacity augmentation through to April 2015.

The new Eustáquio tailings facility was commissioned in the fourth quarter of 2012. It has a planned footprint of 1,300 ha, significantly expanding KBM's water storage and tailings capacity. The dam is being constructed in stages and has a planned final design crest height of 140 m. Passive treatment effectively reduces levels of manganese and other metals in tailings seepage below permit limits. The Eustáquio reservoir complements the storage capacity provided by the Santo Antônio reservoir.

In 2013, a passive treatment system was installed in the Eustáquio creek slightly downstream of the dam. This system is similar to the one installed at Santo Antônio Tailings Storage Facility (TSF).

Acid and Cyanide Management

To reduce the potential of tailings to produce acid, sulphur levels in tailings is reduced via floatation prior to disposal (Section 17.4). Concentrated sulphur is placed in lined ponds (referred to as specific tanks).

A small fraction of the mill feed is recovered as sulphide concentrate. After gold recovery by cyanide leaching, the residual solids (sulphide tailings), including the residual cyanide solution, are permanently stored in lined waste repositories referred to as “specific tanks” that were specially designed to prevent leakage into the groundwater and any contamination of the environment. The potential liabilities of these tailings are related to the high sulphide mineral content and the concentration of cyanide. The total cyanide concentration averages below 50 ppm.

The specific tanks are located close to the plants between the basins of the Santo Antônio and Eustáquio tailings dams. This ensures that the water discharged from the tailings ponds into the Santo Antônio and Eustáquio Creeks is of a quality that complies with the local regulatory standards and World Bank guidelines.

The current management of these sulphide tailings, including the installation of composite liners in the specific tanks and sub-drains to intercept and collect any potential leakage, is considered to provide appropriate containment. Multi-layer covers will be completed on reclaimed specific tanks as part of the closure process. At present there are seven specific tanks that have been closed with an initial compaction layer. Two specific tanks are in operation. Final cover design of specific tanks will be implemented after a decision regarding possible re-processing of specific tank material. In the last three years, KBM has re-processed the tailings disposal at specific tanks I and II, and completely removed the contained sulphur concentrate.

KBM has used cyanide as part of its final hydrometallurgical carbon-in-leach (CIL) circuit, where it is used to leach the gold into solution. Prior to discharge of tailings, cyanide levels are reduced below 50 ppm as required by the "International Cyanide Management Code for the Manufacture, Transport, and Use of Cyanide in the Production of Gold". In the Santo Antônio and Eustáquio tailings dams the residual cyanide is naturally degraded by volatilization and ultraviolet rays from sun light. KBM monitors residual cyanide discharge and degradation within the tailings facilities. All measurements to-date have been below detection limit. This information is forwarded to the Brazilian environmental agency as part of KBM’s operational license (LO) requirements.

In recent years, KBM has implemented a series of improvements to reduce cyanide consumption, including the implementation of automatic feed systems, which has reduced the consumption of this reagent in kg/t ore processed. A further improvement that was introduced during the expansion is a pre-aeration step prior to leaching, which enables the current cyanide consumption levels to be maintained and not increased despite the increase in the amount of concentrate to be processed.

Dust, Vibration and Noise Management

In the last few years, KBM has implemented many actions regarding dust, vibration and noise control, seeking continuous improvement in compliance with internal standards and local legislation.

Air emissions from mining occur primarily as fugitive dust from mining, transportation, waste rock handling and ore crushing from the in-pit crusher.

In order to minimize emissions from fugitive dust from the mine, KBM operates water trucks 24 hours per day. Dust suppressant is also applied annually to avoid particulate dispersion due to wind disturbance. Additionally, some roads are covered with laterite or coarse limestone to reduce dust generation.

In the in-pit crusher, dust is controlled by the use of water sprays during dumping and a back-up bag house when the water system is under maintenance. The stock pile is covered to avoid dust dispersion and contains water sprays to limit dust generation.

Frequent inspections of dust control systems are performed. A program of Opacity Measurement is in place to standardize and facilitate the inspections.

In addition to dust-generating mining activities, there are also point source stack emissions from onsite facilities that perform carbon regeneration, electro-winning, smelting and laboratory processes. Emission controls from point sources are installed and an annual emission inventory is maintained onsite.

As explosives are required for blasting, KBM has an air pressure and vibration management program.

All mine planning takes the noise and vibration impacts associated with blasting into consideration. To minimize disturbance, the daily blast occurs at 3:00 pm on weekdays only, as per an agreement with the residential community. Any changes to this schedule are agreed upon with the community prior to blasting. KBM uses the best technologies and procedures to minimize the amount of vibration in the city of Paracatu. These procedures include careful planning of blast layout patterns and consideration of wind direction to avoid carrying dust towards the city.

In order to reduce the noise levels associated with mining activities, KBM built a noise barrier waste stockpile between the boundary of the pit and the vicinity of the residential community. Besides reducing the noise level in the residential community, this barrier also minimizes the visual impact of the mine. This barrier has been raised incrementally each year to further improve dust and noise management.

KBM conducts daily noise level surveys in the area surrounding the mine during the night shift.

20.2 Permitting

Brazilian environmental policy is executed at the federal, state and municipal levels of public administration. Coordinating and formulating the Brazilian Environmental Policy is the responsibility of the Environmental Ministry. Directly linked to this body is the National Environmental Council (CONAMA), the deliberative and consultative board for environmental policy. CONAMA's responsibility is to establish the rules, standards and guidelines so that environmental licensing can be granted and controlled by the state and municipal environmental agencies. These agencies are part of the National Environmental System (SISNAMA), and the Brazilian Institute for the Environment and Renewable Resources (IBAMA). IBAMA is the government agency under the jurisdiction of the Environmental Ministry and is the agency responsible for executing the Brazilian Environmental Policy at the federal level.

The basic environmental impact assessment process is initiated with the collection of baseline data following the submission of a conceptual mine plan. Baseline data collection is followed with a formal Environmental Impact Assessment (EIA). The Environmental Impact Report (RIMA) is a summary of the EIA presented in language adequate for public communication and consultation. The EIA and RIMA are made available for public review and comment during the public hearings.

Once the EIA/RIMA process is complete, there are three components of the Environmental License that must be completed. The License is issued by the State Agency (SUPRAM NOR) in an integrated process that includes the Forest Agency (IEF), Water Agency (IGAM) and Environmental Agency (FEAM). This integration was consolidated in the Minas Gerais State in 2007 with the creation of eight Regional Agencies. Kinross Paracatu reports to the office in Unaí, located 100 km from Paracatu. The additional permits required for the Environmental License are below:

Previews Permit (LP) - This is relevant to the mining project's preliminary planning stage. The permit contains the basic requirements of the municipal, state and federal agencies for soil use during the location, installation and operation stages.

Requirements must meet regulations, criteria and standards set out in the general guidelines for environmental licensing issued by the State Level (DN 74/2004), providing there is no conflict with federal level requirements.

The Mining Plan and the EIA/RIMA are technical documents required in order to obtain the Previews Permit. This process is concurrent with the request for a mining concession.

This phase is related to the environmental feasibility.

Installation Permit (LI) – This authorizes the installation and implementation of the mining project according to the specifications in the approved Environmental Control Plan. The LI can be requested only after the completion of the LP. The Economic Development Plan (PAE) and the Environmental Control Plan (PCA) are prepared in parallel and approved by the National Department for Mineral Production (DNPM) and the local Environmental Agency, respectively. The Environmental Control Plan (PCA) is based on the Environmental Management System (SGA). These two plans are required for the Installation Permit and the land clearing (deforestation) permit to be issued. At this stage, a closure plan is made and submitted to the DNPM for approval.

Operating Permit (LO) – This authorizes the start of permitted operation activities according to what was established in the Preview and Installation Permits. The LO has the duration of four years and can be renewed. The revalidation permit is granted after the submission of the Environmental Performance Report (RADA) and its evaluation by the environmental agency (SUPRAM NOR). During the operating phase of the Project, the Annual Mining Report (RAL) is submitted by the company for the approval of the DNPM. In the closure phase, after the decommissioning, restoration and environmental monitoring operations are have been completed, the company applies for a Conformity Certificate from the environmental agency and DNPM. As part of these permitting requirements, Kinross Paracatu must submit an environmental report.

Figure 20-1 shows a simplified diagram of the environmental and mining rights, licensing and control processes.

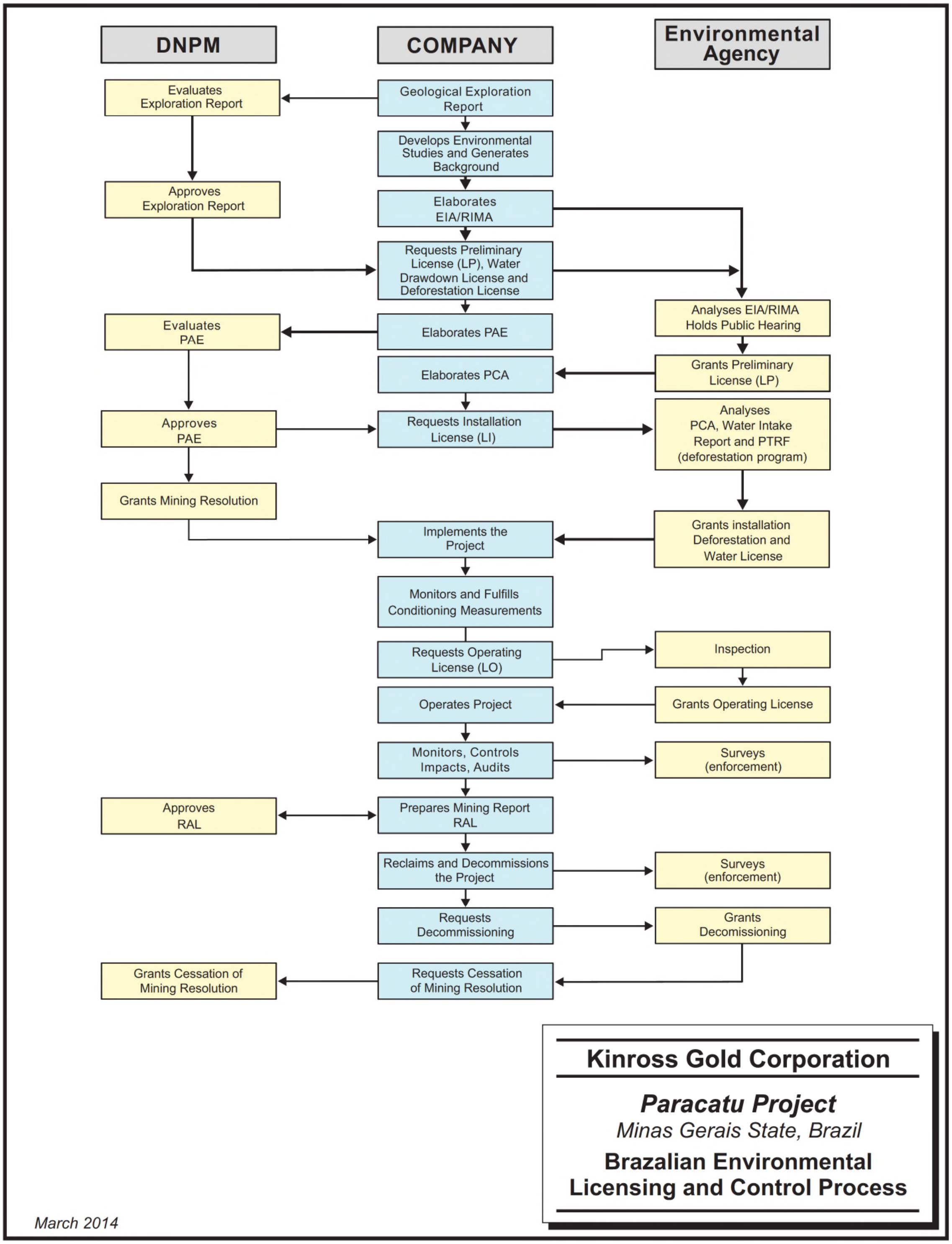

Figure 20-1: Brazilian Environmental Licensing and Control Process

Permitting Status

Environmental permits related to Expansion Plan III (Plant II) have been granted from the Environmental Regulatory Authorities.

The Operation Permit (LO) for the Eustáquio tailings dam was granted in November 2011 with 17 conditions that are ongoing. The first stage of the new tailings facility was finalized in 2010 and the second stage was completed in 2011. Tailings disposal started in April 2012.

The Operation Permit (LO) for a 61 Mt/a throughput pit was granted in July 2010 with 21 permit conditions that are ongoing.

Some other permits for water usage and deforestation were also granted in 2011. One requirement of the Eustáquio water permit was the diversion of certain springs around the tailings facility to ensure that the minimum flow downstream of the TSF is maintained.

In 2012, Kinross was granted the seasonal pump authorization and the deforestation approval to relocate the main power line. The power line supplies power for all the water consumption for the Expansion III project.

Every four years the site must renew its operating permits. This process was most recently completed in July 2013 and resulted in the approval of Paracatu's operating permits for the next four years by the multi-agency body. The only exception applies to the Eustáquio tailings dam, for which the permit was granted at the end of 2012. As part of the operating permit renewal process, the agencies complete a comprehensive review of Kinross Paracatu's environmental and operating performance over the previous four years and determines if the site has been in compliance with their obligations.

20.3 Social and Community Related Requirements

Given the proximity of the mine to the city of Paracatu, the primary concern for community impact relates to dust, vibration and noise, as discussed above. Kinross Paracatu also has various projects designed to focus on the community, with the goal of preserving the cultural identity of the region, and supporting community health and education. This includes an Environmental Education Program that answers current and local environmental questions for the community and employees.

20.4 Mine Closure Requirements and Costs

KBM has a comprehensive and updated closure plan that includes an estimation of closure cost. The plan is based on the closure plan strategy developed in 2010 by Golder Associates using Kinross Guidelines (Golder, 2010). The closure plan covers all affected areas including the mine pit area, tailings dams and ponds that are used to dispose of the sulphide concentrate. KBM will implement specific measures to safely close down its facilities as described below.

Mine Pit Area

Oxidized or non-sulphide areas will be reclaimed by direct treatment of the surface (grading) and growing of grass and legume species. Sulphide-exposed areas will receive a cover of waste material and soil to act as a capillary break and prevent acid mine generation. A drainage system for controlling runoff water will be implemented and maintained until the reclaimed areas become stable. A pit lake will be formed in the west part of the mine. KBM conducted hydrogeological studies to identify the potential risk of impacting groundwater flow and quality. These studies recommended reducing the exposure time of sulphides and addressing poor water quality by pumping water from the tailings dam to the pit lake.

Tailings Dam Areas

The closure of the tailings dam areas will occur in three phases. The first phase is a partial closure of Santo Antônio Dam, which has already been started with the reclamation of the dam wall and borrow pit areas. The second phase is to cover the tailings exposed areas (beaches) close to the discharge point. The third phase will occur at the end of operations with the complete closure and rehabilitation of the Santo Antônio Dam and the Eustáquio Dams. A 1.5 m cover of saprolite is currently planned to be placed over the tailings disposed in the basins. This cover will considerably reduce the risk of exposing any residual sulphides contained in the tailings materials to atmospheric oxygen and reduce the risk of generating acid drainage.

The closure of the tailings dam ponds will occur in segments by installing dykes in specific branches of each pond. The Santo Antônio tailings storage will be relocated upstream during the end of the operational stages by the use of dykes that will later be removed in the final closure stage of the dam to achieve a higher safety coefficient. Peripheral canals will be constructed on both sides of the tailings pond to drain any stored water and to dry the stored tailings.

The current mine plan requires that the last layers to be deposited in Santo Antônio to be from ore with low sulphur content to reduce acid generation. A buttress was installed to increase the safety factor of Santo Antônio Dam as it was raised to its final level in 2011. In 2013, Kinross initiated the development of a new conceptual plan for closure and rehabilitation of Santo Antônio Tailings Dam area. In the same year the installation of a spillway in the Santo Antônio Dam was completed, as there will be no further raises of the primary dam structure.

An evaluation of the final water table is required to properly select the species of vegetation to grow in dry and wet areas. The tailings embankment slopes and borrowing areas downstream of the dam will be adequately rehabilitated with the installation of a drainage system over the benches and with the use of grass and legume species selected to prevent erosion.

CIL Sulphide Concentrate Specific Tanks

The sulphur concentrate disposal facilities (“specific tanks”) require a special strategy for closure in order to prevent atmospheric exposure and potential acid generation. Specific tanks II and III were removed and re-processed, while two specific tanks remain in operation. A total of seven specifics tanks have been closed.

In 2012, a cover design was developed for specific pond closure. First, a layer of oxide overburden material will be directly dumped and compacted over the tailings. After the oxide overburden, a thin layer of coarse material (ground limestone or saprolitic material) will be laid to act as a capillary break. A local clay that has been proven to have a high adsorption capacity of potential contaminants will be installed on top, followed by a layer of fine silt material. Finally, an organic soil will be placed on the top of the surface and selected grass and legume species will be planted. The final covering of specific tanks will be implemented after a decision regarding possible re-processing of specific tank material is made.

In order to reduce the seepage of water to lower levels, and to prevent erosion, a drainage system to collect and divert runoff water will be installed. Periodic maintenance of surface drain systems and of re-vegetation will be required for at least five years after the closure of the specific ponds to ensure sustainable reclamation.

Buildings and Ancillary Facilities

Any remaining buildings that would not be demolished, and ancillary facilities such as roads, power lines and pumps, will be maintained and passed to local Paracatu authorities for municipal use.

Future Land Use and Sustainability of Closure

KBM intends to integrate its closure strategy with local expectations, but there will be restricted use of some areas (sulphide mined areas, tailings dams and specific ponds).

Future potential land uses include the development of parks and the potential for areas to be used for educational and recreational purposes.

KBM has conducted surveys and discussions with the local community to identify perceptions and expectations in relation to site closure. Additionally, KBM has initiated a series of programs to address the social impacts arising from the site closure. For instance, a local agency for developing alternative sources of income has been created with support from the company.

Funding Closure

Reclamation activities that are performed during the life of mine are funded from the mine's cash flow. Reclamation activities performed after a mine has completed operation will be funded by Kinross' portfolio of operating mines.

21. CAPITAL AND OPERATING COSTS

21.1 Capital Costs

Remaining capital costs at Paracatu are primarily sustaining capital, which includes mine equipment replacement and $793 million for the tailings dam expansions. Total sustaining capital costs are $1,453 million in real terms. Mine pre-stripping capital has been treated as an operating cost for the purposes of this Technical Report.

Table 21-1: Sustaining Capital for LOM

Area		Sustaining Capital
Mine Mobile Equipment	(US$000s)	450,127
Mine Other	(US$000s)	34,955
Processing Facilities	(US$000s)	182,987
Tailings Facilities	(US$000s)	792,963
Site Infrastructure	(US$000s)	18,324
Major Development Projects	(US$000s)	57,363
Information Technology	(US$000s)	9,067
Other	(US$000s)	-92,617
Total	(US$000s)	**1,453,170**

21.2 Operating Costs

Operating costs are tracked and well understood. Unit operating costs for the LOM production schedule total US$8.79 per tonne processed.

Table 21-2 below presents a summary of the production and operating costs used in the LOM schedule.

Table 21-2: Paracatu Production and Cost Summary

Area	Unit	Cost
Mining	(US$/t)	$ 2.36
Processing	(US$/t)	$ 5.35
Site Admin	(US$/t)	$ 1.07
Total	**(US$/t)**	**$ 8.79**

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, unless the Technical Report prepared by the issuer includes a material expansion of current production. Kinross is a producing issuer, the Paracatu mine is currently in production, and a material expansion is not included in the current LOM plans. Kinross has carried out an economic analysis of the mine using the estimates presented in this report and confirms that the outcome is a positive cash flow that supports the statement of Mineral Reserves.

23. ADJACENT PROPERTIES

There are no other producing mines near the Paracatu mine. There are undeveloped gold showings in the vicinity of Paracatu, but they have not proven to be viable exploration targets.

No reliance was placed on any information from adjacent properties in the estimation and preparation of the resources and reserves reported in this Technical Report. Adjacent properties are therefore not deemed material to this report.

24. OTHER RELEVANT DATA AND INFORMATION

No additional information or explanation is necessary to make this Technical Report understandable and not misleading.

25. INTERPRETATION AND CONCLUSIONS

Mineral Resource Estimation

- The EOY2013 Measured and Indicated Mineral Resources, exclusive of Mineral Reserves, total 540.2 million tonnes averaging 0.36 g/t Au and contain 6.2 million ounces of gold.

- The EOY2013 Inferred Mineral Resources total 3.2 million tonnes averaging 0.27 g/t Au and contain 28,200 ounces of gold.

- The resource estimate gold cut-off grades range from 0.139 g/t for B1 ore to 0.208 g/t for B2 ore, which requires more intense grinding.

- Mineral Resource estimates have been prepared using acceptable estimation methodologies. The classification of Measured, Indicated, and Inferred Resources conform to Canadian Institute of Mining, Metallurgy and Petroleum Definition Standards for Mineral Resources and Mineral Reserves dated November 27, 2010.

- Protocols for drilling, sampling, analysis, security, and database management meet industry accepted practices. The drill hole database was verified and is reasonable for supporting a resource model for use in Mineral Resource and Mineral Reserve estimation.

- Kinross is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other modifying factors which could materially affect the open pit mineral resource estimates.

Mining and Mineral Reserves

- The EOY2013 open pit Proven and Probable Reserves, including existing stockpiles scheduled for processing, are estimated to be 764 million tonnes at 0.424 g/t Au, containing 10.4 million ounces of gold.

- The reserve estimate gold cut-off grades range from 0.163 g/t for B1 ore material feed (going to Plant I) to 0.244 g/t for B2 ore material feed (going to Plant II), which requires more intense grinding.

- The Mineral Reserve estimates have been prepared using acceptable estimation methodologies and the classification of Proven and Probable Reserves conform to CIM Definition Standards.

- Recovery and cost estimates are based on actual operating data and engineering estimates.

- Economic analysis of the Paracatu Life of Mine (LOM) plan generates a positive cash flow and meets the requirements for statement of Mineral Reserves. In addition to the Mineral Reserves in the LOM plan, there are Mineral Resources that represent opportunities for the future.

- One major change between EOY2013 and EOY2012 is the inclusion of sustaining capital costs as operating costs for the purposes of pit shell calculation. These costs, including waste pre-stripping, mine fleet replacement costs, and tailings dam construction, were traditionally considered as capital costs, but are now included as operating costs. This change has a large impact on the overall reserves calculation since the LG cone calculation takes these added costs into consideration when sizing the overall economic pit. The end result is a smaller overall pit with higher grade.

<u>Process</u>

- Paracatu has two mineral processing plants, Plant I and Plant II. Plant I focuses on treating the softer near-surface B1 ore at a throughput of 20 Mt/a, whereas Plant II focuses on treating the harder B2 ore at a throughput of 41 Mt/a.

<u>Environmental Considerations</u>

- KBM is constantly seeking excellence in its occupational health and safety, environmental and social responsibility system. The site maintains Certifications in ISO 14001, OHSAS 18001, SA 8000 and compliance with the Cyanide Management Code.

- KBM’s Environmental Impact Control Plan and Closure Plan were updated in 2010 to reflect changes in mining activities, processing, and final disposal of waste and tailings.

26. RECOMMENDATIONS

There are no recommendations at this time as Paracatu is a fully operational mine.

27. REFERENCES

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28. DATE AND SIGNATURE PAGE

The effective date of this Technical Report entitled "Kinross Gold Corporation, Paracatu Project, Brazil, NI 43-101 Technical Report" is March 31, 2014.

"Signed and sealed"

John Sims, AIPG Certified Professional Geologist

Dated March 31, 2014